A multi criteria analysis and comparison of primary copper processing options

By

Dion Lusinga

Submitted in fulfilment for the award of the degree of
Master of Science in Chemical Engineering

March 2011

Declaration

I know the meaning of plagiarism and declare that all the work in this document, save for that which is properly acknowledged, is my own.

………………………………..　　　　　　　　　　　　………………………………..

Dion Lusinga

ABSTRACT

The global mining industry is currently under pressure from various stakeholders to implement all its activities, from exploration up to post closure, in a manner that is consistent with the principles of sustainable development. Essentially, this implies that economic, environmental and social concerns must be addressed simultaneously in all aspects and phases of mining and mineral processing. Historically, decision making in the mining industry has been largely influenced by economic objectives and it is not until recently that the mining industry has begun making efforts to address the environmental and social issues that are associated with their operations.

In the present study a methodology and a spreadsheet model were developed with the aim of integrating techno-economic and environmental objectives simultaneously during decision making. The model was developed specifically for the copper industry, mainly due to the wide array of different processing technologies and operational synergies that exist in this industry. It is because of this that integrating techno-economic and environmental concerns during process design and selection and even during the management of copper processing operations is important.

The methodology and model were developed using publicly available technical, logistical and economic information pertaining to copper production, and were applied in a case study which emulated a typical decision making scenario in the copper industry. The case study concerned a custom mine in northern Chile that is currently selling concentrates to custom smelters in China and is considering whether or not it should integrate vertically through its own smelter or POX plant next to the concentrator. The base case results showed that from purely economic perspectives, the continued sale of concentrates to custom smelters in China was the most viable option since this option had the highest Net Present Value (NPV). However, when the environmental impacts of the three processing options were compared, integrating vertically through a local smelter in Chile was the best option available to the mine.

Eco-efficiency indicators, which simultaneously take into consideration economic and environmental concerns, were also used to compare the performances of the various processing options. Similarly to the outcomes of the purely environmental assessment, this analysis showed that integrating vertically through a local smelter in Chile would be the most eco-efficient processing route. However, a sensitivity analysis on the geographical location of the mine showed that if the mine were located in central Chile, as opposed to northern Chile, then integrating vertically through a POX plant would be the most eco-efficient option.

A major limitation to the current study was the lack of detailed technical and economic data in the public domain, however, it is envisaged that industry's involvement in future studies could address this shortcoming.

ACKNOWLEDGEMENTS

I would like to express my deepest gratitude to my supervisors, Dr Jochen Petersen and Dr Jennifer Broadhurst, for their supervision, advice, and encouragement throughout this project. I am also grateful to Professor J-P Franzidis for the support and opportunities he gave me through the Minerals to Metals Initiative.

I would like to thank Mymoena van der Fort (Minerals to Metals) and Sue Jobson (CeBER) for their administrative assistance during the project.

I am also grateful to Grenvil Dunn (Hydromet (Pty) Ltd) and Dr Ray Shaw (Independent consultant) for the technical advice they gave me throughout the project. Allison Kasozi, Brian Chaponda (Lonmin) and Jacquez Bezuidenhout are also thanked for their advice on various aspects of this project.

To my God – Thank You. To my family and friends, thank you for your support and encouragement.

Page

LIST OF FIGURES

LIST OF TABLES

ABBREVIATIONS

AMD	Acid mine drainage
AP	Acidification potential
Cu	Copper
EBITDA	Earning before interest, taxes, depreciation and amortisation
EP	Eutrophication potential
ET	Eco-toxicity
EW	Electrowinning
GWP	Global warming potential
HFC	Hydrofluorocarbon
HT	Human toxicity
IRR	Internal rate of return
M	Molar mass
NMVOC	Non-methane volatile organic compounds
NPV	Net present value
OD	Ozone depletion
PBP	Payback period
PLS	Pregnant leach solution
POX	Total Pressure Oxidation
PP	Price participation
ROI	Return on investment
ROM	Run-of-mine ore
SX	Solvent extraction

1 INTRODUCTION

The world's resource-based industries are currently characterised by a need for minimal resource usage, production efficiency and the development of environmentally benign technologies and process routes. Historically, there has been a tendency in the minerals industry to give more attention to the economic aspects of a company's operations, with little or no consideration to the associated environmental and social aspects. In recent years, however, efforts to avert global climate change and the wide reporting of local environmental disasters have resulted in environmental regulatory bodies, non-governmental organisations (NGOs) and local communities increasingly holding industrial partakers, such as mining companies, accountable for their actions. Deterrents such as hefty fines, closure, imprisonment and sanctions have brought mining companies to the realisation that they have to carry out their operations in a manner that is consistent with the principles of sustainable development. The concept of sustainable development seeks to draw together the economic, social and environmental concerns of stakeholders (including future generations) during decision making and the implementation of human activities within the biosphere (WCED, 1987; United Nations, 2009). Companies refer to a balance between the 3 pillars (i.e. the social, economic and environmental aspects) of sustainability as the triple bottom line (Henriques, 2004; Savitz and Weber, 2006; Harrison, 2008). The concept of sustainable development is illustrated diagrammatically in Figure 1.1.

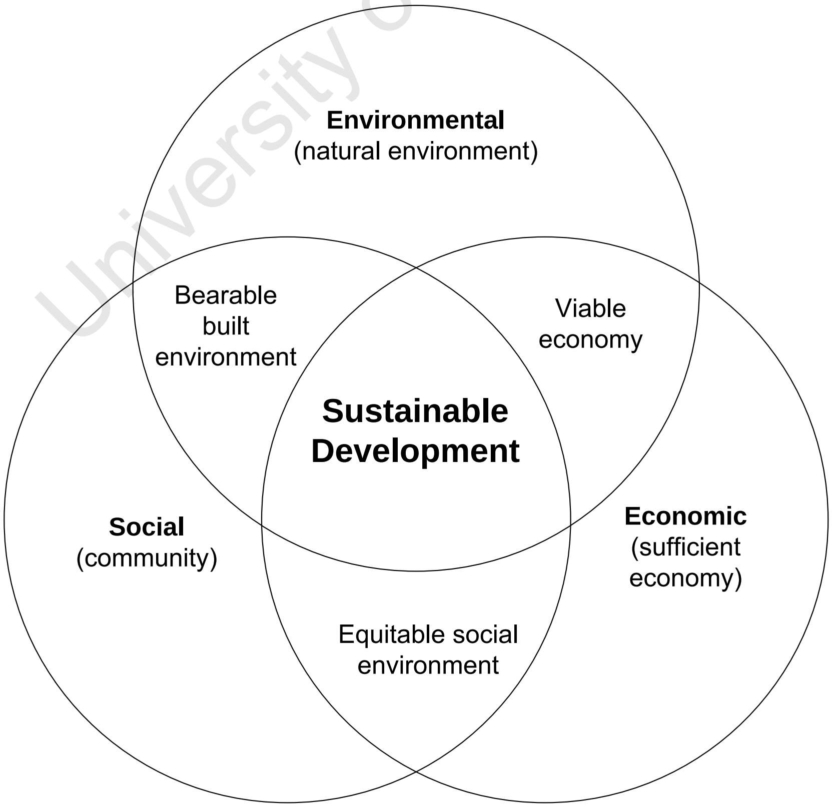


Figure 1.1: Sustainable development (adapted from Harrison, 2008)

Although some mining companies have made efforts to address some of the environmental and social challenges that are associated with their operations, there still remains a collective need in the industry to consider multiple inter-related objectives during the selection, design, operation, and management of minerals beneficiation operations. This is especially true for the early stages of process selection and design, which are typically characterised by data shortages and low certainty.

This thesis attempts to address some of these challenges through the development of a methodology for simultaneously assessing the economic and environmental performances of minerals beneficiation processes, with a focus on primary copper production.

1.1 Background and motivation

This section introduces the concept of environmentally conscious process selection and design, and provides a brief overview of previous studies pertaining to the techno-economic and environmental performances of primary copper processing options.

1.1.1 Environmentally conscious process selection and design

Process selection and design is an iterative process which consists of a number of steps as illustrated in Figure 1.2.

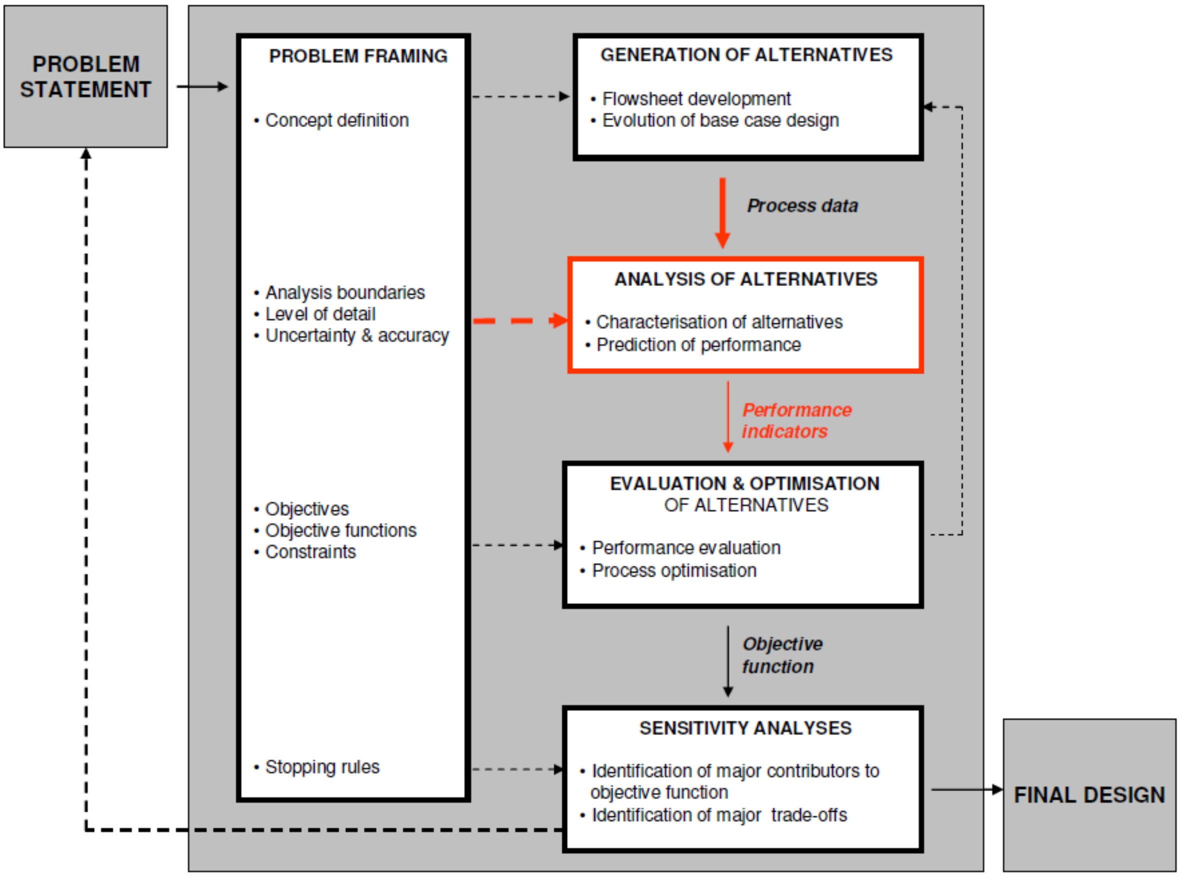


Figure 1.2: Traditional process design methodology (Guma, 2010)

The design process begins when the managing team of a company issues a project brief to a design team, for which the team must come up with a solution. The team frames the project by making critical decisions concerning the objectives, the scope, the evaluation criteria and the stopping rules of the design process (Cano-Ruiz and McRae, 1998). Once the problem has been framed, alternative flowsheets are generated and these are analysed through the development of mass and energy balances as well as equipment sizing (Cano-Ruiz and McRae, 1998; Guma 2010). The data generated during the analysis step is aggregated into suitable indicators and metrics that are used by stakeholders to evaluate the performance of the alternative processes with reference to the desired project objectives and outcomes (Cano-Ruiz and McRae, 1998). Traditionally, indicators such as operating costs, capital costs, net present value (NPV), internal rate of return (IRR), payback period (PBP), and return on investment (ROI) have been used.

The methodology illustrated in Figure 1.2 is applied successively in the various stages of process design, which are detailed in Table 1.1.

Table 1.1: Process design stages and accuracy of cost estimates (Cano-Ruiz and McRae, 1998; Douglas, 1988)

Stage	**Number of alternatives**	**Accuracy of capital cost estimates**
Order of magnitude estimate *(Conceptual)*	Infinite (theoretically)	Based on similar previous cost data; ± 40% accuracy
Study estimate *(Pre-feasibility)*	Infinite (theoretically)	Based on knowledge of major items of equipment; ± 25% accuracy
Preliminary estimate *(Feasibility)*	~ 7 – 15	Based on sufficient data for budget authorisation; ± 12% accuracy
Definitive estimate *(Preliminary)*	~ 3 – 5	Based on almost complete data but before completion of drawings; ± 6% accuracy
Detailed estimate *(Final Design)*	~ 1(for optimisation)	Based on complete eng drawings, specs and site surveys; ± 3% accuracy

Each stage in process design has unique information requirements and degrees of accuracy. The conceptual stage of process selection and design typically requires the least amount of information and produces capital cost estimates that are accurate to within ± 40%. At this stage of process design, there exists an infinite number of alternative flowsheets that the design engineers can generate and evaluate; however, through their experience they can reduce the initial number of flowsheets that they generate (Douglas, 1988; Cano-Ruiz and McRae, 1998). As more alternatives are rejected during the subsequent stages, the number of remaining alternatives decreases, and the quality and quantity of information required along each stage increases. Eventually the design team remains with one design, which, after optimisation, is likely to be used by the interested company.

Traditionally, emphasis has been given to the economic objectives and outcomes of projects and according to Cano-Ruiz and McRae's (1998) review of process design textbooks, little or no attention was given to the environmental objectives of processes. For example, with reference to environmental concerns, Douglas (1988) states, *"Normally, we want to find the process alternative (out of the* 10^4 *to* 10^9 *possibilities) that has the lowest cost, but we must also ensure that the process is safe, will satisfy environmental constraints, is easy to start up and operate etc."* No guidance or methodologies were given in the text on how to design environmentally benign or sustainable processes, and consequently on how decision makers should make tradeoffs between different processes on the basis of techno-economic, environmental and social criteria.

Broadhurst (2007) illustrated diagrammatically how and when environmental and social concerns should be incorporated into the design process (Figure 1.3).

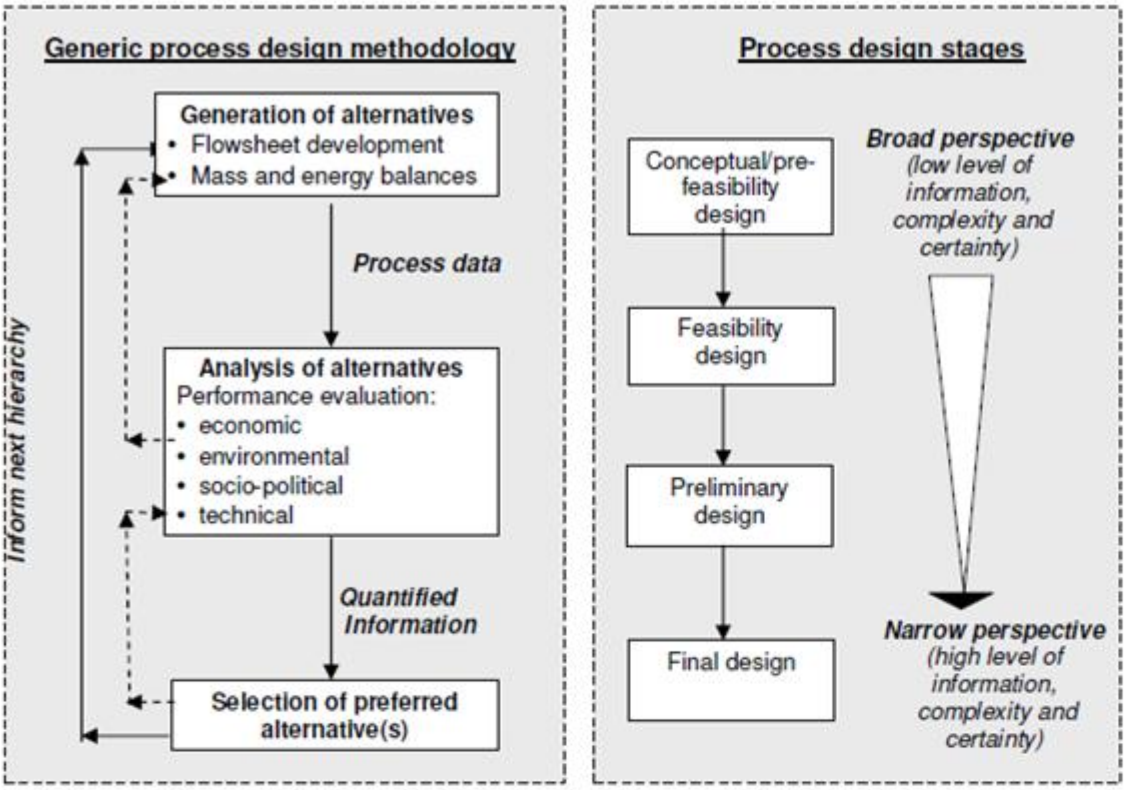


Figure 1.3: Design procedure and information and data requirements (Broadhurst, 2007)

Figure 1.3 shows that the environmental and social objectives of a project should be incorporated into the design methodology from the beginning, and should be used concurrently with economic criteria in evaluating all of the flowsheets that are generated, unlike the traditional design methodology which prioritises economic objectives and considers social and environmental objectives as constraints.

The goals of environmentally conscious design are to produce designs and manage operations that have (Cano-Ruiz and McRae, 1998; Broadhurst, 2010):

- high economic potential
- high conversion of raw materials into product
- efficient resource utilisation
- minimal release of hazardous substances to the environment.

1.1.2 Performance analysis of primary copper processing alternatives

Commercial copper can be produced through a variety of process routes within the fields of pyrometallurgy, hydrometallurgy, and biohydrometallurgy (Chapter 2); and as a result of the large number of process alternatives available, the need for multi-criteria analyses is particularly relevant to the copper sector. This is evidenced by the numerous studies that have been performed to address and compare the environmental impacts of copper production. A summary of notable studies done to this effect is given in Table 1.2.

Table 1.2: A brief overview of previous studies pertaining to the environmental performance of primary copper processing options

Study	Scope / Approach	Limitations
Giurco et al. (2001)	A life cycle assessment of flash, reverberatory and electric smelting, as well as heap leaching	No economic assessment Excluded recently developed alternatives to smelting No consideration of the concentrate supply chain
Norgate (2001)	A life cycle assessment of concentrate treatment processes. Includes Flash smelting and hydrometallurgical processes such as the Intec, POX, BioCOP and Activox processes	No economic assessment Concentrate supply chain considered, but only through the effect of transportation, however country specific analysis not performed

Table 1.2 continued …

Study	Scope / Approach	Limitations
Ayres et al. (2002)	A general report on copper from a life cycle perspective. Discusses issues related to copper mining, processing, wastes, disposal and recycling	No economic assessment Excluded recently developed alternatives to smelting No consideration of the concentrate supply chain
Alvarado et al.(2002)	Investigates the effect of mitigation strategies on Chile's future greenhouse gas and SO_2 emissions	No economic assessment Excluded recently developed alternatives to smelting No consideration of the concentrate supply chain
Kuckshinrichs et al. (2007)	Investigates the effects of recycling copper as well as the effects of technical progress in the industry on the CO_2 and SO_2 emissions related to the world's copper production	No economic assessment Excluded recently commercialised alternatives to smelting No consideration of the concentrate supply chain
Guma (2010)	Focuses on the development of eco-efficiency indicators for the minerals industry	Economic assessment included Excluded recently developed alternatives to smelting No consideration of the concentrate supply chain
Pimentel (2009)	Compares the energy consumption and global warming potential of smelting and leaching operations that receive power from two power grids (SING and SIC) in Chile	No economic assessment Excluded recently developed alternatives to smelting No consideration of the concentrate supply chain

In spite of the valuable effort made to date, one or more of the following gaps were identified in each of the studies in Table 1.2:

- The economic performances of the various available process routes were not considered.
- No connections were made between ore mineralogy and the process routes used to produce the copper.
- With the exception of Norgate (2001), none of the studies considered the concentrate supply chain and its marketing, implying that:
    - The effect of concentrate transportation on the environment was not considered.
    - The impact of the difference in the electricity generation mixes of the copper concentrate mining and treatment countries was neglected.
- The studies tended to be generic; site or geography specific factors and the benefits of possible tradeoffs related to these were not integrated into the studies.
- Many of the studies focused on global warming potential and energy consumption only.
- Typical decision-making scenarios related to copper production were not included.

A summary of key economic assessments performed to date relating to copper production is given in Table 1.3.

Table 1.3: A brief overview of previous studies pertaining to the economic performance of primary copper processing options

Study	**Scope**	**Limitations**
Davenport et al. (2002)	Gives a breakdown of the capital and operating costs of smelting and heap leaching processes	No environmental assessment No consideration of the concentrate supply chain
Peacey et al. (2003)	Shows typical capital and operating costs of heap leaching, smelting, and the Mount Gordon process	No environmental assessment No consideration of the concentrate supply chain
Barr et al. (2005)	Compares the economic performance of smelting and the CESL copper concentrate treatment process	No environmental assessment No consideration of the concentrate supply chain
Robles (2009)	Compares the operating costs of smelting and the new hydrometallurgical alternatives to smelting	No environmental assessment No consideration of the concentrate supply chain

From the economic studies the following gaps were identified:

- With the exception of Barr et al. (2005) and Guma (2010), the studies did not emulate typical decision-making scenarios in the copper industry. Therefore, the studies tended to be generic and did not include profitability indicators such as net present value (NPV), payback period (PBP) and return on investment (ROI).
- The environmental aspects of copper production were not analysed in tandem with the economical aspects.
- Where smelting was considered in previous studies, only one configuration was considered, that is, either the concentrates were treated near the mine by the company's own smelter (vertical integration through an internal smelter) or they were sent or sold to external refineries (custom smelting). No comparisons were made between the two options, i.e. internal smelting vs. custom smelting.

The review of previous key techno-economic and environmental studies on copper processing shows that the studies were largely performed in isolation, and that there is a need for holistic simultaneous techno-economic and environmental comparisons to guide decision-making in the sector towards more sustainable options.

1.2 Problem Statement

Due to the pressures on industry to operate more sustainably, there is a need for simultaneous techno-economic and environmental analyses in the design and selection of process alternatives, as well as in the day-to-day operation and management of minerals beneficiation operations. A review of recent literature (Table 1.2 - Table 1.3 and Chapter 2) concerning process selection and operations in the copper industry has identified shortcomings in this regard. Therefore, there is a need for research that demonstrates how techno-economic and environmental criteria can be used simultaneously in the decision making processes of the copper industry.

1.3 Research Objectives

The overarching objectives of this thesis are to:

1. Develop a methodology for generating and assessing multiple flowsheet options that take into account techno-economic and environmental criteria, using data that is consistent with early stage, design requirements in the copper processing industry.
2. Demonstrate the application of this methodology for the case of primary copper production.

In particular, this study aims to:

1. Identify the key parameters and criteria of significant relevance to the techno-economic and environmental performances of primary copper processes.
2. Build a data base of available input and output inventory data.
3. Develop a model to generate information pertaining to the environmental and techno-economic performance of various process options.
4. Apply this model in a case study which emulates typical decision-making scenarios in the copper industry.

1.4 Thesis structure

The background, problem statement and research objectives of the thesis are presented in Chapter 1. Chapter 2 reviews the pertinent literature related to the techno-economic and environmental aspects of copper production, with the aims of identifying key criteria related to the performance of copper production processes, and collating the relevant available data. Chapter 3 describes the methodology and performance model that were developed to address the objectives of this study, and in Chapter 4 the performance model is applied in the context of a hypothetical case study that emulates typical operational and decision making scenarios in the copper industry. The thesis closes with Chapter 5, in which conclusions are drawn from the study, and recommendations for future work are made, to build on the analyses performed in this study.

1.5 Disclaimer

The core data used in this thesis was sourced and collated from literature & media sources available in the public domain except were expressly indicated otherwise. The results of the case study in this thesis do not reflect the performance of any particular business or organisation. The author does not accept liability for any undesirable outcomes resulting from the use of data presented in this thesis.

2 LITERATURE REVIEW

Copper is second in terms of global production of the non-ferrous metals in the world, after aluminium (Giuirco et al., 2001, Coulas, 2005). In 2008, 34 million metric tonnes of aluminium (NRC, 2009) and 15 million metric tonnes of copper were produced worldwide (ICSG, 2009). Due to its lustre, corrosion resistance, malleability, ductility, and high conductivity of electricity and heat, copper is widely used in the electrical, electronics, telecommunications, construction, transportation, manufacturing and consumer product industries (ICSG, 2009). To understand the production of copper, section 2.1 gives a background of the nature of copper deposits and their ores, and section 2.2 discusses the various process routes used in industry to extract copper from its ores.

2.1 The nature of copper deposits and ores

Igneous, sedimentary and metamorphic rocks are the three main rock groups of the Earth's crust. Igneous rocks are formed through the solidification of magma (molten rocks) originating from the Earth's crust and sedimentary rocks are formed through the surface, physical and chemical weathering of the original magmatic deposits (Press et al., 2004; Broadhurst, 2007). Metamorphic rocks are formed when conditions of high temperature and pressure originating from deep within the earth crust change the physical and chemical nature of igneous, sedimentary and other metamorphic rocks (Press et al., 2004). Mineral deposits can be found in these three main rock groups as well as in other sub classes of the three groups that develop over time due to a variety of geological processes, as described by Press et al. (2004) and Broadhurst (2007). Table 2.1 shows a summary of the mineralogy of the world's copper sulphide deposits. The evolution of the original copper sulphide minerals to copper oxides in these deposits is described later in the text.

Table 2.1: Selected minerals of environmental and economic significance in copper sulphide ore bodies source - data compiled by Broadhurst (2007)

Deposit class	**Porphyry**	**Skarns**	**Polymetalic vein replacement**	**Sedimentary**	**Massive volcanogenic**	**Ultrabasic magmatic**	**Carbonatite magmatic**
Contribution to world's copper production	50%	N/A[1]	7%	24%	7%	4%	N/A
Main ore minerals containing copper	Bornite Chalcopyrite Chalcocite Covellite	Bornite Chalcopyrite Enargite	Bornite Chalcopyrite	Chalcopyrite Chalcocite	Chalcopyrite Chalcocite Covellite	Chalcopyrite	Chalcopyrite
Associated minerals	Galena Sphalerite Molybdenite Pyrite	Arsenopyrite Pyrite Pyrrhotite Sulphosalts	Pyrite Bismutite Stibnite Sphalerite Arsenopyrite	Pyrite Bornite Galena Sphalerite	Pyrite Pyrrhotite Sphalerite Arsenopyrite	Cobaltite Pyrrhotite Arsenides Antimonides PGMs	Pyrite Pyrrhotite Sphalerite Columbite Monazite
Gangue minerals	Quartz Alunite Clays Chlorite Calcite	Quartz Chlorite Amphibole Wollasinite	Quartz Chlorite Clays Muscovite	Dolomite Shales Calscillicates	Quartz Muscovite Chlorite Albite	N/A	Calcite, Dolomite Biotite Ilmenite Ankerite
Co-elements to copper	Ag, Au, As, Bi, Sn, Mo, Zn, Pb, Te	Ag, Au, As, Mo, Sn, Se, Zn, Sb, Bi,	Ag, Au, As, Mo, Sn, Pb, Zn, Sb, Bi	Ag, As, Au, Co, Hg, Zn, Mo, Pb	Al, As, Ag, Ba, Sb, Zn, W, Co	Ni, Co, Mg, Pt, Ni	B, Ba, Pb, P, Zn, Zr, V

The bulk of the world's copper is mined from porphyry copper deposits followed by sedimentary, vein replacement and volcanogenic deposits, in descending order. No data was available concerning the percentage contribution of skarn and carbonatite magmatic deposits to the world's copper production. Porphyry copper deposits are large, low grade, disseminated ore bodies that are found in igneous rocks (Press et al., 2004; Broadhurst, 2007). Vein deposits are intrusive[2] mineral deposits that are found in rock fractures and joints and are formed when mobile hot solutions, such as heated groundwater, cool and precipitate minerals in the rock fractures (Press et al., 2004). Skarns are intrusive deposits

---

[1] No data was available concerning the percentage contribution of skarn and carbonatite magmatic deposits to the world's copper production.

[2] Intrusive – formed underneath the earth's surface.

that are hosted by carbonate rocks such as limestone, and are typically found on the margins of intrusions that create porphyry deposits (Broadhurst, 2007; Mining Basics, 2010 a). Magmatic deposits are sulphide deposits that were created through gravitational segregation and magmatic differentiation[3] of molten magma (Press et al., 2004). Massive volcanogenic sulphides are extrusive deposits that are created when materials from erupting volcanoes cool on the earth's surface (Press et al., 2004).

The copper minerals in these deposits are normally associated with other minerals such as pyrite (FeS_2)[4], arsenopyrite (FeAsS), galena (PbS), sphalerite (ZnS), molybdenite (MoS_2), and small quantities of precious metals such as gold (Au) and silver (Ag). Gold, silver and molybdenum are common by-products of copper mines. Common gangue minerals in the copper deposits are silicates such as quartz (SiO_2), muscovite $H_2KAl_3(SiO_4)_3$ and chlorite $H_4(Mg,Fe)_3Si_2O_9$, as well as carbonates such as calcite ($CaCO_3$) and siderite ($FeCO_3$) (Jansen and Taylor, 2003; Broadhurst, 2007).

In the copper deposits (Table 2.1), copper minerals are normally classified according to the section of the deposit wherein they are found, as oxide, secondary sulphide (supergene) and primary sulphide (hypogene) minerals. The typical structure of a copper deposit is shown in Figure 2.1.

---

[3] Magmatic differentiation occurs as a result of different minerals crystallising at different temperatures.

[4] A major source of Acid Mine Drainage (Akcil and Kolders, 2006; Maluleke, 2006; Hesketh 2010).

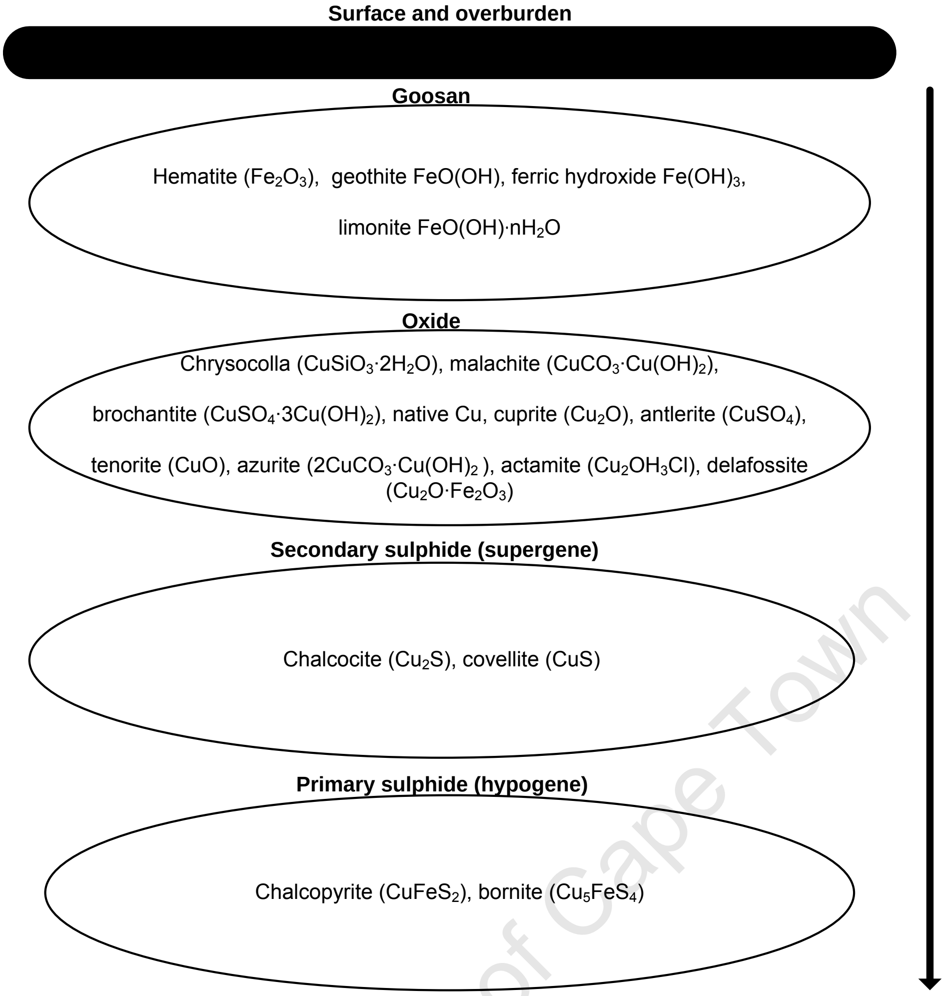


Figure 2.1: The typical structure of a copper ore body (adapted from Davenport et al., 2002; CDAA, 2009)

Originally the minerals are deposited as primary sulphides, of which copper typically exists as chalcopyrite ($CuFeS_2$) and bornite (Cu_5FeS_4)[5], iron as pyrite (FeS_2), lead as galena (PbS) and zinc as sphalerite (ZnS) (Takeuchi and Nambu, 1952; Mining Basics, 2010 b). However, because of exposure to surface water and oxygen the original mineralisation of the primary sulphide zone is chemically altered, resulting in the progressive formation of the goosan, oxide and secondary sulphide layers as shown in Figure 2.1. The formation of the oxide and secondary sulphide layers from the primary sulphide layer is briefly explained as follows (Takeuchi and Nambu, 1952; Farooq, 2010; Mining Basics, 2010 b):

- Original pyrite (FeS_2) near the top of the ore body is oxidised to produce sulphuric acid (H_2SO_4) and ferric ions Fe^{3+} which are oxidising.
- The acid and catalytic ferric ions leach the original primary sulphide minerals in the ore body generating copper sulphate ($CuSO_4$) and more acid and ferric ions. This

---

[5] Bornite (Cu_5FeS_4) may exist as a primary sulphide mineral or it may be formed as a secondary sulphide mineral through secondary enrichment (Takeuchi and Nambu, 1952).

process creates a region in the deposit which is lean in copper and contains precipitated iron in the form of ferric hydroxide $Fe(OH)_3$, hematite (Fe_2O_3), goethite ($FeO(OH)$) and limonite ($FeO(OH)\cdot nH_2O$). This region of the deposit is known as the goosan[6] and is typically of low economic value.

- As the solution, containing copper sulphate ($CuSO_4$) trickles down, some of its copper ions precipitate, resulting in the formation of the oxide layer as shown in Figure 2.1. This layer contains copper oxide minerals such as chrysocolla ($CuSiO_3\cdot 2H_2O$) and malachite ($CuCO_3\cdot Cu(OH_2)$). Malachite is more likely to be formed when the host rock contains carbonate minerals.
- When the solution reaches the water table, the copper sulphate remaining in the solution reacts with the unaltered iron sulphide minerals such as chalcopyrite ($CuFeS_2$) in the primary zone, replacing Fe ions in these minerals with copper. This process is known as secondary enrichment and creates the secondary sulphide zone which contains minerals such as chalcocite (Cu_2S) and covellite (CuS).
- The minerals in the primary sulphide zone ($CuFeS_2$ and Cu_5FeS_4), beneath the enriched secondary sulphide zone remain unaltered.

According to Farooq (2010), secondary enrichment can affect most classes of mineral deposits, however favourable conditions of climate and physiography have to exist for the process to occur.

Grades of copper ore bodies

It was shown in Table 2.1 that the bulk of the world's copper is currently mined from porphyry copper deposits which are commonly found in North and South America, the largest copper mining regions of the world (BCS Incorporated, 2002; CDAA, 2009; Napier-Munn et al., 1999). In 2008, of the 15 million tonnes of copper that were produced globally, Chile and the United States of America contributed 5.3 and 1.6 million tonnes, respectively (ICSG, 2009).

An analysis performed on data compiled for the US Geological Survey (Singer et al., 2002), of the mineral content of the world's porphyry copper deposits showed that the deposits have an average copper grade of approximately 0.5% and a maximum copper grade of 2%. The maximum grade (2% copper content) was found in a relatively small deposit that is located in Papua New Guinea which contained approximately 60 million tonnes of ore. The largest porphyry copper deposit was found in Chile and contained approximately 17 000 million tonnes of ore with an average grade of 0.65% copper. A survey pertaining to the characteristics of other deposit classes was not readily available and a detailed study to obtain such data is beyond the scope of this project.

---

[6] There is no clear distinction between the goosan and oxide layer. Takeuchi and Nambu (1952) refer to the regions containing the iron and copper oxides, collectively, as the oxidised zone.

2.2 Copper beneficiation processes and technologies

Commercial copper is produced through a variety of process routes that are normally selected based on the type of ore (i.e. oxide, primary and secondary sulphide) and ore grade that the company is mining. This is illustrated diagrammatically in Figure 2.2 which summarises the main process routes used for primary copper production. The diagram also indicates the unit operations that are reviewed in this section (the boxes).

