TEXTE
100/2018

The WORLD Model Development and The Integrated Assessment of the Global Natural Resources Supply

For our Environment

Umwelt Bundesamt

TEXTE 100/2018

Environmental Research of the
Federal Ministry for the
Environment, Nature Conservation
and Nuclear Safety

Project No. (FKZ) 3712 93 102
Report No. (UBA-FB) 002711/ENG

The WORLD Model Development and The Integrated Assessment of the Global Natural Resources Supply

by

Harlad Sverdrup
University of Iceland, Reykjavik

Deniz Koca
Lund University, Lund

On behalf of the German Environment Agency

Weitere Publikationen aus dem Forschungsvorhaben 3712 93 102:

TEXTE 48/2018 Potentiale und Kernergebnisse der Simulationen von Ressourcenschonung(spolitik) - Endbericht des Projekts „Modelle, Potentiale und Langfristszenarien für Ressourceneffizienz“ (Sim-Ress)

TEXTE 49/2018 System analysis for environmental policy - System thinking through system dynamic modelling and policy mixing as used in the SimRess project

TEXTE 50/2018 Langfristszenarien und Potenziale zur Ressourceneffizienz in Deutschland im globalen Kontext – quantitative Abschätzungen mit dem Modell GINFORS

Fachbroschüre Ressourcenschonung als Zukunftsaufgabe – Ansatzpunkte für eine systemische Ressourcenpolitik

Imprint

Publisher:
Umweltbundesamt
Wörlitzer Platz 1
06844 Dessau-Roßlau
Tel: +49 340-2103-0
Fax: +49 340-2103-2285
info@umweltbundesamt.de
Internet: www.umweltbundesamt.de

/umweltbundesamt.de
/umweltbundesamt

Study performed by:
University of Iceland
Icelandic System Dynamics Centre (ISDC),
Department of Industrial Engineering,
Hjardarhagi 2-6,
107, Reykjavik, Iceland

Lund University
Centre for Environmental and Climate Research (CEC)
Sölvegatan 37,
22362, Lund, Sweden

Study completed in:
August 2018

Edited by:
I 1.1 Fundamental Aspects, Sustainability Strategies and Scenarios,
Sustainable Resource Use
Ullrich Lorenz

Publication as pdf:
http://www.umweltbundesamt.de/publikationen

ISSN 1862-4804

Dessau-Roßlau, December 2018

Abstract

WORLD6 model was developed and applied to simulate potential future supply and scarcity of a number of natural resources within the system dynamics modelling workpackage of the SimRess project. The main objective of this report is to outline the overall structure of the WORLD6 model and provide a detailed description of the "resources" module implemented in the model.

In the WORLD6 model, resources are clustered under METALS, MATERIALS and ENERGY sub-modules. The metals sub-module includes copper, zinc, lead, technology metals (silver, antimony, bismuth, tellurium, selenium, cadmium, germanium, indium, gallium), manganese, chromium, nickel, iron, stainless steel, speciality metals (molybdenum, rhenium, niobium, tantalum, cobalt, wolfram, platinum group metals (platinum, palladium, rhodium), super alloys) and light metals (aluminium, lithium). The materials sub-module includes sand, gravel and stone, where as the energy sub-module includes fossile fuels more specifically hydrocarbons.

For each of the modelled resources, model simulation results were presented and an associated integrated assessment was provided under separate sections of this report. In general, the WORLD6 model simulation results suggests that most of the supply peaks for metals, materials and fossil fuels will occure around the middle of this century. This will lead to some of the most serious industrial, political and social challenges of our times and will require careful preparation and research in order not to disrupt the functioning of society. Substantially better degree of recycling offers a long term sustainable and secure supply of most resources.

Kurzbeschreibung

Das Modell WORLD6 wurde zur Simulation einer potentiellen zukünftigen Versorgung und Verknappung einer Reihe von natürlichen Ressourcen im Modellierungsarbeitspaket für die Systemdynamik im Rahmen des SimRess-Projektes entwickelt und eingesetzt. Hauptzielsetzung dieses Berichtes ist die Darstellung der Gesamtstruktur des WORLD6-Modells und die Vorlage einer ausführlichen Beschreibung des im Modell verwendeten „Ressourcen"-Moduls.

Beim WORLD6-Modell werden die Ressourcen in die Kategorien METALLE, MATERIALIEN und ENERGIE aufgegliedert. Das Untermodul Metall umfasst Kupfer, Zink, Blei, Technologiemetalle (Silber, Antimon, Wismut, Tellur, Selen, Kadmium, Germanium, Indium, Gallium), Mangan, Chrom, Nickel, Eisen, Edelstahl, Spezialmetalle (Molybdän, Rhenium, Niob, Tantal, Kobalt, Wolfram, Metalle der Platingruppe (Platin, Palladium, Rhodium), Superlegierungen) und Leichtmetalle (Aluminium, Lithium). Das Untermodul Materialien umfasst Sand, Kies und Stein, und zum Untermodul Energie gehören fossile Brennstoffe, insbesondere Kohlenwasserstoffe.

Für jede modellierte Ressource wurden die Modellsimulationsergebnisse präsentiert und es wurde jeweils eine integrierte Beurteilung in einem separaten Abschnitt dieses Berichts vorgelegt. Ganz allgemein legen die Ergebnisse der WORLD6-Modellsimulation nahe, dass die meisten Versorgungs-spitzen bei Metallen, Materialien und fossilen Brennstoffen etwa in der Mitte dieses Jahrhunderts auftreten werden. Dies wird zu einigen der schwersten industriellen, politischen und sozialen Herausforderungen unserer Zeiten führen und sorgfältige Vorbereitungen und Forschungs¬arbeiten erforderlich machen, damit die Funktionsfähigkeit der Gesellschaft nicht zerstört wird. Ein deutlich höheres Niveau beim Recycling eröffnet die Möglichkeit einer langfristig nachhaltigen und sicheren Versorgung mit den meisten Ressourcen.

Table of Contents

List of Figures

List of Tables

Summary

This report is the final deliverable of the system dynamics modelling workpage of the Simress project.

The overall aim with this report is to present the general structure of the current version of the WORLD model – WORLD6 – and provide a detailed description of the "resources" module, which consists of a number of individual resource sub-modules.

This report is structured as follows:

Chapter 1 provides a very brief background information to this report.

Chapter 2 provides an overview of the WORLD6 model. The WORLD6 model incorporates World3 model in its structure and enhances it with a more detailed resource module. Thus, the WORLD6 model provides more detailed simulation results on the global supply of various resources. These resources include metals (i.e. BRONZE (Copper, Zinc, Lead, Silver, Gold and the dependent metals Antimony, Bismuth, Cobalt, Gallium, Germanium, Indium, Cadmium, Tellurium, Selenium), STEEL (Iron, Chromium, Manganese, Nickel, stainless steel), LIGHT METALS (Aluminium, Lithium), SPECIALTY METALS (Platinum group metals, Molybdenum, Rhenium, Niobium, Tantalum, Rare Earth Metals), materials (i.e. phosphorous, stone, sand, gravel) and energy resources (i.e. oil, coal, and gas). The model also provides detailed simulation results on the energy consumption associated to extraction of these resources (see Figure 2.1 in Chapter 2).

Chapter 2 also describes the methodology used in the model development. For each of the resources, systems analysis was used to conceptually model the production, trade and consumption systems. The flow pathways, the causal chains and the whole feedback structure of the systems were mapped using causal loop diagrams and flow charts. The resulting coupled differential equations were transferred to computer codes for numerical solutions. For all of the resources, first a stand-alone model was developed and tested. When that model worked satisfactorily, it was then integrated as a submodule in the WORLD6 model structure. The model was used to first reconstruct the past (1900-2015) to assess performance and robustness of the model. When the performance was satisfactory, then the model was applied to simulate the possible future (2015-2400), having the support of being able to reconstruct the observed past pattern for 115 years. The model was developed iteratively by repeatedly testing it on data gathered from the literature and corporate sources. The iterations were used to set the parameterization to such values that the mining history, observed ore grades and price picture could be reproduced.

Prior to the assessment of resource scarcity and the sustainable supply of resources (later in Chapter 4), scarcity was defined in Chapter 2 as:

- Soft scarcity, where shipments into the market do not match the demand for delivery from the market, leading to an adjustment of price up, reducing demand and sometimes increasing the amount offered to the market. This down-sized demand is the sign of soft scarcity. The adjustment goes on until the offering into the market matches the demanded amount. The presently implemented open market economic mechanisms work well to handle soft scarcity. The previously considered command-and-control regimes did not cope well with scarcity, but had a tendency to worsen the situation.
- Hard scarcity has two variants:
- Economic scarcity, where the demand is downsized by the increased price, but where the demand gets cut to zero by lack of ability to pay, while there is still material available (at high prices). If the use of the resource is a major bulk resource for society, then economic crisis will be the result if the resource is a keystone resource for the economy. Oil and coal are two such examples. If it is for a specialized but not essential technology, then failure of provision or/and technical regression will be the result.

- Physical scarcity is where the supply cannot sufficiently cover the demand. There will be a demand gap, a difference between the adjusted demand and what really can be supplied. Even if money to pay whatever is asked, the material is not available. This is something that we have observed with platinum and rhodium from time to time in platinum metal trading. Failure of provision is the final result. Rhodium is an example where this has occurred on frequent occasions.

Chapter 3 assesses the amount of extractable resources available on the seafloor in relationship to the resources available on land for some key infrastructural and technology metals; Iron, manganese, copper, nickel, cobalt, molybdenum, platinum group metals (PGM), silver and gold.

A literature review was performed to harvest the relevant data, assessments and evaluations. Estimates of technically extractable amount in the oceans graded with estimated extraction costs and degrees of extractability were compiled. Three types of deposits included in the assessment:

1. Manganese nodules on the main sea floor
2. Cobalt crusts on submerged seamounts
3. Massive sulphide deposits associated with black smokers and deep sea hydrothermal activity along mid-ocean ridges and tectonic separation- and subduction-zones

Results present the area cover, metal ore density and contents of different metals in the different nodules, crusts and massive sulphides on the ocean floor. The results from this assessment were used in setting the right input resource sizes for iron, manganese, copper, nickel, cobalt, molybdenum, platinum, gold and silver in the WORLD 6 model development.

The results of this asssessemnt suggest that for all the metals considered, significant amounts are present and located in mainly 5 limited regions of the oceans: two areas in the Pacific Ocean, one in the Atlantic Ocean and one in the Indian Ocean. Most of the tonnage of the interesting metals in nodules and black smokers are located at great water depth, from 1 km to more than 4 km depth, involving huge technological challenges. Only a smaller fraction (5-25%) of the detected amounts can be considered to be extractable under the most favourable of conditions.

Chapter 3 concludes that the role of huge new resources for metals on the ocean floor to replace land supply seems limited for most metals. For some of the main metals in society (iron, copper, platinum group metals, gold, silver), the actual extractable amounts of seafloor resources are significantly smaller than those on land, and mostly more expensive to extract. For some of the key technology metals that are in short supply and with small extractable resources (where every ton of supply counts and when the price is consistently high) the ocean resources may be an interesting and significant source of metals. The metals cobalt, nickel and silver may be in this category, with additional side products of gold, copper or perhaps molybdenum and platinum group metals. There is a large amount of manganese nodules available, however, the manganese price is at present very low, and would in no way pay for the effort to harvest them and then extract the manganese. The extraction of these resources may be commercially viable for the metals identified above, and they may under certain conditions contribute significant amounts to the total supply. Considering the difficulty of mining and based on an ore grade to metal price evaluation, it seems that only cobalt or silver would perhaps be interesting on a commercial level in the near future.

Chapter 4 provides indepth information on the WORLD6 Model's resources module. The resources module consists of: 1) Metals sub-module; 2) Meterials sub-module; and 3) Energy resources sub-module. Sections 4.1, 4.2 and 4.3 of the Chapter 4 present each of the individual resources modelled under these three sub-modules, including brief background information on the resource, underpinned hard data (eg. known individual metals and mineral reserves) used in the WORLD6 model, key model results and vericifaction on available historical data (eg. extraction amounts, price) for the period 1900-2010, as well as major discussions and conclusions.

Section 4.1 in Chapter 4 describes the Metals sub-module of the WORLD6 Model. A number of metals were clustered and modelled under four sub-modules. These are:

1. BRONZE sub-module
2. Copper (Cu), Zinc (Zn), Lead (Pb)
3. Technology metals – (Silver (Ag), Antimony (Sb), Bismuth (Bi), Tellirium (Te), Selenium (Se), Cadmium (Cd), Germanium (Ge), Indium (In) and Gallium (Ga))
4. STEEL sub-module
5. Manganese (Mn), Chromium (Cr), Nickel (Ni), Iron (Fe) and Stainless steel
6. SPECIALITY METALS sub-module
7. Molybdenum (Mo) and Rhenium (Re)
8. Niobium (Nb) and Tantalum (Ta)
9. Cobalt (Co)
10. Wolfram (W)
11. Platinum group metals – (Platinum (Pt), Palladium (Pd), Rhodium (Rh))
12. Super alloys
13. LIGTH METALS sub-module
14. Aluminium (Al)
15. Lithium (Li)

BRONZE Sub-module: Copper, Zinc, Lead (Section 4.1.1) and Technology metals (Ag, Bi, Cd, Ge, In, Sb, Se, Te) (Section 4.1.2)

The consistently declining copper, zinc and lead ore grades over the last 150 years are closely matched by the WORLD6 model. This is a strong indication of the coming scarcity and increased price for copper, zinc and lead. Failure to take this message in would be a very serious mistake. For governance, it would constitute a failure of taking responsibility, for corporations, to ignore key business obstacles or opportunities. In a historical perspective, it will appear as totally unforgivable to have ignored these scientific messages. However, using the BRONZE sub-model embedded inside the WORLD 6 model does give a richer picture, including more details and essential feedbacks. The model is able to reconstruct the past history of mining rates, ore grade decline and approximate metal price levels. The WORLD6 model provides reasonable estimates of the production rates for the technology metals dependent on copper, zinc and lead for their production. We need to carefully distinguish between the primary production from mines and total supply to the market. Long after primary metal mining has been reduced to insignificant levels, supply may be kept up by efficient recycling. The time after 2050 will be the age of urban mining, where more copper, zinc and lead metal is supplied from recycling that from primary extraction from mines. However, for the technology metals, many have at present low rates of recycling and these are much more vulnerable for decline in mining, unless the recycling efficiency can be significantly improved. The conclusion is that there is not an immediate risk for scarcity but that in the longer run (after 2030), scarcity manifested as rising metal prices should be expected and prepared for.

STEEL Sub-module: Manganese, Chromium, Nickel, Iron and Stainless steel (Section 4.1.3)

It is evident from the model simulation results that the world will not run out of iron in the next 400 years. However, it should be noted that by 2200, all of the rich ore grades and most of the high grade ores will be gone. As soon as the rich grade starts to run out, a significant rise in iron price should be expected. This increase comes from the increase in mining costs, ore treatment costs and iron extraction costs.

The model was used to simulate the supply of the individual metals used for stainless steel; iron, manganese, chromium and nickel. In the production of stainless steel nickel is predicted to become limiting metals for production, shifting a part of stainless steel to chromium-based stainless steel. Model results show that the global stock-in-use for stainless steel reaches a peak in 2140, and then decline. Shortage will become very pronounced after 2140, when demand can no longer be satisfied. After 2140, most supply will come from metal recycling. The long term supply of stainless steel is determined by several factors: Nickel is a limited resource, and even with very generous estimates of nickel reserves, it may set limits to how much stainless steel it is possible to make. Based on the model simulation results, after 2050, nickel will start to run out and nickel-based stainless steel production will start to decline after peaking at about 68 million to per year. The missing amount will be replaced by a simpler chromium-manganese stainless steel alloy. The available amounts of molybdenum, vanadium, niobium or cobalt are far too small to act as proper substitutes for nickel.

With increased price because of scarcity, we may expect recycling to go up and soften the decline somewhat. At recycling degrees above 80%, the supply of nickel, chromium and manganese will be sufficient for a century, but eventually run out.

The energy needed for mining, smelting and production of these metals (Fe, Mn, Cr, Ni and stainless steel) amounted to approximately 5% of all global energy available in 2016. By 2060, this will have increased to an amount five times the amount in 2016. It is self-evident that this will cause a huge demand on energy markets and sending energy prices up. The energy needs for the iron and steel sector will be most challenging to satisfy in the time after 2100. This will need future attention from science and policy.

LIGHT METALS sub-module (Aluminium) (Section 4.1.4)

The WORLD6 model simulation results for the aluminium supply, stocks in use and ore reserves suggest that supply to the market will peak around year 2080 and decline after that, reaching 2014 level in about 2300. Recycling will peak around the year 2100 and decrease with the stock-in-society after that. Primary extraction from bauxite will peak in the next decade 2020-2030, unless extra effort is made to increase mining efforts. This will, however, require a higher market price for aluminium. After 2030, recycling or urban mining will be the major source of aluminium. This will be the age of scrap metal, and probably provide the basis for growing many new companies and enterprises. Bauxite reserves may potentially run out, but more than one century from now. The aluminium production from mining will reach a peak about year 2030 and decline begins after 2040, because of diminishing ore grades and increasing energy prices. Aluminium scarcity will manifest itself as increasing metal price, and supply limits may likely come from energy limitations and not only reserves running low. The increased price will change demand and considerations for what aluminium is used for. It does not seem possible to substitute a significant fraction of the iron supply with aluminium. Nor will it be possible to increase the global aluminium production very much more, without upsetting the global energy balance.

Very large bauxite or aluminium reserves are available, but they are not inexhaustible. It is doubtful that we have enough carbon-based energy available to exhaust the reserves, because the amount carbon needed for alumina reduction may be difficult to source. At a production of 250 million ton aluminium per year, the production would consume about 14% of the present total global energy production. That would amount to about 1,600 million ton oil equivalents per year as electricity. The global energy production is to about 85% from fossil fuels, and by 2100 the energy production from fossil fuels may have declined to about half (Campbell 2013). Then the energy consumption used to maintain an aluminium production at the present level will consume an ever increasing fraction of the total global energy. That would significantly upset global energy prices and transfer to the aluminium market price. Thus, we may run out of energy and money to buy it, long before we run out of aluminium metal.

LIGHT METALS sub-module (Lithium) (Section 4.1.5)

Our assessment of the extractable amounts (resources) of lithium shows that the amount is far larger than what the scientific literature suggests it is. Instead of the often suggested URR of 40 million ton lithium, our assessment suggests a maximum resource estimate of 116 million ton lithium, a middle best estimate resource of 73 million ton lithium and a worst case low resource estimate of 34 million ton lithium. In the assessment we quantified extractable amounts across high, low and ultralow ore grades and included resources in salt brines, pegmatite hard minerals, lithium contained in different types of in clays, and accumulated deposits in all countries. The assessment on resource data concludes that 73 million ton lithium is the most plausible resource estimate for what is actually extractable, and this is taken as baseline modelling scenario. This amount is both technically feasible and economically possible.

The model simulation results suggest that a production peak is likely to occur in the period 2050-2055, with a slow decline in supply and a steady increase in price in the decades after 2120. The results also suggest that a maximum level of primary extraction may be reached in 2020, followed by a flat plateau to 2120, and followed a slow decline in primary extraction output.

The low recycling fraction for lithium remains is a technical hurdle to be overcome and a political management challenge. Model results show that it will stabilize around a maximum of 50% by market force mechanisms alone, which is not sufficient for solving the electrical vehicle transition challenge. That is not very good, and some considerations need to go into how that can be significantly improved.

The estimates for maximum number of battery units for vehicles, indicate that we have some challenges to solve:

1. It would be beneficial if the lithium use per battery unit could be reduced. Research and development is needed, to increase battery efficiency as well as capacity and safety. If alternatives are found, these are helpful only if they use less material of elements more abundant than lithium. Alternatively, one should develop batteries where more electricity can be stored per kg lithium.
2. Cobalt will be limiting for high-density batteries as lithium at the present level of use, cobalt abundance (extractable amounts) and production rates are lower than for lithium. However, cobalt extraction is dependent on copper, nickel and platinum extraction, and there are fewer options available for increasing the production significantly. Substitution for cobalt will be important, and manganese and nickel are acceptable substitutes with sufficient supply.
3. The degree of lithium recycling in the global system is far too low, and substantial improvements are required. This will need intensified efforts like those of UNEP, but also involvement of larger economic forces.
4. The estimates for the extractable amounts are uncertain, and the resources need to be far better investigated for extractability and extraction costs. However, finding larger potential resources will not be sufficient, they must also be technically recoverable, and this without large environmental damages. All in all, and Effort Return On Effort Investment (EROEI) must be performed. Substantially better degree of recycling is required to have a more long term secure supply.

SPECIALITY METALS sub-module: Cobalt (Section 4.1.6)

The present use of cobalt shows a low degree of recycling and systemic losses are significant. The extractable resources of cobalt are not very large, about 32 million ton extractable from land-based resources and about 34 million to from ocean seabed resources, a maximum of about 66 million ton. Model results show that some time after 2200, the supply of cobalt will have run out under a business-as-usual scenario, depending on the population scenario adopted. Too much cobalt is lost if only market mechanisms are expected to improve recycling, and unnecessary cobalt is wasted if no policy actions are taken. We can conclude that the market mechanisms alone neither have the goal nor the competence to make cobalt use sustainable without governmental interventions and regulations. A science-based solution-oriented policy is needed to correct the situation before it is too late and too much cobalt has been lost. Failure to take this message in will risk that society one day will be without cobalt. In order to conserve cobalt and allow it to be available for coming generations, present policies must be changed and the large observed losses mitigated within the next decades.

SPECIALITY METALS sub-module: Molybdenum and Rhenium (Section 4.1.7)

The WORLD6 model results show that molybdenum will always be in soft scarcity. But, after price-adjustment to demand, no hard scarcity and failure of provision will occur. The simulations shows a steady decline in molybdenum ore grade with time. Known reserve of about 14 million ton in 2015 peaks at 20 million ton in 2060. Under business-as-usual the molybdenum reserves will provide sufficient supply to about 2150, after which it declines quickly. By 2150, virtually all hidden resources will have been detected and included in minable reserves. The model suggests that by 2250, all known molybdenum reserves will have been exhausted.

Model simulation results show that molybdenum and rhenium will only last for another 150 years under the present metal conservation paradigm. These metals are so important for high performance alloys and advanced technologies, that much better metal conservation efforts need to be made than those that presently are being applied. The present policy that “the market will fix it” has very evidently failed to regulate the issue of molybdenum and rhenium conservation so far. It illustrates the need for governmental participation in creating a fair, stable and transparent market that is sustainable (Sverdrup et al., 2015c). The model performs well in tests against observed data and is thus a valid assessment tool. The model predicts a significant decline in molybdenum and rhenium supply after 2100 with severe scarcity after 2150 under the present regime of recycling. In the molybdenum-rhenium system the losses are too large for the system to be sustainable for more than 150 years. Most chemical uses are not followed by recapture. In metallic uses, a large portion is lost to dilution into general stainless steel scrap.

SPECIALITY METALS sub-module: Niobium and Tantalum (Section 4.1.8)

The model results show a steady decline in niobium and tantalum ore grades with time. Results show that the WORLD6 model is capable of capturing the observed trands in extraction rates, market price and the ore grade. On the systemic level, the results suggest that most chemical uses of tantalum and niobium are not followed by recapture. The present lack of consistent efforts in niobium and tantalum conservation cause an unsustainable supply situation to persist. In the long run, business as usual for these metals will lead to scarcity. At present, too large amounts of tantalum are irreversibly lost or lost into copper due to simplified electronics scrap handling. In metallic uses, a far too large portion of niobium is lost to dilution into stainless steel.

SPECIALITY METALS sub-module: Wolfram (Section 4.1.9)

In the future, worlfram can be expected to have an increasing demand, as there are many advanced technologies where it gives a technical advantage. Two parameters are the most important for the assessment of wolfram scarcity: global average supply per capita per year (necessary to assess whether we can have growth, if we will get global stagnation or global contraction) and global average stock-in-use per capita (necessary to assess wheter we can increase global utility provision, if it will develop into global stagnation or global contraction).

Simulation results show that soft scarcity for wolfram occurs around 2030 and hard scarcity occurs after 2200 because of resource exhaustion. The stock-in-use per capita declines aftetr 2100, first slowly, then fast (after 2220), and service provision is lost. There will be a significant worlfram scarcity problem after 2200.

SPECIALITY METALS sub-module: Super alloys (Section 4.1.10)

WORLD6 model results were used to assess the sufficiency of the molybdenum, rhenium, niobium, tantalum, cobalt and nickel supply for production of superalloys, considering its technical use in rocketry, high performance turbines and advanced jet engines. The assessment suggest that:

1. Military and strategic great power aspects:
    a. Without metals like tantalum, niobium, molybdenum, rhenium and hafnium available to the technologies demanding superalloys for their function, a sufficient number of advanced jet engines necessary for 5th and 6th generation fighter planes will not be available. States desiring these technologies in volume can no longer have that kind of hardware. That could mean loss of military superiority enjoyed by the few nations that have this today. Other sophisticated military hardware also depends on these metals to maintain their edge, and this will not be available to the same extent as before.
    b. Several types of missile engine nozzles and rocket engines will no longer be possible to make without at least 5th generation single crystal superalloys. This will limit military hardware performance, and weaken the military potential of states without own secure access to these metals.
2. Technological aspects:
    a. The limitation that could be caused to advanced jet engine technology is serious for future development of flight technologies and high-temperature jet turbines needed for fuel-efficient engines. Superalloys without the key metals (Made from molybdenum, niobium, tantalum, platinum group metals, zirconium, hafnium, cobalt, chromium, nickel (all which may face shortage risks in the next decades, with the only exception of zirconium and chromium) have definitely poorer technical performance and thus, this could become a problem in a future world with higher volatile hydrocarbon prices. In addition, those metals mentioned are also in scarce supply.
    b. Some very corrosion-resistant alloys can be made with superalloys containing molybdenum, tantalum, niobium, platinum group metals or rhenium and these are important for containment of corrosive molten salts containing halides. Examples of uses are:
        i. In molten salt cooled systems in advanced high-performance nuclear reactors. These are important for the development of the next generation molten salt cooled nuclear power plants based either on uranium 238 or thorium.
        ii. They are important for developing turbines for civil society power generation at higher temperature to increase thermal efficiency.
        iii. They are also important for the development of inert electrodes for smelting of light metals like magnesium or aluminium and for production of silicium and ferrosilicium.

SPECIALITY METALS sub-module: Platinum group metals (platinum, palladium, rhodium) (Section 4.1.11)

Our assessment of the extractable amounts (resources) of platinum group metals are twice as big as earlier anticipated and estimated. This result does not affect the fact that the metals are very scarce and can be expected to remain scarce for the foreseeable future. The assessment suggests an ultimately recoverable resource (URR) of about 216,000 ton platinum group metals (The platinum resource is approximately 99,000 ton, and the palladium resource is approximately 93,000 ton), summing up both primary resources and resources coming through dependent secondary extraction. The sensitivity analysis suggests that the resource size given above is the most probable estimate.

The WORLD6 model simulation results show that peak production from mines will occur in the period 2035-2050, and a peak in supply to society 20-35 years later. The model outputs suggest that diffuse and dissipative losses are at present too large on the global systemic level for the platinum group metals, and that conservation efforts would be needed to be strengthened if we consider the very long term perspective. Because ore grade and cost of extraction is included in the model, effort required per amount platinum group metals extracted increase with time, eventually closing down extraction. In summary, based on the model results:

1. Production of supply and extraction
    a. The platinum group metals extraction will go through a peak in 2035-2050 and market supply reach a peak during 2070-2080. The delay between extraction and supply peak depend on the degree of recycling and the relative size of the loss flows to the extraction flows.
2. Reserves and resources
    a. The total extractable amounts of platinum group metals if we take in the latest resource assessments, consider new technology for mining at great depth, and the resources existing at great depth are about 220,000 ton
    b. The known reserves will stay at about 60,000 ton and slowly decline after 2030.
    c. The ratio between the cumulative amount supplied to society and the cumulative amounts primary extracted is about 2.6, this is called Factor X.

Section 4.2 of Chapter 4 describes the materials submodule of the WORLD6 Model. For this report only sand, gravel, crushed rock and stone were assessed as a part of the materials submodule.

The simulation results under assumed business-as-usual scenario, show that cut stone production will reach a maximum level about 2020-2030 and slowly decline after that. The cause for this is that demand exceeds extraction as well as slow exhaustion of the known reserves of high quality stone. Sand and gravel also show peak behaviour and reach their maximum production rate in 2060-2070. The reason for the peak behaviour is partly driven by an expected population peak in 2065 and increasing prices for sand and gravel, limiting the demand.

While in a global perspective supply may seem inexhaustible, supply and availability is already a growing problem at a local to regional scale, signalling that global trade with sand, gravel and stone will continue to increase. We need better data on production rates, use and recycling to better assess risks of regional scarcity, as well as a model of this kind divided into different regions. The state of the research is not at present at a stage where this is possible without substantial research funding to support it. The materials sub-module of WORLD6 model, given better data, could be used for this as it could be down-scaled to regions.

Section 4.3 of Chapter 4 defines the energy sub-module of the WORLD6 model. For this report only fossil fuels (hydrocarbons) were assessed as a part of the energy submodule.

Only a fraction of the world's detectable resources of fossil fuels is extractable, one of the major reasons for widely different estimates of the resources. There are several reasons for the differences in resource estimates:

1. A significant part of the resources is hidden behind an energy barrier; the energy cost to extract is larger than the benefit of what can be extracted. When the extraction costs exceed the income, then extraction stops.
2. A significant part of detected resources is blocked hidden behind a technological barrier. The resources are located in such places, that the technology to extract it does not exist now nor in the foreseeable future. The resource is unreachable.
3. A significant part is behind a capital barrier. The resource is hypothetically extractable, but there is not enough capital to make the investments, nor the liquid money to buy the product at the price it would imply. Hydrocarbon fossil fuels are out of reach when the oil price reaches 1700 $ per ton oil equivalent.
4. A significant part of the resources is hidden behind an environmental destruction barrier. The resource is of such a poor environmental quality, or the extraction and use have such large environmental detrimental impact, that it remains unfeasible.

Thus, the world does not run out of fossil fuels because of lack of fossil carbon in the ground. The production comes to a stop because the implications of continuing extraction are unacceptable.

Chapter 5 provides overall conclusions that most of the supply peaks for metals, materials and fossil fuels come in the period 2040-2060. This will amount to some of the most serious industrial, political and social challenges of our times and will need careful preparation and research in order not to disrupt the functioning of society. A business-as-usual is a worst-case scenario with respect to material resource use, leading to challenges of material scarcity and a significant potential for society disruptions. With increasing recycling peak extraction of resources can be shifted towards the future, increasing the Factor X, i.e. the number of times one weight unit is used before it is lost.

Zusammenfassung

Dieser Bericht ist das endgültige Ergebnis des Modellierungsarbeitspakets für die Systemdynamik im Rahmen des SimRess-Projektes.

Hauptzielsetzung dieses Berichtes ist die Darstellung der allgemeinen Struktur der aktuellen Version des WORLD-Modells (WORLD6) und die Vorlage einer ausführlichen Beschreibung des „Ressourcen“-Moduls, der sich aus einer Reihe einzelner Ressourcen-Untermodule zusammensetzt.

Dieser Bericht ist wie folgt aufgebaut:

Kapitel 1 enthält sehr kurze Hintergrundinformationen zu diesem Bericht.

Kapitel 2 ist ein Überblick über das WORLD6-Modell. Das WORLD6-Modell umfasst das World3-Modell, baut auf dessen Struktur auf und erweitert diese um ein ausführlicheres Ressourcenmodul. Somit bietet das WORLD6-Modell ausführlichere Simulationsergebnisse über die globale Versorgung mit verschiedenen Ressourcen. Zu diesen Ressourcen zählen Metalle (d. h. BRONZE (Kupfer, Zink, Blei, Silber, Gold und die damit zusammenhängenden Metalle Antimon, Wismut, Kobalt, Gallium, Germanium, Indium, Kadmium, Tellur, Selen), STAHL (Eisen, Chrom, Mangan, Nickel, Edelstahl), LEICHTMETALLE (Aluminium, Lithium), SONDERMETALLE (Metalle der Platingruppe, Molybdän, Rhenium, Niob, Tantal, Seltene Erden), Materialien (d. h. Phosphor, Stein, Sand, Kies) und Energieressourcen (d. h. Öl, Kohle und Gas). Das Modell liefert auch ausführliche Simulationsergebnisse in Bezug auf den Energieverbrauch in Verbindung mit der Gewinnung dieser Ressourcen (siehe Abb. 2.1 in Kapitel 2).

In Kapitel 2 wird auch die bei der Modellentwicklung verwendete Methodologie beschrieben. Für jede der Ressourcen wurde die Systemanalyse für eine konzeptionelle Modellierung der Produktions-, Handels- und Verbrauchssysteme verwendet. Die Ablaufbahnen, die Ursachenketten und die gesamte Feedback-Struktur der Systeme wurden in Form von Ursache- und Folgediagrammen sowie Flowcharts aufgezeichnet. Die sich daraus ergebenden gekoppelten Differentialgleichungen wurden mit der Zielsetzung numerischer Lösungen in Computercodes umgewandelt. Für alle Ressourcen wurde zunächst ein eigenständiges Modell entwickelt und getestet. Als dieses Modell zufriedenstellend funktionierte, wurde es als Untermodul in die Struktur des WORLD6-Modells integriert. Das Modell diente zunächst zur Rekonstruktion der Vergangenheit (1900 bis 2015), damit die Leistung und Stabilität des Modells bewertet werden konnte. Als die Leistung zufriedenstellend war, wurde des Modell zur Simulation der möglichen Zukunft (2015 bis 2400) ausgehend von der Erkenntnis verwendet, dass es gelungen war, die beobachteten Vergangenheitsmuster von 115 Jahren zu rekonstruieren. Das Modell wurde iterativ durch wiederholte Tests mit Daten entwickelt, die aus Literatur- und Unternehmensquellen stammen. Diese Iterationen dienten zur Festlegung der Parameter auf solche Werte, die eine Reproduktion der Bergbauindustrie, der beobachteten Erzqualität und des Preisbilds ermöglichen.

Vor der Beurteilung der Ressourcenverknappung und der nachhaltigen Versorgung mit Ressourcen (siehe Kapitel 4) wurde die Verknappung in Kapitel 2 wie folgt definiert:

- Sanfte Verknappung, bei der die Belieferung des Marktes nicht der Nachfrage des Marktes nach Lieferungen entspricht, wodurch es zu Preiserhöhungen kommt, die wiederum einen Rückgang der Nachfrage und manchmal auch eine Steigerung der dem Markt angebotenen Menge nach sich ziehen. Die geringere Nachfrage ist ein Zeichen sanfter Verknappung. Die Anpassungen gehen weiter, bis das Angebot für den Markt der nachgefragten Menge entspricht. Die gegenwärtig wirksamen wirtschaftlichen Mechanismen des offenen Marktes reichen für die Bewältigung der sanften Verknappung aus. Die bereits früher in Erwägung gezogenen Plan- und Überwachungsregelungen können die Verknappung nicht in den Griff bekommen, sondern zeigten sogar eine Tendenz zur Verschlimmerung der Situation.
- Bei der schweren Verknappung gibt es zwei Varianten:

- Die wirtschaftliche Verknappung, bei der die Nachfrage durch die Preissteigerung sinkt, aber dann durch die fehlende Zahlungsfähigkeit auf Null sinkt, wenn noch Material (zu hohen Preisen) verfügbar ist. Wenn die Nutzung der Ressource von großer Bedeutung für die Gesellschaft ist, tritt eine Wirtschaftskrise ein, falls es sich bei der Ressource um einen Grundpfeiler der Wirtschaft handelt. Öl und Kohle sind zwei Beispiele dafür. Wenn es um eine spezialisierte, aber nicht unverzichtbare Technologie geht, wird der Nachschub ausfallen und eine technische Regression eintreten.
- Eine physikalische Verknappung ist dann gegeben, wenn die Versorgung die Nachfrage nicht ausreichend abdecken kann. Dadurch entsteht eine Nachfragelücke, also ein Unterschied zwischen der angepassten Nachfrage und der tatsächlichen Versorgungsmöglichkeit. Auch wenn Geld für jeden geforderten Preis vorhanden ist, steht das Material nicht zur Verfügung. Derartige Zustände wurden bereits mehrfach bei Platin und Rhodium auf dem Platinmarkt notiert. Als Endergebnis fällt der Nachschub aus. Rhodium ist ein Beispiel dafür, dass sich dies schon mehrfach ereignet hat.

In Kapitel 3 wird die Menge der nutzbaren Ressourcen auf dem Meeresboden im Verhältnis zu den an Land verfügbaren Ressourcen für einige wichtige Infrastruktur- und Technologiemetalle, Eisen, Mangan, Kupfer, Nickel, Kobalt, Molybdän, Metalle der Platingruppe (PGM), Silber und Gold bewertet.

Zur Erfassung der relevanten Daten, Beurteilungen und Auswertungen wurde die entsprechende Fachliteratur herangezogen. Schätzungen der technisch nutzbaren Mengen in den Meeren im Verhältnis zu den geschätzten Abbaukosten und dem Umfang der Nutzbarkeit wurden zusammengestellt. Bei der Bewertung wurden drei Arten von Lagerstätten berücksichtigt:

1. Manganknollen auf dem Hauptmeeresboden
2. Kobaltkrusten auf Seebergen unter Wasser
3. Massive Sulfidvorkommen in Verbindung mit Schwarzen Rauchern und Hydrothermalquellen in der Tiefsee an Höhenzügen mitten im Ozean und tektonischen Trenn- und Subduktionszonen

Aus den Ergebnissen gehen die Bereichsabdeckung, die Metallerzdichte und die Anteile der verschiedenen Metalle in den einzelnen Knoten, Krusten und massiven Sulfidvorkommen auf dem Meeresboden hervor. Die Ergebnisse dieser Beurteilung wurden bei der Festlegung der richtigen Eingabegrößen der Ressourcen für Eisen, Mangan, Kupfer, Nickel, Kobalt, Molybdän, Platin, Gold und Silber bei der Entwicklung des WORLD6-Modells verwendet.

Die Ergebnisse dieser Auswertung legen nahe, dass bei allen berücksichtigten Metallen erhebliche Mengen vorhanden sind und sich hauptsächlich in 5 begrenzten Regionen der Meere befinden: zwei Bereiche im Pazifischen Ozean, einer im Atlantischen Ozean und einer im Indischen Ozean. Die größten Vorkommen der interessanten Metalle in Knoten und Schwarzen Rauchern befinden sich in großer Wassertiefe, von 1 km bis mehr als 4 km tief, was enorme technische Herausforderungen beinhaltet. Nur ein kleinerer Teil (5 bis 25 %) der entdeckten Vorkommen können als unter den günstigsten Bedingungen förderbar eingestuft werden.

Kapitel 3 kommt zu der Schlussfolgerung, dass die Rolle der riesigen neuen Metallressourcen auf dem Meeresboden als Ersatz für die Gewinnung an Land bei den meisten Metallen begrenzt zu sein scheint. Bei einigen der für die Gesellschaft wichtigsten Metalle (Eisen, Kupfer, Metalle der Platingruppe, Gold, Silber) sind die tatsächlich förderbaren Vorkommen auf dem Meeresboden deutlich geringer als an Land und ihre Gewinnung ist meistens kostspieliger. Bei einigen der wichtigsten Technologiemetalle, die Mangelware sind und nur als kleine förderbare Ressourcen vorkommen (wo jede Tonne zählt und bei ständig hohem Preis), können die Vorkommen im Meer eine interessante und wichtige Metallquelle sein. Die Metalle Kobalt, Nickel und Silber können in diese Kategorie fallen, wobei Gold, Kupfer oder vielleicht Molybdän und Metalle der Platingruppe eine Rolle als zusätzliche Nebenprodukte spielen könnten. Es gibt eine Vielzahl von Manganknoten. Allerdings ist der Manganpreis gegenwärtig sehr niedrig und kann den Aufwand für deren Förderung und die anschließende Gewinnung von Mangan keinesfalls abdecken. Die Förderung dieser Ressourcen kann bei den oben genannten Metallen wirtschaftlich tragbar sein. Unter bestimmten Umständen können sie sogar in erheblichen Mengen zur Gesamtversorgung beitragen. Angesichts der Schwierigkeiten im Bergbau und auf der Grundlage einer Auswertung der Erzgüte im Verhältnis zum Metallpreis scheinen in naher Zukunft lediglich Kobalt oder Silber eventuell wirtschaftlich interessant zu sein.

Kapitel 4 enthält tiefgehende Informationen über das Ressourcenmodul des WORLD6-Modells. Das Ressourcenmodul setzt sich wie folgt zusammen: 1) Untermodul Metall; 2) Untermodul Materialien und 3) Untermodul Energie. Die Abschnitte 4.1, 4.2 und 4.3 von Kapitel 4 präsentieren die einzelnen im Rahmen dieser drei Untermodule modellierten Ressourcen einschließlich kurzer Hintergrundinformationen zur Ressource, unterlegt mit den belastbaren Daten (z. B. bekannte Einzelmetall- und Mineralreserven) des WORLD6-Modells, der wichtigsten Modellergebnisse und der Abgleichung mit den verfügbaren historischen Daten (z. B. Fördermengen, Preis) im Zeitraum 1900 bis 2010 sowie der bedeutenden Erörterungen und Schlussfolgerungen.

Abschnitt 4.1 in Kapitel 4 beschreibt das Metall-Untermodul des WORLD6-Modells. Einige Metalle wurden in Gruppen zusammengefasst und unter vier Untermodulen modelliert. Dabei handelt es sich um:

1. Untermodul BRONZE

- Kupfer (Cu), Zink (Zn), Blei (Pb)
- Technologiemetalle – (Silber (Ag), Antimon (Sb), Wismuth (Bi), Tellur (Te), Selen (Se), Kadmium (Cd), Germanium (Ge), Indium (In) und Gallium (Ga))

2. Untermodul STAHL

- Mangan (Mn), Chrom (Cr), Nickel (Ni), Eisen (Fe) und Edelstahl

3. Untermodul SPEZIALMETALLE

- Molybdän (Mo) und Rhenium (Re)
- Niob (Nb) und Tantal (Ta)
- Kobalt (Co)
- Wolfram (W)
- Metalle der Platingruppe – (Platin (Pt), Palladium (Pd), Rhodium (Rh))
- Superlegierungen

4. Untermodul LEICHTMETALLE

- Aluminium (Al)
- Lithium (Li)

Untermodul BRONZE: Kupfer, Zink, Blei (Abschnitt 4.1.1) und Technologiemetalle (Ag, Bi, Cd, Ge, In, Sb, Se, Te) (Abschnitt 4.1.2)

Die in den letzten 150 Jahren ständig sinkende Qualität von Kupfer, Zink und Blei wurde im WORLD6-Modell bestätigt. Dies ist ein starkes Anzeichen für die bevorstehende Verknappung und steigenden Preise für Kupfer, Zink und Blei. Eine Nichtberücksichtigung dieser Botschaft wäre ein sehr schwerer Fehler. Für Regierungen wäre es ein Versagen, die Verantwortung zu übernehmen; für Unternehmen wäre es eine Unterlassungssünde, die wichtigen Geschäftshindernisse oder -möglichkeiten zu ignorieren. Aus historischer Perspektive würde es als absolut unverzeihlich eingestuft werden, diese wissenschaftlichen Botschaften zu ignorieren. Allerdings erhält man durch die Einbeziehung des BRONZE-Untermoduls im Rahmen des WORLD6-Modells ein umfangreicheres Bild mit mehr Einzelheiten und wichtigen Rückmeldungen. Das Modell kann die Vergangenheit der Bergbauwerte, des Erzqualitätsrückgangs und des ungefähren Preisniveaus rekonstruieren. Das Modell WORLD6 liefert vernünftige Schätzwerte für die Technologiemetalle, deren Produktion von Kupfer, Zink und Blei abhängig ist. Wir müssen sorgfältig unterscheiden zwischen der Primärproduktion der Bergwerke und dem gesamten Marktangebot. Lange nach dem Niedergang des Primärmetallbergbaus auf unbedeutende Werte kann die Versorgung durch effizientes Recycling gesichert werden. Der Zeitraum nach 2050 wird zum Zeitalter des städtischen Bergbaus, weil dann mehr Kupfer, Zink und Blei durch Recycling gewonnen werden als durch den Abbau in Bergwerken. Die Technologiemetalle zeichnen sich allerdings gegenwärtig durch einen geringen Recyclinganteil aus und reagieren viel anfälliger auf den Rückgang im Bergbau, sofern die Recyclingeffizienz nicht erheblich verbessert werden kann. Die Schlussfolgerung lautet, dass keine unmittelbare Verknappungsgefahr besteht, diese sich aber langfristig (nach 2030) manifestieren könnte, weil man mit steigenden Metallpreisen rechnen und sich darauf vorbereiten sollte.

Untermodul STAHL: Mangan, Chrom, Nickel, Eisen und Edelstahl (Abschnitt 4.1.3)

Aus den Modellsimulationsergebnissen geht eindeutig hervor, dass es in der Welt in den nächsten 400 Jahren keinen Eisenmangel geben wird. Allerdings sollte beachtet werden, dass bis 2200 alle Reicherzvorkommen und die meisten hochwertigen Erze abgebaut sein werden. Sobald der Rückgang des Reicherzes beginnt, sollte mit einem deutlichen Anstieg des Eisenpreises gerechnet werden. Dieser Anstieg kommt durch die Kostensteigerungen im Bergbau, bei der Erzverarbeitung und beim Abbau von Eisen zustande.

Das Modell diente auch dazu, die Versorgung mit den einzelnen Metalle zu simulieren, die für Edelstahl verwendet werden: Eisen, Mangan, Chrom und Nickel. Bei der Herstellung von Edelstahl wird prognostiziert, dass Nickel zum Problemmetall wird, so dass ein Teil des Edelstahls mit Chrom als Grundlage produziert werden wird. Die Modellergebnisse weisen auch aus, dass der weltweite Lagerbestand für Edelstahl im Jahr 2140 seinen Höhepunkt erreicht und dann zurückgehen wird. Die Verknappung fällt nach 2140 immer massiver aus, wenn die Nachfrage nicht mehr befriedigt werden kann. Nach 2140 stammt der Großteil des Nachschubs aus dem Metallrecycling. Die langfristige Versorgung mit Edelstahl hängt von mehreren Faktoren ab: Nickel ist eine begrenzte Ressource, und auch bei sehr großzügigen Schätzungen in Bezug auf die Nickelreserven können Grenzen für die tatsächlich produzierbare Menge Edelstahl entstehen. Auf der Grundlage der Modellsimulationsergebnisse wird Nickel nach 2050 knapper und die Produktion von Edelstahl auf Nickelbasis wird nach dem Höchstwert von etwa 68 Million Tonnen pro Jahr zurückgehen. Die fehlende Menge wird durch eine einfachere Edelstahllegierung aus Chrom und Mangan ersetzt werden. Die verfügbaren Mengen von Molybdän, Vanadium, Niob oder Kobalt sind viel zu gering als dass sie als angemessener Ersatz für Nickel in Frage kämen.

Angesichts der Preissteigerungen aufgrund von Verknappungen können wir mit einem Recyclinganstieg und dadurch mit einer leichten Abschwächung des Rückgangs rechnen. Bei Recyclingwerten von über 80 % reicht die Versorgung mit Nickel, Chrom und Mangan für etwa ein Jahrhundert aus, bevor auch diese Quelle versiegt.

Die für Bergbau, Verhüttung und Produktion dieser Metalle (Fe, Mn, Cr, Ni und Edelstahl) benötigte Energie belief sich im Jahr 2016 auf etwa 5 % der gesamten weltweit verfügbaren Energie. Bis 2060 wird sich dieser Wert von 2016 auf das Fünffache erhöhen. Es ist offensichtlich, dass dadurch eine riesige Nachfrage aus den Energiemärkten entstehen wird, die für einen Anstieg bei den Energiepreisen sorgen wird. Der Energiebedarf im Eisen- und Stahlsektor wird ab dem Jahr 2100 nur noch extrem schwer zu decken sein. Das macht die zukünftige Aufmerksamkeit der Wissenschaft und Politik erforderlich.

Untermodul LEICHTMETALLE (Aluminium) (Abschnitt 4.1.4)

Die Simulationsergebnisse des WORLD6-Modells in Bezug auf die Aluminiumversorgung, nutzbare Lagerbestände und Erzreserven legen nahe, dass die Marktversorgung ihren Höhepunkt etwa im Jahr 2080 erreichen und danach ungefähr bis zum Jahr 2300 auf den Wert des Jahres 2014 zurückgehen wird. Der Recyclinganteil wird seinen Höhepunkt etwa im Jahr 2100 erreichen und danach entsprechend dem in der Gesellschaft vorhandenen Bestand sinken. Die Primärgewinnung von Bauxit wird im nächsten Jahrzehnt (2020-2030) ihren Höhepunkt erreichen, sofern keine zusätzlichen Bemühungen zur Steigerung der Bergbauerträge unternommen werden. Die Folge davon wird jedoch ein höherer Aluminiumpreis auf den Märkten sein. Nach 2030 wird Aluminium hauptsächlich durch Recycling oder städtischen Bergbau gewonnen werden. Dann erreichen wir das Zeitalter der Altmetalle, das wahrscheinlich die Grundlage für die Entstehung und den Erfolg von vielen neuen Firmen und Gesellschaften bilden wird. Die Bauxitreserven könnten letztendlich auch erschöpft werden, was aber mehr als ein Jahrhundert dauern dürfte. Die Aluminiumproduktion erreicht ihren Höhepunkt etwa im Jahr 2030, um dann ab 2040 zurückzugehen, weil die Erzqualität sinkt und die Energiepreise steigen. Die Aluminiumverknappung wird in einem steigenden Metallpreis zum Ausdruck kommen, wobei die Liefergrenzen wahrscheinlich eher durch Energieeinschränkungen und nicht nur durch abnehmende Reserven entstehen werden. Der höhere Preis wird die Nachfrage verändern und Gedanken darüber auslösen, wofür Aluminium verwendet wird. Es scheint nicht möglich zu sein, einen wesentlichen Teil der Eisenversorgung durch Aluminium zu ersetzen. Es wird auch nicht möglich sein, die weltweite Aluminiumproduktion wesentlich mehr zu steigern, ohne dabei das weltweite Energiegleichgewicht zu stören.

Es gibt sehr große Bauxit- oder Aluminiumreserven, aber auch diese sind nicht unerschöpflich. Es ist zu bezweifeln, dass uns für die Nutzung der Reserven genügend Energie auf Kohlenstoffbasis zur Verfügung steht, weil die für die Tonerdensenkung notwendige Kohlenstoffmenge schwierig zu beschaffen sein dürfte. Bei einer Produktion von 250 Millionen Tonnen pro Jahr würde der Energieverbrauch bei etwa 14 % der gegenwärtigen weltweiten Gesamtenergieproduktion liegen. Der Stromverbrauch würde rund 1.600 Millionen Tonnen Öl pro Jahr entsprechen. Die weltweite Energieproduktion besteht zu etwa 85 % aus fossilen Brennstoffen, und bis 2100 könnte dieser Wert auf rund die Hälfte gesunken sein (Campbell 2013). In diesem Fall würde der für die Aufrechterhaltung der Aluminiumproduktion auf dem heutigen Niveau benötigte Energieverbrauch einen ständig wachsenden Anteil der gesamten weltweiten Energie ausmachen. Dadurch würden die weltweiten Energiepreise erheblich steigen und sich auch auf den Aluminiummarktpreis auswirken. Somit könnten wir keine Energie und auch kein Geld mehr für den Kauf von Energie haben, und zwar lange bevor das Aluminiummetall verbraucht ist.

Untermodul LEICHTMETALLE (Lithium) (Abschnitt 4.1.5)

Unsere Beurteilung der förderbaren Mengen (Ressourcen) ergibt bei Lithium, dass die Vorkommen wesentlich größer sind als dies in der wissenschaftlichen Fachliteratur dargestellt wird. Statt der häufig genannten URR (ultimately recoverable resources - schlussendlich förderbare Ressourcen) von 40 Millionen Tonnen Lithium kommt unsere Beurteilung auf einen maximalen Ressourcenschätzwert von 116 Million Tonnen Lithium, einen mittleren Bestwert von 73 Millionen Tonnen Lithium und einen im schlechtesten Fall zu erwartenden Ressourcenbestand von 34 Millionen Tonnen Lithium. Bei der Beurteilung haben wir die förderbaren Mengen in den Erzgüten hohe, niedrige und besonders schlechte Qualität quantifiziert und dabei Ressourcen in Salzlake, Pegmatit-Mineralien, Lithium in verschiedenen Lehmarten und angesammelte Vorkommen in allen Ländern berücksichtigt. Die Beurteilung der

Ressourcendaten kommt zu der Schlussfolgerung, dass 73 Millionen Tonnen Lithium die plausibelste Ressourcenschätzung der tatsächlich förderbaren Menge ist. Dieser Wert dient als Grundlage für weitere Modellszenarien. Diese Menge ist einerseits technisch machbar und andererseits wirtschaftlich umsetzbar.

Die Modellsimulationsergebnisse legen nahe, dass ein Produktionshöhepunkt wahrscheinlich im Zeitraum 2050 bis 2055 eintritt und die Versorgung in den Jahrzehnten nach 2120 bei gleichzeitigem Preisanstieg zurückgehen wird. Aus dem Ergebnis ist auch ablesbar, dass der Höchstwert bei der Primärgewinnung im Jahr 2020 erreicht wird. Daran anschließend wird sich bis 2120 nicht viel verändern, bis danach ein langsamer Rückgang bei der Primärgewinnung zu erwarten ist.

Der geringe Recyclinganteil bei Lithium bleibt eine technische Hürde, die es zu überwinden gilt, und eine Herausforderung für die politische Verwaltung. Aus dem Modellergebnissen geht hervor, dass sich dieser Wert alleine aufgrund von Mechanismen der Marktkräfte bei etwa maximal 50 % stabilisieren wird, was aber nicht zur Bewältigung der Umstellung auf Elektrofahrzeuge ausreichen wird. Das ist nicht besonders gut, und es bedarf einiger Überlegungen, wie diese Entwicklung erheblich verbessert werden kann.

Die Schätzungen in Bezug auf die Höchstzahl von Batterieeinheiten für Fahrzeuge machen deutlich, dass wir einige Herausforderungen zu überstehen haben:

1. Es wäre von Vorteil, wenn der Lithium-Verbrauch pro Einheit gesenkt werden könnte. Ein erheblicher Forschungs- und Entwicklungsaufwand ist zur Verbesserung von Batterieeffizienz, Kapazität und Sicherheit nötig. Wenn Alternativen gefunden werden, sind diese nur dann nützlich, wenn sie weniger Material von Elementen verbrauchen, die reichlicher vorhanden sind als Lithium. Alternativ sollten Batterien entwickelt werden, in denen mehr Elektrizität pro Kilogramm Lithium gespeichert werden kann.
2. Kobalt wird die Herstellung von hochdichten Lithium-Batterien bei der gegenwärtigen Nutzung begrenzen, weil der Kobalt-Überschuss (förderbare Menge) und die Produktionsraten geringer sind als bei Lithium. Allerdings hängt die Kobaltförderung von der Kupfer-, Nickel- und Platingewinnung ab, und es gibt weniger verfügbare Optionen zur deutlichen Steigerung der Produktion. Ein Ersatz für Kobalt ist sehr wichtig, wobei sich Mangan und Nickel mit ausreichenden Vorräten anbieten.
3. Der Anteil des Lithium-Recyclings im globalen System ist viel zu gering, so dass erhebliche Verbesserungen benötigt werden. Dazu werden intensivere Bemühungen wie die von UNEP notwendig, aber auch größere wirtschaftliche Kräfte müssen einbezogen werden.
4. Die Schätzungen in Bezug auf die förderbaren Mengen sind unzuverlässig und die Ressourcen müssen erheblich besser auf Förderbarkeit und Förderkosten untersucht werden. Eine erfolgreiche Suche nach größeren potenziellen Ressourcen wird jedoch nicht ausreichen. Die Vorkommen müssen auch technisch förderbar sein, und zwar ohne große Umweltschäden. Insgesamt muss ein EROEI-Wert erreicht werden (Effort Return On Effort Investment). Ein erheblich besser Recyclingwert ist Voraussetzung für eine langfristig bessere und sichere Versorgung.

Untermodul SPEZIALMETALLE: Kobalt (Abschnitt 4.1.6)

Die gegenwärtige Nutzung von Kobalt steht im Zeichen eines geringen Recyclingwerts und die systemweiten Verluste sind erheblich. Die förderbaren Kobaltmengen sind nicht besonders groß, etwa 32 Millionen Tonnen aus Ressourcen an Land und etwa 34 Millionen Tonnen auf dem Meeresboden, so dass sich ein Höchstwert von etwa 66 Millionen Tonnen ergibt. Die Modellergebnisse zeigen, dass die Kobaltvorkommen im Zeitraum nach 2200 erschöpft sein werden, wenn abhängig vom den verwendeten Bevölkerungswerten am gegenwärtigen Szenarium nichts geändert wird. Die Kobaltverluste sind zu hoch, wenn man davon ausgeht, dass der Recyclinganteil nur durch Marktmechanismen verbessert werden soll. Unnötig viel Kobalt wird verschwendet, wenn keine Richtlinien aufgestellt werden. Wir können nur schlussfolgern, dass die Marktmechanismen allein weder das Ziel verfolgen noch über die notwendige Kompetenz verfügen, die Kobaltnutzung ohne gesetzliche Vorschriften und Regeln nachhaltig zu gestalten. Eine Richtlinienlösung auf wissenschaftlicher Grundlage wird zur Korrektur der aktuellen Situation benötigt, bevor es zu spät ist und zu viel Kobalt verloren gegangen ist. Sollte diese Aussage nicht berücksichtigt werden, besteht die Gefahr, dass der Gesellschaft eines Tages kein Kobalt mehr zur Verfügung steht. Zur Schonung von Kobalt und Sicherstellung, dass auch zukünftige Generationen darüber verfügen können, müssen die gegenwärtigen Richtlinien geändert und die großen beobachteten Verluste in den nächsten Jahrzehnten abgeschwächt werden.

Untermodul SPEZIALMETALLE: Molybdän und Rhenium (Abschnitt 4.1.7)

Die Ergebnisse des WORLD6-Modells zeigen, dass bei Molybdän immer eine leichte Verknappung bestehen wird. Allerdings wird es nach einer Anpassung des Preises an die Nachfrage keine schwere Verknappungen und Versorgungsausfälle geben. Die Simulationen weisen im Verlauf der Zeit einen ständigen Rückgang bei der Molybdän-Erzgüte aus. Die bekannten Reserven von etwa 14 Millionen Tonnen im Jahr 2015 erreichen ihren Höhepunkt mit 20 Millionen Tonnen im Jahr 2060. Wenn sich an der bisherigen Praxis nichts ändert, ist mit den Molybdän-Reserven eine ausreichende Versorgung bis etwa 2150 sichergestellt. Danach kommt es zu einem schnellen Rückgang. Bis 2150 werden praktisch alle verborgenen Ressourcen entdeckt und in die förderbaren Reserven einbezogen worden sein. Aus dem Modell geht hervor, dass bis 2250 alle bekannten Molybdän-Reserven ausgeschöpft worden sein dürften.

Die Modellsimulationsergebnisse legen nahe, dass Molybdän und Rhenium nur für weitere 150 Jahre reichen werden, wenn die gegenwärtigen Metallschutzparadigmen beibehalten werden. Diese Metalle sind so wichtig für Hochleistungslegierungen und hochmoderne Technologien, dass erheblich bessere Metallschutzbemühungen erforderlich sind als dies bisher der Fall ist. Die gegenwärtige Richtlinie, derzufolge „der Markt es schon richten wird“, ist es bisher ganz offensichtlich nicht gelungen, das Problem des Schutzes von Molybdän und Rhenium zu lösen. Dies verdeutlicht den Bedarf an regierungsseitiger Beteiligung bei der Erstellung eines fairen, stabilen und transparenten und nachhaltigen Marktes (Sverdrup et al., 2015c). Das Modell schneidet in Tests im Vergleich zu bekannten Daten gut ab und ist somit ein brauchbares Beurteilungswerkzeug. Das Modell prognostiziert einen deutlichen Rückgang bei der Versorgung mit Molybdän und Rhenium nach 2100, wobei es nach 2150 beim gegenwärtigen Recycling-Umfang zu einer schweren Verknappung kommen dürfte. Im Molybdän-Rhenium-System sind die Verluste zu groß für eine Nachhaltigkeit von mehr als 150 Jahren. An die meisten meisten chemischen Nutzungen schließt sich keine Wiederverwertung an. Bei metallischer Nutzung wird ein großer Anteil durch Verwässerungseffekte im Edelstahlschrott verloren.

Untermodul SPEZIALMETALLE: Niob und Tantal (Abschnitt 4.1.8)

Die Modellergebnisse weisen im Verlauf der Zeit einen ständigen Rückgang bei der Erzgüte von Niob und Tantal aus. Aus den Ergebnissen geht hervor, dass das WORLD6-Modell in der Lage ist, die beobachteten Trends bei den Abbaukosten, beim Marktpreis und der Erzgüte zu erfassen. Auf der systemweiten Ebene legen die Ergebnisse nahe, dass in den meisten chemischen Nutzungsbereichen von Tantal und Niob keine anschließende Wiederverwertung stattfindet. Der gegenwärtige Mangel an konsequenten Bemühungen zum Schutz von Niob und Tantal sorgen dafür, dass die nicht nachhaltige Versorgungssituation bestehen bleibt. Langfristig wird eine „Weiter-so-Vorgehensweise“ bei diesen Metallen zu einer Verknappung führen. Gegenwärtig gehen viel zu große Mengen von Tantal unwiderruflich verloren oder gehen aufgrund einer vereinfachten Elektronikschrottverarbeitung im Kupfer unter. Bei metallischer Nutzung wird ein viel zu großer Anteil von Niob durch Verwässerungseffekte im Edelstahl verloren.

Untermodul SPEZIALMETALLE: Wolfram (Abschnitt 4.1.9)

Man kann von einer zukünftigen Erhöhung der Nachfrage nach Wolfram ausgehen, weil es bei vielen hochmodernen Technologien mit technischen Vorteilen verbunden ist. Bei der Beurteilung der Wolfram-Verknappung sind zwei Parameter von besonderer Bedeutung: die weltweite Durchschnittsversorgung pro Kopf und Jahr (unverzichtbar bei der Beurteilung des Wachstums, ob eine globale Stagnation oder ein globaler Rückgang zu erwarten ist) und die weltweiten durchschnittlichen Lagerbestände pro Kopf (unverzichtbar bei der Beurteilung, ob wir die weltweite Selbstversorgung steigern können, ob eine globale Stagnation oder ein globaler Rückgang zu erwarten ist).

Aus den Simulationsergebnissen geht hervor, dass eine sanfte Verknappung von Wolfram etwa 2030 und aufgrund der Ressourcenerschöpfung eine schwere Verknappung nach 2200 eintreten wird. Die Lagerbestände pro Kopf gehen nach 2100 zurück, zunächst langsam, dann schnell (nach 2220) und die Selbstversorgung ist nicht mehr vorhanden. Nach 2200 wird es ein schwerwiegendes Wolfram-Verknappungsproblem geben.

Untermodul SPEZIALMETALLE: Superlegierungen (Abschnitt 4.1.10)

Die WORLD6-Modellergebnisse dienten zur Beurteilung der vorhandenen Mengen Molybdän, Rhenium, Niob, Tantal, Kobalt und Nickel zur Herstellung von Superlegierungen unter Berücksichtigung der technischen Verwendung in der Raketentechnik, in Hochleistungsturbinen und in hochmodernen Luftstrahltriebwerken. Die Bewertung läuft hinaus auf:

1. Militär- und strategische Großmachtaspekte:
2. Wenn Metalle wie Tantal, Niob, Molybdän, Rhenium und Hafnium den Technologien nicht zur Verfügung stehen, die Superlegierungen für ihre Funktion benötigen, dann wird es auch keine ausreichende Anzahl hochmoderner Luftstrahltriebwerke für die 5. und 6. Generation der Kampfflugzeuge geben. Staaten, die diese Technologien in großen Mengen benötigen, können dann nicht mehr auf diese Art von Technik zugreifen. Das könnte einen Verlust der militärischen Überlegenheit nach sich ziehen, den einige wenige Staaten gegenwärtig genießen. Andere ausgereifte militärische Ausrüstungen hängen bei der Sicherstellung ihres Vorsprungs ebenfalls von diesen Metallen ab, die dann nicht mehr im bisher gewohnten Umfang zur Verfügung stehen werden.
3. Mehrere Arten von Antriebsdüsen für Flugkörper und Raketentriebwerke werden ohne zumindest die 5. Generation der einkristallinen Superlegierungen nicht mehr hergestellt werden können. Dadurch wird die Leistung der militärischen Ausrüstung eingeschränkt und das militärische Potenzial von Staaten ohne sicheren Zugriff auf diese Metalle geschwächt.
4.
5. Technologische Aspekte:
6. Die Gefahr einer Einschränkung für die hochmoderne Luftstrahltriebwerk-Technologie wirkt sich ernsthaft auf die Entwicklung von Flugtechnologien und Hochtemperatur-Strahlturbinen aus, die für verbrauchsarme Triebwerke benötigt werden. Superlegierungen ohne die wichtigsten Metalle (hergestellt aus Molybdän, Niob, Tantal, Metalle der Platingruppe, Zirkonium, Hafnium, Kobalt, Chrom, Nickel (bei denen in den nächsten Jahrzehnten eine Verknappung eintreten könnte, wobei Zirkonium und Chrom die einzigen Ausnahmen sind) erzielen definitiv eine schwächere technische Leistung. So könnte daraus in Zukunft ein Problem in einer Welt mit höheren und schwankenden Kohlenwasserstoffpreisen entstehen. Außerdem stehen diese oben erwähnten Metalle auch nur spärlich zur Verfügung.
7. Einige ausgesprochen korrosionsfeste Legierungen können mit Superlegierungen hergestellt werden, die Molybdän, Tantal, Niob, Metalle der Platingruppe oder Rhenium enthalten, die wichtig für die Eindämmung von zersetzenden geschmolzenem Salz sind, das Halogenide enthält. Beispiele für denkbare Einsatzbereiche:

8. In mit geschmolzenem Salz gekühlten Anlagen, wie sie in hochmodernen Kernreaktoren zu finden sind. Diese sind wichtig für die Entwicklung der nächsten Generation der mit geschmolzenem Salz gekühlten Kernkraftwerke auf der Grundlage von Uran 238 oder Thorium.
9. Sie haben auch eine große Bedeutung für die Entwicklung von Turbinen zur Stromerzeugung für die Zivilgesellschaft bei höheren Temperaturen zur Steigerung des Wärmewirkungsgrads.
10. Sie werden auch für die Entwicklung von inerten Elektroden zur Schmelzung von Leichtmetallen wie Magnesium oder Aluminium und zur Herstellung von Silizium und Ferrosilizium benötigt.

Untermodul SPEZIALMETALLE: Metalle der Platin-Gruppe (Platin, Palladium, Rhodium) (Abschnitt 4.1.11)

Unsere Einschätzung der förderbaren Mengen (Ressourcen) bei den Metallen der Platin-Gruppe sind doppelt so hoch wie bisher erwartet und geschätzt. Dieses Ergebnis wirkt sich nicht auf die Tatsache aus, dass die Metalle sehr selten sind und in der überschaubaren Zukunft auch knapp bleiben werden. Die Bewertung geht von einer schlussendlich förderbaren Ressourcenmenge (URR) von etwa 216.000 Tonnen Metalle der Platin-Gruppe aus, wobei Platin bei etwa 99.000 Tonnen und Palladium bei etwa 93.000 Tonnen liegt. Dabei wurden sowohl Primärressourcen als auch Ressourcen berücksichtigt, die aus davon abhängigen Sekundärrohstoffen stammen. Diese Sensitivitätsanalyse legt nahe, dass der oben genannte Ressourcenumfang der wahrscheinlichsten Schätzung entspricht.

Die Simulationsergebnisse des WORLD6-Modells ergeben, dass die Spitzenproduktion der Bergwerke in den Jahren 2035 bis 2050 liegen, während der Spitzenwert bei der Versorgung der Gesellschaft etwa 20 bis 35 Jahre später eintritt. Die Modellergebnisse zeigen, dass die Diffusions- und Dissipationsverluste gegenwärtig bei den Metallen der Platin-Gruppe auf der weltweiten systemweiten Ebene zu groß sind. Die Schutzanstrengungen müssten bei einer langfristigen Perspektive gestärkt werden. Weil die Erzqualität und die Förderkosten bei dem Modell berücksichtigt wurden, erhöht sich der Aufwand pro geförderter Metallmenge der Platin-Gruppe im Verlauf der Zeit, bis die Förderung schließlich eingestellt wird. Zusammenfassung auf der Grundlage der Modellergebnisse:

1. Produktion von Versorgung und Förderung
2. Die Förderung der Metalle der Platin-Gruppe erreicht ihren Spitzenwert in den Jahren 2035 bis 2050 und die Marktversorgung ist in den Jahren 2070 bis 2080 am höchsten. Die Verzögerung zwischen Förderung und Versorgungsspitzenwert hängt vom Recyclingumfang und der relativen Größe der Verluste im Verhältnis zu den Fördermengen ab.
3. Reserven und Ressourcen
4. Die gesamte förderbare Menge bei den Metallen der Platin-Gruppe unter Berücksichtigung der neuesten Ressourcen-Schätzungen, der neuesten Technologie im Bergbau in großen Tiefen und der Ressourcen in großen Tiefen liegt bei etwa 220.000 Tonnen.
5. Die bekannten Reserven liegen bei etwa 60.000 Tonnen und nehmen nach 2030 langsam ab.
6. Das Verhältnis zwischen der kumulierten Menge zur Versorgung der Gesellschaft und der kumulierten Menge der Primärförderung beträgt etwa 2,6 und wird Faktor X genannt.

Abschnitt 4.2 von Kapitel 4 beschreibt das Material-Untermodul des WORLD6-Modells. In diesem Bericht werden lediglich Sand, Kies, Schotter und Stein als Bestandteile des Material-Untermoduls beurteilt.

Die Simulationsergebnisse unter der Voraussetzung, dass sich am aktuellen Szenarium nichts ändert, weisen aus, dass die Produktion von Quadersteinen ihren Höchstwert etwa 2020 bis 2030 erreicht und danach langsam sinkt. Ursache dafür sind die Tatsache, dass die Nachfrage die Fördermenge übersteigt, und die langsame Erschöpfung der bekannten Reserven an hochwertigen Steinen. Sand und Kies bewegen sich auch auf einen Höchstwert zu und erreichen ihre maximale Produktionsrate in den Jahren von 2060 bis 2070. Gründe für diese Entwicklung sind einerseits der erwartete Bevölkerungsspitzenwert im Jahr 2065 und andererseits der Preis für Sand und Kies, der die Nachfrage einschränkt.

Obwohl die Versorgung weltweit betrachtet unerschöpflich zu sein scheint, sind Nachschub und Verfügbarkeit bereits jetzt ein wachsendes Problem auf lokaler und regionaler Ebene. Daraus lässt sich ableiten, dass der weltweite Handel mit Sand, Kies und Stein weiter zunehmen wird. Wir benötigen bessere Daten über die Produktionsraten, die Nutzung und das Recycling, damit wir die Risiken der regionalen Verknappung besser beurteilen können, und außerdem ein Modell mit einer Aufteilung in verschiedene Regionen. Der Stand der Forschung lässt dies gegenwärtig nicht ohne eine erhebliche Erhöhung der Forschungsmittel zu. Das Material-Untermodul des WORLD6-Modells könnte mit besseren Daten dafür genutzt werden, weil es sich in Regionen unterteilen lässt.

Abschnitt 4.3 von Kapitel 4 legt das Energie-Untermodul des WORLD6-Modells fest. In diesem Bericht wurden im Rahmen des Energie-Untermoduls lediglich fossile Brennstoffe (Kohlenwasserstoffe) bewertet.

Nur ein Bruchteil der weltweit nachweisbaren Vorkommen fossiler Brennstoffe kann gefördert werden. Dies ist einer der Hauptgründe für die weit auseinander gehenden Schätzungen der Ressourcen. Für die Differenzen bei den Schätzungen der Ressourcen gibt es mehrere Gründe:

1. Ein erheblicher Teil der Ressourcen ist hinter einer Energiebarriere verborgen. Die Förderkosten sind größer als der durch die Fördermenge entstehende Nutzen. Sobald die Förderkosten die Einnahmen überschreiten, wird die Förderung eingestellt.
2. Ein erheblicher Teil der bekannten Ressourcen wird hinter einer Technologiebarriere ausgeblendet. Die Ressourcen befinden sich an Orten, an denen die entsprechende Fördertechnologie weder jetzt noch in der vorhersehbaren Zukunft vorhanden ist. Die Ressourcen sind unerreichbar.
3. Ein erheblicher Teil befindet sich hinter einer Kapitalbarriere. Die Ressourcen können theoretisch gefördert werden, aber es gibt nicht genug Kapital für die notwendigen Investitionen, und auch keine flüssigen Mittel für den Kauf des Produktes zum damit verbundenen Preis. Fossile Kohlenwasserstoff-Brennstoffe sind unerreichbar, sobald der Ölpreis 1.700 $ pro Tonne Öl erreicht.
4. Ein erheblicher Teil der Ressourcen ist hinter einer Umweltzerstörungsbarriere verborgen. Die Ressourcen sind von so schlechter Umweltqualität oder die Förderung und Nutzung haben so verheerende Umweltauswirkungen, dass die Förderung undenkbar ist.

Der Welt stehen die fossilen Brennstoffe also nicht deshalb mehr zur Verfügung, weil es keinen fossilen Kohlenstoff mehr im Boden gibt. Die Produktion wird eingestellt, weil die Auswirkungen der weiteren Förderung nicht akzeptabel sind.

Kapitel 5 enthält die Gesamtschlussfolgerungen, dass die meisten Versorgungsspitzen bei Metallen, Materialien und fossilen Brennstoffen im Zeitraum von 2040 bis 2060 eintreten werden. Dies wird zu einigen der schwersten industriellen, politischen und sozialen Herausforderungen unserer Zeiten führen und sorgfältige Vorbereitungen und Forschungsarbeiten erforderlich machen, damit die Funktionsfähigkeit der Gesellschaft nicht zerstört wird. In Bezug auf die Nutzung der Materialressourcen entspricht eine Fortsetzung der aktuellen Vorgehensweise dem Worst-Case-Scenario, das zu Problemen der Materialverknappung führt und ein erhebliches Potenzial für gesellschaftliche Verwerfungen in sich birgt. Mit mehr Recycling könnte der Spitzenwert bei der Förderung der Ressourcen weiter in die Zukunft verlagert werden. So würde der Faktor X erhöht, d. h. die Häufigkeit der Nutzung einer Gewichtseinheit vor ihrem endgültigen Verlust.

1 INTRODUCTION

1.1 Background

The overall objective of the systems dynamics modelling workpackage in the SimRess project was to further develop the WORLD model and use it as a decision support tool in simulating potential resources scarcities and testing alternative resource efficiency policy approaches.

1.2 Purpose and scope

The aim of this report is to present the overall structure of the existing version of the WORLD model (WORLD6) and provide a detailed description of the resources module by presenting simulation results and integrated assessments for each of the individual resources modelled.

1.3 Report structure

Each of the following chapters present an individual resource sub-module including brief background information on the resource, underpinned hard data (eg. known individual metals and mineral reserves) used in the model, key model results and vericifaction on available historical data (eg. extraction amounts, price) for the period 1900-2010, as well as major discussions and conclusions.

2 WORLD MODEL OVERVIEW

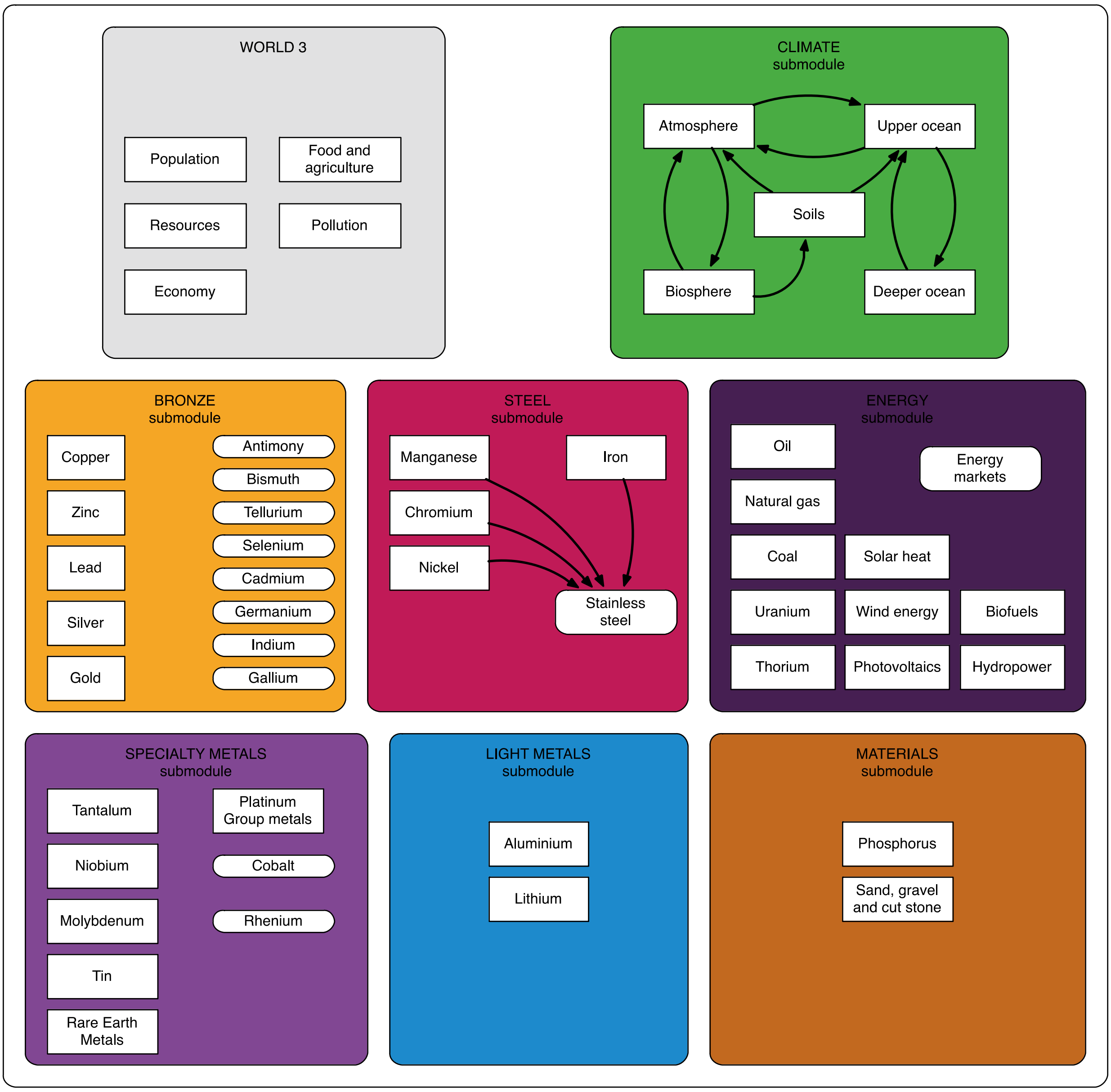
2.1 Introduction

The WORLD model is a system dynamics model that runs iteratively to reflect the complex interactions between population, economy, food production, ecology and resources over an extended time span and aggregated to a global level i.e. all the parameters are global totals or averages, rather than modeling any differentiations between geographies. The WORLD model consists of several (sub-) modules, four of which (population, economy, land/food, and ecology) are identical to the ones presented in the latest instance of the World3 Model (Meadows et al. 2005), which is not much changed from the original version (Meadows et. al., 1972). The data used in World3 model is of two types including historical data of key levels such as population and arable land, and other data derived from the modeling group's own analysis and/or other academic work that support the trends and interrelationships in the model (see Meadows et. al., 1974 for full reference).

Incorporating World3 model in its structure, and enhancing it with a more detailed resources module and an additional simple climate module considering the CO2 emissions due to use of non-renewable energy resources, the current version of the model, WORLD6, provides more detailed simulation results on the global supply of various resources, including metals (i.e. BRONZE (Copper, Zinc, Lead, Silver, Gold and the dependent metals Antimony, Bismuth, Cobalt, Gallium, Germanium, Indium, Cadmium, Tellurium, Selenium), STEEL (Iron, Chromium, Manganese, Nickel, stainless steel), LIGHT METALS (Aluminium, Lithium), SPECIALTY METALS (Platinum group metals, Molybdenum, Rhenium, Niobium, Tantalum, Tin, Rare Earth Metals), materials (i.e. phosphorous, stone, sand, gravel) and energy sources (i.e. oil, coal, and gas), as well as on the energy consumption associated to extraction of these resources (Figure 2.1).

This report provides a general overview of the WORLD model and a detailed documentation of its resources module in the following sections. For a full description of the World3 model we refer the readers to Meadows et. al., 1974.

Figure 2.1. Schematic overview of the modules included in the WORLD6 model



Source: Own Figure, Lund University/University of Iceland

2.2 Methodology

Several methods were used in this study including systems analysis and system dynamics for modelling, and traditional assessment methods for appraisal of resources and reserves for mass inputs.

2.2.1 Systems analysis and system dynamics

In this study, we employed the standard methods of systems analysis and system dynamics modelling (Roberts et al., 1982, Haraldsson and Sverdrup 2005, Haraldsson et al., 2006, Forrester 1971, Meadows et al., 1972, 1992, 2005, Senge 1990).

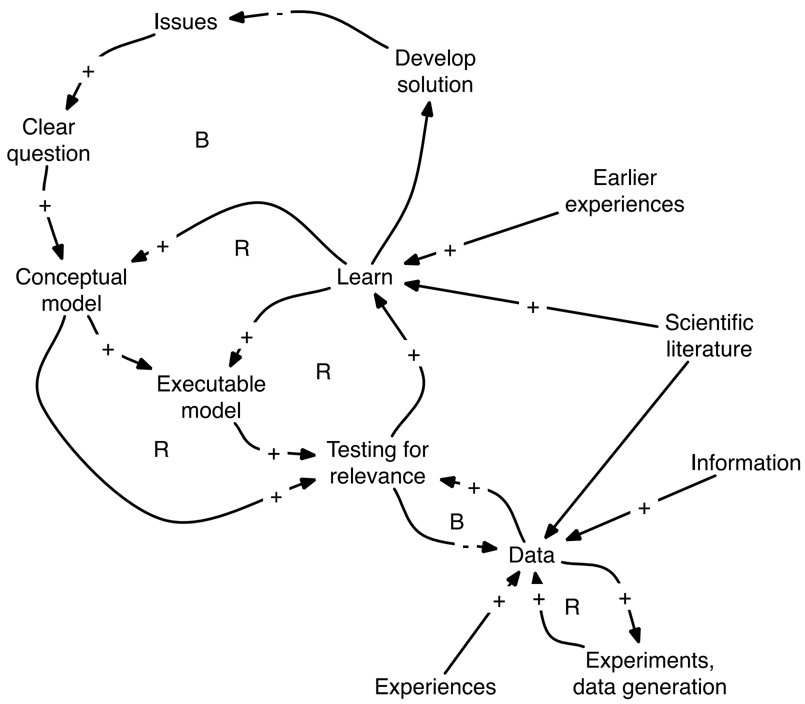
The methodology for modelling follows the order of events as shown in Figure 2.2. Modelling always start with a precise problem statement, followed by a conceptualization, and in that order. Systems analysis is used as the standard methodology for conceptualization. Regardless of computational method chosen (in our case system dynamics modelling), the modelling must without exception be preceded by the conceptualization step.

Figure 2.2. Modelling always start with a precise problem statement, followed by a conceptualization.

Source: Own Figure, Lund University/University of Iceland

We analyse the systems using stock-and-flow charts and causal loop diagrams. The learning loop(Figure 2.3) is the adaptive learning procedure followed in our studies (Senge 1990, Kin 1992, Senge et al., 2008). The conceptualization is where the actual model is developed. The model is the bearer of the knowledge employed and must be completely clear before any computational work can be undertaken. It is fundamental to understand that the causal understanding is the model.

Figure 2.3. The learning loop is the adaptive learning procedure followed in our studies (Senge 1990, Senge et al., 2008).



Source: Own Figure, Lund University/University of Iceland

Systems analysis produces system mapping in terms of causalities (Causal loop diagrams) and flows (Flow charts). Together these define the causalities and flow-paths and the structure of the system. These different types of system maps need to be fully internally consistent, and constitute the design plans for the computational system used.

“Data” enters the procedure late in the process (Figure 2.3). Data comprises both numbers that quantify system states, changes and sizes, as well as structural and contextual information, and qualitative information. If “collecting all the data” takes place before the problem statement and conceptualization, then the data search will be a random effort, and most of the effort risks being wasted and unnecessary. The process must then be stopped and restarted from a problem statement. Data collection occurs simultaneously with the construction of the computations system. Please note that the steps problem statement and conceptualization needs to take place regardless of computational method, including back-of-the-envelope calculations or using statistics packages.

For the computations, system dyanmics modelling is applied by using graphical high level programming software STELLA® by ISEE systems. The entering of the code follows from the causal loop diagrams and flow charts developed in the conceptualization stage. The software tools have no conceptualization power of their own. The computational modelling carries out calculations according to the instructions derived from the conceptual model, only and uniquely that and nothing else. The mass balance expressed differential equations resulting from the flow charts and the causal loop diagrams are numerically solved using the STELLA® modelling environment (Sterman 2000, Senge 1990, Senge et al. 2008, Haraldsson and Sverdrup 2005, 2017, Sverdrup et al. 2014a,b, 2015a,b, 2016a,b). The "data" is divided on harvest into several different categories:

5. System boundary and initial conditions
6. System structures
7. System parameters settings
8. System states

Of these, categories 1-3 are used to parameterize the model before the simulations start. The state data (4), are not used for model initialization, but saved and used for evaluation of model performance. A major feature is to map up how well the embedded understanding actually reproduces the observed development in systems states, and where the actual deviation brings an important message. It is not the objective through extensive calibration of parameters to get a maximum likeness to the system output, as is sometimes done with calibrated statistical models.

2.2.2 Modelling of individual resources submodules

For each of the resources, we used systems analysis to conceptually model the production, trade and consumption system. The flow pathways and the causal chains and the whole feedback structure of the system are mapped using causal loop diagrams and flow charts. The resulting coupled differential equations are transferred to computer codes for numerical solutions.

For all of the resources first a stand-alone model was developed and tested. When that model worked satisfactorily, it was then integrated as a submodule in the WORLD6 model structure. The model is used to first reconstruct the past (1900-2015) to assess performance and robustness of the model. When the performance is satisfactory, then the model is used to simulate the possible future (2015-2400), having the support of being able to reconstruct the observed past pattern for 115 years.

The model was developed iteratively, by repeatedly testing it on data gathered from the literature and corporate sources. This methodology was implemented through out the study and when developing the model. The iterations were used to set the parameterization to such values that the mining history, observed ore grades and price picture could be reproduced. This allows us to see where the intervention points in the system are, and to propose policy interventions (Mason et al. 2011, Haraldsson and Sverdrup 2004).

2.2.3 Defining scarcity

Before we assess resource scarcity and supply sustainability, we need to define what scarcity is in operational terms. Scarcity is defined as:

- Soft scarcity, where shipments into the market do not match the demand for delivery from the market, leading to an adjustment of price up, reducing demand and sometimes increasing the amount offered to the market. This down-sized demand is the sign of soft scarcity. The adjustment goes on until the offering into the market matches the demanded amount. The presently implemented open market economic mechanisms work well to handle soft scarcity. The previously considered command-and-control regimes did not cope well with scarcity, but had a tendency to worsen the situation.
- Hard scarcity has two variants:

- Economic scarcity, where the demand is downsized by the increased price, but where the demand gets cut to zero by lack of ability to pay, while there is still material available (at high prices). If the use of the resource is a major bulk resource for society, then economic crisis will be the result if the resource is a keystone resource for the economy. Oil and coal are two such examples. If it is for a specialized but not essential technology, then failure of provision or/and technical regression will be the result.
- Physical scarcity is where the supply cannot sufficiently cover the demand. There will be a demand gap, a difference between the adjusted demand and what really can be supplied. Even if money to pay whatever is asked, the material is not available. This is something that we have observed with platinum and rhodium from time to time in platinum metal trading. Failure of provision is the final result. Rhodium is an example where this has occurred on frequent occasions.

Scarcity can be offset by increasing supply into the market with increased recycling, or by reducing demands by substitution with other materials. Increasing recycling has both a price and a social component. Substitution is sometimes possible, but often not without issues of functionality. Scarcity is not a new phenomenon, but rather a standard component of economic systems.

3 ON THE METAL CONTENTS OF OCEAN FLOOR NODULES, CRUSTS AND MASSIVE SULPHIDES – AN ASSESSMENT OF THE EXTRACTABLE AMOUNTS

3.1 Introduction

There is concern for the long term sustainability of metal supply to the world. Therefore mining the ocean seafloors have been proposed. Many authors have highlighted risks of resource depletion looming (Meadows et al 1972, 1974, 2005, Hawkins 2001, Bardi 2013, Johnson et al. 2007, Eliott et al. 2014, Elshaki and Graedel 2013, Heinberg 2001, 2011, Mohr et al. 2014, Northey et al. 2014, Kwatra et al. 2012, Nassar et al. 2012, Ragnarsdottir et al. 2011, 2012, Sverdrup and Ragnarsdottir 2014, Sverdrup et al. 2013, 2014a,b,c, 2015a,b,c, Dawkins et al., 2012, UNEP 2011a,b,c,d, 2012, 2013a,b,c, Prior et al., 2012, Northey et al., 2014, Alonso et al., 2007, Ragnarsdottir et al. 2012, Sverdrup et al. 2013, UNEP 2011a,b,c,d, 2012, 2013a,b,c). Metals on the seafloor (Managese nodules, cobalt crusts, massive sulphide deposits) sometimes suggested as a calming message against metal resource scarcity concerns, a "solution-to-all-worries". We would like to inspect a bit closer how realistic this message is, and attempt to substantiate this to tonnage numbers for realistic extractable amounts. We have assessed the land-based resources extensively (Sverdrup and Ragnarsdottir 2014), and thus, time has come to ask if there is some substance in mining the ocean seafloors and how much metal can we reasonably well expect to get out of it. Nodules are found on the flat sea floors, cobalt crusts are associated with volcanism and seamounts, and massive sulphides with deep sea brines, and deep sea hydrothermal wells associated with techtonic division lines, such as the mid Atlantic Ridge.

3.2 Scope

To preliminarily asses the amount of extractable resources available on the seafloor in relationship to the resources available on land for some key infrastructural and technology metals; Iron, manganese, copper, nickel, cobalt, molybdenum, platinum group metals (PGM), silver and gold. We would consider the impact of setting different assumptions on technical feasibility of actually doing extraction from the Ocean floor, using literature descriptions of the available and potential future technology. The results will be used for helping set the right input resource sizes for iron, manganese, copper, nickel, cobalt, molybdenum, platinum, gold and silver in the WORLD 6 model being developed by the authors (Ragnarsdottir and Sverdrup 2011, Sverdrup and Ragnarsdottir 2014, Sverdrup et al., 2015a,b, 2016a,b).

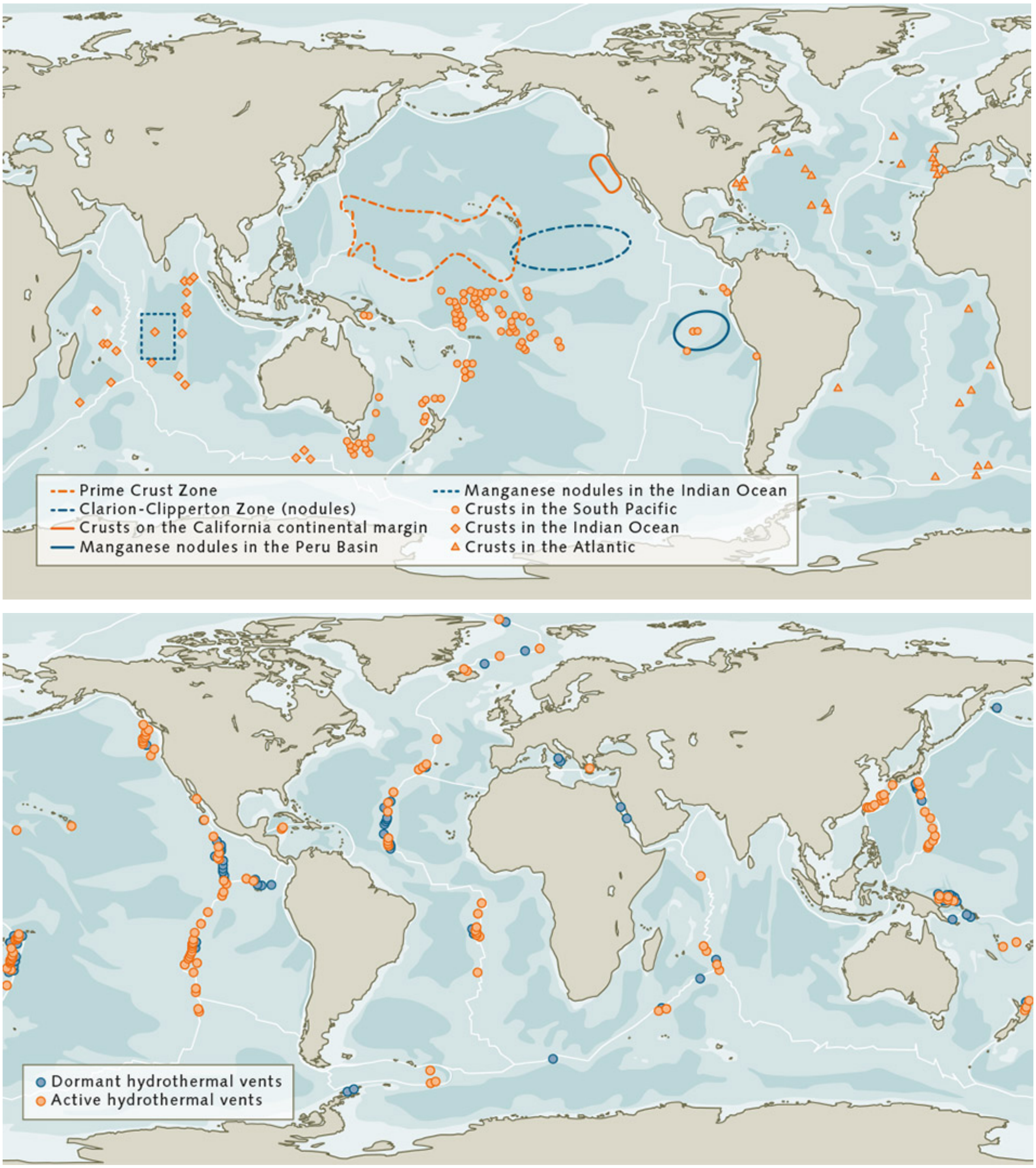
3.3 Methods

The method is to use a literature review and harvest it for data, assessments and evaluations, compiling estimates of how large the technically extractable amount in the oceans will be, grading these with estimated extraction costs and degrees of extractability. We will make simple mass balances in tables and overview tables, using extractability estimates to gain recoverable resources from metal detected from any type of deposit. We will be studying three types of deposits:

1. Manganese nodules on the main sea floor
2. Cobalt crusts on submerged seamounts
3. Massive sulphide deposits associated with black smokers and deep sea hydrothermal activity along mid-ocean ridges and tectonic separation- and subduction-zones

Figure 3.1 indicates the approximate location of detected locations of ocean floor nodules, cobalt crusts and massive sulphides. We will use earlier resource estimates by the authors for these metals based on what is available on land (Ragnarsdottir et al., 2012, Sverdrup 2011, Sverdrup et al., 2012a,b, 2013, 2014a,b,c, 2015a,b,c, 2016a,b,c,d). These were in turn based on a number of scientific studies: Alonso et al., 2007, Alonso 2010, Bardi 2013, Sverdrup and Ragnarsdottir 2014, Berger et al., 2011, Brown et al., 2015, Cailtreaux et al., 2005, Crowson 2011, Dalvi et al., 2004, Dawkins et al., 2012, Davis 2000, Drake Resources 2015, Eckstrand and Hulbert 2007, Eilu 2014, Gordon et al., 2006, Graedel et al., 2004, 2011, Graedel and Erdmann 2012, Geoscience Australia 2009, Hagelücken and Meskers 2009, Halada et al., 2008, Harper et al 2012, Johnson et al., 2007, 2014, Keseler et al., 2013a,b, Kwatra et al., 2012, Mohr et al., 2014a,b, Mudd 2007, 2009a,b, 2010a,b,c, 2012, Mudd et al., 2013a,b, Mudd and Iowitt 2014, Nassar et al., 2012, Newman 2011, Nickless et al., 2014, Norgate and Rankin 2002, Northey et al., 2014, Nuss and Eckelmann 2014, Slack et al., 2010, Shedd 2015, Tilton and Lagos 2007, UNEP 2011a,b, 2012, 2013a,b,c, US EPA 1994, US Department of the Interior 1980, USGS 2014, Wagner and Fettweiss 2001, Walter 2014).

Figure 3.1. Location of detected locations of ocean floor nodules, cobalt crusts and massive sulphides. As can be seen from the maps, these can be found in specific areas of the world's oceans, depending on their geological history.



Source: Modified after Boschen et al. (2013) and World Ocean Review (2013).

3.4 Materials and data sources

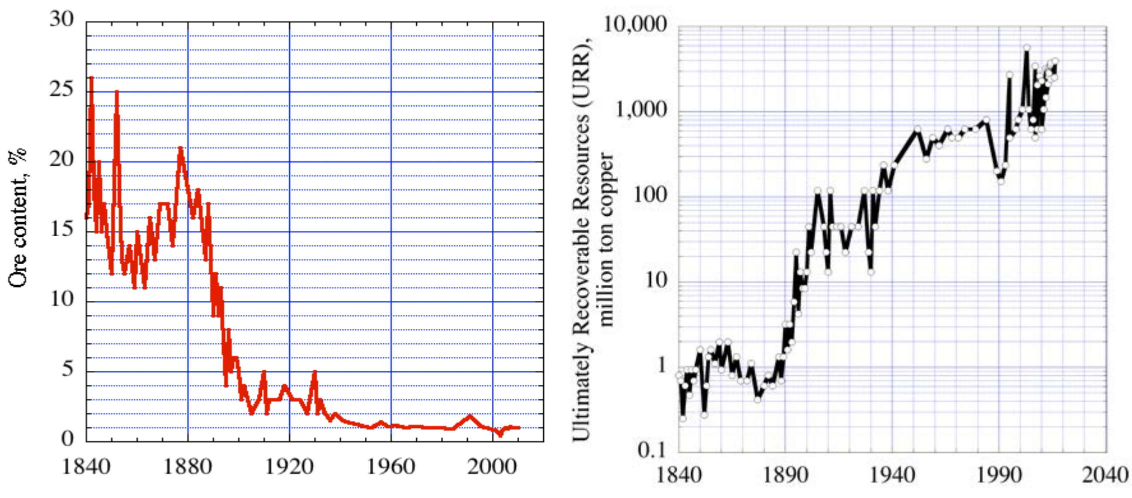
A number of articles have reviewed the amount of resources and their locations in the World's Oceans (Allsopp et al., 2013, Beaudoin et al., 2014, Beckmann 2007, Berger et al., 2011, Bertram et al., 2011, Boschen et al., 2013, Clark et al., 2013, 2010, Fouquet and Scott 2009, Hein 2004, Hein et al., 2009, 2010, 2013, Herzig and Hanninton 1995, Hoagland et al., 2009, Kojima 1999, Mudd et al., 2013, Muinos et al., 2013, Roberts 2012, Schmidt 2015, Smith and Heydon 2013, SPC 2012, Tilton 1983, Zhou 2007, International Seabed Authority 2012, World Ocean Review 2013a,b,c). Table 3.1 shows a general relationship between ore grade, the approximate production cost and minimum supply price to society, as well as impact of price on the recycling in market supply (Adapted after Sverdrup et al., 2015a,b,c, Wellmer 2008, Phillips and Edwards 1976). The cost estimates were generalized and approximated by the authors using literature data. The metal resources were stratified according to such an extraction cost scheme on the ocean floor. Boschen et al., 2013 and Hannington et al., (2014) locates where the deposits have been found.

Table 3.1. General relationship between ore grade, the approximate production cost and minimum supply price to society (Adapted after Sverdrup and Ragnarsdottir (2014), Sverdrup et al., 2014a,b, 2015a,b).

Ore grade	Metal content %	g/ton	Extraction yield, %	Land extraction cost, $/kg	Ocean floor extraction cost, $/kg	Land energy need, MJ/kg
Rich	40-5	400,000-50,000	100	4-15	15-28	30
High	5-1	50,000-10,000	99	15-28	28-50	33-64
Low	1-0.2	10,000-2,000	98	28-50	50-100	120-160
Ultralow	0.2-0.04	2,000-400	91-95	50-100	100-160	400-600
Extralow	0.04-0.01	400-100	80-91	100-160	160-400	700-1,100
Trace	0.01-0.002	100-20	75-80	160-400	400-1,200	3,000-20,000
Rare	>0.002	20-4	55-75	>400	>1,200	>20,000

Hannington et al., (2014), Hein et al., (2012) and Beaudoin et al., (2014) made efforts to estimate the size of the deposits and their average contents (ore grades). Roberts (2012) assesses the location and amount of metals in sulphides. Many samples have been taken from the sea floor, however, how well they actually cover the territory is only known on an overview level. We estimate that this is still sufficient for a preliminary estimate. Figure 3.2 shows the development of copper ore grade with time (Data compiled by Sverdrup et al., 2014, 2017). Most ores mined today have an ore grade of 1% or lower, whereas ocean floor massive sulphides have ore grades from 3-15% copper content. Diagram (a) shows the decline in ore grade in the land-based ultimately recoverable resource for copper. Several other metals (For example iron, nickel, zinc, gold or platinum) show similar patterns. Copper contents in nodules and cobalt crusts are normally lower (Table 3.1). At the same time, the ultimately recoverable resources of metals like copper is converging on a fixed and finite number. The land-based resource is somewhere between 3.5 and 4 billion ton of copper (b). This is what the human society can expect to be able to extract.

Figure 3.2. Development of copper ore grade with time Diagram (a) shows the decline in ore grade observed for land-based copper deposits. Diagram (b) shows how the size of the recoverable copper resource is converging on a finite number somewhere around 4 billion ton copper. This is what the human society can expect to be able to extract, and what it will have to make the best of.



a b

Source: Own Figure, Lund University/University of Iceland

In deep sea mining, the mining extraction operation and actual hauling the material up and processing it out at sea cause larger costs than similar operations on land. Waste management will also be more expensive if serious environmental damage is to be avoided. The different types of metal resources are found at different depths in the oceans;

1. Manganese nodules on the sea floor are found from 1,800 to 4,000 meter depth
2. Cobalt crusts on submerged seamounts are found on 400 to 4,000 meter depth
3. Massive sulphide is found on 800 to 3,500 meter depth in connection with hydrothermal activities, but some deep trenches also develop warm salt brines such as in the Red Sea.

These factors all conspire to make the mining operations technically challenging and expensive. The southern and northern Polar Regions are at the moment off limits because of ice conditions and the rough environment. The Southern Seas are also largely off limits because of their storm-infested waters and bad weather. Part of the Penrhyn area in the Pacific Ocean is host to some of the largest coral reefs on the planet, and large scale mining in significant parts of the area may not be such a good idea from an environmental point of view (Allsopp et al., 2013, Beckmann 2007, Smith and Heydon 2013, Zhou 2007).

3.5 Results

The results have been compiled in Table 3.2, 3.3 and 3.4. Table 3.2 shows the area cover, metal ore density and contents of different metals in the different nodules, crusts and massive sulphides on the ocean floor. Table 3.3 shows the estimated seafloor metal resources, before assessment of technical extractability. Table 3.4 shows the estimated seafloor metal resources, coming to assessment of technical extractability. We found that significant amounts are present located in mainly 5 limited regions of the oceans for all the metals considered, two areas in the Pacific Ocean, One in the Atlantic Ocean and one in the Indian Ocean. Most of the tonnage of the interesting metals in nodules and black smokers are located at great water depth, from 1 km to more than 4 km depth, involving huge technological challenges. Only a smaller fraction (5-25%) of the detected amounts can be considered to be extractable under the most favourable of conditions (Mudd et al., 2013, Allsopp et al., 2013).

Table 3.2. Area cover, metal ore density and contents of different metals in the different nodules, crusts and massive sulphides on the ocean floor. The estimates are very approximate.

Area	Size	Density	Amount	Fe	Mn	Cu	Zn	Ni	Co	PGM	Au	Ag
	mill. km2	kg/m2	billion ton	% weight						g/ton		
Deep sea nodules												
Clarion-Clipperton	9.0	2.3	21	18	22	0.2	0.4	0.4	0.5	5	1	10
Peru Basin	4.5	10	45	18	22	0.2	0.4	0.4	0.5	5	1	10
Penrhyn	0.75	25	19	18	22	0.2	0.4	0.4	0.5	5	1	10
Atlantic	0.20	25	4.8	18	22	0.2	0.4	0.4	0.5	5	1	10
Indian Ocean	0.75	5	3.8	18	22	0.2	0.4	0.4	0.5	5	1	10
Cobalt crusts												
Northeast Pacific	9.0	0.84	7.55	17.8	22.9	0.2	0.4	0.5	0.8	3	20	200
Southwest Pacific	4.5	0.7	3.2	18.1	21.7	0.1	0.2	0.5	0.8	3	20	200
Penrhyn	0.75	0.7	0.53	14.5	20.9	0.2	0.4	0.4	0.5	2	20	200
Atlantic	0.20	0.7	0.15	15.0	20	0.2	0.4	0.4	0.5	2	6	200
Indian ocean	0.75	0.7	0.53	17.0	22.3	0.23	0.4	0.5	0.6	2	20	200
Massive sulphides and brines												
Pacific CC+PB	13.5	0.22	3.000	10	0.2	3.5	13	0.1	0.7	1	3	150
Penrhyn	0.75	1.25	1.000	10	0.5	8.5	7	0.1	0.7	1	3	80
Atlantic	1.50	1.3	2.000	10	0.5	8.5	7	0.1	0.7	1	3	80
Indian Ocean	0.75	0.3	0.250	10	0.5	0.9	4	0.1	0.7	2	2	100

Table 3.3. Estimated seafloor presence of metal resources, before assessment of technical extractability. The estimates are very approximate.

Area	Fe	Mn	Cu	Zn	Ni	Co	Mo	PGM	Au	Ag
	Million ton							Thousand ton		
Nodules										
Clarion-Clipperton	6,000	5,990	226	452	274	44	12	15	52	520
Peru Basin	8,145	9,765	450	900	587	94	26	30	104	1,040
Penrhyn Basin	3,420	4,180	203	406	247	40	10	15	48	480
Atlantic	864	1,056	10	20	62	10	3	4	12	220
Indian Ocean	1,080	1,078	41	82	49	8	2	-	1	10
Sum	19,509	22,069	930	1,860	1,219	196	53	64	217	2,270
Crusts										
Northeast Pacific	1,344	1,714	7.4	1.8	32	50	3.5	12	2	17
Southwest Pacific	2,870	3,668	15.8	4.0	68	107	7.7	24	4	34
Penrhyn	77	111	1.2	2.4	2.1	2.7	0.3	2	-	2

Atlantic	2	27	0.4	0.6	0.5	0.7	0.8	0.5	-	0.5
Indian ocean	90	118	1.3	0.4	2.2	2.9	0.3	3	-	2
Sum	4,383	5,636	26.7	9.2	104.5	163.3	12.6	41.5	6	55.5
Sulphides										
Pacific	300	150	60	405	6	21	21	6	60	600
Penrhyn	100	50	20	49	2	7	7	2	20	200
Atlantic	200	10	170	140	4	14	14	4	40	400
Indian Ocean	25	12	5	10	0.5	2	2	-	5	50
Sum	625	222	85	604	12.5	44	44	12	125	1,250

Table 3.4. Estimated seafloor metal resources, coming to assessment of technical extractability. The estimates are very approximate.

Area	Fe	Mn	Cu	Zn	Ni	Co	Mo	PGM	Au	Ag
		Million ton						ton		
Sum nodule	19,509	22,069	930	1,860	1,219	196	53	63,000	217,000	2,270,000
Sum crusts	4,383	5,636	27	9	105	163	13	41,500	6,000	55,500
Sum sulfides	625	222	85	604	13	44	44	12,000	125,000	1,250,000
Sum	24,517	27,727	1,042	2,473	1,337	403	110	116,000	348,000	3,505,000
Considering different extractability in the total picture of all extractable metal resources. The URR estimates are very approximate										
Ocean, 25%	6,129	6,932	261	618	334	100	26	29,000	87,000	876,300
Ocean, 5%	1,226	1,386	53	124	67	20	6	5,800	17,400	175,260
Land; 2015	340,000	5,600	3,770	2,676	300	32	80	210,000	150,000	1,200,000
How much metal resources do we have on land and ocean floors? The estimates are very approximate										
URR low	341,226	6,986	3,823	2,800	367	52	86	215,800	167,400	1,375,260
URR high	346,129	12,532	4,032	3,294	634	132	106	229,000	237,000	2,074,300
% on land	99-98	81-41	98-92	96-81	82-49	62-24	94-79	97-91	90-66	87-58

3.6 Discussions

Despite huge optimism in the field, no profitable commercial operation has yet been started. Considering the difficulty of mining, it seems that based on an ore grade to metal price evaluation, that only cobalt or silver would perhaps be interesting on a commercial level, with by-products of copper, gold, perhaps also molybdenum and nickel. There are few assessments of how much of the detected seafloor deposits that can actually be extracted. Many reports and prospects available are optimistic stories about new technologies aimed at potential investors and shareholders, and these do not constitute objective assessment of real extractability. The extractability estimates in the scientific literature vary from 5% (Mudd et al., 2013) to 25% in the most optimistic views (Allsopp et al., 2013, Beckmann 2007). We have taken 5-25% as the minimum to maximum range. For iron (1-2% of the resource is located on the sea floor), manganese (19-59% of the resource is located on the sea floor), copper (2-

8% of the resource is located on the sea floor), molybdenum (6-21% of the resource is located on the sea floor), PGM (Platinum Group Metals) (3-9% of the resource is located on the sea floor), gold (10-34% of the resource is located on the sea floor), silver (13-42% of the resource is located on the sea floor), the contribution from seafloor extraction will never be significant for any longer time. The resources on land for these metals are far larger than what is extractable from the sea floors. There are similar proportions for zinc and lead, however, these have so low market price at present that they do not support any deep sea floor mining operations with the present cost situation. Some of the metals with significant fractions located in the ocean floor include; Manganese (19-59% of the extractable resources in the ocean floor), cobalt (38-76% of the extractable resources in the ocean floor), nickel (18-51% of the resource is located on the seafloor), gold (10-34% of the resource is located on the seafloor) and silver (13-42% of the resource is located on the seafloor). This has been listed in Table 3.4.

Considering the importance of cobalt, silver and nickel for production of new advanced technologies, the ocean floor resources provide interesting and significant back-up resources for future global supply. It has been speculated about mining Rare Earth metals and phosphorus from the ocean floors, but at the moment, these are traded at prices where an extraction operation would be very far from profitable. Extraction from ocean floors are subject to the same consideration as is valid for the extraction of low quality reserves and exotic sources of oil, a concept called EROI (Energy Return On Investment)(Hall et al., 2001, 2009, Hall 2008). For rare metals and materials, we would have the same limiting indicator; Utility Return On Effort Investment. When the benefit from extracting the resource is less than the cost, energy use, effort and environmental damage cost, then it will not be extracted. If ocean floor mining is profitable or not in the future, depends completely on the future raw material prices and how the demand will develop in the time to come. To be able to pay for the extraction costs, as well as to do this in a manner that is not destructive to the oceans, several things are still needed.

Firstly, the technology available at present, is technically capable of extracting the ore, but at a high price. But the technology has seen great advances in the recent years. Secondly, many reports discuss the potential environmental impacts of ocean floor mining (Allsopp et al., 2013, Beaudoin et al., 2014, Beckmann 2007, Clark et al., 2010, Clark and Smith 2013, Zhou 2007, Schmidt 2015, International Seabed Authority 2012, World Ocean Review 2013a,b,c). Unless special precautions are taken, huge impacts may be expected on marine ecosystems and water quality. Much work remains to ensure that the mining can be done environmentally sound and without causing irreversible damage to marine ecosystems and without causing deleterious damage to physical structures. This homework has definitely not been done yet, and the industry should use the time to solve this issue, rather than protesting against not being allowed to cause damage to nature.

3.7 Conclusions

The role of huge new resources for metals on the ocean floor to replace land supply seems limited for most metals. For some of the main metals in society (iron, copper, platinum group metals, gold, silver), the actual extractable amounts seafloor resources are significantly smaller than those on land, and mostly more expensive to extract.

For some of the key technology metals in short supply and with small extractable resources, where every ton of supply counts, when the price is consistently high, then the ocean resources may be an interesting and significant source of metals. The metals cobalt, nickel and silver may be in this category, with additional side products of gold, copper or perhaps molybdenum and platinum group metals. There is a large amount of manganese nodules available, however, the manganese price is at present very low, and would in no way pay for the effort to harvest them and then extract the manganese.

The extraction of these resources may be commercially viable for the metals identified above, and they may under certain conditions contribute significant amounts to the total supply. Considering the difficulty of mining, it seems that based on an ore grade to metal price evaluation, that only cobalt or silver would perhaps be interesting on a commercial level in the near future.

4 WORLD MODEL – RESOURCES MODULE

This Chapter provides indepth information on the WORLD Model's resources module. The resources module consists of:

1. Metals sub-module;
2. Meterials sub-module; and
3. Energy resources sub-module.

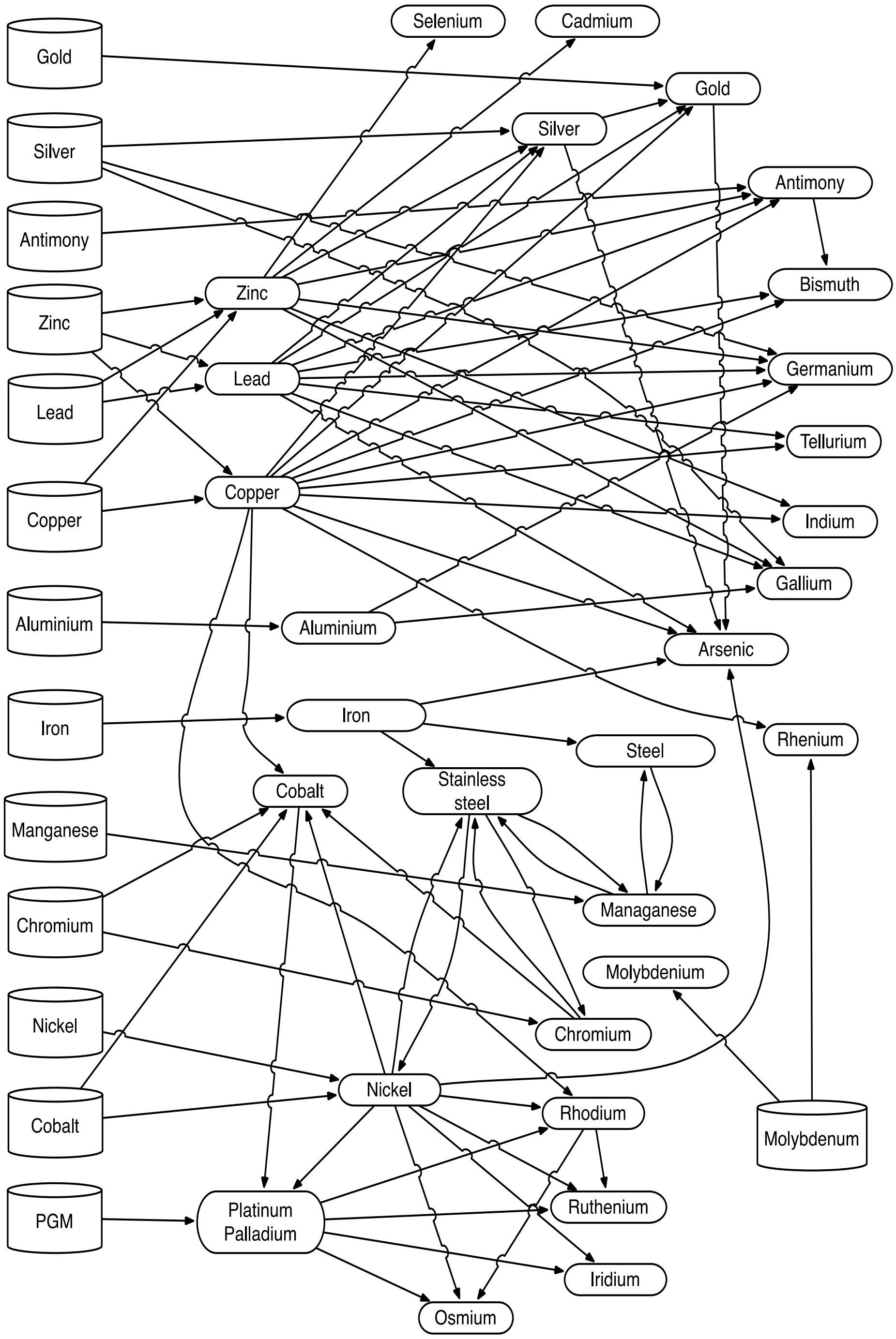
Following sections in this Chapter prodive detailed information on how each of these submodules are formulated in the current version of the WORLD model, and present and assess model simulation results of selected key resources modelled under these sub-modules. The reader should keep in mind that the WORLD Model development is an ongoing process and there may be additional resources implemented to the WORLD model in addition to the ones presented in this report.

4.1 METALS SUB-MODULE

There are a number of metals included in the WORLD Model's metals sub-module as can be seen in Figure 4.1. Among them, we distinguish between metals that are fully independent in their extraction, mother metals, and a number of metals, which are partially or fully dependent on the extraction of some other metal. We clustered and modelled these metals under various sub-modules:

1. BRONZE sub-module
    - COPPER, ZINC, LEAD
    - TECHNOLOGY METALS – (SILVER (Ag), ANTIMONY (Sb), BISMUTH (Bi), TELLIRIUM (Te), SELENIUM (Se), CADMIUM (Cd), GERMANIUM (Ge), INDIUM (In) and GALLIUM (Ga))
2. STEEL sub-module
    - MANGANESE (Mn), CHROMIUM (Cr), NICKEL (Ni), IRON (Fe) and STAINLESS STEEL
3. SPECIALITY METALS sub-module
    - MOLYBDENUM (Mo) and RHENIUM (Re)
    - NIOBIUM (Nb) and TANTALUM (Ta)
    - COBALT (Co)
    - WOLFRAM (W)
    - PLATIMUM GROUP METALS – (PLATINUM (Pt), PALLADIUM (Pd), RHODIUM (Rh))
    - SUPER ALLOYS
4. LIGTH METALS sub-module
    - ALUMINIUM (Al)
    - LITHIUM (Li)

Figure 4.1. Flowchart for the primary production of some metals and the metals whose extraction depend on other metals



Source: Own Figure, Lund University/University of Iceland

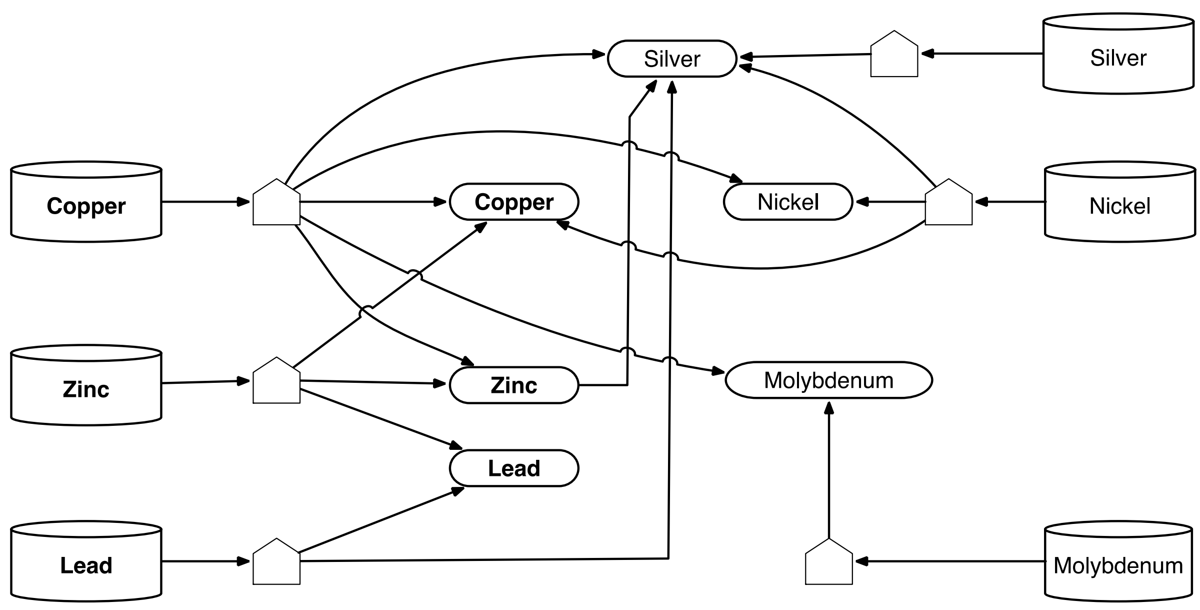
4.1.1 BRONZE sub-module – COPPER, ZINC, LEAD

4.1.1.1 Introduction

The four most important strategic metals for human society are iron, aluminium, copper and zinc. The extraction of copper, zinc and lead is normally done from poly-metallic ores. All metals exist in limited reserves on Earth, the genesis of metal ore is very slow and nowhere near the extraction rates due to present mining rates. There are no really good substitutes for copper, zinc nor lead available, considering their specific technical and chemical properties according to the literature on the subject (Heinberg 2001, Johnson et al., 2007, Graedel et al, 2011, Rauch and Graedel 2007, Norgate and Rankin 2000, 2002, Graedel and Edelmann 2012, Grandell and Thorenz 2014, Goe and Gaustad 2014, Sverdrup et al., 2014). The differences in metal production amounts, and the fact that significant amounts already are booked for important purposes in society, makes substitution a significantly limited option for coping with larger scarcity issues. Likewise, the specialty metals like indium, germanium, gallium, selenium and tellurium are used in applications where there are no obvious substitutes according to the cited literature. Failure to secure them may cause failure of provision of certain technologies. We are not the first to be concerned about the global sufficiency of these metals; this was pointed out already by Meadows et al. (1972, 1974, 1992, 2004), and more recently by Gordon et al. (2006), Rauch and Graedel (2007), Heinberg (2001, 2008, 2011), Laherrere (2010), Morrigan 2010, Northey et al. (2014), Ragnarsdottir et al., (2012) and Sverdrup et al. (2013, 2014a,b, 2015a,b). These studies presented different types of metal supply assessments and expressed worries about a potential scarcity or future peak in production. Further recent assessments are found in reports by the International Resource Panel (UNEP 2011a,b, 2013a,b,c). They discuss recycling of metals in many aspects and are a valuable synthesis of state-of-the-art.

The technology metals indium, bismuth, germanium, gallium, tellurium, cadmium, selenium, antimony and silver are all supplied fully or partly as dependent by-products of copper, zinc and lead primary extraction from poly-metallic ores. Figure 4.1.1.1 shows the pathways of extraction of copper, zinc and lead.

Figure 4.1.1.1. Pathways of extraction of the base metals (i.e. copper, zinc and lead) subject to primary extraction are shown to the left; the technology metals, mostly obtained by secondary extraction, are shown to the right.



Source: Own Figure, Lund University/University of Iceland

4.1.1.2 Objectives and scope

The objective is to develop an integrated dynamic model for the global market for copper, zinc, lead. The model must be able to simulate primary metal production rates from mines, recycling flows, market supply, world market price and to predict how the known and hidden extractable amounts and ore grades will develop with time. The model will be used to assess the long term supply sustainability of copper, zinc and lead and the metals that depend on them for their extraction, as a sub-module in the WORLD6 model (Sverdrup et al., 2012a,b, 2013, 2014a,b,c, 2015a,b,c, 2016a,b, Sverdrup and Ragnarsdottir 2017, Sverdrup 2017). We include the whole global system in a generalized way; simulations cover the recent past and the future (1900-2400).

4.1.1.3 Earlier modelling work

There have been several earlier attempts at modelling copper extraction rates. Roper (2009) used empirical functions for assessing copper, zinc and lead mining rates and estimated when the production would peak. Roper (2009) used systematically low reserve estimates, ignoring hidden resources and the peaks were all predicted to come far too early. Van Vuuren et al., (2009) outlined the principles of a model for the global metal cycling, looking at copper. Glöser et al. (2013) applied a mass and flows model to the global copper cycle. Northey et al. (2014) and Mohr et al. (2014) used a Mass Flow Analysis model for copper, reaching results similar to some of the results obtained in this study (they used a low estimate for the extractable amount). Common for Mass Flow Analysis models are that they do not incorporate feedbacks, nor do they have any market dynamics and cannot generate metal prices internally to the models. They are step-by-step mass balances advanced one year at a time a spread-sheet. Laherrere (2010) used Hubbert's model for copper, doing a valid analysis, but using a too small reserve estimate for the assessment. There is no earlier process-oriented systems dynamics model for copper, zinc or lead known to us, and nothing for the dependent technology metals antimony, bismuth, selenium, tellurium, indium, cadmium, germanium and gallium. We have earlier used simple back-of-the-envelope methods like burn-off rates and peak discovery shift, but these methods only give a diagnostic indication of potential problems. During the course of SimRess project, we have also developed stand-alone systems dynamics models for gold, silver, copper, aluminium, iron, lithium and platinum group metals (Sverdrup et al., 2012, 2014a,b, 2015a,b,c, 2016a,b), rare earths (Kifle et al. 2012), stainless steel and associated metals manganese, chromium and nickel (STEEL) (See following sections). In these models, one of the major advances were the development of a reality-based metal price model, allowing for price estimation from market fundamentals inside the models, without external forcing functions or calibration. This study represents a substantial improvement upon the earlier stand-alone model for copper extraction and supply dynamics (Sverdrup et al., 2014b).

4.1.1.4 Data sources and estimations

Geological resources and reserves

The reserves and resources estimates are based on geological estimates, the interpretation of geological data, and the allocation of extractable amounts according to ore quality, stratified with extraction costs (Tilton 2007, 2009, 2012, Neumeyer 2000, Sweeneye 1992, USGS 2014). All major inputs were extracted from earlier published information (Moskalyk 2003, 2004, Alfantazi and Moskalyk 2003, Graedel et al. 2004, 2005, Spatari et al., 2005, Gordon et al. 2004, 2006, Graedel et al., 2005, Guirco 2005, Brown 2006, Rauch and Graedel 2007, Mudd 2007, 2010, Gerts 2009, Radetzki 2008, 2012, Roper 2009, Geoscience Australia 2009, Rauch and Pacyna 2009, Rauch and Pacyna 2009, Laherrere 2010, MinEX 2010, Crowson 2011a,b, NGEX 2011, Singer 2011, 2013, ICCM 2012, Glöser et al. 2013a,b, Anglo-American Mining Corporation 2012, Prior et al. 2012, Kesler and Wilkinson 2013, Johnson et al., USGS 2013, Dehnavi 2013, Bardi 2013, Copper Development Association 2013, Laherrere 2010, Harmsen et al., 2013, Kesler and Wilkinson 2013, Kavlak and Graedel 2013a,b, International Zinc Association 2013,204a,b, Henckens et al., 2014, Northey et al. 2014, Sverdrup et al., 2014b, Yoshimura et al., 2013). Most actors trade metals through the market, trade takes place geographically dispersed, but is linked through the price systems at the London and New York Metal Exchanges. The available data was closely inspected for inconsistencies and averages and adjustments were made when the input data were not internally consistent. Several of the numbers given in the literature are uncertain as some of them severely mismatch in the overall mass balance. Generally, the recycling numbers often appear inconsistent (UNEP 2011a,b, 2013a,b). The refining process from extracted ore to chemically separable metal concentrate has about 85-90% efficiency for copper and zinc. Further refining runs with 95-97% efficiency, thus the overall efficiency from mine output to metal to market is about 80-87%.

In 2010-2011, the copper stock in the global society was estimated to be about 46 kg per person, the demand on provision from the market was 2.4 kg primary extracted copper per person per year, with 7.3 billion people in the world, corresponding to a demand of about 17.5 million ton of primary produced copper per year. Average global GDP per person was about 7,000 $/capita, the copper primary production was about 17 million ton, and the price was about 9 $/kg in the market. However, the market was supplied with about 30 million to copper metal in total, the excess over primary supplied coming from recycled copper (13 million ton per year). This corresponds to a demand for copper of about 4.1 kg copper per person and year. Brewster (2009) suggested that the total demand of copper could rise to about 10 kg per person per year by 2100 in a world predicted to have about 10 billion people (Gordon et al., 2006, Rauch and Graedel 2007, Rauch and Pacyna 2009, Rauch 2009, Gerts 2009, Gloser et al. 2013), which would correspond to a demand of 100 million ton of copper per year. Even if the global population peaks out at 8 billion people, and the global demand per capita peaks at 6 kg copper per person and year, this would imply supplying about 50 million ton of copper per year to the market, or 3 times more than today. That would be a major challenge, emptying out known and anticipated resources in less than 75 years.

Table 4.1.1.1 shows the energy required to produce copper from different substrates, a comparison between different metals and pathways from raw material to finished metal. This is dependent on a several factors;

1. Oil and coal market price - because metal production process is energy demanding for mining in mountains, moving stone, crushing, smelting and refining and transporting and again fabricating copper, zinc or lead into products.
2. The ore metal content – an indicator of how much rock must be moved in addition to the metal extracted. (Singer 2007, Gordon et al. 2009, Mudd 2007, 2009, 2010, Northey et al. 2014, House et al. 2011, Glöser 2013), and indicating the work we must put in to get the metal out. When the ore quality goes down, the extraction costs rise proportionally with it (Table 4.1.1.2).
3. The flexibility of the supply infrastructure, and delays in increasing or decreasing production. In the past, a mine had a delay in 3-6 years to increase production, at present this has increased to 7-15 years. The increased technical challenges associated with lower ore grades, more complicated operations permission systems and deeper or more remote mining locations are among the causes for this increased delay.

Table 4.1.1.1. Energy required to produce copper metal from different substrates, a comparison between different metals and pathways from raw material to finished metal.

Source of product	Energy use		Climate emission	Water use,
	MJ/kg	Range	CO2/kg	m3/ton product
Recycled scrap	14	6-15	2.2-10	15-20
Sulphide ore, high grade	50	33-64	7	20-50
Sulphide ore, low grade	58	40-60	21	50-60
Sulphide ore, ultralow grade	130	120-160	42	75-100
Sulphide ore, extralow grade	440	400-600	82	100-140
Sulphide ore, trace grade	2,040	1,800-2,600	360	300-500

Table 4.1.1.2. Relationship between ore grade, the approximate production cost and minimum supply price to society. The energy need is approximate. The numbers are approximate for 2012-2015 and are based on data from gold, silver, copper, zinc and uranium mining from many countries.

Ore grade	Ore metal content (Z)		Yield (Y)	Cost (C)	Market price (P)	Energy need (E)
	kg/ton	%	%	$/kg	$/kg	MJ/kg
Rich	400-50	40-5	99-100	4	5-20	30
High	50-10	5-1	98-99	15	20-80	33-64
Low	10-2	1-0.2	95-98	63	80-330	120-160
Ultralow	2-0.4	0.2-0.04	91-95	280	330-1,200	400-600
Extralow	0.4-0.1	0.04-0.01	80-90	1,000	1,200-1,900	700-1,100
Trace	0.1-0.02	0.01-0.002	70-80	1,600	1,900-10,000	13,000
Rare	0.02-0.002	>0.002	50-70	8,000	10,000-50,000	120,000

Basis for estimating how much can be extracted from ore deposits

Fundamental for this study is the estimation of how much metal we can reasonably expect to be able to extract for use. This implies evaluation of how much metal is present in the ore bodies we have and those we expect to find, how much is technically extractable, disregarding price, energy or political obstacles. Finally, how much is extractable in reality, considering extraction costs, willingness to pay, as well as political hindrances over the next three centuries.

The task is to divide this into quantitative estimates of known and hidden extractable amounts, graded into ore quality groups as we have displayed in Table 4.1.1.2. Table 4.1.1.2 shows the relationship between copper ore grade, production cost and minimum supply price to society, as well as impact of price on the recycling in market supply as we have used it in the model. The price range is based the author's own professional experiences from copper, silver and gold extraction (Sverdrup et al., 2013c, 2014a,b), and data from the literature (Mudd 2007, 2009, 2010).

The resource estimates do vary between ore types and metal. Figure 4.1.1.2a shows the size of the extractable copper amount as a function of time, the historical data was taken from House et al. (2011). Copper zinc and lead occur together in many geological deposits, and are mostly extracted from polymetallic ores. The figure shows how the estimates of total extractable copper resources are converging on a value of about 3,000 million ton of copper. All land resources down to 3 km in available deposits and ocean seafloor resources down to an ocean depth of 4,000 meter have been counted in the estimations. Figure 4.1.1.2ab shows data for how the ore grade for copper has gone down systematically over the time since 1880. This is a long term pattern for many metals such as gold, silver, nickel, uranium, zinc, lead, platinum, palladium or uranium). Many examples of such diagrams have been compiled by Mudd (2007, 2009, 2012a,b) for regions, nations and globally defined regions. The figure and its comparative for other metals tell us that the majority of the best deposits have been mined to exhaustion and that mainly the lower quality ore deposits are left. It is a very strong signal suggesting increased risk for scarcity. Without imposing any limitations on extractability, it may appear as if the global copper source would be endlessly large, only by going to lower ore grades (Figure 4.1.1.2c). This is starting from how resources and reserves are defined by the USGS1, but being reformatted to the format needed for the BRONZE model. We define the two types of stocks where metal is held and where it can be extracted from:

1. Hidden resources (These are inferred resources we can assume exists and that will be found and that can be extracted at a later time in the future at some cost as defined in Table 4.1.1.2)
2. Known resources, also sometimes called reserves: These are known resources that have been demonstrated or identified, and are known to be extractable now or in the future.
3. Known waste or scrap is urban ore of metal that is available for recycling once the effort is made

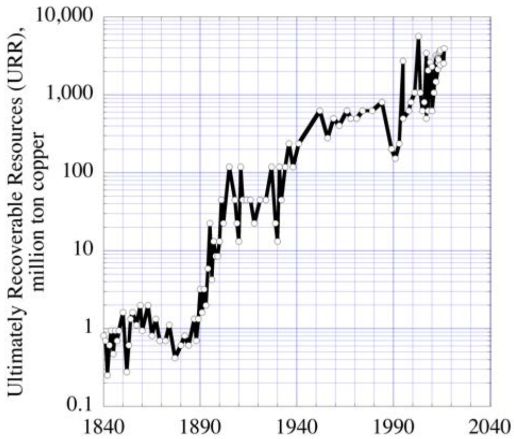
Our judgement is that to hope for doublings or many multiples increases in the available copper resources by mining the ocean floors would be a vain hope, because of the significant technical challenges and the cost of extraction under such conditions.

Figure 4.1.1.2d shows how the extraction cost has gone down due to better and deeper prospecting, new flotation technologies, mechanization of ore benefication, use of dynamite (1900-1920), improved work procedures (1920-1940), increased automatization, more efficient technical mining methods and improved extraction technologies (1950-2010). In the years after 1990, costs were further reduced by moving to low salary countries and where the externalities are not paid for. For many years, this did offset the impact of declining ore grades. From 1930-1980, treatment cost for ore was about 40 $/ton, however, moving to low cost countries, combined with improved methods has lowered that to about 25 $/ton. Lately, it appears as that trend is coming to an end, as there are no new locations with copper deposits where it will be possible to pay workers even less for more work.

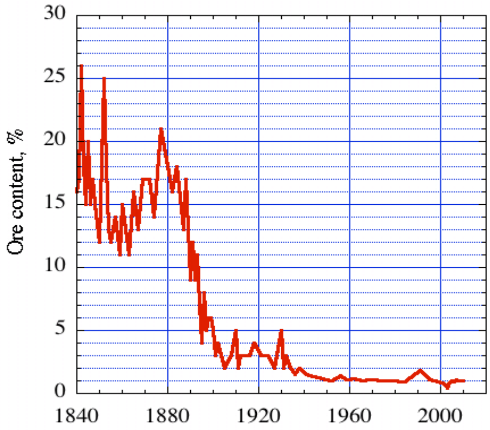
Up to the point where ore quality decline cannot be offset with efficiency and technology gains and further lowering of wages, URR will increase with time (Crowson 2011), but thereafter it converges on a limiting value as can be seen in Figure 4.1.1.2a. The resource estimates seem to level off towards 2.8-3.2 billion ton for copper. Demand should eventually go down as the copper becomes less and less available for trivial and wasteful uses. Uses depending on the metal to be cheap in comparison to the utility value will have problems. The copper ore grade has fallen from a range of 27-15% a hundred years ago to a range of 0.8-3% copper content today (2014), which is partially reflected in the world market copper price through increased production prices.

[1]The USGS (2004) definitions is**: Resource**: A concentration of a naturally occurring mineral in a form and amount such that economic extraction of a commodity is currently or potentially feasible. **Identified Resource**: Resources whose location, grade, quality, and quantity are known or reliably estimated. **Demonstrated Resource**: Resources whose location and characteristics have been measured directly with some certainty (measured) or estimated with less certainty (indicated). **Inferred Resource**: Resources estimated from assumptions and evidence that minerals occur beyond where measured or indicated resources have been located. **Reserve Base**: That part of an identified resource that meets the economic, chemical, and physical criteria for current mining and production practices, including that which is estimated from geological knowledge (inferred reserve base). **Reserves**: That part of the reserve base that could be economically extracted at the time of determination. **Marginal Reserves**: That part of the reserve base that at the time of determination borders on being economically producible. **Undiscovered Resources**: Resources whose existence is only postulated.

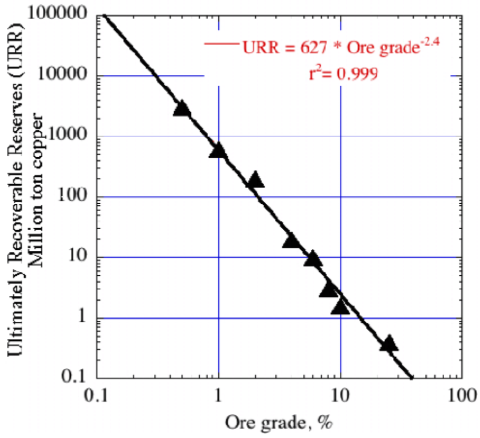
Figure 4.1.1.2. Diagram (a) shows the size of the extractable copper amounts as a function of time (House et al. 2011, Sverdrup et al. 2014b). Diagram (b) shows how the ore grade has gone down with time for copper. Diagram (c) shows how the URR goes up with decreasing ore grade, assuming that every ore grade is fully extractable, with no regard to feasibility. Diagram (d) shows how the extraction cost has decreased with time.



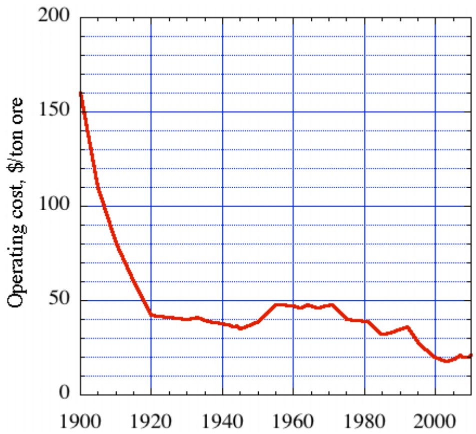
a; URR for copper with time



b; Global copper ore grade with time



c; URR versus copper ore grade cut-off



d: Copper ore extraction cost

Source: Own Figure, Lund University/University of Iceland

Finding the equation for the ultimately recoverable reserves

The Ultimately Recoverable Reserves (URR) is one of the key input parameters in the assessment, and we must put great care into getting good estimates for it. We will work out the example for copper, but in principle, the method is the same for all metals. The (URR) potentially increase when lower grade deposits are included in the estimate. This is shown for copper in Figure 4.1.1.2b, a similar development can be seen for zinc, silver, gold, nickel, uranium and several other metals. The following equation was found to be valid for copper ores (Mudd 2007, 2009, 2012a,b):

$$URR(potential) = kmetal * Zw \qquad (4.1.1.1)$$

Where Z is the ore grade in % weight content, the exponent w=-2.4 and kmetal=627 for copper if URR is in million ton of copper contained. This is only the nominal value of URR and we need to critically assess how much we can extract in reality. The extraction yield (Y) will decrease with declining ore grade as shown in Figure 4.1.1.3b. If we use the data we have collected (Tables 4.1.1.3 and 4.1.1.4), we get the equation (4.1.1.2) for the extraction yield for copper as a function of copper ore metal content Z:

$$Y = \frac{k_Y * Z^n}{1 + k_Y * Z^n} \qquad (4.1.1.2)$$

Where Y is extraction yield (fraction of the weight content recovered from the total ore content), and kY is the technological efficiency constant. It has the value kY=80 when extracted from available data (Figure 4.1.1.3b). The exponent on the ore grade Z seems to be globally valid at n=1.1. The yield may be increased by repetitive extraction, as is the case with the example of rare earth metals up from ore (5-50 separation steps) and by separation of the individual metals from the mix (approximately 1,000 to 4,000 separation steps) or the example extraction of the uranium isotope 235 from mined natural uranium (approximately 5,000 to 10,000 separation steps). The energy, materials and capital cost escalate fast when the number of processing steps multiply, making the whole procedure near impossible for agents that do not have access to huge industrial and energy resources (Figure 4.1.1.3a). From Figure 4.1.1.3a we can determine the equation for energy use in GJ per ton metal produced as related to ore grade in %, yielding the following equation in GJ per ton metal extracted:

$$E = \frac{k_Z}{Z^s} = \frac{3.6 * 10^7}{Z^{1.1}} \qquad (4.1.1.3)$$

Equation (4.1.1.3) was incorporated into the systems dynamics models mentioned here. The correlation for this equation is quite good (Figure 4.1.1.3a) when s=1.1, but this may be depending on interpretation of the data, the exponent may be in the range: 1 < m < 1.2. URR for any metal can be calculated with a combination of equations (4.1.1.1) and (4.1.1.2) to yield for the combination of yield and ore grade alone. The derived formula is:

$$URR(0) = Y * URR(potential) = \frac{k_Y * Z^n}{1 + k_Y * Z^n} * k_M * Z^w \qquad (4.1.1.4)$$

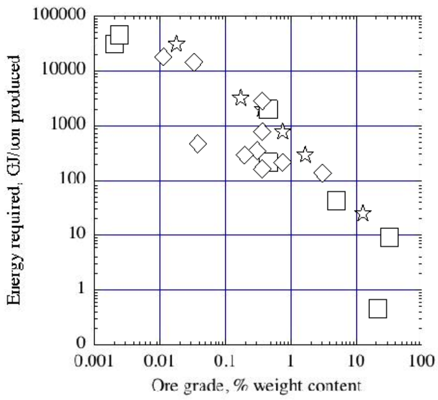
We get:

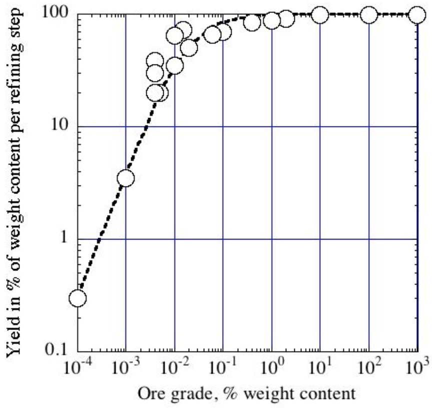
$$URR(0) = \frac{k_Y * k_M Z^{n+w}}{1 + k_Y * Z^n} \qquad (4.1.1.5)$$

We add the energy limitation from equation (4.1.1.3) to equation (4.1.1.4):

$$URR = URR(0) * \frac{1}{E} \qquad (4.1.1.6)$$

Figure 4.1.1.3. Diagram (a) shows how the energy cost rise with declining ore grade, equation (4.1.1.6) as compared to data. The squares, circles and stars represent data from copper, nickel and gold from the literature. Diagram (b) shows how the extraction yield declines with the ore grade with the fit of the yield equation to copper data.





a: Energy cost of extraction b; Yield versus ore grade

Source: Own Figure, Lund University/University of Iceland

If we insert equation 4.1.1.3 into equation 4.1.1.5, we get after rearranging:

$$URR = \frac{k_Y * k_M Z^{n+w+s}}{k_Z * (1 + k_Y * Z^n)} \qquad (4.1.1.7)$$

We set

$$k_{URR} = \frac{k_Y * k_M}{k_Z} \qquad (4.1.1.8)$$

And we get the final equation:

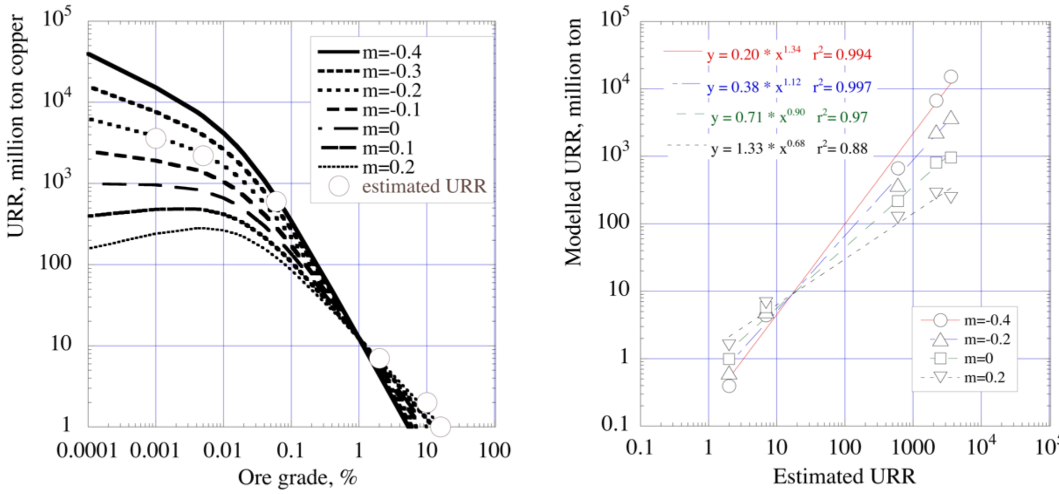
$$URR = \frac{k_{URR} * Z^m}{1 + k_Y * Z^n} \qquad (4.1.1.9)$$

The exponent m is w+n+s, URR is the Ultimately Recoverable Reserve. Equation (4.1.1.9) also defines the exponents m and n, and Equation (4.1.1.3) defines the exponent s. Z is the cut-off ore grade for setting the URR.

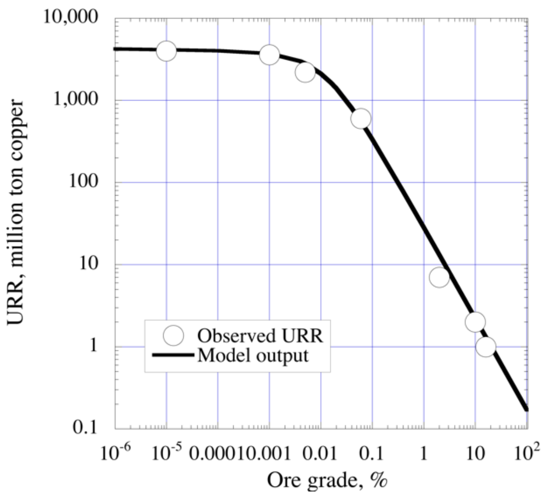
From the data, we have determined for copper that the exponents have the approximate values of w= -2.3, s= +1.1 and n= +1.1, thus m= -2.4 + 1.1 + 1.07= -0.23 for a first approximation. We set n=1.1 and varied m in the range -0.4 to 0.2 in equation (4.1.1.7), this has been plotted up in Figure 4.1.1.4a. When m=0, it would cap URR at a fixed finite value. Equation (4.1.1.6) has been drawn up in Figure 4.1.1.4a The curve should be scalable to URR for any primary extraction of parent metals. The data tested on the equations above suggests that -0.1 < m < 0 and Figure 4.1.1.4b confirms this (r2=0.92 correlation to the 1:1 line for m= -0.01). Figure 4.1.1.4c use the parameters kURR=3,700, kY=130, m= -

0.1 and n= 1.1 for copper. The URR for copper converge on a value of URR= 4,030 million ton of copper. The observed URR data in Figure 4.1.1.2a for copper confirms the shape of the curve generated by Equation (4.1.1.9) and allow us to parameterize it. The expense of making enriched nuclear fuel from uranium or why we cannot extract gold from seawater is explained by these equations. The implication is that below an ore grade of about 0.02-0.01 kg metal per ton rock, the extractable reserves will not increase significantly, but energy costs will escalate. Equation (4.1.1.7) imply that inclusion of large, very low grade deposits will be mitigated by the low yields at low ore grades and the energy costs that increase with declining ore grade (Figure 4.1.1.3a). As we go down in ore grade, it theoretically may look as if the extractable resources increase fast (Figure 4.1.1.2b), but at the same time, the decreasing yield makes this irrelevant as we cannot extract it cost efficiently (Figure 4.1.1.4a).

Figure 4.1.1.4. Diagram (a) shows equation (4.1.1.7) drawn up for different values of the exponent m. The estimate of URR takes into account ore grade, yield of extraction and energy requirement for extraction. Diagram (b) shows a 1:1 plot for different values of m, suggesting that as we can judge from the available data represented by circles in diagram (a), with an apparent best 1:1-fit at m=-0.01. This corresponds to an URR of about 4,030 million ton of copper. (c) shows the plot of the available data together with kURR=3,700, kY=130, m=-0.1 and n=1.1 for copper



a: URR equation b: Testing URR estimates against data

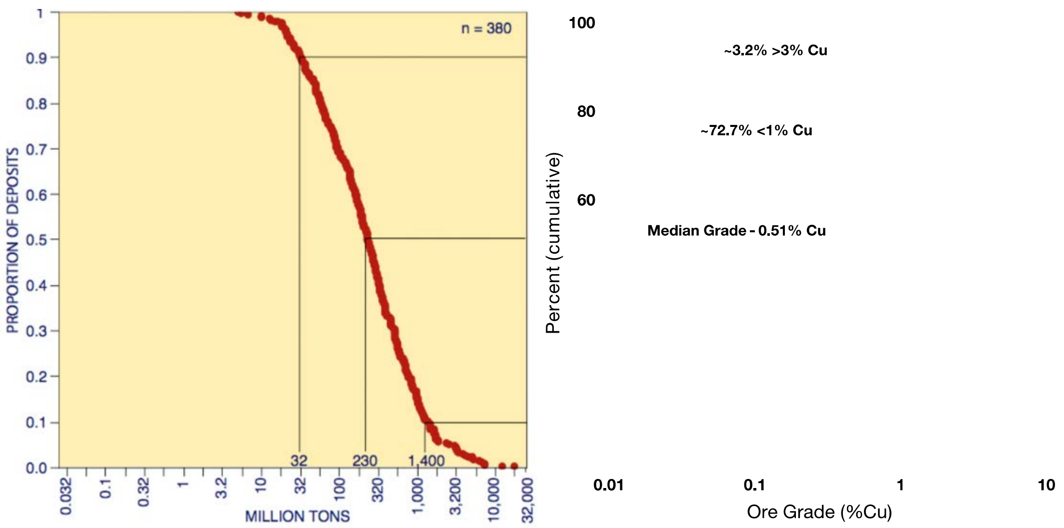


c: Plotting Equation 4.1.1.9 with the best fitting parameters.

Source: Own Figure, Lund University/University of Iceland

Figure 4.1.1.5 shows two diagrams taken from the literature. Figure 4.1.1.5a shows the tonnage distribution of porphyry Cu deposits. Each point represents a deposit and intercepts are at the 90th, 50th, and 10th percentile (from Singer, 2007). Setting the URR for copper to 3.7 billon ton is consistent with this. Figure 4.1.1.5b shows the distribution of ore grades within the copper deposits (Mudd et al., 2013b). These diagrams were used to check the consistency of our estimate of extractable amounts and to distribute it to rich, high, low, ultralow and extra low ore grades.

Figure 4.1.1.5. Diagram (a): Copper tonnage distribution of porphyry deposits. Each point represents a deposit and intercepts are at the 90th, 50th, and 10th percentile (from Singer, 2007). Diagram (b): Distribution of ore grades within the copper deposits (from Singer 2007 and Mudd et al. 2013b). The diagrams were used to assist assigning the URR to different quality classes as shown in Table 4.1.1.5.



a b

Source: Modified after figures by Singer 2007 and Mudd et al. 2013b

Table 4.1.1.3 shows an overview of past mined amounts and extractable amounts for copper, zinc and lead from the literature. Northey et al. (2014) estimated 591 million ton of copper had been mined to 2010, making mined to 2012 about 625 million ton of copper. Mudd et al. (2013b) estimated mined to 2010 to 690 million ton. The extractable amounts of indium, germanium, gallium, tellurium, selenium, bismuth and antimony were given as a % content of these parent metal extractable amounts. Johnson et al., (2014) attempts to estimate the undiscovered copper resources around the world, stating that in addition to identified 2,100 million to, they find about 3,500 million ton of copper resources, making a total of 5,600 million ton in total. They considered porphyry deposits down to 1 km depth and sediments and strata-bound copper down to 2.5 km depth, but they use an average guessed global ore grade. However, the ore grade and extractability remain as unexplained and unknown and the 3,500 million ton cannot be added to the 2,100 million to without a proper extractability assessment. Assuming that 1/3 of the undiscovered copper suggested by the USGS above is in reality extractable, URR would be in the order of already detected resources of 2,100 million ton plus another 1,170 million ton extractable of the undiscovered resources, or about 3,270 million ton of extractable copper. If we assume ½ of this extra amount to be extractable, the estimate would be 3,850 million ton of extractable copper. This would be consistent with our other findings from the literature. The methods we developed for resource size estimating for copper was used in a similar way for zinc and lead. Table 4.1.1.4 shows a summary of the total extractable data we have found, and represents some of the data behind diagram in Figure 4.1.1.2a.

Table 4.1.1.3 shows a review of earlier studies on copper extractable amounts (US Department of Interior 1988, Radetzki 2008, Mudd 2008, 2009, 2010, Commodities Australia 2009, Laherrere 2010, MinEx Consulting 2013, Sverdrup et al. 2014b, USGS 2013, 2014, Kesler and Wilkinson 2013, Copper Development Association Inc. 2013, Johnson et al., 2013). URR is equal to the recoverable resource to date plus the amount extracted until then. Harmsen et al., (2013) and Henckens et al., (2014) assumes an average generalized copper content in the Earth's crust down to 1 km depth, and then assumes 0.1% of that as extractable, based on an assumption made by the International Resource Panel. We are very critical of such estimates as they are based on unsubstantiated assumptions, and because better data with better assumptions are available that partially negates. In summary, using both Figure 4.1.1.4c and the Tables 4.1.1.3-4.1.1.4, we get the estimates for the ultimately recoverable resources (URR) for the three key metals as follows;

Copper; 2,850 million ton < URR= 4,030 million ton < 4,475 million ton

Zinc; 1,362 million ton < URR= 2,676 million ton < 3,000 million ton

Lead; 2,800 million ton < URR= 3,015 million ton < 3,650 million ton

This is also supported by the best-fit-to-data curve in Figure 4.1.1.4a that converges on 4,000 million ton for copper.

Table 4.1.1.3. Estimation of total resources. Overview of past mined amounts of metals and extractable amounts for copper, zinc and lead, expressed in million ton of metal.

Period	Cu	Zn	Pb
Mined before 1800	30	2	50
Mined 1800-1900,	37	50	68
Mined 1900-2010,	617	612	754
Mined to date, integration of mining history curve to 2010	684	664	872
Mined in total 1800-2010 (Mudd et al., 2013a)	690	-	-
Mined in total 1800-2010 (Northey et al., 2014)	591	-	-
Mined 2010-2017	105	85	26
Mined from beginning to 2017	795	749	908
Remaining reserves (USGS 2012)	680	250	736
Sums of mined to date plus remaining reserves	1,475	999	1,744
World remaining resources (USGS 2012)	3,000	2,000	1,900
URR estimate (1); Sums of mined plus USGS 2012 resources	3,759	2,724	2,792
Mined+reserves+resources	4,475	2,999	3,644
Mined and cumulative discoveries for copper, zinc and lead, million ton of metal.			
Period	Cu	Zn	Pb
Mined before 1900	67	52	118
Cumulative discoveries 1900- 2017	2,510	1,302	3,230
Discovered from 2010 to 2017	110	100	25
URR estimate (2); Cumulative discoveries plus mined to 1900	2,687	1,454	3,373

Table 4.1.1.4. Review of earlier studies on copper extractable amounts 1995-2016. Million ton of copper. Note that there is a trend in the data as is visible in Figure 4.1.1.2a.

<table>
<tr><th>Source of estimate</th><th colspan="2">Recoverable resources at the reporting date</th><th>Mined to date</th><th>URR</th></tr>
<tr><td>Sverdrup and Ragnarsdottir (2016)</td><td colspan="2">3,164</td><td>770</td><td>3,934</td></tr>
<tr><td>1. Interpretation of USGS</td><td colspan="2"></td><td></td><td>3,850</td></tr>
<tr><td>2. Interpretation of USGS</td><td colspan="2"></td><td></td><td>3,270</td></tr>
<tr><td>3. Interpretation of USGS (Johnson 2014)</td><td colspan="2"></td><td></td><td>5,100</td></tr>
<tr><td>USGS (2014) global copper resources</td><td colspan="2">3,000</td><td>750</td><td>3,750</td></tr>
<tr><td>Northey et al. (2014)</td><td colspan="2">1,771</td><td>645</td><td>2,416</td></tr>
<tr><td>Johnson et al., USGS, (2014); Known
Johnson et al., USGS, (2014); Hidden</td><td>2,100
1,170-1,750</td><td>3,270-3,850</td><td>643-730</td><td>4,248</td></tr>
<tr><td>Edelstein (2013)</td><td colspan="2">2,100</td><td>730</td><td>2,830</td></tr>
<tr><td>Mudd et al. (2013a)</td><td colspan="2">1,861</td><td>730</td><td>2,591</td></tr>
<tr><td>Copper Development Association (2013)</td><td colspan="2">2,185</td><td>730</td><td>2,915</td></tr>
<tr><td>USGS (2013), Kesler and Wilkinson (2013)</td><td colspan="2">2,246</td><td>730</td><td>2,976</td></tr>
<tr><td>MinEx (2013) Cumulative discoveries</td><td colspan="2">2,391</td><td>730</td><td>3,121</td></tr>
<tr><td>USGS (2012) total global copper resources</td><td colspan="2">2,500</td><td>715</td><td>3,215</td></tr>
<tr><td>Laherrere (2010) peak copper assessment</td><td colspan="2">1,600</td><td>715</td><td>2,315</td></tr>
<tr><td>Commodities Australia (2009)</td><td colspan="2">1,950</td><td>680</td><td>2,630</td></tr>
<tr><td>Radetzki (2008)</td><td colspan="2">1,400</td><td>670</td><td>2,070</td></tr>
<tr><td>Singer (2007)</td><td colspan="2">2,800</td><td>660</td><td>3,460</td></tr>
<tr><td>Singer (1995)</td><td colspan="2">1,512</td><td>560</td><td>2,722</td></tr>
</table>

For a sensitivity analysis, we should run the model for these three variants of total URR; (1); pessimist, (2); average and (3); the optimist scenarios. Table 4.1.1.5 shows the input data to the model used for the simulations. The extractable amounts were set at the beginning of the model simulation in 1800, stratified with respect to ore metal content. The distribution of copper to ore grades is based on the analysis presented in Figure 4.1.1.4 and 4.1.1.5. For zinc and lead, the distribution of URR to ore grades was based on a best fit of the simulations to observations of ore grade development with time, but also assisted by adopting the resource estimation methodology used for copper as outlined above. The estimates are including ocean massive sulphides for copper, zinc and lead. The resource estimates in Figure 4.1.1.2a have a trend in time until about 2000, after which the URR seems to flatten out and show no significant trend with respect to time (Figure 4.1.1.2a, the data was compiled by the authors from literature sources).

Table 4.1.1.5. The resource input data to the BRONZE model.

COPPER							
Copper; porphyric deposits (70% of all)							
Ore grade	------------- Million ton copper -----------					Ore grade %	kg/ton content
	Known	Hidden	Sum	%	Cumula-tive		
Rich	10	4	14	0.5	14	40-5	400-50
High	7	571	578	22	592	5-1	50-10
Low	70	1,313	1,383	53	1,975	1-0.2	10-2
Ultralow	10	403	418	16	2,393	0.2-0.04	2-0.4
Extralow	10	212	222	8.5	2,615	0.04-0.01	0.4-0.1
Sums	107	2,508	2,615	100	-	-	-
Copper; sulphide deposits (30% of all), including ocean massive sulphides							
Ore grade	------------- Million ton copper -----------					Ore grade %	kg/ton content
	Known	Hidden	Sum	%	Cumula-tive		
Rich	5	1	6	0.5	6	40-5	400-50
High	3	111	114	10	120	5-1	50-10
Low	30	270	300	27	420	1-0.2	10-2
Ultralow	5	359	364	33	784	0.2-0.04	2-0.4
Extralow	5	330	335	30	1,119	0.04-0.01	0.4-0.1
Sums	58	1,061	1,119	100	-	-	-
Copper, Extractable ocean massive sulphides and cobalt crusts							
Ore grade	------------- Million ton copper -----------					Ore grade %	kg/ton content
	Known	Hidden	Sum	%	Cumula-tive		
Rich	0	6	6	2	6	40-5	400-50
High	0	45	45	15	51	5-1	50-10
Low	0	95	95	32	146	1-0.2	10-2
Ultralow	0	80	80	27	246	0.2-0.04	2-0.4
Extralow	0	70	70	24	296	0.04-0.01	0.4-0.1
Sums	0	296	296	100	-	-	-
Copper; All deposits on land							
Ore grade	------------- Million ton copper -----------					Ore grade %	kg/ton content
	Known	Hidden	Sum	%	Cumula-tive		
Rich	15	5	20	0.5	20	40-5	400-50
High	10	670	692	19	845	5-1	50-10
Low	100	1,583	1,683	45	2,822	1-0.2	10-2
Ultralow	15	767	782	21	3,419	0.2-0.04	2-0.4
Extralow	15	540	555	15	3,736	0.04-0.01	0.4-0.1
Sums	155	3,579	3,734	100	-	-	-
Ocean floor	0	296	296	-	-	-	-
Total URR	155	3,875	4,030	-	-	-	-
ZINC							
Ore grade	------------- Million ton zinc -------------					Ore grade	kg/ton

	Known	Hidden	Sum	%	Cumula-tive	%	content
Rich	1	28	29	1	29	40-5	400-50
High	5	310	315	12	344	5-1	50-10
Low	1	976	977	37	1,321	1-0.2	10-2
Ultralow	0	1,355	1,355	50	2,676	0.2-0.04	2-0.4
Sum	7	2,669	2,676	100	-	-	-
LEAD							
Ore grade	------------ Million ton lead ------------					Ore grade %	kg/ton content
	Known	Hidden	Sum	%	Cumula-tive		
Rich	20	10	30	1	40	40-5	400-50
High	5	40	45	1.5	75	5-1	50-10
Low	1	1,084	1,085	36	1,160	1-0.2	10-2
Ultralow	210	1,640	1,855	61.5	3,015	0.2-0.04	2-0.4
Sums	20	2,995	3,015	100	-	-	-

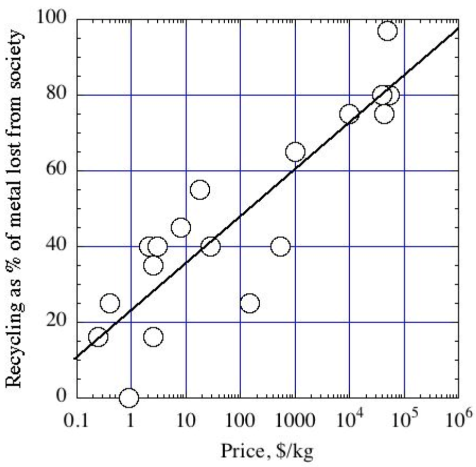
Dependence of recycling on price

Figure 4.1.1.6 shows the dependence of the recycling rate (the fraction of the total supply not coming from mining) as a function of the metal price. This shows what the market mechanisms alone will do. Additional effects can be derived but changing policies and introducing incentives and increasing the costs of not recycling. Or policies and training may over longer times change cultural behaviour, independently of monetary and market mechanisms. We can derive the equation for the market mechanism based recycling as % of total supply to the market. This is used as a basis for the market drive on recycling. The equation determined from Figure 4.1.1.6 is:

$$R = 23.6 + 12.2 * log10(Price) \qquad (4.1.1.10)$$

Where R is the degree of recycling. The correlation is r2=0.61. The figure was made with data from Lenzen (2008) and Prior et al. (2013).

Figure 4.1.1.6. Recycling dependence on price



Source: Own Figure, Lund University/University of Iceland

4.1.1.5 Model description

Structural description

The BRONZE submodel is based on the earlier COPPER, SILVER and GOLD models and experiences learned from them (Sverdrup et al., 2013c, 2014a,b). The BRONZE module in WORLD6 consists of the following parts based on two model published earlier and new systems dynamics parts:

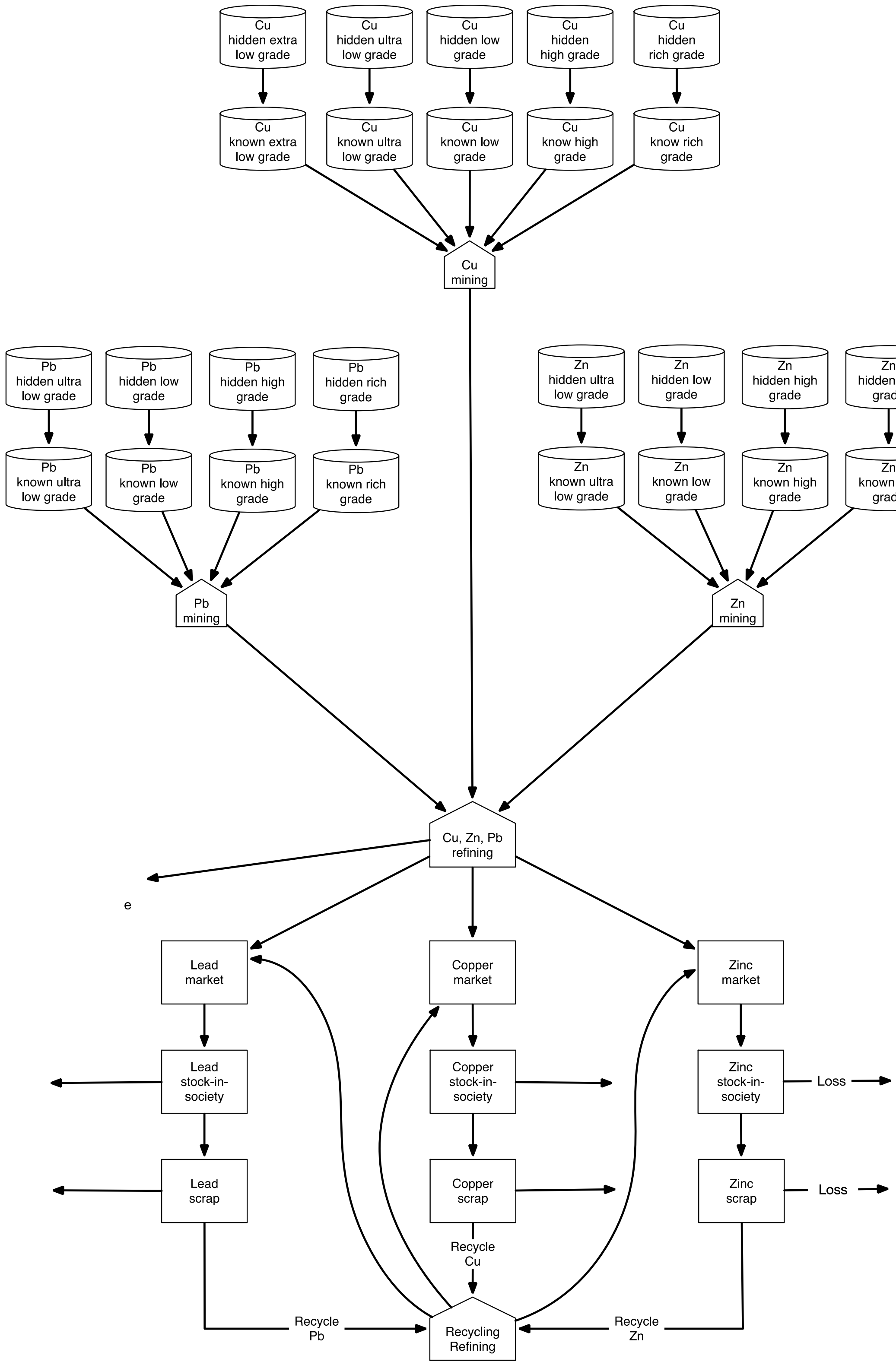
1. The COPPER model (Sverdrup et al., 2014)
2. The ZINC submodule
3. The LEAD submodule
4. The module for technological metals dependent on the extraction from residuals coming from copper, zinc and lead refining and cross-linkages in the system (See Section 4.1.2)

The following modules have a full market dynamics and calculation of the market metal price used for this assessment: Copper, Zinc, Lead, Silver, Gold, Nickel, Molybdenum and Aluminium. The other technology metals have a supply, but no price mechanism as of yet. For these metals, the price has an effect on demand, but not any significant impact on the supply, as this is dependent on the mother metal extraction rate. The technology metals considered are: Antimony, Bismuth, Cadmium, Indium, Gallium, Germanium, Selenium, Tellurium and Silver.

Material flow charts

Figure 4.1.1.7 shows the flow chart for the BRONZE sub-model inside the WORLD 6 model, with the parent ores, the primary extraction and the dependent secondary extraction of many technology metals found in these parent metal ores.

Figure 4.1.1.7. Flowchart for the BRONZE module in the WORLD6 model, including independent and some dependent metals.



Source: Own Figure, Lund University/University of Iceland

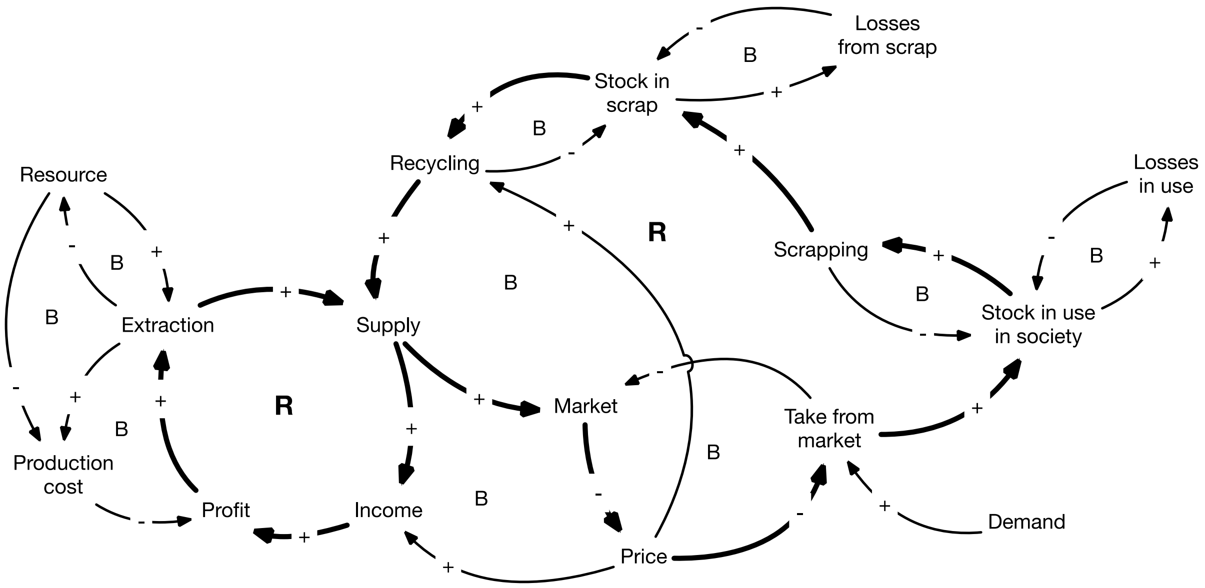
Many metals are extracted from poly-metallic ores, and they are all extracted out, many for new uses in new technologies. The full causal loop diagram for metal mining and copper in particular has been published earlier (Sverdrup et al., 2014a,b, 2015a). The data has been stratified with respect to ore metal content and relative extraction cost. The numbers from Tables 4.1.1.1- 4.1.1.4 and experiences learned from studying corporate reports and scientific literature discussed in the text were reworked into the resource data and the estimates of costs of extraction. This is referred to as the opportunity cost approach (Tilton 2002, 2007, 2009, 2010). We have in the estimation of the extractable amounts of copper (Table 4.1.1.5), considered both porphyric (70% of land deposits, and 65% of all global extractable deposits), sulphide land deposits (30% of land deposits, and 28% of all global extractable deposits) and ocean floor copper resources in cobalt crusts and subsea hydrothermal massive sulphides for copper (7% of all global extractable deposits), using the literature reviewed earlier for this purpose.

