BVT-Merkblatt zum „Management von Bergbauabfällen und Taubgestein"

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mit ausgewählten Kapiteln in deutscher Übersetzung

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Das Bundesministerium für Umwelt, Naturschutz und Reaktorsicherheit und die 16 Bundesländer haben eine Verwaltungsvereinbarung geschlossen, um gemeinsam eine auszugsweise Übersetzung der BVT-Merkblätter ins Deutsche zu organisieren und zu finanzieren, die im Rahmen des Informationsaustausches nach Artikel 16 Absatz 2 der Richtlinie 96/61/EG über die integrierte Vermeidung und Verminderung der Umweltverschmutzung (IVU-Richtlinie) (Sevilla-Prozess) erarbeitet werden. Die Vereinbarung ist am 10.1.2003 in Kraft getreten. Von den BVT-Merkblättern sollen die für die Genehmigungsbehörden wesentlichen Kapitel übersetzt werden. Auch Österreich unterstützt dieses Übersetzungsprojekt durch finanzielle Beiträge.

Als Nationale Koordinierungsstelle für die BVT-Arbeiten wurde das Umweltbundesamt (UBA) mit der Organisation und fachlichen Begleitung dieser Übersetzungsarbeiten beauftragt.

Die Kapitel des von der Europäischen Kommission veröffentlichten BVT-Merkblattes „[hier den vollständigen Namen des BVT-Merkblattes einfügen]“, in denen die Besten Verfügbaren Techniken beschrieben sind [hier die Kapitelnummern der übersetzten Kapitel einfügen], sind im Rahmen dieser Verwaltungsvereinbarung in Auftrag des Umweltbundesamtes übersetzt worden.

Die nicht übersetzen Kapitel liegen in diesem Dokument in der englischsprachigen Originalfassung vor. Diese englischsprachigen Teile des Dokumentes enthalten weitere Informationen (u.a. Emissionssituation der Branche, Technikbeschreibungen etc.), die nicht übersetzt worden sind. In Ausnahmefällen gibt es in der deutschen Übersetzung Verweise auf nicht übersetzten Textpassagen. Die deutsche Übersetzung sollte daher immer in Verbindung mit dem englischen Text verwendet werden.

Das Kapitel „Zusammenfassung“ basiert auf der offiziellen Übersetzung der Europäischen Kommission in einer zwischen Deutschland, Luxemburg und Österreich abgestimmten korrigierten Fassung.

Die Übersetzungen der weiteren Kapitel sind ebenfalls sorgfältig erstellt und fachlich durch das Umweltbundesamt und Fachleute der Bundesländer geprüft worden. Diese deutschen Übersetzungen stellen keine rechtsverbindliche Übersetzung des englischen Originaltextes dar. Bei Zweifelsfragen muss deshalb immer auf die von der Kommission veröffentlichte englischsprachige Version zurückgegriffen werden.

Dieses Dokument ist auf der Homepage des Umweltbundesamtes (http://www.bvt.umweltbundesamt.de/kurzue.htm) abrufbar.

Durchführung der Übersetzung in die deutsche Sprache:

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This document is one of a series of foreseen documents as below (at the time of writing, not all documents have been drafted):

Full title	BREF code
Reference Document on Best Available Techniques for Intensive Rearing of Poultry and Pigs	ILF
Reference Document on the General Principles of Monitoring	MON
Reference Document on Best Available Techniques for the Tanning of Hides and Skins	TAN
Reference Document on Best Available Techniques in the Glass Manufacturing Industry	GLS
Reference Document on Best Available Techniques in the Pulp and Paper Industry	PP
Reference Document on Best Available Techniques on the Production of Iron and Steel	I&S
Reference Document on Best Available Techniques in the Cement and Lime Manufacturing Industries	CL
Reference Document on the Application of Best Available Techniques to Industrial Cooling Systems	CV
Reference Document on Best Available Techniques in the Chlor – Alkali Manufacturing Industry	CAK
Reference Document on Best Available Techniques in the Ferrous Metals Processing Industry	FMP
Reference Document on Best Available Techniques in the Non Ferrous Metals Industries	NFM
Reference Document on Best Available Techniques for the Textiles Industry	TXT
Reference Document on Best Available Techniques for Mineral Oil and Gas Refineries	REF
Reference Document on Best Available Techniques in the Large Volume Organic Chemical Industry	LVOC
Reference Document on Best Available Techniques in the Waste Water and Waste Gas Treatment/Management Systems in the Chemical Sector	CWW
Reference Document on Best Available Techniques in the Food, Drink and Milk Industry	FM
Reference Document on Best Available Techniques in the Smitheries and Foundries Industry	SF
Reference Document on Best Available Techniques on Emissions from Storage	ESB
Reference Document on Best Available Techniques on Economics and Cross-Media Effects	ECM
Reference Document on Best Available Techniques for Large Combustion Plants	LCP
Reference Document on Best Available Techniques in the Slaughterhouses and Animals By-products Industries	SA
Reference Document on Best Available Techniques for Management of Tailings and Waste-Rock in Mining Activities	MTWR
Reference Document on Best Available Techniques for the Surface Treatment of Metals	STM
Reference Document on Best Available Techniques for the Waste Treatments Industries	WT
Reference Document on Best Available Techniques for the Manufacture of Large Volume Inorganic Chemicals (Ammonia, Acids and Fertilisers)	LVIC-AAF
Reference Document on Best Available Techniques for Waste Incineration	WI
Reference Document on Best Available Techniques for Manufacture of Polymers	POL
Reference Document on Energy Efficiency Techniques	ENE
Reference Document on Best Available Techniques for the Manufacture of Organic Fine Chemicals	OFC
Reference Document on Best Available Techniques for the Manufacture of Specialty Inorganic Chemicals	SIC
Reference Document on Best Available Techniques for Surface Treatment Using Solvents	STS
Reference Document on Best Available Techniques for the Manufacture of Large Volume Inorganic Chemicals (Solids and Others)	LVIC-S
Reference Document on Best Available Techniques in Ceramic Manufacturing Industry	CER

ZUSAMMENFASSUNG

Anwendungsbereich

Das vorliegende Dokument befasst sich mit Tätigkeiten im Zusammenhang mit der Bewirtschaftung von Aufbereitungsrückständen und taubem Gestein, die bei der Gewinnung von mineralischen Rohstoffen anfallen und erhebliche Umweltauswirkungen haben können. Insbesondere geht es dabei um Tätigkeiten, die als Beispiele für „gute Praxis" betrachtet werden können. Auf Abbautechniken und auf die Aufbereitung wird nur insoweit eingegangen, als sie für die Bewirtschaftung von Aufbereitungsrückständen und taubem Gestein von Bedeutung sind. Erreicht werden sollen eine stärkere Propagierung entsprechender Praktiken und die Förderung ihrer Anwendung bei allen Tätigkeiten in diesem Sektor.

Der Ausgangspunkt für die Erstellung des vorliegenden Merkblattes ist die Mitteilung der Europäischen Kommission KOM(2000) 664 über „Sicherheit im Bergbau". In der als Reaktion auf die Dammbrüche in Aznalcóllar and Baia Mare vorgelegten Mitteilung werden die Umsetzung eines Maßnahmeplanes auf der Grundlage eines Informationsaustauschs zwischen den Mitgliedstaaten der Europäischen Union und der Bergbauindustrie und die Erarbeitung eines BVT-Merkblatts (BVT: Beste Verfügbare Techniken).vorgeschlagen, Das vorliegende Dokument ist das Resultat dieses Informationsaustauschs. Es wurde in Form einer Initiative der Kommission und in Erwartung der vorgeschlagenen Richtlinie über die Bewirtschaftung von Abfällen aus der mineralgewinnenden Industrie[1] erstellt.

Die genannten Unglücke haben die Aufmerksamkeit der Öffentlichkeit auf die Bewirtschaftung von Sedimentationsbecken und deren Dämme gelenkt. Man sollte dabei nicht vergessen, dass das Abrutschen von Bergehalden und von zu Halden aufgeschüttetem taubem Gestein auch gravierende Umweltschäden verursachen kann. Beide Arten von Aufschüttungen können gewaltige Ausmaße annehmen. Dämme können Höhen von mehreren Dutzend Metern erreichen, Halden sogar über 100 m hoch und mehrere Kilometer lang sein und Hunderte Millionen Kubikmeter Aufbereitungsrückstände oder taubes Gestein enthalten. Nach Angaben im Eurostat-Jahrbuch 2003[2] fallen in der EU der 15 pro Jahr schätzungsweise mehr als 300 Millionen Tonnen Abfälle aus dem Bergbau und aus Steinbrüchen an.

Die folgenden Metalle finden in diesem Dokument Berücksichtigung, sofern sie in der Europäischen Union (EU-15), den Beitrittsländern, den Bewerberländern und der Türkei gewonnen und/oder aufbereitet werden:

- Aluminium
- Cadmium
- Chrom
- Kupfer
- Gold
- Eisen
- Blei
- Mangan
- Quecksilber
- Nickel
- Silber
- Zinn
- Wolfram
- Zink.

[1]) KOM(2003) 319 endgültig vom 2.6.2003. Artikel 4, Absatz 2, und Artikel 19, Absätze 2 und 3, des Richtlinienvorschlags beinhalten Verweise auf beste verfügbare Techniken.

[2]) Eurostat-Jahrbuch 2003, Der statistische Wegweiser durch Europa, 8. Ausgabe, herausgegeben und verfasst von Eurostat, dem Statistischen Amt der Europäischen Gemeinschaften in Luxemburg.

Alle diese Metalle werden im vorliegenden Dokument unabhängig von den produzierten Mengen oder der Aufbereitungsmethode (z. B. mechanische Methoden, Flotation, chemische oder hydrometallurgische Prozesse wie das Laugen) abgehandelt.

Kohle und ausgewählte Industrieminerale werden in diesem Dokument ebenfalls erfasst, d. h.
- Schwerspat
- Borat
- (mittels Flotation aufbereiteter) Feldspat
- Flussspat
- (mittels Flotation aufbereitetes) Kaolin
- (aufbereiteter) Kalkstein
- Phosphat
- Kalisalz
- Strontium
- (mittels Flotation aufbereiteter) Talk.

Kohle wird nur einbezogen, sofern sie aufbereitet ist und hierbei Aufbereitungsrückstände anfallen (und damit unter das vorstehend genannte Thema fällt). Ganz allgemein bedeutet dies, dass Steinkohle berücksichtigt wird, während die in der Regel nicht aufbereitete Braunkohle unberücksichtigt bleibt.

Ölschiefer wird in Estland aufbereitet; dabei fallen große Mengen an Aufbereitungsrückständen an, die bewirtschaftet werden müssen. Aus diesem Grund wurde entschieden, auch Ölschiefer in dieses Dokument aufzunehmen. Da jedoch keine sachdienlichen Informationen zu diesem Thema bereitgestellt wurden, werden Ölschiefer betreffende Fragen ausgeklammert.

Ferner finden folgende Aspekte in diesem Dokument keine Berücksichtigung:
- stillgelegte Abbaustandorte, wobei einige erst kürzlich geschlossene Abbaustandorte dennoch einbezogen werden,
- Abbau, Aufbereitung und Bewirtschaftung von Aufbereitungsrückständen, die bei der Gewinnung von Gas und Flüssigkeiten (z. B. Erdöl und Salzsole) anfallen.

Was die im Abschnitt „Anwendungsbereich“ aufgezählten Minerale betrifft, wird in dem Dokument
- die Bewirtschaftung von taubem Gestein untersucht,
- die Aufbereitung erörtert, sofern sie für die Bewirtschaftung von Aufbereitungsrückständen von Belang ist (wenn diese beispielsweise die Eigenschaften und das Verhalten der Aufbereitungsrückstände beeinflusst),
- die Bewirtschaftung von Aufbereitungsrückständen, z. B. in Becken/hinter Dämmen oder auf Halden, und deren Verwendung als Versatz in den Mittelpunkt gestellt sowie
- auch auf den Oberboden und das Deckgebirge eingegangen, sofern diese bei der Bewirtschaftung von Aufbereitungsrückständen eine Rolle spielen.

Der Bergbau

Der Zweck der bergbaulichen Tätigkeit besteht darin, den Bedarf an mineralischen Rohstoffen zu decken, die verwendet werden, um die Infrastruktur usw. zu entwickeln und die Lebensqualität der Bevölkerung zu verbessern, denn die abgebauten Stoffe werden bei der Herstellung vieler Waren und Materialien vielfach als Rohstoffe eingesetzt. Dazu gehören beispielsweise Erzminerale und Metalle, Kohle und Industrieminerale, die im Chemiesektor oder für Bauzwecke verwendet werden, usw.

Die Produkte der Bergbauindustrie werden gelegentlich direkt weiterverwendet, häufig aber noch veredelt, z. B. in Hüttenwerken.

Übliche Stufen bei allen bergbaulichen Tätigkeiten sind der Abbau, die anschließende Aufbereitung und der Abtransport der Produkte sowie die Bewirtschaftung der Abfälle.

Bei den meisten Erzmineralen werden in Europa – gemessen an der weltweiten Gesamtproduktion – nur geringe Mengen produziert (z. B. Gold: 1 %, Kupfer: 7 %); dasselbe gilt für den Kohlebergbau mit einem Anteil von 6 % an der Weltproduktion. Im Gegensatz zu den meist rückläufigen Zahlen für die Gewinnung von Metallen und Kohle ist bei der Produktion vieler Industrieminerale im europäischen Maßstab ein ständiger Anstieg zu verzeichnen. Für die Mehrzahl der Industrieminerale gilt, dass ein wesentlicher Teil der weltweiten Produktion auf Europa entfällt (z. B. Feldspat: 64 %, Kalisalze: 20 %). Einige Bereiche der europäischen Bergbauindustrie, wie der Metall- und Kohlebergbau, operieren unter sehr schwierigen wirtschaftlichen Bedingungen, hauptsächlich deswegen, weil die Lagerstätten international nicht mehr wettbewerbsfähig sind. Der Metallsektor in der EU hat auch mit dem Problem zu kämpfen, neue rentable Erzvorkommen in bekannten geologischen Regionen finden zu müssen. Während die abgebauten Mengen in den genannten Bereichen immer weiter sinken, steigt hingegen der Verbrauch stetig an. Um die steigende Nachfrage in Europa decken zu können, wird zunehmend auf Importe zurückgegriffen.

Die Größe der in diesem Sektor tätigen Unternehmen weist eine erhebliche Bandbreite auf; sie reicht von Firmen mit einer Hand voll Mitarbeitern bis zu Unternehmen mit mehreren tausend Beschäftigten. Was die Besitzverhältnisse anbelangt, sind weltweit agierende Unternehmen ebenso vertreten wie Industrieholdings sowie staatliche und private Einzelunternehmen.

Bewirtschaftung von Aufbereitungsrückständen und taubem Gestein

Die Bewirtschaftung der bei bergbaulichen Tätigkeiten anfallenden Abfälle sowie der Aufbereitungsrückstände und des tauben Gesteins, denen in diesem Dokument besondere Bedeutung beigemessen wird, stellt für den Betreiber in der Regel eine unerwünschte finanzielle Belastung dar. Eigentlich sollen im Abbaubetrieb und in der Aufbereitungsanlage möglichst große Mengen verkaufsfähiger Produkte gewonnen werden, so dass die Abfallbewirtschaftung und das Umweltmanagement in seiner Gesamtheit ausgehend von den angewandten Prozessstufen gestaltet werden.

Es gibt zahlreiche Möglichkeiten für die Bewirtschaftung von Aufbereitungsrückständen und taubem Gestein. Zu den gängigsten Methoden gehören

- das Einleiten von Aufbereitungsrückständen in Form von Schlämmen in Sedimentationsbecken,
- das Versetzen von Bergwerken und das Verfüllen von Tagebauen mit Aufbereitungsrückständen oder taubem Gestein bzw. eine Verwendung für den Bau von Dämmen aus Aufbereitungsrückständen,
- das Verkippen von mehr oder minder trockenen Aufbereitungsrückständen und Taubgestein auf Halden oder Hängen,
- die Verwendung der Aufbereitungsrückstände oder des tauben Gesteins als Produkt für die Boden- und Baugrundverbesserung, z.B. als Aggregate oder für die Wiedernutzbarmachung,
- die Trockenlagerung von eingedickten Aufbereitungsrückständen sowie
- das Verklappen von Aufbereitungsrückständen in Oberflächengewässern (z. B. Meere, Seen, Flüsse) oder ihr Einleiten in das Grundwasser.

Die Einrichtungen zur Bewirtschaftung von Aufbereitungsrückständen und taubem Gestein können sehr unterschiedliche Ausmaße haben; dabei kann es sich z. B. um Absetzbecken in der Größe eines Swimmingpools ebenso wie um über 1000 ha große Sedimentationsbecken, aber auch um kleine Halden mit Aufbereitungsrückständen bzw. taubem Gestein oder mehrere hundert Hektar große Flächen, auf denen Taubgestein gelagert wird, bzw. um über 200 m hohe Bergehalden handeln.

Für welche Methode der Bewirtschaftung von Aufbereitungsrückständen und/oder taubem Gestein man sich entscheidet, hängt hauptsächlich von der Bewertung folgender drei Faktoren ab:

- Kosten
- Umweltbilanz
- Gefahr einer Betriebsstörung/Havarie.

Zentrale Umweltfragen

Die wichtigsten Auswirkungen, die Einrichtungen zur Bewirtschaftung von Aufbereitungsrückständen und taubem Gestein auf die Umwelt haben, hängen mit der Lage des Standortes und dem relativen Flächenverbrauch zusammen sowie mit den potenziellen Emissionen von Staub und Abwässern während des Betriebs und in der Nachsorgephase. Zudem können Rutschungen oder Verbrüche in Einrichtungen zur Bewirtschaftung von Aufbereitungsrückständen und/oder taubem Gestein schwere Umweltschäden verursachen und sogar Menschenleben kosten.

Voraussetzungen für eine erfolgreiche Bewirtschaftung von Aufbereitungsrückständen und taubem Gestein sind eine ordnungsgemäße Materialbeschreibung, einschließlich genauer Voraussagen zu ihrem Langzeitverhalten, und eine gute Standortwahl.

Emissionen:
Abwässer und Stäube, die kontrolliert oder unkontrolliert aus Einrichtungen zur Bewirtschaftung von Aufbereitungsrückständen und taubem Gestein freigesetzt werden, können für Menschen, Tiere und Pflanzen unterschiedlich giftig sein. Die Abwässer können sauer oder basisch sein und gelöste Metalle und/oder lösliche und aus der Aufbereitung mitgeführte unlösliche komplexe organische Bestandteile sowie gegebenenfalls auch natürlich vorkommende organische Stoffe, wie Humin- und langkettige Karbonsäuren aus den bergbaulichen Tätigkeiten, enthalten. Die in den Emissionen enthaltenen Stoffe sowie deren pH-Wert, Gehalt an gelöstem Sauerstoff, Temperatur und Härte können wichtige Faktoren sein, die ihre Umwelttoxizität beeinflussen.

In den zurückliegenden zwanzig Jahren hat sich allgemein die Erkenntnis durchgesetzt, dass im Bergbau ein unter der Bezeichnung „Sauerwasserbildung" (*acid rock drainage* bzw. ARD) bekanntes Umweltphänomen besteht. Sauerwässer entstehen beim Abbau von Pb, Zn, Cu, Au und anderen Mineralen einschließlich Kohle in Sulfiderzlagerstätten. Obwohl sich saure Wässer in sulfidhaltigen Tagebau- und Untertageaufschlüssen bilden können, wird in diesem Dokument nur die Bewirtschaftung von Aufbereitungsrückständen und taubem Gestein erörtert.

Folgende Schlüsselfaktoren liegen diesen Umweltproblemen zugrunde:

- Oftmals enthalten Aufbereitungsrückstände und/oder taubes Gestein Metallsulfide.
- Sulfide oxidieren, wenn sie Sauerstoff und Wasser ausgesetzt werden.
- Bei der Sulfidoxidation entsteht saures schwermetallhaltiges Sickerwasser.
- Die Sickerwasserbildung erstreckt sich über lange Zeiträume.
- Es besteht ein Mangel an säurepuffernden Mineralen.

Unfallartige Brüche und Einstürze:
Versagen jeder Art von Einrichtungen zur Bewirtschaftung von Aufbereitungsrückständen oder taubem Gestein können sowohl kurz- als auch langfristige Folgen haben. Zu den üblichen kurzfristigen Folgen gehören unter anderem:

- Überflutung,
- Überdeckung/Abdämmung,
- Bruch und Zerstörung,
- Unterbrechung des Zugangs zur Infrastruktur,
- Vergiftung.

Als potenzielle Langzeitfolgen können unter anderem genannt werden:
- Metallakkumulation in Pflanzen und Tieren,
- Bodenverunreinigung,
- Verlust von Menschenleben und/oder Tieren.

Wiederherstellung des Geländes und Nachsorge:
Nach Einstellung des Betriebs muss das Gelände für die spätere Nutzung vorbereitet werden. Normalerweise gehört zur Erteilung einer Betriebsgenehmigung seit einigen Jahrzehnten bereits in der Planungsphase die Vorlage von Plänen für die Stilllegung und die Reinigung des Geländes; diese Pläne müssten also bei jeder Betriebsänderung und bei Verhandlungen mit den Genehmigungsbehörden und anderen Beteiligten regelmäßig aktualisiert worden sein. In einigen Fällen wird es darum gehen, so wenig Spuren wie möglich zu hinterlassen, in anderen könnte eine völlig andere Gestaltung der Landschaft angestrebt werden. Die „Planungsstrategie für die Stilllegung" sieht auch vor, dass die Schließung des Standortes bereits in der Durchführbarkeitsstudie für einen neuen Abbaubetrieb berücksichtigt und anschließend während des Betriebs ständig überwacht und aktualisiert wird. In jedem Fall müssen die Umweltbeeinträchtigungen auf ein Mindestmaß beschränkt bleiben.

Übliche Verfahren und Techniken

Abbautechniken:
Die als Abbau bezeichnete bezeichnete Gewinnung von Bodenschätzen, die anschließende Aufbereitung und die Bewirtschaftung von Aufbereitungsrückständen und taubem Gestein werden zumeist als ein Arbeitsgang betrachtet. Welche Techniken bei der Erzgewinnung, der anschließenden Aufbereitung und beim Umgang mit Aufbereitungsrückständen und taubem Gestein zum Einsatz kommen, hängt von der Abbautechnik ab. Daher sind Kenntnisse über die wichtigsten Abbaumethoden unerlässlich.

Feststoffe werden mithilfe der folgenden vier grundlegenden Abbautechniken gewonnen:
(1) Tagebau
(2) Untertagebau
(3) Steinbruch und
(4) Lösungsbergbau, Untertagelaugung

Die Entscheidung für eine dieser vier Möglichkeiten hängt von vielen Faktoren ab; dazu gehören u. a.
- der Wert des gewünschten Minerals/der gewünschten Minerale,
- die Qualität des mineralischen Rohstoffes,
- Größe, Form und Teufe des Lagerstättenkörpers,
- die Umweltbedingungen im umliegenden Gebiet,
- die geologischen, hydrogeologischen und geomechanischen Verhältnisse im Gebirgskörper,
- die seismischen Bedingungen in dem Gebiet,
- die Lage der Lagerstätte,
- die Löslichkeit des Lagerstättenkörpers,
- die Umweltauswirkungen des Betriebs,
- übertägige Sachzwänge sowie
- die Verfügbarkeit von Flächen.

Mineralogie:
Grundsätzlich kann zwischen den wichtigen Mineralarten wie Oxiden, Sulfiden, Silikaten und Karbonaten unterschieden werden, die durch Verwitterung und andere Einflüsse grundlegenden chemischen Veränderungen unterliegen können (z. B. Umwandlung von Sulfiden durch Verwitterung in Oxide). Die Mineralogie ist durch die Natur vorgegeben und bestimmt vielfach

das nachfolgende Aufbereitungsverfahren der gewünschten Minerale sowie die nachfolgende Bewirtschaftung der Aufbereitungsrückstände und des tauben Gesteins.

Gute mineralogische Kenntnisse sind eine wichtige Voraussetzung für
- eine ökologisch unbedenkliche Bewirtschaftung (z. B. die getrennte Behandlung von säurebildenden und nicht säurebildenden Aufbereitungsrückständen oder Taubgestein),
- einen verringerten Bedarf an Nachbehandlungen (wie der Behandlung von saurem Sickerwasser aus einer Einrichtung zur Bewirtschaftung von Aufbereitungsrückstände mit Kalk) sowie
- mehr Möglichkeiten zur Verwendung von Aufbereitungsrückständen und/oder taubem Gestein als Zuschlagstoffe.

Aufbereitungstechniken:
Mit Hilfe der Aufbereitung soll der in einem Abbaubetrieb gewonnene Bodenschatz in ein absatzfähiges Produkt umgewandelt werden. Die Aufbereitung erfolgt gewöhnlich auf dem Gelände des Abbaubetriebs, wobei es sich bei der Einrichtung um eine Anlage zur Aufbereitung von mineralischen Rohstoffen (durch Sieb- und Brechanlagen oder eine komplexe Aufbereitung) handelt. Der wesentliche Zweck der Aufbereitung besteht darin, die Masse zu verringern und möglichst reine bergbauliche Rohstoffe, reines Wertmaterial (reines Erz) zu den nachgelagerten Verfahrensanlagen (z. B. Verhüttung) zu befördern. Dazu werden Techniken und Methoden eingesetzt, mit denen das/die (gewünschte(n)) Wertmineral(e) vom tauben Gestein getrennt werden. Das dabei gewonnene absatzfähige Produkt wird als Konzentrat, das zurückbleibende Material als Aufbereitungsrückstand bezeichnet.

Die Aufbereitung umfasst verschiedene Verfahren, die auf den physikalischen Eigenschaften des Minerals (z. B. Partikelgröße, Dichte, magnetische Eigenschaften, Farbe) oder den physikalisch-chemischen Eigenschaften (Oberflächenspannung, Hydrophobie, Benetzbarkeit) aufbauen.

Bei der Aufbereitung kommen üblicherweise folgende Techniken zum Einsatz:
- Zerkleinern,
- Klassieren und Hydrozyklonierung,
- Schwerkraftkonzentration,
- Flotation,
- Sortieren,
- Trennen mittels Magnetscheider,
- Trennen mithilfe von elektrostatischen Walzenscheidern,
- Laugung,
- Eindicken,
- Filtern.

Einige dieser Techniken erfordern die Verwendung von Reagenzien. Im Falle der Flotation kann die gewünschte Abscheidung nur in Anwesenheit von Schäumern, Sammlern und Reglern erreicht werden.

Die bei der Aufbereitung angewandten Techniken beeinflussen die Eigenschaften der Aufbereitungsrückstände.

Umgang mit Aufbereitungsrückständen und taubem Gestein:
Zu den wichtigsten Eigenschaften von Stoffen in Einrichtungen zur Bewirtschaftung von Aufbereitungsrückständen und taubem Gestein gehören
- Scherfestigkeit,
- Korngrößenverteilung,
- Dichte,
- Plastizität,
- Feuchtegehalt,

- Permeabilität und
- Porosität.

Sedimentationsbecken sind Oberflächenanlagen zur Bewirtschaftung von Aufbereitungsrückständen in Form von Schlämmen. Diese Art Einrichtung zur Bewirtschaftung von Aufbereitungsrückständen kommt in der Regel bei Material aus der Nassaufbereitung zum Einsatz. Für jeden Bergedamm müssen mehrere Aspekte betrachtet werden, darunter:

- Dämme zur Umschließung von Aufbereitungsrückständen,
- Ableitungssysteme für den natürlichen Ablauf um den Damm herum oder durch diesen hindurch,
- Transport von Aufbereitungsrückständen aus der Aufbereitungsanlage zum Sedimentationsbecken,
- Ablagerung der Aufbereitungsrückstände im Sedimentationsbecken,
- Ableitung von überschüssigem freien Wasser,
- Schutz des umliegenden Gebiets vor schädlichen Umweltbeeinträchtigungen,
- Ausstattung mit Geräten und Überwachungssystemen zur Inspektion des Dammes,
- langfristige Aspekte (d. h. Stilllegung und Nachsorge).

Weitere Techniken für die Bewirtschaftung von Aufbereitungsrückständen und taubem Gestein sind der Versatz, die Aufhaldung, das Eindicken von Aufbereitungsrückständen, die Unterwasserbewirtschaftung und die Suche nach weiteren Verwendungsmöglichkeiten.

In der Regel stellen Abbaubetriebe und die dazugehörigen Aufbereitungsanlagen sowie die Einrichtungen zur Bewirtschaftung von Aufbereitungsrückständen und taubem Gestein schon nach wenigen Jahrzehnten ihren Betrieb wieder ein. Die durch die Abbautätigkeit entstandenen Hohlräume (die nicht Gegenstand der vorliegenden Arbeit sind), die Aufbereitungsrückstände und das Taubgestein können jedoch noch lange nach Einstellung des Betriebs zurückbleiben. Daher muss auf die ordnungsgemäße Stilllegung dieser Einrichtungen, die Wiedernutzbarmachung und die Nachsorge besonderes Augenmerk gelegt werden.

Zu den wichtigsten Aspekten bei der Bewirtschaftung von Aufbereitungsrückständen und taubem Gestein gehören neben der Standortwahl Überlegungen zu Unfällen bei Halden und Dämmen, die Eigenschaften und das Verhalten von Aufbereitungsrückständen sowie das Potenzial für die Bildung von Sauerwasser (ARD).

Angewandte Verfahren und Techniken, Emissions- und Verbrauchsgrenzwerte

In der nachfolgenden Aufzählung werden anhand von Beispielen einige der wichtigsten Fragen im Zusammenhang mit der Bewirtschaftung von Aufbereitungsrückständen erörtert.

- Der bei der Tonerdeverarbeitung anfallende und als „Rotschlamm“ bezeichnete Aufbereitungsrückstand hat einen erhöhten pH-Wert und wird entweder in konventionellen Sedimentationsbecken- bzw. -dammanlagen gelagert und so weit eingedickt, dass ein „Trockenstapeln“ möglich ist, oder im Meer verklappt.
- Aufbereitungsrückstände aus der Gewinnung und Aufbereitung von unedlen Metallen werden meist in Form von Schlämmen in große Sedimentationsbecken eingeleitet. Der Sulfidgehalt geringwertiger mineralischer Rohstoffe ist vielfach höher als der Gehalt an Puffermineralen, so dass die Aufbereitungsrückstände ein Potenzial für die Sauerwasserbildung aufweisen. In manchen Betrieben werden Aufbereitungsrückstände unter Wasser entsorgt, um von vornherein eine Sauerwasserbildung auszuschließen. Andere Abbaubetriebe nutzen Aufbereitungsrückstände teilweise zum untertägigen Verfüllen. Mitunter kommt bei der Schließung von Sedimentationsbecken auch das „Nassabdeckungsverfahren“ zum Einsatz, in anderen Fällen wird die Trockenabdeckung angewandt.

- Grobe Aufbereitungsrückstände aus der Eisenerzgewinnung und -aufbereitung werden auf Halden verbracht, in Form von Schlamm vorliegende Aufbereitungsrückstände in Sedimentationsbecken eingeleitet.
- Bei einigen der in Europa betriebenen Goldminen besteht die Möglichkeit der Sauerwasserbildung. Erfolgt die Goldgewinnung mittels Cyanidlaugung, so wird das Cyanid vor dem Einleiten in den Absetzbecken vernichtet.
- Verschiedene Betriebe, die Industrieminerale gewinnen, produzieren überhaupt keine Aufbereitungsrückstände bzw. verkaufen sie als Zuschlagstoffe.
- In Boratabbaustätten werden grobe Aufbereitungsrückstände zunächst auf Halden gelagert, ehe sie verfüllt werden.
- Ein in diesem Dokument aufgeführter Fluoritproduzent verklappt seine Aufbereitungsrückstände im Meer.
- Der Betreiber einer in diesem Dokument aufgeführten Kaolingrube entzieht den feinen Aufbereitungsrückständen vor dem Transport zu den Halden das Wasser; ebenso verfahren einige Kalkstein- und Kalziumkarbonatproduzenten.
- Ein in diesem Dokument genannter Kalksteinproduzent entsorgt in Schlammform vorliegende Aufbereitungsrückstände in einem ehemaligen Steinbruch.
- Kaliabbaubetriebe verbringen feste Aufbereitungsrückstände auf Halden oder nutzen sie zum Verfüllen. Flüssige Aufbereitungsrückstände werden teilweise in Tiefbrunnen gepumpt und teilweise in Oberflächengewässer eingeleitet. In einem in diesem Dokument beschriebenen Fall werden Aufbereitungsrückstände im Meer entsorgt.
- Im Kohlebergbau werden grobe Aufbereitungsrückstände üblicherweise auf Halden oder in ehemaligen Tagebauen entsorgt. Die zu Schlamm aufbereiteten Feinfraktionen werden entweder in Becken eingeleitet oder gefiltert, In einigen Fällen werden sowohl die gefilterten als auch die groben Aufbereitungsrückstände verkauft. In anderen Fällen werden sie auf Halden verbracht. Verfüllen ist oft nicht rentabel.
- Zu den Unfallverhütungsmaßnahmen gehören u. a. die regelmäßige Überwachung, Betriebs-, Überwachungs- und Wartungshandbücher; unabhängige Prüfungen, Wasserbilanzen, Maßnahmen gegen Bodensenkungen, die Überprüfung der Planungen durch externe Fachleute, der Einsatz von Piezometern und Inklinometern sowie die seismische Überwachung.

In der nachfolgenden Aufzählung werden anhand von Beispielen einige der wichtigsten Aspekte im Zusammenhang mit der Bewirtschaftung von taubem Gestein erörtert.

- Im Untertagebau verbleibt das Taubgestein in der Regel auch unter Tage.
- Ebenso wie Aufbereitungsrückstände weist taubes Gestein aus der Gewinnung und Aufbereitung von unedlen Metallen ein Potenzial für die Sauerwasserbildung auf. Manche Betriebe bewirtschaften Taubgestein, bei dem Sauerwasserbildung auftritt, getrennt von Taubgestein, bei dem keine Sauerwasserbildung erfolgt. Taubgestein ohne Sauerwasserbildung wird entweder als Zuschlagstoff oder beim Bau von Dämmen oder Straßen auf dem Betriebsgelände verwendet bzw. aufgehaldet. Bei einer Stilllegung werden Halden mit taubem Gestein, bei dem die Möglichkeit einer Sauerwasserbildung besteht, mit technisch hergestelltem trockenem Deckmaterial abgedeckt, um die Bildung von Sauerwässern zu verhindern.
- Taubes Gestein aus der Eisenerzgewinnung wird zusammen mit groben Aufbereitungsrückständen auf Halden verbracht.
- Taubes Gestein aus Goldminen wird auf Halden gelagert oder beim Dammbau bzw. zum Verfüllen von Tagebaugruben verwendet.
- In einigen Betrieben, die Industrieminerale abbauen, wird das taube Gestein zum Verfüllen eingesetzt oder als Zuschlagstoff verkauft.
- In vielen Kohleabbaubetrieben wird das taube Gestein zusammen mit den gefilterten feinen Aufbereitungsrückständen auf Halden verbracht. Die endgültige Gestaltung der Halde erfolgt in Abstimmung mit den zuständigen Behörden und den Gemeinden, wobei in die Landschaft integrierte Strukturen angestrebt werden.

Emissions- und Verbrauchsmengen

Der überwiegende Teil des Prozesswassers wird von der Einrichtung zur Bewirtschaftung von Aufbereitungsrückständen in die Aufbereitungsanlage zurückgeführt, wobei der Anstieg des Reagenziengehalts nicht außer Acht gelassen werden darf.

Aufgrund der großen Unterschiede in Bezug auf die Mineralogie, bei den Abbau- und Aufbereitungsmethoden sowie hinsichtlich der Standortbedingungen sind weiter gehende Ausführungen zu den Emissions- und Verbrauchsmengen nicht möglich. Dennoch stellten viele Betreiber diesbezügliche Angaben zur Verfügung, die in Kapitel 3 aufgenommen wurden. In der Regel umfassen diese Informationen Angaben zum Wasserverbrauch und zur Menge des der Wiederverwendung zugeführten Prozesswassers, zur Wasserbilanz, zum Reagenzienverbrauch, zu den Staubemissionen und zu den Emissionen in Gewässer.

Kosten

Kapitel 3 beinhaltet einige Beispielberechnungen zu den Kosten der Bewirtschaftung von Aufbereitungsrückständen und taubem Gestein bei laufendem Betrieb und nach der Stilllegung.

Techniken, die bei der Festlegung von BVT in Betracht zu ziehen sind

In Kapitel 4 wird eingehend über die Festlegung von BVT für die Bewirtschaftung von Aufbereitungsrückständen und taubem Gestein aus bergbaulichen Tätigkeiten informiert.

Dabei sollte der Umfang der aufgenommenen Informationen eine Bewertung der Eignung der Techniken sowohl im Allgemeinen wie auch im konkreten Einzelfall ermöglichen. Den Informationen in diesem Kapitel kommt entscheidende Bedeutung für die Festlegung von BVT zu.

Zu den als BVT eingestuften Techniken finden sich auch Querverweise in Kapitel 5. Die Aufmerksamkeit der Nutzer des Dokuments wird so auf die Diskussion über die relevanten Techniken in Verbindung mit den BVT-Schlussfolgerungen gelenkt; dies kann ihnen bei der Festlegung der Genehmigungsbedingungen anhand von BVT helfen.

Einige der in Kapitel 4 aufgeführten Techniken sind technische Prozesse, andere wiederum gute Betriebspraktiken einschließlich Managementtechniken. Diese Techniken werden in folgender Reihenfolge zu Gruppen zusammengefasst:

- Allgemeine Grundsätze: Grundsätze der guten Bewirtschaftung, Bewirtschaftungsstrategien und Risikobewertung, die sämtlich einen allgemeinen Hintergrund für die erfolgreiche Bewirtschaftung von Aufbereitungsrückständen und taubem Gestein bieten sollen.
- Lebenszyklus-Betrachtung: Die Gefahr des Versagens/des Ausfalls von Anlagen kann zusätzlich durch eine Verpflichtungszusage des Betreibers gemindert werden, während der gesamten Betriebsdauer angemessene, verfügbare technische Verfahren sachgerecht und konsequent bei der Planung, beim Betrieb und bei der Stilllegung von Einrichtungen zur Bewirtschaftung von Aufbereitungsrückständen und taubem Gestein anzuwenden. Zu den für eine gute Technik wesentlichen Instrumenten gehören die Festlegung einer ökologischen Vergleichsbasis, die Beschreibung der Aufbereitungsrückstände und des tauben Gesteins, die Nutzung von Handbüchern über Dammsicherheit und die Durchführung von Dammprüfungen sowie konkrete Stilllegungspläne von Beginn an.
- Emissionsvermeidung und -kontrolle:
  - Bewältigung des Problems der Sauerwasserbildung: Es gibt zahlreiche Möglichkeiten zur Vermeidung, Kontrolle und Behandlung des Sauerwassers (z. B. Abdeckung, Zusatz von Puffermineralen, aktive/passive Behandlung), die speziell für sauerwasserbildende Aufbereitungsrückstände und taubes Gestein entwickelt wurden und die gleichermaßen für die Betriebs- und Stilllegungsphase eines Bergbaubetriebs geeignet sind.

   - Techniken zur Senkung des Reagenzienverbrauchs: Es stehen mehrere Methoden zur Senkung des Einsatzes von Reagenzien zur Verfügung, d. h. die computergestützte Überwachung der Qualität des eingesetzten Stoffes, Betriebsstrategien zur Minimierung des Cyanidzusatzes und Vorsortieren der Einsatzstoffe für die Aufbereitungsanlage.
   - Vermeidung von Wassererosion: Die Wassererosion in Einrichtungen zur Bewirtschaftung von Aufbereitungsrückständen und taubem Gestein kann durch Abdecken der Hänge oder das Stimulieren der Partikelbindung verhindert werden.
   - Vermeidung von Staubemissionen: Staubemissionen treten hauptsächlich an den Spülstränden von Sedimentationsbecken, an der Außenböschung von Dämmen und Halden sowie beim Transport von Aufbereitungsrückständen und taubem Gestein auf. Ein Verfahren zur Verminderung der Staubbildung besteht darin, die Spülstrände von Absetzbecken und andere Hänge feucht zu halten.
   - Techniken zur Minderung von Lärmemissionen: Zu den häufigsten Quellen von Lärmemissionen gehören der Transport, das Abkippen und das Verteilen der Aufbereitungsrückstände mithilfe von Lkw und Förderbändern. Lärmbeeinträchtigungen in Wohngebieten durch Lkw-Verkehr auf der Halde können durch das Errichten von Lärmschutzwänden gemindert werden.
   - Schrittweise Wiedernutzbarmachung/Wiederbegrünung: Halden und Dämme werden vielfach noch während der Betriebsphase wieder nutzbar gemacht/wiederbegrünt. Neben anderen Vorteilen bringt dies auch eine Verkürzung der Stilllegungsphase mit sich.
   - Wasserbilanzen: Die Erstellung einer detaillierten Wasserbilanz spielt bei der Auslegung von Sedimentationsbecken und des Abbaustandortes sowie für die Zeit nach der bergbaulichen Nutzung eine wichtige Rolle. Anhand der Wasserbilanz können das Fassungsvermögen des Beckens und der erforderliche Freibord ermittelt werden (falls das Wasser aus dem Becken nicht direkt in den aufnehmenden Wasserlauf eingeleitet werden kann). Zum Zeitpunkt der Stilllegung erfolgt eine Evaluierung der Wasserbilanz, um die Stilllegungspläne umzusetzen.
   - Ableitung aus Becken: Bei undurchlässigen Becken kann ein Ablaufsystem erforderlich werden, das die Wiederverwendung des Prozesswassers und eine bedarfsgerechte Verringerung der Beckengröße ermöglicht.
   - Behandlung von freiem Wasser: Wird das freie Wasser aus dem Becken nicht direkt in den natürlichen Wasserlauf abgeleitet, müssen Vorkehrungen für das Speichern getroffen werden, d. h. das gesamte freie Wasser muss entweder in die Anlage zurückgeführt werden bzw. unter trockenheißen klimatischen Bedingungen verdunsten können.
   - Behandlung von Sickerwasser: Die Planung von Versickerungsanlagen setzt gründliche Kenntnisse über den hydrogeologischen Zustand des Standortes voraus. In einigen Fällen wird eine Versickerung verhindert. In anderen Fällen wird das Sickerwasser aufgefangen und, sofern es eine gute Qualität aufweist, dessen Einsickern in das Grundwasser ermöglicht.
   - Techniken zur Verringerung der Emissionen in Gewässer: Emissionen in Gewässer können durch Wiederverwendung des Prozesswassers verhindert werden. Diese Möglichkeit besteht nicht, wenn die Abwässer sauer oder basisch sind bzw. suspendierte Feststoffe, gelöste Verbindungen oder Metalle (z. B. Arsen) oder Prozesschemikalien (z. B. Cyanid) enthalten. Je nach der vorkommenden Verbindung können verschiedene Behandlungstechniken angewandt werden.
   - Grundwasserüberwachung: In der Regel erfolgt eine Überwachung des Grundwassers in Gebieten, in denen Aufbereitungsrückstände und taubes Gestein gelagert werden. Die Überwachung erstreckt sich auch auf den Grundwasserspiegel und die Wasserqualität.
- Unfallverhütung:
   - Bewirtschaftung von Aufbereitungsrückständen und taubem Gestein im Bergbaubetrieb: Um Dammbrüche und das Abrutschen von Halden zu vermeiden, sollten Einrichtungen zur Bewirtschaftung von Aufbereitungsrückständen und taubem Gestein am besten in einem geeigneten nahe gelegenen Bergwerk(sbetrieb) angelegt werden, um somit Probleme mit der Stabilität des Damms bzw. der Halde zu umgehen.

- Ableitung natürlicher Abflüsse: Ein Ableitungssystem ist von entscheidender Bedeutung für die Sicherheit von Bergedämmen. Jede Art von Schäden am Damm kann zum Eindringen von Wassermengen führen, für die dieser nicht ausgelegt ist; dadurch kann es zu einer Überspülung kommen, die den Damm insgesamt brechen lässt.
- Vorbereitung der natürlichen Erdoberfläche unter dem Damm: Die natürliche Erdoberfläche unter dem Bergedamm wird in der Regel vollständig von Pflanzenbewuchs und humosem Boden befreit, der abgetragen wird, , um eine geeignete „Gründung" für den Damm herzustellen.
- Beim Dammbau eingesetztes Material: Bei der Entscheidung, welche Materialien beim Dammbau eingesetzt werden sollen, steht an erster Stelle die Überlegung, dass diese Materialien für den vorgesehenen Zweck geeignet sein müssen und dass unter Betriebsbedingungen bzw. klimatischen Bedingungen keine Beeinträchtigung ihrer Festigkeit auftreten darf.
- Absetzen der Aufbereitungsrückstände: Das richtige Ablagern der Aufbereitungsrückstände, insbesondere im feuchten Zustand, ist stets entscheidend für die Stabilität des Bauwerks. Feuchte Aufbereitungsrückstände werden üblicherweise auf der Dammkrone abgekippt und anschließend möglichst gleichmäßig auf dem Damm verteilt, um einen „Spülstrand" für Aufbereitungsrückstände an der Innenseite des Damms anzulegen.
- Techniken, die beim Bau und der Erhöhung von Dämmen angewandt werden: Bergedämme wurden oft aus der Grobfraktion der Aufbereitungsrückstände gebaut, und dies kann noch immer eine sehr geeignete Form des Absetzens von Aufbereitungsrückständen in Schlammform sein. Während der Betriebsphase können sich jedoch sowohl die Eigenschaften der mineralischen Rohstoffe als auch die Aufbereitungsmethode und damit auch die Eigenschaften des Aufbereitungsrückstandes ändern. Das Qualitätsmanagement ist demzufolge während der gesamten Betriebsphase eine sehr wichtige Aufgabe. In der Regel wird daher beim Bau des Damms und oftmals auch beim Erhöhen Füllmaterial eingesetzt, dessen Qualität während des Dammbaus leichter überwacht werden kann. Die Sicherung der langfristigen Stabilität hängt jedoch nicht nur von der Art des beim Bau von Bergedämmen eingesetzten Materials ab, sondern auch davon, wie das geeignete Baumaterial verbaut und verdichtet wird. Im Wesentlichen werden folgende Grundtypen von Dämmen gebaut: konventionelle Dämme bzw. Dämme, die im Upstream-, Downstream- oder Centerline-Dammbauverfahren aufgeschüttet werden (d.h., die Dämme werden nach innen, nach außen bzw. mittig aufgeschüttet und „wandern" in das Becken „hinein" bzw. davon weg).
- Freies Wasser, Freibord, Notüberlauf und Bemessungshochwasser: Freies Wasser kann unter anderem über Überläufe, offene Kanäle, Scheidetürme und Scheidebrunnen abgeleitet werden. Neben geeigneten Freibord- und Notüberlaufsystemen sind dies wichtige Anlagen zur Verhinderung von Unfällen, wie zum Beispiel Dammüberflutungen.
- Drainage von Dämmen: Durchlässige Dämme funktionieren nach dem Prinzip, dass ein Austritt durch den Damm weit unter der Basis der Außenböschung erfolgen sollte. Zu diesem Zweck wird eine interne Drainage verlegt, deren Entwässerungszone im Innern des Damms liegt. Undurchlässige Dämme sind mit vergleichbaren Drainagesystemen ausgestattet, wobei es darum geht, das Versickern durch den Kern und damit die Erosion des Kerns und der Außenböschung des Damms zu verhindern.
- Überwachung des Versickerns: Das kontrollierte Versickern erfolgt durch den Damm und gewährleistet dessen Stabilität dadurch, dass der auf den Damm einwirkende Porendruck verringert wird. Eine wirksame Kontrolle und der reibungslose Ablauf des Versickerns sind sowohl für die Umweltbilanz als auch für die Unfallverhütung von entscheidender Bedeutung.
- Damm- und Haldenstabilität: Ein wichtiges Kriterium für die Stabilität von Halden und Dämmen ist der Sicherheitsfaktor, d. h. das Verhältnis zwischen Scherfestigkeit und Scherbeanspruchung.
- Verfahren zur Überwachung der Stabilität von Dämmen und Halden: Grundlage der gesamten Überwachungstätigkeit ist die Aufstellung eines Überwachungsplans. Die

Überwachung umfasst eine Liste von in bestimmten Abständen durchgeführten Messungen. Der allgemeine Überwachungsplan beinhaltet in der Regel auch Pläne für Inspektionen und Audits/Prüfun-gen. Die Stabilität von Dämmen und Halden hängt auch von der Stabilität der tragenden Schicht ab, d. h. des Untergrunds, auf dem Dämme und Halden errichtet werden.
  - Bewirtschaftung von Cyanid: Neben der Behandlung von aus der Cyanidlaugung stammenden Aufbereitungsrückständen schließt die Bewirtschaftung von Cyanid eine Vielzahl von Sicherheitsmaßnahmen zur Unfallverhütung ein. Die Anlagenplanung umfasst auch mehrere technische Lösungen zur Unfallverhütung.
  - Entwässerung von Aufbereitungsrückständen: Der größte Nachteil bei der Entsorgung von Aufbereitungsrückständen in Schlammform ist deren Beweglichkeit. Bei einem Versagen der Rückhaltestruktur (d. h. des Damms) ist eine Verflüssigung möglich, wobei aufgrund der physikalischen und chemischen Eigenschaften erhebliche Schäden eintreten können. Es gibt zwei Möglichkeiten, dieses Problem zu umgehen: mit trockenen Aufbereitungsrückständen und mit dem Eindicken von Aufbereitungsrückständen.
- Verringerung der Standfläche/des Platzbedarfs: Eine wirksame Möglichkeit zur Verringerung der von Einrichtungen zur Bewirtschaftung von Aufbereitungsrückständen und taubem Gestein beanspruchten Standfläche (d.h. ihres Platzbedarfs) besteht darin, dieses Material ganz oder teilweise zum Verfüllen einzusetzen. Weitere Möglichkeiten wären die Unterwasserentsorgung von Aufbereitungsrückständen, d. h. die Verklappung im Meer, oder das Ermitteln anderer Verwendungsmöglichkeiten für Aufbereitungsrückstände und Taubgestein.
- Abmilderung von Unfallfolgen: Die Folgen eines Unfalls können sowohl durch Notfallpläne als auch durch die Auswertung und Untersuchung von Zwischenfällen abgemildert werden.
- Instrumente des Umweltmanagements: Umweltmanagementsysteme erweisen sich als nützliches Instrument bei der Verhinderung von Verschmutzungen im Zusammenhang mit gewerblichen Tätigkeiten im weitesten Sinne.

BVT für die Bewirtschaftung von Aufbereitungsrückständen und taubem Gestein aus bergbaulichen Tätigkeiten

Im Kapitel über BVT (Kapitel 5) werden die als BVT betrachteten Techniken anhand der in Kapitel 4 enthaltenen Informationen sowie unter Berücksichtigung der Definition des Begriffs „beste verfügbare Techniken“ und der in Anhang IV der IVU-Richtlinie (siehe Vorwort) aufgelisteten Erwägungen festgelegt.

Das BVT-Kapitel ist in einen allgemeinen Teil, der für alle Standorte der Bewirtschaftung von Aufbereitungsrückständen und taubem Gestein gültig ist, und in einen spezifischen Teil für bestimmte Minerale untergliedert.

Entscheidungen zur Bewirtschaftung von Aufbereitungsrückständen und taubem Gestein beruhen auf der Umweltbilanz, der Risikobewertung und der Wirtschaftlichkeit, wobei die Risikobewertung ein standortspezifischer Faktor ist.

Der Vollständigkeit halber werden sämtliche Schlussfolgerungen zu den BVT hier wiedergegeben.

<u>Allgemeine BVT</u>

Mit den BVT
- sollen die in Abschnitt **4.1** aufgeführten allgemeinen Grundsätze angewandt und
- soll die in Abschnitt **4.2** beschriebene Herangehensweise einer Bewirtschaftung über den gesamten Lebenszyklus hinweg („*life-cycle management*“) umgesetzt werden.

Das Life-Cycle-Management erstreckt sich über alle Phasen der Existenz eines Standorts, einschließlich

- der Entwurfsphase (Abschnitt 4.2.1):
  - ökologische Vergleichsbasis (Abschnitt **4.2.1.1**),
  - Beschreibung der Aufbereitungsrückstände und des tauben Gesteins (Abschnitt **4.2.1.2**),
  - Studien und Pläne zu TMF (Abschnitt **4.2.1.3**), die folgende Aspekte abdecken:
    - Dokumentation zur Auswahl der Standorte,
    - Umweltverträglichkeitsprüfung
    - Risikoeinschätzung
    - Einsatzbereitschaftsplan für Katastrophenfälle
    - Ablagerungsplan
    - Wasserbilanz und Wasserhaushaltsplan sowie
    - Außerbetriebnahme- und Stilllegungsplan
  - Auslegung der TMF und der damit verbundenen Bauten (Abschnitt **4.2.1.4**),
  - Kontrolle und Überwachung (Abschnitt **4.2.1.5**),
- der Bauphase (Abschnitt **4.2.2**),
- der Betriebsphase (Abschnitt **4.2.3**), mit den Elementen:
  - OSM-Handbücher (Abschnitt **4.2.3.1**),
  - Revision (Abschnitt 4.2.3.2),
- der Stilllegungs- und der Nachsorgephase (Abschnitt 4.2.4) mit den Elementen:
  - langfristige Ziele der Stilllegung (Abschnitt **4.2.4.1**),
  - spezielle Fragen der Stilllegung (Abschnitt **4.2.4.2**) von
    - Halden (Aufschüttungen),
    - Becken, einschließlich:
      - wasserbedeckte Becken,
      - entwässerte Becken,
      - wasserwirtschaftliche Anlagen.

Des Weiteren soll mit den BVT

- der Reagenzienverbrauch verringert (Abschnitt **4.3.2**),
- die Wassererosion verhindert (Abschnitt **4.3.3**),
- die Staubbildung verhindert (Abschnitt **4.3.4**),
- eine Wasserbilanz aufgestellt (Abschnitt **4.3.7**) und die daraus gewonnen Ergebnisse für die Erarbeitung eines Wasserbilanzplanes genutzt (Abschnitt **4.2.1.3**),
- die Klarwasserbewirtschaftung angewandt (Abschnitt **4.3.9**) und
- das Grundwasser um alle Bereiche mit Aufbereitungsrückständen und taubem Gestein herum überwacht werden (Abschnitt **4.3.12**).

ARD-Management

Zur Charakterisierung der Aufbereitungsrückstände und des tauben Gesteins (Abschnitt **4.2.1.2** zusammen mit Anhang 4) gehört die Bestimmung des Säurebildungspotenzials von Aufbereitungsrückständen und/oder taubem Gestein. Wenn ein Säurebildungspotenzial vorhanden ist, sollen die BVT als erstes die Bildung von Sauerwässern (ARD) verhindern (Abschnitt **4.3.1.2**). Falls das nicht verhindert werden kann, sollen die BVT die Auswirkungen der ARD kontrollieren (Abschnitt **4.3.1.3**) oder die Anwendung von Behandlungsoptionen ermöglichen (Abschnitt **4.3.1.4**). Oft wird eine Kombination aus allen Möglichkeiten verwendet (Abschnitt **4.3.1.6**).

Alle Optionen zur Verhinderung der Entstehung, zur Kontrolle und zur Behandlung von Sauerwässern können in vorhandenen oder neuen Anlagen zur Anwendung kommen. Dennoch werden die besten Ergebnisse bei der Stilllegung dann erreicht, wenn die Pläne für die Stilllegung gleich zu Beginn (d.h. im Entwurfsstadium) für den Betrieb ausgearbeitet werden („Von-der-Wiege-bis-zur-Bahre“-Philosophie).

Die Anwendbarkeit der Optionen hängt hauptsächlich von den am Standort vorherrschenden Bedingungen ab. Faktoren, wie

- die Wasserbilanz,
- die Verfügbarkeit möglicher Abdeckmaterialien und
- der Grundwasserpegel

beeinflussen die am jeweiligen Standort anwendbaren Optionen. Abschnitt **4.3.1.5** liefert Entscheidungshilfen für die am besten geeigneten Stilllegungsoptionen.

Sickerwasser-Management (Abschnitt 4.3.10)
Der Standort für eine Anlage zur Bewirtschaftung von Aufbereitungsrückständen oder taubem Gestein wird vorzugsweise so ausgewählt, dass eine Abdichtung (Auskleidung mit einer Dichtungsbahn) nicht notwendig wird. Wenn das aber nicht möglich ist und sich die Qualität des Sickerwassers als nachteilig sowie dessen Strömungs- bzw. Fließgeschwindigkeit als (zu) hoch erweist, muss das (Einlaufen von) Sickerwasser verhindert, verringert (Abschnitt **4.3.10.1**) oder kontrolliert (Abschnitt **4.3.10.2**) werden (und zwar möglichst in dieser Reihenfolge). Oft findet eine Kombination aus diesen Maßnahmen Anwendung.

Emissionen in das Wasser
Mit den BVT soll(en)

- das Prozesswasser wieder verwendet (siehe Abschnitt **4.3.11.1**),
- das Prozesswasser mit anderen ausströmenden, gelöste Metalle enthaltenden Medien vermischt (siehe Abschnitt **4.3.11.3**),
- Absetzbecken zum Abfangen erodierter Feinstoffe gebaut (siehe Abschnitt **4.3.11.4.1**),
- schwebende Feststoffe und gelöste Metalle vor dem Austrag des gereinigten Abwassers in den Vorfluters entfernt (Abschnitt **4.3.11.4**),
- alkalische Abwässer mit Schwefelsäure oder Kohlendioxid neutralisiert (Abschnitt **4.3.11.6**) und
- Arsen aus den aus den Gruben ausfließenden Abwässern durch die Zugabe von Eisen(III)-Salz entfernt werden (Abschnitt **4.3.11.7**).

Die jeweiligen Abschnitte über Emissions- und Verbrauchswerte in Kapitel 3 enthalten Beispiele über die erreichten Werte, aber es sollte kein Zusammenhang zwischen den angewandten Techniken und den verfügbaren Emissionsdaten hergestellt werden. Deshalb war es in diesem Dokument nicht möglich, aus den mit den BVT auftretenden Emissionswerten Schlussfolgerungen über die BVT selbst zu ziehen.

Die folgenden Techniken sind BVT für die Behandlung säurehaltiger Abwässer (Abschnitt **4.3.11.5**):

- aktive Behandlungsverfahren:
  - Zugabe von Kalk (Calciumkarbonat), Löschkalk oder Ätzkalk,
  - Zugabe von Ätznatron zu ARD mit einem hohen Mangananteil
- passive Behandlungsverfahren:
  - künstliche Feuchtgebiete,
  - offene Kalksteinkanäle/anoxische Kalksteinkanalisation,
  - Ableitungsbrunnen.

Passive Behandlungssysteme stellen eine langfristige Lösung nach der Außerbetriebnahme eines Standortes dar, aber nur, wenn sie als Ergänzung zu anderen (präventiven) Maßnahmen eingesetzt werden.

Geräuschemissionen (Abschnitt 4.3.5)
Mit den BVT sollen:

- kontinuierlich arbeitende Systeme (z.B. Transportbänder, Pipelines) eingesetzt werden,
- eingehauste Bandantriebe in Gegenden eingesetzt werden, wo es örtlich zu Lärm kommt und
- zunächst die Außenböschungen einer Aufschüttung errichtet und dann die Rampen und Arbeitsstrossen soweit wie möglich in den Innenbereich der Halde verlegt werden.

Auslegung des Damms
Zusätzlich zu den in Abschnitt **4.1** und Abschnitt **4.2** beschriebenen Maßnahmen während der **Entwurfsphase** (Abschnitt 4.2.1) für den **Damm eines Sedimentationsbecken** entspricht es BVT:

- einen Zufluss mit einem statistischen Wiederkehrsintervall von 100 Jahren (HQ 100) als Bemessungshochwasserzufluss für die Dimensionierung der Hochwasserentlastungsanlage eines Sedimentationsbeckens mit schwachem Gefahrenpotential anzusetzen,
- einen Zufluss mit einem statistischen Wiederkehrintervall von 5.000 – 10.000 Jahren (HQ 5.000 – 1.000) als Bemessungshochwasserzufluss für die Dimensionierung der Hochwasserentlastungsanlage eines Sedimentationsbeckens mit hohem Gefahrenpotential anzusetzen.

Dammkonstruktion
Zusätzlich zu den in Abschnitt **4.1** und Abschnitt **4.2** beschriebenen Maßnahmen während der **Konstruktionsphase** (Abschnitt **4.2.2**) für den **Damm eines** Sedimentationsbecken entspricht es BVT wenn:

- die alle Vegetation und humushaltiger Erden aus dem natürlichen Untergrund unter dem Staudamm beseitigt wird (Abschnitt 4.4.3) und
- das Dammbaumaterial so ausgewählt wird, das sich für den vorgesehenen Zweck eignet und unter den vorherrschenden betrieblichen und klimatischen Bedingungen standfest ist (Abschnitt 4.4.4).

Dammschüttung
Zusätzlich zu den in Abschnitt **4.1** und Abschnitt **4.2** beschriebenen Maßnahmen während der **Konstruktions- und Betriebsphase** (Abschnitte **4.2.2** und **4.2.3**) für den **Damm eines Sedimentationsbeckens** entspricht es BVT, wenn

- die Risiken eines zu hohen Porendrucks eingeschätzt werden, der Porendruck vor und während jeder Schüttung überwacht wird und die Einschätzung durch einen unabhängigen Fachmann erfolgt;
- unter folgenden Bedingungen herkömmliche Arten von Dämmen verwendet werden (Abschnitt 4.4.6.1), wenn nämlich
  - die Aufbereitungsrückstände für die Dammkonstruktion nicht geeignet sind,
  - das zu speichernde Wasser angestaut werden muss,
  - sich der Standort für die Bewirtschaftung der Aufbereitungsrückstände an einem weit entfernten und schwer zugänglichen Ort befindet,
  - das Wasser mit den Aufbereitungsrückständen wegen des Abbaus eines toxischen Elements (z.B. Cyanid) über einen längeren Zeitraum zurückgehalten werden muss und
  - der natürlich Zufluss in den Staubereich sehr groß ist oder starken Schwankungen unterliegt, so dass das Wasser zu Kontrollzwecken gespeichert werden muss;
- unter folgenden Bedingungen auf das Upstream-Dammbauverfahren zurückgegriffen wird, d.h. die Dämme werden nach innen aufgeschüttet, so dass der Damm ins Sedimentationsbecken „hinein“ wandert (Abschnitt 4.4.6.2), wenn nämlich
  - das seismische Risiko gering ist und
  - Aufbereitungsrückstände für den Bau des Dammes verwendet werden, die zu mindestens 40–60 % aus Material mit einer Korngröße zwischen 0,075 und 4 mm in

ganzen Aufbereitungsrückständen bestehen (was aber nicht auf verdickte bzw. entwässerte Aufbereitungsrückstände zutrifft);
- unter folgenden Bedingungen auf das Downstream-Dammbauverfahren zurückzugreifen, die Dämme werden nach außen aufgeschüttet, so dass der Damm vom Becken „weg" wandert (Abschnitt 4.4.6.3),
  - ausreichende Mengen an Dammbaumaterialien zur Verfügung stehen (z.B. Aufbereitungsrückstände oder taubes Gestein) und
- unter folgenden Bedingungen auf das Centerline-Dammbauverfahren zurückzugreifen, so dass der Damm mittig aufgeschüttet und höher wird (Abschnitt 4.4.6.4), wenn nämlich
  - das seismische Risiko gering ist.

Dammbetrieb

Zusätzlich zu den in Abschnitt **4.1** und Abschnitt **4.2** beschriebenen Maßnahmen während der **Betriebsphase** (Abschnitt **4.2.3**) eines **Sedimentationsbeckens** sollen die BVT genutzt werden, um

- die Stabilität, wie nachfolgend näher spezifiziert, zu überwachen,
- bei Schwierigkeiten den Austrag in ein anderes Becken umzuleiten,
- alternative Anlagen für den Austrag bereitzustellen, möglicherweise in einer anderen Stauhaltung,
- eine zweite Dekantieranlage (z.B. ein Notüberlauf, Abschnitt 4.4.9) und/oder Standby-Pumppontons für Notfälle verfügbar zu machen, wenn der Klarwasserpegel im Becken das vorbestimmte Mindestfreibord erreicht (Abschnitt 4.4.8),
- die Bodenbewegungen mit Tiefenneigungsmessern (Tiefeninklinometer) zu messen und sich Kenntnis über die Porendruckbedingungen zu verschaffen,
- für eine ausreichende Drainage zu sorgen (Abschnitt 4.4.10),
- die Entwurfs- und Konstruktionsunterlagen zu führen sowie alle Aktualisierungen/Änderungen an den Entwürfen und der Konstruktion vorzunehmen,
- wie in Abschnitt **4.2.3.1** beschrieben ein Dammsicherheitshandbuch zu führen, zusammen mit den in Abschnitt 4.2.3.2 erwähnten unabhängigen Überprüfungen und
- das Personal entsprechend zu schulen.

Ableitung des Klarwassers aus dem Becken (Abschnitt 4.4.7.1)

Den BVT entspricht
- das Nutzen eines Ablaufbauwerks auf/in gewachsenem Boden /Fels bei talabariegelnden und an Hängen angelegten Sedimentationsbecken
- das Nutzen eines Mönchbauwerkes
  - in kaltem Klima mit positiver Wasserbilanz und
  - für Becken, die nach dem Paddock-Verfahren betrieben werden
- das Nutzen eines Dekantierbrunnen
  - in warmem Klima mit negativer Wasserbilanz,
  - für Becken, die nach dem Paddock-Verfahren betrieben werden und
  - wenn beim Betreiben des Beckens ein großer Freibord gehalten wird.

Entwässerung der Aufbereitungsrückstände (Abschnitt 4.4.16)

Die Wahl des Verfahrens (zur Gewinnung aufgeschlämmter, eingedickter oder trockener Aufbereitungsrückstände) hängt hauptsächlich von der Bewertung dreier Faktoren ab, nämlich
- den Kosten,
- der Umweltbilanz und
- der Gefahr einer Betriebsstörung.

Die BVT werden genutzt für
- die Bewirtschaftung trockener Aufbereitungsrückstände (Abschnitt 4.4.16.1),
- die Bewirtschaftung eingedickter Aufbereitungsrückstände (Abschnitt 4.4.16.2) oder
- die Bewirtschaftung aufgeschlämmter Aufbereitungsrückstände (Abschnitt 4.4.16.3).

Es gibt viele Faktoren, die die Wahl der entsprechenden Techniken an einem gegebenen Standort beeinflussen, zum Beispiel:

- die Mineralogie des Erzes,
- der Wert des Erzes,
- die Korngrößenverteilung,
- die Verfügbarkeit von Prozesswasser,
- die klimatischen Bedingungen und
- der für die Bewirtschaftung der Aufbereitungsrückstände verfügbare Platz.

Betrieb von Anlagen zur Bewirtschaftung von Aufbereitungsrückständen und taubem Gestein

Zusätzlich zu den in Abschnitt **4.1** und Abschnitt **4.2** beschriebenen Maßnahmen während der **Betriebsphase** (Abschnitt **4.2.3**) von **Anlagen zur Bewirtschaftung von Aufbereitungsrückständen und taubem Gestein** entspricht es BVT

- den natürlichen externen Oberflächenabfluss abzuleiten (Abschnitt 4.4.1),
- Aufbereitungsrückstände oder taubes Gestein zu bewirtschaften (Abschnitt 4.4.1), wobei die Stabilität der Halde bzw. des Dammes in diesem Falle irrelevant ist,
- beim Betreiben aller Halden und Dämme einen Sicherheitsfaktor von mindestens 1,3 anzuwenden (Abschnitt 4.4.13.1) und
- eine progressive Wiedernutzbarmachung (u.a. Wiederbegrünung) vorzunehmen (Abschnitt **4.3.6**).

Überwachen der Stabilität

Mit den BVT sollen

- auf einer Halde/einem Damm für Aufbereitungsrückstände (Abschnitt 4.4.14.2)
  - der Wasserstand,
  - die Qualität und Quantität des Sickerwasserflusses durch den Damm hindurch (auch Abschnitt 4.4.12),
  - die Position des Grundwasserspiegels,
  - der Porendruck,
  - die Bewegung der Dammkrone und der Aufbereitungsrückstände,
  - zur Absicherung der Stabilität des Dammes und der tragenden Schichten die Seismik (auch Abschnitt 4.4.14.4)
  - der dynamischen Porendruck und der Verflüssigungsprozess,
  - die Bodenmechanik sowie
  - die Vorgehensweise beim Einbau der Aufbereitungsrückstände überwacht werden,
- auf einer Halde (Abschnitt 4.4.14.2):
  - die Strossen- bzw. Böschungsgeometrie,
  - die Drainage unter der Kippe und
  - der Porendruck überwacht werden sowie
- auch
  - im Falle eines Beckens/eines Dammes für Aufbereitungsrückstände
    - Sichtprüfungen (Inaugenscheinnahme) (Abschnitt 4.4.14.3),
    - jährliche Überprüfungen (Abschnitt 4.4.14.3),
    - unabhängige Prüfungen (Audits) (Abschnitt 4.2.3.2 und Abschnitt 4.4.14.3) und
    - Sicherheitsbewertungen an vorhandenen Dämmen (SEED) (Abschnitt 4.4.14.3) sowie
  - im Falle einer Halde
    - Sichtprüfungen (Abschnitt 4.4.14.3),
    - geotechnische Überprüfungen (Abschnitt 4.4.14.3) und
    - unabhängige geotechnische Audits (Abschnitt 4.4.14.3) vorgenommen werden.

Verminderung von Havarieauswirkungen
Mit den BVT soll(en)

- eine Notfallplanung vorgenommen (Abschnitt 4.6.1),
- Vorfälle eingeschätzt und nachverfolgt (Abschnitt 4.6.2) und
- die Pipelines überwacht werden (Abschnitt 4.6.3).

Verringerung des Platzbedarfs/der Aufstellfläche
Mit den BVT soll(en)

- die Erzeugung von Aufbereitungsrückständen/taubem Gestein verhindert und/oder verringert werden, wenn das möglich ist (Abschnitt **4.1**)
- die Aufbereitungsrückstände unter den folgenden Bedingungen wieder verfüllt werden (Abschnitt 4.5.1), wenn nämlich
  - der Versatz als Teil des Abbauverfahrens benötigt wird (Abschnitt 4.5.1.1),
  - die zusätzlichen Kosten für die Verfüllung mindestens durch die höhere Erzausbeute ausgeglichen werden können,
  - sich beim Abbau im Tagebau die Aufbereitungsrückstände leichter entwässern lassen (d.h. durch Verdunstung, Dränung und Filtration) und dadurch ganz auf eine TMF verzichtet oder sie in der Größe verringert werden kann (Abschnitte 4.5.1.2, 4.5.1.3, 4.5.1.4, 4.4.1),
  - nahe gelegene ausgekohlte Tagebaue für die Versatzmaterialien zur Verfügung stehen (Abschnitt 4.5.1.5),
  - in Untertagebergwerken große Abbaukammern für die Versatzmaterialien zur Verfügung stehen (Abschnitt 4.5.1.6), wobei mit verschlämmten Aufbereitungsrückständen verfüllte Kammern eine Drainage erfordern (Abschnitt 4.5.1.9) und Bindemittel zur Erhöhung der Stabilität hinzugefügt werden können (Abschnitt 4.5.1.8),
- die Aufbereitungsrückstände in der Form von pastösen Versatzmaterialien wieder verfüllt werden (Abschnitt 4.5.1.10), wenn die zur Wiederverfüllung erforderlichen Bedingungen gegeben sind und wenn
  - die Notwendigkeit zum kompetenten Versatz besteht,
  - die Aufbereitungsrückstände sehr fein(körnig) sind, so dass für den hydraulischen Versatz nicht genügend Material vorhandenen wären und die großen, in ein Becken entsorgten Mengen zu lange brauchten, ehe sie entwässert wären und
  - es wünschenswert wäre, das Wasser von der Grube fernzuhalten bzw. wenn es zu kostenaufwendig wäre, das aus den Aufbereitungsrückständen austretende Wasser abzupumpen (d.h. über eine lange Strecke),
- das taube Gestein unter folgenden Bedingungen wieder verfüllt werden (Abschnitt 4.5.2), nämlich wenn
  - es innerhalb einer Untertagegrube verfüllt werden kann,
  - es in der Nähe eine oder mehrere ausgekohlte Gruben gibt (was manchmal als '*transfer mining*' bezeichnet wird)
  - der Tagebaubetrieb in einer solchen Weise erfolgt, dass das taube Gestein ohne Beeinträchtigung des Abbaubetriebes versetzt werden kann,
- die möglichen Verwendungszwecke für Aufbereitungsrückstände und taubes Gestein untersucht werden (Abschnitt 4.5.3).

Stilllegung und Nachsorge
Zusätzlich zu den in Abschnitt **4.1** und Abschnitt **4.2** beschriebenen Maßnahmen während **Stilllegungs- und Nachsorgephase** (Abschnitt **4.2.4**) von **Anlagen zur Bewirtschaftung von Aufbereitungsrückstände und taubem Gestein** soll(en) mit den BVT

- schon während der Planungsphase Stilllegungs- und Nachsorgepläne, einschließlich Kostenvoranschläge, aufgestellt bzw. erarbeitet werden, die dann im Laufe der Zeit zu aktualisieren sind (Abschnitt **4.2.4**); dennoch ergeben sich Anforderungen an die Sanierung einer TMF über deren gesamten Betriebszeitraum hinweg und können erst während der Stilllegungsphase präzisiert werden,

- bei Dämmen und Halden nach der Stilllegung ein Sicherheitsfaktor von mindestens 1,3 angewandt werden (Abschnitte **4.2.4** und 4.4.13.1), wobei die Meinungen zu Abdeckungen mit Wasser geteilt sind (siehe Kapitel **7**).

Für die Stilllegungs- und Nachsorgephase von Becken für Aufbereitungsrückstände sollen mit den BVT Dämme errichtet werden, die langfristig stabil bleiben, falls für die Stilllegung die Lösung einer Abdeckung mit Wasser gewählt wird (Abschnitt **4.2.4.2**).

Goldlaugung mit Cyanid

Zusätzlich zu den allgemeinen Maßnahmen für alle Standorte, an denen die Goldlaugung mit Cyanid erfolgt, sollen die BVT folgendes erreichen:

- Verringerung des Einsatzes von CN durch
  - Betriebskonzepte, mit denen die Zugabe von Cyanid minimiert wird (Abschnitt **4.3.2.2**),
  - automatische Cyanidkontrolle (Abschnitt **4.3.2.2.1**) und,
  - Peroxid-Vorbehandlung, wenn anwendbar (Abschnitt **4.3.2.2.2**).
- Zersetzung des verbleibenden freien CN vor dem Austrag in das Becken (Abschnitt **4.3.11.8**). **Tabelle 4.13** zeigt Beispiele für an einigen europäischen Standorten erreichte CN-Werte.
- Anwendung folgender Sicherheitsmaßnahmen (Abschnitt 4.4.15):
  - Dimensionierung des Kreislaufs für die Zersetzung des Cyanids mit einer Kapazität, die dem zweifachen der tatsächlichen Anforderungen entspricht,
  - Einbau eines Backup-Systems für die Zugabe von Kalk und
  - Installation eines Reserve-Stromgenerators.

Aluminium

Zusätzlich zu den allgemeinen Maßnahmen für alle Tonerderaffinerien sollen die BVT folgendes erreichen:

- während des Betriebs:
  - Vermeiden des Austrags von Abwasser in das Oberflächenwasser. Das wird durch die Wiederverwendung von Prozesswasser in der Raffinerie erreicht (Abschnitt **4.3.11.1**) bzw. unter trockenen klimatischen Bedingungen durch Verdunstung
- in der Nachsorgephase (Abschnitt **4.3.13.1**):
  - Behandlung des Oberflächenabflusses aus der TMF vor dem Austrag, bis die chemischen Bedingungen eine akzeptable Konzentration für den Austrag in das Oberflächenwasser erreicht haben,
  - Instandhaltung der Zufahrtsstraßen, Drainagesysteme und der abdeckenden Vegetationsschicht (einschließlich einer Wiederbegrünung, wenn notwendig),
  - laufende Probenahmen zur Feststellung der Grundwasserqualität.

Kali

Zusätzlich zu den allgemeinen Maßnahmen für alle Kali-Standorte sollen die BVT folgendes erreichen:

- Wenn der natürliche Boden nicht undurchlässig ist, muss der Boden unter der TMF undurchlässig gemacht werden (Abschnitt **4.3.10.3**).
- Verringerung der Staubemissionen von den Bandtransporten (Abschnitt **4.3.4.3.1**).
- Versiegelung/Auskleidung des Böschungsfußes außerhalb der undurchlässigen Kernzone und Auffangen des Oberflächenabflusses (Abschnitt **4.3.11.4.1**).
- Verfüllen großer Abbaukammern mit trockenen und/oder verschlämmten Aufbereitungsrückständen (Abschnitt 4.5.1.6).

Kohle

Zusätzlich zu den allgemeinen Maßnahmen für alle Kohle-Standorte sollen die BVT folgendes erreichen:

- Verhinderung von Sickerwasser (Abschnitt **4.3.10.4**)
- Entwässerung feiner Aufbereitungsrückstände (< 0,5 mm) durch Flotation (Abschnitt 4.4.16.3)

Umweltmanagement

Eine Reihe von Umweltmanagementtechniken ist als BVT eingeordnet. Der Anwendungsbereich (z. B. die Detailebene) und die Art des Umweltmanagementsystems (z. B. standardisiert oder nicht standardisiert) wird generell auf die Art, die Größe, die Komplexität der Anlage sowie die Reichweite der möglichen Umweltauswirkungen bezogen.

Mit den BVT soll ein Umweltmanagementsystem (EMS) umgesetzt und zugleich eingehalten werden, das je nach den individuellen Umständen die folgenden Inhalte hat (siehe Kapitel 4):

- Definition einer Umweltpolitik für die betreffende Anlage durch die Unternehmensleitung (das Engagement der Geschäftsführung wird als Voraussetzung für die erfolgreiche Anwendung der anderen Merkmale und Eigenschaften des EMS angesehen),
- Planung und Festlegung der notwendigen Vorgehensweisen,
- Umsetzung der festgelegten Vorgehensweisen, wobei folgenden Punkten besondere Aufmerksamkeit zu schenken ist:
  - Organisationsstruktur und Verantwortung,
  - Schulung, Bewusstmachung und Kompetenz
  - Kommunikation
  - Einbeziehung der Mitarbeiter
  - Dokumentation
  - Wirksame Prozesskontrolle
  - Instandhaltungs- und Wartungsprogramm
  - Vorbereitung und Reaktion auf Notfälle
  - Sicherstellung der Einhaltung der Umweltbestimmungen
- Kontrolle der Leistungsfähigkeit des Systems und Ergreifung von Abhilfemaßnahmen, wobei folgenden Punkten besondere Aufmerksamkeit zu schenken ist:
  - Überwachung und Messungen (siehe dazu auch das Merkblatt zur Emissionsüberwachung),
  - Abhilfe- und vorbeugende Maßnahmen,
  - Führung der Aufzeichnungen und Unterlagen,
  - unabhängige (wo praktikabel) interne Revision (Audit) um festzustellen, ob das Umweltmanagementsystem mit den geplanten Arrangements übereinstimmt und es ordnungsgemäß umgesetzt und aufrechterhalten wird oder nicht sowie
- Überprüfung durch das Spitzenmanagement.

Drei weitere Inhalte eines EMS, welche die vorgenannten Schritt für Schritt ergänzen können, werden als zusätzliche Maßnahmen zur Unterstützung der erstgenannten angesehen. Ihr Fehlen wird jedoch nicht als Widerspruch zu den BVT angesehen. Diese drei zusätzlichen Schritte sind:

- Überprüfung und Validierung des Managementsystems und der Vorgehensweise mittels Audits durch ein akkreditiertes Zertifizierungsorgan oder einen externen EMS-Prüfer;
- Ausarbeitung und Veröffentlichung (sowie möglicherweise externe Validierung) einer regelmäßigen Umwelterklärung mit einer Beschreibung aller wesentlichen Umweltaspekte der Anlage, die einen jährlichen Vergleich mit den aufgestellten Umweltzielen und, wie angebracht, mit den Eckwerten der Branche gestattet;
- Umsetzung und Einhaltung eines international anerkannten freiwilligen Systems, wie EMAS und EN ISO 14001:1996. Dieser freiwillige Schritt könnte dem EMS eine größere Glaubwürdigkeit verleihen, was insbesondere auf EMAS zutrifft, das alle o.a. inhaltlichen Merkmale und Eigenschaften in sich vereint. Trotzdem können prinzipiell auch nichtstandardisierte Systeme gleichermaßen wirksam sein, vorausgesetzt, sie sind ordnungsgemäß entworfen und werden ebenso gründlich umgesetzt.

- Insbesondere für die Bewirtschaftung von Aufbereitungsrückständen sowie taubem Gestein wird die Errichtung eines integrierten Risiko-/Sicherheits- und Umweltmanagementsystems als BVT erkannt. In diesem Fall muss das Umweltmanagement gemeinsam mit der in Abschnitt 4.2.1 beschriebenen Risikobewertung, dem dort beschrieben Risikomanagement sowie mit dem in Abschnitt **4.2.3.1** beschriebenen Betriebs-, Überwachungs- und Wartungsmanagement entwickelt und durchgeführt werden.

Techniken in der Entwicklung

Kapitel 6 beinhaltet sechs Techniken, die sich „in der Entwicklung" befinden, d.h. in der Forschungs- oder Entwicklungsphase, und noch nicht kommerziell genutzt werden. Dabei handelt es sich um:

- die gemeinsame Beseitigung von Aufbereitungsrückstände und taubem Gestein aus der Eisenerzgewinnung,
- die Verhinderung einer fortschreitenden Sauerwasserbildung,
- das Cyanid-Recycling mithilfe der Membrantechnologie,
- Trennkammern,
- die Nutzung von Rotschlamm zur Reduzierung der aus der Sauerwasserbildung und aus der Metallbelastung resultierenden Probleme sowie
- die Zerstörung von Cyanid durch Anwendung einer Kombination aus SO_2/Luft und Wasserstoffperoxid.

Diese Techniken wurden in das vorliegende Dokument aufgenommen, um das Bewusstsein für die Notwendigkeit künftiger Überarbeitungen des Dokuments zu schärfen.

Abschließende Bemerkungen

Informationsaustausch

Von der Industrie und den Genehmigungsbehörden wurden viele Dokumente bereitgestellt, die für die in diesem Dokument enthaltenen Informationen als Grundlage dienten. Bulletins der Internationalen Kommission für große Talsperren (*International Commission on Large Dams* - ICOLD) zur Bewirtschaftung von Aufbereitungsrückstände, der kanadische Leitfaden mit dem Titel „*Guide to the management of tailings facilities*" (Leitfaden für die Bewirtschaftung von Aufbereitungsrückständen) und der finnische Leitfaden zum Thema Dammsicherheit können als Grundlage für dieses BVT-Dokument betrachtet werden.

Die Menge und Qualität der Daten in diesem Dokument lässt ein Ungleichgewicht erkennen, d. h. es werden nur wenige Informationen über die tatsächlichen Verbrauchs- und Emissionsmengen von Einrichtungen zur Bewirtschaftung von Aufbereitungsrückstände und taubem Gestein bereitgestellt.

Emissionsdaten von der Metallgewinnung liegen nur aus einzelnen Einrichtungen vor. Es konnte kein Zusammenhang zwischen den angewandten Techniken und den verfügbaren Emissionsdaten hergestellt werden. Daher war es auch nicht möglich, Schlussfolgerungen über die BVT zu ziehen und Aussagen zu den Emissionswerten zu treffen.

Das erreichte Maß an Einvernehmlichkeit

Die Schlussfolgerungen aus der geleisteten Arbeit wurden auf der abschließenden Plenartagung im November 2003 mit einem hohen Maß an Einvernehmlichkeit abgestimmt. Uneinigkeit besteht hinsichtlich des Sicherheitsfaktors für die langfristige Stabilität von Dämmen mit „Nass"abdeckung.

Empfehlungen für die künftige Arbeit

Das Ergebnis dieses Informationsaustausches, d.h. dieses Dokument, stellt einen wichtigen Schritt vorwärts bei der Verringerung der täglichen Umweltverschmutzung und der Verhinderung von Unfällen dar, zu der es durch Anlagen zur Bewirtschaftung von Aufbereitungsrückständen und taubem Gestein kommen kann. Zu einigen Themen sind jedoch die Angaben unvollständig und lassen keine Schlussfolgerungen zu den BVT zu. Die künftige Arbeit könnte sich sinnvollerweise auf die Sammlung folgender Angaben konzentrieren:

- Erweiterung des Anwendungsbereichs, um alle Arten von Abfällen aus bergbaulicher Tätigkeit zu erfassen und Beispiele und Techniken im Hinblick auf andere Minerale einzubeziehen,
- detailliertere Informationen über den Anfall von Aufbereitungsrückstände und taubem Gestein
- Emissionswerte bei der Abwasserbehandlung und beim Cyanidabbau im Zusammenhang mit den BVT,
  - Bewirtschaftung von unter Wasser abgelagerten Aufbereitungsrückständen im Seewasser,
- Wirtschaftliche Daten für viele der in Kapitel 4 vorgestellten Techniken,
- Charakterisierung der Aufbereitungsrückstände und des tauben Gesteins
  - zur Einbeziehung einer größeren Zahl internationaler und nationaler Standards in Anhang 4
  - zur Entwicklung einer Standardmethode für die Beschreibung (Charakterisierung) von Aufbereitungsrückstände und taubem Gestein,
- mehr Leistungsdaten zum Verfahren für die Eindickung von Aufbereitungsrückständen sowie
- neue Techniken zur Behebung des Cyanidproblems.

Ferner muss möglicherweise auch eine Anpassung des BVT-Merkblatts an den endgültigen Anwendungsbereich der Richtlinie über die Bewirtschaftung von Abfällen aus der mineralgewinnenden Industrie erfolgen.

Themenvorschläge für künftige Forschungs- und Entwicklungsprojekte

Der Informationsaustausch hat auch einige Bereiche sichtbar gemacht, wo aus Forschungs- und Entwicklungsprojekten zusätzliche und nützliche Erkenntnisse gewonnen werden könnten. Diese beziehen sich auf folgende Themen:

- Bewirtschaftung über die gesamte Lebensdauer (“Life-cycle-Management”): Die TWG hat die Wichtigkeit der Bewirtschaftung eines Standortes über die gesamte Lebensdauer hinweg (“Life-cycle-Manegement”) erkannt, um ein hohes Niveau bei der Sicherheit und der Umweltbilanz zu erreichen. Dennoch mangelt es immer noch an wirtschaftlichen Daten, aus denen hervorgeht, dass es viel wirtschaftlicher ist, bergbauliche Operationen auf Grundlage eines vollständigen Lebensdauermodells zu gestalten. Die Forschung auf diesem Gebiet muss alle möglicherweise vorhandenen Fallstudien untersuchen, um die Wirtschaftlichkeit der Anwendung des integrierten Life-cycle-Managements im Vergleich mit der konventionellen Herangehensweise (d.h. der Erzielung von Höchstgewinnen während der Betriebsphase) festzustellen.
- Toxizität der Zersetzungsprodukte von Cyanid: Die Toxizität von Cyanid selbst ist ein ausreichend gut untersuchtes Thema, aber es scheint, dass auch einige Zersetzungsprodukte (des Cyanids) von toxikologischer Wichtigkeit sind. Angesichts der Auswirkungen von Verseuchungen an Standorten, wo Cyanid für die Goldlaugung verwendet wird, besteht die Notwendigkeit, auch die Toxizität der Zersetzungsprodukte von Cyanid zu erforschen.

VORWORT

1. Status dieses Dokuments

Dieses Dokument bildet einen Teil einer ganzen Reihe von Dokumenten, in der die Ergebnisse eines Informationensaustausches zwischen den EU-Mitgliedsstaaten und den betroffenen Industriebereichen zum Thema „beste verfügbare Techniken“ (BVT), zu der damit verbundenen Überwachung sowie zu den entsprechenden Entwicklungen vorgelegt werden. *[Das Dokument wird durch die Europäische Kommission gemäß Artikel 19(3) der vorgeschlagenen Richtlinie zur Bewirtschaftung der Abfälle aus der mineralgewinnenden Industrie (Bergbau und Steinbrüche)[3] herausgegeben und muss daher bei der Festlegung der „besten verfügbaren Techniken“ berücksichtigt werden].

*Hinweis: Die Klammer entfällt, sobald die Kommission die Publikationsprozedur abgeschlossen hat.

1.1. Hintergrund

Ausgangspunkt für dieses Dokument ist die Mitteilung der Europäischen Kommission KOM (2000) 664 über die Sicherheit bei bergbaulichen Aktivitäten (im folgenden als die „Mitteilung“ bezeichnet). Wie in Abschnitt 5.5 dieser Mitteilung dargelegt, sind mineralgewinnende Kernaktivitäten nicht durch die Richtlinie 96/61/EG des Rates (IPPC-Richtlinie) abgedeckt. Dennoch fallen bergbauliche Aktivitäten, wie sie am Standort Baia Mare verfolgt werden (die Erzeugung von Metall durch Laugung von Gold), bereits in den Gültigkeitsbereich der IPPC-Richtlinie. Absatz 2.5 (b) des Anhangs I der IPPC-Richtlinie führt “Anlagen zur Herstellung nichteisenhaltiger Rohmetalle aus Erz, Konzentraten oder Sekundärrohstoffen in metallurgischen, chemischen oder elektrolytischen Prozessen” auf.

Die Mitteilung erkennt weiterhin an, dass die IPPC-Richtlinie nicht *alle* Standorte in der Europäischen Union abdeckt, nicht einmal die *meisten* der Standorte, wo Anlagen oder Methoden zur Bewirtschaftung von Aufbereitungsrückständen oder Bergematerial (TMF) arbeiten bzw. eingesetzt werden.

Abschnitt 6 der Mitteilung schlägt einen Nachfolge-Maßnahmeplan vor, der drei Hauptmaßnahmen enthält:

- einen Nachtrag zur Richtlinie 96/82/EC des Rates vom 9. Dezember 1996 zur Kontrolle schwerer Unfallgefahren mit Gefahrstoffen (Seveso-II-Richtlinie)
- eine Initiative zur Bewirtschaftung der Abfälle aus der mineralgewinnenden Industrie
- ein BVT-Referenz-Dokument.

Bei der Entscheidung, ein Technisches Referenz-Dokument zu erstellen, in welchem BVT für die Bewirtschaftung von bergbaulichen Abfällen im Sinne von Artikel 2(6) der IPPC-Richtlinie beschrieben werden, handelte es sich um eine freiwillige Vereinbarung zwischen der Kommission, den Mitgliedsstaaten und dem Bergbausektor.

2. Definition des Begriffs “beste verfügbare Techniken” (BVT)

Um dem Leser den Kontext verstehen zu helfen, in dem dieses Dokument entworfen worden ist, beschreibt dieses Vorwort einige der relevantesten, in der IPPC-Richtlinie enthaltenen Definitionen, einschließlich der Definition des Begriffes “beste verfügbare Techniken”, sowie die für die Richtlinie zur Bewirtschaftung der Abfälle aus der mineralgewinnenden Industrie vorgeschlagenen Bestimmungen. Diese Beschreibung kann keinesfalls vollständig sein und dient lediglich der Information. Sie ist weder rechtlich verbindlich, noch ändert oder beeinträchtigt sie die Bestimmungen dieser Richtlinien.

Der Vorschlag für eine Richtlinie zur Bewirtschaftung von Abfällen aus der mineralgewinnenden Industrie enthält Maßnahmen, Vorgehensweisen und Anleitungen zur weitestgehenden Vermeidung bzw. Verringerung schädlicher Auswirkungen auf die Umwelt

3 KOM(03)319

und der daraus entstehenden Gefahren für die menschliche Gesundheit, die mit der Bewirtschaftung von Abfällen aus der mineralgewinnenden Industrie einhergehen. Dieses Dokument ist auf die Einführung einer solchen Herangehensweise bei der Bewirtschaftung von Aufbereitungsrückständen, taubem Gestein und Bergematerial aus bergbaulichen Aktivitäten gerichtet, in deren Mittelpunkt das generelle Prinzip steht, dass die Betreiber alle entsprechenden Vorbeugungsmaßnahmen gegen Umweltverschmutzung ergreifen sollten, insbesondere durch den Einsatz der besten verfügbaren Techniken, die es ihnen ermöglicht, ihre Umweltbilanz zu verbessern.

Die folgenden Definitionen sind verwandt worden:
Der Begriff "beste verfügbare Techniken", wie in Artikel 2(11) der IPPC-Richtlinie definiert, wird als "das wirksamste und fortgeschrittenste Stadium bei der Entwicklung von Aktivitäten und ihrer Betriebsverfahren beschrieben, die auf die praktische Eignung bestimmter Techniken verweisen, prinzipiell die Grundlage für Emissionsgrenzwerte zu schaffen, die so ausgelegt sind, Emissionen und Auswirkungen auf die Umwelt als Ganzes zu verhindern bzw., wo das praktisch nicht möglich ist, wenigstens zu verringern". Die vorgeschlagene Richtlinie zur Bewirtschaftung von Abfällen aus der mineralgewinnenden Industrie greift diese Definition von BVT auf.

"Techniken" schließt sowohl die verwendete Technologie als auch die Art und Weise ein, wie die Anlage/Einrichtung konstruiert und gebaut ist sowie wie sie gewartet, betrieben und außer Betrieb genommen wird;

"verfügbare" Techniken sind solche, die in einem Maße entwickelt wurden, dass sie in der entsprechenden Branche unter wirtschaftlich rentablen und technisch durchführbaren Bedingungen umgesetzt werden können, wobei bei den Kosten und Vorteilen zu berücksichtigen ist, ob diese Techniken innerhalb der betreffenden Mitgliedsstaaten eingesetzt oder hergestellt werden können, solange sie für den Betreiber in angemessener Weise zugänglich sind;

"beste" bezieht sich auf die die höchste Wirksamkeit bei der Erreichung eines hohen allgemeinen Niveaus beim Schutz der Umwelt als Ganzes.

Des Weiteren enthält Anhang IV der IPPC-Richtlinie eine Liste "allgemeiner oder in besonderen Fällen anzustellender Überlegungen bei der Festlegung der besten verfügbaren Techniken, wobei die wahrscheinlichen Kosten und der Nutzen einer Maßnahme sowie das Prinzip der Vorkehrung und Vermeidung nicht aus den Augen gelassen werden sollten:
1. der Einsatz abfallarmer Technologien;
2. die Verwendung weniger gefährlicher Stoffe;
3. das Fördern der Wiedergewinnung und des Recycelns von im Prozess gewonnener und verwendeter Stoffe sowie von Abfall, wo angebracht;
4. vergleichbare Prozesse, Einrichtungen oder Betriebsverfahren, die erfolgreich auf Branchenebene getestet wurden;
5. technologische Fortschritte und Veränderungen bei wissenschaftlichen Erkenntnissen und im wissenschaftlichen Verständnis;
6. die Art, die Auswirkungen und die Menge der betreffenden Emissionen;
7. die Inbetriebnahmedaten für neue oder vorhandene Anlagen;
8. die benötigte zeitliche Dauer für die Einführung der besten verfügbaren Technik;
9. der Verbrauch an im Prozess eingesetzten Rohstoffen, deren Art (einschließlich Wasser) und ihre Energieausbeute;
10. die Notwendigkeit, die gesamten Auswirkungen der Emissionen auf die Umwelt und die damit verbundenen Gefahren zu vermeiden bzw. auf ein Minimum zu verringern;
11. die Notwendigkeit, Unfälle und Havarien zu vermeiden, sowie die daraus entstehenden Konsequenzen für die Umwelt zu minimieren;
12. die von der Kommission gemäß Artikel 16 (2) bzw. durch internationale Organisationen veröffentlichten Informationen."

Artikel 19(2) der vorgeschlagenen Richtlinie zur Bewirtschaftung von Abfällen aus der mineralgewinnenden Industrie erlegt den Mitgliedsstaaten die Verpflichtung auf abzusichern, dass die zuständigen Behörden über die Entwicklungen auf dem Gebiet der besten verfügbaren Techniken informiert werden bzw. sie diese verfolgen.

3. Ziel und Zweck dieses Dokuments

In Abschnitt 6.3 stellt die Mitteilung fest, dass sich das BVT-Dokument mit Techniken befassen soll, um

- die tägliche Umweltverschmutzung zu reduzieren und
- Unfälle und Havarien zu verhindern bzw. ihre Auswirkungen einzuschränken.

Des Weiteren wird in der Mitteilung festgestellt, dass das BVT-Dokument das Wissen über die Maßnahmen erweitert, die verfügbar sind, ähnliche Unfälle bzw. Havarien (wie die in Baia Mare) zukünftig zu verhindern. Die Genehmigungsbehörden und die Mitgliedsstaaten wären mit diesen Informationen in der Lage zu fordern, dass Anlagen in der Europäischen Union, die Einrichtungen oder Methoden zur Bewirtschaftung von Aufbereitungsrückständen oder Bergematerial nutzen, diese hohen Umweltstandards erfüllen und dass der Branche gleichzeitig die wirtschaftliche und technische Rentabilität erhalten bleibt.

Die Kommission (Generaldirektion für Umwelt) richtete ein Forum für den Informationensaustausch (IEF) ein, wie auch eine Anzahl technischer Arbeitsgruppen unter dem Dach des IEF gegründet wurden. Dem IEF und den technischen Arbeitsgruppen gehören Vertreter der Mitgliedsstaaten wie auch der Industrie an.

Ziel und Zweck dieser Dokumentenreihe ist es, genauestens über den stattgefundenen Informationsaustausch zu berichten und Referenzangaben zur Berücksichtigung für die einbezogenen Behörden zur Verfügung zu stellen, wenn letztere Maßnahmen auf BVT-Basis festlegen. Durch die Bereitstellung entsprechender Informationen zu den besten verfügbaren Techniken wird dieses Dokument zu einem wertvollen Instrument zur weiteren Verbesserung der Umweltbilanz.

4. Informationsquellen

Dieses Dokument stellt eine Zusammenfassung von Informationen und Angaben aus einer Reihe von Quellen dar und spiegelt insbesondere die Sachkenntnis der (Arbeits-)Gruppen wider, die gegründet worden waren, um die Kommission bei ihrer Arbeit zu unterstützen und die durch die Dienste der Kommission verifiziert wurden. Alle Beiträge sind dankbar entgegen genommen worden.

5. Zum Verständnis und zur Nutzung dieses Dokuments

Die in diesem Dokument gegebenen Informationen sind zur Nutzung als Input bei der Bestimmung der jeweils besten verfügbaren Technik in speziellen Fällen gedacht. Wenn eine BVT festgelegt wird und Maßnahmen auf Grundlage einer BVT beschlossen werden, sollte stets das übergreifende Ziel im Auge behalten werden, nämlich die Erreichung eines hohen Niveaus beim Schutz der Umwelt als Ganzes. Bei dem Dokument geht es zwar um eine bestimmte Anzahl von Mineralien bzw. Gebrauchsgütern, aber die hier angewandten Techniken gelten auch für viele andere Einrichtungen. Daher kann dieses Dokument auch auf andere als nur die aufgelisteten Mineralien Anwendung finden, wenn die Fragen ähnlich sind.

Der verbleibende Teil dieses Abschnitts beschreibt die Art der Informationen, die in jedem Abschnitt des Dokuments zur Verfügung gestellt wird.

Die Kapitel **1** und **2** enthalten allgemeine Informationen über Anlagen oder Methoden zur Bewirtschaftung von Aufbereitungsrückständen, taubem Gestein und Bergematerial in der betreffenden Branche sowie über die in dieser Branche genutzten industriellen Prozesse, soweit sie für die Bewirtschaftung der Aufbereitungsrückstände und des Bergematerials relevant sind. Kapitel **3** enthält Daten und Informationen bzgl. der angewandten Techniken sowie der gegenwärtigen Emissions- und Verbrauchslevel, die die Situation in den bestehenden Anlagen zur Bewirtschaftung von Aufbereitungsrückständen, taubem Gestein und Bergematerial in der mineralgewinnenden Industrie zum Zeitpunkt der Erstellung dieses Dokuments reflektieren.

Kapitel **4** beschreibt die Techniken zur Emissions- und Gefahrenverringerung wie auch andere Techniken ausführlicher, die für die Festlegung einer BVT und die Ergreifung von auf einer BVT basierenden Maßnahmen für besonders relevant erachtet werden. Zu diesen Informationen gehören die Verbrauchs- und Emissionslevel, die durch den Einsatz der betreffenden Technik für erreichbar gehalten werden, einige Kostenvorstellungen, medienübergreifende Fragen im Zusammenhang mit dieser Technik sowie das Ausmaß, mit dem die Technik auf die genehmigungspflichtigen Anlagen zur Bewirtschaftung der ganzen Breite an Aufbereitungsrückständen, taubem Gestein und Bergematerial anwendbar ist, z.B. bei neuen, bestehenden, großen oder kleinen Anlagen. Als veraltet geltende Techniken sind nicht erfasst.

Kapitel **5** enthält die Techniken sowie die Emissions- und Verbrauchslevel, die im allgemeinen Sinne als mit einer BVT vereinbar angesehen werden. Der Zweck besteht deshalb darin, allgemeine Hinweise auf die Emissions- und Verbrauchslevel zu geben, die als entsprechender Bezugspunkt und Hilfestellung bei der Festlegung von Maßnahmen auf BVT-Grundlage dienen können. Es muss jedoch betont werden, dass dieses Dokument selbst keine Emissionsgrenzwerte vorgibt bzw. vorschlägt. Bei der Festlegung entsprechender Maßnahmen auf Grundlage der BVT sind die örtlichen, standortspezifischen Faktoren, wie die technischen Merkmale der betreffenden Anlage, ihre geografische Lage und die örtlichen Umweltbedingungen, zu berücksichtigen. Im Falle bestehender Anlagen muss auch die wirtschaftliche und technische Rentabilität einer Ertüchtigung der Anlage berücksichtigt werden. Selbst das übergreifende Ziel der Gewährleistung eines hohen Maßes an Umweltschutz als Ganzes wird oft dazu führen, dass unterschiedliche Arten von Umweltauswirkungen und -einflüssen gegeneinander abgewogen werden müssen, und diese Entscheidungen werden oft durch örtliche Überlegungen beeinflusst.

Obwohl der Versuch gemacht wird, einige dieser Fragen anzusprechen, ist es unmöglich, sie im Rahmen dieses Dokument ausführlich zu behandeln. Die in Kapitel **5** aufgeführten Techniken und Größenordnungen lassen sich daher nicht notwendigerweise auf alle Anlagen übertragen. Andererseits gehört zu der Verpflichtung, ein hohes Maß an Umweltschutz zu erreichen, auch die Einsicht, dass die auf BVT basierenden Maßnahmen nicht nur auf der Grundlage örtlicher Überlegungen festgelegt werden können. Es ist daher von größter Wichtigkeit, dass die in diesem Dokument enthaltenen Informationen von den zuständigen Behörden in vollem Umfange berücksichtigt werden.

Da die besten verfügbaren Techniken mit der Zeit Änderungen unterliegen, wird dieses Dokument je nach Notwendigkeit überprüft und aktualisiert. Zu einer Überprüfung des Dokuments kann es auch im Zusammenhang mit der endgültigen Formulierung der vorgeschlagenen Richtlinie zur Bewirtschaftung von Abfällen aus der mineralgewinnenden Industrie kommen, nachdem diese angenommen wurde. Alle Anmerkungen, Kommentare und Vorschläge sind an das europäische IPPC-Büro am *Institute for Prospective Technological Studies* unter folgender Adresse zu richten:

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Best Available Techniques Reference Document on Management of Tailings and Waste-Rock in Mining Activities

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ANWENDUNGSBEREICH

Grundlage für diese Ausarbeitung ist die Mitteilung der Europäischen Kommission über die 'Sicherheit bei bergbaulichen Aktivitäten' (KOM(2000) 664, endgültige Version). Eine der in dieser Mitteilung vorgeschlagenen Nachfolgemaßnahmen ist die Erarbeitung und Zusammenstellung eines BVT-Merkblatts. Im Abschnitt 6.3 der Mitteilung heißt es, dass das BVT-Dokument darauf gerichtet sein sollte, "ähnliche Unfälle (wie die in Aznalcóllar oder Baia Mare) in der Zukunft zu verhindern" und dass "die Verarbeitung bestimmter bergbaulich gewonnener Minerale und dort anfallender Reststoffe (in den Rahmen des Dokuments) mit einbezogen werden könnte".

Vor diesem Hintergrund wurde eine auf diese Aufgabe orientierte Technische Arbeitsgruppe (TWG) gegründet, die eine Entscheidung zu folgendem Arbeitsumfang traf:

Horizontaler Anwendungsbereich
Die Gewinnung von Gas und Flüssigkeiten (z.B. Öl und Salz aus Sole), deren Verarbeitung sowie die Bewirtschaftung von in diesem Zusammenhang anfallenden Aufbereitungsrückständen werden in dieser Arbeit nicht berücksichtigt, weil sich die dort abspielenden Prozesse deutlich von der Verarbeitung von Trockenerzen unterscheiden, wie auch die Aufbereitungsrückstände andere als die in den zu erfassenden Bereichen sind. Die Laugung von Metallen wird jedoch durch das Dokument abgedeckt.

Der dieser Arbeit zugrunde liegende Leitsatz erfasst die Verarbeitung von Mineralen, die Aufbereitungsrückstände sowie die Bewirtschaftung von taubem Gestein aus der Erzgewinnung, die alle potenziell nachhaltige Auswirkungen auf die Umwelthaben bzw. die als Beispiele für "gute Praktiken" betrachtet werden können. Die Absicht besteht darin, das Bewusstsein für die besten Praktiken bei allen in diesem Bereich ergriffenen Aktivitäten weiter zu schärfen.

In diesem Dokument werden folgende Metalle erfasst, wenn sie in der Europäischen Union („alte" EU-15), in den inzwischen beigetretenen Ländern, in den künftigen Beitrittsländern und in der Türkei abgebaut und/oder verarbeitet werden, d.h.:

- Aluminium
- Cadmium
- Chrom
- Kupfer
- Gold
- Eisen
- Blei
- Mangan
- Quecksilber
- Nickel
- Silber
- Zinn
- Wolfram
- Zink.

Diese Metalle werden unabhängig von der gewonnen Menge oder den eingesetzten Verarbeitungsmethoden erfasst (d.h. unabhängig davon, ob mechanische Verfahren, wie Flotation, oder chemische bzw. hydrometallurgische Verfahren, wie Laugung usw., zur Anwendung kommen).

Die Arbeitsgruppe entschied in Verfolgung des o.a. Leitsatzes, in diesem Dokument auch ausgewählte Industrieminerale und Kohle zu berücksichtigen.

Um die Arbeit auf einen angemessenen Zeitrahmen zu beschränken, wurde beschlossen, nicht alle Industrieminerale zu erfassen und daher eine auf zwei Faktoren beruhende Auswahl zu treffen:
1. eine beträchtliche Gewinnung in den (bisherigen) 15 EU-Staaten sowie in den inzwischen beigetretenen Ländern, in den künftigen Beitrittsländern und in der Türkei sowie
2. die Erzeugung von Aufbereitungsrückständen , die bei unsachgemäßem Umgang mit ihnen große Auswirkungen auf die Umwelt haben könnten.

Dennoch werden zusätzlich zu dieser Klassifizierung noch einige andere Minerale berücksichtigt, wenn die Bewirtschaftung der durch sie entstehenden Aufbereitungsrückstände sowie des tauben Gesteins als Beispiele "guter Praktiken" angesehen werden, die auch auf andere Minerale Anwendung finden könnten.
Auf diesen Grundlagen werden folgende Industrieminerale in dieses Dokument einbezogen:
- Baryte
- Borate
- Feldspat (wenn durch Flotation gewonnen)
- Flussspat
- Kaolin (wenn durch Flotation gewonnen)
- Kalk (wenn verarbeitet)
- Phosphat
- Kalisalz
- Strontium
- Talk(um) (wenn durch Flotation gewonnen).

Es wurde festgestellt, dass Aufbereitungsrückstände nur bei der Verarbeitung von Feldspat und Kaolin entstehen, wenn sie wenn durch Flotation gewonnen werden.

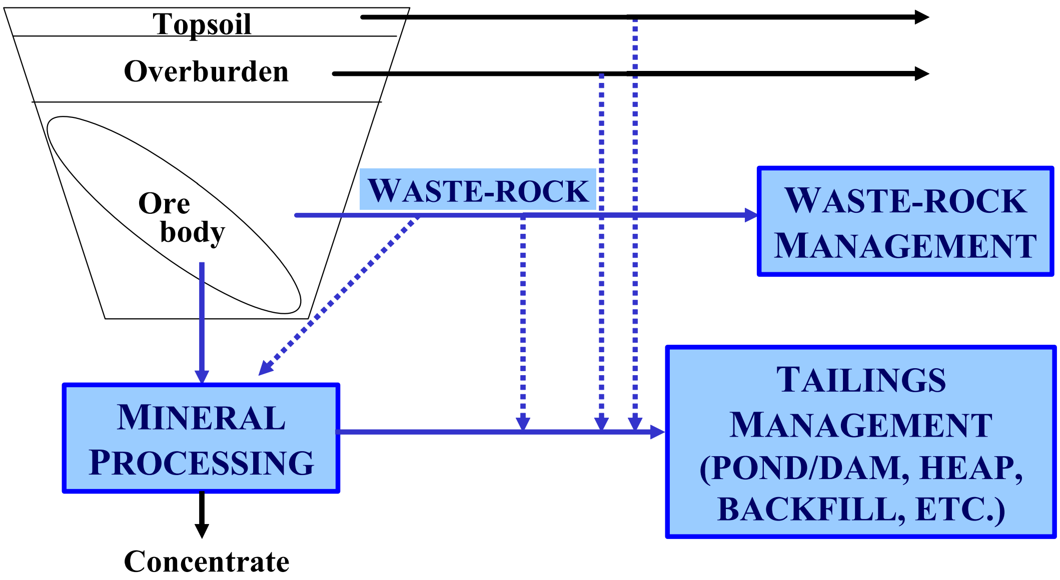
Kohle wird nur berücksichtigt, wenn sie verarbeitet wird und daraus Aufbereitungsrückstände anfallen (womit der o.a. Leitsatz weiter verfolgt wird). Im Allgemeinen bedeutet das, dass Steinkohle (bzw. Anthrazit) erfasst wird, nicht aber Braunkohle, die üblicherweise nicht (weiter) verarbeitet wird.

Der in Estland verarbeitete Ölschiefer hinterlässt große Mengen Aufbereitungsrückstände, die bewirtschaftet werden müssen. Deshalb wurde entschieden, sie mit in dieses Dokument aufzunehmen.

Die Frage der Bewirtschaftung von Aufbereitungsrückständen an stillgelegten Standorten wird in dieser Arbeit nicht berücksichtigt, obwohl einige Beispiele kürzlich stillgelegter Standorte dennoch einbezogen werden.

Vertikaler Anwendungsbereich
In Hinblick auf alle im horizontalen Anwendungsbereich aufgeführten Minerale
- beschäftigt sich das Dokument mit der Bewirtschaftung des tauben Gesteins,
- erfasst das Dokument sowohl Mutterboden als auch Abraum, wenn diese bei der Bewirtschaftung der Aufbereitungsrückstände verwendet werden,
- erfasst das Dokument die für die Bewirtschaftung der Aufbereitungsrückstände relevante Mineralverarbeitung (d.h., in welchem Maße die Mineralverarbeitung die Eigenschaften und das Verhalten der Aufbereitungsrückstände beeinflusst) und
- konzentriert sich das Dokument auf die Bewirtschaftung des tauben Gesteins, z.B. in Becken/ Dämmen, auf Halden oder als Versatzmaterial.

Nachfolgende Darstellung zeigt den vertikalen Anwendungsbereich. Die blau unterlegten Kästen verweisen auf die in diesem Dokument erfassten Prozessschritte.



Darstellung des vertikalen Anwendungsbereiches

In diesem Dokument bezieht sich

‘Bergwerksproduktion’ bei Metallen auf den Anteil von Metall im Konzentrat nach der Gewinnung, in allen anderen Fällen – wenn nicht anders angegeben – auf den gewichtsmäßigen Anteil des Konzentrats nach der Verarbeitung;

‘Europa’ auf die (bisherigen) 15 EU-Staaten sowie die inzwischen beigetretenen Länder, die künftigen Beitrittsländer und die Türkei;

‘TMF’ (‘Tailings Management Facility’) auf eine Anlage oder Methode zur Bewirtschaftung von Aufbereitungsrückständen oder Bergematerial, was im Einzelfall ein Absetzbecken-/Damm-System, Versatzmaterial, eine Kippe mit Aufbereitungsrückständen oder irgendeine andere Art der Bewirtschaftung von Aufbereitungsrückständen sein kann.

Weitere Erklärungen zu den in diesem Dokument benutzten Fachbegriffen sind im Glossar zu finden.

1 GENERAL INFORMATION

Mining is one of mankind's oldest industries. This industry has a significant history throughout Europe. Archaeological investigations at the Los Frailes mine in southern Spain discovered the body of a worker with a copper collar dated 1500 bc. However, there are older examples of mineral working in Europe, including Neolithic flint working, and metalliferous mining dating back to almost 2000 bc. Mining has been undertaken by many civilisations and has in many areas been a source of wealth and importance. A good example in more recent times is the importance of coal mining (together with other 'heavy industries') in Germany for the 'Wirtschaftswunder' after World War II.

In the last few decades, metals and coal mining on a worldwide scale have moved away from underground operations towards larger bulk mining in open pits. As a consequence now larger amounts of residues result from these operations, mainly because the often unwanted topsoil and overburden have to be removed to gain access to the ore. In many cases, the amount of overburden and waste-rock that have to be transported is many times more than the tonnage of ore that is extracted. The amount of tailings generated depends on the content of the desirable mineral(s) in the ore, its grade, and the efficiency of the mineral processing stage in recovering this/these. Another factor is the duration of an operation. As already stated, the total amount of tailings can be very large in comparison to the amount of product, unless there is a sufficient way to use the residues. Grades vary between several grams per tonne of ore to 100 % (i.e. pure metal or mineral). The increase of bulk mining in open pits has also led to mining becoming a more capital intensive business, where in many cases it can take many years before the invested money is 'returned' through the sold product, i.e. typically the concentrates.

The purpose of mining is to meet the demand for metals and minerals resources to develop infrastructure etc. and to improve the quality of life of the population as the extracted substances are the raw materials for the manufacture of many goods and materials. These can be, for example, metalliferous minerals or metals, coal, or industrial minerals that are used in the chemical sector or for construction purposes. At any rate, the management of the residues produced, the topsoil, overburden, and, of special concern in this document, the tailings and waste-rock, presents an undesired financial burden on operators. Typically the mine and the mineral processing plant are designed to extract as much marketable product(s) as possible. The residue and overall environmental management is then designed as a consequence of these process steps.

Some parts of the mining industry, such as metal and coal mining within Europe, operate under severe economic conditions, mainly because the deposits can no longer compete on an international level. The EU metal sector is also struggling from the difficulty of trying to find new profitable ores in known geological regions. Hence the ability for the metal and coal mining sectors to invest in non-productive expenditures such as tailings and waste-rock management, may be constrained. However, despite the reduced mine production in these areas, consumption is steadily increasing. Therefore, to meet this demand imports into Europe are on the increase.

In contrast to the mostly declining production figures in the metal and coal mining sectors, the production of many industrial minerals has been expanding steadily on a European scale.

The following sections try to give an overview of the metal, potash, coal and oil shale mining sectors. In terms of economics, mines will open when it is economical to do so, be mothballed if short-term low prices persist or may even be closed if there is no prospect of their being viable. However, this chapter tries to provide an overview of the economic situation for each different mineral.

The mining production figures used in the following sections originate from the 'world mining data' book [30, Weber, 2001]. Where appropriate these numbers have been revised by the technical working group members.

1.1 Industry overview: metals

For the detailed discussions, this sector is divided into the following sub-sectors:

- aluminium
- base metals (cadmium, copper, lead, nickel, tin, zinc)
- chromium
- iron
- manganese
- mercury
- precious metals (gold, silver)
- tungsten.

The following table shows that for most of these metalliferous ores, European production is small compared to overall world production.

Commodity	Percentage of world production (%)
Iron	3
Bauxite	3
Cadmium	16
Chromium	12
Copper	7
Lead	11
Manganese	0.5
Mercury	17
Nickel	2
Tin	1
Tungsten	11
Zinc	12
Gold	1
Silver	10

Table 1.1: Production of metal concentrates within Europe as percentages of world metal concentrate production in 1999

Over many years in Europe, ore deposits containing metals in viable concentrations have been progressively depleted and few indigenous resources remain. Also, a decreased interest for exploration and development within European due to the relatively high production costs, competitiveness with regard to land use and due to political pressure, together with the discovery of large mineral deposits in other parts of the world, have led to a reduction in European originated concentrates and a subsequent import of concentrates into Europe from a variety of sources worldwide.

Metalliferous ore deposits usually have the minerals finely disseminated within the ore. Also the metalliferous ore minerals within the deposit are mostly irregularly intergrown. To liberate the desired mineral, the ore has to be reduced in size to a fine powder, so that the metalliferous minerals can then be recovered from the ore via different mineral processing techniques, in many cases by froth flotation. Since flotation is a ‘wet’ process the tailings from metal processing are typically in the form of a slurry and are managed in tailings ponds. If the metal(s) is (are) mined in an open pit, large amounts of waste-rock also have to be handled, usually on heaps or dumps.

Most metals are mined as sulphide or oxide minerals. Sulphidic metalliferous minerals often, but not always, contain pyrite, an iron sulphide. Irrespective of the mineral processing method used, some of these metal-sulphide complexes will always be included in the tailings. If air and water have access to the tailings or the waste-rock acids can be formed, that can have a high environmental impact. This phenomenon is called 'Acid Rock Drainage (ARD)' and is explained in detail in Section 2.7. The ARD potential of precious metal ores is often smaller than for massive sulphide ores (usually base metal ores). In general, the sulphur content of bauxite, chromium, iron, manganese and tungsten mineralisations is of minor importance.

1.1.1 Aluminium

In the production of primary aluminium, as a first step the raw material, called bauxite, is refined to alumina. In a second step, the alumina is converted in a smelter to aluminium. The tailings management of the alumina refining is covered in the scope of this work. The smelting part is discussed in the BREF on non-ferrous metals. [35, EIPPCB, 2001]

Bauxite is a naturally occurring, heterogeneous material, primarily composed of one or more aluminium hydroxide minerals, plus various mixtures of silica, iron oxide, titanium oxide, aluminosilicate, and other impurities in minor or trace amounts.

Bauxite is, in most cases, imported from Australia, Brazil, and the equatorial regions of West Africa, principally Guinea and Ghana. The products of alumina refineries are calcined alumina and, in some cases, aluminium hydrate. The alumina is usually shipped to smelters [33, Eurallumina, 2002].

The worldwide demand for aluminium, which directly determines the alumina demand, is currently static after a long period of continuous increase. The annual production of metal aluminium is currently 21 million tonnes, and correspondingly the prduction of alumina metallurgical grade is around 44 million tonnes. [33, Eurallumina, 2002].

There are six European countries that mine bauxite, which altogether produced 2.2 millon tonnes in 2001 [70, EAA, 2002]. However, there are ten alumina plants that refine imported and/or mined bauxite.

The ten alumina refineries in Europe are listed in Table 1.2.

Country	Plant	Production (kt)
France	Pechiney, Gardanne	600
Germany	Aluminium Oxid, Stade	820
Greece	Aluminium de Greece, Distomon	710
Ireland	Aughinish Alumina, Aughinish	1550
Italy	Eurallumina, Sardinia	990
Spain	Alcoa Inespal, San Ciprian	1300
UK	British Alcan, Burntisland	100
Hungary	Ajka	300
Romania	Tulcea Oradea	330 200
TOTAL:		**6800**

Table 1.2: Alumina refineries in Europe alumina production year 1999
[34, EAA, 2002]

The dominant bauxite producer worldwide is Australia, being about 50 million tonnes in 1999. Other producers are Guinea, Brazil, Jamaica, China and India.

The European alumina production of 6.8 million tonnes represents 13 % of the world alumina production. Typically bauxite is refined near the producing mines in order to minimise transport costs, with only high-grade bauxite being shipped to refineries over long distances.

Most of the alumina is sold under long-term contracts, with prices fixed at 11 to 13 % of the metal price fixed for aluminium by the London Metal Exchange (LME). After a period at USD 1500 per tonne, the Al price has now dropped due to recession in the US and Japan. At present, Al is priced at USD 1360 per tonne (average 2002 prices), and is expected to remain little changed for the next two years. Hence, the corresponding alumina price is around USD 164 per tonne [33, Eurallumina, 2002].

The alumina operating cost of the EU producers ranges between USD 160 and 200 per tonne, which is higher than in most non-EU countries [33, Eurallumina, 2002].

The tailings from the refining are a reddish slurry called 'red mud' and a coarser fraction called 'sand'. They have an elevated pH and contain several metal complexes. Of the EU-15 refineries, some apply thickened tailings management of these caustic tailings, some discharge into the Mediterranean, while others utilise conventional tailings ponds and one site manages the red mud in a pond after neutralising the mud with seawater and a flue-gas desulphurisation process. [33, Eurallumina, 2002].

1.1.2 Base Metals (Cadmium, Copper, Lead, Nickel, Tin, Zinc)

Currently base metal prices are low. In many cases, the mineral deposits are relatively complex from a processing point of view. These two factors, combined with the high labour costs in Europe, have led to some temporary and some final closures of mines.

Base metals can often be found jointly, as complex ores, in the same mineral deposit. They are often separated in the mineral processing phase by selective flotation.

There is a big imbalance between European mine production and the European consumption of these metals. A good example is lead, where in 1999 the European consumption was close to 2 million tonnes, which is about 6 times the amount of lead produced from European mines (350000 t) in the same year.

In this section, the subsequent refining, often smelting, will be briefly discussed but, for further details see the BREF on non-ferrous metals industries. [35, EIPPCB, 2001]

<u>Cadmium (Cd)</u>
Cadmium is often found in zinc-concentrate after mineral processing, so the cadmium will be removed at the smelter. In addition, lead and copper ores may contain small amounts of cadmium. [35, EIPPCB, 2001] Cd is always a by-product which is recovered in smelters. There are no cadmium mines that produce a Cd concentrate.

World production in 1999 was about 16500 tonnes of cadmium in concentrates, of which 14.5 % (2400 tonnes) was produced from European mines. The following figure shows the main producers in Europe.

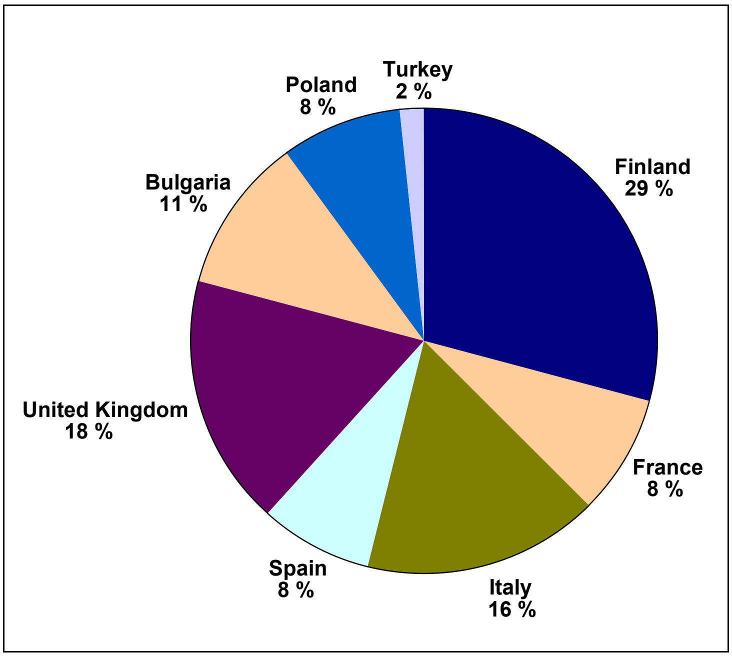


Figure 1.1: Primary cadmium production in Europe in 1999

<u>Copper</u>

Copper is mostly found in nature in association with sulphur. It is recovered from a multistage process, beginning with the mining and concentrating of low-grade ores containing copper sulphide minerals, and followed by smelting and electrolytic refining to produce a pure copper cathode. Worldwide, an increasing amount of copper is produced from the acid leaching of oxidised ores [36, USGS, 2002].

Sulphide minerals are usually recovered using flotation. Oxides, carbonates and silicates are leached.

The world production of copper in 1999 was 12.4 million tonnes. European mine production was 890000 tonnes, which represents 7.2 % of the world production. The following figure shows the main producers in Europe.

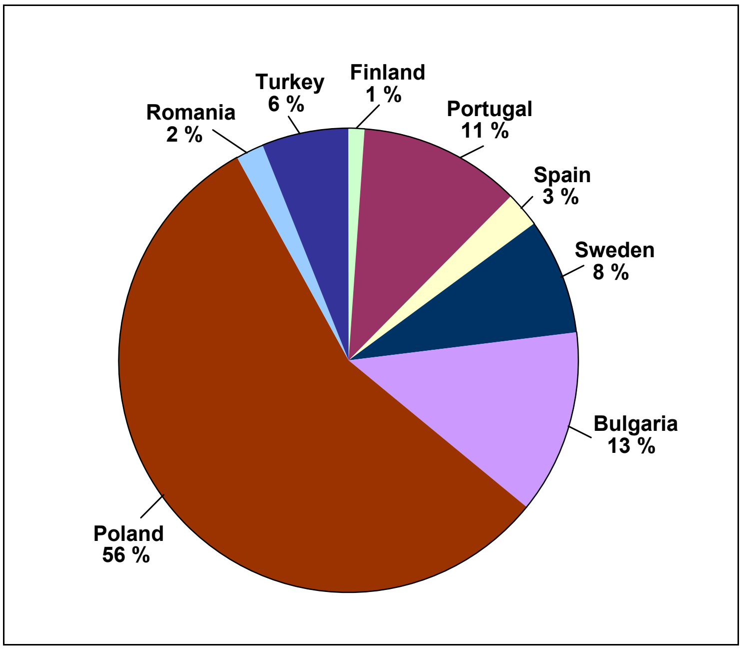


Figure 1.2: Copper mining production in Europe in 1999

While copper prices have begun to recover from their recent lows, they remain at low levels. This provides a challenge for the copper producers, especially the underground mining operations, due to their increased cost for extraction compared to open pit operations. Fortunately, these operations have succeeded over the last decade to significantly reduce their costs to such a point that they are now able to make a profit even at present prices.
[KGHM Polska Miedz, 2002 #113]

Lead
Lead ores occur primarily as sulphides or, nowadays, more commonly in complex ores where it is associated with zinc and small amounts of silver and copper. There have been major changes in the pattern of lead use over the years. The battery industry creates up to 70 % of the demand, which is fairly stable, but other uses for lead are in decline.

Usually the lead concentrate is achieved by selective flotation. The metal is recovered from the concentrate by smelting.
The world mining production of lead in 1999 was 3.3 million tonnes, about 10 % of which (about 350000 tonnes) came from European mines. The following figure shows the main producers in Europe.

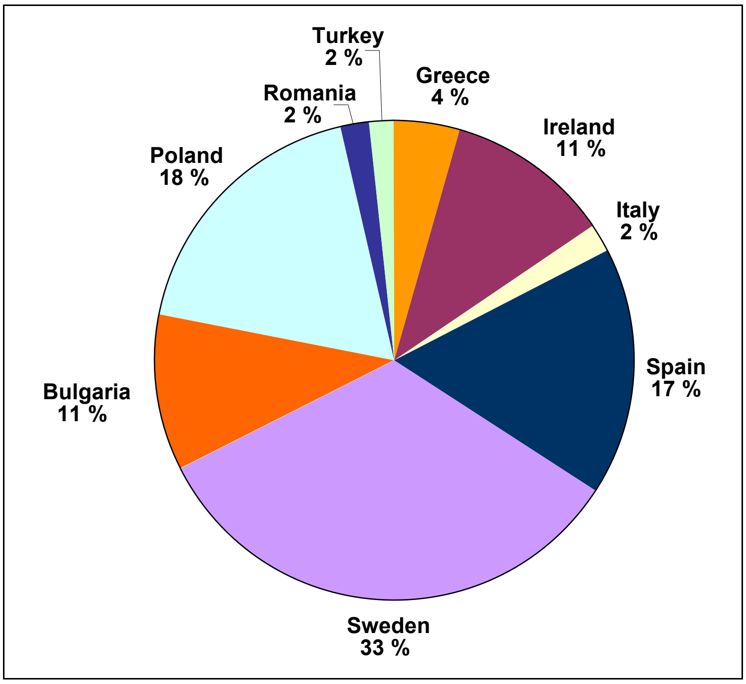


Figure 1.3: Lead mine production in Europe in 1999

Although lead ore is mined in many countries around the world, three quarters of the world output comes from only six countries: China, Australia, US, Peru, Canada and Mexico. Lead extraction in Russia has greatly declined following economic change. Total production has been at a similar level since the 1970s; with new mines being opened or expanded to replace old mines. (Note: all these mines contain at least two metals-lead, zinc, and sometimes silver, gold and copper).

Nickel
Nickel is used in a wide variety of products. Most primary nickel is used in alloys; the most important of which is stainless steel. Other uses include electroplating, foundries, catalysts, batteries, coinage, and other miscellaneous applications. [35, EIPPCB, 2001]

Europe produced only 1.4 % of the world mine production in 1999 (about 1.1 million tonnes). The following figure shows the most important producers in the world.

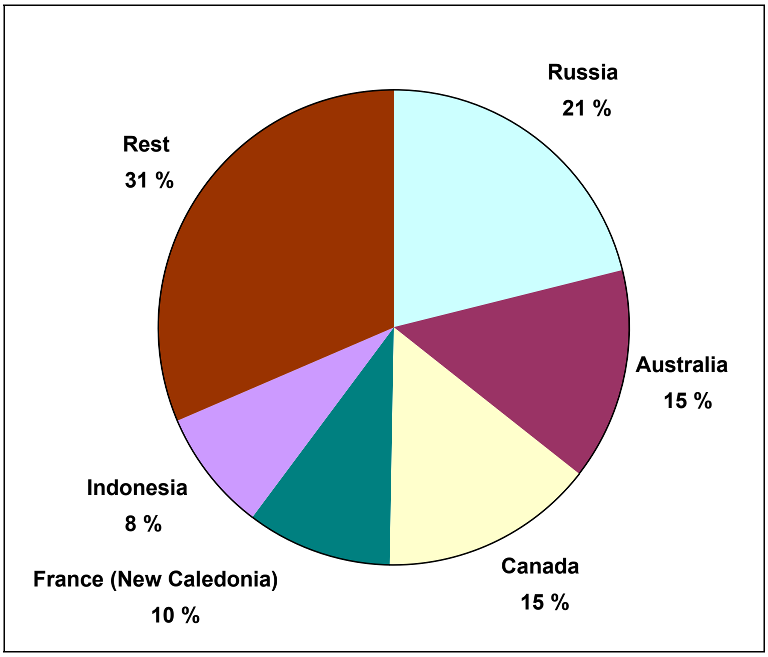


Figure 1.4: World nickel mine production in 2001

There are only two producers in Europe: Greece with 13500 tonnes and Finland with 1000 tonnes in 1999. However, since New Caledonia is part of France, this may also be considered as part of the European production, which would mean that European production provides more than 11 % of the world production.

World production in 2001 was significantly increased due to three new mines being opened in Western Australia. At these sites, nickel is recovered on-site using advanced Pressure Acid Leach (PAL) technology. At least four other Australian PAL projects are in varying stages of development. Competitors are also considering employing PAL technology in Cuba, Indonesia, and the Philippines. In April 2001, a Canadian company launched an innovative PAL project in New Caledonia. If the New Caledonian project is successful, the company will use the technology in Newfoundland to recover nickel and cobalt from sulphide concentrates. The concentrates would come from the Voisey Bay nickel-copper sulphide deposit in north-eastern Labrador. In late 2001, development of the Voisey's Bay deposit was still in limbo, as the Canadian company and the Government of Newfoundland have so far been unable to agree on critical concepts.
[36, USGS, 2002].

Tin

Nearly every continent has an important tin-mining country. Tin is a relatively scarce element, with an abundance in the earth's crust of about 2 ppm, compared with 94 ppm for zinc, 63 ppm for copper, and 12 ppm for lead. Most of the world's tin is produced from placer deposits; at least one-half comes from south-east Asia.
[36, USGS, 2002].

The world tin production in 1999 was about 230000 tonnes. Of this, Europe contributed 1 %. The only European producers are Portugal (2163 tonnes) and the UK (100 tonnes).

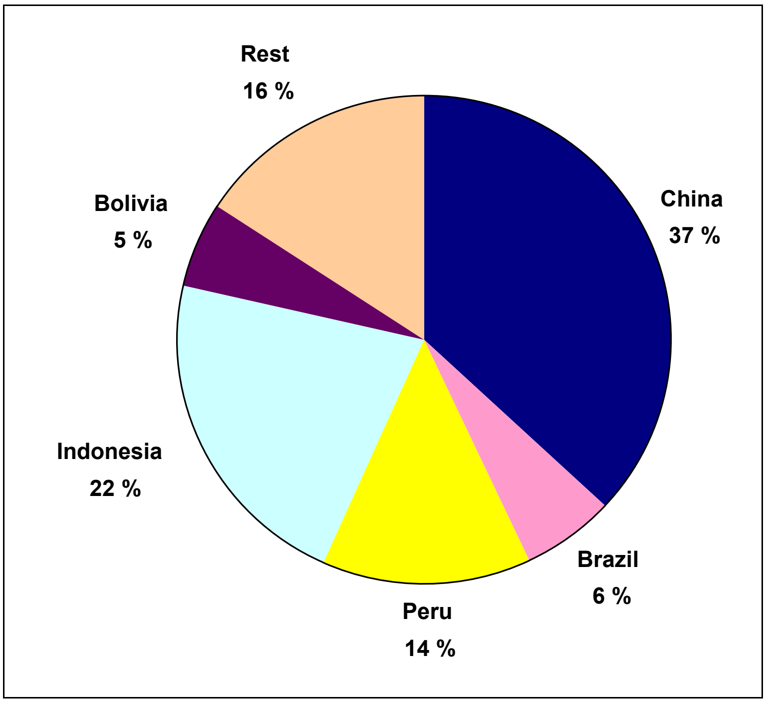


Figure 1.5: World tin mining production in 1999

As can be seen from the figure above, China is by far the largest producer of tin, and also has the largest reserves.

Tin prices continued to decline in 2001. Industry observers attributed lower prices to an oversupply of tin in the market [36, USGS, 2002]. World tin consumption was also believed to have declined somewhat during that year.

Zinc

Sphalerite (zinc iron sulphide, ZnS) is one of the principal ore minerals in the world. Zinc, in terms of tonnage produced, is the fourth most popular metal in world production—being exceeded only by iron, aluminium, and copper.

The zinc is normally recovered from the mined concentrate by leaching and electrowinning.

Europe accounted for 11.8 % of the total world mined production of about 7.5 million tonnes in 1999. The following figure displays the major European zinc producers.

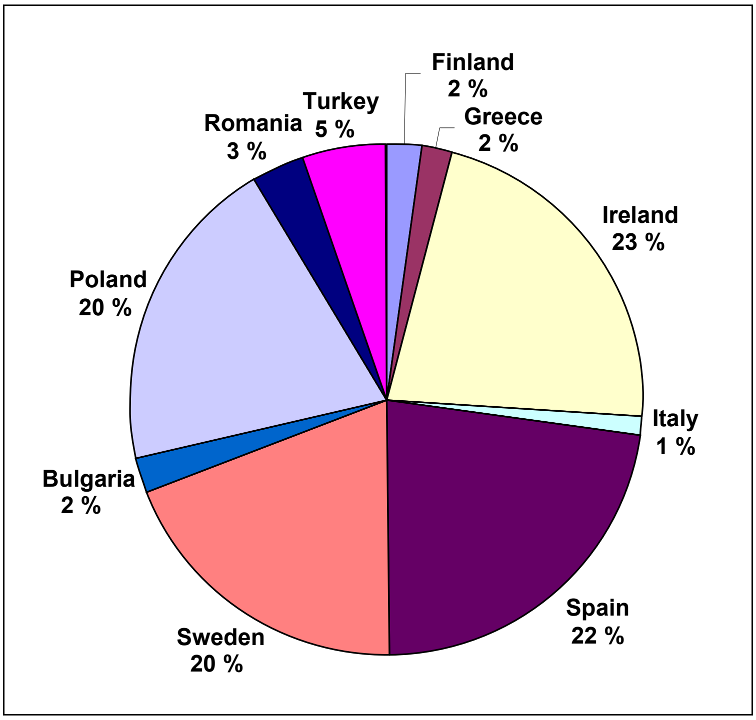


Figure 1.6: Zinc mining production in Europe in 1999

The **tailings from base metal mining activities** can be characterised as follows:

- usually a slurry of 20 - 40 % solids by weight
- containing metals
- containing sulphides
- large amounts produced.

The slurried tailings are managed in ponds. With some underground mines the coarse tailings are used as backfill material.

The sulphide in tailings and waste-rock can oxidise when water and air have access and an acidic leachate is generated. This phenomenon is called Acid Rock Drainage (ARD). Due to ARD, not only is the physical stability of the tailings ponds and dams an issue but so is the chemical stability of the acid generating tailings, both during operation and after the mine closure.

Note, waste-rock is stacked on heaps. The waste-rock from these activities can also have a high environmental impact if it has a net acid generating potential.

1.1.3 Chromium

In Europe, two countries produce significant amounts of ferrochromium; Finland (about 250000 tonnes in 1999 from a single mine) and Turkey (about 430000 tonnes in 1999). Turkey is the fourth largest chromium producer in the world. Greece produces smaller amounts, i.e. 1000 tonnes in 1999. The European mine production represents about 12 % of the world production (5.8 million tonnes in 1999). The three major world producers are South Africa, India and Kazakhstan.

The use of chromium (Cr) to produce stainless steel and non-ferrous alloys are two of its more important applications. Also, chromites, poor in iron and silica, are used for the production of refractory products. Chromite ($FeCr_2O_4$) is the most important chromium mineral, indeed it is the one from which chromium derives its name.

The concentrate from the Finnish mine is shipped directly to a stainless steel smelter owned by the same company.

The slurried tailings are managed in ponds. Currently, at the Finnish site, the waste-rock is managed on heaps. In the future, the operation will turn from an open pit to underground mining, which will almost eliminate the production of waste-rock. All waste-rock will then be used as backfill.

1.1.4 Iron

Iron ore is a mineral substance which, when heated in the presence of a reductant, will yield metalliferous iron (Fe) [55, Iron group, 2002].

Iron ore is the source of primary iron for the world's iron and steel industries. It is therefore essential for the production of steel. Almost all iron ore (i.e. 98 %) is used in steelmaking [36, USGS, 2002].

In the beginning of the 20th Century the US was the world's largest iron ore producer, accounting for about 60 % of the total yearly world output of approximately 45 million tonnes. By the end of the century the world iron ore production had grown to more than one billion tonnes per year.

In 2000, China was the largest producer in gross weight of ore produced, but because its ore was of such low grade, the country's output ranked well below Australia's and Brazil's output, of 171 and 200 million tonnes respectively. Iron ore is mined in about 50 countries. The seven largest of these producing countries account for about three-quarters of the total world production, which was about 560 million tonnes in 1999. Australia and Brazil together dominate the world's iron ore exports, each providing about one-third of the total exports. The European iron ore mining industry is of little significance on a world scale, only generating 3 % of the yearly world production.

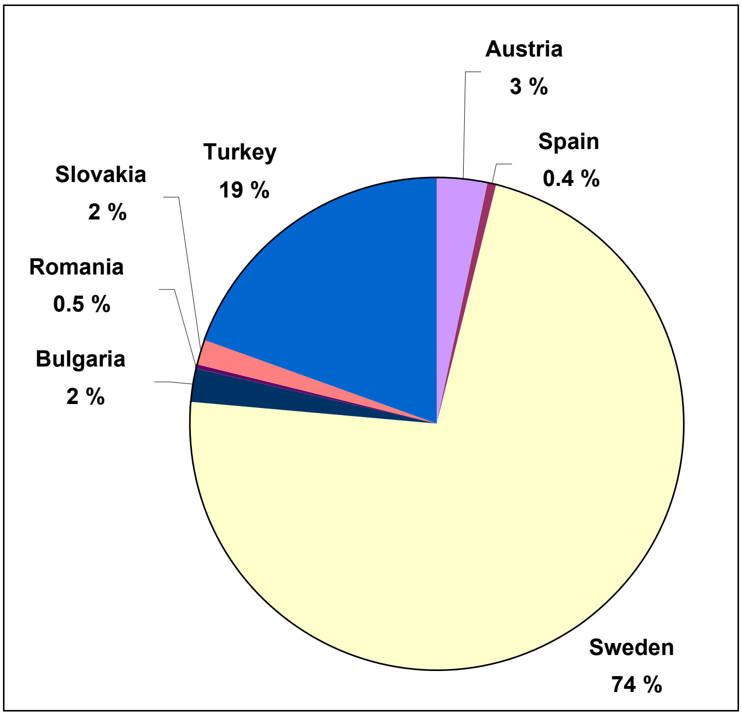


Figure 1.7: Iron mining production in Europe in 1999

The biggest iron ore producing company in the world is CVRD of Brazil. The sales of this group reached a new record of 143.6 million tonnes in 2001. The London-based Rio Tinto group produced 115.8 million tonnes and shipped 110.6 million tonnes in the same year. Corresponding figures for the Australian/South African group BHP Billiton, was 82.6 million tonnes and 84.5 million tonnes respectively in 2001. At present these big three control approximately 70 % of the iron ore market.

Iron ore production in Western Europe is now mainly concentrated in Sweden, as the production of iron ore in the 'minette' regions of France/Luxembourg ceased in the first half of 1990s, as did the iron ore mining in Spain. There are still some small scale operations for domestic use in Turkey, Austria and Norway, the latter also producing some for export. In Eastern Europe, Slovakia, Bulgaria and Romania are represented in the statistics of iron ore producers.

Of the merchant iron ore products, 490 million tonnes in 2000, pellets accounted for about 90 million tonnes. The rest consisted of coarse ores (approximately 70 million tonnes) and fines. Iron ore fines are used as a feed to blast furnaces, after sintering or pelletising processes. Pellets are split up into two types, depending on their use, i.e.for blast furnaces use, or as feed for the expanding Direct Reduced Iron/Hot Briquetted Iron (DRI/HBI) industry.
[49, Iron group, 2002]

The end of the 20th Century saw a wave of company amalgamations in the iron ore industry as producers strove to reduce production costs and become more competitive. This period of consolidations is thought to have come close to an end, though there is still some potential for further acquisitions and mergers. [49, Iron group, 2002]

For iron ore mining in Europe, this metal is only mined in the form of oxides and carbonates and the ores either contain little or no sulphide minerals. The tailings and waste-rock from these operations do not have a net ARD potential. Typically, a coarse tailings fraction is generated which is managed on heaps. The fines are discharged into tailings ponds.

1.1.5 Manganese

Steelmaking accounts for most of the manganese (Mn) demand [36, USGS, 2002].

In some cases, manganese is the prime product of a mine (e.g. Hotazel mine in South Africa or Nikopol mine in the Ukraine), but usually, manganese is associated with other minerals (e.g. iron-carbonates). One positive effect of this association with iron is that in steel production less additional manganese needs to be added [38, Weber, 2002]

The European mine production of 43500 tonnes in 1999 represents 0.5 % of the world production in the same year. The following figures show the European and the largest international producers.

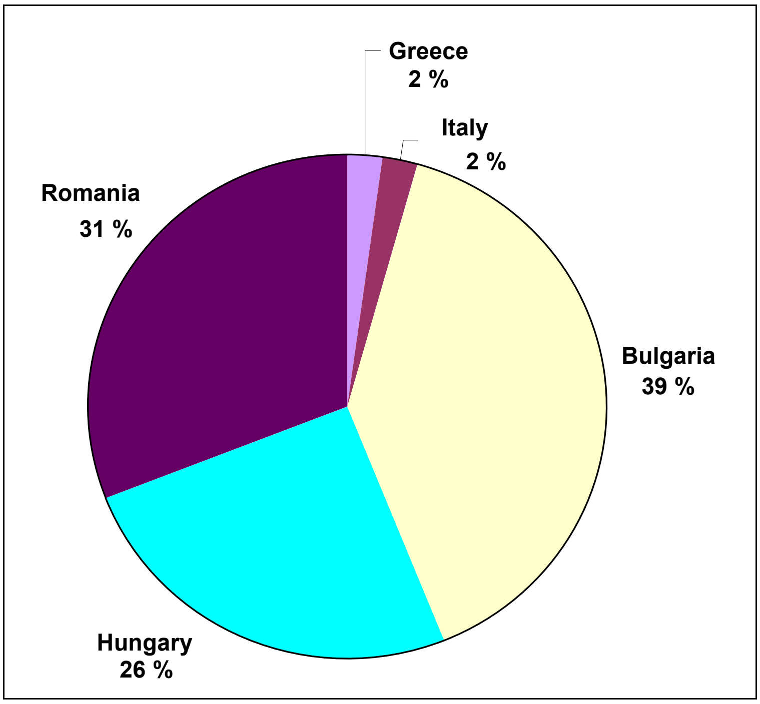


Figure 1.8: Manganese mining production in Europe in 1999

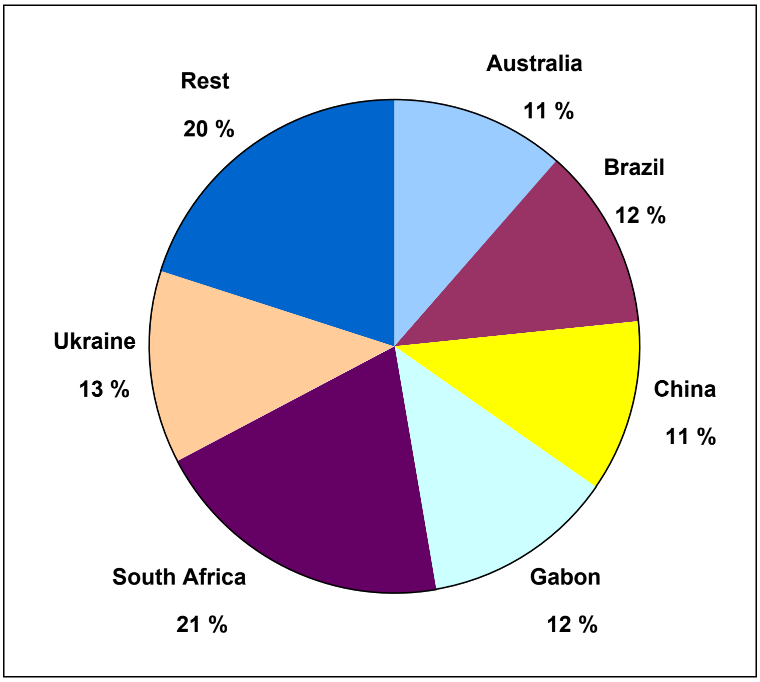


Figure 1.9: World manganese mine production in 1999

The free on board (f.o.b.) price of the manganese ore from the Hungarian operation is USD 42 per tonne.

1.1.6 Mercury

Cinnabar (HgS) is the main ore of mercury. [37, Mineralgallery, 2002]. Mercury is the only common metal that is liquid at room temperature. It occurs either as native metal or in cinnabar, corderoite, livingstonite, and other minerals [36, USGS, 2002].

The only remaining European mercury mine is the Almadén mine in Spain. The mine was subsidised from the Spanish state with a commitment to reduce mining activities. In 1995, EUR

5222 million were paid to the holding company which includes the Almadén mine. In 1999, about 100 persons were directly employed in the mining section of the company. However this mine has now been closed and is unlikely to be recommissioned. Other mines, although mining other metal sulphides, sometimes produce mercury as a by-product. One example is the Pyhäsalmi Oy Mine, which produces Cu-, Zn-, Pyrite concentrates that include Cd, Hg, Au and Ag.

World mercury mining is currently carried out in about ten countries, with the largest quantities coming from Spain and Kyrgyzstan. Over the past ten years the estimated annual world mine production of mercury has averaged about 2500 tonnes, but world production values have a high degree of uncertainty. Annual world mining of mercury is declining and was estimated 1640 tonnes in 2000. In 1999 European production represented 17.4 % of the world production.

Mercury use in Western Europe and North America has declined because of numerous restrictions on the use of mercury-containing products. The chlor-alkali industry will also gradually cease to be one of the major users. At the same time, the supply of secondary and recovered mercury has increased due to environmental regulation.

This leaves most developed countries as net exporters of mercury, which has led to steadily declining mercury prices. The market price since 1990 has been very low: prices in 1997-1999 were around EUR 4 per kg of mercury. The surplus of mercury on the market keeps the price of mercury low, which may encourage additional uses and lead to increased demand on a global scale, in particular outside the OECD. Mercury is exported to developing countries for re-use in gold recovery for use in the production of cosmetics, paints and pesticides, in addition to application types shared with OECD countries, such as in measurement and electrical devices. In this respect, the effects of the continuing exports of mercury by European companies to developing countries, where its use may lead to pollution and adverse health effects, need to be given full consideration. Furthermore, a significant part of the mercury could return to Europe as long-range transboundary air pollution.
[112, Commission, 2002]

Since the tailings contain sulphides, the generation of ARD will be an issue with Hg mines. Older Hg mines, waste-rock heaps and tailings management facilities will also cause problems. ARD and the seepage of heavy metals can be expected for many years if the sites are not properly decommissioned. However Hg in the S form is not water soluble and should therefore remain stable in the tailings and waste-rock.

No information has been provided on the management of tailings and waste-rock at mercury mines.

1.1.7 Precious Metals (Gold, Silver)

Most of the gold and silver produced is used in the manufacture of jewellery but, due to properties such as their high electrical conductivity and resistance to corrosion, they are also increasing by being used as industrial metals.

Of an estimated 140000 tonnes of all gold ever mined, about 15 % is thought to have been lost, used in dissipative industrial uses, or otherwise unrecoverable or unaccounted for. Of the remaining 120000 tonnes, an estimated 33000 tonnes are official stocks held by central banks and about 87000 tonnes is privately held as coin, bullion, and jewellery [36, USGS, 2002].

In some cases gold and silver are directly turned into crude metal at an on-site mineral processing plant as doré, containing typically 75 % gold and 25 % silver. In other cases, gold and silver are found in other metal concentrates and are recovered in the smelting process [36, USGS, 2002], for instance, a considerable amount of silver originates from the desilvering of lead.

Gold occurs in native form (free-gold) or locked in other minerals (pyrite, quartz etc). It can contain a variable amount of silver in solid solution. Gold-silver tellurides can also be a minor addition in commercial gold deposits.

Of approximately 2.5 million kg of gold mined worldwide in 1999, Europe produced only 0.8 %. For silver, European production represented approximately 10 % of the world production.

The following two figures show the main producers of gold and silver in Europe.

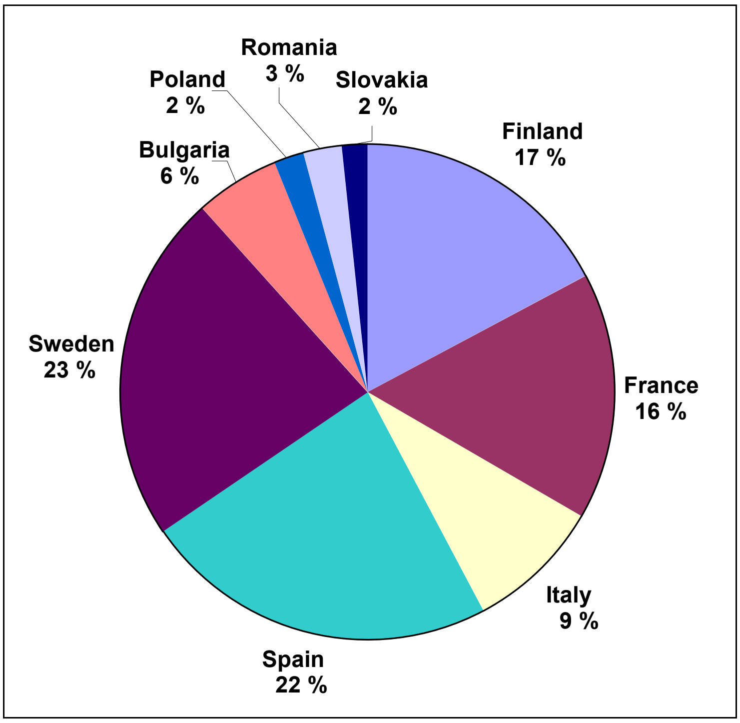


Figure 1.10: Gold mining production in Europe in 1999

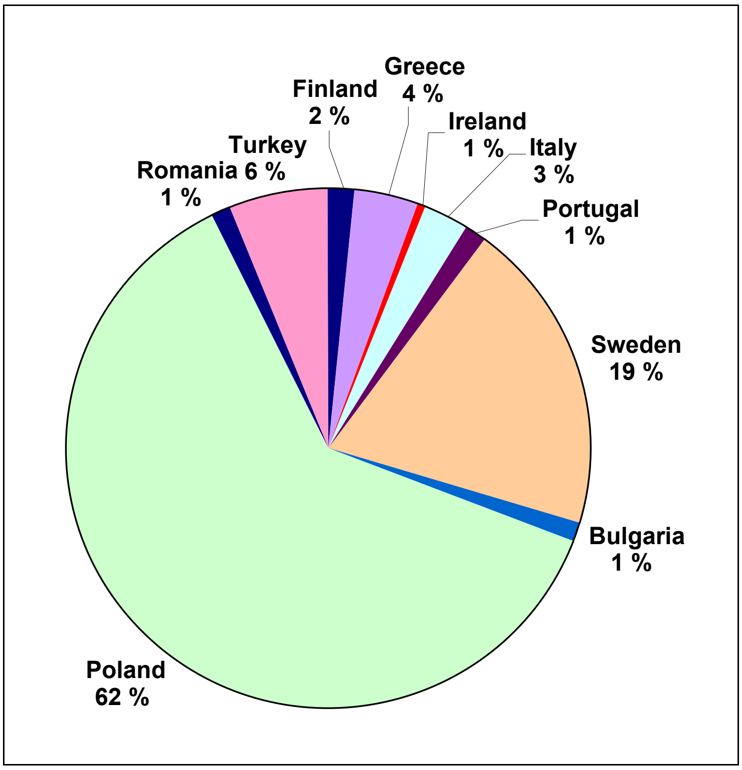


Figure 1.11: Silver mining production in Europe in 1999

Currently there are six gold mines in the EU-15. In Europe, silver is not mined as a main product. Silver is primarily a by-product of lead mining.

A new gold mine in Turkey has been in operation since 2001.

There are several examples of projects where the permitting process has been initiated, e.g. the Svartliden mine in northern Sweden, the Matalikais mine in Greece and the Rosia Montana open pit gold mine project in Romania.

The following figure shows the world gold mining production in 2001.

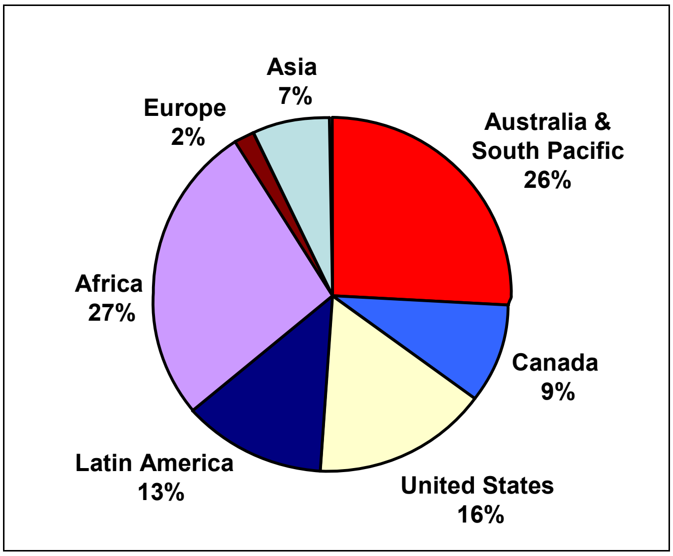


Figure 1.12: World gold mining production in 2001

The use of cyanide (CN) to leach gold has been a much discussed issue in recent years. The Baia Mare accident brought special attention to this technique. In 2000, there were about 875 gold or gold and silver mining operations in the world. This number does not include the contribution from base metal mines where some gold is recovered as a side product at the mine or the smelter. Of those 875 sites, 460 (i.e. 52 %) utilised cyanide, 15 % of them were heap leaches and 37 % used cyanidation in tank leaching. The remaining 48 % used a variety of processes that did not use alternative chemical reagents or lixiviants, but instead used primarily gravity separation and flotation to form a concentrate. These concentrates were then sent to a smelter for final processing [26, Mudder, 2000]. The following figure shows the world distribution of gold or gold and silver mines using cyanidation in 2000.

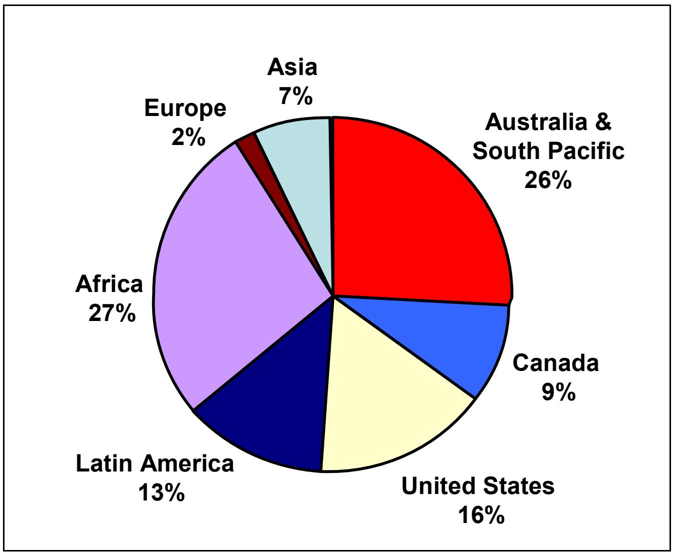


Figure 1.13: World distribution of gold or gold and silver mines using cyanidation in 2000 [26, Mudder, 2000]

During the first nine months of 2001, the Engelhard Corporation's daily price of gold ranged from a low of about USD 257 per troy ounce in April to a high of almost USD 294 in September. For most of the year, this price range was below USD 270. The traditional role of gold as a store of value was able to lift the price of gold out of its low trading range when terrorists attacked the United States in September 2001. In 2001, the Swiss National Bank continued selling 1300 tonnes of gold (one-half of its reserves), and the United Kingdom government completed its drive to sell 415 tonnes of gold from British gold reserves. Concerns about the true position of central bank gold sales, prospects for more consolidations within the gold mining sector, and a lack of renewed investor interest in gold, kept gold prices depressed until the middle of September 2001. Throughout 2002, gold traded steadily USD 300 per ounce.

Gold is a very valuable natural resource. Therefore, it is still worth mining if the ore grade is in the grams/tonne-range. This results in large amounts of tailings being produced in gold mining relative to the amount of gold produced. For instance, at a gold grade of 5 g/t, 200000 tonnes of ore have to be mined to produce 1 tonne of gold (assuming 100 % recovery of gold).

Coarser gold particles can be recovered using gravity separation. However the finer gold particles can often only be recovered by leaching the ore with a cyanide solution. Due to the high toxicity of cyanide, special attention has to be given to the tailings management where this process is applied.

There is research ongoing with the aim of replacing cyanidation with less hazardous techniques. Also new techniques to destroy cyanide in the tailings or to recycle cyanide from the tailings to the process are currently being investigated.
Gold mining tailings are usually in the form of fine slurry which is managed in ponds. All sites within the EU-15 and the Turkish Ovacik mine destroy the cyanide in the tailings prior to discharge into the tailings pond. Both chemical and physical stability of tailings management facilities are of high importance, since the tailings can also have an ARD potential.

1.1.8 Tungsten

The main tungsten bearing minerals are wolframite (Fe, Mn)WO_4 and scheelite ($CaWO_4$).

In 1999, a total of 3000 tonnes of tungsten oxide were produced in Europe. 1800 tonnes WO_3 resulted from Austria and 549 tonnes from Portugal. European production accounted for 11.5 % of the world production in 1999.

The average worldwide consumption of tungsten is 40000 t (W) per year. The main producers are China (>80 %), Canada, Russia, Austria, Portugal and Bolivia [52, Tungsten group, 2002].

Due to low prices, many mines throughout the world have had to close during the last two decades [52, Tungsten group, 2002].

The coarse tailings are managed on heaps; fine tailings in ponds. Depending on the deposit, sulphides are present in smaller or larger quantities; therefore ARD may be an issue.

1.2 Industry overview industrial minerals

For the detailed discussions this sector is divided into different sub-sectors. These are:

- barytes
- borates
- feldspar
- fluorspar
- kaolin
- limestone
- phosphate
- strontium
- talc.

The following table shows that, for most of these minerals, European production, other than metalliferous minerals, presents a major fraction of the world production.

Commodity	Percentage of world production (%)
Barytes	11
Borates	30
Feldspar	64
Fluorspar	5
Kaolin	18
Phosphate	1
Talc	26

Table 1.3: Production of some industrial minerals within Europe as a percentage of world production in 1999

Industrial minerals are recovered in many different ways. Some are sold as mined, i.e. without being processed. In other cases all sorts of mineral processing methods have to be applied to achieve a highly concentrated product. The majority of the mines in the 'Industrial Minerals' sector use only physical treatments (e.g. crushing, washing, magnetic separation, optical sorting, hand sorting, classification, flotation), with only a minority of mines carrying out a chemical treatment of the mineral (e.g. leaching). Hence, the amounts and characteristics of tailings and waste-rock vary significantly. In general these operations are small compared to most metal mines, and the grade of the mineral is usually higher. Therefore, in most cases the amounts of waste-rock and tailings are also smaller. Acid rock drainage is typically not an issue in the industrial minerals sector.

1.2.1 Barytes

Barytes is the naturally occurring mineral form of barium sulphate ($BaSO_4$). It is a relatively low-value industrial mineral. Filler applications can command higher prices after more intense mineral processing. There are also premiums for colour – whiteness and brightness [29, Barytes, 2002].

The EU-15 consumption of barytes is estimated to be around 700000 tonnes, with EU-15 mined production around 340000 tonnes in 2000 and the balance being imported, mainly from China but also from Morocco and India [29, Barytes, 2002].

The following figure shows the main producing countries Europe. The total annual production in Europe is about 715000 tonnes.

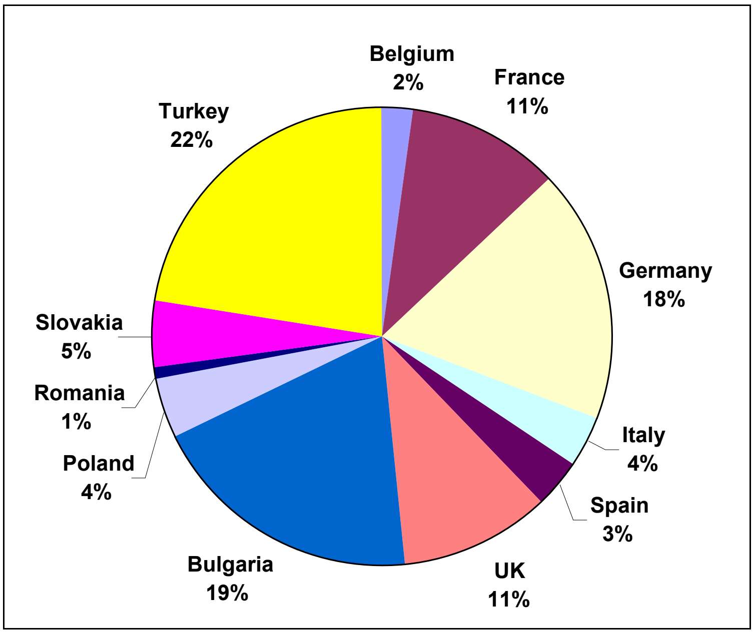


Figure 1.14: Barytes mining production in Europe in 2000

Of the total 6.4 million tonnes production, the US consumed some 2.7 million tonne and EU-15 an estimated 0.7 million tonnes. The following figure shows the main producers in the world.

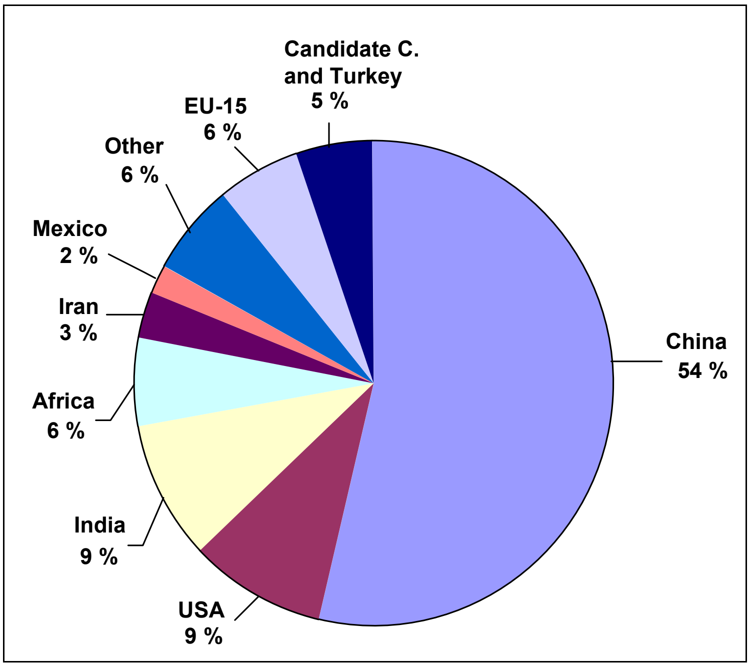


Figure 1.15: World barytes production (production figures) in 2000

Furthermore, imported barytes is processed in the Netherlands.

Quoted prices (*Industrial Minerals* magazine) for oil-well crushed lump are around EUR 55 - 60/tonne rising to EUR 100/tonne for ground material. The mined production output in Europe has remained steady for several years; and provides direct employment for over 400 people and directly contributes over EUR 50 million to the gross domestic product [29, Barytes, 2002].

The average grade of ores mined in the EU-15 is around 50 % $BaSO_4$. This indicates, that to produce 715000 tonnes of barites, about 1400000 tonnes of ore has to be mined. Some of this ore has been sold as other mineral products [29, Barytes, 2002].

Only a small percentage (2 %) of the tailings produced within the EU-15 is discarded as slurry in ponds. Typically coarse tailings are sold as aggregates. Finer tailings are mostly dewatered and also sold or used as backfill in the mine.

1.2.2 Borates

Borates are a group of over 200 naturally-occurring minerals containing boron. Trace amounts exist in rock, soil and water. Elemental boron does not occur in nature but traces of its salts are present almost everywhere in rocks, soil and water. Nevertheless, borate minerals are comparatively rare and large deposits exist in only a few places in the earth's crust (Turkey, US, China, Russia, and South America).
[92, EBA, 2002]

The global supply market for borates, some 4.2 million tonnes, is largely dominated by Turkey (the only European producer), the US, and South America (Argentina, Bolivia, Chile, and Peru). China and Russia produce significant volumes of borates, but export little on to the world market. The world's two leading producers are Eti Bor, which produces in western Turkey, and US Borax in California, which together command perhaps some 75 – 80 % of the supply market.

The Turkish borate producer has an annual production of about 1.2 million tonnes from nine operations (seven open pits, and two underground mines). This represents about 30 % of the world production [36, USGS, 2002].

The Turkish borates industry provides direct employment for over 2150 people and directly contributes over EUR 225 Million to the country's gross domestic product. Quoted prices (Industrial Minerals magazine) for borates range from EUR 270 to 900 per tonne.

In Turkey, the residues of the mines are the tailings from the minerals processing plants and the boron derivatives plants. The tailings are disposed of either on heaps (for coarse clays and calcareous minerals) or in lined tailings ponds (for fine clay particles) near the mines.

1.2.3 Feldspar

Feldspars are common rock-forming minerals, which can become valuable industrial raw materials when occurring in large, easily extractable and processable quantities. By composition, feldspars are aluminosilicates containing potassium, sodium and/or calcium.

More than 60 % of the feldspars produced in the EU-15 are used in the ceramic industry, with most of the remainder being used in glass production. In the manufacture of ceramics, feldspar is the second most important ingredient after clay, acting functionally as a flux.
[39, IMA, 2002].

The feldspar sector is composed of small and medium companies, spread around all EU-15 Members States.

In 1999, a total of 6 million tonnes of feldspar were produced in Europe, which is almost two thirds (64 %) of the total world production. Feldspar recovered by flotation represents about 10 % of the European feldspar production. The following figure shows the major European producers.

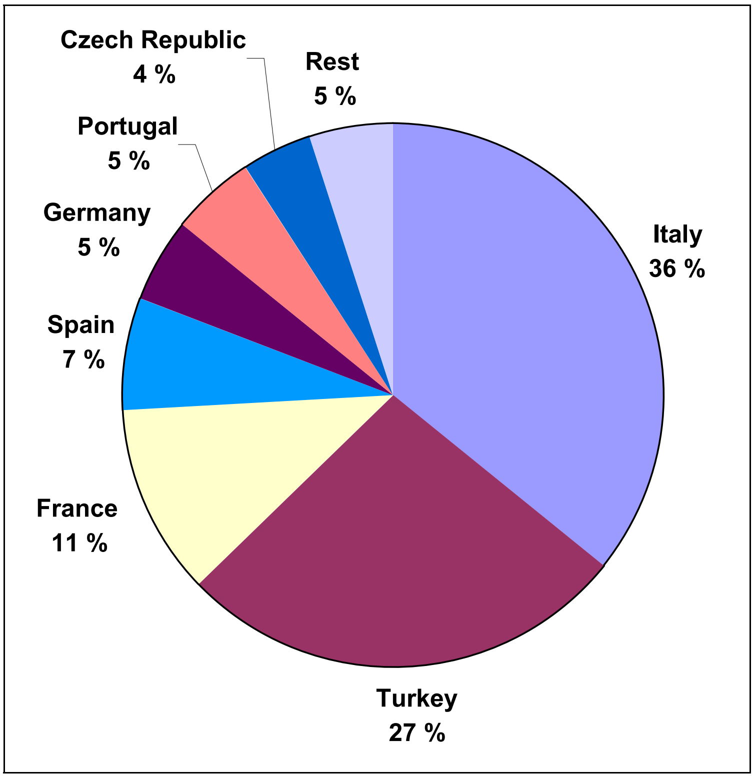


Figure 1.16: Feldspar mining production in Europe in 1999

Minor producers (i.e. <100000 tonnes/yr) include Finland, Greece, Sweden, UK, Poland and Romania.

The feldspar industry in the EU-15 provides direct employment for over 3000 people and directly contributes over EUR 900 million to grass domestic product. The quoted prices (Industrial Minerals magazine) for feldspar are in the range EUR 13 - 205 per tonne. The market for the low-cost sodium-feldspar is mainly local or national because of the proportionally high transport cost. Only a few, higher value feldspars (high grades qualities, i.e. floated feldspar and potassium feldspar) are transported over long distances.

Feldspar production results in tailings heaps made of coarse sand, gravel and rock, as well as tailings ponds for the fine tailings.

1.2.4 Fluorspar

Fluorspar is the industrial name of the mineral fluorite (CaF_2). It is extracted from mines (underground and open pits), with natural concentrations between 20 and 90 % CaF_2. Ore and concentrated marketable products have the same name, i.e. fluorspar. Fluorspar has long been known for the beauty and variety of its colours. Nowadays it is used for its chemical properties (it is a fluoride, and therefore a source of the element fluor) and for its physical properties (e.g. as a fluxing agent).
[43, Sogerem, 2002]

Worldwide production is between four and five million tonnes per year. The main producers are China (2.5 million tonnes), Mexico (0.5 million tonnes), the EU-15 (0.4 million tonnes), and South Africa (0.3 million tonnes). Around 20 countries declared a substantial production in 2000 [43, Sogerem, 2002]. The European producers are displayed in the following figure.

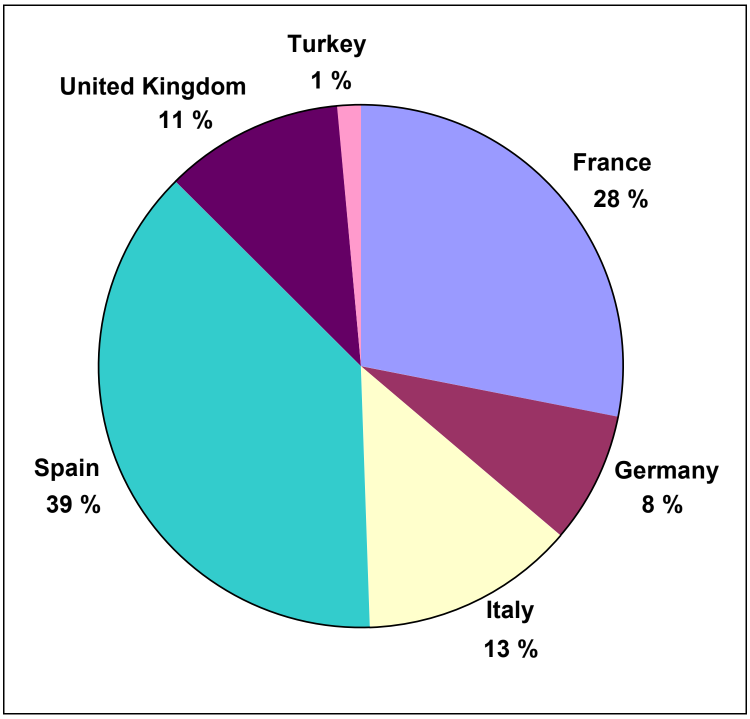


Figure 1.17: Fluorspar mining production in Europe (1999)

At the Sardinian fluorspar/lead sulphide mine the average value of the products are EUR 120 per tonne for fluorspar and USD 190 per tonne for lead sulphide [44, Italy, 2002].

1.2.5 Kaolin

The word kaolin derives from the Chinese "Kao-ling" (High Crest), the name of a hill in central China near where this substance was originally mined for use in ceramics. This is also the origin of the name "China Clay". Since those early days, the use of kaolin has widened to paper, rubber, paints and plastics manufacture [40, IMA, 2002].

In 1999, European kaolin production was about five million tonnes, about 20 % of the world production in the same year. The biggest European producers are listed in the following figure.

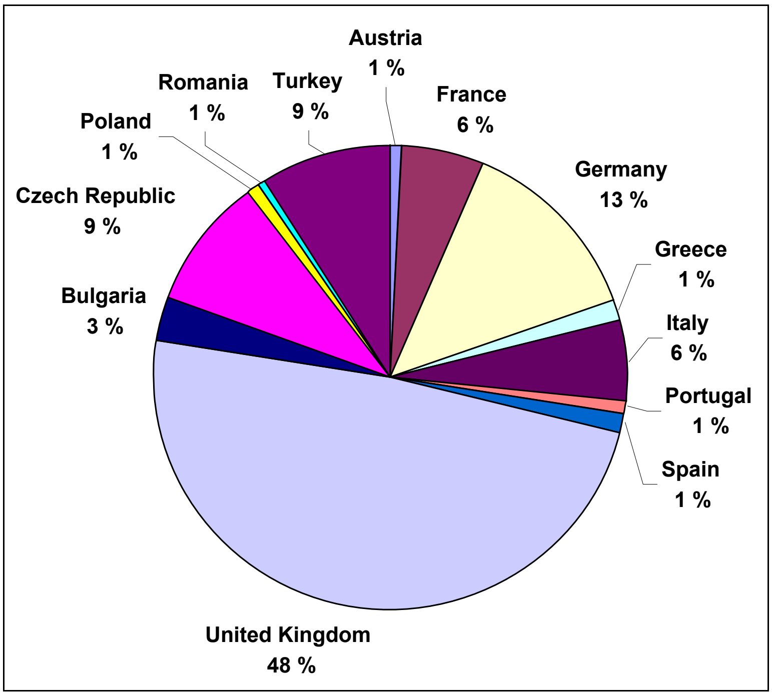


Figure 1.18: Kaolin production in Europe in 1999

In Europe, the kaolin industry provides direct employment for over 6000 people and directly contributes over EUR 1500 million to gross domestic product. The quoted prices (Industrial Minerals magazine) for kaolin are in the range EUR 40 - 375 per tonne.

Kaolin production results in tailings heaps made of coarse sand, gravel and rock, as well as tailings ponds for the fine tailings.

1.2.6 Limestone

Limestone is used in three different ways: as an aggregate, as calcium carbonate and in the cement and lime industry. The aggregates sector will not be discussed, since it does not generate tailings.

The calcium carbonate industry operates mainly with deposits of a grade higher than 96 %. Therefore, there is usually no need for further mineral processing steps. In Europe, only seven plants need to use flotation to separate calcium carbonate from unwanted minerals (mainly graphite and mica). These seven plants account for less than 5 % of the total European calcium carbonate production. Five of these plants do not have tailings ponds, since they use dewatering devices (e.g. thickening and filter press).
[42, IMA, 2002]

The limestone used for the cement and lime sector contains clay impurities that can be washed off. These tailings are stored in ponds.

1.2.7 Phosphate

The only phosphate mine in Europe is the Finnish Siilinjärvi mine. Currently its annual production levels are 800000 tonnes of apatite concentrate ($Ca_5(PO_4)_3$ (F), calcium fluoro phosphate). The main product, the apatite concentrate, is mainly used as a raw material for phosphoric acid production.

Furthermore, 100000 tonnes of calcite concentrate, 10000 tonnes of mica concentrate, 70000 tonnes of micaceous products and 200000 - 300000 tonnes of various crushed rock products are produced annually.

Some nine million tonnes of ore and two to three million tonnes of waste rock are extracted annually.

Tailings from the concentrator are pumped to the tailings dam area. Waste-rock is crushed for it to be used as aggregate in road and dam constructions or stockpiled in waste-rock areas.
[143, Siirama, 2003]

1.2.8 Strontium

Strontium is commonly mined in the form of two minerals, celestite (strontium sulphate) and strontianite (strontium carbonate). Of the two, celestite occurs much more frequently in sedimentary deposits of sufficient size to make development of mining facilities attractive. Strontianite would be the more useful of the two common minerals because strontium is used most often in the carbonate form, but few deposits have been discovered that are suitable for development.
[36, USGS, 2002]

Celestine ($SrSO_4$), is mined in two mines in southern Spain, which together produced about 120000 tonnes of final product in 2000. The other European producer of Strontium ore is Turkey with about 25000 tonnes in the same year. World production in 2000 amounted to about 300000 tonnes. All figures are given in metric tonnes of strontium content. Spain is the second largest producer in the world after Mexico.
[36, USGS, 2002]

1.2.9 Talc

Talc is a hydrated magnesium silicate. Although talc deposits are found throughout the world in various geological contexts, economically viable concentrations of talc are not that common.

The largest producer in the world is China with an annual production of about 1.7 million tonnes followed by the US (0.9 million tonnes) and India (0.6 million tonnes). EU talc production stands at 1.4 million tonnes/year, of which France and Finland account for 70 %. The world talc production is estimated to about 5 million tonnes/year.

The following figure shows the producing countries in current Member States and Candidate Countries. It is difficult to obtain sensible talc production data as it is often grouped with steatite and talc-related materials.

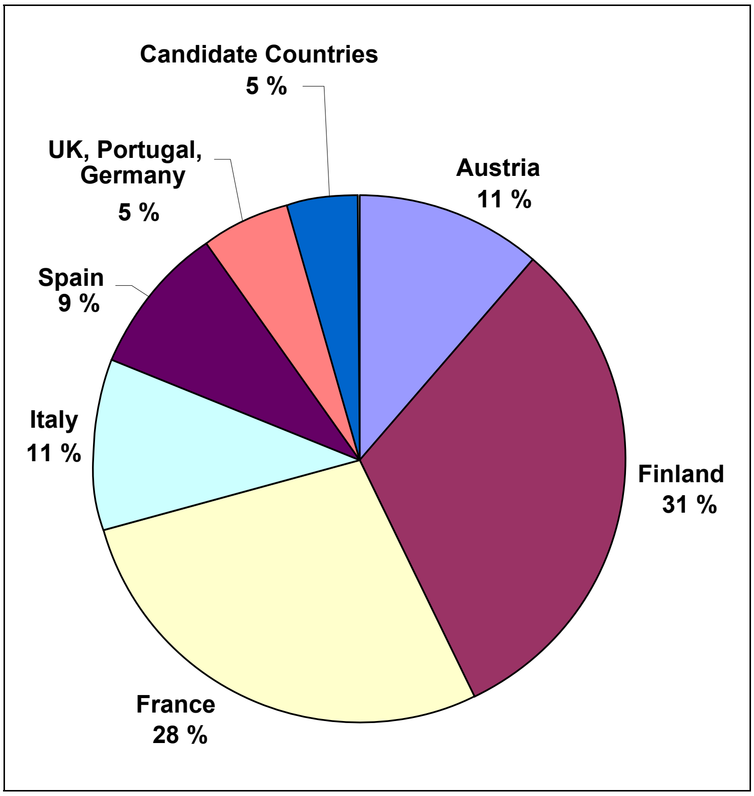


Figure 1.19: Talc mine production in Europe (1999)

Luzenac is the major producer on the European market. The two other main producers are Mondo Minerals and IMI Fabi SpA. Luzenac, which belongs to the group Rio Tinto, is the leading talc producer with sales exceeding 1.4 million tonnes/year. In Europe, Luzenac owns 7 talc deposits and 11 processing plants. Mondo Minerals incorporates the European talc activities of Mondo Minerals Oy (two mines and three processing plants in Finland), Mondo Minerals B.V. in the Netherlands and Norwegian Talc AS. IMI Fabi SpA has its main activities in Italy, with three mines and two comminution plants.

The talc market is undergoing consolidation on both the supply and demand side in response to increasing competition from other minerals and emerging economies, and due to increased market transparency and globalisation pressures. The talc market in Europe is mature with low growth in most sectors, so for many years price increases have been marginal, barely keeping abreast with inflation. Domestic markets are also increasingly subject to pressure from competitively priced imports of high quality grades which can command a price premium, especially from China. Talc's properties (platyness, softness, hydro-phobicity, organophilicity, inertness and mineralogical composition) provide specific functions in many industries. Quoted prices (Industrial minerals magazine) for talc varies from USD 100 - 300 per tonne, depending on the grade, with an average price of USD 210 per tonne. The world market is thus estimated to be USD 1.2 billion per year.

Generally, due to the high purity of the deposit, the talc industry does not generate tailings.

However, in Finnish operations, which actually represent about 33 % of the European talc production, talc is extracted from a talc magnesite rock using flotation. The tailings are managed in ponds.

1.3 Industry overview: potash

Even though potash is an industrial mineral, it was decided by the TWG at the kick-off meeting, that due to the different techniques in the mineral processing and tailings management this mineral would be treated separately in its own section.

The main potash products used as fertilisers (with the nutrients potassium, sulphur and magnesium) are potassium chloride (MOP[4]), potassium sulphate (SOP) and kieserite. These are produced with different values of K_2O[5] (40 - 62 %) and in a fine, standard or coarse grade. Potassium sulphate and sulphates of potash-magnesia are non-chloride potash fertilisers.

About one fifth of the world potash production comes from European mines in France, Germany, Spain and the UK.

The European mine production in 1999 was just over 5 million tonnes K_2O. The following figure shows the production percentages by country.

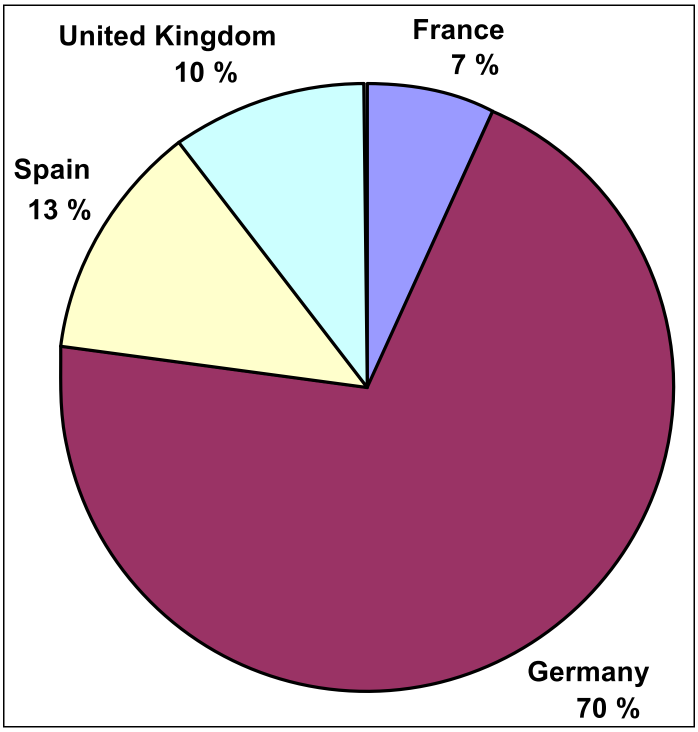


Figure 1.20: Potash mining production (K_2O) in Europe in 1999

The world potash production is dominated by Canada, Russia and Germany, which together account for about 76 % of the total world production. Potassium chloride (KCl), commonly referred to as Muriate of Potash (MOP), is the most common and least expensive source of potash. Potassium chloride accounts for about 95 % of world potash production.
[19, K+S, 2002]

The world potash industry has experienced unstable conditions since the late 1980s (just prior to the economic collapse of the Eastern block countries). Up to that point, average industry operating rates (percentage of production capacity) were exhibiting a slow but steady upward

[4] Muriate of potash (MOP) is the common term for the salt potassium chloride (KCl) and is so named because hydrochloric acid was originally called muriatic acid. The name MOP has stuck with the product even though the name of the acid has since changed.

[5] Potassium oxide is not known to exist owing to its highly reactive properties. However it is used as a convention term for stating the potassium content of a material, e.g. 100 tonnes 95 % KCl (MOP) is the equivalent of 60 tonnes K_2O.

trend that ended abruptly in 1988. The average world operation rate, which had steadily increased to 83 % in 1988, declined gradually to only 56 % of previous levels. During this period, world consumption declined from 31 to 21 million tonnes K_2O.

World potash demand in 2000 was approximately 26 million tonnes of potassium oxide (K_2O) or 42 million tonnes of product (KCl and K_2SO_4). Compared with these figures, the manufacturing capacity was approximately 37 million tonnes of potassium oxide (K_2O) or 59 million tonnes of products. Therefore, a considerable overcapacity exists worldwide.

The economic situation, particularly in developed countries, greatly influences the extent and regional distribution of exports. Both the quantity exported and its distribution among consumers are greatly affected by the state of the importers agriculture, by the demand for (or availability of) convertible currency in the exporting or importing country and by fluctuations in currency exchange rates. Transport costs for potash fertilisers have a considerable bearing on total cost to the consumer. Logistical considerations, therefore, influence the direction and magnitude of imports or exports and contribute to the worldwide overcapacity.
Five methods are used in Europe for managing the tailings, these are:

- storing solid tailings on tailings heaps
- backfilling solid tailings into mined out rooms of underground works
- discharging solid and liquid tailings into the ocean/sea (e.g. marine tailings management)
- discharging liquid tailings into deep wells
- discharging liquid tailings into natural flowing waters (e.g. rivers).

Potash tailings are made up of table salt (sodium chloride) together with a few per cent of other salts (e.g. chlorides and sulphates of potassium, magnesium and calcium) and insoluble materials such as clay and anhydrite. The tailings heaps themselves generate saline solutions when atmospheric precipitation dissolves salt from the tailings material.

1.4 Industry overview: coal

The TWG decided at the kick-off meeting that coal is only included when it is processed and there are tailings produced. Therefore, in this section, only hard coal (or rock coal or black coal) is discussed, whereas lignite (or brown coal), which is usually not processed, is not covered.

Throughout Europe coal is mined under difficult geological conditions, usually underground. The industry is characterised by a high degree of automation. The production from coal mines in the EU-15 has been declining for decades. This is due to the often high cost caused by mining deep-lying and relatively slim deposits, called seams. However, with the accession of new members, overall coal production in the EU will increase. In other parts of the world, large deposits close to the surface can be mined at lower costs. Coal mines in Europe will keep closing. The opening of new underground mines is not foreseeable in the near future. Except for Spain and the UK, where some four million tonnes per year of bituminous coal are mined from open pits, coal is usually extracted by means of underground operations.

As can be seen in the following table, the total hard coal production in Europe in 2001 was 188.2 million tonnes. It can be seen that Poland is the dominant European hard coal producer, providing over 50 % of the total European production in 2001.

Country	1980	1996	1997	1998	1999	2000	2001
France	20194	7314	5779	4739	4033	3166	1971
Germany	94492	53156	51212	45340	43849	37338	30362
Spain	13147	17465	18861	16380	15433	14965	14539
United Kingdom	130096	49307	47123	40045	36356	30465	32512
Total EU-15	**257929**	**127242**	**122975**	**106504**	**99671**	**85934**	**79384**
Bulgaria	267	186	99	118	108	66	20
Czech Republic	288	301	301	301	300	631	630
Hungary	3065	996	959	914	783	754	570
Poland	193121	136385	137100	116381	110443	103173	103896
Romania	8060	4219	3401	2679	2748	3243	3680
Turkey	3602	3029	2291	3994	2705	3110	3719
Total Cand. Countr. and Turkey	**208403**	**145116**	**144151**	**124387**	**117087**	**110977**	**112515**
Total Europe	**466332**	**272358**	**267126**	**230891**	**216758**	**196911**	**191899**
World	**2728475**	**3818221**	**3833233**	**3789727**	**3505000**	**3447248**	**3408945**
Europe as % of world	17 %	7 %	7 %	6 %	6 %	6 %	6 %

Table 1.4: Coal production figures in kt, 1980, 1996-2001
[111, DSK, 2002]

The table highlights the declining production in most European countries, the most radical examples being Germany, France and the UK. In Germany, by the end of 2000, there were only 12 mines left in production (1990: 27 mines, 1980: 39 mines, 1973: 53 mines, 1957: 173 mines).

In the UK, which is the biggest coal producer in the EU-15, there were an average of 41 open pit and 22 underground mines producing at any one time during the year 2002. 15 million tonnes of the UK production originates from open pit (or opencast) sites.

Hard coal in the Czech Republic mainly occurs in the Upper Silesian Basin. Regarding the coal resources within this region, about 15 % are in the Czech Republic, and the balance are in Poland.
[83, Kribek, 2002]

In many cases European production costs are several times the world average. Some mines, even though they cannot compete in the world market, are still in production only because they receive subsidies. However, in the UK, coal mining is in the main competitor with world coal. In 2001, 15 million tonnes of surface mined coal was produced and purchased by the electrical supply industry in competition with imported coals. No subsidies were paid to produce this coal. 17 million tonnes of deep mined coal was produced, again in the main, without subsidies. ‘Selective Operating Aid’ of some GBP 65 million was paid in 2001 to specific mines to enable them to achieve long-term viability and to compete long-term with imported coal.

The tailings from coal mining are the coarse tailings, which are managed on heaps, and flotation slurries, which are either discharged into ponds or, after filtering, onto heaps. The ponds may be small settling basins, which need to be dug out periodically. In other cases coal tailings ponds can cover tens of hectares and may be contained by tailings dams. Coal tailings can contain pyrite and flotation reagents.

Efforts have been made to use coal tailings as construction materials. Due to their low permeability dried flotation fines can also be used as liners for landfills.

Waste-rock is produced by open pit mining and is used to restore the site during extraction (by progressively restoring the coaled out areas) and on completion to produce a satisfactory landform. Waste-rock is also produced in underground mining from driveages etc. and then either remains underground or is stored in spoil heaps above ground.

1.5 European mine and mine waste production

The following tables show the production from European countries. The figures are expressed as percentages of total European production. The numbers used in these two tables are the same as used throughout Sections 1.1 to 1.4. However, these tables allow an easier overview of all the sectors. This table also makes it easier to compare the production figures of different countries.

	FERROUS METALS	NON-FERROUS METALS											PRECIOUS METALS	
	Iron	*Alumina*[1]	*Cadmium*	*Chromium*	*Copper*	*Lead*	*Manganese*	*Mercury*	*Nickel*	*Tin*	*Tungsten*	*Zinc*	*Gold*	*Silver*
	(%)	*(%)*	*(%)*	*(%)*	*(%)*	*(%)*	*(%)*	*(%)*	*(%)*	*(%)*	*(%)*	*(%)*	*(%)*	*(%)*
Austria	3	-	-	-	-	-	-	-	-	-	63	-	-	-
Belgium	-	-	-	-	-	-	-	-	-	-	-	-	-	-
Finland	-	-	29[3]	36	1	-	-	19[3]	7	-	-	2	17	2
France	-	9	8	-	-	-	-	-	-	-	-	-	16	-
Germany	-	12	-	-	-	-	-	-	-	-	-	-	-	-
Greece	-	10	-	-	-	4	2	-	93	-	-	2	-	4
Ireland	-	23	-	-	-	11	-	-	-	-	-	23	-	1
Italy	-	15	16	-	-	2	2	-	-	-	-	1	9	3
Portugal	-	-	-	-	11	-	-	-	-	96	37	-	-	1
Spain	-	18	8	-	3	17	-	81	-	-	-	22	22	-
Sweden	74	1	-	-	8	33	-	-	-	-	-	20	23	19
United Kingdom	-	-	18	-	-	-	-	-	-	4	-	-	-	-
Total EU-15 (t)[2]	12816129	5970000	1900	248149	204749	236646	1972	291	14483	2264	3215	616868	16.27	525.46
Bulgaria	2	-	11	-	13	11	39	-	-	-	-	2	6	1
Cyprus	-	-	-	-	1	-	-	-	-	-	-	-	-	-
Czech Republic	-	-	-	-	-	-	-	-	-	-	-	-	-	-
Estonia	-	-	-	-	-	-	-	-	-	-	-	-	-	-
Hungary	-	4	-	-	-	-	26	-	-	-	-	-	-	-
Latvia	-	-	-	-	-	-	-	-	-	-	-	-	-	-
Lithuania	-	-	-	-	-	-	-	-	-	-	-	-	-	-
Malta	-	-	-	-	-	-	-	-	-	-	-	-	-	-
Poland	-	-	8	-	56	18	-	-	-	-	-	20	2	62
Romania	-	8	-	-	2	2	31	-	-	-	-	3	3	1
Slovakia	2	-	-	-	-	-	-	-	-	-	-	-	2	0
Slovenia	-	-	-	-	-	-	-	-	-	-	-	-	-	-
Turkey	19	-	2	64	6	2	-	-	-	-	-	5	-	6
Total Acceeding Countries, Accession Countries and Turkey (t)	3945719	830000	500	433658	684066	114074	41372	-	-	-	-	273995	2.16	1244.09
Total EUROPE (t)	16761848	6800000	2400	681807	888815	350720	43344	291	14483	2264	3215	890863	18.43	1769.54
World (t)	556777376	53000000	16495	5777378	12364823	3340792	9595182	1673	1071425	228767	28015	7533028	2432.46	17293.21
EUROPE as % of world	3.0	12.8	14.5	11.8	7.2	10.5	0.5	17.4	1.4	1.0	11.5	11.8	0.8	10.2

1) year 2001
2) EU-15 Member States not listed do not produce any of these minerals
3) These figures include the Hg and Cd metallurgical production from imported ore. The figures for Finnish mine production are Cd: 2.5 %, Hg: 1 %.

Table 1.5: European mine production expressed in % of total European production of ferrous, non-ferrous and precious metals in 1999 (unless otherwise indicated)

	INDUSTRIAL MINERALS										COAL
	Barytes	*Boron*	*Feldspar*	*Fluorspar*	*Kaolin*	*Limestone*	*Phosphate*	*Potash*	*Strontianite*	*Talc*	*Hard-Coal*
	(%)	*(%)*	*(%)*	*(%)*	*(%)*	*(%)*	*(%)*	*(%)*	*(%)*	*(%)*	*(%)*
Austria	-	-	-	-	1	-	-	-	-	10	
Belgium	2	-	-	-	-	-	-	-	-	-	
Finland	-	-	1	-	-	-	100	-	-	35	
France	11	-	11	28	6	-	-	7	-	28	
Germany	18	-	5	8	13	-	-	70	-	2	
Greece	-	-	1	-	1	-	-	-	-	-	
Ireland	-	-	-	-	-	-	-	-	-	-	
Italy	4	-	36	13	6	-	-	-	-	10	
Portugal	-	-	5	-	1	-	-	-	-	2	
Spain	3	-	7	38	1	100	-	13	83	8	
Sweden	-	-	1	-	-	-	-	-	-	-	
United Kingdom	11	-	-	11	48	-	-	10	-	2	see Table 1.4
Total EU-15[1]	322762	-	3927357	350176	3906168	2000000	734068	5066880	10590	1260000	
Bulgaria	19	-	-	-	3	-	-	-	-	Candidate Countries: 4	
Cyprus	-	-	-	-	-	-	-	-	-		
Czech Republic	-	-	4	-	9	-	-	-	-		
Estonia	-	-	-	-	-	-	-	-	-		
Hungary	-	-	1	-	-	-	-	-	-		
Latvia	-	-	-	-	-	-	-	-	-		
Lithuania	-	-	-	-	-	-	-	-	-		
Malta	-	-	-	-	-	-	-	-	-		
Poland	4	-	1	-	1	-	-	-	-		
Romania	1	-	1	-	1	-	-	-	-		
Slovakia	5	-	-	-	-	-	-	-	-		
Slovenia	-	-	-	-	-	-	-	-	-		
Turkey	22	100	28	1	9	-	-	-	17		
Total Acceeding Countries, Accession Countries and Turkey (t)	344327	1242228	2004473	4812	1152811	-	-	-	-	60000	
Total EUROPE	666999	1242228	5931830	354988	4773774	2000000	734068	5066880	145000	1442000	
World	6326531	4200000	8950309	4612569	25982207	n/a	67040137	24665640	300000	5620000	
EUROPE as % of world	10.5	29.6	66.3	7.7	18.4	n/a	1.1	20.5	48.3	25.7	

1) EU-15 Member States not listed do not produce any of these minerals

Table 1.6: European mine production expressed in percentage of total European production of industrial minerals and coal in 1999 (unless otherwise indicated)

According to the Eurostat yearbook for 2003, in the EU-15 the following amounts of waste are generated.

Country and reference year	Agriculture and forest (kt)	Mining and quarrying (kt)	Manufacturing industry (kt)	Energy production (kt)	Construction (kt)
Austria 99	0	0	14284	0	25392
Belgium 99	0	619	13779	1287	0
Germany 93	0	67813	65119	25310	131645
Denmark 98	0	0	2783	1469	2962
Spain 99	0	22757	29239	0	22000
Greece 97	7781	3900	6682	9320	1800
France 95	377000	0	101000	0	13700
Finland 99	24000	28000	15910	1274	35000
Italy 97	242	350	22993	0	20587
Ireland 98	64578	3510	5113	450	2704
Netherlands 99	0	333	9779	1546	0
Portugal 99	0	4691	12804	487	63
Sweden 98		63818	19780	0	0
United Kingdom 99	84000	118000	50000	13000	71000

Table 1.7: European waste generation
[139, Eurostat, 2003]

It should be noted that any statistics on mining waste always bear a level of uncertainty, since some ‘residues’ from mining are in some states considered waste and in others they are not.

However, looking at the table above it becomes clear that waste from mining and quarrying represents a significant amount of the total waste generated in the EU-15, i.e. about 20 %.

1.6 Key environmental issues

Proper material characterisation is the basis for successful tailings and waste-rock management. The management of tailings and waste-rock is one part of the entire mining operation, which naturally also includes the actual extraction and the mineral processing stage. Not only do these other parts of the operation influence the management of tailings and waste-rock, in reality the methods of mining and mineral processing actually determine the management and not vice versa.

Tailings and waste-rock management sites go through certain phases from design to after care. It is essential to manage these facilities in a way that makes most sense in all phases of that life cycle.

Another important issue to consider is the adaptation to changes in reality. An example here may be that after 10 years of operation, the sulphide content in the waste-rock coming from the mine could increase to such a level that Acid Rock Drainage (ARD) could become an issue. To avoid this becoming a problem in the longer term, care needs to be taken during the operational phase by possibly mixing this waste-rock with other waste-rock containing buffering minerals or by separately depositing material with ARD potential in an adequate way. In the given example, it would be necessary to project any findings during operation to steps much further down the ‘life cycle road’ and to then act accordingly to achieve the best overall long-term environmental and economic benefit.

Within the mining industry environmental awareness has improved considerably over the last decades. Thus historical operations with large environmental impact cannot be regarded as representative for the prevailing modern management of waste-rock and tailings. A significant improvement has also been achieved concerning the legislative framework, permitting requirements and control. In reality, what this all means is that now the entire life cycle of the mine is considered at all times and the closure of the mine is planned and provided for in an environmentally acceptable way, even before the mine is opened.

1.6.1 Site location

Mining is a unique sector in so much as primarily geology determines the location of the mine. This is a major difference to other industries. An ore can only be mined where the deposit is. Of course the choice of mining method and the exact location of shafts and other infrastructure still have to be made.

The degrees of freedom in terms of choice of location increases the further downstream one goes in the process. The location for the extraction itself is predetermined as mentioned above. Typically, the mineral processing though, is undertaken as close to the actual mine site as possible, due to the often low grade of the ore, which implies that the ore value cannot cover high transport costs. However, this is not true in all cases and in some cases the ore is processed many thousands of kilometres away from the mine. For instance, for bauxite the processing into aluminium is very energy demanding and the transport cost of the ore can be recovered by lower energy costs for the processing in a different location (often though some pre-refining will still be done at the site).

For the management of tailings and waste-rock, the degrees of freedom concerning location is in general again increasing, but as with the mineral processing, it is generally preferable to limit or reduce the transportation cost. However, in many cases tailings are pumped or trucked many kilometres to an appropriate site for deposition.

When it comes to the selection of a tailings and/or waste-rock management site many other factors have to be considered, such as:

- preferable use of existing geographic formations (e.g. existing pits or slopes)
- need to respect the hydrogeological setting of the surrounding area (ground- and surface water)
- adaptation of facility to surrounding area (e.g. dust, noise and odour control if there is a residential population nearby)
- meteorology (e.g. rainfall data)
- geotechnical and geological background (e.g. foundation conditions, seismic risk data)
- natural and cultural environment
- relationship of tailings facility to underground operation
- topography of long-term construction
- proximity to surface water
- proximity to the coast (seawater)
- existing land-use
- local communities
- biodiversity.

Underwater deposition, which is often carried out for tailings with an ARD potential, involves a different set of issues, such as secure surface water supply, a natural or constructed basin, post-deposition use of area, etc.

The proximity to surface water is often a complex issue. On the one hand, if a discharge to surface water is required it is preferable to have the river ‘next door’. On the other hand, it needs to be assessed if this surface water would act as the ideal transport medium of tailings in the case of an accidental release.

In general, a balance has to be maintained between the proximity of the tailings or waste-rock management site to the mineral processing site for economical reasons and other factors such as those listed above. In reality, often the site investigation will result in several 'candidate locations'. The actual decision is then made in the permitting process, often as a compromise between the operator, the permit writers and public concerns.

1.6.2 Material characterisation including prediction of long-term behaviour

The only way of determining the long-term behaviour of tailings and waste-rock is to characterise them properly. This may sound trivial, but it has often been neglected in the past. Too often the focus has been on the saleable concentrate, which generates revenue and not on the remaining residue. However, operators should not forget the negative economic effect that improper tailings and waste-rock management can incur.

From an environmental point of view, the main difference between the mineral in the original deposit and the same mineral, less as much as possible of the desired mineral, in the tailings and waste-rock is the increased availability for physical, chemical and biological processes to affect the mineral. This means that through the treatment of the ore (mainly comminution) the constituents of the tailings and waste-rock are more accessible. The following two examples may further explain this phenomenon:

Sulphide ore, in its natural location (i.e. underground and bound in rock mass), is not exposed to an oxidising environment. The finely ground tailings of this ore, once discarded in a pond, are much more accessible to water and oxygen. The surface area of accessible sulphides is increased by orders of magnitude through the size reduction. This implies that, if not managed properly, the rate of weathering, and thereby the mobilisation of weathering products, may be significantly increased.

Another example is potash ore. These ores consist of potash minerals and rock salt. The deposits are protected from water by impermeable layers (typically of clay and gypsum). The tailings of this same ore, however, consist mainly of rock salt (>90 %) and are typically piled up on heaps. This salt is accessible for precipitation and is washed-off over a long period of time.

Also, the mineral processing of the ore may change the chemical characteristics of the processed mineral and hence the tailings.

Overall, the characteristics that have to be investigated are, e.g.:

- chemical composition, including the change of chemistry through mineral processing and weathering
- leaching behaviour
- physical stability
- behaviour under pressure
- erosion stability
- settling behaviour
- hard pan behaviour (e.g., crust formation on top of the tailings).

Proper material characterisation is the basis for any planning of the management of tailings and waste-rock. Only if this background work is done properly can the most appropriate management measures be applied.

General issues about closure, rehabilitation and after-cares are discussed in Section 2.6. Applied measures are shown in Section 4.2.4.

Each mining operation will have an irreversible impact on the earth's crust. To qualify this impact, baseline studies are carried out to give a point of reference. Baseline studies are described in more detail in Section 4.2.1.1.

1.6.3 Environmentally relevant parameters

The environmentally relevant parameters of tailings and waste-rock management facilities can be subdivided into two categories: (1) operational, and (2) accidental. Both have to be taken into consideration.

During operation the 'typical' emissions to air, water and land have to be considered and techniques to reduce these emissions will be discussed in this document. However, two very important environmental issues which need to be highlighted are:

- the generation of acid rock drainage and
- the occurrence of accidental bursts or collapses.

1.6.3.1 Typical emissions and management of water and reagent

- **Emissions to air** can be dust, odour and noise. Usually the latter two are of less concern unless the tailings or waste-rock are transported with trucks and there is residential housing nearby. Dust can consist of materials such as quartz or any other components found in rocks and minerals, including metals.

- **Emissions to water** can include reagents from mineral processing, such as
  - cyanide
  - xanthates
  - acids or bases resulting in low or high pH
  - solid or dissolved metals or metalliferous compounds (e.g. iron, zinc, aluminium)
  - dissolved salts e.g. NaCl, $Ca(HCO_3)_2$, etc,
  - radioactivity (in coal tailings/waste-rock heaps)
  - chloride (coal mines)
  - suspended solids.

- **Emissions to land** can occur via settled dust or via the seepage of liquids from tailings and/or waste-rock management facilities into the ground. The building and removal of temporary storage piles is one often occurring source of land contamination. This is also true for the construction of industrial areas, railway banks, tailings dams, etc., using waste-rock containing, e.g. ARD producing material.