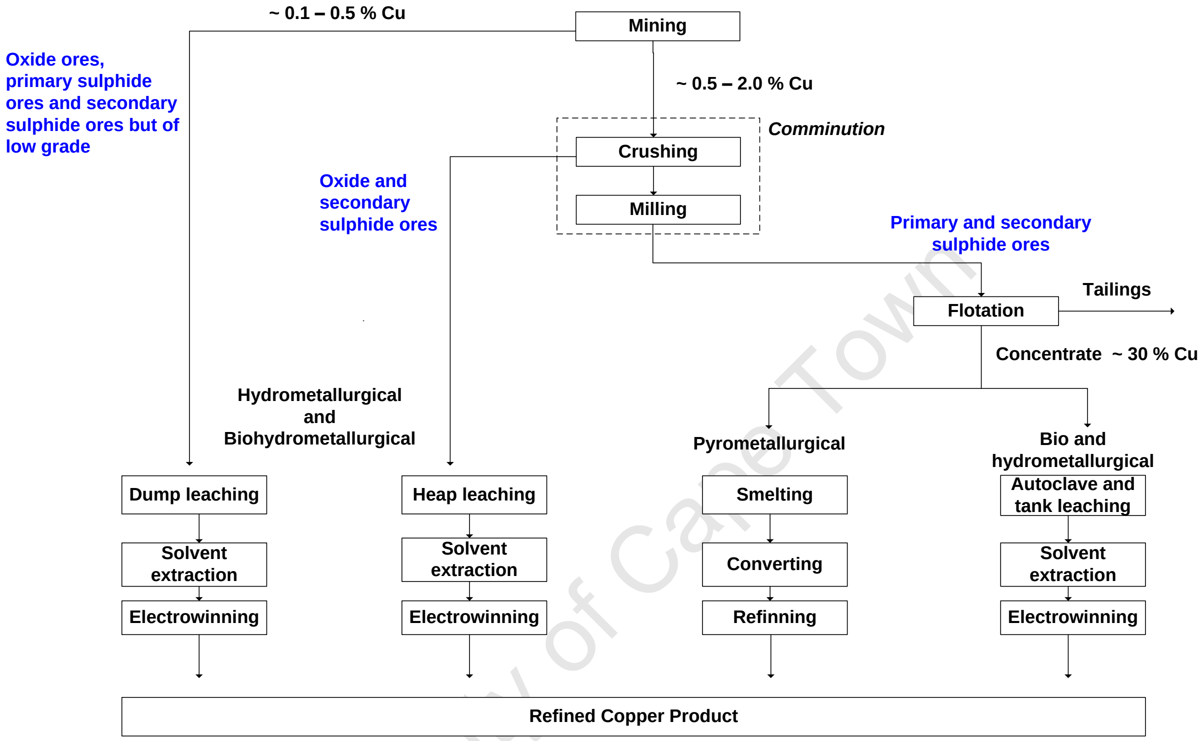


Figure 2.2: A summary of the main processing routes of extracting copper from its ores (adapted from Biswas and Davenport, 1994; Davenport et al., 2002; Broadhurst, 2009)

In 2008, the hydrometallurgical and bio-hydrometallurgical process routes accounted for 17% of the worlds copper production (ICSG, 2009), and it is assumed that the bulk of this production was attributed to ore heap and dump (run-of-mine) leaching, whereas the concentrate leaching processes contributed a negligible amount to this figure since they are not yet widely used. The pyrometallurgical process route which involves smelting and converting copper concentrates and scrap accounted for 83% of the world copper production of which 15% was attributed to copper scrap recycling and 68% to concentrate smelting (ICSG, 2009). The production of copper through each process route outlined in Figure 2.2 is discussed in more detail in the following sections of the literature review.

2.2.1 Mining

Mines are classified in broad terms as either surface or underground mines. Surface mines are operated from the surface or involve operations that are performed in excavations that are completely open (Hartman and Mutmansky, 2002). Examples of surface mining methods are open pit mining, quarrying and in situ leaching. Underground mining involves the creation of holes underground to enable humans to gain access to the ores (Hartman and Mutmansky, 2002). Examples of underground mining methods include room and pillar mining, stope and pillar mining, block cave mining and cut and fill stoping (Hartman and Mutmansky, 2002), each of which are discussed in more detail in the mining engineering texts by Hartman (1992), and Hartman and Mutmansky (2002). In this thesis it is of importance to note that each mining method has its unique costs and environmental impacts, and data relating to a specific mining method should be collected prior to its evaluation in a techno economic and environmental evaluation. In general, surface mining is the main mining method employed for extracting ores in the world (Hartman and Mutmansky, 2002). It is normally used when large, low grade ore bodies that are near the surface are being extracted, whereas underground mining is practised when deep, high grade ore bodies are being extracted (USCOTA, 1988). Of the surface mining techniques, open pit mining is the leading mining method, and because of this, it will be used in the model developed in this study and therefore warrants further discussion.

Figure 2.3 conceptually illustrates the production cycle of open pit mining.

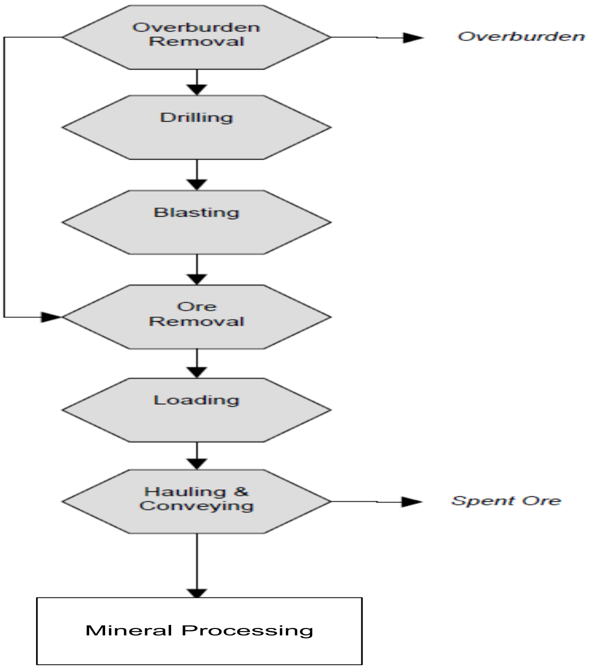


Figure 2.3: Open pit mining flow diagram (adapted from BCS Incorporated, 2002)

Typically, open pit mining is a cyclic process which initially involves drilling holes into the overburden layer so that explosives can be inserted. Thereafter, the explosives are detonated and the overburden and ore are reduced in size to rock pieces that can be loaded by shovels into haulage trucks or trains which transport the ore to the crushers and the waste to dumps (Hartman and Mutmansky, 2002). In non-cyclic mines that have continuous[7] operations, a single machine known as an excavator can replace the drilling and blasting stages by extracting and loading the ores in a single step known as excavation. The typical costs and energy consumption data associated with open pit mining are discussed in sections 2.3 and 2.4 respectively. Section 2.2.2 discusses comminution, which involves reducing the size of the rocks that are extracted during the mining stage.

2.2.2 Comminution: Crushing and grinding

Comminution involves the progressive size reduction of an ore until its valuable minerals are liberated from the gangue minerals (Wills et al., 2006).

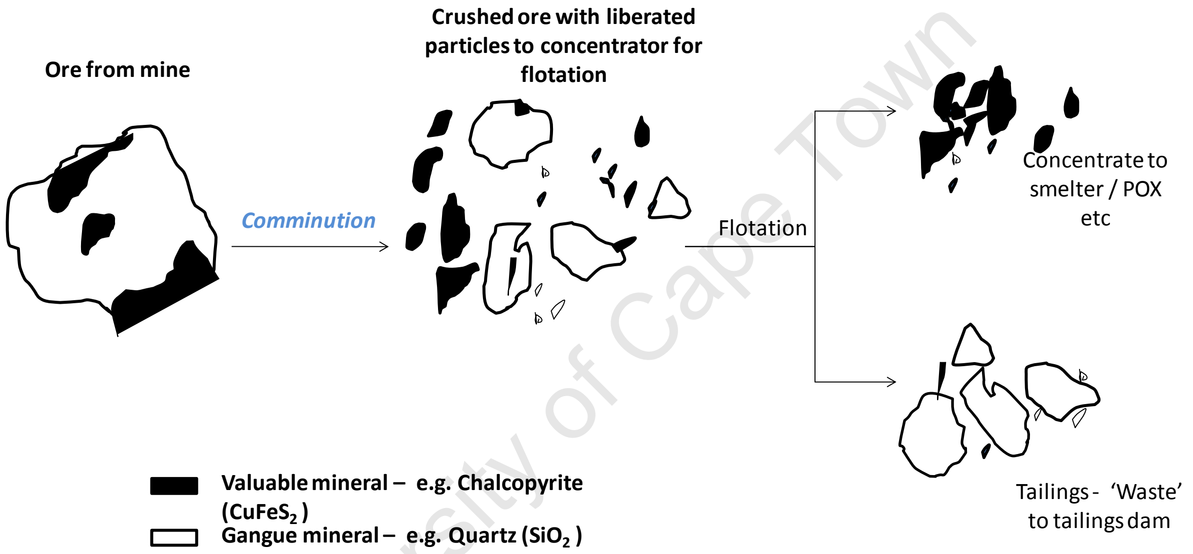


Figure 2.4: Illustration of an ore's path through comminution and flotation

Separating the valuable and gangue minerals during comminution permits the extraction of the valuable minerals through flotation or gravity concentration, prior to their conversion in downstream processing facilities (e.g. a smelter) to commercial products such as copper cathodes and rods. Blasting can be considered as the first stage of comminution (Napier-Munn et al., 1999; Herbst et al., 2003; Wills and Napier-Munn, 2006), however on mineral processing sites comminution normally refers to a series of crushing and grinding stages that utilise various combinations of the equipment listed in Table 2.2.

---

[7] Applied when soft ores are being mined.

Table 2.2: The main comminution equipment

Crushers type	Typical ore feed size (mm)	Grinding Mill type	Typical ore feed size (mm)
Jaw	1600	Autogenous	200
Gyratory	1829	Semi autogenous	200
Roll	1000	Rod	10
Impact	1000	Ball Mills	10
Cone	378		
HPGR	75		

Jaw, gyratory, roll and impact crushers are types of primary crushers. They are used in comminution circuits to reduce the size of run-of-mine rocks (ore from the mine) that can be as large as 1500 mm (1.5 m) to 100 – 200 mm (0.1 – 0.2 m). The primary crusher product is subsequently fed to secondary crushers and sometimes tertiary crushers for further size reduction. Cone crushers can be used for secondary and tertiary crushing. An alternative to using secondary and tertiary crushing is installing an autogenous (AG) or semi autogenous (SAG) mill, which typically eliminate the need for further crushers after the primary crusher. The product from the tertiary crushers or the AG / SAG mills is typically fed to ball mills, which reduce the size of the ore particles to approximately 100 μm which is approximately the optimum economic size typically required for copper flotation (Davenport et al., 2002). The crushers are operated in dry conditions, whereas the mills are operated in a wet environment. Illustrations of how these circuits are set up in industry are shown in Figure 2.5, Figure 3.5 and Figure 3.6. The selection of a specific comminution circuit depends on a number of factors which include ore characteristics (e.g. hardness, abrasiveness, mineralisation etc), plant throughput, size of the feed and products, and the response of the down stream processes[8] (Major, 2002). For the purposes of this project it was important to note that the different comminution circuits that can be employed in the production of copper have different capital costs, operating costs, energy consumptions and hence environmental profiles. Therefore, inventory data related to a specific comminution circuit has to be compiled prior to its use in a techno-economic and environmental analysis.

---

[8] See Napier-Munn et al. (1999), Barratt & Sherman (2002), Callow & Moon (2002), Major (2002), O'Bryan (2002), Utley (2002), Herbst et al. (2003), and Wills and Napier-Munn (2006) for more detailed discussions on the factors that affect comminution circuit selection.

2.2.3 Flotation

Flotation is a physico-chemical process that is used to separate valuable minerals from gangue minerals (Wills and Napier-Munn, 2006). Flotation involves simultaneously feeding the pulp from the comminution circuit, chemical reagents and air bubbles into a mechanically agitated cell. Agitation, through an impeller, prevents the pulp from settling in the flotation cells, improves the dispersion of air bubbles, and promotes particle-bubble collisions, through which sulphide minerals attach to the rising bubbles (Erepan, 2004). The rising bubbles and the attached minerals overflow from the flotation cells in a 3-phase froth (solids, liquids, air) which is collected as a concentrate that is relatively richer in valuable minerals compared to the original feed material. The remaining portion of the feed, which is lean in valuable minerals and contains most of the original gangue material, is known as the tailings stream and is discarded in a tailings dam.

For the case of copper production, the concentrate will mainly consist of copper sulphide minerals such as chalcopyrite ($CuFeS_2$), chalcocite (Cu_2S) and covellite (CuS) (Sanchez-Corrales et al., 2004). Flotation is not a perfect process, therefore gangue minerals such as pyrite (FeS_2 – sulphide gangue) and quartz (SiO_2 – non sulphide gangue) can be found in the concentrate and likewise some copper sulphide minerals are lost in the tailings. The gangue minerals enter the concentrate with some of the valuable minerals that were not completely liberated in the comminution circuit, and through physical entrapment in the valuable minerals that are in the concentrate froth, as well as through entrainment in the water that overflows with the froth (Wills and Napier-Munn, 2006). The reader is referred to Lynch et al. (1981) for detailed graphical illustrations of the separation efficiency and behaviour of chalcopyrite, pyrite and non-sulphide gangue minerals in copper flotation circuits.

Figure 2.5 illustrates the set-up of a conventional copper concentrator which includes a comminution circuit and a conventional flotation circuit.

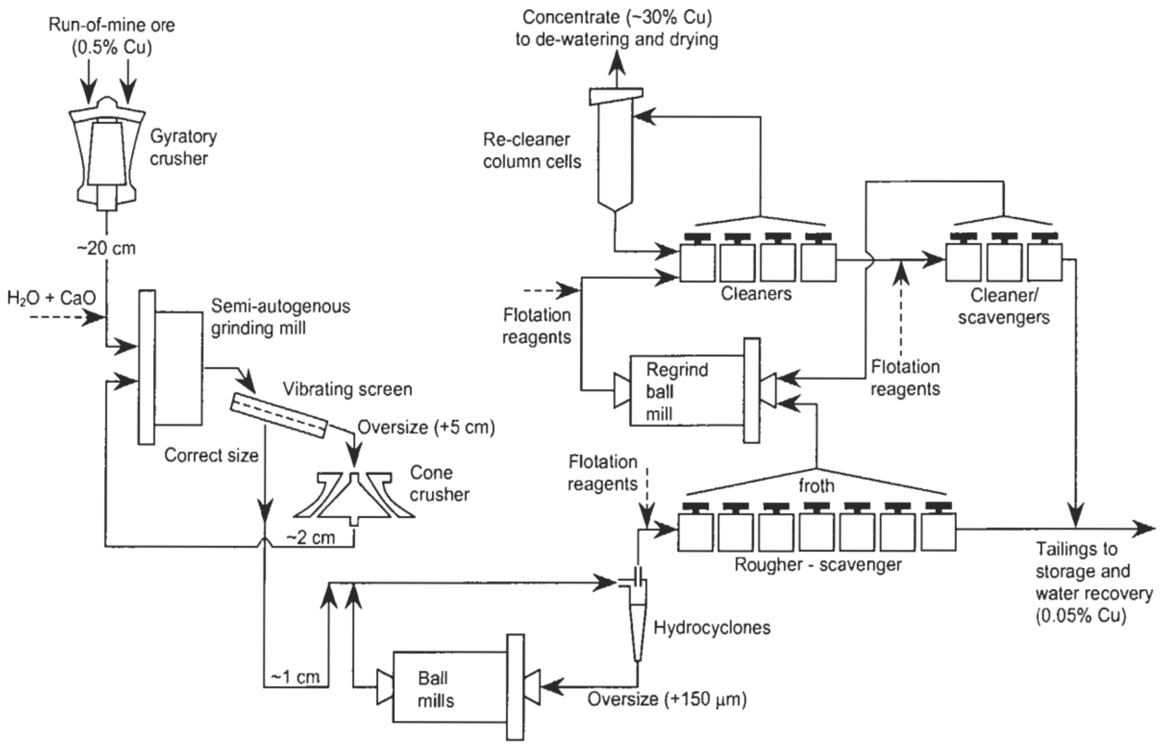


Figure 2.5: Set up of a typical copper concentrator (Davenport et al., 2002)

There are different circuit configurations that can be utilised; however the conventional flotation circuits normally consist of four banks of cells which function as follows (Davenport et al., 2002):

Rougher-Scavengers – Designed to maximise the recovery of valuable minerals from the feed. The valuable product from this bank of flotation cells is a copper concentrate froth containing approximately 15 – 20% Cu. The desired copper grade for a copper concentrate is approximately 30%, therefore the concentrate produced in these cells (rougher-scavengers) is sent to the cleaners for further processing.

Cleaners – Designed to increase the grade of the concentrates from the rougher-scavengers and the cleaner scavengers by depressing the non-copper minerals.

Cleaner scavengers – Designed to recover copper from the cleaner tailings through further depression of Fe and other gangue minerals.

Re-cleaners – Designed to increase the grade of the concentrate from the cleaners. This is where the final concentrate is produced. It typically contains, on a dry mass basis, about 30% copper, 30% sulphur, 30% iron and 10 % gangue.

The concentrate from the flotation cells contains moisture and is dewatered by use of thickeners and filters, to produce a final concentrate that typically has a moisture content of approximately 8% (wet basis). The concentrate can be treated via smelting or through the biohydrometallurgical and hydrometallurgical processes as described in sections 2.2.4 and 2.2.5 below.

2.2.4 Pyrometallurgical extraction

The bulk of the world's primary copper is produced through the pyrometallurgical process route which involves smelting copper concentrates. In the copper industry 'smelting' is the generic term that embodies two sub-processes, namely matte smelting and converting. The main material flows in an integrated smelter and refinery are illustrated in a simplified block flow diagram (Figure 2.6).

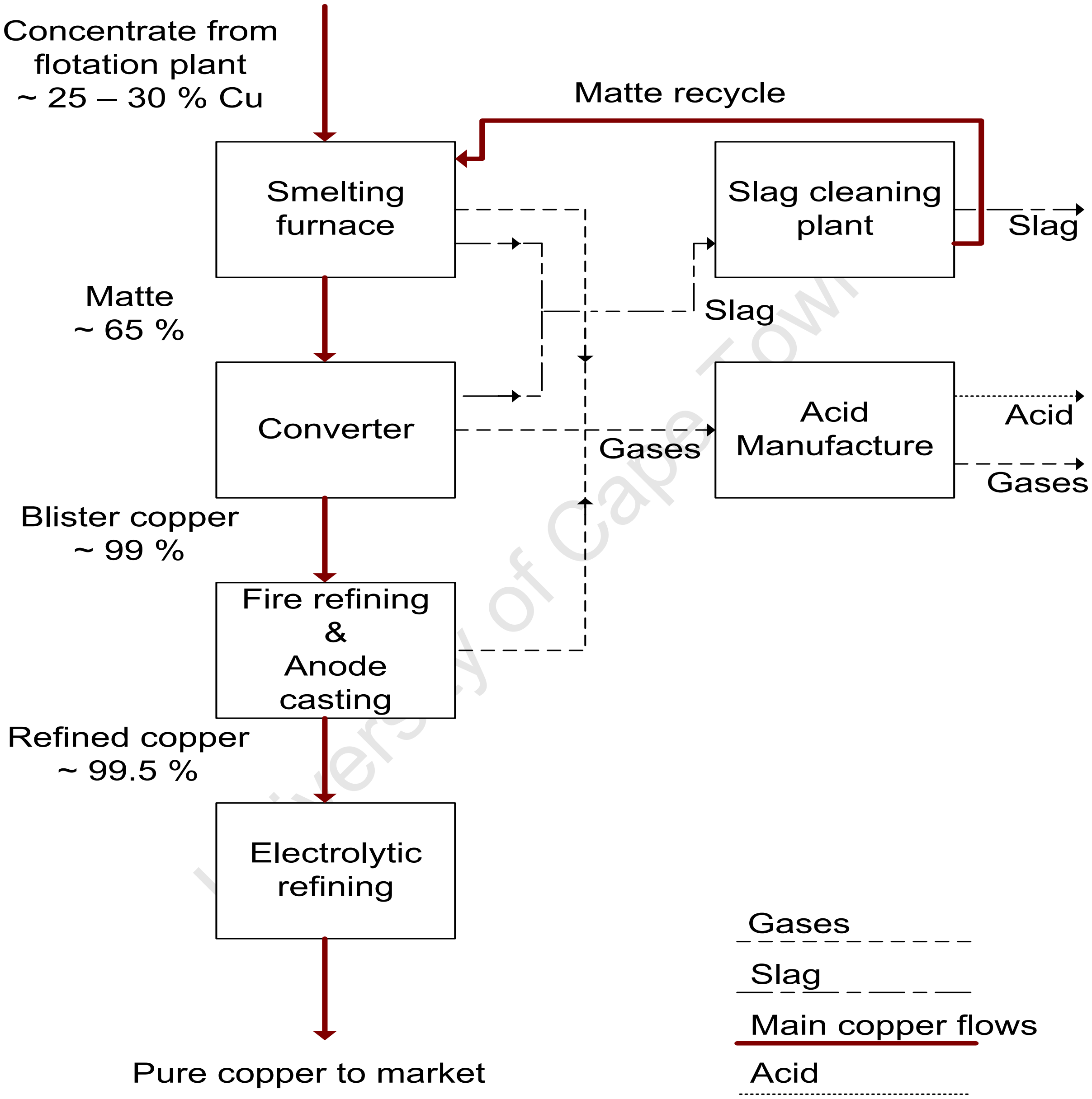


Figure 2.6: Block flow diagram of a copper smelter (adapted from Goonan, 2004)

This section gives a process description of the production of copper through the pyrometallurgical process route and reviews the different copper concentrate smelting and converting technologies that are used in industry.

Smelting

Matte smelting is the process through which copper concentrates are oxidised in the presence of air at 1250°C in a furnace. The products are a Cu-Fe-S matte, slag which is rich in iron and silica (BCS Incorporated, 2002; Davenport et al., 2002), and an off gas that contains 10 – 60% sulphur dioxide (Davenport et al., 2002). Fuels such as natural gas, coal, fuel oil and diesel are added to provide some of the energy required for melting the concentrate. Silica is fed as a flux to create a stable slag and to improve the immiscibility of matte and slag (Davenport et al., 2002). The specific gravities of matte and slag are typically 5 and 3 (Lakernik and Sevryukov, 1960) respectively, therefore the matte sinks to the bottom of the furnace and the slag floats above the matte. The products are tapped from holes in the furnace and processed further.

The reactions that occur in a copper smelting furnace are of the type (Biswas and Davenport, 1994)[9]:

$$2\,CuFeS_2 + {}^{5}\!/_{2}\;O_2 + \mathrm{SiO_2} \rightarrow Cu_2\mathrm{S}\cdot\mathrm{FeS}\;(\mathrm{m}atte\,) + \;FeO\cdot\mathrm{SiO_2}\;(\,slag) + 2\,SO_2 + \;\mathrm{heat} \quad (2.1)$$

Matte (sometimes presented as Cu-Fe-S) almost always exists as an alloy of Cu_2S and FeS, and this is independent of the mineralogy of the (sulphide) feed concentrate (Lakernik and Sevryukov, 1960).

Oxygen is usually added in the form of enriched air (Davenport et al., 2002; Ramachandran et al., 2003) and is an important control parameter in matte smelting. Excessive oxidation in the smelter can improve the matte Cu grade through the oxidative removal of Fe and S, however since the Fe and S in the matte provide the energy for the autothermal operation of the converter, this is not desirable (Davenport et al., 2002). Furthermore, excess oxygen promotes the oxidation of copper in the matte to copper oxide via the following reaction (Davenport et al., 2002):

$$Cu_2S + {}^{3}\!/_{2}\;O_2 \rightarrow Cu_2O + SO_2 \qquad (2.2)$$

The fraction of the Cu_2O that is formed by this reaction, which is not converted back to Cu_2S by reacting with FeS, dissolves in the slag, causing a reduction in the copper recovery from the process (Lakernik and Sevryukov, 1960; Davenport et al., 2002).

FeO is a product of the oxidation reactions that occur in the smelter (see equation 2.1). A high activity of FeO in the slag would cause the following unwanted reaction:

$$FeO + Cu_2S \rightarrow FeS + Cu_2O \;\textit{(Copper loss to slag)} \qquad (2.3)$$

Therefore, the activity of FeO is reduced by adding silica to the smelter prompting the following reaction (Davenport et al., 2002; Lakernik and Sevryukov, 1960).

$$FeO + SiO_2 \rightarrow FeO\cdot SiO_2 \;\textit{(Molten slag)} \qquad (2.4)$$

---

[9] For more reactions the reader is referred to the appendix B.

This reaction forms slag in the form of a stable silicate, and in the process, prevents the oxidation of FeO to Fe_2O_3 and Fe_3O_4 (Lakernik and Sevryukov, 1960).

Smelting technologies

There are about 8 smelting furnace technologies including Teniente, Noranda, Outokumpu, Inco, Reverberatory, Isasmelt, Mitsubishi and Ausmelt smelters, all of which differ in energy demands, operation, techno-economic and environmental performance (Davenport et al., 2002; Goonan, 2004). Some of these smelting technologies can be installed and used with a variety of converting technologies, resulting in different economic and environmental performance outcomes. Outokumpu flash smelters are the predominant copper smelting technology in the world (Ramachandran et al., 2003), and they accounted for 35 – 50% of the world smelting capacity in 2003 (Moskalyk and Alfantazi, 2003).

Outokumpu and Inco smelters utilise flash smelting technology, in which the bulk of the energy required for smelting the concentrates is obtained from the oxidation of Fe and S in the concentrate, making the process autothermal[10]. A flash smelter is shown in Figure 2.7.

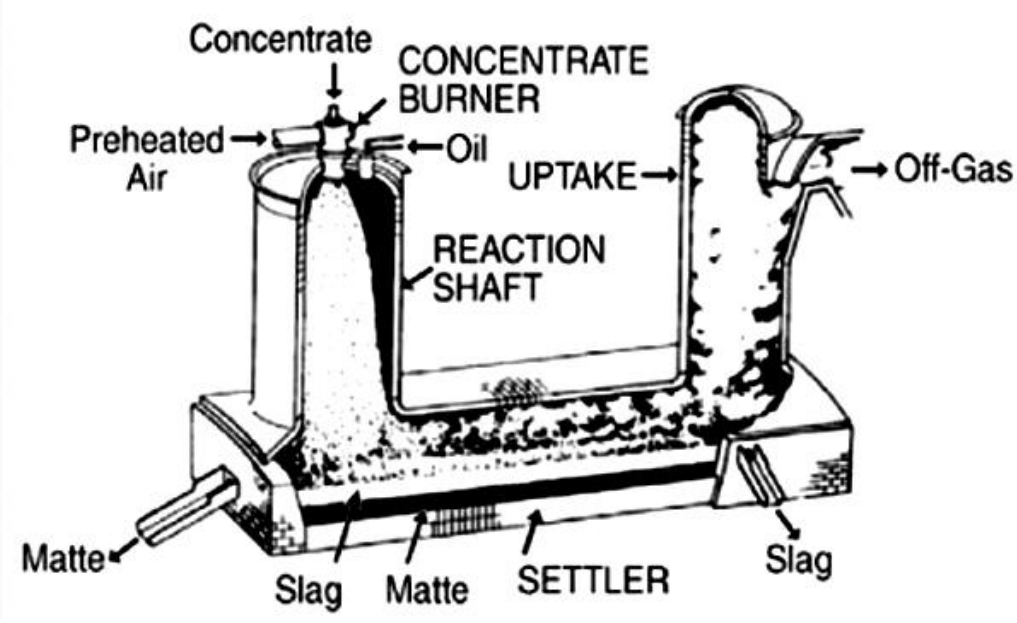


Figure 2.7: An Outokumpu flash smelter. The reactions take place in the solid gas phase found in the reaction shaft (King, 2007)

One of the notable differences between flash smelters and other smelting technologies, referred to as bath smelters, is that the smelting reactions of a flash smelter occur in the mixed gas-solid phase, and not in the molten bath (Davenport et al., 2002). Refer to the reaction shaft in Figure 2.7.

Figure 2.8 shows an Isasmelt smelter (a bath smelter) in which the reactions take place in the molten bath.

---

[10] A small amount of fuel is added in the settling part of the furnace to offset heat losses during the process (Kojo and Storch, 2006).

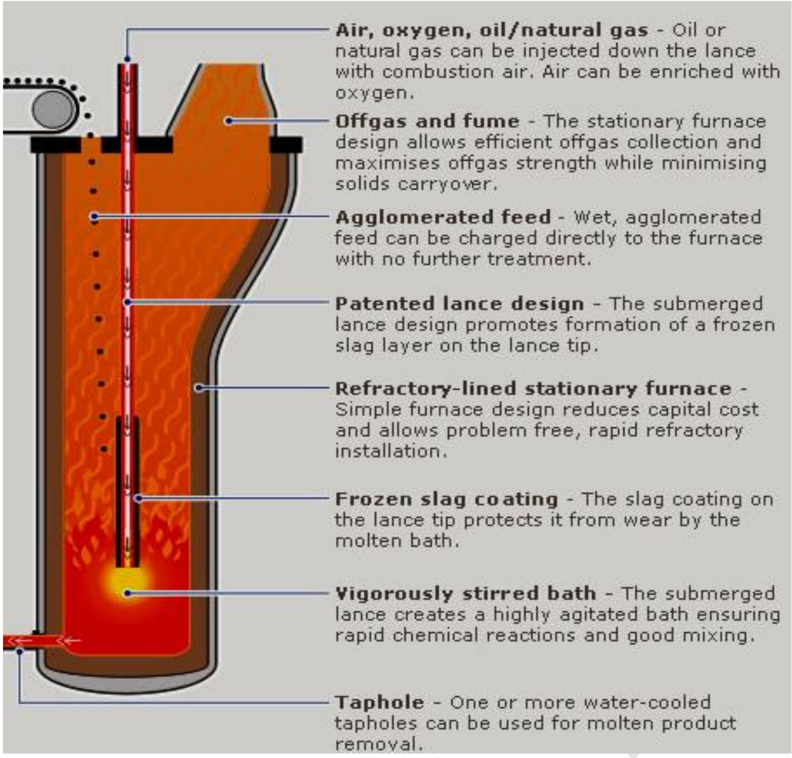


Figure 2.8: An Isasmelt smelter (a bath smelter). The reactions take place in the molten bath (Xstrata, 2006).

An analysis of modern copper smelters (flash and bath smelters) suggests that the bulk of the energy required for smelting is derived from the oxidation of S and Fe in the concentrate. However, the degree of autothermal operation varies with each smelting technology; therefore, the various technologies have different fuel requirements (Davenport et al., 2002; Moskalyk and Alfantazi, 2003)[11]. The specific energy consumption data for each technology is scarce in the public literature.

Until the 1970's, reverberatory furnaces were the dominant smelting technology used in copper concentrate smelting. Unlike modern smelters (Figure 2.7 and Figure 2.8), reverberatory smelters relied mainly on burning coal, oil and natural gas to generate heat, which made them energy intensive. Furthermore, they produced an offgas that was too dilute in SO_2 for its economical recovery as sulphuric acid. The reveberatory furnaces produced offgases that typically contained about 1% volume SO_2, whereas modern smelters produce offgases that contain 10 – 75% SO_2 (Davenport et al., 2002). Therefore, the offgas from reverberatory smelters was vented directly to the atmosphere, a process which became increasingly environmentally unacceptable towards the end of the 20th century (Biswas and Davenport, 1994).

Another notable difference between the copper smelting technologies is the moisture content of their concentrate feed. The concentrates fed to Outokumpu, Inco, Teniente and Noranda smelters are typically dry and therefore less fuel is required in the smelting

---

[11] For a specified concentrate.

process. However this advantage is offset by energy requirements either in the form of steam[12] or fuels, which are combusted to produce hot gases that are used for pre-drying the concentrate (Davenport et al., 2002). Energy savings can be achieved by using steam that is generated in waste heat boilers that utilise the hot off gases that are produced during smelting. The concentrate fed to the Ausmelt and Isasmelt smelters is moist and therefore, unlike the former smelters, a relatively larger amount of fuel is required during the smelting process (Davenport et al., 2002).

Converting

Copper matte (Cu-Fe-S) from the smelting furnace contains Fe, S and other impurities. It has a copper content of between 45 - 75% and is treated further to produce marketable copper (Ramachandran et al., 2003). Molten matte from the smelting furnace is fed, with silica flux and oxygen, into a converter in which the remainder of the Fe and S are oxidised to FeO and SO_2. Again, the activity of FeO in the slag is reduced via its reaction with SiO_2, forming $FeO{\cdot}SiO_2$ (slag). Converting is carried out at about 1200°C in the sequence shown by the following reactions (Davenport et al., 2002):

Slag forming stage

$$FeS + 1.5\,O_2 \rightarrow FeO + SO_2 \tag{2.5}$$

$$3FeS + 5\,O_2 \rightarrow Fe_3O_4 + 3\,SO_2 \tag{2.6}$$

Copper making stage

$$Cu_2S + 1.5\,O_2 \rightarrow Cu_2O + SO_2 \tag{2.7}$$

$$Cu_2S + 2\,Cu_2O \rightarrow 6\,Cu + SO_2 \tag{2.8}$$

The products of the converting stage are blister copper (~ 99% Cu), slag and an offgas containing SO_2 (Figure 2.6).

Converting technologies

There are about 6 converting technologies including Peirce-Smith, Outokumpu, Kennecott, Hoboken, Mitsubishi and Ausmelt converters (Goonan, 2004). Peirce-Smith converters are the dominant copper converting technology in the world (Ramachandran et al., 2003; Moskalyk and Alfantazi, 2003). They operate in batch mode and suffer from inefficiencies in gas capture due to gas leaks during operation (Davenport et al., 2002). Their other drawback is that they allow the SO_2 containing offgas to be diluted with air during operation resulting in a relatively weak off gas being fed to the acid plant (Moskalyk and Alfantazi, 2003; Davenport et al., 2002). Hoboken converters are a modified form of Peirce-Smith converters with improved gas handling features. Mitsubishi, Outokumpu, Kennecott and Ausmelt smelters are continuous converters and have better gas handling features than Peirce-Smith converters (Moskalyk and Alfantazi, 2003; Davenport et al., 2002).

---

[12] Steam is used for heating coils that are put in contact with the concentrate to dry it.

Fire refining and anode casting

The product from the converters is blister copper which is refined in fire to remove the dissolved oxygen from the preceding production stages and the remaining sulphur (Biswas and Davenport, 1994; Davenport et al., 2002). Fire refining entails blowing air into the molten copper from the converters to remove the remaining sulphur, and thereafter, passing a hydrocarbon through the copper to remove dissolved oxygen. This is typically performed in a rotary furnace (Biswas and Davenport, 1994). Removal of the dissolved sulphur and oxygen prevents the formation of SO_2 bubbles during the anode casting stage (Biswas and Davenport, 1994; Davenport et al., 2002). Typical hydrocarbons used for reducing oxygen that dissolves as Cu_2O are liquid petroleum gas, oil and natural gas (Lakernik and Sevryukov, 1960; Davenport et al., 2002).

$$4\,Cu_2O + CH_4 \rightarrow CO_2 + 2\,H_2O + 8\,Cu \text{ (Copper regeneration)} \qquad (2.9)$$

The product from the fire refining stages is then cast into copper anodes by pouring the molten copper into cathode shaped molds that rotate on a casting wheel. The molten anodes are cooled by cold water sprays and are eventually removed from the molds (Davenport et al., 2002).

Electrorefining

The cast copper anodes are then sent to an electrorefinery, where copper anodes are electrochemically dissolved in a H_2SO_4, water and $CuSO_4$ solution, and subsequently deposited onto thin stainless steel sheets that function as starter sheets for the production of copper cathodes that have a purity of over 99.99% Cu (Biswas and Davenport, 1994; Davenport et al., 2002). The dissolution and deposition of copper from the anodes to the cathodes is represented by equations 2.10 to 2.12.

$$Cu\,(s)\;anode \rightarrow Cu^{2+} + 2e^{-} \qquad (2.10)$$

$$Cu^{2+} + 2e^{-} \rightarrow Cu\,(s)\;cathode \qquad (2.11)$$

$$Cu\,(s)\;anode \rightarrow Cu\,(s)\;cathode \text{ (Overall)} \qquad (2.12)$$

The main purpose of electrorefining copper is to remove impurities such as arsenic, bismuth, gold, silver, and nickel. For the purposes of producing commercial grade copper cathodes, these elements are considered as impurities, however they are collected from the bottom of the refining tanks as slimes, and processed further to produce various saleable by-products that can make significant contributions towards a mine’s revenue. A detailed review of the chemistry and behaviour of impurities during electrorefining is beyond the scope of this project; however, the subject is discussed in more detail by Wenzl et al. (2007).

Slag treatment

The slag from the smelting and converting furnaces contains copper and is often cleaned before it is discarded[13]. This cleaning step is often performed in cleaning furnaces that effectively allow more time for the matte particles to separate out from the slag, and facilitate the reduction of copper oxide (Cu_2O) through the addition of a reducing agent (Davenport et al., 2002). Some cleaning furnaces accept only smelter furnace slag, whereas others accept both smelter and converter furnace slags (Davenport et al., 2002; Ramachandran et al., 2003). Typically, pre-treated smelter and converter slags contain about 1 – 8 % copper (Davenport et al., 2002). The final products from the slag smelter are a matte and a final slag containing approximately 60% and less than 1 % copper, respectively (Davenport et al., 2002). The matte is recycled back to the smelting furnace (see Figure 2.6). Equations 2.13 to 2.15 show the slag cleaning reactions (Davenport et al., 2002):

$$C + Cu_2O \rightarrow CO + 2\,Cu^{\circ} \tag{2.13}$$

$$C + Cu_2O + FeS_2 \rightarrow Cu_2S + FeS + CO \text{ (Matte)} \tag{2.14}$$

$$C + Fe_3O_4 \rightarrow CO + 3\,FeO \textit{ (Magnetite reduction)} \tag{2.15}$$

Gas treatment and sulphuric acid production

Offgases containing, amongst other substances, N_2, NOx, CO_2, H_2O, volatile impurities and SO_2 are produced during the smelting and converting stages. The concentration of SO_2 in the offgases typically varies between 10 – 75% (Davenport et al., 2002). Due to stringent environmental regulations and the ability to increase revenues from selling by-products, smelting companies convert the SO_2 in the offgas to sulphuric acid and other sulphur containing products such as gypsum (Davenport et al., 2002).

$$SO_2\,(g) + 0.5\,O_2(g) \rightarrow SO_3(g) \tag{2.16}$$

$$SO_3(g) + H_2O\,(l) \rightarrow H_2SO_4(l) \tag{2.17}$$

$$H_2SO_4 + CaCO_3 + H_2O \rightarrow CaSO_4 \cdot 2H_2O + CO_2 \tag{2.18}$$

Total sulphur capture efficiency varies widely amongst copper smelters i.e. between 60 – 99.9%, however, many plants are now operating in the higher end of this efficiency range (Biswas and Davenport, 1994; Ramachandran et al., 2003; R. Shaw, personal communication, 2010).

---

[13] Some companies discard the slag without cleaning it while others mill it and re-float it (Ramachandran et al., 2003).