Causal Loop Diagram

Figure 4.1.1.8 shows a partial causal loop diagram needed to understand how profits drive mining. B are balancing loops, balancing the reinforcing loops (R) and slowing them down. As the extractable amounts run low, more money must be spent on prospecting. When the hidden extractable amounts run low, then it will get more expensive to find the dwindling extractable amounts remaining out there to be found. It shows how mining rate is driven by demand from society, and promoted by metal price and mining profit to generate supply to the market. The price is determined by how much metal is available in the market. A high metal price will stimulate the mining rate and increase supply to the market, and limit demand. But more supply to the market will increase the amount available and may potentially lower the price. The profit is affected by the mining cost and how that is modified with oil price and ore grade. A lower ore grade implies that more rock must be moved to mine the copper. The implication is that a higher copper price is necessary to keep the copper production up. The price is set relative to how much metal there is available in the market. The traders come to the trading floor with their lots to sell or to buy, and adjusts their amounts as the price increase or decrease. When there is a match, the price is set. If demand is higher than production, the price increase; in the opposite case the price is moved down. This is a self-adjusting mechanism that balances the trade by adjusting the prices until the demand to buy an amount at a price match the offers to sell an amount at a price. In the market, several types of transactions occur:

1. The metal is sold in the market to a buyer, shipped and physically supplied at once. Supply to market and take from market is the same.
2. Forward sale; The metal is sold at once and payment received, but the metal is physically delivered at a later date. Ownership shifts at once, but the money later. Many mines do this to improve liquid funds.
3. The metal is shipped at once but payment is received at a later date. Ownership shifts at once, but the money moves later.

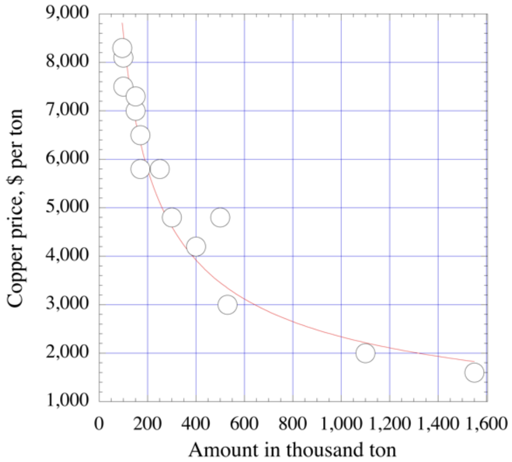
Figure 4.1.1.8. The causal loop diagram showing the market mechanisms and their effect on extraction, supply, demand and actual delivery from the market. This part of the model generates market price inside the model and eliminate the need for externally fed price time series. The price is used to modify the demand to become "Take from market".



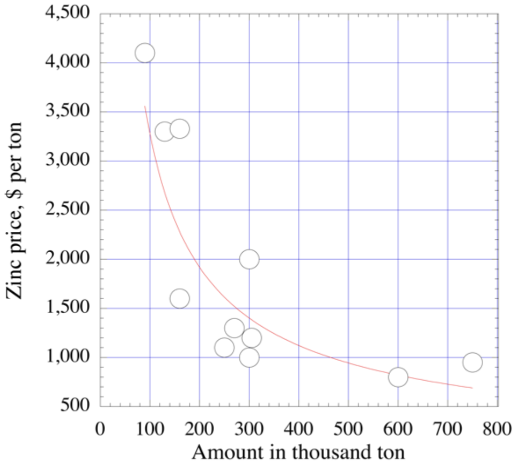
Source: Own Figure, Lund University/University of Iceland

The causality in profits go to income from supply when the amount is supplied and paid at once into the metal exchange warehouse. The same applies with a forward sale, when the metal is paid upfront but physically delivered later. If not, all or part may be paid when the supplied amount has been cleared out from the physical warehouse. This description is based on personal observation on the trading floors at the metal markets in New York and London by the author. For many metals the amounts in the markets are recorded. On investigation, the price is strongly correlated to the market amounts. This has been shown in Figures 4.1.1.9a-d. Figure 4.1.1.9a show the market mechanism price curve for copper, Figure 4.1.1.9b shows the curve for zinc and Figure 4.1.1.9c shows the curve for lead.

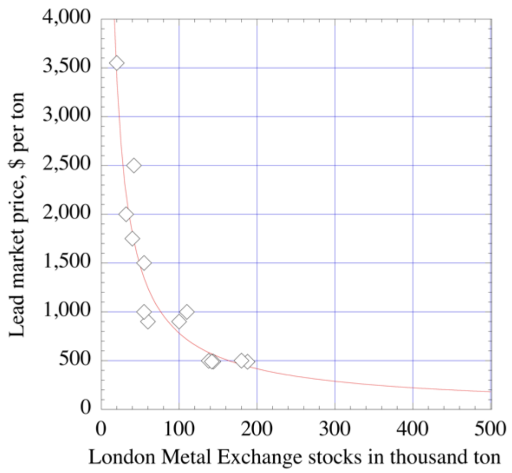
Figure 4.1.1.9. Diagrams (a) show the market mechanism price curve for copper, (b) zinc and (c) lead. The diagrams were generated using market data collected by the authors in the trade market. (The curves were based on market data collected by the author from the trading markets, obtained from market records and experiences in trading activities. The authors have similar curves in use for silver, gold and platinum. (d) The relationship between metal market availability in general and market price, putting all the data together.



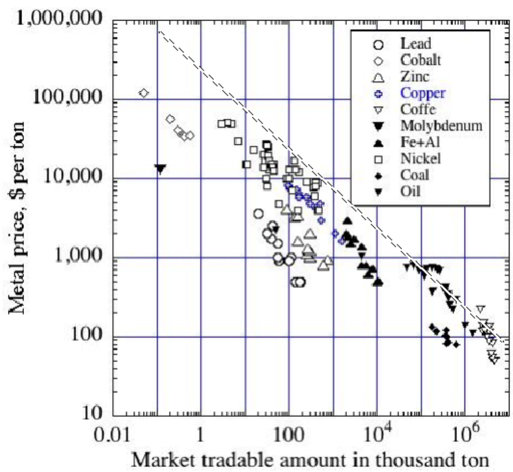
a; Copper



b; Zinc



c; Lead



d; General diagram for many metals

Source: Own Figure, Lund University/University of Iceland

The diagrams were generated using market data collected by the authors in the trade market. Figure 4.1.1.9d shows the curve for the degree of supply as recycled a function of metal price. The price curves found were:

Copper price = 115,000 * (copper market stock)-0.56 r2=0.93

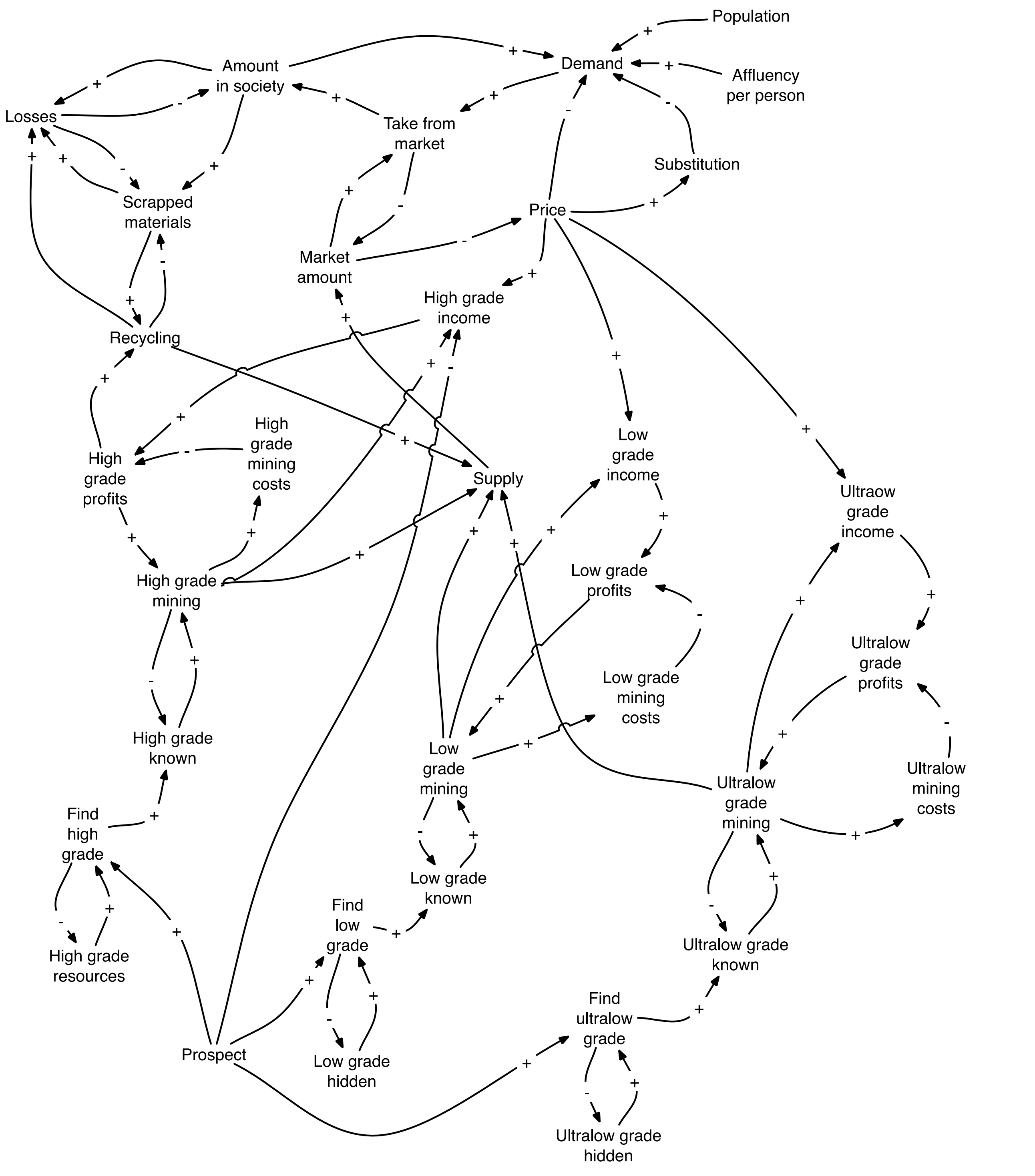
Zinc price = 116,000 * (zinc market stock)-0.77 r2=0.80

Lead price = 51,125 * (lead market stock)-0.91 r2=0.91

When the market stock is in million ton of copper, zinc or lead. With market stock, we imply the amount available in the market for immediate transaction with ownership transfer and if necessary, physical supply. This thus excludes derivatives trade, hedging and forwards, which are not counted as immediately physically deliverable. The scrapping process for stock-in-use in society is not strongly driven by price, but rather from the fact that the infrastructure where it is incorporated has become obsolete. Once the metal is available as scrap not in service, the recycling is driven by profits. After the metal has arrived at the scrap heap, the metal price will have a promotion effect in causing somebody to recover it. The metal price in the market reflects several aspects. One is that the price will not go below the cost of actual production and extraction. And there needs to be profit above the extraction cost, normally about 10-20%. Figure 4.1.1.9d shows the relationship between metal market availability in general and market price, putting all the data together. The solid line is where we think the real price line is. The variation between the different metals reflect how much metal actually passes through the metal exchange, how well the amounts are recorded and some variations in cultural trade behaviour. This is for example known to be different for precious metals. For some of the specialty metals, there is not always a functioning market. When demand is far above supply, it also represents the degree of supply occasionally or short-term not being able to match demand, which is one aspect of scarcity. The potential for exponential growth when the resource exploitation is unrestricted and dynamic market price mechanisms that can limit demand are mechanistically incorporated in our process-oriented models.

Figure 4.1.1.10 shows the causal loop diagram for the mining activity for copper, zinc and lead in the BRONZE submodule of the WORLD6 model.

Figure 4.1.1.10. Causal loop diagram for the mining activity for copper, zinc and lead in the BRONZE sub-module of the WORLD6 model.



Source: Own Figure, Lund University/University of Iceland

The equations and parameter settings

For lead and zinc extraction dynamics, market dynamics and trading details, we did not have the same level of detailed information available as for copper, and a simpler approach was used. Refining is found in the market sector of the model. In the market sector we find refining, trade markets, copper stored in society, scrapped copper, and the actions in/out of refinery, refinery output, market input, and market supply. Recycling is in principle mining from society and from waste dumps and provides a substantial flow of copper in comparison with rock mining. Mining is the action that extracts copper from “known” and puts it to refining. The general mining rate equation used for metals in the WORLD 6.0 model is:

$$r_{Mining} = k_{\ Mining} * m_{Known}^{n} * f(T) * g(P) * h(Y) \qquad (4.1.1.11)$$

where rMining is the rate of mining, kMining is the rate coefficient and mKnown is the mass of the ore body, and n is the mining order, where f(T) is a technology improvement function dependent on time, g(P) is a feed-back from price on the prospecting rate. h(Y) is a rate adjustment factor to account for differences in extraction yield when the ore grade decreases. These functions are given exogeneously to the model. The size of the extractable ore body is determined by the extractions (rMining) and prospecting (rDiscovery). However, it appears from a sensitivity analysis that the results are not very sensitive to the value of n. It turns out that the impact of choosing a different mining model on the final result is small. We get equation (4.1.1.12):

$$\frac{dm_{Known}}{dt} = r_{Discovery} - r_{Mining} \qquad (4.1.1.12)$$

The discovery is a function of how much prospecting we do and how much there is left to find. The amount hidden resource (mHidden) decrease with the rate of discovery (rDiscovery). The rate of discovery is dependent on the amount metal hidden (mh) and the prospecting coefficient kProspecting. The prospecting coefficient depend on the amount of effort spent and the technical method used for prospecting. We get equation (4.1.1.13):

$$-\frac{dm_{Hidden}}{dt} = r_{Discovery} = k_{Prospecting} * m_{Hidden}^{w} * f(Tech) * g(Pop) * j(m_{Known}) \quad (4.1.1.13)$$

The rate w is first order as prospecting is three-dimensional by drilling into the ore body to map it by extent and depth. The modifying functions for technology development f(T) and profit drive on prospecting g(P) are the same as those defined above, but j(mKnown) is a curve expressing the urgency to prospect more, once the known reserves decrease to low levels. Prospecting also depends on how large the known reserves are. The larger they are, the less the urge to drill for more. The basic driving mechanism of basic mining comes from profits and availability of a mineable resource used in the model. The price is set relative to how much iron or steel there is available in the market.

The rate of corrosion of the stock-in-use ad from scrapped metal are defined as Equation (4.1.1.14):

$$r_{Corrosion} = -k_{Corrosion} * m_{Stock-in-use}^{s} \qquad (4.1.1.14)$$

We have assumed corrosion to be a first order process on large scale (s=1, but observed orders are in the range 0.6-0.9, suggesting surface rate control). Scrap is both lost physically by dropping it where it cannot be found and by corrosion:

$$r_{Scrap\ loss} = -(k_{Scrap\ loss} + k_{Corrosion}) * m_{Scrap} \qquad (4.1.1.15)$$

The ore benefication and subsequent smelting yield is defined as:

$$r_{smelting\ supply} = k_{Smelting\ yield} * r_{Mining} \qquad (4.1.1.16)$$

Ore benefication implies that the raw rock extracted from the mine is treated to an ore concentrate. This implies that the rock is milled and enriched using froth flotation, magnetic sorting or other gravimetric separation methods to separate rock minerals from metal-containing parts of the ore. The model as a causal loop diagram for the system is in principle the same as those published earlier for silver, copper and aluminium (Sverdrup et al., 2014a,b, 2015a).

In the model, purchases from the market are driven by demand, put copper into society where it stays until scrapped or removed by wear and losses. A part of the scrapped copper, zinc and lead is recycled and returned to the refinery. Demand in the model is driven by population and copper use per person, but is adjusted up or down with price. Copper stock in use per person as related to global GDP and as it has developed over time. There seems to be a general saturation level at about 10-12 kg per person. Demand is also affected by the market price, which in turn depends on how much is available in the market. The demand is estimated from affluence and global population using outputs of the WORLD 6 model system (Ragnarsdottir et al., 2011, 2012, Sverdrup et al. 2005, 2011, 2012a,c, 2014a,b, 2015a,b, Sverdrup and Ragnarsdottir 2011).

The demand in the model is generated from two sources:

1. Demand from other modules in the WORLD6 model where the metal in question is used.
2. A generic demand (other use) based on use per capita and the total population.

This demand is fed to the model, but adjusted (a) when the price increases:

$$DemandM = Demand0 * g2(Price) \qquad (4.1.1.17)$$

Where D0 is the primary demand from needs before any feedbacks from price or scarcity. The function g2(Price) is the demand modification function using market price as the input. Some metals may reach a saturation level in society. When this happens, demand decreases, and this correspond to soft scarcity as described earlier.

$$Demand_R = Demand_M * 1 - \left(\frac{S}{S_0}\right)^v \qquad (4.1.1.18)$$

DemandR is the real demand after the feedbacks. This is used in the scenarios above to adjust demand. S is the stock-in-use, S0 is the saturation level, and v is the saturation reaction order, we have set this to v=0.5. There seems to be an apparent saturation level at about 12-13 kg copper per capita on a global basis. The corresponding number for zinc appears to be 5 kg capita in developed nations. If this is a real saturation or just a steady state where input of new of material matches the outflow resulting from redundancy, losses and wear, is uncertain. This is linked to the philosophical question if society can tell itself that it has enough, or if the accumulation of stuff just goes on without much reflection on sufficiency (Table 4.1.1.6).

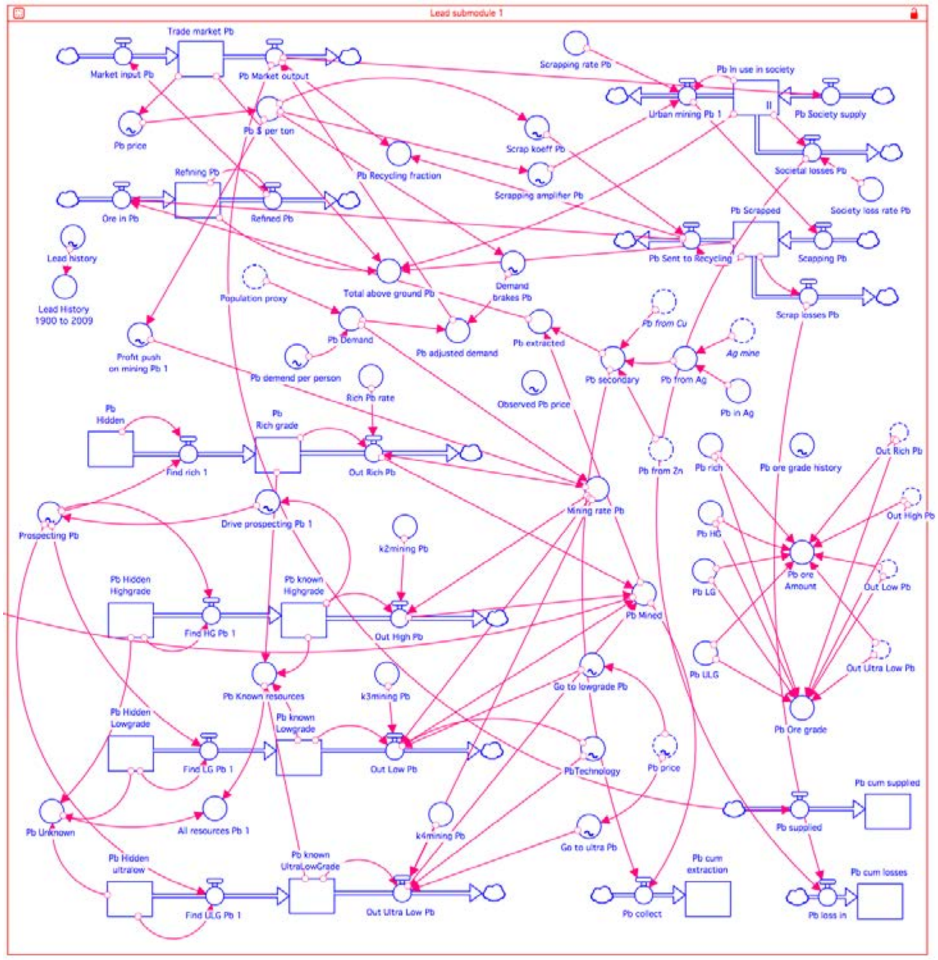
Table 4.1.1.6. Stock-in-use for different metals in 2015. The data was modified after UNEP (2010) and mass balance runs using the WORLD6 model.

Metal	Global average, kg per capita	Developed country kg per capita stock			Developing country, kg per capita	Suggested global saturation level for stock-in-use, kg per capita		
		Average	Future need	Range		High	Mean	Low
Iron	2,200	11,000	12,000	7,000–14,000	2,000	11,000	7,500	4,400
Steel	1,420	7,085	8,000	5,000-9,000	500	7,000	5,000	3,000
Aluminium	80	425		350–500	35	500	330	160
Copper	70	220	300	140–300	30–40	230	190	140
Stainless	15	130	200	80–180	15	150	90	30
Zinc	26	140	200	80–200	20–40	150	100	50
Lead	8	85	0	20–150	1–4	90	65	20
Nickel	3.5	15	140	13-20	1.1	140	15	10
Lithium	0.5	?	50	?	0.05	50	15	3
Cobalt	1	?	50	?	0.10	50	15	3

The STELLA diagram and the computer code

The STELLA diagram for the BRONZE system dynamics model is shown in Figure 4.1.1.11, consistent with the causal loop diagram in Figure 4.1.1.10 and the flow chart in Figure 4.1.1.7. The BRONZE model has several sectors, one for copper, zinc, lead and silver, and one for all the different technology metals. The mining modules where the reserve is divided into known reserves and hidden reserves, and stratified into 5 levels of ore quality; rich high grade, high grade, low grade, ultra low grade and extreme low grade. The input data is shown in Table 4.1.1.5 for copper, zinc and lead. Reserves move from hidden to known because of prospecting. The extractable amounts were set at the beginning of the BRONZE model simulation in 1800, stratified with respect to ore metal content and relative extraction cost based on yield of extraction and energy requirements. The distribution of copper to ore grades is based on Figure 4.1.1.4a. For zinc and lead, the distribution of URR to ore grades was based on a best fit of the simulations to observations of ore grade development with time. The following metals have a full mass balance-based market model, generating demand, supply, price and recycling in the model: Copper, zinc, lead antimony, indium, silver, gallium and aluminium. Selenium, tellurium, cadmium, germanium and bismuth are derived by straight dependency and no market model is used, no market price nor recycling is addressed. The BRONZE model is numerically integrated using a 0.025-year time-step in a 4-step Runge-Kutta numerical method.

Figure 4.1.1.11. An example of a STELLA diagram for the copper module of WORLD6 model. This is only a small section of the model shown in Figure 4.1.1.7.



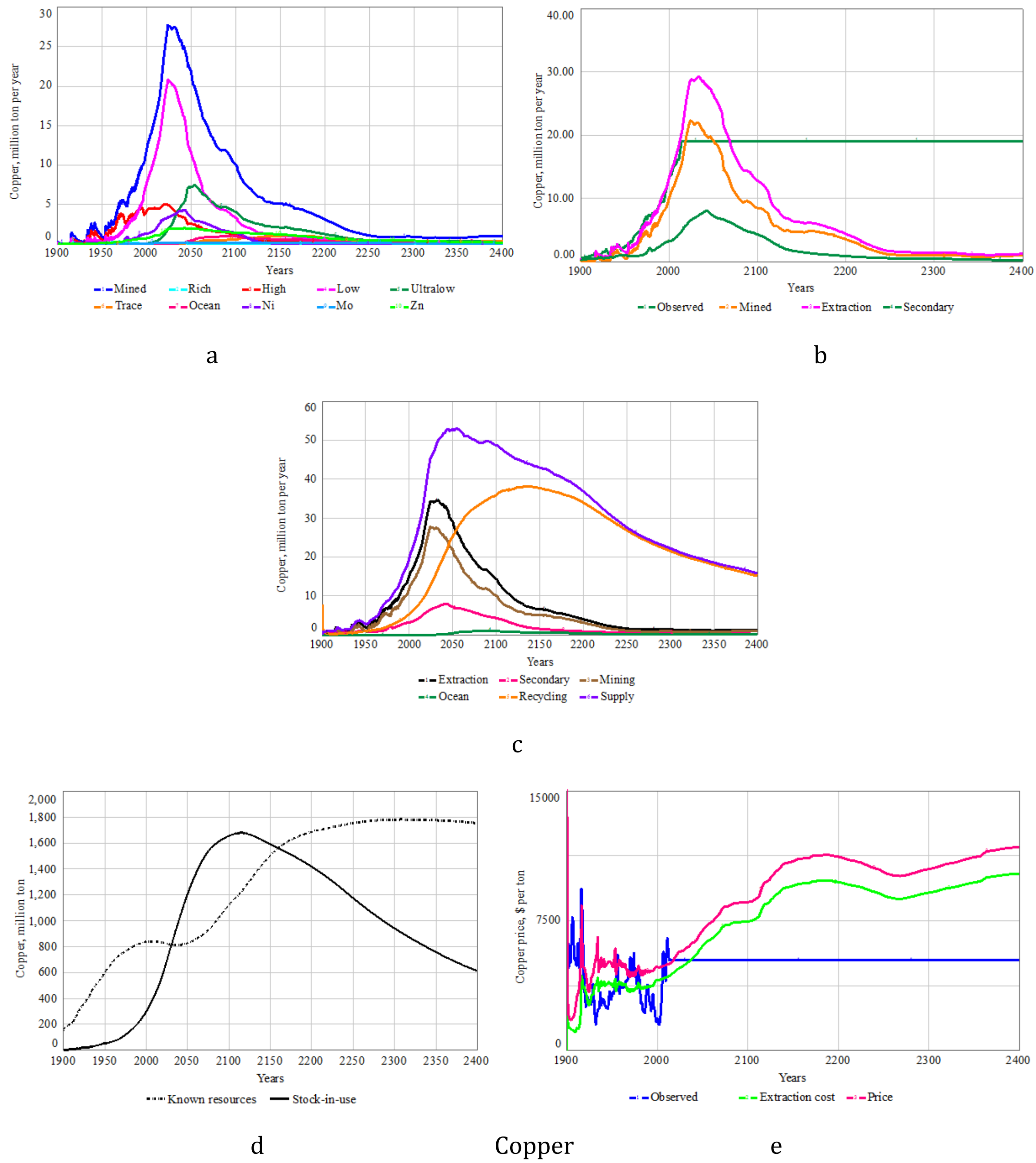
Source: Own Figure, Lund University/University of Iceland

4.1.1.6 Results

Copper mining, extractable amounts and supply to society

Figure 4.1.1.12a shows the simulated copper mining, contributions from different ore grades as well as copper derived from zinc, nickel and molybdenum mining. For copper, the willingness to pay is good because of the importance of copper for many essential functionalities of society. The production is predicted to peak around 30 million ton per year (Globally that corresponds to about 3.2-3.5 kg per person and per year) and to decline after 2045. Figure 4.1.1.12b shows copper supply from mining as compared to the observed mining rate Figure 4.1.1.12c shows the supply to society, the mining rate and recycling. Long after the copper mines have run out, copper will be present in society and be supplied from recycling and urban mining. It can be seen that copper supply to society peaks at about 77 million ton of copper per year in 2120 (7.5 kg copper per person and year). Figure 4.1.1.12d shows the development of known copper extractable amounts over time compared to stock-in-use. It can be seen that whereas copper mining peaks in about 2040, copper supply to society does not peak until 2110 because of copper recycling from the stocks-in-use, about 70 years after the mining rate peaks. The stock-in-use in society reaches a maximum in 2090 at 2,200 million ton of copper according to the calculations. Supply to society is far larger than mining, because of effective recycling of copper. Copper has a high price, and copper recycling is helped by a cultural habit of copper being valuable that promote recycling. After 2055, stock-in-society will be larger than known extractable amounts. Note how recycling will be the major source of copper after 2050. Figure 4.1.1.12e shows the simulated copper price as compared to the observed one.

Figure 4.1.1.12. Copper. Diagram (a) shows the copper mining rate and contribution from different ore qualities, as well as copper derived from zinc, nickel and molybdenum mining. Diagram (b) shows supply from mining and compared to the observed mining rate. Diagram (c) the supply to society, the mining rate and recycling. Diagram (d) comparison of stocks-in-use with the known reserves over time. Diagram (e) shows the simulated price as compared to the observed.

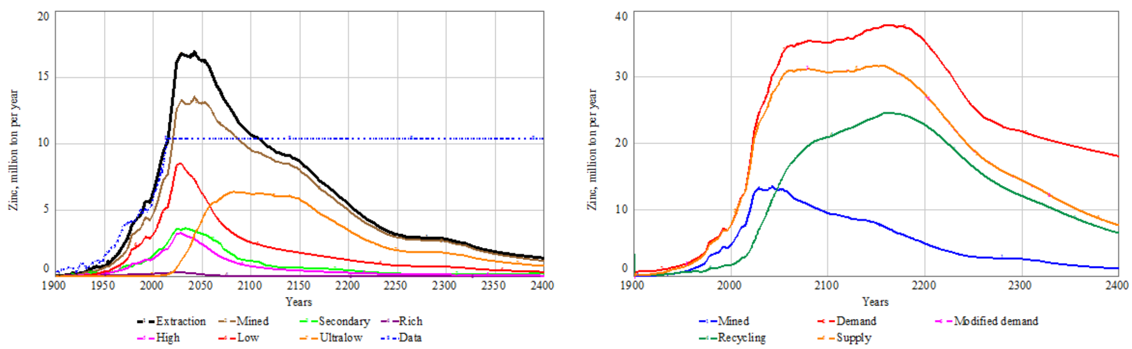


Source: Own Figure, Lund University/University of Iceland

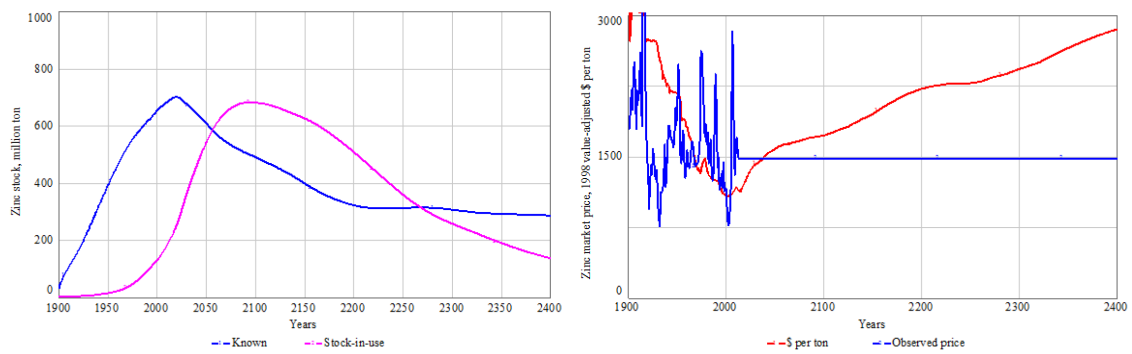
Zinc mining, reserves and supply to society

Figure 4.1.1.13a shows the simulated zinc mining rate along with the contribution from the different zinc ore grades. Figure 4.1.1.13a shows that the zinc supply from mining peaks 2031-2052, the zinc production is estimated to peak at a zinc production of about 14.5 million ton of zinc per year. Figure 4.1.1.13b shows the supply to society, the mining rate and the recycling rate. It can be seen that supply to society peaks in 2092 at about 37 million ton of zinc per year. Supply to society is far larger than mining, because of recycling. According to the simulations around 2048, the best ore grades are consumed first, implying that only poor zinc ore grade extractable amounts are available after 2015. Figure 4.1.1.13c shows the development of different zinc stocks in society and in the known zinc reserves with time. The known extractable amounts peak about 2075. After 2050, stock-in-society will be larger than known extractable reserves. Note how recycling will be the major source of zinc after 2045. The zinc stocks-in-use peak around 2080. Supply to society peaks later than zinc supply from mining because of recycling from the zinc stock-in-use. The stock-in-use peaks at about 615 million ton of zinc. The diagram shows that after 2052, zinc in society will be a larger source of zinc than mining from ore deposits. This will be the era of scrap zinc. The simulations show that using zinc as sacrifice anodes for corrosion protection of other metals is a dissipative loss where the zinc cannot be recycled. That would be a tough challenge, and if that can be reengineered remains to be seen. Figure 4.1.1.13d shows the simulated and observed zinc prices with time.

Figure 4.1.1.13. Zinc. Diagram (a) shows the zinc mining rate and contribution from different ore qualities. Diagram (b) shows mining, supply to society and recycling. Diagram (c) shows known extractable amounts and stocks-in-use in society. Diagram (d) shows the simulated price for zinc as compared to the observed market price in 1988 value adjusted $ per ton.



a Zinc b



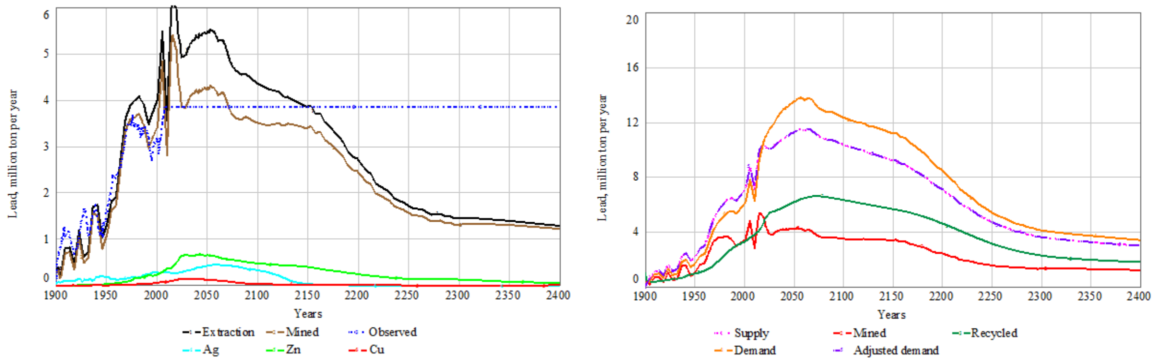
c Zinc d

Source: Own Figure, Lund University/University of Iceland

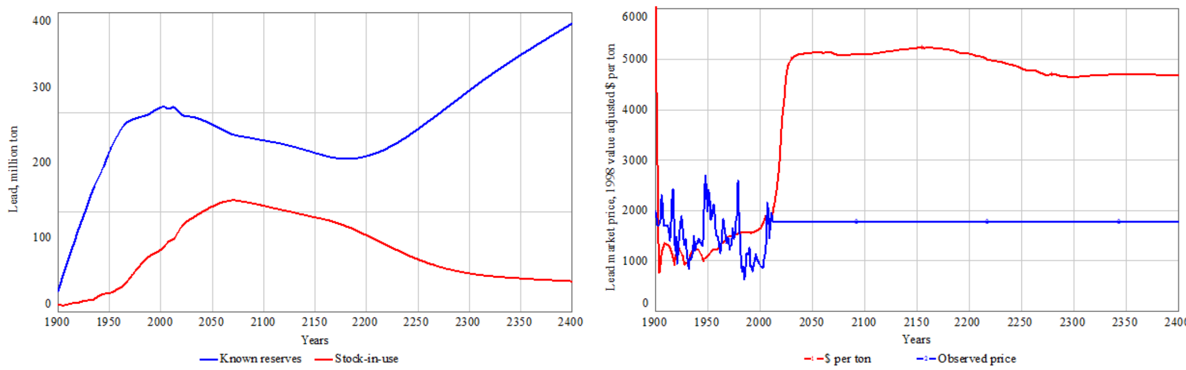
Lead mining, reserves and supply to society

Figure 4.1.1.14a shows the simulated lead supply from mining, the amount of lead coming from copper and zinc mining, and compared to the observed mining rate. Figure 4.1.1.14b shows the supply to society. The supply of lead will reach a peak from mining in about 2030-2045 for copper, supply to society peaks later because of recycling from the stock-in-use, for lead supply to society, the total lead supply peaks at the latest in 2045. Because lead is being phased out from most of its uses, known extractable amounts stay high as extraction goes down due to lower demand. Lead is now banned in paints, colours, additive to vehicle petrol, in ammunitions for hunting and is being phased out from soldering alloys. All this because of its toxicity. The lead reserves and resources may well support a mining rate compared to that of copper and zinc for comparable long as time. Figure 4.1.1.14c shows known exgtractable amount and stocks-in-use in society. Recycling will be the major source of lead after 2050. Figure 4.1.1.14d shows the simulated lead price as compared to the observed prices.

Figure 4.1.1.14. Lead. Diagram (a) Lead mining rates, contribution from zinc and copper mining. Diagram (b) shows the mining rate, supply to the market and the amount from recycling. Diagram (c) shows known extractable amounts and stocks-in-use in society. Diagram (d) shows the simulated price for lead compared to the observed price in 1998 value adjusted $ per ton.



a Lead b



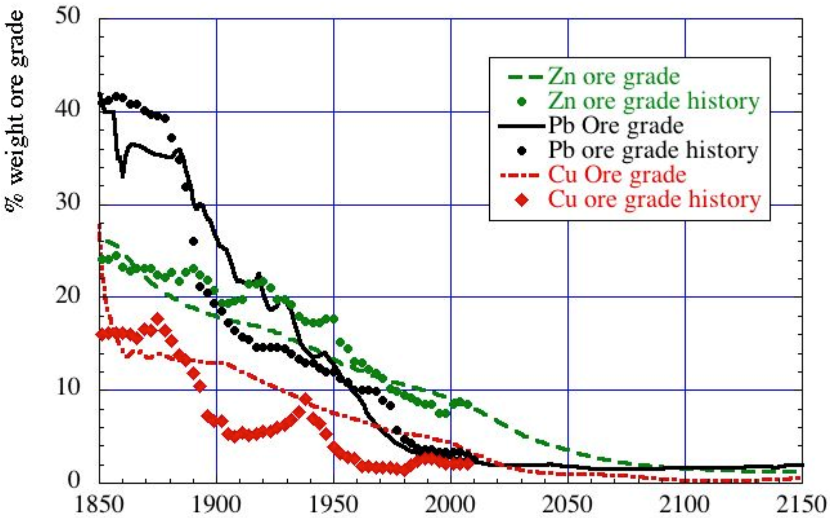
c Lead d

Source: Own Figure, Lund University/University of Iceland

Cu, Zn and Pb ore grades 1800-2200

The model predict the ore grades of copper, zinc and lead to continue to decline, as well as explain the observed ore grade decline of the past. Figure 4.1.1.15 shows ore grades in % for copper, zinc and lead, the field data collected for ore grades in the world. The data are from the publications of Mudd (2007, 2009, 2010) as well as single values picked throughout many publications. The consistently declining ore grades over the last 150 years is closely matched by the model.

Figure 4.1.1.15. The simulated ore grades as compared to the observed data.

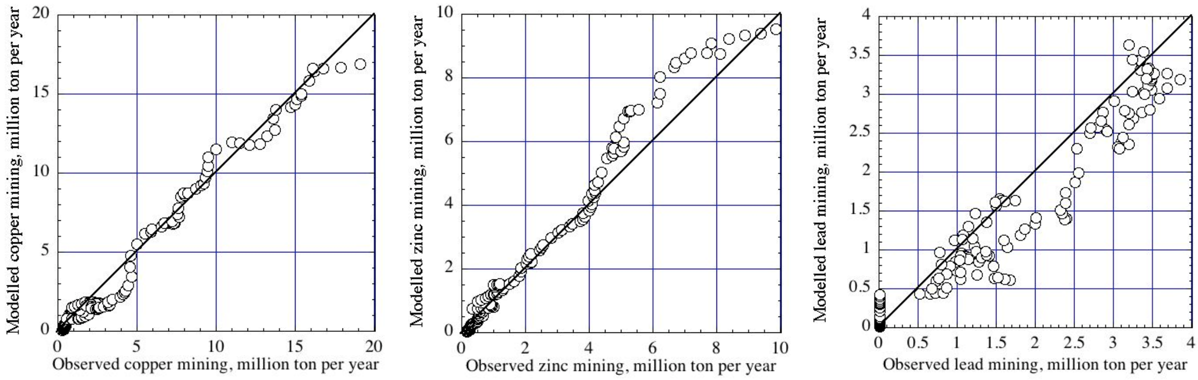


Source: Own Figure, Lund University/University of Iceland

Testing the model

The model has been tested against the recorded mining data derived from the USGS (2013) databases. This can be seen throughout the results section (Figure 4.1.1.12b, 13a and 14a). Figure 4.1.1.16 shows the modelled mining rates as compared to observed data.

Figure 4.1.1.16. Comparison of modelled mining rate and the observed data.



Copper Zinc Lead

Source: Own Figure, Lund University/University of Iceland

Figure 4.1.1.15 shows a testing of the model for ore grade with time for copper, zinc and lead. The ore grade declining trends are very well reconstructed for copper and zinc and acceptable well for lead. The metal market price is well simulated for copper (Figure 4.1.1.12e), zinc (Figure 4.1.1.13d) and lead (Figure 4.1.1.14d). Supply to society is far larger than mining, because of effective recycling of copper. Copper has a high price, and copper recycling profits from a cultural habit promoting recycling. After 2055, stock-in-society will be larger than known extractable amounts. Note how recycling will be the major source of copper after 2050. Table 4.1.1.7 shows an overview of fitness between data and simulations.

Table 4.1.1.7. Overview of the correlations (r2) between observed data and simulations.

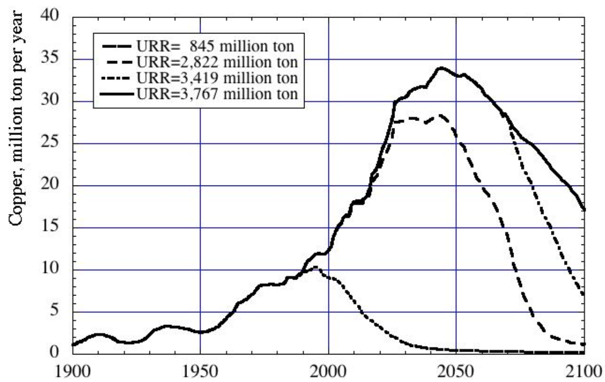
Metal	Mining to data	Ore grade to data	Market amount to price	Market price to data
Copper	0.986	0.86	0.93	0.70
Zinc	0.983	0.81	0.80	0.40
Lead	0.950	0.97	0.91	0.40

4.1.1.7 Discussion

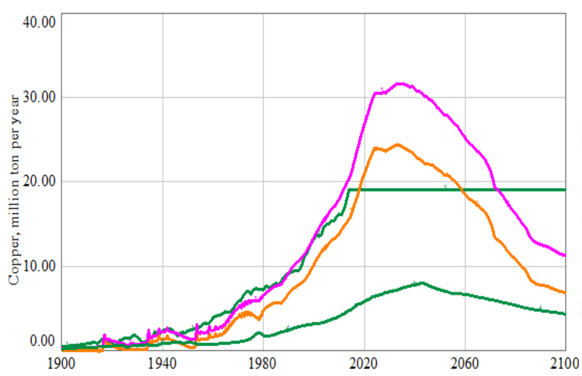
Comparison with Northey et al.'s study

Figure 4.1.1.17 shows a comparison of the model outputs for copper production from mines from the study of Northey et al. (2014) and the copper production from mines of this study (BRONZE) show some significant differences. We get a higher copper production at the peak (31 million ton per year versus 27 million ton per year), the peak comes 2045-2050 in the BRONZE model simulation, whereas Northey et al. (2014) gets the peak 2028-2035 in their model. The main difference between the studies is the URR used as input data to the models and the way the URR is used. Northey et al. (2014) use an URR of 1,800 million ton of copper, but also investigating the sensitivity to setting it at 900 million ton and 2,700 million ton. We use a standard URR of a total of 3,700 million ton for the BRONZE model, graded as shown in Tables 4.1.1.1 and 4.1.1.5, but can also assess the production from a URR of 1,400 million ton and a URR of 2,600 million ton. When the two different models use the same extractable reserve, then the results are very similar. For any policy interpretation, they have the exact same ramifications. The main difference is in how the URR was determined, in our present study, this is a dynamic size, not to be mixed up with the reserves and resources definitions of the USGS mentioned earlier. Depending on the extraction cost and the copper price in the market, increasing price reduce demand, and when mining does not pay off, it stops.

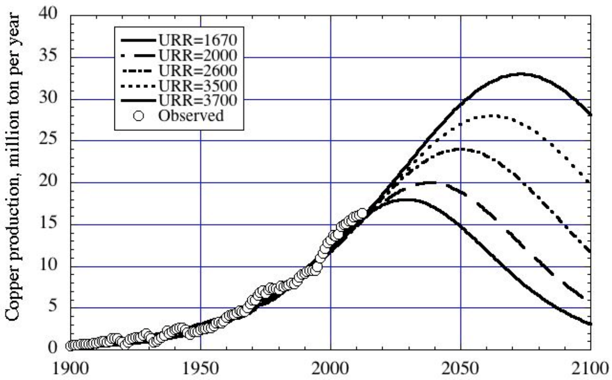
Figure 4.1.1.17. Comparison of (a) the model output for copper production from mines from the study of Northey et al. (2014) and (b) this study. Northey et al. (2014) used three different URR; 900, 1,800, 2,600 million ton. We used 1,400, 2,615 and 3,700 million ton in our BRONZE assessment and 4,030 million ton in our WORLD6 assessment. Diagrams (c) and (d) are the mining rates using the BRONZE model and the Hubbert's model applying different values for URR.



a: Northey et al. (2014) b: BRONZE stand-alone model



c: WORLD6 model



d: Hubbert's model

Source: Own Figure, Lund University/University of Iceland

Reserves and extractable amounts of metal

Table 4.1.1.5 shows a comparison of the traditionally estimated URR pre-determined by using cost-graded ore qualities, correlated with metal content. In Table 4.1.1.3 and Table 4.1.1.4 we have compared different ways to pre-estimate the URR, the ore quality grading method, the standard reserves, resource grading method (USGS), and the reserve assessed using the parent standard URR for copper, zinc, lead. We have compared it to the ultimately extracted amount.

It appears as the method to use an extraction cost-ore quality stratified approach gives a better estimate for extraction calculations and a model that will respond better when put inside an economic model. Such a price model is able to respond to changing economic conditions, which is preferable for the WORLD6 model.

Sustainability of mining

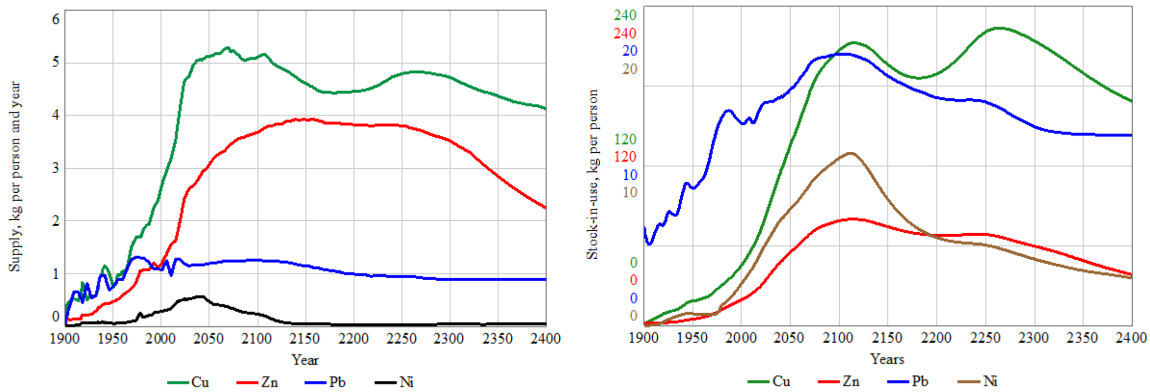
The declining ore grade is a clear indication that the resource has been depleted and that scarcity has already set in. For some time, this was offset by improvements in efficiency and lowering of wages, but that has recently come to an end. In the model, the demand is made on the market, and when the market amount goes low, then the prices go higher. The higher prices push mining, causing the price to increase. The model becomes self-regulating. Thus, the market dynamics are fully expressed in the present model and that gives a significantly smoother production curve and better dynamics. In the discussion of sustainability, we should not that recycling can delay symptoms of scarcity for a significant time even after the production from mines has stopped. The recycled copper, zinc and lead have nothing of the dependent metals preserved and this must be considered in a future strategy for how they are to be sourced.

Resource scarcity

The dependent metals may fast run into physical scarcity because the price has no feedback on the extraction rate of the mother metals copper, zinc or lead. It is important to consider that when the commodity is still relatively cheap, it may for that reason be unnecessarily wasted, when it would in retrospect have been relatively easy to induce better use efficiency and recycling. To wait until there is severe scarcity would be too late for optimal mitigation as unnecessary resources would already have been wasted by then. Others have proposed deep ocean mining (Scott 2001) but efficient and affordable technology for this has failed to surface so far. Different authors point to future risks and possibilities and mainly focus on issues of short term market conditions, better technologies and new markets.

The future will show how these issues will really work out under a different set of conditions with potential scarcity and additional challenges of climate change (Wagner and Fettweis 2001, Ghose 2009, Eliott et al., 2014, Morrigan 2010). The purpose of models is to look into what would be possible in the future. The model does not tell us what the future will be, but can illustrate what futures are possible, and something about how likely they are to happen. This is the only way we can plan ahead of time and avoid getting into unsustainable situations. At present, many studies and model assessments point out that the global resource use is in large overshoot and that we have limited time left for corrective action. And when thermodynamic corrective enforcement has already started, then it may be too late to do the mitigation in an orderly manner. Figure 4.1.1.18 shows the supply in kg per person and year and the stock-in-use, kg per capita, for copper, zinc, lead and nickel as calculated using the WORLD6 model. The stock in use indicates the amount of utility that we have from each metal, whereas the supply per person, suggests how much we have available for maintenance, to replace losses and if possible, for growth. The model suggests that growth for copper use stops about 2030, for zinc about 2140 and for lead it stopped already 1980. The utility of copper peaks in society in 2120, for zinc in 2090, for lead in 2070. The supply level stays above the 2017 level to after 2400 for copper, and declines to the 2017 level in 2350 for zinc. The stock-in-use stays above the 2017 level beyond 2400 for both copper and zinc, but falls below that in 2260 for lead. This suggests that there will not be any acute levels of physical scarcity in the next century. The main reason for this will be the increasing price and the retarding effect this has on demand. A main driver for price increases will be increasing extraction costs. The main reason for the increasing extraction costs will be the decline in ore grade and the increasing energy costs during the period.

Figure 4.1.1.18. Diagram (a) shows the supply in kg per person and year. Diagram (b) shows the stock-in-use, kg per capita, for copper, zinc, lead and nickel as calculated using the WORLD6 model.



Source: Own Figure, Lund University/University of Iceland

4.1.1.8 Conclusions

The model seems to be verified reasonably well against field data. The WORLD 6 model assessment and the Mass Flow Analysis by Northey et al., (2014) reach the same conclusions concerning need for future policies. The consistently declining ore grades, with are perfectly matched by the model, is a strong indication of the coming scarcity and increased price for copper, zinc and lead. Failure to take this message in would be a very serious mistake. For governance, it would constitute a failure of taking responsibility, for corporations, to ignore key business obstacles or opportunities. In a historical perspective, it will appear as totally unforgivable to have ignored these scientific messages. However, using the the BRONZE sub-model embedded inside the WORLD 6 model does give a richer picture, including more details and essential feedbacks. We are able to reconstruct the past history of mining rates, ore grade decline and approximate metal price levels. We get reasonable estimates of the production rates for the technology metals dependent on copper, zinc and lead for their production. We need to carefully distinguish between the primary production from mines and total supply to the market. Long after primary metal mining has been reduced to insignificant levels, supply may be kept up by efficient recycling. The time after 2050 will be the age of urban mining, where more copper, zinc and lead metal is supplied from recycling that from primary extraction from mines. However, for the technology metals, many have at present low rates of recycling and these are much more vulnerable for decline in mining, unless the recycling efficiency can be significantly improved. The conclusion is the there is not an immediate risk for scarcity but that in the longer run (after 2030), scarcity manifested as rising metal prices should be expected and prepared for.

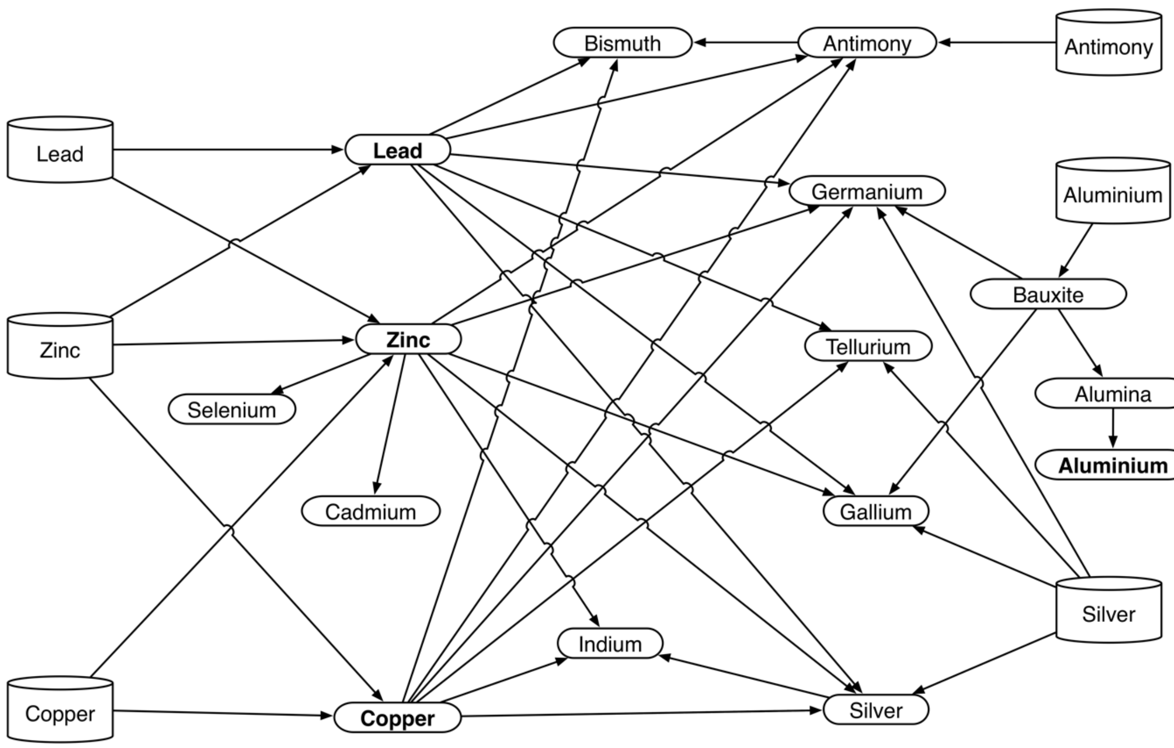
4.1.2 BRONZ sub-module – TECHNOLOGY METALS (Ag, Bi, Cd, Ge, Ga, In, Sb, Se, Te)

4.1.2.1 Introduction

The specialty metals like indium, germanium, gallium, selenium and tellurium are used in applications where there are no obvious substitutes according to the cited literature. Failure to secure them may cause failure of provision of certain technologies. We are not the first to be concerned about the global sufficiency of these metals; this was pointed out already by Meadows et al. (1972, 1974, 1992, 2004), and more recently by Gordon et al. (2006), Rauch and Graedel (2007), Heinberg (2001, 2008, 2011), Laherrere (2010), Morrigan 2010, Northey et al. (2014), Ragnarsdottir et al., (2012) and Sverdrup et al. (2013, 2014a,b, 2015a,b). These studies presented different types of metal supply assessments and expressed worries about a potential scarcity or future peak in production. Further recent assessments are found in reports by the International Resource Panel (UNEP 2011a,b, 2013a,b,c). They discuss re-cycling of metals in many aspects and are a valuable synthesis of state-of-the-art.

The technology metals indium, bismuth, germanium, gallium, tellurium, cadmium, selenium, antimony and silver are all supplied fully or partly as dependent by-products of copper, zinc and lead primary extraction from poly-metallic ores (Figure 4.1.2.1). Thallium and arsenic are not included in this study, even if they also are partly dependent on copper-zinc-lead poly-metallic ores. Cobalt, molybdenum, tantalum, niobium, tin and rhenium are not included here, they are the subject of separate studies and will be reported separately in the later sections of this report. Ocean floor resources for copper, zinc and lead have been omitted, as we will see later in this study, the extractable amounts are insignificant as compared to land-based resources.

Figure 4.1.2.1. The flow chart for the metals focused on here. The metal subject to primary extraction are shown to the left, the technology metals, mostly obtained by secondary extraction, are shown to the right. The complex flow chart makes the modelling of the system a large and complex task. The model was used for assessments for copper, zinc and lead, with adjoining discussions of antimony, bismuth, indium, gallium, germanium, tellurium, selenium, cadmium and silver.



Source: Own Figure, Lund University/University of Iceland

4.1.2.2 Objectives and scope

Our goal was to develop an integrated dynamic model for the global market for copper, zinc, lead and some of the technology metals dependent on their extraction; indium, bismuth, germanium, gallium, tellurium, cadmium, selenium, antimony and silver. The model must be able to simulate primary metal production rates from mines, recycling flows, market supply, world market price and to predict how the known and hidden extractable amounts and ore grades will develop with time. Copper zinc and lead occur together in many geological deposits, and are mostly extracted from poly-metallic ores. The objective is to use an integrated model to assess the long term supply sustainability of copper, zinc and lead and the metals that depend on them for their extraction, as a sub-module in the WORLD6 model. We will explore the metal supply system and explore how to make the supply of the metals modelled more sustainable. We include the whole global system in a generalized way; simulations cover the recent past and near future (1900-2200). Our focus in this study are the metals mentioned, but we pull into the picture the availability of fossil fuels, market dynamics and feedbacks as well as the human population size. This was undertaken by using a preliminary version of a global civilization model presently under development: The WORLD Model (Sverdrup et al., 2012a,b, 2013), and learning from the experiences of other metal models (Sverdrup et al., 2014a,b, 2015a,b).

4.1.2.3 Earlier modelling work

There have been several earlier attempts at modelling copper extraction rates. van Vuuren et al., (2009) outlined the principles of a model for the global metal cycling, looking at copper. Glöser et al. (2013) applied a mass and flows model to the global copper cycle. Northey et al. (2014) and Mohr et al. (2014) used a Mass Flow Analysis model for copper, reaching results similar to some of the results obtained in this study, they used a low estimate for the extractable amount. Common for Mass Flow Analysis models are that they do not incorporate feedbacks, nor do they have any market dynamics and cannot generate metal prices internally to the models. They are step-by-step mass balances advanced one year at a time a spread-sheet. Laherrere (2010) used Hubbert's model for copper, doing a valid analysis, but using a too small reserve estimate for the assessment to be valid. Ragnarsdottir et al. (2012) and Sverdrup et al., (2013) used Hubbert's model for copper, zinc, lead, indium, antimony, bismuth, germanium, gallium, tellurium, selenium and cadmium in an earlier assessment study. There is no earlier process-oriented systems dynamics model for copper, zinc or lead known to us, and nothing for the dependent technology metals antimony, bismuth, selenium, tellurium, indium, cadmium, germanium and gallium. We have earlier used simple back-of-the-envelope methods like burn-off rates and peak discovery shift, but these methods only give a diagnostic indication of potential problems. Earlier, we have developed systems dynamics models for gold, silver, copper, aluminium, iron, lithium and platinum group metals (Sverdrup et al., 2012, 2014a,b, 2015a,b,c, 2017a,b, Sverdrup and Ragnarsdottir 2017, Sverdrup 2017), rare earths (Kifle et al. 2012). In these models, one of the major advances were the development of a reality-based metal price model, allowing for price estimation from market fundamentals inside the models, without external forcing functions or calibration.

4.1.2.4 Data sources and estimations

The reserves and resources estimates are based on geological estimates, the interpretation of geological data, and the allocation of extractable amounts according to ore quality, stratified with extraction costs (Tilton 2007, 2009, 2012, Neumeyer 2000, Sweeneye 1992, USGS 2014). All major inputs were extracted from earlier published information (Moskalyk 2003, 2004, Alfantazi and Moskalyk 2003, Graedel et al. 2004, 2005, Spatari et al., 2005, Gordon et al. 2004, 2006, Graedel et al., 2005, Guirco 2005, Brown 2006, Rauch and Graedel 2007, Mudd 2007, 2010, Gerts 2009, Radetzki 2008, 2012, Roper 2009, Geoscience Australia 2009, Rauch and Pacyna 2009, Rauch and Pacyna 2009, Laherrere 2010, MinEX 2010, Crowson 2011a,b, NGEX 2011, Singer 2011, 2013, ICCM 2012, Glöser et al. 2013a,b, Anglo-American Mining Corporation 2012, Prior et al. 2012, Kesler and Wilkinson 2013, Johnson et al., USGS 2013, Dehnavi 2013, Bardi 2013, Copper Development Association 2013, Laherrere 2010, Harmsen et al., 2013, Kesler and Wilkinson 2013, Kavlak and Graedel 2013a,b, International Zinc Association 2013,204a,b, Henckens et al., 2014, Northey et al. 2014, Sverdrup et al., 2014b, Yoshimura et al., 2013). Most actors trade metals through the market, trade takes place geographically dispersed, but is linked through the price systems at the London and New York Metal Exchanges. The available data was closely inspected for inconsistencies and averages and adjustments were made when the input data were not internally consistent. Several of the numbers given in the literature are uncertain as some of them severely mismatch in the overall mass balance. Generally, the recycling numbers often appear inconsistent (UNEP 2011a,b, 2013a,b). The refining process from extracted ore to chemically separable metal concentrate has about 85-90% efficiency for copper and zinc. Further refining runs with 95-97% efficiency, thus the overall efficiency from mine output to metal to market is about 80-87%.

Basis for estimating how much can be extracted from ore deposits

The extractable amounts of indium, germanium, gallium, tellurium, selenium, bismuth and antimony were given as a % content of the parent metal extractable amounts and this is summed up with Table 4.1.2.1.

Table 4.1.2.1. Estimation of contents of dependent metals used for the BRONZE simulations. These contents represent approximate estimates as there is very little information available on this aspect in the literature. The metric used is fraction of weight. Extract is the amount extracted at the moment, content is an estimate of the average content in the whole resource.

Metal	Cu	Zn	Pb	Al	Ag
	Weight fraction				
Copper	0.84	0.14	0.006	-	0.00032
Zinc	0.14	0.80	0.05	-	0.00040
Lead	0.006	0.05	0.93	-	0.00008
Antimony	0.0065	0.0065	0.0065	-	-
Cadmium	0.002	High grade; 0.020 Low grade; 0.010 Ultralow grade; 0.003	-	-	-
Silver	260*10-6	125*10-6	130*10-6	-	0.90
Bismuth	330*10-6	-	440*10-6	-	-
Selenium	80*10-6	80*10-6	100*10-6	-	80*10-6
Indium	5*10-6	60*10-6	4*10-6	-	20*10-6
Gallium	-	23*10-6	-	8*10-6	-
Germanium	5*10-6	40*10-6	80*10-6	5*10-6	2*10-6
Tellurium	20*10-6	10*10-6	10*10-6	-	400*10-6

Table 4.1.2.2 shows the input data to the model used for the simulations. The extractable amounts were set at the beginning of the model simulation in 1800, stratified with respect to ore metal content. The distribution of copper to ore grades is based on the analysis presented in Figures 4.1.2.1 and 4.1.2.4. For zinc and lead, the distribution of URR to ore grades was based on a best fit of the simulations to observations of ore grade development with time, but also assisted by adopting the resource estimation methodology used for copper as outlined above. The estimates are including ocean massive

sulphides for copper, zinc and lead. We get the estimates for the ultimately recoverable resources (URR) for the three key metals as follows;

Copper; 2,850 million ton < URR= 4,030 million ton < 4,248 million ton

Zinc; 1,362 million ton < URR= 2,676 million ton < 2,800 million ton

Lead; 2,800 million ton < URR= 3,015 million ton < 3,300 million ton

This will be used as illustrated in Table 4.1.2.2 as input data to the model.

Table 4.1.2.2. The resource input data to the BRONZE model.

COPPER							
Ore grade	------------- Million ton copper -----------					Ore grade %	kg/ton content
	Known	Hidden	Sum	%	Cumulative		
Rich	15	5	20	0.5	20	40-5	400-50
High	10	670	692	19	845	5-1	50-10
Low	100	1,583	1,683	45	2,822	1-0.2	10-2
Ultralow	15	767	782	21	3,419	0.2-0.04	2-0.4
Extralow	15	540	555	15	3,736	0.04-0.01	0.4-0.1
Sums	155	3,579	3,734	100	-	-	-
Ocean floor	0	296	296	-	-	-	-
Total URR	155	3,875	4,030	-	-	-	-
ZINC							
Ore grade	------------- Million ton zinc -------------					Ore grade %	kg/ton content
	Known	Hidden	Sum	%	Cumulative		
Rich	1	28	29	1	29	40-5	400-50
High	5	310	315	12	344	5-1	50-10
Low	1	976	977	37	1,321	1-0.2	10-2
Ultralow	0	1,355	1,355	50	2,676	0.2-0.04	2-0.4
Sum	7	2,669	2,676	100	-	-	-
LEAD							
Ore grade	------------- Million ton lead -------------					Ore grade %	kg/ton content
	Known	Hidden	Sum	%	Cumulative		
Rich	20	10	30	1	40	40-5	400-50
High	5	40	45	1.5	75	5-1	50-10
Low	1	1,084	1,085	36	1,160	1-0.2	10-2
Ultralow	210	1,640	1,855	61.5	3,015	0.2-0.04	2-0.4
Sums	20	2,995	3,015	100	-	-	-

Dependence of recycling on price

Table 4.1.2.3 shows the recycling rate in 2012 and in 2100 as the fraction of the total market supply being from recycled waste from society based on a number of reports and assessments (UNEP 2011, 2013 and our own model runs). These values are used in the model as the standard setting.

Table 4.1.2.3. Estimated recycling rate in 2012 and in 2100 as the fraction of the total market supply being from recycled waste from society based on a number of reports and assessments.

Year	Indium	Gallium	Germanium	Bismuth	Tellurium	Antimony	Cadmium	Selenium
2012	40-63%	30-42%	33-50%	<10%	<10%	10-20%	15%	<2%
2100	75%	60%	65%	50%	50%	50%	80%	50%

4.1.2.5 Model description

Structural description

The BRONZE model is based on the earlier COPPER, SILVER and GOLD models and experiences learned from them (Sverdrup et al., 2013c, 2014a,b). The BRONZE model consists of the following parts based on two model published earlier and new systems dynamics parts (Figures 4.1.2.8-4.1.2.10):

1. The COPPER model (Sverdrup et al., 2014)
2. The zinc module
3. The lead module
   a) The module for technological metals dependent on the extraction from residuals coming from copper, zinc and lead refining and cross-linkages in the system.
   b) A simplified mass balance module for bismuth, cadmium, tellurium and germanium
   c) The INDIUM module, based on partly mining and contribution from copper, zinc and lead anode sludge extraction, and including market, stock-in-use and recycling of indium.
   d) The ANTIMONY module, based on partly mining and contribution from copper, zinc and lead anode sludge extraction, and including market, stock-in-use and recycling of antimony.
   e) The GALLIUM model based on a simplified version of the ALUMINIUM model integrated into the BRONZE model, and including market, stock-in-use and recycling of gallium. Gallium is extracted from bauxite processing to alumina.
   f) The SILVER model, contributes silver, and potentially tellurium, indium and gold as by-products of refining.
   g) The GOLD model, deriving gold from copper, zinc, silver and primary mining of gold

For these metals, the price has effect on demand, but not any significant impact on the supply, as this is dependent on the mother metal extraction rate. The technology metals considered are: Antimony, Bismuth, Cadmium, Indium, Gallium, Germanium, Selenium, Tellurium, Silver

Causal loop diagrams

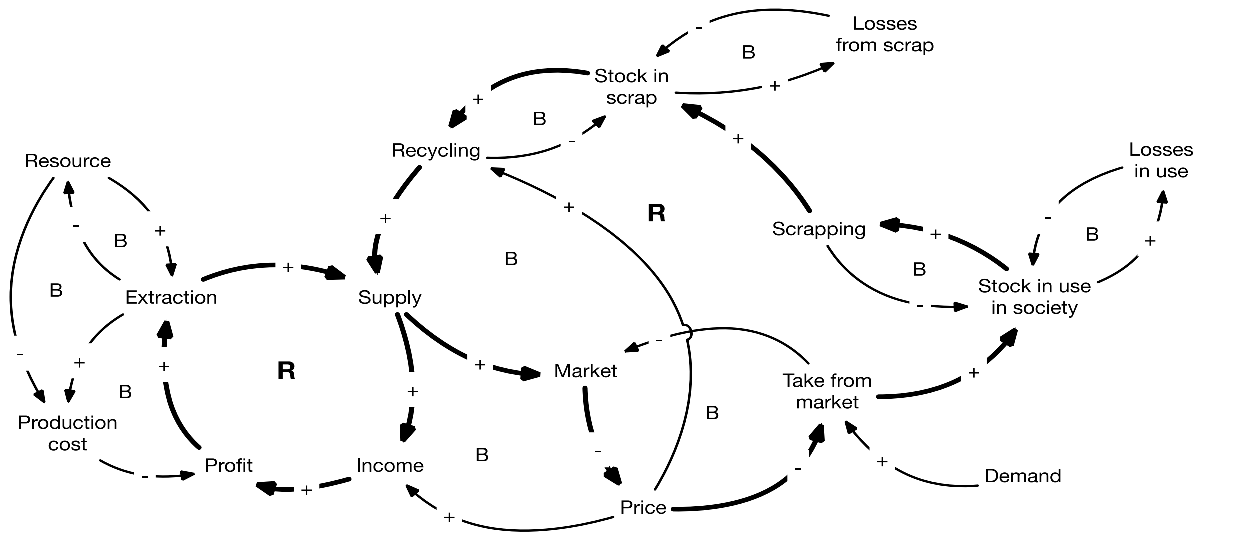
Figure 4.1.2.2 shows a partial causal loop diagram needed to understand how profits drive mining. B are balancing loops, balancing the reinforcing loops (R) and slowing them down. As the extractable amounts run low, more money must be spent on prospecting. When the hidden extractable amounts run low, then it will get more costly to find the dwindling extractable amounts remaining out there to be found. It shows how mining rate is driven by demand from society, and promoted by metal price and mining profit to generate supply to the market. The price is determined by how much metal is available in the market. A high metal price will stimulate the mining rate and increase supply to the market, and limit demand. But more supply to the market will increase the amount available and may potentially lower the price. The profit is affected by the mining cost and how that is modified with oil price and ore grade. The price is set relative to how much metal there is available in the market. The traders come to the trading floor with their lots to sell or to buy, and adjusts their amounts as the price goes up and down. When there is a match, the price is set. If demand is higher than production, the price goes up; in the opposite case the price is moved down. This is a self-adjusting mechanism that balances the trade by adjusting the prices until the demand to buy an amount at a price match the offers to sell an amount at a price. The causality in profits go to income from supply when the amount is supplied and paid at once into the metal exchange warehouse. The same applies with a forward sale, when the metal is paid upfront but physically delivered later. If not, all or part may be paid when the supplied amount has been cleared out from the physical warehouse. This description is based on personal observation on the trading floors at the metal markets in New York and London by the author. For many metals the amounts in the markets are recorded.

The scrapping process for stock-in-use in society is not strongly driven by price, but rather from the objects containing copper, zinc or lead becoming obsolete in some way (e.g. plumbing, machinery, wiring, metal objects). After the metal has arrived at the scrap heap, the metal price will have a promotion effect in causing somebody to recover it. The metal price in the market reflects several aspects. One is that the price will not go below the cost of actual production and extraction. And there needs to be profit above the extraction cost, normally about 20%.

For some of the specialty metals, there is not always a functioning market. When demand is far above supply, it also represents the degree of supply occasionally or short-term not being able to match demand, which is one aspect of scarcity. The potential for exponential growth when the resource exploitation is unrestricted and dynamic market price mechanisms that can limit demand are mechanistically incorporated in our process-oriented models.

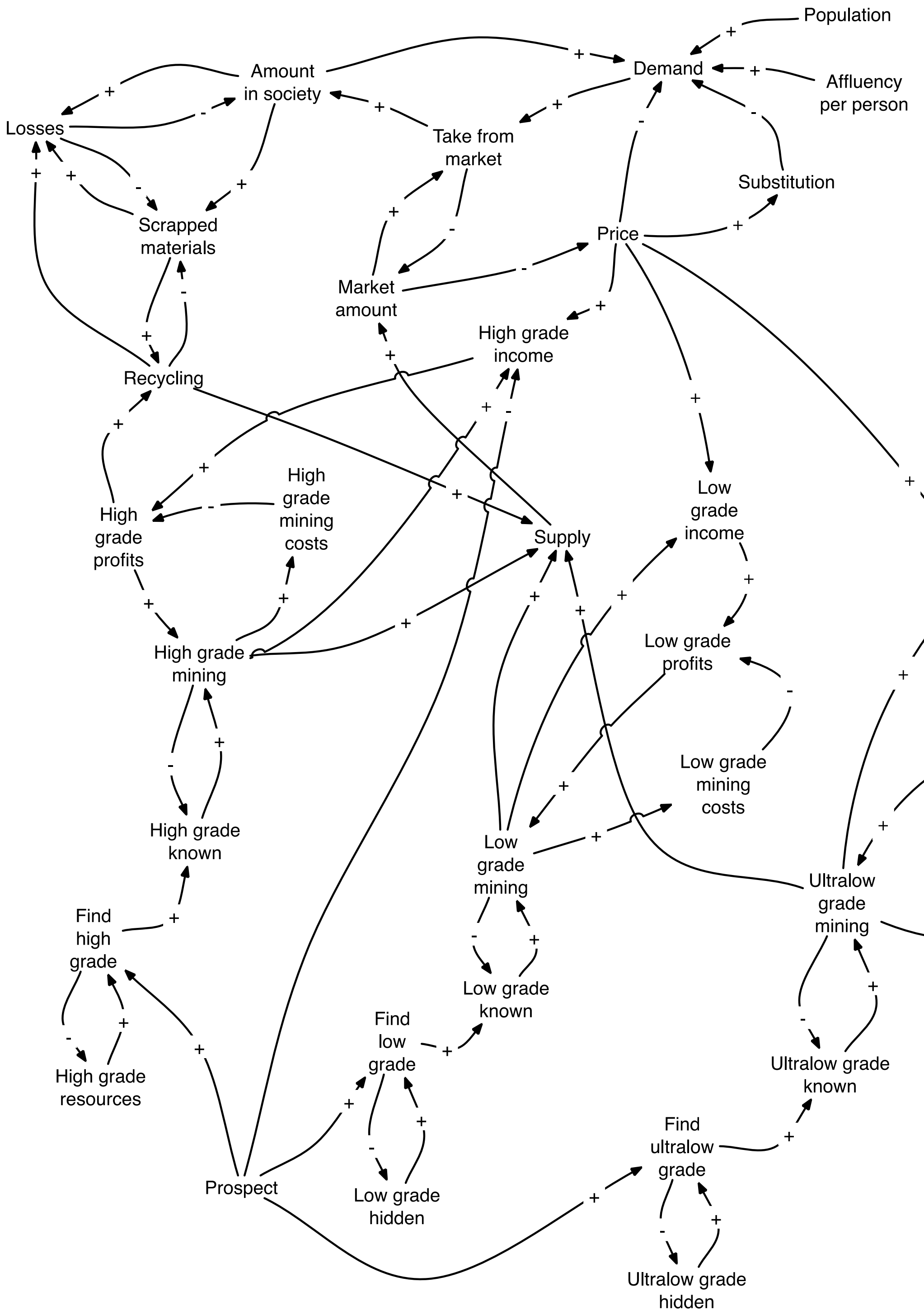
The full causal loop diagram for metal mining and copper in particular has been published earlier (Sverdrup et al., 2014a,b, 2015a) and is shown in Figure 4.1.2.3.

Figure 4.1.2.2. The causal loop diagram showing the market mechanisms and their effect on extraction, supply, demand and actual delivery from the market. This part of the model generates market price inside the model and eliminate the need for externally fed price time series. The price is used to modify the demand to become “Take from market”.



Source: Own Figure, Lund University/University of Iceland

Figure 4.1.2.3. Causal loop diagram for the mining activity for copper, zinc and lead in the BRONZE sub-module of the WORLD6 model.



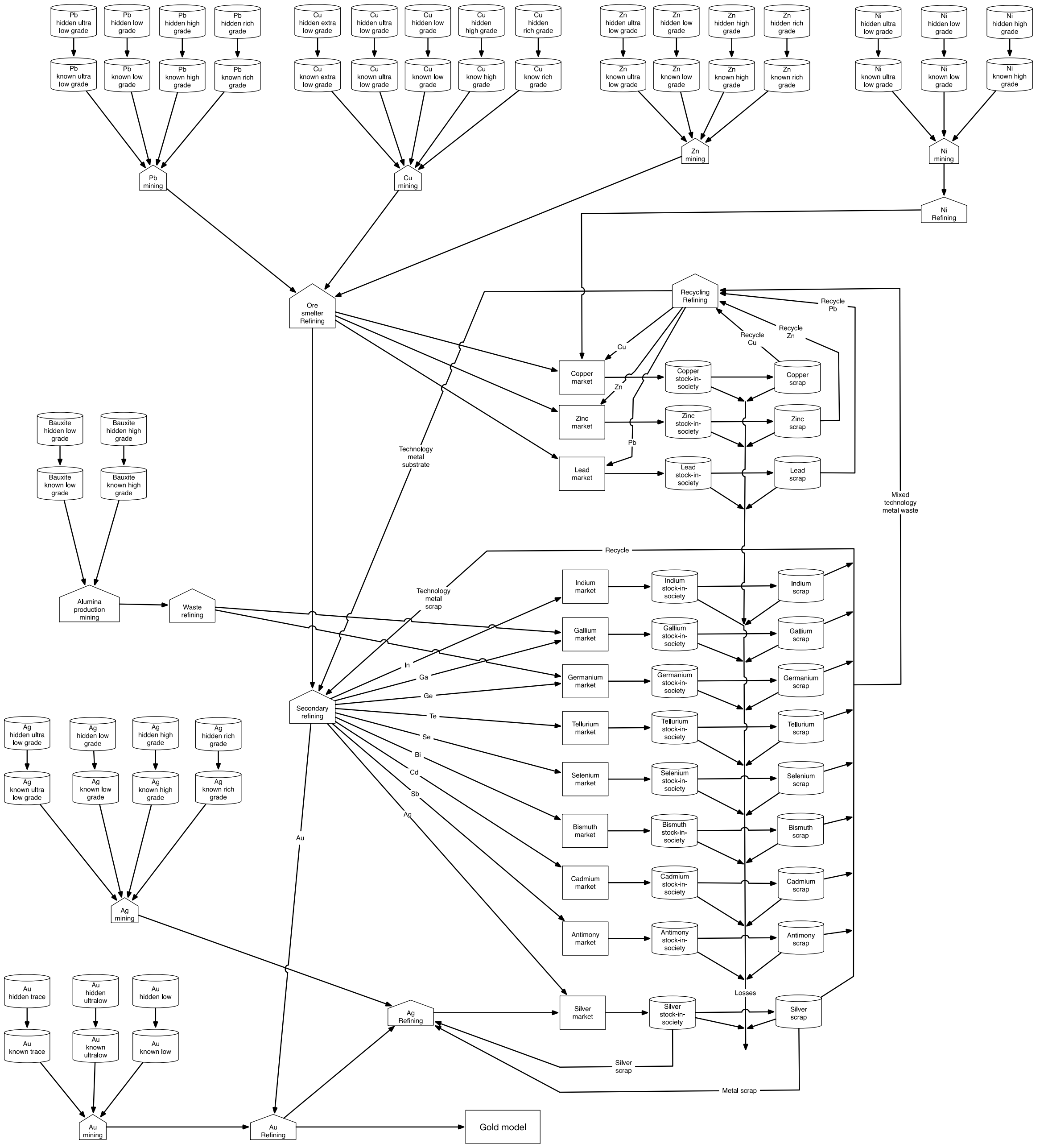
Source: Own Figure, Lund University/University of Iceland

Material flow charts

Figure 4.1.2.4 shows the flow chart for the BRONZE sub-model inside the WORLD 6 model, with the parent ores, the primary extraction and the dependent secondary extraction of many technology metals found in these parent metal ores. Many metals are extracted from poly-metallic ores, and they are all extracted out, many for new uses in new technologies. A full market model was not built for technology metals like germanium, gallium, indium, tellurium, selenium, bismuth or antimony, because the demand cannot readily be estimated well from the available data in a stand-alone model.

The data has been stratified with respect to ore metal content and relative extraction cost. The data from Tables 4.1.2.1-4.1.2.3 and experiences learned from studying corporate reports and scientific literature discussed in the text were reworked into the resource data and the estimates of costs of extraction. This is referred to as the opportunity cost approach (Tilton 2002, 2007, 2009, 2010). We have in the estimation of the extractable amounts of copper (Table 4.1.2.2), considered both porphyric (70% of land deposits, and 65% of all global extractable deposits) and sulphide deposits (30% of land deposits, and 28% of all global extractable deposits) and ocean floor copper resources in cobalt crusts and subsea hydrothermal massive sulphides for copper (7% of all global extractable deposits), using the literature reviewed earlier for this purpose.

Figure 4.1.2.4. Flowchart for a part of the BRONZE model, including independent and dependent metals.

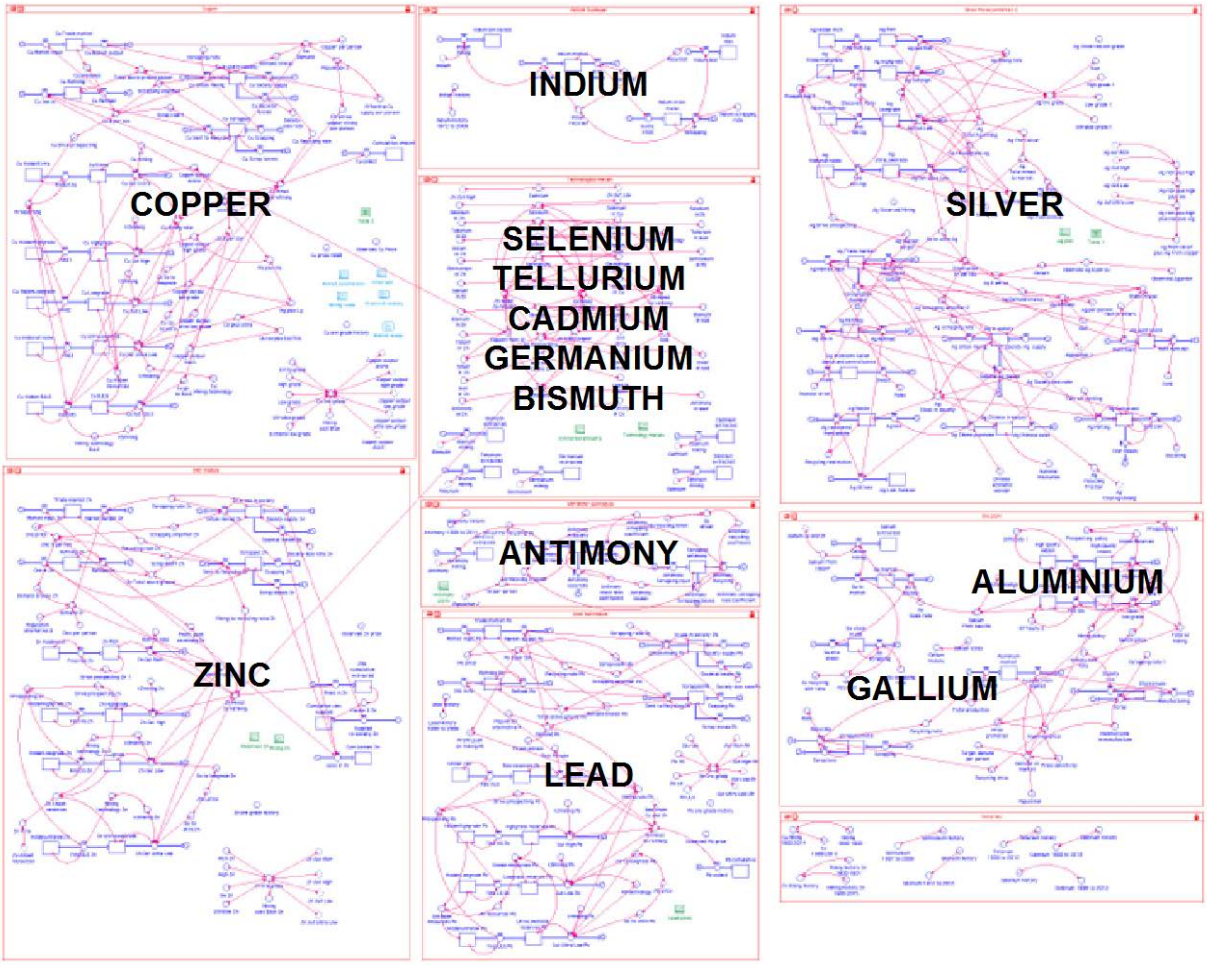


Source: Own Figure, Lund University/University of Iceland

The STELLA diagram and the computer code

The STELLA diagram for the BRONZE system dynamics model is shown in Figure 4.1.2.5, consistent with the causal loop diagram in Figure 4.1.2.3 and the flow chart in Figure 4.1.2.4.

Figure 4.1.2.5. The STELLA diagram for the BRONZE sub-model of WORLD 6 model. The primary mining copper, zinc and lead have their own modules, the dependent metals are calculated from the production of these, with extra sub-modules for antimony mining and with a special module gallium extraction from bauxite benefaction processing to alumina. The whole structure is embedded in the WORLD6 model



Source: Own Figure, Lund University/University of Iceland

The BRONZE model has several sectors, one for copper, zinc, lead and silver, and one for all the different technology metals. The mining modules where the reserve is divided into known reserves and hidden reserves, and stratified into 5 levels of ore quality; rich high grade, high grade, low grade, ultra low grade and extreme low grade. The input data is shown in Table 4.1.2.2 for copper, zinc and lead. Reserves move from hidden to known because of prospecting. The following metals have a full mass balance-based market model, generating demand, supply, price and recycling in the model: Copper, zinc, lead antimony, indium, silver, gallium and aluminium. Selenium, tellurium, cadmium, germanium and bismuth are derived by straight dependency and no market model is used, no market price nor recycling is addressed. The model is numerically integrated using a 0.025-year time-step in a 4-step Runge-Kutta numerical method.

The equations and parameter settings

The general mining rate equation used for metals in the WORLD6 model is:

$$r_{Mining} = k_{Mining} * m_{Known}^{n} * f(T) * g(P) * h(Y) \quad (4.1.2.1)$$

where rMining is the rate of mining, kMining is the rate coefficient and mKnown is the mass of the ore body, and n is the mining order, where f(T) is a technology improvement function dependent on time, g(P) is a feed-back from price on the prospecting rate. h(Y) is a rate adjustment factor to account for differences in extraction yield when the ore grade decreases. These functions are given exogeneously to the model. The size of the extractable ore body is determined by the extractions and prospecting. However, it appears from a sensitivity analysis that the results are not very sensitive to the value of n. It turns out that the impact of choosing a different mining model on the final result is small. The discovery is a function of how much prospecting we do and how much there is left to find. The amount hidden resource decreases with the rate of discovery. The rate of discovery is dependent on the amount metal hidden. The prospecting coefficient depends on the amount of effort spent and the technical method used for prospecting. Prospecting also depends on how large the known reserves are. The larger they are, the less the urge to drill for more according to the causal loop in Figure 4.1.2.3. The basic driving mechanism of basic mining comes from profits and availability of a mineable resource used in the model. The model as a causal loop diagram (CLD) for the system is in principle the same as those published earlier for silver, copper and aluminium (Sverdrup et al., 2014a,b, 2015a). The causal loop diagram maps all significant causal connections in the system. It is important that the links are true causal links and not just correlations or modelled on chain of events. The causal loop diagram together with the flow chart, defines the model.

$$r_{Dependent\ metal} = X_{Dependent\ metal} * r_{Mother\ metal} \quad (4.1.2.2)$$

Table 4.1.2.4 shows the values for the coefficient XDependent metal in Equation 4.1.2.2 as used in the model. It shows the estimation of contents of dependent metals used for the WORLD6 model simulations. These contents represent very approximate estimates, as there is very little information available on this aspect in the literature. It is known that for gallium, the content potential in bauxite is not fully used for a number of reasons, some of them technical. Indium occurs with zinc in sulphide minerals because the two elements have similar atomic radii and similar chemical properties. The main indium sources at present are extraction from zinc and lead. Gallium can be extracted when bauxite is processed to alumina and if the extraction plant has the necessary infrastructure for it, which is not always the case. Lack of infrastructure is often the case with extraction of indium, germanium and tellurium in the copper, zinc, lead and silver refining processes from mother metals. The metric used in Table 4.1.2.1 is fraction of weight. Only a small part of the indium present in all the zinc mined zinc is extracted. On the average, zinc contains about 448 ppm of indium, or corresponding to 0.045% by weight. However, only 60 ppm of the 448 ppm of indium are extracted, the rest is not recovered and is lost with the produced zinc. 448 ppm corresponds to an URR of 1.2 million ton of indium, which at present is being lost. With only 60 ppm of indium recovered this is an URR of 0.161 million ton of indium. In copper, the indium content is on the average 5 ppm, but some mines have as much as 28 ppm. In lead the indium content is on the average 4 ppm.

In the model, purchases from the market are driven by demand, put copper into society where it stays until scrapped or removed by wear and losses. A part of the scrapped copper, zinc and lead is recycled and returned to the refinery. Demand in the model is driven by population and copper use per person, but is adjusted up or down with price. Copper stock in use per person as related to global GDP and as it has developed over time. There seems to be ageneral saturation level at about 10-12 kg of indium per person. Demand is also affected by the market price, which in turn depends on how much is available in the market. The demand is estimated from affluence and global population using outputs of the

WORLD6 model system (Ragnarsdottir et al., 2011, 2012, Sverdrup et al. 2005, 2011, 2012a,c, 2014a,b, 2015a,b, Sverdrup and Ragnarsdottir 2011). There seems to be a saturation level at about 12-13 kg of copper per capita on a global basis. The corresponding number for zinc appears to be 5 kg of zinc capita in developed nations. For gallium, additional material comes from the processing of bauxite ore to alumina. This is generated from the ALUMINIUM module that has been integrated into BRONZE (Sverdrup et al., 2015a), which in turn is a subroutine in the WORLD 6 model.

4.1.2.6 Results

Figure 4.1.2.6 shows the production rate for the technology metals gallium, indium, tellurium, germanium, cadmium and selenium as modelled with the BRONZE model. Table 4.1.2.3 shows the recycling rate in 2012 as the fraction of the total market supply being from recycled waste from society based on a number of reports and assessments. The recycling rate is related both metal price and promotion policies, the simulated recycling for gallium and indium is shown in Figure 4.1.2.6b and Figure 4.1.2.6d. The year 2012 indicates the year the recycling rate was estimated for and 2100 shows the future assumed maximum feasible recycling rate used in the model calculations. The production of the dependent metals is limited to some extent by the fact that only a fraction of the metal refineries is technically equipped for efficient recovery of these metals. Thus, we may have underestimated the potential somewhat. Significant investments would be needed in order to increase production.

Gallium

Figure 4.1.2.6a shows the production and the contribution from bauxite refining to alumina (75% of the gallium) and from zinc production (25% of the gallium) as compared to the recorded mining history (USGS 2014). A eutectic alloy of gallium, tin and indium make an important non-toxic substitute for mercury in many of its applications. The ALUMINIUM model was used for the contribution from bauxite mining, integrated as a sub-module into the BRONZE model. At present, gallium is mostly derived from bauxite processing, a preparation for aluminium production, but other sources like zinc and silver are becoming more important with time. Figure 4.1.2.6b shows the mining rate as compared to the recycling flux and the total supply to the market. The gallium supply to the market has a maximum capacity in 2045. When the source ores decrease in grade, extraction from mining side production becomes more difficult. Light Emitting Diode lighting technology is rapidly spreading as mass products in consumer lighting devices, uses metals like gallium, antimony, indium, arsenic, selenium, yttrium, cerium and germanium. Gallium and indium often a main substance.

Indium

Figure 4.1.2.6c shows the mining rate and the recorded extraction history (USGS 2014). Figure 4.1.2.6d shows the mining rate as compared to the recycling flux and the total supply to the market. Indium is mainly used in LCD screens and LCD lighting, and as a grain stabilizer agent in alloys. Indium production depends on copper and zinc mining, some silver ores also contain significant amounts of indium, and in the longer run, these may be developed at the larger silver refineries. We may expect on-going prospecting for indium to be successful in finding new parent metals for indium or new zinc, tin, copper or iron ore deposits with higher contents. This could potentially increase our estimate of URR by a factor of two. Certain tin deposits in Cornwall have been discovered have as much as 50g to 100g indium per ton ore. Potentially, aluminium deposits with gallium may as well have traces of indium due to the similarity of the two. Indium supply will have peak capacity 2040-2050, whereas mining has a maximum about 2035. Some business analysts (Gibson and Heyes 2013) suggest on a future increase in demand, and if this is larger than the underlying growth in zinc production, the prices can be expected to rise sharply. Indium is used widely in Light Emitting Diodes (LED), and the demand is rising rapidly. Indium prices have in the latest years fluctuated in distinct cycles between 120 and 400 $ per kg, or 120,000 to 400,000 $ per ton.

Tellurium

Tellurium is a very rare metal, on par with platinum in the Earth's crust. Tellurium is mostly derived from copper and lead ore, often from anode sludge after electrolytic refining of blister copper or furnace dust from lead smelting. Certain ores zinc and silver ore can have significant contents of tellurium. The data we have seen has been highly variable, making it difficult to set a global average for the model. Tellurium has found new uses in specialty technologies such as solar panels, and in semiconductors. The demand is fast growing, faster than copper and lead production, and supply will possibly fall short of demand with rising price as the response. Figure 4.1.2.6e shows the observed extraction data and the modelled extraction of tellurium. The model assessment suggests that there is more ore content available for tellurium extraction, and that the challenge can be met but adapting refineries for more extraction.

Germanium

Germanium extraction data and simulation is shown in Figure 4.1.2.6f. Germanium is also special as it comes from several independent sources (See Table 4.1.2.1); copper, zinc and lead mining, as well as from the processing of bauxite for aluminium production. Some Russian coal mines extract germanium as a by-product. For the last decade, germanium production has varied a lot, and the price has seen a lot of fluctuations. Some silver ores contain germanium and this may contribute significantly in the future. Germanium is used for as a chemical catalyst, for light diodes, solar panels and transistors, business areas that will probably increase in the future, and so will demand. If the demand would rise faster than copper and zinc mining, then we may expect germanium supply to be less than demand and the price may increase significantly.

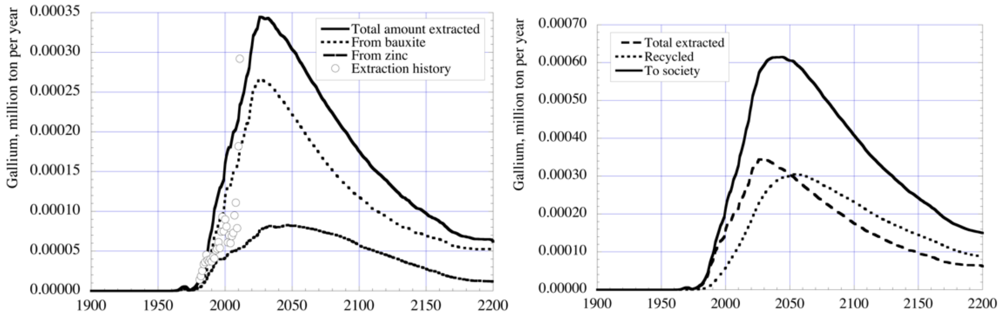
Cadmium

Cadmium extraction is shown in Figure 4.1.2.6g. Cadmium is extracted to full availability from zinc ores. In the calculation for cadmium we have only assumed cadmium to be extractable from high grade (1-0.7% content in zinc) and low-grade zinc ores (0.3-0.7 % content in zinc), but it does not occur in extractable amounts and concentrations in ultralow grade zinc ore. Cadmium is expected to peak soon, but demand is decreasing for its use as pigment (16%), in soldering alloys and in other metal alloys because of health and environmental concerns. It is still used in accumulators, but these may eventually also be phased out. The fit between data and modelled may be interpreted as if we may have assumed a slightly too low cadmium content in the zinc ore.

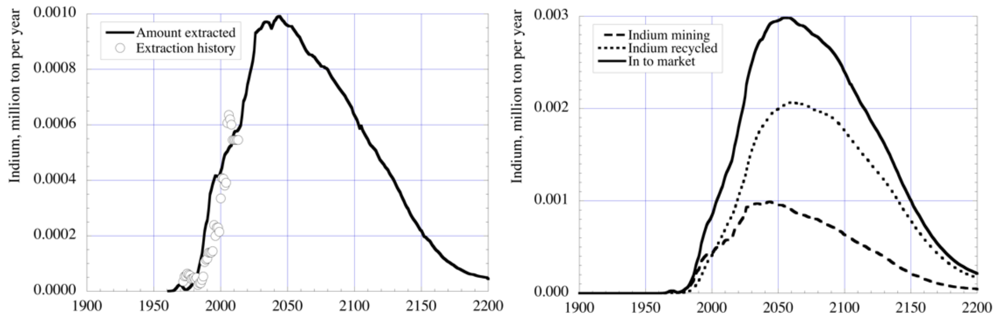
Selenium

Selenium is produced from sulphide ores or copper, nickel and lead and from the production of sulphuric acid. Especially important is anode mud from electrolysis refining of copper. With change of method to electro-winning of copper from ore concentrates, less selenium is extracted from the ore and the extraction sludge is less useful for selenium extraction. The long-term perspective is that selenium extraction is about to stagnate and even decline, despite increased demand. This points into a future with physical shortages and increased prices. Figure 4.1.2.6h shows the observed data and the modelled extraction of selenium.

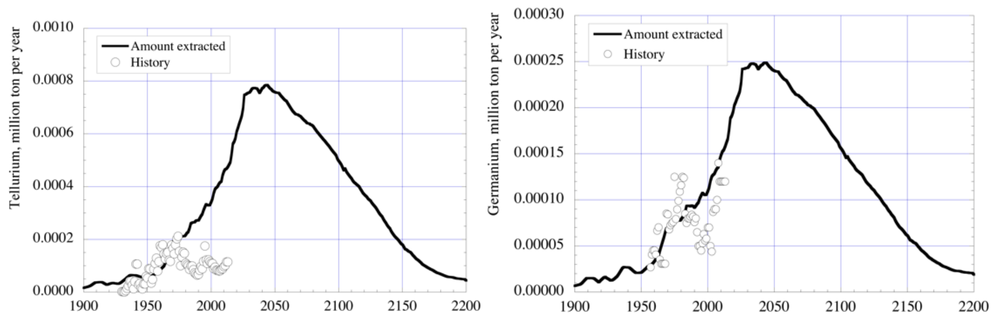
Figure 4.1.2.6. The production rate for the technology metals gallium (Diagram a and b), indium (Diagram c and d), tellurium (Diagram e), germanium (Diagram f), cadmium (Diagram g) and selenium (Diagram h) as modelled with the BRONZE model, as compared to the observed data on extraction from ores. The tellurium estimate is based on the extraction being done at the available extraction capacity. For many years before the invention of the new technologies in electronics, photovoltaic technologies and LED technologies, tellurium demand was low.



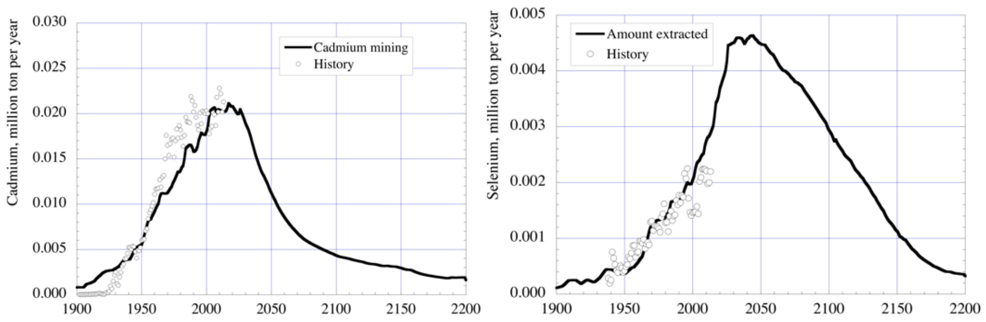
a Gallium b



c Indium d



e: Tellurium f: Germanium



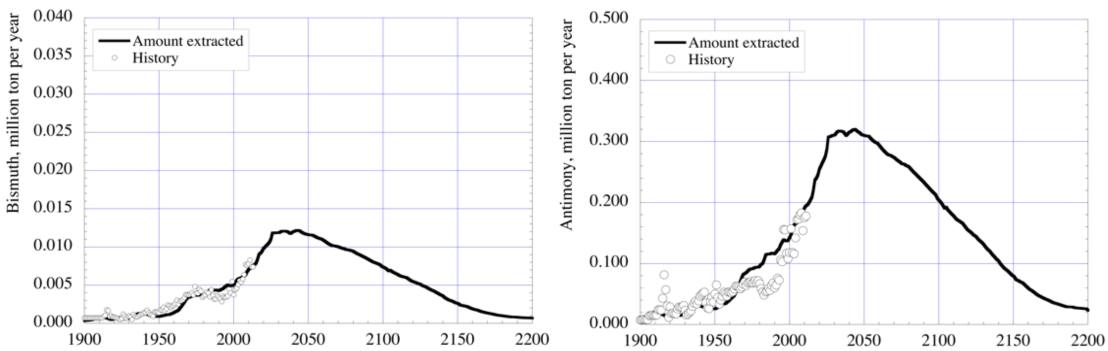
g: Cadmium h: Selenium

Source: Own Figure, Lund University/University of Iceland

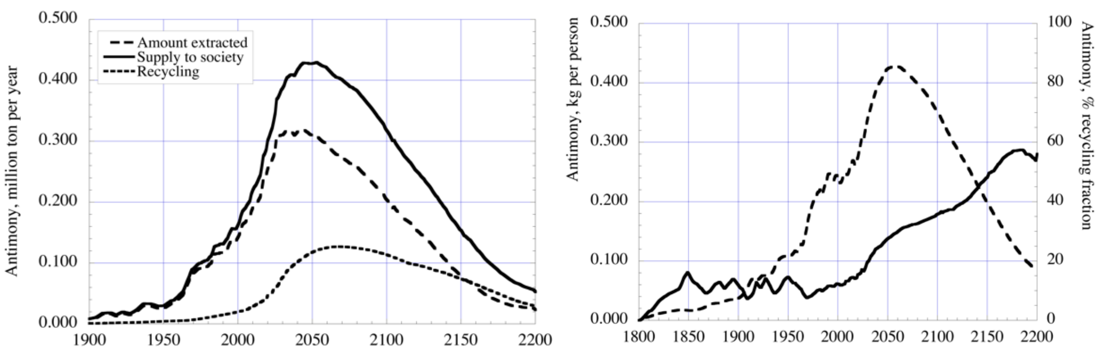
Antimony and bismuth

Antimony has both primary mines (70%) and secondary production (30%). It is the only of these metals that have any recycling to speak of (Butterman and Carlin 2004, UNEP 2013). It is a part of solder used in electronics and gets partly recycled together with tin and lead. Another important use is flame retardant chemicals. The total geological extractable amounts include about 6-8 million ton in primary mines (mostly from ore based on the antimony sulphide called stibnite, USGS 2014) and about 6 million ton in poly-metallic ores (Copper-zinc-lead ores, tin ores, tungsten ores) according to traditional estimates (USGS 2014). Extending the estimate with new results on copper, zinc and lead reserves (Sverdrup et al. 2014b and this study), we can expand the total antimony extractable resource to about 40 million ton (Tables 4.1.2.7 and 4.1.2.8). Figure 4.1.2.7 shows the production rate for the metals bismuth (Figure 4.1.2.7a) and antimony (Figure 4.1.2.7b) from the antimony sub-module in BRONZE with antimony extraction compared to recycling and supply to society. Figure 4.1.2.7c-d shows more specific outputs for the antimony sub-module within the BRONZE model. Figure 4.1.2.7c shows the stocks in society, the accumulated amount extracted and the amount for recycling. Because of recycling, the supply to society will be larger than the extraction rate from mining by-products. After 2050 the contribution from mining to the stock in society will be less than the losses. After this point in time, the stock will decrease. Figure 4.1.2.7d shows the amount antimony per person on a global scale and the % of the supply to society that comes from recycling. Their use is of such a nature, that it is already very diluted in the products, and the metal price is not high enough to warrant recycling efforts. How large the recycling of antimony really is in the world is quite uncertain, but the UNEP-IRP suggests that it is about 10-20% in one of their recent reports (Table 4.1.2.3).

Figure 4.1.2.7. Diagram (a) shows the production rate for the technology metal bismuth (Ragnarsdottir et al. 2011, Sverdrup et al. 2012). Diagram (b) shows antimony extraction compared to data. Diagram (c) shows the extraction rate supply to society and recycling. Diagram (d) shows the amount antimony per person on a global scale and the % of the supply to society that comes from recycling.



a: Bismuth b: Antimony

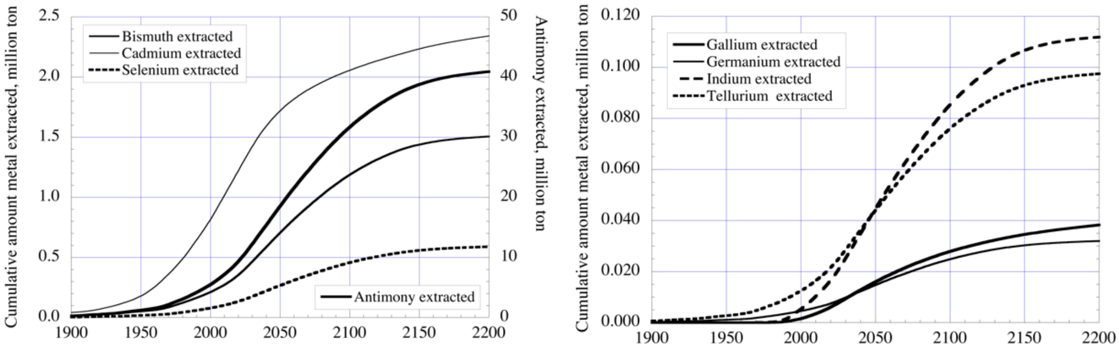


c Antimony d

Source: Own Figure, Lund University/University of Iceland

Figure 4.1.2.8 shows the cumulative amount metal extracted until year 2200 using the model simulation for antimony, bismuth, selenium and cadmium (Figure 4.1.2.8a) and gallium, germanium, indium and tellurium (Figure 4.1.2.8b).

Figure 4.1.2.8. Diagram (a) The cumulative amount metal extracted until year 2200 as recorded in the BRONZE model simulation for antimony, bismuth, cadmium and selenium. Diagram (b) shows cumulative extracted amount for gallium, germanium, indium and tellurium.



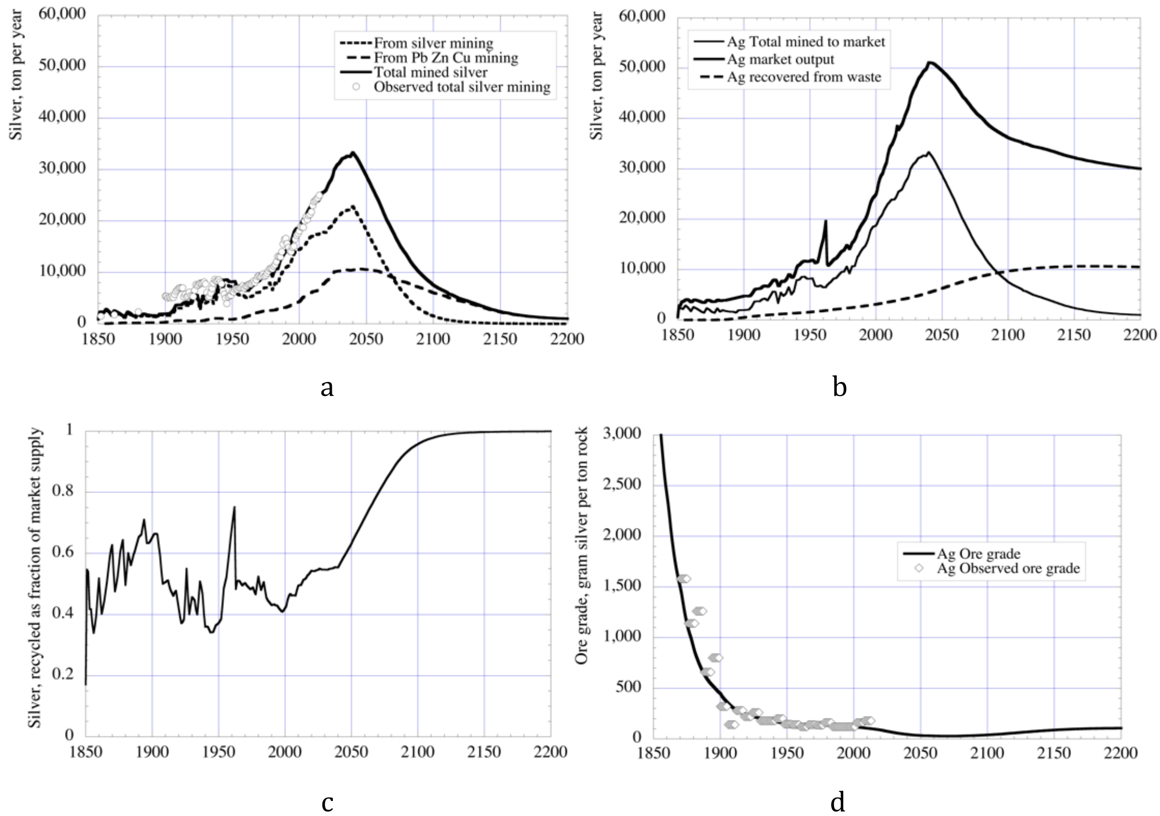
a: Antimony, bismuth, selenium, cadmium b: Gallium, germanium, indium, tellurium

Source: Own Figure, Lund University/University of Iceland

Silver

A significant amount of gold and silver originates from copper, zinc and lead mining (Figure 4.1.2.4). The runs were done by integrating an updated version of the model for silver (SILVER-II) into the model, collecting contributions from silver ores, and additional flow from copper, zinc and lead extraction. Figure 4.1.2.9a shows the simulation results of the amount of silver mined, originated from lead, zinc and copper, and comparison with the observed silver mining. Figure 4.1.2.9b shows the total amount mined and supplied to the society. Figure 4.1.2.9c shows the recycled silver as fraction of market supply. Figure 4.1.2.9d shows silver ore grade with time as compared to the observed data. These are new runs as compared to an older model published in 2014 (SILVER, Sverdrup et al., 2014a), which will be redundant to the model. Most of the gold recovered this way comes from copper mining, but small amounts also originate from zinc and lead. Silver originates from all three main ores as well as the poly-metallic ones. The peak production from these poly-metallic ores is for gold at 793 ton per year in 2045 (25% of the total production) and for silver at 9,700 ton per year in 2045 (35% of the total production). The total silver production at present is 25,000 ton per year, and it will peak at 27,000 ton per year in 2032. Thus, the contribution from this kind of source is significant for both gold (27% of present production) and silver (42% of present production). The total gold production was in 2012-2015 about 2,800-2,900 ton per year, and our models predict that it will probably peak at about 3,300 ton per year and decline after that. The silver production is also near the peak (Sverdrup et al., 2014a). This does not mean that we will be without silver and gold, they have consistently high recycling rates and already about 2025, recycling will probably be the dominant source of these metals.

Figure 4.1.2.9. Diagrams (a) shows the simulation results of the amount of silver mined, originated from lead, zinc and copper, and comparison with the observed silver mining. Diagram (b) shows the total amount mined and supplied to the society. Diagram (c) shows the recycled silver as fraction of market supply. Diagram (d) shows silver ore grade with time as compared to the observed data.

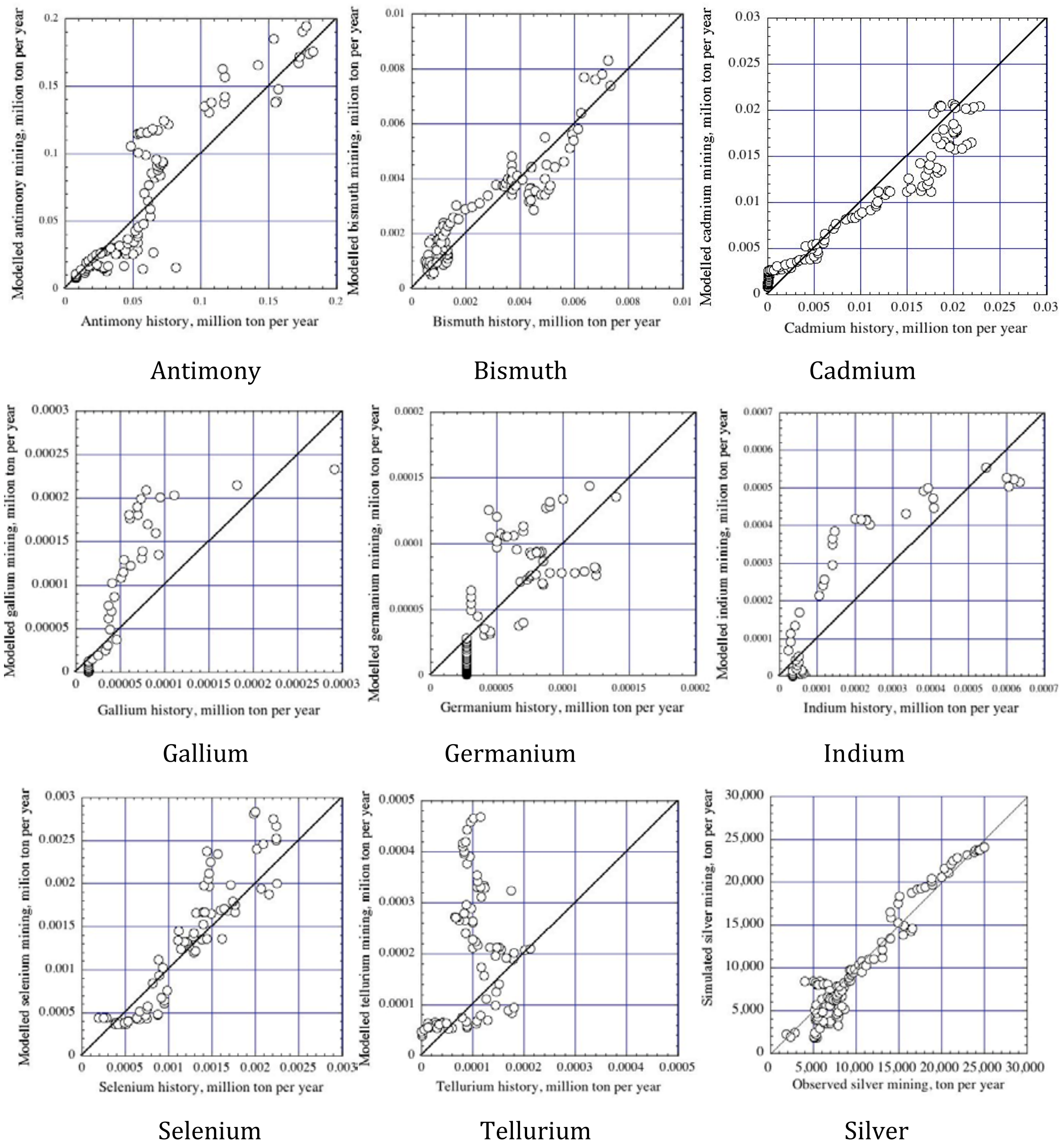


Source: Own Figure, Lund University/University of Iceland

Testing the model

The model has been tested against the recorded mining data derived from the USGS (2013) databases. The model does reproduce the observed mining rates satisfactory when the model is driven by market demand and price dynamics. Figure 4.1.2.10 shows comparisons of the model mining outputs to observed data with a good performance for the technology metals; antimony, bismuth, cadmium, gallium, germanium, indium, selenium, tellurium, for antimony and bismuth. They all showed from good correlation with the observed mining data, with the single exception of tellurium.

Figure 4.1.2.10. Comparison of modelled mining rate as compared to the observed data.



Source: Own Figure, Lund University/University of Iceland

The correlation coefficient for simulated extraction to data are shown in Table 4.1.2.4 and show that the rates are very well reconstructed. For gallium, germanium and indium, the extraction was less than the supply capacity initially, but that that has picked up recently. For germanium, extraction has been variable, and in some periods significantly under the potential capacity. For gallium, there seems to be less extraction than what is predicted and what is technically and economically feasible. This suggests an opportunity not taken by industry, partly out of ignorance, partly because of lack of investment in the proper infrastructure. For cadmium, the underproduction is caused by an environmental ban on cadmium due to its great toxicity. For tellurium, it reflects the past history of a troubled market, and the very recent revival caused by new demand from new technologies. The lower correlation for tellurium is related to the fact that the extraction did not utilize the extraction potential, mostly because of lack of demand for tellurium until very recently.

Table 4.1.2.4. Overview of the correlations (r2) between observed data and simulations for the metals of mainly secondary extraction origin.

Metal	Correlations	Extraction	Metal	Correlations	Extraction
Antimony	0.81	Combined	Germanium	0.69	Secondary
Bismuth	0.88	Secondary	Indium	0.82	Secondary
Cadmium	0.95	Secondary	Tellurium	0.12	Secondary
Gallium	0.74	Secondary	Selenium	0.93	Secondary

4.1.2.7 Discussion

Reserves and extractable amounts of metal

Indium may also be present in additional copper, silver, tin and iron deposits, possibly increasing the extractable amounts to almost 220,000 ton globally. The tellurium content in silver ore is about the same as in copper ore, sometimes more. USGS estimates of selenium reserves indicate about 100,000 ton of selenium as known and extractable. We estimate the total extractable amount to be at least 600,000 ton. The content of the technology metals in the parent metals are very approximate. They are the result of estimations made by the authors for single producers, as very little is available on this on a global level, even if a few producers report useful numbers. These contents may change in the future once better data becomes available. It is known that for gallium, the content potential in bauxite is not fully used for a number of reasons, some of them technical. This may also be the case with indium, germanium and tellurium.

Sustainability of mining

The declining ore grade is a clear indication that the resource has been depleted and that scarcity has already set in. For some time, this was offset by improvements in efficiency and lowering of wages, but that has recently come to an end. In the model, the demand is made on the market, and when the market amount goes low, then the prices go higher. The higher prices push mining, causing the price to increase. The model becomes self-regulating. Thus, the market dynamics are fully expressed in the present model and that gives a significantly smoother production curve and better dynamics. In the discussion of sustainability, we should not that recycling can delay symptoms of scarcity for a significant time even after the production from mines has stopped. The recycled copper, zinc and lead have nothing of the dependent metals preserved and this must be considered in a future strategy for how they are to be sourced.

Resource scarcity

The dependent metals may fast run into physical scarcity because the price has no feedback on the extraction rate of the mother metals copper, zinc or lead. It is important to consider that when the commodity is still relatively cheap, it may for that reason be unnecessarily wasted, when it would in retrospect have been relatively easy to induce better use efficiency and recycling. To wait until there is severe scarcity would be too late for optimal mitigation as unnecessary resources would already have been wasted by then. Others have proposed deep ocean mining (Scott 2001) but efficient and affordable technology for this has failed to surface so far. Different authors point to future risks and possibilities and mainly focus on issues of short term market conditions, better technologies and new markets.

The future will show how these issues will really work out under a different set of conditions with potential scarcity and additional challenges of climate change (Wagner and Fettweis 2001, Ghose 2009, Eliott et al., 2014, Morrigan 2010). The purpose of models is to look into what would be possible in the future. The model does not tell us what the future will be, but can illustrate what futures are possible, and something about how likely they are to happen. This is the only way we can plan ahead of time and avoid getting into unsustainable situations. At present, many studies and model assessments point out that the global resource use is in large overshoot and that we have limited time left for corrective action. And when thermodynamic corrective enforcement has already started, then it may be too late to do the mitigation in an orderly manner.

4.1.2.8 Conclusion

Using the the BRONZE sub-model embedded inside the WORLD 6 model does give a richer picture, including more details and essential feedbacks. We are able to reconstruct the past history of mining rates, ore grade decline and approximate metal price levels. We get reasonable estimates of the production rates for the technology metals dependent on copper, zinc and lead for their production. We need to carefully distinguish between the primary production from mines and total supply to the market. Long after primary metal mining has been reduced to insignificant levels, supply may be kept up by efficient recycling. For the technology metals, many have at present low rates of recycling and these are much more vulnerable for decline in mining, unless the recycling efficiency can be significantly improved. The conclusion is that there is not an immediate risk for scarcity but that in the longer run (after 2030), scarcity manifested as rising metal prices should be expected and prepared for.

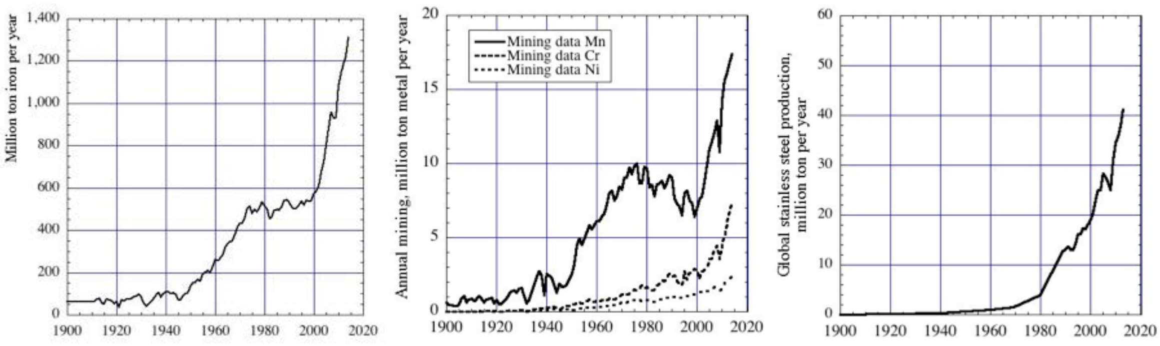
4.1.3 STEEL sub-module – IRON, MANGANESE, CHROMIUM, NICKEL, and STAINLESS STEEL

4.1.3.1 Introduction

The world's most important synthetic metal is steel, which has iron as its main component. 92% of all metal on the globe used in society is made out of iron. In total shipped weight, any other metal is insignificant. Though it may appear that there is a nearly endless amount of iron on our planet, what is available to humans for extraction, is finite and limited, and thus by consequence of thermodynamics, exhaustible. Iron and steel scarcity would be a huge problem for the world economy and the continuation of the modern world.

Iron is the most important metal for human society (Smith 1776, Champion et al. 1984, Rostoker et al. 1984, Rostker and Bennet 1990, Diamond 1997, Distelkamp et al. 2010, Meyer et al, 2012a,b, Bleischwitz et al. 2012, Acemoglu and Robinson 2013, World Steel Association 2012, 2014a,b, Eliott et al. 2014, Kennedy 1987). Iron is one of the big six metals that build the modern world (Heinberg 2001, 2005, Ragnarsdottir et al. 2011, Sverdrup et al. 2013, 2015a). These are in order of yearly production (2014); In Figure 4.1.3.1a it can be seen that iron extraction is about 1,350 million ton per year from mining, but the total supply to society is significantly larger because of recycling (About 2,200 million ton). Aluminium mining is about 50 million ton per year, but 80 million ton per year if recycling is included, copper mining is 17 million ton per year, but as much as 35 million ton per year is supplied to society when recycling is included. Zinc mining is about 12 million ton per year. Manganese mining is about 18 million ton per year, chromium production is about 8 million ton per year (Figure 4.1.3.1b). All other metals than iron combined, make up less than 130 million ton of metal per year in production. Iron is used in alloys with carbon, silicon, manganese, chromium, molybdenum, vanadium and nickel for making different types of alloyed steel. Steel is the major metal in structures, in everyday utensils, cars, ships, houses, bridges all the way to everyday items like teaspoons. 97% of all iron is made into some type of steel (World Steel Association 2014a). Figure 4.1.3.1c shows the approximate global stainless steel production 1900-2014 in million ton of metal per year.

Figure 4.1.3.1. Approximate global iron (a) and stainless steel (c) production 1900-2014 in million ton of metal per year is shown. Extraction data for chromium, manganese and nickel are shown in (b).



Iron Manganese, Chromium, Nickel Stainless steel

a b c

Source: Own Figure, Lund University/University of Iceland

Industrial reduction of iron from the oxide ore is energy demanding. The basic principle is to reduce the oxide with carbon, creating metallic iron and carbon dioxide in the process. Iron rich rocks are found on all continents, but in addition to ore grade and deposit size, factors like existing infrastructure for production, rail transportation possibilities for large tonnage of material, shipping ports and distance to customers is of equal importance. The total production in 2012 was 1,350 million ton per year, the metal yield from the iron ore is at present 47 % iron yield per weight of ore (World Steel Association 2014a,b,c). The weight averaged ore grade from the same source is stated to be 58% iron content, which means some iron is lost to the waste and slag in the process. The numbers imply an average yield of 81% from deposit to metal. Most of the world's production of manganese, chromium, nickel are alloyed into this amount of iron to make steel of different qualities. There are several natural resources used in producing iron, carbon steels and stainless steels:

1. Iron and conventional steel products
    a. Metals
        i. Iron metal extracted from ore
        ii. Manganese and niobium for alloying
    b. Energy for heating, melting and casting
    c. Coke and coal for reduction and carbon addition to the steel
2. Stainless steel
    a. Metals
        i. Iron metal extracted mainly from iron oxide ore requiring carbon
        ii. Manganese extracted mainly from oxide ore requiring carbon
        iii. Chromium extracted from oxide ore requiring carbon
        iv. Nickel extracted mainly from both oxide ores and sulphide ore and subsequent electrolysis
        v. Silica, molybdenum, niobium and cobalt for specialty steel

The annual global production of these categories is summarized in Table 4.1.3.1.

Table 4.1.3.1. Approximate amounts of iron and steel produced and supplied in overview.

Item	Flow of material, million ton per year		
	Primary extraction	Recycling	Supply
Iron as metal	1,500	700	2,200
Cast iron	70	30	100
Carbon steel	1,400	650	2,050
Stainless steels	30	20	50
Sum	1,500	700	2,200

4.1.3.2 Scope and objectives

The scope of this study is to create an integrated simulation tool for the global iron cycle, based on best estimates on available extractable iron amounts and a systems analysis of the systems dynamics of the global iron system, but focused on the additional requirements applicable to steel. The model has the objective to assess the supply of the metals iron, manganese, chromium and nickel, and the use of these to produce iron for casting, regular steel and stainless steel. The STEEL model is incorporated in the WORLD model (Sverdrup et al. 2012), a part of a new global resource, demography and economic model. The STEEL model should make production estimates and market price for iron, stainless steel, and cast iron, demanding that we can model manganese, chromium and nickel for those aspects as well (Sverdrup et al. 2015).

4.1.3.3 Earlier modelling work of iron

Yellishetty et al. (2010) used statistical correlations and they used it for forward extrapolations. The model is statistically based and does not have any feedbacks. Pauliuk et al., (2012, 2013a,b) and Wang et al., (2007) developed a stock and flow model for analysing the global steel market, with emphasis on aspects of recycling, using flow charts and material flow analysis (MFA). Daigo et al., (2010, 2014) did a material flow analysis of chromium and nickel for steel manufacture and zinc used for steel. Moynihan and Allwood (2012) did material flow analysis for steel in the construction sector. Their approach has been on a country-by-country basis, or sector-by-sector, documenting all flows and making mass balances for the global system. Hatayama et al., (2010) developed a stock and flow model, using top-down mass balances that are rolled forward, year-by-year. These models are varieties material flow analysis models operated from spread-sheets, where the feedbacks are either entered manually or absent (see Pauliuk et al 2015, Nakamura et al. 2007, Giljum and Hubacek 2009 for how the models were constructed). Hu et al., (2010) defined a mass balance model to investigate the flow of iron and steel in the Chinese building sector. These are dynamic parallel mass balances, rolled forward in a spreadsheet. However, there are no real feedbacks operated in the model used.

Mohr et al., (2014) and Guirco et al., (2014), also reported by Nickless et al., (2014) estimated of the global iron ore production, based on a variant of the Hubbert's model combined with a mass flow model. They paid extra attention to establish robust estimates of available extractable resources. Roper (2009), Bardi and Lavacchi (2009), Ragnarsdottir et al. (2011) and Sverdrup et al., (2013) applied the Hubbert's model, a purely empirical model. It has certain systemic properties incorporated, but there are no dynamic feedbacks.

Cullen et al., (2012) Pauliuk et al., (2013), Rauch (2009), Rauch and Pacyna (2009) mapped the global steel stocks and flows. These do not constitute dynamic modelling. They are valuable for model testing and validation, but they are not models that can be used for predictions.

We have not found any earlier global systems dynamics model for the global system for iron or stainless steel, involving systems feedbacks from market dynamics, extraction dynamics, market price mechanisms on supply and demand and simultaneously involving the main three alloying metals manganese, chromium and nickel for alloying to stainless steels.

The authors have developed system dynamics studies on copper, silver, aluminium and iron (Sverdrup et al. 2014a,b, Sverdrup et al. 2015a, Sverdrup et al. 2015d) and metals (Mn, Cr, Ni, Mo, Re, Ta, Nb, In, Se, Te, Ga, Ge, Cd, W, Sn, Sb, Bi, Au, Pt, Pd, Rh, Co, Li, REE) in general (Sverdrup et al 2013, 2015c), where the basic principles for modelling the global metal supply, metal extraction and metal reserves depletion were developed. The WORLD6 model has been further developed to incorporate stainless steel production and include the metals manganese, chromium and nickel that are used in alloyed stainless steels. More than 90% of all iron produced is manufactured to steel by alloying it with carbon and silicon, a smaller amount is alloyed with the metals mentioned above to make stainless steel. There have been no prior sustainability concerns over iron, but other metals connected to stainless steel has gotten some attention (Alonso 2007, Eliott 2013). However, carbon steel is not very resistant to corrosion. Iron has impurities of phosphorus, sulphur and nitrogen, and manganese is added to remove these impurities in the manufacturing process. Nickel is expensive, there has been a movement towards lower contents of nickel in some stainless steels. In addition, comes more special alloys for niche applications and adapted to specifications of extreme properties. The most important alloying metals and their ores are:

1. Chromium is the most important component for making stainless steel alloys. It is also used in substantial amounts for chromium plating of metal objects to give them a bright, shiny surface. Louis Nicholas Vauquelin, a French scientist, discovered chromium in 1797. It is derived from chromite rock, the main chromium ore. Chromite rock consists of FeCr2O4 or FeO*Cr2O3 as the main extractable mineral, the chromium content is 47% in the mineral. The high-grade ore has about 55% mineral content of chromite. Bulk ore contains about 26% chromium on a weight basis. 18% of the chromite mineral is iron, thus bulk ore contains about 10% iron. The largest chromium global resources and reserves are found in South Africa, the ores often are poly-metallic, with additional contents of nickel, copper, zinc and sometimes the platinum group metals. Chromium is mainly

produced in South Africa (44%), but also in Kazakstan (16%), India (18%), Zimbabwe (5%), Brazil (2%), Iran (4%) and Finland (4%) significant amounts are made. At present, chromium is a low margin, large volume commodity. Chromium has no substitute in stainless steel, the leading end use, or in superalloys, the major strategic end use. Chromium-containing scrap can substitute for ferrochromium in metallurgical uses.

2. Manganese is mostly used for steel alloys, but approximately 3% of all manganese is used in aluminium alloys at a content of 1%. Manganese is also added to cast iron in about 0.2-1%. Manganese also protects against corrosion. The Swedish scientist Johan Gottlieb Gahn discovered manganese metal in 1784. Manganese is primarily derived from two minerals; pyrolusite (MnO2), a simple oxide found in low concentrations in most soils, and rhodochrosite (MnCO3). 63% of the mineral is manganese, and high grade ore contains 46-63% weight content manganese, low grade is counted as 35-12 % and below 12% is considered as a poor grade. The main South African ores run at 43-38% weight of manganese at present. If even lower grade ores are counted in down to 1%, the technically extractable manganese reserves would be very large. The extraction of lower grades would require a significantly higher price and more energy than what is the case today. Manganese is mined in South Africa (31%), Gabon (21%) and Australia (13%), significant amounts also come from West Africa. 80% of the worlds known resources appear to be in South Africa, their total ore resources were estimated to be perhaps as much as 15 billion ton in 2011. How much of this that is really recoverable, remains unspecified. Assuming it all to be high grade at 46% would probably overstate the actual resource. Assuming 1/3 at 46%, 1/3 at 23% and 1/3 at 13% would indicate a resource of 4.2 billion ton of manganese. Significant resources are found in Ukraine (10%), Australia, India, China, Gabon and Brazil. There is no substitute for manganese in steelmaking. Manganese makes steel harder (0-4% in carbon steel or 8-12% in stainless steel). Manganese is added to aluminium to make it more resistant to corrosion (0.5-1.5%, consuming about 0.4 million ton of manganese per year). Vanadium and molybdenum may do part of the function played by manganese in alloys and in the manufacturing process, but vanadium and molybdenum production rates are far too small to be realistic alternatives. Manganese is present in manganese nodules in the deep ocean floor (Some estimate this to be as much as 500 billion ton of manganese, but only 3-4 billion may be realistically harvestable). So far none have been successful in developing a feasible way to bring them up for utilization, the attempts were abandoned in the late 1970'ies. Some smaller amounts appear to be technically possible to harvest, but only at significantly higher manganese prices than today. At present, manganese is a low margin, large volume commodity.

3. Nickel was discovered in 1751 by the Swedish scientist Axel Fredrik Cronstedt. Nickel is an important component in high quality stainless steel (46% of all nickel goes to this) is used nonferrous alloys and super-alloys (34% for this purpose), electroplating (14% for this purpose), and 6% is used for other uses. It is important for stainless and for nickel plating. No real replacement or substitute exist, but chromium may do part of the job (Alonso et al., 2007). The biggest producers are Phillipines, Indonesia, Russia, United States, Australia and Canada, but also many other countries also produce significant amounts of nickel. Nickel is derived from lateritic ores that come from surficial weathered rinds formed on ultramafic rocks. The main minerals are limonite (Fe,Ni)O(OH) and (Ni,Mg)3Si2O5(OH)5. These make up 73% of the worlds nickel reserves. The second type of nickel ore are the magmatic and hydrothermally affected sulphide deposits, these make up about 30% of all reserves. The main mineral is pentlandite (Ni,Fe)9S8. Presently mined ores have an ore grade of 2.5-1.5% measured as nickel content, but the ore grade is declining in several large mines. The extraction of these ores has been associated with significant sulphur dioxide pollution to the environment. The Russian nickel smelters are large sulphur emitters (Around Noril'sk in Siberia and at the nickel mines in the Kola Penninsula in Northwestern Russia. Australia and New Caledonia account for 45% of the global nickel extractable amount at present. The amount nickel in ore above 1% ore content is estimated at about 130 million ton, of these about 75 million ton are known and verified. Counting in extractable amount of nickel down to 0.2% content would possibly enlarge the total resource to about 300 million ton of nickel, these would partially

become extractable reserves at a nickel price 6-10 times the present. Present estimates of extractable amounts of nickel count deposits down to 2 km mining depth, however, new South African technology may allow mining of nickel down to about 3.5-4 km. The heat increases with increasing mining depth and mining below 4 km depth would demand a very high nickel price to defend the high extraction costs. At present, 4 km mining depth is considered as the technical limit. This increase in mining depth would potentially increase the global extractable amounts to 180-200 million ton of nickel. Significant amounts of extractable platinum group metals are associated with nickel deposits.

4. Iron has been known since prehistory. The etymology of the word "iron", eisen", "iarn", in Indoeuropean languages is believed to be the same as the root word for star. This is suggesting that the first iron known, was meteoritic iron, "fallen stars". The Hittite civilization was the first to make iron in significant amounts and sufficient quality to weapons around 1,700 BC in Anatolia (Modern Turkey). Iron is used for cast iron, carbon steels and alloyed steels. Iron is the metal that is produced in the largest amount of all metals. Iron is the most important of all metals for modern civilization, and no scarcity in iron can be tolerated without large consequences.

Table 4.1.3.2 shows important steel-associated metals and what they are used for. Specialty alloys are made using metals like molybdenum, cobalt, vanadium or niobium.

Table 4.1.3.2. Important stainless steel-associated metals and what they are used for

Metal	Use in steelmaking, % of total metal flow	Non-ferrous alloys, super-alloys, % of total metal flow	Plating, % of total metal flow	Chemicals and catalysts, % of total metal flow	Other uses, % of total metal flow
Manganese	95	-	-	-	5
Chromium	80	-	8	5	7
Nickel	46	34	14	-	6
Molybdenum	82	4	-	14	-
Niobium	90	9	-	-	1
Vanadium	85	10	-	-	5
Cobalt	8	80	2	5	5

If stainless steel is counted as a separate metal for society, then it is the fourth largest (synthetic) metal in supplied weight, after simple carbon steel, cast iron and aluminium. In the 1890's, the German scientist Hans Goldschmidt invented the alumina-thermic process for producing stainless steel on an industrial scale. Several researchers, in particular the Frenchman Guillet developed a number of alloys from 1904 to 1911, laying the foundation for modern stainless steel. The German industrialist Alfred Krupp (1812-1887) invented industrial production on a large scale of modern steel, and took part in the invention of new types of alloyed stainless steel, alloyed into toughness, hardness, heat tolerance and finally stainlessness. Their first stainless patented alloy (1912) was called "Nirosta", which translates as "never-rust". His factories became key players in two World Wars and made the best material for modern warfare hardware, such as modern artillery, battleships, tanks, submarines and other armaments beside many peace-time uses. Carbon steel is the most common form of steel, it typically has from 0.3% to 2% carbon, sometimes small amounts of silicon. However, carbon steel is not very resistant to corrosion. Iron has impurities of phosphorus, sulphur and nitrogen, and manganese is added to remove these impurities in the manufacturing process. For stainless steel, it is alloyed with a range of other metals to increase the resistance to corrosion.

For making high quality carbon steel and stainless steel, four metals are essential, assisted by specialty metals for special properties:

- Iron for bulk of the material
- Chromium for corrosion resistance
- Manganese for removing impurities and gain strength and workability
- Nickel for corrosion resistance, temperature resistance and hardness
- Molybdenum, cobalt, vanadium and niobium for strength, hardness and temperature resistance.

For stainless steels, metals like vanadium (occurs as a contaminant in almost all iron ore) is used for toughness and strength, tungsten, tantalum and niobium for extra hardness and high temperature resistance, cobalt for corrosion prevention. World production of stainless steel was about 46 million ton in 2017. This typically consists of 5-12% manganese, 10-18% chromium, 3-5% nickel and 0.1% molybdenum on the average as seen from the input to the industry. There are different stainless steels of different composition in the market today:

1. Austenitic steel is based on more than 16% chromium, 10-12% manganese and about 8% nickel. This is 70% of all stainless steel, in 2014 about 29 million ton, demanding 2.2 million ton of nickel per year (total production is 2.6 million ton of nickel per year in 2016), 3.5 million ton of manganese per year (Total production is 16 million ton of manganese per year) and 5 million ton chromium per year (Total production is 8 million ton of chromium per year in 2016).
2. Ferritic steel has the composition 11-27% chromium, on the average 18%, about 1-4% molybdenum, on the average 2%, and 0-2% nickel, on the average the content is 0.5% nickel.
3. Marstenitic steel has 12-14% chromium, 0.3% molybdenum and about 0.5% nickel.
4. Duplex steel is corrosion resistant, but has other mechanical qualities; 22% chromium, 5.5% nickel and 2% molybdenum.

4.1.3.4 Estimation of extractable reserves and resources

In our work, the US Geological Survey is an important source of data. But corporate annual reports and other literature sources are significant for getting reasonable estimates. We work with all technically extractable ore content (below referred to as resources) that is classified according to ore grade and extraction costs in our models. The result is that that we will have ever rising cost as ore quality and extraction yields go down (Tilton 2002, 2007, 2009, 2012, Neumayer 2000, Sweeneye 1992, Mudd 2009, Keseler et al., 2012). Data was gathered from a number of sources and earlier studies where it is readily available in open sources; Allen and Behamanesh (1994), Bardi (2013), Chen and Graedel (2012), Cullen et al. (2012), Crowson (2011), Daigo et al., (2010), Gordon et al. (2006), Gordon (1996), Hu et al. (2010), Johnson et al. (2007), MinEx Consultants (2010), Nuss et al. (2014), Pauliuk et al. (2013), Radetzki (2008), Rauch and Graedel (2007), Rauch and Pacyna (2009), Rauch (2009), Raimanaidou and Wells (2014), Stanway (2014), Turner (1990), UNEP (2011a,b, 2013), USGS (2005, 2007, 2008, 2013), United Nations (2003), Wang et al. (2007), World Steel Association (2012, 2014a,b,c,d), Wübbeke and Heroth (2014). Input for the global population to the demand calculation was derived using the FoF-model (Ragnarsdottir et al. 2011, Sverdrup and Ragnarsdottir 2011).

Iron resources:

Table 4.1.3.3 shows the reserves and resources for iron, broken down on countries with the largest endowments.

Table 4.1.3.3. Distribution or iron ore expressed as iron content distributed among known and hidden amounts in ton iron for 2012. All amounts in metric ton iron contained.

Country	Known extractable amounts		Hidden amounts	Sum
	Iron content, ore %	Million ton Fe	Million ton Fe	Million ton Fe
Brazil	52	16,000	51,000	67,000
Russia	45	14,000	45,000	59,000
Canada	38	2,300	42,000	46,300
Australia	48	17,000	24,000	41,000
United States	30	2,100	26,000	28,100
China	31	7,200	10,000	17,200
Zimbabwe	44	400	15,000	15,400
India	64	5,200	8,000	13,200
Venezuela	60	2,400	8,000	10,400
Bolivia	44	2,200	6,600	8,800
Kazakstan	44	2,000	6,300	8,300
Sweden	63	2,200	5,600	7,800
Ukraine	35	2,300	5,000	7,300
Iran	56	1,400	4,000	5,400
Rest of the world	49	4,300	3,400	7,700
Sum	50	81,000	259,000	340,000

Table 4.1.3.4 shows different estimates of the total iron extractable amounts as a basis for potential mining of iron as well as the regionalized iron production and stocks in society, matched to fit the global data for 2012. On the average, the recycled part of the flow is about 5% and on the average about 1% of the stocks recycled per year.

Table 4.1.3.4. Regionalized iron production and stocks in society, matched to fit the global data for 2012.

Grouping	Production million ton/yr	Recycling of production, million ton/yr	R/P %	Stock-in-use, Million ton	Recycling from stock, million ton/yr	R/S %	Supply to society, ton/yr
China	632	35	5.5	4,600	50	1.08	782
Americas	173	8	4.6	5,700	52	0.91	235
Russia, Belarus Ukraina, Kazakstan	119	4	3.4	700	6.7	0.96	126
India, Pakistan, Bangladesh	58	1.7	3.0	1,500	11.3	0.75	69
Southeastern Asia, Oceania	213	6	2.8	2,000	20	1	233
Europe	200	20.5	10.3	5,600	56	1	256
Middle East, West Asia, Africa	67	1.4	2.0	1,400	7	0.5	73
Sum	1,462	76.6	5.2	21,500	203.0	0.94	1,665

Table 4.1.3.5 shows estimates of iron reserves and resources over time by different authors.

Table 4.1.3.5. Different estimates of the total iron extractable amounts as a basis for potential mining of iron. Billion ton of extractable iron.

Source of Estimate	Known reserves	Hidden resources	Dug up to time	From time to 2014	URR
Geological mapping, (Stockwell 1999)	50	270	22	13	305
Geology based (DNP/DM 2000)	50	256	40	12	308
Verhulst's linearization (Roper 2009)	50	300	50	7	357
Rauch and Pacyna (2009)	35	200	30	7	237
USGS mineral resources program (2011)	81	260	40	6	306
Verhulst's linearization (Roper 2014)	-	234	50	-	284
Hubbert's model (Sverdrup et al. 2011)	60	280	38	3	321
Mohr et al., (2014)		195	50	2	247
Guirco et al., (2014)		200	50	2	252
Sverdrup et al., 2016	90	200	50	1	341
This study, reassessment	90	255	55	-	400

Table 4.1.3.6 shows how in 2008, approximately 928 million ton of iron was produced from different ore grades. This has been distributed over different ore grades.

Table 4.1.3.6. Distribution of different production amounts over different ore grades. (approximately 928 million ton of iron production in 2008).

Ore grade, % weight	70-60%	60-50%	50-40%	40-30%	30-20%	20-10%
Million ton per year	500	300	100	75	33	10

Figure 4.1.3.2a shows the evolution of the total iron resource estimates with time. From the plot we estimate that the real extractable iron resource is about 500 billion ton of iron.

Nickel resources:

Table 4.1.3.7 shows estimated global reserves and resources for nickel in million ton of metal content, listed by country, using data compiled from detailed studies.

Table 4.1.3.7. Estimated global reserves (Known) and resources (Hidden) for nickel in million ton of nickel metal content, listed by country, using data compiled from detailed studies.

Region	Known reserves	Hidden resources	Total resource
Australia	19.0	40	59
Indonesia	22.0	52	74
Brazil	9.1	30	39
Russia	7.9	20	28
Cuba	5.5	7	13
South Africa	3.7	10	14
Phillipines	3.1	12	15
Canada	3.0	45	48

China	3.0	5	8
Finland	3.0	7	10
Sweden	2.0	7	9
Madagascar	1.6	12	14
Colombia	1.1	4	5
Greenland	0.2	12	12
Norway	0.2	2	2
Other	6.5	25	32
Sum	81.0	285	366
Seafloor nodules and cobalt crusts			150-274
Sum of all			516-640

Table 4.1.3.8 shows the ultimately recoverable amount nickel, starting from 1900. This was used as the starting point for the model. The nickel deposits are stratified into rich, high, low and ultralow resources. The high, low and ultralow resources are each divided into 0-1 km, 1-2 km, 2-3 km and 3-4 km mining depth as shown below.

Table 4.1.3.8. Estimated ultimately recoverable amount nickel, starting from 1900. This was used as the starting point for the model. The nickel deposits are stratified into rich, high, low and ultralow resources. The high, low and ultralow resources are each divided into 0-1 km, 1-2 km, 2-3 km and 3-4 km mining depth as shown below.

Mining depth	Recoverable metal amount metal in million ton of nickel per ore class				
	Rich	High	Low	Ultralow	Sum
	55-11%	11-2%	2-0.5%	0.5-0.1%	
Hidden 0-1 km	6	40	60	90	166
Hidden 1-2 km	3	30	50	50	133
Hidden 2-3 km	2	20	40	25	87
Hidden 3-4 km	1	10	20	15	46
Sum 0-4 km	12	100	170	180	432
Mining cost, $ per ton	40	120	400	1,750	-

Table 4.1.3.9 shows estimates of the total nickel extractable amounts, estimated at different points in time. This data was used to make the plot in Figure 4.1.3.2b. Figure 4.1.3.2b shows the evolution of the total nickel resource estimates with time. From the plot, we estimate that the real extractable nickel resource is about 500 million ton of nickel. Mudd (2009, 2010b, 2012a,b), Mohr et al., (2014) and Nickless et al., (2014) were used for estimating the nickel reserves. The nickel resources were estimated down to an assumed maximum mining depth of 4 km. The nickel deposits are stratified into rich, high, low and ultralow resources. The resources are divided between the rich-, high-, low- and ultralow grade. Muss and Jowitt (2014) reports resources with 118 million ton in sulphides and 181 million ton in laterites.

Table 4.1.3.9. Different estimates of the total nickel extractable amounts. Million ton of nickel. 1: ore with >1%. 2: ores with less than 1%.

Source of Estimate	Known reserves	Resources	Dug up to time	From time to 2017	URR
	Million ton extractable iron content				
USGS mineral resources (2001)	61	140	20	10	231
Mudd 2009	60	202	25	14	301
Roper 2009	50	200	25	14	289
UNEP (2011)	180		28	12	220
USGS (2012)	75	130 100	30	8	343
Muss and Jowitt 2014	299		35	6	340
Sverdrup et al., 2016	65	230	40	2	352
USGS mineral resources (2017)	78	1301 1302	42	-	380
Sverdrup et al., (2017)	81	285	42	-	408

Chromium resources:

Table 4.1.3.10 shows estimated global reserves and resources for chromite ore, Cr2O3, listed by country, using data compiled from detailed resource studies. 1 ton chromium metal is contained in 3.8 ton of chromite ore (Pariser (2013, USGS 2007). The South African reserves for chromite have different estimates for the total chromite reserves and resources depending on depth of extraction and where the ore quality cut-off is set.

Table 4.1.3.10. Estimated global reserves (Known) and resources (Hidden) for chromite ore, in million ton, listed by country, using data compiled from detailed resource studies (Pariser 2013, USGS 2007). Guesses and expert estimates are in brackets (..) Million ton of chromite. The total estimates for South Africa vary a lot, based on differences in assumptions made in the estimation.

Region	Known extractable chromite reserves	Hidden extractable chromite resources	Dug up total	Total chromite resource
South Africa 1984 **South Africa 2017**	200 3,100	3,100 5,500	87 30	3,387 8,630
Zimbabwe	140	1,000	37	1,177
Kazakstan	320	320	95	735
Russia	(30)	(300)	30	(360)
Canada	49	242	9	291
Turkey	(40)	?	31	220
Phillipines	(5)	?	22	(220)
Albania	0	(200)	18	(200)
Finland	9	?	7	120
India	25	70	21	116
Brazil	5	(25)	7	30
China	(5)	(20)	11	36
Other	(50)	(550)	21	621
Sum	878-3,778	5,807-8,207	374	7,050-12,359

Table 4.1.3.11 shows estimated global reserves and resources for recalculated for chromium over time. The evolution of the estimates over time are shown in Figure 4.1.3.2c. From the plot, we estimate that the real extractable chromium resource is about 3,400 million ton of chromium.

Table 4.1.3.11. Estimated global reserves (Known) and resources (Hidden) for chromium (Pariser 2013, USGS 2007). Million ton of chromium metal. The evolution of the estimates over time are shown in Figure 4.1.3.3b.

Region	Known extractable reserves	Hidden, extractable resources	Total resource	URR
Expert guess for 1900	50	400	450	1,170
USGS, de Young et al., 1984	140	1,300	1,440	1,935
USGS 2007	210	2,915	3,125	3,275
Pariser 2013	250	2,146	2,396	2,441
USGS 2013	210	2,915	3,125	3,170
USGS 2016	126	3,125	3,251	3,270
This study 2017	828	2,142	2,970	3,045

Manganese resources:

Table 4.1.3.12 shows estimated global reserves and resources for manganese, listed by country, using data compiled from detailed resource studies, using the latest available numbers. For South Africa, the estimates for the hidden resources are very different and represents probably a good deal of guessing. If we assume the total ore resource of South Africa to be 15 billion ton of manganese ore, then our estimates imply an average ore grade of 18% to 38% manganese content. The South African reserves for manganese have different estimates for the total manganese reserves and resources depending on depth of extraction and where the ore quality cut-off is set.

Table 4.1.3.12. Estimated global reserves (Known) and resources (Hidden) for manganese in million ton, listed by country, using data compiled from detailed resource studies, using the latest available numbers.

Region	Known extractable reserves	Hidden extractable resources	Total resource	Annual production, million ton per year
Australia	187	514	701	2.5
Mexico	5	200	205	0.2
Russia	30	250	280	0.7
Kazakstan	5	500	505	0.2
South Africa	200	2,500-5,500	2,700-5,700	5.0
Gabon	27	250	277	2.0
Ghana	12	150	162	0.5
Ivory Coast	7	100	107	0.5
Ukraina	140	1,000-1,500	700	0.3
China	44	655	699	3.0
Malaysia	7	14	21	0.2
India	52	380	432	1.0
Brazil	116	450	566	1.7
Burma	11	100	111	0.1

Other	100	500	600	0.7
Sum on land	840	7,063-11,063	7,903-11,903	16.0
Seafloor	300	2,700	3,000	0
Sum	1,230	9,763-13,763	10,903-16,903	18.6

Table 4.1.3.13 shows estimated global reserves and resources in different scientific studies and survey reports. Everything has been converted to million ton of manganese metal content. The evolution of the estimates over time are shown in Figure 4.1.3.2d. From the plot, we estimate that the real extractable manganese resource is somewhere between 8 billion ton and 12 billion ton of manganese. In the simulation, we have used 9 billion ton.

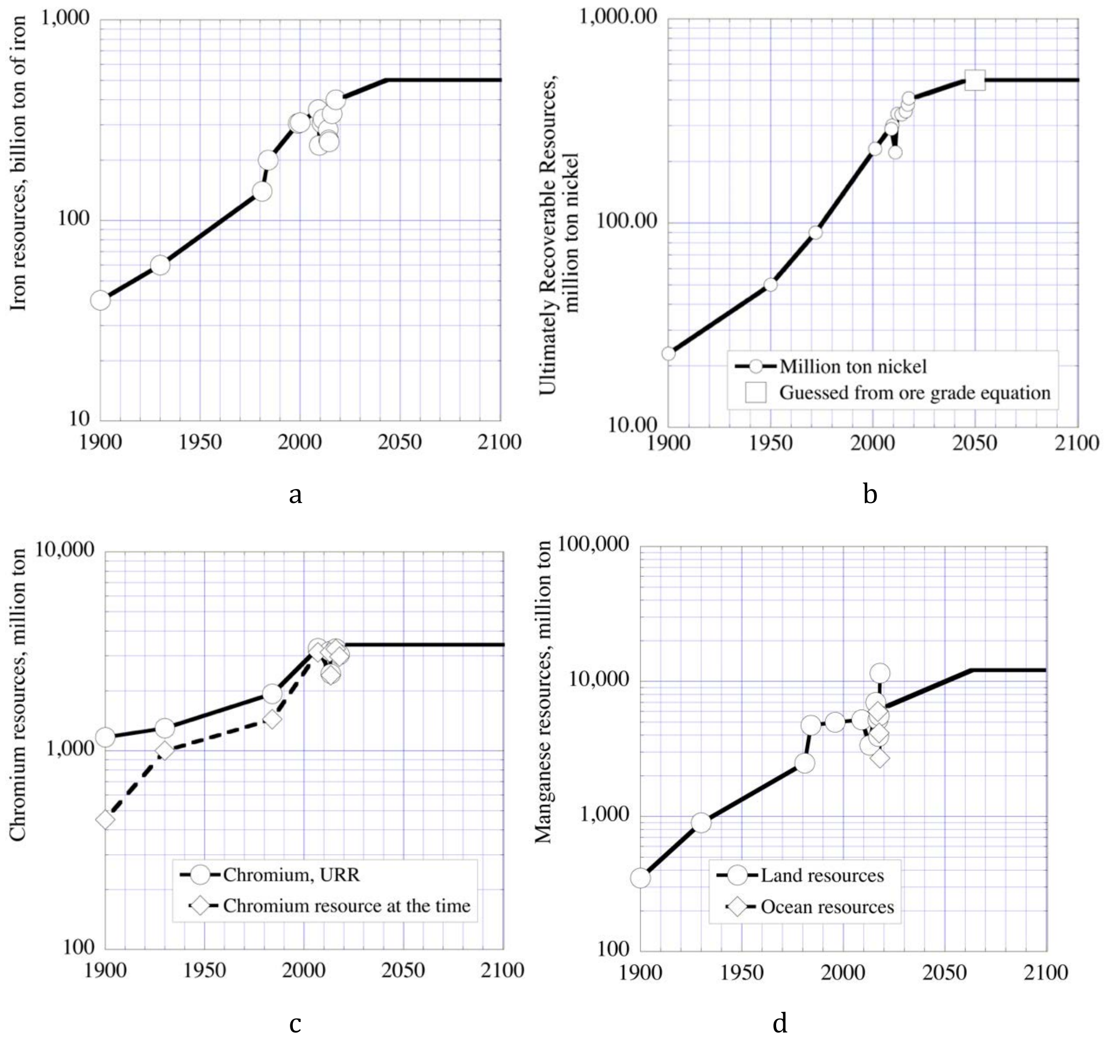
Table 4.1.3.13. Estimated global reserves (Known) and resources (Hidden) in different scientific studies and survey reports. Everything has been converted to million ton of manganese metal content. The evolution of the estimates over time are shown in Figure 4.1.3.3a.

Region	Known extractable reserves	Hidden extractable resources	Total resource	Dug up to present	URR million ton	Seafloor nodules
National Materials Advisory Board 1981	491	1,995	2,486	92	2,578	5,000
USGS, de Young et al., 1984	210	3,120	3,330	114	4,740	-
USGS 1996	-	-	-	-	5,000	-
USGS 2009	-	-	-	-	5,200	-
USGS 2013	210	2,900	3,150	220	3,370	-
Encylopedia Iranica 2016	-	-	-	-	7,000	-
World Ocean Review 2017	630	4,310	5,000	220	5,200	5,990
USGS 2017	690	3,125	3,715	220	3,915	-
Sverdrup et al., 2017	-	-	-	-	5,600	4,160
Sverdrup et al. (here)	1,230	6,673	7,903-11,490	220	8,123-12,710	3,000

4.1.3.5 Setting the extractable amounts

Figure 4.1.3.2 shows the development of estimated ultimately recoverable (a) manganese, (b) chromium, (c) iron and (d) nickel resources with time from data found in the literature (The data has been listed in Tables 4.1.3.4, 4.1.3.8, 4.1.3.11 and 4.1.3.13).

Figure 4.1.3.2. The development of estimated ultimately recoverable (a) manganese, (b) chromium, (c) iron and (d) nickel resources with time from data found in the literature (Tables 4.1.3.4 and 4.1.3.11). We estimate the final URR based on these curves, for nickel to be about 500 million ton, for chromium 3,500 million ton, for manganese, 9 billion ton and iron to be about 500 billion ton.



Source: Own Figure, Lund University/University of Iceland

The prediction of the final URR is using relationships discovered in the study on copper, zinc and lead by Sverdrup et al. (2017). We estimate the final URR based on these curves, for nickel to be about 500 million ton, for chromium to be about 3,400 million ton. For manganese, we estimate it to be maximum 9 billion ton. We estimate the total resource for iron to be about 500 billion ton.

Table 4.1.3.14 shows the estimated ultimately recoverable reserves (URR), starting from 1900. 55% ore grade implies a range of 65-45% metal weight content (Rich ore grade), 11% ore grade implies a range 35-8% metal weight content (High ore grade), 2% ore grade implies the range 2.5-1.5% metal weight content (Low ore grade), 0.5% ore grade implies 1-0.2% metal weight content (Ultralow ore grade). The distribution among different qualities and the production price connected to ore grade play an important role, and with this data, extra attention was paid to checking data quality properly. We assess the ultimately recoverable reserves (URR), the presently extractable amounts, potential re-sources that may become extractable amounts, division between presently known extractable amounts and estimated as hidden extractable amounts, all taken from the literature, from scientific articles and from corporate information. The estimates of URR vary significantly with time, and this is depending on many factors. We have estimated URR as extractable amounts plus mined to present. Global population is together with affluence an important driver of iron demand in the model. Stainless steel has chromium, manganese, nickel as alloy metals (Gordon 1996, EuroInox 2014, International Steel Development Forum 2013, World Steel Association 1999, 2012, 2014a).

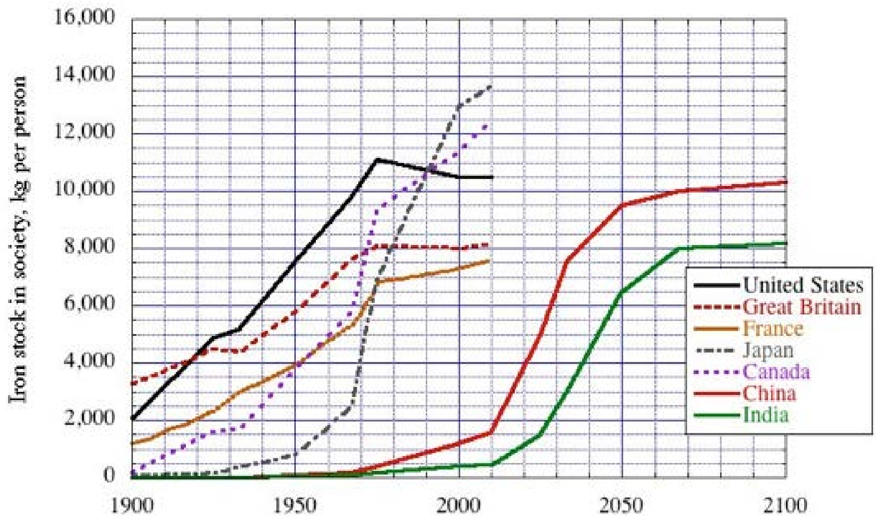
Table 4.1.3.14. Overview of important alloying metals for stainless, heat resistant or high hardness steels. Best estimate according to the compilations made by the authors.

Metal	Million ton metal						
	Known extractable amount metal	Additional amount after price increase	Extractable total amount metal	Extracted before present	URR	Seafloor nodules	Resources of unknown extractability
Fe	56,000	409,000	465,000	45,000	500,000	50,000	400,000
Mn	1,480	6,756	8,236	764	9,000	500,000	16,000
Cr	440	2,560	3,400	475	3,400	10,000	3,000
Ni	80	390	470	30	500	495	100

Harmsen et al., (2013) and Hencksens et al., (2014) give total extractable resources as large as here, but it remains unclear from their publications how those number actually were derived. For nickel their estimate is substantially larger. Henckens et al., (2014) estimates the total original nickel resource (URR) in the first 5 km of the Earths crust, using all land area above sea, to be 1,800 million ton, the remaining amount today to be 1,700 million ton, supposedly to have been reported by UNEP in 2011 (Graedel et al., 2011). However, this must be a mistake, as UNEP reported URR for nickel to be 180 million ton. They must have made a mistake and multiplied the resource by a factor of 10! For chromium, Henckens et al., (2014) reports an estimate of URR of 3,300 million ton, consistent with our estimates. For manganese, Henckens et al., (2014) report an estimate of URR at 24,000 million ton, supposed to come from the UNEP 2011 report. I cannot find that number in that report, so something is very wrong with that estimate. For iron, Henckens et al., (2014) report an estimate of URR at 1,400,000 million ton. This was estimated by guessing a depth and guessing a global average content and conjuring up the number given above. The number is probably a factor of 1,000 off the mark. The manganese and iron amounts would include all known geological potentials, deep sea manganese nodules as well as the minute amounts found in the worlds soils. This is not a serious estimate. In the same article, they also report an estimate of URR of 20 million ton for silver, something we do know with certainty to be an estimate with no basis in reality (Sverdrup et al., 2014a). The costs of extraction of the lower grades and the very deep potential deposits and the technical unavailability of these resources seems to have been ignored in the studies of Harmsen et al., (2013) and Henckens et al., (2014). The UNEP 2011 report states in the conclusions that the estimates in their report are too unreliable for making any estimates of extractable amounts and cannot really be used for sustainability estimates, as the extractability question could not be properly addressed (Graedel et al., 2011).

Figure 4.1.3.3 shows steel stocks in the past and the future expressed as ton iron metal per person is shown in the diagram (World Steel Association 2013, Graedel and Erdmann 2012, Rauch 2009). Numbers in the diagram are used to set the demand per capita, when we use the global population to drive total demand. The available data was closely inspected for inconsistencies and averages and adjustments were made when the available sources for input data were not internally consistent.

Figure 4.1.3.3. Steel stocks in use in society the past and the future expressed as ton iron metal per person is shown in the diagram.



Source: The figure was plotted from data found in World Steel Association (2013), Graedel and Erdmann (2012), Yellishetty et al., (2012) and Rauch (2009).

Energy use

An approximate estimation of the energy used for mining, benefication, smelting and processing metals before they are ready for sale is made. The data used has been shown in Table 4.1.3.15. Total CO2 emissions of a typical integrated steel plant are equal to 1.8 ton CO2 per ton rolled steel coil, of which 1.7 ton CO2 per ton rolled coil is associated with coal use, and the other 0.1 ton CO2 per ton rolled coil is related to lime use (Birat 2010, Dahmus and Gutowski 2014). Energetics, Inc. (2004) gives the range of energy use for coke making as 5.5–6.5 GJ per ton coke.

Table 4.1.3.15. Energy required to produce metal from different substrates, a comparison between different metals and pathways from raw material to finished metal. We have also added the average CO2 contribution from each type of pathway

Metal base	Source of product	Energy use MJ per kg	Steel model parameter	CO2 per kg
Iron	Scrap iron	6-15	10	1
	iron ore	20-25	18	2
	bog ore	100-150	120	12
	Ore, rich grade	20-30	20	2
	Ore, high grade	40-60	44	4.4
	Ore, low grade	80-120	88	8.8
	Ore, ultralow grade	100-250	120	12
Steel	Recycled steel scrap	10-25	10	1
	Fresh metals	6-15	10	1
Stainless steel	Recycled stainless steel scrap	10-25	12	0.2-1
	Fresh metals			
Manganese	Recycled manganese scrap	10-25	15	1
	Ore, rich grade	30-60	45	5
	Ore, high grade	60-120	60	10

	Ore, low grade	120-250	120	20
	Ore, ultralow grade	200-600	220	40
Chromium	Recycled chromium metal scrap	10-25	15	1
	Ore, rich	90-200	90	10
	Ore, high	180-270	180	22
	Ore, low	300-670	350	48
	Ore, ultralow	900-1,900	900	110
Nickel	Nickel metal scrap	140-200	120	12
	Sulphide ore, high	230-270	250	25
	Sulphide ore, low	630-670	650	32
	Sulphide ore, ultralow	1,900-2,100	1,900	150
	Oxide ore, high	180-270	180	25
	Oxide ore, low	300-670	350	32
	Oxide ore, ultralow	900-1,900	900	150

Extraction cost basis

Table 4.1.3.16 shows the approximate cost structure of iron ore production, adapted from the BHP-Billiton website (Data from 2011). The labour costs are typical for China, the other costs are the same everywhere in the world. Industrial countries have three-four times higher labour cost. We have used a factor of 3.5 when scaling the costs from Chinese costs to estimated Australian costs.

Table 4.1.3.16. The table shows the approximate cost structure of iron ore production, adapted from the BHP-Billiton website (Data from 2011).

Approximate Chinese iron mining and extraction costs, $ per ton						
Cost items	Rich grade ore 55% content		High grade ore 25% content		Low grade ore 14% content	
	Price	Energy cost	Price	Energy cost	Price	Energy cost
Labour costs	11.1		15.0		30.0	
Energy and consumable resource materials	13.1	13.1	26.0	26.0	52.0	52.0
Financial costs	3.7		3.7		3.7	
Freight and logistics	4.3	2.0	8.6	4.0	17.2	8.0
Taxation	7.0		7.0		7.0	
Exploration	0.6		0.6		0.6	
Sum on land costs	38.8	15.1	60.9	30.0	110.5	60.0
Ship to overseas users, excluding energy use	12.0					
Ship to overseas users, energy input	8.0	8.0	16.0	32.0	32.0	64.0
World market ore cost	58.8	23.1	76.9	62	142.1	124.0
Approximated Australian iron mining and extraction costs, $ per ton						
Cost items	Rich grade ore 55% content		High grade ore 25% content		Low grade ore 14% content	
	Price	Energy cost	Price	Energy cost	Price	Energy cost

Labour costs	39.0		52.0		105.0	
Energy and consumable resource materials	13.1	13.1	26.0	26.0	52.0	52.0
Financial costs	3.7		3.7		3.7	
Freight and logistics	4.3	2.0	8.6	4.0	17.2	8.0
Taxation	7.0		7.0		7.0	
Exploration	0.6		0.6		0.6	
Sum on land costs	56.7	15.1	97.9	30.0	184.5	60.0
Ship to overseas users, excluding energy use	12.0					
Ship to overseas users, energy input	8.0	8.0	16.0	32.0	32.0	64.0
World market ore cost	88.0	23.1	113.9	62	216.5	124.0

4.1.3.6 Model description

For all the metals, mass balance applies:

$$Mined + Recycled = Accumulated\ in\ society + Recycled + Lost\ irreversibly \qquad (4.1.3.1)$$

“Mined” is what we take from the geological reserves, and there is no other net production of new metal into the system. “Accumulated in society” is metal kept in society and not lost, and “Lost irreversibly” is what is lost in such a way that we cannot retrieve it again, it leaves the system. “Recycled” refers to metals that are circulated in the system. Recycled is present on both sides of the equation, increasing the internal flux, while the external net input may remain low if recycled is large. The more we recycle, the less we need to mine to keep the same amount in society. Iron is sold as the physical metal and we ignore the derivatives trade in our economics model. We assume that the present type of society with international trade and general rule of law and order as it is today will persist over the time considered (1900-2400).