- **Overall management of water and reagents, such as:**
  - Consumption and treatment and/or recycling of
    - reagents (e.g. flotation reagents, cyanide, flocculants) and
    - water

    prior to discharge into tailings facility or surface water
  - management of precipitation and surface water (e.g. gathering in ditches).

It should be noted that emissions to land are a highly site-specific issue and that there are very few default emission scenarios currently available to characterise these emissions.

1.6.3.2 The environmental impact of emissions

Effluents and dust emitted from tailings and waste-rock management facilities, controlled or uncontrolled, may be toxic in varying degrees to humans, animals and plants. The effluents can be acidic or alkaline, may contain dissolved metals and/or soluble and entrained insoluble complex organic constituents from mineral processing, as well as possibly natural occurring organic substances such as humic and long-chain carboxylic acids from the mining operations. The substances in the emissions, together with their pH, dissolved oxygen, temperature and hardness, may all be important aspects in the toxicity to the receiving environment.

Certain reagents, such as cyanides, frothers and xanthates require long retention time, oxidation (air, bacteria, sunlight) and, for xanthates, temperatures above 30 ºC to decompose. Therefore the planning of the mineral processing circuit and the TMF must consider the environmental impacts of these substances and the potential need for extra ponding or treatment to provide for certain reagents' decomposition.
[21, Ritcey, 1989]

The actual environmental impact of emissions to watercourses always depends on many factors such as concentration, pH, temperature, water hardness etc. However, Ritcey [21, Ritcey, 1989] and many other sources, provide tables listing, e.g.:

- maximum and minimum pH levels for various aquatic life form
- ammonia toxicity data
- acute toxicity data for various flotation agents
- toxicity of specific chemicals
- toxicity data for flocculant and coagulants.

These tables can give an impression of the potential impact of certain reagents, but, as mentioned above, the whole picture has to be taken into consideration.

The following table shows the effects of some metals on humans, animals and plants.

Metal	Effect
Arsenic (As)	Highly poisonous and possibly carcinogenic in humans. Arsenic poisoning can range from chronic to severe and may be cumulative and lethal
Cadmium (Cd)	Cadmium is concentrated in tissue and humans can be poisoned by contaminated food, especially fish. Cd may be linked to renal arterial hypertension and can cause violent nausea. Cd accumulates in liver and kidney tissue. It depresses growth of some crops and is accumulated in plant tissue
Chromium (Cr)	Cr^{+6} is toxic to humans and can induce skin sensitisations. Human tolerance of Cr^{+3} has not been determined
Lead (Pb)	A cumulative body poison in humans and live-stock. Humans may suffer acute or chronic toxicity. Young children are especially susceptible
Mercury (Hg)	Hg and its compounds are highly toxic, esp. to the developing nervous system. The toxicity to humans and other organisms depends on the chemical form, the amount, the pathways of exposure and the vulnerability of the persons exposed
Copper (Cu)	Small amounts are considered non-toxic and necessary for human metabolism. However, large doses may induce vomiting or liver damage. Toxic to fish and aquatic life at low levels
Iron (Fe)	Essentially non-toxic but causes taste problems in water
Manganese (Mn)	Affects water taste and may stain laundry. Toxic to animals at high concentrations
Zinc (Zn)	May affect water taste at high levels. Toxic to some plants and fish

Table 1.8: Effects of some metals on humans, animals and plants
[53, Vick, 1990]

1.6.3.3 Acid rock drainage

The past two decades have brought widespread awareness of a naturally occurring environmental problem in mining known as ‘Acid Rock Drainage’ or ARD. Though difficult to reliably predict and quantify, ARD is associated with sulphide ore bodies mined for Pb, Zn, Cu, Au, and other minerals, including coal. While ARD can be generated from sulphide-bearing pit walls, and underground workings [13, Vick,], only tailings and waste-rock are considered in this document.

The key issues that are the root of these environmental problems are:

- tailings and/or waste-rock often contain metal sulphides
- sulphides oxidise when exposed to oxygen and water
- sulphide oxidation creates an acidic metal-laden leachate
- leachate generation over long periods of time.

Unless otherwise mentioned the following information is from [20, Eriksson, 2002].

<u>The basics of ARD</u>
When sulphide minerals come into contact with water and oxygen they start to oxidise. This is a slow heat generating process (kinetically controlled exothermal process) which is promoted by:

- high oxygen concentration
- high temperature
- low pH
- bacterial activity.

The overall reaction rate for a specified quantity of sulphides is also dependant on other parameters such us, for example, the type of sulphides and the particle size, which also governs the exposed surface area. When the sulphides oxidise they produce sulphate, hydrogen ions and dissolved metals.

Tailings and waste-rock consist of the different natural minerals found in the mined rock. In the unmined rock, often situated deep below the ground level, the reactive minerals are protected from oxidation. In oxygen-free environments, such as in deep groundwater, the sulphide minerals are thermodynamically stable and have low chemical solubility. Deep groundwater in mineralised areas, therefore, often has a low metal content. However, when excavated and brought to the surface, the exposure to atmospheric oxygen starts a series of bio-geo-chemical processes that can lead to production of acid mine drainage. Hence, it is not the content of metal sulphides in itself that is the main concern, but the combined effects of the metal sulphide content and the exposure to atmospheric oxygen. The effect of exposure increases with decreasing grain size and, therefore, increased surface area. Hence the sulphides in the finely ground tailings are more prone to oxidation [14, Höglund, 2001].

Tailings and waste-rock are normally composed of a number of minerals, of which the sulphides only constitute one part, if present at all. Therefore, if sulphide oxidation occurs in mining waste, the acid produced may be consumed by acid consuming reactions in varying degrees, depending on the acid consuming minerals available. If carbonates are present in the mining waste, pH is normally maintained as neutral, the dissolved metals precipitate and thus are not transported to the surrounding environment to any significant degree. Other acid consuming minerals include alumino-silicates. The dissolution of alumino-silicates is kinetically controlled and cannot normally maintain a neutral pH in the drainage.

The interaction between the acid producing sulphide oxidation and the acid consuming dissolution of buffering minerals determines the pH in the pore water and drainage, which in turn influences the mobility of metals. If the readily available buffering minerals are consumed, the pH may drop and ARD will then occur.

The release of ARD to surface- and groundwater deteriorates the water quality and may cause a number of impacts, such as depletion of alkalinity, acidification, bioaccumulation of metals, accumulation of metals in sediments, effects on habitats, elimination of sensitive species and unstable ecosystems.

The chemical processes of acid generation and acid consumption are explained in Section 2.7

Weathering at the field scale

ARD may be produced where sulphide minerals are exposed to the atmosphere (oxygen and water) and there are not enough readily-available buffering minerals present. In mining this could be in, e.g., waste-rock deposits, marginal ore deposits, temporary storage piles for the ore, tailings deposits, pit walls, underground workings or in heap leach piles. Historically sulphide-containing material has also been used for construction purposes at mine sites, e.g. in the construction of roads, dams and industrial yards. However, regardless of where ARD production occurs, the fundamental processes behind the generation of ARD are the same.

Figure 1.21 schematically shows some of the most important geochemical and physical processes and their interaction and contribution to the generation of ARD and the possible release of heavy metals from mining waste. As can be concluded from the figure, the ARD and metal release will depend primarily on the sulphide oxidation rate, the potential immobilisation/remobilisation reactions along the flow path and the water flow. However, the sulphide oxidation rate is dependant on redox conditions (Eh), pH, and microbial activity. The pH is, in turn, determined by the sulphide oxidation rate and buffering reactions (carbonate dissolution and silicate weathering). Furthermore, the potentially metal retaining immobilisation reactions that can occur along the flow path are dependant on pH, redox conditions and the sulphide oxidation rate.

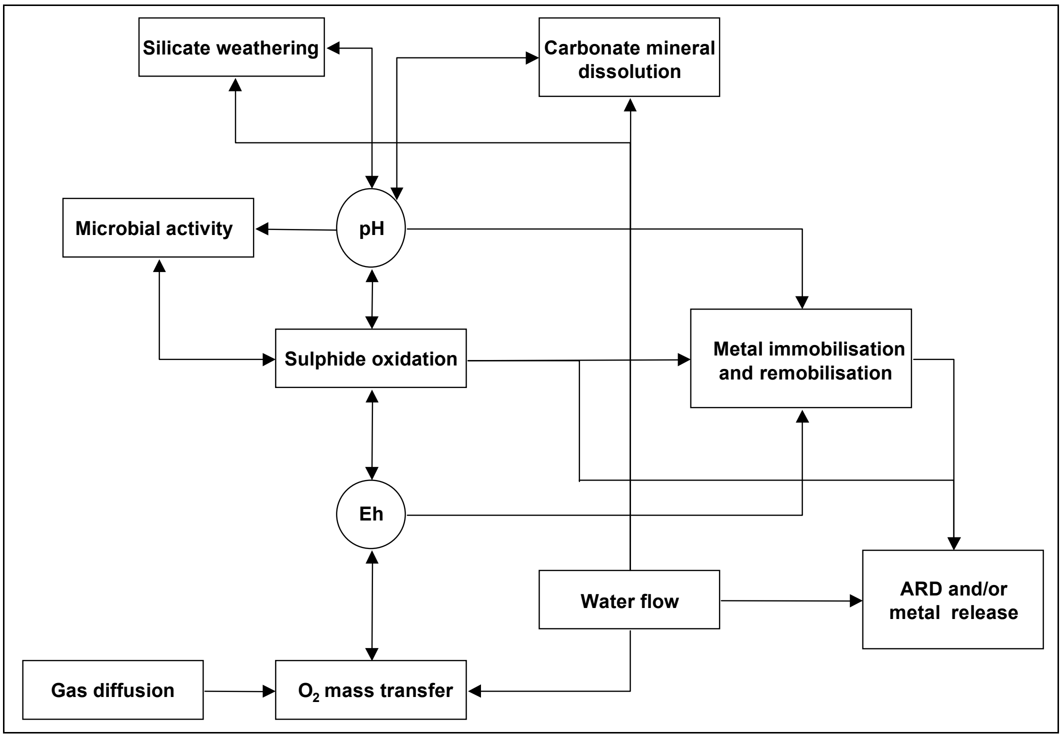


Figure 1.21: Schematic illustration of some of the most important geochemical and physical processes and their interaction and contribution to the possible release of heavy metals from mining waste
[20, Eriksson, 2002]

At the field-scale not only are the temporary variations of material characteristics important for the evolution of the drainage water quality but the spatial variations will also be a factor to take into account. The resulting drainage characteristics depend on a number of additional parameters, such as infiltration rate, evaporation rate, oxygen profile in the deposit, height of the deposit, and the construction of the deposit. Heterogeneities in the material characteristics, such as varying mineralogy and degree of compaction, are other parameters that may affect the drainage water quality. Due to the normally long residence time of the infiltrating water in the deposit, the influence of various immobilisation reactions (precipitation and adsorption) can also be significant. The interaction between the tailings and/or waste-rock and the atmosphere is illustrated schematically in the following figure.

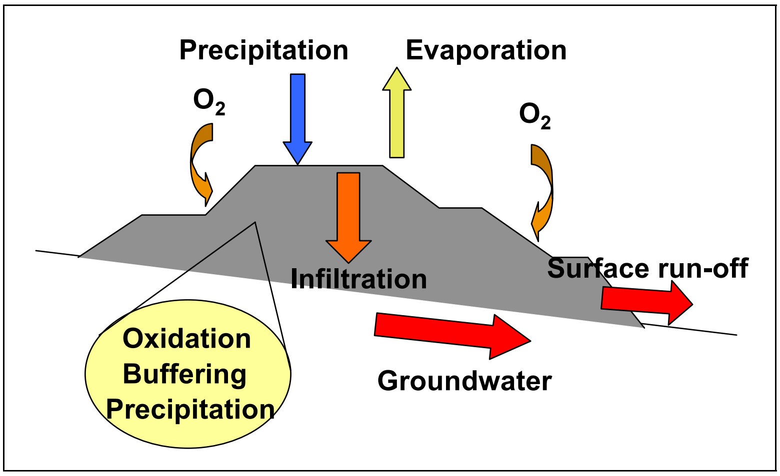


Figure 1.22: Schematic illustration of the drainage water generation as a function of the interaction between the tailings or waste-rock in the facility and the atmosphere
[20, Eriksson, 2002]

1.6.3.4 Accidental bursts or collapses

The bursts or collapses of tailings dams at operations in Aznalcollar and Baia Mare have brought public attention to the management of tailings ponds and tailings dams. However, it should not be forgotten that the collapse of tailings and waste-rock heaps can cause severe environmental damage. The dimensions of either type of tailings management facility can be enormous. Dams can be tens of metres high, heaps even more than 100 m and several kilometres long possibly containing hundreds of millions of cubic metres of tailings or waste-rock. At the other extreme are ponds the size of a swimming pool or heaps smaller than a townhouse.

The following two pictures show the two extremes. Figure 1.23 shows a pond containing 330 Mm^3 of tailings and Figure 1.24 shows a small settling basin.

Figure 1.23: Example of a large tailings pond (330 Mm^3)

Figure 1.24: Example of a small tailings settling basin

Tailings dams are built to retain slurried tailings. In some cases, material extracted from the tailings themselves is used for their construction. Tailings dams have many features in common with water retention dams. Actually, in many cases they are built as water retaining dams, particularly where there is a need for the storage of water over the tailings [9, ICOLD, 2001].

Heaps are used to pile up more or less dry tailings or waste-rock.
The collapse of any type of TMF can have short-term and long-term effects. Typical short-term consequences include:

- flooding
- blanketing/suffocating
- crushing and destruction
- cut-off of infrastructure
- poisoning.

Potential long-term effects include:

- metal accumulation in plants and animals
- contamination of soil
- loss of animal life.

Guidelines for the design, construction and closure of safe TMFs are available in many publications. If the recommendations given in these guidelines were to be closely followed, the risk of a collapse would be greatly reduced. However, major incidents continue to occur at an average of more than one a year (worldwide) [9, ICOLD, 2001]

An investigation of 221 tailings dam incidents has identified the main causes for the reported cases of dam failures. The main causes were found to be lack of control of the water balance, lack of control of construction and a general lack of understanding of the features that control safe operations. It was found that only in very few cases did unpredictable events, such as unexpected climatic conditions or earthquakes, cause the bursts [9, ICOLD, 2001].

1.6.4 Site rehabilitation and after-care

When an operation comes to an end, the site needs to be prepared for subsequent use. Usually, these plans are part of the permitting of the site from the planning stage onwards and should, therefore, have undergone regular updating, depending on changes in the operation and in negotiations with the permitters and other stakeholders. In some cases, the aim is to leave as little a footprint as possible, whereas in other cases, a complete change of landscape may be aimed for. The concept of ‘design for closure’ implies that the closure of the site is already taken into account in the feasibility study of a new mine site and is then continuously monitored and updated during the life cycle of the mine. In any case, negative environmental impacts need to be kept to a minimum.

Some sites can be handed over to the subsequent user after a relatively simple reclamation, e.g. after reshaping, covering and re-vegetation. In other cases, after-care will need to be undertaken for long periods of time, sometimes even in perpetuity.

It is impossible to restore a site to its original condition. However, the operator, the authorities and the stakeholders involved have to agree on the successive use. It will usually be the operators responsibility to prepare the site for this. In order to receive a permit for the closure, the characteristics of the impounded material should be well determined (e.g. amounts, quality/ consistency, possible impacts). As indicated in Section 1.6.3.3 avoiding future ARD is a main concern for the closure design for tailings with a net ARD potential.

2 COMMON PROCESSES AND TECHNIQUES

This chapter aims to provide background information to non-experts in the management of tailings and waste-rock. Together with the specific glossary this chapter should allow the reader to understand the subsequent chapters.

2.1 Mining techniques

The extraction of an ore, (a process called mining), subsequent mineral processing and the management of tailings and waste-rock are, in most cases, considered to be a single operation. Even though this document does not cover the ore extraction, the subsequent mineral processing techniques and tailings and waste-rock management are all highly dependent on the mining technique. Hence, it is important to have an understanding of the most important mining methods.

For the mining of solids, there are four basic mining concepts:
(1) open pit
(2) underground mine
(3) quarry and
(4) solution mining.

The choice between these four alternatives depends on many factors, such as:

- value of the desired mineral(s)
- grade of the ore
- size, form and depth of the orebody
- environmental conditions of the surrounding area
- geological, hydrogeological and geomechanical conditions of the rock mass
- seismic conditions of the area
- site location of the orebody
- solubility of the orebody
- environmental impact of the operation
- surface constraints
- land availability.

Often the uppermost part of an orebody is mined in an open pit, but over time and with increasing depth, the removal of overburden makes this mining method uneconomical, so deeper parts are sometimes mined underground (see figure below). An alternative to continuing the mining underground, is often to stop production altogether, as the processing plant may have designed for large tonnages only, which are difficult to achieve underground. Mining costs are significantly higher underground, which is another reason for often ruling out this possibility. It also may be rejected if the orebody is not continuous enough to allow economical underground mining. Rock stability may also set limits on any underground mining.

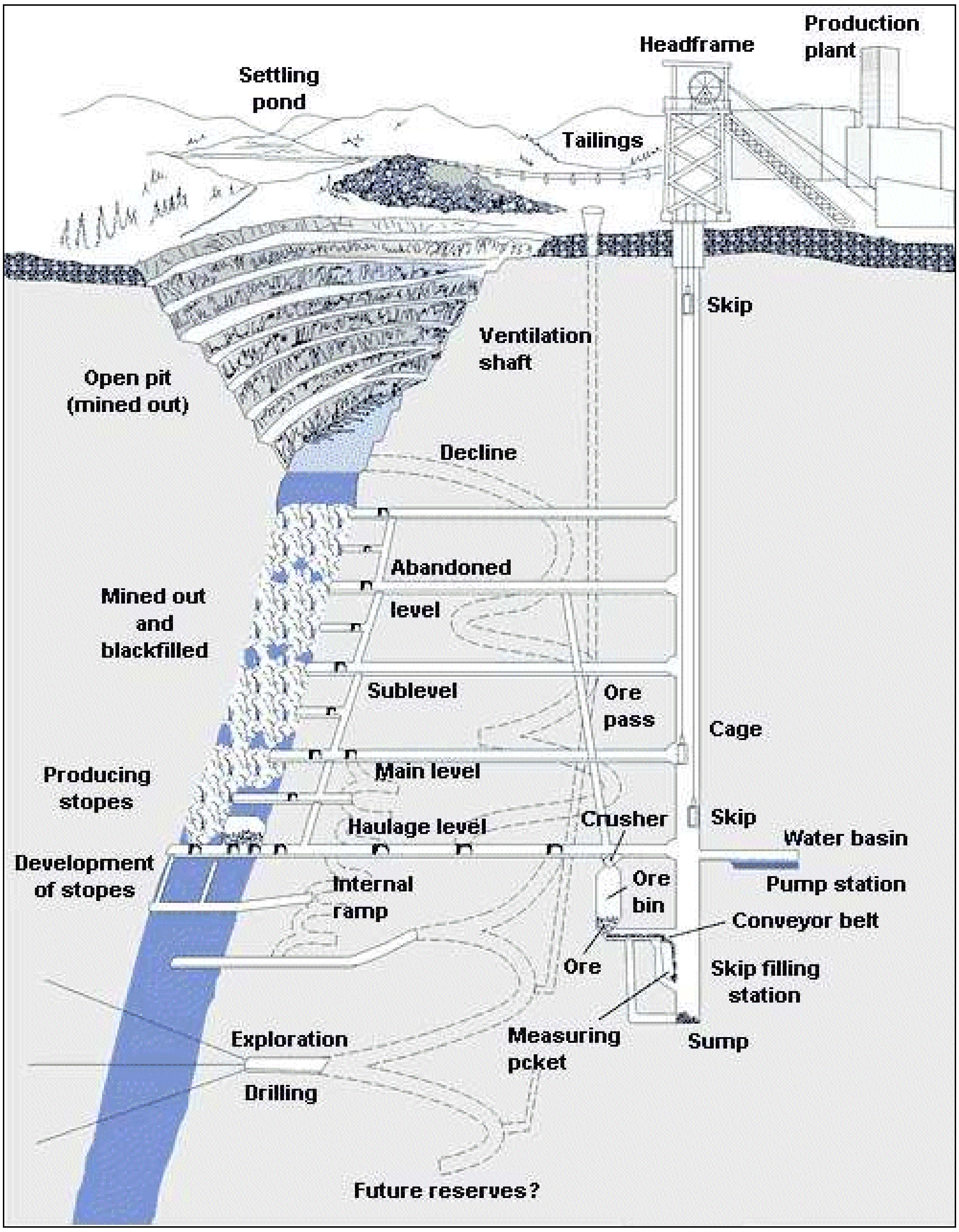


Figure 2.1: Transition from open pit to underground mining
[93, Atlas Copco, 2002]

If open pit is the chosen mining method, it will, in most cases, result in larger amounts of waste-rock. This is indicated in the following two figures. The waste-rock may be deposited close to the open pit, backfilled into the current or nearby mined out open pits or crushed and sold if there is a market for the material.

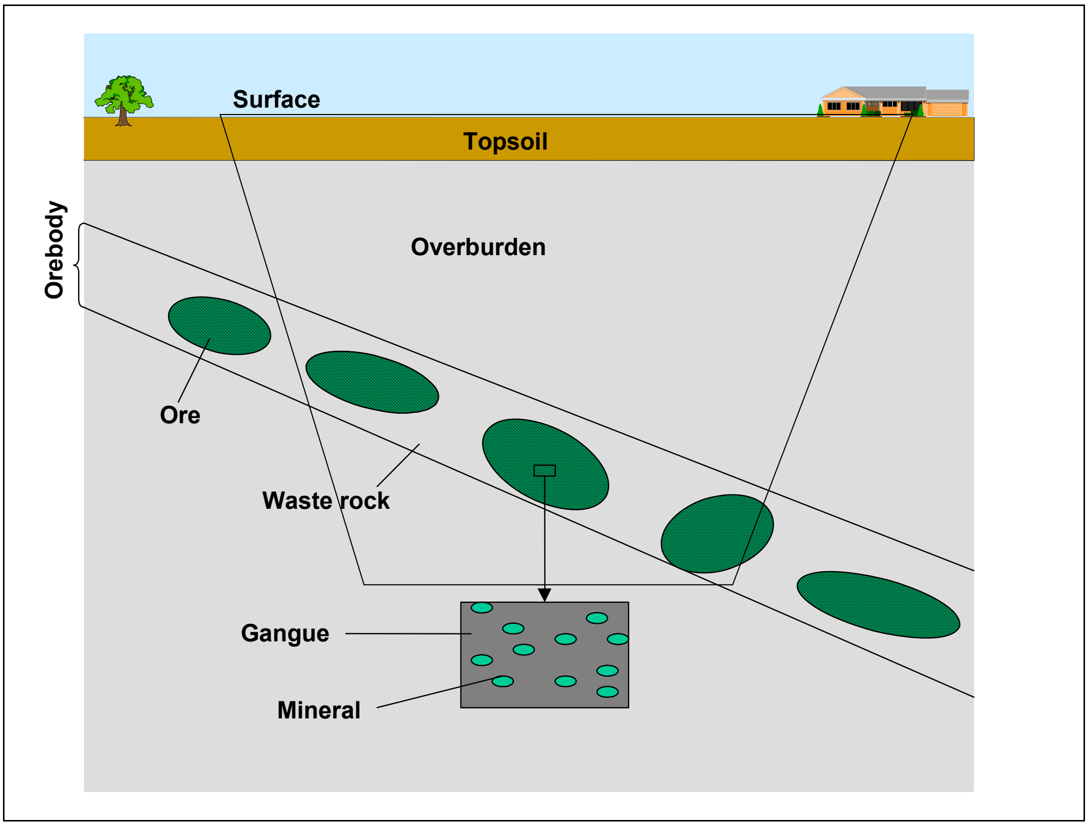


Figure 2.2: Schematic drawing of an open pit

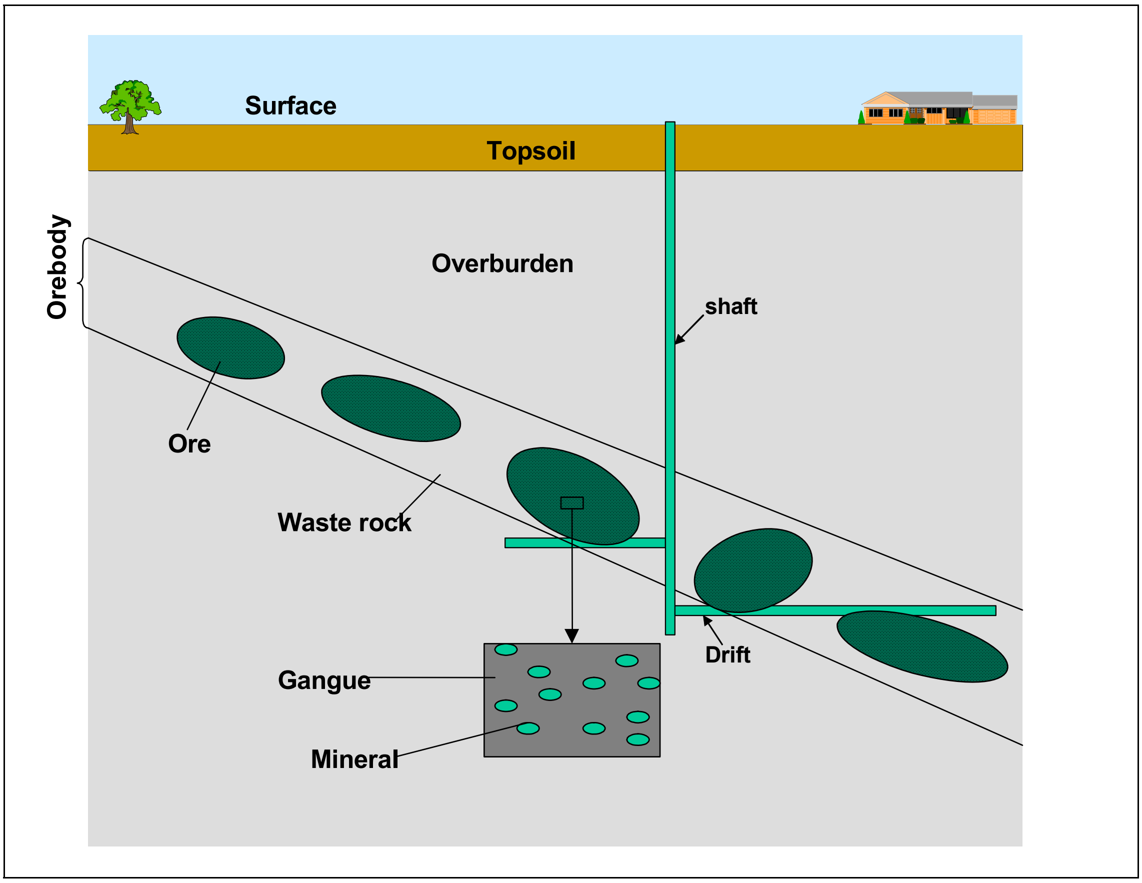


Figure 2.3: Schematic drawing of an underground mine

In the example shown in the above figures the amounts of topsoil, overburden and waste-rock that will have to be moved using the open pit technique are greater than with underground mining. In the latter case, a shaft and drifts are constructed from which the ore can be mined more selectively, meaning areas of waste-rock and/or low grade ore can mostly be left out. The waste-rock that has to be mined is either moved within the mine or hoisted to the surface.

It should be noted that the above drawings only show schematic drawings of one scenario. As will be described in the following section there are many different types of orebodies. Also the grades can vary quite significantly, for example, in most cases, viable industrial minerals deposits have an ore grade of between 50 and 99 %. This is one of the main differences with the metallic ores, where grades are much lower.

For underground mining, it is also possible to backfill mined out areas. This may be difficult to realise in open pit operations that progress vertically while they still being mined, unless the backfill material can be moved to another pit. However open pits that progress horizontally are typically progressively restored.

2.1.1 Types of orebodies

The type of orebody has a big influence on the choice of mining method. The following types of orebodies are known, with classification depending on the form of the orebody or the distribution of the ore:

- seam-type orebody
- vein-type orebody
- massive-type orebody (e.g. massive sulphides with high variations of grade within the orebody; limestone orebodies with very consistent grades)
- disseminated-type orebody (e.g. copper porphyries).

Often the disseminated type has a ‘cap’ of weathered sulphides (hence oxides) on top of a disseminated-type orebody. The ore within this weathered cap is called ‘gossan’.

2.1.2 Underground mining methods

There are many different ways of exploiting an orebody using underground mining methods. The most commonly used underground mining methods are:

Mining method	Application
Longwall mining	Flat, thin seam type ore bodies, soft rock
Room and pillar mining	Inclined massive-type ore bodies and flat seam-type deposits
Sublevel stoping	Steep, large ore bodies (massive- or disseminated-type)
Cut and fill mining	Steep, firm ore bodies, selectivity, mechanisation (seam-, vein-, massive-, disseminated-type)
Sublevel and block caving	Steep, large or massive ore bodies, extensive development effort (mostly massive-, disseminated-type)

Table 2.1: Most important underground mining methods and their areas of application [47, Hustrulid, 1982]

Section 3.1.4.1 provides an example of a highly mechanised underground mining operation using large-scale sublevel caving.

These methods have been widely described in literature (e.g. AIME/SME Underground mining methods handbook, http://sg01.atlascopco.com). The basic objective of selecting a method to mine a particular orebody is to design an ore extraction system that is most suitable under the existing circumstances. This means aiming for lowest operational costs. This decision is based upon both technical and non-technical factors (e.g. high productivity, complete extraction of the ore, safe working conditions).

In 'room and pillar' mining, some of the ore remains unmined and serves as support (the pillars) for the mine shafts. In some cases, backfilling is used to allow subsequent mining of these pillars.

A reduction of tailings can be achieved by using the most selective mining method, i.e. by insuring that only undiluted ore is fed to the mineral processing plant, so that the amount of waste-rock that has to be handled, is minimised. Feeding diluted ore to the mineral processing plant results in a decrease in recovery and, therefore, results in larger amounts of the desired mineral being lost in the tailings.

2.2 Mineralogy

Basically it is possible to differentiate between oxide, sulphide, silicate and carbonate minerals, which, through weathering and other alterations, can undergo fundamental changes (e.g. weathering of sulphides to oxides). Mineral paragenesis and intergrowth are important bases for the subsequent mineral processing and, thereby, the tailings and waste-rock management. Therefore, a basic knowledge of the mineralogical composition is of utmost importance.

Mineralogy is set by nature and determines, in many ways, the subsequent recovery of desired minerals and the tailings and waste-rock management. Mineralogy often changes within an orebody and hence during the life of a mine. Sometimes these changes are well known and can be planned for, sometimes they occur unexpectedly. Some examples are listed below:

- oxides on top and sulphides in deeper lying parts of the orebody, which require completely different mineral processing and tailings management methods
- ore type changing from a copper ore to a zinc ore
- ore type changing from a magnetite to a haematite type iron ore (Malmberget).

Mineralogy has a big influence on the mining technique chosen and the sequencing of mining operations. For example, for gold mining the gossan is mined because it is more easily accessible and naturally enriched and is easier to recover. The deeper lying sulphides have to be oxidised before they can be recovered, which makes the process less profitable. For copper, it is also easier to recover the oxide section, which can easily be leached using sulphuric acid, than the sulphides, which have to be recovered using flotation.

The sulphide content, which is determined by mineralogy, influences the tailings and waste-rock management, because of its acid generating potential (see Section 2.7).

Having a good knowledge of mineralogy is an important precursor for:

- environmentally sound management (e.g. separate management of acid-generating and non-acid-generating tailings or waste-rock)
- a reduced need for end-of-pipe treatments (such as the lime treatment of acidified seepage water from a TMF)
- more possibilities for utilising tailings and/or waste-rock as aggregates.

2.3 Mineral processing techniques

The purpose of mineral processing is to turn the raw ore from the mine into a marketable product.

2.3.1 Equipment

The following information is all taken from [105, Wotruba, 2002].

2.3.1.1 Comminution

Comminution is an essential element of mineral processing. It requires a great deal of expenditure in terms of energy consumption and maintenance. In comminution, the particle size of the ore is gradually reduced. This is necessary for many reasons, e.g.:

- to liberate one or more valuable minerals from the gangue in an ore matrix
- to achieve the desired size for later processing or handling
- to expose a large surface area per unit mass of material, thus aiding some specific chemical reaction (e.g. leaching)
- to satisfy market requirements relating to particle size specifications.

Comminution is composed of a sequence of crushing and grinding processes.

After grinding, the ore, often in slurry form, ‘contains’ the now liberated ore particles and the tailings material which need to be separated in later process steps. The characteristics of the ore, in combination with the equipment used for the crushing and grinding, determine the physical properties of the tailings, such as the particle shape and particle size distribution.

2.3.1.1.1 Crushing

Crushing is the first stage in the comminution process. This is usually a dry operation, which involves breaking down the ore by compressing it against rigid surfaces or by impacting it against hard surfaces in a controlled motion flow.

This process step prepares the ore for further size reduction (grinding) or for feeding directly to the classification and/or concentration separation stages. Tailings are usually not generated in this process step.

Typical types of crushers are:

- jaw crushers
- gyratory crushers
- cone crushers
- roll crushers
- impact crushers.

2.3.1.1.2 Grinding

Grinding is the final stage in the comminution process and requires the most energy of all the mineral processing stages. Because of this, the tendency is to first blast (in the mine) or crush the ore as fine as possible to reduce the amount of larger materials sent to grinding, thereby reducing the overall energy consumption in grinding and, hence, comminution. If possible, grinding is performed 'wet' as this requires less energy, allowing energy savings of up to 30 % compared to dry grinding. In grinding, the particles are usually reduced by a combination of impact and abrasion of the ore by the free motion of grinding bodies such as steel rods, balls or pebbles in the mill.

<u>Tumbling mills</u>
Tumbling mills consist of a rotating cylindrical steel vessel on a horizontal axis, with openings on both ends for feeding and discharging material. The vessel contains tumbling bodies that are free to move as the mill rotates on its horizontal axis (the vessel rotating on hollow trunnions fastened to the end walls). The tumbling bodies include balls, rods, or other shapes and forms, and are made of steel, cast iron, hard rock, ceramic materials or may even consist of the material itself being reduced (pebbles).

The most commonly used **tumbling mills** are:

- rod mills, for product sizes: <1 mm
- ball mills, for product sizes: <100 µm
- autogenous (AG) mills, semi-autogenous (SAG) mills; product size: in combination with ball mills typically <1500 µm; if only AG or SAG mill:<100 µm possible.

Figure 2.4 and figure 2.5 respectively show a ball mill and a grinding circuit, consisting of AG mills and ball mills used for primary and secondary grinding.

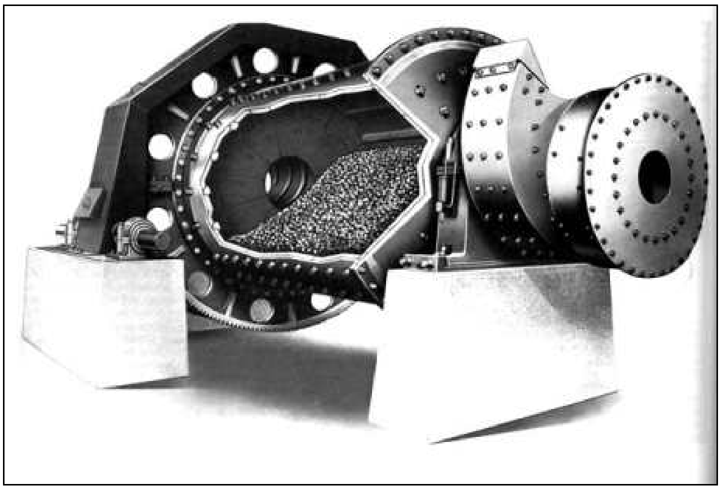


Figure 2.4: Ball mill

Figure 2.5: Grinding circuit with AG mills (primary grinding, right side) and ball mills (secondary grinding, left side)

In rod and ball mills, the grinding media are rods and balls made of steel and sometimes ceramic. Sometimes, conical steel pieces, called cylpeps, are used as a grinding medium in ball mill size mills. As reflected by the name, in AG mills the ore grinds itself. For this purpose, larger ‘pebbles’, i.e. fist size pieces, of the ore are required in the mill. In SAG mills, these pebbles are assisted by a small loading, compared to rod and ball mills, of steel balls.

Tumbling mills are essential for fine grinding of large quantities (e.g. for froth flotation feed or agitation leaching feed).

The degree of grinding is governed by the ore characteristics and the chosen method(s) of extracting the valuable minerals, e.g. flotation requires a fine feed. However, overgrinding will generate ‘slimes’ which can reduce the efficiency of flotation and as a secondary effect could also lead to tailings that take a longer time to dewater and become stable in a pond.

Beside tumbling mills, other important types of grinding equipment are **agitated mills** and **vibrating mills**.

Agitated mills

Agitated mills are used for very fine wet grinding. Agitated mills (or tower mills) are vertical steel cylinders filled with 80 - 90 % grinding media which are agitated by an internal flighted axis. Throughput is a maximum of 100 t/h, feed size <1 mm, and the product size will be 1 - 100 µm.

Vibrating mills

Vibrating mills are used for very fine grinding (dry or wet). Continuous vibrating mills are horizontal steel cylinders filled with 60 – 70 % grinding media, agitated by an eccentric drive. Throughput is a maximum of 15 t/h, for product sizes of <10 µm.

2.3.1.2 Screening

Screening can be defined as a mechanical operation which separates particles according to their sizes and their acceptance or rejection by openings of a screening face. Particles that are bigger than the apertures of the screens are retained, and constitute the oversize. Conversely, those that are smaller pass through the screening surface, forming the undersize. There are many different types of industrial screens, which may be divided into stationary and moving screens. The most important reasons for screening in mineral processing are:

- to avoid undersize material entering the crushers
- to avoid oversize material passing to the later stages in the grinding process or in closed-circuit fine crushing
- to produce material of controlled particle size, e.g. after quarrying.

2.3.1.3 Classification

Classification may be described as the separation of solid particles into two or more products according to their velocities when falling through a medium. The velocity of the particles depends on their size, density and shape. In mineral processing, classification is mostly carried out wet, with water being used as the fluid medium. Dry classification, using air as the medium, is used in several applications (cement, limestone, coal). Classification is normally performed on minerals considered too fine to be separated effectively by screening.

2.3.1.3.1 Settling cones and hydraulic classifiers

Uses: Cones (or settling cones) are mostly used for de-sliming. Hydraulic classifiers in the mineral industry are used either to receive final products (sand industry) or to prepare feed into several particle size ranges for subsequent gravity concentration processes.

Principles and construction: Settling cones are conical vessels, where the pulp is introduced vertically from the top. Coarse particles settle down and leave the vessel through the underflow spigot, fine particles leave the vessel with most of the water over the upper rim (overflow). Hydraulic classifiers use extra water, which is injected into the separating vessel. The direction of water flow is opposite to that of the settling particles. In general, hydraulic classifiers are composed of a sequence of columns, in which a vertical current of water rises inside each column with heavier particles settling out first. A typical hydraulic classifier is the 'Fahrenwald classifier', widely used in the glass and foundry sand industry. New hydraulic classifiers are the 'allflux' or similar models, that combine hydraulic classification with autogenous dense medium thus combining classification with dense medium separation (mostly used to de-coal sands).

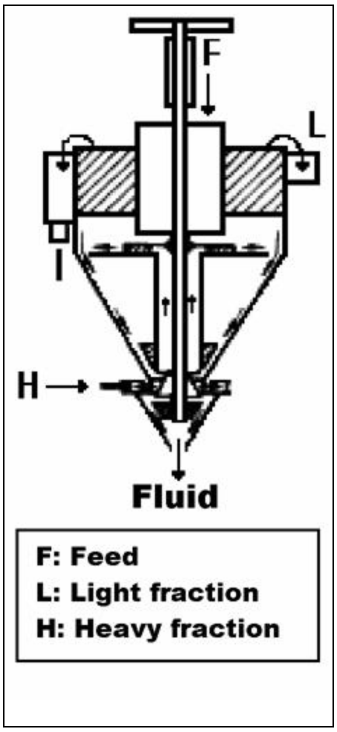


Figure 2.6: Hydraulic classifier

2.3.1.3.2 Hydrocyclones

Uses: Widely applied in mineral processing for fine classification (mostly <100µm), often in a closed-circuit with ball mills for flotation or leaching feed preparation and for special fine final products (kaolin). They are particularly efficient for fine separation sizes, such as de-sliming, thickening and de-gritting.

Principle and construction: A hydrocyclone is a vessel composed of a cylindrical section with a tangential feed entrance, joined to a lower conical part. The feed is accelerated and rotates at high speed within the vessel, transporting the coarse particles by centrifugal forces to the inner wall, from where it moves down along the conical part and leaves the vessel through the underflow spigot. The slower settling fine particles stay in the centre of the fluid, which forms an inner upstream current and leaves the vessel through the central upper discharge opening. To avoid short-cuts, the upstream is collected by an adjustable inner piece of pipe, connected to the overflow outlet (vortex finder).

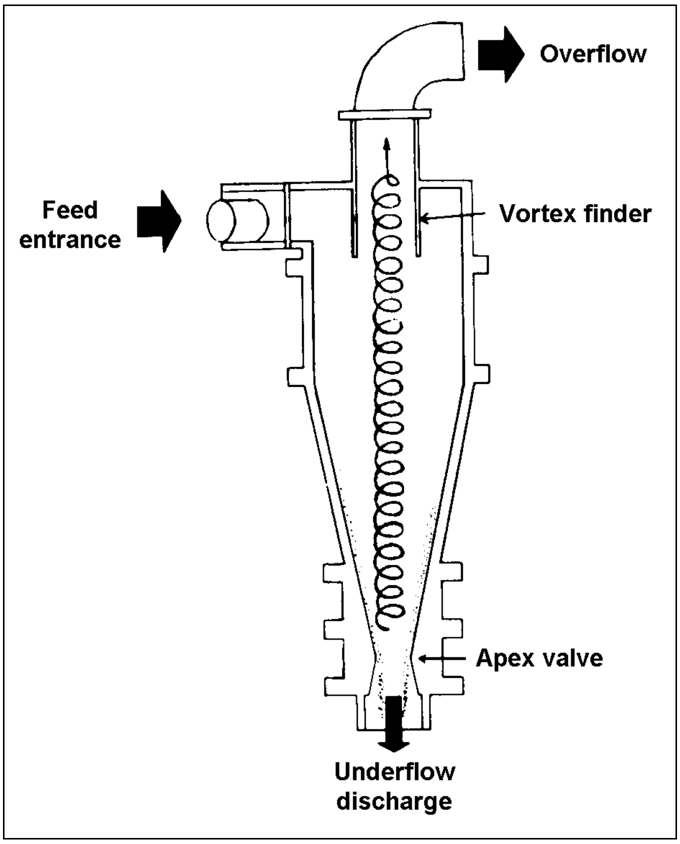


Figure 2.7: Hydrocyclone

The separation size and the throughput depend on the diameter of the hydrocyclone. For larger throughputs hydrocyclones are used in parallel.

2.3.1.3.3 Mechanical classifiers

Uses: Formerly closed-circuit grinding operations, dewatering, washing and de-sliming operations, were frequently used in milling circuits, but they are now gradually being replaced by hydrocyclones. Nowadays, they are mostly used in the sand and gravel industry and in smaller ore processing plants.

Principles and construction: Mechanical classifiers consist of a settling tank with parallel sides and an inclined base, which is equipped with a device that constantly promotes the agitation of the pulp and removal of the settled solids. The feed pulp is fed into the classifier, forming a settling pool in which particles of high falling velocity rapidly fall to the base of the tank. Mechanical rakes or helical screws drag the material deposited on the equipment bottom, upwards. At the same time, the material of lower settling velocity is taken away in a liquid overflow. There are various types of mechanical classifiers available, mainly ‘spiral classifiers’ and ‘rake classifiers’.

General technical data spiral classifiers:
- tank length: 3 – 12 m
- tank width: 0.3 – 6.5 m
- spiral circumferential speed: 10 – 40 m/min
- tank inclination: 14 - 18°
- rate of flow: 10 - 90 m^3/h.

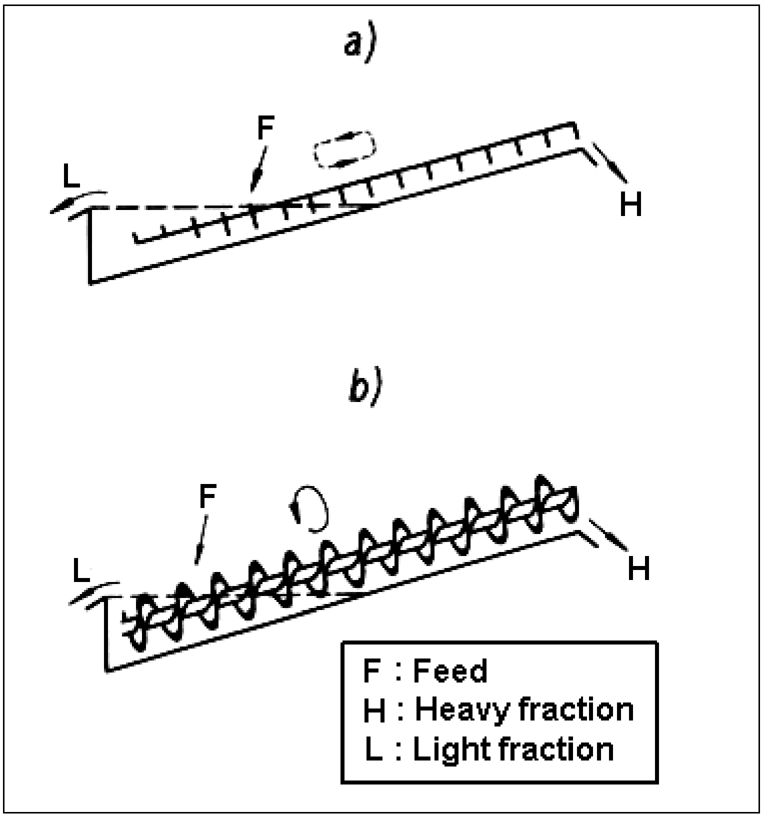


Figure 2.8: Rake and spiral classifiers

2.3.1.4 Gravity concentration

Gravity concentration is a method of separating minerals of different density by the force of gravity or by other forces, such as centrifugal force or the resistance to movement offered by a viscous fluid, such as water or air. The motion of a particle in a fluid is dependent not only on its specific gravity, but also on its size and shape. Advanced gravity concentration has proven itself to be an alternative to flotation and leaching, since, among other reasons, no reagents are required.

2.3.1.4.1 Dense medium separation

<u>Gravitational vessels</u>
Uses: Coal industry, also iron and chromite ore processing.

Principle and construction: Gravitational vessels include containers into which both the feed and dense medium are introduced. The floats are separated by paddles or simply by overflow, while the sinks can be removed by different means according to the separator design. The most complicated part of the separator design is the discharge of the sinks, as the purpose is to remove the sink particles without draining the dense medium by producing disturbing downward currents in the vessel. There are numerous types of gravitational vessels available, such as the 'Wemco cone separator', 'drum separators', or the 'Drewboy bath.

General technical data:
Drewboy bath:

- feed particle size: up to 1000 mm
- rate of flow: 25 – 150 t/h per m of wheel width.

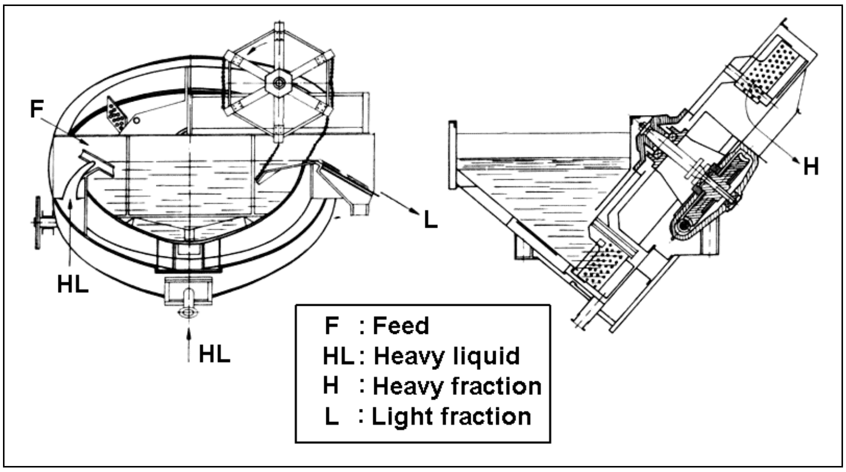


Figure 2.9: Drewboy bath

Centrifugal separators

Uses: Treatment of coal, chromite, baryte, fluorspar, etc. and for the concentration of particles in the intermediate size range, in particular those too small for conventional gravity-type separators but too large for froth flotation.

Principles and construction: In centrifugal separators, centrifugal acceleration aids the gravitational acceleration in separating minerals with low densities from those with high densities. The two most important types of dense medium centrifugal separators are the 'DSM cyclone' commonly called 'dense medium cyclone' and the 'Dyna-Whirlpool (DWP)' and similar types (e.g. the 'Tri-Flow', which is basically a three-product separator consisting of two in-line Dyna-Whirlpools). A similar size to the Dyna-Whirlpool in design but larger in capacity and feed size is the 'Larcodems'-separator.

General technical data:

DSM cyclone (dens media cyclone):

- feed size: metal ores in the size range 0.5 - 10 mm, and coal in the size range 40 - 0.5 mm
- diameter: 250 - 1500 mm
- maximum density: 3 t/m^3
- capacity: up to 30 t/h.

Dyna Whirlpool (DWP):

- feed size: coal, diamonds, tin and lead-zinc ores in the size range 0.5 - 30 mm, barytes, feldspar
- cylinder inclination: 30°
- capacity: 30 - 100 t/h
- diameter: 250 - 400 mm.

2.3.1.4.2 Jigging

Uses: Jigging is used today in pre-concentration or in the sorting process of coarse material (mainly coal). Many large jig plants are in operation in the gold, barytes, coal, cassiterite, tungsten, iron-ore, sand and gravel industries.

Principles and construction: In jigging the ore particles are held up on a perforated screen or plate in a layer many times higher than the thickness of the major particle. This layer or 'bed' is exposed to an alternating increasing and decreasing (pulsating) flow of fluid in an attempt to produce stratification, causing all the high density particles to move to the base of the bed while the low specific gravity particles assemble at the top of the bed. The fluid is commonly water.

There are various types of jigs such as the 'Denver mineral jig', the 'circular jig', the 'Baum jig'and the 'Batac jig'.

General technical data (examples):

Denver Mineral Jig (mostly used for heavy minerals, in milling circuits):

- high frequency: 280 - 350/min
- fine grains: 100 µm – 5 mm
- application: heavy minerals and sulphides
- maximum setting surface: 2 x (60 x 90 cm)
- maximum throughput: 30 t/h.

Batac jig (mostly used for coal):

- width: up to 7 m
- length: up to 6 m
- throughput: up to 1000 t/h.

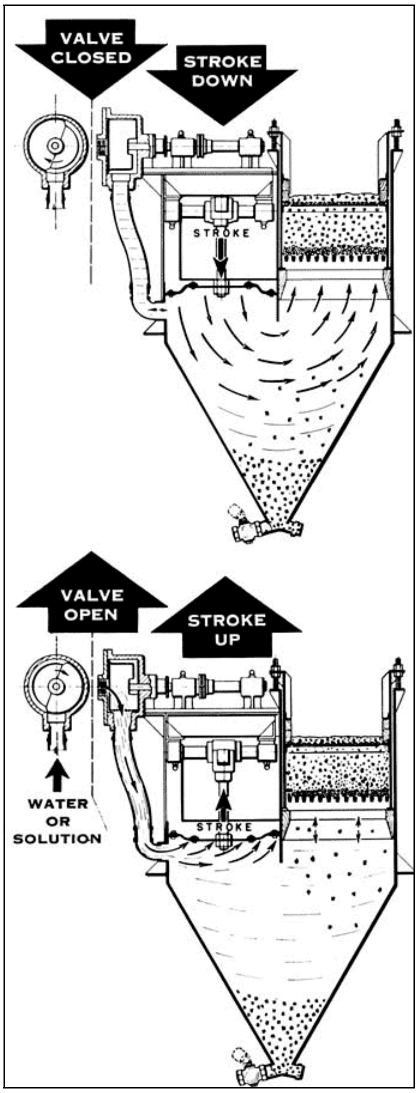


Figure 2.10: Denver mineral jig

2.3.1.4.3 Shaking tables

Uses: Treatment of coal, gold, heavy minerals, tantalum, tin, barite, glass sands, chromite, etc.

Principles and construction: The shaking table can be described as a platform deck with a slight inclination, riffles and a rectangular or rhomboid form. The shaking table is commonly built from wood or fibreglass. Water and solids are fed onto its upper edge. The table vibrates longitudinally as a result of slow forward strokes and quick returns. The minerals move slowly along the table, under exposure to two forces. The first force is caused by the deck movement and the second one by a streaming film of water. The outcome is that the minerals separate on the deck, the lighter, bigger grains being taken to the tailings launder whilst the denser, smaller grains are carried in the direction of the concentrate launder at the far side of the deck. The concentrate can be divided into various products, for example a middling fraction and a high-grade concentrate, by adjustable splitters situated at the concentrate end. The shaking table has various designs and operating variables which regulate the process.

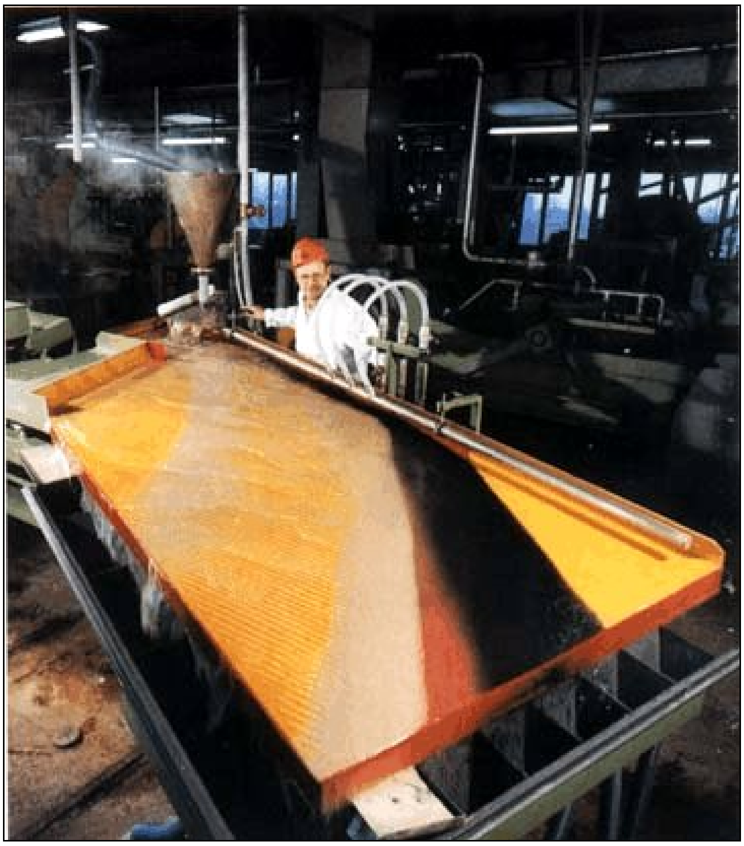


Figure 2.11: Shaking table

2.3.1.4.4 Spirals

Uses: Diverse applications, principally used in the processing of heavy mineral sands, gold, tin, tantalum, glass sands, and fine coal

Principles and construction: Spirals consist of a helical trough with a modified semicircular cross-section. The slurry is fed into the top of the spiral and during its helical course, the grains are stratified as a consequence of different mechanisms such as the differential settling rates of the particles, centrifugal forces and interstitial trickling through the flowing particle layer. Product bands are removed through adjustable splitters along the helix and/or at the lower discharge end of the spiral. Nowadays, several types of spirals are applied for gravity concentration, all developed from the original 'Humphreys spiral'.

General technical data:

- processable particle size: coal: 0.1 – 4 mm, metal ores: 0.02 – 1 mm
- throughput: 1-3 t/h per spiral.

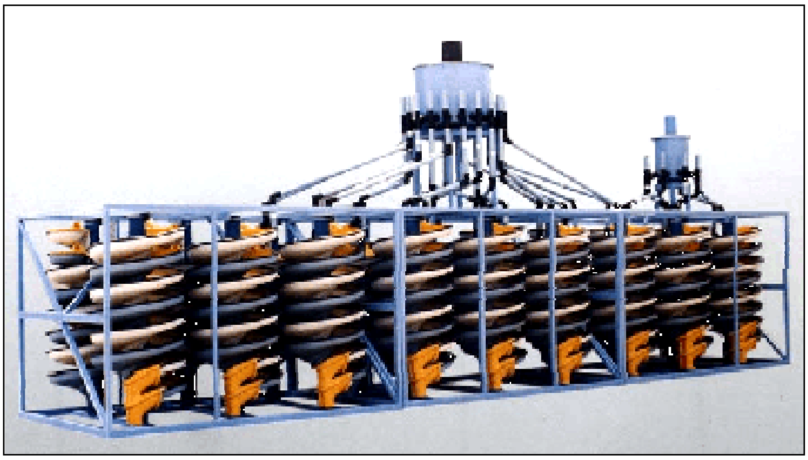


Figure 2.12: Spiral bank

2.3.1.4.5 Cones

Other than the settling cones mentioned in Section 2.3.1.3.1, which classify the feed according to grain size, cones are used for separation according to specific gravity.

Uses: In high-capacity gravity concentration applications for fine material (<1 mm), such as in the treatment of beach sands; pre-concentration of tin, iron and gold, recovery of wolframite and chromite, and in the concentration of magnesite.

Principles and construction: Several stages of upgrading can be carried out in a single unit of equipment, since the equipment consists of several cone sections piled vertically. In the 'Reichert cone', for example, a vertical distributor cone distributes the feed at high pulp density around the periphery of an upturned concentration cone. When the feed flows in the direction of the cone centre the heavy mineral particles separate to the bottom of the film. An annular slot in the base of the concentrating cone withdraws this concentrate while the fraction of the film flowing over the slot, constituting the tailings, falls into the feed box for the second stage.

General technical data:

- cone diameter: 2 m
- solids content: 55 – 65 %
- throughput: 70 – 100 t/h.

Figure 2.13: Reichert cone

2.3.1.5 Flotation

Uses: This is the most important separation technique used in mineral processing for base-metal ores. Originally used to concentrate sulphides, ores of copper, zinc and lead, it is nowadays also used in the treatment of non-metallic ores such as fine coal, fluorite and phosphate, potash, oxides such as cassiterite and haematite; and oxide minerals, such as cerussite and malachite.

Principles and construction: In flotation, the separation of minerals is accomplished by utilising the differences in their physico-chemical surface properties. For instance, after conditioning with reagents, some particles become water repellent or hydrophobic (or aerophilic), while other particles remain hydrophilic. In the selective separation process, the air-bubbles stick to the hydrophobic (or aerophilic) particles, lifting them to the water surface and forming a stable froth, which is removed. The hydrophilic particles remain within the pulp and are discharged. Flotation processes generally consist of several stages to clean the concentrates again and to scavenge the remaining valuable minerals from the tailings.

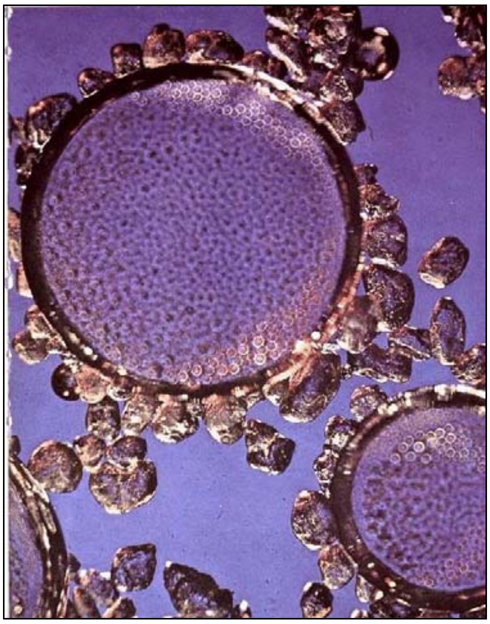


Figure 2.14: Flotation process

Flotation cells
There are two principal types of flotation cells: pneumatic and mechanical.

- mechanical cells are the traditional and most widely used devices applied in flotation plants. They consist of steel vessels which have a mechanically driven impeller that causes the dispersion of the air as small bubbles and agitates the slurry. Several single cells are mounted to a bank. The froth overflows or is removed with mechanical paddles
- there are two main types of pneumatic flotation cells: flotation columns and the short pneumatic flotation cell. Flotation columns consist of a high (up to 15 m) vertical steel cylinder of up to 3 m diameter. The feed pulp enters the cylinder at about three quarters of of the way up. Air enters into the vessel through a sparger at the lower end of the cylinder. Charged froth is washed by water sprays before it leaves the cylinder over the upper rim. The tailings with the hydrophilic particles leave the cylinder through the underflow spigot. Short pneumatic flotation machines do the bubble-particle collision outside the separating vessel in the pulp feeding tube, through various mixing devices or 'reactors', where compressed air is pumped into the pulp. The three-phase mixture enters the separating vessel, where the charged bubbles rise to the upper rim, where they then leave the vessel, while the tailings are discharged at the conical bottom.

Figure 2.15: Mechanical flotation cell

Figure 2.16: Pneumatic flotation cell

2.3.1.6 Magnetic separation

Uses: Tramp iron removal, concentration of ferromagnetic and paramagnetic minerals, cleaning of glass sands

Principles and construction: Magnetic separation is based on the different magnetic properties of minerals. In general, minerals can be divided into three groups according to their magnetic characteristics: diamagnetics, paramagnetics or ferromagnetics. Diamagnetics are materials which are repelled by a magnet and so are not able to be separated magnetically. Paramagnetics are materials that are attracted weakly to a magnet and can be concentrated in 'high-intensity magnetic separators'. Ferromagnetics are also materials attracted to a magnet, but this attraction is much stronger than in paramagnetics. Consequently, 'low-intensity magnetic separators' are applied to concentrate them.

The most commonly used magnetic separators are:

- dry low-intensity separators. These include drum separators principally utilised to concentrate coarse sands (cobbing process); cross-belt separators and disc separators both applied in the processing of sands; and 'magnetic pulleys' used for tramp iron removal
- wet low-intensity separators: drum separators are used to cleanse the magnetic medium in the Dense Medium Separation (DMS) circuits and to treat ferromagnetic sands, bowl traps, magnetising coils and demagnetising coils
- dry high-intensity magnetic separators: induced roll separators are used in the concentration of phosphate ore, glass sands, beach sands, tin ores and wolframite
- wet high-intensity magnetic separators: Jones separator are applied in the treatment of low-grade iron ores containing haematite.

Figure 2.17: Low-intensity drum separators

2.3.1.7 Electrostatic separation

Uses: Concentration of minerals such as ilmenite, rutile, zircon, apatite, asbestos, haematite and potash.

Principles and construction: Electrostatic separation is a method which utilises forces acting on charged or polarised bodies in an electric field to carry out mineral concentration. Different mineral particles, depending on their conductivity, will follow different paths in an electric field, making it possible to separate them. Some significant factors in this process include the mechanical and electrical characteristics of the separator and the size, form, specific gravity, surface condition and purity of the mineral particles. Mineral particles have to be entirely dry and the moisture of the surrounding air must be controlled. Electrostatic separators can be divided into plate electrostatic separators and screen electrostatic separators.

2.3.1.8 Sorting

Uses: Separation of industrial minerals, such as magnesite, barytes, talc, limestone, marble, gypsum, flint; recovery of wolframite and scheelite from quartz; treatment of gold ores, uranium ores and the recovery of diamonds.

Principles and construction: Ore sorting has been carried out since ancient times. Even though 'hand sorting' is nowadays not so common as it once was, mainly because of the large quantities of low-grade ore requiring very fine grinding, it is still applied in remote and underdeveloped countries. The mechanised procedures of sorting can be divided into photometric sorting, radiometric sorting (with uranium ores) and electrical sorting (resistance test, metal detectors).

Photometric sorting is a process, where the ore is separated into different fractions after an optical examination. The feed particles must be coarse enough e.g. (usually greater than about 10 mm) for sorting equipment to effect the desired separation at an acceptable rate. Some detectable characteristics, or combination of properties, must be present to allow a discrimination of the valuable, from the non-valuable, material. The basis of the photometric sorter is a light source and a sensitive photomultiplier, used in a scanning system to detect light reflected from the surfaces of the feed. An electronic circuit analyses the photomultiplier signal, which varies with the intensity of the reflected light, and produces control signals to activate the appropriate valves of an air-blast rejection device to take away certain particles selected by means of the analysing process.

2.3.1.9 Leaching

Uses: Extraction of rock salt, potash, gold (dissolution of native gold in cyanide solutions) and silver, uranium ore (dissolution of uraninite in carbonate solutions), copper and also residual substances.

Principles and construction: Leaching is a method where valuable minerals are selectively dissolved from a material by a lixiviant, normally aqueous solution, resulting in a rich solution (with high concentration of valuable compounds). Afterwards, the valuable mineral needs to be recovered, for instance by precipitation. The valuable mineral or compound can appear in the material being leached in at least three physical forms: as free particle, as multiphase particle in which the valuable mineral is exposed on at least one side to the lixiviant, and as inaccessible material surrounded by gangue material. In the first two cases, the valuable mineral can be directly leached.

There are several techniques of leaching. These can be grouped into fixed bed procedures, such as percolation leaching, heap leaching, and in-situ leaching, as well as leaching in a pulp in movement such as in agitation leaching (tank leaching) and pressure leaching. There is also a 'biological leaching' which uses the bacteria thiobacillus ferrooxidans and thiobacillus thiooxidans.

Figure 2.18: Heap leaching

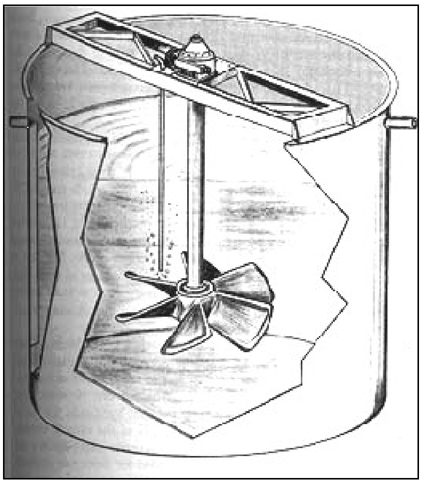


Figure 2.19: Leaching tank

2.3.1.10 Dewatering

Thickening

Uses: Thickening is extensively applied in pre-dewatering of concentrates and in tailings dewatering for water recovery, due to its comparatively low cost and high capacities compared to filtering. Intermediate thickening is also applied in several mineral processing techniques.

Principles and construction: Thickening is a sedimentation process that results in a large increase in the concentration of the suspension and in the formation of a clear liquid. Thickeners are tanks from which the settled and thickened solids are removed at the bottom as an underflow and the clear liquid flows to an overflow point or launder system at the top. They may be batch units, such as the baffle-plate thickener, or continuous units. Continuous thickeners are normally constructed of a cylindrical tank made of steel (mainly less than 30 m in diameter), concrete or a combination of both with the depth ranging from approx. 1 to 7 m and the diameter from approx. 2 to 200 m. In the tank, there will be one or more rotating radial arms, each possessing a series of blades. These blades rake or scrape the settled solids towards the underflow withdrawal point. There are several types of continuous thickeners, for instance bridge thickeners, centre pie thickeners, traction thickeners, tray thickeners and high-capacity thickeners.

General technical data:
Continuous thickener:

- diameter: 2 – 200 m
- diameter/height:
  - small thickener: 1:1 up to 4:1
  - large thickener: up to 10:1

Baffle-plate thickener:

- effective surface lamella thickener: up to 600 m^2

Figure 2.20: Continuous thickener

Filtering

Uses: Dewatering of flotation concentrate, magnetic concentrates and several non-metallic minerals; removing pregnant solution from the leached solid in the cyanide process; washing the dewatered filter cake; clarifying decanted pregnant solution and in collecting precipitate.

Principles and construction: Filtration can be regarded as the process of separating solids from a liquid by means of a permeable septum, which holds the solid but allows the passage of liquid. Filtration often follows thickening, whereby the thickened pulp may be fed to storage agitators where sometimes flocculants are added and from where it is drawn off at a uniform rate to the filters. The most common types of filters employed in mineral processing are 'cake filters' in which the principal requirement is the recovery of large solid amounts from quite concentrated slurries. Cake filters are classed essentially as 'vacuum filters' and 'pressure filters', depending on the means employed for effecting the required pressure difference on the two sides of the porous medium. They may be also 'batch' or 'continuous' types.
The most frequently utilised types of pressure filters are 'filter presses', which are constructed in two main forms: 'the plate-and-frame filter press' and 'the chamber press'. The operating pressure in the plate and frame press can achieve 25 bar.
On the other hand, there are several types of vacuum filters, such as 'continuous drum filters' (made in a wide variety of designs), 'continuous disk filters' and 'horizontal belt filters'.

General technical data:

- plate-and-frame filter press:
  - plate size: up to 2 x 2 m
  - filter surface: maximum 1500 m^2 per machine
- continuous drum filter:
  - filter surface: approximately up to 120 m2
- continuous disk filter:
  - larger filter surface per volume unit: approximately up to 200 m^2

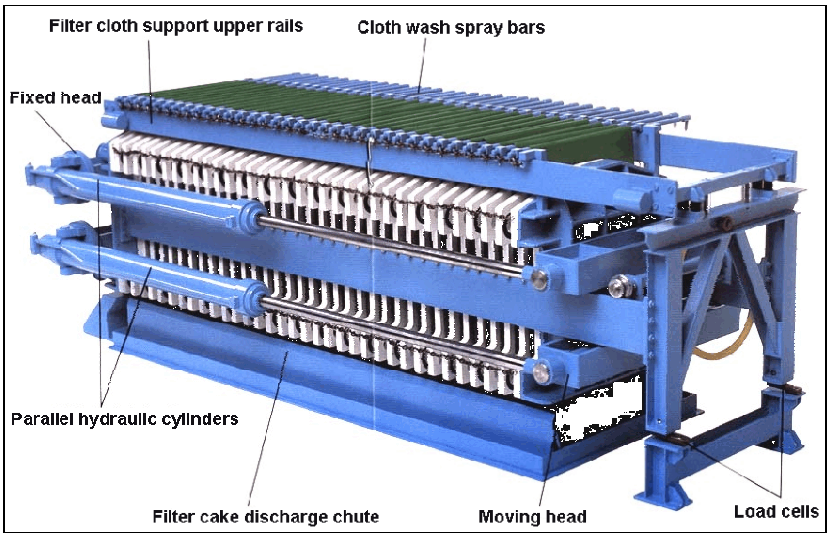


Figure 2.21: Plate-and-frame filter press

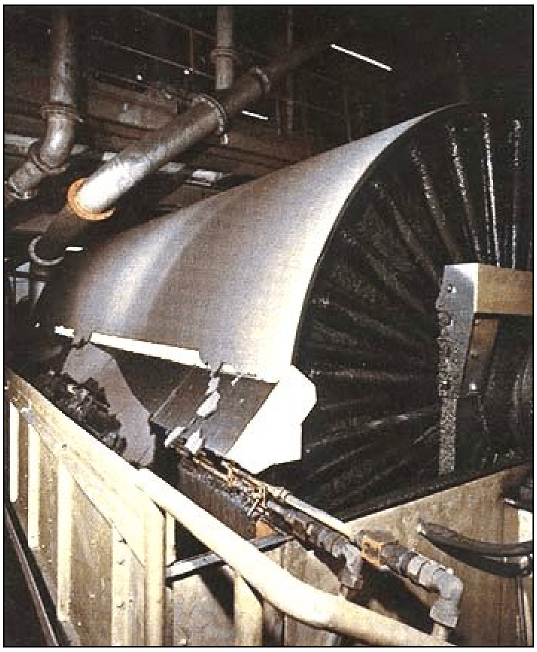


Figure 2.22: Drum filter

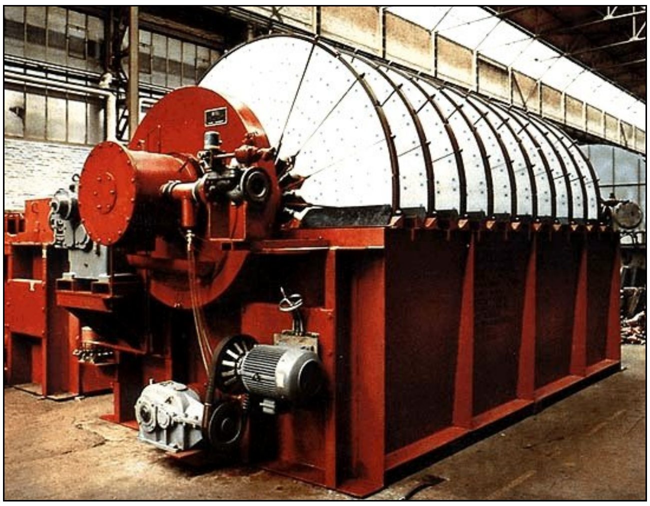


Figure 2.23: Disk filter

Centrifuging
As an alternative to plate-and-frame filter presses, solid bowl centrifuges are used for dewatering.
General technical details include:

- drum diameter: up to 1100 mm
- drum length: up to 3300 mm
- throughput: max. 15 tonnes (dry basis)/hour

Dewatering by means of centrifuges results in a lower solid contents compared to plate-and-frame filter presses. Therefore, the dewatered material behaves more like a jelly than a cake. Flocculants have to be added for optimal results.

2.3.2 Reagents

Flotation reagents
Flotation reagents are the various chemical compounds used in the flotation process, which assure the appropriate conditions for the operation. They are selectively employed according to the ore type. They comprise 'collectors', 'frothers' and 'regulators'.

- collectors: are 'surface-active substances', i.e. organic compounds which adsorb on mineral surfaces, leaving them hydrophobic and making bubble adhesion possible They are divided into ionising or non-ionising compounds. Non-ionising collectors are practically insoluble and cover the surfaces of minerals with an high natural hydrophobicity (mainly coal), to strengthen its water-repellent properties. Ionising collectors dissolve in water and have a heteropolar structure, that means a non-polar group (hydrocarbon group) which has water-repellent properties, and a polar group which attaches to the mineral surface. The type of polar group classifies the collector: anionic (carboxylic, sulphates, sulphonates, xanthates and dithiophosphates), cationic (amine) collectors or amphoteric collectors

- frothers: are reagents that help to keep the stability of the froth, e.g. acids, amines and alcohols

- regulators or modifiers: are reagents which regulate the flotation operation. They are classed as activators, depressants or pH modifiers. Activators allow collector adsorption on minerals by changing the chemical character of the mineral surfaces. Such substances are generally soluble salts. Depressants (water glass, starch, quebracho, etc.) conversely render minerals hydrophilic, thereby stopping them floating. pH modifiers (such as lime, soda and caustic soda for alkalinity, and predominantly sulphuric acid for acidification) control the pH of the pulp, which has an important influence on most process steps (collector and depressant adsorption, etc.)

- flocculants: in German hard coal processing plants, flocculants for industrial use are applied based either on polyacrylates or on polyacrylamides.

2.3.3 Effects on tailings characteristics

Process step	Tailings characteristics								
	Grain size distri-bution	**Generation of fines**	**Specific surface**	**% solids**	**Reagents**	**pH**	**ARD influence**	**Surface properties**	**Particle shape**
Comminution	X	X[1]	X	X[2]	-	-	X	X	X
Screening	X	X[3]	-	-	-	-	-	-	-
Classification	X	X	-	X	-	-	X	-	-
Gravity conc.	-	-	-	X	-	-	X	-	-
Flotation	-	-	-	X[4]	X[5]	X[6]	X	X	-
Magnetic sep.	-	-	-	-	-[7]	-	X	-	-
Electr. sep.	-	-	-	-	X	-	X	X	-
Sorting	-	-	-	-	-	-	X	-	-
Leaching	-	-	-	X	X	X	-	X	-
Thickening	-	-	-	X[8]	X[9]	-	-	X	-
Filtering	-	-	-	X	X	X[10]	-	X	-

1) e.g. agitated mill generates more fines than ball mill
2) crushing dry, tumbling mills and agitated mills wet process
3) excessive screening can lead to generation of fines
4) flotation is a wet process with about 30 - 40 % solids in metal ore processing and 5 - 15 % solids in coal processing, in most cases water will have to be added
5) see 2.3.2 for details
6) raised or lowered
7) usually no reagents, however, for fines sometimes dispersion agents are used for deagglomerisation
8) obviously % solids are reduced by thickening
9) often use of flocculants (see 2.3.2 for details)
10) e.g. by using flocculants such as aluminium sulphate or lime, which change pH

Table 2.2: Effects of mineral processing steps on tailings characteristics

Screening and classification have an indirect influence on the grain size distribution and generation of fines if they are used in a closed-circuit with grinding, such as a ball mill in closed circuit with a cyclone. In this example, the ball mill discharge is fed to a cyclone. The cyclone overflow is of such a grain size that the desired mineral is liberated for subsequent separation or concentration. The cyclone underflow needs further size reduction and is led back to the ball mill. In this example, the classifier ensures that overgrinding in the mill does not occur.

It should be noted that for magnetic (if wet) and gravity separation, the percentage of solids may have to be adjusted, hence the process steps also change the percentage of solids. However, this does not impact upon the tailings management if the tailings go through a thickener, prior to discharge to the pond.

The column on 'ARD influence' highlights process steps that either alter the accessibility to sulphides (i.e. comminution) or change the sulphide content in the tailings (for instance, electrostatic separation can remove part of the pyrite). The ARD influence of flotation can be both positive (sulphides removed to the concentrate) and negative (other minerals removed and the sulphides remain in the tailings). Comminution mainly has the effect of making sulphide minerals more accessible and, thereby, enhances ARD generation.

It is obvious that comminution changes the surface properties. However, in fact all process steps where reagents are added influence the surface properties.

2.3.4 Techniques and processes

2.3.4.1 Alumina refining

Alumina refining is the process that uses bauxite as a raw material to produce alumina. Alumina is a white granular material and is properly called aluminium oxide. The Bayer refining process used by alumina refineries worldwide involves four steps - digestion, clarification, precipitation and calcination.

Alumina is converted into aluminium via smelting, and these techniques are described in the BREF on non-ferrous metals industries. [35, EIPPCB, 2001]

The digestion (dissolution) of aluminium 'hydrate' (e.g. $Al_2O_3{\cdot}3H_2O$) from the bauxite is carried out under pressure in high temperature (around 250 ºC) sodium hydroxide. The insolubles, sand and red mud, are separated by cycloning, decantation, and, after washing and filtration, are deposited in the TMF. The aluminium hydrate is precipitated as a white slurry and dried (calcined) to produce alumina (Al_2O_3), as a white crystalline product in particles of about 90 µm size. Six to four tonnes of bauxite are needed to produce two tonnes of alumina and subsequently one tonne of aluminium [22, Aughinish,].

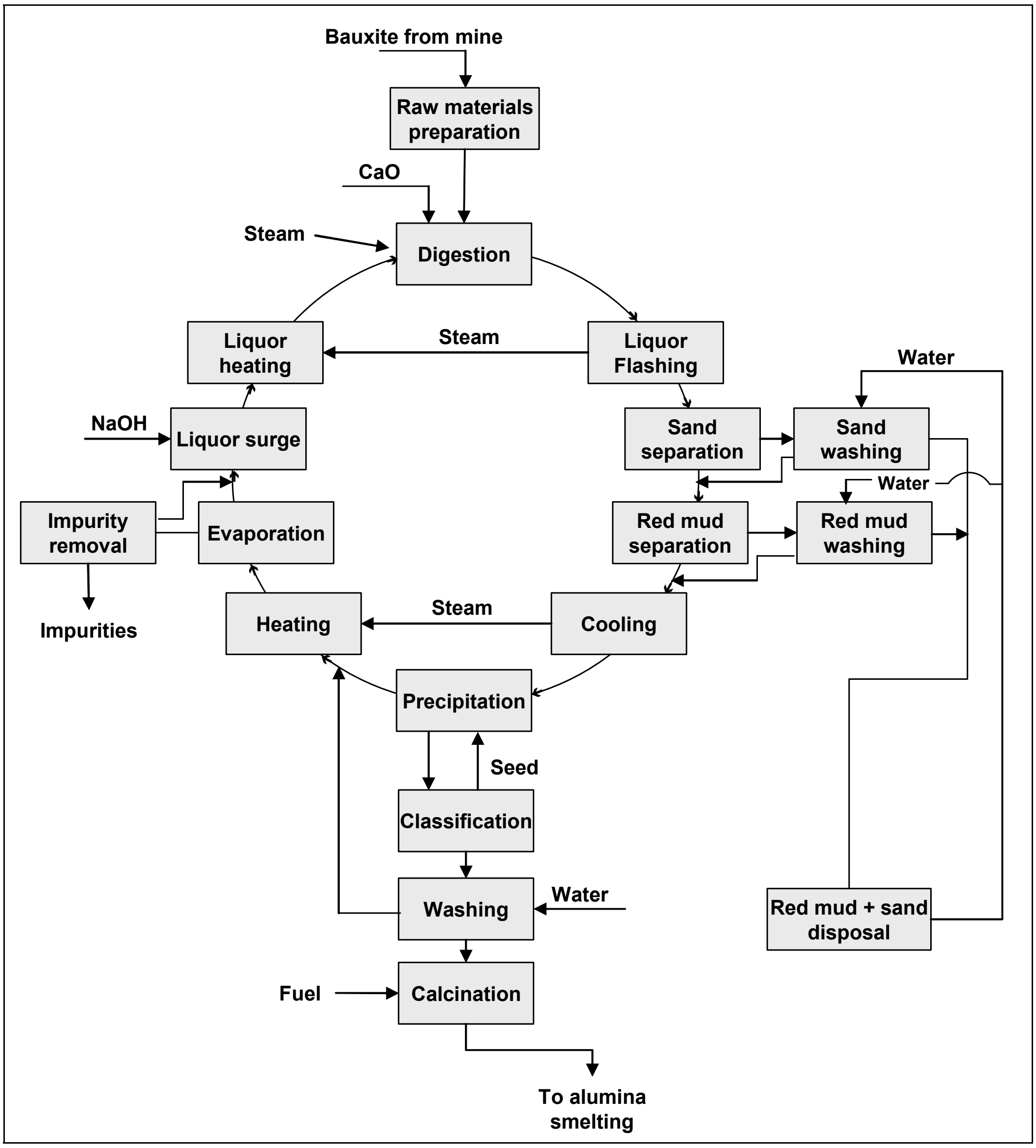


Figure 2.24: Typical flow sheet of Bayer-process

This process is normally carried out close to the mine site but there are sites in Europe where bauxite is converted to alumina at the same site as the aluminium smelter or at stand-alone alumina refineries.

More information about alumina refinery is available at:
http://www.world-aluminium.org/production/refining/.

2.3.4.2 Gold leaching with cyanide

Strictly speaking leaching is less a typical mineral processing technique than a hydrometallurgical process. However, for gold leaching it is applied to run-of-mine ore or is integrated into the other mineral processing steps (e.g. after comminution and gravity separation or flotation). Therefore leaching is generally considered to be part of mineral processing. Although other minerals may be leached and lixiviants other than cyanide are used (e.g. salt can be leached or dissolved with water, copper may be leached with sulphuric acid), due to the high toxicity of cyanide and the public concern about its use in the mining sector, this chapter will focus on the use of cyanide in the leaching of gold. However, it should be noted that cyanide may also be used in the flotation of sulphides, as a depressant for pyrite (FeS_2).
The following text on the use of cyanide for the leaching of gold is taken from the "International cyanide management code for the manufacture, transport and use of cyanide in the production of gold" (www.cyanidecode.org), unless otherwise stated. From this website, information about cyanide chemistry and sampling and analytical methods has been dowloaded and attached in Annex 1.

Use of cyanide in the gold industry
Gold typically occurs at very low concentrations in ores, i.e. less than 10 g/t or 0.001 %. At these concentrations the use of hydrometallurgical extraction processes, i.e. based on aqueous chemistry, are the only economically viable methods of extracting the gold from the ore. Typically hydrometallurgical gold recovery involves a leaching step during which the gold is dissolved in an aqueous medium, followed by separation of the gold bearing solution from the residues or adsorption of the gold onto activated carbon and finally gold recovery either by precipitation or elution and electrowinning (see the following figure).

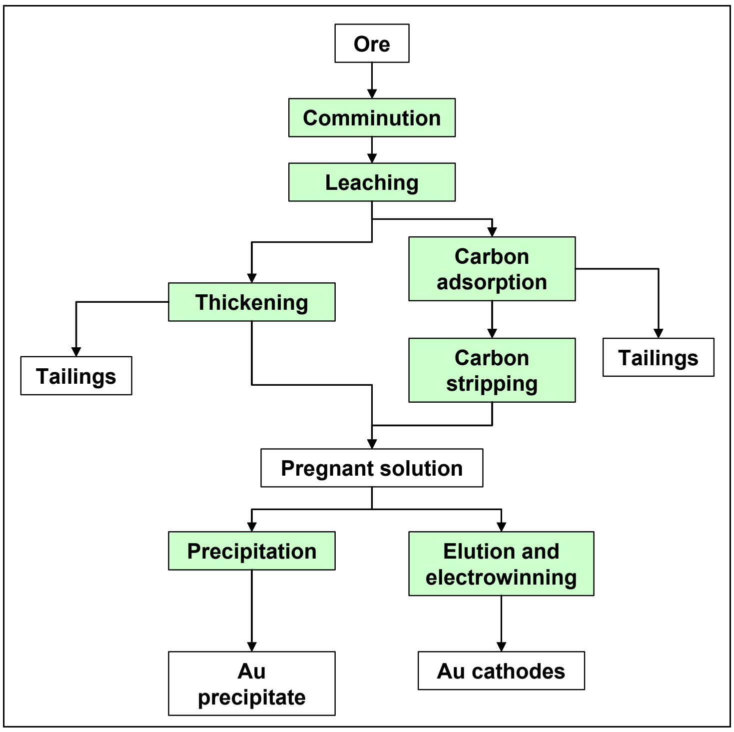


Figure 2.25: The principles of gold recovery by leaching

Often a gravity separation circuit is incorporated into this process after comminution to recover the sufficiently coarse gold particles (>30 µm) prior to leaching. The use of gravity separation in the field of gold recovery is rapidly advancing into ever smaller particles sizes (see Chapter 6).

Gold is one of the noble metals and, as such, is not soluble in water. The presence of a complexant, such as cyanide, which stabilises the gold species in solution, and an oxidant, such as oxygen, are required to dissolve gold. The amount of cyanide in solution required for dissolution may be as low as 350 mg/l or 0.035 % (as 100 % NaCN).

Alternative complexing agents for gold, such as chloride, bromide, thiourea, and thiosulphate are available but form less stable complexes and, thus, require more aggressive conditions to dissolve the gold. These reagents are often more expensive to use and/or also present risks to health and the environment. This explains the dominance of cyanide as still the primary reagent for the leaching of gold from ores.

Ore preparation

The aim of ore preparation is to present the ore to the lixiviant (the aqueous cyanide solution) in a form that will ensure optimum economic recovery of the gold. The first step in ore preparation is crushing and grinding, which reduces the particle size of the ore and liberates the gold for recovery.

Ore that contains free gold may not yield a sufficiently high recovery by means of cyanide leaching only, and may require a gravity recovery process where the free gold is recovered before the remainder of the gold is subject to cyanide leaching.

Gold bearing ores that contain gold associated with sulphide or carbonaceous minerals require additional treatment, besides size reduction, prior to gold recovery. Gold recovery from sulphide ore is poor because the cyanide preferentially leaches the sulphide minerals rather than the gold, and cyanide is consumed by the formation of thiocyanate. These ores are subject to a concentration process, such as flotation, followed by a secondary process to oxidise the sulphides, thus limiting their interaction with the cyanide during the gold leach. Carbonaceous minerals adsorb the gold after it has been dissolved. This is prevented by oxidising the ore prior to leaching. The leaching process may also be modified to counter this effect, by the addition of activated carbon to preferentially adsorb the gold.

Leaching with aqueous cyanide solutions

Gold is leached in aqueous cyanide by oxidising it with an oxidant such as dissolved oxygen and complexing it with cyanide to form a gold-cyanide complex. This complex is very stable and the cyanide required is only slightly in excess of the stochiometric requirement. However, in practice, the amount of cyanide used in leach solutions is dictated by the presence of other cyanide consumers and the need to increase the rate of leaching to acceptable levels.

In practice, the typical cyanide concentrations used range from 300 to 500 mg/l (0.03 to 0.05 % as NaCN), depending on the mineralogy of the ore. The gold is recovered by means of either heap leaching or agitated pulp leaching.

With heap leaching, the ore or agglomerated fine ore is stacked in heaps on a pad lined with an impermeable membrane. The term ‘dump leaching’ is sometimes applied to heap leaching of uncrushed ore. Cyanide solution is introduced to the heap by sprinklers or a drip irrigation system, the solution percolates through the heap, leaching the gold from the ore. The gold bearing solution is collected on the impermeable membrane and channelled to storage facilities for further processing. Heap leaching is attractive due to the low capital cost involved, but is a slow process and the gold extraction efficiency is also relatively low.

In a conventional milling and agitated leaching circuit, the ore is milled in semi-autogenous, ball or rod mills to the consistency of sand or powder. The milled ore is conveyed as a slurry to a series of leach tanks. The slurry is agitated in the leach tanks, either mechanically or by means of air injection, to increase the contact of cyanide and oxygen with the gold and to enhance the efficiency of the leach process. As mentioned earlier, the cyanide dissolves gold from the ore and forms a stable gold-cyanide complex.

The pH of the slurry is raised to pH 10 - 11 using lime, at the head of the leach circuit to ensure that when cyanide is added, hydrogen cyanide gas is not generated and the cyanide remains in solution and hence available to dissolve the gold. The slurry may also be subject to other preconditioning, such as pre-oxidation at the head of the circuit, before cyanide is added.

Where oxygen instead of air is used as the oxidant, it has the advantage of increasing the leach rate and also decreasing cyanide consumption due to the inactivation of some of the cyanide consuming species present in the slurry.

Where carbon is used to recover the dissolved gold, highly activated carbon is introduced into the process, either directly into the leach tanks (referred to as Carbon-in-Leach - CIL) or in separate tanks after leaching (referred to as Carbon-in-Pulp - CIP). The activated carbon adsorbs the dissolved gold from the solution component of the leach slurry, thereby concentrating it onto a smaller mass of solids. The carbon is then separated from the slurry by screening and subjected to further treatment to recover the adsorbed gold, as described below.

Where carbon is not used to adsorb the dissolved gold in the leach slurry, the gold bearing solution must be separated from the solids component of the slurry, utilising filtration or thickening units. The resultant solution, referred to as pregnant solution, is subjected to further treatment (other than by carbon absorption) to recover the dissolved gold, as discussed under gold recovery.
The material from which the gold has been removed by adsorption or liquids/solids separation is referred to as tailings. The tailings are either dewatered to recover the water and residual cyanide reagent, treated to either neutralise or recover cyanide, or sent directly to the TMF (see Section 3.1.6.3).

Recovery of dissolved gold

The gold is recovered from the solution by using cementation on zinc powder (the so-called Merrill-Crowe process) or by first concentrating the gold using adsorption on activated carbon, followed by elution and either cementation with zinc or electrowinning. For efficient cementation, a clear solution is required, which is typically prepared by filtration or counter current decantation. These are capital-intensive processes and have been superseded by processes using adsorption of the dissolved gold onto activated carbon. Adsorption is achieved by contacting the activated carbon with the agitated pulp. This can be done while the gold is still being leached with the Carbon-in-Leach or CIL-process, or following leaching with the Carbon-in-Pulp or CIP-process. Activated carbon in contact with a gold containing pulp can typically recover more than 99.5 % of the gold in the solution in 8 to 24 hours. The loaded carbon is then separated from the pulp using screens that are air or hydro-dynamically swept to prevent blinding by the near-sized carbon particles. This separation of ore particles (typically <100 µm) from the coarser carbon particles (>500 µm) is a lot less capital intensive than the filtration needed when using the Merrill-Crowe technique).

The fine barren ore, i.e. the tailings, is then either thickened to separate the cyanide containing solution for recovery or destruction of the cyanide, or sent directly to the TMF, where the cyanide containing solution is often recycled to the leach plant.

The gold adsorbed on the activated carbon is recovered from the carbon by elution, typically with a hot caustic aqueous cyanide solution. The carbon is then regenerated and returned to the adsorption circuit while the gold is recovered from the eluate using either zinc cementation or electrowinning. This gold concentrate is then calcined, if it contains significant amounts of base metals, or directly smelted and refined to gold bullion that typically contains about 70 – 90 % gold. The bullion is then further refined to either 99.99 % or 99.999 % fineness, using chlorination, smelting and electrorefining. Recently developed processes utilise solvent extraction to produce high purity gold directly from activated carbon eluates, or following intensive leaching of gravity concentrates.

Process operation and the environment
The following are sources of cyanide emissions to the environment:

- CN to air as HCN
- seepage from tailings ponds
- tailings pond discharges required to manage overall water balance.

It is part of normal operation to attempt to optimise process economics. This coincides with the objective of minimising cyanide impact on the environment and cyanide consumption. Process economics are sensitive to the amount of cyanide consumed in the process. Increased cyanide addition may have a 'double-barrelled' effect, meaning the operating costs increase through the extra amounts of cyanide that have to be purchased as well as because of the higher amounts of cyanides that will have to be destroyed or recycled prior to effluent discharge. Cyanide classified as 'consumed' from a process point of view may still be active from an environmental perspective, for instance as may be the case with copper cyanide complexes .

2.4 Tailings and waste-rock management

There are many options for managing tailings and waste-rock. The most common methods are:

- dry-stacking of thickened tailings
- dumping of more or less dry tailings or waste-rock onto heaps or hillsides
- backfilling of tailings or waste-rock into underground mines or open pits or for the construction of tailings dams
- discarding of tailings into surface water (e.g. sea, lake, river) or groundwater
- use as a product for land use, e.g. as aggregates, or for restoration
- discarding of slurried tailings into ponds.

Waste-rock is either managed on heaps or is sometimes dumped on existing hillsides.

The ways in which these different techniques are applied will be discussed in this section.

2.4.1 Characteristics of materials in tailings and waste-rock management facilities

This section has been taken from the UK "Spoil heaps and lagoons" technical handbook [130, N.C.B., 1970].

2.4.1.1 Shear strength

The shear strength is the most important characteristic of any tailings or waste-rock in the design of a heap or dam. Normally the appropriate shear strength parameters necessary to carry out a stability analysis are those related to the effective stress, i.e. the effective cohesion and the effective angle of shearing resistance. Comparatively small variations in the shear strength parameters used may have a significant impact on the safety factor. Therefore, strength tests are carried out on a reasonable number of samples.

2.4.1.2 Other characteristics

Other important characteristics relevant for the stability of a facility are:

- particle size distribution: as this influences shear strength
- density
- plasticity
- moisture content
- permeability. According to their hydraulic conductivity or coefficient of permeability k (in m/s), tailings and waste-rock can be classified in five groups according to DIN 18130 part 1:
  - very high permeability: $>1 \times 10^{-2}$
  - high permeability: 1×10^{-4} - 1×10^{-2}
  - permeable: 1×10^{-6} - 1×10^{-4}
  - low permeability: 1×10^{-8} - 1×10^{-6}
  - very low permeability: $<1 \times 10^{-8}$
- consolidation: the amount and rate of settlement of tailings or waste-rock under load are related to the consolidation characteristic of the soil
- porosity.

2.4.2 Tailings dams

Tailings dams are surface structures in which slurried tailings are managed. This type of TMF is typically used for tailings from wet processing. Ponds consist of 20 – 40 % solids by weight, but levels from 5 – 50 % solids have been known.

The following figure shows a cross-sectional view of a tailings dam and illustrates the water cycle of this type of TMF.

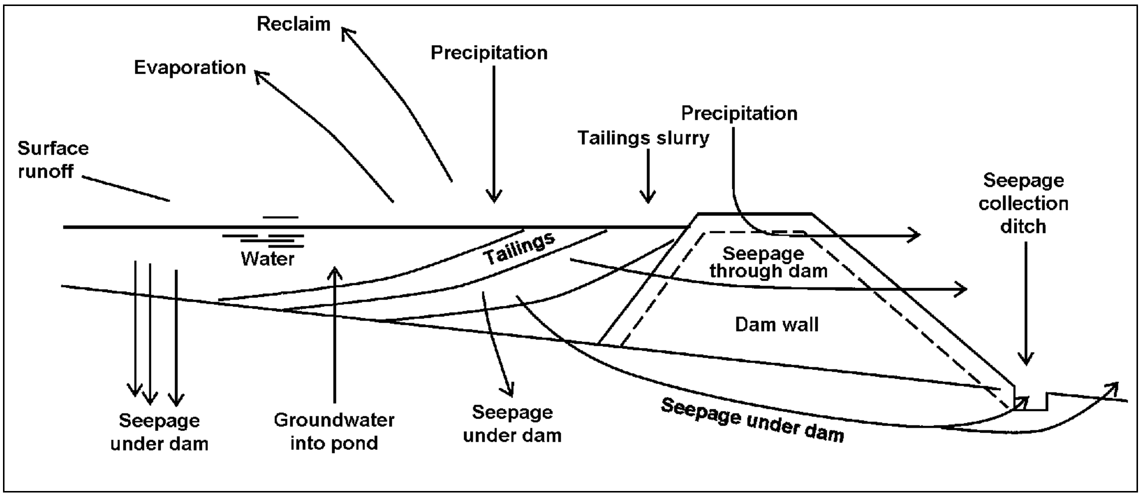


Figure 2.26: Dam water cycle changed from
Changed from [11, EPA, 1995]

The following section on tailings dams is mostly gathered from ICOLD Bulletin 106 [8, ICOLD, 1996]. Other references are mentioned where appropriate.

The vast majority of tailings are managed on land. This entails the selection of a tract of land on which the tailings are stored for an extended period while the tailings are being generated by the mineral processing plant and, unless reclaimed for further treatment, for an indefinite period thereafter. The deposit must be secure against physical damage from outflow and must not pollute the surrounding area, neighbouring water courses, the groundwater, or the atmosphere.

Since the tailings are conveyed as slurry from the plant and may remain as a suspension, or since, thus, may be capable of reverting to a fluid, the deposited mass requires confinement to the extent necessary to prevent the flow of the material out of the designated area. In most tailings ponds, the solids settle out of the slurry after discharge and the pond is, therefore, composed of settled solids and free water. This may be supplemented by natural run-off, in-flowing groundwater or direct precipitation. The free fluid may be returned to the processing plant for re-use, stored in the impoundment for future use, or removal by evaporation or it may be discharged into surface water courses, often after undergoing treatment.

The basic arrangements of tailings dams may be classified as:

- existing pit
- valley site
- off valley site
- on flat land.

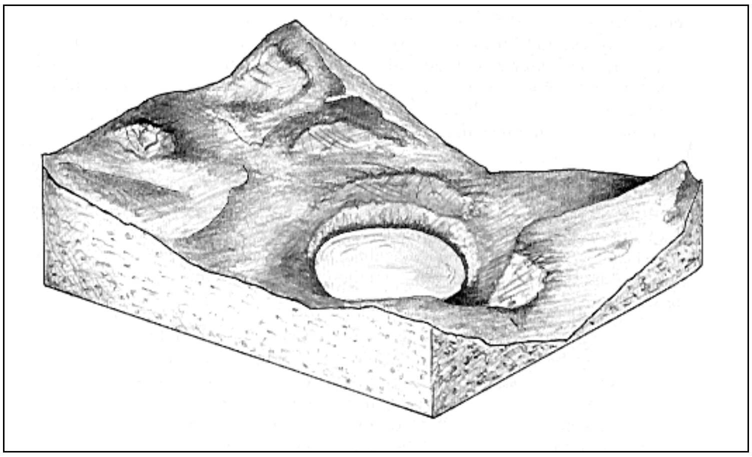


Figure 2.27: Illustration of a tailings pond in an existing pit
[8, ICOLD, 1996]

The following gives an actual example of this type of TMF.

Figure 2.28: Picture of a tailings pond in an existing pit

The following two figures illustrate a valley site and an off-valley site tailings pond.

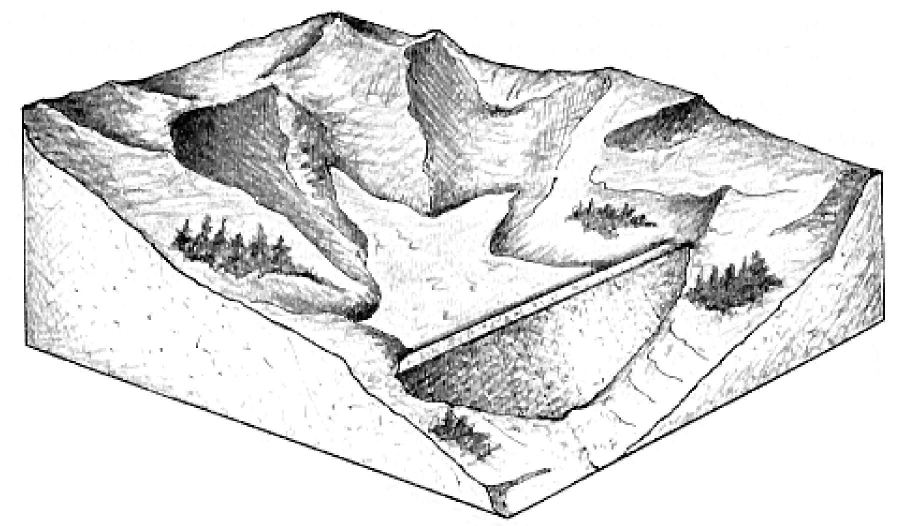


Figure 2.29: Illustration of tailings pond on a valley site
[8, ICOLD, 1996]

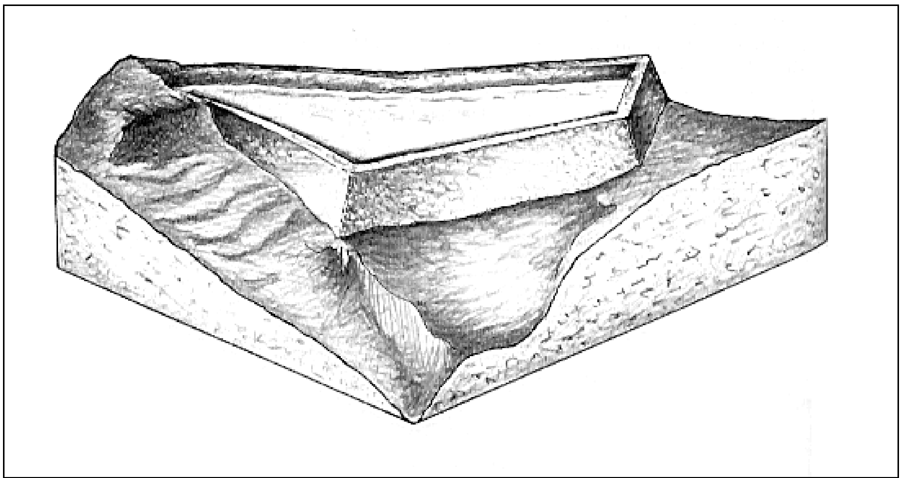


Figure 2.30: Illustration of an off-valley site tailings pond
[8, ICOLD, 1996]

If a tailings pond is built on flat land it is often referred to as a paddock. The following picture gives an impression of paddocks used in South African gold mining operations.

Figure 2.31: Tailings pond on flat land (Courtesy of AngloGold, South African Division)

For each tailings impoundment, several activities need to be considered, including:

- tailings delivery from the mineral processing plant to the tailings dam
- dams to confine the tailings
- diversion systems for natural run-off around or through the dam
- deposition of the tailings within the dam
- evacuation of excess free water
- protection of the surrounding area from environmental impacts
- instrumentation and monitoring systems to enable surveillance of the dam
- long-term aspects (i.e. closure and after-care).

Some of these activities will be discussed in the following sections. Also some aspects of seepage flow and design flood considerations will be introduced. These two aspects have an impact on several of the activities listed above.

2.4.2.1 Delivery systems for slurried tailings

Slurry transport from the plant to the TMF is usually undertaken by pipeline. In some cases, open channel conveyance may be used, as it is cheaper. The pipeline is seldom buried. Occasionally, the slurried tailings are transported from the mineral processing site to the TMF by trucks.

2.4.2.2 Confining dams

The construction material and methods used in forming the dam vary widely to accommodate the particular needs of the selected site, the availability of materials and the financial and operating policies of the entire operation.

Typically, dams are subdivided into three parts:

1. an upstream section which is capable of retaining the tailings without excessive penetration/erosion by the tailings themselves (e.g. compacted sand)
2. a middle section, or core, which provides a passage for seepage water through the structure in a controlled manner and which will not break down or become blocked by fine material (e.g. rock or crushed filter stone) and
3. a downstream section which provides toe strength and stability and which will remain 'dry' under all circumstances (e.g. sand compacted to a high density). In some circumstances, it may be necessary to incorporate artificial membranes (filter cloths) between the main sections of the structure where there is a risk of high seepage and the movement of fine material.

The dam types may be classified as follows:

- non-permeable (water-retention type) dams
  - conventional dam
  - staged conventional dam
  - staged dam with upstream low permeability zone.

- permeable dams
  - dam with tailings low permeability core
  - dams with tailings in structural zone
  - upstream construction using beach or paddock.

These types will be briefly discussed below.

Note that the term **beach** used in conjunction with the management of slurried tailings in a pond means the area of tailings resulting from the settled solid fraction of a tailings slurry in a pond not covered by free water between the edge of free water and the crest of the dam.

The purpose of a beach is to establish an area of 'dry' tailings against the upstream face of retaining dams for two important considerations:

1. to prevent water from reaching the crest of the dam where it could cause erosion of the inside face, or more seriously, lead to excessive leakage through the dam with the subsequent risk of 'piping' and possible damage/collapse of the structure
2. to allow 'natural' separation of the coarser and finer particles of the tailings. Where tailings are discharged into a dam by suspension in water (and most are) the larger sized particles tend to settle out more quickly. As these 'dry' out and consolidate, densities will generally increase over time, thereby adding to the overall stability of the structure as a whole.

The following picture shows an example of a beach at an alumina refinery's red mud pond. The dam's upstream face and crest can be seen on the left hand side and the free water on the right hand side. The red section in the middle is considered the 'beach'.

Figure 2.32: Example of a beach at an alumina refinery's red mud pond

Conventional dam

This type of dam is completely built before tailings are discharged at the site. Hence, tailings cannot be used to build the dam. Conventional dams are constructed where the confinement is to be effected for both tailings and free water during the whole period, from the start of tailings management to the end of the particular site selected.

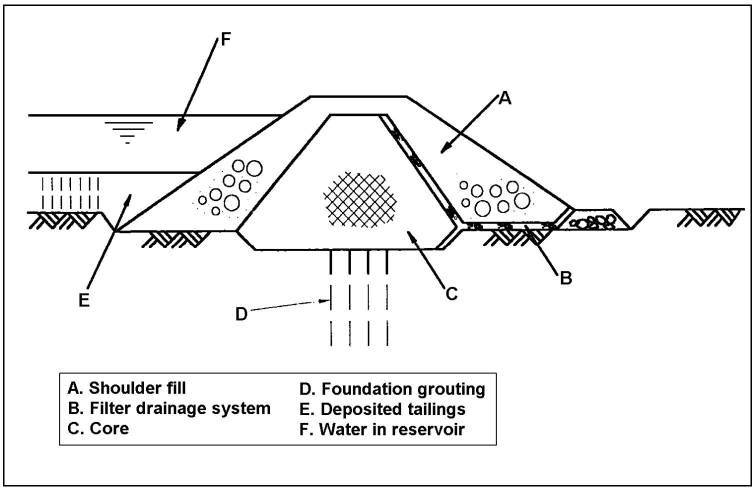


Figure 2.33: Conventional dam
[8, ICOLD, 1996]

The purpose of the shoulder fill is to increase the overall dam strength, but also to protect the core from erosion (wind and water) and from wave action from the free water.

A conventional central core section is illustrated in the above figure but the range of options is varied and similar to that for dams designed to confine water alone. In general though, the dam must be capable of:

- controlling the passage of water
- supporting the loads imposed by the tailings and water in the impoundment
- transmitting the seepage water effectively and without the passage of solids (filtration system).

Staged conventional dam

This is similar to a conventional dam but has a lower initial capital cost by staging the construction so that the costs are spread more evenly over the period of deposition.

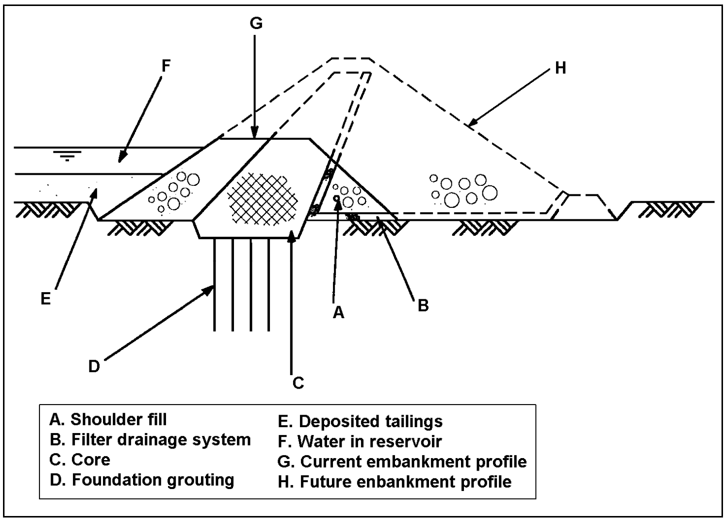


Figure 2.34: Staged conventional dam
[8, ICOLD, 1996]

Staged dam with upstream core

If the deposited tailings lie close to, or above, the level of the free water in the impoundment, the low permeability core zone of the dam may be located on its upstream face. This is possible because the core is protected against erosion and wave action by the tailings.

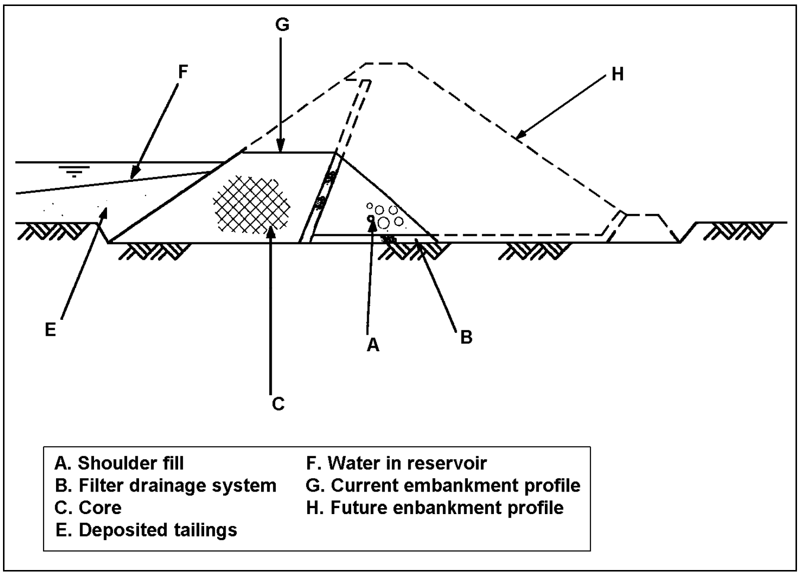


Figure 2.35: Staged dam with upstream low permeability zone
[8, ICOLD, 1996]

Dam with tailings low permeability core zone

Where all or part of the tailings deposition into the pond occurs from the dam a beach of tailings may be formed. It is then possible for the tailings beach alone to provide the less permeable zone of the system.

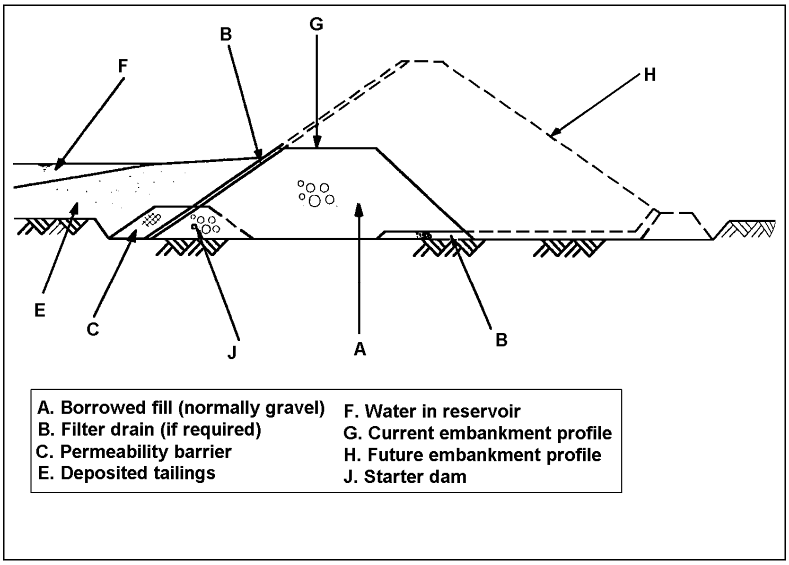


Figure 2.36: Dam with tailings low permeability core zone
[8, ICOLD, 1996]

This arrangement is only possible where the inflow of water will not allow the impoundment water level to rise above the uppermost level of the beach and, therefore, against the more pervious dam material. Therefore, continuous monitoring is required for this kind of arrangement.

For this arrangement, it is necessary to build a low impermeability barrier (C) into the starter dam, until the beach has developed far enough away from the dam itself.

Dam with tailings in structural zone

In this arrangement, tailings are not only used as a water barrier but also as construction material of the dam. In this case, typically the coarser hydrocyclone underflow is for the structural zone and the finer hydrocyclone is discharged into the pond forming the beach.

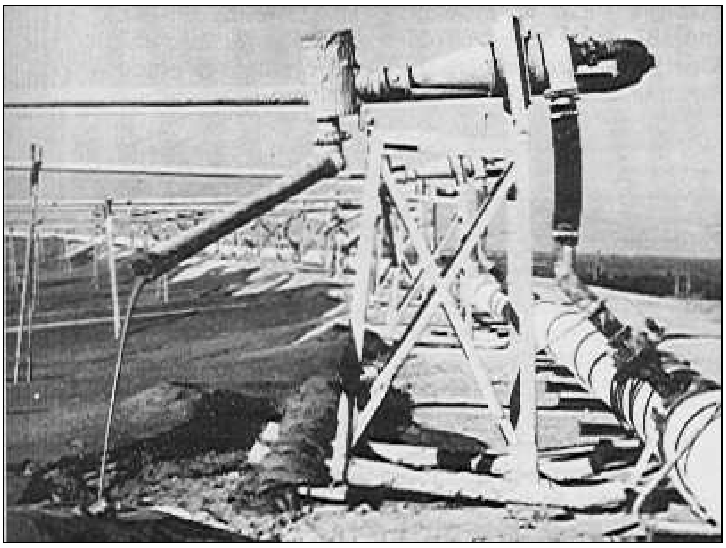


Figure 2.37: Row of hydrocyclones on the crest of a dam

For further information on hydrocyclones please refer to Section 2.3.1.3.2.

There are three main approaches when considering the progressive construction of this type of dam. These are:

- upstream method
- downstream method
- centreline method.

These methods allow for staged construction of the dam, which minimises start-up capital costs. The following figure illustrates these methods.

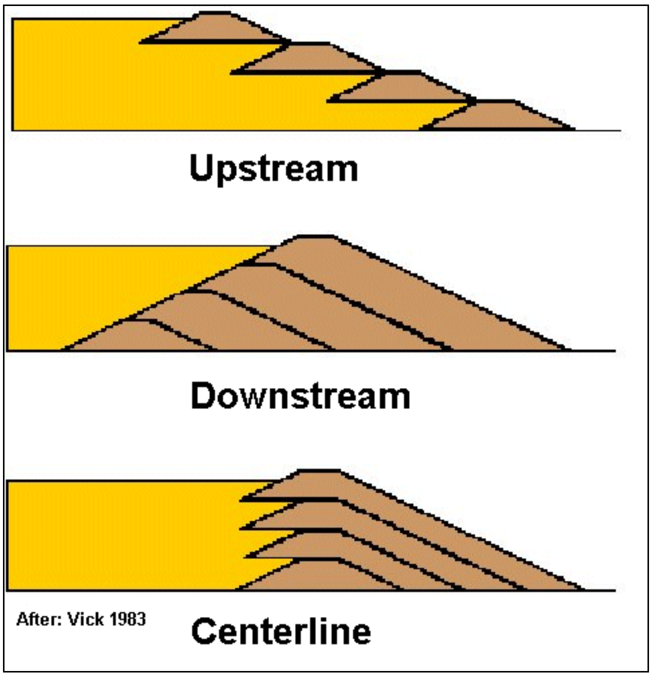


Figure 2.38: Types of sequentially raised dams with tailings in the structural zone
[11, EPA, 1995]

Upstream method using cycloned tailings

This method is very economical in the use of the coarser fraction of the tailings since only a thin outer zone of this material will result.

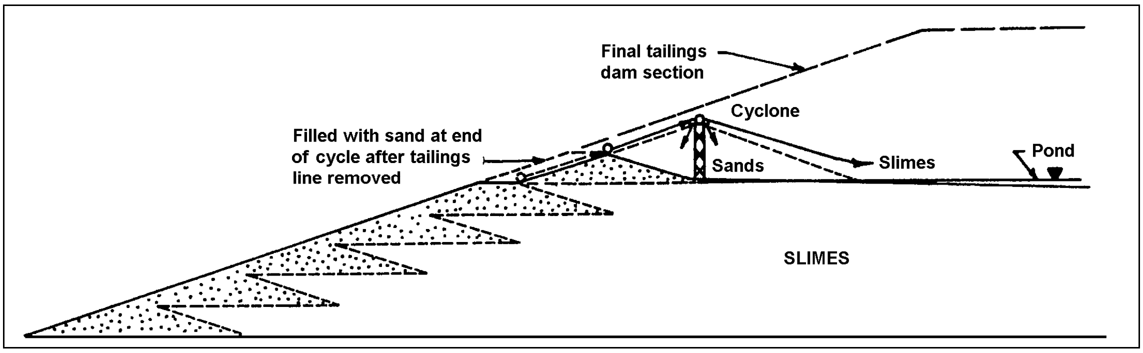


Figure 2.39: Upstream method using cycloned tailings
[11, EPA, 1995]

The following picture shows a dam built using the upstream method. The dam itself consists of borrowed rock-fill, different from the example above where cyclones tailings are used.

Figure 2.40: Dams raised using the upstream method at the Aughinish site

The main disadvantage of this method has, in the past, been the physical stability on the dam and its susceptibility to liquefaction. Care must be taken in order to control the phreatic surface, which can be achieved by correct drainage. Also, the exposed tailings, used to build the dam should not have ARD potential.

Downstream method

The coarse fraction of the tailings, separated by the hydrocyclone, may be used to form the complete structural portion of the dam or a large part of it. The size of hydrocyclone is selected such that a bank of them acting in parallel can deal with the tailings throughput. With the tailings delivery line and the bank of hydrocyclone offtakes located initially on the crest of the starter dam, the underflow is discharged downstream to form the dam, and the overflow is discharged into the impoundment, as illustrated in the following figure.

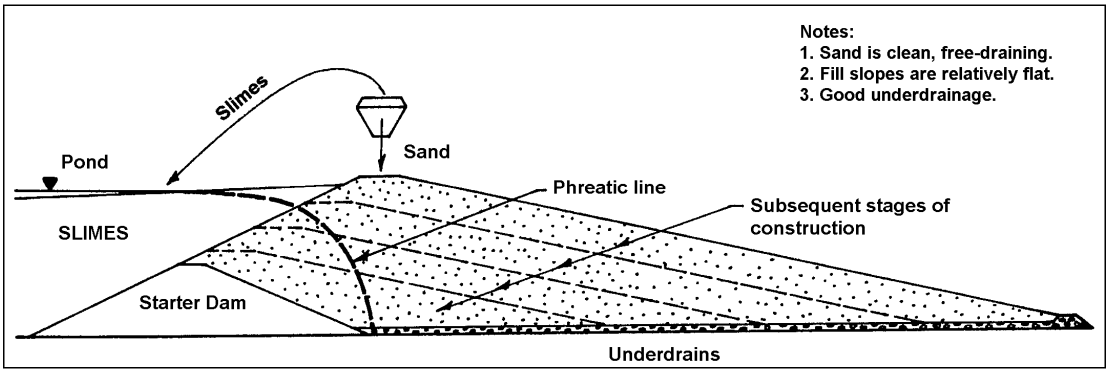


Figure 2.41: Downstream construction of a dam using hydrocyclones
[11, EPA, 1995]

This method is called the downstream method because as the dam height rises, the crest moves downstream.

Centreline method

The downstream method of construction entails the use of a considerable volume of coarse tailings for the dam, and an area of land under the footprint of the dam. Where the proportion of the coarse tailings separated out by cycloning is insufficient to permit the dam to keep ahead of the rise of the impoundment level, the tailings zone may need to be supplemented by a zone of borrowed material. As an alternative to this option, the upstream portion of the dam may be composed of the beach of deposited tailings. This is possible because the upstream face of the dam is progressively supported by the rise of tailings. The resultant structure is illustrated in the following figure and the method is generally termed the centreline method.

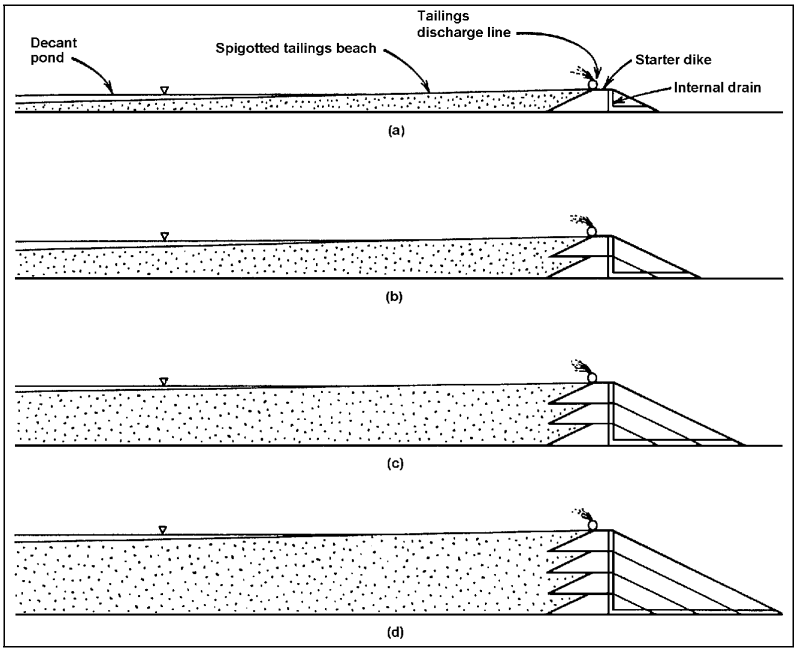


Figure 2.42: Centreline method
[11, EPA, 1995]

Upstream construction using beach or paddock
This traditional tailings dam construction method uses the beach instead of a hydrocyclone to size-sort the tailings. This method makes maximum use of the actual tailings itself for confinement, and may provide the cheapest system of tailings management. The system relies on the formation of a satisfactory beach by control of the deployment of the discharge arrangements and by control of the length of time material is discharged from each point.

2.4.2.3 Deposition in the impoundment

Hydraulic deposition
The tailings are pumped into the tailings pond with 5 to 50 % solids. In some applications, particularly where conventional dams are employed, the discharge of tailings into the impoundment can take the form of a **single-pointed open-ended discharge**. In other cases a more controlled deposition method may be desirable. This may incorporate **line or perimeter discharges** or the use of **hydrocyclones** [21, Ritcey, 1989]. For progressively built tailings dams, the discharge arrangements are dictated by the dam construction method selected.

The increase of density of deposited material is accelerated by the action of drainage and evaporation. Therefore storage efficiency can be increased by **deposition taking place on a beach**.

Thickened deposition
Thickened tailings have a solids content of over 50 %. This enables the storage efficiency, in terms of the storage volume to dam height, to be substantially increased, since the angle of deposition increases with the solids content of the tailings. Equipment used to thicken tailings are thickeners and/or filters.

Special techniques
For very fine tailings, **special techniques** may be employed, such us the addition of coarser particles or flocculants.

In some cases, it is necessary for all the tailings to be deposited under water (e.g. tailings with ARD potential or severe dust problems). This is referred to as **sub-aqueous** deposition.

2.4.2.4 Removal of free water

The aim throughout the development of the impoundment is usually to keep the pool of free water as low and as small as possible as a means of risk management. However, this needs to be balanced against several other objectives, e.g. tailings need a certain amount of time to settle within the pond. Also, in some cases the water has to remain in the dam for a certain period of time in order to allow deterioration of the process chemicals. Water saturation of the tailings may also be required to avoid dusting.

A good balance between the need to keep the pool low, and the contradicting requirements to leave a certain amount of water in the pond, may be utilisation of a clarification pond. This allows the settling of the fines slimes and deterioration of process chemicals, whilst the water level in the actual dam, containing the settled tailings, can be kept to a minimum.

The main requirement for successful removal of the water is the provision of an outlet arrangement, the effective level of which can be adjusted throughout the progressively increasing impoundment level, or of a pump, which can perform a similar function. The removed water is either returned to the mineral processing plant and/or, usually after treatment, discharged into natural water courses.

The outlet structure, or ‘decanting system’ as it is normally termed, is usually composed of two elements:

- an extendible intake, and
- a conduit to convey the discharge away from the dam.

The intake may take the form of a vertical tower, or a sloping chute founded usually in natural ground on a flank of the impoundment and occasionally on the upstream face of the dam.

The following figures show the three basic alternatives:

- decant tower
- decant chute
- pumped decant.

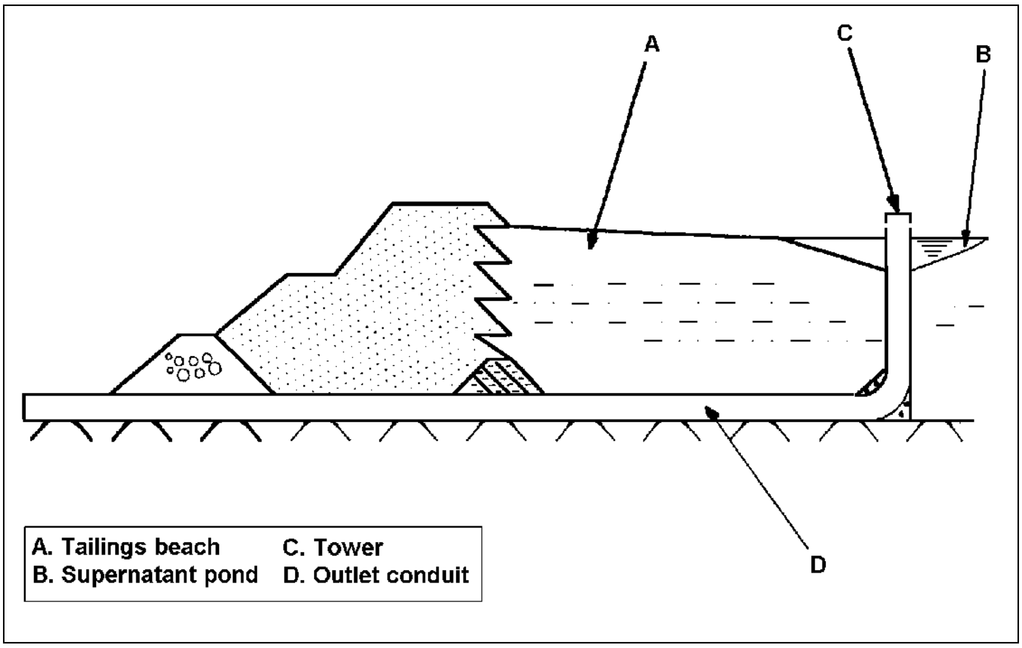


Figure 2.43: Tower decanting system
[8, ICOLD, 1996]

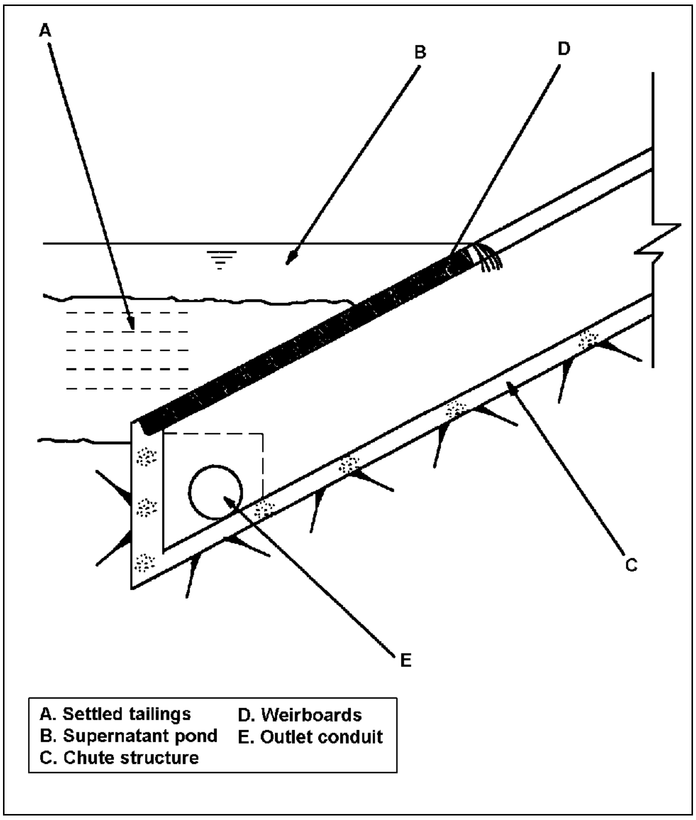


Figure 2.44: Chute decanting system
[8, ICOLD, 1996]

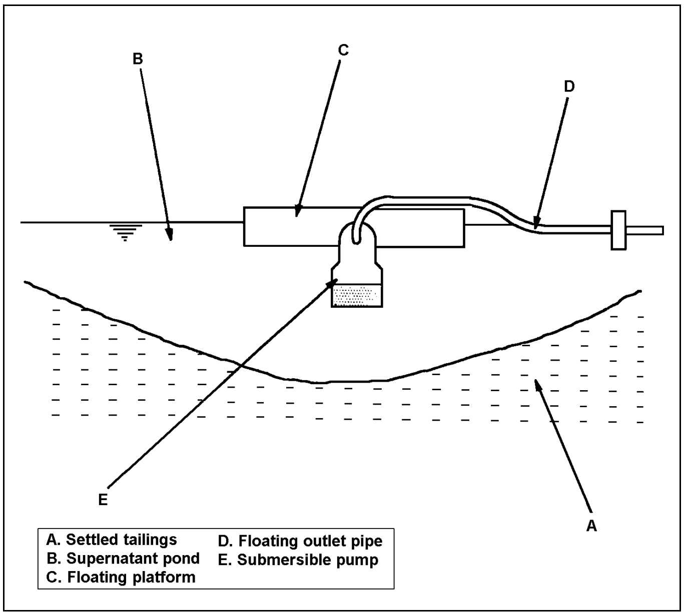


Figure 2.45: Pump barge
[8, ICOLD, 1996]

Other options are:

- drained pond or
- overflow systems:
  - within the dam
  - around the dam.

In addition to the regular means of removing the free water, sometimes emergency overflows are installed. The idea is that in case the regular system fails, the emergency overflow will protect the dam from collapsing entirely. These outlets are typically overflow systems within or around the dam.

Emergency overflows are further discussed in Chapter **4**.

2.4.2.5 Seepage flow

A tailings dam will influence the original groundwater flow pattern by introducing a hydraulic gradient (difference in hydraulic head between two points divided by the travel distance between the points). The following figures show schematic seepage flow patterns for original groundwater flow conditions and for the following basic dam types:

- existing pit
- valley site
- off-valley site
- on flat land.

Introduced in Section 2.4.2.

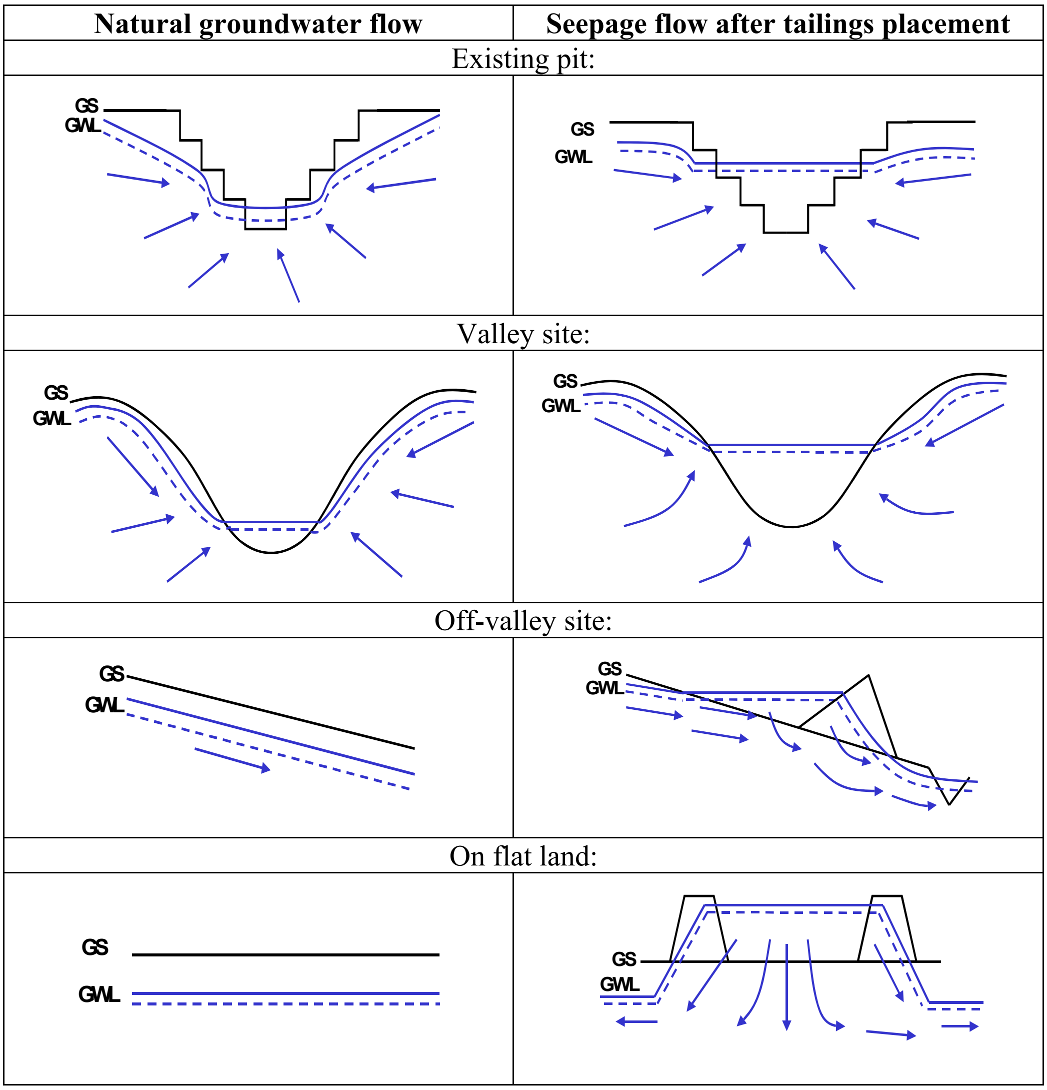


Figure 2.46: Simplified seepage flow scenarios for different types of tailings ponds

It should be noted that these are simplified schematic two-dimensional drawings. In the real world the actual flow pattern will be influenced by factors such as:

- dam properties
- water level in the dam
- permeability of the underlying formations
- ground layering
- original groundwater flow regime.

In Section 4.3.10 management and control of seepage for the various situations is discussed.

2.4.2.6 Design flood

During operation the discharge capacity should be able to handle foreseeable extreme flood events. This is based on the Probable Maximum Flood (PMF), usually defined as the 10000 year flood or two or three times the 200 year flood. The PMF is normally based on a series of local assumptions (e.g. snow smelt period, persistent rain during a number of days, plus the occurrence of an extreme precipitation event) which allow development of a hydrograph. The hydrograph is a curve of the flow (necessary discharge capacity) as a function of time at a certain point of the studied system. As a rule of the thumb one can say that the designed discharge capacity is approximately 2.5 times the highest measured flow at any point.

The Finnish "Dam Safety Code of Practice" (http://www.vyh.fi/eng/orginfo/publica/electro/damsafet/damsafe.htm) at Appendix 12 of this code provides information on how to determine the design flood as well as design outflow.

2.4.3 Thickened tailings

Applying thickened tailings management requires the use of mechanical equipment to dewater tailings to about 50 – 70 % solids. The tailings are then spread in layers over the storage area, to allow further dewatering through a combination of drainage and evaporation [11, EPA, 1995].

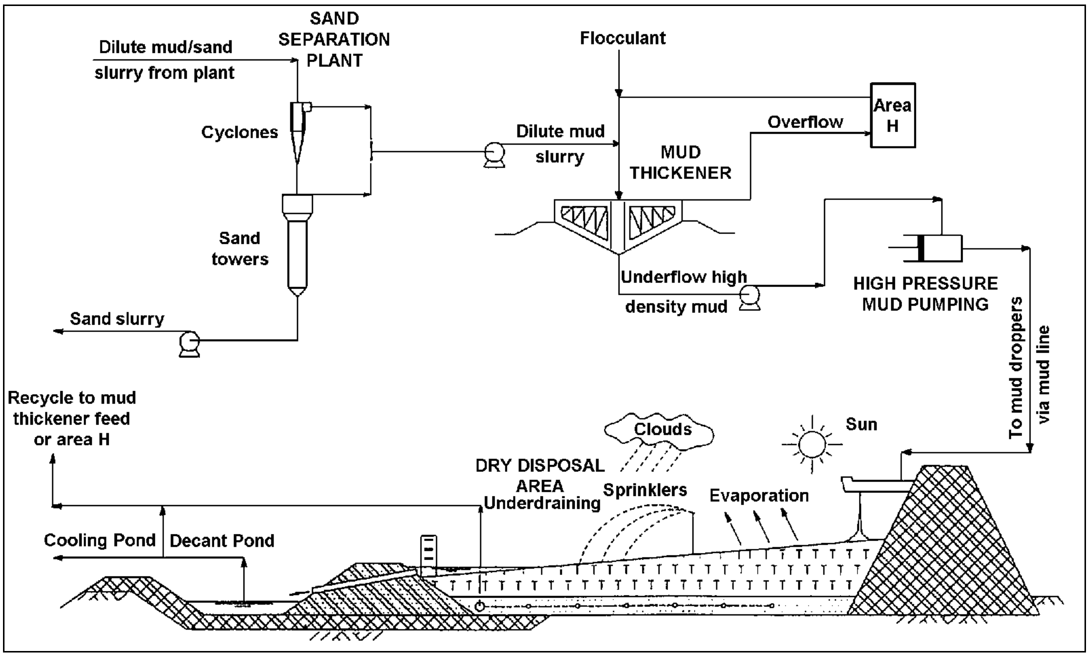


Figure 2.47: Schematic drawing of thickened tailings management operation [11, EPA, 1995]

2.4.4 Tailings and waste-rock heaps

The tailings from potash mining and the coarse tailings from iron and coal mining are often managed on heaps. Large amounts of waste-rock are managed in most metal mines using the open pit mining method.

Delivery is carried out by a conveyor belt or trucks. The heaps are surveyed to monitor for stability of the structure. Surface run-off is collected and treated, if necessary, prior to discharge or it may be diverted into the tailings ponds or separate retention basins. Geotechnically, the coarse tailings and waste-rock are usually stable. The coarseness of the material, the actual action of the truck dumping in itself, the spreading and compacting in thin layers using a tracked machine and sometimes a vibrating roller, all help to stabilise the material during and after deposition. Apart from the heap stability itself, the stability of the supporting strata also has to be considered in the design and operation of heaps.

Dust emissions from heaps can be quite significant. With dumping from conveyor belts, the operation may have to be interrupted in windy conditions. If the tailings or waste-rock are transported by trucks the transport paths may have to be sprayed in dry periods. Progressive reclamation, if possible, helps prevent erosion and dusting.

2.4.5 Backfilling

Backfilling is the reinsertion of materials into the mined-out part(s) of the extraction site. These materials are typically overburden, waste-rock and tailings, either alone or in combination with other structural products (e.g. cement).

If other material, which does not come from the mine operation, such as smelter slags, is inserted into mine voids, this is considered infilling. In some cases, the material being infilled does not serve a geotechnical requirement but is infilled for disposal purposes.

In some cases, mined rocks of a marginal or uneconomic grade may be ‘backfilled into’ or temporarily stored in disused workings. Sometimes this process is referred to as ‘stowing’.

Slurried and dry tailings are sometimes used in underground mines or abandoned pits or in portions of active pits as backfill. In most cases, backfill is used to refill mined-out areas in order to:

- for underground mining:
    - assure ground stability
    - reduce underground and surface subsidence
    - provide roof support so that further parts of the orebody can be extracted and to increase safety
    - provide an alternative to surface disposal
    - improve ventilation.

- for open pit mining:
    - decommissioning/landscaping reasons
    - safety reasons
    - minimise the foot print (e.g. as opposed to building ponds or heaps)
    - minimise risk of collapses by backfilling the pit instead of building a new pond or heap.

Besides the benefits for the mining operation itself (see list above), backfilling also decreases the above ground surface disturbance. Due to the increase in volume from size reduction separations a maximum of about 50 % of the tonnage extracted can be backfilled. This means that in cases where the ore grade is less than 50 % it will not be possible to backfill all the tailings. Hence, a surface TMF, as well as backfilling, may be necessary in these cases.

There are 4 types of mine backfill:

1. dry backfill
2. cemented backfill
3. hydraulic backfill
4. paste backfill.

[94, Life, 2002]

Dry backfill

Dry backfill generally consists of unclassified sand, waste-rock, tailings, and smelter slag. The backfill is transported underground by dropping it down a small shaft (or raise) from the surface directly into a stope or to a level where it can be hauled to a stope with loaders or trucks. Despite its name, the dry backfill usually contains some adsorbed surface moisture.

This type of backfill is suitable for mechanised ‘cut and fill’ or other methods where structural backfill is not required.
[94, Life, 2002]

Cemented backfill

Cemented backfill generally consist of waste-rock or coarse tailings mixed with a cement or fly ash slurry to improve the bond strength between the rock fragments. The methods of placement all involve mixing the rock and cement slurry in a hopper before placing it in voids (e.g. stopes or mined out longwall), or percolating a slurry over the rock after it has been placed. The waste-rock or tailings can be classified or unclassified. Cemented backfill contains a mixture of coarse aggregate (<150 mm) and fine aggregate (<10 mm fraction). The cement slurry concentration is often around 55 % by weight (1:1.2 water/cement ratio).

Cemented backfill is applied for longhole open stoping, ‘undercut and fill’, and other methods where a structural fill is required.
[94, Life, 2002]

Hydraulic backfill

Hydraulic backfill can consist either of classified slurried tailings or naturally occurring sand deposits mined on the surface. The hydraulic backfill is prepared by dewatering the mineral processing tailings stream to a pulp density of approximately 65 - 70 % solids and then passing it through hydrocyclones to remove the "slimes" retaining the coarse fraction for backfill. Fines are removed to improve the drainage capacity of the backfill, leading to an improved stability. The backfill mixture is hydraulically pumped from the surface through a network of pipes and boreholes to the stope. Sand obtained from surface borrow pits will be screened prior to use in a backfill plant to remove oversize particles that could plug the backfill line. Hydraulic backfill can be cemented or uncemented.

The tailings or tailings fraction suitable for hydraulic backfill depends on several factors, e.g.

- grain size distribution
- slope of the grain size distribution (the steeper the better)
- particle shape (flat silicates are not favoured whereas, round shape are).

In general, hydraulic backfill has permeability coefficients in the range of 1×10^{-7} m/s to 1×10^{-4} m/s corresponding to a grain size of about 35 µm – 4 mm. Hydraulic placement of backfill results in a loose fill structure with a void ratio of about 0.70.

In practice, an apparent cohesion often develops in uncemented backfill which increases the shear strength of the backfill. Often a vertical face of 3 - 4 m can be maintained under some mining conditions. Nearby blast vibrations can also act to compress the fill and increase its shear strength. To overcome the lack of true cohesion in the backfill, cement and other binders are added. Note that the backfill strength decreases with water content and the water content

needed to transport hydraulic backfill is far in excess of what is required for cement hydration. Hence, mine operators are moving towards using less water in the fill in order to decrease the cement and binder consumption. Flow velocities in excess of 2 m/s are required to maintain a homogeneous dispersion of the fill components in the slurry.
[94, Life, 2002]

Paste Backfill
The paste backfill is a high density backfill (>70 % solids depending on the density of the solids). In order to pump material at this density, a component of fines is required. As a general rule, the fines content (<20 µm) must be at least of 15 % by weight.

Paste backfill is pumped by piston type pumps of the same type used to pump concrete. Whole mineral processing tailings can often be used to make paste backfill. The final product has a lower void ratio so the backfill is denser.
[94, Life, 2002]

2.4.6 Underwater tailings management

Deep sea/ lake tailings management
In mining areas where tailings are likely to generate acids, deep lake, deep sea or submarine tailings management is sometimes an acceptable method. However, Section 4.5.3 shows an example where the driving force for the application of this technique is the lack of space for tailings deposition on land.

River tailings management
This practice is applied for water soluble materials (e.g. salt). Some potash mines discharge saline waters into rivers. Insoluble tailings are not discharged into running surface waters.

2.4.7 Failure modes of dams and heaps

Usually the following failure modes are considered in developing a tailings management strategy:

- instability
- overtopping of dams
- internal erosion.

Also the long-term safety and failure modes other than complete embankment failure should be considered, such as:

- seepage
- dust
- long-term erosion.

Tailings may retain their hazard potential for a long period of time, which therefore requires efficient measures to contain these hazards in the long term.

From the report of the International Task Force for Assessing the Baia Mare and Baia Borsa accidents it can be seen that usually there is a combination of reasons for the failures of the tailings dams in these cases the accidents were in summary caused by:

- firstly, the use of an inappropriate design
- secondly, by the acceptance of that design by the permitting authorities; and
- thirdly, by inadequate monitoring and dam construction, operation and maintenance.

Design faults:

- a closed-circuit system was used with no specific provision for emergency discharge/storage of excess water
- the dam wall was of inadequate construction, due to lack of homogeneity of the tailings
- the hydrocyclones were non-functional at very low temperatures.

Operational fault:

- failure to observe the design requirements for tailings gradation for dam construction.

[116, Nilsson, 2001]

2.5 Tailings characteristics and tailings behaviour

The tailings characteristics determine the tailings behaviour. In combination with the site location, these factors determine to a large extent the type of management facility. The following table shows how certain tailings characteristics influence the tailings behaviour.

Tailings behaviour	Tailings characteristics								
	Grain size distrib.	Fines	Specific surface	% solids	Reagents	pH	ARD influence	Surface properties	Particle shape
Permeability	X	X	X	-	-	-	-	X	X
Plasticity	X	X	X	-	-	-	-	-	X
Shear strength	X	X	X	-	-	-	-	X	X
Compressibility	X	X	X	-	-	-	-	X	X
Tendency to liquefaction	X	X	X	X	-	-	-	X	X
Chemical properties	-	X[1]	X[1]	-	X	X	X	X	X
Density (in-place and relative)	X	X	X	-	-	-	-	X	X
Consolidation	X	X	X	-	-	-	-	X	X
Dusting	X	X	-	X	-	-	-	-	-
Toxicity of discharge	X[2]	-	X[2]	-	X[3]	X	X	X	-
Tailings delivery	X	X	-	X	-	-	X	-	-
Deposition	X	X	-	X	-	-	X	-	-
Free water management	X	X	-	X	X[3]	X	X	-	-
Seepage flow	X	X	X	X	-	-	-	X	X
Long-term safety	X	X	X	-	-	-	-	X	X
ARD management	X	X	X	-	-	X	X	X	-
Emissions to air	X	X	-	X	-	-	-	-	-
Emissions to water	X	X	-	X	X[3]	X	X	X	-
Emissions to land	X	X	-	X	X[3]	-	X	-	-
Effluent treatment	X	X	X	X	X[3]	X	X	X	X
Dam construction	X	X	X	X	X[3]	X	X	X	X
Monitoring	-	X	-	-	X[3]	X	X	-	-
Closure and after-care	X	X	X	X	X[3]	X	X	X	X

1) because of increased/altered availability
2) if ARD producing tailings and exposed to the atmosphere
3) not necessarily valid if tailings water is removed (i.e. by filtration) prior to tailings discharge

Table 2.3: Effects of tailings characteristics on engineering properties and safety/environmental behaviour of tailings

In combination with Table 2.2, this table shows a connection between the mineral processing technique with the tailings characteristics, the tailings engineering properties and their safety and environmental behaviour. The two tables can be also be read ‘backwards’. This means that by starting at the tailings behaviour one can trace back what mineral processing step has an impact on this feature.

2.6 Closure, rehabilitation and after-care of facility

Usually a mine, together with the mineral processing plant and the tailings and waste-rock facilities, will only by in operation for a few decades. Mine voids (not part of the scope of this work), tailings and waste-rock however, may remain long after the cessation of the mining activity. Therefore special attention needs to be given to the proper closure, rehabilitation and after-care of these facilities.

In many cases, the tailings and waste-rock do not contain any substances that are harmful to the environment. In these cases, during the closure phase the operator will ensure that the water is drained from the tailings pond to safeguard the physical stability, and then the dams will be flattened to allow access for machinery. Ponds and heaps will then be prepared for subsequent use, which in most cases, means covering the ponds and/or heaps with soil and vegetating them. In some cases these facilities may be used again, e.g. for potash mining, the tailings heaps contain more than 90 % salt (NaCl), which can be a future economic resource when other economic deposits are depleted or too distant from their markets. In other cases, the mineral processing techniques may develop in a way that more minerals can be profitably extracted. Keeping tailings materials accessible for possible future exploitation, therefore, may be a desirable objective.

If tailings and waste-rock facilities contain substances that can be hazardous to the environment, other measures need to be taken. These measures are aimed at the stability of the tailings and waste-rock facilities whilst minimising future monitoring.

Generally, the major issues to be considered for the reclamation and closure of tailings and waste-rock management facilities include the long-term:

- physical stability of constructions
- chemical stability of tailings and waste-rock and
- successive land use.

The TMF areas of a mine site should be stable under extreme events such as floods, earthquakes and perpetual disruptive forces, including wind and water erosion, such that they do not impose a hazard to public health and safety or to the environment [12, K. Adam,].

If tailings and/or waste-rock contain sulphide minerals, they may create an acid discharge. Even though Acid Rock Drainage (ARD) is a phenomenon that may occur during operation, it is the time after the closure of the facility when ARD becomes a problem. While in operation tailings impoundments are usually saturated and the voids are filled with water. Therefore, chemical oxidation is limited during operation. It is at the closure phase of an operation, when usually the water level within the tailings drops and air enters the voids, that pyrite oxidation can occur and create a problem.

The rehabilitation of a site usually aims to turn the area into something that the local society needs and can make use of. This, of course, has to be compatible with the long-term stability of the site (see Section 4.2.4.1) [118, Zinkgruvan, 2003].

There can be problems trying to establish vegetation on sites which are acid generating, have an elevated metal content, or have a coarse texture and are unable to retain nutrients or water. Guidance is provided on these topics in several reports, such as 'Restoration and re-vegetation of colliery spoil tips and lagoons'[6], 'The reclamation and management of metalliferous mining sites'[7] and 'Landscaping and re-vegetation of china clay waste'[8].

2.7 Acid Rock Drainage (ARD)

For a more complete and scientifically correct description of all relevant issues regarding ARD generation a large amount of recently published literature is available. Recently published state-of-the-art reports for research purposes, with substantial amounts of literature references included, are available free on the internet (www.mimi.kiruna.se) on: Sulphide oxidation (Herbert, 1998); Predictive modelling (Destouni et al., 1998); Prevention and control of pollution from tailings and waste-rock products (Elander et al., 1998); Laboratory studies of key processes (Herbert et al., 1998); Field studies and characterisation (Öhlander et al., 1998); and on Biogeochemical modelling (Salmon, 1999).

The above-mentioned references are only included to give examples. A significant number of these publications are the result of research initiatives that are currently being undertaken, or that have been undertaken during the last 15 - 20 years, within large research programmes such as MEND, Post-MEND, AFR, MiMi, MIRO, INAP, PYRAMID and ERMITE. Some of the most active countries carrying out the research have so far been Canada, Australia, the United States, Sweden, Norway and the UK.

This section aims to provide a short overview of the chemical processes involved in the generation and consumption of acid.

Note in this section (s) stands for solid phase and and (g) for gas phase.

Sulphide oxidation (acid generation)
Sulphide minerals extracted from the bedrock have been formed under strongly reducing conditions resulting in sulphur being present in its lowest oxidation states. The most commonly occurring sulphides are iron sulphides (pyrite $FeS_2(s)$ and pyrrotite $FeS(s)$). These iron sulphides often coexist with other sulphides of higher economic value such as chalcopyrite ($FeCuS_2(s)$); galena ($PbS\ (s)$); sphalerite ($ZnS(s)$) or with sulphides of very little economic value such as arsenopyrite ($FeAsS_2(s)$). In unaltered bedrock the overlying overburden and groundwater minimise the contact with oxygen. This almost eliminates the oxidation of the sulphides. However, when the sulphides become exposed to an oxidising and humid atmosphere, e.g. by the mining activity, they start to oxidise (weather, dissolve, etc). This process is commonly demonstrated by pyrite ($FeS_2(s)$) oxidation by oxygen and water as:

$$FeS_{2(s)} + 7/2O_2 + H_2O \rightarrow Fe^{2+} + 2SO_4^{2-} + 2H^+ \quad (1)$$

Sulphide oxidation, which is a slow kinetically controlled exothermal process, can also take place with other oxidants such as ferric iron, Fe^{3+} as:

$$FeS_{2(s)} + 14Fe^{3+} + 8H_2O \rightarrow 15Fe^{2+} + 2SO_4^{2-} + 16H^+ \quad (2)$$

Oxidation of sulphides, mainly pyrite, and the processes that influence the oxidation rate of the sulphides have been studied in detail over the last decades. Of the various factors that influence the sulphide oxidation rate, the availability of oxygen has been found to be the most important.

6 Richards, Moorehead and Laing Ltd (1996), Restoration and revegetation of collieryspoil tips and lagoons, United Kingdom, HMSO, Department of the Environment, ISBN 0 11 753315 7

7 Environmental Consultancy University of Sheffield and Richards Moorehead & Laing Ltd (1994) The reclamation and management of metalliferous mining sites, United Kingdom, HMSO

8 Armstrong W.(1993) Landscaping and re-vegetation of china clay wastes – main report, United Kingdom, HMSO, Department of the Environment, ISBN 0 11 752843 9

To sustain a continuous sulphide oxidation, oxygen has to be supplied from the surrounding atmosphere. This is true for sulphide oxidation with oxygen (equation 1) as well as indirectly for sulphide oxidation with ferric iron (equation 2), since oxygen is required for the oxidation of ferrous to ferric iron according to

$$Fe^{2+} + 1/4O_2 + H^+ \rightarrow Fe^{3+} + 1/2H_2O \qquad (3)$$

Ferric iron may contribute to sulphide oxidation (equation 2) or it may hydrolyse and precipitate as ferric oxyhydroxide (dominant at pH>3.5) according to

$$Fe^{3+} + 2H_2O \leftrightarrow FeOOH_{(s)} + 3H^+ \qquad (4)$$

There are also indications that the cycling of iron through the ferrous and ferric oxidation states may potentially be a key process in anaerobic tailings and waste-rock management facilities. Field studies, however, indicate that the overall sulphide oxidation rate is dramatically reduced by applying oxygen diffusion barriers. Bio-geochemical modelling results calibrated to field data from a covered tailings deposit do not indicate that pyrite oxidation by ferric iron plays any significant role in the remediated deposit.

As described above, many factors have been found to influence the sulphide oxidation rate, such as e.g. bacterial activity, pH, Eh (oxygen concentration), temperature and galvanic processes between different sulphides. This has, to a large extent, been investigated and numerical expressions (rate laws) have been developed for pyrite oxidation under various conditions. These rate laws are available in the literature. However, under natural conditions, as e.g. in a tailings or waste-rock facility, these various factors are co-dependent and influenced by other factors such as the available surface area for oxidation determined by the grain size distribution, mineralogy, hydrology and the availability of buffering minerals, etc. and will be described in the following sections.

Dissolution of buffering minerals (acid consumption)

If readily available buffering minerals (carbonates) are present in the tailings or waste-rock, acid produced by the oxidation of sulphide minerals (equations 1 and 2) and the precipitation of iron oxyhydroxide (equation 4) will be consumed by the dissolution of the buffering minerals, demonstrated here by the dissolution of calcite

$$CaCO_{3(s)} + 2H^+ \rightarrow Ca^{2+} + CO_{2(g)} + H_2O \qquad (5)$$

The dissolution of calcite is a fast reaction in comparison to pyrite oxidation and is therefore normally assumed to be in equilibrium (i.e., acid is consumed at the same rate as it is produced). If there are not enough readily available buffering minerals present, or if they are depleted over time, the pH in the drainage may drop and the solubility of dissolved metals will increase. This is what is normally called ARD.

Acid is also consumed by the dissolution of other buffering minerals, such as alumino-silicates, but normally at a slow rate, that cannot keep up with the acid production from sulphide weathering, as the dissolution of the alumino-silicates is kinetically controlled. Acid consumption by dissolution of alumino-silicates is demonstrated below by the dissolution of K-feldspar, muscovite and biotite.

$$KAlSi_3O_{8(s)} + H^+ + 9/2\ H_2O \rightarrow K^+ + 2H_4SiO_4 + 1/2Al_2Si_2O_5(OH)_{4(s)} \qquad (6)$$

$$KAl_2(AlSi_3O_{10})(OH)_{2(s)} + H^+ + 3/2\ H_2O \rightarrow K^+ + 3/2Al_2Si_2O_5(OH)_{4(s)} \qquad (7)$$

$$KMg_{1.5}Fe_{1.5}AlSi_3O_{10}(OH)_{2(s)}+7H^++1/2\ H_2O\rightarrow$$
$$K^++1.5\ Mg^{2+}+1.5Fe^{2+}+2H_4SiO_4+1/2Al_2Si_2O_5(OH)_{4(s)} \qquad (8)$$

3 APPLIED PROCESSES AND TECHNIQUES

The following tables summarise the mineralogies, the mining techniques and the mineral processing for the minerals covered by this document. Furthermore they highlight some examples of the most important factors in tailings and waste-rock management, namely the characteristics of the tailings and waste-rock, the applied management methods, the measures applied to ensure the safety of the facilities and to prevent accidents, and closure and after-care planning.

It should be noted that unless otherwise mentioned, during mineral processing the ore is reduced in size by crushing and grinding. Screening is also often a part of the size reduction circuit.

As part of an accident prevention programme, it is common practice to carry out visual inspections and phreatic surface measurements with piezometers in dams.

In underground operations, the waste-rock usually remains underground.

Unless otherwise mentioned, the waste-rock from open pit operations is managed on nearby heaps, where the drainage water is collected.

It should be noted that both tables summarise the information provided on tailings and waste-rock management. This however does not generally allow an extrapolation of the information, since an operation extracting the same mineral could operate under completely different conditions and would therefore apply different tailings and waste-rock management methods.

Mineral	Mineralogy	Mining technique	Mineral processing	Tailings characteristics	Tailings management	Safety and accident prevention	Closure and after-care
Aluminium	Al_2O_3 SiO_2 Fe_2O_3 CaO TiO_2	Open pit and underground, only one mine in Europe, mostly imported ore	Bayer process	Elevated pH, red mud: d_{80}<10 µm, process sand: d_{80}<1000 µm	Slurried or thickened	Monitoring routine	Dewater and dry cover, discharge treatment
Base metals	Mostly sulphides	Open pit and underground (cut-and-fill, room and pillar, blasthole stoping)	Flotation, at Boliden CN leaching for Au	d_{80}: 50 – 100 µm, often ARD potential	Slurried, at Lisheen subaqueaous, usually large ponds: 35 – 1450 ha, some backfill (coarse fraction)	OSM manual, independent audits, water balance	Dewater and dry cover or wet cover
Chromium	26 % Cr_2O_3	Open pit	Dense medium and magnetic separation	Containing Cr and Ni	Slurried	Independent audits	No plans
Iron	Phosphorous magnetite, iron carbonates	Open pit (Erzberg), underground (large-scale sub-level caving)	Magnetic separation, dense medium separation	No ARD potential, Kiruna: mostly SiO_2 and Fe_xO_y	Fines: slurried, coarse: heaps	OSM manual, independent audits, subsidence measurements	Dewater and dry cover
Manganese	MnO_2	Underground	Only crushing	No tailings			
Precious metals	Complex sulphides, native gold, gossan, etc.	Open pit and underground	CN leaching, spirals, shaking table	Some have ARD potential, in case of CN leach: containing cyanide, complexed metals, cyanate, thiocyanate	Slurried, some backfill (coarse fraction), CN destruction	Risk assessment, stability calculations, planning by external experts, OSM manuals, independent audits, piezometers, inclometers	Dewater and dry cover, wet cover, raised groundwater table
Tungsten	(Fe, Mn)WO_4, $CaWO_4$	Underground (sublevel stoping, sublevel caving, cut-and-fill)	Flotation, dense-medium separation, shaking tables	d_{80}=100 µm, no ARD potential	Slurried, some backfill (coarse fraction)	Involvement of external experts and authorities	Dewater and dry cover
Barytes	$BaSO_4$	Open pit, underground	All techniques e.g. jigging, dense medium, flotation		Often no tailings, fines as slurry, sometimes backfilled, coarse tailings on heaps or sold as aggregates		
Borates	B_2O_3	Open pit, underground	Dissolution, crystallisation, drying/cooling		Coarse tailings first on heaps and then backfilled, slurry in ponds		
Feldspar	Orthoclase, albite, anorthite	Quarrying	Sometimes none, otherwise optical separation, flotation, electrostatic or magnetic separation	Solids contain fine sands and micas, 10 % iron oxides, some flocculants, process water: pH 4.5, some fluoride	Coarse tailings on heaps, slurries are backfilled or ponded	Topographical surveys	

Mineral	Mineralogy	Mining technique	Mineral processing	Tailings characteristics	Tailings management	Safety and accident prevention	Closure and after-care
Fluorspar	CaF_2 (in one case also PbS)	Open pit and underground (cut-and-fill, room and pillar)	Dense medium separation, flotation,	Mostly silica (90 %), Fe and Al oxides	Backfilling and process water re-use, slurries mostly to ponds, in one case fine tailings into sea	Phreatic surface with piezometers	After-care period of 10 yrs expected to monitor heavy metals, fund for closure/after-care costs
Kaolin	Kaolinite, quartz, micas, feldspar residues	Quarrying	No comminution, magnetic separation, flotation	Fine sands and micas, <1 % iron oxides, some flocculants, process water: pH 4.5, some phosphates, sulphates, foam inhibitor	Coarse tailings on heaps, slurries in ponds lined with clay, in one case dewatered fines are transferred to heaps	Seepage flow, vertical and horizontal movements of the dam crest, emergency plans	Dewater and dry cover
Limestone/calcium carbonate	97 – 98 % $CaCO_3$, <1 % $MgCO_3$, <1 % SiO_2	Open pit/quarry	Limestone: washing; calcium carbonate: flotation, magnetic separation	Limestone: <0.25 mm	Slurries in ponds, in one case the pond is a former quarry, sometimes the slurry is dried and the tailings discarded onto heaps	Stability calculation using DIN, quality management during dam construction, changes in dam are logged, annual reviews, independent audits	Dewater and dry cover
Phosphate	Apatite (10 %), phlogopite mica (65 %), carbonates (20 %) and silicates (5 %)	Open pit	Flotation		Slurries in ponds	Water level controls on-line and monitored, with alarms in plant operating system, seepage measurements, dam movement measurement	
Strontium		Open pit	In one case none, in the other dense medium and flotation		Coarse tailings are backfilled, flotation slurry in ponds	New lined pond with dam built to final height	
Talc	Talc, carbonates, chlorites and sulphides		Often only comminution, sometimes flotation		Flotation tailings in ponds, which through dewatering become heaps	Seepage water control, safety manuals, annual review	
Potash	Sylvinite carnallitite hard salt kainitite and other salts	Underground (room and pillar, longwall, sublevel stoping)	Hot leaching, flotation, electrostatic separation, dense medium separation	Liquid and solid tailings, containing sodium chloride with other salts, clay and anhydrite	Solid tailings on heaps, liquid tailings into deep wells or surface waters, in one case marine discharge of liquids and solids, some solid tailings are backfilled	Annual review, slope inclinometers, seismic monitoring	Heaps remain unchanged and dissolve over time
Coal	Carbon, ash, sulphur	In Spain and UK- some open pit, otherwise underground (longwall)	Coarse fractions in jigs or dense medium, flotation for fines	Clay, shale, sandstone, sulphides, some reagents, can be radioactive	Backfilling often too costly, coarse tailings on heaps or in old pits, fines in ponds, sold or filtered and to heaps	In some areas seismic monitoring	Landscape integrated heap design agreed with authorities and communities

Table 3.1: Summary of applied processes in the management of tailings

Mineral	Waste-rock characteristics	Waste-rock management	Safety and accident prevention	Closure and after-care
Base metals	Sometimes has ARD potential	In one case selective management of ARD and non ARD waste-rock, sometimes used for dam construction, in one case backfill, collection of surface run-off	1:3 slope	Vegetative cover, engineered cover to reduce ARD generation
Chromium				Backfilling of all waste-rock underground
Iron	No net ARD potential, possibly ammonium nitrate leachate	On heaps, at one site together with coarse tailings	1:2 slope	Vegetative cover using soil and seeds, long-term seepage monitoring
Precious metals		On heaps, for dam construction or backfill into open pit		At one site covering with topsoil
Barytes		Sometimes sold as aggregate or backfill		
Fluorspar		Backfill		
Kaolin		Collection of surface run-off		
Limestone		Backfilling into old quarry		
Phosphate		Some uses as aggregate		Landscaping plans have been developed with local authorities and communities
Talc			1.3 safety factor, yearly surveys, monthly inspections, periodical risk assessments	Water drainage and vegetative cover
Coal		With coarse tailings on heaps, temporary heaps and later backfilling		Final heap design agreed with authorities and communities with the goal of creating landscape integrated structures

Table 3.2: Summary of applied processes in the management of waste-rock

3.1 Metals

3.1.1 Aluminium

In the section, information about the following alumina refineries is provided:

Plant	Country
Aluminium de Greece, Distomon	Greece, Central
Aughinish Alumina, Aughinish	Ireland, Aughinish
Eurallumina, Sardinia	Italy, Sardinia
Alcoa Inespal, San Ciprian	Spain, Galicia
Ajka	Hungary, Bakony region

Table 3.3: Alumina refineries mentioned in this section

3.1.1.1 Mineralogy and mining techniques

The bauxite deposits of **central Greece** are lenticular bodies in the form of three bauxite layers. Vanadium, manganese, nickel, cobalt, chromium, zinc, copper, phosphorus and sulphides can be found in the ore at low levels or as trace elements. So far the amounts of ore resulting from from underground and open pit mining have been roughly equal, but in the future it is expected that

more underground mining will be carried out because of the increase in the stripping ratio, and due to the emerging environmental aspects connected to open pits.
[90, Peppas, 2002]

In underground mining the ‘room and pillar’ method is applied, sometimes in combination with ‘cut-and-fill’ if the orebody is thicker than 8 m. Ore bodies with a stripping ratio of 6 - 8 m^3 of waste-rock or overburden per tonne of ore are mined in open pits via conventional drilling, blasting and loading [90, Peppas, 2002].

In the Hungarian **Bakony** region six bauxite mines are in operation, which all send their bauxite to the refinery in **Ajka**. The bauxite is of the karst type in the form of lenticular or pod-like deposits. Mining is undertaken by open pit (drilling/blasting/loading) at a stripping ratio of 6.3 m^3/t, or underground using sublevel caving [91, Foldessy, 2002].

The following table shows the chemical composition of bauxite processed in European refineries.

Component	% by weight
Al_2O_3	53 - 60
SiO_2	2 – 25
Fe_2O_3	6.5 - 22
CaO	0.2 – 1.2
TiO_2	2 - 4
LOI[1]	16 - 27

Table 3.4: Chemical composition of bauxites fed to European refineries

3.1.1.2 Mineral processing

As mentioned in Section 2.3.4.1, the Bayer process is used to treat bauxite in all alumina refineries in Europe.

The Bayer process is based on a continuous re-circulation of caustic solution, which acts as the dissolving agent for the hydrate-alumina within the bauxite as well as the transport medium for carrying all the solids through the various process stages. In the first stage of this process, the bauxite is put through a wet grinding stage, resulting in a slurry with 50 % solids. This is preheated to 100 ºC and held in holding tanks to make the silica more reactive. Caustic liquor returning from the previous cycle is then re-concentrated and heated up. At the subsequent leaching (or digestion) phase the bauxite slurry is mixed with the caustic liquor at a high temperature (250 ºC). Gibbsite and Boehmite rapidly dissolve, leaving the inert part of the bauxite (the red mud) undissolved.

Clarification of the pregnant liquor is carried out by thickeners and filtration. The mud is separated in two steps. First, so-called sands (i.e. particles over 150 µm) are removed by cycloning the liquor and separating the solids in screw-classifiers. In the second step the mud is settled in large thickeners.

The clarified pregnant liquor is then pumped to the precipitation phase to produce solid hydrate. The hydrate is then calcined to produce alumina. The liquor is strengthened with fresh soda make-up and returned to the process.

The separated mud is extracted from the decanter’s cone at around 30 % solids, and pumped to a continuous three or four stages counter current washing unit, where most of the caustic liquor accompanying the mud is recovered.

Some alumina plants pump mud from the last washer to the mud pond. Other plants thicken the mud by vacuum filtration or deep thickeners before pumping the mud to the TMF.

3.1.1.3 Tailings management

On a worldwide basis, four to six tonnes of bauxite on average yield on average two tonnes of alumina and one tonne of aluminium. The European refineries that import bauxite use high grade bauxite, in order to reduce shipping costs. The following figure shows typical mass flows for European refineries.

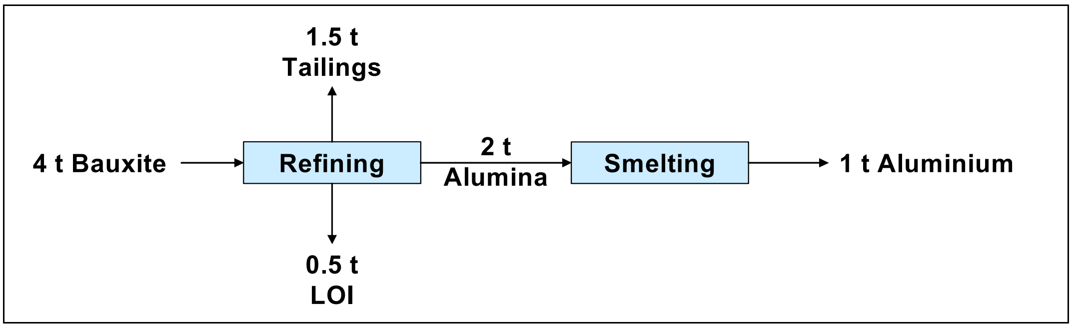


Figure 3.1: Typical mass flow from Bauxite to Aluminium (dry basis)

It should be noted that LOI stands for 'Loss On Ignition' or 'Water Of Crystallisation.'

3.1.1.3.1 Characteristics of tailings

The alumina tailings consist of two major parts. The fine fraction, which accounts for 80 – 95 % of total, called 'red mud' and a coarser fraction, commonly referred to as 'process sand'. These two portions represent 97 – 100 % of the total tailings. In some cases the remaining 3 % consists of salt cake, which can originate from a salting-out liquor purification process and sludge (principally aluminium hydroxide) from the underflow of the clarifier.

Red mud
The following figure shows some red mud size distributions of alumina refineries.

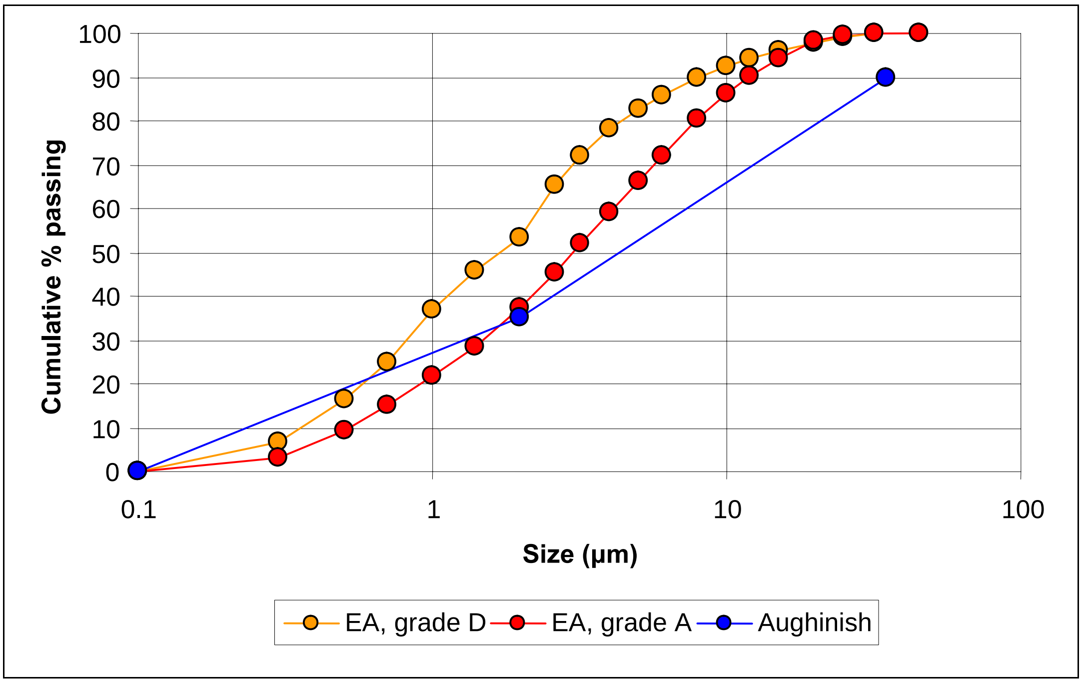


Figure 3.2: Size distribution (particle size vs. cumulative % passing) of red mud at the Sardinian (EA) and Aughinish sites
[89, Teodosi, 2002], [22, Aughinish,]

If the red mud is pumped as thickened tailings, it usually has a solids content of 55 – 60 %. It then 'matures' at the TMF, which for thickened tailings is often referred to as a 'stack', over a period of three to six months to a solids content of 68 – 70 % due to compression and evaporation.

At the **Aughinish** refinery the initial permeability of the red mud is between 1 x 10^{-8} to 1 x 10^{-9} m/s. It decreases as the mud matures. The average density of dry mud solids is 3.1 t/m^3 [22, Aughinish,]. The benefit of this technique is that the tailings are physically stable upon discharge onto the stack. However, precipitation run-off and seepage water will have elevated pH levels, due to the residual caustic liquor, and must therefore be neutralised before release to the environment. Alternatively they can be used in the washing circuit in the alumina plant.

At the **Sardinian site,** the red mud is resuspended to 20 - 25 % solids using fresh seawater and free water from the tailings pond and then pumped to the tailings pond. The neutralisation of the mud is performed by the flue-gas desulphurisation in the wet scrubbing operation, and with the magnesium chloride of the fresh seawater added to the system.

After settling and evaporation, the solids content increases to 65 – 72 %. The ratio of tailings, at the Sardinian refinery, is 0.78 tonnes dry tailings per tonne of alumina. Considering that the slurry consolidates at 60 – 65 % solids in the pond, this corresponds to about 1.3 tonnes of wet material per each tonne of alumina produced or 0.8 m^3/tonne of alumina produced.
[89, Teodosi, 2002].

The neutralisation of the red mud leads to chemical stability of the tailings. The trade-off here is that, as for all slurried tailings impoundments, physical stability of the dams must still be taken care of.

The solids content of the tailings for both options are shown in the following figure.

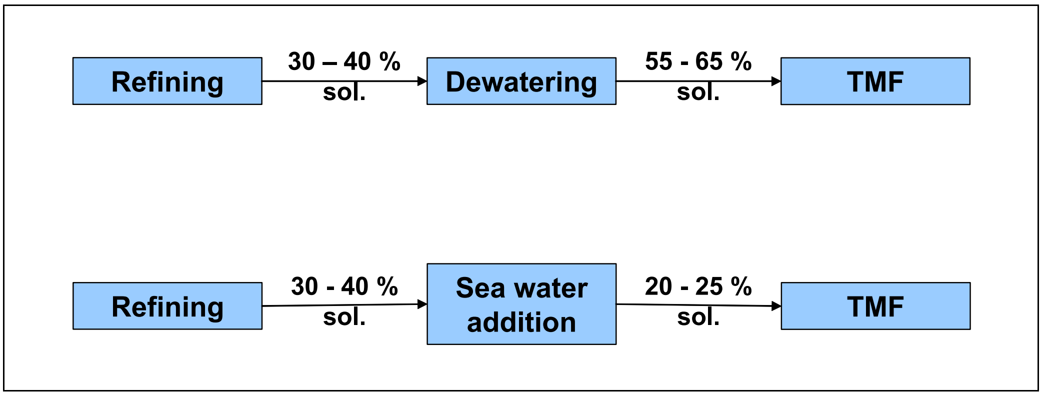


Figure 3.3: Solids content (in % solids by weight) of tailings for thickened and conventional management schemes

In both cases, the tailings mature to about 70 % solids. Generally dewatering can be carried out in vacuum filters (yields 63 % sol., e.g. Aughinish) or in deep thickeners (yields 50 % sol.).

Some chemical analyses of red mud from different sites are shown in the following table.

	Site:		
	Sardinia	**Bakony**	**Aughinish**
Component:	Dry wt. %	Dry wt. %	Dry wt. %
Fe_2O_3	18	40	47
Al_2O_3	26	18	17
TiO_2	6	4	12
SiO_2	20	15	7
Na_2O	12	8	5
CaO	8	7	8
LOI	9	7	3
Misc. Trace elements	1	1	1

Table 3.5: Constituents of red mud
[89, Teodosi, 2002], [91, Foldessy, 2002], [27, Derham, 2002]

Despite repeated washings, the solution entrained within the red mud still contains small amounts of caustic (sodium hydroxide), which causes the elevated pH characteristics, and alumina. Most of the caustic converts to sodium carbonate and sodium bicarbonate on the tailings stack.

The following table shows an example of a more detailed analysis of red mud, including the trace elements.

Analysis of Aughinish Alumina Red Mud

Name	formula		% dry		METHOD OF ANALYSIS
Principal Compounds					X-RAY
Titanium dioxide	TiO_2		9.93 %		
Iron Oxide	Fe_2O_3		46.18 %		
Silica Quartz	SiO_2		8.11 %		
Sodium Oxide	Na_2O		4.39 %		
Calcium Oxide	CaO		4.41 %		
Alumina (aluminium oxide)	Al_2O_3		16.50 %		
Loss on Ignition LOI (includes crystalline water)			9.26 %		
subtotal				**98.78 %**	
Secondary Compounds					FLUORESCENCE
Zirconium dioxide	ZrO_2		0.15 %		
Zinc Oxide	ZnO		0.01 %		
Vanadium pentoxide	V_2O_5		0.17 %		
Phosphorus pentoxide	P_2O_5		0.43 %		
Manganese Oxide	MnO		0.05 %		
Magnesium Oxide	MgO		0.07 %		
Potassium Oxide	K_2O		0.04 %		
Chromium trioxide	Cr_2O_3		0.26 %		
subtotal				**1.18 %**	
BASIC TOTAL =				**99.96 %**	
Misc. Trace elements ex analysis by EOLAS					
Sulphur			0.12 %	**Inductively coupled plasma spectrometry**	
Arsenic (just at detection limit, therefore approximate)			0.005 %		
Tin		<	0.005 %		
Mercury		<	0.005 %		
Antimony			0.019 %		
Lead			0.020 %		
Gallium			0.006 %		
Bismuth		<	0.005 %		
subtotal =				**0.19 %**	
GRAND TOTAL (discrepancy of different dates & methods of analysis) =				**100.15 %**	

Table 3.6: Detailed anaysis of red mud, including trace metals
[32, Derham, 2002]

Process sand

Size distribution curves for process sand are shown in the following figure.

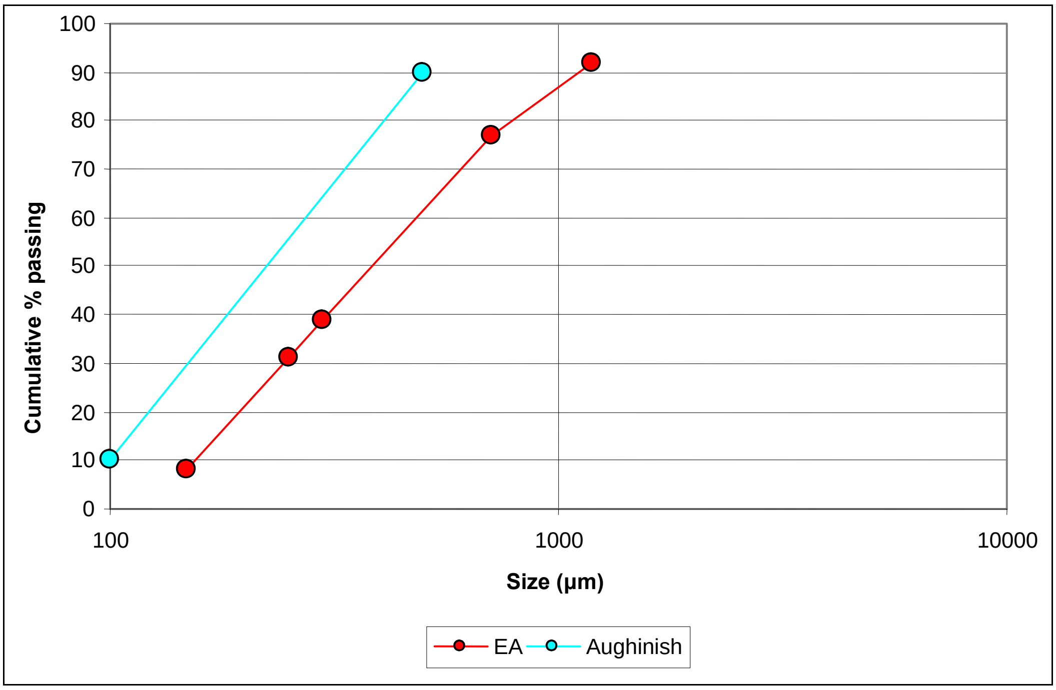


Figure 3.4: Size distribution (particle size vs. cumulative % passing) of process sand at the Sardinian (EA) and Aughinish sites
[89, Teodosi, 2002], [22, Aughinish,]

The following table shows the components of the sand fraction:

	Site:
	Sardinia
Component:	Dry wt. %
Fe_2O_3	14
Al_2O_3	40
TiO_2	3
SiO_2	16
Na_2O	12
CaO	1
LOI	12
Misc. Trace elements	2

Table 3.7: Constituents of tailings sand
[33, Eurallumina, 2002]

The permeability of the sand fraction is estimated to be 100 times higher than that of the red mud [22, Aughinish,].

Others

Salt cake is dumped as a 70 % solids cake. Clarifier sludge is pumped to the stack as a 2 to 3 % solids slurry. Salt cake consists of organic degradation products from humates in the bauxite, including sodium carbonate, sodium sulphate and sodium oxalate.
[22, Aughinish,].

3.1.1.3.2 Applied management methods

For the management of tailings from alumina refining, thickened tailings as well as conventional slurried tailings, are applied. Some refineries discharge the tailings into the sea. Others manage them on land on 'stacks', for thickened tailings, or within dammed ponds, in the case of slurried tailings.

Generally the design of red mud stacks using the **thickened tailings method** includes pervious perimeter rock fill dams and sealing of the underlying surface. A perimeter dam is usually used for the collection of surface run-off and, therefore, typically surrounds the stack. The upstream construction method for the stacks is used, since the dewatered red mud is sufficiently stable.

Due to the very low permeability of the red mud, the principal risk of seepage arises from ponding of caustic surface water run-off in exposed areas prior to covering with mud and seepage from standing water in the perimeter ditch. This can be handled by sealing surface and ditches with liners, such as glacial till or synthetic liners combined with a drainage system. Seepage analysis for typical and worst case conditions are undertaken in order to properly design these facilities.
[22, Aughinish,]

With the **Sardinian** refinery, the red mud is diluted to 20 % solids and used in the flue-gas desulphurisation. The mud slurry to be used in the absorbers needs to have its solid content well diluted, in order to protect the perforated dishes of the absorber against early blockages by plugging by solids deposition.
[89, Teodosi, 2002]

In the Sardinian refinery the following aspects were of importance in the design of the facility:

- short distance between refinery and pond, to reduce pumping costs
- availability of surface area
- need to manage tailings on land, as opposed to discharging into the sea, in order to protect fishery
- vicinity to the sea, because of the need for seawater to neutralise the tailings
- low risk of aquifer contamination
- strong winds in the area, therefore it is beneficial to have wet tailings.

The location of the TMF can be seen in the following figure

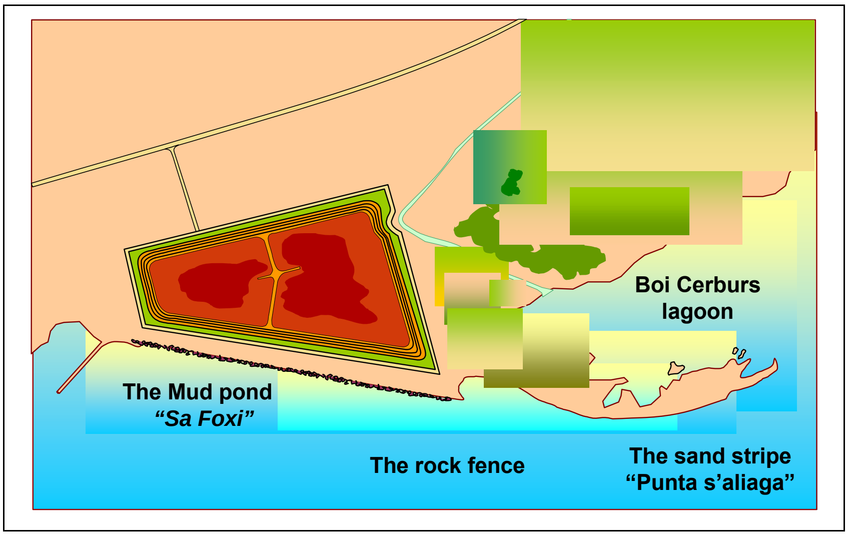


Figure 3.5: Location of TMF at the Sardinian refinery
[33, Eurallumina, 2002]

The 'rock fence' protects the TMF from waving action.

A cross-section of the dam can be seen in the following figure.

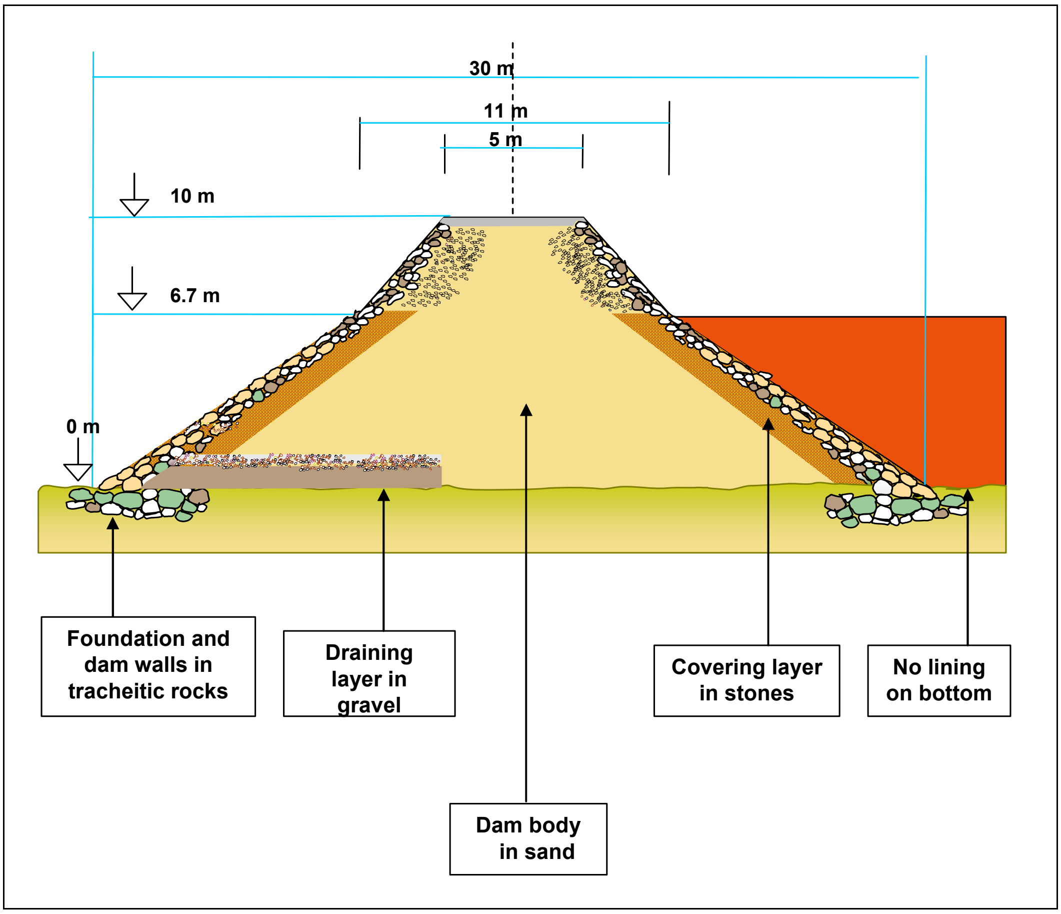


Figure 3.6: Cross-section of tailings dam at Sardinian site
[33, Eurallumina, 2002]

The concept of this original dam design is to drain the tailings water whilst the tailings remain within the impound. Hence good drainage (up to 70 %) is achieved.

Further raises of the dam, using the upstream method, were carried out as shown in the following figure.

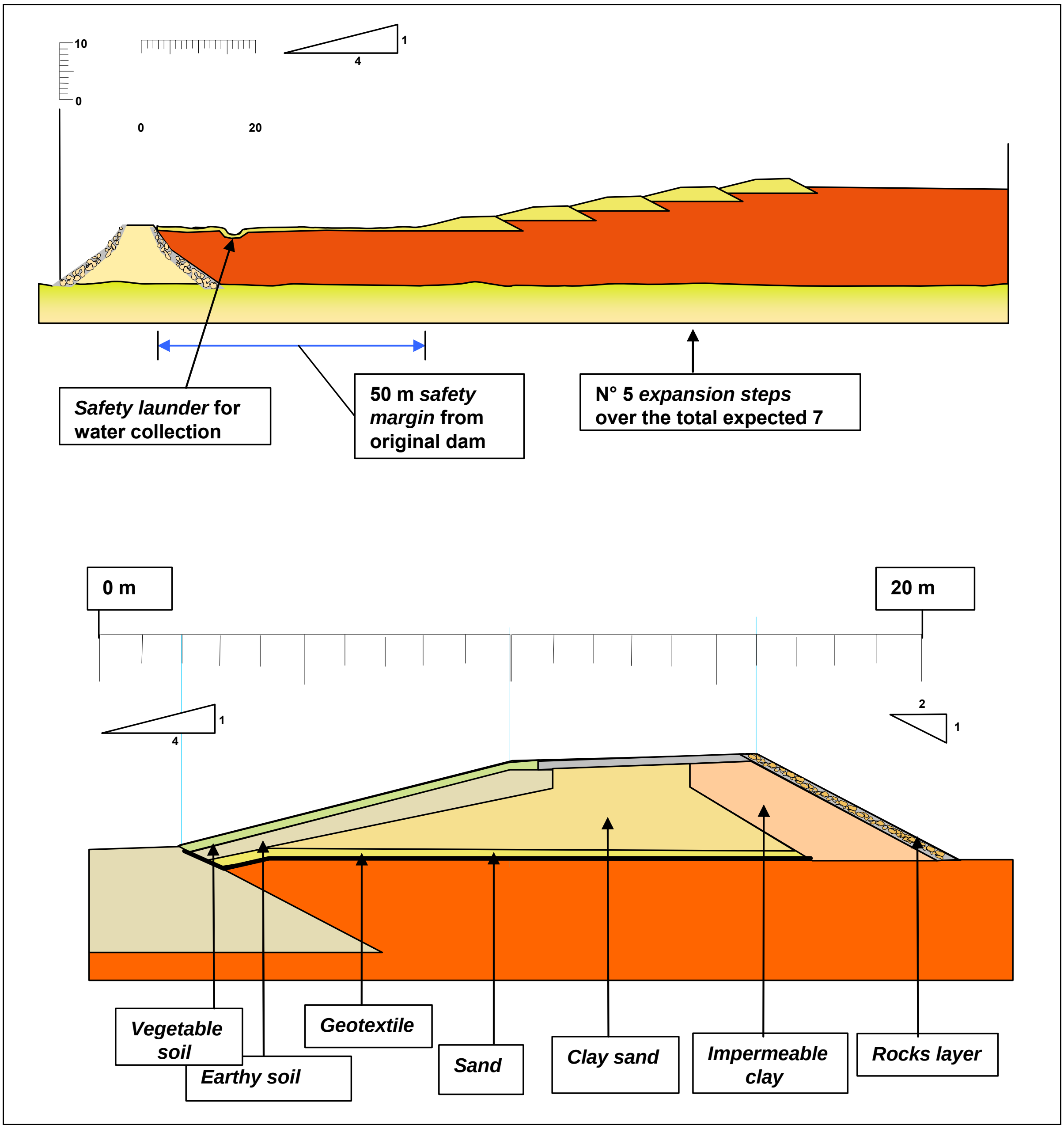


Figure 3.7: Cross-section of dam raises using the upstream method [33, Eurallumina, 2002]

The mud is distributed along the perimeter of the facility with a discharge every 50 m. To achieve an even distribution, different discharge points are used every 24 h. The sands and other process residues are transported to the TMF by trucks and discarded in a dedicated area of the TMF.
[33, Eurallumina, 2002]

In the **Ajka** refinery, 'cassettes', i.e. paddock-style tailings ponds for the collection of red mud, are built from gray slag derived from the nearby thermal power plant. The dams have 1:1 to 1:1.5 slope ratios (see figure below). Their final height is up to a maximum of 10 m. The red mud is transported to the TMF via pipeline at 20 % solids. The distance is 3 - 4 km. The free water from the pond is re-used in the process. The circular movement of the discharge pipe achieves an even distribution of the red mud in the cassette. The free water in the cassettes prevents the development of larger dry surfaces and the drying of the red mud.
[91, Foldessy, 2002]

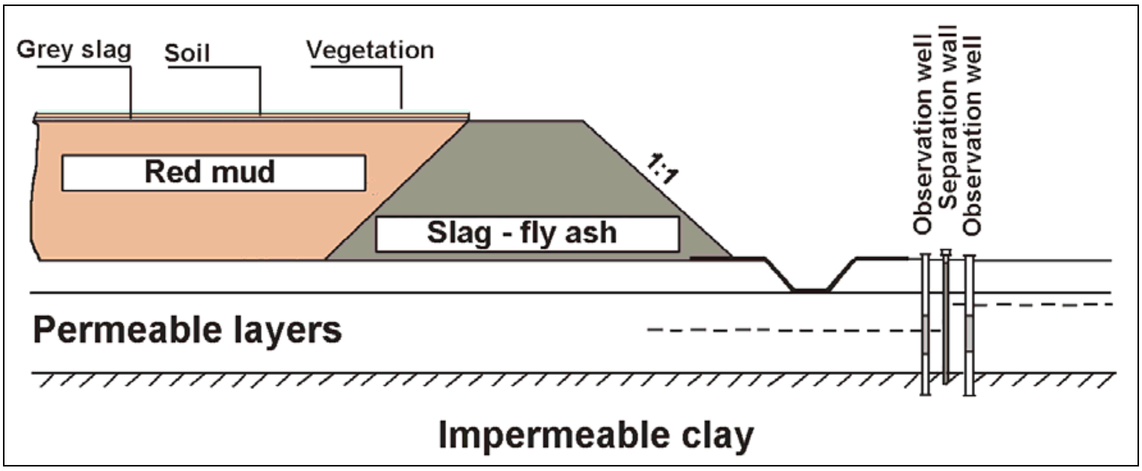


Figure 3.8: Cross-sectional view of TMF at Ajka showing the dam, pond, observation wells, separation wall and ground conditions, as well as the soil cover upon closure
[91, Foldessy, 2002]

An impermeable clay layer is found 10 m beneath the tailings management facilities. For this reason, no sealing was used during the construction of the cassettes. In the 1980s it was revealed that groundwater pollution had developed in the layer between the bottom of the cassettes and the clay layer. To contain this pollution an impervious wall into the impermeable clay layer was built around the cassettes. In the inner side of this sealing wall a drainage system collects seepage water and groundwater, which is then pumped back into the cassette.

In the surrounding area 240 groundwater observation wells were drilled. These serve to measure the level and sample the groundwater for chemical analysis. Groundwater level measurements are repeated monthly, and a chemical analysis of the groundwater samples for 8 - 10 components is done every quarter. This system ensures the early detection of any damage of the separation wall, and also monitors the migration of the pollution plume.
[91, Foldessy, 2002]

At the **Galician** alumina refinery, the initial method of raising the dam was the upstream method. For this, rock and soil was taken from local deposits of granite-quartz rock and fill material. This method has been changed since 1986. The new method, the centreline method, uses the same borrow materials. However, by using this method the available surface area and, therefore, the storage capacity does not decrease with each dam raise (see Figure 3.9).

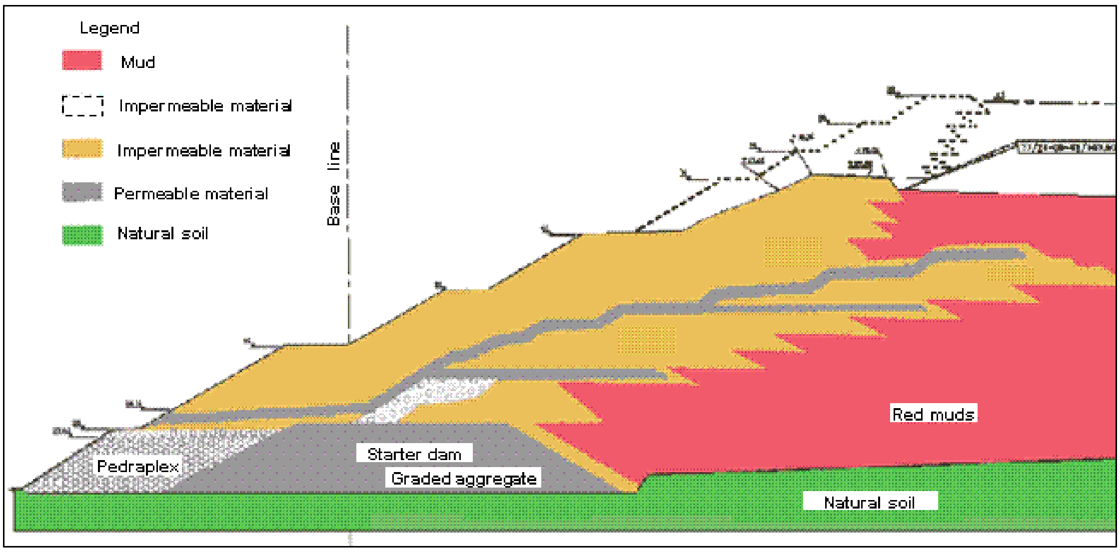


Figure 3.9: Cross-section of the tailings dam at the Galician refinery showing the upstream and centreline methods of increasing the dam height

3.1.1.3.3 Safety of the TMF and accident prevention

The control programme at the Sardinian site includes the following:

- inspection tour of the TMF every two hours
- daily overall inspection inside and outside of the TMF by trained staff
- performance control of external water collecting pumps on a daily basis and recording of the flow measurements
- monthly sampling at external piezometer network, with analysis of pH and metals
- checks of dam stability twice a year
- annual tracing of coastal profile to check erosive trends
- daily change of discharge points
- checking of water balance
- continuous recording of meteorological conditions
- continuous measure of pH upon exit from the mud filtration unit, before pumping to the TMF.

Staff working in the TMF area have been trained during specific annual training courses. An emergency procedure exists.

Seven pumps are spread around the perimeter of the pond so that they can be used should a water leakage occur from the embankment. The water level in the basin is controlled by a thorough monitoring and control of fresh seawater addition to the mud circuit. [33, Eurallumina, 2002]. However, these pumps are not capable of handling a complete failure of the dam.

3.1.1.3.4 Closure and after-care

If thickened tailings management is carried out horizontally on one stack, progressive restoration is not practical, since most of the surface will be used for dumping red mud. During restoration, the stacking slope of 2.5 % allows effective run-off of precipitation without erosion. Furthermore, the stack is accessible for construction equipment [22, Aughinish,]. The mud stack will be restored with a vegetative cover. This has been successfully demonstrated at several sites. Re-vegetation of the perimeter slopes, built with borrowed rock-fill (e.g. limestone) is common practice and usually designed in such a way that the vegetation matches the looks of the surroundings [22, Aughinish,].

Vegetative covers have already been applied successfully to conventional tailings ponds.

At the **Ajka** site, the dewatered tailings are covered by a 0.5 m thick layer of slag from a power plant and another layer of soil [91, Foldessy, 2002].

In the after-care phase, the run-off needs to be treated prior to discharge, until the chemical conditions have reached acceptable concentrations for discharge into surface waters. Also access roads, drainage systems and the vegetative cover (including re-vegetation if necessary) need to be maintained. Furthermore, continued groundwater quality sampling will form part of any closure programme implementation and must be continued.
[22, Aughinish,]

3.1.1.4 Current emission and consumption levels

3.1.1.4.1 Management of water and reagents

At **Aughinish,** the water from the TMF is recycled to the process; at the **Sardinian site** the TMF water is recycled to the mud handling facilities, to better manage the water balance at the pond. In the latter case, it is not possible to re-use the free water in the Bayer process because the saline content of seawater would spoil the caustic leaching solution.

At **Ajka** a total of 1.75 Mm^3 of fresh water are consumed every year, of which 50 % are released into surface water.

The following table lists the reagent consumption of one alumina refinery

	Ajka site
Reagent	Consumption
	g/t
NaOH	79167
H_2SO_4	4167
HCl	50
Hg	3
CaO	39167
Water glass	19333

Table 3.8: Consumption of reagents at Ajka refinery

At the **Sardinian** refinery, the chemical additives added to the process are grouped in the following categories:

a) Lime: the principal process reagent, with a specific consumption of around 40 kg CaO/tonne of alumina, for a number of reactions, namely

- reaction with titanium and phosphorus contained in bauxite, by precipitating them as titanate and phosphate, to protect alumina from the relevant content of impurities
- reaction with sodium carbonate, an impurity present in the liquor, to revert it back into sodium hydroxide
- reaction with sodium oxalate, an organic impurity of the liquor, to transform it into calcium oxalate, which in the solid form is rejected with the process mud
- plus other reactions in the digestion phase, to improve the boehmite (aluminum oxyhydroxide, a source of alumina in the bauxite) extraction, and to promote the transformation of any iron oxide present in the bauxite as goethite into iron haematite, which in the solid form follows the mud, thus minimising iron impurity in the product.

b) Other process reagents:

- humate control agents, used to remove long chain organic matters from the caustic liquor: polyamines in water
- precipitation control agents, mostly used to control the oxalate impurity precipitation: oxygenated hydrocarbons, fatty acids and oxyalkylates
- antifoam agents: hydrogenated fatty acids
- flocculants for mud decantation, to improve mud settling and separation from the rich liquor: polyacrilic product
- flocculants for mud decantation, to improve mud settling in the mud washing circuit: polyactylamidic product
- dewatering agents, to reduce hydrate moisture at the calciners feed: based on nonyl-phenol etoxylate and oxygenated hydrocarbons
- rheology agent, to reduce viscosity of bauxite slurry and to improve its fluid flow properties: acrylic polymers with sulphonic functional groups.

c) Boiler feed water reagents:

- chelate agent, to reduce incrustation inside the boiler tubes, fed with process condensate: functional groups type NTA (nitrilo-tri-acete) or EDTA (ethylen-diamine-tetracetate), which can capture (sequestrate) Ca and Mg, thereby inhibiting their precipitation inside the boiler water circuit
- deoxygenating agent, to treat boiler feed water: sodium-hydrosulphite
- antifoam agent, to treat boiler feed water
- cleaning agent, for boiler water side.

d) Fuel oil treatment:

- dispersant, to improve burners cleanliness
- magnesium oxide, to reduce smoke side
- tar solvent, to reduce deposition of solids.

e) Water treatment:

- dispersant for cooling water, to reduce scaling rate in the circuit and in the tower
- biocide reagent, for water treatment
- sterilising reagent, for water treatment.

f) Reagents for chemical cleaning:

- sulphuric acid, specific consumption about 9 kg/tonne alumina, to clean digestion heater-tubes, and for the final control of mud pH, before its discharge to the pond
- hydrochloric acid, specific consumption ab.0.4 kg/tonne alumina, to clean press cloths
- corrosion inhibitor for H_2SO_4
- corrosion inhibitor for HCl
- antifoam for acid treatment.

The total quantity of all the above-mentioned reagents amounts to nearly 1 kg/tonne alumina. All organic compounds, which largely decompose into CO_2 and water during the high temperature digestion phase.

In the near future, the **Sardinian** refinery will incorporate a treatment plant for the free water from the pond. Currently the pond water balance has been maintained owing to the favourable climatic conditions (i.e. high net evaporation rate) and by recirculating water from the pond to the mud filters to suspend fresh mud. This recirculation has become more and more important at the refinery during the cold season, because of the reduced evaporative surface, as a consequence of the sequentially raised dam, using the upstream method. Once the water treatment plant is operational, it will allow discharge of the free water from the pond into the sea, and consequently it will eliminate seasonal problems with the water.

3.1.1.4.2 Emissions to air

Air pollution may result from the stack gases of the high capacity alumina calcinating kilns. Here electrostatic filters are used to separate the suspended solid particles.