2.2.5 Bio / Hydrometallurgical extraction

For a long time, smelting copper concentrates produced from primary and secondary sulphide ores has been the preferred process route for copper production. In recent years however, smelting sulphide concentrates has generally been perceived negatively from an environmental point of view, mainly due to the historic emissions of SO_2 and CO_2 from the mining industry (Gaines, 1980; Peacey et al., 2003). Direct emissions to the atmosphere were common during the period when reverberatory furnaces were the dominant smelting technology. However, stringent environmental regulations caused radical improvements in smelting technology, resulting in large scale reductions of CO_2 and SO_2 emissions (Ramachandran et al., 2003).

During the period when environmental regulations were becoming more stringent, researchers developed a suite of hydrometallurgical processes that were designed to reduce or eliminate the sulphur dioxide emissions associated with copper production from sulphide concentrates (Peacey et al., 2003; Habashi, 2007). Some of the processes, developed between 1970 and 1995 reached commercialisation but were later abandoned due to technical challenges (Habashi, 2007).

Nevertheless, towards the end of the century a new class of hydrometallurgical and biohydrometallurgical processes for treating copper concentrates emerged, and these processes were developed to address some of the challenges faced by copper mines with regards to (Marsden, 2004; Dreisinger, 2005, 2006 a; Marsden and Wilmot, 2007):

- the high costs of treating and refining concentrates in external refineries (periodically).
- limited smelter capacity when there is excess copper concentrate in the market.
- the high capital cost of installing a new smelter. The capital costs of a smelter and refinery project are estimated to be US$ 3,000 – US$ 6,000 / annual tonne of copper.
- the requirements of acid for heap and stockpile leach operations onsite.
- penalty charges imposed by smelters for treating dirty[14] and low grade concentrates.

---

[14] Copper mines that produce concentrates that contain high levels of As, Bi, Sb, Pb and Zn are penalised by smelting companies that buy their concentrates or treat them on their behalf for impurities that exceed the smelter’s tolerance levels.

The hydrometallurgical concentrate leaching processes are classified through their hydrometallurgical media, as: (Davenport et al., 2002; Peacey et al., 2003; Dreisinger, 2006 a):

- sulphate processes
- sulphate / chloride processes
- chloride (and bromide) processes
- ammonia processes.

Table 2.3 summarises the leaching conditions of the main copper concentrate leaching processes, and the final state of the sulphur that is originally in the concentrates (Davenport et al., 2002; Norgate, 2001; Peacey et al., 2003; Dreisinger, 2005, 2006 a; Outotec, 2007).

Table 2.3: Summary of main hydrometallurgical processes for treating copper concentrates

Technology	Leaching conditions and *(Lixiviant)*	Sulphur product
Total Pressure Oxidation (POX)	220°C and 30 – 40 atm *(H_2SO_4)*	$(SO_4)^{2-}$
Intec	85°C and 1 atm *(NaCl / BrCl)*	S°
BioCOP	Bacterial, 65 – 80°C and 1 atm *(H_2SO_4)*	$(SO_4)^{2-}$
Activox	Fine grind, 90 – 110°C and 10 – 12 atm *(H_2SO_4)*	S° & $(SO4)^{2-}$
Bactec / Mintek	Bacterial, 35°C and 1 atm *(H_2SO_4)*	S & $(SO_4)^{2-}$
CESL	Fine grind, 140 – 150°C and 10 – 12 atm *(H_2SO_4 & Cl^-)*	S°
HydroCopper	80 - 100°C and 1 atm *($CuCl_2$)*	$(SO_4)^{2-}$
Dynatech	150°C and 10 – 12 atm *(H_2SO_4)*	S°

Table 2.3 summarises the operating conditions and the leaching agents *(lixiviants)* of the main concentrate leaching processes that can be used as alternatives to smelters. In these processes, leaching is typically performed in either stirred tank reactors (for the low temperature and pressure processes) or in autoclaves (for the high temperature and pressure processes). It was mentioned previously that one of the main reasons for the development of these hydrometallurgical processes as alternatives to smelting was the elimination of SO_2 gas, which when released to the atmosphere contributes to the formation of acid rain. Table 2.3 also shows the final state of the sulphur that would originally be in the concentrate for the hydrometallurgical processes. In these processes, sulphur is typically converted to either elemental sulphur (S°) which remains in the leach residue and is generally considered environmentally stable, or to sulphate $(SO_4)^{2-}$ containing products such as sulphuric acid (in solution) without the production of SO_2 gas.

With the exception of the Total Pressure Oxidation process and its variant - a medium temperature pressure oxidation process, (see Marsden and Wilmot 2007) - there is no evidence that the other hydrometallurgical concentrate treatment processes are currently in commercial operation. The BioCOP process was used at semi-commercial scale near Chuquicamata (Chile) by BHP Billiton and CODELCO, but the plant was recently shut down for reasons cited as commercial by Gericke et al. (2009). The HydroCopper process is regarded as proven technology since it has been operated successfully in a large scale demonstration plant (Outotec, 2007), however there is no information regarding its current status. It is likely that the other processes are still in the pilot or demonstration plant scale of their development since there is no current information that been published regarding their status. In this study, Total Pressure Oxidation (POX) was selected as the main alternative to concentrate smelting and its economic and environmental performances are compared to the economic and environmental performances of smelting in Chapter 4. POX was selected over the other processes because adequate data was available to enable its modelling. Furthermore, this technology is currently used commercially at Kansanshi Copper Mine in Zambia and it was used commercially at Bagdad Copper Mine in the USA until 2009[15] (First Quantum Limited, 2010; Freeport-McMoRan Copper & Gold Inc., 2010 a). The Total Pressure Oxidation process is described next.

Total Pressure Oxidation (POX)

Total Pressure Oxidation (POX) is a hydrometallurgical process in which copper sulphide minerals in a concentrate, or possibly in a high grade ore, are oxidised to sulphates and sulphuric acid in an autoclave. The typical operating conditions of the autoclave are 210 – 220 °C and 30 atm. This process is currently used commercially at Kansanshi Copper Mine in Zambia and it was used commercially at Bagdad Copper Mine in the USA until 2009 (First Quantum Limited, 2010; Freeport-McMoRan Copper & Gold Inc., 2010 a).

Figure 2.9 illustrates a generic POX flowsheet which is described below.

---

[15] The Total Pressure Oxidation process at Bagdad mine (USA) was converted to a molybdenum processing facility; no reasons for the change are cited in the public media (Freeport-McMoRan Copper & Gold, 2010 a).

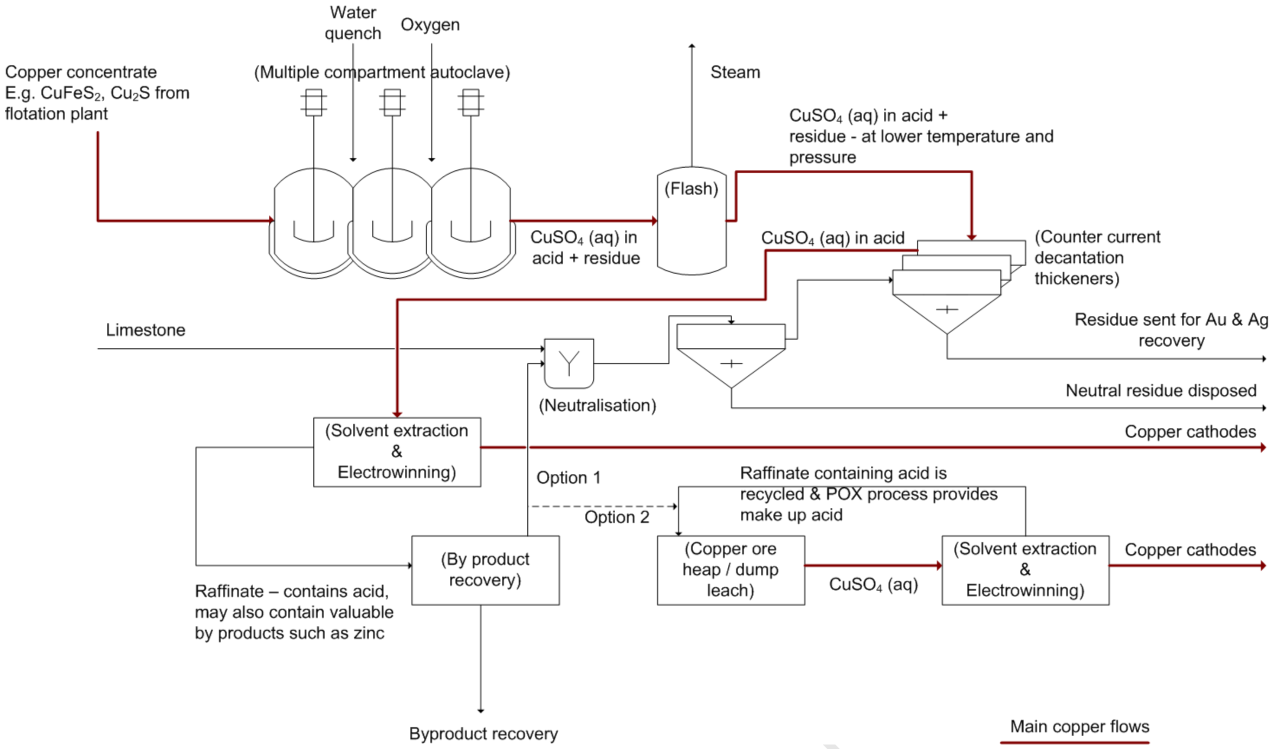


Figure 2.9: Illustration of the POX process (adapted from McElroy and Young, 1999)

The autoclaves are normally designed to have an effective average solids density of 8 – 12% (w/w) (Dunn, 2009 a). The concentrate or ore sulphide minerals are oxidised by reactions of the type (Dreisinger, 2005, 2006 a):

$$\mathrm{CuFeS_2} + O_2 + 2\,H_2SO_4 \rightarrow CuSO_4 + FeSO_4 + 2\,H_2O + 2\,S \tag{2.19}$$

$$FeS_2 + 0.5\,O_2 + H_2SO_4 \rightarrow FeSO_4 + 2\,S + H_2O \tag{2.20}$$

$$FeSO_4 + {}^{1}\!/_{4}\,O_2 + {}^{1}\!/_{2}\,H_2SO_4 \rightarrow {}^{1}\!/_{2}\,Fe_2(SO_4)_3 + {}^{1}\!/_{2}\,H_2O \tag{2.21}$$

$$Fe_2(SO_4)_3 + 3\,H_2O \rightarrow Fe_2O_3 + 3\,H_2SO_4 \tag{2.22}$$

$$S + {}^{3}\!/_{2}\,O_2 + H_2O \rightarrow H_2SO_4 \tag{2.23}$$

$$1.5\,Fe_2(SO_4)_3 + 7\,H_2O \rightarrow H_3OFe_3(SO_4)_2(OH)_6 + 2.5\,H_2SO_4 \tag{2.24}$$

Iron is mainly precipitated as a hematite (Fe_2O_3) (McElroy and Young, 1999; Dreisinger, 2006 b), and hydronium jarosite is precipitated as a co product (McElroy and Young, 1999).

The autoclave discharge is a hot slurry that contains leached Cu in the form of $CuSO_4$. The slurry is fed to a flash tank to reduce its pressure, in stages, to atmosheric pressure and its temperature to the corresponding boiling point, in the process producing steam (Dreisinger, 2005, 2006 a). The slurry is then sent to counter-current decantation thickeners where the pregnant leach solution (PLS), rich in sulphuric acid and $CuSO_4$, is seperated from the solid residue. The solid residue is neutralised and discarded if it does not contain precious metals, otherwise the precious metals can be leached in cyanide and recovered from the solution on activated carbon. The PLS from the thickeners is sent to solvent extraction and electrowinning facilities for copper extraction.

After the copper is extracted, a raffinate which is rich in acid and lean in copper is produced and may contain valuable by-products such as $ZnSO_4$ (Dreisinger, 2005). Therefore, it can be treated to remove the by-products before or after acid neutralisation, or it can be used on heap / ROM leach operations onsite that require acid. This gives a financial credit for acid to the POX process and reduces or eliminates neutralisation costs, depending on the acid requirements of the heap operations.

The chemical reactions that occur in the autoclave typically result in a net heat release that causes the temperature of the pulp in the autoclave to exceed the prescribed process temperature (210 – 220 °C). Temperature control in the autoclave is essential, and it is normally achieved by using cooling coils or water / raffinate quench cooling. However, these cooling techniques typically result in a loss of effective reactor volume and consequently throughput. Hydromet SA recently invented a cooling concept known as the Flash Thickener Recycle that significantly eliminates the aforementioned problems and results in increased solids residence time and optimum utilisation of the installed autoclave. For a detailed discussion on the cooling concepts refer to Dunn (2009).

Heap Leaching

Heap leaching is a relatively low cost method used for extracting copper from oxide and secondary sulphide ores (Peacey et al., 2003; Robertson et al., 2005). It involves irrigating a lixiviant onto a heap of crushed ore that has been stacked on an impervious base, fitted with a solution drainage system (Petersen and Dixon, 2007; Davenport et al., 2002). Alternatively, the heap could be placed on a slightly sloping base to facilitate solution drainage by gravity (Petersen and Dixon, 2007). The pregnant leach solution, which is rich in copper ions, is collected in ponds or tanks before it is sent for copper extraction in solvent extraction and electrowinning facilities. Usually the ore is cured and agglomerated in rotating drums by use of strong sulphuric acid or an acid containing raffinate prior to heap leaching (Petersen and Dixon, 2007; Davenport et al., 2002). Acid curing causes the rapid dissolution of copper from oxide ores and prevents the precipitation of ferric sulphate during the subsequent leaching of sulphide ores (Davenport et al., 2002; Lu et al., 2007). Agglomeration improves the permeability of the ore by causing the fine particles to attach to the coarser particles (Lewandowski and Kawatra, 2007; Lu et al., 2007).

Generally, heap leaching has lower capital and operating costs compared to other copper extraction methods (see section 2.3), however it has the negative drawbacks of low copper recoveries and long leach periods (200 – 300 days)[16] (Robertson et al., 2005). Furthermore,

---

[16] These are the leach periods typically required to achieve 70 – 80% copper extraction in the presence of bacteria, and correspond to secondary copper sulphide ores. Oxide ores leach relatively faster, see Figure 2.11.

chalcopyrite, the most abundant copper mineral, can not be treated economically through heap leaching, unless it is being leached from waste dumps (J. Petersen, personal communication, 2011). Chalcopyrite behaviour during heap leaching is discussed below (see Figure 2.11).

A conceptual diagram of a heap leach operation is shown in Figure 2.10.

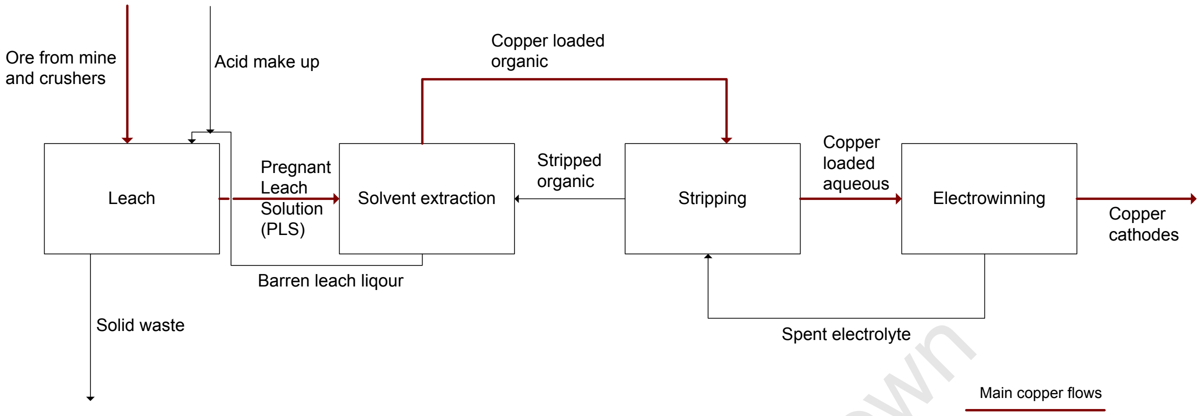


Figure 2.10: Simplified flow diagram of a heap leach operation (adapted form Dresher, 2001)

The oxide and sulphide minerals dissolve through the following type of reactions (Iasillo and Schlitt, 1999; Crundwell, 2003; Petersen and Dixon, 2007):

$$CuSiO_3 \cdot 2H_2O + H_2SO_4 \rightarrow CuSO_4 + SiO_2 + 3\,H_2O \quad \text{for oxide ores} \qquad (2.25)$$

$$Cu_2S + 1.6Fe^{3+} \rightarrow 1.6Fe^{2+} + 0.8Cu^{2+} + Cu_{1.2}S \text{ for sulphide ores} \qquad (2.26)$$

$$Cu_{1.2}S + 2.4Fe^{3+} \rightarrow 2.4Fe^{2+} + 1.2Cu^{2+} + S° \qquad (2.27)$$

$$FeS_2 + 8H_2O + 14Fe^{3+} \rightarrow 15Fe^{2} + 2SO_4{}^{2-} + 16H^+ \qquad (2.28)$$

$$Fe^{2+} + 0.25O_2 + H^+ \xrightarrow{bacteria} Fe^{3+} + 0.5H_2O \qquad (2.29)$$

$$S° + 1.5O_2 + H_2O \xrightarrow{bacteria} H_2SO_4 \qquad (2.30)$$

Leaching rates

The copper dissolution kinetics for copper oxide ores are relatively fast (Bartlett, 1997) compared to the leach kinetics of copper sulphide ores. The catalytic action of added or naturally occurring bacteria, in heap leach operations, can increase the rate of leaching by up to a million-fold (Davenport et al., 2002). The bacteria catalyse the oxidation of ferrous (Fe^{2+}) to ferric (Fe^{3+}) in the presence of oxygen (Bartlett, 1997; Davenport et al., 2002). Fe^{3+} oxidises the copper sulphide minerals (Bartlett, 1997), releasing copper ions into the solution. Pyrite is normally associated with copper sulphide minerals and serves as a major source of Fe ions and acid.

The relative dissolution kinetics of various copper minerals under bacterial heap leach conditions are shown in Figure 2.11.

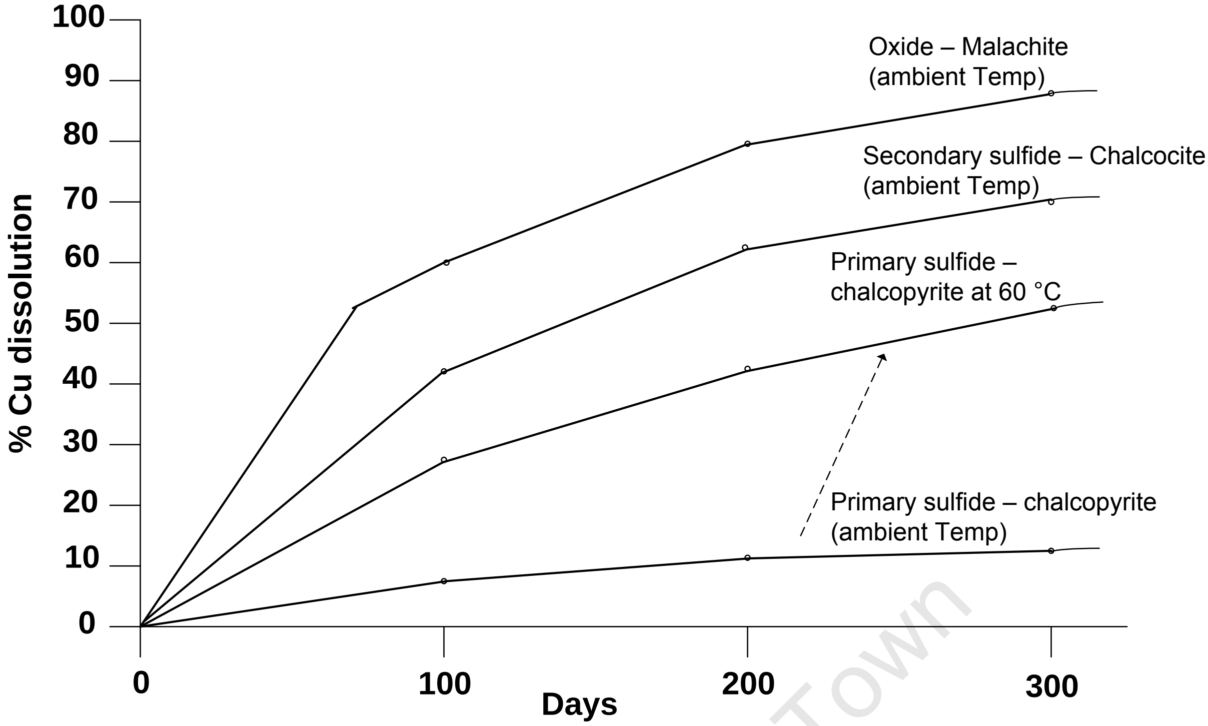


Figure 2.11: A comparison of the leach kinetics of various copper ores during heap leaching (adapted from Robertson et al., 2005)

Under typical plant heap leaching conditions the rate of chalcopyrite extraction is very low; however it is improved when temperatures are elevated. The slow leaching kinetics of chalcopyrite are believed to be caused by the formation of passive copper polysulphide and elemental sulphur layers on the ore particles (Davenport et al., 2002; Dreisinger, 2006 b; Watling, 2006).

Solvent extraction / Electrowinning (SX/EW)

Solvent extraction involves the removal of Cu^{2+} ions from the pregnant leach solution that flows from the leach pads, with an organic substance (HR). This process produces sulphuric acid which is recycled back to the heap in the form a raffinate that has a low copper content (Figure 2.10). The copper that is loaded in the organic liquid is subsequently stripped off through contact with strong sulphuric acid, producing a copper loaded aqueous liquid which is sent to the electrowinning plant were copper is electrochemically extracted as copper cathodes, and water dissociated to O_2 (g) and H^+ (aq) ions. The copper-stripped organic is recycled back to the solvent extraction plant and the copper lean (but acid-rich) aqueous liquid is recycled back to strip more copper (Davenport et al., 2002; Dreisinger, 2006 a).

$$CuSO_4 + 2HR \rightarrow CuR_2 + H_2SO_4 \qquad \text{Solvent extraction} \qquad (2.31)$$

$$CuR_2 + H_2SO_4 \rightarrow CuSO_4 + 2HR \qquad \text{Stripping} \qquad (2.32)$$

$$CuSO_4 + H_2O \rightarrow \mathrm{Cu}^\circ + {}^1/_2\ O_2\ (g) + H_2SO_4 \qquad \text{Electrowinning} \qquad (2.33)$$

$$CuSO_4 + H_2O \rightarrow \mathrm{Cu}^\circ + {}^1/_2\ O_2\ (g) + H_2SO_4 \qquad \text{Overall} \qquad (2.34)$$

Acid consumption and production

The sulphuric acid balance is an important aspect of industrial heap leaching operations. Acid consumption can account for 12.5% of the operating costs of a heap leaching operation (Davenport et al., 2002), depending on the relative price of acid and the acid consumption capacity of the ore. As shown in equation 2.25 and in the overall reactions in ***Appendix D***, sulphuric acid is consumed when copper is leached from its host minerals, however this acid is regenerated in the solvent extraction facilities and is recycled back to the heaps in the form of an acid raffinate for leaching (see equation 2.31 and Figure 2.10). Acid is also produced when sulphur from the sulphide minerals such as pyrite is oxidised (Robertson et al., 2005).

It is well established that gangue minerals are the main acid consuming species in heap leaching operations. Special emphasis is normally given to carbonate minerals because of their relatively high solubility in acid (see equation 2.36). However, silicate and limonite minerals must not be disregarded as they also consume acid (Jansen and Taylor, 2003). Whilst the silicate breakdown reactions consume acid, the products from these reactions react and regenerate acid (see equation 2.37 – 2.40), therefore, net acid consumption is a balance between acid consumption and regeneration in heaps (Jansen and Taylor, 2003).

Carbonate (Calcite) breakdown and precipitation

$$CaCO_3 + H_2SO_4 + H_2O \rightarrow CaSO_4 \cdot 2H_2O + CO_2 \tag{2.36}$$

Silicate (Muscovite) breakdown

$$2\,KAl_3Si_3O_{10}(OH)_2 + 10\,H_2SO_4 \rightarrow K_2SO_4 + 3\,Al_2(SO_4)_3 + 6\,H_4SiO_4 \tag{2.37}$$

Limonite (Goethite) breakdown

$$2\,FeO(OH) + 3\,H_2SO_4 \rightarrow Fe_2(SO_4)_3 + 4H_2O \tag{2.38}$$

Precipitation (Alunite)

$$3\,Al_2(SO_4)_3 + K_2SO_4 + 12H_2O \rightarrow 2\,KAl_3OH_6(SO_4)_2 + 6\,H_2SO_4 \tag{2.39}$$

Alteration (Kaolinite)

$$H_2O + Al_2(SO_4)_3 + 2H_4SiO_4 \rightarrow Al_2O_3 \cdot 2SiO_2 \cdot 2H_2O + 3\,H_2SO_4 \tag{2.40}$$

There is not much empirical data available regarding the acid consumption capacity of the silicate and limonite minerals under heap leaching conditions, however qualitative descriptions of the behaviour of the silicates can be found in the literature (Seyedbagheri et al., 2009). Figure 2.12 illustrates the relative dissolution kinetics of various silicate minerals.

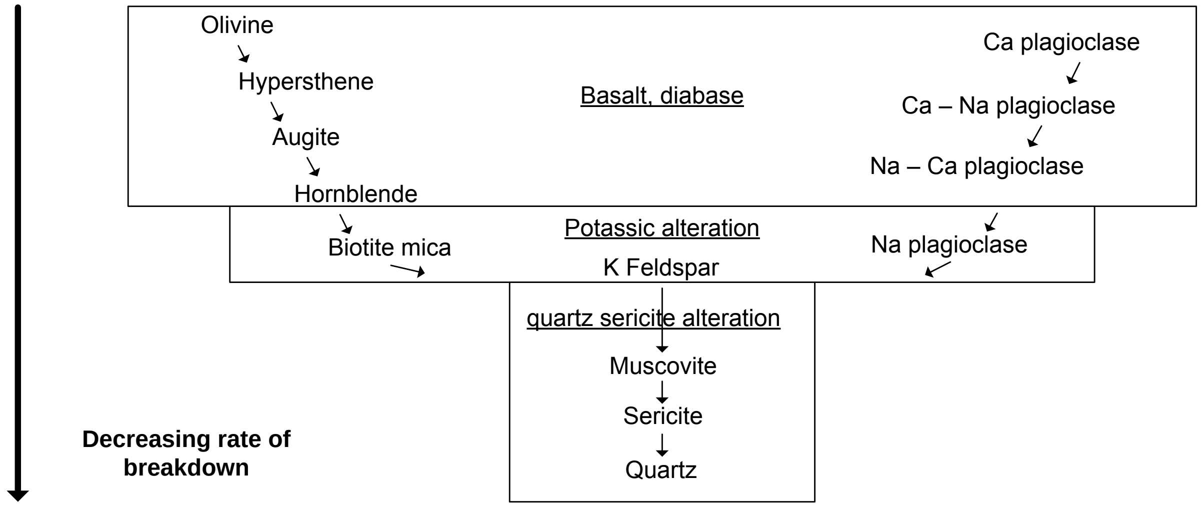


Figure 2.12: The relative dissolution kinetics of various silicate minerals in heap leach operations (adapted from Seyedbagheri et al., 2009)

Net acid consumption in heap leaching operations varies between 1 – 14 t acid [17] / t copper produced (Davenport et al., 2002), however operations on the lower side of the range are more common. The sources of acid for heap leaching operations include:

1. local and offshore smelters that process sulphide-containing concentrates and produce acid as a by-product.
2. hydrometallurgical processes that are alternatives to smelting, that produce an acid-containing raffinate as a by product.
3. acid purchased from companies that use sulphur, from the desulphurisation of flue gas and crude oil, as a raw material for acid production (ESA and EFMA (2000).

The factors that affect acid consumption and production in heap leach processes are discussed in more detail by Jansen and Taylor (2003).

---

[17] These values were obtained from plant data (Davenport el al., 2002). Some heap leach operations are net acid producers.

2.3 The economics of copper production

The technical aspects associated with the production of copper have been reviewed in sections 2.1 and 2.2. This section reviews the economic aspects of copper production.

2.3.1 Capital costs

Typical fixed investment capital costs of a new mine, concentrator, smelter and refinery are shown in Table 2.4

Table 2.4: The fixed investment costs of a new mine, concentrator, smelter, and refinery producing copper from an ore containing 0.75% copper (Davenport et al., 2002)

Section	Fixed investment cost US$ per (annual tonne of copper)
Mine (Open pit)	3,000
Concentrator	2,500
Smelter	2,500
Refinery	500
Total	**8,500**

The capital costs shown in Table 2.4 are at the *study estimate* level of accuracy (± 25%), and they can vary widely for individual projects (Davenport et al., 2002). Using the data presented in Table 2.4, in the year 2002[18], new copper production facilities producing 100,000 tonnes of copper annually would have required a fixed capital investment of US$ 850,000,000 to construct.

During the same period, POX/SX/EW plant cost estimations were between US$ 1,000 – 2,500 per annual tonne of copper (McElroy and Young, 1999; Marsden, 2004). The lower end of the range was for plants that had free solvent extraction (SX) and electrowinning (EW) capacity, perhaps from previous operations (former heap / ROM leach operations) and the high end for plants that required completely new solvent extraction and electrowinning facilities.

---

[18] Current and comprehensive data was not available, and it is scarce in the public literature.

Comparing the price range of a POX plant to that of a new smelter and refinery (US$ 1,000 – 2,500 compared to US$ 3,000 – 6,000[19] per annual tonne of copper), it can be seen that a POX plant typically costs less than a smelter and refinery.

During the same period, Peacey et al. (2003) estimated that the capital costs of a mine, heap leach, solvent extraction and electrowinning operation were US$ 4,000 – 5,000 per annual tonne of copper. The capital costs were distributed as follows:

- Open pit mine – 33.3%
- Ore crushing and heap leach system – 33.3%
- Solution ponds, solvent extraction and electrowinning – 33.3%

Comparing the total fixed capital costs required to construct a new mine, concentrator, smelter and refinery project (US$ 8,500 per annual tonne of copper) to the heap leaching capital costs (US$ 4,000 – 5,000 per annual tonne of copper), it can be seen that ore heap leaching is less capital intensive compared to smelting or leaching concentrates. However, as discussed in section 2.2.5, heap leaching currently suffers from the shortcoming of not allowing for the economical recovery of copper from chalcopyrite ($CuFeS_2$) due to low copper extraction rates (refer to Figure 2.11).

2.3.2 Operating costs

During the same period when the capital costs were quoted (circa 2002), the direct (production) operating costs of a mine, concentrator, smelter and refinery project were approximately US$ 1,100 per tonne of copper and the direct operating costs of a mine, heap leach, solvent extraction and electrowinning project were approximately US$ 700 per tonne of copper. Including indirect costs (capital recovery and interests), the total manufacturing costs of smelter and heap leach projects were approximately US$ 1,500 – 2,200 and US$ 700 – 1,500 per tonne respectively. A more detailed breakdown of these costs is given by Davenport et al. (2002), and a detailed discussion on the typical components of direct and indirect costs is given by Couper (2003). Comparing the operating costs of a smelter and heap leach project shows that heap leaching projects typically cost less than smelting projects to operate, a similar result to the capital cost estimates.

The costs given thus far were quoted to give the reader an order of magnitude appreciation of the costs associated with copper extraction. In practice, the capital and operating costs of extracting copper vary widely for individual operations and as such, each case under review

---

[19] Davenport et al.(2002), as shown in Table 2.4, give smelter and refinery costs of $ 3,000, however the cost range of smelter and refinery projects is $ 3,000 – 6,000 per annual tonne of copper (Peacey et al., 2003; Marsden, 2004; Dreisinger, 2005).

should be treated independently. Figure 2.13, known as a C1 Cash Cost Curve, illustrates this point for the direct operating costs of the world's copper producing mines in 2009.

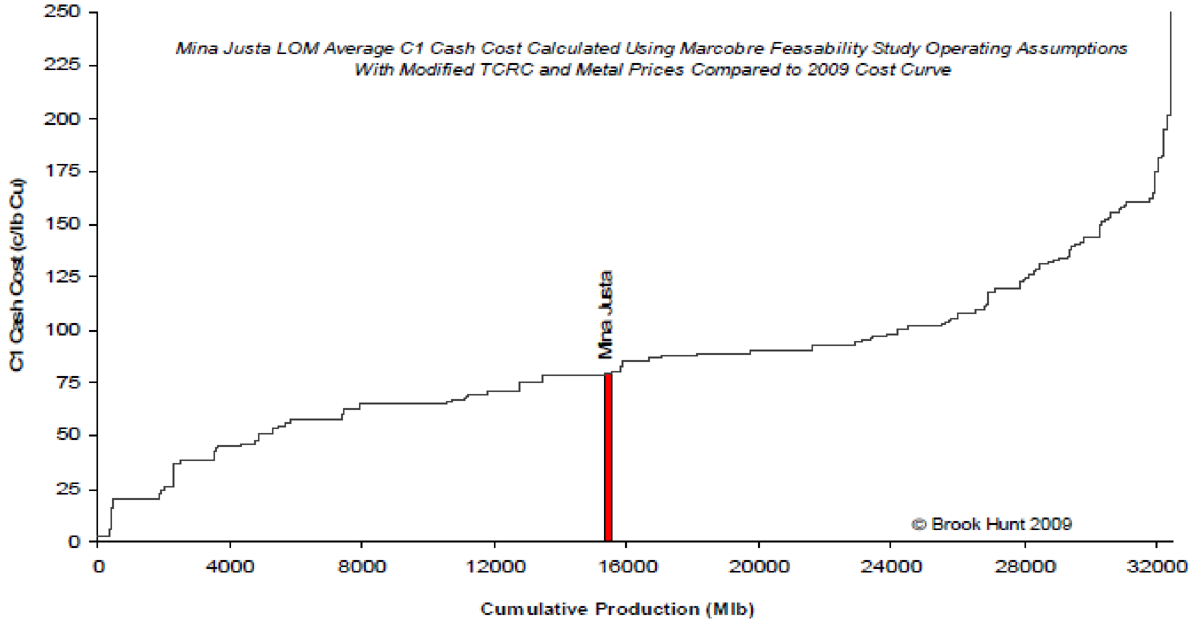


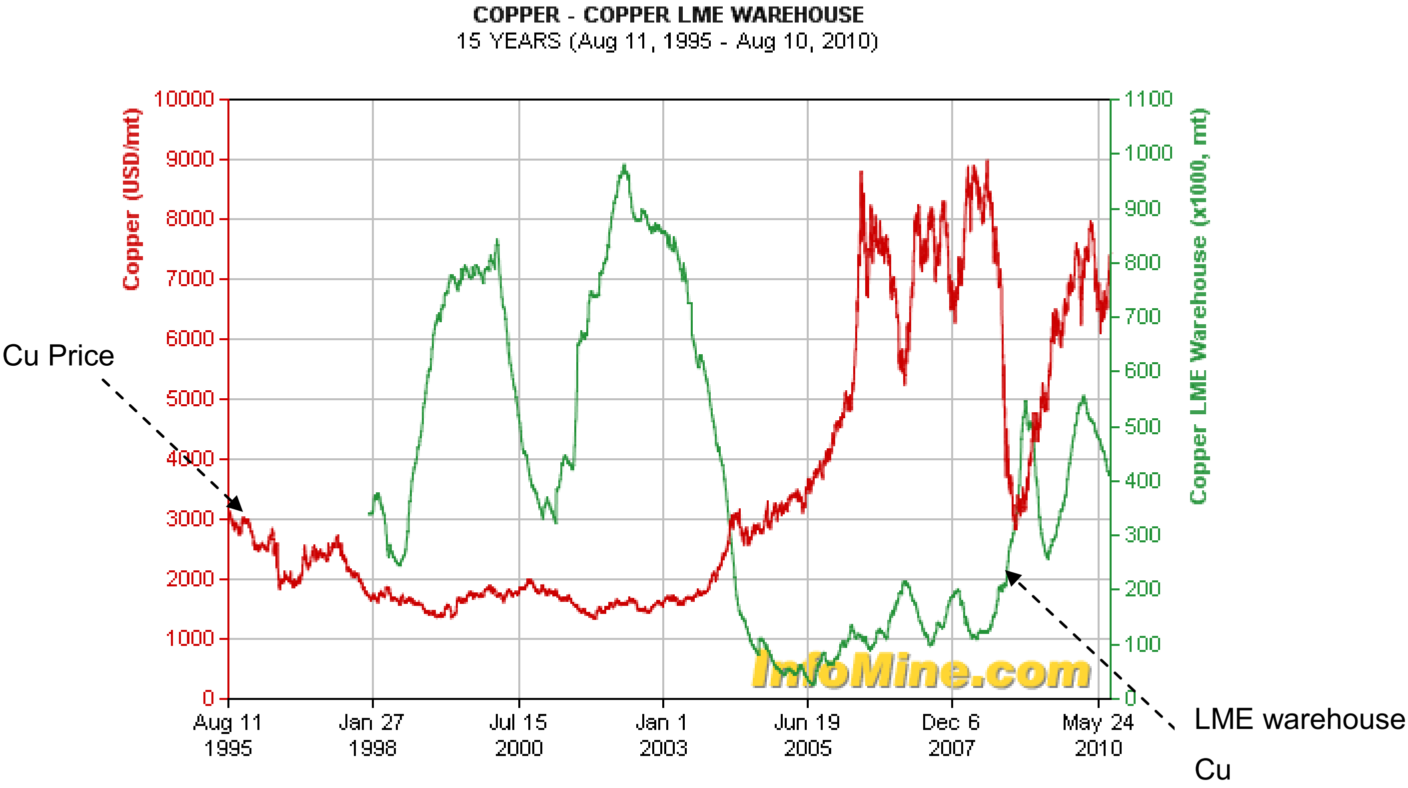
Figure 2.13: Example of a C1 cost curve (Brook Hunt as cited in Chariot Resources, 2010)

A C1 cost curve plots the direct operating costs of individual mines (which include mining, mineral processing, leaching, ocean freight, treatment and refining charges etc) against the cumulative production of copper from different copper mines. The costs shown in Figure 2.13 are net of by-product credits (e.g. gold – Au, silver – Ag and molybdenum – Mo that could be in the copper ore). The strip that is shown on the curve shows that the production costs of Mina Justa mine were approximately 75 cents per pound of copper (US$ 1,650 / tonne of copper) and the width of the strip shows the contribution of Mina Justa mine to the world's total copper output.