The model was developed to estimate supply, extractable amounts stocks, price and stocks, and flow in society in the time interval 1900-2400. The model has the following stocks used to define the coupled non-linear differential equations defined by mass balance:

1. Iron
    a. Mineable stocks;
        i. Known; rich-, high-, low- and ultralow grade ore
        ii. Hidden; rich-, high-, low- and ultralow grade ore
    b. In society, we distinguish;
        i. Iron in smelting
        ii. Trade market
        iii. Stock-in-use
            1. Long term
            2. Intermediate term
            3. Short term
            4. In Stainless steel
        iv. Scrapped
2. Manganese
    a. Mineable stocks;
        i. Known; rich-, high- and low grade ore
        ii. Hidden; rich-, high- and low grade ore

b. In society, we distinguish;
    i. Trade market
    ii. Stock-in-use
        1. In Stainless steel
        2. Other
    iii. Scrapped
3. Chromium
    a. Mineable stocks;
        i. Known; rich-, high- and low grade ore
        ii. Hidden; rich-, high- and low grade ore
    b. In society, we distinguish;
        i. Trade market
        ii. Stock-in-use
            1. as alloy
            2. in plating and other use
            3. In Stainless steel
        iii. Scrapped
4. Nickel
    a. Mineable stocks;
        i. Known rich-, high-, low- and ultralow grade ore
        ii. 0-1 km depth; Hidden high-, low- and ultralow grade ore
        iii. 1-2 km depth; Hidden high-, low- and ultralow grade ore
        iv. 2-3 km depth; Hidden high-, low- and ultralow grade ore
        v. 3-4 km depth; Hidden high-, low- and ultralow grade ore
    b. In society, we distinguish;
        i. Trade market
        ii. Stock-in-use in other use
            1. Long term
            2. Intermediate term
            3. Short term
        iii. Stock-in-use
            1. as plating
            2. other
            3. In stainless steel
        iv. Scrapped
5. Stainless steel
    a. Trade market stock
    b. Stock-in-use
        i. Material in manufacture
        ii. Material in use;
            1. Slow turnover stocks-in-use
            2. Intermediate turnover stocks-in-use
            3. Fast turnover stocks-in-use
            4. Material in recycling processes
    c. Scrap stock

This makes a simple and compact model with 71 coupled differential mass balance equations. In addition to the mass balances comes equations for ore grades, energy use and summing up of cumulative amounts of energy use, waste rock and ore grades. The ore is mined from the grade with the lowest extraction cost. Dependent stocks are cumulative amounts of: mined, amounts rock waste, losses, smelter slag and ore found by prospecting. The mining activity is profit driven, the profit is affected by the mining cost and the market price. A lower ore grade implies that more rock must be moved to mine the iron:

$$r_{Mining} = k_{\ Mining} * m_{Known}^{n} * f(Technology) * g(profit) * p(yield) \qquad (4.1.3.2)$$

where rMining is the rate of mining, k is the rate coefficient and m is the mass of the ore body, and n is the mining order. where f(Technology) is a technology improvement function dependent on time, g(profit) is a feed-back from profits on the mining rate. p(Yield) is a rate adjustment factor to account for differences in extraction yield when the ore grade decreases. These functions are given exogenously to the model. The rate coefficient is modified with ore extraction cost and ore grade. The size of the extractable ore body (Known), mK, is determined by the extractions (rMining) and prospecting (rDiscovery). We get equation (4.1.3.3):

$$\frac{dm_{Known}}{dt} = -r_{mining} + r_{discovery} \qquad (4.1.3.3)$$

The discovery is a function of how much prospecting we do and how much there is left to find. The amount hidden resource (mH) decrease with the rate of discovery (rDiscovery). The rate of discovery is dependent on the amount metal hidden (mHidden) and the prospecting coefficient kProspecting. The prospecting coefficient depend on the amount of effort spent and the technical method used for prospecting. We get equation (4.1.3.4):

$$\frac{dm_{Hidden}}{dt} = -r_{Discovery} = -k_{Prospecting} * m_{Hidden} * f(Technology) * g(profit) \qquad (4.1.3.4)$$

basic mining comes from profits and availability of a mineable resource used in the model. The rate is first order as prospecting is three-dimensional by drilling. The basic driving mechanism of basic mining comes from profits and availability of a mineable resource used in the model. The price is set relative to how much iron or steel there is available in the market. The rate of corrosion of the stock-in-use ad from scrapped metal are defined as:

$$r_{Corrosion} = -k_{Corrosion} * m_{Stock-in-use} \qquad (4.1.3.5)$$

Scrap is both lost physically by dropping it where it cannot be found and by corrosion:

$$r_{Scrap\ loss} = -(k_{Scrap\ loss} + k_{Corrosion}) * m_{Scrap} \qquad (4.1.3.6)$$

The iron, manganese and chromium ore benefication and subsequent smelting yield is defined as:

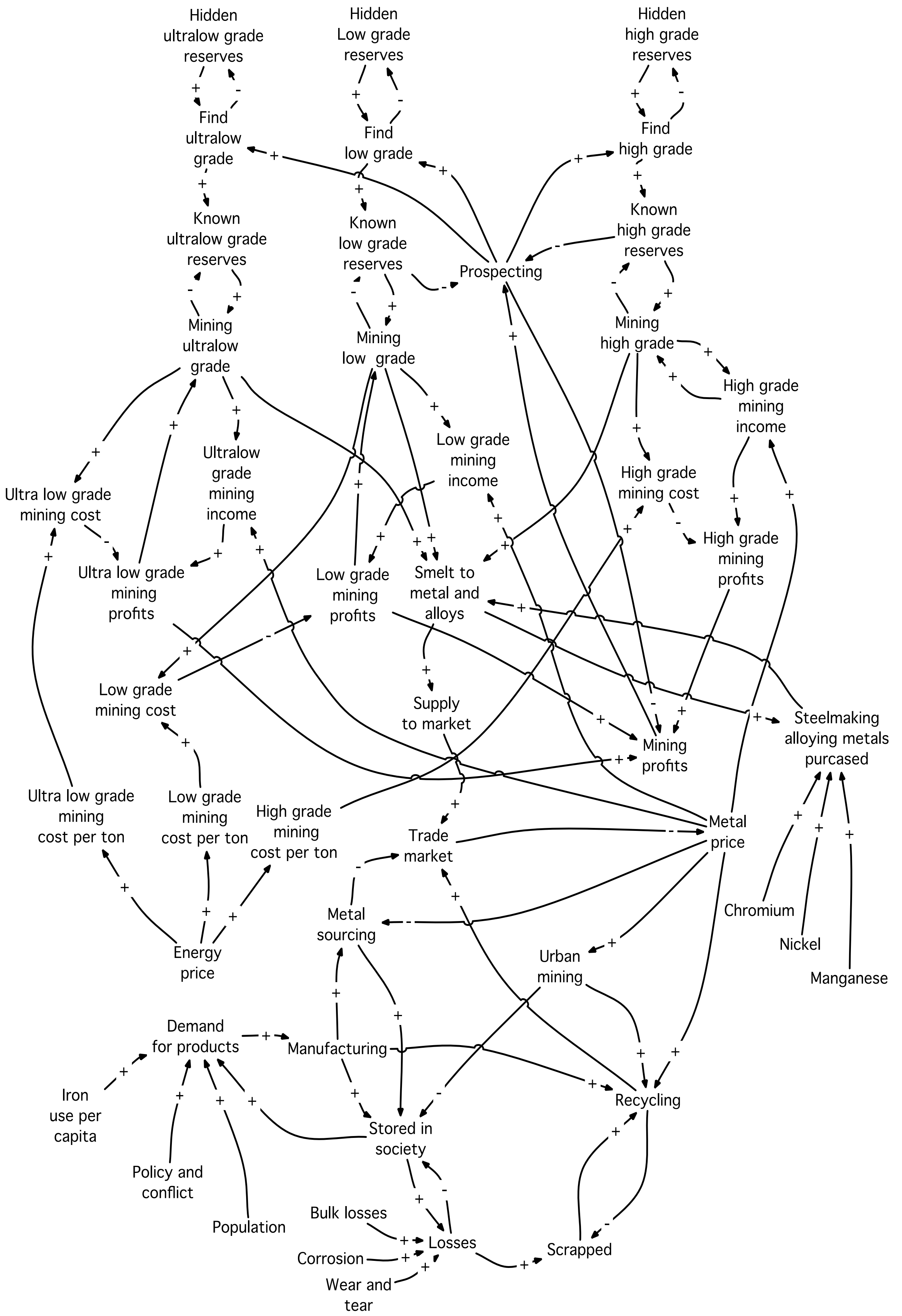
$$r_{smelting\ supply} = k_{Smelting\ yield} * r_{Mining} \qquad (4.1.3.7)$$

For nickel the equation is somewhat different:

$$r_{Nickel} = k_{Ni\,smelting\,yield} * r_{Nickel\,mining} + r_{Ni\,from\,PGM} + r_{Ni\,from\,Cu} \quad (4.1.3.8)$$

The price is set relative to how much iron or steel there is available in the market. The causal loop diagram shown in Figure 4.1.3.4 maps all significant causal connections in the system for the part of the model concerning iron. It is important that the links are true causal links and not just correlations or modelled on chain of events. The causal loop diagram together with the flow chart, defines the model. The rate is first order as prospecting is three-dimensional by drilling. The causal loop diagram shows us the mining operation is driven by operations profit. This profit is driven by iron price and amount extracted, but balanced by the cost of operation. Higher price decrease demand.

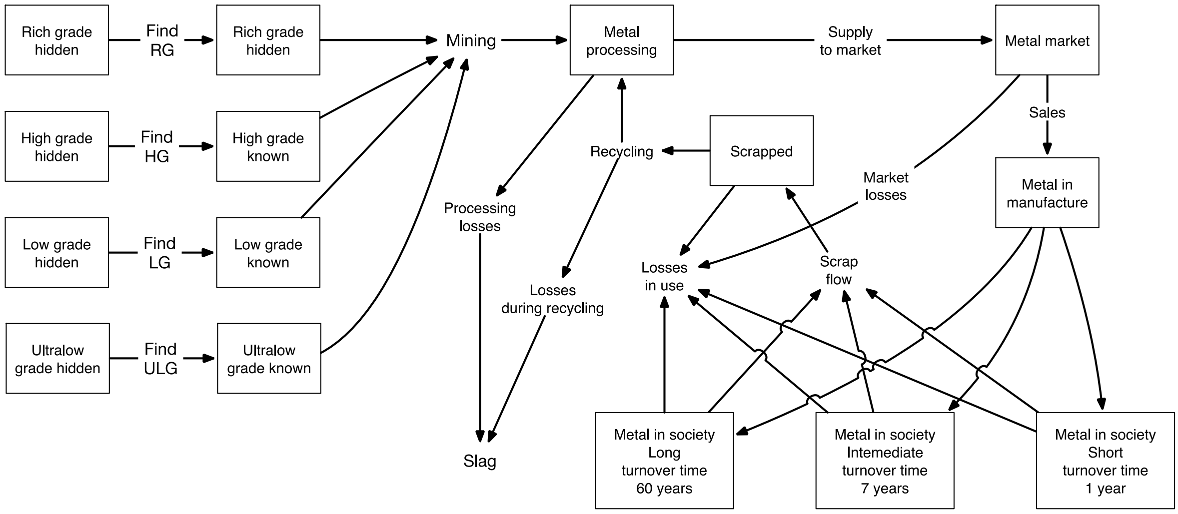
Figure 4.1.3.4. Causal loop diagram for the iron cycle in the model.



Source: Own Figure, Lund University/University of Iceland

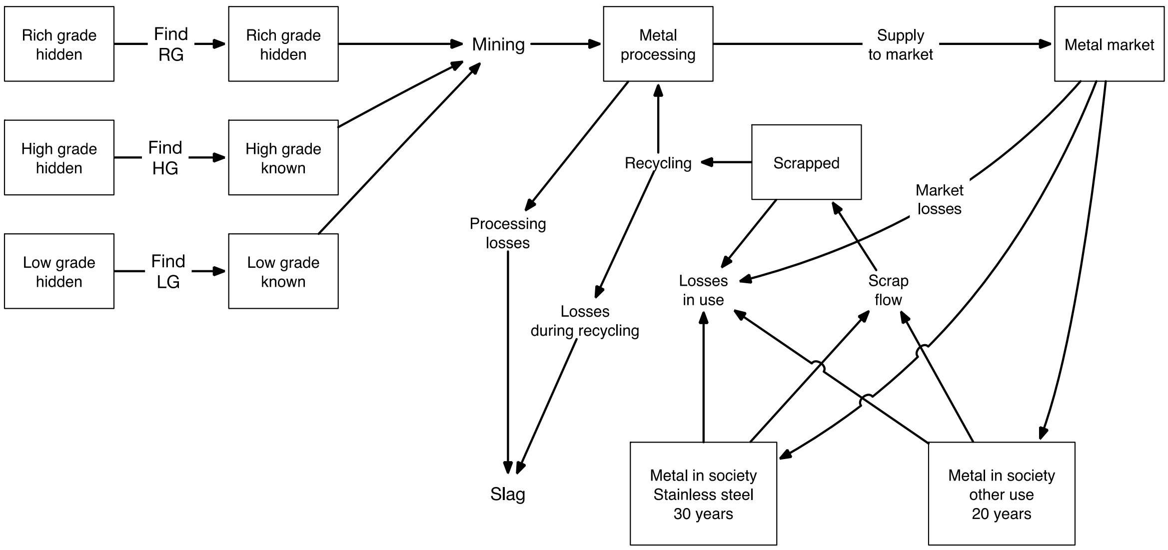
Figures 4.1.3.5-4.1.3.7 show the basic flow chart for iron, manganese and chromium included in the model. Figure 4.1.3.8 shows the more detailed flow chart used for the nickel module in the model. Figure 4.1.3.9 shows the more detailed flow chart used for the stainless steel module in the model. The metals flow from reserves, and when they decline, this may limit stainless steel production. Manganese demand is driven in a large part from both conventional steel making and from stainless steel production. Chromium demand is driven from electroplating, conventional steel making and stainless steel production. Iron demand is driven from by world population and economic development in general.

Figure 4.1.3.5. Basic flow chart for iron included in WORLD6.



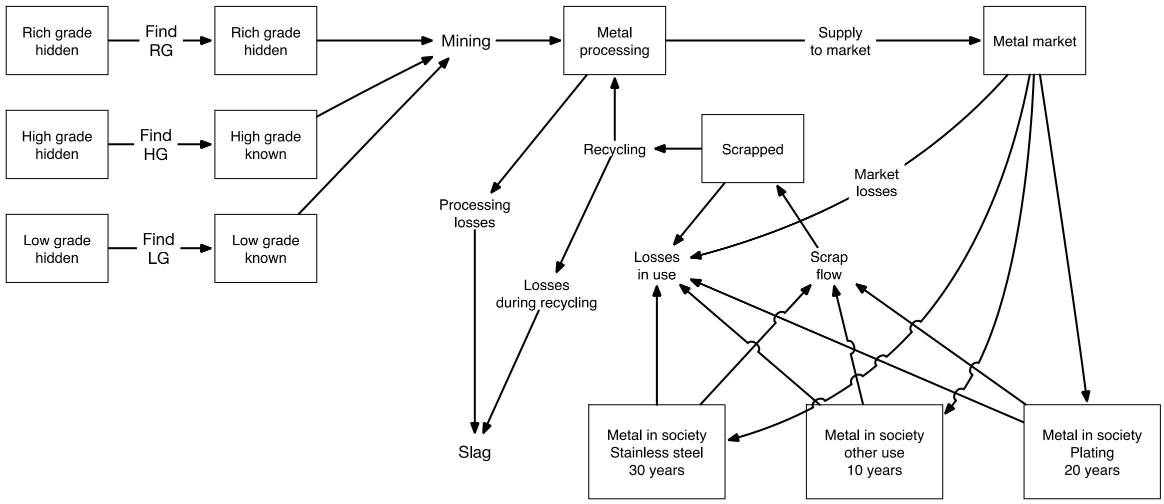
Source: Own Figure, Lund University/University of Iceland

Figure 4.1.3.6. Basic flow chart for manganese included in WORLD6



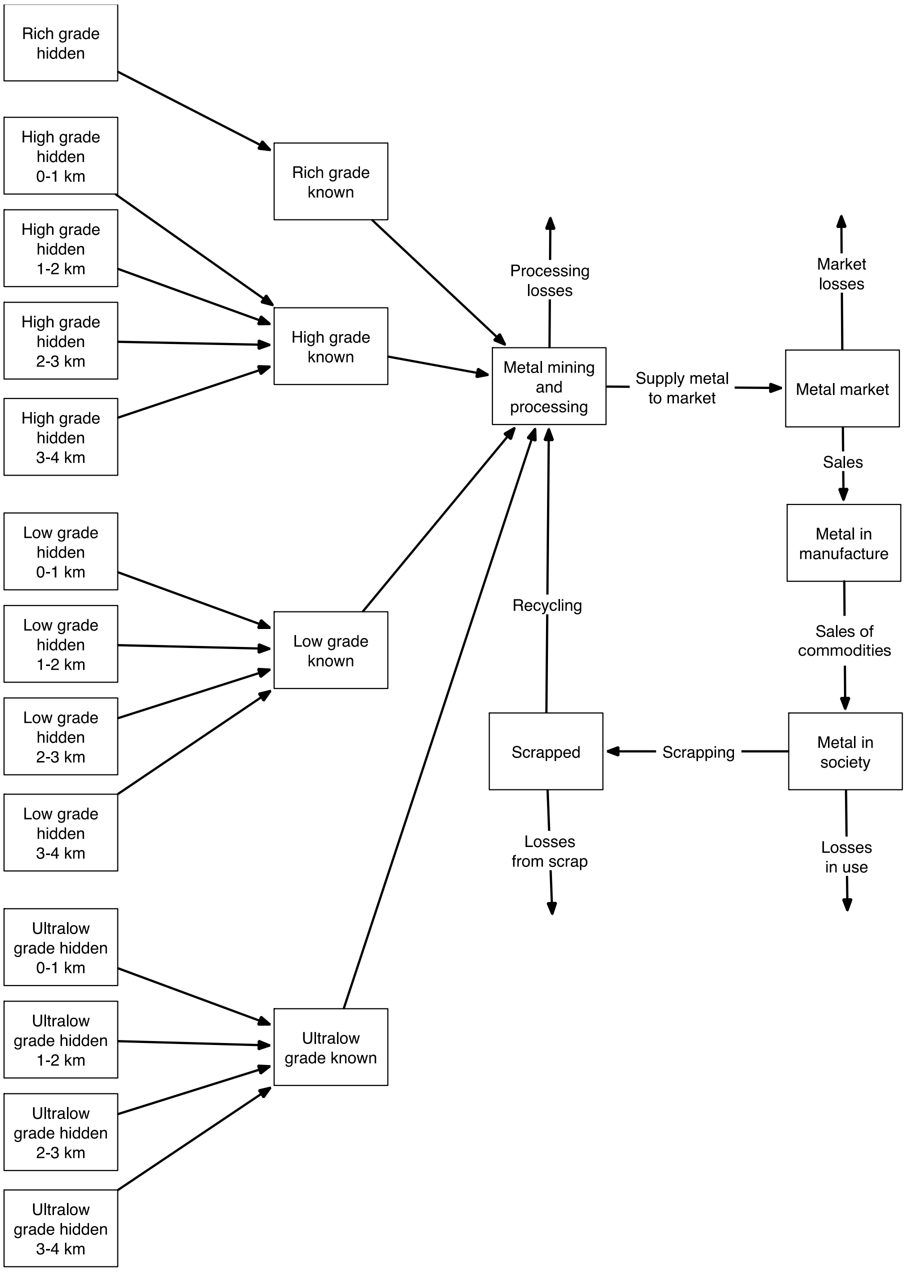
Source: Own Figure, Lund University/University of Iceland

Figure 4.1.3.7. Basic flow chart for chromium included in WORLD6



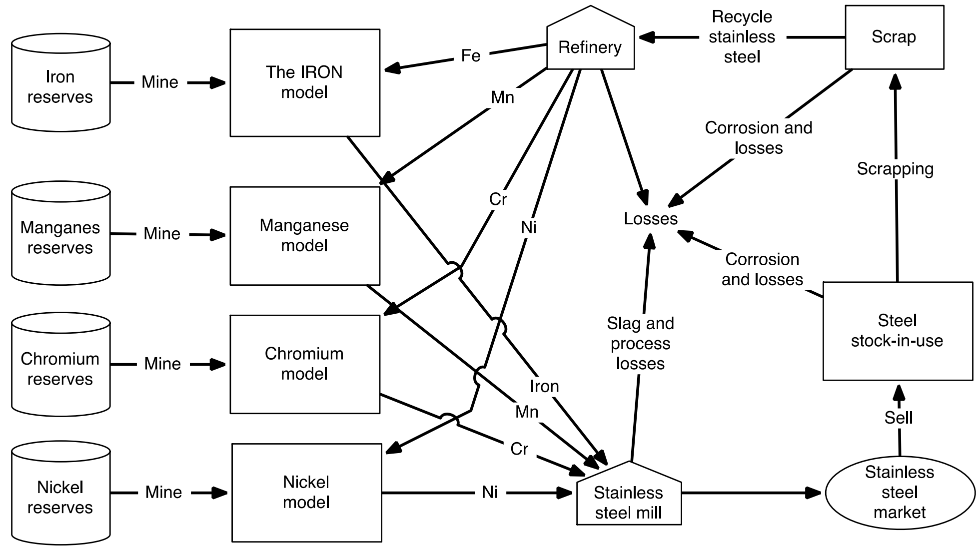
Source: Own Figure, Lund University/University of Iceland

Figure 4.1.3.8. Flow chart for nickel included in WORLD6.



Source: Own Figure, Lund University/University of Iceland

Figure 4.1.3.9. Flow chart for metals used in the production of stainless steel in WORLD6, showing the additions to the model for making alloyed steel using iron, manganese, chromium and nickel.



Source: Own Figure, Lund University/University of Iceland

The profit is calculated as the difference between the income and the extraction costs:

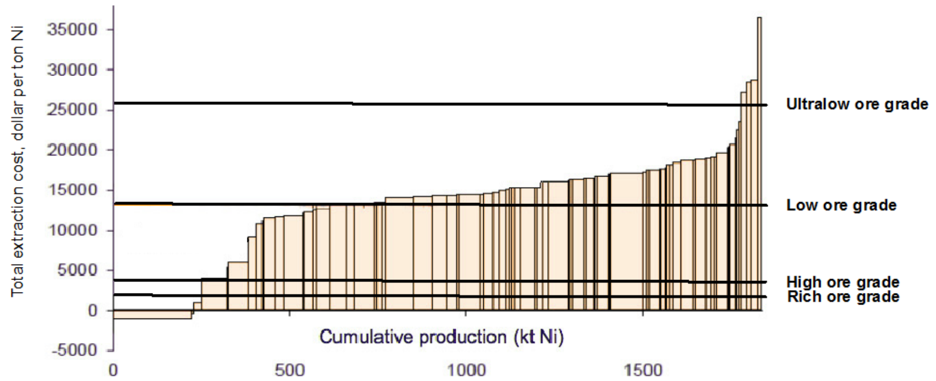
$$Income = Market\ price * supplied\ amount \quad (4.1.3.9)$$

$$Costs = \sum_{i}^{Grades} (Extraction\ cost_i * \ Amount\ produced_i) \quad (4.1.3.10)$$

$$Profit = Income - costs \quad (4.1.3.11)$$

The extraction is stopped for all values of profit less than zero, and scaled up with positive profits. Figure 4.1.3.10. Shows the cost distribution chart from the nickel industry that was used to set the extraction cost functions based on the ore grade. The fact that a resource is extractable, does not necessarily imply that it will get extracted. If recycling is efficient or the price does not reach the level required to make its extraction profitable, it will stay in the ground. This implies that resource exhaustion may not occur, but rather lack of demand at the required price or lack of ability to pay at the required price. All of these are aspects of scarcity.

Figure 4.1.3.10. A cost distribution chart from the nickel industry was used to set the extraction cost functions based on the ore grade.

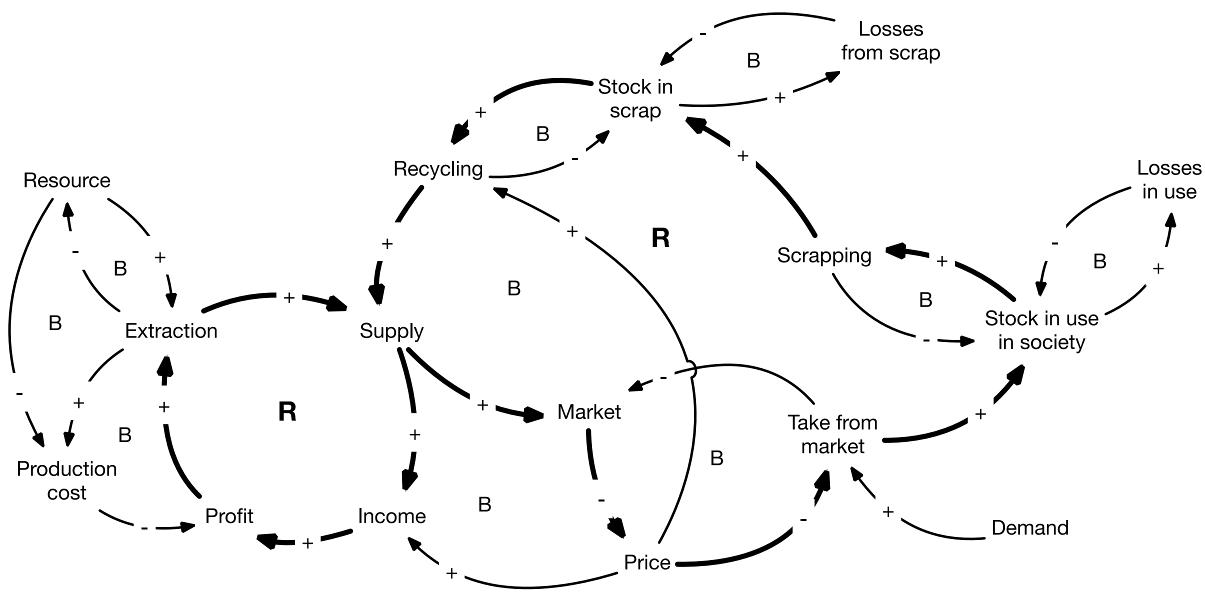


Source: Adapted from http://forexstocknews.com/2018/03/top-5-nickel-miners-to-consider-before-the-nickel-boom/

The Price Model

The general price model applied to manganese, chromium and nickel, it is conceptually the same as that used for silver, gold, aluminium, lithium, platinum and copper used earlier (Figure 4.1.3.11, see Sverdrup et al. 2014, 2015a,b, Sverdrup and Ragnarsdottir 2017, Sverdrup 2017 for further explanations).

Figure 4.1.3.11 shows the generic price model applied in the model.

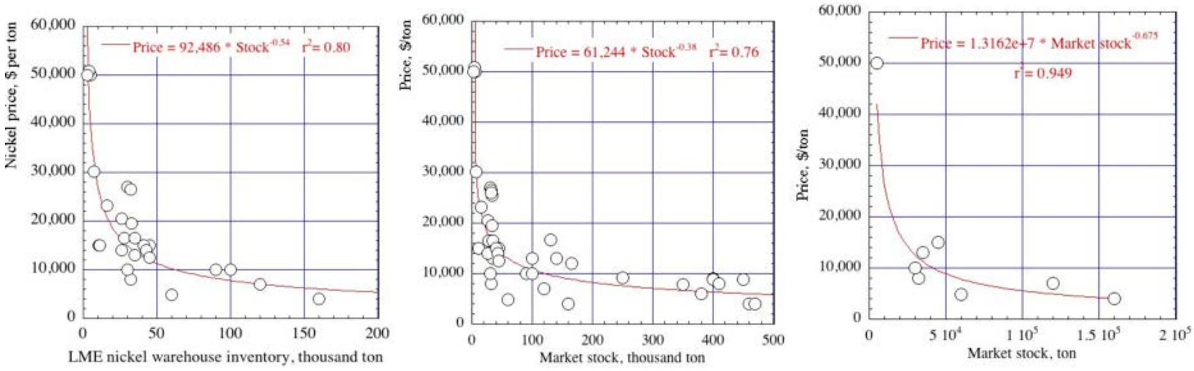


Source: Own Figure, Lund University/University of Iceland

It can be seen how the price affects demand directly, and supply indirectly by affecting mining profits, which in turn affects the mining rate, which again affects supply to the market. This pathway has some delays, causing the price at times to fluctuate for metals with small market stocks that can be mobilized for trade. The implication is that there is not an equilibrium, but that it is the results of market dynamics between demand and supply (mining and recycling), where price equilibrium may never be reached fully. The mining cost is driven by two factors in the model, the oil price and the ore grade. The ore grade determines how much material we must move and treat to make ore concentrate for extracting the metals.

Figure 4.1.3.12 shows the price response curve for nickel, showing the relationship between stock in the market and the market price. We have constructed this diagram by finding data for amounts available in the market and then found the corresponding price for that day. The data was taken from old metal market logs not available in any published form.

Figure 4.1.3.12. The curve for nickel during different time periods showing the relationship between stock in the market and the market price.

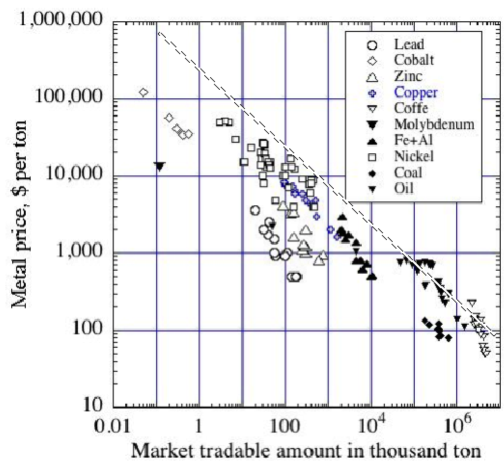


Nickel 2010 Nickel 2012-2015 Nickel 2011

Source: Own Figure, Lund University/University of Iceland

Figure 4.1.3.13 relates global metal production to metal price on a generalized level.

Figure 4.1.3.13. The relationship between metal market availability in general and market price, putting all the data together.



Source: Own Figure, Lund University/University of Iceland

Figure 4.1.3.4 shows the causal loop diagram for the part of the model concerning iron. The basic feature is that demand is taken from the market. The amount in the market sets the price, which has a feedback on supply (higher price gives more mining when the reserves allow this) and demand (higher prices presses down demand). The supply is increased by increasing the mining of iron ore and smelting it to iron metal. When we have no data or very few, maybe only one data point for creating a price curve for a commodity, then we will fit a curve to Equation 4.1.3.12:

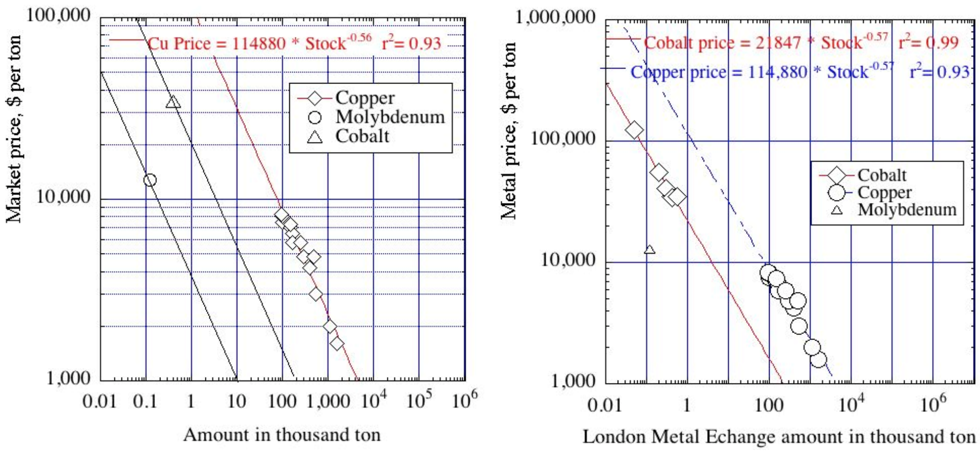
$$Price = k * (Market\ stock)^n \qquad (4.1.3.12)$$

We will demonstrate the procedure as follows using molybdenum and cobalt as examples. The same was done for chromium and manganese.

1. For molybdenum, the following data is available; The London Metal Exchange (LME) stock was 120 ton during March to December 2015, the price was 12,800 $ per ton.
2. For cobalt, there was 400 ton in the LME trade stock in January 2012, the price was then 34,700 $ per ton.

We fitted this information into the diagram for copper, and drew a parallel line as is shown in Figure 4.1.3.14. The figure shows the reconstructed price curve for molybdenum and cobalt, using the copper curve for help. For molybdenum, we do not have good overlapping data for market stocks and price. The molybdenum price is for molybdenum oxide.

Figure 4.1.3.14. The reconstructed price curve for molybdenum and cobalt, using the copper curve for help. a; Approximation of molybdenum and cobalt price curve, (b) the cobalt curve from data.



a; Finding a curve for Co and Mo b; Verifying for Co

Source: Own Figure, Lund University/University of Iceland

For manganese and chromium, it has not been possible to find any estimates of market stocks anywhere, but they are important in the model. In December 2015, the price was 2,200 $ per kg for chromium, and 2,000 $ per ton for manganese. In March 2016, it was 1,580 $ per ton. For manganese, it was 1,630 $ per ton in March 2016. The default equation to use in the model would be:

$$Price = 300{,}000 * (Stock)^{-0.55} \quad (4.1.3.13)$$

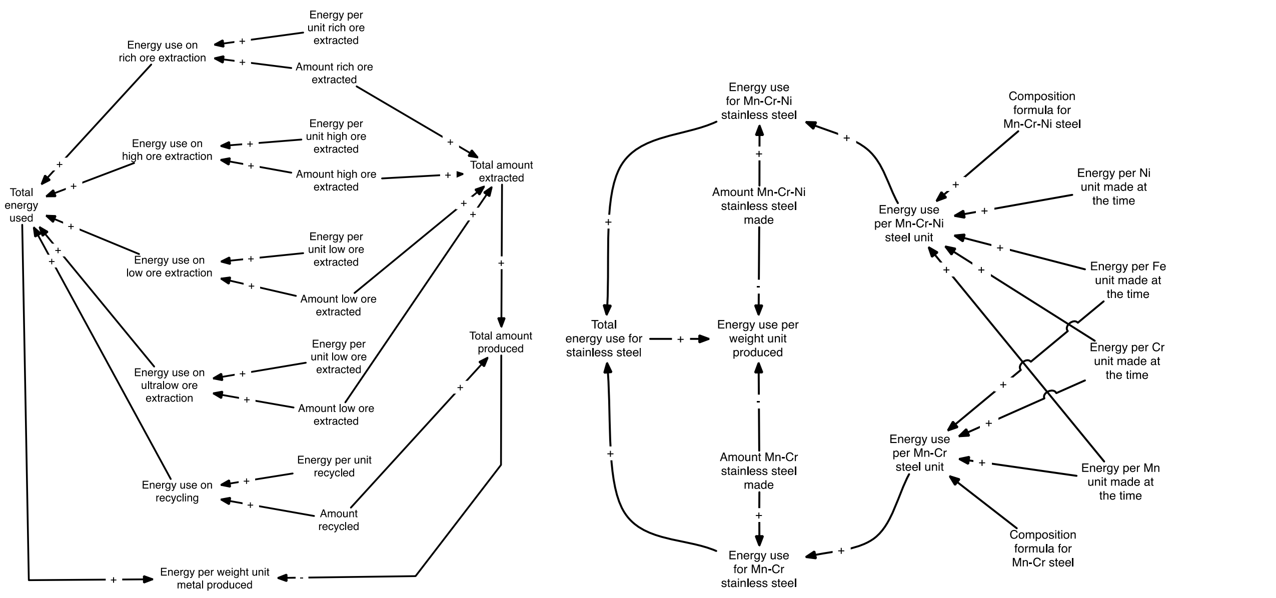
When no information is available, this equation will be used as shown below in an example. For some metals, we do have market stock information can be found; manganese, chromium, cobalt, molybdenum and nickel. We may use the average price to back-estimate the average market stock active in trade and price formation by using the inverse of Equation (4.1.3.13) where k=300,000 and n=0.55. Some values have been shown in Table 4.1.3.17. Cost distribution chart from the nickel industry (Figure 4.1.3.10) was used to set the extraction cost functions based on the ore grade. Extraction cost per ton is used together with supplied amounts and the price to determine the profit. A similar approach was used for manganese, chromium and iron.

Table 4.1.3.17. Summary of price curve parameters conforming to Equation (4.1.3.15). Ratio is the production divided by the typical stock size. Production in 2015 in million ton.

Metal	Production Million ton	Inventory Million ton	Stock location	Ratio	k	n	r2
Iron	1,350	300.000	Society	4.5	2,763,300	-0.94	0.83
Copper	16.7	0.200	LME	80	115,000	-0.56	0.93
Manganese	18.0	0.300	Producers, LME	60	120,000	-0.6	-
Zinc	12.3	0.300	LME	41	116,000	0.77	0.80
Chromium	9.0	0.300	Producers, LME	30	180,000	-0.7	-
Nickel	2.5	0.100	LME	25	61,244	-0.38	0.76
Cobalt	0.067	0.00015	LME, Strategic stocks	450	21,847	-0.57	0.99

Figure 4.1.3.15a shows the causal loop diagram for the calculation of energy use for the stainless steel production in the model. Figure 4.1.3.15b shows the causal loop diagram for the calculation of total energy used for the metals used to make stainless steel, and the specific amount used per weight unit, as it was programmed in the model. The model is based on mass balance expressed differential equations, and solved numerically with a 4-step Runge-Kutta method, using a 1/512-year time-step in the integration (daily).

Figure 4.1.3.15. (a) Causal loop diagram for the calculation of energy use for the stainless steel production. (b) Causal loop diagram for the calculation of total energy used for the metals used to make stainless steel, and the specific amount used per weight unit.

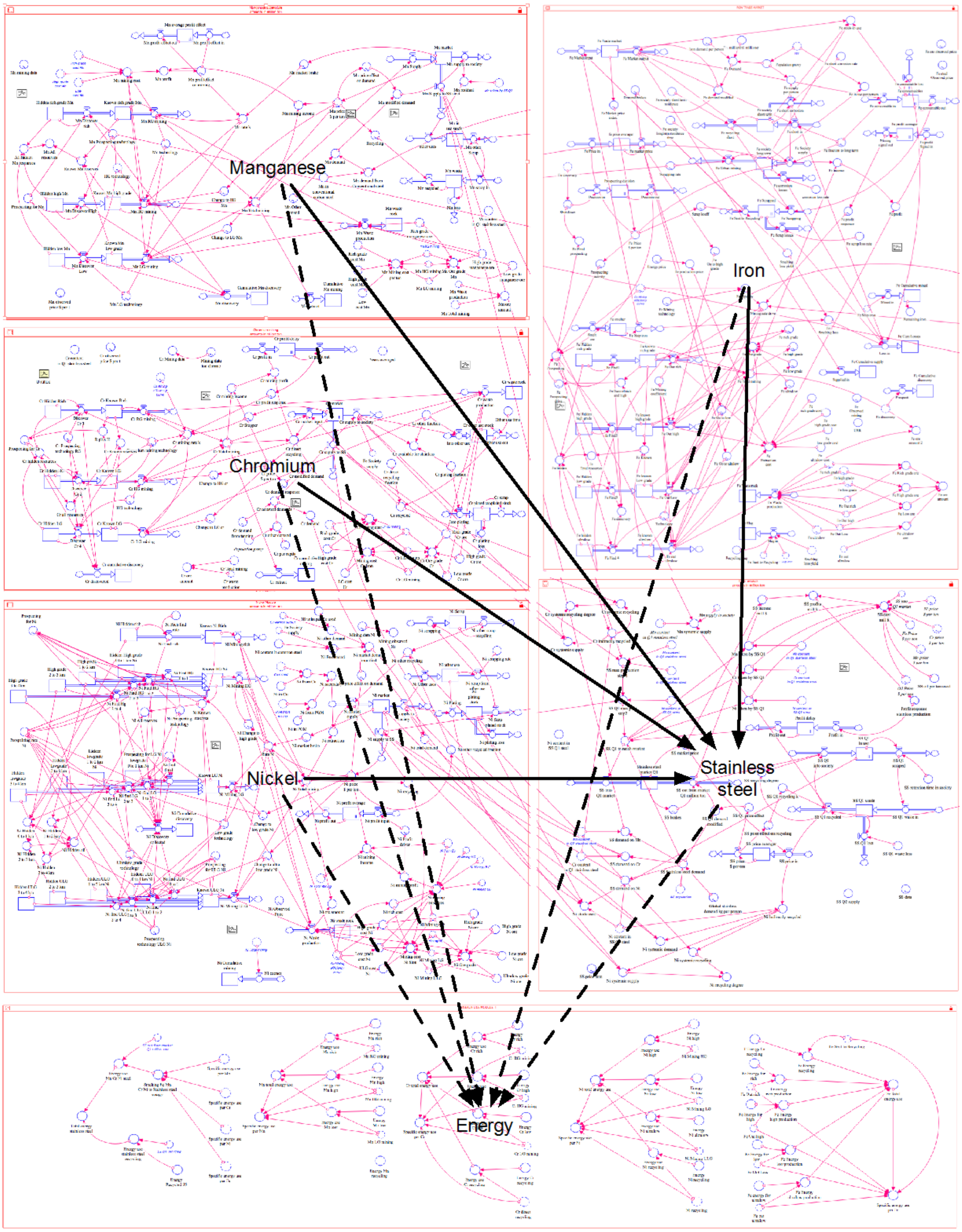


a b

Source: Own Figure, Lund University/University of Iceland

Figure 4.1.3.16 shows the STELLA diagram for the STEEL module in the WORLD6 model.

Figure 4.1.3.16. The STELLA diagram for the STEEL module.



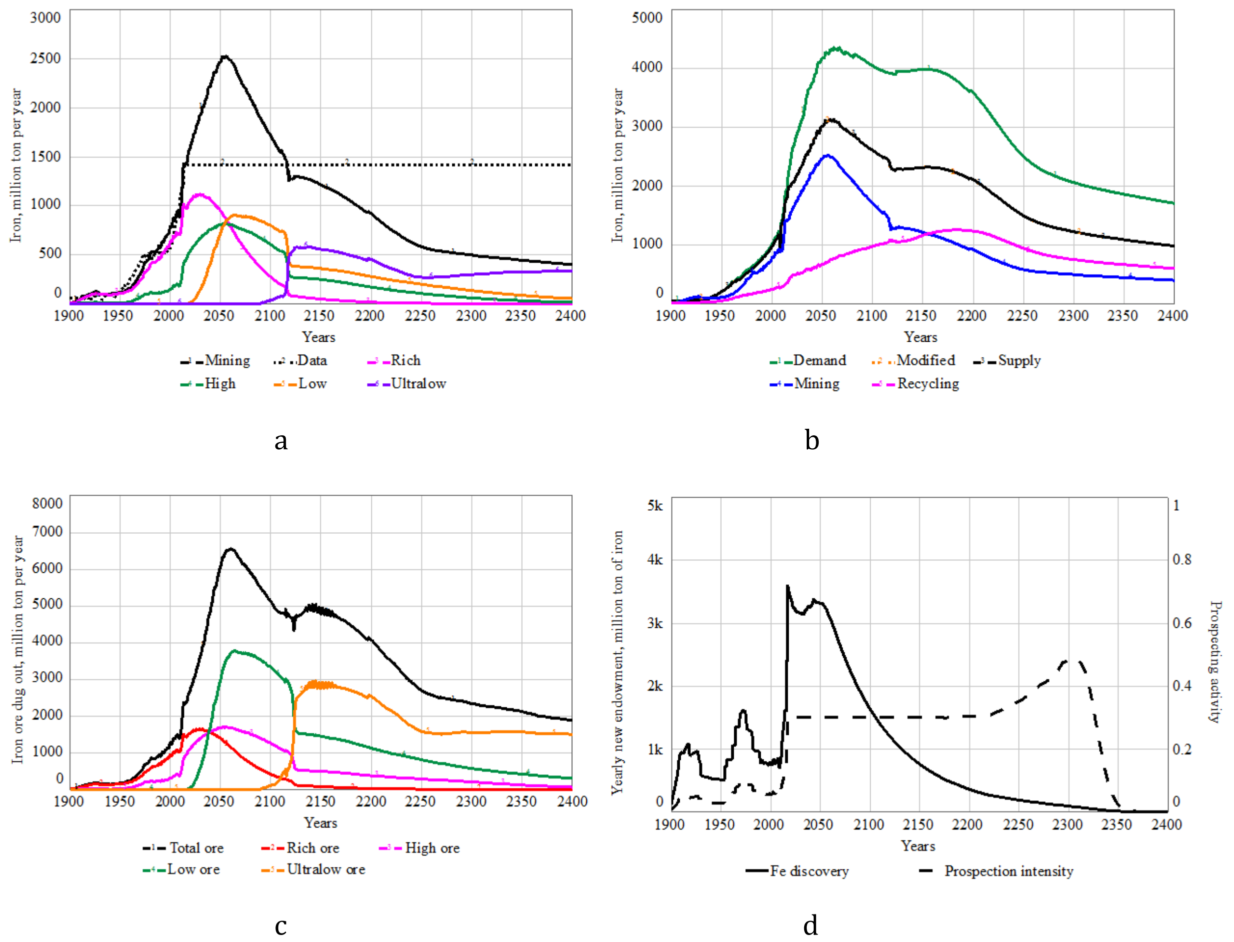
Source: Own Figure, Lund University/University of Iceland

4.1.3.7 Results

Iron

Figure 4.1.3.17 and Figure 4.1.3.18 show outputs from the WORLD6 model simulation outputs for iron, based on URR set at 500 billion ton extractable iron. Figure 4.1.3.17a shows the mining rate for iron, and specified per ore quality. The dots represent the observed mining rate. Figure 4.1.3.17b shows the demand, the demand after modification by the price, the market supply, the mining rate and the amount recovered through recycling. Figure 4.1.3.17c shows the extraction of ore amounts, specified per ore quality in million ton per year. Figure 4.1.3.17d shows the rate of reserve endowment from prospecting and the prospecting activity itself. When the prospecting no more finds new ore, then prospecting is stopped in the model.

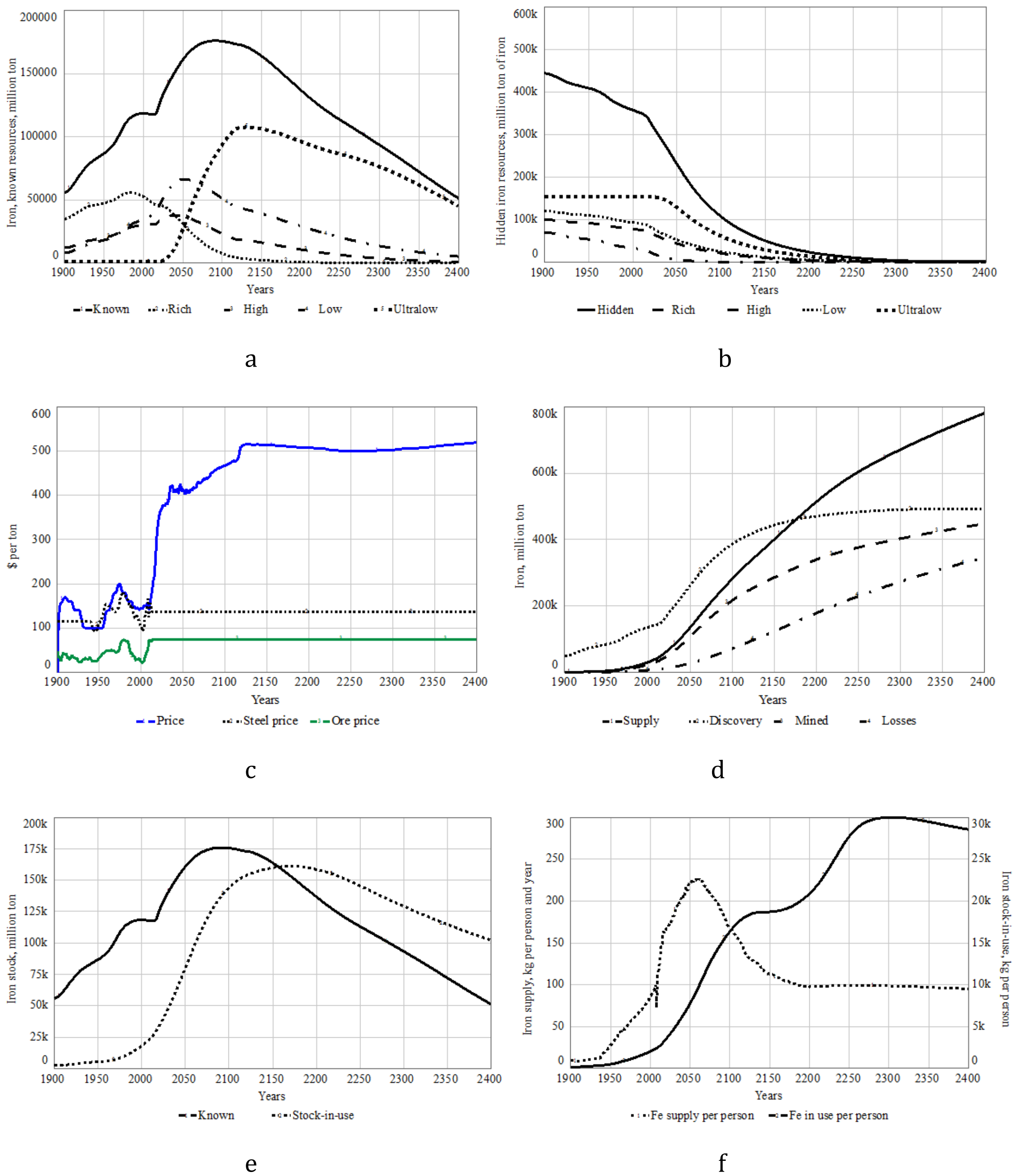
Figure 4.1.3.17. Diagram (a) shows the mining rate for iron, and specified per ore quality. The dots represent the observed mining rate. Diagram (b) shows the demand, market supply, mining and recycling rate, diagram (c) shows the ore amounts taken out according to ore quality. Diagram (d) shows the rate of reserve endowment from prospecting (solid line) and the prospecting activity itself (dotted line).



Source: Own Figure, Lund University/University of Iceland

Figure 4.1.3.18a shows known amounts of reserves distributed among ore qualities. Figure 4.1.3.18b shows the hidden resources distributed among different ore qualities. Figure 4.1.3.18c shows the inflation-eliminated iron price as compared to the available data as compared to available price data. Figure 4.1.3.18d shows the cumulative amounts found by prospecting, mined, supplied and lost for iron. The curves suggest that the Factor X is about 1.8 for iron. Faktor X is the ratio between amount supplied and amount mined. Figure 4.1.3.18e shows amount in stock-in-use in society and known extractable amounts. By 2050, the stock in use will be larger than the known extractable amounts. Amounts are in million ton of iron. Then we will be in the iron scrap age. Figure 4.1.3.18f shows the stock-in-use as kg per person and the annual supply in kg per person. It is evident that the world will not run out of iron in the next 400 years. However, it should be noted that by 2200, all of the rich ore grades and most of the high grade ores will be gone. As soon as the rich grade starts to run out, a significant rise in iron price should be expected as is shown in Figure 4.1.3.18c. This increase comes from the increase in mining costs, ore treatment costs and iron extraction costs. The costs also include capital costs.

Figure 4.1.3.18. (a) shows the known amount of iron in different quality classes of ore. (b) shows the amounts hidden in different ore qualities. (c) shows the iron price as compared to the available data. (d) shows the cumulative amounts mined, supplied, discovered, recycled and lost. (e) shows amount in stock-in-use in society and known extractable amounts. Amounts are in million ton of iron. (f) shows the stock-in-use as kg per person and the annual supply in kg per person.

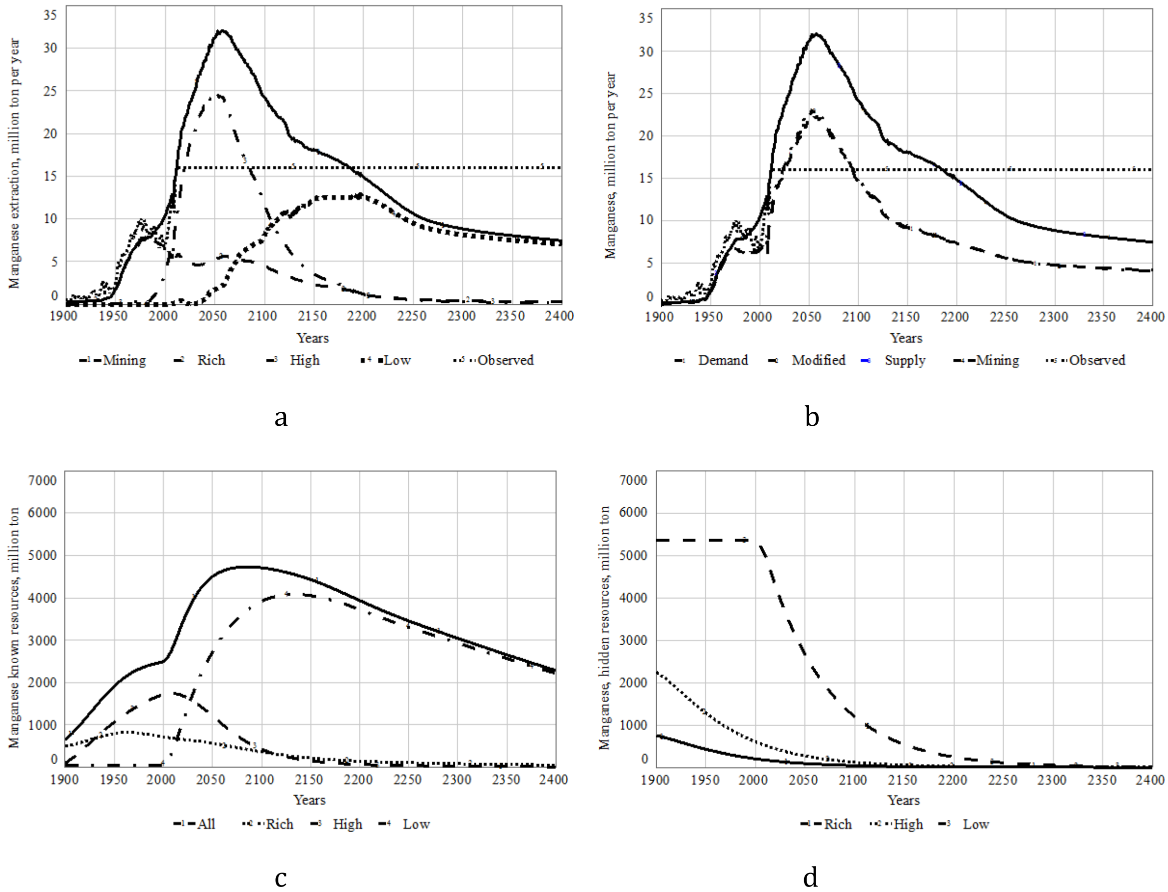


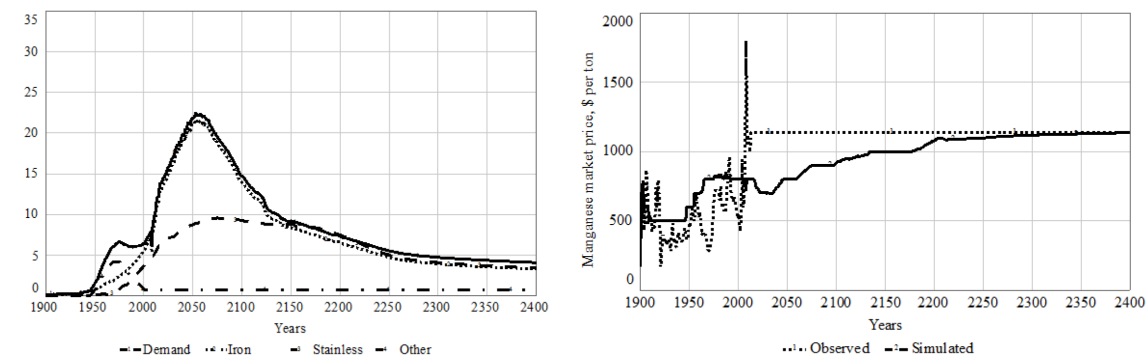
Source: Own Figure, Lund University/University of Iceland

Manganese

Figure 4.1.3.19 shows the WORLD6 model simulation results for manganese, based on URR set to 9 billion ton. Figure 4.1.3.19a shows the mining rate of manganese from different ore grades. Figure 4.1.3.19b shows the full demand from unbridled consumption, the modified demand after adjustment by price, the supply simulated in the model, the estimated mining rate and the observed mining rate according to the data. The simulation is recreating the mining rate quite well (r2=0.96). The observed data were not used to calibrate the model, the mining rate is the result of the demand on the market, driving the price and thus the profit. The demand is mostly driven by the demand from manufacture of stainless steel, the demand from production of carbon steels and other uses play a minor role. Figure 4.1.3.19c shows the development of the known manganese reserves over time, they reach a maximum in 2030, and then decline. Figure 4.1.3.19d shows the amounts of hidden resources of manganese with time. Figure 4.1.3.19e shows supply of manganese to different sectors in society. Figure 4.1.3.19f shows the simulated market price for manganese compared to the data. The simulation of manganese may show different results, depending on the size of the real resource available for mining. The resource estimates are uncertain, but we have chosen to use an estimate close to the top of the range (9 billion ton), consistent with the latest findings in the scientific literature. However, any vale for URR in the range 6-12 billion ton could possibly be defended.

Figure 4.1.3.19. (a) Mining of manganese from different ore grades. (b) demand, modified demand, supply, mining and observed mining rate. (c) known reserves of manganese, (d), hidden resources of manganese. (e) the supply of manganese to different sectors, (f) the simulated market price for manganese compared to the data.





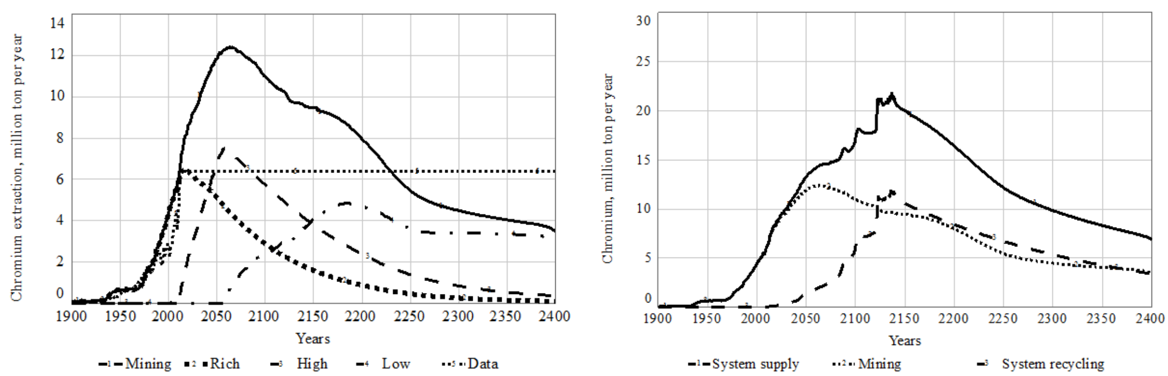
e f

Source: Own Figure, Lund University/University of Iceland

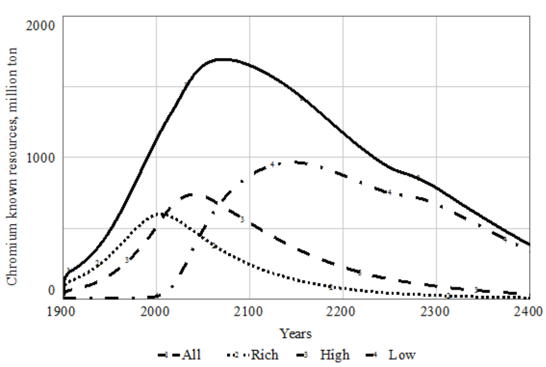
Chromium

Figure 4.1.3.20 shows the WORLD6 model simulation results for chromium, based on URR set to 3.4 billion ton. Figure 4.1.3.20a shows the mining rate of chromium from different ore grades and total mining as compared to the data. Figure 4.1.3.20b shows the systemic supply, mining and system recycling. The systemic supply and recycling include recycling of chromium as a metal and when stainless steel is recycled directly, including the chromium inside it. Figure 4.1.3.20c shows the development of the known reserves of chromium, and Figure 4.1.3.20d shows the hidden resources of chromium with time. The known reserves show a maximum in 2100. Figure 4.1.3.20e shows the total systemic recycling of chromium as fraction of supply. Figure 4.1.3.20f shows the simulated market price for chromium compared to the data. The price compares well with the observed data. The simulation of chromium may show different results, depending on the size of the real resource available for mining.

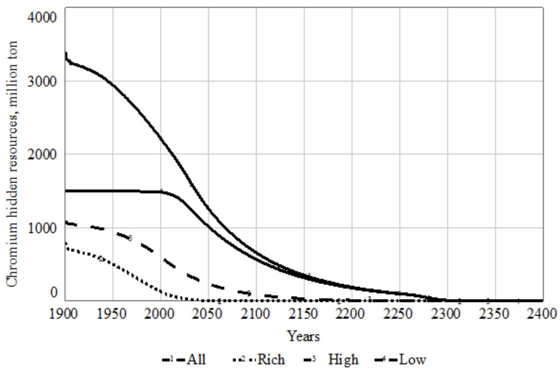
Figure 4.1.3.20. (a) Mining of chromium from different ore grades and total mining as compared to the data. (b) Systemic supply, mining and system recycling. (c) known reserves of chromium, (d), hidden resources of chromium. (e) recycling of chromium as fraction of supply, (f) the simulated market price for chromium compared to the data.



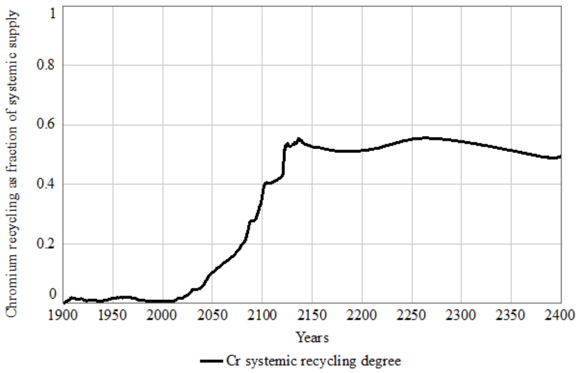
a b



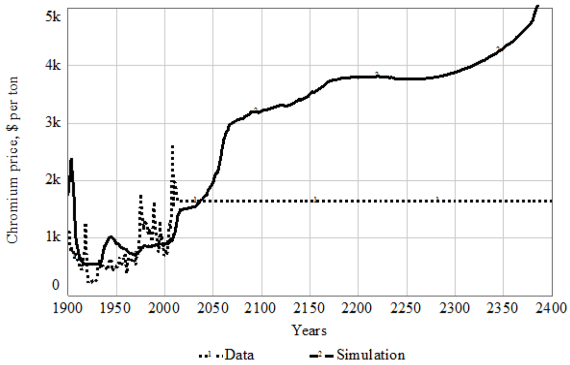
c



d



e



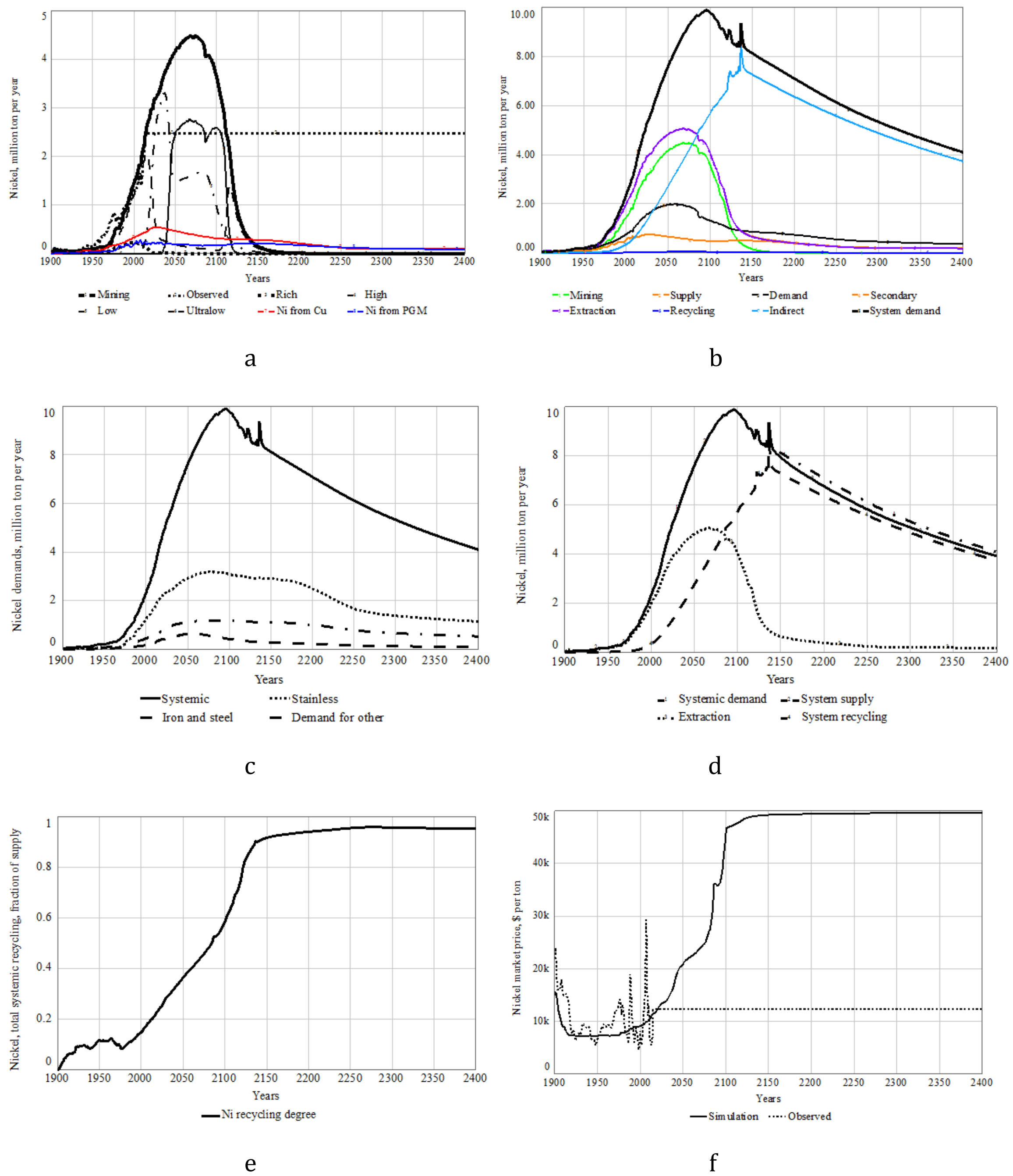
f

Source: Own Figure, Lund University/University of Iceland

Nickel

Figure 4.1.3.21 shows the WORLD6 model simulation outputs for nickel with URR set at 500 million ton. Figure 4.1.3.21a shows the extraction of nickel from different ore grades and secondary sources. Figure 4.1.3.21b shows the model output for demand, demand modified with the price, the mining rate, secondary extraction, recycling and indirect recycling of nickel. Figure 4.1.3.21c shows the supply of nickel to different sectors. Figure 4.1.3.21d shows the systemic supply and systemic demand, showing demand-supply separation. Figure 4.1.3.21e shows the total system degree of recycling, both direct and what follows as a consequence of stainless steel recycling. Figure 4.1.3.21f the simulation of nickel price and comparison with the data.

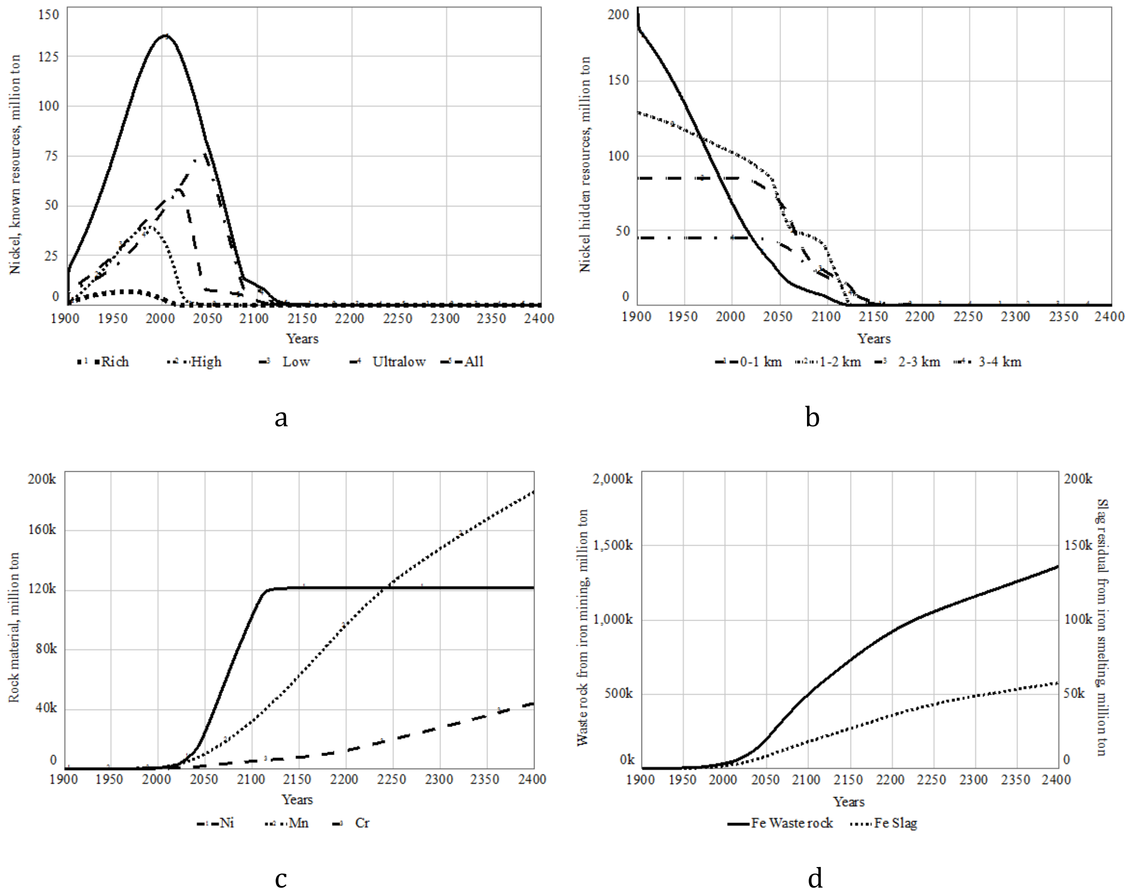
Figure 4.1.3.21. Extraction of nickel from different ore grades and secondary sources. (b) Demand, modified demand, mining, secondary extraction, recycling and indirect recycling of nickel. (c) supply of nickel to different sectors. (d) system demand, supply, total extraction and recycling, It is showing demand-supply separation (e) the degree of recycling as fraction of total supply. (f) The simulation of nickel price and comparison with the data.



Source: Own Figure, Lund University/University of Iceland

Figure 4.1.3.22a shows the known reserves of nickel and Figure 4.1.3.22b the hidden and known reserves of nickel. Figure 4.1.3.22d shows the amounts of waste rock and slag from iron mining and refining. The amounts of waste rock produced in iron mining are very substantial. No account has been taken of removal of overburden to reach the actual ore body, nor any other waste produced in setting up supporting structures to the mines.

Figure 4.1.3.22. (a) Known reserves of nickel and (b), the hidden and known reserves of nickel. (c) shows the cumulative amounts Mn, Cr and Ni extracted. (d) shows the amount of waste rock produced by chromium, manganese and nickel mining, and e shows the amounts of waste rock and slag from iron mining and refining.

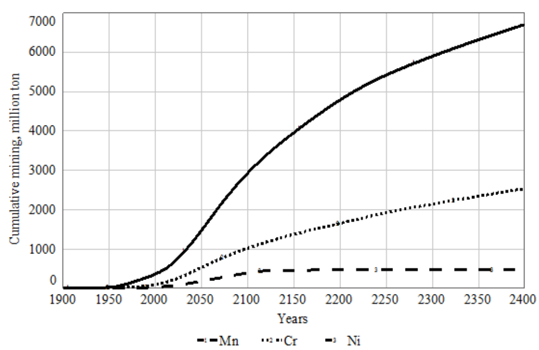


Source: Own Figure, Lund University/University of Iceland

Ore grade and total amounts

Figure 4.1.3.23 shows the cumulative extracted amounts of Mn, Cr and Ni. The amounts to the year 2400 are for manganese about 6.947 billion ton (URR=9 billion ton), for chromium about 2.536 billion ton (URR=3.4 billion ton) and for nickel about 462 million ton.

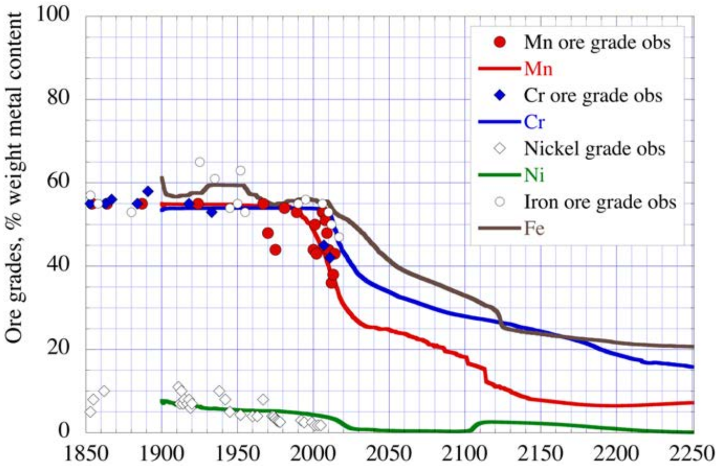
Figure 4.1.3.23. Cumulative amounts of manganese, chromium and nickel extracted in the period 1900 to 2400.



Source: Own Figure, Lund University/University of Iceland

Figure 4.1.3.24 shows the predicted ore grades as mining continues to exhaustion for iron, manganese, chromium and nickel as simulated using the model, compared to the observed ore grades. Decreasing ore grade cause increased extraction costs, requiring a higher market price. Deceasing ore grades are a certain diagnostic sign of mine exhaustion. When the decrease is on global scale, the exhaustion risk is on global level. This is in the process of evolving, and is a very strong warning signal for the future. Ignoring such strong signal would constitute gross neglect.

Figure 4.1.3.24. Ore grades for iron, manganese, chromium and nickel as simulated using the model, compared to the observed ore grades. Mn, Cr, Ni and Fe are the simulated ore grades by the model.

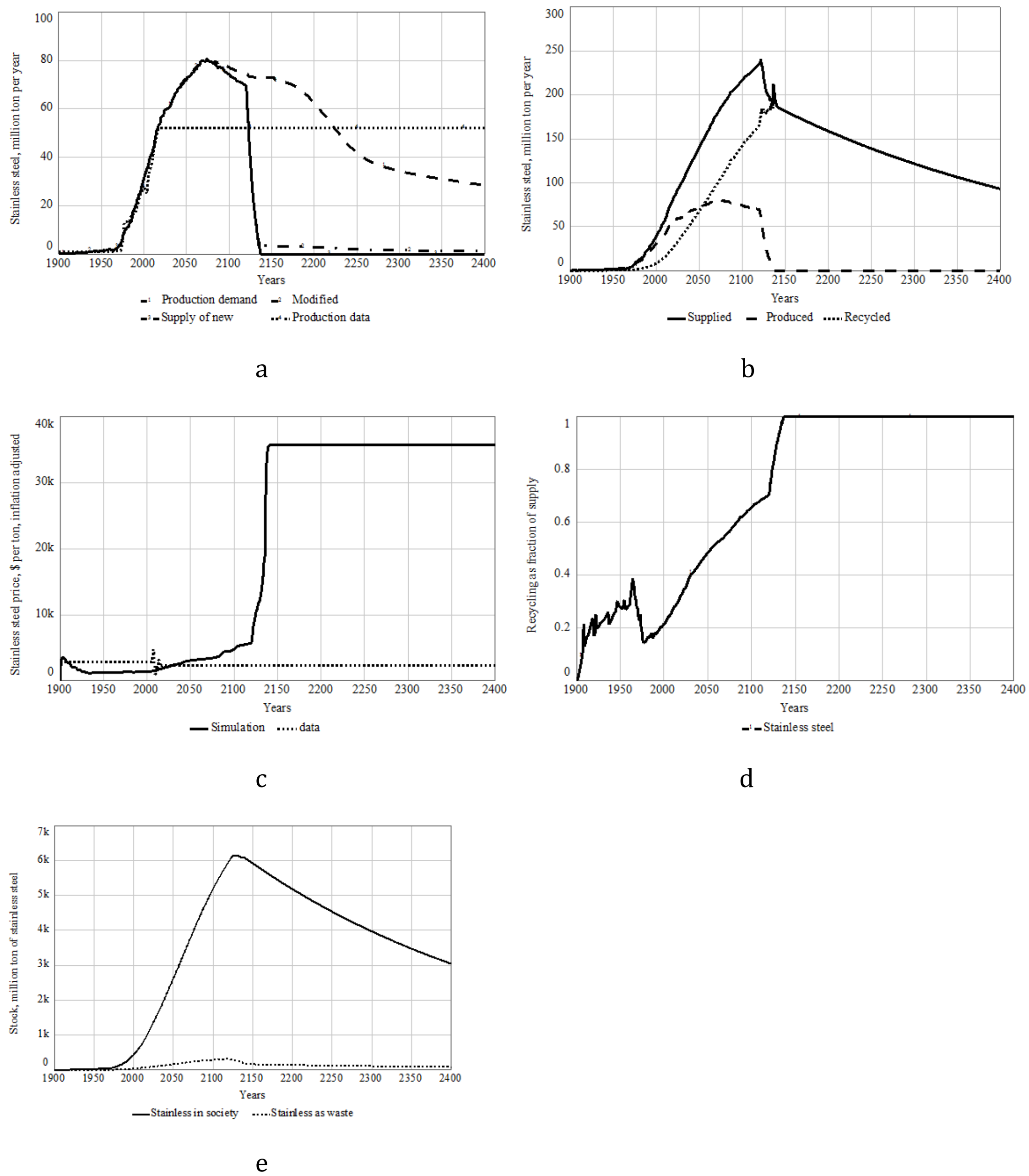


Source: Own Figure, Lund University/University of Iceland

Stainless steel

The model was used to simulate the supply of the individual metals used for stainless steel; iron, manganese, chromium and nickel. The model makes stainless steel from these metals. In the production of stainless steel nickel is predicted to become limiting metals for production, shifting a part of stainless steel to chromium-based stainless steel. Figure 4.1.3.25 shows the production rate for stainless steel and the simulated global stock-in-use for stainless steel. The stock-in-use reach a peak in 2140, and then decline. Figure 4.1.3.25a shows the production of stainless steel. Shortage will become very pronounced after 2140, when demand can no longer be satisfied. Figure 4.1.3.25b shows a comparison of supply, production and recycling. After 2140, most supply will come from metal recycling. Figure 4.1.3.25c shows the simulation of the price as compared to the data. Figure 4.1.3.25d shows the degree of recycling as a fraction of total supply. Figure 4.1.3.25e shows the estimate for stock-in-use and stainless steel as waste metal. The long term supply of stainless steel is determined by several factors: Nickel is a limited resource, and even with very generous estimates of nickel reserves, it may set limits to how much stainless steel it is possible to make. It can be seen that after 2050, nickel will start to run out and nickel-based stainless steel production will start to decline after peaking at about 68 million to per year. This just a decade away with our predicted global population maximum, using the WORLD6 model. The missing amount will be replaced by a simpler chromium-manganese stainless steel alloy. The available amounts of molybdenum, vanadium, niobium or cobalt are far too small to act as proper substitutes for nickel. The stock-in-use in society of stainless steel will reach a maximum in 2085, and then slowly decline. We have plotted extracted amount with time to 2200 AD.

Figure 4.1.3.25. (a) shows the production of stainless steel. (b) shows a comparison of supply, production and recycling. (c) shows the simulation of the price as compared to the data. (d) shows the degree of recycling as a fraction of total supply. (e) shows the estimate for stock-in-use and stainless steel as waste metal.

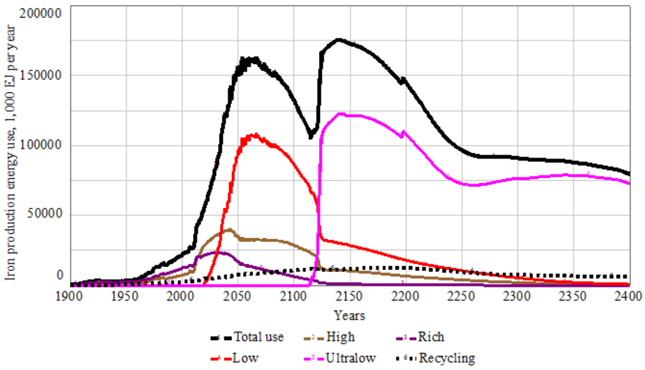


Source: Own Figure, Lund University/University of Iceland

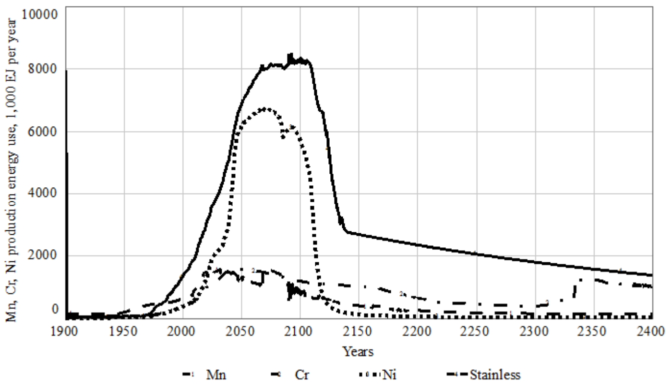
Energy use for metal mining, smelting and production

Figure 4.1.3.26a shows the energy use for making the global iron production. Figure 4.1.3.26b shows the energy use for manganese, chromium, nickel and stainless steel. The average of the range of the energy values in Table 4.1.3.17 were used for the calculations. The energy needed for these metals (Fe, Mn, Cr, Ni and stainless steel) amounted to approximately 5% of all global energy available in 2016. By 2060, this will have increased to an amount five times the amount in 2016. It is self-evident that this will cause a huge demand on energy markets and sending energy prices up. The energy needs for the iron and steel sector will be most challenging to satisfy in the time after 2100.

Figure 4.1.3.26. Diagram a) shows the energy use for mining, benefication, smelting and finalizing production of iron. Diagram b) shows the energy use for making manganese, chromium, nickel and stainless steel. 1 EJ is about 23.9 million ton oil equivalents



a



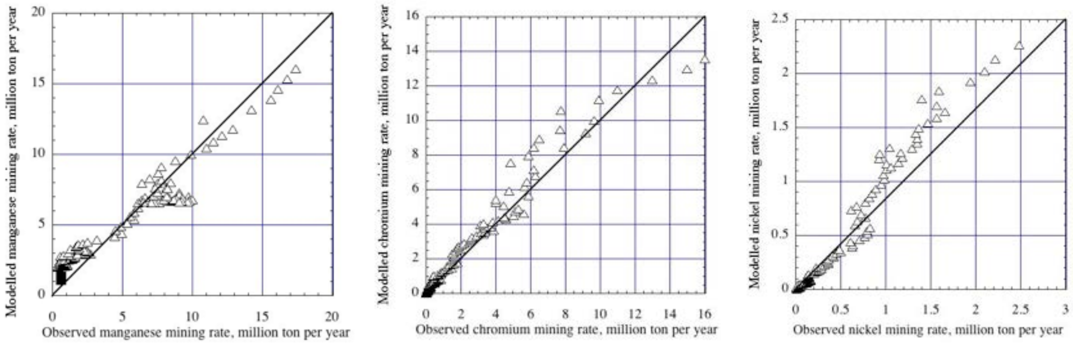
b

Source: Own Figure, Lund University/University of Iceland

Model validation

Data for mining rates were compared to the total mining rate outputs from the model for iron, manganese, chromium and nickel. Before 1880, lower ore grades were also used as manganese occurred in proximity to coal deposits, but the total volume was low compared to the whole production, and it does not show up in the time range we have plotted. Figure 4.1.3.27 shows the simulated ore grades together with the observed data, and the simulations and the data are consistent. The implications of declining ore grades for iron, manganese, chromium and nickel has large consequences. The extraction costs goes up exponentially with declining ore grade. The tests show that the model performs well, the correlation to observed data is better than r2=0.96 for all the metals. For stainless steel we get r2=0.98.

Figure 4.1.3.27. Comparison between the model outputs and the observed data for the mining rate of diagram (a): manganese, diagram (b): chromium, diagram (c): nickel. The diagrams show that the model is quite successful in predicting the mining rate for all the metals.

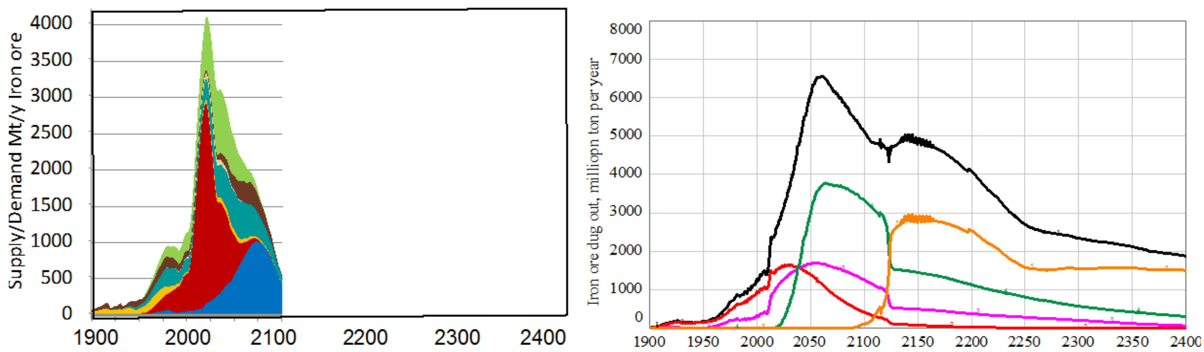


a b c

Source: Own Figure, Lund University/University of Iceland

Figure 4.1.3.28a shows an estimate of iron ore supply made by the team at Monash University in Australia, using a mass flow model (Giurco et al. 2013, Nickless et al., 2014, Mohr et al., 2014). Figure 4.1.3.28b shows our WORLD6 estimate of iron mining from ore, expressed as iron ore amounts (and not iron metal as our other figures). Giurco et al., 2013 used for iron URR=280 billion ton, we used for iron URR=500 billion ton. Their curve only runs to 2100 and is truncated prematurely as compared to our graph. They get a very sharp peak, we get a more rounded shape. The comparison shows that our study and that of our colleagues in Australia (Monash University, Australia) reach the same conclusions, using different modelling approaches, with respect to the sustainability issues.

Figure 4.1.3.28. Diagram (a) shows an estimate of iron ore supply, using a mass flow model (Giurco et al. 2013, Nickless et al., 2014, Mohr et al., 2014). This should be compared to diagram (b) with the output from the WORLD6 model which shows ton iron metal. 2.5 billion ton of iron metal corresponds to about 4.1 billion ton of iron ore assuming 60% content



a: Australian study b: this study

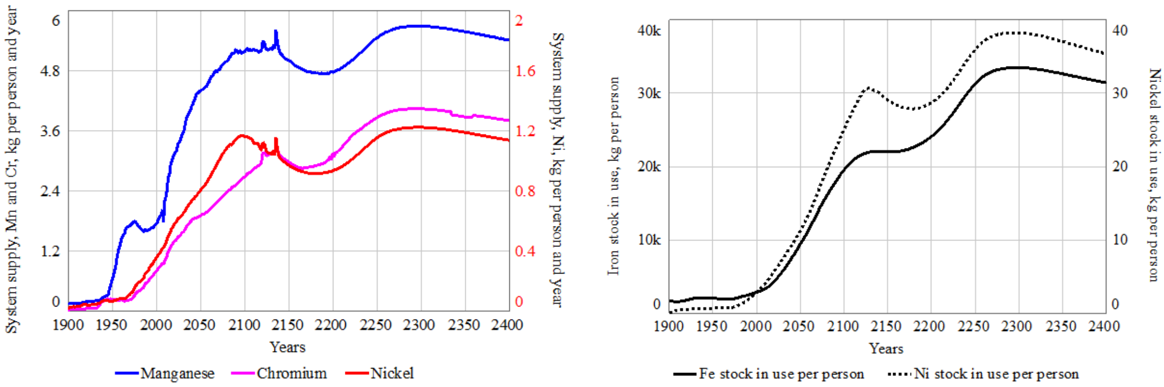
Source: (a) Giurco et al., 2013. (b) Own Figure, Lund University/University of Iceland

4.1.3.8 Conclusions

In this study, we have come up with substantially larger ultimately extractable resources for manganese, chromium, nickel and iron that earlier studies. These estimates have been achieved by reviewing a large number of literature sources and corporate documents and adding the amounts up, having found justification for the numbers used. This sets a new scene for making sustainability assessment, calling for new simulations using the fully integrated WORLD6 model.

The tests show that the model can simulate the observed historical mining rates for the period 1900-2015, based on reserve size, observed mining technology efficiencies and global population. The metal prices are generated endogenously in the model with accuracy sufficient to get the systemic feedback from price dynamics right. The model seems to perform well when tested against field data for mining rates, production rates, ore grades and prices over time. The assessment shows that specifically, nickel may become a limiting resource for making stainless steel. Figure 4.1.3.29 shows the supply per person and the stock in use for the key metals. The figures reveal that despite a peak supply, the development of population predicted at the same time, takes away most of the risk for scarcity.

Figure 4.1.3.29. The supply per person and the stock in use for the key metals.



a: Supply per person and year

b; Stock-in-use

Source: Own Figure, Lund University/University of Iceland

For the WORLD6 model with relation to ferrous metals, we may conclude:

1. The WORLD6model performs well in tests on historical data for mine production, market supply, metal price, ore grade and stock in society during 100 years for iron, chromium, manganese and nickel.
2. The WORLD6 model performance for production amounts and market prices suggests it is a good component in the model.
3. The WORLD6 model test on historical data lends credibility to predictions for the future.

For global sustainability we have the following conclusions from this part of the study:

1. Running out of iron, steel and stainless steel on large global scale will make global climate change appear as a small problem. After 2100, this appear to be a real reality that may happen unless measures to mitigate it are taken.
2. Energy limitations may become a large issue with iron and steel production after 2100. This will need future attention from science and policy.

With increased price because of scarcity, we may expect recycling to go up and soften the decline somewhat. At recycling degrees above 80%, the supply of nickel, chromium and manganese will be sufficient for a century, but eventually run out.

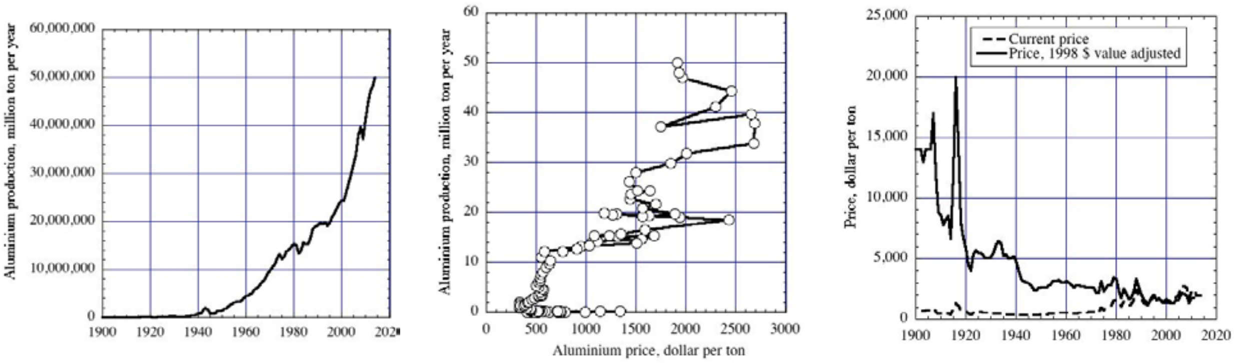
4.1.4 LIGHT METALS sub-module ALUMINIUM

4.1.4.1 Introduction

Aluminium is the second most important metal for modern human civilization. In this paper, we will use integrated model to assess the long term outlook for future aluminium primary production and supply of aluminium to society. Special attention will be made with respect to recycling and aluminium conservation and accumulation in society, considering dynamic feedbacks from market mechanisms and policy. The model developed for this study, generate world market aluminium price internally from the interactions of demand and supply through market mechanisms.

Fifty million ton aluminium metal per year was produced in 2015 (USGS 2015). Overall the aluminium production has grown an average of 2.5% per year for the last 25 years. Figure 4.1.4.1a shows the global aluminium production since 1900 to the present; the price has not gone down with increasing amounts of production, suggesting that the demand is increasing and taking everything that is produced (Figure 4.1.4.1b). Historically, primary aluminium production has been gradually made more energy efficient, and most of the mining is now located in low wage, developing countries. At the same time, no major change in ore quality has occurred yet. Only iron has a larger mine production than aluminium with about 1,450 million ton iron mined per year. Aluminium mining and smelting amounts to about 50 million ton per year at present. This constitutes 97% of all global metalmaking. Before 1920, aluminium was produced in insignificant amounts, but with the development of new production processes, the metal became important when it could be relatively cheaply produced in large amounts.

Figure 4.1.4.1. (a) Global aluminium production since 1900 to the present; the amounts are expressed as million ton. (b) The price has not gone down with increasing amounts of production in the last 3 decades, suggesting that the demand is also increasing and taking everything produced. (c) The price in dollars is shown in dashed line (same as in (b)), and the solid line gives price that is inflation adjusted using 1998 as reference.



a b c

Source: Own Figure, Lund University/University of Iceland

Bauxite is the main ore for aluminium, and by far the most cost efficient source of aluminium extraction. Bauxite, a mixture of aluminium and iron oxides, is dug up from large open pit mines; it formed as a result of weathering of plutonic rocks in tropical or former tropical areas. Bauxite consists of the minerals gibbsite ($Al(OH)_3$), boemite ($AlO(OH)$) and diaspore $AlO(OH)$, and mixed in are kaolinite ($Al_2Si_5(OH)_4$), the iron bearing minerals goetite ($FeO(OH)$) and haematite (Fe_2O_3) and small amounts of anatase (TiO_2). Red mud is the waste after refining bauxite to alumina (Al_2O_3), and the amounts produced are very large; red mud is caustic and represents an environmental hazard. It is formed when bauxite is treated with hot alkali solutions; the resulting aluminium hydroxides are later burned to get alumina. This alumina is then reduced in aluminium smelters in the presence of coal, resulting in roughly 1.2 tons of CO_2 for each ton of aluminium metal (e.g. Sverdrup and Ragnarsdottir 2014).

Making aluminium from other ore than bauxite is not economic at present, but may be in the future if bauxite reserves should run out. A good quality bauxite ore has a low content of alkali metals (CaO, MgO, Na_2O, K_2O), low contents of iron oxy(hydr)oxides (FeO(OH)), Fe_2O_3, Fe_3O_4) and titanium oxide (TiO_2), and especially low content of silica (SiO_2). Bauxite ore quality is in the first stages of declining reserve quality at present, a diagnostic indicator that identifies a need to asses the future of bauxite mining and aluminium supply (Alumina Limited 2012). Nepheline ($NaAlSiO_4$), a feltspatoid, is the only mineral so far used for alumina production (in Russia, about 800,000 ton alumina per year was produced in 2015); per weight nepheline contains 44% alumina, the Russian ore has 24-28% alumina bulk content (Smirnov 1996, 2014; Sverdrup 1990). Kaliophilite ($KAlSiO_4$) is the potassium end member of the same type of mineral and there is a continuous solid solution between them ($Na_xK_{(1-x)}AlSiO_4$); it is an alternative mineral substrates for alumina production. Going on to more tightly bound alumina-silicates for aluminium extraction would increase the energy costs of the aluminium metal production significantly. The cost rises proportionally with the alkali metal-oxygen bonding energy of the minerals. The production pathway is known for aluminium extraction from many aluminium-silicate minerals, but the costs are excessive compared to the present aluminium market price. Although aluminium is very abundant on Earth, most of it is tighly bound into aluminosilicates, requiring prohibitive amounts of energy to take it out of for example granite rock. Therefore, despite making up 8% of the crust, most aluminium is unavailable for extracting the metal. Aluminium production depends on bauxite and feldspatoids that can economically be reduced to metal.

4.1.4.2 Earlier research into modelling of the global aluminium cycle

The Global Aluminium ReCycling model (GARC 2011) was developed by the International Aluminium Institute. It is a Mass Flow Analysis type of model that uses parallel mass balances that are advanced one year at a time. This way, time-dependent trajectories are created as an expansion of modified business-as-usual versions. It does not involve crosslinking between mass balances and cannot accommodate iterative feedback loops in the system. The Mass Flow Analysis models can infer price development through statistical correlations, but looped system causalities are not possible in the methodology. But for some purposes, they are sufficient and quite practical as they are easy to make.

Ramkumar (2014) made a stock-driven, trade-linked, multi-regional model of the global aluminium cycle; it is a semi-dynamic econometric model, based on a regression formula calibrated on times-series. Econometric models normally use statistical relationships instead of causalities and the use of feedbacks is very limited or not existent. Econometric models are unable to generate commodity prices from causalities, and can only predict system behaviour that has been previously observed. Since econometric models operate on statistical correlations, the relationships do not distinguish between correlation and causation, and may at times represent spurious connections.

Many researchers used Material Flow Analysis modelling for metals, including Bangs (2011), Chen and Graedel (2012a,b), Chen and Shi 2012, Ciacci et al., (2013), Gang et al., (2013), Hatayama et al. (2007), Liu and Müller (2013a,b,c) and Müller et al. (2014). Mass Flow Analysis models are simplified and normally linearized models, and can answer relatively simple questions efficiently. However, if we are asking questions related to causalities, non-linearities and feedback effects, they are not a sufficient tool for future scenarios.

Recycling and flows were mapped by Hatayama et al. (2009), Liu and Müller (2013a,b,c), Modaresi et al, (2014), Rauch (2009), Rauch and Pacyna (2009), Løviket al., (2014), UNEP (2010, 2011a,b, 2013a,b), McMillan et al., (2010), Wang and Graedel (2010). They present snapshots of mass flows and some considerations on how they may change, but these efforts do not model any systems dynamics in the global aluminium system. They are however very important for validation of the dynamic models as they describe past record of flows and stocks and record what happened in the past. Hubbert's model was used by Roper 2009, Ragnarsdottir et al., (2011), Sverdrup et al., (2013), and Sverdrup and Ragnarsdottir (2014). Hubbert's model is empirically based and does not include any defined feedbacks in any way. Hubbert's model is a very simplified model, and can answer simple questions of production in a business as usual scenario.

Several features of the aluminium system cannot be investigated unless we use models that incorporate feedbacks in the model formulations (Haraldsson and Sverdrup 2004, Sverdrup et al. 2014a,b, Sverdrup and Ragnarsdottir 2014a,b). Important in the global cycling of major commodity are the factors involved in market dynamics, such as the connection between market price, trade market stocks, and the effects of price on supply and demand. These cannot be considered without having a whole system of feedbacks, which leads to the need for systems dynamics types of models.

4.1.4.3 Scope and objectives

The scope of this study is to create an integrated systems dynamics model for the global aluminium cycle, based on best estimates on available extractable aluminium reserves, primary production rates, market supply rates and the global market price. Our objective was to generate the aluminium price inside the model from market fundamentals and a description of the price formation process, and to include market mechanisms and the dynamic effect on supply and demand. Feedbacks play an important role in our model, as it needs to be able to address what happens in the global aluminium cycle system when the reserves run low or empty, and mining decreases. The ALUMINIUM systems dynamics model presented in this paper assesses the sufficiency and sustainability of the global aluminium supply.

4.1.4.4 Input data and reserve estimates in particular.

The reserves were estimated using data from the published literature; one important source is the US Geological Survey's resources mapping programme. Data and information, qualitative and quantitative was gathered from a number of sources and earlier studies where it is readily available in open sources (Osborn 1948, Turner 1990, Fitzgerald et al., 1990, Smirnov and Tikhonov 1991, Smirnov 1992, 1996, Allen and Behamanesh 1994, Gordon 1996, Stockwell 1999, Norgate and Rankin 2000, 2002, Heinberg 2001, Dahlström et al., 2004, Bardi 2005, 2007, 2008, 2009a,b, 2013, Gordon et al. 2006, Cohen 2007, Johnson et al. 2007, Rauch and Graedel 2007, Tran 2007, Wang et al. 2007, Bardi and Pagani 2008, Radetzki 2008, 2012, Brewster 2009, Geoscience Australia, 2009, Rauch and Pacyna 2009, Roper 2009, Rauch 2009, MinEx Consultants 2010, Morrigan 2010, McMillan et al. 2010, Wang and Graedel 2010, Crowson 2011, Fischer-Kowalski et al. 2011, Bangs 2011, GARC 2011, Graedel et al. 2011, Ragnarsdottir et al. 2011, Chen and Graedel 2012, Cullen et al. 2012, ICMM 2012, Norsk Hydro 2012, Wübbeke 2012, Gang et al. 2013, Liu et al. 2011, 2013, Liu and Müller 2013a,b, Liu et al. 2013, Ciacci et al. 2013, Campbell 2013, Løvik et al. 2014, Müller et al. 2014, Nuss et al. 2014, International Energy Agency 2014, Ramkumar 2014, Reck and Graedel 2012, Wells 2014, Stanway 2014, Sverdrup et al. 2013b, UNCTAD 2000, UNEP 2011a,b,c, 2013a,b,c, USGS 2005, 2007, 2008, 2013, United Nations 2003, World Aluminium Association 2012, 2014). We assess the ultimately recoverable aluminium reserves (URR), as the presently extractable metal amounts along with potential resources that may become reserves at increased prices; we also assess the division between presently known extractable amounts and estimated as hidden extractable amounts, by reviewing the available scientific literature and from corporate information (Heinberg 2001, Rauch and Pacyna 2009, Rauch 2009, Singer 2010, 2011, 2013). The estimates of URR vary significantly with time, and depends on many factors. The extractable amounts may be adjusted based on renewed assessment of how recoverable a resource is, as well as be reassessed based on purely political aspects. We have defined extractable amounts as a resource that is extractable physically, provided the extraction price can be paid. Resources are deposits with aluminium contents that could perhaps in theory be extracted, but where the extractability has not been assessed. The reasons for this may be several, often because they are far outside technical reach, that they are very expensive to extract or somehow blocked from exploitation.

Nepheline is an alternative source for aluminium to bauxite. The known and anticipated global nepheline reserves are considerably smaller than the bauxite reserves (Tables 4.1.4.1-2). In this study URR is distributed between the quality classifications high grade, low grade and nepheline ores. As the aluminium extraction efficiency with declining ore grade ultimately goes down, URR will converge on a final limit with time (Sverdrup et al. 2014b). When the ore grade goes down, we will have to handle larger and larger amounts of rock and the mining cost will go up (Sverdrup et al., 2014b, Mudd 2012, Tilton 2002, 2007, 2009, 2012).

Table 4.1.4.1. Distribution of the world's recoverable amounts of aluminium ore in tons, the ore grades used in the ALUMINIUM model simulations. These are the extractable amounts as estimated for the year 1900 before the large expansion in aluminium production began. Amounts are in million ton aluminium.

Type	Known amounts	Hidden amount	Total amounts	Detected bauxite, but considered as unavailable resources for mining
High Quality	300	9,700	7,000	8,400
Low Quality	400	9,600	10,000	12,500
Nepheline and other minerals	200	3,000	3,200	25,100
Sum	900	22,300	23,200	46,000

Table 4.1.4.2. All known aluminium extractable amounts make up 16.2 billion ton, dug up aluminium is 1.9 billion ton, giving an URR of about 18.1 billion ton expressed as aluminium metal in this estimate.

Country	Reserves fully exploitable	Resources, unassessed access to extraction	Sum of all	Country	Amount fully extractable	Potential resources, unassessed access to extraction	Sum of all
	billion ton aluminium				billion ton aluminium		
Vietnam	11.000	5.400	16.400	Greece	0.600	0.650	1.250
Guinea	7.400	8.600	16.000	Suriname	0.600	0.600	1.200
Australia	6.200	7.900	14.100	Venezuela	0.320	0.350	0.680
Brazil	3.600	2.500	6.100	Russia	0.200	0.250	0.450
Jamaica	2.000	2.500	4.500	Sierra Leone	0.180	0.250	0.430
India	0.900	1.400	2.300	Kazakstan	0.160	0.450	0.610
Guyana	0.850	0.900	1.750	Others	3.300	3.800	7.100
China	0.830	2.300	3.130	Sum	34.600	38.000	72.600
Aluminium					16.200	17.900	34.100

Here we estimate URR as the sum of the presently recoverable amounts plus what has been mined to present (Table 4.1.4.3). Table 4.1.4.1 shows the distribution of aluminium ore expressed as aluminium content, distributed among known reserve qualities and estimated hidden amounts in ton. URR has a range from 15 billion ton aluminium to as much as 46 billion ton aluminium depending on the interpretation of the data. Mining costs are strongly connected to energy prices as expressed by for example the oil price. The lower the ore grade, the less metal is recoverable from the metal ore deposits in terms of yield. This puts an upper limit on the possible operational size of the URR (Lenzen 2008, Prior et al. 2013, Sverdrup et al. 2014a).

Table 4.1.4.3. Different estimates of the total aluminium resources as a basis for mining of alumina according to the literature listed in the text. The reserve base is available as extractable, but only with higher extraction costs and more mining waste.

Source	Million ton expressed as aluminium					
	Known reserves bauxite	Known reserves aluminium content	Reserve base bauxite	Reserve base aluminium metal	Aluminium metal dug up to date	URR
Roper (2009)	-	-	40,000	18,800	1,900	20,700
USGS (2011)	29,000	13,700	44,000	20,700	1,900	22,600
Rauch (2009)	-	16,000	34,040	-	1,900	17,900
Norsk Hydro (2012)	29,000	13,700	65,000	30,550	1,800	46,050
Averages	29,000	14,470	49,900	23,350	1,900	26,813

Input for the global population size to the demand calculation was derived using the FoF-model (Ragnarsdottir et al. 2011, Sverdrup and Ragnarsdottir 2011). The FoF-model uses a model similar to the standard UN population model (UN 2003), but it has been enhanced with a food production module limited by available phosphorus supply (as a food proxy). The available ore deposit data was inspected for inconsistencies and adjustments were made when the available input data were not internally consistent. We had to make expert judgment about what are the most likely parameter values to use on some occasions. Aluminium has a specific density of 2,700 kg per m3, whereas for steel and iron this is about 7,900 kg per m3. In replacement, because of strength and density differences, 1,000 kg of steel can be replaced by approximately 500-600 kg of aluminium.

Table 4.1.4.1 shows the distribution of the world's recoverable amounts of bauxite ore in tons of aluminium, distributed to ore grades as we have set it up in the ALUMINIUM model for starting the runs. Table 4.1.4.2 lists aluminium reserves, either proven or highly probable, by country according to the literature cited above in the text.

Table 4.1.4.3 shows estimates of the global aluminium reserves by different authors at different times (Roper 2009, USGS 2011, Rauch 2009, Rauch and Pacyna 2009, Norsk Hydro 2012). A general impression given by many published studies of aluminium reserve estimates, is that there is a lot of aluminium deposits, but many of these are probably not economic or technically viable reserves, even at substantially higher aluminium prices. The stock-in-use in society is about 500-700 million ton, and we estimate that 1,800-1,900 million ton aluminium has been dug up and produced to metal. This suggests that we have retained as stock in society about 40-50% of the mined aluminium.

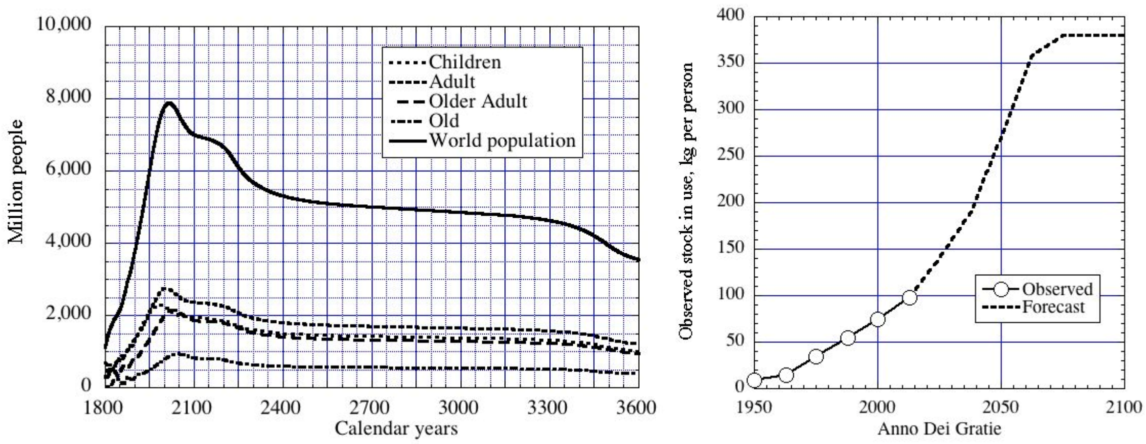
Energy plays an important role for evaluating the impact of aluminium production and for evaluating to which extent the aluminium production can be increased and whether aluminium can be a substitute for other materials like iron, steel, stainless steel, copper, bronze or zinc alloys. The energy use for iron and aluminium production in MJ per kg metal (Smirnov and Tikhonov 1991, Smirnov 1992, 1996, Norgate and Rankin 2000, 2002) is shown in Table 4.1.4.4. This data was used as input to the model simulations. Aluminium is tightly bound to oxygen in the ore and in the rocks where it occurs (Sverdrup 1990). Bauxite is one of the minerals where the extraction energy cost is the lowest. Working from recycled aluminium uses 5-8% of the energy for making aluminium from bauxite ore to metal. The cost of recycling metal from metallic scrap is similar for iron and aluminium, but as soon as the material is derived from ore, the difference is large. The energy cost of making steel from ore is only 7-12% of that of making aluminium from bauxite ore, and this explains why aluminium is more expensive than iron.

Table 4.1.4.4. Energy need for different metal production pathways, depending on starting raw material and final metal product. 1 MJ =2.388 *10-5 ton oil equivalents.

Metal made	Pathway from	Energy need MJ/kg	Multiple of the least energy demanding step
Aluminium	Recycled aluminium scrap	11-17	1
	High quality bauxite	227-300	20
	Low quality bauxite	250-342	25
	Nepheline rock	250-350	30
	Muscovite or feldspar rock	500-1,400	65
Steel	Recycled steel scrap	6-15	1
	Iron scrap	20-40	3
	Iron ore plus alloy metals	40-100	7-10
Iron	Iron ore	20-25	3
	Bog ore	100-150	16

Figure 4.1.4.2a shows the population from the FoF-model (Ragnarsdottir et al. 2011, Sverdrup and Ragnarsdottir 2011), used as input to set world market aluminium demand. The population size goes through a maximum this century and then declines, caused by phosphorus shortages. Our model stricter than the United Nations population model, which assumes that there will never be any resource limitations affecting population growth (birth rate, mortality rate). The FoF-model is less strict than the Limits to Growth model population outputs (Meadows et al. 1972, 1992, 2005), which related population size to bulk available resources in general. The aluminium demand is driven mainly by population size and the affluency. Higher price acts as a brake on demand. Figure 4.1.4.2b shows the stock in use per person in the world, expressed as kg per person. Here we give stocks in use in different countries are for 2013, with the saturation level and time of saturation in brackets: USA 540 kg/person (600 kg/person, 2020), Netherlands 500 kg/person (540 kg/person, 2020), Germany 410 kg/person (540 kg/person, 2023), Australia 370 kg/person (460 kg/person 2023), Japan 320 kg/person (350 kg/person, 2018), France 240 kg/person (380 kg/person, 2030), Great Britain 220 kg/person (380 kg/person 2030), China 55 kg/person (380 kg/person, 2035), India 7 kg/person (250 kg/person, 2080). At a stock-in-use level above 400 kg aluminium per person, the demand seems to stagnate towards the maintenance supply for infrastructure and short-term consumption (Figure 4.1.4.2b). The current global average stock is estimated at about 90-121 kg/person.

Figure 4.1.4.2. (a) Earth population as a function of time. The population was derived from a FoF-model run (Ragnarsdottir et al. 2011, Sverdrup and Ragnarsdottir 2011) and used for input to the ALUMINIUM model. Note that this differs from the standard UN estimates that for political reasons do not allow any food restrictions to populations growth. (b) Aluminium stock in use per person in the world, expressed as kg per person. This was used to generate the demand used as input data to the ALUMINIUM model.



a b

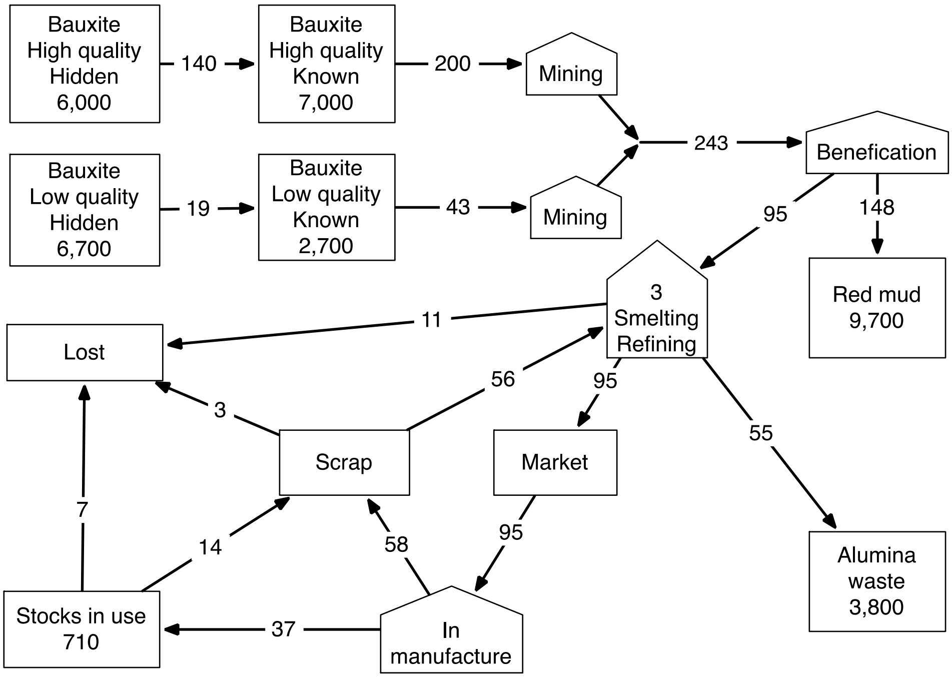
Source: (a) Ragnarsdottir et al. 2011. (b) Own Figure, Lund University/University of Iceland

4.1.4.5 The ALUMINIUM model

Figure 4.1.4.3 shows a simplified flow chart for aluminium for 2000. The numbers adopted using data from Rauch and Pacyna (2009), Rauch (2009) and others and fitted to the structure of the ALUMINIUM model. The ALUMINIUM model is based on mass balance expressed differential equations, and solved numerically in STELLA with a 4-step Runge-Kutta method, using a 0.02 year time-step in the integration from 1900 to 2400. The ALUMINIUM model a number of stocks and flows:

1. Extractable amounts aluminium
    a. High quality ore expressed as aluminium (low silica content)
        i. Hidden
        ii. Known
    b. Low quality ore expressed as aluminium (higher silica content, some contaminants)
        i. Hidden
        ii. Known
2. In society
    a. aluminium trade market stock
    b. stock-in-use aluminium in society
    c. scrapped aluminium not yet lost or recycled

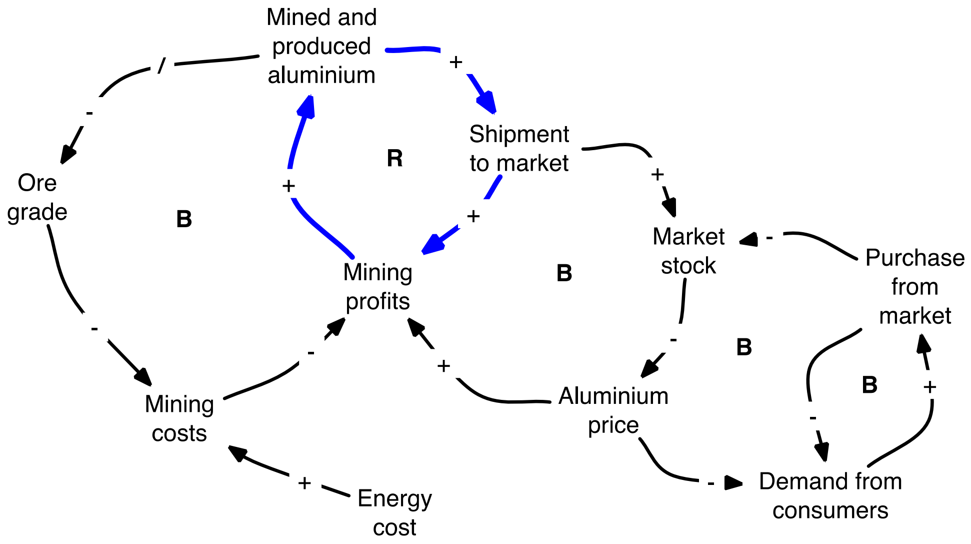
Figure 4.1.4.3. Flow chart for the aluminium system in 2000. The numbers are stocks in million ton aluminium (boxes), and flows in million ton material (bauxite, alumina, waste, after refining: aluminium except for alumina waste) per year (arrows). This structure is mirrored in the STELLA version of the model.



Source: Own Figure, Lund University/University of Iceland

The basic driving mechanism of mining comes from profits and availability of a mineable reserve used in the model, but affected by the mining cost and how that is modified with capital costs, oil price and ore grade. A lower ore grade implies that more rock must be moved to mine the aluminium. The implication is that a higher aluminium price is necessary to keep the aluminium production up. The price must stay above the production costs and is set by the amount in the trade market. The amount in the trade market depends on the balance between deliveries into the market from production and the shipments from the market, in response to world market demand. The causal loop diagram in Figure 4.1.4.4 shows how the mining operation is driven by profit. This profit is driven by the aluminium price and aluminium amount extracted, but balanced by the cost of operation. The cost of operation is mainly determined by two important factors beside cost of investments, the energy price ad the ore grade. The price in the model is set by the amount in the market and the fact that it must stay above the production costs. It depends on the balance between deliveries into the market from production and the shipments from the market in response to world market demand. The price also drives the urge for recycling of aluminium stock from society. The demand is driven mainly by population size and the affluence of society. Higher price acts as a brake on demand.

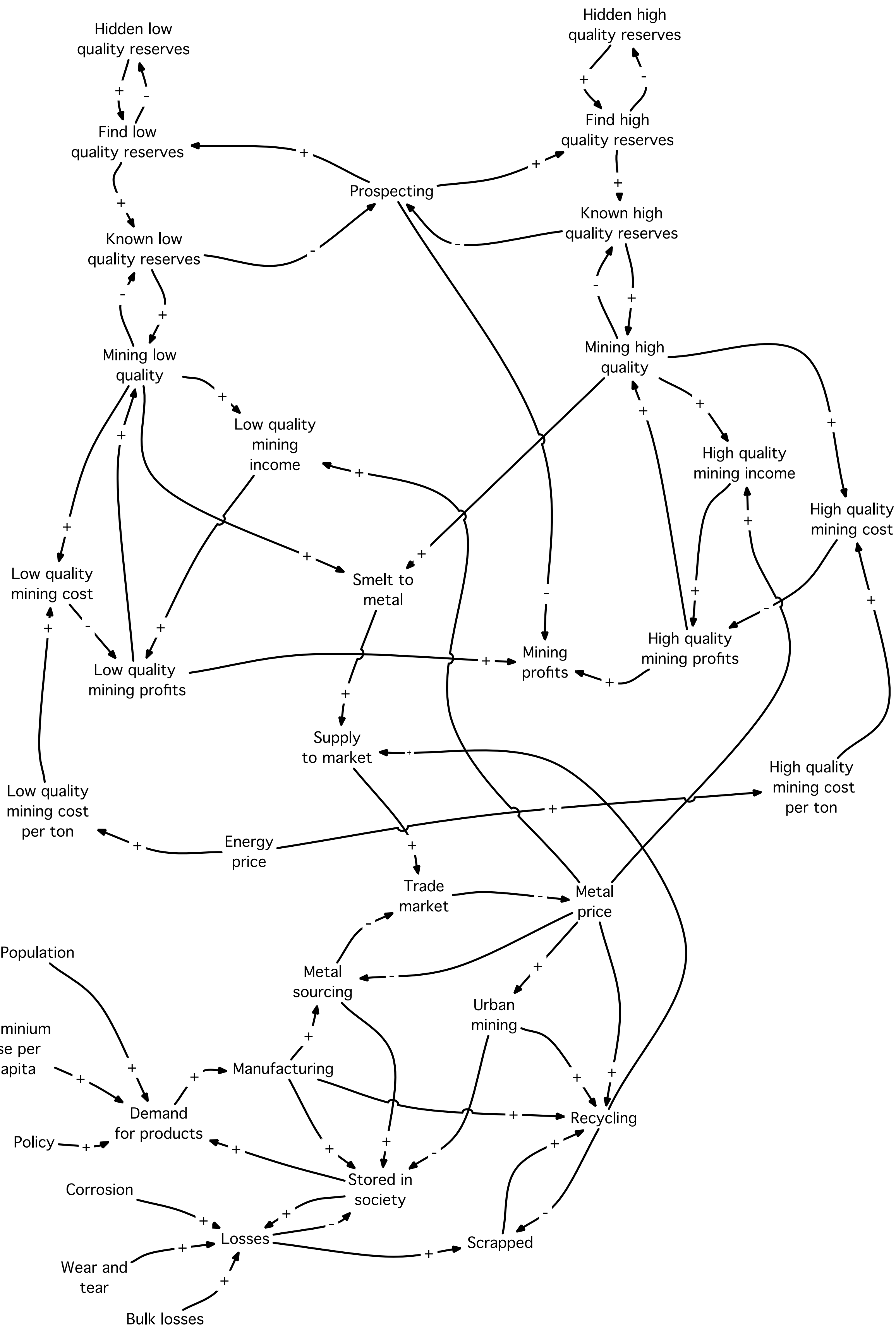
Figure 4.1.4.4. Causal loop diagram of the price model used in the ALUMINIUM model. The model has a core reinforcing part, market with an R. It is surrounded by two coupled balancing loops, marked with B. To one side, the balancing loop has two coupled balancing back-loops. Mining is profit driven.



Source: Own Figure, Lund University/University of Iceland

The pricing mechanism in the model has been adjusted to how the aluminium market has worked in the past and how it has recently changed. Metal trading is supposed to operate as follows: The traders come to the trading floor with their lots to sell or to buy, and adjusts their sales or purchase amounts as the price goes up and down. If demand is higher than production, the price goes up; in the opposite case the price is moved down. This is a self-adjusting mechanism that balances the trade by adjusting the prices until the demand to buy an amount at a price match the offers to sell an amount at a price. The buyer offer to purchase more at a lower price or less at a higher price, and the sellers offer to sell less at a lower price or more at a higher price. When the price and amount match, the price is set. This is based on personal observation on the trading floors at the metal markets in New York and London by the authors. Figure 4.1.4.5 shows the ALUMINIUM model as a causal loop diagram for the whole world aluminium system. In the model, the aluminium ore is divided into two qualities: one high quality bauxite grade with low content of silica as an impurity in the alumina; and the other one low quality with higher silica content, making it more expensive to reduce to aluminium. It consists of low grade bauxite and other types of low grade alumina and nepheline.

Figure 4.1.4.5. The aluminium model shown as a causal loop diagram for the whole world aluminium system.



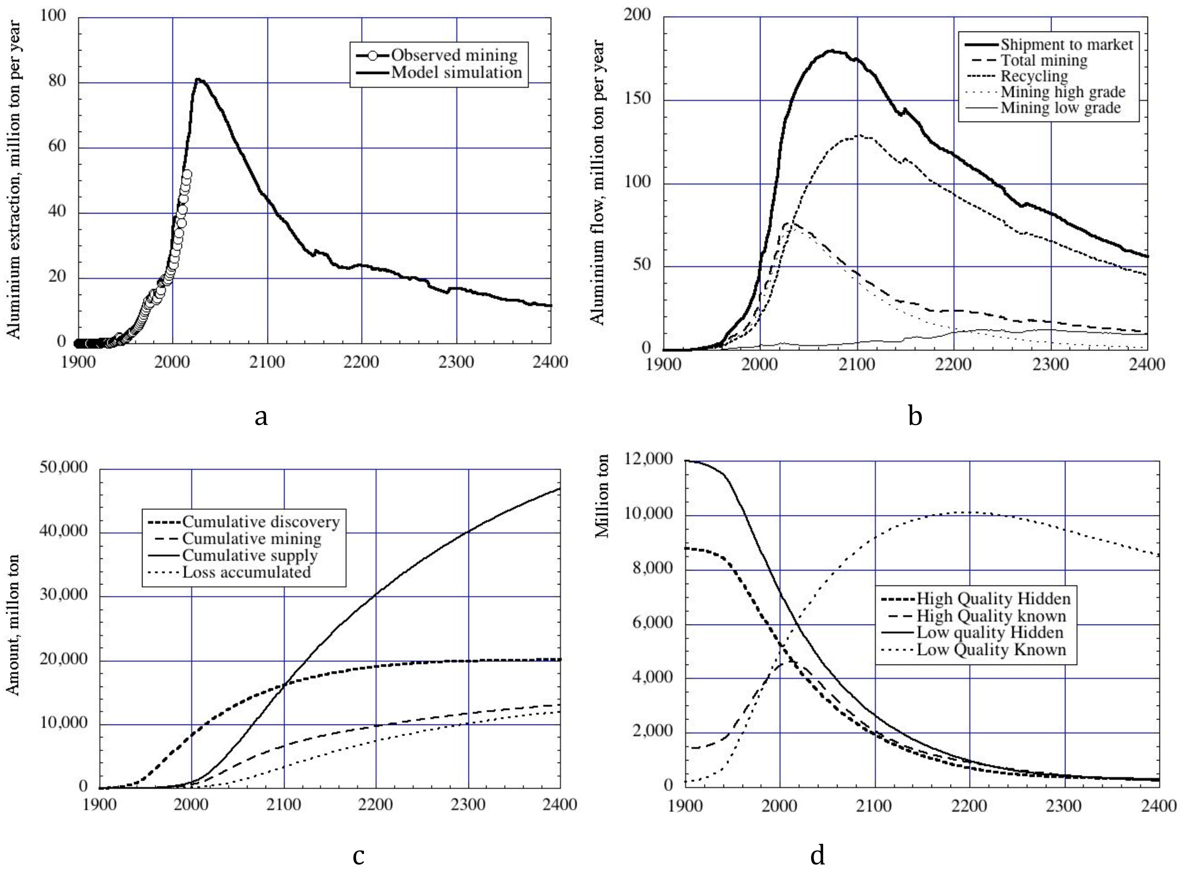
Source: Own Figure, Lund University/University of Iceland

4.1.4.6 Results

Outputs from the ALUMINIUM model

Figure 4.1.4.6 shows the past and future aluminium mining rate, expressed as aluminium primary production from bauxite ore, and supply to the market. The dots represent the observed primary production rate ("aluminium mining"). Figure 4.1.4.6b shows the primary production as compared the amount recycled from society and to the total supply to markets. Figure 4.1.4.6c shows cumulative discovery or extractable amounts, cumulative mining, cumulative supply to society and cumulative losses. Note that the supply to society is considerably larger than the amount mined, because of significant recycling. Figure 4.1.4.6d shows the mining from high and low quality reserves, low quality is prioritized down as long as good quality ore is available. Note that after about 2020, recycling is predicted to become the most important source of aluminium. That will be the time of scrap aluminium.

Figure 4.1.4.6. Diagram (a) shows the total mining and compares observed mining (open circles) and ALUMINIUM model output. Diagram (b) gives the total mining amount, the total supply into the market, and how much comes from recycling. Diagram (c) shows cumulative amounts mined, cumulative supply, cumulative discovered and cumulative losses, expressed as million ton aluminium content. Diagram (d) shows the reserves in the ground, high quality hidden and known, low quality hidden and known and how they develop over time. Amounts are million ton aluminium.

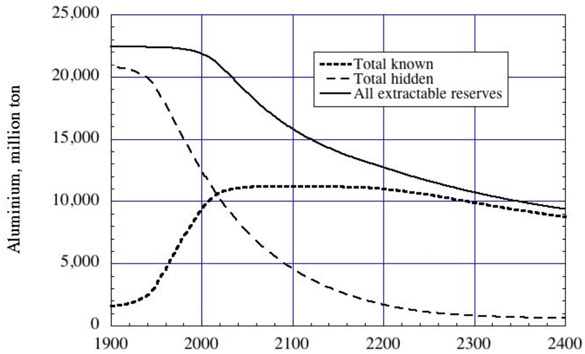


Source: Own Figure, Lund University/University of Iceland

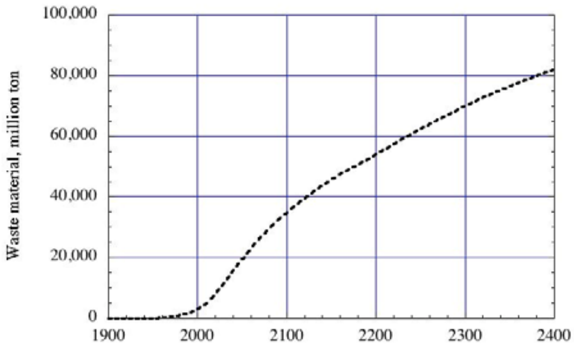
Figure 4.1.4.7a shows an overview over the development of the reserves of aluminium in bauxite, known extractable amounts, hidden extractable amounts and all extractable amounts. Figure 4.1.4.7b shows the cumulative amount of red mud produced from 1900 to 2400, expressed in million ton. Red Mud is a waste by-product of alumina production. Red mud is at present either dumped in nature

(10%), stored in ponds (30%), and stored as dewatered dry piles (60%) that can be reclaimed as land. Normally, bauxite ore contains about 20% aluminium. About 50% of the alumina weight will be converted to aluminium metal in the smelter. Thus, to produce 50 million ton aluminium per year, about 250 million ton of bauxite per year is needed, and 200 million ton wet or solid waste is created (Figure 4.1.4.7b).

Figure 4.1.4.7. Diagram (a) shows total known aluminium in the total known and hidden extractable amounts. All amounts are million ton aluminium metal. Diagram (b) shows the cumulative amount of red mud produced from 1900 to 2400, expressed in million ton red mud waste material.



a

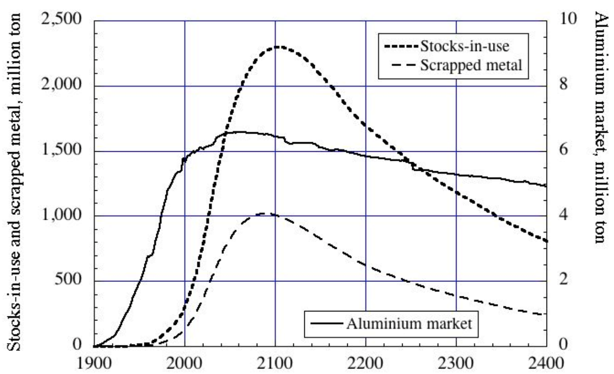


b

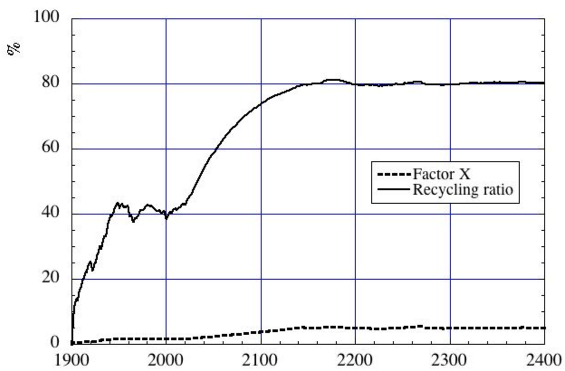
Source: Own Figure, Lund University/University of Iceland

Figure 4.1.4.8a shows the trade market size, the amount scrapped and the amount in the trade market. Figure 4.1.4.8b shows the ratio of recycled material to the flux into the market. The large stock in society allows for a high degree of recycling, even after the mines become exhausted. After 2050, more than half the aluminium metal supplied to society will come from recycled metal (Figure 4.1.4.8b). Overall, today a much smaller proportion is recycled aluminium as we are still building up the stock-in-use, making the recycled fraction appear as low. The recycling rate is frequently urged by state authorities to do more, helping to keep the recycling higher than what just money and market mechanisms alone can do. Such campaigns are taking place in many countries at present. Factor X which is defined as the ratio of the internal flux to the net systems input. Factor X has a value of about 2 to 2.5 until about 2020, when it slowly increases to reach a maximum value of 5 around by 2250.

Figure 4.1.4.8. Diagram (a) shows the ALUMINIUM model outputs for stock-in-use, scrapped metal stock and aluminium market size. Diagram (b) shows the factor X, and the ratio of recycled material to the flux into the market.



a

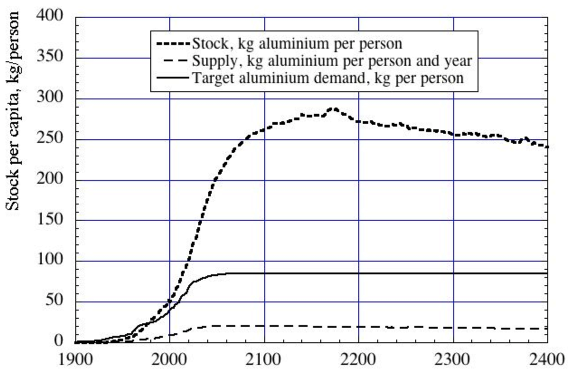


b

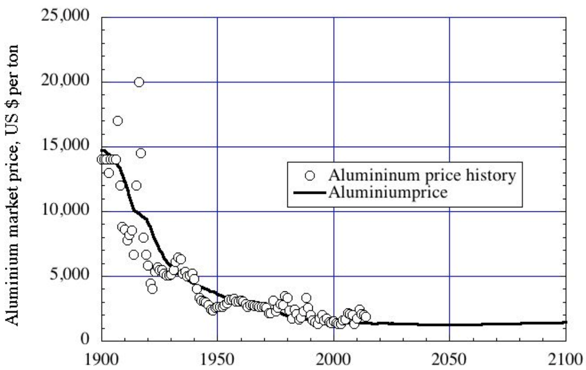
Source: Own Figure, Lund University/University of Iceland

Figure 4.1.4.9a shows the stock-per-person, the supply per person in kg per person and year and the target aluminium demand. The simulation shows that stock per person and supply per person will flatten out and stay constant from about 2045 to 2400. Outputs from the ALUMINIUM model (Figure 4.1.4.9a) suggest that aluminium production will not grow forever, but it will stop at a saturation level of 280 kg/person, a supply of 16 kg per person per year. For comparison, the level of Germany is 410 kg/person, Britain is 220 kg per person and France is 260 kg per person. After 2100, the aluminium market reaches a high level and remains there for a substantial time into the future (UNEP 2010, 2011a,b, 2013a,b). Rauch (2009) mapped stocks-in-use for aluminium, iron, copper and zinc and related this to GDP for some countries. He found that the stock in use was linearly proportional to the country's GDP. Stock of aluminium in use was equal to 13.5*106*GDP. This suggests that the decoupling presented by Fischer-Kowalski et al. (2011) is in reality not taking place for aluminium, and when all externalities are taken into account and have been brought into the model, thene there is no decoupling of primary value production with respect to natural resources. Figure 4.1.4.9b shows the simulated world market aluminium price as compared with the observed data. We get r2=0.81 which is very good, and confirms that we can make valid use of the price predictions inside the model to drive dynamic market feedbacks on demand, supply and production.

Figure 4.1.4.9. Diagram (a) shows the stock-per-person simulated by the ALUMINIUM model, along with the supply per person and the target demand in kg per person. Diagram (b) shows the ALUMINIUM simulation of the aluminium price with the observed data added in (r2=0.81).



a

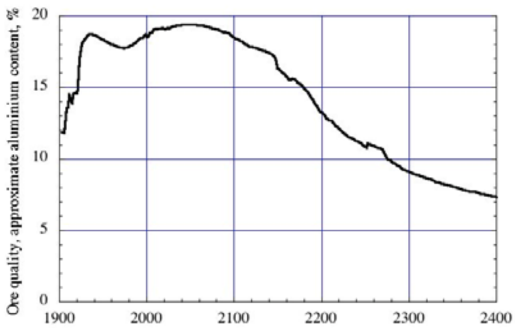


b

Source: Own Figure, Lund University/University of Iceland

The ALUMINIUM model operates with two ore qualities and the diagram shows the development of average ore quality with time. The best ore is mined first, and this causes the ore quality to go down with time. The high quality ore contains about 20% aluminium, we have set the low quality to have an average content of 5%. We can see in Figure 4.1.4.10 that after 2100, the ore quality will start to decrease. This is an indicator for increased cost of production and increased world market price for aluminium. At some point it may become more cost-competitive to mine the large stock-in-society.

Figure 4.1.4.10. Simulated development of ore quality with time



Source: Own Figure, Lund University/University of Iceland

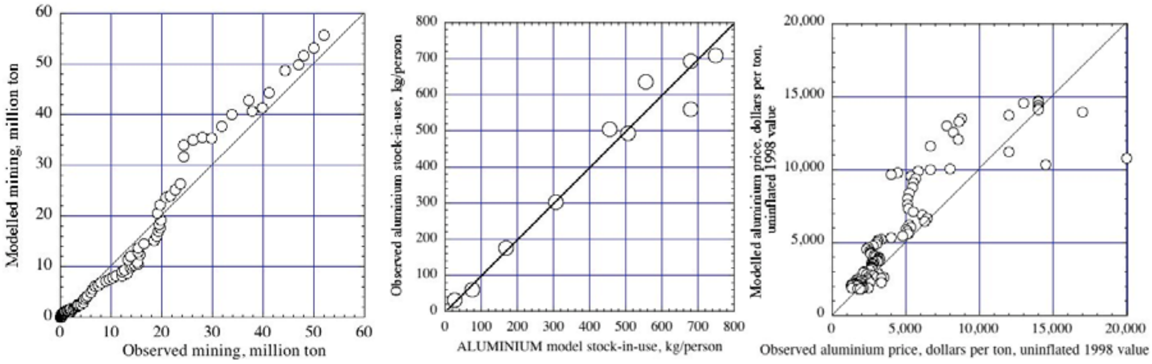
Model field test on data

We have tested the model on data from the past (1900-2015), in order to assess whether the model has a reasonable performance. Since the ALUMINIUM model reliably reconstructs the observations of the past, then it can be used with confidence for future predictions (2015-2400) and for scenarios. We have tested the ALUMINIUM model performance on several aspects:

1. Mining rate with time as recorded 1910-2015 (USGS 2014) (Figures 4.1.4.1, 4.1.4.6a and 4.1.4.11a)
2. The estimated stock in society (Table 4.1.4.5 and Figure 4.1.4.11b)
3. The development of the world market aluminium price (USGS 2014) (Figures 4.1.4.9b and 4.1.4.11c)
4. The recycling rate as a percentage of total supply to the market (Figure 4.1.4.8b and Table 4.1.4.5)
5. The known extractable reserve at present (Table 4.1.4.5)

For the estimated stock in society we use data by UNEP (2010, 2011a,b, 2013a,b), Rauch and Pacyna (2009), and Rauch (2009), (Table 4.1.4.5). Figure 4.1.4.11a shows a comparison of observed and simulated aluminium mining rates. The correlation coefficient is r2=0.97, it is hard to do better. The production curve shows no sharp bends and the test is not difficult. Figure 4.1.4.11b shows a comparison of observed stock-in-use and stock-in-use simulated by the ALUMINIUM model. The correlation between observed stock-in-use and that simulated by the model is r2=0.96. The correlation for the price simulation (Figure 4.1.4.11c) is reasonable, considering how sensitive the price mechanism is to changes, we get r2=0.81.

Figure 4.1.4.11. Comparison of observed and simulated mining rates. Diagram (a) shows fit with data (circles) over time. All amounts are million ton aluminium. Diagram (b) Comparison of all available observed data for stock-in-use and the stock-in-use simulated by the ALUMINIUM model. Diagram (c) shows a comparison of observed value-adjusted aluminium price (circles) and the ALUMINIUM model simulated price.



a b c

Source: Own Figure, Lund University/University of Iceland

Table 4.1.4.5 shows further tests on individual points in time. For some types of aluminium products, the recycling rate can be very much higher, such as beverage cans or scrap generated internally in manufacturing industry (Liu et al. 2013a,b, Løvik et al. 2014). It is estimated that the total extractable resources are at least 55-75 billion metric tons bauxite when assumed but yet undiscovered bauxite resources are included. In recent years, prospecting has increased the reserves more rapidly than the rate of extraction: from 1995 to 2011, 2.7 billon metric tons bauxite were extracted, but in that time the reserves increased from 23 billion metric tons bauxite to 29 billion metric tons bauxite (Norsk Hydro 2012). The average of data on extractable bauxite for 2014, requires the model to start with the rights size for URR, have a reasonable distribution between known and hidden, and make reasonable predictions for discovery and mining.

Table 4.1.4.5. ALUMINIUM model tests on individual points in time for some parameters

Source of information	Total amount extractable bauxite	Stocks in use, million ton	Known reserves, million ton	Stocks in use, kg per person	Recycling ratio %
Rauch and Pacyna (2009), for year 2002		493	16,000	76	
Rauch (2009) for year 2000		504		80	
UNEP (2011) in year 2011		560		80	25
Aluminium Association, in year 2011					32
Ramkumar (2014) in year 2014		710		97	
USGS (2013) in year 2012	22,300		7,000		
Liu et al. (2013) in year 2005		636		94	
Norsk Hydro (2012) for 1950-2011	29,000	694	10,000	100	18-28
Average of data	25,650	600	11,000	88	25
ALUMINIUM model in year 2014	22,400	710	9,700	98	25-30

4.1.4.7 Discussion

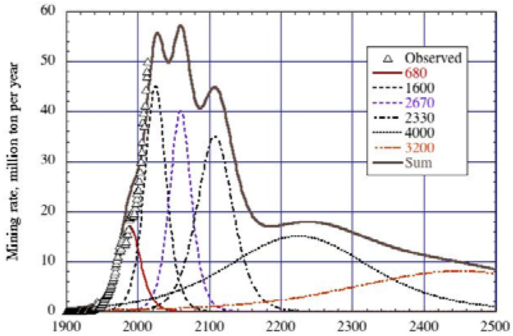
On the prospect of aluminium scarcity

If we combine the ALUMINIUM model outputs (Figure 4.1.4.6) and the Hubbert's model (Figure 4.1.4.12), they suggest peak production to occur somewhere in the period 2025-2050. The ALUMINIUM model suggests that the supply curve reaches maximum about 2080-2090. The supply to society will slowly decline after the peak, and the supply level of 2014 will be reached in the times around 2275 AD (See Figure Figure 4.1.4.6b).

Figure 4.1.4.12. Hubbert's model assessment for aluminium extraction from bauxite.

Hubbert' model parameters			
Reserve, million ton aluminium	b	PMAX, million ton per year	TPeak
680	0.100	17	1986
1,600	0.100	45	2025
2,670	0.060	40	2060
2,330	0.060	35	2108
4,000	0.015	15	2225
3,200	0.010	8	2450
14,480			

Hubbert's model diagram



Source: Own Figure, Lund University/University of Iceland

It is apparent from our study that there will be no aluminium shortage in the near future, and the aluminium price is predicted to stay stable on a relative scale for a long time. Only in the very long perspective, after 3000 AD will the bauxite reserves have been used up at the present rate of depletion and the stock-in-use been depleted by recycling and lower replacement rates. However, before that happens, the ore quality will decline (Figure 4.1.4.10). Based on field fact, the best ore qualities are mined first, causing ore quality to go down. Whether the aluminium supply is sustainable depends on this perspective on time. The reserves of bauxite are still increasing as a result of prospecting, suggesting that we are maybe 20 - 40 years from a definite peak production. Resource scarcity manifests itself gradually through four stages:

1. The first stage comes immediately when the peak has been reached, when demand still goes up, but production is flat or slowly declining. The scarcity is manifested through increased prices. There will be no material shortage, as increased prices will simultaneously decrease demand. A diagnostic indicator is that even if prices go up, production cannot be increased above the former demand (2050-2100).

2. The second stage is when the production decreases, and the price will further increase and there will be capacity limitations to supply. Demand will be further decreased by high prices, but because of limited reserves, supply will not go up because of increased prices (2100-2250).

3. Later, at an advanced stage of extractable resources decline, the ore quality also goes down, pushing production prices up (Figure 4.1.4.10, from 2130 to 2200). To make 50 million ton aluminium per year from a 5% grade ore instead of a 20% grade ore will increase the waste created from 250 million ton per year to 1,000 million ton per year. It is self-evident that that increase in waste will cost money to produce, transport and safely store.

4. The last stage is when the production has gone down significantly, and material supply is restricted, making practices of mass consumption impossible (2250-2400).

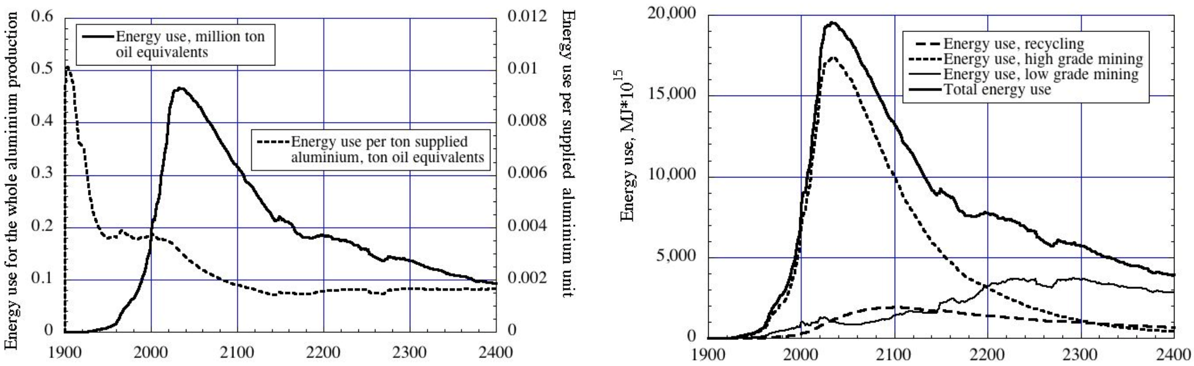
Energy, decoupling and substitution

There is another element that relates to iron and copper. Aluminium may replace these two metals in some of their applications, but far from completely. The production of iron is 1,450 million ton per year, whereas aluminium mining is about 50 million ton per year, the total aluminium supply to society is about 100 million ton per year. It can be seen that aluminium would never be able to replace all iron. Iron may become more expensive after 2100, and if aluminium should be replacing iron on a large scale, then the aluminium supply situation may change completely. If we assume for a hypothetical argument that we can replace half of all iron with aluminium, it would require 670 million ton iron to be replaced by about 300 million ton aluminium. The energy requirement would rise from 670 million x 1,000 x 22 MJ/kg = 14.5 trillion MJ to 300 million x 1,000 x 230 MJ/kg = 69 trillion MJ, which would be 4.8 times more energy. Aluminium takes up about 2.7% of the global energy expenditure today, and increasing this to 13% in a time when fossil fuels supply will go down, seems more than what would be feasible in reality.

At present, the reserves to production ratio is about 400 years for aluminium, but doubling the production to 100 million ton aluminium per year would cut that ratio to half; increasing it to 300 million ton aluminium per year would reduce the reserve to production ratio to 60 years and the production would demand 36% of the present global energy production. It is self-evident that we simply do not have the available energy for such an amount of aluminium production, nor do we have the aluminium reserves to sustain it for any length of time. Then scarcity and high prices may definitely be a future prospect even for aluminium. Copper production is about 16 million ton per year, total copper supply to the market was about 28 million ton per year in 2014 and copper will soon pass through peak production and decline after about 2025-2035 (Sverdrup et al. 2014a). If this is to be replaced with aluminium, then aluminium production must increase with 8-10 million ton aluminium per year. The result of our analysis is that the realism of substitution seems to be limited for the big volume metals. Imagine an extraction rate 5 times the aluminium extraction rate of today (everyone has as much aluminium as every American citizen, implying a global extraction of 250 million ton per year), then the aluminium reserves will have been exhausted by the year of 2200.

Figure 4.1.4.13a shows the energy demand for all aluminium produced, usinging the values shown in Table 4.1.4.4, to calculate the total energy use for aluminium production over time. Figure 4.1.4.13b shows energy use for the amount recycled aluminium, the high grade and the low grade mining contributions, and the sum of all used to supply aluminium. It can be seen that the energy use per ton aluminium decrease with time as the contribution from recycling becomes larger. Then the energy use decreases faster than the production decreases. In 2014, the energy use for aluminium production was 0.320 million ton oil equivalents per year, at the peak in 2050, it will be 0.44 million ton oil equivalents per year. In 2014, the global oil production was 3,700 million ton oil equivalents, the global energy production was 12,400 million ton oil equivalents. Aluminium production requires the energy as electricity, and the energy use for aluminium is about 50% of all hydroelectricity produced in 2014. Of note is that when hydrocarbons are used for electricity generation, but the efficiency in conversion is at best about 40%, and normally this ends up at closer to 30%. If all the aluminium were to be supplied with fossil fuels energy, it would consume about 8% of the total global energy. A significant shortage of fossil fuels predicted by Campbell (2013) after 2080-2100 may upset the aluminium production system and cause significant metal price rises, caused by both energy availability limitations and increased energy price.

Figure 4.1.4.13. The total energy use for aluminium production over time. Diagram (a) shows the total energy use in million ton oil eqivalents, and the energy use per aluminium weight unit produced. Diagram (b) shows energy use for the amount recycled aluminium, the high grade mining, the low grade mining and the sum of all energy used to supply aluminium.



a b

Source: Own Figure, Lund University/University of Iceland

Recycling

The recycling rates given by the ALUMINIUM model are about the same as what is reported in the literature. For some items like beverage cans, the recycling rate is high; for lots of other uses such as for example packaging, it is sometimes non-existent. The present observed recycling estimates are offset by the fact that much aluminium is still being built into our infrastructures, and thus will eventually be recycled, but only after a delay of many decades (Hatayama et al., 2007, 2009). The retention time in society seems to be on the order of several decades, or 30-60 years. However, aluminium in daily consumer goods like beverage cans, only have a market life in terms of weeks or months, and even if recycling in cans is 85%, that is still not good enough to prevent loosing all the aluminium in that cycle within months. For such high turnover use, an alternative is to use have even higher recycling or change material altogether.

Time, long term perspectives and sustainability

For some few people, 100 years is within living memory, and it is the minimum time horizon of professional pension fund management. For our children and grandchildren, these longer perspectives are important (Leslie 1998, Macintosh and Edward-Jones 2000). For aluminium, the delays in the system are long and scenarios that run to the year 2400 seem appropriate. We know for a fact that the principle of mass balance will be valid 400 years into the future. We know that if we think about sustainable society, then a number of basic conditions must be in place, setting our assumptions at known positions for a very long time (Tainter 1988, Leslie 1998, Diamond 2005). We assume that society will be a civilized society under rule of law and democracy. If we put aside wishes for miracles to occur, then we must assume that what we know today will be known then, and that must be sufficient to solve the situation that may come up and the problematic issues that may occur (Kennedy 1987, Diamond 2005). Some of these issues we know all too well (global chemical pollution, climate change, land degradation), and we know that they have long delays.

4.1.4.8 Conclusions

We conclude concerning the aluminium supply, stocks in use and ore reserves as follows:

1. Supply to the market will peak around year 2080 and decline after that, reaching 2014 level in about 2300. Recycling will peak around the year 2100 and decrease with the stock-in-society after that. Primary extraction from bauxite will peak in the next decade 2020-2030, unless extra effort is made to increase mining efforts. This will, however, require a higher market price for aluminium.
2. After 2030, recycling or urban mining will be the major source of aluminium. This will be the age of scrap metal, and probably provide the basis for growing many new companies and enterprises.
3. Bauxite reserves may potentially run out, but more than one century from now. The aluminium production from mining will reach a peak about year 2030 and decline begins after 2040, because of diminishing ore grades and increasing energy prices.
4. Aluminium scarcity will manifest itself as increasing metal price, and supply limits may likely come from energy limitations and not only reserves running low. The increased price will change demand and considerations for what aluminium is used for. The global society will probably run out of money and energy before there is a real lack of bauxite to make aluminium.
5. It does not seem possible to substitute a significant fraction of the iron supply with aluminium. Nor will it be possible to increase the global aluminium production very much more, without upsetting the global energy balance.

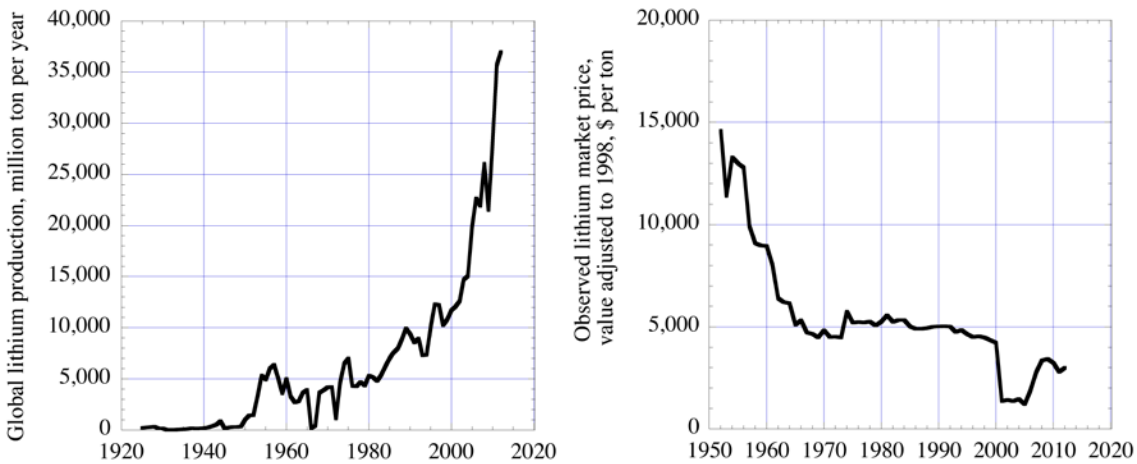
Very large bauxite or aluminium reserves are available, but they are not inexhaustible. It is doubtful that we have enough carbon-based energy available to exhaust the reserves, because the amount carbon needed for alumina reduction may be difficult to source. At a production of 250 million ton aluminium per year, the production would consume about 14% of the present total global energy production. That would amount to about 1,600 million ton oil equivalents per year as electricity. The global energy production is to about 85% from fossil fuels, and by 2100 the energy production from fossil fuels may have declined to about half (Campbell 2013). Then the energy consumption used to maintain an aluminium production at the present level will consume an ever increasing fraction of the total global energy. That would significantly upset global energy prices and transfer to the aluminium market price. Thus, we may run out of energy and money to buy it, long before we run out of aluminium metal.

4.1.5 LIGHT METALS sub-module LITHIUM

4.1.5.1 Introduction

Lithium is an important element for many new technologies, including batteries, ceramic materials and new types of alloys (Nassar et al., 2015). Especially, development of the use of lithium for high-performance batteries have recently opened a new and expanding sector in the market. Figure 4.1.5.1a shows the global lithium production to date (USGS 2015). We estimate from the curve in Figure 4.1.5.1a that about 1 million ton lithium has been extracted to 2015. The lithium demand and production has gone up significantly during the last decade, much because of increased demand from rechargeable batteries and accumulators for power-tools and battery-packs for new electric transportation vehicles (Yoda and Ishihara 1997, Tarascon and Armand 2001, Ning and Popoc 2004, Johnson et al., 2007, Tahil 2007, 2008, 2010; Tytgat et al., 2008, Angerer et al., 2009, Gruber and Medina 2010, Jacobson et al., 2011, Gruber et al., 2011, Heinberg 2011, Jacobson and Delucci 2011, Kushnir and Sanden 2012, Grosjean et al., 2012, Goonan 2012, Stamp et al., 2012, Elshakaki and Graedel 2013, Wang et al., 2014, Li et al., 2013, Gaines 2015, Speirs et al., 2014, Bradley et al., 2014, Richa et al 2014, Sonoc and Jeswiet 2014, Wang et al., 2014, Gaines and Nelson 2015). Figure 4.1.5.1b shows the lithium price for lithium carbonate or lithium hydroxide 1998 value-adjusted dollars per metric ton (USGS 2014). Most lithium is produced to lithium carbonate, lithium hydroxide or lithium chloride for further use. Only a small fraction is produced all the way to metal.

Figure 4.1.5.1. Diagram (a) shows the lithium production to date (USGS 2015) as ton pure lithium. Diagram (b) shows the lithium price for the same time period in $ per ton, in dollar value-adjusted to 1998 (USGS 2015).



a: Production b: Price

Source: Own Figure with data from USGS 2015, Lund University/University of Iceland

Much of the worlds' resources of lithium are limited to a small number of old salt-beds, associated with inland, high altitude salt-lakes or fossil remnants of such lakes (Andes and Rocky Mountains, Tibet Plateau, Anatolian plateau are examples). It is also associated with some lithium-bearing minerals in pegmatite formations (Keseler et al., 2012, King 2015), located on the main continental shields. The main commercial sources of lithium extraction used today or one additional source possible in the future are (Mohr et al., 2012):

1. Brines from salt beds of former or existing salt lakes (Bolivia, Peru, Chile, Argentina, in California, Nevada and Utah, but also in Tibet, China, and in the deserts of Turkey and Iran) (Ober 1994, 2002, Helvaci et al., 2003, Aguilar-Fernandez 2009, Keseler et al., 2012, Munk 2011, Nickless et al., 2014, Tan et al. 2012). There are also deep sea brines, geothermal brines, oil field brines and some lithium in underground rock salt deposits. Contents are very variable between and within

different brines, the contents are typically 0.035-0.3%, but in small zones it may reach 1%. High magnesium content is a problem because it interferes with the extraction process.

2. Lithium-containing minerals where spodumene (Li2Al2SiO6), amblygonite (LiAlPO4F) and lepidolite (K2Li3F3Si7O20) are the most important minerals. Spodumene has 8% weight of lithium in the pure mineral, it is the industrially most important mineral source (Canada, Finland, Sweden, Australia, Russia). When lower grades are considered, a range of minerals would serve as sources for lithium. In lower grade ores, larger lithium resources are known (Kunaz 1994, Eilu 2011, Keseler et al., 2012, Vladimirov et al., 2012, Luong et al., 2013, Nickless et al., 2014, King 2015, Outotec 2015). Typical lithium rock deposits contain 0.5-2% lithium in the rock matrix, this is concentrated up to 2-4% in the enrichment process. The rest of the rock is minerals with no lithium content.

3. Clays that contain lithium. The possibly largest lithium-bearing clay deposits are not yet being exploited on any scale. The clays subject to mining mostly originate from weathered lithium-bearing rock. (Hectorite, smectite, montmorillonite, illite; Starkey 1982, Lien 1985, Crocker et al., 1988, Siame 2011, Keseler et al 2012, Sonoc and Jeswiet 2014, Nickless et al., 2014, Outotec 2015). The clays contain 0.2-0.35% lithium by weight.

Salt brines (saline lakes in the Andes, Turkey and Tibet) are the main source today (50%), but extraction from minerals is also significant (40%), whereas extraction from clays is modest (10%) (Mohr et al., 2012). The salt brine obtained from the saline lakes is leached selectively for lithium in several steps, and in the process magnesium, sodium and potassium salts are produced in addition to lithium hydroxide or carbonate. In the lithium mines of South America, boric acid is often also recovered. For pegmatitic minerals (Bedrock resources), the procedure is different. The mined mineral is crushed and treated with flotation to produce concentrate. The concentrate is heated in a calcination kiln to 1,100° C, making it more reactive to sulphuric acid. A mixture of finely ground calcinated spodumene mineral and sulphuric acid is heated to 250° C, reacting to form lithium sulphate. The lithium sulphate is dissolved in water. The lithium sulphate solution is reacted with soda ash, precipitating insoluble lithium carbonate from the solution. The carbonate is separated and dried for sale or use in the production of other lithium compounds (Colton 1957, Lien 1985, SGS 2010, Swanson 2012). From clays, the lithium is selectively leached using acid or organic solvents, the process requires many steps and is demanding to operate (Crocker et al., 1988).

Extraction from ocean seawater, using evaporation ponds and selective extraction has been discussed, but so far, no project has been realized in any significant scale. The energy demands for such methods are very large, and they are only feasible if energy is very cheap. At the point of writing, energy is cheap because of political processes going on in the Middle East. There is no certainty that cheap oil will persist.

Uses of lithium are in 2010 (Baylis 2010, Walker 2011, Polinares 2012, Fox-Davies 2013, USGS 2014): ceramics and glass; 29%, batteries; 27%, other unspecified uses; 16%, greases and lubricants; 12%, castings and light weight specialty aluminium and magnesium alloys; 5%, air conditioning, cooling media; 4%, polymers; 4%, aluminium production; 3%, drugs; 2%.

4.1.5.2 Objectives and scope

Our goal was develop a simple model for the global lithium cycle in order to assess the sustainability of lithium use in new technologies. Special focus will be paid to production of rechargeable batteries for a future electric vehicle production, and to different strategies for recycling and the sensitivity of the supply to the size of the extractable amounts of lithium from different sources.

4.1.5.3 Earlier modelling work

Ziemann et al., (2010, 2012) is in the process of developing a mass flow model for lithium, where simple linear mass balances are rolled forward one year at a time. Angerer et al., (2009) modelled the global supply, assuming 50 and 85% of all cars to be electric vehicles in the future. Carles (2010) modelled global lithium using a stock-and-flow model. Mohr et al., (2012) developed a model for the global lithium supply, based on modified asymmetric version of Hubbert's type of model, and used it for supply sustainability assessments for the period 1900-2200 AD, looking at three different scenarios for URR. In earlier studies, the authors have developed system dynamics models for a number of metals: gold (GOLD: Sverdrup et al., 2002), rare earths (Kifle et al. 2012), copper (COPPER: Sverdrup et al. 2014b), silver (SILVER: Sverdrup et al. 2014a), aluminium (ALUMINIUM: Sverdrup et al. 2015a), iron, manganese, chromium and nickel (IRON, STEEL, Sverdrup et al 2015b,c) and platinum group metals (PGM Sverdrup et al. 2015d).

The models reported in earlier publications are not true dynamic models and lack the feedback structure expected from a system dynamics model. The earlier models do not include any effect of price on demand, supply or and recycling. The present models are fully integrated dynamics models, overcoming many of the weaknesses of earlier models. In the present model and the earlier models of the authors, one of the major advances were the development of a reality-based metal price model, allowing for price estimation from market fundamentals inside the models, without external forcing functions or calibration, and with feedbacks on mining, demand and recycling.

4.1.5.4 Model description

The LITHIUM model is numerically integrated from 1900-2400 using a 1/18 year time-step in a 4-step Runge-Kutta numerical method of integration in the STELLA® modelling software. In addition to this the general mass balance principle applies with all inputs being balanced by the outputs and the accumulation in society:

$$Production + recycling = accumulatin\ in\ society + losses + recycling \qquad (4.1.5.1)$$

Recycling is present on both sides, and this is how recycling helps keep the flow through society up even when primary production declines. The mining activity is mainly profit driven as the main causal factor, but affected by the mining cost and how that is modified with oil price and ore grade. A lower ore grade implies that more rock must be moved to mine the iron. The implication is that a higher iron price is necessary to keep the iron production up. This implies that the core of the model is based on coupled non-linear differential equations, all based on mass balance.

The mining rate was modelled using the following equation (Levenspiel 1980, Sverdrup and Bjerle 1982, Sverdrup 1983, Sverdrup and Warfvinge 1988):

$$r_{Mining} = k_{\ Mining} * m_{Known}^{n} \qquad (4.1.5.2)$$

where rMining is the rate of mining, kMining is the rate coefficient and mKnown is the mass of the known ore body. n is the mining exponent discussed above. The basic mining rate was set at 6% of the known reserve. The mining coefficient is composed of several variables:

$$k_{Mining} = k_0 * f(T) * g(P) * j(Y) \qquad (4.1.5.3)$$

where fT) is a technology improvement function dependent on time, g(P) is a feed-back from price on the prospecting rate. j(Y) is a rate adjustment factor to account for differences in extraction yield when the ore grade decreases. This becomes important at very low ore grades. These functions are given exogeneously to the model. Where the base mining rate coefficient was set at k0=0.001 and n=1.0. However, it appears from a sensitivity analysis that the results are not very sensitive to the value of n. (See Figure 4.1.5.5 for the value of the price driver with time and Figure 4.1.5.6 for the technology factor). The technology functions will keep the mining intensity low when there is only simple technology available and the demand is not very strong, but increase it significantly when demand increases and the technology is much improved. For the known reserves, we get equation (4):

$$\frac{dm_{Known}}{dt} = r_{Discovery} - r_{Mining} \qquad (4.1.5.4)$$

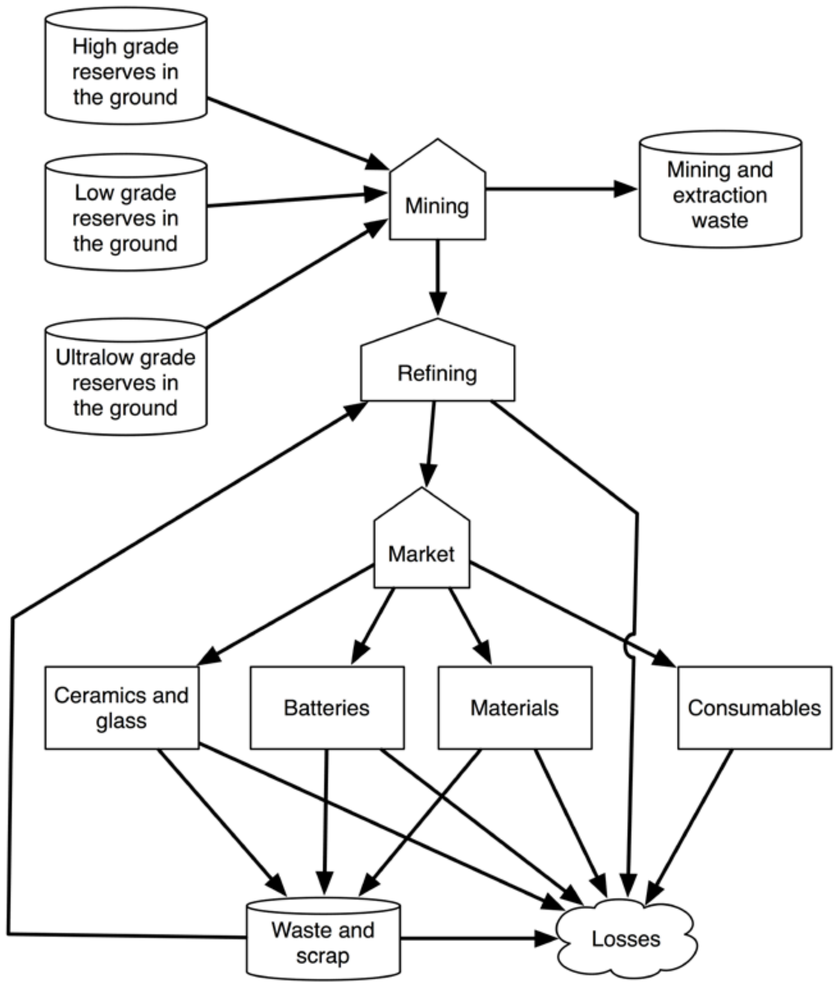
The discovery is a function of how much prospecting we do and how much there is left to find. The amount hidden resource (mHidden) decrease with the rate of discovery (rDiscovery). The rate of discovery is dependent on the amount metal hidden (mh) and the prospecting coefficient kProspecting. The prospecting coefficient depend on the amount of effort spent and the technical method used for prospecting. We get equation (5):

$$\frac{dm_{Hidden}}{dt} = -r_{Discovery} = -k_{Prospecting} * m_{Hidden} * f(T) * g(P) * j(m_{Known}) \qquad (4.1.5.5)$$

The rate is first order as prospecting is three-dimensional by drilling. The modifying functions f(T) and g(P) are the same as those defined above, but j(mKnown) is a curve expressing the urgency to prospect more, once the known reserves decrease to low levels. Prospecting also depends on how large the known reserves are. The larger they are, the less the urge to drill for more

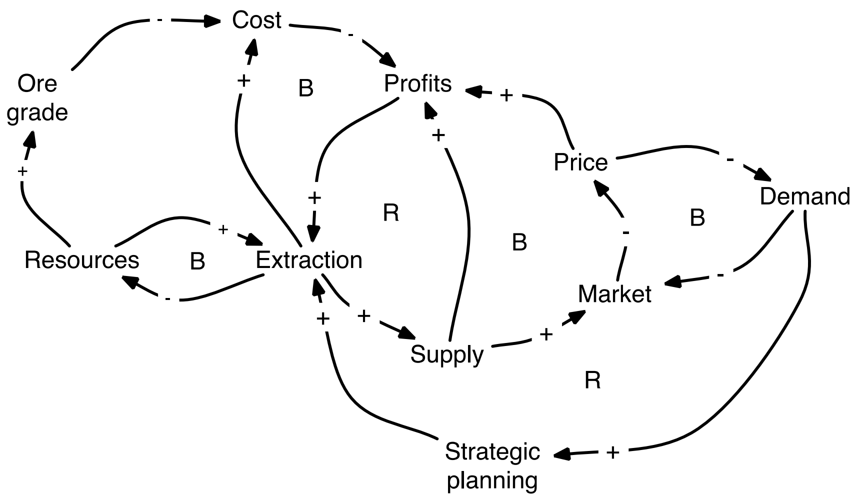
Figure 4.1.5.2 shows the flowchart for the LITHIUM model used in this study. In the model, mining and recycling is profit driven as is shown in Figure 4.1.5.3. The usual terminology of supply and demand is not used. It is based on observations in the New York and London metal markets for the copper, platinum group metal, silver and gold markets from the authors. Sales of extracted metal together with the price, given the income, the extraction and prospecting give the costs, the profit being the difference.

Figure 4.1.5.2. Flowchart for the LITHIUM model developed for this study



Source: Own Figure, Lund University/University of Iceland

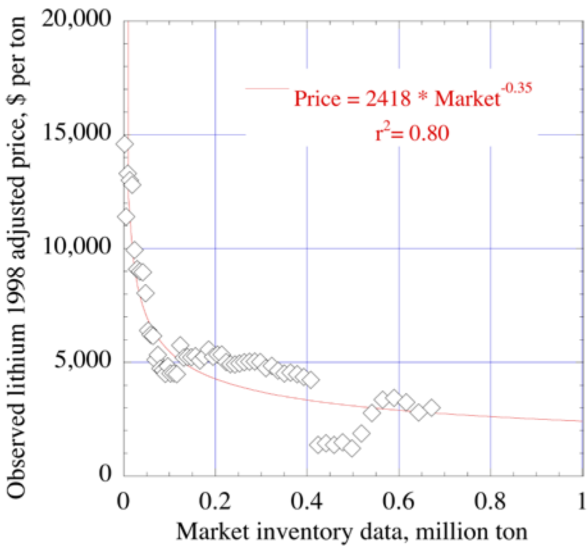
Figure 4.1.5.3. The price mechanism as implemented in the model is shown as a causal loop diagram.



Source: Own Figure, Lund University/University of Iceland

Figure 4.1.5.4 shows the lithium price curve, relating amount available in the market to the price. The price curve was constructed by plotting estimates of market inventory to the spot market price.

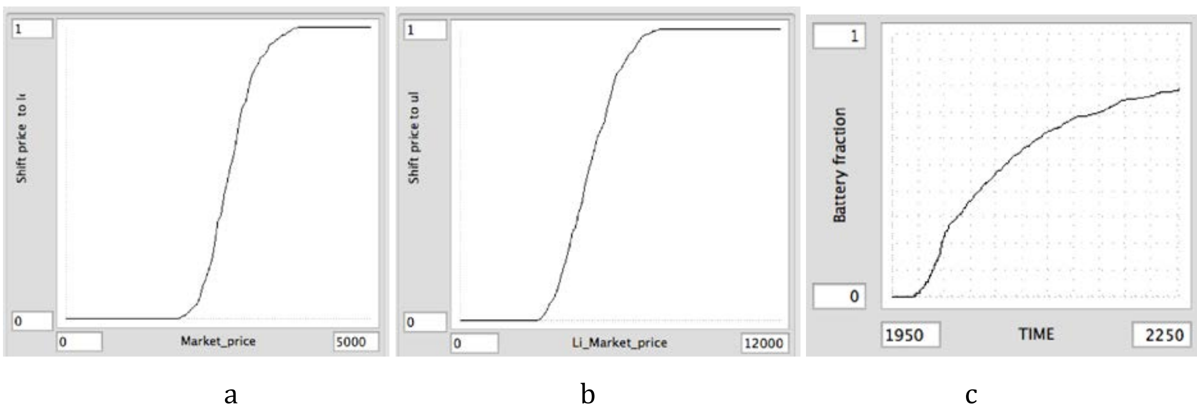
Figure 4.1.5.4. Diagram shows the lithium price curve, relating amount available in the market to the price



Source: Own Figure, Lund University/University of Iceland

The switch cost for starting to extract a lower quality substrate in the model is shown in Figure 4.1.5.5 and in Table 4.1.5.3. Figure 4.1.5.5a shows the switch from high grade to low grade. Figure 4.1.5.5b shows the switch from low grade to ultralow grade. Figure 4.1.5.5c shows the lithium demand from battery production as a fraction of the total demand.