Dust blowing of the TMF can be an issue, in which case in dry periods spraying with water and hay spreading are applied.

3.1.1.4.3 Emissions to water

Groundwater monitoring is carried out in wells around the stacks and ponds. No effluent is discharged into surface waters [22, Aughinish,].

3.1.1.4.4 Soil contamination

Due to the combined very low permeability of both the red mud and underlying estuarine soil (clayey silt) deposits, seepage into the ground is very limited.

3.1.1.4.5 Energy consumption

The energy consumption related to the tailings management at the Sardinian site is caused by the energy used in three pumping stations, to pump:

- the tailings slurried in water (fresh seawater and recycled water from the pond) from the refinery site to the pond, and distribution within the dam; power utilisation approx. 230 kW, 100 % of the time
- the clarified water from the pond back to the refinery to suspend other mud and reduce the usage of fresh seawater, to keep the total water into balance; power utilisation approx. 60 kW, 70 % of the time
- the fresh seawater necessary for the tailings management, both for neutralisation and for the solids suspending purposes; power utilisation approx. 100 kW, 30 % of the time.

[33, Eurallumina, 2002]

At **Ajka** in 2001, the energy consumptions were as follows:

- energy: 127705 MWh or 21 kWh/tonne of feed
- steam: 788300 t or 1.3 tonnes of steam/tonne of feed
- natural gas: 35360000 m^3 or 58.9 m^3/tonne of feed

3.1.2 Base metals

In this section, information about the following base metals sites is provided:

Area	Site	Country
Aitik	Aitik Mine	Sweden
Almagrera	Aguas Teñidas, Sotiel	Spain
Aznalcollar[1]	Los Frailes	Spain
Boliden Mining Area	Maurliden, Petiknäs, Renström, Åkerberg, Kristineberg	Sweden
Cantabria	Mina Reocín	Spain
Garpenberg	Garpenberg Mine, Garpenberg Norra	Sweden
Hitura	Hitura Mine	Finland
Las Cruces Project[2]	Las Cruces	Spain
Legnica-Glogow copper basin	Lubin, Polkowice-Sieroszowice, Rudna	Poland
Lisheen	Lisheen	Ireland
Pyhäsalmi	Pyhäsalmi, Mullikkoräme	Finland
Tara	Tara	Ireland
Zinkgruvan	Zinkgruvan	Sweden
1. Information on closure 2. Currently in permitting stage		

Table 3.9: Base metals sites mentioned in this section

3.1.2.1 Mineralogy and mining techniques

Mineralogy

Cadmium
There are only a few cadmium minerals, such as greenockite (CdS) or otavite ($CdCO_3$ and as CdO). The chemical element cadmium (Cd) can replace zinc (Zn) in the sphalerite mineral. Hence cadmium is often found in the zinc-concentrate after mineral processing. In this case cadmium is removed at the smelter. In addition lead and copper ores may contain small amounts of cadmium. [35, EIPPCB, 2001]

Copper
The most common copper minerals are:

- sulphides:
  - chalcopyrite ($CuFeS_2$)
  - chalcocite (Cu_2S)
  - covellite (CuS)
  - bornite (Cu_5FeS_4).

The yield of chalcopyrite is rather low in terms of atoms per molecule. It is only 25 %, compared to other copper minerals such as chalcocite – 67 %; cuprite – 67 %; covellite – 50 % or bornite – 50 %. However, the large quantities and widespread distribution of chalcopyrite make it the leading source of copper. Chalcopyrite is a common mineral and is found in almost all sulphide deposits.

- oxides: cuprite (Cu_2O).

Cuprite has long been mined as a major source of copper and is still mined in many places around the world today. Of all the copper ores, excluding native copper, cuprite gives the greatest yield of copper per molecule, since there is only one oxygen atom to every two copper atoms [37, Mineralgallery, 2002].

- others, such as:
  - malachite ($Cu_2(CO_3)(OH)_2$)
  - azurite ($Cu_3(CO_3)_2(OH)_2$)
  - chrysocolla, a hydrated copper silicate ($CuSiO_3$ - nH_2O).

Lead
The most important lead mineral for the mining industry is galena (PbS), which can contain up to 1 % silver.

Nickel
Nickel (Ni) is a transition element that exhibits a mixture of ferrous and non-ferrous metal properties. It is both a siderophile (associates with iron) and a chalcophile (associates with sulphur). The bulk of mined nickel comes from two types of ore deposits:

- laterites, where the principal ore minerals are nickeliferous limonite ((Fe, Ni)O(OH)) and garnierite (a hydrous nickel silicate), or
- magmatic sulphide deposits, where the principal ore mineral is pentlandite ($(Ni, Fe)_9S_8$).

The ionic radius of divalent nickel is close to that of divalent iron and magnesium, allowing the three elements to substitute for one another in the crystal lattices of some silicates and oxides. Nickel sulphide deposits are generally associated with iron- and magnesium-rich rocks called ultramafics and can be found in both volcanic and plutonic formations. Many of the sulphide deposits occur at great depth. Laterites are formed by the weathering of ultramafic rocks and are a near-surface phenomenon. Most of the nickel on Earth is believed to be concentrated in the planet's core.
[36, USGS, 2002]

Tin
The only mineral of commercial importance as a source of tin is cassiterite (SnO_2), although small quantities of tin are recovered from complex sulphides such as stanite, cylindrite, frankeite, canfieldite, and teallite.
[36, USGS, 2002].

Zinc
Sphalerite (zinc iron sulphide, ZnS) is one of the principal ore minerals in the world.
Mining of primary sulphide ores dominates the base metal mining for Cu, Zn and Pb in Europe (Las Cruces, once in operation, will be an exception). The sulphide content and the grade of the value mineral vary significantly between the sites.

Some examples of different mineralogical characteristics found and different mining areas are described below.

- at the **Aitik** site, the contact between the main ore zone and the hanging wall is sharp, as the ore is bound in a thrustfault. The contact between the footwall and the ore zone is gradual and grade dependent. The main ore minerals are chalcopyrite, pyrite and pyrrhotite, which occur as dissemination and veinlet deposits. The footwall consists of biotite-amphibole gneiss and intrusions of quartz-monzodiorite (the footwall has less than 0.26 % Cu.) The main ore zone comprises biotite schist/gneiss and muscovite schist. The hanging wall consists of amphibole-biotite gneiss and pegmatite and is barren in copper. The value mineral in the orebody is chalcopyrite. The mean copper concentration in the ore is 0.4 %. Furthermore, the ore contains gold (0.2 g/t) and silver (3.5 g/t) [63, Base metals group, 2002]

- at the **Hitura** nickel mine, the ultramafic complex consists of three separate, closely-spaced serpentinite massives surrounded by magmatised mica gneiss. The main ore minerals are pentlandite, chalcopyrite and pyrrhotite, but in some places mackinawite, cubanite and vallerite are abundant. Pyrite occurs only in joints with [62, Himmi, 2002]

- at the **Las Cruces Project**, which is currently in the planning and permitting phase, the value mineral is chalcocite, a secondary sulphide copper mineral, in massive pyrite [67, IGME, 2002]

- in the **Legnica-Glogow copper basin** the copper ore occurs at depths from 600 to 1200 m in a 40 m thick bed-type polimetallic deposit, where aside from copper minerals other metals, such as silver, gold, platinum and palladium can be found. Ore minerals occur either in the sandstones of the 'Rotliegendes' or 'Weissliegendes' or in the copper-bearing shales and carbonate rocks of the Werra cyclothem, mainly in the dolomites. In this copper deposit in total over 110 ore minerals have been found. The main metalliferous minerals are chalcocite, bornite, chalcopirite, covellite, pyrite and galena. The distribution of mineralisation in the deposit is highly variable

- at the **Lisheen** site**,** the sulphide mineralisation that forms the orebody occurs at the base of dolomitic limestone. The metalliferous minerals are pyrite, marcasite, sphalerite and galena, and, in smaller concentrations, chalcopyrite, tennatite, native silver, arsenopyrite and gersdorffite. The gangue material is dolomite together with barytes, calcite, shale, illite and quartz [75, Minorco Lisheen/Ivernia West, 1995]

- the ore at **Pyhäsalmi** is massive and coarse grained. The ore contains on average 75 % sulphides, made up of 3 % chalcopyrite, 4 % sphalerite, 2 % pyrrhotite and 66 % pyrite, plus minor amounts of galena and sulphosalts. Barytes and carbonates are the main gangue minerals [62, Himmi, 2002]

- the **Neves Corvo** site is a high-grade copper-tin mine in the Iberian Pyrite Belt. The dominant ore minerals in the volcanogenic massive sulphide-type orebody are pyrite chalcopyrite, sphalerite, galena, cassiterite, stannite, tetrahedrite and arseonpyrite [142, Borges, 2003].

Mining techniques

Both underground and open pit mines are represented in the base metal mining sector in Europe. The mining methods used underground are cut-and-fill, room-and-pillar and various other techniques. The ore production capacity in the underground mines is between 65000 and 1100000 tonnes/yr. In open pit mining, the production (ore and waste-rock) in 2001 was between 1200000 and 43700000 tonnes. In underground mining, almost all the waste-rock produced is directly used as backfill in the mine. In some cases, waste-rock was extracted from existing waste-rock dumps and transported under ground. In open pit mining, backfilling was not possible in the majority of the cases, however, at Mina Reocín a mined out part of an open pit was backfilled using waste-rock. Various mines and the mining techniques they apply as well as their ore and waste-rock production are listed in the table below.

Mining area	Mine	Mining method	Ore production (kt/yr)	Waste-rock deposition (kt/yr)
Aitik	Aitik Mine	Open pit	17700	26000[4]
Almagrera	Aguas Teñidas	Underground (cut and fill)	300	0[1]
	Sotiel	Underground	700	0
Boliden Mining Area	Maurliden	Open pit	224.4	875.7
	Renström	Underground (cut and fill)	160.5	-104[*]
	Petiknäs	Underground (cut and fill)	553	-15.7[*]
	Åkerberg	Underground	32	-21[*]
	Kristineberg	Underground (cut and fill)	503.6	4.6[3]
Cantabria	Mina Reocín	Open pit/Underground	1100	2500[2]
Garpenberg	Garpenberg Mine	Underground (cut-and-fill)	310	0
	Garpenberg Norra	Underground (cut and fill)	709	38.4[5]
Hitura	Hitura Mine	Underground (cut and fill)	518.3	0[3]
Legnica-Glogow copper basin	Lubin	Underground (room and pillar)	6808	0[3]
	Polkowice-Sieroszowice	Underground (room and pillar)	10436	0[3]
	Rudna	Underground (room and pillar)	11490	0[3]
Lisheen	Lisheen	Underground (cut and fill)	1110[6]	7
Pyhäsalmi	Pyhäsalmi	Underground (cut and fill)	1097.2	0[3]
	Mullikkoräme	Underground	64	0
Tara	Tara	Underground (blasthole open stoping)[7]	2000[7]	
Zinkgruvan	Zinkgruvan	Underground (cut and fill)	850	0[4]

1. Waste-rock used in backfill + schists from borrow area
2. Waste-rock used to fill out mined out open pit.
3. Waste-rock used in backfill
4. 65 % deposited separately for alternative use
5. Used for dam construction
6. Source: [76, Irish EPA, 2001]
7. Source: [74, Outokumpu,]

[*]: A negative number indicates that waste-rock has been removed from existing deposits and brought underground for backfilling purposes.

Table 3.10: Information on mining technique, ore and waste-rock production of base metal mines Year 2000 figures for Almagrera, Mina Reocín, Pyhäsalmi and Hitura; year 2001 figures for the Aitik, Garpenberg and Boliden mining areas

The **Aitik** site is a typical example of base metals open pit mining, incorporating the following operations:
Drilling: The drilling equipment consists of rotary drill rigs. The bench height is 15 m and subdrilling 3 m. The drilled burden and spacing are 8 m x 10.5 m. The diameter of the drillholes are approx. 300 mm. The rate of drilling is normally about 17 m/h, but in the hard parts of the ore it can be less than 10 m/h. Water is pumped from the open pit at 3 - 15 m^3/min.
Charging and blasting: Emulsion explosive is pumped from a truck into the blast holes. Non-electric detonators are used for the initiation of the blast. The size of each round is about 600 kt and blasting takes place once a week. The benches are planned with a final pit slope angle of 47° in the footwall (following the foliation) and 51 - 56° in the hanging wall.

Loading and transportation: Three rope shovels and two hydraulic shovels are used. A wheel loader completes the loading fleet. The haulage is carried out by 17 trucks (172 t and 218 t trucks).

In-pit crushing: The ore is transported by trucks to the primary crushers in the pit, 165 m below the surface. The ore is loaded onto a conveyor belt from bins below the crusher. The conveyor belt takes the ore to the mineral processing plant. The inclination of the conveyor is 15°, the width 1800 mm and the capacity 4000 t/h. The total stockpile capacity at the surface is around 50000 t.
[63, Base metals group, 2002]

Both **Garpenberg** and **Garpenberg Norra** are underground mines. The techniques used in these mines are described here as examples of base metal underground mining.

The applied mining method is cut-and-fill. The coarse fraction of the tailings is used as backfill and as a platform when mining the ore above. At present, the ore is mined at a depth of between 400 and 870 m in the Garpenberg mine and between 700 and 990 m in Garpenberg Norra.

Blasting is done using emulsion explosives. Loading and hauling is carried out using diesel vehicles. The ore is crushed with an in-pit crusher before it is skipped through a shaft to the surface. A covered 500 m long conveyor belt transports the ore from the Garpenberg mine to the mineral processing plant. For the Garpenberg Norra mine, the ore has to be trucked approximately 2 km to the mineral processing plant.
[64, Base metals group, 2002]

At the Neves Corvo underground mine, four different mining methods are applied depending on the shape of the orebody. All mine voids are backfilled in order to maximise ore extraction and to reduce surface subsidy [142, Borges, 2003].

3.1.2.2 Mineral processing

In the processing of the primary sulphide ores all plants use similar processing techniques, namely:

- crushing;
- grinding
- flotation
- drying of concentrates.

Flotation can be carried in various ways, e.g., by selective flotation or by bulk/selective flotation, depending on the characteristics of the ore, the market demands, the cost of flotation additives, etc. Two possible options for the same mineral processing plant are illustrated in the figures below for the Zinkgruvan mineral processing plant.

The **Zinkgruvan** mineral processing plant, which was constructed in 1977, is located next to the mine. It operates continuously with an annual throughput of 850000 tonnes. The choice of process and technology is based on a large number of test works with the actual zinc and lead ore. Autogenous grinding in combination with bulk/selective flotation (see Figure 3.10 below) of the ore has been chosen as the main process technique and has been used at Zinkgruvan since 1977.

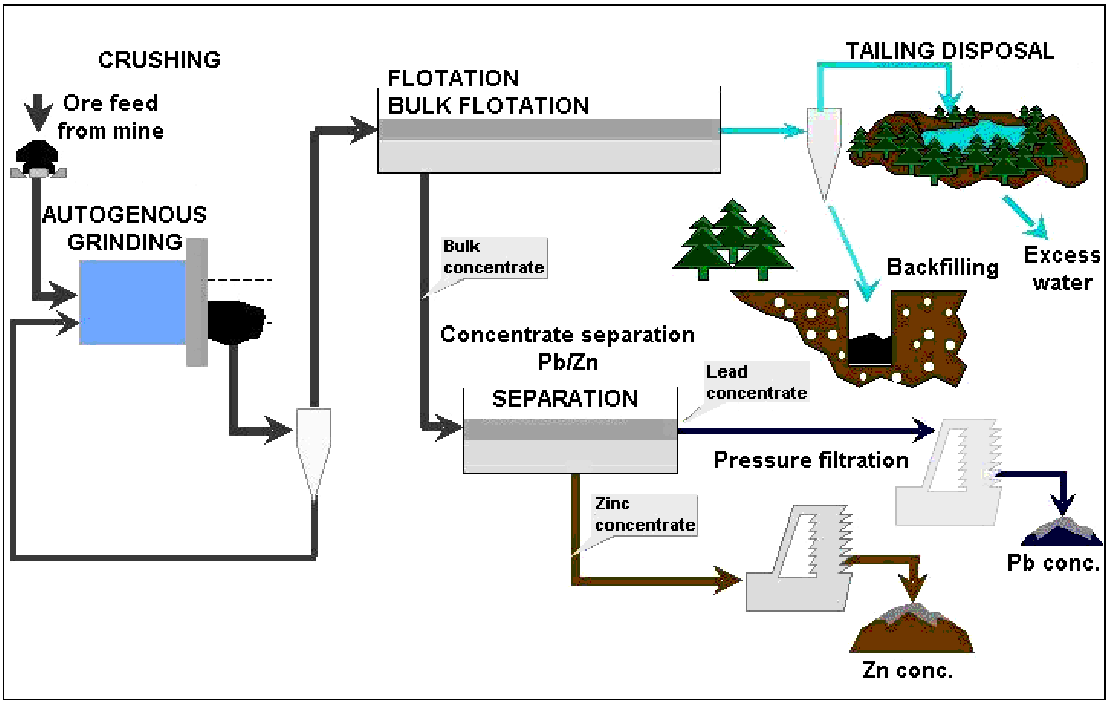


Figure 3.10: Bulk/selective flotation circuit for Zinkgruvan site
[66, Base metals group, 2002]

An alternative flotation method, which could be used if there were changes in the ore composition, would be stepwise selective flotation (see Figure 3.11 below) This would require slightly different process chemicals but is otherwise similarly economical and technically feasible.
[66, Base metals group, 2002]

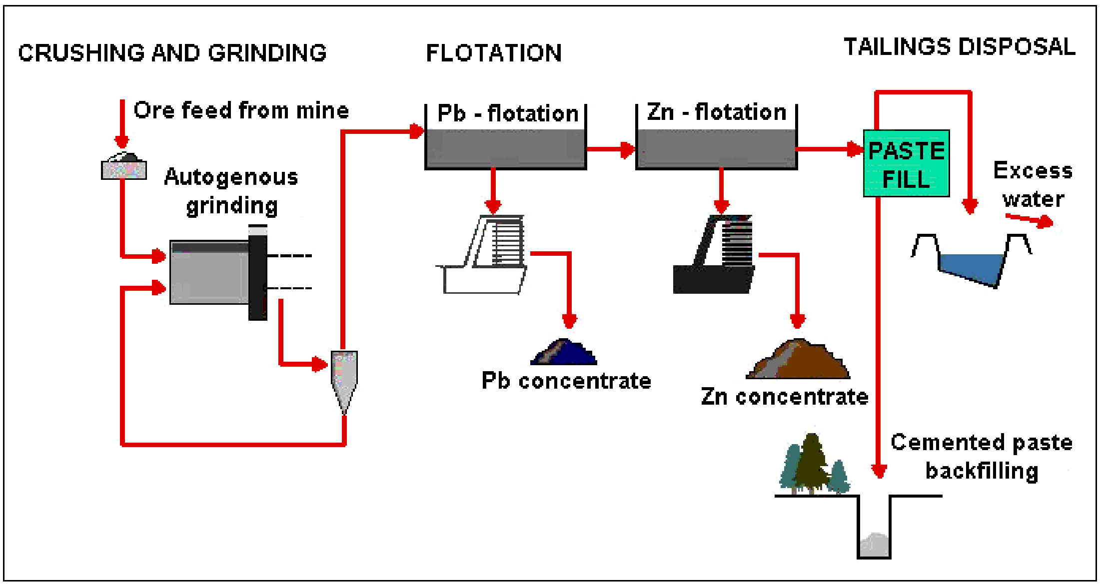


Figure 3.11: Possible selective mineral processing circuit for Zinkgruvan site
[66, Base metals group, 2002]

The mineral processing set-up for the nickel ore at the **Hitura** site is similar to that for the sulphide ores as shown in the figure below.

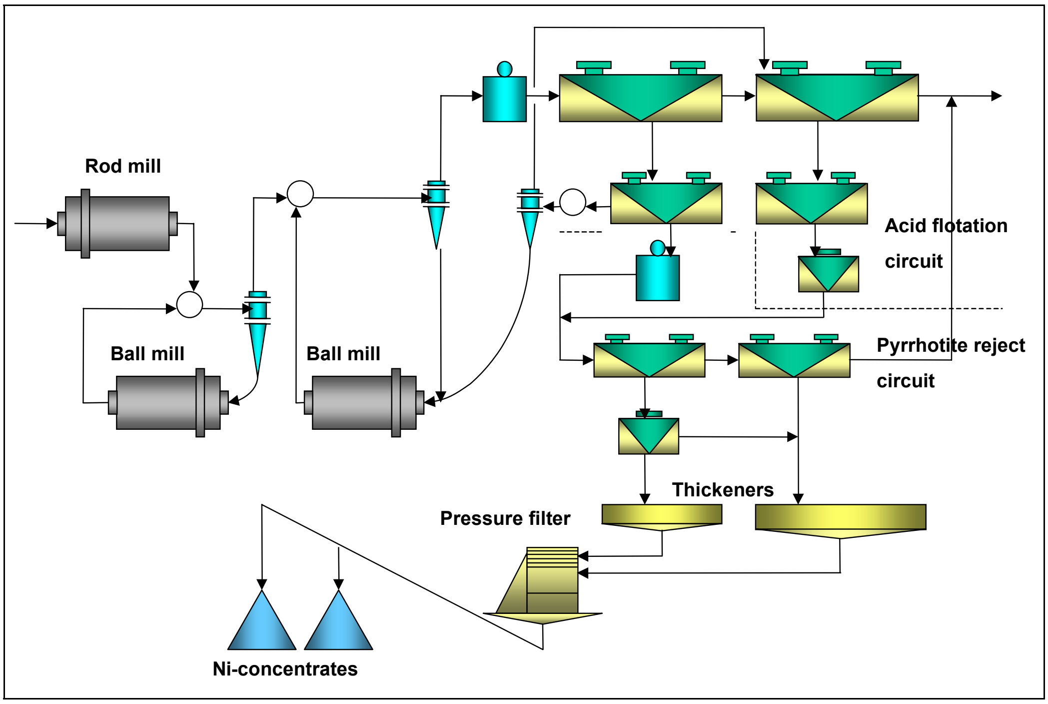


Figure 3.12: Mineral processing flow sheet at Hitura site
[62, Himmi, 2002]

In the **Las Cruces Project** leaching with sulphuric acid is the proposed process method followed by solvent extraction and electrowinning (SX-EW). Tailings will be dewatered using filtration and will be sent to 'dry' lined cells [67, IGME, 2002].

The ores extracted in the **Legnica-Glogow copper basin**, which vary in their lithological and mineralogical composition, are processed in three concentrators (Lubin, Polkowice and Rudna) with a total capacity of approx. 30 million t/yr. In this case, the separation technique most suitable to achieve a maximum recovery of copper and silver, is flotation. Two types of ore are processed: sandstone-carbonate in the Lubin and Rudna facilities, and dolomite-shale in the Polkowice facility.

At **Mina Reocín,** a pre-concentration is done before grinding using gravimetric methods. Tailings are pumped as a slurry to the pond systems. The coarse fraction of the tailings, which is used in the backfilling, is separated from the fines using hydrocyclones [54, IGME, 2002].

3.1.2.2.1 Comminution

Size reduction at all sites is done by crushing and grinding using various types of crushers and mills.

At **Aitik**, two gyratory crushers are used for primary crushing. The intake opening of the crusher is 152 cm and the diameter of the inner surface at the bottom is 277 cm. The fragmentation of the crushed ore depends on the setting of the crusher but normally the width is set to 160 - 180 mm. The largest pieces are thus between 350 and 400 mm but variations occur caused by different ore characteristics. Each day 40000 to 60000 tonnes are crushed and fed to the grinding circuit. This consists of five milling lines, each made up of an AG mill followed by a pebble mill. Each grinding circuit operates in closed circuit with a screw classifier, which feeds back material into the autogenous mill.

This site has several grinding sections, which are described below:

B-section, which comprises two 300 t/h mill lines, is the oldest primary grinding facility. All the mills are run at 75 % of critical speed. C-section is a single 460 t/h line. The AG and pebble mills are run at 76 % and 73 % of critical speed, respectively. D-section, another two 460 t/h lines both run at 75 % of critical speed.

Data B-section:

- two AG mills, 6 m diameter, 10.5 m long, installed power 3600 kW
- two pebble mills, 4.5 m diameter, 4.8 m long, installed power 1250 kW.

Data C section:

- one AG mill, 6.7 m diameter, 12.5 m long, installed power 6600 kW
- one pebble mill, 5.2 m diameter, 6.8 m long, installed power 2500 kW.

Data D-section:

- two AG mills, 6.7 m diameter, 12.5 m long, installed power 6000 kW.
- two pebble mills, 5.2 m diameter, 6.8 m long, installed power 3000 kW.

The total grinding capacity is about 50000 t/d, although the actual throughput depends on the grindability, or the hardness, of the ore. Energy consumption averages around 11 - 12 kWh/t. The grinding is done at 55 % by weight solid material. The finished ground product from the screw classifier has a d_{80} value of 180 µm and about 25 % are smaller than 45 µm.
[63, Base metals group, 2002]

Ore delivered to the **Boliden** mineral processing plant arrives either crushed or uncrushed. A jaw crusher with an opening of 220 mm is installed to crush, if necessary, run-of-mine ore (mostly open pit ore). The size distribution of the ore varies from time to time, from very small rocks to rocks of 200 - 300 mm size. The size variation mainly depends on the ore type.

All ore is stored in four underground bins. Storage capacity varies between 1500 to 4500 tonnes of ore. The underground bins make it possible to blend ores if desired. Underground storage is beneficial during winter, as it minimises freezing problems. Ore from the bins is fed to the mineral processing plant by feeders and conveying belts.

The mineral processing plant uses autogenous grinding. The primary AG mill is followed by a pebble mill, which receives the grinding pebbles through a continuous draw from the output end of the primary mill. Between mills, magnetic separators are installed to clean the pulp of metal scrap from mines. The coarse material is sent back to the mills after screening and hydrocycloning. Both grinding circuits are equipped with Reichert cones, spirals and shaking tables for gravity separation of gold.

The throughput is between 92 and 110 tonnes per hour per circuit depending on the ore. Energy consumption is about 22 kWh/t. The grinding result varies between 50 - 80 % <45 µm.
[65, Base metals group, 2002]

At the **Hitura** mineral processing plant size reduction is achieved via:

- crushing in three stages with a jaw crusher, a gyratory crusher and a cone crusher. The crushing circuit also includes a screen operating in an open circuit
- grinding in three stages with a rod mill (Ø 3.2 x 4.5 m) in the primary stage and two ball mills (Ø 3.2 x 4.5 m) in the following stages.

[62, Himmi, 2002]

The **Las Cruces** project proposes using:

- a primary jaw crusher
- secondary and tertiary cone crushers
- ball mills.

The predicted average grain size after comminution is 100 % <100 µm.
The first stage of crushing in the **Legnica-Glogow copper basin** is carried out underground. In three surface mineral processing plants the ore is first screened. The oversize is crushed in hammer or cone crushers. The screen underflow ground in two stages in rod mills and ball mills. The final grain sizes are as follows:

- in Lubin and Rudna mineral processing plants: 100 % <0.3 mm and 45 – 60 % <45µm
- in Polkowice mineral processing plant: 89 – 92 % <45µm.

At **Lisheen,** the ore is continuously fed from the surface stockpile into a grinding circuit. This consists of a SAG mill, a secondary ball mill, and closed circuit hydrocyclones [73, Ivernia West,].

At the **Neves Corvo** operation, the size reduction of the **copper circuit** is carried out by a primary crusher in the underground mine. Secondary crushing is carried out in the mineral processing plant via two hydroclones in a closed circuit with a screen (350 t/h capacity). Grinding is carried out in a rod mill (3.8 m x 5.5 m, 1600 kW) followed by two ball mills (4.1 m x 6.7 m, 1600 kW each) in closed circuit with hydrocyclones (230 t/h capacity). The feed to the flotation circuit has a d_{80} of 45 µm.

Size reduction for the **tin circuit** commences with the crushing section, which consists of an open circuit jaw crusher and 2 x 4.25 cone crushers, the second in a closed circuit with a 12-mm screen. The plant has a capacity of 80 t/h. The grinding circuit consists of a 3 m x 1.8 m rod mill in open circuit, followed by a 3 m x l.8 m ball mill in closed circuit with a screen sieve, together providing a flotation feed with a d80 of 350µm.
[142, Borges, 2003]

At the **Pyhäsalmi** mine, comminution is carried out by:

- one stage crushing with a jaw crusher located in the underground mine
- autogenous grinding in three stages (balls are used in the tertiary stage)
- in the grinding circuit five ball mills (3.2 x 4.5 m).

[62, Himmi, 2002]

At **Zinkgruvan** a primary crusher is situated underground. From a temporary storage at ground level, normally containing about 9000 tonnes, the ore is transported to the secondary crusher where two size fractions are produced:

- >100 mm as pebbles for the AG mill
- 25 – 100 mm is recycled
- <25 mm to AG mill.

An optimum mixture of the two size >100 mm and <25 mm fractions is then fed to the AG mills. Autogenous grinding is used to generate a product with 90 % <100 µm at 40 % solids.
[66, Base metals group, 2002]

The above information on comminution is summarised in the following table.

	Aitik	Boliden	Hitura	Las Cruces	Legnica-Glogow	Lisheen	Pyhä-salmi	Zink-gruvan
Crushing in pit/ug	In-pit cone	Jaw	Jaw	Jaw	Ug crusher	Ug crusher	Ug jaw	Ug crusher
Crushing in mpp			Cone cone	Cone cone	Hammer cone			Sec crusher
Grinding	AG PM	AG PM	RM BM BM	BM	RM BM CM	SAG BM	3 stage AG BM	AG
Lines	5	2	1		29		1	1
Line capacity (t/h)	500	100	90		86 - 180		150	115
ug = underground jaw = jaw crusher cone = cone crusher mpp = mineral processing plant AG = autogenous grinding mill RM = rod mill BM = ball mill CM = cylpeps mill PM = pebble mill SAG = semi-autogenous grinding mill								

Table 3.11: Equipment types used for comminution, number of lines and throughput

3.1.2.2.2 Separation

At **Aitik,** the flotation is divided into two steps, one circuit for bulk flotation and one cleaning circuit. The bulk flotation consists of four parallel lines of nine mechanical flotation cells in each line. The cleaning step consists of four flotation and 16 mechanical flotation cells.

The feed pulp is conditioned with frothers and collectors and the pH-value is raised to 10.5 by adding lime. In the bulk flotation chalcopyrite and pyrite are floated together. Each flotation line is divided into two steps, where the first four cells are used for rougher flotation and the last five as scavengers. In the rougher flotation a bulk concentrate is achieved with 10 - 15 % Cu. The rougher concentrate from the four lines is fed to the cleaning circuit. The scavenger concentrate (1.3 % Cu) is reground in a pebble mill.

In the cleaning circuit, the chalcopyrite is separated from the pyrite after regrinding and further addition of lime. The rougher concentrate, together with returned products from the separation circuit, is reground in a ball mill in closed circuit with hydrocyclones. The cyclone overflow flows to the columns. The concentrate from column one and two normally holds 20 - 25 % Cu and is mixed together for cleaning in two steps in small mechanical cells. The final concentrate contains 28.8 % Cu, 8 g/t Au and 250 g/t Ag. The concentrate is dewatered with a continuous thickener, drum filters and oil-fired rotating kilns. Dried concentrate is shipped in containers by truck 20 km to the railroad and then by rail 400 km to the smelter.

The mineral processing plant runs on 100 % re-circulated water from the tailings pond system and recovers 90 % of the copper, 50 % of the gold and 70 % of the silver. It is equipped with a distributed control system and an on-line analysis system.
[63, Base metals group, 2002]

At **Hitura** mine the separation is done by flotation. All flotation machines are mechanical. An automatic process control system with two on-line x-ray analysers (six slurry lines) is also installed.

Dewatering is carried out using two continuous thickeners for Ni-conc. (Ø 25 m + Ø 10 m) and a pressure filter (25 m^2).
The reagents added to the process at Hitura are:

- grinding: Sodium Ethyl Xanthate (SEX)
- flotation: H_2SO_4, SEX, frother, carboxymethylcellulose (CMC), lime (cleaning).

[62, Himmi, 2002]

At the **Las Cruces Project** it is proposed to use pressurised leaching with sulphuric acid followed by Solvent Extraction and Electro-Winning (SX-EW) to recover the copper [67, IGME, 2002].

At the mineral processing plant treating the ore from the **Legnica-Glogow copper basin** the flotation process is performed in three stages: rougher, scavenger and cleaner. Additionally so called 'flash flotation' (or skimmer) has been introduced, at the initial grinding and classification stage at the Polkowice and Lubin plants. The concentrate from flash flotation contains 30 – 45 % Cu). At the Rudna plant flash flotation is currently being introduced to replace the rougher flotation.

In all three plants the water consumption is 4.5-5.2 m/t ore.

As collectors, a mixture of Sodium Ethyl Xanthate (SEX), Sodium Isobuthyl Xanthate (SIBX) and hostaflot LET (salt of sodium diethylene ditiophosforic acid) are consumed at a level of 50 – 68 g/t ore. Carflot (a mixture of buthyl ethers and di-, tri-, and tetra-ethylene glycols) is used as a frother (consumption: 22 g/t ore). The pH is neutral (7-8), neither milk of lime, nor any polyelectrolytes are added.
The process is controlled continuously by X-ray analysers.

The recovery level is 87 – 90 % for copper and 83 – 87 % for silver. The final concentrate contains:

- 18 % Cu and 1000 ppm Ag (from Lubin)
- 27.2 % Cu and 480 ppm Ag (from Polkowice)
- 30.5 % Cu and 640 ppm Ag (from Rudna).

The concentrate is dewatered in thickeners, filtration presses (up to 12 – 14 % moisture content) and gas fired drum dryers (up to 8.5 % moisture content) before it is sent to the smelters.
[KGHM Polska Miedz, 2002 #113]

At **Lisheen,** the ground ore is fed into a lead circuit, and then into a zinc circuit. The lead and the zinc circuit use mechanical flotation cells while the zinc circuit also uses flotation columns. The zinc circuit utilises a regrind step to assist in the production of a high-grade concentrate and to maximise metal recovery and an acid leach circuit is also added to ensure low levels of magnesium oxide in the concentrate [73, Ivernia West,]. Process water is recycled and supplemented with water reclaimed from the TMF.

The **copper separation at Neves Corvo** is achieved by flotation. The tin separation is carried out by gravity separation on Holman-Wilfley shaking tables and subsequently by cassiterite flotation [142, Borges, 2003].

At the **Pyhäsalmi** mine the separation is done using a flotation circuit composed of Cu-, Zn and finally Pyrite flotation. All flotation cells are mechanical type.

Backfilling material (coarse fraction of the tailings) is separated from the fine tailings in a hydrocyclone (Ø 500 mm) before pumping the fines to the tailings pond.

Reagents added to the process are:

- grinding: lime, $ZnSO_4$, Sodium isobutyl xanthate (SIBX), frother
- Cu-flot.: lime, $ZnSO_4$, SIBX, frother, NaCN
- Zn-flot.: lime, $CuSO_4$, SIBX, frother, NaCN (cleaning)
- Pyrite-flot.: H_2SO_4, SIBX
- dewatering: flocculant (thickeners), HNO_3, CH_3COOH (filters)
- tailings: lime (neutralisation).

[62, Himmi, 2002]

At **Tara**, sphalerite and galena are selectively floated while pyrite is depressed. Selective removal of galena is enhanced by the collector Sodium Isopropyl Xanthate SIPX. MIBC is added as a frother. During galena flotation sphalerite and pyrite are depressed with quebracho tannin, lignossal, starch and sodium cyanide. In the subsequent flotation of sphalerite, copper sulphate and calcium oxide are added to reactivate the sphalerite and to increase pH. Thiocarbonate and Potassium Amyl Xanthate (PAX) are used as collectors and MIBC as the frother.
[101, Tara mines, 1999]

At **Zinkgruvan** the flotation process is done in two steps, as above, with bulk flotation followed by zinc and lead separation. In the bulk flotation sulphuric acid is added in order to lower pH to approx. 8 from its natural level of approx. 9. As collector for the desired minerals (galena and sphalerite) sodiumisopropylxanthate (SIBX) is used, together with methylisobutylcarbinol (MIBC) as frothing agent. In the bulk flotation circuit separate regrinding is done to improve the purity of the concentrate. The bulk concentrate recovers 98 %, 95 % and 85 % of the total ore content of zinc, lead and silver respectively.

Sodium hydroxide is added to the zinc/lead separation step to increase pH to about 12. The zinc concentrate is directly produced, whilst the lead concentrate requires additional flotation in multiple steps in order to achieve the final lead concentrate.
[66, Base metals group, 2002]

3.1.2.3 Tailings management

Tailings are used in the backfill of most underground operations. At these sites 16 – 52 % of the tailings are backfilled. One site, **Mina Reocín**, backfills an old open pit using 94 % of the tailings. The tailings that are not used for backfilling need to be managed in ponds. For the **Las Cruces project**, it is proposed to deposit dewatered tailings in lined cells. At **Almagrera** the coarse fraction of the tailings (33 %) is roasted and sulphuric acid is produced. The cinders are then leached and copper extracted in an SX-EW process. The cinders are deposited in a cinders dam. The remaining 2/3 of the tailings are deposited into a tailings pond.

Tailings production and the percentage of backfilled tailings at the various mineral processing plants is summarised in the table below.

Site	Mining method	Tailings production (t/yr)	Tailings used in backfill (%)
Aitik	Open pit	17700000	0
Almagrera	Underground	900000	0
Boliden Mining Area	Open pit/underground	1457000	29
Garpenberg	Underground	910000	50
Hitura	Underground	518331	0
Legnica-Glogow copper basin	Underground	27000000	0
Lisheen	Underground	910000	50
Mina Reocín	Open pit/underground	950000	94
Neves Corvo	Undergrounde	1370000	30
Pyhäsalmi	Underground	213816	16
Tara	Underground	1680000	52
Zinkgruvan	Underground	850000	50

Table 3.12: Per cent of tailings backfilled at base metal operations

Almagrera uses waste-rock and rock from quarrying (schist) in the backfill and not tailings. **Mina Reocín** fills out a mined out open pit, which explains the high backfill percentage. **Zinkgruvan** and **Garpenberg** run backfilling operations which utilise 45 – 50 % of the tailings in the backfill. The **Boliden Mining Area** received ore from one open pit and a series of underground mines. If the ore from the open pit is subtracted from the tailings production the percentage of backfilled tailings is 34 %. This value is misleading since during year 2001 large quantities of waste-rock were brought back underground at Renström, Petiknäs and Åkerberg mines (a total of 140000 tonnes of waste-rock was brought back underground during 2001).

Base metal ores usually contain several metalliferous minerals. Often copper, lead and zinc are mined together. Typically base metals are mined as sulphides. Hence, acid rock drainage is a major issue in the management of tailings and waste-rock. Long-term chemical stability is, therefore, a challenge. The tailings are in the form of a slurry and the ponds and dams can be of large dimensions.

A suite of metalliferous complexes and process chemicals are included in the tailings slurry. Hence physical stability is also of major importance for this sector.

3.1.2.3.1 Characteristics of tailings

At **Almagrera,** there are two types of tailings. The fine fraction of the tailings and the cinders resulting from the roasting and leaching of the coarse fraction of the tailings. The tailings are mainly pyrite and ARD generating. The cinders are easily leached with water. The tailings have a 66 % solids content and the compact density of the tailings material is 4.0 t/m^3 (mainly pyrite). Upon discharge into the tailings pond, the tailings have an initial pH of approximately 9 but pH in the pond is around 3.2.

At **Aitik,** the main issue for the closure and decommissioning plans for the tailings pond is the possible acid generating potential. Due to an early assumption that the material would produce ARD, a number of options to change the composition of the material have been investigated. In its crude form, the tailings have an ABA value of -13 kg $CaCO_3$/t, determined by the pyrite content (0.9 % S). Flotation tests and sampling of various products in the mineral processing plant have yielded a range of samples with sulphur contents ranging from 0.12 % for de-pyritised tailings to 31 % for the pyrite flotation product. These samples have been subjected to humidity cell tests in different campaigns.

The results from the kinetic tests and modelling indicate that the silicates in the tailings constitute a substantial acid consuming capacity. More important, however, is the sulphide oxidation rate in the field. The dissolution of silicates is capable of consuming the acid produced by pyrite oxidation in the tailings up to a certain rate. Below that rate, the carbonates are slowly consumed, but above that rate, the carbonates are slowly depleted, after which the silicates alone are unable to neutralise the acid generated.

Field oxygen flux measurements have been carried out to illustrate the material's behaviour in field scale. The results indicate acid production will take place, corresponding to the silicate acid consumption capacity of only the top 20 cm layer of tailings. In lower strata, no acid will be produced, indicating a vast excess of buffering capacity.

In Aitik, where frost conditions prevail for seven months of the year, the kinetics differ significantly from the conditions in the laboratory and during the actual field test. To verify that the tailings do not possess ARD capabilities, column tests have also been carried out, under conditions which are representative for the unfrozen period at Aitik. In this test, the measured oxygen consumption rate was 50 % below the lowest oxygen consumption rate calculated from the sulphate export in the humidity cell experiments.

Parallel to these tests, hydrogeological modelling of the groundwater flow within the pond have shown that over 90 % of the volume will be permanently water saturated, which is technically equal to sub-aqueous tailings management. Only minor areas at the upstream and downstream dams may become unsaturated at times. To address the situation, a solution has been derived suggesting a wetland be established in the lower parts of the tailings pond. Unsaturated areas in the lower parts of the pond would then be avoided, leaving the problem with only a small fraction of the total tailings, at the upstream dam, unresolved at present.

A possible solution for the remaining, upper part of the pond, is pyrite separation and selective management of pyrite (de-pyritisation). Such a solution, however, does not eliminate possible problems, it only concentrates the pyrite into a high-potential acid generating material. This requires a technical solution which is of high quality and low risk. Such a solution could be deposition of this material in the bottom of the mined out open pit upon closure, whereby it would then remain permanently covered by water.
[63, Base metals group, 2002]

The **Boliden** mining area consists of complex sulphide mineralisations. Mining in the area started in 1925 and, to date, approximately 30 mines have been exploited in the area. The tailings in the pond consequently have variable chemical characterisations and physico-chemical properties. The characteristics of the tailings produced today are summarised in the tables below. The fine fraction after cycloning is deposited to the tailings pond and the coarse fraction is used as backfill in the underground mines.

Size	**Total tailings**	**Hydrocyclone overflow to pond**
µm	**Cumulative % passing**	**Cumulative % passing**
350	100	100
250	99.9	100
180	99.7	100
125	97.8	100
88	93.5	95.6
63	85.9	87.8
45	76.6	78.3
20	53.2	54.4
-20	0	0

Table 3.13: Particle size distribution of tailings at the Boliden site
[65, Base metals group, 2002]

The tailings have the following composition before cycloning and CN leaching:

- Au: 0.85 g/t
- Ag: 24.9 g/t
- Cu: 0.10 %
- Zn: 0.40 %
- Pb: 0.13 %
- S: 17.8 %

More than 50 % of the tailings consist of particles less then 2 µm. The tailings slurry pumped to the tailings pond contains 20 - 25 % solids. The density of the tailings, as placed in the pond, is 1.45 t/m^3.
[65, Base metals group, 2002]

At **Mina Reocín**, the tailings are in the form of a slurry, a mixture of water and dolomite, with 65 % solids content and with a solids density of 2.75 t/m^3. The tailings are alkaline at the time of discharge (pH 6.5 to 8) and are reported to be easily compactable and not reactive (due to their alkaline nature).

At **Garpenberg,** the tailings were investigated with regard to composition and weathering characteristics. The methods used were mineralogical investigations, full rock analysis, Acid Base Accounting (ABA) and kinetic weathering tests (extended humidity cell tests conducted between 1995 and 1999) in combination with predictive modelling. All results show that the tailings will not produce ARD. The metal concentrations in the pore water of the tailings will have limited solubility at the naturally high pH within the pond even if the tailings are allowed to weather with full access to atmospheric oxygen. The metals mobilised by sulphide oxidation at the surface of the tailings will be immobilised by absorption and precipitation as they are transported through the tailings. Based on these results, it was concluded that no measures were necessary in order to limit the mobilisation of metals by weathering from the deposit at closure.

The tailings presently produced show large variations in mineralogy as other parts of the orebody are mined with higher sulphide content, primarily higher content of pyrrhotite (FeS). According to sampling and analysis done during 2001, it is predicted that these 'new' tailings will produce ARD (see the detailed analysis in the table below).

Following the development of the weathering, characteristics of the tailings is considered important, even though the planned decommissioning method (flooding) is well suited for potentially ARD producing tailings. Therefore, sampling and testing of the tailings will continue in the future.
[64, Base metals group, 2002]

Element	Concentration (mg/kg)
As	56.3
Ba	338.8
Be	0.45
Ca	30933
Cd	18.6
Co	6.1
Cr	3.2
Cu	317.7
Fe	65533
Li	4.6
Mn	4163
Mo	2.9
Ni	7.8
P	149
Pb	4011
S	44600
Sn	<5
Sr	19.6
V	9.5
Zn	7051

Table 3.14: Average results of tailings analysis at the Garpenberg site (2001)
[64, Base metals group, 2002]

Some of the key information regarding the tailings deposited in the tailings pond can be listed as follows:

- 500000 tonnes of tailings/yr
- discharged into the pond at 20 % solids
- typical particle size distribution (% passing) (d_{50} = 20 µm, d_{80} = 64 µm).

Size (µm)	Cumulative % passing
500	100
350	99.8
250	99.7
180	99.4
125	97.5
90	93.3
63	79.1
45	68.1
20	50.8
10	31.6

Table 3.15: Size distribution of tailings at the Garpenberg site
[64, Base metals group, 2002]

Some of the key information regarding the tailings used as backfill at the Garpenberg are:

- 450000 tonnes of backfill/ year
- 80 - 85 % solids.

Size (µm)	Cumulative % passing
250	96.6
180	86.8
90	46.4
45	18.8

Table 3.16: Typical size distribution of backfilled tailings at Garpenberg site [64, Base metals group, 2002]

At the **Hitura** site the same tailings examinations as at Pyhäsalmi have been performed. The most significant problems with the tailings are the contents of Cu and Ni. The tailings will not produce ARD because the buffering capacity of the tailings is higher than the acid formation potential. The particle size distribution of the tailings is 60 % <74 µm.
[62, Himmi, 2002]

For the **Las Cruces Project** the tailings generated during the estimated lifetime of the project will amount to approximately 4 Mm^3 (or 15 million tonnes). The tailings are pyritic and are predicted to be ARD generating. The average grain size is estimated to be 100 µm. The tailings will be deposited 'dry' after dewatering, with a moisture content of approx. 7 - 8 [67, IGME, 2002].

At the **Legnica-Glogow copper basin** the tailings from all three mineral processing plants are pumped to a single tailings pond at 14-20 % solids. The composition and particle size distribution are shown in the following tables.

		Mineral processing plant		
Element/ compound	Unit	Lubin	Rudna	Polkowice
Cu	%	0.16	0.21	0.26
Pb	%	0.06	0.04	0.026
Zn	%	0.007	0.006	0.004
Fe	%	0.57	0.54	0.48
S (total)	%	0.27	1.12	0.66
S (s^{2-})	%	0.15	1.01	0.12
C (total)	%	2.80	4.14	9.26
C (organic)	%	0.48	0.32	0.54
SiO_2	%	68.03	53.05	18.42
CaO	%	5.43	12.14	26.25
MgO	%	3.15	5.72	6.88
Al_2O_3	%	3.09	4.11	4.58
Mn	%	0.094	0.153	0.190
Na	%	0.26	0.40	0.40
K	%	1.23	1.20	1.17
As	g/t	71	10	37
Ag	g/t	13	7	6
Co	g/t	39	10	21
Ni	g/t	27	16	42
V	g/t	72	38	110
Mo	g/t	15	12	8
Au	g/t	0.002	0.006	0.008

Table 3.17: Chemical analysis of tailings from the Legnica-Glogow copper basin [KGHM Polska Miedz, 2002 #113]

Tailings type:	Particle size >0.1 mm (%)	0.1 - 0.045 mm (%)	<0.045 mm (%)
Sandstone-carbonate ore (processed at Lubin and Rudna)	27 - 36	16 - 35	40 - 60
Dolomite-shale ore (processed at Polkowice)	-	8 - 11	89 - 92

Table 3.18: Particle size distribution of tailings from the Legnica-Glogow copper basin [KGHM Polska Miedz, 2002 #113]

As the tailings have a low concentration of sulphur (S^{2-} <1%) and a high concentration of buffering carbonates (20 – 80 %), no ARD has occurred so far or is bound to occur in the future. [KGHM Polska Miedz, 2002 #113]

The tailings are delivered to the TMF at **Lisheen** with about 35 % solids and contain zinc, lead, some process reagents and metal salts that have a grain size of 80 % <95 µm. The density of the tailings on a dry basis is 3.5 g/cm^2. The in-situ density is about 1.7 g/cm^2. ABA was performed at the permitting state and the tailings are predicted to be acid generating [75, Minorco Lisheen/Ivernia West, 1995].

The tailings at Neves Corvo are relatively fine, with a d_{80} of 30 – 40 µm. The following table shows the minerals present in the tailings:

Mineral	Weight-%
Pyrite (FeS_2)	84 – 90
Arsenopyrite (FaAsS)	3 – 7
Chalcopyrite ($CuFeS_2$)	1.5 – 2.5
Sphalerite (ZnS)	1.0 – 2.5
Tetraedrite, Tenandrite $(Cu, Fe)_{12}(Sb, As)_4S_{13}$	1 – 2
Non-metallic minerals	8 – 12
Others	1 – 2

Table 3.19: Mineralogical composition of tailings at the Neves Corvo site [142, Borges, 2003]

The tailings have a high acid generating potential (AP: 910 kg $CaCO_3$/tonne). It is expected that throughout the lifetime of the mine a total of 42 million tonnes of tailings will be generated, of which 14 million tonnes will be backfilled.
[142, Borges, 2003]

At **Pyhäsalmi,** the chemical composition and leaching behaviour (max. solubility/DIN 38614-S4 by Kuryk`s method and long-term behaviour) of the tailings have been determined in laboratory scale simulation tests. Neutralisation capacity vs. acid formation potential of material has been investigated. Also wind erosion tests have been done on laboratory scale. The most significant problems are the contents of heavy metals (As, Cd, Cu, Pb, Zn) and sulphur, resulting in an ARD generating potential. Alternative processing methods to change the characteristics of tailings have been considered. One example is the selective flotation of pyrite in the tailings to achieve a final S-content of less than 1 %. This is technically possible, but in this case economically not viable. The process would generate a product (pyrite) that is impossible to sell and that requires special techniques and arrangements for its deposit or destruction.

Mixing of peat with the tailings when it is pumped to tailings area to create reducing conditions has also been investigated. The test was stopped because of technical difficulties, but the intention is to continue the investigation on a laboratory scale. The down side of this technique is the fact that a natural resource is ‘consumed’.

The particle size distribution of tailings material is 65 % <74 µm.
[62, Himmi, 2002]

The sphalerite concentrate at **Tara** is washed with sulphuric acid to remove dolomite ($CaCO^3.MgCO^3$). This treatment precipitates magnesium and calcium sulphates, which are added to the tailings stream. The tailings slurry also includes collectors, suppressants and MIBC.
[101, Tara mines, 1999]

At **Zinkgruvan** the tailings mainly contain quartz, feldspar and calcite. Small quantities of sulphides are also present (sulphur content <0.25 %). The calcium content is approximately 8 %. The ratio between sulphur and calcite is <0.1 suggesting that the tailings are well buffered and will not produce ARD. Weathering tests have also shown that the tailings have a low weathering rate. The composition of the tailings is given in the table below.

Mineral	Weight %
SiO_2	62.4
TiO_2	0.3
Al_2O_3	11.8
Fe_2O_3	0.6
FeO	2.9
MnO	0.7
MgO	2.2
CaO	7.0
BaO	0.01
Na_2O	0.6
K_2O	4.9
$H_2O^{110\text{-}350}$	0.1
CO_2	2.1
B_2O_3	0.1
FeS	0.5
ZnS	0.2
PbS	0.1
Other minerals	3.3
TOTAL	**100**

Table 3.20: Chemical analysis of tailings at the Zinkgruvan site
[66, Base metals group, 2002]

Once settled in the pond the tailings have an in-situ permeability of 10^{-5} - 10^{-6} m/s and an in-situ density of 1.35 - 1.45 t/m^3.

3.1.2.3.2 Applied management methods

At **Aitik,** the tailings are pumped to a 14 km^2 (7 km x 2 km) tailings pond. Four pipelines (rubber lined steelpipes) are used for this purpose, although normally only two are in use at any one time. All four lines are equipped with five pumps in series. The total installed power for each line is 2000 kW. The water from the tailings pond feeds into a clean water clarification pond.

The tailings pond is limited by the topography (valley-site type) and four dams, see figure below. The tailings are pumped as a slurry to the discharge area along dam A-B. There the spigotting leads to an accumulation of the coarser particles close to the dam A-B, while the finer fractions successively settle along the pond towards the downstream dam, where separated water is collected. The active water volume in the tailings pond is normally about 2 Mm^3, which occupies about 1/5 of the pond's surface area. The water is discharged using a spillway and a steel lined culvert located at the contact between the dam and the valley side. In the future, a system of open channels in undisturbed ground will be used for discharging the water, eliminating the culvert through the dam.

The clarification pond is located west of the tailings pond, downstream dam E-F and the E-F dam extension. The pond's area is 1.6 km^2 and the holding capacity is about 15 Mm^3. This pond serves as:

- the final treatment step for the process water
- a reservoir for process water
- and as a buffer water for spring snowmelt and precipitation events.

The freezing of the process water during winter is a climatic effect that is of particular importance for the water balance. At excessive precipitation and snowmelt, water is discharged from the pond to the receiving streams. Also, when necessary, discharge of water is possible from the recycling water channel.

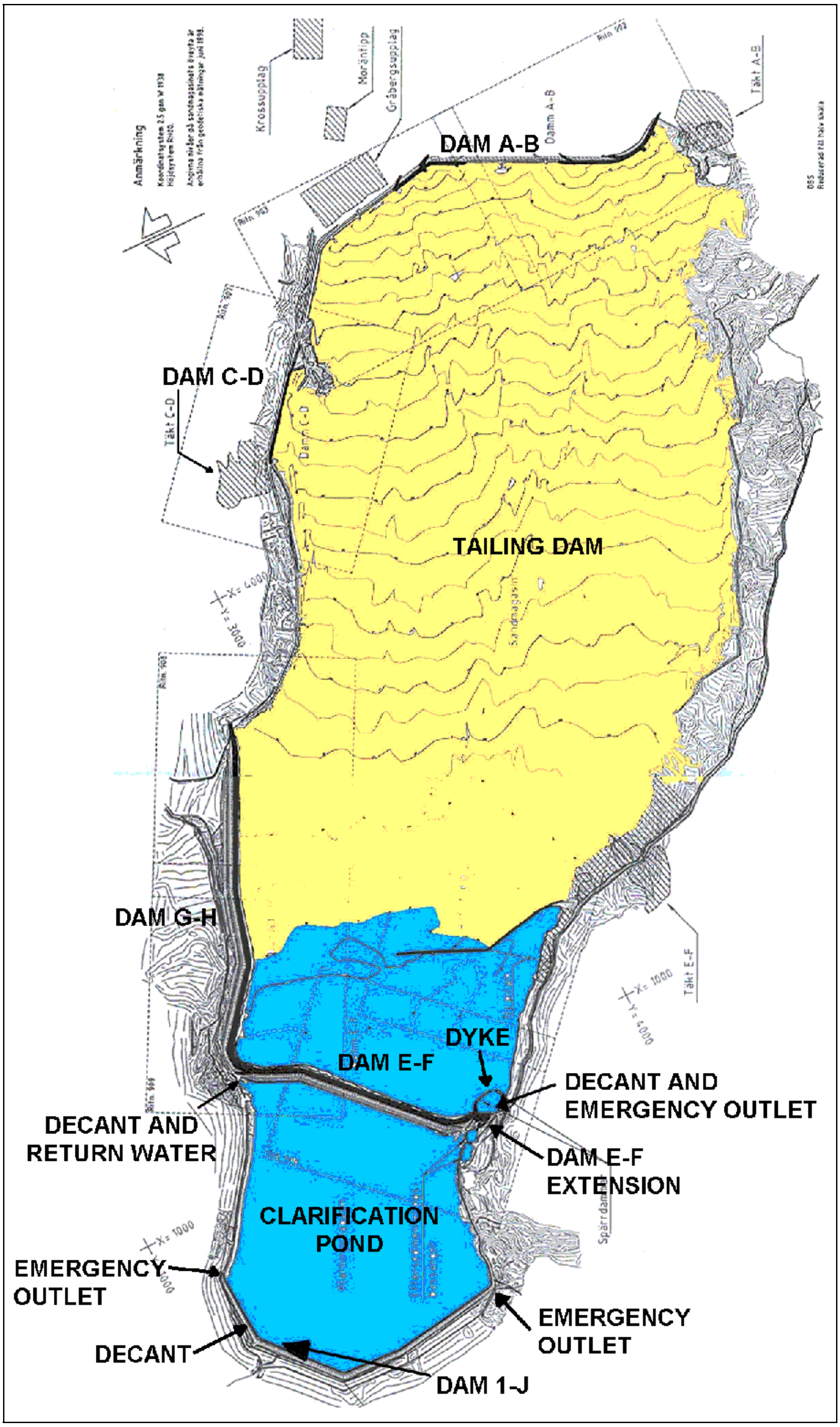


Figure 3.13: Year 2000 situation of Aitik tailings and clarification ponds [63, Base metals group, 2002]

The non-permeable dams surrounding the pond were constructed starting in 1966 and have since been raised mainly applying the upstream method (see figure below). Each raise has been of about 3 m. The material used for the raises have been till for sealing cores and waste-rock for the support fill. For the construction of the E-F dam extension, which started in 1991, the downstream method was used, with the crest of the dam moving outwards from the pond.

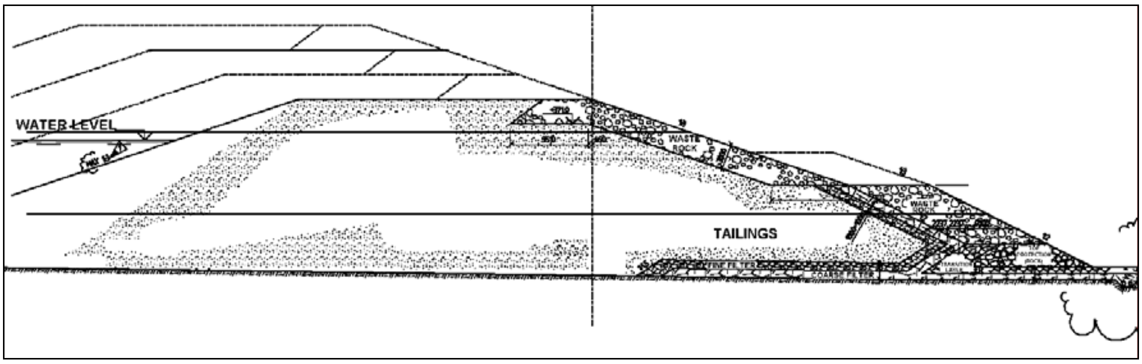


Figure 3.14: Cross-section of dam at Aitik
[63, Base metals group, 2002]

At **Almagrera**, the coarse fraction of the tailings (33 % or 300000 t/yr) are roasted and sulphuric acid is produced. The cinders are then leached with sulphuric acid and copper extracted in a SX-EW process. The cinders are deposited in a cinders pond. The remaining 66 % of the tailings (600000 tonnes fines) are deposited into a tailings pond. The dam is constructed without utilising liners. It is an earth dam with a core of compacted clay. The volume of the dam is 3.2 Mm^3. Leakage through the dam is back pumped into the pond. Clarified water is pumped to a water treatment plant (liming) and treated before discharge. The emergency outlet is constructed in natural bedrock.

The cinders are deposited in a cinders dam.
[61, IGME, 2002]

The tailings management at the **Boliden** area is described in the Section 3.1.6.3.

At **Mina Reocín,** 94 % (900000 out of 950000 t/yr) of the coarse tailings, which are filtered to 15 % moisture content, are used to backfill an old open pit. The remaining 50000 t/yr are deposited in a tailings pond due to the limited filter capacity,. The capacity of the pond is 2.6 Mm^3 and it currently contains approximately 2.5 Mm^3 of tailings. The dams are constructed of borrow material. The pond is built on top of natural soil. The decant water is discharged to the recipient after having passed a series of clarification ponds. No water is recycled back to the mineral processing plant. 100 % of the required 2.2 Mm^3 of process water are pumped from the mine [54, IGME, 2002].

All mining voids (or openings) created at **Garpenberg** are backfilled with waste-rock from development works and tailings. The concentrates constitute about 10 % of the ore processed which means that the other 90 % become tailings. 50 % of the tailings are used for backfilling. When the ore is blasted, crushed and ground, the volume increases by about 60 %, which means that the volume of tailings in Garpenberg is about 145 % of the volume of mined ore. There are no possibilities to backfill more tailings underground due to geometric reasons.

The tailings are cycloned in order to separate fine and coarse particles. The coarse particles are filtered to remove water and to allow transport by trucks. At one mine they are also mixed with cement to stabilise the backfill. After mixing with water, the cemented backfill is transported hydraulically to mined-out areas of the mine and excess water is removed by a draining system.

The tailings pond presently used in the Garpenberg area is located approximately 2 km south-west of the mineral processing plant. Before applying for the latest permit to increase the height of the tailings pond, various alternative tailings management methods were investigated, such as:

- thickened tailings and
- sub-aqueous discharge into a lake.

These alternatives were rejected because of the high cost (thickened tailings) and the public opinion against sub-aqueous deposition.

The presently active part of the tailings pond covers approximately 35 ha. The lifetime of the pond depends on the tailings production rate but is approximately 8 years assuming the present production rate. The tailings have an effective density of 1.5 t/m^3. Currently, the dam is raised using the downstream method (see figure below).
[64, Base metals group, 2002]

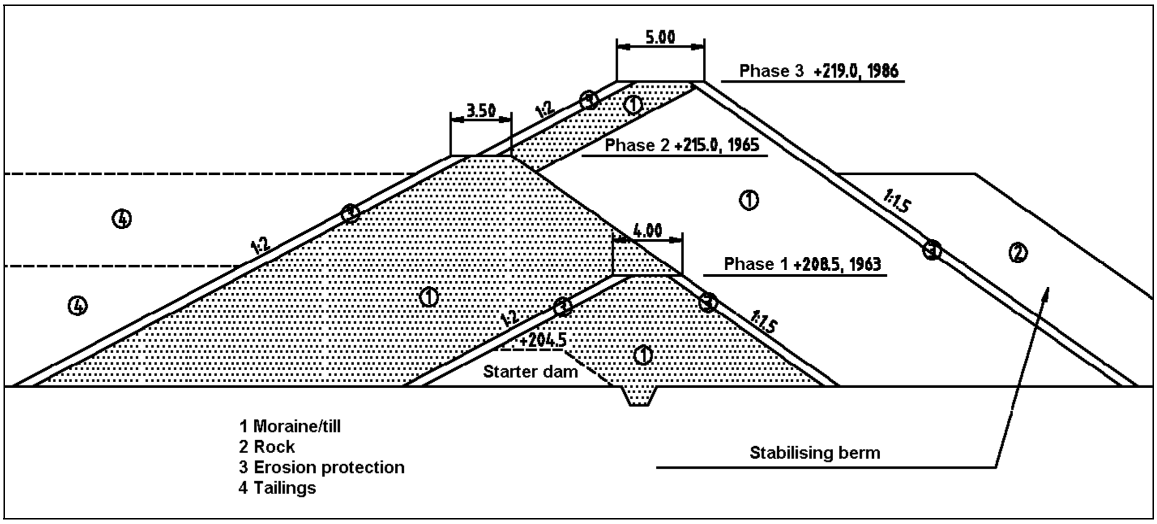


Figure 3.15: Cross-section of dam at Garpenberg before latest raise
[64, Base metals group, 2002]

The operator did some analysis on the potential of used the centreline method and found that this would result in:

- lower operating costs
- the use of less construction material and
- at the same time still fulfilling the stability requirements.

Hence an application to the authorities has been filed asking for a permit for raising the dam using the centreline method.

The water discharge from the pond in 2001 was 4.55 Mm^3. Of this 50 % was re-used in the mineral processing plant. The remaining 50 % was discharged to surface waters. The catchment area for the tailings pond is 1.56 km^2.
[64, Base metals group, 2002]

At **Hitura,** the tailings area, 110 ha in total, is divided into three ponds. The tailings (480000 tonnes in 2000) are discharged into the first pond. The two others are clarification ponds. The solids settle in the first pond and the clarified water is decanted via a tower and led to the next pond from the central part of the tailings pond. Clarified water is re-used in the mineral processing. Only the excess water is fed to the river system. The tailings pond is off-valley-site type. The starter dams are made of moraine. The tailings are distributed with spigots. The dams are raised every 12 to 15 months using tailings.

The dams of the clarification ponds are made of moraine and are lined with coarse gravel to prevent erosion. The distance from the mineral processing plant to the TMF is about 500 m. The distance from the tailings area to the nearest river is about 3 km.

Problems with seepage of water from the tailings pond into the groundwater exist. Groundwater and seepage water are pumped into the pond in order to control the groundwater flow and to minimise any impact.
The annual rainfall in Hitura is about 550 mm. The mean temperature during a year is 1 – 3 ºC The maximum temperature. in summer is 30 ºC and the minimum temperature in winter is – 35 ºC. During five months in a year the temperature is under zero and during six months above zero.

Before construction of the tailings management area, the soil was investigated, but apparently not carefully enough, as in one location infiltration to groundwater occurs. The affected groundwater is monitored in groundwater monitoring wells located downstream of the tailings pond and the back-pumped water is sampled.
[62, Himmi, 2002]

In the **Legnica-Glogow copper basin**, the mining of copper ore began in 1967. All of the tailings, which constitute 93 – 94 % of the extracted ore, have since been stored in tailings ponds, which have been raised using the upstream method. From 1968 until 1980, the first tailings pond of 600 ha, built upstream, was in operation and 93 million tonnes of tailings were stored there. This pond was decommissioned in 1980. It is assumed that this closure may be temporary and that in the future it may be brought back into operation again as spare capacity.

Since 1977 a new tailings pond of 1450 ha has been in operation. Similar to the previous pond it receives tailings from all three mineral processing plants. As all three mines are situated in an inhabited area and the distances between the mines is no further than 20 km, it was decided to find a topographically suitable area and convert it into a tailings pond which could serve all mines. An advantage of this set-up is that it takes into account the different characteristics of the tailings. For example, tailings from the Lubin and Rudna mines are coarse, whilst those from the Polkowice mine are fine, so it is possible to utilise coarse tailings for the dam construction and fine tailings for sealing the bottom of the tailings pond.

The tailings are transported into the tailings pond by pipeline, as a slurry of 14 – 20 % solids. The option to pump thickened tailings was considered in 2001, but the idea was rejected due to economic reasons, especially the capital cost of changing the existing system. The length of the current transport routes from the three mineral processing plants are between 6 and 9 km.

The total amount of tailings stored in the currently operated tailings ponds at the end of 2001 was 550 million tonnes.

Tailings are not utilised as backfill. The coarse fractions which technically match hydraulic backfill standards are required for dam construction. The fine tailings could only be utilised in paste form, which would currently be too expensive.

Some carbonate tailings (150000 t/yr) are used to neutralise diluted sulphuric acid from the copper smelters. The neutralisation process takes place at the Polkowice enrichment plant. The neutralisation product is mixed with the main stream of tailings.

The **previous tailings pond**, which was in operation from 1968 until 1980, was created by construction of an earthen dam across the valley of 600 ha. The characteristics of this dam were as follows: an earthen dam of in situ soils with a 15 cm-thick concrete screen on the internal slope with an inclination of 1:2; dam length 6760 m, max. height 22 m, and a triangular gravel filter drainage system connected with the dam ditch.

Decant water from the pond was collected by means of two decant towers with openings for water, and later transferred by a pipeline located in the gallery. Decant and seepage water were directed to flotation by means of a pump station located downstream of the dam. In the beginning, the tailings pond was filled by pouring the tailings from the dam crest by concrete canals located obliquely on the slopes. Later, tailings were placed directly from the outlets located on the dam crest every 40 m. At first the decant water level was up to 2 m above the tailings level. Even in this early period, some negative phenomena took place within the area downstream of the dam. These included a rising of the groundwater level, which even lead to flooding, and the creation of ground surface overflow zones. A front of water seeping from the tailings pond was created, in many sections below the bottom of the dam ditch, of increased mineralisation content. The water was later transferred into the ditches of the hydrography network of the Zielenica stream in the Oder river basin.

The subject area, prior to construction of the tailings pond, was of a deep groundwater level, on a significant longitudinal slope (11-16 ‰) and with high permeability of the subsoil such as sands.

A drainage system of open ditches that allowed for water outflow into the Zielenica stream and protection against flooding of the industrial zone, roads, railway line and main forest area was constructed to counter the threat. Close to the dam, drainage by a barrier well was made in order to collect polluted water and to lower the groundwater level. The system of tailings deposition then changed. Carbonate tailings from ore flotation by the Polkowice plant of the excess clay-silt fraction was directed close to the watershed side in order to seal the base of the pond. The system of tailings disposal was also changed by introducing outlets every 20 m. This allowed for stabilisation of the beach at a minimum distance of 100 m and the fraction segregation of tailings in this zone.

The measures listed above resulted in a limitation of water infiltration into the subsoil and effective water transfer from the area directly upstream of the dam. As a result of all the measures mentioned above, the water infiltration from the pond was reduced to a level similar to the conditions prior to the construction of the pond.

The consequences of this included losses in groundwater resources (removal of the groundwater intake, which had been situated there before), losses in forest resources (premature cut-out of an area of approximately 45 ha), extra costs for protection measures against pests in the weakened parts of the forest and extra costs of mineral fertilisation and liming. Also, water in the Zielenica stream had, within this section, a much increased general mineralisation to 3300 mg/l.

The tailings pond was located mainly in the Lubin mine area, and partially extended into that of the Polkowice and Rudna mines. In order to protect the dam, a protective pillar was created. The mine disposals could have been exploited by increased mine requirements and increased losses of disposals but there would have been additional requirements related to the exploitation of the tailings pond due to settlement of the area and the possibility of paraseismic vibrations caused by mining activity.

The above-mentioned limitations resulted in a decision to stop any further use of the pond, and to reject a proposed further development by a second stage to a volume of 160 million m^3.

The dam settlement, up to now, has reached a max. 3.25 m, while horizontal displacement had also been observed. Dense and loose zones were detected in the dam body. The deformations are monitored and analysed by the mine staff to meet the needs of the updated exploitation programme in the protective pillar of the dam. From this monitoring, it was decided that the observed deformations created no threat to dam safety.

Construction of the **current tailings pond** started in 1973. The location of the pond was chosen because it was outside the mining activity area, and thus, in contrast to the previous one, it was not subject to direct influence of the mine and, consequently, it did not limit mining operations. The second factor taken into account in selection of the pond location, was its proximity to the mineral processing plants.

The subsoil of the pond is formed by Quaternary deposits to a depth of 30 - 50 m below ground level. Locally, shallow tertiary deposits heavily disturbed by glacial activity are also observed.

To find the best way for filling the tailings pond, the characteristics of the tailings were taken into account. Sandstone tailings were transferred from the dam crest in sections 500 - 700 m long, situated every 20 m, in order to have a beach not smaller than 200 m and to allow for a gravitational segregation of tailings on the beach. More coarse material was deposited on the beach, while the majority of the fine material (0.05-0.002 mm) was transferred into the pond.

Fine carbonate tailings were, at the beginning, transferred by open canals along the natural slopes with the intention of creating a sealing of the bottom. Later, piers were made that transferred the tailings by pipeline to the edge of the pond.

As starter dams, conventional earthen dams were constructed along a 14.5 km perimeter. Since then, the dams have been raised using the coarse tailings stored on the beach. 2.5 m high dams have been made, from the coarse material, by the upstream method and by stages in two-year periods on the entire perimeter, with the pond increasing on average by 1.2 m per year.

The next stage of spigotting the tailings on the beach is carried out in layers not thicker than 25 - 30 cm per day, over several weeks. Usually, after a long break, the cycle of spigotting the tailings is repeated several times (4-7 times). The spigotting of tailings in one section usually takes approximately 15 weeks until the level of the dam is reached. For longer breaks, the surface of the beach is stabilised in order to protect it against wind erosion, by means of a bituminous emulsion water solution. The emulsion is sprayed from a helicopter. Later, the stabilised surface is removed by heavy equipment. This construction by stages allows proper drainage of the tailings and a stable phreatic surface within the dam body. In this section approximately 2/3 of the coarse particle tailings are stored. The longitudinal beach inclination varies from 6.5 ‰ close to the dam to approximately 4.0 ‰ at a distance of 100 m. The dam raises are carried out by bulldozers that also compact the tailings.

Density values in the upper layer are approximately 1.40-1.45 t/m^3, and increase with depth (to 10 m) to approximately 1.60-1.70 t/m^3. The water content varies between 5 – 20 %. The density of tailings is equal to 1.46 t/m^3. Based on piezometric measurements and CPTU soundings, it has been concluded that the pore pressure distribution is not hydrostatic, indicating tailings water seepage into the ground. This amount was assessed to be 0.862 m^3/min in 2000 and 0.690 m^3/min in 2001.

Circumferential drains in the tailings were installed along the majority of the tailings pond perimeter to enable control of the water level in the tailings and in the starter dams. The installation of drains is also foreseen on higher levels.

Values for the permeability coefficient ‘k’ in the beach area and in the pond are as follows:

- in the beach area: k is from 2.0 x 10^{-7} m/s to 2.0 x 10^{-9} m/s
- in the pond: k is from 5.0 x 10^{-8} to 1.0 x 10^{-10} m/s.

Surface water is protected against contamination by:

- intention to seal the bottom of the pond with fine fraction tailings which consolidate naturally
- collecting seepage water along the entire perimeter of the dam
- maintaining a barrier of wells along selected sections
- placing surface water intakes in selected flows at greater distances, and
- applying continuous monitoring of any underground and surface water under the influence of the tailings pond.

The monitoring network of ground and surface water includes over 800 monitoring points. [KGHM Polska Miedz, 2002 #113]

At **Neves Corvo,** the tailings are managed in a pond. The retaining dam is of the conventional type. The original dam core is clay. For the two raises, both using the downstream method, an HDPE liner was used to form the low-permeablity core. The dam has a slope of 1:1.8 (water/tailings side) and 1:1.7 (air side). Downstream of the core is a filter layer.

Due to the high acid-generating potential of the tailings, they are deposited subaqueously. The water cover is maintained at a minimum height of at least 1 m.

The option to apply thickened tailings for closure is currently being investigated.

At **Pyhäsalmi,** 16 % of the tailings are used in the backfilling of the mine, the remaining 84 % (180000 t/yr) are deposited in a tailings pond. This relatively low backfill percentage can be explained by the fact that only the coarse tailings are suitable for backfilling. The total area of the tailings management facilities is about 100 hectares, which includes three tailing ponds. Two of those ponds (pond B and D in the figure below) are used in parallel for settling the solids and to decant clarified water to the third pond (pond C in the figure below). The residence time of the tailing water in the area is about two months.

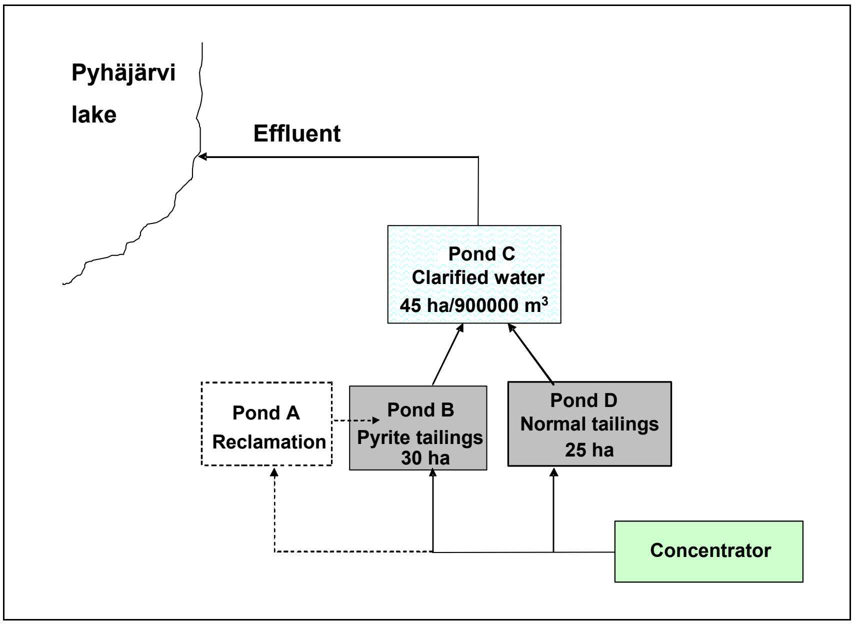


Figure 3.16: TMF set-up at Pyhäsalmi site
[62, Himmi, 2002]

Pond A in the figure above is completely filled and is not in use any more. Reclamation work for this pond was started in 2001. It will be covered with a 80 cm thick layer of soil material (30 cm clay and silt and 50 cm moraine). The central part of the pond will remain under water.

Before construction of the tailings area the soil had been studied. The soil was considered sufficiently impermeable (silt) to prevent leakage to groundwater and also stable enough to carry the load of tailings material. Base line studies were also performed on the downstream lake systems.

The tailings area is built paddock-style on a flat terrain. The base dam is made of moraine. The tailings are distributed from spigots around the first tailings pond and the clarified water is led forward from the centre part of the pond via a decant tower. The necessary raises of the tailings dams are done with tailings material. The dam of the clarification pond is constructed of moraine and lined with broken rock to prevent erosion. The area is surrounded by a ditch used to collect seepage water, which is pumped back to the tailings pond.

The distance from the mineral processing plant to the TMF is about 500 m and the distance to the nearest lake is 200 m.

The annual rainfall in Pyhäsalmi is approx. 650 mm. The climatic conditions are similar to the conditions and the Hitura site.

The tailings management area was designed in the early 1960's and no closure or after-care plans were taken into account in the design stage.

The operational routines include daily control of the facility, regular monitoring of the phreatic surface level in the dams, monitoring of discharged water and audits of the facilities.
[62, Himmi, 2002]

At **Tara** the tailings stream is cycloned. The coarse fraction (52 % of total tailings) is pumped down boreholes to the underground mine as a cement slurry (3 % cement) as backfill. The fine tailings are pumped to the surface tailings pond.
[101, Tara mines, 1999]

At **Zinkgruvan,** the mining method used requires backfilling. Up until 2001 hydraulic backfill had been used. This type of backfilling requires a drainage capacity of the tailings of at least 5 cm/h. This is why the coarse fraction had been extracted from the tailings using hydrocyclones whereby the fraction >50 µm was returned to the mine. In this way approximately 50 % of the tailings were backfilled using hydraulic backfill. The fine fraction of the tailings had been pumped to the Enemossen tailings pond.

A change in mining method to using 'panel stoping' requires paste backfill. This removes the requirement of the drainage capacity of the fill and thereby allows the use of the fine fraction of the tailings in the backfill. In this way, it is anticipated that up to 65 % of the tailings will be possible to backfill. Furthermore, the tailings pumped to the tailings pond will also contain the coarse fraction which will enable the use of the tailings in the construction of the dams. This method is now implemented in Zinkgruvan, so hydraulic backfilling is no longer performed.

Tailings that are not backfilled are pumped together with the process water from the mineral processing plant to the tailings pond, located 4 km south, in pipelines. The solids sediment in the tailings pond and the free water are led by gravity to a clarification pond 1 km from the tailings pond for additional clarification. In order to evenly fill up the tailings pond and to avoid dusting and oxidation of the tailings, the spigotting points are continuously moved along piers constructed of waste-rock. Water is re-circulated back to the mineral processing plant from the clarification pond (see water balance). Water is also discharged through a pipeline and a tunnel to the recipient water body. The tailings pond and the clarification pond are formed by natural basins (valley site type).

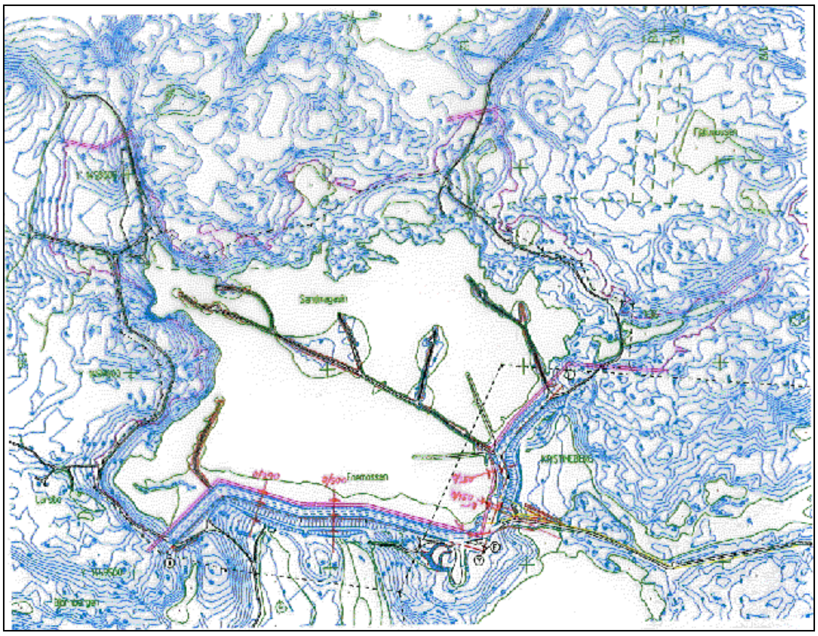


Figure 3.17: Top view of the Zinkgruvan TMF
[66, Base metals group, 2002]

The tailings pond is constructed in a valley and surrounded by natural slopes and two dams. The pond is founded on a peat bog and currently covers approximately 50 ha. At its final height it will cover approximately 60 ha. The embankments are of zoned construction, comprising erosion protection rock-fill on the upstream face, an inclined low permeability till core, a filter layer of sized screen rock and a downstream shoulder of rock-fill. The characteristics of the dams and the tailings pond is given in the table below.

Characteristic data	Dam X-Y	Dam E-F
Used capacity Dec. 2000	5.7	
Permitted capacity (from 1981) (Mm^3)	7.0	
Total tailings pond area (ha)	50	
Total clarification pond area (ha)	16	
Volume of material in dams (m^3)	380000	170000
Material from external borrow area	70000	30000
Dam height (m)	27	17
Crest length (m)	800	400
Crest width (m)	16	16
Dam upstream slope	1:1.5	1:1.5
Dam downstream slope	1:1.5	1:1.5
Width of stabilising berm (m)	7	7
Slope of downstream side of berm	1:1.5	1:1.5

Table 3.21: Characteristic data for the existing dams X-Y and E-F at Zinkgruvan site
[66, Base metals group, 2002]

To avoid dusting and oxidation sub-aqueous discharge is practised. However, to lower the phreatic surface, a 30 - 50 m beach with a height of 0.1 - 0.5 m above the water level close to the dam, is required. When discharging tailings under water, the angle of repose is significantly steeper than for discharge above the water level. In order to evenly fill up the pond the spigotting points are continuously moved along piers constructed into the pond. The beach is irrigated during the dry period of the year (spring-summer-autumn). During periods with no snow and during the winter dusting cannot be entirely avoided, even though several methods of temporary covering etc have been tried.

The decant system is tower-type. Decant water flows by gravity to the clarification pond. 50 % of the decant water is re-used in the mineral processing plant. An emergency outlet is constructed, which automatically discharges the water if the level increases above a certain level. The installed discharge capacity is 0.7 m^3/s (not counting the emergency outlet discharge capacity) which corresponds to the 100 year rain event and a maximum increase of the water level in the pond of 0.5 m.

The E-F and X-Y dams are constructed as conventional dams. The foundation of the dams is natural bedrock partly covered with moraine or peat soil. Excavations were done below the dams, down to natural bedrock or at least 4 m into the moraine, for the connection between the low permeable core of the dam and the underlying foundation. The low permeable core is constructed of compacted moraine from a borrow pit area. The permeability of the moraine is between 1 x 10^{-8} and 1 x 10^{-9} m/s. During the construction of the dams quality control was carried out continuously on the moraine and the filter material, mainly including compaction tests/control and material characterisation (grain size distribution).

Hydrogeological studies of the area show that the bedrock in the area contains several fracture zones. The fractures are permeable and drained which results in seepage from the pond. The water balance for the pond is given in the figure below.

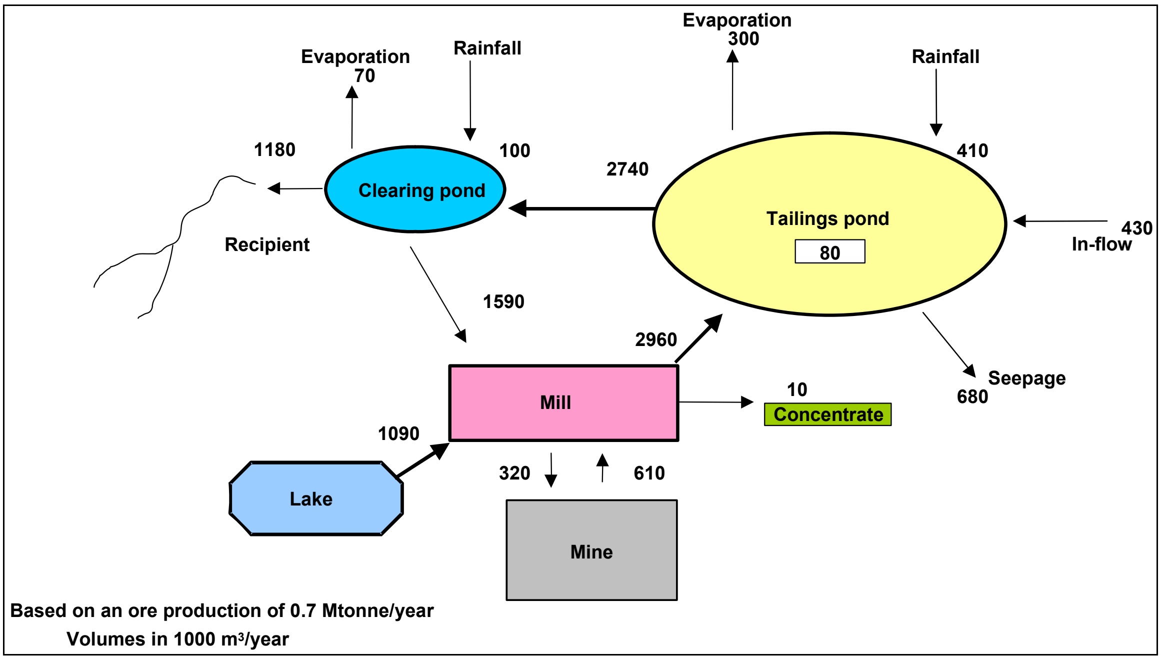


Figure 3.18: Water balance for the Zinkgruvan operation
[66, Base metals group, 2002]

The design of a new TMF at Lisheen

Probably the newest TMF in Europe was constructed recently at the **Lisheen** mine. This pond was constructed on flat land (paddock style) on a peat bog and is fully lined. Even though it is designed for a maximum amount of 10 million tonnes of tailings, it is only expected to contain a total of 6.6 million tonnes of tailings over the project's life [75, Minorco Lisheen/Ivernia West, 1995].

In the design phase of the Lisheen TMF all available primary methods of tailings management were discussed and evaluated. In the decision-making process that led to the preferred method of tailings management, the various methods were investigated on the basis the basic construction requirements and more detailed design criteria for the TMF. This process is described in the following.

Primary tailings management methods
Three primary methods of tailings management were investigated during the design phase, namely depositing them:

- into a surface water body such as a lake, river or sea
- into the mine as backfill
- into a surface tailings pond.

The first of these options was considered environmentally unacceptable. Although lake deposition, under managed conditions, has been accepted as best practice in several northern Canadian operations. However, in this case the operator adopted the philosophy that the most desirable tailings management strategy is to maximise the use of tailings as backfill in the underground workings. This was thought to have the advantages of:

- minimising the volumes of tailings to be managed on the surface
- supporting the hanging wall so that surface subsidence is minimised
- managing the tailings in an underground environment, that will be permanently under water after closure, hence oxidation will be avoided
- maximising the recovery of ore.

The layout of the mine and the sequence of mining make it possible to backfill 6.9 million tonnes tailings underground. The balance of 6.6 million must, therefore, be managed in a surface impoundment.
The topography at Lisheen, within a reasonable distance from the ore processing plant, is such that no significant valleys or hillsides were available as potential tailings pond sites, and, therefore, a ring-dyke impoundment (paddock-style) was proposed.

Other considerations
It had been identified that the tailings have the potential to generate acid if exposed to oxygen, and that the tailings pore water contains some metal ions. These two facts led to the decision that:

- a tailings pond/dam system to retain water so that the tailings are discarded and kept under water was needed
- the tailings need to be dealt with in a pond that is as impermeable as possible to minimise seepage into the groundwater system.

To satisfy these requirements a low, or very low, permeability liner with attenuating capability was considered necessary. The extensive bogs in the area contain peat which has a low permeability, making its use as a component of a composite liner very attractive. Peat has the added advantage in that it can attenuate the release of many of the likely contaminants in any seepage that may occur.

In order to identify the strength of the peat, its permeability in both the uncompressed and compressed states and its attenuation properties, a programme of tests was carried out.

Selection
It had been established that the maximum mass of tailings to be managed on the surface will be 10.0 million tonnes and the TMF should incorporate a low permeability barrier between the tailings and the local groundwater system. Using average topographical features and a reasonable thickness of tailings an area of 80 to 120 ha will be required. This area is based on the conservative in-situ dry density of 1.6 t/m^3, though subsequent design is based on 1.8 t/m^3, and a relatively low average height of approximately 10 m of tailings.

Since the tailings were found to be net acid generating it was decided that the containment facility must prevent oxidation of the pyrite and must be lined to restrict the seepage of water into the groundwater system. Two methods of achieving this were discussed, namely:

to provide a composite artificial liner if the site is on farm land or make use of the low permeability and high attenuation potential of compressed peat, as part of a composite liner, if the site is on a bog.

Methodology
The selection of the site for the TMF involved the assessment of the economic, environmental and engineering considerations. The objectives of the selection process were, thus, to minimise the impacts on the local community and the environment while, at the same time, satisfying the engineering requirements in the most economical way.

The site selection process involved four stages, namely:

1. a regional search for a topographical bowl or valley that would favour a tailings management scheme within a radius of 15 km of the ore processing plant site
2. a localised search to eliminate unsuitable areas within an 8 km radius. This radius was based on pumping considerations and the lack of good topographical sites in the area immediately beyond this radius
3. identification of possible locations
4. a detailed assessment of the possible locations.

[75, Minorco Lisheen/Ivernia West, 1995]

Description of the constructed TMF
The TMF was constructed on a bog which consisted of up to 4 m of peat overlying a glacial till on limestone bedrock. Limestone is a geotechnically competent lower carboniferous dolomitised Waulsortian formation with no major faulting, and a low palaeokarst potential. The site investigation found no open or infilled cavities and, for this reason and due to the minimal drawdown that takes place below the TMF, dewatering of the nearby mine does not cause reactivation of palaeokarst features, even if these are present.

The TMF consists of an earth embankment, which forms a dam around the impoundment area. Complete removal of peat from the embankment footprint was performed and the entire embankment is constructed on firm till or bedrock.

The perimeter of the TMF is a wide embankment consisting of zoned, engineered fill with a cross-section designed and built to act as a water retaining structure. The dams are constructed of compacted fill material from borrow pits with upstream and downstream slopes of 1:3 and 1:2 respectively. The dam crest is 6 m wide to provide access during construction and operation. A cross-sectional view of the dam is shown in the following figure.

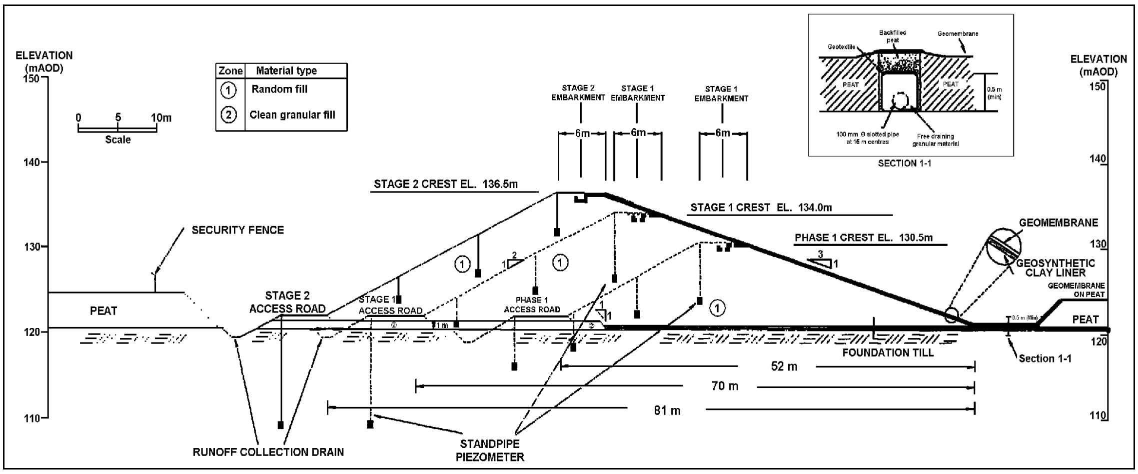


Figure 3.19: Cross-sectional view of dam at Lisheen TMF. Pond is to the right of the dam [75, Minorco Lisheen/Ivernia West, 1995]

The dams have been designed to a maximum height of 15.5 m above the till which lies beneath the bog. This allows for the possibility that additional capacity may be required, due to the discovery of additional ore reserves or reduction in the in-situ dry density of the tailings or change to the backfilling quantities. The dams are constructed initially to a maximum height of 9.5 m to provide for the 2.8 million tonnes of tailings which will be discarded on surface in the first six years of operations.

Most of the impoundment area will be underlain by the bog. Peat in the bog is generally of sufficient thickness and has the required physical and chemical characteristics to limit seepage and remove various metalliferous constituents from the seepage water. When loaded by the tailings, the peat will compress to become a natural liner with a permeability of less than 1 x $^{10-9}$ m/s. The permeability and strength of the peat are adequate to enable it to act with a geomembrane to form a composite liner capable of containing the tailings and its porewater. A small volume of seepage, estimated to be 34 m^3/day, could pass through the composite liner due to punctures in the geomembrane. It is likely that the majority of this water will be collected in the perimeter drains and pumped back into the impoundment.

Around the inner perimeter of the dams, in areas where the peat is less than 1.5 m thick, and on the embankments, a geosynthetic clay liner was placed below the geomembrane, to complete the containment system. A series of 100 mm diameter slotted drainage pipes were installed around the inner perimeter at the level of the base of the peat. These drains will extend from the start of the blanket drain beneath the embankment to 50 m inside the toe of the embankment and will collect some of the water that will be released during compression of the peat and also collect some of the seepage water.

At start-up, prior to deposition of any tailings, the impoundment was covered with water to a minimum depth of 1 m to provide cover over the tailings. Tailings were placed below the water surface by a floating distribution system which was moved slowly back and forth across the impoundment to produce a relatively even layer of tailings so as to minimise differential loading on the peat liner.

Tailings transport water is be returned to the ore processing plant for re-use, and any surplus water in the TMF is treated in the mine water treatment plant prior to discharge into the river system. Due to the net annual precipitation of approximately 450 mm, and the low volumes of seepage water, there is generally a surplus of water in the tailings impoundment.

The seepage and run-off water from the dams are collected in the surface drain around the TMF and pumped back into the impoundment.
[75, Minorco Lisheen/Ivernia West, 1995]

In short, for the design of the liner and the dams, the following factors were considered:

- stability
  - dam stability
  - foundation stability (in this case peat)
- seepage: seepage rates were calculated based on different defect scenarios
- seepage quality: it was concluded that the seepage water in general, meet drinking water standards, partly due to the fact that the peat has the ability to bind metal ions
- decant water and water balance
- tailings conveyance and discharge.

It was decided to discharge the tailings sub-aqueously to avoid oxidation of the sulphides. This will be achieved via floating pipelines (see figure below).

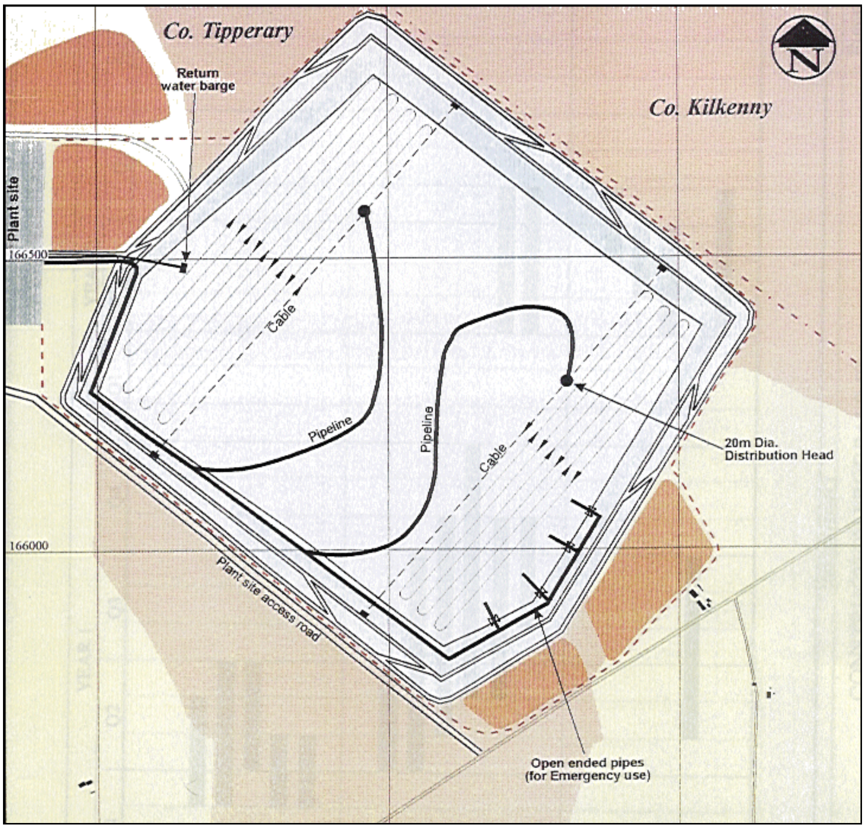


Figure 3.20: Tailings distribution system at Lisheen

The distribution heads at the end of each pipeline are connected to a reversible, electrically driven winch (see figure below) which passes over a main pulley.

Figure 3.21: Electrically driven winch controlling the tailings distribution pipeline at the Lisheen TMF

Lisheen uses an LLDPE (Linear Low Density Polyethylene) membrane as part of the liner system. The following programme was carried out during the installation of the liner:

- soil testing of embankment fill material
- destructive and non-destructive testing of LLDPE liner
- destructive and non-destructive testing of welds on the liner
- geosynthetic clay liner testing
- micro gravity survey for potential kaarst features
- liner leak location survey.

The field quality control documents for the TMF liner included:

- geosynthetics inventory control form
- geomembrane panel deployment log
- geomembrane trial seam log
- geomembrane seam log
- geomembrane seam pressure test log
- geomembrane seam vacuum (spark) test log
- geomembrane defect log
- geomembrane log
- geomembrane destructive test record
- geomembrane seam destructive sample log
- geosynthetic clay liner panel log
- geosynthetic clay liner accessory bentonite test record
- failed destructive sample tracking log.

[41, Stokes, 2002]

However, recent inspections have shown that several leaks and tears have developed in the synthetic liner membrane [76, Irish EPA, 2001]. These, where accessible, have been subsequently repaired.

The operation practises an 'open door policy', which includes:

- environmental information office in the community
- all monitoring data is made available in monthly and annual reports to the authorities
- complaints register
- annual schools project.

[41, Stokes, 2002]

3.1.2.3.3 Safety of the TMF and accident prevention

The tailings ponds at **Aitik**, **Boliden** and **Garpenberg** follow the routines for dam safety worked out within the OMS manual for tailings ponds (see Section 4.2.3.1). Furthermore, each site follows specific monitoring and surveillance routines. For example, at Garpenberg the pore pressure in the dams is monitored on a weekly or monthly basis in 13 piezometers installed in the dam (manual monitoring). Each measured value is compared to an alarm level at which a thorough follow-up investigation is conducted to detect why an abnormal value was obtained. At the discharge point an automatic water level indicator is installed which is coupled to the information system of the mineral processing plant. Every day, the dams are inspected by personnel from the mineral processing plant. The inspections include the slopes, the discharge from the polishing pond and the pipes for sand transportation [63, Base metals group, 2002], [64, Base metals group, 2002, 65, Base metals group, 2002].

At **Pyhäsalmi** and **Hitura**, the underlying soil was investigated before the dam construction commenced. The system has been designed and constructed so that the surface water in the tailings area can be kept in balance and the excess water from precipitation can be removed in a controlled manor, i.e. the ponds have been designed on a calculated water balance. Engineering and stability issues were addressed by external experts before the raising of all the dams at the Hitura site. No formal risk assessments were carried at either site.

The TMF area is controlled daily by the operators of the mineral processing plant and inspected annually by an independent expert and at five-year intervals by the dam safety authority The comments are recorded and stored in a "Dam Safety File", which is compulsory for all types of tailings management areas in Finland. The operational routines applied also include regular monitoring of the phreatic surface level in the dams, monitoring of discharged water and audits of the facilities. A documented emergency plan does not exist, but it is expected, that an emergency plan will be developed in the near future according to new legislation.
[62, Himmi, 2002]

The tailing pond of the operation in the **Legnica-Glogow copper basin** is operated by a separate division called the 'hydrotechnical plant division'. Staff working within the pond area have access to all-terrain vehicles, a hovercraft, a cutter and heavy equipment for earth works (excavators, bulldozers, loaders, tractors, crane). There is a system of communication (wire and wireless) and an alarm system, and the staff co-operate closely with the Mining Rescue Station.

The dam crest is illuminated constantly, since the roads on the dam crest and on the lower shelves of the dams are in continuous use.

The normal water volume in the pond is 5-6 million m^3. The reserve for periodic storage of excess water has a capacity of approximately 8 million m^3, while the additional reserve for rainwater is approximately 1 million m^3. The total available water volume in the pond is therefore 13-14 million m^3. The beach width is maintained at a minimum of 200 m and minimum the freeboard is 1.5 m.

The monitoring of the pond is carried out in co-operation with several external experts. Numerical systems for recording, transfer and storage of the monitoring data are also implemented. Results are analysed and conclusions are then drawn, usually within a year's time-scale.

Supervision is carried out by the designers. Additionally, scientific supervision for the safety of the hydraulic structures has been established. Supervision and consultancy is carried out by a team of independent experts (the IBE – International Board of Experts). The activity of the IBE, co-ordinated by the PGE – Polish Geotechnical Expert, is carried out based on the 'observational method' applied for the long-term development of the tailings pond.

In the period 1992-1999, the IBE prepared a geotechnical report on the safety and development possibilities of the currently operating pond. The report included complex subsoil investigations and determination of geotechnical properties of the tailings. The following design data were established: soil and tailings parameters, seepage conditions, slope stability conditions, and a monitoring programme. Numerous monitoring instruments were installed, stabilising berms were placed in selected sections, and circumferential drains were installed in the tailings.

Control parameter	Monitoring applied/ monitoring frequency
Control of water level in the pond Min. distance between the coast line and the dam crest – 200 m Control of phreatic line location in the tailings and in the dam body ▪ piezometric water level in the body of the starter dam and in the tailings ▪ piezometric water level in the body of the starter dam and in the tailings, in the vicinity of A, B and C pipelines ▪ water level in the vicinity of the circumferential strip drains in the tailings ▪ pore pressure in tertiary clays	Piezometer; measurements three times daily Distance marks + binocular with range-finder ▪ piezometer clusters: 7 cross-sections with continuous measurement and data transfer to the main station ▪ piezometer clusters: 7 cross-sections with manual measurements, every month or, for some piezometers, every 10 days ▪ 12 piezometer clusters in the tailings at the distance of 10 m upstream and 20 m downstream of the drainage axis ▪ piezometers
Discharge measurements of drainage: ▪ ditches ▪ circumferential strip drains in the tailings ▪ drainage of the starter dam ▪ barrier of wells downstream of the dam	▪ once per month ▪ twice per year ▪ twice per year ▪ three times per week
Dam movement	▪ bench marks, twice per year, ▪ inclinometers, once per month
Slope stability	▪ routine visual inspections ▪ exceptional inspections e.g. after heavy vibration and during heavy rain ▪ periodic inspection by a committee in charge of the technical state of the structure (once a month, twice a year) ▪ inspection by the competent authority ▪ system of linear transducers in the body of the starter dam, on the perimeter of the pond, on two levels with signal transfer to the main station
Properties of tailings and the subsoil (according to the programme established by the scientific supervisors and the designer)	Hyson equipment, CPT, CPTU DMT tests, Mostap sampler
Paraseismic activity induced by mining exploitation at the distance of min 800-900 m and max. over 2 km	Accelerometers in five cross-sections with transducers at the toe of the slope and on the crest of the dam and in 1 cross-section in the tailings.
Meteorological conditions within the pond area: rain, temperature, velocity and wind direction, humidity	Meteorological station

Table 3.22: Control parameters and applied monitoring at Legnica-Glogow copper basin [KGHM Polska Miedz, 2002 #113]

As the tailings pond has been classified as a high risk structure, appropriate emergency procedures and an emergency plan have been prepared for failure. The warning system and evacuation shelters for the local population are now under construction in co-operation with the local and state authorities.
[KGHM Polska Miedz, 2002 #113]

At **Zinkgruvan,** a risk classification of the tailings pond and the clarification pond was carried out according to the RIDAS system (Guidelines for dam safety developed by the hydro power industry, see Tabelle 4.2). According to this classification system the dams of the tailings pond (E-F and X-Y) are classified as type 1B and the dams of the clarification pond are classified as type 2.

This classification dictates what (minimum) safety measures and control programmes need to be followed. For the dams at Zinkgruvan some applicable measures are:

- audits of the class 1 dams at least every 3 years and the class 2 dams every 6 years
- class 1 dams need to be able to discharge the 100 year flow event as well as store a class 1 flow event. Class 2 dams need only to be able to discharge the 100 year flow event
- monitoring of class 1 and 2 dams needs to be carried out according to the table below.

Parameter	Consequence class 1B	Consequence class 2
Seepage	X, Continuously	Every 6 months
Movements of the dam crest	X, Every 6 months	(X, Annual)
Movements of the slopes	(X, Every 6 months)	(X)
Pore pressure in the core	(X, Annual)	(X)
Water level in support filling	(X, Every 6 months)	(X)
Water level in the foundation	X, Every 6 months	(X, Every 6 months)
X = measuring should be compulsory where it is feasible. () = measuring is important but can be excluded under some circumstances.		

Table 3.23: Basic measuring regime to be performed at new dams
[66, Base metals group, 2002]

The stability of the two dams have been assessed with the help of external experts. Results show safety factors of 1.5 and 1.6. Nonetheless, a dam safety improvement programme is running, comprising, among other things, installation of piezometer readings, flattening the dam slope from 1:1.5 towards a slope 1:2.5 – 1:3.0 and monitoring of the seepage water flow.

A number of incidents have occurred over the years mainly due to inner erosion of the dams. This has led to changed operating routines with regard to the deposition technique of the tailings in the dam. In order to lower the pore pressure and thereby avoiding any further inner erosion of the dams, a >30 m wide beach is maintained in the upstream side of the dams. The pore pressure level is monitored frequently (monthly, but more often if any abnormal levels are monitored) by installed piezometers in the dams.

A control programme for dam safety has been agreed with the competent authority and contains the following main components:

- yearly external audits of the tailings pond, dams and clarification pond. This inspection also includes pipelines for water and tailings, as well as discharge facilities
- weekly inspection of the dams by the environmental department at the site. At these inspections the dams are checked for possible damage, water levels, ice pressures and high precipitation events. Dam leakage flow is measured at the toe of the dams (stable around 5 - 10 l/s). All observations are registered in a logbook
- yearly environmental audits of the entire site that also include the tailings pond facilities
- yearly inspections by experts from the competent authority
- maintaining regular communication with the consultant who designed the dam.

Since 2001, piezometer readings have been included in the monitoring programme in order to register the hydraulic gradient over the dam. In total 21 manually monitored piezometers have been installed. In addition, three control wells have been constructed to better monitor and control seepage water flow and quality. The dam seepage flow collection and measurement facilities are shown in the figures below. Instrumentation for reading the electrical potential gradient in order to register water streaming through the embankment dams provides an additional method of monitoring the dam conditions.

Figure 3.22: Ditch for collection and flow measuring of seepage water alongside the dam [66, Base metals group, 2002]

Figure 3.23: Another ditch for collection and flow measuring of seepage water alongside the dam [66, Base metals group, 2002]

A dam safety manual is currently being prepared in order to cover all the issues connected to the tailings management. The manual will cover the following areas:

- dam safety organisation
- emergency and contingency plans
- risk assessment, environment impact and consequence classification
- design and construction
- hydrology and decant system
- systematic monitoring
- plans for closing the facility
- official permits and other documents of importance.

[66, Base metals group, 2002]

At **Lisheen** the following monitoring scheme is applied for this TMF:

Location	Parameter	Monitoring Frequency	Analysis Method/Technique
Piezometers in TMF Embankment	Water level pH Conductivity Pb, Zn, As, Fe, Cu, Hg, Co, Cr, Mg, Mn, Cd, Ni, CN, Sulphide & Sulphate	Weekly Weekly Weekly Monthly	Dip Meter Electrometric Electrometric Standard Method Note 1
Hydrostatic pressure cells on base of TMF	Hydrostatic pressure	Monthly	Agreed method (c.f. condition 7.4.12)
TMF Retaining Wall	Standard walk-over condition & stability checks Embankment Settlement/ movement Annual safety inspection report	Weekly Quarterly Annually	Visual Survey of seven fixed movement monitoring stations Agreed standard.
TMF embankment crest	Tailings distribution system	Twice daily	Visual
TMF	Tailings settlement/peat consolidation	Bi-annual	Agreed geophysical methodology. (c.f. Condition 7.4.11)
TMF	Volume of tailings disposed Tonnage of tailings disposed Used Capacity Remaining Capacity	Continuous Monthly Annual Annual	Flow meter Dry Density Agreed method Agreed method
Use of spigot distribution system	Period and volume/tonnage Efficiency of distribution	Continuous during use	Record Log Visual
Tailings distribution heads	Depth to tailings	Continuous	Agreed method (c.f. condition 7.4.13)
TMF Perimeter Drain (min. six N o selected locations).	Water level pH Conductivity Pb, Zn, As, Fe, Cu, Hg, Co, Cr, Mg, Mn, Ni, Cd, CN, SS, Sulphide and Sulphate,	Weekly Weekly Weekly Monthly	Dip meter/gauge Electrometric Electrometric Standard Method Note 1
TMF perimeter groundwater monitoring wells (Inner and outer rings)	Water level pH Conductivity Pb, Zn, As, Fe, Cu, Hg, Co, Cr, Cd, Mg, Mn, Ni, CN, Cl, PO4, Cr, NO_2, NO_3, Na, DS Sulphide & Sulphate,	Monthly Monthly Monthly Monthly	Dip meter/gauge Electrometric Electrometric Standard Method Note 1

Table 3.24: Example of monitoring scheme of TMF
[41, Stokes, 2002]

Annex 2 provides several examples of dam failures, mainly at base metals operations.

3.1.2.3.4 Closure and after-care

The decommissioning plan for **Aitik** focuses on the three main parts of the operations, i.e. the waste-rock areas, the tailings pond and the industrial area, which includes the open pit. With regard to the tailings, the evaluation of the weathering properties are still going on. The results so far indicate that no wet cover is required. The measures planned are therefore limited to fertilising and sowing with herbs, grass and trees to prevent wind erosion of the top layer. Dams around the tailings deposit and the clarification pond will be re-sloped at an angle of 1:3 and the slopes will be sown with grass.
[63, Base metals group, 2002]

At **Aznalcollar** after the accident the emergency programme evolved into a complete decommissioning of the failed dam and the entire pond. This included:

- diversion of the nearby river
- building an impermeable seepage cut-off wall around the north and east sides of the dam
- installation of a hydraulic barrier including a back-pumping system on the inside of the cut-off wall
- cutting and re-sloping the dam to 3:1 and covering it
- remodelling the tailings surface to minimise the infiltration and to control the surface run-off
- construction of a vegetated composite cover over the remodelled tailings surface. Starting from the tailings, the cover consists of a geo-textile layer, 0.5 m waste-rock, 0.1 m blinding layer, 0.5 m compacted clay, 0.5 m protective soil layer and vegetation.

[68, Eriksson, 2000]

The decommissioning plan for the **Boliden** tailings pond is described in Section 3.1.6.3.4.

At **Garpenberg**, according to hydro-geological modelling results, the higher section of the Ryllshyttan tailings pond will be almost completely saturated with groundwater. Limited areas along the west and south dams will have a partly unsaturated top-soil.

According to the decommissioning plan, the tailings pond will be covered with vegetation. With numerous references from other sites, it is anticipated that seeding directly on the tailings surface with the addition of nutrients will be a cost efficient and realistic alternative. If problems occur measures to reinforce vegetation, such as application of an organic cover or similar, will be taken. The areas along the dams that remain unsaturated will be covered if acid conditions develop. The dams, which potentially contain acid producing material, will be covered using a 1.1 m thick engineered soil cover, containing a 0.4 m compacted clay layer as the sealing element. The dams will be re-sloped to 1:2.5 – 1:3.0 before covering and then revegetated. The lower section of the tailings pond (the part that is now active) is situated is such a way that a positive water balance can be guaranteed, to that it will thus remain covered by water.

For several years, contacts have been maintained with a nearby paper mill, regarding possible use of their organic waste products for reclamation purposes. These contacts resulted in a test programme, which was launched after the upper section of the pond was completed in 2000. The paper mill produces organic sludge and a fly ash product, a combination with properties making the material suitable as cover material. The supply of material is sufficient to cover the entire pond area within 5 – 10 years, and provides the potential for a robust and environmentally friendly technical solution.
[64, Base metals group, 2002]

A draft plan for closure and after-care has been developed at **Hitura**, which has not yet been approved by the authorities [62, Himmi, 2002].

At **Lisheen** the closure plans were developed as part of the initial permitting procedures and will be reviewed annually. It is expected that five years active care and ten years passive care will be necessary. For the TMF a permanent water cover, due to the acid generating potential of the tailings, is believed to be the best solution. Erosion protection of the dams will by achieved by vegetation and, if necessary, by a rock cover [75, Minorco Lisheen/Ivernia West, 1995].

A closure funding of about EUR 14 million (incl. perpetual after-care)has been in place with the authorities since construction commenced (i.e.IRL 11 million).
[41, Stokes, 2002]

At **Pyhäsalmi**, the closure plan for the first filled tailings pond (pond A) has been worked out and presented to environmental authorities, but it is not yet officially approved. The closure costs are estimated to be about EUR 1 million for this pond. No detailed plans for the other ponds exist, but the total closure and after-care costs for the Pyhäsalmi tailings area are estimated at EUR 5.4 million. The costs are reviewed every year. The EUR 5.4 million needed for closure have been reserved in the income statement of the company to cover the closure and after-care costs. This money, however, has not been deposited. So, for economical difficulties of the company, no assurance mechanism exists.

Production is planned to continue for at least an additional 15 years. Hence, it will be possible to gather experiences for long-term behaviour of the material and the dams at pond A. This experience will be utilised for planning the closure of the other dams in the future.

How the tailings management area will be monitored in the future, i.e. after the closure, is not yet determined. The main target of the after-care work will be to prevent ARD generation from the tailings (5 – 10 % sulphur) and to avoid the need for collection and treatment of drainage water for an indefinite period of time.

At pond A, the tailings will be covered with 80 cm of soil. The lower layer will be clay and silt material (about 30 cm thick) and the upper layer will be made of moraine. The thickness of the cover was determined taking into account site-specific design criteria and the locally available materials. Other cover materials were also considered, such as peat, sand etc., but the final choice was made based on economical and technical reasons again taking into account the locally available materials. The central part of pond A will remain water covered. A system to control the level of the water surface has to be constructed and will include a decant tower and a culvert. Finally, the surface of the treated area will be covered with suitable vegetation.
[62, Himmi, 2002]

The existing and the indicated ore reserves are estimated to give **Zinkgruvan** a mine life for at least another 15 years of operation. Plans for rehabilitation of the areas affected by the mining operation are designed according to the present status of the rehabilitation technique. Since the technology and the requirements from authorities are changing continuously this closure plan can be considered a model, developed from today's demands and standards.

The rehabilitation of the previous tailings area started in 1982 with the construction of an 18 hole golf course and was finalised in 1991 when a marina, a beach area and residences were arranged in the centre of the area. A monitoring programme for the recipient of water from the golf course area, is now running.

Until the currently operating facilities are decommissioned, the closure plan will be reviewed at least every five years.

The current tailings impoundment is planned to be dewatered and covered. Once the area has been restored and rehabilitated the land will be handed back to the original owners. At that stage it may be used for the same purposes as pre-mining i.e. forestry.

The time schedule for the rehabilitation work depends on the life of the mine and will consequently not be started until the mining operation has ceased, now estimated to be around 2025. Depending on the choice made as to how to extend the tailings impoundment area, which is currently estimated to reach permitted volumes around 2007, the need for rehabilitation of the existing tailings impoundment may occur earlier. If the authorities demand a new tailings impoundment to be constructed then rehabilitation of the existing facilities will be performed.

In the application for a new permit an extension of the existing tailings impoundment is the primary alternative. This facility can technically, by means of raising the dam, handle tailings quantities corresponding to another 25 years of ore production. A dam raise to a height corresponding to the life of the mine will imply that rehabilitation measures are not applied before mine closure. An exception to this is the downstream walls of the dams that may be rehabilitated before final restoration.

A ‘wet’ cover is not possible at the existing pond as the catchment area is too small to guarantee a permanent water surface covering the area. Hence, a ‘dry’ till cover must be arranged in order to reduce infiltration and diffusion and to prevent water and oxygen reaching the tailings.

When the pond has been dewatered the dams will no longer be subject to water pressure. Instead the dam walls can be classified as stable earth-formations with groundwater pressure. From this point on, the dams cannot be flooded and will not be subjected to inner erosion, which are normally the two most common reasons for dam failure. During times of high water flows it is important though, that water is prevented from entering the pond.

Measures will be taken to secure the physical and chemical stability of the dams and the tailings managed within the pond. Long-term stability and access for large equipment can be achieved by flattening the dams slope from the current 1:1.5 to 1:2.5 - 1:3. The major part of the material needed to flatten the slopes will be put in place simultaneously with the continuous raising of the dams.

The slopes and the surface of the pond will be vegetated to withstand erosion and to aesthetically blend into the surroundings.

The final rehabilitation of the tailings impoundment can be summarised as follows:

- excavation of by-pass ditches along the surrounding natural slopes, approximately 2000 m
- dewatering and consolidation of the pond
- contouring of the pond surface
- flattening of the downstream dam slopes
- placing of dust control cover
- placing of the final cover
- revegetation of the cover.

The table below gives the planned cover design. This proposal is based on recommendations from the authorities, international practice and experiences from other rehabilitation projects in similar settings. The design of the cover may change over time, since closure is far into the future. The suggestion below has been chosen in order to fulfil its purpose, with a good margin. It has been assumed that the following materials will be used to form a cover from top to bottom:

0.2 m	Top soil
0.5 m	Protective cover of moraine
0.2 m	Drainage layer of moraine
0.2 m	Dealing cover of material with low permeability
0.2 m	Dust control layer of crushed rock or sand and gravel
-	Tailings

Table 3.25: Structure for cover of Zinkgruvan TMF
[66, Base metals group, 2002]

The water surface of the clearing pond will be lowered to a level that can be maintained by natural precipitation within the catchment area. At this level minor areas with tailings will be exposed, mainly in the upper (south) part of the pond. In these areas it is thought to be sufficient to use a simplified type of the cover compared to the cover used at the tailings impoundment. It is assumed that this simplified cover may consist of 0.2 m of topsoil and another 0.2 m of moraine.
[66, Base metals group, 2002]

3.1.2.4 Waste-rock management

At all sites, where the ore is mined underground, the relatively small amounts of waste-rock from development works remain underground.

3.1.2.4.1 Characteristics of waste-rock

The **Aitik** waste-rock has been subjected to extensive testing such as material characterisation, field-scale transport modelling, hydro-geological tracer tests, mineralogy and geology. The suite of tests performed include:

- whole rock analysis
- mineralogical investigations
- Acid-Base Accounting (ABA)
- kinetic testing, such as batch, column, humidity cell and, large scale column weathering tests
- tracer tests to determine the water flow paths within the waste-rock
- effective surface area determinations.

Field characterisation includes:

- in-situ measurements of oxygen concentration as a function of depth within the heaps
- temperature profiles within the heaps
- field-scale tracer tests
- determination of effective diffusion coefficient
- water flow and quality measurements
- water balances.

All this characterisation work has been used in various scientific exercises and in the waste-rock management planning of the Aitik site. Activities performed are, e.g., predictive modelling of water quality evolution with time, equilibrium and kinetic modelling of pore water and drainage composition, mass-balance calculations, coupled hydro-geological and transport modelling. Due to the extensive test work done, it has even been possible to use the information from Aitik to try to solve one of the biggest scientific challenges within this area - namely the dependency between laboratory tests and actual field conditions.

From these results, it can be concluded that, at Aitik, two types of waste-rock are generated – about 65 % that will not generate ARD and 35 % which have the potential of producing ARD. It is an very small percentage that will actually produce ARD, however it is not feasible to separate this fraction from rock that may produce ARD.

These results led to the decision to try to separately deposit the waste-rock that does not produce ARD and thereby minimise the surface area on which ARD-producing waste-rock is deposited. Since 1999 Aitik mine has used a new waste-rock dump for selective deposition of sulphide free waste-rock. This dump is named ‘the environmental waste-rock dump’. The results have also been used in order to develop an adequate decommissioning plan for the waste-rock dumps.

The environmental waste-rock is frequently tested and has to have less than 0.1 % S and 0.03 % Cu with a NP/AP ratio exceeding 3 to be accepted for use outside the mining area and for deposition in the deposit for 'the environmental waste-rock dump'. Tests conducted by different laboratories have shown that the waste-rock quality is usable as ballast material for roads and railways as well as for use in asphalt.
[63, Base metals group, 2002]

Within the **Boliden** area (five operating mines) waste-rock is managed based on detailed characterisation, mainly focusing on weathering characteristics. ARD producing waste-rock is preferably used directly as backfill. For open pit mining, ARD generating waste-rock is separately deposited and at the Maurliden mine, the ARD generating material is temporarily stored in deposits and will be backfilled into the mined out open pit at closure when it will also be permanently covered by water.
[65, Base metals group, 2002]

The waste-rock at **Mina Reocín** is mainly dolomite (limestone). At the initial stage of the open pit mining, clay (marl) and topsoil were also generated and stored separately for future use during the decommissioning phase.
[54, IGME, 2002]

At **Zinkgruvan**, the mineralogical composition of the waste-rock is given in the table below (based on microscopic analysis). The waste-rock consists of mainly quartz and feldspar (>70 %) and may contain traces of sulphide minerals. The ratio of carbonates to sulphur is >10, therefore the waste-rock has a high buffering capacity and will, therefore, not produce ARD. The waste-rock is regularly sampled and analysed for Zn and Pb content, which, over a large number of samples, have been found to be 0.3 % and 0.2 % respectively. The density of crushed waste-rock is 1.75 t/m^3, whilst the compact density of the rock varies between 2.6 and 2.7 t/ m^3.
[66, Base metals group, 2002]

Mineral	Fraction %	Mineral	Fraction %
Quartz	32.8	Epidote	0.4
Plagioclase	1.0	Zoizit	3.1
Mikrocline	27.3	Calcite	2.5
Biotite	4.3	Titanit	0.3
Muscovite	1.6	Zircon	0.3
Hornblende	11.7	Apatite	0.1
Diopside	9.9	Other	0.5
Garnet	4.2	**Total:**	**100 %**

Table 3.26: Waste-rock mineralogy at Zinkgruvan
[66, Base metals group, 2002]

3.1.2.4.2 Applied management methods

The waste-rock deposits at **Aitik** are situated east and west of the mine and cover an area of approximately 400 ha. In 2001, 26 million tonnes of waste-rock were extracted from the mine, of which 67 % were separately deposited due to their low sulphur and metal content.

Today's strategy is to avoid expanding the stockpile area containing sulphidic waste-rock. In 1999, a new waste-rock dump area was opened. This dump is designated for non-sulphidic waste-rock exclusively, to allow for less extensive decommissioning procedures according to the permit. Furthermore, the quality of the rock opens opportunities for its utilisation as a construction material.

The selective management of waste-rock has been identified as a potential for cost savings and possible revenue if low sulphur material can be isolated. The bedrock from the hanging wall has a lower sulphide content and is therefore more suitable for selective management than rock from other parts of the mining area. The material consists of amphibole-biotite gneiss, which is intruded by pegmatite veins. The amphibole-biotite gneiss is characterised by a varying degree of amphibole banding, with a matrix of amphibole, biotite, quartz and to a lesser extent plagioclase. The pegmatites contain mostly feldspar and quartz. The thrustfault forms a sharp contact between the hanging wall and the ore zone, making it easy to follow the contact. It is known that the hanging wall is barren of copper, and earlier mapping from diamond drill holes shows no change in the bedrock. The analyses carried out show low copper and sulphur content.

A new test procedure to secure the quality of the waste-rock was developed. This included chemical analyses, acid base accounting (ABA-test) and humidity cell tests on drill core material on the future waste-rock. This work led to further investigations. Drill chip samples from the production drilling were collected and tested for several different blasts, with positive results. Today, routines are implemented for testing this type of bedrock for every blast, aiming at rapidly classifying the material for deposition on the new waste-rock dump. The material should be amphibole-biotite gneiss and/or pegmatite. Copper grades, sulphur content and the ABA-test are not to exceed the recommended values. All results are stored in databases.

In the latest waste-rock deposition plan of 1999, conditions for the selective management of various waste-rock fractions are regulated. The criteria for the selective deposition of sulphide free waste-rock are less than 0.1 % S, less than 0.03 % Cu and a NP/AP ratio exceeding 3. Analyses are conducted on accumulated samples from a minimum of eight drillholes representing 150000 t of waste-rock. To secure the quality, any waste-rock within 30 m from the ore zone needs to be excluded.

The decommissioning method involves covering the sulphide free waste-rock dump with 0.3 m of till and/or other material as a vegetative layer. The decommissioning is undertaken progressively, and the establishment of vegetation will start within two years after deposition of each terrace is completed.

Surface run-off and drainage water in collection ditches is collected and re-used in the mineral processing plant as process water. Collection ditches receiving effluents from old sections of the waste-rock dumps currently receive drainage water with a high metal content and low pH. The quality of the water in the diversion ditches is strongly influenced by the local quaternary geology, with elevated sulphide contents in the till.

The hydrogeological investigations showed that the dumps are not hydraulically connected with the pit. The whole area, on which the dumps are located, is covered with a 10 m layer of low permeable glacial till on top of the bedrock. Virtually all the infiltrated water leaves the dumps at the toe, and is easily collected in ditches. Acid drainage with an elevated content of copper was found during the 1970's. Detailed field investigations in 1992 – 1993 estimated the annual amount of copper leaving the dumps to be 80 tonnes, of which 55 tonnes originated from the old marginal ore stockpile. The corresponding overall amount of sulphate was 4000 tonnes annually. During recent years, the bulk part of the marginal ore has been reprocessed and the influence on the pollution load of this undertaking is presently being evaluated.

A critical component of the decommissioning plan was the development of measures addressing the ARD situation. An engineered cover was identified as the only realistic way to deal with the waste-rock dumps, and between 1993 and 1996, a project using modelling tools to design a cover to reduce the flux of water and oxygen into the waste-rock was undertaken. The goal was to achieve a 99 % reduction in oxygen flux into the dump. The hydraulic properties of potential cover materials were measured and a number of cover designs involving layers of moraine and tailings sand were investigated. Following the modelling programme, a cover design was selected for the waste-rock dumps. Physical tests of the glacial till in the area, i.e. the stockpiles and the overburden that has been or will be removed in the future, indicated that this material would be suitable for engineering a cover suitable as a gas diffusion barrier of relevant quality.

A number of possible cover alternatives were evaluated. The results indicated that a 1 m layer of compacted moraine with a hydraulic conductivity of 1.5×10^{-7} m/s would reduce the oxygen transport into the dump to 1.2×10^{-9} kg O_2/m^2s - less than 1 % of the reference case without cover. From this result, the estimation, based on weathering tests, was made that the reduction in copper pollution load would be of the same order of magnitude, resulting in a copper release of less than 1000 kg/yr.

Snow reduces the frost penetration. An estimation of the influence of freezing, which could possibly affect the long-term performance of the cover, was that frost would penetrate the cover to a depth of 0.7 m. The penetration of frost is strongly dependent on the depth of the snow cover, which at Aitik is considerable during winter.

To enhance the establishment of vegetation and to further secure the structure’s resistance to frost penetration, it was concluded, that an additional top layer of 0.3 m of non-compacted till should be applied. An illustration of the decommissioned waste-rock dump and the proposed cover is shown in the figure below.

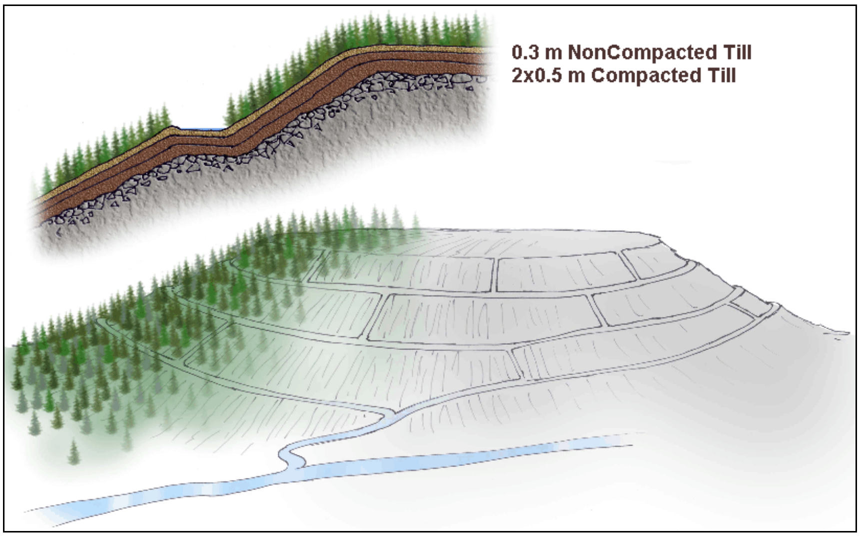


Figure 3.24: Structure of waste-rock dump cover and illustration of the decommissioned waste-rock dump at the Aitik site
[63, Base metals group, 2002]

The 1997 permit allowed Aitik to commence the cover placement in 1997, with a 14 hectares area of the east waste-rock dump. This cover consisted of 1 m of moraine, distributed in two 0.5 m layers, which were compacted individually, and 0.2 - 0.3 m of topsoil. According to the permit, the maximum hydraulic conductivity was 2×10^{-7} m/s, and the compaction rate was 93 % proctor. The surface was finally sown with grass during the autumn of the same year.

To divert surface run-off water, channels were constructed along the benches and down the slopes, using geotextile and till. It soon became obvious, that a different solution regarding the surface water needed to be developed, as erosion from snowmelt water severely damaged the cover. Replacement using new till and erosion resistant waste-rock was an immediate solution, but for future cover steps, surface water management solutions must be designed in a way that does not endanger the integrity of the cover.

The placement of the cover on the slopes, on the other hand, did not constitute any problem. The 1:3 slope is shallow enough to allow normal operation of the conventional construction machinery.

In the coming years, additional sections of the waste-rock dumps will be covered in order to reduce the exposure of the waste-rock to oxidising conditions and to minimise the material handling and costs. Therefore for future mine developments cover placements will be synchronised with the overburden removal.

Since 1999, the Aitik mine has used a new waste-rock dump for selective deposition of sulphide free waste-rock. This dump has so far received 40 million tonnes of waste-rock. It is frequently tested to verify that the permitted values, less than 0.1 % S and 0.03 % Cu and a NP/AP ratio exceeding 3, are met. Tests conducted by different laboratories on the chip value, brittleness, ball mill hardness and particle density have furthermore shown that the waste-rock quality is sufficient for it to be used as ballast material for roads and railways as well as for use in asphalt. [63, Base metals group, 2002]

In the **Boliden** underground mines, large quantities of waste-rock are moved directly to mined out areas within the mine. Only the waste-rock that is not used for backfilling is brought to the surface. In open pit mining all waste-rock has to be brought up to the surface and deposited. At closure, some of the waste-rock, e.g. highly acid generating rock, may be backfilled into the mined out open pit.

During 2001, the following amounts of waste-rock were used for backfill and were deposited within the Boliden mining area.

Mine	Waste-rock used in backfill (kt)	Waste-rock deposited (kt)
Renström	82.1	-104.0
Petiknäs	103.4	-15.7
Kristineberg	127.6	4.6
Maurliden		875.7
Åkerberg	24.3	-21.0

Table 3.27: Amounts of waste-rock backfilled and deposited in the Boliden area

Waste-rock from deposits at the Petiknäs and Åkerberg mines has been backfilled (hence negative values). The waste-rock dumps at the Renström mine have decreased significantly as material from the dumps has been used in the construction of a regional public road.

Generally it can be concluded that the managed waste-rock quantities are relatively limited, with the exception of the Maurliden open pit mine.

The waste-rock is managed based on a detailed characterisation, mainly focusing on weathering characteristics. ARD producing waste-rock is preferably used directly as backfill. For open pit mining, ARD generating waste-rock is separately deposited and for the Maurliden mine, the ARD generating material is temporarily stored in deposits and will be backfilled into the open pit upon closure, where it will then be permanently covered by water. All waste-rock deposits are surrounded by diversion ditches and drainage collection ditches. If required, the drainage can be treated before discharge.

Topsoil and moraine are deposited separately for future use in the decommissioning of the site. [65, Base metals group, 2002]

The Lubin, Polkowice-Sieroszowice and Rudna mines in the **Legnica-Glogow copper basin** produce two types of waste-rock. The first type of waste-rock is generated during the development of the underground mines. Due to the different shape of the deposit in each mine, the amount of the waste-rock varies. Annually, the Lubin mine produces about 450000 t and the Rudna mine about 600000 t. The Polkowice-Sieroszowice mine produces ten times more (6000000 t.), because its deposit is the thinnest (0.4 - 3.5 m) and in many places it is necessary to extract waste-rock and ore at the same time and separate them on site. All waste-rock is utilised as solid backfill in the mined out stopes or for underground road construction.

The other stream of waste-rock which occurs periodically comes from the construction of shafts (e.g. in 2001, 61500 t of waste-rock was extracted for the construction of a shaft at Rudna mine). This material is stored on heaps, which are shaped and reclaimed.
[KGHM Polska Miedz, 2002 #113]

At **Mina Reocín**, the waste-rock is deposited into an mined out part of the open pit. The old waste-rock dumps generated in the initial phase of the open pit mining are covered with soil and re-vegetated. Restoration is done using clay (marl) and top soil separately stored for this purpose [63, Base metals group, 2002].

At **Zinkgruvan**, about 0.2 million tonnes of waste-rock are produced annually in preparation works. At the end of the mine life, ore production will be possible for a couple of years without any waste-rock generation. The waste-rock is used for construction of the tailings dam, as backfill in the mine and is also sold outside the mine. About 0.5 million tonnes of waste-rock is managed on the surface close to the old open pit as a noise barrier around the east part of the industrial area. Any surplus of waste-rock is managed in deposits that are managed by an external entrepreneur who crushes and sells the material to third parties. From 1996 until 2000 58 % of the waste-rock was sold.
[66, Base metals group, 2002]

3.1.2.5 Current emissions and consumption levels

3.1.2.5.1 Management of water and reagents

Water consumption
The following table shows the water consumption and percentages of re-used process water of base metal sites.

Site	Ore processed (tonnes/year)	Water consumption (m^3/tonne)	Re-used in min. proc. plant (%)	Of which from TMF (%)	Of which from mine (%)
Aitik	17700000	1.8	100	100	0
Almagrera	1000000	3.2	0	0	0
Boliden area	1450000	3.2	0	0	0
Garpenberg	984000	2.9	68	100*	0*
Hitura	518331	6.2	100	90	10
Mina Reocín	1100000	2.0	100	0	100
Neves Corvo	1750	0.8	75	100	
Pyhäsalmi	1250000	5.3	0	0	0
Zinkgruvan	850000	2.7	63	73	27
*: mine water first pumped to TMF					

Table 3.28: Water consumption and water use/re-use of base metal sites

Note that at the Pyhäsalmi and Boliden sites water is partially re-used within the mineral processing plant.

The **Aitik** mineral processing plant uses 100 % re-used water from the tailings pond. Under normal conditions the entire water consumption, 31.5 Mm^3/yr, is supplied by re-used water from the tailings pond. Approximately 1.8 m^3 of water per tonne of ore processed is used in the process plant. In the snow smelt period, excess water is normally released from the clarification pond to the recipient. The released water is of good quality and no water treatment is required (see Section 3.1.2.5.3).
[63, Base metals group, 2002]

From the **Garpenberg** mine the mine water is pumped to the mineral processing plant and used as process water before it is pumped together with the tailings to the tailing pond system where water treatment occurs through interaction with the fresh mineral surfaces which effectively

absorb any dissolved metals. From **Garpenberg Norra** the mine water is released to the recipient after clarification. At the Garpenberg mineral processing plant the consumption of used/re-used water was 1.95 Mm^3 during year 2001 and the consumption of freshwater during the same period was 0.93 Mm^3. The discharge from the tailings pond amounted to 4.55 Mm^3. Out of this volume approximately 50 % was re-circulated to the mineral processing plant and re-used as process water. The remaining 50 % was discharged to a lake.
[64, Base metals group, 2002]

At **Hitura**, the clarified water from the TMF is re-circulated to the process. The amount of this water corresponds to almost 100 % of the total amount of water used in the process. This system does not save reagents significantly, because flotation chemicals such as xanthate and frothers are decomposed in the tailings area and the tailings material consumes the sulphuric acid. The water balance is presented in the figure below.

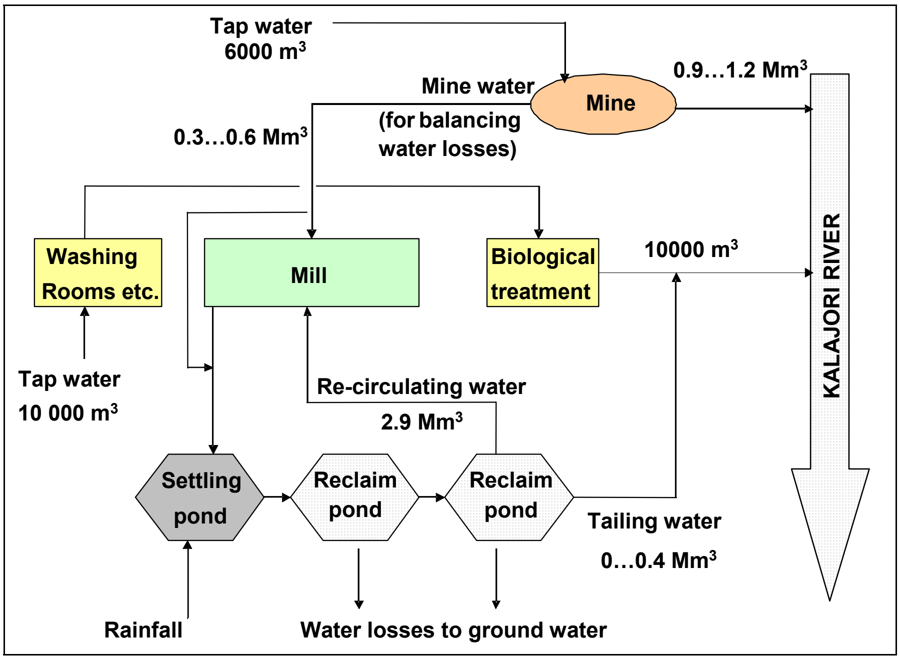


Figure 3.25: Water balance at Hitura
[62, Himmi, 2002]

It can be seen that, depending on rainfall the amount of water from the tailings pond used/re-used in the mill (mineral processing plant) varies between 88 to 100 % (0 to 0.4 Mm^3 to the river).

The mines in the **Legnica-Glogow copper basin** pump a total of about 70000 m^3 per day of mine water. The CL^--content of this water ranges from 0.5 to 127 g/l and the SO_4^{2-}-content is approx. 2 g/l. However, the actual amount of water pumped to the surface is higher, and its salinity is lower, because of additional water streams from backfilling and flush boring. All these waters combined are utilised in the mineral processing plant.
[KGHM Polska Miedz, 2002 #113]

At **Lisheen,** process water is re-used and supplemented with water reclaimed from the TMF [73, Ivernia West,].

At **Pyhäsalmi**, there is no re-use for process water from the TMF area to the process. The reason being gypsum ($CaSO_4$) in the water causing blocking problems in the pipes. There is only an internal re-use of water in the process, where water from the thickener in the pyrite flotation is returned to the grinding circuit to save sulphuric acid in the pyrite flotation and lime in the Cu-flotation. This amount of water is corresponding to 10 % of the total amount needed in the mineral processing plant.

Fresh water is pumped from a lake. The water balance for 2001 is presented in the table below.

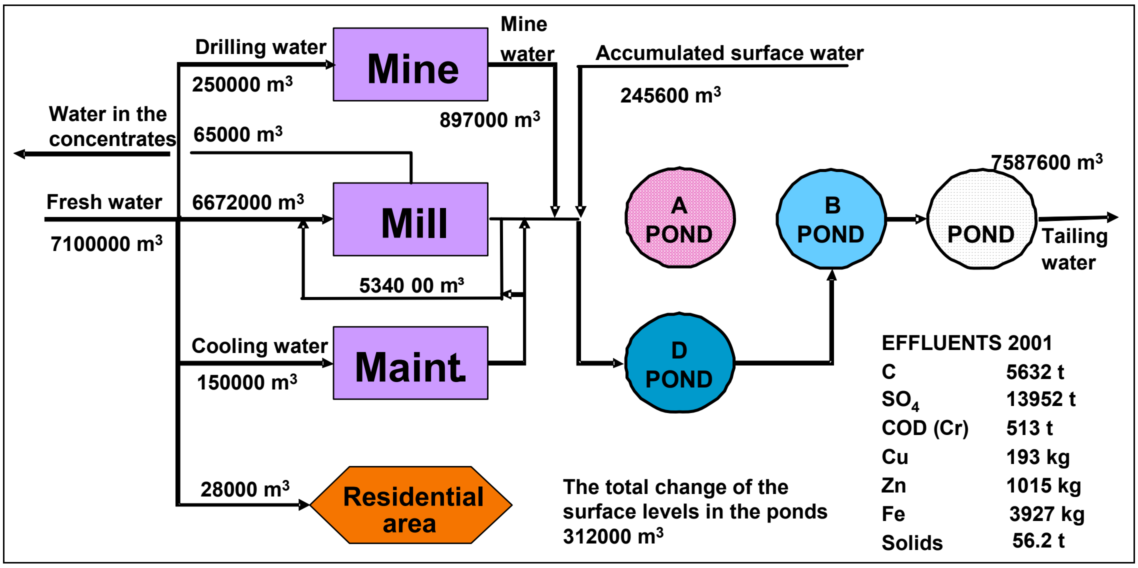


Figure 3.26: Water balance at Pyhäsalmi for the year 2001
[62, Himmi, 2002]

At **Zinkgruvan** the water consumption in the mineral processing plant is approximately 2.7 m^3/tonne or 2.4 Mm^3/yr in total. The water requirement is covered by freshwater supply from nearby lakes and by recycling of water from the tailings pond (partly process water and partly mine water).

The main consumption of water is in the actual process, in the paste fill and for cooling purposes. The entire water balance is illustrated in the following figure.

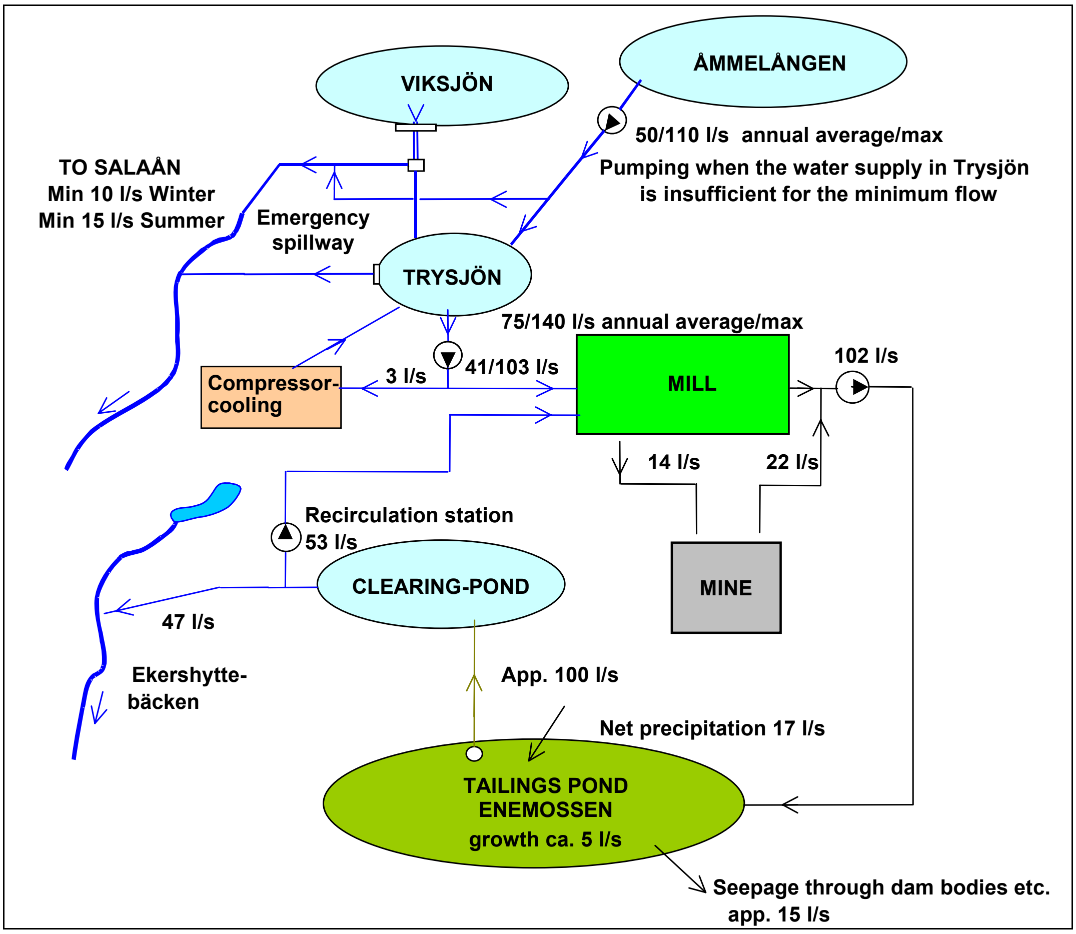


Figure 3.27: Water balance for the Zinkgruvan operations shown as average annual flows and maximum flow during operation
[66, Base metals group, 2002]

Reagent consumption

The following tables shows the reagents used at base metal mineral processing plants. Note that cyanide can be used for two purposes, as a depressant for sphalerite, pyrite and some copper sulphides or as a leachate for gold.

Reagent		Site								
		Aitik	Almagrera	Mina Reocín	Boliden	Garpenberg	Hitura	Lisheen	Pyhäsalmi	Zinkgruvan
		Consumption	Consumption	Consumption	Consumption	Consumption	Consumption	Consumption	Consumption	Consumption
Group:	**Type:**	g/t	g/t	g/t	g/t	g/t	g/t	g/t	g/t	g/t
Collectors					179[1]					
	Xanthates					209	300	135	250	100 - 120
	Thionocarbamate							10.9		
Frothers					28					
	Sylvapine						150		50	
	MIBC							8.8		30 - 40
	Dowfroth							0.9		
Activator										
	Copper sulphate				441	433		876	500	
	Sodium sulphide									
	Sodium hydrosulphide									
Depressants					90					
	Sodium cyanide				310[2]				4	
	Zinc sulphate				92	306		234	400	30 - 50
	Iron sulphate					47				
	Acetic acid								15	
	Sodium chromate				30	10				
	Dithiophosphate							55.1		
pH										
	Lime	408			3448	773	350	4368[5]	9000	
	Sulphuric acid						7500	5609[6]	12000[3]	300 - 500
	Sodium hydroxide				30					400 - 600
	Nitric acid								150	
	Hydrochloric acid				1					
Flocculants										
	CMC						100			
	Other							13.5	1	
Others										
	Soda ash							472		
	"Flotation agents"	19								
	Sulphur dioxide				869[4]					

1. No information about collector type, probably xanthates; 2. Used in cyanide gold leaching; 3. Based on 100 % H_2SO_4
4. For CN destruction after cyanidation; 5. pH and water treatment; 6. pH and to leach

Table 3.29: Consumption of reagents of base metal sites

As an alternative to xanthates as collector there are a number of different brands on the market. These collectors are of the type diaryldithiophosphates. A change into those collectors means for Zinkgruvan a change of the flotation process into a straight selective lead/zinc flotation process. The overall costs for chemicals in that process is twice the costs compared to the actual process used today. This is due to the fact that a set of other chemicals will be used i.e. copper sulphate, sulphur dioxide and slaked lime [66, Base metals group, 2002].

The **copper separation at Neves Corvo** is achieved by flotation. The following collectors are used:

- dithiophosphate, 80 – 120 g/t, pH 10-11
- potassium amyl xanthate (PAX), 30 – 40 g/t, pH 11

The tin separation is carried out providing by gravity separation on Holman-Wilfley shaking tables and subsequently by cassiterite flotation.

3.1.2.5.2 Emissions to air

The emissions to air for the **Boliden** site are discussed in the precious metals section.

The **Aitik** site follows a comprehensive monitoring programme for emissions to air. At the site there are mainly three sources of emissions to air:

- from the drying of the concentrates
- from blasting and diesel vehicles, and
- diffuse dusting from the whole site including the tailings pond.

However, emissions from blasting, diesel vehicles and the drying of concentrates are not part of the scope of this document. It should be noted, though, that drying ovens are gradually being replaced by filters.

The diffuse dust immissions are measured at eight monitoring points at the site as sedimented particles. The collected samples are analysed for copper and the total weight of sedimented particles (normalised towards the surface area of the collector). The results are summarised for the years 1999 to 2001 in the table below.
[63, Base metals group, 2002]

	1999		2000		2001	
Monitoring point	Sedimented particles	Cu	Sedimented particles	Cu	Sedimented particles	Cu
	mg/m^2month	mg/m^2month	mg/m^2month	mg/m^2month	mg/m^2month	mg/m^2month
S 1	1210	1.5	1910	2.5	3030	2.6
S 7	450	0.4	330	0.3	480	0.4
S 8	394420	21.4	55550	19.8	23440	12.7
S 9	1100	0.7	720	0.3	2610	1.0
S 10	920	0.9	750	0.7	540	0.5
S 11	690	0.7	1200	0.8	480	0.5
S 12	1820	0.8	1360	0.8	1000	0.9
S 13	520	0.3	860	0.5	780	0.4

Table 3.30: Measurements of total sedimented particles and Cu at Aitik
[63, Base metals group, 2002]

At **Garpenberg,** there are mainly two sources of emissions to air:

- the drying of concentrates and
- ventilation from the mines (SO_2, NO_2 and CO_2).

[64, Base metals group, 2002].

At **Hitura,** the main sources of emissions to air have been identified as:

- dust from the industrial area including TMF and mineral processing plant
- dust from roads.

The area of influence is monitored at several collecting points.

Dust from the TMF is a problem in dry and windy weather. Attempts have been made to prevent dusting by covering the banks immediately after raising with soil material and using lime slurry on the banks. Also the water surface level in the tailings pond is kept as high as possible in the summer time and tailings distribution is arranged so that the beach area is kept as wet as possible.
[62, Himmi, 2002]

In the **Legnica-Glogow copper basin** there are three types of airborne emissions:

- dust, heavy metals, SO_2 and NO_2 emissions from the ventilation shafts of the underground mines
- dust, heavy metals, SO_2 and CS_2 emission from the three mineral processing plants
- dust emissions from the dry surface portion of the tailings pond.

As to the latter type of emissions, it is the beach, which constitutes a considerable source of dust emissions, especially on windy days. To reduce this dust a water 'curtain' is installed on the crest of the dam. Additionally, to stabilise the surface in sections which are temporarily dry, an asphalt emulsion is sprayed from a helicopter. Currently, additional water 'curtains' are being tested. These are installed inside the pond on the beach at a distance of 150 m, and are put into operation when a dry section, after removing the asphalt cover, is utilised for dam construction.

In the vicinity of the tailings pond, an air monitoring system has been installed. This consists of three continuous measurement stations, one meteorological and one central station. The measurement stations are equipped with FAG airborne dust measurement devices, which measure particulate matter (total). There is also one more station, owned and operated by the local inspection authority. The result for total particulate matter imission are shown in the following table.
[KGHM Polska Miedz, 2002 #113]

Measuring point (distance from the dam)	**Annual medium particulate matter (total, µg/m^3).**			
	Year 1998	Year 1999	Year 2000	Year 2001
Rudna (1000 m SE)	36.3	34.3	29.2	33.6
Kalinówka (600 m NE)	33.9	29.1	28.7	30.2
Tarnówek (500 m SW)	35.7	34.0	31.3	23.9
Local authority's station (1800 m SE)	24.3	18.0	14.8	12.7

Table 3.31: Dust immissions from tailings pond in the Legnica-Glogow copper basin
[KGHM Polska Miedz, 2002 #113]

Furthermore, annual medium concentrations of particulate matter (total) and metals content in ambient air within close proximity (60-2250 m) of the tailings pond are measured. The results for 2001 are shown in the following table.

	Particulate matter (total)	Metal				
		Cu	Pb	Zn	Cd	As
	(µg/m^3)	(µg/m^3)	(µg/m^3)	(µg/m^3)	(µg/m^3)	(µg/m^3)
D_{24}[1]	1.0 - 70.0	<0.01 - 0.07	0.05 - 0.26	0.001 - 1.321	0.0001 - 0.0226	0.0001 - 0.0515
$D_a m$[2]	12.7	0.019	0.099	0.151	0.0007	0.0038
1. the range of 24-hour measurement results 2. medium annual value						

Table 3.32: Annual medium concentrations of particulate matter (total) and metals content in ambient air within close proximity (60-2250 m) of the tailings pond in the Legnica-Glogow copper basin
[KGHM Polska Miedz, 2002 #113]

At **Lisheen,** the emissions to the atmosphere are monitored using the following measurements:

- point source
- ambient air
- dust deposition.

[41, Stokes, 2002]

The emissions in 2001 are listed in the following table.

Parameter	Unit	Quantity
Particulates	kg/yr	3375
Nitrogen oxides	kg/yr	243266
Carbon monoxide	kg/yr	129546
Carbon dioxide	kg/yr	186713872

Table 3.33: Emissions to air at the Lisheen site
[76, Irish EPA, 2001]

At **Pyhäsalmi,** the main sources of emissions to air have been identified as:

- dust and SO_2 from concentrate drying in the mineral processing plant
- dust from the TMF
- dust from concentrates loading area
- dust from roads and industrial area.

Dust emissions are measured at several collecting points. The main purpose is to survey the area of influence. Since June 2001 emissions have also been controlled with an automatic device, which continuously takes measurements.

Dust emissions from the tailings management area are a problem in dry and windy weather. Attempts have been made to prevent this by spraying lime slurry on the banks.
[62, Himmi, 2002]

3.1.2.5.3 Emissions to water

The following table summarises the total emissions to water from base metals sites.

		Site						
		Aitik	Boliden	Garpenberg	Hitura	Legnica-Glogow	Lisheen	Pyhäsalmi
Parameter	Unit	Year						
		2001	2001	2001	2000	2001	2001	2000
Discharge	Mm^3	6.44	11.10	2.60	0.08	21.1	22.9	6.89
Ca	t/yr	-	-	-	-	26164	-	4727
SO_4	t/yr	-	-	-	254	58742	-	12057
COD	t/yr	-	-	-	-	654	51.4	334
Solids	t/yr	-	-	6.2	0.9	633	89.4	47.1
Al	kg/yr	446.0	-	-	-	-	2465	-
As	kg/yr	1.7[1]	156	18	-	422	-	-
Cd	kg/yr	-	1	0.8	-	591	8.1	7
Co	kg/yr	5.3	-	-	-	-	17	-
Cr	kg/yr	0.2[1]	-	25	-	1160	-	-
Cu	kg/yr	36.0	72	40	-	1435	28.5	309
Fe	kg/yr	-	-	-	24	9495	1412	9141
Mn	kg/yr	-	-	-	-	-	565	-
Hg	kg/yr	0.1	-	0.3	-	6.33	0.6	-
Ni	kg/yr	5.1[1)]	-	-	107	-	311.9	-
Pb	kg/yr	0.1	191	52	-	3376	263	-
Zn	kg/yr	34.6	1070	586	-	949	2321	1464
N	t/yr	17.0	-	6.5[2]	-	130	40892	-
CL						176269	-	-
1. Dissolved metals, sample is filtered in the field before it is acidified 2. Year 2000								

Table 3.34: Total emissions per year to water from base metals sites

The annual total discharge from **Zinkgruvan** was 1.5 Mm^3.

Table 3.35 shows the concentrations in the emissions from tailings management facilities.

		Site			
		Aitik	Garpenberg	Legnica-Glogow	Zinkgruvan
Parameter	Unit			Year	
		2001	2001	2001	2001
pH		7.1	10	7.9	7.5
Susp. particles	mg/l	-	2.4	30	3.1
Mineral oil	mg/l	-	0.1		-
Copper (dissolved)	µg/l	2.1	-		-
Copper (total)	µg/l	7.3	15	68	2.7
Zinc	µg/l	1.7	218	45 (total)	220
Lead	µg/l	0.02	20	160 (total)	27.3
Cadmium	µg/l	0.004	0.37	28 (total)	0.3
Arsenic	µg/l	0.3		20 (total)	1.9
Chromium	µg/l	0.004	9	55 (total)	<1.0
Mercury	µg/l	0.009	-	0.3 (total)	<0.1
Iron	µg/l	8	-	450 (total)	-
Aluminium	µg/l	38.5	-		-
N-total	mg/l	2.6	-	6.16 (total)	5.4

Table 3.35: Concentrations in emissions from base metals sites

At **Aitik,** water sampling is carried out at the discharge point (clarification pond) and at 12 sampling stations in the river systems according to the regular monitoring programme. The samples are analysed for several metals, pH, N-total, oil, SO_4-S, conductivity and turbidity. Water was, during the year 2001, only discharged from the clarification pond to the Leipojoki river. No discharge was done from the recycle pond nor from the recycle channel [63, Base metals group, 2002].

The emissions to water from the **Boliden** tailings pond are described in detail in the precious metals section.

Garpenberg follows a broad monitoring programme for surface waters as well as recipient sampling and control, which is carried out within an integrated programme for the catchment area (the main river in the area). This programme contains water sampling analysis, fish investigations, sediment and bottom fauna investigations. The discharge from the tailings pond is sampled by an automatic sampler every two hours and a composite sample is produced monthly.

Sufficient water quality for the process and for discharge is obtained in the tailings pond/clarification pond system. The main contaminants are Zn and N predominantly from the mine water. The mine water contains approximately 4.5 mg/l Zn and up to 50 mg/l of total N. Major reductions in the discharge of Zn to the environment have been obtained by pumping the mine water together with the tailings slurry to the tailings pond, whereby the Zn adsorbs to the mineral surfaces. Laboratory test work has shown that the method effectively reduces the Zn concentration in the mine water from 4.5 mg/l to less than 0.2 mg/l in 40 min. N compounds are partially degraded in the tailings and clarification ponds. In 1998, it was estimated that about 10 tonnes of N was added to the system from the mine water.
[64, Base metals group, 2002].

At **Hitura,** emissions from the TMF to groundwater have been reported. Exact figures are not available. The flow of groundwater has been cut and the contaminated water is back-pumped and led to the river [62, Himmi, 2002].

In the **Legnica-Glogow copper basin** tailings pond to keep the balance of water and its salinity in the circuit, an average of 60000 m^3/d of clarified water, containing 16-20 g/l total suspended solids, must leave the system. The discharged water is pumped to the Oder River by a pipeline of 17 km. The amount of the water is controlled to correspond to the current river flow, so that the sum of chlorides and sulphates in the Oder does not exceed 500 mg/l. To eliminate a local higher concentration of total suspended solids in the river, the discharging system distributes the discharged water at the bottom, across the whole cross-section of the river.

The concentration of suspended solids in water leaving the pond varies, depending on its current volume in the pond and weather conditions. As the suspended solids contain heavy metals, a water treatment plant is temporarily put into operation to clean the discharged water to the level of <50 mg/l.

The purification technology is based on coagulation (with about 300 mg/l ferric chloride) supported with polyelectrolyte praestol (1 mg/dm^3) and sedimentation in a lamella settling tank. Table 3.34 and Table 3.35. show total emission to water and concentrations in the emissions from the tailings management facilities.
[KGHM Polska Miedz, 2002 #113]

At **Lisheen** arsenic is treated with ferric sulphate if the concentration in the discharge is above 0.0048 mg/l. Thereby, the arsenic is precipitated as a meta-stable ferric arsenate compound. Similarly if cyanide is added in the process as a suppressant and the concentrations in the discharge approach 0.048 mg/l, the CN will be destroyed [75, Minorco Lisheen/Ivernia West, 1995].

At **Zinkgruvan,** the tailings and tailings pond system constitutes a very good treatment facility for the process and mine water due to its high adsorbtion capacity. By fully utilising the characteristics of the system and passing all mine and process waters through the system significant reductions in Zn discharge have been achieved over the last 15 year period as illustrated in the figure below.

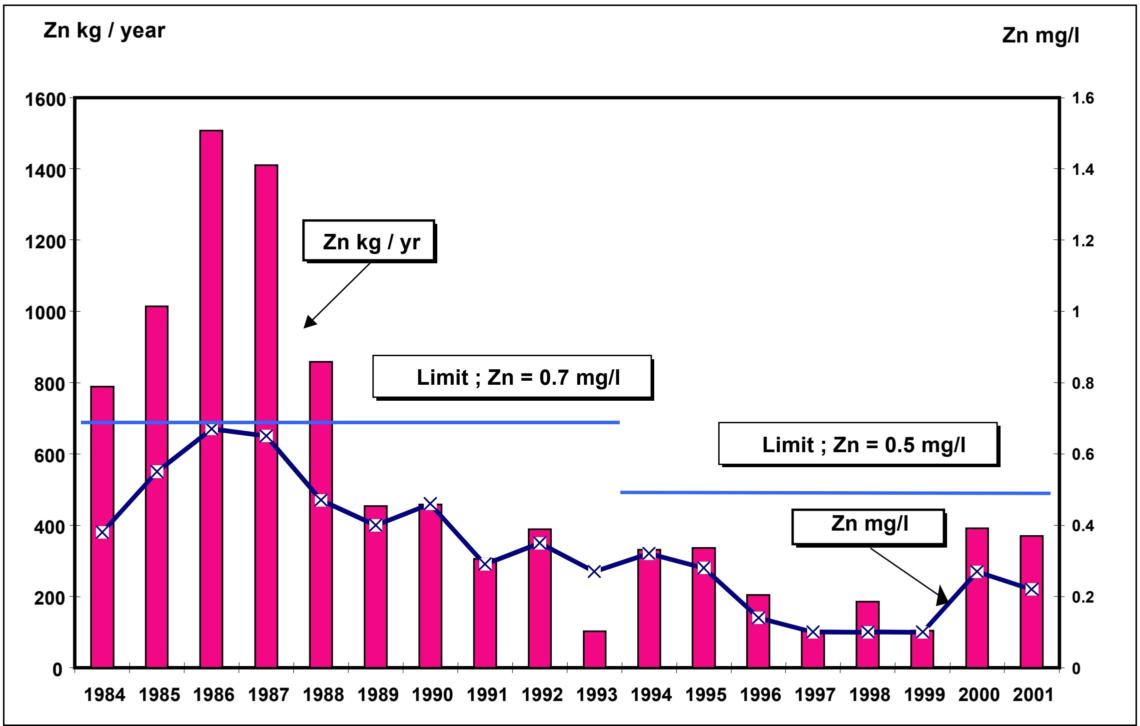


Figure 3.28: Annual average zinc concentration (in mg/l) in excess water from the clearing pond to the recipient and calculated transport (kg/yr) 1984 - 2000
[66, Base metals group, 2002]

3.1.2.5.4 Soil contamination

In an area of about 400 m around the TMF, soil contamination was discovered at **Hitura**. At **Pyhäsalmi,** soil contamination in the close environment of the plant has been observed. This was caused by sulphur (pyrite) dusting. No significant contents of heavy metals or chemicals have been reported in the soil.
[62, Himmi, 2002]

Every year soil contamination is monitored in 54 points located within close proximity (50-2000 m) of the **Legnica-Glogow copper basin** tailings pond. The results obtained from 1996-2001 indicate that a higher concentration of copper in the soil is found only in the closest proximity to the dam. The concentrations of the other metals are at background level.
[KGHM Polska Miedz, 2002 #113]

3.1.2.5.5 Energy consumption

The following table summarises the energy consumption of base metal sites.

Energy consumption	Units	Site						
		Aitik	Boliden	Garpenberg	Hitura	Neves Corvo	Pyhäsalmi	Lisheen
Mine	kWh/t[1]	n/d	n/d	n/d	n/d	21.44	n/d	n/d
Mineral processing plant, total	kWh/t[1]	n/d	n/d	n/d	32.8	36.95	34.9	47.3
	GWh[1]	n/d	n/d	n/d	n/d		n/d	53.4
Grinding	kWh/t[1]	11 - 12	22	n/d	n/d	24.93	n/d	20.6
Dewatering	kWh/t[1]	n/d		n/d	0.22	1.28	3.9	n/d
TMF	kWh/t[1]	2	2	3	1	1.97	1.6	n/d
Waste-rock management	kWh/t[1]	n/d	n/d	n/d	n/d	n/d	n/d	n/d
Total electrical	kWh/t[1]	22.1	n/d	n/d	n/d	n/d	n/d	n/d
Total all energies	GWh	545.5	214.6	123.5	n/d	n/d	n/d	n/d
	kWh/t	30.7	148	126	n/d		n/d	n/d
Ore processed	Million tonnes	17.77	1.45	0.98	0.52	1.75	1.25	1.15
1. Electrical energy								

Table 3.36: Energy consumption at base metal sites

3.1.3 Chromium

This section contains information about the Kemi chromium mine in Finland. All information taken from [71, Himmi, 2002].

3.1.3.1 Mineralogy and mining techniques

Chromite forms in deep ultramafic magmas and is one of the first minerals to crystallise. It is because of this fact that chromite is found in some concentrated ore bodies. As magma slowly cools inside the earth's crust, chromite crystals form and, because of their density, settle at the bottom of the magma and are concentrated there.

The chromium ores at **Kemi** are associated with a mafic-ultramafic layered intrusion within the contact between migmatite granite and schist. The formation starts at the town of Kemi and extends approximately 15 km NE, with a maximum width of 1500 m. The compact chromite-rich horizon appears 50 - 200 m above the bottom of the formation. The thickness of the continuous chromite horizon varies from a few millimetres to a couple of metres, but in the Nuottijärvi-Elijärvi area, the chromite layer contains eight layers, which are economically viable over a distance of 4.5 km. Both host rock is serpentinite and talc-carbonate rock. Idiomorfic chromite is the only ore mineral appearing in economic quantities. The average content of the ore is 26 % Cr_2O_3 and the Cr/Fe ratio is 1.55.

The **Kemi** chromium mine is an open pit mine with a waste-rock to ore ratio of 5.5:1. The mine production in 1999 was approx. 250000 tonnes.

3.1.3.2 Mineral processing

At Kemi, the ore from the mine contains 11 % iron and 25.5 % Cr_2O_3. After the mineral processing the concentrate contains between 35 % Cr_2O_3 in the coarse fraction (lumps) and 44 % of Cr_2O_3 in the fines.
The flow sheet of the Kemi site is given below:

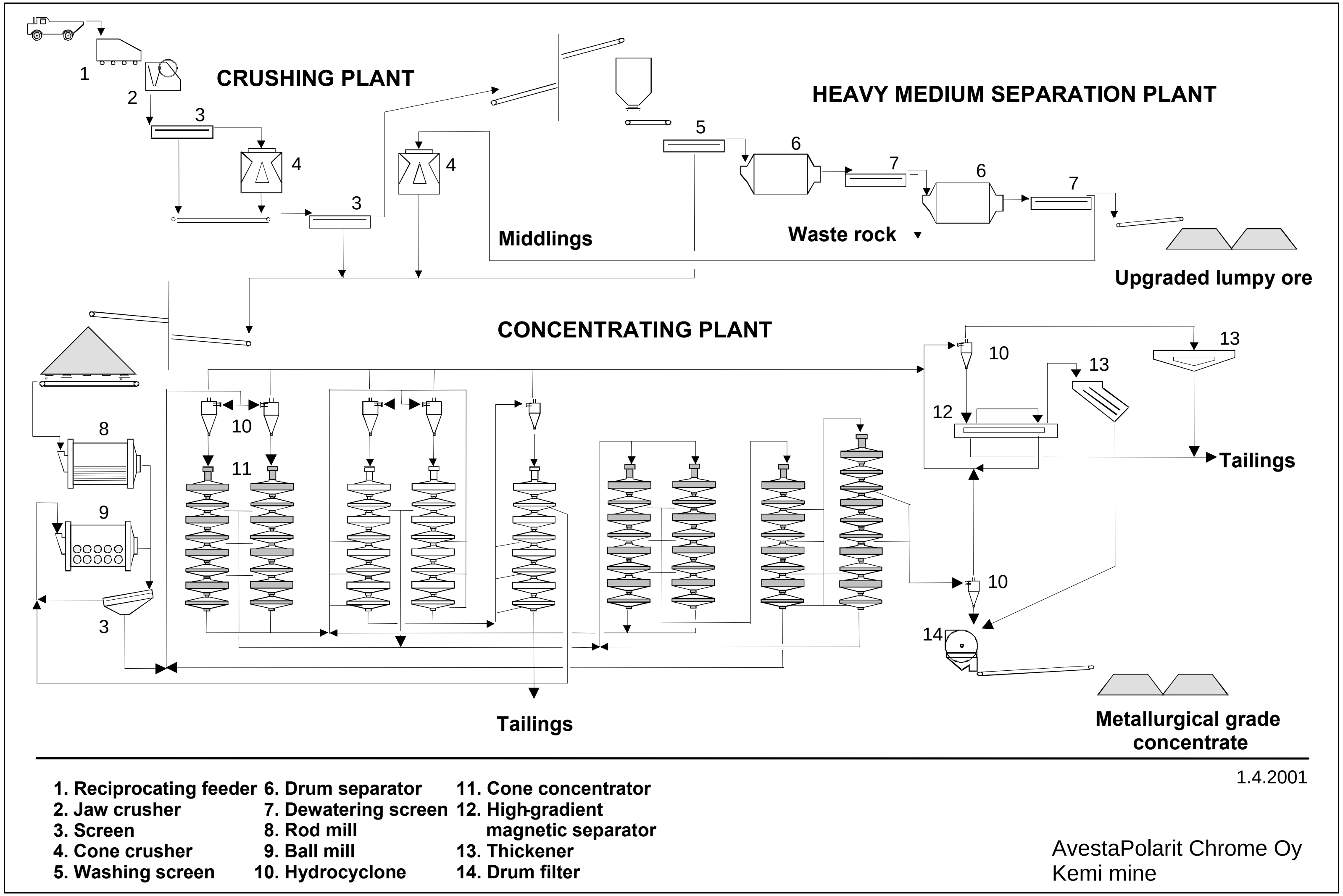


Figure 3.29: Flow sheet of the mineral processing plant at Kemi
[71, Himmi, 2002]

The process steps will be explained in the following sections in more detail.

The mineral processing plant operates at 207 t/h.

Size reduction at Kemi is carried out as follows:

- crushing in three stages with a jaw crusher and two cone crushers
- grinding in two stages with a rod mill (Ø 3.2 x 4.5 m) and a ball mill (Ø 2.7 x 3.6 m).

The following equipment and techniques are used at Kemi to separate the mineral from the gangue:

- two drum separators and three dewatering screens in a dens medium separation plant for lumps
- nine cone separators and a high-gradient magnetic separator in the concentrating plant for fine material.

3.1.3.3 Tailings management

3.1.3.3.1 Characteristics of tailings

The chemical composition of both types of tailings at the Kemi site has been determined and leaching behaviour (max. solubility/DIN 38614-S4 by Kuryk`s method and long-term behaviour) have been investigated in laboratory scale simulation tests. Also laboratory scale wind erosion tests have been done. In the tailings material, the most significant contents are Cr and Ni, which occur as insoluble compounds and are considered by the operator not to cause any negative effects.

3.1.3.3.2 Applied management methods

The Kemi TMF consists of three active and three decommissioned ponds and a total area of 120 ha. The tailings are pumped from the process to a first pond where the solids settle before the free water is led to one of the two clarification ponds. Water is re-used in the process. Excess water is led to the river system. One of the decommissioned ponds has been covered and landscaped, the remaining two await landscaping.

The distance between the mill and the TMF area is about 1 km. A stream runs just beside the ponds. The quality of water in the stream is poor as it comes from a moss area. Very close to the mine and the TMF there is a moss protection area. So, in respect of flora and fauna the area is sensitive. Drainage water leaks directly to the stream without any special collecting ditch and control system.

No baseline studies have been done.

The TMF has been built on flat land with paddock-style dams. The starter dams have been made of moraine and are founded on stable and low permeable soil. The supporting body has been made of broken rock. Where necessary, to improve the stability of the dams, counter banks are built.

The tailings from the process are distributed directly from the tailings pipe around the first tailings pond. The outlet is moved periodically so that the pond will be equally filled. The dams are raised annually with moraine and broken rock as a supporting body. External experts are usually involved when plans to raise the dam are first made.

The dam of the clarification pond is made of moraine and lined with broken rock to prevent erosion.
The tailings management area was designed in 1960`s and no closure or after-care plans were taken into account at that time. However a risk assessment has been performed more recently.

3.1.3.3.3 Safety of the TMF and accident prevention

The system has been constructed so that the surface of the water in the tailings area can be kept in balance and the excess of water, from rainfall etc., can be removed in a controlled manner.

The tailings management area is inspected daily by the operators of the mineral processing plant. The dams are inspected annually by an external expert and at five-year intervals by the dam safety authority. The comments have to be recorded in a Dam Safety Document.

As a result of recent legislation a documented emergency plan must now be created.
[71, Himmi, 2002]

3.1.3.4 Waste-rock management

Currently at **Kemi** waste-rock is deposited in three separate areas close to the mine. From 2003 the mine production will gradually change towards underground mining. The annual amount of waste-rock will, therefore, decrease and by the end of the decade all waste-rock will be directly backfilled in the underground mine. Waste-rock material from the old waste-rock dumps will also be used as backfilling material in the future.

The most important design parameters in the construction of the waste-rock heaps were:

- high stability of strata
- low permeability of the underlying strata
- short transport distance from mine
- good possibilities for material use in the future.

Drainage from the waste-rock dump area is not specifically monitored, but emissions are included in the emission figures (see Section 3.1.3.5.3), relating to calculations made according to regular samples taken from the stream both above and below the mining site.

Part of the drainage water is collected in a ditch and is led with other drainage waters from the industrial area to the tailings management area. There is also a part of the drainage that drains directly to the nearby stream.

3.1.3.4.1 Site closure and after-care

No plans for closure or after-care have been made. Also, no money has been reserved for closure and after-care.

The expected lifetime of Kemi Chromium Mine is tens of years. Therefore, no closure plans have been developed, as it can be assumed that both technical and economic plans will be further developed. There are no legal requirements to reserve money for closure and after-care.

As described above, waste-rock material will be used as the backfilling material in the underground mine in the future. No alternative use for the waste-rock can be foreseen. A plan for landscaping has been made, but no further closure plans exist.

3.1.3.5 Current emissions and consumption levels

3.1.3.5.1 Management of water and reagents

The following table shows the reagents and steel in the mills consumed per tonne of ore processed.

Reagent	Consumption (g/t of ore processed)
Flocculant	13
Steel balls	50
Steel rods	200
Ferrosilicone (for dens media separation)	80

Table 3.37: Consumption of reagents and steel at the Kemi site

In the process there are arrangements to carry out internal re-circulation of process water to minimise fresh water consumption. Re-use of the clarified water from the tailings management area covers almost 100 % of the total demand of water in the process. Sometimes (usually, when a dam raise is ongoing) it is necessary to add fresh water. Excess water from the system is removed to the stream without any further treatment.

A water balance is not available.

3.1.3.5.2 Emissions to air

Dust emissions are not regarded as a significant problem. The mineral processing plant has de-dusting equipment installed. The dust emissions from the mineral processing plant have been estimated to be approx. 1.8 t/yr. The area of influence is assumed to be very limited based on results from moss investigations. At intervals of five years, sampling of moss is carried out for determination of heavy metals and suspended particles.

Dust from the open pit and loading area has been estimated to be around 30 t/yr. Also in this case the area of influence is very limited.

Emissions from the waste-rock dumps to air are not specifically monitored. However, any dusting from the dumps is monitored in an integrated way for all emissions to air in the moss-investigations described above.

3.1.3.5.3 Emissions to water

The discharge to the stream is sampled on a monthly basis and is carried out by an external expert, also taking samples from the surrounding streams.

For the year 2000, the total emissions to surface water are summarised in the table below. The year 2000 was exceptionally rainy and wet, which resulted in extraordinarily high amounts of discharge from the pond system. However, this did not influence the other parameters listed in the table.

Parameter	Units	Amounts
Discharge from pond system	Mm^3	1.67
Ca	t	191
Fe	kg	11000
total solids	t	33
Cr in total solids	kg	79

Table 3.38: Emissions to surface water at Kemi site

3.1.3.5.4 Soil contamination

No significant soil contamination has been reported at Kemi. Limited areas, such as locations of old stockpiles of chromium concentrate, may be contaminated.

3.1.3.5.5 Energy consumption

The energy consumption for the tailings management is given in the table below for the year 2000.

Process step	Electrical energy consumption (kWh/tonne of ore processed)
Mineral processing	16.6
Dewatering	1.5
Tailings management	0.9

Table 3.39: Energy consumption data at Kemi site

3.1.4 Iron

This section includes information about the **Kiruna** and **Malmberget** mines in Sweden and the **Steirischer Erzberg** in Austria

3.1.4.1 Mineralogy and mining techniques

Commercial grade iron ores are mainly mined from proterozoic sedimentary banded iron formations. The major ore minerals are haematite (Fe_2O_3), magnetite (Fe_3O_4) and siderite (in order of importance). The main world producers are Russia, Brazil, China, Australia, India and the US. In Europe, the main producer of iron ore is Sweden. Its occurrences are phophorous magnetite ores, which are related to proterozoic syenite prophyry volcanic activity. Several smaller mines, mainly in Central and Southern Europe (e.g. ‘Steirischer Erzberg’) produce lower grade siderite iron ores (iron carbonates), which are also sediment-related ore formations.

Mining operations normally consist of preparation, including stripping or drifting, drilling, blasting and transportation prior to processing.
[49, Iron group, 2002]

Underground mines
The magnetite orebody in the **Kiruna mine** is about four kilometres long with an average width of 80 m and extends to an estimated depth of around two kilometres at an incline of roughly 60˚. The main haulage level is at a depth of 1045 m. Mining of the orebody between the 1045 m and 775 m levels will continue until about the year 2018. To date, about 940 million tonnes of ore have been extracted from the Kiruna orebody. Approximately 20 - 23 million tonnes of crude ore is mined every year from the ore, sending approximately 5 million tonnes to the coarse tailings facility and 1.7 million tonnes to the fine tailings facility.

The orebody is divided into ten blocks. Each block has its own group of shafts, each consisting of four shafts, except for the two northernmost blocks (the Lake Ore), which have three. In total, the Kiruna mine has 38 such shafts. Each shaft in a group is about 30 m from the next. The ten mining blocks are accessed via five separate ramps. An extension of each ramp is cut into the two neighbouring blocks on one side. By linking the blocks in this way, five smaller "mines" are created. Each block has its own air intake and exhaust shafts. The geographic division of the orebody into five mines enables greater mining efficiency. Since the mines are well separated from each other, ore can be extracted from one mine while blasting or maintenance are taking place in another. The mining operation passed the 775 m level in the summer of 1999. Mining

will take place above the 1045 m level until the year 2018. The orebody between 775 m and 1045 m is divided horizontally into nine slices, each of which is 27.5 m high. The distance between ore passes is 25 m. Each blast brings down about 10000 tonnes of ore.
[49, Iron group, 2002]

The **Malmberget mine** consists of about 20 ore bodies, 10 of which are currently being mined. Most of the ore base is magnetite, but there are also occurrences of non-magnetic haematite ore. Malmberget's newest main haulage level is at a depth of 1000 m. Up until now, about 350 million tonnes have been extracted from the ore bodies. About 12 million tonnes of crude ore are mined from the ore bodies every year, generating 5.6 million tonnes of tailings each year.

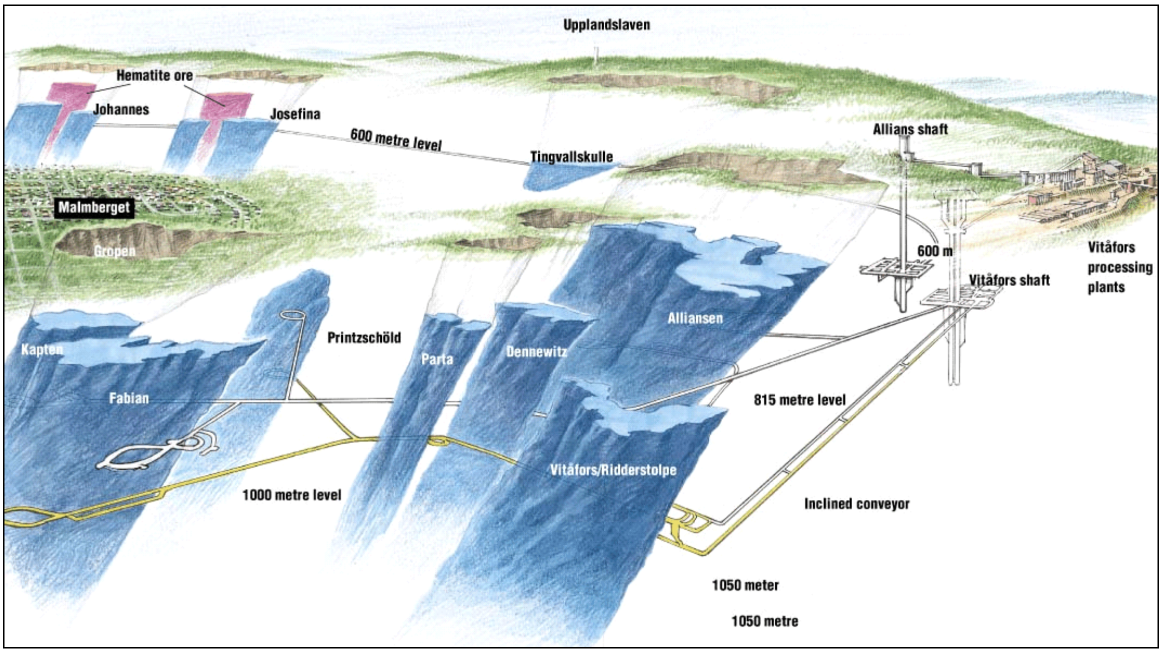


Figure 3.30: Illustration of the Malmberget ore deposit
[49, Iron group, 2002]

The ore field is 4.5 km long in an E – W direction and 2.5 km in a N – S extension. In the western part of the mine, the ores form more or less continuous undulating bands of lens-shaped ore bodies. The ores in the eastern part of the mine exhibit a more complicated, intensively folded, tectonical structure. The ore bodies are steep with great local variations. The thickness of the ore bodies varies between 20 and 100 m. The host rock is acidic to intermediate highly deformed and metamorphosed volcanic rocks, now appearing as 'leptites' (fine-grained feldspar-quartz rocks) and gneisses. The ore field is generally metamorphosed to lower amphibolite faces. In the western part of the ore field, local higher metamorphosed grade occurs.

Both Swedish mines use large-scale sub-level caving as their mining method.

Preparation/Development

At **Kiruna,** the first step is to develop drifts straight into the orebody. Drilling is done with electric-powered hydraulic drill rigs. Rounds of up to 60 holes, each five metres deep, are drilled. These holes are then charged with explosive and blasted. Rounds are blasted during the night. Ore from these blasts is removed with loaders. Then, the next round is drilled, etc., until the entire development drift is ready. Drifts can be up to 80 m long. If necessary, the walls and roofs are reinforced with rock bolts and/or concrete (so-called 'shotcrete'). Once the initial development work is completed, or when a number of cross-cuts in the same area are ready, the next step in the production chain commences, i.e. production drilling and blasting.

Production

When a number of development drifts have been developed, production drilling of a 27.5 m high 'slice' can begin. This is carried out by remote-controlled production drill rigs. The operators control several rigs in the production area by remote control from control rooms. The rig drills upwards into the ore, forming fan-shaped patterns, each with ten holes. The holes are normally 40 - 45 m long, and straight, so that subsequent loading with explosives and blasting can be done efficiently. When a pattern of holes has been drilled, the rig is moved back three metres, then drilling of the next pattern begins. About 20 of these patterns will be drilled in an 80 m long drift. Once this is completed, loading of the holes can begin.

A robot injects explosives into the drill holes in one pattern. Blasting is done every night. Each round brings down about 10000 tonnes of ore. When the blast has been ventilated, loading with wheeled loaders (LHD) can begin. Then, the next pattern is charged, etc. The procedure is repeated until the entire ore pass has been mined out. Electric wheeled loaders load and carry the ore to vertical shafts (ore passes), located along the orebody. Each loader carries a bucket payload of 17 - 25 tonnes and tips its load to an ore pass. By gravity the ore falls down to bins, located just above the main level.

In the **Kiruna** mine remote control electric loaders are also utilised. Here, the operator sits in front of a monitor in a control room, and 'drives' the machines in the production area. The machines navigate with the help of rotating lasers and reflectors on the walls of the drifts. Information, such as the position of the machine, is sent via a number of wireless base stations to the control system in the control room computer.

The main haulage level in the Kiruna mine is at the 1045 m level. Ore is tapped via remote control from the bins into railway cars. A driverless train, consisting of an engine and 24 cars, carries the ore to one of four discharge stations. When the train passes the station, the bottoms of the cars open and the ore falls down into a crusher bin, from which it is fed to one of four crushers. The ore is crushed into lumps of about 100 mm diameter. Nine locomotives and about 185 cars are operated on the main level. Each train carries approx 500 tonnes of ore.

Mining in **Malmberget** takes place at several different levels, as there are many ore bodies. The main haulage levels are at 600, 815 and 1000 m. There are crushers at each level. Twelve large mine trucks, with payload capacities of 70 to 120 tonnes, are operated at these levels. The trucks are driven to vertical shafts. Drivers control loading from inside the cab of the truck. The fully-loaded truck is then driven to a discharge station and the ore is emptied, sideways, into a crusher bin. This is also controlled from the cab of the truck. The ore is fed into the crusher and crushed into lumps of about 100 mm diameter.
[49, Iron group, 2002]

Open pit mines

The valuable mineral at **Steirischer Erzberg** is the iron-mineral siderite and the gangue mineral is ankerite. The iron content of the ore is approx. 21 %.

The Erzberg mine is an open pit operation, with a yearly production of 3.8 million tonnes/yr, of which 1.2 million tonnes is waste-rock. Conventional drilling and blasting are used. Transportation is carried out with wheel loaders and trucks. Within the pit there are 20 benches with an average height of 24 m in operation.
[55, Iron group, 2002]

3.1.4.2 Mineral processing

Typically after extraction, the ore is crushed and ground in various stages to achieve the required size. This is followed by either screening to final products, lumps and fines, or further treatment. The choice of the mineral processing methods depends on the ore type, chemical composition, fineness, etc. The most common methods used are magnetic separation, usually high intensity magnets for concentrating haematite ores and low intensity for magnetite, as well as gravity separation and flotation. The grade of the ore and the treatment method both influence the amount, type and composition of the tailings.

At **Steirischer Erzberg** the mineral processing plant treats 1.7 million tonnes of ore per year of which 0.98 million tonnes becomes concentrate, 0.7 million tonnes coarse tailings (co-deposited together with waste-rock) and 0.1 million tonnes fine tailings. 0.9 million tonnes of ore per year is sold directly as low-grade ore without processing.

3.1.4.2.1 Comminution

The **Kiruna** and **Malmberget** operations include in pit crushers (product 100 % passing 100 mm) and secondary crushing for sinter feed production. In-pit crushing, secondary crushing, AG mill/ball mills and pebble mills are applied for pellet production [49, Iron group, 2002]. At the **Erzberg** operation, two gyratory crushers (product 100 % passing 120 mm) and secondary crushing are applied [55, Iron group, 2002].

3.1.4.2.2 Separation

The **Kiruna and Malmberget** operations use dry magnetic separation (in the so-called 'sorting plant') followed by wet magnetic separation for the sinter feed production. Dry magnetic separation, wet magnetic separation, hydrocycloning and flotation are applied for the pellets production in the so-called 'concentrator' (in Malmberget no flotation is required) [49, Iron group, 2002].

The following figure shows the Kiruna concentrator, which generates the feed for the pellet plant.

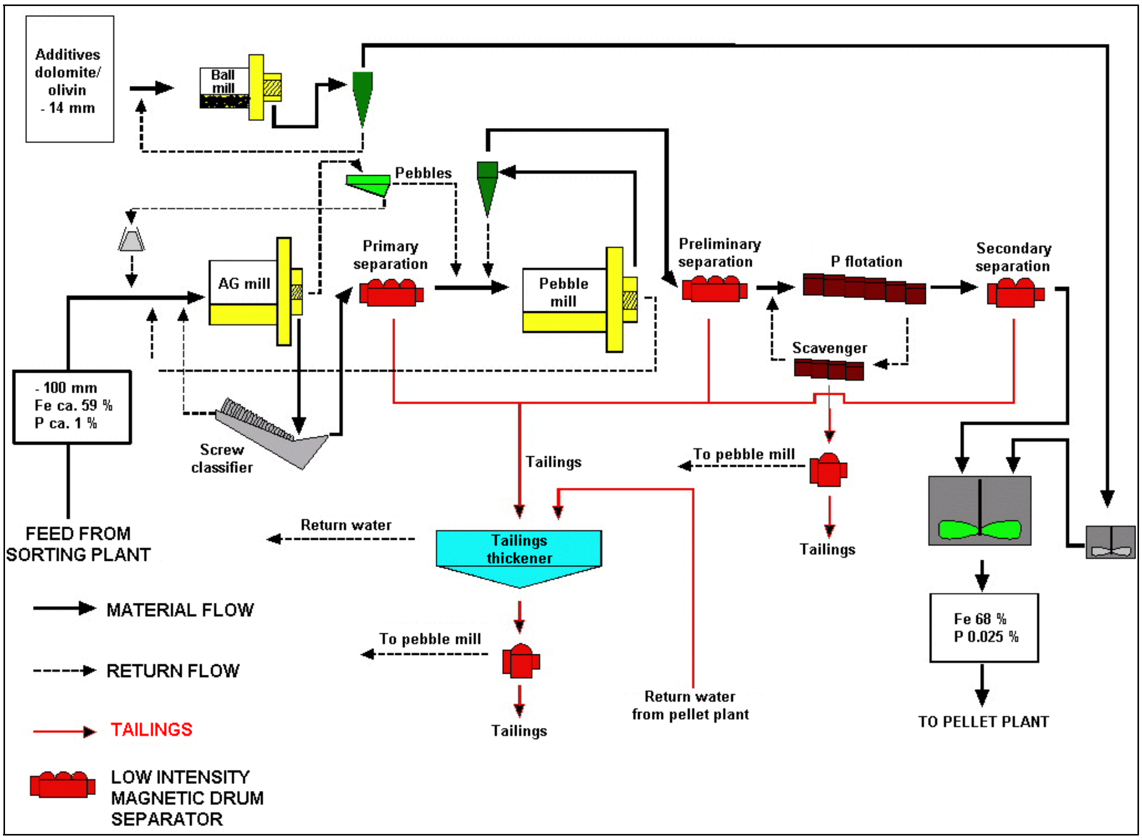


Figure 3.31: Kiruna concentrator

At **Erzberg** the coarse fraction, i.e. 8 - 30 mm and 30 - 120 mm, are separated by dense medium separation. Finer fractions, 1 - 4 mm and 1 - 8 mm, are separated by dry high intensity magnetic separation. The concentrate is further crushed to <8 mm. The fines, 0.1 - 1 mm, are dewatered via screw classifiers and are hauled, together with coarse tailings from the dense medium separation and the high intensity magnetic separation, to heaps within the mining area. Blending of the concentrate with ‘direct ore’ (ore that is not processed) is done in the final crushing and screening.

The process water, which is mainly the overflow from the screw classifiers, is treated in three 32 m continuous thickeners. The overflow is recycled back to the process, whilst the thickened slurry is pumped to tailings pond.
[55, Iron group, 2002]

3.1.4.3 Tailings management

3.1.4.3.1 Characteristics of tailings

Iron ores are usually mined as oxides (e.g. Kiruna and Malmberget) or as carbonates. Two tailings fractions, coarse and fines, are generated in the mineral processing step. The coarse tailings are managed on heaps and the fines are pumped into ponds. The tailings and waste-rock, if the iron is mined as oxides, are not acid generating.

The tailings from iron ore production are well characterised in the **Kiruna** area with regards to

- mineralogy
- geochemistry (kinetic leaching tests, trace element analysis)
- mechanical/geotechnical properties.

The tailings material at **Malmberget** has not been characterised.
[49, Iron group, 2002]
Example results from Kiruna are given in the tables below.

Compounds	Average Concentration (wt. %)
SiO_2	33.82
TiO_2	1.21
Al_2O_3	6.82
MnO	0.15
MgO	6.9
CaO	15.7
Na_2O	2.02
K_2O	1.89
V_2O_5	0.06
P_2O_5	8.1
Fe_xO_y	16.5
Total	**93.17**
Element	
Fe	11.6
P	3.55
S	0.35

Table 3.40: Average concentrations in wet-sorting tailings from Kiruna and Svappavaarra [82, Iron group, 2002]

Element	Wet-sorted tailings (ppm)	Other tailings (ppm)
As	3.67	18.1
Ba	168	205
Be	8.25	6.10
Cd	0.14	0.10
Co	94.2	67
Cr	13.4	23.5
Cu	356	211
Hg	<0.0400	0.060
La	107	331
Mo	15.4	11.8
Nb	11.9	<12.0
Ni	82.4	56.5
Pb	9.35	7.56
S	4990	4130
Sc	48.2	26.7
Sn	36.8	31.1
Sr	30.3	80.4
V	523	290
W	11.9	<12.0
Y	40.6	170
Yb	7.78	15.4
Zn	53.5	42.5
Zr	114	161
Notes: Samples marked with < are below detection limit, the numbers indicate the detection limit		

Table 3.41: Average trace element concentrations for wet-sorting tailings and other tailings material at Kiruna and Svappavaarra
[49, Iron group, 2002]

Geotechnical properties for the tailings material in Kiruna have been investigated for its use as a dam construction material. It was concluded that the tailings need to be cycloned in order to fulfil the requirements for dam construction due to the grain size distribution.

Undisturbed samples of tailings have been taken at different depths in the impoundment in both Kiruna and Svappavaara. The typical results are:

- bulk density 1.71 - 2.30 t/m^3
- calculated dry density 1.66 - 1.97 t/m^3
- density of particles 3.2 t/m^3
- friction angle 19°- 26.5°

Samples of tailings material collected from the gravity separation circuits (excluding particles from the pellet production) show the following grain size distribution:

Size (µm)	Cumulative % passing
700	100
60	75
2	5

Table 3.42: Size distribution of tailings from gravity separation
[49, Iron group, 2002]

Samples of tailings material collected after the separation by screw classifiers show the following, slightly finer, grain distribution:

Size (µm)	Cumulative % passing
60	91
40	80
2	8.8

Table 3.43: Size distribution of tailings after separation by screw classifiers
[49, Iron group, 2002]

Samples are collected on a frequent basis from the tailings deposition stream in order to evaluate the efficiency of the separation method.

3.1.4.3.2 Applied management methods

Note: The coarser part of the tailings which is co-deposited with waste-rock, is regarded as waste-rock and will be described in the waste-rock section (see below).

Kiruna (which has tailings ponds in Kiruna and Svappavaara) and **Malmberget** tailings facilities consist of tailings ponds and subsequent clarification ponds. All operations deposit their tailings using hydraulic methods (pumping in pipelines or by gravity flow in trenches). Conventional earth dams are used for all dams. The core consists of compacted till and filters. Support fill consists mainly of waste-rock. The three tailings ponds are described in detail below, with key information for each tailings pond being summarised in tables as well. All sites follow a very similar tailings management since the material being deposited, as well as the meteorological, geological and hydrological settings, are relatively similar.

At all sites, tailings slurries have a low solids content, ranging from 3 – 5 % to 10 – 15 %. The discharge point has remained in nearly the same location throughout the operation of the tailings ponds. In order to increase the solids content and to change the distribution of the tailings, the use of a mobile discharge point or cyclones have been discussed for future raises of the dams.

The freeboard at the tailings dams are 2 m at two of the facilities and 1.2 m at the third. The freeboard at **Kiruna** and **Malmberget** is based on Swedish guidelines for water retention dams (RIDAS), and includes precipitation, inclined water surface and wave run up. For a class 2 dam it should be possible to decant excess water from a one in a 100 year event, 24 hours rainstorm event, without a rise in the water level. The discharge of tailings into the ponds is controlled by a relatively constant operation system which produces a constant flow of tailings.

The starter dam at the tailings facility in **Kiruna** was originally completed in 1977. The tailings dam was then raised twice in 1984 and 1992 using the centreline method. The current maximum dam height at Kiruna is 15 m. A new rise has been applied for since the impoundment will be full at the end of 2003.

From the tailings pond, water is decanted to the clarification pond through two decants. These decants each consist of two vertical intake towers, with a submerged intake level, due to ice forming on the surface in winter. From the intake towers, horizontal pipes connect into one pipe/culvert (1400 mm in diameter) per decant going under the dam. Downstream of the dam there is a control chamber from where it is possible to regulate the flow. From the clarification pond, the water is decanted in a similar way, although with one change being that downstream of the clarification pond the water is pumped back to the process via a storage pond near the plant or discharged into the recipient. As a result of the new guidelines, a new emergency outlet was constructed for the clarification pond in the year 2000. The emergency outlet consists of a 13.5 m wide channel through the top of the dam near one of the abutments.

The main technical characteristics of the Kiruna tailings dam system are summarised in the following table.

	Tailings dam			Clarification pond	
Dam type	Off-valley site			Off-valley site	
Dam area (km^2)	4.2			0.96	
Tailings volume	9			n/a	
Water volume (Mm^3)	7.4			2.3	
Dam body	C-D	O-R	R-B	R-S	S-F
Dam type	Centreline	Centreline	Centreline	Centreline	Centreline
Highest height (m)	8	15	15	11	13
Dam length (m)	1450	2560	1040	1440	850
Dam width (m)	15	15	15	15	15
Lowest freeboard (m)	2.0[1)]	2.0[1)]	2.0[1)]	2.0[1)]	2.0[1)]
Upstream slope	1:1.8	1:1.8	1:1.8	1:2	1:2
Downstream slope	1:1.4	1:1.4	1:1.4	1:1.5	1:1.5
Volume of dam construction material (Mm^3)	0.66	1.58	0.86	3.00	0.39
Core width	4	4	4	4	4
Fine filter width, (m)	1.5	1.5	1.5	1.5	1.5
Fine filter grain size (mm)	0 - 6 or 0 - 8	0 - 6 or 0 - 8	0 - 6 or 0 - 8	0 - 6 or 0 - 8	0 - 6 or 0 - 8
Coarse filter width (m)	0 - 30 or 0 - 100	0 - 30 or 0 - 100	0 - 30 or 0 - 100	0 - 30 or 0 - 100	0 - 30 or 0 - 100
Support fill and erosion protection material	Waste-rock	Waste-rock	Waste-rock	Waste-rock	Waste-rock
Support fill grain size (mm)	0 - 200	0 - 200	0 - 200	0 - 200	0 - 200
Erosion protection grain size (mm)	0 - 100	0 - 100	0 - 100	0 - 100	0 - 100
Discharge arrangement			2 decant towers	Emergency overflow	2 decant towers

Table 3.44: Characteristics of the Kiruna tailings dam system
[49, Iron group, 2002]

The other tailings facility used for ore from **Kiruna** processed in Svappavaara is Svappavaara tailings facility, 50 km south-east of Kiruna. This facility consists of three ponds, the tailings pond, the first clarification pond and a second clarification pond called the recipient pond. In addition to these constructed ponds, a natural lake, is used as a water resource. All dams are valley-site impoundments.

The recipient pond was the first to be built, and came into operation in 1964. The purpose was to collect the draining water from the tailings settling naturally on the hillside. Water was then decanted from the recipient pond to a lake. Because of the properties of the tailings and due to the terrain, (i.e. steep ground) most tailings settled too close to the downstream dam. Therefore, a second retention dam, i.e. the tailings dam, was constructed to prevent the tailings from settling too close to the recipient dam, which has since then worked as a clarification pond. Later, in 1973, a third dam was constructed right across the tailings impoundment, to keep the tailings in the upstream part and use the downstream part as a clarification pond. This dam is constructed of rock fill as a draining dam. Due to problems with ice an overflow outlet was constructed in this dam in 2001.

From the first clarification pond the water is decanted to the recipient pond by two decants with vertical intake towers and horizontal culverts under the dam. Stop logs at the intake tower regulate the water flow. The decant at the recipient dam is similar to the ones in Kiruna where the water is regulated from the downstream side. From there the water can be pumped back to the process via a lake or discharged to the recipient. Normally, no excess water results as most of the water is re-circulated.

The dams around the tailings pond and clarification pond as well as the rock fill dam dividing the two has been raised several times (11 times in total). For the downstream clarification dam the downstream method has been used and for the tailings dam an the rock fill dam the upstream method has been used. The maximum height today is 21 m and approximately 15 million tonnes (dry weight) of tailings have so far been deposited.
[49, Iron group, 2002]

The technical characteristics of the Svappavaara tailings dam system are summarised in the following table.

<table>
<tr><th></th><th colspan="2">Tailings pond</th><th>Clarification pond</th><th>Recipient pond</th></tr>
<tr><td>Dam type</td><td colspan="2">Off-valley</td><td>Off-valley</td><td>Off-valley</td></tr>
<tr><td>Dam area, km^2</td><td colspan="2">1.2</td><td>0.7</td><td>0.42</td></tr>
<tr><td>Tailings volume, Mm^3</td><td colspan="2">4.5</td><td>1.5</td><td>0.2</td></tr>
<tr><td>Water volume, Mm^3</td><td colspan="2">0.4</td><td>4.5</td><td>0.45</td></tr>
<tr><td>Dam section</td><td>Soil dam</td><td>Blocking dam</td><td>Soil dam</td><td>Recipient dam</td></tr>
<tr><td>Dam type</td><td>Upstream</td><td>Upstream</td><td>Downstream</td><td>Downstream</td></tr>
<tr><td>Max. height, m</td><td>15</td><td>15.5</td><td>21</td><td>10</td></tr>
<tr><td>Dam length, m</td><td>2030</td><td>1100</td><td>2350</td><td>800</td></tr>
<tr><td>Dam width, m</td><td>8.3</td><td>12</td><td>7.2</td><td>6.0</td></tr>
<tr><td>Smallest freeboard, m</td><td colspan="2">2.0</td><td>1.8</td><td>2.5</td></tr>
<tr><td>Upstream slope</td><td>1:2</td><td>1:1</td><td>1:2</td><td>1:2</td></tr>
<tr><td>Downstream slope</td><td>1:1.5</td><td>1:3/1:7</td><td>1:1.5</td><td>1:1.8</td></tr>
<tr><td>Approx. volume of dam construction material used to date, Mm^3</td><td>0.36</td><td>0.5</td><td>0.46</td><td>0.17</td></tr>
<tr><td>Discharge arrangement</td><td></td><td>overflow outlet</td><td>2 decants</td><td>1 decant</td></tr>
</table>

Table 3.45: Characteristics of the Svappavaara tailings dam system
[49, Iron group, 2002]

Tailings pond

- soil dam

The starter dams comprises a homogeneous moraine material with an erosion cover of 0 - 100 mm grain size. The erosion cover is 1 m thick on the downstream slope and 1.5 m thick on the upstream slope. The slope angle is 1:1.5 and 1:2 for the downstream and upstream slopes, respectively. Height increases of the dam have been constructed based on the upstream method with a four metres thick impervious core consisting of moraine material. There is a one metre thick transition layer on both sides of the core with a grain size of 0 - 100 mm. The erosion cover on the downstream side is approximately 0.5 m thick with a grain size range of 0 - 100 mm. The upstream support fill and erosion cover consist of material with a grain-size range of 0 - 200 mm and 0 - 500 mm, respectively. A two metres increase in dam height, using the downstream method, is planned for the summer of 2002.

- blocking dam

The blocking dam is made with a dam of waste-rock without an impervious core. The dam is built with the upstream method with a grain size range of 0 - 500 mm.

Clarification pond
The clarification pond is built with a soil dam constructed as a conventional dam. The starter dam is made with a homogeneous moraine material with an erosion cover consisting of material with a grain size of 0 - 100 mm. The erosion cover is 1.0 m thick on the downstream slope and 2 m thick on the upstream slope. The slope angles are 1:1.5 and 1:2 for the downstream and upstream slopes, respectively. Further height increases have been constructed based on the centreline method.

Recipient pond
The dam at the 'recipient reservoir' is built as a conventional dam and raised using the centreline method. The vertical impervious core consists of, at the top, a 3 m thick moraine material. On both sides of the impervious core, is a 2 m thick fine-sand filter consisting of a material with grain size range of 0 - 32 mm. Outside the fine filter is a coarse filter material with grain size of 8 - 64 mm. On top of the core and the fine filter is a 0.5 metre thick horizontal layer consisting of bark. The support material consists of blasted rocks on both sides. The downstream slope angle is 1:1.8 and the upstream slope angle is 1:2.
[49, Iron group, 2002]

There are five dams within the **Malmberget** mining operation; tailings dam, clarification pond, pond for biological degradation, a reserve pond and a buffer pond. Only the two first dams are described in this document.

