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Resource sustainability assessment of the use of non-ferrous metals in advanced materials applications
Dutch translation of the title:
Duurzaamheidsanalyse van het grondstoffengebruik bij aanwending van non-ferrometalen in geavanceerde materiaaltoepassingen

Suggested citation:

ISBN-number: 978-90-5989-674-1

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Acknowledgements

The text presented to you in the following is the result of a four year long journey. I would like to express my thanks to those who accompanied me on this expedition. First of all I would like to thank Professor Jo Dewulf, who acted as my guide. Time and time again he proved his keen eyes and his advice helped me to look at things from a different angle more than once. Then I had the privilege to have group of advisors from industry with whom the ongoing work was discussed in several meetings over the years. Many thanks for their cooperation in this project go to Marleen Esprit, Bénédicte Robertz and Staf Laget from Umicore.

I would like to express my appreciation for the members of the jury and especially the three members of the reading committee: Marisa Vieira, Professor Eric Pirard, Professor Marc Verhaege. Their comments were of great help in improving the original manuscript. I would also like to acknowledge the financial support I received via a scholarship from the Flemish agency for Innovation by Science and Technology (IWT). Furthermore, I would like to thank the countless people at Umicore and Bebat who collaborated in the case studies that I was directly involved in.

Then I would like to thank the people of our research group and our office. I am rather a lone wolf than a party animal. Nevertheless, whenever I needed assistance with something, someone was there to help. I also very much enjoyed the discussions we had among the ‘exergy group people’. There are a number of memorable moments, which I will take with me: from Christmas quizzes, over office rearrangements and freezing together, to preparing GC-exercises and the sadness when someone left for good.

For helping me take my mind off of work I would like to thank my friends; close and far away. For their constant support I thank my family in Germany of whom I have seen far too little these past years. Luckily we do not rely on letters any more for long distance communication. Thanks also go to the Belgian family for their support. And last but not least thanks also to my home base in Ghent for propping me up now and then.

Pilar Swart, December 2013
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List of Abbreviations

ADP – Abiotic Depletion Potential
AoP – Area of Protection
CED – Cumulative Energy Demand
CEENE – Cumulative Exergy Extraction from the Natural Environment
CExC – cumulative exergy consumption
CExD – Cumulative Exergy Demand
CF – characterization factor
CML - Institute of Environmental Sciences – Leiden University
DNI – direct normal irradiation
EDIP – Environmental Design of Industrial Products
ELU – Environmental Load Unit
EPS2000 – Environmental Priority Strategies version 2000
EPBT – Energy Payback Time
ExPBT – Exergy Payback Time
FEE – Fossil energy equivalents
GWP – global warming potential
HCPV – high concentration photovoltaic
ICEC - Industrial Cumulative Exergy Consumption
ICSG – International Copper Study Group
ILCD – International Reference Life Cycle Data System
IEA – International Energy Agency
IPCC – Intergovernmental Panel on Climate Change
JORC – Joint Ore Reserves Committee
LCA – life cycle assessment
LCC – life cycle costing
LCI – life cycle inventory
LCIA – life cycle inventory assessment
Li-ion – lithium ion
MCI – marginal cost increase
MIPS – Material Intensity Per Unit Of Service
MFA – material flow analysis
MOVPE - metalorganic vapour phase epitaxy
OP – open pit
PGM – platinum group metal
PLS – pregnant leach solution
PV - photovoltaic
RMD – Raw Materials Data
REE – rare earth element
SETAC – Society of Environmental Toxicology and Chemistry
SLCA – social life cycle assessment
SXEW – solvent extraction and electrowinning
UG – underground
UNEP – United Nations Environment Programme
USGS – United States Geological Survey
WTP – willingness-to-pay
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Part I
General Introduction:
Sustainability and the role of primary metals

1 Part I is in parts based on a chapter entitled “Abiotic resource use” prepared by Pilar Swart, Rodrigo Alvarenga and Jo Dewulf. This chapter is submitted for publication in volume 4 (Life Cycle Impact Assessment – LCIA) of the Springer Encyclopedia of Life Cycle Assessment edited by Michael Zwicky
1 The sustainability debate in sensu lato

In modern times Thomas Robert Malthus was one of the first to voice concerns regarding the sustainability of mankind. In his work “An essay on the principle of population” he postulated that “the power of population is indefinitely greater than the power in the earth to produce subsistence for man” (Malthus, 1798). Thus, population growth eventually would be held in check by available resources.

Due to technological advances more particularly the green revolution and our ability to make use of fossil fuel resources, global population growth has continued up till now. Nevertheless, time and again others followed in his foot, warning that the Earth’s carrying capacity might not be sufficient to sustain all of us. In the second half of the 20th century the “The Limits to Growth” (Meadows et al., 1972) was published. Scenarios modelled by the researchers projected that resource constraints and pollution would lead to a collapse of the global system with a sudden decrease of the population. The study triggered an intense debate which seemingly was won by the work’s opponents. Since then the debate had subsided, but in recent years again a number of studies have been published that revisit the projections of the original publication and find that it was not all that wrong up to now (Bardi, 2011; Hall and Day, 2009; Turner, 2008), though there are also those that remain critical (Popper et al., 2005; Radetzki, 2006).

Another milestone in the sustainability debate is the so-called Brundtland report (WCED, 1987) and with it the often quoted definition of sustainable development, which can hardly be avoided when discussing this topic: “Sustainable development is development that meets the needs of the present without compromising the ability of future generations to meet their own needs” (WCED, 1987). How can we know what the future generations will need? How can we handle the uncertainties related to the effects of our behaviour on the means of livelihood for future generations? Well, we cannot know everything with certainty, therefore the precautionary principle can serve as an important guideline along the way (Paterson, 2007; Santillo, 2007).

As if things were not already complicated enough, next to environmental issues, which are usually linked to sustainability in the minds of the general public, there are also other aspects of sustainability namely the economic and the social, which are also acknowledged in the Brundtland report. The three together are often referred to as the three pillars of sustainability. The three pillars are not clearly separated from each other and thus, in an ideal world, all three aspects should be evaluated together. Considering the above, it seems nigh impossible to say that any development is truly sustainable.

On the road to sustainable development we are faced with immense challenges. The world population is still growing and is projected to reach 10.9 billion by 2100 in a medium fertility scenario (United Nations, 2013). Consumption of natural resources is following suit (Dobbs et al., 2011) and climate change and its effects on nature and human societies (IPCC, 2007) do not contribute to making the prospects for life on Earth any more positive. Figure 1-1 and Figure 1-2 show population growth since 1950 and projections up till 2100 and historical development of material use between 1900 and 2005 for a range of resources, respectively.
Given these developments it is not surprising that there is a growing interest into matters concerning sustainability, which is for example reflected by the soaring number of research papers published between 1990 and 2010 with “sustainability” or “sustainable development” as topic, compared to the increase of the number of papers published with generic keywords like “study” or “analysis” (“Web of Knowledge [v.5.10] - All Databases Home,” 2013). Concerns about the carrying capacity of the planet have become more accessible to the general public, for example with the development of assessment methods like the Ecological Footprint (Ewing et al., 2010; Wackernagel et al., 2002).

Figure 1-1 Projections of population growth. (Source: United Nations (2013)).

Figure 1-2 Evolution of material use between 1900 and 2005 (Source: Krausmann et al. (2009)).
How can these problems be approached in practice? There are various tools which should help us to make development more sustainable by providing guidance to decision makers. On the macro level material flow analysis (MFA) (Allen et al., 2009; Eurostat, 2001; OECD, 2008) or indicators like the Human Development Index (United Nations Development Programme, 1990) can give indications on where societies stand compared to others and in which direction they are headed when looking at the evolution over several years. MFA indicators can give information on the flows of specific substances (often pollutants) or the resource flows related to businesses, economic sectors or regions. Material and energy flows related to a specific product are often studied in the framework of life cycle assessment (LCA). LCA is a tool which can be viewed as a type of MFA (OECD, 2008) on the micro-level. LCA studies usually assess a wide range of environmental impacts, mainly related to pollution, but also those related to resources, and include all life cycle stages of a product from the extraction of natural resources, the cradle, over manufacturing, use and end of life, the grave. This is done to avert the problem of reducing impacts at one point, while increasing impacts at another point. All downstream and upstream processes in the technosphere related to the product and their exchanges of material and energy streams with the environment are in principle to be compiled in the so-called life cycle inventory (LCI). Subsequently, these streams have to be classified, e.g. greenhouse gas emissions are assigned to the impact category climate change. Eventually, the impacts are quantified and interpreted. LCA has been around for some decades and has been standardized to some extent in particular with the publication of ISO standards starting in 1997 (Rebitzer et al., 2004), which have since been revised with the publication of the ISO 14040 and ISO 14044 standards (Finkbeiner et al., 2006; ISO, 2006a, 2006b). Additionally, documents have been published by the European Commission giving general guidance on conducting an LCA (EC-JRC, 2010a), and providing a detailed standard for performing assessments of life cycle environmental impacts of products (European Commission, 2013). By now a number of databases are available to speed up the process of compiling LCIs for the product at hand. LCA practitioners do not have to collect data on environmental exchanges on every single upstream and downstream process anymore; instead they can for example use a dataset for electricity supply already available. An overview of LCI databases is provided in Curran (2012). Next to databases also some specialized software programmes have been developed to support LCA (Ciroth, 2012). Notwithstanding the progress that has been made, LCA is still a relatively young discipline and there are still many issues to be solved so development is continuing (Finnveden et al., 2009; Guinée et al., 2011; Reap et al., 2008a, 2008b).

Traditional LCA is limited to environmental issues and thus does not cover all three pillars of sustainability, thus complementary tools covering economic and social aspects are being combined with LCA (Kloepffer, 2008). Life cycle costing (LCC), like LCA, is a life cycle approach. Instead of quantifying environmental impacts, LCC quantifies economic costs of products borne by stakeholders over their life cycle. In principle, externalities can also be included in LCC, but when combined with LCA double counting with environmental impacts considered there should be avoided (Wood and Hertwich, 2013). Guidelines for conducting environmental LCC have been published by the Society of Environmental Toxicology and Chemistry (SETAC) (Swarr et al., 2011). Social life cycle assessment (SLCA) is still in the early development. Nevertheless, guidelines for SLCA are already available from the United Nations Environment Programme (UNEP) (Benoît et al., 2009). The need for a stronger integration of environmental, economic and social aspects in a life cycle sustainability analysis framework has been recognized beyond performing LCA, LCC and SLCA alongside but separate from each other (Guinée et al., 2011).
The aforementioned tools for decision support can for example be applied to new technologies in order to determine whether technologies that are claimed to fulfil human needs in a more sustainable way, actually live up to those promises. Examples of technologies promising to be more sustainable are biobased materials, electric mobility, or energy provision from renewable resources. Technology has certainly helped us to increase the productivity of available natural resources, especially when it comes to land resources and the production of food (Trewavas, 2002). As a result technology vastly increased the carrying capacity of the planet at least temporarily. Eco-innovation is a term used to describe new technologies, but also institutional innovations, that reduce environmental impact (Kemp and Pearson, 2007; OECD, 2009). However, the case of biobased energy has highlighted once more that we always have to think one step further when assessing sustainability of new technologies (Tirado et al., 2010; Woess-Gallasch et al., 2011).

On a more conceptual level it is suggested to make human societies more sustainable by decoupling economic growth from resource use and environmental impacts (Fischer-Kowalski and Swilling, 2011). This is based on the observation that typically economic growth is tight to increased resource use. In principle, decoupling aims to sever this link between economic growth and resource use by introducing innovations that increase resource productivity and, in the case of developing countries, leapfrogging, i.e. developing countries do not follow the same course of development as the more developed countries, but directly employ more sustainable technologies which are already available. Proponents of the steady state economy concept go one step further as they explicitly advocate that physical stocks in society remain constant with only minimal maintenance flows (Daly, 1987), whereas decoupling in a first instance only implies that the material and energy intensity per unit of economic growth decreases.

From the above introduction to the sustainability debate it should be clear that in principle the problems posed by Malthus and others after him are still unresolved and that the right solutions will not present themselves on a silver platter. The population is still growing, pollution problems, especially in the form of climate change, are not all that easily controlled on a global scale and resource constraints need to be overcome. This thesis tries to make a contribution to the topic of sustainability in one very specific area, which is advanced material applications employing non-ferrous metals and impacts on natural resources in general and non-ferrous metal resources in particular.
After environmental pollution had dominated the sustainability debate for a long time, concern for resource sustainability has been rekindled, which is also reflected in the launch of the International Resource Panel in 2007 (“International Resource Panel,” 2013). The panel is hosted by the UNEP and its purpose is to provide scientific information on sustainable resource use and environmental impacts of resource use. Resources may be defined as those elements that are extractable for human use and thus have a functional value for society (Udo de Haes et al., 2002). Resources may be classified according to different typologies:

**Renewable and non-renewable:** Typically renewable resources are defined as resources that are replenished naturally on a relatively short time frame. Examples for renewable resources are fish stocks or solar energy. Non-renewable resources on the other hand are not replenished at all or their renewal is so slow that it can be regarded as negligible on the human time scale. Examples for non-renewable resources are fossil resources or primary forest. It should be noted that though renewable resources are replenished, they can still be exhausted if the extraction rate exceeds the renewal rate.

**Biotic and abiotic:** Biotic resources are materials derived from presently living organisms. In addition to the resource value, they typically have an important role in maintaining ecosystem services and also intrinsic value (examples are tropical hardwood and ivory). Abiotic resources are the product of past biological processes (coal, oil and gas) or physical/chemical processes (deposits of metal ores) (Guinée, 1995; Müller-Wenk, 1998).

**Funds, flows and stocks:** In the case of stocks extraction inevitably leads to the depletion of the resource, i.e. reduction of the available amounts in nature, whereas funds may be depleted but also have a renewal rate which is high enough to allow the resource to recover. Usually biotic resources are categorized as funds, but also groundwater can be regarded as fund resource. Flow resources cannot be depleted. Their availability per unit time however is limited, and thus their extraction is marked by competition (e.g. wind energy) (Heijungs et al., 1997; Lindeijer et al., 2002).

Metals are an important resource for mankind. They are everywhere in use around us: in basic appliances like pots and hammers, in luxury items like jewellery and in advanced material applications like solar cells and battery applications. Metals and metalloids are valuable to us because they have unique properties: like heat and electrical conductivity, mechanical strength or band gaps. Most of the elements in the periodic table are actually metals or metalloids. The provision of metals can be considered one of the services of the natural environment (Zhang et al., 2010b). As in the case of resources in general, metal resource use has increased tremendously over the decades for many metals (Figure 2-1).

Metals occur in the lithosphere, but also in the hydrosphere. The absolute amount of many metals in the continental crust alone would suffice to provide mankind for a long time. From the average copper concentration in the continental crust (Wedepohl, 1995) and its mass (Liu and Rudnick, 2011) it follows that the continental crust contains 550 \(10^{12}\) tonnes of copper. This amount would be equivalent to almost 500 years of primary copper production, even if
production continued to grow with 3% per year. On the flip side, though the average rock in the continental crust contains 7.96% aluminium and 4.32% iron, its copper content is only 25 ppm (Wedepohl, 1995). Considering that the lower the metal concentration in the rock, the more energy is likely to be expended in order to obtain the metal in its pure form, occurrences of metals are much more attractive for metal production when they contain higher amounts of the metals we need. The occurrences in the lithosphere which have high concentrations of specific metals are usually called ores and are the primary metal resources we typically use today instead of the average rock. The concentration of a metal in the ore is called ore grade. The ore grade of the currently exploited ores depends on the targeted metals, ranging from the ppm level for precious metals, over some percentage points for metals like copper, zinc and lead to more than 10% for aluminium, iron or manganese.

![Figure 2-1 Historical development of the world primary production of some metals up to 2010 (Data sourced from Kelly and Matos (2012)).](image)

The amount of metals available for extraction can be evaluated using different definitions, e.g. in its annually published ‘Mineral Commodity Summaries’ the United States Geological Survey (USGS) (2002) makes a distinction between ‘reserves’ and the ‘reserve base’. The reserves are the resources that can currently be extracted economically and the reserve base includes also additional resources that meet certain criteria relevant for mining and production practice (e.g. depth of deposits). Next to the definitions handled by USGS there are also a number of codes intended for public reporting of resources and reserves by companies. In the code prepared by the Australian Joint Ore Reserves Committee (JORC) (2012) more extensive definitions and explanations on the classification of ‘Mineral Resources’ and ‘Ore Reserves’ are provided. For Mineral Resources there have to be “reasonable prospects for eventual economic extraction” (JORC, 2012, p. 11). In addition further differentiations are made based on the extent and source of information available on the Mineral Resources and Ore Reserves. For Mineral Resources these range from ‘Inferred Mineral Resources’, over ‘Indicated Mineral Resources’ to ‘Measured Mineral Resources’. The code further specifies that an Ore Reserve “is the economically mineable part of a Measured and/or Indicated Mineral Resource” (JORC, 2012, p. 16). For ‘Ore Reserves’ a distinction is made between ‘Probable Ore Reserves’ and ‘Proved Ore Reserves’.

There are different theories concerning how the amount of available resources is related to the concentration. Some propose a unimodal distribution, while others suggested a bimodal
distribution at least for some metals (Tilton, 2001). Generic representations of the unimodal and bimodal distributions are depicted in Figure 2-2. In the bimodal distribution the mode at higher concentrations represents the concentrated occurrences of metals, i.e. the resources we are using today, whereas the mode at lower concentrations represents occurrences where metals occur like in the average rock. A distinction between the metals whose distribution likely follows a bimodal curve and those metals that are more likely distributed unimodally observed on the basis of the enrichment factor. The enrichment factor is the ratio between a metal in a more concentrated form, e.g. an ore, and the average crustal abundance of the metal (average concentration in the crust). There are a number of metals that are quite common in the crust, for example iron and aluminium. In fact the ores which are specifically mined to recover those metals require on average an enrichment factor above 6 to make recovery viable (Dill, 2010; Evans, 1993; von Gleich, 2006). In contrast the enrichment factor required on average for copper ores is around 70 and for zinc ores it is in excess of 500 (Dill, 2010).

Figure 2-2 Skinner (1976) suggested that scarce metals are distributed according to a bimodal curve, while only the more abundant metals are distributed according to a unimodal function (Adapted from Skinner (1976)).

For the gap between the two modes Skinner (1979) used the term “mineralogical barrier”. Much higher amounts of energy are expected to be required to process metals from rock on the other side of the mineralogical barrier (Norgate, 2010). As the mode at higher concentrations provides much less metal than the second mode it is not inconceivable that the “mineralogical barrier” is something which is relevant on the human time scale. Based on the distributions shown in Figure 2-3 Gerst (2008) found it more likely that copper indeed follows a bimodal distribution instead of a unimodal one.

Geological processes produced a variety of metal deposits over time (Mukherjee, 2011; Tilton, 2001). There are various deposit types with different characteristics. An important aspect is that often two and more metals are enriched in the same deposit (Dill, 2010; Evans, 1993). Depending on the deposit type different metals can be recovered together. For example, porphyry copper deposits usually contain copper and molybdenum or copper and gold. Sedimentary exhalative deposits are the major source of lead and zinc, while copper and silver may also be recovered. This means that in many cases the production of one metal is not independent of the production of other metals.

Because the geological processes that produced the metal resources exploited today are so slow, compared to the rate with which existing deposits are exploited, metal resources are considered non-renewable. Yet, in contrast to fossil fuels metals are in principle not destroyed
when used, because their functional value lies in the element and we do not exploit metal resources for their energy content with the exception of radioactive elements, like uranium. As a consequence it makes sense to recycle metals after their use. In fact, for a number of metals recycling contributes a substantial amount to their annual supply: 18% of total refined production for copper in 2012 was provided by recycled copper (International Copper Study Group (ICSG), 2013), and recycling satisfied 20% of the gross platinum demand in 2009 (Johnson Matthey, 2009). It is necessary to realize that even though metals are not destroyed, there are some limitations to recycling them. First of all they can be dissipated to the environment when used in certain applications, e.g. zinc based coatings or copper pesticides. Secondly, recycling from products, in which metals usually occur together with other metals, can be technologically challenging (Hagelüken and Meskers, 2010). Thirdly, there has to be an economic or legal incentive for recycling.

Apart from their status as a non-renewable and therefore potentially limited resource, metals are also specifically relevant to the sustainability debate because they are often essential to applications that are targeted at improving the sustainability of human society. Table 2-1 shows selected special and precious metals and their clean energy technology applications, which include photovoltaics (PV) based on a range of active materials and electric vehicle employing lithium ion (Li-ion) batteries. Globally the demand for special metals has increased more than the demand for bulk metals like copper and is forecasted to increase further in many cases (Angerer et al., 2009; Hagelüken and Meskers, 2010; Halada et al., 2008). For example, world production of indium almost doubled between 2000 and 2010, while primary world production of cobalt more than doubled during the same time frame according to data published by the US Geological Survey (Kelly and Matos, 2012).

![Figure 2-3 Grade-tonnage relationships for common rock and copper resources including undiscovered resources (Source: Gerst (2008)).](image)
Table 2-1 Special metals and precious metals used in clean technologies (Source: Speirs et al. (2013)).

<table>
<thead>
<tr>
<th>Material</th>
<th>Symbol</th>
<th>Clean Energy Technology Applications</th>
</tr>
</thead>
<tbody>
<tr>
<td>Silver</td>
<td>Ag</td>
<td>Photovoltaics (c-Si), Concentrated Solar Power (CSP), Nuclear</td>
</tr>
<tr>
<td>Cobalt</td>
<td>Co</td>
<td>Electric vehicle batteries, Biofuels (Fischer–Tropsch process)</td>
</tr>
<tr>
<td>Gallium</td>
<td>Ga</td>
<td>Photovoltaics (CIGS)</td>
</tr>
<tr>
<td>Germanium</td>
<td>Ge</td>
<td>Photovoltaics (a-SiGe)</td>
</tr>
<tr>
<td>Indium</td>
<td>In</td>
<td>Photovoltaics (CIGS, Transparent Conductive Oxide)</td>
</tr>
<tr>
<td>Lithium</td>
<td>Li</td>
<td>Electric vehicle batteries</td>
</tr>
<tr>
<td>Platinum Group</td>
<td>PGMs</td>
<td>Fuel Cells, Catalytic Converters</td>
</tr>
<tr>
<td>Metals</td>
<td></td>
<td></td>
</tr>
<tr>
<td>Rare Earth</td>
<td>REE</td>
<td>Electric vehicle batteries and motors, wind turbine generators, efficient lighting (phosphors)</td>
</tr>
<tr>
<td>Selenium</td>
<td>Se</td>
<td>Photovoltaics (CIGS)</td>
</tr>
<tr>
<td>Tellurium</td>
<td>Te</td>
<td>Photovoltaics (CdTe)</td>
</tr>
</tbody>
</table>

It may come as no surprise that there is a particular concern for the availability of those metals and thus they are the focus of a number of reports that assess the criticality of these metals by determining supply risks and the vulnerability of technologies and economies to supply disruptions (Angerer et al., 2009; Buchert et al., 2009; Committee on Critical Mineral Impacts of the U.S. Economy, Committee on Earth Resources, National Research Council, 2008; Erdmann and Graedel, 2011; European Commission, 2010; Graedel et al., 2011b; Speirs et al., 2013; U.S. Department of Energy, 2010). The list of 14 raw materials critical to the EU (European Commission, 2010) include among others cobalt, gallium, germanium, indium and niobium. It has to be pointed out that these lists are subject to change as they depend on the region for which the criticality is assessed, the time and the methods used to assess the criticality.

The production of special metals and their applications also goes along with the use of other resources. And there is still some work to do to inventory these resource inputs for performing LCAs. For example, the process based life cycle LCI database ecoinvent, which is used by many LCA practitioners, does not yet have LCI datasets for germanium or germanium dioxide in version 2.2 (Swiss Centre for Life Cycle Inventories, 2010) and the dataset for cobalt is associated with high uncertainty (Hischier, 2007). Moreover, for many advanced materials process data to establish LCIs are missing completely or are based only on information in patents or limited process information supplemented by estimations based on physico-chemical data. However, as these advanced material applications usually require high purity materials their production is potentially associated with high auxiliary requirements, which need to be assessed carefully. Thus, while pollution and use of some resources might be decreased as a result of clean technology applications employing advanced materials, these clean technologies lead to an increased use of other resources, i.e. special metals in particular. In addition the auxiliary required in the production of those advanced materials have to be considered.

For a long time metal resources have not been viewed as threatened from a geological point of view, because new resources were found in previously unexplored territories, while advances in technology made it economically viable to extract metals from ever lower graded deposits. Indeed, according to a number of studies real metal prices, which by some are viewed as a measure of scarcity, have been falling over the 20th century (Cuddington, 1992; Krautkraemer, 1998; von Gleich et al., 2006). This trend is exemplified with copper in Figure...
2-4. Cost decreases caused by economies of scale and new technologies are believed to make lower copper prices feasible (Tilton, 2001; von Gleich, 2006). Recently, the significance of the observation of long term price decreases has been disputed by Svedberg and Tilton (Svedberg and Tilton, 2006), because of the use of biased deflators in previous studies. However, the authors could also not ascertain a general upward trend for copper prices in their case study covering data between 1870 and 2000 (Svedberg and Tilton, 2006). Contrary to past price developments there is some concern for a persistent future rise in resource prices (Dobbs et al., 2011) and metal prices in particular (Humphreys, 2010), because of factors including location of new deposits, cost of energy carriers and ore grades.

Figure 2-4 Inflation adjusted price index of copper in US$ and British pounds. (Source: InflationMonkey (2012))

So should we be concerned about the geological supply of metals? Are there ways to quantify how much those resources are impacted by current extraction? The subsequent chapter gives an overview of methods used in LCA to quantify impacts on metal resources.
3
LCA and the assessment of abiotic resource use

In the beginning of this chapter some general terminology and concepts that are used in LCA are presented. The Area of Protection (AoP) ‘natural resources’ is introduced and subsequently selected methods that can be used to assess abiotic resource use and especially methods used for metals in life cycle impact assessment (LCIA) for this AoP are discussed.

A short introduction to the principles of LCA was already given in chapter 1. An LCA study usually starts with defining the goal and the scope of the study. This includes among others setting up system boundaries, i.e. what is the system to be studied, determining the target audience and goal of the study. Another part of this step is defining the functional unit, i.e. the reference relative to which impacts will be expressed. This is especially important if different products are to be compared as for example one litre of paint A might be sufficient to paint 1 m², while 1.2 litre of paint B are required. Once the goal and scope have been adequately defined the inventory data can be collected. This phase is typically quite time intensive. A frequent problem occurs when processes result in more than one product, e.g. a cogeneration plant produces heat and electricity. If for the study only the electricity production is relevant, a number of possibilities exist to deal with this issue. One of them is the so-called allocation. For the example of the cogeneration plant this means that part of the inputs from and outputs to the environment and the industrial system are assigned to the produced electricity and the other part to the produced heat based on an allocation factors. According to ISO 14044 the allocation factors should reflect the relationship between the products and the other inputs and outputs or other relationships between the products. For example, allocation factors might be based on the mass of the co-products or on their economic value.

Once the inventory has been modelled and the exchanges with the environment, also called elementary flows, are compiled in the LCI for all the substances that were considered, the substances can be classified to the impact categories that are to be covered. An overview of common impact categories is given in Figure 3-1. Next to the impact categories usually a distinction is also made between three AoPs: human health, natural environment and natural resources. It is at the AoPs that the actual damages occur. The impact modelling tries to reflect the environmental mechanisms that link emissions and extractions of individual substances to the AoPs. The impacts can be assessed at the midpoint level or at the endpoint level. The environmental mechanism linking substances and an impact category can be different before the midpoint for the various substances, however after the midpoint the mechanism linking the impact category and the AoP is the same for all substances belonging to the same impact category. Characterization factors (CFs) have been developed in the framework of various LCA methodologies. Once they have been established, CFs make it possible to easily quantify impacts. CFs are factors that are to be multiplied with elementary flows to quantify impacts either at the midpoint or the endpoint level. They reflect the impact per unit of elementary flow and are the higher the higher the impact caused by a unit of the elementary flow. The resulting impacts can be expressed in different units depending on the considered impact category, AoP or LCA methodology. At the midpoint impacts are often stated in terms of equivalents of a reference substance, e.g. CO₂-equivalents in the case of climate change.
According to Jolliet et al. (2003b), damages to the AoP natural resources consist in the reduced availability of the corresponding type of resource in the future, which is usually known as ‘resource depletion’. In principle resource depletion is a purely physical concept (van der Voet, 2013), i.e. by extracting natural resources less is left in nature. In contrast to this van der Voet describes ‘resource scarcity’ as a concept that also considers the exploitability of a resource.

Though for some impact categories it makes sense to consider local conditions at the point where emission or extraction takes place, this is usually not the case of the extraction of metals and its impact on the AoP natural resources. However, changes that occur over time, e.g. due to new metal resource discoveries or changed extraction rates, can be of relevance for determining impacts.
LCIA methods for abiotic resource extraction impacts on the AoP natural resources can be classified in different categories, considering some common characteristics. Lindeijer et al. (2002) and Steen (2006) classified the approaches in four categories: (1) Approaches based on energy or mass; (2) Approaches based on ratio of use to deposits; (3) Approaches based on future consequences of current resource extractions; and (4) Approaches base on exergy consumption or entropy production. The International Reference Life Cycle Data System (ILCD) Handbook (EC-JRC, 2011) classified the methods for abiotic resource use in four other categories: (1) Methods that use an inherent property of the material as basis for the characterization; (2) Methods that address the scarcity of resource; (3) Methods focused on water depletion; and (4) Methods that evaluate the depletion of resources at an endpoint level. Based on these previous classifications, the approaches that evaluate abiotic resource use were assigned to one of the following three groups: (1) Resource accounting methods based on inherent properties; (2) Effect on resource availability at midpoint level; and (3) damage to resource availability at endpoint level. In the following the discussion is limited to methods that are relevant for metal resources. Even on the same level the methods for quantifying impacts are very diverse, not only with respect to calculation methods but also with respect to the units. Also the number of metals considered differs between the various methods. For midpoint and endpoint methods a table showing which elements have CFs in which method can be found in Appendix 1.

3.1 Resource accounting methods

These methods are based on inherent properties of the extracted resource and are far from giving a direct quantitative value for environmental damages, but they are still able to provide results on the environmental sustainability of a product due to the philosophy of ‘less is better’.

They generally sum up all the resources consumed/used in the life cycle of a product. In order to provide results in single indicators, the resources are usually represented in common units (e.g. energy); otherwise the same information as given by the LCI would be obtained. Though they quantify how much is removed from natural resources they give no information on future availabilities as they do not relate extraction to available stocks in any way.

3.1.1 Mass

In MFA resources are typically aggregated based on mass. There are different MFA approaches. One of those is the Material Intensity Per Unit Service (MIPS) method (Ritthoff et al., 2002; Spangenberg et al., 1999). Though it is not usually classified as an LCIA method, it shows some similarities to LCA (Finnveden and Moberg, 2005). The MIPS method distinguishes between five resource categories: abiotic raw materials, biotic raw materials, movement of soil (agriculture and forestry, incl. soil erosion), water and air. These categories can be further divided into subcategories. A general guide to MFA was published by the OECD (2008).

3.1.2 Energy

Accounting for primary energy use along the production chain is a concept that was introduced in the 1970s (Boustead and Hancock, 1979). Energy-based resource accounting methods quantify the cumulative energy extracted from the natural environment (i.e. the cradle) to support the technosphere. They account not only for types of energy but also for materials by quantifying their energy content. The results are generated in a unit easily
comprehended by stakeholders (e.g., MJ). However, since metal resources typically have low energy value compared to their value for human use and their availability, the relevance of energy-based resource accounting methods for assessing the sustainability of metal resource availability is limited. These methods have been operationalised for LCA, for instance as the Cumulative Energy Demand (CED) for the ecoinvent database (Althaus et al., 2009). With the exception of uranium the CED method does not account for metals.

### 3.1.3 Exergy

By definition, the exergy of a resource or a system is the maximum amount of useful work that can be obtained from it (Dewulf et al. 2008) and is thus a measure of the quality of energy. Exergy analysis is usually used in industry to assess the efficiencies of processes. The cumulative exergy consumption (CExC), introduced by Szargut et al. (1988), is the exergy of the overall natural resources consumed in the life cycle of a product. Two exergy-based resource accounting methods have been operationalised for use with the process-based ecoinvent database (Swiss Centre for Life Cycle Inventories, 2010): the Cumulative Exergy Demand (CExD) (Bösch et al., 2007) and the Cumulative Exergy Extraction from the Natural Environment (CEENE) (Dewulf et al., 2007). The CExD and the CEENE are able to account for several resources. An important drawback compared to mass or energy based resource accounting methods is that the term exergy is not widely known and thus it is more difficult to communicate the results. There are a number of differences between the CExD and the CEENE methods, including the approach to account for metals and minerals. Whereas in the CExD method the exergy of the extracted ore is accounted for, the CEENE method only considers the exergy value of the mineral species containing the target metal. This means that the CExD exergy value increases with decreasing ore grade, while the CEENE value is independent of the grade. For the economic input-output U.S. 1997 database, the Industrial Cumulative Exergy Consumption (ICEC) is operationalised in Zhang et al. (2010a). Though exergy is a measure for the quality of energy it still suffers from drawbacks of the energy based methods with regard to metals. For example, the exergy of CuFeS$_2$ (chalcopyrite), a copper mineral, is 1530.3 kJ/mol and the exergy of pure gold is 51.1 kJ/mol (Valero Delgado, 2008). Also in terms of mass the exergy of the chalcopyrite (8338 kJ/kg) is higher than the exergy of pure gold (261 kJ/kg). Of course, the exergy contained in the mineral bonds also is valuable, but it does not reflect why society values metals.

### 3.2 Damage to resource availability at midpoint level

The following approaches evaluate the impacts along the environmental mechanism leading to damages in the AoP natural resources at the midpoint level.

#### 3.2.1 EDIP 97 and EDIP 2003

The EDIP method applicable for metal resources is described in Hauschild, Wenzel and Alting (1997) and Hauschild and Wenzel (1998). EDIP stands for Environmental Design of Industrial Products. The approach for ‘abiotic resource use’ in EDIP 2003 and EDIP 97 is the same, except that the values used in the calculations in EDIP 2003 are updated for the year 2004, while in EDIP 97 the data is from 1990 and 1991. Based on the reserves as defined by the USGS, the ‘abiotic resource use’ is evaluated by the scarcity of resources naturally available, which means that even though metals may not disappear after their use (unlike fossil energy), they will no longer be available in their natural deposits, but in other places (e.g. landfill).
To calculate the CF, the authors divided the procedure in two steps. In the first step, called normalization, the global production of a substance $i$ for a specific year (2004 in EDIP 2003) is considered, and this value is divided by the world population from that year. This is done to relate impacts to the impact of an average person. In the second step, which is called weighting in the EDIP literature, the reserve of substance $i$ is divided by the global production from the same substance $i$ in a particular year (2004 in EDIP 2003), providing the supply horizon of the substance, in years. Finally, the CF for the substance $i$ is calculated by the reciprocal of the product between the normalization and the so-called weighting factors (equation 1). It should be noted that according to the ISO 14044 on LCA weighting includes value choices. It could be argued that a factor based on a static reserve range does not depend on value choices. Taking a look at the equation, we can notice that the ‘global production’ can be erased from the equation, and effectively the CF are based solely on the reserves, but normalized for the World population from the year 2004. The method considers aspects of resource scarcity in that it explicitly takes into account reserves, i.e. currently exploitable resources. It is not considered, however, that the amount of reserves is variable and in fact has been increasing for many metals over time (USGS, 2010, 2002).

$$\text{CF}_i = \frac{1}{\left(\frac{\text{Global production}_{i,2004}}{\text{World population}_{2004}}\right) \times \left(\frac{\text{Reserves}_i}{\text{Global production}_{i,2004}}\right)}$$

With this approach the CF for gold is 87 persons/kg and for copper the CF is 0.016 persons/kg.

### 3.2.2 Abiotic Depletion Potential

Guinée (1995) developed the abiotic depletion potential (ADP) as an approach applicable for ‘abiotic resource use’. Later on the approach was implemented in the CML (Institute of Environmental Sciences – Leiden University) LCIA method by Guinée et al. (2002) and further updated by van Oers et al. (2002). These updates were implemented in the CML method in 2009 and 2010. The latest implementation of the CML method can be found on the CML website (http://www.cml.leiden.edu/software/data-cmlia.html). Due to relatively good data availability (for the reference years) a large number of metals and other elements are covered.