In summary, the graph shows that the costs of producing copper vary widely for individual mines, this is because the mines process different ore types (sulphides, oxides etc) and ore grades at different depths, and they purchase reagents, electricity and labour at different rates depending on local and site specific circumstances. Furthermore, some mines have by-products such as gold (Au), silver (Ag) and molybdenum (Mo) which are credited to the operating costs of the copper project, and this reduces the overall operating costs of copper production.

2.3.3 *The selling price of copper*

Figure 2.14 illustrates the price of copper in US$ / tonne of copper from 1995 – 2010 and the amount of copper stored by producers and customers in London Metal Exchange (LME) registered warehouses during the same period.



Cu Price

LME warehouse Cu

Figure 2.14: The price of copper, and the copper inventory stored in LME registered warehouses from 1995 – 2010 (Adapted from Infomine Inc, 2010)

This traditional graph shows that when the price of copper is low (perhaps due to low demand for copper), generally large amounts of copper will be stored in LME registered warehouses, and when the price of copper is high the amount of copper stored in LME warehouses decreases (perhaps due to high demand). When copper mines, smelters or fabricators sell copper cathodes and other copper products (e.g. rods and wires) to customers, they add a cost component known as a premium to the London Metal Exchange price that is shown in Figure 2.14. This relates to cutting, packing, transporting, warehousing and converting the copper cathodes to special products as requested by the customer (Metal prices, 2010). Premiums vary with time and the location where the copper products are required.

2.3.4 Concentrate marketing: Vertical integration and custom smelting

Not all copper mines own the downstream processing facilities (smelter, refinery and fabrication plants) required to convert their concentrates to copper cathodes and other products such as rods and wires. Such mines are known as custom mines and they supply their concentrates to custom smelters (US Gold Corporation, 2007; de Souza, 2010). The concentrates can be treated on behalf of the mine and the metals returned to the mine (toll smelting), or they can be sold to the smelter completely (outright sale), (Lewis and Streets, 1979). In the case of toll smelting, the refined metals (e.g. copper cathodes) can also be sent to any other location chosen by the mine. Mining companies that own their smelting and other downstream processing facilities are known as vertically integrated companies (Crowson, 1998; de Souza, 2010). In 2009, custom concentrate smelting accounted for about 53% of the total copper produced from smelted concentrates[20] (de Souza, 2010).

Some of the parameters that influence the decision of whether or not a mine should integrate vertically are (Crowson, 1998):

- The capital costs of constructing a mine and refinery simultaneously
- The nature of the deposit (oxide, secondary sulphide, primary sulphide)
- The capacity and life of the mine, and consequently of the smelter
- The relative cost of custom smelting and the profitability of the new smelter
- The relative locations of the mine and smelters (influences inland and sea transportation costs for shipping the concentrates).

However, according to Crowson (1998), vertical integration may be the only economic means of opening up a deposit, especially for mines that are located far inland, as is the case for mines in Zambia and DRC as well as Palabora Mining Company in South Africa and Bingham Copper Mine in the USA. Transporting their concentrates to custom smelters could be very expensive (Crowson, 1998).

The revenue[21] received by custom mines for their concentrates after smelting is calculated as follows (Wills and Napier-Munn, 2006):

$$\text{Net Smelter Return} = \text{Payment for contained metal} - (\text{Smelter charges} + \text{Transportation costs}) \quad (2.41)$$

The payment for the metal contained in the concentrate and the smelting charges deducted by a smelter are a function of the terms contained in contracts that are negotiated by individual copper mining companies and smelters. These contracts can be valid for 1 – 5 years, however negotiations may be held periodically to maintain fair conditions for both

---

[20] The balance was attributed to concentrates smelted by vertically integrated mines.

[21] The Net Smelter Return received by the mine is subject to further deductions, i.e. direct and indirect operating costs for mining and concentration before the mine determines its earnings (Wills and Napier-Munn, 2006).

parties (Lewis and Streets, 1979; Crowson, 1998; Cruz-Coke and Aravena, 2004). The terms of a typical copper concentrate contract are reviewed briefly in the following section.

2.3.5 Terms of a base metal concentrate contract: Copper

Individual smelting contracts vary for every mine and smelter, and they may be complex (Wills and Napier-Munn, 2006). Smelting contracts are normally classified as Regular (fixed term) or Spot contracts (Cruz-Coke and Aravena, 2004). Regular contracts are divided into 3 sub classes, namely, long term, medium term and short term contracts (Cruz-Coke and Aravena, 2004). The long term contracts are normally valid for 5 years or more, whereas the latter are held for under 3 years and 1 year, respectively (Cruz-Coke and Aravena, 2004). Some of the clauses in long term contracts can be renegotiated periodically to maintain fair operating conditions for all the parties involved (Lewis and Streets, 1979; Crowson, 1998; Cruz-Coke and Aravena, 2004). Mines usually desire to sell some of their concentrates under spot contracts to take advantage of favourable market conditions when they exist (Cruz-Coke and Aravena, 2004).

The terms outlined and negotiated in smelter contracts are (Lewis and Streets, 1979; Crowson, 1998; Cruz-Coke and Aravena, 2004):

- **The type and duration of the contract**
- **The material/s to be smelted**
    - This is usually a short qualitative description of the concentrate and its origin.
- **The quantity of material**
    - These are details about the quantity of materials to be sold or smelted over a given period.
- **Concentrate delivery and delivery schedule**
    - These are details about the delivery and insurance of the concentrates. They determine who pays for road and sea transportation, insurance, and the passing on of ownership of the concentrate (Title and risk). Details about the delivery schedule are also included.
- **Weighing, sampling, assaying and moisture determination**
    - This section contains details on weighing shipments, sampling concentrates and moisture determination.

- **The price of copper contained in the metal**
    - Typically, the price of copper at the London Metal Exchange, over a specified period, is used.
- **Metallurgical accountability**
    - This is an allowance given to copper smelters for metallurgical losses since they will not recover 100% of the copper and precious metals contained in the concentrates. The standard accountability is normally 96.5% for high grade concentrates (e.g. 30% Cu).
    - Smelters are compensated for processing low grade ores (e.g. lower than 30%) by allowing them to deduct a certain percentage (typically 1%) from the percentage copper content of the concentrate.
    - Similar principles as above are applied to the precious metals that could be contained in the concentrate (e.g. Au and Ag).
- **Treatment and Refining charges (TC/RCs)**
    - These are the charges deducted from the payment for the copper (and by products) contained in the concentrate. These charges are for smelting and refining the concentrate. Figure 2.15 illustrates the combined treatment and refining charges[22].

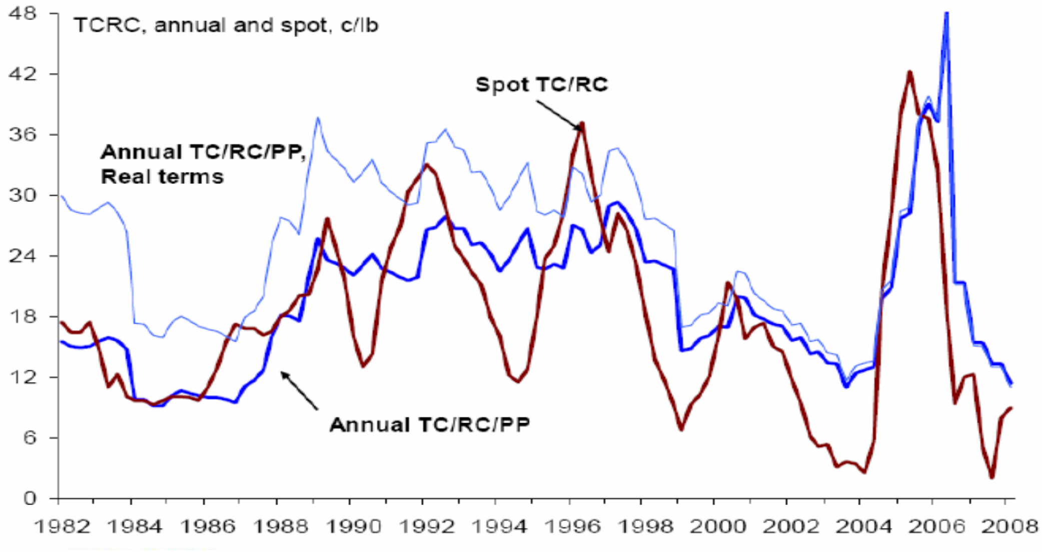


Figure 2.15: The combined treatment and refining charges from 1982 – 2008[23] (CRU International Limited as cited in Soderstrom, 2008)

---

[22] Combined treatment and refining charges are shown on the graph, however treatment and refining charges are normally quoted and shown separately in historical charts. Current charts are scarce in the public literature.

[23] PP on chart – Price participation

- **Penalties (Deductions from payment)**
    - These are penalty charges for concentrates that contain deleterious impurities such as Arsenic, Lead, Bismuth, Antimony and Zinc over specified limits.
    - Some smelters charge penalties for excess moisture (e.g. above ~ 8% moisture) in the concentrate.
- **Escalation of refining charge / Price participation (Optional)**
    - This determines the increase or decrease in the refining charges that were originally agreed upon because of an increase or decrease in the market price of copper during the contract's duration. This is often referred to as price participation (PP). An example of its application is shown below.
    - As an example, when the price of refined copper exceeds US$ 0.90 / lb Cu, the original refining charges that were required by the smelting company will increase by *10% * (New copper price - 0.90) US$ / lb Cu* (Soderstrom, 2008).
    - US$ 0.90 / lb is the historic trigger price, however this clause is currently not in practice (de Souza, 2010).
- **Payment schedule**
    - The payment schedule is set out in this part of the contract.

Other clauses negotiated in the contracts include details on taxes and duties, force majeure[24], and arbitration in the event of disputes. The contract terms outlined above are for the outright sale of a concentrate to a smelter. Similar terms and conditions apply for toll smelting; however toll smelting will include a clause for the delivery of the final products back to the mine or a specified location. Mines usually send out invitations for bids to smelters before selecting the smelter with the best offer (Lewis and Streets, 1979).

---

[24] This clause discusses conditions under which contracts can not be honoured due to unforeseen circumstances and proposes a way forward (Lewis and Streets, 1979; Crowson, 1998).

Having reviewed the main terms of a base metal smelting contract, the revenue stream of a custom smelter that receives concentrates from custom mines is reviewed briefly. Figure 2.16 illustrates a typical revenue stream of a custom smelter.

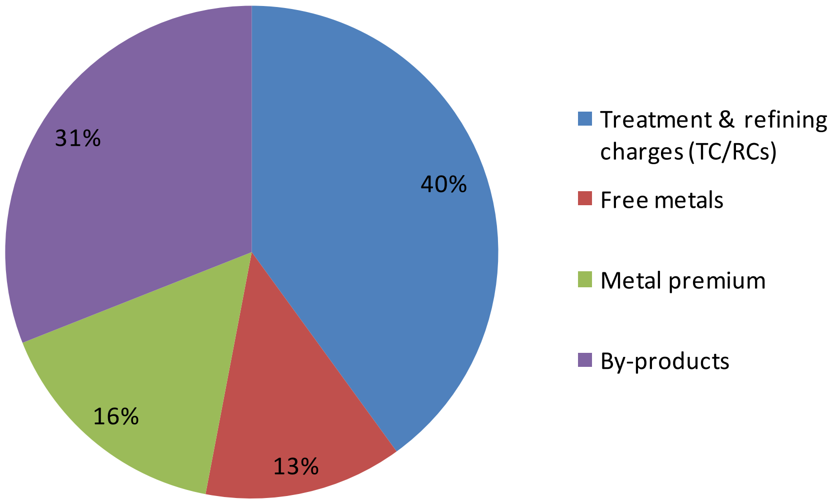


Figure 2.16: A typical revenue stream of a custom smelter, receiving concentrates from a custom mine (adapted from Soderstrom, 2008)

Treatment and refining charges refer to the fees paid by custom mines for concentrate processing. Free metals refer to the difference between the actual copper recovered from the concentrate by the custom smelters (~ 97 - 98%) and the actual copper they pay for to the mine (~ 96.5%). Similar principles apply for gold and silver that could be in the concentrate. Premiums relate to cutting, packing, transporting, warehousing and converting the copper cathodes to special products as requested by the customer (Metal prices, 2010), and by products mainly relates to sulphuric acid that is produced from the sulphur in the concentrate (Soderstrom, 2008)[25].

---

[25] Zinc and lead custom smelters have similar revenue streams, see Nyrstar (2009).

2.3.6 Project evaluations

This section briefly discusses and summarises some of the considerations that must be taken into account during mining project evaluations. Project evaluations in the mining industry are iterative and interdisciplinary in nature, requiring knowledge from the fields of geology, mining, processing, economics, environment and regulations (Hrebar and Gentry, 2003). They typically require information in the following areas:

- Information on the deposit
    - Geology, geometry, geography, exploration
- Information on general project economics
    - Markets, transportation, utilities, land, water, rights, labour, financing, government
- Mining method selection
    - Physical controls, selectivity, preproduction and production requirements
- Processing methods
    - Mineralogy, alternative processes, recoveries, plant layout
- Capital and operating estimates
    - For exploration, working capital, mining, mill, administrative etc

Project evaluations and feasibility studies typically produce information related to the costs and cashflows of a project which are then converted to profitability metrics / indicators that are used for deciding on whether or not an investment should be made. Some of the more common indicators include return on investment (ROI), payback period (PBP), discounted payback period, net present value (NPV), benefit cost ratio (B/C ratio), internal rate of return (IRR), wealth growth rate (WGR) and growth rate of return (GRR) (Smith, 2002; Hrebar and Gentry, 2003). The indicators that will be used in this project are return on investment (ROI), payback period (PBP), net present value (NPV) and internal rate of return[26] (IRR). The methodology used in the case study applied in this project is discussed in more detail in Chapter 3.

---

[26] See Smith (2002) and Hrebar and Gentry (2003) for worked examples of project evaluations, and Peters and Timmerhaus (1991), and Couper (2003) for more detailed discussions on project evaluation indicators.

2.4 Environmental impacts of copper production

The main environmental concerns associated with mining are broadly classified as:

- Energy use
- Emissions to the air (CO_2, SO_2, NOx, N_2O,CH_4 and dusts)
- Solid waste management (covering overburden, mine wastes, tailings, slag and leach residues which can cause acid rock drainage, contamination and land disturbance)
- Liquid waste management, and water management and use
- Resource consumption

These are typically the main environmental issues addressed in sustainability reports that are published annually by mining companies to communicate their progress towards sustainable development in their immediate areas of operation and on a global level (Anglo Platinum, 2009; BHP Billiton, 2009; Freeport McMoRan Copper and Gold Inc., 2010 b).

2.4.1 Energy use in mining operations

The major link between the environment and energy consumption is the release of CO_2 and other greenhouse gases[27] such as CH_4 and N_2O, which are released to the atmosphere when fossil fuels are burned to generate energy (IPCC, 2006). An analysis of recent sustainability reports, published by major mining companies, has shown that the main energy sources for their operations, in descending order, are electricity, liquid fuels, natural gas and coal (Anglo platinum, 2009; BHP Billiton, 2009; Freeport-McMoRan Copper & Gold Inc., 2010 b). This is illustrated for BHP Billiton in Figure 2.17.

---

[27] Greenhouse gases contribute to global warming.

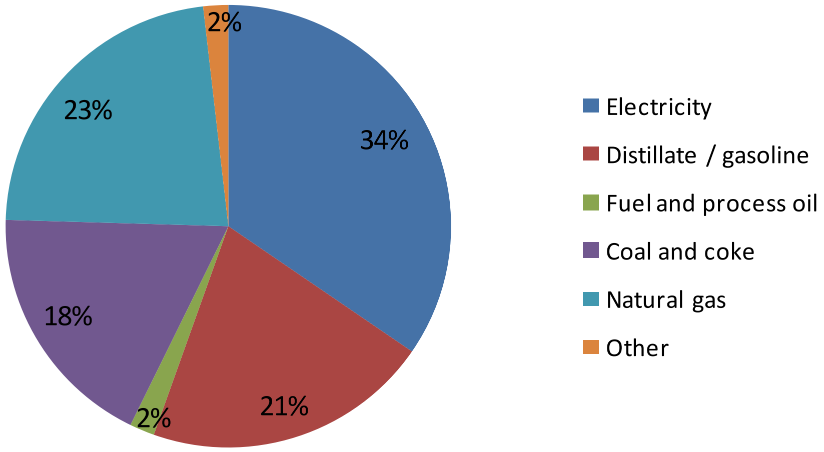


Figure 2.17: BHP Billiton energy use by source / type (adapted from BHP Billiton, 2009)

Figure 2.17 shows BHP Billiton's aggregated energy sources for all their mining operations, which include copper, coal, aluminium, iron ore, lead, uranium etc. An analysis of sustainability reports shows that electricity is a major energy source in the mining industry; hence the gaseous emissions associated with its generation should be included in environmental studies.

Energy consumption of pyrometallurgical extraction

Table 2.5 shows energy consumption data compiled from different sources for the pyrometallurgical extraction of copper. The data represents the total energy consumption in the mine, concentrator, smelter and refinery.

Table 2.5: Energy consumption for pyrometallurgical copper production

Study	Ore grade % Cu	Energy MJ / kg Cu
USCOTA (1988)	0.55	77 – 100
BCS Incorporated (2002)	0.55	85 – 94
Marsden[28] (2008)	0.5	57
Giurco et al. (2001)	1.4	44 and 59
Norgate (2001)	N/A	45
Marsden (2008)	1.5	30

---

[28] Shown here excluding transportation and power plant energy inefficiency, as it was not known whether the other studies included those energy components.

Table 2.5 highlights a number of important issues regarding energy consumption in the copper industry. Firstly, there is no single correct value for energy consumption as it will vary for individual operations due to a number of factors which include ore grade, deposit characteristics, the mining method and operational efficiency. Secondly, a comparison of the data for the grades shown in Table 2.5 indicates that for each grade (i.e. ~ 0.5 Cu and ~ 1.5 % Cu) the energy consumption decreased over time, probably because of progress in energy efficiency in the copper industry.

Figure 2.18 shows a typical distribution of energy requirements for the production of copper via the pyrometallurgical processing route for an ore with an initial grade of 0.55% Cu.

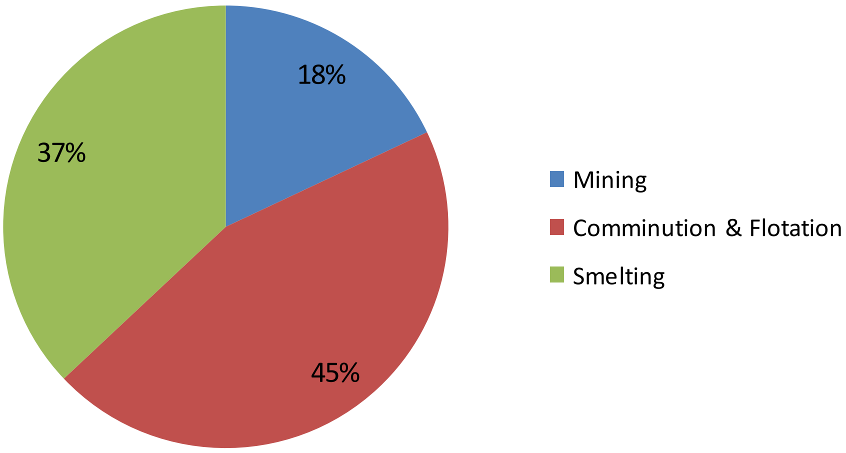


Figure 2.18: Energy distribution of the pyrometallurgical route of copper production[29] (adapted from BCS Incorporated data, 2002. See footnote for USCOTA (1988) data)

The distribution shown in Figure 2.18 is for the pyrometallurgical processing of copper concentrates and is used here for illustrative purposes. Similarly to the total energy consumption data, this distribution will vary with ore body characteristics and the technologies used along the production chain and therefore it is company and site specific. The mining subsystem consumes about 18% of the total energy requirements of the pyrometallurgical processing route, and hauling by trucks consumes about 54% of this energy[30] (BCS Incorporated, 2002). Comminution and concentration consume approximately 45% (BCS Incorporated, 2002; USCOTA, 1988[31,32]) of the energy

---

[29] Initial ore grade 0.5% Cu

[30] The mining subsystem refers to the operations in the open pit mine and excludes milling and subsequent operations. Therefore hauling consumes ~ 54% * 18% = 9.7% of the total energy required for copper production. (18% is from Figure 2.18)

[31] In this report the energy distribution of copper production is; Mining ~ 19 – 25%, Milling and flotation ~ 40 – 52%, smelting and refining ~ 23 - 42%.

requirements of copper production. Comminution is an inefficient and energy intensive operation. It is estimated that only 1% of the energy input into a ball mill is used for mineral liberation (Hinde, 2009; Wills and Napier-Munn, 2006). The rest is lost to the breakage of barren rocks and in the production of heat and noise. Smelting and refining consume about 37% of the total energy requirements required for copper production. For further information on the assumptions made and configurations used in the compilation of this data, the reader is referred to the studies in Table 2.5.

Energy consumption of the Pressure Oxidation process

There is currently little publicly available data for the Pressure Oxidation process and all the other hydro/biohydrometallurgical concentrate treatment processes. However, an analysis of Marsden's (2008) data shows that the energy consumption for producing copper through the POX process is approximately 27 MJ / kg for an initial ore grade of 1.5% Cu and 53 MJ / kg for an initial ore grade of 0.5% Cu. This data includes the energy consumption in the mine, concentrator, and in the oxygen, solvent extraction and electrowinning plants, therefore it can be compared directly to the data in Table 2.5 which is for copper production through the pyrometallurgical (smelting) route. A comparison of the POX and smelting data shows that POX would result in slightly lower (total) energy consumption compared to smelting. However, the analysis shown in this chapter excludes the energy required for transporting the concentrates to offshore smelters and energy lost during the generation and distribution of electricity, thereby allowing for a direct comparison of energy consumption due to production only. The contributions of transportation and electricity generation to energy consumption are explored in Chapter 4.

A review of Norgate's (2001) study showed that the energy consumption for the POX process was 70 MJ / kg of copper, which is considerably higher than Marsden's (2008) values. However, Norgate's (2001) study included precious metals recovery from the leach residues through a lime boil and cyanidation, whereas Marsden's did not. Furthermore, the initial ore grade was not mentioned in Norgate's (2001) study, and differences in initial ore grades could contribute to the variation in the published energy consumption data. Figure 2.19 shows the energy consumption distribution of the POX process.

---

[32] It is not stated in these reports whether or not energy losses associated with electricity generation were taken into account.

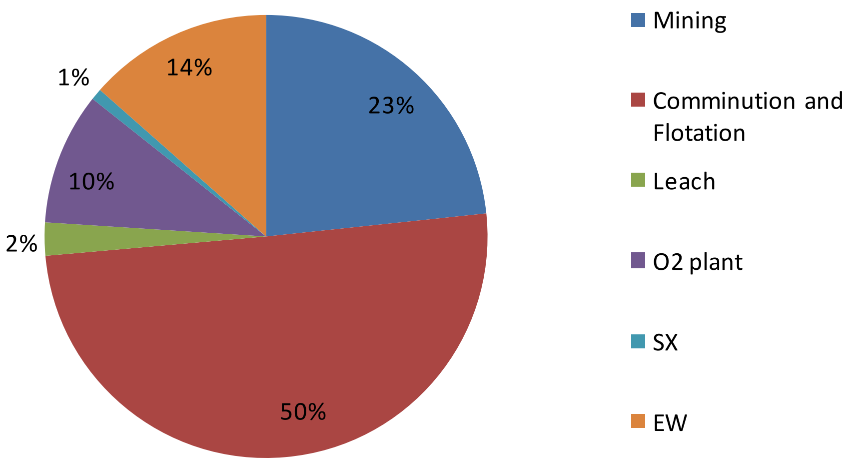


Figure 2.19: Energy distribution of the hydrometallurgical route of copper production[33] (adapted from Marsden, 2008)

Figure 2.19 shows that for the POX process (excluding the mine and concentrator), the actual leach plant (i.e. autoclaves, thickeners, flash) does not consume much energy. Energy, and more specifically, electricity consumption is high in the oxygen and electrowinning plants.

Energy consumption of the heap / ROM leaching processes

Table 2.6 shows available energy data on heap and ROM leaching operations.

Table 2.6: Energy consumption for heap and ROM leaching

Study	**Ore grade % Cu**	**Energy MJ / kg Cu Heap**	**Energy MJ / kg Cu ROM**
Giurco (2001)	0.7	82	N/A
Marsden (2008)	0.5	34	28
Marsden (2008)	0.75	27	23

A comparison of Giurco's (2001) and Marsden's (2008) data for heap leaching an ore with an initial grade of about 0.7%, similarly to the pyrometallurgy data, shows that energy consumption decreased with time. However, these observations may not be universal, and they highlight major variations in publicly available data, indicating that consideration must be given to the sources and quality of data used in environmental performance studies (Stewart and Petrie, 2004).

---

[33] Initial ore grade 0.5% Cu

ROM leaching consumes less energy than heap leaching (Table 2.6). In heap leaching, ores are normally crushed before they are leached, whereas in ROM leaching the ores are leached directly without crushing, thereby reducing total energy consumption. However, the energy advantage of ROM leaching is offset by low copper recoveries[34].

Figure 2.20 and Figure 2.21 display the energy distributions of the heap and ROM leaching processes

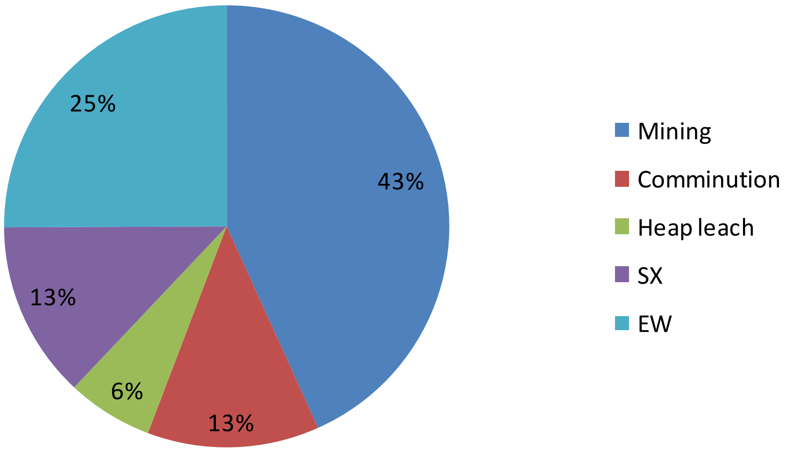


Figure 2.20: Energy distribution of copper production through heap leaching (adapted from Marsden, 2008)

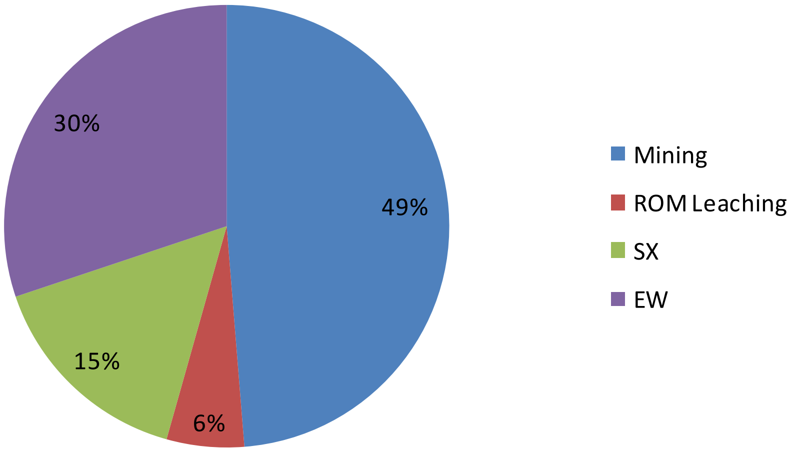


Figure 2.21: Energy distribution for copper production through ROM leaching (adapted from Marsden, 2008)

---

[34] Typical copper recoveries for heap and ROM leaching are ~ 70% and 40%, respectively. See Figure 2.11, Marsden (2008) and BHP Billiton (2004).

For heap and ROM leaching processes, similarly to the POX process energy distribution (Figure 2.19), electrowinning is a major energy consuming subsystem, likewise, the leaching sub-system on its own does not consume much energy.

The applicability of the majority of the published energy sources is currently limited as most of the data is now either outdated or may not be reflective of the current technological advances within the copper industry (Marsden, 2008). Furthermore, an analysis of some of the previous work on energy efficiency in the copper industry has revealed that some of the inventory energy data used in the studies is repeated from older publications. In the present study, Marsden's (2008) energy model was used since it was the most comprehensive and current energy data source available. The energy consumption data used to develop the energy model was obtained from real industrial plants.

These plants treat oxide, and primary and secondary sulphide copper ores depending on the nature of the underlying deposit. Copper concentrates are produced at Bagdad, Sieritta, Chino, Candelaria and Cerro Verde. The concentrates are transported to a smelter / refinery by truck or ship, depending on the location of the mine and smelter. POX, solvent extraction and electrowinning technology, was at the time of publication, practised on a portion of the concentrates produced at Bagdad and on the whole concentrate produced at Morenci to produce copper cathodes onsite. Heap leaching, solvent extraction and electrowinning technology is practised at Morenci and Cerro Verde, whereas ROM leaching, solvent extraction and electrowinning of low grade ores, as a process route, is practised at Bagdad, Morenci, Chino and Tyrone. For more details on the model the reader is referred to Marsden (2008).

2.4.2 Life cycle assessment

This section briefly introduces the concept of Life Cycle Assessment, since it is a commonly used environmental analysis tool in industry which has also been applied at varying levels in previous studies pertaining to the production of copper (Giurco, 2001; Norgate, 2001; Guma, 2010). Also, aspects of the LCA methodology will be used in this study, to quantitatively determine the effect of the gaseous emissions from the processes on the environment.

Life cycle assessment (LCA) is an environmental management and decision support tool which incorporates life cycle thinking into environmental analyses (Crosby-Forbes, 1999; Azapagic, 2005):

Life cycle assessments consist of four stages:

1. Goal and scope definition

   In this stage the purpose of the study, its system boundaries and its functional unit(s) are defined. Consideration is also given to the quality of data used, the assumptions made in the study as well as its limitations. LCA's are usually classified as either ‘cradle to grave’ or ‘cradle to gate’, depending on the chosen system boundaries. In ‘cradle to grave’ studies, the system boundaries include all the stages involved in the life cycle of a product / activity, i.e. extraction of raw materials, manufacturing, transportation, use, reuse, maintenance, recycling and final disposal (see Figure 2.22).

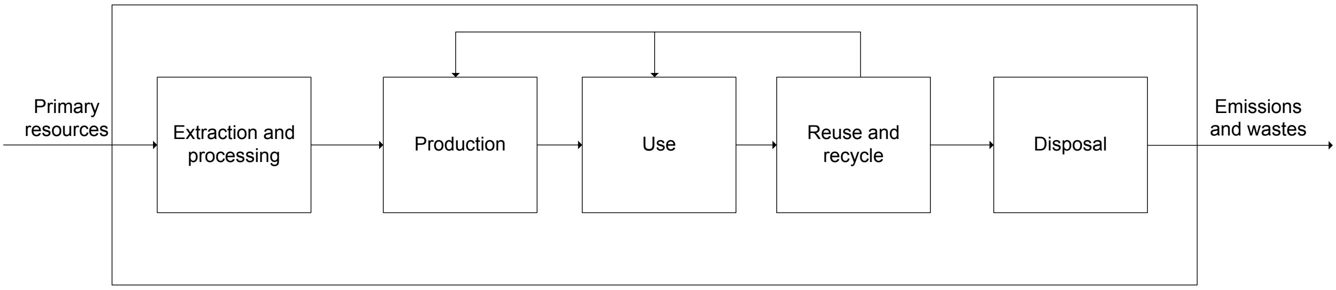


Figure 2.22: Life cycle stages and cradle to grave boundary (adapted from Azapagic, 2005)

The system boundary of a ‘cradle to gate’ LCA includes the extraction of raw materials and the production and manufacture stages of a product. The distribution, use, reuse, maintenance, recycling and final disposal of the product are excluded.

2. Inventory analysis

This is the data collection / gathering stage of the LCA methodology. The first step in this stage is to draw a flowsheet describing the system under analysis. Thereafter, input inventory data into the system and output inventory data from the system are compiled, typically through some form of mass and / or energy balance. Data in the following groups is required:

- Primary fuels and other services such as water and compressed air
- Raw materials
- Deposits to land
- Emissions to air
- Emissions to water

Following this step, environmental burdens (e.g. CO_2, SO_2 emissions) can be quantified for the chosen functional unit (e.g. kg of copper produced). If there is more than one economic product or functional unit, allocation of burdens to each product / co-product may be necessary.

3. Impact assessment

This stage involves aggregating the environmental burdens (e.g. CO_2, SO_2, $CH_{4,}$ Cu, Fe, cyanide emissions), from the inventory analysis stage, into a small number of environmental impact categories that the burdens contribute towards. This is known as the problem oriented approach[35]. The main impact categories are non-renewable resource depletion, global warming potential, acidification potential, eutrophication, human toxicity, aquatic toxicity, ozone depletion and photochemical smog). Energy use and water consumption are sometimes included.

The impact assessment stage of the LCA methodology is performed in four steps. The first step, ***classification***, involves identifying the environmental problem of concern (e.g. global warming potential) and allocating the environmental burdens to the chosen environmental problems. The second step, ***characterisation***, involves quantification of the environmental problem by calculating the contribution of each burden to a particular environmental problem, relative to a particular substance. For example, global warming potential is expressed in units of ***kg of CO_2 equivalent*** by using the following equivalency factors:

- CO_2 – 1
- CH_4 – 21
- N_2O – 310
- HFCs[36] – 400
- Volatile organic compounds – 11

In the third step, ***normalisation***, the impacts (e.g. global warming potential) can be normalised relative to annual global, regional and industry averages, which can help determine the effect of the process on a certain area and in a specified time frame. The last step of the impact assessment stage of the LCA methodology, ***valuation***, involves assigning a weight to each impact category according to its perceived importance, and aggregating the results to a single indicator. Weighting factors can be assigned based on monetary terms and social preferences by stakeholders, experts and the public. Valuation is subjective, and currently, there is no single, widely accepted method for its application.

4. Interpretation

This is the final step of the LCA methodology and it involves identifying the areas in the system that have large contributions towards the environmental impacts under consideration, with the aim of improving the profile of the system. Thereafter, a sensitivity analysis is performed to quantify the effects of the uncertainties that are associated with the assumptions made and the data used during the study. For a more detailed discussion on the LCA methodology the reader is referred to Crosby – Forbes (1999) and Azapagic (2005).

---

[35] In this study the problem oriented approach will be taken, see Chapter 3.

[36] Hydrofluorocarbons

2.4.3 Emissions to the air

A review of the literature has identified a number of studies (Table 2.7 and Table 2.8) in which the environmental impacts of the different copper extraction routes were compared by using the Life Cycle Assessment methodology. Common environmental impact categories used in the studies included Global Warming Potential (GWP), Ozone Depletion (OD), Acidification Potential (AP), Eutrophication Potential (EP), Human Toxicity (HT), Eco-Toxicity (ET) and Smog. Summaries of notable studies are shown in Table 2 7 and Table 2.8, below.

Norgate (2001) compared the GWP and AP of the alternative technologies available for copper concentrate treatment and therefore excluded heap leaching and reverberatory smelting, which is almost obsolete. His study was performed on copper concentrates that contained precious metals (Au and Ag), therefore the environmental performance results were presented inclusive of the contributions of precious metal recovery operations (i.e. a lime boil and cyanidation of leach residues). It was shown that flash smelting had the lowest global warming potential and the highest acidification potential of all the options considered. POX had the highest global warming potential and a relatively high acidification potential. Norgate (2001) considered concentrate marketing and supply chains (custom and integrated smelting), but, only through the inclusion of transportation; a country specific analysis was not performed.

Guirco et al. (2001) compared the GWP, AP, ET and smog for reverberatory smelting, flash smelting and heap leaching. Their study can not be summarised in detail here, but, an analysis of the various scenarios they simulated seems to suggest that, generally, flash smelting had the lowest global warming potential and the highest acidification potential. Heap leaching had the lowest acidification potential. They considered the effect of using various fuels in the electricity mix through a country specific analysis; however, it was assumed that all the processes were integrated. Therefore, the effect of the differences between the electricity mixes of the concentrate's countries of origin and treatment were not analysed, nor was the effect of concentrate transportation on the environment.

Ayres et al. (2002) reported on the life cycle of copper using a life cycle type of methodology, but did not use the life cycle assessment impact categories. However, they discussed the pertinent environmental issues associated with copper, from its production to its disposal and recycling. Other notable studies with similar themes but different scopes include work by Alvarado et al. (2002), Pimentel (2009), and Kuckshinrichs et al. (2007) (see Table 1.2 for a brief review of the scopes of their studies).

None of the aforementioned environmental studies considered the economic aspects of copper production, which are a fundamental component of sustainable development; hence this thesis aims to address this shortcoming by developing a methodology for assessing multiple flowsheet options, taking into account various techno-economic and environmental criteria at the same time. Guma's (2010) study, which included the economic aspects of copper production, is reviewed within the context of Eco-efficiency in section 2.5.

Table 2.7: Selected studies on copper production from concentrates

Study	**Norgate (2001)**	**Giurco et al. (2001)**	**Giurco et al. (2001)**
	Assumptions for concentrate processes		
Ore grade (%)	Not available	0.5	1.4
Cu concentrator recovery (%)	93.7	Not available	Not available
Electricity generation mix	Coal	Coal	Chile mix[37]
Concentrate grade (% Cu)	25	Not available	Not available
Concentrate supply chain	Yes	No	No
Transportation	Yes	Not applicable	Not applicable
Includes precious metal recovery	Yes	No	No
	Results		
	Reverberatory smelting		
Countries	Not applicable	Generic	Chile (Integrated)
SO_2 capture (%)	Not applicable	5	5
GWP (kg CO_2 e / kg Cu)	Not applicable	15	4.3
AP (kg SO_2 e / kg Cu)	Not applicable	2.5	2.5
	Flash smelting		
Countries	Generic	Generic	Chile (Intergated)
SO_2 capture (%)	99	93	93
GWP (kg CO_2 e / kg Cu)	4.3	14.8	3.3
AP (kg SO_2 e / kg Cu)	0.058	0.2	~ 0.16
	POX		
Countries	Generic	Not applicable	Not applicable
SO_2 capture (%)	Not applicable	Not applicable	Not applicable
GWP (kg CO_2 e / kg Cu)	9	Not applicable	Not applicable
AP (kg SO_2 e / kg Cu)	0.046	Not applicable	Not applicable

---

[37] Assumed Chile's electricity generation mix consisted of 30% coal and 70% hydroelectricity.