The tailings pond was constructed in a lake. The tailings pond consists of primarily two dams of different design, the B-A dam and the C-D-E-F dam. Water is funnelled through a decant tower from the tailings pond into the clarification pond. Water is then pumped from the clarification pond back to the processing plant.

The tailings dam at Malmberget was originally constructed in 1977 and has been increased in height five times since then. The height of the dam reaches 35 m. It will be full by the end of 2002 and a height increase has been designed using the upstream method. This height increase will secure the tailings deposition for another 25 years assuming today's production rate of 1.5 million tonnes/yr. The whole pond currently contains approximately 16 million tonnes (dry weight) of tailings.

The following table lists the characteristic data for the Malmberget TMF. The tailings dam and clarification pond were constructed using the natural terrain with a main dam at the end of the valley.
[49, Iron group, 2002]

	Tailings dam		Clarification pond
Dam type	Valley dam		Valley dam
Dam area (foot print)	1.8 Mm^2		0.12 Mm^2
Tailings volume	16.8 Mm^3		n/a
Water volume	0.4 + 1.2 Mm^3		0.25 Mm^3
Dam section	b-a	c-d-e_1-f	j_1-j_2
Dam type	Up/downstream	Downstream	Centre line
Maximum height	13 m	35 m	14 m
Dam length	700 m	2500 m	1100 m
Dam width	40 m	40 m	8.0 m
Smallest freeboard	1.2 m	1.2 m	0.5 m
Upstream slope	1:2	1:2	1:1.5
Downstream slope	1:1.5	1:1.5	1:1.5
Approx. current tailings dam volume	0.2 Mm^3	2.5 Mm^3	0.2 Mm^3

Table 3.46: Characteristic data for the Malmberget tailings and clarification ponds and dams [49, Iron group, 2002]

Tailings dam

The dam was designed to span the width of a lake, thus blocking the lake water. The inside of this blocking dam is designed as an upstream dam to level 271 m (see Figure 3.32). The upstream dam is built with a 7 m thick impervious core of moraine material with permeability of 10^{-8} m/s. The impervious core is slanted 1:1.5. Below and above the impervious core is 1 m thick filter with grain size of 0 - 100 mm and a permeability of 1 x 10^{-3} – 1 x 10^{-4} m/s.

From level 271 m, the dam is built using the downstream method with an inside slope of 1:2 and an outside slope of 1:1.5. Between the support material and the impervious core is a 1 m filter as described above. On top of the core is a 1 m thick erosion layer consisting of material with grain size 0 - 70 mm and permeability of 1 x 10^{-5} m/s.
[49, Iron group, 2002]

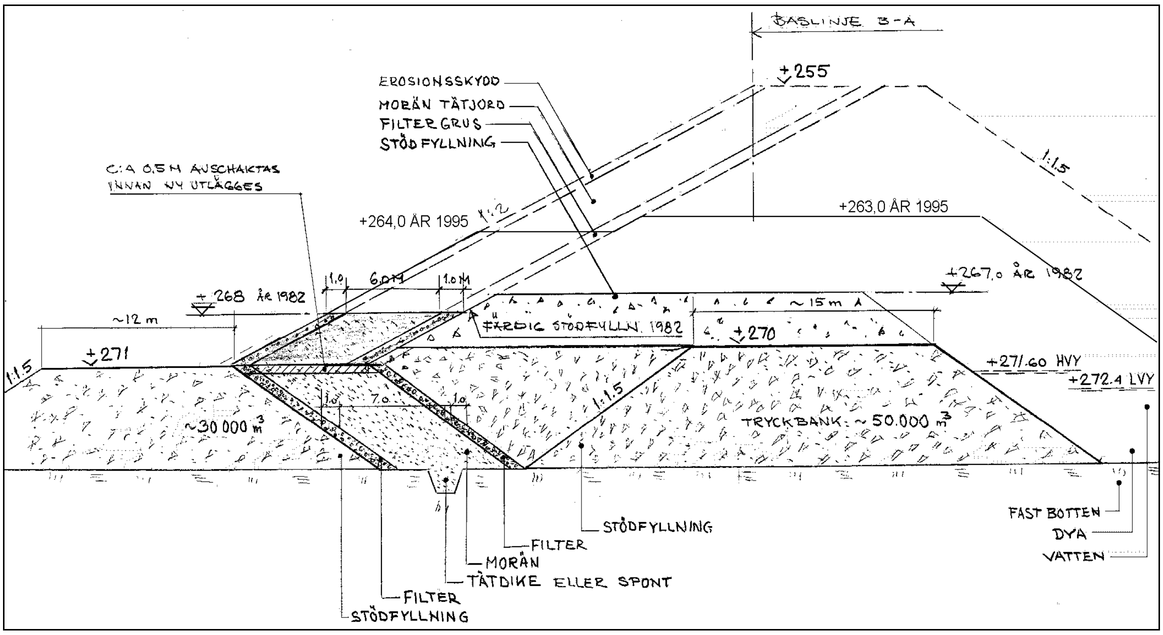


Figure 3.32: Cross-section of Malmberget tailing dam
[49, Iron group, 2002]

Clarification pond
The dam of the clarification pond is designed as a conventional dam with a 4 m thick impervious core of moraine material. On each side of the core is a 1 m thick filter layer. Outside this is a support material and on top an erosion layer. Both the support material and the erosion layer are made of coarse dry tailings material. Outside and inside slope are 1:1.5 [49, Iron group, 2002].

At **Steirischer Erzberg,** the tailings facilities where the fine tailings are deposited cover about 40 ha and are divided into 6 tailings ponds of which 3 are currently in operation. Up to 2002, about 5.2 Mm^3 (9.4 million tonnes) of tailings materials had been deposited in total. An overview of the operation is given in the figure below.

Figure 3.33: Steirischer Erzberg
[55, Iron group, 2002]

The tailings ponds are built on top of the 50 to 100 m high waste-rock dumps and are constructed to be of low permeability but use infiltration areas to drain off the clarified water. The draining water infiltrates through the waste-rock dump and mixes with the water in a stream that flows underneath the dump. This is described in more detail below.

The distance between the processing plant and the active TMF varies between 500 and 2000 m. The tailings have to be pumped from an altitude of 745 m to an altitude of 873 m and 980 m respectively.

In the first half of the 20th century the area served as a waste-rock dump area for the mining operation. This buried the stream in this valley for practically its the total length. The method applied at that time - trail bound transportation with comparatively high dump heights - resulted in a high proportion of big sized blocks at the bases of the dump, due to size segregation. The base of the dump was constructed by removing the topsoil and installation of a bottom layer of big rock blocks. Accordingly, a sufficient permeability for the dewatering of the valley was achieved and has remained intact until now. The majority of the drained water from the dump emerges at the toe of the dump. The dump material is mainly ankerite and limestone.

The fundamental design criteria were stability and tightness to water. All dams are constructed from carbonate tailings (0.15 - 120 mm) and a schist ("Werfener Schiefer") rock layer on the inner side of the dam. Sealing is done by establishing a compressed layer of schist ("Werfener Schiefer") and tailings, which provide, according to the experiences of the company, sufficient impermeability. In order to prove the suitability of the materials and techniques used for dam construction, comprehensive studies were conducted, comprising both in-situ and laboratory tests (geotechnical parameters, permeability, internal friction angle, etc.)

Investigations have shown, that the stability of the dam construction is almost independent of the tailings situation inside the pond, if a sufficiently impermeable seal layer made of compressed schist and tailings is put in place before starting the discharge of tailings. Accordingly the impermeability of the sealing layer is of great importance.

During design and construction, attention was paid to the execution of the sealing layer and the drainage of the tailings water. Depending on the dam material for each pond a particular position is selected for discharge of the water from the pond. These discharge areas are 20 to 30 m in length and consist of weathering resistant materials of appropriate fragmentation in order to warrant the necessary permeability.
[55, Iron group, 2002]

3.1.4.3.3 Development of new deposition methods

The construction of a drained cell pond is currently investigated in **Kiruna** and **Malmberget**. If the results from this test project are positive, the method will be modified to suit large-scale applications. The technique is based on the grading of waste-rocks taking place down slope from the truck dumping. This grading results in a pervious/well draining filter dam. Constrained cells can be constructed with this technique, in which tailings are discharged hydraulically. The filter dam then contains the tailings material, while process water is drained.

A collection ditch or walls will be constructed around the filter dams to collect the draining water. Collected water will be directed towards the existing tailings dam. The suggested location of these draining cell ponds will result in the tailings dam acting as a clarification pond for suspended material transported through the filter dam.

Some of the tailings material will pass through the filter dam to the existing tailings dam. This may result in a need for a height increase of the existing tailings dam during the planned 16 year deposition period, depending on the efficiency of the filter dam. It is necessary to have a high filtering efficiency (sand deposition within the cell) to make the drained cell deposition a viable method. The height increase that may be necessary (max. 1 - 2 m over the 16 years period depending upon the dam efficiency) can be constructed on the existing dam.

One advantage of this draining technique is that an increase of the footprint of the existing tailings dams is not necessary. Also since the drained cell is a ‘dry’ system the tailings can be stacked higher. Because the water from the tailings deposition is drained, failure of the filter dam is less likely. However should it fail, the effect of the failure will be reduced because water content is lower compared to the current system and tailings material escaping the cell ponds will be trapped in the current tailings dam. With the current conventional dam system the coarse tailings are treated like waste-rock and trucked to the waste-rock dump, which is very cost and labour intensive. An economic benefit for the operator is that with this new method both coarse and fine tailings can be pumped to the new TMF as a slurry.
[49, Iron group, 2002]

3.1.4.3.4 Safety of the TMF and accident prevention

At **Kiruna** and **Malmberget** discharge to the tailings dams is controlled by a relatively constant operation system producing a constant flow of tailings. The dams are inspected several times a week in line with guidelines set out in an Operation, Inspection and Maintenance (OIM) manual that has been developed for all three facilities. The inspections include evaluation of water level in the dams and the overflow ditches/funnels. All observations are logged in the field log book so that changes can be evaluated. Monthly and yearly inspections are also implemented according to the OIM manuals. Inspections are performed several times a week by operating personnel, monthly by the manager and yearly by an expert (usually the 'in house' consultant).

A classification of all dams according to hazardousness (human life, environmentally, economically) of a dam failure has been performed following the Swedish guidelines (RIDAS, see Section 4.2.3.1). For the classification, a risk assessment was performed that focused on the worst-case dam failure. Since the material, as described earlier in this document, is chemically stable, the risk of causing environmental hazards is very small.

The OIM manuals developed at Kiruna and Malmberget are described below.

<u>General</u>
In 2001 Operation, Inspection and Maintenance (OIM) manuals, similar to the OSM manuals described in Section 4.2.3.1, were developed for three large tailings dams. These manuals were developed in order to avoid dam failures, or, in the event that a failure takes place to advise on emergency responses to reduce the effect from a dam failure. The three manuals are very similar and will, therefore, be described together. Another objective of these manuals is to facilitate and document future design changes. The manuals are updated yearly.

The content of these manuals are as follows:

- dam design
- dam classification according to hazard (including risk assessment)
- possible actions for safety improvements
- operation, inspection and maintenance routines
- emergency preparedness plan for dam incidents (EPP).

The condition of the dams during operation can be classified in four different levels:

- normal operation, where there is no indication of changes in conditions
- tightened operation, when there may be some indications of dam fractures, high rainfall or process water output etc.
- disturbed operation, when there is an unusually high water level in the dams, distinct dam fractures, and water leakage; and lastly
- incidents, where operation is likely to be be halted.

The following paragraphs describe monitoring/dam inspection routines and dam failure emergency plans (EPPs).

<u>Monitoring and inspections of tailings facility</u>
The phreatic surface is monitored using standpipes installed in selected sections of the different dams. There are nine standpipes at the Kiruna tailings dam, fiftythree at Svappavaara and four for the tailings dams and Malmberget. Measurements are taken manually on a monthly basis as long as readings are stable, otherwise more often. Climate data is received from a weather station located at the nearest airport.

The OIM manuals describe the critical parameters for operation, inspection and maintenance. These include except in the case of dam manuals, decants and outlets, tailings discharge systems, storm water diversion channels, etc. The manuals suggest regular inspections by trained operating personnel three times a week where changes such as erosion on slopes, seepage, material transport in seepage water, which indicate internal erosion are checked. All observations from these inspections are logged in a field book. The manuals require meetings once a week for the OIM personnel, when the information collected during the week is presented and discussed and decisions on dam safety improvements are made if necessary.

A monthly inspection is performed in order to evaluate the safety of the dams and to identify any possible improvements needed to maintain a high level of safety. These inspections are to be performed by the person responsible for the tailings dam together with the operating personnel. In addition to the visual inspections, readings are also taken of the stand pipes, the seepage water and pond water levels.

An expert performs a yearly inspection (audit). At this inspection all the field notes and monthly inspection reports are reviewed and a visual inspection is performed. The report from the inspection summarises all measurements collected throughout the year, evaluates the results, and suggests possible improvements or adjustments to the dams and to the daily and monthly inspections. The yearly inspections also review and evaluate the dam calculations behind the dam designs including the operation and maintenance data.

Emergency Preparedness Plans, EPP, for the four levels of operating conditions listed and described above, have been developed. These levels require different responses, which are summarised below.

Normal Operation: Routines for normal operation in the OIM manual is followed.

Tightened Operation: When the conditions indicate an increased risk of a possible dam incident such as, increased seepage, unusual high water level in the pond etc., the facility will undergo more frequent inspections (every second day or every day) to evaluate if the conditions are improving or getting worse. The person responsible for dam safety notes all observations in the field book.

Disturbed Operation: If there are major changes on the dams, more severe than described above, e.g., extreme climate, severe erosion, internal erosion or erosion along decant culverts, major cracks, sinkholes or settlements the operation is classified as ‘disturbed operation’. At this stage preventive measures are required. The OIM manuals describe possible scenarios and suggested measures for these scenarios and recommend that an expert be consulted if necessary. All observations and measures are to be described in detail in the field logbook by the person responsible for the dam safety.

Incident: If an incident takes place a temporary stop in the mining operation is likely. An action plan to aid in decision making was established as well as both internal and external phone lists. An incident has to be followed up with a report that includes the reason for the incident and what actions were taken to mitigate the incident.

For the safe operation of the tailings ponds located on top of the waste-rock dumps at **Erzberg,** a series of monitoring and supervision measures are provided, focusing on crucial parameters. Parameters observed on a regular basis comprise:

- surface water level inside the dams (piezometer measurements)
- water level in the ponds
- subsidence measurements (surveys).

Operational instructions are also provided and cover:

- visual observations
- drainage control and the documentation of drainage failures and maintenance works
- water monitoring
- monitoring of dam stability by surveying fix points
- monitoring of water-levels within the dams.

The water quality is regularly analysed at sampling points defined by authorities and an internal analyses of water quality is done according to needs. However, due to the fact that the discarded tailings are classified as safe in respect to their geochemical environmental aspects, the environmental monitoring will merely be of a documentational and preventional character.
[49, Iron group, 2002]

3.1.4.3.5 Closure and after-care

For the three large tailings ponds at **Kiruna** and **Malmberget** formal closure plans have not been submitted for approval by the regulatory authority. A closure plan will be developed in co-operation with local and regional regulatory agencies. Those parts of the tailings dam system that might be decommissioned prior to mine closure, will be covered and re-vegetated, and if ponding takes place, water pumping and regrading may be performed.

At **Erzberg**, some small tailings ponds have been decommissioned. No approved closure plan exists for the ponds in operation, however, studies have been conducted and closure concepts have been developed. The methodology used so far for the closed ponds was dewatering and soil covering, followed by re-vegetation. Re-vegetation directly in the dewatered tailings has also been carried out successfully. These measures effectively eliminate dust emissions from the ponds. Water contamination is not an issue (as proven by 30 years of monitoring results) as the tailings are chemically stable and no reagents are used in the mineral processing. The closed ponds are continuously supervised and surveyed. Alternative uses for the tailings material are currently being investigated.

3.1.4.4 Waste-rock management

Two of the mining operations are underground mines (i.e. **Kiruna** and **Malmberget**). As a result, only smaller amounts of **real waste-rock**, as defined for this document, are excavated for access tunnels. However, the dry magnetic separation tailings are included in the discussion of waste-rock, since the management of these coarse tailings is more typical of waste-rock than tailings.

At the **Kiruna** and **Malmberget** operations the coarse tailings are transported on a conveyer from the processing plant to bins and from there hauled to the so called ‘waste-rock’ facility using dump trucks. The coarse tailings are dumped on heaps approximately 15 m high and at the natural angle of repose. In total these two sites manage about 12 million tonnes/yr of ‘waste-rock’ this way.

At **Erzberg**, approximately 1.9 million tonnes/yr of ‘waste-rock’ are managed, 0.7 million tonnes of which are the coarse tailings from the dense media separation and 1.2 million tonnes of actual waste-rock, which comes directly from the open pit mine.

3.1.4.4.1 Characteristics of waste-rock

The **Malmberget** waste-rock (the coarse tailings) has not been characterised, however, the waste-rock at **Kiruna** was tested for leachability and Acid-Base Accounting (ABA), in addition to characterisation of the ore and nativ rock during exploration. Detailed mineralogical and trace element analyses have previously been described under the tailings section (see above). Tests have also been performed to evaluate the amount of unexploded explosives left in the waste-rock material.

The leachability and ABA investigations indicated that the finer fraction of the waste-rock (from the sorting plant) had the highest sulphide content (1.4 - 3 weight. % S). The neutralising capacity from calcite is, however, higher than the acid producing potential from the sulphides.

The leach tests performed (i.e. humidity cell tests), indicate that acid being produced due to sulphide mineral oxidation is neutralised by the calcite. The investigation also indicated that silicate minerals present in the test material also act as neutralisers. The leach tests indicate that sulphate, calcium and magnesium are the main constituents leaching from the waste-rock.

The nitrate/ammonia leaching tests indicate that the ammonium nitrate left over from undetonated explosive, is easily leachable and is primarily leached by the first infiltrating rainwater on the waste-rock.

Geotechnically, the waste-rock is stable. The coarseness of the material and truck dumping stabilise the material during deposition. The chemical weathering is very slow in the northern Sweden sub-alpine climate. The generation of clay minerals due to weathering is extremely slow. Therefore, no alternative deposition method has been considered.
[49, Iron group, 2002]

At the **Erzberg** site, the waste-rock has not shown any sign of leaching and has been mineralogically characterised as follows:

- ankerite
- limestone
- schist ("Werfener Schiefer", "Zwischenschiefer"): quartz 46 %, dolomite 14 %, haematite 6 %, mica 4 %, feldspar 0.18 %, phyrophyllite 30 %
- porphyroid (small amounts): mica 8 %, quartz, 63 %, feldspar 5 %, chlorite 25 %
- fragmentation: 0 - 1500 mm.

Ankerite, limestone and porphyroid are quite resistant to weathering. On the other hand schist shows a rather high degree of weathering, in particular due to the meteorological conditions at the site.
[55, Iron group, 2002]

3.1.4.4.2 Applied management methods

There were no baseline studies performed prior to developing the waste-rock management facilities at two of the sites. However, at one site, an advanced design was carried out based on-site investigations. The locations of all dumps were chosen so as to be as close as practically and technically possible to either the mine or the processing plant.

For two of the sites the waste-rock management facility is located near the processing plant and extends to mined out open pits. In fact, at one site, the coarse tailings from dry magnetic separation were discarded into the mined out open pit over a short period using a conveyor belt system. This is not done any more because of dust problems.

At **Kiruna** and **Malmberget** the waste-rock is deposited on a thin soil cover or directly on bedrock. The bedrock consists of primarily volcanic rocks, trachytes, trachy-andesite, rhyolites, and rhyodacites. These rocks are very competent resulting in little risk for collapse into the underground mining operation [49, Iron group, 2002].

At **Erzberg**, due to the alpine location of the mine, space is scarce. The previous waste-rock dump was in operation up to the middle of the 20th century. After closure the tailings ponds were built in this dump area. As the capacity of the dump was exhausted, it became necessary to find new dumping facilities. Based on investigations done by the operator and in close co-operation with the local community, landowners and involved authorities a new area was identified for the waste-rock dump. This new waste-rock dump is located in a small valley close to the mining operation. The rivulets in this valley were dumped over while care was taken to ensure sufficient permeability for the water. Soil and loose material have been removed down to the competent rock. This formation is permeable and sits on top of an impervious bed, which consists of schist and porphyries. In the valley the base rock consists of porphyries, clay schists and carbonates. The total area of the dump is about 400 ha. Up until 2002 about 550 million

tonnes of waste-rock have been dumped at this facility. The dump extends from the 1230 m level to the toe of the end dam at the 821 m level. The dump comprises several dump areas and has a total vertical extension of more than 400 m. The maximum height of a single dump slope is 70 m. The end dam, which is situated at the lowest part of the valley has a height of 147 m. The distance from the mining faces to the dump varies between 500 m and 1500 m in linear distance. Hauling distances for truck haulage are up to 3 km.
[55, Iron group, 2002].

Design and construction
As mentioned above, Erzberg needed to locate the waste-rock dump area in a valley due to the topography in the area. For planning and operation of the waste-rock management facility particular care was taken due to the specific situation of this dump with respect to:

- dumping at a mountain-slope area
- dumping on top of rivulets
- distance to residents
- alpine climatic conditions.

Therefore the planning of the project considered three key factors:

- ground conditions (geological, hydro-geological)
- waste-rock characteristics
- dumping method.

Many options for dealing with mining, soil mechanics, geology and hydraulic systems were discussed. The following issues were evaluated:

- avoidance of erosion and stability of the dump slopes
- avoidance of accumulation of water behind and inside the dumps
- studies about the flowrate through the dumps at high water flow
- evaluation of the quality of water after percolating through the dumps.

The basis for the design and construction of the waste-rock management facility was developed by an external consultant. According to the concept worked out, the bottom layer of the dump (valley base) consists of large-sized carbonatic rocks. The cross-section of this layer was designed for a flood (100-year event) the water can percolate through the dump without problems and without producing an increase in the flow pressure. In addition, an extensive testing programme was executed by the responsible authority. Over a two year period, penetration tests have been conducted which show that the maximum water flow can be managed if the base of the dump is constructed as proposed.

Based on these expert opinions and investigations the waste-rock management facility was approved by the mining authority in 1969. The approval comprises a series of strict obligations with respect to the design and operation, including:

- before dumping, the ground had to be cleared from vegetation, trees, roots and soil
- the dump must not exceed a general slope angle of 31° upon completion
- the cross-section of the lateral ditch for drainage must be designed large enough to handle run-off waters from the slopes
- the total bottom layer of the dump must be made of carbonatic rock blocks of a size between 400 - 1000 mm and must be at least 1.5 m high
- in the area of the previous bed of the rivulet, block sizes of at least 700 mm should be used
- in the designated discharge zones only carbonatic rocks must be used
- at the toe area of the dump towards the valley a discharging body perpendicular to the valley must be made
- an appropriate monitoring system has to be implemented to check the phreatic surface within the heap
- the total workings for the dam and the separate construction phases have to be well documented.

Both design and construction were evaluated by an external expert on the basis of the existing documentation for the closure in 1996. This evaluation showed that all instructions of the authorities had been followed and that there were no indications of any instabilities of the dump slope.

As described above, the dumps have been designed to allow for a stream to flow underneath. Apart from this the main factor for the waste-rock dump design is hauling distance from the mining area. As described above, the waste-rock and the dry magnetic separation tailings are transported on trucks and dumped within the waste-rock facility. The dumping is based on the natural angle of repose with no further change of the slopes. This has been the historic way of depositing the waste-rock. Since the material is considered to have only a minor impact on surface and groundwater or the surrounding soils, changes to these practices have not been made. The use of conveyer belts or slurry pumping is frequently being evaluated to replace the truck hauling. However, truck hauling has so far been found to be the most efficient and economic way of transporting the waste-rock.
[55, Iron group, 2002]

<u>Operation</u>
The deposition of waste-rock is similar at all sites. The waste-rock is hauled by trucks from the mining faces at distinct benches via the ramp system and from the dump area to the dump positions. The material is directly dumped from the truck over the dump slope or on top of the dump.

At Erzberg, the dump heights vary between 40 and 70 m. With this method, dump slopes will be between 33° and 38°. The overall general slope angle is kept lower than 28° [55, Iron group, 2002].

At the **Kiruna** and **Malmberget** sites the dumps are constructed in 15 m high lifts. The truck dumping method results in a gradation where the larger grain sizes roll down to the bottom of the slope, while smaller grains settle higher up on the slope. This was used in the design of one of the dumps as described above in order to allow for a stream to flow underneath one of the dumps. In addition, there is likely to be some compaction on the top of each lift level due to the driving of the dump trucks. Later on natural compaction of the deeper parts of the waste-rock piles may also take place. None of these different compactions considerably influence the water flow. Most of the rainfall onto the waste-rock is likely to flow vertically through the dumps. When the infiltrating water has percolated through the dumps, a portion of the water will infiltrate the groundwater and a portion will flow on top of the bedrock and be visible as seepage at the toe of the dump. It is common practice to construct ditches at the toe of the waste-rock facility to control the seepage water. At one site, however, the seepage goes directly into the stream that flows under the dump.
[49, Iron group, 2002]

3.1.4.4.3 Safety of waste-rock facility and accident prevention

At two sites the waste-rock is considered to be chemically and geotechnically stable. For that reason, monitoring systems of the waste-rock facilities are not applied.

At the site where the stream flows underneath the waste-rock a monitoring plan is followed including geotechnical monitoring (surveying, piezometer measurements) and environmental monitoring.

3.1.4.4.4 Site closure and after-care

As a part of the permit process for the waste-rock facility, one company has developed a closure plan. As described before, the waste-rock dumps are designed with 15 m lifts. The waste-rock on top of each lift is moved inwards leaving a ledge of 30 m. The reclamation concept is to focus on re-vegetation of the ledges adding soil and seeds in line with the local vegetation. A small rock berm will be constructed at the edge of each ledge. Water will be added to the re-vegetated areas in the early stages of the reclamation project but will not be required later on.

The top of the waste-rock will slope from the centre to the edge of the waste-rock dumps. The dry coarse magnetically-separated tailings will be spread on top of each lift at a thickness of 0.5 - 0.7 m. On top of this coarse tailings material it is suggested to add a 0.2 m thick soil cover. Growth enhancing organic material is also suggested to be added to the soil.

At another site the reclamation measures to be taken after closure are part of the permit by the authorities. These measures are different for distinct areas and comprise landscaping and tree-planting. However, due to the local situation characterised by

- absence of mineralogical soil
- deficit of nutrients (mainly carbonates)
- coarse fragmentation (due to mining technique and weathering resistance)
- temperature gradient
- steep slope angles.

These measures will be difficult to realise.

Due to these difficulties the company initiated a research project with specialists (biologists, reclamation experts, forest experts, mining engineers) to develop improved and site-specific reclamation techniques. Another important goal is to achieve site-specific vegetation in order to gain a sustainable reclamation.

By testing reclamation techniques over a three year period the most appropriate methods were selected. After six years of observing the vegetation progress, it is clear that sustainability of the measures is possible. Hence, the company now has the know-how to apply reclamation in the future with a high potential for success and in an economic manner. The observed and documented effects of progressive re-cultivation of the waste-rock dumps are:

- improvement of water balance (percolation and surface drainage rate)
- improvements of visual impact
- increased habitats for flora and fauna
- improvement of bio-diversity in the area.

The methods developed are also planned to be used for the areas currently in operation.

Long-term supervision for the waste-rock management facility is comprised of frequent monitoring of the seepage line within the end dam.

3.1.4.5 Current emissions and consumption levels

All operators follow established monitoring programmes agreed with the competent authorities.

The operator of the Malmberget and Kiruna sites has implemented a monitoring system for the environmental effects of emissions. The programme contains descriptions of sampling procedures, analysis, and reporting for environmental control. There are instructions and procedures within the company operation system that describe sampling in detail.

Monitoring is carried out according to the following minimum protocol:

- discharge control in one sampling point at least ten times a year. The analysis includes pH, carbonate nitrate, phosphorus, hydrocarbons and metals
- recipient control is based on two sampling points and in one reference location (for background level) at least six times a year. The analysis parameters include pH, carbonate, and phosphorus
- recipient- and surroundings investigations of the recipient environment are carried out every three to five years. The investigations consist primarily of sedimentological and biological evaluations
- evaluation of flooding overflow water from the clarification pond takes place continuously.

[49, Iron group, 2002]

3.1.4.5.1 Management of water and reagents

At **Kiruna** the total water intake into the mineral processing plant was 61 Mm^3 in 2001. Of this 3 Mm^3 was captured surface run-off, 9 Mm^3 mine water and the rest, 49 Mm^3, was water re-used from the clarification pond. For the 23 million tonnes of ore processed in that year, the process uses 2.6 m^3/tonne of ore, of which 80 % are re-circulated from the pond [51, Iron group, 2002].

In the flotation at Kiruna the following amounts of reagents are consumed in a year:

- collector: fatty acid, 290 tonnes
- depressant: sodium silicate, 1500 tonnes containing 94 tonnes Na and 194 tonnes Si
- conditioner: sodium hydroxide, 60 tonnes containing 35 tonnes Na.

The fatty acid, coming from the flotation process, which goes to the tailings corresponds to 250 t/yr (86 % of total consumption), of which approx. 63 % are methylic carbon and 27 % carboxylic carbon. The fatty acids are attached to the mineral phases and are transported to the tailing pond where they sediment and decay. The complete aerobic decay can be described by the formulas below:

$$CH_2\text{-} + 2\ O_2(g) + 2H^+ = CO_2(g) + 2\ H_2O$$

$$COOH + \tfrac{1}{2}\ O_2(g) + H^+ = CO_2(g) + H_2O$$

There is no collection of run-off water/seepage from the waste-rock facilities except for a drainage ditch around parts of the dumps. In these two cases, the seepage flows naturally into the tailings ponds.

At the **Erzberg** operation the mineral processing plant uses 90 % re-circulated water from the screw-classifiers. Drainage water from the tailings ponds percolates through the waste-rock dump and drains into a stream that flows under the dumps. No chemicals are used in the process. The tailings are inert and do not leach nor weather to any notable degree.

None of these operations have completed water balances. At Kiruna, however, as part of a groundwater investigation to estimate sources for contaminants to a lake, the drainage from the waste-rocks into this lake was calculated to be approximately 1.13 Mm^3/yr.

3.1.4.5.2 Emissions to air

The most severe dust problems at the waste-rock dumps occur on dry days from the crushing, transport and dumping of the waste-rock. The haul roads are then watered to reduce this problem and dumping facing populated areas ceases during windy or dry days. At one site, progressive reclamation minimises the open waste-rock dump area and thereby also the possible dust emissions.

Ponds in operation at **Erzberg** are kept water covered or water saturated. This is possible due to the alpine weather conditions with:

- high precipitation rate of about 1200 mm/yr
- short summer period
- protection by nearby mountains against wind.

At **Kiruna** and **Malmberget** sampling of airborne particles is performed continuously at several locations around the three mining operations and within the residential areas. During the winter, the snow is collected at the sampling points and analysed for particles.

Testing of air imissions the last few years at the three sites, indicates that the solid particles measured so far have been less than 220 g/(100 m^2 x 30 days) for Kiruna, 18 - 220 for Malmberget, and <200 for Svappavarra residential area. The solid particles trapped in these tests are primarily from other parts of the mining facility and not from the tailings dams. Snow samples are collected during the winter at several collection points. These samples are analysed for airborne particle distribution and reported yearly.

3.1.4.5.3 Emissions to water

At **Erzberg** water discharges are monitored. No negative effects on the downstream water quality have been detected nor have any threshold values been exceeded.

For the other sites the emission to water is variable for each of the large sites. The following sections give a description for each of the sites. Groundwater samples have been collected in order to evaluate transport of nitrate from the coarse tailings facilities.

At **Kiruna** approximately 9 Mm^3 is discharged yearly from the clarification pond to the surface water system. The yearly average discharge rate is approximately 16.8 m^3/min. The discharge rate over the year is highly variable, and follows the natural drainage cycle, however, with some time delays. The total amount of nitrate and phosphate discharged in 2001 was 116 tonnes and 251 kg, respectively, which is in the range of the discharge over the last 10 years. Discharge concentrations for nitrate are approximately 13 mg/l, and for phosphate, the discharge concentrations are approximately 0.03 mg/l (average concentrations for the year). Nitrate comes from the un-detonated explosives and the phosphate comes from the ore.

The following table shows a complete analysis of the discharge of this site.

Parameter	Conc.	Units
Al	10.7	µg/l
Aliphatics	<0.1	mg/l
Aromatics	<0.2	mg/l
As	0.59	µg/l
Ba	31.35	µg/l
Ca	160.7	mg/l
Cd	0.009	µg/l
Cl	123.8	mg/l
Co	0.18	µg/l
Cr	0.049	µg/l
Cu	1.79	µg/l
F	1.71	mg/l
Fe	0.049	mg/l
HCO_3	1.10	mmol
Hg	<0.002	µg/l
K	35.1	mg/l
Conductivity	139.7	mS/m
Mg	20.05	mg/l
Mn	32.36	µg/l
Mo	53.94	mg/l
Na	80.37	mg/l
Ni	0.92	µg/l
NO_3-N	11.33	mg/l
P	25.54	µg/l
Pb	0.0429	µg/l
pH	8.03	
S	141.1	mg/l
Si	3.684	mg/l
SO_4	431.2	mg/l
Sr	551.1	µg/l
Susp. Solid	3.14	mg/l
Tot-N	12.77	mg/l
Tot-P	0.0274	mg/l
Turbidity	1.871	FNU
Zn	0.924	µg/l

Table 3.47: Average concentrations of an iron ore tailings facility discharge to surface waters for 2001

From the **Svappavarra** facilities there is normally no or only marginal direct water discharge of process water to the recipient water system except for leakage through the dams. For the year 2000, approximately 130000 m^3 water was reported discharged during the period from May 23 to June 14, due to an unusually high water level in the clarification pond. Four sampling points are frequently sampled for water quality in connection with the tailings facility.

Water quality in the tailings ponds complies with Swedish and European water quality standards. Water from the tailings ponds discharges into the clarification ponds. Excess water from the clarification pond is used either as process water or for transport of the tailings to the tailings dams. Excess water from this cycle is discharged to the river system according to the discharge permits. In 2000, approximately 80 % of the excess water entering the clarification pond was re-used in to the processing plant, while 20 % was discharged. The amount discharged is 16.7 m^3/min (yearly average). The water quality discharged to the river systems is classified according to the Swedish Environmental Protection Agency as low concentration waters for all three facilities at Malmberget and Kiruna.

Approximately 6168 m^3 water was discharged from the Malmberget facility into the river. The discharge water and the recipient water were monitored and total mass of constituents discharged are estimated on a yearly basis. The processing water constitutes approximately 2 % of the total flow in the river.

At one of the sites a comprehensive groundwater investigation was performed to evaluate contaminant transport from the waste-rock facility to a nearby lake. Four monitoring wells were installed to depths of 2.5 - 3 m and sampled several times during the summer. The study indicated that there are only minor amounts of constituents transported from the waste-rock facility via the groundwater due to the high acid-buffering capacity of the waste-rock and the sorption capability of the aquifer.

Erzberg has direct discharge of drainage from the waste-rock dumps. After 30 years of monitoring of the surface water, no adverse effects on the surface water quality have been detected.

3.1.4.5.4 Soil contamination

At the **Kiruna** and **Malmberget** sites soil sampling is performed on a regular basis (approximately every five years). This is designed to monitor any contamination originating from atmospheric deposition. The investigation includes analysis/evaluation of ground-growing moss near (at various distances and in various directions) the mine facilities. The investigations focus on metal concentrations. The results of this investigation are compared with regional investigations performed by the competent authorities.

A water balance calculation has been performed for the tailings dams system, including:

- direct precipitation
- surface run-off
- process water discharge
- pump back process water
- evaporation
- discharge to the river system
- groundwater recharge and seepage through the dykes.

Based on this balance the estimated flow into the groundwater from the tailings pond/dam system is 2 m^3/min. However, there is a large uncertainty behind this number since several parameters cannot be measured but must be estimated.

Groundwater studies to evaluate the effect of the groundwater recharge from the TMF have not been performed. However, tailings/clarification pond water quality is monitored regularly, and is considered to have low concentrations. Groundwater contamination from the tailings dam system is unlikely to occur.

There has been no investigation carried out to directly evaluate the possibility of contamination of soil from the waste-rocks facilities. The leaching from these dumps is minor except primarily for nitrate and smaller amounts of sulphate. It is considered not necessary to investigate soil contamination from the waste-rock facility other than airborne particle monitoring and the vegetation investigations updated every five years.

3.1.4.5.5 Energy consumption

One site reported a unit diesel consumption for the haulage of waste-rock: 0.18 litre/tonne (average 2001).

3.1.5 Manganese

In this Section only some information about the Hungarian Úrkút mine is provided.

3.1.5.1 Mineralogy and mining techniques

Pyrolusite (MnO_2) is the most common manganese mineral and is an important ore. The mining term "wad" is used to indicate ores that are a mixture of several manganese oxides, such as pyrolusite, psilomelane and others that are difficult to distinguish. Pyrolusite is an oxidation product of weathered manganese minerals and also forms from stagnant shallow marine and fresh water bog and swamp deposits. Minerals such as rhodochrosite, rhodonite and hausmannite are often replaced by pyrolusite [37, Mineralgallery, 2002].

3.1.5.2 Tailings management

From the several manganese occurrences in Hungary one mine operates at present. This is in Úrkút, where mining started in 1917. The open pit was in operation until 1930, but since 1935 the ore has been mined underground. The mining method is room and pillar stoping combined with sublevel caving.

Until the 1970s the oxide manganese ore was treated in a mineral processing plant. The Mn-rich mud (12 % Mn and 17 % Fe) has long been discarded near the mine (2.5 million tonnes). Presently, the ore is only crushed to less than 10 mm, and sold directly to a single end-user, the Dunaferr Steel Mills in Dunaújváros. No tailings are generated.

The small amounts of waste-rock produces are used to fill the nearby decommissioned open pit.

3.1.6 Precious metals (gold and silver)

The following list shows the current gold mining operation in Europe.

Site	Country
Baia Mare	Romania
Bergama-Ovacik	Turkey
Boliden, Bjoerkdal	Sweden
Orivesi	Finland
Río Narcea, Filón Sur	Spain
Salsigne	France
Sardinia	Italy

Table 3.48: List of current European gold producers known/reported to date

Of the sites listed in the table above Orivesi, Río Narcea, Boliden and Bergama-Ovacik provided information for this section.

3.1.6.1 Mineralogy and mining techniques

Gold and silver are very different in the way deposits occur. Silver is mined entirely as a by-product of base metal or gold mineralisations and is therefore not specifically mentioned in this Section. Gold occurs either as free gold, or as sulphide-related gold.

Various geological settings and mineralogical characteristics are represented in the precious metals sites:

- complex sulphide ores where Cu, Zn and Pb are complementary or even the main value minerals (Boliden)
- sulphide mineralisation comprising pyrite, arsenopyrite, galena and sphalerite where the contained gold is submicroscopic (<1 µm), finely disseminated in the pyrite and arsenopyrite lattices (refractory gold) (Olympias Gold)
- low sulphidation epithermal quartz and breccia veins in andestic host rock (Ovacik Gold Mine)
- strongly altered volcanics: quartz, sericite and andalusite rich rocks or schists (Orivesi)
- native gold with copper sulphides in skarn and brecha jasperizadas (Río Narcea)
- gossan (Filón Sur).

The differing mineralogies require different mining and mineral processing techniques to obtain optimum gold recovery. Underground (with and without backfilling) and open pit mining are applied. The open pits are, in two cases, planned to become underground mines in time. There are several examples where gold is extracted from a tailings stream from a base metal mineral processing plant (i.e Boliden) or from old waste-rock dumps (i.e. Filón Sur) and tailings ponds (i.e. Baia Mare).

3.1.6.2 Mineral processing

Various mineral processing techniques are used, mainly due to their different suitability for different mineralogy. Depending on how the gold occurs in the ore it may be necessary to use different methods to liberate the gold so that it can be extracted. The gold can, in many cases, be recovered in the copper concentrate and separated from the copper in the subsequent smelting process. Native gold can be gravimetrically concentrated and recovered. Gold in its oxide form can be directly leached with cyanide. Refractory gold may require oxidation, e.g. bio-oxidation, in order to liberate the gold and make it accessible for CN leaching.

3.1.6.2.1 Comminution

Common to all operations is that the ore needs to be crushed and ground before the gold can be liberated. In some cases this is done in the previous recovery of base metals. Tank leaching requires a finer grain size in order to allow for relatively short residence times in the leaching tanks. Heap leaching allows for a coarser grain size as the leaching time is much longer. In heap leaching a relatively coarse grain size (even conglomeration may be necessary) is desired to allow for oxygen inflow and to secure a sufficiently high permeability of the heaped material.

The type of equipment used in comminution are various types of crushers, and various types of mills such as dry semi-autogenous mills, ball mills, autogenous mills, etc.

The **Orivesi** mine uses the following equipment in the comminution process:

- crushing in three stages with a jaw crusher, a gyratory crusher and a cone crusher
- grinding in two stages with a rod mill (3.2 X 4.5) and a ball mill (3.2 X 4.5)
- classification with hydro cyclones.

[59, Himmi, 2002]

The **Boliden** comminution circuit is described in Section 3.1.2.2.1. Both grinding circuits are equipped with Reichert cones, spirals and a shaking table for gravity separation of gold.

For the tank leaching operations it is commonly required to reach a grain size of 50 - 80 % <45 µm or in some cases, if the gold is extremely finely disseminated, even below 40 µm to achieve optimum liberation.
[50, Au group, 2002]

3.1.6.2.2 Separation

The mineral processing methods commonly used are:

- flotation, where the gold binds mainly to the copper concentrate (gold recovered from the concentrate in the smelting process)
- heavy-medium separation for lumps using drum separators and dewatering screens
- cone separators and high-intensity magnetic separators for fine material
- Reichert cones, spirals and shaking tables for the gravity separation of gold.

In the schematic figure below, an example of a mineral processing plant is given. This plant, with a relatively low throughput of 35 t/h, produces a concentrate containing 125 g Au/tonne. The leaching of some of the gold concentrate is carried out to reduce the content of impurities (Tellurium (Te) and Bismuth (Bi)). This step aims to dissolve Bi and Te away from the concentrate. The tailings from this process are led to a separated ditch in the old TMF (used during nickel mining phase). Because the water from the leaching process is acidic, lime is added to neutralise it. Bi is precipitated in these circumstances, but most of the Te remains in solution. The leaching process has been in use only when necessary, depending on the ore characteristics. There is no outlet from the ditch, thus the water evaporates and filtrates into the old tailing material. According to analysis on seepage water outside the TMF area, no significant concentrations of Te have been found. Currently the leaching process is not in operation, because the quality of the ore has changed and Bi and Te are no longer problematic.

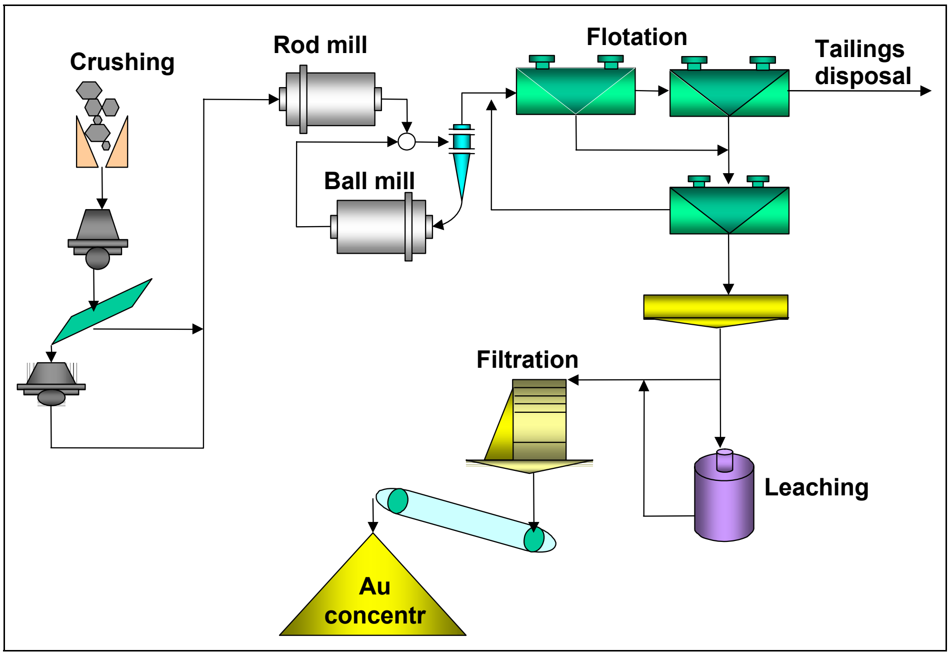


Figure 3.34: Schematic flow sheet of an example gold mineral processing circuit [59, Himmi, 2002]

Leaching of gold is carried out as follows:

- CN leaching in tanks using the Carbon-In-Pulp method (CIP) (e.g. Ovacik Gold Mine)
- CN leaching in tanks using the Carbon-In-Leach method (CIL) (e.g. Boliden and Río Narcea)
- bio-oxidation and pressure oxidation followed by CN leaching using the CIL method (all processes in closed tanks) (e.g. Olympias Gold Project)
- heap-leaching using CN solution followed by Merrill-Crowe process where the gold is precipitated on zinc powder (e.g. Filón Sur).

The leaching processes mentioned above all require further processing in order to achieve a sellable product, i.e. transfer of the gold and silver from the activated carbon into doré containing gold and silver. A complete gold tank leaching plant constitutes of the following principle stages:

- cyanide leaching (CIL-process or CIP-process)
- gold refining (eluation, electrowinning, smelting and doré production)
- cyanide destruction (e.g. oxidation)
- reagents preparation (lime and sodium cyanide).

A complete plant is schematically illustrated in the figure below. This particular plant (Boliden), was commissioned in 2001 and recovers gold and silver from the tailings stream resulting from a base metal mineral processing plant. The system is designed for a throughput of 800000 t/yr with a gold production of 850 kg/yr. The recovery is approximately 80 % of the gold. The recovery of gold increased by 50 % after the installation of gold leaching.

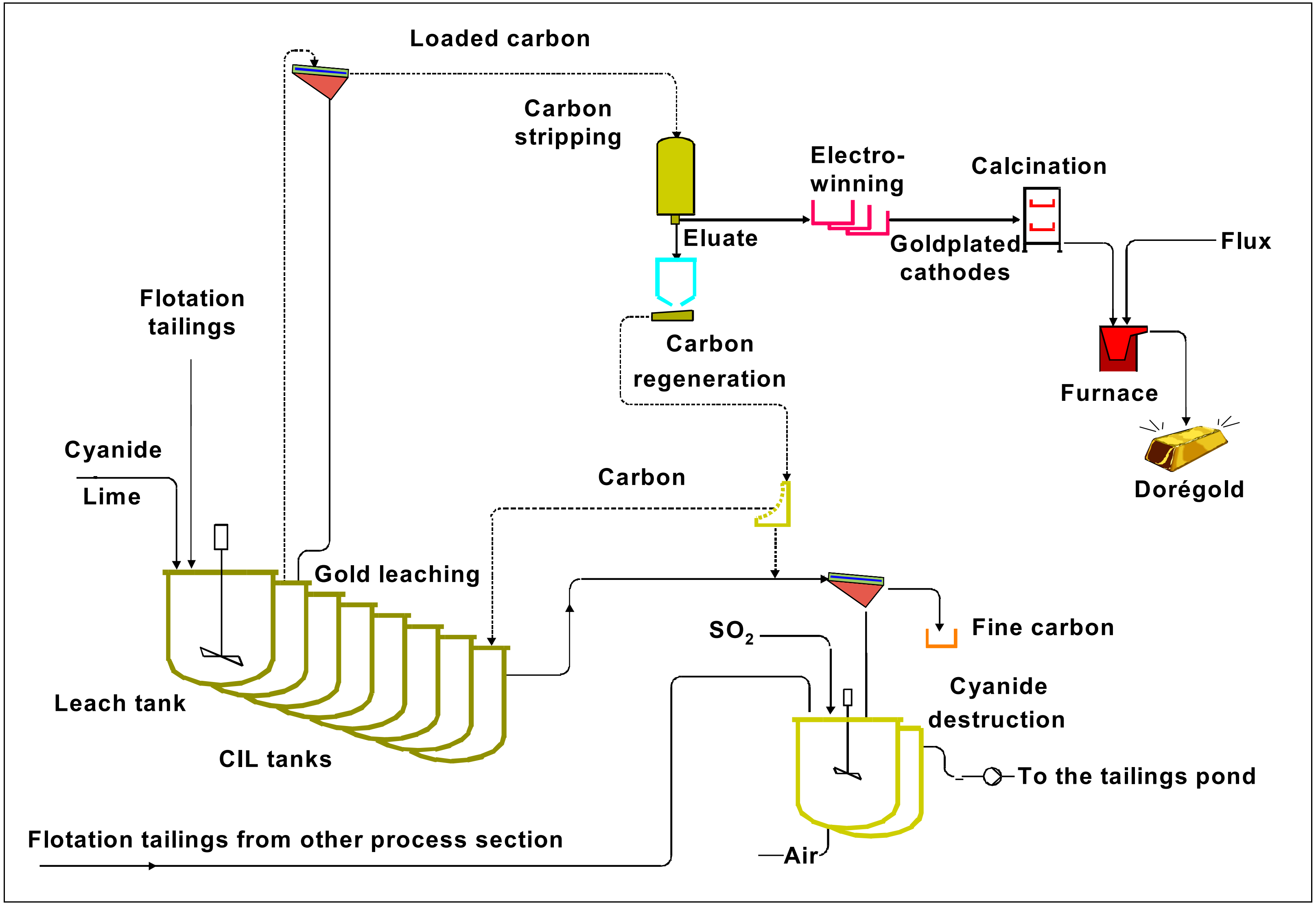


Figure 3.35: Schematic drawing of CIL process
[50, Au group, 2002]

At all sites where tank leaching is practised, the tailings slurry undergoes detoxification prior to discharge into the tailings pond.

3.1.6.3 Tailings management

3.1.6.3.1 Characteristics of tailings

The untreated tailings from gold mineral processing using cyanide contain different compounds, depending on the process used, ore type, cyanide dosage, degree of aeration, etc. The composition of tailings will also change as the ore changes [24, British Columbia CN guide, 1992].

During a CIP/CIL leaching process a small portion is lost to the mineral processing plant atmosphere by volatilisation. Some will react with whatever other cyanide consumers may be present in the ore to produce complexes such as the ferrocyanide, thiocyanate, cyanate and cuprocyanide complexes. During leaching, gold is removed from the solution by adsorption onto carbon, and some cyanide may be removed with it. The remaining unreacted cyanide, together with the products with other cyanide consumers, is discharged with the tailings. The cyanide in the tailings may be treated for cyanide removal (most European sites) or left as is for removal by natural degradation in the tailings pond (international standard). Any cyanide entering the carbon stripping circuit would either be periodically bled back into the leach circuit or destroyed during reactivation of the carbon in the carbon kiln [24, British Columbia CN guide, 1992].

The untreated tailings stream from a CIP/CIL process consists of a tailings slurry with elevated levels of cyanide, complexed metals, cyanate and thiocyanate. It may also contain arsenic and antimony, depending on the ore and mineral processing.

It is common practice to have regular control of other material characteristics (the parameters determined varies somewhat from site to site) such as, e.g.:

- grain size distribution
- solid to liquid ratio
- ARD-characteristics
- mineralogy
- trace element content.

The above-mentioned parameters are used to determine the leaching characteristics of the material which has an important influence in the operational management and suitable decommissioning methods for the tailings. For this purpose all sites using tank leaching have carefully evaluated ARD-generation characteristics for their tailings. The Boliden mineral processing plant, with 18 % sulphur and low carbonate content has to deal with potentially ARD-generating tailings [50, Au group, 2002].

At **Bergama-Ovacik** a detailed characterisation of some samples has shown that the tailings and waste-rock will not produce ARD as illustrated in the figure below.

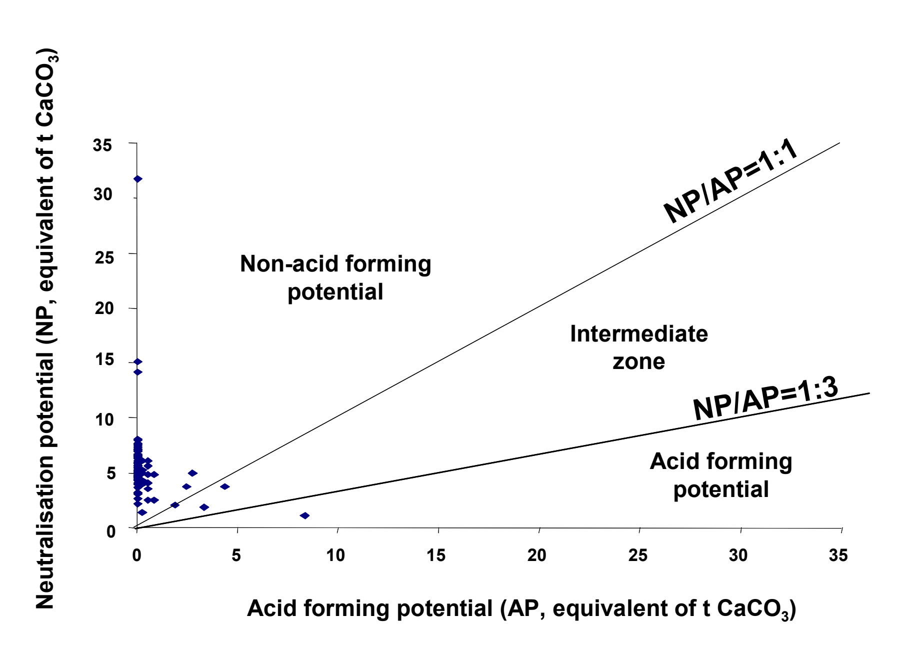


Figure 3.36: Acid forming potential vs. neutralisation potential graph of samples from Ovacik site [56, Au group, 2002]

The following table shows the average results of 99 samples.

	pH	AP*	NP*	NNP*	NP/AP*	S^{2-} %
Average of 99 samples	7.52	0.47	5.5	5.18	4.67	0.02
*:Tonnes $CaCO_3$ equivalent per 1000 tonnes AP: Acid Potential NP: Neutralisation Potential NNP: Net Neutralisation Potential						

Table 3.49: Acid production potential at Ovacık Gold Mine

The **Boliden** mining area consists of complex sulphide mineralisations. Mining in the area started in 1925 and to date approximately 30 mines have been worked in the area. The tailings in the pond consequently have varied chemical characterisations and physico-chemical properties. The characteristics of the tailings produced today are summarised in the tables below. The fine fraction after cycloning is discarded into the tailings pond and the coarse fraction extracted from the hydrocyclones is used as backfill in the underground mines.

Size	Total tailings	Hydrocyclone overflow to pond
µm	Cumulative % passing	Cumulative % passing
350	100	100
250	99.9	100
180	99.7	100
125	97.8	100
88	93.5	95.6
63	85.9	87.8
45	76.6	78.3
20	53.2	54.4
-20	0	0

Table 3.50: Particle size of tailings at Boliden mine
[50, Au group, 2002]

The tailings have the following composition before cycloning and CN leaching:

- Au: 0.85 g/t
- Ag: 24.9 g/t
- Cu: 0.10 %
- Zn: 0.40 %
- Pb: 0.13 %
- S: 17.8 %

More than 50 % of the tailing consist of particles less than 0.002 mm. The tailings slurry pumped to the tailings pond contains 20 - 25 % solids. The density, as placed in the pond, of the tailings is 1.45 t/m^3.
[50, Au group, 2002]

3.1.6.3.2 Applied management methods

At the **Filon Sur** heap leach operation, the tailings (the heap of leached material) are left in-situ and decommissioned. The heaps are built on a pad with a synthetic liner. Leachate or 'pregnant solution' is collected in a small pond before it is pumped to the plant for gold and silver precipitation. The leachate is then pumped to a conditioning pond before it is re-used in the leaching process. Very little information is available at the moment to evaluate how tailings and waste-rock management and decommissioning is done and planned, thus it will not be further described at this stage. No material characterisation is reported [57, IGME, 2002].

All other sites, using CIL or CIP to leach the gold in tanks, produce tailings in a slurry form that, after CN-destruction is applied, are pumped via pipelines to tailings ponds. The commonly used process to destroy CN is the SO_2/air process. In general this treatment results in a total CN concentration in the treated slurry stream of <1 mg/l. One site (**Bergama-Ovacik**) that measures WAD CN reports concentrations <1 mg/l.

Boliden uses the coarse fraction of the tailings as backfill in underground operations. These tailings are extracted from the tailings stream in hydrocyclones situated after the CN-destruction plant. The tailings used for backfill are also analysed for total CN (typically less than 1 mg/l).

50 % of the sites use lined tailings ponds and 50 % use unlined tailings ponds. Various dam types are used to confine the ponds.

At the **Bergama-Ovacik** gold mine, with an ore production of 0.3 million tonnes/yr, the tailings are managed in a 1.6 Mm^3 capacity pond with a 30 m high downstream rockfill embankment and clay-geo-membrane composite lining system. As described earlier the tailings are treated for cyanide destruction and heavy metal precipitation utilising oxidation with SO_2 followed by ferric sulphate treatment [56, Au group, 2002].

A conceptual drawing of the TMF is given below:

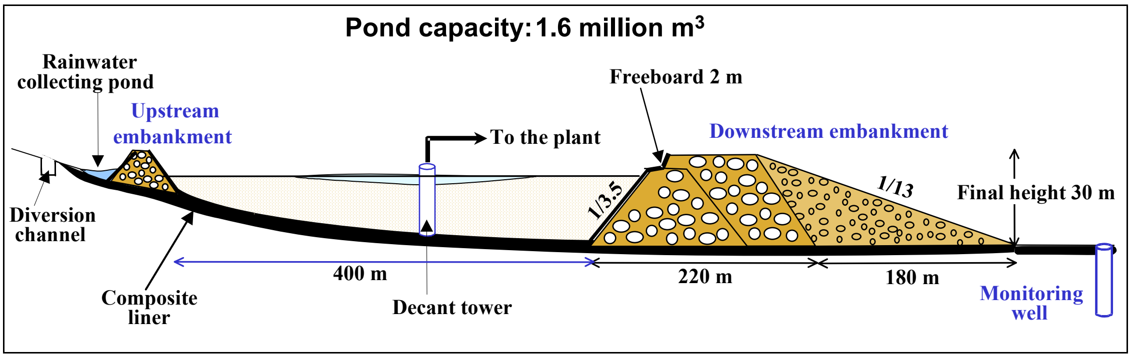


Figure 3.37: Cross-sectional drawing of Ovacik tailings pond
[56, Au group, 2002]

It should be noted that the bottom of the pond as well as the downstream face of the upstream embankment and the upstream face of the downstream embankment are lined.

The lined tailings pond is located in a valley within two hundred metres from the process units. Rock fill dam construction materials (mainly andesites) were obtained from the overburden excavation in the open pit. The region is an arid zone where evaporation plays an active role in the water deficit for the pond during the summer season. The TMF was designed as a 'zero' discharge unit where water in the pond is re-circulated during the operation of the mine. Because of the low cyanide concentration in the pond (less than 1 mg/l WAD), HCN volatilisation is negligible. The geo-technical and seismological investigations in the TMF area before and after the construction revealed the presence of a suitable setting for the rock fill embankments and the reservoir stability. The embankments were constructed as conventional dam structures.

Topsoil was scraped and stored on site for future use in site rehabilitation. During closure of the pond, tailings will be dewatered and the top will be covered with rock and soil and subsequently re-vegetated.

In selecting the TMF location, the main factors taken into consideration were:

- minimised land and soil disturbance
- proximity to the process plant
- use of overburden and waste-rock in the embankments in an efficient way to minimise the footprint
- storage of topsoil for vegetative cover upon closure
- cyanide destruction and heavy metal precipitation for tailings
- re-use of process water in the process
- zero-discharge of water from the TMF.

It was the company policy to select tailings dams of rock fill type for its increased stability and easy maintenance (as opposed to using the coarse tailings). The clay-geo-membrane composite liner system was selected to achieve an effective containment and to expedite the regulatory approval and permitting process.

From the geotechnical point of view, the dams were designed to withstand an earthquake induced horizontal acceleration of 0.6 g. During operation with the placement of the overburden and the waste-rock on the downstream slope of the main dam, the slope changed to less than 10˚, increasing the factor of safety of the dam structure to 2.23 compared to the usual 1.2 used internationally for water retention dams.

The base of the tailings pond is covered with a composite liner system of 50 cm compacted clay, overlain by a 1.5 mm thick High Density Polyethylene (HDPE) geo-membrane, 20 cm of another compacted clay and 20 cm gravel filter layer. Drainage pipes are placed in the filter layer to drain the water towards the decant. The following figure shows the set-up of the composite liner system.
[56, Au group, 2002]

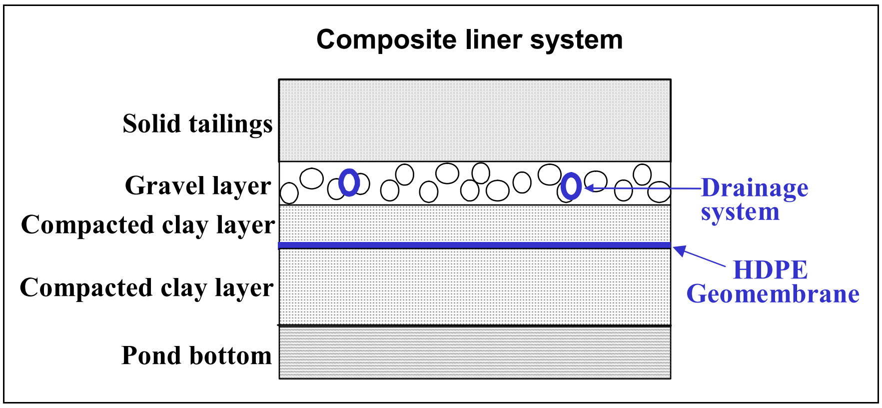


Figure 3.38: Composite liner set-up at Ovacik site
[56, Au group, 2002]

The deposition of tailings is carried out via pipelines discharging into the pond area near the downstream embankment. During the mine operation, a minimum of 2 m of freeboard is provided in the TMF design.

The TMF design includes surface run-off retention behind the upstream dam and a diversion channel for excessive flood waters (for 100-year flood conditions).

The **Boliden** base metal mineral processing plant received a total of 1.58 million tonnes of ore from five different mines during 2001 in order to produce copper, lead and zinc concentrates. Coarse gold is also extracted using shaking tables. Depending on the ore type part of the tailings produced (approx. 50 %) are further processed in the gold leaching plant. The gold leaching plant generated 0.8 million tonnes of tailings in 2001.

Of the five mines, four are underground mines and one is an open pit. The underground mines use the coarse fraction (>125 µm) of the tailings for backfilling. The amount of tailings used for backfilling depends the production level in the mines and the production status. During preparation work in the mines a significant amount of waste-rock is produced and used for backfilling. It should be noted that approx. 33 % of the ore comes from an open pit, where no backfilling is done during operation. Subtracting this amount of ore the percentage of backfilling is close to 50 %.

The tailings that are not used for backfilling are sent to the tailing pond that has been used since the 1950's. Previously the area had a lake. The amount of tailings in the pond is currently approx. 16 Mm^3 and covers a surface area of 260 ha. According to the existing operation levels, the existing tailing pond can be used for four to five more years. The tailings are pumped to the pond and discharged at various outlet points in order to allow for uniform filling of the pond.

The tailings are confined within the pond by five dams. Another dam is also constructed downstream of the tailings pond to cut off the lakes natural outflow and to create an additional clarification volume. The pond area is currently 260 ha and after a dam raise in the summer of 2002 the area will be 280 ha.

The tailings pond catchment area is 8 km^2. The inflow of surface run-off has been estimated to be 1 Mm^3 during a dry year and 3 Mm^3 during a normal year. The pond receives approximately 4.5 Mm^3/yr of process water from the mineral processing plant.

The tailings pond is approximately 3 km from the concentrating plant. Tailings are pumped via two separate pipelines, one to the north and one to the south of the pond. Downstream of the pond, slaked lime is added to the discharged water to increase the pH to 10 - 11. All water from the pond is discharged to waterways downstream. No re-circulation of process water is done at the moment.

Water sampling for monitoring water quality is done on a regular basis according to a control programme. Sampling is done both upstream and downstream of the tailings pond, as well as around the industrial area. Sampling consists of stream analysis and groundwater samples.

The dams were constructed initially in 1979 to +216.2 m as a centreline type dam with a vertical impervious core and support fills on both upstream and downstream sides of the dam. In 1995 the dam was raised to +220 m as a downstream dam (see the figure). A final raise is ongoing to +225 m to be finalised in 2002. A discharge channel constructed in natural ground will replace the current decant tower.
[50, Au group, 2002]

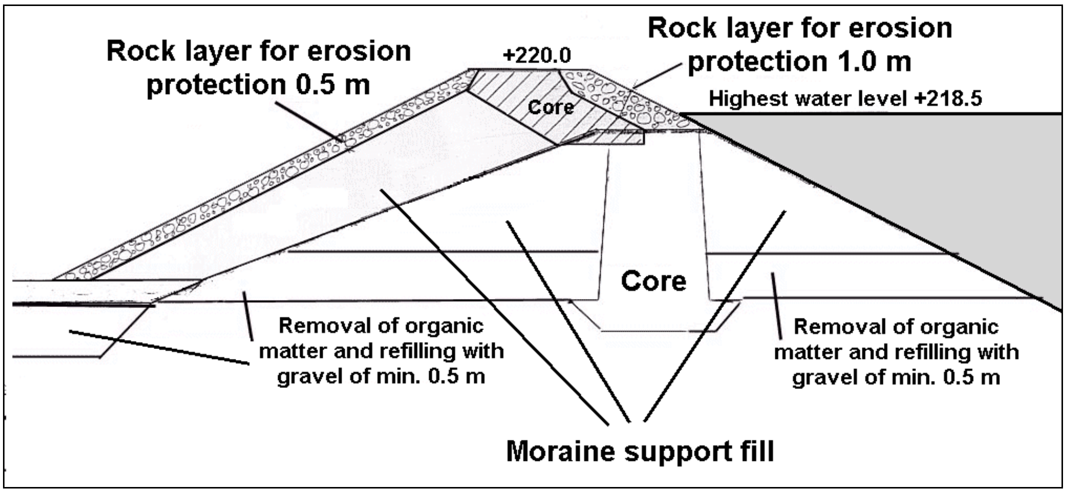


Figure 3.39: Cross-sectional view of dam at Boliden site
[50, Au group, 2002]

Any drainage through and under the dams is collected in a collection ditch and led to the clarification pond. Drainage through and under the other dams is back-pumped into the pond.
[50, Au group, 2002]

The tailings area of the **Orivesi** mine consists of two tailing ponds. The tailings from the process are pumped into the first pond (37 ha), where the solids settle and the clarified water is led forward from the other end of the pond. The second pond (14 ha) is for storing clarified water. Water is re-used in the process and only the excess is led to the river system. The starter dams have been made of moraine. The tailings are spigotted to one side of the first pond and the clarified water is led forward from the other side.

The dams of the clarification pond are made of moraine and lined with broken rock and coarse gravel to prevent erosion. The tailings management area was designed in the beginning of the 1970`s and no closure or after-care plans was taken into account at that time. The tailings pond is, however, used only occasionally when the tailings are not deposited into the old mined-out underground nickel mine.
[59, Himmi, 2002]

A schematic figure of the system is given below.

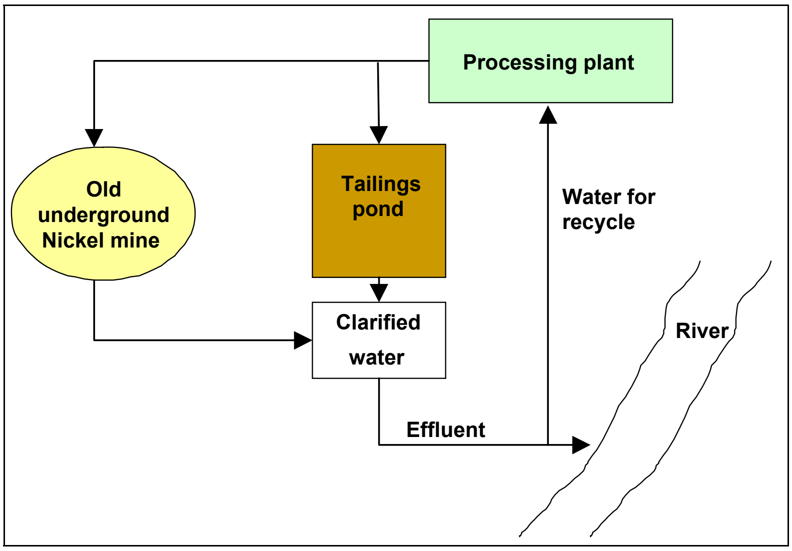


Figure 3.40: Schematic illustration of tailings and effluent treatment at Orivesi mine [59, Himmi, 2002]

The base dam of the tailings pond has been constructed of moraine and there is drainage collection outside the dam to collect seepage water. The necessary raises of the dams are done using moraine for the core and the tailings material as supporting fill.

The TMF was originally constructed for a nickel mining operation. After 20 years of operation the nickel mine was closed, but the mill has been used since to treat gold ore from Orivesi mine located 85 km from the plant. The distance from the mill to the tailings management area is about 500 m. The distance from the tailings area to the river is about 600 m. The surrounding area is not used for agriculture, but the nearest house is only 200 m from the tailings area. The operator does not consider dusting from the tailings management area a problem, because the material on the surface of the area has formed a hard layer. The drainage water is collected by a ditch system and is led directly to a river, because, according to the operator, it does not contain 'significant' contamination.
[59, Himmi, 2002]

At **Río Narcea**, the tailings are deposited into a lined tailings pond after CN-destruction. The present volume of the deposit is 2.4 Mm^3 and the pond is continuously raised according to requirements. The dams are built out of compacted clay with a supporting fill of waste-rock. The pond has an impermeable composite liner system composed of compacted clay and a 1 mm HDPE liner. The pond is surrounded by channels for the diversion of surface run-off. Collected surface run-off is diverted into three sedimentation ponds for clarification before discharge [58, IGME, 2002].

3.1.6.3.3 Safety of the TMF and accident prevention

At the **Bergama-Ovacik** site a full risk assessment has been done, stability calculations have been performed and the design has been carried out by external experts. As described above, the design aims at assuring stability for seismic load, static stability, flood events and any other relevant parameter detected in the risk assessment.

The tailings facility is under daily surveillance for environmental monitoring and structural integrity. The site is routinely audited as per the mother company's environmental policies and an Ovacik Gold Mine Environmental Management System report prepared. The mine will be subject to an annual internal environmental audit programme using the company's assessment process to assess the effectiveness of the environmental management systems and the level of environmental performance at the operation. An external audit by an independent expertise group was conducted during the trial operations.

Similarly, management plans on other issues such as health and safety, tailings storage, mine closure and rehabilitation, emergency action and community relations are in place.
[56, Au group, 2002]

The tailings pond at the **Boliden** site is managed according to an OSM manual (see Section 4.2.3.1) designed according to guidelines for dam safety, developed by the Swedish Association for Hydropower Operators (RIDAS). In 1997, when Boliden initiated a dam safety project for tailings dams it was decided to use RIDAS as a guideline where applicable to tailings dams. Changes would then be made when necessary, rather than developing new guidelines for tailings dams. Other mining companies have followed the same route [50, Au group, 2002].

At the **Orivesi** mine, the tailings facility is inspected daily as part of the operational routines at the site. No formal risk assessment has been done. However, the dam undergoes annual audits by independent experts and every fifth year it is audited by the competent authorities. The comments are recorded in the dam safety document, which is a compulsory document for all similar types of tailings management areas since the mid 1980's.

In the construction phase of the tailings facility the soil characteristics were investigated. The system has been constructed in such a way, that the surface of water in the tailings area can be kept in balance and the excess of water from rainfalls etc. can be removed in a controlled manner. There are no instruments installed to monitor the phreatic level in the dam body. A documented emergency plan does not exist. It is not clear if the environmental impact of the backfilling of tailings has been assessed.
[59, Himmi, 2002]

At **Río Narcea**, the dams are controlled using piezometers and inclinometers. The tailings pond undergoes regular audits by external experts. A risk assessment has been performed [58, IGME, 2002].

3.1.6.3.4 Closure and after-care

At **Bergama-Ovacik** mine rehabilitation will be done concurrent with the operation to the extent practicable. Topsoil removed during construction is retained on site for subsequent rehabilitation. A conceptual mine closure and rehabilitation plan has been prepared and will be reviewed yearly during operation. Upon closure of the mine, the tailings pond area will first be covered by rock, gravel, clay and topsoil and then replanted with trees. Prior to the operation of the mine, a financial assurance bond was submitted to the competent authority to secure rehabilitation and closure in accordance with the operation permit protocol [56, Au group, 2002].

At **Boliden**, a water cover solution has been chosen for the closure of the tailings pond. The dams around the tailings pond have been raised to their final height. The pond will be filled up in five years time after which it will be water covered according to existing permits. Apart from the water cover of the open tailings surface, the dams will be re-sloped to 1:3, covered and re-vegetated, long-term stable outlets will be arranged and breakwaters will be constructed in shallow water depths to avoid re-suspension of tailings by wave action. All dams will receive additional long-term stable erosion protection. The back-pumping of seepage water will be carried out until the water quality has improved sufficiently to allow its direct discharge. Water treatment will be conducted by straight liming at the outlet during the same time period, which is expected to last <8 years.

Water cover as a decommissioning method has been used at various sites within Boliden. The water cover established at Stekenjokk in 1991 has been extensively monitored with subsequent follow-ups in detail, showing very good results.

An alternative decommissioning technique currently being evaluated is wetland establishment. This would allow for a higher sand level in the pond (better use of existing pond), less water stored in the pond (less risk) and a self generating organic oxygen consuming cover the top of the tailings.

Boliden is also trying out an alternative method called 'water saturation' or 'raised groundwater level' which basically is applicable where the natural groundwater level in the tailings is very shallow. By applying a simple soil cover the groundwater level can then be raised to permanently cover the tailings and eliminating sulphide oxidation (see Section 4.2.4).
[50, Au group, 2002]

At **Orivesi** a plan for closure and after-care has been developed recently, concerning the mine site and the industrial area. Only a draft plan has been made concerning the tailings management area. The main idea is to cover old tailings material from the nickel process with the tailing material from the gold process. A total of EUR 0.6 million has been reserved for closure [59, Himmi, 2002].

At **Río Narcea** the tailings pond will be dewatered and covered using soil that has been temporarily stockpiled at the edge of the pond. Re-vegetation will be carried out and the area will be returned to original land use (pasture). Pore water, with a WAD CN concentrations <1 mg/l, will be collected through the installed underdrains in the pond and analysed before discharge.

3.1.6.4 Waste-rock management

At the **Bergama-Ovacik** gold mine the overburden and the waste-rock are andesites which are currently used as rock fill material on the downstream side of the TMF embankment. The waste-rock source at later stages of the mine will be from underground workings (galleries, drifts etc.) and these materials will be used as backfill in the underground voids.

ARD potential and geotechnical property tests were conducted on the waste-rock. These tests revealed that the waste-rock does not have ARD potential and is has adequate properties for use in construction of the rock fill dam and retaining structures. The non-ARD potential of the waste-rock allowed the operator to use this material in the retaining structure of the TMF while providing an optimum use of the storage area requirement at the facility. The waste-rock is transported from the open pit area by trucks and placed on the downstream slope of the TMF embankment and spread evenly and compacted with clay material.

Because of the inert nature of the waste-rock, there is no environmental risk associated with the waste-rock dumping unit at the Ovacik Gold Mine. (according to a probabilistic risk assessment carried out by an independent consultant).
[56, Au group, 2002]

At **Boliden**, the waste-rock is generated at the five mines supplying the mineral processing plant with ore. As these mines are mainly base metal mines, this waste-rock management is described under the section for base metals (see Section 3.1.2.4) [50, Au group, 2002].

At **Filón Sur**, 0.1 million tonnes/yr of waste-rock are generated. There is no information on how this is handled nor any information on the characteristics of this material [57, IGME, 2002].

Orivesi uses all its waste-rock as backfill in the underground operations. No waste-rock is hoisted to the surface [59, Himmi, 2002].
At **Río Narcea**, six million tonnes of waste-rock were produced in 2001. Approximately 20 million tonnes of waste-rock is kept in waste-rock dumps at the site. Topsoil is separately stored so that it can be used in the reclamation of the site. Waste-rock from mine production will be backfilled in mined out open pits as production progressively moves along. The initial waste-

rock dump, from the initial open pit, will be decommissioned in-situ. The waste-rock consists mainly of silicates (granite and sandstone) and various carbonates (limestone) [58, IGME, 2002].

3.1.6.5 Current emissions and consumption levels

In addition to the routine occupational health and safety monitoring, an environmental monitoring programme has been established at the **Bergama-Ovacik** mine. An official monitoring committee assigned by the Turkish Government carries out verification sampling. Environmental monitoring data are compiled in monthly reports and submitted to the competent authorities. These are also opened up to the community through various means including the national press and other public reports. Environmental sampling locations are presented in the figure below. Data collected for the periodical environmental monitoring are the following:

- dust, noise and vibration levels
- WAD CN in tailings water leaving the detoxification unit and at the water intake from the tailings pond
- heavy metals (As, Sb, Cd, Hg, Cu, Pb, Zn, Cr) in the tailings water
- indicator water quality, including WAD CN at the six groundwater monitoring wells located downgradient of the tailings dam
- HCN measurements at various locations at the mine, including the tailings pond area.

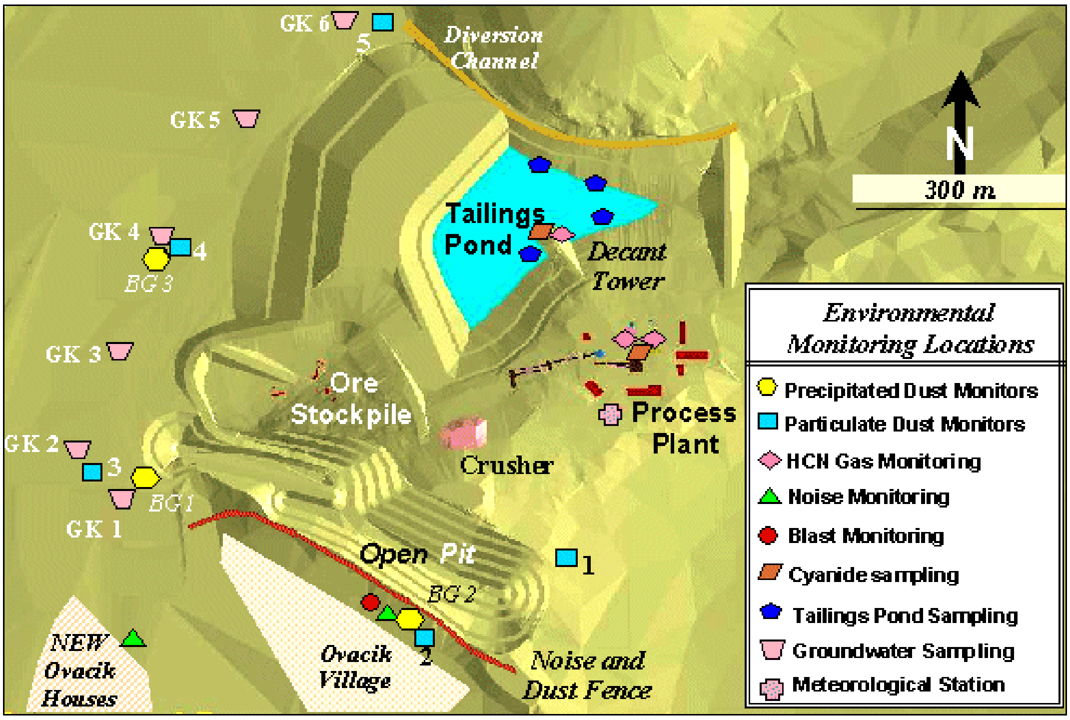


Figure 3.41: Environmental monitoring locations at Ovacik site
[50, Au group, 2002]

The control programme followed at the **Boliden** mineral processing plant consists of:

- surface (numerous monitoring points with varying frequency) and groundwater monitoring (17 monitoring wells with monthly sampling)
- emissions to air (dust and gases)
- CN destruction monitoring (at various points. The discharge from the CN-destruction plant to the tailings pond is sampled six times per day and the discharge from the tailings pond daily)
- noise and vibration monitoring
- recipient investigations.

Environmental monitoring data are compiled in monthly reports and submitted to the regulatory authorities and shared with the community through various means including a local reference group that meets regularly at the site to discuss any issues of concern and for general information.

3.1.6.5.1 Management of water and reagents

The design criteria and management system for **Bergama-Ovacik** tailings pond is set for 'zero' release of water to the receiving environmental media. This is possible as the operation is a net consumer of water (due to the arid climate conditions) and re-uses all the water from the tailings pond in the process. Mean annual rainfall and evaporation of the area are 728 and 2313 mm, respectively (i.e. there is a negative water balance).
The catchment area at the point of the upgradient dam is approximately 0.6 km^2. Maximum possible flood discharge is calculated as 24.6 m^3/s for the first hour of an extreme rainfall event. In the event of such extreme rainfall, the potential floodwaters coming from the catchment area will be stored in the run-off water pond behind the upstream embankment. The accumulated water will be pumped to the tailings pond or the excess water taken directly into the diversion channel, which is constructed along the north side of the pond.

The water consumption at the **Boliden** mineral processing plant is approximately 4.5 Mm^3/yr or 2.9 m^3/tonne of ore. The water is obtained from a lake 2 km north of the mineral processing plant. Some re-circulated water is used in the mill for cleaning and cycloning. Of the total amount of water used in the mineral processing plant about 10.5 % are re-used.

Due to oxidation of thiosalts and depending on the time of the year, the water contained within the pond is of low pH and contains elevated metal concentrations. The discharge from the tailings pond is, therefore, treated in a straight liming installation installed at the outlet of the tailings pond. A small sedimentation pond has been constructed to collect the precipitates. The pond is dredged biannually and the precipitates are deposited within the tailings pond. The flow of the discharged water is measured daily. Discharged water volume from the tailings pond is presented in table below.

Year	1997	1998	1999	2000	2001
Flow (l/s)	254	238	186	218	352
Volume (Mm^3)	8.0	7.5	5.9	6.9	11.11

Table 3.51: Discharged water from Boliden TMF from 1997 - 2001

The following figure illustrates the seasonal variations of the water quality in the tailings pond system and the recipient water body (year 2001 data).

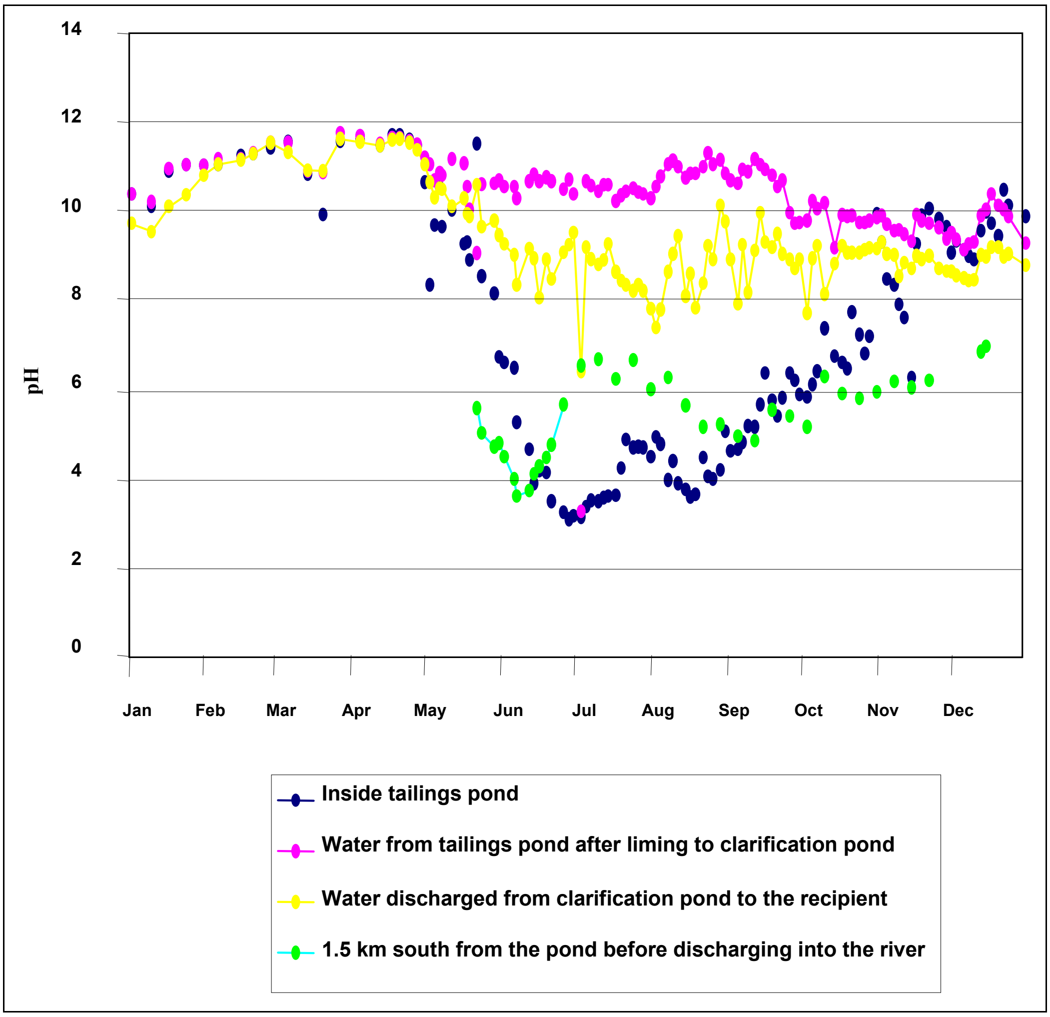


Figure 3.42: Seasonal variations of water quality in the tailings pond and the recipient at Boliden in 2001
[50, Au group, 2002]

The sampling points in the figure above are at four different sampling points: inside the tailing pond, discharge water from the pond after liming to the clarification pond, discharged water from the clarification pond to the recipient and 1.5 km south of the pond before discharging to the river. The pH in the tailings pond during winter seasons is 10 - 11. During spring and summer the pH drops to about 3.5 due to the oxidation of thiosalts and the discharged water is therefore limed to pH 9-11 to neutralise the acid effects as described above.

During 2002, the downstream dam will be raised, the discharge system will be rebuilt and a new system for flow monitoring will be installed. The discharge from the tailings pond will be rearranged from a decant tower to an overflow channel in natural ground. A back-up system for discharging water in the tailings pond is in place and will be raised.

A water balance for the Boliden mineral processing plant, the tailings pond and the surroundings is illustrated in the figure below for a year with average precipitation.

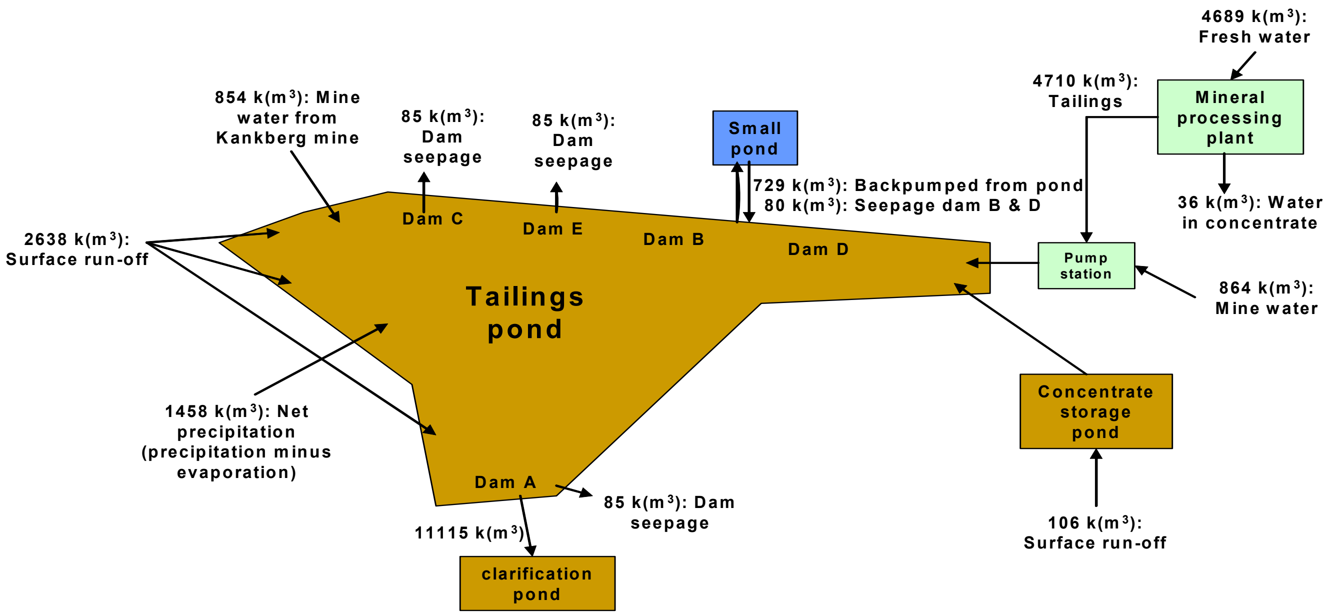


Figure 3.43: Water balance at Boliden site
[50, Au group, 2002]

Within the industrial area there is an old open pit and a shaft under the mineral processing plant. Drained water is pumped to the tailings pond to be treated before discharging to the recipient. Drained water from the tailings pond is pumped back to the pond continuously. A small lake north of the tailings pond is continuously pumped in order to maintain a lower water level than the surroundings and, thereby, to capture any possible seepage and pump it back to the tailings pond. Data such as snow depth, rain and groundwater level are collected for the water balance. The data of water in the concentrates are also used for the water balance. The system is used for monitoring the amount of water in the system.

Discharge from the Boliden tailings pond only occurs through the outlet at dam A. The seepage that occurs through dams B, C, D and E is back-pumped into the pond from the small collection pond (see Figure 3.43).

It should be noted that at the Boliden TMF, dilution through precipitation and surface run-off adds (besides the natural decomposition of CN compounds) to the decreased CN concentration.

Fresh water consumption is monitored continuously in the process system in the mineral processing plant.

At the Boliden gold leaching plant sodium cyanide is used for collecting precious metals. Sulphur dioxide is used in the destruction of cyanide and lime is used for pH-regulation, before discharging to the tailings pond. During 2001 the consumption of chemicals used in the recovery of gold (at a throughput of 0.8 million tonnes) was as follows:

lime (gold and base metals):	5000 tonnes
sulphur dioxide:	1260 tonnes
sodium cyanide:	450 tonnes

The CN that is discharged into the tailings pond undergoes further natural decomposition in the pond system. This is the reason for further decreases in CN concentrations in the tailings pond and, if discharge occurs, in the discharge from the tailings pond. Values from Ovacik site, where there is no discharge to the recipient, shows that the average WAD CN concentration in the discharge to the pond is 0.33 mg/l while the concentration in the pond itself is 0.19 mg/l. At the Boliden site the total CN concentration in the discharge to the tailings pond is on average 0.89 mg/l, while the discharge from the pond contains only an average 0.06 mg/l total CN.
Natural decomposition of possible trace contents of cyanide is assumed to take place in the tailings pond, following a complex scheme of processes.

At the **Orivesi** mine, the clarified water from the tailings management area, including the rainfall water, or from the old underground mine is re-used/used in the process. The mineral processing plant is operating only with this water, without any additional water from natural surface waters. Depending on the rainfall, it is sometimes (but not every year) necessary to remove excess water from the system by leading it to the river. Recycling also saves small amounts of reagents, but the savings are not very significant, because the flotation reagents decompose in the tailings management area. A schematic water balance is presented in the figure below.

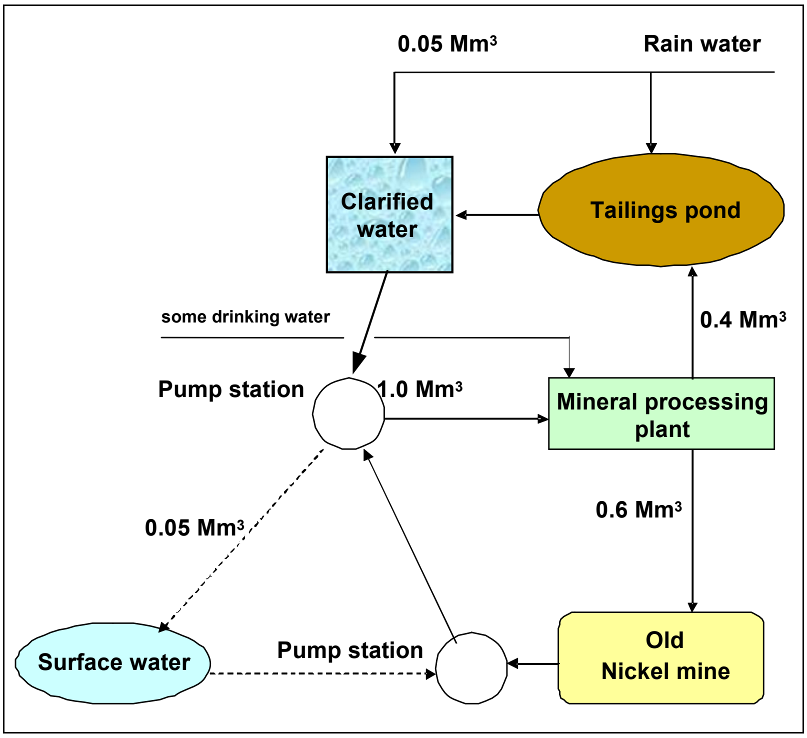


Figure 3.44: Water cycle at Orivesi site
[50, Au group, 2002]

During 2001 the (unit) consumption of reagents at the Orivesi gold mine is given in the table below.

Reagent	Consumption
	(g/t)
SIBX	50
DTP	50
Dowfroth	8
Flocculant	2
Steel balls	1500
Steel rods	700

Table 3.52: 2001 unit reagent consumption at Orivesi mine

3.1.6.5.2 Emissions to air

At **Bergama-Ovacik**, dust and HCN emissions are monitored on a daily basis. Dust emissions are eliminated by surface wetting of the roads and by a scrubber system at the crushers and conveyors. HCN gas is monitored over the leach tanks and on the embankment of the tailings pond, producing monitoring results of nearly zero. A scrubber treats the gas emissions to air from the regeneration oven of the activated carbon.

At the **Boliden** mineral processing plant, the emissions to air are monitored. During the last years the biggest emission source to air, drying of concentrates, has been completely eliminated by the introduction of filters instead of using ovens. The gold leaching plant has a complete purification plant for all ventilation air. This air passes through a wet scrubber where any possible HCN is absorbed in a sodium-hydroxide solution at high pH. The CN laden solution is returned to the CIL-process. The regeneration circuit for the activated carbon is equipped with a wet scrubber where lime is added for pH adjustment.

The emission from the gold leaching plant during year 2001 is summarised in the table below. Apart from the emissions reported in the table below the Boliden mineral processing plant reported emissions of 0.1 tonne particles in suspension.

		Emissions				
Date	Operating hours	Particles	CN_{tot}	Hg	H_2S	SO_2
Regeneration of activated carbon	h	kg	kg	kg	kg	kg
2001 – 10 - 16	30	128.550	0.270	0.000	8.700	1.275
2001 – 11 - 22	30	1.350	0.009	0.006	10.050	1.275
Wet-scrubber						
2001 – 11 - 22	1400		4.200			
2001 – 10 - 16	1400		3.080			
2001 – 07 - 03	1400		0.042			
Ovens						
2001 – 12 - 03	437.5	0.013	0.051			
2001 – 09 - 25	437.5	0.001	0.001			
Total		**129.91**	**7.65**	**0.007**	**18.75**	**2.55**

Table 3.53: Emissions to air from Boliden gold leaching plant

At the **Orivesi** mine dust emissions are not measured, but some dust emission occurs from the crushing plant.

3.1.6.5.3 Emissions to water

No discharge of water occurred from the **Bergama-Ovacik** site during year 2001 therefore no direct emissions. Groundwater monitoring does not indicate any discharge to the groundwater.

The emissions to surface water from the **Boliden** site are summarised in the table below for the last four years (1998 - 2001). The annual average concentrations are given together with total annual load of each element.

Year	**Volume**	**Cu**		**Pb**		**Zn**		**As**		**Cd**	
	Mm^3	µg/l	kg	µg/l	kg	mg/l	tonne	µg/l	kg	µg/l	kg
2001	11.1	7	72	19	191	0.1	1.07	14	156	0.1	1
2000	6.9	10	70	34	235	0.11	0.77	8	55	0.1	3.0
1999	5.9	8	51	10	59	0.2	1.04	10	58.7	0.1	0.6
1998	7.5	22	134	20	100	0.22	1.33	1	7.5	0.2	1.5

Table 3.54: Emissions to surface water from Boliden site

The production at the gold leaching plant started in July 2001. During the remainder of that year a total of 417 kg of CN_{tot} were discharged. Once the plant had reached normal production the average concentration of CN_{tot} in the discharge reached 0.06 mg/l.

At the **Orivesi** mine the total emissions to surface water for year 2000 are given in the table below.

Parameter	Unit	Year 2000
Tailings water discharge	m^3	780000
Ca	t	-
SO_4	t	680
COD	t	-
Solids	t	15
Cu	kg	10
Zn	kg	-
Fe	kg	-
Cd	g	-
Ni	kg	278
Cr	kg	-

Table 3.55: Emissions to water from Orivesi site

A slight increase of metal contents in groundwater (compared with the contents in the baseline study) have been observed after the nickel mine was closed and the groundwater had reached the original level. The tailings water from the current gold process has not increased the metal contents in ground water.

3.1.6.5.4 Energy consumption

The energy consumption for tailings management at **Orivesi** is reported to be 1 kWh/t. The total energy consumption at the site per tonne ore processed is 53.5 kWh/t.

At **Ovacik** mine the monthly total energy consumption (based on the first 10 months of operation) 1500 MWh. Corresponding to the designed throughput of 0.3 million tonnes/yr, this results in a total energy consumption of 60 kWh/t of ore processed.

At the **Boliden** mineral processing plant it is estimated that tailings management consumes about 2 kWh/t.

3.1.7 Tungsten

In this section, information is provided about the Panasqueira mine in Portugal and the Mittersill mine in Austria.

3.1.7.1 Mineralogy and mining techniques

Wolframite ((Fe, Mn)WO_4, iron manganese tungstate) is actually a series between two minerals; huebnerite and ferberite. Huebnerite is the manganese-rich end member of the series while ferberite is the iron rich end member at the other end of the series. Wolframite is the name of the series and the name applied to indistinguishable specimens and specimens intermediate between the two end members. Most specimens found in nature fall within the 20 - 80 % range of the series and these are termed wolframites. Only if they are more pure than 80 % manganese are they called huebnerite and conversely if they are 80 % iron they are called ferberite. Scheelite ($CaWO_4$, Calcium Tungstate) is an important ore of tungsten which is a strategically important metal. Scheelite is named after the discoverer of tungsten, K. W. Scheele [37, Mineralgallery, 2002].

The **Panasqueira** mine in Portugal mines ferberitic type wolframite. In the year 2000, 332000 t of ore were extracted, which yielded 1269 t of wolframite concentrate (75 % WO_3), 12 t of cassiterite concentrate (72 % Sn) and 132 t of chalcopyrite concentrate (28 % Cu).

The Panasqueira orebody occurs as a sequence of almost parallel quartz veins containing, amongst other minerals, wolframite and cassiterite. The mineralised zone has a length of approximately 500 to 1000 metres, and continues 500 metres downwards. The upper parts of the orebody have been mined out. The wolframite mineralisation occurs as very large crystals or large crystal aggregates, usually concentrated towards the margins or, occasionally, close to the mid-line of the quartz vein host. The mineralisation may be accompanied by intense biotite alteration.

At **Panasqueira** the applied mining method is room-and-pillar mining.
[141, Panasqueira, 2003]

In 1975, the mining operation in **Mittersill** started with an open pit operation. In 1979, the underground operation was developed. The open pit was closed in 1986. Today 450000 tonnes of ore are mined yearly in the underground mine with an average WO_3-content of 0.50 %.

The host rock of the Mittersill deposit consists of quartz lenses, laminated quartzites, pyroxenites, orthogneisses, amphibolites, hornblendites and granites. The tungsten bearing mineral at Mittersill is scheelite ($CaWO_4$). The main gangue minerals are quartz, silicates (mica, talc, biotite, hornblende, amphibole, pyroxene, etc.), carbonates, apatite and sulphides. The content of sulphide minerals is <0.5 %. The most frequent sulphide mineral is pyrrhotite. Less frequent are pyrite, chalcopyrite, galena and molybdenite.

The whole mining operation in Mittersill is situated in a protected landscape. Therefore, all the social facilities, workshops and warehouses are installed underground. The ore is crushed underground. The mine and the mineral processing plant are connected by a 3 km long gallery. The ore is transported from the crushing station to the mineral processing plant by a conveyor belt system.

The main mining methods used for the extraction of the massive orebody are:

- sublevel stoping
- sublevel caving
- cut and fill.

The waste-rock which is mined during the development of the orebody is dumped into open stopes underground. There are no waste-rock dumps on the surface. Tailings are used for backfilling of the open stopes.

3.1.7.2 Mineral processing

At **Panasqueira,** the wolframite is recovered by a combination of dense medium separation, shaking tables and flotation. Tin and copper are also removed by flotation. [141, Panasqueira, 2003]

In **Mittersill**, due to the fine intergrowth of scheelite with the gangue minerals, the ore is treated by flotation as using gravity separation would result in high losses of scheelite, making the operation uneconomical. In the following section the circuit at the Mittersill operation is described in more depth.

3.1.7.2.1 Comminution

The ore is crushed to <14 mm by means of a three stage crushing system, situated underground. The crushed ore is then stored in two underground ore bins before being transported to the mineral processing plant by a conveyor belt system situated in a 3 km long gallery. Just beside

the mineral processing plant there is a stockpile dimensioned so as to secure the supply of the process with ore for discontinued production at the crushing plant.
The top size of the feed is further reduced to <10 mm in a one stage crushing system consisting of a cone crusher which operates in closed cycle with a vibrating screen. The crushed ore is stored in two ore silos from where the ore is fed to a single stage ball mill at a feed rate of 80 – 82 t/h. To achieve sufficient liberation of the scheelite from the gangue, the ore has to be ground to 80 % passing 180 µm. The mill discharge is pumped to a classification system, which consists of screens and a hydrocyclone. The fines with a top particle size of 500 µm are pumped to the flotation process, the coarse fraction is recycled to the ball mill.
[52, Tungsten group, 2002]

3.1.7.2.2 Separation

Flotation consists of one rougher bank and four cleaning stages. A concentrate with an average grade of 40 % WO_3 is produced. The rougher tailings are pumped to a hydrocyclone. The cyclone underflow, which contains coarse and intergrown scheelite is recycled to the ball mill for regrinding, the hydrocyclone overflow represents the final tailings stream. The collectors used for flotation are fatty acids (carboxylates), alkyl sulphonates and alkyl sulphate.

A schematic flow sheet of the processing plant is given in the figure below.

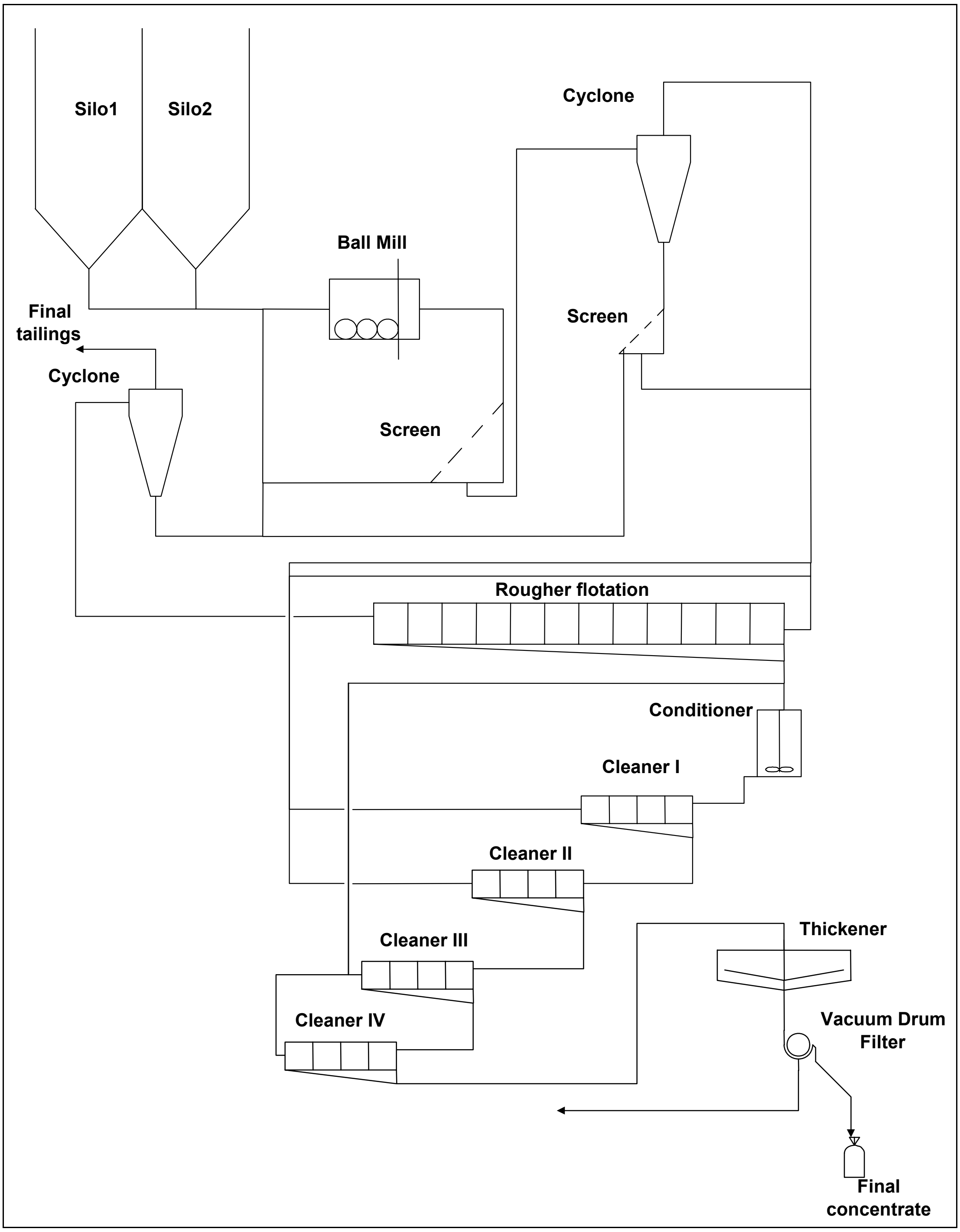


Figure 3.45: Flow sheet of Mittersill mineral processing plant
[52, Tungsten group, 2002]

3.1.7.3 Tailings management

The tailings at the **Panasqueira** operation are managed in ponds [141, Panasqueira, 2003].

The tailings stream at **Mittersill site** represent 99 % of the initial process feed. At the present throughput of 450000 t/yr, a storage volume of 250000 m³ is needed every year.

The Mittersill site operates two tailings management systems:
- a tailings pond, approximately 10 km away from the mineral processing plant in a valley
- a backfilling system, with a maximum capacity of 35 % of the mineral processing plant feed.

The tailings ponds cover an area of 34 ha, of which 20 ha have already been reclaimed.

3.1.7.3.1 Characteristics of tailings

The chemical behaviour of the tailings has been characterised. The test procedures involved:

- performing leachate tests
- determination of the total content of heavy metals by leaching the solids with aqua regia.

The following tables show the results of these tests.

Parameter leachate	Test results
pH	7.8
Conductivity, mS/cm	0.8
Ca, mg/l	10
Mg, mg/l	9
Al, mg/l	0.17
Sb, mg/l	<0.01
As, mg/l	<0.05
Ba, mg/l	<0.5
Be, mg/l	<0.005
B, mg/l	<0.01
Pb, mg/l	<0.05
Cd, mg/l	<0.005
Cr total, mg/l	<0.05
Fe, mg/l	<0.1
Co, mg/l	<0.01
Cu, mg/l	<0.01
Mn, mg/l	<0.01
Ni, mg/l	<0.05
Hg, mg/l	<0.001
Se, mg/l	<0.01
Ag, mg/l	<0.05
Th, mg/	<0.01
V, mg/l	<0.01
Zn, mg/l	<0.5
Sn, mg/l	<0.05
F, mg/l	<0.01
PO_4, mg/l	0.6
SO_4, mg/l	156
CN, mg/kg dry solids	n/d
F, mg/kg dry solids	n/d
NO_3-N, mg/kg dry solids	0.8
Anionic surfactants, mg/kg dry solids	<0.05
Total hydrocarbons-C, mg/kg dry solids	Not detectable
Hydro-Carbons, mg/kg dry solids	Not detectable
Extractable organic halogens, mg/kg dry solids	Not detectable

Table 3.56: Leachate test results of tailings at Mittersill site
[52, Tungsten group, 2002]

Parameter total content	Test results (mg/kg dry solids)
As	7
Cd	<0.5
Co	<0.5
Cr	31
Cu	<0.5
Ni	22
Hg	Not detectable
Pb	12
Zn	82
THC	Not detectable
HC	Not detectable
PAH	Not detectable

Table 3.57: Heavy metal contents of tailings at Mittersill site
[52, Tungsten group, 2002]

The following figure shows the grain size distribution of the feed to the mineral processing plant and the tailings.

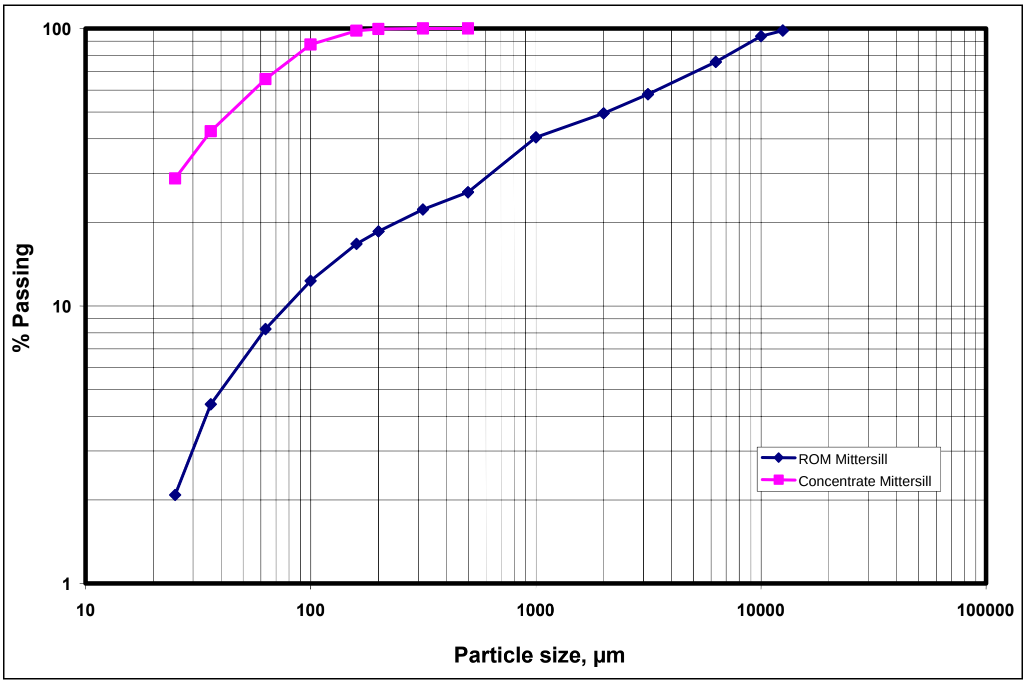


Figure 3.46: Size distribution of feed to mineral processing plant and tailings at Mittersill site
[52, Tungsten group, 2002]

3.1.7.3.2 Applied management methods

The backfilling system was installed in 1987 and consists of a lamella thickener, a piston diaphragm pump and a steel pipeline which connects the mineral processing plant with the different levels of the underground mine. The backfill has to be pumped over a distance of 3000 m and up to a maximum height of 280 m.

The currently operated tailings pond is situated south of the little village of Stuhlfelden. The start-up of the tailings ponds was in 1982. Until this time the first tailings pond, the 'Felbertal' pond, situated just on the opposite side of the mineral processing plant, was in operation. The final height of this first tailings dam was 24 m. The dam was built using the upstream method. Every 8 m a drainage system was installed. The starter dam consists of borrow material, the second and third stage were built using tailings.

The tailings ponds in Stuhlfelden are built using the upstream method. The final height of the tailings dam Stuhlfelden I & II was 16 m. The dams IVA and IVB will reach a final height of 24 m. The starter dams of ponds I and II with a height of 4 m were constructed using borrowed material. The starter dam of tailings pond IVA was built with tailings. To prevent erosion, the surface of the dam is covered with humus and re-vegetated. On one side, the area is limited by a slope. Two roads which cross the slope 30 and 60 m above the pond prevent uncontrolled entering of surface water into the tailings pond area. Prior to construction of the starter dam, the area was investigated by geotechnical engineers. Where necessary, the foundation of the starter dam was reinforced. The construction was surveyed by geotechnical engineers and reviewed by the water and mining authority.

In spring and summer, the water surface in the pond is kept high enough to prevent dust emissions from the tailings pond area. In autumn, water is discharged to the nearby stream. To prevent dusting from the tailings pond area, an automatic sprinkling system was installed. The sprinkling system is started and monitored from the central control room of the plant. During shutdowns of the mineral processing plant, standby teams are on duty to control the tailings pond area. The nearest river, the river Salzach is approximately 600 m away from the tailings ponds.

3.1.7.3.3 Safety of the TMF and accident prevention

The dams are raised in 2.5 m sections every year. The height of the layers applied to the dam surface is 0.5 m. The dam is divided in sections of 50 m. From every profile four samples are taken from the applied layer. The compaction is checked by using the proctor method. From one sample of every profile a particle size analysis is performed. The construction, monitoring, sampling and the data are controlled by a civil engineer and the federal authority.

For monitoring settlements of the tailings pond piezometers were installed. The ground movements are checked yearly. The data are controlled by the federal authority.

Monitoring of the TMF is performed three times a day by the process supervisors. For heavy rainfalls and failure of the barriers, excess water can be discharged through an emergency outlet.

To prevent erosion of the dam by the slurry, the inner surface of the dam is covered by a geo-textile.

3.1.7.3.4 Closure and after-care

It is planned to cover the pond surface with humus and grass. After reclamation the land is given back to the landowners. The tailings of the Mittersill operation readily dewater. It is known from already reclaimed tailings ponds that the tailings dewater and consolidate within a time period, i.e. 2 – 4 years.

Partial reclamation of the tailings pond is already performed during operation. The dam is constructed at the final inclination. The outer dam surface is already covered with humus and reclaimed.

3.1.7.4 Waste-rock management

At Mittersill, the waste-rock which is mined during development of the orebody is dumped into open stopes underground. There are no waste-rock dumps on the surface.

3.1.7.5 Current emissions and consumption levels

3.1.7.5.1 Management of water and reagents

No water is recycled from the tailings pond to the mineral processing plant.

3.1.7.5.2 Emissions to air

The average emissions of dust particulates from the tailings pond area are in the range of 50 mg/(m^2 28 days).

3.1.7.5.3 Emissions to water

The following table shows the parameters measured in the effluent discharged from the tailings pond.

Parameter	Average Values 1997
Temperature, °C	13.8
pH	7.9
Volume of sediment, ml/l	<0.1
Aluminium, mg/l	0.072
Iron, mg/l	0.285
Tungsten, mg/l	<0.1
Nitrite, mg/l	<0.1
Phosphorus, mg/l	<0.1
Chemical oxygen demand, mg/l	32.3
Total hydrocarbons, mg/l	<1

Table 3.58: 1997 averages of parameters measured in discharge from TMF of Mittersill site [52, Tungsten group, 2002]

Monitoring of the effluent of the tailings pond is performed twice a week by the laboratory technicians. When discharging the water into the nearby river, sampling of the water of the river upstream and downstream is performed daily. These samples are analysed in the laboratory of the mill and by a chemical laboratory. A report is sent to the federal authorities every year.

3.1.8 Costs

3.1.8.1 Operation

The following table lists the costs for tailings and waste-rock management.

Operation	Sub-operation	Cost interval	Units	Site/reference
Waste-rock management				
	Hoisting to surface	0.5 - 1	EUR/t	[1]
	Surface transport to dump	0.2 - 0.5	EUR/t x km	[1]
	Dump construction	0.1 - 0.5	EUR/t	[1]
Tailings management				
	Pumping to pond	0.1	EUR/t	[1]
	Tailings distribution	0.05 - 0.3	EUR/t	[1]
	Dust suppression	>0.1	EUR/t	[1]
	Tailings dewatering	1.0 - 4.0	EUR/t	[1]
	Truck transport to mine/dump	0.5 - 1	EUR/t	[1]
	Tailings pumping and maintenance	0.1	EUR/t	[1]
	Dam raises	0.4	EUR/t	[1]
	Water treatment with lime	0.1	EUR/t	[1]
	Monitoring	0.1	EUR/t	[1]
	Total operating cost	0.8	EUR/t	Boliden[2]
	Capital cost for 7 Mm^3 pond	5.34	EUR million	Zinkgruvan[3]
	Capital cost pumps, 100 l/s	0.45	EUR million	Zinkgruvan[3]
	Tailings pumping	0.11	EUR/t	Zinkgruvan[3]
	Pumping water back to processing plant	0.04	EUR/t	Zinkgruvan[3]
	Pipe wear	0.16	EUR/t	Zinkgruvan[3]
	Piers	0.07	EUR/t	Zinkgruvan[3]
	Total operating cost	0.37	EUR/t	Zinkgruvan[3]
	Dam safety monitoring	0.05	EUR/t	Zinkgruvan[3]
	Total operating cost	0.8	EUR/t	Zinkgruvan[3]
	Dam raises	0.5	EUR/t	Río Narcea[4]
	CN destruction	1.0	EUR/t	Río Narcea[4]
	Others (energy, pipes, maint.)	0.5	EUR/t	Río Narcea[4]
	Total operating cost	2.0	EUR/t	Río Narcea[4]
	Total operating cost	0.6	EUR/t	Kemi[5]
	Total operating cost	0.4	EUR/t	Orivesi[6]
	Total operating cost	0.48	EUR/t	Pyhäsalmi[7]
	Total operating cost	0.3	EUR/t	Hitura[7]
	Total operating cost	0.4	EUR/t	Garpenberg[8]

Sources:
1 = [98, Eriksson, 2002]
2 = [65, Base metals group, 2002]
3 = [66, Base metals group, 2002]
4 = [58, IGME, 2002]
5 = [71, Himmi, 2002]
6 = [59, Himmi, 2002]
7 = [62, Himmi, 2002]
8 = [64, Base metals group, 2002]

Table 3.59: Costs for tailings and waste-rock management at metal sites

At the **Boliden** mineral processing plant the operational cost for deposition of tailings is EUR 0.8/t. This figure includes the energy cost for pumping the tailings and maintenance (EUR 0.1/t) and the actual cost to raise the dam (EUR 0.4/t), water treatment of discharged water from the pond (EUR 0.1/t) and monitoring costs (EUR 0.1/t).

At **Garpenberg** the operational cost for the tailing deposition is EUR 0.4/t ore processed. This cost includes pumping costs, raising of dams, maintenance of pipelines and pumps, monitoring etc. However, it does not include decommissioning costs.

Tailings management costs in the **Legnica-Glogow copper basin** are as follows:

Sub-operation	Costs interval	Unit
Tailings pumping to the tailings pond [1)]	0.530	EUR/t
Dam construction	0.060	EUR/t
Pumping water back to the processing plant[1)]	0.333	EUR/t
Dust spraying with asphalt emulsion[2)]	0.031	EUR/t
Air, water, soil and seismic monitoring	0.020	EUR/t
Safety supervision and control procedures (geotechnical monitoring)	0.014	EUR/t
Emergency alarm system	0.0004	EUR/t
Ecological fee for tailings disposal[3)]	0.470	EUR/t
Pumping excess water to the Oder river[4)]	0.064 0.046	EUR/m^3 EUR/t
Purification of discharged water[4)]	0.043 0.031	EUR/m^3 EUR/t
Hydrotechnical monitoring[4)]	0.003 0.002	EUR/m^3 EUR/t
Ecological fees for discharged water[4)]	0.135 0.097	EUR/m^3 EUR/t
Total operating cost	1.634	EUR/t

1. The relevant figures to relate to these costs are shown in the following tables
2. Cost includes cost for emulsion and distribution from helicopter and ground vehicles. The yearly sprinkled surface is about 1080 ha, taking into account that some places are sprinkled more than once.
3. Compulsory fee
4. In 2002 18.9 Mm^3 of water was discharged from the tailings pond, from which 18.6 Mm^3 to the Oder river and 362664 m^3 to the bottom of the pond. The data refer to 1m^3 of discharged water and to 1 t of tailings (1t of tailings refers to 0.721 m^3 of discharged water).

Table 3.60: Tailings management costs in the Legnica-Glogow copper basin
[KGHM Polska Miedz, 2002 #113]

Processing plant	Tailings generated in 2001 (dry Mt/yr)	Horizontal distance (km)	Elevation (m)
Lubin	6.4	13.4	47
Polkowice	8.0	13.7	39
Rudna	12.5	11.2	23

Table 3.61: Relevant tailings generated, distance and elevation between mineral processing plants and the tailings pond in the Legnica-Glogow copper basin
[KGHM Polska Miedz, 2002 #113]

Processing plant	Water returned in 2001 (Mm^3/yr)	Horizontal distance (km)	Elevation (m)
Lubin	26.8	12.1	45
Polkowice	27	9.7	60
Rudna	67	6.4	60

Table 3.62: Relevant amounts of water returned to mineral processing plants, distance and elevation between mineral processing plants and the tailings pond in the Legnica-Glogow copper basin
[KGHM Polska Miedz, 2002 #113]

At **Zinkgruvan** up to the beginning of 1990's the tailings were managed above the water surface which was less expensive as the pipes could be stationary at one fixed point for a long time. Since the start of discharging mainly under the water surface the costs per unit have been more than double. On the other hand the management under water has given a significant reduction of the metal transport from the pond and less dusting from the tailings area.

The operating costs can be divided into the following items (EUR/m^3):

- pumping of tailings: 0.15
- water recycle: 0.05
- pipe arrangements, wear: 0.22
- piers: 0.10

The dam safety monitoring system now underway will add another EUR 0.07/m^3 and may be complemented with other systems as well.
[66, Base metals group, 2002]

The following table shows some further cost information relevant to the management of tailings and waste-rock.

Operation	Sub-operation	Cost	Units	Comment/Site
Dam costs	Dam construction	0.05 - 0.5	EUR/t	Scale. site & method dependent[1]
Lining	HDPE liner, 16 ha	7.5	EUR/m^2	Ovacik[2]
Environmental monitoring	One water sample (surface or GW)	220	EUR/sample	Sampling, sample preparation, shipping, analysis and reporting[1]
Installation of monitoring well	Ground water monitoring well	200	EUR/m	Establishment, drilling, lining and rinsing[1]
Backfill	Transport cost, 15 km Transport cost, 100 km	0.3 0.8	EUR/t EUR/t	[1] [1]
Thickened tailings	Operating costs excluding capital costs Capital cost thickener, (14 m high) Total capital cost Of which for dam construction	0.15 170000 2.2 1.4	EUR/t EUR EUR million EUR million	[3] [3] [3] [3]
Sources: 1 = [98, Eriksson, 2002] 2 = [56, Au group, 2002] 3 = [31, Ritcey,]				

Table 3.63: Cost of other operations relevant to the management of tailings and waste-rock

The following table gives more detailed information on costs for destroying cyanide using the SO_2/air method.

Site	Tonnes/day	Weight % solids	WAD-CN (mg/l)		Operating cost	
			Feed	Treated	USD/tonne	USD/kg WAD-CN
A	2800	35	80	0.30	0.35	2.56
B	920	47	175	0.90	0.77	4.28
C	800	45	120	0.50	0.91	6.06
D	2700	40	290	0.15	0.95	2.40

Table 3.64: Operating cost in USD for CN destruction using the SO_2/air method in 2001
[99, Devuyst, 2002]

The operating costs are actual and include the costs of SO_2, lime, copper sulphate and power. Capital costs for these operations are in the range of USD 360000 to 1.1 million installed. Capital costs include reactor, agitator, air compressor, SO_2 delivery system, and copper sulphate delivery system. It does not include the tailings pump box and pump and the lime system (usually already part of the plant). It assumes the system is outdoors, including reagent systems and air compressor. Therefore no additional building facilities need to be constructed, only site preparation and proper foundations. None of the examples in the table make use of a sulphur burner for the source of SO_2. If this was the case, the capital cost would be much higher (about 80 %), but the operating cost would be reduced by about 60 %. The variation in operating costs is due to unit reagent cost for SO_2, lime, copper sulphate and power.
[99, Devuyst, 2002]

3.1.8.2 Closure

The following table lists cost information related to closure cost.

Sub-operation	Cost interval	Units	Comment/Site
Dump or tailings pond revegetation	0.1 - 0.5	EUR/m^2	Scale dependent[1]
Engineered cover on dump or pond	3.0 - 10	EUR/m^3	Scale and method dependent[1]
Flooding of tailings pond	0.5 - 1	EUR/m^2	Scale and site dependent[1]
Wetland establishment	0.1 - 1	EUR/m^2	Scale and site dependent[1]
Groundwater saturation	0.2 - 2	EUR/m^2	Scale and site dependent[1]
Dewatering of pond	0.7 - 1.2	EUR/m^2	Tara[2]
Revegetation	0.7 - 0.8	EUR/m^2	Tara[2]
Monitoring	1.3 - 1.7	EUR/m^2	Tara[2]
Maintenance	0.1	EUR/m^2	Tara[2]
total reclamation and closure	3.1 - 3.7	EUR/m^2	Tara[2]
Closure (dewatering and cover)	1.8	USD million	Ovacik[3]
Closure (not specified), 37 ha	0.6	EUR million	Orivesi[4]
Closure (water cover, vegetation), 280 ha	1.5	EUR million	Boliden[5]
Closure and after-care, 100 ha	5.4	EUR million	Pyhäsalmi[6]
Rehabilitation	14.4	EUR/m^2	Zinkgruvan[7]
Apirsa actual costs			
Apirsa tailings pond reclamation	18.5	EUR/m^2	Total cost/total area[1]
clay cover placed	2.9	EUR/m^3	Material it self not included[1]
Protection cover placed	3.1	EUR/m^3	Material it self not included[1]
Resloping of dam	0.9	EUR/m^3	<100 m movement of material (bulldozer)[1]
Resloping of dam	4	EUR/m^3	>100m movement of material (loading transport and placement)[1]
Revegetation with grass	0.05	EUR/m^2	Conventional seeding[1]
Saxberget actual reclamation cost			
Composite cover unit cost (1995)	7	EUR/m^2	Total cost/total area[1]
Stekenjokk actual reclamation cost			
Water cover unit cost (1992)	1.5	EUR/m^2	Total cost/total area[1]
Kristineberg actual reclamation costs			
Unit cost water cover	1.5	EUR/m^2	Total cost/total area[1]
Unit cost composite dry cover	6	EUR/m^2	Total cost/total area[1]
Unit cost increased ground water level	4	EUR/m^2	Total cost/total area[1]
Sources: 1 = [98, Eriksson, 2002] 2 = [23, Tara, 1999] 3 = [56, Au group, 2002] 4 = [59, Himmi, 2002] 5 = [50, Au group, 2002] 6 = [62, Himmi, 2002] 7 = [66, Base metals group, 2002]			

Table 3.65: Cost information for closure and after-care of metalliferous mining tailings and waste-rock management

Reclamation and closure costs estimated for the Tara tailings facility are calculated for a five year active monitoring phase, a five year passive monitoring phase and a ten year. long-term monitoring phase. Re-vegetation costs were calculated for a surface are of 66.8 – 85.4 ha with a unit cost of approx. EUR 3200/ha including fertiliser and seed. The costs for monitoring are based on the assumption that one full time staff be employed for a five year so called active care period monitoring phase. Other cost factors included are reclamation performance, agronomic performance assessment (examination of grazing sheep), wildlife monitoring, surface water quality, groundwater quality, dust monitoring, geotechnical monitoring (piezometers and visual inspections).

The decommissioning cost for the Boliden tailings pond are estimated to be EUR 1.5 million. This includes the arrangements for securing a permanent water cover, stabilisation of shallow bottoms, reconstruction of discharge devices, re-vegetation costs, long-term monitoring and management of the water cover. At the last raise, the dams are built to their final long-term stable slope angle and required erosion protection is installed, costs that are not included in the decommissioning costs are given above [50, Au group, 2002].

At Pyhäsalmi deposits of EUR 3.6 million and at Hitura EUR 0.6 million have been reserved in the accounts for closure and after care. The total closure and after-care costs for Pyhäsalmi tailings area is estimated to EUR 5.4 million.

Río Narcea has posted a bond of approximately EUR 3 million which corresponds to the Spanish norm (PTS 2 million/ha).

3.2 Industrial minerals

The term "industrial minerals" covers a wide range of different materials. Their common denominator is that they are all used as functional fillers or as production aids by industry. They are generally reduced in size to a very fine powder before use. The main categories included in this family are talc, calcium carbonate (ground and precipitated), feldspar, kaolin, ball clays, perlite, bentonite, sepiolite, silica, borates, etc. The mineralogical and chemical characteristics, as well as the particle-size distribution of the final product, determine the possible end uses. Quality requirements are usually very precise. The end uses of these minerals are extremely diversified. The geological availability of industrial minerals depends on the categories considered: talc, for instance, is less common than silica sand. However, even for the categories which seem more common, the physico-chemical requirements can be so high and precise that only a limited number of ore bodies can be worked.
[48, Bennett, 2002]

3.2.1 Barytes

The following production sites within the EU-15 were reported to this work:

Site	Country
Barytine de Chaillac, Chaillac	France
Wolfach Dreislar Bad Lauterberg	Germany
Vera, Coto minero Berja	Spain
Foss Mine, Aberfeldy Closehouse Mine, Middleton-in-Teesdale	United Kingdom

Table 3.66: Barytes mines in Europe

3.2.1.1 Mineralogy and mining techniques

Barytes is the naturally-occurring mineral form of barium sulphate ($BaSO_4$).

Within the EU-15, 55 % of the Barytes is produced by underground mining [29, Barytes, 2002].

Barytes deposits worldwide occur in ore bodies as residual, vein-type and bedded formats. Extraction is by both surface and underground techniques dependent on the geology and economics of the region. Each deposit and the most suitable extraction and processing route are very site-specific. Overburden and waste-rock generally remain in-situ, or are sold as construction products or are used in general reclaim/restoration.

3.2.1.2 Mineral processing

There is no standard flow sheet for the industry due primarily to the wide range of products. Mineral processing varies from a simple crush-only aggregate-type operation through to heavy-medium processing, jigging, fine grinding and flotation. At some operations small quantities of finished product are subsequently and separately acid-washed for special sale applications [29, Barytes, 2002]. Optical separation is also used in at least one operation.

The prime requirement for oil-well applications and for several of the filler applications (e.g. sound deadening, nuclear shielding) is high density (4.3 kg/l) and often $BaSO_4$ content (80 – 90 %) is sufficient to meet this. These operations generally only require crushing the run-of-mine material to produce a finished product with no processing waste.

Several other operations only require simple gravity methods to enhance the quality for the finished product, generally jigging or heavy-media separation.

Mineral processing may be necessary:

- for more complex ore bodies
- where the barytes is associated with other minerals (e.g. fluorspar, iron ore)
- where the barytes is finely disseminated in the host-rock (flotation)
- for the chemical industry where grades greater than 97 % BaSO4 are required.

The following flow sheet shows a site using gravity separation using jigs and flotation.

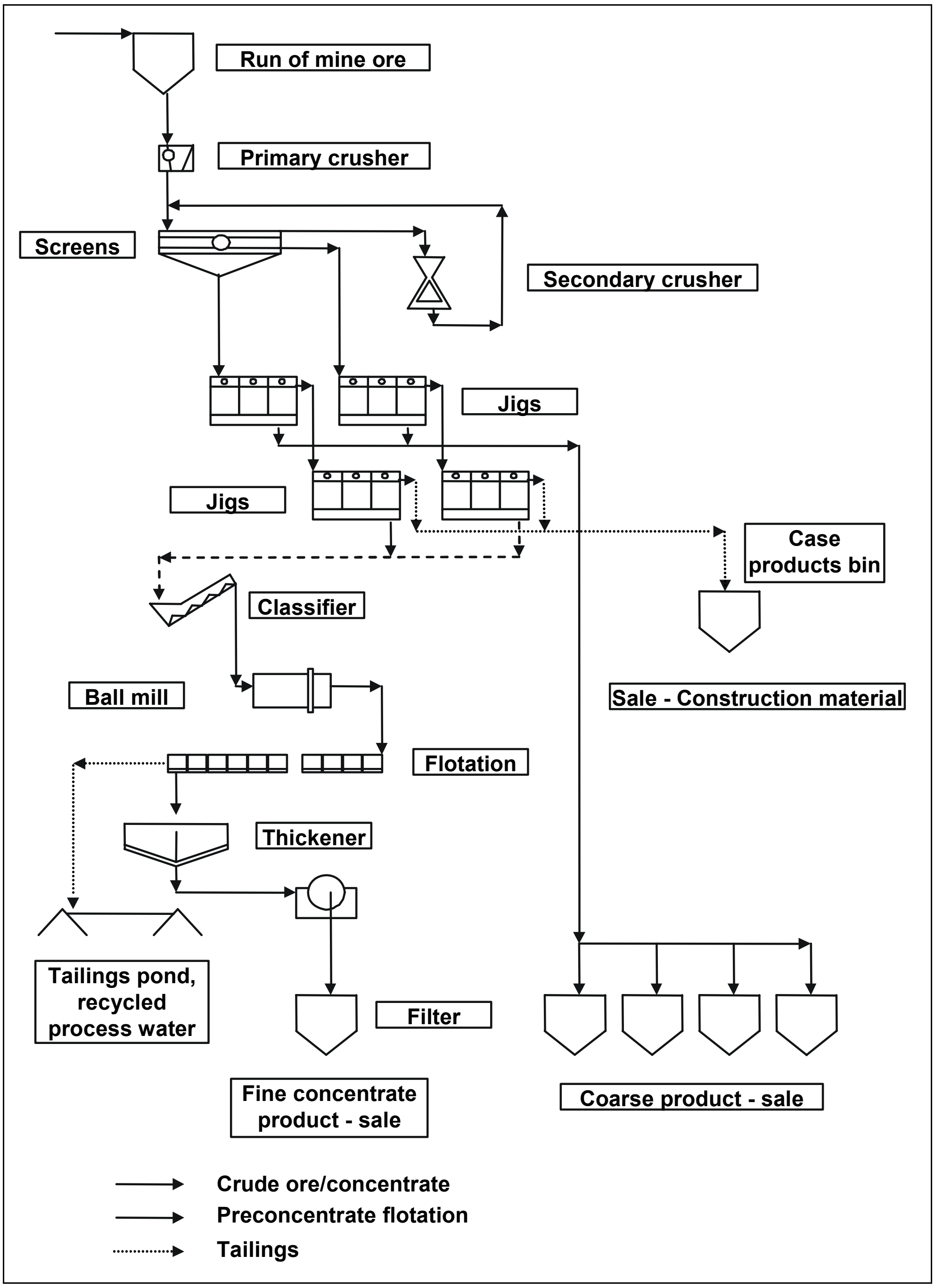


Figure 3.47: Flow sheet of barytes mineral processing plant using jigs and flotation

Sites with flotation operations use standard reagents for processing e.g. alkyl sulphates as collectors and all or some of sodium silicate, quebracho tannin (suppressant for talc and carbon) and citric acid as pulp modifiers [29, Barytes, 2002].

3.2.1.3 Tailings management

The following table shows the tailings management methods that are applied to different mineral processing schemes.

Type of mineral processing	No. of sites	% total output	Tailings management
Crushing Only	2	15	Nil
Crushing + Jigs only	4	23	Nil
Crush + Grind + Flotation	2	22	Dry tailings
Crush + Grind + Flotation	5	40	Wet tailings

Table 3.67: Tailings management methods applied to Barytes mines in Europe
[29, Barytes, 2002]

It can be seen that five sites, which together produce 40 % of the Barytes, use wet tailings management. Two out of these five sites together discard only 12500 tonnes of tailings into small ponds and nearly half of this tonnage is regularly dredged out as a product for land use.

In general it can be said that only a small percentage (2 %) of the tailings produced within the EU-15 are discarded as slurry in ponds. Typically coarse tailings are sold as aggregates. Finer tailings are mostly dewatered and also sold or used as backfill in the mine.

The tailings management options are listed in more detail in the following table.

Size fraction		Amout (kt/yr)
Subtotal >250 - 300 µm (including sales)		77
	<250 - 300 µm dewatered, heap/sale	214
	<250 - 300 backfill	20
	<250 - 300 µm tailings pond, recycle	5.5
	<250 - 300 µm tailings pond	7
Subtotal <250 - 300 µm		255.5
Total		323.5

Table 3.68: Tailings management options at European barytes operations

The operation in **Coto minero Berja** with a total mine production of 150000 t/yr produces three types of tailings:

- coarse tailings (>25 mm): after crushing in a hammer mill and screening
- after density separation the light fraction passes a screw classifier. The coarse fraction of these tailings are backfilled after dewater in basins in the pit (see figure below)
- the slimes from the screw classifier (17000 t/yr dry basis) are dewatered via evaporation in small concrete tailings basins (total capacity of 240 m^3). The dried slimes are then also backfilled in the open pit (see figure below).

Figure 3.48: Dewatering of barytes tailings in the pit
[110, IGME, 2002]

Figure 3.49: Dewatering of tailings in concrete basins
[110, IGME, 2002]

3.2.1.4 Waste-rock management

In general waste-rock remains in-situ, is sold as a construction product or used for site restoration.

At the operation in **Coto minero Berja** the waste-rock (325000 m^3/yr) is transferred with trucks within the mine and backfilled on the mined out site of the open pit and progressively re-vegetated.
[110, IGME, 2002]

3.2.2 Borates

This section includes information about the Turkish borates sites, the only producer of borates in Europe.

3.2.2.1 Mineralogy and mining techniques

The oldest form of boron known is the mineral salt called tincal (sodium tetraborate decahydrate, or simply borax). Other boron-containing minerals that occur naturally and are mined commercially include colemanite (calcium borate), hydroboracite (calcium magnesium borate), kernite (another sodium borate), and ulexite (sodium calcium borate).
[92, EBA, 2002]

3.2.2.2 Mineral processing

Boron minerals coming from open pit or underground mines are crushed into appropriate sizes and are then fed to the mineral processing plant.

The following figure shows a simplified flow sheet of the production of refined boron products.

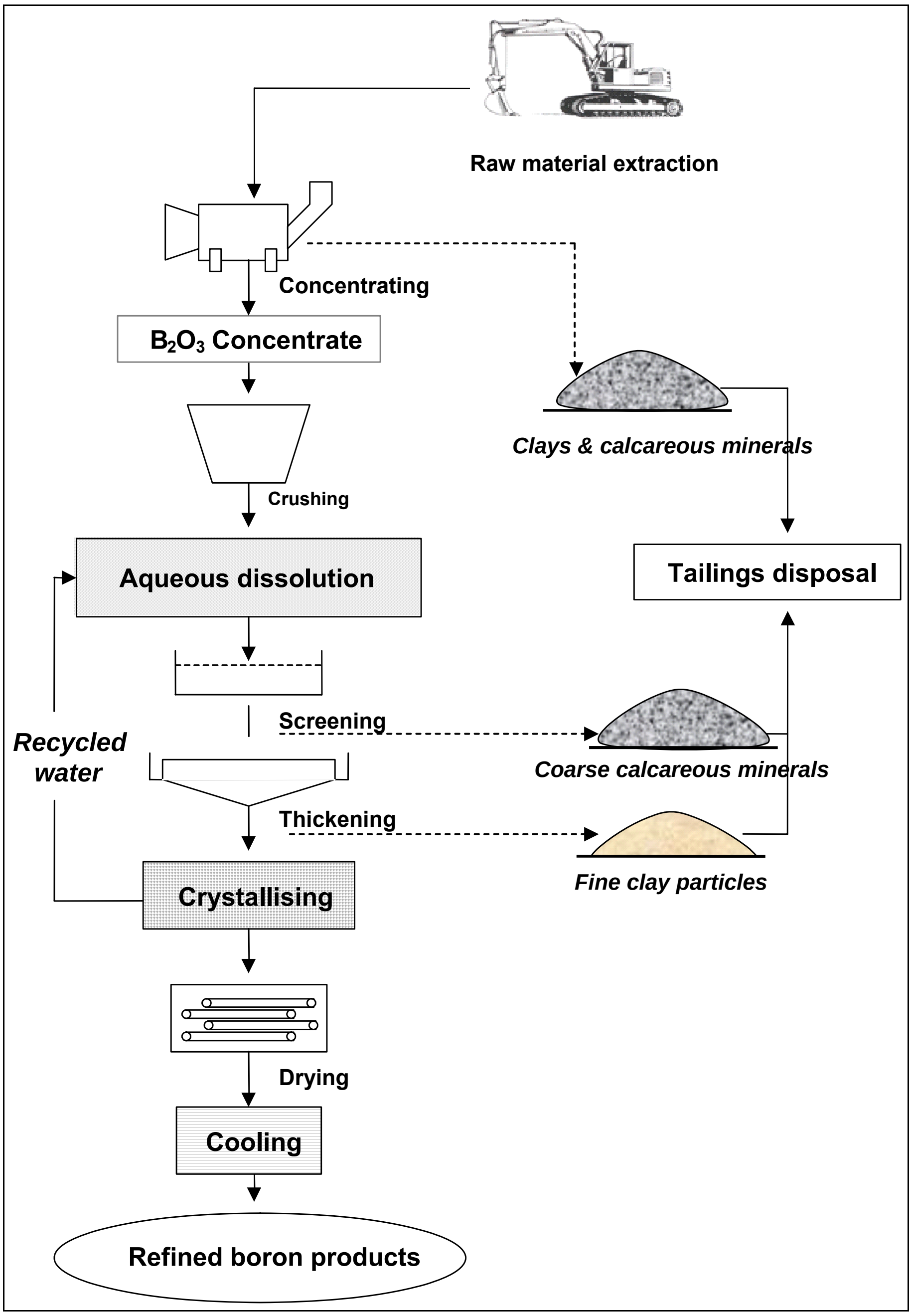


Figure 3.50: Simplified flow sheet of the production of refined boron products
[92, EBA, 2002]

The following table lists the inputs and outputs from the main steps of the borate process:

Process step	Inputs	Outputs
1. Cassifying	Raw material	Clays and calcareous minerals (solid) B_2O_3 concentrate
2. Aqueous dissolution	B_2O_3 concentrate Hot water	Unrefined Borax saturated solution
3. Screening	Unrefined Borax saturated solution	Coarse calcareous minerals Borax solution and fine clays
4. Thickening	Borax solution + fine clays Flocculants	Fine clays particles and flocculants Borax solution
5. Crystallising	Borax solution	Boron refined products (wet)
6. Drying/ cooling	Boron refined products (wet)	Boron refined products (dry)

Table 3.69: Inputs and outputs from the main steps of the borate process
[92, EBA, 2002]

3.2.2.3 Tailings management

In short, the coarse tailings consist of clays and calcareous minerals which are stored on heaps for backfilling purposes. The tailings slurries, which contain fine clays particles and flocculants, are managed in ponds. After the settlement of the clays particles, the water is recycled into the process.

The following table provides a list of the tailings released from the process and the type of management applied to them.

Process step	Tailings generated	Management method
1. Classifying	Clays and calcareous minerals (solid)	Heap
2. Aqueous dissolution	Non	n/a
3. Screening	Coarse calcareous minerals	Tailings ponds
4. Thickening	Fine clays particles & flocculants	Tailings ponds
5. Crystallising	Non	n/a
6. Drying/cooling	Non	n/a

Table 3.70: List of the tailings released from the process and the type of management applied
[92, EBA, 2002]

The tailings from screening and thickening are discharged into lined ponds near the mines. The ponds have five levels, with the first one being at the lowest and the fifth one at the highest level. The tailings pulp from the plant is pumped directly to the second, third and fourth ponds. After the solid particles contained in the tailing pulp settle down in these ponds, the overflow water is transferred progressively into the first pond. The 'clean' water in the first lake is then pumped back in the processing plant. Discharging tailing pulps to the fifth pond has recently started and the water level is increasing at this pond.

The annual quantity of the solid waste is about 350000 - 400000 tonnes and the amount of water for pumping the tailings to the lakes is 300000 – 500000 m^3/year. Total capacity of the current pond system is 14 million m^3.

The following alternatives are under evaluation for the management of tailings in the future:
1. constructing a new pond
2. discharging the solid tailings from the third and fourth pond to the heap area, and re-using the ponds
3. using a decanter system to recover the tailings in a solid form, and discarding them on a heap.

[92, EBA, 2002]

There is a monitoring system for CO, SO_2, NO_x and dust emissions. Boron particles in neighbouring streams, chemical oxygen demand in neighbouring streams, pH and conductivity values of neighbouring streams are measured on a regular basis. The analysis shows that the B_2O_3 content in the water is negligible, and it was demonstrated that this B_2O_3 content was coming from the groundwater being in contact with the deposit.

3.2.3 Feldspar

Unless otherwise mentioned, all information provided in this section originates from [39, IMA, 2002]

3.2.3.1 Mineralogy and mining techniques

Feldspar is by far the most abundant group of minerals in the earth's crust, forming about 60 % of terrestrial rocks. Feldspar minerals are essential components in igneous, metamorphic and sedimentary rocks, to such an extent that the classification of a number of rocks is based on feldspar content. The crystalline structure of feldspar consists of an infinite network of SiO_2 octahedron and AlO_4 tetrahedron. They usually crystallise in the monoclinic or triclinic system.

The mineralogical composition of most feldspars can be expressed in terms of the ternary system orthoclase ($KAlSi_3O_8$), albite ($NaAlSi_3O_8$) and anorthite ($CaAl_2Si_2O_8$). The minerals, the composition of which is comprised between albite and anorthite are known as plagioclase feldspars, while those comprised between albite and orthoclase are called alkali feldspars. This latter category is of particular interest in terms of industrial use.

Feldspar is extracted from quarries by simple excavation (loading shovel). The mineral ore is crushed into the appropriate size and transported to the processing plant by conveyor belts or trucks.

3.2.3.2 Mineral processing

Feldspars are either selectively mined or processed by optical, flotation and/or electrostatic separation, in order to remove the accessory minerals (e.g. quartz, mica, rutile, etc.) present in the ore. The feldspar then undergoes a comminution step. The degree of refining and possible comminution is very dependent upon the final use of the product. For a number of uses, it is perfectly acceptable, and even advantageous, that the product retains some accessory minerals, e.g. quartz, while at the other extreme some applications require extremely pure and fine-grounded grades. Basically, the two properties which make feldspars useful for downstream industries are their alkali and alumina content.

The flotation process is only used by AKW, INCUSA, and SP Minerals. The feldspar recovered by flotation only represents about 10 % of the European feldspar production. The flotation process is essential to get a high quality grade (low iron content and high alumina content) required for some specific and important applications (e.g. TV/computer screens). For instance, although the Italian producer Maffei is the biggest producer in Europe, the three above-mentioned companies supply the Italian market with these high quality grade products.

The essential use of the flotation process may be explained by the following figure:

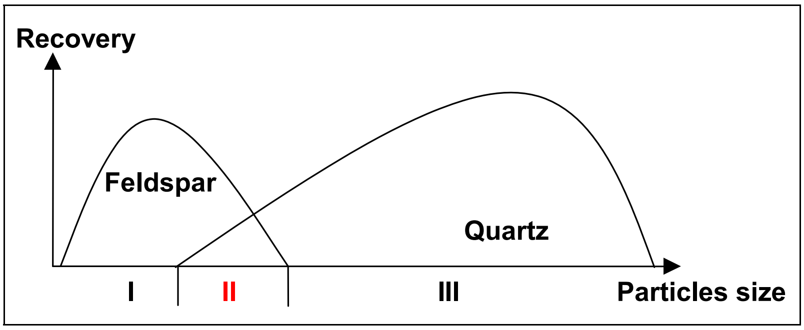


Figure 3.51: Feldspar particle vs. recovery graph
[39, IMA, 2002]

In Sections I and III a primary mechanical separation (hydrocycloning, centrifugation) can be achieved. In Section II, either optical, flotation or electrostatic separation can be used to separate feldspar from quartz, depending on both the intrinsic characteristics of the raw material, and the final product requirements.

The following flow sheet shows the steps involved in the recovery of feldspar.

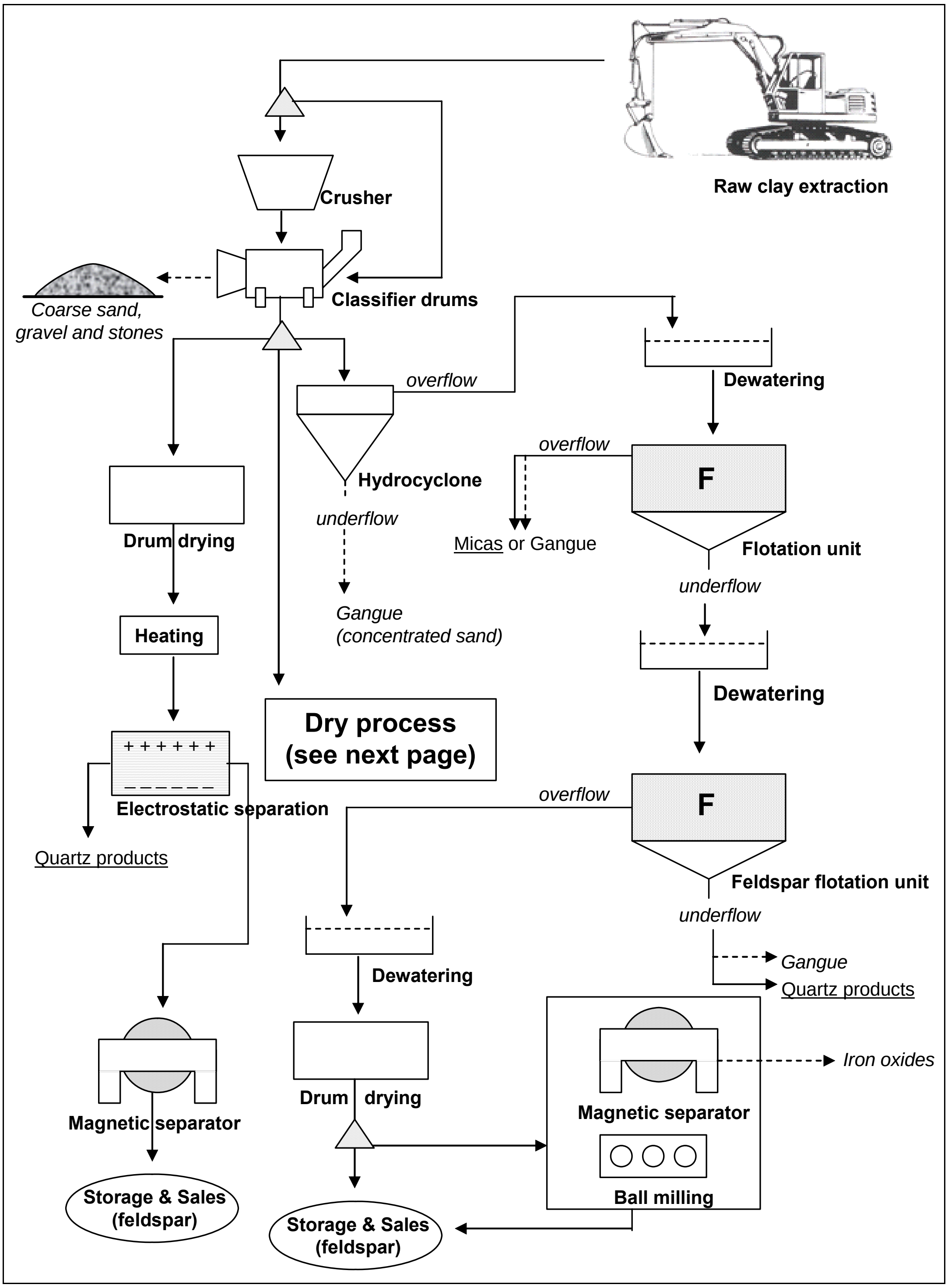


Figure 3.52: Flow sheet for Feldspar recovery using flotation
[39, IMA, 2002]

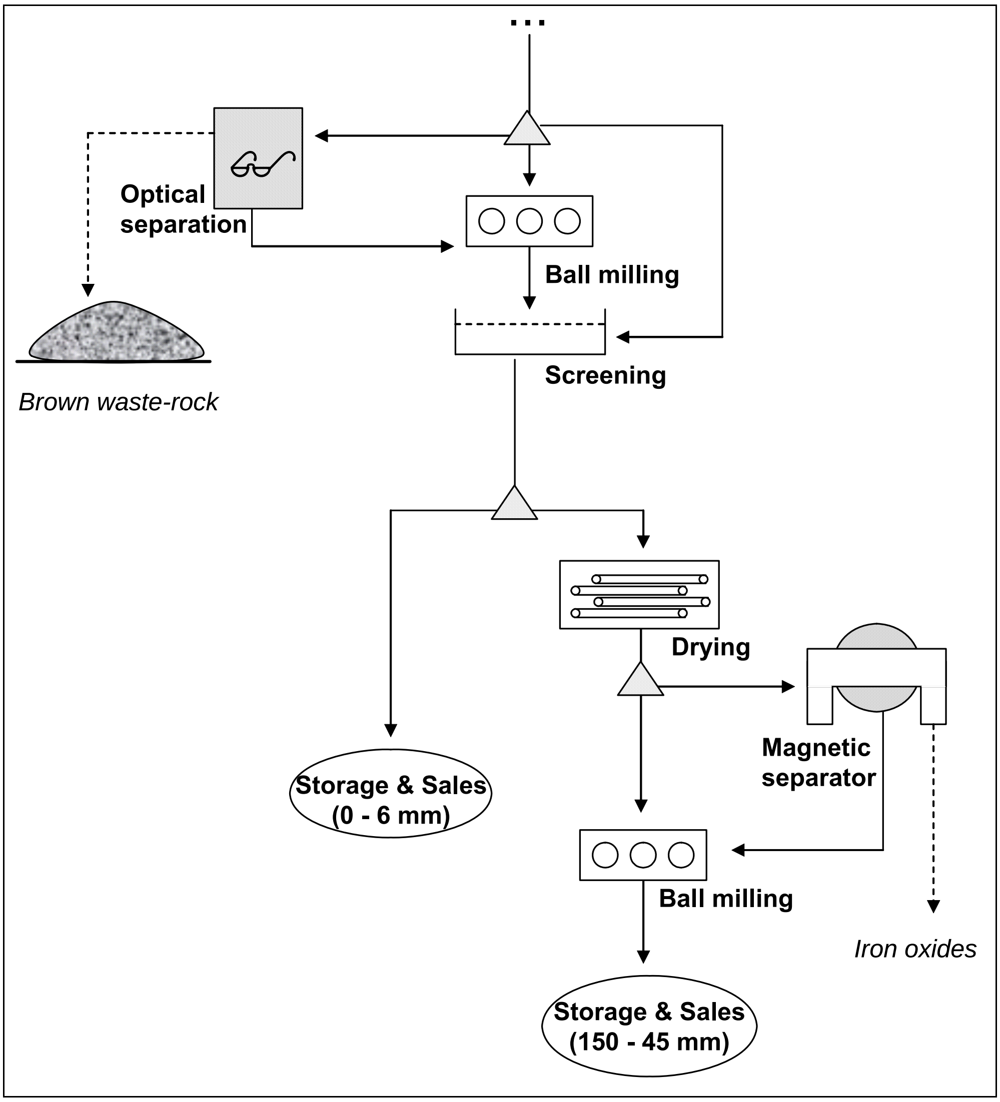


Figure 3.53: Dry processing step in the recovery of feldspar
[39, IMA, 2002]

In the feldspar process, one may distinguish three different flotation steps, namely the micas flotation, the oxides flotation, and the feldspar flotation**. Each of these requires a differnent reagent regime.**

The following table shows the inputs and outputs from the main steps of the feldspar process.

Process step	Inputs	Outputs
1. Milling & classifying	▪ raw material ▪ water	▪ slurry mixture (containing feldspar) ▪ coarse sand, gravel, and stones
2. Hydrocycloning	▪ slurry mixture ▪ water	Overflow ▪ feldspar, fine sand and micas Underflow ▪ gangue: concentrated sand ▪ process water
3. Dewatering by screens or vacuum filters	▪ feldspar, fine sand and micas	▪ feldspar, fine sand and micas ▪ process water
4. Micas or oxides flotation	▪ feldspar, fine sand and micas ▪ foam inhibitor ▪ acids (H_2SO_4) ▪ surfactants	Overflow ▪ micas or oxides Underflow ▪ feldspar, fine sand, quartz ▪ process water
5. Dewatering by screens or vacuum filters	▪ output from the underflow of the previous step	▪ feldspar, fine sand, quartz ▪ process water
6. Feldspar flotation	▪ feldspar, fine sand, quartz ▪ foam inhibitor ▪ acids (HF) ▪ surfactants	Overflow (reverse flotation possible) ▪ feldspar Underflow ▪ fine sand and quartz ▪ process water
7. Dewatering by filters	▪ output from the overflow of the previous step ▪ feldspar (moisture <25 %)	▪ feldspar (moisture <25 %) ▪ process water
8. Drying	▪ feldspar (moisture <25 %)	▪ feldspar (moisture <1 %)
9. Magnetic separation	▪ feldspar (moisture <1 %)	▪ feldspar (moisture <1 %) ▪ iron oxides

Table 3.71: Inputs and outputs from feldspar mineral processing steps
[39, IMA, 2002]

At the operations in the **Segovia region** and in **Finland,** the process used for the separation of the feldspathic sands from the silica sands is that of flotation in a highly acid environment for which hydrofluoric acid is used. The flotation plants are fed with the fraction smaller than one millimetre. The mineral processing plants have a capacity of 2400 t/d.
[110, IGME, 2002]

3.2.3.3 Tailings management

3.2.3.3.1 Characteristics of tailings

An example chemical analysis of a tailings eluate is presented below:

Parameter	Units	Result
pH- eluate after 2 hours	-	7.76
pH- eluate after 8 hours	-	9.06
pH- eluate after 24 hours	-	9.14
pH- eluate after 48 hours	-	9.20
pH- eluate after 72 hours	-	9.04
pH- eluate after 102 hours	-	9.03
pH- eluate after 168 hours	-	8.5
pH- eluate after 384 hours	-	8.0
Cyanide	μg/l	<10
Chloride	mg/l	<10
Fluoride	mg/l	<0.5
Nitrate	mg/l	23
Sulphate	mg/l	101
Arsenic	μg/l	<5
Barium	mg/l	<0.1
Cadmium	μg/l	4
Cobalt	μg/l	<100
Chromium	μg/l	14
Beryllium	μg/l	<1
Mercury	μg/l	<0.1
Nickel	μg/l	2
Lead	μg/l	19
Copper	μg/l	16
Selenium	μg/l	<1
Vanadium	μg/l	<100
Zinc	mg/l	2.4
COD	mg/l of O_2	27

Table 3.72: Example of a chemical analysis of feldspar tailings eluate

The following table shows the characteristics of the materials released from the process.

Process step	Material released from the process	Destination
Comminution and classifying	▪ coarse sand, gravel, and stones	▪ by-product or tailings heap
Hydrocycloning	▪ concentrated sand ▪ process water	▪ by-product or tailings pond
Dewatering by screens or vacuum filters	▪ clear water overflow is directly recycled or used to hold reserves of water.	
Micas flotation	▪ micas ▪ process water	▪ by-product or tailings pond
Oxides flotation	▪ oxides ▪ process water	▪ tailings pond
Dewatering by screening or with vacuum filters	▪ clear water overflow is directly recycled or used to hold reserves of water.	
Feldspar flotation	▪ fine sand, quartz, and micas ▪ process water	▪ by-product or tailings pond
Dewatering in filters	▪ clear water overflow is directly recycled or used to hold reserves of water ▪ process water, tailings pond	
Drying	▪ non	▪ n/a
Magnetic separation	▪ iron oxides	▪ by-product or tailings heap

Table 3.73: Products and tailings from the mineral processing of feldspar
[39, IMA, 2002]

Besides the tailings heaps consisting of coarse sand, gravel and stones, there are tailings ponds which contain:

Solid materials:

- fine sand and micas (50 – 70%)
- some iron oxides (less than 10 %)
- flocculants (in the ppm range)
- fluoride strongly adsorbed or bounded onto the solids.

Liquid (process water)

- water at a pH value of about 4.5
- foam inhibitor (traces)
- fluoride (100 – 1000 ppm).

3.2.3.3.2 Applied management methods

At most sites the tailings are stored in dug out settling basins within the pit, and thus they do not have dams. The bottoms of the ponds are lined with clay layers.

At one of the operations in Segovia, 110000 t/yr of tailings are generated (mine production 600000t/yr). These consist of a sandy fraction (80000 t/yr) and the tailings after flotation. The sandy fraction consists of coarse sands that do not have a market. They are backfilled in the open pit. The flotation tailings are filtered. The filter cake (28000 t/yr) is also backfilled, whereas the remaining slurry is sent to small ponds. The backfilling area in the open pit had been prepared by placing a drainage system to control and sample the drainage water prior to discharging to the river.

The flotation concentrate is led to a treatment facility that generates 200 t/yr of calcium fluoride sludge from neutralisation of the HF-acid using lime. After filtration in a filter press the sludge is backfilled together with the tailings. The flotation tailings stream is not neutralised directly. Instead the tailings pond has four control wells in its periphery from which the seepage water is pumped to the water treatment plant.
[110, IGME, 2002]

Tailings heaps have a natural slope of 30 to 45°.

3.2.3.3.3 Safety of the TMF and accident prevention

The TMFs are controlled visually and by topographical surveys.

3.2.3.4 Current emissions and consumption levels

3.2.3.4.1 Management of water and reagents

1. Micas flotation:

Chemicals used in the process:

Chemicals	pH/concentration
Acid (H_2SO_4)	To adjust to a pH value of about 3
Surfactant	10 - 100 ppm
Foam inhibitors	10 - 100 ppm

2. Oxides flotation:

Chemicals used in the process:

Chemicals	pH/concentration
Acid (H_2SO_4)	To adjust to a pH value of about 3
Surfactant	10 - 500 ppm
Foam inhibitors	10 - 100 ppm

3. Feldspar flotation:

Chemicals used in the process:

Chemicals	pH/concentration
Acid (HF)	pH <3
Surfactant	10 - 500 ppm
Foam inhibitors	10 - 100 ppm
Alkaline solution (CaO, $Ca(OH)_2$, NaOH)	To adjust to a pH value of about 4.5

Water is neutralised with CaO, $Ca(OH)_2$, Na(OH) to pH values of about 7; using calcium ions there is the advantage that the fluoride is bounded and a larger part of it disappears from the balance because the CaF_2 is almost insoluble. After this treatment, the water is added to the waste water-stream.

3.2.3.4.2 Energy consumption

The average energy consumption for the feldspar mineral process is approximately 300 MJ/tonne. However, large discrepancies have been observed from site to site (min: 10 – max: 1800).

3.2.4 Fluorspar

3.2.4.1 Mineralogy and mining techniques

The chemical element F is not rare in the earth's crust (at 0.07 % it is the 13th most abundant element by weight), but naturally occurring concentrations are scarce. The elements fluorine (F) and calcium (Ca) are strongly bound in CaF_2 and this molecule is very stable.
[43, Sogerem, 2002]

The mineralogy of the Sardinian fluorspar/lead sulphide operation can be described as follows:

- fluorspar, with a grade of 26 – 38 %
- lead sulphide, with a grade of 1.5 – 8 %
- barium sulphate
- zinc sulphide
- iron sulphide, as pyrites and marcasite
- calcium carbonate, as calcite
- quartz
- silicates.

Of the above, only the first two are of economic interest; as the liberation size of 6 mm makes the comminution and separation relatively simple, to pre concentrate the mineral in a static dense medium separtion process [44, Italy, 2002].

Mining is carried out both underground and open pit.

In one operation the underground mining method is applied in a vein, cut and fill mining [44, Italy, 2002].

Fluorite mining in Asturias is carried out in three mines using the room and pillar technique. The deposit is of the hydrothermal type, where $CaCO_3$ has been replaced by CaF_2. About 60000 m^3 of waste-rock are generated in the mining operation each year. This waste-rock is backfilled directly in mined out chambers of the mine [110, IGME, 2002].

3.2.4.2 Mineral processing

3.2.4.2.1 Gravity concentration

At the fluorite mine in the Southern Pyrenees, after crushing to <30 mm, the different components of the ore are separated dense medium separation. This process is capable of upgrading the ore from 30 – 60 % CaF_2 to around 90 % CaF_2.

The gravity concentration, a continuous process, is done in a water environment at ambient temperature in closed circuit (hydrocyclones or drums) with automated regulation. The water is re-circulated in a closed circuit. The washed material is sorted by size (2 mm, 5 mm, 25 mm) and stored outside on concrete surface.

All tailings are subsequently processed in the flotation plant described below to increase recovery. The finished product can be sold in wet form and the delivery to the customers is done in covered dump trucks. If it is delivered dried, transportation is done in covered dump-trucks or in silo-trucks.
[43, Sogerem, 2002]

3.2.4.2.2 Flotation

At the fluorite mine in the Southern Pyrenees, after crushing and grinding, the ore with a fluorspar content around 40 % is reduced in size to particles under 1 mm and is then dispersed in water. The fluorite grains are rendered hydrophobic by the surface action of natural fatty acids (oleic acid for example). The ‘fatty’ particles attach to the injected air bubbles to form a froth that is then mechanically skimmed off at the surface of the cells. This froth, containing mainly calcium fluoride, i.e. 97 – 98 % of CaF_2 (dry basis), is washed several times with water. Filtration of the slurry gives a filter-cake with around 10 % moisture.
[43, Sogerem, 2002]

In Asturias, the ore from three mines, 400000 t/yr, is processed in one plant. The distance from the mines to the mineral processing plant is between 18 and 100 km. The plant includes primary and secondary grinding, fine milling and hot flotation.
[110, IGME, 2002]

3.2.4.2.3 The fluorspar/lead sulphide process

The Sardinian Silius Mine mine produces fluorspar and a lead sulphide concentrate. The average rate of production per year is 45000 tonnes of 97 % CaF_2 and 5000 tonnes of 67 % PbS. Silius Mine is the only operating mine in Europe for Fluorspar and Lead Sulphide. The fluorspar product is sold to a chemical plant and the lead sulphide is sold to a smelter located in South West Sardinia

The ore is pre-concentrated at the mine site using gravity concentration. The pre-concentrate with a fluorspar grade of 43 – 50 % is transported via trucks to the mineral processing plant

57 km away from the mine, the reason for this being the availability of large amounts of water, not available at the mine.

The mineral is ground in ball mills to 100 % passing 0.5 mm. The first mineral recovered is the lead sulphide in a 3-stage flotation unit. The reject of this stage is then processed in a 4-stage fluorspar flotation unit. The commercial products are filtered in drum filters.
[44, Italy, 2002]

3.2.4.3 Tailings management

3.2.4.3.1 Applied management methods

In one operation in the Southern Pyrenees, the tailings, containing 1 to 5 % CaF_2, are backfilled into the mine after dewatering with filter-presses, located inside the plant itself. The water is entirely recycled. The coarseness of the tailings is close to the one of the finished concentrated fluorspar, with a particle size less than 350 µm.

The constituents are silica and shale (80 - 90 % SiO_2), and on a smaller scale iron derivatives (5 - 10 % Fe_2O_3: shales, iron hydroxides, iron carbonate), other oxides (1 – 2 % Al_2O_3), iron/copper sulphides, and of course some residual CaF_2 (usually 1 – 5 %).

In another case, that being the operation in Sardinia, the tailings are cycloned in a dense medium to separate the sands from the muds. The sands are settled in 'sand ponds'. The muds are pumped into 'settlements ponds'.

The process water is cleaned in three ponds. The clean water from the third pond is partially recycled and partially discarded into the river. The total volume of the tailings ponds is about 1300000 m^3.

The dried sands are stocked in heaps and are sold for civil construction works; the muds are under evaluation for new uses such as for tiles, cement.

Further developments aim to eliminate the settlement pond by introducing filter press sections.

The tailings facilities are located near the plant very close to the river. The ground where the facilities are located is an alternation of sands and clay layers, with the result that no seepage into the ground occurs.

A conventional dam with a clay nucleus of the classical trapezoid shape contains the tailings. The dam slope is 1:1.5. The dams are raised every three to four years.

A characterisation of the site is in progress to evaluate the chemical situation, the leaching behaviour, and so on. Alternative solutions to the present management will be decided after the results of the study. An important factor to be considered in these conditions are related to the heavy metal contents and the systems to avoid that those metals can migrate into water and surrounding properties.
[44, Italy, 2002]

The tailings at the operation in Asturias are discarded into the sea after removing the coarse, sellable fraction in hydrocyclones [110, IGME, 2002].

3.2.4.3.2 Safety of the TMF and accident prevention

At the fluorspar/lead sulphide operation, the dam slopes and decant system are checked visually on a daily basis. The water coming from the ponds overflow is chemically checked weekly before discharge into the river. The phreatic surface is controlled by means of piezometers. For safety reasons the dam height is limited to 7 - 10 m.

There are no specific emergency plans because the risk of a heavy accident is considered 'basically zero'.
[44, Italy, 2002]

3.2.4.3.3 Closure and after-care

The closure and after-care plan for the fluorspar/lead sulphide operation is currently in progress. The costs of closure are expected to be in the order of several million EUR. Monitoring of the site after the end of the operational life must be carried out for several years (currently about 10 years are foreseen) in order to establish if any migration of heavy metal occurs. There are no arrangements for financial assurance to cover the long-term risk of pollution, but a special fund has been established by the company in the annual balance to finance the closure operations [44, Italy, 2002].

3.2.4.4 Waste-rock management

One operation backfills all waste-rock along with the tailings in the underground operation. The waste-rock comes from the excavation of galleries in rock mass outside of the orebody. The waste-rock is used as backfill, so that the surface heaps are reduced to a minimum and are only used as a temporary deposit [44, Italy, 2002].

3.2.4.5 Current emissions and consumption levels

3.2.4.5.1 Management of water and reagents

In one case, the clean water from the last clarification pond is partially recycled and partially discarded into the river. The total volume of the tailings ponds is about 1300000 m^3 [44, Italy, 2002].

The water is cleaned before the discharge. The reagents used in mineral processing are of vegetal origin (e.g. oleins from olive or pine oil); potentially dangerous reagents are chemically treated before discharge. The water consumption is on average 8000 m^3 per day. [44, Italy, 2002]

At the operation in Asturias the following reagents are used:

- oleic acid, as a collector and frother, 400 g/t
- quebracho tannin, as a depressant for calcite
- sodium carbonate, as a pH adjuster.

[110, IGME, 2002]

3.2.4.5.2 Soil contamination

At the fluorspar/lead sulphide operation, due to the nature of the material processed, heavy metals contamination could occur. The metals contained are lead, zinc, iron and fluor. However, the concentrations are low and emissions are monitored.

3.2.5 Kaolin

3.2.5.1 Mineralogy and mining techniques

Clay minerals are divided into four major groups. One of these is the kaolinite group. This group has three members (kaolinite, dickite and nacrite) and a formula of $Al_2Si_2O_5(OH)_4$. The different minerals are polymorphs, meaning that they have the same chemistry but different structures. The general structure of the kaolinite group is composed of silicate sheets (Si_2O_5) bonded to aluminum oxide/hydroxide layers ($Al_2(OH)_4$) called gibbsite layers. The silicate and gibbsite layers are tightly bonded together with only weak bonding existing between the layers [37, Mineralgallery, 2002].

Kaolinite can be formed as a residual weathering product, by hydrothermal alteration, and as a sedimentary mineral. The residual and hydrothermal occurrences are classed as primary occurrences and the sedimentary occurrences as secondary.

Primary kaolins are those that have formed in-situ, usually by the alteration of crystalline rocks such as granite or gneiss. The alteration results from surface weathering, groundwater movement below the surface, or due to the action of hydrothermal fluids. Secondary kaolins are sedimentary minerals which have been eroded, transported and deposited as beds or lenses associated with other sedimentary rocks. Most of the secondary deposits were formed by the deposition of kaolinite which had been constituted elsewhere. One type of kaolin deposits which can be considered as either primary or secondary, depending on the point of view, are arkosic sediments which were altered after deposition, primarily by groundwater.

Kaolin is extracted from quarries either by hydraulic means or by simple excavation (e.g. by use of a loading shovel).

3.2.5.2 Mineral processing

The processing of kaolin varies greatly from company to company; with each kaolin producer using different equipment and methods. Even when companies use identical methods, they may use them at different stages of the processing.

Kaolin ore, generally composed of kaolinite, quartz, micas, feldspar residues, etc., is commonly wet processed to eliminate the unwanted minerals. The various steps in the processing are:

- placing the ‘ore’ in suspension with water
- recovery of the kaolin fraction through sieving and cycloning
- concentration of the suspension through decantation in basins followed by passing it through filter-presses.

The kaolin properties (brightness, rheology, purity, grain size distribution) can be improved during the treatment, by using magnetic separation, bleaching or centrifugation.

Comminution is usually not necessary. Sometimes during wintertime, crushers (e.g. jaw crushers, cone crushers, roll crushers, hydrocone, etc.) are used to break frozen raw material.

Coarse clay may be used as a low grade filler or a ceramic clay. Alternatively, it can be upgraded by further processing. The flotation process is used to refine coarse clay and to maximise the recovery of kaolin. It can increase the kaolin recovery yield by up to 15 %, which is a significant improvement in the management of this natural resource. Not all producers use flotation. This depends on the product requirements and the characteristics of the deposit.

The following figure shows a typical kaolin process flow sheet

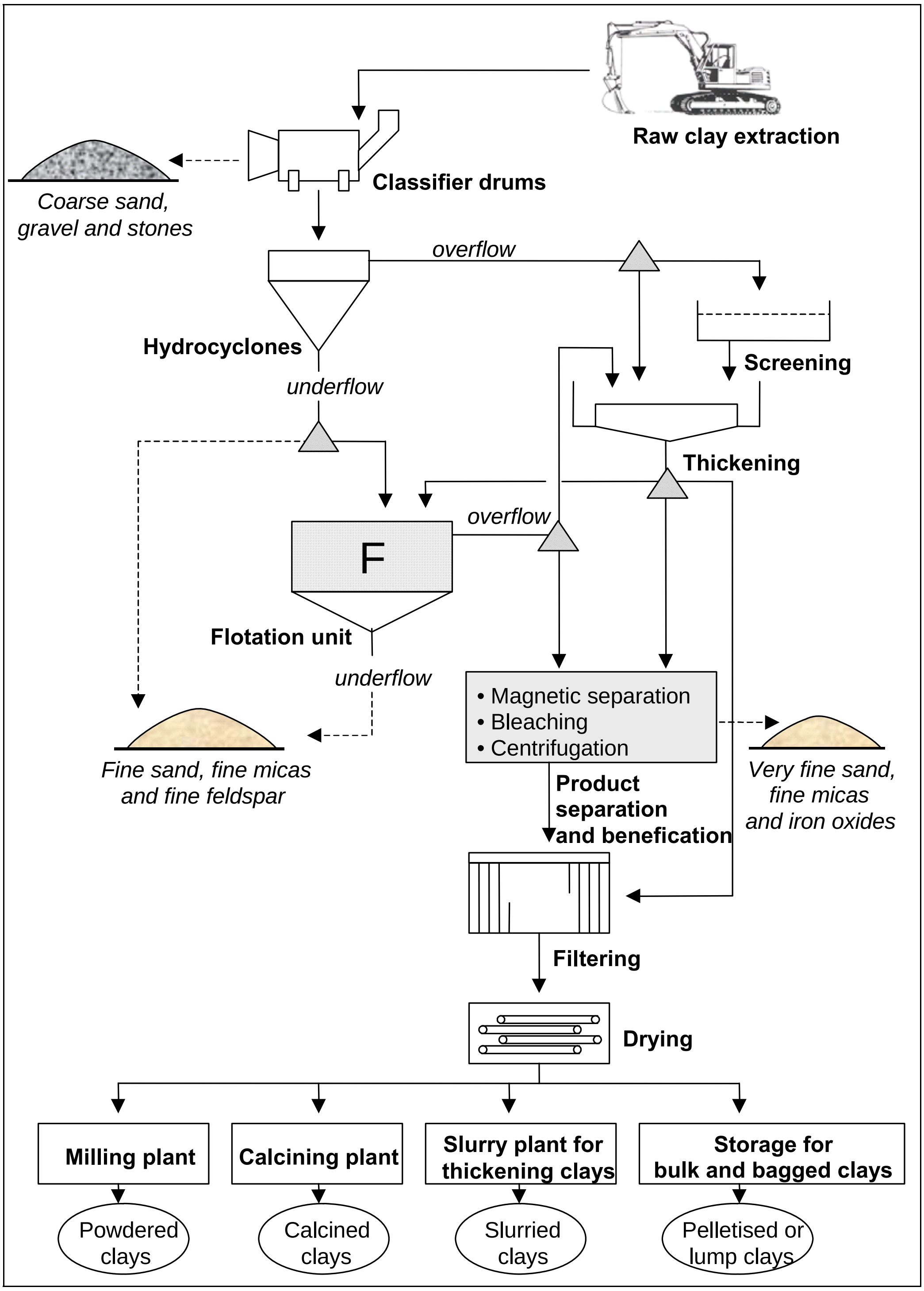


Figure 3.54: Typical kaolin process flow sheet
[40, IMA, 2002]

The essential use of the flotation process can be explained by the following figure:

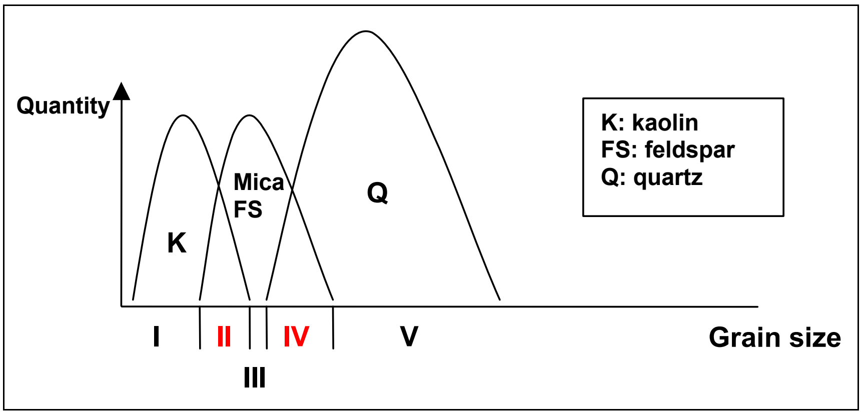


Figure 3.55: Kaolin grain size vs. quantity graph
[40, IMA, 2002]

In Sections I, III, and V, a primary mechanical separation (cycloning, centrifugation) can be achieved.

In Sections II and IV the grain size of different minerals is equal. If there is only a minor difference in specific weight, mechanical separation is not possible. Other differences will then have to be used. At smaller grain sizes (Section II) the only possible separtion method is flotation. At larger grain sizes, Section IV, other methods, such as electrostatic separation of feldspar, is possible.

The following table shows the inputs and outputs from the main steps of kaolin processing.

Process step	Inputs	Outputs
Classifying	Raw material Water	Coarse sand, gravel and stones Slurry mixture (containing kaolin)
Hydrocycloning	Slurry mixture Water	Overflow Kaolin + fine sand, micas, (and feldspar) Underflow Kaolin + fine sand, micas, (and feldspar) Process water
Flotation	Underflow from the hydrocycloning step, or kaolin concentrate Acid (H_2SO_4, H_3PO_4) Surfactants Anti-foam chemicals Alkaline solution (NaOH)	Overflow Kaolin mixture (after acid neutralisation) Underflow Very fine sand, micas, (and feldspar) Process water
Thickening	Overflow from the hydrocycloning step or flotation Flocculent	Kaolin concentrate (15 – 30 % solid content)
Product separation	Kaolin concentrate, or kaolin mixture Magnetic separation Bleaching Sodium hydrosulphite Ozone gas Centrifugation	Kaolin Iron oxides (very small amount) Very fine sand and micas
Filtering	Kaolin, kaolin concentrate	Kaolin (moisture <18 %) Process water
Drying	Kaolin (moisture <18 %)	Kaolin products

Table 3.74: Inputs and outputs in the processing of Kaolin
[40, IMA, 2002]

3.2.5.3 Tailings management

3.2.5.3.1 Characteristics of tailings

Characterisation of the materials released from the process

Process step	Material released from the process	Destination
Classifying	▪ coarse sand, gravel and stones	▪ heap or saleable products (if local market available)
Hydrocycloning	▪ fine sand, micas, (and feldspar) ▪ process water	▪ if it contains feldspar, it is further refined in the feldspar process ▪ mica is a commercial product ▪ fine sand: heap or saleable products (if local market available) ▪ tailings pond
Flotation	▪ very fine sand, micas, (and feldspar) ▪ process water	▪ tailings pond ▪ if it contains feldspar, it is further refined in the feldspar process
Thickening	Clear water overflow is directly recycled or used to hold reserves of water.	
Product separation	▪ very fine sand and micas ▪ iron oxides	▪ tailings pond or ▪ heap (compared to the other outputs, the amount is here negligible - several orders of magnitude less)
Filtering	▪ process water	▪ tailings pond ▪ the filtrate ("process water") can also be recycled (depends on applied flocculants)
Drying		

Table 3.75: Tailings and products from Kaolin mineral processing
[40, IMA, 2002]

Beside the heap of coarse sand, gravel and stones, there are tailing lagoons which contain:

Solid materials:

- fine sand and micas (more than 95 %)
- some iron oxides (less than 1 %)
- flocculants (in the ppm range).

Liquid (process water)

- water at a pH value of about 4.5
- some phosphates
- some sulphates
- foam inhibitor.

3.2.5.3.2 Applied management methods

Beside the heaps of coarse sand, gravel and stones, there are also tailings ponds for the fine tailings. These are a mixture of fine clay particles (95 % of the solid content) associated with some surfactants and foam inhibitors (in the ppm range) in an acidic solution (pH of about 4.5). Usually, tailing ponds are used to clean the water before recycling or discharging to the river. The ponds are lined with impermeable clay layers.

In the **Nuria** operation, the tailings are the ultrafines after classification (2 % of total feed). Flotation is not applied. These fines are dewatered in several concrete settling basins in series (each with a size of about 300 m^2). The basins are dewatered with syphons. In the summertime the dewatered fines are transferred to the waste-rock heap [110, IGME, 2002].

The **Kernick** mica dam is a micaceous tailings facility for the china clay (kaolin) industry in Cornwall, UK. It has been in use for 30 years and is one of the largest tailings dams in Europe. It occupies an area in excess of 55 ha and is 92 m high (above lowest ground level). The dam contains approximately 14 million tonnes of bulk fill which impounds approximately 28 tonnes of micaceous tailings. The structure consists of an embankment constructed around the perimeter of a worked-out china clay pit (quarry) which had been previously backfilled with micaceous tailings. The purpose of the embankment is to impound the tailings above the rim of the quarry.

The china clay industry generates three main types of residues from the deposit matrix:

- waste-rock, known locally as 'stent' which is a mixture of unkaolinised granite and other hard mineral lodes removed by drilling and blasting
- sand tailings, a coarse grained silica sand removed by mechanical separation
- mica tailings, a residue of mica and very fine sand removed by flotation.

The sand tailings and the waste-rock have been used to construct the dam in specific zones separated by transition layers. The waste-rock, evenly graded between 50 mm and 750 mm in size, forms a central core for the capture and drainage of seepage through the structure. The sand tailings, containing no material larger than 150 mm but typically less than 25 mm grain size, forms both the downstream and upstream parts of the main dam. The transition layer, consisting of clean, crushed rock typically between 75 mm and 125 mm, forms a filter layer between the sand tailings and the waste-rock core.

The embankment structure sits on a prepared ground surface which was stripped of all vegetation, topsoil, weathered profile and soft material. The excavation was proof-rolled by vibrating rollers and backfilled with clean sand in order to establish an even working foundation. A 1 m thick drainage blanket of clean stone was laid beneath the entire length and breadth of the rock core and downstream embankment. This blanket incorporates a longitudinal cut-off trench at the base of the rock core in which are situated a number of reinforced concrete (inlet) manifolds. The manifold in return are connected to reinforced concrete conduits used to transmit seepage water beyond the toe of the structure into collector chambers prior to final discharge to the adjacent watercourse.

During construction, the embankment site was protected (separated) from the quarry backfilling operation by a coffer dam built of randomly placed waste materials.

The downstream and upstream sand tailings embankments have been raised in horizontally placed layers approximately 0.5 m thick and compacted by vibrating rollers. The waste-rock core was 'free-tipped' by dump trucks to achieve an even distribution, and has not been compacted (other than by the weight and passage of bulldozers used to level the surface). The transition layer was placed by mechanical shovel to achieve a maximum thickness of 3 m.

The outer face of the embankment has a designed profile of 35°/32° (1:1.5/1:1.7 (V:H)) to which has been added a thin veneer of topsoil as a growing medium for subsequent vegetation. A hydroseeding technique is used to spray the surface with a mixture of grass, legumes, fertiliser, lime and organic binders, which together progressively establish a dense growth of gorse/lupin scrub, typical of unfarmed areas in the south-west of England.

Deposition of the tailings is carried out by using pipelines and spigots around the entire crest of the dam. The hydraulic separation leaves the coarser mica closer to the inside face of the dam with finer particles gradually settling out towards the back-end of the pond, where the free water is decanted by a pump barge.

Decanted water is either:

- re-circulated back into the process operation, or
- released to the watercourse (together with sub-pond drainage).

The performance of the structure (stability) is monitored by survey monuments to observe any horizontal/vertical movement, by piezometers to measure phreatic seepage patterns within and below the embankment, and by weirs to measure gross ground water flow through the final discharge flume.

Additional storage capacity is currently being achieved by surcharging the pond with bunds of compacted sand, placed directly on the ‘dry’ beach - this also creates a landscaped profile to the final surface of the lagoon which will eventually be dewatered and vegetated.
[125, Grigg, 2003]

3.2.5.3.3 Safety of the TMF and accident prevention

The TMFs are controlled visually and by topographical surveys.

3.2.5.4 Waste-rock management

The Nuria operation operates a waste-rock heap of 2.8 Mm^3. The foundation of this heap was first stripped of the topsoil before a drainage system (consisting of perforated pipes covered with gravel and a geotextile) was installed. The surface run-off, containing a large amount of fines, is gathered and collected in a series of sedimentation ponds. The bench height is 15 m with 10 m wide berms [110, IGME, 2002].

3.2.5.5 Current emissions and consumption levels

3.2.5.5.1 Management of water and reagents

The reagents used in the flotation of kaolin are listed in the following table.

Reagent	Average concentration
Acid (H_2SO_4, H_3PO_4)	To reach a pH value of about 2.5
Surfactant	10 - 100 ppm
Foam inhibitors	10 - 100 ppm
Alkaline solution	To neutralise to a pH value of about 4.5

Table 3.76: Reagents used in the flotation of kaolin
[40, IMA, 2002]

3.2.5.5.2 Energy consumption

The average energy consumption for the kaolin mineral process is about 2000 MJ/tonne.
The average diesel consumption of a truck is 25 l/h.

3.2.6 Limestone

3.2.6.1 Mineralogy and mining techniques

From a mineralogical point of view, calcium carbonate falls into three structurally different groups: the calcite and the aragonite groups (both $CaCO_3$), and the dolomite group ($CaMg(CO_3)_2$). Calcite ($CaCO_3$) crystallises in the hexagonal system, but its crystals are extremely varied in habits, and often highly complex. The rhombohedron and the scalenohedron are the most frequent forms. Calcite is one of the most common and widespread minerals on earth, particularly in sedimentary rocks. Aragonite ($CaCO_3$) is formed in a narrow range of physico-chemical conditions. It crystallises in the orthorhombic system, typically in thermal springs. However, aragonite is also formed through biomineralisation processes; mollusc shells, pearls, and the human skeleton is made of aragonite. Dolomite is a double carbonate of calcium and magnesium, with the formula $CaMg(CO_3)_2$. Like calcite, it crystallises in the hexagonal system. It forms by the secondary transformation of calcite sediments in limestone, under the influence of circulating water, through partial substitution of Ca by Mg. These minerals constitute rocks, of which chalk, limestone, marble, and travertine are the most important ones. Chalk is a poorly compacted sedimentary rock, whose diagenesis is incomplete, and which is almost exclusively made up of calcium carbonate (calcite). The sediments from which chalk originates predominantly include compacted coccolithophoridae skeletons (calcareous algae) with limited cement, if any. This rock shows a very fine grain size, and is porous. Limestone is generally used as a generic term which designates a compacted sedimentary rock made of calcium carbonate. It is often used as a synonym for natural calcium carbonate. Marble is a metamorphic rock, which is the result of a re-crystallisation process of limestone, under conditions of high pressure and temperature. True marble has a low porosity and may host calcite crystals of several centimetres. Travertine, which is also called "calcareous tuff" or "spring deposit tuff", results from the chemical or biochemical precipitation of calcium carbonate in thermal springs, as calcite or sometime as aragonite. All these minerals, when of the highest quality, are the source of industrial calcium carbonate.
[42, IMA, 2002]

Limestone is almost exclusively mined in open pits.

The limestone in Flandersbach has the following parameters:

- 97 – 98 % $CaCO_3$
- <1 % $MgCO_3$
- <1 % SiO_2 (quartz)
- sometimes a higher content of shale or mud is included.

[107, EuLA, 2002]

3.2.6.2 Mineral processing

Limestone

At the **Flandersbach** quarry, after blasting, the limestone is transported by trucks to the crusher. There, the waste-rock is separated and dumped into another mined out quarry. The limestone goes to the mineral processing plant, which is essentially a washing plant for separation of ‘mud’ sediment from the limestone. The slurry after the washing plant is pumped into the tailings pond, another nearby mined out quarry.

The amount of raw material from the quarry is between 7 and 8 million tonnes/yr. Nearly 10 % of this raw material is waste-rock. Another 10 % is ‘mud’ sediment which is separated in the washing plant. The amount of sediment pumped into the tailings pond is, therefore, nearly 700000 t/yr. For every tonne of washed limestone 1 m^3 of process water is required.
[107, EuLA, 2002]

Calcium carbonate
The vast majority of the mine production is marketable, as can be seen in the following table.

	Amount (kt)	Per cent
Ore from the quarry (natural calcium carbonate)	16655	100.0
Stock for sale	16100	96.7
Tailings released to the outside	75	0.4
Dust managed on-site	111	0.7
Tailings managed on-site for the rehabilitation of the quarries	369	2.2

Table 3.77: Production figures of calcium carbonate in the EU in 2000

Tailings released to the outside:
These tailings include the flotation residues with the mica (such as phlogobite, biotite, muscovite) and graphite impurities. They are sometimes settled in ponds or directly released to the recipient.

Dust managed on-site:
This dust includes all the tailings coming from the various dust collectors and cleaning systems in the plant bagging stations, etc.

Tailings managed on-site for the rehabilitation of the quarries:
This kind of material consists mainly of off-colour production or ground fillers and pigments outside of the product specification.

The production of Ground Calcium Carbonate (GCC) starts with its extraction. Identifying the right orebody in terms of composition, homogeneity, etc. is essential to the whole production process that will follow; a pure calcium carbonate source needs to be identified. Generally, the processing includes washing, sorting of undesirable by-minerals, grinding, size classification of particles and possibly drying. Depending on the circumstances and intended uses, the order and necessity of those different steps vary. At the outlet of the process, the material is delivered in bags or in bulk (trains, boats, trucks) when dry, or as bulk container from slurries. GCC results directly from the exploitation of pure calcium carbonate ore bodies (ore grade >96 %). The production process maintains the calcium carbonate very close to its original state, resulting in a finely ground product delivered either in dry or slurry form. Blasted raw marble is pre-crushed, and depending on the geology washed and sometimes screened. The fines are normally sold for different applications, such as road making, cement mills etc.

In the dry process, calcium carbonate is ground in ball mills, classified and stored in silos, or bags, before shipped by railway wagons or trucks. The products are mainly used in paint and plastics industries, minor applications are in the chemical industry, for fertilising and desulphurisation. Fillers and pigments for the paper industry are produced as slurries, which are finely dispersed calcium carbonate in water. Crushed material is ground with water in rod mills, or ball mills in open or closed circuit, classified and stored in silos before loaded onto railway wagons or trucks.

Due to the geology and mineralogy some calcium carbonate deposits contain unwanted minerals such as graphite, mica, or schist. To remove these natural impurities, selective mining and optical separation are developed together with other mineral processing steps in order to meet the requirements of the customers. Such mineral processing systems can be flotation or magnetic separation.

When magnetic minerals are bound to the marble, magnetic separation is a successful method to separate those "impurities".

Gangue minerals such as mica (such as phlogobite, biotite, muscovite) lead to abrasion in the paper producing machines, while graphite leads to a grey colour in the pigments. Therefore, product requirements impose to separate these minerals during the production process of the aqueous dispersion by means of flotation. The thickened concentrate is normally dewatered in filter presses.

As with all minerals the flow sheet for the production of calcium carbonate fillers and pigments must be adjusted according to the mineralogical characteristics of the calcium carbonate deposits.

The following figure shows an example calcium carbonate process flow sheet.

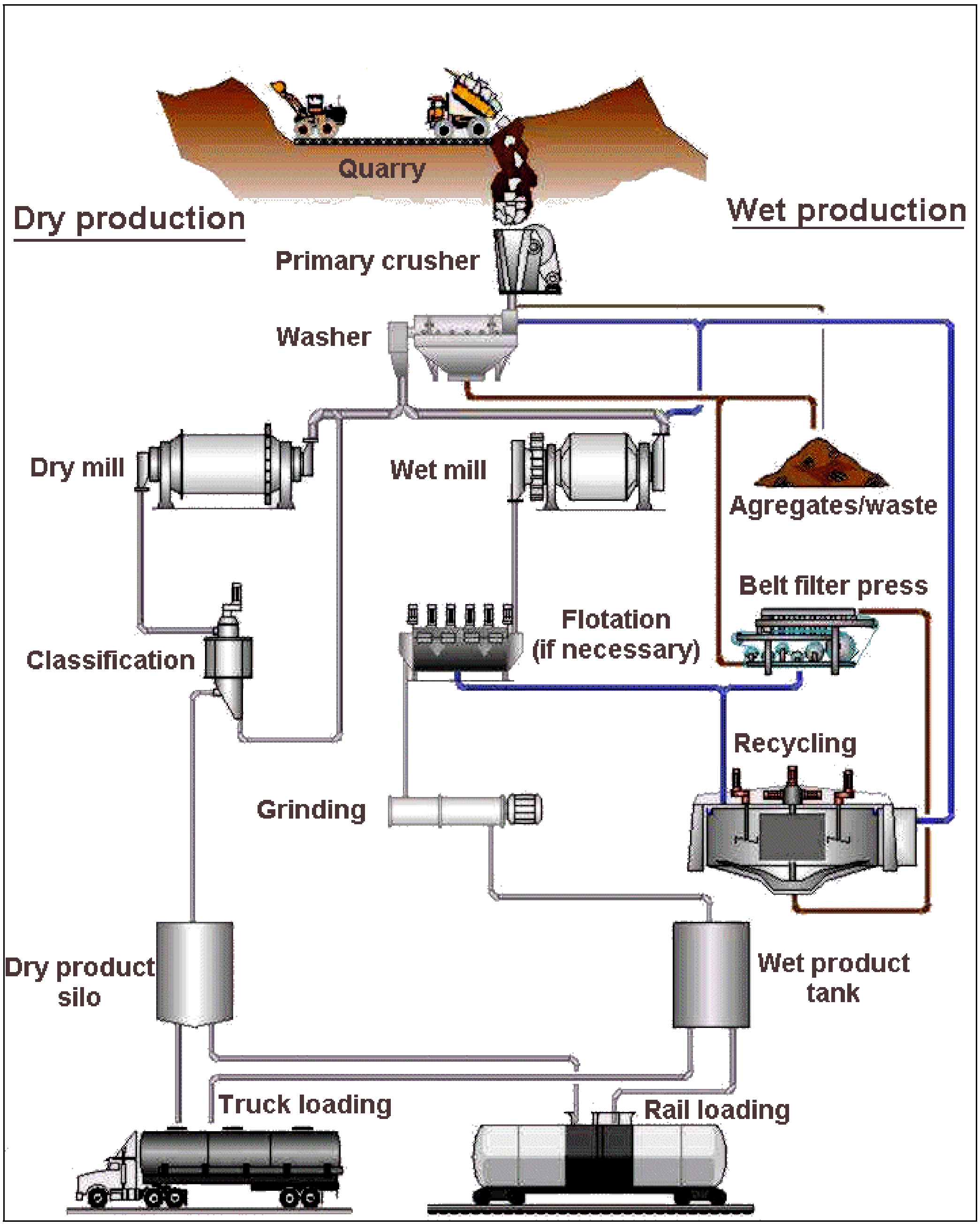


Figure 3.56: Calcium carbonate process flow sheet
[42, IMA, 2002]

3.2.6.3 Tailings management

3.2.6.3.1 Characteristics of tailings

Limestone tailings are a mixture of calcite, dolomite, wollastonite and other very insoluble silicates and very small amounts of heavy metals. The grain size of the tailings is usually less than 0.25 mm.

3.2.6.3.2 Applied management methods

Limestone
The tailings pond of the Flandersbach quarry is installed in a mined-out quarry. The area today is 27 ha. The area in the future will be about 60 ha. The total capacity is over 30 Mm^3. The pond is located close to the mineral processing plant. The pipes for the process water to the pond and for the clarified water back to the mineral processing plant have a length of about 1 km. There is also groundwater inflow into the pond from dewatering of the working quarry. Surplus water is led into a nearby river.
[107, EuLA, 2002]

At the Münchehof quarry the tailings are stored in a pond surrounded by a dam. The following monitoring scheme is applied:

- groundwater level around the dam (monthly measurements)
- phreatic surface in the dam
- seepage water measurements (in a sump from which all drainage water is pumped collectively)
- surveillance of the dam crest and downstream dam toe
- water level in the dam (measured continuously)
- visual inspection by trained staff.

The monitoring scheme is designed in a way that changes of the dam can be recognised in time so that appropriate measures to maintain the stability of the dam can be initiated.
[108, EuLA, 2002]

Calcium carbonate
The calcium carbonate industry uses tailings ponds from which the water is re-circulated to the mineral processing plant. The tailings are a saleable by-product. As far as possible waste-rock and dry tailings are also sold for other applications such as road making, cement and concrete manufacturing, but when there is a lack of customers, those aggregates have to be brought to heaps.

Prior to discarding, the ground is investigated in order to check whether the geology, hydrology, environmental issues and stability fit the requirements set up by the competent authorities. These studies are essential to get the permission for a heap from the competent authorities. The waste-rock and tailings are discharged together in horizontal layers. The end benches are immediately covered with soil and reclaimed with grass and trees according to long-term recreation plans. The evolution of the heap is monitored as well as water quality, groundwater level, and the slope stability if relevant or required by the authorities.

Slurried tailings are either:

- dried (thickener and filter press) and discarded on a tailings heap, or
- discharged to the outside water system (effluent) under conditions controlled by the competent authorities, or
- discharged into a tailings pond (one instance in Europe).

In the latter case the quality of the mineral deposit is such that about one third of the quarried stone is not suitable to the mineral processing plant and was used to construct the 16 m wide starter dam after removal of the huminous material. The slope of the starter dam was 1:1 and the impermeable core is protected against erosion by a layer of 1 - 2 m of 0 - 20 mm material. The impermeable core consists of 2 - 3 m of clay surrounded by a membrane.

Eventually the dam was raised. The starter dam was broadened (+ 12 m) and its height increased (+ 5 m).

Today the total area of the clarification pond is about 45 ha. All tailings discharged at the same point into the pond (single-point discharge). The seepage water through the dam is gathered and pumped back into the pond or, if the free water level in the pond is too high, it is discharged in a controlled manner (quality and amount) into the sewer system, from where it is further discharged into the municipal sewer system.

When the level of the flotation sand rises to a certain level, the discharge is moved and the dry flotation sand is excavated and sold. According to analyses of flotation sand (NEN 7341, NEN 7343 and ISO 11466), the contents of heavy metals are negligible. Also the concentration of flotation reagents is very low and they are very tightly fixed on the mineral particles but easily decompose if liberated.
[42, IMA, 2002]

3.2.6.3.3 Safety of the TMF and accident prevention

The permitting procedure for the TMF at the Münchehof quarry included, according to DIN 19700 T 10, a proof of stability of the dam including static and hydraulic aspects.

The stability calculation is carried out with the following elements:

- geotechnical and hydrogeological modelling
- slope stability
- shear strength
- base failure safety
- safety against pore pressure build-up in the foundation
- overtopping and erosion stability.

Another essential requirement for the dam stability is the suitability of the dam construction material. This is investigated in geotechnical tests. The following parameters are examined:

- friction angle
- specific density
- compressibility
- water content.

During the construction phase quality management was applied to ensure that the parameters that are crucial for the stability of the dam were met. This applies to dam foundation, the dam body and the dam core.
[108, EuLA, 2002]

The control and monitoring of the tailings facilities is done by both industry and the competent authorities. All the constructions (plans, design, etc.) must receive prior approval by the competent authority. The dams are checked every day and all possible changes in the constructions are marked in the control diary. If any leak is noticed, it will be instantly repaired and the information will be sent to the authority. An in-depth inspection is done yearly, and the authority audits the constructions and the record-keeping every five years.
[42, IMA, 2002]

3.2.6.3.4 Closure and after-care

Upon closure of the TMF, the ponds are dewatered and covered with a vegetative cover. [108, EuLA, 2002]

3.2.6.4 Waste-rock management

At the Flandersbach quarry, waste-rock is separated before the washing and dumped into an old quarry [107, EuLA, 2002].

3.2.6.5 Current emissions and consumption levels

3.2.6.5.1 Management of water and reagents

Due to the circulation of process water the consumption of fresh water is low, since only the pore water attached to the product end the evaporated water is lost. The addition of fresh water strongly depends on the climatic conditions (evaporation and rainfall). The Münchehof quarry, for example has to add 437 m^3/d for 23000 m^3 (dry basis) of tailings.
[108, EuLA, 2002]

3.2.7 Phosphate

All information from [143, Siirama, 2003].

3.2.7.1 Mineralogy and mining techniques:

The Siilinjärvi mine is located in Eastern Finland 400 km north-east from Helsinki. The known ore deposit is 16 km long and up to 800 m wide, and is an almost vertical outcrop.

Besides the phosphate mineral apatite (10 %), the ore consists of phlogopite mica (65 %), carbonates (20 %) and silicates (5 %). The ore quality varies strongly throughout the ore body. Apatite is distributed quite evenly through the deposit, but the mica and carbonate distribution varies significantly. Siilinjärvi is one of the world's poorest deposits to be exploited; the average P_2O_5 content in situ is 4 %.

The mining in the open pit is done in 14-metre wide benches. Drilling is done by hydraulic top hammer drilling machines using mainly 203 mm diameter boreholes. Transportation of the blasted ore to the processing plant is carried out by 100-tonne dumper-trucks.

3.2.7.2 Mineral processing

The Siilinjärvi mineral processing flow sheet is shown in the following figure.

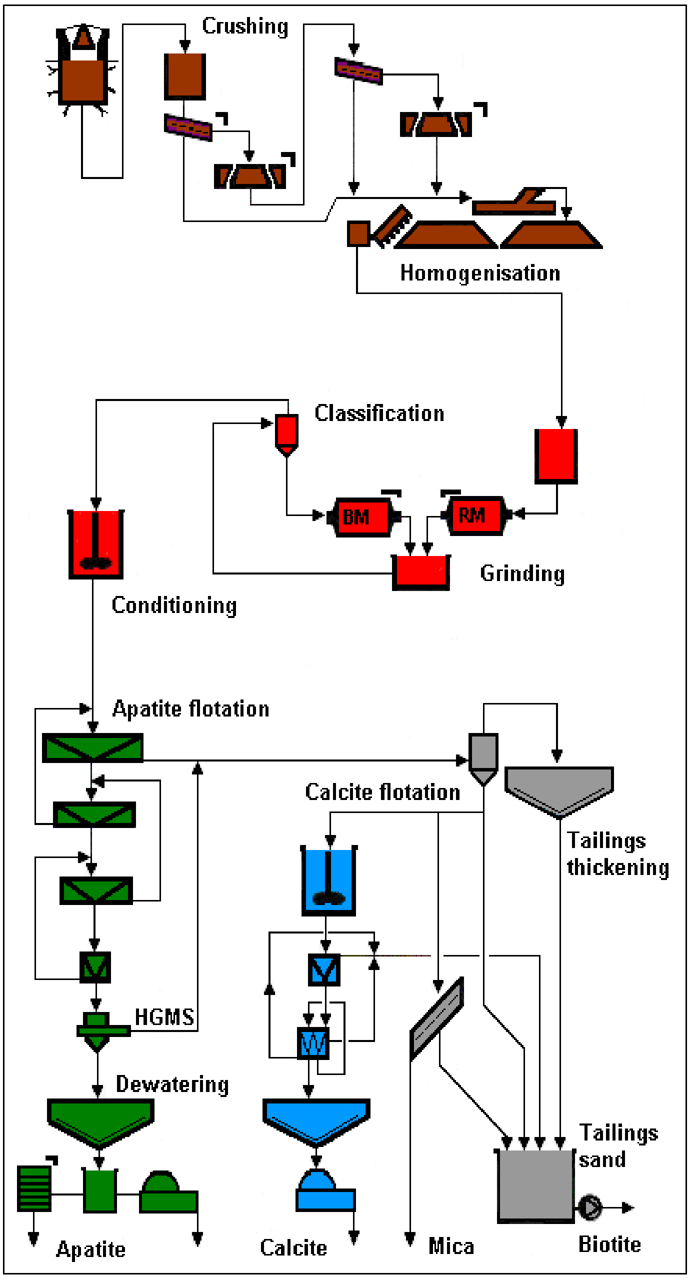


Figure 3.57: Flow sheet of the Siilinjärvi mineral processing plant

Blasted ore is first crushed in three steps, and after homogenisation ground in rod mills (RM) and ball mills (BM). Subsequently, the apatite mineral is recovered by flotation, cleaned and dewatered, before the concentrate is transported by trucks to the phosphoric acid plant. Calcite is removed from apatite tailings, as are mica and other micaceous products. Tailings are pumped to the tailings dam area.