From a conceptual point of view the approach is similar to the approach used for resources in the EDIP methodology, as it is based on use-to-resource ratios. However, the extraction rate is retained in the final calculation. In the original implementation of the ADP method the remaining stocks are quantified in terms of so-called ‘ultimate reserves’. The ‘ultimate reserve’ is the total mass of the metal available on Earth, be it in the Earth’s crust, in the oceans or the atmosphere. For the stocks in the crust the average crustal abundance is multiplied with the mass of the crust up to a depth of 17 km (Guinée, 1995).

As the CFs are to be multiplied with elementary flows of metals in an LCA the amount of remaining resource is squared in order to take into account that extracting 1 kg from a larger resource is not equivalent to extracting 1 kg from a small resource, even if the use-to-resource ratio is the same. Equation 2 represents the generic calculation of the ADP of substance $i$.
expressed in kg of reference substance (Guinée et al., 2002). For metals the reference substance is antimony.

\[ ADP_i = \frac{DR_i}{R_i^2} \times \frac{R_{ref}^2}{DR_{ref}} \]  

(2)

With \( ADP_i \) the abiotic depletion potential of substance \( i \) (kg antimony/kg \( i \)), \( R_i \) the ‘ultimate reserve’ of substance \( i \) (kg \( i \)), \( DR_i \) the extraction rate of substance \( i \) (kg \( i \) yr\(^{-1}\)), \( R_{ref} \) the ‘ultimate reserve’ of the reference substance (kg antimony) and \( DR_{ref} \) the extraction rate of the reference substance (kg antimony yr\(^{-1}\)). According to the updated version of van Oers et al. (2002) the ADP for copper is \( 1.37 \times 10^{-3} \) kg antimony/kg copper and the ADP for gold is \( 5.20 \times 10^{-1} \) kg antimony/kg gold. If reserve base data is used instead of the ‘ultimate reserves’ the ADP of copper is \( 2.50 \times 10^{-3} \) kg copper/kg antimony and the ADP for gold is \( 3.60 \times 10^{-1} \) kg antimony/kg gold.

In the ADP approach resource extractions are related to the total amount of the metal in nature and thus the depletion impact is represented. Even though global annual extraction is considered the scarcity aspect is not prominent as ‘ultimate reserves’ are virtually infinite on the human time scale. The ADP approach was criticized by Müller-Wenk (1998), because the amount of ‘ultimate reserves’ could satisfy human consumption for ‘millions of years’, which would imply that there is no scarcity issue. Moreover, the approach lacks consideration of quality aspects of the resource. Already earlier Guinée (1995) had remarked that what was relevant were the reserves which can eventually be extracted, called ‘ultimately extractable reserves’, likely to be very different from the ‘ultimate reserves’. Guinée (1995) implicitly assumed ‘the ratio between the ultimately extractable and ‘ultimate reserve’ to be equal for all resource types.’ With the updates to the ADP approach, alternative CFs are available which use reserves or the reserve base as defined by the USGS (USGS, 2010) in the reference year 1999 instead of ‘ultimate reserves’. Guinée (1995) implicitly assumed ‘the ratio between the ultimately extractable and ‘ultimate reserve’ to be equal for all resource types.’ With the updates to the ADP approach, alternative CFs are available which use reserves or the reserve base as defined by the USGS (USGS, 2010) in the reference year 1999 instead of ‘ultimate reserves’. For resource depletion the ILCD (EC-JRC, 2011) recommends the use of the CML method at midpoint, in particular the alternative CFs using reserve base. In addition, it is advised to perform a sensitivity analysis using reserves and ‘ultimate reserves’. It should be noted that the USGS has discontinued the reporting of reserve base statistics due to lack of data (USGS, 2010).

### 3.3 Damage to resource availability at endpoint level

The approaches that evaluate actual damages are typically based on the quantification of the effort needed to produce the metal in the future.

#### 3.3.1 Eco-Indicator 99 and Impact 2002+

The approach for metal resource use in Eco-indicator 99 (Goedkoop and Spriensma, 2000) and Impact 2002+ (Jolliet et al., 2003a) is based mainly on Müller-Wenk (1998). The decrease in resource concentration due to extraction is modelled and evaluated by the concept of surplus energy, i.e. the difference between the energy needed to extract a resource now and at some point in the future.

For metals, the authors considered geostatistical models published in literature in order to evaluate the relationship between availability and quality. As an approximation at higher ore grades, it was assumed that the logarithm of cumulative amount of minerals mined (\( Q \)) was linear to the logarithm of the ore grade (\( g \)) (equation 3). The slope (\( m \)) of the curve between
the cumulative amount of minerals mined and ore grade is of key interest: The constant \( c \) does not play a further role in the determination of the CFs.

\[
\log Q = c - m \times \log g
\] 

(3)

In Figure 3-2 we can see that for a metal with a steep slope (e.g. chromium) the factor with which the ore grade changes when a certain amount is mined is smaller than the factor with which the ore grade of copper changes when the same amount is mined. The slope \( m \) is used to calculate the ore grade in the future when the total amount mined is five times the value of today. The assumption is that the energy requirement needed to extract, grind and purify an ore goes up as the grade goes down. The difference in the energy requirement per kg of metal between today and the future is used as CF in the Eco-indicator 99 method.

\[\text{Figure 3-2 Slope of the cumulative amount of mined versus the ore grade (taken from Müller-Wenk (1998), originally from Chapman and Roberts (1983).}\]

According Goedkoop and Spriensma (2000) the damage factor is 36.7 MJ surplus energy/kg copper in ore. As there are no damage factors available for precious metals, a comparison with gold is not possible. In this method the decrease in quality and quantity of metal stocks are considered, which as such are purely physical characteristics of the stocks. Expressing the impact as an increased need in energy requirements, however, links these decreases to an issue directly relevant for society.

3.3.2 ReCiPe

ReCiPe is a Dutch LCIA method created in 2008, which combines the scientific efforts of several institutes. The main information can be found in their report (Goedkoop et al. 2009). This method provides indicators at two levels: midpoint and endpoint. The midpoint indicators for ‘abiotic resource use’ from ReCiPe will be discussed together with the endpoint indicators in this subsection.

The approach of ReCiPe for metals and minerals focuses on the depletion of deposits, instead of individual commodities. Thus, it is taken into account that in many cases several metals can be recovered from the same deposit. Eventually, the CFs are still provided for the
elements as this is necessary to calculate impacts from LCIs for individual products. Curves for the evolution of the so-called value weighted grade of a deposit type are produced based on data published by the USGS for deposit modelling (Cox and Singer, 1986) covering mined and unmined deposits worldwide.

ReCiPe provides midpoint and endpoint indicators for metals. The damage to the AoP natural resources due to the extraction of a certain mineral is evaluated by the additional costs society has to pay due to this extraction, and it is expressed in US$ (present value in 2000). For calculating this additional cost the overall decrease in ore grade of the deposits containing a specific metal, the current global production of the metal and current mining costs are considered. The current mining cost is one fixed value independent of the deposit type. CFs for indefinite time horizons are calculated by adding up the additional costs per year and employing discounting. For this calculation a change in the amount of commodity produced per year is not considered.

The CFs for the midpoint indicators are calculated by an equation similar to the equation from the endpoint indicator (except by the exclusion of some constant factors) and then normalized to the value obtained for iron. Thus at midpoint the CFs are expressed in iron equivalents. Similar to surplus energy method quantity and quality decreases are considered. At the endpoint relevancy of these decreases for society is made explicit by expressing damages in terms of economic cost. Nuclear energy (uranium) is considered together with other metals. With a discount rate of 3% the endpoint CF for copper is 3.06 $/kg and the endpoint CF for gold is 5006 $/kg.

### 3.3.3 Sustainable Process Approach (EPS2000)

The sustainable process approach is an endpoint method implemented in the Environmental Priority Strategies for product development (EPS2000) (Steen, 1999a, 1999b). In the EPS2000 methodology the approach is implemented for metals and minerals, fossil energy, nuclear energy and atmospheric resources. Alternative factors for some metals are available in Steen and Borg (2002).

As for the other impact categories in EPS2000, the idea is to quantify the willingness-to-pay for restoring damage done to the so-called safe-guard subject. The willingness-to-pay is expressed in so-called Environmental Load Units (ELU). In the case of abiotic stock resources, not only the present generation but also future generations are included; therefore the willingness-to-pay is calculated based on hypothetical so-called sustainable processes which could produce resources like those extracted today once these are depleted. Thus, the method can be classified as one based on future consequences of current abiotic resource extraction (Steen 2006; Lindeijer et al. 2002). The calculated costs include direct production costs and external costs due to emissions and resource use. The sustainable processes are assumed to produce resources from average rock (most elements and gravel), from seawater or air. The sustainable processes are further optimized, e.g. by using electricity from solar energy and wood as an energy source. For copper a CF of 208 ELU/kg and for gold a CF of 1,190,000 ELU/kg are reported for EPS2000.

The sustainable process approach has been criticized for its rather long time horizon and the many assumptions associated with it (Müller-Wenk 1998; European Commission 2011). Thus, even though the method attempts to quantify the cost for future generations, this relevancy for society might be questioned.
3.4 Recent developments
A recent extension of the ADP method from Schneider et al. (2011) includes the stocks of metals which have accumulated in the technosphere reflecting the possibility of more extensive urban mining in the future. Vieira et al. (2012) evaluates the resource depletion of metals based on the global ore grade information at midpoint level. The data basis is similar to the one used in the ReCiPe method, but the evolution of the copper ore grade is modelled directly instead of the evolution of a value weighted grade. For now the method provides CFs for three different types of copper deposits.

3.5 Discussion
It seems prudent to give a short overview over the mining process, before discussing the viability of the various methods, so that the origin for some of the criticism will be clearer to the reader. Figure 3-3 illustrates schematically the steps required at mining sites to produce concentrates, which can be further treated to obtain metals, from a polymetallic ore.
Many ores are mined with the purpose to recover more than one metal. This is in particular true for zinc and lead, which are often co-products at mining operations. The ore grade is the concentration of a valuable metal in the ore. Extraction of the ore containing the valuable minerals can be done by open pit (OP) or underground mining (UG), depending on the circumstances. The use of UG or OP mining methods has an impact on the resources required for the mining. Additionally in open pit mining often huge amounts of overburden (i.e. waste rock that is located above the mineral deposit) have to be removed. Non valuable minerals (gangue) are closely mixed with the valuable minerals. To separate the gangue and the different valuable minerals from each other it is first necessary to liberate the minerals by size reduction (comminution). The actual separation often is done by flotation methods. Commination and concentration are often referred to as beneficiation. With decreasing ore grade more ore has to be mined and processed, which increases the auxiliary requirements. The concentration of the valuable metal in the resulting concentrate streams is fairly independent of the grade in the mined ore and thus subsequent processing steps may be deemed independent of the initial ore grades. Some metals occurring at lower concentrations are not usually recovered in a separate concentrate, e.g. silver is recovered with the lead concentrate. The non valuable minerals and also part of the valuable minerals end up in a waste stream (tailings). The recovery efficiency of beneficiation is equal to the fraction of the metals entering the processing that are retained in the appropriate concentrate stream. The higher the recovery efficiency the less ore needs to be mined and processed to recover 1 kg of metal in concentrate and the less valuable metal ends up in the waste stream.

Table 3-1 gives an overview over the most important LCIA methods discussed so far. The resource accounting methods will not be discussed here any further as they do not give any information regarding possible future availabilities.

The remaining methods have their merits, but also suffer from several drawbacks to varying extent. These drawbacks are related to conceptual issues, the implementation, the data sources employed and the metals covered.

Some methods produce their CFs based on the production of metals from average crust concentrations. This is questionable because average crust concentrations are unlikely to be used as resources in the human time frame. In the ADP method of CML the use of the ‘ultimate reserves’ concept is only a proxy for the actually extractable resources. Considering the possibility of a bimodal distribution for some metals, this might not be a good approach. Moreover, some elements/metals are relatively abundant in the Earth’s crust, but only rarely occur in more concentrated forms. Rubidium, for example, is more abundant in the crust than other metals, such as copper, but rubidium does not form minerals of its own and is thus only obtained as a by-product (Buttermann and Reese, 2003). The alternative to use reserves also does not seem advisable because they are more dependent on the economic requirements of mining companies than on actual availabilities of the metal. For example, the production to reserve ratio for zinc has been approximately 1/20 for half of the 20th century (Wellmer and Wagner, 2006). The reserve base is a more a suitable option if such data is available. Still a simple use to resource based method does not take into account changes in production (e.g. increase and eventually decline) (Sorrell et al., 2009) nor the change in the quality of resources with continuing extraction. In addition it is questionable whether the abundance of metals is a relevant issue. In fact, Tilton (2001) claims that there is “a growing consensus ... that the fixed stock paradigm should be retired in favour of an alternative that focuses on the opportunity costs of finding and extracting mineral resources” (p. V-4). The biggest
Table 3-1 Overview of methods used to assess metal resource use.

<table>
<thead>
<tr>
<th>LCIA Method</th>
<th>Basis</th>
<th>Data requirements for CF</th>
<th>Data quality</th>
<th>Metals covered</th>
<th>Resource and demand aspects</th>
<th>Co-mining</th>
<th>Timeframe</th>
<th>References</th>
</tr>
</thead>
<tbody>
<tr>
<td>1. Based on mass (or energy) of the inventoried material (resource accounting method)</td>
<td>MIFs, ...</td>
<td>Mass of material</td>
<td>No-specific CF for the natural resource.</td>
<td>All</td>
<td>Neither quantity, nor quality of remaining resources considered. Demand not considered.</td>
<td>NA</td>
<td>NA</td>
<td>Ritthoff et al., 2002</td>
</tr>
<tr>
<td>2. Based on exergy (or entropy) of the inventoried material (resource accounting method)</td>
<td>CEExD, CEENE, ...</td>
<td>Exergy of material</td>
<td>Exergy of minerals/ores required.</td>
<td>Major/most metals covered.</td>
<td>Neither quantity, nor quality of remaining resources considered. Demand not considered.</td>
<td>NA</td>
<td>NA</td>
<td>Dewulf et al., 2007; Bösch et al., 2007</td>
</tr>
<tr>
<td>3. Based on use to availability ratios of the metal (midpoint)</td>
<td>EDIP 97/ EDIP 2003</td>
<td>Population/Reserves</td>
<td>Data readily available.</td>
<td>Reserve data should be updated regularly.</td>
<td>CIs available for most important metals.</td>
<td>Demand could be considered to be reflected in population. Reserves considered.</td>
<td>NA</td>
<td>Short.</td>
</tr>
<tr>
<td></td>
<td>CML</td>
<td>Use/Total metal content in hydrosphere, atmosphere and Earth's crust</td>
<td>Data readily available.</td>
<td>Especially demand data should be updated more frequently.</td>
<td>Almost all metals covered.</td>
<td>Quantity of one resource category considered, depending on which CML method is used. Demand considered.</td>
<td>NA</td>
<td>Short for reserves. Very long for ultimate reserves</td>
</tr>
<tr>
<td>4. Based on future consequences of extraction of the metal (endpoint)</td>
<td>Surplus energy method</td>
<td>Extra energy required for mining per kg of metal in future</td>
<td>Extensive data requirements.</td>
<td>Original source not traceable, Mining/resource specific data from two books from the 1960s from Chapman &amp; Roberts and de Vries.</td>
<td>Available for most important metals, however no Cf available for PGMs, Ag and Au.</td>
<td>Quantity and quality of remaining resources considered. Cumulative demand taken into account.</td>
<td>Not explicitly taken into account.</td>
<td>Longer term impact of current extraction.</td>
</tr>
<tr>
<td></td>
<td>ReCiPe</td>
<td>Marginal cost increase of mining caused by metal extraction</td>
<td>Extensive data requirements.</td>
<td>Old, incomplete data. Mining/resource specific data from USGS database for deposit modelling published in 1997, based on previously published data. Mining cost data from two mines.</td>
<td>CIs available for most important metals.</td>
<td>Quantity and quality of remaining resources considered. Current/future demand taken into account.</td>
<td>Taken into account by value based allocation. However, procedure leaves room for improvement.</td>
<td>Prolonged impact of current extraction.</td>
</tr>
<tr>
<td></td>
<td>Sustainable process</td>
<td>Environmental impact from producing ore like resources from bedrock</td>
<td>Extensive data requirements.</td>
<td>Bedrock data sufficient quality. Sustainable process data obviously based on assumptions to some extent.</td>
<td>CIs available for almost all metals.</td>
<td>Quality of currently used resources and quality of bedrock considered. Demand not considered.</td>
<td>Taken into account by number-based allocation. Very long, timeframes, relevancy debatable.</td>
<td>Steen, 1999a; Steen, 1999b; Steen and Borj, 2002</td>
</tr>
</tbody>
</table>

Abbreviations: CF = Characterisation factor; NA = Not applicable; PGMs = Platinum group metals
advantages of these methods are that they are still relatively simple and data are quite readily available, which are quite reliable for what they are supposed to represent. This makes the calculation of CFs for many metals possible.

The advantage of ore grade based methods is that they take one aspect of the change in quality of resources into account explicitly and with regard to more reasonable time frames. In the surplus energy approach the time horizon is defined by the cumulative amount of metal resources extracted. According to Müller-Wenk (1998) this approach was chosen because it is difficult to predict the evolution of the annual extraction, which had declined between 1981 and 1995 for a couple of metals. However, this same observation would also mean that the surplus energy for the different metals would not be calculated for the same point in time and could be for example within 100 years for metal A and within 150 years for metal B. How could a higher CF for metal B than for metal A be interpreted in this case?

In the ReCiPe approach the damage assessment quantifies the additional cost to recover the annual global production of a metal induced by the extraction of 1 kg of the metal. At the endpoint CF values for an indefinite time horizon are calculated, but they do not take into account that global annual production might change. If this is not considered, why is the time horizon extended at all? One could argue that the growth might be included in the employed discount rate, so that the actual discount rate would in fact be higher. As long as extraction growth would be the same factor for all metals and only the relative values of the CFs are considered this does not influence the result.

With regard to the supply side, the extension to the ADP method suggested by Schneider et al. (2011) is the only method, which tries to take into account that future generations could also satisfy their needs by utilizing stocks that have accumulated in the technosphere in the past. Urban mining may be one of the factors influencing future demand for primary metals. From a societal point of view it seems to make sense. Future generations will have less of a problem with decreased availability of natural stocks if there are larger amounts of metal available in stocks in the technosphere. The CF could be improved by considering the global dissipation rate in the numerator instead of the global primary production, in this way the use to resource ratio would also again represent the aspect of time till the considered stock is depleted, which was contained in the original approach. Of course at this point, the AoP is not actually natural resources any more in the strict sense.

Apart from the sustainable process approach the marginal cost approach is the only one explicitly taking into account co-production. The implementation in ReCiPe, however, does lead to inconsistencies, which can be easily observed for the hypothetical case of a deposit type that only contains two metals and is the sole source of these metals (see Appendix I for details).

In addition the metal prices which are used to calculate the value weighted grades are not averaged over several years, which is disadvantageous considering the volatility of the metal price market. Another issue with the implementation in ReCiPe lies in the determination of the costs. It was chosen to use one fixed cost value per kg of ore for all metals. Though real data are used for the costs, these data are based on a typical open pit copper mine and one specific very large copper-gold mine (Grasberg), which provides the majority of the ore (Freeport-McMoRan Copper & Gold Inc., 2011). This does not seem realistic given that some metals are mostly mined UG, while others are mostly obtained via OP mining methods, for
which costs per kg of ore tend to be lower (Crowson, 2003). Similarly, the size of a mine (Crowson, 2003; Mutmansky et al., 1992; Nilsson, 1992) and a number of other factors can also impact the costs. The inclusion of the most important factors could make the results more representative of actual mining practice. In the surplus energy method at least different energy demands for the considered metals are taken into account.

Another point related to mining practice which is typically overlooked is the recovery efficiency which differs from metal to metal and from processing route to processing route. This is relevant as, e.g. in the case of ReCiPe the calculations are done based on the metal in the ore, which makes sense because this is also what is typically inventoried in LCI as the system boundary is located at the point of extraction. Yet, further on in the derivation of the CFs global primary production data is used, which is typically mine production data, i.e. only takes into account the metal content of the mine products, which are typically concentrates, while part of the metal is going to the tailings stock.

It would be useful to also crosscheck results with mining practice, e.g. if the choice of a working point affects the results of a marginal approach, it should be similar to the actual current grades. In ReCiPe the working point for determining marginal cost increases decreases in function of extraction is chosen at the median of the total extractable amount of a deposit type in their database. More particularly at 0.5 × the constant of the regression line of value weighted yield versus value weighted grade (see Goedkoop et al. (2009) for details). This is stated to not have an effect on the relative differences in the results. This is only true, however, if one does not consider the possibility that the different deposit types might be mined to different degrees.

The implementation and data used in the sustainable process approach is inherently based on many assumptions due to the long time horizon. The data used in the surplus energy approach are outdated and not traceable. In ReCiPe deposit databases from the USGS are used, similar but newer versions are employed in the method described recently by Vieira et al. (2012). These databases contain quite extensive information on discovered deposits, which have been mined in the past, are still mined or have not yet been developed. The (potential) mining method or recovery route are not recorded in those data files, but could maybe be estimated from available data or other sources.

The reserve data used in the ADP method of CML in principle can be easily kept up to date as these data are regularly updated by the USGS. However, reporting of reserve base values was discontinued by the USGS (USGS, 2002) for the time being due to lack of data.

The number and type of metals covered depends a lot on the method and the data used. The CML method can be considered to be not very data intensive and the required data is mostly easily available for many commodities. In other methods the number of metals covered is much lower due to data constraints. Nevertheless, there is also the question if actually all metals should be treated in the same way, especially by the ore grade based methods, if the nature of the resources and the mechanisms that govern their supply are fundamentally different.
4 Objectives

The objectives of this PhD are related to two aspects of metal use sustainability:

On the one hand the work that will be presented deals with methods used to assess metal use impacts in the AoP natural resources. The question of preserving metal resources is quite distinct from the question of protecting human health and the functioning of ecosystems or preserving other natural resources like water or even preserving landscapes. Society primarily has an interest in metal resources because they can be used to make objects to fulfil various functions. To this end these metals have to be extracted. Of the LCIA methods discussed in the previous chapter those based on ore grade changes most explicitly take into account future extraction of metals. They intend to model the effect that current metal extraction has on economic costs or energy requirements of future metal extraction. The question arises to what extent the models and their implementations reflect mining and mineral processing as they occur in the real world, as the methods are in parts relying on old data, and on assumptions that might be too simplifying. With this in mind the main goal regarding LCIA methods for metal use impacts in the AoP natural resources was to examine current LCIA methods in the light of historical mining and mineral processing data and determine to what extent improvements were possible. The following further questions were to be tackled: Is it possible to develop CFs on the basis of mining data? How do CFs derived from such data compare to the CFs of existing LCIA methods? To which extent is the assumption of the ore grade based LCIA methodologies realistic that a lower ore grade means that the effort to produce the metal is increased? What are the drawbacks of using real world mining data? To tackle these questions the Raw Materials Database (Raw Materials Group, 2013, 2012), annual and sustainability reports from mining companies as well as other literature sources were to be used.

On the other hand the overall resource use for the production of metal products that are used in clean energy technologies is assessed. This other part of the PhD work was dealing with the resource consumption of advanced materials made from metals. These materials are frequently used in clean technologies with the aim to reduce emissions and resource consumption. For example, photovoltaic technology is used to harvest the energy from the sun, which is an amply available renewable energy source. Though the use of batteries as such does not contribute to the reduction of environmental impacts, they play an important role in reducing direct dependence of transportation on fossil fuels. As the processing of metals is typically resource intensive, even more so if high purity material is needed as is the case for advanced materials, it needs to be substantiated that these clean technologies in fact contribute to the reduction of emission and resource impacts. LCA is an important tool to verify the overall environmental impact of products, but as mentioned in the introduction LCI data for advanced materials is often not available or not based on detailed data from producing companies. Thus, the main goal was to establish process and LCI data for two advanced materials used in clean technologies on the basis of data from companies actually producing these materials in order to improve the modelling of the life cycles of the full applications. Two materials were selected: germanium wafers and cathode powders. The germanium wafers are applied in high concentration photovoltaic (HCPV) systems, which convert solar
energy to electricity. The cathode powders on the other hand are used in Li-ion batteries for energy storage. Other important questions were: How can the resource consumption for these intermediate products be related to the full application? Can a meaningful functional unit be chosen? What do the resource fingerprints of the advanced materials look like? Which inputs from the technosphere and which processing steps contribute most to the overall resource consumption? To answer these questions detailed data from producing companies was to be collected for the production of these advanced materials. For background data the LCI and process database was an important source of information.

To reflect that the presented work covers two more or less distinct aspects of the resource sustainability of metal use, it was divided into two parts. Part II deals with LCIA methods and the possibility to use data from existing mining and mineral processing operations to derive CFs. Chapter 5 focuses on ore grade changes for a number of non-ferrous metals and how they compare to CFs from existing LCIA methodologies. Chapter 6 deals with the mining technology and differences in effort requirements related to technology and ore grade. Part III covers the case studies on advanced materials. The germanium wafer study is discussed in chapter 7 and the Li-ion cathode materials are the topic of chapter 8.

In conclusion of this PhD Part IV discusses the results obtained in Part II and Part III (chapter 9), presents a broader look at the elements relevant for metals and their future availability (chapter 10) in order to frame the further development of the assessment of metal use and suggests further research options regarding the case studies (chapter 11).
Part II
Mining data base supported analysis of metal resource use life cycle impact assessment methods in the area of protection ‘natural resources’
Quantifying the impacts of primary metal resource use in life cycle assessment based on recent mining data²

Abstract. The quantification of impacts in the abiotic resource category in LCA is still controversial. However, this is a pertinent issue because of the growing dependence of our industrial society on these resources, particularly on metal resources. One of the important shortcomings of the existing assessment methods used today is that characterization factors are not based on actual mining practice data. In this chapter, a new characterization factor derived from recent (1998-2010) and representative (more than 50% of global primary metal production were analysed) mining data was established for nine metals: copper, zinc, lead, nickel, molybdenum, gold, silver, platinum and palladium. The quantification of this new characterization factor is based on the annual increase in mass of ore required per unit mass of metal in the ore. This quantification relies on the concept that the mining of resources is threatened not by lack of ores but by changing ore characteristics, e.g., the percentage of metal in the ore, mineral type and location. The CFs determined in this study ranged from below 0.1 kg ore kg⁻¹ y⁻¹ for zinc to more than 15,000 kg ore kg⁻¹ y⁻¹ for gold. These results indicate that in 1999, 370,000 kg of ore was required per kg of gold in the ore, whereas in 2008, 530,000 kg of ore was required per kg of gold in the ore (an increase of approximately 4% per annum). When comparing these results with traditional LCIA methods, it was found that in all but one method gold, palladium and platinum have the highest CFs among the nine metals. In all methods based on ore grade changes lead and zinc are the metals with the lowest CFs. However, an important difference in the proposed method is that it assigns higher relative values to precious metals. This suggests that the supply of precious metals may be under more pressure than indicated by other methods, which in the framework of the proposed method implies greater efforts in mining and mineral processing. There is still scope for improvement of the proposed method if more data become readily available.

Keywords: life cycle impact assessment; abiotic resources; metals; mining; ore grade

5.1 Introduction

With some exceptions, primary metal production has grown over the years (Kelly and Matos, 2010) due to increasing welfare, the increasing number of people on the planet and technological applications that rely more and more on special metals, which previously were used only rarely. Although metals are a finite natural resource, increased recycling may offer a way to reduce the need for primary metal production; however, as long as demand for these metals continues to rise (as predicted by various forecasts, e.g., (Angerer et al., 2009; Backman, 2008; Frondel et al., 2007; Halada et al., 2008; U.S. Department of Energy, 2010)),

recycling cannot completely supersede primary metal production. Therefore, currently available resources must be managed carefully.

Based on the observation that existing LCIA methods for evaluating impacts on abiotic metal resources require improvement, especially with respect to the nature of the data used, this chapter assesses the viability of an alternative indicator for the assessment of metal resource impacts that is based on current mining data. None of the LCIA methods discussed in Part I use recent data from actual mining operations for the development of their CFs. If the CFs can be based on actual mining data and incorporate relevant aspects of mining practice, then they will reflect actual mining practice to a higher degree than the existing methods and can therefore be of added value in the characterization of metal resource impacts. As discussed by Mudd (2009a, 2009b), decreasing ore grades can be observed for a number of metals and regions. In addition, decreases in ore grades can potentially lead to increased efforts in mining with subsequent increases in environmental impact (Mudd, 2010, 2009a, 2007a, 2007b; Norgate and Haque, 2010; Norgate and Jahanshahi, 2010; Norgate, 2010). Therefore, there are indications that the ore grade is of more immediate concern than the abundance of the metal in terms of mass. This proposition is consistent with the opportunity cost paradigm (Tilton, 2001), which suggests that the depletion of low-cost deposits forces society to exploit deposits that require more effort, including deposits with lower grades. Eventually, increasing prices will result in decreased demand, even in cases where resources are still available in the ground.

The focus of this chapter is the change in the amount of ore required to produce metals, which may later be combined with the utility requirements for metal production (energy and auxiliaries) to formulate an overarching indicator for evaluating changes in the physical accessibility of primary metals. Conceptually such an indicator would reflect the relative extent to which current primary metal consumption affects efforts in mining and mineral processing for various metals, excluding metals with high concentrations in deposits (e.g., iron, aluminium and manganese), metals that are not mainly recovered from hard rock (e.g., Li) and metals that are principally by-product metals (e.g., In, Ge and Ga). By-product metals were excluded because their supply is largely governed by the demand for the metal they are associated with (Hagelüken and Meskers, 2010). The reasoning behind the development of this indicator is that extraction of a specific metal may cease long before the potential resources have been depleted due to the increasing efforts required for mining (Tilton, 2001).

5.2 Metal mining and energy requirements
For most metals, the ore contains only small amounts of the metal of interest. Therefore, large amounts of ore have to be mined and treated to produce 1 kg of the refined metal. For example, the ore grade of copper is typically below 1% mass (Crowson, 2012). Therefore, at least 100 kg of ore is required to obtain 1 kg of copper. The amount of ore that is processed is an important factor for determining the energy and material requirements of a mine. Other relevant factors include the grain size (the smaller the grain size, the finer the ore has to be ground), the depth of the mine (ventilation for UG mines and hauling distances), distance to ports and available energy sources in the vicinity, and mineral composition (the ease of separation of valuable minerals and the hardness of the rock) (Cochilco, 2009; Davenport et al., 2002; Ennis et al., 2008; Krauß et al., 1999; Lund et al., 2008; Marsden, 2008; Norgate and Jahanshahi, 2011; Norgate and Haque, 2010). Norgate and Haque (2010) collected life-cycle inventory data for bauxite, iron ore and copper concentrate. Those data showed that
most of the energy used during mining and beneficiation is required for loading, hauling and comminution. For UG mining, ventilation is also very important. Transport energy requirements can also be significant. In previous publications, Norgate and colleagues (Norgate and Jahanshahi, 2010; Norgate, 2010) estimated the impact of decreasing ore grades and grain size on the energy requirements and global warming potential (GWP) that was associated with primary metal production.

### 5.3 An indicator framework for metal resources using mine-level data

In the context of this study, specifically chapter 5 and chapter 6, mining refers to the operations performed to remove ore bearing rock from the ground (drilling, blasting, ...) including supporting operations like ventilation as well as loading and hauling the material above ground. When referring to mineral processing this in principle includes all further operations required to obtain a refined metal. The focus lay on a number of metals that are mined and usually undergo more extensive further treatment close to the mine as their concentrations in ores are usually too low to warrant direct shipping.

To identify the relative differences in impacts on resources caused by the extraction of various metals, the changes in the effort required to produce a metal are to be quantified. This approach is based on the concept that demand and therefore production are assumed to result in a decrease in ore resources that are easily accessible and processed. However, changes in technology may offset these cost-increasing effects; e.g., mining operations that use hydrometallurgical processing typically process lower grades (Crowson, 2003) but are also described as having lower costs (Kordosky, 2002). In a simplified model, two factors are considered in the calculation of the resources required per kg of metal extracted: 1) the amount of ore that needs to be processed per unit mass of metal, which is an aspect of ore grade, and 2) the amount of utilities required for mining and processing, which are in parts dependant on the amount of ore that needs to be processed (such as electricity used during beneficiation). In this chapter only the changes with respect to the first factor of ore processing requirements will be quantified. The annual change of ore required per kg of metal will serve as CF. The assessment of the impacts of technology on energy requirements per kg of ore and per kg of metal presented in chapter 6 may contribute to the future development of CFs that take into account the changes in the amount of utilities required per kg of metal.

For the first factor, time-dependent ore grade data are required. Ore grade is a suitable starting point because although only limited ore grade data are available, there are at least some data that have already been compiled and are available for the quantification. Because the ore grade is inversely related to the efforts required for mining, the inverse of the ore grade, i.e., the amount of ore treated per kg of metal in the ore, will be examined. Therefore, it is possible to directly relate the resulting parameters to mining efforts. The annual increase in ore requirements is affected by the annual production of mined resources and their grades. The proposed ore requirement indicator will be based on recent and traceable mining data and will take into account effects of co-mining and distinguish between metal content in the ore and the amount of metal recovered at the mine.

#### 5.3.1 Data sources

Processing and production data of the mines were taken from the database Raw Materials Data (RMD) provided by the Raw Materials Group (Raw Materials Group, 2012). This
database contains data for individual mines, and includes metal production, ore production and metal grades of the processed ore on an annual basis. Ore production data are provided from 1998 onwards. In addition, data are available on the mine type (e.g., UG or OP), metals of economic importance for the mine (1-5 metals), percentages recovered during beneficiation and the country where the mine is located. The availability of such databases means that updating the method with new data on a regular basis is feasible. However, there are some mines for which the required data were not available for all years on the level of the individual mine, but for which the data were available on an aggregated level for a group of mines. The mines in the same group were typically located in the same area and operated by the same company. Data related to three operations were modified from the RMD values based on either company reports (Impala Platinum Holdings Limited, 2003; Orsu Metals Corporation, 2009; Polymetal, 2011) or on data from the USGS (Levine, 2001). For over 1000 mines and mine groups, sufficient information was available to quantify the ore requirements per unit mass of metal processed and the ore requirements per unit mass of metal recovered. The total annual metal production represented by the analyzed data covered more than 50% of global production with some variations for the different metals, when using global production values reported by the USGS (Kelly and Matos, 2010) as a reference. To ensure a minimum level of representativeness overall only metals whose production covered at least 50% of world production were taken into account. Nevertheless, there may be some bias in the data because data availability is dependent on the world region and is much lower for the non-western world, e.g., China. Data were analyzed for the period from 1998 to 2010. For the period prior to 1998 insufficient data was available.

For the allocation of ore production to co-produced metals metal price data were collected from the USGS Mineral Commodity Summaries (USGS, 2009, 2005, 2003). The annual average metal prices were normalized to the year 2000 US$ by correcting the price values with respect to the U.S. consumer price index using data published on the website of the U.S. Bureau of Labor Statistics (U.S. Bureau of Labor Statistics, 2011). Although the U.S. consumer price index may not be the best indicator for calculating real developments in metal prices (Svedberg and Tilton, 2006), it is a suitable method for this study because the main focus is evaluating the price of one metal relative to other metals. The mean of the annual averages for the data was calculated over the period from 1998 to 2007.

Product LCIs were obtained from ecoinvent v2.2 (Swiss Centre for Life Cycle Inventories, 2010) and global mine production data were sourced from Kelly and Matos (2010).

5.3.2 Data processing

By using the non-aggregated data from RMD, it was possible to include the effect of co-mining via allocation on basis of economic value. It was assumed that a mine was producing a specific metal if the metal production data for that metal were available or if the metal was mentioned as being one of the metals of economic importance for the mine. For each mine, ore production, the mass of each metal in the ore and the production of each metal were determined on an annual basis.

As mines typically produce more than one metal, the ore production was allocated between the metals to take into account burden sharing. The overall value of the recovered metals was calculated by multiplying the 10-year average price of the metal with the metal production at the mine and then summing the values for all of the metals produced in the mine. The ore production was allocated to each metal by multiplying the total ore production by the fraction
of value of the metal relative to the overall value. More details on the data processing are available in Appendix 2 sections A.2.1 to A.2.4.