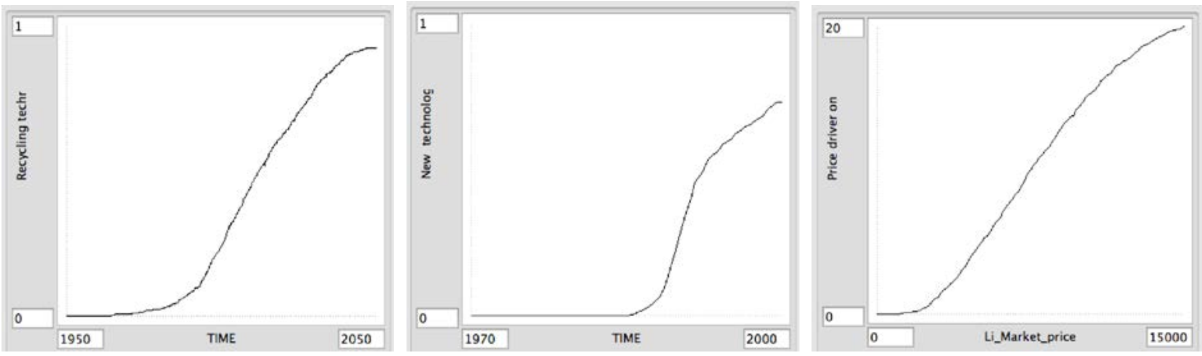
Figure 4.1.5.5. The switch cost for starting to extract a lower quality substrate in the model. Diagram (a) shows the switch from high grade to low grade. The median switch price is 2,650 $ per ton. Diagram (b) shows the switch from low grade to ultralow grade. The median switch price is 5,000 $ per ton. Diagram (c) shows the demand from battery production as a fraction of the total demand.



Source: Own Figure, Lund University/University of Iceland

Figure 4.1.5.6a shows the recycling enhancement technology as function of time. Figure 4.1.5.6b shows the new technology demand, as function of time. Figure 4.1.5.6c shows the price driver function on mining as dependent on the market price.

Figure 4.1.5.6. Feedback curves used in the LITHIUM model. Diagram (a) shows the recycling technology as function of time. Diagram (b) shows the new technology consumer products demand, as function of time. Diagram (c) shows the price driver function on mining as dependent on the market price.



a b c

Source: Own Figure, Lund University/University of Iceland

The LITHIUM model was used to explore the effect of this policy change on supply to society and the overall recycling fraction of supply as well as the Factor X. Factor X is defined as the ratio of total amount supplied to society divided by the amount extracted from the ground. The general consumer demand for products using lithium follows roughly the historic and future simulated population size (Ragnarsdottir et al., 2012). Recycling depends on price (Figure 4.1.5.6a) as shown by the data in Sverdrup et al., (2014a,b) in earlier studies.

A normal lithium battery in a computer or mobile phone can take about 1,000 charges before the performance starts to drop significantly, which corresponds to about 3.3 years of use. We have assumed that significant development will take place and that a battery will last about 7 years on the average, allowing for 2,100 recharges before the battery must be changed and recycled (At present, a battery can take 800-1,000 recharges, so we assume an improvement by 2.1-2.6 times the present). The present type of batteries available in the market do not live up to these expectations, except in the marketing. The present type of batteries also loose their capacity most significantly when the temperatures go down below zero, pointing to much larger batteries will be required for northern conditions in the winter (Gaines 2014, Gaines and Nelson 2015, Angerer et al., 2009).

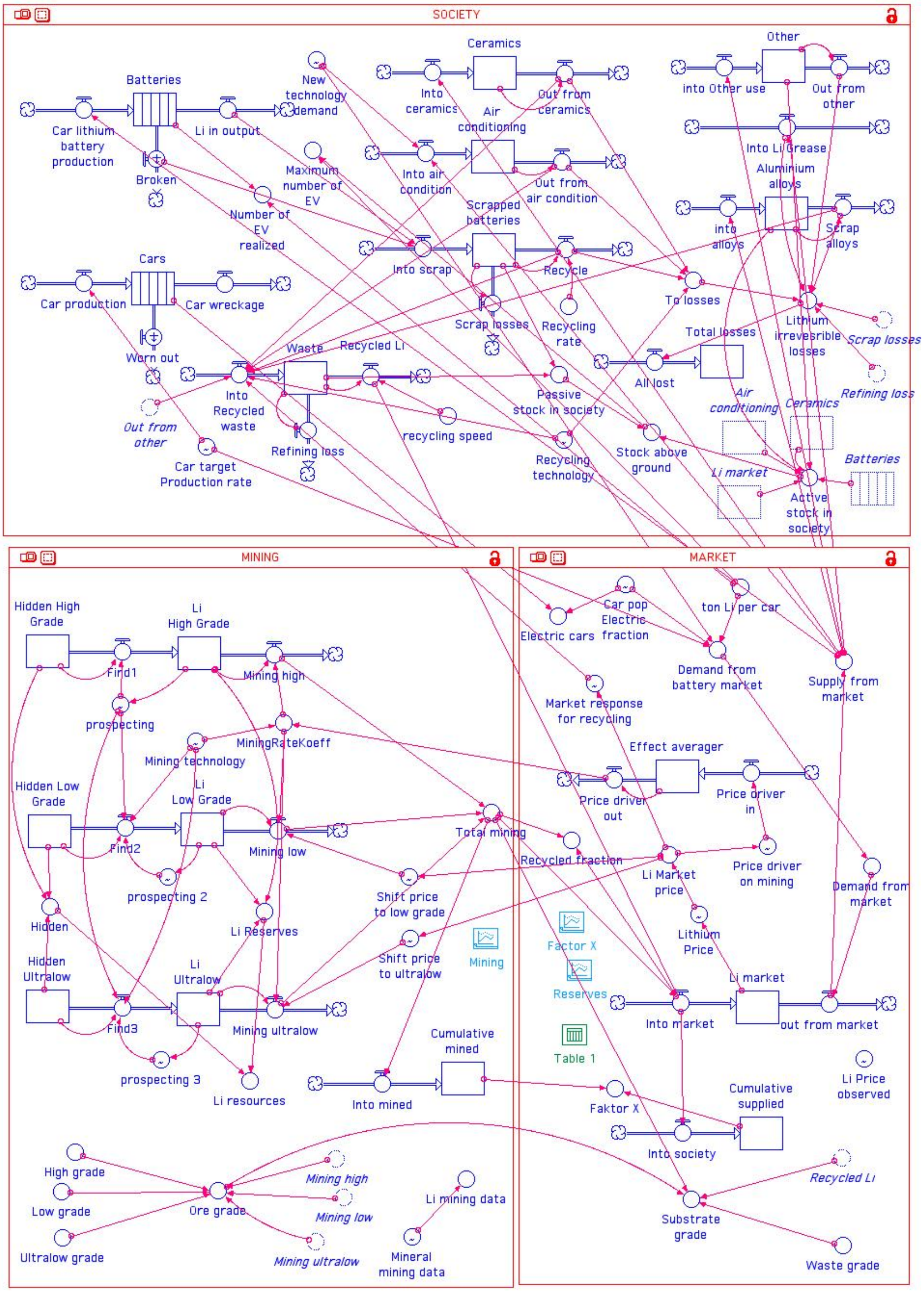
Present electric vehicles have a distance capacity rarely exceeding 150 km, and during winter time in Northern Europe, this must be divided by a factor of three. For the average customer in a thinly populated country, this is far from reasonable performance, and these cars offer no serious competition to fossil fuel-based cars (Bullis 2013). We may assume that research and development may reduce the amount needed for lithium batteries, but to get better reach for the vehicles, substantially larger batteries will be needed, probably offsetting any development in storage capacity. Some earlier (Carles 2010, Angerer et al., 2009, Mohr et al., 2010, 2012) model assessment assume 3-5 kg lithium per battery pack, we have assumed a substantially larger demand as realistic, as indicated in the ranges we have explored. We hold the opinion that a range of at least 300 km must be included in order for such vehicles to provide a viable alternative to hydrocarbon engine and hybrid vehicles.

Demand

Demand was made to a linear combination of a proportionality to the global population and a demand driven by the urge to replace conventional vehicles with electric vehicles. However, the “old applications” of lithium, such air conditioning, alloys, ceramics and lubricants kept their internal ratios, but the battery proportion of demand was allowed to increase as shown in Figure 4.1.5.5c. It shows the demand for batteries were assumed to rise faster than other demand. This shows the demand to battery production as a fraction of the total demand. It assumes that by 2250, 80% of all lithium supplied goes to batteries. Not that it is supplied and not primary extracted, as recycled is a part of supplied.

Figure 4.1.5.7 shows the STELLA software diagram for the LITHIUM model.

Figure 4.1.5.7. The STELLA model diagram for the LITHIUM model. The model has a sector for mining lithium from brines and minerals, one sector for the market and one for society and industrial use. In the model we have rearranged all extractable amounts into high, low and ultralow ore grade, disregarding if this is in salt brines or mineral deposits.



Source: Own Figure, Lund University/University of Iceland

Scenarios tested

We used the model to test the outcome of a number of scenarios. Our scenarios are:

1. Recycling rates tested were 50%, 65%, 80%. Recycling above 80% is difficult for technical reasons and the cost associated with extracting lithium from complex low grade sources.
2. Different possible sizes of URR; 116 million ton, 73 million ton and 34 million ton lithium.
3. Lithium requirement per vehicle unit; 3, 5, 10, and 30 kg lithium per unit. This corresponds to 100, 160, 320 and 400 km vehicle range on one charge.
4. The peak global stock of vehicles may reach 1.2, 1.7 or 2 billion units by 2050. 1.2 billion implies a stabilization at slightly above today's level, 1.7 billion is a 50% increase and 80% increase above the present level

The outputs corresponding to these scenarios will be found later in the results section. As shown in Figure 4.1.5.7 the model has a sector for mining lithium from brines and minerals, one sector for the market and one for society and industrial use. In the model we have rearranged all extractable amounts into high, low and ultralow ore grade, disregarding if this is in salt brines or mineral deposits. In the model, shift to a lower ore grade only occurs when the market price exceed the extraction cost in such a way that the profit stays positive. When the profit is lower or goes negative, the mining rate is slowed down, leading to less material in the market, driving the price up. In practice we have several sources of metal in society; the high, low and ultralow ore grades and the stocks of metal in society that can be recycled. For lithium, we use three different ore grades, as well as recycling.

4.1.5.5 Input data generation

The available data was closely inspected for inconsistencies and averages and adjustments were made when the input data were not internally consistent. Many of the data sources we have used are inconsistent between them and we are sometimes forced to make expert judgment about what are the most likely figures. The available amount for extraction from salt-brines, pegmatitic minerals and lateritic clays have been mapped by several studies (Colton, 1957, US Bureau of Mines 1980, Anstett et al., 1990, Ober 1994, 2004, Kunaz 1994, Yoda and Ishihara 1997, Tarascon and Armand 2001, Helvaci et al., 2003, USGS 2008, 2012, 2014, Tahil 2008, Evans 2008a,b, Wellmer 2008, Fernandez 2009, Yaksic and Tilton 2009, Clarke and Harben 2009, Angerer 2009, Jaskula 2010, Mohr et al. 2010, 2012, Gruber and Medina 2010, Gruber et al., 2011, Jacobson and Delucci 2011, Eilu 2011, Munk 2011, Rodigues and Contreras 2011, Munk 2011, Ziemann et al., 2012, Vikstrøm et al. 2012, Grosjean et al. 2012, Keseler et al., 2012, Polinares 2012, Munaz 2011, Kushnir and Sanden 2012, Tan et al., 2012, Li et al., 2013, Bradley and Jaskula 2014, Gaines 2014, Speirs et al. 2014, Nickless et al., 2014, King, 2015).

The opportunity cost approach was used when evaluating extractability of resources (Tilton 2009, 2012). This was done by assigning an extraction yield and an extraction cost for each ore quality class. Table 4.1.5.1 shows different estimates of reserves and resources by different studies. All amounts are in million metric ton lithium content. Considerable uncertainty is connected to the estimation of the amounts stored in clays. In some studies, this is stated, but not quantified. Some studies exclude lithium in clays altogether.

The study of Mohr at al. (2012) was useful and has a thorough assessment of available extractable amounts. Mohr et al. (2012) paid special attention to the reserves and resources, listing every resource they could find data on. Mohr et al. (2012) recognizes that there may be significant additional resources not yet discovered, considering that large areas have not been prospected for lithium yet. Thus, any estimate limiting itself to what is known from a limited part of the possible territory, risks coming up with a substantially too low estimate. The conclusion to draw from Mohr et al. (2012) is that, even if they come up with a larger estimate than many other researchers, it may still have to be considered as a minimum estimate. We have extrapolated the occurrence of lithium in areas investigated, and tried to guess how much more there could possibly be in comparable areas, such as salt brines of different types, lithium-containing clays or areas with pegmatite rocks, all being helped by literature such the study of Keseler et al. (2012) and more. If this is considered, the global resource converges on slightly less than 120 million ton lithium eventually being technically extractable, disregarding the cost for the moment (Tables 4.1.5.1 and Table 4.1.5.2).

Table 4.1.5.1. Different estimates of reserves and resources by different studies. All amounts are in million metric ton lithium content. Considerable uncertainty is connected to the estimation of the amounts stored in clays, and a base assumption was made. These estimates are placed in brackets. Only Mohr et al., (2012) reports extracted to present, but this coincides with our estimate (Italics) from the production curve, and is fairly certain. Data was adapted using data from the literature (See in particular Mohr et al., 2012), combined with our own assessment. On the average across all the estimates, "reserves" are about 30% of the "resources".

Author	Reserve Estimate "Known"	Resource estimate "known" + "hidden"				Extracted to 2015	URR
		Brine	Mineral	Clays	Sum		
Ziemann et al., (2010, 2012)	16	14	12	(30)	56	0.8	57
Polinares (2012)	13	n.a.	n.a.	n.a	n.a	0.8	n.a
Vikstrøm et al. (2012)	30	n.a.	n.a.	n.a	(60)	0.8	(61)
USGS (2014)	18	24	15	(30)	69	0.8	70
Grosjean et al. (2012)	16	29	15	(30)	74	0.8	75
Mohr et al. (2010, 2012)	19	30	20	5	55	0.75	56
Keseler et al., (2012)	20	24	10	115	149	0.8	150
Yaksic and Tilton (2009)	39	20	20	24	44	0.8	45
Evans (2009)	n.a.	29		(30)	59	0.8	60
Clarke and Harben (2009)	39	(39)		(30)	69	0.8	70
Tahil (2008, 2009)	19	(36)		n.a	66	0.8	67
Gruber and Medina (2010)	15	25	14	(30)	69	0.8	70
Average	22				72		72

Table 4.1.5.2. Input data for the runs with the LITHIUM Model, divided into a high, middle and low estimate for the extractable amount of lithium. Amounts in million metric ton lithium content.

High: Assuming 50% extractability on all resources											
Ore grade	Switch $ per ton Li	Salt-brines			Lithium minerals			Lateritic clays			Sum
		Hidden	Known	Sum	Hidden	Known	Sum	Hidden	Known	Sum	
High	-	6	4	10	4	3	7	10	0	10	27
Low	2,650	13	1	14	7	1	8	15	0	15	37
Ultralow	7,500	20	0	20	15	0	15	17	0	17	52
Sum		39	5	44	26	4	30	37	5	42	116

Middle scenario: Based on Table 4.1.5.1.											
Ore grade	Switch $ per ton Li	Salt-brines			Lithium minerals			Lateritic clays			Sum
		Hidden	Known	Sum	Hidden	Known	Sum	Hidden	Known	Sum	
High	-	6	3	9	6	1	7	4	0	4	20
Low	2,650	6	0	6	8	0	8	4	0	4	18
Ultralow	7,500	10	0	10	10	0	10	5	0	5	25
Sum		22	3	25	25	0	25	13	0	23	73
Low scenario											
Ore grade	Switch $ per ton Li	Salt-brines			Lithium minerals			Lateritic clays			Sum
		Hidden	Known	Sum	Hidden	Known	Sum	Hidden	Known	Sum	
High	-	6	3	9	6	1	7	4	0	4	20
Low	2,650	4	0	4	3	0	3	1	0	1	8
Ultralow	7,500	2	0	2	2	0	4	0	0	0	4
Sum				15			14	5	0	5	34

For the model, the data extracted from the literature has been rearranged to fit the models input data format requirements. The extractable amounts have been sorted according to substrate quality in terms of extraction cost as defined by in Table 4.1.5.3. Table 4.1.5.2 shows the final assembly of the input data for extractable amounts for the model runs. We have adopted 3 different scenarios:

1. A high scenario, where all plausible resources have been included assuming them all to be fully extractable (Keseler et al., 2012). URR was assumed to be 116 million ton lithium.
2. A middle scenario where we in agreement with Mohr et al., (2012) assume 50% of all resources to be extractable, but we come to substantially higher estimate URR than the Mohr et al., (2012) study. URR was assumed to be 73 million ton lithium.
3. A low scenario is consistent with the conventional view as shown in Table 4.1.5.1 (USGS 2008, 2012, 2014, Grosjean et al., 2012, Gruber et al., 20111, Ziemann et al., 2010, 2012, Polinares 2012, Vikström et al., 2012, Mohr et al., 2012). URR was assumed to be 34 million ton lithium.

Table 4.1.5.3. Quality definition of the extraction substrate qualities

Substrate grade	High	Low	Ultralow	Extralow
Lithium weight content	8-3%	3-1 %	1-0.3%	>0.3%
Extraction threshold cost	1,000 $ per ton	2,650 $ per ton	4,000 $ per ton	6,000 $ per ton

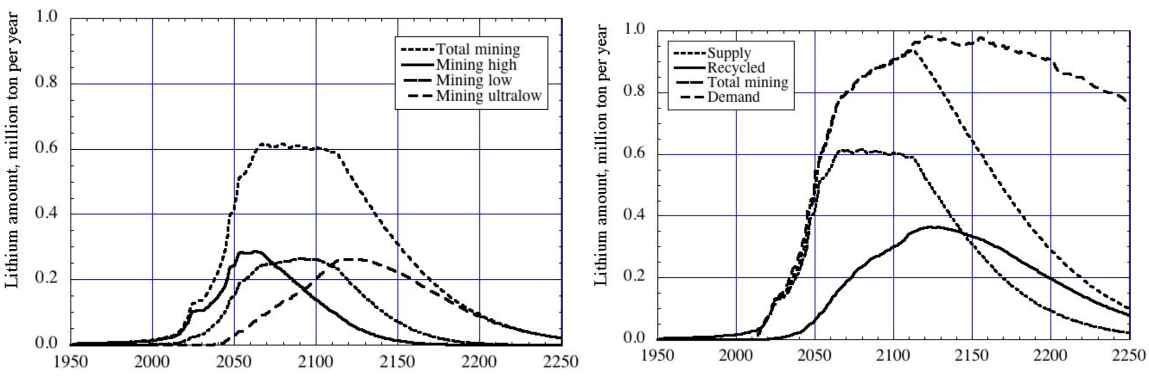
The lithium amounts represent the amounts and their allocation in 1950, when the LITHIUM model run starts. In the Table 4.1.5.2, amounts in different substrate qualities have been considered, and salt brines, pegmatite minerals and lateritic clays been factored in. "Known" is the known reserves, whereas "known" plus "hidden" amounts to what is called known and anticipated resources. The LITHIUM model was applied to all these scenarios. Figure 4.1.5.5 shows the switch cost for starting to extract a lower quality substrate in the model. The category extra-low ore grade can also be defined and its size estimated, but it was not used in this assessment. This would apply to large quantities of pegmatite, deep sea brines and some bottom nodules, and deep subterranean fossil salt-beds (about 15 million ton extra, but at present, this is not really technically feasible to extract). These have been not included in the simulations as they were estimated to be too expensive and energy demanding to extract. Including them would increase the URR from 73 million to 90 million ton lithium in the middle estimate (Table 4.1.5.2). It can be seen from the data in Table 4.1.5.1 that it will be important to include lateritic clays in the estimates. The lateritic clays are low in ore grade but high in contained amount (Crocker et al., 1988, Rodrigruez and Contreras 2011, Siame 2011, Keseler et al., 2012, Grosjean et al., 2012, Sonoc and Jeswiet 2014). However, the extraction costs are high requiring a high lithium price. The environmental impacts of large scale clay mining will be significant. At an ore mineral content of 1%, every ton lithium content produced will generate more than 100 ton of mine waste, and then refining waste comes in addition to that. Our conclusion is that the technically extractable resources are far larger than earlier realized, and that a new assessment of sufficiency and risk for scarcity is needed.

4.1.5.6 Results

The LITHIUM model was run 1900-2250 for the medium extractable amount case shown in Table 4.1.5.2 and plotted up in Figure 4.1.5.8-14, and a small sensitivity analysis of the extractable amount estimate in Figure 4.1.5.14.

Figure 4.1.5.8a displays the mining rate over time in total and from the different ore qualities. It can be seen that the model suggests that extraction will reach a peak in the time period 2065, when high grade deposit production goes into decline. The production will stay level at about 0.6 million ton lithium per year to about 2110, and then decline. This coincides with beginning decline in the known reserves, and that more effort must be made to produce the same amount from a lower grade. The decline in extractable reserves can be counter-reacted by intensifying the extraction rate at which at the same time will increase the speed of high grade ore exhaustion. Figure 4.1.5.8b shows a comparison of the demand, mining rate, the flow from recycling and the total supply to the market. The simulation suggests that by 2020-2025, recycling may become the most important source of lithium, as mining steadily go down with declining ore grade and increasing cost of primary extraction. In 2110 we can see a demand-supply separation, suggesting that after this, scarcity starts to appear. Further runs suggest that after 2400, the world will in principle be without primary production of lithium.

Figure 4.1.5.8. Diagram (a) shows the mining rate over time in total and from the different ore qualities. Diagram (b) shows a comparison of the mining rate, the flow from recycling and the total supply to the market and the demand.

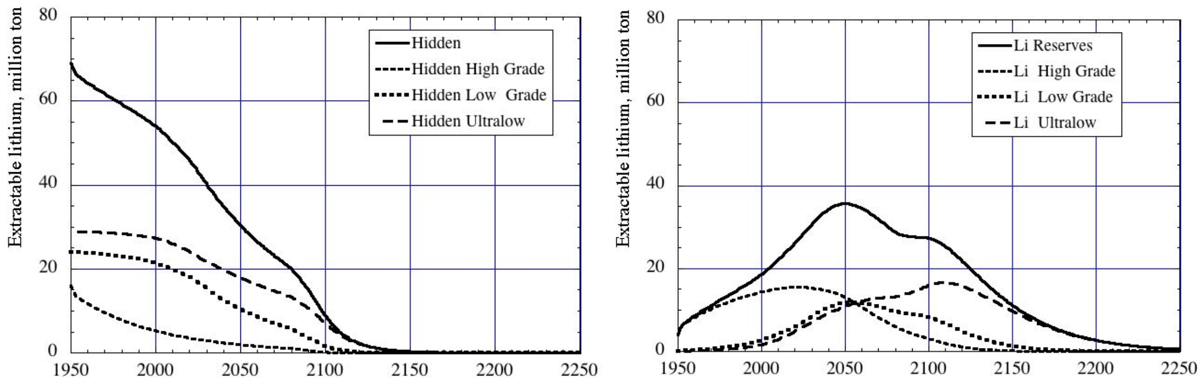


a b

Source: Own Figure, Lund University/University of Iceland

Figure 4.1.5.9a shows the hidden and Figure 4.1.5.9b known reserves of lithium. The model predicts that the known reserve in 2008 is 21 million ton and in 2012 that it is 23 million ton, including salt brines, hard minerals and clays. Table 4.1.5.1 inform us that the reserves estimates range from 13-39 million ton, with an average of 22 million ton lithium.

Figure 4.1.5.9. Simulation of the hidden (a) and known (b) extractable amounts of lithium, including salt brines, hard rock minerals and lithium contained in clays.

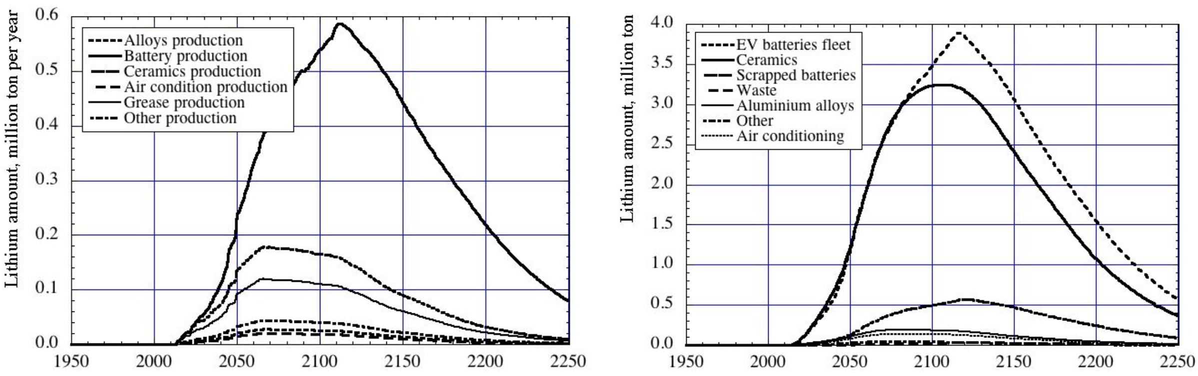


a b

Source: Own Figure, Lund University/University of Iceland

Figure 4.1.5.10a shows the supply rate of lithium in million ton per year to different sectors. Battery production becomes the dominant market in the future. Figure 4.1.5.10b shows the stock in society in the same sectors. Batteries and ceramics dominate in the stocks-in-use. Accumulators represent the largest stock-in-use, as batteries are assumed to last about 7 years, where as the other applications have their specific lifecycles. This is about twice the duration of present batteries, but we have assumed that lifetimes will improve fast in the near future from the present 3 years to 7 years (Burke 2012). The life time for air conditioning material was set at 7 years. The life time for ceramics was set at 20 years. For aluminium alloys with lithium the lifetime in society was set as 5 years on the average. For scrap it is assumed that it takes 1 year before it is recycled back to battery. For “other”, like medications, the residence time for lithium is 1 year. For greases, it is a flow, greases are not stored in significant amounts and all is lost after use. The maximum lithium stock in use for batteries peak at 4 million ton in 2115.

Figure 4.1.5.10. Diagram (a) shows the supply rate of lithium in million ton per year to different sectors. Diagram (b) shows the stock in society in the same sectors.

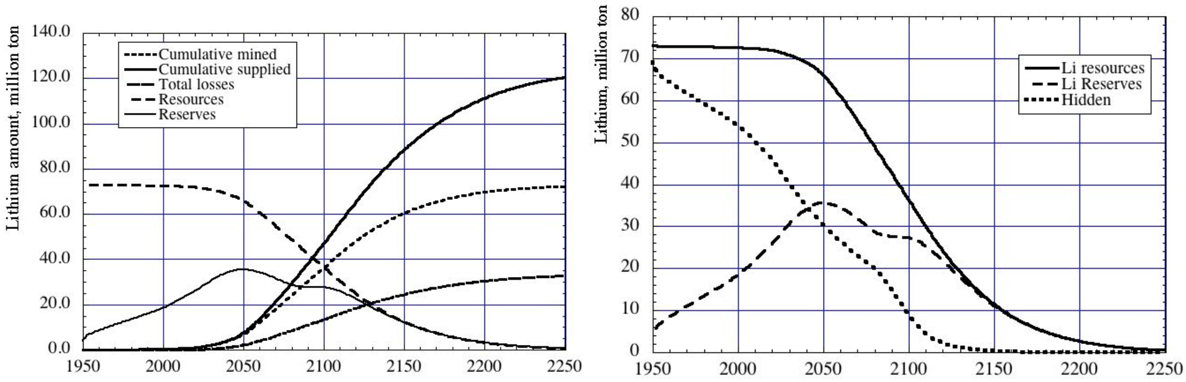


a b

Source: Own Figure, Lund University/University of Iceland

Figure 4.1.5.11a shows the cumulative amounts supplied, mined and lost with time. It can be seen that the amount supplied to the market becomes much higher than the amount mined in the future. This implies that recycling actually increases the amount in use. Figure 4.1.5.11b shows the total reserves, the total resources and the amount hidden.

Figure 4.1.5.11. Diagram (a) shows the cumulative amounts of lithium mined, supplied and lost with time. Cumulative supplied is larger than extracted because of recycling. Every ton of lithium is used about twice. Diagram (b) shows the extractable resources, the known reserves and the hidden resources.

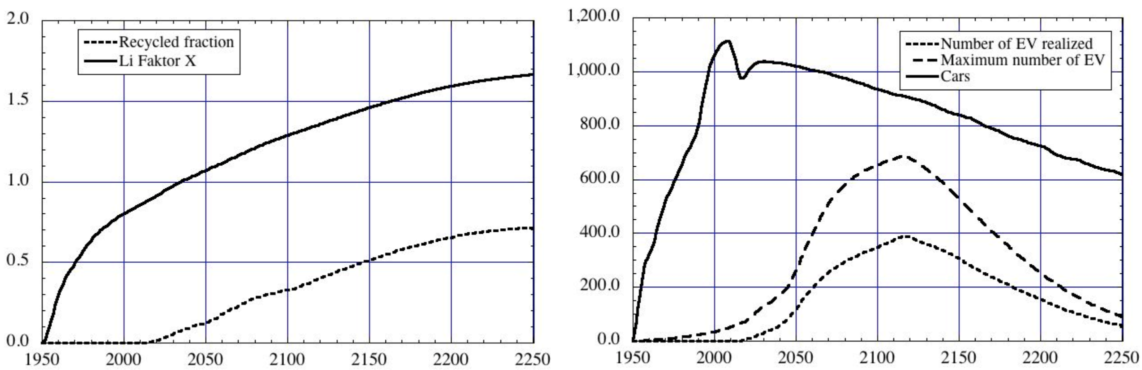


a b

Source: Own Figure, Lund University/University of Iceland

Figure 4.1.5.12a shows the degree of recycling and the factor X. Factor X is the ratio of the total cumulative amount supplied to society to the mined cumulative amount. The recycling (ratio of recycled to the total amount supplied to society) stabilizes at about 50% after 2150 and will stay there as long as the lithium price does not go up further. This is low because of several of the uses of lithium have no or limited recycling. For some of the uses, the use of lithium is present in the product in such a low concentration (medication), bound in such a strong compound (Ceramics), or used in a way that it gets lost (greases, lubricants), that recycling is not very feasible. Figure 4.1.5.12b shows the number of total cars and the projected future demand for vehicle stock in society. The simulations with our model suggest that the demand for electric vehicles (EV) cannot be satisfied. "EV realized" in Figure 4.1.5.12b is the number of EV possible, based on the assumption that about 10 kg of lithium is used per vehicle. If we assume 85% of the active above ground stock is allocated to batteries, a maximum of about 400 million electric vehicles can be maintained.

Figure 4.1.5.12. Diagram (a) shows the Factor X and degree of recycling. Diagram (b) shows the number of total cars and the projected future demand for vehicle stock in society using the standard case with URR=73 million ton. The demand for electric vehicles is also shown. EV realized is the number of EV possible. The maximum number of EV assume 85% of the active above ground stock is allocated to batteries at all times.

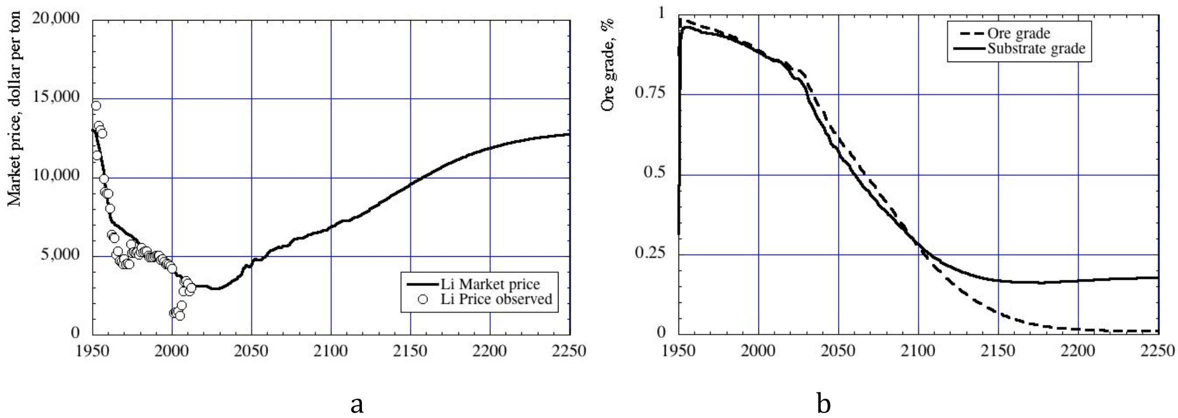


a b

Source: Own Figure, Lund University/University of Iceland

Figure 4.1.5.13a shows the simulation of the lithium market price. For the period 1965-2010, the model simulation matches the observed price level fairly well. The correlation to the observed data is r2=0.86. Figure 4.1.5.13b shows the development of ore grade with time, and the ore substrate grade. The substrate grade is the grade of all the material used for extraction, including scrap as an ore. In the calculation, we assume that the scrap from which recycling takes place contain average about 1% lithium. That is higher than the lowest ore grades. Thus, at some point it (about 2100) will be more cost effective to derive lithium from recycling than from primary extraction.

Figure 4.1.5.13. Diagram (a) the simulated market price as compared to the observed price in value-adjusted dollars to 1998 per ton lithium content. Diagram (b) shows the development on ore grade with time, and the substrate grade.

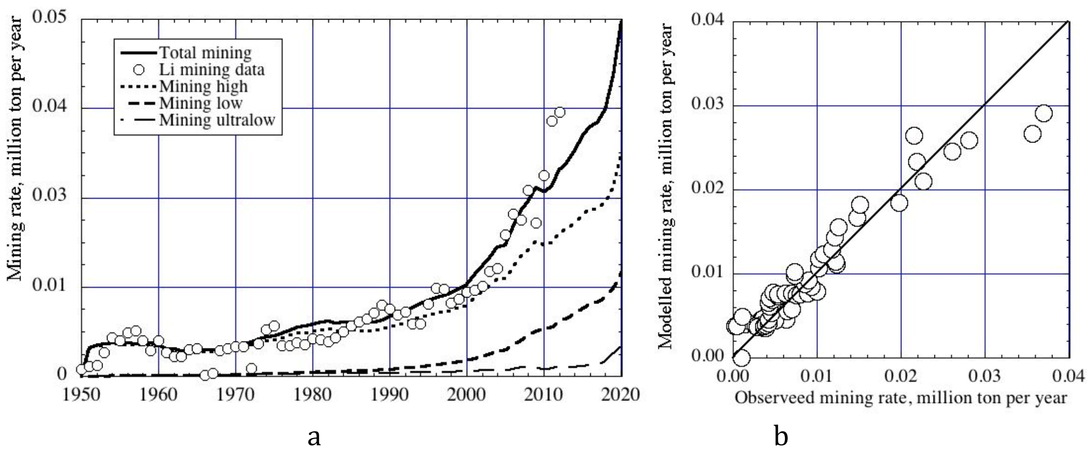


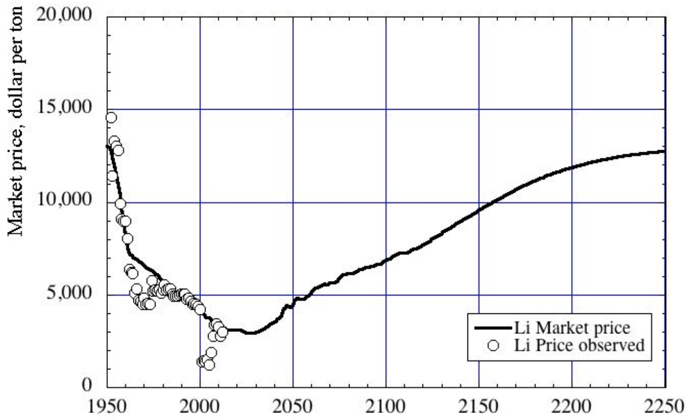
Source: Own Figure, Lund University/University of Iceland

4.1.5.7 Testing the model

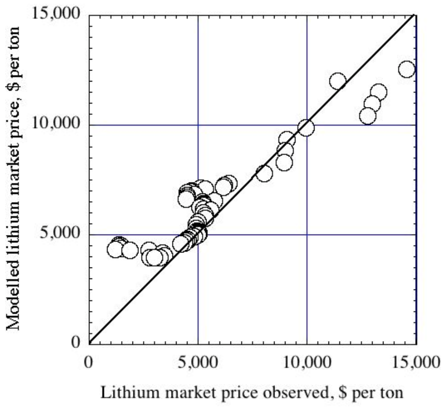
Figure 4.1.5.14 shows the mining 1950-2020 in a different resolution. Figure 4.1.5.14a shows a test of the model on the mining rate against observed data. Figure 4.1.5.14b shows the same data in a 1:1 plot of observed versus simulated with the LITHIUM model. The fit between simulation and observed data is excellent; r2=0.91. Figure 4.1.5.14c and Figure 4.1.5.14d show a test of the model on modelled price versus the observed data. The correlation to the observed data is reasonable; r2=0.86. This test shows that the model is able to reconstruct the past with respect to mining rate and price. This give confidence for making future assessments and predictions.

Figure 4.1.5.14. Test of the model on the mining rate against observed data in diagrams (a) and (b). The fit is excellent, r2(mining)=0.91. Diagrams c and d shows the simulation of the lithium market price in $ per ton, value adjusted to 1998 year dollar value. The correlation to the observed data is good; r2(price)=0.86.





c



d

Source: Own Figure, Lund University/University of Iceland

4.1.5.8 Discussion

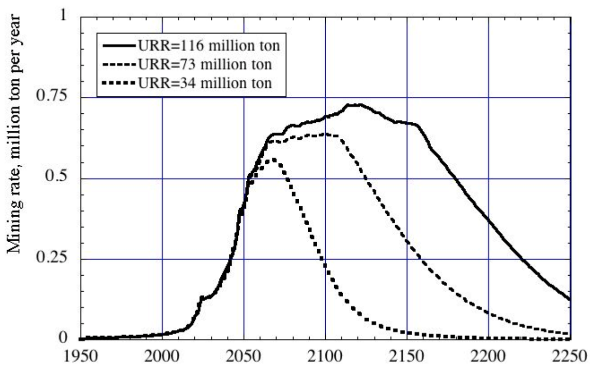
Sensitivity analysis

The sensitivity of the model was explored with a very limited sensitivity analysis, using the high, middle and low scenario for extractable amount as outlined inTable 4.1.5.2. The high scenario assumes the extractable amount to be 116 million ton lithium, the middle scenario 73 million ton lithium and the low scenario sets the extractable amount at 34 million ton lithium. The results for lithium mining rate, supply to the market and the market price in $ per ton is shown in Figure 4.1.5.15a-d. Figure 4.1.5.15a shows the effect of URR on the mining rate, Figure 4.1.5.15b shows the effect of changing URR on the supply rate, Figure 4.1.5.15c the stock in society and Figure 4.1.5.15d the effect on the market price. For lower reserves, the lithium price rises to higher levels earlier. We get for the three resource size scenarios listed earlier:

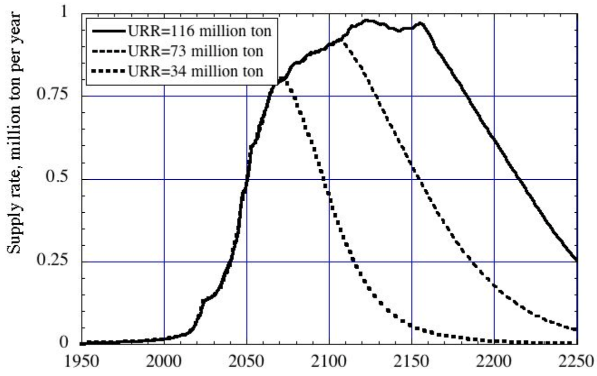
1. URR=34 million ton; The electric vehicle production peaks at 8 million units per year. The total number electric vehicles produced will be 522 million units to 2400.The amount of electric vehicles realized yields a stock of electric vehicles in society of about 305 million vehicles at the peak in 2060-2070. The number declines after that.
2. URR=73 million ton: The electric vehicle production peaks at 8.3 million units per year. The total number electric vehicles produced will be 962 million units to 2400. This implies a relatively level stock from 2070 to a maximum stock in 2120, and a decline after that.
3. URR=116 million: The electric vehicle production peaks at 9.8 million units per year. The total number electric vehicles produced will be 1,570 million units to 2400. The EV stock increases to about 310 million vehicles by 2070 and grow slowly up to 380 million vehicles in 2160. After that, decline sets in.

The number of electric vehicles in the future vehicle fleet will be substantial. For a strategy to replace conventional fossil fuel driven vehicles with electric vehicle to be successful, a strong recycling strategy will be a requirement. The large need to recycle them suggests that here is a significant future business opportunity. The size of the extractable amounts has a significant effect on the lithium mining rate, the level of supply and the market price. We find the middle case the most plausible when we assess the reserve and resource estimates critically. All three cases listed above imply cases of exhaustion of a finite source, with the same result, despite the large difference in extractable amounts. The stock in society stays always below 2.5 million ton lithium, and it is always lower that the known reserves.

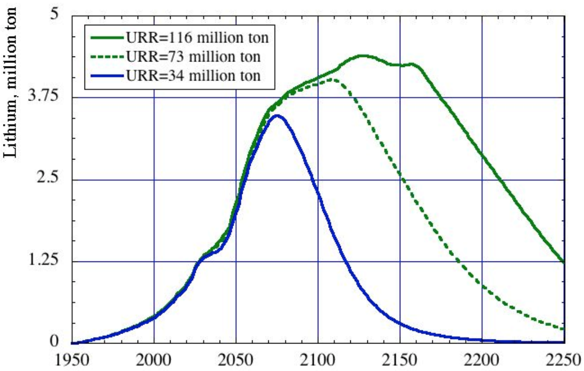
Figure 4.1.5.15. A simple sensitivity analysis with the LITHIUM model using the high (URR=116 million ton) middle (URR=73 million ton) and low (URR=34 million ton) estimates for the extractable amount of lithium as outlined and specified in 4.1.5.2. Diagram (a) shows the mining rate, diagram (b) shows the lithium supply rate, diagram (c) shows the stock in society and diagram (d) shows the lithium market price for the different URR scenarios.



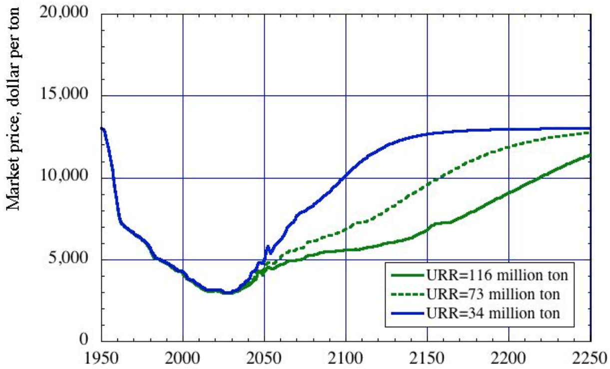
a: Mining rate



b: Supply rate



c: Stock in society



d: Market price

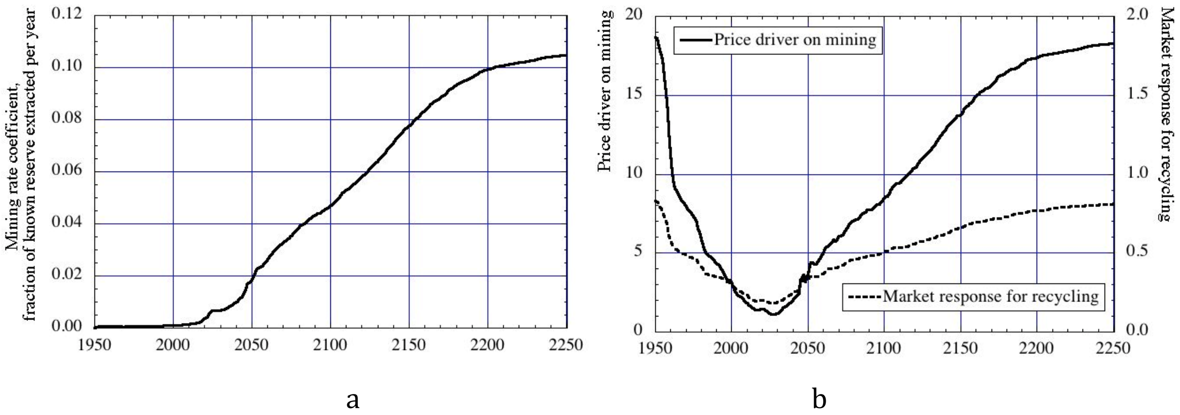
Source: Own Figure, Lund University/University of Iceland

The extractable amount estimates, as illustrated in Table 4.1.5.2, are considerably larger than the literature estimates. The reasons for this are several. We have considered high, low and ultralow grade ores of salt brines and pegmatite minerals. We have included lateritic clays that contain significant amounts of adsorbed lithium on the ion exchange sites of the clay. This can be leached out and extracted using special solvents and then separated from other alkali metals using repetitive chemical methods. The sensitivity analysis we have done here is only a first tentative exploration. At a later stage, more sensitivity analysis will be needed with respect to more parameters, like mining intensity, delay times on feedbacks, enhanced recycling, different future product changes and proportions between lithium uses. Different types of market interventions may be explored.

Sensitivity analysis using a different mining rate

The mining rate is important for the total extraction rate. Figure 4.1.5.16a shows the change in the technology factor with time. This feature is essential for obtaining the good fit to the observations. The mining coefficient is the result of two feedbacks, one is the development of technology, and the other is the push from a higher lithium price. Figure 4.1.5.16b shows the value of the feedback function in the model with time on mining from the price and the feedback function in the model with time on recycling from the price. It can be seen that the market feedback is substantial throughout the whole period. The effect of the global mining output being capped at a global production of 250,000 ton lithium per year and 85% of the production is reserved for electric vehicle batteries can be investigated with the model. The lithium would lasts longer. The production of EV units may stay constant at 400 million electrical vehicles per years for a longer time to 2300-2350.

Figure 4.1.5.16. Diagram (a) shows the value of mining rate coefficient from Equation (4.1.5.3), it tells how intense the mining effort is. Diagram (b) shows the push on mining from the price and the push on recycling from the price.

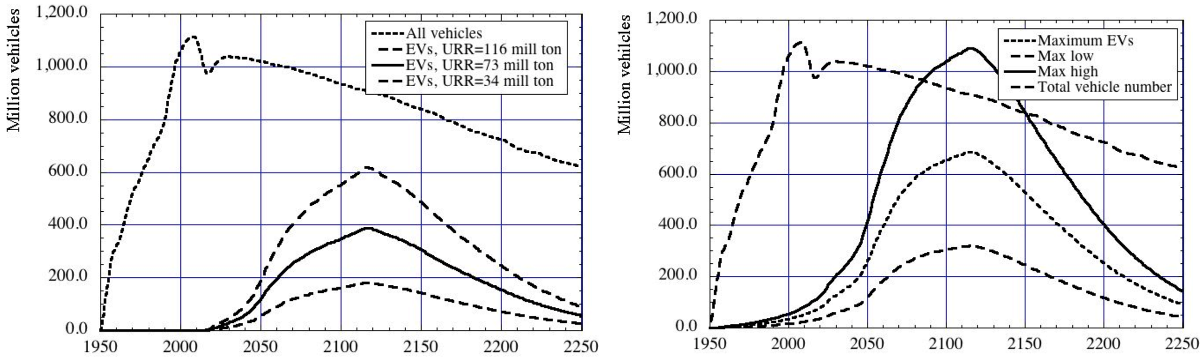


Source: Own Figure, Lund University/University of Iceland

The feasibility of replacing all cars with electric vehicles

The simulations show that all cars in the world today cannot be replaced with electric vehicles without overcoming some substantial hurdles and challenges (See Figure 4.1.5.17 and Table 4.1.5.4). The lithium weights per vehicle battery unit given in the Table 4.1.5.4, it shows the values applied in presently produced batteries. We have assumed a vehicle design range of 500 km under summer conditions. In below zero degrees centigrade conditions, the range is normally less than half of that (Bullis 2013, Gorzelany 2014). Zero degree situations occur in a majority of where people live some time during the year. Table 4.1.5.4 shows the electric vehicle potential as a function of lithium requirement and size of the URR. The total vehicle fleet has a peak at 1.2 billion cars in this scenario. We have assumed three different recycling scenarios; 50% which is the business-as-usual, 65% which is the enhanced recycling scenario, and 85% which is an ambitious scenario. Across that we have taken the three different sizes of URR; 34 million ton lithium which is the literature average estimate, 73 million ton lithium which we think is the best estimate, and 116 million ton lithium which we think is a global maximum for extractable lithium. Moving the increase in battery production forward, does not really change the overall picture of how many, as mass balance principle still apply.

Figure 4.1.5.17. Most likely under the base-line scenario (a) and maximum feasible numbers (b) in the global electric vehicle fleet.



a; Base case b; Maximum technical possible

Source: Own Figure, Lund University/University of Iceland

There are two developments to consider when looking at these estimates. The world contains about 1.2 billion cars today (2016), and we conclude that a maximum stock of 400 million cars can be maintained in the base case for URR and recycling, assuming business-as-usual with respect to recycling and lithium use per car battery unit (10kg). As resources dwindle, the ability to maintain fleet size decline. The maximum occurs around 2120. If a policy change is made where 85% of all lithium is reserved for electric vehicle battery production, we will achieve market sufficiency in about 2060, and be able to maintain this to about 2180-2200, when the ability to supply declines. Depending on different interpretation of the literature and evaluation of future global economy, this may stagnate at the present level or continue to grow up to about 2 billion cars on the globe, corresponding to a stock of about 10-15 billion ton steel locked in vehicles. We may conclude that we need to think in terms of business-as-unusual, and change things in the system. If the trend towards heavier batteries seen recently would continue, we should look towards the row with 30 kg lithium per vehicle unit. We will with all probability see peak fossil fuels occur long before that time, and a decline is as likely as an increase. Assuming higher car production will not lead to more electric vehicles, the mining rate is capacity limited. The number of electric vehicles realized depend on how much lithium is available, and not how many cars that are needed.

Table 4.1.5.4. Electric vehicle potential as a function of lithium requirement and size of the extractable amount available. Basic car demand, has a peak at 1.2 billion cars in this scenario (Figures 4.1.5.17a and b). The positions where 85% of the world's vehicle demand of 1.2 billion cars can be met has been put in light shaded. To meet 85% demand of 2 billion has been put in a darker shade

Size of the extractable amount, million ton lithium	Recycling fraction of supply alternatives used								
	Alternative 1; 50%			Alternative 2; 65%			Alternative 3;80%		
	Lithium requirement per battery unit, kg lithium contained								
	5	10	30	5	10	30	5	10	30
	Millions of electric battery units possible								
116	1,200	610	203	2,884	1,442	481	3,050	1,525	508
73	800	400	133	1,892	946	315	2,000	1,000	333
34	396	198	66	880	440	147	932	466	155

Table 4.1.5.5. Different battery types and their properties and materials requirements (Gaines and Nelson 2010, Gaines 2014).

Components	Description of compo-nents and costs	Wh /kg	500 km range at 190 Wh/km, ingredient weight, kg							Lim it me tal	Use	#
			Li	Co	Ni	Mn	Ti	REE	Unit kg			
Li(Org)O2.	Does not tolerate fast dis-charge. Cathode material 10 $/kg, cost of cathode 40-50 $/kg.	350	32	-	-	-	-	-	50-80	Li	Space	1
LiCoO2	Weight fac-tor be-tween Co and Li is 8.5. The highest risk for self-ig-nition. Cathode material 18 $/kg, cost of cathode 27-36 $/kg.	170	56	476	-	-	-	-	530	Co	Tele PCs EV	2
Li(ConNinMnn)O2 **Li(CovNiwAlz)O2** **n=1/3, v=0.15, w=0.8, z=0.05**	These bat-teries dis-charge and recharge faster. Weight fac-tor Co:Li is 1.5-2.8, Ni:Li is 2.6-8. Cathode material 11 $/kg, cost of cathode 22-30 $/kg.	150	25	63 -	84 200	66 -	- -	- -	240	Co Ni	EV	3
Ag2O/Zn	Uses silver and zinc as main com-ponents	130	-	-	-	-	-	-	330	Ag	Any	4

LiMnO2	Weight factor Mn:Li is 7.9. Cathode material 3.8 $/kg, cost of cathode 10 $/kg.	100	11	-	-	88	-	-	100	Li	EV	5
LiFePO4	Weight factor Fe:Li is 8.5, PO4:Li is 13.4. Cathode material 1.5 $/kg, cost of cathode 20 $/kg.	100	16	136	-	-	-	-	152	Li	EV	6
Li4Ti5O12	Fast charge and discharge, lower fire hazards. Weight factor Ti:Li=6.9. Cathode material 4 $/kg, cost of cathode 10 $/kg.	100	41	-	-	-	354	-	395	Li	EV	7
NiZn	Very simple design. Fewer recharging cycles. Low cost.	100	-	-	500	-	-	-	500	Ni	EV	8
NiREEO2 La0.8Nd0.2Ni2.5Co2.4	Complex mix of Ni, Co and REE.	50-70	-	350	400	-	-	50-300	700-1,000	REE Ni Co	EV	9
NiCd	Simple design. Toxic waste by failure	40-60	-	40	500	-	-	-	1,000	Cd Ni Co	EV	10
Lead accumulators	Needs lead, zinc and manganese, all sufficiently abundant elements	30	-	-	-	100	-	-	1,500	Pb	EV	11

There is one new product type we have not taken up in this study. As more and more countries move towards renewable energy like photovoltaic electricity or windmills, the demand for intermittent electricity storage will rise (Takada et al., 2004, Takada 2013, Tasumisago et al., 2013). One alternative technology for that would be large lithium-based accumulator banks. Since the extent and design of such technology is still largely undecided, we have chosen not to include it here. But if implemented on a large scale, it could compete with units for automotive use.

Changing from business-as-usual to business-as-unusual.

There are a number of factors in the global lithium system that we can attempt to change:

1. It will be a very tough challenge to replace the global fleet of conventional combustion engine based vehicles (Amounts to 1.2 billion in 2016 and increasing) by electric vehicles based on lithium technology batteries or lithium-cobalt technology batteries. In addition would come amounts contained in other appliances, in addition to lithium locked in scrap, products in production, material in transit, and a substantial amount is lost as the degree of recycling remains too low. Table point out that the situation may be changes with powerful policy changes, prioritizing uses, minimizing losses and dramatically increase recycling.
2. We can assume the recycled fraction to increase from the base case of 50% to 65% or 80%. This would depend on a number of factors in how the products are designed, development of efficient recycling and refining technologies, and incentives and limitations designed through policies (Gaines 2014, Li et al., 2013, Outotec 2015 Richa et al., 2014, Sonoc and Jeswiet 2014, Tarascon and Armand 2001, Tytgat et al., 2008, Wang et al., 2014, Ziemann and Schebek 2010).
3. The extractable reserve is not sufficiently well investigated and mapped. Much of the resource estimates are based on good science, but the uncertainties are still so significant that we still only have a vague idea of the order of magnitude. Not really good enough for robust estimates. We think 73 million to is a reasonable estimate, but we cannot prove it, and there is a fair probability that the extractable amount may be both half or the double.
4. If a completely new battery technology should appear that does not need lithium, all our conclusions may change. The basic assumption made here is that these limitations and conclusions apply if we remain dependent on lithium. Inventing a better battery with no need for lithium, would create a new paradigm and need a new assessment.

Some of this discussion has been summed up in Table 4.1.5.4. It shows the electric vehicle potential as a function of lithium requirement and size of the extractable amount available. Lithium batteries, have the cathode based on several substances containing lithium; the most important are from top and down (Takada et al., 2004, Takada 2013, Tatsumisago et al., 2013, Gaines 2014) are shown in Table 4.1.5.5. All the batteries listed have greatly reduced lifetime when recharged fast at high voltage or used for fast discharges.

The numbers given above implies that producing 100,000 ton of lithium cathodes per year will require 850,000 ton per year of cobalt for alternative 2, and 280,000 ton of cobalt per year in alternative 3. Alternative 3 implies that 263,000 ton nickel per year is required. The annual cobalt primary production in 2014 was about 110,000 ton of per year. The total amount cobalt supplied to the market in 2014 was about 150,000 ton, and that is insufficient for both alternative 2 and 3. Thus cobalt will probably become limiting for high-performance battery production before lithium does.

The global nickel production is at present about 2.1 million ton per year and sufficient for the need. Thus, both lithium availability and cobalt availability may limit the maximum production capacity for the best category of batteries. Alternative 4 implies that we need 790,000 ton per year of manganese.

The present production is about 18 million ton manganese per year and this is fully adequate to cover the demand from batteries. For alternative 5, we need for battery production about 100,000 ton production consumption of lithium, a supply of 850,000 ton per year of iron and 1.34 million ton of phosphate per year. The supply of iron is 1,300 million ton per year in 2012 and for phosphate the production in 2012 was about 210 million ton per year. There fully adequate supplied for both phosphorus and iron for handling the battery needs. For alternative 3 there is sufficient amounts of titanium available.

The present lithium cycle is not optimized for efficient recycling in terms of industrial recycling technologies, infrastructures nor in public policies. Many products containing lithium are not being recycled at all today, leading to permanent losses that can never be retrieved. Some of these uses have low or no potential for recycling (glass and ceramics, lubricants, medical components), and possibly these used should be taxed significantly, to limit their use to the absolutely essential. For many of the uses of lithium, there are good substitution alternatives (Elshakaki and Graedel 2013). However, for medical use, the substitutability is poor, and for some uses no substitutes are available. In modern high-performance rechargeable batteries, no good substitute has been found so far, but that may change in the future.

The industrial response-delay to price we have set to 3 years. It is a bit short for a step-up, a bit long for a step-down. Delay for starting new mines used to be 5 years, it is now seen to lengthen to 7-15 years in some cases. Introducing such long delays will create production oscillations and a very volatile price. Because of improvements in recycling, the supply is kept up longer and the decline is slower than the decline in primary extraction. The supply will be sufficient for the demand to the new electric automotive demands, but after 2050, prices may increase significantly because of inability to satisfy demand. If our basic simulation assumptions are right, in particular the size of the extractable amount, the available lithium resources will be largely exhausted by 2300. The supply situation may be improved by additional efforts to increase recycling and product design to promote recycling ease.

Policy advice

This work was funded by the German government for support of a vision called “Industriland Deutschland - 2050” as one of the scenarios in the SIMRESS project. For this reason, our policy analysis will have a focus towards German resource policy. The policy advice derived from our assessment is for Germany in the context of global lithium supply and the effects on electric vehichles availability:

1. Without powerful resource policy changes on the global level, it will not be possible to replace 85% of the global vehicle fleet with electric vehicles based on the lithium battery technology. Strategies should be developed that reduce the ever increasing need for individual vehicles. A national German policy must be set into that global frame. A German domestic policy must be supported by a strong International policy on the resource issue. Research must be done to stay ahead in terms of technical and resource strategic developments.
2. There is probably substantial lithium resources in fossil salt brines in the large rock salt formations under the Central European area, known locations are known both in Northern Germany and in German Bavaria. In an initial stage this should be properly mapped and quantified, probably lead by a government-supported program with drilling to 3-4 km depth. Potentially, Germany may have substantial resources that are technically extractable. If so, some thought should be made that the lithium thus gained is recycled to a maximum degree and that export from it does not exceed the imports that can be recycled.
3. German industry has a globally leading competence in recycling of metals and development of industrial chemical processes, and should look into its potential for creating a substantial commercial platform for recycling lithium batteries on a large scale, serving at least a substantial part of the European demand, but exploring wider scopes. It could potentially make Germany a

major supplier of secondary lithium to the markets. This strategy would secure future German lithium demands.

4. Germany has a significant competence in electric storage technology and vehicle manufacturing and should enhance and support further development of such technologies to become world leading in mobile high-performance electric storage. A transition to front-line electrical vehicle technologies would be essential for the large vehicle industry in Germany. Several large companies may take charge and lead the way, but governance could probably promote innovation among small- and medium-sized enterprises to create a broad industrial base.
5. There are potential substitutions to lithium, however these are far from being usable and available. Several of the substitutes have sufficient technical performance, but with significant impairment to durability, efficiency or performance. Much more research is needed and this needs to be properly funded for a sustained period of time.

Electric temporal storage is an issue for the German National policy of the “Energiwende”. It designed to make Germany less dependent of fossil fuels and increase the fraction supplied by renewables drastically. Coal power and conventional uranium-based nuclear energy will possibly also be phased out. What remains will be different ways to harvest solar energy and lean and efficient energy use. Electricity production today occurs out of phase with consumption, and storage that can quickly take excess production under certain hours and then supply it back during other hours in 24 hour cycles, and well as in biannual storage cycles are in great demand.

4.1.5.9 Conclusions

We have found the extractable amounts (resources) of lithium to be far larger than what the scientific literature suggests it is. Instead of the often suggested URR if 40 million ton lithium, we get by quantifying extractable amounts across high, low and ultralow ore grades and including resources in salt brines, pegmatite hard minerals and lithium contained in different types of in clays, accumulating deposits in all countries, a maximum resource estimate of 116 million ton lithium, a middle best estimate resource of 73 million lithium to and a worst case low resource estimate of 34 million ton lithium.

We conclude from our investigations on resource data that 73 million ton lithium is the most plausible resource estimate for what is actually extractable, and this is our baseline modelling scenario. This amount is both technically feasible and economically possible. The simulations suggest that a production peak is likely to occur in the period 2050-2055, with a slow decline in supply and a steady increase in price in the decades after 2120. The model simulations suggest that a maximum level of primary extraction may be reached in 2020, followed by a flat plateau to 2120, and followed a slow decline in primary extraction output.

The low recycling fraction for lithium remains is a technical hurdle to be overcome and a political management challenge. Our simulations suggest it will stabilize around a maximum of 50% by market force mechanisms alone, which is not sufficient for solving the electrical vehicle transition challenge. That is not very good, and some considerations need to go into how that can be significantly improved.

The estimates for maximum number of battery units for vehicles, indicate that we have some challenges to solve:

1. It would be beneficial if the lithium use per battery unit could be reduced. Research and development is needed, to increase battery efficiency as well as capacity and safety. If alternatives are found, these are helpful only if they use less material of elements more abundant than lithium. Alternatively, one should develop batteries where more electricity can be stored per kg lithium.

2. Cobalt will be limiting for high-density batteries as lithium at the present level of use, cobalt abundance (extractable amounts) and production rates are lower than for lithium. However, cobalt extraction is dependent on copper, nickel and platinum extraction, and there are fewer options available for increasing the production significantly. Substitution for cobalt will be important, and manganese and nickel are acceptable substitutes with sufficient supply.

3. The degree of lithium recycling in the global system is far too low, and substantial improvements are required. This will need intensified efforts like those of UNEP, but also involvement of larger economic forces.

4. The estimates for the extractable amounts are uncertain, and the resources need to be far better investigated for extractability and extraction costs. However, finding larger potential resources will not be sufficient, they must also be technically recoverable, and this without large environmental damages. All in all, and Effort Return On Effort Investment (EROEI) must be performed. Substantially better degree of recycling is required to have a more long term secure supply.

4.1.6 SPECIALITY METALS sub-module – COBALT

4.1.6.1 Introduction

Cobalt belongs to a series of technology metals, which are produced in small annual amounts and are used as key components in important technologies that society depends on. Cobalt production is at present 97% from secondary extraction. In the past, mining of cobalt was from small cobalt mines with rich ore grade in Germany, Norway and Canada.

Cobalt blue is well known and cobalt is used since ancient times as a pigment for glass. Georg Brandt, a Swedish scientist, was the first to isolate cobalt as a new metal (Brandt 1735, 1746, 1748). He detected that cobalt was the causative element in cobalt blue. Cobalt is a silvery, white metal, the specific density is 8.86 g/cm3 and the melting point is high, 1,495oC. It is magnetic and resistant to corrosion. The earliest cobalt use was to colour glass, enamel and pottery.

One of the first industrial cobalt mines was located in Norway in the 16th century (Rosenqvist 1949, Liesmann 1994, Drake Resources 2015), but at the same time mines were developed in Sweden, Saxony (now a federal state in Germany) and Hungary. The first cobalt product was pigment for glass, later some of these mines also produced metal (Ramberg 2008). Today, there are few mines extracting cobalt only (Eckstrand and Hulbert 2007, USGS 2010). There is one mine in Morocco at Bou Azzer that produces about 1,800 ton per year; it has been operating for many years. The reserves are stated at 20,000 ton, the total hidden resources are unknown, but they are with certainty significantly larger (Newman 2011, Shedd 2002, 2012, 2015). The other dedicated cobalt mine opened in 2013 in Idaho, USA, and will produce about 1,500 ton per year of refined pure cobalt, about 1,500 ton per year of copper and about 100 kg of gold per year. The ore grade is 0.6% for cobalt and considered as “high grade cobalt ore”, 0.6% for copper and 0.5 g per ton rock for gold. The total resource is estimated to significantly more than 20,000 ton (Spencer and Searle 2015).

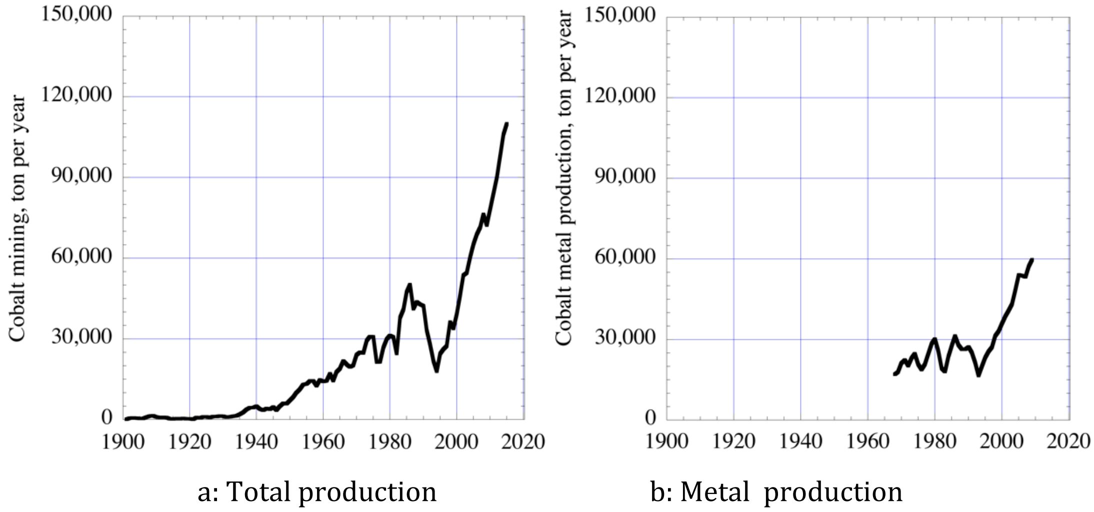
Cobalt has many uses, the most important are listed in Table 4.1.6.1 together with their market share, assumed market prospects and profitability. Some authors have highlighted risks of resource depletion looming (Meadows et al. 1972, 1974, 2005, Heinberg 2001, Johnson et al. 2007, 2014, UNEP 2011a,b,c,d, 2012, 2013a,b,c, Ragnarsdottir et al., 2012, Kwatra et al. 2012, Nassar et al. 2012, Bardi 2013, Elshaki and Graedel 2013, Sverdrup et al. 2013, Roland Berger 2015, Eliott et al. 2014, Mohr et al. 2014a,b, Northey et al. 2014), a few also considering cobalt particularly (Alonso et al., 2007, 2014, UNEP 2011a,b,c,d, 2012, 2013a,b,c, Ragnarsdottir et al. 2012, Sverdrup et al. 2013, Sverdrup and Ragnaradottir 2014). About 55% of the cobalt extraction is a by-product of nickel extraction, 35% of the extraction is a by-product of copper production and 10 % comes from other sources like platinum group metals extraction and chromium refining. Only 2.2% came from primary cobalt mining (Liessmann 1994, Hawkins 2001, Shedd 2002, 2012, 2015, Alonso et al. 2007, Eckerstrand and Hulbert 2007, Slack et al., 2010, Mudd et al., 2013a,b, 2014, Dawkins et al., 2015).

Table 4.1.6.1. Overview of different uses of cobalt. The metallic uses take 56% of the supply, the other uses take 44%.

Product group	Form	Fraction of supply, %	Trend in use and since when		Recycling	Value added to product
Super-alloys	Metallic	22	Increases	1995	Yes	High
Cutting tools	Metallic	11	Increases	1960	Partly	Medium
Magnets	Metallic	12	Increase	1990	Partly	High
Other applications	Metallic	11	Constant	-	No	Low
Batteries	Chemical	22	Increases	2000	Yes	Medium
Chemicals, catalyst	Chemical	9	Increases	1960	No	Low
Resins, pigments	Chemical	13	Constant	-	No	Low

Figure 4.1.6.1a shows the historical cobalt mining in tons per year since 1900 (data taken from USGS 2015). Figure 4.1.6.1b shows the cobalt metal production for the same time period, only about 65% the extracted amounts of cobalt is smelted into metallic form. Since year 2000, there has been a significant increase in metal production, which is used for super-alloys and specialty steels.

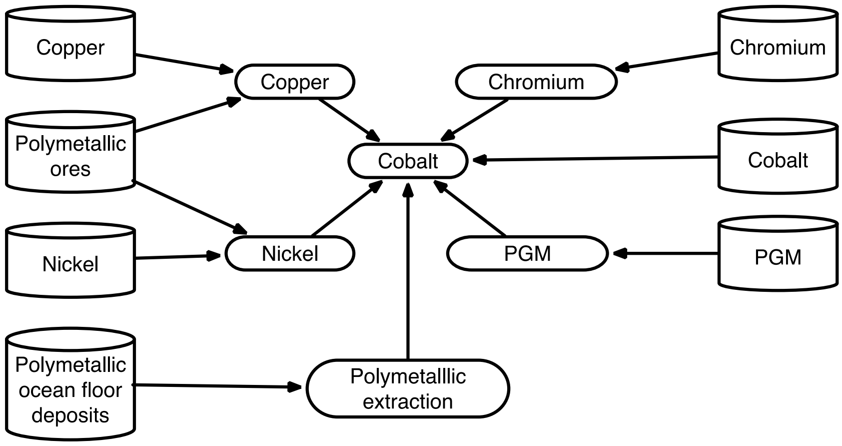
Figure 4.1.6.1. Diagram (a) shows the cobalt mining to date (USGS 2015). Diagram (b) shows the cobalt metal production for the same time period.



Source: Own Figure with data from USGS 2015, Lund University/University of Iceland

The new growing application for cobalt is for rechargeable accumulators (batteries) and in super-alloys for high performance jet turbines. Cobalt is a significant and required component in nickel hydride and lithium ion batteries. A flow chart for cobalt is shown in Figure 4.1.6.2 (note more elaborate flow chart for the whole system are given in Figure 4.1.6.4 and Figure 4.1.6.5). A very small amount (3%) comes from primary mining of cobalt, but this may be expected to increase in the future because of on-going projects and mineral prospecting in unchartered areas of the World (USA, Canada, Morocco). In 1960, most cobalt was used for alloys, but the materials consumption has increased significantly after year 2000. The cobalt use in chemicals is sharply rising with the production of larger lithium ion batteries, where cobalt is a major battery cell-stabilizing compound. The cobalt price has gone up in running prices, and it has fluctuated a lot since 1975. From 1945-1973, cobalt was considered to be a strategic metal and trade was partly government controlled. During that period, some countries had strategic stockpiles, and the market was not really free for cobalt. The market was thus not a free functioning, but one of partial command and control. Much of the cobalt originates from the copperbelt in Central Africa (about 50%), and the volatility in the cobalt supply and price is a reflection of the troubled political situation in that region.

Figure 4.1.6.2. Cobalt flow path from ores to society, from different types of deposits. Connection between the production of many different metals (oval rectangles) and the parent metal ores (cylinders) are displayed as a flowchart. This flowchart was used in the development of the COBALT sub-model in WORLD6. PGM is “Platinum Group Metals”.



Source: Own Figure, Lund University/University of Iceland

4.1.6.2 Objectives and scope

Our goal was to use BRONZE, STEEL and PGM (platinum group metal) sub-models to be operated from inside the WORLD6 system dynamics model and add a cobalt sub-module to the integrated structure. The objective is to use the WORLD6 model to explore what it would take to make the global cobalt supply system more sustainable.

4.1.6.3 Earlier modelling work

There have been several earlier attempts at modelling global cobalt extraction, cobalt demand and market supply rates. Some early predictions were made without any formalized simulation models, they represent educated guesses (Papp et al. 2008, Berger et al. 2011, Spencer and Searle 2011, Dawkins et al. 2015, Wilburn 2015). Roper (2014) used a fit to an empirical model analogous to the Hubbert’s model. When applied to cobalt it becomes a non-linear curve-fitting to a curve where the integral of the curve fits the assumed extractable resource. The Hubbert’s type of model used in singular mode is not applicable for cobalt because of cobalt’s dependence on the extraction of other metals (Sverdrup and Ragnarsdottir 2014a).

Mass Flow Analysis models have been used earlier for sustainability assessments for cobalt (Alonso et al. 2007, Halada et al. 2008, Gerst 2008, 2009, Elshaki and Graedel 2013, Graedel et al., 2004, Glöser et al. 2013a,b). As Flow models are easy to use, but do have some shortcomings we need to be aware of. Mass flow analysis models do neither incorporate systemic feedbacks, nor do they have market dynamics and they cannot generate metal prices internally to the models. Most Mass Flow Analysis models are linear in their fundamental function. Many of them are step-by-step mass balanced advanced one year at a time in spread-sheets (Graedel et al., 2004, Norgate and Rankin 2014, Gerst 2009, Mohr 2012).

We have in earlier studies developed systems dynamics models for a number of metals: gold, silver, copper, rare earths, aluminium, iron, platinum group metals, lithium (Sverdrup et al. 2002, 2014a,b, 2015a,b,c, 2016a,b, Sverdrup and Ragnarsdottir 2014a, 2017, Sverdrup 2017), manganese, chromium and nickel as well as copper, zinc and lead metals with a number of associated minor metals and elements. In these integrated dynamic models, some of the major advances were the development of a reality-based metal price model, allowing for price estimation from market fundamentals inside the models, without external forcing functions or calibration.

Table 4.1.6.2 shows an overview of cobalt demand for several current battery technologies. Limitations in cobalt supply may pose a problem for future production of rechargeable large batteries for electric vehicles, unless efficient substituting new technologies are developed (Sverdrup et al., 2016). If we assume we will need one billion battery units in use in the future, then the cobalt column in Table 4.1.6.2 shows million ton of cobalt required. The estimates can be seen to be in the range of from a very lowest estimate of 40 million ton of cobalt (NiCD battery) up to as much as perhaps as much 470 million ton of cobalt (LiCoO2 battery) (Goll et al., 2015). When the total supplied amount to the 2400 from land-based resources seems to be on the order of 32 million ton of cobalt and ocean resources about the same, we will have a significant problem with making that many battery units. The situation is similar for nickel, from 84 million ton nickel up towards 500 million ton of nickel would be required, when we have no more than maybe 250-300 million ton of extractable nickel in total (Sverdrup and Ragnarsdottir 2017).

Table 4.1.6.2. Different battery types and their properties and materials requirements (adapted from Sverdrup 2016). REE = Rare Earth Elements.

Components	Description of components and costs	Wh/kg	500 km range at 190 Wh/km, ingredient weight, kg						Limiting metal
			Li	Co	Ni	Mn	REE	Battery	
LiCoO2	The highest risk for self-ignition. Cathode material 18 $/kg, cost of cathode 27-36 $/kg.	170	56	476	-	-	-	530	Co
Li(ConNinMnn)O2 **Li(CovNiwAlz)O2** **(n=0.33, v=0.15, w=0.8, z=0.05)**	These batteries discharge and recharge faster. Cathode material 11 $/kg, cost of cathode 22-30 $/kg.	150 150	25 25	63 -	84 200	66 -	- -	240 240	Co, Ni Ni
LiFePO4	Cathode material 1.5 $/kg, cost of cathode 20 $/kg.	100	16	136	-	-	-	152	Li
NiREEO2 **La0.8Nd0.2Ni2.5Co2.4**	Complex mix of Ni, Co and REE.	50-70	-	350	400	-	50-300	850	REE, Ni, Co, Ni
NiCd	Simple design. Toxic waste by failure	40-60	-	40	500	-	-	1,000	Cd, Ni, Ni, Co

4.1.6.4 Data sources and estimations

The reserves estimates are based on classical geological estimates from survey data, and the allocation of extractable amounts according to ore quality, stratified after extraction costs (Sweeneye 1992, Neumeyer 2000, Tilton 1983, 2009, 2012, Tilton and Lagos 2007, Boschen et al. 2013, Singer 2013, Phillips and Edwards 1976, Fischer 2011, US Environmental Protection Agency 1994, US Department of the Interior, Bureau of Mines and Geological Survey, 1980). Some complication is added when ocean seafloor resources are considered, but we have used the available literature for ocean resource estimates and the opportunity cost approach when it was included in the model. Hubbert's model and burn-off rates were rejected for cobalt, as the production is dependent on other metals such as nickel, copper, chromium, platinum group metals as illustrated in Figure 4.1.6.2.

Basis for estimating how much can be extracted from ore deposits

Table 4.1.6.3 shows the relationship between metal ore grade, production cost and minimum supply price to society, as well as impact of price on the recycling in market supply as we have used it in the model. The task is to divide the ultimately recoverable resources (URR) into known resources and hidden resources, graded into ore quality groups as we have displayed in Table 4.1.6.3. Hidden means that the resources exist, but must be found with prospecting before they are moved to "known" and can be mined. This classification we developed for copper, silver and gold initially, has subsequently been applied to all metals (Sverdrup 2016, Sverdrup and Ragnarsdottir 2016, Sverdrup et al., 2014a,b,c, 2017). This is the format needed for the sub-models of the COBALT model system inside the WORLD6 model. The price range is based on data and the author's own professional experiences from copper, silver and gold extraction (Sverdrup et al. 2013c, 2014a,b, but see also Papp et al., 2008, Wellmer and Becker-Platen 2007, Wellmer 2008), and does vary between ore types and metal types. The assessment was done by studying the available scientific literature, and a number of industrial reports and annual reports. All the estimates have then been added up in Table 4.1.6.3. This assessment gives significantly larger estimates of the extractable amounts than conventional overview studies.

Table 4.1.6.3. General relationship between ore grade, the approximate production cost and minimum supply price to society, as well as impact of price on the recycling in market supply (adapted from Sverdrup et al., 2015a,b and a large yet unpublished database by the authors for about 35 different metals and materials). The values are illustrative of typical ranges.

Ore grade	Metal content		Extraction yield %	Extraction cost $/kg	Market recycling %	Energy need MJ/kg
	%	g/ton				
Rich	40-5		100-99	4-15	0-30	30
High	5-1		99-98	15-28	30-40	33-64
Low	1-0.2	10,000-2,000	98-95	28-50	40-48	120-160
Ultralow	0.2-0.04	2,000-400	91-95	50-100	48-54	400-600
Extralow	0.04-0.01	400-100	80-91	100-160	54-60	700-1,100
Trace	0.01-0.002	100-20	75-80	160-400	60-80	3,000-20,000
Rare	>0.002-0.0004	20-4	55-75	>400	80-90	>20,000

Cobalt systematically follows copper, nickel, chromium and platinum group metals in ore deposits (see Figure 4.1.6.2), and thus literature concerning these metals were also reviewed for estimates of extractable cobalt (Berry 2015, Boliden 2011, Brown et al., 2015, Cailteaux et al., 2005, Cobalt Invesriting News 2014, Cobalt 2014, Crowson 2011, Dalvi et al., 2004, Davis 2000, Eilu 2011, Hagelücken and Meskers 2009, Hannis and Bide 2009, Geoscience Australia 2009, Kesler and Wilkinson 2013, 2015, Nuss and Eckelmann 2014, Norgate and Rankin 2002, Prior et al., 2012, Rasilainen et al., 2010a,b, 2012, Idaho Cobalt Mine 2015).

An important basis for the assessment is how much cobalt is available for extraction. We build on three different assessments based on two methods for the estimation. (1) The work of Mudd 2007, 2009a,b, 2010a,b, 2012, Mudd et al. (2013a,b) made an important assessment of the available amounts extractable of cobalt. (2) Harper et al. (2012) and (3) Eckstrand and Hulbert (2007) mapped where these deposits are located and of what origin they are, providing important geological background material. The USGS has renewed their resource assessment and updated it (Slack et al., 2010, Shedd 2015). Further information was obtained by compiling and combining information from Ghose (2009), Gordon et al., (2006), Graedel and Erdmann (2012), Graedel et al., (2011) and the reports by UNEP (2011a,b,c, 2012, 2013a,b). The different estimates yield consistent results:

1. The estimation of extractable amounts of cobalt was done from copper, nickel, platinum group metal and cobalt production numbers. We ignored cobalt content in manganese and iron deposits, as cobalt is not extracted from these ores and the amounts of cobalt contained are limited. Our estimates are as follows:
    a. Total global annual production of cobalt is 110,000 ton per year (USGS 2015). Originating from nickel, the cobalt extraction is 61,000 ton per year. Cobalt content in primary nickel extracted is 3-3.3%. The extractable nickel known reserve is 73 million ton, the total known and hidden cobalt resources have been approximated by the USGS as minimum 182 million ton. Analysing nickel resources in another ongoing study we find it is probably closer to about 300 million ton (Mohr et al., 2014, see also in Table 4.1.6.4). In extractable nickel deposits, there are 6-9 million ton of cobalt (Nickless et al. 2014).
    b. Annual production of copper is 17 million ton per year, the cobalt flow from copper is 39,000 ton per year. The cobalt content in the copper flow is on the average about 0.23-0.25% in the data available. If we estimate that the extractable copper amount is about 3,700 million ton of copper (Henkens et al., 2014, Sverdrup et al. 2014b, 2016a), then the extractable copper deposits contain about 8.5 million ton of cobalt (8.5-9.3 million ton).
    c. Other ores with cobalt are:
        i. Cobalt production from PGM mining activity is about 10,000 ton/year, mostly in South Africa. The ores in South Africa are poly-metallic ores with copper, nickel, gold and platinum, where every metal contributes to the profitability of the extraction operation. Cobalt content in PGM ore is about 20 times the platinum content (40-100 g cobalt/ton rock, 0.4-1%). This suggests that there is about 4-6 million ton of cobalt in the extractable PGM deposits, about 200,000 ton PGM but maybe as much as 300,000 ton PGM, implying perhaps as much as 6-9 million ton of cobalt (Sverdrup et al., 2016b).
        ii. Chromium and manganese smelting and refining could yield maybe as much as 2,000-3,000 ton of cobalt per year (assuming a content of about 0.16%, or about 3-4 million ton). This much is not extracted at present, it would require a change in the chromium production technology in South Africa and other chromium and manganese producing nations.
        iii. Extractable low grade old mine dumps are estimated to contain about 0.3 to 0.5 million ton of cobalt (in South Africa and Peoples Republic of Congo, but also other places), but depending on Ni-Cu-Co content for economics of extraction. These are old dumps where older and less efficient extraction methods were used or where no attention was given to cobalt. This would be 0.5-1 million ton of cobalt all together.
    d. In total this adds up to 23.3-32 million ton of extractable cobalt. The historically extracted amount is somewhere between 2 and 3 million ton of cobalt, this gives us URR= 26-35 million ton.
2. USGS (2012) made an estimate in 2012 based on traditional assessment of reserves and resources, classifying them according to technical extractability and reality-evaluated extractability. The results are as follows:

   a. An estimate of 7.5 mill ton for the cobalt reserve, and 13 million ton of cobalt in the reserves base, total about 20.5 mill ton cobalt in all (average content 0.36%). Their justification for this estimate is available in the USGS website (Johnson et al., 2014, USGS 2014).

3. The Australian geologist Mudd and his coworkers (2013a,b, 2014), at Monash University in Australia, made a new assessment by going through all known and recorded deposits, including the latest data, evaluated their extractability and came up with the following estimates:
   a. 34 million ton of recoverable cobalt, 16-20 million ton of cobalt is estimated to be technically recoverable in: Nickel ores 5.5-6 million ton, poly-metallic ores with copper-zinc-lead, about 10 million ton of cobalt, other ores of varying poly-metallic compositions have an estimated content of about 0.5 million ton of cobalt. Not included by Mudd et al. (2013a,b, 2014) were estimates of about 2,000 million ton sub-sea ore with a cobalt content in the range 0.25-0.35%, translating to an approximate 5-7 million ton cobalt, bringing the total contained cobalt resources to about 32 million ton. These sub-sea deposits were concluded to be out of technical reach at present, and at present also too expensive to mine for their cobalt content. Excluded were also about 0.5 million ton of cobalt in old mine waste dumps, which would be technically feasible resources to recover. In Mudd's (2013) estimate, the implication is that 50% of detectable contained cobalt in deposits is extractable. His bottom line is that we have 16-17 million ton extractable out of about 34 million detectable. In 2014, Mohr et al. (2014b) revised their nickel resources estimate to be at least 300 million ton of extractable nickel. Our own recent estimates confirm this as the probable maximum amount for nickel URR. At a content of 3-3.2% cobalt in the nickel that implies the content is about 9-10 million ton of cobalt in such extractable resources, increasing the land-based URR to be in the range 32-36 million ton.

Table 4.1.6.4. Sources of cobalt from secondary extraction and their concentration in the parent substrate, and estimate of total extractable amounts. The content in the primary cobalt ore relates to rock cobalt content. All amounts are in million ton of cobalt.

Parent source of cobalt for extraction	This study				USGS (2012) (2015)	Mudd (2013)
	Xi Parent material Co content, %	Average parent metal resource	Extractable amount, million ton	Included in the model		
Copper	0.23-0.3	3,767	6-11	yes	10	6-10
Nickel	3-3.3	300	9-10	yes	9	9-10
Chromium	0.16-0.2	2,495	3-5	yes	2	-
PGM	2,000	0.2-0.3	4-9	yes	2	1
Primary cobalt ores	0.3-3.0	Not available	0.5-2	no	1	0.5
Old mine dumps	0.2-0.7	300	0.5-2	yes	-	0.5
Sums			23-39	-	20	22-29
Extracted earlier			3	yes	3	3
URR on land			26-41	-	25.5	25-31
Ocean floor deposits	0.25-0.35	2,000	7-372	yes	1483	5-7
URR, land and ocean			33-78	-	n.a.	30-38

From the overview given in Table 4.1.6.4, we can see that there is consensus on the total land-based extractable amount to be in the range 30-38 million ton of cobalt. Extraction of cobalt is energy-inten-

[2]We have assumed that ¼ is technically extractable, but at considerable extra cost, see Table 4.1.6.6.

[3]No assessment of extractability was done in that study.

sive as can be seen from Table 4.1.6.3, which shows energy requirements for different steps. The COBALT module inside the WORLD6 model was run with the Ultimately Recoverable Resources (URR), divided into quality classes as indicated in Table 4.1.6.3. Fundamental for this study is the estimation of how much metal can be extracted in total. This implies evaluation of how much metal is present, how much is technically extractable, disregarding price, energy or political obstacles. Both Mudd (2013) and USGS (2012, 2015, Slack et al., 2010) include ocean seabed cobalt crusts, but they evaluated the crust cobalt content very differently.

The studies of the USGS are among the most valuable sources of information due to their wide scope, long term perspective and large degree of consistency (Shedd, 2002, 2012, 2015, Papp et al., 2008, Slack et al., 2010, Berger et al., 2011, Newman 2011, Johnson et al., 2014, USGS 1980, 2014). The most recent USGS estimate is for cobalt detected in deposits, but with no serious assessment of extractability. There are a number of corporate prospects and pamphlets available on ocean seafloor mining, all very optimistic. However, these are aimed at potential investors and company shareholders, and may not represent the most down-to-Earth assessments. Ocean floor mining technology is currently being developed, much supported by advances in deep-sea oil exploration. However, such undertakings appear as technologically challenging and as capital intensive (Allsopp et al. 2013, Clark and Smith 2013a,b, International Seabed Authority 2015). Mudd (2013) estimates that a lot of cobalt is present, but very little is technically extractable.

A list of production rates and estimated resources are shown in Table 4.1.6.5. The main cobalt producing country is the Peoples Republic of Congo, in Africa. They produce one third of the global output (about 56,000 ton per year). The production in these countries is associated with copper production. They are followed by a number of countries in the range from 8,000 to 3,000 ton per year. The assessment was done by going through the available scientific literature and a number of industrial reports and annual reports, independent from the sources behind the information in Table 4.1.6.4. All the estimates have then been added up in Table 4.1.6.5. This assessment gives significantly larger estimates of the extractable amounts than conventional overview studies like those of the USGS.

Table 4.1.6.5. Assessment of total production, reserves (Known) and technically extractable resources (Hidden) in million ton of cobalt.

Country	Extraction 2015, ton/yr	Cobalt refining 2015, ton/yr	Deposit type or mother metal	Reserves million ton	Resources million ton
Congo DRC	56,000	3,300	Copper	3.400	14.5
China	7,200	42,000	Copper, Nickel, Iron	0.080	0.4
Canada	7,000	7,000	Copper, Nickel, Cobalt	0.240	0.9
Australia	6,500	6,500	Copper, Nickel	1.200	4.5
Russia	6,300	2,300	Nickel, Copper	0.250	0.9
Cuba	4,200	-	Nickel	0.500	2.1
Phillipines	3,700	-	Nickel	0.500	1.2
Zambia	3,100	5,300	Copper	0.270	1.2
South Africa	3,000	1,400	PGM, Nickel, Copper	1.200	4.5
Brazil	3,000	1,400	Iron, Copper	0.890	2.1
New Caledonia	3,500	-	Nickel	0.370	0.9
Indonesia	-	-	Nickel	0.080	1.0
Morocco	1,800	1,800	Cobalt ore	0.020	0.2
Madagaskar	2,950	3,000	Copper	0.500	1.5
France	-	220	-	-	-

Belgium	-	11,000	-	-	-
Finland	-	11,400	Nickel	0.250	1.2
Norway	-	4,500	Nickel	0.050	0.5
Germany	-	5,000	-	-	0.2
Sweden	-	2,500	Copper	0.100	0.7
Japan	-	3,700	-	-	-
India	-	1,100	Copper	-	0.2
USA	1,260	5,000	Copper, Nickel	0.180	1.0
Other countries	1,750	2,000	Copper, Nickel	0.200	0.34
Cameroon				0.050	0.66
Sum	110,000	118,200		9.180	39.5
Recycling, waste	-	12,000	Metal waste, batteries	2.200	2.5
URR	-	-	-	11.380	43.7
Ocean floor	-	-	Crusts, nodules	1.000	7-37.04
Global total	110,000	129,200	-	12.280	80.7

Table 4.1.6.6 shows the resources of cobalt from secondary extraction and their concentration in the parent substrate, and estimate of total extractable amounts. The content in the primary cobalt ore relates to rock cobalt content. Table 4.1.6.7 shows an overview of deep sea metal resources. The extractability estimates rely on a number of scientific studies (Allsopp et al. 2013, Beaudoin et al., 2014, Tilton 1983, Herzig and Hannington 1995, Clark et al., 2005, 2010, 2013a,b,c, Beckmann 2007, Boschen et al., 2013, Hein et al. 2009, 2010, 2013, Hein et al., 2009, 2010, 2013, Hoagland et al., 2009, Johnson et al., 2014, Martino and Parson 2011, Mudd 2013, Munos et al., 2013, Schmidt and Heydon 2013, Schmidt 2015, Roberts 2012, SPC 2012, Walther 2014, Zhou 2007, World Ocean Review 2015a,b,c) and this study. Table 4.1.6.8 shows the relationship between cobalt ore grade, the approximate production cost and minimum supply price to society, for a high and a low case for recoverability. The size of the cobalt resource is dependent on the estimates for the total copper and nickel resources. Thus, the total extractable resource estimates for these metals must be discussed. Johnson et al. (2014) attempt to estimate the copper resources around the world, stating that in addition to identified 2,100 million ton, they suggest that about 3,500 million ton of copper resources should exist, making a total copper resource of 5,600 million ton (URR). They considered porphyry copper deposits down to 1 km depth and sediments and strata-bound copper down to 2.5 km depth. However, the copper ore grade distribution in the resource and extractability remain unknown, and the proposed additional 3,500 million ton of copper cannot be added to the known resources of 2,100 million ton of copper without a proper extractability assessment. Such an assessment of extractability has not been undertaken. Assuming that 1/3 – ½ of the resources given by Johnson et al. (2013) are in reality extractable, then the total URR for copper would be in the order of 3,270-3,850 million ton of copper.

Table 4.1.6.6. Sources of cobalt from secondary extraction and their concentration in the parent substrate, and estimate of total extractable amounts. The content in the primary cobalt ore relates to rock cobalt content. All amounts are in million ton cobalt metal.

Parent source of	Parent metal resource	Cobalt content in mother material, %	Fraction of mines with Co extraction capability	URR million ton

[4]As much as 148 million ton of cobalt has been detected in deep sea cobalt crusts and manganese nodules (Slack et al., 2015). If we assume that 25% is extractable, that corresponds to a resource of 37 million ton of cobalt. That we probably should see as a high estimate (Smith and Heydon 2007, Hein et al., 2010, 2013). Mudd (2013) assumes that 5% of the ocean floor cobalt is technically extractable (7.5 million ton of cobalt).

cobalt for extraction	Million ton	Medium		Stand-ard	High
Copper	3,767	0.23	Variable; 0-0.8	8	16
Nickel	300	3.2	Variable; 0.2-0.7	10	14
Chromium	2,495	0.12	Variable; 0-0.5	2	4
PGM	0.3	2000	1.0	3	6
Primary ores	100	1.0	1.0	2	5
Ocean floor	148	4.7	1.0	7	37
URR	-	-		32	82

Table 4.1.6.7. Overview of deep sea metal resources

Type	Depth, meter	Metals	Co content, mill ton	Extractable fraction, %	Co extractable mill ton
Manganese nodules	4,000-6,000	Ni, Cu, Co, Mn	38	5-25	1.9-9.5
Cobalt crusts	800-2,500	Co, V, Mo, Pt	110	5-25	5.5-27.5
Sulfide deposits	1,400-3,700	Cu, Pb, Zn, Au, Ag	2	5-25	0.1-0.5
Sums					7.5-36.5

Table 4.1.6.8. Relationship between sea floor copper ore grade, production cost and URR.

Grade	Ore metal content, %	Extraction cost, $/kg	Hidden	Known	Sum
			Million ton cobalt		
Sea floor grade low	1-0.2	50,000	2	1	3
Sea floor grade ultralow	0.2-0.04	80,000	4	0	4
Sum			6	0	7

Additional information concerning mining technologies and technical development was taken from Rosenqvist (1949), Allen and Behamanesh (1994), Liessmann (1994), Hawkins (2001), Wagner and Fettweiss (2001), Shedd (2002, 2012), Alonso et al. (2007), Papp et al. (2008), Berger et al. (2011), Boliden AB (2011), Cobalt Facts (2011), Newman (2011), Harper et al. (2012), Wübbeke (2012), Walter (2014), Wagner and Fettweiss (2001), Dawkins et al. (2015), Searle (2015), and Wilburn (2015). The extractable amounts were set at the beginning of the COBALT model simulation in 1900, stratified with respect to ore metal content and relative extraction cost based on yield of extraction and energy requirements. All major model inputs on resources were extracted from earlier published information.

Harmsen et al. (2013) and Henckens et al. (2014) use an estimate for the total extractable nickel resource at 1,800 million ton, but we can find no scientific support for such a large estimate in the available scientific literature (Mudd et al. 2013, Mudd and Jowitt 2014). With no scientific basis in numbers, it would appear to be a wild guess. Johnson et al. (2013) agrees with this, and states that the copper resource estimates in Harmsen et al. (2013) and Henckens et al. (2014) are not suitable for any sufficiency assessments. We have done an extensive survey of the literature and found that there is more nickel available than what the traditional estimates suggests by about a factor of two. This is shown in

Table 4.1.6.4, where we estimate the nickel resources to be at least 183 million ton and more probably about 300 million ton (Mudd and Jowitt 2014). The full assessment of global nickel resources will be published in a separate study. Table 4.1.6.9 shows the relationship between primary hard rock copper ore grade, the approximate production cost and minimum supply price to society, and URR used in the COBALT model. This is a minor part of the URR for cobalt (URRMines=2 million ton).

Table 4.1.6.9. Relationship between primary hard rock copper ore grade, the approximate production cost and minimum supply price to society, and URR used in the model.

Grade	Ore metal content, %	Extraction cost, $/ton	Hidden	Known	URR
			Million ton cobalt		
Low	1-0.2	28,000	0.290	0.010	0.300
Ultralow	0.2-0.04	50,000	1.000	0	1.000
Trace	0.04-0.01	80,000	0.700	0	0.700
Sum			1.990	0.010	2.000

Table 4.1.6.10 shows an overview of the input data to the COBALT submodel fin WORLD6 or extractable amounts at the starting time for the simulations. Both land-based and ocean sea-floor resources were included. The ocean seafloor extractable resources are 5 times those of Mudd (2013), assuming 25% extractability.

Table 4.1.6.10. Input data to the model for extractable amounts, at the starting time for the simulations. Both land-based (Primary and secondary) and ocean sea-floor resources were included.

Secondary extraction					
Ore grade	------------- Million ton copper -----------			Ore grade Cu %	Kg Cu per ton rock
	Known	Hidden	Sum		
Rich	15	5	20	40-5	400-50
High	10	20	30	5-1	50-10
Low	100	1,250	1,350	1-0.2	10-2
Ultralow	15	1,200	1,215	0.2-0.04	2-0.4
Extralow	15	1,100	1,115	0.04-0.01	0.4-0.1
Sums	155	3,575	3,730		
Ore grade	------------- Million ton nickel ------------			Ore grade Ni %	Kg Ni per ton rock
	Known	Hidden	Sum		
High	29	1	30	5-1	50-10
Low	6	36	42	1-0.2	10-2
Ultralow	3	110	113	0.2-0.04	2-0.4
Sum	38	147	185		
Ore grade	-------- Ton platinum group metals ----			Ore grade PGM %	g PGM per ton rock
	Known	Hidden	Sum		
Extralow	2,000	38,000	40,000	0.01-0.005	10-5
Trace	0	47,000	47,000	0.005-0.002	5-2
Rare	0	64,000	64,000	0.002-0.0004	2-0.4
Nickel ore	5,000	18,000	23,000		5-0.5
Sums			174,000		
Ore grade	------------- Million ton chromium ------			Ore grade Cr	Kg Cr per ton

	Known	Hidden	Sum		rock
Rich	50	700	750	55-5	550-50
High	0	800	800	11-2	110-20
Low	0	1,000	1,000	2-0.5	20-5
Sums	50	2,450	2,500		
Primary cobalt mining					
Ore grade	------------ Million ton cobalt ----------			Ore grade Co %	Kg Co per ton rock
	Known	Hidden	Sum		
Low	0.01	0.290	0.300	1-0.5	10-5
Ultralow	0	1.000	1.000	0.5-0.1	5-1
Extralow	0	0.700	0.700	<0.1	<1
Sums	0.010	1.990	2.000		
Ocean seabed primary cobalt mining					
Ore grade	------------ Million ton cobalt ----------			Ore grade Co %	Kg Co per ton rock
	Known	Hidden	Sum		
Low	1	20	21	1-0.2	10-2
Ultralow	1	14	15	<0.2	<2
Sums	2	34	36		

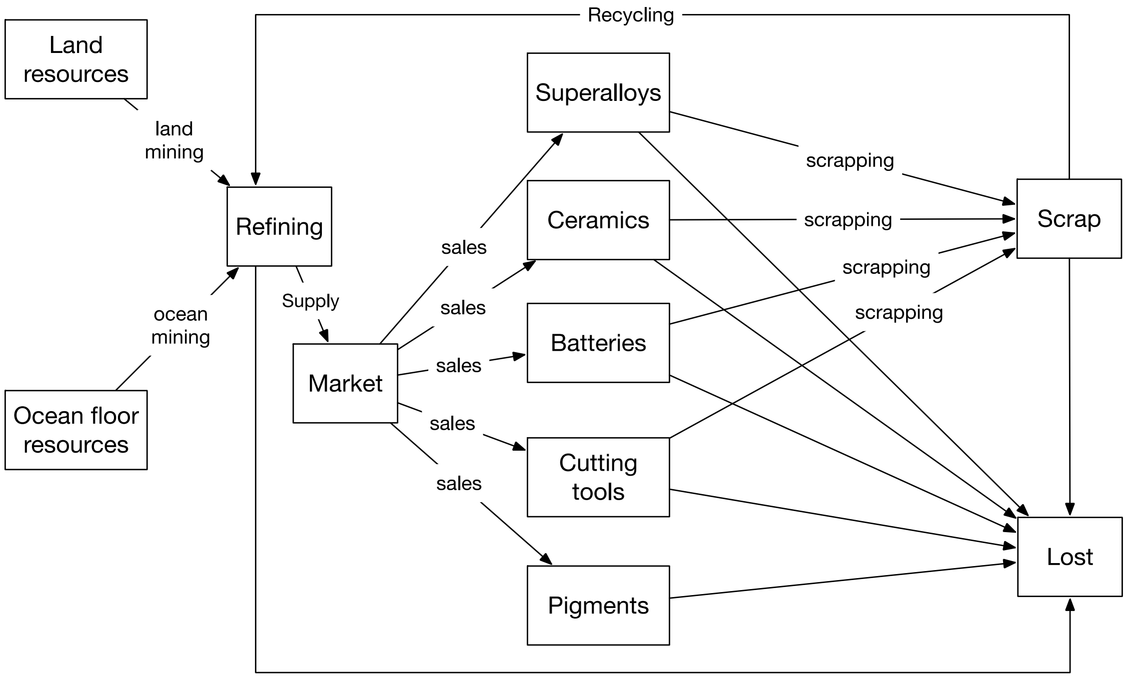
4.1.6.5 Model description

Structural description

The COBALT submodel is a model arrangement inside the WORLD6 model consisting of a system of sub-models. Figures 4.1.6.3 and 4.1.6.4 shows the connection between the production of many different metals and the parent metal ores displayed as a flowchart. The COBALT submodel consists of the following parts based on earlier models (Sverdrup et al. 2014a,b,) and new system dynamics parts (Sverdrup et al. 2015a,b,c) as shown in Figure 4.1.6.4:

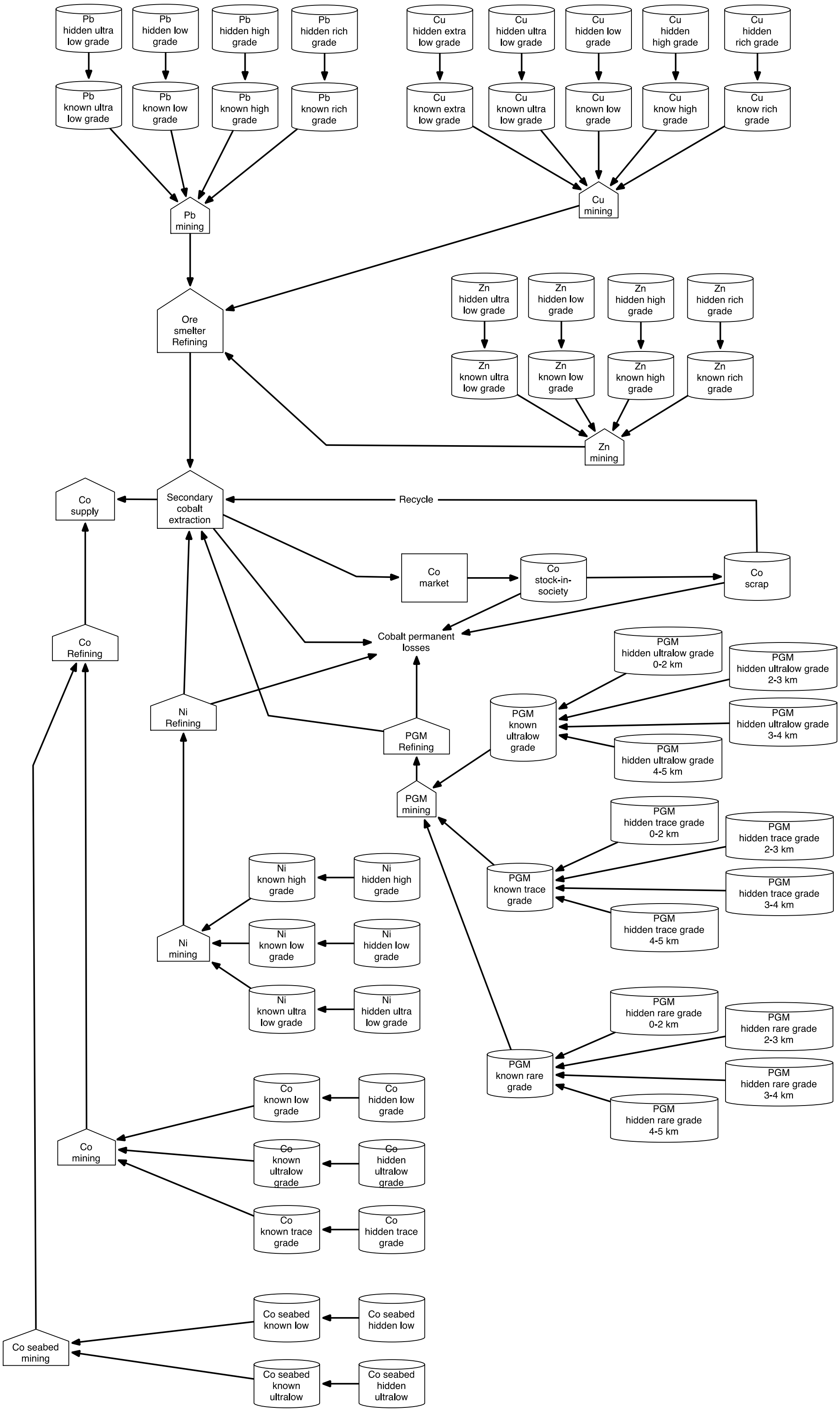
1. BRONZE submodel for, zinc, lead and many dependent metals (silver, antimony, bismuth, indium, gallium, germanium, tellurium, selenium, cadmium), including the COPPER and SILVER submodels (Sverdrup et al. 2014a,b)
2. STEEL sub model for iron, manganese, chromium and nickel, including the IRON submodel.
3. PGM submodel for platinum, palladium and rhodium, in conjunction with the STEEL submodel.
4. A submodel for the simulation of primary mining of cobalt.
5. Inside the COBALT submodel is a newly developed sub-module for the cobalt market cycle.

Figure 4.1.6.3. Flowchart for cobalt flow in society as pictured in the COBALT submodule in WORLD6. This is continued in Figure 4.1.6.5 and reflected in the STELLA structure shown in Figure 4.1.6.6 and Figure 4.1.6.7.



Source: Own Figure, Lund University/University of Iceland

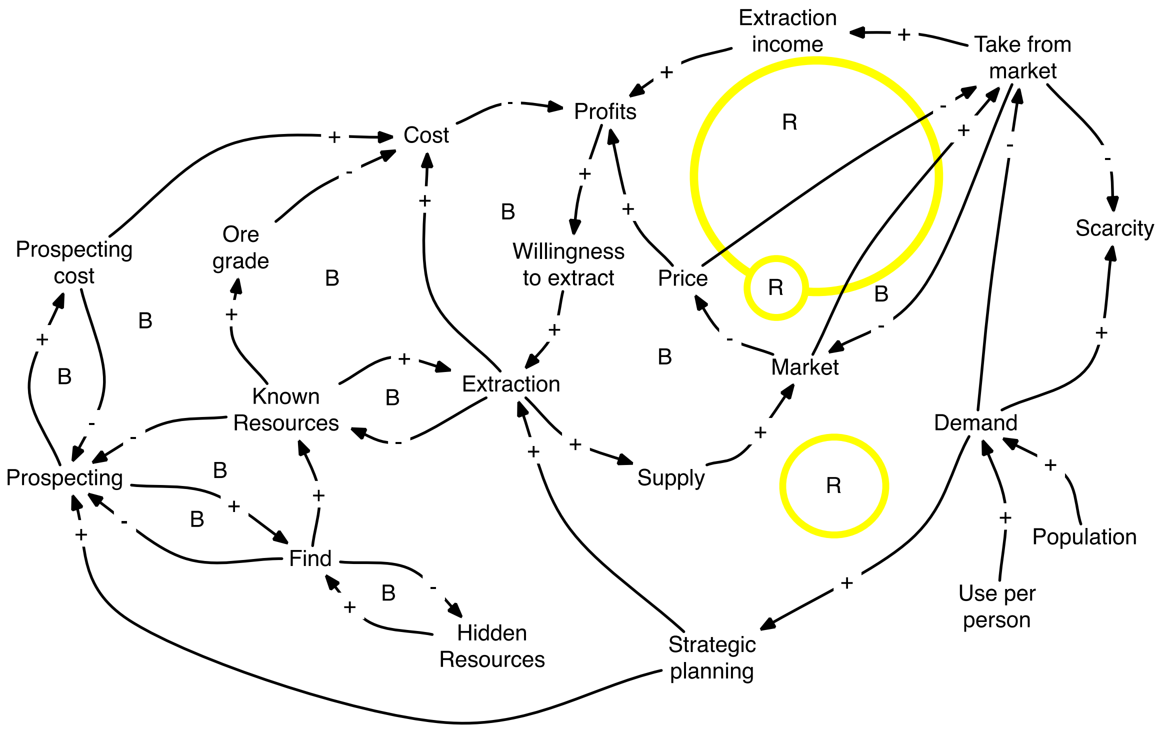
Figure 4.1.6.4. Flowchart for the COBALT submodel, including independent and dependent metals relevant to cobalt. The stocks sorted up into extraction cost classes are shown.



Source: Own Figure, Lund University/University of Iceland

Figure 4.1.6.5 shows the causal loop diagram defining the market dynamics in the model, where the price is an output from market mechanisms involving supply to the market, supply to customers from the market in response to demand. Note that supply from the market is not necessarily the same as the demand.

Figure 4.1.6.5. The price mechanism as implemented in the COBALT submodel is shown as a causal loop diagram. The diagram shows how supply to the market, the market amount, price and take from market and demand are all connected into one system of mutual feedbacks. In the model this is used to calculate the price dynamically at each week. The time step in the model is one week.

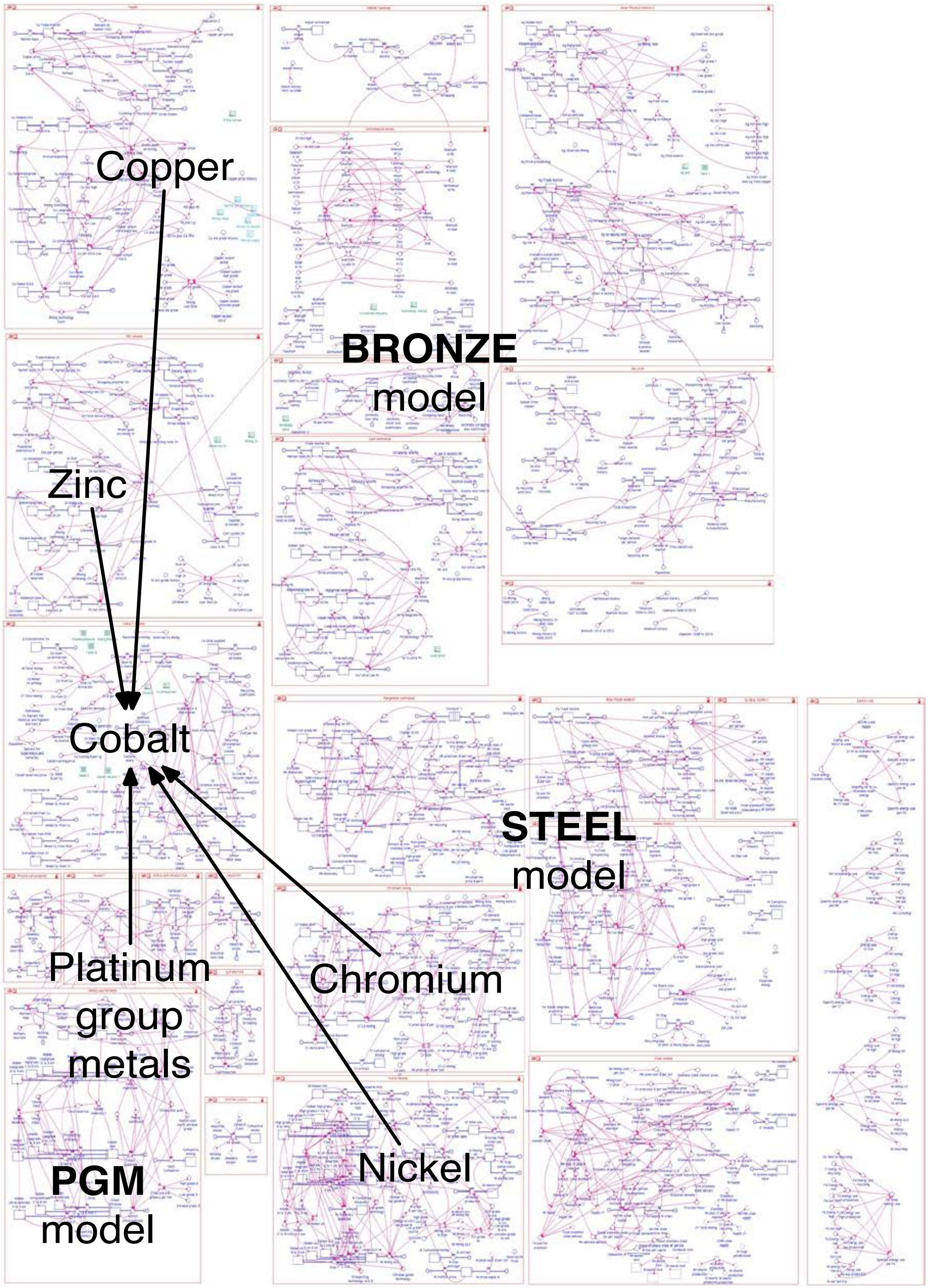


Source: Own Figure, Lund University/University of Iceland

Figure 4.1.6.6 shows the STELLA diagram for the whole COBALT submodel. All of this is embedded into the WORLD6 model, and the COBALT submodel is operated and run from within the WORLD6 model. The simulation of the primary mining of copper, zinc, lead and antimony and silver are in their own modules, the dependent metals are calculated from the production of these primary metals. Recycling from society and waste dumps provides a substantial flow of copper in comparison with rock mining. The COBALT submodel is numerically integrated using a 0.02 year time-step in a 4-step Runge-Kutta numerical method of integration in the STELLA® modelling software (Figure 4.1.6.7). That means we have values every week. Table 4.1.6.10 shows the input data for extractable amounts used to initiate the COBALT submodel. The main extractable amounts of cobalt are associated with nickel, copper and platinum group metals deposits.

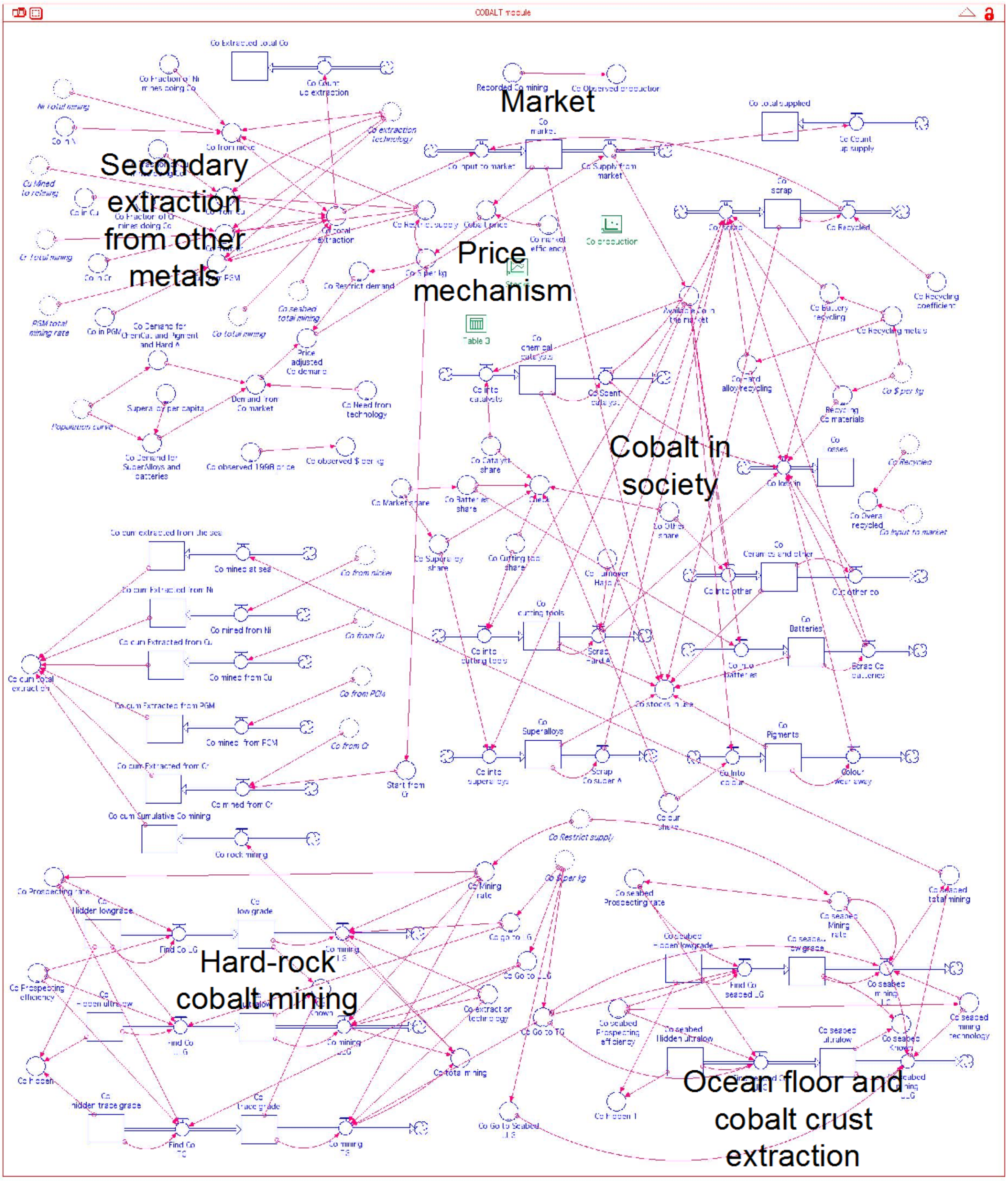
The full causal loop diagram for metal mining and copper in particular has been published earlier (Sverdrup et al., 2014a,b, 2015a,b). Population estimates from the WORLD6 model is used in the demand estimates (Ragnarsdottir et al. 2011, 2012, Sverdrup and Ragnarsdottir 2011, Sverdrup et al. 2012c) (see Figure 4.1.6.9).

Figure 4.1.6.6. The STELLA diagram for the whole COBALT submodel including the linked systems dynamics submodels BRONZE, STEEL and PGM (Sverdrup et al. 2016a) that we have developed earlier as a part of the WORLD6 model system.



Source: Own Figure, Lund University/University of Iceland

Figure 4.1.6.7. The cobalt sub-module of the WORLD6 model as shown in Figure 4.1.6.4 and 4.1.6.5, implemented inside the STELLA software shown in Figure 4.1.6.6. The titles overwritten on the diagram shows the different modules for secondary extraction, using copper, nickel, chromium and platinum group metals as input. In the model is also included hard rock mining in Canada and Morocco and future ocean seabed mining. Figure 3 shows the causal loop diagram implemented in this module.



Source: Own Figure, Lund University/University of Iceland

Rate equations and feedback functions

The rate of cobalt primary production is determined by the following equation:

$$rCobalt = rNi * XNi + rCu + rPGM * XPGM + rCr * XCr \qquad (4.1.6.1)$$

where rcobalt is the rate of cobalt extraction in million ton of metal per year, rNi the contribution from secondary extraction of cobalt from nickel production, rCu is cobalt from copper production, rCr is cobalt from chromium production and rPGM is cobalt from platinum group metals production. The values of the different Xi are given in Table 4.1.6.4, it is a unitless fraction. The extraction is first order proportional to the amount of substrate from the primary mother metals treated in refining. In addition to this the general mass balance principle applies with all inputs being balanced by the outputs and the accumulation in society:

$$Flow\ through\ society = Production + Recycling = Accumulation + Losses + Recycling \qquad (4.1.6.2)$$

For metal recycling we used the following equation (adapted after Sverdrup et al. 2014a,b):

$$R = 0.3 * log10(Price) + f(policy) \qquad (4.1.6.3)$$

for the applied feedback function for recycled fraction. R is the recycled fraction of the total supply, P is the price in US dollars per kg and f(policy) is a unitless function depending on regulations and social training of the agents in the market. With social training, we mean ways to have the consumers and actors have recycling and a more inherent behaviour, independently of monetary drivers. In the model, market price is determined by the amount present as tradable in the market. The market price curve for cobalt is derived from Figure 4.1.6.8a where we have plotted market inventories against the observed global market price for cobalt. The equation is in US dollars per ton:

$$Cobalt\ price = (21{,}867 * Market\ stock - 0.57 - 20) * g(E) \qquad (4.1.6.4)$$

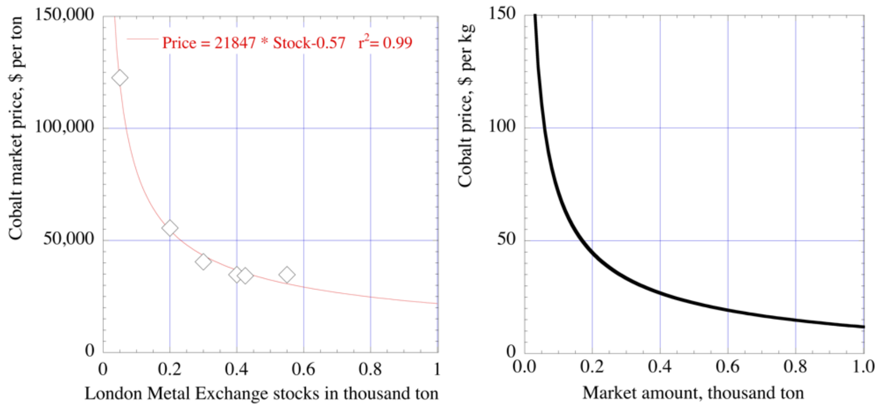
g(E) is the market efficiency, a unitless scaling function that varies with time. It has the value 0.5 until about 1960, and then increases to 1 by 1980, when the metal trade markets became organized and much more efficient, especially for cobalt. The correlation to the observed points between market stock and cobalt price is r2=0.99 (Figure 4.1.6.8a). The more material in the market, the lower the price. The amount of cobalt in the market depends on both the speed of delivery (supply) and the shipping from the market (demand). The cobalt amount taken from the market (F), as promoted by demand, was calculated as follows:

$$F = (DNM * P * (1 - ZM) + DNM * ZM)) * f(D) * g(S) \qquad (4.1.6.5)$$

F is the amount cobalt that is taken from the market. DNM is the specific cobalt demand per person. This amount is limited by what is available, where ZM is the fraction metallic cobalt in the demand. This demand was assumed to be higher for metals than for non-metallic uses. The factors DNM=0.00166 and DM=0.00332 implies a demand of 1.66 gram of cobalt per person for non-metallic uses and 3.32 gram cobalt per person for metallic uses. f(D) is the unitless feedback function limiting demand with higher price and g(S) is technological development, a unitless scaling factor. P is the global population in billion people taken from the population module in WORLD6. The actual rate of cobalt extraction is maximized by the rate of extraction of nickel, copper and PGMs, the parent substrates it is contained in.

Figures 4.1.6.8-13 show some of the feedback functions from the model that we use in the COBALT submodel. Figure 4.1.6.8a shows the empirically determined price curve for cobalt as a function of London Metal Exchange market stock. Figure 4.1.6.8b shows the applied feedback function for the fraction recycled materials as a function of the price, in US dollars, value-adjusted to 1998. The higher the market price, the more recycling efforts will be stimulated.

Figure 4.1.6.8. Diagram (a) shows the price curve found for cobalt, by plotting market tradable inventory against observed price. The red line is the fitted equation. Diagram (b) shows the actual curve modified from (a) and entered in the model as Equation (4). Curve (b) comes down a bit faster at higher market contents than the fit to data in diagram (a). The curves are derived from the private metal trading archives of one of us (Sverdrup) and are in the process of being published.

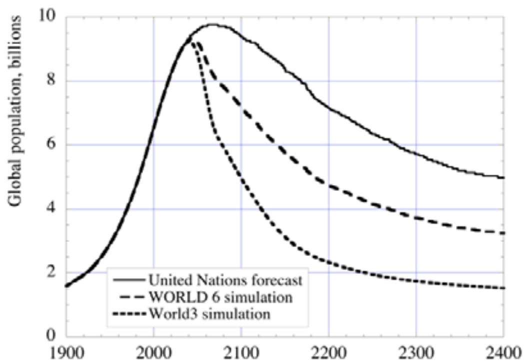


a b

Source: Own Figure, Lund University/University of Iceland

Figure 4.1.6.9 shows the population curve from the World3 sub-module and the alternative United Nations forecast for populations with no resource feedbacks of any kind. The World3 model simulation peak in 2040 at 8.6 billion people, the WORLD6 simulation peak at 9.1 billion people in 2045 and the UN forecast in 2065 at 9.8 billion people.

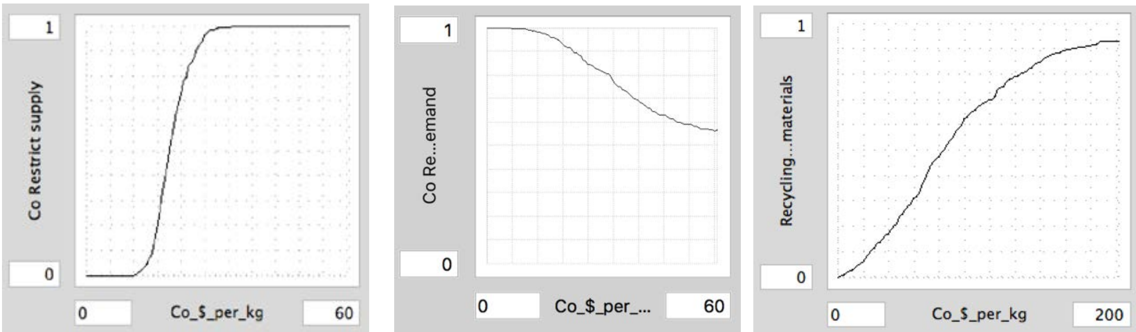
Figure 4.1.6.9. shows the population curve coming from World3 model, (W3 scenario), the new population estimates from WORLD 6 (W6 scenario) and the alternative United Nations forecast for populations with no resource feedbacks of any kind (UN scenario).



Source: Own Figure, Lund University/University of Iceland

Figure 4.1.6.10a shows the fraction of supply as recycled, a function of metal price. It shows how much market mechanisms make recycling as a % fraction of supply increase when the metal price goes up. Adapted with data from Lenzen (2008), Prior et al. (2013) and Dahmus and Gutowski (2007) for Sverdrup et al. (2014a,b, 2017). The diagram was used to parameterize Equation (4.1.6.3). Figure 4.1.6.10a shows the applied price feedback function for the restriction on supply. For the feedback, a one-and-half year running average is used. This running average is adapted, because society is not instantaneous in its response to change in the market price. Figure 4.1.6.10b shows the applied price feedback function for the restriction on demand. The higher the price, the lower the demand. Figure 4.1.6.10c shows the applied feedback function for the fraction recycled materials as a function of price.

Figure 4.1.6.10. The diagrams show the applied price feedback function for the restriction on supply (a) and demand (b) of cobalt. Diagram (c) shows the shows the applied feedback function for the fraction recycled materials as a function of price (see the CLD shown in Figure 4.1.6.3 where in the model this feedback is used).



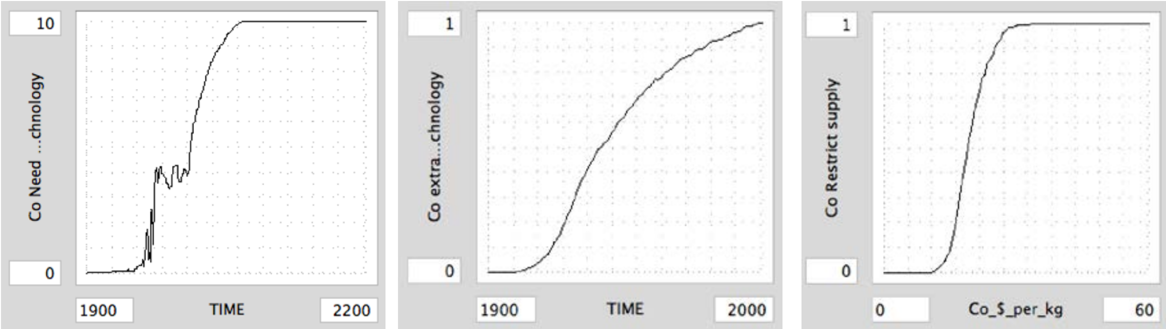
a b c

Source: Own Figure, Lund University/University of Iceland

We use the global population from the population model inside the WORLD6 to estimate the global cobalt demand from the per capita estimation (Sverdrup and Ragnarsdottir 2011) (Figure 4.1.6.11a). We use the predictions internal to the WORLD6 model, as well as the United Nations populations predictions which are politically correct, but probably flawed (Meadows et al., 2004, Greer 2008, Ehrlich and Ehrlich 2009, Heinberg 2001).

Figure 4.1.6.11a shows the applied increase in demand from the technological development of new industrial cobalt uses. The fluctuations reflect the instability of production caused by problems in unstable countries in Central Africa. Prices are in American inflation adjusted dollar value with 1998 as the reference year. The main market model used for cobalt is the same as used in our earlier models (Sverdrup et al. 2013c, 2014a,b, 2015a). Figure 4.1.6.11b shows the efficiency of the cobalt extraction technology and methods, and Figure 4.1.6.11c shows the applied restriction on hard rock cobalt mining and ocean mining from price feedbacks.

Figure 4.1.6.11. Diagram (a) shows the applied increase in demand per person from the technological development of new industrial cobalt uses. This is an input curve. Diagram (b) shows the efficiency of the cobalt extraction technology and methods. Diagram (c) shows the applied restriction on hard rock cobalt mining and ocean mining from price feedbacks.

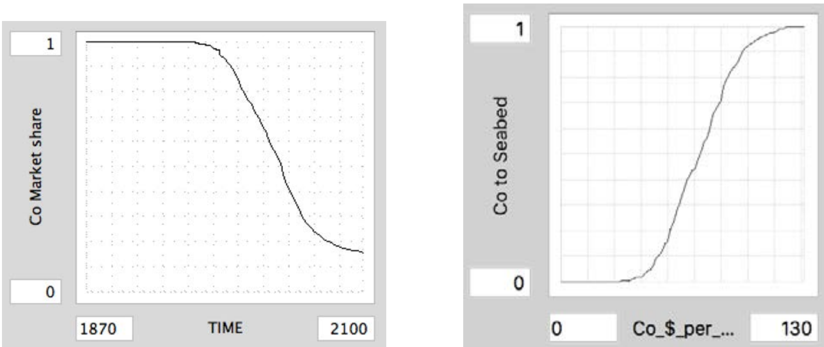


a b c

Source: Own Figure, Lund University/University of Iceland

Figure 4.1.6.12 shows the share of total cobalt supply going to super-alloys and batteries. The demand for cobalt to batteries comes from the battery module in the lithium sub-module of the WORLD6 model. A certain price level for cobalt must be reached before it becomes profitable to start mining from the ocean floor. Figure 4.1.6.12b shows the switching price for changing from land-based mining to ocean floor-based mining.

Figure 4.1.6.12. Share of total cobalt supply going to superalloys and batteries. Diagram (a) shows market share of cobalt; Diagram (b) shows the switching price for changing from land-based mining to ocean floor-based mining.



a b

Source: Own Figure, Lund University/University of Iceland

The market research organization CRU in their Cobalt Market Outlook 2015 (Spencer and Searle 2015), forecast a progressive tightening of the supply-demand balance out to 2020 and beyond; "A serious deficit is anticipated by 2020 which will probably generate a supply response from China and the Democratic Republic of Congo. Beyond 2020, a change in battery cathode chemistry which does not use cobalt could bring the market back into surplus." The market mechanisms cannot extract above the content potential in the mother metal flows. This limits the flexibility of the cobalt production. The primary cobalt production thus only partly responds to market mechanisms. We assume the concentration in the refinery residuals used for cobalt extraction to remain constant, even when the ore grade of the mother metal ore grade decline.

4.1.6.6 Results

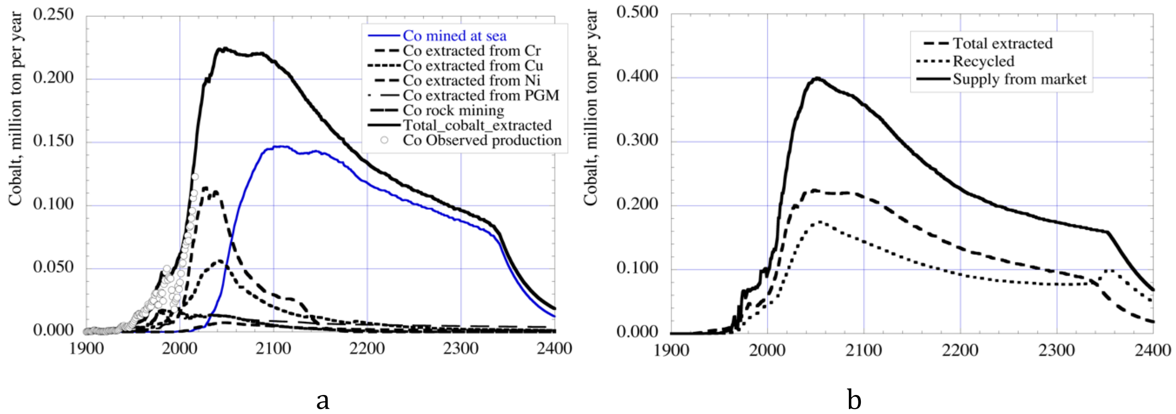
Basic model setup

We ran the COBALT model as integrated in the WORLD6 model with the three different population scenarios shown in Figure 4.1.6.9, the World3 scenario (W3) (using the same way to estimate the global population as was done with the original World3 1992 version), the base case scenario with the WORLD6 model (W6) and the United Nations population scenario (UN). The different scenarios result in different cobalt demands, and thus also different types of cobalt scarcity, three different cobalt total supply scenarios and cobalt market price trajectories as will be evident from the results shown in the following text. Runs with the COBALT submodel were made for the time 1900-2400 in order to see how long the available amount for extraction would last. We have done all the simulations using a resource estimate of URRLand=32 million ton cobalt on land and URROcean=34 million ton at the bottoms of the oceans, in cobalt crusts and massive sulphides, a total of 66 million ton (Table 4.1.6.4). We have included primary hard rock mining, secondary extraction from copper, nickel, chromium and platinum group metals (PGM), and from old mine waste dumps, and from 2020 on from ocean seabed mining, using the ocean seabed resource estimate of Mudd and Jowitt (2013). Including a larger estimate for ocean seabed resources did not change anything significantly before 2200, because of the higher costs of extraction. This favours land-based resources and recycling as primary sources of cobalt until after 2200.

Comparing the base case with the high, intermediate and low population scenarios

Figure 4.1.6.13a shows the primary extraction rate as reconstructed by the COBALT submodel for the past 115 years and predicted for the future from 2015 to 2400. Not all of the mined cobalt goes to metal, but a significant part is used for industrial chemicals, batteries and in pigments. The dots are observed values. It can be seen from Figure 4.1.6.13a that observed mining rates are the same as given by the model. To note is that they are not fitted, the mining data was not used to calibrate the model. Figure 4.1.6.13b shows the supply rate to society for cobalt as compared to the extraction rate and the cobalt recycling rate. The chromite deposits of South Africa and Zimbabwe are used less for cobalt extraction than what could actually be done. The decline is both due to declining resources, but also a decline in population size-driven demand after 2060 (see Figure 4.1.6.9).

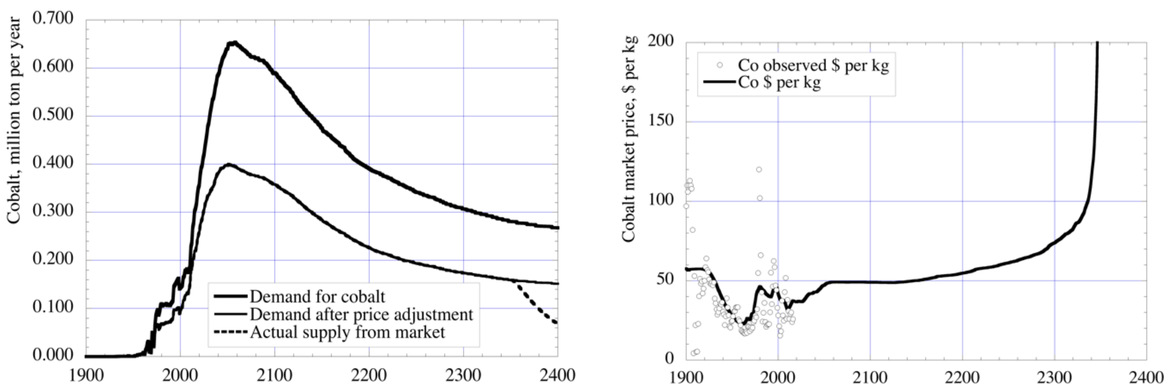
Figure 4.1.6.13. Diagram (a) shows the mining rate as reconstructed by the COBALT model, as well as from which mother metal. The dots show the actual mining rate (the model is not fitted to these). The importance of cobalt supply from secondary extraction from copper and nickel mining is evident. The circles are the observed extraction rates (USGS 2015). Diagram (b) shows the cobalt extraction, the supply to the market and the amount derived from recycling.



Source: Own Figure, Lund University/University of Iceland

Figure 4.1.6.14a shows the price adjusted demand for cobalt. Figure 4.1.6.14b shows the simulation of the price as compared to the observed data. The price shoots up when cobalt runs into hard scarcity 2360.

Figure 4.1.6.14. Diagram (a) shows how the price-adjusted demand (a) separates from the demand from the market. This is what we defined as soft scarcity. From 2360, hard scarcity is visible, when price-adjusted demand cannot be fulfilled. Diagram (b) shows the simulation of the price as compared to the observed data. The price shoots up when cobalt runs into hard scarcity 2360.

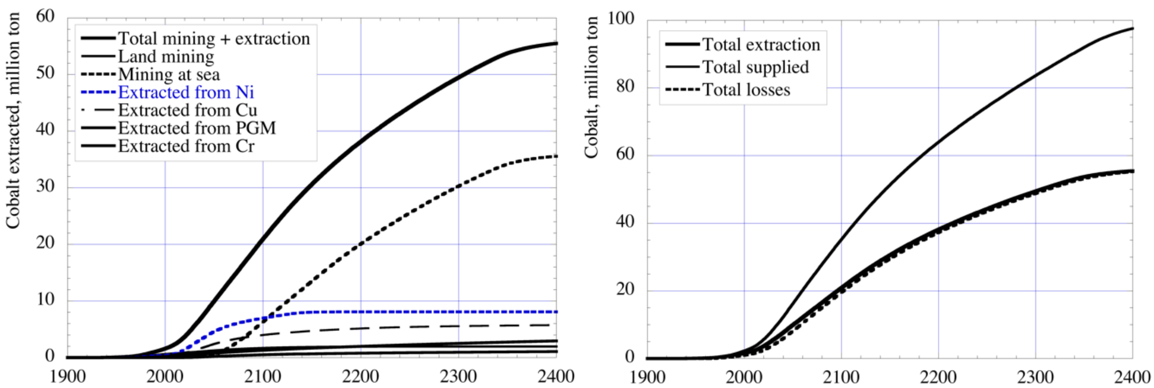


a b

Source: Own Figure, Lund University/University of Iceland

Figure 4.1.6.15a shows the origin of the cumulative cobalt extraction in million ton per year. The total amount of cobalt is extracted from copper mining, nickel mining, chromium mining and platinum group metals mining. Figure 4.1.6.15b shows the cumulative amount cobalt supplied to the markets, the total amount cobalt extracted to the markets and the cumulative cobalt losses to 2400.

Figure 4.1.6.15. The diagram (a) shows the cumulative amounts cobalt extracted from nickel, copper and PGM, and the total extracted amounts. Diagram (b) shows the cumulative amount supplied, the total amount extracted and the cumulative losses to 2400 AD.



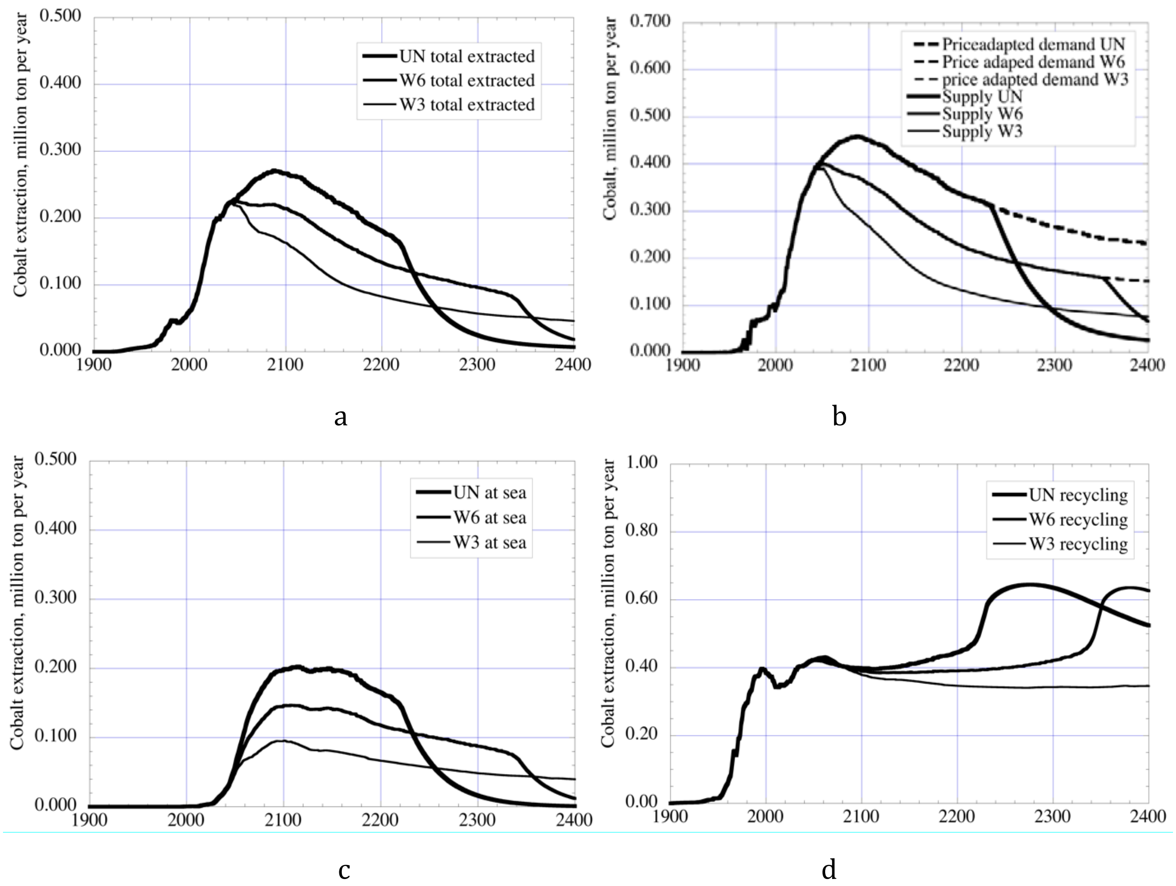
a b

Source: Own Figure, Lund University/University of Iceland

Figure 4.1.6.16a shows the total extracted amount for the different population scenarios explained in the text and in Figure 4.1.6.9. UN is the United Nations population forecast, W6 is the estimate of the WORLD 6 model and W3 is the updated World3 estimate using the model from 1992. It can be seen that the higher population leads to a higher demand, causing a higher extraction and a faster depletion of the extractable resources. Under the UN population scenario, cobalt extraction starts drastically decreasing by 2230, the price rises sharply leading to increase in recycling and mining at sea. The effect is also seen under the WORLD6 population scenario (by 2340), but not in the World3 scenario. Figure 4.1.6.16b shows the price-adapted demand (after adjustment for soft scarcity) and the actual supply from the market. It can be seen how hard scarcity for cobalt sets in under the WORLD6 and UN populations scenario. Hard scarcity sets in in 2230 for the UN scenario and in 2360 in the WORLD6 scenario, but never in the World3 scenario. Figure 4.1.6.16c shows the amount cobalt deriving from mining at sea. Figure 4.1.6.16d shows the recycling degree for the different scenarios. The recycling rates are dependent on the market price and have no additional social mechanisms or governmental actions involved. The recycling works reasonably well for metallic cobalt in super-alloys and other specific metallic uses since 1978-1980 when new and better technology was developed. For other use such as paints, pigments, chemicals or chemical catalysts, recycling is virtually non-existent.

Most of the cobalt in these uses gets lost or goes to landfills. The recycling rates are dependent on the market price, but will not go above 45% unless something more than just market mechanisms are used. In the time period from 2080-2120 the recycling rate increases as a market response to increased market prices. By market forces alone, recycling improves towards 2100, but by then far too much cobalt will have been wasted and lost diffusively. When compared to the recycling rates reported by the International Resource Panel (UNEP 2011a,b,c, 2012, 2013a,b,c), it becomes apparent that estimates of recycling rates at single point in time are very uncertain.

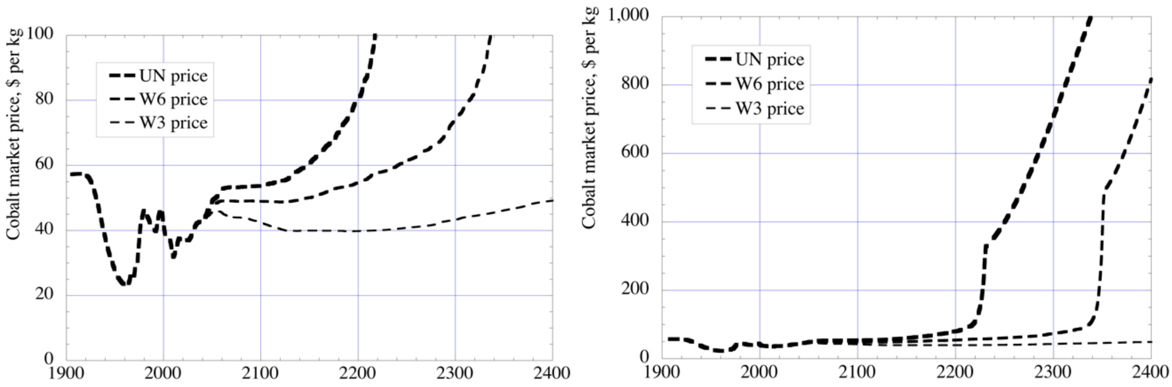
Figure 4.1.6.16. Diagram (a) shows the total extracted amount for the different population scenarios explained in the text and in Figure 4.1.6.9. UN is the United Nations population forecast, W6 is the estimate of the WORLD 6 model and W3 is the updated World3 estimate using the model from 1992. Diagram (b) shows the price-adapted demand and the actual supply from the market. Diagram (c) shows the amount cobalt deriving from mining at sea. Diagram (d) shows the recycling degree for the different scenarios.



Source: Own Figure, Lund University/University of Iceland

Figure 4.1.6.17a-b shows the price under different population scenarios in different scaling. The model gives price in 1998 value-adjusted prices in US dollars per kg cobalt. Scarcity becomes serious after 2050 with sharply increasing cobalt price in the intermediate and high population scenario. For the low scenario, nothing much happens with the price until after 2400, beyond our time horizon. The relationship between total extracted and total supplied is about 1.8, Factor X. Factor X is an indicator for the resource use efficiency in society. Factor X can be changed by changing the degree of recycling.

Figure 4.1.6.17. The diagrams show the price trajectory for cobalt for the different population scenarios, the same diagram but in in different scaling.



a b

Source: Own Figure, Lund University/University of Iceland

4.1.6.7 Discussions

Test and uncertainties

In Figure 4.1.6.13a we show a tests of the model on the cobalt extraction rate to the observed extraction rate (dots in figure), it gives a correlation of r2=0.96. Figure 4.1.6.14b shows the modelled price as US dollars per kg value-adjusted to 1998 for 1900-2020 as compared to the observed running current market price and the observed price in US dollars per kg value-adjusted to 1998 and the observed price in the markets. For the period 1920-1990, we get the level in price right (Figure 4.1.6.14b, r2=0.44), but all the observed price volatility in the market is not captured by the WORLD6 model. The accuracy of the price simulations is quite sensitive to the price-to-market-stock curve used, and the observed price curve shown (Figure 4.1.6.8b) has the best performance. The discrepancy between observed and modelled price is also caused by difficulties in setting historic demand levels, the short term variations in the demand and uncertainties in the demand-price and supply-price feedback functions applied in the cobalt model. The reason for this, may be the inability to capture periods of disruption in major producing countries a-priori as well as the effect or market speculation and financial manipulations which are not included in the model at present.

Some readers may react to the runs being run as far into the future to 2400. This may seem bold, but it must be remembered that in contrast to statistical data driven models, causally based models with coupled mass and energy balanced become quite well confined for substantial periods of time. This is supported by the fact that the same model (WORLD6) has been used to model trajectories from 1850 to the present (165 years) for copper, zinc, lead and iron to give examples, and from 1750 to the present (265 years) for silver and gold, being able to reconstruct the past production and market process with good accuracy (Sverdrup et al. 2002, 2012, 2014a,b). This was accomplished without excessive calibration, and these fundamental settings were kept throughout the runs (Figures 4.1.6.8, 4.1.6.10-12, Table 4.1.6.10).

Output sensitivity to population size, affecting demand

We have used the base run of the WORLD6 model for the estimates. As an alternative, we used the United Nations estimates. They are only based on demographic dynamics, and assume that the resource supply situation is sufficient at all times and will never ever become limiting. In the WORLD6 model, food supply as affected by phosphate availability, agricultural land development including soil erosion and energy availability is considered in the simulation of population dynamics. Cobalt has marginal effect on this, but the population has a large effect on cobalt. This leads to a very different population dynamics as a result of feedbacks from the global mass balances and energy balances. Our opinion is that a population model with no feedback from energy availability or food availability is not very realistic, and probably gives an over-prediction of the long-term sustainable population.

Ore grades and reserves

Over time, the grades for cobalt have declined in parallel with the decline in ore grade of the parent metals nickel, copper and PGM (Mudd 2007, 2009, 2010a,b, 2012, Mudd et al. 2013a,b, Sverdrup et al., 2017). Sudbury (Canada) and other similar nickel sulphide ores have today 1-0.5 kg nickel per ton rock, the laterite ores have 1-10 kg nickel per ton material, the cobalt ore in Katanga, Republic of Congo has a cobalt content 2-10 kg cobalt per ton rock. The bulk cobalt ore grade have gone down about a factor 3-5 over the last 40 years, following the declining ore grade of the mother metal ores for nickel and copper (Mudd 2007, 2009, 2010a,b, 2012, Mudd et al. 2013a,b). As the mother metal ore grades are low in the mother metal deposits, cobalt extraction indirectly implies a large environmental impact. If the cobalt reserves run out, and the metal has been diffusively lost, we will simply have no cobalt available when the stock in society is gone. The systemic cobalt recycling rate at present is far too low to be acceptable from a point of view of long term metal conservation, and it is appropriate to consider policy measures to strengthen cobalt conservation in society.

Substitution

Substitution is partially possible for some of the applications of cobalt (Slack et al. 2010). In magnets, several rare earths provide alternatives, and the use of cobalt there is flexible. Rare earths are more abundant than cobalt, and provide a real and good functional substitute. Substitution in cutting tools and in pigments is trivial and straight forward, and good alternatives exist.

In super-alloys, there are not any good functional substitutes for cobalt, and this may become a problem for large scale production of high performance jet turbines. It may only partly be replaced by nickel. The platinum group metals, niobium or rhenium may in some situations substitute for some of the cobalt, but niobium and rhenium are rarer in nature than cobalt, and thus the capacity for supplying the amounts needed for full substitution are not there. After 1980, super-alloy recycling increased significantly due to technical advances, improving cobalt supply. But further improvements are necessary (Alonso et el. 2007).

For chemical catalysts, some of the used cobalt may be substituted with platinum and palladium, but with a loss of performance and at a significantly higher cost. The platinum group metals are very limited in supply when compared to cobalt, and for any need in larger volumes, they will be totally insufficient. For some uses there is no functioning replacement for cobalt, but some work-around strategies is possible.

In lithium battery technology, cobalt plays an important role for battery durability, number of battery recharging cycles and stability of the storage capacity with time. The substitutes manganese, nickel or rare earths have inferior function, leading to heavier batteries. The use of cobalt in battery technology requires weight amounts comparable to or larger that of lithium and may exceed the available cobalt production rate in the near future.

The need for dynamic modelling

It is important to include dynamic market features and feedbacks in assessment used for policy development. Some policies depend on assumptions on market mechanisms being able to allocate resources most efficiently, and this needs to be modelled with feedbacks involved and tested in order to verify or falsify such assumptions. For those metals we have assessed so far (copper, zinc, lead, silver, gold, platinum, palladium, cobalt, lithium, iron, aluminium, manganese, chromium, nickel, tantalum, molybdenum, rhenium), market mechanisms were inadequate for sustainable management of the materials, calling for substantial governance actions and measures.

Ocean cobalt mining

Ocean bed mining of cobalt was included in the model. Because of cost constraints, the amounts being mined turned out to be insignificant as compared to land-based extraction for the period from now to about 2050. The model predicts that ocean mining will become significant when significant price increases will make extraction payable (Figure 4.1.6.16 and Figure 4.1.6.17). However, because of the high costs and low profit potential as compared to recycling, the amounts never become really significant until after 2050 for any of the scenarios. For the higher population pressures under WORLD6 and UN population estimates, sea mining can become significant and even run those resources to exhaustion. No technology exists for doing subsea mining on a large scale at present, what exist are proposals and early prototypes (Beckmann 2007, Allsopp et al. 2013). The long term environmental challenges have so far no solution (Herzing and Hannington 1995, Beckmann 2007, Zhou 2007, Clark et al., 2010, 2013, Allsopp et al. 2013, Mudd et al. 2013, Smith and Heydon 2013, Schmidt 2015). Large technological and environmental obstacles must be overcome for the ocean seabed alterative to become realistic (Beckmann 2007). The cobalt crusts are located from 800 meter depth to 3,500 meter depth in the oceans, and associated manganese nodules are normally from 1,000 to 5,000 meters depth, and it is not going to be an easy task to get them up to the surface (Clarke et al. 2010, 2013, Allsopp et al., 2013). The work will require advanced robotics and is likely to incur substantial cost that can only be carried out at a price significantly higher than the present market price.

The next step in model development

The next steps are possible, because the COBALT submodel is integrated in the WORLD6 model. By running it in conjunction with the LITHIUM submodel in WORLD6, we can take into account demand from lithium based battery technology. Inside the WORLD6 model are modules handling the metals that go into making super alloys for high performance turbines, air superiority jet fighter high performance engines and rocket engines (nickel, cobalt, niobium, molybdenum, rhenium, tantalum, hafnium and PGM). This allows assessment of any scarcity issues for these types of products.

The issue of population

Throughout the internal market dynamics inside the WORLD6 model, the size of the population is very important. Demand is composed of use per capita, the efficiency of use and the number of consumers. The number of consumers is another name for the global population size. Thus, the issue of population cannot be ignored. We have adopted three population scenarios, the UN forecast, the WORLD6 estimation and the Word3 estimation from 1972. We think the World3 1972 estimation is somewhat too low, and we think the UN estimate is a naïve estimate ignoring all feedbacks from resources like energy, land and phosphorus. Our own opinion is based on what we consider to be the best available estimate according to present scientific knowledge, considering world systems feedbacks, as done with WORLD6.

4.1.6.8 Conclusion

The present use of cobalt shows a low degree of recycling and systemic losses are significant. The extractable resources of cobalt are not very large, about 32 million ton extractable from land-based resources and about 34 million to from ocean seabed resources, a maximum of about 66 million ton. Some time after 2200, the supply of cobalt will have run out under a business-as-usual scenario, depending on the population scenario adopted. Too much cobalt is lost if only market mechanisms are expected to improve recycling, and unnecessary cobalt is wasted if no policy actions are taken. We can conclude that the market mechanisms alone do neither have the goal nor the competence to make cobalt use sustainable without governmental interventions and regulations. A science-based solution-oriented policy is needed to correct the situation before it is too late and too much cobalt has been lost. Failure to take this message in will risk that society one day will be without cobalt. In order to conserve cobalt and allow it to be available for coming generations, present policies must be changed and the large observed losses mitigated within the next decades.

4.1.7 SPECIALITY METALS sub-module – MOLYBDENUM and RHENIUM

4.1.7.1 Introduction

Molybdenum, rhenium are metals that are key for making specialized stainless steel alloys that preserve strength and corrosion resistance at high temperatures, important for advanced technologies used in civil and military applications. Molybdenum and rhenium are associated in nature, and that share a number of special metallurgical and chemical properties. These metals are valuable and sought after for manufacturing of super-alloys and as catalysts for chemical reactions. The demand for these metals is normally higher the supply, and they are already scarce with a high market price. Scarcity of natural resources in the future may cause trouble for continued technology development and the further development of the economy in the long term (Hubbert 1966, Acemoglu and Robinson 2013, Bardi 2013, Busch et al., 2014, Heinberg 2001, Hirschnitz-Garber et al., 2015, Morrigan 2010, Meadows et al., 1972, 1974, Nickless et al., 2014, Nassar et al., 2012, Ragnarsdottir et al., 2015, International Molybdenum Association 2013, 2015, Avon Metals 2014, Robinson and Menzie 2012). This study intends to address some of these issues. They find use in turbine blades, jet engines, rocket engines, nuclear energy technologies and specialized chemical environments.

Super-alloys mainly rely on dispersion of coherent L12 γ'-austenitic precipitates in a face-centred cubic γ metal matrix for excellent high temperature mechanical properties. Corrosion resistance is enhanced by components creating corrosion resistant oxide scales at high operation temperatures. Compositions of super-alloys have continued to evolve to be able to be used under increasingly challenging operation conditions. Additions of rhenium, ruthenium or other exotic materials have significantly improved creep strength and resistance to corrosion. Nickel-based alloys are the most common, but other alloys based on cobalt have been developed in recent years. Further research focuses on more alternatives using high contents of metals such as chromium, niobium, molybdenum or platinum. A strategically important use of these metals is in high temperature section of turbines and jet engines. The superalloys may make up as much as 50% of the jet engine weight, which shows how dependent modern jet engines are on these metals (Reed 2006, Makineni et al., 2015a,b).

4.1.7.2 Objectives and scope

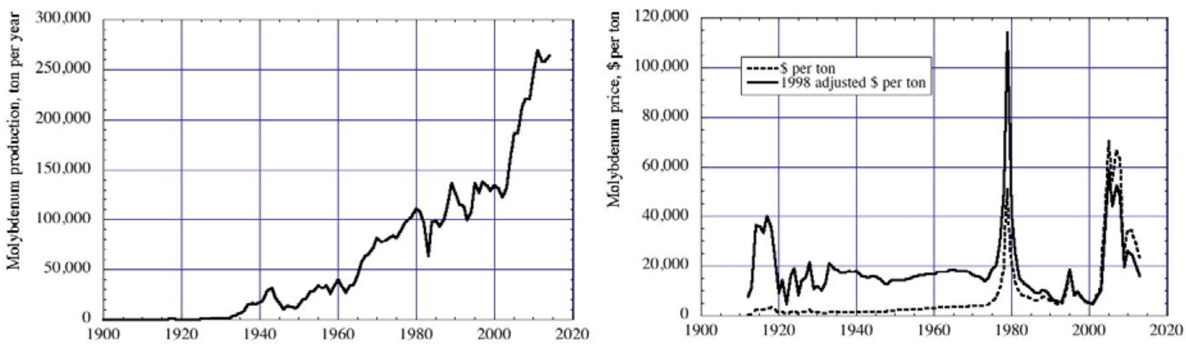
Develop a model for the production and cycling of molybdenum and rhenium in society. We intend to compile estimates of the available reserves and reserves for extraction, the present annual production and historical production and to give an outline of what these metals are used for and where they are strategically important. The objective is to use a systems dynamics model to explore the term sustainability of the supply system and its limitations.

4.1.7.3 Background Molybdenum

In 1778 the Swedish chemist Carl Wilhelm Scheele found a new element, named molybdenum for the mineral in which it resided, and from which can be isolated (Scheele 1778). The Swedish chemist Peter Jacob Hjelm isolated metallic molybdenum in 1781, but published it some years later (Hjelm 1788, Hess 1924, Cunningham et al., 2008, Ludington and Plumlee 2009). Molybdenum is found in wulfenite (PbO*MoO3) and powellite (CaO*MoO3), but the main commercial source for extraction is molybdenite (MoS2), which is a sulphide ore (Cunningham et al., 2008, Ludington and Plumlee 2009, Magyar 2004, Molybdenum 2015). There are two types of molybdenum mines. Primary mines where the main focus is molybdenum extraction and some other metals are made as by-product, and mines where the main focus is most of the time copper, but sometimes also nickel-copper mines, and where molybdenum is a by-product. Some of the first mines for molybdenum and niobium were in Knaben Gruver, Norway. Very little molybdenum was produced before 1930 (Figure 4.1.7.1). About 88% of molybdenum produced today is used in metallurgical applications such as specialty alloys, super-alloys and stainless steels, with the rest of molybdenum used as compounds in chemical applications (12%). The industrial use of molybdenum in structural steel is about 38% of the annual supply, manufacture of stainless steel is 22% of the annual production, tool and high-speed steels consume 8% of it, production of nickel alloys consume 2% of it, cast iron uses 8% of it, molybdenum elemental metal production is about 5% of it, and super-alloys consume 5% of the annual production (International Molybdenum Association 2013, 2015; Koizumi et al. 2003, Molybdenum 2015). The ability of molybdenum to withstand extreme temperatures without significantly expanding or softening makes it useful in applications that involve high temperatures, such as military armour, weapons, aircraft parts, electrical contacts, industrial motors and filaments. Molybdenum has good resistance to chlorine, and stainless steel that is in contact with seawater needs to contain molybdenum. Many types of high-strength steel contain 0.25% to 8% molybdenum. Many high-strength steel alloys contain from 0.25% up to as much as 8% molybdenum. In 2014, more than 45,000 ton per year of molybdenum was used for alloying in stainless steels, tool steels, cast iron and temperature-resistant super-alloys out of a total extraction of 265,000 ton per year. Figure 4.1.7.1 shows the recorded extraction rate for molybdenum and the observed market price. Around 1978-1981 and 2008-2010, the molybdenum market was subject to severe speculation and possibly illegal use of derivatives to manipulate the market, putting normal market price mechanisms out of function. Molybdenum is processed to three categories of products. These are (International Molybdenum Association 2013, 2015, Molybdenum 2015):

1. Technical oxides (MoO2) which are added in the specialty steel-making process
2. Ferromolybdenum (FeMo), is produced with the thermite reduction of technical oxide with iron present. Used for alloying into specialty steel and stainless steels.
3. Pure metal is used for alloying. Several super-alloys cannot tolerate iron content, and for this purpose, pure molybdenum is produced and mixed with the cobalt- or nickel-base alloy. The pure molybdenum metal is made from the technical oxide (MoO2) or the ammonium molybdate ((NH4)2MoO4), by reduction with hydrogen at enhanced temperature. The result is molybdenum sponge that is annealed or sintered to solid pieces. This is then used in the alloying.

Figure 4.1.7.1. The world production of molybdenum, and the inflation-adjusted market price (USGS 2015).

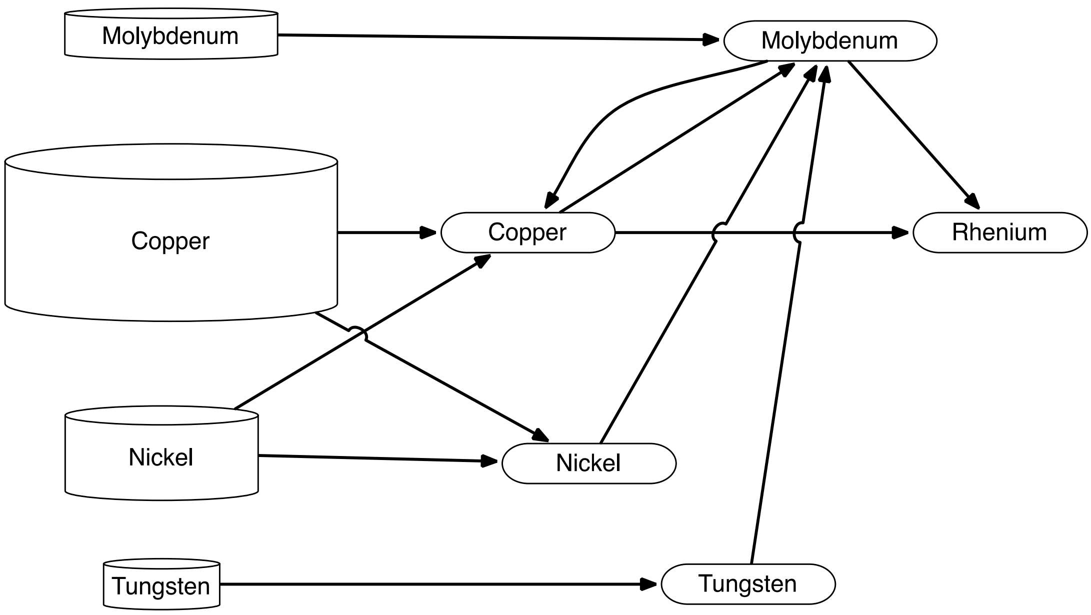


a Molybdenum b

Source: Own Figure with data from USGS 2015, Lund University/University of Iceland

Figure 4.1.7.2 shows a flow chart for where the produced molybdenum and rhenium comes from. Some molybdenum is recycled (Magyar 2004a, International Molybdenum Association 2013, 2015, Molybdenum 2007). Molybdenum is used in super-alloys with rhenium, niobium and tantalum to create rocket nozzles and the high-performance jet engines fight aircrafts designed for total air superiority. With special ceramic coatings on single crystal turbine blades, with special cooling strategies, these blades may tolerate working at 1,600-1,700oC in a high-performance jet engine. It is also used for satellite skins, and ship rudders and guidance planes. Molybdenum has a small neutron absorption cross section, and good corrosion resistance to liquid metal in fusion reactors. The largest producers of primary molybdenum from specifically molybdenum mines are found in Canada, United States, China, Papua New Guinea and Australia. Norwegian and Swedish mines are closed at the moment, these were among the first mines opened. Many large copper and nickel producers have a significant production of molybdenum as a by-product from copper and poly-metallic deposit mining. Molybdenum has very variable ore grade between the different mines where it is excavated and extracted, from the best at 4% through the more usual at 0.3-1.5%, but occasionally molybdenum is still taken out along other metals at lower ore grades.

Figure 4.1.7.2. Flow chart showing where molybdenum and rhenium comes from in terms of ore deposits and mother metals. Containers are extractable resources of mother metal, ovals represents production and arrows are metal pathways.

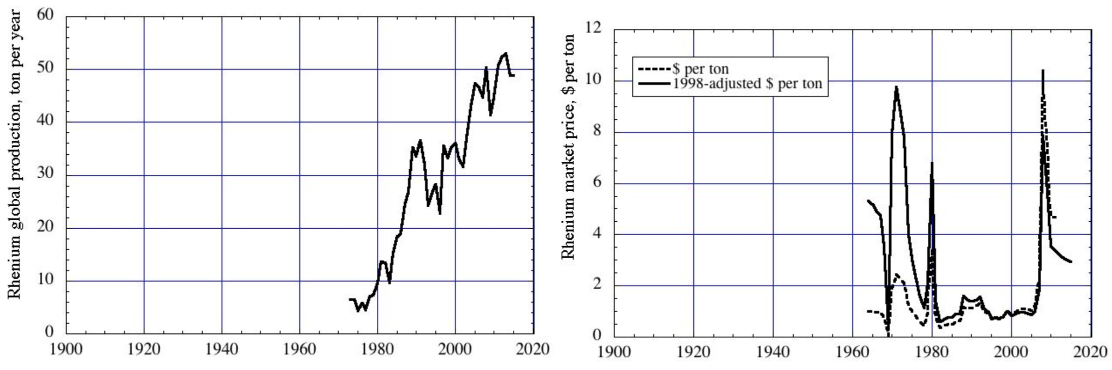


Source: Own Figure, Lund University/University of Iceland

4.1.7.4 Background Rhenium

Rhenium was discovered in 1925 by a team of German scientists; Walter Noddack, Ida Tacke-Noddack, and Otto Berg (Noddack and Noddack 1931). It is a precious metal at room temperatures, and has a notable high melting point and high specific density, in some ways resembling the platinum group metals. Rhenium is found in contents of 0.001-0.2% in molybdenite ore, on the average 0.02% (Magyar 2004b, Fleischer 1959, Polyak 2011), it is very rare. Large scale copper, nickel and tungsten extraction sometimes yield small amounts of rhenium as a by-product when the refinery is set up with the right equipment for it. It is one of the rarest metals on earth and whenever it is found, it is always in very low concentrations, mostly associated with another metal. The annual global rhenium production is 50 ton per year (2014). Figure 4.1.7.3 shows the world production of rhenium, and the inflation-adjusted market price as far as the records go (USGS 2015). Rhenium remained a curiosity metal until it became important for high performance jet engines and high efficiency turbines in the 1980's (USGS 2015, see Figure 4.1.7.3). Before 1970, there was no real trading market for rhenium and no real market price, and the technologies where it is needed were not yet invented. The metal was produced to order by a few specialized firms at internally negotiated prices. The rhenium production in 2015 is not exactly known according to the USGS (2015) commentary, it is somewhere between 48 and 58 ton per year. It is used in rocket engine nozzles and jet turbine blades to the largest part (78%).

Figure 4.1.7.3. The world production of rhenium, and the inflation-adjusted market price (USGS 2015). The production of rhenium is closely related to the molybdenum production and the new demand for it from technologies involving superalloys.



a Rhenium b

Source: Own Figure with data from USGS 2015, Lund University/University of Iceland

Much of the rhenium use and stockpiling is military secrets, thus the difficulty in estimating how much rhenium is really in use and circulation. The rhenium production is dependent on the production of molybdenum and copper mainly. The largest market suppliers are perhaps Poland and Germany, because this is where the refining capacity is best developed. Potential additional future rhenium sources may be nickel, platinum and tungsten mining where it may be extracted as a by-product (Magyar 2004b). It has been estimated that Chile, United States, Peru, Uzbekistan and Kazakstan have the largest geological resources of rhenium (See Table 4.1.7.1, the data was compiled using information from Abischeva and Zagorodnyaya 2013, Berzina et al., 2005, Berzina and Korobeinikov 2007, Cunningham et al., 2008, Figueiredo et al., 2013, Fleischer 1959, Grabezev 2012, Keseler and Wilkinson 2012, Plotinska et al., 2014, Safirova 2013, Roskill 2015, IndexMundi 2015, USGS 2015). Some rhenium is used in chemical catalysts (14% of the supply). A significant part of rhenium is used for chemicals and chemical catalysts. The most important is a rhenium-platinum catalyst for converting low octane fuels to high-octane petrol (Avon Metals 2014).

One would believe that because of the very high price, rhenium would have a high recycling rate, but that does not appear to be the case. Actually, real rhenium recycling is not more than 25% at present. Recycling of rhenium from some of the military uses pose significant chemical challenges and may at times incur high costs. Rhenium recovery from recycling is about 10 ton per year, and reported to be increasing (Anderson et al., 2013).

Table 4.1.7.1. Overview of reserves and resources in 2012 for important alloying metals for superalloys. Amounts shown are expressed in million metric ton of molybdenum.