Table 2.8: Selected studies on heap leaching copper ores

Study	Norgate (2001)	Giurco et al. (2001)	Giurco et al. (2001)
Ore grade	Not applicable	0.5	0.7
Electricity generation mix	Not applicable	Coal	Chile mix
	Results		
	Heap leaching		
Country	Not applicable	Generic	Chile
GWP (kg CO_2 e / kg Cu)	Not applicable	19.8	6.2
AP (kg SO_2 e / kg Cu)	Not applicable	0.1	~ 0.04

2.4.4 Solid waste management

Solid waste management is becoming an increasingly important economic, environmental and social issue in the mining industry, mainly due to increased pressure from regulators on mining companies to carry out their operations in a manner that is consistent with the broad principles of sustainable development. Some of the major environmental concerns that must be taken into account in mining projects regarding solid waste management include:

- *The physical footprint of mining and land disturbance (Affects the ecosystem)*

  The physical footprint created by the extraction and disposal of solids from mining and mineral processing activities is very large and may reduce the productivity of land that was previously productive as wildlife habitat and farmland, thereby affecting local ecosystems (IIED, 2002).

- *The physical stability of the disposal sites*

  There are documented cases in which tailings dams have failed due to poor design and natural causes (Rico et al., 2007). Some of these failures have even led to human death (IIED, 2002, Rico et al., 2007).

- *Leachate generation and the characteristics of the solid waste*

  When water comes into contact with mining wastes, leachates which may have high metal concentrations and extreme pH[38] values are generated. These leachates may come into contact with, and contaminate soil and water sources, leading to ecotoxicity and human toxicity effects (Broadhurst et al., 2007; Hansen, 2004).

---

[38] Wastes containing sulphide minerals typically generate acidic leachates (Acid mine drainage) and slag typically produces alkaline leachates.

Acid mine drainage (acidic leachate generation) remains, according to the IIED (2002), the “most serious and pervasive environmental problem related to mining” which could occur for hundreds of years after mining activities cease (IIED, 2002).

Acid mine drainage

Acid mine drainage occurs when sulphide containing minerals, such as pyrite and pyrrhotite, are oxidised by oxygen in air and water in the presence of catalytic bacteria (Hesketh, 2010; Akcil and Kolders, 2006). The oxidation of the sulphide minerals results in the formation of an acidic leachate and the release of potentially toxic metals. Representative reactions are shown for the case of pyrite (FeS_2), which is generally known as the main source of acid mine drainage (Hesketh, 2010):

$$\mathrm{FeS_2} + {}^{7}/_{2}\ \mathrm{O_2} + \mathrm{H_2O} \rightarrow \mathrm{Fe^{2+}} + 2\,(\mathrm{SO_4})^{2-} + 2\,\mathrm{H^+} \tag{2.42}$$

$$FeS_2 + 7\,Fe_2(SO_4)_3 + 8\,H_2O \rightarrow 15\,FeSO_4 + 8\,H_2SO_4 \tag{2.43}$$

$$Fe^{2+} + {}^{1}/_{4}\ O_2 + H^+ \rightarrow Fe^{3+} + {}^{1}/_{2}\ H_2O \tag{2.44}$$

$$Fe^{3+} + 3\,H_2O \rightarrow Fe(OH)_3 + 3\,H^+ \tag{2.45}$$

Waste rocks from mines, tailings from flotation and spent ore from heap and dump leaching are known sources of acid mine drainage. Open pits and voids from underground mines are also known sources of acid mine drainage. Typically, the residues generated from pyrometallurgically processing copper concentrates (smelting) generate leachates that are alkaline in nature, whereas the residues produced from the hydrometallurgical processing of copper concentrates (e.g. POX) contain elemental sulphur (Table 2.3), and / or sulphates in stable forms such as gypsum and have not been shown to be sources of acid mine drainage.

The effects of acid mine drainage

The acidic conditions that are created in a solid waste deposit enhance the dissolution of toxic elements from stable minerals within the deposit, and water flow through the deposit (e.g. from precipitation) generates an acidic leachate that promotes the transportation and spread of acid and toxic elements into the local soil and ground water systems, eventually leading to ecotoxicity and human toxicity (Hansen, 2004; Hesketh, 2010). Methods of preventing and treating acid mine drainage are not reviewed in the current study, however, the interested reader is referred to Todd and Reddick (1997), Ford (2003), Akcil and Kolders (2006), Zipper et al. (2009) and Hesketh (2010).

2.4.5 Water management

About 97.5% of the Earth's water is salty (sea and brackish water) and 2.5% is fresh water of which about two thirds is frozen, leaving a small fraction of water available to meet human needs (Broadhurst, 2010; Lange, 2010). While on a national and global level the mining industry may not be a large water consuming industry, its water withdrawals could pose a threat to water availability at local and regional levels (Broadhurst, 2010). Furthermore, mining activities have the ability to reduce fresh water quality and availability through leachate generation and the subsequent contamination of water sources as described in the solid waste management discussion. Water management is therefore a major issue in the mining industry, as mismanagement could have an adverse impact on the accessibility of clean water for the people and animals living in the region where the operations are being performed.

Water management strategies and controls are not generic but are site specific and are influenced by the local climate, hydrogeology and topography (Hansen, 2004). Table 2.9 and Table 2.10 show some of the possible sources and sinks for water related to mining operations (both surface and underground mines).

Table 2.9: Primary and worked water sources for the minerals industry (Commonwealth of Australia, 2008)

Raw / Primary water source	Worked (Recycled / Reused) sources
Surface water sources (rivers, lakes, dams and clean runoff)	**Direct:** Tailings thickener overflow
Ground water (surficial and fractured rock aquifers)	Concentrator thickener overflow
Seawater (with desalination common)	Filtration plant filtrate
Rain interception	Site stormwater runoff – roads
Town water utility supply	**Indirect:** Tailings dam decant water
Treated sewage	Irrigation return
Mine dewatering	Acid mine drainage
	Stockpile/waste rock dump run-off and seepage
	Effluent from industrial waste water treatment plant

Table 2.10: Water outputs in the minerals industry (Commonwealth of Australia, 2008)

Water output	Output source / pathway
Evaporation	Evaporation facilities, stores, channels, co-disposal and tailings facilities, dust suppression, ventilation
Seepage	Stores, channels, co-disposal and tailings facilities, underground losses
Leaks	Pumps and pipes
Re-injection	Bore to aquifer, dewatering
Release/Discharge	Stores, dewatering, runoff
Entrainment	Product, tailings, leach residues
Transfer to third party	Pipe, channel, dewatering

The integration and management of these water sources and sinks cannot be summarised in this thesis, however this topic is reviewed in more detail in Commonwealth of Australia (2008). Norgate and Lovel (2004) diagrammatically illustrate the link between some of the water sources and sinks outlined in Table 2.9 and Table 2.10, for a mine (Figure 2.23).

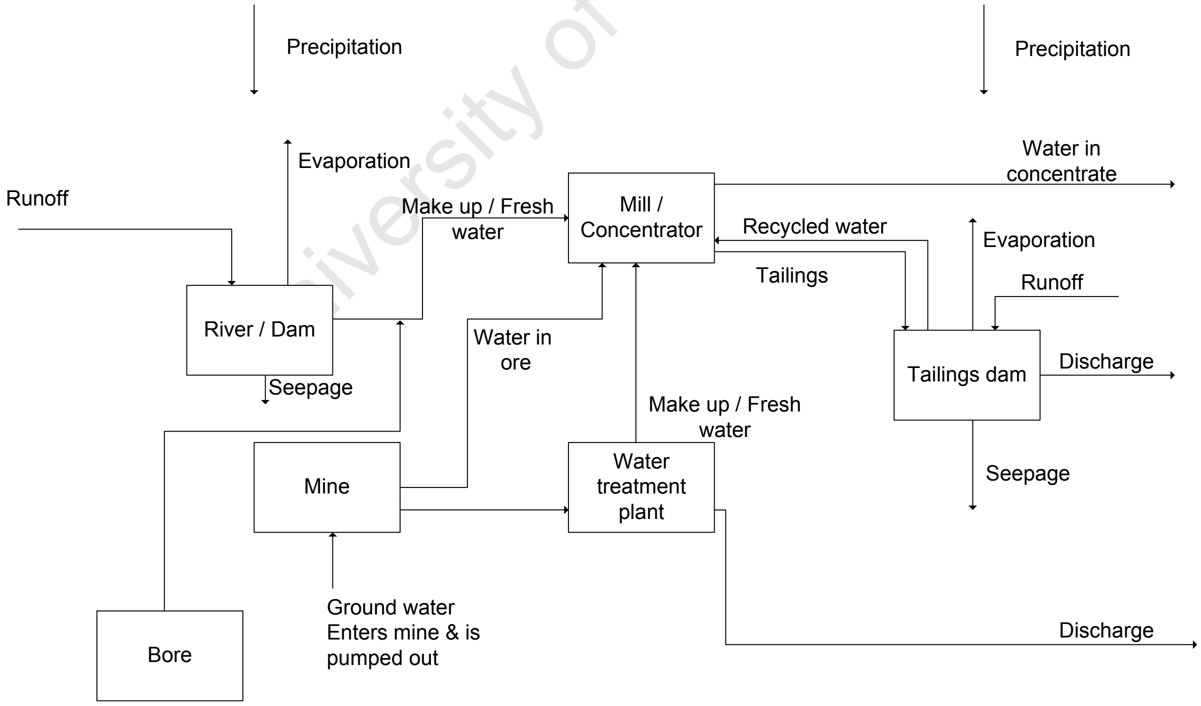


Figure 2.23: Water flows in a mine and concentrator (adapted from Norgate and Lovel, 2004)

According to Davenport et al. (2002), most concentrators do not discharge water, and therefore minimise pollution and fresh water consumption through recycling and reuse.

A review of the literature, for example, Davenport et al. (2002), Mwale et al. (2005), and Wels and Robertson (2003), suggests that the bulk of the water losses and water recoveries in mining operations are associated with tailings disposal. Water losses at the tailings disposal site are attributed to evaporation, seepage (rewetting) and entrapment in the solid pores. In arid regions (e.g. Chile, which is the subject of the case study in Chapter 4), the only significant source of water entering the tailings dam is the water in the tailings stream and not water associated with precipitation.

Data presented by Mwale et al. (2005) for 3 Chilean copper mines showed that for water recycle in the range of 20 – 58%, the copper mines required 0.35 – 0.65 tonnes of fresh water per tonne of ore processed, and data compiled by Brantes (2009) showed that in 2006 Chilean copper concentrators required 0.3 – 2.1 tonnes of fresh water per tonne of ore processed. The average water consumption for that year (2006) was 0.79 tonnes of water per tonne of ore processed. The average smelter water consumption was a further 3.6 tonnes / tonne of copper product. Information concerning the water requirements of the POX process and the other new hydrometallurgical processes is scarce. In 2006, the water consumption for Chilean heap leaching operations varied from 0.08 – 0.25 tonnes / tonne of copper produced (Brantes, 2009).

2.5 Eco-efficiency

The previous sections in this chapter have reviewed the technical, economic and environmental aspects of copper production. In this thesis, eco-efficiency indicators will be used to consolidate the techno-economic and environmental aspects of copper production with the aim of enabling sound business decision making.

According to the World Business Council for Sustainable Development (UNCTAD, 2004), “Eco-efficiency is reached by the delivery of competitively priced goods and services that satisfy human needs and bring quality of life, while progressively reducing ecological impacts and resource intensity.” The council further states that an eco-efficient state occurs when economic activities can be sustained by the “Earth’s estimated carrying capacity” (UNCTAD, 2004).

Quantitatively eco-efficiency indicators are normally defined as (UNCTAD, 2004; UNESCAP, 2009; Guma, 2010):

$$Eco-efficiency = \frac{Financial\ performance}{Environmental\ performance} \tag{2.46}$$

Eco-efficiency indicators can also be depicted graphically as shown in Figure 2.24 and Figure 2.25.

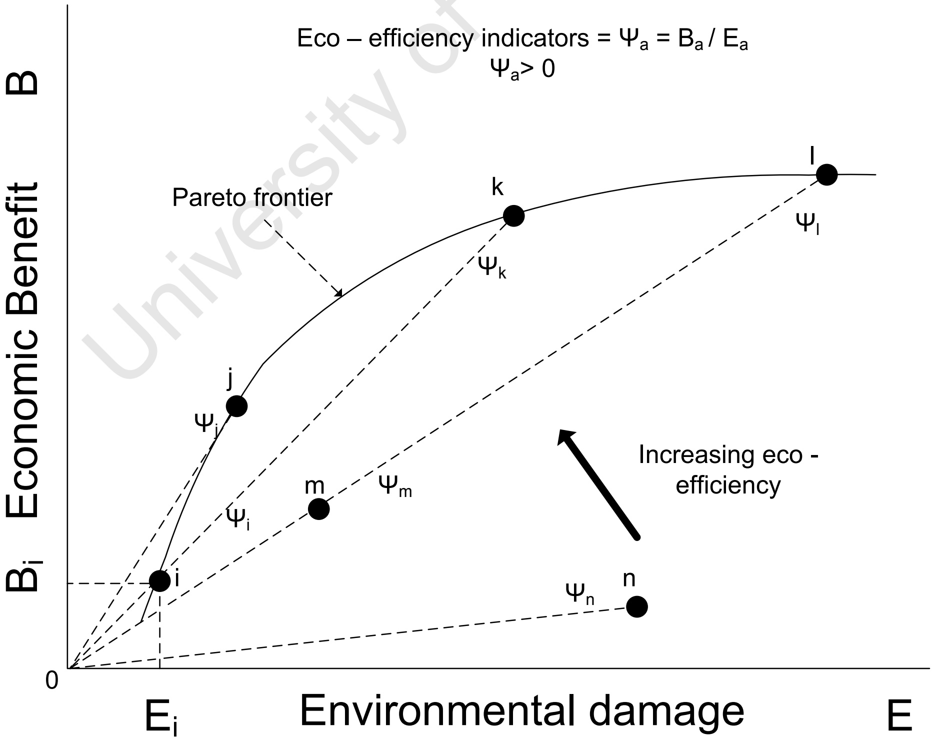


Figure 2.24: Graphical depiction of eco-efficiency indicators when positive economic indicators are expected e.g. profit (adapted from Guma, 2010)

Points **i,j,k,l,m** and **n** are alternative designs that are under consideration in the decision making scenario. The eco-efficiency indicator (Ψ) for each alternative is calculated as follows (Guma, 2010):

$$\Psi_a = {}^{B_a}/_{E_a} = Slope\ of\ line\ joining\ origin\ to\ point\ a;\ where\ a =\ alternatives\ i, j \quad ....(2.47)$$

Process alternatives **i,j,k** and **l** are known as Pareto optimal alternatives and they represent a set of superior process alternatives with regards to the selected environmental and economic criteria e.g. (global warming potential and profit) (Guma, 2010). They form a locus known as the Pareto frontier or Pareto set. Process alternatives **m** and **n** are Pareto sub-optimal alternatives meaning they are inferior options. The numeric eco-efficiency (Ψ_a) of a particular process option, under consideration, can be represented by the slope of the line joining the origin to the coordinates that map the economic and environmental indicators of that option. The highest eco-efficiency is achieved when the ***ratio*** of economic benefit to environmental damage is at a maximum, and on the decision space shown in Figure 2.24, this occurs at the intersection of the Pareto frontier and its tangent line (drawn from the origin). Eco-efficiency increases as the slope of the line joining the mapped process alternatives to the origin increases, i.e. from process option **n** to **j**. On Figure 2.24, alternative **j** is the most eco-efficient option of all the alternatives under consideration.

Figure 2.25 demonstrates how eco-efficiency indicators are mapped when the economic indicators associated with various process alternatives are negative (e.g. when considering losses) (Guma, 2010).

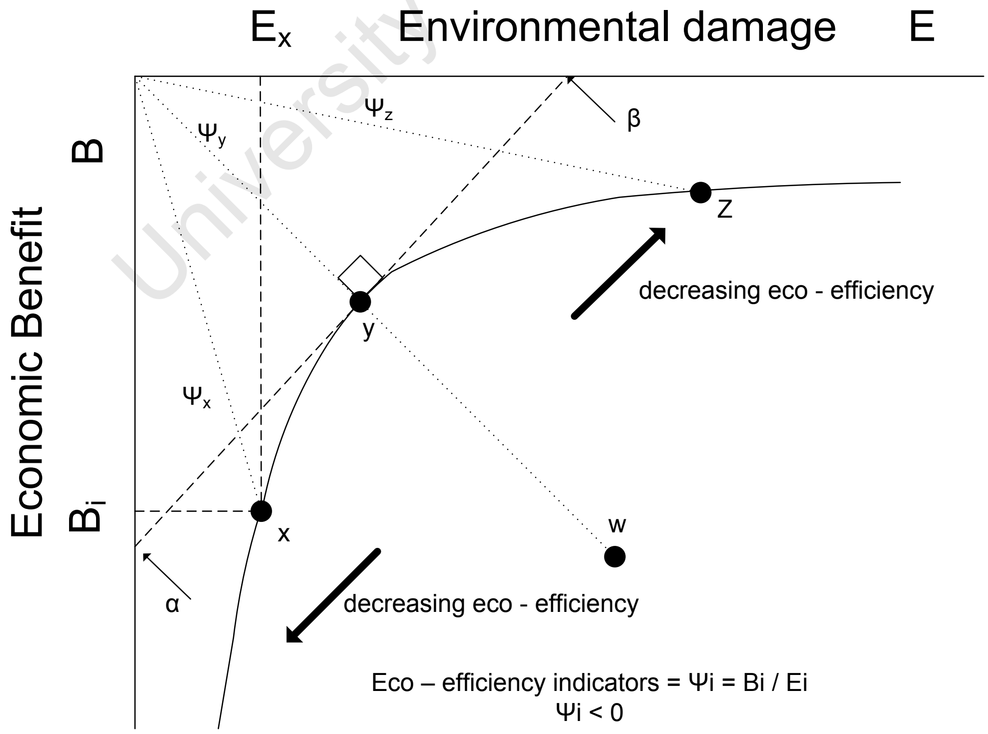


Figure 2.25: Graphical depiction of eco-efficiency indicators where economic indicators are negative e.g. losses (adapted from Guma, 2010)

Since a low economic loss and low environmental damage are required, the alternatives that lie close to the origin are desirable. In this instance (where there is a negative economic indicator) it is the distance of the lines drawn from the origin to the mapped coordinates which determines the level of eco-efficiency, not the slope of the lines (Guma, 2010). Alternatives that have coordinates that are mapped near the origin (i.e. short distance from the origin) are desirable and are more eco-efficient. As such process alternative ‘y’ on Figure 2.25 is the most eco-efficient option since it has the shortest distance from the origin.

Eco-efficiency indicators can be used as a guide in moving towards a sustainable industry, although, they must not overshadow the value of the information given by the original economic and environmental data to avoid, for example, a situation where emission limits set by governing authorities are overlooked because a certain option has the highest eco-efficiency indicator (Guma, 2010).

The objectives of using eco-efficiency indicators are to (WBCSD, 2000):

- Reduce resource consumption

  This includes energy, materials, water and land. Recycling and improving product durability are also encouraged

- Reduce the impact of human activities on nature

  This can be achieved by reducing air emissions, water discharges, wastes and the dispersion of toxic substances

- Increase product or service value

  This involves improving the product’s functionality, flexibility and modularity, thereby fulfilling the customer’s functional needs with fewer resources.

The five environmental categories for which reporting frameworks were created by the United Nations are water use, energy use, global warming, ozone depletion and waste. However, these categories may be extended by individual companies to include categories that address specific industry and regional needs (UNCTAD, 2004). In Guma (2010), eco-efficiency indicators were applied to two case studies. The first case study involved the selection of a process route for producing copper and the second case study involved technology selection for tailings dewatering. In summary, Guma (2010) found that eco-efficiency indicators were useful for guiding decision making in the minerals industry; however he demonstrated that caution has to be taken in their use and interpretation depending on the decision making context. In this study numeric eco-efficiency indicators will be used to link and assess the techno-economic and environmental results that are generated from the model developed in this project. However, the overarching aim of this current study is not to develop or test eco-efficiency indicators but to:

- Develop a methodology for generating and assessing multiple flowsheet options that take into account techno-economic and environmental criteria, using data that is consistent with early stage, design requirements in the copper processing industry.

For a more detailed discussion on the development and use of eco-efficiency indicators, the reader is referred to Guma (2010).

Summary

The literature review has shown that techno-economic and environmental aspects of copper production are usually considered in isolation, especially during the decision making stages of process selection and design. However, at the same time, the literature review highlighted that environmental and economic performances of copper production processes are interlinked, and are normally influenced by technical choices. With the move towards a sustainable industry, there is a need to integrate techno-economic, environmental and social aspects in all activities related to copper production. Chapter 3 proposes a methodology that can be used to perform studies and generate information, simultaneously, pertaining to the techno-economic and environmental performances of copper production. The methodology involves the development of a performance model which is applied on a case study that is presented in Chapter 4.

3 METHODOLOGY

It was shown in Chapters 1 and 2 that previous studies that compared the environmental and techno-economic performances of various copper production methods had considered the environmental and techno-economic aspects in isolation. However a review of the literature has demonstrated that these aspects are interlinked and depend on a number of parameters which include:

1. The mineralogy and the grades of the ores
2. The operations taking place on or near the mine site (e.g. milling, smelting, heap leaching, ROM leaching etc)
3. The technologies used to process the ores and concentrates (e.g. type of crushers, smelters, converters etc)
4. The technical performance of the different technologies (e.g. metal recoveries and final state of sulphur)
5. The management of wastes and water
6. The local climate
7. The structure of the mine (integrated or custom)
8. The electricity mixes of the countries were the operations are performed
9. The price of the copper and its associated by-products (acid, silver, gold, molybdenum etc.)
10. The treatment and refining charges
11. The availability of spare solvent extraction and electrowinning capacity in existing operations
12. The cost of utilities and reagents

Where considered in previous studies, these parameters were not considered holistically, but selectively. In view of the aforementioned shortcomings, and because of the need to move towards a sustainable minerals industry, this thesis attempts to consolidate the techno-economic and environmental performances of the various methods of copper production by developing a simulation model in Microsoft Excel® that takes into account the key parameters that affect the performances of the various production routes. The model will be used to simulate the various processing routes of copper production (see Figure 3.1) and predict the techno-economic and environmental performances of the process routes simultaneously. The model is developed using data that is consistent with the data requirements for the early stages of process design, and therefore the performance predictions are a first order approximation.

This chapter describes the methodology used to develop the model, while Chapter 4 presents a case study where the model is applied and demonstrated.

A simplified overview of the simulation model developed in this study is illustrated in Figure 3.1 prior to a more detailed discussion on its development.

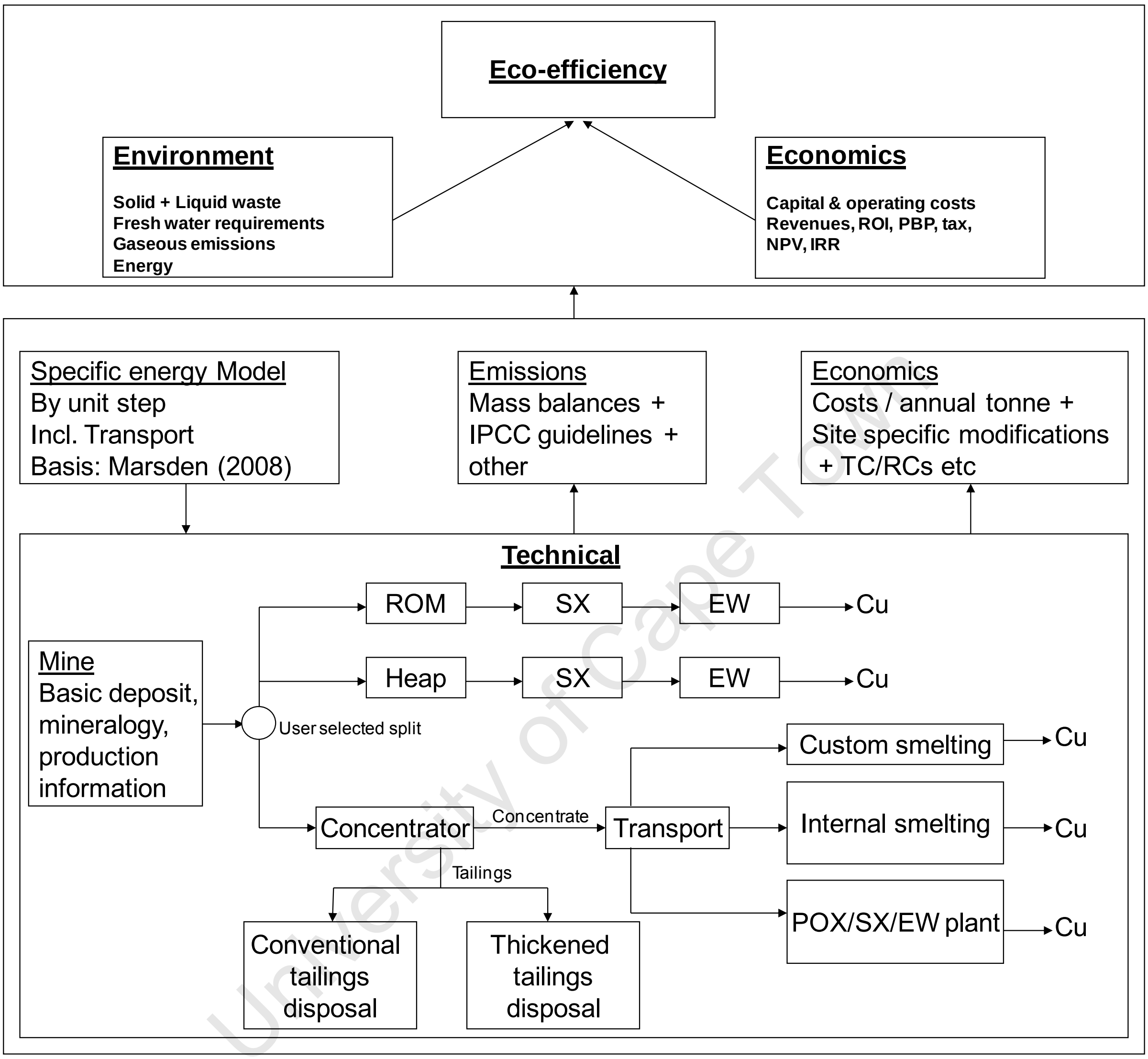


Figure 3.1: Simplified overview of the inventory model used to simulate copper production

3.1 Scope and limitations

The key purpose of this thesis is to develop a methodology for generating and assessing multiple flowsheet options that take into account techno-economic and environmental criteria, using data that is consistent with early stage design requirements for the copper process industry. It is expected that this will be of use to:

1. Decision makers who seek to shift towards a sustainable copper industry.
2. Engineers who may not be familiar with the environmental aspects of copper production.
3. Researchers who are often faced with inventory and data collection, which consumes a lot of time and money.

The model developed in this study was established on the basis of currently available public data, and a quantitative uncertainty analysis was not incorporated into the model. However, where possible, every effort was made to validate the accuracy of the assumptions and data sets used in the study. The model, however, can be adapted to accommodate more accurate input data as it becomes available, ensuring that model outcomes become more reliable and less uncertain.

Methods used to verify the assumptions made during the development of the model included reviewing media sources (newspapers, company reports, technology pamphlets), and personal communication with academics, industrially experienced practitioners and peers. Industrially experienced engineers consulted personally during this study include:

- J. Bezuidenhout (2009 – 2010)
- B. Chaponda (2009 – 2010)
- G. Dunn (2009 – 2010)
- R. Shaw (2010)
- D. Slabbert (2010)

Further communication with industry personnel was carried out by Bezuidenhout on behalf of the author and at the author’s request as direct communication was not possible due to confidentiality issues.

3.2 System boundary definition

The system boundaries considered in the developed model, and consequently in this study, for the production of copper encompass the ore in the ground up to the point where copper cathodes reach the final market. This is in tandem with the economic and environmental analyses performed, and the available information and energy data used in this project. An example of the system boundaries is shown in Figure 3.2, for the case of ores that are sent to a concentrator. The boundaries include operations that are usually under the direct control of mining companies, such as mining, concentration, refining and transportation.

The system boundaries were extended to take into consideration the environmental impacts of electricity generation, since electricity is a significant energy source in mining and mineral processing operations with negative environmental impacts.

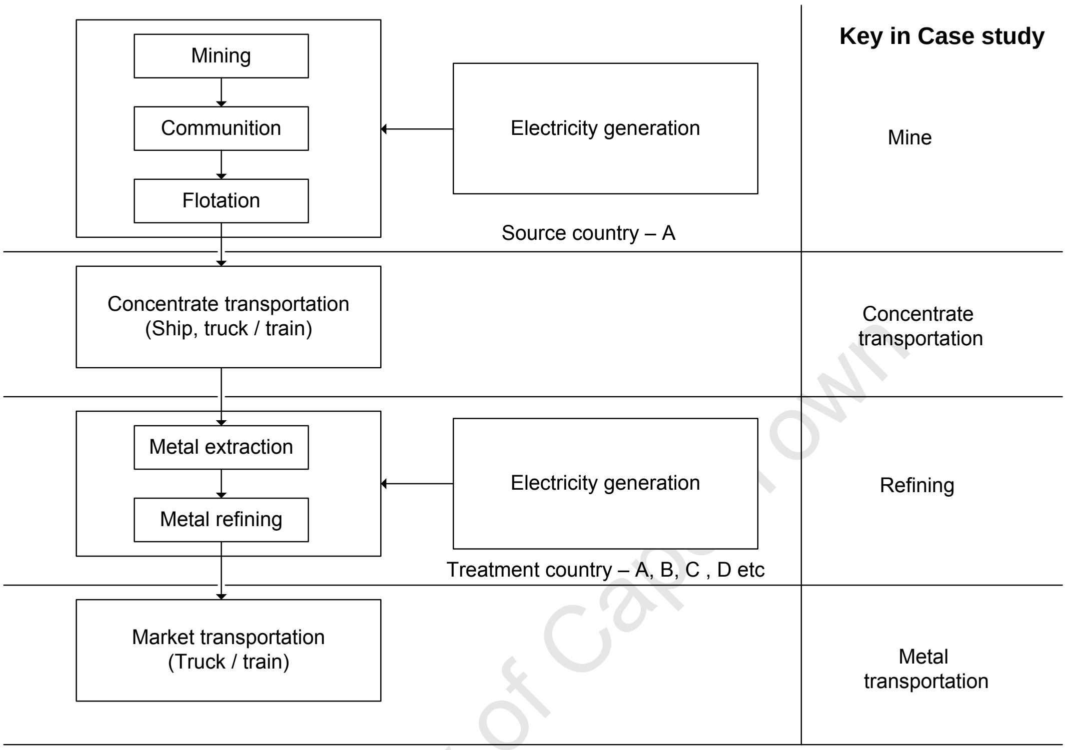


Figure 3.2: System boundary considered in this project

Similarly, for heap and run-of-mine leaching processes, the system boundaries included the mine, leaching operations, solvent extraction and electrowinning plants, transportation of copper to the market and electricity generation.

3.3 Development of process flowsheets and compilation of material inventory data

This section discusses the methodology used to develop the spreadsheet model and how input and output inventory data for the processes outlined in Figure 3.1 are derived from the model. More detailed flowsheets for the process routes that are shown in Figure 3.1 are developed further in the following text.

The material flows of significant environmental and economic importance were identified from the literature (Giurco, 2001; Norgate, 2001; Davenport, 2002; Goonan, 2004; Dreisinger, 2006 a). Mass balance templates were set up for each process route considered in this study, in conjunction with inventory data sourced from the literature; they are a source of material inputs and outputs for all the processes. The scope of the mass balances covers the ore in the ground up to the refined metal (copper cathodes). The mass balance calculations were based on chemical reactions and assumptions from the literature and these are discussed in more detail in sections 3.3.1 – 3.5. Typical equipment efficiencies, reaction conversions and species distributions from the literature were used to emulate the technical performance of each unit. The energy consumption data used in this study was taken from Marsden (2008) and the economic data was derived from the literature and information in the public domain (see section 4.5 for more details).

The spreadsheets that were developed were set up on four levels as described below:

- Input-output data for each process route

  This data includes the material and energy flows associated with copper production for each process route considered in this study. Included are ores, intermediate material flows, chemical reagents, energy flows by fuel type, solid and liquid wastes, and gaseous emissions.

- Environmental performance metrics

  The environmental performance metrics that are included in this study are energy use, global warming potential (GWP), acidification potential (AP) and fresh water consumption.

- Economic performance metrics

  Capital costs, revenues, operating costs, net present value (NPV), internal rate of return (IRR), return on investment (ROI) and payback period (PBP) are the economic indicators used in this study.

- Eco-efficiency indicators

  Eco-efficiency indicators were used in this study for assessing the consolidated environmental and economic performances of the various processes simultaneously. Eco-efficiency indicators were developed using net present value (NPV), energy consumption, global warming potential (GWP), acidification potential (AP) and water consumption.

This methodology is summarised visually in Figure 3.3.

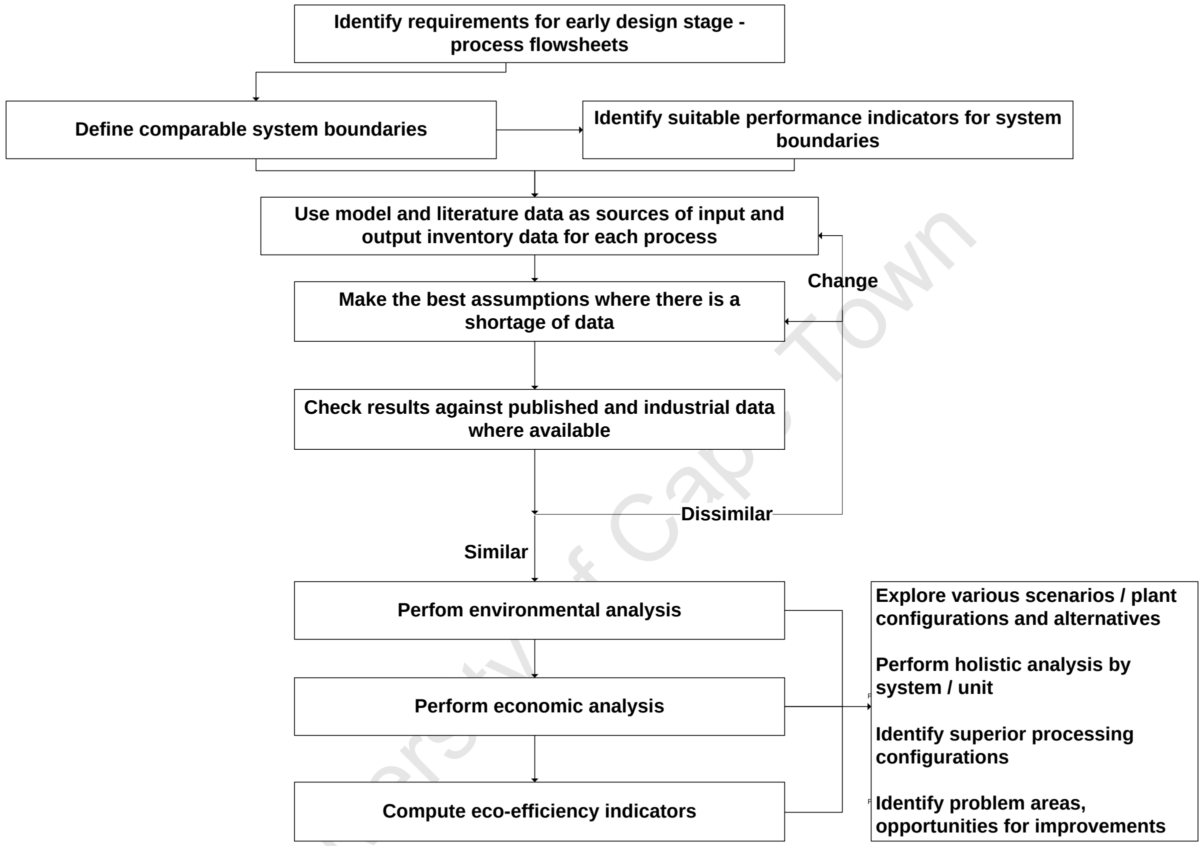


Figure 3.3: Generic overview of project methodology

The following sections describe how the inventory data for each flowsheet is generated and also details the overarching assumptions.

3.3.1 Ore extraction (mining)

A module was set up in the spreadsheets to predict the mass flows from the mine to the concentrator. It was shown in the previous chapters of this thesis that ore mineralogy is an important aspect of process selection and design in the minerals industry. On this basis, a conceptual ore body was developed based on the typical mineralogy of copper deposits. The mineralogy considered in this project consists of the main copper oxide and sulphide minerals, common sulphide gangue minerals and common non-sulphur gangue minerals. The full list of minerals is shown in ***Appendix A*.** Model input variables include the percentage composition of the minerals in the ore and waste, and the annual flowrate of the ore from the mine (tonnes / annum). The grades of copper in the ore and waste are fixed by the selected mineralogy. It was assumed that an open pit mine will be mining the ore because the energy and economic data to support this assumption was available. Furthermore, it is also reported that open pit mining is the predominant mining method in the world for copper (ICSG, 2009). The inventory model was set up in such a way that the mining mode can be switched from open pit to underground mining, provided that the associated energy and economic data are jointly available, as they are interlinked.

A mine stripping ratio of 3 was chosen for the ore that feeds the heap leach operation and the concentrator, and a strip ratio of 1.5 was chosen for the ore that feeds the run-of-mine (ROM) leach process. This was in tandem with the energy model[39] (Marsden, 2008).

The strip ratio was defined as follows:

$$Strip\ ratio = \frac{\dot{M}_{Waste}}{\dot{M}_{Ore}} \tag{3.1}$$

where $\dot{M}_{Waste}$ and $\dot{M}_{Ore}$ are flowrates (tonnes / annum) of the waste and ore respectively. This is illustrated diagrammatically in Figure 3.4, for operations with a strip ratio of 3.