5.3.3 Quantifying changes in specific ore requirements
To ensure a minimum level of representativeness, the changes in the ore requirement were only determined for metals that had data for more than 50 % mass of the global annual mine production for the majority of years during the period from 1998 to 2010. For each metal and year the percentage of global annual mine production that was represented by the data was determined. In the following this percentage will be referred to as coverage. As reference for the global annual mined production data published by the USGS was used (Kelly and Matos, 2010) represented by the data from RMD. Coverage varied between the metals and was generally slightly lower at the beginning and end of the 1998-2010 time period. The coverage was high enough to include the following nine metals in the analysis: copper, zinc, lead, nickel, molybdenum, gold, silver, palladium and platinum. The maximum coverage for cobalt was below 50 % mass, and therefore, it was only considered a co-mined metal in the calculation of the overall value recovered at a mine (see section 5.3.2 and Appendix 2 sections A.2.3 and A.2.4).

Table 5-1 Summary of the basic concepts and main data used in the selected LCIA approaches for metal use.

<table>
<thead>
<tr>
<th>Method name</th>
<th>Basic concept</th>
<th>Main data used and procedure parameter determination</th>
</tr>
</thead>
<tbody>
<tr>
<td>CML</td>
<td>Use to resource ratios</td>
<td>Primary metal production from USGS (reference year 1999) as use and average concentration in crust × mass crust (up to 17 km) as resource + metals in sea water</td>
</tr>
<tr>
<td>EPS2000</td>
<td>Cost of producing ore from average rock</td>
<td>Average concentration in the continental crust used in the modelling of energy requirements</td>
</tr>
<tr>
<td>Eco-indicator 99</td>
<td>Future additional energy requirements for mining and processing. Increases in energy are due to the increased amount of ore to be processed when the grade has decreased</td>
<td>Slopes of the logarithm of cumulated metal production versus the logarithm of grade directly obtained from works by Chapman (table) and De Vries (graph) from the 1980s; coupled with energy requirements per unit mass of ore</td>
</tr>
<tr>
<td>ReCiPe</td>
<td>Marginal increase in future extraction costs due to current extraction; grade decreases when cumulative production increases; increasing costs with decreasing grade</td>
<td>Deposit data from the USGS database for deposit models ordered by specific ore value; cumulated metal value plotted versus specific ore value for each deposit type; weighted averages for the slope and intercept calculated for contained metals</td>
</tr>
<tr>
<td>This study</td>
<td>Annual change in ore required per kg of metal content</td>
<td>Mine data from RMD (1998-2010) used to relate ore requirements at the mine level to the produced metals; total allocated ore mass per unit mass of metal plotted versus the year for all relevant mines, and calculation of the slope</td>
</tr>
</tbody>
</table>

For each metal, the ore processed per unit mass of metal in the ore and the ore processed per unit mass of metal produced were plotted as the dependent variables against time. The time frame covered by the data is rather short and the data vary considerably from year to year. Therefore, only the average annual change in ore requirements for the current time period was
determined instead of developing a more specific model. The average annual change in ore requirements was calculated via simple linear regression and quantification of the coefficients of the independent variable time. Table 5-1 gives a short overview of the method proposed in this study and a selection of existing LCIA methods quantifying impacts on metal resources.

5.4 Results and discussion
In section 5.4.1, the CFs that are based on changes in specific ore requirements will be presented and discussed. These new CFs will then be compared to the CFs produced by existing methods in section 5.4.2. In Appendix 2 graphs depicting annual ore demands (unallocated/allocated) and coverage are available for each metal (Figure A.3 and Figure A.4).

5.4.1 Changes in specific ore requirements
Table 5-2 shows the results for calculating the slope of the regression curve of ore requirements versus time in years. Wide confidence intervals are reported for most metals. This is largely due to the high year to year variability in the calculated ore requirements and the limited number of years that are taken into account. The increases in ore requirements per kg of metal in the ore are used as the CFs for the resources and can be directly applied to the environmental metal resource flows, which are expressed in terms of metal in ore.

From Table 5-2 it can be observed that overall the lower the ore grade (reciprocal of values in column 5), the higher the values for the absolute annual change (CFs in column 2). The increase in annual change for gold was by far the highest of the analyzed metals, with more than 15,000 kg of additional ore required each year per kg of gold in ore. Variability in the data is also much lower for gold than for other metals. Because gold has a low ore grade, a small relative change in grade will result in high absolute changes in the ore requirements. Similarly, other precious metals (Pt, Pd, and to a lesser extent Ag) also exhibited high increases in the annual ore requirements. The slope for molybdenum is in the same order of magnitude as the slope of silver. An analysis of the data for molybdenum mines reveals that the data contain a higher fraction of mines producing no other metal than molybdenum in the second half of the time period than the first half of the time period. The values for zinc and lead are only approximately 0.1 kg of additional ore required each year, which were the lowest absolute increases observed in this study. The calculated slope for copper in ore was somewhat lower than that of nickel but higher than that for lead and zinc. The variability in the data of copper is much higher than for the other metals. In fact, there seems to be an increasing trend for copper ore requirements between 1999 and 2007 and then a decrease between 2008 and 2009, indicating that the interpretation of the slope is less obvious than for the other metals. The decrease observed in the later time period may be related to the financial crisis gaining momentum in late 2008. Similar effects were not clearly observed for zinc and lead.
Table 5-2 Results for the analyzed metals based on the allocation by value for the metal in ore.

<table>
<thead>
<tr>
<th>Metal</th>
<th>Average increase in ore requirements per unit mass of metal in ore^a (kg ore kg metal(^{-1}) year(^{-1}))</th>
<th>Average ore per metal in ore (kg ore kg metal(^{-1}))</th>
<th>Average increase/average ore per metal in ore (year(^{-1}))</th>
</tr>
</thead>
<tbody>
<tr>
<td>CF</td>
<td>Lower bound^b</td>
<td>Upper bound^b</td>
<td></td>
</tr>
<tr>
<td>Cu</td>
<td>0.32</td>
<td>-0.25</td>
<td>0.90</td>
</tr>
<tr>
<td>Zn</td>
<td>0.07</td>
<td>0.02</td>
<td>0.12</td>
</tr>
<tr>
<td>Au</td>
<td>18,005</td>
<td>15,477</td>
<td>20,533</td>
</tr>
<tr>
<td>Ag</td>
<td>44.4</td>
<td>16.3</td>
<td>72.6</td>
</tr>
<tr>
<td>Pt</td>
<td>5244</td>
<td>3,890</td>
<td>6597</td>
</tr>
<tr>
<td>Pb</td>
<td>0.09</td>
<td>0.02</td>
<td>0.15</td>
</tr>
<tr>
<td>Pd</td>
<td>2052</td>
<td>125</td>
<td>3979</td>
</tr>
<tr>
<td>Mo</td>
<td>29.4</td>
<td>10.3</td>
<td>48.4</td>
</tr>
<tr>
<td>Ni</td>
<td>0.92</td>
<td>0.37</td>
<td>1.47</td>
</tr>
</tbody>
</table>

^aThe median of the coverage was determined for each metal. Subsequently, for each metal those years for which coverage was more than 10 percent points below the median coverage for the metal were excluded from further analysis to limit the impact of varying coverage.

^bLower bound and upper bound of the 95% confidence interval of the slope coefficient

Table 5-3 contains some key figures reflecting the representativeness of the data and the results for the period from 1998 to 2010. The metals that were examined are listed in the first column. In the second column, the median of the coverage value over the 13-year time period is reported. This was determined as the coverage value for which for 50% of the years a smaller or the same value was calculated. It can be observed that there is some variation in coverage between the metals and that the nickel coverage in particular is rather low. The number of years that were included in this study is presented in column three and refers to the number of years that were taken into account in the linear regression analysis. The last two columns (four and five) show to what extent it was necessary to use recovery values that were only specific to the metal and not specific to the particular mine, with respect to the metal of interest in the ore (column four), and with respect to the total metal value recovered at the mines producing the metal (column five). Recovery is calculated as the ratio between the metal recovered in the product and the metal in the ore being processed. Mine-unspecific recovery refers to the use of recovery values that were not obtained from the production data of the mine in question, but from averages obtained from other mines in the database. The percentage of mine-unspecific recovery usage is that part of the total metal processed which was calculated based on the mine-unspecific metal recovery data. The percentage of mine-unspecific recovery usage in allocation expresses the percentage of the total value recovered at mines which was calculated from mine-unspecific metal recovery data. The metal specific recoveries, which were used instead of mine-specific recoveries, were calculated from the available data from the other mines (see also Appendix 2 Table A.3). With the exception of the platinum group metals (PGMs), a differentiation for recoveries was made with respect to the main metal produced at the mine. The reported medians were calculated based on the percentage (50%) of number of years which had a mine-unspecific recovery usage equal or below the median value. In contrast to the median of the coverage only the included years were taken into account. For instance, the values in columns four and five indicate that the values calculated for the total amount of molybdenum in ore may be less accurate than is the case for other metals and that the allocation in the case of nickel may not be a close reflection...
of the actual mine production. Another source of uncertainty is observed for the PGMs because only an overall PGM grade was available; however, production data for the individual PGMs were separately available for platinum, palladium and rhodium.

A more detailed evaluation of nickel would be desirable because the average coverage value in the database was lower than for the other metals, and nickel production is currently shifting away from sulphide resources to lateritic resources, resulting in a more complex production process (Jessup and Mudd, 2008). Given the representativeness requirements and the currently available data, this is the best set of actual mining data that can be analyzed and that can serve as a basis for the development of CFs for impacts on metal resources.

Table 5-3 An overview of characteristics of the analyzed data.

<table>
<thead>
<tr>
<th>Metal</th>
<th>Coverage of global production (median, % mass, all years)</th>
<th>Number of included years</th>
<th>Mine-unspecific recovery usage (median, % mass of metal in ore)</th>
<th>Mine-unspecific recovery usage in allocation (median, % value recovered)</th>
</tr>
</thead>
<tbody>
<tr>
<td>Cu</td>
<td>72</td>
<td>11</td>
<td>1</td>
<td>11</td>
</tr>
<tr>
<td>Zn</td>
<td>70</td>
<td>9</td>
<td>&lt;1</td>
<td>12</td>
</tr>
<tr>
<td>Au</td>
<td>67</td>
<td>11</td>
<td>2</td>
<td>7</td>
</tr>
<tr>
<td>Ag</td>
<td>67</td>
<td>12</td>
<td>5</td>
<td>4</td>
</tr>
<tr>
<td>Pt</td>
<td>85</td>
<td>12</td>
<td>4</td>
<td>12</td>
</tr>
<tr>
<td>Pb</td>
<td>67</td>
<td>10</td>
<td>3</td>
<td>10</td>
</tr>
<tr>
<td>Pd</td>
<td>73</td>
<td>12</td>
<td>13</td>
<td>12</td>
</tr>
<tr>
<td>Mo</td>
<td>69</td>
<td>11</td>
<td>26</td>
<td>4</td>
</tr>
<tr>
<td>Ni</td>
<td>61</td>
<td>10</td>
<td>9</td>
<td>20</td>
</tr>
</tbody>
</table>

5.4.2 Comparison with selected existing methods

The CFs presented in the previous section (shown in Table 5-2, column 2) were compared with the CFs from existing LCIA methods to determine whether there were any noteworthy differences. First, a comparison between the individual CFs is presented. This comparison is followed by a comparison of the methods when the CFs are applied to actual metal containing products. When the average annual ore requirements increase, is used as CF, it is multiplied with the amount of metal i (in the ore) required for product Y. The result, which is expressed as the extra mass of ore per year, can be interpreted as the extra amount of ore needed each year to produce product Y, or as part of the annual increase in ore to be mined and treated to produce the total annual supply of metal i due to production of product Y. As there are uncertainties involved, estimates of the actual development were not attempted. Instead, the relative differences between the metals were considered to be more important.

Four methods for quantifying the impacts on metal resources were taken into account. For the Eco-indicator 99 method and the ReCiPe method (midpoint) the ecoinvent v2.2 implementations were applied (Swiss Centre for Life Cycle Inventories, 2010). A more recent version of the CML default method from November 2010 was included, and factors for the EPS2000 method were taken directly from the documentation (Steen, 1999a, 1999b).
Subsequent to a direct comparison of the CFs, all of the methods were applied to ecoinvent datasets to compare the calculated impact values on metal resources attributable to a selection of metal products including their supply networks.

To compare the CFs of the different methods, which are all originally expressed in different units, all of the CFs were converted to Cu-equivalents, i.e., the CFs for each method were expressed relative to copper. Cu-equivalents were chosen because copper has the highest annual world production among the studied metals. For example, the CF for silver given in Table 2 is 44.4 kg ore kg silver$^{-1}$ year$^{-1}$, and the CF for copper is 0.32 kg ore kg copper$^{-1}$ year$^{-1}$. Expressed in Cu-equivalents, the CF of silver becomes 139 kg copper kg silver$^{-1}$. The results are as shown in Table 5-4. As discussed in chapter 3 the ReCiPe and the Eco-indicator 99 methods infer increases in metal production costs or energy requirements on the basis of decreases in ore grade, the CML method is based on use to resource ratios and the factors in the EPS2000 method are based on a scenario where metals are produced from average rock. It should be noted that the Eco-indicator 99 method does not include CF values for gold, silver, platinum and palladium. It was found that in all but the CML method gold, palladium and platinum have the highest CF values among the nine metals. In the CML method the CF value of silver is higher than the CF value of palladium. In all other methods silver was found to have the lowest CF of the precious metals.

Table 5-4 The relative CFs for the impact on metal resources of selected LCIA methods and results for the annual increases in ore requirements presented in this study are compared. All CFs are converted to Cu-equivalents.

<table>
<thead>
<tr>
<th></th>
<th>This study (Cu-equivalents)</th>
<th>ReCiPe (Midpoint, ecoinvent 2.2) (Cu-equivalents)</th>
<th>CML (default, Nov 2010) (Cu-equivalents)</th>
<th>Eco-indicator 99 (ecoinvent 2.2) (Cu-equivalents)</th>
<th>EPS2000 (Steen, 19999b) (Cu-equivalents)</th>
</tr>
</thead>
<tbody>
<tr>
<td>Au</td>
<td>55,926</td>
<td>1638</td>
<td>38,100</td>
<td>n/a</td>
<td>5721</td>
</tr>
<tr>
<td>Pt</td>
<td>16,287</td>
<td>3813</td>
<td>1622</td>
<td>n/a</td>
<td>35,721</td>
</tr>
<tr>
<td>Pd</td>
<td>6375</td>
<td>89.3</td>
<td>417.7</td>
<td>n/a</td>
<td>35,721</td>
</tr>
<tr>
<td>Ag</td>
<td>138</td>
<td>6.70</td>
<td>866.8</td>
<td>n/a</td>
<td>259.6</td>
</tr>
<tr>
<td>Mo</td>
<td>91.2</td>
<td>4.86</td>
<td>13.0</td>
<td>1.12</td>
<td>10.2</td>
</tr>
<tr>
<td>Ni</td>
<td>2.86</td>
<td>0.29</td>
<td>0.04</td>
<td>0.65</td>
<td>0.77</td>
</tr>
<tr>
<td>Cu</td>
<td>1.00</td>
<td>1.00</td>
<td>1.00</td>
<td>1.00</td>
<td>1.00</td>
</tr>
<tr>
<td>Pb</td>
<td>0.27</td>
<td>0.04</td>
<td>4.64</td>
<td>0.20</td>
<td>0.84</td>
</tr>
<tr>
<td>Zn</td>
<td>0.22</td>
<td>0.05</td>
<td>0.39</td>
<td>0.11</td>
<td>0.27</td>
</tr>
</tbody>
</table>

(n/a – not available)

For the method proposed in this study the value obtained for nickel relative to copper was found to be higher than in the other methods. The CML method assigns a relatively high CF value to lead. In addition, nickel has a very low relative CF value in the CML method. In all ore grade based methods lead and zinc have lower CF values than nickel and copper. The CFs obtained for zinc and lead are rather close both for the method used here and for the ReCiPe method. There are bigger differences between the CFs of lead and zinc for the other methods. This may be due to some important similarities between the approach used here and the ReCiPe approach: 1) Both methods are based on ore grades. 2) Lead and zinc often occur together in deposits and both methods used metal prices to account for this co-production. Though the Eco-indicator method is also based on ore grades, co-production and metal prices were likely not considered.

In the EPS method lead has a CF value higher than nickel, but still lower than copper. Furthermore, molybdenum and silver had similar values in the method proposed in this study.
and in the ReCiPe method. The differences between the methods based on ore grade changes and the other two methods are likely related to the fact that the crustal abundance of a metal can be relatively low, while its ore grades can be relatively high, and vice versa. Though the ore requirement change is not necessarily high only because the ore grade is low, there clearly is some correlation. For example, copper has a higher average concentration than lead in the continental crust (Wedepohl, 1995) but typically has lower ore grades than lead. The obvious distinctions between the methods based on ore grade changes are the magnitudes of the relative differences between the metals. The CFs in the ReCiPe method were on average much lower in terms of the Cu-equivalents than in the method proposed in this study.

With the exception of some prominent differences between the methods, such as the CML method not being based on ore grades, it is hard to determine the exact cause of the differences observed in the CF values because not only the methodologies but also the time reference and type of data are quite diverse. Part of the difference between the values for platinum and palladium in the ReCiPe approach and the approach in this study can be attributed to the higher relative platinum price used in the ReCiPe method.

Overall, it can be concluded that the CFs determined for the precious metals in this study are rather high compared with the established methods. In addition, it is noteworthy that nickel has a higher value than copper, which is not the case in any of the other methods.

Figure 5.1 The metal resource impacts of electronic components including their supply networks are compared when using the indicator developed in this study versus approaches using selected LCA resource indicators. The results are converted to Cu-equivalents.

In addition to the direct comparison of the CFs, the CFs were also applied to ecoinvent datasets for electronic components. This made it possible to consider the metal content in the product, as well as other metals used in the supply chain of the electronic component. For this comparison, only those environmental metal resource flows for which an indicator value was quantified in the present study were included, i.e., copper, gold, silver, lead, zinc, nickel, molybdenum, platinum and palladium. The CF values of this study were assigned to the relevant resource flows available in ecoinvent. In doing so, differences in grades between these flows were disregarded because the metal market is assumed to be working globally, and therefore, only the amount of metal that is taken from the ground in total is of importance.
It was assumed that the beneficiation recoveries considered in ecoinvent were similar to the ones used in this study for the ore requirement calculations.

The results for each method were expressed in terms of Cu-equivalents, i.e., the cumulated results for each product were divided by the CF for copper (in ore) for each method. The results are presented in Figure 5-1. The outcome for these products was largely dominated by their precious metal content. As the Eco-indicator 99 method does not contain CFs for the precious metals, its results are very low and were excluded. The rankings for the method proposed by this study (the indicator based on increases in ore requirements) and the ReCiPe method were found to be the same. For both methods the highest values were obtained for the printed wiring circuit board. However, the method presented in this study resulted in much higher values in terms of Cu-equivalents due to the relatively high CF values for precious metals. In both the CML and the EPS2000 method the highest impact values were calculated for the inductor. Thus, when conducting an LCA in practice the results depend substantially on which method is chosen. The results obtained with methods like the EPS2000 and the CML method will not represent the more direct implications of metal resource extraction.

5.5 Concluding remarks

By using actual and recent mining data, which were available for the period from 1998 to 2010, CF values for metal resources could be established for nine metals. The rankings found in the various approaches show similarities, especially for the methods that are based on ore grades. An important difference between this study and the existing LCIA methods is that the CFs from this study assign relatively higher importance to precious metals. If more data on ore grades become readily available, then the method developed in this study can be further expanded and improved with respect to coverage of global primary production, number of metals included and the time frame considered. Of course, any method that is based on metal resources will always be limited because considerable sections of the planet remain unexplored and technological advances are difficult to predict.

It would be desirable for the method presented in this study to include other aspects relevant to the calculation of the efforts required during primary metal production, such as an inclusion of utility requirements that take into account the mine type (UG versus OP) and processing technique (e.g. concentrate production with subsequent pyrometallurgical treatment versus leaching, solvent extraction and electrowinning) because changes in ore grades are also associated with these aspects. In the next chapter some of these aspects and their impacts on energy requirements will be explored for the case of copper.
6 Modelling fossil energy demands of primary non-ferrous metal production: The case of copper

Abstract. Some LCIA methods are based on ore grade changes, but typically do not consider the impact of changes in primary metal extraction technology. To characterize the impact of technology changes for copper we modelled and analyzed energy demand, expressed in fossil energy equivalents (FEE) per kg of primary copper taking into account the applied mining method and processing technology. The model was able to capture variations in reported energy demands of selected mining sites (FEE: 0.07 to 0.84 MJ-eq./kg ore) with deviations of 1 to 30%. Applying the model to a database containing global mine production data resulted in energy demand median values of around 50 MJ/kg Cu irrespective of the processing route, even though median values of ore demands varied between processing routes from ca. 35 (UG, conventional processing) to 200 kg ore/kg Cu (OP, solvent extraction and electrowinning), as high specific ore demands are typically associated with less energy intensive extraction technologies and vice versa. Thus, only considering ore grade in LCIA methods without making any differentiation with regard to employed technology can produce misleading results.

Keywords: life cycle assessment; ore grade; energy demand; copper; abiotic resources

6.1 Introduction

The methods used in LCIA for evaluating the impacts of primary metal use in the AoP natural resources are diverse. Especially, evaluation at the endpoint is rather uncertain. The reasoning behind some of the methods is that decreasing ore grades, and hence increases in specific ore demand, potentially increase the efforts needed to produce metals from primary resources (Goedkoop and Spriensma, 2000; Goedkoop et al., 2009; Jolliet et al., 2003a; Müller-Wenk, 1998), with the most recent approaches proposed by Vieira et al. (2012) and in chapter 5 of this thesis. What these methods do not consider are changes in mining technology, which make it possible to exploit lower grade deposits economically.

In the case of copper an important technological change is the shift away from UG to OP mining methods during the 20th century (Crowson, 2003). Today OP mining is the most common method for mining copper (International Copper Study Group (ICSG), 2010). While ore grades are typically lower at OP mines than at UG copper mines (Davenport et al., 2002), OP mines are usually benefitting from economies of scale. Another technological change taking place in the second half of the 20th century is the increasing share of primary copper metal being produced via solvent extraction and electrowinning (SXEW) rather than via the conventional route, where concentrates are produced, which are smelted and subsequently electrorefined. Today about 20% of primary copper are produced via the SXEW route, while it was about 10% in the beginning of the 1990s (Schippers et al., 2011). The typical recovery

efficiency, i.e. the percentage of metal recovered in the product, differs between SXEW and the conventional route. Ore grades at these operations also tend to be lower than at mining operations employing conventional ore processing methods (Crowson, 2003).

In spite of these technological transitions the LCIA methods that are based on ore grade decreases limit their models for resulting changes in efforts to fixed amounts of energy, respectively costs per unit mass of ore. In the surplus energy method (Goedkoop and Spiersma, 2000; Müller-Wenk, 1998) there is at least a distinction between different metals while in the ReCiPe method (Goedkoop et al., 2009) one fixed unit cost per kg of ore is used. Vieira (2012) distinguishes between different deposit types for copper which might make it possible to assume certain mining methods, e.g. porphyry copper deposits are today typically mined via OP methods (Ayuso et al., 2010). However, these deposits contain sulphide and oxide ores and the preferred process route depends on the specific mineralization and also changes with advances in technology (Bartos, 2002; Schippers et al., 2011). Nickel is another metal for which the existence of fundamentally different resource types is relevant for the employed primary production technology. Even though 60% of continental nickel resources occur in laterites and only 40% in sulphide deposits (USGS, 2010), for a long time most primary nickel was still supplied from sulphide ores, because laterites require more complex treatment (King, 2005; Norgate and Jahanshahi, 2011). In 2010 primary nickel production from laterites exceeded nickel production from sulphide sources for the first time (CSIRO, 2012).

To determine whether the limited consideration of technological aspects of primary metal production affects the assessment of the impact of ore grade changes an attempt was made to model energy demands of copper mining and mineral processing based on data available in a commercial mining database containing annual production data for individual mining operations. For the modelling a three-step procedure was followed. First, a model distinguishing between different mining and mineral processing methods was set up based on literature. Then, the model was validated by applying it to mining operations for which the energy demands had been reported by mining companies in sustainability reports and other similar publications. Finally, the model was implemented by applying it to data available in a commercial mining database to obtain an impression of the results on a global scale.

6.2 Materials and methods

As mentioned in the introduction primary copper can be produced via different routes. A simplified scheme of the production routes is presented in Figure 6-1. Ore containing copper and other metals is mined via UG or OP methods depending on the properties of the deposit. Sulphide ore is typically treated via the conventional route, whereas oxide ore and certain types of sulphide ore are treated via the SXEW route. During conventional processing the size of the ore is first reduced (comminution). Then concentrates are produced, typically by flotation methods. Comminution and concentration are also referred to as beneficiation. Often other metals than copper are recovered to concentrates as well. While beneficiation typically occurs close to the mine, smelting and refining takes place further away. Refining via SXEW typically takes place at the mining site. Before solvent extraction the mined and often also crushed ore is leached. Leaching can take several months. Therefore, there can be considerable time lag between the initial mining and processing and the actual recovery of the copper cathode at the mining site. The product of the leaching is the so-called pregnant leach
solution (PLS). Via solvent extraction copper is concentrated in an acid solution and then deposited as copper cathode in the electrowinning step.

![Simplified representation of refined copper production from ores.](image)

In the following the term operational unit will be used to denote an entity which can be classified according to the following typology: 1) UG mining and conventional processing (UG-conventional), 2) OP mining and conventional processing (OP-conventional), 3) UG mining and hydrometallurgical processing (UG-SXEW) or 4) OP mining and hydrometallurgical processing (OP-SXEW). The modelling took into account among others the following aspects: mine type (OP or UG), processing route (conventional or SXEW), co-production. Mining sites employing both the conventional and the SXEW methods are thus split up into two entities.

### 6.2.1 Fossil energy equivalents

Usually energy is provided to the operations via fuels (mainly diesel) or electricity. When electricity from the grid is not available, electricity is produced locally. In the framework of this study energy provided by fuels and energy provided by electricity are aggregated. Diesel generators are frequently used in remote locations or as back-up electricity source (Bleiwas, 2011; Paraszczak and Fytas, 2012; U.S. Department of Energy, 2002), e.g. at the Raglan Nickel mines (Canada) and Red Dog zinc/lead mine (Alaska). Taking this into account the assumption is made that all electricity is produced locally via diesel generators. The assumed efficiency of electricity production from diesel generators is 39.5% (Bleiwas, 2011). Thus all electricity requirements (MJ) are divided by 0.395 and added to the fuel requirements (MJ) to obtain the energy requirements in terms of fossil energy equivalents (FEE).

Electricity demands for smelting and electorefining are also converted to FEE, even though in reality electricity production with diesel generators is not typical for these processes. The assumption of a local electricity supply via diesel generators production can be seen as a worst case scenario. It also prevents masking of the impact of mining and mineral processing technology, which is the focus of this study, by differences in electricity supply.
6.2.2 Model parameters and independent variables

For each mining and processing step presented in Figure 6-1 energy requirements were to be determined via the model. Different parameters (specific FEE values) and independent variables (masses handled) were employed depending on the characteristics of the operational unit in question. For UG mining the mass of ore mined is the determining factor for the energy demand. This is different for OP mining as considerable amounts of waste material have to be moved in addition to the ore. The waste to ore ratio in OP mining varies usually between one and four (Krauß et al., 1999). Therefore energy demands for OP mining are to be determined based on the total material moved instead of considering only the ore mined. In the case of the conventional processing route requirements were modelled as dependent on the mass of ore treated and a differentiation was made between three ore types: 1) copper ore - copper is the main metal and neither zinc, nor lead nor nickel are recovered alongside copper; 2) polymetallic ore - zinc and lead are recovered next to copper; 3) nickel-copper ore - nickel is recovered next to copper. Smelting and refining requirements were modelled as being dependent on the mass of refined copper cathode. As no input and output data for specific smelters and refineries were to be collected a default Cu recovery for these steps of 97% (Krauß et al., 1999; Marsden, 2008) was assumed. The smelting and refining energy demands were included in the model even though they can be considered independent of the ore grade. This was done because the PLS, which is obtained at the leaching in the SXEW processing flow sheet is not equivalent to a concentrate, but the copper cathodes produced via electrowinning and electrowinning can be viewed as equivalent products. For the SXEW route a differentiation was made between ore that is crushed and then heap leached and mined ore that is leached without further size reduction. Mined ore prior to treatment is also called run-of-mine ore, thus in the following the leaching of this ore will be referred to as run-of-mine leaching. Run-of-mine leaching is typically performed on very low grade material. SXEW requirements were modelled as dependent on the mass of copper cathode recovered and assumed to be the same for PLS from heap leaching and from run-of-mine leaching. The specific FEE values are constant parameters, which were obtained from literature sources (De la Vergne, 2003; Lund et al., 2008; Marsden, 2008; Norgate and Haque, 2010). The amounts of material moved, ore mined, ore treated and copper cathode are variables, which depend on the operational unit and the year in question. Flowcharts (Figure A.5) for determining the appropriate equations as well as a table (Table A.8) containing values of specific energy demands, i.e. the parameters, can be found in Appendix 2.

The following types of operational units were not included in the model: units employing the conventional route and having gold as their main metal, units recovering copper, zinc or lead and nickel, units employing leaching having gold or silver as their main metal. Some of these combinations usually do not occur in practice, while in other cases e.g. operational units leaching gold, other methods are used than those that are used at copper SXEW operations.

6.2.3 Model validation

To confirm that the model could be used to determine energy demands of mining operations it was applied to several operational units for which actual energy requirements could be collected from sustainability reports and similar publications (EHS reports, performance tables) by mining companies. When sustainability reports are mentioned in the following this also includes other similar reports unless otherwise noted. Data were collected for 17 operational units. The names of the operational units and references for the collected data can be found in Appendix 2 Table A.9. In principle, data were collected for the time window
1998-2010, but this was not possible for all operational units. For the modelling of the conventional route smelting and refining was not included, because these steps usually do not take place at the mining site and are thus also not reported along with the energy demand of mining operations. In the sustainability reports the energy demands were reported in various mass, volume ore energy units. The factors used for converting to MJ can be found in Appendix 2 (Table A.10). When energy demands were reported as specific energy demand, e.g. MJ per kg of copper, they were converted to absolute annual requirements. Eventually, the annual fuel and electricity energy demands were converted to FEEs (MJ-eq.) and added up.

The collection of data required to determine the independent variables of the model was done in the same way for the model validation and the final model implementation. RMD (Raw Materials Group, 2013) was the mining database which provided the basis of the data needed. Total ore production, metal production data and information on the five metals of major importance at the operational unit were sourced from RMD. However, data on waste material mined and a differentiation between heap leaching ore and run-of-mine leaching were only rarely available, respectively not available at all in this database. Where needed and available, the missing data were obtained from sustainability reports, annual reports, production reports or other reports with relevant information. Data from these reports were also used to crosscheck the information available in RMD if there were indications of inaccurate data, e.g. when the copper recovered in concentrates at conventional operational units was much more (10%) than the copper contained in the processed ore. For a full list of all reports used as additional sources of information and more details on handling of missing data the reader is referred to Appendix 2 (sections A.2.11 and A.2.12). Eventually, the total FEE based on the reported energy requirements and the total FEE calculated based on the model could be compared for 91 datasets representing 17 operational units.

6.2.4 Model implementation on mining database level

Not all datasets available in the mining database were included in the implementation. A selection was made according to the following requirements: 1) Copper was produced at the operational unit; 2) ore production data were available; 3) information on the processing route was available in RMD or could be determined based on publications from mining companies; 4) metal production or ore grade data were available; 5) a model was available for the operation; 6) data were from the time window 1998-2010. In addition some operational units were excluded if data were unclear or inconsistent, e.g. for one year the ore production reported in RMD is much higher than in years before or after and no other information is readily available confirming or definitely disproving this or there were changes in mining type (OP or UG) and it is not clear when this occurred.

Though the required data were compiled in a similar manner as described above for the validation of the model, some additional aspects were taken into account in the implementation. In some cases conventional and SXEW processing takes place at the same site, but most data were available separately for each processing route. When this concerned an OP mine then the material mined was allocated over these two operational units according to the economic value which could be recovered from the processed ore. To determine the economic value the recoverable metal production was multiplied by a 10-year average price (See Appendix 2 Table A.4 and Table A.5). For operational units using conventional processing the recoverable value was simply based on the reported metal production. As the long leaching time at SXEW operational units, introduces a considerable time lag between
As the model implementation was targeted at copper only, allocation was used if more than one metal was recovered at an operational unit. In these cases mining, comminution, concentrating and leaching energy requirements were allocated to copper according to economic value, which was calculated based on recoverable metal production for the SXEW route and recovered metal production for the conventional route. As the energy demand for copper cathode production was to be determined for both routes, smelting and refining requirements were added to the energy demand at the mining site for conventional operational units. Finally, allocated energy requirements of each step were added up to obtain a year total for each operational unit and this annual energy demand was divided by the copper production in that year. Per processing route the results were then checked for extreme outliers, which could indicate inaccurate or incomplete data. This could lead to correcting or retaining the raw data (independent variables) if another trustworthy source could be found, or lead to excluding the dataset, if parts of the data seemed doubtful, e.g. because they differed considerably from the data of other years for the mine. Of course, it was taken into account that time lag in the leaching step could lead to extreme specific energy demands per unit mass of copper produced.

To estimate the coverage of global primary copper production and the contribution of the four possible processing routes to global primary copper production and global SXEW production (Cochilco, 2011, 2007), as well as global OP and global UG share (Raw Materials Group, 2012) were obtained from literature and RMD. On the basis of the observation that for the 1998-2010 time window only relatively few UG-SXEW operations could be identified in RMD, the contribution of these types of operations to the global primary copper production was assumed to be negligible for this period.

6.3 Results and discussion

6.3.1 Validating the energy model with collected energy data

As described above the model was validated by applying it to a selection of mines for which energy consumption data were available from sustainability reports to compare reported and modelled energy consumption. Table 6-1 shows reported ore production and energy consumption converted to FEEs aggregated over all years with sufficient available data for each operational unit. The third column shows the FEE reported, i.e. the FEE that was calculated from the reported energy consumption values collected from the reports. The fifth column shows the FEE modelled, i.e. the FEE that was calculated based on the developed model for mining and mineral processing energy consumption. With the exception of the OP-SXEW units the FEE values only take into account energy consumption up to the production of concentrates. For the OP-SXEW units production of copper cathodes is included. As expected the specific FEE demands per unit mass of ore (fourth column) clearly differ between OP and UG operational units, even though concentrates are the final product in both mining and initial processing on the one hand and eventual production of CCs on the other hand, the recoverable value for those operational units was determined based on ore production and ore grade data if possible. The average recovery was calculated based on the aggregated data on this operational unit available in RMD for all years in the time window. If this approach was not possible, the metal production was used in spite of the time lag. For long established operations this can still deliver reasonable estimations of the amount of metal which can be recovered from the ore.
cases. Considering that Cu cathodes and not only concentrates are produced at the SXEW mining sites the specific FEE demand per kg of ore is rather low in comparison to the OP-conventional specific FEE demand per kg of ore. Variations can also be observed inside the groups. Specific FEE demands range from 0.41 to 0.84 GJ-eq./t ore for UG-conventional, from 0.20 to 0.55 GJ-eq./t ore for OP-conventional and from 0.07 to 0.35 GJ-eq./t ore for OP-SXEW. These variations can be caused by actual differences in the conditions and characteristics of the operational unit that are not reflected in the grouping, e.g. ore grade or waste-to-ore ratio, but also by differences in the system boundaries applied for determining the energy consumption reported in the sustainability reports or even by reporting errors.

Table 6-1 Aggregated results of the comparison between FEEs calculated from reported consumption data of electricity and fuels and modelled consumption.