3.2.7.3 Tailings management

There are two tailing dam areas at Siilinjärvi. One is Raasio dam area (150 ha), which was used during the start-up phase of the operations, but today is only only used as a temporary stand-by pond and as a part of the closed water circulation system. The TMF which has been used since 1982 is the Musti dam area (over 800 ha).

The Musti area is located 5 km from the mineral processing plant and is on off-valley type dam, built on a sloped landscape (east side up to 30 m higher than the west side). Due to the repeated dam raises, almost the entire facility is now surrounded by the dam.

The tailings within the pond are crushed and milled rock (i.e. sand), consisting mainly of phlogopite mica, which can be considered inert. After settling, the clarified water is pumped via Raasio back to the process water pumping station from where the plant gets its process water, with the surplus water being pumped via chemical treatment to the nearby lake. Water pumped to the lake is treated with water purification chemicals, with a pH reduction to 7 to allow for efficient sedimentation of the solids.

The dam is a staged conventional downstream type (see Section 2.4.2.2), built out of moraine, with crushed waste-rock as a filter and blasted rock as support part.

The operation of the tailings dams at Siilinjärvi includes the following programmes and routines:

- control programmes:
  - water level controls on-line and monitored, with alarms in the plant operating system
  - regular measures of the amount of circulating and surplus waters
  - daily inspection of the area
  - seepage measurements
  - dam movement measurement
- risk assessments:
  - according to Finnish dam safety law
- ensuring continuity throughout the mine life:
  - planning ahead 10 - 15 years
  - continuously carrying out dam construction programmes and filling estimates
  - owning the land
  - applying for permissions years in advance
  - maintaining a good relationship with the permitting authorities and also the people living around the mine
- using the downstream method to raise the dams
- emission controls:
  - water quality control in seepage, surplus and circulation water
- continuous free water surface control (amounts and quality)
- emergency plans:
  - based on Finnish dam law, a simulation of a total collapse has been done together with permitting and rescue authorities

3.2.7.4 Waste-rock management

The waste rock from the open pit is used as a raw material of crushed rock products or as a structural material in soil engineering constructions (roads, dams, railroads). The excess of waste rock is stockpiled to certain areas around the open pit.

The waste rock stockpiles are landscaped following a landscaping plan, which is used during piling. The landscaping plans have been done together with local authorities and with the input of people living around the mine.

3.2.7.5 Current emission and consumption levels

Emissions to air are not measured, but observations of dusting are recorded.

The excess water, which cannot be returned to the mineral processing plant, is discharged into the river system, where the phosphate load and BOD and solids are measured. The half-yearly floating average in these watercourses is about 1.5 kg of phosphate per day.

3.2.8 Strontium

3.2.8.1 Mineralogy and mining techniques

There are two open pit mines in the south Granada area in Spain. In one case the orebody is very pure and massive. The ore is extracted using drilling and blasting. At the other site the deposit is irregular and not as pure. There, the ore is mined selectively with excavators, so that practically no waste-rock is generated.
[110, IGME, 2002]

3.2.8.2 Mineral processing

The ore from the pure massive orebody is of such high grade that only classification is needed to obtain the final product.

At the other operation the characteristics of the deposit require the installation of a mineral processing plant incorporating grinding, classification and concentration. The latter is carried out by dense media, to obtain a pre-concentrate, and finally fine grinding and flotation.
[110, IGME, 2002]

3.2.8.3 Tailings management

There are two types of tailings from the mineral processing step at the Granada sites: One coarse fraction from the dense-media pre-concentration and the fines tailings from flotation.

The coarse tailings are backfilled into the open pit where they are used in the site restoration.
The flotation fines, in the form of a slurry are managed in a tailings pond. At the pond currently in operation, the tailings are cycloned, with the coarse fraction being used in the structural zone of the dam, while the fines are discharged into the pond (see figure below). The current pond, with a surface area of 14 ha, 17 m height and containing 700000 m^3 of tailings, will soon be replaced by a new impoundment.

This new construction follows a completely different approach, namely that:

- a flat area has been excavated on a hillside
- the dam has been constructed to its final height using the excavated rock and borrow material
- the foundation of the new pond has been lined with PVC, under which has been placed another geotextile layer to protect the liner from possible punctures by direct contact with the natural bedrock.

With a total capacity of 800000 m^3, this new TMF has an expected lifetime of 10 years.

The following picture illustrate the old and the new site.

Figure 3.58: Old strontium TMF with tailings in structural zone
[110, IGME, 2002]

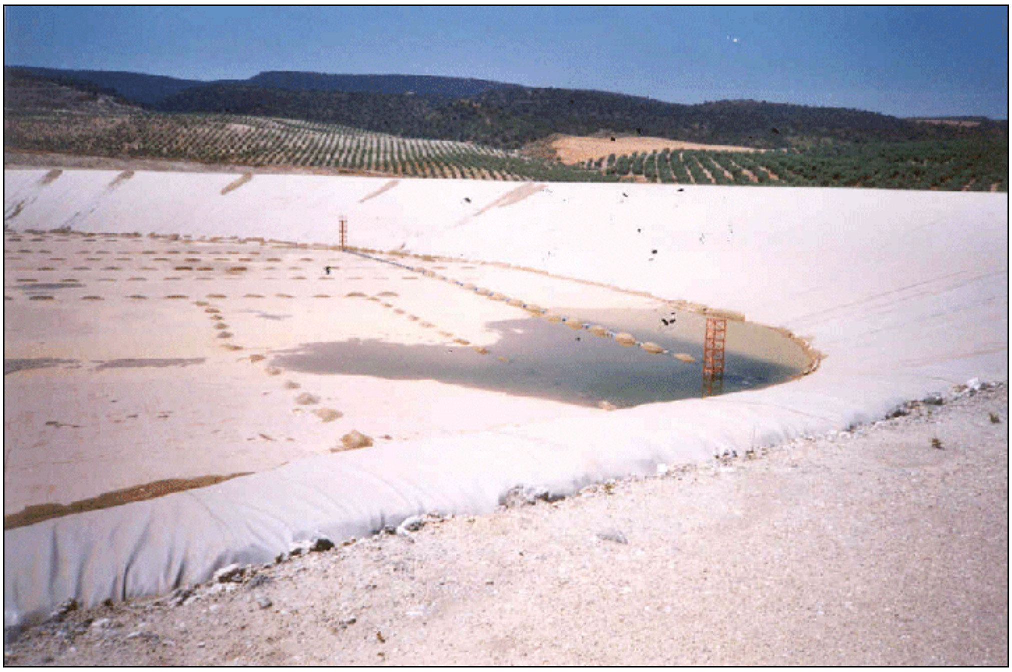


Figure 3.59: New strontium TMF with a synthetic liner and decant towers
[110, IGME, 2002]

3.2.9 Talc

3.2.9.1 Mineralogy and mining techniques

Talc is a hydrated magnesium silicate; it is the softest mineral known in nature. Talc occurs mainly in two forms: schistose talc-magnesite rock and massive pure talc. There is no specific mining technique for the excavation of this kind of mineral, because the choice of the technique depends on the structure of the orebody.

Talc deposits in Finland are located on the early Proterozoic schist belt in Eastern Finland. Talc deposits are related to Mg-rich ultramafic rocks which have been altered to talc-carbonate rocks. The schist belt is about 2 billion years old and the talc was formed during the Svecokarelian orogeny some 1.8 billion years ago. Talc is extracted from a talc magnesite rock which is mainly composed of talc, carbonates (magnesite and dolomite), chlorite and sulphide minerals. Oxides and sulpharsenides are present as trace minerals. The amount of talc varies from 45 to 60 % and carbonates from 35 to 45 % while chlorite (5 %) and sulphides (1-3 %) are just minor components. Some parts of the deposits are relatively sheared where the talc ore is also schistose and fine grained. Talc is typically fine-grained (0.05-0.2 mm) and platy, chlorite occurs in a similar form while carbonates are much coarser (up to several mms or cms in diameter). On the other hand, some parts are massive with relatively coarse grained talc (up to 1 millimetre) and carbonates. Talc carbonate rock is typically greyish occasionally with a greenish or reddish colour, whereas talc itself is typically greenish or very pale, almost white mineral. Talc ore must be ground before the flotation to liberate different minerals and flotation is needed to achieve a high purity and brightness of the end-product.

3.2.9.2 Mineral processing

When using dry processes (67 % of the European production), no tailings are generated. All the raw materials are used and sold with different grade specifications. The flotation process is only used to treat the Finnish ores, which represent about 33 % of the total European talc production. The use of the flotation process is imposed by the characteristics of the Finnish deposits.

The following flow sheet shows the process for the Finnish operation using flotation.

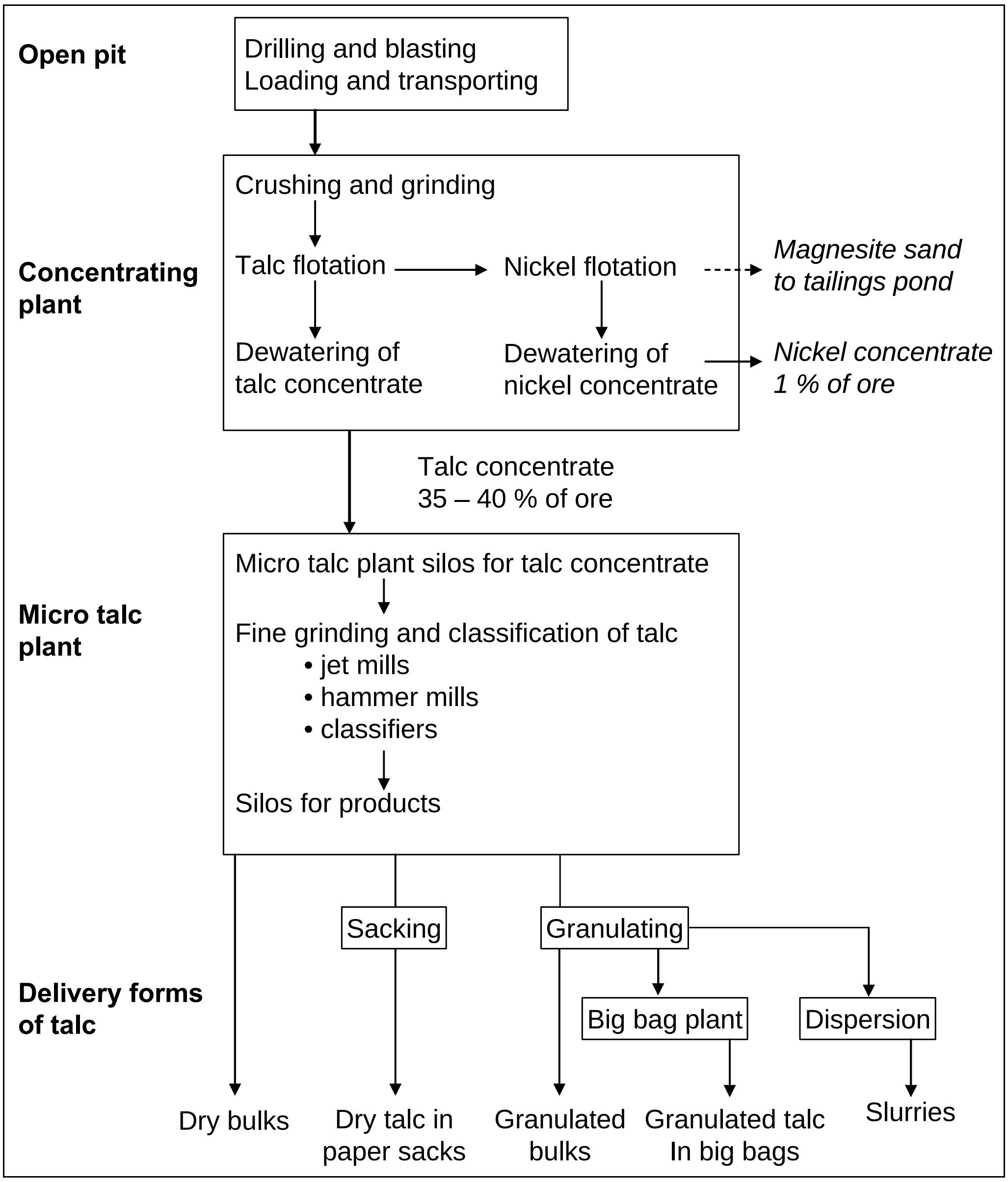


Figure 3.60: Talc process flow sheet using flotation

The process chemicals used in the flotation are Montanol, Na Xanthate and CMC.

3.2.9.3 Tailings management

Three tailings ponds are in use with a total current volume of about 10 Mm^3 and dam heights up to 17 m. Part of the tailings are discarded onto a heap (currently 1 Mm^3).

The heap is constructed as follows:
Tailings slurry is pumped into a pond with a decant tower in the centre. The tailings are distributed from the surrounding dams into the pond so that the tailings sand settles close to the dams and can be used as construction material to increase the height of the dam. Clear free water is discharged through the decant tower. By systematically changing the discharge points of the tailings slurry, the height of the whole area can be increased by 5 - 10 m. The outer slopes of the dams are covered with soil to prevent dusting and to promote vegetation. After dewatering the tailings the pond can be considered a heap.

The operational monitoring is done as follows:
Every day the tailings areas are visually checked and the necessary level monitoring is carried out and recorded. When necessary, monitoring is carried out (Ni and As analysis) of the tailings pond water, before draining it as waste water. During the snow melting season visual checks are made of the tailings areas and the dams on every shift. Annual monitoring of dams is carried out in summertime and all data are filed in dam safety manuals including dam condition, seepage water assessment, etc.

According to the Finnish dam safety regulations a dam safety manual is required for each tailings pond. An inspector from the competent authority visits the tailings area every five years and carries out a visual check of the dams and inspects the collected operational monitoring.
The dam safety manuals include the tailings area and dam maps, design values and stability calculations of tailings dams, classification criteria, inspection and monitoring documents, risk assessment of tailings areas, etc.

The water management of the three plants can be described as follows:

- Sotkamo plant: the process water needed for flotation comes from recycled water from the tailings ponds. Recycling percentage is close to 100 %. Additional water to the process water system comes from the adjacent open pit mine (nickel containing), fresh water system of the steam boiler and rainwaters collected on-site. This additional amount of water is drained from the tailings pond to the local lake

- Vuonos plant: the process water needed for flotation comes about 50 % from the recycling water in the tailings ponds. Additional water to the process water system comes from the local lake, adjacent old open pit mine (nickel containing), the fresh water system of the steam boiler and rainwater collected on-site. This additional amount of water is drained from the tailing pond to the local lake. Process water is used also in the production of some paper talc qualities

- Kaavi plant: the process water needed for flotation is comes to 100 % from the local lake. Additional water to the process water system comes from the fresh water system of the steam boiler and rainwaters collected on-site. No recycling of process water from tailings ponds is available. All process water is treated and drained from the tailings pond to the local lake. The waste water permit states that a recycling system has to be operational at latest by the end of 2003.

3.2.9.4 Waste-rock management

Trucks are used to haul and dump the waste-rock to the heaps which are designed with a safety factor of at least 1.3. The heaps are surveyed yearly by an external topographic contractor and (inspected) monthly by mine staff. Risk assessments are periodically done by the operator.

The heaps are permitted with a final rehabilitation project including water drainage and vegetal planting (trees and local grass).

3.2.10 Costs

In European feldspar operations the average cost for moving solid residues to a heap within a site amounts to EUR 0.80 and the average diesel fuel consumption of a truck is 28 l/hour.

For the flourspar/lead zinc operation the overall cost for tailings management in several ponds, 1300000 m^3 in total volume, is around EUR 210000/yr; this includes energy consumption and maintenance of the section.

For kaolin operations the average cost for moving tailings to a heap within a site amounts to EUR 1/tonne (if done internally) and EUR 2/tonne (if done by a contractor).

Approximate costs per m^3 of water are, in the dewatering system EUR 0.10/m^3 and, in the water cycle of the limestone plant at **Flandersbach,** another EUR 0.10/m^3.
[107, EuLA, 2002]

At the Finnish talc operation, the cost of trucking tailings is EUR 2 per tonne and km.

3.3 Potash

The applied techniques for potash are very much different than all other industrial minerals, hence a separate Section has been dedicated to discussing potash. Unless otherwise mentioned the information has been submitted by the potash subgroup [19, K+S, 2002]. This contribution describes potash sites in Germany, Spain and the UK.

3.3.1 Mineralogy and mining techniques

Potash deposits were formed by the evaporation of seawater. Their composition is often affected by secondary changes in the primary mineral deposits. More than 40 salt minerals are known, which contain some or all of the small number cations Na^+, K^+, Mg^{2+}, and Ca^{2+}, the anions Cl^- and SO_4^{2-}; and occasionally Fe^{2+} and Br^-, as well. The most common minerals are listed in Table 3.78.

Mineral name	Chemical composition
Anhydrite	$CaSO_4$
Carnallite	KCl x $MgCl_2$ x $6H_2O$
Gypsum	$CaSO_4$ x $2H_2O$
Halite	$NaCl$
Kainite	KCl x $MgSO_4$ x $11H_2O$
Kieserite	$MgSO_4$ x H_2O
Langbeinite	K_2SO_4 x $2MgSO_4$
Leonite	K_2SO_4 x $MgSO_4$ x $4H_2O$
Polyhalite	K_2SO_4 x $MgSO_4$ x $2CaSO_4$ x $2H_2O$
Sylvite	KCl

Table 3.78: Most common salt minerals in potash deposits

The most important salt minerals are halite, anhydride, sylvinite, carnallite, kieserite, polyhalite, langbeinite and kainite. Gypsum and/or anhydride occur at the edges of salt deposits and in the overlying strata.

Potash salt deposits always consist of a combination of several minerals (Table 3.79). The German term "Hartsalz" (hard salt) refers to the greater hardness of sulphate- and magnesium-containing potash minerals.

Marine salt minerals	Main compounds
Sylvinite	Sylvite, halite
Carnallitite	Carnallite, halite
Hard salt	Sylvite, halite, kieserite and/or anhydrite
Kainitite	Kainite, halite

Table 3.79: Marine salt minerals

In the following, to avoid confusion, the term sylvinite will be used for the mineral mixture of sylvite and halite, which usually occur together.

Salt deposits in Central Europe are the result of intensive evaporation of marine water more than 250 million years ago. Over millions of years, the original salt deposits were covered with other sediments, such as clay, limestone and anhydride. Tectonic influences left them as flat layers (sub-horizontal deposits) or deformed them into steeply dipping deposits (see figures below).

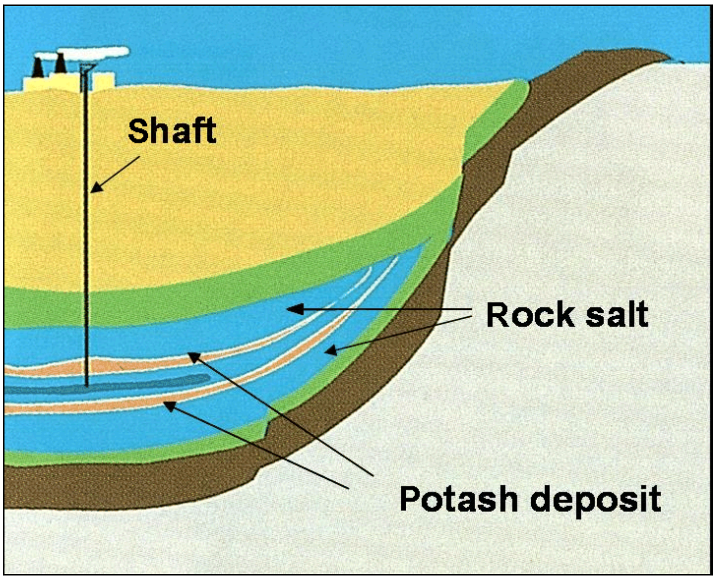


Figure 3.61: Sub-horizontal potash deposit

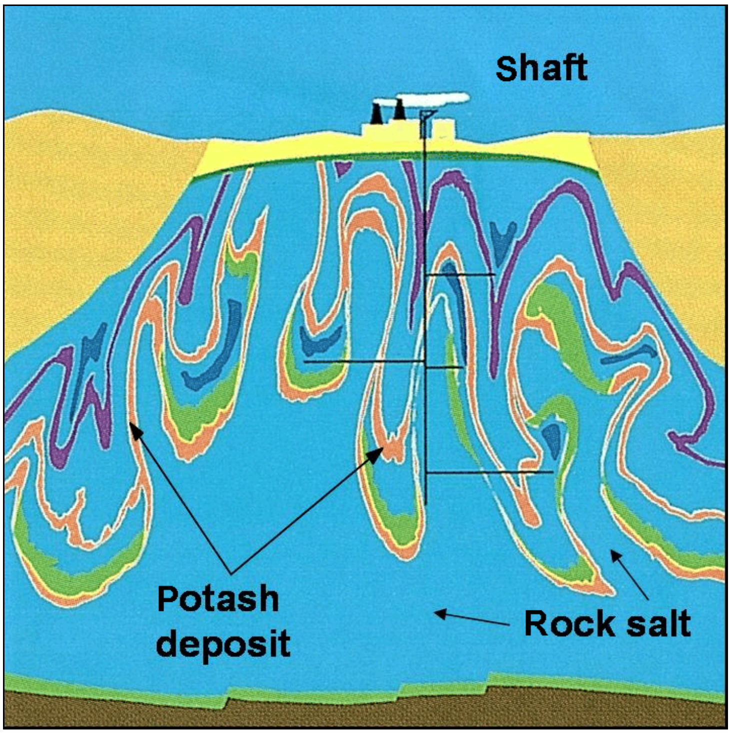


Figure 3.62: Steeply dipping potash deposit

Potash is usually extracted by room and pillar and sometimes longwall mining. Sometimes the ‘solution mining’ method is also applied. However, today solution mining is only of minor local importance in Europe. Open pit mining is not an option, due to the water solubility of potash.

Room and pillar mining
With this method the height of stopes is about two to three metres. Usually 25 - 60 % of the ore can be extracted from the mine. The pillars remain unmined. Two ways of applying this method are currently practised:

- drilling and blasting: Drilling machines are used to cut small diameter boreholes over a distance of 7 m to 30 m in the face, either horizontally (sub-horizontal/flat deposit) or vertically (steep deposit). The holes are filled with explosives (prills of ammonium nitrate with 3 % mineral oil) and the rock is blasted. The fractured salt is hauled by loaders to underground pre-crushing stations where it is crushed to a size which can be transported by conveyor-belts

- continuous mining: An excavation machine with a rotating head, the so-called 'continuous miner', is used to mine the ore in a size which can be transported directly by conveyor belts. The following surface operations are similar to the drill and blast mining method. Bolts are placed in the roof of the underground galleries for support and to protect the workers and the equipment.

At present, potash mining in Germany is carried out in depths between 400 and 1200 m. The ore is always transported in pre-crushed form by conveyor-belts to intermediate underground storage prior to hoisting with skips.

Longwall mining
This is the same method commonly used to mine coal deposits in Europe.

Sublevel stoping
In steeply dipping deposits in Northern Germany, sublevel stoping (also called 'funnel mining') is carried out. Entry drifts are driven one above the other at intervals of 15 - 20 m, and the remaining potash salt is mined by drilling vertical boreholes and then blasting. The blasted ore falls into the main level underneath. The mined-out room, 100 – 250 m in height, is usually backfilled with salt tailings.

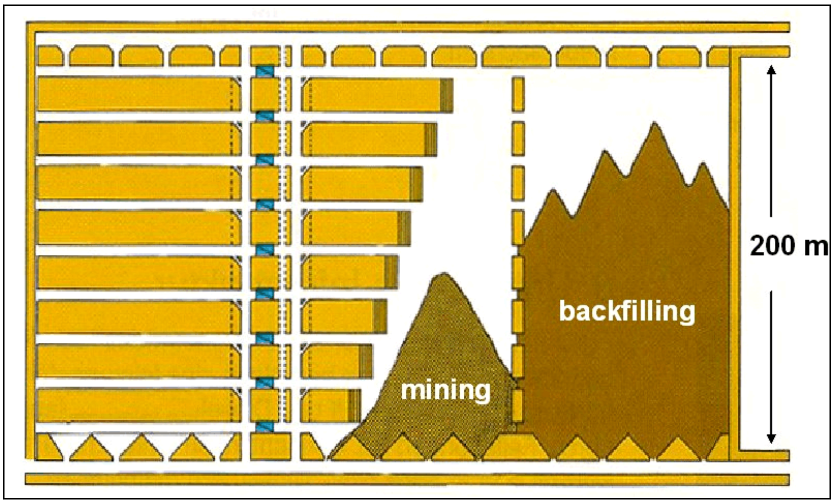


Figure 3.63: Sublevel stoping with backfill in steep potash deposits

Solution mining
KCl-unsaturated brine is injected in a borehole into the salt deposit to dissolve potassium chloride. The KCl-saturated brine is pumped back to the surface. The saturated solution crystallises and precipitates by evaporation of the brine in huge evaporator-vessels. A second separation process - e.g. flotation or re-crystallisation - follows to purify potassium chloride and sodium chloride as marketable products.

Exploited potash deposits in Europe
The exploited potash deposits in Europe were mainly formed in the Permian period, which took place in a vast evaporite basin, called the Central European Basin. This basin extends from North-East England to Central Poland and Lithuania, and from Central Germany to the northern part of the North Sea. The Alsacian and Spanish deposits were formed in the Tertiary period and are isolated basins.

France
The deposit in Alsace contains two sylvinite seams in a marl-rock salt series. The upper layer has a thickness of up to 2 m and contains 19 - 25 % K_2O; the lower, up to 5.5-m-thick layer, with 15 – 23 % K_2O, also contains 15 % insolubles (clay, anhydrite, and dolomite). Mining is carried out at comparatively high rock temperatures at a depth of 500 - 1000 m in flat or slightly inclined seams that have been disturbed by faults. The last producing mine was closed in 2003.

Germany
In the Werra and Fulda areas, the Hessen and Thuringia potash seams of the Werra series are mined (hard salt and carnallitite in level deposits at a depth of 400 - 1000 m with a thickness of 2 - 5 m, containing 9 – 12 % K_2O and 4 – 20 % $MgSO_4$:. The Stassfurt potash seam of the Stassfurt series was mined in the Harz-Unstrut-Saale area (hard salt and carnallitite at a depth of 500 - 1000 m and a thickness of 5 m, containing 20 % K_2O). The last potash mines, extracting hard salts of the Stassfurt series closed in 1991 for economic reasons. The potash seams Ronnenberg and Riedel of the Leine series are mined in the Hanover area in salt diapirs (sylvinite in inclined deposits at a depth of 350 - 1400 m with a thickness of 2 - 40 m, containing 12 – 30 % K_2O). Finally, potash is mined on the Massif of Calvörde near Zielitz (at a depth of 350 - 1200 m, Ronnenberg sylvinite inclined at <18 - 25°, thickness of up to 10 m, containing 14 – 20 % K_2O).

Spain
Deposits are located in two areas of the Ebro Basin. In Catalonia and Navarra, potash salts lie above the rock salt. These deposits are up to 15 m thick in Catalonia and up to 10 m in Navarra. Above this occurs an interbedded deposit of rock salt, carnallitite, marl, and anhydrite. Only the sylvinite seams A and B are mined. These are up to 4 m in total thickness at a depth of 1020 m, some deposits are level and some inclined. The crude salt contains 12.5 – 14 % K_2O.

United Kingdom
In Cleveland, a level deposit of sylvinite is extracted, which correlates with the German Riedel seam, both petrographically and stratigraphically (average thickness of 7 m, containing 25 % K_2O at depths of 800 - 1300 m).

3.3.2 Mineral processing

The processing of potash generally involves a series of steps including size reduction (crushing/grinding), separation (hot leaching-crystallisation, flotation, electrostatic separation) and de-brining. These steps are described below.

3.3.2.1 Comminution

The salt minerals in run-of-mine potash ore are intergrown to varying extents. Before the minerals can be separated and the useful components recovered, the raw salt must be sufficiently reduced in size to liberate the desired mineral from the gangue.

For the hot leaching process, a maximum grain size limit of 4 - 5 mm is adequate. For mechanical processing (e.g. flotation), the potash minerals must be ground to a degree of liberation >75 %. For sylvinite minerals and hard salts, this is achieved by grinding to a maximum size of 0.8 - 1.0 mm.

Various grain size fractions are produced in mills and different types of screens. In the first stage impact- or hammer mills generally produce particles of about 4 - 12 mm, depending on the raw material and the processing method used. The final fine grinding stage works with rod mills (when wet) or under dry conditions with roller mills or impact crushers (see figure below). The selection of the equipment used is based on minimising the generation of fines and ultrafines which have a negative influence on the subsequent separation, e.g. in flotation the reagent consumption increases significantly with the amount of fines due to the larger specific surface.

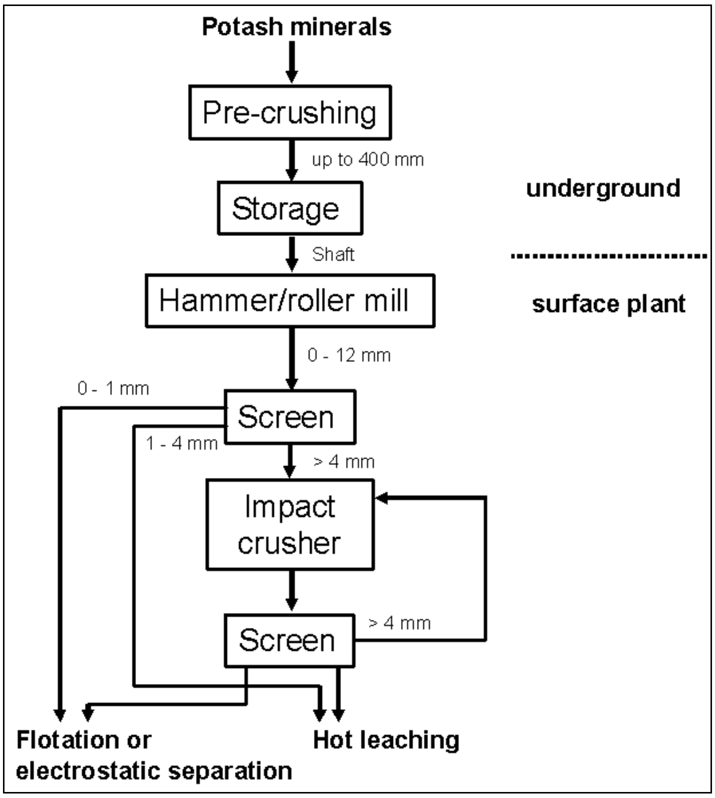


Figure 3.64: Dry grinding and screening (schematic) of potash ore
[19, K+S, 2002]

3.3.2.2 Separation

If potash is mined ‘mechanically’, i.e. not by solution mining, there are four methods which can be applied for separating the desired salts from the gangue:

1. hot leaching
2. flotation
3. electrostatic separation
4. heavy-medium separation.

For all wet processes (i.e. 1,2,4) de-brining is necessary.

The following sub-sections describe these process steps.

3.3.2.2.1 Hot leaching process

For the hot leaching process, two different processes are used, depending on the composition of the salt minerals. In the **sylvinite hot leaching process,** the other salts present besides KCl and NaCl play only a minor role in process solutions. The **hard salt leaching process** solutions contain appreciable amounts of $MgSO_4$ and $MgCl_2$. For carnallite-containing hard salts or unique carnallitite, preliminary carnallite decomposition must be carried out if the amount of carnallite present exceeds a critical level of about 20 - 30 %.

In both processes the potash minerals, ground to a fineness of <4 - 5 mm, are stirred in a continuous dissolver with leaching brine heated to just below its boiling point. The leaching brine (with a temperature of approx. 110 °C) is the preheated mother liquor from the crystallisation stage of a previous process cycle. The potassium chloride should be extracted from the minerals as completely as possible, and the resulting product solution should be nearly saturated. The tailings consist of two fractions of different particle size. The coarse fraction is removed from the dissolver and de-brined. The fine fraction (e.g. slime) is removed from the dissolver along with the crude solution. After separation in a clarifier, the fine fraction is filtered off.

The tailings are washed with water or plant brines low in potassium chloride to remove the adhering crude solution, which has a high potassium chloride content. The tailings are then discarded by stacking or backfilling in the mine. If kieserite needs to be separated, the tailings are transported to further mineral processing (e.g. flotation). The filtrate of tailings-dewatering is recycled to the re-circulating brine.

The hot, clarified, solution is cooled by evaporation in the vacuum station. Evaporated water must be replaced to avoid crystallisation of undesired sodium chloride. The desired potassium chloride crystals, formed by cooling the crude solution stage by stage (down to about 25 °C), are separated from the mother liquor and further processed. The mother liquor (saturated with KCl and NaCl at 25 °C) is heated and recycled to the dissolver as leaching brine. The layout of a leaching plant including crystallisation is shown in the figure below.

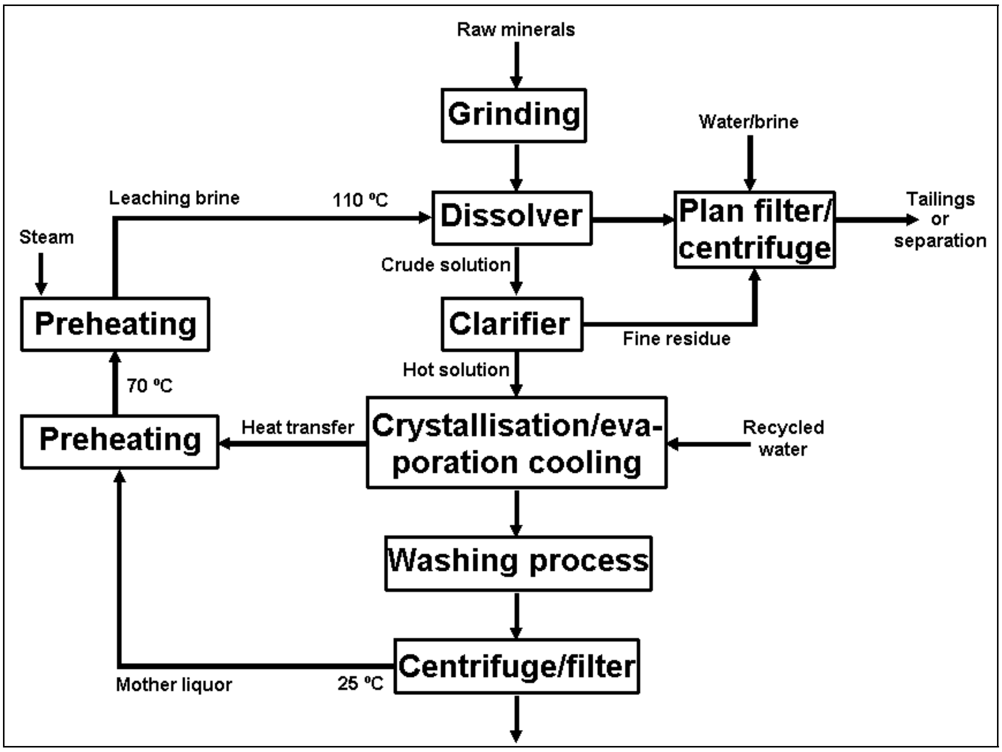


Figure 3.65: Flow diagram of the hot leaching-crystallisation process used for the production of KCl from potash minerals (schematic)

This simple process is used for the treatment of sylvinite minerals only. The mineral processing of hard salt minerals is more complicated. With higher magnesium salt contents, the temperature dependence of the solubility of NaCl becomes undesirable and the yield of potassium chloride decreases.

In many plants, especially in Canada, where flotation is the main production process, small hot leaching plants are also operated, in which the product "fines" (<0.2 mm) are re-crystallised, or potassium chloride is separated from flotation tailings or thickened clay slurries. These procedures lead to a considerable improvement in total yield and result in a very pure, completely water-soluble product. The hot leaching process is necessary to generate pure potassium chloride products for chemical or pharmaceutical uses.

3.3.2.2.2 Flotation

In the German potash industry, potash flotation as well as kieserite flotation is used. After grinding or previous separation-processes the fine size fraction (0 - 1 mm) is added to an aqueous, saturated potassium/kieserite and sodium chloride solution. As a frother pine oil is added. Rotating paddles scrape the potassium chloride or kieserite bearing froth from the surface of the mechanical cells for further treatment. The most satisfactory collecting agents are long chain alkylammoniumchlorides.

The following figure shows a schematic illustration of the mineral processing of the raw minerals or intermediates, carried out in rougher and cleaner flotation cells.

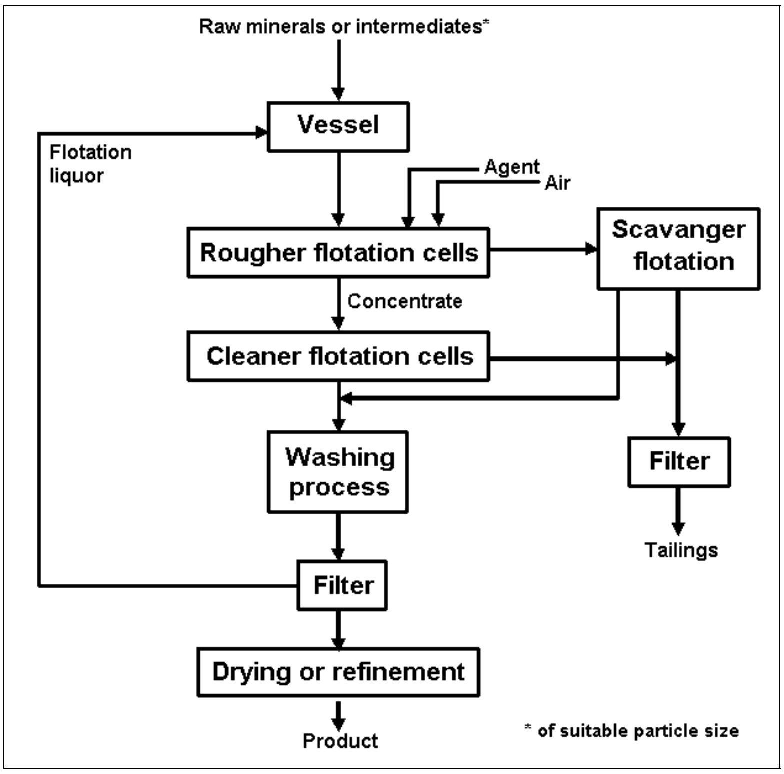


Figure 3.66: Flow diagram of a flotation plant (schematic)

3.3.2.2.3 Electrostatic separation

Crushed and ground raw salt is conditioned to achieve greater retention of the electrostatic charge by heating to less than 100 ºC. The crystals are coated with an organic agent such as a primary fatty acid, a derived salt, ester or amine. Depending on the aim of the separation 20 to 100 g of conditioning agents per tonne of raw salt are applied.
The ground mineral is electrostatically charged, under a specified relative humidity, by friction in a heated fluidised bed (see figure below). Separation of the halite minerals occurs when the charged crystals fall under gravity through an electric field of about 120000 volts in a free fall separator. The separation process is controlled by adjustable flaps, that are placed in the bottom of the separator (see Section 2.3.4). The middlings are reconditioned and recycled.

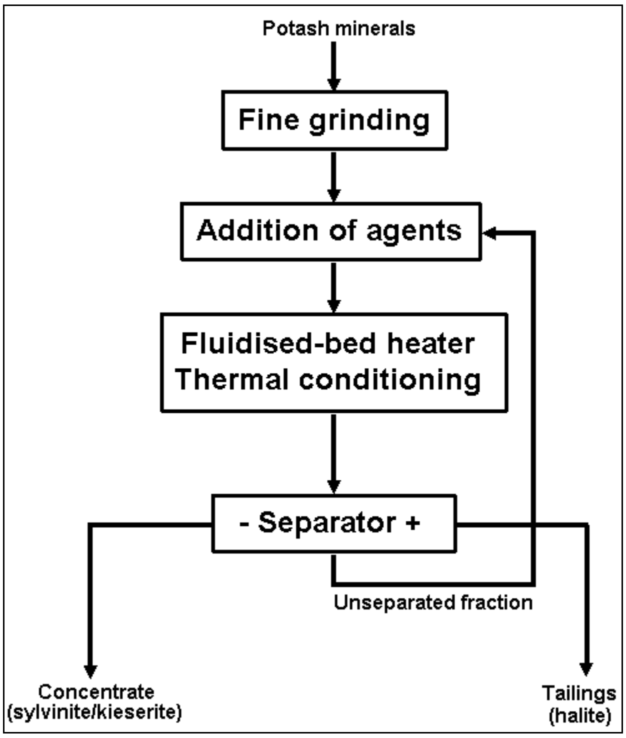


Figure 3.67: Flow diagram of an electrostatic separation process (schematic)

In most cases, a multi-stage separation or treatment is used. The solid tailings (sodium chloride/halite) are stacked directly on the tailings heap. Other options, such as firstly separating the sylvinite and carnallite from the kieserite, are also possible and are applied at other plants.

3.3.2.2.4 Heavy-media separation

Halite has a higher density than sylvinite (specific gravity 2.13 g/cm^3 versus 1.9 g/cm^3 for sylvinite). Commercial dens media operations use a very finely divided weighting agent, typically ferrosilicon or magnetite of a fine grade, which is slurried to create an artificial dens medium at the specific gravity required for separation. After separation, the magnetite or ferrosilicon is recovered by magnetic separation and re-circulated to the system.

A plant of this type operates in Canada. This process is also applied for the separation of langbeinite (specific gravity 2.83 g/cm^3) from sylvinite/halite a plants in New Mexico and the US. At present, this technique is not used in Europe.

3.3.2.3 De-brining

The products and tailings from all potach treatment processes, except for the dry electrostatic process, are obtained as suspensions/slurries with various solid contents and must be de-brined – after first being thickened in circular thickeners. The equipment used includes centrifuges, plan filters, drum filters and belt filters, especially for de-brining fine tailings (moisture content of about 9 - 14 %) and, when it is necessary, to wash the filter cake. The choice of equipment is determined mainly by the particle size of the material to be treated and the content of other minerals such clay.
For coarse products and tailings, vibrating screens and screw screen centrifuges are commonly used.

3.3.3 Tailings management

The mineral processing of potash minerals leads to over 78 % solid or liquid tailings (see figure below).

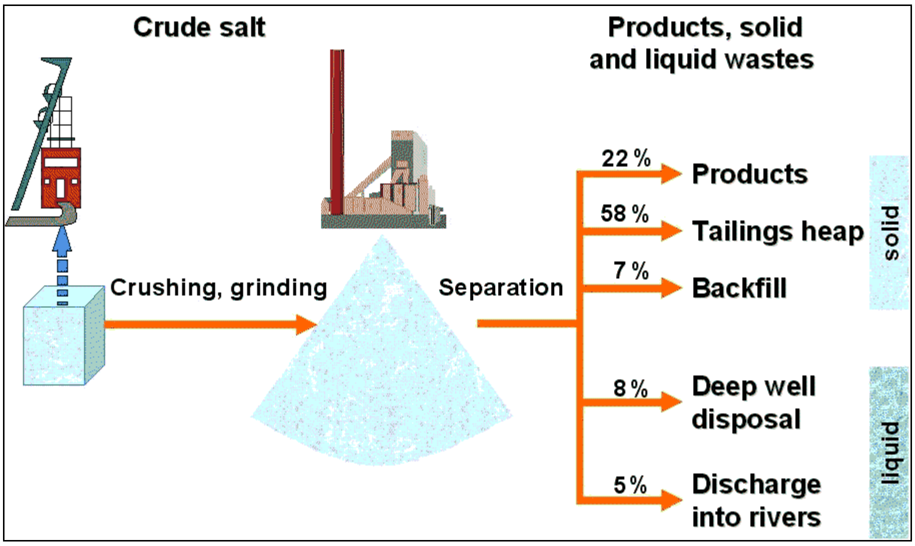


Figure 3.68: Distribution of products, solid and liquid tailings after mineral processing

Six methods for managing process water and/or tailings are applied:

- discarding solid tailings onto heaps
- backfilling solid tailings underground into mined out stopes
- discarding slurried tailings on tailings piles (only carried out in Canadian/US-Potash Mines)
- applying marine tailings management of solid and liquid tailings
- pumping liquid tailings into the ground (deep well tailings management)
- discharging liquid tailings into rivers.

3.3.3.1 Characteristics of tailings

Solid potash tailings consist of sodium chloride with a few per cent of other salts and insoluble materials such as clay and anhydrite (see figure 'sylvinite tailings'). Hard salt tailings additionally contain about 5 % kieserite (see figure 'hard salt tailings').

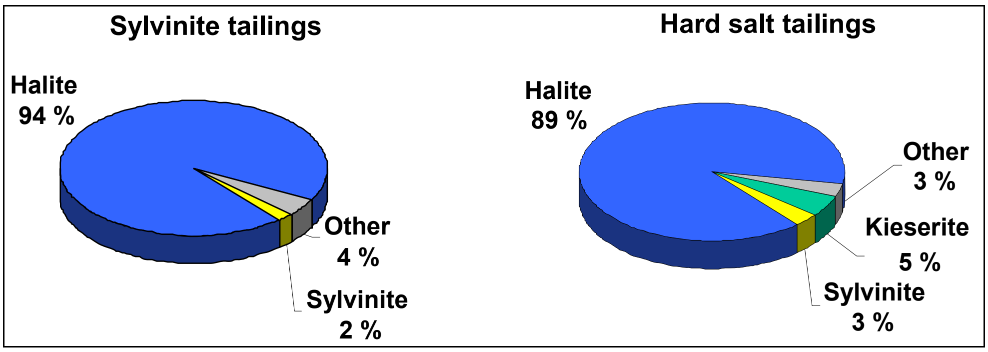


Figure 3.69: Mineral composition of sylvinite and hard salt tailings

The stacked tailings harden immediately, and the density of the tailings increases to nearly the same density as underground due to compaction. This has been shown by measurements from borehole-samples of tailings heaps. Heaps are stacked with an angle of repose of about 37 °

(natural soil angle: 25°). Therefore, no problems with the slope stability of the heap occur, if the underlying ground is stable. There is a wide experience in stacking potash tailings. The first heaps going up to 200 m in height were started about 30 years ago. Smaller heaps with tailings from potash mining exist from the beginning of potash-mining in about 1890.

Precipitation dissolves the tailings heaps slowly and over a long period of time. As a result of compaction and hardening, the interior of potash tailings heaps is impermeable to water. Water and generated brines flow down in an outer sphere around the inner impermeable core. To protect soil and groundwater, the outer seam of heaps outside the impermeable core zone is carefully sealed and the brines are collected in sealed ditches around the heap. The slope of the heap consists of hardened rock salt without any erosion after compaction and re-crystallisation.

The dissolved NaCl needs careful management to reduce its impact on the local environment. However, the tailings usually contain insignificant amounts of heavy metals and other trace elements or substances.

Liquid potash tailings are essentially the same material as in sylvinite tailings (90 % NaCl) but which have been dissolved in fresh- or seawater for transport to a suitable receptor. For discharges into surface waters or through long pipelines (i.e. as in Spain), the suspended solids content is usually very low.

3.3.3.2 Applied management methods

The amount of tailings generated by a potash mine depends primarily on the potash seam configuration, rock stability and mineral composition. These are all natural conditions that vary between mines and deposit and sometimes even within a deposit. As a result, there is no standard model of mines in terms of processing and generation of products and tailings. Each mine has its own specific conditions affecting solid or liquid tailings generation and management. Also, these specific conditions can change over the lifetime of a mine. However, economic reasons mean that operators will seek to minimise the amount of gangue materials mined and processed.

For **solid tailings**, the management of the tailings on heaps and by backfilling underground are applied. The tailings from the hot leaching and flotation process with sodium chloride as the main compound are dewatered by centrifuges, filters and then transported by conveyor-belts to the tailings heap. In addition, in Germany, the dry electrostatic separation process allows dry management of tailings on tailings heaps.

For **liquid tailings**, management of the tailings involves deep well discharge (under specified geological conditions) and/or discharging into surface waters. Under special geographical conditions, marine discharge of solid and liquid tailings is applied.

3.3.3.2.1 Tailings heaps

About 21 million tonnes of potash tailings are stacked by the German potash industry every year. Large tailings heaps are built with quantities of 25 to 130 million tonnes, altitudes of 90 to 240 m with a footprint of 47 to 110 ha.

The largest tailings heaps, their location, altitude, size, the quantity of tailings and the main compounds are shown in Table 3.80.

Plant/facilities	Location	Altitude (m)	Size (ha)	Quantity (million tonnes)	Main compound	Remarks
Hattorf	Werra-area	160	47	59	Halite	
Wintershall	Werra-area	240	55	99	Halite	
Unterbreizbach	Werra-area	42	4.6	<1	Kieserite	Currently reprocessed
Neuhof-Ellers	Fulda- area	180	70	80	Halite	
Sigmundshall	Hannover area	150	26	25	Halite	
Zielitz	Zielitz Zielitz	50 90	53 110	 130	Halite Halite	

Table 3.80: Tailings heaps of the German potash mines

The following figure shows a typical salt tailings heap in Germany.

Figure 3.70: Aerial view of a salt tailings heap

Environmental impact studies including baseline studies are a necessary part of the design of these heaps. They include research into different site aspects, such as:

- stability of the heap
- stability of the supporting strata
- water protection (ground- and surface water, water quality and supply)
- dust emissions
- technical operations
- wildlife habitat
- rehabilitation and after-care
- control and monitoring systems.

It is necessary to ensure the **stability of the heap** to avoid possible movements of parts of the heap. The rock salt hardens rapidly, due to the sufficiently low moisture content of the stacked material. Therefore no significant erosion occurs and additional support around the heap is not necessary. In essence, the stability of the tailings heap is ensured by the application of fundamental civil engineering rules.

The **stability of the supporting strata** is controlled regularly by seismic monitoring (see monitoring and control systems, below), which search for and determine seismic, seismic-acoustic and geo-mechanic facts. Survey of pillars and the determination of the mineral compounds are used to calculate and observe the stability of the mined out rooms.

To ensure **water protection** the following items are taken into consideration:

- water balance (groundwater and surface water)
- detected aquifer strata
- watersheds
- water impermeability of supporting strata
- possibility of process water re-use
- water supply and distribution management
- quantity and management of accumulated drainage water
- salt quantities to be managed
- land requirements for stacking.

The interior of potash tailings heaps is impermeable to water. Water and generated saline solutions only flow down in an outer sphere around the inner impermeable (see). The toe of the heaps outside the impermeable core zone is carefully sealed and the solutions are collected.

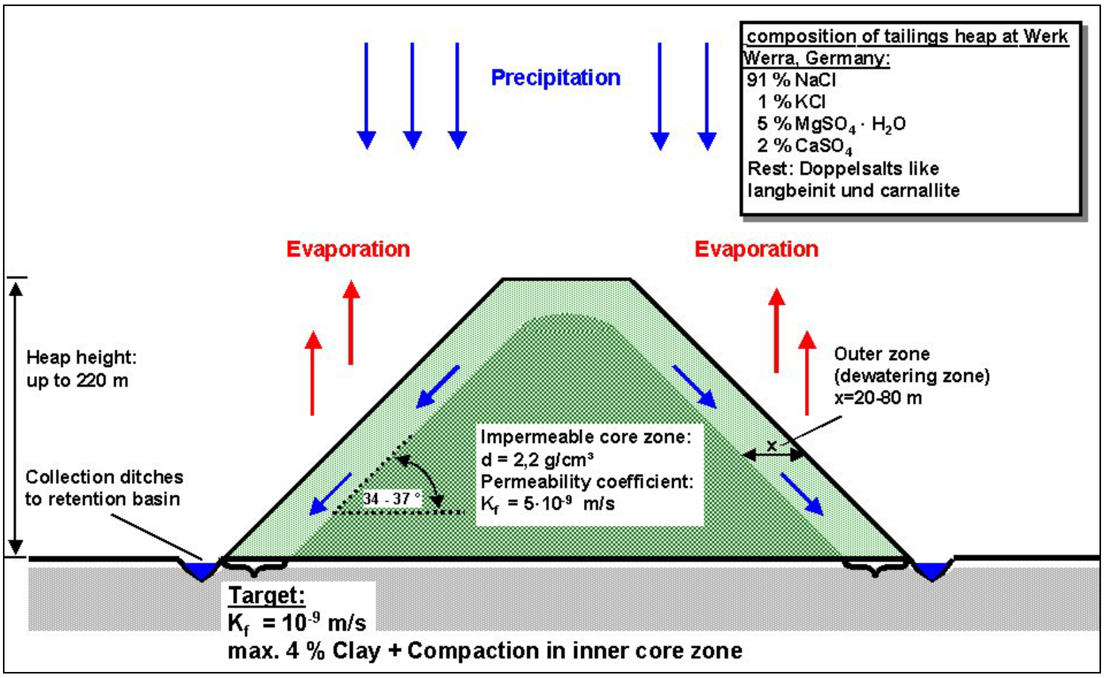


Figure 3.71: Schematic drawing of a tailings heap in German potash mining

After collecting the brine in the retention basin for intermediate storage and depending on the received water quality, the liquid is pumped to the river or into the ground (deep well discharge). In some cases the collected brines are re-used for processing (e.g. granulation, recycled processing brine). In general, only small amounts of collected brine are re-used.

Since the water flow from precipitation runs down the heap underneath the surface (see blue arrows in figure above) erosion at the surface does not occur. If possible, saline drainage from the heaps is kept separately from surface run-off. This is one way to minimise salt water contamination of soil and groundwater.

Another objective is to reduce land use by stacking the tailings to a maximum of height. In this operational technique (see below), the design used (conical/longitudinal heap) and the natural angle of repose are critical to obtain this.

The commonly applied technique uses conveyor-belts, continuously stacking the tailings on a heap, which is located near the processing plant. After the addition of a small amount of processing brine to the dry tailings from electrostatic separation the moisture of the stacked combined tailings results to the aimed 5 - 6 %. The stacked salt hardens immediately because of compaction and re-crystallisation.

The **technical operations** for stacking have been applied and optimised over more than 30 years.

The salt tailings are stacked using conveyor-belts and spreader systems, this allows steeper, higher stacking than wet stacking. Up to 1200 tonnes per hour solid tailings are stacked on one heap. These enormous amounts of material are piled near the processing plant, to minimise material transport over long distances or through communities.

The distribution of tailings on the heap is performed by combination of several conveyor-belts. Depending on the type of construction chosen, the discharging belt can be slewed, adjusted in height and, if necessary, be telescoped. A low discharge height is preferred. A last short underlying conveyor-belt, arranged below the main conveyor-belt is reversible (see figure below), which is particularly effective in avoiding dust in windy conditions. Dust control is not an issue with tailings from the wet separation processes as the residual moisture content of 5 - 10 % is sufficient to eliminate dust problems and to cause rapid consolidation with the tailings heap.

Figure 3.72: Photo of a conveyor belt with an underlying reverse belt

Processing, and therefore also tailings discharge, is carried out continuously day and night. The employees usually work in rotating shifts. Continuous working systems create less dust and noise and material transport over long distances is not required.

Possible effects on **wildlife**, both at present and for future, developments need to be examined, carefully considered and, as far as possible avoided during the operation.

The **controlling and monitoring regime** examines seismic events or subsidence of the surface as a result from mining activities. The stability of the supporting strata and underground mined rooms can be measured by seismic monitoring.

At the surface, different controlling and monitoring systems are applied e.g. for groundwater protection, determination and control discharging brine to the river and the mineral processing process, dust emissions, energy consumption, water supply etc.

Several locations have slope inclinometers which are used to study the deformation and stability of the tailings heap. Slope stability needs less monitoring on tailings heaps that are confined by natural topography.

3.3.3.2.2 Tailings piles

Commonly the tailings in **Canadian/US** plants are pumped as a slurry with 20 – 35 % solids to the top of the tailings piles in the tailings management area. The slurry flows down the gentle back slope of the pile with the slimes settling out at the toe. Low containment dykes are built to confine the discharge of brine to the surrounding area. At present, the tailings piles are generally about 50 m in height. Due to this low height compared to tailings heaps large areas are occupied by this tailings management method.

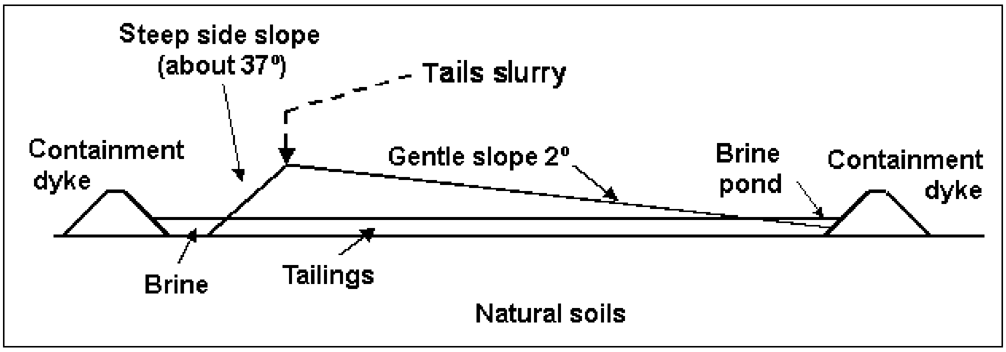


Figure 3.73: Typical cross-section of Canadian tailings piles (schematic)

3.3.3.2.3 Backfill

The second method of tailings management for solid tailings is **underground**. This method is applied in steeply dipping deposits in Northern Germany and in the potash mines of New Brunswick in Canada. Since the bulk density of the tailings is much lower than that of the original potash ore, only a part of the tailings can be accommodated by the space left after extraction of the crude salt.

In most potash plants, where the mineral is mined from flat deposits, backfill is not carried out for economic reasons.

A similar method, although less important for active European mines, is backfilling of tailings as a slurry. The tailings slurry is returned underground to fill up the potash cut-and-fill stopes, which are shaped as ‘domes’. However, the applicability of this option, amongst other reasons, depends on the existence of suitable geological formations (i.e. local steeply dipping deposit).

At one plant, Unterbreizbach in the Werra-region, brine is added to the tailings and the resulting slurry is pumped for backfill.

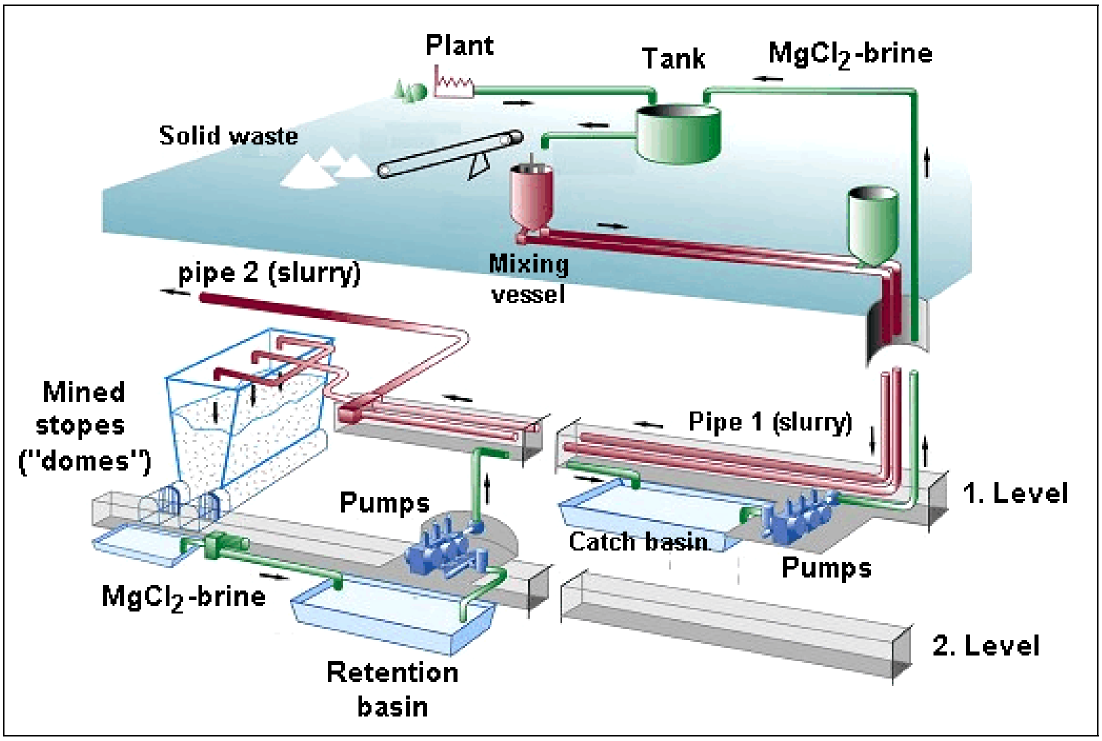


Figure 3.74: Backfill system of solid tailings (sodium chloride) at the plant Unterbreizbach, Germany

The Unterbreizbach plant differs from the other potash plants with flat deposits in various aspects:

Geology:

- the exploited seam Thuringia contains a very thick layer of carnallite above the hard salt seam. When the carnallite is mined, a series of empty "domes" are left.

Mineral processing:
- a combination of thermal dissolution process and the flotation of kieserite is used.

Tailings management:

- salt tailings (solid sodium chloride) from the flotation of kieserite are slurried with MgCl2-brine (salt-saturated) from the thermal dissolution process and pumped underground for backfill. The efficiency of the backfill system could be increased with a second pipe. The brine is recovered underground and pumped back to the surface for re-use.

In the UK, backfilling a proportion of the insoluble tailings as a slurry is being investigated. In this instance suitable geological conditions and appropriately configured mine workings dictate the volume available for placement. Similar trials in Spain failed because of the poor geological conditions.

3.3.3.2.4 Surface water discharge

At the operations in Germany and Catalonia brine from production, sometimes mixed with small amounts of salt water from the tailings heap, is collected in lined retention ponds from where the brine is discharged into surface water (e.g. river). The following figure shows one of these basins.

Figure 3.75: Water retention basin of German potash mine

In Germany the surface water discharge is combined with deep well discharge (see following section).

3.3.3.2.5 Deep well discharge

Pumping salt solutions back into the ground is possible if certain geological requirements are met. The geological formation required for this purpose must possess sufficient porosity and permeability, and must have no contact with formations that can be used for water supply.

In the German potash industry a combination of river and deep well discharge are used. As much water as possible is discarded into the river system. This is determined by the set threshold for chloride in the river taking into account the total discharge of all potash mines (see figure below). All excess water is pumped into the deep wells.

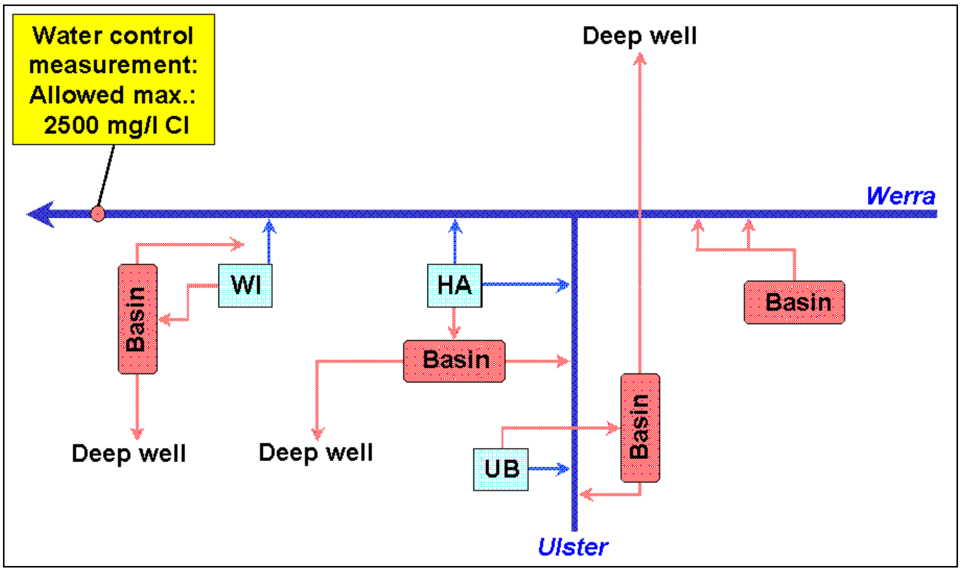


Figure 3.76: Management of three potash mines (WI, HA, UB) in the Werra area, Germany

3.3.3.2.6 Marine tailings management

At the Cleveland Potash operation, the ore is crushed and separated into the potash and tailings fractions. The tailings consist primarily of sodium chloride with small quantities of calcium sulphate and clay. These naturally occurring components are mixed with seawater and discharged into the North Sea through a long outfall pipeline.

Discharges into the North Sea are controlled by the OSPAR Commission (OSPARCOM, http://www.ospar.org/eng/html/welcome.html) and in this case, permitted by the UK regulatory body. Meaning that guidance concerning discharges into the North Sea developed by OSPARCOM was adopted by the UK government, which used the information for the permitting and monitoring requirements. Extensive baseline studies of the receiving body were conducted including bathymetry, benthic flora and fauna, water quality and the state of the important local fisheries. Continual monitoring of the quantity and quality of the discharge ensure that all parameters remain below consented values. Trace element analysis of the ores, products and effluents solids allow mass balances to provide checks on the flow and other monitoring data.

Continuing annual surveys of all the parameters are conducted by external experts to ensure that the effects of the discharge are determined and kept to a minimum. Audit samples are taken by the regulatory body for independent confirmation of the company results. Annual stakeholder meetings ensure that the results of the monitoring are communicated to all interested bodies and that they have the opportunity to influence the direction and content of future monitoring programmes.

3.3.3.3 Safety of the TMF and accident prevention

In the design of the TMFs, the following factors are considered:

- examination of ground stability
- examination of heap stability
- reduction of permeability of supporting strata if the average permeability coefficient exceeds e.g. 1 x 10 – 9 m/s, but site-specific and depending on the findings of the environmental impact assessment
- avoidance of artificial sealing layers with low shearing strength (has a negative effect on heap stability)
- application of moist tailings but with a moisture content below about 10 %.

Inspections of tailings heaps are routinely carried out by the operator. These include yearly surveillance of the heaps and observation of ditches and basins.

3.3.3.4 Closure and after-care

For **rehabilitation and after-care**, the description of the current state and future development of the facility including the tailings management area, and the closure plans of the mining operation are compiled in the form of a detailed plan.

After permitting of the monitoring and surveillance plan for closure, the operation facilities from the plant must be removed. However, the tailings heaps remain unchanged for a long period of time. A fund to cover future maintenance cost is financed from operational costs before closure.

3.3.4 Waste-rock management

Since potash mining is only carried out underground, the amounts of waste-rock arising are relatively small. The waste-rock remains underground in mined out areas of the mine. Usually this underground movement of waste-rock is referred to as ‘stowing’ or ‘backfilling’.

3.3.5 Current emission and consumption levels

The quantities of emissions and effluents vary from mine to mine. They are also in some respect a function of natural conditions - the components of the exploited deposit and the mined minerals. Site-specific contributions - the form of mineralisation, the grade and liberation of the material, the mixture of mineral constituents in the mined deposit - are always unique. Depending on the mined ore and the desired products a process is chosen with solid and liquid tailings in varying proportions. Emissions and effluents are also a function of management and processing method.

3.3.5.1 Management of water and reagents

In general, it is possible to dissolve all solid tailings and discharge the resulting solution including insolubles into natural water systems (e.g. marine tailings management in UK).

Tailings heaps generate saline solutions when atmospheric precipitation dissolves the salt. This run-off water is collected in sealed ditches around the tailings heap and pumped into sealed retention basins. From these retention basins the saline water is discharged into natural flowing waters (e.g. rivers) or pumped into the ground (deep well tailings management).

The sealings of ditches and retention basins are inspected to avoid soil and groundwater salinisation. Furthermore, the water of groundwater wells in the surrounding of a tailings heap is periodically analysed to verify its quality.

No addition of water is applied for backfilling. For the backfill of slurries at the Unterbreizbach plant, processing brine is combined with solid tailings. The brine is used as a transportation medium only and is recycled. Processing brine is re-used for different applications in mineral processing to minimise the consumption of water.

In solid tailings no significant amounts of reagents are detectable. The only reagents used result from the electrostatic separation or the flotation process. These processing methods work with a low content of organic compounds (salicylic acid, fatty amines).

The main components of the liquid brine are inorganic salts, while the presence of organics (TOC) and heavy metals is negligible. This is a consequence of the deposit-formation by the evaporation of seawater about 250 million years ago.

3.3.5.2 Emissions to water

No noticeable amounts of trace elements, heavy metals or organic substances can be detected in the surface run-off from the heaps. The main components of surface run-off are salts such as sodium, magnesium, potassium and calcium chlorides and sulphates. The volume of surface run-off from the heap depends on land consumption, precipitation (per year) and the components of the salt tailings.

If the mineral kieserite ($MgSO_4 \cdot H_2O$) is one component of the mined salt, some kieserite will be in the tailings, too. Upon contact with rainwater, kieserite is hydrated and thus binds some of the rainfall. In consequence, the water binding capacity of a tailings heap from potash mining is strongly dependent on the specific minerals content.

A second important factor influencing the amount of surface run-off is the evaporation of water, which depends on several factors such as temperature, humidity, wind speed, colour of the tailings, sunshine intensity, etc.

3.4 Coal

In this section contributions about practices in Spain, the Ruhr, Saar and Ibbenbüren areas in Germany and the Ostrava and Karviná areas in the Czech Republic are included. Furthermore, comments from the UK have been added.

3.4.1 Mineralogy and mining techniques

All of Germany's hard coal resources are carboniferous in age. While the Saar and Ibbenbüren Basins represent remnants of larger coalfields, the Ruhr contains massive resources that dip towards the North Sea. Current working areas are located in depths ranging between 900 and 1500 m. Conditions in the Saar Basin are more complex than in the Ruhr.

The high-quality coke, gas and steam coals typically contain 6 – 9 % ash, and less than 1 % sulphur, although some seams require extensive washing before sale. The Niederberg mine and the Ibbenbüren deposit contain anthracite, which is coal with a fixed-carbon content between 92 % and 98 % (on a dry, mineral-matter-free basis).

Longwall faces of up to 400 m are now in service. The seams worked range in thickness from 1.0 - 4.0 m, with ploughs being used in the thinner seams and shearers in thicker applications.

Hard coal in the Czech Republic mainly occurs in the Upper Silesian Basin. The major fault, called the Orlova fault, divides the Czech part of the Upper Silesian Basin into the western section (the Ostrava part), which is older and of paralic character of sediments and coal seams, and the eastern section (the Karviná part), which exhibits a limnic character of the sediments as well as the coal. The western part consists of several thin coal seams of high grade coking coal, whereas the eastern part is characterised by abundant thick seams containing mixed coking coal and highly volatile steam coal. Some of the characteristics of hard coal include a carbon content of more than 73.4 %, less than 50 % volatile matter, and a dry (ash free) calorific value that exceeds 24 MJ/kg.

Mining in the Ostrava part of the basin has reached depths of about 1000 m, which together with complex and unfavourable mining and geological conditions makes economic mining extremely difficult. Consequently, the Ostrava mines have been gradually abandoned. The majority of mines in the eastern part have sufficient reserves which can be extracted at much lower costs. However, this coal is of low grade, as far as coking properties are concerned.

Relatively large reserves of coal were verified south of the original Upper Silesian basin, particularly near Frenštát pod Radhoštěm, where carboniferous sediments are buried under Miocene sediments and the Beskydy napes. Here, the coal can be extracted from depths of 800 to 1300 m under difficult geological and mining conditions. As the deposit is situated on the border of a protected landscape area, conflicts of interests may arise with Beskydy protection.
[83, Kribek, 2002]

Most operations in Europe are based around longwall mining, using both shearers and ploughs for production. Most mines operate in several seams, with each unit operating several faces. In Germany, an increasing number of longwalls are controlled remotely from the surface, high

levels of automation allowing saleable outputs of up to 20000 t/d per longwall [79, DSK, 2002], [83, Kribek, 2002].
In the UK (about 15 million tonnes/year) and Spain coal is also mined in open pit mines [84, IGME, 2002]

3.4.2 Mineral processing

In general, after the extraction step, particle size ranges from pieces of more than one metre in diameter to ultrafine grains (<5 µm). In the three German coalfields of Ruhr, Saar and Ibbenbüren a wide range of coal qualities are mined, from anthracite at the Ibbenbüren colliery with 6 % volatile matters (VM) up to the high volatile bituminous coals of the Ensdorf underground mine with more than 36 % VM. In 2000, 12 coal processing plants with feed rates between 950 and 1700 t/h were in operation in these coalfields. [79, DSK, 2002].

In most cases the coarse (>10 mm) and fine fraction (0.5 – 10 mm), are separated in jigs. The finest fraction <0.5 mm is separated by flotation. In some cases, the fraction >10/30 mm is separated from heavier gangue by dense media separation.

A typical flow sheet can be seen in the following figure:

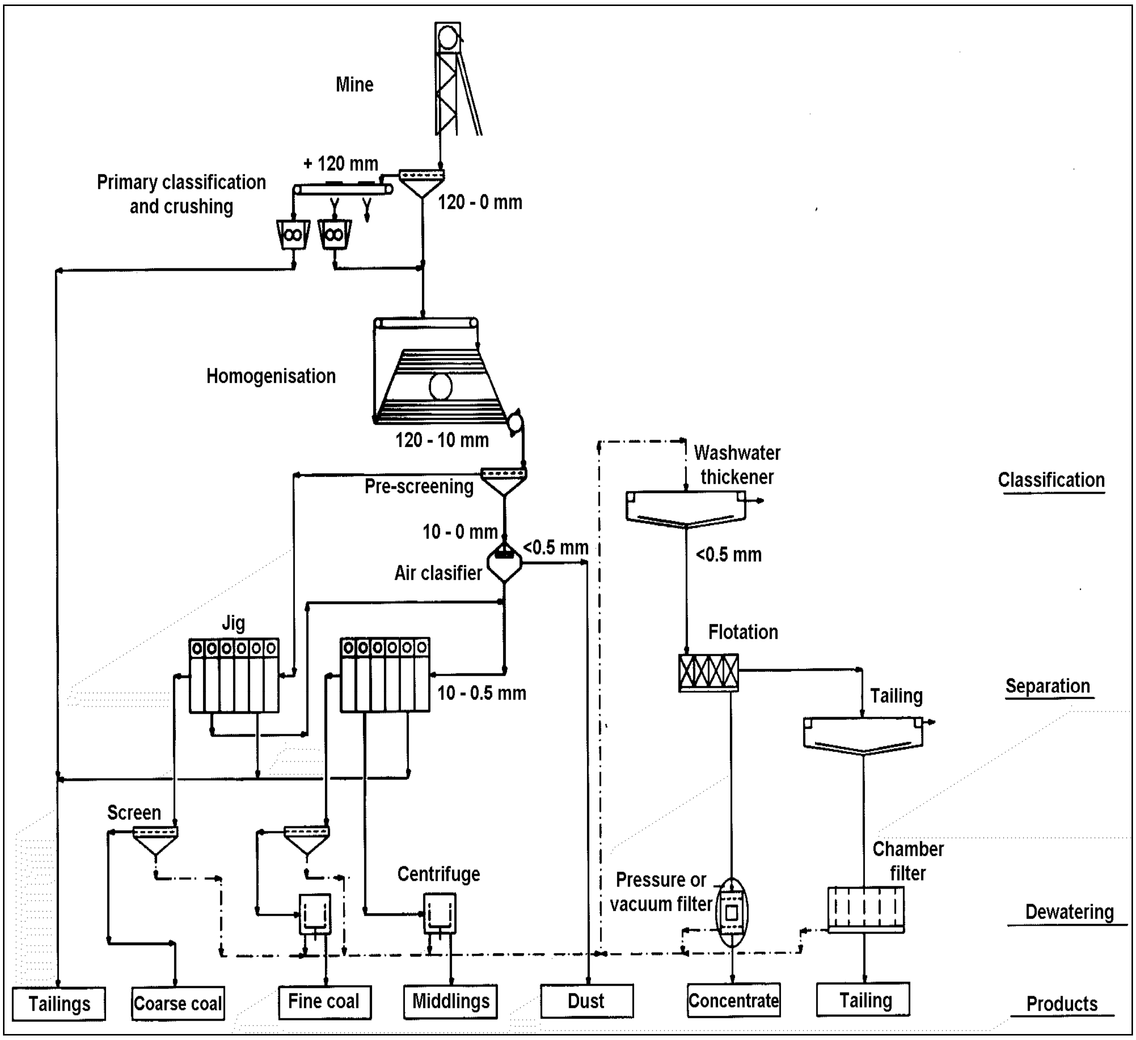


Figure 3.77: Standard flow sheet for coal mineral processing
[79, DSK, 2002]

There is also one site that uses hydrocyclones instead of flotation of the fines [83, Kribek, 2002].

3.4.3 Tailings management

3.4.3.1 Characteristics of tailings

Typically tailings from the Ruhr, Saar and Ibbenbüren areas in Germany consist of 55 - 60 % clay shale, 30 - 40 % sandy clay shale and 5 - 15 % sandstone (Prosper-Haniel mine) [79, DSK, 2002]. Hard coal deposits can be influenced by maritime 'footprints', when formed as paralic basins, i.e. in a marginal marine environment. Freshwater coal basins formed in a river delta, so-called limnic basins, lack such evidence. Amongst environmentally relevant substances embedded in intermediary layers, chloride and pyrite are the most important ones. Precipitation coming into contact with tailings material takes up the salt and is acidified through sulphuric oxidation. As a consequence, the pH-value of a leachate or surface water so influenced decreases (ARD, see Section 2.7).

The fine flotation tailings from the Ruhr, Saar and Ibbenbüren coal mines <0.5 mm with a >77 % solids and a homogeneous mineralogical composition were tested in detail. In physical and chemical tests with long-term considerations including environmental impact assessment, it has been proven that flotation tailings can be used for the construction of surface liners even achieving the stringent requirements of the German Technical Standard for the construction of liners for landfills [80, DSK, 2002]. In laboratory tests pure flotation slurries from hard coal processing can reach k–coefficients of around 5 x 10^{-9} m/s. In-situ tests resulted in k_{f-}oefficients of ~ 2 x 10^{-7} m/s. These k-coefficients do not reach values required by TASi/LAGA standards for mineral liners (k_f = 5 x 10^{-10} m/s) and surface seals for landfill category I (k = 5 x 10^{-9} m/s). [79, DSK, 2002].

In the Ostrava and Karviná areas the coarse tailings are handled on heaps and the fines from flotation are sent to basins or ponds. In one case, a level of radioactivity of 75.5 ± 6.9 Bq/kg was measured in the tailings [83, Kribek, 2002].

Two other important aspects that need to be considered in the management of coal tailings are:

1. coal tailings can have increased contents of naturally occuring radiactive materials (NORM) due to their genuine strata
2. and may cause similar ARD problems as sulphide containing metal ores, because of the pyrite content of the coal.

3.4.3.2 Applied management methods

In the Ruhr, Saar and Ibbenbüren areas, a total of 23 tailings heaps and 7 tailings ponds are currently in operation [79, DSK, 2002]. Considerable amounts of tailings from coal mining have to be handled (about 33 million tonnes in the Ruhr, Saar and Ibbenbüren areas in 2000), since they can amount up to around 50 % of raw production. Principally, three management options are available:

- internal application, i.e. for underground backfill or construction projects linked to mining operations (e.g. compensation measures for mining-induced ground subsidence such as heightening of bridges or embankments)
- external application, i.e. commercial products, such as bulk mass material or base material in construction sector and civil engineering
- management on dumps and in ponds.

As a rough guide, around one quarter of all rock and tailings in the Ruhr, Saar and Ibbenbüren area is sold for internal and external purposes (see Section 4.5.3), whilst the remainder is managed on dumps (or heaps) and in ponds (see figure below).

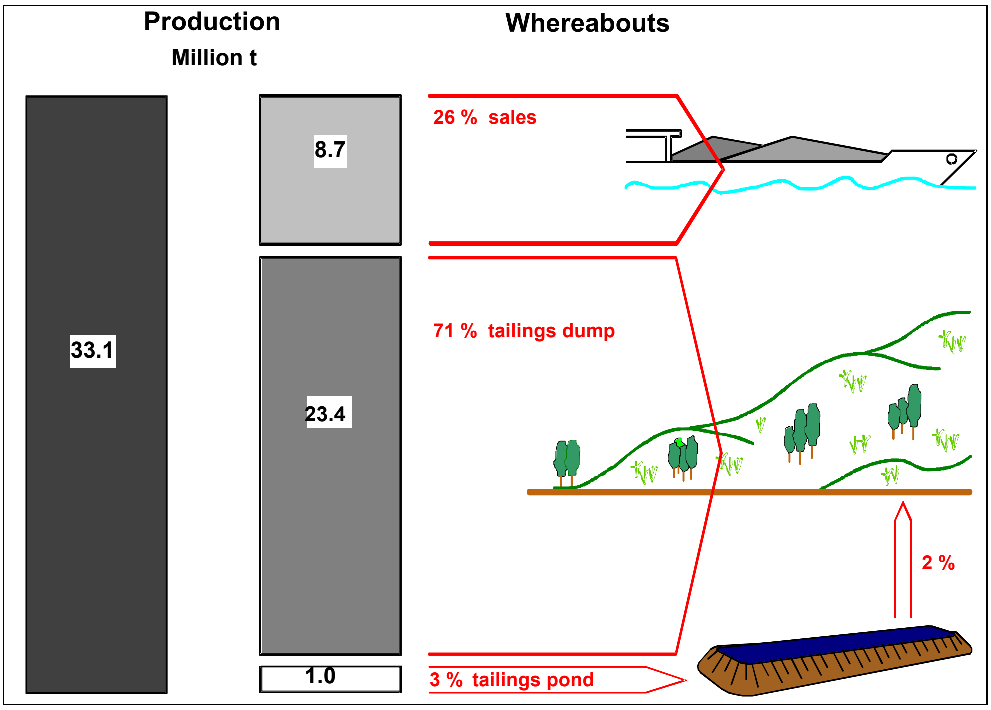


Figure 3.78: Tailings production and applied management methods in the Ruhr, Saar and Ibbenbüren areas in year 2000
[79, DSK, 2002]

At the Prosper-Haniel colliery, flotation tailings, which amount to around 13 to 18 % of total tailings, are transported with trucks on public roads [79, DSK, 2002].

Dewatering of fine tailings
Fine tailings <0.5 mm from flotation are thickened to 25 – 50 % solids. Provided sufficient area for final deposition in engineered ponds is available, fine tailings are directly transported via pipelines or trucks to these facilities. When the deposition of fine tailings on heaps is considered, e.g. for reasons of restricted area capacities, they have to be further dewatered in order to reach a sufficient structural stability.

In principle, three methods can be applied for further reducing the thickened tailings' water content:

- plate-and-frame filter presses, usually featuring more than 1000 m² of filter area (see Section 2.3.1.10)
- in those cases, where a higher water content is acceptable, solid bowl centrifuges (see also Section 2.3.1.10), e.g. used for dewatering flotation tailings
- sedimentation ponds (temporary stowing in ponds, i. e. around three years).

Dewatering by means of sedimentation ponds is carried out as follows: in phase one, the first pond is filled with thickened tailings which then start to settle. In phase two, the pond's content advances in settling, and in phase three, dried tailings are excavated either for deposition on heaps or for external use, e.g. as a construction material. Depending on climatic conditions, each phase can last up to one year. This in turn means, that a set of sedimentation ponds usually consists of three or more adjacent ponds.

In **Spanish coal mines** the coarse material in discarded onto heaps or used as backfill or as filling material in other areas. Flotation slurries are either:

- filtered and sold, or
- filtered and discarded with the coarse tailings, or
- discharged as slurries into tailings ponds.

[84, IGME, 2002]

3.4.3.2.1 Tailings heaps

As shown in the following figure in the year 2000 some 23.4 million tonnes of tailings, out of a total of 33.1 million tonnes, from the Ruhr, Saar and Ibbenbüren area were discarded onto tailings heaps.

The development over time of the tailings heap design in the Ruhr, Saar and Ibbenbüren areas is shown in the following figure.

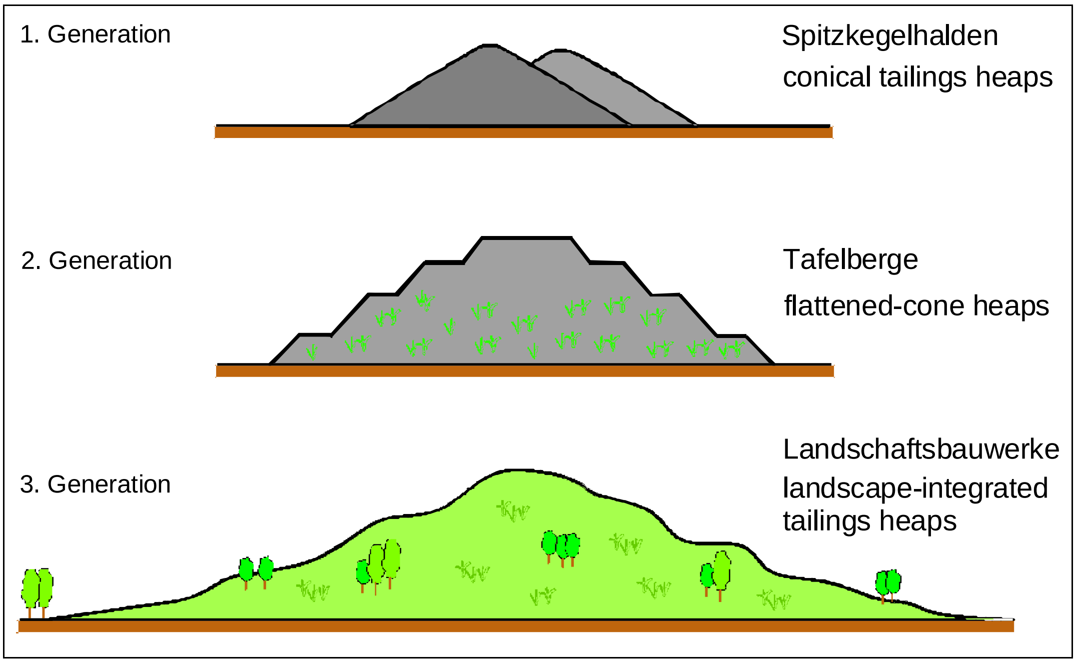


Figure 3.79: Development of tailings heap design in the Ruhr, Saar and Ibbenbüren areas [79, DSK, 2002]

Since the 1970s, the third generation of tailings dumps - so-called landscape-integrated earth constructions – has been established. Since then, these heaps have been accepted as essential landscape elements in the densely populated industrial regions of Ruhr and Saar owing to their high recreational and ecological value.

Principally, tailings are dumped onto the heaps in layers. The thickness of layers ranges from 0.5 to 4.0 m. Compaction is achieved by way of the trucks' rolling wheels and via vibration roolers to reduce, as much as possible, penetration by oxygen or precipitation into the dump body and, thus, minimising or even preventing the generation of ARD by pyrite oxidation.

As an example, the tailings heaps at the Prosper Haniel colliery in the Ruhr region are described:

Currently, operation of the Haniel tailings heap is in its final stage whilst dumping at the new 'Schöttelheide' heap commenced in 1998. Both facilites are so-called "third generation" tailings management facilities (see). The following table provides some information about the sizes of the two heaps.

	Haniel tailings heap	Schöttelheide tailings heap
Start of operations	1963	1998
Final area (ha)	108	66.7
Current area (ha)	108	10.0
Final elevation (m above ground)	126	62
Current elevation (m above ground)	99	5
Overall capacity (million m³)	57.3	15.8
Residual capacity (million m³)	6.3	15.2

Table 3.81: Tailings heaps at the Prosper Haniel colliery in the Ruhr region

Haniel tailings heap

The landscaping of the tailings heap's top section included the construction of an amphitheatre on the top with a seating capacity for 750 persons. To date, this tailings heap represents a unique landscaped earth structure for the Ruhr district with a high cultural interest.

As opposed to the previous planning approval, the heap's flanks hitherto planned for forestation are now solely sown. This, in turn, requires more than 20 ha of forestry compensation measures in the tailings heap's vicinity.

Schöttelheide

For the permitting of the new Schöttelheide heap, the following information was collected:

- water management:
  - hydrologic study, including a groundwater model
  - drainage concept for the tailings heap's surface
  - plan of a hydraulic/subsurface drainage system in the heap's rim area
  - study of the hydrochemical processes in the drainage system, with respect to operational safety
  - compensation measures for balanced water management, retention and discharge of precipitation and leachate emanating from the tailings heap

- dumping:
  - dumping plan, including essential calculations on structural stability and subsidence
  - expert opinion on fire protection during the dumping phase

- emissions, immissions:
  - expert opinion on emissions and immissions of dust
  - expert evaluation on noise emissions and immissions

- climate:
  - expert opinion regarding possible effects on local climatic conditions in the tailings heap's vicinity

- environmental impact study:

- regional development plans:
  - regional development plans for constructing a landscaped earth structure including modelling and recultivation plan
  - regional development plans for tailings truck transport track

- recreation:
  - control of recreational activities at the tailings heap location

- forestry:
  - transition of woodland.

At the beginning of preparatory works, recovery of the growable topsoil from the entire ground area was carried out.

For the Schöttelheide facility, the ring drainage method is applied (see Abb. 4.16). Above the drainage system, a ditch runs along the dump's toe, which collects the surface run-off and transports it to the sedimentation ponds.
With the exception of the Schöttelheide facility's Western area, the underlying ground is impermeable. In a small area only, the ground moraine has hydrological 'windows'. These were sealed by means of compacted tailings material.

Surface run-off, seepage and groundwater are collected in a retention lagoon and discharged by means of a pressure pipeline to the Emscher River.

For documentation and evaluation of the effects resulting from the impacts on the groundwater cycle system, a comprehensive groundwater monitoring system is run, using precipitation measurements as well as surface water and groundwater surveillance. For this purpose, new observation wells were sunk. This set of measures allows the operator to discuss possible changes in groundwater composition with experts at any point and to rapidly initiate necessary measures.

The final heap will consist of two hill tops, with heights of 52 and 62 metres and will rise smoothly out of the surrounding woodlands. Only the lowermost slope is constructed with an incline of 1:2 in the bordering forest areas. The entire heap surface will be made accessible by a large trail system for recreational purposes, which is integrated into the heap's surroundings. The surface will be covered in part by topsoil; some parts; however, will remain 'black' by tailings material.

Planting will be carried out with autochthonous trees and shrubs, i.e. plants that can be found in the surrounding area. Recultivation is scheduled to commence as soon as possible and will progress successively.

For the sake of tipping and of other construction measures, e.g. a retention lagoon, approx. 15 hectares of woodland had to be cut. Around 46.6 hectares are reforested on the heap itself and additional silvicultural replacement measures have to be carried out in the surrounding area.

In the **UK** the tailings heaps are raised to a profile agreed with the competent authorities and are soiled and landscaped on completion. Surface run-off and discharge to watercourses are required to meet specified limits to minimise water quality impacts.

The coarse tailings, typically several hundred thousand tonnes per year, from coal mines in the **Ostrava and Karviná areas** are transported to the heap on conveyor belts or with trucks. In other cases they are used in the reclamation of old tailings basins or for landscaping of subsidence areas.
[83, Kribek, 2002]

3.4.3.2.2 Tailings basins/ponds

Often the fine slurry from flotation is pumped to sedimentation basins (e.g. caused by ground subsidence) or engineered ponds. The settling of tailings is occurs in several ponds/basins in series. The settled tailings are excavated periodically and refloated or sold. The clarified overflows are mostly recycled to the mineral processing plant [83, Kribek, 2002], [84, IGME, 2002].

<u>Hahnwiese pond</u>
The following text describes the experiences of operating a coal tailings pond in an area influenced by underground coal mining, both from the past and scheduled for the future.

The technical features are as follows:

- dam volume: 1.6 million m³
- largest height of dam above strata of the valley: 36 m
- length of dam across the crest: 636 m
- width of dam crest c. 40m, planned as base for future raises
- slopes: 1:2 (water side/upstream), 1:3 (air side/downstream)
- impounding volume: 2.2 million m³.

By calculating the ground movement elements in the planning area originating from past mining activities in two nearby mining districts, the effects had to be assumed as follows:

Ground movement element	Max. amount affecting the investigation area
Subsidence, m	~4 m at dam crest ~5.5.m at dam toe
Elongation (mm/m)	2 - 8 mm/m
Compression (mm/m)	2 - 4 mm/m in dam area

Table 3.82: Effects on TMF resulting from past mining activities

Additionally, the effects of future mining activities were taken into consideration.

Additional investigations within the planning process included:

- assessment of geologic subsurface conditions, including an analysis of existing fracture systems
- development of a groundwater model.

The dam is a staged dam with an upstream core and a filter drainage system. A cut-off trench system made up of interlocking sheet piling sealed at the joints ("sealing lip") represents the central sealing element. A grout curtain aims at preventing seepage underneath the dam. This system can cope well with deformations from mining-induced ground movements.

The dam concept for impounding structures exposed to mining-induced ground movements aims to allow for dangerous situations by installing a redundant control system. Measurements and observation programmes are important means for identifying irregularities at the impounding structure as well as during operations. Only by early identification, can directed measures be undertaken and major damages at the tailings pond system prevented. The concrete measures for improving situations detrimental to operational safety and stability are listed below.

Source of concern	Observation	Observed by or at	Resulting risk	Possible measures, follow-up
G + M	High ground water mobility in dam areas, in underground	Phreatic surface gauging points, hydraulic gauging station	Water losses from pond, erosion problems	Mud layers and/or injections, directional mud discharge, installation of mud discharge banks
G + M	Rise of water below the dam	Gauging stations in conjunction with drainage outflow	Scour, undermining, headward/regressive erosion	Sealing measures in underground, grout curtain, drainage checks
M	After core seal elongation: water rise in downstream dam side, saturation line	Gauging stations, drainage	Imperfection of core seal, erosion	If possible, resealing of core, eventually drainage cleaning, reinforcing the dam toe; layers of suitable material after rise of saturation line
G	Sedimentation in drainage pipes	Inspection with TV camera	Reduced water flow up to backwater, and thus affecting saturation line in the dam	Rinsing/cleaning, removing sediments by mechanical means or chemicals (e.g. acidic solution)
M	Subsidence, slumping, settlement of earth dam	Levelling, gauge measurements	Overflow at dam crest	Raising the dam, if necessary by extending internal core seal (incl. spillway)
M	Cleats, fissures, contraction joints in underground, in dam and in impoundment area	Visual observations, linear measurements, if necessary pond bed topographic survey	Scour, erosion	Filling or sealing with impervious material (e. g. loam)
M	Movement at abutment of spillway bridge	Visual observations, inclination measurement, position measurement	Loss of the spillway bridge's necessary support moment	Adaptation of bridge abutment
M	Movement at sockets of surplusing works	Special measurements at spillway/pipelines	Damages at sockets of spillway, water spill in pipeline trench, by-passing, headward/regressive erosion	Pipeline enhancing, if necessary by inserting inliner
M	Movement of spillway	Position/tilt measurement	Damages at link to pipeline, by-passing	Pipeline enhancing, if necessary by inliner
M	Movements at safety elements of tailings pipeline (stretcher)	Geometric control of stretcher	Leakage, headward/regressive erosion	Pipeline re-adjustment within stretcher limits
G	Precipitation in drainage outlet pipes	Inspection with TV camera	Reduction of tube diameter, blockage/backwater followed by erosion	Rinsing, mechanical cleaning
G + M	Indications of earth dam failure	Cracks in dam with quickly regressing erosion in conjunction with failure of sealing elements and drainage	Failure of dam, dam collapsing	Quick emergency relief via spillway (down to mud level)
Source of concern: G: general, given by circumstances and operations M: initiated by mining activities				

Table 3.83: Tailings ponds influenced by mining-induced ground movements: Catalogue of potential risks and counter measures

3.4.3.3 Safety of the TMF and accident prevention

The Ostrava and Karviná area has a high seismic risk. Therefore seismic events are monitored [83, Kribek, 2002].)

3.4.3.4 Site closure and after-care

Basically, five types of subsequent utilisation of tipping locations are common in the Ruhr, Saar and Ibbenbüren areas:

- forestal utilisation
- agricultural utilisation
- installations for leisure and recreational purposes
- secondary biotopes
- new industrial areas.

Land availability is very limited in the densely populated areas of the Ruhr and Saar coalfields. Areas under use for industrial purposes such as tailings management have to be reintegrated into the landscape as rapidly as possible.

The dumped tailings are sampled immediately after dumping, after the two years and after three years as far as required. Per each 2500 m² dump area, three samples from depths between 0 and 20 cm are taken and blended for a representative mixed sample. One sample is taken from a depth between 40 to 50 cm. Investigation of sample material includes pH-value determination to identify the acidification grade, total sulphur content (first sample) and total alkalinity content. For the second samples, the contents of P_2O_5, potassium, calcium and magnesium accessible to plants are determined. These results are taken into account for the soil cover and the revegetation.
[79, DSK, 2002]

The subsequent utilisation of a tailings location results from a balanced consideration of the ecological, environmental, recreational, and economic aspects. As shown by the example of an amphitheatre (Bergtheater (”mountain theatre”)) erected on the Haniel tailings heap, also cultural and sports aspects can be taken into account. Further examples include a large hall structure built on the Prosperstrasse tailings heap for downhill skiing, and the exposed location of an art monument, such as the Tetraeder (”tetrahedron”) on the Beckstrasse landscaped earth structure.

Tailings management heaps in Germany’s coal districts are often designed by landscape architects taking into account many ideas from the public.

Ongoing re-vegetation already during operation can be accelerated by different measures (see Section 4.3.6). After completion of the slope areas, the dump surface is sown with herbs seed. The herbs layer assists the heap’s integration into the landscape, prevents erosion to a major extent and contributes to humus formation in the uppermost soil layer. Sizing and composition of seed mixture is dependent on local situation at individual dumps, on ground structure and on climatic influences. For wet sowing, water is used as carrier. Apart from the seed, fertiliser, soil amelioration agents and mulch, mixed with water, can also be applied.

Next step shrubs and trees are chosen only after evaluation of soil investigations. Selecting the plants and designing the planting scheme is done in close co-operation with forestry authorities. Plant material, in most cases, is taken from tree nurseries after a growing period of three years and planted with a narrow spacing between 1 m x 2 m to 1 m x 1 m.

Apart from the vegetation measures described above, by landscaping wet and dry bi-topes, small water courses as well as by creating areas left to natural succession, reclamation in the

Ruhr, Saar and Ibbenbüren areas aims at creating the basis for a variety of flora and fauna habitats.
[79, DSK, 2002]
A regional closure plan for the landscaping of mines and tailings management facilities in the Ostrava and Karviná area has been developed [83, Kribek, 2002].

3.4.4 Waste-rock management

The small amounts of waste-rock from underground operations are managed with the coarse tailings on the heaps.

Normally waste-rock arising from UK open pit mines is managed in temporary heaps in accordance with the technical requirements of the Health & Safety at Quarries:- Quarries Regulations 1999 – Approved Code of Practice. After removal of coal deposits, the waste-rock is then returned to the void and restored in accordance with the Planning Consent. Note that removal of the overburden from site is normally specifically prohibited by the Minerals Planning Authority

Waste-rock heaps are raised to a profile agreed with Mineral Planning Authorities in the UK and are soiled and landscaped on completion. Surface run-off and discharge to watercourses are required to meet specified limits to minimise water quality impacts.

3.4.5 Current emission and consumption levels

3.4.5.1 Management of water and reagents

The reagents used in the flotation of coal are mixtures, whose composition is only partially known. Also they are subject to the variations of any product from large scale refinery processes. In most cases mixtures of certain light oil fractions (collectors) or alcohols (frothers) together with emulsifiers are used. The flotation reagents used can contain traces of up to 50 different substances.

Whilst the salt and metal contents of coal and their leachability are well known, the content of organic chemicals is not so well documented. It is assumed that most contaminants will accumulate on the fine flotation tailings because of their large specific surface. Organic contaminants can originate from flotation reagents, as mentioned above, but also from hydraulic oils used in the mining operation.

Conventional methods of analysing the content of organic chemicals in the coal tailings are prone to errors, firstly because they are not suitable for such small concentrations but also because these methods dissolve naturally present hydrocarbons. However by means of radioactive tracing (i.e. by using ^{14}C) it can be shown that 1 kg of flotation tailings contains 120 mg of flotation reagents. This 'load' decreases with increasing ash contents of the tailings.
[102, Diegel, 1994]

Although flotation reagents can accumulate on the surface of fine tailings, they remain immobilised. By applying along-term monitoring of surface run-off and percolation water from tailings heaps, it was demonstrated that no water contamination occurred due to the organic constituents of the flotation reagents. This is attributed to the tight binding of the organic components and to the compacted construction of the entire heap.

In German hard coal processing plants, flotation reagents based on hydrocarbons or alcohols are applied. For flocculation, reagents based on polyacrylates or polyacrylamides are used.

In addition to fine coal, the following lists some typical reagents used in coal mineral processing plants in the US:

- anionic or cationic flocculants
- lime
- natural and modified starches
- caustic starch
- sulphuric acid as pH adjuster
- alum (aluminium sulphate) as pH adjuster
- anhydrous ammonia.

[81, MSHA, 2002]

The clarified water from basins/ponds in the Ostrava and Karviná area are re-used in the mineral processing plant. Surplus water is discharged to surface water.

In flotation the agent Flotalex, which is a mixture of alcohols and mineral oil, is used in concentrations of 0.25 - 0.35 kg/t. As a flocculant an organic agent based on polyacrylamide is added.
[83, Kribek, 2002].

3.4.5.2 Emissions to air

To minimise dust and noise emissions from dumping tailings transport and spreading operations, ramps and working benches are transferred into the heap's inner area as far as possible and are shielded by embankments or hollows [79, DSK, 2002].

3.4.5.3 Emissions to water

Fine tailings from flotation are often managed in ponds and basins (e.g. Ostrava and Karviná area). Most of the clarified water is re-used in the mineral processing plants. However, in some cases surplus water is discharged to surface water. The amounts of discharge per year and the concentrations of emissions to surface water are shown in the following table.

Parameter	Units	Site				
		Paskov	CSA	Lazy	Dukla	CSM
Discharge	Mm^3	0.2	2.0	1.6	4.0	0.27
COD	mg/l	22208	16985	19.19	50.91	1920.2
BOD	mg/l		2333	4.34	6.54	20.65
Total soluble matter[1]	mg/l		1310			
Soluble inorganic salts[2]	mg/l	687833				
non-soluble matter	mg/l	131667	7166	9.88	20.58	285.4
P total	mg/l	0.04				
$N\text{-}NH_4$	mg/l	0.06	0.33	0.2	1.48	
Cl	mg/l		382.5			
SO_4	mg/l	204.5	290.5			
PO_4	mg/l		0.055			
Phenols	mg/l		0.1			
Fe	mg/l			0.17	0.22	
Mn	mg/l			0.09	0.14	
Hg	µg/l	0.9				
Cd	µg/l	0.5		<0.005	<0.005	
CN total	µg/l		6			
FN	mg/l		0.1			
pH			8	8	7.61	

1 total soluble (not suspended) matter (organic and inorganic) obtained from the sample after filtering and washing with distilled water
2 soluble inorganic salts are determined after oxidation of the total soluble matter fraction with H_2O_2 using the gravimetric method

Table 3.84: Amount of discharge and concentrations of emissions from tailings ponds/basins in the Ostrava and Karviná area in 2000
[83, Kribek, 2002]

4 BEI DER FESTLEGUNG VON BVT ZU BERÜCKSICHTIGENDE TECHNIKEN

Dieses Kapitel beschreibt eine Reihe von Techniken zur Vermeidung oder Reduzierung von Emissionen sowie Techniken zur Vermeidung bzw. Minderung von Zwischenfällen entsprechend Kapitel 6.3 der Kommunikation (COM (2000) 664). Sie sind ausnahmslos verfügbar und werden eingesetzt.

4.1 Allgemeine Grundsätze

Erfolgt bei der Planung des Gesamtbetriebs (Bergwerk, Aufbereitungsanlage, Bewirtschaftungsanlagen für Rückstandsaufbereitung und taubes Gestein) bereits die Berücksichtigung der Beschaffenheit der Aufbereitungsrückstände und werden die unterschiedlichen chemischen, physikalischen und biologischen Wechselbeziehungen durch bergbauliche Tätigkeit und Verarbeitung berücksichtigt, können die Umweltprobleme und Kosten durch die Bewirtschaftung von Aufbereitungsrückständen und taubem Gestein reduziert werden [21, Ritcey, 1989]. Zudem ist die Bewirtschaftung der Aufbereitungsrückstände und des tauben Gesteins, einschließlich des Wasserhaushalts, in der Regel ein integraler Teil des Gesamtlebenszyklus eines Betriebes und von so grundlegender Bedeutung wie die eigentlich Förderung [45, Euromines, 2002].

Die professionelle Bewirtschaftung von Aufbereitungsrückständen und taubem Gestein beinhaltet die Bewertung alternativer Möglichkeiten für:

- die Minimierung der anfallenden Menge an Aufbereitungsrückständen und taubem Gestein, z. B. durch die Wahl der geeignetsten Abbauweise (Tagebau, Untertage, verschiedene Verfahren des Untertagebergbaus)
- die Maximierung der alternativen Verwendungsmöglichkeiten für Aufbereitungsrückstände und taubes Gestein, z. B. durch Verwendung:
  - als Zuschlagstoffe
  - bei der Wiederurbarmachung anderer bergbaulicher Standorte
  - als Verfüllung
- Behandlung von Aufbereitungsrückständen und taubem Gestein als Teil der Minimierung der Umwelt- und Sicherheitsgefahren, z. B.
  - Depyritisierung
  - Einsatz als Pufferstoff.

Die wirksamste Methode der Reduzierung der anfallenden Menge an taubem Gestein ist der Abbau im Untertagebetrieb statt im Tagebau. Der Abbau über Tage mag ökonomische Vorteile gegenüber dem Untertagebergbau haben, doch ändert sich dies grundlegend mit der Optimierung des Abbaus der vorhanden Bodenschätze. So ist es häufig möglich, einen wesentlich größeren Teil der Lagerstätte im Tagebau zu nutzen.

Wie aus Kapitel **2.1** ersichtlich, ist jedoch eine Vielzahl von Gesichtspunkten bei der Entscheidung über das anzuwendende Bergbauverfahren, z. B. Über- oder Untertagebau oder kombinierte Formen dieser beiden wichtigsten Alternativen, zu berücksichtigen. Anfall und Bewirtschaftung von taubem Gestein sind Aspekte, Sicherheit, Arbeitsbedingungen, Kosten, Optimierung des Einsatzes von Ressourcen, Stabilität, geometrische Form und Tiefe der Lagerstätte usw. sind weitere bei der Entscheidung über die günstigste Abbaumethode zu berücksichtigende Aspekte. Nach welcher Methode auch gearbeitet wird, es liegt in keinem Fall im Interesse des Betreibers, taubes Gestein in größeren Mengen als notwendig zu produzieren, da dessen Bewirtschaftung für den Bergbaubetrieb den Einsatz von Rohstoffen bedeutet und einen Kostenfaktor darstellt, so dass die Gesellschaft damit wenig oder keinen Gewinn erwirtschaftet.

Zumeist wird eine **Risikoeinschätzung** vorgenommen, mit der die eingesetzten Techniken hinsichtlich ihrer Eignung für die bestehenden Bedingungen unter ökologischen, sicherheitsrelevanten, technischen und technologischen Gesichtspunkten bewertet werden. Zur Identifizierung möglicher Ursachen für das Versagen von TMF und damit zur Vermeidung weiterer Probleme, steht die Gesamtbetrachtung stets unter der Frage "*Was wäre, wenn*?". Das heißt, es müssen verschiedene Szenarien betrachtet und auf deren Grundlage der möglichen Folgen Havarie- und Notfallpläne entwickelt werden. Vor allem aber – und darauf kommt es an – müssen diese den Beschäftigten bekannt und von diesen verstanden werden.

(Aus Gründen der Zugänglichkeit der Lagerstätte, aus Sicherheitsgründen usw.) unvermeidbare Aufbereitungsrückstände und taubes Gestein, für das keine alternative Verwendungsmöglichkeit besteht (z. B. bedingt durch physikalische oder chemische Eigenschaften, Transportkosten, fehlender Markt), erfordern ein geeignetes Bewirtschaftungskonzept mit folgender Zielorientierung:

- Sicherheit, Stabilität und Effektivität der Bewirtschaftung von Aufbereitungsrückständen und Taubgestein bei Minimierung des Risikos der störfallbedingten kurz-, mittel- oder langfristigen Einbringung in die Umwelt
- Minimierung von Menge und Toxizität der kontaminierten Freisetzung / Sickerung von der Bewirtschaftsungsanlage
- fortschreitende Reduzierung des Risikos über die Zeit.

Bei Anfall verschiedener Arten von Aufbereitungsrückständen und taubem Gestein wäre eine sortenreine Erfassung eine Vorleistung für eine mögliche spätere Rückgewinnung für alternative Einsatzzwecke oder Weiterverarbeitung. Auf der anderen Seite könnte das Mischen der verschiedenen Aufbereitungsrückstände und/oder tauben Gesteine eine positive Möglichkeit der ökologischen Bewirtschaftung, z. B. mit der Zielstellung der Minimierung von Sauerwässern, sein.

4.2 Lebenszyklus-Management

Eine effektive Minderung des Ausfallsrisikos ist nur möglich durch die Anwendung und konsequente Umsetzung wirksamer technolgischer Methoden bei Auslegung, Betrieb und Stilllegung von TMF für deren gesamte Betriebsdauer.

4.2.1 Auslegungsphase

Zur Sicherung einer umweltbewussten Bewirtschaftung von Aufbereitungsrückständen und taubem Gestein ist es wichtig, dass der Betrieb bereits in der Anfangsphase auf eine spätere Stilllegung hin konzipiert wird und der Quantifizierung der langfristigen Umweltfunktionen und Auswirkungen von TMF/WRMF entsprechende Beachtung geschenkt wird. Die folgende Abbildung zeigt den Informationsfluss für eine 'Stilllegungsplanung'.

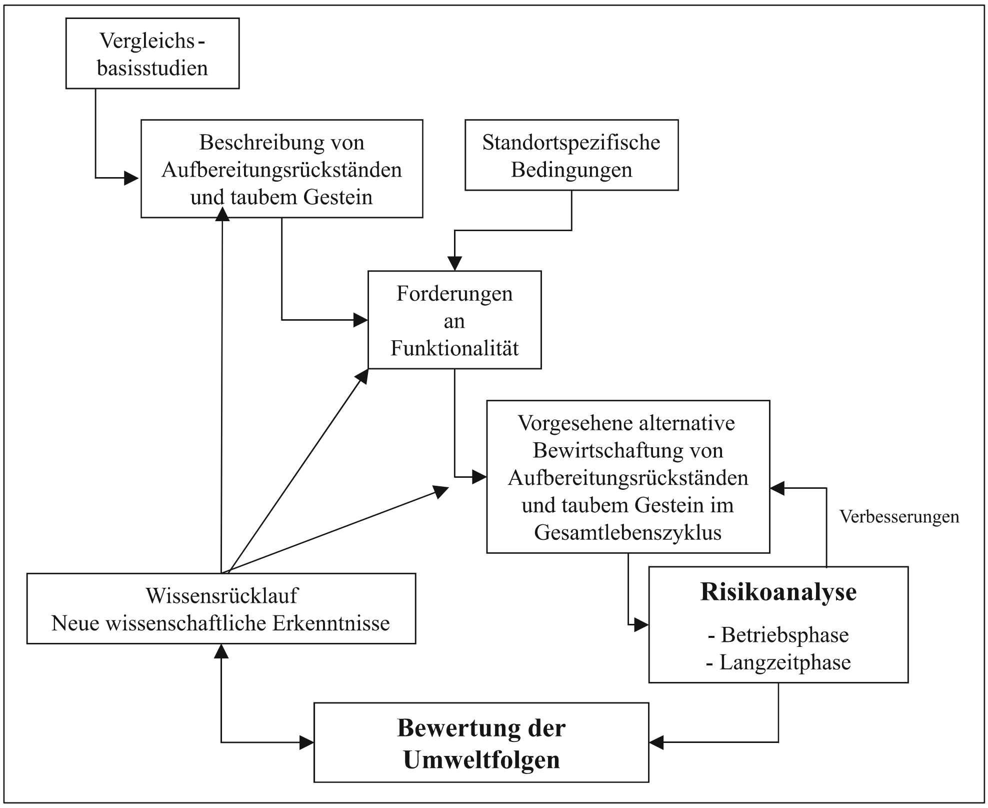


Abb. 4.1: Darstellung des Informationsflusses für eine 'Stilllegungsplanung

In diesem Kapitel werden die Überlegungen in der Planungsphase einer TMF bzw. Bewirtschaftungsanlage für taubes Gestein (WRMF) beschrieben. Soweit nicht anders angegeben, sind die Angaben dem "Canadian guide to the management of tailings facilities", [18, Mining Association of Canada, 1998] dem "Framework for mining waste management" [45, Euromines, 2002] und mündlichen Mitteilungen von TWG-Mitgliedern entnommen.

4.2.1.1 Ökologische Vergleichsbasis

Es folgt eine Zusammenschau der bei der Erfassung und Ordnung von Umweltdaten für Standortwahl, Auslegung und Betrieb von TMT / WRMF einzubeziehenden Überlegungen. Diese Ausgangsinformationen bilden die Grundlage für die Entwicklung von Stilllegungsplänen und Umweltüberwachungsprogrammen. Ausführlichere Darlegungen können spezifischen ökologischen Bewertungsrichtlinien entnommen werden.

- Vorhandene Ressourcen und deren Nutzung – Die vorhandenen Ressourcen und Bodennutzungen im Bereich der Rückstandsaufbereitungsanlage und innerhalb des größeren potentiellen Einflussbereiches müssen definiert werden, insbesondere:
  - Nutzung von Boden und Gewässern:
    - Aktuelle und historische Nutzung, einschließlich für Erholungszwecke, als Parks, Wohnbereiche, zur Trinkwassergewinnung, unter archäologischen Gesichtspunkten, für Bergbau, Holzeinschlag, Landwirtschaft, Jagd und Fischerei
  - Landbesitzverhältnisse:
    - Sicherung des Erwerbsrechts auf das benötigte Land für die Errichtung einer TMF/WRMF
    - Feststellung der Besitzverhältnisse an Grund und Boden sowie Lagerstätten

- Wissenschaftliche Vergleichsdaten – Die ökologischen Ausgangsdaten für den vorgesehenen Standort der Anlage für die Rückstandsaufbereitung müssen erfasst werden, insbesondere:
-
  - Physikalische Daten:
    - Klima (z. B. Temperatur, Wind, Niederschläge, Verdunstung, regelmäßige Überschwemmungen, Niederschlags- und Oberflächenabfluss, Luftqualität)
    - Wasser (z. B. Hydrologie, Verlauf von Wassereinzugsgebieten, Strömungsmuster, Abflüsse, Gewässertiefen, Tiefenhydrogeologie und Grundwasserqualität, Qualität von Oberflächenwasser und Sedimenten)
    - Geländeformen
    - Geologie und Geochemie (z. B. Oberflächenablagerungen (Art, Ort, Dichte, Durchlässigkeit), Schichtung, Geomorphologie, Mineralogie, Vorkommen von Zusatzelementen)
    - Topographie (z. B. topographische Regional- und Detailkarten, Stereoluftbildaufnahmen, Satellitenbilder)
    - Böden (z. B. Bodenbeprobung und Analyse)
    - natürliche Gefahren (Erdrutsche, Lawinen, seismische Vorgänge, Überschwemmungsgefahr, Frosteinwirkungen)
    - Angaben zu früheren Bergbaustandorten in der Nähe von oder unter TMF/WRMF

  - Biologische Daten:
    - Beschreibung des Ökosystems
    - Terrestrische Daten (z. B Flora, natürliche Weidegründe, Fauna, gefährdete und bedrohte Arten, Migrationsarten)
    - Gewässerdaten (Benthos, Makroinvertebrate, Fische, Wasserpflanzen)

- Sozioökonomische Vergleichsdaten – Sozioökonomische Vergleichsdaten mit Relevanz für den Projektbereich der Anlage zur Rückstandsaufbereitung, einschließlich:
  - historische Hintergrunddaten
  - Bevölkerung
  - regionale Wirtschaft (z. B. Gesundheit, Bildung, Kultur, Demographie)
  - Herausarbeitung von potentiellen sozioökonomischen Problemen im Zusammenhang mit dem Projekt für die Anlage zur Rückstandsaufbereitung.

Eine Vergleichsstudie der Ausgangssituation wird in der Regel als Teil der Umweltverträglichkeitsprüfung (EIA) erstellt.

In dieser Untersuchung der Ausgangssituation werden die potenziell durch einen Standort gefährdeten Ressourcen definiert und Daten zur Beschreibung dieser Ressourcen bereit gestellt. Damit liefert die Studie Informationen, auf deren Grundlage die ökologischen Einflüsse des Projektes bestimmt werden können und eine Datenbank zur Bewertung künftiger Veränderungen der Umweltqualität [25, Lisheen, 1995]. Eine gut ausgeführte Ausgangsstudie liefert dazu wertvolle Daten für die weitere Auslegung, das Layout und die Planung des Standortes.

Es ist anzumerken, dass der Inhalt einer Ausgangsstudie in jedem Fall getrennt festzulegen ist. So wird zum Beispiel der Umfang von Art und Größe des vorgesehenen Betriebs bestimmt. Die Bestimmung von Metallanteilen wäre in Fällen, bei denen Belastung durch Metalle von vornherein ausgeschlossen werden kann, nicht sinnvoll.

Anhang 3 zeigt an einem realen Beispiel den Umfang einer vor kurzem durchgeführten Ausgangsstudie.

4.2.1.2 Charakterisierung von Aufbereitungsrückständen und taubem Gestein

Entscheidend für die richtige Bewirtschaftung von Aufbereitungsrückständen und taubem Gestein ist die adäquate Beschreibung der Rückstände. Die Beschreibungsergebnisse bestimmen die Bewirtschaftung der Aufbereitungsrückstände und des tauben Gesteins im Betrieb (Ablagerung, Schutzmaßnahmen usw.), bei Stilllegung (Voraussetzungen und Techniken der Stilllegung) sowie in der Nachsorgephase (Einschätzung des Langzeitverhaltens).

Im Idealfall erfolgt die zuverlässige Beschreibung von Aufbereitungsrückständen und taubem Gestein vor Betriebsaufnahme und die Ergebnisse werden in vollem Umfang in die Auslegung der Bewirtschaftungseinrichtungen und die Bewirtschaftungspläne einbezogen. Die Beschreibung umfasst physikalische und chemische Beschaffenheit, die Aussagen zum kurz-, mittel- und langfristigen Auflösungs-/Verwitterungsverhalten (Freisetzung von Elementen) sowie zum geotechnischen Verhalten gestatten. Bei dieser Aktivität, die oft entsprechend der erhaltenen Ergebnisse in Etappen abläuft, wird mit einer Reihe von Methoden von der relativ einfachen Analyse über relativ komplexe Auslaugungsversuche bis zu komplexen Interpretations- und Vorhersagemodellen gearbeitet.

Die folgenden Beschreibungen von Erz, taubem Gestein (bei Verwendung zum Dammbau oder bei Bewirtschaftung in der gleichen TMF), Aufbereitungsrückständen und Aufbereitung finden Eingang in die Auslegung der TMF/WRMF:

- Charakterisierung von Erz und taubem Gestein:
  - Umfang der Lagerstätte
  - Mineralogie
  - chemische Eigenschaften
  - physikalische und technologische Eigenschaften
  - Säurebildungspotenzial
  - auslaugbare Verunreinigungen
  - Erze und Veränderungen der Erzqualität über den Nutzungszeitraum des Bergwerks
  - Armerz und Gangart nach Qualität und Abbau
  - kinetische Prüfungen
  - Korngrößenverteilung
  - hydrologische Eigenschaften[9]

- Charakterisierung der Aufbereitungsrückstände, einschließlich der allgemeinen Beschreibung der physikalisch-chemischen Eigenschaften, z. B.:
  - Tages-/Jahresdurchsatz und Gesamtmenge
  - Größenverteilung
  - feste oder abgeschlämmte Aufbereitungsrückstände, Teigdichte (prozentualer Anteil an Feststoffen)
  - Feststoffdichte
  - Stabilität/Plastizität
  - Chemie der flüssigen Phase
  - Säurebildungspotenzial
  - geochemische Charakterisierung (Metallgehalt, Laugungsverhalten)
  - Porenwasser
  - Konsolidierungsverhalten
  - kinetische Prüfung
  - Mineralogie
  - hydrologische Eigenschaften[10].

[9] Mineralogisch-hydrologische Eigenschaften sind entscheidend bei der Durchführung geochemischer Untersuchungen (Wasserqualität, Reaktionsverhalten und Massenbelastung (Walder u. a. in Vorb., Environmental Geochemie of Ore Deposits, S. 250 ff)

[10] Mineralogisch-hydrologische Eigenschaften sind entscheidend bei der Durchführung geochemischer Untersuchungen (Wasserqualität, Reaktionsverhalten und Massenbelastung (Walder u. a. in Vorb., Environmental Geochemie of Ore Deposits, S. 250 ff)

- Charakterisierung der Aufbereitung:
  - eingesetzte Reagenzien mit Konzentration und Menge
  - Bedarf an Kreislaufwasser
  - Prozesse in der Aufbereitungsanlage (z. B. Cyanabbau)
  - sonstige Zuflüsse in das Absetzbecken
  - Rohrleitungen und ähnliche Konstruktionen
  - Möglichkeit des untertägigen Verkippens oder Verfüllens von Gruben
  - Verhältnis der übertägigen Bewirtschaftung von Aufbereitungsrückstände zu Verfüllung.

[18, Mining Association of Canada, 1998]

Der Aufbau von Methoden einer kosteneffektiven Bewirtschaftung von Aufbereitungsrückständen und taubem Gestein erfordert genaue Aussagen zum Verhalten dieser Rückstände in der natürlichen Umwelt. Zur Charakterisierung von Bergbaurückständen und Bewertung des Potenzials von bergbaulichen Aufbereitungsrückständen und taubem Gestein zur Entwicklung von Azidität oder Metall belastetem Schmutzwasser stehen international zahlreiche Prüfverfahren und Prognosemethoden zur Verfügung und werden angewendet. Die Zuverlässigkeit dieser Methoden wird von der Wertigkeit zahlreicher wichtiger chemischer und mineralogischer Variablen und Faktoren, unter denen die Entsorgung der Abfälle erfolgt und der Entwicklung von umfassend dokumentierten Standardmethoden für die Charakterisierung der bergbaulichen Mineralrückstände und sonstiger Minerale bestimmt.

Eine Zusammenfassung der zur Verfügung stehenden Methoden für die geotechnische und geochemische Charakterisierung von Aufbereitungsrückständen und taubem Gestein enthält Anhang 4. Zur Bewertung der möglichen Qualität und der Strömung des Austragswassers werden die Ergebnisse dieser Charakterisierung mit den relevanten Daten (d. h. für die Vergleichsstudie erfasste physikalische Daten) eines bestimmten Standortes zusammengefasst. Bei der Auswertung werden die verschiedenen Skalisierungseffekte zwischen Labor und Feldmaßstab berücksichtigt. In den meisten Fällen basieren Aussagen zum Verhalten verschiedener Bewirtschaftungsoptionen auf Rechenmodellen.

4.2.1.3 Studien und Pläne zu TMF/WRMF

Es folgt eine Zusammenfassung von Studien und Plänen aus der Erarbeitung des Auslegung für eine reale TMF/WRMF bis auf eine relevante Detailebene der einzelnen Phasen (Konzeption, Vorauslegung und Detailauslegung) mit anschließender Weiterführung durch die gesamte Betriebphase bis zu Stilllegung:

- Dokumentation der Standortwahl
- Umweltverträglichkeitsprüfung
- Risikoeinschätzung
- Einsatzbereitschaftsplan für Katastrophenfälle
- Ablagerungsplan
- Wasserbilanz und Haushaltsplan und
- Außerbetriebnahme- und Stilllegungsplan.

Die oben genannten Planinhalte sind in jedem Falle Mindestforderungen. Praktisch können fallbezogen bei Bedarf weitere Themen berücksichtigt werden.
[18, Mining Association of Canada, 1998]

Die genannten Themen werden im Folgenden vertieft.

Standortwahl

Der Betreiber wählt den gewünschten Standort aus und erarbeitet eine dokumentierte Begründung für die Standortwahl, einschließlich Diskussion untersuchter und verworfener Alternativstandorte. Ferner sind Probleme der Sensibilisierung der Öffentlichkeit im

Zusammenhang mit dem Projekt (d. h. Forderungen interner und externer Interessengruppen) aufzuzeigen. Zu Fragen der Standortwahl zählen:

- Umweltaspekte:
  - Notwendigkeit der Schmutzwasserbehandlung
  - Emissionen in Oberflächenwasser
  - Emissionen in Grundwasser (hydrogeologischer Sicherheitsbereich)
  - historische Nutzung des Wassereinzugsgebietes
  - ökologische Hintergrundbedingungen
  - Auswirkungen auf Vegetation, Tierbestand und Wasserbewohner
  - natürliche Flora und Fauna
  - archäologische Gesichtspunkte
  - potenzielle Emissionen in die Luft
  - Aspekte des Erscheinungsbildes
  - Wasserbilanzkonzeption

- Planungsaspekte:
  - Zugang (Straßenbau)
  - Entfernung von der Aufbereitungsanlage
  - relative Höhenlage gegenüber der Aufbereitungsanlage
  - Abstand zu Wohngebieten und Bereichen menschlicher Tätigkeit
  - Topographie
  - bestehende Nutzung von Boden und Ressourcen
  - Besitzverhältnisse und Schürfrechte
  - Transportkorridore, Energieleitungen usw.
  - Aspekte von Wassereinzugsbereich und Oberfläche
  - Volumenmenge
  - Verhältnis Fassungsvermögen des Absetzbeckens zu Vorhaltungskapazität
  - Geologie, einschließlich potenzieller Erzlagerstätten
  - Verfügbarkeit von Baustoffen
  - Konflikte mit bergbaulicher Tätigkeit
  - Voraussetzungen für Dammgründung
  - Voraussetzungen für Beckengründung
  - abflussseitige Gefahren
  - Hydrologie
  - Grundwasser, Sickerung von Kontaminationsstoffen
  - potenzielle Einflussbereiche
  - Gefahren für Mensch und Umwelt
  - Wasserhaushaltsplan und vorläufige Wasserbilanz
  - Betriebsplan
  - Ablagerungsplan
  - vorläufige Speicher- und wasserwirtschaftliche Bauten
  - vorläufiger Kostenplan auf der Grundlage vorläufiger Prämissen
  - konzeptionelle Risikoeinschätzung
  - Gesundheits- und Arbeitsschutzbewertung

- Aspekte von Außerbetriebnahme/Wiederurbarmachung:
  - Wasserablaufberechnung
  - potenzielle Wiederurbarmachung
  - physikalisch-chemische Langzeitstabilität
  - Möglichkeit der Errichtung von Dauerdrainage
  - Verminderung und/oder Kontrolle des Sauerwasserabflusses und anderer Belastungen
  - Vermeidung von Staubemissionen
  - Voraussetzungen für langfristige Wartung, Überwachung und Behandlung

- Aspekte zu Aufschluss-, Betriebs- und Stilllegungskosten:
  - Investitionskosten

- Kosten für den Transport der Aufbereitungsrückstände
- Betriebs- und Wartungskosten für die Anlage für Aufbereitungsrückstände
- Stilllegungskosten
- Kosten je Tonne verarbeitetes Erz.

Umweltverträglichkeitsprüfung

Für die Zustimmung der verschiedenen Interessenvertreter sowie der Genehmigungsbehörden zum Standort einer neuen TMF/WRMF ist es häufig erforderlich und auch eine gesetzliche Forderung, eine Umweltverträglichkeitsprüfung (EIA) durchzuführen. In EU-Mitgliedsländern ist die Durchführung einer EIA durch die Richtlinie des Rates 97/11/EG vom 03. 03. 1997[11] in Änderung von Richtlinie 85/337/EWG vom 27. 06. 1985 zur Umweltverträglichkeitsprüfung bei bestimmten öffentlichen und privaten Projekten[12] geregelt. Die Richtlinie gestattet es den Mitgliedsländern, bei bestimmten Aktivitäten über die Notwendigkeit einer EIA zu entscheiden. Jedoch sind nach Anhang I der Richtlinie Steinbrüche und Tagebaue mit einer Abbaufläche von mehr als 25 Hektar zur Durchführung einer Umweltverträglichkeitsprüfung verpflichtet. Anhang II der Direktive legt fest, dass die Mitgliedsländer frei entscheiden können, ob für untertägige Gruben und kleinere Steinbrüche und Tagebaue eine EIA erforderlich ist. Die Angaben, die der Betreiber in der EIA machen muss, sind in Anhang IV der Umweltverträglichkeitsrichtlinie geregelt. Auf der Internetseite http://europa.eu.int/comm/environment/eia/home.htm sind umfangreiche Informationen und Anleitungen zur Durchführung von EIA abrufbar.

Mit Hilfe von Vergleichsstudien werden die bestehenden Bedingungen vor Betriebsaufnahme eines neuen Standortes ermittelt. Damit bilden sie die Grundlage für eine mögliche anschließende Bestimmung und Abschätzung der Auswirkungen. Der genaue Umfang von Vergleichsstudie und Umweltverträglichkeitsprüfung wird in der Regel in einer von der Genehmigungsbehörde durchzuführenden Prüfung festgelegt. Bei Bedarf werden weitere Beteiligte gehört.

Der Prozess der Umweltverträglichkeitsprüfung erfordert das Einbringen von Kenntnissen zum Projekt im Auslegungszustand, der natürlichen und sozialen Umwelt, in der das Projekt angesiedelt ist sowie die Anhörung von Beteiligten und Betroffenen. In der Phase der Umweltverträglichkeitsprüfung werden Anlagen für Aufbereitungsrückstände zumeist als Komponenten eines größeren, integrierten Projekts betrachtet. Nachfolgend eine Zusammenfassung wichtiger Aspekte in bezug auf Aufbereitungsrückstände, die in einer Umweltverträglichkeitsprüfung Berücksichtigung finden müssen:

- bestehende Umweltsituation
- Aspekte der Aufbereitungsrückstände bei der Mineralverarbeitung
- Standortwahl bei Anlagen für Aufbereitungsrückstände und taubes Gestein mit eingehender Begründung des gewählten Standortes
- Konzeptauslegung der Anlage für Aufbereitungsrückstände und taubes Gestein.

Die Umweltverträglichkeitsprüfung arbeitet die wahrscheinlichen Umweltfolgen der Anlage für Aufbereitungsrückstände und/oder taubes Gestein heraus, z. B.:
- physikalische Auswirkungen
- Physiographie
- Klima und mögliche Folgen von Klimaveränderung
- Qualität der Luft
- Lärm
- Hydrologie
- Hydrogeologie
- Wasserqualität

[11] Amtsblatt Nr. L 073 vom 14. 03. 1997

[12] Amtsblatt Nr. L 175 vom 05. 07. 1985

- biologische Auswirkungen
- Wasserlebewesen
- Vegetation
- wildlebende Tiere
- archäologische Auswirkungen
- sozioökonomische Folgen
- Auswirkungen auf die Flächennutzung.

Risikoeinschätzung

In großen Teilen von Kapitel **3** wird deutlich, dass Methoden zur Vermeidung von Unfällen in der Tat auf Risikoeinschätzungen beruhen. Ferner wird durch die Novellierung der Seveso-II-Richtlinie[13] und die Initiative zur Bewirtschaftung von Abfällen aus der Rohstoffindustrie die Durchführung einer Risikoeinschätzung in naher Zukunft zu einem rechtlichen Gebot für manche oder alle Bewirtschaftungsanlagen für Aufbereitungsrückstände und taubes Gestein.

Zur Gesamtrisikoeinschätzung gehört die Prüfung der einzelnen Betriebsgefahren in enger Zusammenschau mit der Charakteristik der Aufbereitungsrückstände und des tauben Gesteins, den physikalisch-chemischen Eigenschaften sowie weiteren wichtigen Aspekten, z. B. Beschaffenheit des Erzes und der Lagerstätte. Auf dieser Grundlage können die kosteneffektivsten Methoden zur Minderung des Gefährdungspotenzials auf ein vertretbares Niveau entsprechend den bestehenden Bedingungen festgelegt werden. Wie in Kapitel 4.2.3.1 ausgeführt, erfolgt in manchen Fällen die Einstufung von TMF oder WRMF, z. B. nach den Folgen eines möglichen Dammbruchs.

Zur Risikoeinschätzung gehört nicht nur die Bestimmung der 'Risikoquellen', sondern auch eine Bewertung der Wahrscheinlichkeit eines möglichen Ausfalls sowie des Schweregrades der möglichen Folgen eines solchen Ausfalls. Damit ist klar, dass die Risikoeinschätzung die Grundlage für die Erarbeitung einer Risikoeinschätzungsstrategie und aller entsprechenden Maßnahmepläne und Verfahren (einschließlich Kommunikation, Haftungsverhältnisse, Abmilderung und Notfallmaßnahmen) bildet.

Die Risikoeinschätzung (und das Risikomanagement) erfassen alle Phasen des Lebenszyklus der TMF/WRMF. Jedoch wird - je nach Zielstellung, Komplexität des betreffenden Problems und Umfang der vorliegenden Daten - nicht jede Phase mit der gleichen Intensität analysiert.

Generell enthalten Risikoeinschätzungen die folgenden Aspekte:

<u>Umfang und Zweck der Einschätzung</u>

In dieser Phase werden alle an der Risikoeinschätzung beteiligten Partner identifiziert.

<u>Risikoeinschätzungsteam</u>

Ein erfahrenes, interdisziplinäres Risikoeinschätzungsteam bestimmt die potenziellen Ausfallweisen, Wahrscheinlichkeiten und Konsequenzen von Ausfallereignissen. In der Regel gehören zu diesem Team TMF/WRMF-Auslegungsspezialisten, der Bauauftragnehmer, Betreiber, Umwelt- und Managementvertreter sowie bei Detaileinschätzungen eine Fachkraft für Risikoeinschätzung. Die ökologische Einschätzung wird von Umweltmitarbeitern und Spezialisten, in bestimmten Fällen unter Einbeziehung von Fachkräften aus dem Gesundheitswesen und Kosteningenieuren, erstellt. Die Einbeziehung von Betriebsmitarbeitern ist entscheidend für eine Risikoeinschätzung bestehender Anlagen für Aufbereitungsrückstände/taubes Gestein. Auf ihr Wissen und ihre Erfahrungen mit der Anlage kann nicht verzichtet werden.

[13] Richtlinie des Rates 96/82/EG vom 09. 12. 1996 zur Beherrschung der Gefahren schwerer Unfälle mit gefährlichen Stoffen

Bewertungskriterien
Es müssen Kriterien als Anleitung für die Bewertung der Erkenntnisse und der Festlegung von Grenzen tolerierbarer und nicht tolerierbarer Risiken festgelegt werden. Hohe Wahrscheinlichkeit, einschließlich der von hohen Folgeausfällen, ist ein wichtiges Kriterium. Aber auch geringe Wahrscheinlichkeit in Verbindung mit der Wahrscheinlichkeit hoher Folgeausfälle muss u. U. in die Betrachtung einbezogen werden. Potenzielle Auswirkungen auf die Gesundheit und Sicherheit von Menschen, Umweltfolgen oder Auswirkungen auf die Betriebsführung (z. B. Abschaltungen, Leumund, Sachschäden) sind zu berücksichtigen.

Methodologie
Die Risikoeinschätzung kann nach qualitativen (subjektive Einschätzung der Wahrscheinlichkeit, Folgen und Gesamtrisiko) oder quantitativen Faktoren (numerische Angabe von Wahrscheinlichkeit und Folgekosten) erfolgen. Eine einfache qualitative Einschätzung ist für die Bewertung einer Reihe von potenziellen TMF/WRMF-Standorten ausreichend. Für umfangreiche technische Modernisierungsprojekte bestehender Anlagen sollte einer eingehenden quantitative Bewertung der Vorzug gegeben werden.

Zu den häufig angewendeten Methoden der Risikoeinschätzung gehören:
- Prozess-/Systemchecklisten
- SystemAuslegungsmodelle
- Sicherheitsprüfungen
- relatives Ranking
- vorläufige Gefahrenanalysen
- "Was-wäre-wenn"-Analysen
- HAZOP-Studien
- Ausfallszenarien, Auzswirkungen (und Kritikalitäts-Analysen - FMEA, FMECA)
- probabilistische Simulationsanalysen
- Fehlerbaumanalysen
- Ereignisbaumanalysen
- Ursache-Folge-Analysen und Humanfehleranalysen.

Potenzielle Auslöser und Ausfallszenarien
- Überströmung von Dämmen durch:
  - Erdrutsch in das Becken mit Entstehung einer den Damm überspülenden Welle
  - Überströmung des Dammes durch Wellenbildung
  - Ausfall des Ableitungssystems und Eintritt von Wasser in das Becken, Überlastung von Überlaufkanal bzw. Aufnahmekapazität oder Ausfall einer externen Strömungsableitung und Eintritt von Wasser in das Becken
  - Füllstand in Absetzbecken erreicht die Dammkrone
  - Abfluss vom oberen Überlauf des Absetzbeckens zur Reduzierung der Inanspruchnahme von Dammhöhe
  - Blockierung der Duchlassbauwerke
  - Niederschlag übersteigt Aufnahmefähigkeit
  - fehlende Beachtung der Wasserbilanz

- Fehlende Dammstabilität (aufstromig bzw. abstromig) durch:
  - Sickerung führt zu innerer Erosion und Abtrag von Dammmaterial (d. h. Filterausfall)
  - Sickerung führt zu Erhöhung des Porendrucks mit nachfolgender flacher Instabilität
  - nichtseismische Verflüssigung des Damms infolge Belastung oder erhöhtem Porendruck
  - seismische Tätigkeit mit:
    - Verflüssigung von Dämmen
    - Verflüssigung von Aufbereitungsrückständen mit nachfolgender Erosion
    - Verflüssigung von Aufbereitungsrückständen und dadurch Angriff horizontaler Druckkräfte am Damm
    - Deformierung von Dämmen

- Ausfall der Sickerwege führt zu erhöhtem Porendruck mit nachfolgendem Erdrutsch
- Erhöhung des Konstruktionsporendrucks mit nachfolgender Bewegung der Dammschräge
- Erschöpfung der Aufnahmekapazität an unverdichtetem Material durch Ersteinbringung oder Einschluss von Regen und Schneeschmelzwasser im Damm, Dammsetzungen, Überströmung
- Erosion der Dammfläche durch unkontrollierten Niederschlag oder Schneeschmelze

- Instabilität der Dammgründung:
  - Karsteinbruch unter Damm/Halde
  - Einbruch durch Senkungsbewegungen der Grube, wodurch die Aufbereitungsrückstände in die Grube oder Hohlräume abfließen
  - Gleitbewegungen an gering druckhaftem Erdreich oder der Grenzfläche der Abdichtung
  - Verdichtung von gering druckhaftem Erdreich mit Rissbildung im Damm
  - Anstieg des Konstruktionsporendrucks mit Bewegung der Gründung
  - Sickerung in das Grundwassser durch schlecht ausgeführte Membran oder durchlässiges Gebirge, Vermeidung der vorgesehenen Sickerungswege
  - siesmische Verflüssigung der Gründung; seismische Verformung der Gründung; nichtseismische Verflüssigung der Gründung

- Konstruktive Probleme:
  - innere Erosion im Bereich von Durchlassbauwerken oder Abschlämmrohr, Mönchausfall
  - Pumpenausfall durch Stromausfall
  - Ausfall von Rohrleitung oder Kanal
  - Blockierung des Überlaufkanals durch Erdrutsch
  - Blockierung des Überlaufkanals durch Eis

- Ausfall der Spannungsversorgung.

Ausfallswahrscheinlichkeit

Die Ausfallwahrscheinlichkeit der einzelnen potenziellen Ausfallszenarien beruht auf den mit der Anlage vorliegenden Erfahrungen, Erfahrungen mit vergleichbaren Anlagen, technologischen Analysen und professioneller Einschätzung.

Ausfallfolgen

Die Ausfallfolgen bei den einzelnen Ausfallszenarien werden eingeschätzt, einschließlich Auswirkungen auf Gesundheit und Sicherheit der Beschäftigten, von Auftragnehmern und der Bevölkerung, Umweltfolgen, einschließlich Betrachtung der Assimilationskapazität und der Umweltsensitivität des Standortes sowie die Folgen für den Geschäftsbetrieb.

Bericht

Die Ergebnisse der Risikoeinschätzung werden in einer für die Beschäftigten und das Management verständlichen und aussagekräftigen Form dargestellt und zusammengefasst. Wichtig ist die Darstellung in einer Weise, dass sie von allen Beteiligten umfassend verstanden wird.

Risikoeinschätzung

Im Ergebnis der Risikoeinschätzung entsteht eine Liste der identifizierten und eingeschätzten Risiken. Der Risikoeinschätzung schließt sich die Planung der die Risiken mindernden Maßnahmen an. Prinzipiell kann man Risiken auf zweierlei Art und Weise begegnen: (1) mit Maßnahmen zur Reduzierung der Wahrscheinlichkeit des Ausfalls oder (2) Maßnahmen zur Milderung der Auswirkungen eines potenziellen Ausfalls. Die möglichen Maßnahmen zur Risikominderung werden eingeschätzt und ein Plan mit Terminen und Zuständigkeiten erstellt. Eine wichtige Komponente bei der Minimierung der Folgen von Ausfällen ist die Erstellung eines Einsatzbereitschaftsplan für Katastrophenfälle.

Einsatzbereitschaftsplan für Katastrophenfälle

Es ist übliche Praxis, auf Katastrophenfälle vorbereitet zu sein und entsprechende Notfall- und Einsatzbereitschaftspläne für derartige Fälle vorzubereiten. Zur Einsatzbereitschaft gehört die Vorbereitung auf Ereignisse am Standort ebenso wie auf Ereignisse, von denen der Sandort indirekt betroffen ist, z. B. Dammbrüche. Notfall- und Einsatzbereitschaftspläne für Katastrophenfälle sind regelmäßig zu prüfen, zu üben und im Unternehmen sowie an alle potenziell betroffenen externen Parteien zu verteilen.

Der Einsatzbereitschaftsplan für Katastrophenfälle des Standortes bezieht in der Regel die Belange der Anlage für Aufbereitungsrückstände in den allgemeinen Standorteinsatzbereitschaftsplan ein. Er hat u. a. folgenden Inhalt:

- Planungskoordinator, Team- und Gesellschaftsstruktur
- Aufbau der Notfallorganisation, Aufgaben und Zuständigkeiten
- Rechtsgrundlagen, Normen, Alarmierungs- und Berichtspflichten
- verfügbare Ressourcen
- gegenseitige Hilfevereinbarungen
- Plan der Öffentlichkeitsarbeit
- Rufnummern
- Festlegung eines Kommunikationssystems für Mitteilungen und Folgekontakte
- Risikoanalyse für Auswirkungen am Standort und außerhalb desselben
- Karten und Tabellen physischer und Umwelt relevanter Freisetzungen (einschließlich Anlagenausfall)
- Grundlagen für die Aktivierung des Notfallplanes und Regelung der Entscheidungsbefugnis
- Ausbildung der Einsatzkräfte
- Untersuchung und Auswertung von Zwischenfällen und Unfällen
- Wiederherstellung der Betriebssicherheit.

Für Einrichtungen, für die Artikel 9 der Seveso-II-Richtlinie[14] gilt, d. h. die zur Erstellung eines Sicherheitsberichts verpflichtet sind, ist der Betreiber ebenfalls verpflichtet, einen internen Notfallplan mit Maßnahmen zur Verhütung schwerer Unfälle innerhalb des Betriebes zu erstellen.

Nach der Richtlinie sind Notfallpläne mit folgenden Zielstellungen zu erarbeiten:

- Begrenzung und Beherrschung von Unfällen zur Minimierung ihrer Folgen und der Begrenzung der Schäden für Mensch, Umwelt und Einrichtungen
- Durchführung der erforderlichen Maßnahmen für den Schutz von Mensch und Umwelt vor den Folgen schwerer Unfälle
- Weitergabe der notwendigen Informationen an die Öffentlichkeit und die im Bereich betroffenen Dienste und Behörden
- Vorbereitung für die Beseitigung von Schäden und die Wiederherstellung der Umwelt nach schweren Unfällen.

Notfallpläne müssen die Angaben nach Anhang IV der Seveso-II-Richtlinie enthalten.

Die Publikation “APELL for Mining” des Umweltprogramms der Vereinten Nationen (UNEP) enthält weitere Informationen zur Katatsrophenbereitschaft (http://www.uneptie.org/pc/apell/publications/publication_pages/mining.html).

Zwei wichtige Forderungen der Seveso-II-Richtlinie gelten der Erstellung von internen und externen Notfallplänen und der Information der Öffentlichkeit. Notfallpläne sind Bereitschaftsmaßnahmen mit dem Ziel der Beherrschung und Begrenzung von Unfällen sowie zur Minderung ihrer Folgen, zur Begrenzung des Schadens für Beschäftigte des Betriebes und die Bevölkerung außerhalb der Einrichtung sowie zur Begrenzung des Schadens für

[14] Richtlinie 96/82/EG des Rates vom 09. 12. 1996 zur beherrschung der Gefahren von schweren Unfällen mit gefährlichen Stoffen, Amtsblatt L 10 vom 14. 01. 1997, S. 13-33

Einrichtungen und die Umwelt. Die Unterrichtung der Öffentlichkeit besteht in der aktiven Information über die geplanten erforderlichen Verhaltensmaßregeln bei Unfällen sowie passiver Information, welche die interessierte Öffentlichkeit auf Anfrage vom Betreiber der Anlage und/oder den öffentlichen Behörden erhalten kann. Trotz der Unterschiedlichkeit von Seveso II und APELL und der verschiedenen Ansätze, ergänzen beide einander. APELL ist eine Anleitung zur praktischen Umsetzung bestimmter Kernforderungen der Seveso-II-Richtlinie.
[135, Wettig, 2003]

Ablagerungsplan
Es wird ein Ablagerungsplan für Aufbereitungsrückstände/taubes Gestein für die vorgesehene Nutzungsdauer der Grube erstellt. In den Ablagerungsplänen können periodische Erhöhungen der TMF/WRMF vorgesehen sein, um den langfristigen Ablagerungsbedarf von Aufbereitungsrückständen und/oder taubem Gestein zu sichern, eine ausreichende Speicherkapazität für flüssige Stoffe vorzuhalten und für die vorschriftsmäßige Behandlung des freien Wassers im Betrieb der Grube zu sorgen.

Im Plan ist bereits eine mögliche spätere Veränderung der Forderungen und/oder Aufstockung der Kapazität zu berücksichtigen. Zur Erstellung des Ablagerungsplans müssen Informationen über Menge und Dichte der Aufbereitungsrückstände, Wassergehalt und Produktionsangaben auf der Grundlage von Prozess/Aufbereitungsanlagen und die Wasserbilanz vorliegen. Ebenso sind Vorkehrungen für Unwägbarkeiten und Notfälle zu treffen. Die Hauptvariablen sind zu prüfen und periodisch oder anderweitig regelmäßig zu aktualisieren.

Von gleicher Wichtigkeit sind Baubeschreibungen und die Aufnahme der Daten der errichteten und erweiterten Anlage, die in regelmäßigen Anständen geodätisch zu vermessen ist.

Wasserbilanz und Wasserbewirtschaftungsplan
Die Betrachtung des Wasserproblem erfolgt im Zusammenhang mit der Grube. Auf diese Weise entsteht eine integrierte Wasserbewirtschaftung. Im Rahmen des Wasserbewirtschaftungsplanes werden standortbezogen Normen, Ziele, Betriebs- und Notfallpläne und Methoden wie folgt entwickelt:

- Rechtsvorschriften
- Risikoeinschätzung
- Überwachung der hydrologischen Prozesse
- Betriebsüberwachung
- Notfallüberwachung
- Wasserversorgung
- Bodenerosion
- Wasserqualität
- Rechnermodelle
- Leistungskennziffern und
- Training und Forschung.

[97, Umwelt Australia, 2002]

Hydrologie
Hydrologiedaten, einschließlich Darstellung des Einzugsbereichs bzw. der Einzugsbereiche des Standortes der Aufbereitungsrückstände und aller potenziellen natürlichen und prozessbezogenen Wasserquellen, dienen der Erstellung einer Wasser-/Kontaminationsbilanz und dem Auslegung der Komponenten der Anlage für Aufbereitungsrückstände. Im ersten Schritt erfolgen Festlegung und Dokumentierung der Auslegungsparameter, danach die Überwachung der realen Erfahrung zur Erkennung von Abweichungen, Kontrolle von Projektionen und Erkennung potenzieller Probleme.

Bemessungswasserstand
Entsprechend den aktuellen Auslegungsnormen und in Abstimmung mit den Regulierungsbehörden erfolgt die Bemessung des angenommenen höchsten Wasserstands. Dieses Problem ist konsequent durch alle Phasen der Nutzungsdauer der Anlage führen. Speicherbedarf sowie Auslegung von Betrieb und Überlaufkanal müssen an der Hydrologie des Enzugsgebietes ausgerichtet sein.

Wasserbilanz
Es ist eine Wasserbilanzstudie durchzuführen. Der Bedarf der ständigen Datenerfassung für den Abgleich zwischen Mineralienaufbereitungsanlage und TMF-Wasserbilanz ist zu definieren.

Bewirtschaftungsplan für Oberflächenwasser/Grundwasser
Der Wasserbewirtschaftungsplan mit entsprechenden Entwürfen und Konzepten muss bei Bedarf folgende Aspekte berücksichtigen:

- Sickerwassererfassung
- Ausspeicher- und Rückpumpanlagen
- Behandlungs-/Austraganlagen, einschließlich Wassertransportanlagen
- Wasserrückhalte- und Austragkonzept, einschließlich Betriebsparameter.

Emissionsbilanz und Freisetzung
Die Emissionsbilanz enthält Angaben zu den voraussichtlichen Emissionen in Boden, Luft und Grundwasser. Es wird ein Plan für die Minimierung der Emissionen erarbeitet.

Schmutzwasserkriterien
Festlegung von Schmutzwasserkriterien für die TMF/WRMF unter Berücksichtigung der gesetzlichen Vorgaben und Betriebsgenehmigungen und Erlaubnisse mit folgendem Inhalt:

- gelöste Substanzen und Trübstoffe
- Schwebstoffe
- Qualität des Schmutzwassers
- Austragzeiten
- bakterielle und biologische Belastung
- Toxizität.

[18, Mining Association of Canada, 1998]

Außerbetriebnahme- und Stillegungsplan
Stilllegungspläne und Leistungskennziffern werden bereits in der frühen Phase de Anlagenauslegung erstellt und danach regelmäßig und kontinuierlich für die gesamte Nutzungsdauer der Anlage in Vorbereitung der endültigen Außerbetriebnahme und Stillegung kontrolliert und aktualisiert. Für die Stilllegung gelten in der Regel Vorschriften. Die nachstehende Auflistung enthält einige allgemeine Aspekte, die bei der Erarbeitung von Stilllegungsplänen zu berücksichtigen sind. Unter Umständen schließt sich an die Stilllegung eine langfristige Nachsorge an. Für diese sind der Stillegung vergleichbare Pläne und Kontrollen zu erarbeiten.

Elementе eines Stilllegungsplans

- Bestimmung der Hintergrunddaten, z. B.:
  - Standorthistorie
  - Infrastruktur
  - Prozessablaufkontrollen
  - Systembetrieb
  - Mineralogie
  - Topographie

- Hydrologie/Wasserhaushalt
- Hydrogeologie
- Bodenpotenzial
- Wiederurbarmachung
- Folgenabschätzung
- langfristige Wartung
- Geotechnik
- Chemie und Geochemie
- Überwachungsprogramm
- Schmutzwasserbewirtschaftung bzw. Behandlungsbedarf, wo zutreffend
- Kommunkation
- Finanzierung
- Abstimmung mit Interessengruppen
- potenzielle Flächennachnutzung, Stilllegungstechnologie (z. B. Trocken- oder Nassabdeckung, Flutung, Feuchtgebiet, ständige Behandlung, Vegatationsdecke).

<u>Aspekte der Stabilität von TMF/WRMF zur Beachtung bei Stilllegungspläne:</u>
Stilllegungspläne erfordern eine gründliche Neubewertung der Anlage und ihrer Stabilität unter Stilllegungsbedingungen. Alle Aspekte der Anlage und der physikalisch-chemischen Stabilität müssen beachtet werden. Besonders die Ist-Leistung der Anlage im Betrieb, einschließlich:

- Deformierung
- Sickerverhalten
- Gründungen und Seitenwälle

sind im Vergleich zu den Auslegungsprojektionen sowie den projizierten Bedingungen nach erfolgter Stilllegung zu prüfen. Auslegunglasten können sich nach Außerbetriebnahme und Stilllegung ändern.

Die Überwachung und Inspektion aller Bauwerke werden bis zur Außerbetriebnahme kontinuierlich fortgesetzt sowie bei Bedarf auch danach. Der bestehende Überwachungs- und/oder Inspektionsbedarf der verbleibenden Bauwerke ist festzulegen.

Es werden Maßnahmepläne für Mängel bei der Qualität der Stilllegung und/oder Problemen bei der Einhaltung der Stilllegungsspezifikation erarbeitet. Die Folgen der Stilllegung von Einrichtungen für Notfall- und Katastrophenbereitschaft sowie bei Bedarf die Aktualisierung dieser Pläne sollten ebenfalls einbezogen werden. Die ständige Verfügbarkeit der Auslegungs-, Bau- und Betriebsunterlagen für nach der Stilllegung verbleibende Bauten ist zu sichern.

4.2.1.4 Auslegung der TMF/WRMF und der damit verbundenen Bauten

Die folgende Aufzählung trifft unter Umständen nicht auf alle Standorte oder Gegebenheiten zu. Daher obliegt die Entscheidung, welche Aspekte zu berücksichtigen sind, dem Betreiber und der Genehmigungsbehörde. Standortspezifische Bedingungen können die Einbeziehung anderer oder zusätzlicher Kriterien erfordern. Kriterien für die Betriebsphase sowie die Nachstilllegungsphase werden betrachtet. Davon abweichende Kriterien im Ergebnis der Auslegungswerte können für die Betriebs- und die Langzeitphase gelten.

Die Angaben zu TMF/WRMF-Standorten wurden der Literatur sowie Feld- und Laboruntersuchungsprogrammen entnommen.

<u>Hydrologie und Hydrogeologie</u>

- hydrologische und hydrogeologische Untersuchungen
- Wasserbilanz, Wasserqualität
- Bemessungshochwasser
- Freibord
- Wasserabdeckung

- Oberflächenabfluss und Ableitungsmaßnahmen im Einzugsgebiet
- Ablagerungsplan
- Erosionsbewirtschaftungsplan.

Gründungen, Geologie und Geotechnik
- Geomorphologie
- Regional- und Lokalgeologie, Verwerfungen
- Stratigraphie
- Muttergestein und Bodenbeschaffenheit
- Geotechnische Angaben, z. B.:
  - Komprimierbarkeit
  - Scherfestigkeit
  - Reibungswinkel
  - Korngrößenstruktur
  - Dichte
  - Plastizität
  - Brüche
  - Verflüssigungspotenzial
  - Duchlässigkeit
  - Erosionspotenzial
  - hydraulische Rissbildung.

Baustoffe
Die Verfügbarkeit natürlich vorkommender Baustoffe ist einzuschätzen, ebenso die technisch-technologische Beschaffenheit dieser potenziellen Baustoffe, Aufbereitungsrückstände, Verpress-/Beton- und anderer potenzieller (natürlicher und künstlicher) Dichtungsstoffe, d. h. zu:
- Korngrößenstruktur
- Dichte
- Volumen
- Scherfestigkeit
- Duchlässigkeit
- Säurebildungspotenzial
- chemische Reaktivität (Säurebildungspotenzial, Reaktion mit Wasser im Absetzbecken, Thiosalzbildungspotenzial)
- Wind- und Wassererosionspotenzial.

Potenziell schädliche Auswirkungen von Aufbereitungsrückständen und/oder Prozesswasser auf Baustoffe werden bestimmt. Umwelteinflüsse, Stabilität und Sanierungsbedarf von Baustoffen werden ebvenfalls in dieser Phase einbezogen.

Topographie
Dies betrifft regionale und topographische Kartierung und Luftbildaufnahmen.

Spezielle Umweltfragen
Seismische Gefahren, seismische Dämpfung von Gründungsschichten und Baustoffen, Verflüssigungspotenzial von Gründungsshichten und Baustoffen, klimatische Bedingungen sind bei der Einschätzung zu berücksichtigen, einschließlich:
- zu erwartende Extremwerte
- Einwirkung von Wind und Wellen
- Auswirkungen von Dauerforst
- Frost.

Sickerverhalten
Die unter Umwelt- und baustatischen Gesichtspunkten maximal zulässigen Sickermengen werden bestimmt. Die Forderungen für durchlässige im Vergleich zu nicht durchlässigen Stoffen und Baumethoden werden festgelegt und ein Sickerbewirtschaftsungsplan erstellt.

Stilllegung
Die Wahl bzw. voraussichtliche Wahl des Stilllegungsverfahrens einer TMF/WRMF kann Einfluss auf die Auslegung haben und ist daher in der Auslegungsphase zu beachten.

Erforderliche Auslegungsparameter
- Anlagenklassifikation (wenn nach geltendem Recht vorgesehen)
- Stabilität
- Erdbebenkriterien
- Sicherheitfaktoren
- Auslegungdurchlässigkeiten
- Sauerwasser
- Wild
- Staubentwicklung
- Stilllegungsaspekte.

Diese Parameter werden in den folgenden Abschnitten näher dargestellt.

Stabilität
Die Stabilität von Gründung, Anlage und sonstigen Bauten unter den Bedingungen von Bau, Betrieb und Stilllegung, unter statischen und dynamischen Bedingungen, einschließlich Einwirkungen von Wellen, Frost/Eis und großer Abflussmengen (bei Absetzbecken) sind zu analysieren. Solldichte und Verdichtung festzulegen.

Vorbereitung der Gründung
Die Bedingungen für die Vorbereitung der TMF/WRMF-Gründung sind vor der Bauausführung festzulegen, einschließlich:
- Abtragen der Vegetation, einschließlich kommerzieller Holzeinschlag
- Aushub organischer Böden
- Schlitzwände
- Beherrschung und Rückhaltung von Grundwasser
- Reinigung und Schlämmebehandlung von Muttergestein
- Hochdruckvermörtelung
- Ableitungsbrunnen
- Ableitungskanäle
- Entwässerung
- Stabilität
- Möglichkeit der Baudurchführung
- sonstige spezifische Bedingungen für die Baudurchführung.

Sickerwasseranalyse und Bewirtschaftung
Die Bedingungen für die Beherrschung des Sickerwassers, einschließlich Eintritt in das Grundwasser, werden eingeschätzt, ebenso die Chemie und das Säurebildungspotenzial des Wassers. Die Durchführung entsprechender Maßnahmen wird geplant, z. B.:
- Filterauslegung
- Schlitzwände
- Zementmörtelabdichtung
- Grabenbau
- geringe Kernduchlässigkeit
- Auffangbrunnen.

Verbundene Bauten
Die folgenden Optionen sind bei Bedarf auszulegen:

- Überlaufkanäle
- Türme
- Rohrleitungen (z. B. Rückschlagventile, Sekundärrückhalte)
- Bedingungen bei Wasserhöchstständen
- Klappen und Ventile
- Syphone
- Pumpen
- Vorkehrungen für natürliche Gefahren (z. B. Schmutz, Biber, Kaninchen, Eissperre).

TMF/WRMF-Auslegung

- Anlagenart (z. B. Halde, Damm (Art des Damms))
- Auslegungsphilosophie
- Kriterien für Hauptelemente.

TMF/WRMF-Bauplan
Es wird ein Plan für die Durchführung des Baus der TMF/WRMF und späterer Erhöhungen, einschließlich Abfolge und Bedingungen für Stabilitätsüberwachung, erstellt. Es werden eine Baumethodologie, Terminplan und Kostenplan entwickelt. Potenzielle Umwelteinflüsse durch den Bau entsprechend der vorgesehenen Auslegung werden bestimmt.

TMF/WRMF-Überwachungssysteme

- Piezometer
- Inklinationsmesser
- Setzungsmesser
- Überwachung des Sickerwasserstroms
- Temperatur (Dauerfrost, Frosteindringtiefe, Heizung)
- Überwachungsmethoden.

Fehlerzustandsanalyse
Potenzielle Fehlerzustände der TMF/WRMF werden analysiert: in der Bauphase, während des Betriebs, im Endzustand sowie nach der Stilllegung.

4.2.1.5 Kontrolle und Überwachung

Es muss ein umfassender Kontrol- und Überwachungsplan erstellt werden, der den gesamten Lebenszyklus des Standortes hinsichtlich Kontrolle der Emissionen und deren Auswirkungen sowie entsprechende Überwachung vorsieht.

Plan der Qualitätssicherung /Qualitätskontrolle (QS/QK)
Es ist gute Praxis, in den Phasen von Bau, Betrieb und Stilllegung folgende Unterlagen zu führen und aufzubewahren:

- Bauzeichnungen und Bausbestandsunterlagen, einschließlich Änderungen
- Prüfergebnisse
- Besprechnungsprotokolle
- Baufotos
- Überwachungsaufzeichnungen.

Baukontrolle
Typische Elemente eines Baumanagementsystems beinhalten:

- Planung und Terminierung
- Vermessungskontrolle (Layout, Bestandsunterlagen)
- Überwachung der Vermörtelung

- Überwachung der Gründungsvorbereitung
- Qualitätskontrollen der Baustoffe
- Kontrolle der Verdichtung
- gerätetechnische Überwachung und Datensynthese
- Führung von Unterlagen
- Bausicherheit
- Bauumweltkriterien.

Vermeidung von Staubfreisetzung

Die Freisetzung von Staub aus der Anlage für Aufbereitungsrückstände ist so gering wie möglich zu halten. Eine Möglichkeit ist das Befeuchten der Aufbereitungsrückstände und/oder der Einsatz kurz- oder langfristiger chemischer oder organischer Abdeckungen.

Kontrolle von Anlagen zur Bewirtschaftung von Aufbereitungsrückständen

- häufige Leistungsüberwachung - Sichtprüfung
- Grundwasserdruck (Porenwasserdruck)
- Sickerwasser
- Verformung (Setzverhalten und Stabilität)
- Witterungseinflüsse
- seismische Ereignisse (nachträglich)
- spezielle Kontrollprogramme nach schweren Ereignissen (Erdbeben, Sturn, Frostaufbruch, Überschwemmung)

- Anzeichen für Instabilität:
  - ‘weiche Zonen’ und Bodenfließen am Dammfuß
  - Schmutzablagerungen im Sickerwasser
  - erhöhtes Sickerwasseraufkommen
  - neue Sickerbereiche
  - Längs- und Querrisse
  - Setzung.

- Bereiche, die besondere Aufmerksamkeit verlangen:
  - Überlaufkanäle
  - Abflussbauwerke
  - Entwässerungs- und Entlastungsbrunnen
  - Betonbauten
  - Rohre und Leitungen durch Dämme
  - Steinschüttungen
  - Syphone
  - Wehre
  - Bäume und Tierbaue.

Überwachungspläne für Stabilitätsprogramme

- Standort von Kontrollstationen
- Zeitplanung (Kontrollzeiträume, Inspektionen)
- Art der Überwachung (Sichtkontrolle, Maßnahmen und Parameter)
- erforderliche gerätetechnische Ausstattung (z. B. Piezometer) mit Festlegung des Verwendungszwecks
- Prüfverfahren, Datenerfassung und Auswertung
- für die Überwachung verantwortliche Personen
- Datenspeicherung und Berichtswesen
- Bewertungskriterien für Überwachungsprogramme.

Wasserqualitätsplan

- Hydrologie:
  - schwere Sturm- und Dürreereignisse

  - benötigte Informationen und Parameter für die Wasserbewirtschaftung
  - Kriterien für die Beherrschung der Wasserstände innerhalb sicherer Grenzen, einschließlich erforderlicher täglicher oder jahreszeitlich bedingter Wasserstandskontrollen

- Wasserbeherrschung mit folgenden Zielstellungen:
  - Sicherheit der Wasserbewirtschaftung innerhalb der Systemgrenzen
  - Verhinderung/Beherrschung/Instandsetzung von Schäden an allen Bauwerken
  - Durchführung von erforderlichen Kontrollen und Prüfungen nach Änderungen von Auslegung oder Methoden während oder nach der Bauphase, bei höherem Wasserstand im Absetzbecken als kritisch zulässig sowie nach schweren Stürmen und der Frühjahrsschmelze.

- Randsickerwasser
  - Einschätzung des Sickerpotenzials im Bereich der Aufbereitungsrückstände
  - Festlegung von Umfang und Kenndaten tolerierbarer Sickerung
  - Erarbeitung von Maßnahmeplänen bei Abweichungen gegenüber der Auslegungssickerung
  - Messung der Leistung, einschließlich Sickerkontrolle innerhalb der Auslegungsgrenzwerte
  - Überwachung und Kontrollen zur Sicherung der Leistung der Systeme innerhalb der Auslegungsgrenzwerte.

<u>Ablagerungsplan für Aufbereitungsrückstände</u>
Damit wird die effektive Nutzung der Kapazität für Aufbereitungsrückstände und die effektive Stilllegung der Anlage gesichert. Lang- und Kurzzeitpläne für TMF/WRMF-Erhöhungen sind ebenfalls im Plan erfasst. In festgelegten Intervallen erfolgt die Kontrolle des Ablagerungsplans für die Aufbereitungsrückstände und einer Füllkurve (Kurve Volumen/Höhe) als Vergleich von Soll- und Istzustand.

4.2.2 Bauphase

Für bestimmte Anlagen für Aufbereitungsrückstände und Taubgestein ist die Differenzierung zwischen Bau- und Betriebsphase nicht eindeutig, da die Bautätigkeit in vielen Fällen während des Betriebs weitergeführt wird oder wieder aufgenommen wird (z. B. Erhöhung des Damms). Der Bau der Anlage ist umfassend zu dokumentieren und folgt dem in der Auslegungsphase erstellten Bauplan. Die Bestandsdokumentation beinhaltet alle Veränderungen gegenüber dem Bauplan.

Während des Baus der Anlage und für die Zukunft:
- werden Bestandsunterlagen und aktuelle Verfahrensdokumente geführt, in denen alle Abweichungen gegenüber der Originalauslegung sowie bei Bedarf eine Kontrolle der Auslegungskriterien dokumentiert sind,
- wird der Bau von einem unabhängigen qualifizierten Technologie-/Geotechnikspezialisten überwacht,
- werden Unterlagen mit den Ergebnissen aller durchgeführten Prüfungen (z. B. Verdichtung) für den Bau und während desselben vorschriftsmäßig geführt.

[45, Euromines, 2002]

4.2.3 Betriebsphase

Als die beiden Haptursachen für Unfälle bei TMF wurden festgestellt:

- Mängel der Wasserbewirtschaftung
- fehlende Kenntnis der für die Betriebssicherheit wichtigen Aspekte.

[9, ICOLD, 2001, p. 6]

Damit ist klar, dass ein erfolgreiches Betriebsmanagement die wichtigste Voraussetzung für den sicheren Betrieb einer TMF/WRMF ist.

Die Geotechnologie hat die notwendigen Fortschritte für die Auslegung sicherer und zuverlässiger Dämme gemacht. Jetzt ist das Management der TMF/WRMF der entscheidende Unterschied zwischen problemlosem Betrieb und potenzieller Katastrophe.

Folgende Maßnahmen dienen in der Regel der Vermeidung von Unfällen:
- Überwachung der phreatischen Grundwasserschicht mit optimal platzierten Piezometergeräten und offenen Standrohren
- Maßnahmen zur Ableitung von Wasser und Aufbereitungsrückständen aus dem Bereich des Absetzbeckens in Problemsituationen
- Schaffung von alternativen Abläufen, möglichst in ein anderes Becken
- Schaffung von Notüberläufen und/oder Reservepumpenpontons für Notfälle
- messtechnische Erfassung der Bewegungen des Baugrundes mit Tiefenneigungsmesser und Kenntnis der Porendruckverhältnisse
- Schaffung ausreichender Abflussverhältnisse
- Führung von Unterlagen über Auslegung und Bau und Dokumentierung aller Aktualisierungen/Änderungen bei Auslegung/Bau
- Qualifikation und Ausbildung des Personals

[9, ICOLD, 2001]
und weiter:
- Sicherung der Kontinuität der Technologie des Damms und
- in bestimmten Fällen unabhängige Prüfung des Damms mit Freigabe durch einen externen Prüfer.

Der Betrieb der Anlage erfolgt entsprechend dem Managementplan für Aufbereitungsrückstände und Taubgestein, den Betriebsanweisungen und dem Überwachungsplan der Anlage. Alle Änderungen dieser Pläne sind zu dokumentieren und zu analysieren. Die Überwachungsdaten sind regelmäßig auszuwerten und bei Bedarf entsprechende Maßnahmen einzuleiten. Interne und externe Kontrollen (Prüfungen) werden in bestimmten Fällen durchgeführt.

Folgende Maßnahmen dienen der Aufrechterhaltung der Betriebssicherheit:
- Der Anfall von Aufbereitungsrückständen und Taubgestein erhält von der Geschäftsführung den gleichen Stellenwert wie die Produktion vermarktungsfähiger Produkte
- Sicherung einer effektiven Betriebskontrolle und Überwachung
- Erfassung von Anfallmengen und Beschaffenheit von Aufbereitungsrückständen und Taubgestein
- klare Festlegung von Aufgaben und Verantwortlichkeiten für die Bewirtschaftung von Aufbereitungsrückständen und taubem Gestein an qualifiziertes Personal
- regelmäßige Inspektion der technischen Einrichtungen durch eine qualifizierte Fachkraft mit Erfahrung in Aufbereitungsrückständen und taubem Gestein und Bestätigung, dass alle signifikanten Risiken beim laufenden Betrieb der Einrichtungen erfasst sind und auseichend beherrscht werden
- Bereitstellung und Einhaltung von Betriebsanweisungen in der Landessprache des Betreibers. Diese Anleitungen enthalten alle Überwachungsaspekte.
- Speicherung und ordnungsgemäße Verwaltung von Betriebsunterlagen, z. B. Wasserstandserhöhungen, abgelagerte Menge, Sickermengen, Wasserverbrauch (evtl. meteorologische Daten) usw.
- unverzügliche Meldung aller in der Auslegung berücksichtigten Betriebsbedingungen außerhalb des Standorts der Anlage an den Konstrukteur oder Prüfung durch einen qualifizierten Techniker
- entsprechendes Training des Betriebspersonals, einschließlich Anfangsfehlerdiagnose

- besondere Aufmerksamkeit für die ständige Arbeit mit dem Wasserbewirtschaftungsplan
- effektive Festlegungen und Einhaltung für Meldung von Störungen
- Erstellung und ständige Anpassung effektiver Notfallpläne.

[45, Euromines, 2002]

4.2.3.1 OSM-Handbücher

Verschiedene Dammbetreiber arbeiten nach Sicherheithandbüchern. Diese sind auch als OSM-Handbücher (Betrieb, Überwachung und Wartung) bekannt [50, Au group, 2002]. Themen von OSM-Handbüchern sind u.a.:

- Organisation der Dammsicherheit
- Einsatzbereitschaftsplan für Katastrophenfälle
- Klassifikation nach den Folgen des Dammversagens
- Dammbau
- Hydrologie
- Umwelt
- Betrieb
- Überwachung
- Genehmigungen
- Berichte.

[50, Au group, 2002]

TMF/WRMF-Sicherheitsorganisation
Die Organisation der Dammsicherheit sieht den Einsaztz eines Dammsicherheitsbeauftragten für jeden Standort vor. Zur Unterstützung dieses Beauftragten kann ferner ein Sicherheitskoordinator benannt werden, der auf TMF/WRMF spezialisiert ist und sich ausschließlich der Sicherheit der TMF/WRMF widmet. Für Betrieb, Überwachung und Wartung setzt der Beauftragte Beschäftigte des eigenen Unternehmens ein, meist die Personen, welche für die Umweltbeprobung und Überwachung der Speicheranlagen für Aufbereitungsrückstände zuständig sind.

Einsatzbereitschaftsplan für Katastrophenfälle (EPP)
Für jede TMF/WRMF besteht bei Unfällen an der Anlage ein EPP. Dieser umfasst Listen der im Havariefall zu benachrichtigenden Personen im Betrieb und bei Behörden. Ferner sind dort Berater und Auftragnehmer genannt, die mit dem Standort vertraut sind und bei Bedarf kurzfristig Unterstützung leisten können. Der EPP beinhaltet ferner Beispiele für zu ergreifende Maßnahmen in verschiedenen denkbaren Situationen. Generell sind Beauftragter und Koordinator in jedem Fall zu konsultieren und in alle wichtigen Entscheidungen und Maßnahmen zum Damm einzubeziehen. Der Beauftragte trifft die letzte Entscheidung über die in einer gegebenen Situation durchzuführenden Aktivitäten.

Risikoeinschätzung der TMF/WRMF
In manchen Fällen erfolgt die Einstufung der TMF/WRMF nach den Folgen eines möglichen Versagens (nicht nach der Versagenswahrscheinlichkeit). In Schweden haben sich die Betreiber von Dämmen für Aufbereitungsrückstände zur Übernahme des RIDAS-Systems von den Betreibern von Wasserstaudämmen entschieden. Nach den möglichen Folgen sieht dieses System vier Klassen vor: 1A, 1B, 2 und 3 entsprechend den folgenden Tabellen. Die Tabelle ist in zwei Klassen geteilt, wobei die Klassifikation der Gefahren für den Menschen getrennt von den Risiken für Objekte, Infrastruktur und Umwelt aufgeführt ist.

Klasse	Folgen
1A	Direkte Gefahren für Leib und Leben
1B	Nicht vernachlässigbare Gefahren für Leib und Leben oder schwere Verletzungen

Tabelle 4.1: Klassifizierung nach Gefahr für Leib und Leben und schwere Verletzungen

Klasse	Folgen
1A	Direkte Gefahren mit: ▪ schweren Schäden an wichtiger Infrastruktur, wichtigen Bauten oder erhebliche Gefährdung der Umwelt, und ▪ schweren wirtschaftliche Schäden (>EUR 10 Mio)
1B	Ergebliche Gefahren mit: ▪ schwerne Schäden an wichtiger Infrastruktur, wichtigen Bauten oder erhebliche Gefährdung der Umwelt, und ▪ schweren wirtschaftlichen Schäden (>EUR 10 Mio)
2	Nicht vernachlässigbare Gefahren mit: ▪ erheblichen Schäden an Infrastruktur, wichtigen Bauten, Gefährung der Umwelt oder von Drittobjekten (<EUR 0,5 Mio)
3	Vernachlässigbare Gefahren mit: ▪ erheblichen Schäden an Infrastruktur, wichtigen Bauten, Gefährung der Umwelt oder von Drittobjekten.

Tabelle 4.2: Klassifizierung nach Schäden an Infrastruktur, Umwelt und Sachobjekten aus: Svensk Energi AB, 2002. RIDAS, Kraftföretagens riktlinjer för Dammmsäkerhet (überarb. 2002). Svensk Energi - Swedenenergy - AB

Diese Klassifizierung bildet die Grundlage für Betrieb und Überwachung. Sie bestimmt die Grenzen für den erforderlichen Freibord und die Kapazität des Überlaufkanals, d. h. den Sicherheitbereich zwischen maximalem Wasserstand und Dammkrone bzw. die maximale Ablaufleistung.

Das schwedische RIDAS-System ist vergleichbar der norwegischen Klassifikation entsprechend der folgenden Tabelle:

Klasse	Folge	betroffene Wohneinheiten
1	geringe Gefahr	0
2	signifikante Gefahr	0 - 20
3	hohe Gefahr	über 20

Tabelle 4.3: Klassifizierung von Dämmen nach norwegischem Recht [116, Nilsson, 2001]

Entsprechende Kartierungen und Standortbesichtigungen dienen als Grundlage für die Bewertung. Klasse 3 und 2 beinhalten Folgen für Wohngebiete und Gefahren für Menschen. Ferner berücksichtigt diese Klassifizierung z. B.:

- potenzielle Schäden an Hauptstraßen und Eisenbahnlinien
- ökonomische und ökologische Schäden.

Damit unterliegt die Klassifizierung der Folgen in bestimmtem Umfang subjektiven Faktoren. Die Klassifizierung und eventuelle Umklassifizierung erfolgt durch die dafür Verantwortlichen und muss von den zuständigen Behörden genehmigt werden.
[116, Nilsson, 2001]

Nach spanischem Recht erfolgt die Einstufung ebenfalls nach Gefahrenkriterien, wie aus der folgenden Tabelle erhellt:

Damm-kategorie	Gefahren für			
	Menschen	**wichtige Dienste**	**Sachschäden**	**Umweltschäden**
A	ernst für mehr als 5 Wohnungen	ernst	sehr ernst	sehr ernst
B	ernst für 1 - 5 Wohnungen	-	ernst	ernst
C	möglicher Verlust von Menschenleben (keine Wohnungen)	-	mäßig	

Tabelle 4.4: Klassifizierung von Dämmen nach spanischem Recht
[116, Nilsson, 2001]

Ein ähnlicher Ansatz ist nach finnischer Rechtslage vorgesehen. Je nach Gefahr erfolgt eine Klassifizierung der Dämme als P, N, O, T, wobei P die Klasse mit der höchsten Wahrscheinlichkeit des Verlustes an Menschenleben, Schäden für Umwelt oder Sachschäden ist.
[117, Forestry, 1997]

Bau von TMF/WRMF
Jede TMF/WRMF ist ausführlich zu beschreiben. Vom Vordamm bis zur aktuellen Höhe sind alle Phasen des Bauablaufs und der verwendeten Baustoffe, Name des Bauausführenden, beim Bau auftretende Probleme, Art des Überlaufkanals, Volumen an abgelagerten Aufbereitungsrückständen / Taubgestein und Wasser usw. zu dokumentieren. Damit sind alle sicherheitsrelvanten Informationen der TMF/WRMF rasch wieder auffindbar.

Hydrologie
Jeder Damm muss mit einem Mindestfreibord, für eine maximale Wellenhöhe und eine Mindeskapazität des Überlaufkanals ausgelegt sein. Das heißt, alle Dämme der Klasse 1A bzw. 1B nach dem RIDAS-System müssen, abgesehen von der Wasserspeichermenge, für eine Kapazität des Überlaufkanals ausgelegt sein, wie sie für einen Jahrhundertsturm benötigt wird. Ferner müssen diese Dämme auch für 'Abfluss Klasse 1' (was etwa einem Jahrzehntausendsturm entspricht) ausgelegt sein, wobei eine ausreichende Wassserspeicherung bis zu einer sicheren Höhe möglich ist. Dämme der Klasse 2 nach dem RIDAS-System werden für einen Jahrhundertsturm ausgelegt. Für Dämme der Klasse 3 besteht keine spezielle Forderung.

Umwelt
Für jede TMF/WRMF und jede Grube wird ein Umweltüberwachungsprogramm mit Festlegungen zu Beprobung, Auswertung und Meldung an die Behörden erstellt.

Betrieb
Der reibungslose Betrieb der TMF/WRMF ist eine wichtige Voraussetzung für hohe Zuverlässigkeit und Sicherheit. Es erfolgen ausführliche, aktuelle Anweisungen für die Betriebsweise der Anlage zur Einhaltung der Auslegungsparameter, Beachtung der Eigenschaften der Aufbereitungsrückstände und Einhaltung des Vorgaben für Prozesswasser und klimatische Bedingungen. Alle in der Anlage und der Einrichtung beschäftigte Personen

müssen mit diesen Anweisungen vertraut sein. Damit ist gründliche Einweisung eine wichtige Forderung.

Überwachung

Kontrolle und fehlerloser Betrieb der Anlage sind zweifellos die wichtigsten Forderungen für eine hohe Dammsicherheit. Zur Kontrolle sind entsprechende Geräte erforderlich, deren Einsatz wiederum Kompetenz der zuständigen Personen bei der Auswertung der Messergebnisse und der Festlegung der richtigen Schlussfolgerungen voraussetzt.

Grundsätzlich finden regelmäßige Kontrollen auf vier unterschiedlichen Ebenen statt. Das Vorgehen ist stufenweise, von der täglichen Inspektion bis zur umfassenden Sicherheitsüberprüfung (Revision) in langen Zeitabständen:

1) Routineinspektionen am Standort
2) Kontrolle
3) Jahres- bzw. Zweijahresinspektion
4) Revisionen.

Standortinspektionen werden bei jeder Anlage in unterschiedlichen Abständen durchgeführt, die von drei Mal täglich bis mehrere Tage pro Woche schwanken. Meist werden die täglich durchgeführten Inspektionen durch Mitarbeiter der Anlage oder die für die Umweltbeprobung zuständigen Personen vorgenommen.

Kontrollen erfolgen monatlich oder mindestens aller drei Monate durch den Beauftragten oder eine dafür benannte Person.

Jahresinspektionen werden durch den Koordiniator oder eine externe Fachkraft durchgeführt. Der Prüfer prüft alle Ereignisse und Maßnahmen seit der letzten Inspektion und erstellt dazu ein Protokoll. Teil der Jahresinspektion ist die Kontrolle des vollständigen OSM-Handbuches.

Umfassende Revisionen finden in der Regel in Abständen von mehreren Jahren statt. Sie umfassen die Prüfung des gesamten vorhandenen Ablagematerials sowie aller Inspektionen, eine Begehung und die Kontrolle des OSM-Handbuches. Darüber wird ein Protokoll erstellt, in dem der Zustand der TMF/WRMF festgestellt wird. Revisionen werden im folgenden Abschnitt ausführlicher dargestellt.

Genehmigungen

Allgemein werden alle für die TMF erteilten Genehmigungen zentral aufbewahrt, wo sie rasch zur Hand sind, wenn die Einhaltung des Betriebs der Anlage entsprechend den erteilten Genehmigungen geprüft wird.

Berichte und Protokolle

Generell werden alle Berichte und Protokolle zur Sicherheit der TMF/WRMF an einem zentralen Ort abgelegt, wo sie bei Bedarf rasch abrufbar sind. Die Aufzeichnungen zu allen Überwachungsübungen sind nach Prioritäten zu ordnen und in Maßnahmepläne umzusetzen.

Weitere Angaben zur Sicherheit von TMF/WRMF

Nach Fertigstellung des Sicherheitshandbuches ist der Umsetzung der OSM-Handbücher vor Ort und der Einweisung des Anlagenpersonals große Aufmerksamkeit zu widmen. Bei einem Beispiel wurden als erster Schritt alle Handbücher vor Ort vorgelegt. Anschließend erfolgte eine vierstündige Einweisung des Personals und weiterer Personen in allen Anlagen mit Dämmen. Daran schloss sich eine drei- bis viertägige Schulung mit theoretischer Einweisung, praktischen Übungen, Durchsprache der bestehenden Bedingungen (vorhandene Einsatzkräfte und Technik), wobei ausreichend Zeit für Diskussionen blieb. Implementierung von OSM-Handbüchern und Weiterbildung des Personals sind laufende Aufgaben im Zusammenhang mit den Jahresinspektionen. Das Inspektionsergebnis wird dem Personal vermittelt, weitere Schulungsmaßnahmen können sich anschließen.
[50, Au group, 2002]

Die Vorteile dieses Dokumentationssystems sind:

- Dokumentation über alle wichtigen Aspekte der TMF/WRMF werden in übersichtlicher Weise erfasst
- Alle Informationen sind jederzeit zugänglich, was die Übergabe bei Wechsel der Zuständigkeit oder des Betreibers erleichtert
- Bei Unfällen ist der direkte Zugriff auf die entsprechenden Informationen gesichert.

Nachteile sind:

- In Ländern mit schwacher Rohstoffindustrie könnte es schwierig sein, geeignete Fachkräfte für die Durchführung der Revision zu finden.
- Für Kleinbetriebe stellen die Kosten der Revision eine Belastung dar.
- Wichtig ist ein ständiger verwaltungstechnischer Prozess und damit Personaleinsatz zur Aktualiserung der Unterlagen.

[118, Zinkgruvan, 2003]

OSM-Handbücher gelten für alle Fälle, in denen die Gefahr erheblicher Schäden an der Infrastruktur, wichtigen Bauten, Gefährdung der Umwelt oder fremder Sachwerte nicht vernachlässigt werden kann und wo sich Klarwasser im Absetzbecken befindet. In manchen Fällen dient eine bstimmte Größe von Absetzbecken oder Halde als Maß für die Unterscheidung zwischen Gefahren, die vernachlässigt werden können oder nicht. Zum Beispiel liegt diese Grenze nach deutschen Gesetzen bei einem Gesamtspeichervolumen von 100000 m^3 und einer Dammhöhe von 5 m.

Es ist nicht möglich, verlässliche Angaben über den für die Erstellung und Pflege der OSM-Handbücher erforderlichen Personal- und Kostenaufwand zu machen. Es kann jedoch festgestellt werden, dass die Kosten etwa in Höhe derer für andere Managementsystem liegen. Zwei Einflussfaktoren bei den Kosten sind der Umfang an bereits in der Auslegungsphase des Standortes erfassten Informationen und die Größe des Betriebs.

4.2.3.2 Revision

Zweck der unabhängigen Revision von TMF/WRMF ist die regelmäßige Beurteilung von Leistung und Sicherheit der Anlage durch qualifizierte und erfahrene Fachkräfte, die nicht bei Auslegung oder Betrieb der Anlage mitgewirkt haben.

Gründe für die Durchführung von Revisionen sind:

1. Wenn sich Ausfälle trotz vorhandener Technologie zum Bau und sicheren Betrieb von Anlagen für Aufbereitungsrückstände und taubem Gestein wiederholen. In diesem Fall sind die Ausfallursachen meist Fehler in der Auslegungsphase oder beim Betrieb der Anlage [9, ICOLD, 2001]. Menschliches Versagen und Baumängel können damit als Faktoren nicht aus der Analyse ausgegliedert werden, was ein Zweitgutachten angebracht erscheinen lässt.
2. Häufig werden bei unabhängigen Revisionen nicht nur menschliche Fehler erkannt, sondern die nicht mit ‘Betriebsblindheit’ geschlagene Fachkraft betrachtet die Anlage aus einem anderen (objektiveren) Winkel, den das Anlagenpersonal mit der Zeit verloren haben kann.
3. Da die bei Auslegung, Bau und an anderen Projekten in der Anlage eingesetzten Fachkräfte stets in bestimmtem Umfang von der Bergbaugesellschaft abhängig sind und ihre Arbeit damit eher der eines betriebsinternen Auftragnehmers oder Beraters der Bergbaugsellschaft gleicht, wird der Auftragnehmer bzw. Berater mit der Zeit ‘einer von ihnen’, was unterschwellig Entscheidungen beeinflussen kann, auch wenn die Absicht zu Objektivität besteht. Aus diesem Grunde werden Revisionen meist von Fachkräften ohne vorherige Verbindung zum jewieligen Standort durchgeführt.
4. Revisionen sind wichtig und müssen regelmäßig durchgeführt werden. Die Zeitabstände zwischen zwei Revisionen schwanken vorwiegend nach der Gefahreneinstufung der Anlage. Andere Einflussfaktoren sind Schütthöhe, Bau und Ablagerungsmethode, Sicherheitorganisation des Damms, Erfahrung der Gesellschaft und Hinweise durch den

firmeninternen Berater. Die die Revision durchführende Person bzw. das Team legt den Termin für die folgende Inspektion in Absprache mit der Bergbaugesellschaft fest.

Revisionen umfassen alle für die Gesamtsicherheit der TMF/WRMF relevanten Aspekte, z. B.:

- aktuelle Auslegung, Auslegung entsprechend Genehmigungen und einschlägigen Normen, Bestands- und Änderungsdokumentation
- bisherige Bau-/Ablagerungsphasen entsprechend der Auslegung
- bisherige Probleme und Unfälle
- künftige/geplante Auslegung entsprechend den einschlägigen Normen
- laufende Bau- und Ablagerungsaktivitäten entsprechend den einschlägigen Normen
- Überwachung von:
  - Sicker-, Oberflächen- und Grundwasserproben (Häufigkeit, Probenahmeort, Analysewerte)
  - Porendruck
  - Eichung der Geräte
  - Auswertung von Ergebnissen und Unterlagen dazu
  - Maßnahmeplan bei Überschreitung der Grenzwerte
- TMF/WRMF-Sicherheitsorganisation der Grube, d. h. Benennung eines Verantwortlichen, Aufgaben und Zuständigkeiten für Einsatzkräfte, Ausbildungsprogramm, Unfallmeldesystem
- Angemessenheit von Betriebshandbuch, Handbuch für Betrieb, Wartung und Überwachung (OMS-Handbuch) oder Gleichwertiges, einschließlich Methoden der Ablagerung und Dammerhöhung, Absetzbecken und Wasserbewirtschaftung, Beherrschung von Sickerwasser und Staubentwicklung, Zufahrtstraßen, Überwachung, Dokumentation und Überarbeitung des Handbuchs
- Gesamtwasserbewirtschaftung der Anlage
- Überwachung entsprechend geltenden Normen
- Risikoeinschätzung, Unfälle, unkontrollierte Sickerung
- Gefahreneinschätzung, einschl. Unfälle mit Todesfolge, ökologische und ökonomische (oder gesellschaftsrelevante) Aspekte
- Einsatzbereitschaftsplan für Katastrophenfälle, Evakuierungsplan, Aufstellung mit allen erforderlichen Angaben für Sicherheitpersonal und Notdienste
- Außerbetriebnahmeplan, einschließlich Gefahrenanalyse, Langzeitstabilität, sicherer Rückhalt von toxischem Material, Bodenfruchtbarkeit und Optik.

Die Qualifikation zur Durchführung von Revisionen ist je nach Gefahrenklasse der Anlage unterschiedlich, hängt jedoch auch von der Verfügbarkeit von Fachkräften in der Region ab. Umfasst die Revision mehrer Disziplinfelder, muss in der Regel ein Expertenteam gebildet werden. Bei Dämmen für Aufbereitungsrückstände ist die Geowissenschaft generell von besonderer Bedeutung. Je nach örtlichen Bedingungen sind andere Wissenschaften, z. B. Hydrologie und Hydrogeologie, vertreten. Die die Revision durchführende Person oder Personen müssen Fachkräfte mit ausgewiesener Erfahrung auf bestimmten Wissensgebieten sein. Günstig kann die Zusammenarbeit mit Fachkräften aus dem Ausland sein, mit denen neues Wissen und neue Ansichten kommen.
[119, Benkert, 2003]

Die aktuellen Normen für Revisionen in unterschiedlichen Teilen der Welt sind in Anhang 5 beschrieben.

4.2.4 Stillegungs- und Nachsorgephase

Meist erfolgt die Stilllegung von Anlagen für Aufbereitungsrückstände und/oder taubes Gestein zeitgleich mit der Stilllegung einer Grube. Daher ist ein integrierter Stilllegungs- und Nachsorgeplan zu erstellen und umzusetzen. In diesem Abschnitt liegt der Schwerpunkt jedoch auf bestimmten Teilbereichen dieser Aufgabe (d. h. nicht auf der Grube, sondern auf den

Einrichtungen für die Bewirtschaftung von Aufbereitungsrückständen und Taubgestein). Bei Bedarf oder wo es sich anbietet, erfolgen Verweise auf den Gesamtstilllegungsplan. Es ist übliche Praxis, dass vor der endgültigen Stillegung des Standortes eine Auswertung der einzelnen Wiedergewinnungsaktivitäten in der Betriebsphase der Grube vorgenommen wird. Die folgenden Fragen werden dabei für die Vorphasen berücksichtigt, werden jedoch einer erneuten Betrachtung im Vergleich zur Bestandssituation am Standort unterzogen und die Stilllegungspläne entsprechend angepasst:

- Berücksichtigung der Stilllegungkosten bei der Bewertung von Alternativen
- Durchführung einer Risikoeinschätzung im Zusammenhang mit Stilllegungsplänen
- Bestehen von Stilllegungsplänen während der gesamten aktiven Nutzungsphase der Anlage, regelmäßige Aktualisierung bei Änderungen gegenüber der Auslegung und im laufenden Betrieb
- bei Bedarf Auslegung der Einrichtungen für eine vorzeitige Stilllegung
- Auslegung der Nachsorge für ein Minimum an aktiver Bewirtschaftung
- regelmäßige Überarbeitung und Aktualisierung des in der Planungsphase erstellten Stilllegungsplans in der Auslegungs- und Betriebsphase der Grube.

[45, Euromines, 2002]

Von großer Bedeutung bei der Erstellung des Stilllegungsplans ist die vorgesehene nachbergbauliche Nutzung der Flächen. Eine erfolgreiche spätere Nutzung des Standortes einer Anlage für Aufbereitungsrückstände verlangt eine ausgewogene Betrachtung von Aspekten der Ökologie, Umwelt, Erholung und Ökonomie. Alle Beteiligten (z. B. Betreiber, Genehmigungsbehörden, nichtstaatliche Organisationen, Kommunen) sind in diese Diskussion einzubeziehen.

Die oben genannten OSM-Handbücher haben auch Geltung für die gesamte Stillegungs- und Nachsorgephase.

4.2.4.1 Langfristige Ziele der Stilllegung

Die folgenden drei Klassen von Versagensmechanismen sind bei der Auslegung langfristig stabiler Anlagen für die Bewirtschaftung von Aufbereitungsrückständen und Taubgestein zu beachten:

1. Versagen des Damms in der Gründung oder Ausfall der Bewirtschaftungsanlage selbst
2. Extremereignisse, z. B. Überschwemmung, Erdbeben und starke Winde
3. Langsam wirkende Zerstörungskräfte, z. B. Erosion durch Wasser und Wind, Frost und Eiskräfte, Verwitterung von Versatzmaterial und Eindringen von Vegetation und Tieren.

[6, ICOLD, 1996]

Die in diesem Abschnitt verwendete Literaturstelle [100, Eriksson, 2002] basiert wesentlich auf der Anleitung der MIRO (1998) "A TECHNICAL FRAMEWORK FOR MINE CLOSURE PLANNING" und dem aktuellen Bericht von MiMi (1998) zu "Prevention und control of pollution from tailings and waste-rock products". Beide Dokumente werden Interessierten zur Lektüre empfohlen, weil sie einen guten Überblick über das Gebiet geben und zahlreiche nützliche Ideen enthalten.

Die Hauptkriterien der Stilllegungsprozesse, von der Ausgangsplanung bis zur eigentlichen Durchführung, sind in der folgenden Tabelle zusammen gefasst.

Problem	Ziele der Stilllegung
Physikalische Stabilität	Alle verbliebenen anthropogenen Strukturen sind physikalisch stabil.
Chemische Stabilität	Nach der Stillegung verbleibende physische Strukturen sind chemisch stabil.

Biologische Stabilität	Die biologische Umwelt wird wiederhergestellt als ein für den Standort typisches, natürliches, ausgeglichenes Ökosystem oder sie verbleibt in einem Zustand, in dem die natürliche Widerherstellung und/oder Wiederherstellung einer biologisch diversifizierten, stabilen Umwelt angeregt und ermöglicht wird.
Hydrologische und hydrogeologische Umwelt	Ziel der Stilllegung ist die Verhinderung des Eindringens und nachfolgende Belastung der abstromigen Umwelt, einschließlich Oberflächen- und Grundwasser, durch physikalische oder chemische Schadstoffe.
Geografische und klimatische Einflüsse	Stilllegung entspricht den Forderungen und Vorgaben des Standortes für Klima (z. B. Niederschlag, Sturmereignisse, jahreszeitliche Extreme) und geografische Faktoren (z. B. Nähe zu menschlichen Wohnorten, Topographfie, Zugang zur Grube)
Lokale Empfindlichkeiten und Optionen	Stilllegung optimiert die Möglichkeiten für die Wiederurbarmachung der Flächen; die Aufwertung der Flächennutzung wird bei Bedarf und/oder ökonomischer Duchführbarkeit berücksichtigt.
Flächennutzung	Wiederherstellung erfolgt mit Optimierung der schließlichen Flächennutzung und entspricht dem Bedarf des Umlandes und der lokalen Kommunen.
Gelder für Stilllegung	Für die Durchführung des Stilllegungsplanes müssen ausreichende und bei Bedarf abrufbare Gelder vorhanden sein.
Sozio-ökonomische Aspekte	Die Aspekte der lokalen Kommunen, deren Existenzgrundlage eventuell an den Arbeitsplätzen und den wirtschaftlichen Erträgen der bergbaulichen Tätigkeit abhängt, sind zu berücksichtigen. Angemessene Maßnahmen zur Maximierung der sozio-ökonomichen Folgen der Stilllegung sind zu treffen.

Tabelle 4.5: Zusamenfassung der Stilllegungskriterien
[100, Eriksson, 2002]

<u>Physikalische Stabilität</u>
Alle nach Stillegung der Grube verbleibenden anthropogenen Strukturen müssen physikalisch stabil sein. Sie dürfen bei Versagen und physikalischer Verschlechterung keine Gefahr für die Gesundheit und Sicherheit darstellen und müssen weiter den bei der Auslegung vorgesehenen Zweck erfüllen. Die Strukturen dürfen keiner Erosion unterliegen oder müssen ortsfest sein – es sei denn, eine Ortsveränderung erfolgt ohne Gefährdung der Sicherheit oder Schäden für die Umgebung. Das bedeutet, dass mögliche Extremereignise, z. B. Überschwemmung, Sturm, Erdbeben, ebenso wie sonstige ständig wirkende natürliche Kräfte, wie Erosion, in der Auslegungsphase in vollem Umfang zu berücksichtigen sind und entsprechende Sicherheitsfaktoren einbezogen werden müssen. Ziel der Überwachung von Strukturen ist der Nachweis, dass keine Verschlechterung des physikalischen Zustandes oder Deformation eingetreten ist.
[100, Eriksson, 2002]

Abweichungen von der üblichen Praxis entstehen in verschiedenen Bereichen. Diese werden im Folgenden betrachtet:

Extremereignisse
Dämme für Aufbereitungsrückstände werden so ausgelegt, dass sie unter den Bedingungen einer bestimmten Überschwemmung oder einer bestimmten Erdbebenstärke, z. B. der größten vermutlichen Flut (PMF) oder des größten anzunehmenden Erdbebens (MCE), ihre Stabilität bewahren. Die entsprechenden Auslegungswerte werden im Rahmen der bekannten meteorologischen und seismische Daten für die Region festgelegt und sind damit eine Funktion des Wissensstandes zum Zeitpunkt der Festlegung. Mit weiterer Verbesserung der Kenntnis technischer Faktoren und der Wechselwirkung zwischen großen Überschwemmungen und Erdbeben ist der Wissenstand ist jedoch ständiger Veränderung unterworfen. Damit ändern sich auch die Parameter der originalen Auslegung mit der Zeit und nehmen an Größe zu. Jederzeit

kann ein noch größeres Ereignis als das größte bisherige eintreten, in keinem Fall ein im Ausmaß geringeres. Der höchste Kostenaufwand für die Dammsicherheit entsteht bei herkömmlichen Staumauern von Wasserkraftwerken für die Verbesserung der Überlaufkanäle und Gründungen entsprechend den sich ändernden höheren Vorgabewerten. Bei manchen Bauwerken für Aufbereitungsrückstände (z. B. zahlreichen Absetzbecken) bedeutet diese Ertüchtigug eine ständige Aufgabe unter Nachsorgebedingungen. Anders wäre die Berücksichtigung der Prognosedaten für Extremereignisse nicht möglich.
[13, Vick,]

Mehrere wichtige zeitabhängige geotechnische Faktoren führen zu einer Verbesserung der Stabilität. Besonders kommt es sowohl in Aufbereitungsrückständen nach Abschluss der Setzungsvorgänge als auch bei losen Grobaufschüttungen zu einer signifikanten Verteilung erhöhter Porwenwasserdrücke. Normal führt dies zu einer Konsolidierung der Schüttung, Verbesserung der Scherfestigkeit und Reduzierung der Duchlässigkeit (besonders in vertikaler Richtung). Dies ist besonders bei Aufbereitungsrückständen mit Deckschichten, die nachträglich weiter geschüttet werden, der Fall. Eine ordnungsgemäße Entwässerung vorausgesetzt, erhöht sich der Sicherheitsfaktor für eine mögliche Instabilität in nahezu jedem Fall mit der Zeit und wird durch Anlegen und Wachstum der entsprechenden Vegetation weiter erhöht.

Die Senkungseffekte durch angrenzende und unterliegende bergbauliche Tätigkeit sowie die potenzielle Grundwassererhebung in Nähe des Dammes bzw. der Halde nach Beendigung des Bergbaus sowie deren wahrscheinliche Auswirkungen auf die Stabilität bedürfen ebenfalls der Berücksichtigung.

Kumulative Schäden

Ein in Verbindung stehender Faktor betrifft kumulative Schäden durch das wiederholte Auftreten von Extremereignissen oder fortschreitende Prozesse, wie innere Erison, die im Laufe der Zeit zu einer Verschlechterung der Dammstabilität führen. Bei Erdbeben entspricht es der üblichen Sicherheitspraxis, am Damm entstandene Schäden unverzüglich nach dem Schadensereignis instand zu setzen. Bei Anlagen für Aufbereitungsrückstände kann eine Instandsetzung aus mechanischen Gründen nicht möglich sein. Bei herkömmlichen Dämmen kann zur Reparatur größerer Schäden das Ablassen des Speicherinhalts erforderlich sein. Gleichzeitig ist dies eine wirksame Notfallmaßnahme. Bei Speicheranlagen mit festen Aufbereitungsrückständen kann der Füllstand nicht gesenkt werden. Zudem wirken auf einen Damm für Aufbereitungsrückstände auf unbestimmte Zeit wiederholt Extremereignisse ein, deren Anzahl von Zeit und Häufigkeit des Auftretens bestimmt wird. Für schwere Erdbeben in manchen Bergbauregionen ist die Größenordnung lediglich die von Jahrhunderten. Ein Beispiel für die kumulative Wirkung durch seismische Erschütterungen ist der La Villita Damm in Mexiko. Bei diesem kam es innerhalb von nur 30 Jahren bei vier getrennten Ereignissen starker seismischer Erschütterung zu zunehmend größeren Setzungserscheinungen im Bereich der Dammkrone. Kumulative Schäden sind auch die Folge einfacher Alterungserscheingungen. Kein Betonbauwerk - Überlaufkanal, Abflussanlage oder Tunnelauskleidung – hält ohne ständige Wartung und Reparaturen ewig.
[13, Vick,]

Klimawandel

Die Folgen langfristiger Klimaveränderungen sind von allergrößtem Interesse und ein großer Unsicherheitsfaktor. Zur Sicherung der zeitlich unbegrenzten Stabilität von Dämmen für Aufbereitungsrückstände müssen diese Änderungen bei Wasserstand und Kapaizät des Überlaufkanals in irgend einer Weise zuverlässig berücksichtigt werden, wozu selbst Klimaexperten nicht in der Lage sind. Klimatische Veränderungen können auch auf andere Weise beeinflussend auf die physikalische und chemische Stabilität wirken. Bei manchen Gruben in der arktischen und subarktischen Region wird davon ausgegangen, dass die Frostbedingungen die ARD-Reaktion senken. Die Stabilität bestimmter Dämme für

Aufbereitungsrückstände in diesen Gebieten hängt vom Dauerfrostboden ab. Es steht außer Frage, dass permanente Tauchung das Aufkommen an Wasser in ausreichender Menge auch in Trockenperioden voraussetzt, ganz abgesehen von künftigen Änderungen des Klimas.
[13, Vick,]

Daher ist es wichtig, die potenziellen Auswirkungen von Klimaveränderungen als Teil der Umweltverträglichkeitsprüfung zu betrachten (siehe Abschnitt 4.2.1.3), wenn dies von Relevanz für das Langzeitverhalten der gewählten Bewirtschaftungsoption ist.