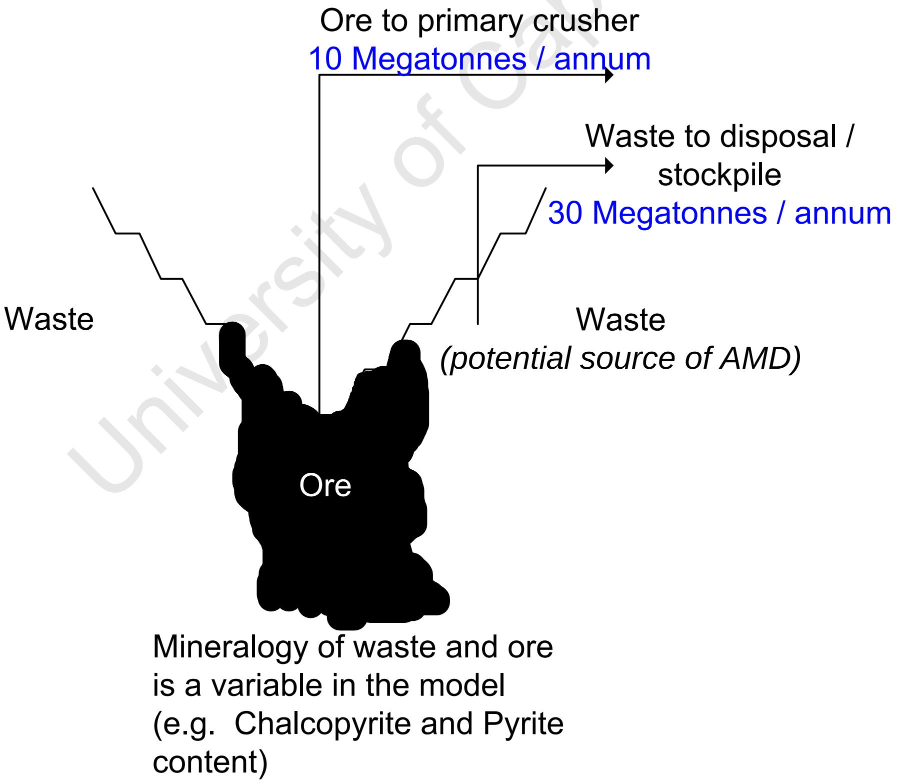


Figure 3.4: Mining operations with a strip ratio of 3 (adapted from University of Pretoria, 2010)

The strip ratios were selected in accordance with the energy model (Mardsen, 2008). However, they can be defined by the user, depending on the availability of economic and

---

[39] Refer to Goonan (2004) to see typical mine strip ratios.

energy data to support alternative stripping ratios. The outputs from this module are the ore and waste flow rates from the mine, which are used in other modules to calculate the associated energy requirements, and environmental and economic indicators. These calculation steps are discussed in more detail in sections 3.4 to 3.6.

3.3.2 Comminution

In the simulation model the ores from the mine go through a series of crushing and grinding stages until the minerals are liberated from the gangue. The oxide and secondary sulphide ores that are processed via heap leaching are normally crushed to approximately 12 mm, which is typically the economic ore size for copper extraction (Davenport et al., 2002). It was assumed that the ores are crushed in a 3-stage comminution circuit (see Figure 3.5).

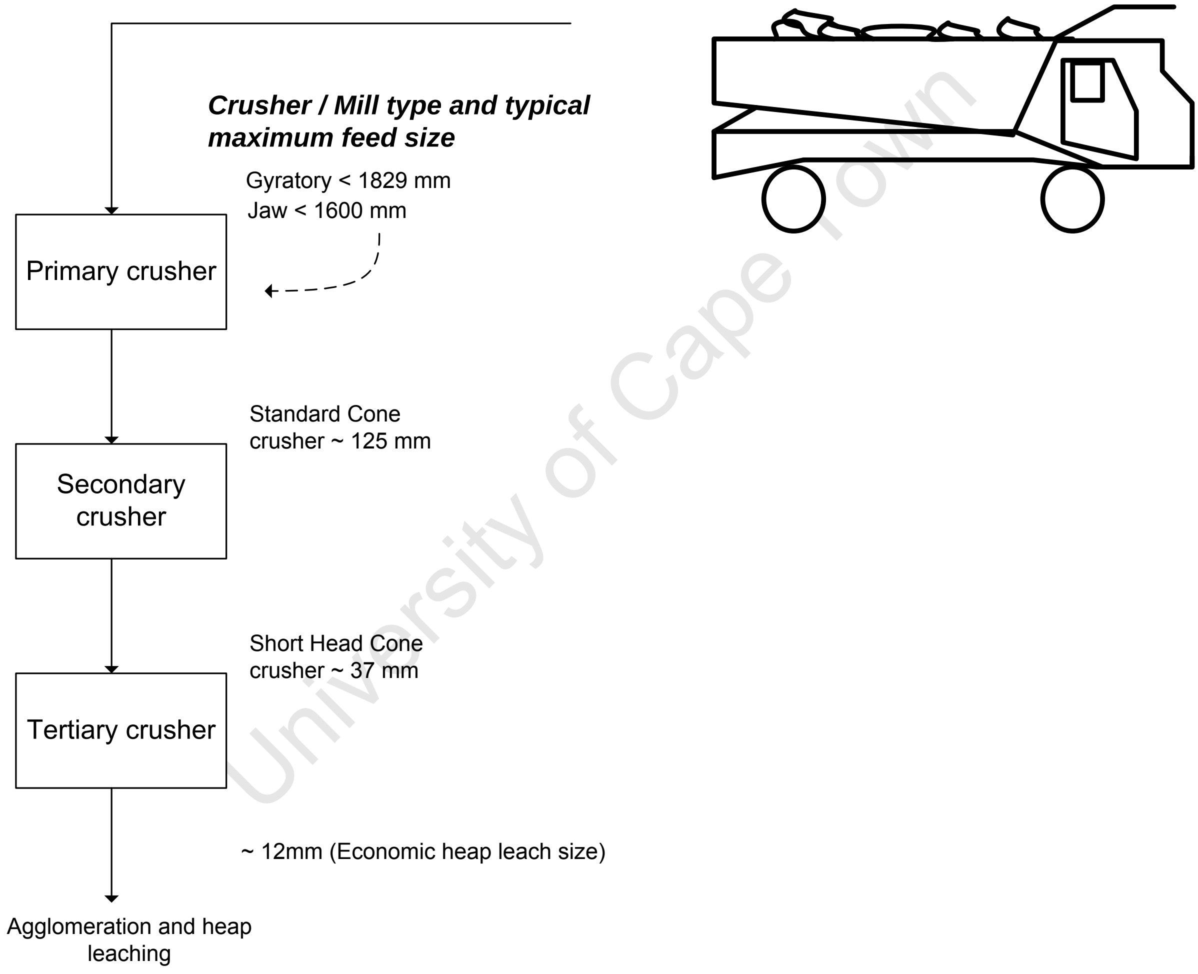


Figure 3.5: Simplified comminution circuit for ore fed to heap leaching operation (adapted from Marsden, 2008)

The primary and secondary sulphide ores which are amenable to flotation (concentration) are normally crushed and ground to a final size of about 50 – 150 µm. In the developed simulation model this can be achieved through crushing and grinding in the comminution circuits shown in Figure 3.6.

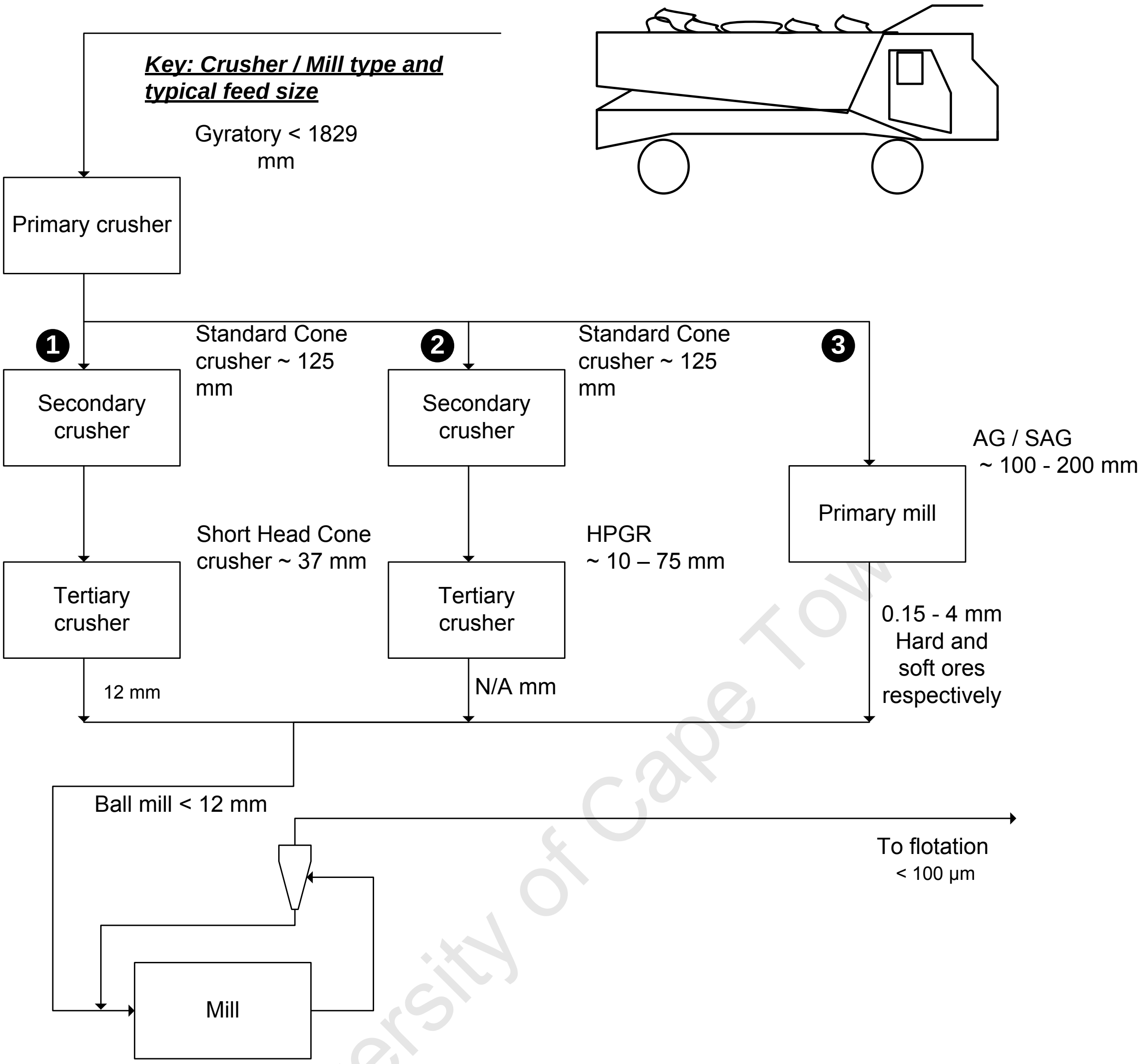


Figure 3.6: Comminution circuit options for ores that are fed to the concentrator[40] (adapted from Wills and Napier-Munn, 2006; Marsden, 2008)

The energy requirements for the comminution circuits shown in Figure 3.5 and Figure 3.6 are calculated from the energy model as described in section 3.4.

---

[40] The dominant comminution circuit for primary sulphide copper ores is the circuit that utilises a SAG mill ***(option 3)***. Older plants (pre 1975) are installing them in place of older circuits while newer plants almost always include them in their designs (Callow and Moon, 2002 and Wills; Napier – Munn, 2006).

3.3.3 Concentrator

Figure 3.7 illustrates the concentrator around which mass balances were modelled.

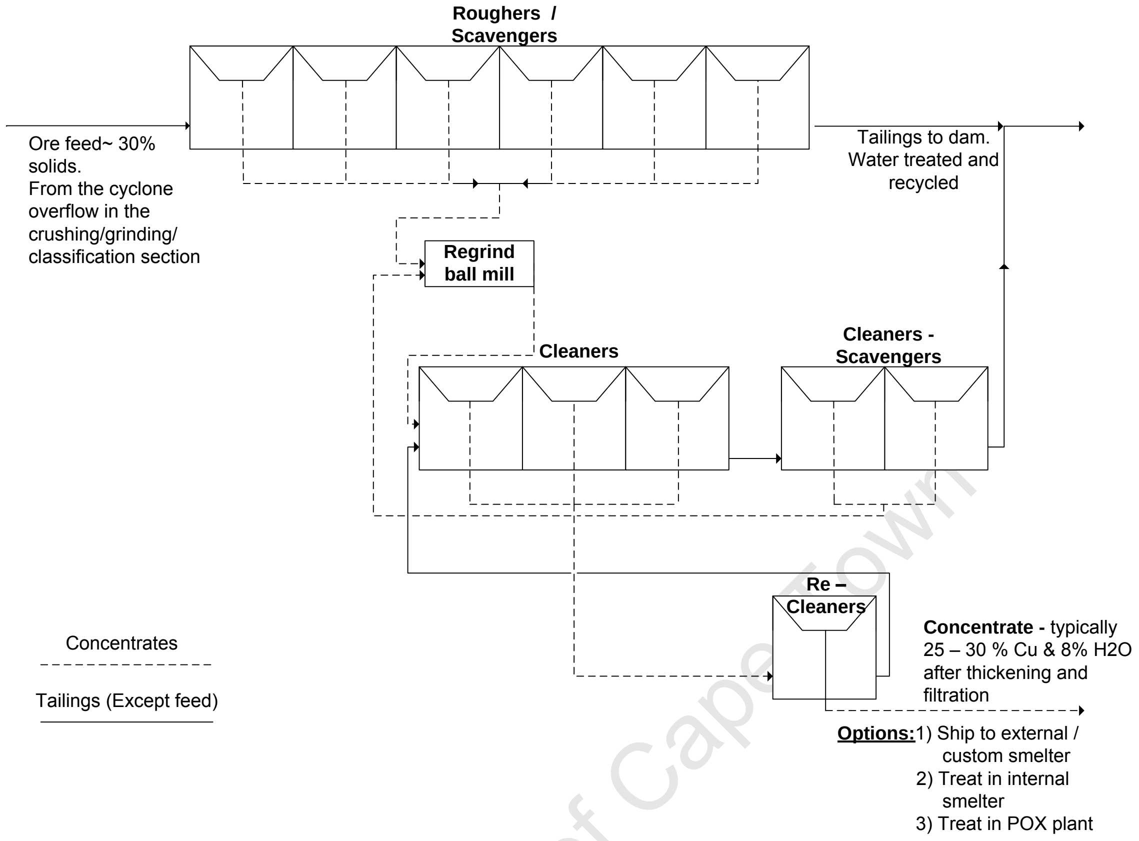


Figure 3.7: Schematic of flotation circuit used in the inventory model

In this module, standard recovery rates of minerals to the concentrate are selected by the user. The recovery of mineral *i* is defined in the model by equation 3.2:

$$Recovery_i = \frac{\dot{M}_{i,C}}{\dot{M}_{i,F}} \tag{3.2}$$

where $\dot{M}_{i,C}$ is the flowrate of mineral ***i*** in concentrate ***C*** and $\dot{M}_{i,F}$ is the flow rate of mineral ***i*** in the feed stream ***F***.

Inputs into the model are the total flow rate of the ore from the comminution circuit, the mineral composition of the ore and the selected recoveries. Model outputs include the flow rates and compositions of the final concentrate and tailings.

3.3.4 Water management and tailings disposal

The fresh water requirements of the concentrator were calculated by assuming that the bulk of the water that enters the flotation circuit goes through the feed stream to the roughers. There are minor water additions in the flotation circuit used for washing the concentrate into the launders, however it was assumed that this additional water is negligible because the final concentrate constitutes approximately 2 – 3% of the initial feed to the roughers (Davenport et al., 2002; Mwale et al., 2005). Furthermore, the bulk of the water added to the concentrate is recovered and recycled internally when the concentrate is thickened and filtered to a final water content of approximately 8% (D. Slabbert, personal communication, 2010). In performing the water balance around the concentrator and tailings management facility (dam) the model water losses occur due to evaporation from the thickeners that are in the concentrator and evaporation from the tailings management facility (dam). Additional water is lost due to seepage and entrainment[41] during tailings disposal. These water losses are replaced by fresh water from primary sources such as dams, boreholes and mine dewatering, as well as other sources such as rainfall and surface run-off. It was assumed that the water recovered from the thickeners, filters and the tailings dam ponds is recycled back to the mill and eventually ends up in the rougher feed and the cycle is repeated.

In the model the tailings can be either disposed directly in a conventional tailings dam or through the thickened tailings disposal method. Figure 3.8 illustrates the two tailings disposal methods that were modelled.

---

[41] Water trapped in the solids’ void spaces (Mwale et al., 2005).

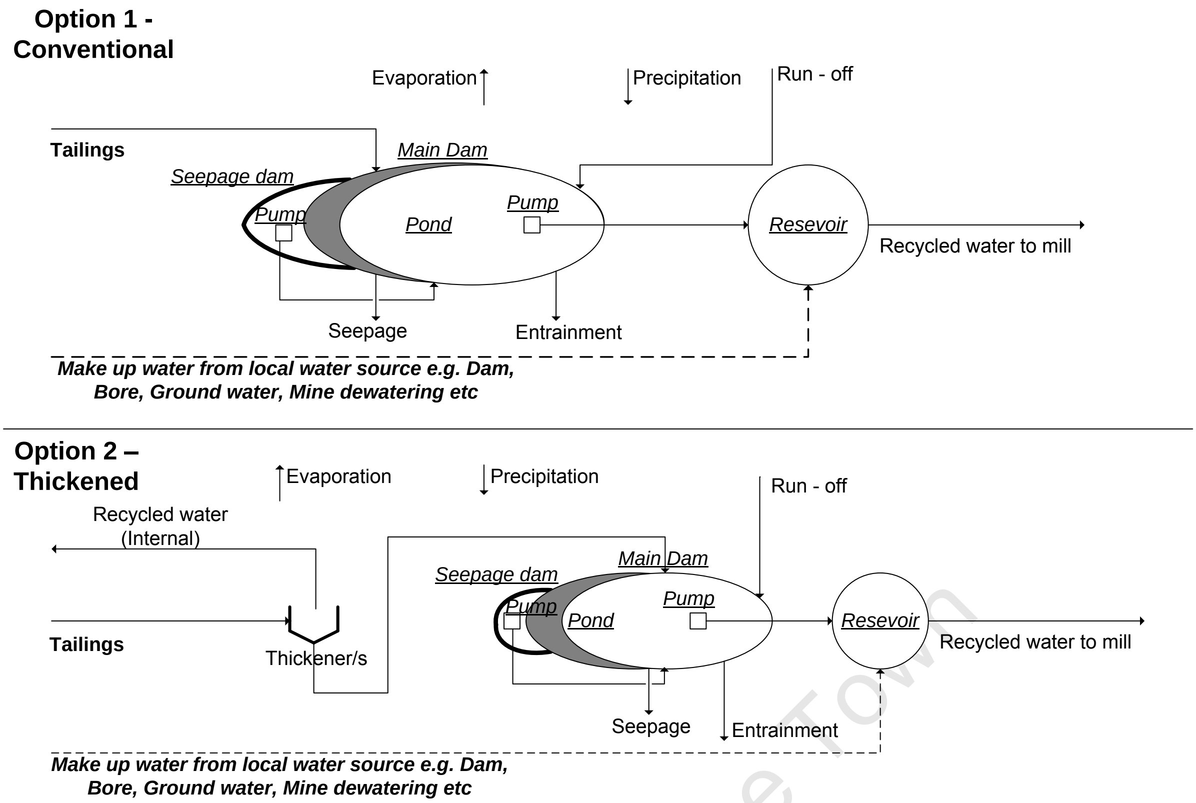


Figure 3.8: Tailings disposal methods (adapted from Chambers et al.,2003; Mwale et al., 2005; Engels and Dixon-Hardy, 2009)

Table 3.1 shows a summary of the model input variables and the mass balance outputs.

Table 3.1: Model variables for calculating the mine / concentrator fresh water requirements

Inputs / Variable	Outputs
Rougher feed moisture content (%)	Water loss (Annual tonnages)
Initial concentrate moisture content (%)	Fresh water requirements (Annual tonnage)
Final concentrate moisture content (%)	Re-use rate (%)
Concentrate thickener pulp density (%)	Mass flow rates (All streams – Annual tonnage)
Concentrate thickener evaporation rate (%)	
Tailings thickener pulp density (%)	
Tailings thickener evaporation rate (%)	
Rainfall (mm)	
Run-off (Annual tonnage estimate)	
Tailings dam evaporation rate (%)	
Seepage and entrainment rate (%)	

3.3.5 Smelter

Figure 3.9 illustrates the smelting facility around which mass balances were performed.

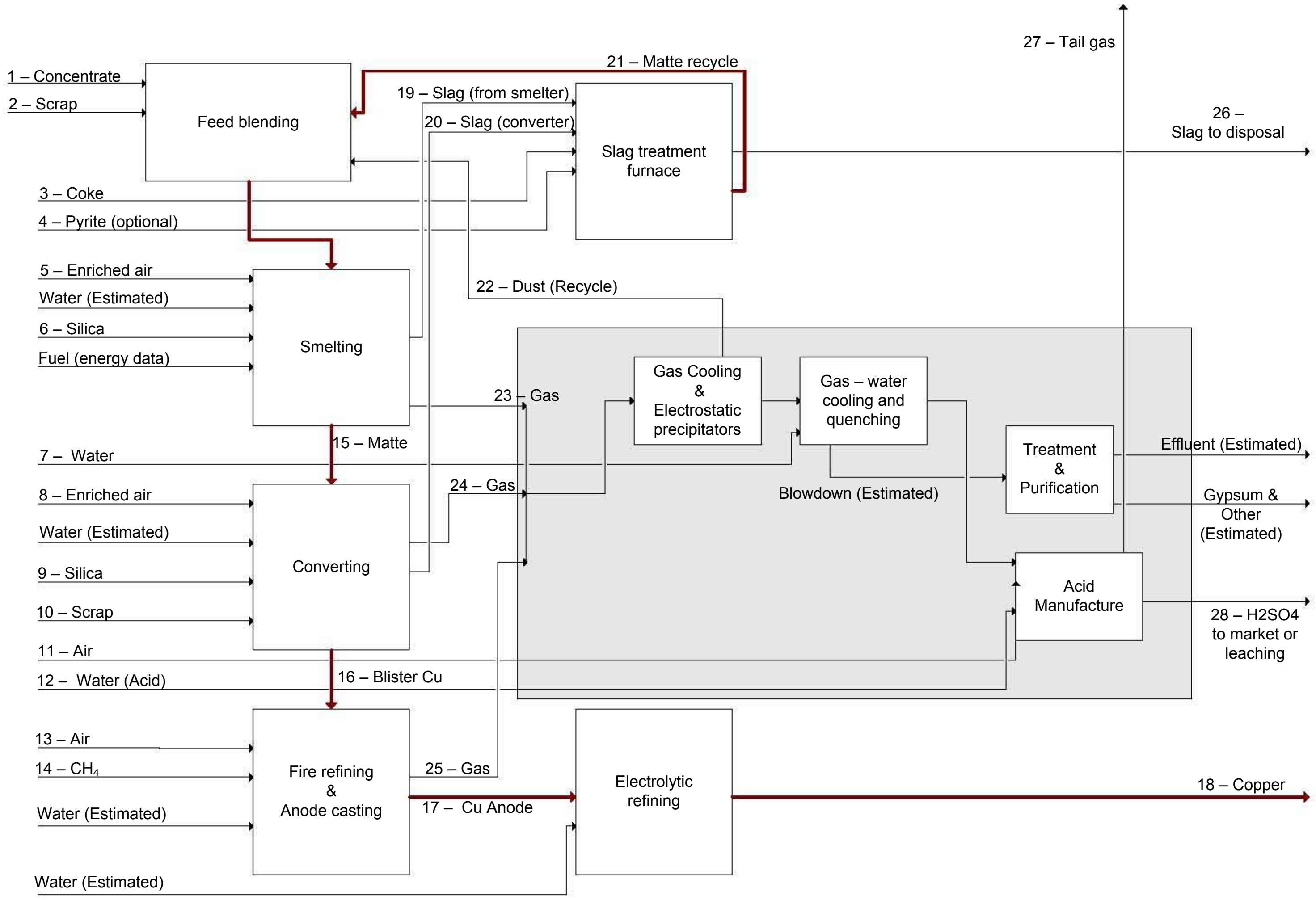


Figure 3.9: Smelter, refinery and acid plant (adapted from Goonan, 2004)

The feed streams to the smelter that contain copper are copper concentrate, scrap, slag treatment – matte, and recycled smelter and converter dusts[42]. The economic outputs from the smelter are copper cathodes and sulphuric acid. While gold and silver were included in the mass balance templates, they did not form part of this case study and were therefore not modelled in detail. Nevertheless, based on distributions from the literature (Davenport et al., 2002) the model can reasonably predict their behaviour in the refinery.

The mass balances performed around the smelter were based on chemical reactions selected from the literature (Lakernik and Sevryukov, 1960; Davenport et al., 2002; Sanchez-Corrales et al., 2004). Species distributions into the various streams were also obtained from the literature (Davenport et al., 2002). The detailed chemical reactions and species distributions can be seen in ***Appendix B***.

It is important to remind the reader that the main objective of smelting (prior to converting) is to produce a copper matte (Cu_2S and FeS) of ideally 63% Cu content. Therefore, in the model, it is assumed that all the Cu_2S formed in the smelting furnace is not oxidised further in the smelting furnace but will only react in the converter where Cu_2S is converted to blister copper (Cu) and SO_2 gas. FeS is a product of smelting, however some of it is oxidised while in the smelting furnace to remove some Fe and S as FeO and SO_2 respectively, leaving behind a sufficient quantity for the near autothermal operation of the converter. The extent of FeS oxidation is controlled by the oxygen input into the smelter, which is an important matte grade control parameter in the model (Davenport et al., 2002).

The model inputs parameters and outputs for the smelter are summarised in Table 3.2.

Table 3.2: Model variables for calculating smelter mass flows

Inputs / Options	Outputs
Stoichiometric equations	Mass flows of streams (internal & external)
Concentrate, scrap, etc (flow & composition)	Cu, Fe, S, SiO2 grade in concentrate
Oxidation extents / reaction conversions (%)	Acid production, sulphur by products (flows)
Distributions of elements to streams (%)	Flux, pure O_2 requirements
Oxygen enrichment (%)	Blowdown (acid plant)
SO_2 conversion to acid, acid strength (%)	Mass balance check for each unit
Recycle streams (dusts)	Copper produced (Annual tonnage)
Slag cleaning (Optional)	CO_2 from carbonate destruction
Matte grade control (through FeS oxidation)	

---

[42] Recycling the dusts helps to force arsenic out in the slag (R. Shaw, personal communication, 2010).

3.3.6 Pressure Oxidation (POX)

The copper concentrate from the concentrator can be fed simultaneously into the smelting and POX plants. This allows for a simultaneous analysis and comparison of the techno-economic and environmental performance of both the smelting and POX processes. Figure 3.10 illustrates the flowsheet around which mass balances were performed for the POX process. The mass balances were based on chemical reactions and assumptions from the literature (Dreisinger, 2005, 2006 a; Long and Dixon, 2006) and the reactions are shown in ***Appendix C***. The Excel spreadsheets were set up as shown by Dreisinger (2006 a) and a summary of the model parameters is shown in Table 3.3.

Table 3.3: Summary of model variables for calculating POX mass flows

Inputs / Options	Outputs
Stoichiometric equations	Mass flows of streams (internal & external)
Concentrate (flow & mineral composition)	Limestone and pure oxygen requirements
Oxidation extents / reaction conversions (%)	Pregnant leach solution (Cu, Fe, H_2SO_4) composition
Species heat capacities, Formation enthalpies	Reagents
Operating conditions (Temp & P for all units)	Quench cooling requirements (flow & energy)
Pulp densities, washing efficiencies	Copper produced (Annual tonnage)
Option to extract zinc (as by-product)	CO_2 from carbonate destruction

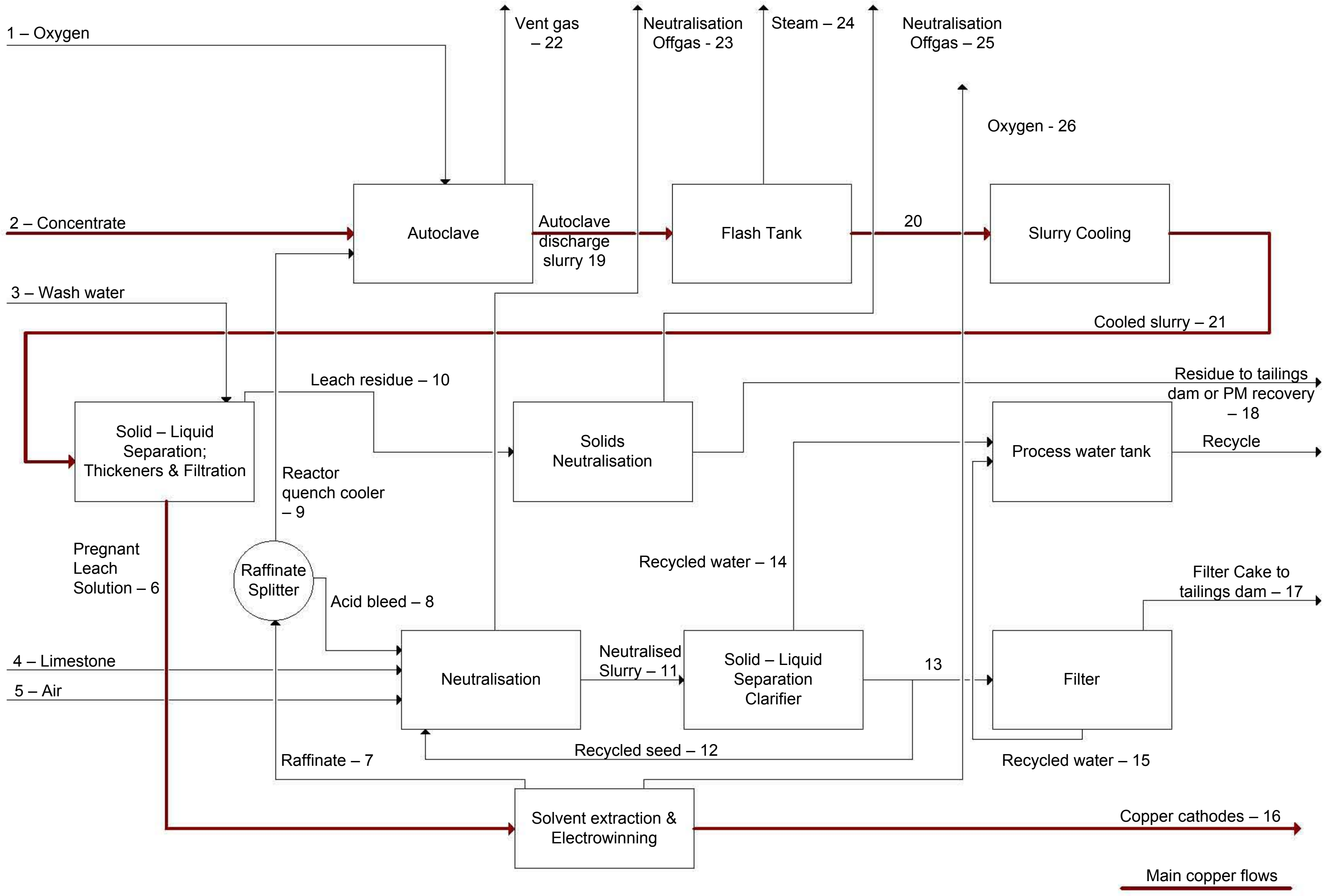


Figure 3.10: POX plant with neutralisation of the acid in the raffinate (Adapted from Dreisinger, 2006 a)

The model was set up in such a way that the acid in the raffinate from the POX plant can be neutralised (see Figure 3.10) or used beneficially in ROM / heap leach processes that require acid on site or next to the site, thereby avoiding acid neutralisation costs. The latter configuration is shown in Figure 3.11.

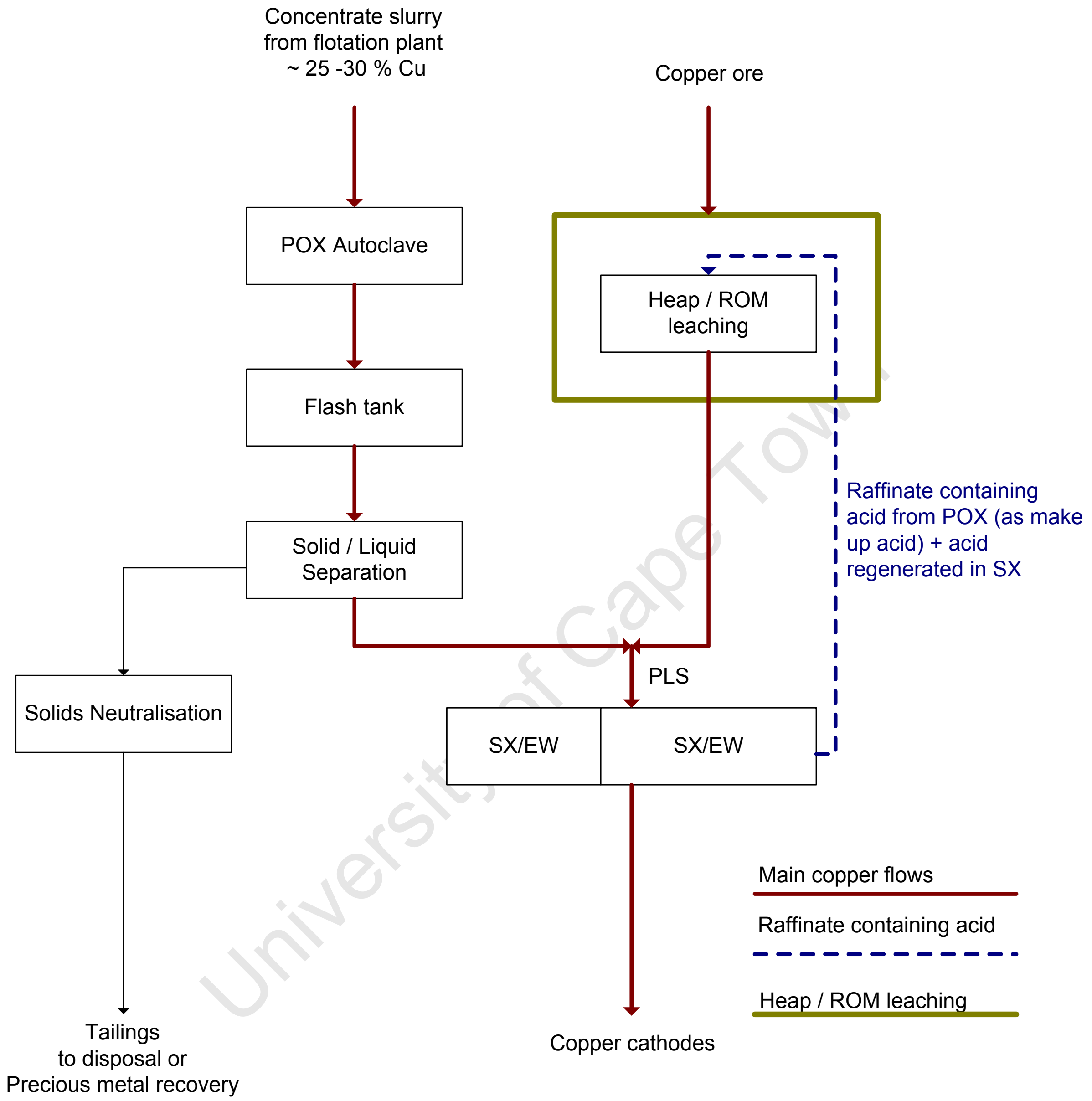


Figure 3.11: Simplified block flow diagram of combined POX and heap leach operations (adapted from Marsden and Wilmot, 2007; Marsden, 2008)

A financial credit, using the long term market value of acid, is given to the POX process when the acid is used in heap / ROM leaching processes onsite.

3.3.7 Heap / ROM leaching

In the inventory model the oxide and secondary sulphide copper ores[43] from the mine are fed to a heap leach operation for copper extraction. Since the ore feed rate to the plant and its mineralogical composition are known, standard extraction rates can be specified for each mineral. Typical extraction rates can be seen in Figure 2.11 in Chapter 2. It is widely accepted for heap leaching that copper extraction rates of 70 – 80% are feasible in the complete leach cycle of copper ores. It was assumed that the complete leach cycle is completed annually and that quartz is not soluble in acid during the during the leach period. The model input variables and outputs are shown in Table 3.4.

Table 3.4: Heap / ROM leach model input variables and outputs

Inputs / Variables	Outputs
Stoichiometric equations	Pregnant leach solution composition
Ore (flow & mineral composition)	Make up / Fresh acid requirements
Oxidation extents / reaction conversions (%)	Surface area of heap
Lixiviant composition	Solution pH
Heap height	Copper production (Annual tonnage)
Bulk density	Solid waste and its mineral composition
Irrigation rate	

The composition of the pregnant leach solution is calculated by performing mass balances around the heaps based on the chemical reactions shown in ***Appendix D***.

The pregnant leach solution is then fed to the solvent extraction and electrowinning plants in which copper is extracted and converted to copper cathodes. The fresh acid requirements are calculated as the difference between the flowrate of acid in the lixiviant and the acid in the raffinate that is returned from the solvent extraction plant.

During heap leaching soluble ions build up in the raffinate since it is continuously recycled. However, Brierley and Brierley (1999) seem to suggest that the accumulation of the ions in the system is controlled by the entrainment of the solution in the heaps that are taken off line, serving as a bleed, and precipitation in the heap. In the present mass balances ion build-up is controlled by precipitation and an active bleed stream.

---

[43] Secondary sulphide copper ores can also be fed to the concentrator prior to smelting or POX.