<table>
<thead>
<tr>
<th>Operational unit</th>
<th>Total ore production (1000 t)</th>
<th>Total FEE reported (TJ-eq.)</th>
<th>FEE reported/ore (GJ-eq./t)</th>
<th>Total FEE modelled (TJ-eq.)</th>
<th>FEE reported/FEE modelled (GJ-eq./GJ-eq.)</th>
</tr>
</thead>
<tbody>
<tr>
<td><strong>UG-conventional</strong></td>
<td></td>
<td></td>
<td></td>
<td></td>
<td></td>
</tr>
<tr>
<td>El Teniente</td>
<td>279,800</td>
<td>114,860</td>
<td>0.41</td>
<td>141,202</td>
<td>0.81</td>
</tr>
<tr>
<td>Neves Corvo</td>
<td>10,556</td>
<td>7,422</td>
<td>0.70</td>
<td>6,834</td>
<td>1.09</td>
</tr>
<tr>
<td>Garpenberg</td>
<td>7,372</td>
<td>6,215</td>
<td>0.84</td>
<td>4,773</td>
<td>1.30</td>
</tr>
<tr>
<td>Golden Grove</td>
<td>3,007</td>
<td>1,960</td>
<td>0.65</td>
<td>1,947</td>
<td>1.01</td>
</tr>
<tr>
<td>Cayeli</td>
<td>7,915</td>
<td>5,507</td>
<td>0.70</td>
<td>5,124</td>
<td>1.07</td>
</tr>
<tr>
<td>Pyhäsalmi</td>
<td>6,952</td>
<td>3,780</td>
<td>0.54</td>
<td>4,501</td>
<td>0.84</td>
</tr>
<tr>
<td><strong>OP-conventional</strong></td>
<td></td>
<td></td>
<td></td>
<td></td>
<td></td>
</tr>
<tr>
<td>Aitik</td>
<td>106,831</td>
<td>24,794</td>
<td>0.23</td>
<td>25,021</td>
<td>0.99</td>
</tr>
<tr>
<td>Alumbrera</td>
<td>211,809</td>
<td>53,899</td>
<td>0.25</td>
<td>51,584</td>
<td>1.04</td>
</tr>
<tr>
<td>Highland Valley Copper</td>
<td>386,722</td>
<td>78,836</td>
<td>0.20</td>
<td>86,630</td>
<td>0.91</td>
</tr>
<tr>
<td>Aguablanca Polymetallic</td>
<td>6,841</td>
<td>3,744</td>
<td>0.55</td>
<td>3,075</td>
<td>1.22</td>
</tr>
<tr>
<td>Los Pelambres</td>
<td>201,370</td>
<td>43,297</td>
<td>0.22</td>
<td>47,030</td>
<td>0.92</td>
</tr>
<tr>
<td><strong>OP-SXEW</strong></td>
<td></td>
<td></td>
<td></td>
<td></td>
<td></td>
</tr>
<tr>
<td>Cerro Colorado</td>
<td>100,702</td>
<td>29,719</td>
<td>0.30</td>
<td>29,293</td>
<td>1.01</td>
</tr>
<tr>
<td>El Tesoro</td>
<td>39,304</td>
<td>13,904</td>
<td>0.35</td>
<td>16,784</td>
<td>0.83</td>
</tr>
<tr>
<td>Lomas Bayas</td>
<td>277,479</td>
<td>18,821</td>
<td>0.07</td>
<td>24,855</td>
<td>0.76</td>
</tr>
<tr>
<td>Mantoverde</td>
<td>28,194</td>
<td>5,021</td>
<td>0.18</td>
<td>5,521</td>
<td>0.91</td>
</tr>
<tr>
<td>Minera Spence</td>
<td>48,363</td>
<td>15,470</td>
<td>0.32</td>
<td>19,256</td>
<td>0.80</td>
</tr>
<tr>
<td>Zaldivar</td>
<td>264,769</td>
<td>59,086</td>
<td>0.22</td>
<td>64,712</td>
<td>0.91</td>
</tr>
</tbody>
</table>

In column six the reported FEE demand and predicted FEE demand are compared by dividing the one by the other. In some cases the modelled and the reported FEE demands are quite close with only 1% difference, but in other cases there are bigger differences of up to 30%. A 30% deviation can still be considered as low when it is taken into account that the reported energy to ore ratios differ by more than 100%. While for the UG-conventional and OP-conventional operational units modelled FEE demand was sometimes higher and sometimes lower than the reported FEE demand, most of the modelled energy requirements for the OP-SXEW operations in the selection are higher than the reported ones. Nevertheless, some of the differences influencing energy requirements seem to be reflected in the model for SXEW. For
example the reported specific FEE demands for Lomas Bayas were much lower than for the other SXEW operational units. This was also reflected in the modelling a low FEE demand due to a low waste to ore ratio and leaching of run-of-mine ore.

A perfect fit cannot be expected for each operation for several reasons some of which have already been mentioned above: 1) System boundaries for the predicted energy requirements will likely differ from those applied in reporting energy and fuel requirements in the various reports. 2) Energy requirements found in the reports can be wrong or incomplete. The authors tried to limit the likelihood of this by looking at reports from several years and only considering operational units for which more than one year of data were available (excluding the year of commissioning the operational unit). 3) Unusual, events can effect energy requirements and productivity. 4) A number of important variables could not be taken up in the model: like the depth of the mine, hardness of the rock or mine size. 5) Data on ore production, metal production, material mined can be wrong. Overall, the model seems satisfactory for the conventional route while there might be a tendency to overestimate energy demand for SXEW operational units.

In the surplus-energy method of Eco-indicator 99 and the marginal cost method of ReCiPe only the efforts up to and including the concentrate production are considered. This might have been reasonable when SXEW was still rather unimportant. The energy demand calculations for the surplus-energy method are based on values published in 1983. In the early 1980s leaching and SXEW was still mainly applied to existing waste piles (Tilton and Landsberg, 1997), i.e. the material was not mined to be leached, and represented less than 10% of the total primary copper production. Thus, it is not surprising that only mining, comminution and concentrating are included in the energy demand for the surplus energy method, whose energy demand calculations are based on values published in 1983. The fixed gross energy demand values employed in the surplus energy method are based on the assumption of a specific mix of OP and UG mining. If these shares never change, this assumption does not pose a problem. Crowson (2003), however, reported that in 1985 almost 30% of the ore treated at conventional copper mines had been extracted from UG operations and that by the year 2000 this figure had halved. In addition, a fixed energy demand per unit mass of ore can be a problem due to technological differences at lower ore grade mines (see section 6.1). The ReCiPe method does not work with energy requirements, but with operating costs in US$/kg ore, which are also fixed. Energy costs represent a large share of total operating costs, but other costs, like costs for consumables and labour costs, are also important (Davenport et al., 2002; Krauß et al., 1999). In contrast to the surplus-energy method no differentiation is made between different metals concerning efforts per unit mass of ore. This can be relevant when comparing cost increases for copper with cost increases for zinc, which is usually mined UG, or with cost increases for metals typically mined at high grades which do not necessarily require concentration like iron.

6.3.2 Implementing the energy model by applying it to a selection of mines from a mining database

Overall, the model equations were applied to more than 1000 datasets in the time window 1998-2010, representing 171 distinct operational units. These datasets cover between 44 and 58% of the global annual primary copper production, or between 40 and 53% of the global copper production via the conventional route and between 64 and 81% of the global SXEW production. There were only three UG-SXEW operational units in the selection, confirming
that this combination is presently very uncommon. Therefore, UG-SXEW operational units were excluded from the analyses presented below.

It should be noted that values are expressed per kg of refined copper and not per kg of copper in ore. The advantages of this approach are twofold: 1) The inclusion of more operational units is possible as more data were available for copper production than for copper ore grades. 2) The energy requirements per kg of refined copper are eventually more relevant to society than the energy requirements per kg of copper in ore. A disadvantage is that the reference flow in LCIs is typically copper in ore. Still as the total amount of ore is usually also evaluated in the framework of the presented study it is possible to contrast ore requirements, on which the ore grade based LCIA methods rely to determine energy or economic costs, with energy requirements determined based on a more detailed model.

Allocated ore demand and energy demand per unit mass of copper were calculated for each processing route. Clear differences can be observed between the three processing routes. For the UG-conventional route a little more than half of the copper production between 1998 and 2010 occurred at an ore demand of 35 kg ore/kg copper or less, whereas for the OP-conventional and the OP-SXEW route less than 5% copper were produced at such a low ore demand. The highest ore demand can be observed for the OP-SXEW route where the median is at around 200 kg ore/kg Cu, i.e. for 50% of the total copper production 200 kg of ore or less was processed per kg of copper. The median is located at around 100 kg ore/kg Cu for the OP-conventional route. Graphs of the distributions per processing route are provided in Figure A.6 in Appendix 2.

The distributions of specific FEE demands of copper cathode production for the three analyzed combinations (UG-conventional, OP-conventional and OP-SXEW) over the time period from 1998 to 2010 are shown in Figure 6-2. As for the ore demand median values were calculated based on the total analysed copper production per processing route. The lowest median values (47 and 48 MJ/kg Cu) of specific energy consumption for Cu-cathode production were observed for the UG-conventional and the OP-SXEW operational units. For UG-conventional the specific FEE demands are strongly influenced by allocation as copper is typically not the main metal. The median value calculated for OP-conventional was somewhat higher with 56 MJ/kg Cu. As the operational units have to be economical and are all in principle trading on the same global market this clustering around a similar value could be expected. Of course, energy only makes up part of the total costs of copper production.

For the UG-conventional group two peaks are visible in the charts of Figure 6-2. This might be related to metals being co-produced. Copper was typically the main metal at the OP operational units in the selection and Cu was typically the only metal recovered at the OP-SXEW sites. Some uncertainties in the results for OP-SXEW arise due to limited data availability concerning the amount of ore that was heap leached and the amount of leached run-of-mine ore though for most OP-SXEW datasets information could be found to determine the amounts of ore for heap leaching and ore run-of-mine leaching separately. In other cases only the amount of ore for heap leaching was available, even though also leaching of run-of-mine material took place. Similarly, only the total amount of material treated was known in some cases without any further information available. Then it was usually assumed that all material was crushed and heap leached. Overall, it can be concluded that the median values of specific ore demands differ by a factor of two to four, while medians of specific FEE demands are much more similar, with less than 20% difference between the processing routes.
Figure 6-2 Total copper production of the considered operational units between 1998 and 2010 in function of specific FEE demand.

In Figure 6-3 specific FEE demands per unit mass of copper for each operational unit are plotted versus allocated ore demand. A distinction is made between the three routes. If the points formed a line, then a less detailed model only distinguishing between the three routes would give comparable results. The variation for the OP-SXEW route seemed to be most pronounced, this might be due to more variability regarding heap leaching and leaching or run-of-mine material and waste to ore-ratios. For the OP-SXEW operational units the impact of mining energy requirements is on average higher than for the ones following the OP-
conventional route. For the UG-conventional route a distinction between the recovered metals is clearly visible in the chart.

Figure 6-3 Specific energy requirements in function of allocated ore requirements to produce 1 kg of Cu cathode per processing route.

The model does not include any allocation on sulphuric acid production during the pyrometallurgical treatment of sulphide copper ore, even though it can be an important product at the smelter. For example, sulphuric acid production makes up 20% of the income of Chilean copper smelters (Schippers et al., 2011). If this would be factored in, pyrometallurgical route and hydrometallurgical route would be much closer matched because part of the energy requirements for the smelter would be allocated to the production of sulphuric acid and thus the energy requirements allocated to copper produced via the pyrometallurgical route would be reduced. In addition, only energy requirements in form of fossil fuels and electricity were considered. Explosives, chemicals and other consumables were not taken into account. In principle, with an allocation on sulphuric acid the model could be refined. For other missing elements more extensive additional modelling and data collection would be required.

As the share of total copper produced via SXEW was much higher in the selection than reported from other sources, shares needed to be corrected to be able to estimate the development of global energy requirements. It was assumed that the average specific FEE demand per unit mass of copper for each processing route for the selection was representative for the totality of copper producing mining operations in the world following the same processing route. For each year the specific global FEE demand was determined as follows: Specific global FEE demand year\(_i\) = global share SXEW year\(_i\) × specific FEE demand OP-SXEW year\(_i\) + global share UG year\(_i\) × specific FEE demand UG-conventional year\(_i\) + global share OP-conventional year\(_i\) × specific FEE demand OP-conventional, whereby the share of OP conventional is determined as the remainder of the global primary refined production. The results of this estimation presented in Figure 6-4 indicate a slight increase in specific FEE demand between 1998 and 2010 from 55 to 58 MJ/kg Cu. Specific allocated ore requirements per unit mass of copper were calculated in an analogous fashion as the global specific FEE
demand. Remarkably, the changes in FEE demand seem to be steadier than the changes in specific ore requirements. If a fixed specific value for economic or energy costs per kg of ore was used, like in the surplus energy method or the ReCiPe method, the specific demand per unit mass of copper would follow the development of the specific ore demand per unit mass of copper, which gives a quite different impression. Nowadays, lower grade ores are typically associated with technologies that are less energy intensive, leading to a partial decoupling between ore demand and energy demand. This supports the notion that a future LCIA method attempting to model changes in energy demand should try to include at least some aspects of mining technology. At any rate known technologies could be considered, e.g. by taking into account depths of deposits and the form in which the copper occurs. Of course, a decrease in ore grade cannot simply be offset by a switch from conventional processing to SXEW processing as the ore also has to be sufficiently susceptible to the treatment method. Still, it cannot be said that a lower global ore grade equals higher production costs, without taking into account the employed extraction technology.

Figure 6-4 Timelines of approximated specific FEE (A) and ore demand (B) for global copper cathode production based on the selection from the mining database.

6.3.3 Concluding remarks

In principle, the model could be applied to the data in the mining database and the results seemed reasonable overall. The data available in the commercial mining database were however not sufficient and additional data had to be collected. The presented model for determining the energy demand based on the amounts of material mined, ore processed and metal produced already takes into account a number of factors, but is still limited as a more detailed model would also require a more extensive data collection. Some additional aspects might be more easily implemented than others. If the aim was to predict actual global energy demand for primary copper production, the model could be improved by considering the actual local electricity production and by making refinements for the modelling of smelting and refining. Additionally an inclusion of explosives and steel used in the comminution could be considered. Recently some big OP mines have transitioned or are planned to transition to UG mining (Chadwick, 2008; Henao, 2012; Page, 2001; Rio Tinto, 2005; Xstrata PLC, 2013, as the pits become too deep to be mined economically via OP mining methods. In these cases
often block-caving is applied, which is a low cost UG mining method. This is also something which might be considered in a more advanced model.

Based on the presented model and the employed data it could be concluded that on a global level changes in ore requirements are not sufficient to derive changes in energy demand, i.e. considering grade data independent from extraction technology could lead to quite different conclusions regarding the evolution of the efforts required for primary copper production than when taking into account grade data and (possible) extraction technologies. It might be possible to take some aspects of technology into account in LCIA methods, e.g. by considering the depth of deposits and the mineralogy. Also for nickel a more specific modelling would be desirable.
Part III
Case studies – Advanced materials for clean technology applications
Assessment of the overall resource consumption of germanium wafer production for high concentration photovoltaics

Abstract. The overall resource requirements for the production of germanium wafers for III-V multi-junction solar cells applied in concentrator photovoltaics have been assessed based on up to date process information. By employing the CED method and the CEENE method the following resources have been included in the assessment: fossil resources, nuclear resources, renewable resources, land resources, atmospheric resources, metal resources, mineral resources and water resources. The CED has been determined as 216 MJ and the CEENE has been determined as 258 MJ_{ex}. In addition partial energy and exergy payback times have been calculated for the base case, which entails the installation of the high concentration photovoltaics (HCPV) in the Southwestern USA, resulting in payback times of around 4 days for the germanium wafer production. Due to applying concentration technology the germanium wafer accounts for only 3% of the overall resource consumption of an HCPV system. A scenario analysis on the electricity input to the wafer production and on the country of installation of the HCPV has been performed, showing the importance of these factors on the cumulative resource consumption of the wafer production and the partial payback times.

Keywords: Payback Time, germanium wafer, high concentration photovoltaics (HCPV), life-cycle assessment, Cumulative energy demand (CED), Cumulative Exergy Extraction from the Natural Environment (CEENE)

7.1 Introduction

The World’s hunger for energy and the ensuing consumption of fossil fuels have lead to an amplification of the Earth’s greenhouse effect and we are now faced with the challenges of climate change. Though the connection between fossil fuel consumption and climate change is widely accepted, fossil fuels still are by far the main resource (more than 80%) of our energy supply, and this while global energy consumption is still on the rise (IEA, 2010). According to the Fourth Assessment Report of the IPCC the production of useful energy from renewable energy sources, like solar energy, is one of the key mitigation technologies (IPCC, 2007). Solar energy can be harvested to produce heat, fuels or electricity, and it is indeed the most abundant energy resource available on Earth (World Energy Council, 2007).

High concentration PV (HCPV) technologies are a promising alternative to conventional non-concentrating PV technologies to convert solar energy to electricity. Most of the currently applied HCPV technologies are based on high efficiency III-V multi-junction solar cells. In the past this type of multi-junction cells has been applied in space applications where its higher efficiency and lower thickness compared to silicon based cells contribute to a lower mass per produced electricity, the latter being a critical factor outweighing the higher production costs of multi-junction cells as compared to silicon cells.

Over the years also terrestrial applications of high efficiency multi-junction cells have been developed. These employ mirrors or lenses to concentrate the direct sunlight on the highly efficient cells. The additional material inputs are counterbalanced by savings of costly semiconductor material owing to the concentration of sunlight. For example, for an HCPV module with 1 m² of aperture area and a concentration factor of 100, in principle, only 0.01 m² of solar cell surface area is required in the installation for collecting the direct sunlight incident on that 1 m². The concentration factors are commonly in the order of several hundred suns. As they use direct sunlight only, HCPV systems are best suited for sunny regions with little cloud cover and require a tracking device to maximise system output. In 2008 the first larger systems were installed for demonstration purposes, most of which were still based on silicon technology (Wiesenfarth et al., 2012). In the subsequent years newly installed capacity increased reaching about 40 MW in 2012, which almost exclusively were systems based on multi-junction technology. Total installed capacity reached 86 MW in 2012. Newly installed capacity has been increasing rapidly, but is still quite small in comparison to total terrestrial photovoltaic technology of which more than 100 GW were installed in 2012 (Masson et al., 2013). For a recent review on the technological development and further details on the market situation of HCPV-systems see Wiesenfarth et al. (2012) and Bett et al. (2013).

As depicted in Figure 7-1 the III-V cells consist of three layers mainly made up of elements from the 3rd and 5th group of the periodic table. The cells are produced by deposition of two active layers on a suitable substrate via metalorganic vapour phase epitaxy (MOVPE). In
triple junction solar cells germanium is used as the substrate and also acts as an active layer, capturing solar energy at lower energy levels in the red and infrared region of the solar spectrum. Under concentration the cell efficiencies of III-V cells lie in the order of 40%. On laboratory scale efficiencies up to 44.7% have been reported for terrestrial concentrator solar cell (Bett et al., 2013). More technical information on triple junction solar cells is provided by Bett et al. (2007).

Superior quality of the germanium crystal is a prerequisite to achieve such high cell efficiencies. The production of germanium wafers requires (1) germanium containing feedstock and (2) auxiliaries. Globally 120 t (2009) (USGS, 2011) of primary refined germanium are produced annually. In 2008 60% of the annual germanium production was obtained from residues from processing of zinc ores and most of the remainder from ashes produced by coal combustion (Bleiwas, 2010). The latter source offers a great potential, but is not yet economically viable to a greater extent. It has been estimated that the annual emissions of germanium from coal combustion to the atmosphere, amount to as much as 200 times the primary refinery production (Rosenberg, 2009). In his review Rosenberg further explains that the environmental contamination with germanium can be considered insignificant.

Next to the germanium feedstock many other resources are required to produce high purity germanium crystals and eventually germanium wafers. This includes energy as well as various chemicals. The overall cumulative resource consumption is an important indicator of sustainability of production. The CED method briefly mentioned in chapter 3 takes into account the cumulative input of energy resources from nature. CED can give an adequate impression of the environmental performance of products as it is also correlated to other aspects of environmental sustainability (Huijbregts et al., 2010). To also include non-energy resources in the assessment of the resource consumption, while preserving one single unit of quantification, the exergy concept for quantifying resource inputs was applied. The CEENE-method (Dewulf et al., 2007) is one of the methods using exergy for quantifying resource consumption. As already mentioned in chapter 3 CEENE stands for Cumulative Exergy Extraction from the Natural Environment and quantifies the total amount of exergy the natural environment is deprived of for the production of a good or service. Both the CED and the CEENE method are based on a life cycle approach, which is necessary to obtain a realistic picture of the global environmental impact of a product.

A number of studies have been published that quantify the energy requirement to produce (and install) PV systems in general and HCPV systems in particular (Alsema, 2000; Fthenakis and Alsema, 2006; Fthenakis and Kim, 2011; Mohr et al., 2007; Peharz and Dimroth, 2005; Raugei et al., 2007). However, no detailed analysis of the resource consumption of the germanium wafer production processes has been conducted so far. In Peharz and Dimroth (2005), for example, the CED of germanium wafers is based on an estimation of only the electricity consumption by the producing company. Reports of CED for PV systems are often accompanied by the calculation of the energy payback time (EPBT) of the systems. The energy payback time is defined as the time it takes to compensate the primary energy input during the systems life cycle by the electricity production of the PV system. The EPBT can be partitioned into different components, each representing one element of the life cycle, and the sum being equal to the overall EPBT, because the denominator, i.e. the primary energy equivalent of the annually substituted electricity production of the PV installation, is the same for each of these elements. These components of the EPBT might be called partial EPBT.
EPBT is a widely used indicator for PV systems. An overview of some published EPBTs for HCPV systems and non-concentrator systems together with the irradiation for which they were determined is given in Table 7-1. It is remarkable that the range of reported EPBT is quite wide, this has not only to do with the differences in obtained primary energy needs for the production of the systems, but also with the conditions under which the systems are operating/assumed to be operating.

### Table 7-1 Published energy payback times of PV systems.

<table>
<thead>
<tr>
<th>System</th>
<th>EPBT (years)</th>
<th>Annual irradiation (kWh/m²)</th>
<th>Reference</th>
</tr>
</thead>
<tbody>
<tr>
<td>Concentrator systems (range EPBT: 0.6-2.6 years)</td>
<td></td>
<td></td>
<td></td>
</tr>
<tr>
<td>Flatcon (III-V multi-junction)</td>
<td>0.75</td>
<td>1900 (DNI)</td>
<td>(Peharz and Dimroth, 2005)</td>
</tr>
<tr>
<td>Concentrix Solar Flatcon (III-V multi-junction)</td>
<td>0.8</td>
<td>1794 (DNI)</td>
<td>(de Wild-Scholten et al., 2010)</td>
</tr>
<tr>
<td>Amonix (single crystal silicon)</td>
<td>1.3</td>
<td>2480 (DNI)</td>
<td>(Fthenakis and Kim, 2011)</td>
</tr>
<tr>
<td>SolFocus (III-V multi-junction)</td>
<td>0.6</td>
<td>2500 (DNI)</td>
<td>(Reich-Weiser et al., 2008)</td>
</tr>
<tr>
<td>Amonix (III-V multi-junction)</td>
<td>0.86</td>
<td>Southwestern USA</td>
<td>(Fthenakis and Kim, 2010)</td>
</tr>
<tr>
<td>Amonix (III-V multi-junction)</td>
<td>1.5</td>
<td>1794 (DNI)</td>
<td>(de Wild-Scholten et al., 2010)</td>
</tr>
<tr>
<td>III-V multi-junction</td>
<td>2.00</td>
<td>1513 (direct)</td>
<td>(Nishimura et al., 2010)</td>
</tr>
<tr>
<td>III-V multi-junction</td>
<td>2.64</td>
<td>1263 (direct)</td>
<td>(Nishimura et al., 2010)</td>
</tr>
<tr>
<td>Non-concentrator systems (range EPBT: 1.0-5.0 years)</td>
<td></td>
<td></td>
<td></td>
</tr>
<tr>
<td>Multi-Si (rooftop)</td>
<td>2.2</td>
<td>1700 (global)</td>
<td>(Fthenakis and Alsema, 2006)</td>
</tr>
<tr>
<td>mc-Si PV</td>
<td>1.73</td>
<td>1701 (global)</td>
<td>(Nishimura et al., 2010)</td>
</tr>
<tr>
<td>CdTe (roof top)</td>
<td>1.0</td>
<td>1700 (global)</td>
<td>(Fthenakis and Alsema, 2006)</td>
</tr>
<tr>
<td>CdTe (ground-mount)</td>
<td>1.1</td>
<td>1700 (global)</td>
<td>(Fthenakis and Alsema, 2006)</td>
</tr>
<tr>
<td>Thin-film GaAs</td>
<td>5.0</td>
<td>1000 (global)</td>
<td>(Mohr et al., 2007)</td>
</tr>
<tr>
<td>Thin-film GaInP/GaAs</td>
<td>4.6</td>
<td>1000 (global)</td>
<td>(Mohr et al., 2007)</td>
</tr>
</tbody>
</table>

a DNI - direct normal irradiation
b 6.9 kWh/m²d⁻¹.
c Most probably DNI, although not explicitly stated.

Richards and Watt (2007) suggest the energy yield ratio as an alternative measure to be used by the PV community to evaluate and compare the performance of PV systems. The energy yield ratio expresses the equivalent primary energy produced by the system over its lifetime per equivalent primary energy needed to produce this system. This concept has previously been discussed by Johansson (1992). The application of the energy yield ratio allows for a straightforward comparison of systems with different lifetimes.

The main purpose of this chapter is the quantification of the resource consumption of the germanium wafer production process designed for HCPV systems including upstream
processes in the life-cycle. This will be expressed in terms of CED as this is a widely used indicator, but also in terms of CEENE as this allows the inclusion of non-energy resources. In addition, the resource input will be put in context of its application, by assuming an HCPV system in which these wafers are employed. Partial energy and exergy payback times will be calculated and put into relation to reported overall EPBTs. Furthermore, the influence of place of installation of the HCPV system on the partial energy and exergy payback time will be examined. Also the variation of the results in function of the electricity used in the production will be analysed.

7.2 Material and Methods

7.2.1 Germanium wafer production

Data on the germanium wafer production have been acquired from Umicore, which is a leading producer of this kind of wafer. These data were produced based on models of a new production line in Quapaw, USA, and include the refining of the germanium and the preparation of the actual wafer. The germanium wafer for which production data have been inventoried has a diameter of 100 mm, a thickness of 150 µm and weighs 6.16 g. An illustration of the production process is given in Figure 7-2. A more detailed description of germanium wafer production can also be found in Depuydt et al. (2007). In the refining stage germanium tetrachloride is produced by chlorinating germanium dioxide. The GeCl$_4$ is subsequently purified and hydrolysed to GeO$_2$ (A). After reduction high purity germanium is obtained during zone refining (B). The supportive processes include mainly processes aimed at recovering germanium from various recycling streams and by-products production (C). A single crystal is produced via the Czochralski crystal pulling technique and subsequently cropped and cut at length (D). The single crystal is cut into wafers with a wire saw after rounding and flat grinding of the crystal. The germanium wafers are marked with a laser to make them traceable. A number of surface treatments assure that damage free wafers are obtained: edge grinding, surface grinding, etching (E). Eventually the wafers are polished, cleaned, dried and inspected before leaving the production process (F). During the processing a large amount of germanium containing waste streams are produced. Though these streams are recycled, some of the germanium is lost, as not all recycling steps result in 100% recovery of the metal.

Umicore also provided data to estimate the resource consumption for the production of the GeO$_2$ input to the germanium wafer production. The GeO$_2$ production starts from a Ge containing leach residue from the zinc industry. The process is a combination of hydrometallurgical and pyrometallurgical steps with major inputs being acid and energy. The production of GeO$_2$ from the leach residue is rather resource intensive. It is expected that in the future germanium sources will be used that require less resources for refining, e.g. wastes from end-of-life products. The leach residue itself is treated as waste and thus no resource input is allocated to its production in accordance with ISO 14044. Data for upstream processes have been taken from ecoinvent (version 2.1) (Swiss Centre for Life Cycle Inventories, 2009). For the CED method the ecoinvent implementation has been applied (Althaus et al., 2009). A distinction is made between renewable (biomass, geothermal - converted, solar – converted, kinetic in wind - converted, potential (in barrage water) - converted) and non-renewable (fossil, nuclear, primary forest) primary energy sources. The contribution of the primary forest category to the non-renewable resources is typically very low. The CEENE method is based on Dewulf et al. (2007). In this method a distinction
between eight resource categories is made: (1) Renewable resources, (2) Fossil fuels, (3) Nuclear energy, (4) Metal ores, (5) Minerals and mineral aggregates, (6) Water resources, (7) Land occupation and transformation and (8) Atmospheric resources. Renewable resources do not include biomass as the production of biomass is included via the land requirements in the land occupation and transformation category. Somewhat different system boundaries and the inclusion of non-energy categories like metal ores typically lead to CEENE values being higher than CED values.

Figure 7-2 Overview of germanium wafer production depicting the grouping of CED and CEENE inputs.

7.2.2 HCPV system
A system offered by Concentrix Solar (Concentrix Solar, 2010) was chosen as HCPV system. Concentrix Solar is a spin-off from Fraunhofer ISE and thus the system modelled in the
The present study can be assumed to be comparable with the Flatcon system studied by Peharz and Dimroth (2005). The system has a modular design. Figure 7-3 visualizes how these components are combined to form the system, starting from the germanium wafer from which triple junction cells are manufactured. The cell units are the basic building blocks of the system. Each cell unit contains one triple junction solar cell and one lens. 98 cell units are combined together in a module. One wafer is required to produce 8.2 modules (Lerchenmüller, 2010a, 2010b), while one system consists of 90 modules, which are mounted on a tracker. One system has an aperture area of 28.8 m² and a nominal AC power of 6.2 kW. The system is equipped with an inverter to convert the produced DC power to AC power. Fresnel lenses concentrate the sunlight on the solar cells. To assure that the direct sunlight is perpendicular to the surface of the lenses a two-axis tracker follows the sun’s movement during the day. The energy from direct solar radiation at the Earth’s surface that is incident on a plane perpendicular to the sun’s beam is also called direct normal irradiation. More information can be found on the website of Nasa’s Surface Meteorology and Solar Energy Program (SSE, 2010). Annual average direct normal irradiation data have been taken from Trieb et al. (2009), who compiled the data from the SSE database.

![Figure 7-3 Building blocks of the HCPV system.](image)

**7.2.3 Partial energy and exergy payback time**

EPBTs are calculated as the ratio of the primary energy equivalent needed (1) to produce the materials which make up the HCPV system, (2) to manufacture the PV system, (3) to transport the materials during the life cycle, (4) to install the system and (5) to dispose of the system at the end of life to the net primary energy equivalent produced per year. The latter being equal to the primary energy equivalent of the generated electricity minus the primary energy equivalent of operation and maintenance (Fthenakis and Kim, 2011; Peharz and
Dimroth, 2005; Richards and Watt, 2007). In contrast to the generally used EPBT, a partial EPBT will be used here.

\[
\text{partial EPBT} = \frac{\text{PE}_{\text{Ge-wafer}}}{\text{PE}_{\text{net-electricity}}} = \frac{\text{CED}_{\text{Ge-wafer}} \cdot N_{\text{sys}}}{\text{CED}_{\text{NR electricity}} \cdot E_{\text{net-electricity}}} \tag{4}
\]

where PE_{Ge-Wafer}, primary energy equivalent of the germanium wafers required for the PV system (includes transport of materials to the production facility, takes into account germanium wafer losses in the production of the modules) (MJ); PE_{net-electricity}, primary energy equivalent of the net electricity generated per year (includes losses due to DC to AC power conversion, and electricity consumption of tracker), if this electricity was produced from fossil or nuclear resources according to the country mix (MJ.year^{-1}); CED_{Ge-wafer}, Cumulative Energy Demand per wafer (MJ.wafer^{-1}); N_{sys}, number of wafers per PV system; CED_{NR electricity}, specific Cumulative Energy Demand per kWh of non-renewable-based electricity production in the country where the PV system is installed (MJ.kWh^{-1}); E_{net-electricity}, net electricity generated by the PV-system per year (kWh.year^{-1}).

This partial EPBT expresses the time needed for the PV system to compensate the primary energy input for the production of the germanium wafers via electricity production. In addition also a partial Exergy Payback Time (ExPBT) has been defined, similar to the partial EPBT:

\[
\text{partial ExPBT} = \frac{\text{CEENE}_{\text{Ge-wafer}} \cdot N_{\text{sys}}}{\text{CEENE}_{\text{NR electricity}} \cdot E_{\text{net-electricity}}} \tag{5}
\]

where CEENE_{Ge-wafer}, Cumulative Exergy Extraction from the Natural Environment for the production of one germanium wafer (MJ.ex.wafer^{-1}); CEENE_{NR electricity}, Cumulative Exergy Extraction from the Natural Environment of the non-renewable-based electricity production in the country where the PV system is installed (MJ.ex.kWh^{-1}). It has also been considered to work with the yield ratio. However, a yield ratio only calculated for the germanium wafer would not make much sense. Partial EPBTS or ExPBT can be added up to come to an overall EPBT or ExPBT, but this is not possible for the yield ratio. Also the EPBT and ExPBT are suitable for evaluating different localities for the installation of PV-systems when lifetimes are the same.

### 7.2.4 Base Case

In the base case the germanium wafer production is localised in the USA and the HCPV system is assumed to be installed in the Southwestern USA. Where no USA specific datasets were available in the ecoinvent database, European datasets have been used. Initial assessment showed the importance of the hydrochloric acid input on the overall results. Therefore, this dataset (including the production of the chlorine) has also been adapted to use US electricity supply. For sunny regions with an annual direct normal irradiation (DNI) of 2500 kWh.m^{-2} the electricity yield of a system per installed nominal AC power is 2600 kWh.kW_{nom}^{-1} according to company information. Degradation of electricity production over time has not been taken into account.

### 7.2.5 Scenario analysis

Next to the base case also a scenario analysis has been performed on the one hand with respect to the location of production of the germanium wafer, and on the other hand with respect to the location of installation of the HCPV system. Thus, in a first step parameters of the germanium wafer production have been adjusted to account for a change in location. This
concerns in particular the electricity inputs. Typically, the direct energy input, here in the form of electricity, is a major contributor to the overall environmental impact of production processes. Additionally, datasets for electricity production in a number of countries are readily available in the ecoinvent database. Included are 32 datasets, most of which are for European countries, but also include some non-European countries, like China, or the USA. The Norwegian supply mix has the lowest CED (4.71 MJ.kWh⁻¹), resp. CEENE (5.99 MJₑₓ.kWh⁻¹), while the Greek supply mix has the highest CED (16.00 MJ.kWh⁻¹), resp. CEENE (19.22 MJₑₓ.kWh⁻¹), according to the ecoinvent database. The direct electricity input to the production of the germanium wafer and the direct electricity input to the production of the GeO₂ input account for almost 60% of the resource consumption in the base case. As hydrochloric acid is a major input to the GeO₂ production, it was also included in the scenario analysis by changing the direct electricity input to the hydrochloric acid production and also the upstream processes for chlorine production, as for the latter electricity input accounts for more than 60% of the resource consumption. Direct electricity input to the wafer production, together with direct electricity and hydrochloric acid input to the GeO₂ production represent approximately 75% of the resource consumption in the base case. The inclusion of cokes production, another important input to wafer production, respectively GeO₂ production, did not have much influence, because electricity is not an important input to cokes production and upstream processes like coal production. The resource consumption of cokes is dominated by the CED, respectively CEENE, of the raw material coal which accounts for approximately 90% of the total.

The location of installation is another parameter relevant for the EPBT and ExPBT of HCPV systems. The place of installation of the PV system determines on the one hand the irradiation input the modules receive and on the other hand the electricity production that is substituted. Thus, in a second step the importance of these two factors was studied by selecting a number of locations representing a range of average annual DNI and CEDₑₓ electricity, resp. CEENEₑₓ electricity values for the local non-renewable-based production mix. The net-electricity production has been estimated based on the base case where 2600 kWh is produced per year per installed nominal AC-power.

7.3 Results and Discussion

7.3.1 Resource consumption and payback times for the germanium wafer (base case)

The assessment of the germanium wafer production showed that the electricity input is the major cause of resource consumption (Table 7-2). The production of the GeO₂ from the leach residue is another important contributor, this is for a large part due to electricity input, but is also due to acid input to the process. The overall resource consumption per wafer is 216 MJ (97% non-renewable resources) for the CED method, respectively 258 MJₑₓ (79% fossil and nuclear resources) for the CEENE method. As already mentioned, CEENE values are typically higher than CED values, due to inclusion of additional resources and other system boundaries. The charts in Figure 7-4 show that fossil resources are the most important resources required for all stages of the germanium wafer production, the second most important being nuclear resources. This reflects for a large part the resource input for electricity production. The most important process stages in terms of resource consumption are the refining stages which require a substantial amount of energy for heating purposes.
Table 7-2 CED and CEENE results for the production of one wafer, split up into different processing steps.