Geologische Gefahren
Die Auslegung von Dämmen für Aufbereitungsrückstände erfolgt unter Berücksichtigung der zum Zeitpunkt des Baus bekannten geologischen Gefahren. Auf nicht absehbare Zeit wirken auf sie alle am jeweiligen Standort auftretenden geomorphologischen Prozesse (z. B, Erdrutsche, Steinschlag, Vulkantätigkeit, Karsteinstürze). Wie bei Extremereignissen, sind die Schadwirkungen dieser Prozesse lediglich eine Frage von Zeit und Häufigkeit des Auftretens, ein besonders bei geologischen Großerscheinungen schwer zu prognostizierender Faktor. Selbst die weniger dramatischen Prozesse der Alluvialablagerung lassen Wasserspeicher mit der Zeit verlanden, wenn Ablagerung und Schutt nicht kontinuierlich beseitigt werden.
[13, Vick,]

Chemische Stabilität
Bewirtschaftungsstandorte für Aufbereitungsrückstände und Taubgestein und die Bauwerke innerhalb derselben müssen über den gesamten Lebenszyklus chemisch stabil bleiben. Das bedeutet z. B., die Folgen chemischer Veränderungen oder Bedingungen, die zur Laugung von Metallen, Salzen oder organischen Verbindungen führen, dürfen keine Gefahren für die Gesundheit oder Sicherheit der Bevölkerung darstellen oder zu einer Verschlechterung der Umweltressourcen führen. In der Praxis müssen Aspekte wie die kurz- und langfristigen Einflüsse von Veränderungen der Geochemie von Aufbereitungsrückständen, des Abflusses von Sickerwasser aus Absetzbecken und Taubgesteinkippen sowie Grubenversatz oder die Ableitung von Oberflächenwasser am Standort, in Betracht gezogen werden. Bei bestehender Prognose von kontaminierten Abflüssen sind entsprechende reduzierende Maßnahmen (z. B. Absetzung oder passive Behandlung über Pflanzenkläranlagen) vorzusehen, um die Folgen der Kontamination zu reduzieren oder zu vermeiden, sofern diese zu negativen Umwelteinflüssen führen können. Ziel der Überwachung ist der Nachweis, dass keine negativen Auswirkungen (z. B. erhöhte Konzentration mit Überschreitung der gesetzlichen Grenzwerte) für Gewässer, Böden und Luft an dem geschlossenen Standort bestehen.
[100, Eriksson, 2002]

Bei **sulfidischen Aufbereitungsrückständen/taubem Gestein** besteht das wichtigste Ziel der Stilllegung in der Aufrechterhaltung der chemischen Stabilität der Aufbereitungsrückstände/des Taubgesteins durch Vermeidung der Freisetzung von Oxidationsprodukten in die Umgebung durch Verhinderung von Oxidationsreaktionen oder Vermeidung des Transports dieser Substanzen über die natürlichen Standortgrenzen hinaus oder beides. Natürliche Vorgänge können einen nachhaltigen Einfluss auf die Realisierung dieser Zielstellung haben. So können zum Beispiel Maßnahmen zur Einschränkung der Infiltration in die Ablagerung Vorrang vor anderen, wie z. B. dem Einbau von **Sohledämmschichten** mit geringer Durchlässigkeit mit entsprechendem hydraulischen Gradienten zur Unterstützung des Transports von Kontaminationsstoffen (so genannter ‚Badewanneneffekt') erhalten.
[13, Vick,]

Biologische Stabilität
Die biologische Stabilität des geschlossenen Standorts steht in einem engen Zusammenhang mit der vorgesehenen Flächennutzung. Die Stabilität der Umgebung dagegen wird in erster Linie bestimmt durch die physikalisch-chemischen Eigenschaften des Standortes. Alle drei stehen untereinander in Wechselwirkung, denn die biologische Stabilität kann einen signifikanten Einfluss auf die physikalische bzw. chemische Stabilität haben. Beispielsweise erschweren Pflanzenwurzeln die Erosion durch Bindung der Erdoberfläche. Die Entwicklung einer

gesunden Pflanzendecke über einer Pflanzenkläranlage erhöht die Oberflächentiefe von organischen Substanzen, wodurch anoxische Bedingungen geschaffen werden, wie sie für die Wasserbehandlung erforderlich sind. Der Rückbau von Standorten ist in der Mehrzahl der Fälle mit der Wiederurbarmachung großer rekultivierter Flächen verbunden, die für ein nachhaltiges Pflanzenwachstum häufig keine guten Bedingungen bieten. Daher ist es wichtig, dass die Methoden von Bodenverbesserung und Kultivierung oder von bodenbildendem Material zusammen mit den ausgewählten Arten zur Ausbildung einer nachhaltigen Pflanzendecke führen. Diese sollte der vorgesehenen Flächennutzung entsprechen und kann für die Aufrechterhaltung der physikalisch-chemischen Stabilität des Standortes, z. B. durch Stabilisierung der Bodendecke und Verhinderung von Erosion, von großer Bedeutung sein. Ziel der Überwachung ist der Nachweis, dass zum einen nicht nur der Pflanzenwuchs erfolgt, sondern sich über mehrere Vegetationsperioden hinweg zu einer selbsterhaltenden Pflanzengemeinschaft entwickelt hat.
[100, Eriksson, 2002]

Nach gängiger Sicherheitspraxis werden die schädigenden Auswirkungen von Bodenwühlern und das Durchdringen des Erdreichs mit Wurzeln als Aspekte betrachtet, die ständige Aufmerksamkeit erfordern. Andere Probleme können eher unerwartet auftreten. So ist z. B. der Biber als Symboltier Kanadas überall im Land verbreitet. Seine Habitate sind den Ingenieuren und Biologen bestens bekannt. Die Neigung des Tieres, an Orten mit fließendem Wasser aktiv zu werden, ist als ernst zu nehmendes Problem für die Langzeitstillegung von Dämmen für Aufbereitungsrückstände erkannt. Das Tier baut Dämme, die Ableitungsbauwerke blockieren können. Dies wurde in der Vergangenheit bereits als Ursache für das Versagen solcher Dämme für Aufbereitungsrückstände nachgewiesen. In Europa ist zu beachten, dass der europäische Biber, der etwa seit 1870 in Schweden als ausgerottet gilt, in den 1920er Jahren wieder eingeführt wurde und sich prächtig vermehrt.

Diese Faktoren zeigen im Detail deutlich, dass die Langzeitsicherheit von Dämmen von kontunuierlicher Wartung, Umbau und Reparatur abhängt und andererseits wie schwierig es ist, langfristig Stabilität zu sichern.
[13, Vick,]

Flächenfolgenutzung
Generell wird die Flächenfolgenutzung geschlossener Standorte von folgenden Faktoren bestimmt:
- vorbergbauliche bzw. aktuelle Flächennutzung des Umlandes
- voraussichtliche Veränderungen bei der Flächennutzung des Umlandes
- voraussichtliche Nutzung des Grubenstandortes nach Betriebseinstellung
- Möglichkeit der weiteren Nutzung von Infrastruktur und Einrichtungen
- Ausmaß der Umweltfolgen
- Notwendigkeit der Sicherung vor physikalisch-chemischen und biologischen Gefahren (anthropogenen und natürlichen Ursprungs).

Spezielle Probleme im Zusammenhang mit der festgelegten Langzeitbewirtschaftung müssen bei der Festlegung der Flächennachnutzung ebenfalls Berücksichtigung finden.

Damit ergeben sich für die Mehrzahl der Standorte verschiedene in Betracht zu ziehende Optionen, z. B.:
- natürliche Rekolonisierung der Fläche durch lokale Vegetation
- Anlage von kommerziell genutzten Forstpflanzungen
- Nutzung durch Landwirtschaft
- Ansiedelung alternativer Industrien
- Nutzung der Infrastrukturanlagen als Teil der wirtschaftlichen Entwicklung der Region.

Wie immer die Entscheidung fällt, die Flächen werden in der Regel wieder nutzbar gemacht, so dass Flächennutzung und Morphologie des Standortes mit der Umgebung bzw. der vorbergbaulichen Nutzung kompatibel sind. Die Ansiedelung von Industrie und anderen Unternehmen am Standort ist damit nicht ausgeschlossen.

4.2.4.2 Spezielle Fragen der Stilllegung

Aufschüttungen
Die Geometrie und damit die Stabilität von Aufschüttungen hängt von der Art des Materials in der Aufschüttung, der Schüttungsmethode und der Topographie ab.

Potenzielle Probleme und Gefahren bei Aufschüttungen sind:
- instabile Böschungen
- Bildung von toxischer Lauge mit abstromiger Kontamination
- Bildung von Sauerwasser
- Verschmutzung von Oberflächenwasser und/oder Grundwasser
- Brand/Selbstentzündung
- Schäden für Tiere, native Fauna und Gefährdung der Bevölkerung
- Staubbelastung und Winderosion
- optische Beeinträchtigung.

Es ist üblich, die Geologie vor Betriebsaufnahme vollständig zu erkunden. Besteht die Gefahr seismischer Aktivität oder anderer natürlicher oder vom Menschen verursachter destabilisierender Ereignisse, sind alle Maßnahmen und Bauwerke entsprechend auszulegen und auszuführen.
[100, Eriksson, 2002]

Becken/Dämme
Geschlämmte Aufbereitungsrückstände werden generell in Rückhalteinrichtungen, z. B. in Auffangbecken, geleitet, wo sie von der Umgebung getrennt sind und damit potenzielle Auswirkungen auf diese Umgebung vermieden werden. Für den Bau der Speicheranlagen werden in der Regel natürliche Geländeformen und Dämme genutzt, innerhalb derer die Bewirtschaftung der Aufbereitungsrückstände beherrscht werden kann.

Die Entscheidung über die Art und Standort der Speicheranlage wird von folgenden Faktoren bestimmt:
- Topographie
- natürliche Gefahren
- lokales Klima und Wasserbilanz
- Volumen der Aufbereitungsrückstände
- Konsolidierung der Aufbereitungsrückstände
- Toxizität der Aufbereitungsrückstände
- Umweltprobleme von Aufbereitungsrückständen und Prozesswasser
- Menge an geeigenten Abdeckstoffen
- vorhandener Mutterboden
- Wirtschaftlichkeit.

Potenzielle Probleme und Gefahren bei Absetzbecken sind u.a.:
- instabile Böschungen mit Einsturz und Dammversagen
- Sickerung oder Laugung mit abstromiger Kontamination
- Bildung von Sauerwasser
- Verschmutzung von Oberflächenwasser und/oder Grundwasser
- Schäden für Tiere, native Fauna und Gefährdung der Bevölkerung
- Staubbelastung und Winderosion.

Es ist üblich, die Geologie vor Betriebsaufnahme vollständig zu erkunden. Besteht die Gefahr seismischer Aktivität oder anderer natürlicher oder vom Menschen verursachter

destabilisierender Ereignisse, sind alle Maßnahmen und Bauwerke entsprechend auszulegen und auszuführen. Damit muss ein umfassender Bericht über die Hydrologie und Geochemie sowie die geotechnischen Aspekte des Standortes erarbeitet werden.
[100, Eriksson, 2002]

Wasserabdeckung
Bei der Auslegung von Absetzbecken müssen die Dämme ein vertretbares Maß an Sicherheit für die Betriebsperiode sowie die Zeit nach der Stilllegung bieten. In vielen Fällen ist eine ständige Wasserabdeckung oder ein Feuchtgebiet über den abgelagerten Aufbereitungsrückständen zur Vermeidung der Mobilisierung von Schadstoffen und/oder aus optischen Gründen angestrebt.

Die folgenden Abschnitte beschreiben die Auslegung von Dämmen mit Langzeitstabilität mit der Möglichkeit einer **ständigen Wasserabdeckung**.

Die Methode der Wasserabdeckung zur Sauerwasserbewirtschaftung wird in Abschnitt 4.3.1.2.1 ausführlich beschrieben.

Absetzbecken stellen eine potenzielle Gefahr für die Umwelt während des Betriebs sowie in der Nachstilllegungsphase dar. Zur Vermeidung von negativen Folgen für die Umwelt müssen Absetzbecken physikalisch und chemisch stabil sein. Dazu müssen zwei Bedingungen erfüllt sein:
1. Der Damm muss während der Betriebs- und der Nachstilllegungsphase hinreichende Stabilität besitzen.
2. Material mit einer möglichen negativen Auswirkung auf die Umwelt muss umweltverträglich gelagert werden.

Nach Einstellung des Betriebs muss das Absetzbecken auf sichere und optisch ansprechende Weise in die Landschaft integriert werden.
Enthalten die Aufbereitungsrückstände Sulfide, die bei Kontakt mit Luft und Wasser in einem langsamen Prozess oxidieren und dabei Säure und gelöste Metalle bilden, ist die Oxidation der Sulfide zu vermeiden, z. B. durch ständige Abdeckung der Aufbereitungsrückstände mit Wasser. In diesem Fall ist das Absetzbecken entsprechend der Langzeitstabilität des Dammes auszulegen und zu errichten und die Bedingungen für die ständige Wasserabdeckung der Fläche zu schaffen.
Die folgenden Voraussetzungen müssen für eine ständige Wasserabdeckung erfüllt sein:
- Es muss Wasser in ausreichender Menge zum Nachfüllen des Absetzbeckens zur Verfügung stehen, damit die Wasserabdeckung und eine stabile Wasserchemie jederzeit gewährleistet sind.
- Die Stabilität des Dammes muss eine hineichende Sicherheit in der Betriebs- und Nachstilllegungspase bieten.

In Bezug auf Stabilität verlangen die Langzeitaspekte eine entsprechende Dimensionirung des Damms für die gegebene Auslegung. 'Langzeit' bedeutet im Normalfall 'bis zur nächsten Eiszeit' oder 'mehrere Tausend Jahre'. Nach aktuellem Wissensstand sind zur Erfüllung der Forderung 'Langzeitstabilität' eines Dammes folgende Fragen zu beachten:
- Stabilität der Böschung
- Überströmung der Dammkrone
- Instabilität der Gründung und innerhalb des Damms
- Extremereignisse, z. B. Überschwemmungen, Erdbeben oder Sturn
- langsam wirkende Zerstörungsprozesse durch Sickerwasser, Niederschlag, Frost, Eis, Vegetation usw.

[126, Eriksson, 2003]

Langzeitstabilität von Dammböschungen bei Auslegung für ständige Wasserspeicherung
Nach Erfahrungen und Untersuchungen natürlicher Formationen, vergleichbar denen bei Dämmen für Aufbereitungsrückstände, ist bekannt, dass Böschungen mit einem geringeren

Verhältnis als 1:3 (Höhe/Breite) sich bisher als stabil gegen Wasser und Winderosion, Frost und Verwitterung seit den letzten 10000 Jahren (d. h. seit der letzten Eiszeit) erwiesen haben. Winkel unter 1:3 fördern auch die Vegetation, was die Wirkung langsamer Zerstörungsprozesse verringert.
[127, Benkert, 2002]

Zwischen dem Kern mit geringer Permeabilität und der Schüttung werden vertikale Filter vorgesehen. Der abstromige Dammfuß besitzt einen Filter (Beschreibung des Zwecks des Filtermaterials in Abschnitt **2.4.2.2**) und kann durch Grobmaterial weiter gestützt werden. Zur Überwachung von Menge und Qualität des Sickerwassers ist abstromig am Dammfuß ein Sammler vorzusehen (eventuell auch zur Erfassung des Sickerwassers, sofern es während der Betriebsphase nicht der Einleitungsvorschrift entspricht).
[126, Eriksson, 2003]

Überströmung
Die Gefahr der Überströmung hängt von den örtlichen Wetterverhältnissen und der Größe des Einzugsgebietes ab. Im Betrieb muss die Abflussleistung die bei vorhersehbaren extremen Zuflüssen abzuführenden Mengen aufnehmen können (z. B. PMF, siehe Abschnitt **2.4.2.6**). In der Regel beträgt die Abflussleistung das 2,5fache des maximalen Wasserstandes bei Messung an beliebigen Punkten. Wird für die Stillegung des Absetzbeckens eine Lösung mit permanenter Wasserabdeckung gewählt, muss das Abflussbauwerk (Abfluss) Langzeitstabilität besitzen und vorzugsweise als Überlaufkanals im natürlichen Boden und nicht duch den Damm errichtet werden. Der Abfluss mit Langzeitstabilität muss mit einer ausreichenden Sicherheit in der Lage sein, extreme Flutereignisse aufzunehmen, ohne Gefährdung der benötigten Abflussleistung durch Eis, umstürzende Bäume, Äste usw. Damit ist nach diesen Forderungen die Errichtung eines Abflussbauwerkes von sehr größer Breite für die Langzeitphase erforderlich.

Als Folge der Forderung nach einem ausreichenden Freibord besteht unter normalen klimatischen Bedingungen wahrscheinlich ein sehr breiter Bereich zwischen dem Rand des Beckens (Klarwasser) und der Dammkrone (der so genannte Spülrand). Dieser Bereich der Aufbereitungsrückstände wird bei Stilllegung mit einer undurchlässigen Materialschicht belegt, um Infiltration, Belüftung und Verwitterung zu vermeiden. Die Vorteile einer großen ‚Spülrandbreite' sind Verbesserung der Böschungsstabilität und Reduzierung der Gefahr innerer Erosion infolge der flachen phreatischen Grundwasserschicht und der Strömungslinien.

Instabilität
Ein Sicherheitfaktor von 1,5 wird für eine ausreichend niedrige Langzeitwahrscheinlichkeit von Instabilität in Untergrund, Gründung und im Damm selbst angenommen. Abschnitt 4.4.13.1 enthält weitere Beispiele für Sicherheitsfaktoren und ihre Bestimmung. Ferner muss bei Nassabdeckung der Neigungswinkel des hydraulischen Gradienten weniger als 50 % des Reibungswinkels des Materials, aus dem der Damm besteht, betragen.

Extremereignisse
Die dynamische Stabilität der Dammauslegung muss hinsichtlich maximal zu erwartender Erdbebendynamik des betreffenden Standorts geprüft werden. Ein Sicherheitsfaktor von 1,5 wird als ausreichend für die dynamische Stabilität angesehen. Starke Winde erzeugen Wellen, die den aufstromigen Damm und die Dammkrone schädigen können. Bei der Berechnung der Auslegungswellenhöhe sind Winddaten für den Standort heranzuziehen. Die Auslegungswellenhöhe bestimmt den notwendigen Erosionsschutz an der aufstromigen Böschung und erhöht u. U. den benötigten Freibord. Erosionsschutz ist für die Langzeitphase ebenso wie für die Betriebsphase erforderlich.
[126, Eriksson, 2003]

Langsam ablaufende Zerstörungsprozesse
In der Langzeitphase können Dämme durch langsam ablaufende Prozesse, z. B. Sickerung, Erosion, Temperatur, Frost, Eis, Vegetation usw. geschädigt werden.

Der mit hoher Wahrscheinlichkeit wichtigste Langzeitprozess für die Stabilität des Damm ist die Sickerung durch den Damm. Sickerung durch den Damm kann zu innerer Erosion führen, einer häufigen Schadensursache bei großen Kraftwerkstaumauern. Innere Erosion kann jedoch vermieden/verhindert werden, wenn die Neigung des hydraulischen Gradienten (d. h. die Porendrucklinie) so niedrig wie in den natürlichen Formationen ist, die gegen die Grundwasserströmung stabil sind. Generell ist ein Bodenböschung gegen interne Erosion stabil, wenn die Neigung des hydraulischen Gradienten weniger als die Hälfte des Reibungwinkels des Bodenmaterials beträgt.

Entsprechend muss ein Damm mit Landzeitstabilität so konstruiert sein, dass die Neigung des hydraulischen Gradienten weniger als die Hälfte des Reibungwinkels des Bodenmaterials beträgt. In diesem Fall kann davon ausgegangen werden, dass der Damm unter Grundwasserdruck und nicht unter statischem Wasserdruck steht und damit eine ausreichende Sicherheit gegen innere Erosion gegeben ist. Diese Bedingung wird wahrscheinlich bei der Dimensionierung der Dammbreite Verwendung finden.

Schäden durch Erosion, Temperatur und Vegetation können durch Verwendung von Material mit Langzeitstabilität beim Bau des Dammes und der Herstellung der Böschungen mit einem ausreichend kleinen Winkel vermieden werden. Ein Böschungswinkel von 1:3 (vertikal/horizontal) gilt als langzeitstabil, weil solche Böschungen in der natürlichen Landschaft vorkommen. Diese natürlichen Böschungen sind seit sehr langer Zeit natürlicher Erosion, Temperatur, Vegetation usw. ausgesetzt, in den skandinavischen Ländern seit der letzten Eiszeit (ca. 10000 Jahre) und trotz dieser langen Zeitperiode weisen sie sehr wenige Anzeichen von Veränderung auf. Das augenfälligste Zeichen von Veränderung sind Oxidation und Laugung der obersten 0,5 m Boden. Darunter ist die Moräne praktisch unverändert. Daher kann angenommen werden, dass aus solchem Material bestehende Dämme diesen Prozessen widerstehen können. Ähnliche Überlegungen gelten für für andere Materialien in anderen Teilen Europas.
[126, Eriksson, 2003]

Die folgende Abbildung zeigt einige typische Beispiele von Dammauslegungen für permanente Wasserabdeckung. In dieser Abbildung befinden sich die groben Aufbereitungsrückstände neben dem Damm.

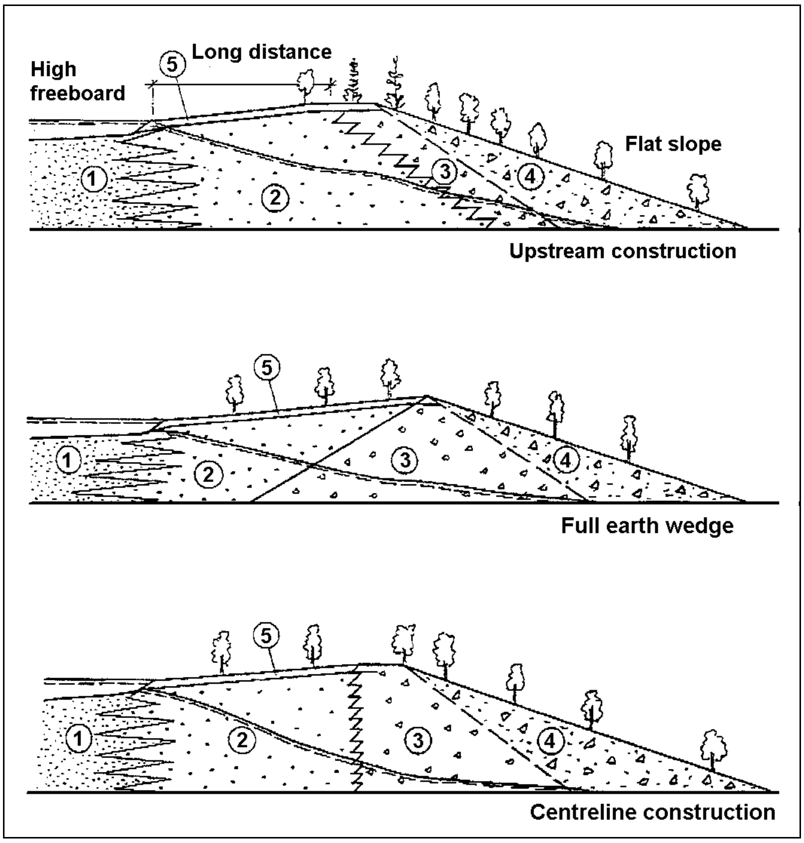


Abb. 4.2: Dämme für permanente Wasserabdeckung
1. feine Aufbereitungsrückstände, 2. grobe Aufbereitungsrückstände, 3. Stützschüttung, 4. Stützschüttung, langzeitstabil, 5. dichte Abdeckung und Erosionsschutz
[6, ICOLD, 1996]
High freeboard – hoher Freibord
Long distance – großer Abstand
Upstream construction – Upstream-Bauverfahren
Full-earth wedge – Erdvollkeil
Centerline construction – Centerline-Bauverfahren

<u>Entwässerte Becken</u>
Bei Stilllegung erhöht sich durch die Absenkung der phreatischen Grundwasserfläche die Stabilität der Böschung und reduziert die Gefahr innerer Erosion. Folgende Aspekte sind zur Vermeidung der oben genannten potenziellen Gefahren und Probleme zu berücksichtigen:

- Die Außenböschungen der Dämme werden zur Sicherung eines angemessenen Sicherheitsfaktors für Langzeitstabilität und seismische Belastungszustände umgebaut.
- Sickerung ist durch ausreichende Drainage zu beherrschen.
- Es müssen Vorkehrungen für die Sammlung und Ableitung des Oberflächenabflusses getroffen werden.
- Der Damm muss Langzeitstabilität gegen langsam wirkende Zerstörungsprozesse besitzen.
- Bei Aufbereitungsrückständen mit ARD-Potenzial ist eine geeignete Abdeckung zur Vermeidung/Verhinderung von Infiltration und Diffusion erforderlich (siehe Abschnitt 4.3.1).

Vorhandene Systeme für Sturmwasserableitung können für größere Kapazität und Dauerhaftigkeit ertüchtigt werden, so dass Erosion der Ablagerung bei starkem Niederschlag vermieden wird. Mönche und Mantelrohre müssen in einem Zustand gehalten werden, in dem sie kein potenzielles Langzeitrisiko darstellen. Es ist gängige Praxis, Mantelrohre mit einer Zementkapsel zu verschließen. Die Oberfläche des Damms wird so gestaltet, dass ein akzeptables Verhältnis zwischen Niederschlag und Verdunstung erreicht wird. In Gebieten mit hohen Niederschlägen kann ein Überlaufkanal zum Abfluss des überschüssigen Wassers auf der Dammoberfläche erforderlich sein.

Die folgende Abbildung zeigt einige typische Beispiele von Dämmen von entwässerten Becken. In dieser Abbildung befinden sich die groben Aufbereitungsrückstände neben dem Damm..

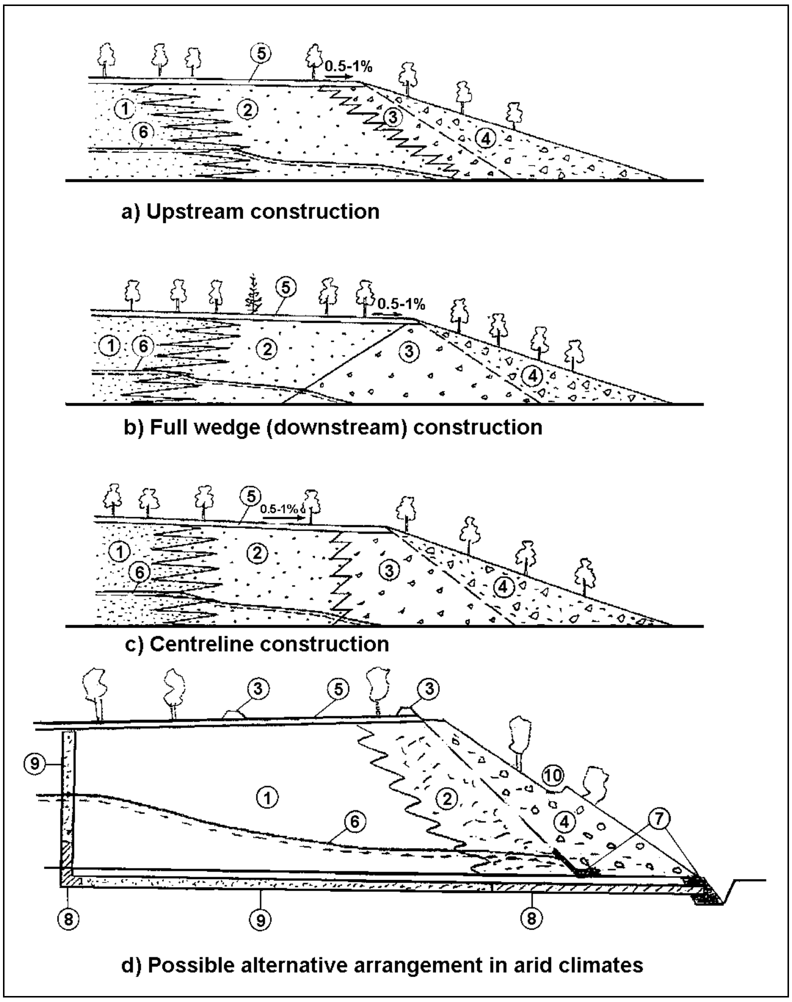


Abb. 4.3: Dämme for entwässerte Becken
[6, ICOLD, 1996]
Upstream construction – Upstream-Bauverfahren
Full wedge (downstream) construction – Vollkeil (Downstream-) Bauverfahren
Centerline construction – Centerline-Bauverfahren
Possible alternative arrangement in arid climates – Mögliche Alternative Lösung in Trockengebieten

Wasserwirtschaftliche Anlagen

Zu wasserwirtschaftlichen Anlagen gehören alle Einrichtungen an oder im Zusammenhang mit Grubenstandorten, die für Kontrolle, Speicherung, Behandlung und Transport von Wasser als Prozesswasser oder für die Hausversorgung eingesetzt werden sowie für die Ableitung und Behandlung von Überschusswasser. Dazu zählen:

- Becken/Dämme
- Rückhaltebecken
- Überlaufkanäle
- Zulaufbauwerke
- Ableitungsgräben
- Durchlassbauwerke
- Rohrleitungen
- Pumpstationen
- Behandlungsanlagen

- Absetzbecken
- Entwässerungssysteme.

Zu potenziellen Problemen und Gefahren im Zusammenhang mit der Stilllegung von wasserwirtschaftlichen Anlagen gehören:
- Verschmutzung von Oberflächenwasser und/oder Grundwasser
- unkontrollierter Wasserabfluss, der zu Überschwemmung und/oder Veränderungen der natürlichen hydrologischen Bedingungen führt
- Schäden, einschließlich Körperverletzung und/oder Tod von Tieren, nativer Fauna und Menschen.

Meist wird ein Bestandsverzeichnis aller Ausrüstungen und Einrichtungen am Standort oder im Einsatz für das Handling und/oder die Aufbereitung von Wasser aus dem Standort erstellt. Ihr Zustand wird dokumentiert und der Verbleib auf Karten und Lageplänen verzeichnet. Vor der Stilllegung werden umfassende Informationen zu den hydrologischen Bedingungen und der entsprechenden bergbaulichen Tätigkeit erhoben. Besteht die Gefahr seismischer Aktivität oder anderer natürlicher oder vom Menschen verursachter destabilisierender Ereignisse, müssen alle Maßnahmen und Bauwerke entsprechend ausgelegt und ausgeführt werden.

Wasserwirtschaftliche Anlagen werden außer Betrieb genommen und nach Möglichkeit am Standort rückgebaut, damit nicht zu stark belastetes Wasser abgeleitet wird. Es entspricht guter Praxis, wartungsbedürftige Anlagen während der Stilllegungsphase rückzubauen, besonders wenn Sicherheit, Stabilität oder die Umwelt gefährdet werden können. Entsprechend den Außerbetriebnahmeplänen des Standorts werden alle weiter nutzbaren Komponenten in die vorgesehene nachbergbauliche Flächennutzung, das System der Wasserbewirtschaftung und/oder die Drainageanlagen des Gebietes integriert.

Mit hoher Wahrscheinlichkeit hat die Wasserbewirtschaftung am Grubenstandort zu Veränderungen der natürlichen hydrologischen Verhältnisse geführt. Die Wasserspeicherung der Rückhalteanlagen verändert in der Regel die natürlichen Oberflächengewässer sowie Strömungsgeschwindigkeit und Durchflussmengen in natürlichen Wasserkanälen. Die Wiederherstellung der natürlichen hydrologischen Bedingungen beinhaltet die Einstellung der Wasserhebung aus unterirdischen Brunnen zur Flutung des Grubenbaus und Heben und Behandeln dieses Wassers, bis keine Gefahr für die Qualität des Grundwassers mehr besteht. Große Teile der aufgeschlossenen Oberfläche der unterirdischen Teile der aufgegebenen Grube können pyrithaltig sein und müssen vor der Erstflutung der Grube durch Oxidation behandelt werden. Verunreinigungen können mit Wasser aus der Grube gewaschen, besonders Sulfate und Metalle reduziert und damit die Gefahr der Kontamination verringert werden. Dieser Vorgang wird fortgesetzt, bis die normale Grundwasserqualität erreicht ist.
[100, Eriksson, 2002]

Stilllegung von Bewirtschaftungsanlagen für Aufbereitungsrückstände und Taubgestein mit nichtreaktiven Aufbereitungsrückständen und/oder taubem Gestein

Für nichtreaktive Aufbereitungsrückstände und/oder Taubgesteine müssen bei der Stilllegung folgende wichtige Gesichtspunkte beachtet werden:
- physikalische Langzeitstabilität
- Landschaftsgestaltung und Wiederurbarmachung
- Verhinderung von:
  - Erosion
  - Staubbildung.

An zahlreichen Standorten erfolgt die Landsachaftsgestaltung der Dammaußenbereiche bereits während des Dammbaus. Bei der Stilllegung werden die phreatischen Grundwasserspiegel dann mit Hilfe eines Überlaufsystems unter der obersten Schicht der Aufbereitungsrückstände gehalten, um Erosion im Bereich des Dammfußes zu verhindern. Die Aufbereitungsrückstände werden mit Lehm, Erde und Gras abgedeckt. Büsche und Bäume werden gepflanzt.

Die folgende Abbildung zeigt typische Abdeckungen von TMF. Optionen 1 und 2 finden bei nichtreaktiven Aufbereitungsrückständen Anwendung.

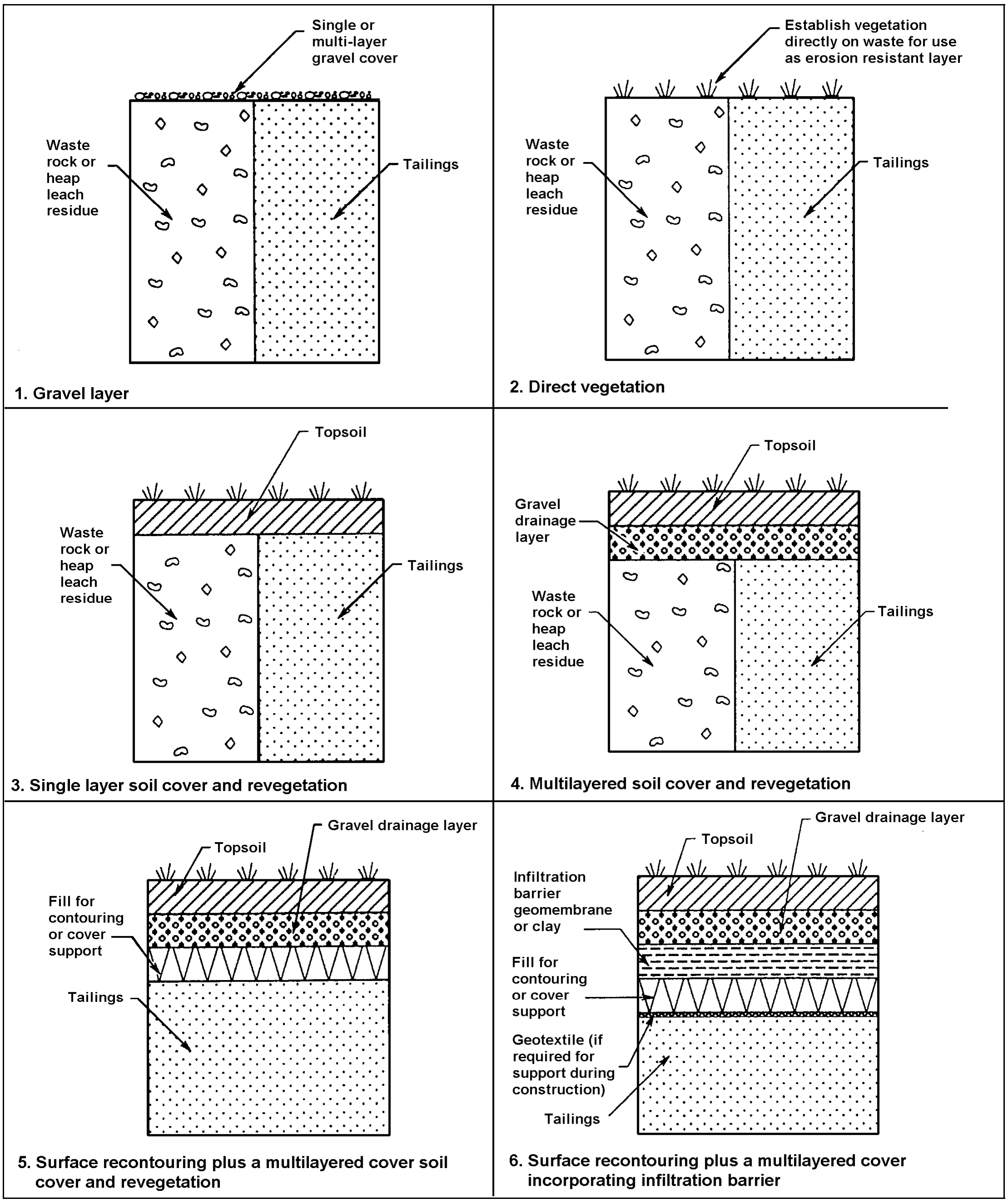


Abb. 4.4: Typische Abdeckungen von Bewirtschaftungsflächen von Aufbereitungsrückständen
[11, EPA, 1995]

1. Gravel layer – Kiesschicht

Single or multi-layer gravel cover – Ein- oder mehrschichtige Kiesabdeckung
Tailings – Aufbereitungsrückstände
Waste rock or heap leach residue – Taubgestein oder Haldenlaugungsrückstände

2. Direct vegetation – Direkte Vegetationsdecke

Establish vegetation directly on waste for use as erosion resistant layer – Errichtung der Vegetationsdecke direkt auf den Rückständen als Schutzschicht gegen Erosion
Waste rock or heap leach residue – Taubgestein oder Haldenlaugungsrückstände

3. Single layer soil cover and revegeation – Einschichtige Erdabdeckung mit Wiederurbarmachung

Topsoil - Mutterboden

Tailings – Aufbereitungsrückstände
Waste rock or heap leach residue – Taubgestein oder Haldenlaugungsrückstände

4. Multilayered soil cover and revegatation – Mehrschichtige Erdabdeckung und Wiederurbarmachung

Topsoil – Mutterboden
Gravel drainage layer – Kiesdrainageschicht
Waste rock or heap leach residue – Taubgestein oder Haldenlaugungsrückstände
Tailings – Aufbereitungsrückstände

5. Surface recontouring plus a multilayered cover soil cover and revegetation – Oberflächengestaltung plus mehrschichtige Erdabdeckung und Wiederurbarmachung

Topsoil – Mutterboden
Gravel drainage layer – Kiesdrainageschicht
Fill for contouring or cover support – Konturfüllung bzw. Träger für Abdeckung
Tailings – Aufbereitungsrückstände

6. Surface recontouring plus a multilayered cover soil cover incorporating infiltration barrier– Oberflächengestaltung plus mehrschichtige Erdabdeckung mit integrierter Infiltrationssperre

Topsoil – Mutterboden
Gravel drainage layer – Kiesdrainageschicht
Infiltration barrier geomembrane or clay – Geomembran oder Ton (Lehm) als Infiltrationssperre
Fill for contouring or cover support – Konturfüllung bzw. Träger für Abdeckung
Geotextile if required for support during construction – Geotextilschicht, wenn beim Bau als Träger erforderlich
Tailings – Aufbereitungsrückstände

4.3 Verhinderung und Kontrolle von Emissionen

4.3.1 Die Bewirtschaftung von Sauerwässern (ARD)

Die Bewirtschaftung von Aufbereitungsrückständen oder taubem Gestein mit Sauerwasserpotenzial folgt in der Regel einem Risikoansatz. Bei der Risikoeinschätzung sind korrekte Beschreibung und Kenntnis des Materials von entscheidender Bedeutung. Der Bewirtschaftungsprozess ist zyklisch und beginnt in der Planungsphase der Grube mit ständiger Aktualisierung und Prüfung über die Nutzungsdauer der Grube. Der Ansatz für den Bewertungsprozess folgt dem Prinzip ‘Von der Wiege bis zur Bahre’, d. h. jede Option für die Bewirtschaftung von Aufbereitungsrückständen und taubem Gestein in der Betriebsphase der Grube muss auch eine gangbare Stilllegungsstrategie vorsehen. Die Erstcharakterisierung des Materials erfolgt in der Planungsphase der Grube. Die Ergebnisse dieser Erstcharakterisierung werden jedoch ständig geprüft und durch Beschreibungen des Material in der Betriebsphase der Grube bestätigt.

Dieser Abschnitt orientiert sich an dem Bericht über “Prevention and control of pollution from tailings and waste-rock products” MiMi (1998) [95, Elander, 1998]. Mehrere Fallstudien wurden zusätzlich berücksichtigt. Der komplette Bericht kann im Internet nachgelesen werden.

Für die Vermeidung, Kontrolle und Behandlung von potenziell Sauerwasser bildenden Bergbauabfällen stehen verschiedene Optionen sowohl für die Betriebs- wie auch für die Stilllegungsphase von Gruben zur Verfügung.

Abschnitt **2.7** beschreibt die Prozesse bei der Entstehung von Sauerwässern.

4.3.1.1 Vorhersage des Sauerwasserpotentials

In **Ovacik** ergab sich aus der ausführlichen Charakterisierung von Proben, dass Aufbereitungsrückstände und taubes Gestein keine Sauerwässer erzeugen.

Nachstehende Tabelle zeigt die durchschnittlichen Ergebnisse von 99 Proben.

	pH	AP*	NP*	NNP*	NP/AP*	S^{2-} (%)
Durchschnitt von 99 Proben	7,52	0,47	5,5	5,18	4,67	0,02
*:kg $CaCO_3$ Gleichwert/t AP: Säurebildungspotenzial NP: Neutralisationspotenzial NNP: Netto-Neutralisationspotenzial						

Tabelle 4.6: Säureproduktionspotenzial der Goldmine von Ovacik
[56, Au group, 2002]

Die Charakterisierung von Aufbereitungsrückständen und Taubgestein (siehe Abschnitt 4.2.1.2 zusammen mit Anhang 4) umfasst:

- die Bestimmung des Säurebildungspotenzials (AP) auf der Basis von Gesamtschwefel oder Gehalt an Sulfid-S
- die Bestimmung des Neutralisationspotenzials (NP).

Wenn NP/AP $\leq$1:1, gilt, dass die Probe ein Säurebildungspotenzial besitzt. Bei NP/AP $\geq$ 3:1 gilt, dass die Probe kein Säurebildungspotenzial besitzt.

4.3.1.2 Optionen zur Verhinderung von Sauerwässern

Die Grundlage für Maßnahmen zur Verhinderung von Sauerwässern ist die Charakterisierung der Aufbereitungsrückstände und des Taubgesteins sowie ein umfassender Bewirtschaftungsplan, in dem die Menge an Aufbereitungsrückständen und Taubgestein, die eine spezielle Behandlung erfordert, festgelegt wird und minimiert ist. Zahlreiche Methoden der Verhinderung konzentrieren sich auf die Minimierung der Sulfidoxidation und damit auf die Primärmobilisierung von Verwitterungsprodukten. Das kann einher gehen mit der Minimierung des Sauerstofftransports zu den Sulfiden durch Einbringen einer Sauerstoffstransportsperre (Abdeckung). Bei den Abdeckungen handelt es sich im Normalfall um Varianten zweier Grundkonzepte: (1) ‘Wasserabdeckung’ oder ‘Feuchtabdeckung’ (d. h. Flutung), oder (2) ‘Trockenabdeckungen’. Eine dritte Variante, ‘Sauerstoff verzehrende Abdeckungen’, wurde ebenfalls entwickelt und wird angewendet. Weitere präventive Methoden zielen auf die Beseitigung von Sulfidstoffen aus Aufbereitungsrückständen oder Taubgestein (Depyritisierung), die Einbringung von Pufferstoffen, Minimierung der Aktivität von Bakterien oder Minimierung der durch Verwitterung angegriffenen Mineraloberfläche. Die Oxidation von Sulfidstoffen während des Betriebs kann z. B. durch Unterwasserbewirtschaftung von Aufbereitungsrückständen minimiert werden.

Verhinderungsmethode	Prinzip
Wasserabdeckung und Unterwasseraustrag	Verwendung einer Klarwasserabdeckung als Diffusionssperre gegen Sauerstoff. Die Sauerstoffdiffusion ist 10^4 Mal geringer als in Luft
Trockenabdeckung	Verwendung einer durchlässigen Schicht mit hohem Wassergehalt an Diffusionssperre gegen Sauerstoff
Sauerstoff verzehrende Abdeckung	Verwendung einer durchlässigen Schicht mit hohem Wassergehalt als Diffusionssperre gegen Sauerstoff. Dazu hat die durchlässige Schicht einen hohen Anteil an organischer Substanz, die beim Abbau Sauerstoff verzehrt und damit den Sauerstofftransport zu darunter liegenden Sulfiden weiter reduziert
Pflanzenkläranlage	Die Errichtung von Pflanzenkläranlagen ist eine Methode der Stilllegung nach dem gleichen Prinzip wie die Wasserabdeckung, jedoch mit geringerer Wassertiefe, da die Pflanzendecke die Sohle stabilisert und damit eine erneute Suspendierung der Aufbereitungsrückstände vermieden werden kann
Erhöhung des Grundwasserspiegels	Die darunter befindlichen Sulfidstoffe beliben konstant unterhalb des Grundwasserspiegels infolge Rückhalt des Wassers durch: ▪ erhöhte Infiltration ▪ verringerte Verdunstung ▪ erhöhten Strömungswiderstand ▪ Kapillarkräfte
Depyritisierung	Trennung des Pyrits von den Aufbereitungsrückständen und getrennte Ablagerung des Pyrit (z. B., unter Wasser)
selektive Materialbehandlung	Selektive Bewirtschaftung von verschiedenen Aufbereitungsrückständen oder Taubgesteinfraktionen entsprechend Zusammensetzung und Eigenschaften, z. B. Trennung der Stoffe nach Sauerwasserbildungspotenzial für getrennte Behandlung

Tabelle 4.7: Methoden und Wirkprinzipien zur Verhinderung von Sauerwässern

4.3.1.2.1 Wasserabdeckung

Wasserabdeckung oder 'Feuchtabdeckung' ist ein Stilllegungsverfahren, bei dem Klarwasser als Diffusionssperre für Sauerstoff dient. Der Sauerstoffdiffusionskoeffizient in Wasser ist 10^4 Mal kleiner als in Luft. Damit wird deutlich: Wenn eine Wasserabdeckung möglich ist, kann Suldifoxidation nahezu ausgeschlossen werden. Voraussetzungen für eine Wasserabdeckung sind:

- eine positive Wasserbilanz, die konstant eine Mindestwassertiefe garantieren kann
- Dämme mit physikalischer Langzeitstabilität (sofern für die Ablagerung der Aufbereitungsrückstände nicht wie in manchen Fällen eine Grube oder ein natürliches Gewässer benutzt wird)
- Abläufe mit Langzeitstabilität und ausreichender Ablaufkapazität auch bei Extremereignissen

- eine Wassertiefe innerhalb des Beckens, die die Resuspension der Aufbereitungsrückstände durch Wellenbildung verhindert (die erforderliche Wassertiefe kann mit Wellenbrechern verringert werden)
- dass die Aufbereitungsrückstände wasserlöslich sind.

Weiterhin ist es von Vorteil, wenn ein natürlicher Strom in das Becken zufließt, d. h. ein Strom, der organische Stoffe, Flora und Fauna, in das außer Betrieb genommene System einbringt. Damit wird die Wirksamkeit der Wasserabdeckung durch die Schaffung einer zusätzlichen Diffusionssperre durch Sediment weiter verbessert und kann die Rekolonisierung des Systems beschleunigen.

Wasserabdeckungren sind eine Stilllegungsoption für Absetzbecken jeder Art (z. B. für die Ablagerung 'normaler' Aufbereitungsrückstände oder die Unterwasserablagerung in der Betriebsphase).

Zwei Beispiele für Standorte, an denen diese durchgeführt wurde, sind Stekenjokk und Kristineberg.

Stekenjokk ist ein Pionierprojekt für die Außerbetriebnahme von Absetzbecken mit sulfidischen Aufbereitungsrückständen. Die Außerbetriebnahme erfolgte 1991, womit weitaus mehr als 10 Jahre Erfahrungen vorliegen. Das Außerbetriebnahmeprojekt Stekenjokk wurde ausführlich von Broman und Göransson (1994) beschrieben. Die in Stekenjokk durchgeführten Maßnahmen sind schematisch in der folgenden Abbildung dargestellt (aus Broman und Göransson, 1994).
[100, Eriksson, 2002]

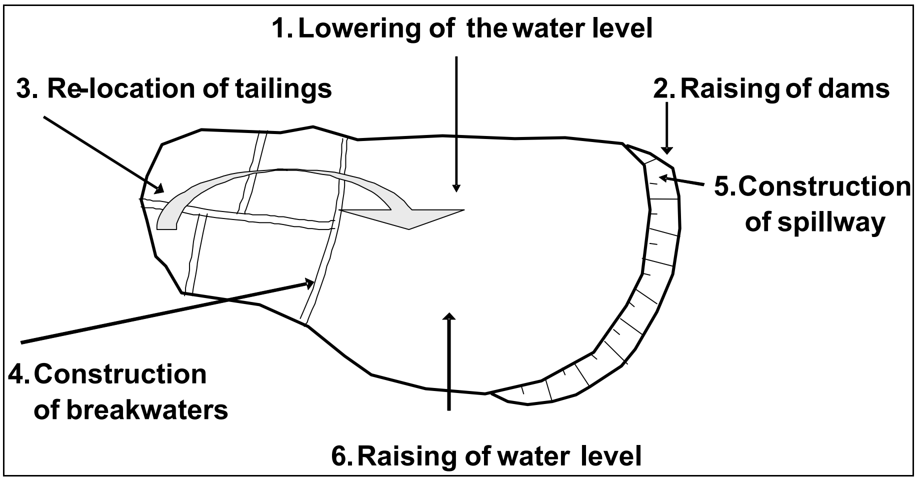


Abb. 4.5: Durchgeführte Maßnahmen am TMF Stekenjokk
[100, Eriksson, 2002]

1. Absenkung des Wasserspiegels
2. Dammaufschüttung
3. Umlagerung der Aufbereitungsrückstände
4. Bau von Wellenbechern
5. Bau des Überlaufkanals
6. Anstieg des Wasserspiegels

Die Wirksamkeit der duchgeführten Maßnahmen wird über die Zeit verfolgt und ausgewertet. In den Berichtsdaten für die ersten 8 Jahre der Nachsorge und unter der Annahme, dass Sulfat als Tracersubstanz für Sulfidoxidation verwendet werden kann, wurde eine Massenbilanzrechnung vorgelegt. Die Analyse macht deutlich, dass die Wasserabdeckung die Sulfidoxidation der abgelagerten Aufbereitungsrückstände wirksam reduziert. Ausgedrückt als Sauerstofftransport durch die Wasserabdeckung in die Aufbereitungsrückstände, entspricht der obere Grenzwert des Sulfataustritts des Absetzbeckens einer Obergrenze des effektiven

Sauerstofftransports von 1 x 10^{-10} kg O_2/m^2s. Dieser Wert entspricht dem von technisch betreuten Verbundlösungen mit Trockenabdeckung bzw. ist diesem überlegen. Diese Ergebnisse beweisen, dass die Ziele des Außerbetriebnahmeprojekts übertroffen wurden. Ähnliche Ergebnisse wurden kürzlich auch für Untersuchungen von Aufbereitungsrückständen bei Unterwasserablagerung in natürlichen Seen berichtet. Im Vergleich zur Trockenabdeckung ist die Wasserabdeckung wirksam und kosteneffektiv.

Die durchgeführten Wasserabdeckungen verursachten Investitionskosten von USD 2/m^2 im Vergleich zu USD 12/m^2 für untersuchte Trockenabdeckungen. Ferner konnte auf Entnahmegruben zur Gewinnung von Abdeckmaterial verzichtet werden.

Die Unsicherheit bei Wasserabdeckungen gilt der Langzeitstabilität des Damms. Einige Aspekte der Langzeitstabilität von Anlagen, bei denen die Wasserabdeckungsmethode angewendet wird, werden in Abschnitt 4.2.4.2 besprochen.

Es kann eingewendet werden, dass es nicht möglich ist, die Sulfidoxidation vollständig auszuschließen, da die Wasserabdeckung stets eine gewisse Sauerstoffkonzentration aufweist. Die Ergebnisse verdeutlichen jedoch, das die Sulfidoxidation in Stekenjokk so gering ist, dass sie vernachlässigt werden kann. Es wurden stetige Trends einer fallenden Sulfatkonzentration des Absetzmaterials im Becken ermittelt. Nach 10 Jahren liegt die Sulfatkonzentration im Schmutzwasser des Absetzbeckens noch immer nahe den Hintergrundwerten.

Die Haupterfahrungen dieses Standortes sind folgende:

- Die in Stekenjokk herrschenden extremen Winterbedingungen erwiesen sich als besondere Schwierigkeit bei diesem Projekt. Abnorme Anstiege des Wasserspiegels im Absetzbecken (die im Extremfall zur Überströmung des Kerns führen könnten) wurden in der Spätperiode des Winters festgestellt. Untersuchungen haben gezeigt, dass die Ursache ein teilweiser Verschluss des Ablaufbauwerks durch Eis war. Dies führte zu einem kompletten Umbau des Ablaufs. Das neue Ablaufbauwerk wurde aus Muttergestein errichtet und hat einen wesentlich tieferen Ablaufkanal, so dass das Wasser selbst bei extremen Eissituationen weiter abfließen kann (in Stekenjokk wurden Eisdicken von mehr als 2 m dokumentiert).

- Im Frühjahr 1998 wurden Anzeichen von “Trübung” im Sickerwasser an einer Stelle des Dammfußes festgestellt. Dies wurde als Anzeichen einer möglichen inneren Erosion gedeutet. Eine als Filter ausgelegte Stabilisierungsstrosse wurde unverzüglich am Dammfuß errichtet. Im Ergebnis der Untersuchung zeigt sich jedoch, dass die „Trübung“ auf die Bildung von Aluminosilikaten zurückzuführen war (als Folge der Auflösung von Silikaten zur Pufferung der Sulfidverwitterung). Damit hatte keine innere Erosion stattgefunden.

- Im Jahre 1998 wurde das Absetzbecken Stekenjokk im Zusammenhang mit der Erstellung des Dammsicherheitshandbuches (OSM-Handbuch) für Stekenjokk einer umfassenden Sicherheitsrevision unterzogen. Im Ergebnis der Revision wurde die Errichtung eines zusätzlichen Ablaufbauwerks empfohlen, um auch bei Blockierung des Hauptablaufs durch Eis den Ablauf zu sichern. Der Bau erfolgte noch im selben Jahr. Der Sicherheitablauf tritt automatisch in Funktion, sobald der Wasserspiegel einen bestimmten Stand erreicht.

- Am Dammkörper wurden nach Abschluss der Stilllegungsarbeiten keine Stabilisierungsmaßnahmen durchgeführt, die Dammböschung wurde auf das Verhältnis 1:2.5 (vertikal:horizontal) angepasst. Im Jahre 1994 wurde entschieden, die abstromige Böschung mit einer Moränenabdeckung zu versehen, weil festgestellt worden war, dass der Damm der Verwitterung ausgesetztes Sulfidmaterial enthielt, was das abstromige Wasserleben beeinträchtigte.

Die Außerbetriebnahme am Absetzbecken am Standort **Kristineberg** 4 ist noch nicht abgeschlossen, die durchgeführten Maßnahmen werden jedoch im Rahmen des Mimi-

Forschungsprojekts präzise verfolgt und im Internet unter www.mimi.kiruna.se dargestellt. [100, Eriksson, 2002].

Die Aufrechterhaltung einer Wasserabdeckung und Wartung eines Damms ohne Bewirtschaftung über einen langen Zeitraum ist ein Problem.

Weitere Informationen wurden durch die Untersuchung natürlicher Gewässer gewonnen, die seit relativ langer Zeit für die Unterwasserablagerung von Aufbereitungsrückständen genutzt werden. Fraser und Robertsson (1994) berichteten, dass Aufbereitungsrückstände, die in Mandy Lake zwischen 1943 und 1945 unter Wasser abgelagert wurden, wenig oder keine Zeichen chemischer Reaktionen auch nach 46 Jahren auf dem Grunde dieses Sees aufweisen. Ähnliche Ergebnisse liegen aus Untersuchungen für den Buttle Lake (Vancouver Island) vor.

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Robertson J. D., Tremblay G. A. und Fraser W. W., 1997. Subaqueous tailing disposal: A sound solution for reactive tailing. In proceedings of the fourth International Conference on Acid Rock Drainage, vol 3; 1029-1041.

St Arnaud L., 1994. Water covers for the decommissioning of sulphidic mine tailings impoundments. Bureau of Mines Special Publication, SP 06 A-94, 279-287.

4.3.1.2.2 Trockenabdeckung

Die Bezeichnung ‘Trockenabdeckung’ besagt nicht, dass die Abdeckung kein Wasser enthält. Sie verdeutlicht lediglich den Unterschied zwischen dieser Methode und der ‘Wasserabdeckung’. Trockenabdeckungen oder Bodenabdeckungen werden auch bei anderen Abfallstoffen häufig als Abdecklösung eingesetzt. Nach Beendigung der Abbautätigkeit und Einstellung der aktiven Ablagerung von Aufbereitungsrückständen wird das Wasser über der Fläche der abgelagerten Aufbereitungsrückstände abgelassen und die Oberfläche trocknet. Dabei bleibt ein großer Teil der feinerkörnigen Aufbereitungsrückstände weich und wassergesättigt. Dann wird über die Fläche eine Decke mit geringer Durchläsigkeit gelegt und zur Verbesserung des Oberflächenabflusses mit einem Gefälle versehen. In manchen Fällen werden durchlässige Schichten als Drainage, zur Überwachung oder als Kapillarbruch angelegt. Prinzipiell werden mit einer solchen Abdecke zwei Ziele verfolgt:

(1) Sie beschränkt den Sauerstoff von den oberflächlichen Aufbereitungsrückständen und die Sauerstoffdiffusion in Hohlräume und reduziert die Reaktionsrate und damit die Bildung von Sauerwaaser und
(2) damit hilft die Abdeckung auch bei der Verhinderung von Wasseransammlung und reduziert die Infiltration von Oberflächenwasser und damit den Transport von Reaktionsprodukten.

In der Praxis sind diese Ziele jedoch aus verschiedenen Gründen schwer und auch nur teilweise zu realisieren. Zudem besteht die Möglichkeit, dass geeignetes Abdeckmaterial lokal nicht zur Verfügung steht. Die Kosten und Probleme der Erdarbeiten auf der weichen Oberfläche der Aufbereitungsrückstände können erheblich sein.
[13, Vick,]

Eine allgemeine Methode für die Auslegung einer Trockenabdeckung besteht im Aufbringen mehrerer Schichten unterschiedlicher Böden, z. B. Ton (Lehm), Schluff, Sand und Kies. Die Wirksamkeit der Abdeckung hängt vom Feuchtigkeitsgehalt der Abdeckschichten ab. Die Gesamtmächtigkeit der Abdeckschichten beträgt normalerweise zwischen 0,3 – 3,0 m und die Duchlässigkeit der Dichtungsschicht zwischen 1 x 10^{-7} – 1 x 10^{-9} m/s.

Verschiedene Untersuchungen haben gezeigt, dass das Verhältnis zwischen der Diffusionrate und dem Grad der Wassersättigung hoch und in hohem Maße nichtlinear ist. Die folgende Abbildung zeigt das Verhältnis zwischen dem effektiven Diffusionskoeffizienten für porigen Stoff und einer gegebenen Wassersättigung und Diffusion in Luft nach Collin [140, Collin, 1987].

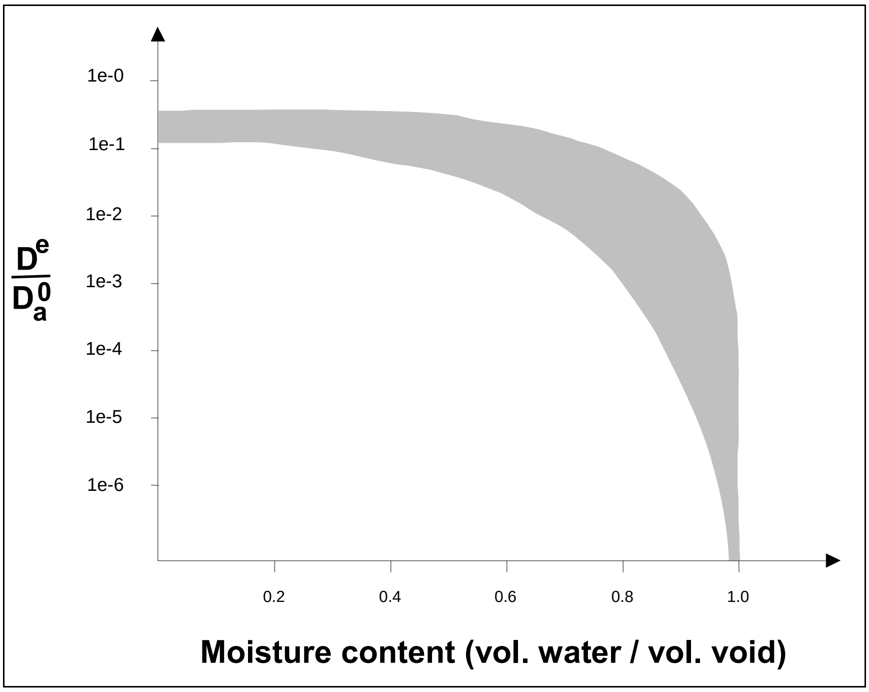

Abb. 4.6: Verhältnis zwischen dem effektiven Diffusionskoeffizienten in einem teilweise mit Wasser gesättigten poriger Stoff und Diffusion in Luft
Moisture content (vol. water / vol. void) – Feuchtigkeitsgehalt (Wasservolumen/Hohlraumvolumen)

Bevor das Absetzbecken abgedeckt werden kann, muss es entwässert werden, damit der Sand konsolidieren kann. Je nach Eigenschaften des Sandes kann die Konsolidierung eine lange Zeit in Anspruch nehmen. Damit ist es in manchen Fällen erforderlich, zur Verhinderung von Staubbildung in der Konsolidierungsphase eine zusätzliche Schicht auf die Aufbereitungsrückstände aufzubringen. Zur Verhinderung von Wasseransammlung ist es üblich, Umgehungskanäle anzulegen und die Oberfläche des Absetzbeckens umzugestalten. Im Indealfall erhält die Oberfläche eine Gefälle von 0,5-1,0 % zu den Rändern des Absetzbeckens.
[66, Base metals group, 2002]

Die Dichtungsschicht wird durch Aufbringen einer Schutzlage vor Austrockung und mechanischer Beschädigung geschützt. Die Schutzlage wird mit Vegetation versehen.

Die kurzfristige Wirksamkeit einer Trockenabdeckung kann langfristig durch verschiedene zerstörerische Prozesse, die zu Rissen und anderen Unterbrechungen der Sperrschichten führen, leiden. Solche Prozesse sind Erosion, Frosteinwirkung, Austrocknung, Setzungsunterschiede, das Durchdringen mit Wurzeln, Bodenwühler und Einwirkungen durch den Menschen [95, Elander, 1998].

Die einfachste Bodenabdeckung ist die mit unspezifischem, unverdichtetem Erdreich wie in Beispiel A (siehe Abb. 4.7). Unter skandinavischen Bedingungen wird auch bei einer Schicht aus Geschiebeton von 1,0 bis 1,5 m die Oxidationsrate noch um 80 – 90 % reduziert. Zur Erhöhung der Wirksamkeit der Abdeckung sind verschiedene Maßnahmen möglich. Der Nachteil einer einzelnen unspezifischen Bodenabdeckung auf den Aufbereitungsrückständen besteht darin, dass die Menge des Infiltrationswassers nur wenig reduziert wird (~10 %), während die Reduzierung der Sauerstoffdiffusion ebenfalls begrenzt ist, wenn der Grundwasserspiegel nicht bis auf das Niveau der Abdeckung angehoben wird.

Besitzt das vorhandene Deckmaterial bei Verdichtung eine relativ geringe Wasserdurchlässigkeit, besteht eine Option zur Verbesserung der einfachen Bodenabdeckung (Beispiel A) darin, die Abdeckung in oder auf 2 oder mehr Schichten ein- bzw. aufzubringen und diese Schichten einzeln zu verdichten. Damit reduziert sich die Bodendurchlässigkeit und der Sättigungsgrad erhöht sich, wodurch wiederum der effektive Sauerstoffdiffusionskoeffizient kleiner wird.

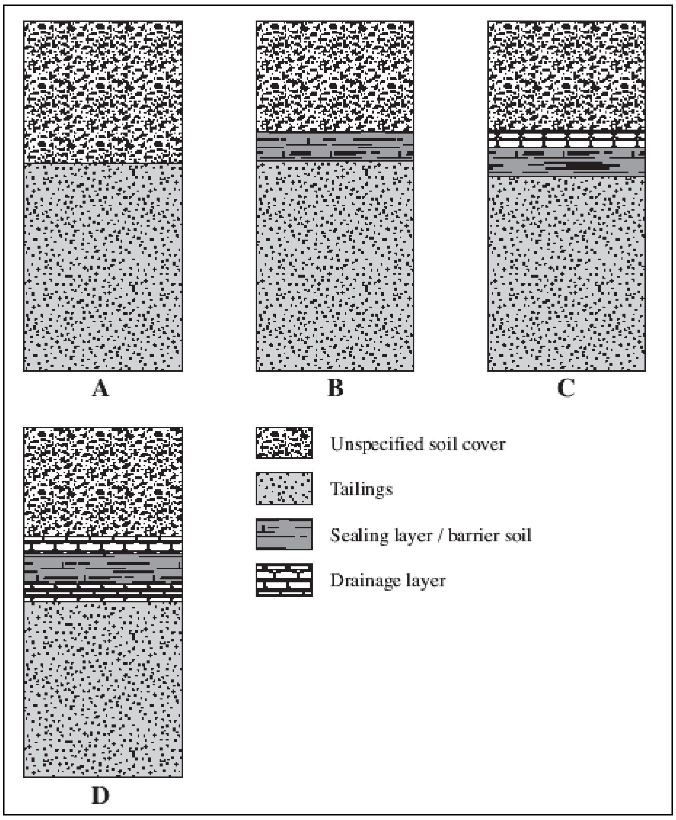

Abb. 4.7: Vier unterschiedliche Auslegungen für Bodenabdeckungen
Unspecific soil cover – unspezifische Bodenabdeckung
Tailings – Aufbereitungsrückstände
Sealing layer / barrier soil – Dichtungsschicht – Bodensperre
Drainage layer - Drainageschicht

Eine technisch anspruchsvollere Bodenabdeckung besteht aus einer verdichteten Dichtungs-schicht mit niederiger Wasserdurchlässigkeit, z. B. Ton oder toniger Geschiebestoff (aus dem größere Gesteinsbrocken entfernt wurde) wie in Beispiel B in Abb. 4.7. In Kristineberg in Nordschweden wurde durch Einsatz dieser Abdeckungsmethode die Oxidation um schätzungsweise >99 % reduziert, nachdem 0,5 m verdichteter Ton und 1,5 m Geschiebeton als Schutzlage auf die Aufbereitungsrückstände aufgebracht wurden (die tatsächlich in Kristineberg eingebaute Abeckung bestand aus 0,3 m verdichtetem tonigen Moränenstoff und 1,2 m Schutzlage aus unsortiertem Moränenstoff). Die Wasserinfiltration wurde um >95 % reduziert, die aus der Rückhalteanlage mit der Lauge austransportierte Menge an Metallen wurde geschätzt um >99,8 % verringert. Die erforderliche Mächtigkeit der Schutzlage hängt von den Bedingungen vor Ort (Frost, Austrocknung, Niederschlag usw.), der lokalen Flora und Fauna, d. h. Durchwurzelung, Bodenwühlern usw. und der Beschaffenheit des vorhandenen Stoffs für die Herstellung der Schutzlage ab. In Europa kann die Mächtigkeit der Schutzlage zwischen 0,5 m (z. B. in Aznalcóllar, Spanien; Bestimmung durch Trockenperioden) und 1,5 m (z. B. in Saxberget und Kristineberg, Schweden; Bestimmung nach Frosteindringtiefe plus einem Sicherheitfaktor) betragen. Temperaturmessungen in der Abdeckung in Kristineberg haben eine Frosteindringtiefe bis zu 0,9 m ergeben.

Eine Drainageschicht auf der Dichtungslage (Beispiel C in Abb. 4.7) reduziert die Infiltration weiter, weil der hydraulische Gradient niedriger gehalten wird, jedoch wird dadurch mit sinkendem Wassergehalt in der Abdeckschicht der Sauerstofftransport in die Aufbereitungsrückstände verstärkt und ein kontraproduktiver Effekt erzielt. Eine Groblage zwischen der Dichtungslage und den Aufbereitungsrückständen (Beispiel D in Abb. 4.7) kann als Kapillarbrecher wirken und die Entwässerung durch Kapillartransport nach unten und den möglichen diffusiven Transport von gelösten Elementen nach oben verhindern. Bei Entwässerung der Schicht mit geringer Wasserdurchlässigkeit besteht eine erhöhte Gefahr der Rissbildung und des nachfolgend verstärkten Transport von Sauerstoff. Zur Verhinderung der Vermischung des gröberen Stoffe und der feineren umgebenden Stoffe wird meist ein Geotextil dazwischen gebaut. Das hat jedoch Auswirkungen auf die Langzeitfunktion, da die Haltbarkeit eines synthetischen Materials über einen Zeitrahmen von Jahrtausenden unsicher ist. Bei Ausfall der Geotextilie durch Alterung oder mechanische Setzung, kommt es wahrscheinlich zu Vermischungserscheinungen der Lagen und Beeinträchtigung oder Ausfall der Funktion der Drainageschicht. Zur Vermeidung von Erosion der schützenden Geschiebetonschicht auf der Dichtungslage wird eine Vegetationsschicht auf der Schutzschicht angelegt. Wichtige Fragen ergeben sich, z. B. ob die Wurzeln der am Standort vorhandenen Pflanzen nicht irgendwann in späterer Zeit in die sanierte Rückhalteanlage eindringen, die Schicht mit niedriger Permeabilität durchdringen und mit welcher Mächtigkeit der Schutzlage dies verhindert werden kann. Auch Frost- und Tauwirkungen müssen berücksichtigt werden, sie können Risse und die Bildung von Makorporen begünstigen, was zu erhöhter Bodendurchlässigkeit führt. Nach Aufbringen der Schutzlage wird gegen Erosion meist eine Grasnarbe angelegt. [136, Carlssons, 2002]

Abschnitt 4.3.6 befasst sich mit Problemen von Wiederurbarmachung und Sanierung.

Beispiele für Standorte, an denen Trockenabdeckungen implementiert wurden, sind Apirsa (Aznalcollar), Aitik, Saxberget, Kristineberg und Enåsen.

Die Außerbetriebnahme der Absetzbecken der Grube in Saxberget in Mittelschweden, die zwischen 1994 und 1996 mit einer kombinierten Trockenabdeckung erfolgte, ist in der Literatur beschrieben. Zwei getrennte Becken waren für unterschiedliche Zeiträume in Betrieb, das Westbecken in der Zeit von 1930 - 1958 und das Ostbecken von 1958 - 1988. Das Westbecken hat eine Größe von 18 ha, das Ostbecken ist mit 35 ha etwa doppelt so groß. Insgesamt befanden sich 4 Mio t Aufbereitungsrückstände in den Becken. Die Zusammensetzung war ca. 2 % S, weniger als 1 % Zn und 0.5 – 1 % Calcit. Diese mineralische Zusammensetzung macht deutlich, dass das Material potenziell Sauerwässer erzeugt, auch wenn der Ablauf der Aufbereitungsrückstände im Ostbecken aktuell einen nahezu neutralen pH-Wert aufweist. [137, Lindvall, 1997]

Die Becken befinden sich in der permeablen Glazialformation, für die ein rasches Absinken des Grundwasserspiegels angenommen wurde, sobald die Zufuhr von Schlämmen der Aufbereitungsrückstände eingestellt würde. Dann würden Aufbereitungsrückstände in großer Menge dem Luftsauerstoff ausgesetzt sein. Während der Produktionsphase wurde von einer Mobilisierung von 3 t/a Zink ausgegangen. Untersuchungen ergaben, dass nach Erschöpfung der verfügbaren Pufferstoffe mit einer beträchtlichen Zunahme der Schadstoffbelastung zu rechnen sei, wenn die Sauerstoffzufuhr zu den Stoffen nicht kontrolliert erfolgte.

Die Modellierung der künftigen Mobilisierung des Metalls ergab eine jährliche Mobilisierung bis zu 600 t Zink in den Becken. Bedingt durch Niederschlag und Adsorptionsprozesse bei neutralem pH-Wert, wurde davon ausgegangen, dass sich die Menge für mehrere Jahre bei 3 t/a Nettotransport stabilisieren würde. Jedoch verlangte die Prognose einer hohen künftigen Schadstoffbelastung die Durchführung von Abhilfemaßnahmen. Da wegen der hydrogeologischen Bedingungen eine Flutung der Becken ausgeschlossen war, blieb als einzige realistische Option das Einbringen einer Abdeckung zur Reduzierung des Sauerstoffeintrags in die Aufbereitungsrückstände.

Da das vorgesehene Projekt erst das zweite und zudem das größte seiner Art in Schweden war, lagen zum Zeitpunkt der Entwicklung der Maßnahmepläne keine zuverlässigen praktischen Erfahrungen vor. Damit mussten mehrere Optionen in Betracht gezogen werden. Generell erfolgte die Auslegung der Abdeckung nach Grundsätzen auf der Grundlage des Untersuchungsprogramms der schwedischen Umweltschutzbehörde mit dem Ziel von Langzeitlösungen für Bergbauabfälle, die geringen Wartungsaufwand erforderten. Damit bestanden für die Abdeckung mindestens zwei Forderungen: (1) geringe Duchlässigkeit der Dichtungslage und (2) eine Schutzlage auf der Dichtungslage.

Am Beispiel von Saxberget erfolgte die Abdeckung der Aufbereitungsrückstände mit 0,3 m verdichtetem Geschiebeton als Dichtungslage und 1,5 m unsortiertem Geschiebeton als Decklage. Auf der Decklage befinden sich Gras und Birken als Vegetation.

Der wichtigste Punkt war die Dichtungslage. Dazu wurden verschiedene Lösungsansätze entwickelt. Wegen günstiger hydraulischer Eigenschaften sah eine den Einsatz von verdichtetem Klärschlamm aus Siedlungsabfällen vor. Aus praktischen Zwängen, hauptsächlich Zeitgründen, wurde diese Alternative aber verworfen.

Eine weitere Option bestand im Einsatz von Flugasche aus Kraftwerken in Form von 'Cefyll', einem betonartigen Produkt, das nach entsprechender Untersuchung bei einem ähnlichen Projekt zum Einsatz gekommen war. Der wesentlichste Nachteil dieser Alternative waren die Kosten, denn die Flugasche (Elektrizitäts- und Wärmekraftwerke auf Kohlebasis in der Region Stockholm) hätte aus zu großen Entfernungen antransportiert werden müssen.

Untersuchungen von Vorkommen an glazialem Geschiebelehm in der näheren Umgebung ergaben das Vorhandensein großer Mengen von Geschiebelehm in der Nähe des Grubenstandorts. Da dieser Stoff ausgezeichnete Wasserdurchlässigkeitseigenschaften besitzt und die Kosten die niedrigsten aller möglichen Alternativen waren, entschied man sich für dieses Dichtungsmaterial.

Das Modell von Sauerstoff- und Wassertransport in Verbindung mit Löslichkeitsberechnungen ergab Werte für den Metalltransport. Auf der Grundlage dieser Berechnungen wurden folgende Daten für die Duchlässigkeit der Dichtungsschicht festgelegt: 0,3 m mit einer Duchlässigkeit von 5×10^{-9} m/s.

Die Größe der Schutzlage war umstritten. Die Bergbaugesellschaft ging davon aus, dass 1 m Geschiebelehmaushub einen ausreichenden Schutz gegen Frost und Durchwurzelung böte. Die Umweltbehörde favorisierte eine Lage mit größerer Mächtigkeit. Schließlich einigte man sich auf eine Schutzlage von 1,5 m.

Die Anlage des Bereichs mit Aufbereitungsrückständen folgte möglichst dem umgebenden Gelände. Oberflächenwasser fließt in einem kleinen Strom ab, der am Westbecken entlang führt. Der Abfluss vom Westbecken erfolgt als Überlauf in das Ostbecken und bildet ein größflächiges flaches Feuchtgebiet. Auf diese Weise wird die Wassersättigung der Dichtungsschicht aufrecht erhalten und die Landschaft erhält ein attraktives und abwechslungsreiches Gesicht. Überschüssiges Wasser wird über ein gepflastertes Ablaufbauwerk über die frühere Dammböschung abgeleitet.

Die Nachbetreuung zeigt einen positiven Trend bei der Entwicklung der Schadstoffbelastung aus dem Gebiet. Es ist jedoch zu früh, endgültige Schlüsse hinsichtlich der Wirksamkeit der Abdeckung zu ziehen.
[100, Eriksson, 2002]

Zur Verhinderung von Erosion wird das Oberflächenwasser erfasst und kontrolliert abgeführt.

Nach dem Versagen des Absetzbecken in Aznalcóllar, der noch 96 % der vor dem Unfall vorhandenen Aufbereitungsrückstände enthielt, wurde auf einer Fläche von 150 ha eine Trockenabdeckung des Absetzbeckens durchgeführt. Der aktive Teil dieser Abdeckung bestand aus einer Dichtungslage von 0,5 m verdichtetem Ton ($k=10^{-10}$ m/s) und einer Schutzlage von 0,5 m. Zusätzlich zur Abeckung wurde eine Stabilisierungsstrosse vorgesehen, um weitere Bewegungen des Damms zu verhindern, zur Erhöhung des Sicherheitsfaktors wurde die Dammkrone abgesenkt, die Dämme auf ein Verhältnis 1:3 (vertikal:horizontal) verändert, zur Kontrolle des Oberflächenabflusses Drainagekanäle auf der Abdeckung gebaut, eine Schlitzwand um das Absetzbecken errichtet und schließlich zur Erfassung von Drainagewässern aus der Entwässerung der Aufbereitungsrückstände ein Netz von Rückpumpbrunnen innerhalb der Schlitzwand geschaffen. Die Kosten dieses Projekts beliefen sich auf USD 37 Mio (EUR 22/m^2). Ferner wurde ein umfassendes Überwachungsprogramm zur Kontrolle der Wirksamkeit der durchgeführten Maßnahmen installiert.

Die folgende Aufnahme zeigt die für die Grube Apirsa angewendete Lösung.

Abb. 4.8: Sammel- und Abführungskanal des geschlossenen Absetzbeckens von Apirsa

4.3.1.2.3 Entsorgung reaktiver Aufbereitungsrückstände unter Wasser

Die Entsorgung von Aufbereitungsrückständen unter Wasser wird auch als subaquatische Entsorgung bezeichnet. Ziel dieser Entsorgungsmethode ist die Minimierung des Kontaktes der Aufbereitungsrückstände mit Luftsauerstoff und damit die Minimierung der Oxidation der reaktiven Stoffe, besonders der Oxidation von Sulfiden. Ziel ist normalerweise eine konstante Wasserabdeckung der Aufbereitungsrückstände in der Betriebsphase und nach Stilllegung.

Die Wirksamkeit der Entsorgung von Aufbereitungsrückständen unter Wasser beruht wesentlich auf vier Mechanismen, die Robertson u. a. (1997) wie folgt zusammenfassen:

1. Verringerte Menge an vorhandenem Sauerstoff aus zwei Gründen: (1) Die gesättigte Sauerstoffkonzentration in Wasser ist 25000 Mal gringer als in Luft. (2) Der Sauerstoffdiffusionskoeffizient in Wasser ist 10000 Mal kleiner als in Luft. Damit steht für Oxidationsreaktionen sehr wenig Sauerstoff zur Verfügung und der Sauerstofftransport verläuft sehr langsam.
2. Sulfidreduktion. Bei geringer Sauerstoffkonzentration im Wasser verbrauchen Sulfat reduzierende Bakterien Sulfat und erzeugen damit Schwefelwasserstoff, der mit den meisten Metallen leicht reagiert und schwerlösliche Ausfällungen bildet.
3. Abfangen von Oxid. Dabei erfolgt die Bildung von Eisen- und Manganoxid als wirksame Stoffe zur Absorbierung einer Vielzahl gelöster Metalle.
4. Sedimentsperre. Nach Einstellung der Produktion entwickelt sich eine natürliche Sedimentschicht auf den abgelagerten Aufbereitungsrückständen, welche die Wechselwirkung zwischen den Aufbereitungsrückständen und dem darüber befindlichen Wasser wirksam minimiert.

Die Methode der Entsorgung unter Wasser ist im Rahmen des kanadischen Forschungsprogramms MEND eingehend untersucht worden. Das abschließende Ergebnis dieses Forschungsprojekts war die Erstellung und Veröffentlichung der Richtlinien für die Entsorgung reaktiver Aufbereitungsrückstände in Wasser in künstlich errichteten Rückhalteanlagen (MEND, 1998). Darin werden ausführlich alle relevanten Aspekte der Auslegung von Standorten für die Entsorgung von Aufbereitungsrückständen unter Wasser dargelegt. Zahlreiche ausführliche Publikationen zur Geochemie von wasserbedeckten Aufbereitungsrückständen in Stekenjokk und den Wasserabdeckungen von Kristineberg, hauptsächlich von Öhlander, Ljungberg und Holmström (z. B. Ljungberg, 1999; Holmström, 2000) wurden von der Universität Luleå vorgelegt.

Die Entsorgung von Aufbereitungsrückständen unter Wasser oder durch Flutung kann prinzipiell in künstlich errichteten Rückhaltebauwerken (Absetzbecken), gefluteten Tagebauen, natürlichen Seen oder unter Meeresbedingungen erfolgen. Die ökologische und politische Komplexität erhöht sich in der Reihenfolge der genannten Optionen. Normalerweise wird eine der beiden folgenden Entsorgungsmethoden angewendet:

- eine schwimmende Rohrleitung, welche die Aufbereitungsrückstände unter die Wasseroberfläche in die Entsorgungsanlage einbringt, die im Normalfall ortsveränderlich ist, damit die Aufbereitungsrückstände über die Anlage verteilt werden,
- eine im Wasser verlegte Rohrleitung, welche die Aufbereitungsrückstände unter die Wasseroberfläche einbringt.

Bei der Entsorgung von Aufbereitungsrückständen mit oder ohne Einschluss auf See reduzieren sich die ingenieurtechnischen Anforderungen (d. h. es muss kein Damm errichtet oder gewartet werden), verbessert sich die chemische Stabilität und wird weniger Landfläche beansprucht. Damit entfallen Sicherheitsfragen der Dämmung bei der Deponierung auf See oder in Gewässern. Häufig wird die Meeresbewirtschaftung von Aufbereitungsrückständen als gefährlich angesehen, da die Verteilung der Schadstoffe in der Umwelt weder prognostiziert noch kontrolliert oder korrigiert werden kann. Ein weiteres Problem besteht im mangelnden Wissen um die Meeresumwelt, und damit lassen sich die Risiken nur schwerlich abwägen.

Die Entsorgung unter Wasser kann die wirksamste Methode zur Verhinderung der Oxidation von Sulfiden sein. Damit verbessert sich die Wasserqualität während des Betriebs, der Aufwand für die Wasserbehandlung entfällt oder wird geringer.

Die Entsorgung unter Wasser minimiert den Stoffeinsatz bei der Stilllegung und vermeidet die Notwendigkeit von ausgedehnten Entnahmegruben zur Bereitstellung der erforderlichen Abdeckstoffe. Weitere Vorteile der Entsorgung unter Wasser sind z. B. die Vermeidung von Staubemission durch den Wegfall des Spülrandes sowie eine verbesserte Optik.

Die Kosten für die Entsorgung unter Wasser sind geringfügig höher als die herkömmliche Verfüllung über Wasser, da ein höherer Aufwand bei den täglichen Anpassungen zur optimalen Befüllung des Beckens erforderlich ist. Die Kosten der endgültigen Außerbetriebnahme sind drastisch niedriger.

Bei der Entscheidung über die Eignung dieser Methode müssen verschiedene Kriterien berücksichtigt werden. Die hydrologischen Bedingungen sind entscheidend. Es muss eine **positive Wasserbilanz** vorliegen. Das für die Ablagerung unter Wasser verfügbare Volumen muss ausreichend groß sein. Bei großen Gruben werden dafür große und tiefe Seen oder Zugang zum offenen Meer benötigt oder es müssen große Dämme errichtet werden, was nicht immer möglich ist.

Die Grube Lökken in Norwegen arbeitet mit ständiger Ablagerung unter Wasser. Die Grube Lisheen wendet diese Methode aktuell ebenfalls an. Wasserabdeckungen oder andere Methoden der Flutung von Aufbereitungsrückständen, Taubgestein und Gruben mit Wasser werden als Außerbetriebnahmemethode erfolgreich praktiziert und sind in der Literatur beschrieben (z. B. Eriksson u. a., 2001; Pedersen u. a., 1997; Amyot und Vézina, 1997). Eine ausführliche Leistungsuntersuchung zu Wasserabdeckungen wurde im Rahmen des MiMi Forschungsprojekts durchgeführt (http://mimi.kiruna.se).

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[122, Eriksson, 2003]

4.3.1.2.4 Sauerstoffverzehrende Abdeckungen

Bei Sauerstoff verzehrenden Abdeckungen wird eine permeable Lage mit hohem Wasseranteil als Diffusionssperre für Sauerstoff eingebaut. Die dünne permeable Schicht und wahrscheinlich auch die Schutzlage haben hohe organische Anteile die beim Abbau Sauerstoff verbrauchen und damit den Sauerstofftransport zu den darunter liegenden Sulfiden reduzieren. Die Verfügbarkeit geeigneter organischer Stoffe in ausreichenden Mengen ist für die Anwendung dieser Methode Voraussetzung.

Beispiele für Standorte mit dieser Abdeckungsmethode sind Galgberget (Mittelschweden) und Garpenberg (Mittelschweden).

[95, Elander, 1998] beschreibt die Außerbetriebnahme des Absetzbeckens Galgberget mit einer Sauerstoff verzehrenden Abdeckung wie folgt[15]:

Eine Beispielanlage ist Galgbergsmagasinet, ein Absetzbecken in Falun, Schweden, wo eine Abdeckung mit hohem Anteil an organischen Stoffen aus Schlamm einer Papierfabrik, Flugasche und Holzabfällen errichtet wurde. Oben auf das Absetzbecken wurde eine 1 m dicke Schicht aus einem Gemisch aus Flugasche und Schlamm aus der Papierfabrik aufgebracht, in zwei Lagen verdichtet und anschließend mit einer 0,5 m mächtigen Schicht aus Holzabfällen und grobem Geschiebeton abgedeckt. Teils wegen des Sauerstoffverbrauchs in der Abdeckung und teils duch eine Wirkung als physikalische Sperre in der gering permeablen, verdichteten Lage aus dem Gemisch aus Flugasche und Schlamm der Papierfabrik soll diese Abdeckung eine wirksame Sperre gegen den Sauerstofftransport bieten. Die Wasserdurchlässigkeit des Gemisches wurde im Labor mit $\leq 5 \times 10^{-9}$ m/s bestimmt. Auch das Wasserrückhaltevermögen wurde bestimmt und als ausreichend für die Aufrechterhaltung eines hohen Sättigungsgrades in der Sperre eingeschätzt. Weitere mögliche positive Wirkungen sind die Inhibierung der acidophilen Laugungsbakterien wegen des hohen Anteils an Calciumhydroxid in der Flugasche, der den pH-Wert des Sickerwassers erhöht und die Ausildung von nachhaltigen Bedingungen für Sulfat reduzierende Bakterien zur Erzeugung von Schwefelwasserstoff zur Ausfällung von Metallen fördert. Es besteht jedoch auch die Gefahr, dass es durch die gemeinsame Anwesenheit von organischen Verbindungen und Eisenhydroxiden im oberen (oxidierten) Teil der Ablagerung zur bakteriellen Reduzierung von Eisen kommen könnte, was gleichfalls ausgefällte Schwermetalle wieder in Lösung bringen würde. Die andauernden Untersuchungen lassen erkennen, dass sich die Oxidation von Sulfiden verlangsamt hat und der pH-Wert am Standort höher als der des Vergleichsstandortes ist. Hinweise auf signifikante bakterielle Sulfatreduktion wurden bisher nicht festgestellt.

Ein weiteres Beispiel für eine Sauerstoff verzehrende Abdeckung ist die Sanierung der Grube von East Sullivan in Quebec. Ferner wurden in einer kombinierten Labormodell- und Pilotstudie drei unterschiedliche organische Stoffe (Torf, kalkstabilisierter Klärschlamm und kompostierte feste Siedlungsabfälle) auf ihre Wirksamkeit als Sauerstoff verzehende Abdeckungsstoffe untersucht (Elliot u. a. 1997).

4.3.1.2.5 Pflanzenkläranlagen

Pflanzenkläranlagen als Stilllegungsverfahren arbeiten nach dem gleichen Prinzip wie Wasserabdeckungen, jedoch mit geringerer Wassertiefe, weil die Pflanzendecke die Sohle stabilisiert und eine Resuspendierung der Aufbereitungsrückstände verhindert. Die geringere Wassermenge im Absetzbecken verringert die potenzielle Gefahr eines Versagens des Damms. Die Voraussetzungen sind die gleichen wie für Wasserabdeckungen mit der zusätzlichen Forderung der Zugabe von organischer Substanz zur Verbesserung der Pflanzenvegetation im Becken.

[15] Aus MiMi Bericht (1998) zu "Prevention und control of pollution from Aufbereitungsrückstände und taubem Gestein products Der gesamte Bericht kann unter:http://www.mimi.kiruna.se hruntergeladen werden.

Das Hauptanliegen bei der Pflanzenkläranlage ist nicht die Behandlung des Wassers, sondern die Schaffung einer sich selbst generierenden und nachhaltigen Abdeckung, welche eine geringere Wassertiefe erfordert und als Sauerstoff verzehrende Sperre wirkt, wenn organische Stoffe auf den wassergesättigten Aufbereitungsrückständen aufgebracht werden.

Verschiedene Kohlen-TMF in Großbritannien sind zu Pflanzenkläranlagen umgestaltet worden, allen voran das Absetzbecken Nr. 8 in Rufford. Dies wurde in “The prospect for reservoirs in the 21st century” (Proceedings of the tenth conference of the BDS held at the University of Wales, Bangor on 9-12 September 1998): Ed. Paul Tedd: Thomas Telford, 1998 ISBN 0 7277 2704 4 an die British Damm Society und ebenso an die Institution of Mining und Metallurgy (Nottinghamshire und South Midlands Branches) mitgeteilt und “International Mining und Minerals”: January 2001 No. 37. ISSN 1461-4715 veröffentlicht. Eine aktualisierte Version (Juni 2001) wurde der 3rd British Geotechnical Association Geoenvironmental Conference an der University of Edinburgh im September 2001 vorgelegt und in “Geoenvironmental Engineering – Geoenvironmental impact management”: Ed. R.N. Yong & H.R. Thomas: Thomas Telford, 2001 ISBN 0 7227 3033 9 veröffentlicht.

Beispiele für Standorte, an denen Pflanzenkläranlagen beabsichtigt/geplant werden, sind Lisheen und Kristineberg.
[100, Eriksson, 2002]

4.3.1.2.6 Anheben des Grundwasserspiegels

Bei dieser Methode wird eine Schicht von geringer Mächtigkeit aufgebracht mit dem Ziel der Anhebung des phreatischen Grundwasserspiegels über das Nievau der Aufbereitungsrückstände und damit Verhinderung der Oxidation. Diese Lösung ist eine Mittelform zwischen Trocken- und Nassabdeckung zur Erzielung von Wassersättigung ohne offene Absetzbecken.

Neben der Reduzierung der Mächtigkeit der Abdeckung hat diese Methode den Vorteil, dass keine Verdichtung der Abdeckung erforderlich ist und die Anforderungen an die Qualität des Abdeckungsstoffs wesentlich geringer sind.

Einsetzbar ist diese Methode bei TMF, deren phreatischer Grundwasserspiegel bereits nahe der Oberfläche der Aufbereitungsrückstände liegt.

Die Kosten liegen höher als bei der Wasserabdeckung, jedoch (wegen der geringeren Mächtigkeit der Abdeckungslage) unter denen einer Trockenabdeckung.

Angewendet wird die Methode bei zwei Becken in Kristineberg, die beide stark verwittertes Material enthalten. Da das Material vollständig mit Wasser gesättigt ist, findet keine weitere Oxidation statt. Dies wird ohne die üblichen Probleme bei Flutung (z. B. Probleme der Dammstabilität) erreicht. Die Grundlage für eine derartige Maßnahme ist die sorgfältige Modellierung des Grundwassers unter Berücksichtigung des Einflusses von Oberflächenbewirtschaftung und Grundwasserhebungsdämmen.
[100, Eriksson, 2002]

4.3.1.2.7 Entpyritisierung

Diese Methode ist vergleichbar mit dem selektiven Handling von Material, erfolgt jedoch als Teil der Aufbereitung in der Aufbereitungsanlage. Pyrit kann durch Flotation abgeschieden und separat behandelt werden. Die Methode Sie ist anwendbar, wenn das ARD-Potenzial des größten Teils der Aufbereitungsrückstände durch Reduzierung des Pyritanteils erheblich verändert (d. h. das Sauerwasserpotenzial beseitigt) werden kann. Die entschwefelten Aufbereitungsrückstände erfordern weniger umfangreiche Außerbetriebnahmemaßnahmen.

Die Abscheidung der Sulfide erfolgt vornehmlich durch Flotation. Pyrit kann mit Xanthaten und Schaumbildnern in einem getrennten Flotationssystem mit guter Ausbeute aus siliciumhaltigen Aufbereitungsrückständen gewonnen werden.

In verschiedenen Anlagen wird die Pryitflotation zur Gewinnung des Pyrit als Schwefellieferant für die Produktion von Schwefelsäure betrieben. Das Verfahren ist bekannt. Es werden saure ebenso wie alkalische Prozesse eingesetzt. Das Pyritprodukt ist sehr reaktionsfreudig und erfordert damit sehr sorgfältige Auslegungsmaßnahmen für die Entsorgung. Geeignete Alternativen für pyritisches Produkt könnten Ablagerung unter Wasser in stillgelegten Tagebauen, in Hohlräumen von Gruben oder in Absetzbecken an Standorten sein, an denen der Grundwasserspiegel das Material konstant bedeckt.

Als Medien übergreifende Auswirkungen zu berücksichtigen sind:
- gringer zusätzlicher Einsatz von Energie und Reaktionsmittel für die Pyritflotation
- Energiestrafe bei separater Bewirtschaftung von hoch pyrithaltigen und depyritisierten Aufbereitungsrückständen.

Flotation und separate Bewirtschaftung des Pyrit verursachen erhebliche Kosten.

Die Wirtschaftlichkeit dieser Methode wird durch den abzuscheidenden Pyritanteil bestimmt. Ist der Anteil zu hoch, sind die Kostenauswirkungen negativ. Ein Kriterium ist, dass dass der resultierende Pyritanteil entsprechend niedrig für die Absicherung der Pufferung ist.

Beispielanlagen sind die Aufbereitungsanlage Nr. 1 in Bolidens, die bis 1991 eine kommerzielle Pyritproduktion betrieb und die Aufbereitungsanlage Pyhäsalmi, in der die Pyritproduktion noch läuft. Es ist nicht bekannt, ob Pyrit als Teil des Sanierungsplanes abgeschieden wird.

Im Kupferbergwerk Aitik gilt Depyritisierung als Schlüsselelement für den Stilllegungsplan des Absetzbeckens. Auf Grund der hydrogeologischen Modellierung wird für diesen Standort angenommen, dass nur ein kleiner Teil des außer Betrieb genommenen Absetzbeckens in Trockenperioden entwässert wird. Der Plan sieht vor, die Depyritisierung der Aufbereitungsrückstände in den letzten Produktionsjahren durchzuführen, so dass die obere Schicht der Aufbereitungsrückstände einen gringeren Schwefelgehalt aufweist. Teil dieses Konzepts ist die Abscheidung und separate Bewirtschaftung der Pyritfraktion mit Schwefelgehalt zwischen 30 und 35 Prozent und deren Einlagerung in einen separaten Teil des Absetzbeckens. Die Ablagerung des Pyrits erfolgt in einem Becken mit permeablen Dämmen, die gleichzeitig mit den umgebenden Bauwerken errichtet werden. Die Fläche des Pyritbeckens nimmt 0,5 – 1 % der Gesamtfläche (6 - 12 ha) ein. Das Becken wird hauptsächlich mit Wasser gesättigt sein, bei einer Stilllegung könnte die Abdeckung mit Erdreich /Trockenabdeckung erfolgen.

4.3.1.2.8 Selektives Handling von Materialien

Das selektive Handling von Materialien ist ein Erfordernis für die Wirtschaftlichkeit im Betrieb. Durch die selektive Ablagerung von reaktiven und nichtreaktiven Aufbereitungsrückständen oder taubem Gestein, kann die Außerbetriebnahme des nichtreaktiven Teils erheblich reduziert werden. Es könnte selbst möglich sein, alternative Einsatzmöglichkeiten für die nichtreaktive Fraktion zu finden.

Als ein Beispiel wird nachfolgend die selektive Bewirtschaftung von Taubgestein mit bzw. ohne Sauerwasserbildungspotenzial besprochen:
Die geologischen Formationen an Sulfiderzlagerstätten weisen häfig zonare Strukturen auf, wobei die erznahen Schichten einen erhöhten Pyritanteil besitzen. Im Tagebau ist es in manchen Fällen möglich, Taubgesteinsarten unter Verwendung geochemischer Eigenschaften als Kriterium selektiv zu bewirtschaften. Sorgfältige geologische Kartierung und Folgeanalysen von Bohrgut liefern erforderliche Informationen für die Klassifikation. Auf dieser Grundlage ist

es dann möglich, Taubgestein mit bzw. ohne Sauerwasserbildungspotenzial voneinander zu trennen.

Die Bedingungen für Betrieb und Außerbetriebnahme bei Taubgestein hängen vom Netto-Sauerwasserbildungspotenzial ab. Taubes Gestein ohne dieses Potenzial erfordert weniger umfangreiche Maßnahmen für die Außerbetriebnahme als Sauerwasser bildendes Taubgestein.

Wird Taubgestein nicht selektiv gehandhabt, muss die Bildung von Sauerwässern insgesamt verhindert werden. Bei selektiver Handhabung ist die Bewirtschaftung der ARD-produzierenden Taubgesteinsfraktion wegen der geringeren Mengen (im Vergleich zur Gesamtmenge an Taubgestein) einfacher.

Die selektive Bewirtschaftung von Taubgestein erfordert keine komplexe Technologie, lediglich leistungsfähige Methoden bei Informationserfassung und Verwaltung des Materials entsprechend diesen Ergebnissen.

Taubgestein mit niedrigem Schwefelanteil kann die Vorgaben für Baustoffe und Zuschläge erfüllen und damit gebrochenen Stein ersetzen.

Die selektive Bewirtschaftung verursacht in der Betriebsphase höhre Kosten. Bei Stilllegung der Grube können die Sanierungskosten jedoch gesenkt werden.

Die Einsatzmöglichkeit wird von der Geologie, dem Abbauverfahren und den geochemischen Eigenschaften jeder Art von Taubgestein bestimmt.

Mehrere Gruben in aller Welt arbeiten nach der selektiven Bewirtschaftung von Taubgestein. Die Grube Aitik in Boliden ist ein Beispiel für eine Großanwendung.

Ein weiteres Beispiel für selektive Bewirtschaftung von Materialien ist die Goldmine von Ridgeway in South Carolina (USA). Dort wurde in der Anfangszeit des Abbaubetriebs nichtreaktives Material gelagert und bei Schließung der Grube als Teil der Abdeckung der Aufbereitungsrückstände eingesetzt.
[120, Sawyer, 2002]

4.3.1.3 Kontrollmöglichkeiten

Wenn Verwitterungsreaktionen nicht verhindert werden können (wie dies z. B. in der Betriebsphase einer Grube möglich sein könnte), muss die Migration von Sauerwässern kontrolliert werden. Zu den Methoden, die auf die Minimierung des Transports von Verwitterungsprodukten aus der Ablagerung in die Umwelt orientieren, gehören u. a. die Ableitung von unbeteiligtem Oberflächenwasser, die Erfassung von beteiligtem Oberflächenwasser und die Kontrolle des Grundwasserstroms. Eine Minimierung der Infiltration in die Ablagerung kann oft mit einfachen Abeckungen erreicht werden. Weitere mögliche Methoden sind – wie aus nachfolgender Tabelle ersichtlich – Mischen und die Zugabe von Pufferstoffen.

Kontrollmethode	Angewendetes Prinzip
Mischen	Zugabe von Aufbereitungsrückständen und taubem Gestein mit einer hohen Pufferleistung zu Stoffen mit Sauerwasserbildungspotenzial, damit kann der pH-Wert im neutralen Bereich gehalten werden.
Zugabe von Pufferstoffen (Kalkung)	Verbesserung der Pufferleistung bei Stoffen mit Sauerwasserbildungspotenzial, damit kann der pH-Wert im neutralen Bereich gehalten werden.
Verdichtung und Bodenversiegelung	Durch kombinierte Verdichtung und Versiegelung der unteren Schichten wird die Bildung von Sauerwasser minimiert und unkontrollierte Sickerung in den Untergrund verhindert (siehe Abschnitt 4.3.10.4)

Tabelle 4.8: Methoden zur Kontrolle und Wirkprinzip der Sauerwässerbildung

4.3.1.3.1 Zugabe von Puffersubstanzen

Die Zugabe von Pufferstoffen (z. B. Kalk) erfolgt in der Regel vor der Trockenabdeckung. Sie dient der Immobilisierung der zum Zeitpunkt der Außerbetriebnahme der Grube vorhandenen Verwitterungsprodukte.

Theoretisch ist es ebenfalls möglich, dies als Methode für die Außerbetriebnahme einzusetzen, da die Zugabe von Pufferstoff in ausreichender Menge die Absenkung des pH-Wertes und damit die Produktion von Sauerwässern verzögern oder sogar vermeiden würde. Die Erzielung einer derart langfristigen Pufferwirkung in Ablagerungen mit Sauerwasserbildungspotenzial erfordert in der Regel Pufferstoffe in großer Menge, deren Verbringung an den Einsatzort mit unvertretbar hohen Kosten verbunden wäre.
[100, Eriksson, 2002]

Dieses "Mischen" ist nur möglich, wenn der Pufferstoff bereits am Standort vorhanden ist oder vorzugsweise als Abfall am Sandort anfällt. Andernfalls sind die Transportkosten zu hoch.

4.3.1.4 Behandlungsmöglichkeiten

Während der Betriebsphase der Grube oder in Fällen, in denen die Minimierung der Sulfidoxidation nicht ohne weiteres möglich ist, kann es notwendig sein, das Drainagewasser vor der Einleitung in die Umwelt zu sammeln und zu behandeln. Diese Behandlung könnte entweder passiv (z. B. Pflanzenkläranlagen oder anoxische Kalksteindrainage) oder aktiv in einer Wasseraufbereitungsanlage (direkte Kalkung, HDS-Prozess, usw.) erfolgen. Bei Stilllegung kann es erforderlich sein, das Drainagewasser auch noch nach Aufbringen einer Abdeckung zu behandeln, bis die Folgen der Freisetzung der Drainage in die Umwelt tolerierbar geworden ist.

Methoden der Schmutzwasserbehandlung werden in Abschnitt 4.3.11 beschrieben.

4.3.1.5 Entscheidungen zur Stilllegung von ARD-verursachenden Standorten

In letzter Zeit sind verschiedene Anleitungen für die Stillegungsplanung von Gruben entwickelt wordem (z. B. MIRO, 1999, “A technical framework for mine closure planning”,. Mineral Industry Research Organisation, Technical Review Series No. 20). Die folgende Abbildung enthält einen der in der Literatur angegebenen Entscheidungsbäume für die Auslegung der Stilllegung einer potenziell ARD-verursachenden Ablagerung von Aufbereitungsrückständen und taubem Gestein.

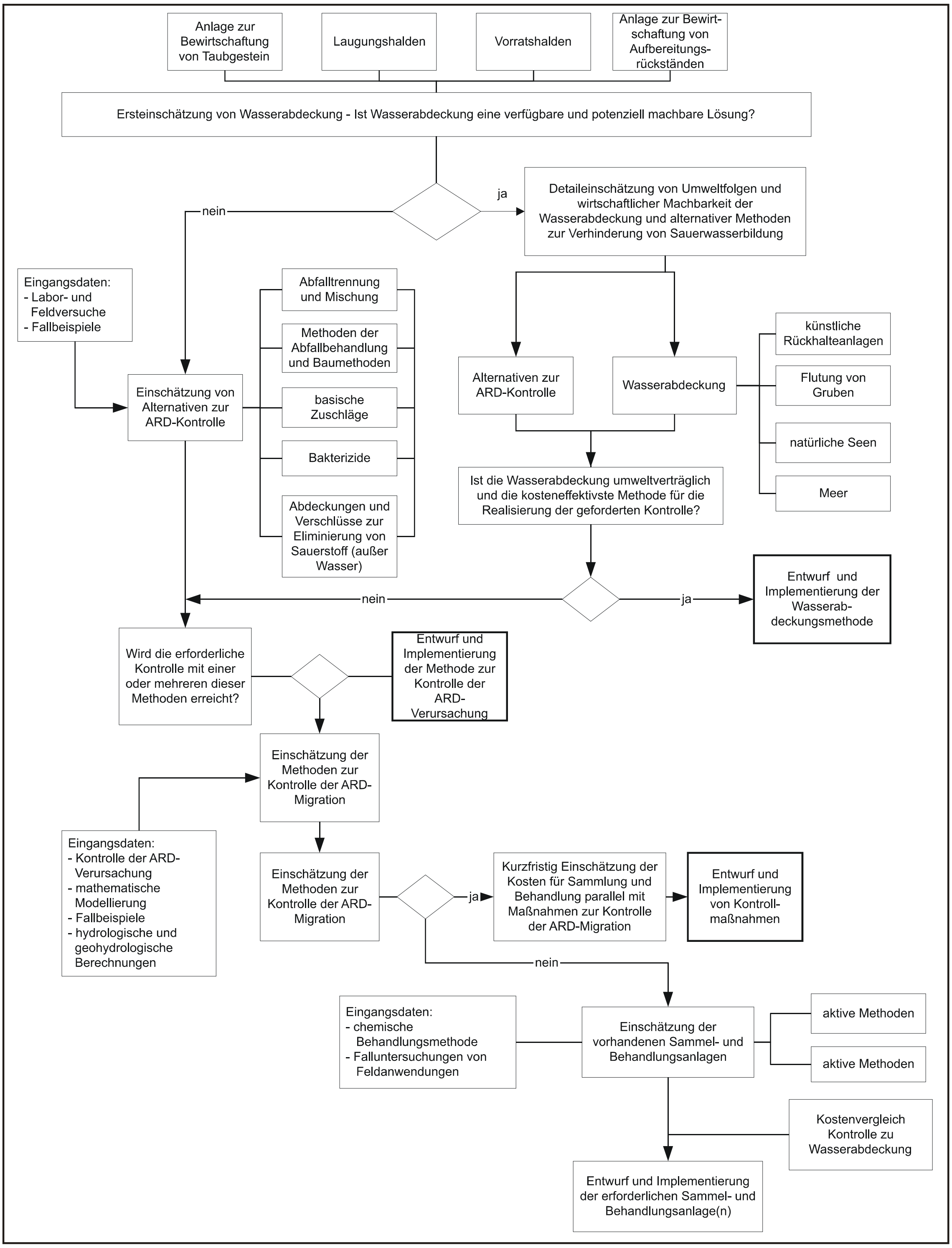

Abb. 4.9: Entscheidungsbaum für die Stilllegung einer potenziell ARD-verursachenden Anlage zur Bewirtschaftung von Aufbereitungsrückständen und Taubgestein
[20, Eriksson, 2002]

Je nach Mineralogie, physikalischen, chemischen und biologischen Eigenschaften, kann die Sulfidoxidation über einen längeren Zeitraum erfolgen. Dies wird bei der Auslegung von Bewirtschaftungseinrichtungen für poztenziell ARD-verursachende Aufbereitungsrückstände und Taubgestein berücksichtigt.

4.3.1.6 Die Bewirtschaftung von Sauerwässern in Talkum-Anlagen

Diese Thematik ist generell nicht relevant für Industrieminerale, ausgenommen die Talkumvorkommen in Finnland. In diesen speziellen Fällen besteht die ARD-verursachende Fraktion des Taubgesteins aus Schwarzschiefer. Normales Taubgestein mit Carbonatgehalt verursacht keine Sauerwässer.

Zur Verhinderung oder Reduzierung der ARD-Bildung werden in diesem folgende Techniken angwendet:

Selektive Bewirtschaftung von Taubgestein mit bzw. ohne ARD-Potenzial
Das Taubgestein ist meist carbonathaltiges minderwertiges Talk-Magnesitgestein oder Schwarzschiefer. Schwarzschiefer enthält ARD-verursachende Minerale (Sulfide). Bei der Errichtung von Taubgesteinaufschüttungen liegt ARD-Taubgestein in Carbonatgestein, welches als Puffer gegenüber den Sauerwässern des Schwarzschiefers wirkt. Taubgesteinaufschüttungen erfordern eine sorgfältige Planung mit langfristiger Berücksichtigung der Bewirtschaftung von ARD-Taubgestein bei möglichst niedrigen Kosten.

Reduzierung von Infiltration
Bei der Errichtung von Taubgesteinaufschüttungen werden die Böschungen geglättet und mit Moränenstoff aus lokalen Vorkommen abgedeckt. Das verringert die Erosion und fördert die Vegatationsbildung. Das Aufbringen einer Moränenabdeckung mit sorgfältig geplanter Erfassung von Oberflächenabfluss und Vegetation vermeidet den größten Teil der Regen- und Schneeschmelzwässer (75 %) beim Vordringen zum Taubgestein. Das Sickerwasser von den Aufschüttungen wird gesammelt und mit Kalk behandelt, sofern es noch sauer und metallhaltig ist.

Reduzierung der ARD-Bildung in Absetzbecken
Während des Betriebs der Absetzbecken wird der größte Teil der Aufbereitungsrückstände mit Klarwasser bedeckt, so dass sich die ARD-verursachenden Minerale (Sulfide) in einem oxidationsfreien Zustand befinden und damit Sauerwassersickerung nur in beschränktem Umfang erfolgt. Bei den Aufbereitungsrückständen handelt es sich zumeist um Magnesit (Mg-Carbonat), einen Pufferstoff, der im Becken sauerwasserfreie Bedingungen erzeugt. In manchen Werken befinden sich Aufbereitungsrückstände mit Sulfidanteilen aus alten Kupferminen unterhalb der jetzigen Magnesitschichten. Sulfidhaltige Schichten werden so ausgelegt, dass sie nach der Einstellung des Betriebs durch Abdeckung der Absetzbecken mit einer Trockenabdeckung aus lokalen Moränenvorkommen in einem stabilen Zustand bleiben. Regen und Schneeschmelzwässer sammeln sich im alten Becken, so dass der Grundwasserspiegel auf einem Niveau gehalten werden kann, das die Oxidation der alten sulfidischen Aufbereitungsrückstände verhindert. Sickerwässer aus Absetzbecken werden gesammelt und außerhalb des Beckenbereichs oder mit Pflanzenklärtechnologie behandelt.

Pflanzenklärbecken zur Behandlung von Sickerwässern aus Absetzbecken oder Taubgestein-aufschüttungen
Bei der Pflanzenkläranlagentechnologie (siehe Abschnitt 4.3.11.5) sammeln sich die Sickerwässer in einer Pflanzenkläranlage, die in einem nicht mehr genutzten Becken oder auf Feuchtgebieten in Nähe des Betriebsstandortes angelegt wird. Durch Einsatz neutralisierender Baustoffe (Carbonatgestein) und natürliche spezifische Vegetation werden die Metalle in den Sickerwässern ausgefällt und die Klarwässer können in lokale Flüsse/Seen eingeleitet werden.
[131, IMA, 2003]

4.3.2 Verfahren zur Senkung des Verbrauchs von Reagenzien

Es werden Anstrengungen zur Reduzierung der Menge an Zusatzreagenzien unternommen. Dies senkt die Kosten und dient der Umwelt. In vielen Fällen wird der Erzeinsatz ständig auf chemische Zusammensetzung überwacht, wodurch die Zugabe von Reagenzien automatisch optimiert werden kann.

Allgemein wird – soweit technisch und ökonomisch sinnvoll - der Einsatz von biologisch abbaubaren Chemikalien befürwortet. Bei den meisten Reagenzien ist ein Recycling wegen der starken Bindung an die Oberfläche der Teilchen nicht möglich [131, IMA, 2003].

4.3.2.1 Computergestützte Prozesssteuerung

Die computergestützte Prozesssteuerung ist ein Schlüsselfaktor für die Optimierung der Ausbeute bei Aufbereitung und Reagenzieneinsatz. Nach Einführung von Prozesssteuerungssystemen wurden Reduzierungen des Reagenzienverbrauchs bis zu 30% berichtet. Diese Steuerungssyteme erfassen alle relevanten Prozessdaten in einem Rechnersystem und zeigen sie auf Monitorbildschirmen in Leitzentralen und an anderen strategischen Stellen an. Die Steuerung kann vollautomatisch erfolgen, wobei die Dosierung der Chemikalien automatisch erfolgt oder sie ist halbautomatisch, wobei der Betreiber die Einstellungen der Dosierung von Chemikalien entsprechend der vom Rechner ausgegebenen Informationen vornimmt.

Vorteile:
- Der Prozess kann auf hoher Ebene automatsich gesteuert werden, wodurch eine Optimierung des Reagenzieneinsatzes möglich ist
- Prozesseinstellungen können unkompliziert vorgenommen werden.

Nachteile:
- hohe Investitionskosten
- Voraussetzung ist eine hohe Computerkompetenz des Betreibers.

Beim Flotationsprozess müssen regelmäßige Produktanalysen vorgenommen werden. Damit müssen die Dosierungsmengen unter Umständen mit sehr kurzem Zeitvorlauf angepasst werden. Es befinden sich mehrere Online-Analysegeäte auf dem Markt, die jedoch ausnahmslos bisher keine positiven Ergebnisse beim Einsatz von Industriemineralen gebracht haben.
[131, IMA, 2003]

Der Erfolg des Flotationsprozesses wird durch richtige Auswahl und korrekten Einsatz der Chemikalien bestimmt. Jede Reduzierung der vorgeschriebenen Chemikalien kann signifikante finanzielle Auswirkungen auf die Produktion haben. Es ist jedoch ebenso erforderlich, den Einsatz von Chemikalien aus ökonomischen und ökologischen Gründen auf ein Minimum zu reduzieren. Dazu werden häufig oder sogar kontinuierich die Erzqualitäten bestimmt, so dass die Reagenzienzugabe entsprechend angepasst werden kann. Neuere Techniken auf diesem Gebiet sehen den Einsatz von Kameras zur Online-Überwachung des Schaums auf den Flotationszellen vor. Zusammen mit Expertensystemen führt dies zur Optimierung der Prozessbedingungen und damit zu höheren Ausbeuteraten und einen günstigeren Einsatz von Reagenzien [69, Nguyen, 2002].

4.3.2.2 Betriebskonzepte zur Minimierung der Cyanidzugabe

Für die Minimierung der Cyanidzugabe werden folgende Betriebsstrategien angewendet:
- Schritte zur Reduzierung des Cyanidverbrauchs durch andere Komponenten, z. B. Kupferminerale, Pyrrhotin usw.
- Versuche zur Rückhaltung von Cyanid im Kreislauf anstelle der Freisetzung in das Absetzbecken. Dies kann – wenn möglich - durch Waschen der Aufbereitungsrückstände erreicht werden.

- Einsatz einer strengen Kontrolle der Wasserzugabe in den Kreislauf zur Reduzierung der Notwendigkeit des Abzugs der Lösung zur Aufrechterhaltung einer Wasserbilanz. In ariden Gebieten sind abflusslose Einrichtungen möglich.
- Strikte Überwachung der Cyanidkonzentration im Prozess und in den Aufbereitungsrückständen, um die Cyanidzugabe auf ein Minimum zu reduzieren. In manchen Anlagen sind Online-Analyseanlagen installiert worden (z. B. automatische Cyanidkontrolle, siehe unten). Diese Einrichtungen können mit automatischen Dosierungsanlagen für Reagienzien zusammen geschaltet werden.
- Verbesserung der Belüftung in der Laugung und/oder Zuführung von Sauerstoff oder anderen Oxidationsmitteln zur Maximierung der Lösungsrate.
- Einsatz einer Vorbelüftung (z. B. Einsatz von Wasserstoffperoxid, siehe unten) des aufgeschlämmen Erzes vor der Cyanidlaugung zur Oxidation der Cyanid verzehrenden Bestandteile, die anschließend eingedickt und dem Prozess entzogen werden können.
  [24, British Columbia CN guide, 1992]
- Nach Möglichkeit Einsatz der Schwerkrafttrennung und Laugung des Konzentrats aus diesem Prozess. Schwerkraftkonzentration ist heute bis zu einer Korngrößenstruktur von 30 µm anwendbar.

4.3.2.2.1 Automatische Cyaniddosierung

Bis etwa vor zehn Jahren war es allgemein üblich, Cyanid zur Cyanidlaugung ausschließlich manuell durch Einstellen eines Ventils zuzugeben. Damit kam es häufig zu Überdosierungen mit Verlusten von CN. Ein typischer Wert für den Canidverlust war 10 %, jedoch waren Werte bis zu 30 % möglich.

Das manuelle Verfahren hat zudem den Nachteil, dass Proben nur in Abstänen von mehreren Stunden gezogen werden, wodurch eine lange Zeit vergehen konnte, ehe die durchgeführte Einstellung wirkte. Außerdem wir die entnommene Probe manuell gefiltert und danach zur Bestimmung des optischen Endpunktes mit Silbernitrat manuell titriert, wobei das Ergebnis aber dennoch wegen des subjektiven Faktors der Betreiberintervention fehlerhaft sein kann.

Mit Einführung der automatischen Cyaniddetoxifikation ist es möglich, Proben aller etwa 5 - 15 min zu nehmen und die Cyanidkonzentration entsprechend automatisch und sofort auf den Sollwert einzustellen. Auf diese Weise ist es häufig möglich, bei gleicher Goldausbeute 10 – 20 % der Cyanidmenge gegenüber dem manuellen Betrieb einzusparen.

Zahreiche kleine Goldminen arbeiten noch immer mit manueller Dosierung, da eine bestimmte kritische oder Schwellenkonzentration von ca. 500 t NaCN jährlich für die Ökonomie des Betriebs erforderlich ist. Oberhalb dieser Schwelle rechnet sich diese Methode jedoch für die meisten Betreiber.

Kurz gesagt, hat die Methode folgende Vorteile:
- Einsparung von CN
- niedrigere Kosten für die CN-Zerstörung.

In Rio Narcea (El Valle) wird die automatische Cyaniddosierung eingesetzt.

Die Kosten für ein derartiges automatisches System betragen ca. EUR 100 000, hängenjedoch von der Betriebsgröße ab.

4.3.2.2.2 Vorbehandlung mit Peroxid

Obwohl nicht generell einsatzbar, besitzen zahlreiche Erze im aufgeschlämmten Zustand (häufig, jedoch nicht ausschließlich, Sulfiderze) stark reduzierende Eigenschaften mit dem Ergebnis, dass die übliche Belüftung oder Sauerstoffzufuhr für eine ausreichende Menge an gelöstem Sauerstoff und/oder die Oxidationseigenschaften für die Oxidation von Gold nicht ausreicht. Dies ist bei der Cyanlaugung erforderlich, da ansonsten eine Laugung des Goldes mit Cyan nicht möglich ist oder der Vorgang mit extrem geringer Geschwindigkeit abläuft.

Erfolgt die Belüftung mit Wasserstoffperoxid (H_2O_2) statt Luft oder Sauerstoff, kann die Goldausbeute verbessert werden. Eine positive Nebenwirkung ist die Reduzierung des Cyaneinsatzes, da die Sulfide geringere Mengen davon verbrauchen.

Diese Methode ist allgemein anwendbar bei sulfidhaltigen Erzen. Jedoch bedarf es einer ausführlichen mineralogischen Untersuchung mit Labortest zur Bestimmung des für diese Behandlung geeigneten Erzes.

Der Verbrauch von Wasserstoffperoxid beträgt häufig ca. 1 kg H_2O_2/t behandeltes Erz. Die Kosten für H_2O_2 liegen um EUR 600/t (70 %) H_2O_2.

Die Investitionskosten für die Behandlungsanlage betragen ca. EUR 100 000, können jedoch mit Durchsatz, Wasserstoffperoxidverbrauch und Mineralogie des Erzes starken Schwankungen unterliegen.

4.3.2.3 Vorsortierung

Durch Vorsortierung des Aufgabegutes in die Aufbereitungsanlage entweder manuell (unbewaffnetes Auge) oder mit optischen Mitteln ist es möglich, bestimmte Fraktionen, die sich für die weitere Aufbereitung nicht eignen, zu eliminieren. Diese grundlegenden Methoden sind in der Verarbeitung von Industriemineralen weit verbeitet. Dazu haben diese Methoden keinen Einfluss auf die Umwelt und können wenig kostenaufwendig sein. Die eliminierten Fraktionen können häufig beim Bau von Dämmen für Aufbereitungsrückstände oder als Baustoffe Verwendung finden. Die Wahl zwischen manueller und optischer Sortierung hängt von der Beschaffenheit des Erzes ab.

4.3.3 Verhinderung von Wassererosion

Wassererosion von Aufbereitungsrückständen oder Taubgestein in der Betriebsphase von Bewirtschaftungseinrichtungen können durch folgende Techniken verhindert werden:

- Abdecken der Hangflächen der Rückhalteeinrichtung mit einer Schutzschicht, z. B. Kies, einer Decke aus Erdreich oder Gras, Geotextil oder Grasnarbe oder eine synthetische Abdeckung
- Impregnierung der Oberflächenschicht der Aufbereitungsrückstände mit einer Chemikalie, die Wasser abweisen kann oder Partikel bindet, z. B. eine Siliciumverbindung, Zement, Bitumen oder Bentonit
- Nutzung der chemischen Eigenschaften der Aufbereitungsrückstände, z. B. der sulfidhaltigen, zur Verbesserung des Bindungsverhaltens.

4.3.4 Verhinderung von Staubbildung

Die nachfolgende Tabelle nennt Möglichkeiten zur Entsorgung **fester** Aufbereitungsrückstände von Dämmen oder Aufschüttungen und Optionen zur Verhinderung dieser Vorgänge.

Verbreitung der Feststoffe möglich durch:	Gegenmaßnahme:
Winderosion der Oberflächen des Rückhaltebauwerks: ▪ Dammkrone/Aufschüttung ▪ Böschungen von Dämmen/Aufschüttungen ▪ Oberfläche der Spülränder	▪ Dammkrone und Böschungen könenn wie bei Wassererosion behandelt werden ▪ Oberfläche kann Windbrecher, Sprühen von Wasser, Anwendung von Bindestoffen, d. h. Aufsprühen von Bitumenemulsion [8, ICOLD, 1996], Oberflächenmulch [11, EPA, 1995], Kalkschlempe erfordern ▪ In extremen Fällen müssen Aufbereitungsrückstände unter Wasser deponiert werden ▪ Oberflächenbewuchs, schwimmend oder auf inaktiven Flächen ▪ häufiger Wechsel der Einleitungsstellen um den Rand zur Sicherung der ständigen Befeuchtung der Oberfläche [11, EPA, 1995].

Tabelle 4.9: Dispersion von festen Aufbereitungsrückständen durch Winderosion aus Bewirtschaftungsanlagen für Aufbereitungsrückstände und Taubgestein und mögliche Gegenmaßnahmen

4.3.4.1 an Spülrändern

Zur Minimierung der Staubentwicklung an Spülrändern wird die Fläche gewöhnlich feucht gehalten. Beispielsweise wird bei drohender Staubbelastung **Wasser** auf Rotschlamm gesprüht. Das ist kosteneffektiver als das Aufbringen einer verrottenden Vegetation, z. B. Heu, auf den Rotschlamm. Abdeckungen, wie z. B. Heu, behindern die optimale Reifung der Rotschlammablagerungen. Am Standort Aughinish sind Sprinkleranlagen über die gesamte TMF verteilt und werden mit dem Füllstand der Aufbereitungsrückstände angehoben. Eine solche Lösung kann nur angewendet werden, wenn die Aufbereitungsrückstände für Fahrzeuge zugänglich sind, d. h.. bei eingedickten Aufbereitungsrückständen.

Sprinklerbehandlung des Spülrandes kombiniert mit der ständigen Bewirtschaftung der Einbringungsstelle der Aufbereitungsrückstände auf den Spülstrand ist in der Regel ausreichend. Sprinklerbehandlung wird meist nur bei eingedickten Aufbereitungsrückstandes angewendet.
Vorteile:

- Es kann Wasser aus der TMF verwendet werden
- kein hoher Kostenaufwand.

Nachteile:

- Frostprobleme in kalten Klimaten
- arbeitsaufwändig.

Eine weitere Methode zur Vermeidung von Staubemission ist die **Abdeckung des Spülrandes** mit staubfreien Stoffen, z. B. Mutterboden, Ligninverbindungen, Stroh oder Bitumen. Diese Methode eignet sich nur für Spülränder, die abschnittweise, nicht aber kontinuierlich erhöht werden. Der Spülrand muss eine ausreichende Stabilität für den Geräteeinsatz zum Aufbringen der Stoffe haben. Andernfalls stehen alternative kostenintensive Methoden, z. B. Einsatz von Hubschraubern zum Aufbringen der Stoffe, zur Verfügung. Die Abdeckung mit

Vegetationsdecken, z. B. Baumrinde oder Heu, kann sehr effektiv sein, behindert jedoch die Reifung der Bestände an Aufbereitungsrückständen. Die Technologie des Aufbringens dieser Stoffe auf sehr weiche, aber reifende Aufbereitungsrückstände ist bei Entwicklung und Durchführung sehr kostenintensiv.

Vorteile:
- Sind die Stoffe einmal eingebacht, ist das Problem der Staubemissionen auf lange Zeit gelöst.

Nachteile:
- Eine kontinuierliche Erhöhung der Spülränder ist nicht möglich.
- Beim Erhöhen des Damms muss eventuell das staubfreie Material entfernt werden.
- Der Spülrand muss eine ausreichende Stabilität für den Geräteeinsatz zum Aufbringen der Stoffe haben.

[118, Zinkgruvan, 2003]

Beim Absetzbecken im **Kupferrevier Legnica-Glogow** wird der Wasserspiegel im Abetzbecken konstant im Abstand von mindestens 200 m von der Dammkrone gehalten. Der Spülrand ist eine erhebliche Quelle von Staubemissionen, besonders an windreichen Tagen. Zur Verringerung der Staublast wurde ein 'Wasservorhang' auf der Krone errichtet. Zusätzlich wird zur Stabilisierung der Oberfläche in zeitweise trockenen Abschnitten eine **Asphaltemulsion** vom Hubschrauber aus aufgesprüht. Gegenwärtig werden zusätzliche 'Wasservorhänge' erprobt. Diese werden innerhalb des Beckens auf dem Spülrand im Abstand von 150 m aufgestellt und in Betrieb genommen, wenn ein trockener Abschnitt (nach Abtragen der Asphaltdecke) für den Dammbau eingesetzt wird.

In **Pyhäsalmi** wird zur Verhinderung der Winderosion feiner Partikel aus den Aufbereitungsrückständen das **Sprühen von Kalkschlempe** eingesetzt. Die Substanz wird mit Geräten versprüht, die ursprünglich für den Einsatz in der Landwirtschaft gedacht waren. Die Anlage besteht aus einem auf einen Traktor montierten Tank sowie Pumpe und Schläuchen. Mit diesem Gerät kann Kalkschlempe in mehr oder weniger gleichmäßigen Lagen auf die Zielflächen aufgebracht werden. Beim Trocknen bildet der Kalk eine feste Oberfläche, die die gesamte niederschlagsarme Sommerperiode hindurch hält. Nach durchgeführten Sichtprüfungen hat diese Methode zu einer signifikanten Verringerung der Staubbildung geführt. Zuverlässige Daten als Nachweis des erreichten Nutzens liegen jedoch dazu nicht vor.

Es ist anzumerken, dass die Versprühung der Kalkschlempe in Pyhäsalmi ausschließlich der mechanischen und physischen Vermeidung der Staubbildung dient und keine chemischen Vorgänge verhindert (d. h. Neutralisierung von Sauerwässern). Mit verbesserter Technik könnten die Ergebnisse homogener und von besserer Wirksamkeit sein. Die Kosten dieses Verfahrens betragen ca. EUR 1500/ha, was im Vergleich zur geforderten Fläche (5 – 6 ha) und der Notwendigkeit der jährlichen Wiederholung (im Frühjahr) vergleichsweise hoch ist.

Eine organisatorische Möglichkeit zur Reduzierung/Verhinderung der Staubemission ist der häufige Wechsel der Einbringungsstellen entlang des Randes, um die Oberfläche ständig feucht [11, EPA, 1995] oder die Aufbereitungsrückstände ständig mit Wasser bedeckt zu halten (siehe Abschnitt 4.3.1.2.1 und 4.3.1.2.3).

4.3.4.2 an Böschungen

Eine Möglichkeit zur Vermeidung der Staubentwicklung an Dammböschungen besteht darin, die Böschungen mit grob zerkleinertem Taubgestein zu belegen.

Vorteile:
- Kostengünstig, wenn der Betrieb über überschüssiges Taubgestein verfügt.
- Dammstabilität wird durch die zusäzliche Masse des Taubgesteins erhöht.

Nachteile:
- Zusätzliche Kosten für Brechen und Einbbau.
  [118, Zinkgruvan, 2003]

4.3.4.3 beim Transport

Gewöhnlich erfolgt der Transport von Aufbereitungsrückständen und taubem Gestein über Rohrleitungen (nur verschlämmte Aufbereitungsrückstände), Bandanlage oder Lkw. Bei Transport (von aufgeschlämmten Aufbereitungsrückständen) kommt es zu keiner Staubemission.

4.3.4.3.1 per Band

Die folgende Tabelle enthält verschiedene Ansätze für die Reduzierung der Staubemissionen in Kalibetrieben, in denen Aufbereitungsrückstände (mit Bandanlagen) transportiert und auf Halden abgelagert werden.

Ansatz		Reduzierung der Emission
Primärmaßnahmen		• Einsatz von Aufbereitungstechnik, die möglichst wenig Feingut erzeugt • Sprühen der Aufbereitungsrückstände
Sekundär-maßnahmen	organi-satorisch	• kontinuierliche Aufbereitung • Verkürzung der Transportentfernungen • Behandlung möglicher Quellen von Lärmemissionen • Logistik von Einspeicherplätzen
	technisch	• Einsatz von Windschutzvorrichtungen (z. B. Abdecken der Bandanlage) • Reduzierung der Abwurfhöhe auf ein Minimum • Quer-/Rückwärtsband • Befeuchtung der festen Aufbereitungsrückstände
Tertiäre Maßnahmen		• Kein Abwerfen bei hohen Windgeschwindigkeiten

Tabelle 4.10: Möglichkeiten der Reduzierung von Staubemissionen beim Transport

In den deutschen Kalibetrieben werden feste Aufbereitungsrückstände aus der elektrostatischen Abscheidung in geschlossenen Räumen befeuchtet. Die Aufbereitungsrückstände werden auf Bandanlagen gefördert und mit einem Feuchtegehalt von ca. 5 – 6 % abgelagert. Dies führt zu geringen Staubemissionen infolge der Rekristallisation der Oberflächenschicht. Die einzige Luftbelastung entsteht durch Salzstaub beim Abwurf der Aufbereitungsrückstände auf die Abraumhalde, besonders vom Föderband auf die Halde bei sehr starken Winden. Um dies zu vermeiden, wird der Abwurf umgehend eingestellt, wenn die Windgeschwindigkeit einen bestimmten Grenzwert überschreitet. In den vergangenen Jahren lag der maximale Messwert der Staubemission mehrerer Emissionsmessstellen (Staubberwachung und Kontrollsystem) im Bereich der Abraumhalden unter 60 mg/m^2/d.

Übergabestationen werden meist gekapselt ausgeführt und die Luft in Filtern behandelt [131, IMA, 2003].

4.3.4.3.2 per Lkw

Zur Staubunterdrückung werden meist mehrere Methoden angewendet, z. B.:
- Besprühen der Schaufel/Kübel des Laders beim Laden
- Besprühen der Schaufel des Lkw

- Befeuchten der Straßen, Sprühen beim Entladen
- Direktes Besprühen der Lkw mit Wasser und/oder Sprinkler entlang des Fahrweges [131, IMA, 2003]
- Festlegen einer Geschwindigkeitsbegrenzung auf 30 km/h [142, Borges, 2003].

Bei Konzentrattransporten müssen die Fahrzeuge zur Reinigung der Reifen häufig Wasserstrecken durchfahren, in einigen extremen Fällen werden die Fahrzeuge vor dem Verlassen der Verladestelle gewaschen.

In Rio Narcea wurde eine Reihe von Staubbeprobungsanlagen für Überwachungszwecke entlang des Randes der Grube von El Valle aufgestellt. Die dabei erfassten Daten werden monatlich ausgewertet. Dieses Überwachungssystem arbeitet parallel zum bestehenden Gesundheits- und Arbeitsschutzprogramm, in dem bereits eine Staubüberwachung durch individuelle Beprobung vorgesehen ist.

4.3.5 Verfahren zur Senkung der Lärmemissionen

Die häufigsten Quellen für Lärmemissionen sind Transport, Abwurf und Verteilung bei Einsatz von Lkw und Förderbandanlagen.

In Zinkgruvan wurden ca. 0,5 Mio t Taubgestein auf eine Fläche nahe der alten Grube als Lärmschutz im östlichen Teil des Industriegebietes abgesetzt.

In den Gruben im Ruhrgebiet und an der Saar werden Rampen und Arbeitsstrossen zur Minimierung von Staub- und Lärmemission durch Transport, Abwurf und Verteilung der Aufbereitungsrückstände soweit wie möglich in den Innenbereich der Halde verlegt [79, DSK, 2002].

In manchen Fällen wird zuerst eine Außenböschung angelegt, um Lärm, Staub und den Einsatz der Geräte außer Sichtbereich der Umgebung zu legen, da generell von unsichtbaren Quellen eine geringere Wirkung ausgeht. Bei dieser Methode wird zuerst die Aufschüttung außen so behandelt, dass eine rasche Vegetation erfolgt, die dann als entsprechender Lärmschutz dient. Betroffenen Nachbarn zufolge ist das akustische Warnsignal beim Rückwärtsfahren der Dumper der störendste Lärm.
[131, IMA, 2003]

Diese Methode wird in Abb. 4.14 dargestellt.

In den deutschen Kaligruben erfolgen Verarbeitung und Bewirtschaftung der Aufbereitungsrückstände in einem kontinuierlichen Prozess. Der Transport zur Abraumhalde erfolgt auf Bandanlagen. Diese Lösung ist mit weniger Lärmbelästigung verbunden als der Transport per Lkw.

Kontinuierlich arbeitende Anlagen sind nicht immer möglich oder praktisch im Einsatz. Bei der Bewirtschaftung von Taubgestein, besonders bei großen Gesellschaften, gibt es verschiedene Abwurfstellen, so dass Lkw häufig die einzig praktikable Lösung sind.

Die Fahrzeuge verlangen eine entsprechende Wartung, damit sie sich stets in einem einsatzbereiten Zustand befinden.

Bandantriebe sind in der Regel gekapselt [19, K+S, 2002].

4.3.6 Progressive Rekultivierung/Wiederurbarmachung

Die progressive Rekultivierung/Wiederurbarmachung während der Betriebsphase hat folgende Vorteile:

- Verteilung der Kosten über einen längeren Zeitraum mit der Möglichkeit der Deckung aus den Erlösen der Grube
- Stilllegungaktivitäten können in den täglichen Betriebsablauf der Grube integriert werden.
- Die Implementationsphase der Stilllegung verkürzt sich.
- Überwachungsprogramme werden in das normale Umweltmanagement integriert.
- Erfolgreiche Techniken können in den endgültigen Stilllegungsplan integriert werden.
- Negative Umweltfolgen werden minimiert.

Es ist nicht möglich, die progressive Rekultivierung für alle Flächenfunktionen in einem einzigen Produktionsstandort durchzuführen. Beispielsweise wäre das der Fall, wenn an einem Standort Reifung und Konsolidierung der Aufbereitungsrückstände beschleunit werden sollen, besonders wenn der umlaufende Damm/Deich nach der Upstream-Baumethode errichtet werden soll.

Aufschüttungen werden häufig fortschreitend wieder urbar gemacht: Das hat den zusätzlichen Vorteil der Verminderung der Erosion. Zum Beispiel werden in der Talkumgrube in Finnland die Taubgesteinshalden und Absetzbecken fortschreitend mit lokal entnommenem Moränenstoff abgedeckt und Vegetation angelegt [131, IMA, 2003].

Werden Aufbereitungsrückstände auf Aufschüttungen deponiert, können diese in horizontalen Lagen aufgebaut werden. Das ermöglicht dem Betreiber die unmittelbare Gestaltung der abschließenden Böschungen nacheinander, so dass Staubemissionen vermieden werden. Rekultivierung/Wiedergewinnung erfolgen entsprechend der später vorgesehenen Nutzung der Fläche, der in umliegenden Gebieten vorhandenen Vegetation und den Bedürfnissen der Anwohner. Ziel ist, dass durch eine rasche Erstbesiedelung mit Pionierarten (Gräser, Gebüsch, Bäume) die Staubbildung erfolgreich verhindert wird, wertvolle Biotope für unterschiedliche Pflanzen- und Tierarten entstehen und die Kosten für den Betreiber in einem vertretbaren Rahmen bleiben.
[131, IMA, 2003]

Die laufende Wiederurbarmachung bereits in der Betriebsphase kann durch verschiedene Maßnahmen beschleunigt werden:

- Loses Verkippen von Aufbereitungsrückständen bis zu einer Tiefe von 2m im äußeren Bereich als Voraussetzung für gute Durchwurzelung
- Mischen mit anderen Stoffen, z. B. Flugasche aus Kraftwerken, Kalk und Dolomit. Auf diese Weise können Pufferwirkung, Wasserrückhaltevermögen und Nährstoffangebot verbessert werden.
- Aufbringen von 5 bis 10 cm Kulturboden. Zur Förderung einer raschen und dauerhaften Vegetation wird bevorzugt eine dicke Erdschicht (ca. 1,8 m, wenn die Beschaffenheit der Aufbereitungsrückstände dies erfordert) oder einer dünnen Lage (5 - 10 cm) aufgebracht. In den meisten Fällen stehen die entsprechenden Erdstoffe in ausreichender Menge zur Verfügung. Damit wird die Wurzelbildung der Pflanzen unterstützt und Büsche können direkt in die Aufbereitungsrückstände gepflanzt werden. Das hat den Vorteil, dass sich die Jungpflanzen an die Bodenbedingungen in den Aufbereitungsrückständen selbst gewöhnen und eine natürliche Bewurzelung stattfindet, die die Pflanzen auch in Trockenperioden ausreichend mit Feuchtigkeit versorgt.
- Ausbringen von mineralischen Düngern als Ausgleich des herrschenden Nährstoffmangels. Organischer Dünger enthält Nährstoffe in organischer Bindung, die jedoch durch mikrobiellen Abbau freigesetzt werden. Ferner verbessern sie die Bodenstruktur, aktivieren Bodenorganismen und verbessern das Wasserrückhaltevermögen.
- Aufbringen von Oberflächenmulch zur Verbesserung des Schutzes bei ungünstigen Klimabedingungen sowie zur Humusanreicherung und Verbesserung der Wasserrückhalte-

leistung, besonders in den ersten Phasen der Vegetation. Als Mulchmaterial können Stroh oder Heu, aber auch Holfschnitzel verwendet werden.

- Bei extremer Trockenheit Beregnung nur in den Nachtstunden
[79, DSK, 2002]
- Verwendung entsprechend behandelter Stoffe, z. B. Klärschlamm, Rinde, organische Abfälle und/oder Aschen mit ausreichend großer Pufferleistung und hohem Mineralanteil zur Unterstützung in der Anfangsphase der Wiederurbarmachung. Diese Stoffe wurden mit Erfolg an verschiedenen Standorten, z. B. Garpenberg und Falun, eingesetzt. Wichtig ist, dass Abfälle, wie z. B. Klärschlamm oder Bioabfälle, zur Minimierung von Krankheitskeimen erst nach entsprechender Behandlung eingesetzt werden. Die Einhaltung von EU- und Landesvorschriften für den Einsatz dieser Abfälle ist zu gewährleisten. Bei Verwendung von Klärschlamm ist die hohe Belastung mit Schwermetallen im Schlamm zu beachten.

In manchen Gebirgsgegenden werden Aufschüttungen durch Verkippen am Berghang unter Nutzung des natürlichen Schüttwinkels errichtet (siehe Abb. 4.10). In diesem Fall kann die Rekultivierung des Hangs erst nach Abschluss der Verkippungen erfolgen.

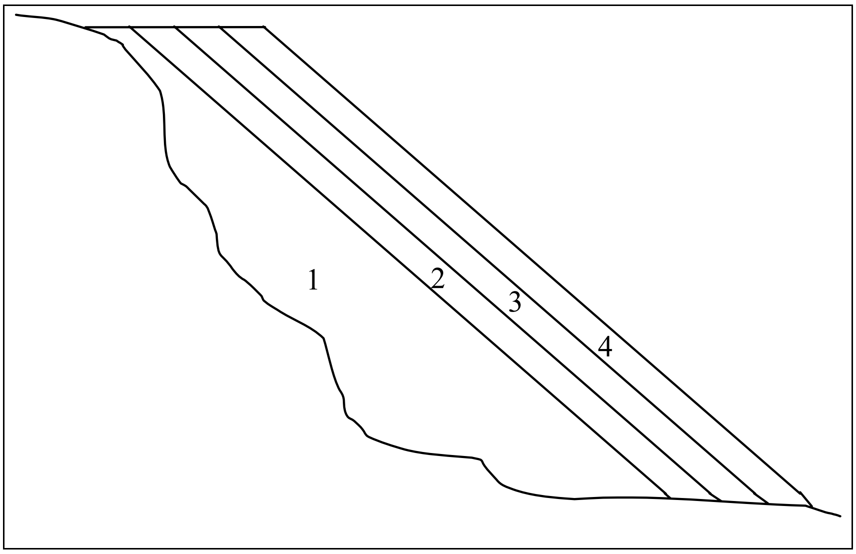

Abb. 4.10: Beispiel für Hangverkippung
[131, IMA, 2003]

Abb. 4.10 zeigt, wie durch kontinuierliches Verkippen (Schicht 1, Abdeckung durch Schicht 2, Schicht 2 mit Abdeckung durch Schicht 3 usw.) am Hang die Rekultivierung während des laufenden Betriebs verhindert. Die Wiederherstellung des Hangs ist kostenintensiv und der Platzbedarf für die Herstellung sicherer Verhältnisse, Ermöglichung der Wiederurbarmachung usw. erheblich größer. Eine weitere Option ist das Schütten in Stufen, deren Breite die Bearbeitung des Hangs jeweils einer Stufe erlaubt. Auf diese Weise erfolgt die Einbringung des Stoffs bereits etwa am Ort des endgültigen Verbleibs (siehe Abb. 4.11).

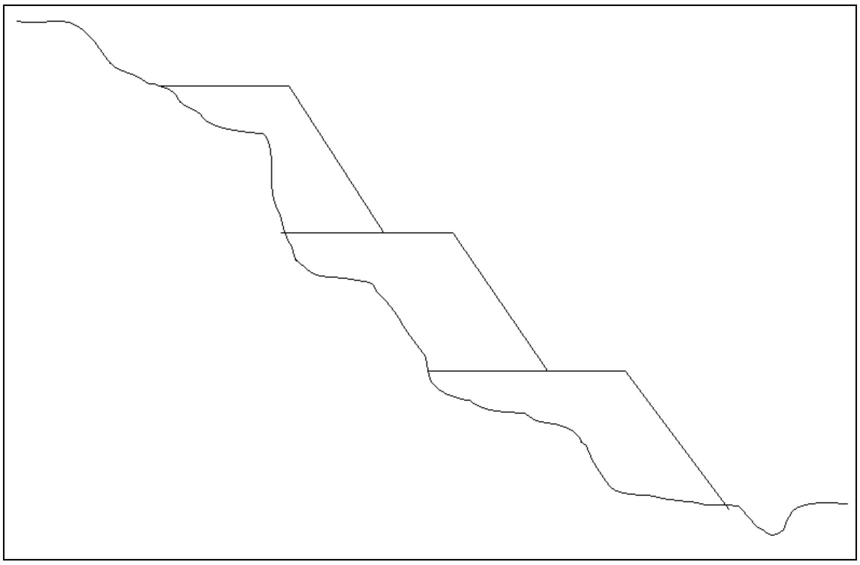

Abb. 4.11: Beispiel einer alternativen Hangverkippung

Generell erfolgen zur Vermeidung von Schäden an Dämmen und Aufschüttungen Rekultivierung und Wiederurbarmachung entsprechend der Stabilität der Dämme. Daher müssen Auslegung und Konrolle durch Fachkräfte erfolgen, da Vegetation potenziell zu ernsten Stabilitätsproblemen führen kann (d. h. Wurzeln können zur Zerstörung des Dammbauwerks führen). Auch muss bei der Wiederurbarmachung die Möglichkeit der Überwachung, z. B. durch Vermessung, jederzeit in Betracht gezogen werden.

4.3.7 Wasserbilanzen

Die Erstellung einer umfassenden Wasserbilanz ist für die Auslegung von Absetzbecken, Grubenstandorten und nachbergbauliche Aktivitäten von Bedeutung. Für die Wasserbilanz sind u. a. folgende Punkte zu beachten:

- Austragleistung des Beckens
- benötigter Freibord (wenn das Wasser aus dem Becken nicht direkt in das aufnehmende Bauwerk fließt)
- benötigte Wasserbehandlungskapazität
- für den Grubenbetrieb benötigte ausreichende Wassermenge und die Wasserqualität
- Behandlung von Überschusswasser
- vom System nicht erfasste Wassermenge (Sickerung durch taubes Gesteins und Aufbereitungsrückstände an die Oberfläche bzw. in das Grundwasser).

Bei Stilllegung wird die Wasserbilanz für die Umsetzung des Stilllegungsplanes und für die Bestimmung der Elementemasse aus der TMF bewertet. Einige Komponenten der Wasserbilanz für eine TMF sind in Abb. 4.11 dargestellt. Dazu muss die Wasserspeicherkapazität des Dammbaustoffs bestimmt werden.

Die folgende Abbildung zeigt einen Schnitt durch einen Damm für Aufbereitungsrückstände und verdeutlicht den Wasserkreislauf für diese Art von TMF.

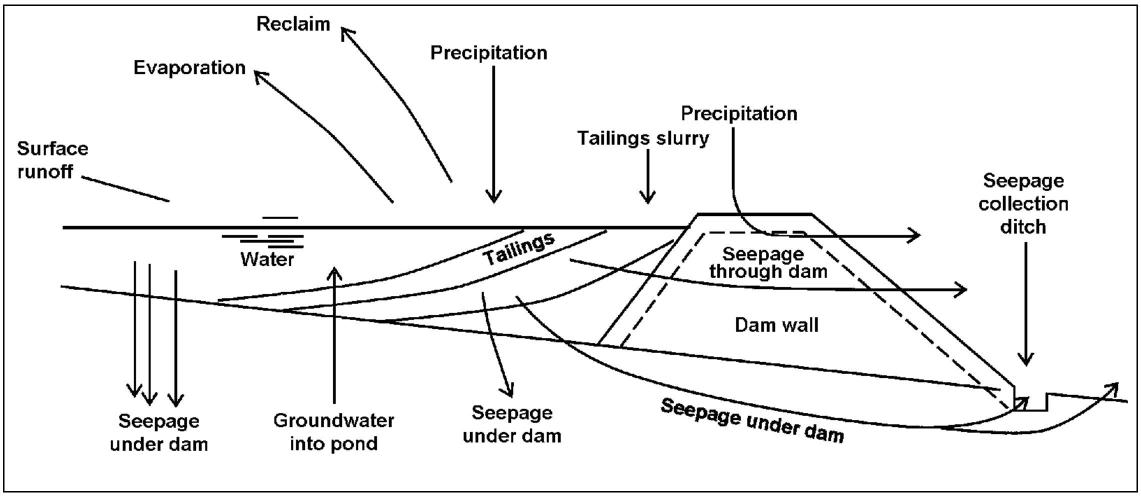

Abb. 4.12: Wasserkreislauf im Damm
nach [11, EPA, 1995]
Surface runoff – Oberflächenablauf
Evaporation – Verdunstung
Reclaim - Rückgewinnung
Tailings slurry – aufgeschlämmte Aufbereitungsrückstände
Precipitation – Niederschlag
Water – Wasser
Tailings – Aufbereitungsrückstände
Seepage through Damm – Sickerung durch Damm
Seepage collection ditch – Sammelgraben für Sickerwasser
Dam wall – Dammkörper
Seepage under dam – Sickerung unter Dammsohle
Groundwater into pond – Grundwasser in das Becken

In den schwedischen Eisenerzgruben wurden Wasserbilanzberechnungen für Dammsysteme für Aufbereitungsrückstände durchgeführt. Dabei wurden berücksichtigt:

- Niederschlag
- Oberflächenabfluss
- Prozesswassermengen
- Rückgewinnung von Prozesswasser
- Verdunstung
- Einleitung in das Flusssystem
- Sickerung unter der Dammsohle und durch den Damm.

Auf der Grundlage der Wasserbilanz konnte der geschätzte Ablauf aus Absetzbecken/Damm in das Grundwasser berechnet werden. Jedoch ist dieser Wert mit einer gewissen Unsicherheit behaftet, da mehrere Parameter nicht messbar sind und daher geschätzt werden müssen.

Weitere Beispiele für Wasserbilanzen können Abb. 3.25, Abb. 3.26, Abb. 3.27, Abb. 3.43, Abb. 3.44 entnommen werden.

4.3.8 Entwässerung von Absetzbecken

Am Standort Ovacik wurden die Sohle des Absetzbeckens und die Dämme mit Hilfe eines Verbundsystems aus 50 cm verdichtetem Ton, darüber eine Lage hochdichtes Polyethylen (HDPE) 1,5 mm als Geomembran, nochmals 20 cm verdichteter Ton und 20 cm Kieslfilterschicht undurchlässig gemacht. Hier werden Drainagerohre in die Filterschicht gelegt, die das Wasser zum Abfluss leiten. Abb. 4.13 zeigt den Aufbau dieser Verbundsystems.
[56, Au group, 2002]

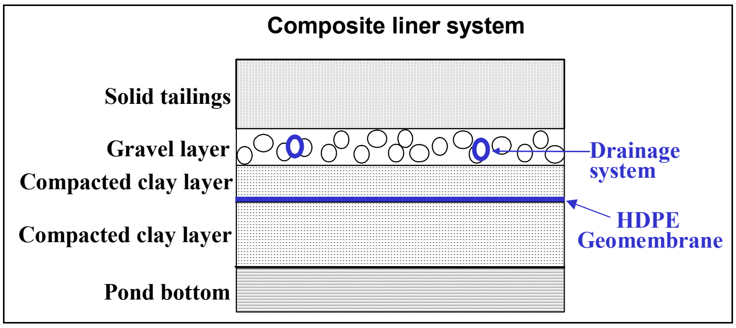

Abb. 4.13: Verbunddichtungssystem am Standort Ovacik
[56, Au group, 2002]
Composite liner system – Verbunddichtungssystem
Solid tailings – feste Aufbereitungsrückstände
Gravel layer – Kiesschicht
Compacted clay layer – verdichteter Ton
Pond bottom – Sohle des Absetzbeckens
Drainage system – Drainagesystem
HDPE Geomembrane – HDPE-Geomembran

Dieser Aufbau eignet sich für kleine, undurchlässige Becken, bei denen das Prozesswasser im Kreislauf geführt wird. Der Vorteil dabei ist, dass das Wasser bei Eintritt in das Drainagesystem gefiltert wird. Die Alternative wäre eine größere Klärfläche. Damit ließe sich mit diesem System die Größe des Absetzbeckens reduzieren.

Diese Lösung kann gegenüber einem zusätzlichen Klärbecken oder einem größeren Absetzbecken bevorzugt werden, wenn das Prozesswasser mit Schadstoffen (z. B. Cyan) belastet ist.

Die Kosten einer solchen Drainageanlage sind jedoch hoch. Im Falle von Ovacik betrugen die Kosten für den Einbau der HDPE-Geomembran EUR 7,50 /m^2 (Jahr 2001) für eine Fläche von 16 ha (siehe **Table 3.63**).

Weitere Nachteile sind die fehlende Instandsetzungsmöglichkeit bei Verstopfung der Drainage und dass die kleinere Fläche zu Lasten höherer Dämme geht.

4.3.9 Bewirtschaftung von Klarwasser

Wird das Klarwasser im Absetzbecken nicht direkt in natürliche Wasserläufe eingeleitet, müssen Vorkehrungen getroffen werden, die sichern, dass das gesamte Klarwasser in die Anlage zurückgeführt wird oder in trockenen, heißen Gebieten verdunstet. Das Abflusswasser kann in einem Klär- oder Entnahmebecken unterhalb des Absetzbeckens gespeichert werden und muss in manchen Gebieten vor Einleitung in natürliche Gewässer behandelt werden.

4.3.10 Bewirtschaftung von Sickerwasser

Die Grundlagen des Sickerwasserstroms sind in Abschnitt **2.4.2.5** beschrieben.

Eine Voraussetzung für die Auslegung der Bewirtschaftung von Sickerwasser sind umfassende Kenntnisse der hydrogeologischen Bedingungen des Standorts. Dazu werden im Normalfall zur Überwachung von Strom, hydraulischem Druckgefälle und Beschaffenheit der Grundwasser führenden Schicht Piezometer angeordnet und kontrolliert. Nach Auswertung der Daten können Entscheidungen über durchzuführende Maßnahmen getroffen werden.

Sickerwasser durch den Damm wird in Gräben gesammelt, wo Strömungsgeschwindigkeit und Qualität überwacht werden. In der Regel fängt der Graben auch den Abfluss in den Untergrund ab.
Ist das **Sickerwasser in den Untergrund (oder unter der Dammsohle)** von guter Qualität, kann es im Untergrund versickern. Andernfalls sind Bestimmung der Grundwasserqualität, Hebung und Behandlung des Wassers erforderlich. Die wichtigste Maßnahme zur Vermeidung von Sickerung in den Untergrund und in das Grundwasser ist die Bestimmung eines geeigneten Ortes, d. h. einer, an dem das Grundwasser in das Becken statt aus diesem heraus fließt. In diesem Falle sind die hydraulischen Bedingungen der Vermeidung von Infiltration in das Grundwasser erfüllt. Weitere Möglichkeiten der Bewirtschaftung von Sickerwasser in den Untergrund sind entweder der Versuch der vollständigen Abdichtung des Bodens mit Tondichtung oder einer synthetischen Membran oder einer Kombination beider Maßnahmen. An einigen Standorten ist das Vorhandensein natürlich vorkommender Tonschichten ausreichend für die wirksame Verhinderung der Sickerung in den Untergrund. Abdichtungen werden immer häufiger angwendet. Kritiker verweisen jedoch auf den ‘Badewanneneffekt’ als zu beachtendes Langzeitproblem, was bedeutet, dass die Abdichtungen die Flüssigkeiten eine bestimmte Zeit lang zurückhalten, diese jedoch irgendwann einmal abfließen lassen.

Das Abpumpen der Sickerwässer ist eine weitere Möglichkeit der Emissionskontrolle in das Grundwasser, vorausgesetzt, es wird beachtet, dass das Abpumpen eventuell auch nach Schließung der Rückhalteanlage für Aufbereitungsrückstände fortgeführt werden muss. Die Notwendigkeit des Abpumpens nach der Stilllegung sollte daher bei der Sanierungs- und Stilllegungsplanung berücksichtigt werden.

4.3.10.1 Vermeidung und Verringerung von Sickerwasser

Die wirksamste Methode zur Vermeidung des Abflusses von Sickerwasser in den Untergrund ist die richtige Standortwahl, d. h. in einem Abflussgebiet, in dem eine undurchlässige hydraulische Sperre vorhanden ist oder wo hydrogeologische Bedingungen herrschen, die dazu führen, dass Grundwasser in das Absetzbecken fließt. So könnten zum Beispiel Taubgesteinflächen oder Absetzbecken auf natürlichem Feuchtland errichtet werden, wo der Untergrund auf natürliche Weise undurchlässig ist.

Muss der Eintritt von Sickerwasser in den Untergrund verhindert werden, und eine natürliche Sperre ist nicht vorhanden, kann die Sohle des Absetzbeckens mit Ton oder anderem Dichtungsstoff undurchlässig gemacht werden, so dass die Wasserpenetration unter 10^{-8} m/s beträgt. Dafür müssen humöse Stoffe vor Herstellung der Abdichtung entfernt werden. In manchen Fällen liegen die Duchlässigkeitswerte unter 10^{-8}.
[131, IMA, 2003]

Abdichtungssystem sollen die Sickerung von Lauge durch die Sohle der Speicheranlage der Aufbereitungsrückstände verhindern. Alle Abdichtungssysteme verhindern letzendlich das Auslaufen nicht. Die Menge hängt ab von:
- der hydraulischen Druckhöhe über der Abdichtung
- der Mächtigkeit und Wirksamkeit des Abdichtungsstoffs
- der Dauer der Einwirkung der hydraulischen Druckhöhe auf die Abdichtung.

Es ist wichtig, die hydrogeologischen Bedingungen des Standortes und die geochemischen Eigenschaften der zu bewirtschaftenden Aufbereitungsrückstände zu beachten [11, EPA, 1995].

Der Einsatz von Abdichtungen ist ein häufig diskutiertes Thema. Ihr größter Vorteil besteht in der möglicherweise erheblichen Reduzierung des Sickerwassers. Kritiker halten dem entgegen, dass es nicht möglich ist, eine Prognose darüber zu geben, für welchen Zeitraum die Abdichtungen wie vorgesehen funktionieren. Eine Alternative ist die Behandlung des Sickerwassers von Beginn der Nutzung an.
Abb. 4.14 zeigt die verschiedenen vorhandenen Abdichtungssysteme.

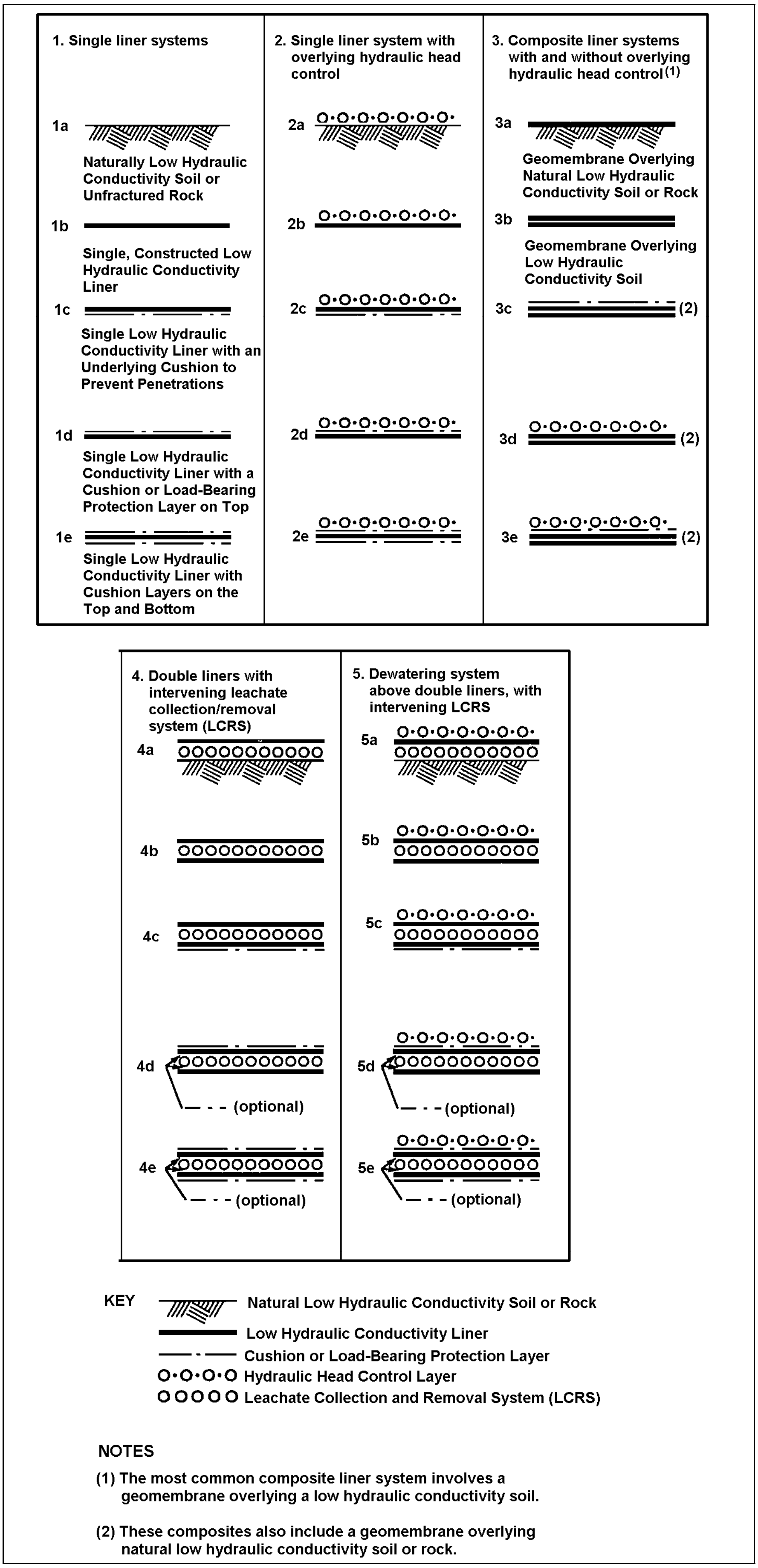

Abb. 4.14: Vorhandene Abdichtungssysteme
[11, EPA, 1995]

1. Einfachabdichtungssysteme
1a) Erdreich mit natürlich geringer Wasserdurchlässigkeit oder Gestein ohne Bruchbildung
1 b) gefertigte Einfachabdichtung mit geringer Wasserdurchlässigkeit
1 c) Einfachabdichtung mit geringer Wasserdurchlässigkeit und Unterkissen zur Verhinderung von Durchdringung
1 d) Einfachabdichtung mit geringer Wasserdurchlässigkeit mit aufliegender Kissen- oder tragender Schutzschicht
1 e) Einfachabdichtung mit geringer Wasserdurchlässigkeit mit Kissenlagen darauf und darunter

2. Einfachabdichtungssystem mit aufliegender Kontrolle der hydraulischen Druckhöhe

3. Kombinierte Abdichtungssysteme mit und ohne aufliegender Kontrolle der hydraulischen Druckhöhe(1)
3 a) Geomembran auf Erdreich mit natürlich geringer Wasserdurchlässigkeit oder Gestein
3 b) Geomembran auf Erdreich mit natürlich geringer Wasserdurchlässigkeit

4. Doppelabdichtungen mit Zwischenlage für Sammlung/Ableitung von Lauge

5. Entwässerungssystem auf Doppelabdichtung, mit Zwischenlage für Sammlung/Ableitung von Lauge

Zeichenerklärung:
Natual Low Hydraulic Conductivity Soil or Rock - Erdreich oder Gestein mit natürlich geringer Wasserdurchlässigkeit
Low Hydraulic Conductivity Liner – Abdichtung mit geringer Wasserdurchlässigkeit
Cushion or Load-Bearing Protection Layer - Kissen- oder tragende Schutzschicht
Hydraulic Head Control Layer – Lage zur Kontrolle der hydraulischen Druckhöhe
Leachate Collection and Removal System (LCRS) - System für Sammlung/Ableitung von Lauge

Anmerkungen:
(1) Das am häufigsten verwendete kombinierte Abdichtungssystem besteht aus Erdreich oder Gestein mit natürlich geringer Wasserdurchlässigkeit und aufliegender Geomembran
(2) Diese Kombinationen bestehen ebenfalls aus einer Geomembran auf Erdreich mit natürlich geringer Wasserdurchlässigkeit oder Gestein

Wie jedoch bereits im letzten Abschnitt erwähnt, können Maßnahmen zu Einschränkung der Infiltration in die Ablagerung gegenüber **Sohleabdichtungen** mit gringer Duchlässigkeit bevorzugt werden, wenn ein entsprechendes hydrauliches Druckgefälle den Schadstofftransport fördert (der sogenannten 'Badewanneneffekt') [13, Vick,].

Ein Anwendungsgebiet von Abdichtungen sind Becken, bei denen:

- das Prozesswasser ansonsten in den Untergrund sickern würde (z. B. Becken auf flachem Land wie in Abb. **2.4.2.5**, keine hydraulische Sperre) und
- angestrebt wird, dass das Prozesswasser während des Betriebs im Becken verbleibt, z. B.:
  - um das Prozesswasser erneut zu nutzen,
  - weil das Wasser belastet ist (z. B. CN)
  - zur Vermeidung von Staubentwicklung durch ständige Sättigung des Spülrandes und
- es nicht erforderlich ist zu sichern, dass Aufbereitungsrückstände nach der Stilllegung wassergesättigt bleiben.

Zeitweilige Becken (nur während des Betriebs) mit goldhaltiger Prozessflüssigkeit in CN-Laugung und Haldenlaugung werden ebenfalls häufig abgedichtet, um Sickerung der CN-belasteten Lösung in den Untergrund zu vermeiden. In vielen Fällen werden dabei Doppelabdichtungen verwendet.

Es ist faktisch unmöglich, eine belastete Abdichtung zu reparieren. Abtragen des Stoffs ist keine praktische Lösung. Das Niederbringen von Bohrungen im betroffenen Gebiet (so man es denn lokalisiert!) und Einpressen von Bentonit ist sehr schwierig und teuer. Genauigkeit ist dabei ein Hauptproblem.

Unter anderen als Reparaturgesichtspunkten betrachtet, sind Abfanggräben oder hydraulische Sperren am TMF-Rand weitere Möglichkeiten, sind jedoch sehr kostenintensiv und müssten in Anbetracht der Größe der meisten Absetzbecken auch sehr große Bauwerke sein. Sie sind in der Tiefe ebenfalls begrenzt, so dass diese Sperren bei Infiltration des Sickerwassers in das Muttergestein von sehr geringer Wirkung wären. Heben und Behandeln des Sickerwassers wäre eine weitere Möglichkeit, jedoch sehr kostenintensiv und eventuell nur möglich, wenn die Grube in Betrieb ist, weil nur in dieser Phase eine Behandlung lokal möglich sein könnte. Dies ist keine Langzeitlösung, sie wäre nicht durchzuhalten.

Ein weiterer entscheidender Aspekt ist, dass die Sickerung durch die hydraulische Druckhöhe nicht vollständig kontrolliert wird. Entfällt diese, kommt es zu keiner oder sehr geringer Sickerung. Drainage und Bedeckung der Aufbereitungsrückstände verringert oder verhindert die Ausbildung der Druckhöhe und damit die Sickerung. Dies ist vermutlich die praktischste Lösung der Sickerproblematik in geschlossenen Anlagen.

Eine Abdichtung kann in keinem Fall garantiert jegliche Sickerung vermeiden. Bestimmte Öffnungen oder Konstruktionsfehler sind unvermeidbar. Was die Abdichtung jedoch erreicht, ist die Reduzierung der Sickerung auf ein Maß, dass für die Umwelt damit durch Verdünnung und Verteilung oder Abbau toleriert werden kann.

Bei der Auslegung abgedichteter TMF ist es erforderlich, die Möglichkeit der Sickerung in Betracht zu ziehen und auszuschließen, dass geringe Sickermengen (im Rahmen der üblichen Industriefaktoren für Baumängel bei Abdichtungen) zu erheblicher Umweltverschmutzung führen. Andernfalls wäre eine Art Sekundärrückhaltung (oder Sammelschicht für Sickerwasser) erforderlich (z. B. Ton, Torf, Bentonit usw.). In vielen Fällen sind die Aufbereitungsrückstände von so großer Feinheit, dass sie nach der Verdichtung eine ähnliche Duchlässigkeit wie mineralische Abdichtungen haben. Das heißt, die sekundäre Rückhaltung kommt von innen. Dies ist günstiger, wenn die Aufbereitungsrückstände über Drainage entwässert werden. Die Verdichtung kann viele Jahre nach der Schüttung in Anspruch nehmen, bis die Aufbereitungsrückstände in ausreichender Tiefe und/oder entwässert sind. In diesem Fall dient die synthetische Abdichtung als Hauptrückhaltung bis sich die Aufbereitungsrückstände verfestigt haben. Danach wirken die Aufbereitungsrückstände generell als wirksame Sperre. Daher ist die Langzeiterwartung an die synthetische Abdichtung nicht das Hauptpoblem.

4.3.10.2 Sickerwasserkontrolle

Es können zwei Arten von Kontrollmaßnahmen betrachtet werden, d. h.:

- Sickersperren und
- Rückführanlagen.

Sickersperren dienen der Verhinderung der Sickerung in den Untergrund und bestehen aus Schlitzwänden, Schlammwänden und Zementmörtelabdichtungen.

Jedoch müssen in jedem Fall mögliche Nachteile dieser Maßnahmen im Zusammenhang mit der Stabilität des Dammes für die Aufbereitungsrückstände betrachtet werden.

In einigen Fällen kann es günstiger sein, eine Rückführanlage statt Sickersperren vorzusehen. Rückführanlagen sammeln den Sickerwasserfluss, statt ihn zu behindern und ermöglichen damit, dass das Sickerwasser behandelt oder auf eine Weise entsorgt werden kann, welche die Umwelt nicht schädigt. Die Rückführanlage könnte aus Sammelgräben und Brunnen bestehen.
Die Vorteile und Grenzen von Maßnahmen zur Sickerkontrolle sind in Tabelle 4.11 dargestellt.

Maßnahmen zur Sickerkontrolle	Art	Vorteile	Grenzen
Sickersperren	Schlitzwand	Wenig kostenintensiv; gute Kontrollmöglichkeit.	Unpraktisch für Gründungen gesättigter Sperren; wirksam nur bei durchlässigen Schichten geringer Mächtigkeit.
	Schlammwände	Sperre mit geringer Duchlässigkeit kann errichtet werden.	Hohe Kosten; nicht besonders geeignet für steiles Gelände oder steinigen Untergrund; undurchlässige Grenzschicht unten erforderlich.
	Zementmörtelabdichtungen	Sperre kann bis in große Tiefe gebaut werden; keine Beeinflussung durch die Standorttopographie	Hohe Kosten; begrenzte Wirksamkeit wegen Duchlässigkeit der Mörtelzone; Zementmörtelung nur praktisch für grobe Böden mit breiten Felsspalten.
Rückführanlagen	Sammelgräben	Wenig kosntenintensiv; geeignet für jede Dammart	Effektiv bei durchlässigen Schichten geringer Mächtigkeit, jedoch günstig auch noch für andere.
	Sammelbrunnen	Größere Tiefen möglich; geeignet als Hilfsmaßnahme	Kostenintensiv; Wirksamkeit hängt ab von der Beschaffenheit der lokalen Grundwasser führenden Schicht.

Tabelle 4.11: Übersicht der Maßnahmen zur Sickerwasserkontrolle

Es sei angemerkt, dass unter realen Bedingungen die Sickerwasserkontrolle oft mehrere der in der Tabelle genannten Methoden umfasst. Auch ist außer Sperren, die allein zur Kontrolle des Sickerwassertransports errichtet werden, die Behandlung der Schadstoffe im Sickerwasser durch bestimmte reaktive Sperren möglich.

4.3.10.3 Halden mit Kali-Aufbereitungsrückständen

Bei Abraumhalden von Kalisalzbergwerken muss die Wasserduchlässigkeit des Bodens von Fall zu Fall bestimmt werden (Vergleichsbedingungen). Meist sind die bestimmten Bodenkomponenten ausreichend undurchlässig, um eine Verschmutzung des Grundwassers zu verhindern. Andernfalls muss der Untergrund unter den Kaliabraumhalden abgedichtet werden, z. B. durch Verbesserung des natürlichen Bodens durch Zugabe von bis zu 4 % Ton. Der Ton wird auf den natürlichen Boden ausgebracht, eingearbeitet und zur Herstellung der erforderlichen Undurchlässigkeit verdichtet. Nach der Behandlung wird der Duchlässigkeitskoeffizient geprüft und bei Unzulänglichkeit der Vorgang wiederholt.

Der Haldenfuß der Aufschüttungen außerhalb der undurchlässigen Kernzone wird abgedichtet und die Lösungen gesammelt.

Lange Erfahrung beim Verkippen von Aufbereitungsrückständen aus Kalibergwerken ist erforderlich, um die geeignetsten Methoden für die Bewirtschaftung der Aufbereitungsrückstände zu entwickeln. Beispielsweise kann der Einbau von Tonabdichtungen unter der Halde zu Stabilitätsproblemen führen. Für die Erweiterung einer Abraumhalde im Revier Fulda in Deutschland verlangten die Behörden die Abdichtung des Untergrundes mit einer künstlichen Tonabdichtung von 0,6 m. Beim Aufschütten der Halde in diesem abgedichteten Bereich wurden schnelle Bewegungen des Teils der Halde über der Tonabdichtung in einem Ausmaß beobachtet, dass die Sicherheit der Beschäftigten auf und vor

der Halde bedroht war und der Betrieb eingestellt werden musste. Nach einer Untersuchung kam man zu dem Schluss, dass Stoffe mit geringer Scherfestigkeit nicht zur Abdichtung des Untergrundes unter Kaliabraumhalden eingesetzt werden sollten.
[19, K+S, 2002]

4.3.10.4 Halden mit Kohle-Aufbereitungsrückständen

An Ruhr, Saar und in Ibbenbüren befinden sich Kohleaufbereitungsrückstände, die schichtweise auf die Halden verkippt wurden. Die Mächtigkeit der Schichten beträgt zwischen 0,5 und 4,0 m. Verdichtet wird durch die Räder der Transportfahrzeuge und Rüttelwalzen, um die Durchdringung mit Sauerstoff oder das Eindringen von Niederschlag in den Haldenkörper möglichst zu vermeiden und damit die Entstehung von Sauerwässern durch Pyritoxidation zu minimieren.

Das Prinzip der Errichtung einer Abraumhalde ist in Abb. 4.15 dargestellt. Der Bau erfolgt in vier Abschnitten in den Schüttungsphasen. Der erste Schritt umfasst den Bau eines äußeren Walls, der sofort wieder mit Vegetation versehen wird und als Schutz für die späteren Schüttungen von Aufbereitungsrückständen im inneren Bereich dient.

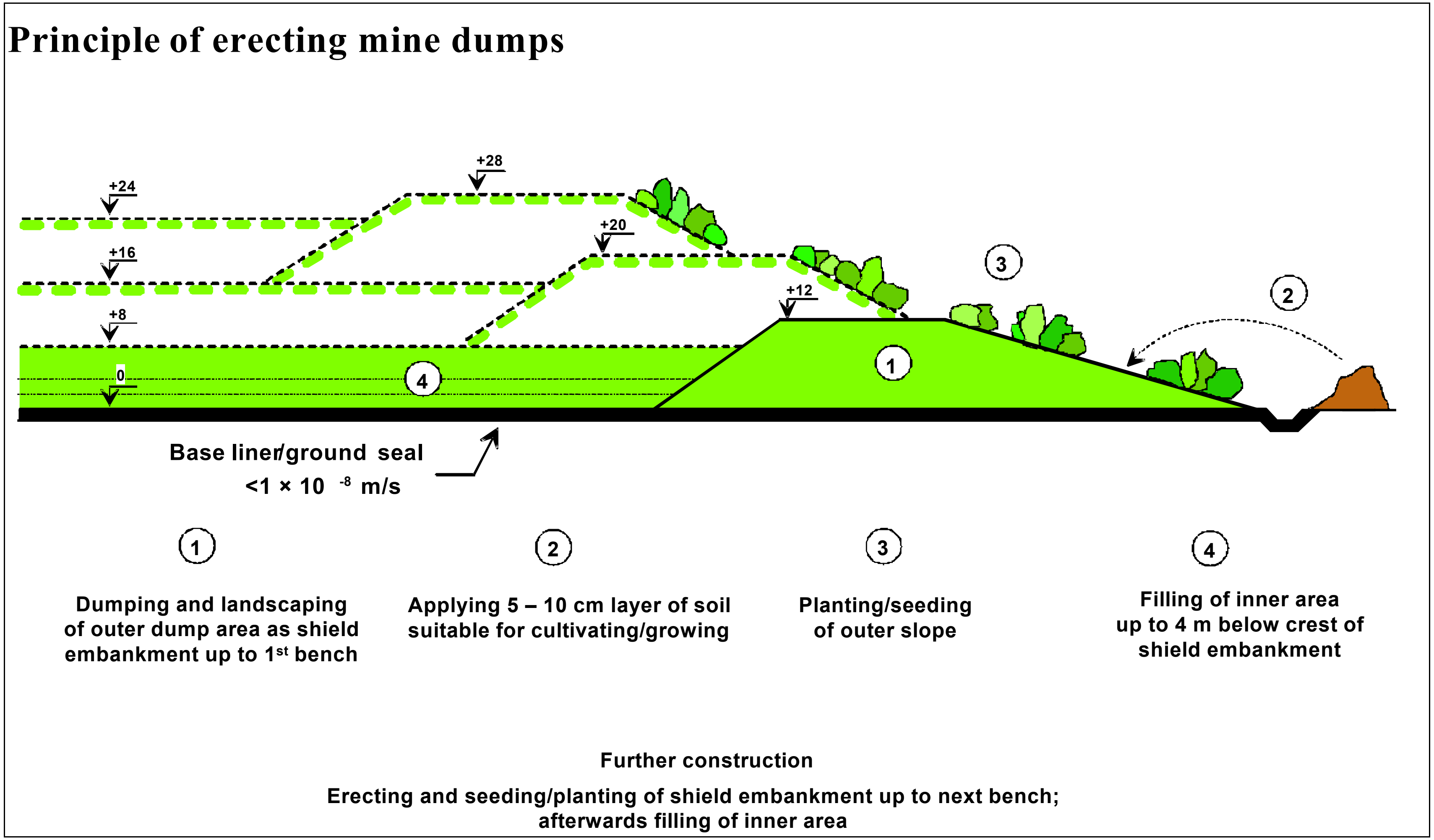

Abb. 4.15: Schematische Darstellung des Aufbaus einer Abraumhalde an Ruhr und Saar sowie in Ibbenbüren [79, DSK, 2002]

Aus Lysimeteruntersuchungen ist bekannt, dass Sickerwasser aus Kohleabraumhalden Elemente in Lösung enthalten kann. Ergebnisse dieser Untersuchungen haben gezeigt, dass Chlorid ausgewaschen werden kann und Sulfat-, Calcium- und Magnesiumwerte durch die Pyritoxidation ansteigen können. Die Bildung von Sauerwässern ist möglich. Wenn dies geschieht, können der Abfall des pH-Wertes und die Verschlechterung der Pufferleistung der Aufbereitungsrückstände oder der Grundwasser führenden Schicht zur Mobilisierung von Spurenelementen in den Aufschüttungen führen.

Als Folge dessen ist der Grundwasserschutz das wichtigste Umweltanliegen bei Bau und Betrieb von Halden. Es gibt vier Hauptmaßnahmen zum Schutz des Grundwassers vor möglichen Haldenschmutzwässern (Abb. 4.10).

Je nach Standortbedingungen kommen spezifische Lösungen zum Einsatz, d. h. Einzelmaßnahmen oder Kombinationen verschiedener Maßnahmen können angewendet werden.

Kürzlich wurde festgestellt, dass sich eine ältere Abraumhalde in einem solchen Umfang ‘selbstverfestigt’ hatte, dass der innere Haldenkörper absolut trocken war.
[79, DSK, 2002]

An Ruhr und Saar sowie in Ibbenbüren bestehen die Kohleaufbereitungsrückstände aus einem Gemisch von Kieselerde und Ton. Im Laufe der Zeit verfestigt sich dieses Gemisch weiter. Der Kieselerdeanteil ist so beschaffen, dass die Befeuchtung des Tons zu keinen Stabilitätsproblemen der Böschungen führt.

Oberflächenabfluss und Sickerung aus den Aufbereitungsrückständen werden gesammelt und zu den aufnehmenden Oberflächengewässern geleitet.

Maßnahmen zur Vermeidung nachteiliger Auswirkungen auf das Grundwasser- und Drainagesystem

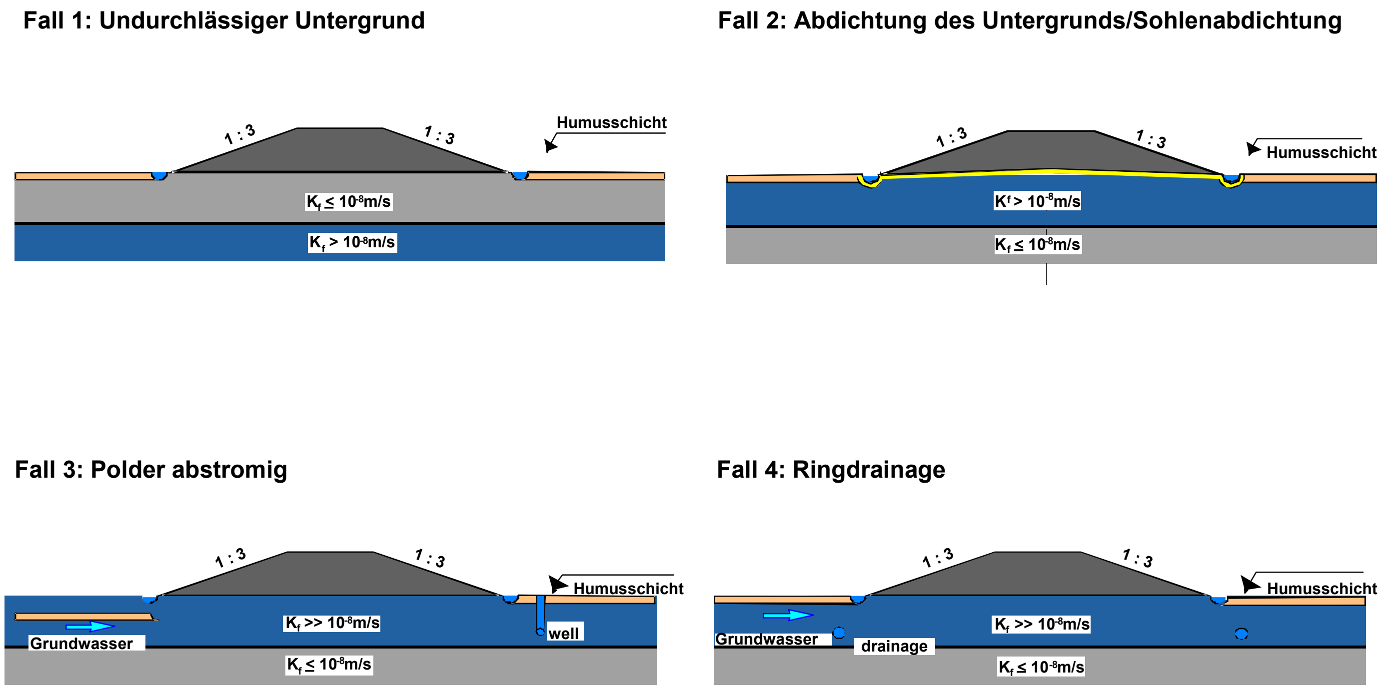

Abb. 4.16: Auslegung von Abraumhalden – Optionen zur Vermeidung nachteiliger Auswirkungen für Grund- und Oberflächenwassersystem [79, DSK, 2002]

4.3.11 Verfahren zur Verminderung von Emissionen in Gewässer

4.3.11.1 Wiederverwendung von Prozesswasser

Eine Möglichkeit zur Reduzierung der Wasseremissionen ist die Wiederverwendung des Prozesswassers. Das wird in mehreren Anlagen erfolgreich praktiziert. Nur der nicht wieder verwendbare Überschuss, z. B. wegen

- Schneeschmelze
- Sättigung mit magnesiumhaltigen Salz (bei Kaliwerken) [19, K+S, 2002]
  wird entweder – wie bei mehreren Kaligruben - in Tiefbrunnen verpumpt oder in Oberflächengewässer eingeleitet.

Die Wiederverwendung von Prozesswasser kann unmöglich sein, wenn die Konzentration von Reagenzien/Komponenten die Aufbereitung stört (z. B. Calciumsulfat in Wasser, was zu Verstopfungsproblemen in den Rohren führen kann).

4.3.11.2 Waschen von Aufbereitungsrückständen

Reagenzien bei der Flotation von Silikaten werden von Silikatteilchen stark adsorbiert. Die Aufbereitungsrückstände der Flotation werden jedoch mit geklärtem Prozesswasser gewaschen, damit mögliche freie Reagenzien gebunden werden. Aufbereitungsrückstände mit Silikatpartikeln binden die im Abwasser noch vorhandenen restlichen freien Reagenzien. Damit führt ein nachgeschalteter Entwässerungsprozess zu sauberem Wasser, das keine Reagienzien mehr enthält und eingeleitet oder dem Prozess erneut zugeführt werden kann.
[131, IMA, 2003]

4.3.11.3 Behandlung gelöster Metalle

Die Adsorptionsfähigkeit fein vermahlener Aufbereitungsrückstände hat eine reinigende Wirkung bei Wasser mit in Lösung befindlichen Metallen (z. B. aus der Grube kommendes Wasser oder Austragswasser aus Taubgesteinshalden). Wird daher Grubenwasser in den Strom der Aufbereitungsrückstandes gegeben, lagern sich die gelösten Metalle an der mineralischen Oberfläche an. An mineralischen Oberfächen adsorbierte Metalle verbleiben in dieser Form, solange der pH-Wert günstig ist (z. B. >7 bei Zink, >5 bei Kupfer). Zur Verbesserung des Kontakts zwischen den gelösten Metallen und der Partikeloberfläche der Aufbereitungsrückstände wird das Grubenwasser vor dem Verpumpen in den TMF in den Sumpf des Aufbereitungsrückstandsstroms aufgegeben.

Dies ist ein einfaches System unter Ausnutzung der Adsorptionswirkung ‘natürlicher Stoffe. Diese Methode kann leicht bei den meisten TMF angewendet werden. Nachrüstung ist kein Problem.
[118, Zinkgruvan, 2003]

Vermischen des Stroms der Aufbereitungsrückstände (einschließlich Prozesswasser und feste Aufbereitungsrückstände) und anderer Wässer mit Anteilen an gelösten Metallen (z. B. Austragswasser aus Taubgesteinaufschüttungen, Grubenwasser – kein Frischwasser) ist in der Betriebsphase anwendbar, wenn:

- der Strom der Aufbereitungsrückstände einen basischen pH-Wert aufweist und frisch vermahlende Minerale enthält (das ist normal der Fall bei Aufbereitungsrückständen aus einem Flotationsprozess),
- die Pufferkapazität des Stroms der Aufbereitungsrückstände erheblich größer als die Säurebildungskapazität des Zugabewassers ist,
- die Zugabe des metallhaltigen Wassers in den Pumpen für Aufbereitungsrückstände in einer solchen Weise in den Strom der Aufbereitungsrückstände erfolgen kann, dass eine

ausreichend lange Kontaktzeit und Vermischung mit den Aufbereitungsrückständen öglich ist.

Das als günstig eingeschätzte Verfahren ist die Flockung.

Die Vorteile des Verfahrens sind:
- Es ist eine sehr wirksame Wasserbehandlungsmethode.
- Keine Kosten für Bau, Betrieb und Wartung einer Wasseraufbereitungsanlage während des Betriebs der Grube.
- Keine Notwendigkeit der Schlammbewirtschaftung (die bei herkömmlicher Wasseraufbereitung anfällt).
- Das Verfahren kann problemlos schwankende Zuflussmengen verarbeiten und ist in allen Temperaturbereichen wirksam. Dies ist wichtig, das das Prozesswasser normalerweise eine erhöhte Temperatur hat.

Eine Variante dieses Verfahrens wird im Kupferrevier Legnica-Glogow angewendet, wo Säuren aus dem Hüttenbetrieb mit Aufbereitungsrückständen zur Neutralisation und Immobilisierung der Metalle (z. B. Arsen) gemischt werden.

4.3.11.4 Schwebstoffe und gelöste Bestandteile

In Ablaufwässern erfolgen feste Emissionen in das Wasser entweder als Feststoffteilchen oder gelöste Komponenten. Die erfolgreiche Wasseraufbereitung muss die Reduzierung der schwebenden Feststoffe mit der Beseitigung der schädlichen gelösten Anteile an Schadstoffen verbinden.

Die Wasserbehandlung kann entweder im offenen Becken oder in errichteten Aufbereitungsanlagen erfolgen. Die eingesetzten Prozesse sind Ausfällung der gelösten Elemente, hauptsächlich Metalle und Abscheidung der Niederschläge und Teilchen. Für Fällungsreaktionen werden entweder Sulfid oder Kalk oder beide Stoffe gemeinsam eingesetzt. Zur Abscheidung von Niederschlägen und Feststoffen kommen Schwerkraft- oder Zwangssedimentation zur Anwendung. Schwerkraftabscheidung kann in Becken oder Eindickern erfolgen.

Der erzeugte Schlamm verlangt ordnungsgemäße Bewirtschaftung und Ablagerung. Im Idealfall kann die Ablagerung als Teil des Versatzes der Grube erfolgen.

Wasseraufbereitung – wie notwendig auch immer – verursacht erhebliche Kosten.

Jede Bergbaugesellschaft muss ein geeignetes System der Wasseraufbereitung vorsehen. Die Forderungen an das System werden von der standortspezifischen Wasserqualität und den aufzubereitenden Mengen bestimmt. Die Wahl der Technologie hängt auch von den Bedingungen vor Ort ab.

Das Reinigungsverfahren zum Ausfällen von Schwebstoffen im Kupferrevier Legnica-Glogow beruht auf Koagulation (mit ca. 300 mg/l Eisenchlorid) mit Unterstützung durch Polyelektrolyt Praestol (1 mg/dm^3) und Sedimentation in einem Lamellenklärer. [KGHM Polska Miedz, 2002 #113]

4.3.11.4.1 Sedimentationsbecken

Beim Verkippen von Aufbereitungsrückständen aus der Flotation oder anderer Aufbereitungsrückstände mit Feinanteilen auf Halde, können durch Feststoffe und Eluate Emissionen in das Wasser gelangen. Emissionen von Feststoffen in Wasser durch schwere Niederschläge können erfolgreich durch den Bau von Sedimentationsbecken entlang der Straßen

und vor dem Oberflächengewässer, in das die Einleitung erfolgt, verhindert werden. Der Bau hängt von der maximalen Niederschlagsmenge, Fläche und Neigung, Strömungsmenge, Größe der Feststoffe usw. ab. Zur Dokumentation ist die Überwachung des Feststoffanteils erforderlich, jedoch entsprechend den Bedingungen vor Ort. Häufigkeit und Art der Messungen werden nach den in der Geotechnologie-/Umweltstudie getroffenen Festlegungen definiert und über die Nutzungszeit der TMF angepasst.
[131, IMA, 2003]

Das Innere von **Kalirückstandshalden** ist undurchlässig für Wasser. Wasser und entstehende Salzlösungen fließen im äußeren Bereich um den undurchlässigen inneren Kern herum ab. Der Haldenfuß der Aufschüttungen außerhalb der undurchlässigen Kernzone wird sorgfältig abgedichtet und die Lösungen gesammelt.

Diese Art eines Sammelbeckens ist geeignet, wenn die direkte Einleitung des Oberflächenabflusses in den Untergrund nicht umweltgerecht wäre.

In der **Kohlenabraumhalde** Schöttelheide verläuft ein Graben entlang des Fußes der Halde, in dem sich der Oberflächenabfluss sammelt und in das Absetzbecken fließt, bevor das Wasser in das Aufnahmegewässer eingeleitet wird. Dies ist wegen des hohen Anteils an Schwebstoffen erforderlich.

In der Betriebsphase einer Halde ist es im Normalfall erforderlich, den Oberflächenabfluss in Gräben am Haldenfuß zu sammeln. Die weiter erforderliche Bewirtschaftung des gesammelten Wassers hängt von der Wasserqualität des Oberflächenabflusses ab. Ist das Wasser von guter Qualität und enthält geringe Konzentrationen an Schwebstoffen, kann eine direkte Einleitung in das Aufnahmegewässer erfolgen. Ist die Wasserqualität gut, der Schwebstoffgehalt jedoch erhöht, kann es ausreichen, das Wasser durch ein Absetzbecken/Abscheider zu leiten um die Schwebstofflast vor Einleitung zu reduzieren. In manchen Fällen ist eine zusätzliche Behandlung erforderlich. Der erfasste Oberflächenabfluss kann häufig als Prozesswasser genutzt werden.

4.3.11.5 Behandlung säurehaltiger Wässer

Die Methoden der Wasseraufbereitung zur Beseitigung oder Reduzierung der Azidität und Schwermetallniederschläge aus belasteten Wässern können in zwei Gruppen unterteilt werden: (1) aktive und (2) passive Aufbereitung:
(1) Aktive Aufbereitung umfasst die Neutralisierung der säurebelasteten Wäser mit basischen Chemikalien. Jedoch können die Chemikalien teuer und Bau und Betrieb der Aufbereitungsanlage kostenintensiv sein.
(2) Passive Aufbereitung beinhaltet den Bau einer Aufbereitungsanlage, die natürlich vorkommende und wenig Wartung erfordernde chemische und biologische Reaktionen zur Unterstützung der Sauerwasserbehandlung nutzt. Passive Kontrollmaßnahmen umfassen anoxische Entwässerungen, Kalksteinkanäle, basische Anreicherung des Grundwassers und die Ableitung der Drainage über künstliche Feuchtgebiete oder andere Absetzbauwerke.

Es besteht auch eine Möglichkeit, aktive und passive Aufbereitungsmethoden miteinander zu kombinieren (z. B. Kalkung und künstliche Feuchtgebiete).

<u>Aktive Aufbereitung - Chemikalien</u>

- Kalkstein (Calciumcarbonat)
  Die Vorteile des Einsatzes von Kalkstein sind gringe Kosten, einfache Anwendung und Bildung eines dichten und leicht zu handhabenden Schlamms. Nachteile sind langsame Reaktion, Wirkungsverlust des Systems durch Umhüllung der Kalksteinpartikel mit ausgefälltem Eisen, Schwierigkeit der Behandlung von Sauerwässern mit hohem Verhältnis von Eisen-(II) / Eisen-(III) und Unwirksamkeit bei der Entfernung von Mangan. Ein typisches Fließbild einer Sauerwasseraufbereitungsanlage ist in Abb. 4.17 dargestellt.

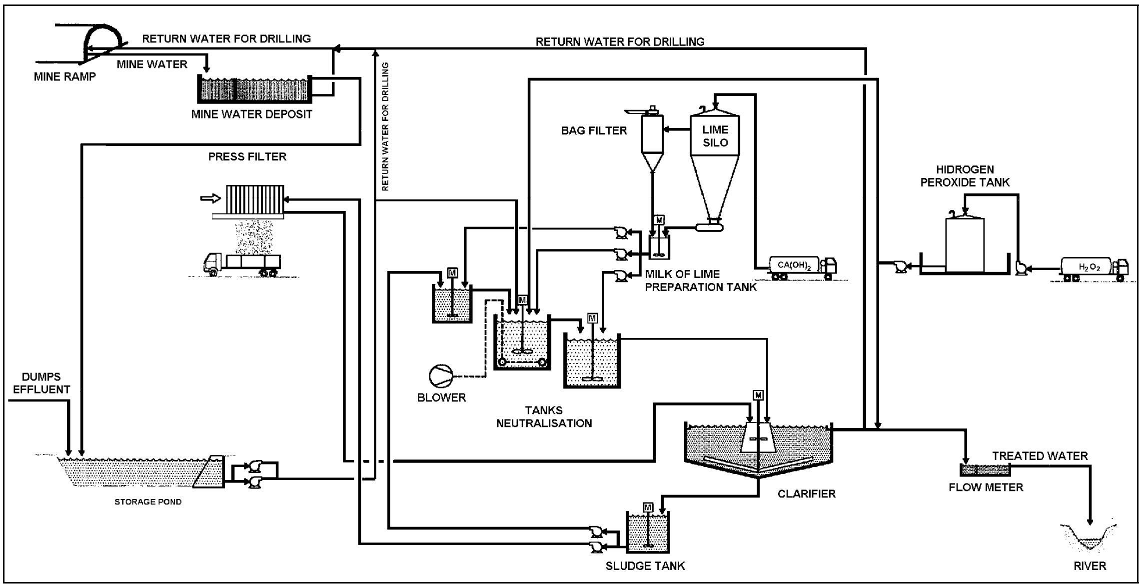

Abb. 4.17: Fließbild einer Wasseraufbereitungsanlage für Prozesswasser mit niedrigem pH-Wert
(aus Almagrera)
Mine ramp – Grubenausfahrt
Mine water – Grubenwasser
Return water for drilling – Rücklaufwasser zum Bohren
Mine water deposit – Grubenwasserspeicher
Press filter – Pressfilter
Dumps effluent – Haldenabfluss
Storage pond – Speicherbecken
Blower – Gebläse
Tanks neutralization – Neutralisationsbehälter
Bag filter – Taschenfilter
Lime silo – Kalksilo
Milk of lime preparation tank – Ansatzbehälter für Kalkmilch
Sludge tank – Schlammbehälter
Clarifier – Klärbehälter
Hydrogen perioxide tank – Wasserstoffperoxidtank
Flow meter – Durchflussmengenmesser
Treated water – aufbereitetes Wasser
River - Fluss

Entsprechend der Darstellung im obigen Fließbild werden Gruben- und Prozesswasser gemeinsam aufbereitet. Das ist nicht immer der Fall.

- Löschkalk (Calciumhydroxid)
  Im Normalfall ist Löschkalk das Neutralisationsmittel im Kohlebergbau, denn er ist einfach und sicher in der Handhabung, wirksam und relativ kostengünstig. Die wesentlichsten Nachteile sind die großen Mengen von anfallendem Schlamm (im Vergleich zu Kalkstein) und die hohen Investitionskosten wegen der Größe der Aufbereitungsanlage [85, EPA, 2002]. Löschkalk wird als Neutralisationsmittel im deutschen Kohlebergbau nicht eingesetzt, da saure Sickerwässer aus Aufschüttungen nicht vorkommen.

- Sodaschmelze (Natriumcarbonat)
  Presslinge aus Sodaschmelze sind besonders wirksam bei der Behandlung kleiner ARD-Abflüsse in abgelegenen Gebieten. Hauptnachteile sind die höhren Reagenzkosten (im Verhältnis zu Kalkstein) und das schlechte Absetzverhalten des Schlamms

- Ätznatron (Natriumhydroxid)
  Ätnatron ist besonders wirksam für die Behandlung von geringen Abflussmengen an abgelegenen Standorten und für die Behandlung von ARD mit hohem Mangananteil. Hauptnachteile sind die hohen Kosten, Gefahren beim Umgang mit der Chemikalie, schlechte Eigenschaften des Schlamms und Probleme mit dem Gefrieren bei niedrigen Temperaturen.

- Ammoniak
  Wasserfreies Ammoniak ist wirksam bei der Behandlung von Sauerwässern mit hohen Anteilen an Eisen(II) und/oder Mangan. Ammoniak kostet weniger als Ätznatorn und hat viele von dessen Vorteilen. Jedoch ist Ammoniak schwierig und gefährlich im Umgang und kann die biologischen Bedingungen abstromig der bergbaulichen Tätigkeiten beeinträchtigen. Die möglichen nicht Standort gebundenen Auswirkungen sind Toxizität für Fische und andere Wasserlebensformen, Eutrophierung und Nitrifikation. Fischarten haben generell eine geringe Toleranz gegenüber nicht ionisiertem Ammoniak und die Toxizitätswerte können durch pH-Wert, Temperatur, gelösten Sauerstoff und andere Faktoren beeinflusst werden. Der Einsatz von Ammoniak ist nicht in allen Gebieten erlaubt und wo der Einsatz gestattet ist, sind zusätzliche Überwachungsmaßnahmen erforderlich.

Passive Behandlung

- Künstliche Feuchtgebiete
  Künstliche Feuchtgebiete nutzen boden- und wasserbürtige Mikroben zusammen mit Feuchtgebietpflanzen zur Beseitigung von gelösten Metallen aus Haldenabflüssen. Anders als die chemische Behandlung sind Feuchtgebiete passive Systeme, die keine oder nur sehr geringe ständige Wartung erfordern. Hierbei handelt es sich um eine relativ neue Behandlungsmethode, bei der viele spezifische Mechanismen und Wartungsvoraussetzungen noch nicht voll analysiert wurden. Die optimale Größe und die Konfigurationskriterien befinden sind noch Gegenstand von Untersuchungen. Alte, stabile, natürlich entstandene Feuchtgebiete sollten unberührt bleiben, denn die Säuerungsprozesse können zum Beispiel durch das Anlegen von Gräben erneut in Gang gesetzt werden.

  Zulaufende Wässer mit hoher Metallkonzentration und niedrigem pH-Wert fließen durch die aeroben und anaeroben Bereiche des Ökosystems Feuchtgebiet. Metalle werden durch Ionenaustausch, Adsorption, Absorption und Fällung mit geochemischer und mikrobieller Oxidation und Reduktion ausgeschieden. Der Ionenaustausch erfolgt durch Kontakt der Metalle im Wasser humose oder andere organische Stoffe im Feuchtgebiet. Zu diesem Zweck angelegte Feuchtgebiete besitzen häufig kein oder wenig Erdreich und sind statt dessen aus Stroh, Mist oder Kompost hergestellt. Durch Bakterien ausgelöste Oxidations- und Reduktionsreaktionen, die in den aeroben bzw. aneroben Zonen ablaufen, spielen eine wichtige Rolle bei der Ausfällung von Metallen als Hydroxide und Sulfide. Ausgefällte und adsorbierte Metalle setzen sich in ruhenden Becken ab oder werden bei der Perkolation des Wassers durch das Medium oder die Pflanzen ausgefiltert.

  Zulaufendes Wasser mit Resten von Explosivstoffen oder anderen Kontaminationsstoffen fließt durch und unter die Kiesoberfläche eines Kies-basierten Feuchtlands. Schwimmpflanzen machen das Feuchtgebiet zu einem gekoppelten aerob-anaeroben System. Die anaerobe Zelle benutzt Pflanzen in Kombination mit natürlichen Mikroben zum Abbau der Schadstoffe. Die aeroebe Zelle, auch als reziprozierende Zelle bekannt, verbessert die Wasserqualität weiter durch ständige Exposition mit den Pflanzen und die Bewegung des Wassers zwichen den Zellbereichen.

  Feuchtgebietbehandlung ist eine Langzeittechnologie mit vorgesehenem kontinuierlichen Betrieb über Jahre.

  Feuchtgebiete werden zur Behandlung von sauren Grubenwässern eingesetzt, die beim Erz- oder Kohleabbau entstehen. Diese Rückstände können hohe Metallkonzentrationen aufweisen und sind sauer. Der Prozess kann für die Verarbeitung neutraler und basischer Aufbereitungsrückstandslösungen angepasst werden. Die Technologie der Feuchtgebietbehandlung muss entsprechend den Unterschieden bei Geologie, Gelände, Zusammensetzung der Spurenmetalle und Klima angepasst werden. Im Feuchtgebiet wird Eisen in der Regel mit besserem Erfolg entfernt als Mangan. Der größte Nutzen der Feuchtgebiete scheint bei der Behandlung geringer Wassermengen im Größenbreich von einigen zehn Litern pro Minute zu liegen [85, EPA, 2002]. Die folgenden Faktoren können Anwendbarkeit und Wirksamkeit des Prozesses einschränken:

- Die Langzeitwirksamkeit künstlicher Feuchtgebiete ist nicht hinreichend geklärt. Feuchtgebietalterung könnte ein Problem sein, das mit der Zeit zur Reduzierung der Schadstoffabbauleistung beitragen kann.
- Die Kosten für die Errichtung eines künstlichen Feuchtgebietes sind von Projekt zuu Projekt stark unterschiedlich und können für viele Standorte finanziell nicht zu schultern sein.
- Temperatur- und Zulaufschwankungen wirken sich auf die Funktion des Feuchtgebiets aus und können dazu führen, dass der Schadstoffabbau mit unterschiedlichen Raten erfolgt.
- Kältere Bedingungen verlangsamen die Rate, mit der das Feuchtgebiet die Schadstoffe beseitigen kann.
- Bei Zulauf großer Wassermengen können die Eliminierungsmechanismen eines Feuchtgebietes überfordert sein, Trockenperioden können zu Pflanzenschäden führen und die Funktion des Feuchtgebietes stark einschränken.

[124, US FRTR, 2003]

Erstauslegungs- und Errichtungskosten können beträchtlich sein und bis zu mehreren Zehntausend Euro erfordern.

- Offene Kalksteinkanäle/anoxische Kalksteinabflüsse
  Hierbei handelt es sich um die einfachste errichtete passive Aufbereitungsmethode mit offenen, Kalkstein gefüllten Gräben (anoxische Abflüsse sind abgedeckt). Die Auflösung des Kalksteins verstärkt die Alkalinität und hebt den pH-Wert. Umhüllung des Kalksteins durch Eisen- und Aluminiumniederschläge beeinträchtigt die Wirksamkeit dieser Behandlungsmethode.

- Ableitungsbrunnen
  Sauerwasser wird in einen "Behälter" oder "Brunnen" abgeleitet, der zerkleinerten Kalkstein enthält. Die Umhüllung durch ausgefälltes Eisen wird durch die Turbulenz der Strömung durch den Brunnen verhindert. Erfordert regelmäßigen Austausch des Kalksteins.

[85, EPA, 2002]

Passive Aufbereitungssysteme sind häufig wegen Problemen bei Kapazität, besonders bei Strömung, Möglichkeit der Behandlung von Wässern mit hoher Azidität, jahreszeitlichen Schwankungen, Schwankungen der Zuflussmengen usw. nicht sehr günstig in der Anwendung. Sie können aber sehr gut eine Langzeitlösung nach der Außerbetriebnahme eines Standortes sein, wenn sie zusammen mit anderen (vorbeugenden) Maßnahmen angwendet werden.

4.3.11.6 Behandlung alkalischer Gewässer

In der sardinischen Aluminiumoxid-Raffinerie werden die basischen Wässer, welche mit dem Schlamm die Wasch- und Filteranlagen verlassen, nach folgenden Methoden auf einen pH-Wert von 10 eingestellt:

- Entschwefelung der SO_2-reichen Verbrennungsgase
- Zugabe von Salzwasser zur Reaktion von $MgCl_2$ mit Ätznatron
- Schwefelsäure bei Bedarf.

In der galizischen Aluminiumoxid-Raffinerie wird Wasser aus dem Rotschlammbecken (Klar- und Sickerwasser) gesammelt und zur Aufbereitungsanlage gepumpt (siehe Abb. unten). Der erste Schritt beinhaltet die Neutralisierung des Wassers durch Zugabe von Schwefelsäure. Der optimale pH-Wert liegt bei 6,85. Bei diesem Punkt wird das im Wasser enthaltene Aluminium unlöslich und fördert die Sedimentation. Nach der Neutralisation gelangt das Wasser über einen Überlauf in den Flockungstank. Das Klarwassser wird der Raffinerie wieder zugeführt.

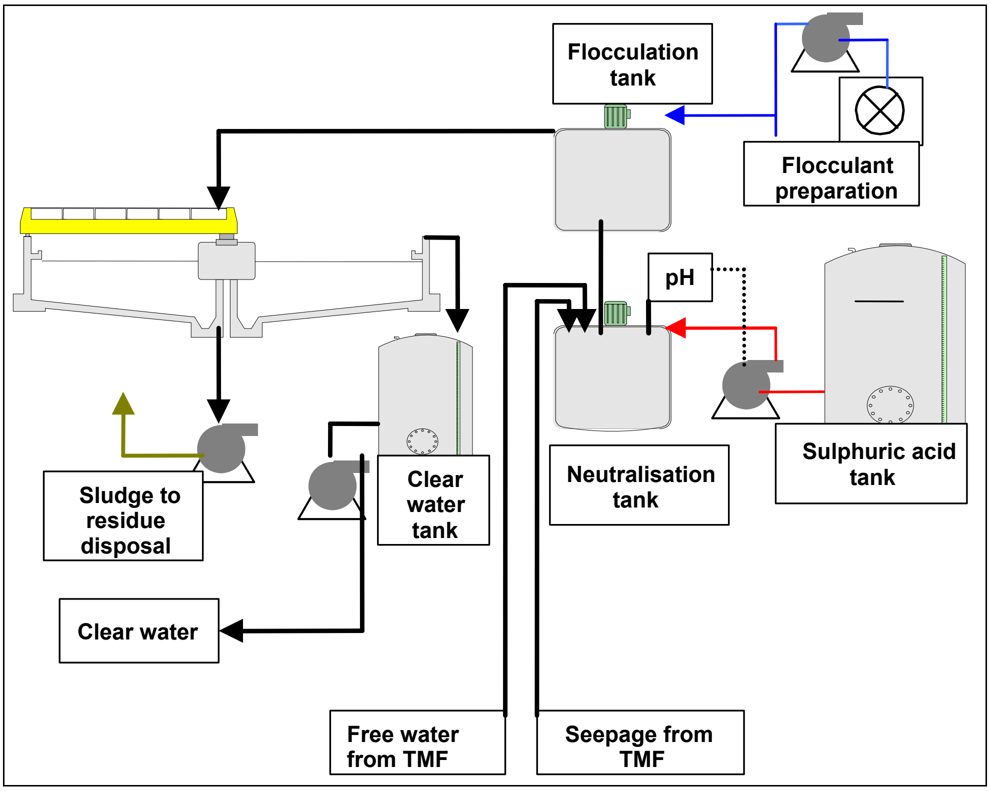

Abb. 4.18: Aufbereitung von alkalischem Wasser in einer Aluminiumraffinerie

In anderen Fällen wird der pH-Wert mit Kohlendioxid herabgesetzt.