3.3.8 *Water requirements*

The fresh water requirements for the mine and concentrator were calculated using the mass balances described in sections 3.3.1 – 3.3.4. The fresh water requirements for the smelter and refinery were obtained from the mass balances described in section 3.3.4, and for activities that cannot be modelled solely by mass balances, such as cooling and cathode washing, estimates from the literature (MIM Holdings, 2001; Sterlite Industries India Limited, 2008) were used. The fresh water consumption data for the POX process were obtained solely from the mass balances as described in section 3.3.6 because, unlike the smelting process, which is the predominant copper production route in the world, information pertaining to the POX process is currently scarce in the public literature since the process is not widely used. Water consumption data for heap and ROM leaching was obtained from the mass balances described in section 3.3.7 and from COCHILCO (2008) and Brantes (2009).

3.4 Energy inventory compilation

To calculate the energy flows associated with the material flows that have been discussed in section 3.3, Marsden's (2008) energy model was incorporated into the spreadsheets as a sub-model. Marsden's (2008) model has energy data for copper production for all the processes that are outlined in Figure 3.1 and Figure 3.12, covering all the main units in the 'minerals to metals' chain, i.e. from ore in the ground to the saleable copper cathodes at their market. Marsden's (2008) model can be considered as a template of the spreadsheets developed in this project since users have the option to enter their own energy consumption data.

Marsden's (2008) energy model is used in the simulation model to estimate energy consumption for each process unit in energy units of kWh and kJ. A conceptual illustration of the algorithm used in the spreadsheets is shown in Figure 3.12 and sample calculations are in ***Appendix H***.

Calculations were set up to convert the energy consumption data to equivalent mass and volume units (kg and litres) for each fuel, and Marsden's (2008) model was extended to account for offsite fuel consumption during electricity generation in the countries where the mine and smelter are located. Furthermore, because of the wide variation in coal characteristics and its dominant use in electricity generation, the type of coal used was made a variable in the model.

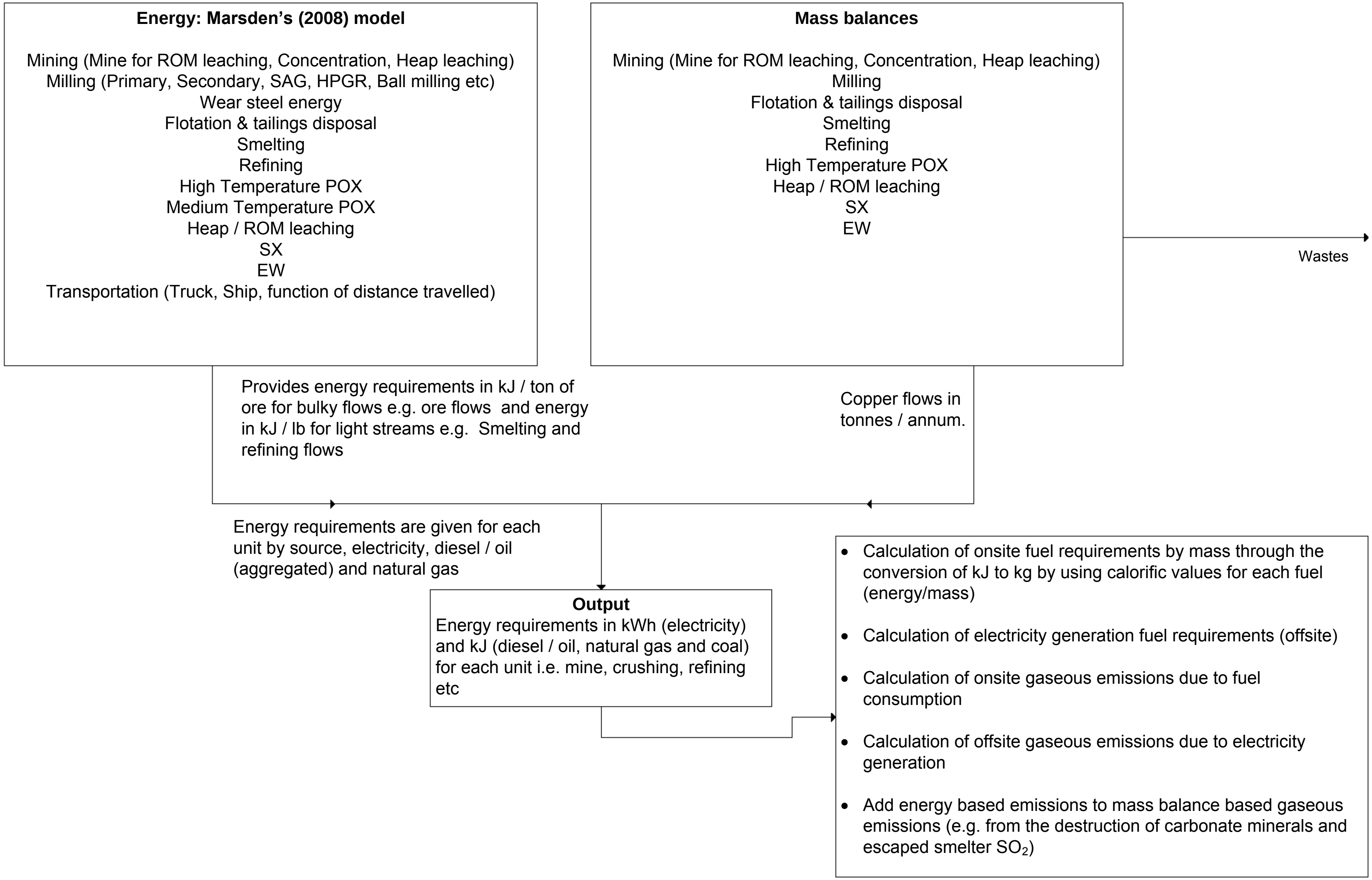


Figure 3.12: Conceptual illustration of the algorithm used to calculate the energy requirements of each process

3.5 Calculation of emissions

The gases considered in this study are CO_2, CO, SO_2, NOx, N_2O, NMVOC[44] and CH_4. Using energy consumption data derived from the model and emissions data from the Intergovernmental Panel on Climate Change (IPCC, 1996, 2006), spreadsheets were set up to calculate the gaseous emissions associated with energy consumption for each process route that was modelled.

Since concentrates can be treated in the country where they are produced, or in an external country, the electricity mixes and generation efficiencies for the source and treatment countries are model variables. The electricity mix for each country consists of coal, diesel, natural gas, biomass, hydro, nuclear and renewable energy. It was assumed that hydro, nuclear and renewable[45] energy sources do not cause significant environmental disturbances towards any of the impact categories analysed in this study. In Marsden's energy model, no distinction is made between diesel and oil, therefore, for the purposes of this study, low sulphur diesel was selected as the fuel of choice where diesel or oil was used as an energy source. The exception to this was the oil used in ships which was modelled as 1.5%, sulphur containing, heavy fuel oil.

The equivalence factors used for calculating the global warming and acidification potential of each process route were taken from Azapagic (2005, 2010). Figure 3.13 gives a conceptual illustration of the methodology used to calculate the gaseous emissions and consequently global warming and acidification potential for each process route and sample calculations are shown in ***Appendix H***.

---

[44] Non-methane volatile organic compounds

[45] Renewable sources in this project are limited to solar and wind energy and currently exclude bio fuels.

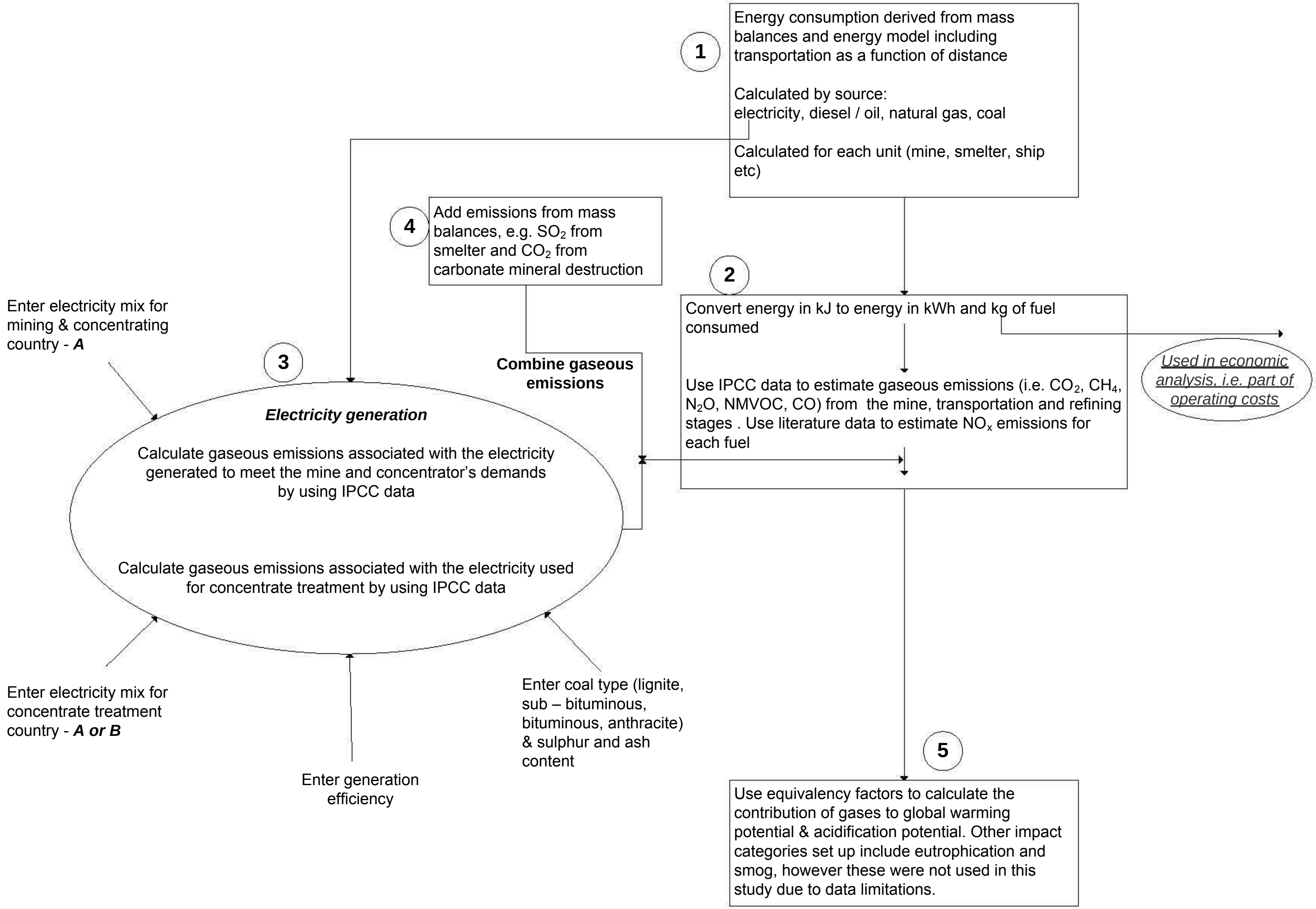


Figure 3.13: Conceptual illustration of the algorithm used to calculate the gaseous emissions of each process

3.6 Economic assessment

3.6.1 Capital cost estimates

Order of magnitude estimates were used for the capital cost estimations in this project; therefore the capital costs are accurate to ±40%. In particular, the *fixed investment per annual tonne of capacity* method was used to calculate the total capital investment required for each process (see Couper (2003) for a detailed discussion on capital cost estimates). For the purposes of this study this approach was considered reasonable because:

1. This research is being undertaken using data that is suitable for the conceptual stages of process selection and development where the objectives are normally to screen ideas and assist in long range planning (Couper, 2003).
2. Order of magnitude costs estimates can be obtained from the literature without soliciting bids from equipment vendors (Couper, 2003).
3. The information used in this model can be updated as the user obtains more accurate and current information that reflects the data requirements of the user's particular study.

With the exception of custom smelting, which does not require a capital investment from the custom mine, the capital costs of all the integrated processes were obtained from the literature (McElroy and Young, 1999; Davenport et al., 2002; Peacey et al., 2003; Marsden, 2004; Dreisinger, 2005).

3.6.2 Operating cost estimates

In the model the operating costs for each process route are generally calculated using equation 3.3:

$$Operating\ costs\ \left(\frac{\$}{annum}\right) = consumables\,\&\,reagents\ +\ power\,\&\,fuels\ +\ labour\ +\ maintenance\ +\ general\ and\ administration\ +\ selling\ expenses \qquad (3.3)$$

The consumables and reagent flows are obtained from the mass balances and their current costs can be obtained from public sources (news, newspapers, websites, company reports and vendors). The fuel and electricity requirements are obtained from the energy sub-model and their associated costs can also be obtained from public sources. Operating costs for cost components such as labour, maintenance and general and administration are inputs into the model and these costs can be obtained from the literature and private sources when available (see for example, McElroy and Young (1999), Davenport et al. (2002) and Peacey et al. (2003)). Depreciation is considered in the discounted cash flow analysis (see section 3.6.5).

A conceptual illustration of the economic analysis performed in this study is shown in Figure 3.14.

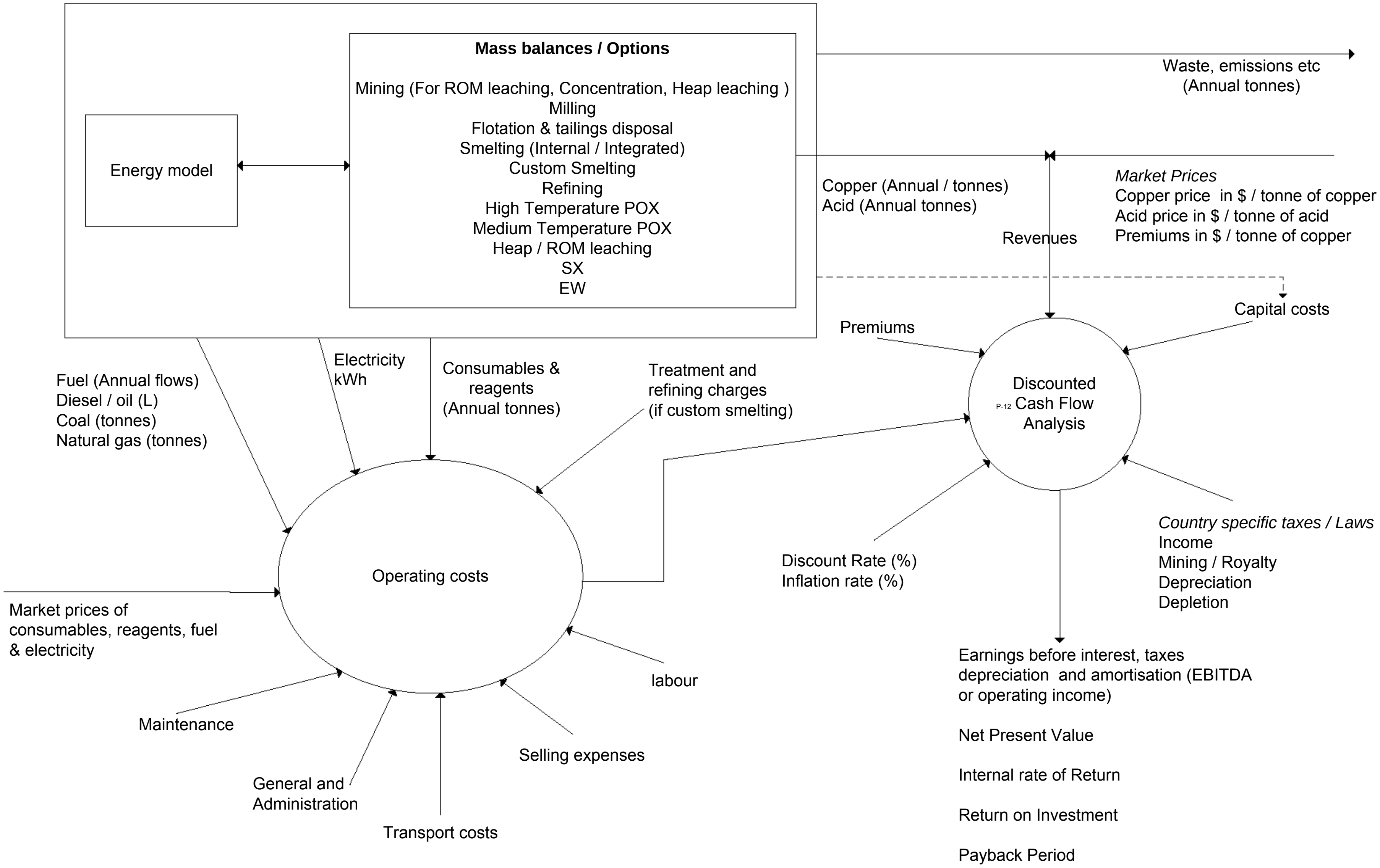


Figure 3.14: Conceptual illustration of the algorithm used in the economic assessment

The details of the methodology used in the economic evaluations of the process routes considered in this study are discussed in section 3.6.5.

3.6.3 Custom smelting

In a situation where the mine does not own a smelter or a POX plant, its operating costs for treating its concentrates are calculated by simulating the financial and logistical clauses of a copper concentrate contract with custom smelters (see sections 2.3.4 and 2.3.5). In this option (custom smelting) the copper concentrate from the mine is transported by truck and ship to custom smelters and refineries for processing into copper cathodes. The mode of transportation and distance to the smelter are variables in the model and are a function of the location of the smelter. A summary of other model variables is shown in Table 3.5.

Table 3.5: Input and output variables for custom smelting

Inputs / Variables	**Outputs**
Concentrate (flow and composition)	Dry concentrate Cu content (%)
Custom smelter accountability (%)	Wet concentrate Cu content (%)
Acceptable copper content (%)	Payment for copper receivable by mine (i.e. value of copper in the concentrate prior to deductions)
Unit deduction for low grade concentrates (%)	Deductions by the smelter for:
Treatment charge & refining charge	***Smelting***
PP[46] base treatment charge	***Refining***
PP amount (%)	***Price participation***
Maximum allowable content of penalty elements (%)	***All penalties***
Penalty amounts ($/tonne/ % excess) for excess As, Pb, Bi, Zn, Sb, moisture	

The costs of transporting the concentrates are also included in this module as a variable. Cost items such as transportation costs, treatment and refining charges and individual concentrate contract terms are usually not readily available to the general public and when they are available they are aggregated. The costs used in this project were obtained from Marsden and Wilmot (2007), Soderstrom (2008) and a general review of copper industry news.

---

[46] Price participation.

3.6.4 Revenues

In the model, the revenue[47] generated by a custom mine from selling copper concentrates to custom smelters is calculated using equation 3.4 (Wills and Napier-Munn, 2006):

$$Net\ Smelter\ Return = \\ Payment\ for\ contained\ metal - (Smelter\ charges + Transportation\ costs) \qquad (3.4)$$

For integrated mines the revenue generated by the mine from selling copper cathodes and sulphuric acid, and from premium payments is calculated as follows:

$$Sales\ revenue \left(\frac{\$}{annum}\right) = Copper\ cathodes \left(\frac{tonnes\ Cu}{annum}\right) * LME\ price\ of\ copper \left(\frac{\$}{tonne\ Cu}\right) + \\ Acid \left(\frac{tonnes\ acid}{annum}\right) * Price\ of\ acid \left(\frac{\$}{tonne\ acid}\right) + Copper\ cathodes \left(\frac{tonnes\ Cu}{annum}\right) * \\ Premium \left(\frac{\$}{tonne\ Cu}\right) \qquad (3.5).$$

3.6.5 Economic evaluation

For the economic evaluation of the processes considered in this work, a discounted cash flow analysis was modelled, and after-tax cash flows were used as recommended by Smith (2002) for mining projects. The calculation of the cash flows is summarised in Table 3.6.

---

[47] The Net Smelter Return received by the mine is subject to further deductions, i.e. direct and indirect operating costs for mining and concentration before the mine determines its earnings (Wills and Napier-Munn, 2006).

Table 3.6: Methodology for calculating cash flows (adapted from Smith, 2002)

Operation	Component
+	Gross revenue
-	Treatment/ Refining / Price Participation
-	Royalties
-	Operating costs
Total	**Net operating income (EBITDA)**[48]
-	Capital expenditures as spent
-	Working capital
-	Mining & Income taxes*
Total	**Cash Flow**
Tax calculation (Separate)	
+	Net operating Income
-	Cash depreciation[49]
Total	**Net taxable income**
Total	**Mining & income taxes***

Other assumptions that were made in the financial analysis performed in this study are (Smith, 2002):

- The metal has a constant price over the life of the project.
- The discounted cashflow analysis includes inflation for the purposes of tax calculations; and the metal price, operating and capital costs were inflated at the same rate (see Table 4.2 in section 4.5). However, the resulting cashflows were deflated so that the NPV and IRR could be expressed in real or ‘today’s’ dollars.
- No debt financing

---

[48] EBITDA – Earnings before interest, taxes, depreciation and amortisation.

[49] Other studies may include depletion in tax calculations, see for example, Hrebar and Gentry (2003). However, some countries do not allow a tax deduction for depletion, e.g. Chile, which is the subject of the case study in Chapters 4 and 5 (Omalu and Waelde, 2000; Cariola Díez Pérez-Cotapos & Cía Ltda., 2010; PKF, 2010). Therefore, depletion is not considered in the current study. Depletion refers to the reduction of mineral / oil / gas reserves. It is a tax deductable amount for reserves that is similar to depreciation for equipment.

- After-tax cash flows were used

The economic indicators that can be obtained from the discounted cash flow analysis are net present value (NPV), internal rate of return (IRR), return on investment (ROI) and pay back period (PBP). Country specific royalties, income and mining tax rates vary widely, however, they can be obtained from government offices and documents, as well as company reports.

3.7 Analysis of Eco-efficiency indicators

Numeric eco-efficiency indicators were used in the case study applied in this project since positive economic outcomes are expected and desired. The eco-efficiency indicators were computed as follows:

$$Eco-efficiency_{\mathrm{j}} = \frac{\mathrm{NPV}}{\sum_{\mathrm{i}=1}^{\mathrm{n}} Environmental\ Performance_{\mathrm{j,i}}} \tag{3.6}$$

where *j* is the environmental performance indicator (e.g. acidification potential), and *i* is the year number during the project’s estimated life.

The financial performance indicator used in the computation of all the eco-efficiency indicators is net present value (NPV), and the environmental performance indicators used are energy consumption, global warming potential (GWP), acidification potential (AP) and fresh water consumption.

4 CASE STUDY APPLICATION

It was shown in Chapter 2 that several hydrometallurgical technologies were developed to compete with smelting for the treatment of copper concentrates. In Chapter 3 a methodology and model for comparing different processing technologies were developed. In this chapter the methodology and model are applied to a case study in which custom smelting is compared to integrated smelting and POX. The case study is explained in more detail below.

4.1 Case study definition and scope

In this case study, an active custom mine that is located in Chile is currently producing copper concentrates that are being sold to custom smelters in China. In an attempt to improve its economic and environmental performance, the mine is currently considering investing in its own smelter or POX plant next to the mine. The mine also has a heap leaching operation near the concentrator that is currently importing acid from offshore suppliers, and it is projected that the acid produced by an integrated smelter or POX plant will be used, completely, in the heap leaching operations. Therefore, the integrated smelter or POX plant would receive credits for the acid[50], and the need to neutralise the acidic raffinate from the POX plant would be eliminated.

In summary, the 3 options that will be compared in this study are:

1. The outright sale of copper concentrates to custom smelters in China and the importation of acid for the heap leaching operations ***(the current company practice / status quo).***
2. The construction of smelting and refining capacity near the mine and concentrator ***(vertical integration)*** and the use of by-product acid on the heap leaching operations.
3. The construction of a Total Pressure Oxidation (POX), solvent extraction and electrowinning plant near the mine and concentrator ***(vertical integration*)** and the use of by-product acid on the heap leaching operations.

The scope of the case study is illustrated schematically in Figure 4.1.

---

[50] The long term market value of acid was used ($ 100 / tonne of acid, see Table 4.2).

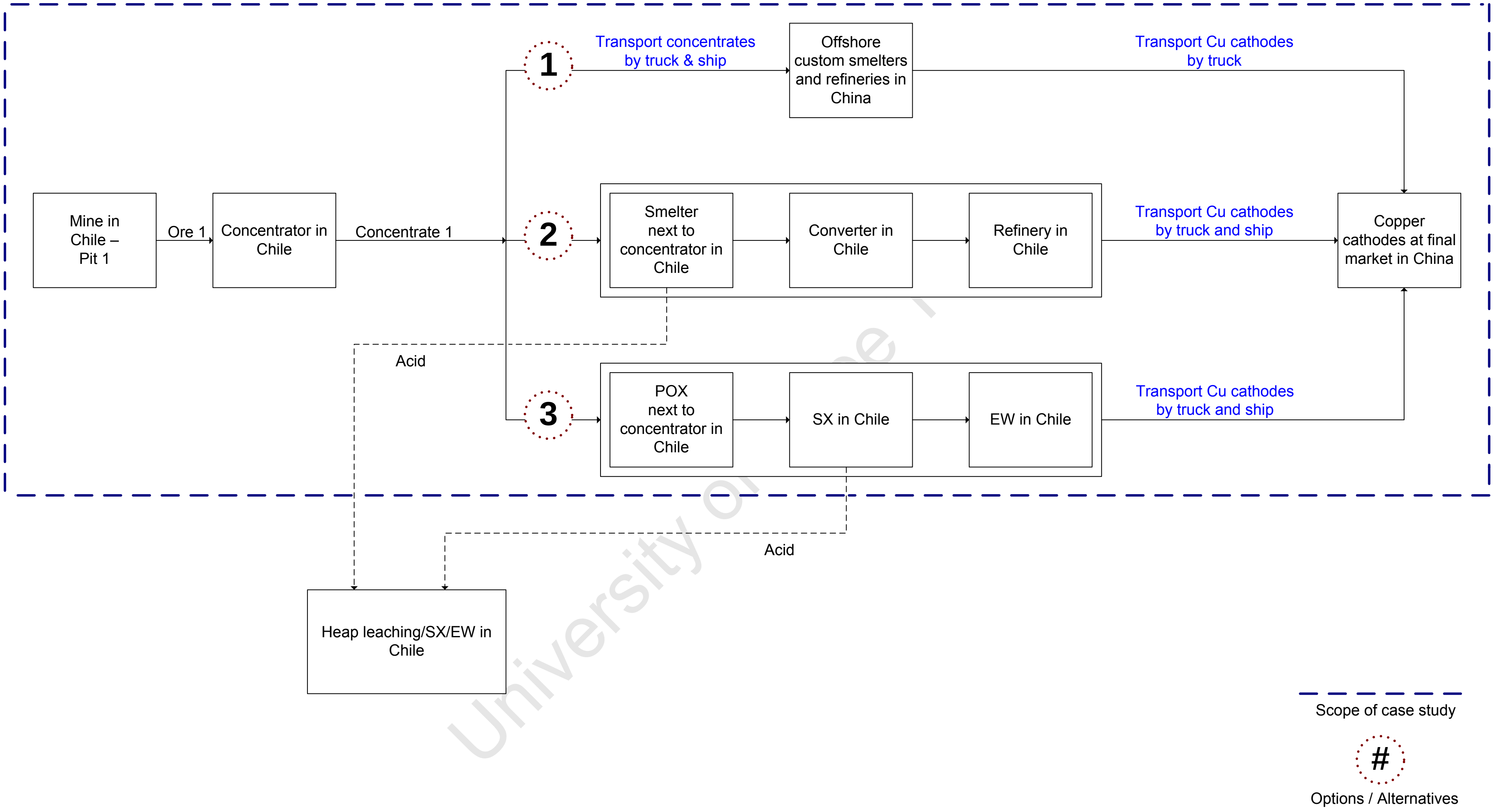


Figure 4.1: Conceptual illustration of the scope of the case study and the options under consideration

4.2 Aims of the case study

The overarching aim of this case study is to apply the proposed methodology and developed inventory model to a typical decision making scenario in the copper industry and thus demonstrate that techno-economic and environmental indicators can be used simultaneously during preliminary process selection to influence decision making. The case study will also highlight some of the techno-economic and environmental trade-offs that exist between smelting and POX.

4.3 Case study: environmental and economic indicators

The three options under consideration will be compared on the basis of the following indicators:

Environmental – Energy consumption, global warming potential (GWP), acidification potential (AP) and water consumption

Economic – Net present value (NPV), internal rate of return (IRR), payback period (PBP) and return on investment

Eco-efficiency – Numerical Eco-efficiency indicators will used, and in this case study they are developed based on the NPV of each project (numerator) and the aforementioned environmental indicators (denominator), taken over a project life of 20 years.

4.4 Sensitivity analyses parameters

Sensitivity analyses will be performed on the SO_2 capture efficiency of smelters, the geographical location of the mine, the price of copper, treatment and refining charges, the price of acid and the cost of energy.

4.5 Summary of key assumptions and input data for the base case

- The techno-economic and environmental results that are presented in this case study are for a mine with an ore production capacity of 45,000,000 tonnes per annum. From this ore the downstream processes (concentrator, smelters and POX plant) produce approximately 300,000[51] tonnes of copper per annum. See Table 4.1 for the actual production flowrates.
- All sensitivity analyses will be performed against the base case scenario described in secctions 4.1 to 4.5.
- Emissions associated with offsite power generation are included.

---

[51] A smelter and a POX plant have different copper recoveries for a common copper concentrate, and a custom smelter will pay for a certain amount of copper in the concentrate. See Dreisinger (2006 a), Marsden (2008) and Soderstrom (2008). This affects the revenues received by the mine. The recoveries used in the case study are shown in Table 4.1.

- CO_2 emissions from carbonate destruction during concentrate smelting and leaching are included in the global warming potentials.
- SO_2 emissions from smelters are included in the acidification potential.
- Other key assumptions are shown in Table 4.1 to Table 4.3.

Table 4.1: Summary of model input parameters

Mineral	**Ore composition %**	**Ore (Tonnes / annum)**	**Recovery to concentrate (%)**	**Concentrate composition (%)**	**Concentrate (Tonnes / annum)**
	Input	Output	Input	Output	Output
FeAsS	0.015	6,750	50.0	0.333	3,380
$CuFeS_2$	1.245	561,000	90.0	49.8	505,000
FeS_2	4.000	1,800,000	10.0	17.8	180,000
Cu_2S	0.400	180,000	90.0	16.0	162,000
SiO_2	93.44	42,100,000	0.175	7.27	73,600
$CaCO_3$	0.400	180,000	2.00	0.356	3,600
$MgCO_3$	0.200	90,100	2.00	0.178	1,800
$FeCO_3$	0.300	135,000	2.00	0.267	2,700
H_2O				8.00	81,000
Total	**100**	**45,000,000**		**100**	**1,010,000**

Average production data	
Ore flowrate (tonnes / annum)	45,000,000
Ore grade (%)	0.75
Concentrate flowrate (tonnes / annum)	1,010,000
Overall Cu recovery in concentrator (%)	90.0
Cu recovery in integrated smelter (%)	96.8
Cu recovery in integrated POX plant (%)	98.6
Custom smelter payable copper (%)	96.65
Overall Cu recovery (%); ore to cathodes (Integrated smelting)	87.1
Overall Cu recovery (%); ore to cathodes (Integrated POX)	88.7
Integrated smelter copper cathodes (tonnes / annum)	294,397
Integrated POX plant copper cathodes (tonnes / annum)	299,900
Smelter SO_2 capture (%)	99.9
Comminution circuit – Primary crusher, SAG mill, ball mill	

Concentrate composition		
Element	Wet (%)	Dry (%)
Cu	30.0	32.6
As	0.150	0.170
Ca	0.140	0.150
S	30.2	32.8
Fe	23.7	25.7
O	11.4	4.62
C	0.100	0.100
Si	3.40	3.69
H	0.900	
Mg	0.050	0.060
Total	100	100

Table 4.2: Summary of main economic input parameters

Component		Unit	Component	Amount	Unit
Price of copper[52]	7,500	US$ / tonne Cu	Income Tax (Chile DL 600 option)	42	%
Premiums	115	US$ / tonne Cu	Mining Tax	5	%
Cost of Electricity	0.09	US$ / kWh	Inflation	3	%
Cost of Natural gas	182	US$ / tonne	Discount (hurdle) rate	10	%
Diesel price	1.38	US$ / L	Depreciation (Straight line)[53]	20	Years
Acid price	100	US$ / tonne acid			

---

[52] Price of copper sourced from Infomine Inc (2010), premiums from Reuters (2011 a), cost of electricity from Reuters (2011 b), cost of natural gas from USEIA (2011), price of diesel from MMI (2010), price of sulphuric acid from Bloomberg (2010), Chile income and mining taxes from PKF (2010), Chile inflation target from USDS (2010), discount rate from Smith (2002).

[53] Assumed for this study

Table 4.2 continued …

Total capital costs[54]				
Integrated smelter/refinery capital investment	5,262	US$ / annual tonne Cu		
Integrated POX/SX/EW capital investment	3,124	US$ / annual tonne Cu		
Operating costs[55]				
Mine/Concentrator/Custom smelting (C)	1,635	US$ / tonne Cu payable	74.14[56]	c/lb
Mine/Concentrator/Smelter/Refinery (I)	1,537	US$ / tonne Cu produced	69.74	c/lb
Mine/Concentrator/POX/SX/EW (I)	1,837[57]	US$ / tonne Cu produced	83.33	c/lb
Mine/Concentrator/POX/SX/EW (I)	1,707[58]	US$ / tonne Cu produced	77.42	c/lb
POX neutralisation savings	130	US$ / tonne Cu		
Smelter acid credit	317	US$ / tonne Cu produced		
Pox acid credit	280	US$ / tonne Cu produced		

---

[54] Refer to Appendix G for more details on the capital and operating costs.

[55] (C) – custom, (I) - integrated

[56] These costs are outcomes of the modelling performed in this study, and can be compared to the operating costs of actual mining companies shown in Figure 2.13. This is done to check the accuracy of the model outcomes. By-products such as gold, silver and molybdenum are not considered in this study. If they were considered the operating costs shown here would be lower on the cost curve due to the credits derived from the by-products.

[57] This is the operating cost of the Mine/Concentrator/POX/SX/EW process route when the acid raffinate from the POX plant is neutralised i.e. there is no heap leaching operation near the mine.

[58] This is the operating cost of the Mine/Concentrator/POX/SX/EW process route when its acid raffinate is not neutralised but is used in the heap leaching operation.

Table 4.2 continued …

Custom smelting contract pricing clauses[59]		**Unit**
Accountability	96.65	%
Subject to a unit minimum deduction	1	%
Acceptable concentrate Cu	30	%
Treatment charge	77	$ / DMT[60]
Refining charge	0.17	$ / kg payable Cu
As penalty	30	$ / DMT / % excess
As tolerance	0.2	%
Transport costs (to China)	80	$ / DMT

---

[59] Smelter accountability sourced from Soderstrom (2008), arsenic penalty from CODELCO (2009), treatment and refining charges from Trading Markets (2011) and Marsden (2007), transport costs estimated using data from Barr et al. (2005) and Sylwestrzak et al. (2009).

[60] Dry metric tonne

Table 4.3: Summary of logistical input parameters[61]

System	Mine & Concentrator in Chile	Custom smelter & refinery in China	Integrated smelter and refinery in Chile	Integrated POX/SX/EW in Chile
Company's final product		Cu concentrates	Cu cathodes	Cu cathodes
Port of loading concentrates or cathodes	N/A	Antofagasta, Chile	Antofagasta, Chile	Antofagasta, Chile
Port of discharging concentrates	N/A	Jiujiang, China	N/A	N/A
Port of discharging cathodes	N/A	N/A	Jiujiang, China	Jiujiang, China
Distance from concentrator in Chile to port in Chile (km by truck)	N/A	100	100	100
Distance from port of loading in Chile to port of discharge in China (km by ship)	N/A	19,364	19,364	19,364
Distance from port of discharge to smelter in China (km by truck)	N/A	100 (concentrates go to smelter)	0 (cathodes go straight to market)	0 (cathodes go straight to market)
Distance to market - cathodes (truck)	N/A	100	100	100

---

[61] All port and shipping data sourced from Farnel Capital (2010).

Table 4.3 continued …

System	Electricity generation efficiency	Mine & Concentrator in Chile	Custom smelter in China	Integrated smelter in Chile	Integrated POX in Chile	Sensitivity analysis: Integrated mine concentrator, smelter, POX in different location within Chile
	Region	Northern Chile	Central China	Northern Chile	Northern Chile	Central Chile
	Efficiency[62] (%)			Electricity mix (%)[63]		
Nat Gas	40	58.6	0	58.6	58.6	27.1
Coal	35	33.5	74	33.5	33.5	8.9
Nuclear	35	0.0	0	0.0	0.0	0.0
Hydro	85	0.4	26	0.4	0.4	52.3
Renew	30	0	0	0	0	0.2
Oil	35	7.5	0	7.5	7.5	9.7
Biomass	35	0	0	0	0	1.8
Total		100	100	100	100	100

---

[62] Electricity generation efficiency data sourced from IEA (2008), Woodbank Communications (2005). Where:

$$Electricity\ generation\ efficiency = {}^{Electricity\ produced\ by\ power\ plant}/_{Energy\ input\ into\ power\ plant}$$

[63] Chile and China's electricity mixes were sourced from Eggers (2009) and Hou et al. (2010), respectively.

4.6 Base case results and discussion

4.6.1 Economic performance results

A summary of the mine's economic performance results for each of the three concentrate processing options is displayed in Table 4.4.

Table 4.4: Summary of the mine's economic position with each option

	Custom smelter – China	**Integrated smelter – Chile**	**Integrated POX – Chile**
Capital costs $ Million	N/A	1,644	940
Revenue $ Million / annum	3,334	3,479	3,531
Costs $ Million / annum	885	812	902
EBITDA[64] $ Million / annum	2,449	2,667	2,629
NPV ($ Million)	8,345	7,797	8,201
Relative (Incremental) NPV ($ Million)	N/A[65]	-547	-144
IRR [66] (%)	N/A	3.74	7.38
ROI (%)	N/A	6.90	11.40
PBP (Years)	N/A	11.39	8.08

The total capital investment of a smelter and refinery complex would be greater than the capital investment required for a POX, solvent extraction and electrowinning plant ($ 1,644 million vs. $ 940 million). Since the two processes have similar copper recoveries from a common concentrate (96.8 and 98.6%, respectively), this result is a reflection of the specific capital costs that are shown in Table 4.2. The revenues generated through integrating vertically with a smelter or POX plant would be greater than the revenues generated from selling concentrates to custom smelters in China. This is because the mine would receive premiums for the sale of copper cathodes and economic credits for the acid produced onsite when it integrates vertically (see Figure 4.1 and Table 4.2), whereas if concentrates were

---

[64] EBITDA – Earnings before interest taxes depreciation and amortisation, IRR – internal rate of return, ROI – return on investment, PBP – payback period

[65] Not applicable. The custom smelting option is the reference process for calculating the relative / incremental cash flow.