<table>
<thead>
<tr>
<th>Process</th>
<th>Input per wafer (Ø 100mm)</th>
<th>CED</th>
<th>CEENE</th>
</tr>
</thead>
<tbody>
<tr>
<td></td>
<td></td>
<td>MJ</td>
<td>%</td>
</tr>
<tr>
<td><strong>Chlorination and Hydrolysis</strong></td>
<td></td>
<td></td>
<td></td>
</tr>
<tr>
<td>GeO₂ material</td>
<td>1.1E-02 kg</td>
<td>73</td>
<td>34.0</td>
</tr>
<tr>
<td>Electricity</td>
<td>4.8E-02 kWh</td>
<td>1</td>
<td>0.3</td>
</tr>
<tr>
<td>Auxiliaries</td>
<td>7.4E-01 kg</td>
<td>3</td>
<td>1.4</td>
</tr>
<tr>
<td><strong>CED/CEENE 1</strong></td>
<td></td>
<td>77</td>
<td>35.7</td>
</tr>
<tr>
<td><strong>Zone Refining</strong></td>
<td></td>
<td></td>
<td></td>
</tr>
<tr>
<td>GeO₂ material</td>
<td>1.0E-04 kg</td>
<td>1</td>
<td>0.3</td>
</tr>
<tr>
<td>Electricity</td>
<td>4.2E+00 kWh</td>
<td>54</td>
<td>25.1</td>
</tr>
<tr>
<td>Water</td>
<td>1.5E+01 kg</td>
<td>10</td>
<td>4.8</td>
</tr>
<tr>
<td>Auxiliaries</td>
<td>3.5E-02 kg</td>
<td>1</td>
<td>0.5</td>
</tr>
<tr>
<td><strong>CED/CEENE 2</strong></td>
<td></td>
<td>66</td>
<td>30.8</td>
</tr>
<tr>
<td><strong>Supportive Processes metal refining</strong></td>
<td></td>
<td></td>
<td></td>
</tr>
<tr>
<td>Electricity</td>
<td>5.7E-02 kWh</td>
<td>1</td>
<td>0.3</td>
</tr>
<tr>
<td>Water</td>
<td>2.0E-01 kg</td>
<td>0</td>
<td>0.1</td>
</tr>
<tr>
<td>Natural gas, cokes &amp; fuel oil</td>
<td>1.7E-02 kg</td>
<td>1</td>
<td>0.4</td>
</tr>
<tr>
<td>Auxiliaries</td>
<td>7.7E-02 kg</td>
<td>1</td>
<td>0.5</td>
</tr>
<tr>
<td><strong>CED/CEENE 3</strong></td>
<td></td>
<td>3</td>
<td>1.3</td>
</tr>
<tr>
<td><strong>Czochralski Ge crystal growth</strong></td>
<td></td>
<td></td>
<td></td>
</tr>
<tr>
<td>Electricity</td>
<td>2.8E+00 kWh</td>
<td>36</td>
<td>16.9</td>
</tr>
<tr>
<td>Auxiliaries</td>
<td>2.1E-01 kg</td>
<td>2</td>
<td>0.9</td>
</tr>
<tr>
<td><strong>CED/CEENE 4</strong></td>
<td></td>
<td>38</td>
<td>17.8</td>
</tr>
<tr>
<td><strong>Wafer manufacturing</strong></td>
<td></td>
<td></td>
<td></td>
</tr>
<tr>
<td>Electricity</td>
<td>4.4E-01 kWh</td>
<td>6</td>
<td>2.6</td>
</tr>
<tr>
<td>Auxiliaries</td>
<td>2.0E-01 kg</td>
<td>9</td>
<td>4.1</td>
</tr>
<tr>
<td><strong>CED/CEENE 5</strong></td>
<td></td>
<td>15</td>
<td>6.7</td>
</tr>
<tr>
<td><strong>Cleaning &amp; Inspection</strong></td>
<td></td>
<td></td>
<td></td>
</tr>
<tr>
<td>Electricity</td>
<td>7.4E-01 kWh</td>
<td>10</td>
<td>4.4</td>
</tr>
<tr>
<td>Natural gas, cokes &amp; fuel oil</td>
<td>3.0E-02 kg</td>
<td>2</td>
<td>0.8</td>
</tr>
<tr>
<td>Auxiliaries</td>
<td>3.3E-01 kg</td>
<td>5</td>
<td>2.5</td>
</tr>
<tr>
<td><strong>CED/CEENE 6</strong></td>
<td></td>
<td>17</td>
<td>7.7</td>
</tr>
<tr>
<td><strong>Total resource input per wafer</strong></td>
<td></td>
<td>216</td>
<td></td>
</tr>
</tbody>
</table>

*a Auxiliaries typically include hydrochloric acid, nitrogen, oxygen and a number of other mainly inorganic materials*
Figure 7-4 CED results (A) and CEENE results (B) per production process cluster.

With equation 4, respectively 5, the EPBT and ExPBT of the base case have been determined as 0.01 years or 4.0 (EPBT) and 4.2 days (ExPBT). The relevant parameters are compiled in the following: Annual electricity yield at a DNI of 2500 kWh.m⁻² per year for a system with 6.2 kW_nom equals 16120 kWh of electricity produced. According to data extracted from
ecoinvent the CED\textsubscript{NR} electricity, resp. CEENE\textsubscript{NR} electricity, for electricity produced from fossil and nuclear resources in the USA is 13.3 MJ.kWh\textsuperscript{-1}, resp. 15.4 MJ\textsubscript{ex}.kWh\textsuperscript{-1}. For one system 11 wafers have to be manufactured.

The payback time results are similar, regardless whether all resource categories or only non-renewable, respectively only fossil and nuclear resources are considered. More visible difference can be expected when the electricity mixes of country of production and country of installation have differences in their profile. The EPBT of 4 days is considerably less than the 18 days of EPBT for a germanium wafer (Figure 7-5) that can be derived from the paper of Peharz and Dimroth (2005), even though they considered only electricity consumption in the germanium wafer production. However, Peharz and Dimroth reported an estimated electricity consumption of 100 MJ per wafer, which is about three times the electricity consumption in the production process of the present study. Indeed, there have been improvements in electricity consumption of the new installation via optimisation of the feed, the processes and the equipment. Moreover, it should be taken into account that the value of 100 MJ electricity per wafer was just an approximate value. An additional factor in the EPBT calculation by Peharz and Dimroth concerns the conversion of the electricity input to primary energy equivalents. The applied conversion factor was 3.1 MJ primary energy per MJ electricity (Peharz, 2010), resulting in a CED of 310 MJ, which lies 94 MJ above the here calculated CED per wafer. Also, the annual irradiation in Southern Spain is lower than in the Southwestern USA. However, based on the available data also the number of wafers required for one HCPV-system per aperture area is lower. Possible explanations for this reduced requirement of wafers include an increased yield during the PV cell production and a higher concentration factor. An increased yield would imply that less germanium wafer material is lost, while a higher concentration factor would result in less cell area per aperture area and thus a reduced need of germanium wafer material in the application.

Figure 7-5 CED values for components of a HCPV system installed in Spain based on Peharz and Dimroth (2005).
7.3.2 Approximated resource consumption and EPBT of HCPV system

Based on the CED values for other system components, like inverter, lenses, frame and MOVPE reported by Peharz and Dimroth (2005) (Figure 7-5) and the here calculated CED of the germanium wafer required for a system an overall EPBT can be estimated. The HCPV system studied by Peharz and Dimroth and the one studied here have a comparable nominal power and aperture area. Taking into account the differences in aperture area results in an overall CED of 87 GJ, this value is obtained by taking the CED of all inputs from Peharz and Dimroth, except for the germanium wafer CED, multiplying this with a factor of 1.125 which is the ratio of the aperture areas of the two systems, and adding the germanium wafer CED from this study. With the here assumed base case annual DNI of 2500 kWh.m\(^{-2}\) and CED of replaced electricity of 13.3 MJ.kWh\(^{-1}\), an overall EPBT of 0.4 years is calculated. Based on the detailed inventory for the germanium wafer production performed for this study it can be concluded that the wafer accounts for only 3% of the total CED of the HCPV system due to the small amount of wafer needed. In the original study of Peharz and Dimroth the share of the germanium wafers of the total CED was 6%.

An energy yield ratio can be calculated based on the guaranteed system lifetime. Concentrix Solar gives a system guarantee of 20 years which is the same as conventional PV systems. The same HCPV system lifetime is also assumed by Nishimura et al. (2010) for all system components except for the inverter, for which they assumed a lifetime of 15 years. Taking a lifetime of 15 years for the converter and 20 years for the rest of the components and assuming no degradation results in an energy yield ratio of 49 MJ per MJ invested. Richards and Watt (2007) report energy yield ratios between 3 and 14 for small silicon based systems, albeit under conditions of lower irradiation.

It should be noted that in the original study of Peharz and Dimroth production was assumed to take place in Germany and that the total CED includes transportation of the system from Germany to Spain and that the total CED has not been adjusted in above calculations. Also, it is likely that in the production of the other system components efficiency gains have been achieved. This may lead to an overestimation of the overall EPBT. The calculated overall EPBT is 0.4 years for an installation in Southwestern USA. This is a lot lower than the EPBT of 0.86 years reported for the Amonix system for which installation also was assumed to be in the Southwestern USA (Fthenakis and Kim, 2010). The discrepancy cannot only be explained by the germanium wafer production process however, as the germanium wafer does not contribute enough to the overall EPBT.

7.3.3 Scenario analysis: Variation of results in response to changing location of germanium wafer production and site of HCPV system installation

As described in a previous section, direct electricity inputs to the production process, the upstream GeO\(_2\) material production process and the hydrochloric acid and chlorine production have been varied from lowest CED, respectively CEENE value (4.71 MJ kWh\(^{-1}\), 5.99 MJ\(_{ex}\) kWh\(^{-1}\), Norway) of supplied medium voltage electricity available in ecoinvent 2.1 to highest available (16.00 MJ kWh\(^{-1}\), respectively 19.21 MJ\(_{ex}\) kWh\(^{-1}\), Greece).

Changing the electricity mix for the germanium wafer production to the mix supplied in various countries has a considerable effect on the overall CED, resp. CEENE value per wafer.
The CED varies between 122 and 253 MJ per wafer, while the CEENE varies between 155 and 308 MJ_per_wafer. For the selection of countries half the CED values lie between 181 MJ and 208 MJ, respectively half the CEENE values lie between 218 and 243 MJ. Thus the values are clustered in the midfield. The base case results for the USA are higher than the midfield range.

For the EPBT (equation 4), respectively ExPBT (equation 5), also the place of installation of the PV system is of relevance. To examine the impact of the factors of average annual DNI and the local conventional electricity supply a number of countries, respectively regions have been selected that represent some variation on both variables (Table 7-3). Other factors that also play a role (wind speed, dust settling on the lenses and maintenance) have not been taken into account. The selected average annual DNI ranged from 1000 kWh.m² to 2500 kWh.m², while the resource consumption for providing one kWh from the current local electricity production from fossil and nuclear resources varied from 11.5 MJ to 13.3 MJ and from 11.0 MJ to 18.1 MJ, respectively.

Table 7-3 Representation of the countries/regions chosen for installation of the solar system and the used DNI values and CEENE and CED values of conventional non-renewable-based electricity production.

<table>
<thead>
<tr>
<th>Country/Region</th>
<th>Average annual DNI</th>
<th>Electricity (conventional production)</th>
<th>CEDNR electricity</th>
</tr>
</thead>
<tbody>
<tr>
<td></td>
<td>(kWh.m²)</td>
<td>CEENENR electricity</td>
<td>CEDNR electricity</td>
</tr>
<tr>
<td></td>
<td></td>
<td>nuclear &amp; fossil resources %</td>
<td></td>
</tr>
<tr>
<td>Poland</td>
<td>1000</td>
<td>17.5</td>
<td>13.3</td>
</tr>
<tr>
<td>Belgium</td>
<td>1000</td>
<td>11.2</td>
<td>11.5</td>
</tr>
<tr>
<td>Southern France</td>
<td>1500</td>
<td>11.8</td>
<td>13.1</td>
</tr>
<tr>
<td>Southern Spain</td>
<td>2100</td>
<td>11.0</td>
<td>11.6</td>
</tr>
<tr>
<td>Southwestern USA</td>
<td>2500</td>
<td>15.4</td>
<td>13.3</td>
</tr>
<tr>
<td>Northern USA</td>
<td>1100</td>
<td>15.4</td>
<td>13.3</td>
</tr>
<tr>
<td>Eastern China</td>
<td>1500</td>
<td>18.1</td>
<td>12.5</td>
</tr>
<tr>
<td>Western China</td>
<td>2500</td>
<td>18.1</td>
<td>12.5</td>
</tr>
</tbody>
</table>

In Figure 7-6 the results of the scenario analysis are displayed. The horizontal axis shows the CED, resp. CEENE, per germanium wafer, whose variation results from the variation of the direct electricity input used in the production of the germanium wafer, GeO₂ production, hydrochloric acid production and chlorine production. The vertical axis shows the calculated partial EPBT, resp. ExPBT in days. The lowest EPBTS resp. ExPBTs, are obtained for production of the wafer in Norway and installation in Southwestern USA, resp. Western China. It should be kept in mind though that transport of components to the site of installation was not included in the calculations. While installation in Southwestern US is represented by the lowest line in the EPBT graph, installation in Western China takes this place in the ExPBT graph. This switch has to do with the different profile of the electricity mix. The Chinese production mix is dominated by production from fossil resources, this is also the case


for the US mix, but the share of production from nuclear resources is much higher for the latter mix. Figure 7-6 also shows that the EPBT, respectively ExPBT, increase is steeper for lower average annual DNI at the same substituted electricity (USA and China) and for lower resource consumption in the production of the substituted electricity at the same average annual DNI (compare Belgium and Poland) because the slope is the reciprocal of the CED resp. CEENE recovered per day. The more CED resp. CEENE is recovered per day, the flatter does the slope become and the less important does the location of production become. The range of EPBT, resp. ExPBTs, is quite large: from approximately two days up to 14, respectively 17 days. For constant electricity input to the germanium wafer production the range is somewhat lower, but still the magnitude of the highest value is several times the magnitude of the lowest value. This highlights the importance of installing HCPV systems at locations with a high annual solar irradiation, and the relevance of the local electricity mix. Additionally, a distinction can be made between renewable resource and non-renewable resource consumption. Principally, payback times could also be defined for specific resources, like water, or other environmental interventions. This can give additional insight into the suitability of a location for PV systems. Two graphs of additional results of the scenario analysis are available in the Appendix 3 section A.3.1.

7.4 Concluding remarks

Renewable resources will become more and more important for our energy supply. However, no technology is 100% renewable. For harvesting solar energy costly advanced materials are needed. For the production of germanium wafers a high amount of electricity input is required for only a small mass output. Nevertheless, developments like HCPV, which makes it possible to use a smaller amount of the advanced material, can vastly improve the balance or resource consumption versus resource saving. For PV-systems, but also other energy technologies making use of renewable resources, the benefit of their installation does for a large part depend on local conditions. In this chapter it has been shown that the EPBT, respectively ExPBT, varies considerably depending on the location of installation. These kinds of results can give a first impression on possible savings. However, for actual large scale application of PV-technologies the consequences for the local market would have to be modelled to a greater depth. For future research it might also be interesting to calculate payback times not only for resource consumption, but also for other environmental impacts.
Figure 7-6 Partial EPBT (A) and partial ExPBT (B) in function of the location of the installation of the PV system, determining the average annual DNI and the resource consumption profile of the substituted electricity, and in function of the CED, resp. CEENE value per wafer, which varies according to the direct electricity input germanium wafer production, GeO$_2$ production, hydrochloric acid production and chlorine production.
Resource demand for the production of different cathode materials for lithium ion batteries

Abstract. With the expansion of the Li-ion battery market, new materials for Li-ion cathodes are constantly being developed. The main objectives of the presented study were to quantify the natural resource use of the production and recycling of cathode materials for Li-ion batteries based on data directly provided by industry and to assess the impact of differences in composition, cathode material properties and production technology. The availability of industry data made it possible to compile relatively detailed inventories for recycling of cobalt and nickel from waste Li-ion batteries, the production of precursors and the cathode material production. The CEENE method was employed to assess the cumulative resource use for the production of five cathode materials. As functional unit one kWh of dischargeable energy supplied over the cycle life of the cathode was chosen. In addition, the results were compared per kg of cathode and per kWh supplied during one cycle to get a better view on the impact of the property differences of the cathode materials. The results per kg of cathode showed that natural resource use was comparable for all (290-346 MJ\textsubscript{ex}/kg) but one (622 MJ\textsubscript{ex}/kg) of the cathode materials. The high resource use for this cathode material was caused by different process conditions in the cathode material production. Overall the metal supply and the energy use during cathode material production were the main drivers of natural resource use. Due to the diverging characteristics of the cathode materials the relative results in terms of the functional unit (0.39-0.70 MJ\textsubscript{ex}/kWh) differed considerably from the results per kg of cathode material. For example, while the resource use for one of the cathodes was relatively high per kg of material, it was similar to the resource use of two other cathode materials per kWh (cycle life). One of the cathodes exhibited low resource use compared to the other four when the resource use was expressed per kg (347 MJ\textsubscript{ex}), and per kWh (cycle life) (0.39 MJ\textsubscript{ex}). Two of the cathode materials were developed with the target to reduce cost of feedstock metals while maintaining performance. Indeed, those two cathodes showed low resource use per kg (290-343 MJ\textsubscript{ex}) and per kWh (one cycle) (377-463 MJ\textsubscript{ex}).

Keywords: lithium ion battery; cathode; cumulative exergy; process inventory; life cycle assessment

8.1 Introduction

By now the Li-ion batteries and lithium polymer batteries make up the large majority of the rechargeable battery market (Goonan, 2012). For quite a while now Li-ion batteries have been employed in mobile devices like laptops and mobile phones. In Europe mobile phones and portable PCs are almost exclusively equipped with Li-ion batteries, while nickel metal hydride batteries still power two thirds of the power tools (2009) (Avicenne, 2010). On top of powering mobile equipment they are also more and more employed in (hybrid) electric vehicles, which might be a “promising technology” (Grünig et al., 2011) for reducing...
greenhouse gas emissions. Currently electric vehicles are supported by policies around the world (Lindquist and Wendt, 2011) to boost their market penetration. Electric vehicles still only present a small share of global vehicle sales, but the sales of electric vehicles have doubled between 2011 and 2012 (IEA and EVI, 2013). While lead-acid and nickel metal hydride batteries are still important in this market, Li-ion batteries are expected to being the dominant technology in this market by 2017 (RECHARGE, 2013). Another field of application for Li-ion battery technology are energy storage systems, which will be required to facilitate the balancing of electricity generation and load when the share of electricity production from is increasing further (Ferreira et al., 2013).

A Li-ion battery consists of one or more cells. Figure 8-1 depicts a cylindrical cell. Cells may also be produced in a prismatic form. The two electrodes, made up of active material, conductive material and a binder, are connected via an electrolyte, which allows ions to move between the electrodes (Ketterer et al., 2009). The electrodes are electronically separated by a membrane. During discharge Li-ions move through the electrolyte from the anode to the cathode. At the same time electrons move from the anode to the cathode via the external circuit. On charging the movement occurs in the opposite direction.

Figure 8-1 Schematic drawing of a cylindrical Li-ion battery cell (copyright Umicore).

As mentioned above Li-ion batteries are involved in technological changes that are at least to some extent intended to reduce humanities impact on the environment by reducing our dependence on fossil fuels. As the processing of metals, which are constituents of many battery components, is typically quite energy intensive, the question of the resource use in their production is quite important. Indeed, a number of life cycle studies have been performed already with Li-ion batteries as main target (Dunn et al., 2012; Gaines et al., 2011; Majeau-Bettez et al., 2011; Notter et al., 2010; Zackrisson et al., 2010) or as component in an application (Rydh and Sandén, 2005; Samaras and Meisterling, 2008). According to some of those studies the cathode, which in addition to the active material also contains some other materials, is an important component of a Li-ion battery also with respect to environmental impacts. According to Majeau-Bettez et al. (2011) the cathode paste production causes more than 35% of the total global warming impact of a Li-ion battery, while in the study of Notter et al. (2010) the cathode active material accounts for 12.5 % of the CED and 13.8 % of the GWP of the Li-ion battery. Unfortunately, industry data for the modelling of the LCIs of
an anode and cathode materials is usually lacking so that the inventories are approximated based on patents and process descriptions. Regarding battery recycling some more industry data are available, which have been used to model LCIs (Dunn et al., 2012; Fisher et al., 2006). These data are not very complete though or important treatment steps to obtain a material which can be used again for Li-ion cathode production are missing.

The first and foremost objective of the study was to establish gate-to-gate inventories of production processes for Li-ion battery cathode active materials based on primary industry data. In the following the term cathode material will be used to refer to the cathode active material. As new cathode materials are constantly developed, five different cathode materials were selected representing different stages of development and having different properties. In addition, industrial Li-ion battery recycling processes, recovering usable nickel and cobalt products, were to be inventoried. Subsequently, the cumulative natural resource use to manufacture the Li-ion cathode materials from these recovered nickel and cobalt streams was to be quantified. No specific battery application was targeted due to the diverse nature of the cathode materials. Therefore, no application specific functional unit has been chosen. Nevertheless, a functional unit was to be selected which included at least the more generic characteristics of the cathode materials in order to capture aspects with respect to the service delivered by the various chemistries. The different cathode materials were to be compared, while keeping in mind their different stages of development and slight differences in application. In parts this study is an update and extension of a previous paper by Dewulf et al. (2010), which dealt with only one cathode material and only quantified resource use per kg of material.

### 8.2 Methodology

#### 8.2.1 Cathode materials

In collaboration with an industrial partner, a producer of cathode materials and recycler of Li-ion batteries, five cathode materials were selected for this assessment (Table 1). These five cathode materials will be denoted as cathode 1, cathode 2 etc. New cathode materials are constantly developed. This is reflected in the selection, which encompasses cathode materials at different stages of development, including commercial production. This meant that the data collected for some of the cathodes were representative for full scale production, whereas the data (properties and process inventory) with respect to the other cathode materials may still change when full production is reached and processing has been optimized.

The selected materials varied considerably in composition and properties (Table 8-1) and also differed in their target applications. All but one of the cathode materials had a layered crystal structure. Cathode 3 had a spinel structure instead. In layered structures ions can diffuse along two dimensions, while in spinel structures three dimensions are available (Yoshio and Noguchi, 2009). The dischargeable energy was determined under standardized conditions with a C-rate of 0.2C and 100% depth of discharge, i.e. the battery is discharged completely over five hours. Cycle life here means the number of charge and discharge cycles a standardized battery using the specified cathode material can undergo before it reaches 80% of its initial capacity. Cycle life was also determined at 0.2 C. Due to its high volumetric energy density cathode 5 is particularly suitable for portable applications for which small battery size is very important. For automotive applications batteries have to supply much higher amounts of energy in order to have a sufficient driving range. This requires more
cathode material. In order to limit costs, cathode materials with lower cobalt content are developed for automotive applications. Cathodes 3 and 4 are examples of this trend. For those cathode materials the aim was to reduce the amount of expensive feedstock materials required, while still attaining a high performance. The cathodes had relatively higher manganese and lower cobalt and nickel contents than the other cathode materials and at the same time offered high energy densities. Additionally, some of the cathode materials could be operated at a higher voltage compared to the others. Cathode 2 was similar to cathode 1, but was designed with a focus on achieving a higher energy density than cathode 1, while keeping the costs similar.

Table 8-1 Data on the selected cathode materials assessed in this study.

<table>
<thead>
<tr>
<th>Application</th>
<th>Type</th>
<th>Global shipments (‘000 t)</th>
<th>Dischargeable energy (Wh/kg) (0.2C, room temperature)</th>
<th>Cycle life (0.2 C, room temperature)</th>
<th>Comment</th>
</tr>
</thead>
<tbody>
<tr>
<td>Cathode 1 Portable electronics,</td>
<td>NMC-layered</td>
<td></td>
<td></td>
<td></td>
<td>High cycle life</td>
</tr>
<tr>
<td>automotive</td>
<td></td>
<td></td>
<td></td>
<td></td>
<td></td>
</tr>
<tr>
<td>Cathode 2 Automotive</td>
<td>NMC-layered, high Ni</td>
<td></td>
<td></td>
<td></td>
<td>High cycle life</td>
</tr>
<tr>
<td>Cathode 3 Automotive</td>
<td>NMC-spinel, high Mn</td>
<td>24.6</td>
<td></td>
<td>540</td>
<td>1000 High voltage (&gt;4 V)</td>
</tr>
<tr>
<td>Cathode 4 Automotive</td>
<td>NMC-layered, high Mn</td>
<td></td>
<td></td>
<td>875</td>
<td>600 High gravimetric energy density; low cycle life (High density (&gt;3.5 kg/l))</td>
</tr>
<tr>
<td>Cathode 5 Portable electronics</td>
<td>LCO-layered</td>
<td>30</td>
<td></td>
<td>539</td>
<td>1000</td>
</tr>
</tbody>
</table>

8.2.2 Functional unit
A generic functional unit was selected because the cathode powder is only one part of the battery and as the five cathode materials have different target applications. As illustrated in Figure 8-2 other battery components (Ketterer et al., 2009; Zhang and Ramadass, 2013) and electricity are required in addition to the cathode active material to fulfil the function of delivering energy, but these additional elements were out of scope of this study. Nevertheless, the objective was to choose a functional unit and not express resource use in terms of cathode powder mass only, because the selected cathode materials clearly have different specifications, which are relevant for their functional value in a battery. The functional unit is defined as 1 kWh of dischargeable energy supplied by a Li-ion battery over its cycle life. Cycle life is defined as the number of charge and discharge cycles the cathode can undergo in a standardized battery design before the capacity has dropped to 80% of its initial value. The energy delivered over the cycle life has been determined as follows: energy delivered over cycle life = capacity × voltage × cycle life × 0.9. The factor 0.9 was used to model the drop in
capacity over the cycle life from 100 to 80%. Thus it is assumed that the average capacity is equal to $0.9 \times$ capacity.

![Diagram of a Li-ion battery](image)

**Figure 8-2 Cathode powder on its own, cannot actually fulfil a function. Other battery components and an electricity supply to charge the battery are needed in order to provide energy in an application.**

Capacity, voltage and cycle life were determined under standardized conditions, with a C-rate of 0.2C, i.e. the battery is discharged completely over five hours, for cathodes 1 to 4. As cathode 5 is typically used in portable electronics, only test data for a portable electronics design were available, which was not applicable for the majority of the cathodes. Therefore, the behaviour of cathode 5 in a standardized design as for the other cathodes was estimated by industry experts based on extensive laboratory data. The given energy density is that part of the stored energy that is available for use in practice. The influence of battery components other than the cathode material and of charge/discharge conditions on capacity, cycle life and other battery properties was controlled by employing standardized conditions. It should be noted that in an application the battery design would be optimized for the specific system, which can have a strong impact on cycle life and overall performance.

A similar functional unit was chosen by Majeau-Bettex et al. (2011) for the battery as a whole, though they did not consider the drop in capacity. In addition, Majeau-Bettex et al. assumed the cycle life for the NMC type cathode to be 3000 cycles, which is much higher than cycle lives used in this study. This might be related to different conditions for the determination or another definition of cycle life. When selecting a cathode material for a specific application, it has to be kept in mind that energy delivered over the cycle life is not the only relevant factor for the performance of a battery. Voltage, gravimetric energy density and volumetric energy density of the battery also have to be considered in order to optimize performance. As the specific application was out of scope of this study, however, not all the factors that might be relevant for a specific application could be taken into account.

### 8.2.3 Exergy and CEENE method

The thermodynamic unit of exergy (Szargut, 2005) was used in this study for two purposes. (1) During the modelling of multi-output processes usually exergy of intermediates or products was used for allocation purposes. (2) For the assessment of natural resource use the CEENE method developed by Dewulf et al. (2007) was applied. The method evaluates natural resource extraction with eight categories: Renewable resources (excl. biomass), fossil energy,
nuclear energy, metal ores, minerals and mineral aggregates, water resources, land occupation and transformation (incl. biomass) and atmospheric resources. For each of these categories the cumulative resource extraction is quantified and expressed in MJ of exergy. The result is a resource fingerprint.

### 8.2.4 Closed loop production scenarios

This study focused on the production of cathode material and subsequent recycling of cathode material contained in a battery, including transport of intermediates. Battery manufacture, use phase as well as collection and sorting of end-of-life batteries were not included. In a closed loop scenario it was assumed that all cathode material which had been produced also entered the recycling. In the recycling processes nickel and cobalt were recovered from the cathode. Figure 8-3 shows an overview of the nickel and cobalt flows considered in this study. Losses taken into account (<7% of input) occur during cobalt refining and nickel purification. In the model the losses were substituted by input of primary cobalt and nickel. All lithium and manganese inputs were considered to be originating from primary resources as they are currently not recovered in the recycling processes under study.

The data available for the modelling of primary metal production came mainly from literature or databases (see Table 8-2 for details). The available detail of this data was lower than for the industry data used for the recycling of cobalt and nickel and consequently also assumptions could not be adapted on the same level.

![Figure 8-3 Simplified scheme of the nickel and cobalt flows in the closed loop scenario. Stages with a dashed border were out of scope of this study. Lithium and manganese were always considered to be provided from primary resources irrespective of the scenario.](image)

### 8.2.5 Process inventories

Industry data were available for the precursor production, the cathode material production and the recycling processes. For metal production from virgin resources and background processes it was necessary to rely on other data sources. For background processes usually ecoinvent v2.2 (Swiss Centre for Life Cycle Inventories, 2010) datasets were used. Additional information on background processes can be found in Appendix 3 section A.3.5.
**Industry data**

The production of the cathode materials consists of two major processes. The first process is the precursor production. The precursor is an intermediate in the production of the active cathode material. The selected cathode materials are produced from different types of precursors containing the transition metals cobalt, nickel and manganese. The feedstock transition metals are either provided in metallic form, as a salt or as a salt solution. The precursor of cathode 5 is produced via a gas phase process, whereas the other precursors are produced via liquid phase processes. In both cases removal of impurities remaining in the feedstock is very important for the quality of the final cathode material product, as impurities can cause battery swelling and compromise safety (Guoxian and MacNeil, 2011) as well as performance. The cathode material production is principally a heat treatment and rather energy intensive. For precursor production and cathode material production primary data from industry were collected.

Next to inventories of the cathode production, inventories of battery recycling processes were compiled. This included the smelting of Li-ion batteries (Van der Vorst 2011, internal report). This pyrometallurgical process results in an alloy containing among others nickel, cobalt and copper. The inputs and outputs of the smelter were based on a model for an NMC-layered battery. In practice the feed to the smelter is always a mix of batteries of different compositions. In the study it was assumed that the NMC-layered battery represented the average feed provided to the smelter. The results showed that the smelter only contributes a minor part to the overall resource use of the closed loop system, so this assumption was not expected to have a considerable effect on final results. No burden was allocated to the waste batteries supplied to the smelter. The auxiliary inputs to the smelter were allocated to the valuable metals recovered in the alloy (Co, Ni, and Cu) on a mass basis.

The alloy from the smelter is one of the metal feeds to a cobalt refining process which recovers a CoCl$_2$ solution, NiCO$_3$ and a copper residue. The latter goes to further treatment for recovery of copper. For the cobalt refining process data provided by the industrial partner for the year 2010 were used to model the inventory. The quantities of all auxiliary inputs were available at sub-process level, except for the electricity input, for which data were only available for the process as a whole. The cobalt refining is a hydrometallurgical process, which means that the chemical requirements are typically much more important in terms of resource inputs than energy/electricity requirements. Inputs were allocated on copper, nickel and cobalt leaving the process in a non-waste stream. Allocation was performed by subdividing the overall process and using the exergy content of the cobalt, nickel and copper containing streams leaving the sub-processes to calculate allocation factors per sub-process. In this way inventories, excluding the metal feedstock, have been set up for 1 kg of Ni (in NiCO$_3$), 1 kg of Co (in CoCl$_2$ solution) and 1 kg of Cu (in residue).

In a subsequent process step NiCO$_3$ is transformed into NiSO$_4$ crystals and a minor amount (2.5%) of NiSO$_4$ solution. The NiCO$_3$ is only part of the nickel feedstock entering this process. As in the case of the cobalt refining process the source of the nickel feed was neglected when establishing an inventory of the process which recovers 1 kg of nickel (in NiSO$_4$ crystals). The overall resource use for recycled nickel (in NiSO$_4$.6H$_2$O crystals) was obtained by adding up the inventories for obtaining Ni (in alloy) from waste Li-ion batteries, for separating Ni (in NiCO$_3$) from the alloy and for recovering Ni (in NiSO$_4$) in the nickel purification. For the sub-inventories the metal feedstock was disregarded and the nickel losses
at the cobalt refining and the nickel purification were taken into account when adding up the
sub-inventories.

The inventory of cobalt (in CoCl₂ solution) was established in a similar fashion. Some of the
precursor production processes do not use CoCl₂ solution as an input but CoSO₄·H₂O crystals.
Due to lack of data the process for converting CoCl₂ to CoSO₄ crystals was not taken into
account in line with the recommendation to accept a data gap if information of suitable quality
is not available (EC-JRC, 2010a). In the closed loop scenario all recycled metal input to the
precursor production was assumed to be in the form of sulphate crystals or solution (cathode 5).

Secondary data for virgin metal feedstock
In the closed loop scenarios more than 93% of nickel and more than 98% of cobalt were
recovered from the recycling processes. Therefore the selection of the inventory for the nickel
and cobalt feedstock derived from virgin resources was only of minor importance. In contrast
to nickel and cobalt, manganese was not recovered to a refined metal stream in the analyzed
recycling processes. Table 8-2 gives an overview of the data used to establish life cycle
inventories for the various metal inputs. A more in depth discussion can be found in the
following paragraphs.

Table 8-2 Datasets for primary nickel, cobalt and manganese products used in the various
scenarios.

<table>
<thead>
<tr>
<th>Short description</th>
<th>Reference</th>
</tr>
</thead>
<tbody>
<tr>
<td>Co metal</td>
<td>Mining of copper-cobalt ore and production of cobalt hydroxide at Ruashi mine, electrefining to cobalt (Ecobalance, Inc., 2000; Metorex Limited, 2011; Peek et al., 2009; Roomanay and Gediga, 2011)</td>
</tr>
<tr>
<td>CoSO₄</td>
<td>Leaching of cobalt metal at site of use. Industry data (see text)</td>
</tr>
<tr>
<td>Ni metal</td>
<td>Production of class I nickel metal from mine to final product. (Ecobalance, Inc., 2000; Nickel Institute, 2012)</td>
</tr>
<tr>
<td>MnSO₄</td>
<td>Manganese concentrate partially reduced, leached, purified and crystallized. (SRK Consulting China Ltd et al., 2010; Swiss Centre for Life Cycle Inventories, 2010)</td>
</tr>
</tbody>
</table>

Although the ecoinvent dataset for nickel is probably reasonable to use when not assessing
emissions (Classen et al., 2009), an LCI dataset from a study commissioned by the Nickel
Institute (Ecobalance, Inc., 2000) and updated in 2003 (Nickel Institute, 2012) was used
instead as it was based directly on data supplied by producing companies. The LCI contained
the most important elementary flows relevant for the CEENE method with the exception of
those related to land use.

In the existing precursor production, 85% of the nickel was provided as nickel metal, which
was leached, and 15% of the nickel was provided to the process in the form of nickel sulphate
crystals. For the closed loop it was assumed that the required 6.5% of primary nickel were
provided as metal and leached.