Metal	Known extractable amount	Additional extractable amount	Extractable amount metal	Extracted before 2012	URR	Wild cards	
						Resources of unknown extractability	Seafloor nodules
Molybdenum	14	34	48	7	55	40	50
Rhenium	0.005	0.010	0.015	0.0014	0.016	0.005	0.010
Cobalt	8	15	23	6	29	30	40
Vanadium	14	20	34	5	39	250	20

Niobium	6.6	40	46.6	5	51.6	10	10
Tantalum	0.1	0.4	0.5	0.02	0.52	0.3	?
Zirconium	10	152	162	18	180	60	-
Hafnium	0.1	3	3.1	0.05	3.15	0.6	-

4.1.7.5 Data and materials

Production rates

The reserves estimates are based on classical geological estimates, and the allocation of extractable amounts according to ore quality, stratified after extraction costs (Singer 1993, 1995, 2007, Singer and Menzie 2010, Tilton 2007, 2009, 2012, Neumeyer 2000, Sweeneye 1992, Sverdrup et al., 2015a,b,c,d). Each category is mined when the market price allows for it. The data was gathered by going through country reports and other published literature. The numbers add up to about double of the estimates from the USGS (2015). Assemble data on reserves and resources, a number of literature sources were consulted (Alves et al., 2015, Bastos-Neto et al., 2009, Berger et al., 2009, Beurlen et al., 2014, Biruaba-rema et al., 2014, Chicharro et al., 2014, Eckstrand and Hulbert 2007, Fritz et al., 2013, Ginzburg and Fel'dman 1974, Gupta and Suri 1994, Hulsbosch et al., 2014, International Business Publications, 2013, 2015, Kravchenko and Pokrovsky 1995, Lele and Bhardwaj 2014, Lehmann et al., 2014, Keseler and Wilkingson 2013, Kwatra et al., 2012, Ludington and Plumlee 2009, Melcher et al., 2015, Möller 1989, Polinares 2012, Safirova 2013, Nassar et al., 2012, Mudd and Jowitt 2014, Mudd 2009, Nickless et al., 2014, Nuss et al., 2014, Polyyak 2011, Sverdrup et al. 2013, Tilton and Lagos 2007, UNEP 2011a,b,c, 2013a,b,c, USGS 2009, 2015, Vulcan 2013). There are a number of sources available for reserves, most have been summarized by the United States Geological Survey in their yearbooks (USGS 2015). Table 4.1.7.2 shows an overview of important alloying metals for superalloys. Table 4.1.7.2 also shows a compilation of some estimates of peak production date estimates and to which degree the extraction is primary hard-rock mining or secondary and dependent on mother metals.

Table 4.1.7.2. Overview of important alloying metals for superalloys. s.d.a.p. = Systems dynamics model assessment planned for later point in time.

Metal	Production in 2015 ton per year	Annual extraction as % of URR	Dynamic model peak year estimate	URR Million ton	Dependent Extraction	Hubbert's Model valid?
Molybdenum	270,000	0.5	2032	55	Partly	Yes
Rhenium	50	0.3	2035	0.016	Yes	No
Cobalt	120,000	0.4	2040	29	Yes	No
Vanadium	80,000	0.2	s.d.a.p.	39	Partly	No
Niobium	65,000	0.3	2055	52	No	Yes
Tantalum	1,200	0.4	2045	0.5	Partly	No
Zirconium	1,530,000	0.9	s.d.a.p.	180	Partly	Yes
Hafnium	10,000	1.0	s.d.a.p.	1	Yes	No
Nickel	2,500,000	0.9	2030	280	No	Yes
Tungsten	85,000	0.3	s.d.a.p.	30	No	Yes

Deducting from resource data the need for a specific simulation model structures

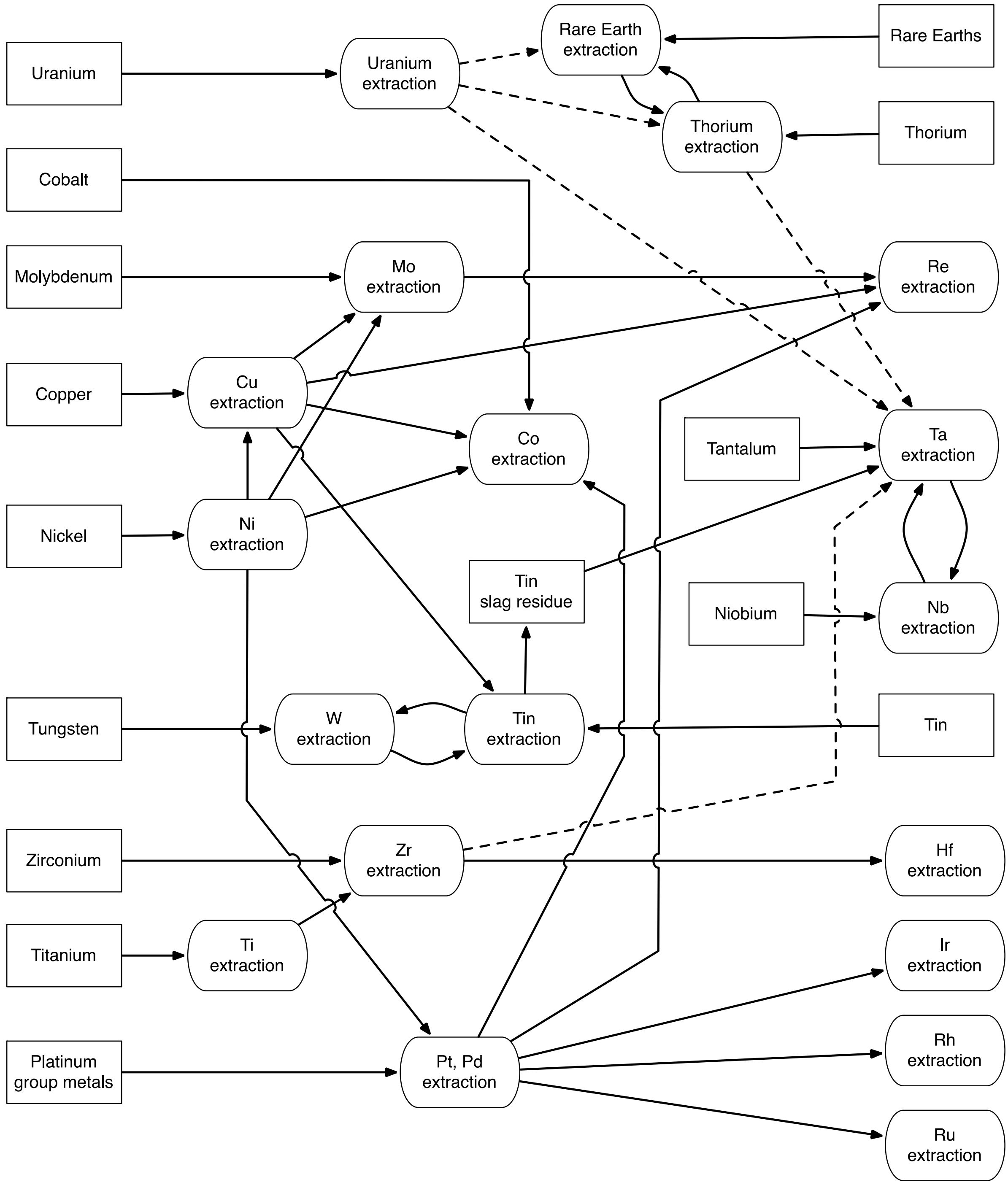
By looking at the numbers presented in Table 4.1.7.1 and 4.1.7.2 and based on Figure 4.1.7.4, we may make a number of conclusions on what sources contribute to the supply of different metals:

1. For molybdenum, 76% is primary extraction, 21% comes from copper extraction and 2% from nickel mining.
2. For rhenium, extraction from molybdenum supplies 78%, 22% comes from copper and about 3% from nickel and 1% from tungsten extraction. There is no primary extraction. Small amounts come from platinum group metal refining.

The following model structure can be foreseen for modelling the supply taken from the authors library of metal models (Sverdrup et al., 2014a,b, 2015a,b,c, 2016, a,b,c):

1. Molybdenum modelling is combined with secondary supply from copper and nickel extraction, using the BRONZE and STEEL sub-modules.
2. Rhenium is modelled as completely dependent on molybdenum, copper and nickel mining and extraction, using the BRONZE and STEEL sub-modules.

Figure 4.1.7.4. Overview of the extraction of the metals that get to be used in superalloys and the couplings between them. The flow chart was a step in developing the final model structure for these metals. Dotted lines represent potential but not yet used production possibilities



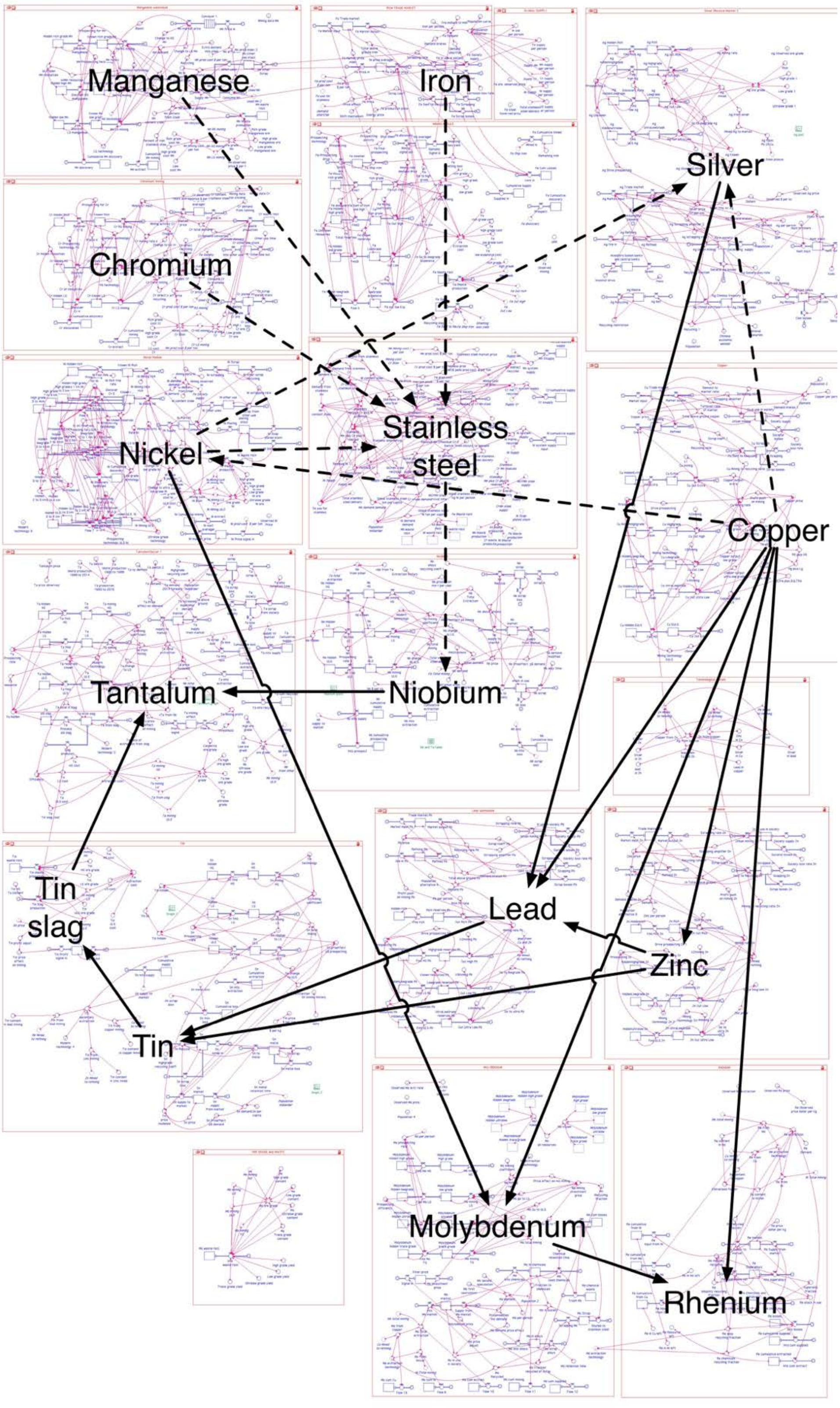
Source: Own Figure, Lund University/University of Iceland

The extraction of niobium and tin is important for tantalum, but not the other way around. Tungsten, niobium and tin have their own mines and contributions from the other metals are insignificant. There are no primary rhenium mines in operation, and only three places on Earth are known to have small, but potentially mineable rhenium deposits for rhenium's sake only. Chile is by far the largest producer with 26-30 ton of rhenium per year in production from their copper and molybdenum deposits. In California, the El Dorado gold mine has small amounts of platinum group metals and rhenium that sometimes follow platinum group metals. Ukraine has a potential deposit of significant size in the Carpathian Mountains that may one day be put into production. Russia is a significant producer with about 5 ton of rhenium per year from their copper-nickel deposits (Safirova 2013). Ore grades for rhenium resemble those of gold and platinum group metals, in the range from 26 gram per ton at the very best to 3-4 gram per ton, which was the norm in 2010-2015. Most rhenium is produced from the roasting of molybdenum concentrates derived from porphyric copper mining. Only some copper plants have molybdenum roasters, and only a few have the ability to extract rhenium. In the roasters, molybdenum sulphide (MoS_2) is roasted to remove the sulphur and to produce molybdenum trioxide (MoO_3) for the steel industry. In the roasting process, rhenium particles form Re_2O_7, which passes up with the flue gasses where the sulphur from the molybdenum roasting is scrubbed out to produce sulphuric acid (Vulcan 2013). Rhenium recycling has traditionally been low, but has in later years gained increased attention because of the high price and increased demand for military applications (Anderson et al., 2013, Papp 2014, Olbrich et al., 2009, UNEP 2013c).

4.1.7.6 The model description

The flow pathways, the causal chains and feedbacks loops in the global iron system were mapped using system analysis, and the resulting coupled differential equations were transferred to computer codes for numerical solutions, using the STELLA® environment. This is shown in Figure 4.1.7.5. The upper part of the metal model consists of the BRONZE sub-model in WORLD6, where molybdenum and rhenium has been integrated. The model combines the extraction of many primary and secondary metals. From this, the necessary demand drivers and mining drivers in the copper sub-model can be estimated. Nickel resides in the STEEL model and takes with it stainless steel, iron, chromium and manganese in order to operate. From this the demands drives for nickel originate. The model was used for quantification of flows, scenarios and future predictions. The model was developed to estimate supply, extractable amounts stocks, iron price and stocks, and flow in society in the time interval 1900-2400.

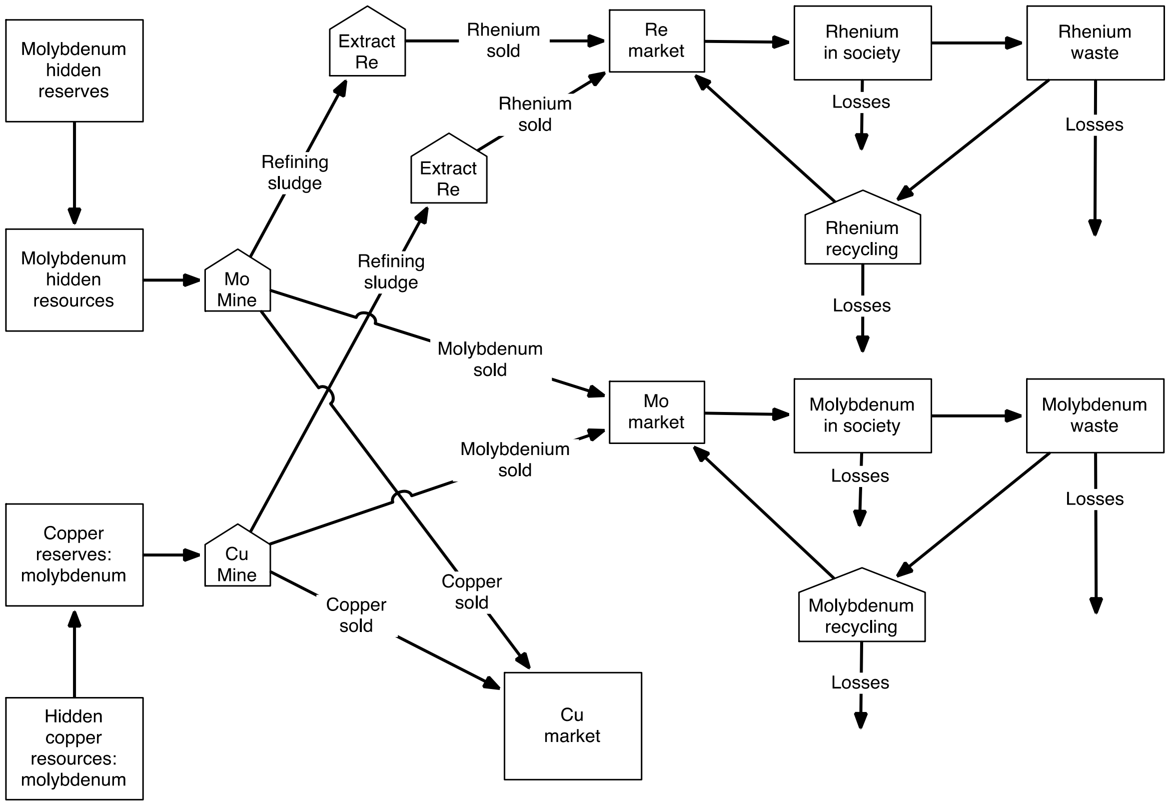
Figure 4.1.7.5. The STELLA diagram for the model structures used. The upper part consists of the WORLD6 model, where molybdenum and rhenium has been integrated. Nickel is simulated as a part of the WORLD6 model and takes with it stainless steel, iron, chromium and manganese in order to operate.



Source: Own Figure, Lund University/University of Iceland

Figure 4.1.7.6 shows a basic flow chart applied in the model. Molybdenum flows from molybdenum ore bodies and from molybdenum produced as a by-product of copper mining. Not all copper deposits contain molybdenum, and not all copper refineries are equipped to extract molybdenum and whatever rhenium that comes with it. Molybdenum and rhenium flow into the market, gets stored in products in society and are eventually scrapped or lost. The losses are of two types: One type is dissipative losses, where the metal is irretrievable lost in general waste, landfills or by wear and tear and lost irreversibly. A significant part or molybdenum and rhenium is also lost into steel scrap that may be recycled as iron, but the rhenium and molybdenum is no longer recognized and separated from the iron, thus it is from a functionality point of view, lost. Rhenium and molybdenum are expensive metals and a significant part of the scrapped metal is recycled and refined. During the last decade recycling technologies for rhenium and molybdenum have been significantly improved and it appears that this trend is continuing.

Figure 4.1.7.6. Flow chart for molybdenum and rhenium from mine to final dissipation.



Source: Own Figure, Lund University/University of Iceland

The part of the WORLD6 structure used for this study has the following stocks used to define the coupled differential equations defined by mass balance:

1. Molybdenum
   a. Mineable 4 stocks, and 4 with hidden resources;
      i. Known high, low, ultralow and trace grade ore
      ii. Hidden high, low, ultralow and trace grade ore
   b. In society, we distinguish 4 stocks;
      i. Trade market
      ii. Stock-in-use as alloy
      iii. Stocks-in-use as chemicals and catalysts

         iv. Scrapped molybdenum
2. Rhenium
   a. In society, we distinguish 4 stocks;
      i. Trade market
      ii. Stock-in-use as alloy
      iii. Stocks-in-use as chemicals and catalysts
      iv. Scrapped rhenium
   b. Rhenium production
      i. Molybdenum mining and refining, contributing rhenium
      ii. The BRONZE model; Copper mining contributing molybdenum and rhenium
      iii. The STEEL Model; Nickel mining contributing rhenium
      iv. The PGM submodel; PGM mining contributing rhenium

The rate of change in all stocks is defined by inputs and outputs, conforming to the equation

$$\frac{dm}{dt} = \ inputs + produced - outputs \qquad (4.1.7.1)$$

The inputs are from recycling, and produced from primary extraction. Accumulated is what is accumulated in society either as stocks in use or as useless scrap. Outputs will be what is lost or recycled. Note that recycled is present on both sides, and thus increase the flow through society. To meet a certain supply to society, the larger the recycling, the smaller the primary extraction must be to meet the demand. The non-linear, coupled differential equations that are simultaneously solved numerically with a 4-step Runge-Kutta algorithm. Figure 4.1.7.5 shows the STELLA® diagram for the model. The ore is mined from the grade with the lowest extraction cost, but with some overlap between the ore grade classes. Dependent stocks are cumulative amounts of: mined, amounts rock waste, losses, smelter slag and ore found by prospecting. The mining activity is profit-driven, the profit is affected by the mining cost and the market price. A lower ore grade implies that more rock must be moved to mine the metals. In the model, mining and recycling is profit driven. Sales of extracted metal together with the price, given the income, the extraction and prospecting give the costs, the profit being the difference. In the model, shift to a lower ore grade only occurs when the market price exceeds the extraction cost in such a way that the profit stays positive. When the profit are lower or goes negative, the mining rate is slowed down or stopped, leading to less material in the market, driving the price up.

There are several sources of metal in during extraction; the high, low and ultralow ore grades and the stocks of metal in society that can be recycled. For molybdenum, we have molybdenum mines with four different ore grades, and secondary extraction from nickel, copper and platinum group metal ores, as well as recycling of a fast stock of chemicals and catalysts and a slow stock of metallic alloys (Figure 4.1.7.4). The mining rate follows a rate equation depending on the mineable reserve, profitability of the operation and the available mining technology:

$$r_{mining} = k * \ m_K^n * \ f(profit) * g(technology) \qquad (4.1.7.2)$$

where rmining is the rate of mining, k is the rate coefficient and mK is the mass of the ore body known and available for extraction, and n is the mining process order. The rate coefficient is modified with ore extraction cost and ore grade. In the model, a delay in mining rate change is considered by using a forward rolling two-year average of the market price. g(technology) is a technology factor accounting for the invention of technologies used in efficient mining, refining and extraction. f(profit) is a feed-back function of market price, increasing mining at higher price and lowering it at lower metal prices.

It was defined by the following equation:

$$Profit = Income\ from\ sales - mining\ costs \qquad (4.1.7.3)$$

Where the costs are defined as:

$$Total\ costs\ =\ Mining\ costs\ +\ refining\ costs\ +\ prospecting\ costs \qquad (4.1.7.4)$$

$$Rre = rMo * XRe(Mo) +\ rCu * XRe(Cu) +\ rW * XRe(W) +\ rNi * XRe(Ni) \qquad (4.1.7.5)$$

In this equation, mining costs, refining costs and prospecting costs all include both variable operations costs and capital costs for infrastructures and equipment, as well as a 10% profit margin. The income is defined as:

$$Income\ from\ sales\ =\ Amount\ sold * market\ price - sales\ costs \qquad (4.1.7.6)$$

The profit is driven by metal price and amount extracted, but balanced by the cost of operation. The mining rate is related to the state of the technology, and it has two alternatives: (1) the technology and the financial capacity determines the mining rate and the ore body is always sufficient to supply anything attempted to be extracted, (2) the mining rate is unrelated to the technical and financial capacity and limited by the access to the ore body. There are many different definitions of recycling available (Graedel and Allenby 2003, UNEP 2011b, 2013a). The size of the extractable ore body is determined by the rate of extractions (rmining) and the rate of prospecting (rdiscovery):

$$\frac{dm_K}{dt} = r_{mining} + r_{discovery} \qquad (4.1.7.7)$$

We use the following equation for the rate of discovery:

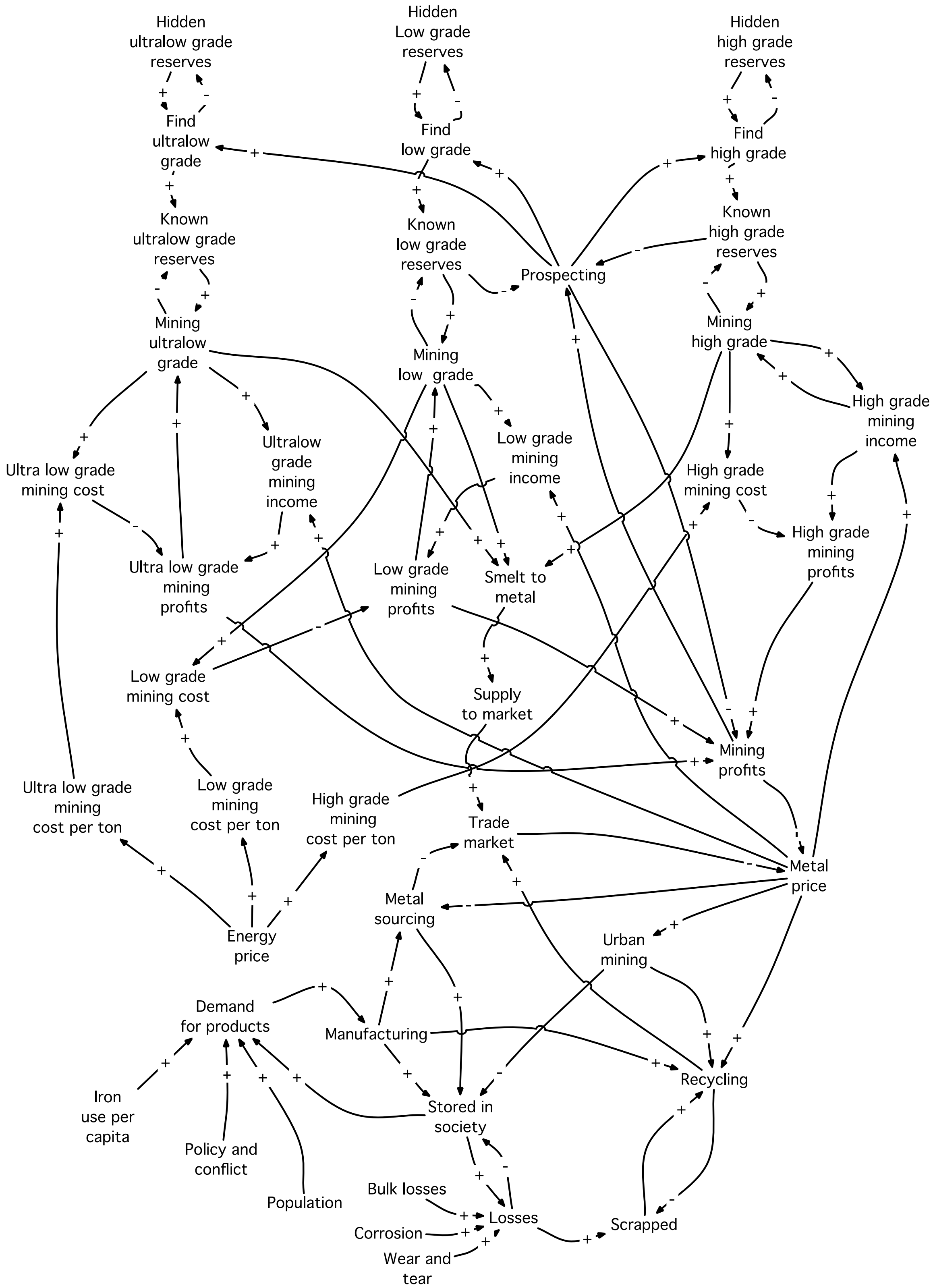
$$r_{discovery} = \frac{dm_H}{dt} = -k_{prospecting}\ *\ m_H\ \ * g(technology) \qquad (4.1.7.8)$$

The discovery is a function of how much prospecting we do and how much there is left to find. There are many different definitions of recycling available (Graedel and Allenby 2003, UNEP 2011b). The amount hidden reserve (h) decrease with the rate of discovery. The rate of discovery is dependent on the amount metal hidden (h) and the prospecting coefficient kprospecting.

The prospecting coefficient depend on the amount of effort spent and the technical method used for prospecting. The rate is first order as prospecting is three-dimensional by drilling. mH is the amount hidden in the resource. The basic driving mechanism of mining comes from profits and availability of a mineable resource used in the model.

The price is set relative to how much iron or steel there is available in the market. f(technology) is a function accounting for improved prospecting techniques coming with modern society and technological development. Figure 4.1.7.7 shows the molybdenum sub-module model as a causal loop diagram (CLD). The causal loop diagram maps all significant causal connections in the system. It is important that the links are true causal links and not just correlations or modelled on chain of events. The causal loop diagram together with the flow chart, defines the model.

Figure 4.1.7.7. The mining model as a causal loop diagram for the whole world system. The causal loop diagram describes the causal relationships and the feedback loops in the global supply system.



Source: Own Figure, Lund University/University of Iceland

The contribution from tungsten mining to molybdenum and rhenium extraction was so small that it was ignored in the final model. The extraction rates of molybdenum (RMo) and rhenium (Rre) in the model were calculated as follows:

$$RMo = rNi * XMo(Ni) + rCu * XMo(Cu) + rMo \qquad (4.1.7.9)$$

Where Xj(i) is the fraction of metal j in mother metal i, and rj is the mining rate of metal j.

The causal loop diagram shows us the mining operation is driven by operations profit as can be seen in Figure 4.1.7.7. For the purpose of clarity, the recycling fraction displayed in results were calculated as follows in this study:

$$Recycling\ fraction = \frac{Flow\ of\ recycled\ metal}{Supply\ from\ primary\ extraction +\ Flow\ of\ recycled\ metal} \qquad (4.1.7.10)$$

Which is equivalent to:

$$Recycling\ fraction = \frac{Supply\ to\ market - Primary\ extractionl}{Supply\ to\ market} \qquad (4.1.7.11)$$

The cost of the mining and extraction operation is mainly determined by two important factors beside cost of investments, the energy price and the ore grade. The price is determined by two factors, that it must stay above the production costs and by the amount in the market.

Defining Factor X

Factor X is used to estimate the resource use efficiency of a material. Factor X vary over time, and thus an alternative measure would be Factor x:

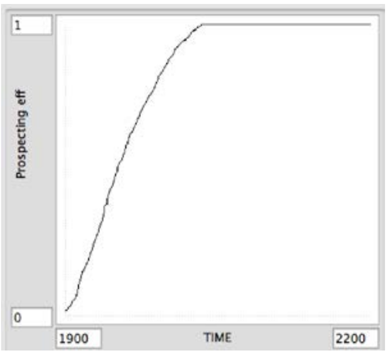
$$Factor\ x = \frac{Market\ supply\ rate}{Extraction\ rate} \qquad (4.1.7.12)$$

We have considered the time from 1930 to 2400. Production before 1930 was insignificant.

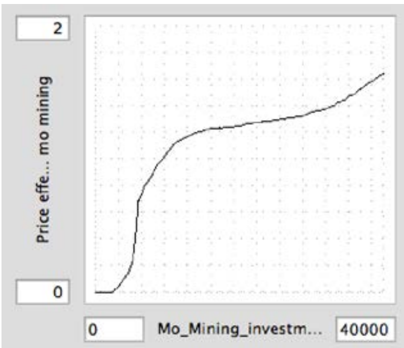
Feedback functions in the model

A number of feedback response curves are employed in the multi-metal model. These are shown in Figure 4.1.7.8. These curves quantify our assumptions on future global demand per person and change in extraction efficiency with time. The causal loop diagram in Figure 4.1.7.7 shows how the extraction operation is driven by profit. This profit is driven by the aluminium price and aluminium amount extracted, but balanced by the cost of operation. The cost of operation is mainly determined by two important factors besides cost of investments, the energy price and the ore grade. The price in the model is determined by the amount in the market and the fact that it must stay above the production costs. It depends on the balance between deliveries into the market from production and the shipments from the market in response to world market demand. The price also drives the urge for recycling of metal stock from society. Higher price acts as a brake on demand. Metal trading is supposed to operate as follows: The traders come to the trading floor with their lots to sell or to buy, and adjusts their sales or purchase amounts as the price goes up and down. If demand is higher than production, the price goes up; in the opposite case the price is moved down. This is a self-adjusting mechanism that balances the trade by adjusting the prices until the demand to buy an amount at a price match the offers to sell an amount at a price. The buyer offer to purchase more at a lower price or less at a higher price, and the sellers offer to sell less at a lower price or more at a higher price. When the price and amount match, the price is set. This is based on personal observation on the trading floors at the metal markets in New York and London by the authors. We model the price as a rational process, assuming the price to be set through an arbitration between market partners. We have not included random effects and irrationality. This profit is driven by metal price and amount extracted, but balanced by the cost of operation. The cost of operation is mainly determined by two important factors besides cost of investments, the energy price and the ore grade. In addition, we have the condition that it must stay above the production costs for the operations to be profitable. The amount in the market in turn depend on the balance between deliveries into the market from production and the shipments from the market in response to world market demand. The price drives the urge for recycling of metal stock from society. The R in the causal loop diagram indicate the reinforcing loops, driving the mining activity and the supply to the market. We have collected a lot of data over the years from the metal trading arenas, and these have been used to create the relationships between amount tradable metal in the market arena and the market price. Some of this is easily accessible from the London, Now York and Shanghai metal trade exchanges, in addition we have used private trading archives collected over the years. From such data, price curves were constructed.

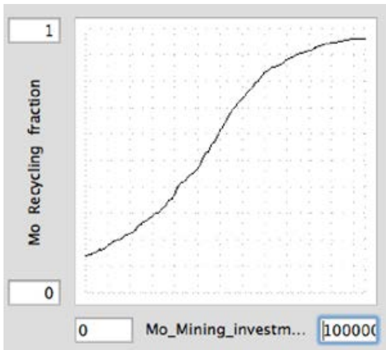
Figure 4.1.7.8. Molybdenum. Feedback response curves employed in the model for molybdenum



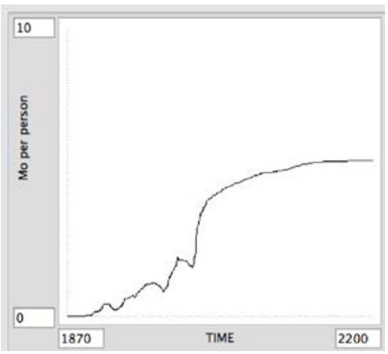
A: Molybdenum prospecting technology efficiency



B: Molybdenum price effect on mining rate

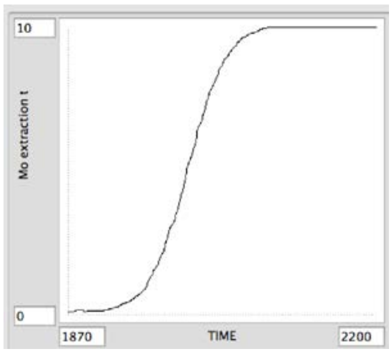


C: Molybdenum recycled fraction based on investment

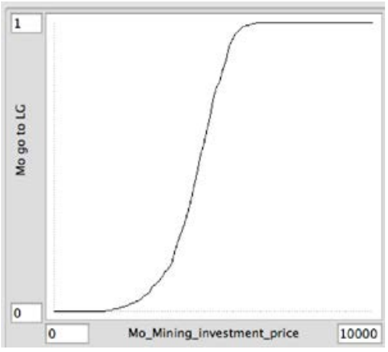


A: Global average demand, kg molybdenum per person

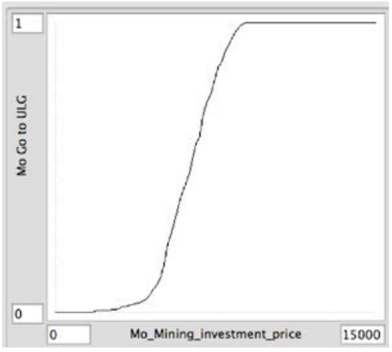
B: Molybdenum price effect on global average demand



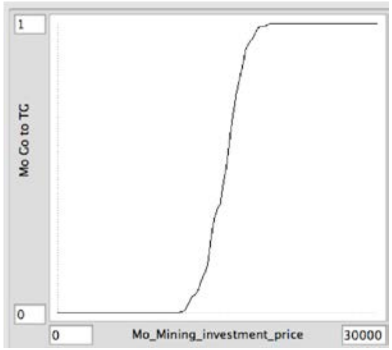
C: Molybdenum extraction technology efficiency



D: Molybdenum threshold price for low grade ore mining



E: Molybdenum threshold price for ultralow grade ore mining



F: Molybdenum threshold price for trace grade ore mining

Source: Own Figure, Lund University/University of Iceland

In the market, several types of transactions occur:

1. The metal is sold in the market to a buyer, shipped and physically supplied at once. Supply to market and take from market is the same.
2. Forward sale; The metal is sold at once and payment received, but the metal is physically delivered at a later date. Ownership shifts at once, but the money later. Many mines do this to improve liquid funds.
3. The metal is shipped at once but payment is received at a later date. Ownership shifts at once, but the money moves later.

The causality in profits go to income from supply when the amount is supplied and paid at once into the metal exchange warehouse. The same applies with a forward sale, when the metal is paid upfront but physically delivered later. If not, all or part may be paid when the supplied amount has been cleared out from the physical warehouse. When we have no data or very few, maybe only one data point for creating a price curve for a commodity, then we will fit a curve to Equation 14:

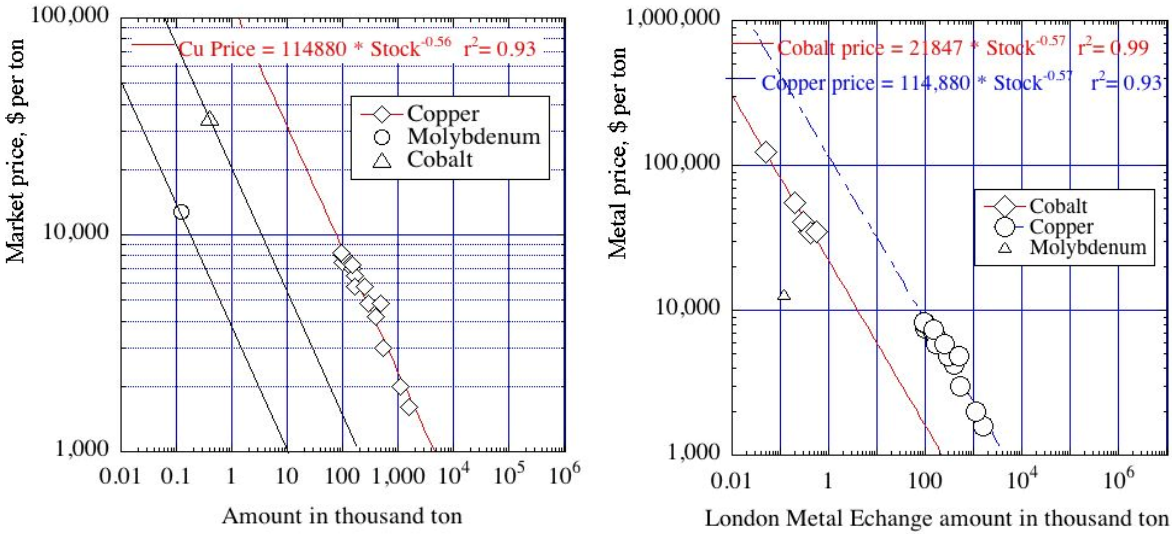
$$Price = k * (Market\ stock)^n \qquad (4.1.7.13)$$

We will demonstrate the procedure as follows:

1. For molybdenum, the following data is available; The London Metal Exchange (LME) stock was 120 ton during March to December 2015, the price was 12,800 $ per ton.
2. For cobalt, there was 400 ton in the LME trade stock in January 2012, the price was then 34,700 $ per ton.

Figure 4.1.7.9 shows the reconstructed price curve for molybdenum and cobalt, using the copper curve for help. For molybdenum, we do not have good overlapping data for market stocks and price. For cobalt, we have some overlap and can thus check the performance of the method. The comparison shows that the assumption of the same curve shape, but a different coefficient is valid. The molybdenum price is for molybdenum oxide. Ferro-Molybdenum has about the same cost. Pure molybdenum metal costs a bit more. When no information is available, this equation will be used.

Figure 4.1.7.9. The reconstructed price curve for molybdenum and cobalt, using the copper curve for help. For molybdenum, we do not have good overlapping data for market stocks and price. a; Approximation of molybdenum and cobalt price curve, (b) the cobalt curve from data.



a; Finding a curve for Co and Mo b; Verifying for Cobalt

Source: Own Figure, Lund University/University of Iceland

Input data and parameters

Table 4.1.7.3 shows the settings nickel. The extractable amounts were set at the beginning of the simulation in 1900, stratified with respect to ore metal content. The average ore grade for the high grade molybdenum was set at 5%, for low grade at 1%, for ultralow grade at 0.2% and for trace grade at 0.02%. For rhenium, no initial resource quantification is used, the metal is totally dependent on the concentration in the different mother metals (Tilton and Lagos 2007, Robinson and Menzie 2012, Sverdrup et al., 2014a,b, c, 2015a,b). For copper, the extractive potential is 3,665 million ton, however, by 2400 AD, only 2,700 million ton of the total copper resource will have been extracted (Sverdrup et al., 2014b).

Table 4.1.7.3. Estimated ultimately recoverable amount nickel, starting from 1900. This was used as the starting point for the STEEL model.

Mining depth	Recoverable metal amount metal in million ton per ore class					
	Rich	High	Low	Ultralow	Extra low	Sums
	55%	11%	2%	0.5%	0.1%	all
Hidden 0-1 km	4	60	45	30	30	169
Hidden 1-2 km	3	30	25	20	15	93
Hidden 2-3 km	2	20	20	10	10	62
Hidden 3-4 km	1	10	10	8	5	24
Sum 0-4 km	10	120	100	68	60	348
Mining cost, $/ton	40 $/ton	120 $/ton	400 $/ton	1,750 $/ton	3,500 $/ton	

The main producing countries of molybdenum have been listed with their annual production and reserves and resources in Table 4.1.7.4. To put this together, a number of literature sources were consulted (Abischeva and Zagorodnyaya 2013. Berzina et al., 2005, Crowson 2011a,b Cunningham et al., 2008, Eckstrand and Hulbert 2007, Figueiredo et al., 2013, Grabezev 2012, Hess 1924, Fleischer 1959, International Business Publications 2013, 2015, International Molybdenum Association 2013, 2015, Keseler and Wilkingson 2013, Kwatra et al., 2012, Ludington and Plumlee 2009, Magyar 2004a,b, Molybdenum 2007, Nassar et al., 2012, Mudd and Jowitt 2014, Mudd 2009, Nickless et al., 2014, Nuss et al., 2014, Polyyak 2011, Robinson and Menzie 2012, Roskill 2010, 2015, Sverdrup et al. 2013, Tilton and Lagos 2007, UNEP 2011a,b,c, 2013a,b,c, USGS 2009, 2012, 2013, 2014, 2015, Vulcan 2013). There are a number of sources available for molybdenum reserves and resources, most have been summarized by the United States Geological Survey in their yearbooks (USGS 2015). Reserves are the known and proven reserves, omitting that there probably is far more material in the ground further in or down, as well as in areas not investigated. These resources, which we call "hidden" in this study, represent what is called resources. Resources are defined as that amount that can in due time be found and extracted, even if some or much of it would require a higher price and more effort. Some of it may require technological innovations and a significantly higher price to cover the extraction costs. For molybdenum, this is a number far larger than 9 million ton, it is at least 33 million ton, and possibly as much as 55 million ton (See Cunningham et al., 2008 and Table 4.1.7.2). The numbers have been assembled from data from the USGS (2015) and other literature sources listed. The molybdenum ores have grades from 4% to 0.01% weight molybdenum content. Rhenium is about 0.02-0.025% in molybdenum metal. The procedure is to first separate molybdenum from the anode sludge and then afterwards, separate out rhenium from molybdenum (International Molybdenum Association 2013, 2015, Olbrich et al., 2009).

Table 4.1.7.4 Estimation of extractable amounts of molybdenum, based on literature sources and new estimates of the extractable resources of the mother metals nickel, copper and tungsten.

State	Production 2013	Mined to present	Reserves (Known)	Total Resources	URR	Comment and source of information. Some from unpublished industrial documents.
	Ton/year	Million ton				
China	94,000	-	8.5	20.0		New data from China
United States	64,000	-	2.7	5.7		New data from industry
Chile	38,000	-	1.2	2.5		New data from industry
Peru	18,000	-	0.6	2.0		Guessed total resource
Mexico	12,000	-	0.2	0.4		Guessed total resource
Canada	4,500	-	0.4 1.5	0.9 3.0		USGS; underestimated New data from industry Guessed total resource
Russia	2,500	-	0.2 3.0	0.5 6.0		USGS; underestimated New data from industry Guessed total resource
Others	20,000	-	1.6 5.0	5.0 12.0		USGS; underestimated Guessed total resource
Sum	280,000	6.5	11.0 20.1	36.5 49.1	42.0 55.6	Minimum Best estimate

Table 4.1.7.5 shows an overview of degrees of molybdenum recycling. The numbers are approximate annual estimates for 2010-2012. Much of the scrap metal is recycled within the metal manufacturing plants and never reach a refinery or the metal trading markets. This holds true for super-alloys, high-speed tool steels, stainless steels and some of the cast iron, and is true for both molybdenum and rhenium.

Table 4.1.7.5. Overview of recycling degrees of molybdenum. The numbers are approximate annual estimates for 2010-2012. All amounts in metric ton.

Product	Recycling fraction of supply %	Recycling, ton per year	Metal use % of total supply	Metal use Ton per year	Primary metal production, ton per year
Super-alloy	58	10,440	5	18,000	-
High-speed steels	56	16,100	8	28,800	-
Tool steel alloy	44	11,100	7	25,200	-
Stainless steel	38	30,100	22	79,200	-
Cast iron	18	5,180	8	28,800	-
Structural steel	13	17,800	38	136,800	-
Chemicals	5	2,160	12	43,200	-
Summary	26	92,880		360,000	280,000

This implies that the degree of recycling appears as invisible to the outsider. Much of the structural steel and cast iron is down-cycled into general steel where it gets diluted and lost. The recycled rhenium and molybdenum chemicals go back to a refinery and are recycled over the metal markets. Table 4.1.7.6 shows the estimation of molybdenum and rhenium primary extraction, metal production and recycling by country. We have made some assumptions, as data on resources amounts are to a large degree only guesswork and data is missing. The data is very uncertain, and for many regions, rhenium resource data is missing or not available to us. The production in 2015 is not exactly known according to the USGS (2015) commentary, it is somewhere between 48 and 58 ton per year. A negative number suggests that rhenium is derived from secondary material from this country somewhere else. In the table, we have back-calculated from molybdenum contents as well as estimated military needs in United States, Russia, China and India, using military resource needs assessment literature. Production, supply and consumption and extraction estimates for India, China and Russia are no longer available, and represents our estimates based on corporate annual reports, secondary unofficial sources, older literature and back-calculations using trading and industrial statistics. Their military needs for rhenium can be roughly estimated through indirect means from open sources. For rhenium, we estimate the degree of recycling to be about 30% of the total market supply, or 42% of the primary extraction estimate. For some countries, the rhenium production is a strategic secret and the published numbers may be distorted. The molybdenum production was included as an indicator of rhenium extraction potential. Most molybdenum deposits contain rhenium, some copper-nickel deposits contain rhenium and it occasionally occurs in uranium deposits. The metal resources were distributed to high grade, low grade and ultralow grade ores.

Table 4.1.7.6. Estimation of molybdenum and rhenium primary extraction, metal production and recycling by country. A negative number suggests that rhenium is derived from secondary material from this country somewhere else. Amounts in metric ton.

Country	Molybdenum production in 2013-2015, ton	Rhenium extraction by country of origin in 2013-2015, ton	Rhenium supply by country in 2013-2015, ton	Rhenium recycling by country of origin in 2013-2015, ton
China	110,000	5.0	12.5	7.5
USA	61,000	4.3	10.0	5.7
Chile	37,000	20.0	26.0	6.0
Peru	17,000	5.0	-	-5.0
Mexico	11,000	4.1	-	-4.1
Canada	9,000	1.7	-	-1.7
Armenia	6,500	0.4	-	-0.4
Iran	6,300	-	-	-
Turkey	5,000	-	-	-
Russia	4,800	5.0	7.0	2.0
Mongolia	2,000	-	-	-
Kazakstan	500	-	3.0	3.0
Uzbekistan	-	5.0	2.5	-2.5
Poland	5,000	2.6	7.6	5.0
Germany	-	-	6.0	6.0
Estonia	-	-	3.0	3.0
India	-	-	1.0	1.0
Other	5,000	2.5	-	-2.5
World	280,100	55.6	78.6	23.0

Table 4.1.7.7 shows an overview of the input data used for rhenium in the simulations. All amounts shown are in metric ton.

Table 4.1.7.7. Overview of the input data used for rhenium in the simulations. Amounts in metric ton

Metal	Mother metal URR Million ton	Rhenium, % in mother metal	Rhenium content, ton
Copper	3,665	0.00006	2,200
Nickel	300	0.000064	180
Molybdenum	54	0.02-0.025	16,500
Rhenium	-	-	18,880

Table 4.1.7.8 shows an overview of production rates and metal prices in 2014 for the STEEL model.

Table 4.1.7.8. Overview of production rates and metal prices in 2014 for the STEEL model. Amounts in million ton

Metal	Ton per year		Typical weight content in stainless steel, %	Metal price $/ton
	Mining rate in 2012	Estimated maximum extraction rate		
Iron	1,320,000,000	3,200,000,000	50-80	450
Manganese	18,000,000	24,000,000	12	2,200
Chromium	8,000,000	15,000,000	18	2,500
Nickel	1,700,000	3,000,000	8	12,000

Table 4.1.7.9 shows the estimated ultimately recoverable amount metal, starting from 1900 for the STEEL model. Values in parenthesis are not used in this assessment. The highest ore grades are consumed first in the model, the switch to a lower ore grade takes place when the price has passed a predefined price level. Reserves are the known and proven reserves. These we call "Known" in this study. The yet undiscovered extractable amounts (Resources), which we call "hidden" here, represent what is called resources. Resources are defined as that amount that can in due time be found and extracted, even if some or much of it would require a higher price and more effort. Through prospecting "hidden" is converted to "known" when it is found. But technically it will sooner or later be available, provided we will take to effort to extract it.

Table 4.1.7.9. Estimated ultimately recoverable amount metal, starting from 1900 for the STEEL model. Values in parenthesis are not used in this assessment. Amounts in million ton

Metal	Type of resource	Ore grade classes, recoverable metal amount metal in million ton and average metals content in %				
		Rich 55%	High 11%	Low 2%	Ultralow 0.5%	URR
Iron	Known	30,000	12,000	8,000	(120,000)	50,000
	Hidden	52,000	80,000	158,000	(500,000)	290,000
	Sum	82,000	92,000	164,000	(620,000)	340,000
Manganese	Known	240	200	50	(1,400)	490
	Hidden	1,010	1,400	1,800	(3,000)	4,210
	Sum	1,250	1,600	1,850	(3,400)	4,700
Chromium	Known	5	0	0	(400)	5
	Hidden	1,200	800	500	(1,200)	2,500
	Sum	1,205	800	500	(1,600)	2,505
Nickel	Known	1	0	0	0	1
	Hidden	5	120	100	68	293
	Sum	6	120	100	68	294

The yield in Table 4.1.7.10 is the fraction of the ore content that actually can be extracted from the whole content. Yield data from Mudd (2009) and from data compiled by Sverdrup et al. (2016) was used for estimating the nickel reserves down to 3 km mining depth.

Table 4.1.7.10. Relationship between ore grade, the approximate production cost and minimum supply price to society. The numbers are approximate for 2012-2015 (Sverdrup et al., 2015a).

Ore grade	Ore metal content		Yield	Cost	Price	Energy need
	kg/ton	%	%	$/kg	$/kg	MJ/kg
Rich	400-50	40-5	100	4	5-20	6
High	50-10	5-1	99	15	20-80	15
Low	10-2	1-0.2	98	63	80-330	100
Ultralow	2-0.4	0.2-0.04	91	280	330-1,200	1,000
Trace	0.4-0.1	0.04-0.01	80	1,000	1,200-1,900	10,000
Rare	0.1-0.02	0.01-0.002	70	1,600	1,900-10,000	100,000
Faint	0.02-0.002	>0.002	50	8,000	10,000-50,000	1,000,000

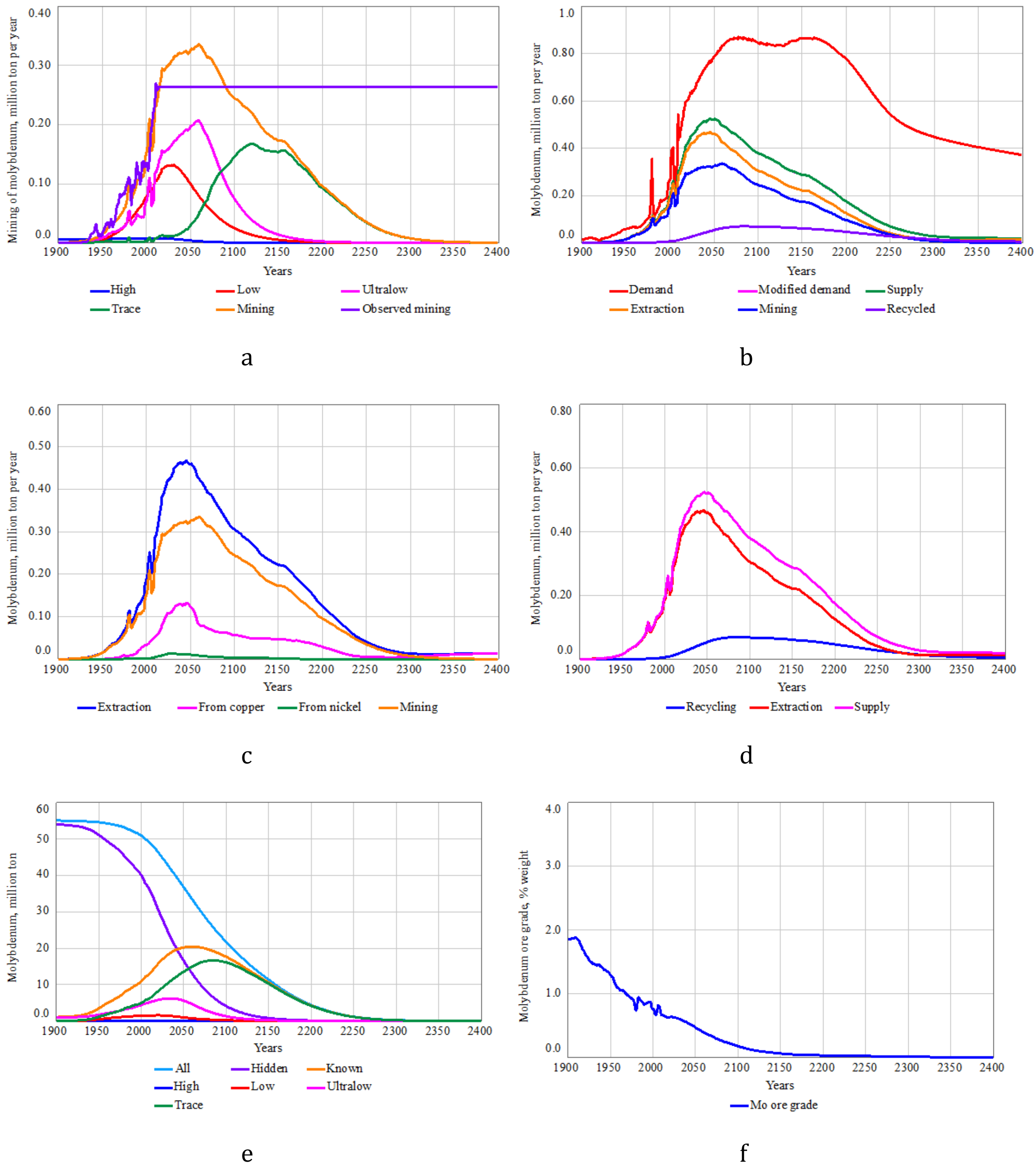
4.1.7.7 Results

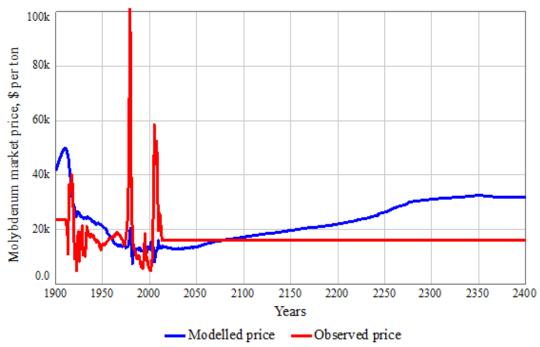
Model simulation results for molybdenum

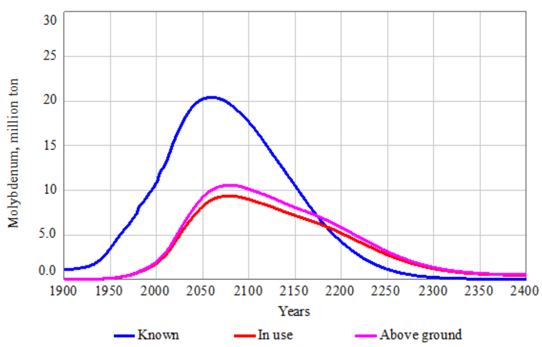
The model outputs for the time period 1900 to 2400 were used to create the simulation output plots. The time interval was chosen as it shows the rise, peak and fall of the metal supply in that period. Figure 4.1.7.10 shows mining rates in terms of source and from what kind of ore grade extraction takes place in molybdenum mining is shown in the upper row to the left. Figure 4.1.7.10b shows a comparison of the demand, price-modified demand, total supply to market, total extraction rate and amount recycled. Figure 4.1.7.10c shows extracted amounts from molybdenum mines, and by-product separation from copper and nickel mining. The diagram to the Figure 4.1.7.10d shows a comparison of extracted, supplied and recycled. Figure 4.1.7.10e shows the development of the known and hidden resources in different qualities. Figure 4.1.7.10f shows the simulated ore grades. Figure 4.1.7.10g shows the simulation of price as compared to data. Figure 4.1.7.10h shows a comparison of known reserves and the amounts in use in society.

The results show that molybdenum will always be in soft scarcity, but that after price-adjustment to demand, the no hard scarcity and failure of provision will occur. The molybdenum sub-model in WORLD6 model suggests that by 2150, virtually all hidden resources will have been detected and included in minable reserves. For molybdenum, we get a known reserve of about 14 million ton in 2015, which peaks at 20 million ton in 2060. The model suggests that by 2250, all known molybdenum reserves will have been exhausted. Unfortunately, there are no consistent data for the observed ore grade available for testing the performance of the model. The trend is declining from the beginning of molybdenum mining. The general price level and long term trends are well reproduced, but the short-term variations are not really captured. The price estimate is based on the dynamic price model shown earlier. The short-term demand variations and the pressure from speculation is unknown and thus very difficult to include in the model. The discrepancy in the 1980 to 1990 was probably due to the finals stages of the cold war arms race. The large discrepancy during 1996-2010 is most certainly due to market speculations. The price curves can be used for strategic planning, but are useless for short term trading and speculative trade. Figure 4.1.7.10b shows the remaining stock of molybdenum and the known reserves as compared to the stock-in-use in society. After 2060, the stock-in-use will be larger than the remaining resource. This will be the age of scrap molybdenum.

Figure 4.1.7.10. Molybdenum. Mining rates in terms of source and from what kind of ore grade extraction takes place in molybdenum mining is shown in diagram (a). Diagram (b) shows a comparison of the demand, price-modified demand, total supply to market, total extraction rate and amount recycled. Diagram (c) shows extracted amounts from molybdenum mines, and by-product separation from copper and nickel mining. Diagram (d) shows a comparison of extracted, supplied and recycled. Diagram (e) shows the development of the known and hidden resources in different qualities. Diagram (f) shows the simulated ore grades. Diagram (g) shows the simulation of price as compared to data. Diagram (h) shows a comparison of known reserves and the amounts in use in society.







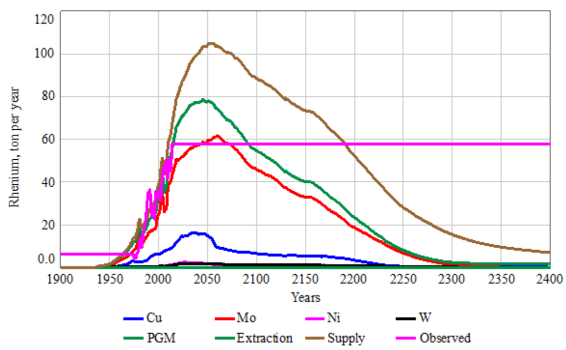
g h

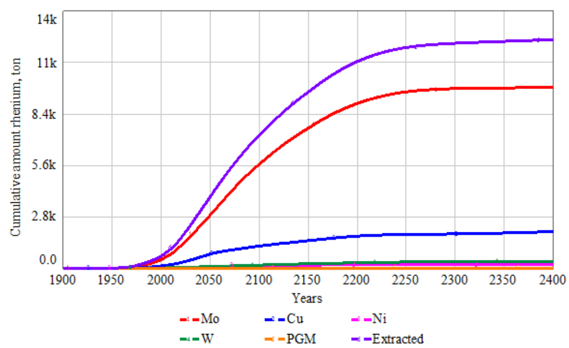
Source: Own Figure, Lund University/University of Iceland

Model simulation results for rhenium

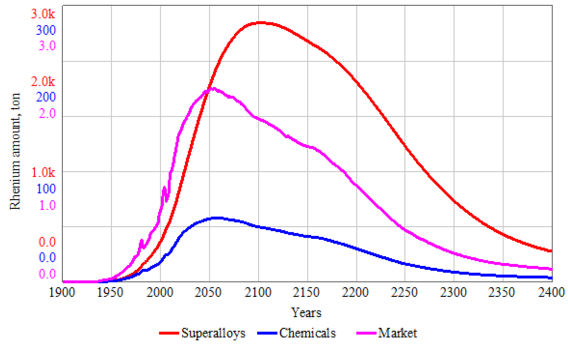
Figure 4.1.7.11a shows the observed extraction and where it comes from in terms of mining or extraction from a mother metal. The diagram on the upper left shows a comparison of total extraction, demand, supply to the market and recycled for rhenium. Figure 4.1.7.11b shows the development of the cumulative amounts of rhenium extracted from different metals. Figure 4.1.7.11c shows the amounts in superalloys, chemcials and in the trading market for rhenium. Figure 4.1.7.11d shows the cumulative amounts supplied, extracted and lost. Figure 4.1.7.11e shows the stock in use, the total amount of resources left, and what is left in molybdenum deposits, copper deposits and nickel deposits. Figure 4.1.7.11f shows the estimated fraction of supply coming from recycled material. In the molybdenum-rhenium sub-model, we lead all recycling rhenium over the market, but in reality, some of this is recycled internally in the industry. Figure 4.1.7.11g shows the simulation of the price and compares this to the observed price for rhenium. There is not much connection except for the approximate level. A possible reason for this is that there is no real functioning market for rhenium because of the limited amount and the very limited number of actors.

Figure 4.1.7.11. Model simulation results for rhenium

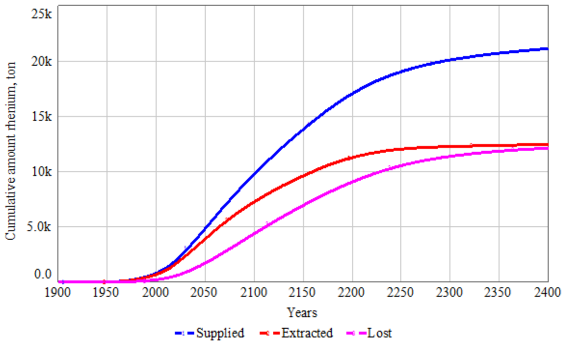




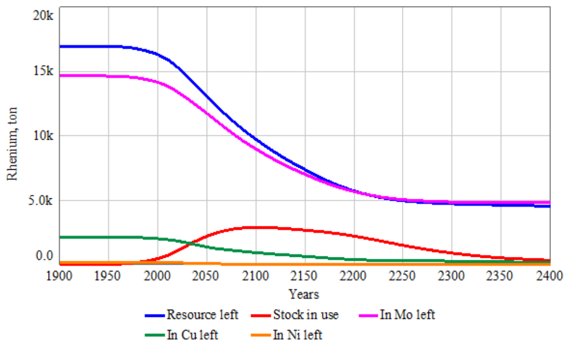
a b



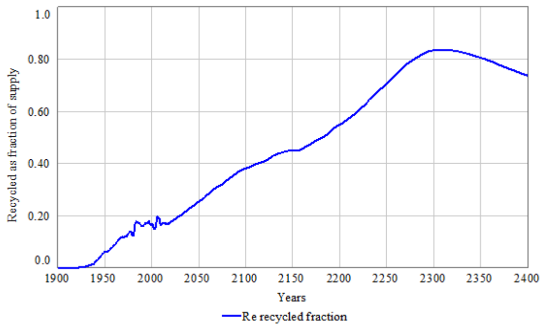
c



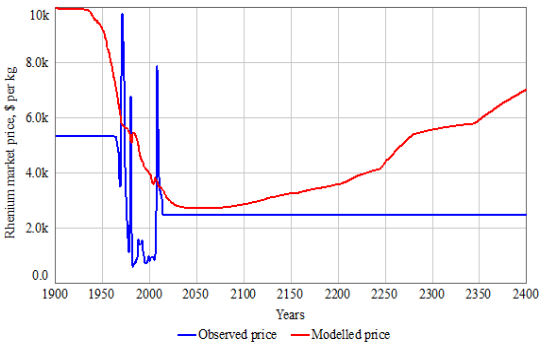
d



e



f



g

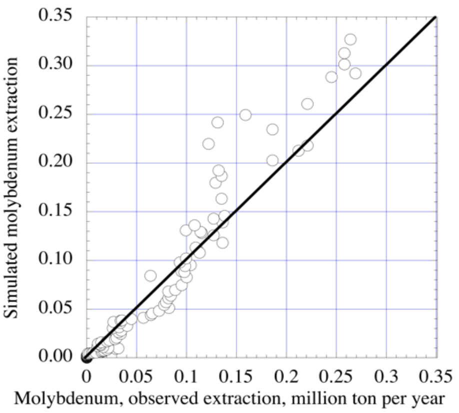
Source: Own Figure, Lund University/University of Iceland

Rhenium is a key strategic metal for high performance jetfighters, and the big powers have sufficient reasons for controlling the market and controlling the market access. How large the industrial internal recycling of rhenium really is difficult to estimate and there are no scientific reports available on it. Rhenium catalysts for the chemical industry are almost completely recycled in a closed cycle, and this rhenium does not really reach the open market. On the average, only 10-20% of the rhenium used today is recycled according to the present setting of the model. Many other settings are possible, however, there is little information to assist in finding the right setting. This will remain as one area for further investigations. What we have estimated was based on conversations with colleagues in the industry, back-calculations and interpretation of data found in secondary sources and industrial internal documents. Super-alloys have trend of increasing recycling, whereas this is weaker in rhenium chemicals. After 2130, the stock in society will be larger than the remaining geological resource.

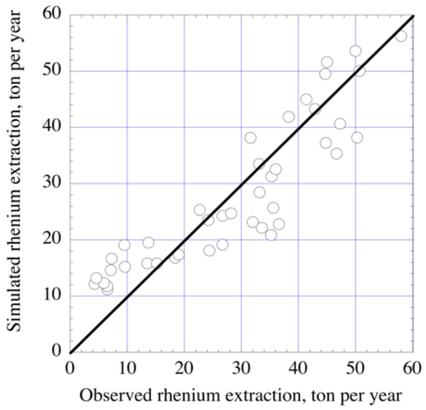
4.1.7.8 Testing the model outputs on observed data

The performance of the model was tested on observed values for extraction, market price and ore grade. Data on extraction for 1900-2012 is available from the United States Geological Survey website (USGS 2015). The same site has metal prices for the period 1964-2015. This was used in the test. The test is shown in Figure 4.1.7.12a shows a test of the ability of the model to recreate the observed molybdenum mining rate. The correlation between the simulated extraction and the observed data shows a correlation coefficient of r2=0.97. The molybdenum price is modelled well with respect to the approximate level and some trends, but some extreme events are not captured (Figure 4.1.7.11g). The reason for this is that the mechanisms that caused the large price fluctuations were not included in the model. The speculation that occasionally took place in the market were not included in the model. Figure 4.1.7.12b shows the simulation of the rhenium extraction as compared to the observed data (circles). The correlation between the simulated rhenium extraction rate and the observations rate is r2=0.81. It appears that the model recreates the mining rates with very good accuracy. No good data is available on ore grade and stocks-in-use in society, preventing a model performance test on these aspects. The price is not captured well, probably because there has not been any real free market for rhenium for strategic and military reasons.

Figure 4.1.7.12. Test of the ability of the model to recreate molybdenum mining rate (r2=0.97), and rhenium mining rate (r2=0.81).



a; Molybdenum



b; Rhenium

Source: Own Figure, Lund University/University of Iceland

4.1.7.9 Discussions

Recycling and lack thereof

The recycling degree of the metals included in super-alloys is far too low for these alloys for a number of reasons. Far too low from a sustainability point of view, where a finite resource is sought to be preserved. Exhaustion of the primary resource by 2200 and total lack of it after 2300 is by no definition of sustainability acceptable. Many of the super-alloys have such good corrosion resistance, as well as tantalum and rhenium alone, that they can be considered as precious metals. Generally, they are also very temperature resistant, creating additional challenges for recycling by chemical separation. Recently, better chemical methods have been developed for recycling superalloys.

Another pathway is by sorting alloys by contents during recycling and reconstitute alloys by corrective re-alloying and hot fusion. Niobium is lost by dilution into recycled iron and stainless steel scrap.

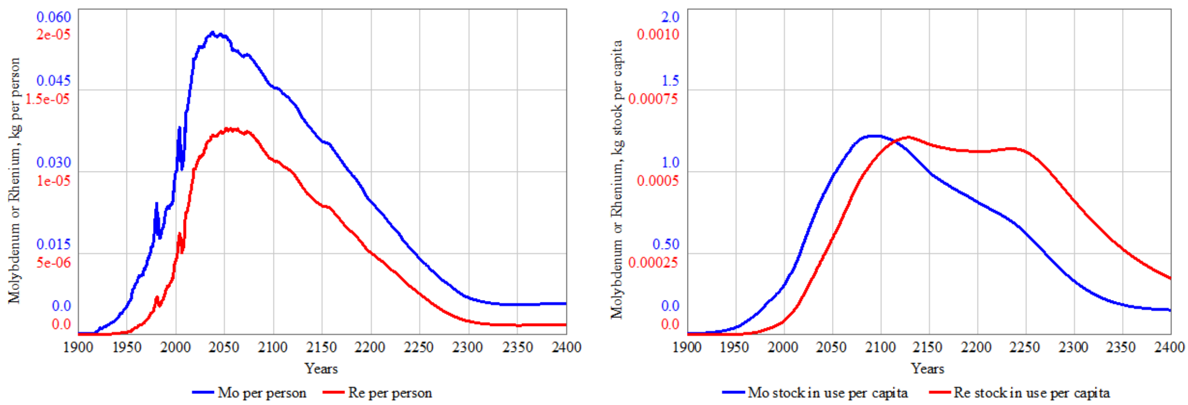
Risks of scarcity

The diagrams suggest that for a period, molybdenum may serve as a substitute for niobium, but not the other way around because niobium decrease before molybdenum and that the amount niobium supplied is significantly less that molybdenum. Tantalum and rhenium decrease at the same time. However, the tantalum production is 10 times larger than the rhenium production and may serve as a substitute when possible. Rhenium and tantalum face hard scarcity because of reserve exhaustion. Molybdenum face soft scarcity because of lower ore grades and higher extraction prices.

Loosing rhenium availability implies a step back towards lower performance for superalloys. Business-as-usual will lead to increased risk for scarcity. This study shows that failure to close the molybdenum and rhenium cycles substantially better than today may have serious effects on the long-term supply ability for these materials. After 2150, they will be unavailable in the amounts they are today under a business-as-usual scenario. The implications are very evident for modern civil and military aviation and the production of high technology military equipment in particular.

Figure 4.1.7.13a shows the supply per capita for molybdenum and rhenium. Molybdenum peaks in 2045 and rhenium in 2052 and both decline after that. Figure 4.1.7.13b shows the stock-in-use for the two metals. The stock-in-use per person is a proxy for the utility in society that these metals provide. It shows that molybdenum utility peaks in 2075, and decline after that, passing 2015 level after 2350. Rhenium stay on a plateau from 2125 to 2250, after which it declines.

Figure 4.1.7.13. The diagram on the left shows the supply per capita for molybdenum and rhenium. The diagram to the right shows the stock-in-use for the two metals.



a b

Source: Own Figure, Lund University/University of Iceland

Factor X

Table 4.1.7.11 shows a summary of extracted, supplied, recycled and lost material to 2400 AD for molybdenum and rhenium, and the Factor X achieved under business as usual.

Factor X is the cumulative amount of metal supplied to the market, divided by the cumulative amount extracted from the ground (Equation 4.1.7.12). The low value of Factor X suggests that the material used efficiency is poor for both molybdenum and rhenium at present.

1. A factor X of 1.3 for molybdenum should be considered as poor performance from the perspective of material conservation and transition to a circular economy. A Factor X value of 2-2.5, on par with the value seen for copper, should be a target for molybdenum (Recycling rate of 50-

60%). That would preserve the global molybdenum for much longer, as well as stabilize the price better than at present.
2. The Factor X value if 1.1 for rhenium, must be characterized as unacceptable for such an important metal. A more appropriate target for Factor X for rhenium would be a value at least better than 2, and preferable having 3 or 4 as the long-term goal (Recycling of 65-75%). That would be approaching the standard for gold, a metal with a price similar to rhenium.
3. Any rhenium use with less recycling than 50% should be taxed to a cost more than the cost of recycling.

Much molybdenum is lost into scrap iron. More molybdenum may be recycled without complex metal separation processing by more careful sorting of scrap according to alloy. Then the alloy can be re-alloyed and recycled. This should be possible for many alloys that today are diluted into unsorted scrap iron. It is apparent that the recycling of these metals is far too low even if we include internal industry recycling. It should be made policy to preserve these metals far better than today.

Table 4.1.7.11. Summary of cumulative extracted, supplied, recycled and lost material to 2400 AD for molybdenum and rhenium, and the Factor X achieved under business-as-usual. Amounts in ton

Metal	Metric ton metal				Factor X
	Extracted	Supplied	Recycled	Lost	
Molybdenum	63,700,000	85,000,000	21,300,000	63,400,000	1.3
Rhenium	16,800	18,200	2,400	16,400	1.1

Both policy, social processes and market mechanisms are needed

Should a global metal scarcity occur, the reader should well take notice that it is realistic to assume that “the market will not fix it”. This is a fact sometimes disliked by many economists as well as some politicians, for them derived from an ideological conviction. There are sufficiently many empirical cases available to prove beyond any reasonable doubt that scarcity issues are not efficiently solved by the market mechanisms alone. That is a scientifically provable fact, and political opinion would be redundant to that fact. Failure to take such experiences in, constitutes aspects of fundamentalism, and is damaging in the political work of governance. We must conclude that the “market mechanisms alone”-approach as a solution to avoid scarcity is a scientifically proven dead end (Fukuyama 2006, Acemoglu and Robinson 2013, Heinberg 2001, Bardi 2013, Sverdrup et al., 2015a,b,c). Still insisting on such an approach, is inviting failure and irreparable scarcity situations (Ostrom et al., 1994, Acemoglu and Robinson 2013). It should be met with the resistance it scientifically deserves. There are no internationally unified policies for recycling. In the international arena there are several bodies of the UN working on the issue (UNEP) and the International Resource Panel. Other important organizations are either happily ignoring this message, or behave in a way that suggests straight denial (OECD, IEA, OPEC). In the political arena, there seems to be little real interest in natural resource sustainability issues. We suggest scientists have a responsibility of speaking about this with a load and clear voice as we can see how the “soft” approach has resulted in nothing until now.

Reflections on the time span considered

Some may react to the long time span we have adopted, we sometimes hear exclamations like “you cannot do that” or “it is impossible to run predictions that long”. There is actually a necessity to run the models this long, because of the long delays in the system. As we can see, the events occur between 2050 and 2250. Models that are confined by mass balance are much longer stable, for the simple reason that mass balances are one of the fundamental first principles of the world. It is also important to make a distinction between predictions that are normative, and scenarios, that explore what would be possible outcomes from a set of assumptions about the future.

4.1.7.10 Conclusions

The study shows that molybdenum and rhenium will only last for another 150 years under the present metal conservation paradigm (Figures 4.1.7.6, 4.1.7.11, 4.1.7.8). These metals are so important for high performance alloys and advanced technologies, that much better metal conservation efforts need to be made than those that presently are being applied. The present policy that "the market will fix it" has very evidently failed to regulate the issue of molybdenum and rhenium conservation so far. It illustrates the need for governmental participation in creating a fair, stable and transparent market that is sustainable (Sverdrup et al., 2015c). The model performs well in tests against observed data and is thus a valid assessment tool. The model predicts a significant decline in molybdenum and rhenium supply after 2100 with severe scarcity after 2150 under the present regime of recycling. In the molybdenum-rhenium system the losses are too large for the system to be sustainable for more than 150 years. Most chemical uses are not followed by recapture. In metallic uses, a large portion is lost to dilution into general stainless steel scrap. The model predicts a steady decline in molybdenum ore grade with time. Under business-as-usual the molybdenum reserves will provide sufficient supply to about 2150, after which it declines quickly.

4.1.8 SPECIALITY METALS sub-module – NIOBIUM and TANTALUM

4.1.8.1 Introduction

Molybdenum (Mo), rhenium (Re), niobium (Nb) and tantalum (Ta) are metals that are key for making specialized stainless steel alloys that preserve strength and corrosion resistance at high temperatures, important for advanced technologies used in civil and military applications. These metals are valuable and sought after for manufacturing of super-alloys and as catalysts for chemical reactions. The demand for these metals is normally higher the supply, and they are already scarce with a high market price. Scarcity of natural resources in the future may cause trouble for continued technology development and the further development of the economy in the long term (Acemoglu and Robinson 2013Bardi 2013, Busch et al., 2014, Heinberg 2001, Hirschnitz-Garber et al., 2015, Morrigan 2010, Meadows et al., 1972, 1974, Nickless et al., 2014, Nassar et al., 2012, Ragnarsdottir et al., 2015, Robinson and Menzie 2012). This study intends to address some of these issues for niobium and tantalum. They find use in turbine blades, jet engines, rocket engines, nuclear energy technologies and specialized chemical environments. A strategically important use of these metals is in high temperature section of turbines and jet engines. Tantalum and niobium occur in the carbonatite type of polymetallic ores, often associated with rare earths, thorium, uranium, yttrium and zirconium. Tantalum and niobium also occur in pegmatites. The major niobium commodities are ferroniobium, niobium metal, ore, and oxide. The major tantalum commodities are tantalum unwrought and wrought alloys, metal, sponge, ore, and scrap. There is a trend that many metals are extracted from polymetallic deposits, rather than single metal ores.

4.1.8.2 Objectives and scope

Develop a model for the production and cycling of tantalum and niobium in society. We intend to estimate the available resources and reserves for extraction, the present annual production and historical production and to give an outline of what these metals are used for and where they are strategically important. The objective is to use a systems dynamics model to explore the term sustainability of the supply system and its limitations.

4.1.8.3 Earlier use of system dynamics

We have employed these tools and methods earlier for assessments made for natural resources in general (Sverdrup et al., 2012b, 2013, 2015b), copper, silver, gold, aluminium, iron, lithium and the platinum group metals (Sverdrup et al., 2014a,b, 2015a,b,c, 2016a,b), and further studies are being prepared for (1) stainless steel, nickel, manganese, chromium, (2) copper, lead, zinc, and (3) technology materials like indium, selenium, antimony, gallium and germanium, and (4) molybdenum and rhenium. There are other simpler modelling methods available (Hubbert's model, Mass Flow Analysis, Flow-sheeting with spread-sheets, but we have avoided these as they lack all market dynamics and most of the feedbacks known to be present in the real global mining, supply and metal trading systems (Mohr 2012, Diago et al., 2010, Bardi and Lavacchi 2009, Sverdrup et al., 2015a, Sverdrup and Ragnarsdottir 2014).

4.1.8.4 Background Tantalum

Tantalum was reported as a new metal in 1802 by the Swedish chemist Anders Gustav Ekeberg, who was professor of chemistry at Uppsala University. It is a very rare metal, the estimated resource is of the same size as that of platinum group metals and gold. It has a very high melting point (3,010oC) and high specific density. It is very chemically resistant up to 150oC, at low temperatures it is a precious metal. The potentially major primary tantalum mining source is Australia, although both there and in in North America, the main mines are shut down or in waiting for better prices. Other producing regions include Brazil, Ethiopia, Central Africa, Nigeria, Mozambique, and China. The main traded commodity is tantalum metal. The typical ore grade is 0.05-0.03% Ta2O5 in the bulk tin ore called cassiterite (0.035-0.02% tantalum weight), old tin smelting slags and other poly-metallic ores. The single largest operating primary tantalum mine is the Mibra operation of Companhia Industrial Fluminense in Nazareno, Brazil.

In the 1980's most tantalum originated as a by-product of tin smelting and from reworking of tin smelter slags. Thailand and Malaysia accounted for nearly 40% of the world's tin production of 236,000 ton of tin in 1980. The tin ores of Malaysia and Thailand were rich in tantalum and slags resulting from the tin smelting contained between 10-15% tantalum oxide (7-10% tantalum content). But by 1994, only 20% was derived from this source, and it has decreased further as other sources have been developed and the slag stocks has declined.

In 2011, 51% of the extracted tantalum came from professional hard rock mining, 16% from artisanal mining, 10% from tin slags and 23% from undefined sources in central Africa (Minor Metals Trading Association 2015). The tantalum supply market is volatile, and in recent years, Africa has become a much more prominent source of tantalum, much from unofficial mining on a small scale, normally outside any regulations or governance. Table 4.1.8.1 shows the approximate production in a number of countries. As regards secondary supply, there are several sources of tantalum extraction:

1. Old low tantalum grade tin slags (About 0.02-0.04% tantalum content in the original ore, 5-10% of the extraction flux and declining), primarily from South East Asia, mostly processed in China and Europe
2. From niobium mining (About 10% of the extraction flux, tantalum content varies from 0.2% to 0.05%).
3. From tungsten (wolfram) mining (About 0.02% content, 1-2% of the extraction flux)
4. From the United States National stockpile (Sold off at irregular intervals, not a source to be relied on for the future)
5. From extraction from industrial and consumer scrap. Increasing with separation technologies. Comparable to a high grade ore as substrate.

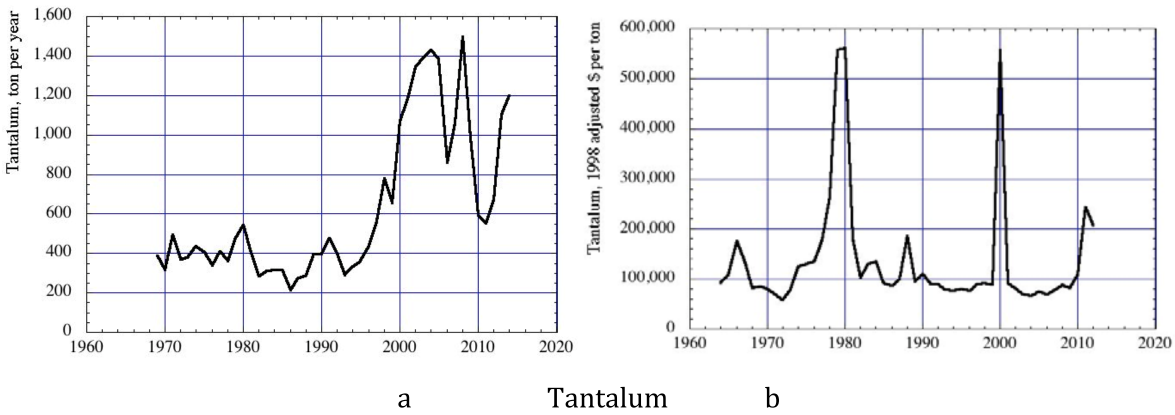
Tantalum's main use at present is in capacitors for consumer electronics, in competition from aluminium, niobium, and multi-layer ceramics. 45-55% of the tantalum demand is from capacitor production. Capacitors made of tantalum remain optimum for various applications.

Tantalum is used in sputtering targets, corrosion-resistant products, superconductors, carbides for cutting tools, chemicals for high end optics and is used in super-alloys for turbine blades in jet engines and industrial gas turbines. Tantalum has found some use for chemical tubing and reaction vessel equipment expected to operate under corrosive conditions. Tantalum is totally resistant to all acids at temperatures up to 150oC and at normal pressures, and tantalum alloys can be outstanding in this role, competing with titanium, zirconium and platinum group metals. High price for these specialized alloys has been seen as been a limiting factor, as the initial investment requires high capital input.

Overall tantalum demand has fluctuated recently between 1,400-2,000 ton of tantalum per year, but following the industrial downturn of 2008-2009, demand plummeted, prices went down, and the production fell to just over 600 ton/year. In 2014-2016 it was back to old levels of about 1,300 ton per year. It is estimated that the total supply including recycling was about 1,650 ton per year, implying that about 20% of the supply originate from recycling (Minor Metals Trade Association 2015).

Figure 4.1.8.1a shows the world production of tantalum and Figure 4.1.8.1b the market price. The production and the price show large variations, and a recent decline in tantalum production. The Tantalum price is higher than the niobium price, they normally stand at a ratio of Ni:Ta = 10:, their ratio of production was 46:1, suggesting that tantalum is not expensive. Tantalum is a specialty metal which earlier had a very narrow application, this has changed. It is not certain that there is a proper market price mechanism operational for tantalum. The tin ore called Cassiterite is a high gravity, polymetallic ore, it is extracted from small placer deposits by informal and artisanal mining when outside China. China and Indonesia are the largest producers at the moment (See Table 4.1.8.1 with an overview for tantalum, niobium, tungsten and tin). In nature, tin and tungsten tend to follow each other for geochemical reasons as well as because they have a high specific density and are also in nature enriched by naturally occurring gravimetric processes (Placer ores). Tin is for similar reasons sometimes associated with these metals in gravimetric deposits. Tin ore from certain deposits contain some tantalum, and reprocessing tin slag has supplied a significant amount of the metal in the past.

Figure 4.1.8.1. The global extraction of tantalum (a) and the inflation-adjusted market price in $ per ton (b). Data compiled from USGS (2015).



Source: Own Figure with data from USGS 2015, Lund University/University of Iceland

Table 4.1.8.1. Tantalum (ta), niobium (Nb), tungsten (W) and tin (Sn) annual metal production distributed among the major producing countries in the time 2012-2014. The largest niobium and tantalum resources are probably located in Brazil and Australia. The amounts are metric ton metal. Tin and tungsten are often associated in the same deposits. Some tin ores have significant contents of tantalum, that gets enriched in the smelter slag after removal of the tin. Niobium generally has a significant content of tantalum and vice versa.

Geographical location	Deposit type	Ta Tantalum	Nb Niobium	W Wolfram	Sn Tin
Rwanda	Ta+Nb Sn+W	250	65	 700	 2,000
Congo	Ta+Nb Sn+W	180	25	 800	 3,000
Mozambique	Ta+Nb	85	34	-	-
China	W Sn	30 30		68,000	 125,000
Nigeria	Ta+Nb	60	40		?
Ethiopia	Ta	40	7	?	?
Burundi	Ta	14	?		?

Brazil	Nb+Ta Sn+Ta	110 30	58,000		 12,000
Australia	Nb W+Sn	0	200	 600	 6,100
USA	Ta	40	?	?	?
Canada	Nb+Ta W	50	4,400	 2,200	 ?
Indonesia	Sn	-	-	-	84,000
Peru	Sn	-	-	-	23,700
Vietnam	W	-	-	3,000	
Bolivia	Sn+W	-	-	1,300	18,000
Portugal	W	-	-	700	-
Austria	W	-	-	850	?
Burma	Sn	-	-	-	11,000
Russia	Diverse sources	?	2,200	3,600	600
Thailand	Sn			?	1,930
Others	Ta, Nb, W, Sn Cu, Zn, Pb	191	29	250	5,000 5,600
Sum, ton metal per year		1,200	65,000	82,000	296,000

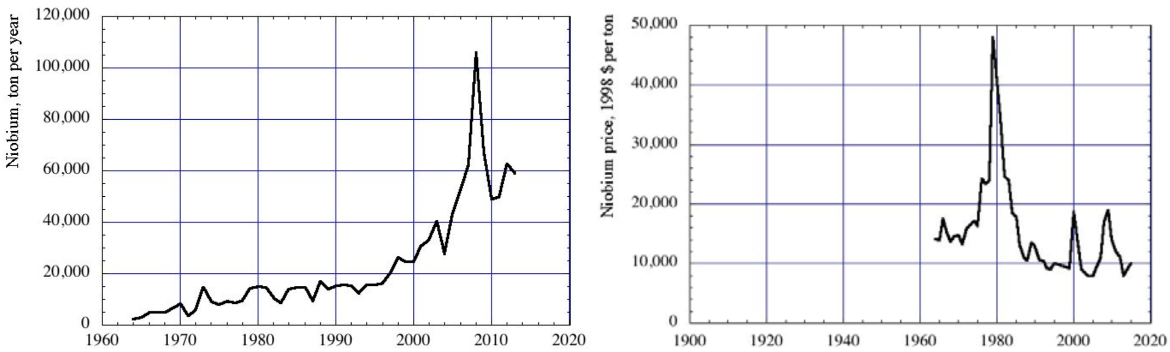
Background Niobium

The English chemist Charles Hatchett found niobium in 1801 and named it columbium (Hatchett 1802a,b). Columbium was the old name and the IUPAC determined name is now niobium. Niobium is an important metal for alloying into specialized stainless steels (20%) and super-alloys (25%). It shares many mechanical and chemical properties with tantalum. The melting point is high, 2,468oC. It is used in super-alloys along with tantalum, molybdenum, nickel, cobalt and rhenium for high performance uses. In some types of general carbon steels, it is used in concentrations of 0.1 to 0.01%, this accounts for 50% of all niobium use (Lele and Bhardwaj 2014). Only a few countries produce significant amounts of niobium. Brazil (89% of the supply), Canada (7%) and Russia (3%) are the most prominent producers (Table 4.1.8.1). Niobium is traded mainly as ferroniob for addition in iron smelting, but also as niobium metal for superalloy production. The main niobium source is the mineral pyrochlore.

The largest extractable resources are located in Brazil, Australia, Canada, Nigeria and Norway. The largest known deposit to be mined is at Araxa in Minas Gerais in Brazil, the average ore grade is 2.5-3% Nb_2O_5. The resource at Araxa, Brazil, is estimated to contain about 460 million ton of niobium ore, which translates to 9 million ton of niobium metal content. This corresponds to about 1.75-2.1% niobium weight content. Another important deposit is operated by Anglo-American at a typical ore grade of 1.4% Nb_2O_5. This resource contains 20 million ton of niobium ore, corresponding to 0.2 million ton of niobium metal, about 1% niobium metal weight content), and Niobec, in Quebec, Canada with a typical grade of 0.41% Nb_2O_5, the resource is 8,000 ton of niobium metal contained, the ore grade is 0.29% niobium metal weight content. The niobium metal and ferroniob market prices have been volatile in a slowly declining trend. Other uses like chemicals only consume a minor fraction (5%). Figure 4.1.8.2 shows the world production of niobium and the market price. The mineral columbite ($FeNb_2O_6$) and coltan ($Fe(Ta,Nb)_2O_6$). The term “Coltan” derives from “columbium-tantal minerals”.

Coltan minerals are occur in pegmatite intrusions, and in alkaline intrusive rocks. Less common are the niobate minerals of calcium, uranium, thorium and the rare earth elements. Niobates are exampli-fied by the minerals pyrochlore (Ca2Nb2O6(OH)) and euxenite (CaNb1.95Ta0.05O6). The largest de-posits of niobium have been found associated with carbonatites and as a constituent of pyrochlore. Typically, niobium ore contains traces of As, Bi, Sn, W, Ti, Hf, Sc, Rare Earth Elements, V, Pb, Zn, Zr, Th and U (Lehmann et al. 2014). The extraction of such extra metals are gaining increasing importance for mine profitability in many places. For niobium, Brazil is a dominating producer (89%). For tungsten, China is a dominant producer (83%).

Figure 4.1.8.2. The global extraction(a) of niobium and the inflation-adjusted market price in $ per ton (b). Data compiled from USGS (2015).



a Niobium b

Source: Own Figure with data from USGS 2015, Lund University/University of Iceland

4.1.8.5 Methods and theory used

Reserves

The methods used in this study are several. We have conducted systems analysis on the supply and extraction systems for tantalum-niobium-tin and wolfram, using causal loop diagrams and flow charts (Haraldsson and Sverdrup 2005, Senge 1990, Sterman 2000). We mapped how the reserves contain the metals in different amounts and how their extraction is linked in a system. The reserves estimates are based on classical geological estimates, and the allocation of extractable amounts according to ore quality, stratified after extraction costs (Sverdrup et al., 2015a,b,c,d). Each ore grade category for any of the metals was mined when the market price allows for it.

4.1.8.6 Data and materials

Production rates

The data was gathered by going through country reports and other published literature. The numbers add up to about double the estimates from the USGS (2015). Table 4.1.8.2 shows tantalum, niobium, wolfram and tin production distributed among the major producing countries. For niobium, Brazil is a totally dominating producer (90%), for wolfram, China is totally dominating production at the mo-ment (83%), for tin, Indonesia and China are the dominating producers (28% and 42%). Tantalum and tin mining is distributed among many small and non-industrialized producers, often on the side of the official economy. Table 4.1.8.2 and Table 4.1.8.3 show an overview of production in 2012 estimates for tantalum, niobium, wolfram and tin, and the dependency on other metals extraction.

Table 4.1.8.2. Overview of production in 2012 estimates for tantalum, niobium, tungsten and tin. All amounts shown are expressed in metric ton per year. *=Tin is the dominating secondary source for wolfram but other poly-metallic ores produce some wolfram.

Metal	Production 2012-2015 Ton per year	Source type of ore							
		Nb		Ta		W		Sn	
		% in other metal	Flow ton/year	% in other metal	Flow Ton/yr	% in other metal	Flow, ton/yr	% in other metal	Flow ton/year
Wolf-ram	82,000	0.2	160	0.013	10	-	73,116	1	820
Nio-bium	65,000	-	64,480	0.2	195	0.02	20	0.02	20
Tan-talum	1,200	33	400	-	875	0.3	4	0.5	6
Tin*	330,000	0.02	60	0.04	120	3.0	8,860	-	286,000
Cop-per Zinc Lead	18,000,000 12,000,000 4,000,000	- - -	- - -	- - -	- - -	- - -	- - -	- - -	44,000
Sums	-	-	65,000	-	1,200	-	82,000	-	330,849

Table 4.1.8.3. Niobium and tantalum extraction degree of dependency quantified.

Metal	Production 2012 ton/year	Niobium in other metal %	Niobium flow ton/year	% of total Niobium extraction	Tantalum in other metal %	Tantalum flow ton/year	% of total tantalum extraction
Wolf-ram	82,000	0.2	160	0.3	0.013	10	1
Nio-bium	65,000	n.a.	64,480	99.0	0.2	195	16
Tanta-lum	1,200	30	400	0.6	n.a.	875	73
Tin	300,000	0.02	60	0.1	0.04	120	10
Sums	-	-	65,000	1.0	-	1,200	-

Deducting from resource data the need for a specific simulation model structures

By looking at the numbers presented in Table 4.1.8.2, we may make a number of conclusions on what sources contribute to the supply of different metals:

1. For niobium, secondary extraction is negligible in volume (1%). Tantalum mining is a minor source
2. For tantalum, the contribution from niobium (16% of extraction) and reprocessing tin slags (10% of extraction) are important, secondary extraction is about 27% of the global production.

3. For wolfram, secondary production is about 4-6% of the global production, mostly from tin mining.
4. For tin, secondary production contributes to about 15-20% of the global production, mostly from copper, lead and zinc refining, but a small amount tin comes from wolfram mining (1%), a total of 16% of the global production.

It is evident from the table that the extraction of niobium, and tin is important for tantalum, but not the other way around. Wolfram, niobium and tin have their own mines and contributions from the other metals are insignificant. For niobium, secondary extraction is less than 1% of the supply (Table 4.1.8.2 and Table 4.1.8.3). For tantalum, the contribution from niobium (16% of extraction) and tin (10% of extraction) are important enough to model it. Wolfram and tin mining contribute small amounts of tantalum and niobium at present and wolfram modelling will not be included here. Thus, no wolfram model module is needed. The conclusion is, that the following model structures can be foreseen for modelling the supply of niobium, tantalum and tin, taken from the authors library of metal models (Sverdrup et al., 2014a,b, 2015a,b,c, 2016, a,b,c):

1. Niobium mining and market supply modelling: One unit module for niobium, supported by the STEEL module to estimate the niobium demand.
2. Tantalum mining and market supply modelling; One unit module for tantalum, and support modules including the niobium sub-module, supported by the STEEL sub-module to estimate the niobium demand, and a separate module for tantalum extraction from tin slags.
3. Tin mining is modelled in a separate module, but supported by the earlier developed BRONZE model. From the mining or high grade tin ores, a tantalum-containing slag is produced. This is processed in China and Germany to extract tantalum metal.

It is evident from Table 4.1.8.2 that the extraction of niobium and tin is important for tantalum, but not the other way around. Tungsten, niobium and tin have their own mines and contributions from the other metals are insignificant.

The extractable reserves and resources of niobium and tantalum

Assemble data on reserves and resources, a number of literature sources were consulted (Alves et al., 2015, Bastos-Neto et al., 2009, Berger et al., 2009, Beurlen et al., 2014, Biruabarema et al., 2014, Chicharro et al., 2014, Eckstrand and Hulbert 2007, Fritz et al., 2013, Ginzburg and Fel'dman 1974, Gupta and Suri 1994, Hulsbosch et al., 2014, International Business Publications, 2013, 2015, Kravchenko and Pokrovsky 1995, Lele and Bhardwaj 2014, Lehmann et al., 2014, Keseler and Wilkingson 2013, Kwatra et al., 2012, Ludington and Plumlee 2009, Melcher et al., 2015, Möller 1989, Polinares 2012, Safirova 2013, Nassar et al., 2012, Mudd and Jowitt 2014, Mudd 2009, Nickless et al., 2014, Nuss et al., 2014, Polyyak 2011, Sverdrup et al. 2013, Tilton and Lagos 2007, UNEP 2011a,b,c, 2013a,b,c, USGS 2009, 2015, Vulcan 2013). There are a number of sources available for reserves, most have been summarized by the United States Geological Survey in their yearbooks (USGS 2015). The USGS (2015) list the niobium reserves as about 9 million ton. This is only a smaller part of the story, as the resources, the amount existing but not yet found ("hidden"), is what matters.

Tantalum is mined both from hard rock tantalum deposits, from old tin smelting slags and as a by-product from niobium mining. There are also small contributions from tungsten and gold mining from skarns, but these volumes are insignificant for the global supply. Much of the mining in Central Africa is done on a small scale, and with a low degree of mechanization. The area where it is mined is troubled with civil war and failed governance. Table 4.1.8.4 to Table 4.1.8.7 show an overview of reserves and resources important alloying metals for super-alloys. Table 4.1.8.4 shows tantalum and niobium reserves and resources by country 2012. The data was gathered by going through country reports and other published literature. The actual data in Table 4.1.8.4 for both adds up to significantly more than what the USGS (2015) estimates are available for tantalum and niobium extraction. This indicates that the USGS has significantly underestimated the resource size. Table 4.1.8.5 shows our overview of remaining reserve in 2012 estimates for tantalum, niobium, tungsten and tin. Note that our estimates of the resources and ultimately recoverable resources are significantly larger than the estimates of the USGS (2015). Table 4.1.8.6 shows a compilation of reserves and resources in 2012 for niobium and tantalum but also other important alloying metals for making superalloys. Table 4.1.8.7 shows an overview of important alloying metals for superalloys.

Table 4.1.8.4. Tantalum and niobium reserves and resources by country 2012. The data was gathered by going through country reports and other published literature.

Location	Tantalum, ton			Niobium, million ton		
	Known reserves	Hidden resources	Total resources	Known reserves	Hidden resources	Total resources
South America	87,000	200,000	287,000	4.0	30.0	34.0
Australia	45,000	90,000	135,000	0.3	4.7	5.0
Canada	4,000	10,000	14,000	0.2	1.8	2.0
USA and Mexico	3,000	5,000	8,000	0.0	0.2	0.2
China/East Asia	7,800	4,000	11,800	0.1	0.3	0.4
Central Africa	3,200	5,000	8,200	0.5	1.5	2.0
Other Africa	12,500	10,000	22,500	0.5	3.0	3.5
Russia, Ukraine, Kazakhstan	5,000	10,000	15,000	0.5	1.5	2.0
Europe	5,000	5,000	10,000	0.0	0.5	0.51
Rest of the world	7,500	15,000	22,500	0.5	1.5	2.0
Sum	180,000	354,000	534,000	6.6	45.0	51.6

Table 4.1.8.5. Overview of remaining reserve in 2012 estimates for tantalum, niobium, tungsten and tin. All amounts shown are expressed in metric ton. Note that our estimates of the resources and ultimately recoverable resources are significantly larger than the estimates of the USGS.

Metal	USGS reserve estimate	USGS resource estimate	URR, our estimate	Dug up before 2010	Typical workable ore grades	Average	Lower cut-off in ore
	Extractable, million ton				Ore grade, % weight content		
Wolfram	3.3	-	20	2.7	0.1-1.5	0.4	0.10
Niobium	4.3	5.4	55	1.5	0.1-4	0.4	0.01
Tantalum	0.150	0.26	0.6	0.11	0.01-0.4	0.03	0.01
Tin	5	50	110	11.0	0.02-1	0.05	0.01

Table 4.1.8.6. Overview of reserves and resources in 2012 for important alloying metals for superalloys. Amounts shown are expressed in million ton contained.

Metal	Known extractable amount	Additional extractable amount	Extractable amount metal	Extracted before 2012	URR	Wild cards	
						Resources of unknown extractability	Seafloor nodules
Molybdenum	14	34	48	7	55	40	50
Rhenium	0.005	0.010	0.015	0.0014	0.016	0.005	0.010
Cobalt	8	15	23	6	29	30	40
Niobium	6.6	40	46.6	5	51.6	10	10
Tantalum	0.1	0.4	0.5	0.02	0.52	0.3	?
Zirconium	10	152	162	18	180	60	-
Hafnium	0.1	3	3.1	0.05	3.15	0.6	-

Table 4.1.8.7. Overview of important alloying metals for superalloys

Metal	Production in 2015 ton per year	Annual extraction as % of URR	Dynamic model peak year estimate	URR Million ton	Dependent Extraction	Hubbert's Model valid?
Molybdenum	0.27	0.5	2032	55	Partly	Yes
Rhenium	50	0.3	2035	0.016	Yes	No
Niobium	65,000	0.3	2055	52	No	Yes
Tantalum	1,200	0.4	2045	0.5	Partly	No

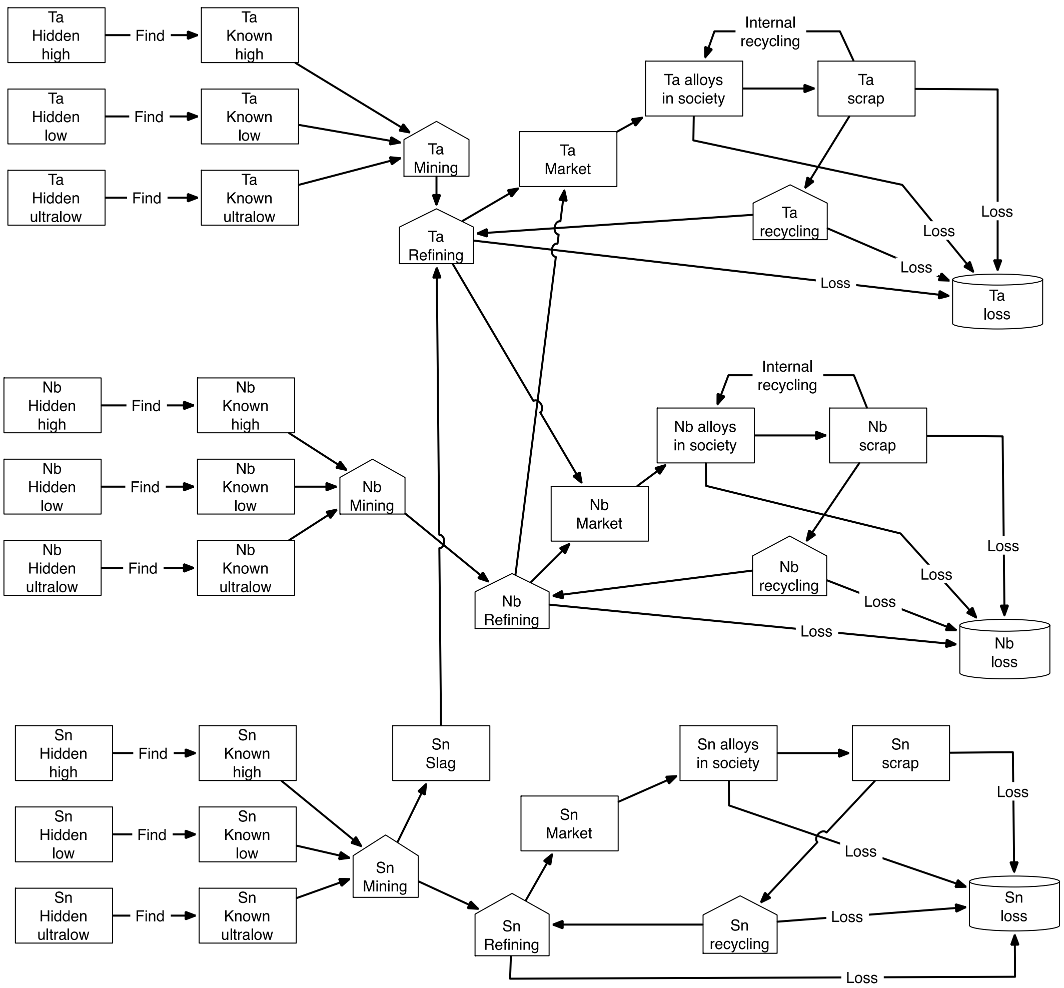
4.1.8.7 The model description

Figure 4.1.8.3 shows a basic flow chart for the Ta-Nb-Sn submodel. Metal flows from ore bodies and from metal produced as a by-product of other metal mining. Metal flow into the market, gets stored in products in society and are eventually scrapped or lost. The losses are of two types: One type is dissipative losses, where the metal is irretrievable lost in general waste, landfills or by wear and tear and lost irreversibly. A significant part or tantalum and niobium is also lost into steel scrap that may be recycled as iron, thus it is from a functionality point of view, lost. The model has the following stocks used to define the coupled differential equations defined by mass balance:

1. Tantalum
    a. Mineable stocks;
        i. Known
            1. high grade,
            2. low grade and
            3. ultralow grade ore
        ii. Hidden
            1. high grade,
            2. low grade
            3. ultralow ore

        iii. Tin Slag
            1. Hidden Ta in tin slag
            2. Ta known in tin slag
    b. In society, we distinguish four above-ground stocks;
        i. Trade market
        ii. Stock-in-use in society
        iii. Scrapped tantalum
        iv. Waste rock produced
2. Niobium
    a. Mineable stocks;
        i. Known
            1. high grade,
            2. low grade
            3. ultralow grade ore
        ii. Hidden
            1. high grade,
            2. low grade
            3. ultralow grade ore
    b. In society, we distinguish four above-ground stocks;
            1. Trade market
            2. Stock-in-use in society
            3. Scrapped niobium
            4. Waste rock produced
3. Tin
    a. Mineable stocks;
        i. Known
            1. high grade,
            2. low grade
            3. ultralow grade ore
        ii. Hidden
            1. high grade,
            2. low grade
            3. ultralow grade ore
    b. In society, we distinguish four above-ground stocks;
            1. Trade market
            2. Stock-in-use in society
            3. Scrapped tin
            4. Waste rock produced
4. The BRONZE submodule of WORLD6
5. The STEEL submodule of WORLD6

Figure 4.1.8.3. The flow of tantalum, niobium and tin in the model.



Source: Own Figure, Lund University/University of Iceland

All stocks have defined inputs and outputs, conforming to the generic mass balance equation:

$$Acccumulated = \frac{dS}{dt} = inputs + produced - outputs \tag{4.1.8.1}$$

The inputs are from recycling, and produced from primary extraction. Accumulated is what is accumulated in society either as stocks in use or as useless scrap. Outputs will be what is lost or recycled. Note that recycled is present on both sides, and thus increase the flow through society. To meet a certain supply to society, the larger the recycling, the smaller the primary extraction must be to meet the demand. The ore is mined from the grade with the lowest extraction cost, but with some overlap between the ore grade classes. The mining activity is profit-driven, the profit is affected by the mining cost and the market price. A lower ore grade implies that more rock must be moved to mine the metals. In the model, mining and recycling is profit driven as is shown in Figure 4.1.8.4. Sales of extracted metal together with the price, given the income, the extraction and prospecting give the costs, the profit being the difference. In the model, shift to a lower ore grade only occurs when the market price exceeds the extraction cost in such a way that the profit stays positive. When the profit is lower or goes negative, the mining rate is slowed down, leading to less material in the market, driving the price up. In practice, we have several sources of metal in society; the high, low and ultralow ore grades and the stocks of metal in society that can be recycled. The mining rate follows a rate equation depending on the mineable reserve, profitability of the operation and the available mining technology:

$$r_{mining} = k * m_K^n * f(profit) * g(technology) \quad (4.1.8.2)$$

where rmining is the rate of mining, k is the rate coefficient and mK is the mass of the ore body known and available for extraction, and n is the mining process order. The rate coefficient is modified with ore extraction cost and ore grade. In the model, a delay in mining rate change is considered by using a forward rolling two-year average of the market price. g(technology) is a technology factor accounting for the invention of technologies used in efficient mining, refining and extraction. f(profit) is a feed-back function of market price, increasing mining at higher price and lowering it at lower metal prices. It was defined by the following equation:

$$Profit = Income\ from\ sales - Mining\ costs - refining\ costs - prospecting\ costs \quad (4.1.8.3)$$

In this equation, mining costs, refining costs and prospecting costs all include both variable operations costs and capital costs for infrastructures and equipment, as well as a 10% profit margin. The income is defined as:

$$Income\ from\ sales\ =\ Amount\ sold * market\ price - sales\ costs \quad (4.1.8.4)$$

The profit is driven by metal price and amount extracted, but balanced by the cost of operation. The mining rate is related to the state of the technology, and it has two alternatives: (1) the technology and the financial capacity determines the mining rate and the ore body is always sufficient to supply any-thing attempted to be extracted, (2) the mining rate is unrelated to the technical and financial capacity and limited by the access to the ore body. There are many different definitions of recycling available (Graedel and Allenby 2003, UNEP 2011b, 2013a). The size of the extractable ore body is determined by the rate of extractions (rmining) and the rate of prospecting (rdiscovery):

$$\frac{dm_K}{dt} = r_{mining} + r_{discovery} \quad (4.1.8.5)$$

We use the following equation for the rate of discovery:

$$r_{discovery} = \frac{dm_H}{dt} = -k_{prospecting} * m_H * g(technology) \quad (4.1.8.6)$$

The discovery is a function of how much prospecting we do and how much there is left to find. There are many different definitions of recycling available (Graedel and Allenby 2003, UNEP 2011b). The amount hidden reserve (h) decrease with the rate of discovery. The rate of discovery is dependent on the amount metal hidden (h) and the prospecting coefficient kprospecting.

The prospecting coefficient depend on the amount of effort spent and the technical method used for prospecting. The rate is first order as prospecting is three-dimensional by drilling. mH is the amount hidden in the resource. The basic driving mechanism of mining comes from profits and availability of a mineable resource used in the model. The price is set relative to how much iron or steel there is available in the market. In the equation, f(technology) is a function accounting for improved prospecting techniques coming with modern society and technological development. The extraction rates of niobium (RNb) and tantalum (RTa) in the model were calculated as follows:

$$RNb = rNb \qquad (4.1.8.7)$$

And

$$RTa = rNb * XNb(Ta) + rSn * XSn(Ta) + rTa \qquad (4.1.8.8)$$

Where Xj(i) is the fraction of metal j in mother metal i, and rj is the mining rate of metal j. The causal loop diagram in Figure 4.1.8.4 shows us the mining operation is driven by profit.

For the purpose of clarity, the recycling fraction displayed in results were calculated as follows in this study:

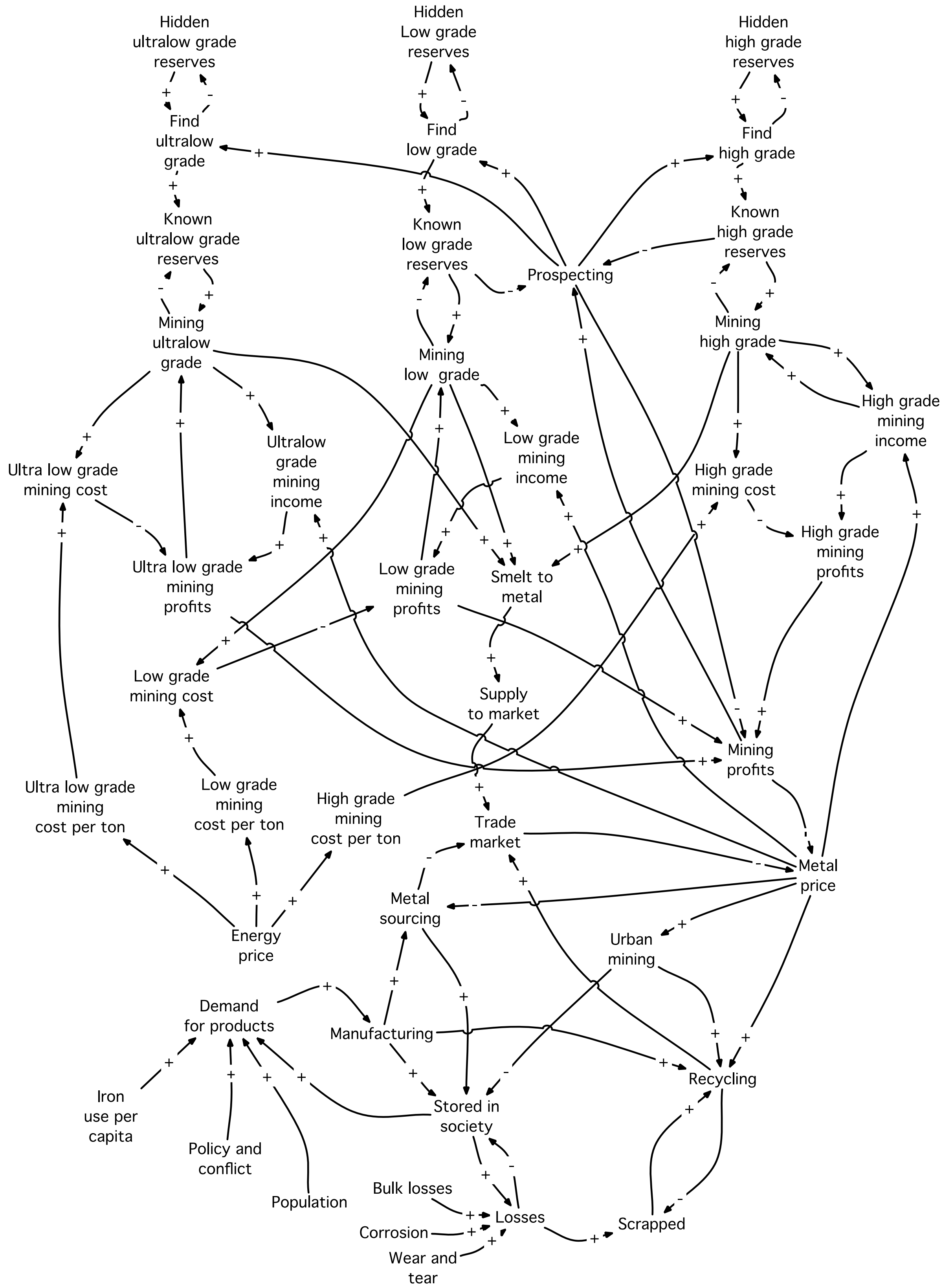
$$Recycling\ fraction = \frac{Flow\ of\ recycled\ metal}{Supply\ from\ primary\ extraction + \ Flow\ of\ recycled\ metal} \qquad (4.1.8.9)$$

Which is equivalent to:

$$Recycling\ fraction = \frac{Supply\ to\ market - Primary\ extractionl}{Supply\ to\ market} \qquad (4.1.8.10)$$

The cost of the mining and extraction operation is mainly determined by two important factors besides cost of investments, the energy price and the ore grade. The price is determined by two factors, that it must stay above the production costs and by the amount in the market.

Figure 4.1.8.4. The mining model as a causal loop diagram for the whole world system. The causal loop diagram describes the causal relationships and the feedback loops in the global supply system.



Source: Own Figure, Lund University/University of Iceland

4.1.8.8 Modeling market price

The extraction operation is driven by profit. This profit is driven by the price and amount extracted, but balanced by the cost of operation. The cost of operation is mainly determined by two important factors beside the cost of investments, that is the energy price and the ore grade. The market price in the model is determined by the amount metal available for transaction in the market. It depends on the balance between deliveries into the market from production and the shipments from the market in response to world market demand. The price also drives the urge for recycling of metal stock from society. Higher price acts as a brake on demand. The traders come to the trading floor with their lots to sell or to buy, and adjusts their sales or purchase amounts as the price goes up and down. If demand is higher than production, the price goes up; in the opposite case the price is moved down. This is a self-adjusting mechanism that balances the trade by adjusting the prices until the demand to buy an amount at a price match the offers to sell an amount at a price. The buyer offer to purchase more at a lower price or less at a higher price, and the sellers offer to sell less at a lower price or more at a higher price. When the price and amount match, the price is set. This profit is driven by metal price and amount extracted, but balanced by the cost of operation. In addition, we have the condition that it must stay above the production costs for the operations to be profitable. The amount in the market in turn depend on the balance between deliveries into the market from production and the shipments from the market in response to world market demand. The price drives the urge for recycling of metal stock from society. The R in the causal loop diagram indicate the reinforcing loops, driving the mining activity and the supply to the market. We have collected a lot of data over the years from the metal trading arenas, and these have been used to create the relationships between amount tradable metal in the market arena and the market price. Some of this is easily accessible from the London, Now York and Shanghai metal trade exchanges, in addition we have used private trading archives collected over the years. From such data, price curves were constructed. In the market, several types of transactions occur:

1. The metal is sold in the market to a buyer, shipped and physically supplied at once. Supply to market and take from market is the same.
2. Forward sale; The metal is sold at once and payment received, but the metal is physically delivered at a later date. Ownership shifts at once, but the money later. Many mines do this to improve liquid funds.
3. The metal is shipped at once but payment is received at a later date. Ownership shifts at once, but the money moves later.

The causality in profits go to income from supply when the amount is supplied and paid at once into the metal exchange warehouse. The same applies with a forward sale, when the metal is paid upfront but physically delivered later. If not, all or part may be paid when the supplied amount has been cleared out from the physical warehouse. When we have no data or very few for the relationship between the market instantly tradable amount and the price for a metal, maybe only one data point for creating a price curve for a commodity, then we will fit a curve to Equation 11:

$$Price = k * (Market\ stock)^n \tag{4.1.8.11}$$

For some metals, we do have market stock information can be found; manganese, chromium, tantalum, rhenium, niobium or tungsten are examples.

4.1.8.9 Defining Factor X

Factor X is used to estimate the resource use efficiency of a material. It is defined as the cumulative amount supplied to the market divided by the cumulative amount extracted from the ground:

$$Factor\ X = \frac{\int_{1930}^{2400} Supplied\ metal}{\int_{1930}^{2400} Extracted\ metal} \qquad (4.1.8.12)$$

Where we have considered the time from 1930 to 2400. Production before 1930 was insignificant. Factor X vary over time, and thus an alternative measure would be Factor x:

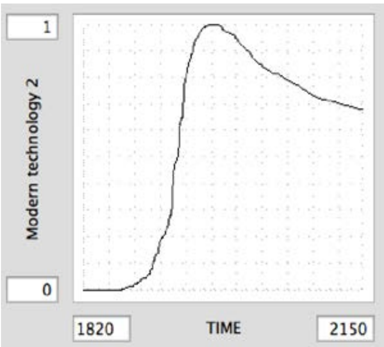
$$Factor\ x = \frac{Market\ supply\ rate}{Extraction\ rate} \qquad (4.1.8.13)$$

This measures the Factor X for the moment and change with time as the extraction rate and the recycling efficiency varies with time.

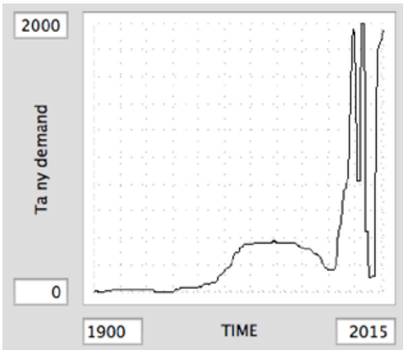
Feedback functions in the model

A number of feedback response curves are employed in the multimetal model. These are shown in Figure 4.1.8.5 and Figure 4.1.8.6. These curves quantify our assumptions on future global demand per person and change in extraction efficiency with time Figure 4.1.8.5 shows the feedback response curves employed in the tantalum sub-module. Figure 4.1.8.6 shows the feedback response curves employed in the niobium sub-module.

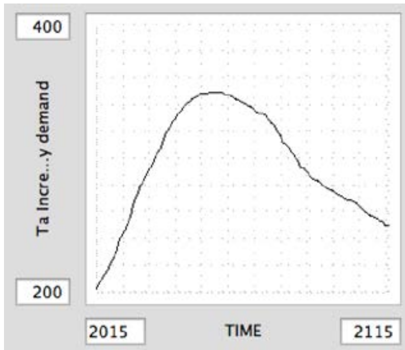
Figure 4.1.8.5. Feedback response curves employed in the tantalum sub-module.



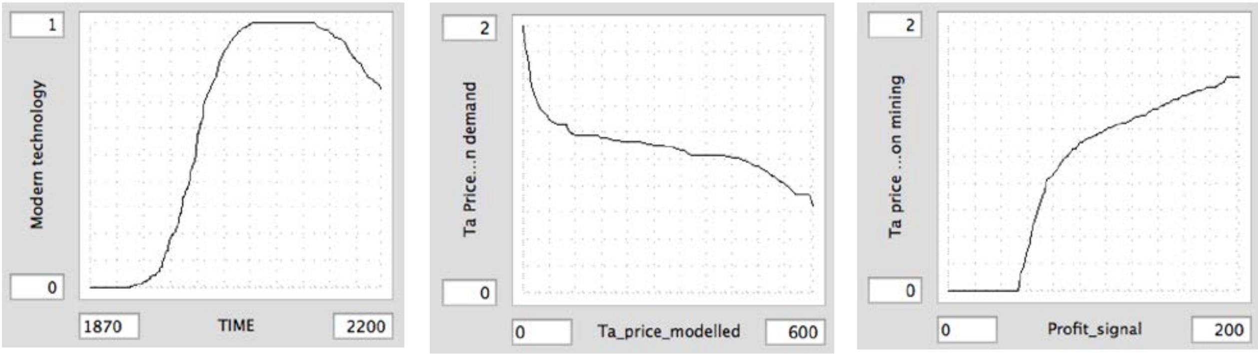
A: Tin slag extraction tantalum technology efficiency



B: Global average tantalum demand, kg per person



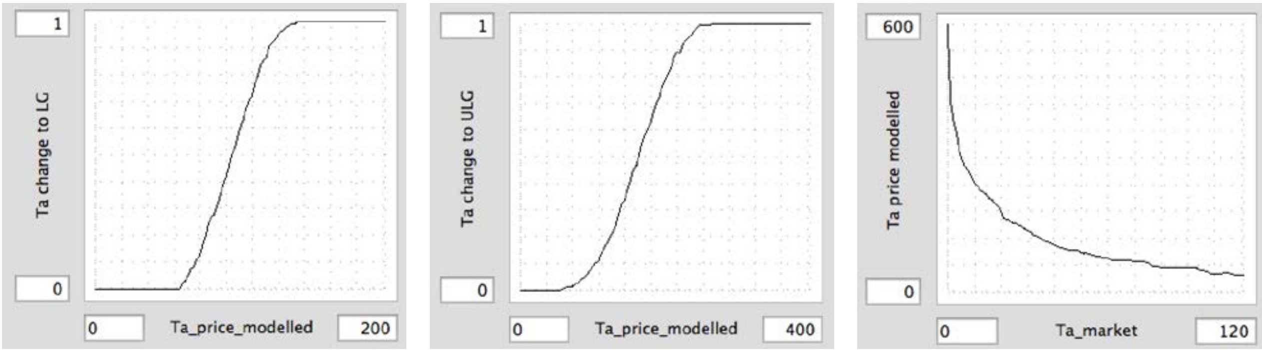
C: Tantalum additional superalloy demand



D: Tantalum hard rock extraction technology efficiency

E: Price effect on global average tantalum demand

F: Profit effect on tantalum mining rate



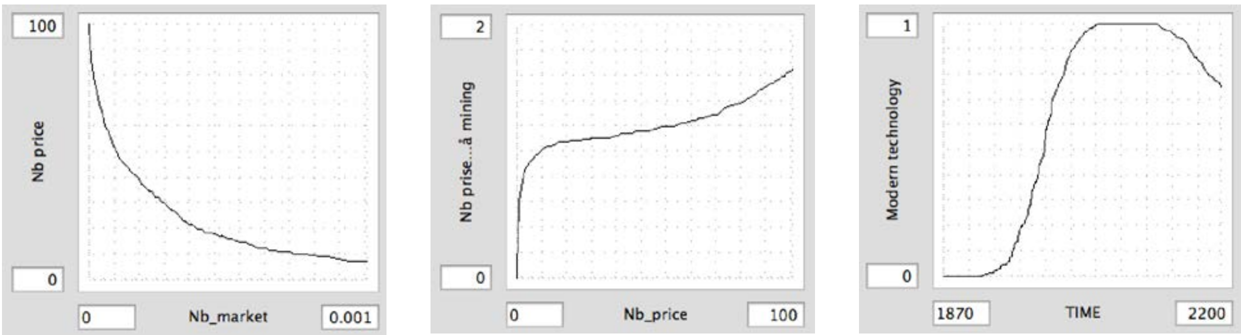
G: threshold price for low grade ore tantalum mining

H: threshold price for ultralow grade ore tantalum mining

I: Price curve for tantalum

Source: Own Figure, Lund University/University of Iceland

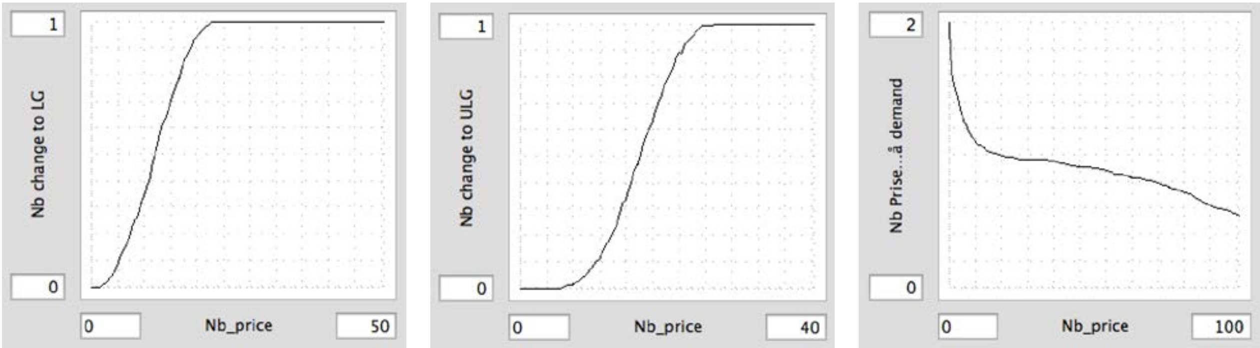
Figure 4.1.8.6. Feedback response curves employed in the niobium sub-module.



A: Price curve for niobium.

B: Price effect on niobium mining rate

C: Niobium extraction technology efficiency



D: Threshold price for low grade ore niobium mining

E: Threshold price for ultralow grade ore niobium mining

F: price effect on global average niobium demand

Source: Own Figure, Lund University/University of Iceland

Input data and parameters

Before the model can be run, it needs to have its input parameters set properly, and the resource database set up. The available extractable resources are stratified as shown in Table 4.1.8.8. The feedback parameterization has been shown earlier in Figure 4.1.8.5 and Figure 4.1.8.6. The extractable amounts were set at the beginning of the simulation in 1900, stratified with respect to ore metal content. The average ore grade for the high grade was set at 5%, for low grade at 1%, for ultralow grade at 0.2% and for trace grade at 0.02% (Sverdrup et al., 2014a,b, c, 2015a,b).

Table 4.1.8.8. Input data to the model for extractable amounts of niobium and tantalum

Ore grade	Niobium, Million ton			Kg per ton content	% weight metal
	Known	Hidden	Sum		
High	0.1	10	10.1	50-10	3.2
Low	0.0	16	16.1	10-2	0.3
Ultralow	0.0	24	24.1	2-0.2	0.01
Sums	0.1	50	50.1		
Ore grade	**Tantalum, Million ton**			**Kg per ton content**	**% weight metal**
	Known	Hidden	Sum		
Ultralow	0.005	0.075	0.080	0.5	0.05
Trace	0.000	0.150	0.150	0.1	0.01
Rare	0.000	0.250	0.250	0.05	0.005
Tin slag	0.000	0.022	0.022	5	0.5
Sum	0.005	0.497	0.502		

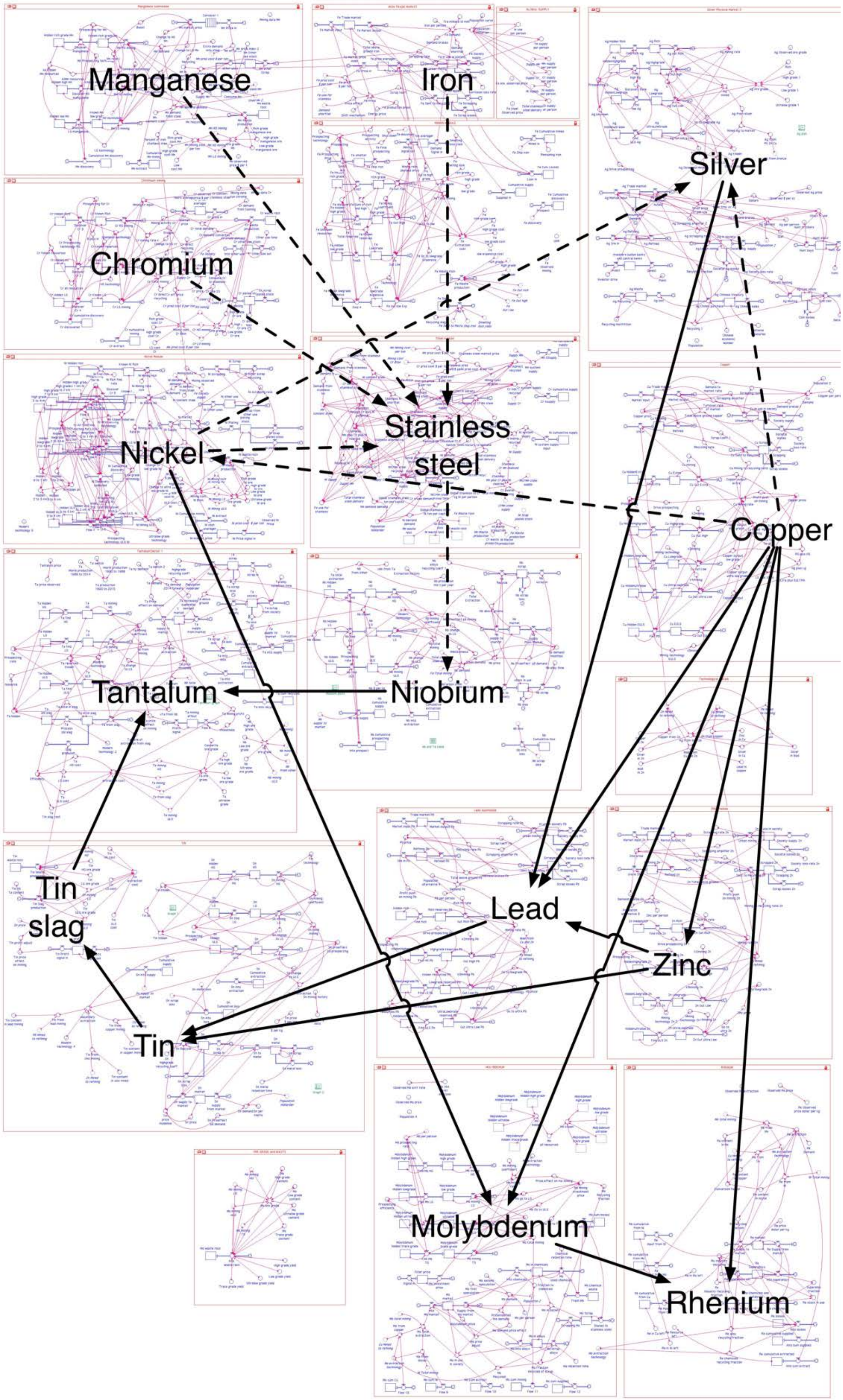
A number of literature sources were consulted (Abischeva and Zagorodnyaya 2013. Berzina et al., 2005, Crowson 2011a,b Cunningham et al., 2008, Eckstrand and Hulbert 2007, Figueiredo et al., 2013, Grabezev 2012, Hess 1924, Fleischer 1959, International Business Publications 2013, 2015, Keseler and Wilkingson 2013, Kwatra et al., 2012, Ludington and Plumlee 2009, Nassar et al., 2012, Mudd and Jowitt 2014, Mudd 2009, Nickless et al., 2014, Nuss et al., 2014, Polyyak 2011, Robinson and Menzie 2012, Sverdrup et al. 2013, Tilton and Lagos 2007, UNEP 2011a,b,c, 2013a,b,c, USGS 2009, 2012, 2013, 2014, 2015). Reserves are the known and proven reserves, ignoring that there probably is far more material in the ground, further in or down, as well as in areas not investigated. These resources, which we call "hidden" in this study, represent what is called hidden resources. Hidden resources are defined as that amount that can in due time be found and extracted, even if some or much of it would require a higher price and more effort. Some of it may require technological innovations and a significantly higher price to cover the extraction costs. The numbers have been assembled from data from the USGS (2015) and other literature sources listed. Table 4.1.8.9 shows the input data to the model for extractable amounts of niobium and tantalum. Amounts in million ton contained. The tantalum resources were distributed to ultralow, trace and rare ore grades. There are no high grade or low grade ore deposits known for tantalum. The highest ore grades are consumed first in the model, the switch to a lower ore grade takes place when the price has passed a predefined price level. Reserves are the known and proven reserves. These we call "Known" in this study. The yet undiscovered extractable amounts (Resources), which we call "hidden" here, represent what is called resources. Resources are defined as that amount that can in due time be found and extracted, even if some or much of it would require a higher price and more effort. Through prospecting "hidden" is converted to "known" when it is found. But technically it will sooner or later be available, provided we will take to effort to extract it. Some niobium and tantalum ores are difficult to process, and metal yields are many times as low as 50-60%.

Table 4.1.8.9. Summary of cumulative extracted, supplied, recycled and lost material to 2400 AD for, niobium and tantalum and the Factor X achieved under business-as-usual. All amounts in ton

Metal	Metric ton metal				Factor X
	Extracted	Supplied	Recycled	Lost	
Niobium	12,500,000	20,000,000	-	-	1.6
Tantalum	120,000	80,000	40,000	100,000	1.2

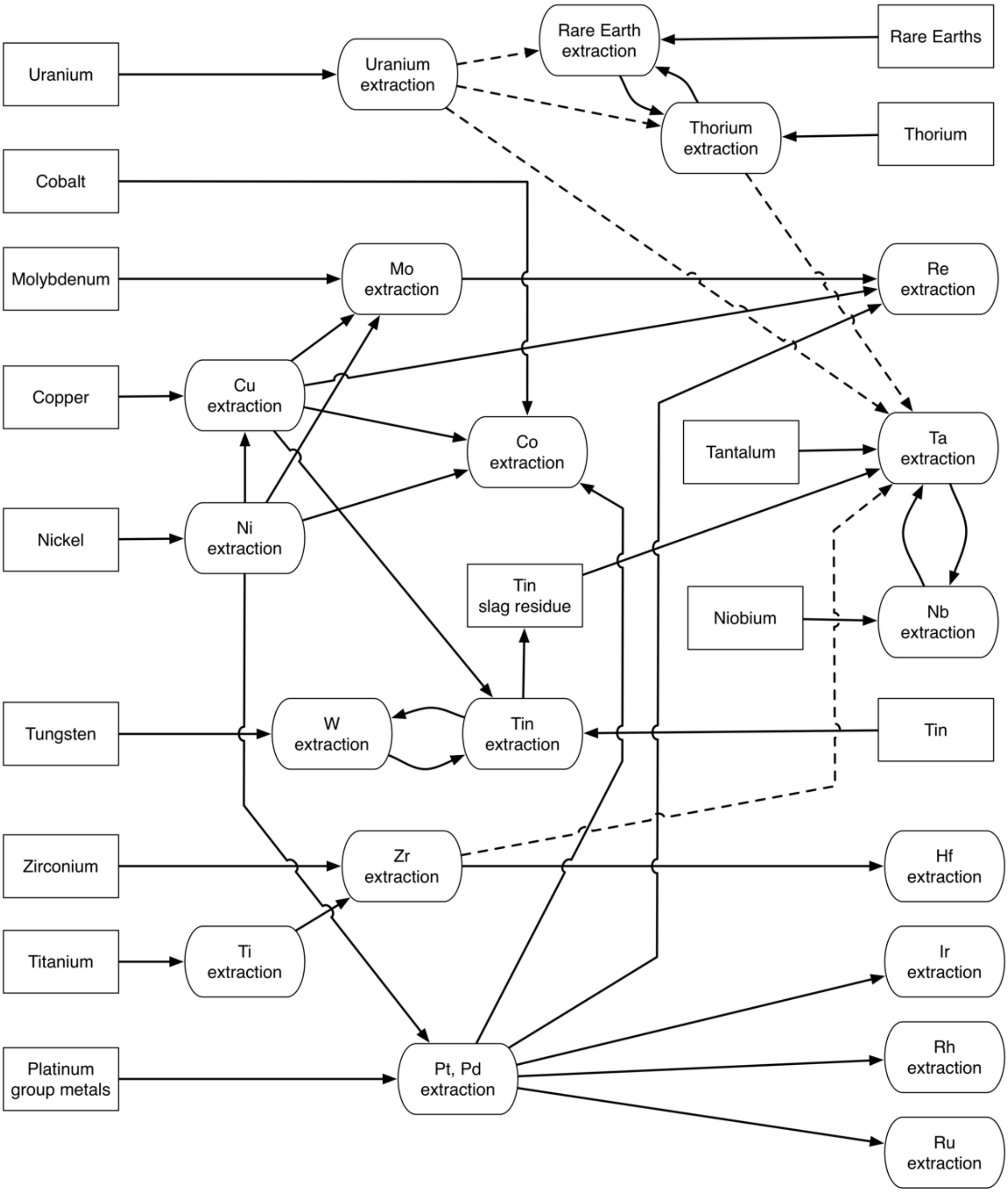
Figure 4.1.8.7 and Figure 4.1.8.8 show the Stella diagram and the flow chart diagram used in the modelling of niobium and tantalum in the WORLD model.

Figure 4.1.8.7. The STELLA diagram for the part of the WORLD6 model used in this study.



Source: Own Figure, Lund University/University of Iceland

Figure 4.1.8.8. Overview of the extraction of some the metals (Co, Mo, Ni, W, Zr, Ti, Hf, PGM, Re) that gets to be used in superalloys as well as some of the carrier metals (Cu, U, Th, REE, Sn) and the couplings between them. The flow chart was a step in developing the final model structure for these metals. Dotted lines represent potential but not yet used production possibilities



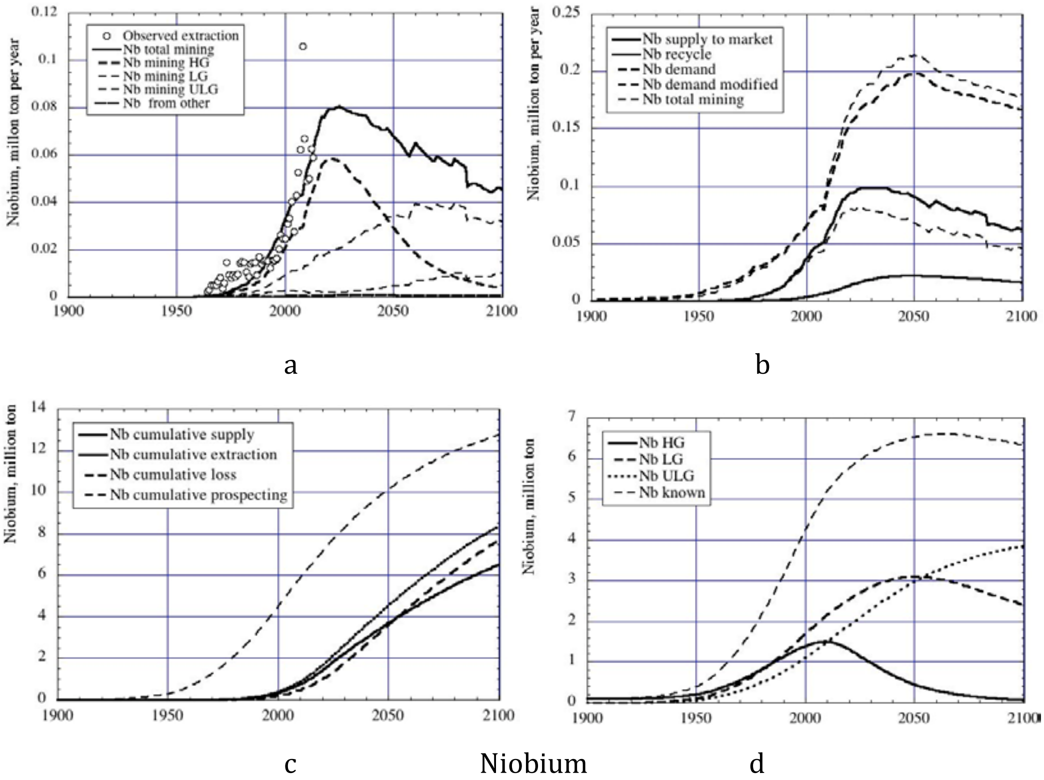
Source: Own Figure, Lund University/University of Iceland

4.1.8.10 Results

Model simulation results for niobium

Figure 4.1.8.9a shows the mining rates for niobium according to the model as compared to the observed rate. The curves also show the contribution from high grade ore, low grade ore and ultralow grade ore. The correlation between the observed and the simulated mining rate is r2=0.71. Figure 4.1.8.9b shows supply rates of niobium, comparing mining, supply, demand and demand after price adjustment. Figure 4.1.8.9c shows the cumulative supply rates of niobium, primary extraction, recycling and losses. Figure 4.1.8.9d shows the stocks remaining of known high grade, low grade and ultralow grade.

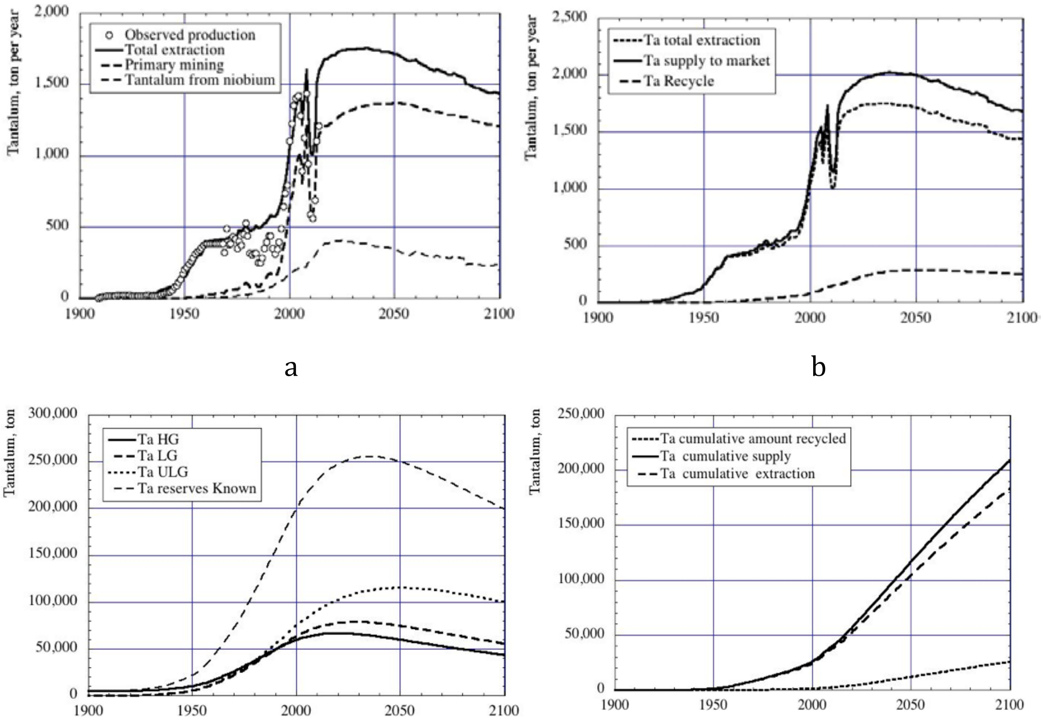
Figure 4.1.8.9. (a) shows mining rates for niobium from high grade ore (HG), low grade ore (LG) and ultralow grade ore (ULG) as compared to the observed rate, simulations from 1900-2100. (b) shows a comparison between the primary niobium extraction, the supply, the flow from recycling, the niobium demand and demand after price adjustment. (c) shows the cumulative supply rates of niobium, comparing mining, supply, demand. (d) shows the stocks remaining of known high grade, low grade and ultralow grade.



Source: Own Figure, Lund University/University of Iceland

Figure 4.1.8.10a shows the mining rates for tantalum according to the model, as compared to the observed rate. The correlation between the observed and the simulated is r2=0.93. Figure 4.1.8.10b shows the supply rates of tantalum, comparing mining, supply, demand and demand after price adjustment. Figure 4.1.8.10c shows the stocks remaining of known high grade, low grade, ultralow grade according to the model. Figure 4.1.8.10d shows the cumulative amounts recycled, extracted and supplied to society. Much tantalum is lost irreversibly, and the stock-in-society will always remain low.

Figure 4.1.8.10. (a) mining rates for tantalum according to the model, as compared to the observed rate, from 1900-2100. (b) shows supply, mining and recycling (c) shows the stocks remaining of known high grade, low grade and ultralow grade according to the model. (d) shows the cumulative amount lost, extracted and supplied to the market.

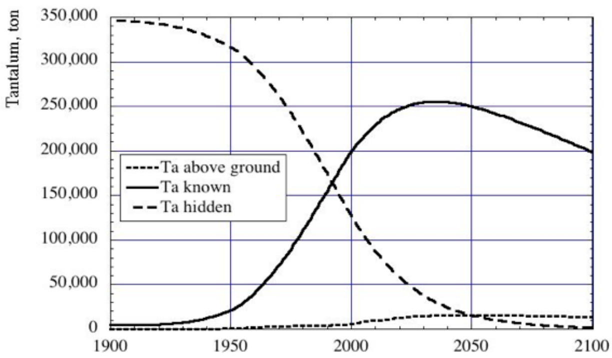


Tantalum

Source: Own Figure, Lund University/University of Iceland

Figure 4.1.8.11 shows a comparison of known reserves and stock-in-use for Tantalum.

Figure 4.1.8.11. Comparison of tantalum stock in society, and the known/hidden tantalum reserves according to the model simulations from 1900-2100.



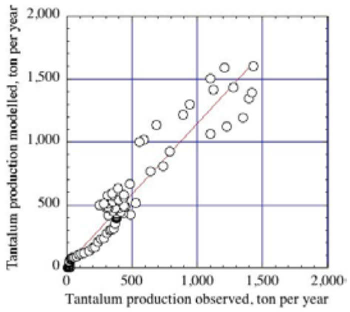
Tantalum

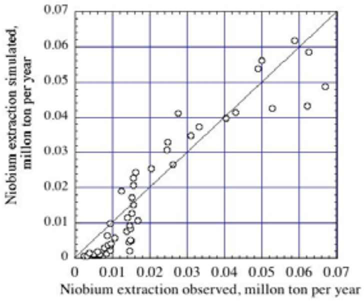
Source: Own Figure, Lund University/University of Iceland

4.1.8.11 Testing the model outputs on observed data

The performance of the model was tested on observed values for extraction, market price and ore grade. Data on extraction for 1900-2012 is available from the United States Geological Survey website (USGS 2015). The same site has metal prices for the period 1964-2015. Figure 4.1.8.12 shows the model simulation extraction rates for tantalum and niobium as compared to observations. The correlation is good, r2=0.93 for tantalum and r2=0.74 for niobium for the mining rate.

Figure 4.1.8.12. Comparison of simulated extraction according to the model versus the observed for tantalum (a) and niobium (b). The correlation is good, r2=0.93 for tantalum and r2=0.74 for niobium.



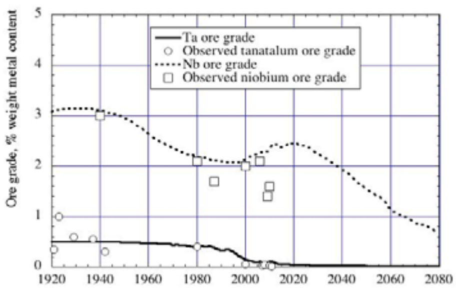


a: Niobium b: Tantalum

Source: Own Figure, Lund University/University of Iceland

No good time-series data is available for verifying for ore grade, but we have been able to pick some values from the web and literature. Figure 4.1.8.13 shows the predicted ore grade for niobium and tantalum mining as compared to some observations. The speculation that occasionally took place in the market was not included in the model. Present ore grades for niobium seems to be in the range 2.5%. The typical tantalum ore grade is in the range 0.1-0.03% at the moment. The slag from cassiterite tin ore smelting contains up to 10-20% Ta2O5, corresponding to 2.5-3% weight tantalum in the original ore. The simulations were compared to the observed ore grades for niobium (triangles, r2=0.59) and tantalum (circles, r2=0.69). The dataset is small and may not be representative for all extraction sites.

Figure 4.1.8.13. The predicted ore grade for niobium and tantalum mining, according to the model. The circles represent single data points on observed ore grades for niobium (triangles, r2=0.59) and tantalum (circles, r2=0.69).

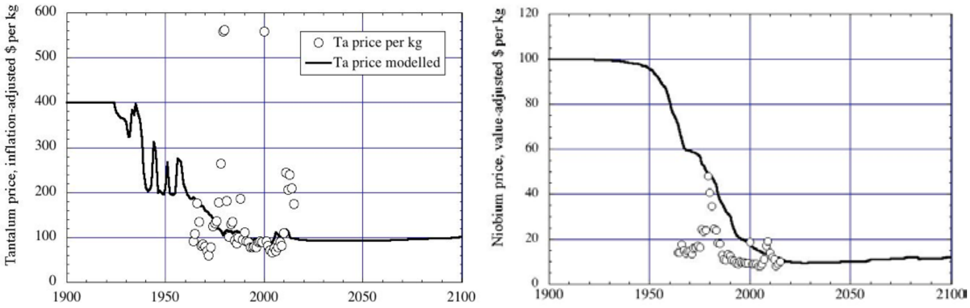


Tantalum and Niobium

Source: Own Figure, Lund University/University of Iceland

Figure 4.1.8.14a shows the market price, observed price and the market amount for tantalum. The general market price level is reproduced, but the short term variations are not captured. Before 1960, there was no real functioning tantalum market, and an observation to simulated comparison would not be valid for evaluation. Figure 4.1.8.14b shows the niobium market price, observed price and the market amount for niobium. There is a time-shift mismatch in the prediction, but the general pattern is reproduced. As in the case of tantalum, prior to 1960, there was no real functioning niobium market, and an observation to simulated comparison would not be valid for evaluation.

Figure 4.1.8.14. shows the market price in inflation-adjusted $ per kg, observed price and the market amount for tantalum and niobium.



a Tantalum b Niobium

Source: Own Figure, Lund University/University of Iceland

4.1.8.12 Discussions

Recycling and lack thereof

The recycling degree of the metals included in super-alloys is far too low for these alloys for a number of reasons. Far too low from a sustainability point of view, where a finite resource is sought to be preserved. Exhaustion of the primary resource by 2200 and total lack of it after 2300 is by no definition of sustainability acceptable. Many of the super-alloys have such good corrosion resistance, as well as tantalum and rhenium alone, that they can be considered as precious metals. Generally, they are also very temperature resistant, creating additional challenges for recycling by chemical separation. Recently, better chemical methods have been developed for recycling superalloys. Another pathway is by sorting alloys by contents during recycling and reconstitute alloys by corrective re-alloying and hot fusion. For tantalum and niobium, most chemical uses as salts leads to irreversible losses. Much tantalum is also diluted into recycled copper from electronics. Niobium is lost by dilution into recycled iron and stainless steel scrap.

Risks of scarcity

The supply of molybdenum, rhenium, niobium, tantalum have a maximum supply in the period 2035-2045, whereas the maximum for platinum (2060) and cobalt (2100) occur a bit later. The diagrams suggest that for a period, molybdenum may serve as a substitute for niobium, but not the other way around because niobium decrease before molybdenum and that the amount niobium supplied is significantly less than molybdenum. However, the tantalum production is 10 times larger than the rhenium production and may serve as a substitute when possible. The substitutability of tantalum and niobium as well as some of the other key metals used in superalloys (Hf, Re, Mo, Ir) is limited and only with loss of technical functionality (Weisz et a., 2015, Graedel et al., 2015a,b). Niobium and molybdenum face soft scarcity because of lower ore grades and higher extraction prices. Loosing rhenium and tantalum availability implies a step back towards lower performance for superalloys. Tantalum has a global rarity on par with gold and the platinum group metals. Thus, there will never be much of it, and it will most probably be more expensive than at present in the future. Even nickel may become limiting as the alloy base in the longer perspective. It is evident that substantial, low grade resources for tantalum and rhenium are not being extracted today. These ore are dependent on many metals being simultaneously as well as the requirement that a sophisticated extraction and refining infrastructure and knowledge is in place. Only a few countries have this at the moment.

General discussions

Table 4.1.8.11 shows a summary of extracted, supplied, recycled and lost material to 2400 AD for niobium and tantalum, and the Factor X achieved under business as usual. Factor X is the cumulative amount of metal supplied to the market, divided by the cumulative amount extracted from the ground (Equation 4.1.8.12). Many alloys that today are diluted into unsorted scrap iron. It is apparent that the recycling of these metals is far too low even if we include internal industry recycling. It should be made policy to preserve these metals far better than today. There are sufficiently many empirical cases available to prove beyond any reasonable doubt that scarcity issues are not efficiently solved by the market mechanisms alone. We must conclude that the “market mechanisms alone”-approach as a solution to avoid scarcity is a scientifically proven dead end (Fukuyama 2006, Acemoglu and Robinson 2013, Heinberg 2001, Bardi 2013, Sverdrup et al., 2015a,b,c). Still insisting on such an approach, is inviting failure and irreparable scarcity situations (Ostrom et al., 1994, Acemoglu and Robinson 2013). Most countries do not have any strategic governmental policy on resources in general (metals, materials, fossil fuels). There are no internationally unified policies for recycling. In the international arena, there are several bodies of the UN working on the issue (UNEP) and the International Resource Panel. Other important organizations are either happily ignore such messages, or behave in a way that suggests straight denial (OECD, IEA, OPEC). However, in the political arena, there seems to be little real interest in natural resource sustainability issues, and the agenda is often set by the least interested and the ones mostly against. Progress has been very slow.

4.1.9 SPECIALITY METALS sub-module – WOLFRAM

4.1.9.1 Introduction

Wolfram is an exotic metal used in modern times with the coming of high technology and sophisticated metallurgy. It also occurs in pegmatites, mostly as a sulphide. Wolfram is also known as tungsten in most English-speaking countries, but outside the English speaking, the name is wolfram, and that is the recognized official name for the element by the International Union of Pure and Applied Chemistry (IUPAC). Thus, we will use wolfram instead of tungsten from now.

In 1759 the Swedish chemist Axel Fredrik Crohnstedt discovered an unusual mineral he called "tungsten" (That is a heavy stone in Swedish) and was convinced it contained a new element. But he had trouble isolating it. The element was recognized in 1781 by the Swedish scientist Carl Wilhelm Scheele in Uppsala (Scheele 1791). However, Spanish chemists, who rediscovered the element in 1783, the brothers Elhuyar de Suvisa, got the international recognition for it.

Wolfram was isolated as a metal in 1783 by Scheele, the same year the Spanish chemists also did that. Wolfram is the metal with the highest melting point of all metals (3,422oC), and it makes very hard alloys and carbides. The dominating use is hard cutting tools (61%), next comes metallic special alloys (20%), wire and plate (11.4% and some for chemicals (7.3%). Wolfram is also important for military armament and heavy armour-piercing ordinance.

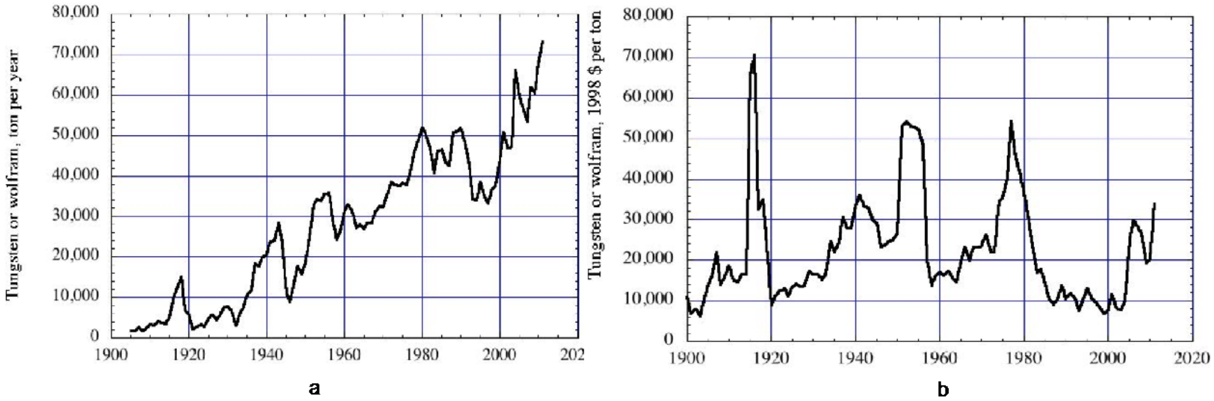
4.1.9.2 Objectives and scope

The goal was to develop a model for the production and cycling of wolfram in society and to include this as a module in the WORLD model. The two significant uses in the WORLD model will be for hard cutting tool bits and for superalloys. The objective is to use the validated systems dynamics model to explore the system and explore what it would take to make the global Wolfram supply system more sustainable. Not much emphasis is put on the consumer side in this stage of development of WORLD6, once the supply simulations have been developed that will follow in later studies.

4.1.9.3 Resource estimates

Wolfram has a limited production and is produced in limited amounts. The wolfram market price is high enough so that it is not used in any large amounts. Figure 4.1.9.1 shows the world production of wolfram and the market price. China is the dominating producer at the moment (83% of the global extraction). The market price shows very distinct pattern of volatile price changes. The main wolfram ores are scheelite ($CaWO_4$) with about 30% of the supply and wolframite ($FeWO_4$) with about 70% of the total supply from the resource. Secondary extraction is marginal for wolfram (6%) but increasing, but recycling is very significant (35%).

Figure 4.1.9.1. The world production of wolfram, and the market price. The market price shows very distinct cyclic pattern of price changes.



Source: Own Figure data from USGS 2015, Lund University/University of Iceland

New estimates of the extractable resources were made by the authors, independently of earlier attempts. Earlier data was also considered in the synthesis towards the final result. To put this together, a number of literature sources were consulted. We used: Andrews 1955, Chiarro et al., 2014, Crowson 2011, Keseler and Wilkingson 2013, Kwatra et al., 2012, Ludington and Plumlee 2009, Lifton 2006, Heinberg 2001, Hughes 1990, International Business Publications 2015, Lehmann et al., 2014, Kravchenko and Pokrovsky 1995, lele and Bhardwaj 2014, Nassar et al., 2012, Mudd and Jowitt 2014, Mudd 2009, Nickless et al., 2014, Nuss et al., 2014, Polyyak 2011, Sverdrup et al. 2013, Tilton and Lagos 2007, Schubert and Lassner 2010, Seddon 2013, Shedd 2003, Sverdrup and Ragnarsdottir 2013, 2014, UNEP 2011a,b,c, 2013a,b,c, USGS 2009, 2010, 2011, 2012, 2013, 2014, 2015, 2016, Vulcan 2013, Shiyu 1991, Dalnedra Vostok 2016, Visser 2002, Walser 2002, Woodcock and Hamilton 1993, USGS Minerals Yearbook 2016, Leal-Ayala et al., 2015. There are a number of sources available for reserves, most have been summarized by the United States Geological Survey in their yearbooks (USGS 2015, it is freely available on the internet). Typically, wolfram ore contains traces of As, Bi, Sn, Ti, Hf, Sc, REE, V, Pb, Zn, Zr, Ta, Nb, Th and U (Lehmann et al. 2014). Wolfram is also present in polymetallic ores, which contain more wolfram (40 million ton) than straight wolfram ores (20 million ton). In the past (before 2015) about 3 million ton of wolfram had been extracted.

USGS estimate for 2014 the wolfram reserves to be about 9 million ton, up from 3.3 million to a few years earlier. This is, however only a smaller part of the story. Reserves are the known and proven reserves, omitting that there probably is far more material in the ground further in or down. These, which we can “hidden” here represent what is added to reserves and called resources. Resources are defined as that amount that can in due time be found and extracted, even if some or much of it would require a higher price and more effort. But technically it will sooner or later be available, provided we will take to effort to extract it. The types of deposits being mined at present are shown in Table 4.1.9.1.

Table 4.1.9.1. Wolfram production, reserves and resources in several countries. Numbers in italics are our own estimates from corporate information where published estimates are failing.

Country	Wolfram extraction, ton per year	Wolfram reserves, million ton	Wolfram resources, million ton
Ruwanda	1.000	0.650	0.300
Congo	800	0.500	0.800
China	71,000	1.900	5.000
Brazil	-	0.055	0.800

Australia	600	0.200	0.400
Canada	3,000	0.290	1.200
USA	-	0.100	1.000
Kazakstan	-	0.300	3.500
Vietnam	5,000	0.100	1.800
Bolivia	1,300	0.053	0.120
Portugal	630	0.004	0.050
Spain	730	0.005	0.015
Austria	870	0.010	0.058
Britain	600	0.050	0.150
Russia	2,800	0.250	3.800
Others	250	1.200	2.100
Undiscovered	-	-	6.835
Sum	88,500	6.212	21.165

Table 4.1.9.2, Table 4.1.9.3 and Table 4.1.9.4 show an overview of remaining wolfram reserves and recoverable resources in 2012 estimates. All amounts are in metric ton. New data on resources were also taken from industrial unpublished sources and from the British Geological Survey 2011. Wolfram is supplied from many countries that do not at present mine any wolfram resources of their own, but which may very well do that if the wolfram price would increase somewhat from the present level. Many of these countries have smelter capacity and purchase their ore in the markets.

Table 4.1.9.2. Overview of remaining reserve in 2012 estimates. All amounts in metric ton. New data on resources from industrial unpublished sources and the British Geological Survey 2011.

Metal	Production 2015 ton per year	USGS reserve estimate	USGS resource estimate	URR at present, our estimate	Dug up before 2010	Typical workable ore grades	Grade	Lower cut-off in ore
		Million ton metal contained				% weight content		
Wolfram	88,000	9	-	16-30	2.7	0.1-1.5	0.4	0.10
Niobium	63,000	4.3	5.4	16-25	1.5	0.1-4	0.4	0.01
Tantalum	1,400	0.150	0.26	0.32-0.4	0.11	0.01-0.4	0.03	0.01
Tin	330,000	5	50	60.3	23.0	0.02-1	0.05	0.01

Table 4.1.9.3. Overview of recoverable resources, including all occurrences known or anticipated. All amounts in metric ton of metal.

Metal	Mother metal URR, Million ton	W in other metal, fraction	W Flow t/yr	Production 2012 t/yr	Secondary W URR Million ton
Wolfram	20	-	-	88,000	-
Niobium	55	0.004	300	65,000	0.220
Tantalum	0.6	0.003	-	1,200	0.002

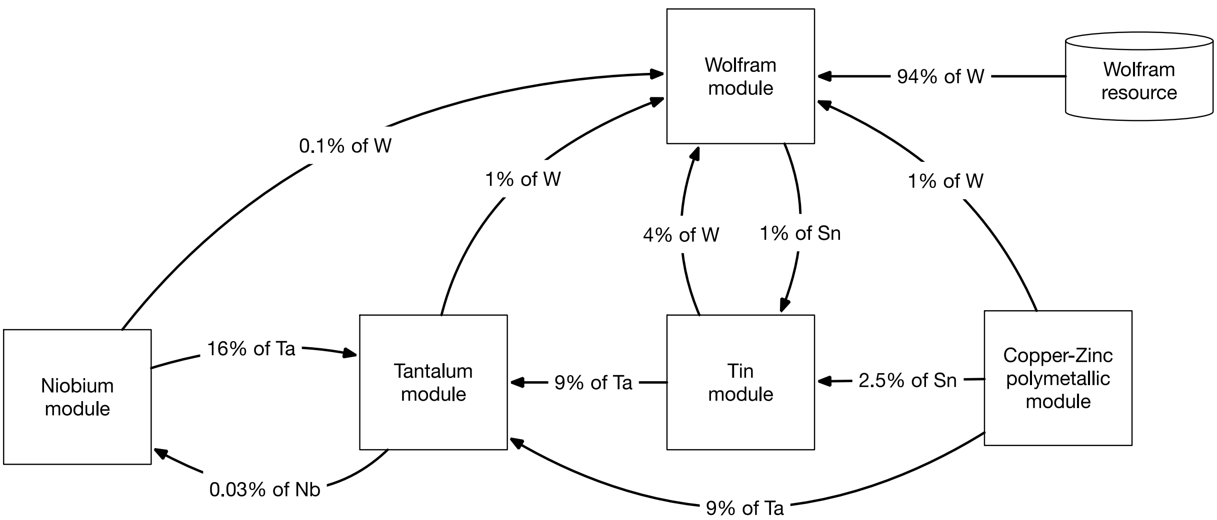
Tin	60	0.008	1800	300,000	0.480
Copper	3,600	0.00004	610	17,000,000	0.144
Zinc	2,700	-	240	12,000,000	-
Lead	3,000	-	-	4,000,000	-
Polymetallic	-	-	-	-	1.154
Sum	-	-	-	-	2.000

Table 4.1.9.4. Overview of recoverable resources, including all occurrences known or anticipated. All amounts in metric ton of metal.

Metal	Wolfram, ton	Tin, ton
Primary deposits	21,165,000	20.250,000
Secondary amount	2,000,000	40,100,000
Dug up 1900- 2015	3,000,000	13,000,000
Total URR in 1900	26,165,000	74,350,000
Dug up before 1900	100,000	10,000,000
Total URR ever	26,265,000	87,350,000

35% of all wolfram is recycled at present, and this may increase in the future. For wolfram, secondary production from tin is about 4%, with small contributions from copper, zinc, tantalum and niobium mining as shown in Figure 4.1.9.2. In nature, tin and wolfram tend to follow each other in deposits for geochemical reasons as well because they have a high specific density and are also in nature enriched by naturally occurring gravimetric processes (placer ores). Wolfram has important use as a component of superalloys, mainly for increasing the melting point and the alloy hardness.

Figure 4.1.9.2. A flow diagram for wolfram in the model. The model combines the sub-modules for tantalum, niobium, wolfram and copper-zinc-lead and tin.



Source: Own Figure, Lund University/University of Iceland

4.1.9.4 The model description

The flow pathways, the causal chains and feedbacks loops in the global tantalum-niobium-wolfram-tin system were mapped using system analysis, and the resulting coupled differential equations were transferred to computer codes for numerical solutions in the STELLA® environment. Figure 4.1.9.3 shows the causal loop diagram for the market module in the WORLD6 model. There it can be seen how the mining is profit driven, including both costs and income from metal sales. The model was developed to estimate supply, extractable amounts stocks, price and stocks, and flow in society in the time interval 1900-2400. Figure 4.1.9.2 and Figure 4.1.9.4 show basic flow charts for the model. The model has the following stocks used to define the coupled differential equations defined by mass balance:

1. Wolfram
    a. Mineable stocks;
        i. Known
            1. high grade,
            2. low grade
            3. ultralow grade ore
        ii. Hidden
            1. high grade,
            2. low grade
            3. ultralow grade ore
    b. In society, we distinguish 3 wolfram stocks;
        i. Trade market
        ii. Stock-in-use in society
        iii. Waste
    c. Dependent 5 stocks of wolfram in other metals:
        i. Stocks in copper
        ii. Stocks in zinc
        iii. Stocks in tantalum
        iv. Stocks in niobium

The model has 9 dependent stocks for secondary extraction, and 18 independent stocks, solved from the differential equations defined by the mass balance. Further dependent stocks in the model such used for counting up cumulative amounts of mined, supplied, recycled amounts, rock waste, losses, smelter slag and ore found by prospecting. All stocks have defined inputs and outputs, conforming to the equation

$$inputs + produced = \ accumulated + outputs \qquad (4.1.9.1)$$

This becomes the differential equations solved:

$$\frac{dm}{dt} = inputs + produced \ - \ outputs \qquad (4.1.9.2)$$

The inputs are from returns from recycling, and produced from primary and secondary extraction. Accumulated is what is accumulated in society either as stocks in use or as scrap. Outputs will be what is lost or recycled. Note that recycled is present on both sides, and thus increase the flow through society. To meet a certain supply to society, the larger the recycling, the smaller the primary extraction must be to meet the demand. Dependent stocks are cumulative amounts of: mined, amounts rock waste, losses, smelter slag and ore found by prospecting. The mining activity is profit-driven, the profit is affected by the mining cost and the market price (see Figure 4.1.9.5 for the exact model used). A lower ore grade implies that more rock must be moved to mine the metals. In the model, mining and recycling are profit driven as is shown in Figure 4.1.9.3. Sales of extracted metal together with the price, given the income, the extraction and prospecting give the costs, the profit being the difference. In the model, shift to a lower ore grade only occurs when the market price exceeds the extraction cost in such a way that the profit stays positive. When the profit is lower or goes negative, the mining rate is slowed down, leading to less material in the market, driving the price up. In practice, we have several sources of metal in the society; the high, low and ultralow ore grades and the stocks of metal in society that can be recycled. For wolfram, mining occurs from three different ore grades, and which ore that is mined depend on the net profit as the difference between price and extraction cost. The lower the ore grade, the higher the extraction cost will be. Metal is also obtained by secondary extraction from tin ores, as well as recycling of a fast stock of chemicals and catalysts and a slow stock of metallic alloys (Figure 4.1.9.2). The mining rate follows a rate equation depending on the mineable reserve, the profitability of the operation and the available mining technology:

$$r_{mining} = k * m_K^n * f(profit) * g(technology) \qquad (4.1.9.3)$$

where rmining is the rate of mining, k is the rate coefficient and mK is the mass of the ore body known and available for extraction, and n is the mining process order. We have chosen to set the mining order at n=1 consistent with alternative 1a. For open pit mining, the mining rate is proportional to one surface on the side of a rectangular shape. The rate coefficient is modified with ore extraction cost and ore grade. In the model, a delay in mining rate change is considered by using a forward rolling two year average of the market price. g(technology) is a technology factor accounting for the invention of technologies used in efficient mining, refining and extraction. f(profit) is a feedback function of market price, increasing mining at higher price and lowering it at lower metal prices.

$$r_{mining} = k * m_K^n * f(profit) * g(technology) \qquad (4.1.9.4)$$

Where the costs are defined as:

$$Profit = Income\ from\ sales - mining\ costs \qquad (4.1.9.5)$$

Where the costs are defined as:

$$Total\ costs = Mining\ costs + refining\ costs + prospecting\ costs + capital\ costs \qquad (4.1.9.6)$$

In this equation, mining costs, refining costs and prospecting costs all include both variable operations costs and capital costs for infrastructures and equipment, as well as a 10% profit margin. The income is defined as:

$$Income\ from\ sales\ =\ Amount\ sold * market\ price - sales\ costs \qquad (4.1.9.7)$$

There are many different definitions of recycling available (Graedel and Allenby 2003, UNEP 2011b, 2013a). The size of the extractable ore body is determined by the rate of extractions (rmining) and the rate of prospecting (rdiscovery):

$$\frac{dm_K}{dt} = r_{mining} + r_{discovery} \qquad (4.1.9.8)$$

The discovery is a function of how much prospecting we do and how much there is left to find. The amount hidden reserve (h) decrease with the rate of discovery. The rate of discovery is dependent on the amount metal hidden (h) and the prospecting coefficient kprospecting. The prospecting coefficient depend on the amount of effort spent and the technical method used for prospecting. We use the following equation for the rate of discovery:

$$r_{discovery} = \frac{dm_H}{dt} = -k_{prospecting} * m_H * g(technology) \qquad (4.1.9.9)$$

The rate is first order as prospecting is three-dimensional by drilling. The amount hidden in the resource is represented by mH is. The basic driving mechanism of basic mining comes from profits and availability of a mineable resource used in the model. The price is set relative to how much iron or steel there is available in the market. g(technology) is a function accounting for improved prospecting techniques coming with modern society and technological development. The extraction rates of wolfram (RW) in the model were calculated as follows:

$$RW = rW + rSn * XSn(W) \qquad (4.1.9.10)$$

where Xj(i) is the fraction of metal j in mother metal i, and rj is the mining rate of metal j. The important mother metal is only tin, and the primary resource for wolfram. The causal loop diagram shows us the mining operation is driven by operations profit as can be seen in Figure 4.1.9.3. This profit is driven by metal price and amount extracted, but balanced by the cost of operation. The mining rate is related to the state of the technology, and it has two alternatives: (1) either the technology and the financial capacity determines the mining rate and the ore body is always sufficient to supply anything attempted to be extracted, or (2) the mining rate is unrelated to the technical and financial capacity and limited by the access to the ore body.