4.3.11.7 Behandlung von Arsen

Spurenmetalle werden aus Grubenschmutzwässern wirksam durch Zugabe von Eisen(III)-Salzen entfernt. Arsen wird entweder als Calcium- oder Eisen(III)-arsenat durch Fällung abgeschieden. Durch Fällung wird Arsen entweder als Calcium- oder Eisen(III)-arsenat abgeschieden. Arsenite können ebenfalls ausgefällt werden, sind jedoch generell besser löslich und weniger stabil als Arsenate. Arsenithaltiges Schmutzwasser wird vor der Fällung generell oxidiert, um auf diese Weise zu sichern, dass Arsenat vorherrscht. Prozesswasser aus der Verarbeitung von arsenhaltigen Erzen kann unterschiedliche Mengen an Arsen(III) und -(V)-oxianionen, Arsenite und Arsenat enthalten. Die Gegenwart von Metallionen wie Kupfer, Blei, Nickel und Zink schränkt die Löslichkeit von Arsen wegen der Bildung schwerlöslicher Metallarsenate ein.

Stabilität und Löslichkeit dieser Arsenate hängt vom Verhältnis Eisen/Arsen ab. Je größer das Verhältnis, umso unlöslicher und stabiler das Präzipitat. Eisen(II)-arsenat ist noch relativ löslich, dagegen sind die basischen Arsenate mit einem Eisen-/Arsen-Molverhältnisvon acht oder mehr im pH-Bereich von ca. 2 bis 8 um Größenordnungen weniger löslich. Gelöste Arsenkonzentrationen von 0,5 mg/l oder weniger können durch Fällung mit Eisen(III) erreicht werden.

Die Fällung unlöslicher Eisen(III)-arsenate ist mit hoher Wahrscheinlichkeit begleitet von der Fällung anderer Metalle, z. B. Selen. Dabei kommt es zu Wechselwirkungen zwischen den verschiedenen Metallgattungen und dem Niederschlag aus Eisenhydroxid. Das macht Eisen(III)-Salze äußerst wirksame Spülmittel für die Entfernung von Spurenverunreinigungen. Damit kann durch Kontakt mit Eisenhydroxid die Konzentration von Arsen und zahleichen weiteren Elementen, z. B. Antimon und Molybdän, auf unter 0,5 mg/l reduziert werden. Dieser Prozess schließt im Normalfall die Zugabe von löslichem Eisen(III)-salz in das Prozesswasser ein, danach die Zugabe von Base in ausreichender Menge zur Anregung der Bildung von unlöslichem Eisenhydroxid. In vielen Fällen enthält das Prozesswasser ausreichend viel Eisen, wodurch lediglich die Zugabe einer Base erforderlich ist, um die Ausfällung von Eisenhydroxid auszulösen.
[78, Ron Tenny, 2001]

Finnische Talkum-Magnesiterze enthalten Arsenminerale. Bei der Verarbeitung des Talkum-Magnesiterzes (Vermahlung und Flotation) geht ein Teil des Arsens im Prozesswasser in Lösung. Arsen wird durch Zugabe von Ferrisulfat ($Fe_2(SO_4)_3$) als Fe-As-Verbindungen ausgefällt. Liegt der pH-Wert bei oder unter 6, kann Arsen vollständig ausgefällt werden. Ist der pH-Wert des Prozesswassers höher (in einem Fall wurde 7 – 8 berichtet), muss weiteres Ferrisulfat zur Senkung des Arsens auf einen akzeptablen Wert (unter 0,4 mg/l) zugegeben werden. Die gleichzeitige Ausfällung von Nickel und Arsen ist schwierig, daher wird ein Zweistufenverfahren benötigt.
[131, IMA, 2003]

4.3.11.8 Behandlung von Cyan

Im Weltmaßstab ist der natürliche Abbau noch immer die am weitesten verbreitete Methode der Behandlung von Cyan in Abwassern der Goldlaugung, obwohl häufig eine Ergänzung durch andere Aufbereitungsverfahren erfolgt. In trockenen und sonnenreichen Klimaten, z. B. Südafrika, ist der natürliche Abbau gewöhnlich die einzige Behandlungsmethode.

Die folgende Tabelle enthält die aktuell bei der Cyandbehandlung angewendeten Alternativen.

Behandlungsverfahren	Stufe	Anwendung	Bemerkungen
Natürlicher Abbau ▪ Neutralisation durch CO_2-Absorption ▪ HCN-Volatilisierung ▪ Dissoziation Metallcyanid-komplex ▪ Präzipitation Metall-cyanid	**C**	**TP, SW**	Anwendung ist begrenzt auf standortspezifische Faktoren (z. B. trocken, sonnig) und Vorschriften
Oxidationsprozesse			
▪ alkalische Chlorierung	**C**	**TP, SW**	Ersetzt durch SO_2-Luft und H_2O_2 aus Kostengründen, Unmöglichkeit der Entfernung von Eisen
▪ SO_2/Luftprozess	**C**	**TP, SW**	Universelle Anwendung, Schlammbehandlung kann zu erhöhtem Verbrauch von Reagenzien führen
▪ Wasserstoffperoxid	**C**	**SW**	Nicht anwendbar bei Schlempen wegen des hohen Reagenzienverbrauchs
Adsorption ▪ Adsorption an Aktivkohle	**D**	**SW**	Beschränkt auf niedrige CN-Konzentrationen, standortspezifisch
Biologische Behandlung ▪ Biologischer Abbau	**C**	**SW**	Beschränkt auf niedrige CN-Konzentrationen, standortspezifisch, kann zusätzliche Wärmezufuhr erfordern.
Cyanrecycling			
▪ AVR	**C**	**TP**	▪ Nicht sehr praktisch bei Schlämmen ▪ Hohe Investitionskosten ▪ Erfordert ausreichend viel wiedergewinnbares Cyan zur Deckung der Betriebskosten im Verhältnis zum gewonnenen Cyan. Freies Cyan ist leicht, danach zunehmend schwieriger bei Zink, Kupfer und Nickelcyanid zurück zu gewinnen. Die Fällung von CuCN verringert die Cyanausbeute ▪ Wird gewöhnlich bei Rückgewinnungsversuchen unter 30 mg/l Cyan unwirtschaftlich. Daher die Notwendigkeit der Entfernung/Zerstörung von Cyan nach AVR [109, Devuyst, 2002]
TP = Einleitung in Absetzbecken SW = Einleitung in Oberflächenwasser C = kommerziell D = Entwicklung			

Table 4.12: Angewendete CN-Behandlungsverfahren

Mehrere andere Optionen für die Cyanausbeute sind in Entwicklung, müssen jedoch noch durch Pilotstudien und industriellen Einsatz bestätigt werden. Das 'Sart-Verfahren arbeitet mit Natriumsulfid in Lösung zur Freisetzung des Cyans aus Zn und Cu, was zur Ausbeute eines Überlaufcyans als Eindicker führt, das direkt dem Recycling zugeführt werden kann. Das 'Hannah-Verfahren' arbeitet nach dem gleichen Prinzip, verwendet jedoch zur Entfernung von Cyan den Ionenaustausch in Lösung oder Pulpe durch Strippen des Cyans vom Harz, danach Ausfällung von Zn und Cu mit Natriumsulfid. Dabei entsteht ein höher konzentrierter Cyanstrom für das Recycling mit der Möglichkeit einer höheren Wiedergewinnungsrate.[109, Devuyst, 2002]

Das SO_2/Luftverfahren, das an allen europäischen Standorten zur Vorbehandlung des Schlamms vor Verkippung in die TMF angewendet wird, wird gewöhnlich durch folgende Reaktionen beschrieben:

Oxidation:

$CN^-_{frei} + SO_2 + O_2 + H_2O => OCN^- + H_2SO_4$
$M(CN)_4^{2-} + 4SO_2 + 4O_2 + 4H_2O => 4OCN^- + 4H_2SO_4 + M^{2+}$
mit $M^{2+} = Zn^{2+}, Cu^{2+}, Ni^{2+}, Cd^{2+}$ usw.

Neutralisation mit Kalk:

$H_2SO_4 + Ca(OH)_2 => CaSO_4 \times 2H_2O$

Fällungsreaktion:

$M^{2+} + Ca(OH)_2 => M(OH)_2 + Ca^{2+}$
$2\ M^{2+} + Fe(CN)_6^{4-} => (M)_2\ Fe(CN)_6$
mit M= Zn, Cu, Ni, Cd, Fe usw.

Die Anwesenheit von Kupferionen katalysiert diese Reaktionen. Es erfolgt Bindung an das Cyan mit Bildung stabiler Kupfer(I)-komplexe, die mit dem INCO-Verfahren durch Oxidation sowohl von Kupfer als auch Cyan zerstört werden können. Je höher die Kupferkonzentration, umso stabiler diese Komplexe. Andererseits erfordert der höhere Kupferanteil im Erz mehr Cyan für die Laugung, und bei Aufrechterhaltung der Wirksamkeit der CN-Zerstörung entsteht eine höhere Konzentration an Restcyan.

Die Rolle des Schwefeldioxids ist nicht vollständig geklärt, es wird jedoch angenommen, dass bestimmte Zwischenverbindungen entstehen, welche die Reaktionen beschleunigen. Am Standort **Bergama-Ovacik** wird Ferrisulfat zur weiteren Stabilisierung von Schwermetallen eingesetzt.

Die Sauerstoffverteilung ist an die Viskosität gebunden. Ist diese hoch, ist weniger Sauerstoff gelöst und die Reaktionskinetik verlangsamt sich.

Die CN-Zerstörung ist in der Lage, die Konzentration von Wad-CN im Schlamm von 140 mg/l auf unter 2 mg/l zu reduzieren, wenn der Kupfergehalt im Erz nicht zu hoch ist. Enthält der Zulauf zur Cyanlaugung mehr als 0,1 % Cu, ist es nicht möglich, diese niedrigen Wad-CN-Werte in den Aufbereitungsrückständen zu realisieren. Bei hohen Kupferkonzentrationen können mehrere Stufen für die CN-Zerstörung erforderlich sein.

Tabelle 4.13 enthält die CN-Konzentrationen für verschiedene Standorte [50, Au group, 2002].

Standort:	**Boliden**	**Ovacik**	**Rio Narcea**
Lauge: Freies Cyan (mg/l) pH	 120	 200 10.5	 400 - 450 (NaCN) 10,5
Messhäufigkeit	täglich	2 h	kontinuierlich on-line
Min.	70	180	
Max.	50	220	
Austritt aus Detoxifikation: Freies Cyan (Wad-CN) Gesamtcyan pH	 0,87	 0,33 0,4 7 - 8	 0 10 - 30 8,5
Messhäufigkeit	1/d SIS-Verfahren, 3/d Pikrinverfahren	2 h	3 h
Min	0,31 (gesamt)	0,06 (Wad)	1 (Wad)
Max	1,94 (gesamt)	0,88 (Wad)	40 (Wad)
In TMF Freies Cyan Wad-CN GesamtCyan pH	 0,3	 0,23 0,39 7 - 8	 0 20 - 30 8,5
Messhäufigkeit	sporadisch	täglich	täglich
Min.	0,05 (gsamt)	0,04 (Wad)	10 (Wad)
Max.	0,74 (gesamt)	0,71 (Wad)	30 (Wad)
TMF-Ablauf: Freies Cyan Wad-CN Gesamtcyan pH	 0,06	kein Ablauf	kein Ablauf, Rückführung Drainage zu Becken 0 0,5 – 1,0 8 – 8,5
Messhäufigkeit	täglich		täglich
Min.	0		0,2 (Wad)
Max.	0,33		2 (Wad)

Tabelle 4.13: CN-Werte von europäischen Standorten mit Cyanlaugung

In der Aufbereitungsanlage Boliden erfolgte im Jahr 2001 die Überwachung der CN-Zerstörung und der Wasserqualität des Abflusses aus Aufbereitungsrückständen und Absetzbecken. Die Ergebnisse zeigten, dass 99,5 % des CN_{frei} zerstört wurden. Ein weiterer Abbau des CN erfolgt natürlich im Absetzbecken. Ähnliche Ergebnisse werden von Ovacik und Rio Narcea berichtet.

Bisher konzentriert sich die Bewirtschaftung von Cyan hauptsächlich auf die Zerstörung des Cyan durch Systeme mit Einzeldurchgang. Es ist jedoch möglich, das Cyan **zurück zu gewinnen und wieder zu verwenden**, wodurch es zur Minimierung des Gesamtcyanverbrauchs und Senkung der Betriebskosten kommt. Wiedergewinnung und Wiedereinsatz von Cyan verringern die CN-Konzentration im Becken und reduzieren die Kosten für die Zerstörung des CN [106, Logsdon, 1999].

CN-Wiedergewinnung und Wiedereinsatz werden seit den 1930er Jahren angewendet. Ein Verfahren mit der Bezeichnung 'AVR' (Acidifizierung/Volatilisierung/Reneutralisation) wird an mehreren Standorten mit Erfolg eingesetzt. Obwohl das Verfahren große Mengen an Säuren und Basen erfordert, verbraucht es weniger Energie als Hydrolyse/Destillation. Auch sind die Volatilisierungsraten höher [104, Young, 1995].

Abschnitt 4.4.15 beschäftigt sich mit Fragen der Cyanbewirtschaftung zu Vermeidung/Verminderung von Unfällen.

4.3.11.9 Durchlässige reaktive Sperren

Durchlässige reaktive Sperren sind durchlässige Zonen, die einen reaktiven Behandlungsbereich für Abfang und Behandlung der Schadstofffahne beinhalten oder bilden. Sie beseitigen Schadstoffe aus dem Grundwasserstrom auf passive Weise durch physikalische, chemische oder biologische Prozesse.

Eine komplette, kontinuierliche, durchlässige reaktive Sperre (PRB) wurde im August 1995 im Gefällebereich einer inaktiven Rückhalteanlage für Bergbauaufbereitungsrückstände am Standort der Nickel Rim Mine in Sudbury, Ontario, Kanada errichtet. Nickel Rim wurde als Grube zwischen 1953 und 1958 betrieben. Vorwiegend wurden Kupfer (Cu) und Nickel (Ni) gewonnen. Oxidationsvorgänge laufen in den Aufbereitungsrückständen seit ca. 40 Jahren ab. Die aus den Aufbereitungsrückständen austretende Grundwasserfahne gelangt in einen nahe gelegenen See. Hauptschadstoffe vor Ort sind Nickel (Ni), Eisen (Fe) und Sulfat. Die Ausgangskonzentrationen lagen bei 2400 - 3800 mg/l Sulfat, 740 - 1000 mg/l Fe und bis zu 10 mg/l Ni.

Die belastete Grundwasser führende Schicht hat eine Mächtigkeit von 3 - 10 m und besteht aus fluvioglazialem Sand. Die Grundwasser führende Schicht ist auf ein schmales Tal beschränkt, das beiderseits sowie darunter von Muttergestein begrenzt ist. Die Fließgeschwindigkeit des Grundwassers innerhalb der Grundwasser führenden Schicht wird auf 15 m/a geschätzt.

Der Einbau der PRB erfolgte quer durch das Tal im Firstenstoßbau. Die Sperre durchquert das gesamte Tal und ist 15 m lang, 4 m tief und 3,5 m breit. Sie besteht aus einem reaktiven Gemisch aus Siedlungskompost, Laubkompost und Holzschnitzeln. Zur Verbesserung der Wasserdurchlässigkeit wurde das Gemisch mit Feinkies versetzt. Sowohl auf der ansteigenden wie auf der abfallenden Seite des reaktiven Materials wurden Pufferzonen aus Grobsand vorgesehen. Zur Minimierung des Eindringens von Oberflächenwasser und Sauerstoff wurde oben auf die Sperre eine Tonlage von 30 cm aufgebracht. Weitere Sanierungsmaßnahmen am Standort der Nickel Rim Mine beinhalteten Sulfatreduzierung und Metallsulfidausfällung durch das Vorhandensein organischer Stoffe.

Entlang dem Transekt paparallel zur Grundwasserströmung wurden Überwachungsbrunnen errichtet. Proben wurden einen Monat und danach erneut neun Monate nach der Errichtung entnommen. Das Durchströmen der Sperre führte zu einer Reduzierung der Sulfatkonzentrationen aus 110 - 1900 mg/l. Die Eisenkonzentrationen fielen auf <1-91 mg/l. Der Anteil an gelöstem Nickel reduzierte sich auf <0,1 mg/l innerhalb und nach der Sperre. Ferner stieg der pH-Wert über die Sperre von 5,8 – 7,0. Insgesamt machte die Sperre die Grundwasser führende Schicht von einer Säure erzeugenden zu einer Säure verzehrenden Schicht. Die Überwachung soll noch für mindestens drei Jahre mit zwei Probenahmen jährlich fortgeführt werden.

Die Kostehn betragen ca. USD 30000. Dies beinhaltet Auslegung, Bau, Material und die reaktive Mischung,
[123, PRB action team, 2003]

An einem finnischen Standort wurde vor kurzem eine PRB aus Kalkstein und Torf in einem offenen Graben um den Steinbruch errichtet. Die Ergebnisse lassen erkennen, dass mit Hilfe dieser Anlage Erstreduzierungen bei Metallen von ca. 90 % erreicht wurden. Mit der Zeit wird es zur Verstopfung der Anlage kommen, und das reaktive Material muss erneuert werden. Die Geschwindigkeit der Verstopfung wird von den Bedingungen, z. B. Konzentration an Metallen und Feststoffen und der Wassermenge, bestimmt. Die Errichtungskosten von Anlagen dieser Art werden auf ca. EUR 100/m^3 geschätzt. Die Kosten der Erneuerung des Materials liegen etwa im gleichen Bereich.

Diese Methode ist anwendbar bei Sanierungsbecken, bei denen mehrere Jahre nach der Stilllegung noch geringe Mengen an Sauerwässern festgestellt werden. Eine alternative passive Behandlungsmethode ist der Einsatz von Feuchtgebieten. PRB-Sperren können für saure und basische Wässer eingesetzt werden, wenn einzelne Schadstoffe durch bakterielle Reduktion beseitigt werden können.

Damit diese Methode Erfolg hat, müssen die Strömungsverhältnisse genau bekannt sein, so dass sicher ist, dass das Wasser auch tatsächlich durch die Sperre fließt.

Die Arbeitsbakterien benötigen einen pH-Wert im Bereich 5 - 7. Der pH-Wert der ARD ist gewöhnlich niedriger, daher ist zur Sulfidausfällung eine Erhöhung notwendig (z. B. durch Zugabe von Kalkstein). Jedoch werden bei zu hohem pH-Wert Metalle ausgefällt, was zu rascher Verstopfung führen kann. Daher muss die PRB zu ihrer Wirksamkeit gut auf das zu behandelnde Abwasser abgestimmt sein.

Die PRB hat eine begrenzte Behandlungskapazität und muss periodisch erneuert werden.

4.3.12 Grundwasserüberwachung

Das Grundwasser wird gewöhnlich im Umkreis aller Flächen mit Aufbereitungsrückständen und taubem Gestein überwacht. Die Höhe des Grundwasserspiegels und die Wasserqualität werden regelmäßig kontrolliert.
[131, IMA, 2003]

Bei einer großen TMF im Kupferrevier Legnica-Glogow besteht das Überwachungsnetz von Grund- und Oberflächenwasser aus über 800 Überwachungsstellen.
[KGHM Polska Miedz, 2002 Nr. 113]

Generell wird der Überwachungsbedarf nicht von der Größe des Absetzbeckens, sondern durch die spezifischen hydrogeologischen Bedingungen am Standort bestimmt. Auf flachem Land errichtete Becken erfordern meist eine größere Zahl von Überwachungsstellen als Becken an Orten mit günstigerer Gestaltung der Grundwasserströmungsverhältnisse.

4.3.13 Nachsorge

4.3.13.1 TMF für roten Aluschlamm

In der Nachsorgephase muss der Oberflächenabfluss vor der Verkippung aufbereitet werden, bis die chemischen Bedingungen eine tolerierbare Konzentration für die Einleitung in Oberflächengewässer haben. Zudem müssen Zufahrtstraßen, Drainageanlagen und Pflanzendecke (einschließlich bei Bedarf von deren Erneuerung) instand gehalten werden. Ferner ist die kontinuierliche Bebrobung der Grundwasserqualität Teil der Durchführung des Stilllegungsprogramms und muss weiter verfolgt werden.
[22, Aughinish,].

4.4 Unfallverhütung

4.4.1 Bewirtschaftung von Aufbereitungsrückständen oder taubem Gestein in Gruben

Um das Einstürzen von Erzhalden zu verhindern, ist der ideale Standort für die Errichtung einer Anlage zur Bewirtschaftung von Aufbereitungsrückständen oder taubem Gestein eine in der

Nähe befindliche geeignete Grube, da in diesem Falle die Stabilität des Damms bzw. der Halde keine Frage ist. Im Allgemeinen finden sich aber solche Standorte nicht in der Nähe der Anlage.

Es ist auch darauf zu achten, dass das Grundwasser nicht verunreinigt ist.

Beim Bauxitabbau wird das taube Gestein zum großen Teil wieder direkt in die abgebauten Tagebaue zurückverfüllt, was den Platzbedarf für die Standfläche verringern hilft und zur Wiederurbarmachung der Gruben beiträgt.

4.4.2 Verlegung des natürlichen Ablaufs

4.4.2.1 bei Becken

Eine Verlegung des natürlichen Ablaufs kann erforderlich werden, um

- den notwendigen Freibord aufrechtzuerhalten,
- eine Verunreinigung des natürlichen Ablaufs mit Prozessflüssigkeiten oder Chemikalien zu verhindern und/oder
- das Wasservolumen in solchen Stauhaltungen zu verringern, die mit Verdunstung arbeiten, um überschüssiges Wasser abzuführen, anstatt es behandeln und ableiten zu müssen.

Zur Verlegung des natürlichen Abflusses werden drei Standardverfahren eingesetzt, deren Auswahl normalerweise von der Topographie des Standortes und der erwarteten Strömungsgeschwindigkeiten abhängt:

- Kanäle oberhalb des Damms und um ihn herum,
- Rohre unter dem Damm und
- Tunnel durch die Seiten des Dammes.

Das Verlegungssystem ist für die Sicherheit des Dammes mit den Aufbereitungsrückständen kritisch. Ein Ausfall an irgendeiner Stelle kann dazu führen, dass Wassermengen in die Stauhaltung eingeleitet werden, für die sie nicht ausgelegt ist, so dass es zum Überlaufen und möglicherweise zur Gefahr des völligen Ausfalls des Dammes kommt. Der Errichtung der Bauten für die Verlegung des natürlichen Ablaufs ist daher bereits bei der Planung höchste Priorität einzuräumen.

Im Allgemeinen gehören zur Auslegung von Aufschichtungen aus Rotschlamm, bei denen eingedickte Aufbereitungsrückstände eingesetzt werden, durchlässige Steindämme an den Außenseiten und die Versiegelung der darunterliegenden Oberfläche. Die Aufschichtung ist normalerweise durch einen Außendamm für das Sammeln des Oberflächenablaufs eingegrenzt.

In **Ovacik** gehört zur Auslegung der TMF eine Rückhaltung für Oberflächenwasser hinter dem vorgelagerten Damm. In **Río Narcea** ist das Becken von Kanälen zum Sammeln und Ableiten des Oberflächenwassers umgeben. Das so gesammelte Oberflächenwasser wird zur Klärung in ein segmentiertes Becken eingeleitet, ehe es von dort abgelassen wird. Auf ähnliche Weise wird das Oberflächenwasser in der Kaolinabbaustätte **Nuria** gesammelt, das viele Feinanteile enthält, wenn es in eine Reihe von Sedimentationsbecken geleitet wird.

Dennoch ist es nicht immer möglich bzw. notwendig, das gesamte Oberflächenwasser aufzufangen. In **Kiruna** belief sich zum Beispiel 2001 die Gesamtmenge des in die Rohstoffaufbereitungsanlage einlaufenden Wassers auf 61 Mio. m^3, wovon 3 Mio. m^3 als Oberflächenwasser aufgefangen wurden. Ein anderes Beispiel ist die **Region Boliden**, wo der Einzugsbereich des Absetzbeckens für Aufbereitungsrückstände 8 km^2 beträgt. Der Zufluss an Oberflächenwasser beträgt schätzungsweise etwa 1 Mio. m^3 in einem trockenen Jahr und 3 Mio. m^3 in einem normalen Jahr. Das Becken nimmt jährlich etwa 4,5 Mio. m^3 Prozesswasser aus der Rohstoffaufbereitungsanlage auf.

Bei TMFs in der Kaliindustrie sollte die Entwässerung der Salzlösung von den Halden so weit wie möglich getrennt von der Oberflächenwasserentwässerung erfolgen.

4.4.2.2 bei Halden

Wasser ist mit großer Wahrscheinlichkeit die größte Ursache für die Instabilität einer Halde für Aufbereitungsrückstände oder taubem Gestein sowie für den Boden darunter, da es zu erhöhtem Porendruck und einer Verringerung der Scherfestigkeit kommen kann. Daher ist alles, was möglicherweise zum Ansteigen des Wassers oder des Porendrucks in einer Halde und ihrem Untergrund führen kann, potenziell eine Ursache abnehmender Festigkeit. Besondere Aufmerksamkeit ist der Entwässerung um die Halde herum zu widmen, um das Einfließen von Grundwasser in die Halde und das Anstauen von Wasser am Fuße der Halde zu verhindern. Bei abfallendem Boden werden die Abläufe normalerweise nahe der Bergseite der Einrichtung gebaut. Um die Kapazität zu berechnen, werden folgende Faktoren berücksichtigt:

- das Einzugsgebiet oberhalb der Ableitung/des Ablaufs,
- das Vorhandensein von Quellen,
- Abläufe/Ableitungen aus der landwirtschaftlichen Nutzung,
- natürliche Oberflächenwasserläufe, die durch die Halde unterbrochen werden.

[130, N.C.B., 1970]

In der Region Boliden sind alle Ablagerungen von taubem Gestein von Ableitungs- und Auffanggräben für die Entwässerung umgeben. Das Sickerwasser kann, wenn erforderlich, vor der Ableitung behandelt werden.

Am Standort Kemi wird ein Teil des Sickerwassers aus der Halde mit taubem Gestein in einem Graben gesammelt und mit anderem Wasser aus Entwässerungseinrichtungen eines Industriestandortes in den Bewirtschaftungsbereich von Aufbereitungsrückstände geleitet, während ein anderer Teil des Wassers aus der Entwässerung direkt in einen nahe gelegenen kleinen Graben eingeleitet wird.
[71, Himmi, 2002]

4.4.3 Vorbereitung des natürlichen Untergrundes unter dem Damm

Der natürliche Boden unter dem Rückhaltedamm (nicht aber notwendigerweise der Boden unter den eigentlichen Aufbereitungsrückständen) wird normalerweise von jeglichem Bewuchs sowie vom Humus befreit, um eine angemessene ‘Gründung’ für das Bauwerk zu schaffen. Diese beräumte Oberfläche muss nach Quellen oder Grundwasser abgesucht werden, die dann durch ein entsprechendes Drainagesystem „behandelt“ werden müssen (zum Beispiel Gräben mit Rohren für die Bodenentwässerung, die von abgestuften Steinen umgeben und durch künstliche Membranen geschützt sind).
[131, IMA, 2003]

4.4.4 Baumaterial für den Damm

Die Hauptüberlegung bei der Auswahl des Baumaterials für den Damm ist, dass das Material geeignet sein muss und unter den jeweiligen Betriebs- und klimatischen Bedingungen nicht an Festigkeit verlieren darf. Zum Beispiel sind horizontale, aus Sand und Steinen errichtete Schichten, die durch das Überfahren mit Lkw und Bulldozern sowie ein weiteres Mal durch Rüttelwalzen verdichtet wurden, unter den meisten Umständen genügend feste Strukturen für die Eindämmung von Aufbereitungsrückstände, selbst solcher, die hydraulisch als wässrige Suspension abgelagert werden.

4.4.5 Absetzen der Aufbereitungsrückstände

Das ordnungsgemäße Absetzen („Einstapeln“) der Aufbereitungsrückstände ist, besonders im nassen Zustand, stets kritisch für die Stabilität des Bauwerks. Nasse Aufbereitungsrückstände werden normalerweise von der Dammkrone so gleichmäßig wie möglich rund um den Damm herum abgeworfen und ausgetragen, um einen "Spülstrand" von Aufbereitungsrückständen gegen die innere Fläche des Rückhaltedamms zu schaffen. Das führt normalerweise dazu, dass sich die gröberen Fraktionen der Aufbereitungsrückstände am nächsten zur Dammaufschüttung absetzen, während sich die Feinpartikel stärker im Trübwasser des Beckens absetzen.
[131, IMA, 2003]

Die Grundlagen des Absetzens von Aufbereitungsrückständen werden in Abschnitt **2.4.2.3**erklärt.

4.4.6 Verfahren zum Bau und zur Erhöhung von Dämmen

Dämme für Aufbereitungsrückstände wurden üblicherweise aus den gröberen Fraktionen der Aufbereitungsrückstände selbst errichtet. Das kann auch immer noch ein angemessener Weg zur Eindämmung des Aufbereitungsrückständeschlamms sein. Dennoch können sich die Qualität des Erzes wie auch die Aufbereitungsmethoden ändern und damit die Eigenschaften der Aufbereitungsrückstände auch. Deshalb ist und bleibt das Qualitätsmanagement eine schwierige und komplizierte Frage über die gesamte Lebensdauer des Betriebs einer Anlage hinweg. Folglich neigt man dazu, zunächst einen Vordamm zu bauen, der dann oft mit Füllmaterial erhöht wird, dessen Qualität während des Dammbaus leichter überwacht werden kann. Trotzdem ist nicht nur die Art Materials wichtig, was für den Damm für die Aufbereitungsrückstände verwendet wird, sondern auch dessen Anordnung und geeignete Verdichtung, um eine langfristige Stabilität zu erzielen.

Die Verfügbarkeit von Material (z. B. geeignete Aufbereitungsrückstände, Füllmaterial) zur Aufschüttung des Damms kann ebenfalls ein Problem sein. Bei gleicher Dammhöhe ist die benötigte Menge an Baumaterial beim Downstream-Dammbauverfahren um vieles größer als beim Upstream-Dammbauverfahren (siehe untenstehende Abbildung 4.19).

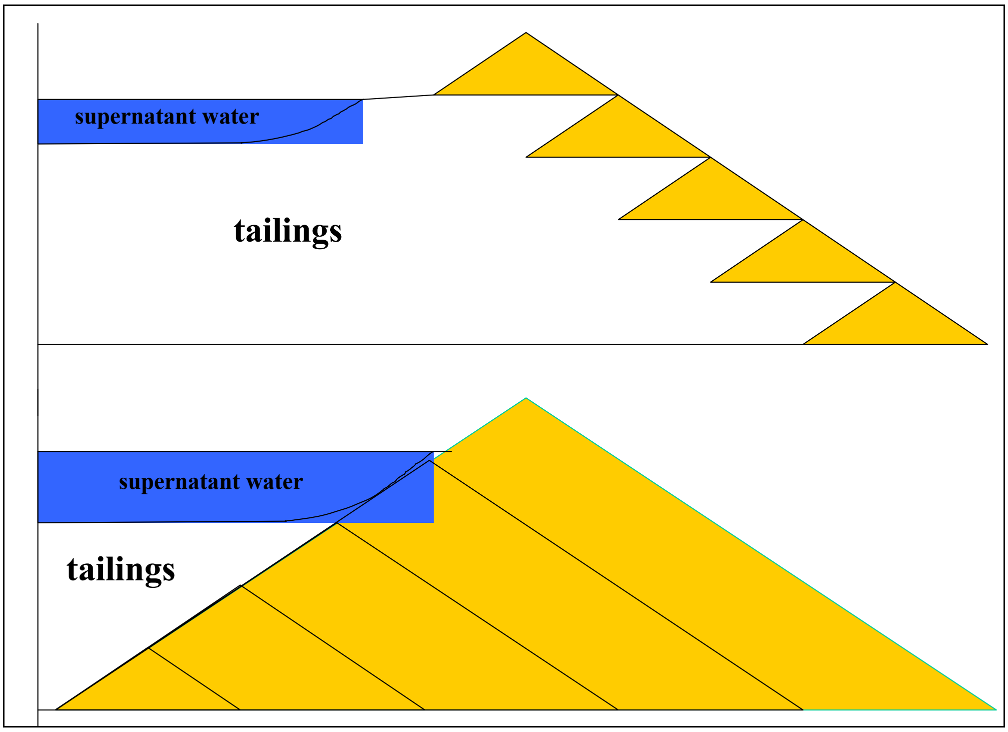

Abb. 4.19: Schematischer Vergleich des Upstream-Dammbauverfahren mit dem Downstream-Dammbauverfahren

Wenn das Dammbaumaterial in Entnahmegruben abgebaut werden muss, erhöht sich der Platzbedarf für die Standfläche der Grube und es müssen für das Downstream-Dammbauverfahren größere Mengen an Baumaterial in die TMF geschafft werden.

Tabelle 4.14 fasst die unterschiedlichen Möglichkeiten für den Bau und die Erhöhung von Dämmen für Aufbereitungsrückstände zusammen.

Dammart	Anwendbarkeit	Eignung für eine Ableitung	Eignung für die Wasserhaltung	Einschränkung bei Erhöhung	Baumaterial	Erdbebenfestigkeit	Damm-Kosten
Konventioneller Damm bzw. Wasserrückhaltung	Geeignet für alle Arten Aufbereitungsrückstände	Geeignet für alle Ableitungsverfahren	Gut	Nicht von den Eigenschaften der Aufbereitungsrückstände abhängig	Füllmaterial aus natürlichem Boden	Gut	Hoch
Upstream-Dammbauverfahren	Bei Einsatz von Aufbereitungsrückständen: mind. 40 - 60 % Sand (0,075 -4 mm) in allen Aufbereitungs-rückständen[1]. geringe Pulpdichte zur Förderung der Korngrößen wünschenswert	Periphere Ableitung und brunnengesteuerter Spülstrand notwendig, mittige Ableitung für eingedickte Aufbereitungsrückstände	Unter bestimmten Bedingungen geeignet	Weniger als 5 m p.a. am besten, um ungenügende Verfestigung und Aufbau von Porendruck zu vermeiden	Natürlicher Boden, Sand von Aufbereitungsrückständen oder taubem Gestein zusammen mit natürlichem Boden oder taubem Gestein	Schlecht in stark erdbebengefährdeten Gebieten	Gering
Downstream-Dammbauverfahren	Geeignet für jede Art von Aufbereitungsrückständen	Schwankt je nach der Auslegung	Gut	Keine	Sand, Aufbereitungsrückstände oder Grubenabfall, wenn genügend vorhanden, sonst natürlicher Boden.	Gut	Hoch
Centreline-Dammbauverfahren	Sande oder Feinstoffe mit geringer Plastizität	Periphere Ableitung notwendig	Nicht für ständige Lagerung empfohlen, zeitweilige Flutaufnahme akzeptabel bei entsprechender Auslegung	Eventuell Höhenbeschränkung für einzelne Niveauanhe-bungen	Sand von Aufbereitungsrückständen oder taubem Gestein, wenn ausreichend vorhanden, sonst natürlicher Boden	Akzeptabel	Mittel
1.) nicht bei eingedickten Aufbereitungsrückständen							

Tabelle 4.14: Vergleich der Dammbauverfahren
[11, EPA, 1995]

Die Grundlagen der Dammbauverfahren sind in Abschnitt 2.4.2.2 vorgestellt worden.

4.4.6.1 Herkömmliche Dämme

Der Vorteil der Nutzung eines **konventionellen, bereits in voller Höhe errichteten Dammes** vor Beginn des Absetzens von Aufbereitungsrückständen besteht darin, dass der Damm in einem nur kurzen Zeitraum errichtet wird, während dessen auch die Qualitätskontrolle leichter durchzuführen ist. Diese Dämme sind aber oft teuer, weshalb man üblicherweise auf im Upstream-Dammbauverfahren errichtete Dämme zurückgekommen ist. Bei diesem Bauverfahren kommt es auf eine ständige Überwachung und Einschätzung an.

Diese Art Damm findet Anwendung, wo
- die Aufbereitungsrückstände nicht für den Dammbau geeignet sind,
- für die Haltung des Wassers bei Anlagen und anderen Verwendungszwecken eine Stauhaltung erforderlich ist, normalerweise saisonal,

- sich der Standort für die Bewirtschaftung der Aufbereitungsrückstände an einer entfernten und schwer zugänglichen Stelle befindet,
- das Wasser aus den Aufbereitungsrückständen zwecks Zersetzung toxischer Elemente (z. B. Cyan) über einen größeren Zeitraum hinweg zurückgehalten werden muss und/oder
- der natürliche Zufluss in die Stauhaltung entweder groß ist oder stark schwankt und zu dessen Kontrolle eine Wasserhaltung benötigt wird.

Vorteile:
- Der Damm wird unter Beaufsichtigung in relative kurzer Zeit errichtet.
- Während des Betriebs wird kaum Überwachung benötigt.
- Schutz gegen Verunreinigung durch Wasser- und Winderosion.

Nachteile:
- Hoher Kapitalbedarf, ehe die Anlage betriebsbereit ist.
- Alle Baumaterialien müssen herangeschafft werden, sofern zur Auffüllung des Dammes kein taubes Gestein aus der Grube verwendet werden kann.

Abgestufte konventionelle Dämme sind undurchlässig und werden während der gesamten Lebensdauer der TMF erhöht. Ein Nachteil im Vergleich mit konventionellen Dämmen ist, dass sich ihr Bau über einen viel längeren Zeitraum hinzieht, was auf Grund von Veränderungen beim Personal und den Auftragnehmern zu Qualitätsverlusten führen kann, da die Aufrechterhaltung einer gleichmäßigen Qualitätskontrolle schwierig ist.

4.4.6.2 Das Upstream-Dammbauverfahren

Das Upstream-Dammbauverfahren ist die billigste Methode, denn es wird für eine bestimmte Höhe das wenigste Baumaterial benötigt. Der größte Nachteil dieses Verfahrens besteht in der physikalischen Stabilität und der Anfälligkeit des Dammes gegenüber einer Verflüssigung. In der Entwurfsphase muss der Kontrolle des Grundwasserspiegels Aufmerksamkeit geschenkt werden, was durch die Schaffung eines ausreichend breiten Spülstrandes sowie die richtige Entwässerung und einen ordnungsgemäßen Betrieb erreicht werden kann. Das für den Bau des Dammes eingesetzte Material sollte kein ARD-Potential enthalten.

Beachten Sie, dass sich das in Tabelle 4.14 erörterte Upstream-Dammbauverfahren auf die Bewirtschaftung konventioneller anstatt eingedickter Aufbereitungsrückstände bezieht. Der Vorschlag, dass die Aufbereitungsrückstände einen 40-60-prozentigen Anteil an Sandfraktionen haben müssen, ist bei eingedickten Aufbereitungsrückständen nicht erforderlich. Zum Beispiel sind bei Rotschlamm-Aufbereitungsrückständen, für die das Upstream-Dammbauverfahren eingesetzt wird, oft schon die Sandfraktionen getrennt, die dann in der Mitte der Aufbereitungsrückstände deponiert werden. Daher sind die am Außenrand zu Stabilitätszwecken analysierten Aufbereitungsrückstände nur die feinen Schluff- bzw. Schlickfraktionen.

Tabelle 4.14 findet auch auf Dämme Anwendung, bei denen es zu einer jährlichen Erhöhung in Größenordnungen von 4 bis 5 m kommt. Bei Rotschlamm-Aufbereitungsrückständen dürfte diese Größenordnung nur 1 bis 2 m pro Jahr betragen. Die Eignung zur Ableitung bezieht sich auf die periphere Ableitung beim konventionellen Anstauen und auf die mittige Ableitung der Aufbereitungsrückstände, wenn sie eingedickt sind.

Jedoch mag selbst die Anwendung des Upstream-Dammbauverfahrens noch vorteilhafter als andere Verfahren, besonders das Downstream-Dammbauverfahren, sein, da nämlich der Grundwasserspiegel dazu neigt, niedrig zu bleiben. Abb. 4.20 veranschaulicht das durch einen Vergleich eines Dammes, der nach dem Upstream-Dammbauverfahren aus im Zyklon eingedickten Aufbereitungsrückständen gebaut wurde, und eines Dammes zur Rückhaltung von Wasser mit einem undurchlässigen Kern, der im Downstream-Dammbauverfahren errichtet wurde.

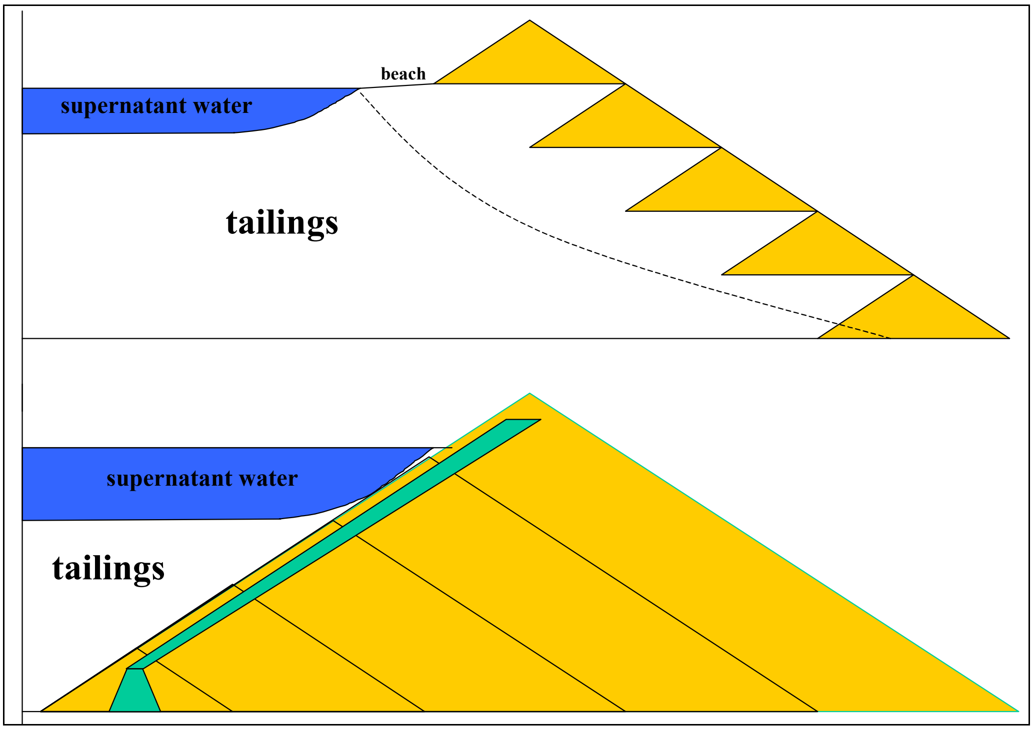

Abb. 4.20: Vereinfachter Vergleich des Grundwasserspiegels für den Dammbau für Aufbereitungsrückstände beim Upstream-Dammbauverfahren und beim Downstream-Dammbauverfahren
supernatant water – aufschwimmendes Wasser
tailings – Aufbereitungsrückstände

Beachten Sie, dass es sich um eine vereinfachte Darstellung handelt. Der im Upstream-Dammbauverfahren gebaute Damm sollte eine Böschung von weniger als 1:3 im Vergleich mit dem Downstream-Dammbauverfahren haben und der Spülstrand sollte breiter als die Höhe des Dammes sein. Der stufenförmig konstruierte konventionelle Damm mit seinem Kern davor sollte mit den dargestellten Filtern und Ablässen ausgerüstet sein, denn dort fließt das Wasser heraus.

Das Upstream-Dammbauverfahren ist möglicherweise für eine Nassabdeckung ungeeignet (siehe Abschnitt **4.3.1.2.1**), wenn das Klarwasser zu schnell abläuft und die Stauhaltung überschwemmt. Andererseits kann das Upstream-Dammbauverfahren zu einer geeigneten Dammstruktur führen, die das Wasser auf Grund der hohen Stabilität des Dammes und seines niedrigen hydraulischen Gradienten zurückhält.

Für die Wasserspeicherung müssen folgende Bedingungen erfüllt sein:
- Vermeidung des Überlaufens und Aufrechterhaltung eines ausreichenden Freibords (siehe Abschnitt 4.4.8),
- Bereitstellung ausreichender Ableitungskapazitäten für den Notfall (bei Sturmereignissen, zu denen es in Schweden alle 10 000, in Österreich und Deutschland alle 100 Jahre kommt [siehe Abschnitte 4.4.9 und 4.4.10]),
- zur Vermeidung einer Verflüssigung den Fuß des Dammes ungesättigt halten,
- gute Regelung des Wasserstandes im Becken (in Bezug auf die Wasserbilanz),
- Überwachung des Grundwasserspiegels innerhalb des Dammes zur Absicherung, dass die beabsichtigten Ergebnisse erreicht werden.

Im Allgemeinen wird eine Stabilitätsberechnung zur Bestimmung des Grundwasserspiegels verwendet.

In jedem Falle muss auch ein Gutachten eingeholt und eine Bewertung durch Experten vorgenommen werden, z. B. die unabhängige Überarbeitung der Auslegung, wofür die Genehmigungsbehörden eine gutachterliche Beurteilung verlangen.

4.4.6.3 Das Downstream-Dammmbauverfahren

Wie aus obenstehender Abbildung 4.20 ersichtlich, hält der undurchdringliche Kern beim Downstream-Dammmbauverfahren das Klarwasser an seinem Platz. Nachlassende Dichte des Kerns kann die Stabilität des Dammes gefährden.
Wird Füllmaterial eingesetzt, kann ein möglicher negativer intermedialer Effekt sein, dass im Vergleich zum Upstream-Dammbauverfahren viel größere Mengen aus den Entnahmegruben abgebaut werden müssen, um den gleichen Höhenzuwachs zu erreichen.

4.4.6.4 Das Centreline-Dammmbauverfahren

In vielen Fällen scheint das **Centreline-Dammmbauverfahren** ein guter Kompromiss zwischen den seismischen Risiken und den Kosten zu sein. Bei Verwendung dieses Dammbauverfahrens verringert sich bei Dammerhöhungen weder das verfügbare Oberflächenareal noch die damit verbundene Staukapazität (siehe Abbildung 3.9)

4.4.7 Bewirtschaftung von Klarwasser

Bei einem durchlässigen Damm wird das Klarwasser normalerweise weit entfernt von der Dammkrone gehalten, um das Böschungsgefälle niedrig zu halten [131, IMA, 2003].

4.4.7.1 Ableiten des Klarwassers

Das Standardverfahren für die Ableitung des Klarwassers ist in Abschnitt **2.4.2.4** dargestellt worden.

In Aitik wird das Wasser über einen **Überlauf** und einen mit Stahl ausgekleideten Umlaufkanal abgeleitet, die sich talseitig am Damm befinden. Künftig wird für die Ableitung des Wassers ein System **offener Kanäle** auf natürlichem Boden benutzt, was den Umlaufkanal durch den Damm hindurch unnötig macht. Die meisten anderen Metallbergwerke in Nordeuropa benutzen ebenfalls diese Art Konstruktion (z. B. Pyhäsalmi, Hitura, Zinkgruvan, Kiruna, Malmberget).

Es ist nicht möglich, für ein weidekoppelartiges Absetzbecken einen offenen Kanal auf natürlichem Boden zu bauen.

Einlaufbauwerke (Mönche) haben sich unter Frostbedingungen mit einer positiven Wasserbilanz gut bewährt. Sie müssen aber so ausgelegt sein, dass sie dem Druck der Aufbereitungsrückstände über die gesamte Lebensdauer der Anlage hinweg standhalten. Da das Wasser durch die Erdanziehungskraft fließt, werden keine Pumpen benötigt, so dass keine ständige und sichere Energieversorgung erforderlich ist (wie es bei Pumpen der Fall wäre). Ein Nachteil dieser Methode besteht darin, dass der Umlaufkanal den Damm perforiert und ihn damit schwächt.

In Ovacik wird eine Abart des Mönchs eingesetzt, die am besten als **'Einlaufbrunnen'** beschrieben werden kann. Das Klarwasser wird über einen fast mittig im Becken angeordneten Einlaufbrunnen abgezogen. Das Einlaufsystem besteht aus einer perforierten Rohrleitung, die von einer Steinpackung umgeben ist (siehe Abbildung 4.21 unten). Dabei handelt es sich um ein dauerhaft angeordnetes und leicht zugängliches System. Anders als bei Ablauftürmen durchdringt kein Rohr den Damm. Das geklärte Wasser wird in die Rohstoffaufbereitungsanlage gepumpt.

Dieses System kann in kleinen Anlagen eingesetzt werden, die in trockenem Klima arbeiten, die kein Wasser ableiten und wo ein hoher Freibord aufrechterhalten wird. Es ist auch hier notwendig, Oberflächenwasser abzuleiten.

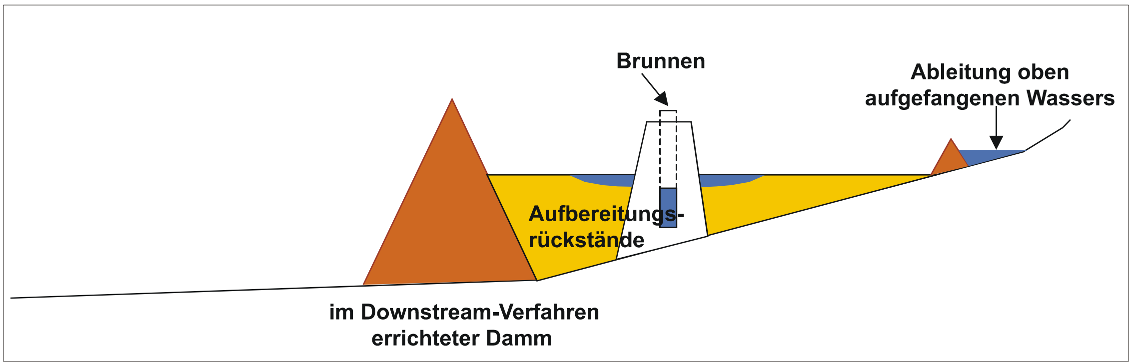

Abb. 4.21: Einlaufbrunnen am Standort Ovacik

Der Kies um den Turm dient als Stabilisator und Wasserfilter und hält die Feinpartikel zurück. Ein weiteres Merkmal dieses Systems sind die Entwässerungsrohre, die grätenartig auf dem Boden des Beckens angeordnet und mit dem Turm verbunden sind, um das abgesetzte feste Material abzuleiten und zu verfestigen.

Bei einem kleinen Becken in Ovacik wurde ein System mit Schuten bzw. Pramen für ungeeignet gehalten, weil das die häufige Bewegung der Schuten bedeutet hätte, um das Klarwasser abzupumpen, da die Austragungspunkte öfter geändert werden.

Ein Nachteil ist, dass der hohe Kiesanteil für das Festigen und Filtern bereits einen Großteil des Dammes füllt. Seine Filterkapazität ist auch beschränkt und mag bei großen Mengen weder praktikabel noch effizient sein.

4.4.8 Freibord

In den Eisenerzgruben Kiruna und Malmberget beträgt der Freibord der Dämme mit den Aufbereitungsrückständen bei zwei Anlagen 2 m und bei einer dritten 1,2 m. Der Freibord beruht auf schwedischen Richtlinien für Wasserrückhaltedämme (RIDAS) und schließt Niederschlags- und Oberflächenwasser sowie erhöhte Wasserstände durch Wellenbildung ein. Bei einem Damm der Klasse 2 und einem 24-stündigen aller 100 Jahre vorkommenden Regensturm sollte die Ableitung ohne Anstieg des Wasserstandes erfolgen können. Die Einleitung der Aufbereitungsrückstände in das Becken wird durch ein relativ konstantes Betriebssystem kontrolliert, das einen konstanten Fluss der Aufbereitungsrückstände erzeugt.

In der Goldgrube Ovacik schafft die Auslegung der TMF einen Freibord von mindestens 2 m.

Im Bereich der Industrieminerale beträgt der Freibord mindestens 1 m um zu gewährleisten, dass das Becken zusätzlich zur normalen Zuführung des Prozesswassers stets eine plötzliche Überflutung auffangen und mit ihr fertig werden kann (siehe Abschnitt 4.4.10).
[131, IMA, 2003]

Bei der TMF in der Kupfergrube Legnica-Glogow wird ein Freibord von mindestens 1,5 m vorgehalten.

Entsprechend dem “Merkblatt für Dammsicherheit” wird der Freibord für Dämme mit hohem Risiko von der größtmöglichen Höhe der Wellen bzw. von der Tiefe der Frostgrenze abgeleitet [129, Finnland, 1997].

4.4.9 Ablassen in Notfällen

Die Auslegung der Absetzbecken für Aufbereitungsrückstände und die Ablasseinrichtungen ziehen alle vorhersehbaren extremen Ereignisse in Betracht, wozu extreme Niederschläge und Schneeschmelzen gehören. Dennoch wird das Risiko weiter durch den Einbau von Notablässen verringert. Notablässe sind für den automatischen Betrieb ausgelegt: Wenn das Wasser einen vorher bestimmten kritischen Pegelstand erreicht, wird das überschüssige Wasser (das nicht durch die normalen Ableitungsvorrichtungen ausgetragen werden kann) ohne Beeinträchtigung der Integrität des Dammes abgeleitet. Auf diese Weise verhindern Notablässe übermäßig erhöhte Wasserstände innerhalb des Dammes bzw., in einem extremen Szenario, das Überlaufen des Dammes, was zu einem katastrophalen Ausfall des Dammes führen könnte.

Das Nichtvorhandensein solcher Notablässe im Design des Absetzbeckens für Aufbereitungsrückstände in Baia Mare war der Grund für die dortige Katastrophe. Hätte es Notablässe gegeben, wären nur kleine Mengen an CN-haltigem Wasser und überhaupt keine Aufbereitungsrückstände freigesetzt worden.

Das am weitesten verbreitete System besteht aus einer Anzahl von großdimensionierten Rohren (damit sie nicht blockiert werden können), die durch den Deich geführt werden. Diese Rohre werden auf einem solchen Pegelstand eingebaut, dass damit der vorbestimmte Mindest-Freibord stets eingehalten wird. Zur Verhinderung von Erosion am Austragende dieser Ablässe wird diese Anordnung verwendet, da sie das Erosionsrisiko des Dammkörpers auch unter extremen Bedingungen ausschaltet.

Als Alternative dazu können Überläufe auch kontrolliert über den Dammkörper oder konstruiert in natürliches Terrain vorgenommen werden, wobei die zweite Möglichkeit nur für Dämme in talartigen Gegenden gilt. Bei solchen Systemen ist der Erosionsschutz von ausschlaggebender Bedeutung.

4.4.10 Bestimmung des Bemessungshochwassers für Absetzbecken

Im Rahmen der Richtlinien für Wasserrückhaltedämme (RIDAS) (siehe **Tabelle 4.1** und **Tabelle 4.2**) wird, ähnlich wie bei der Vorgehensweise bei einer vermutlich höchsten Flut (PMF), für Dämme, deren Ausfall verheerende Konsequenzen haben kann (Klasse 1), eine deterministische Herangehensweise vorgeschlagen, wobei der Schwerpunkt auf dem kritischen Timing der die Flut hervorbringenden Faktoren liegt. Der Input für den Niederschlag beruht jedoch nicht auf Schätzungen über die vermutlich höchste Niederschlagsmenge (PMP), sondern eher auf einer Einschätzung der (bisher) beobachteten maximalen Regenfälle. Für einen Damm mit geringem Gefahrenpotential wird die aller 100 Jahre auftretende Flut bzw. Überflutung als Bemessungsgrundlage benutzt. Zu den typischen Maßnahmen bei dieser Herangehensweise gehören die Vergrößerung der Überlaufkapazität, um extreme Zuflussmengen sicher ableiten zu können, wobei eine zeitweilige Speicherung über dem normalen Hochwasserpegel durch die Erhöhung der Dammkrone ermöglicht wird. Diese Richtlinien sind für die Bedingungen bei Wasserkraftwerken ausgearbeitet worden, wo es normalerweise große Einzugsbereiche gibt. Bei Dämmen für Aufbereitungsrückstände sind die Einzugsbereiche hingegen oft ziemlich klein, so dass diese Richtlinien diesbezüglich fortgeschrieben werden müssten.
[115, Mill, 2001]

Entsprechend dem finnischen ‘Merkblatt für Dammsicherheit’ bestimmt die Gefahren-risikoklasse eines Dammes den Wert für das Bemessungshochwasser. Bei Dämmen der höchsten Risikokategorie (P) beruht das Bemessungshochwasser für die Dimensionierung der

Überläufe auf einem Zeitraum von 5000 – 10000 Jahren, in denen sich eine solche Flut wiederholen kann, bei den beiden 'niedrigeren' Kategorien (N, O) auf Zeiträumen von 500 - 1000 bzw. 100 - 500 Jahren.

Die Wahl des Verfahrens zur Bestimmung der Bemessungshochwasserkapazität hängt in erster Linie von den vorhandenen hydrologischen Daten ab.

4.4.11 Entwässerung von Dämmen

4.4.11.1 Durchlässige Dämme

Wird ein Damm ohne jegliches internes Entwässerungssystem gebaut, treten Bedingungen wie in 4.22 a) dargestellt auf. In der Praxis wird das Austreten von Sickerwasser aus der Außenböschung und eine Sättigung des äußeren Dammfußes vermieden, da dies zur Laugung und damit zur Instabilität des Dammes führt, wenn die Böschung nicht ausgesprochen flach ist.

Durchlässige Dämme beruhen auf dem Prinzip, wonach das Sickerwasser weit vor dem Fuß der Außenböschung nach unten abgeflossen sein sollte, was durch die Installation eines internen Drainagesystems erreicht werden kann, wobei sich die Drainagezone im inneren Abschnitt des Dammes befindet.

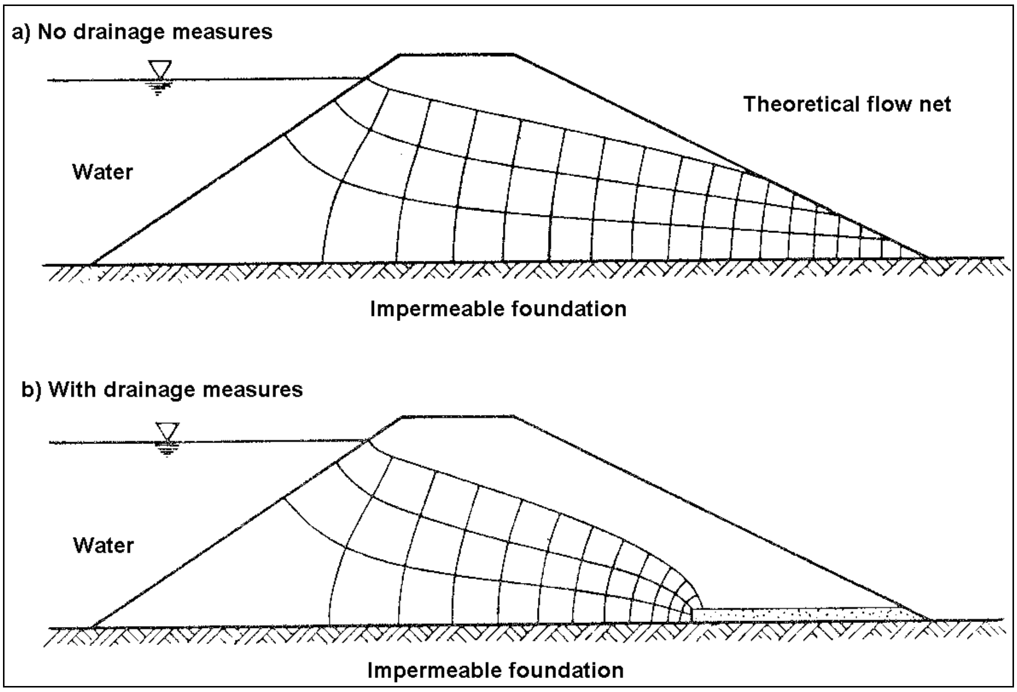

Abb. 4.22: Damm ohne und mit Drainagesystem
[130, N.C.B., 1970]
No drainage measures – keine Entwässerungsmaßnahmen
Theoretical flow net – theoretisches Strömungsnetzwerk
Water – Wasser
Impermeable foundation – undurchlässiger Untergrund
With drainage measures – mit Entwässerungsmaßnahmen

Es muss darauf geachtet werden, dass das Entwässerungssystem (manchmal auch als der Filter bezeichnet) nicht durch die Aufbereitungsrückstände blockiert wird.

Die Grundwasserbedingungen sind ebenfalls zu berücksichtigen. In einigen Fällen kann es notwendig werden, ein Drainagesystem zu konstruieren, das sowohl für das Grundwasser als auch die Entwässerung des Absetzbeckens ausgelegt ist.
[130, N.C.B., 1970]

Figure 3.6 zeigt ein Beispiel für einen durchlässigen Damm mit Drainagesystem.

Beim Damm von Kernick Mica ist der Sand aus den Aufbereitungsrückständen und dem tauben Gestein zum Bau spezifischer Zonen des Dammes verwendet worden, die durch Übergangsschichten getrennt werden. Das taube, gleichmäßig zwischen 50 mm und 750 mm klassierte Gestein bildet einen zentralen Kern für das Sammeln und Ableiten des Sickerwassers durch die Dammstruktur. Der Sand in den Aufbereitungsrückständen, der keine Korngrößen von mehr als 150 mm, sondern normalerweise von unter 25 mm enthält, wird zur Errichtung von Teilen des Hauptdamms sowohl im Downstream- als auch im Upstream-Dammbauverfahren verwendet. Die Übergangsschichten, die aus sauberem, gebrochenem Gestein mit einer üblichen Größe zwischen 75 mm und 125 mm bestehen, bilden eine Filterschicht zwischen dem Sand der Aufbereitungsrückstände und dem Kern aus taubem Gestein.

4.4.11.2 Undurchlässige Dämme

Es sollte beachtet werden, dass undurchlässige Dämme auch ein Entwässerungssystem ähnlich dem in Abbildung 4.22 dargestellten haben. In diesem Falle hat der Filter die Aufgabe, das durch den Kern fließende Sickerwasser davon abzuhalten, den Kern und die Außenböschung des Dammes zu erodieren. Einen für diese Art Damm typischen Filter zeigt Abbildung 3.14.

4.4.12 Überwachung des Sickerwassers

Durch den Damm dringendes Sickerwasser, wie in Abschnitt **2.4.2.5** dargestellt, sollte nicht als negativ per se betrachtet werden. Es ist wichtig, dass ein kontrollierter Abfluss des Sickerwassers durch den Damm erfolgt, um dessen Stabilität zu gewährleisten, indem der Porendruck auf den Damm verringert wird. Dennoch ist es sowohl unter dem Gesichtspunkt des alltäglichen Umweltschutzes als auch der Unfallverhütung wichtig, das Sickerwasser gut zu kontrollieren und richtig zu bewirtschaften.

Die Kontrolle des Sickerwassers dient dem Management jeder Art von Dammkonstruktion. Die Überwachung des normalen Sickerwasserflusses durch Damm und ein gutes Verständnis der darum herum ablaufenden Prozesse gleichermaßen (Meteorologie, Wasserstand im Absetzbecken etc.) geben rechtzeitig Hinweise darauf, ob es am Damm zu Problemen kommen kann. Ein verstärkter Durchfluss in Kombination mit schwebenden Partikeln im Sickerwasser deutet auf den möglichen Beginn einer inneren Erosion, während abnehmender Durchfluss auf ein mögliches Verstopfen der Entwässerung/der Filter hindeutet.

Auf Grund des jeweiligen hydraulischen Gradienten (hydraulischer Druckunterschied) zwischen dem Absetzbecken und seiner Umgebung kommt es zum Austritt von Sickerwasser, nicht nur durch den Damm, sondern auch in das natürliche Erdreich, das für die Rückhaltung der Aufbereitungsrückstände benutzt wird. Die Unterschiede in den hydrologischen Bedingungen zwischen den Standorten macht eine spezielle Evaluierung jedes einzelnen Standortes notwendig. Je nach Ergebnis dieser hydrogeologischen Untersuchung und der Notwendigkeit des Auffangens des Sickerwassers sind verschiedene Schutz- und Auffangmöglichkeiten verfügbar. In vielen Fällen wird eine Kombination der verfügbaren Möglichkeiten bevorzugt.

In Abschnitt **4.3.10** wird die Sickerwasserkontrolle unter ökologischen Gesichtspunkten erörtert.

4.4.13 Damm und Haldenstabilität

Die Stabilität der Böschungen von Dämmen und Halden ist abhängig von bestimmten Faktoren wie:

- der Reibungswinkel, die Sättigung des Wassers, der Grundwasserspiegel, der Porendruck,
- die Geometrie des Querschnitts,
- die Festigkeitsparameter des Materials (Scherfestigkeit im Vergleich mit der Scherbeanspruchung) und seiner Gründung sowie der sich daraus ergebende Sicherheitsfaktor.

4.4.13.1 Sicherheitsfaktor

Der Sicherheitsfaktor einer Böschung wird als Verhältnis zwischen verfügbarer Scherfestigkeit und der für das Erhalten des vorhandenen Gleichgewichts erforderlichen Scherbeanspruchung definiert.
[75, Minorco Lisheen/Ivernia West, 1995]

Entsprechend dem finnischen "Merkblatt für Dammsicherheit" sollte der Gesamtsicherheitsfaktor von Dämmen im Zustand ständigen Sickerwasserflusses mindestens 1,5 betragen. Im Endstadium des Baues und als Sicherheit gegen ein plötzliches Abfallen des Wasserstandes sollte der Gesamtsicherheitsfaktor nicht unter 1,3 sinken.
[129, Finnland, 1997]

Die Stabilität der beiden Dämme am Standort Zinkgruvan ist durch externe Spezialisten untersucht worden, die Sicherheitsfaktoren von 1,5 und 1,6 feststellten.

Die Halden für taubes Gestein beim finnischen Talkumunternehmens sind mit einem Sicherheitsfaktor von mindestens 1,3 ausgelegt.

Während des Betriebs der Goldgrube von Bergama-Ovacik, bei dem Abraum und taubes Gestein auf der im Downstream-Dammmbauverfahren errichteten Böschung des Hauptdammes deponiert wurden, änderte die Böschung ihre Neigung auf weniger als 10˚, wodurch sich der Sicherheitsfaktor der Dammkonstruktion im Vergleich zu dem für Wasserrückhaltedämme international üblichen Faktor von 1,2 auf 2,23 erhöhte.

In Deutschland werden Wasserdämme und Absetzbecken für Aufbereitungsrückstände entsprechend den Industrienormen EN DIN 19700 T10-15 und DIN 4084 errichtet, wobei ein Sicherheitsfaktor je nach den unterschiedlichen Arten der Bemessungslasten zwischen 1,3 und 1,4 gefordert wird. Zusätzliche Belastungen (wie Lkw-Verkehr auf dem Damm, Schnee usw.) müssen zusätzlich berücksichtigt werden.

Entsprechend der von der Österreichischen Kommission für Großdämme ausgearbeiteten Richtlinien muss der Sicherheitsfaktor für die Stabilität von Böschungen bei normaler Belastung mindestens 1,3 betragen. Zur Sicherheit gegen statische Bodenverflüssigung wird ein Sicherheitsfaktor von mindestes 1,5 verlangt. Die Sicherheit gegen interne Erosion, die Berechnung erwarteter Beanspruchungen und Bewegungen, die Stabilität gegen langfristiges Altern sowie Aspekte der dynamischen Bodenverflüssigung (z. B. auf Grund von Erdbeben) müssen ebenfalls berücksichtigt werden. Bei all diesen Untersuchungen kommt es darauf an (z. B. durch Labor- und Feldtest), von konservativen und gesicherten Annahmen für die örtlichen geotechnischen Parameter sowie für die Materialien und Gründungen auszugehen.
Wie bereits in Abschnitt 4.2.4 über langfristig stabile Dämme erwähnt, wo ein Verfahren mit Wasserabdeckung zur Anwendung kommt, wird ein Sicherheitsfaktor von 1,5 normalerweise für ausreichend gehalten.

4.4.13.2 Stabilität von Halden mit Kaolin-Aufbereitungsrückständen

Folgende Kriterien gelten für den Bau einer stabilen Halde mit Kaolin-Aufbereitungsrückständen:

- Die Einstapelung erfolgt auf einer entwässerten, humusfreien Oberfläche, um ein Abrutschen einzuschränken.
- Das Material muss vor dem Einstapeln ausreichend getrocknet worden sein, was Eindicken voraussetzt.
- Es können Partikelgrößen bis 80 µm abgesetzt werden.

Zur Erhöhung der Sicherheit in TMFs ist es notwendig, eine detaillierte Studie zur Stabilität des Untergrundes, zur vorgesehenen Höhe, zur Grundwassersituation, zu den langfristigen Wetterbedingungen und zur vorgesehenen Zusammensetzung der Aufbereitungsrückstände (Art, Korngrößen, Prozentsätze usw.) zu erarbeiten.

Das Absetzen und Einstapeln kann nach der Vorbereitung des Untergrundes (Entfernen der Humusschicht sowie schwacher und weicher Schichten) schichtweise beginnen, wobei die Endböschung sofort und daran anschließend ausgeformt wird. In direkter Berührung mit dem Unterboden eingestapeltes taubes Gestein muss zur Sicherung der Durchlässigkeit grobkörnig sein (gesprengtes Gestein). Geneigte untenliegende Böschungen werden terrassenförmig ausgeformt, um ihre Stabilität zu erhöhen. Das Sickerwasser von den Halden wird abgeleitet.
[131, IMA, 2003]

4.4.13.3 Stabilität von Dämmen mit Kalk-Aufbereitungsrückständen

Zum Genehmigungsverfahren für die TMF des Kalksteinbergwerkes Münchehof gehörte nach DIN 19700 T 10 ein Stabilitätsnachweis für den Damm unter statischen und hydraulischen Gesichtspunkten.

Die Stabilitätsberechnung erfolgt mit folgenden Elementen:

- geotechnisches und hydrogeologisches Modellieren,
- Böschungsstabilität,
- Scherfestigkeit,
- Ausfallsicherheit des Unterbaus,
- Sicherheit gegen den Aufbau von Porendruck im Fundament,
- Überlaufen und Erosionsstabilität.

Eine andere wichtige Anforderung für die Dammstabilität ist die Eignung des für den Dammbau eingesetzten Materials, was in geotechnischen Tests untersucht wird. Dabei werden die folgenden Parameter geprüft:

- Reibungswinkel,
- spezifische Dichte,
- Kompressibilität und
- Wassergehalt.

Das Qualitätsmanagement während der Bauphase diente der Absicherung der Einhaltung dieser für die Dammstabilität so wichtigen Kriterien und erstreckte sich auf die Dammgründung, den Dammkörper sowie den Dammkern.
[108, EuLA, 2002]

4.4.14 Verfahren zur Überwachung der Damm- und Haldenstabilität

4.4.14.1 Erstellung eines Überwachungsplanes

Zur Beobachtung der Stabilität eines Dammes gehören die Überwachung der Messgeräte (entweder online oder in bestimmten Abständen), Inspektionen (täglich/wöchentlich/monatlich) und detaillierte Kontrollen/Überprüfungen in größeren Abständen (zwischen 1 und 20 Jahren).

Ein solcher Überwachungsplan wird auf Grundlage einer Analyse kritischer Faktoren, potentieller Ausfallszenarien und von Anzeichen für Funktionsstörungen erstellt. Die Häufigkeit der Überwachung hängt von den Konsequenzen eines Ausfalles (Havarie) ab.

Zum Überwachungsplan gehören normalerweise auch

- eine Beschreibung des Ziels der Überwachung einzelner Parameter,
- die Bewertungskriterien für die Auswertung der Ergebnisse,
- die Festlegung einer Person/einer Stelle für die Überwachung, die Datenzusammenstellung und -auswertung sowie für die Berichterstattung und
- ein Zeitplan für die Überarbeitung des Planes.

4.4.14.2 Messungen und Geräte zur Überwachung von Dämmen mit Aufbereitungsrückständen, Häufigkeit der Überwachung

Zur Überwachung der Dammstabilität gehört ein Überwachungssystem zur Bewertung der tatsächlichen Stabilität der TMF, einschließlich der Dammstrukturen.

Tabelle 4.15 enthält Beispiele für üblicherweise vorgenommene Messungen, die verwendeten Messgeräte und die Häufigkeit der Prüfungen/Messungen von Hand sowie die angezeigten Abstände zwischen der entsprechenden Überwachungsmaßnahme selbst.

Messung	Messgeräte	Häufigkeit
Wasserstand im Absetzbecken	Pegelstandsanzeige, Doppler	wöchentlich, täglich oder online
Sickerwasserableitung durch ▪ den Damm selbst, ▪ die Gründung, ▪ die Aufleger	▪ Wehre oder Behälter/Tanks ▪ Porenwasserdruckmesser ▪ Grundwasserbrunnen	wöchentlich, täglich oder online
Sickerwasserproben	Probenahme und Trübenmessung	monatlich oder wöchentlich
Position des Grundwasserspiegel	Piezometer (gewöhnlich offenes Standrohr)	monatlich oder wöchentlich
Porendruck	Piezometer oder Bourdonrohr	monatlich oder wöchentlich
Bewegung der Dammkrone und der Aufbereitungsrückstände	Geodätische Bezugspunkte am Spülstrand (fertig gestellter Damm) und Dammkrone, Luftaufnahmen, GPS	jährlich oder halbjährlich
Seismik	Beschleunigungsschreiber für heftige Bewegungen	bei Ereignissen (erfolgt nicht am Standort)
Dynamischer Porendruck und Verflüssigung	Schwingsaiten-Piezometer	jährlich
Bodenmechanik	Bodensonde für die Dichte und Scherfestigkeit	jährlich (nur während der Entwurfsphase)
Absetzprozesse mit Aufbereitungsrückstände n	Scherfestigkeit, Kompressibilität, Verfestigung, Probenahme von Korngröße und -dichte, Breite des nicht unter Wasser stehenden Spülstrandes als Hinweis auf den Grundwasserspiegel durch Luft- oder Satellitenaufnahmen	jährlich (nur in der Entwurfsphase)

Tabelle 4.15: Typische Messungen, ihre Häufigkeit und die eingesetzten Messgeräte zur Überwachung von Dämmen für Aufbereitungsrückstände. Nachbearbeitet aus [7, ICOLD, 1996]

Tabelle 3.22, Tabelle 3.23 und Tabelle 3.24 enthalten Beispiele von Messungen bei der Verarbeitung und Aufbereitung einiger Grundmetalle (unedle Metalle).

Messung	Messgeräte	Häufigkeit
Strossen-/Böschungsgeometrie	GPS	halbjährlich
Drainage unter der Halde	Wehre/Spitzkerben	jährlich
Porendruck (wo potentielles Risiko besteht)	Piezometer/Standrohre	jährlich

Tabelle 4.16: Typische Messungen, ihre Häufigkeit und die eingesetzten Messgeräte zur Überwachung von Halden

4.4.14.3 Inspektionen, Kontrollen und Überprüfungen

Der Gesamtüberwachungsplan enthält üblicherweise auch Pläne für Inspektionen, Kontrollen und Überprüfungen.

Tabelle 4.17 enthält ein mögliches Überwachungsprogramm für die Betriebs- und Nachsorgephase.

Art der Einschätzung	Häufigkeit		Personal
	Betriebsphase	Nachsorgephase	
Inaugenscheinnahme	täglich	halbjährlich	Dammbetreiber, nach der Stilllegung möglicherweise das Nachfolgepersonal
Jährliche Überprüfung	jährlich	jährlich	Ingenieur
Unabhängige Prüfung (Audit)	aller 2 Jahre	aller 5 - 10 Jahre	Unabhängiger Experte
Sicherheitsbewertung des vorhandenen Dammes (SEED)	15 - 20 Jahre	15 - 20 Jahre	Team unabhängiger Experten

Tabelle 4.17: Regime für die Bewertung von Dämmen für Aufbereitungsrückstände während der Betriebsphase und in der Nachsorgephase

Art der Einschätzung	Häufigkeit		Personal
	Betriebsphase	Nachsorgephase	
Inaugenscheinnahme	täglich	halbjährlich	Haldenbetreiber, nach der Stilllegung möglicherweise das Nachfolgepersonal
Geotechnische Überprüfung	jährlich	aller 2 Jahre	Ingenieur
Unabhängige geotechnische Prüfung (Audit)	aller 2 Jahre	aller 5 - 10 Jahre	Unabhängiger Experte

Tabelle 4.18: Regime für die Bewertung von Halden mit Aufbereitungsrückständen während der Betriebsphase und in der Nachsorgephase

Die für die Nachsorgephase angegebenen Abstände sind für den Anfangszeitraum nach der Stilllegung relevant. Auf Grundlage der Ergebnisse können die Abstände im Laufe der Zeit in einem solchen Maße ausgedehnt werden, dass Inspektionen, Kontrollen und Überprüfungen überhaupt nicht mehr erforderlich sind, wenn die Wiederherstellung/Wiederurbarmachung vollständig abgeschlossen ist. Des Weiteren gestatten die Überwachungsprogramme in der Nachsorgephase eine Überprüfung, ob alle Ziele der Stilllegung und die langfristige Funktion erreicht worden sind. Wenn die Ziele der Stilllegung nicht erfüllt worden sind, müssen im Zeitraum der Umsetzung Abhilfemaßnahmen ergriffen werden.

Zur Routine bei der Einschätzung der Stabilität von Dämmen und Halden kann folgendes gehören:

Inaugenscheinnahme – erfolgt durch einen erfahrenen Betreiber/eine erfahrene Aufsichtsperson nach einer vorher festgelegten ‘Checkliste’, die sich auf zu inspizierende Merkmale konzentriert, die wahrscheinlich Probleme hervorbringen, wenn sie nicht korrigiert werden (zum Beispiel blockierte Überläufe, schadhafte Pumpen, übermäßige Erosion, zu viel Feuchtigkeit am Fuße der Böschung usw.). Diese Checklisten beruhen auf Merkmalen, die dem erfahrenden Beobachter sofort ins Auge springen und innerhalb eines angemessenen Zeitraumes leicht korrigiert werden können. Durch diese Art eines einfachen Inspektionsregimes kann eine Halde oder ein Absetzbecken in guter Tagesform gehalten werden, d. h. die täglichen Inspektionen sollten nicht auf Dingen beruhen, die eine detaillierte wissenschaftliche Herangehensweise erforderlich machen. Es ist wichtig abzusichern, dass über diese täglichen Inspektionen zu Referenzzwecken ein Nachweis geführt wird und dass eine Vorgehensweise zur Information einer kompetenteren Person existiert, falls die Inspektion Unnormales oder für den Betreiber Bedenkliches zutage fördert, was er nicht selbst beheben kann.

Anhang 6 enthält ein Beispiel für Dinge, die während der täglichen Inspektion kontrolliert werden sollten.

Jährliche Überprüfungen – umfassen eine vollständige topographische Vermessung des Baukörpers mindestens einmal im Jahr oder öfter, wenn der Baukörper groß ist und sich in ständiger Entwicklung befindet. Nach Abschluss dieser topografischen Vermessungen müssen genaue Pläne und Querschnitte des Baukörpers erstellt werden, die alle in eine wieder abrufbare Datenbank eingespeichert werden.

Mindestens ein Mal im Jahr muss eine genaue Vermessung von auf dem Bauwerk (besonders in Absetzbecken für Aufbereitungsrückstände) errichteten "Beobachtungspunkten" oder "-plateaus" vorgenommen werden, bei der Hinweise auf horizontale oder vertikale Bewegungen kontrolliert werden. Wichtig ist dabei, dass diese Beobachtungspunkte ein Bezugssystem haben, das sich auf festem Boden jenseits der Standfläche des Bauwerks befindet. Bei Bauwerken, wo das Risiko eines hohen Grundwasserspiegels oder potenzieller Sickerwasserzonen besteht (was in Absetzbecken für Aufbereitungsrückstände wahrscheinlicher ist), muss sowohl innerhalb der o.g. Oberflächenstruktur der Anschüttung der Aufbereitungsrückstände als auch im Unterboden unterhalb des Bodenniveaus ein System vertikaler Piezometer installiert werden. Diese Piezometer müssen mindestens einmal in jeder ‘Saison’, d. h. im Winter, Frühjahr, Sommer und Herbst, abgelesen werden, um saisonale Unterschiede aller Art, besonders im Grundwasserdurchfluss, aufzuzeichnen. Wenn in einem Absetzbecken absichtlich ein hoher Wasserstand gehalten wird (zum Beispiel zur Staubbekämpfung), mag es notwendig werden, diese Piezometer öfter abzulesen. Diese Messwerte werden sinnvollerweise im Computer gespeichert und den Querschnittszeichnungen zugeordnet, damit die Sickerwasserbewegungen des Bauwerks schnell und einfach identifiziert werden können. Wo Sickerwasser aus der Anschüttungsstruktur austritt bzw. durch ein Entwässerungssystem fließt (z. B. Rohre/Steinfilter usw.), sind diese Systeme gewöhnlich mit Messwehren oder -rohren ausgestattet, so dass jede Zu- oder Abnahme festgestellt und für Referenzzwecke aufgezeichnet werden kann. Diese Systeme müssen mindestens einmal pro Saison kontrolliert werden, wobei dann ein kompetenterer Mitarbeiter über plötzliche oder unnormale Veränderungen in Kenntnis gesetzt werden sollte.

Anhang 6 enthält ein Beispiel für Dinge, die während der jährlichen Überprüfung kontrolliert werden sollten.

Unabhängige Prüfungen (Audits) – müssen an bereits in Betrieb befindlichen Bauwerken mindestens einmal aller zwei Jahre durchgeführt werden. Diese Prüfungen (Audits) werden durch eine Sachverständigengruppe durchgeführt, zu der oft ein unabhängiger Gutachter gehört. Zu diesen Einschätzungen gehört die Überprüfung aller verfügbaren Daten, der täglichen Inspektionsberichte, der Überwachungsberichte, der Piezometermessungen usw., um sich eine Meinung über die Stabilität des Bauwerks sowohl zum Zeitpunkt der Einschätzung als auch im Zeitraum bis zur nächsten Begutachtung bilden zu können. Wenn diese Begutachtung irgendwelche Hinweise erbringt, die Anlass zu grundsätzlichen Bedenken geben, ist der Gutachter verpflichtet, dem Betreiber diese Bedenken einschließlich seiner Empfehlungen zur Beheben des Problems zu übermitteln.
[131, IMA, 2003]

SEED-Audits – (Safety Evaluation of Existing Damms/Sicherheitseinschätzung vorhandener Dämme) werden aller 15 - 20 Jahre durch ein Team unabhängiger Gutachter durchgeführt und umfassen die Durchsicht aller den Damm betreffenden Dokumentationen, die Infragestellung aller grundlegenden Annahmen, die zur Auslegung und zum Bau des Dammes führten, sowie eine Einschätzung der Übereinstimmung zwischen der Auslegung und dem tatsächlichen Zustand des Dammes. Die Audits sollten auch eine Überprüfung aller verfügbaren Daten, der täglichen Inspektionsberichte, der Überwachungsberichte, der Piezometer-Messungen usw. einschließen. Bei dieser Art Überprüfung müssen alle neuen Entwicklungen auf dem Gebiet der Dammsicherheit und der Hydrologie berücksichtigt werden. Vorschläge zur Modernisierung des vorhandenen Dammes werden mit neuen Erkenntnissen und Entwicklungen verglichen. Das Ergebnis dieser Überprüfung versetzt die Gutachter in die Lage, sich eine Meinung über die Stabilität des Bauwerkes zu bilden.
Weitere Angaben sind auch in Abschnitt **4.2.3.2** enthalten.

4.4.14.4 Stabilität der stützenden Bodenschichten

Selbst die stabilste Anlage für Aufbereitungsrückstände oder taubes Gestein wird ihren Dienst versagen, wenn die Gründung, auf der sie errichtet ist, nicht stabil (genug) ist. Es ist daher wichtig, in der Planungsphase auch die Eignung der stützenden Bodenschichten zu untersuchen (siehe Abschnitt **4.2.1.4**).
Während des Betriebes von Halden für Kali-Aufbereitungsrückstände wird die **Stabilität der stützenden Bodenschichten** zum Beispiel regelmäßig durch seismische Überwachungsmaßnahmen kontrolliert, wobei seismische, seismisch-akustische und geomechanische Ereignisse bzw. das Absinken der Oberfläche im Ergebnis bergbaulicher Aktivitäten erforscht und bestimmt werden. Die Vermessung von Stützpfeilern und die Bestimmung der mineralischen Verbindungen werden zur Berechnung und Beobachtung der Stabilität bergbaulich ausgebeuteter Räume verwendet (siehe Abschnitt **3.3.3.2**).

Diese Art Überwachung ist für den Betrieb von Anlagen geeignet, wo es in der Vergangenheit seismische Ereignisse gegeben hat bzw. die sich in der Nähe von Untertagebergwerken befinden.

4.4.15 Bewirtschaftung von Cyanid

Zusätzlich zur Behandlung des Cyanids (siehe Abschnitt **4.3.11.8**), verlangt die CN-Laugung und die Bewirtschaftung von CN im Allgemeinen eine große Anzahl von Sicherheitsmaßnahmen, um Unfälle und negative Auswirkungen auf die Umwelt zu verhindern. Zur Auslegung der Anlagen gehören auch verschiedene technische Lösungen, die auf die Verhinderung von Unfällen und umweltschädlichen Auswirkungen gerichtet sind, wie

- der Einbau eines integrierten Zersetzungskreislaufes für Cyanid in die Laugungsanlage, wobei dieser Kreislauf eine zweifache Bemessungskapazität der tatsächlichen Anforderung hat;
- die Nutzung eines ganzen Absetzbeckensystems für die Aufbereitungsrückstände als zweite Behandlungsstufe, gewissermaßen als Backup für den Cyanid-Zersetzungskreislauf;
- die Kombination der Aufbereitungsrückstände aus der Flotationsanlage (für das Extrahieren von Grundmetallen) mit dem Schmutzwasser aus dem Goldlaugungskreislauf vor der Zersetzung des Cyanids, um einen Anstieg des pH-Wertes zu verhindern, der die Auflösung bereits ausgefällter Cyanidkomplexe verursachen könnte;
- der Einbau eines Backup-Systems für den Zuschlag von Kalk;
- die Umleitung des Laugungskreislaufes in ein Sammelbecken mit einem Volumen, das der Aufnahmekapazität eines Laugungsbehälters entspricht;
- der Einbau der Laugungsbehälter in eine Betonwanne mit einem sie umgebenden Bankett (Berme), das/die auch als Stoßbarriere dient; das Fassungsvermögen der Wanne ist größer als das Volumen eines Laugungstankes und der Boden ist beheizt, um die Schnee- und Eisbildung im Winter zu verhindern;
- das Aufstellen offener Laugungsbehälter im Freien;
- der Einbau von Notstromgeneratoren und
- das Zurückpumpen aller Überlaufreste und Verschüttungen in den Kreislauf.

Weitere Angaben zur Cyanid-Bewirtschaftung enthält die Website des *International Cyanide Management Code* für die Herstellung, den Transport und die Verwendung von Cyanid bei der Goldproduktion: www.Cyanidcode.org

4.4.16 Entwässerung der Aufbereitungsrückstände

Aufbereitungsrückstände in Schlammform enthalten typischerweise 20 – 40 Gew.-% Feststoffe, aber es hat auch schon Aufbereitungsrückstände mit nur 5 – 50 % Feststoffe gegeben. Alle Aufbereitungsrückstände in Schlammform werden normalerweise durch Dammbauten eingedämmt (siehe Abschnitt **2.4.2**), was zugleich auch der kostengünstigste Weg ihrer Bewirtschaftung ist.

Weitere Vorteile dieser Art des Umgangs mit den Aufbereitungsrückständen sind:
- Auf Grund der Sättigung mit Wasser kommt es nicht zur Staubbildung durch Aufbereitungsrückstände (was sich ändern kann, sobald als Teil des Spülstrands der Sonne und dem Wind ausgesetzt sind).
- ARD wird verhindert.

Der größte Nachteil beim Umgang mit schlammartigen Aufbereitungsrückständen ist ihre Mobilität. Wenn das Rückhaltebauwerk (d. h. der Damm) zusammenbricht, verflüssigen sie und können auf Grund ihrer physikalischen und chemischen Eigenschaften beträchtlichen Schaden anrichten. Zur Vermeidung dieses Problems sind einige Alternativen entwickelt worden, d. h. 'trockene' Aufbereitungsrückstände und eingedickte Aufbereitungsrückstände (siehe Abschnitte **4.4.16.1** und **4.4.16.2**)

Wie sich aus **Table 3.59** und **Table 3.60** ergibt, schwanken die Bewirtschaftungskosten für schlammartige Aufbereitungsrückstände zwischen € 0,30 und € 1,60 je Tonne trockener Aufbereitungsrückstände.

4.4.16.1 'Trockene Aufbereitungsrückstände'

In der Grube Greens Creek in den USA werden die Aufbereitungsrückstände eingedickt und danach gefiltert. Der so entstandene Filterkuchen enthält etwa 12 % Feuchtigkeit. Etwa die Hälfte der gefilterten Aufbereitungsrückstände wird nach Zumischung von 3-5 % Zement wieder als Verfüllmaterial unter Tage eingestapelt. Die verbleibenden Aufbereitungsrückstände werden per Lkw in eine oberflächig angelegte Stauhaltung verbracht, wo sie entsprechend der Spezifikationen verdichtet werden, die Wasser- und Sauerstoffinfiltration auf ein Minimum reduzieren.

Die Aufbereitungsrückstände sind feinkörnig (80 % nur 20 – 30 µm) und benötigen einen teuren Entwässerungsprozess, der als Druckfiltration bekannt ist. Das Verfahren mit 'trockenen Aufbereitungsrückständen' erwies sich auf Grund eines nicht zur Verfügung stehenden geeigneten Platzes für ein konventionelles Absetzbecken für Aufbereitungsrückstände und auf Grund der Vorgaben für die Verfüllung der Grube als das einzig praktische (und wirtschaftliche) für Greens Creek.

Die Gesamtbetriebskosten für die Entsorgung der 'trockenen' Aufbereitungsrückstände in Greens Creek belaufen sich wahrscheinlich bei 1 000 t Aufbereitungsrückständen am Tag auf etwa US-$ 4-6/t (2002). In diese Kosten eingeschlossen sind Verdickungsreagenzien, Druckluft (hauptsächlich strombetrieben) für die Druckfilter, Lohnkosten für Betrieb und Wartung sowie die Transportkosten für die Aufbereitungsrückstände in die 15 km entfernte oberflächig angelegte Stauhaltung. Das ist viel teurer als die klassische Entsorgung 'nassen' Schlamms, wo die Aufbereitungsrückstände verrohrt (oft nur durch die Schwerkraftwirkung) zu einem Absetzbecken transportiert werden, wo sie sich absetzen können. Das klare Wasser wird dann zurück zur Aufbereitungsanlage gepumpt.

In der Zinkanlage Asturiana belaufen sich die Kosten für die Entwässerung in Bandfiltern (Umlauffilter) gegenwärtig auf € 0,95 je Tonne Erz. Die Investitionskosten für die Filtrationsanlage beliefen sich auf € 3,5 Mio.

Am Standort Neves Corvo werden Kosten von € 2,50 je Tonne Erz für die Entwässerung mit Druckfiltern erwartet. Diese Aufbereitungsrückstände mit einen Netto-Säurebildungspotential werden (innerhalb von 8 Monaten) schnell abgedeckt, um eine Oxidation zu vermeiden.

Trockene Aufbereitungsrückstände werden in großem Umfang auch noch in der Gold- und Silberlaugungsanlage La Coipa in Chile entsorgt, wo täglich 15 000 t von Aufbereitungsrückständen mit Vakuum-Bandfiltern entwässert und dann per Band zu einer Stapelanlage im Bereich der Stauhaltung transportiert werden. Die Kosten sind viel niedriger als in Greens Creek, weil

- die Aufbereitungsrückstände gröber sind und im Vakuum anstatt mit Druckfiltern gefiltert werden können,
- die Anlage wirtschaftlicher ist (vgl. 15 000 t/d mit nur 1 000 t/d) und
- die Standortbedingungen besser sind (flaches, trockenes Wüstenklima anstatt feuchtes, Bergklima).

[120, Sawyer, 2002]

Bei Aufbereitungsrückständen mit einem ARD-(Sauerwasser-)Potential kann das Verfahren mit den trockenen Aufbereitungsrückständen zu unumkehrbarer Oxidation führen, was dem Prinzip der ARD-Verhinderung entgegen läuft. Es ist praktisch nicht durchführbar, diese Aufbereitungsrückstände zur Verhinderung ihrer Oxidation abzudecken.

Bei hydrometallurgischen Vorgängen (d. h. Laugung) ist dieses Verfahren Teil des Prozesses. Zusammen mit den natürlich verfügbaren Tonschichten für die Ablagerung der Aufbereitungsrückstände, kann dieses Verfahren durchaus anwendbar sein (siehe Abschnitt **6.4**).

Es kann sich in solchen Fällen als günstig erweisen, wo der verfügbare Platz beschränkt ist. Dennoch müssen mögliche medienübergreifende Auswirkungen, wie der Energieverbrauch für die Filtration, die Emissionen der Lkw und mögliche Staubemissionen, in Betracht gezogen werden.

Bei der Gewinnung von geringwertigen Mineralen verbietet sich dieses Verfahren in dem Maße von selbst, sobald die Kosten für die Bewirtschaftung der Aufbereitungsrückstände die des Erzwertes übersteigen.

Es gibt verschiedene gesetzliche Bestimmungen, die verlangen, dass Aufbereitungsrückstände mit ARD-potenzialhaltigem Wasser jederzeit gesättigt gehalten werden müssen, so dass dieses Verfahren in vielen Fällen keine verfügbare Option darstellt.

Die Kosten bei diesem Verfahren steigen sprunghaft, je kleiner die Korngröße wird.

In allen europäischen Kali-Unternehmen werden die Aufbereitungsrückstände trocken bewirtschaftet.

4.4.16.2 Eingedickte Aufbereitungsrückstände

Eine Option für die sicherere Bewirtschaftung von Aufbereitungsrückständen ist das Absetzen pastöser (oder eingedickter) Aufbereitungsrückstände statt der Entsorgung von Schlamm [116, Nilsson, 2001].

Die Grundlagen dieses Verfahrens sind in Abschnitt **2.4.3** vorgestellt worden. Das Wichtige daran ist, dass die Bewirtschaftung eingedickter Aufbereitungsrückstände den Einsatz mechanischer Geräte zur Entwässerung der Aufbereitungsrückstände verlangt, so dass ein Feststoffanteil von etwa 50 - 70 % verbleibt. Die Aufbereitungsrückstände werden dann im Ablagerungsbereich schichtweise verteilt, so dass sich ein weiterer Entwässerungseffekt durch eine Kombination aus Drainage und Verdunstung einstellt [11, EPA, 1995].

Der Hauptunterschied zu ‘trockenen’ Aufbereitungsrückständen, wie sie im vorstehenden Abschnitt beschrieben sind, besteht im Feststoffanteil nach der Entwässerung. Beim Verfahren mit ‘trockenen’ Aufbereitungsrückständen werden letztere so weit gefiltert, bis ein ‘Kuchen’ mit etwa 12 % Feuchtigkeit zurückbleibt. Eingedickte Aufbereitungsrückstände hingegen werden nur soweit entwässert, bis eine 'Paste' mit zwischen 30 und 50 % Feuchtigkeit zurückbleibt (d. h. ein Feststoffanteil von 50 – 70 %).

Der größte Vorteil dieses Verfahrens besteht darin, dass die Aufbereitungsrückstände weniger mobil sind, was insofern von Nutzem ist, falls es zu einem Dammbruch kommt.

Andere Vor- und Nachteile sind:

Vorteile:
- verringerte Wartungs- und Stilllegungskosten,
- größere Absetzkapazität bei gleicher Höhe des Dammumfanges (trifft nicht auf die Bewirtschaftung von Rotschlamm zu),
- niedrige Anfälligkeit für Verflüssigung, damit höhere Erdbebenwiderstandsfähigkeit,
- keine Notwendigkeit für ein Klarwasserabzugssystem,
- verringerte Sickerwassereindringung in das umgebende Terrain,
- Abscheidung des meisten Wassers in der Mineralaufbereitungsanlage, daher geringere Notwendigkeit des Recycelns des Wassers aus dem Absetzbecken. [77, Robinsky, 2000]

Nachteile:
- Der Transport der eingedickten Aufbereitungsrückstände kann sich als schwierig und teuer erweisen; das Eindicken sollte daher in einer Anlage am Absetzbecken selbst vorgenommen werden. **[21, Ritcey, 1989]**
- Die ausgetrocknete Oberfläche kann zur Staubbildung führen, weshalb ggf. ein Bewässerungssystem erforderlich werden kann. **[21, Ritcey, 1989]**
- Es muss ein besonderes System zum Sammeln des ablaufenden Oberflächenwassers und des Drainagewassers gebaut werden, wobei das gesammelte Wasser wiederum ordnungsgemäß bewirtschaftet werden muss.

Außer der Tatsache, dass es sich hierbei um ein Verfahren zum Austragen der Aufbereitungsrückstände handelt, ist es auch zur Abdeckung vorhandener konventioneller Absetzbecken mit Aufbereitungsrückständen empfohlen worden [21, Ritcey, 1989]. Das Verfahren zum Absetzen eingedickter Aufbereitungsrückstände kann unter folgenden Umständen besonders vorteilhaft sein:
- flache Topographie, die das Anlegen eines breiten kegelförmigen Absetzbeckens mit flachen Böschungen erlaubt,
- wo der Bau eines konventionellen Dammes auf Grund der örtlichen Bedingungen teuer wird
- und wo die Aufbereitungsrückstände fein sind, so dass keine groben Fraktionen zur Verfügung stehen.

[21, Ritcey, 1989]

Unter folgenden Bedingungen lässt sich dieses Verfahren nicht anwenden:
- weniger als 15 % der Aufbereitungsrückstände haben eine Korngröße von <20 µm (auf Trockenbasis), [138, Verburg,]
- wenn die Aufbereitungsrückstände ein säurebildendes Potential haben.

Eine Publikation behauptet, dass eingedickte Aufbereitungsrückstände auch gut sind, wenn sie ein säurebildendes Potential haben, was durch die Tatsache gerechtfertigt wird, dass die Feinstoffe in der homogenen Mischung eingedickter Aufbereitungsrückstände eine hohe kapillare Sogwirkung entwickeln, die wiederum die Aufbereitungsrückstände in einem gesättigten Zustand halten. Dadurch verhindern sie die Säurebildung [77, Robinsky, 2000]. Das wird jedoch öfter bestritten und ist auch noch nicht branchenweit im großen Maßstab nachgewiesen worden.

Abbildung 4.23 zeigt einen Vergleich zwischen einem Absetzbecken mit eingedickten Aufbereitungsrückständen und einem mit herkömmlichen Aufbereitungsrückständen in gleicher geologischer Umgebung.

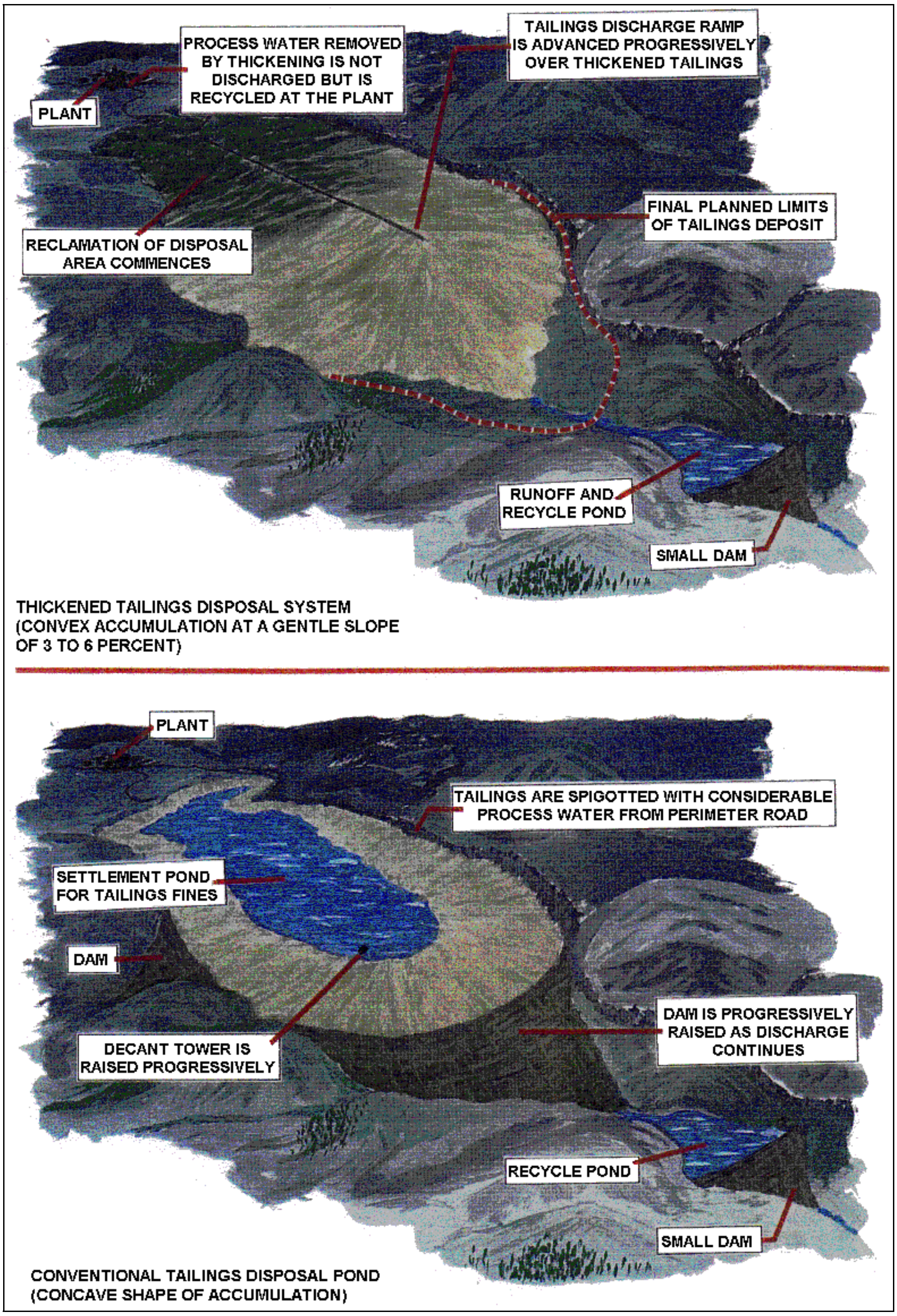

Abb. 4.23: Vergleich von Absetzbecken mit eingedickten Aufbereitungsrückständen und mit herkömmlichen Aufbereitungsrückständen in der gleichen geologischen Umgebung [77, Robinsky, 2000]

Process water removed by thickening is not discharged but recycled at the plant – Das beim Eindicken entstandene Prozesswasser wird nicht ausgetragen, sondern in der Anlage recycelt.

Tailings discharge ramp is advanced progressively over thickened tailings – Sich allmählich beim Eindicken der Aufbereitungsrückstände vorwärtsbewegende Austrageböschung für Aufbereitungsrückstände

Plant – Anlage

Reclamation of disposal area commences – Wiedernutzbarmachung des Absetzbereichs beginnt

Final planned limits of tailings deposit – Endgültige geplante Begrenzung für das Absetzen der Aufbereitungsrückstände

Runoff and recycle pond – Ablauf-/Ableitungs- und Wiedergewinnungsbecken

Small dam – Kleiner Damm

Thickened tailings disposal system (convex accumulation at a gentle slope of 3 to 6 per cent) – Entsorgungssystem für eingedickte Aufbereitungsrückstände (konvexer Aufbau mit einer leichten Anböschung von 3 – 6 Prozent)
Tailings are spigotted with considerable process water from perimeter road – Einleitung der Aufbereitungsrückstände über ein Spukpfeifensystem mit beträchtlichen Mengen an Prozesswasser von der Außenstraße
Settlement pond for tailings fines – Absetzbecken für feinkörnige Aufbereitungsrückstände
Dam – Damm
Decant tower is raised progressively – Das Einlaufbauwerk (Mönch) wird allmählich erhöht
Dam is progressivel raised as discharge continues – Mit dem weiteren Absetzen wird der Damm allmählich erhöht
Recycle pond – Wiedergewinnungsbecken
Conventional tailings disposal pond (concave shape of accumulation) – Herkömmliches Absetzbecken für die Entsorgung von Aufbereitungsrückständen (konkaver Aufbau)

Eingedickte Aufbereitungsrückstände werden mit einem Feststoffanteil von zwischen 50 und 70 % abgesetzt. Das bedeutet, dass sie mehr Wasser enthalten, als das Porenvolumen der Aufbereitungsrückstände speichern kann, sodass ein Teil des Wassers auf irgendeine Weise wieder aus der Anlage ausgetragen werden muss.

Die Betriebskosten für Anlagen mit eingedickten Aufbereitungsrückständen sind im Vergleich mit Anlagen, wo schlammartige Aufbereitungsrückstände bewirtschaftet werden, beim Einsatz einer tief wirkenden Eindickanlage etwa 25 % und beim Einsatz von Filtern 40 % höher.

Bei der **Veredelung von Alumina** können die wesentlichen Unterschiede zwischen der Verwendung von eingedickten und schlammartigen Aufbereitungsrückständen wie folgt zusammengefasst werden:

Die **Bewirtschaftung schlammartiger Aufbereitungsrückstände** bringt die Behandlung von viel mehr Wasser mit sich, das mit dem Schlamm kommt. Dieses Verfahren hat den Vorteil, dass sich der Schlamm mit Standardzentrifugalpumpen bei relativ niedrigem Leitungsdruck leicht pumpen lässt. Das zur Aufschlämmung verfügbare Wasser kann Seewasser sein, wenn es in der Umgebung der Veredelungsanlagen verfügbar ist, wobei die verbleibenden Ätzmittel neutralisiert werden müssen. Die Aufbereitungsrückstände können über verhältnismäßig große Entfernungen (mehrere Kilometer) zwischen der Veredelungsanlage und dem Absetzbecken gepumpt werden, ohne dass die Gefahr eines Druckabfalls in den Leitungen besteht.

Die **Bewirtschaftung eingedickter Aufbereitungsrückstände** muss mit dem gewissenhaften Entzug der ätzenden Stammlauge einhergehen, da bei der Bewirtschaftung des Absetzbeckens keine weitere Neutralisierung erfolgt. Die Dichte und Viskosität der eingedickten Aufbereitungsrückstände (manchmal auch ‘Paste’ genannt) ist so hoch, dass die Entwässerung vorzugsweise in der TMF erfolgt, es sei denn, das Absetzen erfolgt nahe der Veredelungsanlage. Wenn die beiden Anlagen jedoch etwas weiter voneinander entfernt sind, erfolgt das Pumpen bei geringer Dichte vor dem Entwässern am Absetzbecken, um den dicken Schlamm erst am Fuße des Absetzbeckens einzuleiten. In diesem Falle muss das überschüssige Wasser den ganzen Weg zurück in die Veredelungsanlage gepumpt werden. Deshalb werden bei diesem Verfahren zusätzliche Investitionen für eine Hochdruckpumpstation mit Membranpumpen oder für den Einbau und den Betrieb einer tief wirkenden Eindickungsanlage am Absetzbecken benötigt, wenn letzteres weit von der Veredelungsanlage entfernt ist.

Bei der Verdichtung des dekantierten und gealterten Schlammes gibt es keine wesentlichen Unterschiede zu den ‘gereiften’ Aufbereitungsrückständen. In beiden Fällen liegt ein Feststoffanteil von etwa 70 % vor.

4.4.16.3 Entwässerung feinkörniger Kohleaufbereitungsrückstände

In einigen Kohlebetrieben werden feinkörnige Aufbereitungsrückstände von <0,5 mm aus der Flotation erst auf einen Feststoffanteil von 25 – 50 % eingedickt. Vorausgesetzt, es ist ausreichend Platz in künstlich errichteten Absetzbecken für die endgültige Deponierung vorhanden, werden die aufgearbeiteten feinkörnigen Aufbereitungsrückstände über Rohrleitungen oder per Lkw - je nach Entfernung und Mengen - in diese Anlagen transportiert. Wenn entschieden wird, die feinen Aufbereitungsrückstände auf Halden abzusetzen, z. B. aus Gründen der verfügbaren Bodenkapazitäten, müssen diese Aufbereitungsrückstände weiter entwässert werden, um die nötige strukturelle Stabilität zu erreichen.

Für die weitere Verringerung des Wassergehalts in den eingedickten Aufbereitungsrückständen werden im Prinzip drei Verfahren angewandt:
- Kammerfilterpressen mit üblicherweise mehr als 1000 m² Filterfläche, wodurch der Feststoffanteil auf 75 - 80 % steigt (siehe Abschnitt **2.3.1.10**),
- Vollmantelzentrifugen, wodurch der Feststoffanteil auf 50 – 70 % steigt (siehe Abschnitt **2.3.1.10**) oder
- Absetzbecken, wodurch der Feststoffanteil auf 50 – 70 % steigt (zeitweise Lagerung in diesen Becken, siehe Abschnitt **3.4.3.2.2**).

4.5 Verringerung des Platzbedarfs für die Standfläche

Der Einsatz eingedickter oder trockener Aufbereitungsrückstände kann bei der Bewirtschaftung den Platzbedarf für die Standfläche verringern helfen (siehe oben). Ansonsten ist der wirksamste Weg zur Verringerung des Platzbedarfs für die Standfläche von Einrichtungen zur Bewirtschaftung von Aufbereitungsrückständen und taubem Gestein das Verfüllen aller dieser Materialien oder Teilen davon. In vielen Fällen sei jedoch darauf hingewiesen, dass auf Grund steigender Mengen abgebauten Materials eine Oberflächenbewirtschaftung immer noch erforderlich wird, selbst wenn ein groß Teil der Aufbereitungsrückstände und/oder des tauben Gesteins verfüllt wird.
Bei allen Arten der Verringerung des Platzbedarfs für die Standfläche müssen auch die ökologischen Auswirkungen eingeschätzt werden.

4.5.1 Verfüllen von Aufbereitungsrückständen

Abschnitt **2.4.5** enthält bereits eine grundlegende Beschreibung des Verfüllens.

Mögliche Gründe für die Nutzung des Verfüllens sind

- im Untertagebau:
  - die Schaffung einer Arbeitsplattform für den Abbau des darüber liegenden Erzes (d. h. Firstenstoßbau,
  - die Sicherung der Untergrundstabilität,
  - die Verringerung der Setzungserscheinungen im Untergrund und an der Oberfläche,
  - die Schaffung einer Abstützung für das Hangende zum Abbau weiterer Teile der Erzlagerstätte und zur weiteren Erhöhung der Sicherheit,
  - die Schaffung einer Alternative zur Oberflächenentsorgung,
  - die Verbesserung der Belüftung;

- im Tagebau:
  - die Außerbetriebnahme/die Landschaftsgestaltung,
  - Sicherheitsgründe,
  - die Minimierung des Platzbedarfs für die Standfläche (als Gegenstück zum Bau eines Absetzbeckens oder einer Halde),

- die Minimierung des Risikos durch Verfüllung der Grube statt des Baues eines neuen Absetzbeckens bzw. einer Halde.

Wichtig ist, alle verfügbaren Optionen sorgfältig zu untersuchen, da das Verfüllen nicht in jedem Falle auch die Lösung mit den geringsten Auswirkungen auf die Umwelt ist.

Die großen im Untertageabbau entstehenden Stollen unterhalb der Oberfläche sind zusammen mit dem Verfüllen der Aufbereitungsrückstände eine ideale Abbauform, da feste oder schlammartige Aufbereitungsrückstände leicht in die großen Öffnungen verkippt werden können. Die ansonsten viel kleineren beim Strebbau, beim Kammerbau und beim Firstenstoßbau entstehenden Hohlräume führen zu erhöhten Verfüllkosten. Das Verfüllen kann auch dann noch Anwendung finden, wenn das Erz hochwertig ist und das Verfüllen eine höhere Gewinnungsrate erlaubt, denn die Sicherheitssäulen können nach Verfüllen der vorher entstandenen Hohlräume ebenfalls abgebaut werden. Beim Bruchbau ist ein Verfüllen nicht möglich, da die Hohlräume sofort mit ab- und herunterfallendem Material verfüllt werden.

Ein weiteres Anwendungsgebiet ist das Verfüllen in der Nähe gelegener bereits erschöpfter Tagebaue oder anderer 'Öffnungen', während das Verfüllen von schlammartigen Aufbereitungsrückständen in noch in Betrieb befindliche Gruben normalerweise nicht möglich ist.

Unter wirtschaftlichen Gesichtspunkten ist hydraulisches Verfüllen die interessanteste Option. Wenn aber das Abbauverfahren das Verfüllen voraussetzt, um eine schnellere Stabilisierung zu erreichen, muss hier möglicherweise Zement zugesetzt werden. In den meisten Fällen machen die Zementkosten das Verfüllen dann unwirtschaftlich. Daher werden in verschiedenen Abbaubetrieben alternative Bindemittel eingesetzt. Je nach Lage vor Ort stehen diese Materialien zu geringen Kosten oder gar kostenlos zur Verfügung. An einem Standort betrugen die Kosten für die der Grube zugelieferte Tonne Flugasche € 17–18 (2003).

Der Transport von Aufbereitungsrückständen in bergbaulich erschöpfte Gruben ist normalerweise nur dann wirtschaftlich, wenn sich die Gruben innerhalb weniger Kilometer im Umkreis befinden und die Aufbereitungsrückstände durch Rohrleitungen transportiert werden können.

In europäischen Untertagebergwerken für Grundmetalle werden die Aufbereitungsrückstände (16 – 52 % aller Aufbereitungsrückstände) gewöhnlich wieder verfüllt. In Pyhäsalmi werden 16 % der Aufbereitungsrückstände zum Verfüllen der Grube eingesetzt, die verbleibenden 84 % (180 000 t/Jahr) werden in einem Absetzbecken für Aufbereitungsrückstände deponiert. Dieser relativ geringe Anteil an verfüllten Aufbereitungsrückständen kann mit der Tatsache erklärt werden, dass nur grobe Aufbereitungsrückstände für das Verfüllen geeignet sind.

4.5.1.1 Verfüllen als Teil des Abbauverfahrens

Das in Garpenberg und Garpenberg Norra angewandte Abbauverfahren ist der Firstenstoßbau), wobei die groben Fraktionen der Aufbereitungsrückstände (manchmal auch als hydraulischer Sandversatz bezeichnet) verfüllt und als Plattform benutzt werden, wenn darüber Erz abgebaut wird. Abbildung 4.24 veranschaulicht, wie das verfüllte Material beim Firstenstoßbau eingesetzt wird.

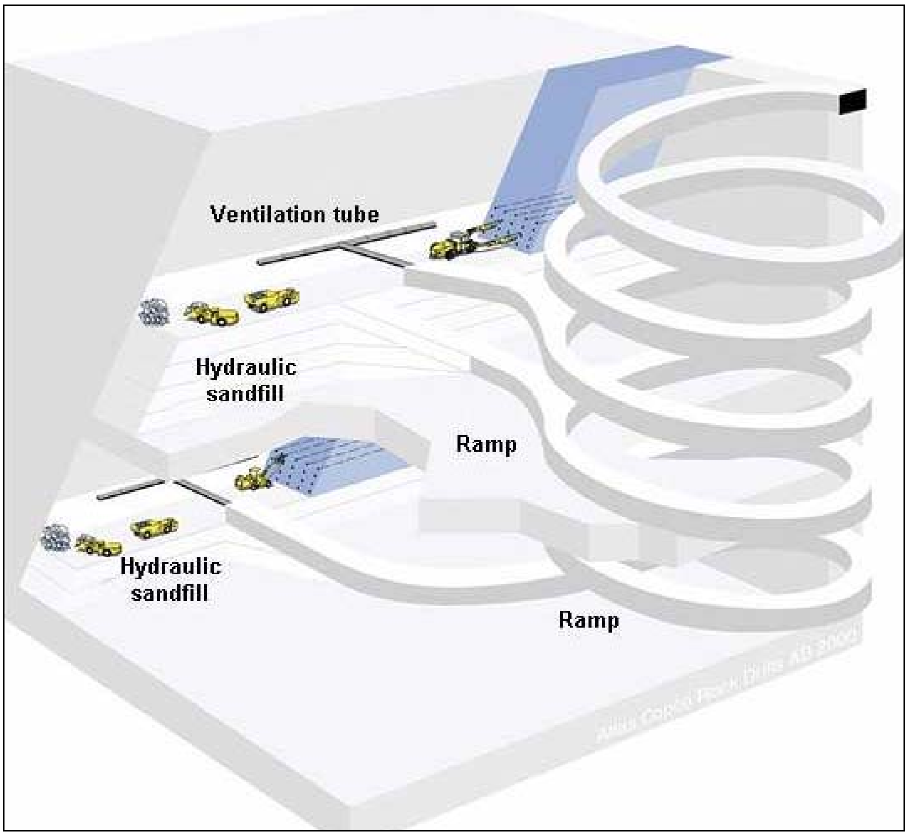

Abb. 4.24: Firstenstoßbau unter Verwendung des verfüllten Materials (hydraulischer Sandversatz) als Arbeitsplattform für den Abbau von Erz [93, Atlas Copco, 2002]
Ventilation tube – Ventilationsrohr
Hydraulic sandfill – hydraulischer Sandversatz
Ramp – Abschrägung/Anböschung

Alle durch den Abbau entstandenen Hohlräume (oder Öffnungen) in Garpenberg werden mit taubem Gestein aus Aufschlussarbeiten und mit Aufbereitungsrückständen verfüllt. Die Konzentrate enthalten etwa 10 % des aufbereiteten Erzes, so dass 90 % Aufbereitungsrückstände entstehen, von denen wiederum 50 % verfüllt werden. Beim Sprengen, Zerkleinern und Mahlen des Erzes erhöht sich sein Volumen um etwa 60 %, so dass das Volumen der Aufbereitungsrückstände in Garpenberg etwa 145 % des Volumens des abgebauten Erzes beträgt. Aus geometrischen Gründen gibt es keine Möglichkeit, mehr von den Aufbereitungsrückständen unter Tage zu verfüllen.

Das in Zinkgruvan angewandte Abbauverfahren erfordert ebenfalls Verfüllen.

4.5.1.2 Verfüllen in kleineren Tagebauen

In einer kleinen Barytgrube in Spanien werden die feinen Aufbereitungsrückstände in einem Betonbecken entwässert und der ‘Kuchen’ wird dann von Lkw in den Tagebau verkippt. Dieses Verfahren ist bei kleinen Gruben und unter klimatischen Bedingungen tauglich, wo die Aufbereitungsrückstände schnell entwässert werden können [110, IGME, 2002].

4.5.1.3 Verfüllen gefilterter Aufbereitungsrückstände

In einem Flussspatunternehmen in den südlichen Pyrenäen werden die Aufbereitungsrückstände, die 1 - 5 % CaF_2 enthalten, in die Grube verfüllt, nachdem sie mit Filterpressen entwässert wurden.

4.5.1.4 Teilverfüllung in Tagebauen

Ein Feldspatunternehmen in Segovia erzeugt 110 000 t Aufbereitungsrückstände im Jahr (Förderung: 600 000 t/Jahr), die aus sandhaltigen Fraktionen (80 000 t/Jahr) und den Aufbereitungsrückständen nach der Flotation bestehen. Die aus Grobsand bestehenden sandhaltigen Fraktionen, für die es keinen Markt gibt, werden in den Tagebau verkippt. Die Aufbereitungsrückstände aus der Flotation werden gefiltert. Der Filterkuchen (28 000 t/Jahr) wird ebenfalls verfüllt, während der verbleibende Schlamm in kleine Absetzbecken transportiert wird. Der Verfüllbereich im Tagebau wurde durch den Einbau eines Drainagesystems vorbereitet, um das Drainagewasser vor dem Auslassen in den Fluss durch Probenahme kontrollieren zu können.

4.5.1.5 Verfüllen in erschöpfte Gruben

Das Absetzbecken für Aufbereitungsrückstände im Kalkbergwerk Flandersbach ist in einem erschöpften Steinbruch errichtet worden, dessen Fläche heute 27 ha und zukünftig etwa 60 ha beträgt. Die Gesamtkapazität beläuft sich auf mehr als 30 Mio. m³. Das Absetzbecken befindet sich ganz in der Nähe der Rohstoffaufbereitungsanlage. Die Rohrleitungen für das Prozesswasser zum Absetzbecken und für das Klarwasser zurück zur Rohstoffaufbereitungsanlage haben eine Länge von etwa 1 km. Es fließt auch Grundwasser aus der Entwässerung des Steinbruchs in das Absetzbecken ein, das überschüssige Wasser wird in einen nahe gelegenen Fluss abgeleitet.
[107, EuLA, 2002]

4.5.1.6 Verfüllen in unterirdische Abbaukammern

Im Kalibergbau wird das Verfüllen in der steilen Lagerung angewandt, wo ein Weitungsbau mit Versatz stattfindet (und auch als *„funnel mining“* bezeichnet wird. Die ausgebeuteten, etwa 100 – 250 m hohen Abbaukammern werden mit salzhaltigen Aufbereitungsrückständen wieder verfüllt.

4.5.1.7 Verfüllen in unterirdische Kohlegruben

Verfüllen ist auch beim Untertagekohleabbau eine Möglichkeit und kann durch Rückführung der Aufbereitungsrückstände in die Abbaureviere unter Tage erfolgen, indem die vorher entstandenen und als ‘Versatzfeld’ oder ‘Alter Mann’ bezeichneten Hohlräume wieder verfüllt werden. Beim Kohlebergbau hängt das Verfüllen von einer Reihe geologischer und technischer Bedingungen ab, damit es sich auch rechnet. Da der in den Aufbereitungsrückständen aus der Steinkohle enthaltene Tonanteil in den Rohrleitungen zu Verstopfungen führen kann, wenn mit Wasser gepumpt wird, sind in den Kohlerevieren an der Ruhr, an der Saar und in Ibbenbüren in der Vergangenheit pneumatische Verfüllverfahren bevorzugt worden.

In den 70er Jahren des 20. Jh. sind Verfüllverfahren für flach abfallende Flöze entwickelt worden, die die Integration des Verfüllverfahrens in die Abbau-, Förder- und Strebausbautechnologie ermöglicht. Die Anwendung dieses pneumatischen Verfüllverfahrens stößt bei geringen Flözneigungen und Flözmächtigkeiten von unter 1,90 m an seine Grenzen. Verschiedene Versuche, Verfüllverfahren auch in kleineren Kohleflözen anzuwenden, sind gescheitert.

Die Investitionskosten für eine adäquate Verfüllinfrastruktur in den Revieren an Ruhr, Saar und in Ibbenbüren sind auf bis zu € 40 Mio. berechnet worden. Weitere Berechnungen haben ergeben, dass sich die Betriebskosten für das Verfüllen auf € 20 pro t geförderter Kohle belaufen, wovon je die Hälfte auf Lohn- und auf Materialkosten entfallen.

Die Anwendung des Verfüllverfahrens bringt auf Grund des hohen Investitionsbedarfs und der steigenden Betriebskosten beträchtliche finanzielle Belastungen mit sich (auch im Ergebnis von Leistungsverlusten bei der Kohleförderung). Das Verfüllen wir daher nur für solche Fälle in Betracht gezogen, wo es wirtschaftlich tolerierbar und in Hinblick auf die Boden-/Oberflächen-situation aus ökologischen Gründen notwendig ist. Daher wird in den Kohlerevieren an Ruhr, Saar und in Ibbenbüren momentan nicht verfüllt.

Einige potentielle Vorteile des pneumatischen Verfüllverfahrens sind:

- Verringerung von Oberflächenabsenkungen um etwa 50 % im Vergleich zum Bruchbau und daher Verringerung des internen und externen bergbaulichen Schadens an der Oberfläche,
- Verringerung des an der Oberfläche zu bewirtschaftenden Volumens an Aufbereitungsrückständen,
- Verlängerung der betrieblichen Lebensdauer vorhandener bzw. geplanter Deponien,
- Kosteneinsparung bei Bewirtschaftung von Aufbereitungsrückständen an der Oberfläche,
- Besseres Handling des durch die Gesteinsschichten entstehenden Drucks,
- Vorteile für das Belüftungssystem der Grube und Verbesserung der klimatischen Bedingungen unter Tage,
- unter bestimmten Bedingungen auch eine Verringerung der Wasseraufnahme unter Tage.

Diese Vorteile müssen aber auch im Lichte einer Reihe von Nachteilen betrachtet werden:

- Die Absenkungsbewegungen dauern normalerweise im Vergleich mit dem Bruchbauverfahren länger (was zu Verzögerungen bei der Widerherstellung der Oberfläche (Oberflächensanierung) bzw. wiederholten Schäden an bereits reparierten Objekten führen kann).
- Die Stillstandszeiten im Kohleabbau auf Grund von Verfüllmaßnahmen (z. B. Schäden an Rohrleitungen mit Verfüllmaterial) können zu einer ungünstigen Abbaudynamik führen, d. h. zu Laständerungen (Verlangsamung/Beschleunigung der Bewegungsprozesse in den Gesteinsschichten und an der Oberfläche).
- Felder mit Verfüllmaterialien neben Abbaufeldern erzeugen den Effekt von Abbaukanten, die äquivalent zu einer Dehnungsspitze an der Bodenoberfläche sind.
- Erhöhte Gefahr eines Gebirgsschlages im Vergleich zum Bruchbauverfahren.
- Der Ein- bzw. Aufbau eines Verfüllsystems in einer vorhandenen Grube ist sehr schwierig und teuer (was die Dimensionen der unterirdischen Transportwege und Eingänge angeht).
- Das notwendige zweite Bandfördersystem für den Transport der Aufbereitungsrückstände in die dem Kohletransport entgegenlaufende Richtung erfordert hohe Investitionen.
- Notwendig ist die genaue Synchronisierung der Anlieferung von Aufbereitungsrückständen mit der Kohleförderung.
- Das Verfüllverfahren schränkt den Betrieb an der Abbaufront (Ortsbrust) in Bezug auf die Vortriebsgeschwindigkeit und die Förderkapazität im Abbaufeld ein und erfordert manchmal sogar alternative Abbaufelder.
- Durch die Aufbereitungsrückstände in Schächten entsteht ein zusätzliches Gefahrenpotenzial beim Personaltransport.
- Das Verfüllen erhöht die Produktionskosten um mindestens € 20 pro Tonne Kohle [79, DSK, 2002].
- Die Gefahren für die Sicherheit und den Arbeitsschutz steigen auf Grund der Enge auf den Transportwegen und im Strebraum, wenn dem Verfüllmaterial gefährliche Abfälle (z. B. Flugasche) beigegeben werden.

4.5.1.8 Zugabe von Bindemitteln

Zur Verbesserung einer ungenügenden Bindigkeit beim hydraulischen Verfüllen werden manchmal Zement und/oder andere Bindemittel beigegeben. Bei diesen Bindemitteln handelt es sich um Flugasche oder Schlacke aus großen Verbrennungsanlagen, Abfallverbrennungsöfen oder Schmelzerein, die auch den Zement ganz oder teilweise ersetzen können. Die Eignung alternativer Bindemittel hängt vom Calciumoxidanteil ab, der die Endhärte und die

Reaktionszeit bestimmt. Oft kann eine größere Menge dieser Bindemittel erforderlich werden, um die gleiche Härte wie mit Zement zu erreichen. Zu Problemen beim Einsatz dieser Materialien kann es möglicherweise durch eine schwankende Qualität, hohe pH-Werte und das Vorhandensein von Schwermetallen oder löslicher Elemente kommen.

4.5.1.9 Entwässerung verfüllter Abbaukammern

Hydraulisch verfüllte Abbaukammern unter Tage müssen entwässert werden. Abbildung 4.25 zeigt ein Beispiel für ein Drainagesystem in einem Untertagebergwerk.

Abb. 4.25: Drainagesystem in Verfüllmaterial

4.5.1.10 Verfüllen pastöser Aufbereitungsrückstände

Ein spezieller Weg des Verfüllens ist der Einsatz pastöser Aufbereitungsrückstände (siehe Abschnitt **2.4.5**). Bei diesem Verfahren werden alle Aufbereitungsrückstände (nicht nur die groben Fraktionen) mit Zement vermischt, so dass ein pastöses Gemisch entsteht. Das führt zu einer Erhöhung der Dichte, so dass mehr Aufbereitungsrückstände in den Hohlräumen unter Tage abgelagert werden können [118, Zinkgruvan, 2003]. Es wird erwartet, dass auf diese Weise im Vergleich zu den 50 % beim hydraulischen Verfüllen bis zu 65 % der Aufbereitungsrückstände wieder verfüllt werden können. Verschiedene Gruben gehen zum Einsatz pastösen Verfüllmaterials über, da im Vergleich zum herkömmlichen hydraulischen Verfüllen nur ein geringerer Zementanteil (3 – 6 %) notwendig ist, um die gleiche Festigkeit zu erreichen, dem Druck des Hangenden standzuhalten.
[94, Life, 2002]

Vorteile, die außer einem erhöhten Anteil an verfülltem Material mit diesem Verfahren erzielt werden können:

- Bei Grubenwasser mit einem niedrigen pH-Wert steigt der pH-Wert im Grubenwasser auf Grund des Einsatzes von Zement.
- Weniger Wasser als beim traditionellen hydraulischen Verfüllen.
- Größere Stabilität, da die Hohlräume nicht nur mit Aufbereitungsrückständen, sondern auch mit Zement verfüllt werden.

Nachteile:

- die Kosten für den Bau der Verfüllanlage für pastöse Aufbereitungsrückstände,
- zusätzliche Kosten für den Zement,

[118, Zinkgruvan, 2003]
- es wird ein technisches Eindämmungsbauwerk erforderlich.

Das Verfüllen pastöser Aufbereitungsrückstände ist eine Möglichkeit in Fällen, wo
- der Bedarf für kompetentes Verfüllen besteht,
- die Aufbereitungsrückstände sehr fein sind, so dass nicht genug Material für das hydraulische Verfüllen vorhanden wäre (in diesem Falle würden die in ein Absetzbecken transportierten großen Mengen an Feinstoffen nur langsam entwässern) und
- es wünschenswert ist, Wasser aus der Grube fernzuhalten oder wo es zu teuer wäre, das von den Aufbereitungsrückständen abgepumpte Wasser abzupumpen (d. h. über eine große Entfernung).

Zum Verfüllen bestimmte Aufbereitungsrückstände müssen in Eindickanlagen von Filtern entwässert werden, was teurer wird und mehr Energie verbraucht als das hydraulische Verfüllen.

Die Kosten für die Lieferung von Zement an den Standort der Grube belaufen sich normalerweise auf US-$ 80/t (2002). In einer Filteranlage entwässerte Aufbereitungsrückstände haben einen durchschnittlichen einen Feuchtigkeitsgehalt von etwa 15 - 20 %. Die Beigabe von 3 - 5 Gew.-% Zement wäre in diesem Falle zusammen mit der natürlichen Feuchtigkeit für das Entstehen einer ziemlich steifen und stabilen Mischung ausreichend. Die Kosten beliefen sich daher auf etwa US-$ 2,40 – 4,00 pro Tonne an solcherart verfüllten Aufbereitungsrückständen (3/100*80 = US-$ 2,40/t).
[120, Sawyer, 2002]

4.5.2 Verfüllen tauben Gesteins

Im **Untertagebetrieb** wird taubes Gestein häufig als Verfüllmaterial verwendet, um die Stabilität zu erhöhen und den Abbau der Bodenschätze zu ermöglichen. Damit verringert sich die Notwendigkeit, taubes Gestein (Abraum) erst an die Oberfläche zu bringen.
Das Verfüllen von taubem Gestein in erschöpfte Tagebaue wird im Bergbau oft praktiziert. Es verringert die Notwendigkeit, Halden mit taubem Gestein erweitern zu müssen, minimiert damit den Platzbedarf für die Standfläche und kann oft kostengünstig sein. Dieses Verfahren wird angewandt, wenn
- das taube Gestein direkt, ohne Zwischenlagerung und erneuten Umschlag in den erschöpften Tagebau verfüllt werden kann, wenn sich der verfügbare Tagebau in angemessener Entfernung befindet (worauf manchmal mit dem Begriff '*transfer mining*' Bezug genommen wird) und
- es gegen Ende der Lebenszeit des Tagebaues in der Grube bestimmte Bereiche gibt, wo taubes Gestein dauerhaft ohne Sicherheitsrisiken und ohne Behinderung der Abbautätigkeit in anderen Bereichen abgelagert werden kann. In einem solchen Fall wird das taube Gestein dauerhaft in solchen Bereichen deponiert anstatt an die Oberfläche verbracht.

Des Weiteren besteht die Möglichkeit, bei der Stilllegung Teile des tauben Gesteins zurück in den Tagebau zu bringen, wenn das wirtschaftlich und ökologisch sinnvoll ist. Das erfolgt normalerweise aus Gründen der Stabilität oder wenn Teile des tauben Gesteins ein Nettopotenzial an Sauerwasser (ARD) haben. Diese Massen würden dann dauerhaft unter Wasser deponiert sein, wenn sich der Tagebau auf natürliche Weise mit Wasser füllt. Eine solche Form der Deponierung kommt aber nicht in Betracht, wenn
- die Kosten für die Verbringung des tauben Gesteins zurück in den Tagebau höher sind als die entsprechenden Stilllegungsverfahren,
- die Verbringung des tauben Gesteins in die Grube im Vergleich mit der Stilllegung der Halden an taubem Gestein an der Oberfläche keine wesentliche Steigerung der Stabilität sowie ökonomische oder ökologische Vorteile mit sich bringen würde, und

- der Abbau eingestellt wurde, ehe die Grube erschöpft war, da es dann in der Grube abgesetztes taubes Gesteins beträchtlich schwieriger und teurer machen würde, die verbleibende Vererzung in der Zukunft einschätzen zu können, wenn die erforderlich würde.

Von den Kohletagebauen in Großbritannien erzeugtes taubes Gestein wird während des Kohleabbaus auf provisorischen Halden bewirtschaftet. Nach dem Ende des Abbaus der Kohlelagerstätten wird das taube Gestein in die Hohlräume verfüllt und die Landschaft saniert.

Vorsicht ist bei der Deponierung von taubem Gestein mit oxidiertem Sulfid in einem Tagebau geraten, da es relative hohe Säuregrade und große Mengen aufgelösten Metalls freisetzen kann. In solchen Fällen ist es wichtig, einen hohen pH-Wert im Grubenwasser abzusichern, was u.a durch Kalkbeigaben erreicht werden kann.

4.5.3 Bewirtschaftung von Aufbereitungsrückständen unter Wasser

In der Calciumkarbonatanlage im norwegischen Hustadmarmor werden die Aufbereitungsrückstände in einen tiefen und geschützten Fjord verklappt, da kein geeignetes Land für die Bewirtschaftung der Aufbereitungsrückstände zur Verfügung steht.

Das zur Kontrolle der ökologischen Auswirkungen im Bereich der Ablagerung und um diesen Bereich herum eingesetzte Überwachungsprogramm erfasst die folgenden Parameter:

- Wasseranalysen:
  - Feststoffgehalt (Trübung),
  - Salzgehalt (Seewasser),
  - Sauerstoffgehalt,
  - Temperatur.

- Sedimentanalysen:
  - Anteile an Calciumkarbonat,
  - Anteile an Feinstoffen,
  - biologische Aktivität,
  - Anteil an Flotationsreagenzien.

- Flachwasser:
  - biologische Aktivität,
  - visuelle Dokumentation (Fotos).

Außerdem werden andere Messungen vorgenommen und Maßnahmen ergriffen, um geeignete Modelle zur Vorhersage künftiger Entwicklungen usw. erarbeiten zu können. Dazu gehören:
- Messungen der Meeresströmung,
- Tiefenmessungen und Volumenberechnungen,
- Videoaufnahmen des Meeresbodens,
- die Rekultivierung des Meeresbodens nach Ende der Ablagerungen.

Zum Betriebsablauf und zur Bewirtschaftung der Aufbereitungsrückstände gehören:
- das Waschen der Feinanteile aus dem Eintrag in die Flotation,
- eine computergestützte Prozesskontrolle,
- on-line-Analysen der chemischen Zusammensetzung des Eintrages in die Flotation und der Aufbereitungsrückstände zur Optimierung der Kalkspatrückgewinnung und des Reagenzverbrauchs,
- Produktion marktfähiger Nebenprodukte aus den Feinanteilen der Aufbereitungsrückstände,
- die Rückführung des Prozesswassers,
- die Schaffung einer hohen Dichte bei den zu deponierenden verschlämmten Aufbereitungsrückständen,

- die Verwendung von Seewasser als Transportmittel für die verschlämmten Aufbereitungsrückstände,
- die Vermeidung des Eintritts von Luft in das Pumpsystem,
- die Vermeidung von Leckagen und Verschüttungen an der Meeresoberfläche,
- die Verwendung eines Rohrauslasses für die Aufbereitungsrückstände 20 m unter der Meeresoberfläche.

Wartung:
- ein redundantes Pump- und Rohrleitungssystem in den Verklappungsbereich,
- vorbeugende Instandhaltung an den für die Aufbereitungsstoffen notwendigen Systeme,
- regelmäßige Unterwasser-Inspektionen an den Rohren für die Aufbereitungsrückstände,
- regelmäßige Unterwasser-Inspektionen am Auslassbereichs aus den Rohren,
- Einschätzung der Notwendigkeit einer Umsetzung der Rohre für den Transport der Aufbereitungsrückstände.

Die Vorteile, wie in Abschnitt **4.3.1.2.2** aufgeführt, sind:
- Verringerung der technologischen Anforderungen (keine Dämme erforderlich),
- höhere chemische Stabilität,
- Verringerung des Platzbedarfs für die Standfläche an Land.

Dieses Verfahren lässt sich dort anwenden, wo die verschlämmten Aufbereitungsrückstände eine hochdichte „Abwasserfahne“ bilden, die auf den Meeresboden sinkt und oberhalb des Rohrauslasses klares Wasser zurücklässt.

4.5.4 Andere Verwendungsmöglichkeiten für Aufbereitungsrückstände und taubes Gestein

In einigen Kohleabbaubetrieben werden feinkörnige Aufbereitungsrückstände von <0,5 mm aus der Flotation zunächst auf einen Feststoffgehalt von 40 – 51 % eingedickt. Um ihre Eignung für das Absetzen auf Halden mit groben Aufbereitungsrückständen zu erreichen, werden sie in Rahmenfilterpressen mit mehr als 1 000 m² Filterbereich oder mit Zentrifugen weiter entwässert (siehe Abschnitt **2.3.1.10**). Der Wasseranteil der in den Zentrifugen entwässerten Feinstpartikel der Aufbereitungsrückstände ist etwa zwei Mal so hoch wie der aus den Rahmenfilterpressen. [79, DSK, 2002].

Aufbereitungsrückstände aus der Kohleförderung (grobe und feine) werden oft als Zuschlagsstoffe oder für andere externe Zwecke verwendet [79, DSK, 2002], [84, IGME, 2002]. Einige spezielle Beispiele sind nachfolgend aufgeführt:

- Erdbau, d. h. als Grundmaterial für geeignete Projekte im Erdbau. Spezielle Anwendungsbereiche sind
  - Dämme und Anschüttungen für Straßen, Lärmschutzwälle und andere Erdbauten,
  - Bodenverbesserungsmaßnahmen (Bodenaustausch und mechanische Bodenmelioration),
  - die Bodenverfestigung durch Einbau von Kohleaufbereitungsrückstände und hydraulischen Bindemitteln,
  - andere Erdbauten (z. B. Deiche, Landschaftsgestaltung und landschaftsgestalterische Bauten),
  - Auskleidungen (z. B. Boden und Oberflächenauskleidungen/-abdeckungen bei Kippen und Deponien sowie Brunnenschachtauskleidungen).

- hydraulisches Engineering:
  - Füllmaterial für nicht mehr genutzte Hafenbecken,
  - Deichbau entlang von Flüssen,
  - Erweiterungen und Sicherheitsmaßnahmen am deutschen Kanalsystem.

- Deponiebau: mit Ton vermischte Aufbereitungsrückstände als Boden- und Oberflächenauskleidung/-abdeckung als Alternative zu natürlichem Ton.

- weitere Anwendungen:
  - in geringem Umfang in der Ziegelindustrie,
  - gegenwärtige Entwicklungsaktivitäten sind auf den Einsatz von Kohleaufbereitungsrückständen bei keramischen Prozessen gerichtet.

Die groben Aufbereitungsrückstände aus den schwedischen Eisenerzgruben sind auch für externe Verwendungszwecke geeignet.

In Kalkbetrieben ist der Einsatz von Aufbereitungsrückständen als Filtersand getestet worden und hat mit sehr feinem Material gute Ergebnisse erbracht. Die Kalkstein-Aufbereitungsrückstände entfernen das Feinmaterial anderer Abwasserströme [131, IMA, 2003].

Folgende unterschiedliche Beispiele stammen aus gegenwärtigen Anwendungen in der deutschen Barytindustrie:

- Verwendung innerhalb der Branche:
  - Einsatz von gebrochenem tauben Gestein für den Straßenbau,
  - grobe Aufbereitungsrückstände (< 1 – 16 mm) und taubes Gestein als Verfüllmaterial,
  - mittelfeine Aufbereitungsrückstände (< 4 mm) als Zuschlagstoffe für Spritzbetonmischungen für die Bauindustrie als Ersatz für Sand,
  - entwässerte feine Aufbereitungsrückstände (< 1 mm) als Verfüllmaterial.

- Verwendung außerhalb der Branche:
  - gebrochenes taubes Gestein für den Bodenaustausch zur Verbesserung des Bodenzustandes unter Fundamenten in der Bauindustrie, als Zuschlagstoff in der Bauindustrie (Kiesauftrag, Kiesstraßen),
  - grobe abgesetzte Aufbereitungsrückstände (1 – 16 mm) al Zuschlagstoff in der Bauindustrie (Verfüllung von Rohrleitungsgräben, Kiesauflage für Parkflächen usw.),
  - mittelfeine abgesetzte gesiebte Aufbereitungsrückstände (1 – 4 mm) als Zuschlagstoff für Pflaster,
  - feine Aufbereitungsrückstände (< 1 mm) als Füllmaterial für die Zementindustrie.

4.6 Minderung von Unfallursachen und -folgen

4.6.1 Notfallplanung

Anlagen zur Bewirtschaftung von Aufbereitungsrückständen und/oder taubem Gestein verfügen gewöhnlich über dokumentierte Notfallpläne für kritische Situationen und Havarieszenarien. Die Notfall- und Havariemaßnahmen werden zusammen mit den zuständigen Behörden geplant.

Notfallpläne enthalten die im Falle möglicher oder tatsächlicher Havarien zu ergreifenden Maßnahmen. Dieser Plan enthält das Organigramm des Betreibers und führt im Einzelnen alle Verantwortlichkeiten jeder Person sowie Kontakte und Verbindungen mit externen Organisationen auf. Der Notfallplan enthält darüber hinaus auch einen Maßnahmeplan mit einer Beschreibung operativer Maßnahmen und verfügbarer Ressourcen zur Eindämmung der Folgen soweit dies möglich ist.

Ziel der Notfallplanung ist es,
- das Risiko eines Versagens der Strukturen und des Bauwerks der Abfallbehandlungsanlage zu verringern sowie von den Menschen und von der Umwelt Schaden abzuhalten,
- die Notwendigkeit des Improvisierens im Krisen- oder Havariefall zu verringern,
- die optimale Nutzung der verfügbaren Ressourcen abzusichern,
- die Verantwortlichkeiten auf jeder Ebene zu benennen und festzulegen sowie

- abzusichern, dass jede Person innerhalb der Organisation sowie die Öffentlichkeit und die betreffenden Behörden mit den notwendigen Informationen versorgt werden.

HINWEIS: Siehe des Weiteren die in Artikel 6 Abs. 4 des Kommissionsvorschlages für eine Richtlinie über die Bewirtschaftung von Abfällen aus der mineralgewinnenden Industrie (KOM 2003, 319 endgültig).

In der Vorbereitungsphase werden bereits alle anormalen Vorkommnisse, die ein Schadensrisiko für Menschen, die Einrichtung und/oder die Umwelt darstellen, soweit wie möglich identifiziert, ausgewertet und analysiert. Die Ergebnisse bilden die Grundlage für die Notfallplanung.

Siehe auch Abschnitt 4.2.1.3 und Artikel 6 sowie Anhang I in der vorgeschlagenen Richtlinie (COM(2003)319 endgültig) zur Bewirtschaftung von Abfällen aus den extraktiven Industrien.

Abschnitt 4.4.16 befasst sich mit Fragen der Bewirtschaftung von Cyan, die auf eine Verhinderung von Unfällen bzw. der Minderung ihrer Ursachen und Folgen gerichtet sind.

4.6.2 Auswertung von und Nachfolgemaßnahmen bei Vorfällen

Um Lehren aus Vorfällen ziehen zu können, die sich bereits ereignet haben, ist es wichtig, über ein System zur Dokumentierung aller Informationen und der Vorgehensweisen im Nachgang zu solchen Vorfällen verfügen zu können. Wenn sich ein Vorfall ereignet, wird er nicht nur gemeldet, sondern er wird auch dokumentiert, z. B. wie und warum es dazu kam. Gleichzeitig werden Vorschläge entwickelt, wie eine Wiederholung verhindert werden kann. Es werden Personen für die Umsetzung der vorgeschlagenen Maßnahmen verantwortlich gemacht und Termine für den Abschluss der Maßnahmen festgelegt.

Vorteile:
- Sowohl kleinere als auch größere Vorfälle werden gemeldet und dokumentiert.
- Ist das System computergestützt, kann leicht nachverfolgt werden, ob und dass die zur Verhinderung einer Wiederholung des Vorfalls festgelegten Maßnahmen ausgeführt werden bzw. wurden.
- Es kann leicht festgestellt werden, ob eine bestimmte Art von Vorfällen vermehrt und wiederholt auftritt.

Nachteile:
- Die Einrichtung und volle Entwicklung eines solchen Systems erfordert eine Menge Arbeit.

[118, Zinkgruvan, 2003]

4.6.3 Bruch von Rohrleitungen mit Aufbereitungsrückständen

In der Anlage **Neves Corvo** sind die Rohrleitungen für Aufbereitungsrückstände an beiden Enden mit Strömungsmessern ausgerüstet (d. h. sowohl an der Rohstoffaufbereitungsanlage als auch am Auslass). Treten Differenzen zwischen den Strömungsraten an beiden Enden auf, wird Alarm ausgelöst. Druckmessungen an Rohrleitungen mit Aufbereitungsrückständen sind schwierig zu interpretieren, da es oft zu Schwankungen in der Strömung (und damit auch beim Druck) kommt.

Entlang der Rohrleitung mit den Aufbereitungsrückständen ist ein Graben gezogen geworden, der im Falle eines Bruchs an der Rohrleitung den auslaufenden Schlamm in Notbecken ableitet. Diese Notbecken sind so ausgelegt, dass sie die Aufbereitungsrückstände eines Produktionszeitraumes von 8 Stunden aufnehmen können, was etwa der Gesamtzeit entspricht, den Zufluss herunterzufahren.

Wo die Rohrleitungen mit den Aufbereitungsrückständen einen Fluss kreuzen, befindet sich unter ihnen ein Auffangblech, das Aufbereitungsrückstände auffängt und sie in Notfallbecken leitet.
[142, Borges, 2003]

Es gibt andere Verfahren, die ein ähnliches Ausmaß an Umweltschutz erreichen, z. B.:
- Der Einsatz doppelt verrohrter Leitungen statt Auffangbleche in ökologisch empfindlichen Bereichen.
- An vielen Standorten gibt es ein oder mehrere Ersatzleitungen, deren Kapazität sofort zugeschaltet werden kann. Das ersetzt den Bau großer Notfallbecken über den gesamten Zeitraum, der für das Herunterfahren der Anlage benötigt würde.

Ein Graben unter der Rohrleitung kann zu verstärktem Eindringen von Prozesswasser und möglicher Schadstoffe führen und die Wartung der Rohrleitung schwierig gestalten. Einen Graben oder ein Auffangblech zu warten kann auch unter klimatischen Bedingungen schwierig sein, wo sich der Graben regelmäßig mit Wasser, Eis oder Schnee füllt.

4.7 Managementwerkzeuge für den Umweltschutz

Beschreibung

Die beste Umweltbilanz wird gewöhnlich durch den Einbau der fortgeschrittensten Technologien und ihren Betrieb auf der wirksamsten und effizientesten Weise erreicht. Das wird auch durch die Definition des Begriffes ‘Techniken’ anerkannt, nämlich *“sowohl die eingesetzten Technologien als solche auch die Art und Weise, wie die Anlage ausgelegt, gebaut, gewartet, betrieben und stillgelegt ist bzw. wird”.*

Ein Umweltmanagementsystem (EMS) für Anlagen zur Bewirtschaftung von Aufbereitungsrückständen und taubem Gestein ist ein Werkzeug, das Betreiber zur Auseinandersetzung mit und Klärung von Fragen bzgl. der Auslegung, des Baus, der Wartung, des Betreibens und der Stilllegung der Anlagen auf systematische und anschauliche Weise verwenden können. Zu einem EMS gehören der organisatorische Aufbau, die Verantwortlichkeiten, praktischen Methoden, Vorgehensweisen, Prozesse und Ressourcen für die Erstellung, Umsetzung, Aufrechterhaltung, Überprüfung und Überwachung von Umweltgrundsätzen. Umweltmanagementsysteme sind dort am wirksamsten und effizientesten, wo sie integraler Bestandteil des Gesamtmanagements und des Betriebes einer Anlage zur Bewirtschaftung von Aufbereitungsrückständen sind.

Innerhalb der Europäischen Union haben sich viele Organisationen freiwillig zur Umsetzung eines Umweltmanagementsystems auf Grundlage der Norm EN ISO 14001:1996 oder des Öko-Management- und Prüfprogramms EMAS der EU entschieden. EMAS schließt die Anforderungen an Bewirtschaftungssysteme nach EN ISO 14001 ein, legt aber zusätzlich großen Wert auf die Einhaltung der gesetzlichen Vorschriften, auf die Umweltbilanz und die Einbeziehung des Personals; das Programm verlangt drüber hinaus eine externe Verifikation des Bewirtschaftungssystems und die Validierung einer öffentlichen Umwelterklärung (EN ISO 14001 sieht eine Eigenerklärung als Alternative zur externen Verifikation vor). Es gibt auch viele Betriebe, die die Einführung eines nichtstandardisierten EMS beschlossen haben.

Während sowohl die standardisierten Systeme (EN ISO 14001:1996 und EMAS) als auch die nichtstandardisierten (‘kundenspezifischen`) Systeme im Prinzip das *Unternehmen* als eine Einheit auffassen, verfolgt dieses Dokument eine viel engere Herangehensweise und erfasst bei weitem nicht alle Aktivitäten des Unternehmens z. B. hinsichtlich seiner Erzeugnisse und Leistungen.

Ein Umweltmanagementsystem (EMS) kann folgende Komponenten enthalten:
(a) Definition von Umweltgrundsätzen,
(b) Planung und Aufstellung entsprechender Ziele,
(c) Umsetzung und Praktizierung von Vorgehensweisen,
(d) Kontroll- und Korrekturmaßnahmen,
(e) Überprüfung des Systemmanagements,
(f) Erstellung einer regelmäßigen Umwelterklärung,
(g) Validierung durch ein Zertifizierungsgremium oder externe EMS-Prüfer,
(h) Überlegungen zur Gestaltung der Anlagenstilllegung am Ende ihres Betriebszeitraumes,
(i) Entwicklung noch sauberer Technologien,
(j) Benchmarking/Leistungsvergleiche.

Diese Komponenten werden nachstehend etwas ausführlicher erklärt. Für weitere Informationen zu (a) - (g), die alle zu EMAS gehören, wird der Leser auf die nachfolgenden Literaturangaben verwiesen.

(a) Definition von Umweltgrundsätzen

Das Spitzenmanagement ist für die Definition von Umweltgrundsätzen für eine Anlage verantwortlich und muss sicherstellen, dass diese Grundsätze

- in Hinblick auf die Art, das Ausmaß und die Umweltauswirkungen der (beabsichtigten oder durchgeführten) Aktivitäten angemessen sind,
- eine Verpflichtung zur Verhinderung der Umweltverschmutzung und entsprechende Kontrollmaßnahmen enthalten,
- eine Verpflichtung enthalten, die einschlägigen und anwendbaren Umweltgesetze und -bestimmungen sowie alle anderen Anforderungen einzuhalten, denen sich der Betrieb verpflichtet fühlt,
- ein Rahmenwerk für das Aufstellen & Überprüfen ökologischer Zielstellungen schaffen,
- dokumentiert und allen Mitarbeitern übermittelt werden,
- der Öffentlichkeit und allen interessierten Kreisen zur Verfügung gestellt werden.

(b) Planung, d. h.:

- Vorgehensweisen zur Identifizierung der ökologischen Aspekte der Anlage, um jene Aktivitäten bestimmen zu können, die wichtige Auswirkungen auf die Umwelt haben oder haben könnten und um diese Informationen auf dem aktuellen Stand zu halten;
- Vorgehensweisen zur Identifizierung rechtlicher und anderer Anforderungen, denen sich der Betrieb verpflichtet fühlt und die auf die ökologischen Aspekte der betrieblichen Aktivitäten anwendbar sind sowie sich den nötigen Zugriff auf sie zu verschaffen;
- Aufstellung dokumentierter Umweltziele und deren Überprüfung unter Berücksichtigung der rechtlichen und anderen Anforderungen sowie der Ansichten der daran interessierten Kreise;
- Aufstellung eines Umweltmanagementprogramms und dessen regelmäßige Aktualisierung, einschließlich der Festlegung von Verantwortlichkeiten zur Erreichung der Ziele für jede relevante Arbeitsstelle und Funktionsebene sowie der Mittel und eines Zeitrahmens für ihre Erfüllung.

(c) Umsetzung und Praktizierung von Vorgehensweisen

Es ist wichtig, Systeme zu haben, die sicherstellen, dass die Vorgehensweisen bekannt sind, verstanden werden und eingehalten werden. Zu einem wirksamen Umweltmanagement gehören daher:

(i) Hierarchien und Verantwortlichkeiten

– das Festlegen, Dokumentieren und Übertragen von Rollen, Verantwortlichkeiten und Vollmachten, was die Ernennung eines speziellen Managementvertreters einschließt;
– das Bereitstellen von Ressourcen, die für die Umsetzung des Umweltmanagementsystems und seine Kontrolle wichtig sind, einschließlich der Bereitstellung von Personal mit Spezialkenntnissen, von Ausrüstungen und finanziellen Mitteln.

(ii) Ausbildung/Schulung, Erhöhung des Umweltbewusstseins und der Sachkompetenz
– Identifizierung des Ausbildungs-/Schulungsbedarfs um zu gewährleisten, dass alle Mitarbeiter, deren Arbeit die ökologischen Auswirkungen der betrieblichen Aktivitäten wesentlich beeinträchtigen kann, auch angemessen ausgebildet und geschult worden sind.

(iii) Kommunikation
– Aufstellen und Umsetzung von Verfahrensweisen für die interne Kommunikation zwischen den verschiedenen Funktionsebenen und Arbeitsstellen der Einrichtung, von Vorgehensweisen, die den Dialog zwischen interessierten Kreisen außerhalb des Betriebes befördern, sowie von Vorgehensweisen für die Entgegennahme und Dokumentierung relevanter Mitteilungen von interessierten Kreisen außerhalb des Betriebes sowie, wo angemessen, das Reagieren darauf.

(iv) Einbeziehung der Mitarbeiter
– Einbeziehung der Mitarbeiter in den auf ein hohes Niveau der Umweltbilanz abzielenden Prozess durch Anwendung entsprechender Formen der Mitbestimmung, wie das Vorschlagswesen, projektbezogene Gruppenarbeit oder Umweltausschüsse.

(v) Dokumentation
– Erarbeitung und Vorhalten aktueller Informationen in Papier- oder elektronischer Form zur Beschreibung der Kernelemente des Managementsystems und ihres Zusammenwirkens sowie Hinweise auf die damit in Verbindung stehende Dokumentation.

(vi) Effiziente Prozesskontrolle
– adäquate Kontrolle der Prozesse in allen Betriebsarten, d. h. während der Vorbereitung, des Anfahrens, des Routinebetriebes, der Stilllegung und während anormaler Zustände;
– Identifizierung der wichtigsten Leistungsindikatoren und der Verfahren zur Messung und zur Kontrolle dieser (z. B. Strömung, Druck, Temperatur, Zusammensetzung und Menge)
– Dokumentieren und Analysieren anormaler Betriebsbedingungen zur Identifizierung der eigentlichen Ursachen sowie das Behandeln der Ursachen um zu gewährleisten, dass sich solche Ereignisse nicht wiederholen (was durch eine Kultur geschehen kann, in der die Identifizierung der Ursachen wichtiger ist als die Suche nach einem „Sündenbock“).

(vii) Wartungsprogramm
– Aufstellen eines gegliederten Wartungsprogramms auf Grundlage der technischen Beschreibung der Ausrüstung, der Normen usw. sowie für Ausfälle und deren Konsequenzen;
– Untersetzung des Wartungsprogramms durch entsprechende Aufzeichnungssysteme und diagnostisches Prüfen;
– klare Zuweisung von Verantwortlichkeiten für die Planung und Durchführung der Instandhaltung.

(viii) Auf Notsituationen vorbereitet sein und reagieren
– Festlegen und Umsetzen von Vorgehensweisen zur Identifizierung des Unfallpotentials und für die Reaktion auf Unfälle und Notsituationen sowie für die

Verhinderung bzw. Minderung von mit ihnen einhergehender umweltschädlicher Auswirkungen.

(d) Kontroll- und Korrekturmaßnahmen, d. h.:

(i) Überwachen und Messen
- Erarbeitung und Umsetzung dokumentierter Vorgehensweisen zur regelmäßigen Überwachung und Messung der Haupteigenschaften der betrieblichen Aktivitäten, die wichtige umweltrelevante Auswirkungen haben, einschließlich der Aufzeichnung von Informationen zur Rückverfolgung, für relevante betriebliche Kontrollen und für Nachprüfungen, wie die Anlage die Umweltziele einhält *(siehe auch das Merkblatt zur Emissionsüberwachung);*
- Aufstellen und Umsetzen einer dokumentierten Vorgehensweise für die periodische Einschätzung der Einhaltung der relevanten Umweltgesetze und -bestimmungen.

(ii) Abhilfe- und Präventivmaßnahmen
- Aufstellen und Umsetzen von Vorgehensweisen für das Festlegen von Verantwortlichkeiten und Vollmachten für die Behandlung und Untersuchung der Nichteinhaltung von Genehmigungsauflagen, anderer rechtlicher Anforderungen sowie der festgelegten Ziele, für das Ergreifen von Maßnahmen zur Minderung der dadurch entstanden schädlichen Auswirkungen sowie für die Einleitung und Durchsetzung von Abhilfe- und Präventivmaßnahmen, die der Größe des Problems und den auftretenden Auswirkungen auf die Umwelt angemessenen sind.

(iii) Unterlagen/Aufzeichnungen
- Aufstellen und Umsetzen von Vorgehensweisen für die Identifikation, das Vorhalten und die Bereitstellung lesbarer, identifizierbarer und nachverfolgbarer Umweltaufzeichnungen, einschließlich der Ausbildungs- und Schulungsunterlagen sowie der Audit- und Überprüfungsergebnisse.

(iv) Audit
- Erarbeitung und Umsetzung eines Programms/von Programmen sowie Vorgehensweisen für die von unparteiischen und objektiven Mitarbeitern (interne Audits) bzw. externen Prüfern (externe Audits) durchzuführenden periodischen Audits des Umweltmanagementsystems, wozu auch Diskussionen/Beratungen mit dem Personal, die Inspektion der Betriebsbedingungen sowie die Überprüfung der Aufzeichnungen und der Dokumentation gehören und die in einen schriftlichen Bericht münden. Diese Programme bzw. Vorgehensweisen legen den Umfang, die Häufigkeit und die Methodik der Audits sowie die Verantwortlichkeiten für und die Anforderungen an die Durchführung der Audits und die zu berichtenden Ergebnisse fest, um bestimmen zu können, ob das Umweltmanagementsystem den geplanten Arrangements entspricht und ordnungsgemäß umgesetzt und durchgeführt wird;
- Fertigstellung des Audit bzw., wenn zutreffend, des Auditzyklus in Abständen von nicht größer als drei Jahren, je nach Art, Ausmaß und Komplexität der Aktivitäten, je nach Bedeutung der damit verbundenen Auswirkungen auf die Umwelt sowie je nach Wichtigkeit und Dringlichkeit der bei vorangegangenen Audits festgestellten Probleme und der Geschichte ökologischer Probleme – komplexere Aktivitäten mit schwerwiegenderen Auswirkungen auf die Umwelt werden öfter einem Audit unterzogen;
- Einrichten entsprechender Mechanismen zur Absicherung, dass die Auditergebnisse nachbereitet werden.

(v) Periodische Auswertung der Einhaltung der gesetzlichen Bestimmungen
- Überprüfung der Einhaltung der anwendbaren Umweltgesetzgebung und der dem Betrieb in der/den Umweltgenehmigung(en) erteilten Auflagen;
- Dokumentierung der Auswertung.

(e) Überprüfung des Systemmanagements, d. h.:

- Überprüfung des Umweltmanagementsystems durch das Spitzenmanagement in von ihm festgelegten Abständen, um die anhaltende Eignung, Adäquatheit und Wirksamkeit des Umweltmanagementsystems zu gewährleisten;
- Absichern, dass die notwendigen Information erfasst werden, um den Management diese Einschätzung zu ermöglichen;
- Dokumentation der Überprüfung.

(f) Erstellung einer regelmäßigen Umwelterklärung:

- Erstellung einer Umwelterklärung, die den durch die Anlage erreichten Ergebnissen hinsichtlich der ökologischen Zielstellungen besondere Aufmerksamkeit schenkt. Die Umwelterklärung wird in regelmäßigen Abständen erstellt - einmal jährlich oder in längeren Abständen, je nach der Schwere der Emissionen, des Umfangs der Abfallerzeugung usw. – und nimmt Rücksicht auf die Informationsbedürfnisse der entsprechenden interessierten Kreise und steht der Öffentlichkeit zur Verfügung (z. B. in elektronischen Veröffentlichungen, in Büchereien usw.).

Für die Erstellung einer Umwelterklärung kann der Betreiber auf vorhandene umweltrelevante Leistungsindikatoren zurückgreifen, wobei gewährleistet sein muss, dass die gewählten Indikatoren

i. eine exakte Bewertung der Leistung der Anlage vornehmen,
ii. verständlich und eindeutig sind,
iii. einen Vergleich von Jahr zu Jahr ermöglichen, um die Entwicklung der Umweltbilanz der Anlage einschätzen zu können,
iv. je nach Bedarf einen Vergleich mit nationalen oder regionalen Bezugswerten bzw. innerhalb der Branche erlauben und
v. je nach Bedarf einen Vergleich mit den gesetzlichen Bestimmungen erlauben.

(g) Validierung durch ein Zertifizierungsgremium oder externe EMS-Prüfer:

- Die Überprüfung und Validierung des Managementsystems, der Vorgehensweise beim Audit und der Umwelterklärung durch ein zugelassenes Zertifizierungsgremium oder einen externen EMS-Prüfer kann, wenn ordnungsgemäß durchgeführt, die Glaubwürdigkeit des Systems erhöhen.

(h) Überlegungen zur Gestaltung der Anlagenstilllegung am Ende ihres Betriebszeitraumes

- Berücksichtigung der ökologischen Auswirkungen einer endgültigen Stilllegung der Anlage im Entwurfsstadium für eine neue Anlage, da eine vorausschauende Betrachtung die Stelllegung leichter, sauberer und billiger macht;
- Die Stilllegung stellt ein ökologisches Risiko für die Verunreinigung des Landes (und des Grundwassers) dar und erzeugt große Mengen festen Abfalls. Die vorbeugenden Techniken sind zwar prozess-spezifisch, aber die allgemeinen Überlegungen sollten folgendes beinhalten:
  i. Vermeidung von Bauten unter Tage,
  ii. Einbeziehung von Elementen, die eine Demontage ermöglichen,
  iii. Auswahl von Oberflächenabschlüssen, die leicht dekontaminiert werden können,
  iv. Einsatz einer Geräteanordnung, die eingebundene Chemikalien minimiert und das Entwässern oder Auswaschen ermöglicht,
  v. Entwerfen flexibler, autarker Einheiten, die die stufenweise Schließung ermöglichen,
  vi. Verwenden biologisch abbaubarer und wieder verwendbarer (recycelbarer) Materialien, wo immer möglich.

(i) Entwicklung noch sauberer Technologien:

- Umweltschutz sollte ein inhärentes Merkmal aller vom Betreiber unternommenen Aktivitäten bei der Entwicklung verfahrenstechnischer Lösungen sein, da Techniken, die im frühestmöglichen Entwurfsstadium integriert werden, sowohl wirksamer als auch billiger sind. Der Überlegung, noch sauberere Techniken zu entwickeln, kann zum Beispiel durch F&E-Aktivitäten und Studien zum Durchbruch verholfen werden. Als eine Alternative zu den betriebsinternen Aktivitäten können, wenn angemessen, auch Arrangements getroffen werden, Aufträge an andere Betreiber oder an auf diesem Gebiet tätige Forschungsinstitutionen zu vergeben, um auf der Höhe der Zeit zu bleiben.

(j) Benchmarking/Leistungsvergleiche, d. h.:

– Vornahme systematischer und regulärer Vergleiche mit nationalen oder regionalen Bezugswerten bzw. innerhalb der Branche bzgl. des Wirkungsgrades der Energie, der Energiekonservierungsaktivitäten, der Auswahl von Inputmaterialien, der Emissionen in die Luft und der Einleitungen ins Wasser (z. B. Verwendung des Europäischen Schadstoffemissionsregister [*European Pollutant Emission Register*, EPER]), des Wasserverbrauchs und der Erzeugung von Abfall.

Standardisierte und nichtstandardisierte EMSs

Ein EMS kann in Form eines standardisierten oder nichtstandardisierten ("kundenspezifischen") Systems existieren. Die Umsetzung und Wahrung eines international anerkannten standardisierten Systems, wie EN ISO 14001:1996, kann dem EMS eine größere Glaubwürdigkeit verleihen, besonders wenn es einer ordnungsgemäß durchgeführten externen Verifikation unterworfen wird. EMAS schafft zusätzliche Glaubwürdigkeit auf Grund der Zusammenarbeit mit der Öffentlichkeit in Form der Umwelterklärung und des Mechanismus, der die Einhaltung der anwendbaren Umweltgesetzgebung gewährleistet. Dennoch können nichtstandardisierte Systeme prinzipiell gleichermaßen wirksam sein, vorausgesetzt, sie sind ordnungsgemäß entworfen und werden ebenso ordnungsgemäß umgesetzt.

Erreichter Nutzen für die Umwelt

Die Umsetzung und Beibehaltung eines EMS lenkt die Aufmerksamkeit des Betreibers auf die Umweltbilanz einer Anlage. Insbesondere die Aufrechterhaltung und Einhaltung eindeutiger betrieblicher Vorgehensweisen sowohl in normalen als auch unnormalen Situationen und den damit verbundenen Verantwortlichkeitshierarchien sollten gewährleisten, dass die Auflagen in der Genehmigung für die Anlage und andere Umweltziele jederzeit erfüllt werden.

Umweltmanagementsysteme gewährleisten typischerweise die ständige Verbesserung der Umweltbilanz der Anlage. Je schlechter die Ausgangssituation, desto wichtigere kurzfristige Verbesserungen können erwartet werden. Wenn die Anlage bereits eine gute Gesamtumweltbilanz hat, hilft das System dem Betreiber, dieses hohe Leistungsniveau aufrechtzuerhalten.

Medienübergreifende Auswirkungen

Umweltmanagementtechniken werden entworfen, um sich den Gesamtauswirkungen auf die Umwelt zu widmen.

Betriebsdaten

Es liegen keine speziellen Angaben vor.

Anwendbarkeit

Die oben beschriebenen Komponenten können normalerweise auf alle Anlagen angewandt werden. Der Umfang (wie die Detailebene) und die Natur des EMS (z. B. standardisiert oder nichtstandardisiert) beziehen sich im Allgemeinen auf die Art, die Größe und die Komplexität der Anlage sowie auf die Reichweite der ökologischen Auswirkungen, die sie haben mag.

Wirtschaftlichkeit

Es ist schwierig, die Kosten und den wirtschaftlichen Nutzen der Einführung und Umsetzung eines guten EMS genau zu bestimmen. Im Folgenden wird eine Anzahl von Studien vorgestellt. Diese Studien sind jedoch nur Beispiele und ihre Ergebnisse sind nicht völlig kohärent. Sie sind möglicherweise nicht für alle Branchen in der EU repräsentativ und sollten daher mit Vorsicht genossen werden.

Eine 1999 durchgeführte schwedische Studie hat alle 360 ISO-zertifizierten und EMAS-registrierten Unternehmen in Schweden untersucht. Bei einer Antwortquote von 50 % kam die Studie u.a. zu folgenden Schlussfolgerungen:

- Die Kosten für die Einführung und das Umsetzen eines EMS sind hoch, wenngleich auch nicht unangemessen hoch, außer im Falle ganz kleiner Unternehmen. Es wird erwartet, dass die Ausgaben im Laufe der Zeit sinken.
- Ein höheres Maß an Koordination und Integration des EMS mit anderen Managementsystemen wird als möglicher Weg der Kostensenkung angesehen.
- Die Hälfte aller Umweltziele zahlen sich innerhalb eines Jahres durch Kosteneinsparungen und/oder höhere Einnahmen aus.
- Die größten Kosteneinsparungen wurden durch sinkende Energiekosten, die Behandlung von Abfall und bei Rohstoffen erzielt.
- Die meisten Unternehmen denken, dass sich ihre Marktposition durch das EMS gefestigt hat. Ein Drittel der Unternehmen meldet steigende Einnahmen auf Grund des EMS.

In einigen Mitgliedsstaaten wird eine geringe Überwachungsgebühr berechnet, wenn die Anlage zertifiziert ist.

Eine Anzahl von Studien[16] zeigt, dass es eine Umkehrrelation zwischen der Unternehmensgröße einerseits und den Kosten für die Einführung und Umsetzung eines EMS andererseits gibt. Eine ähnliche Umkehrrelation besteht beim Zeitraum für die Rückzahlung des investierten Kapitals. Beide Elemente implizieren ein weniger günstiges Kosten-Nutzen-Verhältnis bei der Einführung eines EMS in kleineren und mittleren Unternehmen (SMEs) im Vergleich zu größeren.

Nach einer Schweizer Studie können die durchschnittlichen Kosten für die Ein- und Durchführung der Norm ISO 14001 schwanken:

- bei einem Unternehmen zwischen 1 und 49 Mitarbeitern: CHF 64 000 (EUR 44 000) für das Einführen des EMS und CHF 16 000 (EUR 11 000) pro Jahr für das Umsetzen;
- bei einem Industriestandort mit mehr als 250 Mitarbeitern: CHF 367 000 (EUR 252 000) für die Einführung des EMS und CHF 155 000 (EUR 106 000) pro Jahr für das Umsetzen.

Diese Durchschnittswerte stellen nicht notwendigerweise die tatsächlichen Kosten für einen existierenden Industriestandort dar, denn die Kosten hängen auch in starkem Maße von der Anzahl anderer ausschlaggebender Faktoren ab (Schadstoffe, Energieverbrauch, ...) sowie von der Komplexität des zu untersuchenden Problems.

[16] Z. B. Dyllick und Hamschmidt (2000, 73) zitiert in Klemisch H. und R. Holger, *Umweltmanagementsysteme in kleinen und mittleren Unternehmen – Befunde bisheriger Umsetzung,* KNI Papers 01/02, Januar 2002, S 15; Clausen J., M. Keil und M. Jungwirth, *The State of EMAS in the EU. Eco-Management as a Tool for Sustainable Development – Literature Study,* Institut für Ökologische Wirtschaftsforschung (Berlin) und Ökologie – Institut für Internationale und Europäische Umweltpolitik (Berlin), 2002, S. 15.

Eine jüngere deutsche Studie (Schaltegger, Stefan und Wagner, Marcus, *Umweltmanagement in deutschen Unternehmen - der aktuelle Stand der Praxis*, Februar 2002, S. 106) weist die folgenden Kosten für EMAS für unterschiedliche Branchen aus. Es kann festgestellt werden, dass diese Zahlen viel niedriger sind als die in der Schweizer Studie angegebenen. Das ist eine Bestätigung der Schwierigkeit, die Kosten für ein EMS bestimmen zu können.

Kosten für die Einführung (EUR):

Minimum	- 18750
Maximum	- 75000
Durchschnitt	- 50000

Kosten für die Validierung (EUR):

Minimum	- 5000
Maximum	- 12500
Durchschnitt	- 6000

Eine Studie des Deutschen Unternehmerinstituts/Arbeitsgemeinschaft Selbständiger Unternehmer UNI/ASU aus dem Jahre 1997 (*Umweltmanagementbefragung - Öko-Audit in der mittelständischen Praxis - Evaluierung und Ansätze für eine Effizienzsteigerung von Umweltmanagementsystemen in der Praxis*, Bonn) liefert Informationen über die mit EMAS erzielten durchschnittlichen jährlichen Einsparungen und über den durchschnittlichen Amortisationszeitraum. Zum Beispiel wurden bei Umsetzungskosten von EUR 80 000 durchschnittliche jährliche Einsparungen von EUR 50 000 festgestellt, was mit einem Amortisationszeitraum von anderthalb Jahren zusammenfällt.
Externe, sich auf die Verifikation des Systems beziehende Kosten können mit Hilfe der vom *International Accreditation Forum* (http://www.iaf.nu) herausgegebenen Richtlinien abgeschätzt werden.

Triebkräfte für die Einführung und Umsetzung eines EMS

Umweltmanagementsysteme bieten eine Reihe von Vorteilen, zum Beispiel:
- verbesserter Einblick in umweltrelevante Aspekte des Unternehmens,
- verbesserte Grundlage für das Treffen von Entscheidungen,
- verbesserte Motivation beim Personal,
- zusätzliche Möglichkeiten der Verringerung der Betriebskosten und der Verbesserung der Erzeugnisqualität,
- verbesserte Umweltbilanz,
- verbesserte Firmenreputation,
- geringe Haftungs- und Versicherungskosten sowie Kosten bei Nichteinhaltung der gesetzlichen Bestimmungen,
- größere Attraktivität für Mitarbeiter, Kunden und Investoren,
- größeres Vertrauen bei den Regulierungsbehörden, das zu einer geringeren regulatorischen Aufsicht führen könnte,
- verbesserte Beziehungen zu Umweltschutzgruppierungen.

Beispielanlagen

Die oben unter (a) bis (e) beschriebenen Merkmale sind Elemente der Norm EN ISO 14001:1996 und des Öko-Management- und Prüfprogramms der Europäischen Gemeinschaft (EMAS), während die unter (f) und (g) aufgeführten Merkmale aus dem EMAS stammen. Diese beiden standardisierten Systeme werden in einer Reihe von Einrichtungen verwandt, die nach der IVU-/IPPC-Richtlinie arbeiten. Im Juli 2002 waren beispielsweise 357 Betriebe in der Chemie- und Chemieerzeugnisindustrie der EU (NACE-Code 24) registriert, die EMAS eingeführt hatten, wovon die meisten Anlagen nach den IVU-/IPPC-Richtlinien betrieben werden.

In Großbritannien (UK) hat die *Environment Agency of England and Wales* 2001 eine Befragung unter Betrieben durchgeführt, die nach IPC-Richtlinien arbeiten (den Vorläuferrichtlinien von IPPC). Es stellte sich heraus, dass 32 % der Betriebe, die die Umfrage beantwortet hatten, nach ISO 14001 zertifiziert waren (was 21 % aller IPC-Betriebe entspricht) und 7 % bei EMAS registriert waren. Alle Zementwerke im UK (etwa 20) sind nach ISO 14001 zertifiziert und die Mehrzahl bei EMAS registriert. In Irland, wo die Aufstellung eines (nicht notwendigerweise standardisierten) EMS für den Erwerb von IPC-Lizenzen vorgeschrieben ist, haben etwa 100 von etwa 500 lizenzierten Anlagen ein EMS entsprechend ISO 14001 eingeführt, während sich die anderen 400 Anlagen für ein nichtstandardisiertes EMS entschieden haben.

Die folgende Aufzählung enthält Beispiele für Umweltmanagementsysteme, die in Europa angewandt werden:

- alle Kaolin-Betriebe im Vereinigten Königreich (UK) und die Grube Lisheen in Irland sind nach ISO 14001 zertifiziert,
- alle Genehmigungen in Irland verlangen ein EMS irgendeiner Art,
- die *Global Mining Initiative* und die damit in Verbindung stehenden Foren für *Minerals Mining & Sustainable Development* befürworten EMS,
- Die Grube S&B Industrial Minerals S.A in Griechenland ist nach ISO 14001 zertifiziert, die Grube Stratoni Mines, die sich im Besitz von TVX Hellas befindet, wurde nach ISO 14001 zertifiziert.

Verweisliteratur

(Vorschrift (EG) Nr 761/2001 des Europäischen Parliaments und des Rates, die eine freiwillige Teilnahme von Betrieben am Öko-Management- und Prüfprogramms (EMAS) gestattet, OJ L 114, 24/4/2001, http://europa.eu.int/comm/environment/emas/index_en.htm)

(EN ISO 14001:1996, http://www.iso.ch/iso/en/iso9000-14000/iso14000/iso14000index.html; http://www.tc207.org)

5 BESTE VERFÜGBARE TECHNIKEN FÜR DIE BEWIRTSCHAFTUNG VON AUFBEREITUNGSRÜCKSTÄNDEN, BERGEMATERIAL UND TAUBEM GESTEIN AUS BERGBAULICHEN AKTIVITÄTEN

5.1 Einleitung

Zum Verständnis dieses Kapitels und seines Inhalts wird die Aufmerksamkeit des Lesers erneut auf das Vorwort zu diesem Dokument und insbesondere auf dessen Abschnitt 5 "Zum Verständnis und zur Nutzung dieses Dokuments" gelenkt. Die in diesem Kapitel dargestellten Techniken und Leistungsebenen sind durch einen sich wiederholenden Prozess bewertet worden, der die folgenden Schritte umfasste:

- Identifikation der Hauptfragen bzgl. der Umwelt, der Risiken und der Sicherheit für den Bereich,
- Untersuchung der für die Behandlung dieser Hauptfragen relevanten Techniken,
- Identifikation der unter Umweltaspekten höchsten Leistungsfähigkeit auf Grundlage der in der Europäischen Union und weltweit verfügbaren Daten,
- Untersuchung der Bedingungen, unter denen diese Leistungen erreicht wurden, wie Kosten, medienübergreifende Auswirkungen sowie die in die Umsetzung dieser Techniken involvierten Haupttriebkräfte und
- Auswahl der im allgemeinen Sinne besten verfügbaren Techniken (BVT) für diesen Bereich.

Eine entscheidende Rolle bei jedem dieser Schritte und bei der Art und Weise, wie diese Informationen hier präsentiert werden, spielte das fachliche Urteil des Europäischen IPPC-Büros und der Technischen Arbeitsgruppe (TWG).

Auf der Grundlage dieser Einschätzung werden in diesem Kapitel Techniken dargestellt, die für den Bereich als Ganzes als geeignet angesehen werden und in vielen Fällen die gegenwärtig erreichbare Leistungsfähigkeit an einigen Standorten innerhalb des Bereichs widerspiegeln. Wo Leistungsebenen dargestellt werden, ist das so zu verstehen, dass diese Ebenen die Leistung in Hinblick auf Umwelt- und Sicherheitsbelange repräsentiert, die im Ergebnis der Anwendung der beschriebenen Techniken in diesem Bereich erwartet werden können, wobei die Ausgewogenheit von Kosten und Nutzen innerhalb der Definition der BVT nicht außer Acht gelassen werden darf. In einigen Fällen mag es durchaus technisch möglich sein, bessere Emissions- oder Verbrauchswerte zu erreichen, aber sie werden auf Grund der Kosten oder anderer (medienübergreifender) Erwägungen nicht als BVT für den Bereich als Ganzes für zweckmäßig erachtet. Dennoch können solche Werte in einigen speziellen Fällen durchaus als gerechtfertigt angesehen werden, wo es besondere Triebkräfte gibt.

Die mit der Nutzung der BVT im Zusammenhang stehenden Emissions- und Verbrauchswerte müssen stets zusammen mit den vorgegebenen Referenzbedingungen (z.B. durchschnittliche Zeiträume) betrachtet werden.

Wo verfügbar, werden die Daten bzgl. der Kosten zusammen mit einer Beschreibung der im vorherigen Kapitel dargestellten Techniken vorgelegt. Sie erlauben eine ungefähre Vorstellung von der Höhe der anfallenden Kosten. Dennoch hängen die tatsächlichen Kosten für die Anwendung einer bestimmten Technik in starkem Maße von der speziellen Situation ab, was zum Beispiel Steuern, Honorare und technische Eigenschaften des betreffenden Standortes angeht. Es ist nicht möglich, in diesem Dokument solche standortspezifischen Faktoren in vollem Umfange zu bewerten. Liegen keine Angaben bzgl. der Kosten vor, werden die Schlussfolgerungen zur Wirtschaftlichkeit bestimmter Techniken aus Beobachtungen an vorhandenen Standorten gezogen.

Es ist beabsichtigt, dass die allgemeinen BVT in diesem Kapitel eine Bezugsgröße bilden, an der die gegenwärtige Leistung einer vorhandenen Anlage oder einer vorgeschlagenen neuen Anlage gemessen wird. Auf diese Weise helfen die BVT bei der Bestimmung entsprechender

Bedingungen auf Grundlage der BVT für die jeweilige Anlage. Es lässt sich absehen, dass die neuen Anlagen so ausgelegt werden können, dass ihre Leistungsfähigkeit mindestens genauso gut wie oder besser als die der hier allgemein vorgestellten BVT sein wird. Darüber hinaus lässt sich denken, dass die vorhandenen Anlagen auf das gleiche oder sogar ein besseres Leistungsniveau wie das der BVT gebracht werden, wenn sich die Techniken im betreffenden Falle als technisch und wirtschaftlich umsetzbar erweisen.

Auf jeden Fall gibt es auf Grund der verschiedenen Arten der Vererzung, der verfügbaren Abbaumethoden und Aufbereitungstechniken sowie der unterschiedlichen geologischen, geotechnischen, hydrologischen und morphologischen Bedingungen, die von Fall zu Fall und von Standort zu Standort anders sind, einen Bedarf für standortspezifische Lösungen hinsichtlich der Entwurfs-, Konstruktions-, Betriebs-, Stilllegungs- und Nachsorgephase sowie für die ständige Kontrolle und Überwachung der Bewirtschaftung von Aufbereitungsrückständen und taubem Gestein.

Wenngleich auch dieses Dokument keine rechtlich verbindlichen Normen setzt, ist es doch dazu gedacht, Informationen über erreichbare Leistungs-, Emissions- und Verbrauchswerte als Anleitung für die Branche, die Mitgliedsstaaten und die Öffentlichkeit zur Verfügung zu stellen, falls die vorgegebenen Techniken eingesetzt werden.

Für die Bewirtschaftung von Aufbereitungsrückständen und taubem Gestein beruhen die Entscheidungen für eine bestimmte BVT auf

- ihrer Leistungsfähigkeit unter Umweltaspekten,
- dem damit einhergehenden Risiko und
- der Wirtschaftlichkeit.

Insbesondere die Berücksichtigung des Risikos ist ein ausgesprochen standortspezifischer Faktor.

5.2 Allgemeines

Mit den BVT

- sollen die in Abschnitt **4.1** aufgeführten allgemeinen Grundsätze angewandt und
- soll die in Abschnitt **4.2** beschriebene Herangehensweise einer Bewirtschaftung über den gesamten Lebenszyklus hinweg („*life-cycle management*“) umgesetzt werden.

Das Life-Cycle-Management erstreckt sich über alle Phasen der Existenz eines Standorts, einschließlich

- der Entwurfsphase (Abschnitt **4.2.1**):
  - ökologische Vergleichsbasis (Abschnitt **4.2.1.1**),
  - Beschreibung der Aufbereitungsrückstände und des tauben Gesteins (Abschnitt **4.2.1.2**),
  - Studien und Pläne zu TMF (Abschnitt **4.2.1.3**), die folgende Aspekte abdecken:
    - Dokumentation zur Auswahl der Standorte,
    - Umweltverträglichkeitsprüfung
    - Risikoeinschätzung
    - Einsatzbereitschaftsplan für Katastrophenfälle
    - Ablagerungsplan
    - Wasserbilanz und Wasserhaushaltsplan sowie
    - Außerbetriebnahme- und Stilllegungsplan
  - Auslegung der TMF und der damit verbundenen Bauten (Abschnitt **4.2.1.4**),
  - Kontrolle und Überwachung (Abschnitt **4.2.1.5**),
- der Bauphase (Abschnitt **4.2.2**),
- der Betriebsphase (Abschnitt **4.2.3**), mit den Elementen:
  - OSM-Handbücher (Abschnitt **4.2.3.1**),
  - Revision (Abschnitt **4.2.3.2**),
- der Stilllegungs- und der Nachsorgephase (Abschnitt **4.2.4**) mit den Elementen:
  - langfristige Ziele der Stilllegung (Abschnitt **4.2.4.1**),

- spezielle Fragen der Stilllegung (Abschnitt **4.2.4.2**) von
  - Halden (Aufschüttungen),
  - Becken, einschließlich:
    - wasserbedeckte Becken,
    - entwässerte Becken,
    - wasserwirtschaftliche Anlagen.

Des Weiteren soll mit den BVT

- der Reagenzienverbrauch verringert (Abschnitt **4.3.2**),
- die Wassererosion verhindert (Abschnitt **4.3.3**),
- die Staubbildung verhindert (Abschnitt **4.3.4**),
- eine Wasserbilanz aufgestellt (Abschnitt **4.3.7**) und die daraus gewonnen Ergebnisse für die Erarbeitung eines Wasserbilanzplanes genutzt (Abschnitt **4.2.1.3**),
- die Klarwasserbewirtschaftung angewandt (Abschnitt **4.3.9**) und
- das Grundwasser um alle Bereiche mit Aufbereitungsrückständen und taubem Gestein herum überwacht werden (Abschnitt **4.3.12**).

ARD-Management

Zur Charakterisierung der Aufbereitungsrückstände und des tauben Gesteins (Abschnitt **4.2.1.2** zusammen mit Anhang 4) gehört die Bestimmung des Säurebildungspotenzials von Aufbereitungsrückständen und/oder taubem Gestein. Wenn ein Säurebildungspotenzial vorhanden ist, sollen die BVT als erstes die Bildung von Sauerwässern (ARD) verhindern (Abschnitt **4.3.1.2**). Falls das nicht verhindert werden kann, sollen die BVT die Auswirkungen der ARD kontrollieren (Abschnitt **4.3.1.3**) oder die Anwendung von Behandlungsoptionen ermöglichen (Abschnitt **4.3.1.4**). Oft wird eine Kombination aus allen Möglichkeiten verwendet (Abschnitt **4.3.1.6**).

Alle Optionen zur Verhinderung der Entstehung, zur Kontrolle und zur Behandlung von Sauerwässern können in vorhandenen oder neuen Anlagen zur Anwendung kommen. Dennoch werden die besten Ergebnisse bei der Stilllegung dann erreicht, wenn die Pläne für die Stilllegung gleich zu Beginn (d.h. im Entwurfsstadium) für den Betrieb ausgearbeitet werden („Von-der-Wiege-bis-zur-Bahre“-Philosophie).

Die Anwendbarkeit der Optionen hängt hauptsächlich von den am Standort vorherrschenden Bedingungen ab. Faktoren, wie

- die Wasserbilanz,
- die Verfügbarkeit möglicher Abdeckmaterialien und
- der Grundwasserpegel

beeinflussen die am jeweiligen Standort anwendbaren Optionen. Abschnitt **4.3.1.5** liefert Entscheidungshilfen für die am besten geeigneten Stilllegungsoptionen.

Sickerwasser-Management (Abschnitt 4.3.10)

Der Standort für eine Anlage zur Bewirtschaftung von Aufbereitungsrückständen oder taubem Gestein wird vorzugsweise so ausgewählt, dass eine Abdichtung (Auskleidung mit einer Dichtungsbahn) nicht notwendig wird. Wenn das aber nicht möglich ist und sich die Qualität des Sickerwassers als nachteilig sowie dessen Strömungs- bzw. Fließgeschwindigkeit als (zu) hoch erweist, muss das (Einlaufen von) Sickerwasser verhindert, verringert (Abschnitt **4.3.10.1**) oder kontrolliert (Abschnitt **4.3.10.2**) werden (und zwar möglichst in dieser Reihenfolge). Oft findet eine Kombination aus diesen Maßnahmen Anwendung.

Emissionen in das Wasser

Mit den BVT soll(en)

- das Prozesswasser wieder verwendet (siehe Abschnitt **4.3.11.1**),
- das Prozesswasser mit anderen ausströmenden, gelöste Metalle enthaltenden Medien vermischt (siehe Abschnitt **4.3.11.3**),

- Sedimentationsbecken zum Abfangen erodierter Feinstoffe gebaut (siehe Abschnitt **4.3.11.4.1**),
- Schwebstoffe und gelöste Metalle vor dem Austrag des gereinigten Abwassers in den Vorfluters entfernt (Abschnitt **4.3.11.4**),
- alkalische Abwässer mit Schwefelsäure oder Kohlendioxid neutralisiert (Abschnitt **4.3.11.6**) und
- Arsen aus den aus den Gruben ausfließenden Abwässern durch die Zugabe von Eisen(III)-Salz entfernt werden (Abschnitt **4.3.11.7**).

Die jeweiligen Abschnitte über Emissions- und Verbrauchswerte in Kapitel 3 enthalten Beispiele über die erreichten Werte, aber es sollte kein Zusammenhang zwischen den angewandten Techniken und den verfügbaren Emissionsdaten hergestellt werden. Deshalb war es in diesem Dokument nicht möglich, aus den mit den BVT auftretenden Emissionswerten Schlussfolgerungen über die BVT selbst zu ziehen.

Die folgenden Techniken sind BVT für die Behandlung säurehaltiger Abwässer (Abschnitt **4.3.11.5**):

- aktive Behandlungsverfahren:
  - Zugabe von Kalk (Calciumkarbonat), Löschkalk oder Ätzkalk,
  - Zugabe von Ätznatron zu ARD mit einem hohen Mangananteil
- passive Behandlungsverfahren:
  - künstliche Feuchtgebiete,
  - offene Kalksteinkanäle/anoxische Kalksteinkanalisation,
  - Ableitungsbrunnen.

Passive Behandlungssysteme stellen eine langfristige Lösung nach der Außerbetriebnahme eines Standortes dar, aber nur, wenn sie als Ergänzung zu anderen (präventiven) Maßnahmen eingesetzt werden.

Geräuschemissionen (Abschnitt 4.3.5)

Mit den BVT sollen:

- kontinuierlich arbeitende Systeme (z.B. Transportbänder, Pipelines) eingesetzt werden,
- eingehauste Bandantriebe in Gegenden eingesetzt werden, wo es örtlich zu Lärm kommt und
- zunächst die Außenböschungen einer Aufschüttung errichtet und dann die Rampen und Arbeitsstrossen soweit wie möglich in den Innenbereich der Halde verlegt werden.

Auslegung des Damms

Zusätzlich zu den in Abschnitt **4.1** und Abschnitt **4.2** beschriebenen Maßnahmen während der **Planungsphase** (Abschnitt **4.2.1**) für den **Damm eines Absetzbeckens** sollen die BVT

- die Dimension einer einmal in 100 Jahren vorkommenden Überschwemmung zur Grundlage für die Festlegung der Ablasskapazität von Dämmen mit schwachem gefahrenpotenzial in Notfällen und
- die Dimension einer einmal in 5 000 – 10 000 Jahren vorkommenden Überschwemmung zur Grundlage für die Festlegung der Ablasskapazität von Dämmen mit hohem gefahrenpotenzial in Notfällen nehmen.

Dammkonstruktion

Zusätzlich zu den in Abschnitt **4.1** und Abschnitt **4.2** beschriebenen Maßnahmen während der **Konstruktionsphase** (Abschnitt **4.2.2**) für den **Damm eines Absetzbeckens** sollen die BVT

- zur Beseitigung aller Vegetation und humushaltiger Erden aus dem natürlichen Untergrund unter dem Staudamm (Abschnitt **4.2.3**) und
- zur Auswahl eines Dammbaumaterials eingesetzt werden, das sich für den vorgesehenen Zweck eignet und unter den vorherrschenden betrieblichen und klimatischen Bedingungen nicht nachgibt (Abschnitt **4.4.4**)

Dammschüttung

Zusätzlich zu den in Abschnitt **4.1** und Abschnitt **4.2** beschriebenen Maßnahmen während der **Konstruktions- und Betriebsphase** (Abschnitte **4.2.2** und **4.2.3**) für den **Damm eines Absetzbeckens** sollen die BVT genutzt werden, um

- die Risiken eines zu hohen Porendrucks einschätzen sowie den Porendruck vor und während jeder Schüttung zu überwachen; die Einschätzung sollte durch einen unabhängigen Fachmann erfolgen;
- unter folgenden Bedingungen herkömmliche Arten von Dämmen zu verwenden (Abschnitt 4.4.6.1), wenn nämlich
  - die Aufbereitungsrückstände für die Dammkonstruktion nicht geeignet sind,
  - das zu speichernde Wasser angestaut werden muss,
  - sich der Standort für die Bewirtschaftung der Aufbereitungsrückstände an einem weit entfernten und schwer zugänglichen Ort befindet,
  - das Wasser mit den Aufbereitungsrückständen wegen des Abbaus eines toxischen Elements (z.B. Cyanid) über einen längeren Zeitraum zurückgehalten werden muss und
  - der natürlich Zufluss in den Staubereich sehr groß ist oder starken Schwankungen unterliegt, so dass das Wasser zu Kontrollzwecken gespeichert werden muss;
- unter folgenden Bedingungen auf das Upstream-Dammbauverfahren zurückzugreifen, d.h. die Dämme werden nach innen aufgeschüttet, so dass der Damm ins Becken „hinein“ wandert (Abschnitt **4.4.6.2**), wenn nämlich
  - das seismische Risiko gering ist und
  - Aufbereitungsrückstände für den Bau des Dammes verwendet werden, die zu mindestens 40–60 % aus Material mit einer Korngröße zwischen 0,075 und 4 mm in ganzen Aufbereitungsrückständen bestehen (was aber nicht auf verdickte bzw. entwässerte Aufbereitungsrückstände zutrifft);
- unter folgenden Bedingungen auf das Downstream-Dammbauverfahren zurückzugreifen, die Dämme werden nach außen aufgeschüttet, so dass der Damm vom Becken „weg“ wandert (Abschnitt 4.4.6.3),
  - ausreichende Mengen an Dammbaumaterialien zur Verfügung stehen (z.B. Aufbereitungsrückstände oder taubes Gestein) und
- unter folgenden Bedingungen auf das Centerline-Dammbauverfahren zurückzugreifen, so dass der Damm mittig aufgeschüttet und höher wird (Abschnitt 4.4.6.4), wenn nämlich
  - das seismische Risiko gering ist.

Dammbetrieb

Zusätzlich zu den in Abschnitt **4.1** und Abschnitt **4.2** beschriebenen Maßnahmen während der **Betriebsphase** (Abschnitt **4.2.3**) eines **Absetzbeckens** sollen die BVT genutzt werden, um

- die Stabilität, wie nachfolgend näher spezifiziert, zu überwachen,
- bei Schwierigkeiten den Austrag in ein anderes Becken umzuleiten,
- alternative Anlagen für den Austrag bereitzustellen, möglicherweise in einer anderen Stauhaltung,
- eine zweite Dekantieranlage (z.B. ein Notüberlauf, Abschnitt **4.4.9**) und/oder Standby-Pumppontons für Notfälle verfügbar zu machen, wenn der Klarwasserpegel im Becken das vorbestimmte Mindestfreibord erreicht (Abschnitt **4.4.8**),
- die Bodenbewegungen mit Tiefenneigungsmessern (Tiefeninklinometer) zu messen und sich Kenntnis über die Porendruckbedingungen zu verschaffen,
- für eine ausreichende Drainage zu sorgen (Abschnitt **4.4.10**),
- die Entwurfs- und Konstruktionsunterlagen zu führen sowie alle Aktualisierungen/Änderungen an den Entwürfen und der Konstruktion vorzunehmen,
- wie in Abschnitt **4.2.3.1** beschrieben ein Dammsicherheitshandbuch zu führen, zusammen mit den in Abschnitt **4.2.3.2** erwähnten unabhängigen Überprüfungen und
- das Personal entsprechend zu schulen.

Ableitung des Klarwassers aus dem Becken (Abschnitt 4.4.7.1)

Die BVT

- nutzen einen Überlauf auf natürlichem Boden aus innerhalb und außerhalb von Tälern gelegenen Becken
- nutzen einen Mönch
  - in kaltem Klima mit positiver Wasserbilanz und
  - für weidekoppelartige Becken
- nutzen einen Dekantierbrunnen
  - in warmem Klima mit negativer Wasserbilanz,
  - für weidekoppelartige Becken und
  - wenn beim Betreiben des Beckens ein großer Freibord gehalten wird.

Entwässerung der Aufbereitungsrückstände (Abschnitt 4.4.16)

Die Wahl des Verfahrens (zur Gewinnung aufgeschlämmter, eingedickter oder trockener Aufbereitungsrückstände) hängt hauptsächlich von der Bewertung dreier Faktoren ab, nämlich

- den Kosten,
- der Umweltbilanz und
- der Gefahr einer Betriebsstörung.

Bei der Bewirtschaftung von Aufbereitungsrückständen nutzen die BVT

- die Bewirtschaftung trockener Aufbereitungsrückstände (Abschnitt 4.4.16.1),
- die Bewirtschaftung eingedickter Aufbereitungsrückstände (Abschnitt 4.4.16.2) oder
- die Bewirtschaftung aufgeschlämmter Aufbereitungsrückstände (Abschnitt 4.4.16.3).

Es gibt viele andere Faktoren, die die Wahl der entsprechenden Techniken an einem gegebenen Standort beeinflussen, zum Beispiel:

- die Mineralogie des Erzes,
- der Wert des Erzes,
- die Korngrößenverteilung,
- die Verfügbarkeit von Prozesswasser,
- die klimatischen Bedingungen und
- der für die Bewirtschaftung der Aufbereitungsrückstände verfügbare Platz.

Betrieb von Anlagen zur Bewirtschaftung von Aufbereitungsrückständen und taubem Gestein

Zusätzlich zu den in Abschnitt **4.1** und Abschnitt **4.2** beschriebenen Maßnahmen während der **Betriebsphase** (Abschnitt **4.2.3**) von **Anlagen zur Bewirtschaftung von Aufbereitungsrückständen und taubem Gestein** sollen die BVT

- den natürlichen externen Oberflächenabfluss ableiten (Abschnitt **4.4.1**),
- Aufbereitungsrückstände oder taubes Gestein in Gruben bewirtschaften (Abschnitt **4.4.1**), wobei die Stabilität der Halde bzw. des Dammes in diesem Falle irrelevant ist,
- beim Betreiben aller Halden und Dämme einen Sicherheitsfaktor von mindestens 1,3 anwenden (Abschnitt **4.4.13.1**) und
- eine progressive Sanierung/Wiederbegrünung vornehmen (Abschnitt **4.3.6**).

Überwachen der Stabilität

Mit den BVT sollen

- auf einer Halde/einem Damm für Aufbereitungsrückstände (Abschnitt 4.4.14.2)
  - der Wasserstand,
  - die Qualität und Quantität des Sickerwasserflusses durch den Damm hindurch (auch Abschnitt 4.4.12),
  - die Position des Grundwasserspiegels,
  - der Porendruck,
  - die Bewegung der Dammkrone und der Aufbereitungsrückstände,
  - zur Absicherung der Stabilität des Dammes und der tragenden Schichten die Seismik (auch Abschnitt 4.4.14.4)

  - der dynamischen Porendruck und der Verflüssigungsprozess,
  - die Bodenmechanik sowie
  - die Vorgehensweise beim Einbau der Aufbereitungsrückstände überwacht werden,
- auf einer Halde (Abschnitt 4.4.14.2):
  - die Strossen- bzw. Böschungsgeometrie,
  - die Drainage unter der Kippe und
  - der Porendruck überwacht werden sowie
- auch
  - im Falle eines Beckens/eines Dammes für Aufbereitungsrückstände
    - Sichtprüfungen (Inaugenscheinnahme) (Abschnitt 4.4.14.3),
    - jährliche Überprüfungen (Abschnitt 4.4.14.3),
    - unabhängige Prüfungen (Audits) (Abschnitt 4.2.3.2 und Abschnitt 4.4.14.3) und
    - Sicherheitsbewertungen an vorhandenen Dämmen (SEED) (Abschnitt 4.4.14.3) sowie
  - im Falle einer Halde
    - Sichtprüfungen (Abschnitt 4.4.14.3),
    - geotechnische Überprüfungen (Abschnitt 4.4.14.3) und
    - unabhängige geotechnische Audits (Abschnitt 4.4.14.3) vorgenommen werden.

Verminderung von Havarieauswirkungen

Mit den BVT soll(en)

- eine Notfallplanung vorgenommen (Abschnitt 4.6.1),
- Vorfälle eingeschätzt und nachverfolgt (Abschnitt 4.6.2) und
- die Pipelines überwacht werden (Abschnitt 4.6.3).

Verringerung des Platzbedarfs/der Aufstellfläche

Mit den BVT soll(en)

- die Erzeugung von Aufbereitungsrückständen/taubem Gestein verhindert und/oder verringert werden, wenn das möglich ist (Abschnitt **4.1**)
- die Aufbereitungsrückstände unter den folgenden Bedingungen wieder verfüllt werden (Abschnitt 4.5.1), wenn nämlich
  - der Versatz als Teil des Abbauverfahrens benötigt wird (Abschnitt 4.5.1.1),
  - die zusätzlichen Kosten für die Verfüllung mindestens durch die höhere Erzausbeute ausgeglichen werden können,
  - sich beim Abbau im Tagebau die Aufbereitungsrückstände leichter entwässern lassen (d.h. durch Verdunstung, Dränung und Filtration) und dadurch ganz auf eine TMF verzichtet oder sie in der Größe verringert werden kann (Abschnitte 4.5.1.2, 4.5.1.3, 4.5.1.4, 4.4.1),
  - nahe gelegene ausgekohlte Tagebaue für die Versatzmaterialien zur Verfügung stehen (Abschnitt 4.5.1.5),
  - in Untertagebergwerken große Abbaukammern für die Versatzmaterialien zur Verfügung stehen (Abschnitt 4.5.1.6), wobei mit verschlämmten Aufbereitungsrückständen verfüllte Kammern eine Drainage erfordern (Abschnitt 4.5.1.9) und Bindemittel zur Erhöhung der Stabilität hinzugefügt werden können (Abschnitt 4.5.1.8),
- die Aufbereitungsrückstände in der Form von pastösen Versatzmaterialien wieder verfüllt werden (Abschnitt 4.5.1.10), wenn die zur Wiederverfüllung erforderlichen Bedingungen gegeben sind und wenn
  - die Notwendigkeit zum kompetenten Versatz besteht,
  - die Aufbereitungsrückstände sehr fein(körnig) sind, so dass für den hydraulischen Versatz nicht genügend Material vorhandenen wären und die großen, in ein Becken entsorgten Mengen zu lange brauchten, ehe sie entwässert wären und
  - es wünschenswert wäre, das Wasser von der Grube fernzuhalten bzw. wenn es zu kostenaufwendig wäre, das aus den Aufbereitungsrückständen austretende Wasser abzupumpen (d.h. über eine lange Strecke),
- das taube Gestein unter folgenden Bedingungen wieder verfüllt werden (Abschnitt 4.5.2), nämlich wenn

  - es innerhalb einer Untertagegrube verfüllt werden kann,
  - es in der Nähe eine oder mehrere ausgekohlte Gruben gibt (was manchmal als '*transfer mining*' bezeichnet wird)
  - der Tagebaubetrieb in einer solchen Weise erfolgt, dass das taube Gestein ohne Beeinträchtigung des Abbaubetriebes versetzt werden kann,
- die möglichen Verwendungszwecke für Aufbereitungsrückstände und taubes Gestein untersucht werden (Abschnitt 4.5.3).

<u>**Stilllegung und Nachsorge**</u>
Zusätzlich zu den in Abschnitt **4.1** und Abschnitt **4.2** beschriebenen Maßnahmen während **Stilllegungs- und Nachsorgephase** (Abschnitt 4.2.4) von **Anlagen zur Bewirtschaftung von Aufbereitungsrückstände und taubem Gestein** soll(en) mit den BVT

- schon während der Planungsphase Stilllegungs- und Nachsorgepläne, einschließlich Kostenvoranschläge, aufgestellt bzw. erarbeitet werden, die dann im Laufe der Zeit zu aktualisieren sind (Abschnitt 4.2.4); dennoch ergeben sich Anforderungen an die Sanierung einer TMF über deren gesamten Betriebszeitraum hinweg und können erst während der Stilllegungsphase präzisiert werden,
- bei Dämmen und Halden nach der Stilllegung ein Sicherheitsfaktor von mindestens 1,3 angewandt werden (Abschnitte 4.2.4 und 4.4.13.1), wobei die Meinungen zu Abdeckungen mit Wasser geteilt sind (siehe Kapitel **7**).