For cobalt a dataset was compiled based on a carbon footprint study for the Ruashi mine
(Metorex Limited, 2011; Roomanay and Gediga, 2011). This concerned only one mine and
the inventory was rather limited as the report itself only accounted for diesel and electricity
inputs. To supplement diesel and electricity inputs at the Ruashi operation a stoichiometric
amount of magnesia, used for the precipitation of cobalt hydroxide at Ruashi, was
inventoried. Additional electricity requirements for electorefining of cobalt were estimated based on Peek et al. (2009). On the one hand the electricity requirements might be underestimated due to not including inputs for the re-dissolution of the cobalt hydroxide, on the other hand an overestimation is possible as the data compiled by Peek et al. (2009) are more than 15 years old, with the likely exception of one operation, which is among those with relatively low electricity consumption. Cumulative water inputs were assumed to be the same as accounted for nickel in the study of the Nickel Institute. Land use was not included. As the data were not based on several mining operations, the resulting dataset cannot be regarded as being of high quality, but at least it represented a possible route for cobalt production and made it possible to control assumptions regarding allocation. Though there is an ecoinvent dataset it was not chosen as it was based on the data for nickel mining and beneficiation (Hischier, 2007) and the documentation in the report is limited.

As in the case of nickel, no dataset was available for cobalt sulphate salt, which was used in the precursor production of most of the cathode materials. The assumption was that cobalt sulphate was obtained via leaching of cobalt on the site of use. This was actually the practice for the primary metal supply to the precursor production of cathode 5 and for all other precursors the major part of the nickel was also leached. Thus, the assumption of leaching cobalt metal was plausible. Electricity consumption for leaching was based on data provided by industry. Steam requirements for heating were not taken into account, to avoid double counting with the subsequent heating step of the precursor production of cathodes 1-4. A stoichiometric amount of sulphuric acid was considered.

The virgin production of manganese is less resource intensive than in the case of nickel or cobalt, as the mined ores typically have a higher grade. There was no dataset for the production of manganese sulphate available in ecoinvent. On the basis of the manganese concentrate dataset from ecoinvent and information found in a technical report (SRK Consulting China Ltd et al., 2010) for a plant producing manganese sulphate monohydrate an inventory for the compound was approximated. Manganese ore is mixed with coal to reduce the MnO$_2$ to MnO. The reduction step is necessary as MnO$_2$ is not soluble in sulphuric acid. The reduced manganese oxide is subsequently leached and impurities are removed. Finally, manganese sulphate monohydrate is crystallized by heating the solution and evaporating the water (Pincock, Allen & Holt, 2008). Subsequently, the crystals are dried with hot air. At the plant an average purity of 98.45% was reached in 2009, compared to a purity of 97.5% of the manganese feed of the precursor production. As ecoinvent was used for the manganese ore land use of mining is included.

Data concerning the process inventories for the virgin nickel, manganese and cobalt products can be found in Appendix 3 sections A.3.2 to A.3.4.

### 8.3 Results and discussion

#### 8.3.1 Aggregated process inventories for cathode production and recycled metals

Table 8-3 shows the inventoried streams for the precursor and cathode material production per kg of cathode material. Inputs required for transport between production sites are not included in this table. The total metal content, which included the lithium and manganese, was similar for all of the cathode materials, the only other element contained in the materials being
oxygen. During the heat treatment of the cathode material production heat was provided via electricity. Though cathode 2 was an NMC-layered material, like cathode 1 and cathode 4, it required much more electricity due to specific process conditions during the cathode material production. Thus it seems not straightforward to estimate heat requirements in production based only on some basic properties of the cathode material. It is also notable that the electricity requirements for the heat treatment were much higher than the energy consumption inventoried by Majeau-Bettez et al. (2011), who only considered 0.55 MJ of heat supplied per kg of NMC-cathode material during the cathode material production. The industry data suggested electricity requirements in excess of 2.5 kWh (9 MJ) per kg of cathode - even 16 kWh/kg (57.6 MJ/kg) for one of the cathodes - for the heat treatment alone, not considering crushing, milling and other minor electricity inputs during cathode material production. Notter et al. (2010) inventoried 15.3 MJ of heat provided, which is in the order of magnitude of the firing electricity requirements inventoried in this study. Notter et al. (2010) based their energy requirements on specific heat capacities and the reaction enthalpy. In contrast, in the present study data was based on industrial production or on estimations based on pilot plant runs or laboratory experiments of the producer. An important factor for the energy requirements were the process conditions in the firing step, which as in the case of cathode 2 can increase energy requirements considerably. Notter et al. (2010) assumed that the process took place in a rotary kiln with heat supplied with natural gas, whereas in the process under study the oven was heated via electricity. This difference can have an effect on the cumulative resource consumption due to additional losses occurring during electricity production.

The data in Table 8-3 also show that the use of inorganic chemicals for cathode 5 was lower than for the other cathode materials, this was caused by use of chemicals during the precursor production. As cathode 5 was the only cathode material whose precursor was produced via a gas phase process, it is also the only one for which natural gas use was inventoried.

In Table 8-4 aggregated and allocated gate-to-gate inventories for the recycling processes up to a useable product are presented. Values presented here are sums of the allocated gate-to-gate inventories for smelter and the refining installation and as such do not include transport between those two. The tabulated nickel and cobalt input to the recycling per kg of recovered metal were based on the metal losses during the refining and purification. The inputs of electricity and organic chemicals were higher for recycled cobalt (by almost 70% and more than 50%, respectively) due to the higher exergy value of the cobalt stream leaving the cobalt refining process, which meant that a higher share of inputs was allocated to that stream. Natural gas use was slightly higher for nickel (5%) due to higher losses compared with the cobalt processing. Use of inorganic chemicals and water was higher for nickel (by 87% and more than 300%, respectively) due to the nickel precipitation step in the cobalt refining, of which all inputs could be allocated to nickel only and due to additional chemical input during the nickel purification, which converted the nickel carbonate stream into nickel sulphate.
Table 8-3 Aggregated gate-to-gate inputs for the precursor production and subsequent cathode material production for one kg of cathode material.

<table>
<thead>
<tr>
<th>Gate-to-gate inputs</th>
<th>unit</th>
<th>cathode 1</th>
<th>cathode 2</th>
<th>cathode 3</th>
<th>cathode 4</th>
<th>cathode 5</th>
</tr>
</thead>
<tbody>
<tr>
<td>Electricity</td>
<td>kWh</td>
<td>8.61</td>
<td>25.7</td>
<td>8.63</td>
<td>12.2</td>
<td>8.36</td>
</tr>
<tr>
<td>Steam</td>
<td>kg</td>
<td>4.33</td>
<td>4.40</td>
<td>4.53</td>
<td>3.35</td>
<td>1.64</td>
</tr>
<tr>
<td>Natural gas</td>
<td>kg</td>
<td>0</td>
<td>0</td>
<td>0</td>
<td>0</td>
<td>0.474</td>
</tr>
<tr>
<td>Inorganic chemicals</td>
<td>kg</td>
<td>0.916</td>
<td>0.937</td>
<td>0.968</td>
<td>1.16</td>
<td>0.205</td>
</tr>
<tr>
<td>Water</td>
<td>kg</td>
<td>20.0</td>
<td>19.9</td>
<td>20.9</td>
<td>33.0</td>
<td>21.6</td>
</tr>
<tr>
<td>Others (Compressed air, PE big bag, activated carbon, oxygen)</td>
<td>kg</td>
<td>5.89x10^{-4}</td>
<td>5.90x10^{-4}</td>
<td>5.97x10^{-4}</td>
<td>6.40x10^{-4}</td>
<td>4.90</td>
</tr>
<tr>
<td>Feedstock metals (elemental, in salts and solutions)</td>
<td>kg</td>
<td>0.670</td>
<td>0.671</td>
<td>0.650</td>
<td>0.673</td>
<td>0.674</td>
</tr>
</tbody>
</table>

- This does not include transport between the precursor production and cathode material production plants.
- Active compound
- Includes water from solutions
- This includes only metals actually used in the product

Table 8-4 Gate-to-gate inputs for smelting and subsequent refining of 1 kg of cobalt and 1 kg of nickel from batteries. The data from the smelter was compiled by Van der Vorst (2011, internal report) based on industry data.

<table>
<thead>
<tr>
<th>Gate-to-gate inputs</th>
<th>Co</th>
<th>Ni</th>
</tr>
</thead>
<tbody>
<tr>
<td>Electricity</td>
<td>3.68x10^8</td>
<td>2.17x10^9</td>
</tr>
<tr>
<td>Steam</td>
<td>8.09x10^9</td>
<td>7.84x10^9</td>
</tr>
<tr>
<td>Natural gas</td>
<td>9.21x10^{-2}</td>
<td>9.70x10^{-2}</td>
</tr>
<tr>
<td>Heat</td>
<td>1.76x10^8</td>
<td>1.85x10^8</td>
</tr>
<tr>
<td>Inorganic chemicals</td>
<td>7.12x10^8</td>
<td>1.33x10^1</td>
</tr>
<tr>
<td>Organic chemicals</td>
<td>1.76x10^{-2}</td>
<td>1.14x10^{-2}</td>
</tr>
<tr>
<td>Water</td>
<td>1.63x10^2</td>
<td>5.39x10^2</td>
</tr>
<tr>
<td>Others (Compressed air, PE big bag, activated carbon, oxygen)</td>
<td>9.59x10^9</td>
<td>1.94x10^1</td>
</tr>
<tr>
<td>Feedstock metal (in battery)</td>
<td>1.02x10^8</td>
<td>1.07x10^9</td>
</tr>
</tbody>
</table>

- Does not include transport between the smelter and refining site
- Active compound
- Includes water from solutions
- Battery itself was assumed to be burden free, transport of the battery was out of scope, only the amount of metal in question, i.e. Co or Ni, is reported here
8.3.2 Natural resource use

The results are not only presented in terms of the functional unit but also per kg of cathode material and per kWh dischargeable energy during one cycle after the first charging of the cathode. This is done to be able to compare with results reported elsewhere and to highlight the effects of composition and property differences.

In the bar chart of Figure 8-4 the overall CEENE value including transport per kg of cathode is displayed for all five cathode materials. There are some notable differences between the cathode materials when comparing the contributions of metal supply (lithium, cobalt, nickel and manganese), and other processing requirements of precursor production and cathode material production to the overall CEENE value. For some cathode materials the supply of the metal feedstock caused the highest overall natural resource use, while for other cathode materials other processing requirements were more important, especially due to considerable energy requirements of the cathode material production step. In the case of cathode 1 and cathode 5 the supply of feedstock metals caused more than 49% of the resource use. For all other cathode materials this share was lower. Generally, a higher nickel and cobalt content also meant a higher contribution of the metal supply to the overall CEENE. This is remarkable as more than 90% of cobalt and nickel are supplied via the recycling processes. It should be considered though that the recycling includes extended refining to cope with the diverse feed. Additionally, much more detailed data was available for the cobalt and nickel refining than for the manganese sulphate production from primary resources. The cathode material production step caused more than 50% of the natural resource use for cathode 2 and cathode 4, due to the high electricity consumption during firing and low nickel and cobalt content, respectively. Electricity use in precursor production and cathode material production caused between 34 and 60% of the overall CEENE. Thus, while it makes sense to reduce the amount of costly metals, the other processing requirements should not be disregarded, which is exemplified by cathode 2.

![Figure 8-4 Overall CEENE per kg split between CEENE due to providing metal feedstock and CEENE due to other processing requirements (transport included).](image)

With the exception of cathode 2 (622 MJex/kg), all cathode materials had a similar resource use for 1 kg of final product (290-347 MJex/kg). The much higher CEENE for cathode 2 is...
due to specific process conditions in the cathode production. In the preceding study by Dewulf et al. (2010) the CEENE consumption of a cathode material similar to cathode 1 was assessed. The CEENE values per kg obtained there in the recycling scenario are only a bit higher (387.4 MJ$_{ex}$/kg) than those obtained in this study (346.5 MJ$_{ex}$/kg) even though some aspects were modelled differently. From the information provided by Notter et al. (2010) it could be derived that the CED of fossil and nuclear energy (Althaus et al., 2009) for the lithium manganese oxide cathode active material was about 80 MJ/kg. CED values for elementary fossil and nuclear resource flows are not too much different from the corresponding CEENE values. For the cathode materials inventoried here the CEENE for fossil and nuclear resources varies from 250 to 480 MJ$_{ex}$/kg. This indicates a higher resource consumption than for the cathode material assessed by Notter et al. (2010). This is within expectations considering that heat was generated electrically in the process assessed in this study and by gas in the process modelled by Notter et al. and taking into account that the cathode materials assessed in this study contain nickel and cobalt next to manganese. In this context it should be noted that lithium manganese cathode material can currently not be used on its own in commercial Li-ion batteries due to capacity fading, which might be overcome by doping or blending with other cathode materials (Liu et al., 2013; Rao, 2013; Smith et al., 2012). Furthermore, as mentioned already in section 8.3.1 the inventory for the cathode material in Notter et al. (2010) is not based on actual industry data and is thus less likely to reflect actual resource consumption also due to incompleteness. Unfortunately, Majeau-Bettez et al. (2011) do not report CED values for the cathode active material.

Comparisons between the resource fingerprints of five cathode materials are shown in Figure 8-5 per kg of cathode, per kWh (one cycle) and per kWh (over cycle life). The cumulated inputs extracted from nature were converted into exergy and grouped into the eight categories as described in section 8.2.3. For all cathode materials the extraction of fossil resources contributes the highest share (> 50%) to the total CEENE value. This is typical for many industrial processes. Due to the high contribution of nuclear energy to the electricity mix of Korea, where the cathode materials are produced from the precursor, the nuclear resource use is also rather high (International Energy Agency, 2011). Water has a rather low exergy value per unit mass, but is often used in large amounts in industrial processes, especially in hydrometallurgical ones. In fact, water is to some extent reused in other operations on some of the production sites. The collection of detailed data for this, however, was out of scope of the study. As the CEENE is similar per kg of cathode material for all but one of the cathode materials the variation visible in the bottom chart of Figure 8-5 is mainly due to the different properties of the cathode materials.

The aim of the development of cathode 3 and cathode 4 was the reduction of the use of the more expensive metals nickel and especially cobalt, while maintaining a high energy density. Indeed, the absolute CEENE contributions due to metal supply were below 140 MJ$_{ex}$/kg for both cathodes, while they were above 170 MJ$_{ex}$/kg for the other cathode materials (Figure 8-4). When the CEENE results were expressed per kWh supplied during one cycle (Figure 8-5), both cathode materials even exhibit the lowest CEENE values (463 and 377 MJ$_{ex}$/kWh) of all cathode materials. The overall CEENE value of cathode 3 (0.51 MJ$_{ex}$/kWh cycle life) was still the second lowest when looking at the results in terms of kWh of energy supplied over the cycle life, but it was considerably higher than the CEENE value of cathode 1 (0.39 MJ$_{ex}$/kWh cycle life). Cathode 4 even had a CEENE value of 0.70 MJ$_{ex}$/kWh cycle life, which was almost double the CEENE value of cathode 1. Thus, while the high energy density of cathode 4 led to relatively low resource consumption per kWh supplied during the initial
charge and discharge cycle, the low cycle life compared to cathode 1 meant that the resource consumption relative to cathode 1 was increased considerably. This could change if the cycle life of cathode 4 was improved. Remarkably, the resource use of cathode 2, which was the highest per kg and per kWh supplied in one cycle, had a CEENE value per kWh supplied over the cycle life similar to the CEENE values of the other cathode materials with the exception of cathode 1. This was due to its high cycle life compared to the other cathode materials. Cathode 1 came out best with regard to resource use per kWh supplied over the cycle life. The resource consumption of cathode 5 was high compared to the resource consumption of cathode 1, this might be explained by the material being specifically targeted at portable electronics (see Table 1) for which specific battery designs are used, which affects performance of the cathode, and for which a high volumetric energy density is of utmost importance.

As the data for some of the cathode materials were not based on industrial production, while the data for other cathode materials were, there is obviously some uncertainty involved when comparing the CEENE results. This should be limited though due to the process data relying on expert judgment of the producer. An additional source of uncertainty with respect to the comparison is the difference in the quality of the data that was available for the precursor production of cathode 1 on the one hand and the other cathodes on the other hand. The uncertainties caused by the allocation necessary at the recycling processes are limited as cobalt is by a large margin the main output of the process also in terms of mass - only 166 g of nickel (in nickel carbonate) and 79 g of copper (in residue) are produced per kg of cobalt (in cobalt chloride) – and 60% of the resource use allocated to nickel at the cobalt refining is due to the carbonate precipitation, which was fully allocated to nickel as the nickel stream is already separated from the cobalt stream when it is precipitated. For the CoCl₂-solution produced at recycling a conversion process to CoSO₄·H₂O crystals was not included in the assessment. Based on the steam consumption of the crystallization process of NiSO₄·6H₂O crystals it was estimated that the removal of water would result in a ca. 10% higher CEENE value per kg of Co. Though smelting in general is regarded as an energy intensive process, the smelter contributed only 1-4% to the total resource use. There are several reasons for this. The smelter valorises large parts of the energy content of the battery; this includes the energy content of plastics and other inorganic materials. So in total (including gas cleaning) only an additional energy input of about 4.5 MJ is supplied to the smelter per kg of alloy. This is 2-3 times less than the energy consumption of a copper smelter per kg of copper (Coursol et al., 2010; Marsden, 2008). Furthermore subsequent processes require a lot of chemicals, whose production also requires other resources. Then there are further heat treatments during the production, especially the heat treatment during the final production step requires a lot of energy and this is provided in the form of electricity, which means that losses during electricity production contribute considerably to the high CEENE value of this step. Therefore, the use of the NMC-model of the smelter for the other cathode materials is not expected to have had a considerable impact on the overall results.
Figure 8-5 Cumulated results for the closed loop scenario expressed in MJ of CEENE for the cathode material per kg (A), kWh over one cycle (B) and per kWh of energy delivered over the cycle life of the cathode (C).
An uncertainty analysis on the basis of data quality indicators (Weidema and Wesnæs, 1996) was performed with Simapro 8 to get an idea on the overall uncertainty. The results are represented in Table 8-5. The degree of uncertainty is somewhat higher for those cathode materials still in development. It should be noted that the uncertainties are to some degree correlated as for example all cathodes depend heavily on the electricity input accounted for Korea. To give an example: The confidence intervals of cathode 1 and cathode 4 below are overlapping, however when a simultaneous uncertainty analysis is performed the CEENE value of cathode 4 is always lower than the CEENE value of cathode 1.

Table 8-5 Results of the uncertainty analyses for the cathode materials.

<table>
<thead>
<tr>
<th>Cathode</th>
<th>Mean MJex/kWh (one cycle)</th>
<th>95% confidence interval lower bound MJex/kWh (one cycle)</th>
<th>95% confidence interval upper bound MJex/kWh (one cycle)</th>
<th>Coefficient of Variation</th>
</tr>
</thead>
<tbody>
<tr>
<td>Cathode 1</td>
<td>621</td>
<td>481</td>
<td>833</td>
<td>14%</td>
</tr>
<tr>
<td>Cathode 2</td>
<td>993</td>
<td>687</td>
<td>1440</td>
<td>19.3</td>
</tr>
<tr>
<td>Cathode 3</td>
<td>465</td>
<td>325</td>
<td>679</td>
<td>19.8</td>
</tr>
<tr>
<td>Cathode 4</td>
<td>378</td>
<td>255</td>
<td>566</td>
<td>20.9</td>
</tr>
<tr>
<td>Cathode 5</td>
<td>591</td>
<td>471</td>
<td>751</td>
<td>11.5</td>
</tr>
</tbody>
</table>

As shown in the charts of Figure 8-5 the differences in energy density and cycle life of the cathodes are essential for the comparison and possibly more important than uncertainties in the LCI. Four of the cathodes had comparable resource consumption per kg of material, but the picture changes when energy density and cycle life are included. This implies that the remainder of the battery, especially the anode and the electrolyte are crucial, as they also determine the performance in the final application. Thus for an LCA involving battery applications the properties of the battery have to be determined thoroughly. With regard to the final application also other properties like the voltage and volumetric energy density need to be considered when selecting a cathode material. Therefore the presented results could only be a guideline when selecting a cathode for a specific application.

8.4 Concluding remarks

The study was based on primary industry data provided for cathode material precursor production, cathode material production and battery recycling, which made it possible to have a rather accurate assessment of natural resource requirements pertaining to commercial Li-ion cathode material production. In the closed loop scenario the production of the metal feedstock and energy use during cathode material production were the main contributors to the resource use for the cathode material production. Thus, even though the metal content is important, process conditions should not be neglected. Though the recycled nickel and cobalt contributed considerably to the overall resource use for the cathode materials that contained high shares of those metals, the properties of the cathode in terms of energy density and cycle life were much more important for the resource footprint than their metal composition as such when the functional properties of the cathode materials were considered. As cathode material properties and especially cycle life depend on the overall battery design and the usage pattern, the results are considerably influenced by the conditions under which the cathode material properties were determined and are thus not definite.
Part IV
Synthesis – General discussion and perspectives
Discussion of the results of Part II and Part III

In part II of this thesis the ore grade based LCIA methods used for assessing impacts in the AoP natural resources were examined. In this context the decreasing availability of a metal resource was interpreted as increasing effort needed to obtain the metal from resources in the natural environment. In chapter 5 intermediate CFs were established for nine metals. These CFs were calculated on the basis of ore grade data covering the years 1998-2010 from a mining database. For a large part the CFs were based on purely physical data related to the mined deposits and the applied method could therefore be described as a depletion method in accordance with the distinction made by (van der Voet, 2013). However, it also contains elements of scarcity. First of all the processed ore grades are in part determined by economic circumstances. For example, low prices of metals mean that at some mines operations might be suspended or mines might even be closed. Also the cut-off grade, i.e. the lowest grade that will be mined and processed depends on economic considerations (Cairns and Shinkuma, 2003; Wellmer et al., 2008). Furthermore, the method used metal prices to allocate ore production to the metals produced at a mine, thus also reflecting economic aspects. The comparison with CFs from a selection of established LCIA methods showed similar trends overall: high CFs for precious metals and particularly low values for zinc and lead. These similarities may be attributed to the higher absolute increase of ore that has to be mined at lower grades for the same relative increase and the differences of average grades/concentrations of the included metals. Relative to copper the CFs of the precious metals calculated from the mining data were more than one order of magnitude higher than the midpoint CFs of the ReCiPe method.

In chapter 6 the analysis was extended beyond only looking at ore grades by considering energy requirements. Chemicals and other materials were not included yet. The development of energy requirements for copper production was estimated based on data from the mining data base and data collected from literature. As the explicit consideration for increased energy requirements directly reflects societal concerns, a method based on this approach can be quite clearly categorized as one representing a measure for scarcity. Based on the data and the model it was confirmed that there are differences in the ore grade between mines employing different mining and processing methods and it was shown that on a global level energy demand and ore demand do not necessarily evolve in the same direction, which means that not sufficiently taking into account technology in LCIA methods for impacts on metal resources might result in misleading CFs. The most apparent advantage of using mining data is that they represent actual ore grade developments as they occur in practice. Hence, energy demands can directly be estimated from mine type and processing route. Co-product metals could be taken into account by allocating material moved and ore treated based on metal value. The use of mining data, however, also means that the results do not represent a projection into the future and might at most be considered valid over a short time frame of less than 10 years. Projections might be improved by additionally considering projects that are currently in development. Such projects are also contained in the mining database. Potentially data on mine type, processing route, annual mine capacity and annual capacity for the main metals are available. Nevertheless, additional data collection would be required. Another drawback of using actual mining data was that the time frame for which data was available in
the database was rather short covering about 10 years and that average ore grades varied considerably from year to year. The inclusion of technology and processing route is desirable, but also very data intensive.

In principle, a method based on (increased) energy needs could also be applied for other resources than metals. This has also already been used for fossil fuels in the Eco-indicator 99 and ReCiPe methods and suggested for water by Pfister et al. (2009). If a surplus energy method was used for other resources, however, this would still not imply that a direct comparison of the results would be possible. There are several constraints: 1) Any determination of the surplus energy requirements would involve uncertainties and those will be even more different between those different resource categories. 2) The functional value of the various resource types is quite different. Whereas the majority of fossil fuel consumption is for energy production, metals can be used in a variety of applications providing all kinds of functions. Sufficient water supply on the other hand is required to fulfil basic human needs and to maintain the functioning of eco-systems. 3) The increased energy needs can be envisioned as a form of having to fall back on substitutes. In the case of metals they are in essence distinct as the substitutability depends on the specific application. Thus it is considered that higher quality metal resources are gradually substituted by lower quality metal resources. For fossil fuels this is somewhat different. With respect to the major function of fossil fuels, energy supply, fossil fuels can be replaced by various renewable energy sources.

As soon at those would be considered, however, the question would arise if e.g. the solar radiation or the energy in the wind should be included in such an indicator. In the short run a method limited to fossil fuels substituting each other can still have its merits. The substitutability for fossil water is much more limited and depends a lot on local conditions. An indicator based on desalination as back-up technology is therefore rather a theoretical concept and not something that will occur in practice (Pfister et al., 2009).

For the future determination of CFs derived from mining data several smaller improvements would be desirable. Year to year variability in calculated average ore or energy demands is on the one hand caused by actual changes in demands at individual mines, the closure and opening of mines, but also by variability in data availability. Thus, a better control of variability caused by unavailability of data would be needed. Generally, it would also be good if overall data availability could be improved. Coverage was rather low especially for nickel and data availability also still depends to some extend on the location of the operational unit. With time it should also be possible to establish longer time series on a global level for more metals. With regard to the data in the used database it would be useful to have timestamps not only for production data, but also for relevant information like mine type or mine status. Of course, it would even be better if not so many data was missing, e.g. with regard to recovery route or material moved. An ideal database could include additional standardized data pertaining to energy and other auxiliary requirements which companies to some extent already compile for sustainability reports or similar.

In part III the results of two case studies of advanced materials were presented. The resource consumption of germanium wafer and Li-ion cathode materials were assessed in chapters 7 and 8 respectively. In LCAs industry data for these advanced material intermediates are often lacking. For the case studies data was obtained directly from industry for the production of germanium wafers from lead processing residues and for the production of cathode materials with nickel and cobalt feedstock recycled from waste batteries. Process inventories and resource fingerprints employing the CEENE method were established.
The case study of the germanium wafers also included the compilation of an inventory for germanium dioxide production. The results of the case study confirmed that the germanium wafer only causes a relatively small part of overall resource use for an HCPV system due to the use of concentrator technology and that a germanium wafer has very short partial payback times of a couple of days in terms of energy and exergy. For the calculation of payback times it was assumed that the produced electricity replaces non-renewable based electricity. Though this introduces elements of a consequential LCA, i.e. an LCA that is aimed at modelling the consequences of decisions instead of just describing potential impacts of the current product system (Finnveden et al., 2009), the study is not a consequential LCA. In essence all data used is average data, i.e. representing the average burden of producing a unit of a product. For a consequential LCA marginal data would have been required, which would represent the change in environmental exchanges if a small change in the output of products occurred.

The study on Li-ion cathode materials showed that changes in material compositions can deliver advantages for resource consumption, but that the main drives for these advantages are the changed properties of the materials. Considerable advances are possible via improving cycle life and energy density. Nevertheless, specific material compositions may also require processing conditions that counteract benefits of improved properties or reduced content of specific metals.

Unfortunately, it was not possible to employ the CFs developed in chapter 5 on the case studies. One reason why such an assessment was not included for the Li-ion cathode material case study (chapter 8) is that almost all of the nickel feedstock and cobalt feedstock were assumed to be obtained from recycling anyway. The other reason is that no ore grade based CFs were established for the metals contained in the cathode materials with the exception of nickel. There are different reasons why CFs were not established for those metals: For cobalt no CF could be established in chapter 5 due to lack of data in the employed database. In this case the lack of data may be due to the fact that cobalt is often only recovered as a by-product metal. For nickel a CF could be established, but the data availability was the lowest among the assessed metals. With regard to the future of primary nickel production ore grade may be of limited use due to the majority of the remaining resources being present in laterites and thus requiring other processing techniques, as has been mentioned in chapter 6. For manganese the data availability was also low and it was initially not included in the assessment as its mineralization is different from the other metals assessed in the study (Skinner, 1979). Detailed information on lithium extraction was not included in the database that was used to obtain the values used as intermediate CFs. Moreover, ore grade is not a suitable measure for lithium, because currently most lithium is obtained from brines (Talens Peiró et al., 2013). Brines also have a grade, but the production steps for extracting metals from brines differs considerably from those required to extract metals from ores, meaning that direct comparisons with ore grades do not make much sense. Primary germanium is only recovered as a by-product form zinc/lead ores, but could also be sourced from coal dust. The case of by-product metals is discussed in more detail in the following chapter 10. That chapter also tries to give some suggestions for subsequent research into methods quantifying impacts on metal resources that more explicitly take into account aspects that go beyond the natural stock. Chapter 11 provides perspectives regarding the case studies.
In chapter 3 impacts in the AoP natural resources were described as being characterized by the reduced availability of resources in the future. The explicit mentioning of the future reminds of the definition of sustainable development, which is concerned with the means available for future generations. In contrast to the subjects of the other AoPs the resource category does not deal directly with living systems. Mineral resources are not directly required by humans or ecosystems. What we need is the functional value of the metal. Not having a specific metal available can have very diverse consequence due to the variety of metal applications. This implies that the choice of an endpoint for this AoP is more arbitrary than for the other AoPs and should maybe be extended beyond resource availability in nature.

Furthermore, the availability of metal resources to future generations does not even solely depend on the amount and quality of metal left in stocks in nature. Figure 10-1 gives an illustration of some of the other aspects that are relevant next to the metal resources present in the natural environment. These aspects include also other pillars of sustainability, i.e. economics and social issues, as well as technology. Technology has been explicitly included in the figure because it plays an essential role in determining supply and demand. It depends on the available technology, which metals are required to fulfil specific functions. Thus, technology influences demand and eventually also serves to fulfil the functions satisfying human needs. The functions metals and their applications fulfil are very diverse including fertilizers, energy carriers, energy production, energy storage in general, medical applications, transportation, information technology, construction, machinery. So the eventual impact of a reduced availability can vary considerably from metal to metal. Therefore LCIA endpoint methods are also actually only representing a proxy for the final impact. Technology is also required to supply metals, i.e. to extract them from nature and from technosphere stocks and to transform them in a way that they can be used in applications. It depends on the available technology whether extraction from natural deposits is possible at all and whether it is possible at a reasonable cost. Whether the costs are reasonable also depends on the income the mining and mineral processing companies can make, i.e. on metal prices. Metal prices in turn depend on and influence supply and demand.

Thus, metal availability seems to be an issue spanning several aspects of sustainability, which makes this matter rather complex and also makes it different from the issue of emission impacts. Though emission impacts may be affected by technology and some of the other aspects of sustainability, these are not an essential element to determine their impact. Metals are, in the majority of cases, provided by profit seeking private businesses; technology is needed to provide the metals and demand for a metal can change drastically depending on technological developments. Once emitted to the environment a substance can affect the ecosystem or humans without further interventions from non-environmental sources. Another related difference is that cost increases due to grade decreases are at least to some extent inherently internal to the companies providing the metals, while emission costs are external unless they are controlled by authorities. Nevertheless, metal resources are a service provided by nature and thus it makes sense to include them in LCA to obtain a more complete picture of the sustainability of a product, especially as such resource impacts are typically not
included in economic analyses. The objective of this chapter is to give an overview of the available information concerning these various aspects which impact the metal supply. Furthermore a closer look will be taken at by-product metals, for which the supply economics differ somewhat from main/co-product metals, which might necessitate a different approach for assessing impacts of their usage.

10.1 Supply from nature
To be able to estimate future availability of metal resources it would be convenient if we would know how much is out there and at what quality. The problem is that not all of the Earth has been explored sufficiently to know for sure and that other dynamic factors also influence what might be considered a resource and what not. A distinction can be made between reserves, resources and geopotential (Sinding-Larsen and Wellmer, 2012), whereby reserves are economic, resources are currently uneconomic and the geopotential the metal present in currently unknown deposits, which might be discovered and extracted in the future.

Various methods have been used to approach the quantification of what is or will be available. A non-exhaustive overview of data available for copper is given in Table 10-1. The average crustal abundance can be used to calculate the total amount of metal present in the crust. This was the approach chosen by Guinée (1995) to calculate the so-called ‘ultimate reserves’. As already mentioned in previous chapters the total amounts of metals present in the crust is very high compared to current production levels. As a comparison with the other values presented in Table 10-1 shows, the amount is also several orders of magnitude higher than estimates
considering deposits, where metals occur in more concentrated form. Guinée himself argues that not all of the ‘ultimate reserves’ will be actually available for extraction and introduces the term ‘ultimate extractable reserves’. Kesler and Wilkinson (2008) made an estimate of the total amount of copper available in deposits up to 3.3 km depth based on a tectonic diffusion model for porphyry copper deposits. This estimate can be considered to represent ‘ultimate extractable reserves’.

The USGS publishes reserve and resource estimates in their annual publication ‘Mineral Commodity Summaries’, whereby the reserve estimates represent what is currently economically feasible to mine and the deposits included in the resource estimates have to satisfy some physical conditions with respect to grades, depth etc. The reserves only include discovered and well explored deposits and thus their physical availability and quality are fairly certain, but the reported estimates depend on metal prices, legal conditions etc. which can change. The resource estimates also include undiscovered resources. In the Mineral Commodity Summaries 2010 (USGS, 2010) unconventional resources like those found in the deep sea were also quantified. In addition the USGS also has published reports on various deposit types containing copper, which also contain estimates of initial resource amounts and grades of known deposits (Cox et al., 2007; Mosier et al., 2009; Singer et al., 2008). These deposits may already be in production, so that the remaining resources are somewhat lower than the total resource amount accounted for in those reports.

Mudd et al. (2013) collected data from company reports to estimate the currently known and remaining resources of copper. Even though these data include not only measured but also indicated and inferred resources, they can be considered to be quite certain as they are based on geologic knowledge of specific ore bodies. Another advantage is that the collected data also include information on average grades. Mudd et al. (2013) also assessed the additional resources becoming available in the near future by the opening of new mines or expansion of existing mines. The estimates by the USGS and Mudd et al. (2013) are about an order of magnitude smaller than the estimate for extractable copper presented by Kesler and Wilkinson (2008).

Table 10-1 Different estimates for copper available in natural deposits.

<table>
<thead>
<tr>
<th>Type of resource</th>
<th>Amount</th>
<th>Grades included</th>
<th>Reference</th>
</tr>
</thead>
<tbody>
<tr>
<td>Total metal in rocks in crust</td>
<td>$3.90 \times 10^{14}$ t</td>
<td>NA</td>
<td>Kesler (2010)</td>
</tr>
<tr>
<td>Accessible resources in deposits in crust, 3.3 km based on tectonic diffusion model</td>
<td>$8.39 \times 10^{10}$ t</td>
<td>no</td>
<td>Kesler and Wilkinson (2008)</td>
</tr>
<tr>
<td>Resources (USGS definition)</td>
<td>$\geq 3 \times 10^9$ t (land)</td>
<td>No (maybe in other reference)</td>
<td>U.S. Geological Survey (2010)</td>
</tr>
<tr>
<td></td>
<td>$0.7 \times 10^9$ t (deep sea nodules)</td>
<td></td>
<td></td>
</tr>
<tr>
<td>Resource (as defined in statutory codes) collected from company reports</td>
<td>$1.7809 \times 10^9$ t (1.861 $\times 10^9$ t, incl. Chinese resources)</td>
<td>yes</td>
<td>Mudd et al. (2013)</td>
</tr>
<tr>
<td>Mined and remaining resources</td>
<td>$2.1 \times 10^9$ t</td>
<td>yes</td>
<td>Cox et al. 2007, Singer et al. 2008, Mosier et al. 2009</td>
</tr>
</tbody>
</table>

Resource data has been used to establish cumulative grade-tonnage relationships on a global level. This approach has been employed in the surplus-energy method (Goedkoop and Spriensma, 2000; Müller-Wenk, 1998), the marginal cost approach (Goedkoop et al., 2009) and the ore grade decrease method proposed by Vieira et al. (2012). If such models are used to predict ore grade development it should be made sure that the resulting relationships and
the chosen working point are more or less in agreement with past mining grades and tonnages, e.g. in the marginal cost approach the working point is chosen quite arbitrarily, assuming that all inventoried deposit types have been mined to the same percentage. In the approach proposed by Vieira et al. (2012) the ‘reserve base’ is derived from the recoverable copper resources up to a depth of 3.3 km estimated by Kesler and Wilkinson (2008). Kesler and Wilkinson (2008) base their estimate of all copper deposits on their own estimate of the total endowment of porphyry copper deposits and another estimate by Singer (1995) according to which porphyry copper deposits represent 57% of world discovered copper. Kesler and Wilkinson assume that this percentage is also true for the undiscovered deposits. There is no indication in their paper that this ratio does not also apply to the deposits up to 3.3 km. However, in the paper by Vieira et al. (2012) porphyry copper deposits make up 75% of what they call the ‘reserve base’. Furthermore the working points are at rather high amounts of total copper mined on the one hand and on the other hand result in rather high current grades at least in the loglogistic model.