There are many different definitions of recycling available (Graedel and Allenby 2003, UNEP 2011b, 2013a). For the purpose of clarity, the recycling fraction displayed in results were calculated as follows in this study:

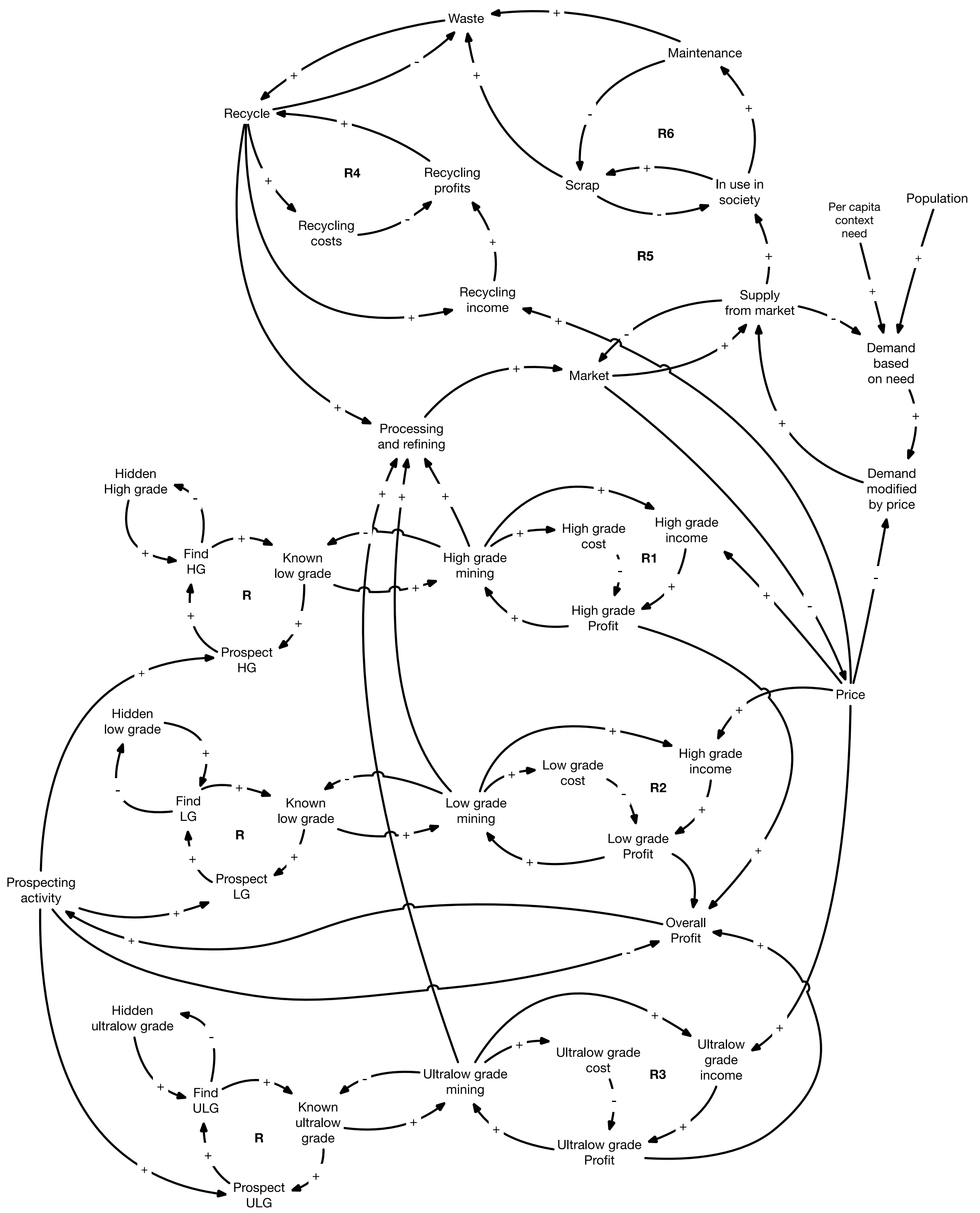
$$Recycling\ fraction = \frac{Flow\ of\ recycled\ metal}{Supply\ from\ primary\ extraction + Flow\ of\ recycled\ metal} \qquad (4.1.9.11)$$

The cost of the mining and extraction operation is mainly determined by two important factors beside cost of investments, the energy price and the ore grade. The price is determined by two factors, that it must stay above the production costs and by the amount in the market. A number of feedback response curves are employed in the model is shown in Figure 4.1.9.3. The wolfram demand was estimated using the demand from specialty steel, lighting technology, an empirical function and a general proportionality to population. The equation used is in million ton of wolfram per year:

$$Wdemand = 0.07 * Nbdemand + 0.01 * f(LT) + WOther + Population * 0.005 \qquad (4.1.9.12)$$

Where f(LT) is a technology development scaling function and wolframs use in glow filaments.

Figure 4.1.9.3. The causal loop diagram for the market module in the WORLD6 model. The causal loop diagram maps the causal relationships in the system as they will be pictured in the model. The R's in the diagram shows the reinforcing loops. These are the loops that keep the system running.



Source: Own Figure, Lund University/University of Iceland

Factor X is used to estimate the resource use efficiency of a material. It is defined as the cumulative amount supplied to the market divided by the cumulative amount extracted from the ground:

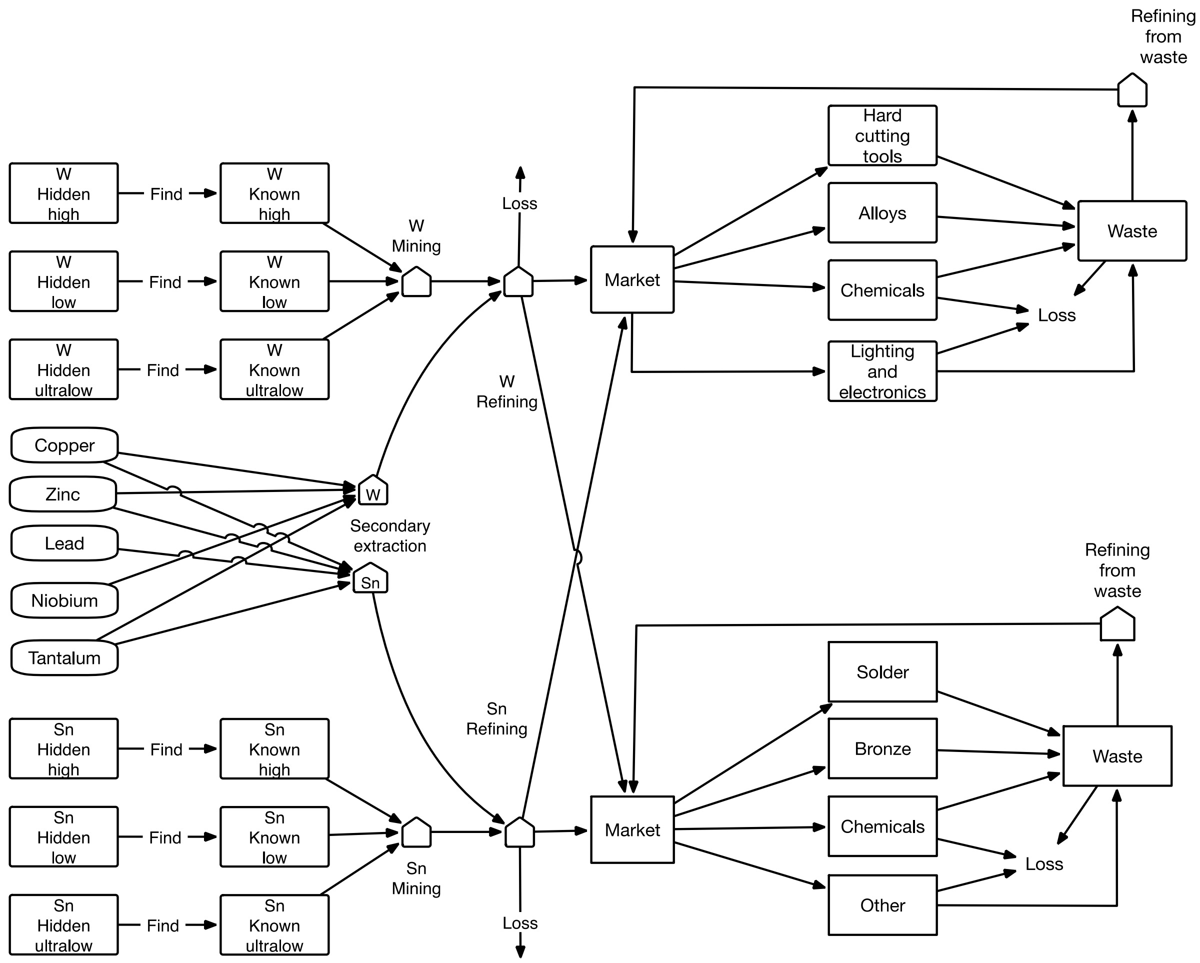
$$Factor\ X = \frac{\int_{1930}^{2400} Supplied\ metal}{\int_{1930}^{2400} Extracted\ metal} \qquad (4.1.9.13)$$

Where we have considered the time from 1930 to 2400. Production before 1930 was insignificant. Factor X vary over time, and thus an alternative measure would be the instant Factor x:

$$Factor\ x = \frac{Market\ supply\ rate}{Extraction\ rate} \qquad (4.1.9.14)$$

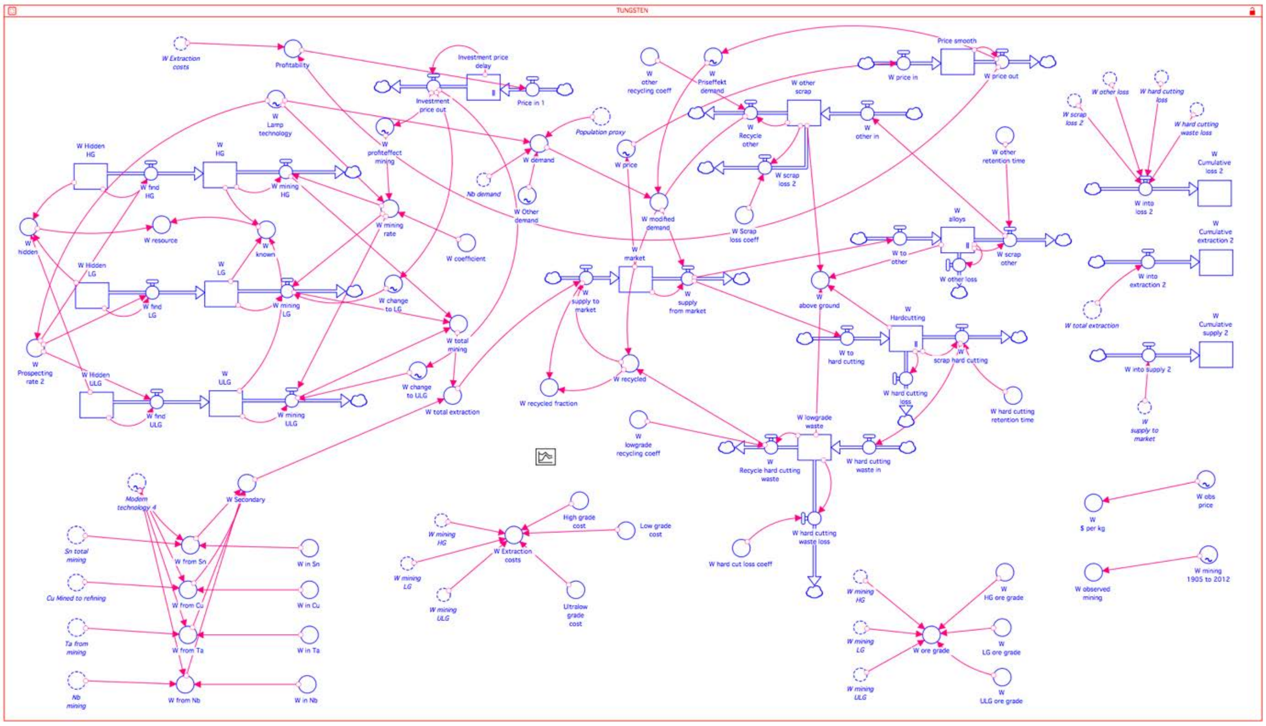
This measures the Factor x for the moment and change with time as the extraction rate and the recycling efficiency varies with time. Figure 4.1.9.4 and Figure 4.1.9.5 shows the flowchart and STELLA® diagram, respectively, for wolfram submodule inside the WORLD6 model.

Figure 4.1.9.4. Flowchart for the whole model showing how wolfram and tin production is linked. In this chart a simplified market module has been used.



Source: Own Figure, Lund University/University of Iceland

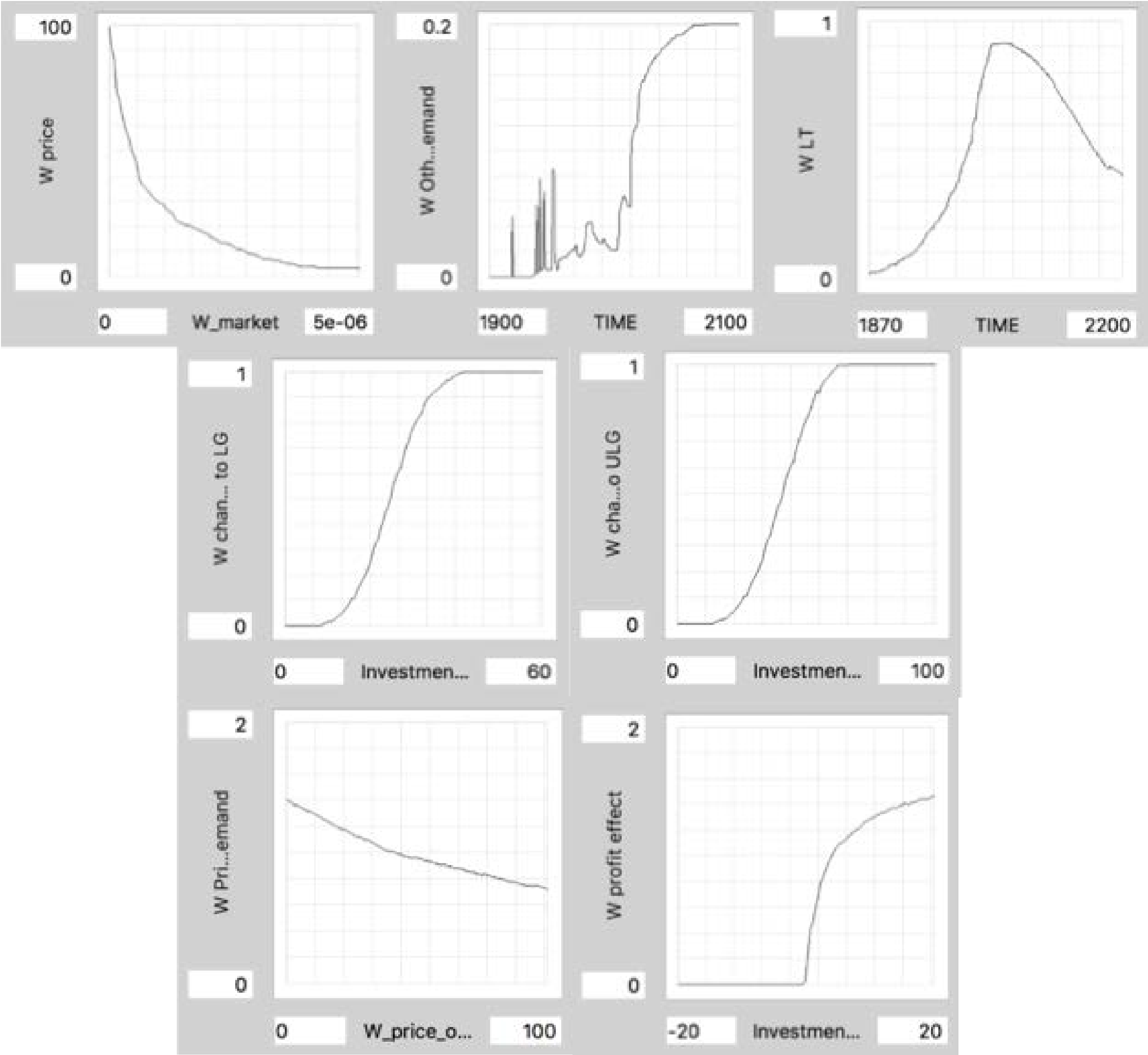
Figure 4.1.9.5. The W submodule in the WORLD6 model taken from the STELLA software. The module follows the logic of the causal loop diagram in Figure 4.1.9.3 and the flow chart in Figure 4.1.9.4.



Source: Own Figure, Lund University/University of Iceland

Figure 4.1.9.6 shows 7 diagrams. In the upper row from left to right the market amount to price relationship, the demand per capita, the intensity of wolfram use in lighting technology, in the middle row the shift price from high grade ore to low grade ore. In the bottom row the price modification of demand and the profit drive on mining. The causal loop diagram shows us the mining operation is driven by operations profit as can be seen in Figure 4.1.9.3. This profit is driven by metal price and amount extracted but balanced by the cost of operations. Related to the state of the technology has two alternatives: either the technology and the financial capacity determines the mining rate and the ore body is always sufficient to supply anything attempted to be extracted, or the mining rate is unrelated to the technical and financial capacity and limited by the access to the ore body. The cost of the mining and extraction operation is mainly determined by two important factors besides the cost of investments, the energy price and the ore grade. The price is determined by two factors, that it must stay above the production costs and by the amount in the market. A number of feedback response curves are employed in the model. These are shown in Figure 4.1.9.3. The price curves relate tradable amount in the market to the market price. These curves were developed from data on copper, silver, gold and platinum trading by the authors personal notes and experiences from trading floors in London and New York.

Figure 4.1.9.6. Parameterization of the wolfram module. The market mount to price relationship for wolfram (W).



Source: Own Figure, Lund University/University of Iceland

4.1.9.5 Input data and parameters

The feedbacks in the modules have been shown in Figure 4.1.9.3. The parameterization is consistent with what we have done earlier for metals. The market amount to price relationship turns out to be very similar among many metals. The same is the case for the price effect on demand and the profit push on mining. There are fundamental market settings (Sverdrup et al., 2012b, 2013, 2015b, 2014a,b, 2015, 2016, 2017, Sverdrup 2017). Before it's possible to use the model to simulate results, it needs to have its input parameters set properly, and the resource database set up. The available extractable resources are stratified as shown in Table 4.1.9.5. The feedback parameterization has been shown earlier in Figure 4.1.9.6. This uses a relationship between ore grade and extraction cost. The extractable amounts were set at the beginning of the simulation in 1900, stratified with respect to ore metal content. The average ore grade for the high grade was set at 5%, for low grade at 1%, for ultralow grade at 0.2% and for trace grade at 0.02%.

Table 4.1.9.5. Input data to the model. Reserves and resources in 1900 and estimated reserves and resources in 2011 for wolfram and tin.

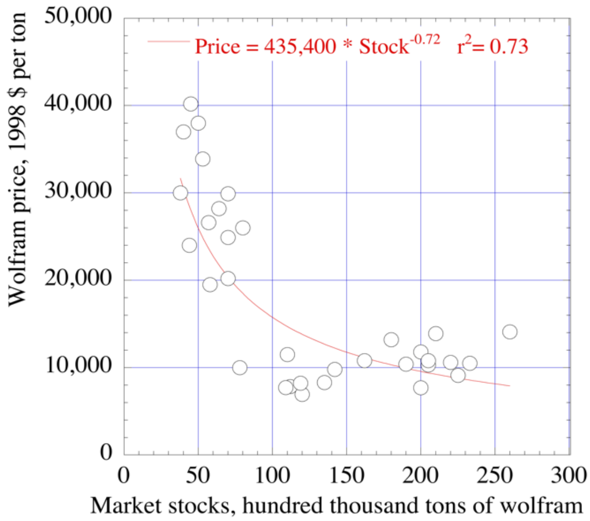
Ore grade	Wolfram 2011, Million ton				Wolfram 1900, Million ton			kg/ton content
	Known	Hidden	Dug up	Sum	Known	Hidden	Sum	
High	1.1	0.9	1	3.0	0.6	5.4	6	50-10
Low	3.0	5.0	1	9.0	1.0	8.0	9	10-2
Ultralow	2.0	9.0	0	11.0	0.0	12.0	12	2-0.2
Sum	6.1	14.9	2	23.0	1.6	25.4	27	
Ore grade	Tin 2100, Million ton				Tin 1900, Million ton			kg/ton content
	Known	Hidden	Dug up	Sum	Known	Hidden	Sum	
High	1	5	7	12	7	4	12	50-10
Low	15	15	3	33	3	30	33	10-2
Ultralow	10	20	0	30	0	30	30	2-0.2
Sum	26	45	10	75	10	64	75	

4.1.9.6 Results

Market price curve

Figure 4.1.9.7 shows the price curve determined from wolfram market data. The data originate from the authors private metal market archives. The price in 1998 ($) were plotted against market amounts including the US strategic stockpile plus the London Metal Exchange stocks and the tradeable stocks held at private corporations, traders and producers. The correlation between the market amounts and price is r2=0.72. The London Metal Market stocks are in the range 20,000 to 50,000 ton.

Figure 4.1.9.7. The market amounts related to the market price. Data from the authors private archives.



Source: Own Figure, Lund University/University of Iceland

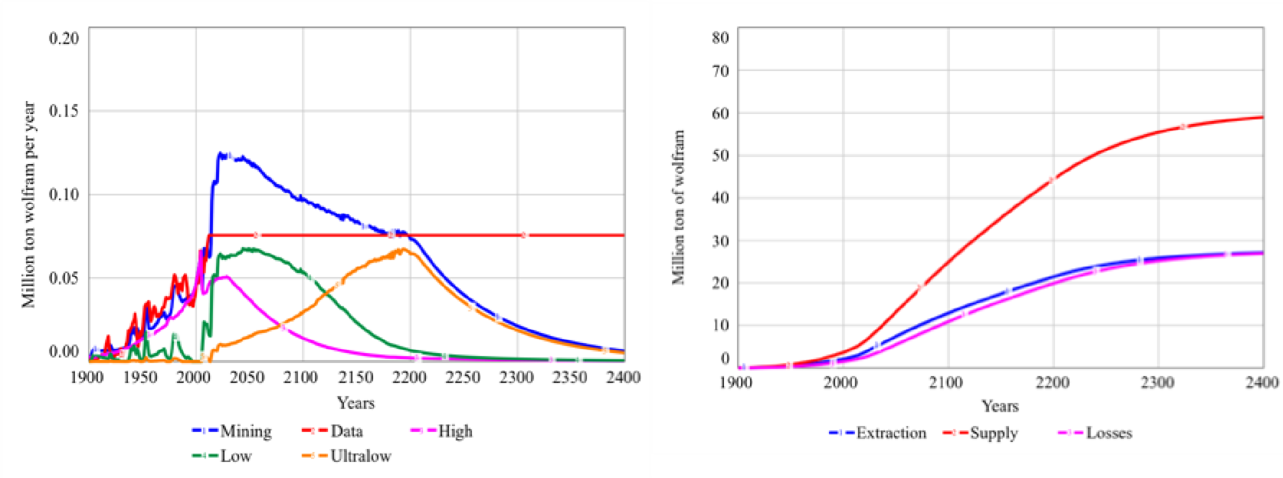
Resource estimates

The resource estimates developed in the Table 4.1.9.1 to Table 4.1.9.4 are important results of this work, enabling the creation of the input data used to run the model as presented in Table 4.1.9.5.

Wolfram dynamics

Wolfram production, supply, recycling and market price were modelled for the time period of 1900 to 2400. Figure 4.1.9.8a shows the mining rate for wolfram and how the different ore grades contribute. It can also be seen how well the historical mining rate is reproduced. Figure 4.1.9.8b shows cumulative amounts of wolfram extracted, supply and losses.

Figure 4.1.9.8. Model outputs from WORLD6 for wolfram and how the different ore grades contribute to the mining result. Cumulative amounts of wolfram extracted, losses, and supply.

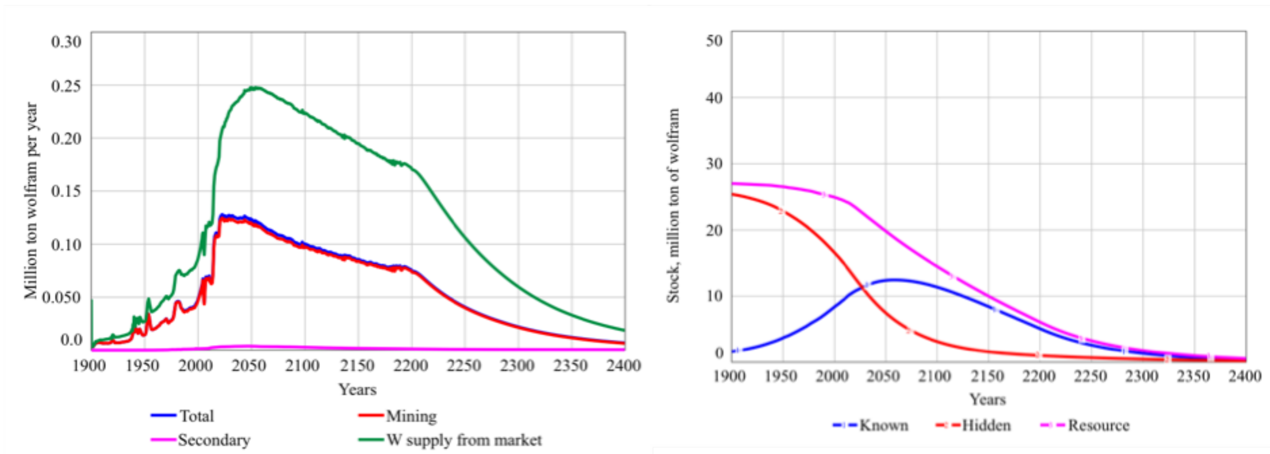


a b

Source: Own Figure, Lund University/University of Iceland

Figure 4.1.9.9a shows the total production, the secondary extraction and the mining of wolfram. Figure 4.1.9.9b shows the development of the known, hidden and total resources over time.

Figure 4.1.9.9. The total production, the secondary extraction and the mining of wolfram. The diagram to the right shows the development of the known, hidden and total resources over time.

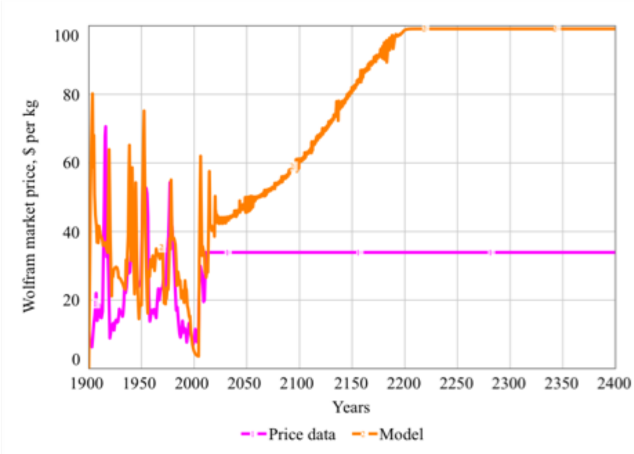


a b

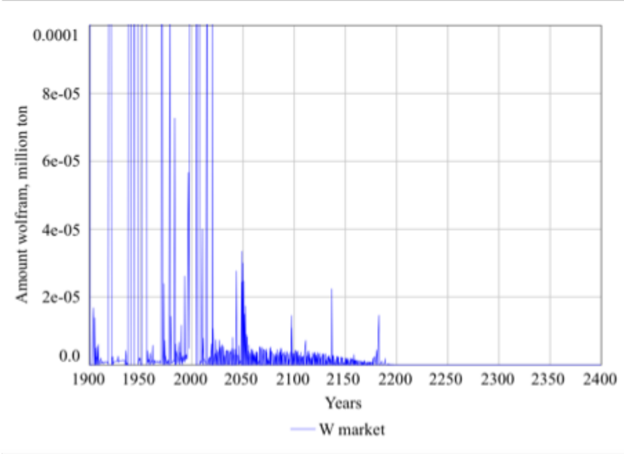
Source: Own Figure, Lund University/University of Iceland

Figure 4.1.9.10a shows a comparison of the observed market price and the modelled market price. Figure 4.1.9.10b shows the amounts of metal available in the trade market for transactions.

Figure 4.1.9.10. Comparison of the observed market price and the modelled market price. The diagram on the right shows the amounts of metal available in the trade market for transactions.



a

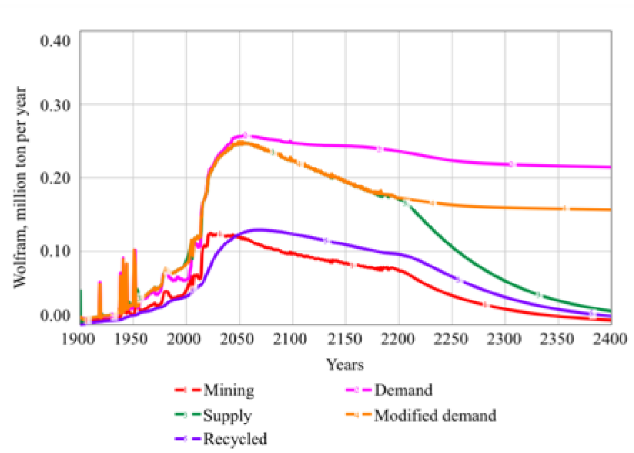


b

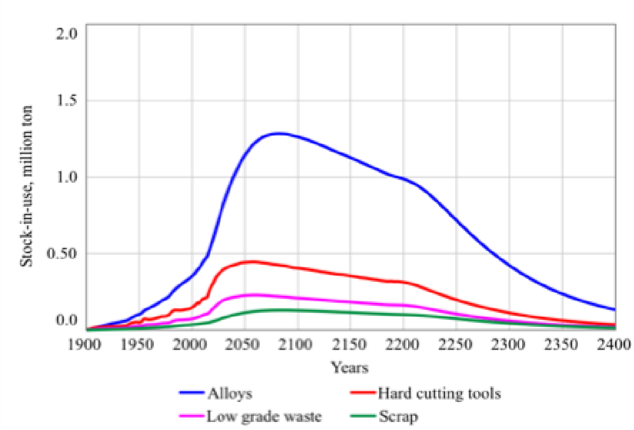
Source: Own Figure, Lund University/University of Iceland

Figure 4.1.9.11a shows the demand for wolfram, the demand after modification by the price, which closely overlaps on the actual supply, as compared to the mining rate and the recycling rate. Figure 4.1.9.11b shows the stocks in use in alloys, hard cutting tools, low grade waste and metallic scrap. Wolfram does not seem to run any serious scarcity based on the demand assumptions we are using.

Figure 4.1.9.11. Diagram (a) shows the demand for wolfram, the demand after modification by the price. Diagram (b) shows stock in use in alloys, hardcutting tools, low grade waste and metallic scrap.



a

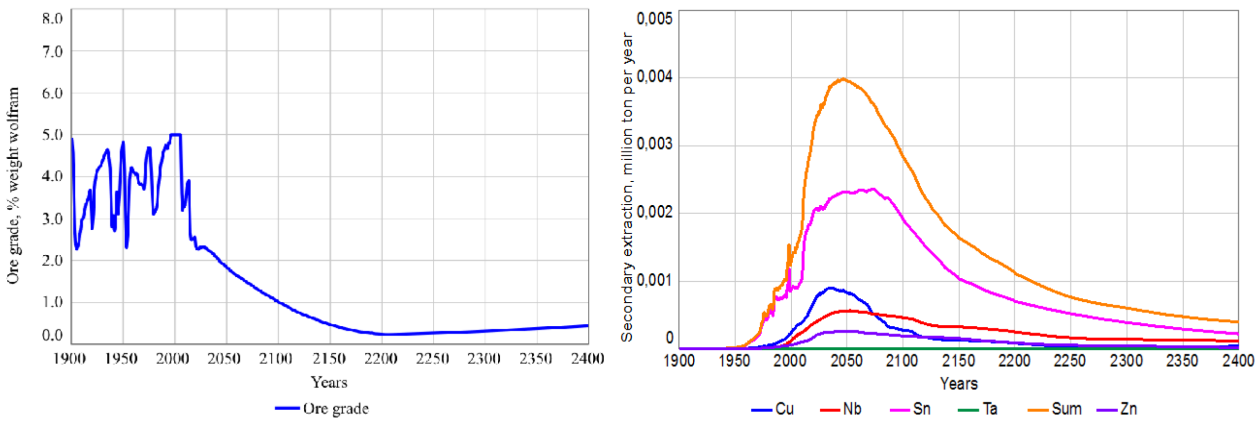


b

Source: Own Figure, Lund University/University of Iceland

Figure 4.1.9.12a shows the ore grade in terms of %weight Wolfram for the period of 1900-2400. Figure 4.1.9.12b shows the origin of the secondary extraction from copper (Cu), from zinc (Zn), from niobium (Nb), from tantalum (Ta) and from (Sn).

Figure 4.1.9.12. Diagram (a) shows the ore grade in terms of % weight Wolfram. Diagram (b) shows the origin of the secondary extraction from copper (Cu), from zinc (Zn), from niobium (Nb) and from tantalum (Ta).

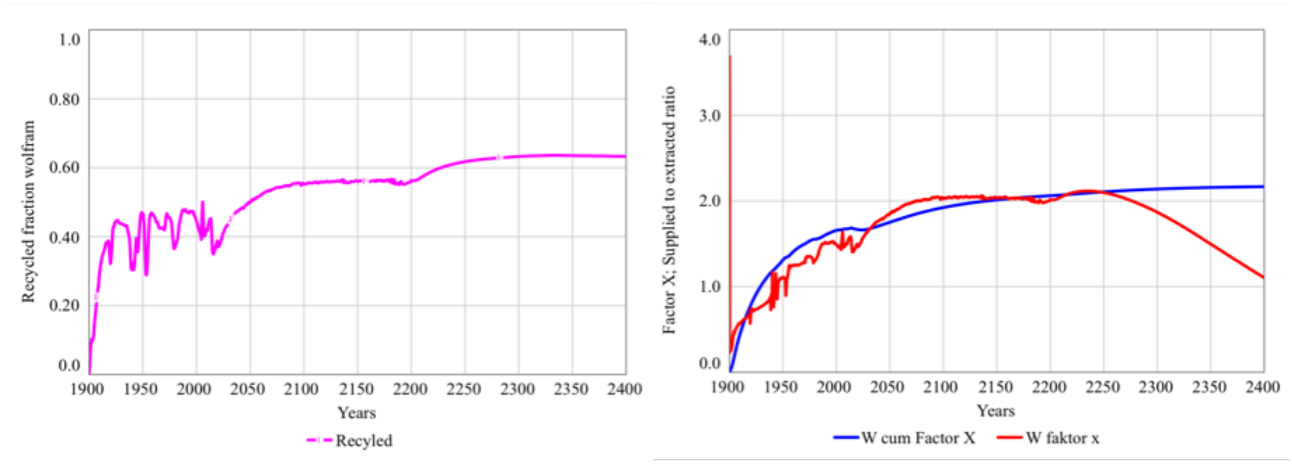


a b

Source: Own Figure, Lund University/University of Iceland

Figure 4.1.9.13a shows the fraction of total suppy being recycled material. Presently the estimated recycling is 35%, which is also reflected by the model output. Figure 4.1.9.13b shows the Factor X for wolfram. Wolfram recycling is at present 35-40% and is expected to rise to more than 50%, implying a factor X of more than 2.

Figure 4.1.9.13. The simulated wolfram recycling degree as a fraction supply and the Factor X for wolfram. Factor X is the ratio of supplied to net extracted.



a b

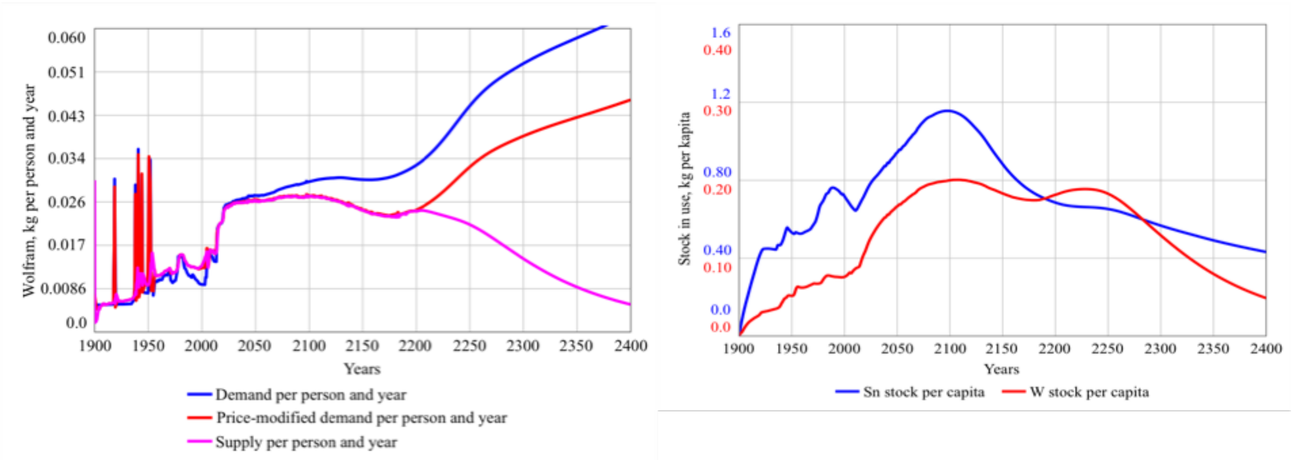
Source: Own Figure, Lund University/University of Iceland

4.1.9.7 Discussions

For the future, wolfram can be expected to have an increasing demand, as there are very many advanced technologies where it gives a technical advantage. Two parameters are the most important for the assessment of scarcity. The first one is global average supply per capita per year and the second one is global average stock-in-use per capita. These two we must have as plots before we can assess the nature of the potential future scarcity. Supply per person per year is necessary to assess if we can have growth, if we will get global stagnation or global contraction. Stock-in-use per person is necessary to assess if we can increase global utility provision, if it will develop into global stagnation or global contraction.

Figure 4.1.9.14a is the key diagram for the scarcity assessment. Here we can relate demand, demand modified by price and the amount supplied by the system. When these curves are not overlapping, then there is some type of scarcity, soft or hard. When demand and demand modified by price separates, we have soft scarcity. Soft scarcity occurs after 2030 for wolfram. When demand modified by price and supply separates, we have hard scarcity in 2200 because of resource exhaustion. Figure 4.1.9.14b, shows the wolfram stock-in-use per capita is an indicator for service provision per person as calculated by the model. The difference between the supply and the demand, suggests the degree of scarcity.

Figure 4.1.9.14. Wolfram supply in kg per person and year compared to the demand before any price adjustments is shown in diagram (a). Diagram (b) shows wolfram stock-in-use per capita,which is an indicator for service provision per person.



a b

Source: Own Figure, Lund University/University of Iceland

Sensitivity

The model does as a fact reproduce the observed mining rate, price and seems to be doing the ore grade well, even if there is little data for the last parameter. This does point out to us that the main causative elements have indeed been incorporated in the model. This suggests that further complication of the model may not necessarily make it better. The model is quite sensitive to the demand function used. The demand was estimated in the model as a curve depending on metal uses in other parts of the model and a component dependent on population only, but modified with inputs on connected demands from elsewhere in the WORLD model. This module is in itself of some interest for wolfram, but further also has a role to play in the WORLD6 model where the integrated impact of potential shortages and price dynamics on different technological developments will be assessed.

4.1.9.8 Conclusions

There is imminent risk for hard scarcity for wolfram in the future. We conclude that we can model the observed pattern for mining, extraction, ore grades and market price, based on mass balance and market principles. The complexity of the WORLD6 model does allow for many cross-linking aspects to be considered which were not possible with earlier, and cruder concepts.

4.1.10 SPECIALITY METALS sub-module – SUPER ALLOYS

4.1.10.1 Introduction

In modern society, some of the most important and promising uses of metals are like tantalum and niobium, but also other metals are in the use of superalloys (Co, Cr, Hf, Zr, Ti, Re, Mo, Ru, Ir, Al, W). Superalloys are metal mixes that have high strength at elevated temperatures and also good corrosion resistance at such conditions. A superalloy has high mechanical strength, resistance to thermal creep, excellent corrosion resistance also at high temperatures and surface structural stability. Chemical and petrochemical processing, combustion power plants, and oil and gas industries use superalloys in their technical installations. They find use in turbine blades, jet engines, rocket engines, nuclear energy technologies and specialized chemical environments. Superalloys mainly rely on dispersion of coherent L12 γ'-austenitic precipitates in a face-centered cubic γ metal matrix for excellent high temperature mechanical properties (Collier et al., 1988, Harada 2012). Corrosion resistance is enhanced by components creating corrosion resistant oxide coatings at high operation temperatures. The resistance to corrosion also makes them difficult and expensive to decompose and refine to single metals for recycling, as well as difficult to melt, and reuse. Thermal barrier used to reduce heat conductance and allow for higher temperatures (Clarke et al., 2012).

Compositions of superalloys have continued to evolve to be able to be used under increasingly challenging operation conditions. Additions of rhenium, ruthenium or other exotic materials have significantly improved creep strength and resistance to corrosion. Nickel-based alloys are the most common, but other alloys based on cobalt have been developed in recent years. Further research focuses on more alternatives using high contents of metals such as chromium, niobium, molybdenum, platinum, iridium or rhodium. The most frequent use is in high temperature section of turbines and jet engines. The superalloys may make up as much as 50% of the turbine weight (Reed 2006, Makineni et al., 2015a,b, Donachie and Donachie 2002). Tantalum and niobium are exotic metals that only found a use in modern times with the coming of high technology and sophisticated industrial metallurgy. Scarcity of natural resources in the future may cause trouble for continued technology development and the further development of the economy in the long term (Hubbert 1966, Acemoglu and Robinson 2013, Bardi 2013, Busch et al., 2014, Heinberg 2001, 2011, Hirschnitz-Garber et al., 2015, Morrigan 2010, Meadows et al., 1972, 1974, Nickless et al., 2014, Nassar et al., 2012, Ragnarsdottir et al., 2015, International Molybdenum Association 2013, 2015, Avon Metals 2014, Robinson and Menzie 2012).

The superalloys are used in a rapidly expanding range of applications from aerospace; disks, bolts, shafts, cases, blades, vanes, combustion chambers, thrust reversers. They are much used in civilian power plants; bolts, blades, stack gas reheaters, in high performance turbines for better thermal efficiency of energy conversion, in automotive vehicles for turbo-changers and exhaust valves. They are also becoming important in medical components because of their good resistance to corrosion and strength. In nuclear power generating systems they are used where repairs can only take place with huge difficulty because of radiation challenges such as for control rod drives, valve stems, springs, ducting and such places where great strength and corrosion resistance is needed. Chemical industry it is used for chemical reaction vessels, piping, valves, bolts and pumps. They are used in space vehicles in aerodynamically heated skins, rocket nozzles, turbo-pumps, rocket engine compressors and service turbines (Donachie and Donachie 2002, Sims et al., 2008). Price and available amounts seems to be the limitation to wider use.

4.1.10.2 Objectives and scope

The goal was to assess the sufficiency of the molybdenum, rhenium, niobium, tantalum, cobalt and nickel supply for production of superalloys, considering its technical use in rocketry, high performance turbines and advanced jet engines. Further, superalloys are being considered as candidates for inert electrodes for aluminium and light metal smelting.

4.1.10.3 Methods and theory used

Metals and metal uses

Table 4.1.10.1 shows a small part of the periodic table showing the position of the refractory metals used for composing superalloys. In addition, comes materials like boron, carbon, silicon and aluminium added in minute amounts. All these metals are characterized by high melting points and hardness, and some of them by good resistance to corrosion both alone and in combination. Table 4.1.10.2 shows the refractory metals and their approximate abundance on earth from the perspective of how much we estimate to be extractable for humans. The resource estimates are new estimates by the authors (Sverdrup and Ragnarsdottir 2014, Sverdrup et al., 2016a,b,c, forthcoming 2017a,b,c) for this study. We have also indicated if these metals are in soft scarcity (compensated with higher price) or hard scarcity (compensated with higher price and with limited amounts that can actually be delivered. Typical for this are the platinum group metals, which at times cannot be delivered at the amounts desired). Superalloys are alloys developed for heat resistance, shape stability at high temperature and corrosion resistance. They have either iron, cobalt or nickel base as the majority metal, and then additions of other metals that contribute to strength, chemical resistance to corrosion, thermal stability and resistance to fatigue failure. The key metals with their important properties for making superalloys are:

1. Nickel (superalloy), cobalt (superalloy) and sometimes iron (stainless steel), these metals often make up the base of the alloy with more than 50% of the weight. They provide the basic stability. Nickel increase resistance to acids and alkali. They set the basis for the melting point of the alloy. Cobalt provides resistance to carburization and sulphidation, as well as general good corrosion resistance.
2. Chromium, nickel provide resistance to oxidative corrosion, and high temperature oxidation and sulphurization.
3. Titanium, zirconium and hafnium provide strength and reduce intergranular corrosion caused by heating cycles and hardening, and enhances the effect of hardening. Hafnium increase the melting point of the alloy.
4. Molybdenum and wolfram provide increased melting point. Molybdenum increase resistance to reducing acids and resistance to chloride corrosion.
5. Wolfram increase the melting point and increase resistance to reducing acids, increases strength and weldability.
6. Tantalum, niobium and rhenium provide corrosion resistance for both acids and alkaline conditions and promote increased melting point. Niobium reduce intergranular corrosion from carbide formation, and provide strength at increased temperatures.
7. Cobalt, platinum, palladium, rhodium, iridium, osmium and ruthenium provide corrosion resistance and increased melting point. Iridium increases strength and stabilize against cracking.

Table 4.1.10.1. A small part of the periodic table showing the position of the refractory metals.

Ti	V	Cr	Mn	Fe	Co	Ni
Zr	Nb	Mo	(Tc)	Ru	Rh	Pd
Hf	Ta	W	Re	Os	Ir	Pt

Table 4.1.10.2. Summary of the physical properties of some of the refractory metals used for superalloys. The resource estimates are new estimates by the authors (Sverdrup et al., 2017a,b,c) for this study.

Name	Niobium	Molybdenum	Tantalum	Wolfram	Rhenium
Melting point °C	2,477	2,623	3,017	3,422	3,186
Boiling point °C	4,744	4,639	5,458	5,555	5,596
Density g·cm−3	8.57	10.28	16.69	19.25	21.02
Young's modulus GPa	105	329	186	411	463
Vickers hardness MPa	1,320	1,530	873	3,430	2,450
Corrosion resistance	Good	Moderate	Very good	Poor	Very good
URR, mill ton	52	55	0.5	25	0.02
Type of scarcity	Soft	Soft	Soft	Soft	Physical
Name	Nickel	Cobalt	Palladium	Iridium	Hafnium
Melting point °C	1,455	1,495	1,555	2,446	2,233
Boiling point °C	2,730	2,927	2,963	4,130	4,603
Density g·cm−3	8.91	8.83	12.0	22.6	13.31
Young's modulus GPa	200	211	121	528	78
Vickers hardness MPa	638	1,043	600	2,200	1,760
Corrosion resistance	Moderate	Moderate	Excellent	Excellent	Excellent
URR, mill ton	290	64	0.17	0.01	0.6
Type of scarcity	Soft	Soft to physical	Soft	Physical	Physical
Name	Iron	Chromium	Ruthenium	Osmium	Zirconium
Melting point °C	1,538	1,907	2,234	3,033	1,855
Boiling point °C	2,862	2,671	4,150	5,012	4,377
Density g·cm−3	7.9	7.19	12.45	22.6	6.5
Young's modulus GPa	211	279	447	480	88
Vickers hardness MPa	608	1,060	2,160	3,950	1,800
Corrosion resistance	Poor	Moderate	Excellent	Excellent	Excellent
URR, mill ton	360,000	2,500	0.035	0.008	420
Type of scarcity	Sufficient	Sufficient	Physical	Physical	Sufficient
Name	Manganese	Vanadium	Rhodium	Platinum	Titanium
Melting point °C	1,246	1,910	1,964	1,768	1,855
Boiling point °C	2,061	3,407	3695	3,825	4,377
Density g·cm−3	7.2	6.0	10.7	21.5	6.5
Young's modulus GPa	198	128	380	168	88
Vickers hardness MPa	196	640	8,000	550	1,800
Corrosion resistance	Poor	Moderate	Excellent	Excellent	Excellent
URR, mill ton	4,700	50	0.035	0.15	3,400
Type of scarcity	Sufficient	Soft	Physical	Physical	Sufficient

The examples of the composition of typical superalloys are shown in Table 4.1.10.3. The first superalloys were discovered in the 1940's, with a demand from the development of the first jet engines and rocket engines. These alloys had a cobalt basis (65%) and used high contents of chromium (23%) with additions of molybdenum and niobium. They tolerated higher temperatures better than the nickel-based stainless steel type of alloys available then. As the arms race steadily demanded higher performance jet engines, superalloys rapidly evolved. Nickel-based alloys soon became the dominant approach. The aid of thermal barriers, corrosion coatings or active surface cooling systems can improve the performance significantly. Lately, new gamma-phase stabilized cobalt and niobium have been developed (Bian et al., 2015). Further, new platinum group metal-based alloys and new thermal are being developed. With the discovery of the gamma-phase stabilization and single crystal casting, nickel alloys became better than the cobalt-based alloys. The diagram was drawn based on data from the literature (Albrecht 1989, Bacos et al., 2009, Boot-Morrison et al., 2008, Caron and Lavigne 2011, Collier et al., 1988, Cornish et al., 2004, 2011, 2012, Geddes et al., 2010, Harada 2012, Graedel et al., 2015a, Yeh et al., 2015, Kablov et al., 2012, Kawagishu et al., 2012, Makineni et al., 2014, 2015a,b, Meng et al., 1984, Mottura and Redd 2014, Mukherij and Rösler 2010, Reed 2006, Schneibel 2005, Tanaka et al. 2003, Tien et al., 1989, Vaßen et al., 2010, Wahl and Harris 2014, Yamabe-Mitarai et al., 2002, 2004, Zhao et al., 2015).

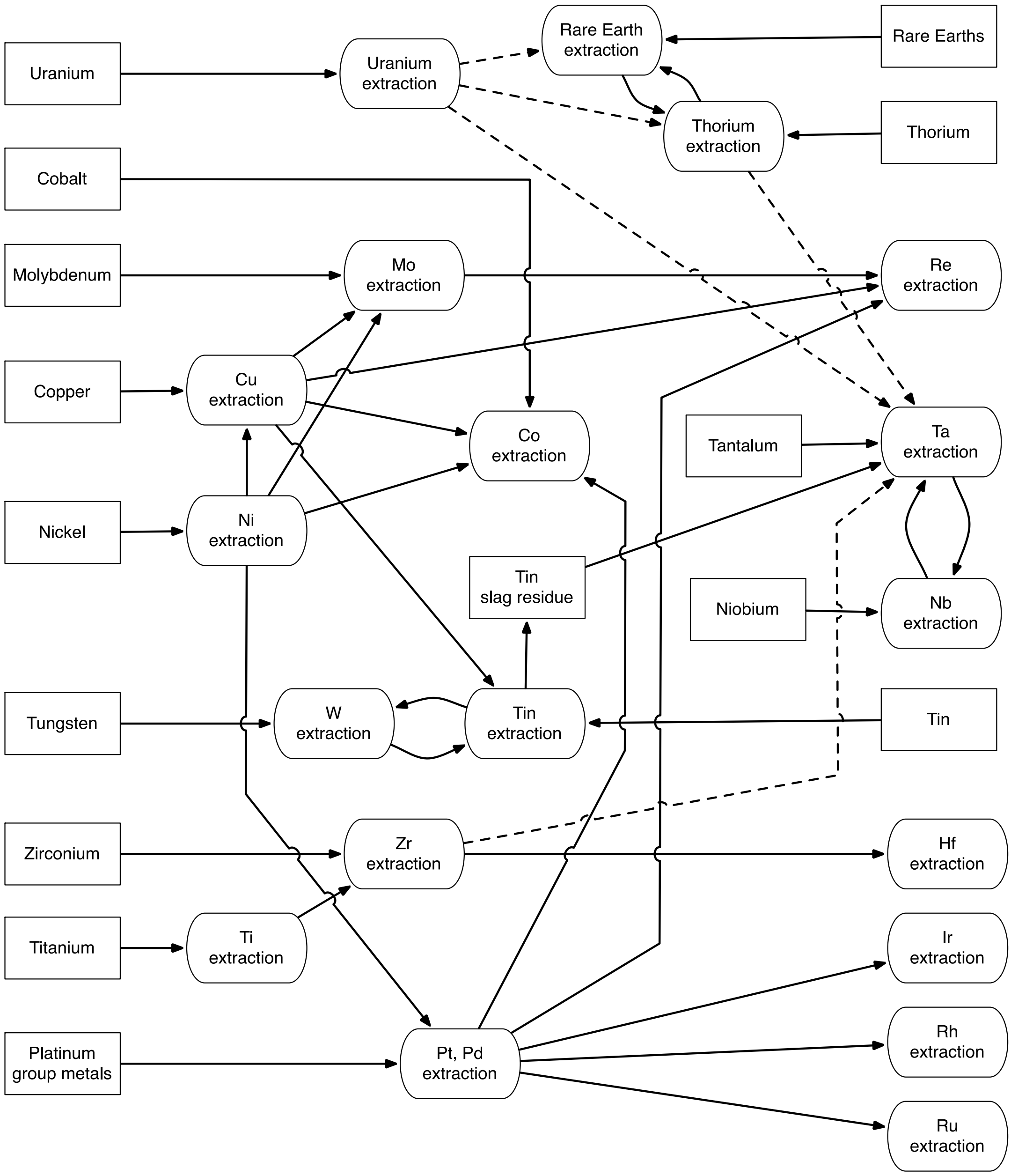
Recent developments after 2000 with discovery of gamma-phase stabilized cobalt alloys with ruthenium and rhenium enhancers, and cobalt alloys may catch up or even gain better performance than the nickel alloys (Helmink et al., 2000, Mottura and Reed 2014). The metals used for minor components are molybdenum, niobium, rhenium, wolfram, tantalum, zirconium or hafnium. The base metal sets the starting point for the melting temperature of the alloy created. Most alloys are eutectic and have lower melting points than the end members. The coating protects against corrosion at these conditions as well as acting as thermal barriers (Adapted and reworked using data from Graedel et al., 2015a, Dexclaraux and Serre 2003, Makineni et al., 2015a,b,c, Vaßen et al., 2010, Booth-Morrison et al., 2008, Geddes et al., 2010, Harris et al., 1992, Helmink et al., 2000, Meng et al., 1983, Tien et al., 1989, Albrecht 1989, Tanaka et al., 2003, Yamabe et al., 1998, 2002, Cornish et al., 2012, Cornish and Chown 2011, Gu et al., 200, Makineni et al., 2015a,b,c). When the alloys are combined with active cooling systems and thermal ceramic barriers, this may increase the maximum temperature by 200-300 degrees over what the naked alloy would have been able to tolerate. Recent research seem to suggest that there will be an upper temperature around 2,200oC where the ceramic coatings start to loose their mechanical strength and be susceptible to corrosion (Gaballa 2012, Clarke et al., 2008, Makineni 2015a,b). Most alloys are nickel-based at present, but cobalt-based alloys are going through improvements and seems to be set for a come-back. Characteristic for all these components are that they have high melting points, low vapour pressure at the melting points (Table 4.1.10.2) and that they tend to make very hard alloys (Table 4.1.10.3). The potential would be further improvement in high temperature performance and corrosion resistance at high temperatures and high pressures. There is a complex interdependency between the metals used for superalloys. A flow diagram illustrating the flows and the interdependencies is shown in Figure 4.1.10.1. Figure 4.1.10.2 shows how the superalloys are created in several steps, either from primary material or from recycled material.

Table 4.1.10.3. Compositions of different super-alloys, with earlier at the top and modern at the bottom.

Alloy	Contents in % weight of the metals used in the alloy.																
	Cr	Co	Mo	Fe	W	Al	Ti	Zr	Hf	Ta	Nb	Re	Ru	Pt	C	Ni	Si
Iron-based alloys examplified																	
IN-COLOY 26-6MO	20		6.5	47												25	
IN-COLOY 864	21		4.3	39												34	
Nickel-based alloys examplified																	
T4	8	10	6	-	-	6	1	0.08	-	3	-	-	-	-	-	65.8	
N1	8	10	6	-	-	6	1	0.08	-	-	2.15	-	-	-	-	66.7	
S1	14	16	8.5	-	-	-	-	-	-	1	-	-	-	-	-	71.5	-
CMSX-3	8	5	0.6	-	8	5.6	1	-	0.1	6	-	-	-	-	-	65.2	-
CSMX-4	6.5	9	0.6	-	6	5.6	1	-	0.1	6.5	-	3	-	-	-	61.7	-
CM4670 C	2.7	3.2	0.4	-	5	5.7	0.018	0.025	1.2	8	0.05	6	-	-	-	67.7	-
TMS-196	4.6	5.6	2.4	-	5.0	5.6	-	-	0.1	5.6	-	6.4	5.0	-	-	59.7	-
TMS-238	4.6	6.5	1.1	-	4.0	5.9	-	-	0.1	7.6	-	6.4	5.0	-	-	58.8	
K648d	33.4		2.8		4.7	1.1	1.1				1.1				0.06	54.7	
GH169	17.7		3	18.1		0.5	0.94				5.4				0.02	54.3	
Cobalt-based alloys examplified																	
Co-1	23	65	5	-	6	-	-	-	-	-	2	-	-	-	0.3	3	-
Co-2	25	67	5	-	-	-	-	-	-	-	2	-	-	-	1	8	
Co-3	3	50	5	-	-	10	-	-	-	-	2	-	-	-	-	30	-
Iron-based alloys examplified																	

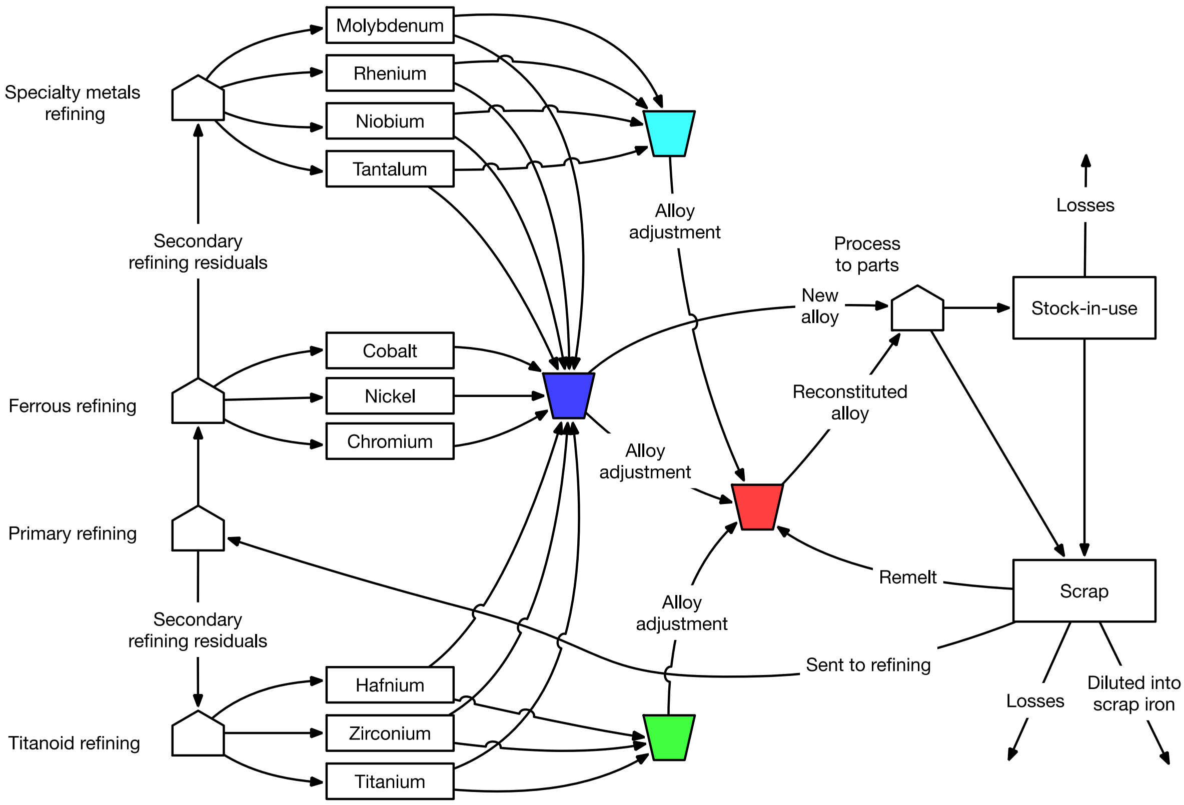
718	23	-	8	61	6	-	-	-	-	-	2	-	-	-	-	-	-
Niobium-based alloys examplified																	
Nb-1	-	-	5	-	15	-	-	-	5	-	54	-	-	-	5	-	16
Platinum-based alloys examplified																	
PGM-1	3	-	-	-	-	11	-	-	-	5	-	-	-	75	-	6	-
PGM-2	3	-	-	-	-	14	-	-	-	-	-	-	3	80	-	-	-
PGM-3	3	-	-	-	-	13	-	-	-	-	-	-	-	77	-	-	7
Iridium-based alloys examplified																	
								Zr	Hf	V	Nb	Re	Ru	Pt	Ir		
Ir-1								-	-	-	25	-	-	-	75		
Ir-3								-	12	12	25	-	-	-	39		
Ir-4								6	8	12	20	-	-	-	54		

Figure 4.1.10.1. A flow chart showing the production pathways for the key metals that are used for making superalloys. These metals are defined by Table 4.1.9.2. The extraction of these metals is linked in an intricate flow chart that is also reflected in the WORLD6 model.



Source: Own Figure, Lund University/University of Iceland

Figure 4.1.10.2. Superalloys are produced in several steps, either from primary material or from recycled material. Depending on the quality of the incoming scrap, the alloys are either fully decomposed and refined or the alloy is recycled as alloy, the composition corrected and the metal is recast into parts. The processing of the alloy to parts is in itself a complicated process. The metal is cast and grown to a single crystal, then shaped and in several steps given different types of protective surface layers.

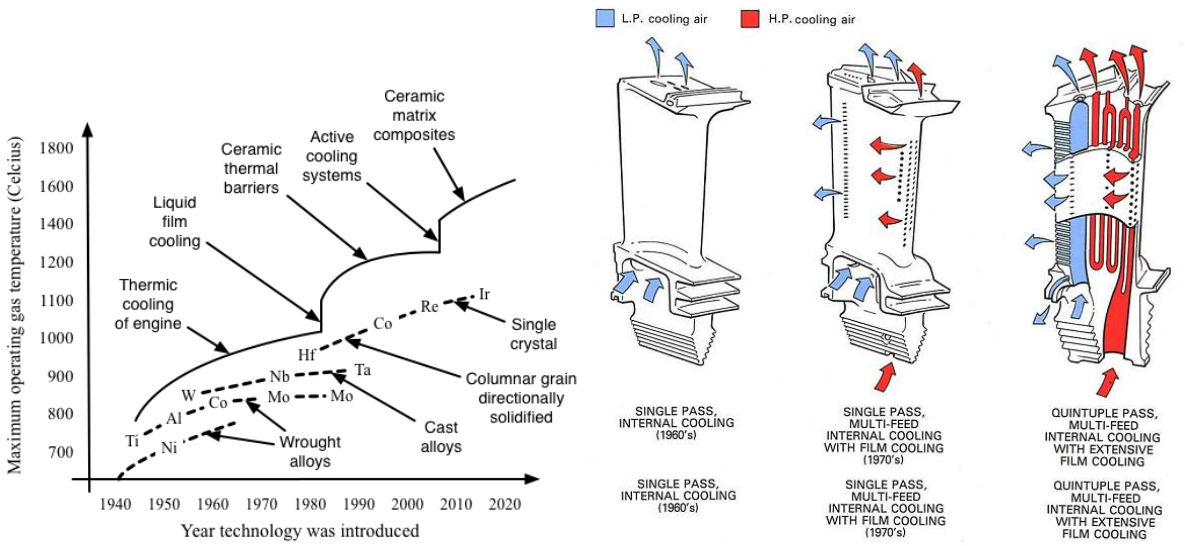


Source: Own Figure, Lund University/University of Iceland

The development started with the development of the first jet engines in Britain and Germany during the second world war. The first alloys were wrought iron-nickel-chromium alloys, in principle variants on stainless steel. Soon, molybdenum and niobium were also employed in the alloys. The first stage of a modern turbine faces temperatures up to about 1,400 °C at present. Modern military jet engines, like the French Snecma M88 which has maximum operating turbine temperature of 1,590 °C (Dexclaraux and Serre 2003). Rocket combustion chambers and nozzles may face even higher temperatures for shorter periods of time. For building such high technology components, superalloys are used, and when needed in combination with protective coatings. In order to increase gas temperatures, engine cooling, liquid surface films, thermal barriers, ceramic material coatings and active cooling systems are employed. Figure 4.1.10.3 shows the role of different metals and cooling techniques for increasing the maximum operating temperature for superalloy components. We have plotted them as more and more metals are added into the alloys and the maximum operating temperature increase.

Depending on the quality of the incoming scrap, the alloys are either fully decomposed and refined or the alloy is recycled as alloy, the composition corrected and the metal is recast into parts. The testing indicator for Figure 4.1.10.3 was 100 hours at 1,400 bar pressure without aid of thermal coatings or extra cooling systems. The role of different metals for increasing the maximum operating temperature for superalloys, partly those listed in Table 4.1.10.2 is shown in Figure 4.1.10.3. The dotted lines show the upper use temperature for the alloy alone. The whole drawn line shows the limit for the gas temperature, when different cooling strategies are used (engine cooling, liquid surface films and internal closed cycle turbine blade cooling systems), ceramic thermal barriers (They protect against heat and chemical attack. The evaluation measure was exposure for 100 hours at 1,400 bar pressure. Single crystal alloys are relying heavily on the use of rhenium, and another rare metal, hafnium, is the only one offering something similar. The alloys are mainly nickel and cobalt-based, however other metal bases are being explored. Some of the metals are available in very small amounts (Total resource is less that 1 million ton for tantalum, rhenium, palladium, rhodium, platinum, ruthenium, osmium and hafnium), and will thus never allow for use in mass production (Rhenium, osmium, iridium, tantalum, platinum, palladium, ruthenium, rhodium, hafnium). Further increases are researched and new elements like the platinum group metals (Pt, Ru, Rh, Ir, Os) may enter the scene at some point. However, these are scarce metals in very short supply.

Figure 4.1.10.3. The role of different metals for increasing the maximum operating temperature for superalloys. The dotted lines show the upper use temperature for the alloy alone. The whole drawn line shows the limit for the gas temperature, when different cooling strategies are used, ceramic thermal barriers. The picture on the right illustrate different cooling methods and how they have evolved in the last 60 years. The evaluation measure was exposure for 100 hours at 1400 bar pressure.

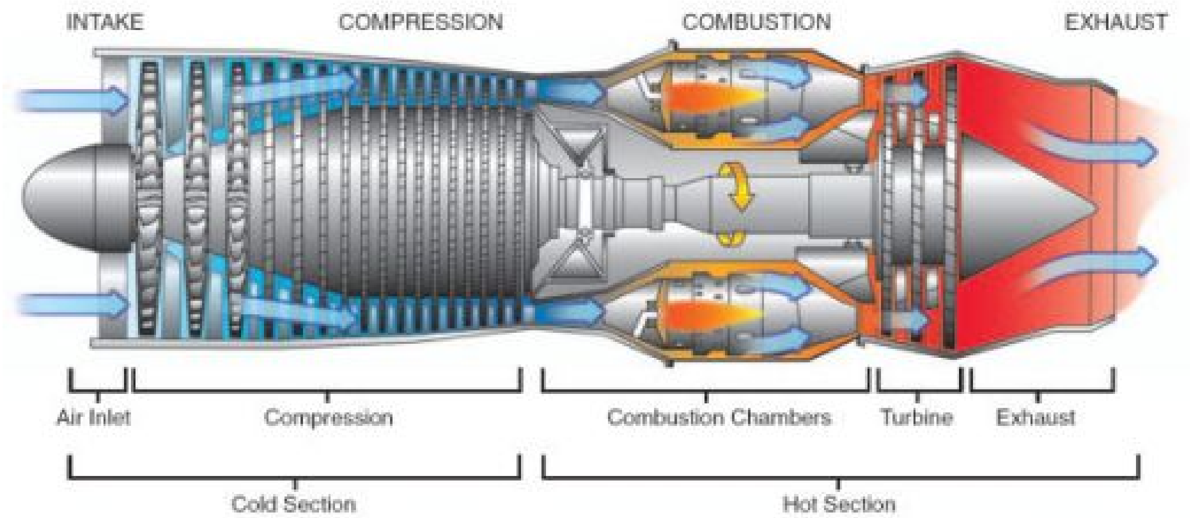


Source: Own Figure, Lund University/University of Iceland & http://s5.picofile.com/file/8115081434/29.jpg

Figure 4.1.10.4 shows a cut though a turbofan jet engine. Titanium is ideal for its strength and density at low temperature (blue in Figure 4.1.10.4). At high temperatures, titanium is replaced by the superalloys (red in Figure 4.1.10.4). Different metals are used in a jet engine depending on what kind of stress they are exposed to. In the front of the compression section at the beginning of the engine, the gasses do not get very warm, they stay below 150oC. Dominant in this section is to use aluminium, titanium and different types of stainless steel. In the later part of the compression zone, it may get as hot as 1,000oC because of the adiabatic compression. Here a high strength at high temperature material must be used. Nickel- or cobalt-based superalloys are used. The combustion chambers can get up to 1,800oC and superalloys and several different types of coatings are used for this part. Additions of aluminium and titanium are not needed for strength in the combustion section, as there are no moving parts, but good tolerance to heat, corrosion and creep is required. Metals like wolfram, molybdenum, niobium, tantalum, rhenium and hafnium are needed in the superalloy to increase the heat tolerance of the superalloys. Ceramics and metal-ceramics composites are used in the combustion section to increase heat resistance and insulation and corrosion resistance. The ceramics have poor resistance to shocks and movement, but there are few of these in the combustion compartments (Donachie and Donachie 2008).

A bit further down in the engine, in the hot turbine, the gasses have cooled only slightly, and the metal must have extraordinary strength, good corrosion resistance implying superalloys. The turbine blades are often single crystal blades. Earlier in jet engine development, stainless steels were used, but as higher performance is demanded, superalloys with several layers of barriers are used (Donachie and Donachie 2008). The exhaust nozzle use Inconel or stainless steel alloys (nickel-chromium-iron alloys). When the temperature over the turbine is allowed to increase from 500oC to 1,600oC, then the thermic conversion efficiency of the engine increase from about 27-30% to 48-52%. Superalloys have been composed for heat resistance, shape stability at high temperature and corrosion resistance (Donachie and Donachie 2008, Clarke et al., 2012, Harada et al., 2012, Kwagishu et al., 2015).

Figure 4.1.10.4. The figure shows a cut though a military jet engine of the type found in jet fighters. Turbofan engines are rather used in civilian aircraft such as those built by Boeing or Airbus. Different metals are used in a Rolls-Royce jet engine depending on what kind of stress they are exposed to. Titanium is ideal for its strength and density at low temperature (blue). At high temperatures, it is replaced by superalloys (red).



Source: https://usercontent1.hubstatic.com/816606_f520.jpg

4.1.10.4 Model description

Data and materials

Table 4.1.10.4 shows an overview of important alloying metals for superalloys. s.d.a.p. = Systems dynamics model assessment planned for later point in time. n.a.= estimate not available at present. Amounts shown are expressed in million metric ton contained. It also shows a compilation of some estimates of peak production date estimates and to which degree the extraction is primary hard-rock mining or secondary and dependent on mother metals.

Table 4.1.10.4. Overview of important alloying metals for superalloys. The metal production numbers are taken from a number of earlier studies by the author. s.d.a.p. = Systems dynamics model assessment planned for later point in time. n.a.= estimate not available at present. Amounts are in million metric ton contained.

Metal	Metal production in 2015 in ton per year	Annual extraction as % of URR	Dynamic model peak year estimate	URR Mill. ton	Dependent Extraction	Hubbert's Model valid?

Molyb-denum	0.27	0.5	2032	55	Partly, Cu	Yes
Rhenium	50	0.3	2035	0.016	Yes, Mo	No
Cobalt	120,000	0.4	2040	29	Yes, Cu, Ni, Cr PGM	No
Vana-dium	80,000	0.2	s.d.a.p.	39	Partly, Fe	No
Niobium	65,000	0.3	2055	52	No	Yes
Tanta-lum	1,200	0.4	2045	0.5	Partly, Nb, Sb	No
Titanium TiO2	310,000 6,700,000	0.0090 0.1860	2080 Sufficient	3,600	No	Yes
Zirco-nium ZrO2	45,000 1,530,000	0.0011 0.3731	Cost limited Sufficient	410	Partly, Ti	Yes
Hafnium	80	0.0008	Cost limited Sufficient	11	Yes, Zr, Ti	No
Nickel	2,500,000	0.9	2030	300	No	Yes
Wolfram	85,000	0.3	s.d.a.p.	30	No	Yes

The metal extraction model description

The flow pathways, the causal chains and feedbacks loops in the global tantalum-niobium-tin system were mapped using system analysis, and the resulting coupled differential equations were transferred to computer codes for numerical solutions, using the STELLA® environment for the WORLD6 model. The model is solved numerically with a 4-step Runge-Kutta method, using a 1/60 year time-step (weekly time-step). The model was developed to estimate supply, extractable amounts stocks, price and stocks, and flow in society in the time interval 1900-2400. An important feature is that linkages between different types of extractions, such as illustrated in Figure 4.1.10.1 and Figure 4.1.10.2 are incorporated. The following 3 main modules were used:

1. The SPECIALTY metals submodules
2. The STEEL submodule
3. The BRONZE submodule

Dependent stocks are cumulative amounts of: mined, amounts rock waste, losses, smelter slag and ore found by prospecting. The mining activity is profit-driven, the profit is affected by the mining cost and the market price. For the purpose of clarity, the recycling fraction displayed in results were calculated as follows in this study:

$$Recycling\ fraction = \frac{Flow\ of\ recycled\ metal}{Supply\ from\ primary\ extraction + \ Flow\ of\ recycled\ metal} \quad (4.1.10.1)$$

The cost of the mining and extraction operation is mainly determined by two important factors beside cost of investments, the energy price and the ore grade. The price is determined by two factors, that it must stay above the production costs and by the amount in the market. The profit is driven by metal price and amount extracted, but balanced by the cost of operations (Sverdrup et al., 2016a,b). Related to the state of the technology has two alternatives:

1. Either the technology and the financial capacity determines the mining rate and the ore body is always sufficient to supply anything attempted to be extracted, then the extraction rate has a low order of dependence on the known reserves (n=0.5-0)

2. Or the mining rate is unrelated to the technical and financial capacity and limited by the access to the ore body by its location or from restrictions by society. The extraction rate has a higher order of dependence on the known reserves (n=1.0-0.5)

The cost of the mining and extraction operation is mainly determined by three important factors;(1) the cost of investments, (2) the energy price for the operation and (3) the ore grade. The price is determined by the fact that the income from sales in the market must stay above the production costs including the capital investment costs and a reasonable profit margin. The price curve in the model relate the actual tradable amount in the market to the market price being paid. These curves were developed from data on copper, silver, gold and platinum trading by the authors personal notes and experiences from being on the metal exchange trading floors in London and New York, as well as data for nickel, zinc, lead, iron, oil and coffee. In the model, the market price is set every week throughout the simulation. During this period the global population is expected to reach a peak and start to decline (Sverdrup et al., 2011). In the same period, it is expected that the global energy production will go through a peak and decline. As a basis for this analysis we also look back on earlier work done to verify the performance of the WORLD6 model on specific groups of metals.

Strategic considerations leading to metal demand

With the new fifth generation jet fighter planes and very high performing jet turbine engines, a significant amount of superalloys will be needed for turbine blades (Polyak 2011, Roskill 2010, 2015, Avon Metals 2014). The United States and its allies in NATO intend to build about 3,100 new F-35 during the next 10 years (50% of the total present production). In addition, comes other 1,000 US high performance military aircraft. Possibly, other big military powers (China, Russia, France, Great Britain, Ukraine, India) will possibly need superalloys for their comparable number of aircraft with possibly 3,000 or more of high-performance military type of jet engine turbines for the next 10 years and the demand for the next 20 years will be 8,600-10,000 aircraft. Rolls Royce estimated in 2011 that 149,000 jet engines will be demanded during the next 20 years, used to power 68,000 civilian jet aircrafts of different types (Donachie and Donachie 2008, INSG 2013). Military aircraft come in addition to these estimates. Superalloy weight in a typical modern jet engine has stabilized at 40-50% of the engine weight (Donachie and Donachie 2008, INSG 2013). This may lead to a specialty metals demand that is larger than the actual supply at present.

The civilian aircraft's new generations high-performing jet engines use superalloys in the turbine blades (Airbus A360, A380, Boeing 777, Boeing 787 and several coming new models from other vendors). The Boeing Corporation in Seattle estimates that 38,000 new civilian aircraft will be taken into service by 2034 (next 20 years) worldwide (1,900 commercial aircraft per year) (Boeing website statement in 2017). That corresponds to 30,000-60,000 ton of superalloy of at least 4th generation performance during the period, or 1,500-3,000 ton superalloy per year will be needed, at 6% tantalum, that corresponds to about 90-180 ton per year (Donachie and Donachie 2008, INSG 2013).

Table 4.1.10.5 shows a summary of superalloy demand for military and civilian aircraft, as well as other uses such as turbines for land-based power production during the next 20 years, 2017-2037. The superalloy demand for military and civilian aircraft depends on engine weight, number of engines per aircraft and the lifetime of the superalloy parts inside the engines, and how many retrofits of the engines that will be needed. These are based on rough estimates. Each new civilian aircraft is assumed to be twin engine planes with an average engine weight of about 4 ton per engine.

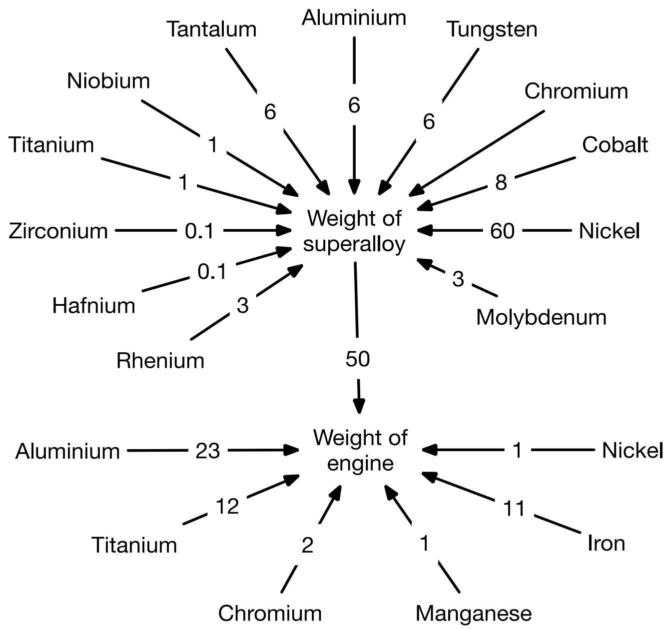
Table 4.1.10.5. Summary of superalloy demand during the next 20 years, 2017-2037

Item	Superalloy use, ton metal			
	Civilian aerospace	Military technology	Technical and chemical	Sum
Demand next 20 years	370,200	28,030	15,000	413,230
Annual demand	18,510	1,402	750	20,662

Figure 4.1.10.5 was used to estimate the demand for different metals in the manufacture of jet engines, to be used as input to the demand in the WORLD6 model.

We have taken two of the newest high performance engines as used their weight specifics; the military engines by Pratt and Whitney 135 and General Electric and Rolls Royce F136 engines. These are used in the F-22, a twin engine high performance superiority jet fighter plane, its total empty weight is 19.7 ton, twin engine of each a weight of 1.8 ton, total engine weight in the plane is 3.6 ton. The superalloy weight is about 1.8 ton. F-35 with a single engine, has a single engine, aircraft weight is 13.4 ton, the engine weight 1.7 ton, the superalloy weight is about 0.9 ton. Typically, the turbine rotates at 3300 rotations per minute, resulting in a tip speed of 1,700-2,00 km h-1. Typically, an engine will last about 10,000 flights before it is scrapped. Civilian aircraft engines vary a lot in size and weight. Much efforts have been made in later years to bring down overall weight, increase thrust to weight ratio and increase overall fuels use efficiency. We have looked into the latest series created by Rolls Royce in the Trent series. This of jet aircraft engines are used in planes from Airbus; A330, A340, A350, A380 and Boeing 777 and 787. They are all twin-engine aircraft. Typical engine weight for these turbofan engines range from 3.5 ton per engine to 7.1 ton per engine depending on the aircraft size and year of production. In 1950, only 10% of a jet engine weight would be superalloy, by 1985 this fraction had reached 50% of the engine weight (Donachie and Donachie 2002). In 2017, this was still the case, with a small upward trend. The flow of superalloy components is shown in Figure 4.1.10.5. This is based on averaging constituents in the newer alloys in the market (Table 4.1.10.3). There the contribution from each metal to the superalloy is shown in % weight of the metal, as well as the metals used for the rig and body of the engine. The superalloys make up about 50% of the engine weight, due to the high density of the constituents (8.8-9.2 g cm-3).

Figure 4.1.10.5. Diagram explaining the flow of metals to each jet engine. This was used in the WORLD6 model.

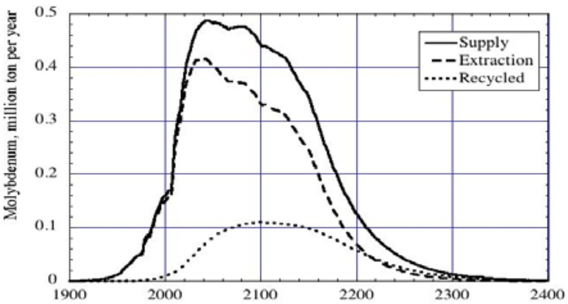


Source: Own Figure, Lund University/University of Iceland

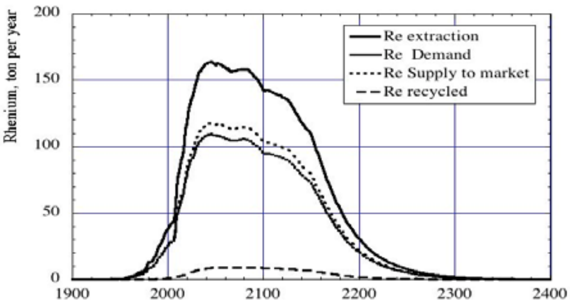
4.1.10.5 Results

The WORLD6 model was run for the time period 1900-2400. Figure 4.1.10.6 shows the extraction, production and market supply results for molybdenum, rhenium, niobium, tantalum, cobalt, nickel. The different metals have different supply profiles, and the supply of most of them shows a fall in extraction and supply levels around 2200.

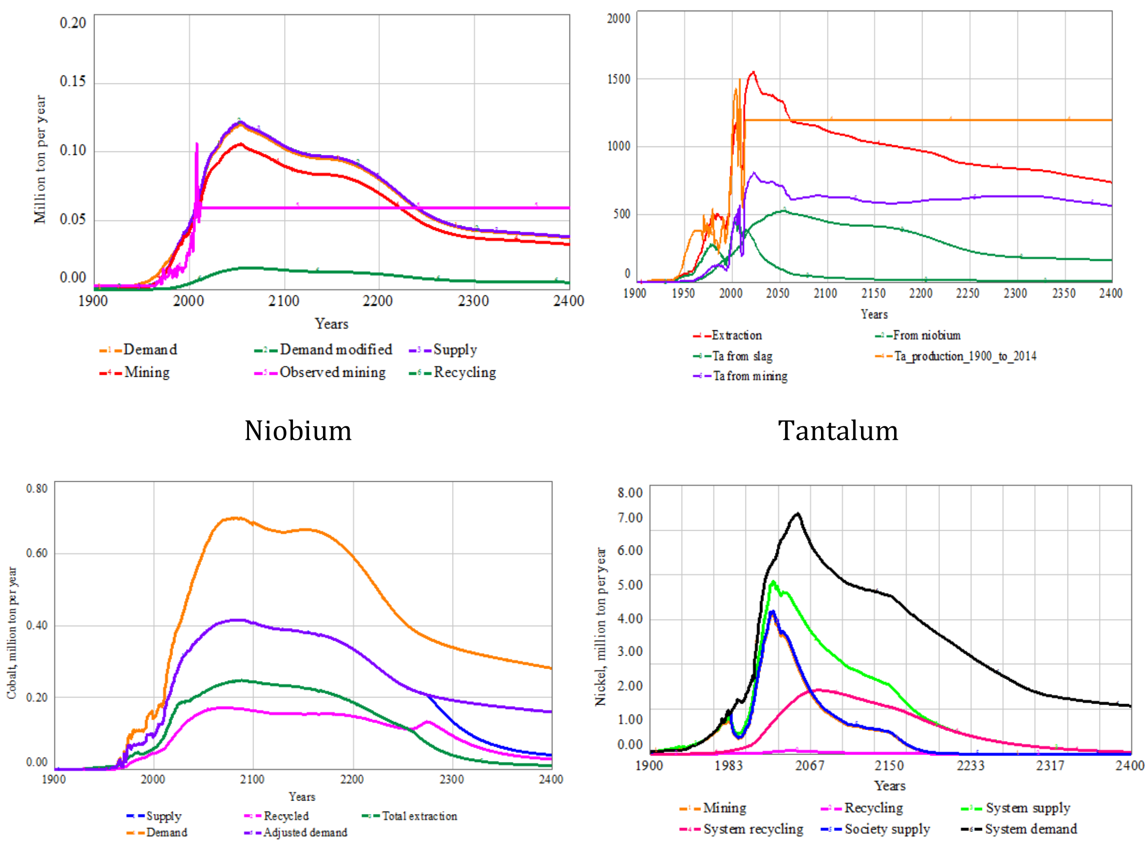
Figure 4.1.10.6. Comparing the extraction, production and market supply results for molybdenum, rhenium, niobium, tantalum, cobalt, nickel.



Molybdenum



Rhenium



Niobium
Tantalum
Cobalt
Nickel

Source: Own Figure, Lund University/University of Iceland

Table 4.1.10.6 shows cumulative extracted, supplied, recycled and lost material to 2400 AD for molybdenum, niobium, tantalum and rhenium, and the Factor X achieved under business-as-usual.

Table 4.1.10.6. Summary of cumulative extracted, supplied, recycled and lost material to 2400 AD for molybdenum, niobium, tantalum and rhenium, and the Factor X achieved under business-as-usual. All amounts shown are in metric ton

Metal	Metric ton metal in total				Factor X
	Extracted	Supplied	Recycled	Lost	
Molybdenum	63,700,000	85,000,000	21,300,000	63,400,000	1.3
Rhenium	16,800	18,200	2,400	16,400	1.1
Niobium	12,500,000	20,000,000	-	-	1.6
Tantalum	120,000	80,000	40,000	100,000	1.2
Cobalt	42,000,000	23,000,000	19,000,000	20,0000	1.8
Platinum	480	1,080	600	-	2.2
Nickel	309,000,000	620,000,000	311,000,000	200,000,000	2.0

4.1.10.6 Discussion

Strategic considerations for the future world

1. Military and strategic great power aspects:

   a. Without metals like tantalum, niobium, molybdenum, rhenium and hafnium available to the technologies demanding superalloys for their function, a sufficient number of advanced jet engines necessary for 5th and 6th generation fighter planes will not be available. States desiring these technologies in volume can no longer have that kind of hardware. That could mean loss of military superiority enjoyed by the few nations that have this today. Other sophisticated military hardware also depends on these metals to maintain their edge, and this will not be available to the same extent as before.
   b. Several types of missile engine nozzles and rocket engines will no longer be possible to make without at least 5th generation single crystal superalloys. This will limit military hardware performance, and weaken the military potential of states without own secure access to these metals.

2. Technological aspects:
   a. The limitation that could be caused to advanced jet engine technology is serious for future development of flight technologies and high-temperature jet turbines needed for fuel-efficient engines. Superalloys without the key metals (Made from molybdenum, niobium, tantalum, platinum group metals, zirconium, hafnium, cobalt, chromium, nickel (all which may face shortage risks in the next decades, with the only exception of zirconium and chromium) have definitely poorer technical performance and thus, this could become a problem in a future world with higher volatile hydrocarbon prices. In addition, those metals mentioned are also in scarce supply.
   b. Some very corrosion-resistant alloys can be made with superalloys containing molybdenum, tantalum, niobium, platinum group metals or rhenium and these are important for containment of corrosive molten salts containing halides. Examples of uses are:
      i. In molten salt cooled systems in advanced high-performance nuclear reactors. These are important for the development of the next generation molten salt cooled nuclear power plants based either on uranium 238 or thorium.
      ii. They are important for developing turbines for civil society power generation at higher temperature to increase thermal efficiency.
      iii. They are also important for the development of inert electrodes for smelting of light metals like magnesium or aluminium and for production of silicium and ferrosilicium.

The implications of point one above are that some countries may risk military inferiority if they do not take better care of their specialty metals and household with then significantly better than today. It implies that they must be conserved and recycled when that is not market profitable for the moment, for the reason that if it is not done, the material will have been lost when the price has increased through scarcity enough to promote recycling. The market will react too late, and thus foresight and long term governance is needed. Several civilian industrial uses will risk to face similar technological quality reductions if access to these metals would become restricted. Tantalum and niobium are important for super-alloys, and these will rise in price and have supply amount restrictions if recycling and conservation efforts are not implemented. The potential problems listed above may be efficiently mitigated by significantly improving recycling of the critical metals like rhenium, niobium, hafnium and tantalum (See Figure 4.1.10.2 for the importance of the metals). This would involve solving some technical challenges with respect to resistance to melting and hydro-chemical processing of the chemically very resistant alloys in question.

The role of the market

We have been living in a society for the last two decades dominated with a political climate where “market solutions” have been thought to efficiently self-regulate all markets. It is becoming more and more evident that this is not correct, and that unregulated markets tend to develop into oligopoly (Fukuyama 2006, Roberts 2014), leading to manipulated prices and unchecked speculations. There are sufficiently many empirical cases available to prove beyond reasonable doubt that scarcity issues are not efficiently solved by the market mechanisms alone, and that this approach as a solution is a dead end (Fukuyama 2006, 2012, 2014, Acemoglu and Robinson 2013, Heinberg 2001, Bardi 2013, Sverdrup et al., 2015). Such an approach is inviting failure and will cause irreparable scarcity situations (Ostrom et al., 1994, Acemoglu and Robinson 2013). It should be met with the resistance it scientifically deserves. Most countries do not have any strategic governmental policy on resources, and there are no International unified policies for recycling.

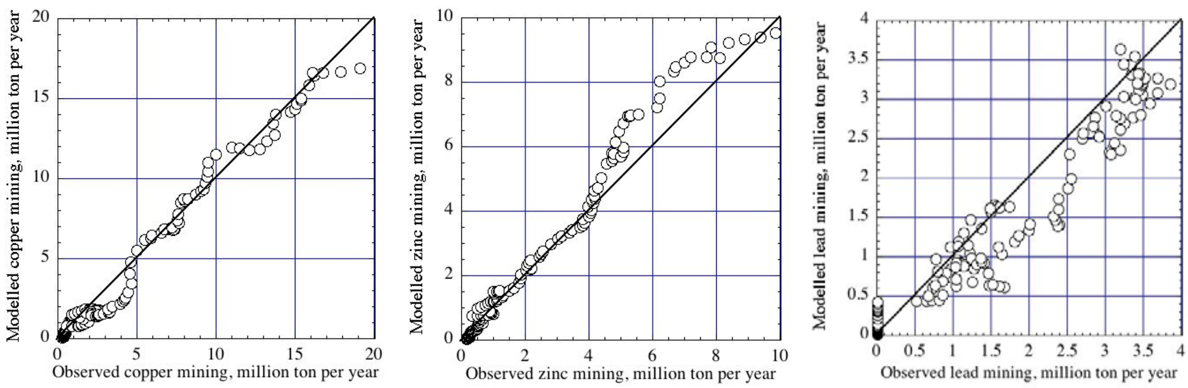
Reflections on testing against observed data.

There are several ways to check the validity and limitations of the parameterization of the model. One is that the input parameters are comparable to real world observations of the same parameters. That sounds very easy and fine, however, is not so obvious and easy when it comes to actually do that. Every measurement is a model too, thus we almost every time end up with comparing a calculation model against a measurement model. The other way is to check how well the full integrated model is capable of reconstructing the system outputs. When a model is capable of reconstructing the past with good success, then that is the nest validation that the model actually works. Discussions over what is politicall correct or reasonable is then redundant.

The WORLD model has been subjected to a number of such field data tests. A few of those available have been shown below:

The Figure 4.1.10.7 shows the results for copper, zinc and lead. It shows the mining rate for the metals with the observed values plotted in.

Figure 4.1.10.7. Comparison of the model outputs for mining rate to data for copper, zinc and lead.

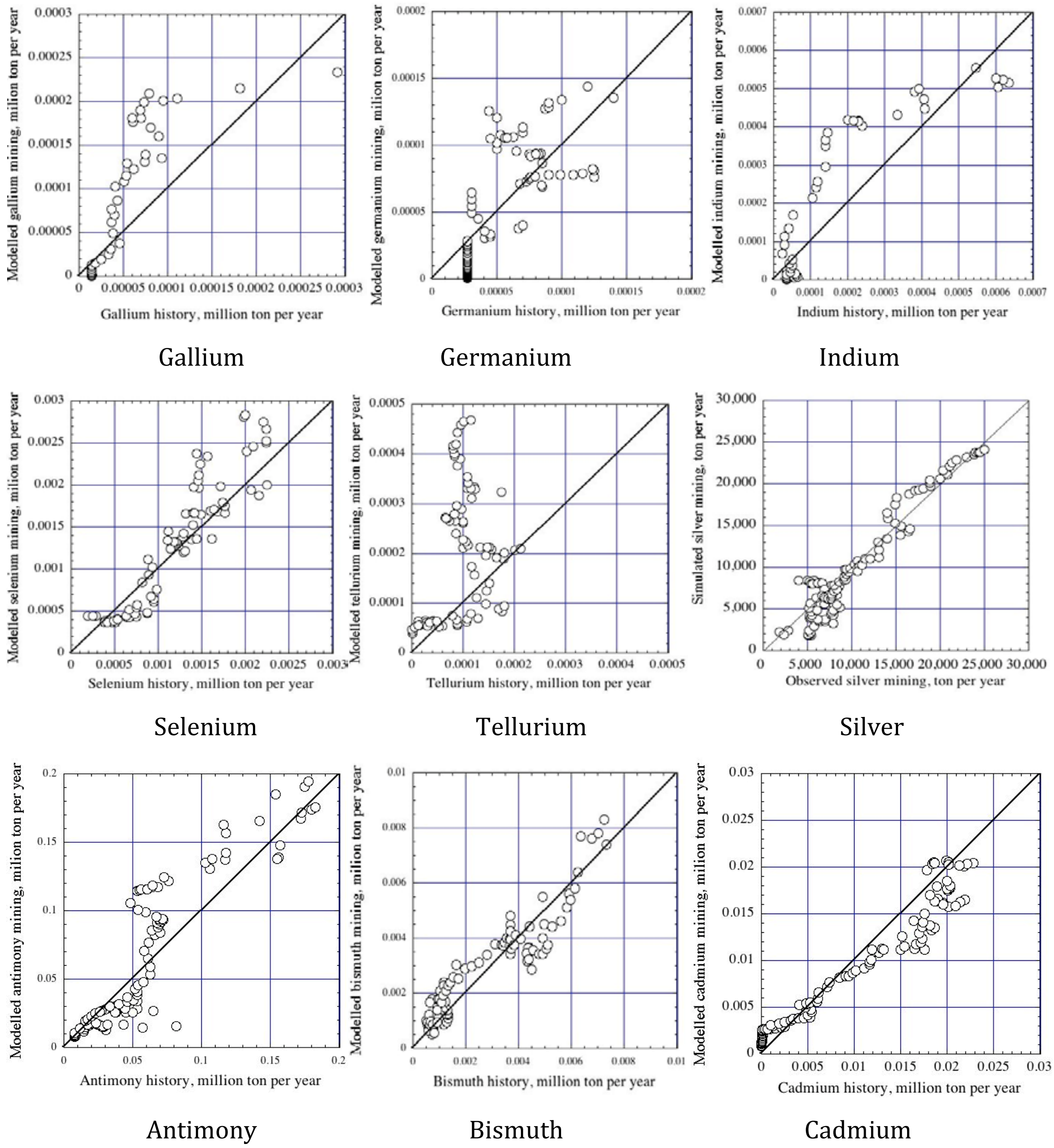


Copper Zinc Lead

Source: Own Figure, Lund University/University of Iceland

Figure 4.1.10.8 shows the production rate for the technology metals gallium, germanium, indium, selenium, tellurium, silver, antimony, bismuth and cadmium as modelled with the BRONZE model, as compared to the observed data on extraction from ores.

Figure 4.1.10.8. Comparison of modelled mining rate and data for gallium, germanium, indium, selenium, tellurium, silver, antimony, bismuth and cadmium



Source: Own Figure, Lund University/University of Iceland

From the comparison of modelled mining rate to the observed data for gallium, germanium and indium, the correlation coefficient is r2(Gallium)=0.74, r2(Germanium)=0.69, r2(Indium)=0.82. For selenium, tellurium and silver the correlation coefficient is r2(Selenium)=0.93, r2(Tellurium)=0.12 and r2(silver)=0.92. For antimony, bismuth and cadmium the correlation coefficient is r2(bismuth) =0.88, r2(antimony) =0.81 and r2(cadmium) =0.95.

4.1.11 SPECIALITY METALS sub-module – PLATINUM GROUP METALS (Platinum, palladium, rhodium)

4.1.11.1 Introduction

It is a known fact that the platinum group metals (PGM) platinum, palladium and rhodium are scarce natural resources and that these metals are of great importance for modern technologies and modern advanced chemistry. The platinum group metals (PGM) comprise the metals platinum, palladium, rhodium, ruthenium, iridium and osmium. The platinum group metals production is dominated by five major actors; South Africa, Russia, United States, Zimbawe and Canada. South Africa is the dominant producer of platinum (92%). For palladium, Russia (40%) and South Africa (37%) together share and dominate the market (77% of the total). South Africa dominates the rhodium market (80%), where the other big three producers have minor contributions. Several authors have previously expressed concerns about a potential scarcity and a future peak in platinum group metals production from mining and discussed the associated challenges and possibilities for the platinum group metals mining industry and the supply to the market (Jochens 1980, Allen and Behemanesh 1994, Cawthorn 1999, 2010, Wagner and Fettweiss 2001, Heinberg 2001, Gordon et al., 2006, Alonso 2010, Alonso et al. 2007a,b, 2009, 2012, Geoscience Australia 2009, Thomas 2009, Yang 2009, Glaister and Mudd 2010, Graedel et al., 2011, Mudd 2010, 2012a,b, Schooldermann and Martelehner 2011, Wäger et al., 2011, 2012, Gordon et al., 2011, Graedel and Erdmann 2012, Nuss 2014a,b, Radetzki 2012, Elshkaki 2013, UNEP 2013b, Bardi 2013, Eliott et al. 2013, 2014, Nansai et al., 2014). These have addressed different parts of the system, but not really taken a systemic grip and modelled the whole system and how platinum group metals circulate in the system. For that, this study is the first to put it all together and assess it as a dynamic system with feedback.

4.1.11.2 Objectives and scope

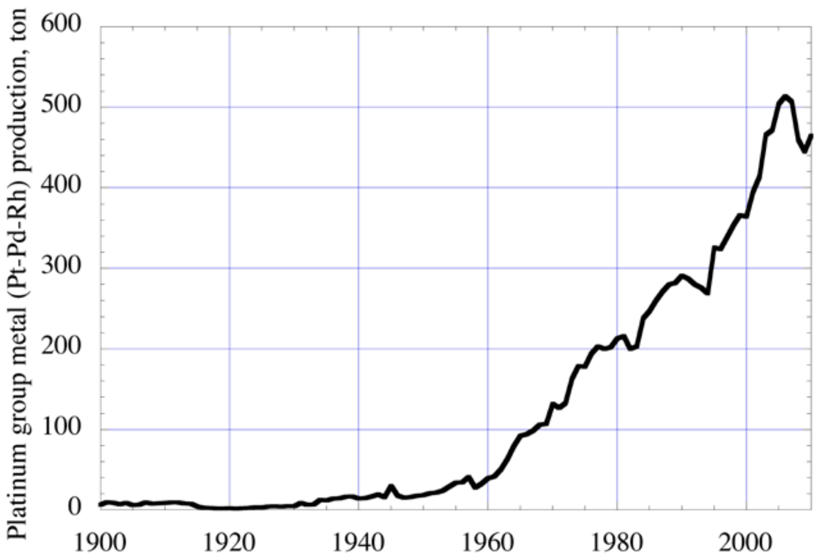
Our objective is to model the price of platinum and the physical availability, and to do this based on a systemic mapping of platinum group metals flows in the world platinum group metal trade system. The model we have developed has a wider scope than earlier developed models, and is based on a careful mapping of the whole system of causal feedback links. The wider scope implies that the causal links connect the physical flow dynamics to the market dynamics of supply, demand and market price modelled endogenously from basic principles. We have developed a tool for investigating the dynamics of the platinum market in relationship to what happens in the extraction, supply and consumption system. Our aim is to assess the long term sustainability of the platinum group metals supply to society, and assess the risk for shortage of platinum group metals expressed as decreased production and increased prices. The historic production curve shown in Figure 4.1.11.1 seems to suggest that platinum group metal mining production reached a peak in 2005-2010. We are asking a number of questions:

1. Production and extraction
    a. Will the production go through a peak?
    b. Are we at the platinum group metals production peak or not?
    c. Was the production peak in 2005 or will it occur later, or never?
    d. What would a system dynamics model assessment give?
2. Reserves and resources
    a. How large are the total extractable amounts of platinum group metals if we take in the latest resource assessments, consider new technology for mining at great depth, and the resources existing at great depth?
    b. How do the known reserves relate over time to the total estimated resources?

c. Can ore grade and extraction prices be included into the extraction dynamics and how does that influence the amount eventually extracted?
d. What is the ratio between the cumulative amounts supplied to society and the cumulative amounts primary extracted?

We assess the supply for the whole globe in a generalized way, and in simulations that cover the past, present and the future (1900-2400). We undertook a renewed survey and assessment of the extractable reserves and resources considering new technologies developed for deep mining as well as all extractable amounts regardless of price. The scope of this paper is to do this for the platinum group metals. We investigate how much the price is determined by supply and demand dynamics in the market. Gold is often found along with platinum, in particular in South Africa. The gold system has it's own intrinsic dynamics, and we have chosen not to include gold in this particular study (Sverdrup et al., (2013a). Ruthenium, iridium and osmium have largely been ignored in this study, their production volumes are relatively small.

Figure 4.1.11.1. Global mine production of platinum group metals in ton, for use in society of platinum. Have we reached the maximum production from mines or not? What does the continuation of production look like when we consider remaining reserves? Was the peak in 2005, or will it occur later? Data from USGS (2012-2015).



Source: Own Figure with data from USGS 2012-2015, Lund University/University of Iceland

4.1.11.3 Background information

Data was taken from a number of sources (von Gruenewaldt 1973, 1976, Hulbert and Gruenewaldt 1982, Suthpin and Page 1986, Prendergast 1988, Prendergast and Wilson 1989, Quiring (1962), Kunilov, 1994, Cawthorn 1999, 2010, Hilliard 2001, Williamson 2003, Wilburn and Bleiwas 2004, Gauthier et al., 2004, Dalvi et al., 2004, Cailteaux et al., 2005, Gotthelf 2005, Alonso et al., 2007a,d, 2009, 2012, Papp et al., 2008, Geoscience Australia, 2009, Oberthür 2011, Oberthur et al., 2003a,b, 2013; Gordon et al., 2006, 2005, Hagelücken 2006, 2012, Eckstrand and Hulbert 2007, Weatherstone 2008, Ragnarsdottir 2008, Hagelücken and Meskers 2009, Ragnarsdottir et al., 2012, Radetzki 2008, 2012, Saurat and Bringezu 2009a,b, Levine and Wallace 2009, Locmelis et al., 2010, Glaister and Mudd 2010, Mudd 2010, 2012a,b, Butler 2010, 2011, 2012, Rasilainen et al., 2010, Eilu 2011, Oberthür 2011, Rasalainen et al., 2012, Wilburn 2012, Wilburn and Bleiwas 2004, Raisalainen et al., 2010a,b, 2012, Elshkaki 2013, Nassar 2013, 2015, Polinares 2012a,b, UNEP 2011a, 20013b,c, Williamson 2003, Wäger et al., 2011, Wilburn 2012, Vermeulen et al., 2013, Royal Bafokeng Platinum 2013, Zientek et al., 2014, Russian Platinum 2014, Platinum Group Metals 2015, Brown et al., 2015,). The United States Geological Survey (USGS) database online (USGS, looked up in 2008, 2009, 2012, 2014, 2015, British Geological Survey 2009, Wilburn 2012, Stillwater Mining Mining Company 2010, 2011, 2012, 2015, Noril'sk Nikel 2012, 2014) was used for assembling data on extractable platinum group metal amounts and production; general flow and stock data were taken to a large extent from the JM Platinum Review booklets (Butler 2010, 2011, 2012) and industrial flow and market data from branch organization statistics and unofficial files (Brewster 2009, Polinares 2012a,b, Anglo-American plc. 2012, 2013, Russian Platinum 2014, Mineweb 2015, IPA 2015, Implats 2015, Stillwater Palladium 2015, Stillwater Mining Company 2010, 2011, 2012, Norilsk Nickel 2012, 2014, Sverdrup's personal notes).

Figure 4.1.11.1 shows recorded platinum group metal production to date (adapted from data downloaded from USGS 2014). Table 4.1.11.1 shows the approximate platinum fluxes in society. It shows the use of platinum in different sectors. Two sectors stand out for the physical size of consumption, the use of platinum, palladium and rhodium in car catalysts and the use of platinum in jewellery. There are further significant inconsistencies in the available published numbers that appear when they are all compiled together and compared against some of the unofficial numbers we have access to. The reasons for these inconsistencies are unknown, but possible explanations are uncertainties in the flux estimates, uncertainty about how large the accumulation in society really is and that some companies may be withholding or distorting numbers for reasons of business secrecy and competition.

Table 4.1.11.1. Approximate platinum fluxes in society. Extract=supply flow from mining, Recycle=recycling from society, Supply=supply to markets, sometimes called demand by market in other texts. Loss=losses, Accumulate=accumulation in society as stock-in-use and as scrap. Numbers in bold are numbers that can be reasonably well verified. Sources of the information are; Johnson Matthey (2015), Butler (2010, 2011, 2012), USGS (2014, 2015), Anglo-American (2012), International Platinum Association (2015), Nassar (2013, 2015), additional information was taken from Harald Sverdrup's personal notes from 1981-2015

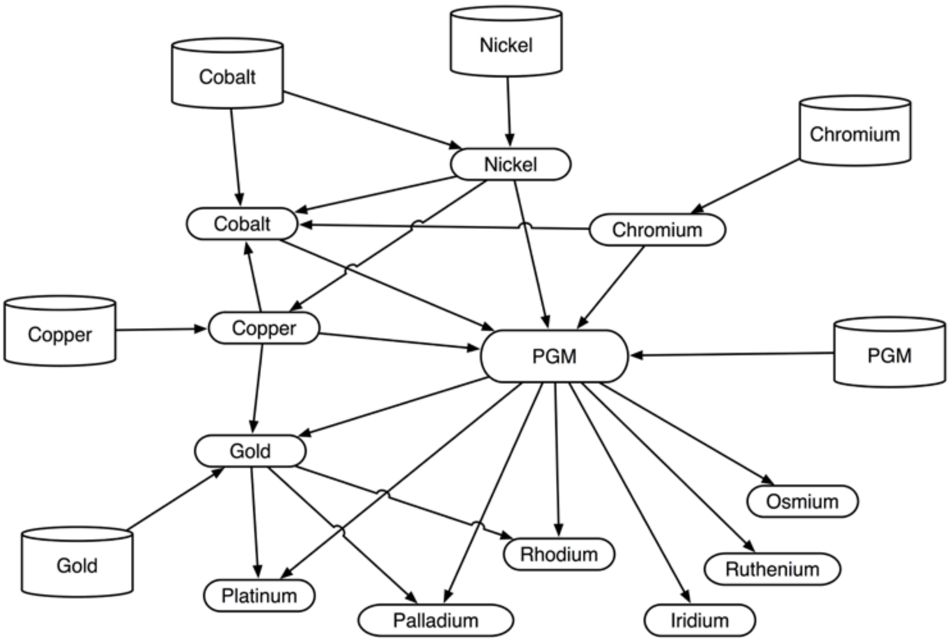
Sector	Use	Platinum, ton per year				
		Extract	Recycle	Supply	Accumulate	Loss
Chemical catalyst	Nitric acid	4	20	20	0	4
	Synthesis	1	3	8	0	1
	Technology	6	6	12	0	0
	Automotive	100	40	100	5	35
Jewellery	Vanity	61	24	85	2	1
Investment	Investment	1	1	2	1	0
Mechanical uses	Other	2	12	10	0	4
	Electronics	6	1	5	1	1
	Glass	6	6	7	0	5
	Dental	1	6	7	-1	1
Sum	All uses	188	119	257	8	52
Sector	**Use**	**Palladium, ton per year**				
		Extract	Recycle	Supply	Accumulate	Loss
Chemical catalyst	Nitric acid	11	11	16	0	6
	Synthesis	8	6	8	0	2
	Technology	0	3	3	0	0
	Automotive	158	88	216	-20	50
Jewellery	Ornamentation	6	6	12	-1	1
Investment	Investment	0	2	3	1	0

Mechanical uses	Other	0	0	0	-1	1
	Electronics	23	13	33	-11	4
	Glass	0	0	0	0	0
	Dental	4	12	16	0	4
Sum	All uses	214	141	307	-32	68
Sector	**Use**	**Rhodium, ton per year**				
		Extract	Recycle	Supply	Accumulate	Loss
Chemical catalyst	Nitric acid	2	6	8	0	1
	Synthesis	1	1	2	0	2
	Technology	0	1	1	0	
	Automotive	18	9	25	0	2
Jewellery	Ornamentation	-	-	-	-	-
Investment	Investment	0	1	1	0	0
Mechanical uses	Other	0	1	1	0	1
	Electronics	0.4	0	0.2		0.2
	Glass	0.6	1	1	0	0.6
	Dental	0	0	0	0	0
Sum	All uses	22.5	15	39.2	0	14
Sum all PGMs		424	275	603	-24	134

In Russia, the platinum group metal reserves have always been regarded as a state secret. In 2013 the platinum budget as shown in Table 4.1.11.1 runs a small net accumulation in society, palladium is possibly running a significant deficit, implying society stocks are decreasing and the rhodium budget seems to balance (Data were largely taken from Johnson Matthey 2015, Butler 2010, 2011, 2012). The numbers shown in the table imply that 65% of the mined amount or 46% of the amount supplied to the market is recycled. The amounts lost every year is 32% of the mined amount or 22% of the supplied amount, which is too much if the long term metal conservation goal is to have platinum available for the next 1,000 years. One way is to model the system with a set of interconnected mass balances, where also the accumulation at several points in the system can be accounted for, something that one cannot do in a simple table. Platinum group metals both originate from platinum group metal mines as in South Africa, but significant amounts come from nickel, manganese, chromium, copper and cobalt mining (Figure 4.1.11.2). Where the undiscovered reserves and resources will be found in the future we can only speculate about, but certain areas where the geology resembles the known deposits may be pointed out on all continents (Eckstrand and Hulberth 2007, Wellmer 2008, Vermeulen et al., 2013, Zientek et al., 2014).

A reserve is a known resource stock and with present market and technology, extractable amounts in location and amount. The term "Resource" is known and anticipated extractable amounts regardless of extraction cost. We have assessed the remaining extractable amounts, the ultimately recoverable reserves (URR) and the stocks-in-use, and done a preliminary assessment using burn-off-time and Hubbert's model in an earlier study (Ragnarsdottir et al. 2012). Although the large platinum mines in South Africa are called platinum mines, they are in reality poly-metallic mines where a number of valuable metals are extracted. Figure 4.1.11.2 shows the platinum group metals flow in general from deposits to metal.

Figure 4.1.11.2. Flow chart for platinum group metals extraction. The platinum metal group elements are produced from a number of sources as shown in the flow chart. Barrel=Resource stock, Oval=Extraction



Source: Own Figure, Lund University/University of Iceland

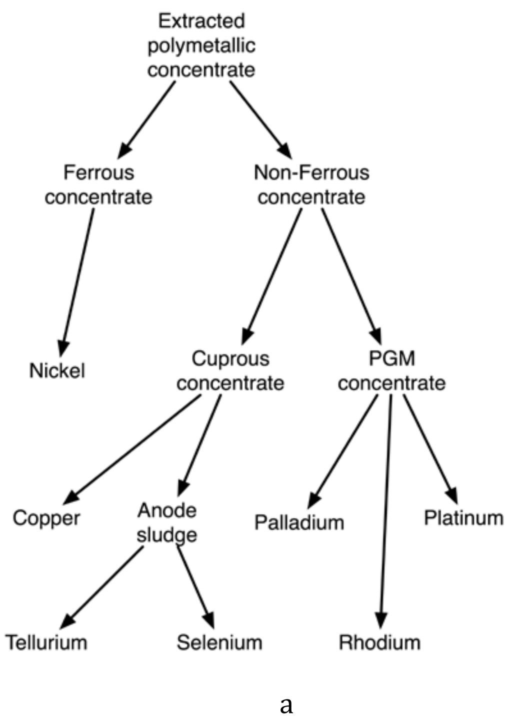
Figure 4.1.11.3 shows the flow chart for extracted metals at the American Stillwater Mining Corporation (Stillwater 2011) and the flow at Noril'sk Nikel, Russia, a major platinum and palladium producer. These flowcharts are typical of a poly-metallic platinum mine. All metals produced contribute to and are necessary for the profitability of the mine. More and more metals are sourced this way as key ore grades decrease, technology improves and metal prices increase. This flow chart is applicable to the South African mines as well any other platinum group metals mine, the only thing different there is that the platinum and palladium contributions are larger. There are dedicated platinum group metals mines, but in reality, the platinum metal group elements are produced from a number of sources as shown in the flow chart. The extractable amounts of platinum group metals from ores are graded according to extraction cost and metal content (Sweeneye 1992, Neumayer 2000, Tilton 2002, 2007, 2009, 2012).

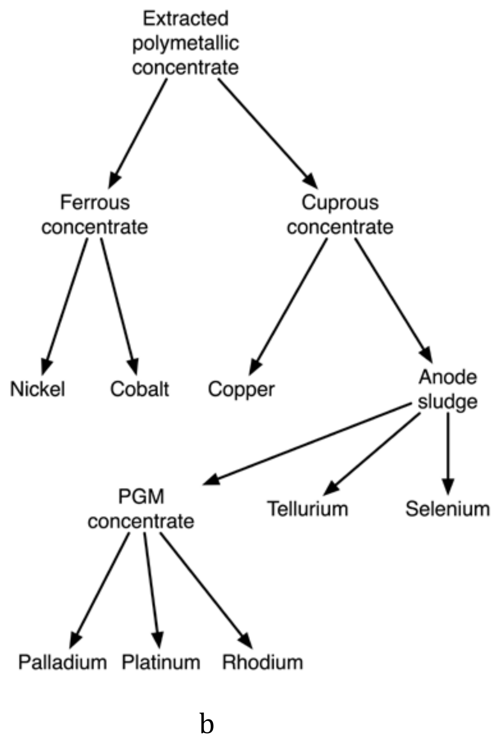
South African mines are based on platinum deposits, chrome deposits and manganese deposits, yielding also cobalt and nickel and substantial amounts of copper and gold. Russia, Canada and United States produce platinum group metals from mainly from copper and nickel ores. The Russian company Noril'sk Nickel produces 300,000 ton per year of nickel, 370,000 ton per year of copper, 2,400 ton per year of cobalt, 21 ton per year of platinum and 81 ton per year of palladium. The Noril'sk ore deposits reach down to about 2 km depth at Noril'sk, the Kola Peninsula ore deposits may be deeper. The economic revenue from the metals flow would in the case of Noril'sk Nickel be: Nickel: 3,750 mill $ per year, copper: 2,220 million $ per year, palladium: 2,025 million $ per year, platinum: 1,050 million $ per year and cobalt: 65 million $ per year, a total of 9,110 million $ per year (Noril'sk Nikel 2012, 2014, Platinum Group Metals 2015, Implats 2015, Russian Platinum 2014, Walters 2014).

Stillwater Mining Company, United States of America, is a poly-metallic mine; the extracted amounts of platinum is 3 ton per year, palladium is 8 ton per year, copper is 21,000 ton per year, nickel is 10,000 ton per year, silver is 20 ton per year, gold is 2 ton per year all contribute to the income of the Stillwater mine. The revenue contribution is estimated as: palladium 200 million $ per year, platinum 150 million $ per year, nickel 250 million $ per year, copper 252 million $ per year, silver 80 million $ per year, gold 80 million $ per year, with an annual revenue of about 1,000 million $ per year (British Geological Survey 2009, Noril'sk Nickel 2012, 2014, Stillwater Mining Company 2010, 2011, 2012, 2015, Walters 2014). The Stillwater deposit reach down to at least 3 km depth. As shown above, all metals produced contribute to and are at some mines also necessary for the profitability of the mines. More and more metals are sourced this way as key ore grades decrease, technology improves and prices increase. For the South African platinum mines, the flows charts are similar, but the value contribution from platinum group metals and gold play a more prominent role than at Stillwater and Noril'sk Nikel.

The Sudbury ore body in Canada (Xstrata, Vale Canada Ltd, Inco Ltd) is a major deposit mined for nickel, copper and platinum metals. It is an ancient (2.3 billion years; Walters 2014) meteoritic crater, and the mineable deposits reach down to at least 4 km or more. Possibly, it is an old meteorite strike. Large nickel-copper deposits have also been recently found in the Phillipines and Indonesia, and these show promise of significant contents of platinum group metals (Mudd 2012a, Elshkaki 2013, Walters 2014).

Figure 4.1.11.3. Diagram (a) shows the metal flow chart for the Stillwater Mining Corporation, United States (Stillwater 2011). Diagram (b) shows the flow chart for the metal production at Noril'sk Nickel in Russia. Similar carts apply for the mines in South Africa and Zimbabwe.





Source: Own Figure, Lund University/University of Iceland

4.1.11.4 Earlier modelling of the global platinum system

TIAX LLC (2003) produced a report on platinum sufficiency for fuel cells to the United States Department of Energy, based on an econometric model. The model used calibrated polynomials based on observed time-series of fluxes. The model involved no causalities or mass balances, and no systemic feedbacks. Saurat and Bringezu (2009a,b) made a Material Flow Analysis model simulation based on econometric principles for platinum group metals supply and evaluated it for potentials for price increases in the future. An earlier assessment was made by the authors used a simpler methodology (Ragnarsdottir et al. 2012, Sverdrup et al. 2013c) based on burn-off rates.

Knoeri et al., (2013) reports on the development of a dynamic agent-based model for platinum supply and the market, but no further results have been reported. Busch et al., (2014) used Material Flow Analysis models for a range of metals, including platinum in order to address risk of scarcity. Boudreau et al. (2009) developed a systems dynamics model for platinum supply and price, and related it to demand for fuel cell technology. The model uses one single platinum extractable reserve and a low URR estimate derived from the USGS (2008) estimate. However, the model uses a feedback system between market stock, supply, take from the market and platinum price to investigate the future price behaviour of the platinum price, similar to what we do here. The Boudreau model shows a dynamic price response. The model has several simplifications to make it small and easy to use. Alonso (2010) and Alonso et al. (2007a,b, 2009, 2012) developed a system dynamic model for platinum supply to the market and relates this to demands from electronics manufacturers. The actual structure of the Alonso model is graphically visible in her PhD thesis from MIT, where the causal loops of the full systems dynamics model are shown, but no further information has been published yet. The authors state that they have metal supply that depends on price, that the price vary with the difference between supply and demand and that demand goes down when the price goes up. Supply and demand is fed exogenously to the model. The metal price and changes in supply and demand are generated endogenously in the model. The actual price model appear to be similar to the one used in our study. They study the generic behaviour of the model, how it responds to disturbances in supply and demand. They do no attempt validation against field data.

None of the models above address the dynamics of the extractable amounts in the ore deposit and feedbacks to extraction costs or the actual dynamics of the mining operations (Graedel and Allenby 2003, Elshkaki 2013, Busch et al., 2014). Thus, the earlier models have some features that are necessary and useful for us, but none of the models are sufficient in describing the global platinum group metals system, they cannot simulate market prices and they do not have the sufficient detailed feedbacks between market dynamics, price and the physical flows.

4.1.11.5 Methods and theory

General methods

The methods are those of systems analysis and systems dynamics, combined with a review of the literature, complemented by unpublished sources from within the platinum group metals industry. Many of these information sources are sometimes inconsistent between them and we therefore make expert judgment on what we think are the most likely estimate. We use different types of methods in order to estimate the scarcity time horizon for platinum group metals (Ragnarsdottir et al. 2012, Sverdrup et al. 2013c):

1. Production peak flowing the prospection endowment peak can be used. Heinberg (2001), Ragnarsdottir et al. (2012) and Sverdrup et al. (2013) have shown that there is a systematic shift of 40±3 years between peak discovery of a natural resource and the production peak. The method is purely empirical but shows good consistency when tested on historic data.

2. System dynamic modelling was used to simulate the platinum group metals cycling system. The flow pathways and the causal chains and feedbacks loops in the system are mapped using systems analysis, and the resulting coupled differential equations were transferred to computer codes using the STELLA® modelling tool (Senge 1990, Sterman 2000, Sverdrup and Svensson 2004, Haraldsson and Sverdrup 2004, Haraldsson et al., 2007, Sverdrup et al. 2012, 2014a,b, 2015a,b).

We draw flow charts, quantifying mass flow through the system (Graedel and Allenby 2003), and causal loop diagrams for the feedback systems. The Hubbert's model was not used in this study. We assess PGM scarcity and supply sustainability. Then we need to define what it really is in operational terms. Our definitions of scarcity are:

1. Soft scarcity, where supply to the market do not match the supply from the market, leading to smaller amounts in the market and higher prices, which in turn reduce demand. This down-sized demand is the sign of soft scarcity.
2. Monetary scarcity, where the demand is downsized by the increased price, but where the actual take from the market gets cut to zero by lack of ability to pay. Economic crisis will be the result
3. Hard scarcity is where the supply cannot sufficiently cover the demand. There will be a demand gap, a difference between the adjusted demand and what really can be supplied. Failure of provision is the final result

The platinum group metals have at all times been in soft scarcity and during some periods in time, they have also been at or near physical scarcity (Hard scarcity).

Key assumptions and data sources

The platinum group metal market is affected by the fact that it is a free market within set physical limits. We apply the following fundamental assumptions in this study: The laws of thermodynamics are universally valid; the principles of mass and energy balance apply everywhere with no exception. We assume that the published data on platinum group metals reserves and resources in different kinds of ore bodies have the correct order of magnitude. We consider physical platinum group metal transactions and we ignore paper metal and derivatives trade in our model. We use a dynamic price model, based on demand and supply and the price arbitration that takes place at the metal trading floors (Gotthelf 2005, Sverdrup et al., 2012, 2014a,b, 2015a,b,c). Platinum group metal ore grade has been sinking during the last two decades (Glaister and Mudd, 2010, Mudd 2012a,b). The recorded known reserves seemed to peak at 34,000 ton between 2005 and 2012 for South African mine districts (Anglo-American 2012, 2013). However, we need to remember that reserves are the known and detected amounts, and that the real extractable amount may be significantly larger. With platinum group metals this is very much so. Table 4.1.11.1 and Table 4.1.11.2 show an overview of extracted amounts during certain periods and recoverable resources looking backwards and forwards in time, expressed in ton of platinum group metals. URR is to a certain extent a moving target, depending on where we put the cut-off in ore content, based on extraction cost traded off against the market price. However as ore grades go down, extraction yields also go down, and below a certain ore grade, the yields become so low that adding large very low grade nominal amounts do no longer significantly contribute to the effectively extractable reserves, and the value of URR does not increase any more. As the ore grade declines, more and more ore bearing rock must be mined in order to extract platinum group metals, and in the end it becomes prohibitively expensive to mine the metal. Declining yield and increasing energy cost caps the size of URR. During 1900-2010, ore grades for platinum group metals have gone down, on the average from around 7-10 g/ton rock at the beginning of the 20th century to a present grade of 2-5 g/ton and the ore grade trend is still downwards (Glaister and Mudd, 2010, Mudd 2012a,b, British Geological Survey 2009). Platinum group metals resources are either reported as 3E (Platinum, palladium and gold), the term 4E which implies platinum, palladium, rhodium and gold, and the term 6E imply platinum, palladium, rhodium, ruthenium, iridium and gold. In this study, we have excluded gold from all numbers.

Table 4.1.11.2. Overview of mined amounts of PGMs during past periods and extractable amounts looking backwards and forwards in time, expressed in ton of PGMs down to 2 km mining depth.

Item of balance	Metric ton platinum group metals.
Mined before 1900, Quiring (1962), the authors estimate	170
Mined 1900-2001 (USGS 2013 and Mudd 2012b)	16,900
Extractable reserve base left (USGS 2013)	68,000
Sum URR	85,070

Extractable PGM amounts from ores

We have attempted to estimate the future final extractable amounts of platinum group metals as this is an important input to the assessment. Earlier estimates for how much is extractable have normally been very conservative, only counting what is proven reserves. These estimates are used for short term financial planning and for informing shareholders in publicly listed companies. They are less suitable for long-term assessments of what is really there for the next centuries. We have included in our estimates, all resources that are technically possible to extract, and then graded them with respect to extraction costs (Tilton (2002, 2007, 2009, 2012). The available data is not always internally consistent, this has been left in the tables in order to reflect the inherent certainty/uncertainty in the data. Important assessments of the platinum group metals resources available for mining have been done by Cawthorn 1999, 2010, Mudd 2012b and Zientek et al., 2014). The South African Bushveld Igneous Complex is the main global source of platinum and can be expected to remain so for the foreseeable future (Cawthorn 1999, 2010, Zientek et al., 2014). Table 4.1.11.2 shows and overview of mined amounts of PGM during past periods and recoverable amounts looking backwards and forwards in time, expressed in ton of platinum group metal. The USGS (2008, 2013, 2015) estimates that there is about 63,000 ton left in known reserves in 2012, assuming mining to 2 km depth. By including this with other extractable amounts in poly-metallic ores, we estimate 68,000 ton platinum group metal is remaining in rock and if we add to that the amount already dug out, 17,000 ton platinum group metal, URR is 85,000 ton platinum group metal. However, this is the conventional wisdom, and new information may change this significantly (Zientek et al., 2014). Table 4.1.11.3 shows extractable amounts implied per country other than South Africa for 2014 as estimated by the authors. In this we include reserves, reserve base and technically extractable metal inferred from prospecting. The numbers on reserves and resources are not always readily accessible as some of the countries do not wish to reveal their extractable amounts estimates. Other and yet undiscovered extractable amounts are based wholly on our own estimates and there is nothing substantial published on it anywhere. The author's opinion is that the Canadian and American extractable PGM amounts along with "others", are somewhat understated by maybe as much as a factor of two or three. This is supported by more recent assessments such as that of Zientek et al., (2014).

Table 4.1.11.3. The World outside South Africa. Overview of smaller producers and extractable amounts platinum group metals other than those in South Africa, 2014 according to Butler (2009, 2010, 2011), who worked with the data of Cawthorn (1999, 2010) and Mudd (2012b), assuming 2 km maximum mining depth.

Region	Ore source of platinum group metals	Production, metric ton PGM per year	Metric ton platinum group metals				
			Known reserves 2012	Extracted to 2012	URR		
					Zientek et al., (2014)	Cawthorn (2010)	Mudd (2012)
Russia	Ni, Cu	200	1,600	1,600	12,200	3,200	12,600
USA	PGM, Ni, Cu	30	1,350	1,000	3,000	2,350	-
Canada	Ni	13	620	1,030	3,000	1,650	-
Zimbabwe	PGM	12	8,600	100	16,000	600	8,700
Others	Ni, Cu, PGM		1,900	500	14,500	2,400	4,100
Sum				4,203	48,700	10,200	25,400

Table 4.1.11.4 shows an overview of the different URR estimates of platinum group metals for South Africa as estimated by Cawthorn (1999, 2010) and Mudd (2012b), assuming a maximum mining depth of 2 km. During 2000-2015, new technology and new engineering resulted in the ability to mine down to below 5 km depth in South Africa. At present, this is only done for platinum group metals, gold (going towards 5 km depth) and copper-nickel deposits (going towards 3 km depth). Mudd (2012b) estimated the total South African resource to be about 65,000 ton platinum group metals, consistent with Cawthorn (2010). The total global resource to be about 90,000 ton platinum group metals, leading to an URR of 105,000 ton when the extracted amount to 2015 is included (Mudd 2012b, Zientek 2014).

Table 4.1.11.4. Estimation of total extractable platinum and palladium amounts based on field assessment (Cawthorn 1999, 2010, Mudd 2012b) down to 2 km extraction depth.

Item	Extractable amounts metric ton		
	Known	Hidden	Known and unknown
South Africa Platinum	6,233	29,260	35,494
South Africa Palladium	3,611	22,115	25,726
South Africa Rhodium	590	2,570	3,160
Sum South Africa	10,434	53,945	64,379
Mined 1900-2010	15,000	-	15,000
Platinum and palladium contained in nickel	3,400	5,000	8,400
Other undefined reserves	2,000	1,000	3,000
Sum	41,268	49,111	90,380

Table 4.1.11.5 shows the estimation of total extractable platinum and palladium based on a new assessment of available reserves and resources assuming a maximum mining depth of 5 km and assuming the rhodium content in the platinum to be the same all the way down (von Gruenewaldt 1973, 1977, Hulbert and von Gruenwaldt 1982, Cawthorn 1999, 2010, Eckstrand and Hulbert 2007, Alonso et al., 2009, 2010, 2012, Anglo-American Platinum plc 2012, Mudd 2012a,b, Geoscience Australia 2009, USGS 2013, Zimasset 2014, Harmony Mining 2014, Zientek et al., 2014, Implats 2007, 2015). The Bushweld Igneous Complex ore body is up to 9 km thick in places.

Table 4.1.11.5. South Africa. Extension of the Cawthorn (1999, 2010, Mudd 2012b, Zientek et al., 2014) South African reserve and resource estimates for platinum and palladium down to a technical mining depth of 5 km. The amounts are approximate. Amounts in metric ton platinum group metals.

Metal	Type	Mining depth					PGM sum
		0-2 km	2-3 km	3-4 km	4-5 km	0-5 km	
Platinum	Reserves	6,323	3,150	1,500	800	11,773	
	Resources	29,206	14,500	7,000	3,500	54,206	
	Sums	35,529	17,650	8,500	4,300	65,979	65,979
Palladium	Reserves	3,611	1,800	900	450	6,761	
	Resources	22,115	11,000	5,500	2,800	41,415	
	Sums	25,726	12,800	6,400	3,250	48,176	48,176
Rhodium	Reserves	560	290	140	70	1,060	
	Resources	2,600	1,300	600	300	5,400	
	Sums	3,160	1,590	740	370	6,460	6,460
PGM sum		64,415	32,040	15,640	7,920	120,616	120,615

Table 4.1.11.6 shows an extension of the Zimbabwean extractable amounts in the Great Dyke and extractable amounts estimates for platinum, palladium and rhodium down to a technical mining depth of 3 km assuming the rhodium content in the platinum to be the same all the way down (Implats 2007, Oberthür 2011, Oberthür et al., 2003a,b, 2013, Zimasset 2014, Zientek et al., 2014). The deposits are located in the Great Dyke, a deposit with significant amounts of nickel, copper, platinum group metals and gold reaching diagonally through the area of Zimbabwe. Implats (2007) estimate the resources to be 6,780 ton platinum group metals, and the reserves to be 1,370 ton for platinum group metals. Mudd (2012b), Zientek et al., (2014) estimates the total Zimbabwean platinum group metals known resource to about 7,700-8,200 ton platinum group metals. The amounts are approximate because of the uncertainties in estimating the resources at depth. Most of the platinum group metals are located at shallow depth, but the Great Dyke in Zimbabwe extends down to 3 km in limited extent (The cross-section is shaped like a triangular wedge pointing down, see Figure 4.1.11.4).

Table 4.1.11.6. Zimbabwe. Extension of the platinum group metal extractable amounts estimates down to a technical mining depth of 3 km and assuming the rhodium content in the platinum to be the same at all depths (Zimasset 2012, Mudd 2012b, Zientek et al., 2014). Amounts in metric ton platinum group metals.

Metal	Mining depth			
	0-1 km	1-2 km	2-3 km	0-3 km
Platinum	2,530	1,240	600	4,370
Palladium	2,304	1,050	500	3,854
Rhodium	330	110	60	500
Sum	5,164	2,400	1,160	8,724

Table 4.1.11.7 shows an overview of platinum group metals occurrence and stocks in 2013. In Table 4.1.11.7, we are extending the Butler (2009, 2010, 2011), Eckstrand and Hulbert (2007), Mudd (2012b) and Cawthorn (1999, 2010) estimates down to a technical mining depth of maximum 5 km for the South African, Russian, Canadian and American deposits, assuming the resources to be extractable provided the price is high enough. All the mines are poly-metallic and yield significant amounts of

nickel and copper. Data from Table 4.1.11.5 and Table 4.1.11.6 have been included. The data in Table 4.1.11.7 confirm how dependent the world is on South Africa for supply of platinum group metals. The annual production of platinum group metals is at present 470 ton per year.

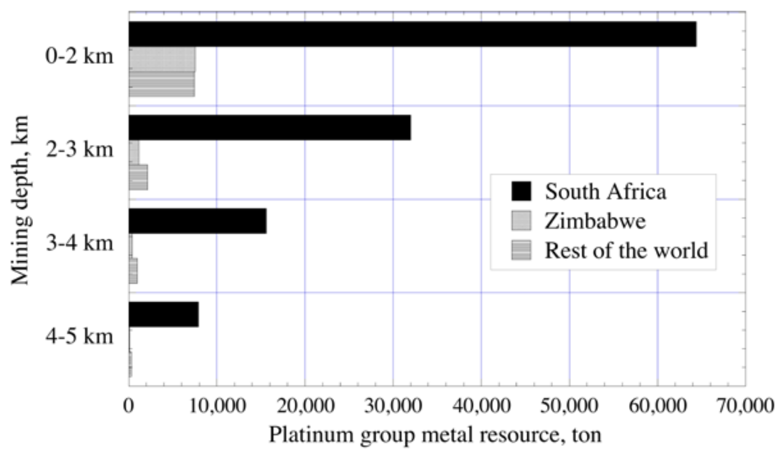
Finland has a relatively small area in the north with both reef and contact type of platinum group metals ores, and on the 90% certainty level, their total resource is at least 3,620 ton platinum group metals (Rasalainen et al., (2013). This makes Finland the 6th largest platinum group metals resource known globally. These deposits are on the poly-metallic type (copper-nickel-gold-silver- platinum group metals, sometimes cobalt and chromium), where the whole sum of metal harvest defines the ultimate profitability of the mining. This is included in the term "Other" in the tables. Similar amounts of platinum group metals may be hiding in the Indonesian and Phillipinese nickel deposits. Such considerations may suggest that the category "other" is underestimated, and that URR for platinum group metals is at least 220,000 ton, and maybe as much as almost 300,000 ton platinum group metals.

Table 4.1.11.7. Extending the resource estimates Levine and Wallace (2005), Butler (2009, 2010, 2011), Mudd (2012b) and Cawthorn (1999, 2010) with estimates ton platinum group metals down to a maximum mining depth of 5 km for both platinum group metals and nickel

Region	Ton platinum group metals (PGM), metric ton					
	Extractable 0-2 km depth	Extractable 2-4 km depth	Extractable 4-5 km depth	Extracted to 2012	Still hidden PGM resources	URR
South Africa	64,000	48,695	7,920	15,800	25,000	161,415
Zimbawe	7,364	1,360	0	100	7,800	16,524
Russia	3,600	1,020	0	1,600	5,600	12,820
USA	1,350	650	200	1,000	1,500	4,700
Canada	620	380	400	1,030	4,200	6,620
Others	1,900	1,100	600	500	10,000	14,100
Sum	78,844	53,205	9,120	20,030	54,100	216,179

Figure 4.1.11.4 shows the distribution of extractable PGM resources with mining depth using the available data. The figure suggests that delving deeper will be mitigated by diminishing returns, and that very little PGM is available below 3 km depth. This has both geological and technical reasons.

Figure 4.1.11.4. The distribution of extractable PGM resources with mining depth using the available data shown in Tables 4.1.11.5-4.1.11.10.



Source: Own Figure, Lund University/University of Iceland

Table 4.1.11.8 shows an analysis of the relative rarity of platinum as compared to other platinum group metals. The South African platinum group metal mines contain 86% of the reserves. The platinum group metal extraction is based on nickel-copper mines in Norilsk, Russia, in the Canadian Nickel mines in the Sudbury area, and recently discovered deposits in the Phillipines (Mudd 2012b, Eckstrand and Hulbert 2007), with significant extractable amounts of nickel-dominated ores containing platinum group metals (This accounts for 7% of the global extractable platinum group metals amounts). All the other mines hold 7% of the global extractable platinum group metals amounts). Doubling the platinum group metals resources outside South Africa would seem to be reasonable, as many areas have not been sufficiently prospected for platinum group metals, but that the geological preconditions for finding them are on a generalized level present. This would increase the global URR from about 220,000 ton (Table 4.1.11.7 and Table 4.1.11.9) to about 245,000 ton platinum group metals (Table 4.1.11.7 and Table 4.1.11.10). Industrial recovery rate relates to closed industrial cycles like catalysts for fertilizer, glass fiber production and similar uses. Total use recovery is the global systemic recovery rate and is estimated from the difference between the total market supply and the net extraction from geological sources.

Table 4.1.11.8. Overview of platinum group metals resources in 2013 down to 5 km mining depth and some additional data. Adapted after Mudd (2012b), Cawthorn (1999, 2010), Zientek 2014). All amounts shown are in metric ton, production and recoveries are in ton per year.

Metal	Pt	Pd	Rh	Ru	Ir	Os	PGM
Production, ton/yr	180	210	27	22	10	1	450
Extractable , ton	91,500	84,300	7,610	5,980	1,200	380	190,770
Cumulative mining, ton	8,330	9,240	1,300	950	150	50	20,030
URR, ton	99,830	93,540	8,910	6,930	1,350	430	210,800
Stock in society, ton	4,400	4,000	480	310	50	6	9,246
Industrial recovery	80%	70%	70%	50%	40%	40%	-
Total use recovery	65%	20%	10%	5%	5%	2%	-
Price: US $/troy ounce	1,100-1,650	500-700	1,100-2,200	85-100	1,100	450	-

Table 4.1.11.9 shows our classification of the platinum group metals extractable amounts into quality types, depending on ore grade for the year 2015. Amounts in ton platinum group metals as estimated based on data from the Table 4.1.11.3 to Table 4.1.11.9. An issue is how low large the remaining nickel resource really is. We have worked with a total URR of 185 million ton as our baseline estimate, giving a content of 46,100 ton platinum group metals (Corresponding to about 250 gram PGM per ton nickel or 0.025% in the nickel metal extracted, corresponding to PGM contents in the ore at 0.5-2 gram per ton ore). However, the range is 190-300 million ton for the nickel resource, with a probability of the higher estimate being correct. If the total nickel resource is 300 million ton, the PGM held in nickel would be about 75,500 ton PGM instead of 46,000 ton as suggested in Table 4.1.11.10. The extraction always occur from the reserves with the lowest extraction costs. When the price rises enough to cover higher extraction costs, the extraction will be initiated on lower ore qualities. The switch cost level is the median point on the curve where the model changes into start mining of the lower ore grade or from the low grade to the ultralow grade.

Table 4.1.11.9. Classification of the platinum group metals approximate extractable amounts into qualities depending on ore grade for the year 2015. The switch cost level is where the model starts mining a lower ore grade. The cost threshold for mining depth, is the additional cost of going to a deeper mining level. The amounts are approximate. To get URR, we need to add the amount mined to 2015 or about 20,000 ton. Amounts in metric ton platinum group metals.

Ore quality	Ore grade switch cost, $ per troy ounce	Cost threshold for mining depth, $ per troy ounce				
		300.-	600.-	1,000.-	1,600.-	-
		0-2 km	2-3 km	3-4 km	4-5 km	0-5 km
High	-	14,000	8,000	6,000	4,000	32,000
Low	2,800	18,000	12,000	10,000	6,000	46,000
Ultralow	3,800	28,000	26,000	22,000	16,000	92,000
Nickel-copper ore	-	13,000	6,000	5,000	4,000	28,000
Sum	-	71,400	37,400	29,000	19,000	193,600
Mined to now						20,000
URR						213,600

Table 4.1.11.10 shows the classification of the platinum group metals reserves into types of technically extractable amounts, and specified at different mining depths, all for the initiation of the PGM Model for the year 1900.

Table 4.1.11.10. PGM model input for potentially extractable amounts classified as hidden and known and distributed to depth categories at the beginning of the simulation in 1900. Amounts in metric ton PGM

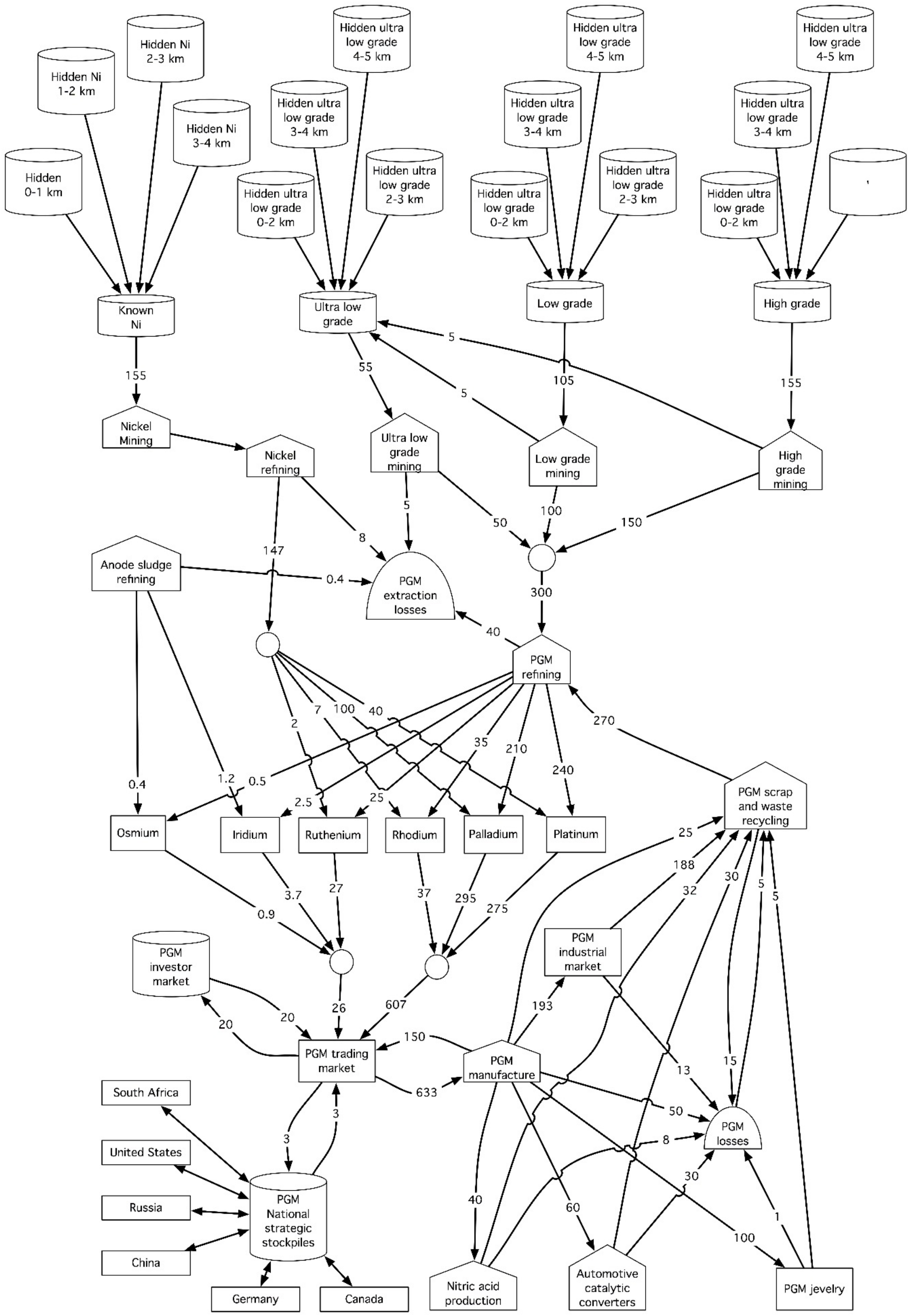
Ore quality	Ore grade PGM/ton	Hidden 0-2 km	Hidden 2-3 km	Hidden 3-4 km	Hidden 4-5 km	Sum all hidden 0-5 km	Known 1900 0-2 km	URR Sum
High	10-2.0	14,000	8,000	6,000	4,000	44,000	1,400	33,400
Low	2.0-0.5	18,000	12,000	10,000	6,000	46,000	500	46,500
Ultralow	0.5-0.2	28,000	26,000	22,000	16,000	92,000	0	92,000
Nickel	0.006%	16,000	12,000	10,000	8,000	46,000	100	46,100
Sum		77,000	48,000	46,000	39,000	210,600	6,400	217,000

4.1.11.6 The integrated dynamic PGM Model description

The model structure

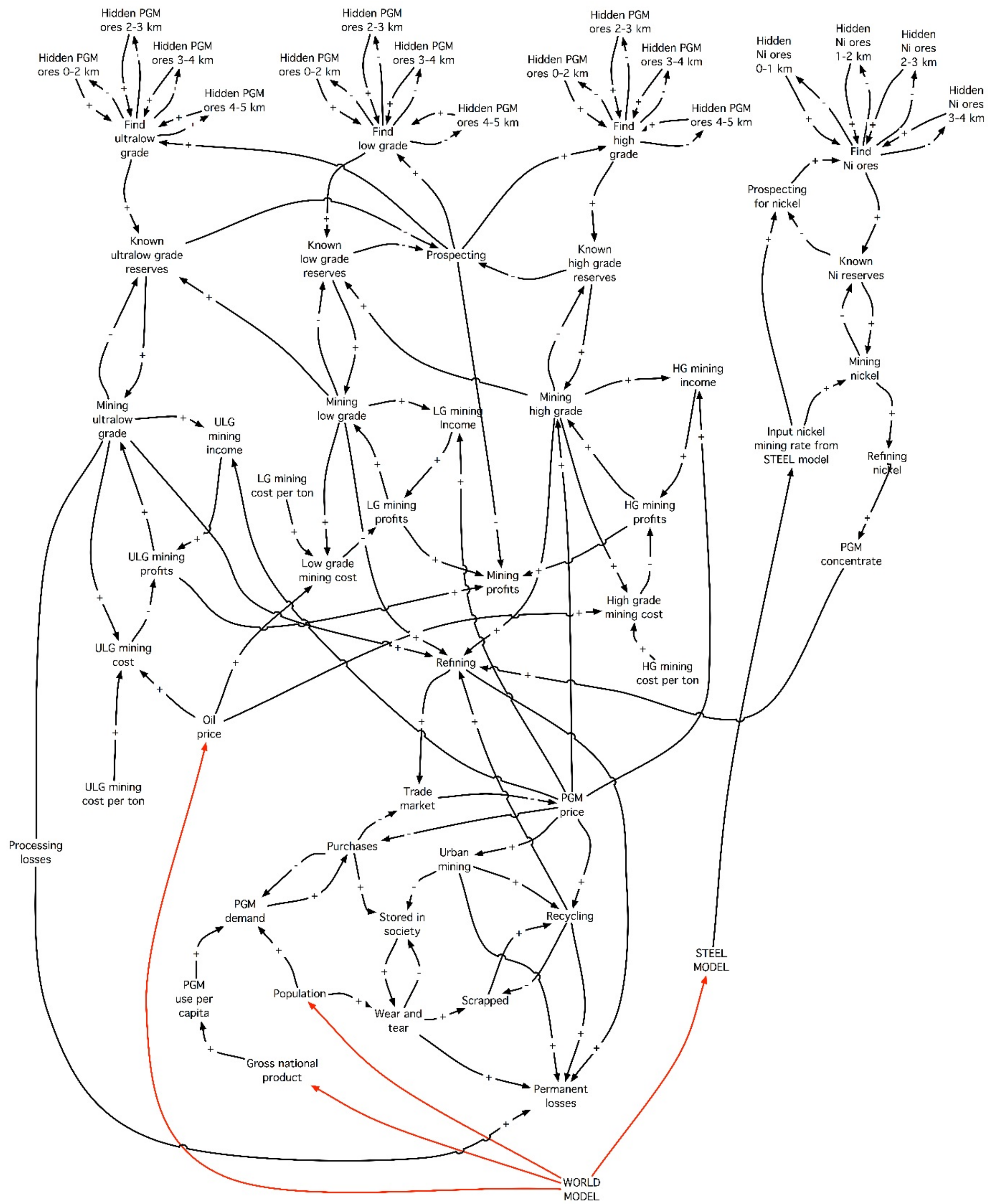
The PGM model used here follows the flow chart in Figure 4.1.11.5 and implements the causal loop diagram in Figure 4.1.11.6 and Figure 4.1.11.7. Figure 4.1.11.6 shows the causal loop diagram for the PGM model, connecting ore grades, mining activity, mining, profits and market price and how platinum group metals flow to different sectors. Figure 4.1.11.7 shows the causal loop diagram for the PGM model showing the mechanisms for mining towards greater depth. Mining at greater depths increase the extraction cost, demanding a higher platinum group metals price to be profitable. Profit is what drives mining at any depth and any ore grade. The mining is profit driven in the model, based on metal price and mined amount, minus costs. The price depends on the amount platinum group metals in the market, which in turn is determined by supply to the market by mining and recycling and sales from the market depending on price and demand. The mining operation cost have a limiting feedback on the mining rate. Known reserves do not promote extraction, but have a limiting effect when they decline. Prospecting leads to discoveries, which remove platinum group metals from unknown reserves to known reserves. Recycling works as mining of the stock of platinum group metals in society, and the causal loop is reinforcing and driven by price.

Figure 4.1.11.5. Flow chart for the platinum group metals market and turnover. The causal loop diagram in Figure 4.1.11.6 was used together with the flow chart here to create the STELLA model. The small numbers on the arrows are fluxes in ton per year.



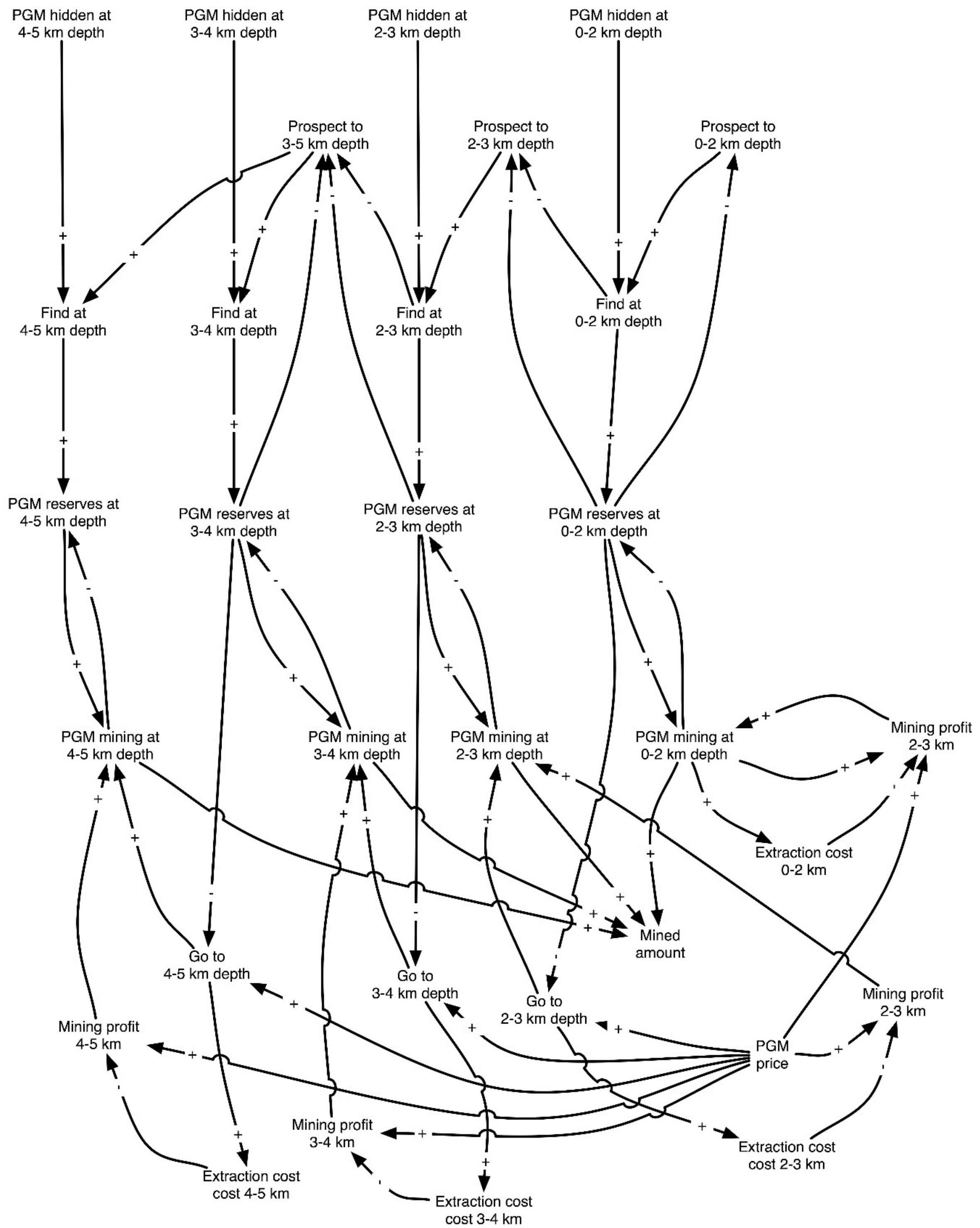
Source: Own Figure, Lund University/University of Iceland

Figure 4.1.11.6. Causal loop diagram for the platinum group metals model (PGM). The causal loop diagram here and in Figure 4.1.11.7 maps all the causal links in the system, and together with the flow chart in Figure 4.1.11.5, serves as a blueprint for building the model in the STELLA software.



Source: Own Figure, Lund University/University of Iceland

Figure 4.1.11.7. Causal loop diagram for the platinum group metals model (PGM) showing the mechanisms for mining towards greater depth.



Source: Own Figure, Lund University/University of Iceland

We employ the following 27 different reservoirs in the PGM-model in the metal mass balancing, and for setting up the system of coupled differential equations in the STELLA environment (See the supplementary material for the STELLA diagram and the feedback curves quantified):

1. Trading market amount of platinum group metals

2. Industrial platinum in
    a. Other uses, mainly industrial processes
    b. Platinum group metals catalysts in nitric acid factories
    c. Platinum group metals Refining plants
3. Automotive catalytic converters containing platinum group metals
4. Jewellery made in platinum group metals
5. Short term investors in platinum group metals
6. Different extractable amounts platinum group metals in the ore deposits:
    a. high grade platinum group metal extractable amounts
        i. Known 0-5 km
        ii. Hidden 0-2 km
        iii. Hidden 2-3 km
        iv. Hidden 3-4 km
        v. Hidden 4-5 km
    b. low grade platinum group metal extractable amounts
        i. Known 0-5 km
        ii. Hidden 0-2 km
        iii. Hidden 2-3 km
        iv. Hidden 3-4 km
        v. Hidden 4-5 km
    c. ultra-low platinum group metal grade extractable amounts
        i. Known 0-5 km
        ii. Hidden 0-2 km
        iii. Hidden 2-3 km
        iv. Hidden 3-4 km
        v. Hidden 4-5 km
    d. platinum group metal contained in nickel extractable amounts
        i. Known 0-5 km
        ii. Hidden 0-1 km
        iii. Hidden 1-2 km
        iv. Hidden 2-3 km
        v. Hidden 3-4 km

The produced platinum group metals is divided up according to source ore, either nickel-copper based or platinum group metals-based, into the six metals making up the platinum group metals; platinum, palladium, rhodium, ruthenium, iridium and osmium. The differential equations in the PGM-model are solved with a 4-step Runge-Kutta numerical integration routine with a time-step of 0.1 year for 1900-2400. The PGM model has several sectors reflecting different activities, as can be seen from Figure 4.1.11.7and Figure 4.1.11.8. These are:

1. The mining and refining sector, involving the platinum mines and the nickel mines supplying platinum group metals
2. The private consumption sector involving jewellery and investment stocking of physical metal
3. The fertilizer industry, using platinum group metals for catalyst
4. The industrial sector, involving other industrial catalysts and other use of platinum group metals
5. The market sector, involving the trade market and the setting of price
6. A sector summing up all systemic irreversible losses of platinum group metals

In the mining sector we have represented the different mining depths, but we only enter the deeper levels when the technology is ready. The ore basis in the model is based on either a platinum group metal ore (Platinum group metals ore) which have contents of platinum, palladium, rhodium, ruthenium, iridium, often bordering on chromite mineralization and where platinum is the dominant metal followed by somewhat less palladium and trace amounts of rhodium, ruthenium, iridium and osmium. The other type of ore is nickel-copper-dominated poly-metallic ores with contents of platinum, palladium, rhodium, ruthenium, iridium and osmium; mostly palladium, platinum and rhodium, and trace amounts of the other. The mining rate may be modelled in several ways: The first class is where the geometry and mass of the ore body plays a role:

1. Related to the geometry of mining:
    a. The ore body is penetrated from all directions all at once, or it is a liquid that flows towards the extraction points. Then the rate may depend on the whole volume, and the mining will be fast. The rate of mining will be proportional to the mass of the ore body for a fixed set of infrastructure, thus n=1
    b. The ore body is extracted from the outside and on the surface of the ore body, then the mining rate will be dependent on the surface of the ore bod. This also applies if the ore body is tunnelled into. The rate proportional to the mass of the ore body to the power of 2/3, thus n=0.67
    c. The ore body is approached from one direction only and the ore is extracted along a constant surface from through the ore body. Open pit mining proceeds like this. Then the rate of mining will be proportional to the length of the ore body. That is equivalent to proportional to the mass of the ore body to the power if 1/3, thus n=0.33
2. Related to the state of the technology
    a. The technology and the financial capacity determines the mining rate and the ore body is always sufficient to supply anything attempted to be extracted.
    b. The mining rate is related to the technical and financial capacity and limited by the access to the ore body as outlined under items 1a-c above.

This yields the following equation:

$$rmining = k * mn \qquad (4.1.11.1)$$

where r is the rate of mining, k is the rate coefficient and m is the mass of the ore body, and n is the mining order. We have chosen to set this at n=2/3 consistent with alternative (b) above. The rate coefficient is modified with ore extraction cost and ore grade. In case of the platinum mining, we have an approach where the mining occurs from the top of the reserve and down, which would suggest equation 2 or 3. And the ore body is determined by the extractions (rmining) and prospecting (rdiscovery):

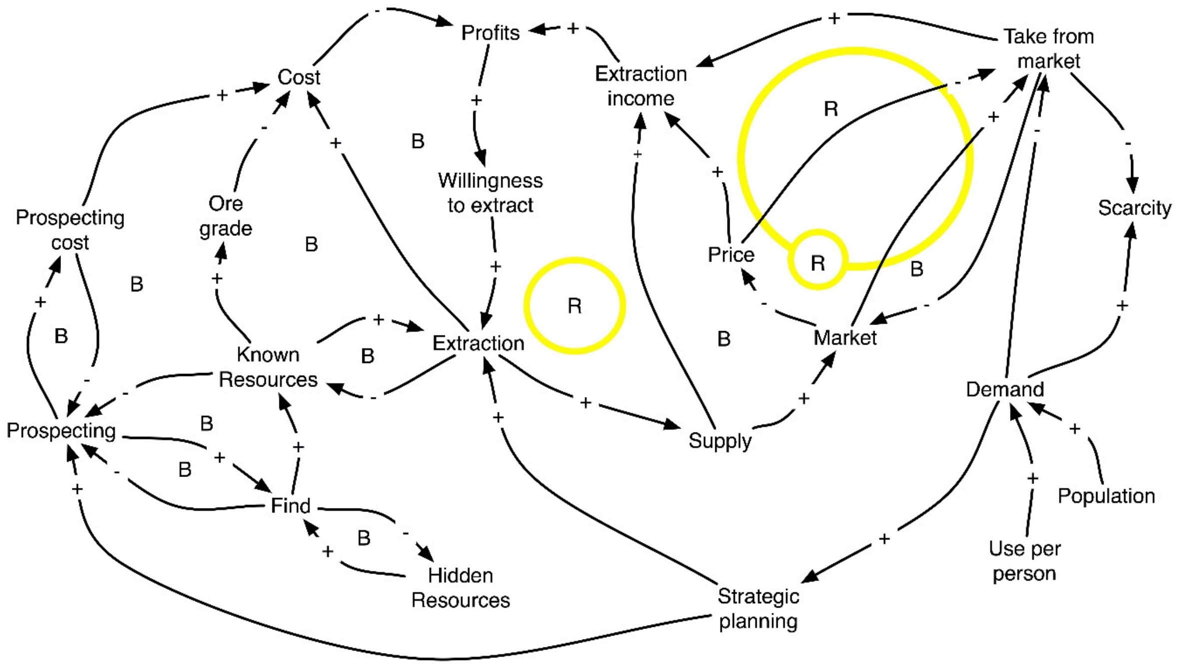
$$\frac{dm}{dt} = r_{mining} + r_{discovery} \qquad (4.1.11.2)$$

The discovery is a function of how much prospecting we do and how much there is left to find. The amount of hidden amount (mh) decreases with the rate of discovery:

$$\frac{dm_{hidden}}{dt} = -r_{discovery} = -r_{prospecting} * m_{hidden} \qquad (4.1.11.3)$$

For the PGM-model, the long-term average market price is an important feedback parameter for production, and together with the production cost, it determines how low an ore grade can be extracted (Figure 4.1.11.8). A sensitivity analysis, suggests that the choice of mining model does not have a large impact. Because of delays and inertia in the system of production, short-term variations in metal prices or operation costs do not affect the actual mining rate as much as the long term level in operations costs and metal market price. Thus, the price signal is averaged over a running 1.5 year period, before it is used as feedback for the mining rate.

Figure 4.1.11.8. Price mechanism as implemented in the PGM model is shown as a causal loop diagram. The usual terminology of supply and demand is not used. It is based on personal observations in the New York and London metal markets for the platinum group metal, silver and gold market under many years.



Source: Own Figure, Lund University/University of Iceland

The price mechanism

The price mechanism used in the model is based on observations in New York and London in the platinum, palladium and gold metal trading market. This is based on personal observations on the trading floors at the markets by the authors and shown in Figure 4.1.11.8. The reinforcing loops in the system have been shown with yellow rings. The usual terminology of supply and demand is not used as they are compound terms and thus inaccurate and often misunderstood. At the metal market, the sellers meet the buyers and present their supplied lots at a seller price. The buyers present their purchase wishes and their buyer's price offers. The prices and lots are adjusted when there is a discrepancy. The seller offers more at a lower price or less at a higher price; the buyers do likewise. When there is a match, the price is set. One of the authors, Sverdrup, has 10 years of personal experience from a major precious metal company in Norway, trading platinum, palladium, gold and silver at the global metal exchanges. The price model reflects this experience, and has some similarity to the market model developed by Alonso et al., (2007a,b, 2012) and Alonso (2010). System demand comes from consumption per person and the number of consuming persons. The demand come through the systems of manufacturing and supply of consumer goods and provision of services. The price curve relating market price to Figure 4.1.11.8 also shows the dynamics of prospecting and transfer of resources from "hidden" to "known". As known resources decline, this will drive the urge to search for more (known). More prospecting will lead to finding more as long as there is "hidden" to be found. Finding less, induce us to prospect more, but as prospecting costs rise, we may eventually stop. This also implies the following; the peak in finding comes before the peak in extraction, and the peak in prospecting comes after the peak in extraction. The peak in supply comes after the peak in extraction. Peak in prospecting costs is before peak in extraction. During the period 1958 (or earlier) to at least 1979, there was an International platinum group metal trading cartel. The platinum cartel was composed of some of the major platinum trading companies and some of the large primary producers. It allowed them to keep control over the market in a way similar to the monopoly DeBeers once had on diamonds. In the 1980's the cartel lost their grip on the platinum market, and it appears to the outsider as if it largely fell apart as it became redundant.

Recycling in the model

Recycling is an important source of platinum group metals and the degree of recycling is related to the market price. Higher price gives more recycling, lower price gives lower. The is also a cultural factor in recycling, where the awareness of the high value of platinum group metals make people less likely to throw it away. Of all commodities, this is most evident with gold. It also implies that we have two ways to affect the rate of recycling; Firstly price, and secondly cultural behaviour and knowledge.

The resources and reserves in the model

South Africa has by far the largest proportion of the global known platinum group metals reserve and resources in the world. There are several reasons for this. One reason is the special geological conditions that lead to the creation of the Bushweld Igneous Complex, also known as the Witwatersrand. Another reason is that the deposit is thick, at least 9 km. The South African mining industry has been able to develop technology to mine to at least 4 km depth and hoping to extend this to 5 km in the next future. Table 4.1.11.10 shows the PGM model input for extractable amounts classified as hidden and known and distributed to depth categories at the beginning of the simulation in 1900. During 1960-1980, mines were driven down to 2 km depth, where the rock surface has a surface temperature between 40-50oC. From 1980 to 2000, the mines reached 3 km depth. In the time period from 2000 to 2015, the mines are being extended and will reach production at 4 km depth, and to 5 km by 2020, with all levels in operation by 2020. At below 3 km depth, the rock surface temperature reaches 50-60oC. Below 4.5 km depth, this can be expected to reach 70-80oC (Zientek et al., 2014). That demands more protective gear, more automatization, demands more pumping energy and more installations to secure cooling for the work crews down in the mine (Jochens 1980, Zientek et al., 2014). The realm below 5 km will probably not be reachable for humans in the near future, and whether these levels can ever be accessed will depend on how well remote controlled robotization will succeed. If workers can be convinced to work under such unpleasant conditions remains untested. At these depths, handling the heat is both technically complicated and expensive. Virgin rock temperature of 75oC is considered as the technically feasibility limit by AngloAmerican Platinum pty (Zientek et al., 2014), corresponding to about 5 km depth. The mountain pressure at depths below 2 km depth is enormous, and rock burst caused the mountain pressure is common, representing a significant risk for the workers. A rock burst occurs when the mountain pressure cracks the rock and an explosion-like jet of rock is ejected into the mineshaft, killing anybody standing in the way. The South African Elandsrand platinum and gold mine reaches down to 3.6 km depth in 2015, with plans of going to at least 4 km, possibly more. Other gold and platinum mining companies are considering advancing further down too.

The South African Northland gold mine reaches down to 4.0 km depth, the TauTona gold mine reaches to 3.9 km depth, and the Mponeng gold mine to reaches 4.1 km depth, proving that these are feasible mining depths. This will be sustained provided the metal extracted is valuable enough, the ore grade sufficiently good and the country stable enough to provide the sophisticated infrastructural support required. The TauTona gold mine operates at an ore grade of 7.5 g per ton rock and the extraction cost is about 900 $ per troy ounce gold recovered. That implies that at the present gold price of 1,200 $ troy ounce, ore grades below 6.8 gram per ton cannot be mined. Noril'sk Nickel in Russia operated mines to 2.1 km depth in 2015. In Sudbury, Ontario, Canada area, the copper-nickel mines reach down to about 3.5 km depth at present. In terms of mining, these mines are related to the South African mines in the sense that the metals extracted are in demand and valuable. Both Noril'sk Nickel and the Mining Companies in Sudbury mine copper, gold, silver, nickel and platinum group metals. Both will have the potential to delve below 4 km if they are well managed (Kunilov 1994).

4.1.11.7 Results

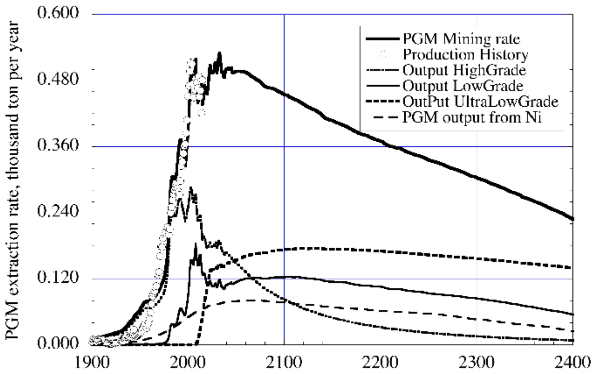
Peak prospecting estimate

The timing of platinum group metals peak extraction and peak supply was taken from the model outputs. The platinum group metals discovery peaked in the period 1980-1985 according to available data records, and prospecting yields have declined ever since. This points to a peak in platinum group metals production in 2020-2025, when we add 40±3 years to the peak discovery years. This appears to be consistent with the observations and the model results described below.