Für die Stilllegungs- und Nachsorgephase von Becken für Aufbereitungsrückstände sollen mit den BVT Dämme errichtet werden, die langfristig stabil bleiben, falls für the Stilllegung die Lösung einer Abdeckung mit Wasser gewählt wird (Abschnitt **4.2.4.2**).

5.3 Goldlaugung mit Cyanid

Zusätzlich zu den in Abschnitt 5.2 aufgeführten allgemeingültigen Maßnahmen für alle Standorte, an denen die Goldlaugung mit Cyanid erfolgt, sollen die BVT folgendes erreichen:

- Verringerung des Einsatzes von CN durch
  - Betriebskonzepte, mit denen die Zugabe von Cyanid minimiert wird (Abschnitt **4.3.2.2**),
  - automatische Cyanidkontrolle (Abschnitt **4.3.2.2.1**) und,
  - Peroxid-Vorbehandlung, wenn anwendbar (Abschnitt **4.3.2.2.2**).
- Zersetzung des verbleibenden freien CN vor dem Austrag in das Becken (Abschnitt **4.3.11.8**). **Tabelle 4.13** zeigt Beispiele für an einigen europäischen Standorten erreichte CN-Werte.
- Anwendung folgender Sicherheitsmaßnahmen (Abschnitt 4.4.15):
  - Dimensionierung des Kreislaufs für die Zersetzung des Cyanids mit einer Kapazität, die dem zweifachen der tatsächlichen Anforderungen entspricht,
  - Einbau eines Backup-Systems für die Zugabe von Kalk und
  - Installation eines Reserve-Stromgenerators.

5.4 Aluminium

Zusätzlich zu den in Abschnitt 5.2 aufgeführten allgemeinen Maßnahmen für alle Tonerderaffinerien sollen die BVT folgendes erreichen:

- während des Betriebs:
  - Vermeiden des Austrags von Abwasser in das Oberflächenwasser. Das wird durch die Wiederverwendung von Prozesswasser in der Raffinerie erreicht (Abschnitt **4.3.11.1**) bzw. unter trockenen klimatischen Bedingungen durch Verdunstung
- in der Nachsorgephase (Abschnitt **4.3.13.1**):
  - Behandlung des Oberflächenabflusses aus der TMF vor dem Austrag, bis die chemischen Bedingungen eine akzeptable Konzentration für den Austrag in das Oberflächenwasser erreicht haben,

- Instandhaltung der Zufahrtsstraßen, Drainagesysteme und der abdeckenden Vegetationsschicht (einschließlich einer Wiederbegrünung, wenn notwendig),
- laufende Probenahmen zur Feststellung der Grundwasserqualität.

5.5 Kali

Zusätzlich zu den in Abschnitt 5.2 aufgeführten allgemeinen Maßnahmen für alle Kali-Standorte sollen die BVT folgendes erreichen:

- Abdichten des Bodens unter der TMF, wenn der natüliche Boden nicht undurchlässig ist (Abschnitt **4.3.10.3**).
- Verringerung der Staubemissionen von den Bandtransporten (Abschnitt **4.3.4.3.1**).
- Versiegelung/Auskleidung des Böschungsfußes außerhalb der undurchlässigen Kernzone und Auffangen des Oberflächenabflusses (Abschnitt **4.3.11.4.1**).
- Verfüllen großer Abbaukammern mit trockenen und/oder verschlämmten Aufbereitungsrückständen (Abschnitt 4.5.1.6).

5.6 Kohle

Zusätzlich zu den in Abschnitt 5.2 aufgeführten allgemeinen Maßnahmen für alle Kohle-Standorte sollen die BVT folgendes erreichen:

- Verhinderung von Sickerwasser (Abschnitt **4.3.10.4**)
- Entwässerung feiner Aufbereitungsrückstände (< 0,5 mm) durch Flotation (Abschnitt 4.4.16.3)

5.7 Umweltmanagement

Eine Reihe von Umweltmanagementtechniken ist als BVT eingeordnet. Der Anwendungsbereich (z. B. die Detailebene) und die Art des Umweltmanagementsystems (z. B. standardisiert oder nicht standardisiert) wird generell auf die Art, die Größe, die Komplexität der Anlage sowie die Reichweite der möglichen Umweltauswirkungen bezogen.

Mit den BVT soll ein Umweltmanagementsystem (EMS) umgesetzt und zugleich eingehalten werden, das je nach den individuellen Umständen die folgenden Inhalte hat (siehe Kapitel **4**):

- Definition einer Umweltpolitik für die betreffende Anlage durch das Spitzenmanagement (das Engagement des Spitzenmanagements wird als Voraussetzung für die erfolgreiche Anwendung der anderen Merkmale und Eigenschaften des EMS angesehen),
- Planung und Festlegung der notwendigen Vorgehensweisen,
- Umsetzung der festgelegten Vorgehensweisen, wobei folgenden Punkten besondere Aufmerksamkeit zu schenken ist:
  - Organisationsstruktur und Verantwortung,
  - Schulung, Bewusstmachung und Kompetenz
  - Kommunikation
  - Einbeziehung der Mitarbeiter
  - Dokumentation
  - Wirksame Prozesskontrolle
  - Instandhaltungs- und Wartungsprogramm
  - Vorbereitung und Reaktion auf Notfälle
  - Sicherstellung der Einhaltung der Umweltbestimmungen
- Kontrolle der Leistungsfähigkeit des Systems und Ergreifung von Abhilfemaßnahmen, wobei folgenden Punkten besondere Aufmerksamkeit zu schenken ist:
  - Überwachung und Messungen (siehe dazu auch das Merkblatt zur Emissionsüberwachung),
  - Abhilfe- und vorbeugende Maßnahmen,

    - Führung der Aufzeichnungen und Unterlagen,
    - unabhängige (wo praktikabel) interne Revision (Audit) um festzustellen, ob das Umweltmanagementsystem mit den geplanten Arrangements übereinstimmt und es ordnungsgemäß umgesetzt und aufrechterhalten wird oder nicht sowie
- Überprüfung durch das Spitzenmanagement.

Drei weitere Inhalte eines EMS, die die vorgenannten Schritt für Schritt ergänzen können, werden als zusätzliche Maßnahmen zur Unterstützung der erstgenannten angesehen. Ihr Fehlen wird jedoch nicht als Widerspruch zu den BVT angesehen. Diese drei zusätzlichen Schritte sind:

- Überprüfung und Validierung des Managementsystems und der Vorgehensweise mittels Audits durch ein akkreditiertes Zertifizierungsorgan oder einen externen EMS-Prüfer;
- Ausarbeitung und Veröffentlichung (sowie möglicherweise externe Validierung) einer regelmäßigen Umwelterklärung mit einer Beschreibung aller wesentlichen Umweltaspekte der Anlage, die einen jährlichen Vergleich mit den aufgestellten Umweltzielen und, wie angebracht, mit den Eckwerten der Branche gestattet;
- Umsetzung und Einhaltung eines international anerkannten freiwilligen Systems, wie EMAS und EN ISO 14001:1996. Dieser freiwillige Schritt könnte dem EMS eine größere Glaubwürdigkeit verleihen, was insbesondere auf EMAS zutrifft, das alle o.a. inhaltlichen Merkmale und Eigenschaften in sich vereint. Trotzdem können prinzipiell auch nichtstandardisierte Systeme gleichermaßen wirksam sein, vorausgesetzt, sie sind ordnungsgemäß entworfen und werden ebenso gründlich umgesetzt.

Insbesondere für die Bewirtschaftung von Aufbereitungsrückständen sowie taubem Gestein soll mit den besten verfügbaren Techniken (BVT) ein integriertes Risiko-/Sicherheits- und Umweltmanagementsystem verwirklicht werden. Das Umweltmanagement muss daher gemeinsam mit der in Abschnitt 4.2.3.1 beschriebenen Risikobewertung/dem dort beschrieben Risikomanagement sowie mit dem in Abschnitt **4.2.3.1** beschriebenen Betriebs-, Überwachungs- und Wartungsmanagement entwickelt und durchgeführt werden.

6 EMERGING TECHNIQUES FOR THE MANAGEMENT OF TAILINGS AND WASTE-ROCK IN MINING ACTIVITIES

6.1 Co-disposal of iron ore tailings and waste-rock

The operator of the Swedish iron ore operations at Kiruna and Malmberget has, for several years, worked on the development of alternative methods of transporting and depositing their so-called 'waste-rock' (dry coarse tailings <100 mm) and tailings from concentrating operations (fine tailings <3 mm). The objectives of this research have primarily been to bring down the significant investment and operation costs of trucking (currently used for waste-rock) and of dam constructions (currently used for the fine tailings).

A major test has been conducted, where a mixture of dry tailings and wet tailings was pumped with heavy duty slurry pumps. The tests and site-specific evaluation showed that the operation was not competitive with traditional transportation techniques, mainly due to wear in pumps and pipelines. The resulting co-disposal, however, showed that the slurry stream formed a rounded moraine-like formation, similar to those created by melting ice during the withdrawal of the glacial ice. The density of the deposited material was found to be higher than that of conventionally placed material, i. e. the use of available volume is more efficient. In addition, it was concluded that if measures are taken in order to control the groundwater level in the deposit, stable and high deposits may be created with this disposal method.

The promising properties of the co-deposited waste-rock and tailings have encouraged research into ways to achieve the advantages of co-disposal combined with traditional transportation techniques. One operator has developed the concept of drained-cell disposal and has carried out laboratory, pilot-scale and full-scale tests to develop applicable design criteria, to evaluate the operational, hydraulic and geotechnical aspects and to investigate the influence of cold climate on the stability of the deposit.

The drained-cell disposal is now evaluated in pre-studies in the Malmberget and in Kiruna mine sites.

In Italy, the technique to build up tips in layers of different permeabilities was successfully used for the disposal of the overburden material in the S.Barbara (Arezzo - Tuscany) coal mine facilities. The high permeability layers can drain water off, so that the time required for pore pressure dissipation in the low permeability layers is dramatically reduced. This technique improved the short-tem stability of the tips, providing sufficient safety for an acceptable building velocity.

6.2 Inhibiting progress of ARD

Artificial coatings have been found to form an impermeable and protective coating on sulphide surfaces, inhibiting the progress of acid rock drainage (ARD). This research programme intends to review the feasibility of the process of forming oxidation-protecting coatings on sulphides using reagents or electrolytic processes. This protective layer must be consistent in order to minimise access of one of the ingredients that generate ARD. The focus of the research will be on obtaining coating layers that resist the ageing process. Electrolytic studies will also be conducted in order to oxidise the surface of exposed sulphides in waste-rocks or tailing dams thereby creating a passivation layer.
(from http://www.mining.ubc.ca - Study of formation of passivation coatings on sulphide-rich waste-rock: a way to hinder ARD propagation).

6.3 Recycling of cyanide using membrane technology

The recycling of cyanide using membrane technology, which is currently under development, is planned to be applied to the gold metallurgical extraction process, where the efficacy of cyanide use is hindered due to the presence of copper (and similar metals such as zinc and silver). The presence of these metals causes an increase in the consumption of cyanide, a lowering of the gold recovery efficiency and also poses a heightened environmental management issue for the tailings.

The technique is an hybrid of the membrane and electrowinning technologies, which allow for the recovery of metallic copper and the simultaneous liberation of free cyanide from the copper-cyanide complexes. The free cyanide may then be recovered and returned to the front end of the milling process with beneficial savings. The process may be installed in the tailings circuit prior to discharge to the tailings pond or in the returned water circuit recovered from the tailings dam.

The component technologies of the process are well tried and tested in industry. Initial cost estimates show that the process is potentially very attractive compared to alternative approaches such as resin exchanges processes, precipitation and acidification processes.

The basic flow diagram for this process consists of three parts:

1. a solids removal step to provide a clean liquor for subsequent processing
2. a membrane step that concentrates the copper-cyanide complexes. This step also recovers a portion of the free cyanide
3. a metal recovery unit (MRU) that deposits the copper electrolytically, thereby liberating a portion of the WAD-cyanide as free cyanide.

This technique is planned to be applied to any process stream that contains free cyanide and/or cyanide complexed with copper or similar (weak acid dissociable, or WAD-cyanide). This may be either the tailings stream prior to the tailings dam or in the recovered water from the tailings dam.

This technique for recovering cyanide from gold tailings can be easily retrofitted to existing gold plants. The feed for the process is either the tailings liquor or the tailings return. This process provides a number of process benefits. Consumption of reagents is low compared to processes for the destruction of cyanide. Cyanide that would otherwise be lost to the circuit is able to be recovered from the tailings and re-used, reducing the cyanide inventory on site, and also the costs of purchasing the cyanide and the cyanide destruction. Copper metal is recovered as a by-product.

There are no limitations on the treatable cyanide WAD concentrations, although the efficiency of the process is dependent on the chemistry of the tailings stream.

There are also obvious environmental benefits. The amount of cyanide and copper in the tailings stream is reduced significantly prior to cyanide destruction or disposal of the waste to tailings storage facilities. This results in reduced environmental risk to wildlife and waterways. Recovery of cyanide reduces the amount of make-up cyanide purchased, stored and handled on site.

6.4 Lined cells

At the **Las Cruces Project,** the proposed deposition method of the tailings is dry deposition in impermeable cells. It is proposed that the cells are constructed as blocks of 100 x 100 m with a height of 25 m. The deposited tailings are proposed to be continuously covered by clay. The final encapsulation will be done using a multi-layer cover, utilising the clay (marl) extracted in the uncovering of the orebody. The cells are proposed to be constructed with an impermeable base, constructed by various layers of clay, maybe supported by a synthetic liner and drainage

layers. A system for the capturing of drainage will be installed and the drainage treated for re-use in the process or discharge.
[67, IGME, 2002]

6.5 Utilisation of treated red mud to remediate problems of ARD and metals pollution

A technology has been developed whereby the process residue (i.e. the'red mud') from alumina refineries is subjected to a proprietary treatment forming a material which posesses residual alkalinity and a capability to trap metal ions. This capacity is based on the complex quantity of minerals present in the material (such as haematite, boehmite, gibbsite, sodalite, quartz, cancrinite, and many others). This material can be used to treat acid and metals contamination of soils and water.

The material has been used at commercial scale for treating 1.5 Mm^3 of tailings water at the Mt. Carrington mine, Australia. The water had a significant metal load (Al, As, Cd, Cu, Fe, Pb, Mn, Ni, Zn) and after the treatment the metal removal from the water was found to vary from a minimum of 90 to 99.9 %. It has been also found that the bound metals can neither be taken up by vegetation nor be released by leaching.

The possible applications identified so far, are the following:

- neutralisation and decontamination of ARD water
- neutralisation of tailings and waste-rock with an ARD potential
- control of acid or metal-rich spills by means of filtering barriers
- removal of phosphate from water
- removal of arsenic and other metals from groundwater.

The material can be used either as a dry powder, or as slurry. The fine particles can then be easily decanted in the bottom of a calm area, such as in a basin, and periodically be collected and taken away. The sediment has a high specific gravity and it can remain in the basin bottom for long periods of time, without creating any problems. The trapped metals are locked in a stable form as demonstrated by leaching tests. The exhausted material can then be disposed of in a landfill or can also be left in place, as a bottom layer in the basin, as a further protection of the groundwater against metals contamination. The material can also be used in a porous pelletised form and in a sand filter arrangement.

6.6 Combination of SO_2/air and hydrogen peroxide technique to destroy cyanide

One technique currently being developed which uses the synergies between the SO_2/air process (applicable to slurries) and the hydrogen peroxide technique (not applicable to slurries) for the destruction of cyanide. The benefit of this new technique is its flexibility to accommodate changes in feed chemistry. Depending on site-specific conditions, the process offers capital and operating savings over traditional SO_2/air plants.

7 ABSCHLIESSENDE BEMERKUNGEN

Zeitliche Einordnung der Arbeit

Die erste Plenartagung der Technischen Arbeitsgruppe (TWG) fand im Juni 2001 statt. Der erste Entwurf des Dokuments wurde der TWG dann im September 2002 zur Beratung zugesandt. Nachdem die Kommentare und Anmerkungen dazu ausgewertet und in das Dokument eingearbeitet worden waren, wurde ein 2. Entwurf einschließlich der Vorschläge für Schlussfolgerungen zu den BVT im Mai 2003 versandt. Die abschließende Plenartagung der TWG wurde für November 2003 anberaumt. Nach der Tagung gab es eine kurze Absprache zu den Kapiteln mit den Schlussbemerkungen und der Zusammenfassung, ehe die endgültige Version des Dokuments erstellt wurde.

Informationsquellen

Viele Unterlagen wurden seitens der Industrie und der Behörden als Grundlage für die in diesem Dokument zu verarbeitenden Informationen bereitgestellt. Bulletins von der 'Internationalen Kommission für Große Talsperren' (ICOLD) bzgl. der Bewirtschaftung von Aufbereitungsrückständen, die kanadische 'Richtlinie für Anlagen zur Bewirtschaftung von Aufbereitungsrückständen' sowie das französische und das finnische 'Merkblatt zur Dammsicherheit' können als Ecksteine für dieses Dokument zu den besten verfügbaren Techniken (BVT) betrachtet werden. Die Metall- und Mineralindustrie sowie der Kohlebergbau steuerten wertvolle Informationen zu speziellen Operationen und den angewandten Techniken bei. Diese Angaben wurden durch Informationen aus Irland, Schweden, Spanien, Portugal, Finnland, Griechenland, Italien, Österreich und Deutschland ergänzt. Befahrungen vor Ort fanden in Irland, Deutschland, Österreich, Spanien, Schweden, in der Türkei und in Polen statt. In die Konsultationsrunden zu jedem Entwurf flossen das spezielle Feedback von den Betreibern, Anmerkungen zur Anwendbarkeit und Umsetzbarkeit einiger Techniken sowie weitere Betriebs- und Kostendaten ein. Besondere Aufmerksamkeit wurde über das gesamte Projekt hinweg der Einbeziehung jener neuen Mitgliedsländer gewidmet, die aktiv Bergbau betreiben. Dadurch kam es zu einer aktiven Teilnahme Polens, der Tschechischen Republik und Ungarns am Projekt.

Zur Förderung des Meinungsaustauschs wurden in Schweden, Irland, Polen und Bulgarien Workshops organisiert. Des Weiteren fanden in Österreich und - wiederholt während des gesamten Projekts - in Brüssel Beratungen der Untergruppen statt. Alle diese Ereignisse produzierten zusätzliche Betriebsdaten und wertvolle technische Informationen.

Lücken und Schwachstellen

Während der Eröffnungsberatung war entschieden worden, auch Informationen zur Bewirtschaftung von Aufbereitungsrückständen und taubem Gestein aus den Ölschieferabbaubetrieben Estlands einzubeziehen. Leider wurden dazu keine relevanten Angaben zur Verfügung gestellt.

Zunächst konzentrierten sich die meisten Beiträge auf die Aufbereitung von Steinen und Erden und die allgemeine Bewirtschaftung von Aufbereitungsrückständen und taubem Gestein. Bitten um weitere detailliertere Angaben erbrachten zwar meist die gewünschten Einzelheiten bzgl. der eingesetzten Techniken, aber der Verzug bei der Einarbeitung der übermittelten Details führte zu Verzögerungen bei der Zusammenstellung und der Verteilung des ersten Entwurfs.

Menge und Qualität der Daten in diesem Dokument sind insofern nicht ganz ausgeglichen, als nur wenige Angaben zu den tatsächlichen Verbrauchs- und Emissionswerten der Anlagen für die Bewirtschaftung von Aufbereitungsrückständen und taubem Gestein aus der Verarbeitung von Industriemineralen zur Verfügung gestellt wurden.

Die Emissionsdaten aus der Metallindustrie beruhen auf dem betrieb einzelner Anlagen und Einrichtungen. Eine Korrelation zwischen den angewandten Techniken und den verfügbaren Emissionsdaten konnte nicht hergestellt werden. Deshalb konnten keine Schlussfolgerungen bzgl. der BVT im Zusammenhang mit Emissionswerten gezogen werden.

Die TWG stellte auch kaum Informationen zur Schadensminderung bei Havarien/Unfällen zur Verfügung.

Das erreichte Maß an Einvernehmlichkeit

Die Schlussfolgerungen aus der geleisteten Arbeit wurden auf der abschließenden Plenartagung im November 2003 mit einem hohen Maß an Einvernehmlichkeit abgestimmt.

Die zur Diskussion auf der Abschlusssitzung anstehenden Hauptfragen betrafen
- die Menge an erzeugten Aufbereitungsrückständen und taubem Gestein,
- die mit den BVT in Verbindung stehenden Emissionswerte bei der Abwasserbehandlung und beim Cyanidabbau,
- Verfahren zur Abdeckung von Absetzbecken nach deren Stilllegung,
- Sicherheitsfaktoren für Halden und Dämme,
- die Überwachung von Dämmen und Halden,
- Verfahren für die Entwässerung der abgesetzten Schlämme sowie
- Verfahren für die Konstruktion und den Bau von Dämmen für Absetzbecken.

Die TWG war nicht in der Lage, Angaben zur Menge an Aufbereitungsrückständen und taubem Gestein zu machen, die durch den Bergbau erzeugt werden bzw. wurden. Deshalb wurden nur allgemeine Daten aus dem jährlichen Eurostat-Bericht in das Kapitel 1 aufgenommen.

Bei der oben erwähnten Abwasserbehandlung und beim Cyanidabbau im Prozess der Goldlaugung konnte sich die TWG nicht auf Emissionswerte bei den CN-Koncentrationen im Zusammenhang mit den BVT einigen, die bei der Einleitung in Absatzbecken auftreten. Die TWG fand die im Richtlinienvorschlag zur Bewirtschaftung von Abfällen aus der Rohstoffindustrie[17] enthaltenen Bestimmungen zu dieser Frage ausreichend. In Artikel 13(4) der Richtlinie wird gegenwärtig ein Wert von 10 ppm WAD_{CN} innerhalb von 10 Jahren nach Umsetzung der Richtlinie verlangt.

Auf der abschließenden Plenartagung wurde entschieden, dass sowohl die 'trockene' als auch die 'nasse' Abdecktechnik gleichermaßen Verfahren sind, die die Bildung von Sauerwässern (ARD) bei der Stilllegung eines Absetzbeckens verhindern. Schweden stellte zur Untermauerung dieser Argumentation zusätzliche Daten zur Verfügung und formulierte die entsprechenden Passagen. Daher werden beide Verfahren als BVT zur Verhinderung der Bildung von ARD anerkannt.

Die TWG einigte sich auf einen allgemeinen Sicherheitsfaktor von 1,3 als BVT beim Betrieb und bei der Stilllegung von Dämmen und Halden. Dennoch gab es geteilte Ansichten bzgl. des Sicherheitsfaktors für stabile Langzeitdämme mit Wasser- bzw. 'nasser' Abdeckung. Ein Mitgliedsstaat sowie einige aus der Industrie kommenden Mitglieder der Arbeitsgruppe schlugen einen Wert von mindestens 1,3 vor. Dieser Vorschlag wurde damit begründet, dass es nicht praktikabel ist, den Sicherheitsfaktor von 1,3 (während des Betriebs) auf 1,5 (nach Stilllegung des Absetzbeckens) zu ändern, dass 1,3 sowieso für 'sicher genug' gehalten wird und dass sich der Wert in Übereinstimmung mit der aktuellen Gesetzgebung befindet. Dennoch schlugen die anderen Mitgliedsstaaten sowie einige aus der Industrie kommende Mitglieder

17 KOM(2003) 319 final, 2.6.2003. Hinweise auf die BVT sind Artikel 4(2), 19(2) und 19(3) des Richtlinienvorschlags enthalten

einen Wert von 1,5 mit der Begründung vor, dass dadurch die Überwachung in der Nachsorgephase verringert werden könne. Des Weiteren empfehle die Internationale Kommission für Große Talsperren (ICOLD) ebenfalls einen Wert von 1,5. Deshalb kam es zu keiner einstimmigen Entscheidung bzgl. der BVT beim Sicherheitsfaktor für stabile Langzeitdämme mit Wasserabdeckung.

Die TWG entschied, dass das Eintrocknen und das Eindicken von Aufbereitungsrückständen sowie das Absetzen verschlämmter Aufbereitungsrückstände in Abhängigkeit von vielen Faktoren (wie Korngröße, Klima usw.) alles BVT sind.

Hinsichtlich der Verfahren für den Bau und die Erhöhung von Dämmen für Absetzbecken einigte sich die TWG auf Grundlage der in diesem Dokument enthaltenen sowie weiterer auf der abschließenden Beratung verfügbar gemachten Informationen auf die Anwendbarkeit des Upstream-Dammbauverfahrens. Herkömmliche Dämme sowie die Upstream- Downstream- und Centerline-Dammbauverfahren werden unter spezifischen Bedingungen alle als BVT angesehen.

Empfehlungen für die künftige Arbeit

Das Ergebnis dieses Informationsaustausches, d.h. dieses Dokument, stellt einen wichtigen Schritt vorwärts bei der Verringerung der täglichen Umweltverschmutzung und der Verhinderung von Unfällen dar, zu der es durch Anlagen zur Bewirtschaftung von Aufbereitungsrückständen und taubem Gestein kommen kann. Zu einigen Themen sind jedoch die Angaben unvollständig und lassen keine Schlussfolgerungen zu den BVT zu. Die Hauptfragen sind in den Abschnitten 7.2 und 7.3 dargestellt.

Die künftige Arbeit könnte sich sinnvollerweise auf die Sammlung folgender Angaben konzentrieren:

- Erweiterung des Anwendungsbereichs: In ihrer Auftaktberatung entschied die TWG, den Anwendungsbereich so einzugrenzen, dass dieses Dokument im Rahmen des vorgegebenen Zeitplanes erarbeitet werden konnte. Bei einer künftigen Überarbeitung sollte dieses Dokument in zwei Richtungen erweitert werden: (1) die Liste der erfassten Minerale sollte erweitert werden und (2) das Dokument sollte dem endgültigen Anwendungsbereich der Richtlinie zur Bewirtschaftung von Abfällen aus der Rohstoffindustrie angepasst werden, nachdem diese angenommen wurde.
- Erzeugung von Aufbereitungsrückständen und taubem Gestein: Dem Dokument mangelt es an Angaben über die Menge an Aufbereitungsrückständen und taubem Gestein, die in Europa erzeugt wird. Es wäre ideal, wenn Mengenangaben zu all den Mineralen aufgenommen werden könnten, die in diesem Dokument berücksichtigt sind. Diese Angaben müssen aus laufenden Projekten durch die Industrie und die Mitgliedsstaaten erfasst werden.
- Emissionswerte bei der Abwasserbehandlung und beim Cyanidabbau im Zusammenhang mit BVT: Es mangelt an Angaben über die Emissionswerte bei der Abwasserbehandlung und beim Cyanidabbau. Diese Daten sollten bei laufendem Betrieb erfasst und danach analysiert sowie mit den bereits in Kapitel 3 enthaltenen Angaben in einer solchen Weise verglichen werden, dass man sich über Emissionswerte im Zusammenhang mit BVT einigen kann. Für den Cyanidabbau gibt es in Europa nur wenige Standorte als Beispiele. Deshalb sollten Informationen über den Betrieb solcher Standorte außerhalb Europas, insbesondere in Australien, Kanada und in den USA, gesammelt werden, wo größere Erfahrungen auf diesem Gebiet vorliegen. Es sollte ein Informationsaustausch mit der Industrie und den Behörden dieser Länder vereinbart werden, um solche Informationen zu erfassen, die für die Festlegung der Emissionswerte im Zusammenhang mit den BVT erforderlich sind.
- Bewirtschaftung von unter Wasser abgelagerten Aufbereitungsrückständen im Seewasser: Diese Technik wurde aufgenommen, nachdem es dazu eine Anmerkung zum zweiten Entwurf gab. Während ihrer abschließenden Beratung kam die TWG zu der

Schlussfolgerung, dass weitere Informationen erforderlich sind, um entscheiden zu können, ob es sich bei dieser Technik um eine BVT handelt. Im Moment fehlen eine klare Beschreibung der Technik und Angaben zur Anwendbarkeit, zu den medienübergreifenden Auswirkungen und zur Wirtschaftlichkeit dieser Technik. Diese Informationen müssen von der Industrie und den Mitgliedsstaaten aus laufenden Projekten gewonnen werden, ehe in vollem Umfang eingeschätzt werden kann, ob es sich um eine BVT handelt. Während der Abschlussberatung kamen diesbezüglich verschiedene Punkte zur Sprache, z.B.:

- die Technik kann durchaus eine mögliche Alternative zur Deponierung auf dem Land darstellen, wenn das Material nicht schädlich und kein Land zur Deponierung vorhanden ist;
- in diesem Gebiet wird die Bewirtschaftung eingedickter Aufbereitungsrückstände entwickelt, was deren weitere ‚Bewegung' ausschließt, wenn sie erst einmal unter Wasser deponiert wurden;
- die Deponierung (von Abfällen) in der See und in Seen ist schon oft angewandt worden, weshalb es Daten für die Entwicklung dieser Technik geben sollte;
- diese Technik ist möglicherweise nur in Fjorden anwendbar, da es ansonsten schwierig sein kann, weitere Bewegungen der Aufbereitungsrückstände vorherzusagen:
- im Zusammenhang mit der Ölhavarie der "Prestige" wurde festgestellt, dass im Falle eines Fehlers der Schaden nicht wieder gutzumachen ist.

- Wirtschaftliche Daten: Es mangelt an wirtschaftlichen Informationen über viele der in Kapitel 4 dargestellten Techniken. Diese müssen durch die Industrie und die Mitgliedsstaaten aus laufenden Projekten beschafft werden.
- Charakterisierung der Aufbereitungsrückstände und des tauben Gesteins:
  - In Annex 4 werden einige Standards aufgeführt, die zur Charakterisierung von Aufbereitungsrückständen und taubem Gestein herangezogen werden können. Das sind für die Messung geotechnischer Eigenschaften aber hauptsächlich BS- und ASTM-Standards. Es müssen jedoch mehr internationale und nationale Standards mitverwendet werden, um den Einsatz dieser Methoden in unterschiedlichen Mitgliedsländern zu ermöglichen.
  - Annex 4 listet viele Methoden für die Charakterisierung von Aufbereitungsrückständen und taubem Gestein auf und beschreibt sie. Es muss aber eine Standardmethode entwickelt werden, die generell in Europa eingesetzt werden kann und dort akzeptiert wird, damit ein entsprechendes Niveau der Charakterisierung allen tauben Gesteins und aller Aufbereitungsrückstände erreicht werden kann. Desgleichen muss eine Korrelation zwischen den Ergebnissen der Charakterisierung auf der einen Seite und dem langfristigen Umweltverhalten der Aufbereitungsrückstände und des tauben Gesteins auf der anderen unter tatsächlichen, d.h. 'in der Realität vorgefundenen' Bedingungen hergestellt werden.
- Eingedickte Aufbereitungsrückstände: Das vorliegende Dokument beschreibt Techniken zur Entwässerung verschlämmter Aufbereitungsrückstände sowie zur Bewirtschaftung verschlämmter Aufbereitungsrückstände in Absetzbecken. Das Dokument enthält dabei relativ wenige Angaben zur Technik des Einschlämmens von Aufbereitungsrückständen, da diese Technik im Bergbausektor erst vor kurzem eingeführt wurde. Es wird erwartet, dass diese Technik in Zukunft eine breitere Anwendung finden wird. Auch hier gilt wieder: Wenn Leistungsdaten vorhanden sind, müssen sie in dieses Dokument aufgenommen werden, da dies dann eine genauere Beschreibung der Anwendbarkeit dieser Technik möglich macht.

- Phytoremediation von Cyanidemissionen: Im Falle des Goldbergbaus resultieren die Umweltrisiken heutzutage aus dem Einsatz von Cyanidlösungen, die für Tiere und Pflanzen hochtoxisch sind. Ein Weg zur Minimierung der durch bergbauliche Emissionen entstehenden Umweltauswirkungen ist möglicherweise die Abwasserbehandlung in Pflanzenkläranlagen, d.h. die Phytoremediation in Pflanzenkläranlagen. Phytoremediation bedeutet die Nutzung von Grünpflanzen zur Stabilisierung oder Entfernung von Schad- und Giftstoffen aus Böden, Sedimenten oder Wasser. Bisherige Studien zur Phytoremediation von Schwermetallen und organischen Schad- und Giftstoffen haben die generelle Eignung von Pflanzen nachgewiesen, Substanzen aus Böden oder Wasser aufnehmen zu können. Das Ziel eines gerade laufenden Forschungsprojektes besteht darin, Pflanzen mit einem hohen Biomasseanteil zu finden, die eine große Aufnahmekapazität an Cyanid und eine geringe Anfälligkeit für Cyanid sowie toxische Metalle generell in sich vereinen. Das Ziel besteht nach erfolgten Laborstudien darin, Experimente in Modell-Pflanzenkläranlagen im Feldmaßstab vorzubereiten und durchzuführen, um eine Biosanierungstechnologie für den Einsatz in der Industrie zu entwickeln. Die Ergebnisse dieses Forschungsprojekts[18] sollten bei einer künftigen Überarbeitung in dieses Dokument eingearbeitet werden.

Die EG startet und unterstützt durch ihre RTD-Programme (Forschungs- und Technikentwicklungsprogramme) eine Reihe von Projekten, die sich mit sauberen Technologien sowie neuen Technologie- und Managementstrategien für Abwasserbehandlung und -recycling beschäftigen. Diese Projekte können potenziell einen nützlichen Beitrag bei der künftigen Überarbeitung dieses Dokuments leisten. Die Leser werden daher gebeten, das EIPPCB über alle Forschungsergebnisse in Kenntnis zu setzen, die für den Anwendungsbereich dieses Dokuments relevant sind (vgl. dazu auch das Vorwort zu diesem Dokument).

Themenvorschläge für künftige Forschungs- und Entwicklungsprojekte

Der Informationsaustausch hat auch einige Bereiche sichtbar gemacht, wo aus Forschungs- und Entwicklungsprojekten zusätzliche und nützliche Erkenntnisse gewonnen werden könnten. Diese beziehen sich auf folgende Themen:

- Bewirtschaftung über die gesamte Lebensdauer (“Life-cycle-Management”): Die TWG hat die Wichtigkeit der Bewirtschaftung eines Standortes über die gesamte Lebensdauer hinweg (“Life-cycle-Manegement”) erkannt, um ein hohes Niveau bei der Sicherheit und der Umweltbilanz zu erreichen. Dennoch mangelt es immer noch an wirtschaftlichen Daten, aus denen hervorgeht, dass es viel wirtschaftlicher ist, bergbauliche Operationen auf Grundlage eines vollständigen Lebensdauermodells zu gestalten. Die Forschung auf diesem Gebiet muss alle möglicherweise vorhandenen Fallstudien untersuchen, um die Wirtschaftlichkeit der Anwendung des integrierten Life-cycle-Managements im Vergleich mit der konventionellen Herangehensweise (d.h. der Erzielung von Höchstgewinnen während der Betriebsphase) festzustellen.
- Toxizität der Zersetzungsprodukte von Cyanid: Die Toxizität von Cyanid selbst ist ein ausreichend gut untersuchtes Thema, aber es scheint, dass auch einige Zersetzungsprodukte (des Cyanids) von toxikologischer Wichtigkeit sind. Angesichts der Auswirkungen von Verseuchungen an Standorten, wo Cyanid für die Goldlaugung verwendet wird, besteht die Notwendigkeit, auch die Toxizität der Zersetzungsprodukte von Cyanid zu erforschen.

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GLOSSAR

1. ALLGEMEINE TERMINI, ABKÜRZUNGEN, AKRONYME UND SUBSTANZEN

DEUTSCHER BEGRIFF	ENGLISH TERM	BEDEUTUNG
A	**A**	
Ableitungen (für Oberflächenwasser)/Fanggräben	diversions	Für Absetzbecken sind Ableitungen/Fanggräben üblicherweise kleine Auffanggräben, die den Überlauf aus einem Wassereinzugsgebiet aufnehmen und stromabwärts um das Absetzbecken und den Damm herum ableiten
Abraum/ Deckgebirge	overburden	Schicht natürlich gewachsenen Bodens oder massiven Gesteins auf einem Bodenschatz. Beim Abbau im Tagebau muss der Abraum vor der Gewinnung des Bodenschatzes abgetragen werden (siehe Abb. G1).
Abraum-Erz-Verhältnis	stripping ratio	Die in einer bestimmten Einheit ausgedrückte Menge an taubem Gestein oder Abraum, die entfernt werden muss, um die in einer bestimmten Einheit ausgedrückte Menge an Erz zu gewinnen, im Allgemeinen ausgedrückt in Kubikmetern (m^3) an taubem Gestein bzw. Abraum zu Tonnen (t) an Roherz.
Abraumhalde, Kippe	tailings heap/ spoil heap	Technische Anlage zur Lagerung von Aufbereitungs-rückständen oder Bergematerial auf der Landoberfläche. Trockene Entsorgung von Aufbereitungsrückständen auf der Landoberfläche.
Absetzbecken	tailings pond, lagoon	Technische Anlage für das Management von Aufbereitungsrückständen aus der Erzaufbereitung sowie für das Reinigen und Recyceln von Prozesswasser, die in den meisten Fällen durch einen Damm geformt werden. Absetzbecken enthalten in der Hauptsache Aufbereitungsrückstände mit unterschiedlichen Anteilen an Klarwasser.
(Erz-)Ader/Gang	vein	geringmächtige, komplexe Struktur von Erzanhäufungen umgeben von Gangart
aerob	aerobic	ein biologischer, sich bei Vorhandensein von Sauerstoff abspielender Prozess
akute Auswirkung	acute effect	Ein schädlicher Effekt für jeden lebenden Organismus, im Ergebnis dessen sich extrem schnell schwerwiegende Symptome entwickeln, die oft wieder verschwinden, sobald die Einwirkung aufhört.
akute Toxizität	acute toxicity	akute Toxizität
Alkali	alkali	Protonenakzeptor (Emprotid). Eine Substanz, die mehr oder wenig bereitwillig Wasserstoffionen in einer wässrigen Lösung aufnimmt.
Alkalinität	alkalinity	Maß der Kapazität einer Lösung, eine starke Säure zu neu-tralisieren.
anaerob	anaerobic	ein biologischer, sich bei Nichtvorhandensein von Sauerstoff abspielender Prozess
Anlage/ Einrichtung	facility	Stationäre technische Einheit, in der Aufbereitungsrückstände und/oder taubes Gestein bewirtschaftet werden und in der andere damit direkt verbundene Aktivitäten stattfinden, die eine technische Verbindung mit den ansonsten am Standort verfolgten Aktivitäten haben, Emissionen verursachen und Verunreinigungen in der Umwelt erzeugen könnten.

Anlagen (oder Methoden) zur Bewirtschaftung von Aufbereitungs-rückständen oder Bergematerial (TMF)	tailings management facilities (TMF)	Bezieht sich auf die Tatsache, dass Aufbereitungsrückstände aus den Aufbereitungsprozessen entsorgt/gelagert oder wieder verwendet werden müssen. Das jeweils gewählte Verfahren hängt neben vielen anderen Faktoren von den physikalischen Eigenschaften (grobkörnig oder feinkörnig) und der Behandlung des Erzes (trocken oder nass) ab. Typische Anlagen bzw. Methoden zur Bewirtschaftung von Aufbereitungsrückständen sind ▪ Absetzbecken, ▪ Halden/Kippen, ▪ Versatz, ▪ Wiederverwertung (als Baumaterial), ▪ die erneute Aufbereitung (d.h. die erneute Gewinnung eines Erzes durch neue, verbesserte Aufbereitungsverfahren).
Anlagen zur Bewirtschaftung von taubem Gestein oder Bergematerial (WRMF)	waste-rock management facility (WRMF)	Anlage/Einrichtung, wo taubes Gestein entsorgt, gelagert und in manchen Fällen behandelt wird, einschließlich Auslaugebereiche für Taubgestein.
assimilative Kapazität	assimilative capacity	Die Fähigkeit eines natürlichen Wasserkörpers, Abwasser bzw. toxische Materialien ohne schädliche Auswirkungen auf das Leben unter Wasser bzw. Schaden daran aufzunehmen.
Aufbereitung (Veredlung, Erzaufbereitung, Aufbereitung von Steinen und Erden)	mineral processing (beneficiation, ore dressing, mineral dressing, milling)	Prozesse zur Herstellung marktfähiger mineralischer Erzeugnisse (Konzentrate) aus Erz. Diese werden normalerweise auf dem Bergwerksgelände durchgeführt, wobei die Anlage als Aufbereitungsanlage (Hütte, Anreicherungsanlage) bezeichnet wird. Der essentielle Zweck besteht in der Massenreduzierung des Erzes, das zu den Folgeprozessen transportiert und durch sie (z.B. durch Schmelzen) weiterverarbeitet werden muss, wobei bestimmte Verfahren zur Trennung der gewünschten Mineralien von der Gangart und dem Nebengestein angewendet werden. Das dabei entstehende marktfähige Erzeugnis wird Konzentrat genannt, das zurückbleibende Material sind die Aufbereitungsrückstände. Die Aufbereitung beinhaltet verschiedene Prozesse und Vorgehensweisen, die auf den physikalischen Eigenschaften des Minerals (z. B. Korngröße, Dichte, magnetische Eigenschaften, Farbe) bzw. physikalisch-chemischen Eigenschaften (wie Oberflächenspannung, hydrophobe Eigenschaften, Benetzbarkeit) beruhen.
Aufbereitungsanlage (Hütte, Anreicherungsanlage)	mineral processing plant (mill, concentrator)	Anlage/Einrichtung, in der die Mineralien aufbereitet werden
Aufbereitungsrückstände, Grob-/Feinberge	tailings, coarse/fine discard	Erz, aus dem heraus so viel wie möglich des gewünschten Minerals gewonnen wurde. Aufbereitungsrückstände bestehen hauptsächlich aus taubem Gestein und können Prozesswasser, Aufbereitungschemikalien sowie Anteile nicht gewonnener Minerale enthalten. Hinweis: Im britischen Kohlesektor wird dieser Begriff wie folgt verwandt: Grobberg: die gröberen (und trockeneren) Fraktionen des

		Abgangs, die nach der Aufbereitung der Masse des gewonnen Materials aus der Abscheidung des gewünschten Produkts im Nass- oder Trockenverfahren übrig bleiben. Feinberg: die feineren (und feuchteren) Fraktionen des Abgangs, die aus den eingedickten oder geflockten suspendierten Feststoffen im Waschwasser entstehen, das zur Aufbereitung und Trennung des gewünschten Produkts vom Feinberg durch Waschen oder Flotation des gewonnen Materials verwendet wird.
Aufbereitungssand	tailings sand	Sand, der aus den Aufbereitungsrückständen gewonnen wird, die für den Bau des Damms des Absetzbeckens verwendet wurden. Der Sand wird häufig durch das Abscheiden aus den Aufbereitungsrückständen im Zyklonverfahren gewonnen.
Aufschluss	liberation	Freisetzung wertvoller Mineralien aus der Gangart.
Ausbeute	recovery	In Prozent ausgedrücktes Verhältnis eines Bestandteiles eines Konzentrats (oder bei Kohle die abschließende Fördermenge) zur Gesamtmenge desjenigen Minerals, das anfänglich vor der Gewinnung als Einsatzmaterial für die Aufarbeitung vorhanden war. Ein Maßstab zur Feststellung der Effizienz des Bergbaus, des Abbaus und der Aufbereitung.
Ausbringen/ Flotationsausbringen	yield	Auf Trockenbasis berechnetes und als Prozentsatz ausgedrücktes Masseverhältnis zwischen Konzentrat und Aufbereitungs- bzw. Flotationsaufgabe.
Ausbringung	percentage extraction	Verhältnis vom aus der Erzlagerstätte gewonnenen Bodenschatz zum Rohstoffvorkommen, ausgedrückt als Prozentsatz der ursprünglichen Menge des Bodenschatzes am Fundort.
autogenes Mahlen	autogenous grinding	Das Nachmahlen von Erz in der Trommel eines sich drehenden Zylinders ohne Kugeln und Stangen.
Azidität	acidity	Maß für die Kapazität einer Lösung, eine starke Base zu neutralisieren.
B	**B**	
Backenbrecher	jaw crusher	Eine Maschine zur Verringerung der Materialgröße durch Aufschlag oder durch Brechen zwischen einer festen und einer schwingenden Platte.
Bakterizid	bactericide	ein Schädlingsbekämpfungsmittel zur Bekämpfung oder Vernichtung von Bakterien
Belüftung	aeration	das Vermischen einer Flüssigkeit mit Luft (Sauerstoff)
Bergbau/Abbau (von Rohstoffen)	mining	Methoden und Techniken zur Erzgewinnung aus dem Boden, einschließlich der Nebenbetriebe (wie Halden, Werkstätten, Transporteinrichtungen, Bewetterung) und der im Bergwerk selbst oder in dessen unmittelbarer Nähe ausgeübten Nebentätigkeiten.
bergbauliche Tätigkeiten	mining operation	Jede Form des Abbaus von Erz, aus dem mineralische Stoffe gewonnen werden, und wo die unternehmerische Absicht in der Erzielung eines Betriebsgewinns bzw. im schrittweisen Aufbau eines profitablen Unternehmens besteht.
Bergwerksproduktion	mine production	Bei Metallen der Anteil von Metall im Konzentrat nach der Gewinnung, in allen anderen Fällen – wenn nicht anders angegeben – der gewichtsmäßige Anteil des Konzentrats nach der Verarbeitung des Minerals.

bestehende Anlage	existing operation	Eine in Betrieb befindliche Anlage oder eine Anlage, die entsprechend der Rechtslage vor dem Inkrafttreten dieser Richtlinie genehmigt war bzw. nach Ansicht der zuständigen Behörde Gegenstand eines umfassenden Genehmigungsverfahren ist, vorausgesetzt, die Anlage wird nicht später als ein Jahr nach Inkrafttreten dieser Richtlinie in Betrieb genommen.
Betreiber	operator	Jede natürliche oder juristische Person, auf die die Verantwortung für die Kontrolle, den Betrieb und die Instandhaltung eines Bergwerks, einer Aufbereitungsanlage, eines Absetzbeckens und/oder ähnlicher Einrichtungen, einschließlich der Nachsorgephase, übertragen worden ist. Wo sich dies aus der nationalen Gesetzgebung ergibt, eine Person, der entscheidende wirtschaftliche Macht über das technische Funktionieren eines Bergwerks, einer Aufbereitungsanlage, eines Absetzbeckens und/oder ähnlicher Einrichtungen, einschließlich der Nachsorgephase, bzw. entscheidender Einfluss darauf übertragen worden ist.
Biochemikalien	bio-chemicals	Chemikalien, die entweder natürlich vorkommen bzw. identisch mit natürlich vorkommenden Substanzen sind. Beispiele sind Hormone, Pheromone (Ektohormone) und Enzyme. Biochemikalien fungieren durch ihre nichttoxische, nichtlethale Vorgehensweise als Pestizide (Schädlingsbekämpfungsmittel) und stören damit das Paarungsverhalten von Insekten, regeln das Wachstum oder wirken als Abschreckungsmittel.
Biodiversität/ biologische Vielfalt/ Artenvielfalt	biodiversity	Die Anzahl und Verschiedenheit unterschiedlicher Organismen in ökologischen Komplexen, in denen sie auf natürliche Weise vorkommen. Organismen sind auf vielen Ebenen organisiert, die von kompletten Ökosystemen bis zu biochemischen Strukturen reichen, und die die Molekularbasis für Vererbbarkeit sind. Der Begriff umfasst somit unterschiedliche Ökosysteme, Arten und Gene, die für eine gesunde Umwelt vorhanden sein müssen. Eine große Anzahl der Arten charakterisiert die Nahrungskette und bildet die vielfachen Beziehungen zwischen Raub- und Beutetieren.
biologisch abbaubar	biodegradable	durch Mikroorganismen physisch und/oder chemisch zersetzbar; zum Beispiel sind viele Chemikalien, Nahrungsmittelreste, Baumwolle, Wolle und Papier biologisch abbaubar.
Biologische Pestizide	biological pesticides	bestimmte Mikroorganismen, einschließlich Bakterien, Pilze, Viren und Protozoen, die die auf sie angesetzten Schädlinge wirksam bekämpfen. Diese Pestizide haben normalerweise keine toxischen Auswirkungen auf Menschen und Tiere und hinterlassen keine giftigen oder anhaltend wirkenden chemischen Rückstände in der Umwelt.
biologische Auslaugung	bio-leaching	Prozess, in dem Mineralien mit Hilfe von Bakterien herausgelöst werden
Bioremediation (Biosanierung)	bioremediation	der Einsatz lebender Organismen (z.B. Bakterien) zur Reinigung von Ölteppichen oder zur Entfernung anderer Schadstoffe aus der Erde, dem Wasser und dem Abwasser.
Bioverfügbarkeit	bio-availability	Eigenschaft einer Substanz, die sie zugänglich und potentiell fähig macht, die Gesundheit eines Organismus

		zu beeinträchtigen. Hängt von den standortspezifischen Bedingungen ab.
Brechen	crushing	Zerkleinerungsprozess, in dem die Korngröße von Fördererz auf ein solches Maß reduziert wird, dass es gemahlen werden kann. Dies wird durch das Andrücken des Erzes gegen starre Oberflächen bzw. durch den Aufprall auf Flächen in stark eingeengten Bewegungsabläufen erreicht.
BREF	BREF	BVT-Merkblatt
BSB	BOD	biologischer Sauerstoffbedarf: die Menge gelösten Sauerstoffs, der von Mikroorganismen zur Dekompostierung organischer Stoffe benötigt wird. Die Maßeinheit ist mg O_2/l. Der BSB wird in Europa üblicherweise nach 3 (BSB_3), 5 (BSB_5) oder 7 (BSB_7) Tagen gemessen.
BVT	BAT	beste verfügbare Technik(en)
C	**C**	
CSB	COD	chemischer Sauerstoffbedarf: die Menge an Kaliumdichromat, ausgedrückt als Sauerstoff, die zur chemischen Oxidation von im Abwasser enthaltenen Substanzen bei ca. 150º C benötigt wird.
D	**D**	
d50, d80	d50, d80	Ein oft in der Mineralverarbeitung verwendeter Wert zur Beschreibung der Korngrößenverteilung. Er verweist auf die Korngröße, die in der betreffenden Probe zu 50 % bzw. 80 % kleiner als angegeben ist.
Damm eines Absetzbeckens	tailings dam, lagoon bank	Technisches Bauwerk zur Rückhaltung und zum Absetzen von Aufbereitungsrückständen und Prozesswasser. Die festen Bestandteile der Aufbereitungsrückstände setzen sich im Absetzbecken ab. Das Prozesswasser wird üblicherweise wieder verwendet.
diffuse Emission	diffuse emission	Emissionen, die vom direkten Kontakt flüchtiger oder leicht staubiger Substanzen mit der Umwelt (Atmosphäre, unter normalen Betriebsbedingungen) herrühren und sich aus ▪ dem der Ausrüstung eigenen Design (z. B. Filter, Färber), ▪ den Betriebsbedingungen (z. B. während des Transportes von Material zwischen Behältern) ▪ der Art der Betriebstätigkeit (z. B. Instandhaltungsarbeiten) oder ▪ der graduellen Freisetzung in andere Medien (z. B. in das Kühl- oder Abwasser) ergeben. Flüchtige Emissionen sind eine Teilmenge von diffusen Emissionen.
diffuse Quellen	diffuse sources	Mehrere Quellen ähnlicher diffuser oder direkter Emissionen, die mehrfach vorhanden und innerhalb eines definierten Bereichs verteilt sind.
Drainagewasser-Zusammensetzung	drainage chemistry	Konzentrationen von gelösten Bestandteilen in der Drainage, einschließlich Elementekonzentrationen, chemischer Stoffe und anderer wässriger chemischer Parameter.
Dränung/Drainage	drainage	Art und Weise, wie Gewässer eines Bereiches existieren und sich bewegen, einschließlich der

		Oberflächenwasserströmung und die Grundwasserpfade. Ein Sammelbegriff für alle konzentrierten sowie diffusen Wasserströme.
durchlässige reaktive Barriere	permeable reactive barrier	Eine durchlässige Zone, die ein reaktives Behandlungsgebiet enthält oder schafft, um eine Schadstofffahne zu unterbrechen und zu sanieren. Sie beseitigt durch physikalische, chemische oder biologische Prozesse auf passive Art Schadstoffe aus dem Grundwasserflusssystem. [123, PRB action team, 2003]
Durchlässigkeit/ Permeabilität	permeability	Fähigkeit eines Gesteins oder eines nicht versteinerten Materials zum Durchlassen strömungsfähiger Medien.
E	**E**	
Effektives Neutralisationspotenzial (ENP)	effective neutralisation potential (ENP)	Der Anteil des Neutralisationspotenzials, das die Säurebildung und Säureabgabe durch Beibehaltung eines Drainage -pH-Wertes von 6,0 oder mehr neutralisiert.
Eindickung	thickening	Prozess zur Trennung der festen und flüssigen Phase zur Erhöhung der Konzentration von Feststoffen in einer Suspension durch Sedimentation. Die Eindickung wird von der Bildung eines reinen Feststoffes begleitet.
EIPPCB	EIPPCB	Europäisches IPPC-Büro
Eiserner Hut	gossan	Erz im oberen Teil eines sulphidischen Erzkörpers, das zu Oxiderz verwittert ist
Emission	emission	Die direkte oder indirekte Freisetzung von Substanzen, Vibrationen, Hitze oder Lärm aus individuellen oder diffusen Quellen der Anlage in die Luft, in das Wasser oder in den Boden.
Emissionsgrenzwerte	emission limit values	Die Menge, ausgedrückt in bestimmten spezifischen Parametern, Konzentrationen und/oder im Ausmaß einer Emission, die während eines oder mehrerer Zeitabschnitte nicht überschritten werden darf.
"End-of-Pipe"-Verfahren	“end-of-pipe” technique	Ein Verfahren, das die am Ende freigesetzten Emissionen bzw. den Endverbrauch durch einige zusätzliche Prozesse reduziert, aber nicht den eigentlichen Ablauf des Kernprozesses. Synonyme: "Sekundärmaßnahme", "Abatement-Techniken" (= nicht prozessintegrierte Verfahren). Antonyme: "prozessintegrierte Techniken", "Primärmaßnahme" (ein Verfahren, das zum Teil die Art und Weise ändert, wie der Kernprozess abläuft und damit die Rohemissionen bzw. die Verbrauchswerte senkt).
EOP	EOP	End-of-pipe (= nicht prozessintegriert)
Erosion	erosion	Lösen und nachfolgende Entfernung von Gesteins- oder Oberflächenmaterial durch Wind, Regen, Welleneinwirkung, Frost, Tauwetter oder durch andere Prozesse.
erste Brechstufe	primary crushing	Prozess zur Zerkleinerung von Erz in kleinere Stücke, um es zur weiteren Aufbereitung vorzubereiten und/oder um es zur Aufbereitungsanlage befördern zu können. In unterirdischen Bergwerken befindet sich der Erstbrecher oft unter Tage oder zu Beginn der Aufbereitungsanlage.
Erz	ore	Mineral bzw. verschiedene Mineralien zusammen von qualitativ wie quantitativ ausreichendem Wert, das/die wirtschaftlich abgebaut werden kann/können. Die meisten Bodenschätze (einschließlich im weitesten Sinne Kohle) sind einerseits Mischungen abbaubarer Mineralien und andererseits als Gangart beschriebenes Nebengestein (siehe Abb. G1).

Erzlagerstätte	orebody (mineral deposit)	Natürlich vorkommende, aus einer Anhäufung des begehrten Minerals einerseits und des taubem Gesteins andererseits bestehende geologische Struktur, aus der das Mineral gewinnbringend oder zumindest angemessen gewinnversprechend abgebaut werden kann (siehe Abb. G1)
Europa	Europe	Die aktuellen EU-Mitgliedstaaten (EU-15) und die EU-Beitrittsländer (siehe Abschnitt 2 dieses Glossars)
Eutrophierung/ Nährstoffanreicherung in einem Gewässer	eutrophication	Die Verunreinigung eines Wasserkörpers durch Schmutz- und Abwasser, durch aus dem Boden eingespülte Düngemittel sowie durch industrielle Abwässer und Abfälle (anorganische Nitrate und Phosphate). Diese Verbindungen stimulieren das Wachstum von Algen, reduzieren den Sauerstoffgehalt im Wasser und töten Tiere mit einem hohen Sauerstoffbedarf.
F	**F**	
Feuchtraumzellentest	humidity cell test	Kinetisches Prüfverfahren, das hauptsächlich zum Messen der Säurebildung und der Neutralisation von sulfidhaltigem Gestein dient.
finanzielle Garantie	financial guarantee	Mittel aus verschiedenen Quellen, die von der Regulierungsbehörde als Ausgleich für Schließungskosten verwendet werden können.
flächiges Einspülen in dünnen Lagen	sub-aerial method of deposition	In Nordamerika weithin verwendeter Begriff für ein Verfahren mit dem Spuckpfeifensystem, bei dem mittels Sprührohren dünne Lagen von Aufbereitungsrückständen über einen vorher geschaffenen Spülstrand verteilt werden.
Flockungsmittel	flocculant	Chemischer Stoff, der das Aggregieren oder Verklumpen suspendierter Partikel verursacht. Diese größeren Agglomerate sinken schneller nach unten. Flockungsmittel werden zum Aggregieren kleiner Partikel eingesetzt, deren langsames Absetzverhalten es ansonsten schwierig machen würde, sie zu entfernen.
Flora und Fauna	biota	alle lebenden Organismen in einem bestimmten Gebiet
Flotation	flotation	Eine Form der Trennung von Mineralien von der Gangart auf der Basis der unterschiedlichen Oberflächenreaktionen auf gewisse Reagenzien (oder alternativ auf der Basis der Grenzflächenchemie der mineralischen Partikel in der Lösung). Die Reagenzien binden sich an das betreffende Mineral und bilden eine wasserabweisende Oberfläche. Das Mineral steigt dann mit zugeführter Luft zur Oberfläche der Flotationszelle auf, wo es als Schaum abgeschöpft werden kann. Wenn es um das Aufschwimmen der Gangart geht, wird dieser Prozess umgekehrte Flotation genannt.
Flöz	seam	Horizontal gelagerte Schicht von Mineralen (typisch für Kohle und einige Arten von Salzlagerstätten). Auf Grund tektonischer Überprägungen können Flöze sowohl als Falzung wie auch steilgelagert vorkommen.
flüchtige Emission	fugitive emission	Emission, die durch undichte Ausrüstungen/Lecks verursacht wird: Emission in die Umwelt, die auf den allmählichen Verlust der Dichtheit bei Ausrüstungen zurückzuführen ist, die für die Aufnahme von (gasförmigen oder flüssigen) strömungsfähigen Medien vorgesehen sind. Entstehung grundsätzlich durch einen Druckunterschied und die daraus entstandene Leckage. Beispiele für flüchtige Emissionen sind Leckagen an einem

		Flansch, einer Pumpe, aus versiegelten oder abgedichteten Ausrüstungen…
FOB (frei an Bord)	free on board (f.o.b.)	Preis einer Warenlieferung an einen Kunden inklusive aller Schiffs- oder Lkw-Ladekosten
Freibord	freeboard	Vertikaler Abstand (Höhe) zwischen dem normalen Höchstbetriebsspiegel (Pegel) eines Absetzbeckens und der Dammkrone, dessen Zweck die Bereitstellung von Auffangkapazitäten bei Hochwasser oder beim plötzlichen Einströmen von Wasser ist.
freies Cyanid	free CN	Das nicht in komplexen Ionen gebundene Cyanid, sowohl das molekulare HCN und das Cyanidion [24, British Columbia CN guide, 1992]
G	**G**	
Gangart	gangue	Die Bestandteile des Erzes, die wirtschaftlich unerwünscht sind, beim Abbau aber nicht vermieden werden können (siehe Abb. G1).
Gehalt	grade	Dimensionsloser Anteil eines Bodenschatzes, oft ausgedrückt in Prozent, Gramm pro Tonne (g/t) oder in Teilen pro Million (ppm).
Geochemie	geochemistry	Lehre von der Chemie geologischer Materialien und des Zusammenwirkens der geologischen Materialien mit der Umwelt.
Geologie	geology	Wissenschaft von der Erde, ihrer Geschichte und der Änderungen, die vor sich gegangen sind bzw. noch sich noch abspielen, von den Gesteinen und unverfestigten Materialien, aus denen die Erde besteht sowie von ihrer Herausbildung und Umwandlung.
Gesamt-Cyanid	total CN	Der Gesamtgehalt aller, in den verschiedenen Verbindungen in einer wässrigen Lösung existierenden Cyanide. [24, British Columbia CN guide, 1992]
Gewinnung im Tagebau	open-pit (open-cast) mining	An der Erdoberfläche stattfindender Bergbau. Die bergbaulichen Tätigkeiten und die Umwelt stehen über ein großes Gebiet hinweg miteinander in Berührung.
Gewinnungsmetho den	extraction methods	Es gibt grundsätzlich vier Methoden, Erze zu gewinnen, nämlich ▪ im Tagebau, ▪ im Bergbau unter Tage ▪ durch Aussolung (Lösungsabbau) und ▪ im Steinbruchbetrieb.
Glühverlust (LOI)	loss on ignition	Bezeichnet bei chemischen Analysen den Gewichtsverlust, der sich durch das Erhitzen der Materialprobe auf eine hohe Temperatur und nach dem vorzeitigen Trocknen bei einer Temperatur ergibt, die gerade über dem Siedepunkt von Wasser liegt. Der Gewichtsverlust beim Trocknen schlechthin wird freies Wasser, der Gewichtsverlust bei Temperaturen über dem Siedepunkt von Wasser wird Glühverlust genannt.
größtes denkbares Erdbeben (MCE)	maximum credible earthquake (MCE)	Hypothetisches Erdbeben, das bei seismischen Ereignissen von den potenziellen regionalen und lokalen Quellen erwartet werden kann und das die schwersten Untergrundvibrationen vor Ort erzeugen könnte.
Grundwasser	groundwater	Teil des unterirdischen Wassers innerhalb der Sättigungszone, zu unterscheiden von Oberflächenwasser
grundwasserführen	aquifer	Eine grundwasserführende Gesteinsschicht (einschließlich

de Schicht		Kies und Sand), die Wasser in einer nutzbaren Menge in einen Brunnen bzw. an eine Quelle abgibt.
Grundwasserspiege l	phreatic surface	Die Fläche zwischen der Sättigungszone und der Luftzone; d.h. die Oberfläche eines uneingeschränkten Grundwasserkörpers, dessen Druck gleich dem der Atmosphäre ist.
H	**H**	
halbautogenes Mahlen	semi-autogenous grinding	Das Nachmahlen von Erz in der Trommel eines sich drehenden Zylinders mit nur wenigen Kugeln und Stangen.
hydraulischer Gradient	hydraulic gradient	Unterschied im hydraulischen Druck zwischen zwei Punkten geteilt durch die Entfernung zwischen diesen Punkten.
Hydrogeologie	hydrogeology	Lehre von den Grundwassersystemen und -kreisläufen (Wechselbeziehung zwischen geologischen Materialien und Prozessen mit Wasser)
Hydrologie	hydrology	Lehre vom Vorkommen, von der Zirkulation, der Distribution, der Bewegung und von den chemischen und physikalischen Eigenschaften der gesamten Wasserumwelt sowie von der Reaktion mit ihr
I	**I**	
IEF	Information Exchange Forum (IEF)	Forum für Informationsaustausch (informeller Beratungskreis für die IVU-[IPPC-]Richtlinie)
Immission	immission	Auftreten und Menge einer umweltverschmutzenden Substanz bzw. eines solchen Geruchs oder Lärms
Industrieminerale	industrial minerals	Nichtmetallisches Erz, Nichtbrennstoffe bzw. nicht-edelmetallisches Gestein sowie mineralische bzw. nichtversteinerte Materialien von ökonomischer Verwertbarkeit. Industriemineralien werden vorrangig im Bausektor oder in der chemischen und verarbeitenden Industrie verwendet. Beispiele sind Baryt (Schwerspat), Borat, Feldspat, Fluorit (Flussspat), Kaolin, Kalkstein, Phosphat, Kali, Strontium und Talkum.
Infiltration	infiltration	Eindringen von Wasser in eine poröse Substanz
innere Erosion oder Suffision	piping	Überwiegend unterirdische, durch fließendes Wasser verursachte Erosion in Lockermassen, die auf Grund nicht mehr vorhandener Partikel zur Bildung von länglichen Hohlräumen führt.
IVU (IPPC)	Integrated Pollution Prevention and Control (IPPC)	Integrierte Verhinderung und Kontrolle der Umweltverschmutzung
J	**J**	
K	**K**	
Kammer-Filterpresse	chamber filter press	Gerät, um Schlämme zu entwässern.
Kegel-, Kreisel- oder Walzenbrecher	gyratory crusher	Ein Vorbrecher, der aus einer vertikalen Spindel besteht, deren Fuß an einem exzentrischen Lager innerhalb eines konischen Gehäuses befestigt ist. Das Oberteil trägt einen konisch geformten Brecherkopf, der sich exzentrisch in einer konischen maw dreht.
Kegelbrecher	cone crusher	Maschine zur Reduzierung der Größe des Materials mittels

		eines Stumpfkegels, der sich innerhalb einer äußeren Kammer um seine Längsachse dreht, wobei sich der Raum zwischen der äußeren Kammer und dem Kegel verjüngt (konisch zuläuft).
keine Daten	n/d	keine Daten
Klarwasser	free water	Der Wasserbereich in einem Absetzbecken oberhalb der abgesetzten Aufbereitungsrückstände (Schlämme), der normalerweise durch Abpumpen oder Dekantieren entfernt wird.
Klarwasserabzugsl eitungen	decant lines	Rohrleitungen, die aus dem Absetzbecken dekantiertes Klarwasser durch oder über den Damm des Absetzbeckens bzw. um ihn herum zu einem stromabwärts gelegenen Sammelpunkt leiten.
kleine Halde	tip	Im britischen Bergbausektor benutzter Ausdruck für Halden oder Absetzbecken, die aus Aufbereitungsrückständen eines Bergwerks oder Steinbruchs bestehen.
kleine und mittlere Unternehmen	small und medium enterprises (SME)	der „Mittelstand“, die mittelständische Wirtschaft
Konzentrat	concentrate	Vermarktbares Produkt mit einem erhöhten Grad des Wert-minerals nach der Trennung in einer Aufbereitungsanlage.
L	**L**	
Langzeitphase	long-term phase	Für Aufbereitungsrückstände benötigter Zeitraum nach dem Ende der Rehabilitationsphase, um ausreichend inaktiv zu werden, so dass sie keine problematischen Aus-wirkungen mehr auf die Umwelt haben.
Lauge	leachate	Durch Laugung gewonnene Lösung, z. B. Wasser, das durch lösliche Substanzen enthaltenden Boden gesickert ist und das gewisse Anteile dieser Substanzen aufgelöst enthält.
Laugung	leaching	Passage einer Flüssigkeit/eines Lösemittels durch poröses oder gebrochenes Material für die Gewinnung bestimmter Komponenten aus der Flüssigkeitsphase. Gold kann zum Beispiel durch Haldenlaugung aus porösem Erz oder aus pulverisiertem Abraum gewonnen werden. Andere Methoden sind die Tanklaugung von Erzen, Konzentraten oder Aufbereitungsrückständen sowie die In-situ-(Untertage-) Laugung.
Lebenszyklus	life-cycle	Gestaltung, Bau, Betrieb, Überwachung, Schließung und Sanierung einer Anlage/Einrichtung und die Nachsorge dafür.
leicht freisetzbares Cyanid	weak acid dissociable cyanide (WAD CN)	Leicht freisetzbares Cyanid repräsentiert Cyanide, die beim Rückfluss einer Schwachsäure (Halbsäure) zerfallen, gewöhnlich bei einem pH-Wert von 4,5. [24, British Columbia CN guide, 1992]
Lithologie	lithology	Aufbau von Gesteinen, einschließlich physikalischer und chemischer Eigenschaften, wie Farbe, mineralogische Zusammensetzung, Härtegrad und Körnung.
Löslichkeit	solubility	Die Menge einer gelösten Substanz, die sich in einem bestimmten Volumen an Lösungsmittel, bei einer bestimmten Temperatur und bei einem bestimmten Druck in einer gesättigten Lösung bildet. Der Grad, zu dem Ver-bindungen löslich sind, hängt von ihrer Fähigkeit und der Fähigkeit der anderen gelösten Stoffe ab, in einer besonderen Drainage-Chemie Ionen und wässrige

		Komplexe zu bilden.
Lysimeter	lysimeter	Vorrichtung zur Wassergewinnung aus den Bodenporen und zur Bestimmung der löslichen Bestandteile, die aus der Drainage entfernt wurden.
M	**M**	
Mahlen	grinding	Zerkleinerungsprozess, der eine feine Korngröße ergibt (< 1 mm), wobei die Verringerung der Größe durch Abrasion und Aufschlagen erreicht und manchmal auch durch die freie Bewegung von losen Mahlmitteln, wie Stäben, Kugeln und Steine, unterstützt wird.
medienübergreifende Auswirkungen	cross-media effects	Die Berechnung/Bewertung medienübergreifender Auswirkungen von Wasser-/Luft-/Bodenemissionen, des Energieverbrauchs, des Verbrauchs von Rohstoffen sowie von Lärm und Wasserentnahme (d.h., alles was von der IVU-Richtlinie (= IPPC-Richtlinie) verlangt wird).
Mischen	blending	Mischen des Rohstoffes, um Inputmaterial mit einer gleich bleibenden Qualität für spätere Verarbeitungsprozesse zu erzielen
Mönch	decant tower	Einlaufbauwerk, das jeweils entsprechend dem Ansteigen des Absetzbeckens erhöht wird. Der Mönch schöpft das klare Wasser von der Oberfläche des Absetzbeckens ab und leitet es über die Klarwasserabzugsleitung ab.
multimediale Effekte	multi-media effects	siehe medienübergreifende Auswirkungen
Mutterboden	top soil	Die oberste, humusreiche Bodenschicht auf einer Lagerstätte, die vor Beginn des Abbaus entfernt werden muss. (siehe Abb. G1)
N	**N**	
Neutralisation	neutralisation	Erhöhung des pH-Wertes saurer Lösungen bzw. Verringerung des pH-Wertes alkalischer Lösungen hin zu einem annähernd neutralen pH-Wert (ca. pH 7) durch eine Reaktion, in der sich die Hydrogenionen einer Säure und die Hydroxylionen einer Base zu Wasser verbinden.
Neutralisationspotential (NP)	neutralisation potential (NP)	Allgemeiner Ausdruck für die Kapazität einer Probe oder eines Materials, Azidität zu neutralisieren.
O	**O**	
Oberflächenabfluss	run-off	Der Teil des Niederschlags und der Schneeschmelze, der nicht versickert, sondern an der Oberfläche abfließt.
Ökosystem	ecosystem	Gemeinschaft von Organismen und deren unmittelbare physikalische, chemische und biologische Umwelt
P	**P**	
phreatisch	phreatic	sich auf das Grundwasser beziehend
Primärmaßnahme	primary measure/ technique	Technik, die zum Teil die Art und Weise ändert, wie der Kernprozess abläuft, und damit die Rohemissionen bzw. die Verbrauchswerte senkt (siehe End-of-pipe-Verfahren).
Probe	sample	Repräsentative, aus der Lagerstätte oder aus den nachgeschalteten Prozessen stammende Menge an Feststoffen für die Durchführung analytischer Testverfahren. Die Menge des Feststoffes und die Anzahl der Proben, die aus der Lagerstätte oder den nachgeschalteten Prozessen entnommen werden, müssen statistisch relevant sein.

prozessintegriert	process-integrated (PI)	prozessintegriert
Pumpponton/ -schiff	pump barge	Schwimmender Träger für die Pumpen auf dem Absetzbecken, die zur Wiedergewinnung des Klarwassers dienen, das in der Aufbereitungsanlage wieder verwendet wird
Q	**Q**	
Quebracho	quebracho	flüssiger Extrakt aus der Rinde des gleichnamigen Baumes; enthält bis zu 65 % die Gerbsäure Tannin, die als Sedativum bei der Schaumflotation (Schaumaufbereitung) für oxidierte Minerale eingesetzt wird.
R	**R**	
Reibungswinkel	friction angle, angle of friction	Winkel zwischen der Vertikalen und einer Oberfläche und der resultierenden, auf einen auf einer Oberfläche sitzenden Körper einwirkenden Kraft, bei der der Körper anfängt zu rutschen.
Restgold	refractory gold	Das enthaltene Gold ist sub-mikroskopisch (< 1 µm) und in den Sulfidmineralgittern fein verteilt.
Rohförderung	run of mine (ROM)	aus der Grube geförderter, unaufbereiteter Rohstoff (Erz).
Rohstoffvorkommen/Bodenschätze	mineral resources	Konzentration oder Vorkommen natürlicher, fester, anorganischer oder fossiler organischer Materialien in oder auf der Erdkruste in solch einer Form und Menge und in solch einem Gehalt oder einer Qualität, dass sich vernünftige Aussichten für den wirtschaftlichen Abbau bieten. Lagerstätte, Menge, Gehalt, geologische Eigenschaften und die weitere Verfügbarkeit eines mineralischen Rohstoffs sind durch spezifische geologische Beweise und entsprechendes Fachwissen bekannt bzw. werden daraus abgeleitet oder interpretiert.
Rückführungsleitungen	reclaim lines	Rohrleitungen, die zum Ableiten des aus dem Absetzbecken zurückgewonnenen Wassers in die Aufbereitungsanlagen verwendet werden.
Rückführungssystem	reclaim system	Verschiedene Bestandteile, die zu einem System gehören, das zur Wiedergewinnung von Wasser aus dem Absetzbecken und zu dessen Weiterleitung an die Aufbereitungsanlage gebaut wurde. Dazu können Pumppontons, Rückpumpleitungen, Mönche und Klarwasserabzugsleitungen gehören.
S	**S**	
Säure	acid	Protonendonator. Eine Substanz, die mehr oder weniger bereitwillig Wasserstoffionen in einer wässrige Lösung abgibt.
saure Bergbaugewässer (AMD), Sauerwässer (ARD)	acid mine drainage (AMD), acid rock drainage (ARD)	Säurehaltige Sicker- bzw. Grubenwässer aus Tagebauen, Untertage-Bergwerken, taubem Gestein oder Anlagen für Aufbereitungsrückstände, die freie Schwefelsäure und gelöste metallische Sulfatsalze enthalten, die im Prozess der Oxidation der darin enthaltenden Sulfidmineralien bzw. Zusatzstoffe entstehen. Die Säure löst Mineralien aus dem Gestein und verändert die Qualität des Dränwassers weiter.
Säure-Base-Analyse (ABA)	acid-base accounting (ABA)	Säure-Base-Analyse (ABA) ist ein Prüfverfahren, mit dem das Säure neutralisierende und das Säure bildende

		Potenzial von Gesteinsproben bestimmt wird.
Säurebildung	acid generation	Entstehung von Azidität (eines Säuregrades bzw. -gehalts), unabhängig von der Auswirkung auf das angrenzende Porenwasser bzw. ob das Material generell Säure produzierend oder neutralisierend ist.
Säurebildungspotential (AP)	acid potential (AP)	Maximales Säurebildungspotential einer Probe. Die Berechnung des AP (bzw. des MPA) ist ein integraler Bestandteil der Säure-Base-Analyse.
Schacht	shaft	Öffnung, die von der Erdoberfläche aus meistens senkrecht oder schräg nach unten durch verschiedene Bodenschichten führt. Ein Schacht kann zur Bewetterung, zur Entwässerung und/oder für Personen- und Materialtransporte (beispielsweise Erz, taubes Gestein) dienen. Er verbindet die Oberfläche mit den Arbeitsplätzen unter Tage.
Schadstoff	pollutant	Einzelne Substanz oder Substanzgruppe, die die Umwelt schädigen oder beeinträchtigen kann.
Scherfestigkeit	shear strength	Der innere Widerstand eines Körpers gegen Scherkräfte, wozu typischerweise die Reibung und ein reibungsunabhängiges Element, die Kohäsion, gehören.
Schlagbrecher	impact crusher	In Schlagbrechern erfolgt die Zerkleinerung des Materials hauptsächlich durch den Aufprall von Schlägern bzw. Hämmern, auf die die frei durch die Brecherkammer fallenden Gesteinsstücke auftreffen und die sie bei hoher Geschwindigkeit gegen feststehende Oberflächen werfen.
Schlamm/ Schlämme	slurry	eine Suspension aus Flüssigkeit und Feststoffen
Schlitzwand	cut-off trench	Ein unter einer Grundfläche bzw. innerhalb eines Dammes angeordneter undurchlässiger Wall, eine Einfassung bzw. eine andere Struktur zur Verhinderung oder Reduzierung von Verlusten durch Versickerung entlang einer Bauwerkstrennfläche oder durch poröse bzw. gebrochene Schichten. Kann aus Beton, kompaktiertem Ton, durch verriegelnde Spundwände oder entlang einer Reihe von in Löchern injizierten Zementmörtels geschaffen werden.
Schmutzwasser	effluent	Strömungsfähiges Medium (Luft oder Wasser mit Verunreinigungen), das eine Emission bildet.
Schüttwinkel	angle of repose	Die maximale Neigung, bei der eine Anhäufung losen oder körnigen festen Materials beim Schütten oder Absetzen auf Halde oder am Hang ohne nachzurutschen liegen bleibt bzw. sich setzt.
Sekundärmaßnahme	secondary measure/ technique	siehe "End-of-Pipe"-Verfahren
Setzmaschine	jig	Gerät, in dem Materialien im ständigen Fluss durch ihre unterschiedliche Dichte getrennt werden.
Sickerwasserrückhaltedamm	seepage recovery dam	Ein kleiner Damm zur Wasserrückhaltung unterhalb des Hauptdamms des Absetzbeckens, dessen Aufgabe darin besteht, alle Oberflächen- und Sickerwässer, die am Hauptdamm vorbeifließen, abzufangen, zu sammeln und in das Absetzbecken zurückzuleiten.
Siebung	screening	Trennung von Material in unterschiedliche Korngrößen.
Sortierung	separation	Aufbereitungsverfahren zur Trennung der Rohförderung in Konzentrate und Aufbereitungsrückstände.
spezifische Emission	specific emission	Auf eine Referenzbasis bezogene Emissionsgröße, wie die Produktionskapazität oder die tatsächliche Produktion (z.B. Masse pro hergestellter Tonne oder Einheit)

Spukpfeifensystem	spigotting	Verfahrensweise, bei der die Verarbeitungsrückstände über eine große Anzahl von Ausflussöffnungen oder Austragsrohren in ein Absetzbecken eingeleitet werden. Dieses Verfahren gewährleistet eine ziemlich gleichmäßige Verteilung der Aufbereitungsrückstände über den Spülstrand des Absetzbeckens, der die innere wasserseitige teildurchlässige Zone des Absetzbeckendamms bildet.
Spülleitung	tailings line	Rohrleitung zum Transport von Aufbereitungsrückständen von der Aufbereitungsanlage zum Absetzbecken.
Spülstrand	tailings beach	Bereich zwischen dem Klarwasserpegel und der Dammkrone in einem Absetzbecken für Aufbereitungsrückständen, die sich aus abgesetzten Feststoffen aus Aufbereitungsschlämmen dort bilden und nicht von Klarwasser bedeckt sind.
Spülstrandüberflutung	drowning the beach	Schnelles Ansteigen des freien Wassers im Absetzbecken, das den halbdurchlässigen, oberhalb des Dammes gelegenen Spülstrand erreicht oder gar überflutet und dazu führt, dass Klarwasser gegen den Damm des Beckens drückt.
Steinbruch	quarry	*Das gesamte Gebiet unter der Kontrolle eines Betreibers, der eine Tätigkeit ausführt, zu der die Erkundung, der Abbau, die Aufbereitung sowie die Lagerung von Mineralien gehört, einschließlich der üblicherweise dazugehörenden Infrastrukturen und der Abfallwirtschaft, nicht jedoch ein Bergwerk, von dem der Steinbruch sich dadurch unterscheidet, dass er üblicherweise oben und vorn offen ist und für die Gewinnung von Baumaterialien, wie Schiefer, Kalkstein, Kies und Sand, genutzt wird.*
Stilllegung	decommissioning	Prozess, durch den der Bergbaubetrieb eingestellt wird.
Sümpfung/ Entwässerung	dewatering	Prozess des Entfernens/der Ableitung des Wassers aus einem untertägigen oder übertägigen Bergwerk bzw. aus dem umgebenden Gestein oder den Lockermassen/Böden (Sümpfung). Dieser Begriff wird weithin auch für die Reduzierung des Wassergehaltes in Konzentraten, Aufbereitungsrückständen und Aufbereitungsschlämmen benutzt (Entwässerung).
T	T	
taubes Gestein, Abgang, Abraum	waste-rock, discard, dirt, spoil	Jene Teile der Lagerstätte ohne oder nur mit geringen Mengen an Erz, die nicht gewinnbringend bergbaulich genutzt oder verarbeitet werden können (siehe Abb. G1)
Technische Arbeitsgruppe	technical working group (TWG)	
U	**U**	
Überwachung/ Monitoring	monitoring	Prozess zur Beurteilung oder Bestimmung des tatsächlichen Wertes und der Variationen einer Emission oder eines anderen Parameters, der auf systematischen, periodischen oder stichprobenartigen Überwachungsprozeduren, Inspektionen, Probenahmen und Messungen oder anderen Bestimmungsmethoden beruht, die zur Bereitstellung von Informationen über ausgestoßene Mengen von emittierten Schadstoffen vorgesehen sind und/oder über in diesem Zusammenhang vorkommende Trends.
ultrabasisch	ultramafic	hauptsächlich aus mafischen Mineralien

		zusammengesetztes Eruptivgestein, z.B. monomineralische, aus Hyper-sthen, Augit oder Olivin zusammengesetzte Gesteine
Umwelt	environment	In wechselseitiger Beziehung stehende physikalische, chemische, biologische, soziale, geistige und kulturelle Komponenten, die das Wachstum und die Entwicklung lebender Organismen beeinflussen.
Untertagebergbau	underground mining	unter Tage durchgeführte Rohstoffgewinnung. Die Lagerstätte wird dazu mit Schächten, Rampen oder Stollen erschlossen.
V	**V**	
verbundene Strukturen, dazugehörige Arbeiten, Verrichtungen und Zubehör	associated structures, appurtenant works, auxiliary works, appurtenances	Alle Strukturen, Bauteile, Komponenten und Anlagen, die funktionell zur Kippe gehören, u.a. Überläufe und Überlaufkanäle, Dekantiertürme und Rohrleitungen, Regeneratpumpen, Wasserkanäle, Umgehungsbauten usw.
Verdichtung	compaction	Prozess, der zu einer Volumenreduzierung führt. Diese Veränderung rührt typischerweise von Auflasten her, die von außen her eine höhere Dichte der Feststoffe bewirken. Insbesondere in feinen Böden erfordert dies einen Abfluss von Porenwasser. Eine größere Verdichtung führt oft zu einer erhöhten Verfestigung.
Verdunstung	evaporation	Physikalischer Prozess, im Verlaufe dessen eine Flüssigkeit in Gas umgewandelt wird.
Verflüssigung	liquefaction	Phänomen, das gewöhnlich in lockeren, gesättigten Böden auftritt, wenn der zusätzliche Porenwasserdruck (z. B. durch ein Erdbeben) gleich dem ursprünglichen Einschlussdruck ist und sich der Boden wie eine dichte Flüssigkeit verhält, die unfähig ist, der Scherspannung zu widerstehen.
Verminderung	mitigation	Auf die Vermeidung, Kontrolle oder Verringerung der Schwere ungünstiger physikalischer, chemischer, biologischer und/oder sozioökonomischer Auswirkungen einer Tätigkeit gerichtete Aktivität
vermutlich höchste Flut (PMF)	probable maximum flood (PMF)	Das gravierendste Niederschlags- und/oder Schneeschmelzereignis, das an einem bestimmten geografischen Ort für denkbar möglich gehalten wird. Eine standortspezifische Bestimmung auf der Grundlage des möglichen Ausmaßes meteorologischer und hydrologischer Ereignisse und Bedingungen. Zu den Variablen gehören die Dauer, das Gebiet und die Jahreszeit. Wird üblicherweise als die einmal in 10 000 Jahren vorkommende Flut oder das Zwei- bis Dreifache einer alle 200 Jahre vorkommenden Flut definiert.
vermutlich stärkstes Erdbeben (PME)	probable maximum earthquake (PME)	Eine geotechnische Kenngröße, die bestimmt wird durch das größte, an dem Standort aufgezeichnete Erdbeben, durch das größte aufgezeichnete Erdbeben an einem Standort in ähnlicher Lage, für die historische Daten verfügbar sind, oder ein einmal in 10 000 Jahren vorkommendes Erdbeben, das aufgrund früherer Erdbeben in der Region statistisch vorhergesagt werden kann.
Versatz/ Versatzmaterial	backfill	Rückfüllung von Material in den Hohlraum/die Hohlräume einer abgebauten Erzlagerstätte. Als Versatzmaterial können taubes Gestein oder Aufbereitungsrückstände aus der Erzaufbereitungsanlage verwendet werden. In den

		meisten Fällen wird der Versatz zur Verfüllung abgebauter Bereiche benutzt, um ▪ die Bodenstabilität zu sichern, ▪ Bodensenkungen und -setzungen zu verhindern bzw. zu verringern, ▪ das Hangende abzustützen, damit weitere Bereiche der Erzlagerstätte bei gleichzeitiger Gewährleistung der Sicherheit abgebaut werden können, ▪ eine Alternative zur Aufhaldung über Tage zu schaffen ▪ und die Bewetterung zu verbessern.
Verwitterung	weathering	Prozesse, im Verlaufe derer Partikel, Gestein und Minerale altern, wenn sie der Oberflächentemperatur und Druck sowie atmosphärischen Bestandteilen, wie Luft, Wasser und biologischen Aktivitäten, ausgesetzt werden.
Vordamm	starter dam	Der Vordamm wird vor Beginn der bergbaulichen Aktivitäten errichtet und dient als Ausgangspunkt für den Bau des eigentlichen Dammes des Absetzbeckens.
W	**W**	
Walzenmühle	roll crusher	Eine Art Nachbrecher, der aus einem schweren Rahmen besteht, an dem zwei Walzen angebracht sind. Diese werden angetrieben, so dass sie sich gegenläufig aufeinander zu drehen. Das von oben zugeführte Gestein gelangt zwischen die sich bewegenden Walzen, wird gebrochen und unten wieder ausgestoßen.
Wasserbilanz	water balance	Prozess, in dem das gesamte in das Absetzbecken eingeleitete und ihm entnommene Wasser sowie die Wasserverluste in einer solchen Weise definiert und beschrieben werden, dass damit die Nettozunahme bzw. der Nettoverlust an Wasser im Absetzbecken bestimmt werden kann.
(Grund-) Wasserspiegel	water table	Gibt das Niveau an, bei dem der Druck des fließfähigen Mediums gleich dem atmosphärischen Druck ist, d.h. der Wasserspiegel, der die Zone des Sickerwassers von der gesättigten Zone trennt (wo der Druck des fließfähigen Mediums größer als Null ist).
Wiedergewinnung (Wiederherstellung, Rekultivierung)	reclamation (rehabilitation, recultivation)	Wiederherstellung von Land und der Umweltwerte eines bisher bergbaulich genutzten Standortes nach dem Abbau des Bodenschatzes. Zur Wiederur- und -nutzbarmachung kommt es üblicherweise, sobald der Bodenschatz von einer Lagerstätte abgebaut wurde. Der Prozess beinhaltet die Wiederherstellung der Landschaft in ihrem etwaigen ursprünglichen Zustand durch Wiederaufbringung des Mutterbodens und durch Anpflanzen einer regionaltypischen Vegetation.
Windsichter	air classifier	Maschinelle Anlage zum Abscheiden von Feinstaub (mit einer Partikelgröße von <0.05 mm) vom trockenen Inputmaterial (<10 mm) bzw. Gerät zur Abscheidung feiner und grobkörniger Feststoffe aus einem Luftstrom.
X	**X**	
Y	**Y**	
Z	**Z**	
Zementmörtelabdic	grout curtain	Ein Bereich, in dem Zementmörtel (pumpfähige

htung		Zementschlämme) injiziert worden ist, um eine Barriere um eine Ausgrabung herum oder unter einem Damm zu bilden, so dass kein Grundwasser durchsickern bzw. -fließen kann
Zerkleinerung	comminution	Zerkleinerung eines Erzes durch Brechen oder Mahlen auf eine solche Korngröße, dass als Zerkleinerungsprodukt eine Mischung aus relativ reinen Mineralpartikeln und Gangart entsteht. Um ein relativ reines Konzentrat zu erzeugen, ist es notwendig, das Erz fein genug zu mahlen, um die gewünschten Minerale freizusetzen.
Zukunftstechno-logien	emerging techniques	Techniken mit einem inhärenten Potenzial zur Verbesserung der Umweltbilanz, die jedoch noch nicht kommerziell eingesetzt und genutzt werden, sondern sich noch im Forschungs- und Entwicklungsstadium befinden.
zurückgewonnenes Wasser	reclaim water	das aus TMFs, Wasseraufbereitungsanlagen oder Aufbereitungsanlagen für Minerale zur Wiederverwendung in einer Aufbereitungsanlage zurückgewonnene Wasser
Cyanidlaugung	cyanidation, cyanide leaching	Methode zur Gewinnung von Gold oder Silber aus gebrochenem oder gemahlenem Erz durch Auflösung in einer schwachen Lösung aus typischerweise Natrium-, aber auch aus Kalium- oder Kalziumcyanid. Die Edelmetalle werden dann ▪ entweder durch Ausfällen auf Zinkstaub (Merrill-Crowe-Verfahren) ▪ oder durch Adsorption auf Aktivkohle in einem Reaktor im CIL-Verfahren (carbon in leach) bzw. im CIP-Verfahren (carbon in pulp) aus der wertstoffhaltigen Lösung gewonnen.

Reagenzien (Prüfmittel):

Kurzform	**Vollname**
	Sammler:
SIBX	Natriumisobutylxanthat
SIPX	Natriumisopropylxanthat
SEX	Natriumäthylxanthogenat
PAX	Kaliumamylxanthat
DTP	Dithiophosphat
	Schaumbildner/Schäummittel:
MIBC	Methylisobutylkarbinol (Methylisoamylalkohol)
	Passivierendes Mittel:
CMC	Karboxymethylzellulose

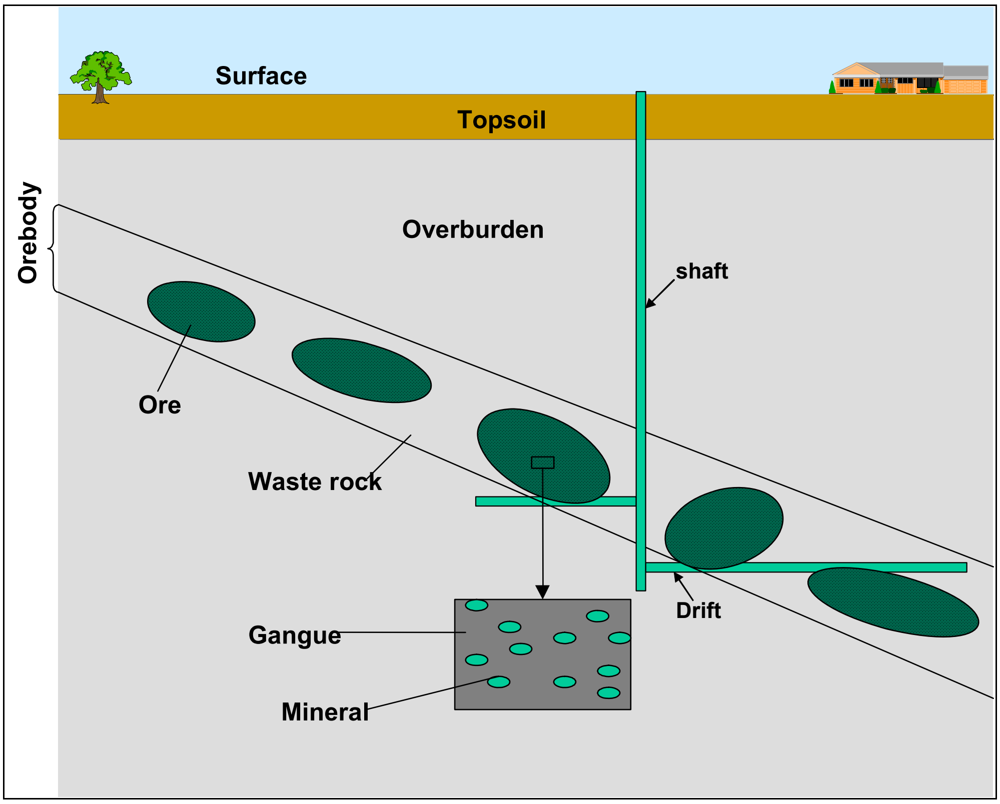

Abb. G1: Schematische Darstellung einer Erzlagerstätte

[Legende zu Abb. G1:]

Surface	Landoberfläche	Orebody	Erzlagerstätte
Topsoil	Mutterboden	Overburden	Abraum
Shaft	Schacht	Ore	Erz
Waste rock	taubes Gestein	Drift	Stollen
Gangue	Gangart	Mineral	Mineral(ien)

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- http://imcg.wr.usgs.gov/dmmrt/
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2. LISTE DER EU-MITGLIEDSSTAATEN (EU der 15)

Kurzname	Vollname	Abkürzung	Währung[2]	Währung ISO-Code[3]
Österreich	Republik Österreich	A	Euro	EUR
Belgien	Königreich Belgien	B	Euro	EUR
Deutschland	Bundesrepublik Deutschland	D	Euro	EUR
Dänemark	Königreich Dänemark	DK	Dänische Krone (Pl.: Kronen)	DKK
Spanien	Königreich Spanien	E	Euro	EUR
Griechenland	Hellenische Republik	EL	Euro	EUR
Frankreich	Französische Republik	F	Euro	EUR
Finnland	Republik Finnland	FIN	Euro	EUR
Italien	Italienische Republik	I	Euro	EUR
Irland	Irland	IRL	Euro	EUR
Luxemburg	Großherzogtum Luxemburg	L	Euro	EUR
Niederlande	Königreich der Niederlande	NL	Euro	EUR
Portugal	Portugiesische Republik	P	Euro	EUR
Schweden	Königreich Schweden	S	Schwedische Krone (Pl.: Kronen)	SEK
Vereinigtes Königreich	Vereinigtes Königreich von Großbritannien und Nordirland	UK	Pfund Sterling	GBP

1. Ehemalige Währungen (vor dem Euro)
   - Österreich – Österreichische Schilling (ATS)
   - Belgien – Belgische Franc (BEF)
   - Deutschland – Deutsche Mark (DEM)
   - Spanien – Spanische Peseta (ESP)
   - Griechenland – Griechische Drachme. Pl.: Drachmen (GRD)
   - Frankreich – Französischer Franc (FRF)
   - Finnland - Finnmark (FIM)
   - Italien – Italienische Lira (ITL)
   - Irland – Irisches Pfund (IEP)
   - Luxemburg – Luxemburgischer Franc (LUF)
   - Niederlande – Niederländischer Gulden (NLG)
   - Portugal – Portugiesischer Escudo (PTE)
2. ISO 4217, wie durch das Secretariat-General empfohlen (SEC(96) 1820).
3. Länderliste (Stand 26.06.2002)

3. NEUE MITGLIEDSSTAATEN (2004)

Kurzname	Vollname	LAND ISO Code[1]	Währung	Währung ISO-Code[2]
Zypern	Republik Zypern	CY	Zyprisches Pfund	CYP
Tschechische Republik	Tschechische Republik	CZ	Tschechische Krone	CZK
Estland	Republik Estland	EE	Estnische Krone	EEK
Ungarn	Republik Ungarn	HU	Forint	HUF
Lettland	Republik Lettland	LV	Lat	LVL
Litauen	Republik Litauen	LT	Litas	LTL
Malta	Republik Malta	MT	Maltesische Lira	MTL
Polen	Republik Polen	PL	Zloty	PLN
Slowakei	Slowakische Republik	SK	Slowakische Krone	SKK
Slowenien	Republik Slowenien	SI	Tolar	SIT

4. WEITERE LÄNDER

Kurzname	Vollname	LAND ISO Code[1]	Währung	Währung ISO-Code[2]
Australien	Australischer Bund	AU	Australischer Dollar	AUD
Bulgarien	Republik Bulgarien	BG	Lev	BGN
Kanada	Kanada	CA	Kanadischer Dollar	CAD
Island	Republik Island	IS	Isländische Krone	ISK
Japan	Japan	JP	Yen	JPY
Neuseeland	Neuseeland	NZ	Neuseeländischer Dollar	NZD
Norwegen	Königreich Norwegen	NO	Norwegische Krone	NOK
Rumänien	Rumänien	RO	Rumänischer Lei	ROL
Russland	Russische Föderation	RU	Neuer Rubel/ Russischer Rubel	RUB; RUR
Schweiz	Schweizerische Eidgenossenschaft	CH	Schweizer Franken	CHF
Türkei	Republik Türkei	TR	Türkische Lira	TRL
Vereinigte Staaten	Vereinigte Staaten von Amerika	US	US-Dollar	USD
1. ISO 3166 2. ISO 4217				

5. LISTE CHEMISCHER ELEMENTE

NAME	SYMBOL	NAME	SYMBOL
Actinium	Ac	Hydrargyrum (Quecksilber)	Hg
Aluminium	Al	Molybdaenum (Molybdän)	Mo
Americium	Am	Neodymium (Neodym)	Nd
Stibium (Antimon)	Sb	Neon	Ne
Argon	Ar	Neptunium	Np
Arsenicum (Arsen)	As	Niccolum (Nickel)	Ni
Astaton	At	Niobium	Nb
Barium	Ba	Nitrogenium (Stickstoff)	N
Berkelium	Bk	Nobelium	No
Beryllium	Be	Osmium	Os
Bismutum (Wismut)	Bi	Oxygenium (Sauerstoff)	O
Boron (Bor)	B	Palladium	Pd
Bromum (Brom)	Br	Phosphorus (Phosphor)	P
Cadmium (Kadmium)	Cd	Platinum (Platin)	Pt
Calcium	Ca	Plutonium	Pu
Californium	Cf	Polonium	Po
Carboneum (Kohlenstoff)	C	Kalium	K
Cerium (Zer, Cer)	Ce	Praseodymium	Pr
Zäsium	Cs	Promethium	Pm
Chlorum (Chlor)	Cl	Protactinium	Pa
Chromium (Chrom)	Cr	Radium	Ra
Cobaltum (Kobalt)	Co	Radon	Rn
Cuprum (Kupfer)	Cu	Rhenium	Re
Curium	Cm	Rhodium	Rh
Dysprosium	Dy	Rubidium	Rb
Einsteinium	Es	Ruthenium (Ruthen)	Ru
Erbium	Er	Rutherfordium	Rf
Europium	Eu	Samarium	Sm
Fermium	Fm	Scandium	Sc
Fluorum (Fluor)	F	Selenium (Selen)	Se
Francium	Fr	Silicium	Si
Gadolinium	Gd	Argentum (Silber)	Ag
Gallium	Ga	Natrium	Na
Germanium	Ge	Strontium	Sr
Gold	Au	Sulfur (Schwefel)	S
Hafnium	Hf	Tantalum (Tantal)	Ta
Helium	He	Technetium	Tc
Holmium	Ho	Tellurium (Tellur)	Te
Hydrogenium (Wasserstoff)	H	Terbium	Tb
Indium	In	Thallium	Tl
Jodum (Jod)	I	Thorium	Th
Iridium	Ir	Thulium	Tm
Iron	Fe	Stannum (Zinn)	Sn
Krypton	Kr	Titanium (Titan)	Ti
Lanthanum (Lanthan)	La	Wolfram	W
Lawrencium	Lr	Uranium (Uran)	U
Plumbum (Blei)	Pb	Vanadium (Vanadin)	V
Lithium	Li	Xenon	Xe
Lutetium	Lu	Ytterbium	Yb
Magnesium	Mg	Yttrium	Y
Manganium (Mangan)	Mn	Zincum (Zink)	Zn
Mendlevium	Md	Zirkonium	Zr

6. DEZIMALE VIELFACHE UND TEILE VON SI-EINHEITEN

Symbol	Vorsatz	Term	Zahl
Y	Yotta	10^{24}	1 000 000 000 000 000 000 000 000
Z	Zeta	10^{21}	1 000 000 000 000 000 000 000
E	Exa	10^{18}	1 000 000 000 000 000 000
P	Peta	10^{15}	1 000 000 000 000 000
T	Tera	10^{12}	1 000 000 000 000
G	Giga	10^{9}	1 000 000 000
M	Mega	10^{6}	1 000 000
k	Kilo	10^{3}	1000
h	Hekto	10^{2}	100
da	Deka	10^{1}	10
-----	-----	1 Einheit	1
d	Dezi	10^{-1}	0,1
c	Zenti	10^{-2}	0,01
m	Milli	10^{-3}	0,001
µ	Mikro	10^{-6}	0,000 001
n	Nano	10^{-9}	0,000 000 001
p	Piko	10^{-12}	0,000 000 000 001
f	Femto	10^{-15}	0,000 000 000 000 001
a	Atto	10^{-18}	0,000 000 000 000 000 001
z	Zepto	10^{-21}	0,000 000 000 000 000 000 001
y	Yocto	10^{-24}	0,000 000 000 000 000 000 000 001

ANNEXES

ANNEX 1

Cyanide chemistry

This section provides a brief overview of the chemistry of cyanide. Cyanide chemistry is complex, and those seeking more detailed information should consult the list of reference materials found at www.cyanidecode.org.

Cyanide Species

The term cyanide refers to a singularly charged anion consisting of one carbon atom and one nitrogen atom joined with a triple bond, CN. The most toxic form of cyanide is free cyanide, which includes the cyanide anion itself and hydrogen cyanide, HCN, either in the gaseous or aqueous phase. At a pH of 9.3 - 9.5, CN and HCN are in equilibrium, with equal amounts of each present. At a pH of 11, over 99 % of the cyanide remains in solution as CN, while at pH 7, over 99 % of the cyanide will exist as HCN. Although HCN is highly soluble in water, its solubility decreases with increased temperature and under highly saline conditions. Both HCN gas and liquid are colourless and have the odour of bitter almonds, although not all individuals can detect the odour.

Cyanide is very reactive, forming simple salts with alkali earth cations and ionic complexes of varying strengths with numerous metals. The stability of these salts is dependent on the cation and the pH. The salts of sodium, potassium and calcium are highly soluble in water, and since they readily dissolve to form free cyanide, they are quite toxic themselves. Operations typically receive cyanide as solid or dissolved NaCN or $Ca(CN)_2$. Weak or moderately stable complexes such as those of cadmium, copper and zinc are classified as weak-acid dissociable (WAD), with equal concentrations of the complex and of its component metal and cyanide ions existing at a pH of approximately 4.0. Although metal-cyanide complexes themselves are less toxic than free cyanide, their dissociation releases free cyanide. Even in the neutral pH range of most surface water, WAD metal-cyanide complexes are sufficiently soluble so as to be environmentally significant.

The differing stabilities of various cyanide salts and complexes under varying pH conditions results in different potential environmental impacts and interactions with regard to their acute or chronic effects, attenuation and re-release. Cyanide forms complexes with gold, mercury, cobalt and iron that are very stable under mildly acidic conditions. However, both ferro- and ferricyanides release free cyanide when exposed to direct ultraviolet light in the presence of water.

Cyanide-metal species also form complexes with alkali or metalliferous cations, such as potassium ferricyanide ($K_3Fe(CN)_6$) or copper ferricyanide $Cu_3(Fe(CN)_6)_2$. The solubility of these complexes varies with the metal cyanide and the cation. Nearly all alkali salts of iron cyanates are very soluble, and if one of these double salts does dissociate to the cation and the cyanide-metal complex, the complex itself may then further dissociate to produce free cyanide. Heavy metal salts of iron cyanides form insoluble precipitates.

The cyanide ion also combines with sulphur to form thiocyanate, SCN. Thiocyanate dissociates under weakly acidic conditions, but is typically not considered to be a WAD species because it has similar complexing properties as cyanide itself. Thiocyanate is chemically and biologically oxidised to carbonate, sulphate and ammonia.

The oxidation of cyanide, either by natural processes or from treatment of effluents containing cyanide, can produce cyanate, OCN. Cyanate is less toxic than HCN, and readily hydrolyses to ammonia and carbon dioxide.

Cyanidation
The process of extracting gold from ore with cyanide is called cyanidation. The reaction, known as Elsner's equation, is:

$$4\ Au + 8\ NaCN + O_2 + 2\ H_2O \Leftrightarrow 4\ NaAu(CN)_2^- + 4\ NaOH$$

Although the affinity of cyanide for gold is such that it is extracted preferentially, cyanide will also form complexes with other metals from the ore, including copper, iron and zinc. The formation of strongly bound complexes such as those with iron and copper will tie up cyanide that would otherwise be available to dissolve gold.

Copper cyanides are moderately stable, and their formation can be a cause of both operational and environmental concerns. High copper concentrations in the ore increase costs and lower recovery efficiencies by requiring higher cyanide application rates to compensate for the reagent that complexes with copper rather than gold. The process water or tailings from such an operation can have significantly higher cyanide concentrations than would otherwise be present in the absence of copper.

Cyanidation is also adversely affected by the presence of free sulphur or sulphide minerals in the ore. Cyanide will preferentially leach sulphide minerals, and will react with sulphur to produce thiocyanate. These reactions will also enhance the oxidation of reduced sulphur species, lowering the solution pH and volatising HCN.

Sampling and analytical methods from the CN Code
This text provides general background information on the sampling and analysis of the various forms of cyanide in aqueous samples at gold mining operations. It is not intended to be an all-inclusive reference for cyanide sampling or analysis.

General information
This text places emphasis on proven and reliable methods used globally for the monitoring of process solutions and environmental compliance at gold mining operations. Other analytical procedures do exist for the measurement of cyanide that are capable of generating acceptable results; these alternative procedures can be substituted for the traditional methods included in this document.
The mining industry, regulators and most service laboratories generally use the following guidelines for cyanide species.

Free Cyanide (CN_F)
Only hydrogen cyanide and the cyanide ion in solution can be classed as "free" cyanide. The proportions of HCN and CN^- in solution are determined according to their equilibrium equation; this is influenced by the solution pH. For the analysis of free cyanide, the methods used:

- should not alter the stability of weaker cyanide complexes, as they may otherwise be included in the free cyanide result
- should be clear of interferences due to the presence of high concentrations of more stable cyanide complexes or other cyanide forms. If not, the interference must be quantified and allowed for in the result.

Weak Acid Dissociable Cyanide (WAD - CN)
Unlike the definition of "free cyanide" which identifies the specific cyanide species being measured, WAD cyanide refers to those cyanide species measured by specific analytical techniques. WAD cyanide includes those cyanide species liberated at moderate pH of 4.5, such as HCN(aq) and CN^-, the majority of Cu, Cd, Ni, Zn, Ag complexes and others with similar low dissociation constants. For the analysis of WAD cyanide, the methods used:

- should be free from interferences due to the presence of high concentrations of more stable cyanide complexes or other cyanide forms. If not, the interference must be quantified and allowed for in the result.

Total Cyanide (CN_T)
This measurement of cyanide includes all free cyanide, all dissociable cyanide complexes and all strong metal cyanide including ferro-cyanide $Fe(CN)_6^{-4}$, ferri-cyanide $Fe(CN)_6^{-3}$, and portions of hexacyano cobaltate $Co(CN)_6^{-3}$, as well as those of gold and platinum. Only the related or derived compounds cyanate (CNO^-) and thiocyanate (SCN^-) are excluded from the definition of total cyanide. For the analysis of total cyanide, the method used:

- must be shown to be capable of quantitatively determining all stable complexes of cyanide, including the cobalt cyanide complex. If methods determine other analytes as well (e.g. SCN^-), those analytes need to be determined separately and allowed for in the total result.

Sampling
The importance of sampling and sample handling, prior to delivery to the laboratory, is summarised by the following statement.

The results of analysis can be no better than the sample on which it was performed.

While the taking of either aqueous or solid samples may appear easy, the collection of correct samples, both in terms of location and with respect to the analytes to be monitored, is fraught with difficulties. Any sampling must have as its aim the collection of a representative portion of the substance to be analysed. When this portion is presented for analysis, the parameters to be determined must be present in the same concentration and chemical or biological form as found in the original environment from which the portion was removed.

Samples representative of a site, or of a portion of a site, provide information that is often extrapolated to include the whole area under investigation. This is true whether the entity being sampled is a contaminated section of land, surface water, an industrial outfall, or a drum containing waste material. Therefore, samples must be representative of the specific entity being sampled, but not necessarily representative of the entire area of which it is part.

The overall objectives of a sampling programme must be considered in the development of the sampling plan. Sampling may be performed for one of several purposes:

- to determine the maximum, minimum and average values for a near steady state stream, with the aim of monitoring compliance versus set specifications (process control, environmental criteria). Such data can illustrate the likelihood and magnitude of occurring non-compliance provided enough data points have been analysed from the samples. Process, residue, and effluent stream analyses could have this type of objective. Even aquifer sampling (boreholes) would fit this description. Often the relative mass-flows have to be known for proper data integration
- to determine the maximum, minimum and average values from the analysis of "batch streams" such as treated backfill portions or detoxified waste batches, which usually require a minimum of one data point per batch to insure a representative sample. The major objective remains one of compliance and/or verification of effective management procedures for the batch streams involved.

Additionally the following issues need to be taken into consideration:

1. non-steady state events following a cyclic pattern are often influenced by several parameters, and these parameters in themselves may also be susceptible to cyclic changes. In other words, these confounding factors create a complex situation that requires careful analysis and planning to obtain a representative sample
2. the cycle periods have to be known along with many other factors of influence in the "system". A typical example would be the sampling of tailings surface liquid (or solids), decant liquids or return dam bulk liquid. All of these "sample populations" undergo massive cyclic fluctuations through influence of chemical and physical changes from process management tailings surface events and seasonal climatic conditions
3. it will be apparent that the cycle periods are not in any way synchronised and hence seemingly random data might be obtained. An objective for such sampling campaigns could be the establishment of a predictions database based on the understanding of the fundamental principles. This means that a complete, non-biased sampling effort across the longest cycle needs to be performed at least once. Alternatively, once such principles are known, selected samples taken at certain times could be analysed for monitoring purposes.

While many sampling strategies may be developed, the main, basic approaches to sampling are depicted in the following table.

Approach	Number of samples	Relative bias potential	Basis of site selection
Judgemental	Small	Very large	Prior history, visual assessment, and/or technical judgment
Systematic	Large	Small	Consistent grid or pattern
Random	Very large	Very small	Simple random selection

Table Annex 1.1: Basic sampling approaches

Sample Preservation

Once samples are removed from their natural environment, chemical or biological reactions can occur which change the composition of the sample, so it is best to analyse the sample as quickly as possible. Preservation of the sample will keep the parameter of interest in the same form as it was prior to the removal from its surroundings. No single preservation technique will preserve all parameters, so each parameter of interest must be considered and preserved specifically.

While most soil samples require exclusion of light, air and warmth to preserve the integrity of the sample, aqueous samples require a more concerted effort. Samples of aqueous cyanide species are potentially very reactive and toxic, so safety precautions such as gloves and protective clothing must be rigorously observed. Due to their reactivity, sample solutions must be tested on site prior to cyanide analysis to preserve them against the main interfering substances, oxidising matter and sulphides.

The presence of oxidising matter is detected by potassium iodide/starch test papers. Whereby a drop of the sample is placed on a moist test paper strip; a blue colouration of the test paper indicates the presence of sufficient oxidising matter to potentially react with the cyanide present during transport. Oxidising agents must be reduced prior to sending the sample to the laboratory.

Procedure for removing the oxidising matter:

1. remove and retain any solids by decantation or pressure filtration
2. add sodium arsenite and mix. About 0.1 g/l is usually sufficient
3. retest, and if test strip is discolored, retreat as per Step 2
4. return solids to sample solution and raise pH to 12 by adding 1 - 2 pellets of solid sodium hydroxide.

The presence of sulphides is indicated by lead acetate paper turning black, whereby a drop of the sample is placed on the test paper previously moistened with a drop of acetic acid and if the paper darkens, sulphides are indicated. Sulphides are removed by reaction with lead carbonate.
Procedure for removing the sulphides
1. remove any solids by decantation or pressure filtration and hold
2. add lead carbonate (about 0.1 g/l) and mix
3. remove formed lead sulphide by pressure filtration and discard PbS precipitate
4. retest sample solution. If test strip is discolored, retreat as in Steps 2 and 3
5. return solids to sample and raise pH to 12 with solid sodium hydroxide.

Samples should be stored in a dark place at about 4 ºC, such as in an esky (cool box) during transport and then refrigerated at the laboratory. Soil samples for cyanide analysis (in cores or jars) must be wrapped in dark plastic and kept cool at 4 ºC without further treatment.

Transport and storage

Once correctly preserved and packaged, samples should be sealed and each container (bottle or jar) individually placed in a sealed plastic bag. All samples should then be packed in an esky (cool box with some ice bricks) to keep them cool during transport. Shipment to the analytical laboratory should occur as soon as practical by overnight truck or airfreight courier. It is essential that the sampling protocol be recorded and a chain of custody included with the shipment to allow tracking prior to and during storage and analysis.

Analytical Procedures

A quality laboratory with necessary technician experience can achieve good results with many different methods. The modified automated SFAA method using the McLeod microstill may be the method of choice for the most advanced laboratories, however global uniformity, availability and cost factors indicate that the analytical methods listed as “Primary” in the following table may be used.

Analyte	Method	Comments
Free cyanide	$AgNO_3$titration	Preferred method For process solutions primarily above 1 mg/l LQL1: 1 mg/l HCN(aq), CN-, Zn(CN)x, parts of Cu(CN)4
	$AgNO_3$titration with potentiometric endpoint determination	Alternate method Precise method of endpoint determination Measures same species as primary method
	Micro diffusion of HCN from static sample into NaOH (ASTM D4282)	Alternate method Close to "free cyanide"
	Ion Selective Electrode	Alternate method Close to "free cyanide"
	Direct colo-rimetry	Alternate method HCN(aq), CN-, Zn(CN)x, parts of Cu(CN)4 + ?
	Amperometric determination	Alternate method Measures same species as primary method
WAD cyanide	Manual distillation pH 4.5 + potentiometric or colorimetric finish (ISO/DIS 6703/2, DIN 38405 Part 13.2: 1981-02)	Preferred method LQL1: 0.05 mg/l HCN(aq), CN-, Zn/Cd/Cu/Ni/Ag(CN)x Better results than ASTM method in presence of high copper concentration
	Amenable to chlorination (CN Total - non-chlorinatable part) (ASTM D2036-B, US-EPA 9010)	Alternate method Measures same species as primary method
	SFIA in-line micro-distillation pH 4.5 + colo-rimetric finish (ASTM D4374)	Alternate method Measures same species as primary method
	FIA In-line ligand exchange + amperometric finish (US-EPA OIA-1677)	Alternate method Measures same species as primary method
	Picric Acid, Colo-rimetric determination	Alternate method Measures same species as primary method
Total cyanide	Manual batch distillation + titration/potentiometric or colorimetric finish (ISO/DIS 6703/1, DIN 38405 Part 13.1: 1981-02)	Preferred method LQL[1]: 0.10 mg/l HCN(aq), CN^-, Zn/Cd/Cu/Ni/Ag/Fe$(CN)_x$, parts of Au/Co/Pt/Pd$(CN)_x$
	SFIA, in-line UV irradiation, micro-distillation + colorimetric finish (ASTM D4374)	Alternate method Measures same species as primary method
1: LQL, Lower Quantitation Level, is defined as about 3 times Detection Level or 10 times the Standard Deviation at near blank level.		

Table Annex 1.2: Primary and alternate analytical methods

For these primary methods, the table also provides a Lower Quantification Level, representing the concentration that all laboratories should be able to reliably determine. Laboratories with a proven record of working with alternative methods, such as those based on automated standard methods, should be encouraged to continue with those methods but should establish crossreferences for each site by applying the suggested methods.

To insure that the mine receives quality analytical service, the chosen laboratory must:

- use experienced to staff perform the analyses
- be certified by the respective national accreditation body for all analytical methods
- have sound quality control procedures in place
- be able to prove the quality of their data by participation in proficiency tests.

Trained analysts and supervisory staff with an expertise in cyanide chemistry methods are critical to consistent and reliable results, as they will be aware of the potential interferences inherent in each method.

The preferred methods for analytical determination of the different types of cyanide are briefly summarised below:

Free cyanide
The preferred method for the analytical determination of free cyanide is silver nitrate titration. Silver ions are added to the solution to complex the free cyanide ions. When all free cyanide is consumed as silver cyanide complex, the excess silver ions indicate the endpoint of the titration. The analytical equipment used for the titration is rather simple. To accurately determine the cyanide concentration, a normalised silver nitrate solution is dosed with a manual or automatic burette, which should be capable of measuring volumes to an accuracy of better than 0.005 ml. Several techniques can be used for the endpoint determination. The easiest possibility is to use an indicator such as potassium iodide or rhodanine that changes colour upon the appearance of free silver ions. It is important that the first colour change is used as the endpoint indication because the silver ions tend to liberate cyanide ions from other complexes, which leads to a disappearance of the colour. The potentiometric endpoint detection is a more accurate way to determine the endpoint as a more easily identified peak signal is produced.

'Weak Acid Dissociable' cyanide (WAD-CN)
The preferred analytical method to determine weak acid dissociable cyanide is the distillation method according to ASTM or ISO/DIS. These methods create chemical conditions which allow the WAD-CN to be liberated as dissolved hydrogen cyanide gas which is then carried in an air stream to a caustic soda absorption where the WAD-CN appears as CNF. As the hydrogen cyanide is adsorbed in a much smaller volume than the original sample solution, the CNF concentration to be analysed is typically at least 10 times higher than the original WAD-CN concentration in the sample solution. The CN_F concentration in the distillation product sample is then determined using silver nitrate titration as described above.

The methods according to ASTM and ISO/DIS are similar. However, the results from ISO/DIS method are more accurate than those from the ASTM method for samples containing a high concentration of copper cyanide.

Total Cyanide
The preferred analytical method to determine total cyanide is the distillation method according to ASTM or ISO/DIS. The applied method is in principle very similar to the distillation method described for weak acid dissociable cyanide. However, strongly acidic conditions and elevated temperatures are required to liberate the cyanide ion from the stable cyanide complexes such as ferri- and ferrocyanides.

Complete descriptions of these analytical procedures can be found in the following references:
DIN 38405-13: 1981-02, German Standard Methods for the Analysis of Water, Waste Water and Sludge - Anions (Group D) - Determination of Cyanides (D13), German Standards (DIN Normen, Beuth Verlag GmbH, Burggrafenstr. 6, 10787 Berlin/Germany).

South African Water Quality Guidelines, Volumes 1 - 7, Department of Water Affairs and Forestry, 1996.

Standard Methods For The Examination Of Waters and Wastewater, APHA-AWWA-WEF, 20th Edition, Washington DC, 1998.

Water Quality - Determination of Cyanide - Part 1: Determination of Total Cyanide ISO/DIS 6703/1, International Organization of Standardization.

Water Quality - Determination of Cyanide - Part 2: Determination of Easy Liberated Cyanide ISO/DIS 6703/2, International Organization of Standardization.

USEPA "Methods and Guidance for Analysis of Water", United States Environmental Protection Agency (USEPA), June 1999.

ANNEX 2

In this annex, several dam failures are described. The descriptions provide useful suggestions for the safe management of tailings management facilities.

The Aitik dam failure

On the night of September 8, 2000, a dam failure occurred at the Aitik site. The failure took place in a section of the dam which separated the tailings pond from the downstream clarification pond. The event led to the discharge of 2.5 Mm^3 of water from the tailings pond section into the clarification pond. The subsequent rise of the water level in the clarification pond, 1.3 m, caused a discharge of 1.5 Mm^3 clarified water into the receiving streams. This resulted in a temporary rise of the suspended solids content in the river system downstream.

The event occurred in spite of manual and automatic monitoring systems in accordance with a recently developed OSM-manual.

Two theories to explain this event have been developed:

According to the first theory, the filter layers in the dam were not performing properly, so that the pore pressure within the pond increased causing erosion or sliding failures in the support fill, eventually resulting in a complete dam failure. Detrimental leaks with elevated pore pressure as a result may also have occurred:

- along the discharge culvert through the dam
- through the narrow upper section of the impermeable core
- underneath the sheet pile at the culvert
- through cracks in the bedrock
- from the right side of the breach.

According to the second theory, inner erosion occurred along the discharge culvert, possibly in combination with openings in the joints between culvert elements and/or collapse of the culvert. Break-in of water and soil to the culvert, probably caused a sinkhole in the dam with flow directly from the pond into the culvert. The failure escalated and ended with overtopping of the dam and, eventually, a complete failure.

It will probably not be possible to fully eliminate one theory in favour of the other, mainly because the dam was completely eroded away. The operator, however, interprets the results as leakage in connection with the culvert being the main cause of the failure. The reasons for this conclusion are:

- the culvert was founded on gravel, 16 – 50 mm, and at the last reconstruction covered by filter cloth. Leaks through joints and/or in the gravel have occurred which is proven by investigations after the accident, when tailings were found, that did originate from the accident
- the culvert was not equipped with a longitudinal reinforcement, and could therefore not withstand tension.

In addition, some conditions indicate that high pore pressure was not the main cause:

- as late as the evening before the failure, no visible leaks could be observed along the toe of dam E-F extension. This indicates that the failure occurred rapidly
- calculations show that before the accident the dam had a safety factor exceeding 1 even at increased pore pressure.

The operator has therefore concluded that leaks and/or collapse of the culvert were the most likely causes of the accident. However, it cannot be ruled out, that also increased pore pressure caused by deficient filter function may have contributed to the accident. [63, Base metals group, 2002]

One consequence of this accident is that a more stable culvert has been constructed, which will in the future be replaced by an overflow system built around the dam in natural rock. Future dams will be built with two filter layers, are coarse and the other fine.

The Aznalcollar dam failure

The Aznalcollar event has been described in many places. Here only the main causes for the failure and the conclusions will be described.

On the night between April 24 and 25, 1998, a 600 m section of the downstream dam of the tailings pond suddenly slid up to 60 m. The slide created a breach in the dam through which water and tailings were flushed out. In a few hours, 5.5 Mm^3 of acid and metal rich water flowed out of the pond. The amount of tailings that was spilled has been estimated to be between 1.3 and 1.9 million tonnes. Due to the fine particle size of the tailings (d_{80} <45 µm) they were easily transported in suspension with the flood wave.

The direct cause of the accident was a fault in the marls 14 m below the ground surface (see figure below). This fault was the result of surplus pressure in the interstitial water of the clay due to the weight of the dam and the tailings deposit.

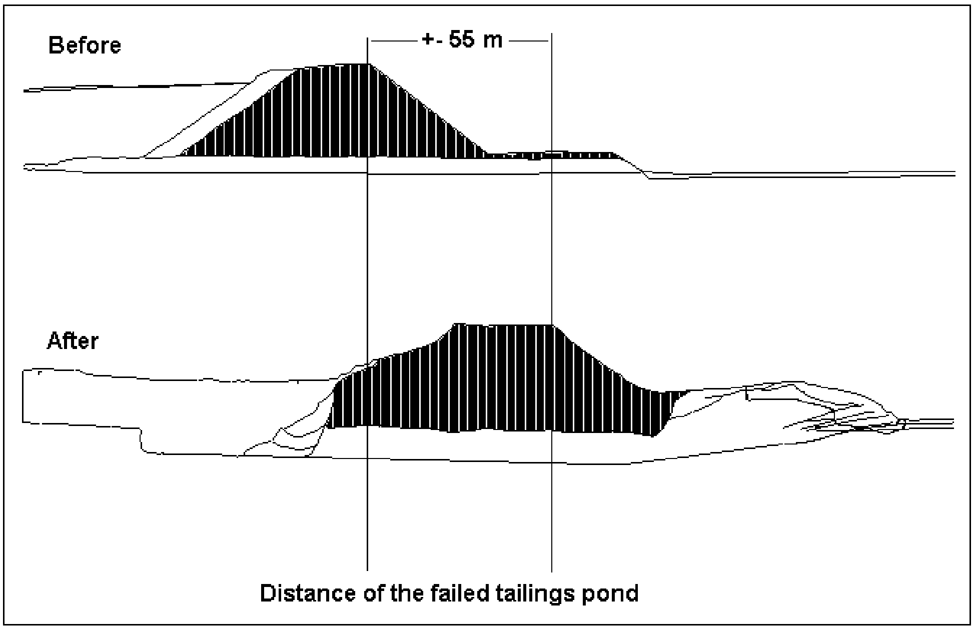

Figure Annex 2.1: Cross-section of the tailings dam
[68, Eriksson, 2000]

One of the conclusions from investigations of this incident was that a good baseline study, conducted before the accident, would have significantly facilitated the evaluation of the effects of the accident [68, Eriksson, 2000]. Another conclusion that can be drawn is that a close and thorough investigation of a TMF's foundation needs to be prepared and evaluated prior to the dam construction.

Further examples

There were many examples of impoundments retained by dams built by the upstream method at the copper mines in Chile. That country is subject to earthquakes and failures were not uncommon. A well known example is that of the El Cobre old dam, built to a height of 33 m between 1930 and 1963, with a downstream slope of 1 on 1.2 to 1.4. Two years after its construction stopped, the area was struck by the 1965 La Ligua earthquake, which occurred during daylight. Eye witnesses said dust clouds came up from the dam, obstructing it from view as it failed, releasing the liquefied tailings to flow down the valley, engulfing the miners' village and continuing for a further 5 km. Many lives were lost. This failure and others in Chile, have been described by Dobry and Alvarez (1967).

The release of dust is typical of the failure of dry loess slopes and is caused by the volume reduction on shearing that occurs in loose particulate materials. Air from the voids is expelled, carrying dust with it. The downstream face of the dam had clearly been relatively dry before it disintegrated allowing the release of all the unconsolidated tailings slurry.

In Japan, the Mochikoshi impoundment was being built in a hollow near the top of a hill to store gold mine tailings and was retained by three tailings dams. These were being built by the upstream method from very sound starter dams made from the local volcanic soil. The dams were raised by building dykes from the volcanic soil placed on the beach and compacted. The impoundment was subjected to a ground acceleration of 0.25g from the Iso-Oshima earthquake of magnitude 7.0 that occurred on 14th January 1978. The highest of the three dams failed during the main shock, releasing 80000 m^3 of tailings contaminated by sodium cyanide through a breach 73 m wide and 14 m deep. The tailings flowed 30 km and into the Pacific Ocean. The second highest dam failed next day, 5 hrs. 20 mins. after an aftershock, releasing a further two to three thousand cubic m of tailings through a breach 55 m wide and 12 m deep. These and other earthquake related failures led to recommendations that the downstream method of construction should be used in earthquake areas, rather than the upstream method. In this method, coarse material, possibly from cyclones, is placed on the downstream part of the dam where effective drainage measures can be employed and the fill can be compacted. Alternatively the dam can be built from borrowed fill, as with water retaining dams.

Stava

At 12.23 on 19th July 1985, two tailings dams, one above the other and both built by the upstream method, collapsed. A total quantity of 190000 m^3 of tailings slurry was released and flowed, initially at a speed of 30 km/hr down into the narrow, steep sided valley of the Rio Stava, demolishing much of the nearby small village of Stava and continued, at increasing speed, estimated to have been 60 km/hr to another small town, Tesero about 4 km downstream, at its junction with the Avisio River in northern Italy. (The only surviving eye-witness, a holiday maker, had the horrifying experience of watching the disaster from the hillside and saw the hotel where his family were taking lunch being swept away by the torrent of tailings.) The damage caused by tailings is very much more than would be caused by the same flood of water, because the tailings are so heavy. Where water could flood a building, tailings can push it over and sweep it along with the flow. This failure caused 269 deaths.

The tailings dams were for a fluoride mine that was begun in 1962 and were sited on a side slope of 1 on 8. The decant was in the form of a concrete culvert laid up the sloping floor, with coverable openings about every 0.5 m vertical rise. Water from the pond decanted into the openings, which were covered, one by one, as the level of the tailings rose. The lower dam was built by the upstream method to a slope of 1 on 1.23. When it reached a height of 19 m, the second dam was begun at the upstream end of the impoundment and built to a slope of 1 on 1.43. When it reached a height of 19 m, further planning consent was required. This was given on the condition that a 5 m wide berm was constructed at that level and permission given for the dam to be built to a height of 35 m. Construction continued at the same slope of 1 on 1.43 and the failure occurred when it was 29 m high. The cause is thought to be due to a combination of blockage and leakage from the culvert under the toe of the upper dam, thereby raising the phreatic surface sufficiently to cause a rotational slip. Six months before the failure, a local slide occurred in the lower portion of the upper dam on its right side, in the area where the decant pipes pass underneath the dam, due to freezing the service pipe during a period of intense frost, according to Berti et al (1988). For the next three months, water was observed seeping from the area of the slide. A month before the failure, the decant pipe underneath the lower impoundment fractured allowing the free water and liquid mud from the pond to escape towards the Stava river, creating a crater above the point of fracture. A bypass pipe had to be installed through the top portion of the lower dam, and the broken decant pipe blocked to restore use of the system. During this operation the water level in the upper impoundment was lowered as far as possible, then just four days before the failure, both ponds were filled and put back into normal operation. 53 minutes before the failure a power line crossing below the impoundments failed, then only 8 minutes before, a second power line failed. The tailings from the failure reached Tesero about 4 km distance, within a period of 5 to 6 minutes. As a result of this failure, the strict Italian law governing the design and construction of water retaining dams, according to Capuzzo (1990), is being extended to include tailings dams.

Merriespruit

The Virginia No 15 tailings dam had been built by the 'paddock' method that is used extensively in South Africa's gold mining industry. It was a long dam encircling and retaining an impoundment of 154 ha holding 260x106 m^3 of gold mine tailings containing cyanide and iron pyrite. The foundation soil was clay and drainage was required under the dams. General experience was that drains were often blocked by iron oxides and other residue. The impoundment formed one of several similar impoundments of the Harmony Gold Mine near Virginia in the Orange Free State. The suburb of Merriespruit containing about 250 houses had been built near the mine in 1956. Virginia No 15 lagoon was begun in 1974 and a straight northern section of the dam nearest the suburb was placed only 300 m from the nearest houses. Dam construction and filling of the lagoon continued until March 1993, when the section of the dam closest to the houses was 31 m high.
The summer of 1993/4 in the Orange Free State had been particularly wet and on the night of Tuesday 22nd February 1994 there were violent thunderstorms over Virginia and a cloudburst when 40 mm of rain fell in a very short time. The water level in the lagoon rose due to direct catchment: there was no stream or other natural source of water that came into the lagoon, which while operational, had a launder system that removed the transportation water decanted from the tailings slurry that had been delivered into the impoundment. During the early evening at about 19.00, water was found running down the streets and through gardens and an eye witness saw water going over the crest of the dam above the houses. The mining company and contractor were informed, but when their representatives reached the site it was already dark. One of the contractor's men rushed to the decants and found water lapping the top rings but not flowing into the decants. He removed several rings to try to get the water flowing, but the main pool was next to the north dam crest with no direct connection to the decants. At the same time, another contractor's man was near the downstream toe of the dam, and saw blocks of tailings toppling from a recently constructed buttress that had been built against a weak part of the dam. An attempt was made to raise the alarm, but before anyone had been contacted, there was a loud bang, followed by a wave of liquefied tailings that rushed from the impoundment into the town.

A breach 50 m wide formed through the dam, releasing 2.5x106 m^3 of tailings that flowed for a distance of 1960 m, covering an area of 520x103 m^2. The flow passed through the suburb where the power of the very heavy liquefied tailings demolished everything in its path, houses, walls, street furniture and cars, carrying people and furniture with it. (According to newspaper reports, people already in bed at about 21.00 hours when the mudflow struck, found themselves floating in their beds against the ceiling. 400 survivors spent the night in the Virginia Community Hall, a kilometre away. Hetta Williamson said that her husband had gone back in daylight to their former home and found nothing but the foundations.) It is remarkable that only 17 people were killed.

Apparently this north section of the enclosing dam had been showing signs of distress for several years, with water seeping and causing sloughing near the toe. A drained buttress constructed from compacted tailings had been built against a 90 m long section, but continued sloughing had caused the mine to stop putting the normal flow of tailings into the impoundment more than a year before the failure, i.e. the impoundment had been closed. At that time the freeboard was, according to the contractor, a respectable 1 m. But sloughing at the toe

continued, and construction of the buttress was continued. Not long before the failure, slips had occurred in the lower downstream slope just above the buttress. In fact, although the placement of tailings had been stopped, waste water containing some tailings continued to be placed and the water overflowed into the two decants. Unfortunately there formed a sufficient deposit of further tailings to cut off the decants and cause the main pool of water to move towards the crest of the north part of the dam, leaving a freeboard of only 0.3 m, and water was still being pumped into the impoundment from the mill on the night of the failure. Evidence of what had been going on since supposed closure was provided by satellite photography. A Landsat satellite passed over the area every 16 days and the infrared images revealed the positions of the tailings and the water pool.
Government Mining Regulations that had come into force in 1976, required a minimum freeboard of 0.5 m to be maintained at all times for this type of impoundment, to enable a 1 in 100 year rainstorm to be safely accommodated without causing overtopping. Evidence of the level of tailings in the Virginia No 15 lagoon showed that the tailings had been brought up to within 15 cms of crest level prior to abandoning this storage in March 1993. Had the government regulations required inspection of the dam, particularly at closure, the very small freeboard would have been noticed and a further raising of the dam crest enforced to prevent overtopping in the event of a maximum probable precipitation.

Baia Mare
The expanding city of Baia Mare in Romania was beginning to encroach on old mining areas where there were disused impoundments of tailings. Removal of these impoundments and their retaining tailings dams would both release valuable land for city development and allow extraction of remaining metals from the old tailings. The scheme at Baia Mare involved construction of a new impoundment and a new efficient processing plant that would accept tailings removed from the old impoundments. Initially three were to be reworked and pipelines were laid out to transmit water from the new impoundment to be used for powerful jets that would cut into the old tailings, producing a slurry that would go to the new processing plant for extraction of remaining metals, with the tailings from it flowing to the new impoundment. The system used the same water going round and round with no interference with the environment.

The site for the new impoundment, well away from the city, was on almost level ground, with its main axis 1.5 km long, sloping down only 7 m from NE to SW with a width of about 0.6 km, as indicated by Fig.3. An outer perimeter bank 2 m high with 1 on 2 side slopes, as shown by Fig.4, was built from old tailings, and the whole area of about 90 ha, lined by HDPE sheet, anchored into the crest of the perimeter bank. Drainage was installed to collect any seepage, that would be pumped back so that there should be no escape of contaminated water into the environment. About 10 m inside the perimeter, starter dams were built, also with 1 on 2 side slopes, to heights of about 5.5 m along the SW lower edge of the impoundment, tapering down to 2 m height about half way along the sides, with the remainder around the NE end of the impoundment, about 2 m high. Cyclones mounted along the crest of the SW starter dam and part way along the side starter dams accepted the tailings piped from the new processing plant, discharging the coarser fraction on to the downstream side to fill the space to the parameter dam, and raise the whole dam, with the main volume of fine tailings slurry being discharged into the impoundment. Collected water was discharged into the central decant, drained through a 450 mm diameter outfall pipe embedded under the HDPE liner and pumped back to operate the monitoring jets in the first of the old impoundments, 6½ km away, and close to the city. Cyanide was used in the new processing plant for the extraction of gold, so that the tailings and water in the new impoundment contained considerable amounts of cyanide. No water should leak from the pipework circuit, although the water used in the cutting jets flowed over the unlined floor of the old impoundment where it could soak into the ground. First discharge into the impoundment was in March 1999, and during the summer everything worked well, particularly during June, July and August when the average evaporation was 142 mm per month, although the delivered tailings did not contain quite as much coarse material as had been envisaged and the rate of height increase of the dams was lower than intended.
During the winter, however, conditions became greatly changed. The temperature fell below zero on 20 December and remained low during most of January, freezing the cyclones and

producing a layer of ice over the impoundment, which became covered by snow. Tailings from the processing plant was warm enough to keep the operation working, but there was no further height increase for the dams because the cyclones were out of action. Precipitation during September to January averaged 71 mm per month and fell as rain and snow on both the whole area of the impoundment but also on the old tailings impoundments that were being worked. This extra water was stored in the impoundment causing the level to rise under the now thick layer of ice and snow.

On 27th January there was a marked change in the weather. The temperature rose above zero and there was a fall of 37 mm of rain. The ice and snow covering melted and the dams, half way along the sides of the impoundment, where they were only starter height, were lower than the developing water level. At 22.00 hours on 30th January 2000, a section overtopped, washing out a breach 25 m long that allowed the escape of about 100000 m^3 of heavily contaminated water that flowed following the natural slope of the area, towards the river Lapus. This in turn fed into the rivers Somes, Tisa and Danube, eventually discharging into the Black Sea. A very large number of fish were killed with serious consequences for the fishing industry for a time. The Hungarian authorities estimated the total fish kill to have been in excess of one thousand tonnes. Water intakes from the rivers had to be closed until the plume of toxic contaminate had passed and for some time afterwards until the purity of the water could be confirmed. The cyanide plume was measurable at the Danube delta, four weeks later and 2000 km from the spill source.

The concept of a closed system in which none of the process water should escape into the environment should have been excellent, with the new tailings impoundment completely lined with plastic sheeting and provision for the collection of any seepage. Unfortunately no provision had been made for the additional water that would accrue from precipitation, nor had the problems of working at low temperatures been addressed. The scheme was one that could have worked well in the hot and dry conditions found in some parts of Australia and South Africa.

ANNEX 3

Example of aspects covered by a baseline study

The following sub-sections provide an example of a baseline study for a tailings pond recently performed in Europe [25, Lisheen, 1995]. These studies have become a standard procedure and are often a legal requirement. They are necessary as a point of reference in order to quantify the impact of an operation.

Archaeology and local history/cultural patrimony

This section of the baseline study investigates whether archaeological findings can be expected based on historical information. It gives answers to the questions whether any important findings may be inhibited or even promoted by a new development. From the perspective of the operator repeated archaeological findings can significantly slow down the development of a site. There may be public concern about the loss of archaeologically significant sites but many authorities will accept their preservation by professional excavation and published record.

Socio-economic

The level of employment is considered and trends for the future are discussed briefly. Major sources of employment are listed. Hence predictions can be made of the prosperity of the investigated area for the future.

Health

The typical lifestyle (e.g. eating habits) in the region is examined, mortality rates are listed and compared to "average" conditions (e.g. country/world average) and the possible reasons for abnormal findings discussed.

Infrastructure

This section describes the road, railway, shipping and airway situation. Furthermore the access to water and electricity is described. This section may also mention the waste collection in the area.

Traffic

Local traffic situation is quantified. Traffic flow compared to other areas or country average may be investigated.

Climatology

Data such as annual rainfall, prevailing winds (strength and predominant direction), humidity and air and soil temperatures are presented. If useful, these figures may be compared to other regions.

Air quality

The results of a baseline sampling programme are presented here. The levels measured are shown and the origins determined. The values measured include total dust and metals.

Geology
This part describes the geology of the mineral deposit and the nearby area. It usually includes:

- deposit depth
- strata dip
- mineral assemblage
- dimensions of the deposit
- mineable resources
- description of topsoil, overburden, bed and waste-rock.

Landscape
The countryside of the study area is described here. Will the site be in the mountains or on flat pastureland? Are there many trees and/or hedges? The visual impact of the new development may be mentioned in this context.

Ecology
This section describes, e.g.:

- the soil of the area
- woodlands of interest
- species in the habitats studied
- diversity of herbs and woods
- plant species
- diversity of birds and mammals
- any special ecological designations near the site.

Noise
Day and night time noise levels, measured for the study are often listed as 12 hour averages.

Soils and soil suitability
The overall quality of soils must be investigated in the area possibly effected by the development and compared to other areas. The field survey includes soil characteristics, quality and suitability for grassland and crop and stock production. In the UK, the assessment of soils is undertaken by a recognised system for evaluating the characteristics and quality of soils, i.e. the Agricultural Land Classification System.

Soil and herbage sampling
This section investigates the soil fertility status of the area. It includes the measurements of trace elements (i.e. magnesium, copper, molybenum, manganese, cobalt, zinc, lead, cadmium) and other nutrient elements such as phosphorus, nitrogen, potassium, calcium, sulphur, iodine, selenium. These values are compared to other areas and anomalies are analysed. Special attention is dedicated to establish baseline levels of any constituents that may be altered by a future mining operation.

Crop and animal production
Surrounding farms are examined for the productivity of their field crop and grassland. Their types numbers and conditions of livestock are examined at the same time. A recognised methodology is needed for evaluating the cropping and livestock systems in the area which takes account of variables such as farm management skills, level of fixed costs, inputs and agricultural land classification. Comparison of yields without taking these other factors into account would be misleading.

Soil moisture
The purpose of this part of the study is to address the concern that dewatering of a mine may adversely affect the growth of crops and other aboveground vegetation including scrubs and trees. To achieve this a survey on the movement of water in soils and the possible relationship between the depth of the watertable and the soil water balance may be carried out.

Veterinary
Within an appropriate area herds are surveyed for trace elements and other important elements in blood silage and milk. Also a 12 month animal health survey may be part of this section.

Hydrogeology
All factors influencing groundwater flow should be mentioned here, including aquifer/aquitard systems, faults and fault zones as well as any other geological features influencing groundwater flow. The existence of hydraulic barriers and hydraulic conduits should be discussed. Other issues mentioned in this section may be groundwater levels and transmissivity (hydraulic conductivity x thickness).

Groundwater quality
This part of the study analyses the groundwater chemistry. The water is typically sampled in wells and piezometers. If contaminated groundwater is found, the possible source of the contamination should be identified (e.g., agricultural practices, other industrial activities, etc.).

Surface water quality
The results from a baseline surface water sampling programme are presented here. The sampling points should be selected to provide a baseline for what part of the catchment area may potentially be affected by discharges from the proposed development.

Typically, the overall quality of the water is discussed as well as the levels of organic pollution, nutrient and trace metal levels. Possible sources of contamination are identified.

Surface water hydrology
In order to determine the assimilation capacity of the receiving waters, flow data of all surface waters potentially influenced by the project are required. “Knowledge of the surface water hydrological characteristics is also important in establishing the recharge and discharge relationship between the rivers/streams and the groundwater system.” [25, Lisheen, 1995]

Fisheries, fish population and spawning
A fish stock assessment in representative sections of the main watercourses within the survey area is part of this section. This assessment includes tissue analysis and density measurements of the existing species. In addition the measures such as the mean redd count (Redds are the nests that mated adult salmon build in the gravel, where they deposit and cover their eggs. A redd count is the number of such nests counted in the river at the end of the spawning season. The number of redds is a good indication of the health of a salmon run) can provided for each of the watercourses as a means of investigating the spawning activities in these rivers.

Surface water flora and macroinvertebrate fauna
Selected plant and macroinvertebrate species may be utilised as indicators of the water quality. To investigate these aquatic ecological surveys and the water chemistry, surveys are carried out. This part of the study should list the flora and macroinvertebrate fauna encountered and the implications of their existence and/or lack of existence.

ANNEX 4

This Annex gives an overview of used characterisation methods for waste-rock and tailings, however, it is not a complete compilation of all existing methods and does not give sufficient guideline on when to use what method. Therefore, it should be regarded as a compilation of which characterisation methods could be relevant in a specific case. Furthermore, it should be regarded as a starting point for further work in order to arrive at a methodology that can be generally applied and accepted within Europe in order to reach a relevant level of characterisation of all waste-rock and tailings.

Characterisation of tailings and waste-rock samples

A summary of methodologies available for the geotechnical and geochemical characterisation of tailings and waste-rock, and for predicting drainage quality, is presented.

Available methodologies for characterisation of tailings and waste-rock samples

Sampling

To ensure the reliable environmental characterisation of tailings and waste-rock and the design of cost-effective remediation and reclamation schemes, appropriate sample collection and preparation procedures need to be defined. These procedures will depend upon whether the programme is related to the following:

- baseline study
- pre-mine planning
- mine operations
- reclamation/closure plan.

Sampling may involve any of the following:

- point samples: These can be either a single grab sample chosen to represent a single waste deposit, or random samples taken from various source points, generally within a predetermined area
- linear samples: Continuous sampling over an interval in a line such as channel samples, profile sampling of overburden, or throughout boreholes as discrete, disturbed, undisturbed or drillcore samples
- panel samples: These are planar samples made up of multiple chips collected from a surface with dimensions
- bulk samples: Sampling of a large mass of material which will be sectioned and split into fractions with samples taken from these various fractions.

Sampling theory and practice is addressed by Pitard (1993), and sampling methodology specifically related to tailings is given in MEND (1989) and Runnels et al. (1997). Sampling guidelines and protocols are beyond the scope of this report and are not discussed further.

Geotechnical parameters

A range of field and laboratory tests is required for the characterisation of tailings and for potential additives in order to derive an understanding of their likely geotechnical behaviour. Physical and geotechnical properties of tailings may be derived from bulk sampling from the mineral process for prediction and control of the deposition process, or from disturbed/undisturbed samples from as-deposited material. The properties will include grain size, moisture content, specific gravity, sedimentation characteristics, in situ and relative density, permeability, plasticity, compressibility, consolidation, shear strength, and stress parameters. Variations in these properties are all known to affect both the geotechnical and geochemical behaviour of tailings and impact on design, stability and drainage of the impoundment as described in the CLOTADAM Green Paper (Knight Piésold, January 2002).

Due to the importance of the geotechnical characteristics of soils in civil engineering and dam design, a number of standard procedures have been developed. Many of these soil testing standard procedures, which include ISO, CEN, national standards and others, are applicable to tailings. In addition, a number of non-standard test procedures are in use for the determination of specific, tailings-related physical and geotechnical parameters.

Geotechnical testing of tailings can be divided into four generic groups:

- individual sample (single) tests
- combined geotechnical tests
- process specific tests, and
- model specific tests.

The normal suite of laboratory geotechnical testing for basic characterisation of tailings is presented below.

Method	Standards operations procedures
Moisture content	BS 1377-2, ASTM D2216
Specific gravity (Particle density)	BS 1377-2, ASTM D854,
Atterberg limits (plastic and liquid limits)	BS 1377-2, ASTM D4318
Soil classification (hydrometer and sieving)	BS 1377
Proctor (compaction) test	BS 1377-4, ASTM-D698, D1558, D558
Dry density	BS 1377-4, ASTM C127-88
Falling head permeability	KH Head: Procedure 10.7.2, BS 1377-6, ASTM D2434, D5084
Oedometer testing	BS 1377-7, ASTM D 3999

Table Annex 4.1: Geotechnical characterisation of tailings – basic characterisation

Full characterisation of the tailings involves a variety of testwork. Standard test methods and guidelines on available procedures are outlined below.

<u>Geotechnical testing</u>
Geotechnical tests to identify individual parameters for tailings include:

- index testing
- desiccation testing
- permeability testing
- strength testing
- consolidation testing
- settlement testing.

Method	Available standards
Index Testing	
Moisture content	BS 1377-2, ASTM D2216
Specific gravity (particle density)	BS 1377-2, ASTM D854
Atterberg limits (Plastic and liquid limits)	BS 1377-2, ASTM D4318
Grain size distribution	BS 1377-2, ASTM D2487, D422
Proctor (compaction) test	BS 1377-4, ASTM-D698, D1558, D558
Dry density	BS 1377-4, ASTM C127-88
Soil classification	BS 1377,
Desiccation testing	
Desiccation test	Mahar and O'Neill (1983)

Permeability testing	
Permeameter	ASTM D5887
Falling head	KH Head: Procedure 10.7.2, BS 1377-6, ASTM D2434, D5084
Strength testing	
Unconfined compressive strength	BS 1377-7, ASTM D2166
Consolidation testing	
Triaxial testing	BS 1377-5, ASTM D2435
Oedometer testing	BS 1377-7, ASTM D 3999
Rowe cell	Sheahan and Watters (1996)
Settlement testing	
Setting velocities	BS 812-103, no ASTM, pipette method
Mudline testing (drained and undrained)	Developed – standard procedures in preparation

Table Annex 4.2: Geotechnical characterisation of tailings - single tests

Mudline tests to determine the settled density of tailings samples on deposition, both in an undrained and drained state, have been developed (Reference Knight Piesold Personal Communication), a standardised procedure for which is being prepared within the Clotadam Project.

Index testing

Index tests provide essential geotechnical characterisation of tailings, and have the advantage of being easy to perform with a quick turn-around time and are thereby relatively cheap. Index properties can provide a rapid tailings classification.

Particle size determination

Tailings are commonly in the top three of the following four grain size distribution groups:

- clay - materials less than 2 µm
- silt - materials lying between 2 µm to 63 µm
- sand - materials lying between 63 µm and 2 mm, and
- gravel materials lying between 2 mm and 60 mm.

Test procedures for grain size analyses typically include a combination of sieves and a hydrometer.

Atterberg Limits (Plastic Limit and Liquid Limit)

Atterberg Limits assess material plasticity and hence provide a fundamental test of the tailings consistency. The water content at which the tailings ceases to act as a liquid and becomes a plastic solid is known as the liquid limit. The definitive method is the Cone Penetration Test, where a sample is tested for a range of moisture contents. From the cone penetration readings obtained, a graph of water content versus penetration is plotted, and the liquid limit is taken as the moisture content that corresponds to a cone penetration of 20 mm.

With decreasing moisture content, the limit at which plastic failure changes to brittle failure is known as the plastic limit. The plasticity index is defined as the range of moisture content over which a material behaves in a plastic manner. Generally, the finer the soil the greater its plasticity index.

Desiccation testing

Air drying tests are carried out on slurry samples to determine the effect of atmospheric evaporation on deposited tailings following initial settlement and removal of the supernatant water. The test, therefore, simulates the sub-aerial deposition of tailings. Continuous monitoring is carried out of the sample weight and volume to define a relationship between the dry density,

moisture content, volume reduction, evaporation and the degree of saturation of the tailings material. The test may include the measurement of shear strength, using a laboratory shear vane. An absolute relationship between the dry density and moisture content exists up to a breakaway point at which the degree of saturation falls below 100 %. At this stage, negative pore pressures develop and act to further consolidate the tailings. At a limiting saturation point, no further bleeding of the material occurs with further drying occurring due to the drawing of water from the voids. At this point, cracking of the sample occurs and hence the final dry density and moisture content are typically calculated by interpolation.
The standard air-drying test is undertaken using a 1 litre container with no underdrainage. For the majority of the samples the lack of underdrainage has no significant effect on the drying rate or final density of the sample. Where tailings samples contain significant amounts of salt in the water, the formation of a salt crust can inhibit drying.

Permeability testing
Standard tests are available for the determination of the Coefficient of Permeability (Hydraulic Conductivity) of a material. These tests provide a measure of the drainage characteristics of tailings.

Strength testing
Strength testing can provide basic characterisation data and also design parameters for consideration in the closure designs of tailings facilities. Consolidated undrained and drained triaxial tests use cylindrical samples.. Samples typically have an aspect ratio of 2:1 and are sealed by a rubber membrane attached by rubber 'O' rings to a base pedestal and a top cap. Pore pressures measurements can be made during testing and consolidation. Undrained Triaxial testing of the samples is typically carried out at multistage cell pressure increments, to determine the shear strength characteristics over a range of effective confining stresses. From a Mohr-Coulomb curve, the fundamental geotechnical parameters can then be determined, i.e. the effective angle of resistance (friction) and the effective cohesion.

Consolidation testing
Consolidation tests are used to assess the behaviour, particularly settlement and drainage characteristics of a material with respect to changes in loading. Test results provide void ratio/log pressure ratios, coefficient of consolidation, coefficient of volume compressibility and swelling pressures. Consolidation parameters are of significance to the operation, water management and closure design of a TMF.

The consolidation of the tailings can be described by two parameters. The first parameter is the coefficient of consolidation (cv) which describes the rate of excess pore pressure dissipation and hence the rate of gain in effective stress of the tailings. This measure of the rate of consolidation implies that higher values mean rapid consolidation. The second parameter is the coefficient of volume compressibility (mv) that is the volume change per unit volume per unit increase in effective stress. The quotient of the two coefficients alongside the material's unit weight can be used as a calculation of the permeability. These two coefficients can also be used alongside other geotechnical parameters to perform settlement and drainage time models using suitable analytical software.
Consolidation is generally carried out in a standard fixed ring oedometer at varying pressure (effective stress) increments. Each pressure increment is double the previous, and maintained for approximately 24 hours. Routine measurements of settlement should be recorded with time during each loading stage. Once settlement ceases or becomes negligible during loading the confining pressure is increased to the next stage. Typically for tailings confining the pressures range from 0.2 kPa to 400 kPa. For low-density samples such as tailings, the Rowe cell or a specially adapted oedometer may be used for consolidation testing. These test cells permit sample placement and testing at an initial solids content approximating the state of the tailings prior to consolidation (determined from the slurry settlement tests).

Settlement tests

Drained and undrained settling tests allow the modelling of the sub-aqueous and sub-aerial phases of the tailings deposition and provide an indication of overall density achievement during placement. The tests indicate not only deposited density, but also the rate of interstitial supernatant release used for water balance modelling purposes.

Undrained settling test

The undrained settling test estimates the density at which the tailings settle in an undrained, sub-aqueous environment. Tests are performed on slurrified tailings placed in a 1 litre graduated cylinder. The rate of settling and change in volume of the tailings as the supernant water bleeds to the surface are recorded. The dry density of the settled voids is calculated once the change in settled volume remains constant.

Drained settling test

The drained settling test provides an indication of the dry density that will be achieved from underdraining the tailings. Tests are carried out in similar fashion to that of an undrained settling test, but the cylinder includes the provision for bottom drainage and the recovery of downward seepage. The rate of settling and change in volume of the tailings is recorded with time, as the supernant water bleeds to the surface and drains from the base. To minimise the development of a vertical gradient across the sample, it is recommended that supernant water is continually decanted from the surface. The dry density of the settled voids is calculated once the change in settled volume remains constant.

Settling velocity

Particle settling velocities of the fine tailings solids (particle sizes <0.074 mm) are determined using the data from the hydrometer portion of the grain size analysis. Alternatively, they may be determined by the measurement of the time it takes for the particle to fall a distance of 500 mm through distilled water. The results may be used for calculating friction losses for design of a tailings slurry pipeline. To determine the tailings delivery pipeline details, the per cent solids in the total product and particulate settling velocity are utilised for the slurry transport analysis.

Geotechnical modelling testwork

A practice adopted occasionally for the modelling of tailings deposits involves the sequential testing of a sample in order to simulate tailings facility conditions. In particular, this can involve a combination of settling air-drying, consolidation and strength tests. The combined test aims to reflect the fact the development of sub-aerially deposited tailings in which any combination of two of the following dewatering processes are always taking place:

- settling
- air drying
- consolidation.

Test methods are outlined above, although the practice of settling and consolidating samples in a combined testing apparatus are not standardised.

Specialist testing

For the design of tailings disposal, a set of additional process specific tests are recognised, including:

- wind tunnel testing
- dewatering testing
- filter leaf testing
- gravity thickening testing.

In the course of modelling tailings ponds, centrifuge testing is also occasionally carried out. Such testing is standardised for soils in ASTM D-425.

Chemical-mineralogical analyses

Chemical analyses

Chemical analyses include methods to analyse tailings and waste-rock samples for
(1) elements and compounds present in minerals which generate and/or neutralise acid
(2) trace metals, and
(3) whole rock constituents which, in conjunction with x-ray diffraction analyses, can be used to quantify mineralogical composition

The procedures to be selected are dependent on the mineralogy of the examined tailings and waste-rock sample.

Sulphur and carbonate analyses

Of particular importance are acid-producing sulphur species and acid-neutralising carbonate species. Acid producing sulphur species include sulphides associated with iron sulphide minerals (usually pyrite and pyrrhotite) and sulphates associated with jarosites, alunites and efflorescent sulphate minerals. Trace metal sulphides will contribute to drainage acidity, if following their oxidation in the presence of water and oxygen, the associated trace metals precipitate as hydroxides, oxides, or carbonates. These minerals are of interest because they can contribute trace metals to drainage. Jarosites and alunite must be distinguished from non-acid-producing sulphate minerals such as gypsum and anhydrite.

Calcium and magnesium carbonate minerals are important in determining the neutralisation capacity of a waste material, because their dissolution can neutralise acid. It is necessary to distinguish these minerals from carbonates of iron and manganese which, under oxidising conditions, will yield no net acid neutralisation.

Sulphur determinations

Existing analytical techniques, such as those using a combustion furnace (e.g. LECO furnace) with a subsequent quantification of the sulphur dioxide evolved, are capable of accurately determining the total sulphur content of the material under study. However, given the different forms in which sulphur can occur in tailings and waste-rock, e.g. sulphide sulphur, elemental sulphur, sulphates etc., and their different potentials for acid production, an analytical scheme to speciate sulphur would be most beneficial for the environmental characterisation of sulphidic tailings and waste-rock. Other sulphur species are often determined by treating the sample to remove a specific sulphur phase. Such a method involves digestion of the sample with sodium carbonate to remove sulphate minerals. Sulphide sulphur is determined as the difference between the total sulphur and the $S(SO_4)$. This procedure presents some limitations depending on the mineralogical composition of the examined tailings and waste-rock. For example, minerals such as orpiment (As2S3) and realgar (AsS) will dissolve to some degree during digestion, leading to underestimation of the sulphide content, while jarosites and alunite may also not completely dissolve in the digestion, leading to an overestimation of the sulphide sulphur.

Carbon determinations

Standard Techniques using a combustion furnace can be also used for the determination of total carbon content (carbon present as carbonate, organic carbon, and graphite). Carbon species are often determined by treating the sample to remove a specific carbon phase, and using a determination of total carbon on the original and treated sample to determine the change in carbon content resulting from the extraction. A method to determine carbonate content, involves heating of the sample at 550 ºC for one hour to drive off organic carbon as carbon dioxide (Lapakko, 2000). The carbonate carbon is estimated as the total carbon in the residue, and tends to be slightly lower than the initial carbonate content, due to some loss of carbonate during pyrolysis. The difference in temperatures at which carbon species decompose can be also used to speciate carbon (Hammack, 1994). Transition metal carbonates (e.g. siderite, $FeCO_3$, and rhodochrosite, $MnCO_3$) decompose, yielding CO_2, in the range of 220 ºC to 520 ºC. Calcite decomposes above 550 ºC whereas, dolomite decomposition occurs at 800 ºC to 900 ºC. A second method to determine carbonate content is referred to as "Acid Insoluble Carbon" (Lapakko, 2000). After analysed for carbon, the sample is digested with hot 20 % HCl, dried, and rinsed three times with distilled water to remove residual chloride, which can interfere with the subsequent analysis for total carbon. The residual solid is analysed for total carbon and assumed to be organic carbon. The carbonate carbon content is the difference between the initial total carbon analysis and acid insoluble carbon.

Total major (whole rock), minor and trace metals
Analytical techniques for determining metal concentrations in tailings and waste-rock samples can be generally categorised as non-destructive or destructive. Non-destructive techniques analyse the sample directly, leaving it intact. In contrast, destructive techniques dissolve the sample and the resultant aqueous solution is submitted for analysis by one of several methods.

Non-destructive techniques
Non-destructive techniques include instrumental neutron activation analysis (INAA) and X-ray fluorescence spectrometry (XRF). Wavelength dispersive XRF (WDXRF) is used to determine the contents of elements with atomic numbers less than or equal to 26, generally referred to as major elements or whole rock constituents, although it can be also used for elements of higher atomic numbers. Energy dispersive XRF (EDXRF) is used for the determination of elements with atomic numbers greater than 26, having the additional benefit of being transportable for field use. XRF is the most widely used non-destructive technique.

Destructive techniques
Acid digestion, sintering, and fusion are destructive techniques used to dissolve the samples, with the resultant solution/residue being analysed for the metals under study by one of several techniques.

An aqua regia (hydrochloric and nitric acids) digestion is commonly used to attack sulphides, as well as some oxides and silicates, and to determine the trace metal concentrations. A "near total" low temperature, atmospheric-pressure digestion using a combination of hydrofluoric, hydrochloric, nitric and perchloric acids can also be employed. Sintering and fusion, with subsequent digestion, can solubilise a wider variety of minerals, however, they are generally more appropriate for determination of whole rock components than trace elements. Aqua region digestion is used to determine the maximum concentration of elements that might become available under severe acidic conditions.

The most common methods for analysis of digestates are flame atomic absorption spectroscopy (F-AAS), graphite furnace-atomic absorption spectroscopy (GF-AAS), inductively coupled plasma-atomic emission spectroscopy (ICP-AES), and inductively coupled plasma-mass spectrometry (ICP-MS) (Hall 1995). The first two methods analyse solutions for a single element at a time, whereas with the ICP methods solutions are analysed for multiple components simultaneously.

Mineralogical analyses

The petrographic or mineralogical examination of samples is usually conducted by X-ray diffraction (XRD) techniques and transmitted and reflected light microscopy, often combined with image analysis. More specialised techniques including scanning electron microscopy (SEM) and electron probe microanalysis (EPMA) are also employed, when more detailed analyses of specific mineralogical components are required. Such techniques are particularly useful in the determination of the chemical composition of sulphide oxidation products such as rims, inclusions and amorphous (non-crystalline) species.
Transmitted light microscopy utilises thin (30 μm) sections of samples and reflected light microscopy utilises polished mounted samples. Samples may be prepared from drill-core samples, or from tailings and representative samples of treated material, or from fragmented material such as humidity cell feed and residue samples.

Transmitted light microscopy is used to examine those minerals that transmit light in thin section, and these include most of the gangue or non-metallic minerals that may have a neutralising capability. Reflected light microscopy is used to examine those minerals that do not transmit light in thin section, but reflect light to varying degrees when polished. Such minerals include metal sulphides that may oxidise to generate acid.

Both types of microscopy are used to identify individual mineral grains to determine mineral grain size and size distribution, and to identify mineral grain spatial interrelationships. Grain size, size distribution and grains associations, are often examined, with the assistance of image analysis techniques combined with the above microscopes The reaction products of sulphide oxidation (rimming of grains) are readily observed, as are many other characteristics of mineral grains (such as inclusions) not readily seen by other analytical techniques. These capabilities of microscopic examination are extremely useful in ARD studies of both tailings and waste-rock.

Metal partitioning

The concentration of a trace metal in a tailings and waste-rock does not necessarily reflect its potential for release to the environment. The phase in which trace metals exist determines how readily available they are for release to the environment. Sequential extractions testwork developed and used primarily for the chemical speciation of metals in soils and sediments (Tessier et al., 1979), can provide useful information about the mode of occurrence and mobility of trace elements. Recently, sequential extractions have been increasingly applied to tailings and waste-rock in order to study the partitioning of metals (Leinz et al., 2000), as well as the retention of mobilised elements by secondary phases (McGregor et al., 1995; Dold, 2001), as these are parameters characteristic of the overall environmental behaviour of the examined material. As an example, a 7-step sequential extraction for tailings and waste-rock reported by Leinz et al. (2000) is given in the following table.

Phase	Sample/ extraction medium	Conditions	Duration
Water-soluble	0.25g sample + 0.25 g silica gel + 25 ml of de-ionised water	Shaking/ ambient temperature	2 h
Ion-exchangeable	Residue of 1st extraction + 25 ml 1M sodium acetate	Shaking/ ambient temperature	1 h
Carbonate	Residue of 2nd extraction + 25 ml 1M sodium acetate buffered with acetic acid, pH: 5.0	Shaking/ ambient temperature	2 h
Fe-$MnOx_{am}$	Residue of 3rd extraction + 25 ml 0.25 M hydroxylamine hydrochloride in 0.25M HCl	Water bath/ 50 ºC	30 min
$FeOx_{cryst}$	Residue of 4th extraction + 25 ml 4 M HCl	Water bath/ 94 ºC	30 min
Sulphide	Residue of 5th extraction + 2 g sodium chlorate +10 ml conc. HCl Separation and dilution to 25 ml with deionised water Residue + 25 ml 4N HNO_3	 Boiling water bath	45 min 40 min
Silicate	Digestion of residue with 10 ml of each conc. HNO_3, $HClO_4$ and HF + 25 ml 4M HCl	220 ºC 100 ºC	 30 min

Table Annex 4.2: Example of a 7-step sequential extraction for tailings and waste-rock.

Acid base accounting

Procedures

Static Acid Base accounting tests are short term (usually measured in hours or days) and relatively low cost tests developed to provide an estimate of a tailings and waste-rock's capacity to produce acid and its capacity to neutralise acid. These tests do not consider parameters such as the actual availability of acid-producing and acid-neutralising minerals and differences between the respective dissolution rates of acid-producing and acid-neutralising minerals. Thus, these tests are commonly used as a screening tool, and their implications are subject to further verification.

The most common of such procedures include:

- Sobek Acid Base Accounting (ABA) procedure (Sobek et al., 1978)
- BC Research Inc. Initial Test procedure (Bruynesteyn and Duncan, 1979)
- Net Acid Production (NAP) test (Coastech Research Inc., 1989)
- Net Acid Generation (NAG) test (Miller et al., 1997)
- modified Acid Base Accounting (ABA) procedure (Lawrence and Wang, 1997)
- Lapakko Neutralisation Potential Test procedure (Lapakko, 1994)
- Peroxide Siderite Correction for Sobek ABA method (Skousen et al., 1997).

Despite individual procedural differences, all these methods involve:

- determination of the Acid Potential (AP) based on the total sulphur or sulphide-S content
- determination of the Neutralisation Potential (NP) including:
  - the reaction of a sample with an inorganic acid of measured quantity
  - the determination of the base equivalency of the acid consumed
  - the conversion of measured quantities to a Neutralising Potential in g/kg or kg/tonne or tonne/1000 tonne calcium carbonate ($CaCO_3$).

Initially the most commonly-used static test was the standard ABA (Sobek et al., 1978). Variations of ABA commonly applied nowadays include the modified ABA (Lawrence and Wang, 1997), NAG test (Miller et al, 1997) and the B.C. Research Initial Test (Bruynesteyn and Duncan, 1979).
As described above, the static tests quantify the acid potential (AP) using either total sulphur or sulphide-sulphur content. The total sulphur content (Standard ABA) overestimates the actual AP of samples containing substantial non acid-producing sulphate minerals (e.g. barite or gypsum). On the other hand, the sulphide-sulphur measurement (modified ABA), will underestimate the actual AP of samples containing substantial amount of acid-producing sulphate minerals (e.g. melanterite or jarosite). Knowledge of the tailings and waste-rock sulphate mineralogy will indicate whether the sulphate minerals present, if any, are acid producing and allow selection of the more appropriate AP quantification. However at present it is accepted that the AP is calculated based on sulphide sulphur.

Different static test methods can produce markedly different neutralisation potential values (NP) for the same sample. Protocol variables which may contribute to these differences include tailings and waste-rock particle size (tailings are typically run "as received"); the type and amount of acid added (i.e. digestion pH), temperature and the endpoint pH of the "back titration", if a back titration is used. The extent to which protocol variables will affect the measured NP is dependent on the sample mineralogy. The conditions and minerals reported to dissolve by various ABA procedures are summarised in the following table. It is noted that carbonates are considered as the most reactive acid neutralising minerals, whereas minerals including plagioclase feldspars, K-feldspar, muscovite and quartz are slow weathering minerals.

The Net Acid Production (NAP) (Coastech Research Inc., 1989) and Net Acid Generation (NAG) (Miller et al., 1997) tests are based on the principle that hydrogen peroxide accelerates the oxidation of iron sulphide minerals. The acid consequently produced dissolves neutralising minerals present, and the net result of the acid production and neutralisation can be measured directly. This test does not require sulphur determinations and is, therefore, more readily conducted in a field laboratory than other static tests. Based on previous studies, the application of NAP to wastes with sulphur content higher than 10 % may underestimate the acid generation potential due to incomplete oxidation (Adam et al., 1997).

Procedure	Acid	Amount of acid added	End pH of acid addition	Test duration	Test temperature	Minerals dissolved
Sobek	Hydrochloric	Determined by fizz test	0.8 - 2.5	Until gas evolution ceases (~3 h)	Elevated (90 °C)	Mineral carbonates Ca-feldspar, pyroxene, olivine (forsterite-fayallite) Some feldspars anorthoclase>orthoclase >albite ferromagnesians – pyroxene hornblende, augite, biotite
BCRI initial	Sulphuric	To reach pH 3.5	3.5	16 - 24 h	Ambient	Ca + Mg carbonates. Possibly chlorite, limonite
Modified ABA	Hydro-chloric	Determined by fizz test	2.0 - 2.5	24 h	Ambient	Ca + Mg carbonates Some Fe carbonate, biotite, chlorite, amphibole olivine (forsterite-fayallite)
Lapakko	Sulphuric	To reach pH 6.0	6.0	Up to 1 week	Ambient	Ca + Mg carbonates
Sobek – siderite correction	Procedure as for Sobek, but with peroxide correction for siderite					Ca + Mg carbonates, excludes Fe+Mn carbonates. Otherwise as per Sobek.

Table Annex 4.3: Most common procedures for acid base accounting

Screening assessment criteria

Two parameters need to be calculated to classify materials in terms of acid drainage generation potential. These are:

- the Net Neutralisation Potential (NNP), which is the difference in value between the neutralisation potential (NP) and the Acid Potential (AP), expressed in kg $CaCO_3$/t of material and
- the neutralisation potential ratio (NPR), which is the ratio of NP value to AP value.

The former is the parameter preferably used for the characterisation of tailings and waste-rock stemming from the Appalachian coal mines, and the latter for Western Canadian metalliferous mines. Materials with sulphide minerals whose net neutralising potential is negative are likely to be an acid drainage source. Exceptions are possible if the sulphide content of material is very low and/or if there are slow dissolving, non-carbonate sources of alkalinity. Based on the NPR values, the Acid-Base Accounting screening criteria recommended by the British Columbia Ministry of Employment and Investment of Canada are given in the following table (Price et al., 1997).

The above guidelines define a “grey zone” for NPR ranging between 1 and 4. The acid drainage potential of materials that fall in the grey zone is considered uncertain and kinetic testwork has to be conducted to further characterise them with regard to acid generation potential. It is noted that the British Columbia guidelines recommend that the neutralisation potential is determined based on the expanded version of the Sobek method (Modified ABA) and acid potential is determined based on the sulphide sulphur content of the samples.

Potential for ARD	NPR	Comments
Likely	<1:1	Likely ARD generating
Possibly	1:1 – 2:1	Possibly ARD if NP is insufficiently reactive or is depleted at a faster rate than sulphides
Low	2:1 – 4:1	Not potentially ARD generating unless significant preferential exposure of sulphides or extremely reactive sulphides in combination with insufficiently reactive NP
None	>4:1	No further testing is required unless material is going to be used as a source for alkalinity

Table Annex 4.4: ARD potential related to neutralisation potential ratio (NPR)

An alternative approach is to use Modified ABA (Lawrence and Wang, 1997) together with the mineralogy of the sample as the basis of a reliable ARD screening programme. Modified ABA has a lower risk of misclassification of the examined waste samples into the wrong category and comprises a cost-effective screening test.

Kinetic tests

Kinetic tests are performed for sulphide tailings and waste-rock that according to static test results are characterised as potentially acid generating or fall in the zone of uncertainty. Kinetic tests can also be used to determine the metal leachability of trace elements of environmental concern. It is required to estimate the acid generation rate and quality of drainage of these materials, information that is considered as critical for the environmentally safe management of tailings and waste-rock. A number of laboratory kinetic tests have been developed with humidity cells, columns and lysimeters (see table below), being the three most commonly used laboratory methods for determining kinetic acid drainage characteristics of drill-core samples, waste-rock and tailings. All kinetic testwork procedures involve two main stages, i.e. subjection of sample to periodic leaching and collection of drainage for analysis.

	Type	Procedure	Comments
1	Humidity cells (ASTM D5744-96)	Sample mass: 1 kg Oxidative wet/ dry cycles Test duration: 20 weeks minimum	• standard procedure • determination of acid generation/neutralisation rates • real conditions may not be simulated.
2	Column test	Operating conditions specific to examined material or disposal site Simulation of oxidizing, reducing environment	• flexible, allowing simulation of field conditions • long duration
3	Lysimeters test	Simulation of field conditions	• no standardised practice • long duration

Table Annex 4.5: Laboratory kinetic tests

The humidity cell is a standard kinetic test (ASTM D5744-96) recommended by the government of British Columbia, Canada for the prediction of the geochemical behaviour of tailings and waste-rock. It is usually referred as an accelerated weathering procedure, since it is designed to accelerate the natural weathering rate of potentially acid generating samples and reduce the length of time for which testwork must be run. A cell 203 mm in height by 102 mm diameter is specified for material 100% passing 6.3 mm (crushed core or waste-rock and coarse tailings) and a cell 102 mm in height by 203 mm diameter is specified for material passing 150 µm (fine tailings). The humidity cell operational procedure is a cyclic one during which the sample is subjected to three days of dry air permeation, three days of humid (water saturated) air permeation and one day of water washing with a fixed volume of water, i.e. 500 ml for 1 kg of sample. The sample mass used is about 1 kg and a minimum test duration of 20 weeks is recommended.

Column testwork may be undertaken to determine the geochemical behaviour of waste-rock and tailings disposed on the surface and exposed to atmospheric weathering (sub-aerial disposal) or disposed underwater cover (sub-aqueous disposal). Unlike the humidity cell procedure, there is little, if any, standardisation of the column testwork procedure, thereby allowing considerable flexibility. This flexibility allows column testwork to be highly site or material specific with regard to material particle size, sample mass and volume, wet/dry cycles, volume of water washing, etc. Columns for sub-aerial and sub-aqueous testwork are typically 76, 102 or 152 mm in diameter and range from about 1 m to more than 3 m in height.

Lysimeters may be also used to determine the acid generation/neutralisation rates of sulphidic tailings and waste-rock and assess the drainage quality. Like the column kinetic test, the lysimeter test allows the simulation of the conditions encountered at the site. Lysimeters have usually larger diameter and smaller height compared to columns. (e.g. 30 or 70 cm in diameter and height 30 to less than 100 cm)

It is noted that a humidity cell will usually determine if a given sample will produce acidity but will not define when the sample will turn acid, or the on-site drainage quality. On the other hand, the column and/ or lysimeter test procedure may simulate field conditions and as a result, may be used to give indications of on-site drainage quality, i.e. they can enable the determination of lower and higher bound. Monitoring parameters in a kinetic test include mass/ volume of leachates, pH, conductivity, redox potential (mV), acidity/ alkalinity, sulphate and dissolved metals.

Presence of soluble salts

Paste pH is a common and simple field test, used to assess the presence of soluble acid salts on tailings and waste-rock. Most methods use a 1:1 weight ratio of distilled water to air dried solids, and pH is measured at the mixture. Sample mass and equilibration time of the water-solids mixture prior to pH measurement vary among methods. The procedure described by MEND (1990) determines pH of a mixture of 10 g sample (-60 mesh) and at least 5 ml distilled water (water addition is adequate to saturate, but not cover, the sample). The Acid Concentration Present test is slightly more complicated but supplies an estimate of acidity present rather than simply pH (Lapakko, 2000). A mixture of 20 g sample (-200 mesh) and 50 ml deionised water is agitated, the initial pH is recorded, and the mixture is titrated to pH 7 with NaOH.

The standard paste pH test was developed by the US EPA, (Method 9045C).

Metal leaching tests

Procedures

Although acid generation has received the most attention for sulphide and coal active and/or abandoned mines, leachable metals comprise potential source of contaminants in tailings and waste-rock drainage. Numerous leaching procedures have been developed worldwide addressing various management scenarios, leaching properties and tailings and waste-rock types. Tests have been developed to account for variability in the ratio of leaching fluid to solid materials, chemical composition of the leaching fluid, testing of monolithic and granular materials, as well as stabilised and solidified materials. A summary of leaching test procedures used in Europe, the US and Canada are given in the following table.

Leaching test methods can be divided into 2 general categories:

- extraction tests, in which leaching takes place with a single specified volume of leaching fluid, and
- dynamic extraction tests, in which the leaching fluid is renewed throughout the test.

Test protocols frequently incorporate a particle size reduction, in order to increase the amount of surface area available for contact, thereby reducing the amount of time required to reach a steady state condition. Examples of extraction tests used for regulatory purposes include:

- US EPA Toxicity Characteristic Leaching Procedure (TCLP, Method 1311)
- British Columbia Special Waste Extraction Procedure, SWEP (MELP, 1992)
- German standard DIN 38414-S4
- French standard AFNOR X 31-210
- Swiss TVA Eluate test
- EN 12457/1-4.

The most commonly used for the last two decades are the TCLP and SWEP tests, which were developed to simulate leaching in sanitary landfills and which involve a leaching with acetic acid. This acid, comprising the decomposition product of organic wastes found in municipal landfills, has a strong capacity to dissolve lead. Given that, in the disposal sites of the mining industry, the co-disposal of tailings and waste-rock with organic wastes, does not normally take place, leachability testing with acetic acid, is not considered as the recommended practice for the characterisation of tailings and waste-rock. Extraction tests using deionised water as the leaching medium, such as DIN 38414-S4, modified SWEP etc. more closely approximate the conditions in a tailings and waste-rock management facility.

Most recently, and within the application of the Landfill Directive (1999/31/EC), the European Standard EN 12457/1-4 was developed, and applied for the classification of waste material accepted for disposal at Landfills (COM(2002) 512 final), also using de-ionised water as the leaching medium.

In dynamic extraction protocols, the leaching fluid is renewed, either continuously or intermittently, to further drive the leaching process. Because the physical integrity of the studied material is usually maintained during the test, and the information is generated as a function of time, dynamic extraction tests provide information about the kinetics of contaminant mobilization. In general, dynamic extraction tests can be categorised as:

- serial batch tests,
- flow-around tests,
- flow-through tests, and
- soxhlet tests.

In a serial batch test, a portion of a crushed, granular sample is mixed with leachant and agitated for a specified time period. At the end of the time period, the leachate is separated and removed, fresh leachant added, and the process repeated until the desired number of leaching periods has been completed. The concentrations of contaminants measured in the serial leachates can provide kinetic information about contaminant dissolution. Examples include the Multiple Extraction Procedure (US EPA Method 1320); the Availability Test (NEN 7341) and Serial Batch Test (NEN 7349) from the Netherlands.

Flow-around tests use either monolithic samples, or samples that are somehow contained. The sample is placed in a test vessel, with space around the sample, and leachant is added so that it flows around the sample. The leachant may be renewed continuously and sampled periodically, or it may be replaced intermittently. In either case, the liquid to solid ratio is expressed as the ratio of volume of leachant to surface area of sample. Examples of flow-around tests include the ANSI 16-1 and the Monolithic Diffusion test (NEN 7345) from the Netherlands.

Flow-through tests differ from flow-around tests in that the leachant flows through the sample rather than around it, thereby simulating conditions in the disposal of tailings and waste-rock. Flow-through tests, such as the kinetic tests used in ARD testwork, are usually constructed as columns or lysimeters, and can be set up to mimic site-specific conditions. These tests, however, pose particular experimental challenges, such as channelling, flow variations caused by the hydraulic conductivity of the waste, clogging of the system by fine particulates, and biological growth in the system. Examples of flow-through tests include:

- Dutch standard column test (NEN 7343)
- ASTM D 4874-95 Column Test and
- Nevada Meteoric Water Mobility Procedure (MWMP), which allow testing for large masses and coarse particle sizes of material.

The above classification of leaching tests is directly related with the operating procedures applied, i.e. extractive, or dynamic. Another way to categorise leaching tests is in relation to their aim of application and practice. In this context, tests can be classified as:

- characterisation tests, aiming to investigate the leaching behaviour of materials under a variety of exposure conditions (typical testing run from a few days to weeks or even a month)
- compliance tests, which are generally of much shorter duration, usually aiming at a direct comparison with threshold values (test duration up to one or two days), and finally
- on-site verification tests, aimed at verifying a previous evaluation of a charge or batch arriving at a processing plant and/or tailings or waste-rock management facility.

The two last categories have been adopted in CEN, the European Standardisation Organisation, as the basis for the development of a standard leach test. As previously noted, the recently developed European Standard EN 12457 (Van der Sloot et al., 1997, EN 2002) is an extraction test proposed for the leaching of granular wastes and sludges with deionised water at a compliance level.

α/α	Organisation/ Country	Standard	Application	Leaching medium	Maximum particle size	Liquid: solid ratio	No of extractions	Test duration
Extraction tests								
1	US EPA	Ep Tox	Classification of wastes in terms of toxicity	Acetic acid 0.04 M, pH:5.0	9.5	16:1	1	24 h
2		TCLP		Acetic acid pH:2.88 or pH: 4.93	9.5 mm	20:1	1	18 h
3		SPLP	Assess impact of wastes	Synthetic acid rain	9.5 mm	20:1	1	18
4	British Columbia	SWEP		Acetic acid, 0.5 N, pH: 5.0	9.5 mm	20:1	1	24 h
5	Special waste regulation	Modified SWEP		Deionised water	9.5 mm	20:1	1	1 h
6	Environment Canada*	ELT	Granular wastes	Deionised water	150 µm	4:1	1	7 days
7	German	DIN 38414 S4	Sludges and sediments	Deionised water	10 mm	10:1	1 or more	24 h
8	France	AFNOR X-31-210	Granular wastes	Deionised water	4 mm	10:1	1	24 h
9	CEN/ TC 292	EN 12457	Granular wastes and sludges	Deionised water	90 % <4 mm	2:1 up to 10:1	1 or 2	24 h
10	Materials characterisation center, 1984	MCC -1P	High-level radioactive waste	Deionised water	Monoliths	Volume/surface area: 10-200 cm	1	Not determined

Table Annex 4.6: Leaching test procedures for wastes
(EPA/625/6-89/022, Van der Sloot et al., 1997)
*** Environment Canada and Alberta Environmental Center (1986)**

(continued)

α/α	Organisation/ Country	Standard	Application	Leaching medium	Max grain size	Liquid: solid ratio	No of extractions	Test duration
Dynamic tests								
13	US EPA	MEP Serial batch test	Granular wastes	α) acetic acid β) synthetic acid rain	9.5 mm	16:1 20:1	1 ≥9	24 h
14		MWEP	Granular wastes/ monoliths	Deionised water	9.5 mm or monolith	10:1	4	18 h per extraction
15	The Netherlands	NEN 7341 Availability test	Dutch waste management Max leachability	Deionised water at a) pH: 7.0 and b) pH:4.0	125 μm	50:1	2	3 h per extraction
16		NEN 7343 Column test	Mineral wastes - Simulate leaching in the short and medium term (<50 years)	Deionised water with HNO_3 at pH:4.0	4 mm	0.1:1 to 10:1	7	21 days
17		NEN 7345	Tank leaching test for monoliths and stabilised wastes	Deionised water	0.1×0.1 ×0.1 m >40 mm	5:1	8	6h to 64 days
18		NEN 7349 Serial batch test	Long-term leaching behaviour of wastes	Deionised water adjusted with HNO_3 at pH:4.0	4 mm	20:1 up to 100:1	5	23 h per extraction
19	Switzerland	TVA-eluate test Serial batch test	Granular and monolithic wastes	Deionised water, CO_2 atmosphere pH:5-6	Not specified	10:1	2	24 h per extraction
20	Sweden	ENA shake test Serial batch test	Mineral wastes-Simulation of initial pore water quality	Deionised water adjusted to pH: 4.0 with H_2SO_4	20 mm	4:1	4	24 h per extraction
21	UK Waste Research Unit	WRU	Waste disposal in inorganic environment or municipal landfill	Deionised water or CH_3COOH at pH:5.0	10 mm	1 to 10 pore volumes	5	2 –80 h
22	The Nordic Countries	Nordtest Serial batch	Granular waste materials	Deionised water with HNO_3 at pH:4.0	90% <4 mm	2:1 up to 50:1	1 or 2 or 3	24 h
23		Nordtest Availability test	Granular waste materials	Deionised water at a) pH: 7.0 and b) pH:4.0	125 μm	100:1	2	3 h per extraction
24		Nordtest Column test	Granular waste materials	Deionised water with HNO_3 at pH:4.0	4 mm	0.1:1 up to 2:1	4-5	20 days
25	ANS 1986	ANS-16.1	Low level/ hazardous wastes	Deionised water	Monoliths	Volume/surface area: 10 cm	11	2 h to 90 days
	Nevada Mining Association	MWMP		Deionised water	5 cm	1:1	1	24 h

Methodology for tailings and waste-rock characterisation

Environmental characterisation of tailings samples

Based on the techniques developed for assessing the environmental behaviour of mining wastes, as described in Section 1, of this Annex one possible methodology for characterisation of tailings and waste-rock is shown in the following figure.

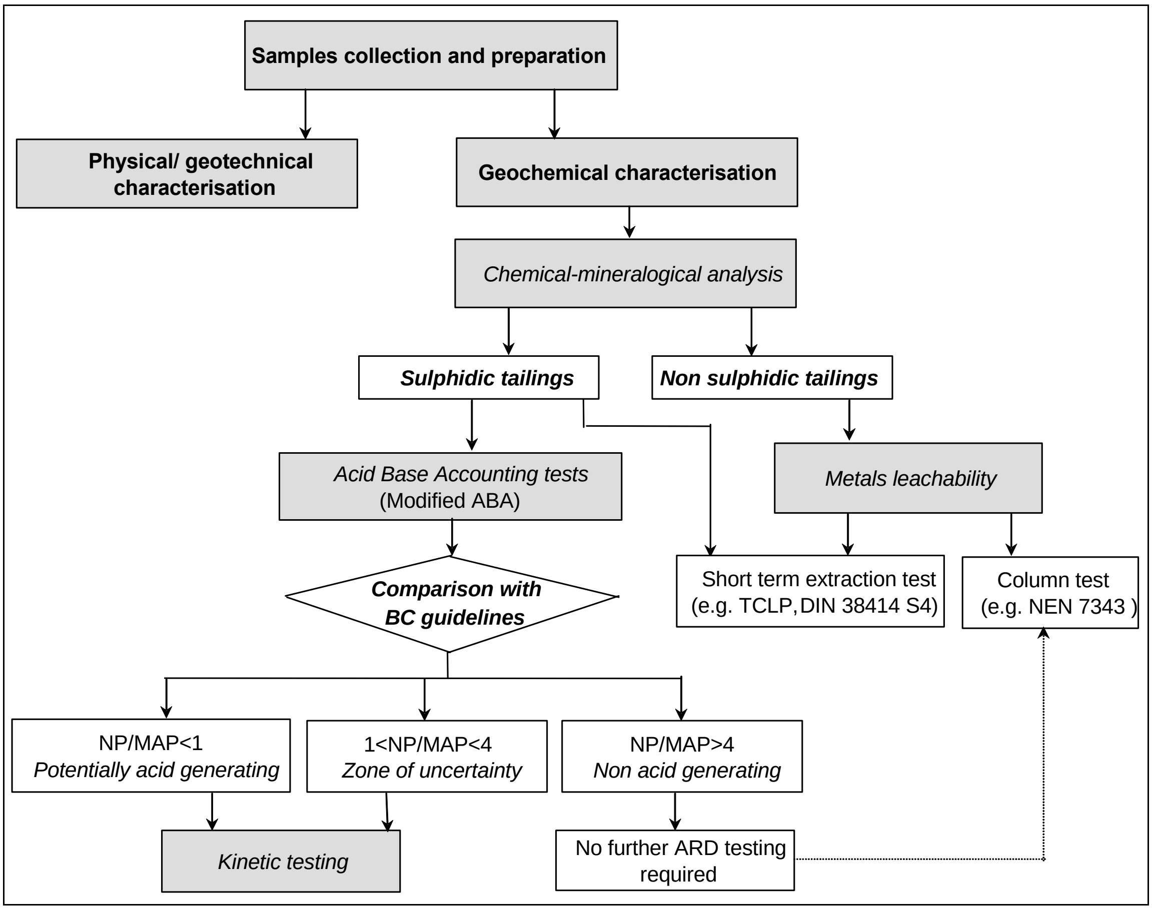

Figure Annex 4.1: Possible methodology for characterisation of tailings and waste-rock

Standard operating procedures

Standard operations procedures (SOPs) describe the way specific tests and methods are performed. These include sampling, sample preparation, calibrations, measurement procedures, and any test that is done on a repetitive basis. "Standard" means that it specifies the way the operation is to be done on each occasion, which may or may not be a procedure developed by a standards organisation. However, when such a standard procedure is available, laboratories, research organisations, and mining industries are advised to consider them since they represent peer judgement and can provide a basis for comparability of data among different user laboratories.

While the use of SOPs may provide a continuity of measurement experience, no methodology should be used without judgement. Its applicability should be reconsidered at each and every use. If used infrequently, it may be necessary for the researcher and/or operator to make a sufficient number of preliminary measurements to demonstrate attainment of statistical control of the measurement process on each occasion.

Standard Operating Procedures (SOPs) for the characterisation of tailings samples are listed in the following 2 tables. The majority of these procedures can be also applied for the characterisation of waste-rock.

Parameter	SOP*	Method
Particle size distribution	BS 1377: Part 2: 1990	Wet/dry sieving method
Particle density	BS 1377: Part 2: 1990	Gas jar method/pycnometer method
Moisture content	BS 1377: Part 2: 1990	Oven drying method
Dry density/moisture content relationship	BS 1377: Part 4: 1990	Compaction method
Consolidation column test	To be specified	
Permeability - triaxial cell - falling head	BS 1377: Part 6: 1990 KP (Appendix A.1.1)	Triaxial cell method Falling head method
*An equivalent to British preferably European standard methodology can be followed.		

Table Annex 4.7: Standard Operating Procedures

Parameter	SOP	Comments
Acid base accounting	Modified ABA (Appendix B.1.1)	Recommended
Soluble salts	1. Paste pH 2. British Columbia Modified Special Waste Extraction Procedure	Recommended Recommended
Leachability 1. Toxicity characteristic leaching procedure (TCLP) 2. Synthetic precipitation leaching procedure (SPLP) 3. Leachability by water 4. Leachability by water 5. Sequential method	 USEPA #1311 USEPA #1312 DIN 38414 S4 EN 12457 Needs standardisation	 Optional Optional Recommended Optional
Kinetic Testing 1. Humidity cells 2. Column protocol	 Modified from Morin and Hutt, 1997and ASTM D5744-96 Developed by GSG	 Column or humidity cell testing selectively applied.
Chemical Analysis 1. Flame atomic absorption spectra.(F-AAS) 2. Graphite-furnace atomic absorption Spectra.(GF-AAS) 3. Inductively coupled plasma-atomic emission-spectroscopy (ICP-AES) 4. Inductively coupled plasma-mass spectroscopy (ICP-MS)		
Mineralogy 1. X-ray fluorescence spectrometry (XRF) 2. X-ray diffraction (XRD) 3. Scanning electron microscopy (SEM) 4. Transmitted light microscopy (TLM)		

Table Annex 4.8: Standard Operating Procedures

Characterisation of additives

For the environmentally safe management of waste-rock and tailings during operation and closure, the application of additives to prevent and mitigate acid and contaminated drainage formation may be required. The characterisation of additives will depend on the type and the specific objectives of additive application. The additives can be grouped in the following categories:

- alkaline materials (e.g. limestone, lime), for the addition of neutralising capacity
- pozzolanic materials (e.g. fly ash), for the addition of neutralising capacity, the modification of the geotechnical properties of disposed wastes/tailings and for the formation of low permeability of covers and barriers
- clays (e.g. bentonite), for the formation of low permeability barriers and covers and
- organic materials (e.g. biological sludge), mainly to facilitate the during the post closure period, or to enhance the maintenance of anaerobic conditions within the material.

Some methods for the characterisation of additives are given in the following table:

Parameter	Method	Alkaline materials	Pozzolanic materials	Clays	Organic materials
Moisture	BS 1377: 2 1990	√	√	√	√
Grain size analysis	BS 1377: 2 1990	√	√	√	√
Swell index	ASTM D 5890	-	-	√	-
Chemical analysis	AAS/ ICP/ Titration/ gravimetric methods	CaO, MgO, Al_2O_3, CO_2, SO_3, SiO_2, Fe, Mn, LOI	Al_2O_3, Fe_2O_3, CaO, MgO, K_2O, Na_2O, TiO_2, SiO_2, SO_3, CO_2, LOI Trace elements content: Pb, Zn, Cd, As, Mn, Ni, Cr		Organic carbon, nitrogen, phosphorus, LOI, heavy metals content
Free calcium oxide content	EN 451-1	√	√	-	-
Mineralogical analysis	XRD/ Optical microscopy	√	√	√	-
Neutralisation potential	Modified ABA	√	√	√	-
Cation exchange capacity	Olphen 1977	-	-	√	√
Metals leachability	TCLP DIN 38414 S4	-	√	√	√

Table Annex 4.9: Some methods for the characterisation of additives

Development of rehabilitation techniques

Physical and geochemical stabilisation

For the development and evaluation of rehabilitation and closure strategies for tailings or waste-rock management facilities, it is general practice for laboratory kinetic tests, similar with those conducted during the environmental characterisation programme, to be performed. For example, the effectiveness of alkaline additives in preventing acid generation from sulphidic tailings and waste-rock can be evaluated with humidity cell tests, columns (ROLCOSMOS, 2001), as well as lysimeters (PRAMID, 1996).

For the development and evaluation of low permeability barrier layers, involving a mixing of the tailings or waste-rock with selected additives, the methodology should include geotechnical and geochemical characterisation of the potential cover system, as shown in the following figure. This methodology, applied previously for sulphidic tailings - fly ash/ bentonite composite cover systems is based on the guidelines given by international environmental agencies for:

- design and evaluation of landfill covers (U.S EPA/625/4-91/025)
- stabilisation/solidification treatment of wastes (U.S EPA 625/6-89/022), and
- prediction and prevention of acidic drainage in a sulphide mine (Environment Australia, 1997).

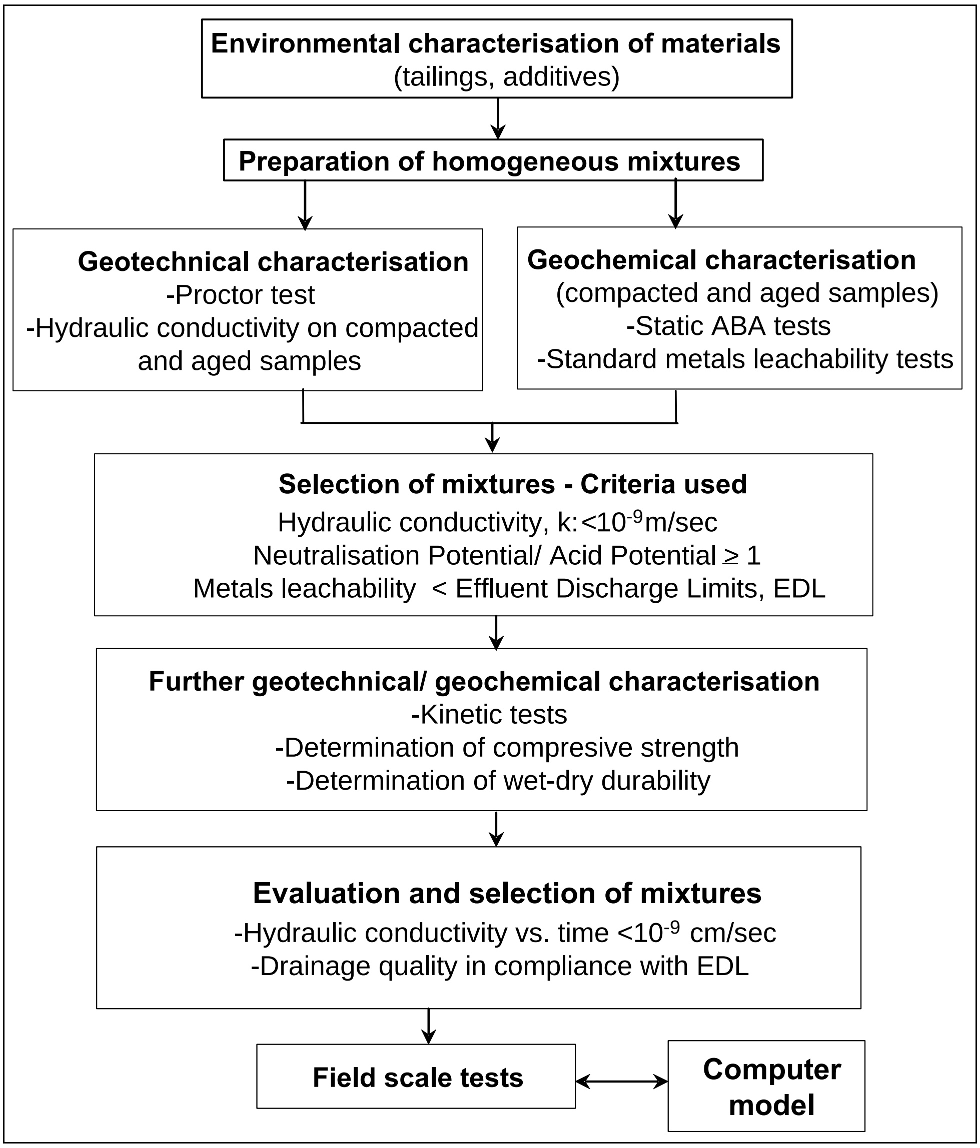

Figure Annex 5.1: Environmental characterisation of materials

Following the physical characterisation of tailings and additives, i.e. moisture, grain size analysis, the geotechnical tests performed with the composite mixtures aim to determine critical parameters used in the development of surface barriers, such as the standard Proctor compaction curve and the hydraulic conductivity. Where additives are employed which exhibit time-dependent behaviour such as bentonite/tailings or cement/tailings mixes, standard procedures for maturing the sample prior to testing are required. Such standardised test procedures are currently in preparation (CLOTADAM 2003).

The moisture vs. dry density relationship (compaction curve) can be determined according to the standard and/ or modified Proctor method (BS 1377 part 4, ASTM D698, D1557, D558). The influence of the maturation of the tailings/additive mixture on compaction delay time and moisture content of the mixtures is normally considered.

Hydraulic conductivity measurements can be conducted on samples compacted at their optimum moisture (OM) and maximum dry density and cured for 7 and/ or 28 days at a relative humidity>90 %, at room temperature. Curing of the samples is very important since it allows the pozzolanic and cementation reactions to proceed, thereby effecting the physical and geochemical stabilisation of material. The hydraulic conductivity of samples compacted wet and dry at OM is also measured, in order to determine the conditions leading to the lower hydraulic conductivity. Hydraulic conductivity can be determined with (BS 1377 Part 7) and/or the falling head method (BS 1377 Part 5/6, ASTM D 5084).

To evaluate the geotechnical suitability of the different mixtures tested, the hydraulic conductivity measurements can, e.g. be compared with the value recommended in the European legislation for low permeability liners and covers, i.e. $k \leq 10^{-9}$ m/sec. For composite mixtures complying with the above criterion, further geotechnical characterisation can be conducted, including compressive strength (ASTM D2166) and dry-wet durability tests (ASTM D559) to determine their strength characteristics and evaluate the long-term physical integrity. The US EPA considers a stabilised/solidified material with a strength of 50 psi (345 kPa), to have a satisfactory unconfined compressive strength. (U. EPA/625/6-89/022). This minimum guideline has been suggested for providing a stable foundation for materials placed upon it, including construction equipment and cover material. The minimum required unconfined compressive strength for the tailings-additive mixtures should be evaluated on the basis of the design loads to which the material will be subjected.

Geochemical tests

The geochemical tests that can be performed on compacted and aged composite mixtures, and include:

- modified ABA method to determine the acid generation potential of sulphidic tailings
- standard metal leachability tests - DIN 38414 S4 method.

Revegetation of tailings disposal areas

A number of specific chemical tests can be conducted to characterise the treated or untreated tailings materials as a growth medium for plants growth. These tests include: chemical analysis, acidity, salinity/sodicity and elements content in the soil solution. A detailed description of tests to be performed is given in:

"Methods of Soils Analysis. Part 2: Chemical and Microbiological Properties", 2nd edition, American Society of Agronomy Inc., Soil Science Society of America Inc., Madison, Wisconsin, US, 1982.

Chemical analysis

Apart from the determination of the heavy metals content previously described, a number of inorganic elements, essential for plants growth, can also be determined during the development of the revegetation scheme. They include:

Determination of total carbon, inorganic and organic carbon:

The total carbon in soils is the sum of both the organic and inorganic C. Most organic C is present in the organic matter fraction of the soil, which consists of micro-organism cells, plant and animal residues at various stages of decomposition, stable "humus" synthesised from residues, and highly carbonised compounds such as charcoal, graphite and coal. Inorganic C is largely found in carbonate minerals.

Determination of P, N and K

The presence of these elements on plant growth media is vital. Their potential deficiency can be mitigated by the addition of the suitable fertilizers. The type and fertilizers quantity should be determined taking into account their presence in the soil. Standard procedures for the determination of P, N and K content in soils will be followed.

Potential acidity and pH

There are several different methods available for measuring pH, including direct measurement in the saturating paste, measurement in the saturation extract and measurement to dilute extracts (i.e. liquid to solid ratios equal to 1, 2, 5 etc.). The more representative measurement for soil pH (as well as for the electrical Conductivity and for the content of soluble salts) is the saturation paste/extract, since it resembles better the field conditions. However, measurement methods in other than saturating conditions are often applied, since they are easy and provide higher quantity of leachate solution, thereby allowing the execution of additional analyses (e.g. sulphates and heavy metals concentration in the extract).

Potential acidity/alkalinity is determined by back titration with base or acid to a predefined end point.

EC and soluble salts

Similarly to pH measurement, electrical conductivity (EC) can be measured either in the saturation paste or in the saturation and diluted extracts. Soluble salts are determined by measuring their concentration in the extract. The Sodium Adsorption Ratio (SAR) is calculated as follows:

$SAR = Na/((Ca+Mg)/2)^{1/2}$

where Na, Ca and Mg are all expressed as meq/l

CEC and ESP

Cation Exchange Capacity (CEC) is a measure of the ability of soil to retain cations in an exchangeable form. Most of this exchange capacity originates from the clay and organic matter components of the sample. The capacity to retain cations in an exchangeable form arises from a negative charge on clay minerals and organic matter. This negative charge balance is neutralised by an equivalent number of exchangeable cations. Procedures for determining CEC in non-calcareous or non-gypsiferrous samples and in calcareous and gypsiferrous samples are different. The Exchangeable Sodium Percentage (ESP) is the ratio of the sodium exchangeable cations to the total cations exchanged.

ANNEX 5

Current standards for auditing in different parts of the world
Independent audits should commence with a review of the design and operation of the facility against the standards as set down by the regulators of the country in question and the undertakings by the mine in their own documentation.

In this respect, the standards of various countries are summarised as follows:

Australia
The Australian guidelines "Guidelines on the Safe Design and Operating Standards for Tailings Storage" and "Guidelines on the Development of an Operating Manual for Tailings Storage" both produced by the Australian department of Minerals and Energy, Western Australia defines standards for routine inspections and operational audits. A complimentary document is "Tailings Dam HIF Audit" that describes the components of an independent audit according to the Australian standards. This document can be found at http://notesweb.mpr.wa.gov.au/MODAMS/MDWebAnalysisReps.nsf/ca94b16fd41d002 and the guidelines ISBN 0 7309 7808 7 and ISBN 0 7309 7805 2.

Canada
The Canadian guidelines "A Guide to the Management of Tailings Facilities" and "Developing an Operation, maintenance and Surveillance manual for Tailings and Water Management" both produced by the Mining Association of Canada suggest that periodic inspections and reviews, audits, independent checks and comprehensive independent reviews need to be carried out as part of the surveillance programme. The documents can be found at www.mining.ca

South Africa
The primary document controlling a mining companies tailings disposal activities in South Africa is the Department of Mineral and Energy Mandatory Code of Practice for Mine Residue Deposits (MRDs) (available on the website www.dme.gov.za (publications)). This code requires each and every mine to set out in writing its intended standards and procedures for the protection of the health and safety of workers, and for the reduction of the risk of damage to persons and property.
Environmental aspects pertaining to the MRD are addressed in each mining companies Environmental Management Programme Report (EMPR), also required in terms of South Africa's Minerals Act (also available at the above web site).
Water quality aspects are controlled by the National Water Act, and a series of six Guideline Documents, M1 to M6.
The design of MRDs in South Africa is guided by SABS 0286: Code of Practice for Mine Residue Deposits.

Sweden
Generally all mining companies have programmes for daily, monthly and yearly inspections/audits, but there are no requirements on independent audits.

ANNEX 6

Pro Forma TMF Checklist For Visual Inspections

Name/Number of TMF:
Inspected by:
Designation:
Date/Time:

General items	Specific criteria	Defective? Comments	Yes/No
Roadways and access	Condition of roads and ramps		
	Damage and erosion of sides		
Trenches	Flow efficiency		
Drain outlets	Flow efficiency		
Outer perimeter	Evidence of spillage		
	Evidence of seepage		
	Presence of wet areas		
Slurry behaviour	Slurry flowrate		
	Slurry density		
Freeboard	Pond position		
	Pond depth		
	Wall freeboard		
Decant Facility and access	Clarity of discharge fluid		
	Structural integrity of decant		
Return water storage	Available capacity		
	Return water pumps		
Tailings Delivery system	Deposition position		
	Condition of pipes and valves		
Monitoring	Damage to instruments		
	Read according to programme		
Gates and fencing	General condition Signage in place and legible		

Pro Forma TMF Checklist for Annual Review

Name/Number of TMF   Date/Time:
Audit by: Designation:
Company: Signature:

General items	Specific criteria	Defective? Yes/No	Comments
Roadways and access	Perimeter roads		
	Access ramps		
Effluent and storm water trenches	Vegetation growth		
	Erosion of sides		
	Flow efficiency		
	Animal damage		
	External wet spots		
Drain outlets	Flow efficiency		
	Breakages		
	Animal life		
Outer perimeter	Evidence of spillage		
	Evidence of seepage		
	Presence of wet areas		
	Vegetation growth		
Outer wall and basin	Quality of wall construction		
	Evidence of cracking		
	Slope geometry		
	Deposition tonnage		
	Slurry density		
	Rate of rise		
	Available capacity		
	Pond depth and position		
	Freeboard		

General item	Specific criteria	Defective? Yes/No	Comments
Decant facility and access	Adequacy of catwalk/access		
	Structural integrity of decant		
	Pond wall position/integrity		
	Pond control		
Tailings delivery system	Operation and control		
	Condition of pipes and valves		
	System effectiveness		
Monitoring instruments	Obvious damage		
	Abnormal trends		
	Read according to program		
	Interpretation of results (eg stability analyses)		
Return water storage facility	Storage level		
	Degree of siltation		
	Condition of wall		
	Spillway condition		
	Decant facility		
	Pumps, valves and pipes		
Rehab. work	Monitor against program		
	Fertiliser applications		
	Performance of vegetation		
Water quality	Clarity of decant water		
	Water chemistry testing		
	General condition		
Gates and fencing	Security requirements		
	Signage in place and legible		
	Gates		
General	Routine logs completed		
	Monitoring carried out		
	Emergency preparedness		

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