For bulk metals other than copper similar data is available or can be compiled. For many special metals the situation is somewhat different due to limited data availability, which is also related to them often being by-products and thus being less relevant for the economics of mining. Mudd et al. (2013) make an assessment of the recoverable resources for cobalt, which is also often a by-product metal. Their approach might also be applicable for other by-product metals. The appeal of the ‘ultimate reserves’ concept is that this data is easily available also for the more exotic elements.

Though most metals are extracted from the ground, this is not always the case. Lithium for example is currently predominantly recovered from brines and might be recovered from seawater in the future (Yaksic and Tilton, 2009). Deep sea nodules and other marine resources are unconventional sources of metals like copper, cobalt, nickel and manganese. In addition some metals might also be sourced from coal, e.g. germanium or even rare earth elements (REEs) (Seredin and Finkelman, 2008; Seredin et al., 2013).

While reserves and resources for some metals seem to be distributed over a bigger number of countries, this seems less the case for other metals. Often discussed is the case of the REEs of which production is currently dominated by China. However, other countries that have REE reserves are ramping up their own primary production (USGS, 2011). In the short run supply also depends on mine capacity and its utilization rate. If mine capacity utilization is not too high, increases in demand can be satisfied faster. High capacity utilization was an important factor in the unusually high metal prices during the first decade of the 21st century (Humphreys, 2010).

### 10.2 By-product metal supply

Many of the special metals are by-product metals, i.e. they are associated with other metals, which are also called carrier metals, and typically occur in very low amounts in deposits. The economic value of the by-product metal stream at mines, smelters and refineries is typically small, even if the price per kg of the by-product metals it usually much higher than the price of e.g. zinc or copper. Hagelüken (2011) explains this quite clearly: While as for other metals an increase in demand leads to increasing prices, mining companies will not increase production of by-product metals unless demand for the carrier metal increases as well, because oversupply of the carrier metal would lead to its price decreasing.
Nevertheless, supply of by-product metals can still be increased by increasing their recovery or by increasing the recycled amounts. Indeed, the recovery of indium has increased over the years. At one point however, increased recovery from primary resources might not be possible anymore. Then the only option that remains to increase the supply directly, is by increased recycling, i.e. production from the technosphere (Figure 10-2). In that case the primary by-product metal supply is constrained by the primary supply of the carrier metal.

**Figure 10-2 Major/co-product metal supply versus constrained by-product metal supply.**

Hence, it might make sense to make a differentiation in LCIA between by-product and main/co-product metals. For main/co-product metals primary supply can be adjusted to demand. Decreasing quality of deposits means that more effort has to be invested unless recovery technology is improved. Prices of main/co-product metals have to make economic mining operations feasible. On the other hand the economic value of by-product metal production is already so low that increased mining efforts would not mean that their prices would have to increase to make their production feasible. This does not mean that their prices would not increase if primary production of the carrier metals decreased. In a situation where primary annual supply is already constrained changes in mining efforts do not seem adequate to reflect changes in the availability of those metals, not even considering the difficulty if one would try to properly allocate any efforts to those metals, for which data on the level of mining is scarcely available.

### 10.3 Supply from stocks in the technosphere

Especially countries that are not gifted with extensive natural metal resources of their own are interested in the recovery of metals from EOL products in order to reduce their dependency on metal imports. Recycling is one of the means brought forward by the European Commission (2011, 2008) to reduce the EU’s consumption of primary raw materials. As mentioned in chapter 2 current metal supply relies in parts on recycling. Though for a number of metals, especially special metals, current recycling rates for EOL products are negligible (Graedel et al., 2011a; Zimmermann and Gößling-Reisemann, 2013). This can be related to often low fractions of those metals in waste flows, complex materials, insufficient economic incentives and poor collection systems (Buchert et al., 2009; Hagelüken, 2013).
Graedel and co-workers have published a great number of works quantifying the stocks of many metals currently in use: copper (Glöser et al., 2013; Lifset et al., 2002), zinc (Gordon et al., 2004; Graedel et al., 2005), silver (Johnson et al., 2006) etc. Some related studies have been published by others as well, e.g. Pauliuk et al. (2013), Glöser et al. (2013), Rauch (2009), Saurat and Bringezu (2008).

In addition to the stocks in use also recovery of metals previously discarded to landfills and waste reservoirs is an option which is explored. Kapur and Graedel (2006) estimated that copper in this type of deposits amounts to $3.93 \times 10^8$ t compared to copper stocks in use of $3.30 \times 10^8$ t. The sum of these estimates is somewhat higher than the current copper reserves of $5.4 \times 10^8$ t (USGS, 2010). The deposited copper is equal to about 25 times the current primary copper production. However, the economics of landfill mining with the sole purpose of materials recovery are not yet clear (Krook et al., 2012). Reworking of mining waste, however, is already common practice. Enhanced landfill mining is proposed as a concept to facilitate future recovery of currently non recyclable material (Jones et al., 2013).

10.4 Technology to recover metals from natural and technosphere stocks

What impact might technological development have on the availability of resources from nature and the technosphere? Mining companies have an interest in more efficient technology in order to cut energy costs. To what extent can ore grade decreases be compensated? As authorities seek to improve recycling and new technologies are developed, what are the prospects of improved recycling rates?

As observed in chapter 6 different processing routes in metal production also lead to differences in energy consumption and more particular the relationship between ore grade and energy consumption is more complex as lower grades usually are associated with different production routes. Thus it can be expected that at least part of the impacts of ore grade decreases can be counteracted by changing the employed technology. Radical technological change is not very common in the mining industry due to high capital costs and long lead times (Warhurst and Bridge, 1996), but in that respect it does not differ from other mature industries (Bartos, 2007).

Research and development spending in the mining industry had decreased in the first decade of the 21st century (Bartos, 2007; Filippou and King, 2011). Continuous efficiency improvements are to a large extent limited to incremental improvements, from automation and economies of scale (Filippou and King, 2011). Productivity gains have been achieved from employing ever larger equipment, but further gains via economies of scale might be limited (Bartos, 2007). A number of challenges lie ahead of the industry, which might require further innovation (Filippou and King, 2011; Johnson, 2013). Production costs need to be controlled in the face of rising energy costs, more remote locations, lower grades, mineralogically more complex resources and carbon taxes. Some big open pit mines have been or will be converted to UG mines employing a low cost UG mining techniques like block caving, with the advantage of reduced disturbance on the surface and costs comparable to large scale open pit mines (Chadwick, 2008; Page, 2001). Ghose (2009) projects that overall UG mining output will exceed OP mining output by 2050.
Metal extraction assisted by biotechnology is already practiced, but could become more widespread in the future to process lower grade ores economically (Johnson, 2013; Schippers et al., 2013, 2011). It has been estimated that 38% of the primary copper leached in 2010 was produced via bioleaching of sulphidic copper minerals (Schippers et al., 2011). Biomining is also used on industrial scale in the production of gold, uranium, cobalt, zinc and nickel. It has been suggested to use hydraulic fracturing technology to facilitate in situ biomining (Filippou and King, 2011; Johnson, 2013) and to use in situ biomining to recover metals from ore residues in block caving mines (Schippers et al., 2013). Thus, even though the quality of ore deposits might be decreasing, there are prospects of at least partial compensation of these negative developments.

Due to new technologies recycling rates of metals might further improve in the future, especially for special metals. For example, the Japanese government has provided large funding schemes for the reduction of its dependency on REE imports (China Daily, 2012). In the wake of this initiative already two processes for the recovery of rare earths from hard disk drives and vehicles have been announced (The American Ceramic Society, 2012). Biotechnology might also be used to recover metals from waste streams: printed circuit boards (Zhang et al., 2012). Maybe even more important than recovery technology will be the efficient collection of WEEE and other waste streams. Frondel et al. (2007) make estimations of future recycled content and based on this estimate the demand for primary metals.

10.5 Demand for metals

For bulk metals past developments have been fairly consistent with general economic development and for the time being, it seems reasonable that on average demand for them will grow further more or less steadily. For example for copper several demand forecasts up to 2050 have been published in the 21st century (Table 10-2). Different methods are used to come to such projections. Frondel et al. (2007) base their estimations on sectoral growth rates and evaluations of technological change. Backman (2008) and Halada et al. (2008) base their projections on relationships between GDP/capita and metal demand/capita combined with population and GDP estimates. Projections of copper demand for specific future technologies have been made by Angerer et al. (2009). The average growth rates for global copper demand are quite similar with 3 respectively 2.7%, while the average growth rate for the future technologies selected by Angerer et al. is somewhat higher at 4.1%.

Similar projections are also available for a number of other bulk metals. There are even projections available for special metals for which demand is much more dependent on technological developments. Table 10-3 gives an overview over selected studies. The selection has been limited to studies covering more than one metal.

For the special metals often employed in emerging technologies some differences in the projections can be observed. For example, while the forecast of Halada et al. (2008) till 2020 roughly equates to an average annual increase of only 3%, Buchert et al. (2009) forecast an average annual increase of 5-10%. Likewise, for cobalt the projection of Halada et al. (2008) for 2020 is almost 50% above the high growth scenario in Buchert et al. (2009). Unfortunately, the coverage of the bulky metals like copper in the studies looking at technological change is limited, so it is not clear if these would lead to different results from the more generic approaches. According to Frondel et al. (2007) technological change does not have an impact on copper demand as opposing trends cancel each other.
For now it seems that for metals mainly used in applications outside of basic infrastructure like housing, machinery, electricity transmission, demand projections based on the assessment of market developments for advanced material applications seems to be more suitable than more general methods based on historic trends or GDP developments only.

Table 10-2 Global copper demand projections up to 2050.

<table>
<thead>
<tr>
<th>Demand type</th>
<th>Scope</th>
<th>Annual growth (%)</th>
<th>Projected Amount (t)</th>
<th>Reference</th>
</tr>
</thead>
<tbody>
<tr>
<td>Total (refined)</td>
<td>Global, 2004-2025</td>
<td>2.7</td>
<td>25.5×10^6 (2025)</td>
<td>(Frondel et al. 2007)</td>
</tr>
<tr>
<td>Total (refined)</td>
<td>Global, 2050</td>
<td>ca. 3</td>
<td>42×10^6 (2050)</td>
<td>(Backman 2008)</td>
</tr>
<tr>
<td>Total (refined)</td>
<td>Global 2000-2050</td>
<td>ca. 3</td>
<td>45×10^6 (2050)</td>
<td>(Haldada et al. 2008)</td>
</tr>
<tr>
<td>Demand for future technologies (solders, electric engines, RFID tags, high temperature superconductors)</td>
<td>Global, 2006-2030</td>
<td>4.1</td>
<td>3.7×10^6 (2030)</td>
<td>(Angerer et al. 2009)</td>
</tr>
</tbody>
</table>

Table 10-3 Studies containing explicit demand forecasts for metals. In some cases the forecasts are directly compiled from other publications. If the referenced publication is already listed here, the metal is not identified as being covered by the referencing work.

<table>
<thead>
<tr>
<th>Scope</th>
<th>Metals covered</th>
<th>Comment</th>
<th>Publication</th>
</tr>
</thead>
<tbody>
<tr>
<td>Global, 2004-2025</td>
<td>Cu, Al, Zn, Cr, Ge, V, Pt, Ta, steel, magnetite, graphite, fluor spar</td>
<td>Demand scenarios based on possible development of markets</td>
<td>(Frondel et al. 2007)</td>
</tr>
<tr>
<td>Global, 2050</td>
<td>Al, Cu, Fe, Pb, Ni, Zn</td>
<td>Country level population, GDP/capita + metal/capita</td>
<td>(Backman 2008)</td>
</tr>
<tr>
<td>Global, 2000-2050</td>
<td>Fe, Al, Cu, Mn, Zn, Cr, Pb, Ni, Si, Sn, rare earths, Mo, Li, Sb, W, Ag, Co, In, Au, Ga, Pt and Pd</td>
<td>Country level population, GDP/capita + metal/capita</td>
<td>(Haldada et al. 2008)</td>
</tr>
<tr>
<td>Global, 2006-2030, specific future technologies only</td>
<td>Cu, Cr, Co, Ti, Sn, Sb, Nb, Ta, PGMs, Ag, REE, Se, In, Ge, Ga</td>
<td>Based on expected growth of specific future technologies</td>
<td>(Angerer et al. 2009)</td>
</tr>
<tr>
<td>Global, 2007-2020</td>
<td>In, Ru, Ga, Te, Li, Co, REE</td>
<td>Demand scenarios based on possible development of markets</td>
<td>(Buchert et al. 2009)</td>
</tr>
<tr>
<td>Global, 2010-2020</td>
<td>Hf, Mo, Nd, Ni, Nb, Se, Ag, Sn, V</td>
<td>Demand scenarios based on possible development of markets</td>
<td>(Moss et al. 2011)</td>
</tr>
<tr>
<td>Global, 2010-2025</td>
<td>Co, Li, La, Ce, Te, In, Ga, Eu, Tb, Y</td>
<td>Demand growth for non-clean energy technologies based on global economic growth projections. Four scenarios for demand for clean energy technologies</td>
<td>(U.S. Department of Energy 2010)</td>
</tr>
</tbody>
</table>

10.6 Conclusions and perspectives

The purpose of chapter 5 and chapter 6 was to confront LCIA methodologies with actual mining data. Especially, based on the results of chapter 6, it would be recommended to include more thorough technology assessments for the different metals in the development of CFs. Ideally, this would also include forward looking assessments, but considering current technology differences would already lead to improvements. A problem for the development of CFs in this area has been the limited data availability, especially with respect to possible future developments. As more and more studies are concerned with metal availability, e.g. in the framework of criticality studies, material flow analyses and the International Resource Panel, also somewhat more information becomes available for developing LCIA methods.
concerned with the future availability of metals. Data on some aspects that might be considered are presented in Table 10-4 for a selection of metals. It should be noted that for some of the metals a more extensive (literature) research is advised to determine the most appropriate values for the aspects considered in the table. For example, the demand increase is in parts based on Halada et al. (2008), whose method might be not so well suited for metals of which a large fraction is used in emerging technologies.

Considering that by-product metals availability is governed by somewhat different supply mechanisms, a dedicated method is warranted, which takes into account that their supply from nature is potentially constrained. It would be interesting to also include an assessment of additional primary production potential for by-product metals if their recovery was increased. For example, values regarding the non-recovered fraction are available for indium (Mikolajczak and Jackson, 2012). In the case of germanium coal is possibly an additional primary resource with a huge potential (Bleiwas, 2010). Talens Peiro et al. (2011) estimated potential production for a number of other metals.

As recycling will likely become a more important source of metals in the future, it makes sense to incorporate recycling considerations, respectively information on dissipation. For metals that are not predominantly produced as by-products this could be done in a more indirect way e.g. by determining the demand for primary supply based on gross demand and recycling potentials. For by-product metals that are supply-constrained potential supply from recycling EOL products and from urban mining are in fact even more important than for other metals, because their supply from natural resources is quite inelastic. Use of these metals can even increase potential future annual supply locally, if they are not dissipated. Instead of considering primary demand and primary supply only during the development of CFs, annual dissipation and/or annual supply from the technosphere could be included explicitly. As these metals are typically also special metals for which demand is strongly dependent on technological developments, long-term projections are very uncertain already anyway, so the time horizon of CFs should be limited to around 10 years.

To incorporate aspects of metal supply other than the naturally available deposits, LCIA might learn from approaches employed in the various criticality reports. For example, in the methodology of Graedel et al. (2011b) by-product metal supply is rated as being associated with higher risks. While the geopolitical factors, which are usually included in metal criticality studies, are not relevant if one looks at the global metal availability, they are important for the future local metal supply and are thus interesting for industries and local authorities. Supply concentration should however be evaluated based also on future production potential (Speirs et al., 2013), e.g. by taking into account reserves. The challenge will be to translate these approaches in a way that they make sense on a per kg of metal basis. One possibility might be to normalize criticality scores to annual production of the metal.

It would be interesting if information on grades and energy requirements could be combined with changes in demand and recycling in a dynamic model, like the one used in the determination of the depletion time according to Graedel et al. (2011b) and Nasser et al. (2011), to determine the impact of mining 1 additional kg of metal today over a specified period. For new products that are expected to require a substantial amount of annual production of a metal a fixed characterisation factor is not appropriate. In these cases an assessment considering impacts of increased demand and interactions with other metal
markets is more suitable, see e.g. the analysis of the introduction of lead free solder by Verhoef et al. (2004).

**Table 10-4 Elements that might be incorporated in the development of CFs for impacts on metal resource availability.**

<table>
<thead>
<tr>
<th></th>
<th>Average annual increase of demand (up to 2020/2025)(^{a})</th>
<th>Loss fraction(^{b})</th>
<th>By-product fraction(^{c})</th>
<th>Geographical concentration of primary reserves(^{d})</th>
</tr>
</thead>
<tbody>
<tr>
<td><strong>Gold</strong></td>
<td>ca. 2</td>
<td>30</td>
<td>ca. 30</td>
<td>Green: &lt;2.5%; Yellow: 2.5-3.5%; Red: ≥3.5%; Data sources: (Frondel et al. 2007, Halada et al. 2008, Buchert et al. 2009, Moss et al. 2011).</td>
</tr>
<tr>
<td><strong>Nickel</strong></td>
<td>4-5</td>
<td>38-45</td>
<td>Mainly main or co-product</td>
<td>Green: &lt;2.5%; Yellow: 2.5-3.5%; Red: ≥3.5%; Data sources: (Frondel et al. 2007, Halada et al. 2008, Buchert et al. 2009, Moss et al. 2011).</td>
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<td><strong>Zinc</strong></td>
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<td>Green: &lt;2.5%; Yellow: 2.5-3.5%; Red: ≥3.5%; Data sources: (Frondel et al. 2007, Halada et al. 2008, Buchert et al. 2009, Moss et al. 2011).</td>
</tr>
</tbody>
</table>

**Most important scarce metals according to global production value (excluding fertilizer (ICMM 2012))**

- **Gold**
- **Copper**
- **Silver**
- **Nickel**
- **Zinc**

**Metals most often classified as critical (Erdmann and Graedel 2011)**

- **Rare earth elements**
- **Niobium**
- **Tungsten**
- **Ruthenium**
- **Rhodium**
- **Platinum**
- **Indium**

**Metals included in previous case studies not yet mentioned above**

- **Lithium**
- **Cobalt**
- **Manganese**
- **Germanium**


\(^{b}\) Approximate losses during production, use and EOL as percentage of extracted amount unless otherwise noted. Green: <30%, Yellow: 30-50%, Red: ≥50%. Data sources: (Buchert et al. 2009, Graedel et al. 2011, Wittmer et al. 2011, Zimmermann and Gößling-Reisemann 2013).


\(^{d}\) Green: more than three countries have more than 50% of global reserves, Yellow: up to three countries have more than 50% of the reserves, Red: less than three countries have more than 50% of the reserves. Data sources: (U.S. Geological Survey 2010)
The case studies presented in this thesis were focussed on the resource consumption of intermediate products used in clean energy applications. In the case of the germanium wafers their contribution to the resource consumption of the HCPV-system was estimated. Nevertheless, to get a better understanding of the sustainability impact of the final implication the process inventories of both the germanium wafer production and the cathode production could be implemented in broader case studies that include the final application. In addition, it would be interesting to extent the case studies to LCIA methods beyond resource consumption and maybe even include other sustainability aspects to obtain a more complete picture.

Especially for the Li-ion cathode materials case and extension to some full applications that also consider different consumer use patterns would be of interest. In this way the importance of the differences in the properties of the cathode materials would possibly be highlighted.

To assess the resource consumption of the germanium wafer production an inventory for germanium dioxide production had been established. However, this inventory could still be improved based on a more detailed analysis of the process. Moreover, the process inventory reflected only a specific production route for germanium dioxide. The concept of energy respectively exergy payback times might be extended to specific resources or emissions as a way to convey results to stakeholders as time is something they can relate to rather easily. For example, the ratio between total CO$_2$ equivalents emitted during the lifetime of a photovoltaic system and the CO$_2$ saved by replacing other electricity sources might be calculated. This might also be possible for water, but then it has to be considered that the impacts of water consumption are much more localized. For the comparison of complete systems with different lifetimes yield ratios would be preferable though.

Eventually, also the assessment of the special metal use in the applications regarding future metals availability would be of interest, e.g. Angerer et al. (2009) had already compared the current metal supply and future demand for future technologies.
12

Summary

With the continuing global population growth, climate change and aspirations of developing nations to follow in the footsteps of the developed nations, it is no wonder that the sustainability topic is so prominent today. Since the metals boom during the first decade of the 21st century also metal resource availability has gained renewed interest from politics and academia. This thesis tried to contribute to some of the issues regarding the sustainability of metal use.

Part I served as an introduction to the topics of this thesis. Chapter 1 gave a general overview of the sustainability debate and presented life cycle assessment (LCA) as a tool, which can be used to evaluate aspects of the sustainability of products. Chapter 2 focused on aspects of the sustainability of metal use. It was noted that in terms of mere abundance there probably is not an issue regarding metal availability, however the quality of the available resources might become problematic, e.g. because ore grades are decreasing. Another important observation from this chapter was that the primary production of many special metals, which were hardly used at all only some decades ago, has increased even more than the primary production of more traditional metals like copper. These special metals are often used in clean technologies, like batteries and photovoltaics, which are also important for increasing the sustainability of our societies. Metal processing is often rather energy intensive, so an analysis of the benefits of clean energy technologies also needs to include the production of the materials they are made of. However, life cycle inventory (LCI) data for the production of the advanced materials used in those clean technologies that is based on actual industry data is rare. With a focus on metals, life cycle impact assessments (LCIA) methods for the area of protection (AoP) ‘natural resources’ were reviewed in chapter 3. It was concluded that methods should take into account the quality of metal resources. The current methods that are based on ore grade decreases were found to be lacking especially with respect to the data used. It was therefore questioned whether they would be able to represent actual mining practice. The objectives of this thesis were located in two areas: On the one hand it was assessed how data on actual mining operations could be used to contribute to the development of LCIA methods for impact quantification on metal resource availability. On the other hand the resource use for the production of advanced materials used in clean energy technologies was to be assessed.

Part II focused on the use of mining data to determine changes in efforts needed for metal production. In chapter 5 ore grade decrease was used as an intermediate characterization factor (CF) for metal resource use. For nine metals CF values were determined based on mining data ranging from 1998 to up to 2010. The resulting CFs were compared to existing LCIA methods and it was found that at least for the nine metals results were somewhat similar with respect to ranking. However, the CF values that had been obtained for precious metals were much higher in relative terms than for the other metals. It was suggested in chapter 5 that eventually ore grade and technological assessment could be combined to obtain a more complete indicator for the increase in efforts required due to decreasing deposit quality. Hence, in chapter 6 the relationship between ore grade, technology and energy requirements was further assessed for copper based on mining data. In the traditional ore grade based LCIA
methods ore grade and energy or economic costs per kg of ore are two independent variables. No distinction is made between copper production via the pyrometallurgical and the hydrometallurgical route. To determine if a differentiation between processing routes has an impact on energy requirements, energy requirements for global copper production between 1998 and 2010 were modelled based on mining data in chapter 6. On the basis of the used data, it was found that even though energy requirements and ore requirements are linked the contribution of other factors seems to be relevant for the overall energy requirements on a global level.

In part III two case studies were presented. Chapter 7 dealt with the resource consumption of germanium wafer production. High electricity inputs in the production mean that the resource footprint is sensitive to the used electricity mix. Though the production of germanium wafers is resource intensive their payback time in terms of resources is only a couple of days if they are used in high concentration photovoltaic systems as only relatively small amounts of germanium wafer are required in this application. The resource consumption of five lithium-ion cathode active materials was assessed in chapter 8. As in the case of the germanium wafer electricity was an important contributor to the resource footprint. Overall the properties of the materials had an important impact on the resource consumption in terms of the functional unit. Nevertheless, it was also found that one of the materials required specific process conditions, which considerably increased the resource consumption for this material vis-à-vis the other cathode materials in the study. Hence, it is advised to properly determine the cathode material properties in function of the application and employing processing data reflecting actual production conditions in LCAs of lithium ion battery applications.

In part IV previous results were discussed and outlooks were given for further research. Though contributions to the sustainability assessment of metal resource use could be made in this thesis, there is still work to be done. In chapter 10 a number of aspects were explored regarding the further development of LCIA methods for impacts on metal resources. Also with regard to the case studies there are possibilities for further research. These were briefly discussed in chapter 11.
Appendices
## Appendix 1

### General Introduction

### A.1.1 List of characterization factors of metals and metalloids for the AoP natural resources

**Table A.1**

<table>
<thead>
<tr>
<th>Ultimate reserves</th>
<th>Reserves/Reserve base</th>
<th>Marginal costs</th>
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<th>Sustainable process</th>
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A.1.2 Metal co-production in the marginal cost method of ReCiPe

For a description of the ReCiPe methodology see (Goedkoop et al., 2009). The presented reasoning will be limited to the CFs based on US dollars, but in extension also applies to the CFs based in kg. Assume a hypothetical case of a deposit type which is the sole source of the two metals A and B and contains a $ of A per b $ of B. Then the marginal cost increase (MCI) of the commodities A and B will be equal to the MCI of the deposit. In ReCiPe parameters (slope and constant) on metal level are calculated as weighted averages from the parameters of the linear regression on deposit level; weights based on value weighted yields of metal A from the deposits:

\[
\text{MCI}_{\text{deposit}} = -4x \times \frac{M_{\text{deposit}}}{c_{\text{deposit}}}^2
\]

\[
\text{MCI}_A = -4x \times \frac{M_A}{c_A}^2
\]

\[
\text{MCI}_B = -4x \times \frac{M_B}{c_B}^2
\]

\[
M_A = Y_{A,\text{deposit}} \times M_{\text{deposit}} / Y_{A,\text{deposit}} = M_{\text{deposit}}
\]

\[
c_A = Y_{A,\text{deposit}} \times c_{\text{deposit}} / Y_{A,\text{deposit}} = c_{\text{deposit}}
\]

\[
M_B = Y_{B,\text{deposit}} \times M_{\text{deposit}} / Y_{B,\text{deposit}} = M_{\text{deposit}}
\]

\[
c_B = Y_{B,\text{deposit}} \times c_{\text{deposit}} / Y_{B,\text{deposit}} = c_{\text{deposit}}
\]

\[
\text{MCI}_{\text{deposit}} = \text{MCI}_A = \text{MCI}_B
\]

\[
P_{\text{deposit},5} = P_{A,5} + P_{B,5}
\]

Definition of characterization factor (CF) on deposit level:

\[
\text{CF}_{\text{deposit},5} = \text{MCI}_{\text{deposit}} \times \sum_{t=1}^{T} P_{\text{deposit},5} \times \frac{1}{(1 + d)^t}
\]

\[
= \text{MCI}_{\text{deposit}} \times \sum_{t=1}^{T} (P_{A,5} + P_{B,5}) \times \frac{1}{(1 + d)^t}
\]

Definition of CF on level of the metals:

\[
\text{CF}_{A,5} = \text{MCI}_A \times \sum_{t=1}^{T} (P_{A,5} \times \frac{1}{(1 + d)^t})
\]

\[
= \text{MCI}_{\text{deposit}} \times \sum_{t=1}^{T} P_{A,5} \times \frac{1}{(1 + d)^t}
\]

\[
\text{CF}_{B,5} = \text{MCI}_B \times \sum_{t=1}^{T} P_{B,5} \times \frac{1}{(1 + d)^t}
\]

So now assume a product which contains the metals A and B in the same ratio as they occur in the deposit. For example, one needs a + b $ of the deposit or x $ of A and y $ of B. If one used the CF on deposit level the impact would be calculated as:
Impact = \((a + b) \times MCI_{\text{deposit}} \times \sum_{t=1}^{T} (P_{A,t} + P_{B,t}) \times \frac{1}{(1 + d)^t} = a \times MCI_{\text{deposit}} \times \sum_{t=1}^{T} \frac{1}{(1 + d)^t} + b \times MCI_{\text{deposit}} \times \sum_{t=1}^{T} \frac{1}{(1 + d)^t}\)

This clearly gives another result than the calculation of the impact on deposit level.
Appendix 2
Mining data base supported analysis of metal resource use
life cycle impact assessment in the area of protection abiotic
resources

A.2.1 Data available in the Raw Materials Database

Table A. 2 Excerpt of column headers for datasets in the Raw Materials Database.

<table>
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<th>Country</th>
<th>Cut-off (% or gpt)</th>
<th>Recovery Main metal (%)</th>
<th>Grade Au in 2008 ore prod (gpt)</th>
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<tr>
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<td></td>
<td>Recovery Metal 2 (%)</td>
<td>Grade Ag in 2008 ore prod (gpt)</td>
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<tr>
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<td></td>
<td>Recovery Metal 3 (%)</td>
<td>Grade Cu in 2008 ore prod (%)</td>
</tr>
<tr>
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<td></td>
<td>Mill capacity (t/d)</td>
<td>Grade Mo in 2008 ore prod (%)</td>
</tr>
<tr>
<td></td>
<td></td>
<td>Recovery method</td>
<td>Grade Zn in 2008 ore prod (%)</td>
</tr>
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<td>Prod Cu 1993 (kt)</td>
<td>Expec. annual ROM (Mt)</td>
<td>Grade Pb in 2008 ore prod (%)</td>
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<td>Prod Cu 1994 (kt)</td>
<td>Mine capacity (Mt/yr)</td>
<td>Grade Ni in 2008 ore prod (%)</td>
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<td>Prod Cu 1995 (kt)</td>
<td>Expec. annual metal cap</td>
<td>Grade Co in 2008 ore prod (%)</td>
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<td>Prod Cu 1999 (kt)</td>
<td>Ore resources (Mt)</td>
<td>Grade Cr2O3 in 2008 ore prod (%)</td>
</tr>
<tr>
<td></td>
<td>Prod Cu 2000 (kt)</td>
<td>Ore reserves (Mt)</td>
<td>Grade U3O8 in 2008 ore prod (%)</td>
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<td></td>
<td>Prod Cu 2001 (kt)</td>
<td>Ore prod 2009 (Mt)</td>
<td>Grade diamond in 2008 ore prod (ct/t)</td>
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<tr>
<td></td>
<td>Prod Cu 2002 (kt)</td>
<td>Ore prod 2008 (Mt)</td>
<td>ROM prod 2008 (Mt)</td>
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<tr>
<td></td>
<td>Prod Cu 2003 (kt)</td>
<td>Cut-off (% or gpt)</td>
<td>Ore prod 2007 (Mt)</td>
</tr>
<tr>
<td></td>
<td>Prod Cu 2004 (kt)</td>
<td>Recovery Main metal (%)</td>
<td>Grade PGM in 2007 ore prod (gpt)</td>
</tr>
<tr>
<td></td>
<td>Prod Cu 2005 (kt)</td>
<td>Recovery Metal 2 (%)</td>
<td>Ore prod 2006 (Mt)</td>
</tr>
<tr>
<td></td>
<td>Prod Cu 2006 (kt)</td>
<td>Recovery Metal 3 (%)</td>
<td>Grade Cu in 2006 ore prod (%)</td>
</tr>
<tr>
<td></td>
<td>Prod Cu 2007 (kt)</td>
<td>Mill capacity (t/d)</td>
<td>Grade PGM in 2006 ore prod (gpt)</td>
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<tr>
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<td>Prod Cu 2008 (kt)</td>
<td>Recovery method</td>
<td>Ore prod 2005 (Mt)</td>
</tr>
<tr>
<td></td>
<td>Prod Cu 2009 (kt)</td>
<td>Expec. annual ROM (Mt)</td>
<td>Grade PGM in 2005 ore prod (gpt)</td>
</tr>
<tr>
<td>Main metal</td>
<td></td>
<td>Metal 2</td>
<td>Ore prod 2004 (Mt)</td>
</tr>
<tr>
<td>Metal 2</td>
<td></td>
<td>Mine capacity (Mt/yr)</td>
<td>Grade PGM in 2004 ore prod (gpt)</td>
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<tr>
<td>Metal 3</td>
<td></td>
<td>Expec. annual metal cap</td>
<td>Ore prod 2003 (Mt)</td>
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<td>Metal 4</td>
<td></td>
<td>Expec. annual metal2 cap</td>
<td>Grade Ag in 2003 ore prod (gpt)</td>
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<tr>
<td>Metal 5</td>
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<td>Expec. annual metal3 cap</td>
<td>Grade PGM in 2003 ore prod (gpt)</td>
</tr>
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<td>Ore resources (Mt)</td>
<td>Geological model</td>
<td></td>
<td></td>
</tr>
<tr>
<td>Ore reserves (Mt)</td>
<td></td>
<td></td>
<td></td>
</tr>
</tbody>
</table>
It should be noted that this is only an excerpt as there are some more data items that can be retrieved. For example, annual production of a number of other metals than copper is available. Annual data is reported from 1998 onwards. Furthermore, not all data is available for all mines. Metal production is mostly available. Ore production is already available to a lesser extent. As can be seen also some data is not available on an annual basis, e.g. mine type mainly OP or UG.

### A.2.1 Calculated non-mine specific recovery values

Average metal specific recoveries have been calculated from RMD, which were used as default recovery values, if no mine specific information was available. For determination the non-mine specific recoveries a distinction was made between which metals were produced as main metal, except for PGMs. The metal specific recoveries were calculated by weighting with the metal production of the mine.

**Table A.3 Average recovery values determined based on RMD.**

<table>
<thead>
<tr>
<th>Metal recovered</th>
<th>Main metal at mine</th>
<th>Recovery (mass%)</th>
</tr>
</thead>
<tbody>
<tr>
<td>Mo</td>
<td>Mo</td>
<td>79</td>
</tr>
<tr>
<td>Mo</td>
<td>Cu</td>
<td>59</td>
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<td>Zn</td>
<td>Zn, Pb</td>
<td>84</td>
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<tr>
<td>Zn</td>
<td>Other than Zn or Pb</td>
<td>71</td>
</tr>
<tr>
<td>Pb</td>
<td>Pb, Ag</td>
<td>86</td>
</tr>
<tr>
<td>Pb</td>
<td>Other than Pb or Ag</td>
<td>73</td>
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<tr>
<td>Cu</td>
<td>Cu</td>
<td>82</td>
</tr>
<tr>
<td>Cu</td>
<td>Other than Cu</td>
<td>82</td>
</tr>
<tr>
<td>Ag</td>
<td>Ag</td>
<td>84</td>
</tr>
<tr>
<td>Ag</td>
<td>Other than Ag</td>
<td>74</td>
</tr>
<tr>
<td>Au</td>
<td>Au, Ag</td>
<td>86</td>
</tr>
<tr>
<td>Au</td>
<td>Other than Au or Ag</td>
<td>71</td>
</tr>
<tr>
<td>Ni</td>
<td>Ni</td>
<td>83</td>
</tr>
<tr>
<td>Ni</td>
<td>Other than Ni</td>
<td>na</td>
</tr>
<tr>
<td>Co</td>
<td>Other than Co</td>
<td>52</td>
</tr>
<tr>
<td>PGM</td>
<td>PGM</td>
<td>83</td>
</tr>
</tbody>
</table>

*na – no data*

### A.2.2 Data processing step 1

Data was analysed for mines and mine groups producing molybdenum, zinc, copper, lead, silver, gold, nickel, cobalt and PGMs. Mines producing other metals were not considered for now.