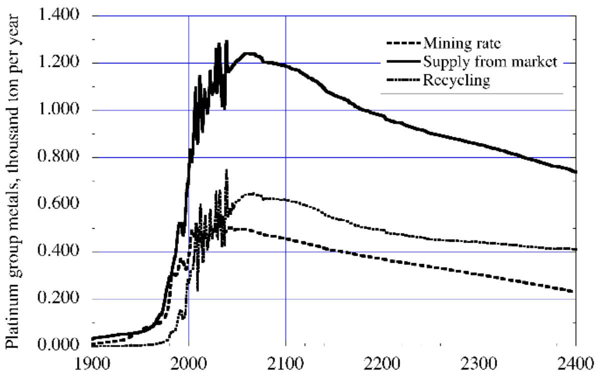
Dynamic modelling

Figure 4.1.11.9a shows the mining rate results from the PGM model during the period 1900-2300. Figure 4.1.11.9b shows the mining rate compared to the recycling and the market supply to society. The supply to society is substantially larger than the mining rate. The turnover of the platinum stocks-in-society is relatively fast (a few months to years for industrial use, for jewellery 40 years), and the stock-in-society is small, thus supply to society follows the extraction pattern with time. The peak production behaviour in the mining rate is not very pronounced. The mining rate is predicted to stay nearly flat from about 2010 to 2060, and then a gradual decline sets in. This is consistent with the data, that the mining production has apparently reached a maximum level and now move sideways in time-charts. South Africa seems to plan for sustained production levels, but not really for any major increase in platinum group metals output. The approach taken is to go down towards deeper mining.

Figure 4.1.11.9. Diagram (a) Mining rate of platinum group metals. The fit to the observations is r2=0.93 for the mining rate. Diagram (b): The mining rate was compared to the trade market output and recycling as modelled with the PGM model. Amounts shown are in thousand ton per year.



a

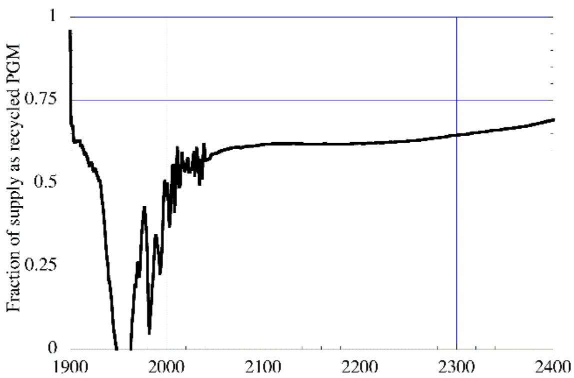


b

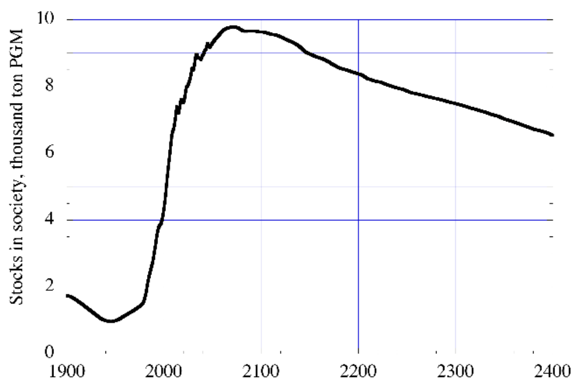
Source: Own Figure, Lund University/University of Iceland

Figure 4.1.11.10a shows the platinum group metals recycling rate, as the ratio between recycled amount to the total supply. After 2050 recycling of landfills and scavenging from the stock in society becomes an important means of platinum group metals supply. Figure 4.1.11.10b shows the stock-in-use in society over time. The low rates until 1960 are partially caused by an accumulation larger than recycling in society. The biggest contributor to low recycling efficiency after 2000 is the relatively low recycling of automotive catalysts. Table 4.1.11.1 discussed earlier indicate the same low systemic recycling rate as the model. Jewellery has a systemic delay of about half a century and there is substantial accumulation in society over time.

Figure 4.1.11.10. Diagram (a) shows The diagram (b) shows the platinum group metal recycling rate, defined as ratio between total supply minus mining to the total supply. The ratio is very variable over time because of the effect of transient accumulation in the system. Diagram (b) shows the total stock in society.



a

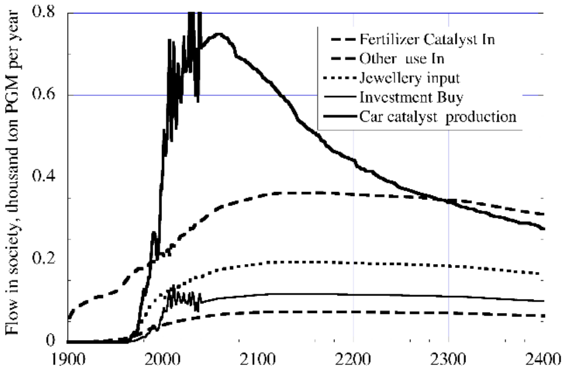


b

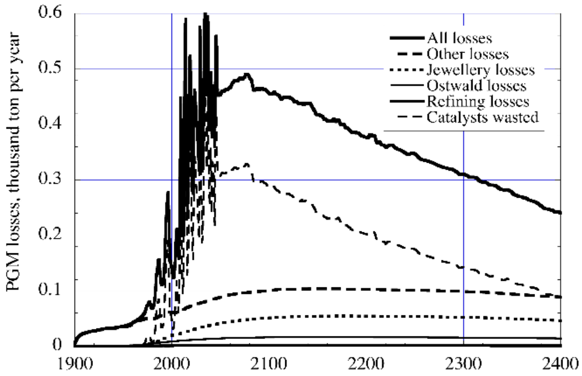
Source: Own Figure, Lund University/University of Iceland

Figure 4.1.11.11a shows the simulation of the platinum group metals flow of platinum group metals in society, as how much goes to car catalysts, fertilizer factories, industrial use and as jewellery during the period 1900-2300. The peak production is estimated to occur around 2055, peak supply to the market is about the same 2055-2060. Figure 4.1.11.11b shows the losses from the system. Car catalysts use the largest amounts platinum group metals, and the poor recycling of the automotive catalytic converters contribute the largest losses in the platinum group metal system. The two most significant uses are car catalyst and industrial use excluding nitric acid and cyanide production (Ostwald and Andrussow processes). The stock in society reaches a maximum of 9,800 ton in 2100, up from about 8,000 ton in 2015 according to the simulations. This corresponds to about 8-10 years supply to the market. The present level of losses are not sustainable from a point of platinum group metals conservation and needs to be mitigated, if not, it will be a contributor to let the society's stock of platinum group metals run out. The sustainability of the platinum supply hangs on if these losses can be significantly reduced. The losses from car catalytic converters are by far the largest.

Figure 4.1.11.11. Diagram (a) shows the platinum group metals flows in society. Diagram (b) shows the platinum group metals losses from the different sectors.



a

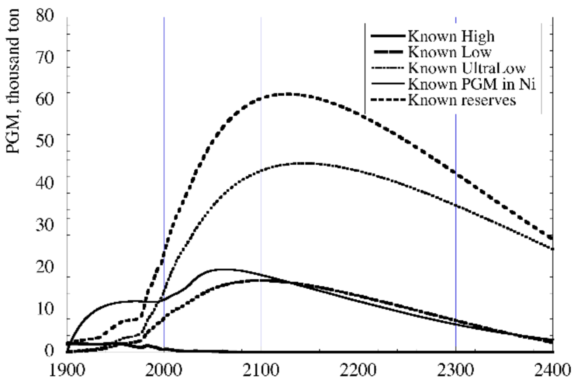


b

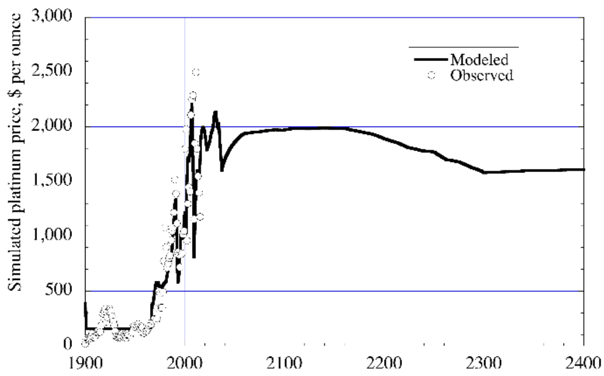
Source: Own Figure, Lund University/University of Iceland

Figure 4.1.11.12a shows the known extractable amounts during the period 1900-2200 including mining at all depths. Figure 4.1.11.12b shows the platinum market price in dollars per troy ounce, value-adjusted to 1998. The dots are observed values for the market price (USGS 2015). The price is predicted to stay volatile in the 1,100 to 1,800 $/oz range until about 2040, after which it will stabilize around the 1,200 $/oz level. The running average price is used as feedback variable to the mining rate. The reason for this is that it takes time to change the mining rate in the field. The estimate of extractable platinum group metal amounts coincide with the observations related earlier in this study.

Figure 4.1.11.12. Diagram (a) shows the known extractable amounts in ore deposits. (b) The market price of platinum, in dollars per troy ounce, value-adjusted to 1998. The dots show the historically recorded market price.



a

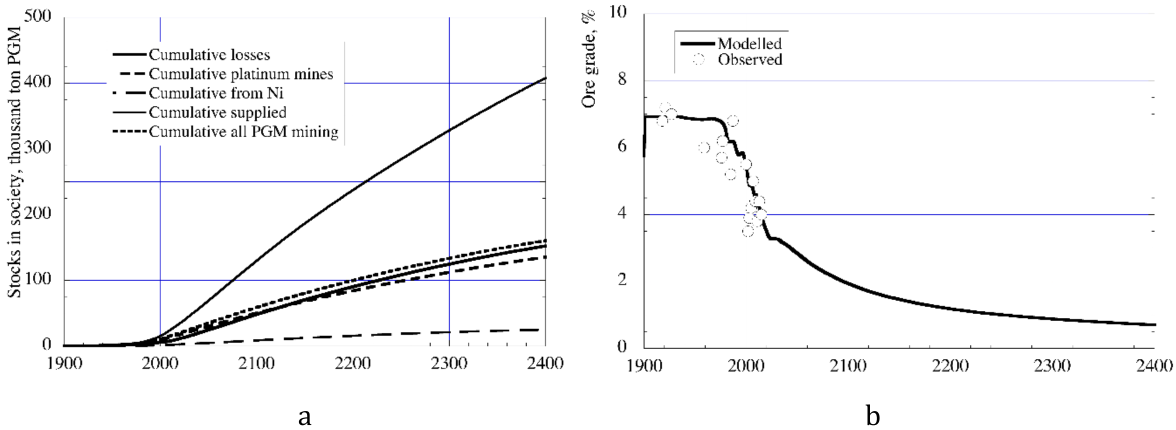


b

Source: Own Figure, Lund University/University of Iceland

Figure 4.1.11.13a shows the cumulative extractable amounts of PGM, as well as cumulative losses and mining. The ratio of total supplied to mined is about 2.6. It implies that every kg of platinum group metal is used 2.6 times before it is lost. Considering that platinum and palladium are scarce precious metals with little or no substitutability, the high loss rates not long term sustainable. Figure 4.1.11.13b shows the observed ore grade and the simulated ore grade with time. The ore grade decline steadily in the years after 1980, down to an average level of 1.2-0.6 gram PGM per ton rock. The stratification of the URR into different ore grades as shown in Table 4.1.11.10 is required to give this good a fit to the available data. The observed ore grades were adapted from Mudd (2010, 2012a,b) as well as different corporate annual reports. A decreasing ore grade increases the amounts of rock that must be extracted from the mine, to be milled and processed. At an average ore grade of 1 gram platinum group metals per ton rock in the future, the rock volume will be four times larger that today, increasing the production cost correspondingly. A simultaneous increase in oil prices will further amplify this price increase. The predicted ore grade, correlates favourably with the observed ore grade. In general, the simulations show that too much of the platinum group metals is lost to diffusive and non-recoverable losses on global level.

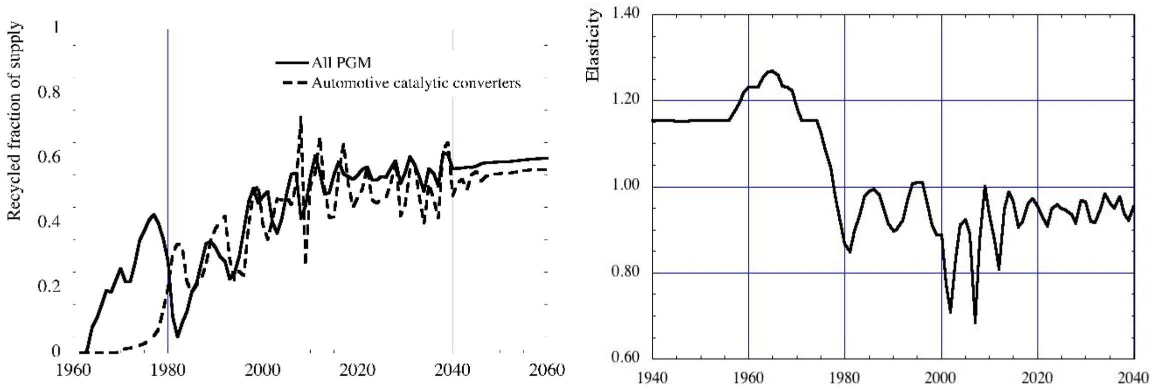
Figure 4.1.11.13. Diagram (a) shows the development of known extractable PGM amounts, cumulative losses, cumulative mining and total extractable amounts of platinum group metal in nickel ore. Amounts are shown in thousand ton platinum group metals. Diagram (b) shows the simulation of the ore grade over the period 1900 to 2400, using South African data for validation. Ore grade is gram platinum group metals per ton rock excavated.



Source: Own Figure, Lund University/University of Iceland

Figure 4.1.11.14a shows the overall recycling rate compared to the specific recycling rate for automotive catalytic converters. The model suggests that the automotive catalytic converter recycling of platinum was about 35% in 2014. In UNEP (2009, 2011a,b,c, 2013a,b,d) it is suggested that recycling of automotive catalytic converters is in the range of 25-40%. Figure 4.1.11.14b shows the price feedback on demand and supply (elasticity). When the value drops below 1, the demand was reduced because the price went up, at the same time as the supply went up from market stocks. When the line is above 1, it implies an oversupply with lower prices. When the line is below 1 it implies the supply is not meeting the demand, the demand that is reduced by higher price, the higher price in turn being caused by the scarcity. The mining rate does not go up instantly the same way, as there is a 1.5 year delay in scaling up production built into the model.

Figure 4.1.11.14. Diagram (a) shows the overall recycling rate and the specific recycling rate for automotive catalytic converters as fraction of total supply and total flux to catalytic converter production. Diagram (b) shows the price feedback on demand and supply (elasticity) in the PGM model.



a b

Source: Own Figure, Lund University/University of Iceland

4.1.11.8 Model tests

All models need to be tested in order to assess the quality of later predictions. Only when they reasonably reconstruct the past, may we conclude that they will make useful predictions for the future. Here we perform a test both for the integrated dynamic PGM-model. We have tested the model on three aspects:

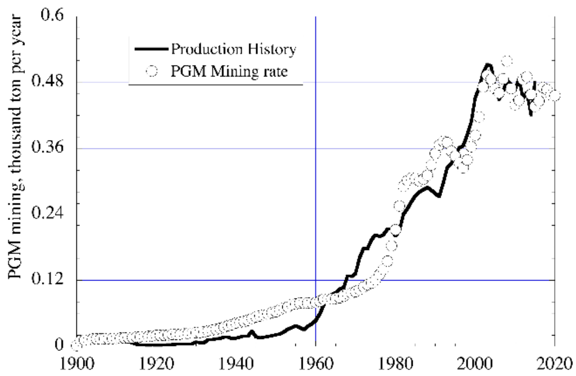
1. Mining rate: Figures 4.1.11.15a-b show a comparison of the modeled past reconstruction 1900-2015 and observed values of platinum group metals production. The fit to the observations is r2=0.93 for mining rate.

2. Platinum market price: Figures 4.1.11.15c-d show tests of the model on predicted versus the observed price (USGS 2015). The fit to the observations is r2=0.80 for the reconstruction of historical platinum price. For reconstruction of the historical price development, the general trend is reproduced, but the detailed and short-term fluctuations in price are not reproduced. This accuracy in price prediction is sufficient for the internal feedback on mining and recycling in the model, and thus important for the overall model performance. The supply of platinum has a very small society buffer, and depends to a large degree on the supply form mining. The price increase is mostly due to reductions in ore grade and extraction costs.

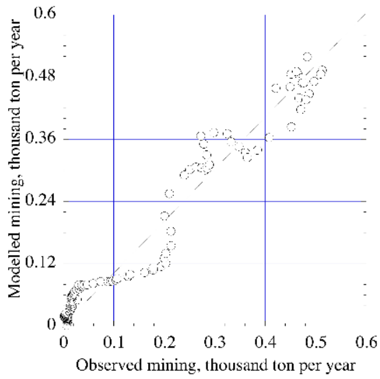
3. Ore grade: Figures 4.1.11.15e-f show a test of the PGM-model simulated ore grade versus the observed data for ore grade (Data compiled from Mudd 2010, 2012a,b). The correlation between observed ore grade data and simulated values was r2=0.77. The test on ore grade is a test on whether we succeeded in distribution the extractable amounts to the right depths and correctly among high, low, ultralow and extra-ultralow ore grades.

We conclude that the validation of the PGM-model was successful in the test, and we can use the model for long-term production level predictions with some confidence. The stock above ground (market, refining, stock-in-use, known scrap, catalytic converters) was estimated to 7,900 ton in 2014, consistent with the value suggested (9,900 ton). It is important for a model user to remember that while some parameters can be robustly set for the future (finite reserves, mass balance compliance, limiting efficiencies), other fluxes and stocks are almost impossible to set at a verifiable size. Then model simulation outputs may actually be the most consistent and accurate way to get an estimate.

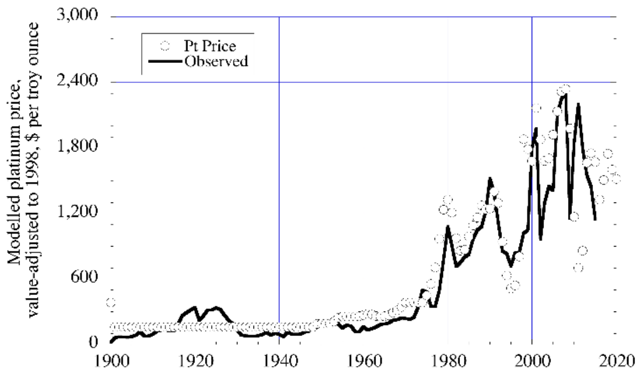
Figure 4.1.11.15. The simulation of the platinum group metals mining rate with time is shown in diagram (a). The fit to the observations is r2=0.93 for the mining rate (Diagram b). The simulation of the platinum price with time is shown in diagram (c) as compared to the observed. The fit to the observations is r2=0.8 (d). Diagram (e) shows the simulated ore grade compared to the observed. Diagram (f) shows the plot of observed versus modelled ore grade. The fit to the observations is r2=0.77



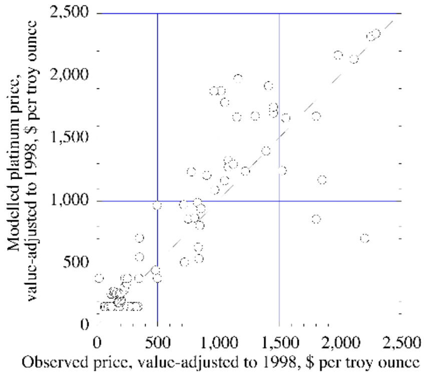
a



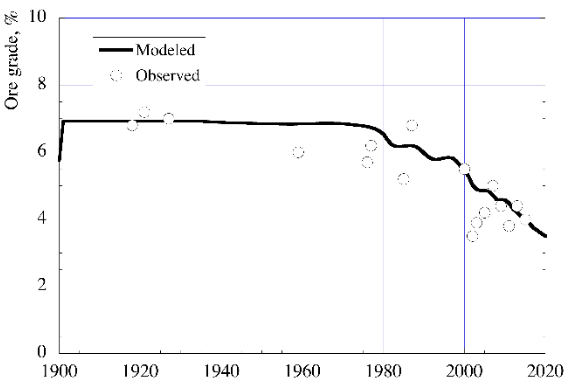
b



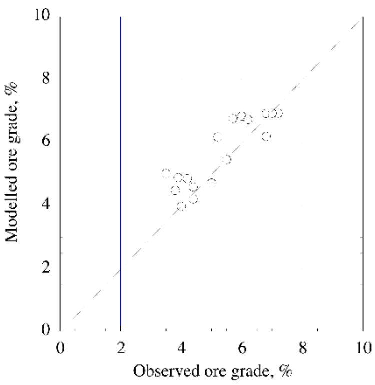
c



d



e



f

Source: Own Figure, Lund University/University of Iceland

4.1.11.9 Discussion

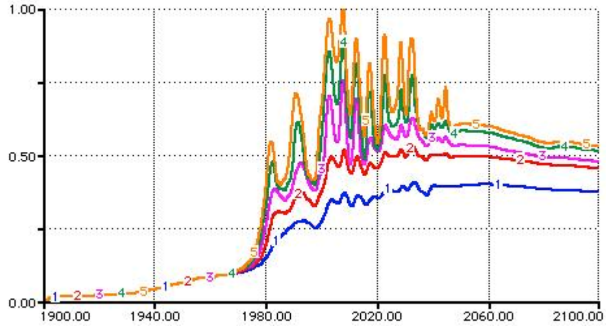
Sensitivity runs

A sensitivity analysis was made to investigate the effect of different demands in the platinum group metals market. Figures 4.1.11.16a-f show the results from a sensitivity run with the PGM-model. The standard scenario demand was multiplied with a factor rising in a scale: 0.4, 1.0, 1.6, 2.2, 2.8. In the model, increased demand push on the mining rate, mainly through the price (See the causal loop diagram in Figure 4.1.11.6). The sensitivity run shows that the model is sensitive to demand across ore grades, recovery rates, mining rate, supply to society and market price. The platinum group metals system is a very dynamic system, and demand and supply are matched through the actions of market mechanisms. Thus, demand if always met after modification. Price modifies both demand and supply. Figure 4.1.11.16a shows the response of the mining rate as demand increases. As demand increases the market becomes more volatile, and the same pattern is seen in the supply diagram (Figure 4.1.11.16b) and in the market price. It can be seen how higher demand creates more price volatility, something that is also actually observed in the markets. However, the mining rate responds to the increase in demand with an increase in price which through feedback drive up the price with a little delay. Higher prices also tend to drive up recycling.

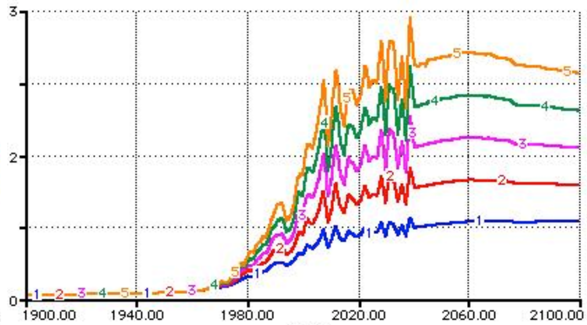
Under the business-as-usual scenario, at least 150,000 ton of the 216,000 ton platinum group metals resources will be extracted by 2400. With the highest demand, 2.8 times the present demand, about 200,000 ton of the resource of 216,000 ton platinum group metals will have been extracted by 2400. Higher demand leads to higher mining rates, higher prices, faster depletion of the resources, and more complete extraction.

The total recycling fraction (Figure 4.1.11.14) and the recycling fraction in the automotive converter sectors (Figure 4.1.11.16d) show a similar behaviour. The driver behind the volatility is the price simulation which is shown in Figure 4.1.11.16e. The ore grade development from all of this is shown in Figure 4.1.11.16f. The sensitivity run defines an envelope where the platinum price is most likely to stay. The price seems to vary in the band from 800-3,000 $ per troy ounce. The base case fluctuates in the range 1,000-1,400 $ per Troy ounce. For the years 1990 to about 2060, the platinum group metals market will remain vey volatile, with large fluctuations in price. In the present model, trading for pure speculation and derivatives trade were not included. From the gold market it is well known that speculation and derivatives trade tend to destabilize the price and create much more noise in the price signal (Sverdrup et al., 2013a). Supply is affected by the increases in demand, and pushing the recovery rates up by increasing the price. The mining rate responds to the increased price volatility as the demand increases. The standard run fits the observed data best.

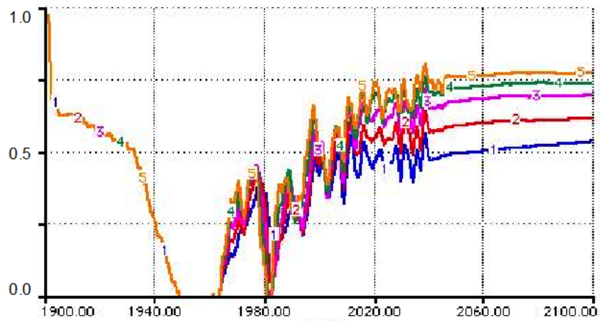
Figure 4.1.11.16. A sensitivity was made with the model using the inbuilt sensitivity feature in STELLA. The standard scenario platinum group metals demand was multiplied with a factor, represented by a line as follows: 1=0.4, 2=1, 3=1.6, 4=2.2, 5=2.8 times the standard run. The results are shown for a; mining rate, b; supply to the market, c; fraction of total supply being recycled, d; the degree of catalytic converter recycling, e; platinum market price and f; ore grade.



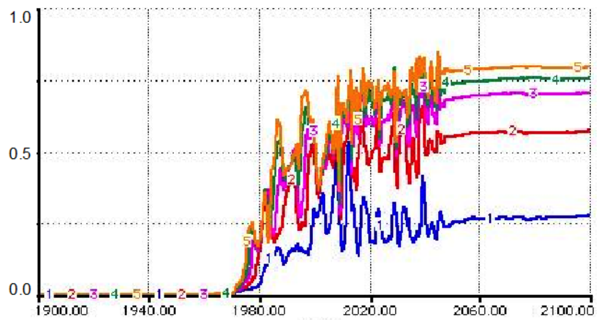
a: Mining rate, thousand ton per year



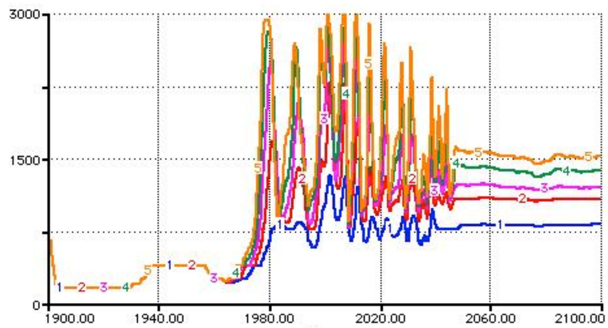
b: Supply to the market, thousand ton per year



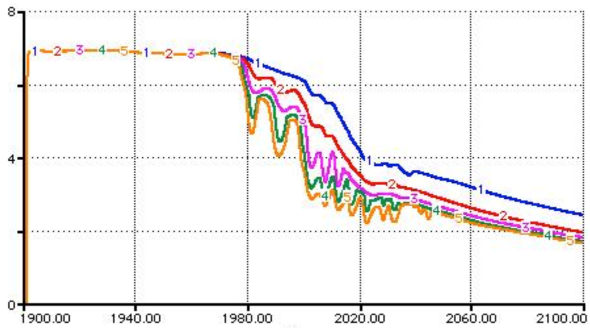
c: Fraction of supply from recycling



d: Catalytic converter recycling, fraction



e: Price, $ per troy ounce



f: ore grade, % of extracted rock

Source: Own Figure, Lund University/University of Iceland

Table 4.1.11.11 shows an analysis of the relative rarity of platinum as compared to other metals in ton. Data sources for gold, silver and copper were taken from Sverdrup et al. (2013, 2014a, 2014b). Considering the observed ore grades and the amount already extracted, the comparison with gold (URR=320,000 ton) suggests that the URR obtained for platinum group metals (URR=216,000 ton) is about right. The recycling rate of platinum stand out as particularly unsatisfactory considering its strategic importance and the high value. Market mechanisms are obviously not able to handle the situation alone.

Table 4.1.11.11. An analysis of the relative rarity of platinum as compared to other metals in ton. Data for gold (Sverdrup et al. 2013), Silver (Sverdrup et al. 2014a) and copper (Sverdrup et al. 2014b).

Metal	Mining 2015	Annual market supply 2014	Reserve known	URR	Annual mining, fraction of reserves	Stock-in-society 2014	Available for trading 2014	Recycling
	ton per year		ton		%	ton		%
PGM	490	680	90,000	216,000	0.5	7,000	3,000	55-60
Gold	3,000	5,500	80,000	320,000	4.5	190,000	12,000	80-92
Silver	23,000	40,000	1,200,000	3,200,000	2.0	1,200,000	430,000	65-80
Nickel	2,000,000	3,000,000	73,000,000	185,000,000	2.5	58,000,000	4,000,000	50-60
Copper	16,000,000	25,000,000	670,000,000	3,600,000,000	2.4	320,000,000	42,000,000	60-65

Extractable amounts

The reserves in this simulation were defined according to the opportunity cost principle, where all resources have a price at which they can be extracted. Some of the technically extractable amounts may thus be out of reach because of price limitations. Figure 4.1.11.16b shows the relationship between metal price and the recycled amount according to the simulations done with the PGM model, using the opportunity cost principle. Contrary to what Tilton (2002, 2007, 2010) claims, the a-priory set URR is larger than the finally extracted amount even if the opportunity cost principle is used in the case of platinum group metals (Neumeyer 2000). The reason for this is simple, Tilton makes a theoretical claim, but we have actually modelled it and tested if the theory really works when coupled to the systems feedbacks, and the outputs tested on observed data. With the whole technically extractable amounts included, but the actual extraction activity being based on a cost graded resource, a more dynamic model is obtained, where URR is a variable and not a fixed number from the beginning. We agree with Tilton (2010, 2012a,b) in the statement that the opportunity cost grading approach to extractable amounts in the model is a better representation of reality in the mine. In most cases we do not run into metal scarcity because of lack of metal ores, but because of lack of money or lack of energy for the extraction.

Peak production, recycling and sustainability

Platinum group metals are and have always been scarce metals. If somebody wants to understand how scarce metals behave and what it means that a metal is scarce, these metals are prime examples. They have limited physical availability, they have very high price because of very limited availability in the market, and the price level is very volatile because of a small buffering stock in society. The unsustainable present use of the platinum group metals are a future threat to the long-term supply to the society. The PGM model outputs suggest that we are in the peak production years at present and that scarcity may set in during the next decades. Figure 4.1.11.14a shows the platinum group metal recycling rate, and Figure 4.1.11.14b shows the recycled fraction, defined as ratio between total supply minus mining divided by the total supply. Significant amounts of platinum group metals are lost through low recycling rates of automotive catalytic converters. On the global scale recycling degrees vary from 40-

60% for Europe and to 10-30% for the rest of the world (Eliott et al., 2014, 2013; UNEP 2011a,b,c, Wäger et al., 2012, 2013, Yang 2009). An important use of platinum group metals is as catalyst in the Ostwald process for producing nitric acid. The high-pressure, high-load variant of the nitric production of the Ostwald process may loose up to 20-30% of the weight during each production campaign that lasts 3-5 months. Some of the platinum group metal lost is recovered afterwards (30-60% of the losses), but a significant amount is lost dissipatively in the equipment, in process waste and with the product (Hoke 1940, Hagelücken 2006, 2012; Steinlechner and Antrekowitsch 2015). Platinum is a far to important metal for society too loose, and proper efforts must be put in place to secure real long term platinum conservation. The available data from platinum metal recycling companies, suggest that once metal-containing material reaches the recycling plants, minimal amounts are lost, as the processes are very efficient (96-99% recovery). The knowledge of the platinum group metals value is pervasive at the recycling companies (Wäger et al., 2011, 2012, Sverdrup, personal communication). Jewellery has a long turnover in society, typically jewellery in gold or platinum stay from 40 to 60 years in society before it is returned.

Technical and political risks

The heavy reliance on South Africa and Russia as main suppliers of platinum, palladium and rhodium, involve significant supply risks. The risks are several, technically, platinum group metal mining and extraction can be challenging technically. But these main producing countries have also significant social and political risks (Risk & Policy Analysts Limited 2012, Eliott et al., 2013, 2014). Technically, the platinum group metal mines in South Africa have reached the depth limit for how far down it is possible to go (5 km). For some of the known reserves, the limiting factor may be technical difficulties in mining as well as cost limitations and safety concerns. The deepest mines reach down to 3,900 m at present (Canada and South Africa), and the rock temperature is 55-60oC at that depth. Approaching 5 km mining depth appears to be the next frontier, and attempts to delve that deep is already in progress (Mudd et al., 2012, Zientek et al., 2014). It appears to be technically feasible, but at a higher cost and significant risk for the workers if technical failures occur. A few gold mines and some platinum mines are running shafts down below the 4 km mark already. South Africa is a young democracy, with very substantial problems of high social stresses in society. Corruption, bureaucratic abuse of power, inefficient state institutions, partisan politics and social tensions are rising from persisting large inequalities and difficulties with service provision from the government. This is eroding social trust and causing risk for political instability in the long term (Eliott et al. 2013, 2014, International Platinum Association 2015, Implats 2015, Fukuyama 2014, Mineweb 2015). The post-apartheid government gained democracy and inherited a reasonably efficient state with efficient state institutions that used to be based on meritocracy and strong governance from the earlier administration. The new government has had problems in maintaining the integrity of the meritocracy as a basis for positions in the state institutions, and this may pose problems for the future if the degree of corruption seen today persist or increase (Fukuyama 2014). In the future, efforts to improve governance and curb corruption in South Africa has to be expanded. The state needs to embrace more of the public participation into governance, limit corruption and stay effective, accountable and fair, something that has been very difficult to achieve (Fukuyama 2014).

Russia has challenges with some of the reserves being located in arctic regions, making mining conditions difficult. The production rate is to a large degree dependent on the mining and extraction of nickel and copper, and to a smaller degree on platinum content in the ore. Russia has a cultural heritage burden from the past with tendencies towards dictatorship, repression, institutional inefficiency, political disorganization, with substantial problems with corruption, abuse of power, lack of proper justice systems and social tensions rising from inequalities and difficulties with service provision from the government. The problems are in several ways paralleling the problems of South Africa. Russia has gone from being a flawed democracy to become a consolidated dictatorship during the period 2012-2015 (Freedom House 2015, Fukuyama 2014). Under such conditions, technical challenges are not the main limiting factor, but rather the shortcomings of society and lack of quality government.

The Phillipines and Indonesia may become significant suppliers of platinum group metals in the future, once the mining of their nickel deposits gets established and operative. The United States, Canada and Australia with their more stable societies appear to be the long term fall-back positions for supply platinum group metals. Canada has substantially more potential to be explored for new platinum group metals deposits, considering what is known from it's geology.

4.1.11.10 Conclusions

The platinum group metals are among the rarest metals on Earth. For these metals, platinum, palladium, rhodium, iridium, ruthenium, and osmium, has always been scarce, reflected in the constantly very high price. The price volatility is typical of a scarce metal with a small stock-in-use in society and low flexibility for accommodating rapid changes in demand. Our result is that the extractable resources are twice as big as earlier anticipated and estimated. This result does not affect the fact that the metals are very scarce and can be expected to remain scarce for the foreseeable future. We find an ultimately recoverable resource (URR) of about 216,000 ton platinum group metals (The platinum resource is approximately 99,000 ton, and the palladium resource is approximately 93,000 ton), summing up both primary resources and resources coming through dependent secondary extraction. The sensitivity analysis suggests that the resource size given above is the most probable estimate.

In this study a global PGM-model was developed and tested and found to reasonably well represent the observed trends for the last 110 years. We conclude that the model suggests that peak production from mines will occur in the period 2035-2050, and a peak in supply to society 20-35 years later. The model outputs suggest that diffuse and dissipative losses are at present too large on the global systemic level for the platinum group metals, and that conservation efforts would be needed to be strengthened if we consider the very long term perspective. Because ore grade and cost of extraction is included in the model, effort required per amount PGM extracted increase with time, eventually closing down extraction. We can summarize the conclusions:

1. Production of supply and extraction
    a. The platinum group metals extraction will go through a peak in 2035-2050 and market supply reach a peak during 2070-2080. The delay between extraction and supply peak depend on the degree of recycling and the relative size of the loss flows to the extraction flows.
2. Reserves and resources
    a. The total extractable amounts of platinum group metals if we take in the latest resource assessments, consider new technology for mining at great depth, and the resources existing at great depth are about 220,000 ton
    b. The known reserves will stay at about 60,000 ton and slowly decline after 2030.
    c. The ratio between the cumulative amount supplied to society and the cumulative amounts primary extracted is about 2.6, this is called Factor X.

We are able to successfully reconstruct the price of platinum from 1900 to the present in broad terms, as well as the development in ore grade with time.

4.2 MATERIALS SUB-MODULE

Materials submodule of the WORLD Model consists of two sub-modules:

(1) Sand, gravel, crushed rock and stone; and

(2) Phosphorous (work in progress – not subject to this report)

4.2.1 SAND, GRAVEL, CRUSHED ROCK and STONE (SGS) sub-module

4.2.1.1 Introduction

A conventional view is that there is for practical purposes endless resources of sand, gravel and stone on the Earth for human use. Given that we are in the midst of the Great Acceleration in resource extraction and use (Steffen et al. 2015) this view needs to be critically examined. Construction materials, such as sand, gravel, stone aggregates and rock are fundamental for human development and wellbeing. Materials play a central role in the economy and stony aggregates are one of the largest material flows humans move around in terms of weight. Sand and gravel occur naturally either as naturally sorted aggregates or as mixed aggregates. Some measure of crushing is also frequently employed. USGS (2015) and UNEP GEAS (2014) estimate that sand, gravel and stone materials use in construction amounts to about 47-59 billion tons a year, the range show the uncertainty in the estimate. Sand and gravel, account for both the largest share (from 65-85%). The world's use of aggregates for concrete can be estimated at 26-30 billion ton a year for 2012 (BMI 2014). Another large driver in North America for sand demand was the activity of fracking for shale gas (BMI 2014). China, India, Brazil, USA and Turkey are currently the world's biggest concrete producers, with China and India accounting for two-thirds of total global production. In the past 20 years, cement demand in China has increased fourfold compared to a growth of about 58 % in the rest of the world, (Giljum et al., 2009, 2011, UNEP 2014). There is significant concerns about the sustainability of present global material extraction rates, including the issue of sand, gravel and stone extraction rates (Bardi 2013, Giljum et al 2000, 2011, Heinberg 2011, Krausmann et al., 2009, Meadows et al., 1972, 1974, 2005, Morrigan 2010, Peduzzi 2014, Sverdrup and Ragnarsdottir 2014)

Until recently, most construction sand was mined from riverbanks and local pits. With the dramatic increase in demand, industrial scale marine and beach sand mining are increasingly common, along with sand and gravel made from industrially crushed stone (Merwede 2014, OSPAR 2003, Radzevičius et al., 2010, Robinson and Brown 2002, Velegrakis et al., 2010, Suthpin et al., 2002, Morrow 2011, Krause et al., 2010, Ravishankar 2015, Anthoni 2000, Ashraf et al., 2011).

While many the world's deserts are rich in sand, much of the material is often too well sorted, have the wrong shape (the grains may be round and polished) or too fine grained for use in construction materials. Crushed sand and gravel are characterised by the fact that the particles are sharp edged thereby binding well with cement owing to their shape. The more polished and rounded the particles of deserts and certain beaches are the less suitable for construction and in particular as fillers in cement. Thus, a large part of natural sand and gravel deposits found will be unsuitable for construction and building because of wrong physical properties.

Sand mining, irrespective of where it occurs, usually has major environmental impacts. Earlier modelling approaches have been attempted, however, the methods used has some significant shortcomings. Econometric input/output table generated time series work as a short term extrapolation tool, but lacks stocks and thus cannot handle delays. Only a limited amount of feedbacks are possible in such models, and the inclusion of market dynamics mechanisms is not possible. Material Flow Analysis can deal with stocks, but only step forward in a spread-sheet manner. Thus, no market dynamics or systemic feedbacks can be included (Graedel and Allenby 2003, Moll et al., 2002, Hirschnitz-Garber et al., 2015, Sverdrup and Ragnarsdottir 2014, Pauliuk et al., 2015). In the WORLD 6 model being developed by the authors, most major resources (metals, rock materials, fossil energy, renewable energy, phosphorus, agricultural land, population) are modelled together with their causal connections through energy consumption, infrastructures and mass balances. The WORLD 6 model generates resource supply to global markets and generates global supply and global market prices endogenously.

4.2.1.2 Objective

The objective is to make and test a first global assessment of long-term use and availability of sand, gravel, stone and rock for human use through an integrated global model for the extraction and market supply of sand, gravel (glacial, fluvial and marine) and cut quality stone from rock for construction needs. The model will be tested on independent observed data – production and market price – to ensure that the model developed has satisfactory performance. The purpose is to make a simplified model that still will perform satisfactorily well when compared to the available data. A secondary objective is to link such a model within the WORLD6 model being developed by the authors. The test here will be module performance inside the WORLD6 model to assessed by comparing observed data and consistency with the outputs of the GINFORS model. Finally, global production rates, prices and supply of sand, gravel, crushed rock and stone will be assessed using the model. The standard for evaluating the performance of the model will be by comparison to observed data on sand, gravel and cut stone extraction rates, rates of sand and gravel from crushing of stone and recorded market price for sand, gravel and cut stone.

It is outside the scope here to make a sensitivity analysis of the Sand-Gravel-Stone (SGS) sub-model in WORLD6 and exploring its sensitivity. We will analyse the model qualitatively using the available causal loop diagrams, but the rest will be the subject of a future study. It is also outside the scope of this study to investigate different policy interpretations or future policy options. That will be the subject of a future study. The model is run for the time interval from 1900 to 2400 AD, or 500 years; 115 years of known history to evaluate the model and validate its performance and for 385 years under assumed business-as-usual conditions. We have adopted the long time-span because of the long delays in the system (residential time in use of 100 years) and that a proper run should normally cover at least three system delay times to evolve through all the inherent dynamics of the system. An approach of using business-as-usual was adopted. Alternative pathways are thinkable, but this will be the subject of later studies.

4.2.1.3 Methods and theory

Reserves

Owing to data constraints several approaches to quantification of reserves and resources have been taken. The reserves estimates are based on classical geological survey estimates, and the allocation of extractable amounts according to ore quality, stratified after extraction costs (Singer 1993, 1995, 2007, Singer and Menzie 2010, Tilton 2007, 2009, 2012, Neumeyer 2000, Sweeneye 1992, Sverdrup et al., 2015a,b,c,d). However, an extensive compilation of regional reserve and resource assessments have not yet been done, making our reserve and resource estimates difficult to do with accuracy. For sand, gravel and stone the data availability is less straightforward in the sense that while there is a wealth of information at small local scales, but few regional summaries and compilations there is almost none at the global scale. Our resource estimates for sand, gravel and stone are very approximate, and are of a back-of-the-envelope type of approximation that may need to be revised dramatically in the near future (Singer 1993, Kostka 2011, Kogel et al., 2006).

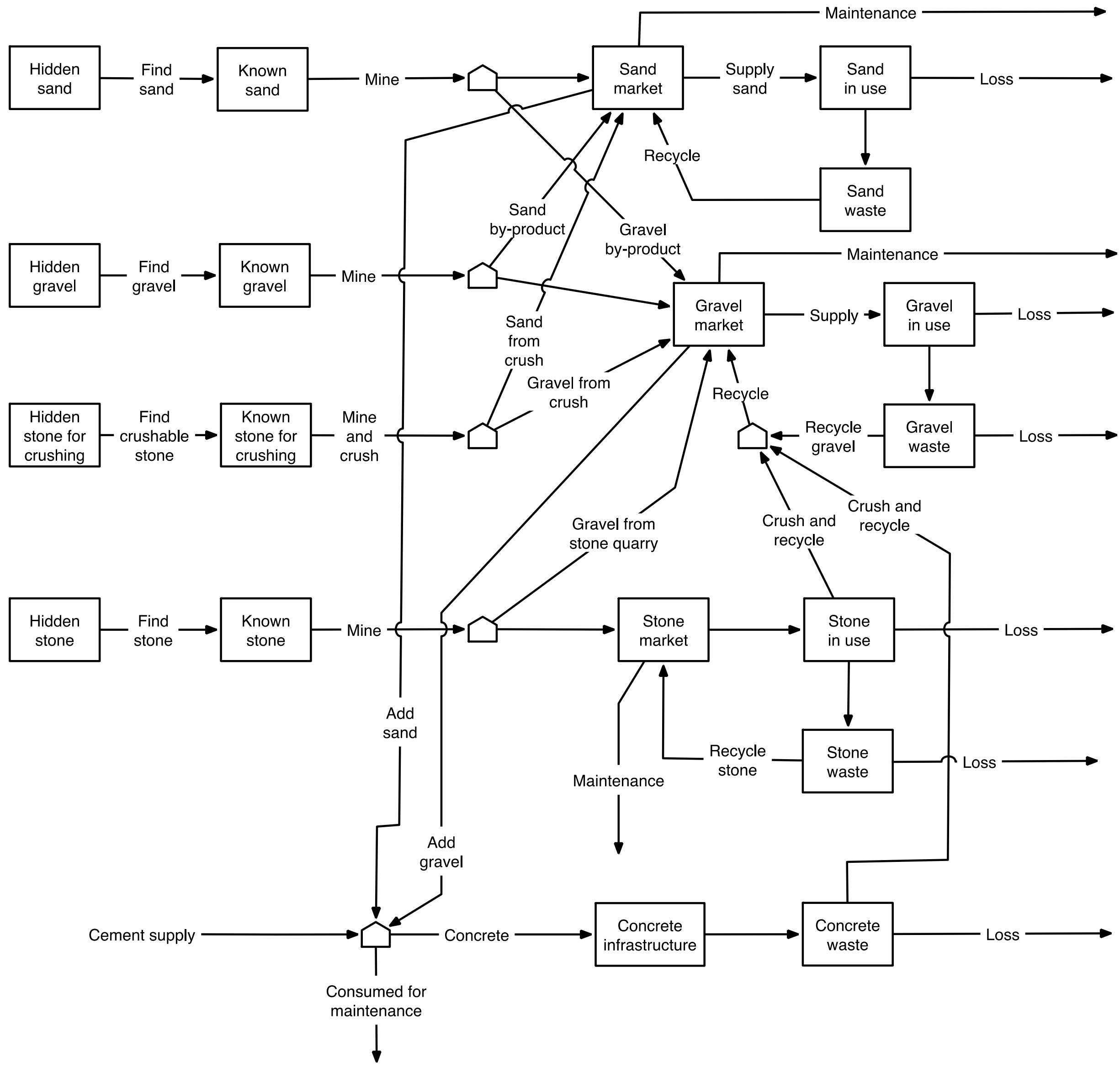
4.2.1.4 Model description

Figure 4.2.1.1 shows a basic flow chart for the SGS model. The SGS model has the following stocks used to define the coupled differential equations from mass balance:

1. Sand
    a. Mineable stocks;
        i. Known sand

        ii. Hidden sand
    b. In society, we distinguish 2 above-ground stocks;
        i. Sand in trade market
        ii. Sand stock-in-use in society
2. Gravel
    a. Mineable stocks;
        i. Known gravel
        ii. Hidden gravel
    b. In society, we distinguish 2 above-ground stocks;
        i. Gravel in trade market
        ii. Gravel stock-in-use in society
3. Stone for crushing to sand and gravel
    a. Minable stocks:
        i. Known stone for crushing
        ii. Hidden stone for crushing
4. Cut stone for construction, with gravel and sand as by-products
    a. Mineable stocks;
        i. Known stone
        ii. Hidden stone
    b. In society, we distinguish 2 above-ground stocks;
        i. Stone in trade market
        ii. Stone stock-in-use in society
5. Sand, gravel and stone embedded into concrete structures, with gravel as by-product from infrastructure demolitions:
    a. In society, we distinguish 2 above-ground stocks
        i. Sand, gravel, stone and concrete embedded into concrete infrastructures in use in society
        ii. Concrete structure waste from demolishing old structures

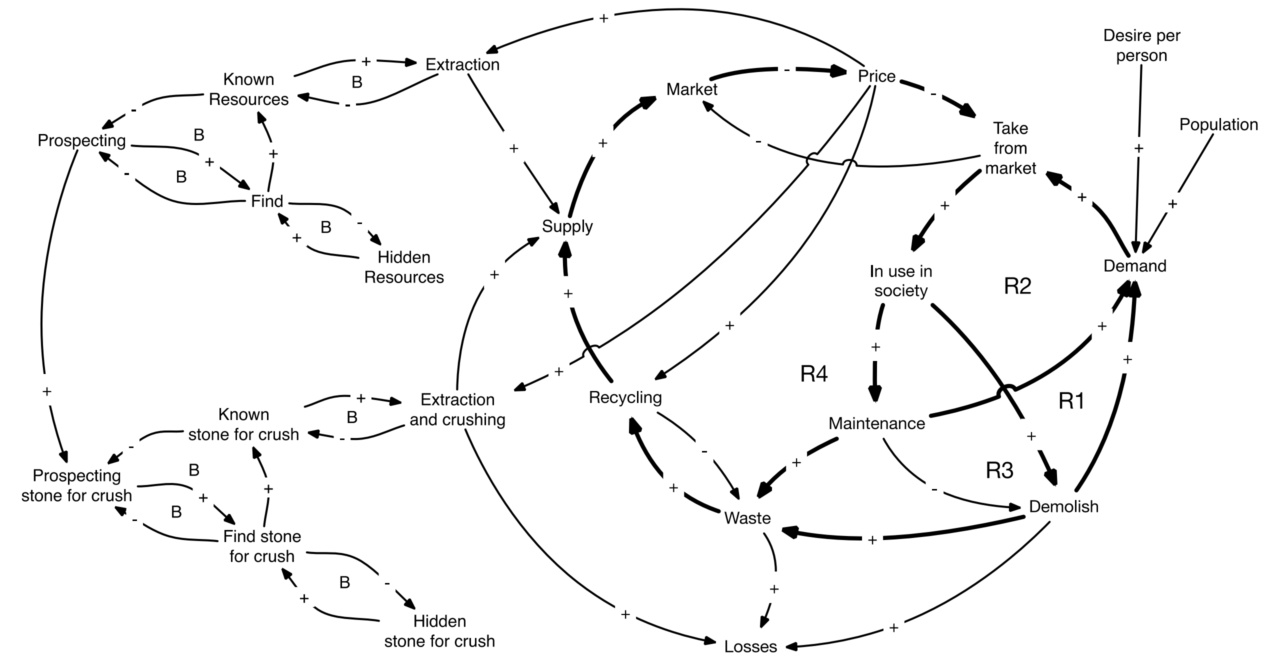
Figure 4.2.1.1. The flowchart for the sand-gravel and stone model (SGS). Maintenance flow are assumed to be replacement for lost material and thus do not add to stock-in-use. Known corresponds to known reserves. All known plus all hidden corresponds to total resources.



Source: Own Figure, Lund University/University of Iceland

This makes a model with 16 linked material stocks, resulting in a system of 16 linked differential equations to be solved. The mining activity is price-driven, the price is the market price. Figure 4.2.1.2 shows the price-extraction driving mechanism used in the SGS model. In the model, the market price is set twice every week throughout the simulation. The extraction is in the real world driven by operations profit. This implies that the main driver is the difference between the income from materials sales (market price times shipped amount) and the extraction cost. The cost is estimated as extracted amount times the total cost, where the total cost is made up of three components; labour cost, capital expenses costs and energy costs. We have tried this out (Figure 4.2.1.3), as well as a simpler model where we use the price as a proxy for the profit (Figure 4.2.1.2). Both approaches work well. The extraction for sand, gravel and cut stone also competes with recycling for sand, gravel and cut stone, mostly on a cost basis (Figure 4.2.1.2).

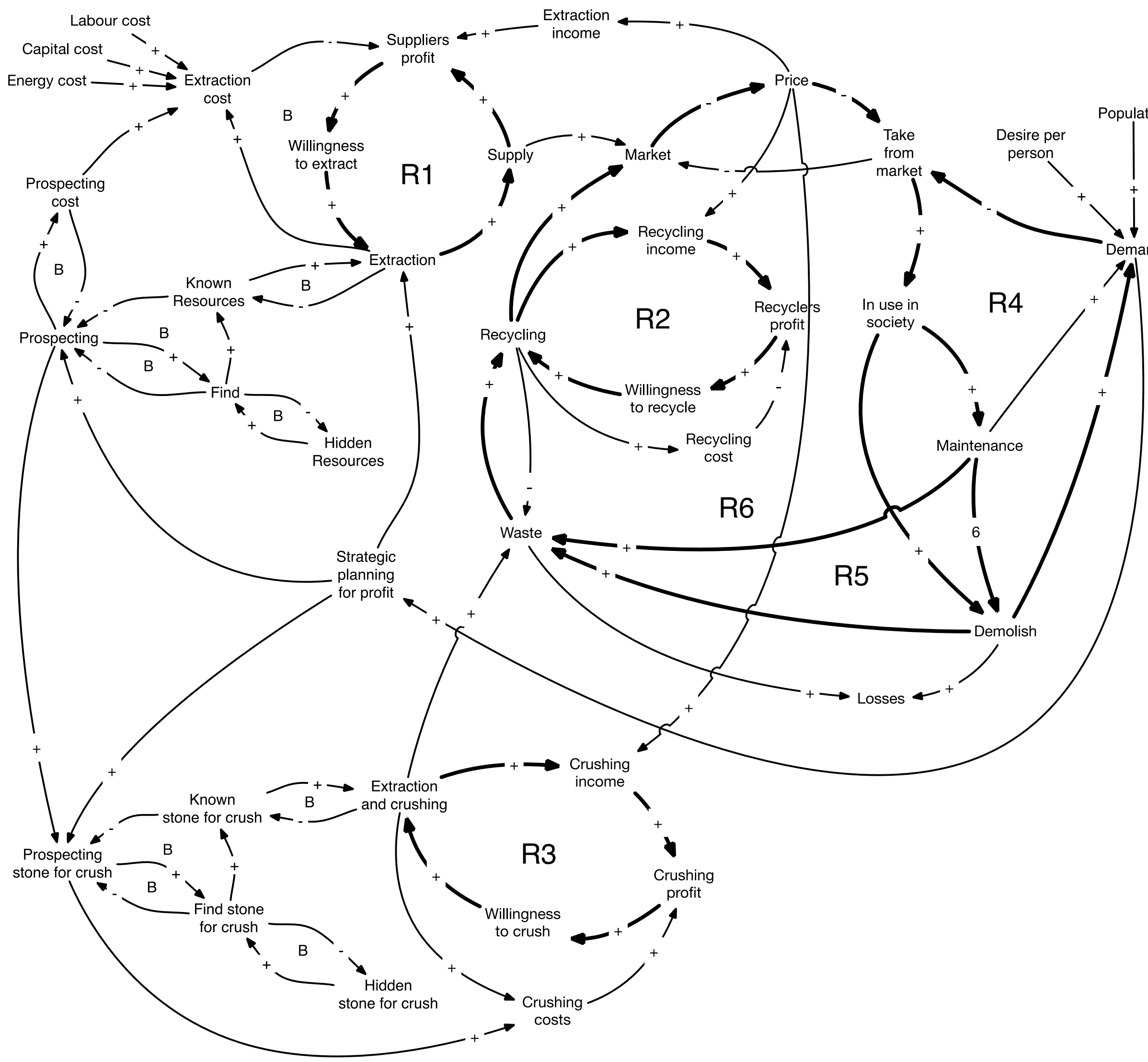
Figure 4.2.1.2. The basic causal loop diagram applied for the simplified SGS model. The causal loop diagrams for sand, gravel, stone for crushing and cut stone were linked as is shown in the flow chart in Figure 4.2.1.1. The actual model consists of 4 such coupled causal loop diagrams for sand, gravel, crushed stone and cut stone as Figure 4.2.1.1 will demand. The bold arrows show the reinforcing. The reinforcing loops (R) keeps the system running. The balancing loops (B) act as brakes in the system. The system is driven by demand from general consumption, demolition and maintenance (R1-R2) and income through price and pushed by demand (R3-R4).



Source: Own Figure, Lund University/University of Iceland

Figure 4.2.1.3 shows the full causal loop diagram for the complex version of the SGS model. The actual model consists of 4 such coupled causal loop diagrams for sand, gravel, crushed stone for sand and gravel and cut stone. The reinforcing loops (R) keeps the system running. The model is driven by profits (R1-R3) and through demand from population, maintenance and demolishing of old structures. The balancing loops (B) act as brakes in the system. The model was simplified to become like Figure 4.2.1.2. The simplified model had basically the same dynamic behaviour as the complex one, but has better stability and demand less parameterization.

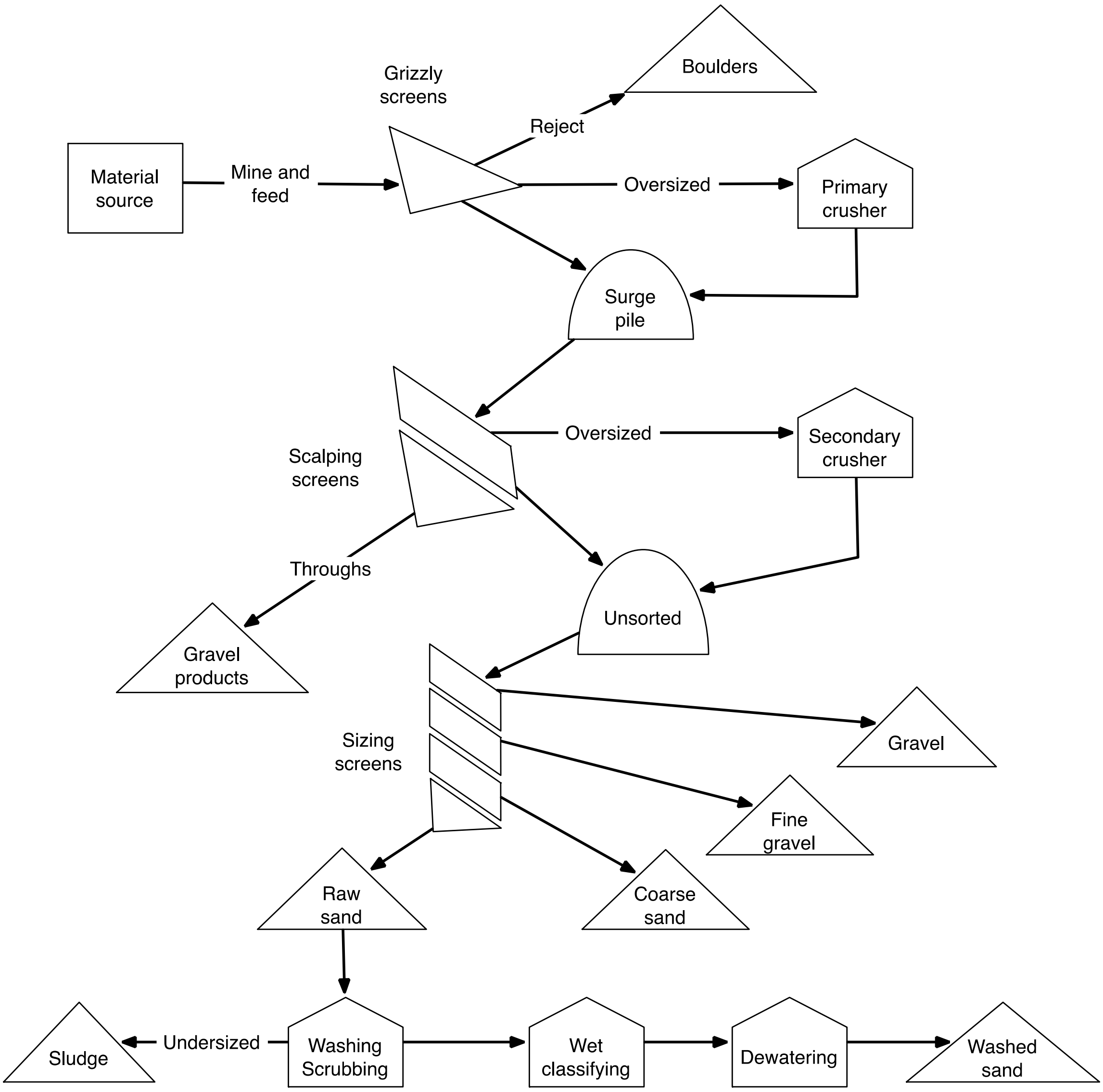
Figure 4.2.1.3. The full causal loop diagram for the full SGS model. The actual SGS model consists of 4 such coupled causal loop diagrams for sand, gravel, crushed stone for sand and gravel production and cut stone. The reinforcing loops (R) keeps the system running. The model is driven by profits (R1-R3) and through demand from population, maintenance and demolishing of old structures (R4-R6). The balancing loops (B) act as brakes in the system. The model was simplified to become like Figure 4.2.1.2. The simplified model had basically the same dynamic behaviour as the complex one, but has better stability and demand less parameterization.



Source: Own Figure, Lund University/University of Iceland

Figure 4.2.1.4 shows the flowchart used for the sand-gravel and stone model (SGS).

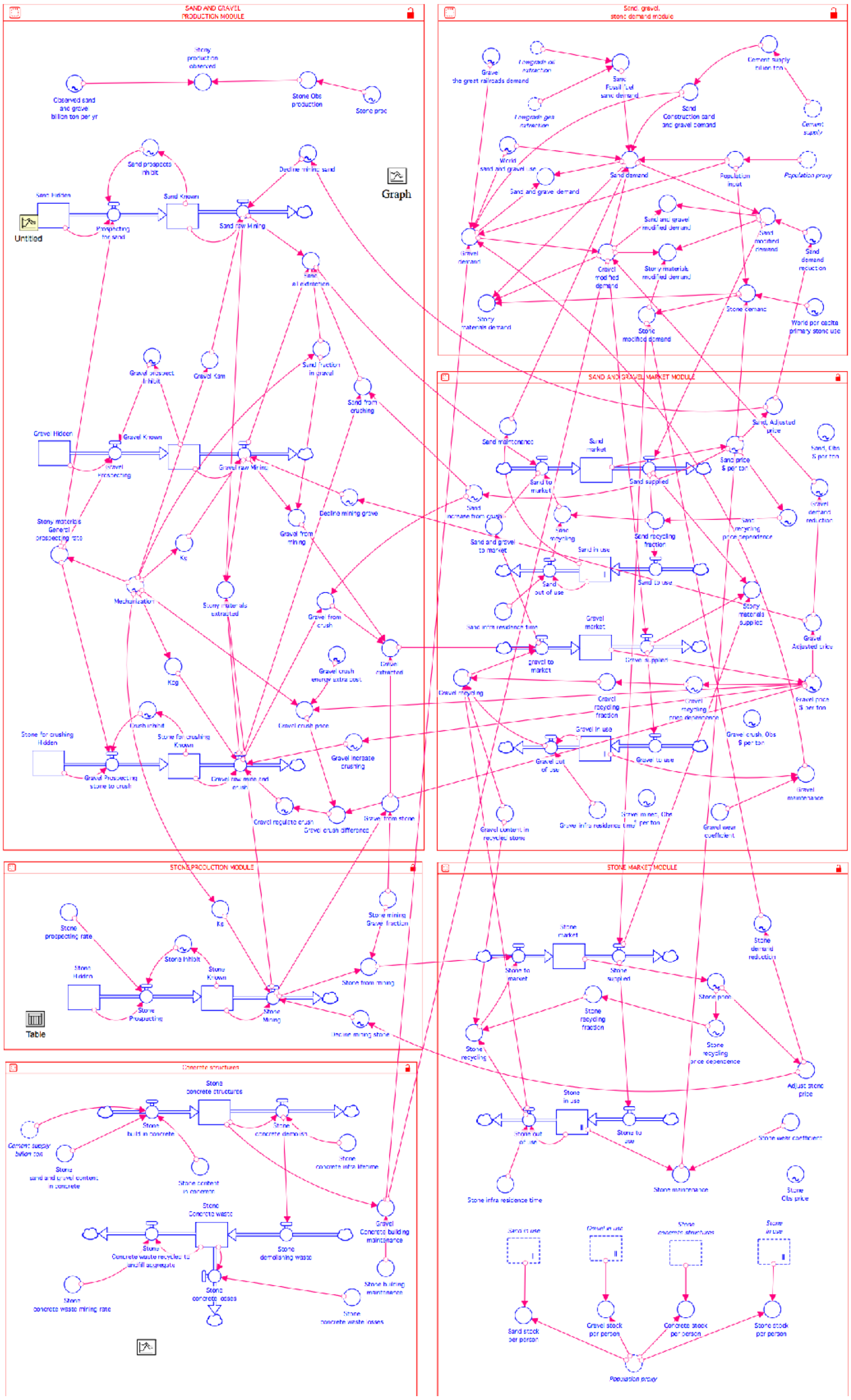
Figure 4.2.1.4. In the model, crushed stone is processed to sand and gravel. The simple flowchart for aggregates processing from mixed substrate to sand fractions and gravel products in a typical industrial operation is shown.



Source: Own Figure, Lund University/University of Iceland

Figure 4.2.1.5 shows the STELLA model diagram for the SGS-model. The SGS-model uses a 4-step Runge-Kutta integration method, with a 1/100 year time-step (3.6 days). In the model, we have assumed that sand and gravel stay 100 years in the infrastructure before the structure is destroyed, based on an evaluation of research literature (Hsu 2009, Korre and Durucan 2007). For cut stone we have assumed that the residence time in society is 100 years. The average life-time on a concrete infrastructure unit is set to 60 years, based on data from United States, Germany and China.

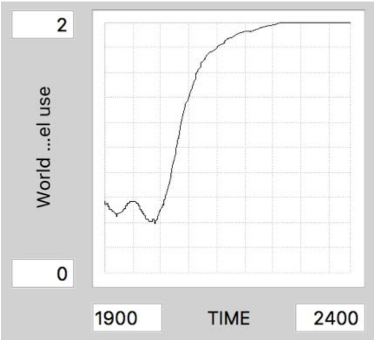
Figure 4.2.1.5. The sand-gravel and stone (SGS) model, in the STELLA modelling environment.



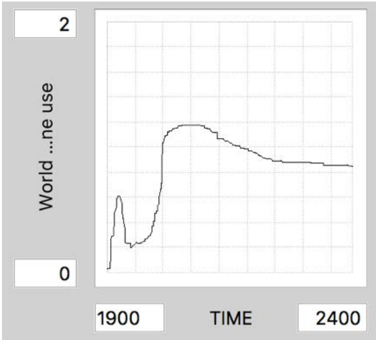
Source: Own Figure, Lund University/University of Iceland

The SGS model was embedded into the WORLD6 model. In the model, the profit is generated by sales to the market, but reduced with extraction costs and prospecting costs. There are three reinforcing loops in the system. The reinforcing loop marked R1 in Figure 4.2.1.3 in the supplementary material is driven by the profits-extraction-supply loop. The reinforcing loop R3 is driven by the supply-market, taken from market-society-demolish-recycle loop. The most important balancing loops; B in Figure 4.2.1.2 are two, the first when the known reserves become depleted, the other when the hidden resources become exhausted and the known reserves can no longer be supplemented with new material. The final loop is when the waste has been exhausted and the last resource runs out. Price is calculated internally in the model as a result of the feedbacks illustrated in Figure 4.2.1.6 and Figure 4.2.1.7.

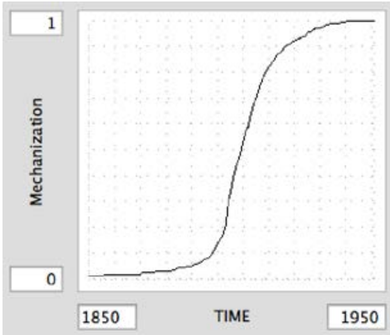
Figure 4.2.1.6. Feedback response curves employed in the SGS model.



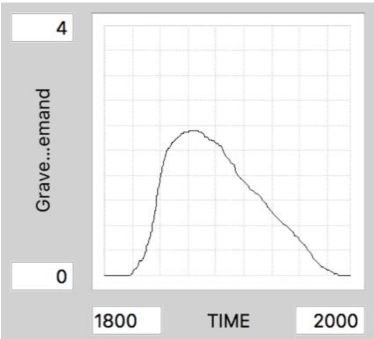
A: Global average demand, kg sand and gravel per person



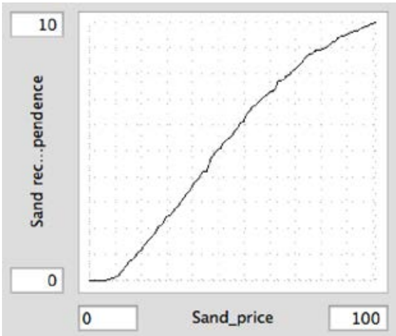
B; Global average demand, kg stone per person



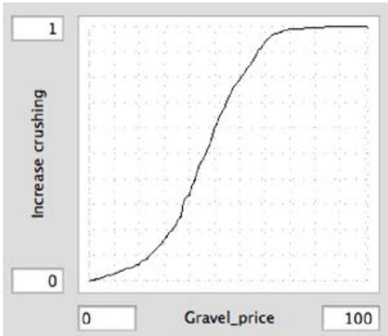
C: The effect of mechanization on work efficiency



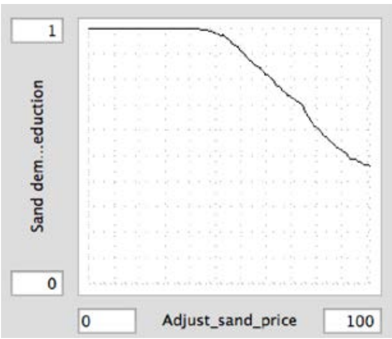
D: The great railroad expansion



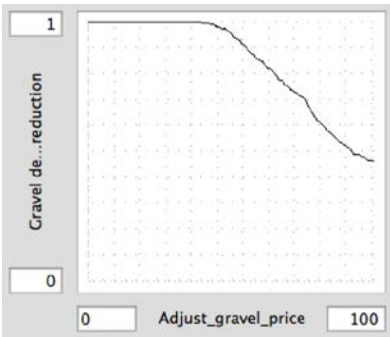
E: Sand shift price to go increase sand production from crush



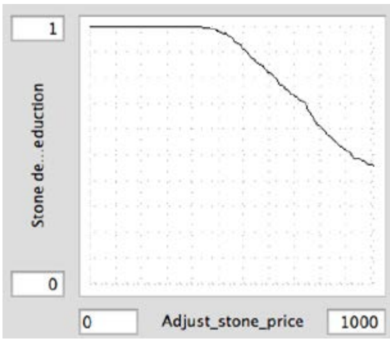
F: Gravel shift price to increase gravel production from crush



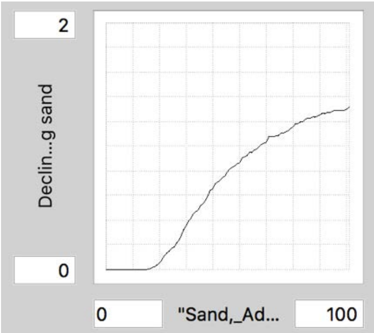
G: Price feedback on sand demand



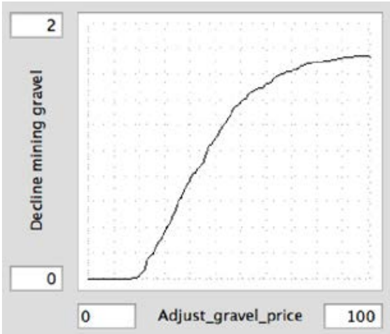
H: Price feedback on gravel demand



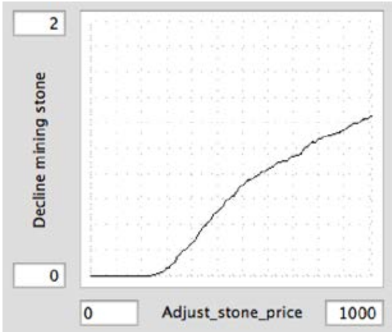
I: Price feedback on stone demand



J: Price feedback on sand mining



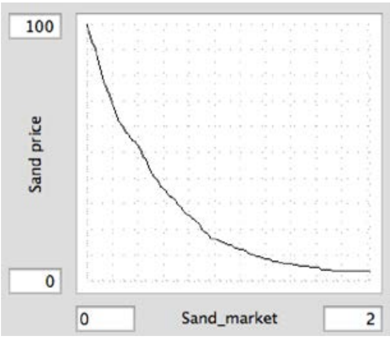
K: Price feedback on gravel mining



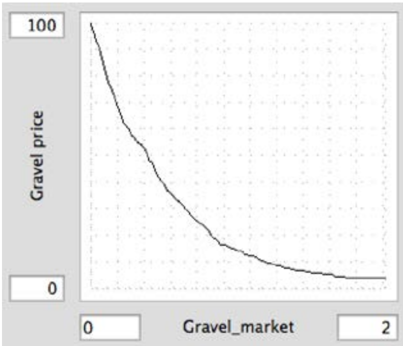
L: Price feedback on stone mining

Source: Own Figure, Lund University/University of Iceland

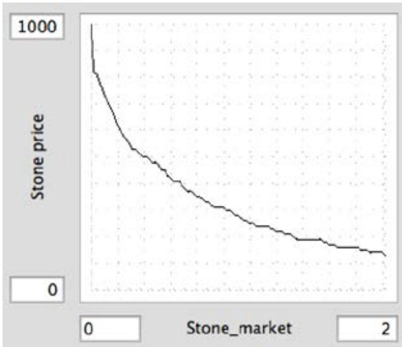
Figure 4.2.1.7. More feedback response curves employed in the SGS model



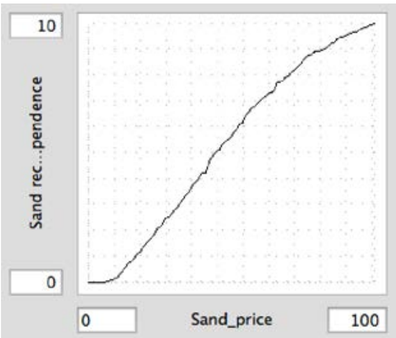
A: Sand price curve



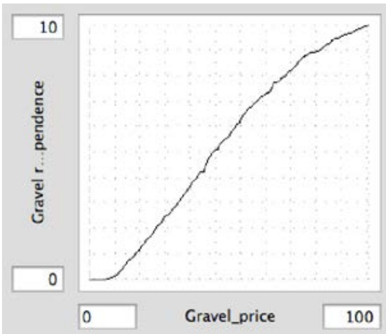
B: Gravel price curve



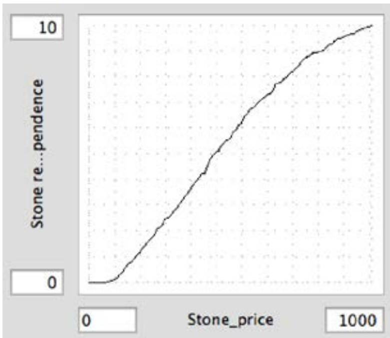
C: Stone price curve



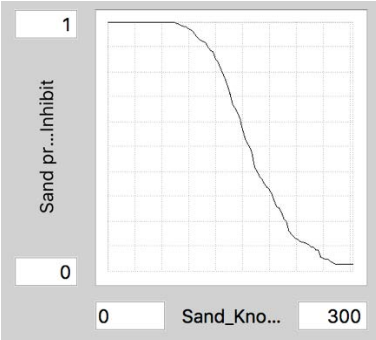
D: Price feedback on sand recycling



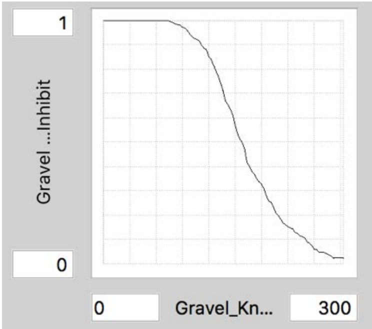
E: Price feedback on gravel recycling



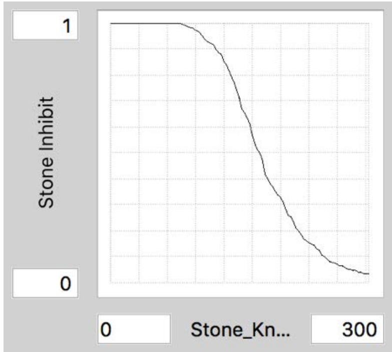
F: Price feedback on stone recycling



G: The feedback from known sand to prospecting.



H: The feedback from known gravel to prospecting.



I: The feedback from known stone to prospecting.

Source: Own Figure, Lund University/University of Iceland

Three parameters intervene to create price dynamics; the effect of market volume on price, and the effect of price on supply, demand and recycling. The characteristic curves for how this is expressed are shown in Figure 4.2.1.6 and Figure 4.2.1.7. The material mining rate was estimated with the following equation:

$$r_{Mining} = k_{\,Mining} * m^{n}_{Known} * \; f(price) * g(T) \qquad (4.2.1.1)$$

where r is the rate of mining, k is the rate coefficient and m is the mass of the ore body, and n is the mining order. The mining order depend on the difficulty of access and the access or the technological capacity is the main limiting factor. When technology dominates, it becomes zeroth order, when the resource availability limits, depending on the geometry n will be in the range 0-7-1. f(price) is a feed-back function of price, increasing mining at higher extraction profits and lowering it at lower metal prices (See the causal loop diagram in Figure 4.2.1.2). g(T) is a technology factor accounting for the invention of technologies used in efficient mining, refining and extraction of metal. We have chosen to set the mining process order at n=1 as most material is extracted in open pit mining. The rate coefficient is modified with ore extraction cost and ore grade. There are many different definitions of recycling available (Graedel and Allenby 2003, UNEP 2011b, 2013a). For the purpose of clarity, the recycling fraction displayed in the results section was calculated as follows in this study:

$$Recycling\ fraction = \frac{Flow\ of\ recycled\ metal}{Supply\ from\ primary\ extraction + \ Flow\ of\ recycled\ metal} \qquad (4.2.1.2)$$

The cost of the mining and extraction operation is mainly determined by two important factors beside cost of investments, the energy price and the ore grade. The size of the extractable ore body is determined by the rate of extractions (rmining) and the rate of prospecting (rdiscovery):

$$\frac{dm_{Known}}{dt} = -r_{mining} + r_{discovery} \qquad (4.2.1.3)$$

The resource discovery is a function of how much prospecting we do and how much there is left to find. The amount hidden reserve (mHidden) decrease with the rate of discovery. The rate is first order as prospecting is three-dimensional by drilling. The driving mechanism of mining comes from profits and availability of a mineable resource used in the model. The rate of discovery is dependent on the amount sand, gravel or stone hidden (mH) and the prospecting coefficient kprospecting. The prospecting coefficient depend on the amount of effort spent and the technical method used for prospecting.

$$\frac{dm_{Hidden}}{dt} = -r_{discovery} = -k_{prospecting} * m_{Hidden} \quad (4.2.1.4)$$

In the model, crushed stone is processed to sand and gravel. The yield is determined by the following equation:

$$r_{Product} = Y_{Product} * r_{Feed} \quad (4.2.1.5)$$

And for the recycling from stock-in-use-in-society, the following equation was used:

$$\frac{dm_{Society}}{dt} = -\frac{1}{t_{Society}} * m_{Society} \quad (4.2.1.6)$$

where tSociety is the average retention time in society.

$$\mathrm{r_{Recycling}} = \mathrm{x_{Recycling}} * \mathrm{r_{society\ outflow}} * \mathrm{g(price)} \quad (4.2.1.7)$$

where xrecycling is the fraction of the flow out of stock-in-use that is recycled. g(price) is a feedback function, increasing recycling when the commodity market price increases, improving recycling profits (See Sverdrup et al., 2014a). A simple flowchart for crushed stone to general aggregates processing to sand fractions and gravel products is shown in Figure 4.2.1.4. The diagram shows the process in far more detail than actually pictured in this flow chart. The parameterization of the significant feedbacks used in the SGS model, has been shown in the supplementary material. Table 4.2.1.1 shows the base parameter settings of the model. These parameters define the equation coefficients in the equations given above. The parameters have been set using generic extraction rate coefficients for the different processing rates (Lewis and Clark 1964, Pohl 2011, Darling et al., 2011). Other coefficients were taken from Graedel and Allenby (2003).

Table 4.2.1.1. Base parameter settings of the rate equations in the SGS model. The values were taken from estimates in the available scientific literature.

Parameter	Sand	Gravel	Stone for crushing to sand and gravel	Stone for building
Mining rate coefficient, fraction (kmining)	0.025	0.02	0.02	0.015
Mining rate order (n)	1	1	1	1
Prospecting coefficient, fraction (kProspecting)	0.005	0.005	0.03	0.035
Base recycling fraction, (xrecycling)	0.05	0.05	-	0.1
Society retention time, years, (tSociety)	100	100	-	100
Yield gravel in product, fraction, (Ysand)	0	0.85	0.9	0.5

Yield sand in product, fraction, (Ygravel)	0.85	0.15	0.1	0
Yield stone in product, fraction, (Ystone)	0	0	0.25	0.75
Yield in stone recycling	0	0.5	0	0.5

Table 4.2.1.2 shows the typical composition of concrete. The use of concrete is an important driver of sand, gravel and stone demand. Gravel use for concrete is 4.6 times the weight of the cement and sand use for concrete is about 3 times the weight of the cement used.

Table 4.2.1.2. Typical composition of concrete showing the importance of sand and gravel in construction. The values were taken from estimates in the available scientific literature.

Material	% volume content	Specific material density kg/m3	% weight content of concrete	% weight content of average reinforced materials in buildings
Sand	26	2,700	31	30
Gravel	41	2,700	49	48
Portland cement	11	2,200	12	11
Water	16	1,000	7	6
Air	6	1	0	0
Reinforcement iron	-	5,500	-	3

Demand

The demand was modelled based on a number of parameters and their values drawn from a number of references (Bolen 2011, Distelkamp et al., 2010, Korre and Durucan 2007, Kostka 2011, Krausmann et al., 2009, Merwede 2014, Oijens 2014, Robinson and Brown 2002, Gutowski et al., 2013, Chilamkurthy et al., 2016). The following parameters were evaluated for setting the demand:

1. World cement production, which correlates straight to world construction activity and infrastructure maintenance. Cement demand per person and year follows a pattern with increasing demand during the transition to industrial society with a peak and a decline down to a maintenance level. This is paralleled by the demand development for iron and steel (Cullen et al., 2012, Guirco et al., 2013, Hu et al., 2010, Moynihan and Allwood 2012, Pauliuket al., 2012, 2013, Stanway 2014). The cement demand per capita has peaked and declined in most of the industrial countries, China and India are in their major transitional stages now, and Africa is about to start. Cement demand is taken from another part of the WORLD6 structure.
2. The global expansion of roads and railroads during 1850-1960 used large amounts of gravel. It is given to the model as an exogenous input curve.
3. Maintenance of stony material-containing infrastructures in use in society. This is based on an annual decay of the stock in-use in society; sand, gravel, stone and embedded in concrete.
4. World fracking activity for oil and natural gas extraction, dominated by use in North America. Fracking is a way to extract oil and gas from deposits where these are not otherwise extractable. The ground is hydraulically fracked, expanded and the cracks propped open using sand. In the United States of America, fracking takes 50-60% of the domestic sand demand. Fracking activity is generated inside the energy module in WORLD6 as a part of oil and natural gas production.
5. Global general consumption patterns from new construction in diverse infrastructures, replacement construction, sand and gravel as volume filler in polymers and other materials.

This is taken together into a general equation that has the following shape as in Equation 4.2.1.8:

$$D = k1 * rC + k2 * rF + k3 * rR + k4 * rFF + rM + k5 * EA * N \qquad (4.2.1.8)$$

Where D is demand, rC is the rate of cement production, rR the rate of gravel use for roads and railroads, rFF the rate of fossil fuel production using sand, rM is the rate of sand, gravel or cut stone use for maintenance of stony material infrastructures, k1, k2, k3, k4, k5 are stony material use coefficients calibrated to the year 2000 for the specific commodity or activity, EA is economic affluency, and N is the global population in number of persons. The numbers to calibrate this relationship came from a number of sources, exemplified by Gutowski et al., (2013), Chilamkurthy et al., (2016) and CemNet (2014) as well as commercial market analysis such as those referred to by NewsChannel110 (2014). The maintenance demand for material is calculated as:

$$rM = kM * Mi \qquad (4.2.1.9)$$

where kM is the decay rate of the infrastructure and Mi is the amount of sand, gravel or cut stone in the infrastructure. An inherent assumption is that the decay of the stock-in-use is compensated for by maintenance. When the life time of an infrastructure is passed, it is demolished. The average lifetime is set at 100 years for sand and gravel in infrastructures and at 200 years for cut stone. We have also looked at the retention times for iron and steel in buildings, giving indications for concrete and stony materials (Cullen et al., 2012, Guirco et al., 2013, Hu et al., 2010, Moynihan and Allwood 2012, Pauliuket al., 2012, 2013, Stanway 2014). The resulting demand is shown in Figure 4.2.1.6 for sand and gravel combined and for cut stone.

Input data

The input data in terms of resource size, mining rates and other key parameters were quite difficult. Large parts of the sand, gravel and stone production is outside the official economy or in a grey zone, and only partially represented in the public statistics and UN or USGS databases. There is not much data available, and what is available is very uncertain. No good global synthesis is available. We have pulled together what we could find in the scientific literature, in corporate brochures and branch organization websites, and made a synthesis of that to our best estimate as presented in Table 4.2.1.1 adn Table 4.2.1.2, and shown in Table 4.2.1.3 and Table 4.2.1.4.

Table 4.2.1.3. Overview of primary and secondary mining of resources. Amounts in billion ton per year. The values were taken from estimates in the available scientific literature.

Material	URR, available for extraction, trillion ton	Dependent material, secondary production	Yield %	Production 2012, million ton/year
Silicate-based stony materials	12	Gravel	60-80	2,000
	125	Crushed stone	60-80	4,920
	12	Cement-sand	50	3,600
		Sand (excluding cement)	80	1,300
	42	Cut stone	30-50	70
	1,000	Rubble material	80-100	8,000
	1,191	Sum		15,000
Limestone	10 mill km3, >80%	Cement and mortar	50-80	3,930
	20 mill km3, 80-50%	Limestone gravel material	60-80	1,700
	20 mill km3, 50-20%	Limestone	50	12,000
	100 mill km3, <20%	Limestone	25	?
Sum all sorts of stony materials, billion ton per year				52,520

Table 4.2.1.4. Ultimately recoverable resource estimates for the input data to the SGS model. The values were taken from estimates in the available scientific literature.

Material	Yield grade of material % of the excavated weight becoming product	Billion ton of stone material extractable as final useful product		
		Hidden	Known	URR
Sand	80	12,000	2	12,050
Gravel	60	12,000	100	12,100
Stone to crush for gravel and sand	60-80	120,000	5,000	125,000
Stone to cut for construction	30-50	40,000	2,000	42,000

4.2.1.5 Results

Reserves and resources

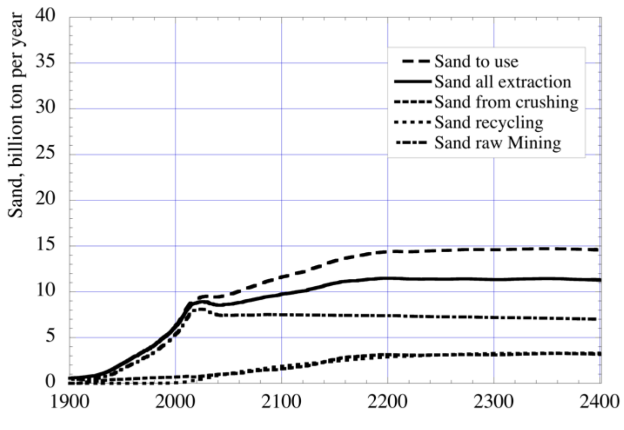
There are no published reserve and resources estimates of sand, gravel and quality stone for construction at the global scale, thus we can give no proper reference for it. However, estimates were made anyhow, based on the available information (Singer 1993, 1995, 2007, 2010, 2011, 2013, Harben and Kuzwart 1996, Chen et al., 2006, Kogel et al., 2006, Korre and Durucan 2007, Bohlen 2009, Velegrakis et al., 2010, Kostka 2011, Krausmann et al., 2009, Langer 2011, Bliss et al., 2012, Merwerde 2014). The resources are hypothetically huge, but a large portion are economically and geographically unavailable because of lack of transport infrastructure or being located unavailable by occurring in built up areas, with other types of major infrastructure, or conflicting with agricultural use. Further significant amounts of sand and gravel are located in protected areas and natural reserves or are physically, technically or logistically challenging to extract. Thus, a significant amount of the resources is currently out of reach because of difficulties of extraction, remoteness, conflicting land-use and significant parts are socially unavailable. The available resources have thus been estimated based on the available information. If we assume that the materials are exploited at a rate of 2-3% of known reserves, suggests reserves of about 1.6-2.5 trillion ton. Resources are at about 5 times known reserves for many other resources, adopting this ratio for sand, gravel and rock materials suggest an extractable resource of about 8-12.5 trillion ton. Table 4.2.1.2 shows the typical composition of concrete. Table 4.2.1.3 shows an overview of primary and secondary mining of resources. We have assumed that 1 km3 of calcite limestone is 2.7 billion ton of stone. Yield is the % weight of the content that end up as final product. Table 4.2.1.4 shows the Ultimately Recoverable Resource (URR) estimates for the SGS model. The resources include both known reserves and different types of resources we have reasons to assume are there, but where the exact location and quality has not yet been identified. Only industrial quality for sand and gravel have been considered. For stone, only stone prime quality that can be manufactured to quality building stone has been assessed. For limestone, we have done a very preliminary resource estimate (Bliss et al., 2012). The reserve and resource estimates are shown in Table 4.2.1.1, Table 4.2.1.3 and Table 4.2.1.4. The resources were estimated looking at area underlain by limestone rock.

Model simulation results

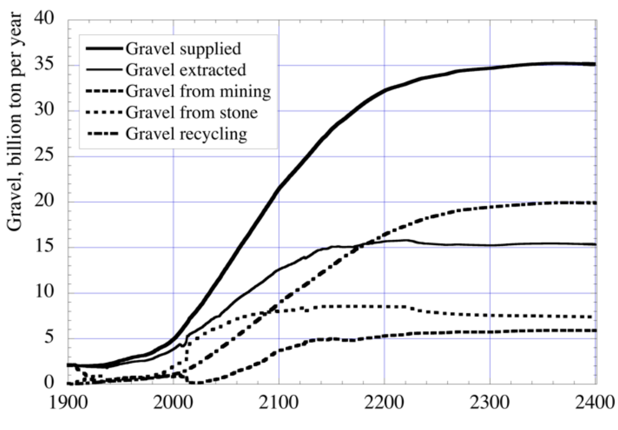
The SGS model was run from 1900 to 2200. The results of the SGS model simulations are shown in the Figure 4.2.1.8 to Figure 4.2.1.13 for the time period from 1900 to 2200 under business-as-usual conditions.

Figure 4.2.1.8 shows the model outputs for extraction, supply, recycling for (a) sand, (b) gravel, (c) stone and (d) all stony materials aggregated. The fit for the stone production data to the simulation is r2=0.52. Records for validation are available from the USGS database available on the web (USGS 2015). The amounts shown are in billion ton of material, the flows are in billion ton of material per year.

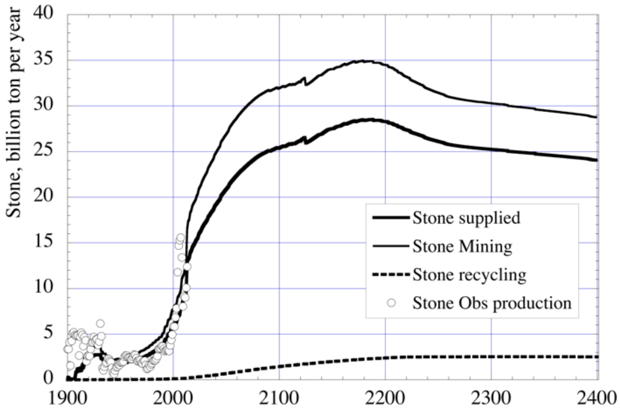
Figure 4.2.1.8. The model outputs for mining, from crushing and as by-product from sand, gravel, stone or concrete, supply, recycling, for (a) sand, (b) gravel, (c) stone and (d) concrete. The amounts shown are in billion ton of stony material, flows are in billion ton of material per year.



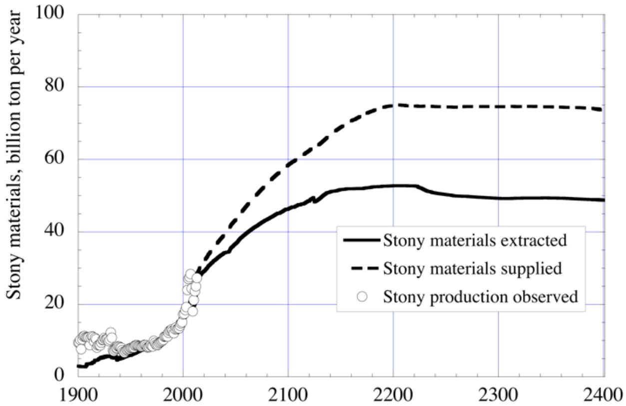
a: Sand production



b: Gravel production



c: Stone production

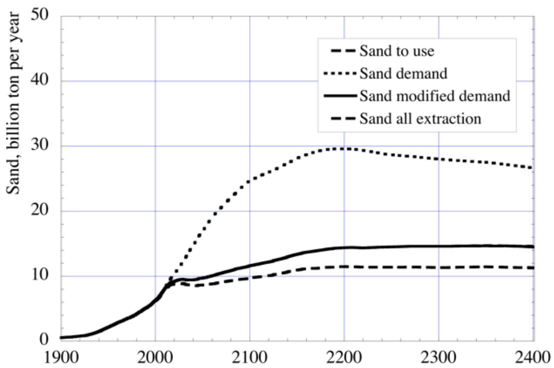


d; Production of stony materials

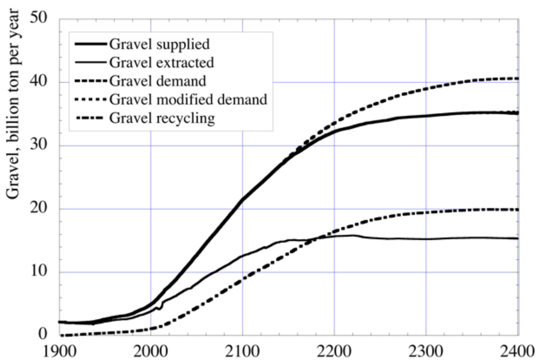
Source: Own Figure, Lund University/University of Iceland

Figure 4.2.1.9 shows the model outputs, market demand and modified demand for (a) sand, (b) gravel, (c) stone for construction and (d) all stony materials aggregated. The circles represent the observed data. The "observed data" in this case are very uncertain estimates, and the available number are not properly published and substantiated, thus the validation is only qualitative. The simulations seem to behave correctly; the simulation fits the observations on global total stony materials produced quite well, if we can assume the available data is valid. The simulation of total stony materials extraction seems, likewise, to fit the observations quite well, the correlation coefficient is r2=0.73. From the diagram, we can see that whereas we will not run out of physical supply of sand, gravel and stone (Hard scarcity), we will encounter increased prices and a peak production followed by a near constant production, suggesting future soft scarcity.

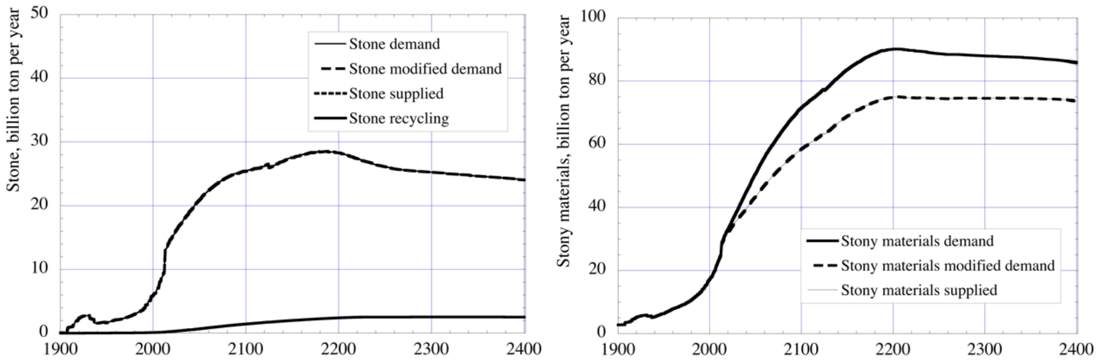
Figure 4.2.1.9. The model outputs market demand and degree of supply sufficiency (a) sand, (b) gravel, (c) stone for construction and all stony materials aggregated (d). In diagram (d), line 5 represents the observed data it should be ignored after 2015. The amounts shown are in billion ton of material, the flows are in billion ton of material per year.



a: Sand



b: Gravel



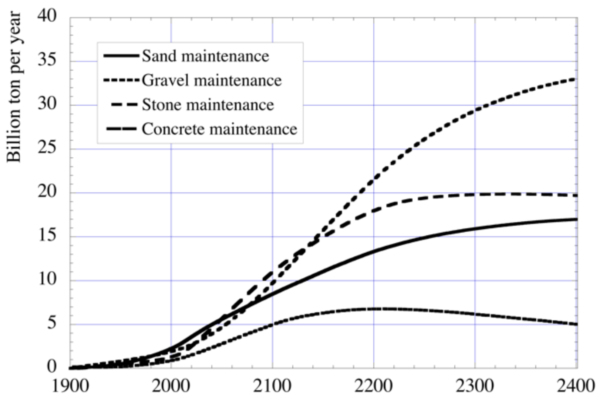
c: Stone

d: All materials

Source: Own Figure, Lund University/University of Iceland

Figure 4.2.1.10 shows the model outputs for maintenance of the infrastructures built with stony materials.

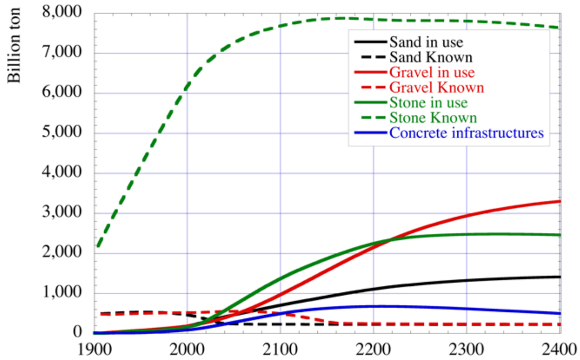
Figure 4.2.1.10. Use of stony materials for maintenance of infrastructures as estimated by the model for sand, gravel, stone and concrete infrastructures.



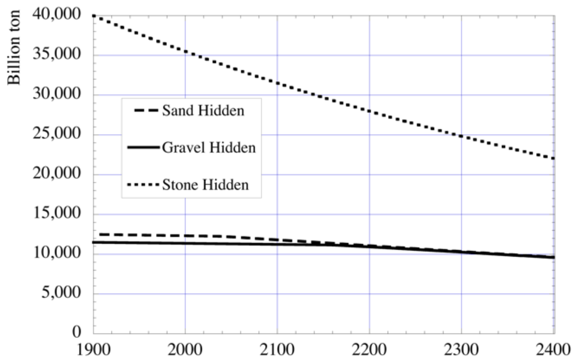
Source: Own Figure, Lund University/University of Iceland

Figure 4.2.1.11 shows materials (sand, gravel, stone) in use in society and known reserves (a) and hidden extractable resources (b). The stock-in-use is important for the maintenance flows. It can be seen that we will not run out of sand, gravel or stone in a very long time. But that if the demand stays on a high level, it will eventually be a finite resource that can be depleted.

Figure 4.2.1.11. The model outputs for materials (sand, gravel, stone) as known reserves and stocks-in-use in society. The flows are in billion ton of material per year. In concrete, the weight of the concrete itself is also included. Diagram (b) shows the hidden resources declining slowly with time.



a: Known reserves and stocks-in-use

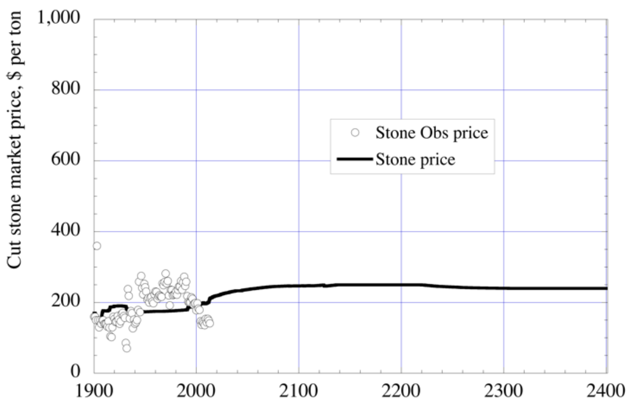


b: Hidden resources

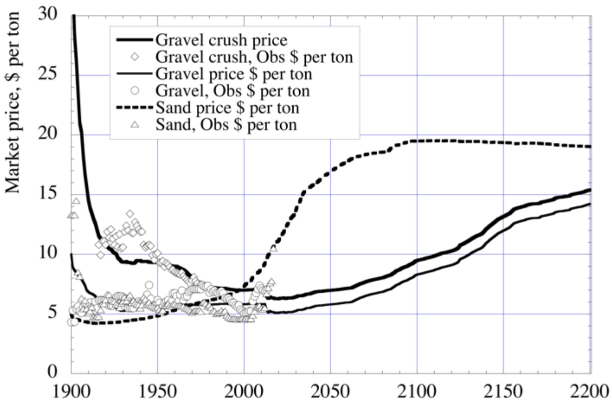
Source: Own Figure, Lund University/University of Iceland

Figure 4.2.1.12 shows the model outputs for price in the market, the simulated and observed price for stone, gravel and sand. The comparison with observed data for price shows a satisfactory fit. The results show that the world is not running out of stone, but that we may run out of sand, gravel and cut stone of the right quality for many purposes. The model suggests future increases in price for sand, gravel, crushed aggregates and cut stone. The world market price reconstruction for sand and cut stone is quite successful, for crushed stone to sand and gravel less so, even if the order of magnitude is correct. This is done under the assumption that there is a functioning global market for sand, gravel and cut stone. There is such a global market, but the prices show huge variations locally, depending on transportation costs and local cost conditions. Considering the inherent inaccuracies one would think was in this approach, it is amazing how well the produced amounts and global prices are simultaneously modelled. The prices are expressed as 1998 inflation-adjusted dollars.

Figure 4.2.1.12. Commodity market price. The model outputs for the simulated and observed price for stone (a), gravel and sand (b). The observed price for sand, gravel and stone are shown.



a: Cut stone



b: Sand and gravel

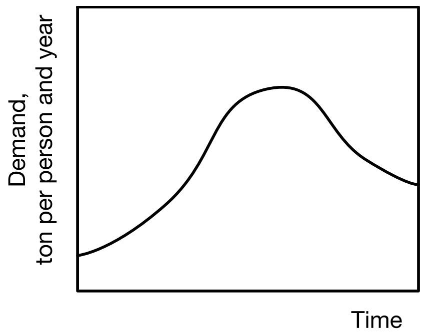
Source: Own Figure, Lund University/University of Iceland

4.2.1.6 Discussion

The peak shape of the curves

The curves have a behaviour consisting of a period of strong growth, an end of growth and stabilization at a stable level. The reason for this is the way demand is driven by population and the typical demand per person and year curve as shown in Figure 4.2.1.13.

Figure 4.2.1.13. The demand per person follows a typical pattern with rise, peak and decline to a maintenance level. Most industrialized countries are in the decline to maintenance phase whereas developing countries are on the rise or close to the peak, depending on what stage they are in their development.



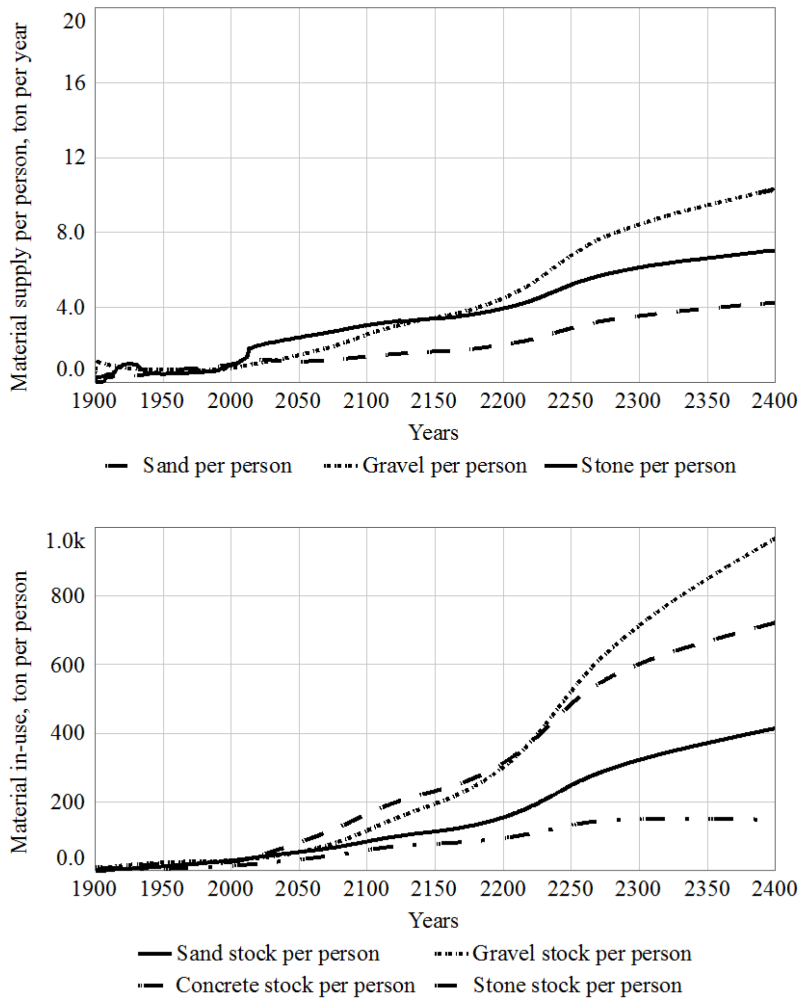
Source: Own Figure, Lund University/University of Iceland

Figure 4.2.1.14 shows the material supply expressed as ton per person per year, and as stock-in-use per person. It can be seen that the use per person cannot grow exponentially indefinitely. The following parameters affected the shape of the curve the most:

1. The population over time development, and with it general consumption. This is an important determinant for demand.
2. The shape of the demand per person and year curve, and the approach to infrastructural saturation. The shape of this curve was taken from the scientific literature and UNEP reports from the International Resource Panel.
3. The prospecting activity level, determining how fast “hidden” is transferred to “known”. As long as prospecting and finding matches the extraction, production can be kept up or grow. If not known” will decline and extraction with it.

It also shows that 2020 supply level can be kept for a significant long time. Sand, gravel and cut stone run into soft scarcity because of rising prices as the response to increased demand. Cut stone approaches physical scarcity for short periods after 2100. The world will not run out of sand, gravel or cut stone, but high prices will limit demand through feedbacks. The amount of these materials extracted annually are very large, and as the price increases, it will be foreseeable that resources unavailable under present social conditions or restrictions to extraction. These conditions and restrictions may be challenged and brought under pressure to be released for exploitation as the price increases. In the model a technology development curve is used, this was adapted after results like those presented by Gutowski et al., (2014).

Figure 4.2.1.14. Material supply expressed as ton per person per year, and stock-in-use as ton per person.



Source: Own Figure, Lund University/University of Iceland

Uncertainties and certainties

The amounts of sand, gravel and stone on the Earth are truly enormous, but what part of this exists in extractable form is dependent on materials having the desired mechanical or chemical properties. Major uncertainties in the output from the model are associated with the lack of reliable global sand, gravel and stone resource estimates. Furthermore, and this need to be emphasised, the very long time perspective of the model runs as such is a complicating factor. Very long-term demographic, economic and technological development are obviously hard to foresee on century scales. Likewise, what is extractable resources from a social perspective will be highly dependent on social and environmental development. To be very clear, we do not attempt to prognosticate future resource use or availability – we attempt, tough, to provide a scenario assessment, based on a business-as-usual run, of potential future resource scarcity horizons. We attempt to do this with a simplified model to obtain a sense of orders of magnitude and relevant times involved. The model was not intended to capture all details, and it seems to work surprisingly well at the global level.

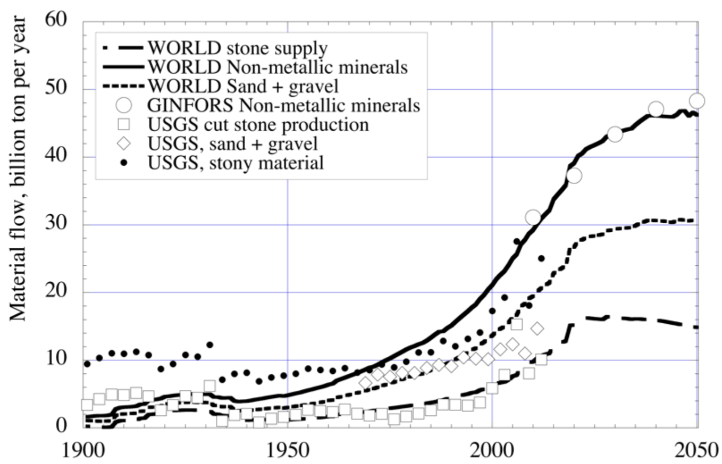
The only performance measures available for validation are: the ability to predict the past mining trajectory, and the market price (Figure 4.2.1.12) for these commodities. When considering the difficulties in the input data, and the challenge of getting the market response curves correct, the model performs surprisingly well (Figure 4.2.1.8 and Figure 4.2.1.16).

While we can securely assume mass balance principles to be valid at all times which is adding robustness to mass balance based models like the SGS model, other factors are less straightforward. It appears more than likely that values with regard to landscape, nature conservation, recreation and perceived resource needs and balance and trade-offs between different needs, societal actors and sectors will be significantly different from today in the long perspective and time-frame covered by our calculations. We are not attempting to model such societal change. Technological change (substitution, increased resource efficiency etc.) will have impact, but probably less so than changes in values, on resource availability as sand, gravel and rock resources to a large degree are used as bulk materials. The apparent model output should be seen as a representation of an illustrative future under assumed "business-as-usual" conditions rather than a projection of a likely future as such. Nevertheless, seen as a scenario, the results allow a better informed discussion about the magnitudes of future resources, their long-term use and sustainability than the alternative of no attempt at assessment.

Testing the model on data

The success when testing the model suggests that the SGS model already now has about an adequate level of complexity and that the key parameters seems to have been set at appropriate value. Figure 4.2.1.15 and Figure 4.2.1.16 show the result of the SGS model integrated into the WORLD6 model and tested against observed data on production of sand, gravel and rock materials as reported by the United States Geological Survey Minerals database for 2015. The forecasts made with the GINFORS model was also tested. The test shows that the SGS model performs very well within the WORLD6 model, and that the outputs are consistent with the GINFORS outputs.

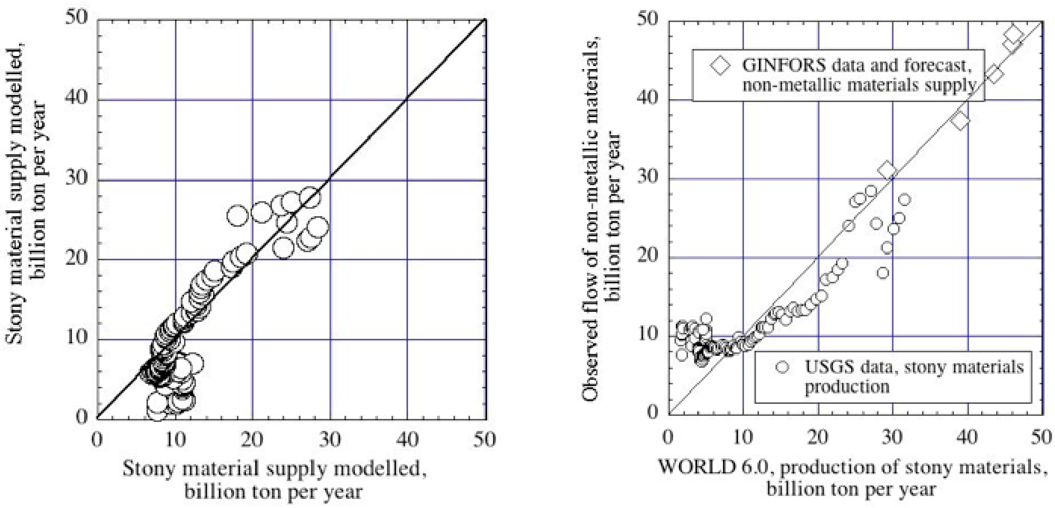
Figure 4.2.1.15. The SGS model was integrated into the WORLD6 model and tested against observed data on supply to the market as reported by the United States Geological Survey Minerals database for 2015. The plot shows that the model is able to reconstruct the right orders of magnitude for the production when compared to data



Source: Own Figure, Lund University/University of Iceland

Figure 4.2.1.16 shows a plot of modelled production of rock materials versus observed total rock materials extraction amounts using the SGS model as a stand-alone model. The plot shows that the model is sufficiently accurate for assessing global production rates. Figure 4.2.1.16 shows the outputs from the SGS model when it is integrated into the WORLD model and tested against data. The correlation between the WORLD6 simulation and the USGS data is r2=0.76, which is better than the performance of the SGS model when it is run as a stand-alone model (r2=0.72). The consistency between the GINFORS forecast and the WORLD6 is r2=0.98 (Figure 4.2.1.16). The test of the SGS incorporated in WORLD6 against the GINFORS model outputs and the USGS data pooled together has the correlation coefficient r2=0.86.

Figure 4.2.1.16. A simple plot of modelled versus observed total stony material extraction amounts using the SGS model as a stand-alone model is shown in diagram (left). Diagram (right) shows the outputs from the SGS model when it is integrated into the WORLD6 model and tested against data.



Source: Own Figure, Lund University/University of Iceland

A successful test of an integrated complex model like the SGS model inside the WORLD6 model on field data, makes the discussion over what could be wrong from a theoretical point of view less relevant, giving more emphasis on simulation performance with respect to testing recorded extraction data. A next step, but outside the scope of this paper, will be to run the model through a number of sensitivity runs in order to assess the robustness and variability of the outputs. At this stage, it is obvious to the user that the results are quite sensitive to the demand created by aggregated per capita consumption as well as by maintenance of built infrastructures. It would be of priority to analyse the effect of different resource magnitudes (varying those shown in Table 4.2.1.2, Table 4.2.1.3 and Table 4.2.1.4) and extraction rates (varying those shown in Table 4.2.1.1).

Sand, gravel and rock scarcity

The used volumes of sand, gravel, crushed rock and stone are truly huge. Extracting, moving, crushing, shaping these materials at the present volumes requires large amounts of energy. When energy eventually becomes expensive and/or transport distances, i.e. transport costs, increase significantly the price of these products will go up. The curves in Figure 4.2.1.8, exhibits a rapid growth, stagnation and almost flat development for sand, gravel and stone with time. But from Figure 4.2.1.11, it appears that even it known reserves may decrease, hidden reserves are far from being exhausted. In fact, they seem to be able to last for several centuries. Hence, on a global scale stone for crushing stone to sand and gravel fractions appears not deplete significantly for the next centuries.

While there is no imminent prospect of stony building materials becoming globally scarce this is unlikely to be the case at other scales. Natural sand and gravel are indeed limited finite resources, and they may be excavated to depletion, in particular at a regional scale around populated centres. Locally, sand and gravel scarcity is already an observable fact (UNEP GEAS 2014, Ashraf et al., 2011, Ooijens 2014, Morrow 2011, OSPAR 2003, Ravishankar 2015), putting demand pressure for long distance supply into the global markets, and potentially causing environmental impacts where it is extracted.

The primary substitute for natural sand and natural gravel is industrially produced sand and gravel from crushed stone and rock. This is nearly inexhaustible from a point of view of having enough raw material, while not necessarily in the longer run as the long-term supply of sand and gravel from crush is likely to be limited by the future energy supply, availability of long range transport, ability to pay, availability of resources from a social context and not only rock quality. Sand, gravel and crushed rock long-distance transportability, between extraction site and final use, is for economical and energy reasons limited – 50 billion ton of stony material cannot be moved without significant use of energy and not without impacts of roads, noise and pollution. As the price increase, the feedback from price tend to reduce demand, thus implying “soft scarcity”. In the continuation, the prices may rise even more owing to competition with other land-use, prices for mining rights and further increases in energy pricing, leading to unaffordability; a transition to economic scarcity, rather than physical scarcity. In a distant future, the primary extractable resource may in practical terms end up being exhausted. Recycling rates remain low in the sand, gravel and stone use systems, much because of the low commodity cost. Currently the drive to recycle for economic reasons is not particularly strong, however, that will change when resource prices increase.

4.2.1.7 Policy implications

Given the long time-perspective of this study and the substantial uncertainties with regard to important data general policy recommendations would be premature. Still, we may speculate based on common sense, generic knowledge and evaluations based on our outputs.

We need to consider that management of the resources in question for practical and economic reasons operate at a regional as well as at a global scale. Scarcity increases efforts of globalization, shifting to other sources of supply and to increased efficiency, minimizing transient stocks. Nevertheless, there is an apparent agreement in the available publications that we are moving towards a possible risk of more widespread sand and gravel soft scarcity, at least regionally.

We think that increasing prices will drive the market towards more globalization, a process already in progress. The conditions and restrictions imposed on and limiting extraction are often rooted in the local communities and regions. The policy challenges will involve how these different scales interact, and on their relative strength.

We would suggest that there is a need to put some effort into getting better regional overviews of available reserves and resources. Such assessments need to include informal or illegal extraction which currently is omitted from official estimates. Similarly, better data on recycling and re-use of bulk materials are needed.

Further, we would suggest that there is a need to assess the large difference between what is actually present and what may be technically and socially viable to extract, as well as how this relates to available future energy. The amounts extracted, processed and transported run in the size of 30 billion ton per year, and the amount energy used for such a task is significant.

4.2.1.8 Conclusions

The developed model performs well and, given input data constraints and uncertainty, successfully reproduces production and global market price when compared with independently observed data. The modelling of sand, gravel and stone resources have reached a level of complexity with this model where few further improvements can be made until better data become available in open scientific sources. At present, this is very limited, making assumptions necessary. The simplifications done to the model from the full concept still retain a good level of performance and show the same dynamics as the full model, but with better stability. For this reason, the simplified version is the best practical modelling option.

The simulation, under assumed business-as-usual conditions, show that cut stone production will reach a maximum level about 2020-2030 and slowly decline after that. The cause for this is that demand exceed extraction as well as slow exhaustion of the known reserves of high quality stone. Sand and gravel also show peak behaviour and reach their maximum production rate in 2060-2070. The reason for the peak behaviour is partly driven by an expected population maximum in 2065 and later slow decline and increasing prices for sand and gravel, limiting the demand.

The developed SGS model appears to perform well enough when compared to observations to justify for it to be included in the WORLD6 model. The outputs from the SGS model when embedded in WORLD6 shows a slightly better performance against observed data. The outputs appear to be consistent with the GINFORS forecasts in the interval from 200-2050.

While in a global perspective supply may seem inexhaustible supply and availability is already a growing problem at a local to regional scale, signalling that global trade with sand, gravel and stone will continue to increase. We need better data on production rates, use and recycling to better assess risks of regional scarcity, as well as a model of this kind divided into different regions. The state of the research is not at present at a stage where this is possible without substantial research funding to support it. The SGS model, given better data, could be used for this as it could be down-scaled to regions.

4.3 ENERGY SUB-MODULE

The WORLD model attempts to model the world systems in a way similar to what the WORLD3 model once did (Meadows et al. 1972, 1992, 2004, Randers 2013), however, with a more complete description of each individual and significant resource.

The energy sub-module consists of:

1. Fossil fuels

a) Hydrocarbons
b) Nuclear fuels (work in progress – not subject to this report)

2. Technology-based alternative energies (work in progress – not subject to this report)

    a) Photovoltaic
    b) Windmills
    c) Geothermal power

3. Traditional alternative energy sources (work in progress – not subject to this report)

    a) Hydropower
    b) Biofuels

4.3.1 FOSSIL FUELS – Hydrocarbons

4.3.1.1 Introduction

In principle, it can be demonstrated that four resources (hydrocarbons, iron, copper and phosphorus) are important in determining the fate of our civilization (Gever et al., 1986, Campbell and Laherrere 1998, Campbell 2013, Heinberg 2001, 2011, Greer 2008, Bardi 2009, Morrigan 2010, Sverdrup et al., 2013, 2014a,b, 2015a,b, 2017a). Of these, hydrocarbons have taken the main role as the largest source of energy. We do well to remember that phosphorus, copper, iron and hydrocarbons (oil, coal, gas) all depend on fossil deposits, that are not renewed at an significant rate. Under business-as-usual, only iron has a significant (30-40%), but still insufficient recycling rate. For phosphorus, present recycling rate is about 16-20%, which means the circularity of the phosphorus use is insignificant. (Ragnarsdottir and Sverdrup 2011, Sverdrup et al. 2011, Sverdrup and Ragnarsdottir 2014). Hydrocarbons, oil, gas and coal, are to the major part irreversibly consumed when used, they are not recycled at all. All other resources can be said to being dependent on or inferior to these. Thus, the future of energy for societies across the world, is tightly connected to the future of society (Gever et al., 1986, Heinberg 2001, Greer 2008, Bardi 2009, Morrigan 2010, Sverdrup and Ragnarsdottir 2014).

4.3.1.2 Objectives and Scope

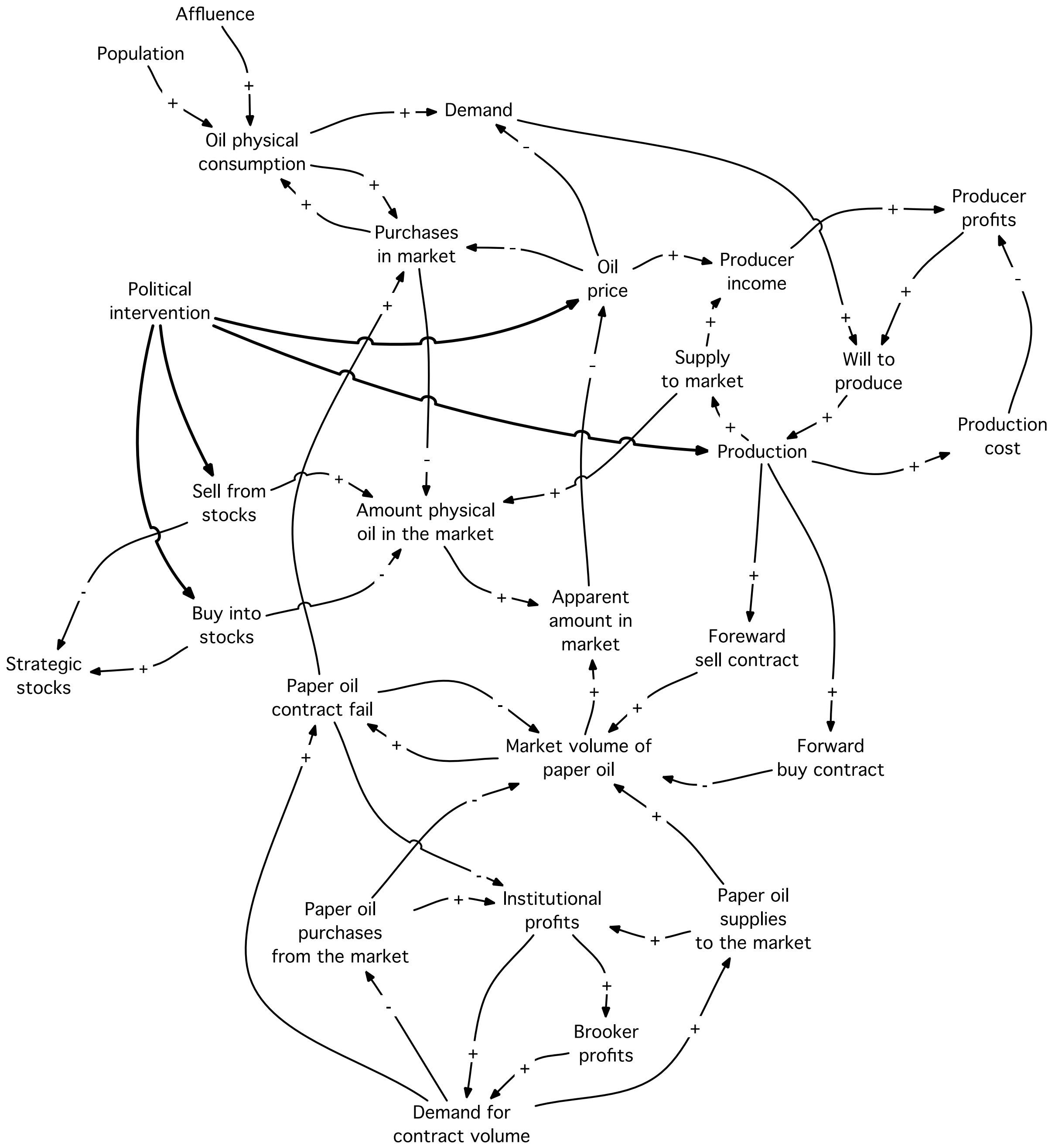
Create a simple fossil energy extraction model to be implemented in the WORLD model. The submodels for oil, gas and coal should reflect variable reserve quality and extraction cost, use reserve sizes as realistic as possible, and be easy to parameterize and run. The extraxtion rate should be demand driven and reserve supply capacity and extraction efficiency dependent.

4.3.1.3 Model description

The fossil hydrocarbon model has three parts, one for oil, gas and coal each. There is a separate module for nuclear fuels mining and nuclear energy generation. It involves Uranium based nuclear fission Traditional U235 fission, new breeder technology using reprocessed uranium-based nuclear fuel waste and redundant nuclear arms materials and thorium based breeder technology. We have not included fusion, as we consider the technology too be too far away from any feasible energy producing process so far.

Figure 4.3.1.1 shows the causal loop diagram for the oil market. The gas and coal market has the same mechanisms involved. The price is determined by two factors, that it must stay above the production costs and by the amount in the market, which in term depend on the balance between deliveries into the market from production and the shipments from the market in response to world market demand. The price drives the urge for energy savings, installation or alternative or renewable energy kinds and heat recovery from society. The political intervention is a substantial part of the market, making the energy price difficult to model. Most of the time, the market operates as a free market, but only in terms of within the conditions set by the existing political intervention. Energy is a strategic asset, especially fossil hydrocarbons, where considerations about spheres of power-interests, military power and political influences play important parts.

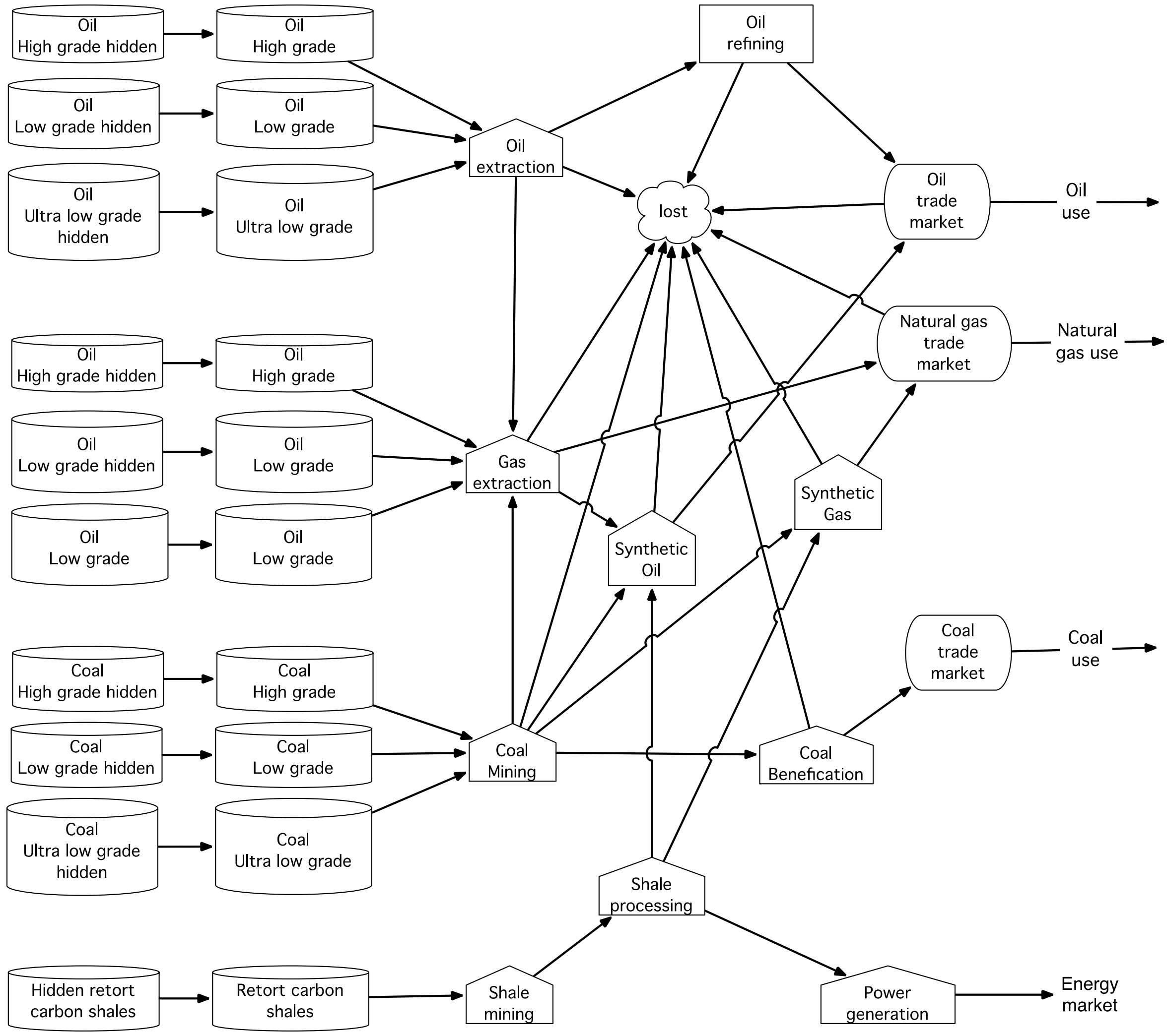
Figure 4.3.1.1. Causal loop diagram for the oil market.



Source: Own Figure, Lund University/University of Iceland

Figure 4.3.1.2 shows a flow chart showing the internal reserve quality structure inside the hydrocarbon energy module. Oil, natural gas and coal has three reserve qualities. Some natural gas is coproduced with conventional oil. Natural gas can be converted to oil. Coal can be converted to oil or gas when the coal price is less than 60% of oil, this is attractive. Retort shale can be converted to gas or oil or combusted directly to heat and electricity. The profit is estimated as the difference between income from supply and the cost of extraction. The profit is used as the driver for extraction and production (From McGlade and Ekins 2015).

Figure 4.3.1.2. A flow chart showing the internal reserve quality structure inside the hydrocarbon energy module ENERGY.



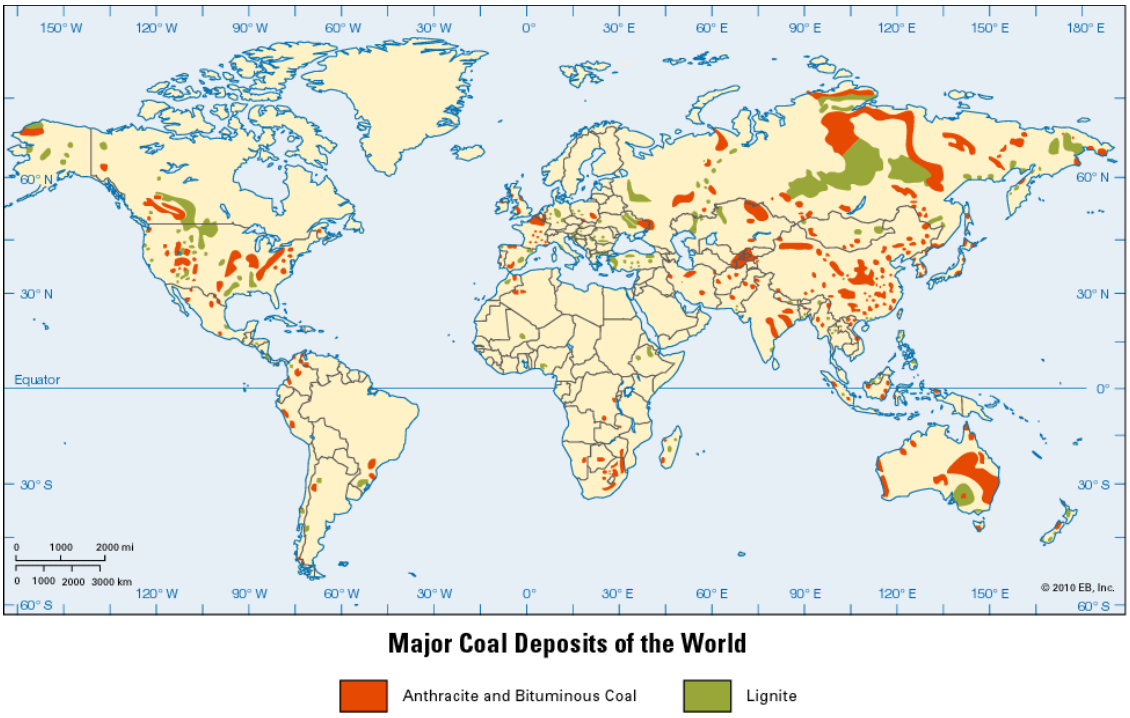
Source: Own Figure, Lund University/University of Iceland

4.3.1.4 Reserves and resource analysis.

We use literature studies for assessing the recoverable resources for oil, heavy oil, tar, tight oil and shale oil, natural gas, shale gas and tight gas, and had coal, soft coal, brown coal, shale carbon resources (Owe et al. 2012, Al-Husseini 2010, BP statistics 2013, OPEC 2013; Leonard 2012, Campbell 2009, 2013, Campbell and Laherrere 1998, Aleklett et al., 2012). Many of the assessments of "how much" focus on proven reserves, extractable at the present moment. This runs the risk of significantly underestimation what is ultimately recoverable in the period 1900 to 2400. Fewer studies are available on the total resource, and how much of that is technically extractable without too much consideration of monetary cost. Another limitation is the energy requirement for extraction. When that exceeds the amount energy gained in the extraction, then it does not pay off to continue. This is called the concept orf EROI (Energy return on investment) (Hall et al., 2001, 2009, Hall 2008).

Figure 4.3.1.3 shows the map of the coal deposit regions of the world. Coal is the hydrocarbon with the largest stock of energy available. Coal, gas and oil reserves and resources are fairly well correlated on a large scale.

Figure 4.3.1.3. Map of the coal deposit regions of the world.



Source: http://www.goldendragoncapital.com/major-coal-deposits-of-the-world/

We assess the ultimately recoverable resources (URR), the presently extractable reserves, potential resources that may become reserves, division between presently known and estimated as hidden, all taken from the literature, from scientific articles and from corporate information (Heinberg 2001, Singer 2010, 2011, 2013).

The estimates of URR vary significantly with time, and this is depending on many factors. Firstly, assessing the size is difficult, as the reserves have not been completely mapped. The reserves may also be adjusted based on renewed assessment of how recoverable a resource is, as well as be reassessed based on purely political aspects. It needs not signify that extraction facilities are in place and operative. We have used the estimates made by the trio Jean Laherrere, Colin Campbell and Kjell Aleklett as the main guidance, but considered other estimates as well.

The available data was closely inspected for inconsistencies and averages. Many of the sources we use are at times inconsistent and we have to make expert judgment about what are the most likely parameter values to use. It is important to realize that the maximum production occurs when the peak in energy comes, not the maximum of ton material pumped. Table 4.3.1.1 shows an asssessment of anticipated global resources by EIA for 2012 and 2013, with no regard of extractability. Cost of this is not recoverable for several reasons. Some are behind an energy barrier, where it costs more energy to extract them than they contain, these resources are all not exploitable, regardless of how large they appear to be.

Table 4.3.1.1. Asssessment of anticipated global resources, with no regard of extractability. Cost of this is not recoverable for several reasons. Some are behind an energy barrier, where it costs more energy to extract them than they contain, these resources are all not exploitable with a net energy gain, regardless of how large they appear to be.

Source	Resources at date, billion ton oe						
	Oil high grade	Lower grades oil	Gas high grade	Lower grades gas	Coal high grades	Coal lower grades	Total
EIA 2012	765	540	200	648	2,500	15,500	20,122
EIA 2013	689	456	220	572	2,500	15,500	17,315
Extracted to date	250		95		163		508
Sum	1,395		887		18,163		

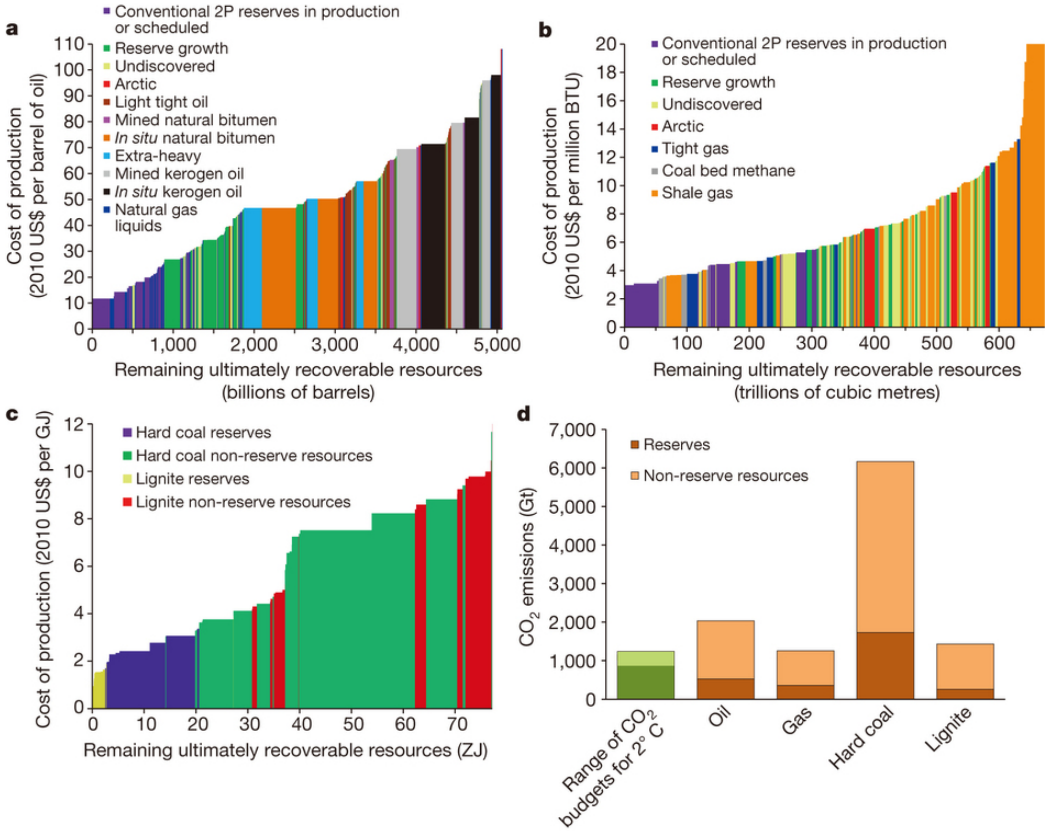
Table 4.3.1.2 shows fossil fuel extractable amounts in billion ton oil oe used as input data to the WORLD6 model.

Table 4.3.1.2. Fossil fuel extractable amounts, in billion ton oil oe for 2015, used to estimate URR from 1900.

Fuel	Known	Hidden	Sum	Extracted to now	URR from 1900
Oil	54	244	298	228	526
Gas	194	703	897	100	997
Coal	1,126	4,368	5,494	166	5,660
Sum	1,374	5,315	6,689	494	7,183

Figure 4.3.1.4 shows the cost to reserve distribution curves used to parameterize the model. These were used to set the value for the extraction cost for rich, high grade, low grade and ultralow grade reserves (known) and resources (hidden).

Figure 4.3.1.4. Cost to reserve distribution curves used to parameterize the model. These were used to set the value for the extraction cost for rich, high grade, low grade and ultralow grade reserves (known) and resources (hidden). The profit is estimated as the difference between income from supply and the cost of extraction. The profit is used as the driver for extraction and production (McGlade and Ekins 2015).



Source: McGlade and Ekins, 2015. https://www.nature.com/articles/nature14016/figures/1

Table 4.3.1.3 and Table 4.3.1.4 show two alterntive estimates of fossil fuel extractable amounts. Table 4.3.1.3 shows alternative 1 with low estimate of fossil fuel extractable amounts, in billion ton oil oil equivalents. This was used as input data to the low scenario in the WORD6 model simulation. Table 4.3.1.4 shows alternative 2 with high estimate of fossil fuel extractable amounts. This was used as input data to the high scenario in the WORD6 model simulation.

Table 4.3.1.3. Alternative 1: Low estimate. Fossil fuel extractable amounts, in billion ton oil oil equivalents. Input data to the WORD6 model.

Fuel		High grade	Low grade	Ultralow grade	Sums
Oil	Known	1	1	1	3
	Hidden	370	230	100	700
Gas	Known	1	1	1	3
	Hidden	270	160	60	490
Coal	Known	23	11	0	34
	Hidden	400	890	1220	2,510
Sum		1,120	860	540	3,040

Table 4.3.1.4. Alternative 2; High estimate. Fossil fuel extractable amounts, in billion ton oil oil equivalents. Input data to the WORD6 model.

Fuel		High grade	Low grade	Ultralow grade	Sums
Oil	Known Hidden	1 370	1 230	1 100	3 700
Gas	Known Hidden	1 432	1 360	1 200	3 992
Coal	Known Hidden	23 500	11 1,600	0 3,560	34 5,660
Sum		1,327	2,203	3,862	7,392

4.3.1.5 Results

Price curves

Figure 4.3.1.5 shows the oil market stocks and the oil price. In March 2016, the oil price was 41 $ per barrel and the Cushing5 market inventory was 69 million barrels of oil, corresponding to 9.4 million ton of oil. The Cushing comparative market inventory is the present stock compared to the running 5-year average. Cushing, Oklahoma is the largest oil-storage tank farm in the world. It has about 73 million barrels of working capacity (2016), about 13 % of total US storage. Several important oil pipelines converge there as oil moves from production sites to refineries on the Gulf Coast. The total world stock would be approximately 290 million ton of oil. That corresponds to about 19% of an annual production, or 10 weeks of global use. The Cushing oil stock index was recalculated to the actual stock in absolute numbers:

$$Cushing\ Relative\ Index = CS_{MAX} - \left(\frac{Cushing\ Oil\ Stocks - Average\ stocks}{Average\ stocks}\right) \qquad (4.3.1.1)$$

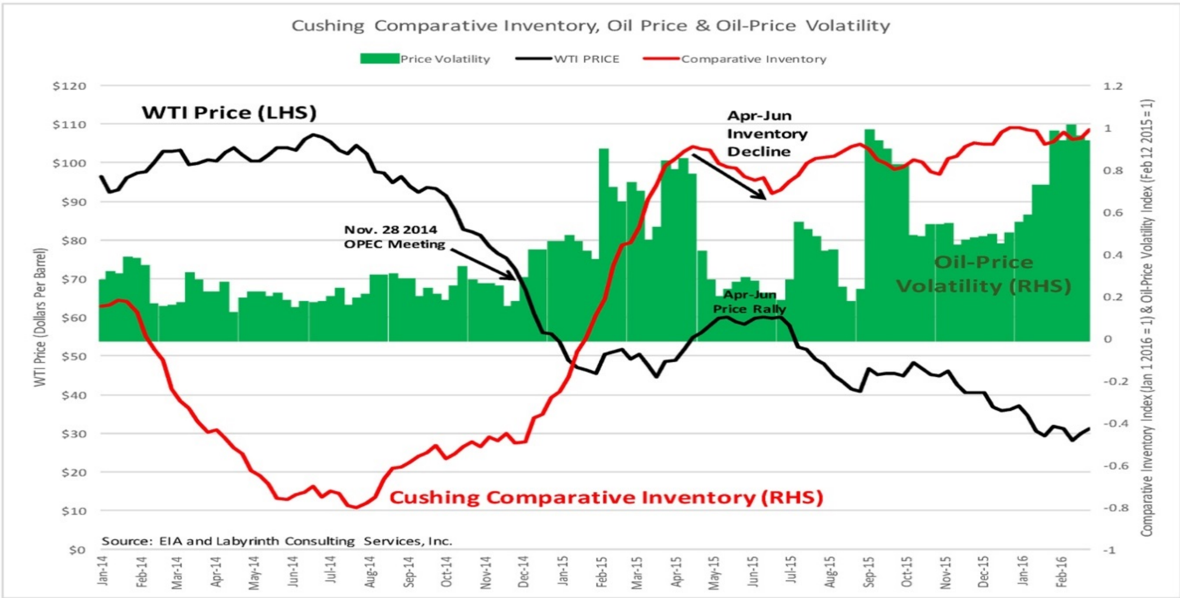
This equation can be inverted to:

$$\text{Cushing stocks} = \text{Average stocks} * \left(1 - (\text{Cushing Relative Index} - \text{Cushing Stock}_{\text{MAX}})\right) \qquad (4.3.1.2)$$

To scale up the Cushing Inventory data to global scale, we made a simple approximation. The Cushing storage capacity is about 13% of the US storage capacity. The United States oil consumption is about 25% of the global oil consumption. That gives us a factor of 31. We can generate the global inventory as approximately 31 times the Cushing inventory. There are some shortcomings in such an approach we should be aware of. At the lower end, we may expect it to work fine. When global stocks are low, the Cushing stocks will also be low. However, at the high end, the global stocks may continue go up, even if the Cushing Oil Storage facility in Oklahoma is full. Then it stops being a good indicator, and we should perhaps adjust the data for this. Thus, at the high end of the Cushing stock level, we expanded the global stocks a bit more for levels when the oil price goes below 35 $ per barrel and when the Cushing facility is more than 90% full. We can then draw up the price curve for oil.

[5] Cushing Oil Tank Terminal in Oklahoma is the largest oil storage facility in the United States and accounts for 13% of the US oil storage capacity. At present, it has a storage capacity of about 73 million barrels of oil.

Figure 4.3.1.5. The oil market stocks and the oil price. In march 2016, the oil price was 41 $ per barrel and the Cushing market inventory was 69 million barrels, corresponding to 9.4 million ton of oil.



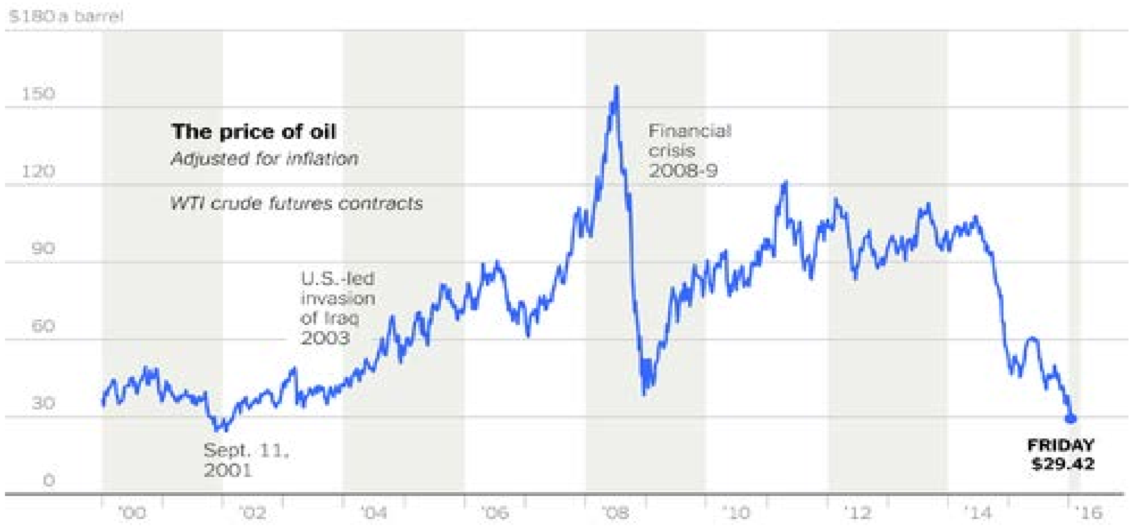
Source: EIA and Labyrinth Consulting Services, Inc. http://www.artberman.com/what-really-controls-oil-prices/

Figure 4.3.1.6 shows the level inventory at Cushing Oklahoma over time. Figure 4.3.1.7 shows the global oil price in $ per barrel. In the calculations we assumed that there are 7.33 barrels per ton oil.

Figure 4.3.1.6. The inventory at Cushing Storage Facility, Oklahoma during the last 10 years from 2006 to 2016.

Source: U.S. Energy Information Administration

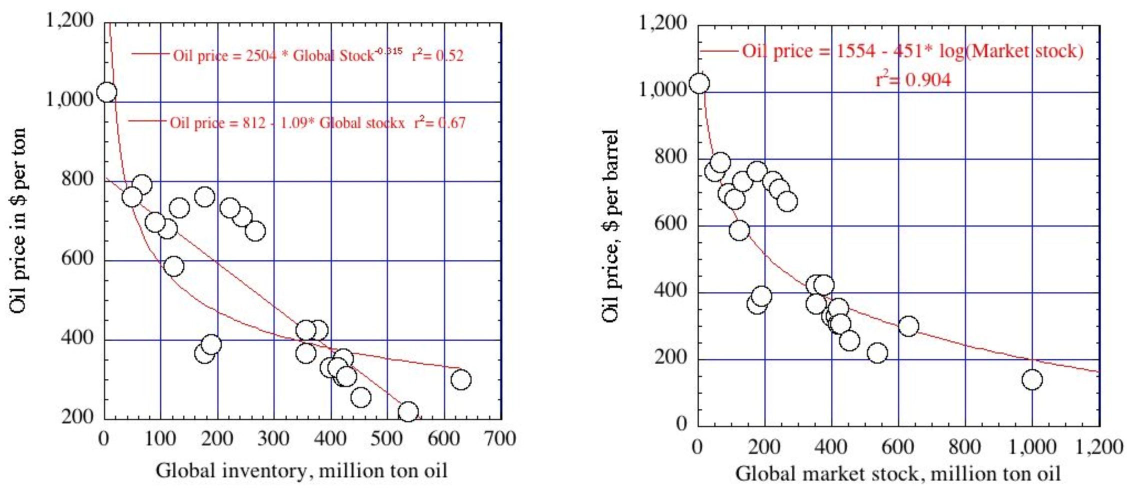
Figure 4.3.1.7. The global oil price in $ per barrel during 2000 to 2016. There are 7.33 barrels per ton of crude oil.



Source: U.S. Energy Information Administration

Figure 4.3.1.8 shows the empirically derived relationships between the global market tradable oil inventory and the global market price for oil. We get two possible oil price equations where the coefficient of correlation to the data, r2=0.52 for the power relationship and r2=0.67 for a linear relationship. For most commodities, the price curves are concave curves, and we would prefer to use such a curve for oil as well. Conceptually, we would like to have a curve that does not cross the axis and assume a price of zero.

Figure 4.3.1.8. Possible price curves for the global oil price derived from the Cushing Inventory data and the global oil price is shown in diagram (a). Diagram (b) shows the price curve when the Cushing Inventory Index has been scaled to global scale. We adjust for the oil price at very low and very high global oil stocks not really reflected in the Cushing Inventory Index.



a; First approach for oil price graph

b: Final oil price graph

Source: Own Figure, Lund University/University of Iceland

At zero global stock, the curve should asymptotically approach endlessly high price, at endless amounts of oil, the price should asymptotically approach zero. This estimating an oil price of 200 $ per ton (15 $ per barrel) at a global market stock of 1,000 million ton and a global oil price of 1,000 $ per ton (80 $ per barrel) at a global market stock of 50 million ton gives an r2=0.904, and the equation for the oil price in $ per ton oil (Figure 4.3.1.9):

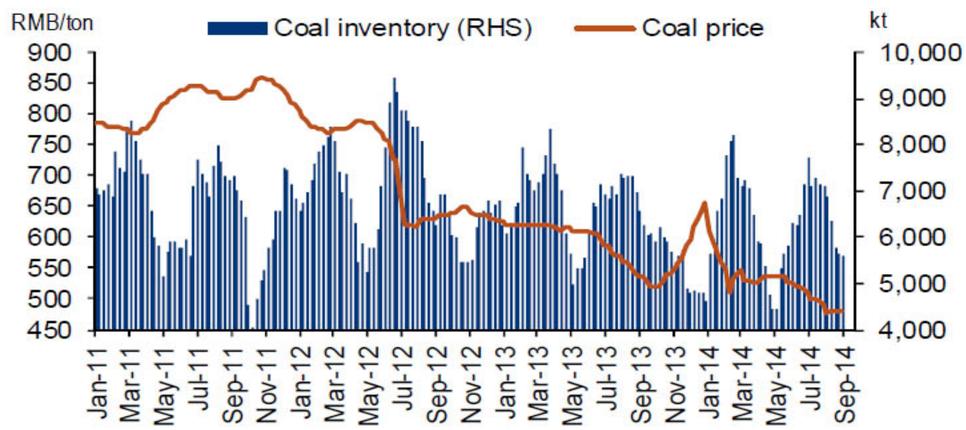
$$Oil\ price = 1{,}554 - 451 * log(Market\ stock) \qquad (4.3.1.3)$$

Or if we should be consistent with the pattern found for other commodities:

$$Oil\ price = 1{,}709 * Stock^{-0.26} \qquad (4.3.1.4)$$

Which has a r2=0.81. The correlation coefficient is r2=0.90 after the scaling and final data scaling modifications. Figure 4.3.1.9 shows the price for coal in RMB in China and the Chinese coal market stocks. The Chines available and tradable market stocks typically value in the range from 4 million ton to 9 million ton of coal. The global market stocks vary in the range from 220-400 million ton of bituminous coal, sub-bituminous coal market stocks vary around 250-450 million ton in total. It is not easy to find exact data, and the Chinese harbour trade warehouses are not well correlated to world inventory. From the diagram on Figure 4.3.1.9, some parts of the price response curve may be recovered.

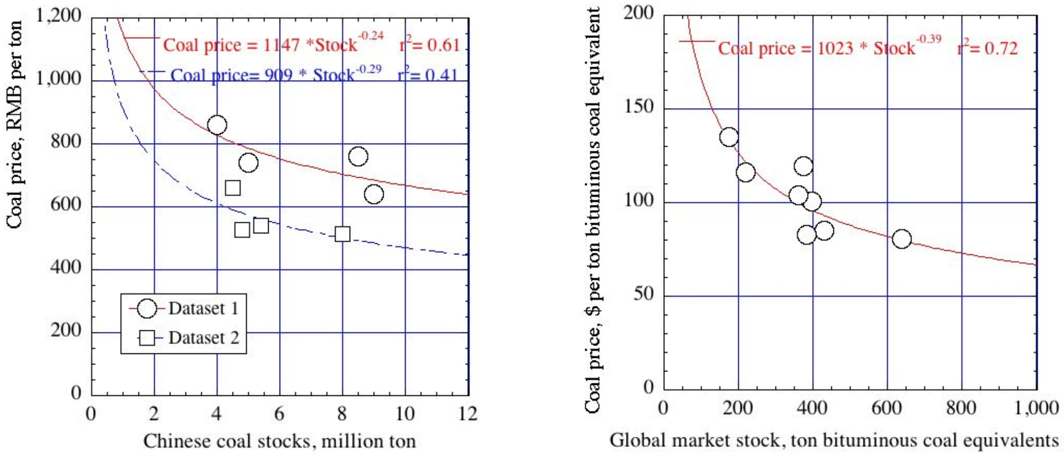
Figure 4.3.1.9. The price for coal in RMB in China and Chinese coal stocks. Data from several sources were used to create the price curve for coal. Bituminous and subbituminous coal was looked at.



Source: BofA Memil Lynch Global Research, Wind. http://climateerinvest.blogspot.com/2014/09/commodity-prices-in-china-continue.html

The global coal price curve is shown in Figure 4.3.1.10. Figure 4.3.1.10a, shows parts of the curves from raw data, Figure 4.3.1.10b shows the curve after some adjustments to compensate for different types of coal in the market and a shift due to inflation. The curve is approximate and has significant uncertainty associated with it. The correlation between the modified data and the coal price appear as having a correlation with data was r2=0.72 for the final curve.

Figure 4.3.1.10. Possible price curves for the global coal price derived from Chines data is shown in diagram (a). Diagram (b) adjusts for different coal qualities and scales it to global scale.



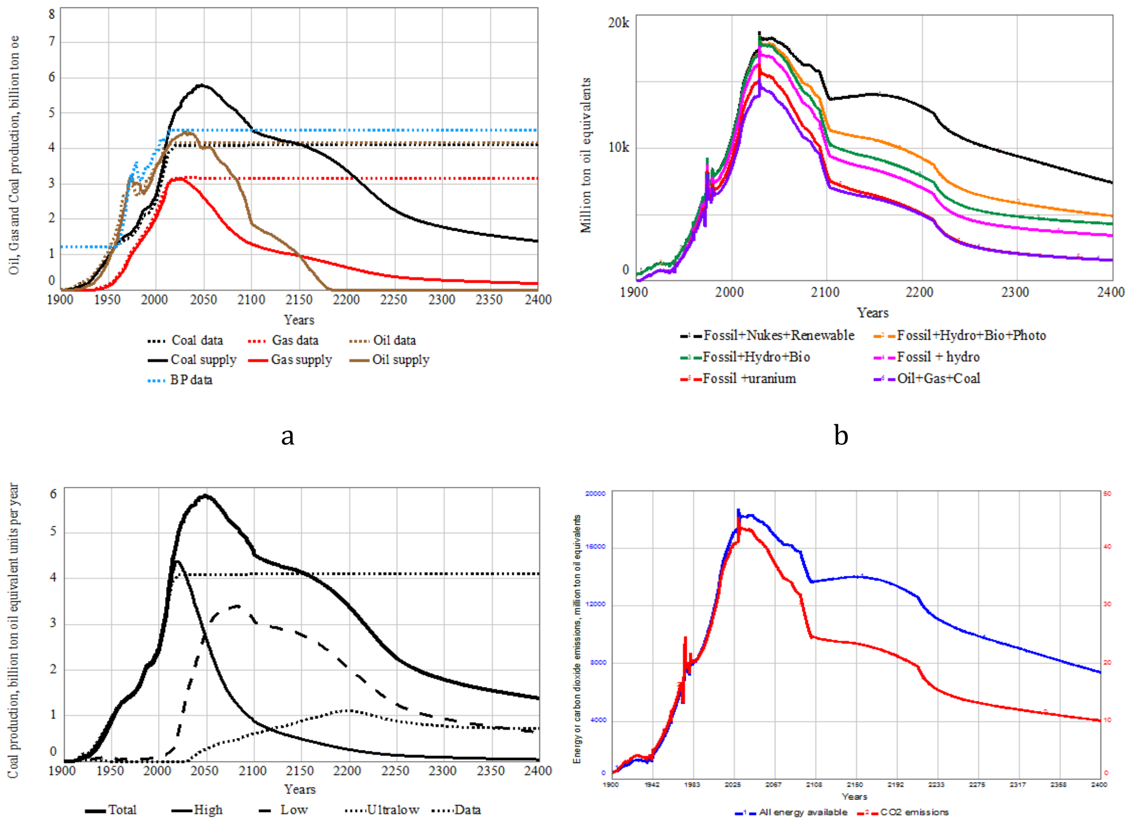
a; First approach for coal price graph b: Final coal price graph

Source: Own Figure, Lund University/University of Iceland

Simulations

Figure 4.3.1.11 shows the total global energy production 1900-2400 as reconstructed and predicted by the energy model. The main limiting factor in the model are the available reserves of fossil fuels. It can be seen that we now live in the time of peak energy. It is easy to show that this is a measurable fact, a consequence of the reality of applying mass balances.

Figure 4.3.1.11. Total global energy production 1900-2400 as reconstructed and predicted by the energy model. The input data from Table 4.3.1.3.

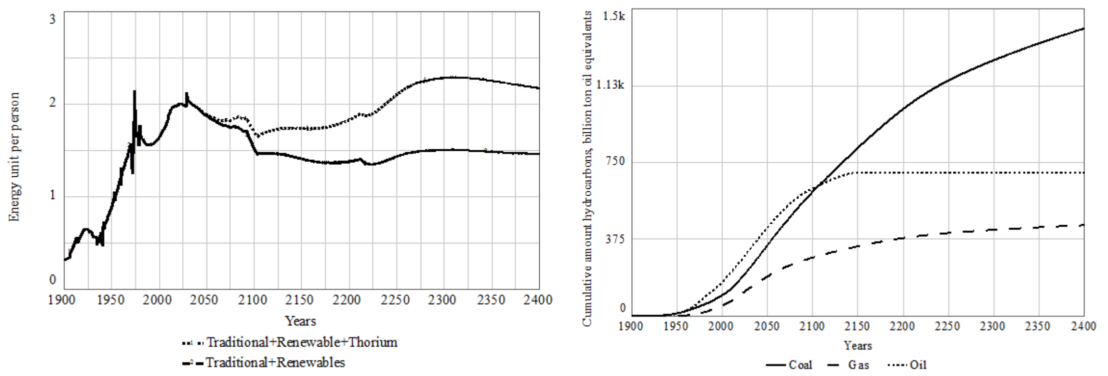


c d

Source: Own Figure, Lund University/University of Iceland

Figure 4.3.1.12a shows the available energy per capita in the world. The solid line represents all energy, fossil and renewable, but excludes thorium power. The dotted line includes energy based thorium power production. We anticipate that thorium energy will one day be built, based on need and a competitive price as well as the fact that it does not allow for nuclear arms material to be produced. Figure 4.3.1.12b shows the simulated oil and coal market prices. The price is calculated every day internally in the model from demand and supply.

Figure 4.3.1.12. (a) shows the available energy per capita in the world. The solid line represents all energy, fossil and renewable, but excludes thorium power. The dotted line includes energy based thorium power production. (b) The energy price for oil and coal as compared to the data for oil. The input data from Table 4.3.1.3.

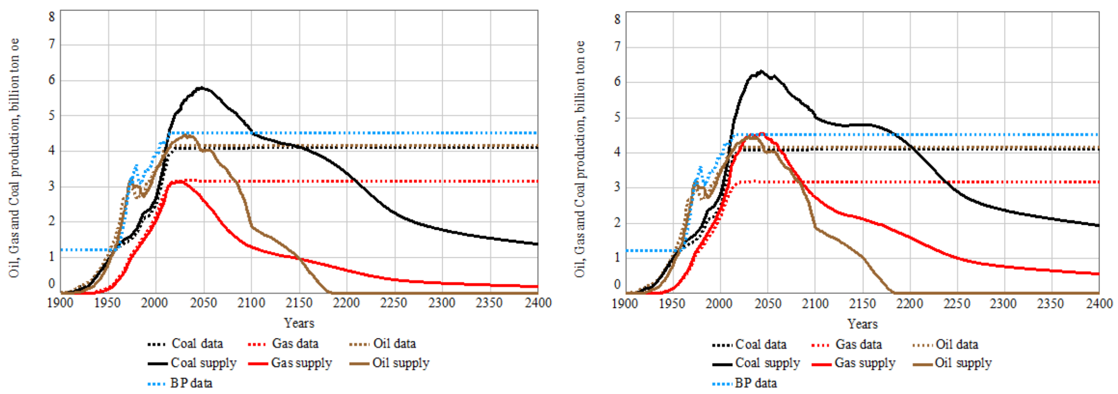


a b

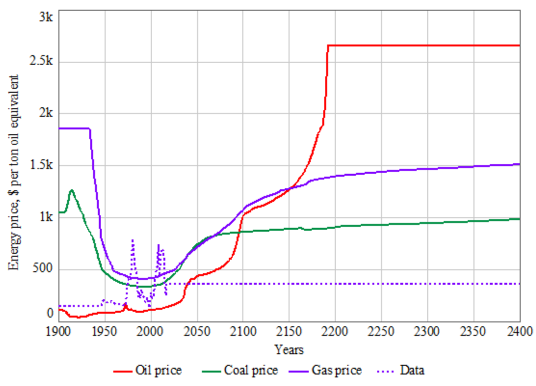
Source: Own Figure, Lund University/University of Iceland

Figure 4.3.1.13 shows a comparison between the high and the low resource scenarios outlined in Table 4.3.1.3 and Table 4.3.1.4. The difference is substantial, and has not a significant impact on price.

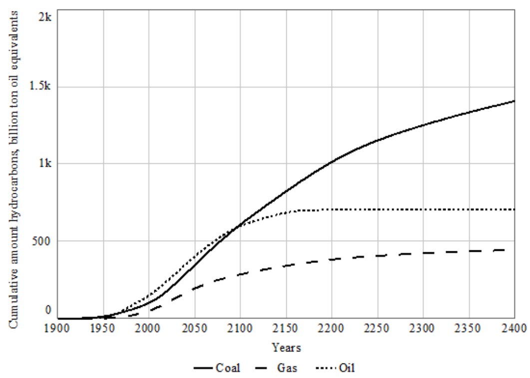
Figure 4.3.1.13. A comparison between the high and the low resource scenarios outlined in Table 4.3.1.3 and Table 4.3.1.4.

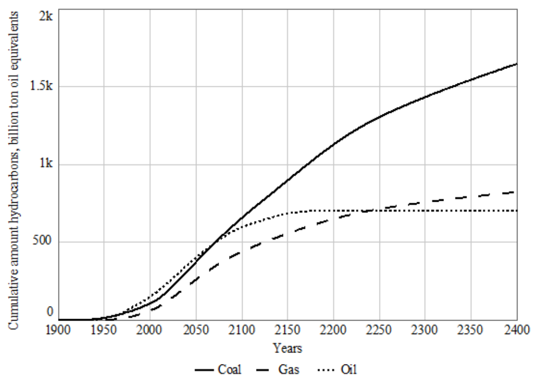


Low High



Low High





Low High

Source: Own Figure, Lund University/University of Iceland

4.3.1.6 Conclusions

Only a fraction of the world's detectable resources of fossil fuels is extractable, one of the major reasons for widely different estimates of the resources. There are several reasons for the differences in resource estimates are:

1. A significant part of the resources is hidden behind an energy barrier; the energy cost to extract is larger than the benefit of what can be extracted. When the extraction costs exceed the income, then extraction stops.
2. A significant part of detected resources is blocked hidden behind a technological barrier. The resources are located in such places, that the technology to extract it does not exist now nor in the foreseeable future. The resource is unreachable.
3. A significant part is behind a capital barrier. The resource is hypothetically extractable, but there is not enough capital to make the investments, nor the liquid money to buy the product at the price it would imply. Hydrocarbon fossil fuels are out of reach when the oil price reaches 1700 $ per ton oil equivalent.
4. A significant part of the resources is hidden behind an environmental destruction barrier. The resource is of such a poor environmental quality, or the extraction and use have such large environmental detrimental impact, that it remains unfeasible.

Thus, the world does not run out of fossil fuels because of lack of fossil carbon in the ground. The production comes to a stop because the implications of continuing extraction are unacceptable.

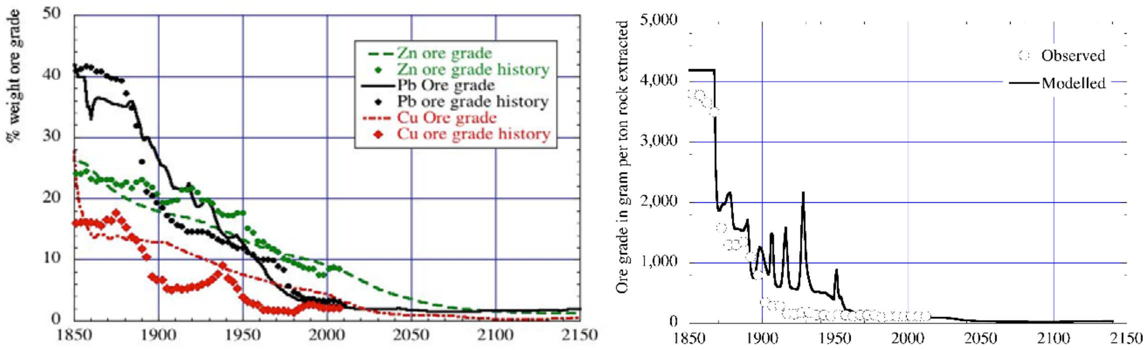
5 OVERALL CONCLUSIONS

5.1 Challenges in the coming decades

5.1.1 Decreasing ore grades in many metals and minerals

Figure 5.1 shows the simulated and observed ore grades for zinc, lead, copper and silver using the WORLD model. The same pattern is observed for gold, nickel, niobium, platinum and uranium (Northey et al. 2014; Sverdrup, Koca, and Ragnarsdottir 2014). In addition to the metals shown in the figure declining ore grades are also expected by WORLD model runs in the near future for phosphorus and metals such as iron, chromium, manganese, bauxite, niobium, molybdenum, gold and uranium. This indicates that the costs of extraction of metals and materials will potentially go up dramatically, as the ore grades go further down in the future. The energy costs of extraction rise exponentially when ore grades decline.

Figure 5.1. Simulated and observed ore grades for copper, zinc and lead (left) and silver (right), using the WORLD model

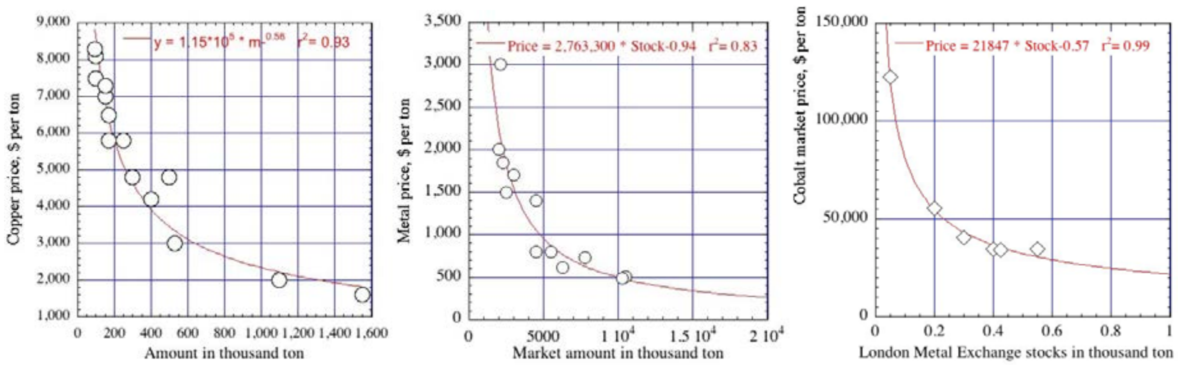


Source: Own Figure, Lund University/University of Iceland

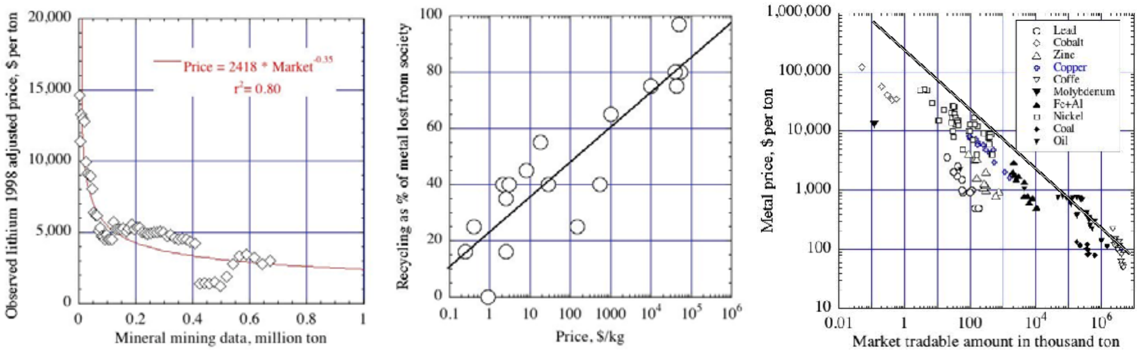
5.1.1.1 Modelling prices for market feedbacks

In order to model market prices dynamically, we used curves in the simulations, as given in Figure 5.2 showing the relationship between metal market availability in general and market price. The solid line is where we think the real price line is. These curves are essential for being able to model the market price dynamically.