Before the mass of ore needed per unit mass of metal processed was determined for each year in the period 1998-2010, some initial processing of the available data was necessary (Figure 2). For each mine and year for which ore production was available, the amount of each metal processed and the amount of each metal recovered were determined. The mass of metal i processed was calculated by multiplying ore production (mined/milled) with the ore grade of metal i for the specific year and project. If grade data was not available the metal mine production has been used to estimate the metal content of the ore, based on average recovery data for the metal and mine for other years or default recovery values. If metal production
data was available this was used as amount of metal recovered unless the metal production exceeded the amount of metal processed. In the latter case the average recovery of the mine and metal or default recoveries were used to estimate metal recovered from grade and ore production data. Metal recovery was set to 0, if (a) it could not be assumed that the metal was produced at the mine, i.e., neither was the metal one of the metals named as being of economic importance for the mine nor was production data available, or (b) if the production of the metal was less than 5% of the product of ore grade and ore production, respectively if the average recovery of the mine for this metal was below 5%. The production of the metal at the latter mines is of little relevance for the global production of the metal, but can have a high influence on the overall amount of ore required per metal in ore, as little ore is allocated to it, while a rather large amount of metal is assumed to be in the ore. For PGMs no individual ore grades were available, instead only the overall PGM grade was given. In that case the processed amount has been estimated either based on the total amount of PGMs processed and the production of the individual PGMs (platinum, palladium, rhodium) or total PGM production and ratios of palladium and platinum production for the specific countries as calculated from USGS Minerals Yearbook, if no metal production data for the individual PGMs was available. The latter procedure was used only in a very few cases. The exact procedure for PGMs has not been included in the scheme shown in Figure A.1 to limit its complexity.
Figure A.1 Flowchart of step 1 in the data processing.
A.2.3 Data processing step 2

In this step (Figure A.2 Step 2 of the data processing: Calculation of overall annual data per metal (total mass recovered, total mass processed, total ore production, total allocated ore production, specific ore requirements). Figure A.2) first for each metal (i) the mines (k) at which it is recovered are identified. Then for each year (j) the total mass of metal i processed and recovered and the total ore production at the identified mines are computed. Next is the calculation of the total amount of ore production allocated to metal i, which is expected to be lower than the total amount of ore to the extent that other metals are recovered at the mines. Allocation has been performed based on the economic value of the recovered metals. The economic value is expressed in terms of value of the refined metal and as such is not the real economic value of the metal mine production, but a refined metal equivalent economic value. The economic value per unit mass of the metal is represented by 10 year averaged prices expressed in year 2000 US$/kg (Table A.4, Table A.5). When for a metal neither ore grade nor production data was available though the metal was named as being of economic importance the recovered amount was set to zero. This can occur either because there just is no data available concerning the metal in question for that mine and year, or indeed because the metal has not been produced (yet) at the mine. For the allocated ore amounts annual sums over all datasets with a positive amount of metal recovered have been calculated. The ratio of total mass of ore allocated to metal i in year j over the total amount of metal i processed in year j then represents the average specific ore requirements per unit mass of metal i in ore for that year.
Figure A.2 Step 2 of the data processing: Calculation of overall annual data per metal (total mass recovered, total mass processed, total ore production, total allocated ore production, specific ore requirements).

<table>
<thead>
<tr>
<th></th>
<th></th>
<th></th>
<th></th>
<th></th>
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<th></th>
<th></th>
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<tbody>
<tr>
<td>$/2000</td>
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<td>1.13</td>
<td>1.10</td>
<td>1.07</td>
<td>1.04</td>
<td>1.03</td>
<td>1.00</td>
<td>0.97</td>
<td>0.95</td>
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</table>

Table A.5 Prices per kg of commodity (refined metal). Original price data sourced from USGS (2009, 2005, 2003)

<table>
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<tr>
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<th></th>
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<th></th>
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</thead>
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<td>Ag $</td>
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<td>13213</td>
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<td>35.19</td>
<td>52.76</td>
<td>23.37</td>
<td>15.23</td>
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<td>6.72</td>
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<tr>
<td>Ni $</td>
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<td>30.91</td>
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<tr>
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<td>2.58</td>
<td>2.58</td>
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<td>2.58</td>
<td>2.58</td>
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<td>1.34</td>
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<td>1.34</td>
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</tr>
<tr>
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<td>6544</td>
<td>7489</td>
<td>6527</td>
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<td>19635</td>
<td>22243</td>
<td>11677</td>
<td>9316</td>
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<td>Pd $</td>
<td>11081</td>
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<td>8868</td>
<td>5770</td>
<td>6827</td>
<td>6108</td>
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<td>$2000</td>
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<td>29076</td>
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<td>63980</td>
<td>29076</td>
<td>19928</td>
<td></td>
</tr>
</tbody>
</table>
A.2.4 Illustrative example for step 2 data processing

In the following the production data for 2005 of the Kidd Creek Mine and the Storliden mine (Table A.6) will serve as an example to explain data processing step 2 for the case of copper. To calculate the allocated amount of ore per kg copper, it is first checked that both mines actually produced copper. As this is the case, the mass of copper processed of both mines can be summed to obtain the total amount copper processed \((46009 \text{ t Cu} + 11803 \text{ t Cu} = 57812 \text{ t Cu})\). The next step is the calculation of the amount of ore production allocated to copper, which is expected to be lower than the total amount of ore when other metals are recovered alongside copper. The amount of ore treated has been allocated based on the economic value of the recovered metals. The total value recovered is calculated by multiplying the mass of metal recovered by its metal price and summing the resulting products for all recovered metals (Kidd Creek: \(42700 \text{ t Cu} \times 2750 \text{ US$}/\text{t Cu} + 120000 \text{ t Cu} \times 1340 \text{ US$}/\text{t Cu} + 114.4 \text{ t Cu} \times 203800 \text{ US$}/\text{t Cu} = 3.0 \times 10^8 \text{ US$}\)). The used metal prices are the 10 year averaged prices of refined metal expressed in year 2000 US$/kg. This is a simplification which assumes that the price ratios of the refined metals are comparable to those of the metal in unrefined form. The fraction of value contributed by copper has been calculated by multiplying the amount of copper recovered with its price and dividing this by the total value recovered (Kidd Creek: \((42700 \text{ t Cu} \times 2750 \text{ US$}/\text{t Cu}) / 3.0 \times 10^8 \text{ US$} = 0.39\)). The ore production allocated to the copper is then obtained by multiplying this fraction with the total ore production (Kidd Creek: \(0.39 \times 2312000 \text{ t ore} = 900580 \text{ t ore}\)). The same can be done for the Storliden mine resulting in an ore production allocated to copper of 125087 t ore. Summing the allocated ore production for both mines results in the total ore production allocated to Cu in 2005: 1025668 t. The unit mass of ore allocated to copper per unit mass copper in ore would then be: 1025668 t ore/57812 t Cu = 17.7 kg ore/kg Cu.

Table A.6 Production data of the Kidd Creek and Storliden mines for 2005.

<table>
<thead>
<tr>
<th>Mine</th>
<th>Ore production (t)</th>
<th>Cu processed (t)</th>
<th>Zn processed (t)</th>
<th>Au processed (t)</th>
<th>Ag processed (t)</th>
<th>Cu recovered (t)</th>
<th>Zn recovered (t)</th>
<th>Au recovered (t)</th>
<th>Ag recovered (t)</th>
</tr>
</thead>
<tbody>
<tr>
<td>Kidd Creek</td>
<td>2312000</td>
<td>46009</td>
<td>141032</td>
<td>0</td>
<td>180.3</td>
<td>42700</td>
<td>120000</td>
<td>0</td>
<td>114.4</td>
</tr>
<tr>
<td>Storliden</td>
<td>319000</td>
<td>11803</td>
<td>34771</td>
<td>0.128</td>
<td>11.2</td>
<td>10800</td>
<td>32000</td>
<td>0.091</td>
<td>8.3</td>
</tr>
</tbody>
</table>

A.2.5 Importance of co-mining

In mining rarely only one metal is recovered at the mine. Accordingly efforts are related to the mining and processing of a number of metals together. As an example Table A.6 shows the metals which are mined alongside copper and the relative amounts recovered, as well as the relative value of the refined metal equivalent. This shows that though copper is obviously the dominant metal in terms of mass and even value, the other metals contribute considerably and thus ore production and as a consequence the environmental interventions caused by the ore treatment cannot all be attributed to copper alone.
Table A.7 Copper co-mining, characteristics for the period 1999-2009.

<table>
<thead>
<tr>
<th>Metal</th>
<th>Average mass% of total metal mass recovered at Cu recovering mines</th>
<th>Average value% of the total refined metal value recovered at copper recovering mines</th>
</tr>
</thead>
<tbody>
<tr>
<td>Cu</td>
<td>74</td>
<td>56</td>
</tr>
<tr>
<td>Zn</td>
<td>18</td>
<td>7</td>
</tr>
<tr>
<td>Pb</td>
<td>5</td>
<td>1</td>
</tr>
<tr>
<td>Ni</td>
<td>3</td>
<td>10</td>
</tr>
<tr>
<td>Mo</td>
<td>&lt;1</td>
<td>4</td>
</tr>
<tr>
<td>Ag</td>
<td>&lt;1</td>
<td>3</td>
</tr>
<tr>
<td>Au</td>
<td>&lt;1</td>
<td>5</td>
</tr>
<tr>
<td>Others</td>
<td>&lt;1</td>
<td>15</td>
</tr>
</tbody>
</table>

Allocation has a considerable effect on the amount of ore/kg or metal in ore. This is exemplified for zinc in Figure A.3. The amount of ore allocated per kg of Zn is only about half the amount of ore considered, when no allocation is performed. Moreover, allocation also flattens out variation in ore grade due to changes in the extent of co-mining. Similar observations can be made for other metals (See Figure A.4). The results are a good illustration that mines can operate economically at lower individual ore grades, if several metals are recovered from the ore.

Figure A.3 Comparison of specific ore requirements for zinc in case of no allocation and in case of value based allocation.
A.2.6 Ore demand: annual results per metal

Graphs A to I:

For the average kg of ore per kg of metal i in ore the data point represents the ratio between total ore processed with metal i recovery and the total amount of metal i in this ore. This can also be seen as an average of the ratio of ore processed to metal i in ore at individual operational units weighted by the ore production. The whiskers represent the range between the first and the third quartile of the unweighted ore to metal i ratio of individual operational units. The percentiles are determined based on the cumulated amount of copper in ore. For example, the 25% percentile represents the ore to metal ratio for which 25% of the total copper processed were processed at operational units with an equal or lower ore to metal ratio.

It should be noted that the reported ranges reflect the wide variability in ore grades depending on the characteristics of the deposit and also on the allocation. Even though current average ore grades for copper are around 0.8% (Crowson, 2012), this does not mean that all mines have grades similar to that. For example for 2008 there are a total of 219 operational units in RMD with Cu ore grade data, of this 76 have a grade below or equal to 0.5% copper and 34 have copper grades greater than or equal to 2%. This range still exceeds the change in average copper ore grades between 1950 and today (Gerst, 2008).

Graphs J to Q:

Next to the allocated ore demand per copper in ore, allocated ore demand per copper recovered and unallocated ore demand per copper in ore are depicted. In addition the percentage of global production of the metal covered by the data is represented.

A

![Graph showing kg ore/kg Cu (in ore) and % of global production covered by data with years from 1998 to 2010.]

- Coverage
- Ore per Cu (in ore)

Total number of mines per year over all years:
133-270

Total number of mines per year over years included in CF calculation:
177-270

Years included in CF calculation:
1999-2009
B

Total number of mines per year over all years:
28-75

Total number of mines per year over years included in CF calculation:
48-75

Years included in CF calculation:
2000-2009

C

Total number of mines per year over all years:
15-29

Total number of mines per year over years included in CF calculation:
15-29

Years included in CF calculation:
1998-2008
Total number of mines per year over all years:
76-156

Total number of mines per year over years included in CF calculation:
107-156

Years included in CF calculation:
1999-2007

Total number of mines per year over all years:
64-128

Total number of mines per year over years included in CF calculation:
74-128

Years included in CF calculation:
1998-2006
Total number of mines per year over all years: 294-416

Total number of mines per year over years included in CF calculation: 357-416

Years included in CF calculation: 1999-2009

Total number of mines per year over all years: 135-218

Total number of mines per year over years included in CF calculation: 169-218

Years included in CF calculation: 1998-2009
Total number of mines per year over all years:
17-34

Total number of mines per year over years included in CF calculation:
17-34

Years included in CF calculation:
1999-20010

Total number of mines per year over all years:
20-35

Total number of mines per year over years included in CF calculation:
20-35

Years included in CF calculation:
1998,2000-2010

Percentage of global production covered by the data.

kg of ore processed per kg of copper in ore

kg of processed ore allocated to copper per kg copper in ore

kg of processed ore allocated to copper per kg of copper recovered
Percentage of global production covered by the data.

- kg of ore processed per kg of nickel in ore
- kg of processed ore allocated to nickel per kg nickel in ore
- kg of processed ore allocated to nickel per kg of nickel recovered

Percentage of global production covered by the data.

- kg of ore processed per kg of molybdenum in ore
- kg of processed ore allocated to molybdenum per kg molybdenum in ore
- kg of processed ore allocated to molybdenum per kg of molybdenum recovered

Percentage of global production covered by the data.

- kg of ore processed per kg of zinc in ore
- kg of processed ore allocated to zinc per kg zinc in ore
- kg of processed ore allocated to zinc per kg of zinc recovered
Percentage of global production covered by the data.

- kg of ore processed per kg of lead in ore
- kg of processed ore allocated to lead per kg lead in ore
- kg of processed ore allocated to lead per kg of lead recovered

Percentage of global production covered by the data.

- kg of ore processed per kg of gold in ore
- kg of processed ore allocated to gold per kg gold in ore
- kg of processed ore allocated to gold per kg of gold recovered

Percentage of global production covered by the data.

- kg of ore processed per kg of silver in ore
- kg of processed ore allocated to silver per kg silver in ore
- kg of processed ore allocated to silver per kg of silver recovered
Figure A.4 Derived annual unallocated and allocated ore requirements as well as annual coverage for nine metals.
Figure A.5 Equations used to estimate fossil energy equivalent requirements at the mining and processing stage up to copper cathode. FEE values are constant parameters, while the MN_{ij}, OM_{ij}, OT_{ij}, and CC_{ij} depend on the specified operational unit i and the year j for which the energy requirement is to be estimated.

\[
\begin{align*}
\text{Energy mining} & = \text{FEE}_{\text{op}} \times \text{OM}_{ij} \\
\text{Energy processing} & = \text{FEE}_{\text{op}} \times \text{OT}_{ij} + \text{FEE}_{\text{smelting}} \times \text{CC}_{ij} \\
\text{Energy processing} & = \text{FEE}_{\text{op}} \times \text{OT}_{ij} + \text{FEE}_{\text{smelting}} \times \text{CC}_{ij} \\
\text{Energy processing} & = \text{FEE}_{\text{op}} \times \text{OT}_{ij} + \text{FEE}_{\text{smelting}} \times \text{CC}_{ij} \\
\text{Energy processing} & = \text{FEE}_{\text{op}} \times \text{OT}_{ij} + \text{FEE}_{\text{smelting}} \times \text{CC}_{ij} \\
\text{Energy processing} & = \text{FEE}_{\text{op}} \times \text{OT}_{ij} + \text{FEE}_{\text{smelting}} \times \text{CC}_{ij} \\
\text{Energy processing} & = \text{FEE}_{\text{op}} \times \text{OT}_{ij} + \text{FEE}_{\text{smelting}} \times \text{CC}_{ij}
\end{align*}
\]
A.2.8 Constant parameters used in equations

Table A.8 Electricity and fuel demands used for the calculation of the total fossil energy equivalent (FEE) parameters for each process step considered in the models are reported. For conversion of electrical energy to FEE the assumption was made that the electricity was produced by diesel generators. When more than one literature source was used the arithmetic mean of the values was taken.

<table>
<thead>
<tr>
<th>Process step</th>
<th>Unit</th>
<th>Electricity demand</th>
<th>Direct fossil energy - fuels</th>
<th>Reference</th>
<th>FEE electricity demand</th>
<th>Total specific FEE</th>
</tr>
</thead>
<tbody>
<tr>
<td>A: Underground mining</td>
<td>MJ/kg ore</td>
<td>0.07</td>
<td>0.12</td>
<td>Norgate TE, Haque N. Energy and greenhouse gas impacts of mining and mineral processing operations. Journal of Cleaner Production 2010;18(3):266–274</td>
<td>0.21</td>
<td>0.32</td>
</tr>
<tr>
<td>B: Open pit mining</td>
<td>MJ/kg material</td>
<td>0.001</td>
<td>0.012</td>
<td>Marsden JO. Energy Efficiency &amp; Copper Hydrometallurgy. In: Young CA, Taylor PR, Anderson CG, Choi Y, editors. Hydrometallurgy 2008 - Proceedings of the Sixth International Symposium. Phoenix Arizona: Society for Mining, Metallurgy, and Exploration (SME); 2008,. p. 29–42.</td>
<td>0.004</td>
<td>0.016</td>
</tr>
<tr>
<td></td>
<td>Process Description</td>
<td>MJ/kg ore</td>
<td>Upper Bound</td>
<td>Lower Bound</td>
<td></td>
<td></td>
</tr>
<tr>
<td>---</td>
<td>--------------------------------------------------------------------------------------</td>
<td>-----------</td>
<td>-------------</td>
<td>-------------</td>
<td></td>
<td></td>
</tr>
<tr>
<td>C3</td>
<td>Comminution + Concentration (Ni/Cu ore)</td>
<td>0.11</td>
<td>0.32</td>
<td>0.32</td>
<td></td>
<td></td>
</tr>
<tr>
<td>F1</td>
<td>Comminution + heap leaching</td>
<td>0.016</td>
<td>0.044</td>
<td>0.044</td>
<td></td>
<td></td>
</tr>
<tr>
<td>F2</td>
<td>Run-of-mine leaching</td>
<td>0.004</td>
<td>0.011</td>
<td>0.011</td>
<td></td>
<td></td>
</tr>
<tr>
<td>G</td>
<td>Solvent extraction</td>
<td>4.35</td>
<td>12.16</td>
<td>12.16</td>
<td></td>
<td></td>
</tr>
<tr>
<td>H</td>
<td>Electrowinning</td>
<td>7.15</td>
<td>21.28</td>
<td>21.28</td>
<td></td>
<td></td>
</tr>
<tr>
<td>D</td>
<td>Smelting</td>
<td>5.05</td>
<td>14.09</td>
<td>23.03</td>
<td></td>
<td></td>
</tr>
<tr>
<td>E</td>
<td>Refining</td>
<td>1.59</td>
<td>4.45</td>
<td>6.17</td>
<td></td>
<td></td>
</tr>
</tbody>
</table>

The comminution value from Marsden (2008) was calculated as an arithmetic average of three comminution routes described there. From the other references overall values for beneficiation were taken as reported.
### A.2.9 Mines used for validation

Table A.9 The operational units used for validation of the model were grouped according to the combination of mining and processing route. The second column specifies the years for which data was collected and the third column names the sources of data. Operational units following the UG-SXEW route could not be included. Till now they are rather rare.

<table>
<thead>
<tr>
<th>Operational unit</th>
<th>Years</th>
<th>References for energy consumption</th>
</tr>
</thead>
<tbody>
<tr>
<td><strong>UG-conventional</strong></td>
<td></td>
<td></td>
</tr>
<tr>
<td>Pyhäsaalmi</td>
<td>2006-2010</td>
<td>Inmet Mining. 2010 sustainability report. Inmet Mining; 2011</td>
</tr>
<tr>
<td><strong>OP-conventional</strong></td>
<td></td>
<td></td>
</tr>
</tbody>
</table>
OP-SXEW

- Highland Valley Copper
  
<table>
<thead>
<tr>
<th>Year</th>
<th>Description</th>
</tr>
</thead>
<tbody>
<tr>
<td></td>
<td>Aguablanca Polymetallic</td>
</tr>
<tr>
<td></td>
<td>Antofagasta. Sustainability Report 2009. 2010</td>
</tr>
<tr>
<td></td>
<td>Los Pelambres</td>
</tr>
<tr>
<td></td>
<td>Antofagasta. Sustainability Report 2009. 2010</td>
</tr>
</tbody>
</table>

- Cerro Colorado
  
<table>
<thead>
<tr>
<th>Year</th>
<th>Description</th>
</tr>
</thead>
<tbody>
<tr>
<td></td>
<td>BHP Billiton. Informe de Sustantabilidad 2009 - Pampa Norte. BHP Billiton; 2010</td>
</tr>
<tr>
<td></td>
<td>El Tesoro</td>
</tr>
<tr>
<td></td>
<td>Antofagasta. Sustainability Report 2009. 2010</td>
</tr>
<tr>
<td></td>
<td>Antofagasta. Reporte de Sustentabilidad 2010 - Minera El Tesoro. Antofagasta; 2011</td>
</tr>
<tr>
<td></td>
<td>Lomas Bayas</td>
</tr>
<tr>
<td></td>
<td>Division Norte de Chile Xstrata Copper. Memoria de Sostenibilidad 2007 - Division Norte de Chile Xstrata Copper. Division Norte de Chile Xstrata Copper; 2008</td>
</tr>
<tr>
<td></td>
<td>Xstrata Copper North Chile Division. 2006 Sustainability Report Xstrata Copper North Chile Division. Division Norte de Chile Xstrata Copper; 2007</td>
</tr>
<tr>
<td></td>
<td>Xstrata Copper North Chile Division. North Chile Sustainability report 2008. Division Norte de Chile Xstrata Copper; 2009</td>
</tr>
<tr>
<td></td>
<td>Mantoverde</td>
</tr>
<tr>
<td></td>
<td>Minera Spence</td>
</tr>
<tr>
<td>2007-2009</td>
<td>BHP Billiton. Informe de Sustantabilidad 2009 - Pampa Norte. BHP Billiton; 2010</td>
</tr>
<tr>
<td></td>
<td>Zaldivar</td>
</tr>
</tbody>
</table>
A.2.10 Conversion factors fuels for validation data

Table A. 10 The conversion values were sourced from VITO (2009).

<table>
<thead>
<tr>
<th></th>
<th>unit</th>
<th>net</th>
</tr>
</thead>
<tbody>
<tr>
<td>Petrol</td>
<td>MJ/l</td>
<td>32.3</td>
</tr>
<tr>
<td>Diesel</td>
<td>MJ/l</td>
<td>35.9</td>
</tr>
<tr>
<td>Light fuel oil</td>
<td>MJ/l</td>
<td>36.8</td>
</tr>
<tr>
<td>Light fuel oil</td>
<td>MJ/kg</td>
<td>42.3</td>
</tr>
<tr>
<td>Natural gas</td>
<td>MJ/l</td>
<td>0.0378</td>
</tr>
<tr>
<td>Natural gas</td>
<td>MJ/kg</td>
<td>46.8</td>
</tr>
<tr>
<td>LPG</td>
<td>kg/l</td>
<td>0.54</td>
</tr>
<tr>
<td>LPG</td>
<td>MJ/kg</td>
<td>45.8</td>
</tr>
<tr>
<td>Diesel</td>
<td>MJ/kg</td>
<td>42.75</td>
</tr>
<tr>
<td>Petrol</td>
<td>MJ/kg</td>
<td>43</td>
</tr>
<tr>
<td>Kerosene</td>
<td>MJ/kg</td>
<td>43</td>
</tr>
<tr>
<td>Kerosene</td>
<td>MJ/l</td>
<td>34.4</td>
</tr>
<tr>
<td>Cokes</td>
<td>MJ/kg</td>
<td>29</td>
</tr>
<tr>
<td>Coal</td>
<td>MJ/kg</td>
<td>30.9</td>
</tr>
</tbody>
</table>

A.2.11 Handling of missing data at validation and implementation

When it could not be determined whether run-of-mine leaching was performed or how much of it was leached at an operation, it was assumed that all ore production would be crushed and heap leached. If the number of years for which no data was available was limited, it was assumed that the ratio heap leached ore and run-of-mine ore was the same as in other years for the same operational unit. If the amount of material mined, respectively the amount of waste or the waste to ore ratio, were not at all available for OP mines the waste to ore ratio listed in RMD was used, which is not year specific. If this was also unavailable a default waste to ore ratio of three (Marsden, 2008) was assumed. When the lack of data for mined waste rock only concerned a limited number of years, then the ratio of material moved to ore processed of the same operational unit for the neighbouring years was used to determine the mass of material moved in a year for which no data could be found.

A.2.12 List of reports used supplementary to mining database RMD

Jönsson H. AITIK 36 – A world class copper project north of the arctic circle. 2010


M.I.M. Third Quarter Production Results for 9 Months to 31 MARCH 1999. MIM Holdings Ltd; 1999.


Penoles. MINERA MADERO, S.A. DE C.V. Production-Producción [Internet]. Penoles; 2012b. [cited 2013 Feb 5] Available from:
disp/prodextranet001670.xls


landing/varvara/operations.aspx

anual-report-2010.pdf


Sally Malay Mining. Quarterly Report for the period ending 31 December 2005. Sally Malay Mining; 2006a

Sally Malay Mining. Quarterly Report for the period ending 31 June 2006. Sally Malay Mining; 2006b

Sally Malay Mining. Quarterly Report for the period ending 31 December 2006. Sally Malay Mining; 2007a
Sally Malay Mining. Quarterly Report for the period ending 31 June 2007. Sally Malay Mining; 2007b

Sally Malay Mining. Quarterly Report for the period ending 31 December 2007. Sally Malay Mining; 2008


A.2.13 Copper production distribution in terms of specific ore demand

Figure A.6 Total copper production of the considered operational units between 1998 and 2010 in function of specific ore demand. In case of the OP-SXEW route there was also some copper production above 1000 kg ore/kg Cu, which is not shown in the respective chart, but this only represented less than 1% of the total.
Appendix 3
Case studies

A.3.1 Additional results of the scenario analysis performed for chapter 7.

Figure A.7 Variation of the CEENE fingerprint of the germanium wafer production in function of the electricity mix used in the production itself and the most important upstream processes.

NO = Norway, CH–Switzerland, AT = Austria, IT = Italy, FR = France, BE = Belgium, RO = Romania, NL = Netherlands, FI = Finland, ES = Spain, JP = Japan, DE = Germany, CZ = Czech Republic, US = United States of America, CN = China, PL = Poland, GR = Greece.
A.3.2 Process inventory data for primary cobalt

Table A.11 Inventory for 1 kg of Co (mining and refining up to Co(OH); allocated between copper and Co(OH); based on exergy content)

<table>
<thead>
<tr>
<th>Input</th>
<th>Comment</th>
<th>Amount</th>
<th>Unit</th>
<th>Input</th>
</tr>
</thead>
<tbody>
<tr>
<td>Diesel</td>
<td>(Roomanay and Gediga, 2011); allocation exergy</td>
<td>0.22</td>
<td>kg</td>
<td>diesel, at regional storage (RER)</td>
</tr>
<tr>
<td>Electricity</td>
<td>(Roomanay and Gediga, 2011); allocation exergy</td>
<td>13.2</td>
<td>kWh</td>
<td>own approximation: electricity, medium voltage (Africa)</td>
</tr>
<tr>
<td>Cobalt (in ore, in ground)</td>
<td>60% cobalt recovery</td>
<td>1.67</td>
<td>kg</td>
<td></td>
</tr>
<tr>
<td>Magnesium oxide</td>
<td>stoichiometric</td>
<td>0.68</td>
<td>kg</td>
<td>magnesium oxide, at plant</td>
</tr>
<tr>
<td>Water</td>
<td>Total water input</td>
<td>1929</td>
<td>kg</td>
<td>tap water, at user (RER) + direct input from environment</td>
</tr>
</tbody>
</table>

Table A.12 Transport distances assumed for primary cobalt.

Transport: Central Africa to Asian plant
Ship: 11000 km
Road: 2000 km

Transport: Central Africa to European plant
Ship: 11500 km
Road: 2000 km
Table A.13 Inventory leaching of 1 kg of cobalt.

<table>
<thead>
<tr>
<th>Input</th>
<th>Comment</th>
<th>Amount</th>
<th>Unit</th>
<th>Ecoinvent dataset</th>
</tr>
</thead>
<tbody>
<tr>
<td>Sulphuric acid</td>
<td>stoichiometric</td>
<td>1.68</td>
<td>kg</td>
<td>sulphuric acid, liquid, at plant (RER)</td>
</tr>
<tr>
<td>Water</td>
<td>stoichiometric</td>
<td>0.271</td>
<td>kg</td>
<td>water, deionised, at plant (CH)</td>
</tr>
<tr>
<td>Electricity</td>
<td>Electricity for pumping</td>
<td>0.08</td>
<td>kWh</td>
<td>electricity, medium voltage, at grid (CN)</td>
</tr>
<tr>
<td>Steam</td>
<td>not accounted to avoid double counting with heat input in subsequent mixing of metal sulphates at the precursor production as that heat input is based on cobalt provided in crystal form</td>
<td></td>
<td></td>
<td></td>
</tr>
</tbody>
</table>

A.3.3 Process inventory data for primary nickel

Table A.14 Inventory for 1 kg Ni in nickel sulphate hexahydrate.

<table>
<thead>
<tr>
<th>Input</th>
<th>Comment</th>
<th>Amount</th>
<th>Unit</th>
</tr>
</thead>
<tbody>
<tr>
<td>Class I nickel metal</td>
<td>(Ecobalance, Inc., 2000)</td>
<td>1</td>
<td>kg</td>
</tr>
<tr>
<td>Nickel purification</td>
<td>inputs for 1 kg of Ni in NiSO4.6H2O</td>
<td>1</td>
<td>unit</td>
</tr>
</tbody>
</table>
### A.3.4 Process inventory data for primary manganese

Table A.15 Available information for Chinese manganese sulphate monohydrate plant (SRK Consulting China Ltd et al., 2010)

<table>
<thead>
<tr>
<th>Input</th>
<th>Comment</th>
<th>Amount</th>
<th>Unit</th>
<th>Ecoinvent dataset</th>
</tr>
</thead>
<tbody>
<tr>
<td>Manganese ore</td>
<td>Average recovery of the plant and average grade of ore</td>
<td>5.3</td>
<td>kg</td>
<td>manganese concentrate, at beneficiation (GLO)</td>
</tr>
<tr>
<td>Coal powder</td>
<td>20% coal powder mixed to ore</td>
<td>1.1</td>
<td>kg</td>
<td>hard coal supply mix (CN)</td>
</tr>
<tr>
<td>Sulphuric acid</td>
<td>Ore to acid ratio of</td>
<td>3.85</td>
<td>kg</td>
<td>sulphuric acid, liquid, at plant (RER)</td>
</tr>
<tr>
<td>Heat crystallisation</td>
<td>Heated to 105°C and evaporation of all water, except the crystal water</td>
<td>22.5</td>
<td>MJ</td>
<td>heat, unspecified, in chemical plant (RER)</td>
</tr>
</tbody>
</table>

### A.3.5 Background processes of the cathode material production not directly obtained from the ecoinvent database

Most datasets for background processes were taken from the ecoinvent database.

Notable exclusions are:

1) The electricity and steam provided to the cobalt refining and nickel purification.
2) Electricity provided to the lithiation process situated in Korea.
3) Electricity used in the cobalt mining and refining.
4) Compressed air at 3 and 1.5 bar.

1) Both inputs were assumed to be fully supplied by the combined heat and power (CHP) installation onsite. For the CHP electricity and steam production per kWh of natural gas were made available. Exergy was used to allocate the natural gas between the produced steam and electricity. The CHP installation as such was not considered.
2) Electricity mix Korea: Information on the electricity mix was obtained from the International Energy Agency website (International Energy Agency, 2011a). The last 2% of electricity production sources were neglected. This mix was used as a
basis to calculate a CEENE value for the electricity provided to the lithiation taking place in Korea.

Table A.16 Korean electricity mix based on International Energy Agency (2011a).

<table>
<thead>
<tr>
<th>Electricity source</th>
<th>Contribution</th>
</tr>
</thead>
<tbody>
<tr>
<td>%</td>
<td>kWh</td>
</tr>
<tr>
<td>Nuclear</td>
<td>33%</td>
</tr>
<tr>
<td>Oil</td>
<td>4%</td>
</tr>
<tr>
<td>Natural gas</td>
<td>16%</td>
</tr>
<tr>
<td>Coal &amp; Peat</td>
<td>47%</td>
</tr>
<tr>
<td>Total</td>
<td>100%</td>
</tr>
</tbody>
</table>

3) Electricity mix Africa: Information on the electricity mix was obtained from the International Energy Agency website (International Energy Agency, 2011b). The last 3% of electricity production sources were neglected. This electricity mix was used as a basis to calculate a CEENE value for the electricity provided to the mining and refining of cobalt containing ore into cobalt.

Table A.17 African electricity mix based on International Energy Agency (2011b)

<table>
<thead>
<tr>
<th>Electricity source</th>
<th>Contribution</th>
</tr>
</thead>
<tbody>
<tr>
<td>%</td>
<td>kWh</td>
</tr>
<tr>
<td>Hydropower</td>
<td>16%</td>
</tr>
<tr>
<td>Oil</td>
<td>13%</td>
</tr>
<tr>
<td>Natural gas</td>
<td>30%</td>
</tr>
<tr>
<td>Coal &amp; Peat</td>
<td>41%</td>
</tr>
<tr>
<td>Total</td>
<td>100.00%</td>
</tr>
</tbody>
</table>

4) Linear extrapolation from 6 bar gauge resource consumption based on ecoinvent (Steiner and Frischknecht, 2007). For the extrapolation resource consumption at 0 bar gauge was set to zero.

Some background processes were not included or represented by somewhat different datasets due to lack of data:

- The input of de-ironed groundwater was represented by the ecoinvent dataset for tap water, as no detailed data on the processing or pumping requirements was available.

- Waste water treatment for waste water from foreground processes was only accounted for up to the physical treatment included in the process plants.

- Treatment of solid wastes was not included for the possible solid wastes, e.g. from precipitation processes of physical waste water treatment, from the foreground processes.
The sources of data for the calculation of some of the primary metal datasets were already referenced in the main text and are not discussed further here.


InflationMonkey, 2012. Copper Price is as Expensive as it was in the 1970s – Inflation Adjusted Historical Copper Price since 1900 in Pounds Sterling and US Dollars [WWW Document]. InflationMonkey. Available from http://www.inflationmonkey.blogspot.be/2012/05/copper-price-is-as-expensive-as-it-was.html [accessed 7 December 2013],


Pillot, C., 2013. The Rechargeable Battery Market and Main Trends 2012-2025. 30th International Battery Seminar & Exhibit. Fort Lauderdale, Florida, USA.


Rao, M.C., 2013. LiMn2O4 cathodes for solid state lithium-ion batteries - Energy storage and conversion. Journal of Optoelectronics and Biomedical Materials 5, 9–16.


Curriculum Vitae

Pilar Swart
Department of Sustainable Organic Chemistry and Technology
Faculty of Bioscience Engineering
Coupure Links 653
9000 Ghent, Belgium

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Education & Research

2009-2013  PhD research, Bioscience Engineering, Ghent University (Belgium)
PhD topic: Resource sustainability assessment of the use of non-ferrous metals in advanced materials applications
Promoters: Prof Jo Dewulf (University Ghent)
           Marleen Esprit (Umicore)
Funding body: Flemish agency for Innovation by Science and Technology (IWT).

2007-2009  MSc, Environmental Sanitation, Ghent University (Belgium)
Master thesis: Valorisation of industrial organic solvent waste
Promoters: Prof Jo Dewulf (University Ghent)
           Dr.ir.W. Aelterman (Janssen Pharmaceutica)

2004-2007  Bachelor studies, Chemical technology and material science, Ghent University (Belgium)

2001-2004  Sociology, Justus Liebig University Giessen (Germany)

International peer-reviewed publications


**Submitted papers**

Swart, P., Dewulf, J., Biernaux, A., 2013. Resource demand for the production of different cathode materials for lithium ion batteries. Submitted for publication to the Journal of Cleaner Production.

**Book chapters**


**Contributions to conferences**


**Teaching activities**

2009-2012 Ghent University (Belgium), Clean technology, theoretical exercises

2011-2013 Ghent University (Belgium), Environmental chemistry, theoretical exercises

2010 Ghent University (Belgium), Environmental chemistry, practical exercises
### Tutoring activities

<table>
<thead>
<tr>
<th>Year</th>
<th>Name</th>
<th>Thesis Title</th>
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<tbody>
<tr>
<td>2012-2013</td>
<td>Six, Lasse</td>
<td>Natural resource consumption analysis of collection, separation and recycling of end-of-life batteries in Belgium, master thesis.</td>
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<td>2011-2012</td>
<td>Biernaux, Alexis</td>
<td>Environmental sustainability assessment of Lithium-ion battery production, master thesis.</td>
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<td>2010-2011</td>
<td>De Keulenaere, Bram</td>
<td>Sustainability assessment of the production of amalgam spheres for the manufacture of fluorescent lamps, master thesis.</td>
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<td>2010</td>
<td>Tran, Phuong Ha</td>
<td>Environmental impact assessment of the Vietnamese surface finishing industry, master thesis.</td>
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