Figure 5.2. The relationship between metal market availability in general and market price



Copper | Aluminium and iron | Cobalt

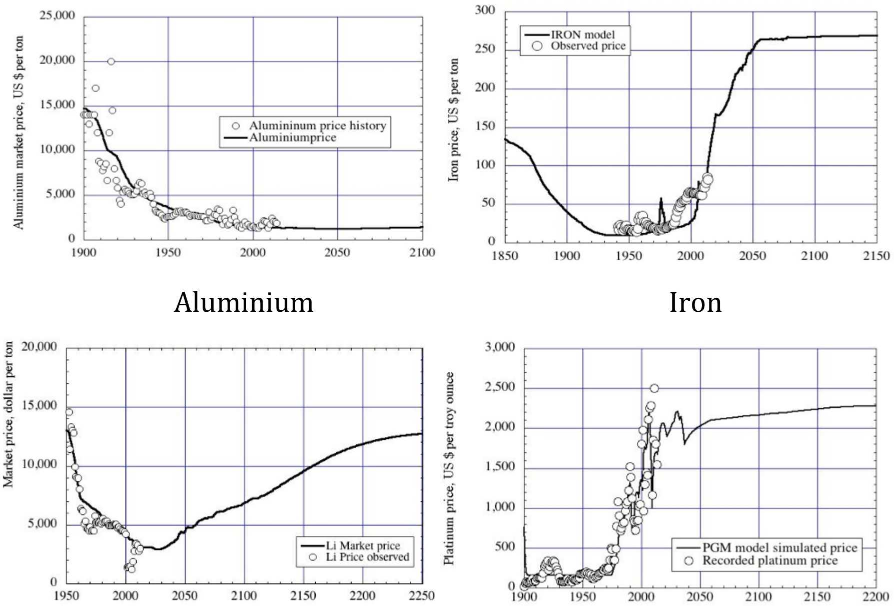


Lithium | Recycling vs price | All metals combined

Source: Own Figure, Lund University/University of Iceland

Examples of price histories simulated by the WORLD model for certain metals are shown in Figure 5.3. The ability to reconstruct and predict price reasonably well is important for being able to model the effect on market dynamics on resource extraction and supply to society. The price modelling (black solid line) is fitting reasonably well to observed data.

Figure 5.3. Examples of price histories simulated by the WORLD 6 model for aluminum, iron, lithium and platinum. Aluminium and iron are the large global bulk metals used in large scale infrastructures. Platinum and lithium are examples of technological metals used in advanced energy technologies. They can be substituted, but not without loss of functionality.



Aluminium | Iron

Lithium | Platinum

Source: Own Figure, Lund University/University of Iceland

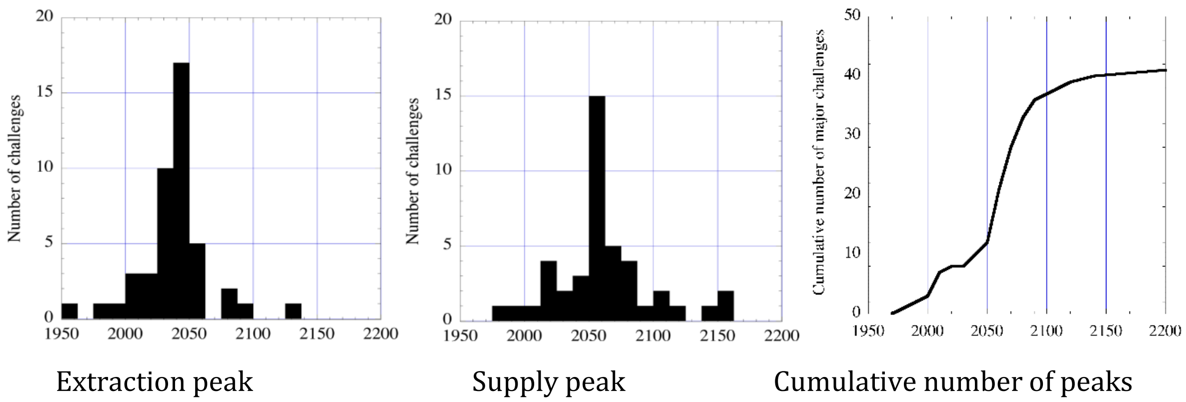
5.1.2 Extraction and supply rate peaks to inform policy

Under a business-as-usual scenario, many challenges loom ahead. Figure 5.4 shows the distribution of dates of extraction rate peaks and the supply rate peaks for 41 different metals, materials and fossil energy sources (Some of the metals, materials and fossil energy sourcs are listed in Table 5.1). In the table, we have also listed best estimates for the approximate recycling rate[6]. It can be seen that these are low for several of the key elements. This illustrates the present problem that much material passes through the system once and disappears without any serious attempt at recovery. As much as Business-as-Usual is a very problematic scenario, this may change fast if we assume recycling rates to be higher.

As shown in Figure 5.4 the number of extraction peaks reaches a maximum in 2040-2050, while the number of supply peaks is greatest in the period 2050-2060. Considering the policy outlook perspective, these results suggest that it will be necessary to have a longer time horizon than up to 2050 only. With regard to the experiences from international coordination of other policies (i.e. the European Long Range Transboundary Air Pollution Convention), we do know that it may take about 20-25 years from the first start to a fully implemented policy in society. Around 2030, the resource challenges will start to pile up towards the maximum in 2045. Calculating backwards, this implies that the process that must lead to an implemented policy by 2040 must start 20-25 years earlier, which corresponds to 2015-2020.

The WORLD model also contains a sub-module for production of fossil fuels, such as oil, gas, coal. The Fossil hydrocarbons have each been divided into three quality categories in the model, covering conventional oil, oil shale and fracking, and similar division for natural gas. Coal is also distinguished as hard coal, loft coals and carbon shales. Great emphasis was put on making more realistic estimates of the ultimately recoverable resources, considering technical feasibility, and declining reservoir qualities, paralleling the decline in ore grade as seen for metals. Special workshops were made with resource geologists from the main oil companies to get as realistic estimates as possible, potentially avoiding some of the politically motivated modifications sometimes seen to such estimates. The extractable amounts derived this way end up somewhat lower that the politically correct estimates published by many international agencies.

Figure 5.4. Distribution of number of extraction rate peaks (left), supply rate peaks (centre) and cumulative number of peaks (right) for 41 different metals, materials and fossil energy resources



Source: Own Figure, Lund University/University of Iceland

[6] Defined as the systemic recycling, the ratio between supplied amounts and extracted amount.

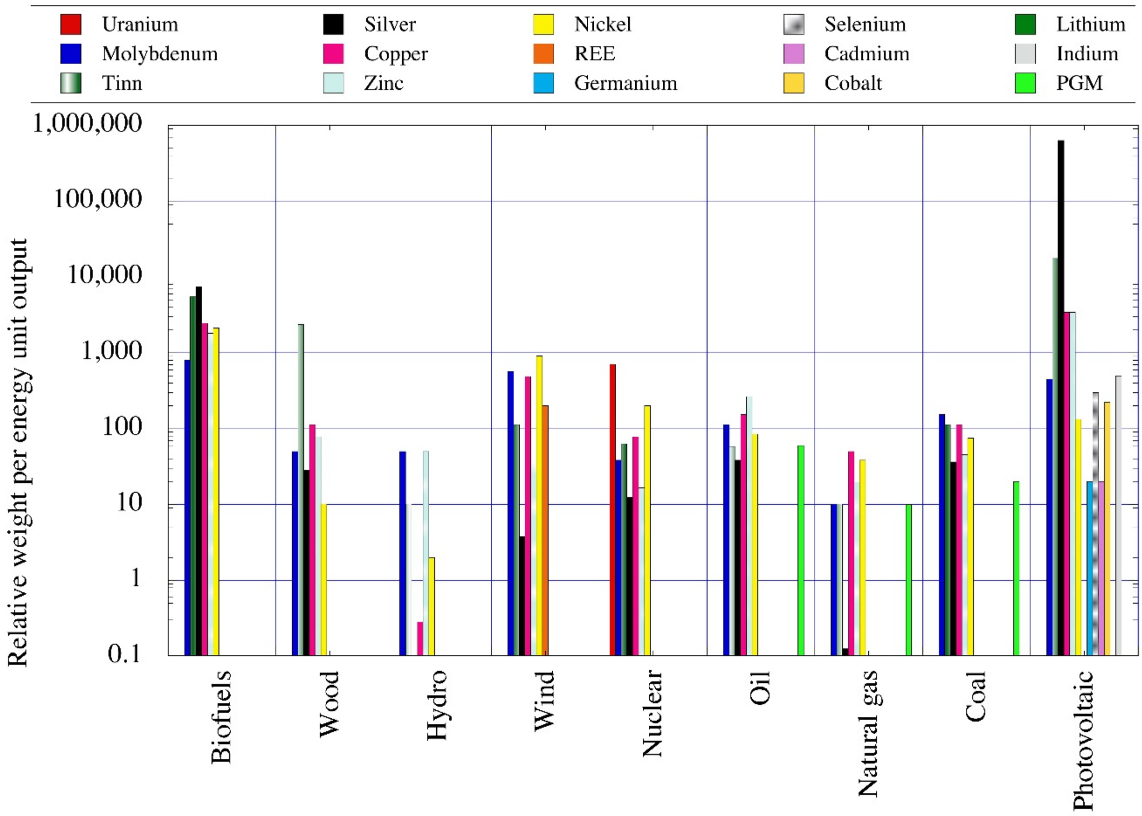
Table 5.1. Overview of peak extraction rate, peak supply rate and approximate recycling degree

Metal	Extraction peak year	Supply peak year	Recycling degree (%)	Metal	Extrac-tion peak year	Supply peak year	Recycling degree (%)
Oil	2012	2014	0	Titanium	2038	2060	40
Gas	2016	2016	0	Tellurium	1984	2060	0
Coal	2020	2018	0	Phosphorus	2035	2060	16-25
Cad-mium	2010	2020	80	Palladium	2042	2065	60
Gold	2016	2036	85-90	Aluminium	2030	2070	75
Cobalt	2026	2040	40	Iron	2052	2072	60
Gallium	2026	2042	5-15	Stainless steel	2052	2070	65
Silver	2038	2045	70	Manganese	2053	2072	45
Sele-nium	2042	2050	0-5	Tantalum	2035	2078	60
Cut stone	2040	2050	20	Molybdenum	2038	2080	40
Lead	2041	2051	65	Rhenium	2042	2080	40
Niobium	2045	2052	60	Uranium	2035	2080	50
Tin	2046	2055	40	Zinc	2046	2090	20
Anti-mony	2048	2056	5-15	Chromium	2051	2110	22
Indium	2042	2055	20-40	Copper	2044	2120	60
Rho-dium	2034	2058	60	Lithium	2060	2142	10-20
Germa-nium	2042	2058	20-30	Sand	2075	2150	30
Bismuth	2044	2059	5-15	Gravel	2130	2150	20
Nickel	2028	2060	50-60	Rare Earths	2045	2280	15
Plati-num	2036	2060	70	Thorium	2090	2400	90

5.2 Materials and metals critical for new or alternative technologies

Figure 5.5 shows different material needs for different technologies. The diagram shows amount of material per energy unit supplied. The materials requirement for different technologies is very different. Note that the scale on the Y-axis is logarithmic. It can be seen that hydropower has the smallest material resource requirement of them all and that the elements needed are not among the most scarce or rare. It also appears that fuels from food crops, wind power or photovoltaic power are among the highest users of materials, and several of those materials are available in very limited amounts. Not shown here, but this also applies to energy conversion technologies like fuel cells (Use of platinum group metals, nickel and cobalt) or new battery technologies (lithium, cobalt, nickel and Rare Earths). For such technologies the material availability would quickly become limiting when such items would go into mass production. It appears from the numbers that while fossil hydrocarbons have largely sustainable energy production technology in terms of material requirements, they do not have a sustainable source of the actual energy. Many of the alternative clean energy production technologies like wind power and photovoltaic electricity use sustainable energy sources, but these technologies are highly dependent on material and metals, which are not that sustainable. Only hydropower and wood as the source of biofuels may become sustainable across all categories. In Table 5.2, we have summarised this up for different energy production methods and added in electric vehicles (batteries) and fuel cells that produce electricity from hydrogen and oxygen.

Figure 5.5. Material use for different technologies involved in energy production and energy extraction



Source: Own Figure, Lund University/University of Iceland

The material needs for different technologies were compiled from a large number of references of which we have listed some examples, but not all (Alonso et al., 2007, Elshkaki and Graedel 2015, Graedel et al., 2015a,b, Grandell and Thorenz 2014, Hallada et al., 2008, Sverdrup and Ragnarsdottir 2016a, b; see also the review made in Sverdrup and Ragnarsdottir 2014c). It must also be remembered that these estimates are approximate, as the technologies varies a lot in their composition and resource use, as well as they change over time.

Table 5.2. Assessment of total sustainability for different energy technologies

Extraction or production method	Energy source is sustainable	Materials use is sustainable	Environmentally sustainable	Socially sustainable	Is it totally sustainable?
Hydrocarbons	No	Yes	No	Yes	No
Biofuels	By design	Yes	By design	By design	By design
Wood	Yes	Yes	By design	Yes	By design
Wind energy	Yes	Limits	By design	By design	Limits
Photovoltaic	Yes	Limits	Yes	Yes	Limits
Uranium energy	No	No	No	No	No
Thorium energy	Partially	No	Partially	Partially	No
Fusion	Unknown	No	Unknown	Unknown	Unknown
Hydropower	Yes	Yes	By design	Yes	By design
Solar heat	Yes	Yes	Yes	Yes	Yes
Geothermal heat	Yes	Yes	By design	Yes	By design
Geothermal conversion to electricity	No	Partially	No	Yes	No
Fuel cells	By design	No	Yes	Yes	No
Electric vehicles	By design	Limits	Yes	Yes	Yes

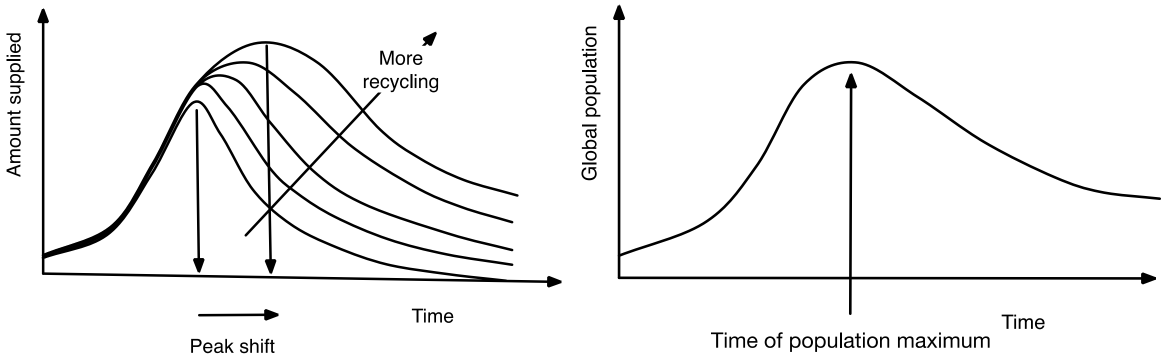
Legend: Green is sustainable, orange implies it can be sustainable if it is carefully designed to be so, brown indicates that with design and great care in can be sustainable, and purplish red suggests that it is not sustainable.

In the above Table 5.2, sustainability in the four columns means

- For energy: The energy source is endless
- For resources: The source is enormously larger than the use
- For environment: Impacts are below critical load
- For society: Does not lead to social unrest

Business-as-usual is a worst-case scenario and the situation will cause huge challenges if nothing is done. As demonstrated by Figure 5.6, with no recycling, the supply curve and the extraction curve will be the same. With increasing recycling, the supply curve can be lifted above the extraction curve and the peak is shifted towards the future, increasing the Factor X, i.e. the number of times one weight unit is used before it is lost. With the present recycling degrees and including market feedback on the recycling processes, we get the middle curve in Figure 5.6, shifting the maximum in the supply curve to about 20 years after the time for extraction maximum. There is a shift between extraction peak and supply peak. This depends on several factors, where the most important ones are recycling, retention time in society and irreversible losses in the system.

Figure 5.6. Relationship between extraction, supply and recycling (left) and typical population graph extending after 2200 (right)



Source: Own Figure, Lund University/University of Iceland

At a Factor X of 5, the supply curve will peak about 25-35 years later than with no measures at all. The model simulations suggest that a Factor X between 2 and 4 is feasible for most materials and metals, but that Factor X values in excess of 6 seem challenging, not least because they are becoming very energy-expensive. If the time for supply maximum can be moved to a time after the time of population maximum, then the challenge will have been significantly reduced. This will need a combination of strongly increased recycling, substantially better material use efficiency and possibly through maintained societal utility with less material intensity.

A business-as-usual (BAU) is a worst case scenario with respect to material resource use, leading to challenges of material scarcity and a significant potential for society disruptions.

However, the room for improvement upon the BAU scenario is substantial, given that initiatives and efforts are made. It appears that Germany's long-term focus on the Energiewende is focusing on the right type of overarching goals, but that it needs to be integrated with resource policy as regards the availability and sustainability of technology and other materials needed. In this context, energy production technologies can be dependent on:

1. Unsustainable energy source, but sustainable technology material use (as is the case for fossil hydrocarbon fuels)
2. Sustainable energy source, but unsustainable technology material use (for high technology energy production solutions)
3. Sustainable energy source and technology material use (for hydropower and to a debatable extent also forest-based biofuels). Energy biofuels based on intensive food-type of crops are not materials sustainable with respect to several materials, fertilizers and metals

The degree of sustainability of most of the alternative technologies to fossil fuel extraction is conditional on the design and modes of operation. To be effective in time, it is urgent that policies are implemented at once. Starting now may be expected to have impact around the time of 2040-2050, just in time to take on the challenges. Hence, to reach a goal of maximum 2oC global temperature increase, an accelerated policy pathway may have to be considered. Nonetheless, the timeframe considered for policy strategies and assessment of implications of policies should extend beyond 2050.

5.3 Final Conclusions

Based on the WORLD model run results, it would be safe to conclude that most of the supply peaks for metals, materials, fossil fuels, and the main nutrient phosphorus come in the period 2040-2060. This will amount to some of the most serious industrial, political and social challenges of our times and will need careful preparation and research in order not to disrupt the functioning of society. Business-as-usual is a worst-case scenario and the situation is only hopeless if nothing is done. As demonstrated by Figure 5.6 with no recycling, the supply curve and the extraction curve will be the same.

6 REFERENCES

Abischeva, Z.S., Zagorodnyaya, A.,N., 2013. Rhenium of Kazakstan. Review of technologies for rhenium recovery from mineral raw materials in Kazakstan. Set of powerpoint slides. 19 pages http://www.technetium-99.ru/Presentations/Abisheva.pdf

Acemoglu, D., Robinson, J.A., 2013. Why nations fail. The origins of power, prosperity and poverty. Profile Books Ltd, London. 529pp. ISBN 978-1-84668-430-2

Aguilar-Fernandez, R. 2009. Estimating the opportunity cost of lithium extraction in the Salar de Uyuini, Bolivia. Master of Environmental Management degree in the Nicholas School of the Environment of Duke University. 58 pp

Albrecht, W W. 1989. Production, Properties and Application of Tantalum, Niobium and Their Compounds. In: Möller, P, Cerný, P and Saupé, F. (eds.) Lanthanides, Tantalum and Niobium. Berlin Heidelberg: Springer-Verlag.

Aleklett, K., 2007. Peak oil and the evolving strategies of oil importing and exporting countries. Discussion paper 17. Joint Transport Research Centre, OECD, International Transport Forum.

Aleklett, K., Lardelli, M., Qvennerstedt, 2012. Peeking at peak oil. Springer. Heidelberg. 335 pp

Alfantazi, A.M., Moskalyk, R.R., 2003. Processing of indium: a review. Minerals Engineering 16: 687-694.

Al-Husseini, M. 2009. World production of conventional petroleum liquids to 2030: A comparative overview. GeoArabia 14: 215-267.

Allen, D.T., Behamanesh, N. 1994 Wastes as Raw Materials. 68-96 in; The Greening of Industrial Ecosystems, Allenby, B.R. and Richards, D.J., (eds.), National Academy Press, Washington, D.C.

Allsopp, M., Miller, C., Atkins, R., Rocliffe, S., Tabor, I., Santillo, D., Johnston, P., 2013. Review of the current state of development and the potential for environmental impacts of seabed mining operations. Greenpeace Research Laboratories Technical Report. 03-2013. 50pp. https://www.oceanfdn.org/sites/default/files/seabed-mining-tech-review-2013.pdf

Allen, D.T. and Behamanesh, N., 1994. Wastes as Raw Materials. 68-96 in; The Greening of Industrial Ecosystems, Allenby, B.R. and Richards, D.J., (eds.), National Academy Press, Washington, D.C.

Alonso, E., Frankfield, J.G., Kirchain, R. 2007. Material availability and the supply chain: Risks, effects, and responses. Environmental Science and Technology 41:6649-6656

Alonso, E. Field, F.R., Roth, R., Kirchain, R.E., 2009. Strategies to address risks of platinum scarcity for supply chain downstream firms. Sustainable Systems and Technology, 2009. ISSST '09. IEEE International Symposium. 1-6. © 2010 Institute of Electrical and Electronics Engineers. http://dx.doi.org/10.1109/ISSST.2009.5156762

Alonso, E. 2010. Material scarcity from the perspective of manufacturing firms: Case studies of platinum and cobalt. PhD thesis from Materials Science and Engineering, MIT, Boston. 216pp http://msl.mit.edu/theses/Alonso_E-thesis.pdf

Alonso E., Field F.R., Kirchain R.E. 2012. Platinum availability for future automotive technologies. Environmental Science and Technology. 46:12986-12993.

Alumina Limited 2012 Built on bauxite. Annual review 2012. Corporate investors information brochure. http://www.aluminalimited.com/uploads/Alumina-Annual-Review-FINAL.pdf

Alves, A., R., dos Reis Couthino, A., 2015. The evolution of the niobium production in Brazil. Materials Research 18:106-112. DOI: http://dx.doi.org/10.1590/1516-1439.276414

Anderson, C.D., Taylor, P.R., Anderson, C.G. 2013. Extractive metallurgy of rhenium: a review, Minerals & Metallurgical Processing, 30:59-73.

Andrews, M.G., 1955. Tungsten, the story of an indispensable metal. The wolfram Institute, Washington DC.

Angerer, G., Marcheider-Weidemann, F., Wendl, M., Wietschel, M. 2009. Lithium für Zukunftstechnologien—Nachfrage und Angebot unter besonderer berücksichtigung der Elektromobilität; Fraunhofer Fraunhofer ISI: Karlsruhe, Germany, 2009

Anglo-American Platinum plc. 2012 Fact Book 2011/12. Platinum. 19pp. Johannesburg, South Africa,

Anglo-American Platinum plc 2013. Ore reserves and mineral resources report 2013. 52pp. http://www.angloamerican.com/~/media/Files/A/Anglo-American-PLC-V2/investors/reports/aa-rr-report-2013.pdf

Anglo-American Mining Corporation 2013 Fact book on Copper. 13pp PDF. www.angloamerican.com/~/media/Files/A/Anglo...Plc/.../copper_2012.

Anstett, T.F.; Krauss, U.H.; Ober, J.A.; Schmidt, H.W. 1990, International Strategic Minerals Inventory Summary Report—Lithium; USGS: Denver, CO, USA.

Anthoni, J.F., 2000. Oceanography: Dunes and beaches http://www.seafriends.org.nz/oceano/beach.htm

Aquaknow 2014. Sand mining- the ‘high volume – low value’ paradox. <http://www.aquaknow.net/en/news/sand-mining-high-volume-low-value-paradox/>

Ashraf, M.A., Maah, M.J., Yusoff, I., Wajid, A., Mahmood, K., 2011. Sand mining effects, causes and concerns: A case study from Bestari Jaya, Selangor, Peninsular Malaysia. Scientific Research and Essays 6: 1216-1231.

Asian Metal 2017. Tungsten: resource distribution and production http://metalpedia.asianmetal.com/metal/tungsten/resources&production.shtml

Avon Metals 2014. Rhenium. On the use of rhenium in jet engine parts for F16 and F-15, as well as F-35 and F-22. http://www.avonmetals.com/rhenium-re/4585681007

Babakina, O.A., and Graedel, T.E., 2005. The Industrial Platinum Cycle for Russia: A Case Study of Materials Accounting. Working Paper, Yale School of Forestry and Environment. www.yale.edu/environment/publications

Bacos, M.-P.; Dorvaux, J.-M.; Landais, S.; Lavigne, O.; Mévrel, R.; Poulain, M.; Rio, C.; Vidal-Sétif, M.-H. 2011. 10 years-activities at Onera on advanced thermal barrier coatings. J. Aerosp. Lab. 3: 1–14.

Bangs, C.E. 2011. Rolling out aluminium to meet IPCC 2050. A dynamic MFA approach for greenhouse gas emission projections for the aluminium industry. Masters Thesis, Norwegian University of Science and Technology, Trondhjem, Norway. 76pp

Bardi, U. 2005 The Mineral Economy: A Model for the Shape of Oil Production Curves. Energy Policy, 33, 53-61.

Bardi, U., and Yaxley, L. 2005 How general is the Hubbert curve? The case of fisheries. In. Proceedings of the 4th International ASPO Conference, Lisbon. Portugal, (Aleklett, K. (Ed.)). Available online: http://europe.theoildrum.com/files/bardiyaxleyaspo2005.pdf (accessed August 12, 2009).

Bardi, U. 2007 Peak oil's ancestor: the peak of British coal production in the 1920s. The Association for Study of Peak Oil (ASPO): Uppsala, Sweden, 2007.

Bardi, U., and Pagani, M. 2008 Peak Minerals. The Oil Drum: Europe. Available online and accessed in 2009 and again 2014: http://www.theoildrum.com/node/3086.

Bardi, U. 2008 Peak oil and The Limits to Growth: two parallel stories, The Oil Drum: Europe

Bardi, U, 2009. The fifth problem: peak capital. The Oil Drum: Europe

Bardi, U., Lavacchi, A., 2009. A Simple Interpretation of Hubbert’s Model of Resource Exploitation. Energies, 2:646-661; doi:10.3390/en20300646

Bardi, U. 2013. Extracted. How the quest for mineral wealth is plundering the planet. The past, present and future of global mineral depletion. A report to the Club of Rome. Chelsa Green Publishing, Vermont. 299pp. ISBN 978-1-60358-541-5

Bartlett A.A. 1999 An Analysis of U.S. and World Oil Production Patterns Using Hubbert-Style Curves. Mathematical Geology.

Bastos-Neto, A C, Pereira, V P, Ronchi, L H, Lima, E F., Frantz, J C. 2009. The World-Class Sn, Nb, Ta, F, (Y, Ree, Li) Deposit and the Massive Cryolite associated with the Albite-Enriched facies of the Madeira A-type Granite, Pitinga Mining District, Amazonas State, Brazil. The Canadian Mineralogist, 47, 1329–1357.

Baylis, R. 2010. The lithium market: 2009 review and outlook. Industrial Minerals Research, Roskill Information Services Ltd. Lithium Supply and Market, 26-28 January 2010 meeting at Caesars Palace, Las Vegas, US. 28 page presentation.

Beaudoin, Y., Bredbenner, A., Baker, E., Roche, C., Bice, E., Pendleton, L, 2014. Thematic focus: Environmental governance, Resource efficiency. Wealth in the Oceans: Deep sea mining on the horizon? www.unep.org/geas. 12pp. GEAS 2014

Beckmann, A. 2007. The physical environment of cobalt-rich ferromanganese crust deposits, the potential impact of exploration and mining on this environment, and data required to establish environmental baselines. In: (Ed. ISA) Polymetallic sulphides and cobalt rich ferromanganese crust deposits: establishment of environmental baselines and an associated monitoring programme during exploration, Proceedings of the International Seabed Authority's workshop, Kingston, Jamaica, 6-10 September 2004, 227–250.

Berger, V.I, Singer, D.A., Orris, G J. 2009. Carbonatites of the World, Explored Deposits of Nb and REE — Database and Grade and Tonnage Models. USGS Open File Report.

Berger, V.I., Singer, D.A., Bliss, J.D., Moring, B.C., 2011, Ni-Co laterite deposits of the world; database and grade and tonnage models: U.S. Geological Survey Open-File Report 2011-1058. http://pubs.usgs.gov/of/2011/1058/

Berry, C., 2015. A Closer Look at Nickel: An Unsustainable Current Reality? The Disruptive Discoveries Journal. http://www.discoveryinvesting.com/blog/2015/10/20/a-closer-look-at-nickel-an-unsustainable-current-reality

Bertram C., Krätschell, A., O'Brien, K., Brückmann, W., Proelss, A., Rehdanz K., 2011. Metalliferous sediments in the Atlantis II deep - Assessing the geological and economic resource potential and legal constraints. Resources Policy 36, 315-329.

Berzina, A., Sotnikov, V.I., Economou-Eliopopulos, M., Eliopopulos, D.M., 2005. Distribution of rhenium in molybdenite from porphyry Cu–Mo and Mo–Cu deposits of Russia (Siberia) and Mongolia. Ore Geology Reviews 26:91-113.
DOI: 10.1016/j.oregeorev.2004.12.002

Berzina A., Korobeinikov A. 2007. Rhenium and precious metal (Pt, Pd, and Au) abundances in porphyry Cu-Mo deposits of Central-Asian Mobile Belt. Act Petrologica Sinica 23: 1957 -72.

Beurlen, H., Thomas, R., Rodrigues da Silva, M.R. et al. 2014 Perspectives for Li- and Ta-mineralization in the Borborema Pegmatite Province, NE-Brazil: A review. J. South American Earth Sci. 56, 110-127. DOI: 10.1016/j.jsames.2014.08.00

Bian, H., Xu, X., Li, Y., Koizumi,Y., Wang, Z., Mingwei Chen, M., Kenta Yamanaka, K., Chiba, A. 2015. Regulating the coarsening of the γ' phase in superalloys. NPG Asia Materials 7, e212; doi:10.1038/am.2015.96

Birat, J.P., 2010. Steel sectoral report - Contribution to the UNIDO roadmap on CCS1. Available at www.unido.org/fileadmin/user.../Stee_sectoral_%20report.pdf

Biryabarema, F., Biester, M., Pohl, W.L. 2014 The geochemical signature of rare-metal pegmatites in the Central Africa Region: Soils, plants, water and stream sediments in the Gatumba tin-tantalum mining district, Rwanda. J. Geochem. Exploration. Special Volume on Mining and the Environment in Africa 144, 539-551. doi: 10.1016/j.gexplo.2014.01.025

Bleischwitz, R., Meyer, B., Giljum, S., Acosta, J., Distelkamp, M., Meyer, M., Pirgmaier, E., Schütz, H., Ritsche, D. 2012: Wirtschaftliches Wachstum mit weniger Ressourcennutzung - Die absolute Entkopplung ist möglich. Ökologisches Wirtschaften 2/2012, S. 30-33.

Bliss, J.D., Hayes, T.S., Orris, G.J., 2012. Limestone—A Crucial and Versatile Industrial Mineral Commodity. USGS Fact Sheet 2008–3089. 4pp

Blossom, J.W., 2000 Rhenium. 64.1-64.4. USGS 2000 Mineral Yearbook, Washington DC.

BMI Research 2014. Global Industry Overview - Sand Miners On Solid Ground - October 2014 http://www.mining-insight.com/global-industry-overview-sand-miners-solid-ground-oct-2014

Boeing 787 Dreamliner 2017: http://barrieaircraft.com/images/boeing-787-8-dreamliner-01.jpg Airbus A380 2017; http://helicecluster.com/aero/airbus-a380.htm.

Bolen, W.P, 2011. Sand and gravel, construction. 2009, Minerals Yearbook, Update. USGS, Washngton DC. 64.1-64.18. Domestic survey data and tables were prepared by H.A. Fatah, M.L. Jackson, and F.H. Morgan, statistical assistants.

Boliden AB, Rönnskärsveket, Sundsvall, Sweden. Metals for the future. Corporate brochure 2011. http://www.boliden.com/Operations/Exploration/Ore-Reserves-and-Mineral-Resources/ http://www.slideshare.net/miningontop/base-metals-outlook-for-supply-and-demand

Booth-Morrison, C., Noebe, R.D., Seidman, D.N., 2008. Effects of a tantalum addition on the morphological and compositional evolution of a model Ni-Al-Cr superalloy. In: Reed, R:C, Green, K.A., Caron, P., Gabb, T.P., Fahrmann, M.G., Huron, E.S., Woodard, S.A., Superalloys 2008. The Minerals, Metals & Materials Society.

Booth-Morrison, C., Noebe, R.D., Seidman, D.N., 2008. Effects of a tantalum addition on the morphological and compositional evolution of a model Ni-Al-Cr superalloy. In: Reed, R:C, Green, K.A., Caron, P., Gabb, T.P., Fahrmann, M.G., Huron, E.S., Woodard, S.A., Superalloys 2008. The Minerals, Metals & Materials Society.

Boschen, R.E., Rowden, A.A., Clark, M.R., Gardner, J.P.A., 2013. Mining of deep-sea seafloor massive sulfides: A review of the deposits, their benthic communities, impacts from mining, regulatory frameworks and management strategies. Ocean & Coastal Management 84: 54-67

Boudreau, J., Choi, E., Datta, R., Rezhdo, O., Saeed, K., 2009. Platinum Supply and the Growth of Fuel Cell Vehicles Proceedings of the Systemd Dynamics Society 2009 Conference. 31pp. http://www.systemdynamics.org/conferences/2009/proceed/papers/P1256.pdf

Bowman, K.A. Green, M. McLean, S. Olson, and J.J. Schim. The Minerals, Metals &Materials Society, http://cannonmuskegon.com/wp-content/uploads/2015/02/Advanced-Superalloy-and-Tailored-Microstrucres-for-Integrally-Cast-Turbine-Wheels-2000.pdf

Bradley, D., and Jaskula, B., 2014, Lithium—For harnessing renewable energy: U.S. Geological Survey Fact Sheet 2014–3035, 2 p., http://dx.doi.org/10.3133/fs20143035.

Brandt, G. 1735 Dissertatio de semimetallis. Acta Literaria et Scientiarum Sveciae, 4:1–10.

Brandt, G., 1746 Rön och anmärkningar angäende en synnerlig färg — cobolt. Kongliga Svenska vetenskapsakademiens handlingar, 7, 119–130

Brandt, B. 1748 Cobalti nova species examinata et descripta, Acta Regiae Societatis Scientiarum Upsaliensis, 1st series, 3:33–41

Brewster, N. Outlook for commodity markets. Rio Tinto Inc. MF Global Seminar, set of ppt slides. 2009. www.slideshare.net/RioTinto/mf-global-seminar-15-oct-2009

Briskey, J.A., 2005. Indium in Zinc-Lead and Other Mineral Deposits—A Reconnaissance Survey of 1118 Indium Analyses Published Before 1985. USGS. Mineral Resources Program—Eastern Mineral Resources Team. Open-File Report 2005-1209

British Geological Survey 2009. Platinum. Definitions, mineralogy and reserves. 31pp

Brown, R.D., 2000. Bismuth. minerals.usgs.gov/minerals/pubs/commodity/bismuth/110400.pdf

Brown, T.J., Wrighton, C.E., Raycraft, E.R., Shaw, R.A., Deady, E.A., Rippingale, J., Bide T., Idoine, N., 2015. World mineral production 2009–13, 88pp. Keyworth, Nottingham British Geological Survey. ISBN 978-0-85272-848-2

Buckman, R.W., 2000. New Applications for Tantalum and Tantalum Alloys. Journal of Materials Research (JOM) March 2000: 40-41

Bullis, K., 2013. Electric Vehicles Out in the Cold. MIT Technology Review. http://www.technologyreview.com/news/522496/electric-vehicles-out-in-the-cold/

Buncic, D., Moretto, C., 2014. Forecasting copper prices with dynamic averaging and selection methods. Discussion paper no. 2014-30. Universität St. Gallen, Switzerland. 75pp.

Burke, A. 2012. Fast charging tests (up to 6oC) of lithium titanate cells and modules: Electrical and thermal response. Institute of Transportation Studies, University of California, Davis. https://trid.trb.org/view.aspx?id=1217323.

Busch, J., Steinberger, J.K., Dawson, D.A., Purnell, P., Roelich, K. 2014 Managing Critical Materials with a Technology-Specific Stocks and Flows Model. Environmental Science and Technology, 48:1298-1305. ISSN 0013-936X

Butler, J. Platinum 2010 interim review Platinum Metals review. 2010, 55:41-42

Butler, J. Platinum 2011 interim review Platinum Metals review. 2011, 56:47-48

Butler, J. Platinum 2012 interim review. Johnson Matthey. 40 pp booklet. Royston, Hertfordshire, UK. ISSN 0268-7305, 2012.

Butterman, W.C., and Carlin, J.F. 2004. Mineral commodity profiles; Antimony. United States Department of the Interior. US Geological Survey. Open-File Report 03-019. 35pp. http://pubs.usgs.gov/of/2003/of03-019/of03-019.pdf

Cailteaux, J.K.H, Kampunzu, A.B., Leroughe, C., Kaputo, A.K., Milesi, J.P., 2005. Genesis of sediment-hosted stratiform copper–cobalt deposits, central African Copperbelt. Journal of African Earth Sciences 42:134–158

Campbell, C.J., Laherrère, J.H., 1998 The End of Cheap Oil. Sci. Amer., March, 78-83.

Campbell, C.J. 2013. Campbell's Atlas of Oil and Gas Depletion. 2nd ed. 788pp

Carles, L. 2010. Modelling long-term global lithium stocks and flows. Master Thesis, Ecole Polytechnique Federale de Lausanne: Lausanne, Switzerland.

Caron, P., Cornu, D., Khan, T., de Monicault, J.M., Development of a hydrogen resistant superalloy for single crystal blade application in rocket engine fuel pumps. In; Superalloys 1996, Eds; Kissinger, R.D, Deye, D.J., Anton, D.L., Cetel, A.D. Nathal, M.V., Pollock, T.M., Woodford, D.A., 53-61. The Minerals, Metals and Materials Society 1996. http://www.tms.org/superalloys/10.7449/1996/Superalloys_1996_53_60.pdf

Caron, P., Lavigne, O., 2011. Recent studies at Onera on superalloys for single crystal turbine blades. Journal of Aerospace Lab 3:1-14.

Cawthorn, R.G. 1999 The platinum and palladium resources of the Bushveld Complex. South African Journal of Science 95:481-490

Cawthorn, R.G. 2010 The platinum and palladium resources of the Bushweld Complex. Platinum Metals Review 54:205-215

Chen, B.C., Ramakrishnan, R., Shavlik, J.W., Tamma, P., 2006. Bellwether Analysis: Predicting Global Aggregates from Local Regions. VLDB '06, September 12–15, 2006, Seoul, Korea. Copyright 2006 VLDB Endowment, ACM 1-59593-385-9/06/09

Chen W.Q., Graedel, T.E. 2012 Anthropogenic cycles of the elements: A Critical Review Environmental Science & Technology 46: 8574-8586

Chen, W.Q., Graedel, T.E., 2012 Dynamic analysis of aluminum stocks and flows in the United States: 1900-2009. Ecological Economics, 81, 92-102.

Chen, W.Q., Shi, L., 2012 Analysis of aluminum stocks and flows in mainland China from 1950 to 2009: Exploring the dynamics driving the rapid increase in China's aluminum production. Resources Conservation and Recycling 65, 18-28.

Chicharro, E., Martın-Crespo, T., Gomez-Ortiz, D. et al. (2014) Geology and gravity modeling of the Logrosan Sn-W ore deposits (Central Iberian Zone, Spain). Ore Geology Rev. 65: 297-304. doi: 10.1016/j.oregeorev.2014.10.005

Ciacci, L.; Chen, W. Q.; Passarini, F.; Eckelman, M.; Vassura, I.; Morselli, L., 2013 Historical evolution of anthropogenic aluminum stocks and flows in Italy. Resources Conservation and Recycling, 72, 1-8.

Clark, M., Smith, S., 2013a. Chapter 3.0: Environmental Management Considerations, in: Baker, E., Beaudoin, Y., (Eds.), Deep Sea Minerals: Sea-floor Massive Sulphides, a physical, biological, environmental, and technical review. Volume 1A, Secretariat of the Pacific Community.

Clark, M., Smith, S., 2013b. Chapter 3.0: Environmental Management Considerations, in: Baker, E., Beaudoin, Y., (Eds.), Deep Sea Minerals: Manganese Nodules, a physical, biological, environmental, and technical review. Volume 1B, Secretariat of the Pacific Community.

Clark, M.R., Heydon, R., Hein, J.R., Petersen, S., Rowden, A., Baker, E., Beaudoin, Y., 2013. Deep Sea Minerals: Cobalt-rich Ferromanganese Crusts, A physical, biological, environmental, and technical review. Baker, E. and Beaudoin, Y. (Eds.) Vol. 1C, Secretariat of the Pacific Community. ISBN 978-82-7701-121-9. 52 pages.

Clark, M.R., Rowden, A.A., Schlacher, T., Williams, A., Consalvey, M., Stocks, K.I., Rogers, A.D., O'Hara, T.D., White, M., Shank, T.M. and Hall-Spencer, J. 2010 The ecology of seamounts: structure, function, and human impacts. Annual Review of Marine Science 2, 253–278.

Clarke, D.R., Oechsner, M., Padture, N.P., 2012 Thermal-barrier coatings for more efficient gas-turbine engines. MRS Bulletin 37:891-898

Clarke, G.M. and Harben, P.W., 2009. Lithium availability wall map. http://www.worldcat.org/title/lithium-availability-wall-map-june-2011/oclc/746758418

Cobalt / Investing News 2014. The top 10 list of cobalt production by country. Investing News Network website. http://investingnews.com/daily/resource-investing/critical-metals-investing/cobalt-investing/top-cobalt-producing-countries-congo-china-canada-russia-australia/

Cobalt Facts 2011. Properties and main uses (3 pages), Metalurgical uses (15 pages), Magnetic alloys (6 pages), Cobalt supply and demand 2010 (8 pages), Cobalt in chemials (12 pages), Cobalt in cemented carbides (5 pages), Cobalt in electronics (6 pages). The Cobalt Development Institute. http://www.thecdi.com/cobaltfacts.php

Cohen, J.E., 1995, Population growth and Earth's human carrying capacity. Science 269, 341-346.

Cohen, C., 2007, Earth's natural wealth, an audit. New Scientist May 23rd, 2605:34

Collier, J.P., Selius , A.O., Tien, J.K. 1988. On developing a microstructurally and thermally stable iron-nickel base superalloy In: Superalloys 1988 Eds: S. Reichman, D.N. Duhl, G. Maurer, S. Antolovich, C. Lund. The Metallurgical Society

Colorado School of mines, Division of Economics and business. Colden, CO. http://econbus.mines.edu/working-papers/wp201205.pdf

Colton, J.W., 1957. Recovery of lithium from complex silicates. In Handling and uses of the alkali metals; In: Advances in Chemistry; American Chemical Society: Washington, DC. http://pubs.acs.org | doi: 10.1021/ba-1957-0019.ch001

Cornish, L.A., Chown, L.H., 2011. Platinum-Based Alloys and Coatings: Materials for the Future? Chapter 15; 337-370. Advances in Gas Turbine Technology (Ed) Benini, E., ISBN 978-953-307-611-9, 538 pages, Publisher: InTech, Chapters published November 04, 2011 under CC BY 3.0 license DOI: 10.5772/664

Cornish, L.A., Shongwe, M.B. Odera, B., Odusote, J.K., Witcomb, M.J., Chown, L.H., Rading, G.O., Papo M.J., 2012. Update on the development of platinum-based alloys for potential high-temperature applications. The Southern African Institute of Mining and Metallurgy Platinum 2012. pages 905-923.

Crocker, L., Lien, R.H., Davidson, C.F., May, J.T., Edlund, V.E., 1988. Lithium and its recovery from low-grade Nevada clays. Bulletin 691, 44 pages. United States Department of the Interior. And United States Buereau of Mines. Washington DC.

Cronstedt, Axel F. 1751. Rön och försök, Gjorde Med en Malm-art från Los Kobolt Grufvor i Farila Socken och Helsingeland". Kongl. Svenska Vetenskaps Academins Handlingar. 12: 287–292

Cronstedt, Axel F. 1754. Fortsättning af rön och försök, Gjorde Med en Malm-art från Los Kobolt Grufvor. Kongl. Svenska Vetenskaps Academins Handlingar. 15: 38–45.

Crowson, P.C.F., 2011. Mineral reserves and future minerals availability. Mineral Economics. 24:1-6.

Cuddington, J.T., Zellou, A., 2012. A simple mineral market model: Can it produce super cycles in prices? Working Paper 2012-05.

Copper Develoment Association Inc. 2013 Copper in the USA: Bright Future Glorious Past. http://www.copper.org/education/history/g_fact_future.html

Crowson, P.C.F., 2011a. Mineral reserves and future minerals availability. Mineral Economics. 2011;24:1-6.

Crowson, P.C.F., 2011b. Some observations on copper yields and ore grades. Resources Policy 37;59–72

Cullen, J.M., Allwood, J.M., Bambach, M.D., 2012 Mapping the global flow of steel: From steelmaking to end-use goods. Environmental Science & Technology 46:13048-13055, DOI: 10.1021/es302433p

Cunningham, C.G., Zappettini, E.O., Vivallo, S., Waldo, V.S., Celada, C.M., Quispe, J., Singer, D.A., Briskey, J.A., Sutphin, D.M., Gajardo M., M., Diaz, A., Portigliati, C., Berger, V.I., Carrasco, R., and Schulz, K.J., 2008, Quantitative mineral resource assessment of copper, molybdenum, gold, and silver in undiscovered porphyry copper deposits in the Andes Mountains of South America: U.S. Geological Survey Open-File Report 2008-1253, 282 p.

Dahlström, K., Ekins, P., He, J., Davis, J., Clift, R., 2004 Iron, Steel and Aluminium in the UK: Material Flows and their Economic Dimensions. Downloaded from: http://www.psi.org.uk/publications/archivepdfs/environment/finalprojectreport.pdf.

Daigo, I., Matsuno, Y., Adachi, Y. 2010. Substance flow analysis of chromium and nickel in the material flow of stainless steel in Japan. Resources, Conservation and Recycling 54: 851-863.

Daigo, I., Osako, S., Adachi, Y., Matsuno, Y, 2014. Time-series analysis of global zinc demand associated with steel. Resources, conservation and Recycling 82:35-40.

Dalnedra Vostok 2016. 20 июля 2012 года. Соболиное, месторождение. Источник: http://nedradv.ru/ajax/cards/cards.cfm?id_obj=a56a2a04b5f2a9dbee29af0a78087aff. nedradv.ru

Dalvi, A.D., Bacon, W.G., Osborne, R.C. 2004. The past and the future of nickel laterites. PDAC 2004 International Convention, Trade Show & Investors Exchange March 7-10, 2004. 27pp.

Darling, P. (Editor) et al., 2011. SME Mining Enginering Handbook, Third Edition 3rd Edition. Society for Mining, Metallurgy, and Exploration. SBN-13: 978-0873352642

Davis, J.R. 2000 ASM Specialty Handbook: Nickel, Cobalt, and their alloys. ASM International.

Dawkins, E. Chadwick, M., Roelich K., Falk R., 2012. Metals in a Low-Carbon Economy: Resource Scarcity, Climate. Change and Business in a Finite World. Stockholm Environment Institute Project Report. Stockholm, Sweden. 56pp. http://www.sei-international.org

Dehnavi, P.Y., 2013. Global cycle of gallium production, use and potential recycling. TRITA-LWR Degree Project 13:23. ISSN 1651-064X. Royal Institute of Technology, Stockholm, Sweden.

Demarthe, J.M., Rousseau, A.M., and Fernandez, F.L., 1990, Recovery of specialty metals, mainly germanium and indium, from zinc primary smelting, in Lead-Zinc '90, Anaheim, Calif., February 18-21, 1990, Proceedings: Warrendale, Pa., The Minerals, Metals & Materials Society, p. 151-160.

Deutsches Ressourceneffizienzprogramms (ProgRess)—Programm zur Nachhaltigen Nutzung und zum Schutz natürlicher Ressourcen; Bundesministerium für Umwelt, Naturschutz und Reaktorsicherheit (BMU; German Federal Ministry for the Environment, Nature Convervation, and Nuclear Safety): Berlin, Germany, 2012 (in German).

Dexclaux, J. and Serre, J. 2003. M88-2 E4: Advanced New Generation Engine for Rafale Multirole Fighter. AIAA/ICAS International Air and Space Symposium and Exposition: The Next 100 Years. 14–17 July 2003, Dayton, Ohio. AIAA 2003-2610

De Young, J.H., Sutphin, D.M., Cannon, W.F., 1984. International strategic minreals inventory summary report-Manganese. USGS Circular 930-A, 26pp

De Young, J.H., Lee, M.P., Lipin, B.R., 1984. International strategic minreals inventory summary report-Chromium. USGS Circular 930-B, 26pp

Diamond, J. 1997 Guns, Germs, and Steel: The Fates of Human Societies. W.W. Norton and Co. ISBN 0-393-06131-0

Diamond, J. 2005. Collapse. How societies choose to fail or survive. Penguin Books

Distelkamp, M., Meyer, B., Meyer, M. 2010: Quantitative und qualitative Analyse der ökonomischen Effekte einer forcierten Ressourceneffizienzstrategie. Kurzfassung der Ergebnisse des Arbeitspakets 5 des Projekts „Materialeffizienz und Ressourcenschonung" (MaRess), Ressourceneffizienz Paper 5.2, ISSN 1867-0237, Wuppertal. http://ressourcen.wupperinst.org/downloads/MaRess_AP5_3_Zusammenfassg.pdf

Donachie, J.M. and Donachie, S.J., 2002. Superalloys, A technical guide. Second edition. The Materials Information Society. Materials Park, Ohio. www.asmininternational.org.450pp

Dong, J., Bi, Z., Wang, N., Xie, X., Wang, Z., 2008. Structure Control of a New-type High-Cr Superalloy IN: Superalloys 2008 Reed, R.C, Green, K.A., Caron, P., Gabb, T.P., Fahrmann, M.G., Huron, E.S., Woodard, S.A., The Minerals, Metals & Materials Society 41-51

Drake Resources 2015. Nickel in Scandinavia. 4ppp. Internet download, may 2016.http://www.drakeresources.com.au/assets/DRK_nickel_handout_copy_1.pdf

Eckstrand, O.R., and Hulbert, L.J. 2007, Magmatic nickel-copper-platinum group element deposits, in Goodfellow, W.D., (Ed.), Mineral Deposits of Canada: A Synthesis of Major Deposit Types, District Metallogeny, the Evolution of Geological Provinces, and Exploration Methods: Geological Association of Canada, Mineral Deposits Division, Special Publication No. 5:205-222.

Edelstein, D.L. 2011, Copper, in Metals and minerals: U.S. Geological Survey Minerals Yearbook, 1:20.1–20.25, http://minerals.usgs.gov/ minerals/pubs/commodity/copper/ myb1-2011-coppe.pdf.

Ehrlich, P., Ehrlich, A., 2009. The population bomb revisited. The Electronic Journal of Sustainable Development 1, 63-71.

Eilu, P., 2011, Metallic mineral resurces of Fennoscandia. Geoscience for Society 125th Anniversary Volume Edited by Keijo Nenonen and Pekka A. Nurmi Geological Survey of Finland, Special Paper 49: 13–21

Ekeberg, A.G, 1802. Uplysning om ytter-jordens egenskaper, i synnerhet i jämförelse med beryll-jorden: Om de fossilier hvari förstnämnde jord innehålles, samt om en ny uptäckt kroop af metallisk natur. Tantalum," Kongliga Vetenskaps Academiens nya Handlingar, 23:, 68–73

Eliott, M. et al., 2013. Aluminium for steel; aluminium, plastics fiber optics or steel and graphene for copper; palladium for platinum; iron for nickel; potential rare earths substitution. In: Supercycle hangover; The rising threat of substitution. pages 8-10. EY Report. http://www.ey.com/GL/en/Industries/Mining---Metals/

Eliott, M. et al., 2014. Business risks facing mining and metals 2014–2015. EY Report 56pp. http://www.ey.com/GL/en/Industries/Mining - Metals/Business-risks-in-mining-and-metals

Elshakaki, A., Graedel, T.E. 2013. Dynamic analysis of the global metals flows and stocks in electricity generation technologies. Journal of Cleaner Production 59:260-273

Elshkaki, A., Graedel, T.E., 2015. Solar cell metals and their hosts: a tale of oversupply and undersupply. Appl. Energ. 158, 167–177.

Elshkaki, A., Graedel, T.E, Ciacci, L., Reck, B.K., 2016. Copper demand, supply, and associated energy use to 2050. Global Environmental Change 39: 305-315.

Elsner, H., Melcher, F. Schwarz-Schampera, U., Buchholz, P. 2010: Elektronikmetalle - zukünftig steigender Bedarf bei unzureichender Versorgungslage?- Commodity Top News Nr. 33: 13 p., Bundesanstalt für Geowissenschaften und Rohstoffe, Hannover.

Encyclopedia Iranica 2017. Mining in iIran I. Mines and mineral resources. http://www.iranicaonline.org/articles/mining-in-iran-i

Energetics, Inc. 2004. Steel Industry - Energy Bandwidth Study. Available at www1.eere.energy.gov/industry/steel/pdfs/steelbandwidthstudy.pdf

Energy, Information Administration 2007. U.S. Crude Oil, Natural Gas, and Natural Gas Liquids Reserves, 2006 Annual Report. DOE/EIA-0216.

EuroInox 2014. International Stainless Steel Forum Austenitic chromium-manganese stainless steels, A European approach. The European Stainless steel development Association. Materials and application Series. 12:2-20. ISBN 978-2-87997-321-0

Evans, R.K., 2008a, Lithium in abundance – World lithium reserve. A report on the world's Lithium resources and reserves. 5pages http://lithiumabundance.blogspot.com/

Evans, R.K. 2008b. An Abundance of Lithium Part Two. 2008. Available online: http://www.evworld. com/library/KEvans LithiumAbunance pt2.pdf

Figueiredo, M.O., Silva, T.P., Batista, M.J., Veiga, J.P. and de Oliveira, D.P. 2013 Opportunities for Recovering Critical Raw Materials from Mine Wastes: The Type-Case of Rhenium in Residues from the Exploitation of Old Portuguese Mines. Wastes: Solutions, Treatments and Opportunities, 2nd International Conference, Braga/Portugal, September 2013, Poster, Book of Abstracts, 163-164.

Fisher, K.G., 2011. Cobalt processing developments. The Southern African Institute of Mining and Metallurgy 6th Southern African Base Metals Conference 2011. 22 pages

Fischer-Kowalski, M., and Swilling, M., von Weizsäcker, E.U., Ren, Y., Morigichi, Y., Crane, W., Krausmann, F., Eisenmenger, N., Giljum, S, Heindricke, P., Romero Lankao, P., Sinban Manalang, A. 2011 United Nations Environment Programme. Decoupling natural resources use and environmental impacts from economic growth. A report of the working group on decoupling to the International Resource Panel. 150pp

Fitzgerald, P., French, G., Fletcher, E.C.G., Welch, B., Hyland, M., 1990. The production of aluminium. VIII-Metals-D-Aluminium production. 10pp. Otago, New Zealand. http://nzic.org.nz/ChemProcesses/metals/8D.pdf

Fleischer, M., 1959, The geochemistry of rhenium, with special reference to its occurrence in molybdenite, Economic Geology, 54:1406-1413.

Forrester J. 1971. World Dynamics. Pegasus Communications, Waltham, Massachusetts

Fouquet, Y., and Scott, S.D., 2009. The science of seafloor massive sulfides (sms) in the modern ocean a new global resource for base and precious metals. 13pp Offshore Tecnology Conference OTC Paper Number : OTC- 19849-PP

Fox-Davies Inc. 2013. The lithium market. 15 pp. Fox-Davies Capital, New York

Fritz, H., Abdelsalam, M., Ali, K.A., et al. 2013 Orogen styles in the East African Orogen: A review of the Neoproterozoic to Cambrian tectonic evolution. J. African Earth Sciences 86, 65–106.

Fukuyama, F. 2006. After the Neo Cons: Where the Right went Wrong. 345pp. Profile Books, 2006. ISBN 1-86197-922-3

Fukuyama, F., 2011. The origins of political order. From prehuman times to the French revolution. 550pp. Farrar, Strauss and Giroux. New York. Profile Books ISBN 978-1846682568

Fukuyama, F., 2014. Political Order and Political Decay: From the Industrial Revolution to the Globalization of Democracy. Farrar, Straus and Giroux, 2014. ISBN 978-0-374-22735-7

Fujitsuka, N., Daigo, I., Goto, Y., Nakajima, K., Matsuno, Y. 2013. Dynamic modelling of the world steel cycle toward 2050. 6th International Conference on life cycle management in Gothenburg 2013. LCM2013. 4 pages extended abstract.

Gaballa, O.G., 2012. Processing development of 4TaC-HfC and related carbides and borides for extreme environments. Graduate Theses and Dissertations. Paper 12635. PhD Thesis. 125pp. Iowa State University. Digital Repository @ Iowa State University. http://lib.dr.iastate.edu/etd

Gaines, L., 2014, The future of automotive lithium-ion battery recycling: Charting a sustainable course. Sustainable Materials and Technologies 1–2:2–7

Gaines, L, Nelson, P., 2015. Lithium-ion batteries: Possible material demand issues. US Department of Energy Laboratories. University of Chicago. Argonne National Laboratory. 2 page pamphlet.

Gang, L. Colton, E. B., Müller, D. B. 2013. Stock dynamics and emission pathways of the global aluminium cycle. Nature Climate Change 3, 338–342 doi:10.1038/nclimate1698

GARC 2011. Global Aluminium Recycling Model. London: Global Aluminium. Recycling Committee, International Aluminium Institute.

Gauthier, M., Gardoll, S., Duchesne, J-C., 2004. Quest for palladium: Application of the Groves hypothesis to Precambrian terranes of North America. Geology: 32:593-597.

Geddes, B., leon, H., Huang, X., 2010. Superalloys; Alloying and performance. ASM International, Materials Park, Ohio. 167pp, ISBN 13: 978-1-61503-040-8

Geoscience Australia. 2009 Australia’s identified mineral resources in 2009. Canberra, Australia.

George, M.W., and Wagner, L.A., 2009, Selenium recycling in the United States in 2004, chap. T of Sibley, S.R., Flow studies for recycling metal commodities in the United States: U.S. Geological Survey Circular 1196–T, T1–T7, available only at http://pubs.usgs.gov/circ/circ1196t/. 12pp.

Gerst, M., 2008 Revisiting the cumulative grade-tonnage relationship for major copper ore types. Economic Geology 2008, 615-628.

Gerst, M., 2009 Linking Material Flow Analysis and Resource Policy via Future Scenarios of In-Use Stock: An Example for Copper. Environmental Science and Technology, 6320-6325.

Gever, J., Kaufmann, R., Skole, D., Vorosmarty, C., 1986. Beyond oil, the threat to food and fuel in the coming decades. Ballinger Publishing Co. Cambridge, MA, 304 pp

Ghose, A. K., 2009. Technology vision 2050 for sustainable mining. Procedia Earth and Planetary Science 1:2-6.

Gibson, C., Hayes, T. 2011. Sector research; Indium and gallium. Edison Investment Research. San Fransisco CA.

Giljum, S., Hinterberger, F., Bruckner, M., Burger, E., Frühmann, J., Lutter, S., Pirgmaier, E., Polzin, C., Waxwender, H., Kernegger, L., Warhurst, M., 2000. Overconsumption? Our use of the world̕ s natural resources. 36pp. © SERI, GLOBAL 2000, Friends of the Earth Europe, September 2009

Giljum, S. and K. Hubacek. 2009. Conceptual foundations and applications of physical input-output tables. In Handbook of input-output economics in industrial ecology, edited by S. Suh, pp. 61–75. Dordrecht, the Netherlands: Springer

Giljum, S., Hinterberger, F., Lutz, C., Meyer. B., 2008: Accounting and modelling global resource use: material flows, land use and input-output models. In: Suh, S. [ed.]: Handbook of input-output economics for industrial ecology, Dordrecht.

Ginzburg, A. I., and L. G. Fel’dman 1974. Mestorozhdeniia tantala i niobiia. In Rudnye mestorozhdeniia SSSR, vol. 3. Moskva.

Giurco, D. 2005. Towards sustainable metal cycles, the case of copper. Ph D Thesis. Department of Chemical Engineering, University of Sydney. 322pp.

Giurco, D., Cohen, B., Langham, E., Warnken, M. 2012. Backcasting energy futures using industrial ecology. Technol. Forcasting Soc. Change 78: 797–818

Giurco, D, Mohr, S., Mudd, G.M., 2013. Resources and supply-demand over the very long term. GSA’s 12. Anniversary Annual Meeting & Expo. 39 powerpoint slides. Pardee Keynote Symposium P12: Resourcing Future Generations 27-30 Oct 2013, Denver USA

Glaister, B.J., Muss, G.M., 2010. The environmental costs of platinum-PGM mining and sustainability: Is the glass half-full or half-empty? Minerals Engineering 23:438-450.

Glöser, S., Soulier, M., Espinoza, L.A.T., 2013 Dynamic Analysis of Global Copper Flows. Global Stocks, Postconsumer Material Flows, Recycling Indicators, and Uncertainty Evaluation. Environmental Science & Technology, 47, 6564-6572.

Glöser, S., Soulier, M., Espinoza, L.T., Faulstich, M., 2013. Using Dynamic Stock & Flow Models for Global and Regional Material and Substance Flow Analysis in the Field of Industrial Ecology: The Example of a Global Copper Flow Model. The 31st International Conference of the System Dynamics Society Cambridge, Massachusetts USA 20pp.

Goe, M., Gaustad, G., 2014. Identifying critical materials for photovoltaics in the US: A multi-metric approach. Allied Energy 123:387-396.

Goll, D., Loeffler, R., Herbst, J., Karimi, R., Pflanz, U., Stein, R., Schneider, G. 2015. Novel permanent magnets by high throughput experiments. JOM 67: 1336-1344.

Goonan, T.G., 2012, Lithium use in batteries: United States Geological Survey Circular 1371, 22 pp., available at http://pubs.usgs.gov/circ/1371/

Gordon, R.B. 1996, American Iron 1607-1900, The Johns Hopkins University Press

Gordon, R.B. Lifset, R.J. Bertram, M. Reck, B. Spatari, S. Graedel T.E. 2004. Where Is All the Zinc Going ?: The Stocks and Flows Project, Part 2 JOM

Gordon, R. Bertram, B., Grädel M., 2006 Metal stocks and sustainability. Proceedings of the National Academy of Science; 103:1209-1214

Gorzelany, J. 2014. The Cold Truth: Icy Temps Can Slash An Electric Car's Range By More Than Half. Forbes Magazine. http://www.forbes.com/sites/jimgorzelany/2014/03/24/the-cold-truth-icy-temps-can-slash-an-electric-cars-range-by-more-than-half/

Gotthelf, P. 2005, Precious metals trading –How to profit from major market movements. John Wiley and son, New York. 320pp

Grabezev, A.I., 2012. Rhenium in porphyry copper deposits of the Urals. Geology of ore deposits 55:13-26. Geologiya Rudnykh Mestorozhdenii, 2013, 55:16–32.

Graedel T. E., and Allenby, B. R. 2003. Industrial Ecology, second edition. 363pp. Pearson Education Inc. New Jersey. AT&T.

Graedel, T.E., van Beers, D., Bertram, M., Fuse, K., Gordon, R., Gritsinin, A. 2004. A. Multi-level cycle of anthropogenic copper. Environmental Science and Technology 38:1242-1252.

Graedel, T.E., Beers, D., Bertram, M., Fuse, K., Gordon, R.B., Gritsin, A., Harper, E.M., Kapur, A., Klee, R.J., Lifset, R., Memon, L., Spatari, S., 2005. The multi-level cycle of anthropogenic zinc. Journal of Industrial Ecology, 9: 67–90

Graedel, T.E, Buckert, M., Reck, B.K., 2011 Assessing mineral resources in society. Metal stocks and recycling rates. UNEP, Paris, ISBN 978-92-807-3182-0

Graedel, T.E., Barr, R. , Cordier, D., Enriquez, M., Hagelüken, C., Hammond, N.Q., Kesler, S., Mudd, G., Nassar, N., Peacey, J., Reck, B.K., Robb, L., Skinner, B., Turnbull, I., Ventura Santos, R., Wall1, F., Wittmer D., 2011. Estimating Long-Run Geological Stocks of Metals. UNEP International Panel on Sustainable Resource Management Working Group on Geological Stocks of Metals. Working Paper, 32pp. http://www.unep.org/resourcepanel /Portals/24102/PDFs/GeolResourcesWorkingpaperfinal040711.pdf

Graedel, T.E. and Erdmann, L., 2012. Will metal scarcity impede industrial use? Metals Research Society Bulletin 37:325-333.

Graedel, T.E., Harper, E.M., Nassar, N.T., Reck, B.K., 2015a. On the materials basis of modern society. Proc. Natl. Acad. Sci. U. S. 112, 6295–6300.

Graedel, T.E., Harper, E.M., Nassar, N.T., Nuss, P., Reck, B.K., 2015b. Criticality of metals and metalloids. Proc. Natl. Acad. Sci. U. S. 112, 4257–4262.

Grandell, L., Thorenz, A., 2014. Silver supply risk analysis for the solar sector. Renewable Energy 69:157-165.

Greer, J.M., 2008. The Long Descent. A Users Guide to the End of the Industrial Age. New Society Publishers, Gabriola Island, Canada.

Grjotheim, K. and Welch, B.J.; 1988. Aluminium Smelter Technology - a Pure and Applied Approach (2nd. ed.); Aluminium-Verlag, Køln.

Grosjean, C., Miranda, P.H., Perrin, M., Poggi, P. 2012 Assessment of world lithium resources and consequences of their geographic distribution on the expected development of the electric vehicle industry. Renewable and Sustainable Energy Reviews 16:1735–1744

Gruber P., and Medina, P., 2010. Global lithium availability. A constraint for electric vehicles?. A practicum submitted in partial fulfillment of the requirements for the degree of Master of Science (Natural Resources and Environment) at the University of Michigan. 76 pages.

Gruber, P,W., Medina, P.A., Keoleian, G.A, Kesler, S.E., Everson, M.P., Wallington, T.J., 2011. Global Lithium Availability A Constraint for Electric Vehicles? Journal of Industrial Ecology 12:1-16

Gu, Y. F. Harada, H., Ro, Y., 2004. Chromium and chromium-based alloys: Problems and possibilities for high-temperature service. JOM 56: 28-33

Guirco, D., Littleboy, A., Boyle, T., Fyfe, J., White, S., 2014. Circular Economy: Questions for responsible minerals, additive manufacturing and recycling of metals. Resources 2014, 3, 432-453; doi:10.3390/resources3020432

Gutowski, T.G., Sahni, S., Allwood, M., Ashby, M.F., Worrell, E., 2016. The energy required to produce materials: constraints on energy-intensity improvements, parameters of demand. Philosophical Transactions of the Royal Society A 371: 20120003. http://dx.doi.org/10.1098/rsta.2012.0003

Gürmen, S. and B. Friedrich. 2004. Recovery of Cobalt Powder and Tungsten Carbide from Cemented Carbide Scrap – Part I: Kinetics of Cobalt Acid Leaching. Aachen, Germany: IME Process Metallurgy and Metal Recycling, RWTH Aachen University.

Hall, C., Lindenberger, D., Kummel, R., Kroeger, T., Eichhorn, W. 2001. The need to reintegrate the natural sciences with economics. BioScience, 51, 663-667.

Hall, C. 2008 Energy return on investment and our economic future. In Proceedings of the 33rd International Geological Congress, Oslo, Norway, August 2008.

Hall, C.A.S., Balogh, S., Murphy, D. J. R. 2009 What is the minimum EROI that a sustainable society must have? Energies, 2, 25-47.

Hagelüken, C., Buchert, M., Stahl, H., 2005. Materials flow of platinum group metals. GFMS Ltd., London.

Hagelücken, C. 2006. Markets for the catalyst metals platinum, palladium and rhodium. Metal 60:31-43.

Hagelüken, C, Meskers, C.E.M. 2009. Complex life cycles of precious and special metals. In; Linkages of Sustainability. Strüngmann Forum Report vol. 4. T. Graedel, E. van der Voet (eds.) Cambridge, MA, MIT Press 2009

Hagelücken, C. 2012, Recycling the platinum group metals: A European perspective. Platinum Metals Review, 56:29-35

Halada, K., Shimada, M., Ijima, K., 2008. Forecasting of the consumption of metals up to 2050. Materials Transactions. 49:402-410.

Hannington M., Jamieson J., Monecke T. Petersen S. 2014 Seafloor Massive Sulfide Deposits. Towards an estimate of global sms deposits. 51 pp. http://fileserver.futureocean.org/forschung/r3/Hannington_SMS_resource_potential.pdf

Hannis, S., and Bide, T., 2009. Cobalt. Commodity Profiles; British Geological Survey, Keyworth, Nottingham, NG12 5GG, United Kingdom

Harada, H., 2012. Development of advanced single crystal superalloys for aeroengines and gas turbines. Japan Foundry Society, Inc. Technical lecture. 10pp. http://doc.assofond.it/13th_World_Conference_Investment_Casting/lectures/TechnicalEngineeringLecture/T01.pdf

Haraldsson H.V. and Sverdrup H.U. 2004. Finding Simplicity in complexity in biogeochemical modelling. In: Wainwright, J and Mulligan, M. (Eds.) Environmental Modelling: A Practical Approach, 211-223. J. Wiley and Sons Ltd., Chichester.

Haraldsson, H. V. 2004. Introduction to systems thinking and causal loop diagrams. Technical report, Lund University, Department of Chemical Engineering. Reports in Ecology and Environmental Engineering, pp. 1–49.

Haraldsson H. and Sverdrup, H., 2005, On aspects of systems analysis and dynamics workflow. Proceedings of the systems dynamics society, July 17-21, 2005 International Conference on systems dynamics, Boston, United States of America. 1-10 pages. http://www.systemdynamics.org/conferences/2005/proceed/papers/HARAL310.pdf

Harben, P.W., and Kuzvart, M., 1996, Industrial minerals - a global geology: London, U.K., Industrial Minerals Information Plc., 476 pp

Harmsen, J.H.M., Roes, A.L., Patel, M.K., 2013. The impact of copper scarcity on the efficiency of 2050 global renewable energy scenarios. Energy 50: 62-73

Harper, E.M. 2008. A Product-Level Approach to Historical Material Flow Analysis: Tungsten as a Case Study. Journal of Industrial Ecology 12, no. 5/6

Harper, E.M., Kavlak, G., Graedel, T.E. 2011. Tracking the Metal of the Goblins: Cobalt's Cycle of Use. Environ. Sci. Technol. 2012, 46, 1079−1086. dx.doi.org/10.1021/es201874e

Harper, E.M., Kavlak, G., Graedel, T.E. 2012. Tracking the metal of the goblins: Cobalt's cycle of use. Environmental Science and Technology 46:1079−1086.

Harris, K., Erickson, G.L., Brentnall, W.D., Aurrecoechea, J.M., 1992. Development of the rhenium containing superalloys CMSX-4 and CM-186LC for single crystal blade and directionally solidified vane applications in advanced turbine engines 10 pages facsimile of article. http://www.tms.org/superalloys/10.7449/1992/Superalloys_1992_297_306.pdf

Hatayama, H.; Yamada, H.; Daigo, I.; Matsuno, Y.; Adachi, Y., 2007 Dynamic substance flow analysis of aluminum and its alloying elements. Mater. Trans., 2518-2524.

Hatayama, H., Daigo, I, Matsuno, Y., Adachi, Y., 2010. Outlook of the world steel cycle based on the stock and flow dynamics. Environmental Science and Technology 44:6457-6463.

Hatchett, C., 1802a. An Analysis of a Mineral Substance from North America, Containing a Metal Hitherto Unknown. Philosophical Transactions of the Royal Society of London 92: 49–66. doi:10.1098/rstl.1802.0005. JSTOR 107114.

Hatchett, C., 1802b. Eigenschaften und chemisches Verhalten des von Charles Hatchett entdeckten neuen Metalls, Columbium. Annalen der Physik 11: 120–122. Bibcode:1802AnP....11..120H. doi:10.1002/andp.18020110507.

Hawkins, M. 2001. Why we need cobalt. Applied Earth Science: Transactions of the Institution of Mining & Metallurgy, Section B 110;2:66–71. doi:10.1179/aes.2001.110.2.66.

Helmink, R.C., Testin, R.A., Price, A.R., Pachman, R., Erickson, G.L., Harris, K., Nesbittt, J.A. Radavich J.F., 2000. Advanced superalloys and tailored microstructures for integrally cast turbine wheels In: Superalloys 2000. Edited by TM. Pollock, R.D. Kissinger, R.R. Bowman, K.A. Green, M. McLean, S. Olson, and J.J. Schim. The Minerals, Metals &Materials Society, http://cannonmuskegon.com/wp-content/uploads/2015/02/Advanced-Superalloy-and-Tailored-Microstrucres-for-Integrally-Cast-Turbine-Wheels-2000.pdf

Helvaci, C., Mordogan, H., Colak, M., Gündogan, I., 2003, Presence and Distribution of Lithium in Borate Deposits and Some Recent Lake Waters of West-Central Turkey. International Geology Review, 45:101-114

Henckens, M.L.C.M., Driessen, P.P.J., Worrell, E., 2014. Metal scarcity and sustainability, analyzing the necessity to reduce the extraction of scarce metals. Resources, Conservation and Recycling 93:1–8

Hein 2004. Cobalt-rich ferromanganese crusts: Global distribution, composition, origin and research activities. chapter 5; In: Cobalt-Rich Ferromanganese Crusts: Global Distribution, Composition, Origin and Research Activities. Minerals other than polymetallic nodules of the International Seabed Area. 188-281: proceedings . ISBN: 976-610-647-9 (pbk)

Hein, J.R., Conrad, T.A., Dunham, R.E. 2009. Seamount characteristics and mine-site model applied to exploration- and mining-lease-block selection for cobalt-rich ferromanganese crusts. Marine Georesources and Geotechnology, 27, 160-176, DOI: 10.1080/10641190902852485.

Hein, J.R., Conrad, T.A., Staudigel, H. 2010. Seamount mineral deposits, a source of rare-metals for high technology industries. Oceanography, 23, 184-189.

Hein, J.R., Mizell, K., Koschinsky, A., Conrad, T.A., 2013. Deep-ocean mineral deposits as a source of critical metals for high- and green-technology applications: Comparison with land-based resources. Ore Geology Reviews 51: 1–14

Heinberg, R., 2001. Peak Everything: Waking Up to the Century of Decline in Earth's Resources 270pp, Clairview.

Heinberg, R., 2011, The end of growth. Adapting to our new economic reality. New Society Publishers, Gabriola Island, Canada.

Helmink, R.C., Testin, R.A., Price, A.R., Pachman, R., Erickson, G.L., Harris, K., Nesbittt, J.A. Radavich J.F., 2000. Advanced superalloys and tailored microstructures for integrally cast turbine wheels In: Superalloys 2000. Edited by TM. Pollock, R.D. Kissinger, R.R.

Herzig, P.M., Hannington, M.D., 1995. Polymetallic massive sulphides at the modern seafloor; A review. Ore Geology Reviews 10, 95-115.

Henckens, M.L.C.M., Driessen, P.P.J., Worrell, E., 2014. Metal scarcity and sustainability, analyzing the necessity to reduce the extraction of scarce metals. Resources, Conservation and Recycling 93:1–8

Hess, F. L. 1924. Molybdenum deposits. A short review. United States Geological Survey. Bulletin 761. 49pp. Washington D.C.

Hilliard, H. E. 2001. Platinum recycling in the United States in 1998. Circular 1196-B (2001). at http://pubs.er.usgs.gov/publication/cir1196B

Hirsch, R.L., Bezdek, R., Wending, R. 2005 Peaking of world oil production: Impacts, mitigation and risk management. The Hirsch report 2005. www.netl.doc.com/publ/other/pdf/oil_publ_net/pdf/

Hirschnitz-Garber, M., Langsdorf, S.,; Sverdrup, H.; Koca, D.; Distelkamp, M., Meyer, M., 2015. Integrated modelling for resource policy assessment – the SimRess-project. Proceedings of the 2015 Word Resources Forum, 11-15 September. Davos, Switzerland. Accepted.

Hitch, M., Bamber, A., Oka, P. 2015. Presorting of high grade molybdenum ore-A case for enhanced small mine development. International Journal of Engineering and Applied Science 2:129-135.

Hjelm, P.J., 1788. Versuche mit Molybdäna und Reduction der selben Erde. Kongliga Vetenskaps Academiens Handlingar 49:268-278

Hoagland, P., Beaulieu, S., Tivey, M., Eggert, R.G., German, C., Glowka, L., Lin., J, 2009. Deep-sea mining of seafloor massive sulfides. Marine Policy 34: 728–732

Hoke, C.M. 1940. Refining precious metal waste. Gold-silver-platinum. A handbook for the leweler, dentist and small refiner. Metallurgical Publishing Co. New York. 362pp

Horwath, A. 2004. Construction materials and the environment. Annu. Rev. Environ. Resour. 2004. 29:181–204. doi: 10.1146/annurev.energy.29.062403.102215

Hsu, F-C., Eidvige, C.D., Matsuno, Y. 2011. Estimating in-use steel stock of civil engineering and building in China by nighttime light image. Proceedings of the Asia-Pacific Advanced Network, 31:58-68. http://dx.doi.org/10.7125/APAN.31.7 ISSN 2227-3026

Hu, M., Pauliuk, S. Wang, T., Huppes, G., ven der Voet, E., Müller, D., 2010. Iron and steel in Chinese residential buildings; A dynamic analysis. Resources, Conservation and Recycling 54: 591-600.

Hubbert, M. K., 1956. Nuclear energy and the fossil fuels. Presented to the spring meeting of the southern district, Division of production. American Petroleum Institute. Plaza Hotel, San Antonio, Texas. Publication no. 95, Shell Development company. Exploration and productions research division, Huston Texas.

Hubbert, M. K. 1962. Energy Resources. A Report to the Committee on Natural Resources; National Academy of Sciences, National Research Council: Washington, DC, USA, 1962; p. 54.

Hubbert, M. K., 1966. History of Petroleum Geology and Its Bearing Upon Present and Future Exploration. AAPG Bulletin 50:2504-2518.

Hubbert, M. K., 1972. Estimation of Oil and Gas Resources. In: U.S. Geological Survey, Workshop on Techniques of Mineral Resource Appraisal, 16-50. Denver, U.S. Geological Survey.

Hubbert, M. K., 1976. Exponential growth as a transient phenomenon in human history. Paper presented to World Wildlife Fund, Fourth International Congress, The Fragile Earth, Towards strategies for survival. San francisco. Other issues 75-84

Hubbert, M.K. 1982. Techniques of Prediction as Applied to Production of Oil and Gas, United States Department of Commerce, NBS Special Publication 631, May 1982. Denver, U.S. Geological Survey

Hughes, F.E. (Ed) 1990 Geology of the Mineral Deposits of Australia and Papua New Guinea, Monograph 14, Australasian Institute of Mining and Metallurgy, Melbourne.

Hulbert, L.J., and von Gruenewaldt, G., 1982, Nickel, copper, and platinum mineralization in the Lower Zone of the Bushveld Complex, south of Potgietersrus: Economic Geology, 77:1296–1306.

Idaho Cobalt Mine, United States of America 2015. http://www.mining-technology.com/projects/idaho-cobalt-mine/ Web-page with mining news.

Implats (Impala Platinum Holdings Limited) 2007. Annual report, Mineral resources and mineral reserves. http://www.implats.co.za/im/files/ar/2007/reserves_resources.htm

Implats (Impala Platinum Holdings Limited) 2015. Corporate news website. http://www.implats.co.za/implats/index.asp

IndexMundi 2015 http://www.indexmundi.com/minerals/?product=rhenium

INSG 2013. Nickel-based superalloys. INSG secretariat briefing paper. 2013:20. 9pages. http://www.insg.org/%5Cdocs%5CINSG_Insight_20_Nickel_Alloys_2013.pdf

International Business Publications, 2013; Ukraine; Mineral, mining sector investment and business guide. Volume 1. Strategic informations and regulations. 230 pp. International Business Publications, Washington DC, USA. ISBN 978-1-4330-5184-2

International Business Publications, 2015. Russia and newly independent states (NIS) 2015; Mining and mineral handbook. . Volume 1. Russia mining industry: Strategic information, regulations, contacts. 312pp. Published by the International Business Publications, Washington DC, USA. ISBN 978-1-329-09121-4

International Council of Mining and Metals (ICMM) 2012. Trends in the metals industry..Mining's contributions to sustainable development.. In brief. London. 16pp

International Energy Agency (IEA) 2014. Key world energy statistics. Energy data centre of the IEA, www.iea.org/statistics. Paris, France Downloadable as pdf. 82pp

International Molybdenum Association 2013 Excerpt from the MolyReview 2013. Molybdenum scrap saves resources. 3pp. International Molybdenum Association. http://www.imoa.info/download_files/molyreview/excerpts/13-2/Molybdenum_scrap_saves_resources.pdf

International Molybdenum Association 2015. A sustainable world with molybdenum. 28 pages. http://www.imoa.info/download_files/sustainability/Sustainability-brochure-2015.pdf

International Platinum Association 2015. International Platinum Group Metals Association. Website with sector news. http://ipanews.com/

International Resource Panel (IRP-UNEP) 2012. Responsible Resource Management for a Sustainable World. Findings of the International Resource Panel. 32pp. ISBN 978-92-807-3278-0. 2012

International Seabed Authority 2015. Cobalt-rich crusts. 4 pages. https://www.isa.org.jm/files/documents/EN/Brochures/ENG9.pdf

International stainless steel forum 2014.The stainless steel family. 5 pp International Stainless Steel Forum Rue Colonel Bourg 120 B-1140 Brussels Belgium, www.worldstainless.org

International Zinc Association 2014 Zinc statistics. http://www.zinc.org/basics/zinc_statistics

International Zinc Association, 2014 Zinc coatings protecting steel, www.zinc.world

International Zinc Association, 2013. Zinc recycling, closing the loop, www.zinc.org

Jacobson, M., Delucchi, M., 2011. Providing all global energy with wind, water, and solar power, Part I: Technologies, energy resources, quantities and areas of infrastructure, and materials. Energy Policy 39:1154-1169

Jaskula, B. 2010. Lithium. Mineral Commodity Summaries. United States Geological Survey. USGS website. http://minerals.usgs.gov/minerals/pubs/commodity/

Jin, T., Wang, W.Z., Sun, X.F., Hu, Z.Q., 2010. Role of Rhenium in Single Crystal Ni-Based Superalloys, Materials Science Forum, Vols. 638: 2257-2262

Jochens, P.R., 1980. The energy requirements of the mining and metallurgical industry in South Africa. Journal of the South African Institute of Mining. 331-343.

Johnson, J., Harper, E.M., Lifset, R., Graedel, T.E., 2007. Dining at the periodic table: Metals concentrations as they relate to recycling. Environmental Science and Technology 41:1759-1765

Johnson, K.M., Hammarstrom, J.M., Zientek, M.L., Dicken, C.L. 2014. Estimate of undiscovered copper resources of the world, 2013: U.S. Geological Survey Fact Sheet 2014–3004, 3 p., http://dx.doi.org/10.3133/fs20143004.

Johnson Mathey 2015. Market tables. http://www.platinum.matthey.com/services/market-research/market-data-tables

Jorgenson, J., 2009, Germanium recycling in the United States in 2000, chap. V of Sibley, S.R., Flow studies for recycling metal commodities in the United States: U.S. Geological Survey Circular 1196–T, T1–T7, available only at http://pubs.usgs.gov/circ/circ1196t/. 15pp

Jorgenson, J.D., and George, M.W., 2004. Indium. USGS Mineral Commodity Profile. Open-File Report 2004-1300. 24pp.

Kablov, E.N., Demonis, I.M., Petrushkin, N.V., 2012. Materials and Technologies for New Generation Aeroengines http://eu-cass2011.conferencecenter.ru/cs/upload/gF76bMq/papers/papers/823-1252-1-RV.pdf

Kalenga, M., Xiaowei, P., Tangstad, M. 2013. Manganese alloys production: Impact of chemical compositions of raw materials on the energy and materials balance. Manganese Fundamentals. Proceedings of the thirteenth International Ferroalloys Congress on efficient technologies in ferroalloy industry June 9 - 13, 2013 Almaty, Kazakhstan. Pages 647-675.

Kasikov, L.G. and Petrova, A.M., 2009. Processing of Deactivated Platinum-Rhenium Catalysts. Chemical and metallurgical processes of crude ore and anthropogenic and secondary raw materials integrated processing. Theoretical Foundations of Chemical Engineering, 43:544–552.

Kavlak, G., Graedel. T.E., 2013. Global anthropogenic selenium cycles for 1940-2010. Resources, conservation and recycling 73:17-22.

Kavlak, G., Graedel. T.E., 2013. Global anthropogenic tellurium cycles for 1940-2010. Resources, conservation and recycling 76:21-26.

Kawagishu, K., Yeh, A., Yokokawa, T., Kobayashi, T., Koizumi, Y., Harada H., 2012. Development of an oxidation-resistant high-strength sixth-generation single-crystal superalloy TMS-238. In Huron, E., Reed, R.C., Hardy, M.C., Mills, M.J., Montero, R.E, Potella, P.D., Telesman, J., Superalloys 2012; 12th International Symposium. The Minerals, Metals and Materials Society. 189-196.

Kennedy, P., 1987, The rise and fall of the great powers: Economic change and military conflict from 1500 to 2000. Random House

Keramidas, K, Kitous, A., Griffinn, B., 2012. Future availability and demand for oil, natural gas and key minerals. Polinares working paper nr. 30. EWU policy on natural resources.

Kesler, S.E., Gruber, P.W., Medina, P.M., Keoleian G.A., Everson M.P., Wallington, T.J. 2012. Global lithium resources: Relative importance of pegmatite, brine and other deposits. Ore Geology Reviews 48:55–69

Kesler, S.E. and Wilkinson, B.H. 2013. Earth's copper resources estimated from tectonic diffusion of porphyry copper deposits. Geology; 41:255-258.

Kesler, S.E. and Wilkinson, B.H. 2013. Tectonic-Diffusion Estimates of Global Mineral Resources: Extending the Method – Granitic Tin Deposits. In: Jenkin, G.R.T., Lusty, P.A.J., McDonald, I., Smith M.P., Boyce, A.J., Wilkinson, J.J. (eds) Ore Deposits in an Evolving Earth. Geological Society, London, Special Publications, 393. First published online October 7, 2013, doi 10.1144/SP393.6

Kifle, D., Sverdrup, H., Koca, D., Wibetoe G., 2012. A Simple Assessment of the Global Long Term Supply of the Rare Earth Elements by Using a System Dynamics Model. Environment and Natural Resources Research; 3:1-15. 2013 ISSN 1927-0488E-ISSN 1927-0496. URL: http://dx.doi.org/10.5539/enrr.v3n1p77

King, H. 2015. Spodumen, an important source of high purity lithium and a gemstone. 5 pages. Geology.com. www.geology.com/minerals/spodumene.

Kim, D.H., 1992. Toolbox: Guidelines for Drawing Causal Loop Diagrams, The Systems Thinker, Vol. 3, No. 1; 5–6

Knoeri, C., Wäger, P.A., Stamp, A., Althaus, H.J., Weil M., 2013 Towards a dynamic assessment of raw materials criticality: Linking agent-based demand - with material flow supply modelling approaches. Science of The Total Environment 461-462:808-812. Presented at the World Resources Forum, Davos, Switzerland. http://publicationslist.org/data/christof.knoeri/ref-10/Knoeri_etal_2011_WRF.pdf

Kogel, J.E., Trivedi, N.C., Barker, J.M., Krukowski, S.T., (Eds)., 2006, Industrial minerals and rocks, 7th ed.: Littleton, Colorado, Society for Mining, Metallurgy, and Exploration, 1548 p.

Koizumi, Y., Kobayashi, T., Jianxin, Yokokawa, T., Harada, H., Aoki, Y., Arai, M., 2003, Proceedings of the International Gas Turbine Congress 2003 Tokyo, 1-6. November 2-7, 2003

Korre, A., Durucan, S., 2007. Life Cycle Assessment of Aggregates. EVA025 –Final Report: Aggregates Industry Life Cycle Assessment Model: Modelling Tools and Case Studies. Waste & Resources Action Programme. The Old Academy. 21 Horse Fair, Banbury, Oxon OX16 0AH. 41pp.

Kostka, S. 2011. Sand, Gravel, and Crushed Stone: The World's Building Blocks. 26 ppt presentation. Minnesota Department of Natural Resources. http://www.d.umn.edu/prc/MMEW/2011%20MMEW%20PPTs/Aggregate%20SK.pdf

Krause, C., Diesing, M., Arlt, G., 2010. The physical and biological impact of sand extraction: a case study of the western Baltic Sea. Journal of Coastal Research 51:215-226.

Krausmann, F., Gingrich, S., Eisenmenger, N., Erb, K-H., Haber, H., Fischer-Kowalski, M., 2009. Growth in global materials use, GDP and population during the 20th century. Ecological Economics 68:2696-2705.

Kravchenko, S.M. and Pokrovsky, B.G. 1995. The Tomtor Alkaline Ultrabasic Massif and Related REE-Nb deposits, Northern Siberia. Economic Geology, 90, 676– 689.

Kunaz, I., 1994. Lithium resources. In Industrial minerals and rocks. United States Geological Survey, Washington DC. Society for mining, metallurgy and exploration Inc. Littleton, Colorado. Pages 599-613.

Kunilov, V. Y, 1994, Geology of the Noril'sk region—The history of discovery, prospecting, exploration and mining of the Noril'sk deposits, in Lightfoot, P.C., and Naldrett, A.J., eds., Proceedings of the Sudbury-Noril'sk Symposium: Ontario Geological Survey Special Volume 5:203–216.

Kushnir, D and Sandén, B., 2012. The time dimension and lithium resource constraints for electrical vehicles. Resources Policy 37:93-103

Kojima, K., 1999, Report on the cobalt-rich manganese crust resources in the waters of the Republic of the Marshall Islands: Based on the results of the cooperation study project on the deep sea mineral resources in selected offshore areas of the Sopac Region. SOPAC Technical Report 293

Kvakkel, J.H., Auping, W.L., Pruyt, E., 2014. Comparing Behavioral Dynamics Across Models: the Case of Copper. Systems dynamics Conference Proceedings. http://www.systemdynamics.org/conferences/2014/proceed/papers/P1337.pdf

Kwatra, S., Jun, C., Warren, S., Yang, M-Y., Graedel, T.E, 2012 Criticality of the geological copper family. Environmental Science and Technology, 46:1071−1078.

Laherrere, J., 2010. Copper peak. Posted by de Sousa, L., 2010 The oildrum Europe. April 1, 2010. 6307:1-27. http://europe.theoildrum.com/node/6307

Langer, W.H., 2011. A General Overview of the Technology of In-Stream Mining of Sand and Gravel Resources, Associated Potential Environmental Impacts, and Methods to Control Potential Impacts. Open-File Report OF-02-153. USGS. 17pp. Washington DC.

Langer, W., 2002. Managing and protecting aggregate resources. Open-File Report 02-415. USGS Denver, Colorado. 16pp.

Langer, W. 2014. Aggregates; Construction sand and gravel. Chapter 14; 159-170 report. http://www.segemar.gov.ar/bibliotecaintemin/LIBROSDIGITALES/Industrialminerals&rocks7ed/pdffiles/papers/014.pdf

Langsdorf, S.; Hirschnitz-Garbers, M.; Distelkamp, M.; Meyer, M.; Koca, D.; Sverdrup, H.; Lorenz, U. 2015. Integrated modelling for resource policy assessment – the SimRess-project approach. Proceedings of the 2015 Word Resources Forum, 11-15 September. Davos, Switzerland. Accepted.

Leal-Ayala, D.R., Allwood, J.M., Petavratzi, E., Brown, T.J., Gunn, G., 2015. Mapping the global flow of tungsten to identify key material efficiency and supply security opportunities. Resources. Conservation and Recycling 103, 19–28, http://dx.doi.org/10.1016/j.resconrec.2015.07.003.

Lehmann, B., Halder, St., Ruzindana Munana, J., Ngizimana, J. de la Paix, Biryabarema, M. 2014 The geochemical signature of rare-metal pegmatites in Central Africa: Magmatic rocks in the Gatumba tin–tantalum mining district, Rwanda. J. Geochem. Exploration (Special Volume on Mining and the Environment in Africa) 144, 528-538.

Lele, A., Bhardwaj, P., 2014. Strategic materials. A resource challenge for India. Pentagon Press, Institute for Defence Studies & Analyses. 221 pages, New Delhi, India. ISBN 978-81-8274-9

Lenzen, M., 2008. Life cycle energy and greenhouse gas emissions of nuclear energy: a review. Energy Convers. Manag. 49, 2178-2199.

Levenspiel, O. 1980, The coming of age of chemical reaction engineering, Chemical Engineering Science 35:1821-1839.

Levine R.M., and Wallace, G.J., 2009. The Mineral Industries of the Commonwealth of Independent States. 2005 Minerals Yearbook. U.S. Geological Survey (December 2007).

Lewis, R.S. and Clark, G.B. 1964. Elements of Mining. John Wiley & Sons Inc; 3rd edition. 780pp. ISBN-13: 978-0471533313

Li, L., Dunn, J.B., Zhang, X.X., Gaines, L., Chen, R.J., Wu, F., Amine, K., 2013. Recovery of metals from spent lithium-ion batteries with organic acids as leaching reagents and environmental assessment. Journal of Power Sources 233:180-189

Lien, R.H., 1985. Recovery of Lithium From a Montmorillonite-Type Clay. United States Department of the Interior, United States Bureau of Mines. RI 8967. 31 pages

Liessmann, W. 1994 Vom Kobalterz zum Köningsblau. Zur Geschichte des Skuteruder Kobaltbergbaues und des Modumer Blaufarbenwerkes in Südnorwegen. Emser Hefte 15: 1-64.

Lifton, J., 2006. The trouble with tungsten. Hard Assets, Resource Investor website. http://www.resourceinvestor.com/2006/01/31/trouble-tungsten

Liu, G., Bangs, C.E., Müller, D. B., 2011 Unearthing Potentials for Decarbonizing the U.S. Aluminum Cycle. Environmental Science & Technology 2011, 45, 9515-9522.

Liu, G., and Müller, D.B., 2012. Addressing sustainability in the aluminium industry: a critical review of life cycle assessments. Journal of Cleaner Production 35:108-117.

Liu, G., Bangs, C.E., Müller, D. B. 2013. Stock dynamics and emission pathways of the global aluminium cycle, Nature Climate Change 3, 338–342, doi:10.1038/nclimate1698

Liu, G., Müller, D.B. 2013a. Centennial Evolution of Aluminum In-Use Stocks on Our Aluminized Planet. Environ. Sci. Technol., 2013a, 47, 4882–4888. DOI: 10.1021/es305108p

Liu, G. Muller, D.B., 2013b Mapping the Global Journey of Anthropogenic Aluminum: A Trade-Linked Multilevel Material Flow Analysis. Environmental Science & Technology 2013, 47, 11873-11881.

Locmelis, M., Melcher F., Oberthur, T., 2010 Platinum-group element distribution in the oxidized main sulfide zone, Great Dyke, Zimbabwe. Mineralium Deposita 45:93-109

Løvik, A. N, Modaresi, R., Müller, D.B. 2014. Long-term strategies for increased recycling of automotive aluminum and its alloying elements. Environmental Science & Technology 48, 4257-4265

Ludington, S. and Plumlee, G.S., 2009, Climax-type porphyry molybdenum deposits: U.S. Geological Survey Open-File Report 2009–1215, 16 p.

Luong, V.T., Kang, D.J., An, J.W., Kim, M.J., Tran, T., 2013. Factors affecting the extraction of lithium from lepidolite. Hydrometallurgy 134–135:54–61

MacIntosh, A., Edwards-Jones, G., 2000 Discounting the children's future? Geophilos 1, 122-133.

Magyar, M.J., 2004. Molybdenum. 11pp. US Geological Survey Yearbook —2004

Magyar, M.J., 2004. Rhenium. 4pp. US Geological Survey Yearbook —2004

Makineni, S.K.; Nithin, B.; Chattopadhyay, K. 2015a. Synthesis of a new wolfram-free γ–γ′ cobalt-based superalloy by tuning alloying additions. Acta Materialia 85:85–94.

Makineni S.K., Nithin B., Chattopadhyay K. 2015. A new wolfram-free c–c' Co–Al–Mo–Nb-based superalloy Scr Mater 98:36-48.

Makineni, S.K.; Samanta, A.; Rojhirunsakool, T.; Alam, T.; Nithin, B.; Singh, A.K.; Banerjee, R.; Chattopadhyay, K. 2015b. A new class of high strength high temperature Cobalt based γ–γ′ Co–Mo–Al alloys stabilized with Ta addition. Acta Materialia 97:29–40.

Maps of the World 2012. World sand and gravel producing countries. http://www.mapsofworld.com/minerals/world-sand-and-gravel-producers.html

Marsden, J.O., 2008. Energy efficiency and copper metallurgy. In: Hydrometallurgy 2008: Proceedings of the Sixth International Symposium, Young, C.A., (Ed) 29-43.

Martchek, K.J., 2006. Modelling more sustainable aluminium; Case study. Sustainable Management of natural resources. Hüthig Jehle Rehm Gmbh, Landsberg, Germany

Martino, S., and Parson, L.M., 2011. A comparison between manganese nodules and cobalt crust economics in a scenario of mutual exclusivity. Marine Policy 36:790–800

Mason, L., Prior, T., Muss, G., Giurco, D., 2011. Availability, addiction and alternatives: three criteria for assessing the impact of peak minerals on society. Journal of Cleaner Production 19:958-966.

McGlade, C. and Ekins, P., 2015. The geographical distribution of fossil fuels unused when limiting global warming to 2 °C. Nature 187-190. doi:10.1038/nature14016. From Nature http://www.nature.com/nature/journal/v517/n7533/fig_tab/nature14016_F1.html.

McMillan, C., Moore, M., Keoleian, G., Bulkley, J., 2010 Quantifying US aluminum in-use stocks and their relationship with economic output. Ecological Economics, 2606-2613.

Meadows, D.H., Meadows, D.L. Randers, J., Behrens, W. 1972 Limits to growth. Universe Books, New York.

Meadows, D.L., Behrens III, W.W., Meadows, D.H., Naill, R.F., Randers, J., Zahn, E.K.O. 1974 Dynamics of Growth in a Finite World. Massachusetts: Wright-Allen Press, Inc.

Meadows D.H, Meadows D.L., Randers J. 1993 Beyond the limits: confronting global collapse, envisioning a sustainable future. Chelsea Green Publishing Company.

Meadows D.H, Randers, J., Meadows D.L. 2005 Limits to growth, the 30 year update Earthscan, Sterling VA.

Melcher F., Graupner T., Gäbler H.-E., Sitnikova M.A., Henjes-Kunst F., Oberthür Th., Gerdes A., Dewaele S. (2015) Tantalum–(niobium–tin) mineralisation in African pegmatites and rare metal granites: Constraints from Ta–Nb oxide mineralogy, geochemistry and U–Pb geochronology. Ore Geology Rev. 64: 667-719. http://dx.doi.org/10.1016

Memary, R., Giurco, D., Mudd, G., Mohr, S., Weng, Z. 2012. Copper case study: Australian resources, technology and future scenarios. Prepared for CSIRO Minerals Down Under Flagship, by the Institute for Sustainable Futures (University of Technology, Sydney) and the Department of Civil Engineering (Monash University), May 2012. ISBN 978-1-922173-45-4.

Meng, Z, Sun, G., Li, M., 1984. The strengthening effect of tantalum in nickel-base superalloys. 10 pages electronic facimile of article, Shanghai Iron and Steel Research Institute - 20 pp. http://www.tms.org/Superalloys/10.7449/1984/Superalloys_1984_563_572.pdf

Merivale, C., 2002. Lithium in ceramics. CerMa/ACS Southeast Regional Joint Meeting Sept 11, 2002 in Asheville, N.C. 7pp.

Merwede, 2014. Ooijens, S., Update of sand and gravel resources and extraction worldwide. 28 page slideshow. http://www.metso.com/miningandconstruction/MaTobox7.nsf/DocsByID/A9260A9C59A15848C2257D87004694A7/$File/Manufactured%20Sand.pdf

Meyer, B., Meyer, M., Distelkamp, M. 2012: Modeling green growth and resource efficiency: New results. Mineral Economics, 24, 145-154.

Meyer, B., Lutz, C. 2007. Resource productivity, environmental tax reform and sustainable growth in Europe. The GINFORS Model. Model overview and evaluation. http://www.petre.org.uk/pdf/sept08/petrE_WP3%202%20Ginfors.pdf. petrE (see http://www.petre.org.uk/papers.htm) is part of the Anglo-German Foundation research policy initiative: Creating sustainable growth in Europé (see http://www.agf.org.uk/currentprogramme/CreatingSustainableGrowthInEurope.php) 21pp.

Milford, R.L., Pauliuk, S., Allwood, J.M., Müller, D.B., 2013. The roles of energy and materials efficiency in meeting steel industry CO2 targets. Environmental science & Technology 47: 3455-3462.

MinEx Consultants 2010. www.minexconsulting.com/publications.html

Minor Metals Trade Association 2015. http://www.mmta.co.uk/tantalum-market-overview

Modaresi, R., Pauliuk, S., Løvik, A.N., Müller, D.B., 2014. Global carbon benefits on material substitution in passenger cars until 2050 and the impact on the steel and aluminium industries. Environmental Science and Technology 48:10776-10784

Mojarov, A. 2013. The Iron Ore Market Situation – views of UNCTAD. Powerpoint presentation shown at the 74th session of the OECD Steel Committee 1-2 July 2013, Paris, France. UNCTAD Trust Fund on Iron Ore Information. www.unctad.org/commodities

Mohr, S.H.; Mudd, G.M.; Giurco, D. 2010. Lithium Resources and Production: A Critical Global Assessment; Report prepared for CSIRO Minerals Down Under Flagship. Department of Civil Engineering, Monash University Institute for Sustainable Futures, University of Technology: Sydney, Australia

Mohr, S.H., Mudd, G.M., Giurco, D., 2012. Lithium Resources and Production: Critical Assessment and Global Projections. Minerals 2:65-84; doi:10.3390/min2010065

Mohr, S. and Evans, G. 2013. Projections of Future Phosphorus Production. PHILICA.COM Article number 380. 23pp

Mohr, S. and Ward, J. 2014. Helium production and possible projections, Minerals 4: 130–144.

Mohr, S., Mudd, G.M., Weng, Z., Giurco, D., 2014a. Modelling future copper ore grade decline based on a detailed assessment of copper resources and mining. Resources, Conservation and Recycling 83:190-201.

Mohr, S., Giurco, D., Yellishetty, M., Ward, J., Mudd, G. 2014b. Projection of iron ore production, Natural Resources Research. 11pp, DOI: 10.1007/s11053-014-9256-6

Mohr, S. H., Jianliang, W., Ellem, G., Ward, J., Giurco, D. 2014c. Projection of World fossil fuels by country, submitted to Fuel.

Moll, S., S. Bringezu, A. Femia, and F. Hinterberger. 2002. Ein Input-Output-Ansatz zur Analyse des stofflichen Ressourcenverbrauchs einer Nationalökonomie. Ein Beitrag zur Methodik der volkswirtschaftlichen Materialintensitätsanalyse. In F. Hinterberger, and H. Schnabl, eds. Arbeit-Umwelt-Wachstum. Nachhaltigkeitsaspekte des sektoralen Strukturwandels. Book on Demand, Norderstedt.

Molybdenum 2007. 15 pages. Mining Journal Supplement. http://www.auzex.com/pdfs/molybdenum_datasheet.pdf

Morfeldt, J., Nijs, W., Silveira, S., 2015, The impact of climate targets on future steel production e an analysis based on a global energy system model. Journal of Cleaner Production 103:469-482

Morrigan, T. 2010. Peak energy, climate change and the collapse of global civilization. The current peak oil crisis. 2nd edition, Global Climate Change, Human Security & Democracy, Orfalea Center for Global and International studies, University of California, Santa Barbara, United States of America. 247pp. 2010. www.global.ucsb.edu/climateproject

Morrow, D. 2011. Why manufactured sand?. Results, Minerals & Aggregates 26-27.

Moskalyk, R.R., 2003. Gallium: The backbone of the electronics industry. Minerals Engineering 16:921-929

Moskalyk, R.R., 2004. Review of germanium processing worldwide. Minerals Engineering 17:393-402.

Mottura, A., Reed, R.C, 2014. What is the role of rhenium in single crystal superalloys? MATEC Web of Conferences 14, 01001. DOI: 10.1051/matecconf/20141401001. Owned by the authors, published by EDP Sciences. 6pages.

Moynihan, M., Allwood, J. M., 2012. The flow of steel into the construction sector. Resources, Conservation and recycling. 68: 88-95.

Mudd, G.M., 2007. Gold mining in Australia: linking historical trends and environmental and resource sustainability. Environmental Science and Technology 10: 629-644.

Mudd, G.M. 2009, Historical Trends in Base Metal Mining: Backcasting to Understand the Sustainability of Mining. Proceedings of the “48th Annual Conference of Metallurgists”, Canadian Metallurgical Society, Sudbury, Ontario, Canada, August Issue.

Mudd, G M, 2009, Nickel Sulfide Versus Laterite, The Hard Sustainability Challenge Remains. Proc. “48th Annual Conference of Metallurgists”, Canadian Metallurgical Society, Sudbury, Ontario, Canada, August 2009.

Mudd, G.M., 2010. The Arrival of Peak Lead: Peak Environmental Impacts? LEAD Action news, 11:1. ISSN 1324-6011. 6 pages. http://www.lead.org.au/lanv11n1/lanv11n1-8.html

Mudd, G.M. 2010a. The environmental sustainability of mining in Australia: key megatrends and looming constraints. Resources Policy 35:98–115.

Mudd, G.M. 2010b. Global trends and environmental issues in nickel mining: Sulfides versus laterites. Ore Geol. Rev. 38:9–26.

Mudd, G.M. 2010c. The environmental sustainability of mining in Australia: Key mega-trends and looming constraints. Resources Policy 35:98-115.

Mudd, G.M., 2012. Key trends in the resource sustainability of platinum group elements. Ore Geol. Rev. 46:106–117.

Mudd, G.M., and Weng, Z., 2012, Base metals, in Letcher, T., and Scott, J.L., eds., Materials for a sustainable future: United Kingdom, Royal Society of Chemistry, p. 11–59.

Mudd, G.M., Weng, Z., Jowitt, S.M., 2013a. A detailed assessment of global Cu resource trends and endowments. Econ. Geol. 108-132.

Mudd, G.M., Weng, Z., Jowitt, S.M., Tirnbull, I.D., Graedel, T.E., 2013a. Quantifying the recoverable resources of by-product metals: The case of cobalt. Ore Geology Reviews 55: 87–98

Mudd, G.M., Weng, Z., Jowitt, S.M. 2013b. A Detailed Assessment of Global Cu Resource Trends and Endowments.. Economic Geology, 108:1163–1183

Mudd, G.M., and Jowitt, S.M. 2014b A Detailed Assessment of Global Nickel Resource Trends and Endowments. Econ. Geol . 109:1813-1841

Muiños, S.B., Hein, J.B., Frank, M., Monteiro, J.H., Gaspar, L., Conrad, T., Pereira, H.G., Abrantes, F. 2013 Deep-sea Fe-Mn Crusts from the Northeast Atlantic Ocean: Composition and Resource Considerations, Marine Georesources & Geotechnology, 31:1, 40-70

Mukherij, D Rösler, J., 2010 Co-Re-based alloys for high temperature applications: Design considerations and strengthening mechanisms. Journal of Physics: Conference Series 240 (2010) 012066 doi:10.1088/1742-6596/240/1/012066. 15th International Conference on the Strength of Materials (ICSMA-15)

Munk, L., 2011. Final Technical Report: G10AP00056 - Lithium Brine Resources: A Predictive Exploration Model. Research supported by the U.S. Geological Survey (USGS), Department of the Interior, under USGS award number [G10AP00056].

Müller, D.B., Wang, T., Duval, B., Graedel, T.E., 2006. Exploring the engine of anthropogenic iron cycles. Proceedings of the National Academy of Science 103, 16111-16116, doi: 10.1073/pnas.0603375103

Müller, E., Hilty, L. M, Widmer, R., Schluen, M., Faulstich, M., 2014, Modeling stocks and flows: a review of dynamic material flow analysis methods Environmental Science & Technology 48, 2102-2113

Nakamura, S., K. Nakajima, Y. Kondo, and T. Nagasaka. 2007. The waste input-output approach to materials flow analysis: Concepts and application to base metals. Journal of Industrial Ecology 11: 50–63.

Nansai, K. Nakajima, K., Kagawa, S., Kondo, Y., Suh, S., Shigetomi, Y., Oshita, Y. 2014. Global Flows of Critical Metals Necessary for Low-Carbon Technologies: The Case of Neodymium, Cobalt, and Platinum. Environmetal Science & Technology 48:1391-1400

Narayan, R. 2015. Reducing copper theft in telecommunications and electrical industry. http://www.iceweb.com.au/Electrical-Web/Earthing/Reducing%20Copper%20Theft%20.pdf

Nashawi, I.S., Malallah, A., Al-Bisharah, M., 2010. Forecasting world crude oil production using multicyclic Hubbert model. Energy Fuels 24, 1788-1800

Nassar, N.T., Barr, R., Browning, M., Diao, Z., Friedlander, E., Harper, E.M., Henly, C., Kavlak, G., Kwatra, S., Jun, C., Warren, S., Yang, M-Y., Graedel, T.E, 2012 Criticality of the geological copper family. Environmental Science and Technology, 46:1071−1078.

Nassar, N.T., Graedel, T.E., Harper, E.M., 2015. By-product metals are technologically essential but have problematic supply. Sci. Adv. 2015, 1: doi: 10.1126/sciadv.1400180

National materials advisory board 1981. Manganese reserves and resources of the world and their industrial implications. NMAB-374, Washington DC: https://books.google.se/books?id=3CIrAAAAYAAJ&pg=PA7&lpg=PA7&dq=manganese+resources+data&source=bl&ots=Fdm2XC-KaQ&sig=UPUlMbUohvJJ_wzgxG0MPLuftio&hl=en&sa=X&ved=0ahUKEwiP-iKH9hczTAhXFvhQKHXYODTEQ6AEIOTAD#v=onepage&q=manganese%20resources%20data&f=false

Neumeyer, E., 2000. Scarce or abundant? The economics of natural resource availability. Journal of Economic Surveys 14: 307-336.

Newman, H.R., 2011. The mineral industries of Morocco and Western Sahara. USGS 2011 Mineral yearbook. 11 pages. http://minerals.usgs.gov/minerals/pubs/commodity/cobalt/

NGEX Resources Inc. 2011 Copper booklet. 32pp

Nicholson, P, Evans, K., 1998. Evaluating new directions for the Lithium market. JOM 27-29.

Nickless, E., Bloodworth, A., Meinert, L., Giurco, D., Mohr, S., Littleboy, A. 2014. Resourcing Future Generations White Paper: Mineral Resources and Future Supply. International Union of Geological Sciences. 44pp.

Ning, G., Popoc, B.N. 2004. Cycle life modeling of lithium-ion batteries. Journal of the Electrochemical Society, 151:1584-1591

Noddack, I. and Noddack, W. 1931 Die Geochemie des Rheniums. Zeitshrift für Physikalische Chemie, A154, 207-244.

Norgate, T.E. and Rankin, W. J., 2000 The role of metals in sustainable development. Conference on minerals processing and extractive metallurgy, 133-138

Norgate, T. E. and Rankin, W. J., 2000, Life Cycle Assessment of Copper and Nickel Production, Proceedings, Minprex 2000, International Conference on Minerals Processing and Extractive Metallurgy, 133-138.

Norgate, T.E. and Rankin, W. J., 2002 The role of metals in sustainable development, Proceedings, Green Processing 2002, International Conference on the Sustainable Processing of Minerals, 49-55.

Norgate, T.E. and Rankin, W.J., 2002, An Environmental Assessment of Lead and Zinc Production Processes, Proceedings, Green Processing 2002, International Conference on the Sustainable Processing of Minerals, 177-184.

Noril'sk Nickel 2012. Investor presentation 2012. http://www.nornik.ru/assets/files/2012/NN-presentation-_December%281%29.pdf. 28pp

Noril'sk Nickel 2014. Investor presentation 2014. http://www.nornik.ru/assets/files/2014/NN-presentation-_December%281%29.pdf. 72 pp.

Norsk Hydro ASA 2012. Aluminium, environment and society. A corporate brochure from Norway's largest aluminium smelter. Oslo, Norway

Northey, S., Mohr, S., Mudd, G. M., Weng, Z., Giurco, D., 2014. Modelling future copper ore grade decline based on a detailed assessment of copper resources and mining. Resources, Conservation and Recycling 83: 190-201.

Nowak, I.; Ziolek, M. 1999 Niobium Compounds: Preparation, Characterization, and Application in Heterogeneous Catalysis. Chemical Reviews 99: 3603–3624. doi:10.1021/cr9800208

Nuss, P. and Eckelmann, M.J., 2014. Life Cycle Assessment of Metals: A scientific synthesis. PLoS ONE 9: e101298. doi:10.1371/journal.pone.0101298

Nuss, P., Harper, E.M., Nassar, N.T., Reck , B. K., Graedel T.E. 2014. Criticality of Iron and its principal alloying elements. Environ. Sci. Technol., 48:4171-4177, DOI: 10.1021/es405044w

Nuss, P., Eckelmann, M.J., 2014. Life Cycle Assessment of Metals: A Scientific Synthesis. PLoS ONE 9: e101298. doi:10.1371/journal.pone.0101298

Ober, J. A. 1994 Lithium. US Bureau of Mines, Department of the Interior, Washington DC. 7 pages

Ober, J. A. 2002, Lithium. US Bureau of Mines, Department of the Interior, Washington DC. 7 pages

Oberthür, T., Weiser, T.W., Gast, L. 2003a Geochemistry and mineralogy of platinum-group elements at Hartley Platinum Mine, Zimbabwe. Part 1: Primary distribution patterns in pristine ores of the main sulfide zone of the Great Dyke: Mineralium Deposita 38:327-343

Oberthür, T., Weiser, T.W., Gast, L. 2003b Geochemistry and mineralogy of platinum-group elements at Hartley Platinum Mine, Zimbabwe. Part 2: Supergene redistribution in the oxidized main sulfide zone of the Great Dyke and alluvial platinum-group minerals: Mineralium Deposita 38:344-355

Oberthür, T., 2011 Platinum-group elements mineralization of the main sulphide zone, Great Dyke, Zimbabwe. Reviews in Economic Geology 17:329-349

Oberthür, T., Melcher, F., Buchholz, P., Locmelis, M., 2013 The oxidized ores of the Main Sulphide Zone, Great Dyke, Zimbabwe: turning resources into minable reserves–mineralogy is the key. The Journal of The Southern African Institute of Mining and Metallurgy 113:191-211

OECD 2017. List of tin, tantalum and tungsten (3Ts) smelters as of 2011. https://www.oecd.org/daf/inv/mne/3TsSmelterList.pdf

Oijens, S. 2014. Update of sand and gravel resources and extraction worldwide. IHC, Merwede. 28 pages. https://www.ciria.org/CMDownload.aspx?ContentKey=a9eebdd5-025a-4b12...

Olbrich, A., Meese-Marktscheffel, J., Jahn, M., Zertani, R., Stoller, V., Erb, M., Heine, K.H., and Kutzler, U., 2009, Recycling of superalloys with the aid of an alkali metal salt bath, US Patent Application No. 0255372 A1, Washington, D.C., U.S. Patent and Trademark Office.

Opiekun, Z.A., 2012 Mechanical Properties of the Thermal Barrier Coatings Made on Cobalt Alloy MAR-M509 In; Superalloys. Chapter 14: 321-336. InTech. http://dx.doi.org/10.5772/61100

Osborn, F.L. 1948. Our Plundered Planet. Pyramid Publications, New York, USA.

OSPAR, 2003. Agreement on Sand and Gravel Extraction, OSPAR Agreement 2003-15. http://www.ospar.org/documents/dbase/decrecs/agreements/03- 05e_Reporting%20format%20Chlor%20alkali.doc

Ostrom, E., Walker, J., Gardner, R. 1994. Rules, Games, and Common-Pool Resources. Ann Arbor: University of Michigan Press.

Outotec 2015. Outotec lithium production technologies. 4pages, www.outotec.com.

Owen, N.A., Inderwildi, O.R., King, D.A., 2010. The status of conventional world oil reserves – Hype or cause for concern.?. Energy policy 38: 4743-4749.

Padture, N. P., Gell, M., Jordan, E. H. 2002, Thermal Barrier Coatings for Gas-Turbine Engine Applications. Science 296:280–284

Patzek, T.W., Croft, G., 2010. A global coal production forecast with multi-Hubbert cycle analysis. Energy 35, 3109-3122

Papp, J. F. 1994. Chromium life cycle study. II Series. Information circular. United States Buerau of Mines. 9411. HD9539.CAP 1994 338.2'74643-dc20 94-29994 CIP. 100pp.

Papp, J.F., 2004, Chromium use by market in the United States: Proceedings of the Tenth International Ferro Alloys Congress, 1-4 February, Cape Town, South Africa. http://minerals.usgs.gov/minerals

Papp, J.F., 2006, Chromite, in Industrial Minerals and Rocks—Commodities, Markets, and Uses, 7th Edition: Society for Mining, Metallurgy, and Exploration, Inc., Littleton, CO, p. 309-333.

Papp, J F., 2007, Chromium – A national mineral commodity perspective: Reston, VA, U.S. Geological Survey Open-File Report 2007-1167, 49 p.

Papp, J.F., Bray, E.L., Edelstein, D.L., Fenton, M.D., Guberman, D.E., Hedrick, J.B., Jorgenson, J.D., Kuck, P.H., Shedd, K.B., and Tolcin, A.C., 2008, Factors that influence the price of Al, Cd, Co, Cu, Fe, Ni, Pb, Rare Earth Elements, and Zn: U.S. Geological Survey Open-File Report 2008-1356, 61 p. http://pubs.usgs.gov/of/2008/1356/pdf/ofr2008-1356.pdf

Papp, J.F. 2010b: Tantalum. In: Mineral commodity summaries 2010. U.S. Geological Survey, pp. 162-163.

Papp, J.F. 2013: Niobium (columbium) and tantalum. In: U. S. Geological Survey minerals yearbook - 2011. Revised 2013. U. S. Geological Survey. pp. 52.1-52.14.

Papp, J., 2014 Recycling metals, Advanced release. Pages 61.1-61. USGS 2013 Minerals Yearbook. Washington DC.

Pariser, H. H, 2013. Chromite: World distribution, uses, supply & demand, future. 2013 PDAC Convention, Toronto, March 2013 "Ontario's Ring of Fire: Unlocking Potential & Creating Opportunity" 23 pp.

Pauliuk, S., Wang, T., Müller, D.B., 2013. Steel all over the world: Estimating in-use stocks of iron from 200 countries. Resources, Conservation and recycling 71: 22-30.

Pauliuk, S., Milford, R.L., Müller, D.B., Allwood, J. M. 2013. The steel scrap age. Environmental Science and Technology, 47, 3448-3454.

Pauliuk, S., Wang, T., Müller, D., 2012. Moving towards the circular society, the role of stocks in the Chinese steel cycle. Environmental Science & Technology 46:148-151

Pauliuk, S., Wood, R., Hertwich, E.G. 2015. Dynamic models of fixed capital stocks and their application in industrial ecology. Journal of Industrial Ecology 19:104-116

Peduzzi, P., 2014. Sand, rarer than one thinks. UNEP Global Environmental Alert Service (GEAS). 15 pp. www.unep.org/geas.

Phillips, W.G.B. and Edwards, D.P. 1976. Metal prices as a function of ore grade. Resources Policy 2:167—179.

Plachy, J, 2009, Cadmium recycling in the United States in 2000, chap. O of Sibley, S.R., Flow studies for recycling metal commodities in the United States: U.S. Geological Survey Circular 1196–T, p.T1–T7, available only at http://pubs.usgs.gov/circ/circ1196t/. 14pp

Platinum Group Metals. 2015 Corporate for Platinum Group Metals Ltd. website for industry news. http://www.platinumgroupmetals.net/

Plotinska, O.Y. Grabezhev, A.I., Seltmann, R., 2014. Porphyry Deposits of the South Urals: Rhenium Distribution. Acta Geologica Sinica 88: 584-586

Polinares 2012a. Fact sheet: Platinum Group Metals. Polinares working paper no. 35. http://www.polinares.eu/docs/d2-1/polinares_wp2_annex2_factsheet1_v1_10.pdf

Polinares 2012b. Tercero Espinosa, L.A. The contribution of recycling to the supply of metals and minerals. Polinares: Working paper no. 20. http://www.polinares.eu/docs/d2-1/polinares_wp2_chapter8.pdf

Polinares 2012. Factsheet: Lithium. Polinares working paper n. 38. EU policy on natural Resources. 16 pages.

Pohl, W.L., 2011 Economic Geology, Principles and Practice: Metals, Minerals, Coal and Hydrocarbons— an Introduction to Formation and Sustainable Exploitation of Mineral Deposits. 663 Pages, Wiley-Blackwell.

Polinares 2012a. Fact sheet; Copper. Polinares working paper n. 40, March 2012. Polinares EU policy on natural resources. 18pp. www.polinares.eu. 2012

Polinares 2012b. Fact sheet; Indium. Polinares working paper n. 39, March 2012. Polinares EU policy on natural resources. 17pp. www.polinares.eu. 2012

Polinares 2012. POLINARES working paper n. 41 March 2012 Fact Sheet: Iron Ore http://www.polinares.eu/

Polyak, D.E., 2011, USGS Mineral Commodity Summary: Rhenium, Reston, VA: U.S. Geological Survey.

Prendergast, M.D., 1988, The geology and economic potential of the PGE-rich Main Sulphide Zone of the Great Dyke, Zimbabwe, in Prichard, H.M., Potts, P.J., Bowles, J.F.W., and Cribb, S.J., eds., Geo-platinum '87: Barking, Essex, Elsevier Applied Science, 281–302.

Prendergast, M.D., and Wilson, A.H., 1989, The Great Dyke of Zimbabwe II—Mineralization and mineral deposits, in Prendergast, M.D., and Jones, M.J., eds., Magmatic sulphides—The Zimbabwe volume: London, The Institution of Mining and Metallurgy, 21–42.

Prior, T., Daly, J., Mason, L., Giurco, D., 2013. Resourcing the future: Using foresight in resource governance. Geoforum, 44:316-328

Prior, T., Giurco, D., Muss, G., Mason, L., Behrish, J., 2012. Resource depletion, peak minerals and the implication for sustainable resource management. Global Environmental Change. 22:577-587.

Radetzki. M. A. 2008 Handbook of Primary Commodities in the Global Economy. Cambridge: Cambridge University Press. 233 pages, 2008

Radetzki, M. 2012 The perseverance of the ongoing metal and mineral boom. Mineral Economics, Raw Materials Report. ISSN 2191-2203. Miner Econ. DOI 10.1007/s13563-012-0020-2

Ragnarsdottir, K.V., Sverdrup, H. 2001. Challenging the planetary boundaries II: Basic principles of an integrated for phosphorus supply dynamics and global population size Applied Geochemistry 2011; 26: S307-S310

Ragnarsdottir, K.V. 2008 Rare metals getting rarer. Nature Geoscience 1:720-721

Ragnarsdottir, K.V., Sverdrup, H, Koca, D. 2012 Assessing long-term sustainability of global supply of natural resources and materials. Chapter 5; 83-116. In; Sustainable Development–116. Energy, Engineering and Technologies–Manufacturing and Environment. Ghenai, C. (Ed.). www.intechweb.org

Ramanaidou, E.R. and Wells, M.A. 2014. Sedimentary Hosted Iron Ores. In: Holland, H.D. and Turekian, K.K. (Eds.), Treatise on Geochemistry. Oxford: Elsevier. 313-355. http://dx.doi.org/10.1016/B978-0-08-095975-7.01115-3

Ramberg, I.B., 2008. The making of a land: The geology of Norway. Geological Society. ISBN 978-82-92394-42-7.

Ramkumar, S. 2014, Stock-driven, trade-linked, multi-regional model of the global aluminium cycle. Masters thesis in Industrial Ecology, Norwegian University of Science and Technology. Trondhjem, Norway. 74pp.

Rasilainen, K., Eilu, P., Halkoaho, T., Iljina, M., Karinen, T., 2010a. Quantitative mineral resource assessment of undiscovered PGE resources in Finland. Ore Geology Reviews 38:270-287

Rasilainen, Kalevi; Eilu, Pasi; Halkoaho, Tapio; Iljina, Markku & Karinen, Tuomo 2010b. Quantitative mineral resource assessment of platinum, palladium, gold, nickel, and copper in undiscovered PGE deposits in mafic-ultramafic layered intrusions in Finland. Geological Survey of Finland, Report of Investigation 180. 338pp.

Rasilainen, K., Eilu, P., Äikäs, O., Halkoaho, T., Heino, T., Iljina, M., Juopperi, H., Kontinen, A., Kärkkäinen, N., Makkonen, H., Manninen, T., Pietikäinen, K., Räsänen, J., Tiainen, M., Tontti, M. & Törmänen, T. 2012. Quantitative mineral resource assessment of nickel, copper and cobalt in undiscovered Ni-Cu deposits in Finland. Geologian tutkimuskeskus, Tutkimusraportti 194 − Geological Survey of Finland, Report of Investigation 194, 514 pages

Rauch, J.N. 2009. Global mapping of Al, Cu, Fe, and Zn in-use stocks and in-ground resources. Proceedings of the National Academy of Sciences, 1-6. Downloaded in 2014 from: www.pnas.org/cgi/doi/10.1073/pnas.0900658106

Rauch, J.N., Pacyna J.M., 2009. Earth's global Ag, Al, Cr, Cu, Fe, Ni, Pb, and Zn cycles, Global Biogeochemical Cycles, 23, GB2001 doi:10.1029/2008GB003376.

Rauch, J.N., and Graedel T.E., 2007 Earth's anthrobiogeochemical copper cycle, Global Biogeochem.Cycles, 21, GB2010. doi:10.1029/2006GB002850.

Ravishankar, S. 2015. Illegal beach sand mining of minerals in Tamil Nadu may be a scam worth Rs 1 lakh crore. Metals and mining. The economic times. http://articles.economictimes.indiatimes.com/2015-02-01/news/58675766_1_beach-sand-periyasamypu-ram-tuticorin-ashish-kumar

Rayzman, V. L., Pevzner, I. Z., Sizyakov, V. M., Ni, L.P., Igor K. Filipovich, I.K., Andrei V. Aturin, A. V. 2010. Extracting Silica and Alumina from Low-Grade Bauxite. Journal of the Materials, Metals and Materials Society; JOM:2010:47-51

Reck, B., Graedel, T.E., 2012 Challenges in Metal Recycling. Science 337:690, pp. 7.

Reed, R. C. The Superalloys: Fundamentals and Applications. Cambridge, UK: Cambridge UP, 2006

Reynolds, D. B. 1999, The Mineral Economy: How Prices and Costs Can Falsely Signal Decreasing Scarcity. Ecol. Econ. 31, 155-166.

Richa, K., Babbitt, C.W., Gaustad, G., Wang, X., 2014. A future perspective on lithium-ion battery waste flows from electric vehicles. Resources, Conservation and Recycling 83:63– 76

Risk & Policy Analysts Limited 2012. Stockpiling of Non-energy Raw Materials, Final Report March 2012. Directorate-General Enterprise and Industry, EU, Brussels, Farthing Green House, 1 Beccles Road, Loddon, Norfolk, NR14 6LT, UK. 327pp

Roberts, N., Andersen, D.F., Deal, R.M., Shaffer, W.A., 1982, Introduction to Computer Simulation: A System Dynamics Approach Productivity Press, Chicago

Roberts, S. 2012. Deep-Sea and Sub-Seafloor Resources: A Polymetallic Sulphide and Co-Mn Crust Perspective. The LRET Research Collegium Southampton, 16 July – 7 September 2012. 36 pages

Roberts, P.C; 2014: The Failure of Laissez Faire Capitalism and Economic Dissolution of the West Clarity Press, 192pp, ISBN: 978-0-9860362-5-5

Robinson, R., Brown, M., 2002. Sociocultural Dimensions of Supply and Demand for Natural Aggregate—Examples from the Mid-Atlantic Region, United States, U.S. Geological Survey Open-File Report 02-350.

Robinson, G. R., Jr., and Menzie, W. D. 2012, revised 2013. Economic filters for evaluating porphyry copper deposit resource assessments using grade-tonnage deposit models, with examples from the U.S. Geological Survey global mineral resource assessment: U.S. Geological Survey Scientific Investigations Report 2010–5090–H, v. 1.1, 21 p. and data tables, available at http://pubs.usgs.gov/sir/2010/5090/h/.

Rodriguez, J.A.E., Contreras, J.L.L.S ., 2011. An assessment of lithium resources. In: Lithium, Technology, performance and safety. Tabaris, F., (Ed). 1-34.. San Francisco, Nova Science Publishers Inc.

Roland Berger 2015. Trend compendium 2030. Scarcity of resources. September 2015. 145 pages. http://www.roland-berger.com/expertise/trend_compendium_2030/

Rolls Royce Trent Turbofan Engine 2017; https://s-media-cache-ak0.pinimg.com/originals/fe/ed/5c/feed5c954e18581cc905e69769c571f6.jpg

http://airway.uol.com.br/wp-content/uploads/2010/01/trent_xwb_960_720.jpg

https://en.wikipedia.org/wiki/Rolls-Royce_Trent.

Roper, L.D. 2009. World mineral reserves. Website with Hubbert's type variant of resource assessment:. http://www.roperld.com/science/minerals/Reserves.htm

Rosenau-Tornow, D., Buchholz, P., Riemann, A., Wagner, M. (2009): Assessing long-term supply risks for mineral raw materials – a combined evalutation of past and future trends. Resources Policy 34:161-175.

Rosenqvist, T. 1983, Principles of Extractive Metallurgy, McGraw-Hill Book Company

Roskill 2010. The Economics of Rhenium (Roskill Reports on Metals and Minerals). 150 pages.

Roskill 2015. Rhenium. Market outlook to 2020, 10th edition. 174 pages. ISBN: 978 0 86214 620 7. http://roskill.com/product/rhenium-market-outlook-to-2020-10th-edition-2015/

Rostoker, W. Bronson, B. Dvorak, J. 1984, The Cast-Iron Bells of China, Technology and Culture, The Society for the History of Technology. 25:750-767, doi:10.2307/3104621, JSTOR 3104621

Rostoker, W. Bronson, B. 1990, Pre-Industrial Iron: Its Technology and Ethnology, Archeomaterials Monograph No. 1

Royal Bafokeng Platinum 2013. RBPLAT integrated annual report: Mineral resources & mineral reserves 26pages. http://www.bafokengplatinum.co.za/im/files/reports/2013/RBPlat-AR-2013-reserves-resources.pdf

Russian Platinum 2014. Reserves. http://www.russian-platinum.ru/about/company?setlang=2

Quiring, H., 1962, Platinmetalle: Platin, Palladium, Iridium, Osmium, Rhodium, Ruthenium. F. Friedensburg, Ed., Die Metallischen Rohstoffe. Ihre Lagerungsverhältnisse und ihre wirtschaftliche Bedeutung (Ferdinand Enke Verlag Stuttgart), vol. 16.

Safirova, E. 2013 The mineral industry of Russia. USGS 2011 Minerals yearbook. 22pp Washington DC.

Saurat, M. and Bringezu, S, 2009a Platinum Group Metal flows of Europe, Part I. Flows. Journal of Industrial Ecology, 13:406-421.

Saurat, M., Bringezu, S., 2009b Platinum Group Metal flows of Europe, Part II: Exploring the technological and institutional potential for reducing environmental impacts. Journal of Industrial Ecology 13:406–421

Sato, A., Harada, H., Kawagishi, K., Kobayashi, T., Koizumi, Y., Yokokawa, T., Zhang, J-X., 2008. A 5th generation superalloy with balanced high temperature properties and processability. In Superalloys 208, Reed, R, Green, K.A., Caron, P., Gabb, T.P., Fahrmann, M.G., Huron, E.S., Woodard, S.A., (Eds). The Minerals, Metals & Materials Society. 131-139

Scheele, C.W., 1774 Om brunsten, eller magnesia, och dess egenskaper. Kongliga Vetenskaps Academiens Handlingar, 35 : 89-116.

Scheele, C.W. 1778 Försök med Blyerts, Molybdæna Kongliga Vetenskaps Academiens Handlingar, 39:247-255.

Scheele, C.W., 1781 Tungstenens bestånds-delar, Kongliga Vetenskaps Academiens Nya Handlingar 2:89-95.

Schneibel, J.H., 2005. Beyond Nickel-Base Superalloys. ORNL Institute brochure, 12pp. http://aprs.ornl.gov/~webworks/cppr/y2001/pres/122106.pdf

Schmidt, C.W, 2015. Going deep - Cautious Steps toward Seabed Mining. Focus. Environmental Health Perspectives. 123:A235-243

Schneibel, J.H., 2005. Beyond Nickel-Base Superalloys. ORNL Institute brochure, 12pp. http://aprs.ornl.gov/~webworks/cppr/y2001/pres/122106.pdf

Schodde, R., 2010, The key drivers behind resource growth: An analysis of the copper industry over the last 100 years: 2010 MEMS Conference Min- eral and Metal Markets over the Long Term: Phoenix, Arizona, USA, 26 p.

Schooldermann, H., Martlehner, R. 2011. Minerals and scarcity in manufacturing: the ticking time bomb. PwC report. www.pwc.com/resourcescarcity

Schubert, W-D., Lassner, E., 2010. ITIA Newsletter: 6. Cemented carbides, a success story. 12pp. International Tungsten Industry Association, 4 Heathfield Terrace, London W4 4JE, UK. www.itia.info

Schwela, U. 2010: The state of tantalum mining. In: Mining Journal special publication - Tantalum, 2-5.

Scott, S.D., 2001. Deep ocean mining. Geoscience Canada 28: 87-96.

Scrosati, B., Garche, J., 2010. Lithium batteries, Status, prospects and future. Journal of Power Sources. 195, 2419-2430.

Scrosati, B, Hassoun, J., Sun, Y-K., 2011. Lithium-ion batteries. A look into the future. Energy Environ. Sci., 2011,4, 3287-3295

Searle, P.J., 2015, Cobalt—Market balance in 2020s?: The Cobalt Conference, Cobalt Development Institute, Toronto, Ontario, Canada, May 20–21, presentation, 21 p.

Seddon, M., 2013. World Tungsten Report 2013. 12pages, Metal Pages Ltd. mark@tungstenresearch.com, https://www.argusmedia.com/~/media/Files/PDFs/Samples/WorldTungsten_201311.pdf/?la=en

Senge P. The Fifth Discipline. The Art and Practice of the Learning Organisation. Century Business, New York. 1990

Senge P.M., Smith B., Schley S., Laur J., Kruschwitz N. 2008. The Necessary Revolution: How Individuals and Organisations Are Working Together to Create a Sustainable World. Doubleday Currency, London,

Seppelt, R., Manceur, A.M., Liu, J., Fenichel, E.P., Klotz, S., Synchronized peak-rate years of global resources use, Ecology and Society 19: 50-64. 14pages http://dx.doi.org/10.5751/ES-07039-190450

SGS 2010. Hard rock lithium processing. 3 pages. http://www.sgs.com/~/media/ Global/Documents/Flyers%20and%20Leaflets/SGS-MIN-WA109-Hard-Rock-Lithium-Processing-EN-11.pdf

Shaw, R., Goodenough, K., Gunn, G, Brown, T., Rayner, D., 2011. Niobium-Tantalum. In: Minerals UK. 27 pages, British Geological Survey, Nottingham, Great Britain

Shedd, K.B., 2002. Cobalt. USGS 2011 Mineral yearbook. 18 pages. http://minerals.usgs.gov/minerals/pubs/commodity/cobalt/

Shedd, K.B., 2003. Tungsten. U.S. Geological Survey Minerals Yearbook—2003; 79.1-79.18

Shedd, K.B., 2012. Cobalt. Advanced release, USGS 2011 Mineral yearbook. 22 pages. http://minerals.usgs.gov/minerals/pubs/commodity/cobalt/

Shedd, K.B., 2015. U.S. and Global Cobalt Statistics and Information from the USGS National Minerals Information Center. The Cobalt Conference 2015 Toronto, Ontario, Canada 20–21 May 2015. http://minerals.usgs.gov/minerals/pubs/commodity/cobalt/. Power Point presentation, 17pages.

Shiyu, Y. 1991. Classification and type association of tin deposits in Southeast Yunnan Tin Belt. Chinese Journal of Geochemistry. 10: 21–35. doi:10.1007/BF02843295.

Siame, E., 2011. Recovery of lithium from china clay waste using a combination of froth flotation, magnetic separation, roasting and leaching. Submitted by Edward Siame to the University of Exeter as a thesis for the degree of Doctor of Philosophy in Earth Resources

Sims, C.T., Stoloff, N.S., Hagel, W.C., 2008. Superalloys II; High temperature materials for aerospace and industrial power. Wiley Interscience, New York. 680pp.

Singer, D. A. 1993. Basic concepts in three-part quantitative assessments of undiscovered mineral resources: Nonrenewable Resources, 2:69–81.

Singer, D.A. 1995. World class base and precious metal deposits—A quantitative analysis: Economic Geology, 90:88–104.

Singer, D.A. 2007 Short course introduction to quantitative mineral resource assessments: U.S. Geological Survey Open-File Report 2007–1434 [http://pubs.usgs.gov/of/2007/1434/].

Singer, D.A., Menzie, W.D., 2010. Quantitative Mineral Resource Assessments—An Integrated Approach. Oxford University Press, New York 219.

Singer, D.A., 2011 A log-normal distribution of metal resources. J. China Univ. Geosci. 36, 1–8.

Singer, D. 2013 The lognormal distribution of metal resources in mineral deposits. Ore Geology Reviews 55:80-86.

Slack, J.F., Causey, J.D., Eppinger, R.G., Gray, J.E., Johnson, C.A., Lund, K.I., and Schulz, K.J., 2010, Co-Cu-Au deposits in metasedimentary rocks—A preliminary report: U.S. Geological Survey Open-File Report 2010–1212, 13 p. (Available at http://pubs.usgs.gov/of/2010/1212/.)

Smirnov, V.S. 1996.Alumina production in Russia, part 1, Historical background. JOM 48:24-26. http://www.tms.org/pubs/journals/JOM/9608/Smirnov-9608.html

Smirnov, V.S., and N. N. Tikhonov. 1991. Main Trends of Alumina Industry Development in USSR. Paper presented at the international conference Scientific and Technical Progress in Non Ferrous Metallurgy, St. Petersburg, 17 September, 1991.

Smirnov, V.S. 1992. Raw Materials of Russian Aluminum Industry—New Investment Projects. Paper presented at the international conference Russian Aluminum Industry—Strategy of Development and New Investment Projects, St. Petersburg, 15-18 September, 1992.

Smith, A., 1759, The theory of moral sentiments, Reprint by Promethus Books, Amherst New York 2000

Smith, A., 1776. An Inquiry into the Nature and Causes of the Wealth of Nations, Reprint by the University Of Chicago Press, Chicago

Smith, V.L. 2009; Economics of Production from Natural Resources. Energies, 2:661

Smith, S. and Heydon, R., 2013. 4.0 Processes related to the technical development of marine mining. In: SPC 2013. Deep Sea Minerals: Cobalt-rich ferromanganese crusts,A physical, biological, environmental, and technical review. Baker, E. and Beaudoin, Y. (Eds.) Vol. 1C, Secretariat of the Pacific. 41-49

Sonber, J.K., Murthy, T.S.R.Ch., Subaramanian, C., Kumar, S. Fotedar, R.K., Suri, A.K. 2010 Investigations on synthesis of HfB2 and development of a new composite with TiSi2. International Journal of Refractory Metals and Hard Materials 28:201–210. doi:10.1016/j.ijrmhm.2009.09.005

Sonoc, A, Jeswiet, J., 2014. A review of lithium supply and demand and a preliminary investigation of a room temperature method to recycle lithium ion batteries to recover lithium and other materials. 21st CIRP Conference on Life Cycle Engineering, Procedia CIRP 15:289 – 293

Spatari, S., Bertram, M., Gordon, R.B, Henderson, K., Graedel, T.E., 2005. Twentieth century copper stocks and flows in North America: A dynamic analysis. Ecological Economics 54: 37-51.

SPC 2012 Pacific ACP States Regional Legislative and Regulatory Framework for Deep Sea Minerals Exploration and Exploitation. http://www. smenet.org/docs/public/FinalDeepSeaMineralsProjectReport.pdf

Spencer E., and Searle, P., 2015, Cobalt market outlook. CRU. http://www.crugroup.com/market-analysis/products/cobaltmarketoutlook?TabId=130087&Info=Cobalt+Market+Outlook&ReturnUrl=null

Speirs, J., Contestabile, M., Houari, Y., Gross, R. 2014. The future of lithium availability for electric vehicle batteries RenewableandSustainableEnergyReviews 35:183–193

Stanway, D. 2014. China steel output near peak, say executives, in bad news for miners. http://www.reuters.com/article/2014/03/06/china-parliament-steel-idUSL3N0LT13W20140306.

Stamp, A., Lang, D.J., Wäger, P.A., 2012. Environmental impacts of a transition toward e-mobility: the present and future role of lithium carbonate production. Journal of Cleaner Production 23: 104-112

Starkey, H.C, 1982. The role of clays in fixing lithium. A study of some methods by which lithium is retained in clay minerals. Geological Survey Bulletin 1278-F. United States Government Printing Office, Washington.

Steffen, W., Broadgate, W., Deutsch, L., Gaffney, O., Ludwig, C. 2015. The trajectories of the Anthropocene: The Great Acceleration. The Anthropocene Review, 1-18. DOI: 10.1177/2053019614564785

Sterman, J. D., 2000. Business Dynamics, System Thinking and Modelling for a Complex World. Irwin McGraw-Hill: New York

Stillwater Palladium 2015. Corporate website. http://www.stillwaterpalladium.com/refining.html

Stillwater Mining Company 2012. Palladium Fundamentals dominates catalytic converters –Confirming positive outlook. 5 pages. Pdf from website. http://www.corporatereport.com/stillwater/Palladiam_Fundamentals_9-26-2012.pdf

Stillwater Mining Company 2011. The case for Palladium, 2011 Update. http://www.stillwatermining.com/market/the-case-for-palladium-2011-update.html

Stillwater Mining Company 2010. The case for Palladium. Accessed at the Stillwater website: http://www.stillwatermining.com/market/case-for-palladium-2010.html

Stockwell, L.E. 1999 World Mineral Statistics 1993-97: Production, Exports, Imports, Keyworth, UK.

Sutphin, D.M., and Page, N.J, 1986, International strategic minerals inventory summary report—Platinum-group metals: U.S. Geological Survey Circular 930-E, 34 (http://pubs.er.usgs.gov/publication/ cir930E?currow=87.)

Sverdrup, H., Svensson, M. 2002 Defining sustainability. In: Developing principles for sustainable forestry, Results from a research program in southern Sweden. Sverdrup, H. and Stjernquist, I. (Eds.) Managing Forest Ecosystems 5:21-32 Kluwer Academic Publishers, Amsterdam.

Sverdrup, H., Svensson, M. 2004. Defining the concept of sustainability, a matter of systems analysis. In: M. Olsson; G. Sjöstedt (Eds.); Revealing Complex Structures - Challenges for Swedish Systems Analysis, 122-142. Kluwer Academic Publishers.

Sverdrup, H. 2011. Peak minerals, peak energy, peak food, peak wealth, moving towards the end of a golden age. The Balaton Bulletin 1:23-24

Sverdrup, H, Ragnarsdottir, KV, Koca, D. 2011. Challenging the planetary boundaries I: Assessing the sustainable global population and phosphate supply, using a systems dynamics assessment model. Applied Geochemistry 2011; 26: S311-313, doi:10.1016/j.apgeochem.2011.03.088

Sverdrup, H., Koca, D., Granath C., 2012a Modeling the gold market, explaining the past and assessing the physical and economical sustainability of future scenarios. In; Schwanninger, M., Husemann, E., and Lane, D., Proceedings of the 30th International Conference of the System Dynamics Society, St. Gallen, Switzerland, July 22–26, 2012. Model-based Management 5:4002-4023. University of St. Gallen, Switzerland; Systems Dynamics Society. ISBN: 9781622764143. Curran Associates, Inc.

Sverdrup, H., Koca, D., Ragnarsdottir, K.V. 2012b The World 5 model; Peak metals, minerals, energy, wealth, food and population; urgent policy considerations for a sustainable society. In; Schwanninger, M., Husemann, E., and Lane, D., Proceedings of the 30th International Conference of the System Dynamics Society, St. Gallen, Switzerland, July 22–26, 2012. Model-based Management 5:3975-4001, ISBN: 9781622764143 Curran Associates, Inc.

Sverdrup, H., Koca, D., Ragnarsdottir, K.V. 2013 Peak metals, minerals, energy, wealth, food and population; urgent policy considerations for a sustainable society. Journal of Environmental Science and Engineering. 2013; 499-534. ISSN 2159-581X

Sverdrup, H., Koca, D., Ragnarsdottir, K.V. 2014a Investigating the sustainability of the global silver supply, reserves, stocks in society and market price using different approaches. Resources, Conservation and Recycling. 83;121-140.

Sverdrup, H., Ragnarsdottir, K.V., Koca, D., 2014b. On modelling the global copper mining rates, market supply, copper price and the end of copper reserves. Resources, Conservation and Recycling 87:158-174

Sverdrup, H.U., and Ragnarsdottir, K.V., 2014c. Natural Resources in a planetary perspective. Geochemical Perspectives Vol. 2, number 2, October issue 2014. 2:1-156. European Geochemical Society.

Sverdrup, H., Koca, D., Ragnarsdottir, K.V. 2015a Aluminium for the future: Modelling the global production, assessing long term supply to society and extraction of the global bauxite reserves. Resources, Conservation and Recycling. 103:139-154

Sverdrup, H., Ragnarsdottir, K.V., Koca, D., 2015b. An assessment of metal supply sustainability as an input to policy: Security of supply extraction rates, stocks-in-use, recycling, and risk of scarcity. Journal of Cleaner Production; http://dx.doi.org/10.1016/j.jclepro.2015.06.085

Sverdrup, H., Koca, D., Ragnarsdottir K.V. 2015c. Defining a free market: Drivers of unsustainability as illustrated with an example of shrimp farming in the mangrove forest in South East Asia. Journal of Cleaner Production, DOI: 10.1016/j.jclepro.2015.06.087

Sverdrup, H., Ragnarsdottir, K.V., Koca, D., 2016a. Modelling the global copper, zinc and lead mining rates and co-extraction of technology metals, supply, price and extractable amounts using the BRONZE model. 38 pp. Submitted to journal.

Sverdrup, H., Ragnarsdottir, K.V., 2016b. The future of platinum group metal supply; An integrated dynamic modelling for platinum group metal supply, reserves, stocks-in-use, market price and sustainability. Resources, Conservation and Recycling

Sverdrup, H., Koca, D., Ragnarsdottir, K.V., 2016c Modelling the long term global iron mining, market supply, stocks-in-use in society and reserves, using the IRON model. Journal of Natural Resources.

Sverdrup, H., Ragnarsdottir, K.V., 2016d. Modelling global extraction, supply, price and depletion of the extractable geological resources with the LITHIUM model. Resources, Conservation and Recycling

Sverdrup, H., Ragnarsdottir, K.V., 2017. The future of platinum group metal supply; An integrated dynamic modelling for platinum group metal supply, reserves, stocks-in-use, market price and sustainability. Resources, Conservation and Recycling 114: 130-152.

Sverdrup, H., 2017. Modelling global extraction, supply, price and depletion of the extractable geological resources with the LITHIUM model. Resources, Conservation and Recycling 114: 112-129

Swanson, S. 2012. Mineralogy of spodumene pegmatites and related rocks in the tin–spodumene belt of North Carolina and South Carolina, USA. The Canadian Mineralogist 50:1589-1608

Sweeneye, J.L., 1992. Economic theory of depletable resources: An introduction. In: Handbook of Natural resource and energy economics: Vol 3, Chapter 17, 759-854.

Tahil, W. 2007 The trouble with lithium: implications of future PHEV production for lithium demand. Meridian International Research; 22 pages. Available from: http://www.evworld.com/ library/lithium_shortage.pdf

Tahil, W. 2008. The Trouble with Lithium 2. Under the Microscope. Meridian International Research, May 29, 2008.

Tahil, W., 2010. How much lithium does a Li-ion EV battery really need? Meridian International Research www.meridian-int-res.com 11 pages.

Takada, K., 2013. Progress and prospective of solid-state lithium batteries. Acta Materialia 61 (2013) 759–770

Takada, K., Inada, I., Kajiyama, A., Kouguchi, M., Sasaki, H., Kondo, S., Michiue, Y., Nakano, S., Tabuchi, M., Watanabe, M. 2004. Solid state batteries with sulfide-based solid electrolytes. Solid State Ionics 172:25–30

Tanaka, R, Kasama, A., Fujikura, M., Iwanaga, I., Tanaka, H., Matsumura. Y., 2003. Research and Development of Niobium-Based Superalloys for Hot Components of Gas turbines. Proceedings of the International Gas Turbine Congress 2003 Tokyo, November 2-7, 2003, 1-5.

Tan, H., Chen, J., Rao, W., Zhang, W., Zhou, H., 2012. Geothermal constraints on enrichment of boron and lithium in salt lakes: An example from a river-salt lake system on the northern slope of the eastern Kunlun Mountains, China. Journal of Asian Earth Sciences 51: 21–29

Tarascon, J.M., Armand, M. 2001. Issues and challenges facing rechargeable lithium batteries. Nature 414: 359-368

Tatsumisago, M., Nagao, M., Hayashi, A., 2013. Recent development of sulfide solid electrolytes and interfacial modification for all-solid-state rechargeable lithium batteries. Journal of Asian Ceramic Societies 1:17–25

The DME. 2003. Platinum-group metal mines in South Africa. Issued by the Minerals and energy department, 30pp, ISBN: 0-9584257-6-0

Thomas, C.E. 2009. Will there be enough platinum? 4 pages. WEB document. http://www.cleancaroptions.com/Will_there_be_enough_platinum.pdf

TIAX LLC 2003. Platinum availability and economics for PEMFC commercialization. https://www1.eere.energy.gov/hydrogenandfuelcells/pdfs/tiax_platinum_appendix.pdf. DOE Report number DE-FC04-01-AL67601

Tien, J.K., Collier, J.P., Vignoul, G., 1989 The role of niobium and other refractory elements in superalloys. In: Superalloy 71 S-Metallurgy and Applications Edited by E.A. Loria The Minerals, Metals & Materials Society, pages 553-567. 14pages facimile. http://www.tms.org/superalloys/10.7449/1989/Superalloys_1989_553_566.pdf

Tilton J.E. 1983, The impact of seabed nodules mining: a qualitative analysis. Research report. Laxenburg, Austria: International Institute for Applied System Analysis (IIASA). 77 p. Available at: /www.iiasa.ac.at/Admin/PUB/ Documents/RR-83-033.pdfS

Tilton, J.E. 2002. On borrowed time? Assessing the threat of mineral depletion. JHup Books. www.jhupbooks.com

Tilton, J.E., and Lagos, G., 2007, Assessing the long-run availability of copper. Resources Policy 32:19-23

Tilton, J.E. 2012. Is mineral depletion a threat to sustainable mining? Views III. Society of Economic Geologists Newsletter. 82:3pp

http://www.tms.org/superalloys/10.7449/1989/Superalloys_1989_553_566.pdf

Tran, A. 2007. Quest for calcined bauxite. M: 2007: 32-41

Turner, G.M. 2008 A comparison of the limits to growth with thirty years of reality. Socio-economics and the environment in discussion: CSIRO Working papers, 2008 Series 2008-2009

Turner, G. M., 2012. On the Cusp of Global Collapse? Updated Comparison of The Limits to Growth with Historical Data GAIA 21:116 – 124

Turner, G. 2014 Is Global Collapse Imminent?, MSSI Research Paper No. 4, Melbourne Sustainable Society Institute, The University of Melbourne. ISBN: 978 0 7340 4940 7

Turner, T. 1900, The Metallurgy of Iron (2nd ed.), Charles Griffin & Company, Limited

UNCTAD. (2000). Recent and planned changes in production capacity for bauxite, alumina and aluminium. http://unctad.org/en/docs/poitcdcomd27.en.pdf

Tytgat J, Lopez I, van Damme G. 2008 End of life management and recycling of rechargeable lithium ion, lithium-polymer and nickel metal hydride batteries. An industrial award-winning comprehensive solution. Umicore.: http://www.batteryrecycling.umicore.com/download/UmicoreBattery/Recycling2008.pdf;

United Nations 2003. World Population Prospects: The 2002 Revision, vol. I, Comprehensive Tables (United Nations publication, Sales No. E.03.XIII.6).

UN Comtrade, 2014. Import of Natural sand except sand for mineral extraction as reported. United Nations Commodity Trade Statistics Database. http://comtrade.un.org

UNEP 2011a. Recycling Rates of Metals – A Status Report, A Report of the Working Group on the Global Metal Flows to the International Resource Panel. Graedel, T.E., Allwood, J.; Birat, J.-P.; Reck, B.K.; Sibley, S.F.; Sonnemann, G.; Buchert, M.; Hagelüken, C. 48pp.

UNEP, 2011b. The International Resource Panel. Edited by Graedel, T.E., Buchert, M., Reck, K., Sonnemann, G., UNEP Metal stocks in society; Scientific synthesis. Recycling rates of metals: A status report. 32pp. ISBN 978-92-807-3182-0.

UNEP 2011c: The UNEP yearbook 2011. UNEP division of early warning and assessment. Nairobi, Kenya

UNEP 2011d: Metal stocks in society – Scientific synthesis, A Report of the Working Group on the Global Metal Flows to the International Resource Panel. Graedel, T.E. 52pp.

UNEP 2012: International Resource Panel: Responsible Resource Management for a Sustainable World. Findings of the International Resource Panel. 32pp. ISBN 978-92-807-3278-0. 2012

UNEP 2013a Metal Recycling: Opportunities, Limits, Infrastructure, A Report of the Working Group on the Global Metal Flows to the International Resource Panel. Reuter, M.A.; Hudson, C.; van Schaik, A., Heiskanen, K., Meskers, C., Hagelüken, C. ISBN: 978-92-807-3267-2. 320pp

UNEP 2013b Environmental Risks and Challenges of Anthropogenic Metals Flows and Cycles, A Report of the Working Group on the Global Metal Flows to the International Resource Panel. van der Voet, E.; Salminen, R.; Eckelman, M.; Mudd, G.; Norgate, T.; Hischier, R. ISBN: 978-92-807-3266-5. 234pp

UNEP 2013c. The International Resource Panel. (Reuter, M., Hudson, C., Schalk, A., Heiskanen, K., Meskers, C., Hagelüken, C.) UNEP Metal recycling; Opportunities, Limits, Infrastructure. 317pp. ISBN 978-92-807-3167-2. 2013

UNEP GEAS 2014; Sand, rarer than one thinks. Downloaded document. 15 pages. http://www.unep.org/pdf/UNEP_GEAS_March_2014.pdf

Unknown, 2009. Iron ore pricing emerges from stone age, Financial Times, October 26, 2009

US Department of the Interior, Bureau of Mines and Geological Survey, 1980. Principles of a Resourcel Reserve Classification for Minerals (Washington, DC: Geological Survey Circular 831

US-EPA 1993, Markets for recovered aluminium. FPA/530 SW 90 077A

US Environmental Protection Agency. 1994. Technical Resource Document Extraction and benefication of ores and minerals. 127pp. EPA 530-R-94-011.

US Department of the Interior, Bureau of Mines and Geological Survey, 1980. Principles of a Resource Reserve Classification for Minerals. Washington, DC: Geological Survey Circular 831

US-EIA 2013. Technically Recoverable Shale Oil and Shale Gas Resources: An Assessment of 137 Shale Formations in 41 Countries Outside the United States Analysis and projections.

USGS, 2008, 2009, 2010, 2011, 2012, 2013, 2014, 2015 Commodity Statistics for a Number of Metals (Consulted Several Times 2008–2015). United States Geological Survey. <http://minerals.usgs.gov/ minerals/pubs/commodity/.

USGS 2009; United States Geological Survey, 2009, Mineral commodity summaries 2009: Appendix C, 195 p.

USGS, 2013b. Sand and gravel (construction) statistics, in: Kelly, T.D., Matos, G.R., (Eds.), Historical statistics for mineral and material commodities in the United States. U.S. Geological Survey Data Series 140, Reston.

U.S. Geological Survey National Mineral Resource Assessment Team. 1998 assessment of undiscovered deposits of gold, silver, copper, lead, and zinc in the United States. U.S. Geological Survey circular; 1178, Series TN23.U57 1999

Vaßen, R.; Jarligo, M.O.; Steinke, T.; Mack, D.E.; Stöver, D. 2010. Overview on advanced thermal barriercoatings. Surf. Coat. Technol. 205:938–942

van Oss, H. 2014. Cement. Mcs-2014-cemen.pdf. USGS website. U.S. Geological Survey, Mineral Commodity Summaries 38-39

van Vuuren, D.P., Strangers, B.J., de Vries, H.J. 1999. Long-term perspectives on world metal use—a system-dynamics model. Resources Policy, 25, 239–255.

Velegrakis, A.F., Ballay, A., Poulos, S., Radzevicius, R., Bellec, V., Manso, F., 2010. European marine aggregates resources: Origins, usage, prospecting and dredging techniques. Journal of Coastal Research 51:1-14.DOI: 10.2112 / SI51-002.1.

Vermeulen, J., du Plessis, D., Mohanlal, K., Ackhurst, C., van Geunen, G., 2013. RBPlat integerated annual report: Mineral resources and mineral reserves. Publicized mineral resources and mineral reserves 2013. Royal Bakofeng Platinum.

Vikstrøm, H., Davidsson, S, Höök, M., 2013. Lithium availability and future production outlooks. Applied Energy 110:252-266.

Visser, W, 2002, Tungsten production and consumption in the CIS, in Outlook for wolfram production and applications in the 21st century: International Wolfram Symposium, 9th, Pittsburgh, PA, September 30- October 4, 2002, Presentation, 22 p.

Vladimirov, A.G., Lyakhov, N.Z., Zagorskiy, V.E., Makagon, V.M., Kuznetsova, L.G., Smirnov, S.Z., Isupov, V.P., Belozerov, I.M., Uvarov, A.N., Gusev, G.S., Yusupov, T.S., Annikova, I.Y., Beskin, S.M., Shokalskiy, S.P., Mikheev, E.I., Kotler, P.D., Moroz, E.N., Gavryushkina O.A., 2012. Lithium Deposits of Spodumene Pegmatites in Siberia. Chemistry for Sustainable Development 20:3-20

von Gruenewaldt, G., 1973, The Main and Upper Zones of the Bushveld Complex in the Roossenekal area, eastern Transvaal: Transaction of the Geological Society of South Africa, 76:207–227.

von Gruenewaldt, G., 1977, The mineral resources of the Bushveld Complex: Minerals Science and Engineering, 9:83–95.

Vulcan, T. 2008; Rhenium: Son of moly. 4 pages. http://www.hardassetsinvestor.com/interviews/1002-rhenium-son-of-moly.html?showall=&fullart=1&start=3

Vulcan, T. 2013; Hard Assets Investor 2012 Rare metal rhenium's toughness under heat & pressure key attraction as super-alloy & turbine material. 6 pages. http://www.hardassetsinvestor.com/features/4666-rare-metal-rheniums-toughness-under-heat-a-pressure-key-attraction-as-super-alloy-a-turbine-material.html?showall=&fullart=1&start=4

Wagner, H., and Fettweis, G.B.L. 2001 About science and technology in the field of mining in the Western world at the beginning of a new century. Resources Policy 27:157-168.

Wahl, J., Harris, K., 2014. New single crystal superalloys – overview and update. MATEC Web of Conferences 14, 17002 DOI: 10.1051/matecconf/20141417002. 6pp.

Walser, P, 2002, World Tungsten mining activities, in Outlook for Wolfram Production and Applications in the 21st Century: International Wolfram Symposium, 9th, Pittsburgh, PA, September 30-October 4, 2002, Presentation, 69 p.

Walston, W.S., 2004. Coating and surface technologies for turbine aerofoils, in K. A. Green, T. M. Pollock, H. Harada et al., Superalloys Warrendale, PA: The Minerals, Metals and Materials Society (TMS), 579–588.

Walther, J.V. 2014. Earth's natural resources. 429pp. Jones and Bartless Learning, Burlington, Massachusetts.

Wang, T, Müller, D.B. Graedel, T.E. 2007. Forging the Anthropogenic Iron Cycle. Environ. Sci. Technol. 41: 5120-5129 DOI: 10.1021/es062761t

Wang, J. and Graedel, T.E., 2010 Aluminium in-use stocks in China: a bottom up study. J. Mater. Cycles Waste. Manag. 12; 66-82. DOI 10.1007/s10163-009-0271-3

Wang, X., Gaustad, G., Babbitt, C.W., Richa, K., 2014. Economies of scale for future lithium-ion battery recycling Infrastructure. Resources, Conservation and Recycling 83:53– 62

Watkins, C. and McAleer, M., 2004. Econometric modelling of non-ferrous metal prices. Journal of Economic Surveys 18: 651-703

Weatherstone, N., 2008, International standards for reporting of mineral resources and reserves—Status, outlook, and important issues: World Mining Congress & Expo 2008, 10 pp., www. crirsco.com/isr_mineral_resources_reserves0908.pdf.

Weisz, H., Suh, S., Graedel, T.E., 2015. Industrial Ecology: The role of manufactured capital in sustainability. Proceedings of the National Academy of Sciences 112; 6260-6264. www.pnas.org/cgi/doi/10.1073/pnas.1506532112

Wellmer, F-W., Becker-Platen, J.D., 2007. Global Nonfuel Mineral Resources and Sustainability. 16pp., Proceedings, Workshop on Deposit Modeling, Mineral Resource Assessment, and Sustainable Development U.S. Department of the Interior | U.S. Geological Survey. http://pubs.usgs.gov/circ/2007/1294/paper1.html

Wellmer, F.-W. 2008: Reserves and resources of the geosphere, terms so often misunderstood. Is the life index of reserves of natural resources a guide to the future? – Z. dt. Ges. Geowiss., 159:575-590. Stuttgart, Schweizerbart'sche Verlagsbuchhandlung.

Wilburn, D.R., and Bleiwas, D.I., 2004, Platinum-group metals—World supply and demand: U.S. Geological Survey Open-File Report 2004–1224, 152 pp. (Also available at http://pubs.usgs.gov/of/2004/1224/2004-1224.pdf.)

Wilburn, D.R., 2012, Global exploration and production capacity for platinum-group metals from 1995 through 2015 (version 1.1): U.S. Geological Survey Scientific Investigations Report 2012–5164, 34 pages. http://pubs.usgs.gov/sir/2012/5164/.

Willburn, D.R., 2015. Cobalt mineral exploration and supply from 1995 through 2013. Scientific Investigations Report 2011–5084. (24 pages) USGS. http://minerals.usgs.gov/minerals/pubs/commodity/cobalt/

Williams, N. 2008. The exhaustion of mineral resources: a truism or a state of mind? Proceedings of the 33rd International Geological Congress, Oslo, August.

Williamson, A. 2003. Russian PGM stocks. The LBMA Metals Conference, Lisbon, Portugal.

Woodcock, J.T. and Hamilton, J.K. (Eds) 1993 Australasian Mining and Metallurgy : The Sir Maurice Mawby Memorial, Volume 2, The Australasian Institute of Mining and Metallurgy, Melbourne.

World Copper; Resources and Technology. 1988. In: U.S. Congress, Office of Technology Assessment. Copper: Technology and Competitiveness, OTA-E-367. Washington, DC: U.S. Government Printing Office, September 1988. Library of Congress Catalog Card Number 87-619893, pages 87-180. https://www.princeton.edu/~ota/disk2/1988/8808/880807.PDF

World Ocean Review 2015. Mineral resources; Manganese nodules. 2 pages. http://worldoceanreview.com/en/wor-3-overview/mineral-resources/manganese-nodules/

World Ocean Review 2015. Mineral resources; Cobalt crusts; 2 pages. http://worldoceanreview.com/en/wor-3-overview/mineral-resources/cobalt-crusts/

World Ocean Review 2015. Mineral resources; Massive sulphides in smoky depths. 3 pages. http://worldoceanreview.com/en/wor-3-overview/mineral-resources/manganese-nodules/

World Steel Association 2012. The white book of steel. ISBN 978-2-930069-67-8

World Steel Association 2013. World steel in figures 2013. 17pp

World Steel Association 2013. Economic Sustainability, Policy and Indicators Association 2013. 12pp. ISBN 978-2-930069-75-3. www.worldsteel.org

World Steel Association 1999 Steel statistical yearbook. International iron and steel institute. Committee on economic studies, Brussels , published online 2000. 248pp, ISBN 2-930069-40-6 www.worldsteel.org

World Steel Association 2014a. A handbook of world steel statistics. Committee on statistics, Brussels 1978 69pp. www.worldsteel.org

World Steel Association 2014b. Crude steel production 1980-2012. www.worldsteel.org

World Steel Association 2014c. Blast furnace iron production 1980-2012. www.worldsteel.org

World Steel Association 2014d. Direct reduced iron production 1980-2012. www.worldsteel.org

Worrell E, Martin N, Price LK. 1999 Energy efficiency and carbon dioxide emissions reduction opportunities in the US iron and steel sector. Berkeley, CA: Ernst Orlando Lawrence Berkeley National Laboratory.

Wübbeke, J. 2012 China's Mineral and Metals Industry: On the path towards sustainable development? Pacific News 38;1-4

Wübbeke, J. and Heroth, T., 2014 Challenges and political solutions for steel recycling in China. Resources, Conservation and Recycling. 87:1-7.

Wäger, P.A., Lang, D.J., Wittmer, D., Bleischwitz, R., Hagelüken, C., 2012. Towards a more sustainable use of scarce materials. A review of intervention options along the metal cycle life. Gaia 21:300-309.

Wäger, P, Widmer, R., Stamp, A., 2011. Scarce technology metals – applications, criticalities and intervention options. EMPA. Federal Office of the Environment Bern, Switzerland.

Yaksic, A. and Tilton, J.E., 2009. Using the cumulative availability curve to assess the threat of mineral depletion: The case of lithium. Resources Policy. 34:185-194

Yamabe-Mitarai, Y., Ro, Y., Maruko, T., Harada, H., 1998. Ir-Base Refractory Superalloys for Ultra-High Temperatures. Metallurgical and Materials Transactions 29A:537-549

Yamabe-Mitarai, Y., Gu, Y.F., Harada, H. 2002. Two-Phase Iridum-Based Refractory Superalloys. Their developmentand possibilities as high temperaturestructural materials. Platinum Metals Rev. 46:74-81

Yamabe-Mitarai, Y., Gu, Y.F., Völkl, R., Harada, H. 2004. Platinum-Group-Metal-Based Intermetallics as High-Temperature Structural Materials. JOM. 56: 34-39.

Yang, C-J., 2009. An impending platinum crisis and its implications for the future of the automobile. Energy Policy 37:1805-1808.

Yeh A.C., Tsao T.K., Chang Y.J., Chang K.C., Yeh J.W., Chiou, M.S., Jian, S.R., Kuo, C.M., Wang, W.R., Murakami, H., 2015 Developing new type of high temperature alloys–high entropy superalloys. International Journal of Metallurgical & Materials Engineering 1:107-111. doi: http://dx.doi.org/10.15344/2455-2372/2015/107

Yellishetty, M., Mudd, G.M., 2014. Substance flow analysis of steel and long term sustainability of iron ore resources in Australia, Brazil, China and India. Journal of Cleaner Production 84: 400-410

Yellishetty, M., Ranjith, P.G., Tharumarajah, A., 2010. Iron ore and steel production trends and material flows in the world: Is it really sustainable? Resources, Conservation and Recycling 54: 1084-1094.

Yellishetty, M., Mudd, G.M., Ranjith, P.G., 2011. The steel industry, abiotic resource depletion and life cycle assessment: a real or perceived issue? J. Clean. Prod. 19:78-90

Yellishetty, M., Mudd, G., Mason, L., Mohr, S., Prior, T., Giurco, D. 2012. Iron resources and production: technology, sustainability and future prospects. Prepared for CSIRO Minerals Down Under Flagship, by the Department of Civil Engineering (Monash University) and the Institute for Sustainable Futures (University of Technology, Sydney), October 2012. ISBN 978-1-922173-46-1.

Yoda, S.,and Ishihara, K. 1997. Global energy prospects in the 21st century: a battery-based society. Journal of Power Sources. 68:3-7

Yoshimura, A., Daigo, I., Matsuno, 2013. Global substance flow analysis of indium. Materials Transactions , 54:102-109.

Yu, X.X., Wang, C.Y., Zhang, X.N., Pan, P., Zhang, Z., 2014. Synergistic effect of rhenium and ruthenium in nickel-based single-crystal superalloys. Journal of Alloys and Compounds 582:299–304

Ziemann, S., Weil, M., Schebek, L., 2012. Tracing the fate of lithium-The development of a material flow model. Resources, Conservation and recycling 63:26-34.

Ziemann S, Weil M, Schebek L. 2010 Dissipative application of lithium–lost for the future? In: 4th international seminar on society and materials (SAM4). Nancy: Sovamat; 2010. Ziemann S., Weil M., Schebek L. (Editors) Energietechnologien der Zukunft: die Problematik der Rohstoffverfügbarkeit am Beispiel von Lithium. Technikfolgenabschätzung – Theorie und Praxis 19:46–55

Zhao, H., Levi, C.G., Wadley, H.N.G. 2014. Molten silicate interactions with thermal barrier coatings. Surface and Coatings Technology doi:10.1016/j.surfcoat.2014.04.007

Zhao, H., Yu, F., Bennett, T.D., Wadley, H.N.G. 2017. High temperature coatings for superalloys. Virginia Institute of Technology website. http://www.virginia.edu/ms/research/wadley/high-temp.html

Zhou, G, Li, F., Cheng, H-M., 2014. Progress in flexible lithium batteries and future prospects Energy Environ. Sci., 7:1307-1338

Zhou, H. 2007. The chemical environment of cobalt-rich ferromanganese crust deposits, the potential impact of exploration and mining on this environment, and data required to establish environmental baselines in the exploration areas. In: (Ed. ISA) Polymetallic sulphides and cobalt rich ferromanganese crust deposits: establishment of environmental base- lines and an associated monitoring programme during exploration, Proceedings of the International Seabed Authority's workshop, Kingston, Jamaica, 6-10 September 2004, 257–267.

Zhao, H., Levi, C.G., Wadley, H.N.G. 2014. Molten silicate interactions with thermal barrier coatings. Surface and Coatings Technology doi:10.1016/j.surfcoat.2014.04.007

Zientek, M.L., Causey, J.D., Parks, H.L., Miller, R.J., 2014, Platinum-group elements in southern Africa—Mineral inventory and an assessment of undiscovered mineral resources: U.S. Geological Survey Scientific Investigations Report 2010–5090–Q, 142pp http://dx.doi.org/10.3133/sir20105090Q

Zimasset 2014. Zimasset awareness seminar for members of the Parliament of Zimbabwe 12-13. March 2014. Powerpoint presentation. Downloaded from: http://www.awepa.org/news/members-parliament-zimbabwe-participate-two-day-seminar-zim-asset/

Zimmerman, T., Gößling-Reisemann. S. 2014. Recycling Potentials of Critical Metals-Analyzing Secondary Flows from Selected Applications. Resources 3:291-318; doi:10.3390/resources3010291