[66] IRR, ROI and PBP are calculated using the incremental cashflows that would arise from investing in a smelter and a POX plant.

sold to custom smelters, the premiums and the acid credit would remain with the custom smelters (see Figure 2.16). Furthermore, the recoveries of copper in an integrated smelter and POX plant are greater than the payable copper percentage from custom smelters (96.8 and 98.6% vs 96.65%).

The mine would have the highest operating costs when it integrates vertically with a POX plant next to the mine ($ 902 million / annum), followed by custom smelting in China ($ 885 million / annum) and integrated smelting next to the mine in Chile ($ 812 million / annum), consecutively. Therefore, on the basis of operating costs only, the mine would not benefit from integrating vertically through a POX plant over custom smelting in China. However, as discussed previously, integrating vertically with a POX plant or a smelter would result in higher revenues. The significance of the increase in revenues is reflected in the mine's earnings as displayed in Table 4.4, which shows that the mine would generate higher earnings if it integrated locally with a smelter or POX plant, compared to custom smelting in China (see EBITDA in Table 4.4). Although there would be an increase in the mine's earnings if it integrated vertically, the incremental cashflows obtained from integrating with a POX plant or a smelter should be sufficient to pay back the capital investment required to construct either the smelter or the POX plant. In this study a discounted cash flow analysis of the mine's cash flows, over the current estimate of the project's life (20 years), was performed to assess the economic feasibility of integrating vertically compared to custom smelting in China. The results of the discounted cash flow analysis are also presented in Table 4.4.

A comparison of the mine's NPVs for each option shows that selling concentrates to custom smelters in China is the best option (from economic perspectives) since this option has the highest NPV. When the mine acts as a custom mine, selling concentrates to custom smelters in China, its earnings do not have to support capital repayment because no capital investment would be required, unlike when it constructs its own internal smelter or POX plant. Following the NPV of the custom smelting option are the NPVs of the POX plant and integrated smelting options, consecutively. Therefore, integrating vertically would not be economically viable since the incremental NPVs or benefits of investing in integrated smelting and POX plants would be negative relative to custom smelting in China (- $ 547 and - $ 144 million, respectively).

The IRRs of constructing a POX plant or a smelter are 7.38% and 3.74%, respectively. These rates are lower than the 10% discount rate (Smith, 2002) assumed for each process in current study (see Table 4.2), therefore, these processing options would not be worth the investment in the long run[67].

---

[67] Damodaran (2011) estimates that the weighted average cost of capital (WACC) of the mining and metals sector is 9.62% (using a sample of 69 mining companies); therefore, these IRRs would also be less than the industry's current minimum acceptable hurdle rate (WACC).

No reliable information was available at the time of writing on typical returns on investment (ROIs) and acceptable payback periods (PBPs) that mines would expect for such projects.

It is also important to remind the reader that the economic outcomes presented in this thesis are not universal, but are for the configurations that are shown in Figure 4.1. Other configurations and projects would have to be analysed independently, using data and information that are specific to those particular projects.

As an illustration of this point, the economic outcomes of the POX process are displayed in

Table 4.5 for the case in which there is no heap leaching operation onsite (or a local market for the acid produced from the POX plant), i.e. the POX process does not receive an economic credit for its acid, and costs associated with acid neutralisation have to be taken into account. The base case results are also shown here for ease of reference when comparing the two scenarios.

Table 4.5: Economic performance of the POX process when there is no heap leaching operation onsite

	Custom smelter – China	**Integrated smelter – Chile**	**Integrated POX with heap – Chile (Base case)**	**Integrated POX with no heap – Chile**
Capital costs $ Million	N/A	1,644	940	994
Revenue $ Million / annum	3,334	3,479	3,531	3,402
Costs $ Million / annum	885	812	902	951
EBITDA[68] $ Million / annum	2,449	2,667	2,629	2,451
NPV ($ Million)	8,345	7,797	8,201	7,739
Relative (Incremental) NPV ($ Million)	N/A	-547	-144	-605

If the mine invested in a POX plant and there was no heap leaching operation available to utilise its acid, the mine’s NPV would be $ 7,739 million, which is less than any of the NPVs of the base case configurations. The mine’s NPV would decrease by $ 605 million relative to custom smelting in China over the life of the project. Comparing the relative NPVs of the POX process, operated with and without heap leaching, shows that the POX process is

---

[68] EBITDA – Earnings before interest taxes depreciation and amortisation, IRR – internal rate of return, ROI – return on investment, PBP – payback period

more economically competitive to custom smelting when it is operated in the former configuration than it is when operated in the latter, which is an observation that agrees with the opinions of McElroy and Young (1999), and Dreisinger (2005), concerning the economic feasibility of the POX process. When there is a heap leaching operation onsite, the acidic raffinate from the POX plant can be used to meet the acid requirements of the heap, thereby generating an economic credit for the POX project; in addition, neutralisation costs are avoided. However, when there is no heap leaching operation onsite (or near the POX plant), the acidic raffinate has to be neutralised, which increases the capital and operating costs of the POX process and eliminates the acid credit.

In summary, this analysis shows that the process rankings that were shown in base case are not universal, but vary according to the circumstances surrounding a particular operation.

Summary

A purely economic assessment of the concentrate processing options under consideration in this thesis has shown that custom smelting in China is more economically attractive than investing in a local smelter or a POX plant in Chile. However, a review of the literature suggests that smelters and refineries with capacities of 300,000 tonnes of copper per annum can, together, create employment for approximately 1,000 people, which would be a social benefit for Chile if the mine integrated vertically (USCOTA, 1988; Peacey et al., 2003; Aurubis, 2010). No specific numbers were available pertaining to employment generation from the POX process.

Although, the results presented in this study showed that custom smelting was the best option available to the mine, by way of illustration through the POX process, it was demonstrated that the results shown in this assessment do not apply to every mine, and that each mine would have to perform its individual economic assessment based on its own set of techno-economic and logistical circumstances.

4.6.2 *Environmental performance results*

The reader is referred back to section 4.5 to view the underlying assumptions made to generate the environmental performance results presented in this section.

Energy consumption

Figure 4.2 graphically illustrates the energy consumption of each process route divided into subsystems, from ore extraction up to the point were copper cathodes would reach the market in China[69, 70].

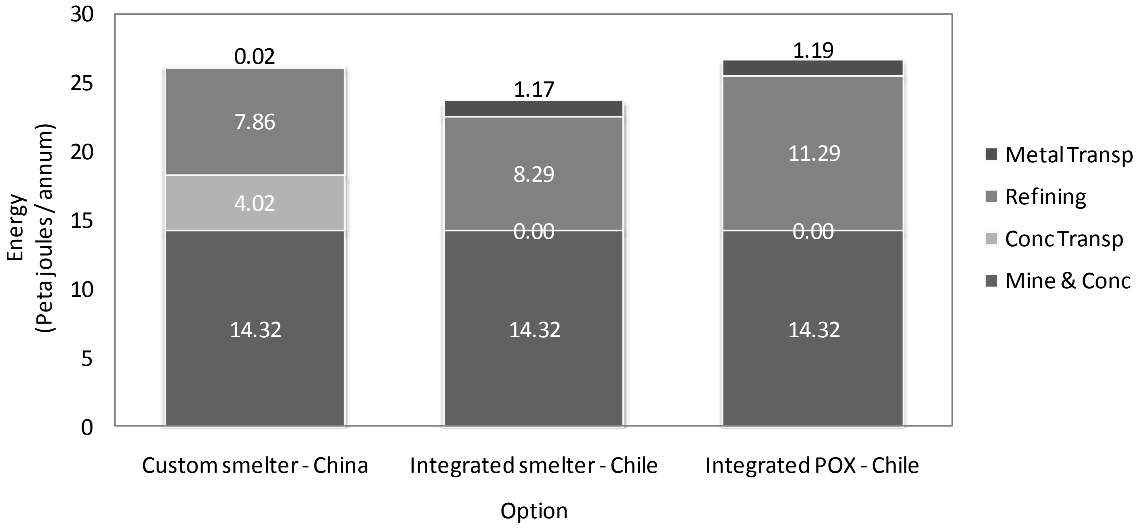


Figure 4.2: Annual energy consumption for each process option

Integrating vertically with a POX plant would result in the largest energy consumption of the three options under consideration. This option would actually increase the mine's total energy consumption by 2.19% relative to custom smelting in China. In contrast, integrating with a smelter in Chile would decrease the mine's energy consumption by 9.33%, making this route the best option when solely considering energy consumption.

Since the mine and concentrator are located in Chile and are common to all three concentrate processing options, it follows that their sectional contributions towards the total energy consumption of all the processing options are equal at 14.32 petajoules / annum. This subsystem has the largest energy consumption in the entire ore to cathode chain. Transporting concentrates to custom smelters in China by truck and ship consumes 4.02 petajoules / annum, which is equal to 15.3% of the total energy consumption of that processing route. Integrating vertically and transporting cathodes instead to the market in

---

[69] Petajoule = 1^15

[70] The total energy consumption results shown in Figure 4.2 correspond to specific energy consumption values of 89.20, 80.76 and 89.35 MJ / kg Cu, in the order shown in Figure 4.2. These values have similar orders of magnitude to the energy consumption data shown in Table 2.5.

China results in lower energy consumption, i.e. ~ 1.20 petajoules / annum, which is approximately 30% of the energy required to transport the concentrates.

Comparing the refining subsystems of the smelting options in Figure 4.2 shows that hydrometallurgical processing (POX) consumes more energy than pyrometallurgical processing (11.29 petajoules and 8.29 petajoules, respectively). However, it is important to remind the reader about the context within which the results in the current study are presented. These results are inclusive of energy losses (inefficiencies) associated with electricity generation in the power plant (see section 4.5). When these losses are excluded, smelting consumes more energy than POX (5.09 petajoules and 4.32 petajoules, respectively). However, when the inefficiencies associated with electricity generation are taken into consideration, POX consumes more energy because it is almost entirely depended on electricity as an energy source, owing largely to electrowinning and not the actual leaching process itself (see Figure 2.19).

Global warming potential

The annual global warming potentials (GWP's) of each processing route, divided by subsystem, are shown in Figure 4.3[71].

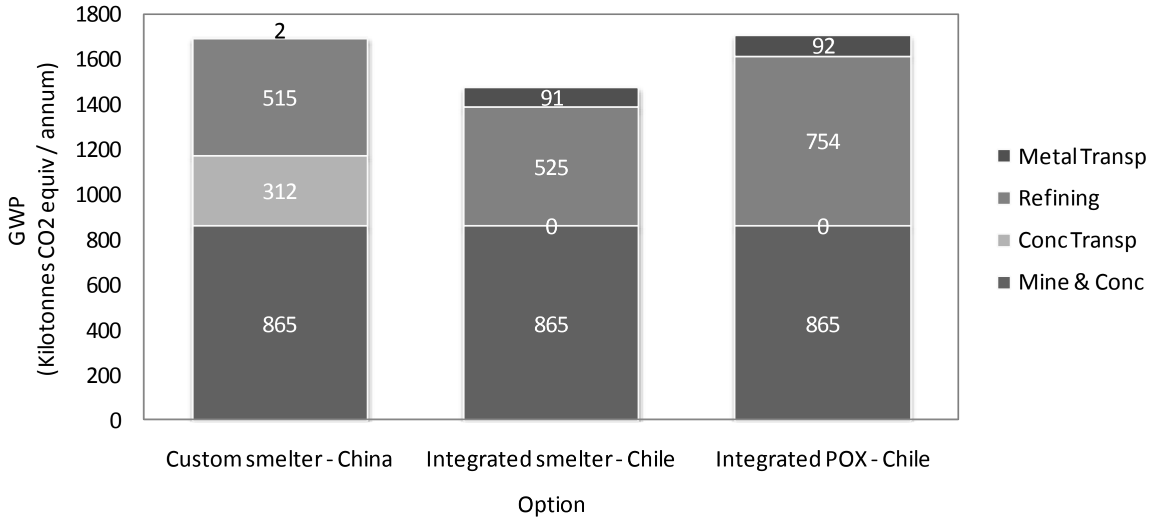


Figure 4.3: Annual global warming potential for each process option

Integrating with a smelter in Chile would result in a 12.6% decrease in total global warming potential, compared to custom smelting in China, owing largely to the reduction in global warming potential when cathodes are transported to China instead of concentrates. In contrast, integrating vertically with a POX plant would increase the mine's global warming potential by 1.00%, making POX the least desirable option.

---

[71] The total global warming potentials shown in Figure 4.3 correspond to specific global warming potentials of 5.76, 5.03 and 5.70 kg CO_2 equivalent / kg Cu, in the order shown in Figure 4.3, and are within the global warming potential range shown in Table 2.7, i.e. 3.3 – 15 kg CO_2 equivalent / kg Cu.

Comparing the refining subsystems of smelting in China and Chile shows that a smelter and refinery in northern Chile would contribute more to global warming than a smelter and refinery in central China (525 and 515 kilotonnes CO_2 equivalent / annum respectively). This is because mines and smelters in northern Chile obtain electricity from a power grid that has an electricity generation mix that consists of 99.6% carbon containing fuels (see Table 4.3 for the detailed mixes), whereas custom smelters in central China receive electricity from a grid that uses 72% carbon sources[72].

Another important observation can be made from Figure 4.3 when comparing the smelting and POX refining subsystems of operations located in Chile, only. Treating concentrates in a POX plant in Chile would contribute more to global warming than smelting in Chile (754 and 525 kilotonnes CO_2 equivalent / annum). Since POX is highly dependent on electricity as an energy source, in a location which is greatly dependent on carbon based sources for electricity generation, and taking into account electricity generation losses, it is likely that POX would contribute more to global warming than smelting[73].

---

[72] It was assumed that similar smelting and refining technologies would be used in China and Chile.

[73] Taking into consideration the relative energy consumptions of both processes, as previously discussed.

Acidification potential

Figure 4.4 displays the acidification potentials of the three options available to the mine[74].

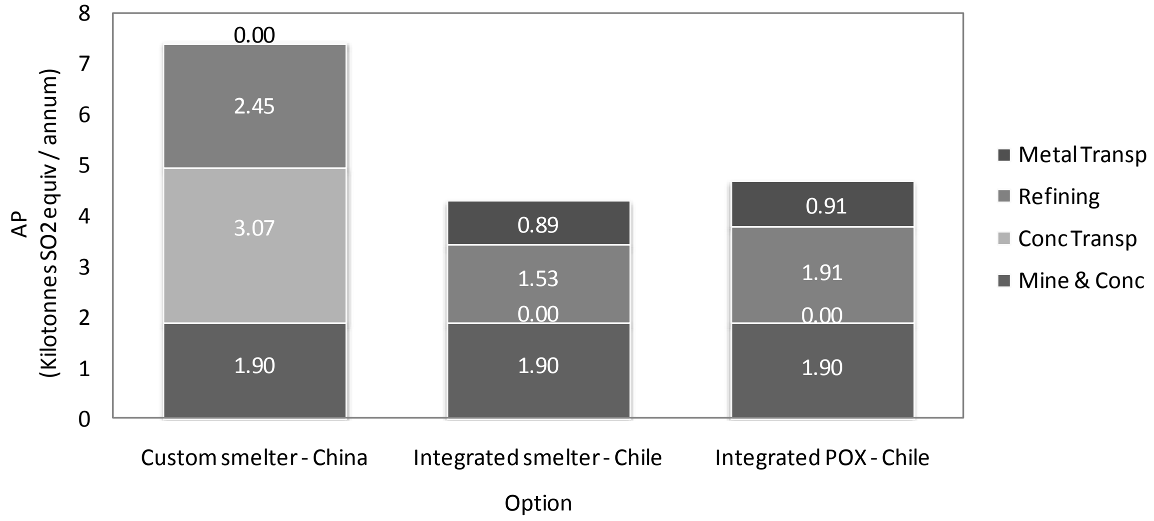


Figure 4.4: Annual acidification potential for each process option

In section 2.2.4 it was shown that smelter SO_2 capture efficiency varies widely, between 60 – 99.9%, although many smelters are now in the higher end of this range (Ramachandran et al., 2003). For the purposes of the current study it was assumed that if the mine were to invest in a new smelter in Chile, it would invest in a smelter with a high SO_2 capture efficiency (99.9%) and that it is currently selling concentrates to custom smelters in China with similar SO_2 capture efficiencies and will continue to do so in the future. The effect of SO_2 capture efficiency is analysed in section 4.7.1.

Based on the base case assumptions, integrating vertically with a new smelter in Chile would result in the lowest acidification potential, compared to the other options, and this option would be followed by a POX plant in Chile and custom smelting in China, in that order.

The contribution to acidification potential of concentrate transportation to China (3.07 kilotonnes SO_2 equivalent / annum) is higher than the acidification potential associated with all the other subystems shown in Figure 4.4. This is because heavy fuel oil used in ships typically contains approximately 1.5% sulphur, which is higher than the sulphur content of the diesel used in trucks and for other general purposes, which in this study was assumed

---

[74] The total acidification potentials shown in Figure 4.4 correspond to specific acidification potentials of 0.025, 0.015 and 0.016 kg SO_2 equivalent / kg Cu, in the order shown in Figure 4.4. These values have similar orders of magnitude to the values shown in Table 2.7, i.e. 0.046 – 0.2 SO_2 equivalent / kg Cu. Higher SO_2 capture efficiencies were used for modern smelters in this study, i.e. 99.9% whereas previous studies range from 5 – 99%.

to contain 15 ppm sulphur (ultra low sulphur diesel - ULSD) (Chevron, 2011; Kamarainen, 2010).

Water consumption

Total fresh water consumption rates, divided into subsystems, are displayed in Figure 4.5[75].

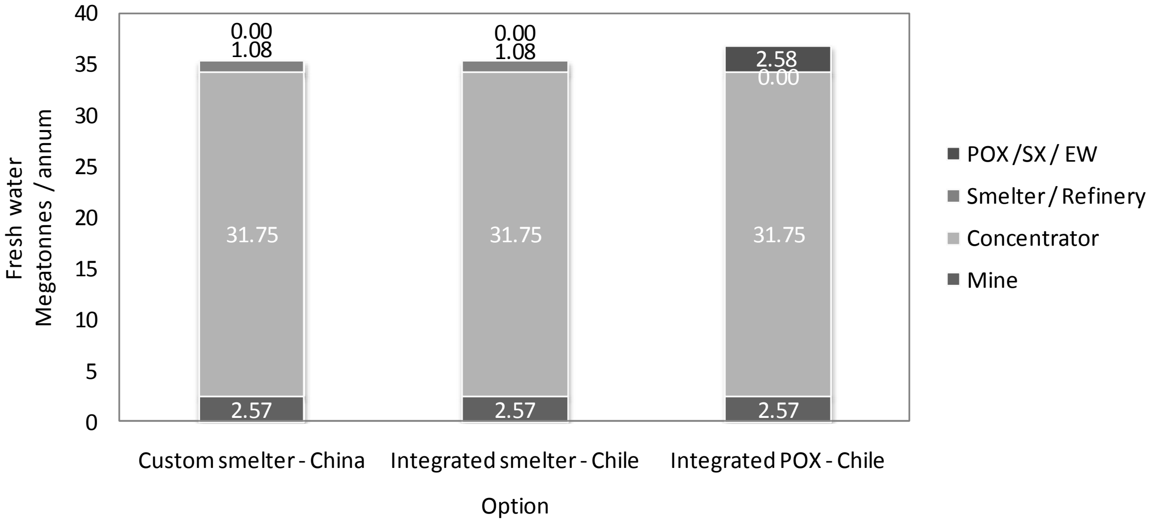


Figure 4.5: Annual water consumption for each process option

For all the three options compared in this study, the concentrator consumes approximately 90% of the total water required to produce copper cathodes from a copper ore with a grade of 0.75% copper (see Table 4.1). This is significantly more than the water required on the mine site (~ 7%), and the smelting and POX sites (~ 3%). This shows that the water required for smelting and POX is relatively insignificant, compared to the water for the upstream processes, especially the concentrator. It was assumed that smelting concentrates in Chile or China would have similar water consumption rates.

Figure 4.5 also shows that the POX process consumes 2.39 times the amount of water consumed by the smelting options (2.58 megatonnes / annum, compared to 1.08 megatonnes / annum). A possible reason for this result is that it was assumed that the steam produced in the flash tank, after the POX autoclave, is vented directly to the atmosphere (see Figure 3.10). However, when it was assumed that the steam is used internally as an energy source and that the water derived from it is re-used (or is used by external industrial plants, with the POX plant receiving water credits), the POX water

---

[75] The specific fresh water consumption of the concentrator shown in Figure 4.5 is 0.71 tonnes of fresh water / tonne of ore (0.75% Cu). This is within the range reported by COCHILCO (2008), i.e. 0.3 – 2.1 tonnes of fresh water / tonne of ore (0.75% Cu). The specific smelter & refinery water consumption is 3.06 tonnes of fresh water / tonne of Cu, which is in the same order of magnitude as the smelter water consumption rate reported by COCHILCO (2008), i.e. 3.60 tonnes of fresh water / tonne of Cu.

consumption decreases from 2.58 to 0.48 megatonnes / annum, which is equivalent to an 81.4% decrease in fresh water consumption.

4.6.3 Eco-efficiency performance

The eco-efficiencies of the three processing options available to the mine are presented in this section. The eco-efficiency indicators were calculated by dividing the net present value (NPV) of each project by each environmental indicator (energy consumption, GWP, AP, fresh water consumption) taken over the estimated project life (20 years). The y axis in Figure 4.6 represents the economic contribution ($ million) towards the NPV of each project option for every petajoule of energy used to extract copper from a mine in Chile up to the point were copper cathodes reach the market in China, during the current estimated project lifespan (20 years). Process options with high eco-efficiencies are desirable.

Energy consumption eco-efficiency

Figure 4.6 displays the energy eco-efficiencies of the three processing options.

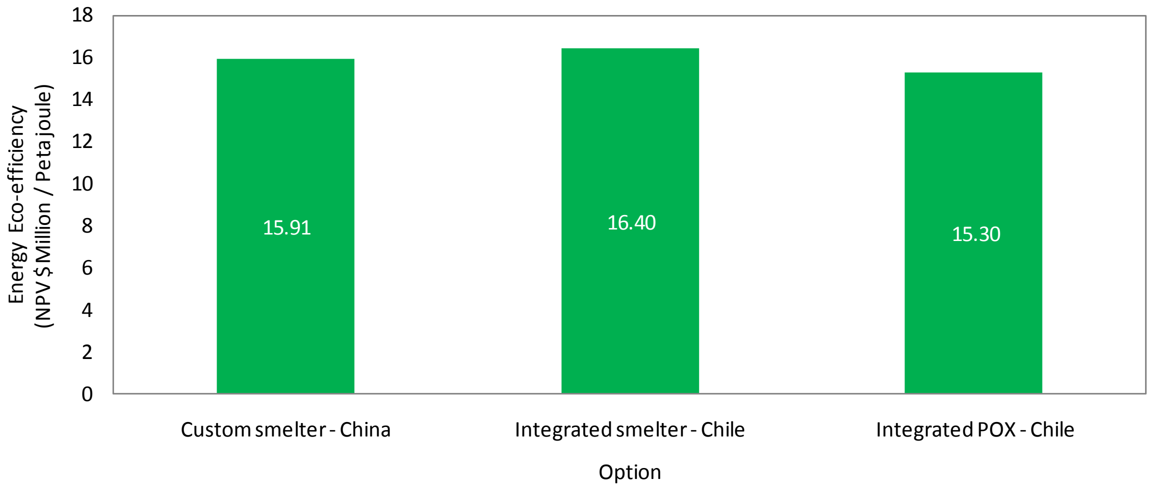


Figure 4.6: Energy eco-efficiency

An integrated smelter in Chile would be the most energy eco-efficient option available to the mine, followed by custom smelting in China and an integrated POX plant in Chile, in that order. A similar ranking was observed when a purely environmental analysis on energy consumption was performed (Figure 4.2). However, a different ranking was observed when purely economic objectives (Table 4.4) were considered. Custom smelting in China was the best option (highest NPV), followed by a POX plant and integrated smelting in Chile, consecutively. Figure 4.7 displays the global warming potential eco-efficiencies of the three options available to the mine.

Global warming potential eco-efficiency

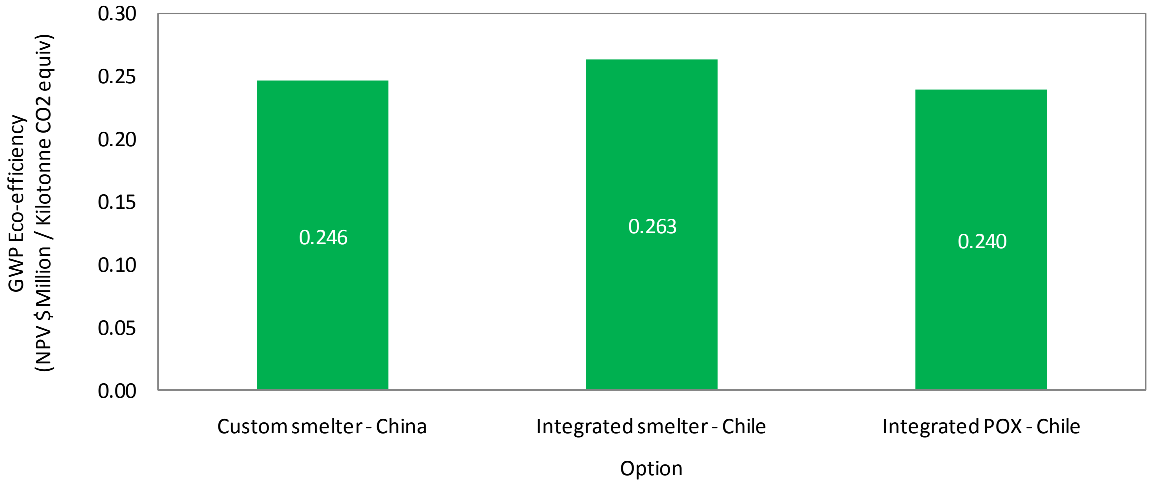


Figure 4.7: Global warming eco-efficiency

Similarly to the outcomes of the energy consumption eco-efficiency analysis, integrating vertically with an internal smelter in Chile would be the most eco-efficient option available to the mine, this time with regard to global warming potential (GWP). This option would contribute $ 0.263 million to the project's NPV for each kilotonne of CO_2 equivalent released during the project's 20 year lifespan. Operating with a POX plant in Chile would be the least eco-efficient option, however the difference between the eco-efficiencies of operating with a POX plant and operating as a custom mine are marginal (0.06 $ million / kilotonne of CO_2 equivalent).

Acidification potential eco-efficiency

Figure 4.8 displays the acidification potential eco-efficiency indicators.

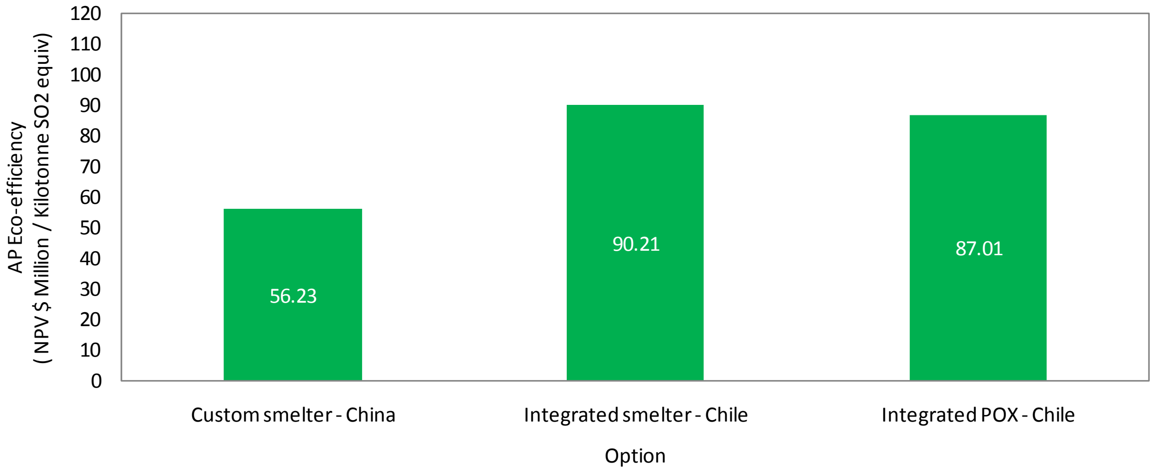


Figure 4.8: Acidification potential eco-efficiency

Again, similarly to the outcomes of the energy consumption and global warming eco-efficiency comparisons, integrating vertically with a smelter in Chile would be the most eco-efficient option, contributing \$ 90.21 million to the integrated smelter project's NPV for each kilotonne of SO_2 equivalent released during the projects lifespan. Integrating vertically through a POX plant follows closely (\$ 87.01 million / kilotonne of SO_2 equivalent), and custom smelting thereafter, with the lowest acidification potential eco-efficiency (\$ 56.23 million / kilotonne of SO_2 equivalent). The large difference observed between the eco-efficiencies of the custom smelting option and the integrated options is a strong reflection of the results shown in Figure 4.4, which showed that the acidification potential of the custom smelting option was notably higher than the acidification potentials of the integrated options.

Water consumption eco-efficiency

Figure 4.9 displays fresh water consumption eco-efficiencies for the three options under consideration.

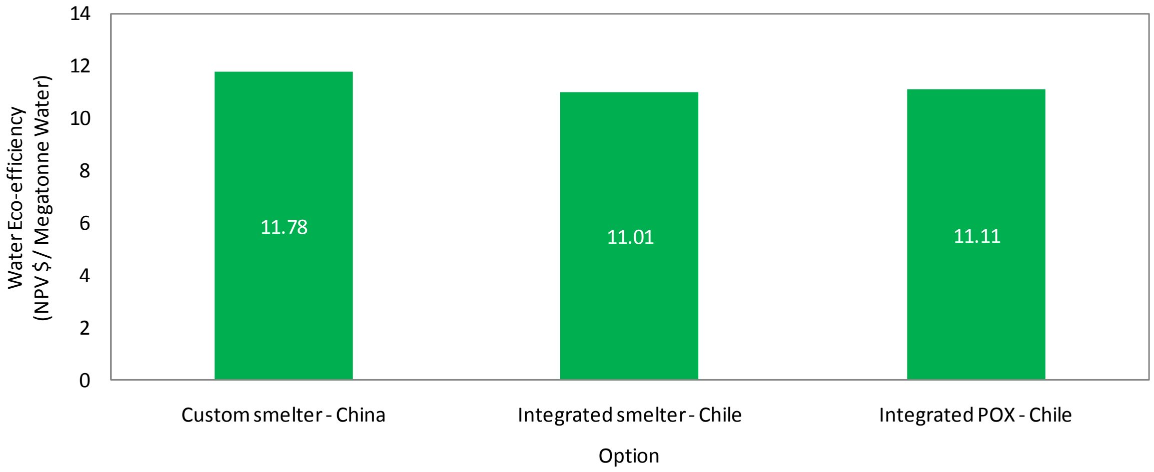


Figure 4.9: Fresh water consumption eco-efficiency

In contrast to the other eco-efficiency categories shown above, integrated smelting in Chile would have the lowest fresh water eco-efficiency, whereas custom smelting in China would have the highest water eco-efficiency. In the purely environmental analysis, as summarised in Figure 4.5, selling concentrates to smelters in China and smelting the concentrates in Chile would result in equal water consumption rates. However, because the NPV of the option which involves selling concentrates to smelters in China is higher than the NPV of an integrated smelter in Chile, selling concentrates to custom smelters in China is more eco-efficient with regard to water consumption. The custom smelting project would contribute \$ 11.78 million to the mine's NPV for each megatonne of water consumed in the production of copper cathodes, whereas an integrated smelter would contribute \$ 11.01 million to the mine's NPV for each megatonne of water consumed during the estimated project life.

4.6.4 Summary

As a summary of the preceding discussion, Table 4.6 ranks the processes from the most eco-efficient to the least eco efficient option. A single, overall score was given to each process, and the process with the highest score was considered to be the most eco-efficient option, whereas, the option with the lowest score was considered to be the least eco-efficient option. To calculate the overall score for each process, the eco-efficiency indicators that were presented in section 4.6.3 were normalised by dividing each eco-efficiency indicator by the copper industry average eco-efficiency indicators for each category, which for the purpose of this study, pending global eco-efficiency data, were assumed to be the average eco-efficiency indicators of the two smelting options considered in this study, since these options are already practised widely in industry, whereas, POX is not. For each processing option, the normalised scores were then multiplied by weighting factors which represented the significance of each eco-efficiency category in the final decision, and the products were summed up to give an overall score for each option. The weighting factors used in this study are shown in Table 4.6, and they reflect the views of the author on the relative significance of each eco-efficiency category in this decision making scenario, and they are used in the present hypothetical case study for the sake of completeness. Valuation is subjective (Crosby-Forbes, 1999) and there is no single, widely accepted method for its application (Azapagic, 2005). Different stakeholders (e.g. businesses, the general public, academics etc) will have different preferences; however, such preferences can easily be accommodated in the model.

In assigning weights to the eco-efficiency indicators used in this study, the author took the following into consideration:

- The majority of Chile's copper mines and the bulk of the world's known copper reserves are located in very dry areas, i.e. within the Atacama desert, therefore, water availability is a major issue for Chile's copper mining industry (COCHILCO, 2008; Gaete, 2009).
- At the time of writing, there were concerns from Chilean companies over rising energy costs (Bodzin, 2011; Seitz, 2011 a; Seitz, 2011 b). The increase in energy costs was caused by an increase in international crude oil prices and a drought that affected the country's hydroelectricity production (Seitz, 2011 b).
- Global warming is an international environmental problem (United Nations, 2010).

Table 4.6 displays the normalised eco-efficiency indicators, the weighting factors, the overall eco-efficiency scores and the overall rankings of the processing options.

Table 4.6: Final ranking of processes based on eco-efficiency indicators

Eco-efficiency category	**Weight**	**Custom smelter – China**	**Integrated smelter – Chile**	**Integrated POX – Chile**
		Normalised eco-efficiency		
Energy	0.3	0.98	1.02	0.95
GWP	0.3	0.97	1.03	0.94
AP	0.1	0.77	1.23	1.19
Water	0.3	1.03	0.97	0.97
Total	1.0			
Overall eco-efficiency score		**0.97**	**1.03**	**0.98**
Overall rank		**Worst**	**Best**	**Middle**

With the exception of the normalised acidification potential eco-efficiency category, all the normalised eco-efficiency indicators of the integrated options are within ± 7% of the normalised custom smelting eco-efficiencies. However, the difference between the normalised acidification potential eco-efficiency of the custom smelting option and the integrated options is approximately + 55%, relative to custom smelting, which influences the decision in favour of the integrated options. Based on the selected weighting factors of each eco-efficiency category, integrating vertically through a smelter in Chile would be the most eco-efficient option available to the mine, followed thereafter by an integrated POX plant in Chile and custom smelting in China, consecutively (see overall scores). However, the overall scores of the integrated options are within + 7% of the overall score of the custom smelting option.

Assuming acidification potential was also a major environmental issue in Chile and increasing its weighting factor to a point such that the weighting factors are all equal, i.e. 0.25 in each category, sways the decision more in favour of the integrated options since they are more eco-efficient in acidification potential than the custom smelting option. This is shown in Table 4.7.

Table 4.7: Final ranking of processes with different weighting factors

Eco-efficiency category	**Weight**	**Custom smelter – China**	**Integrated smelter – Chile**	**Integrated POX – Chile**
			Normalised eco-efficiency	
Energy	0.25	0.98	1.02	0.95
GWP	0.25	0.97	1.03	0.94
AP	0.25	0.77	1.23	1.19
Water	0.25	1.03	0.97	0.97
Total	1.0			
Overall eco-efficiency score		**0.94**	**1.06**	**1.01**
Overall rank		**Worst**	**Best**	**Middle**

Increasing the weighting factor of acidification potential in the decision making scenario does not change the overall rankings of the processes, but, it does increase the differences between the overall scores of the custom smelting option and the integrated options i.e. from + 6.19% and + 1.03%[76], relative to custom smelting, when using the initial weighting factors, to + 12.77% and + 7.45% when using the current weighting factors.

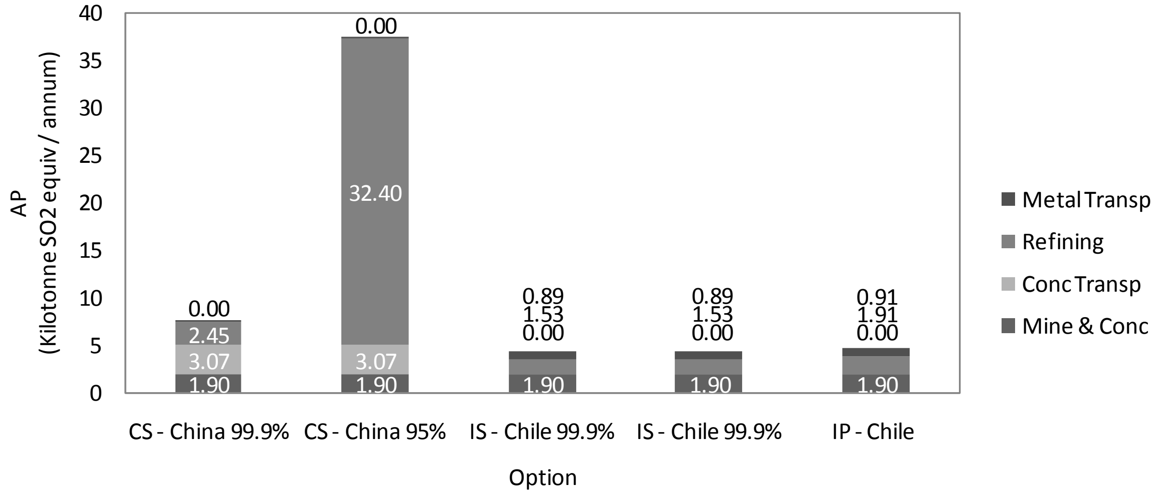
---

[76] The percentages refer to the integrated smelter and POX plants, respectively.

4.7 Sensitivity analysis

4.7.1 Smelter SO_2 capture efficiency

This section examines the effect of SO_2 capture efficiency on the acidification potential of smelters. In the base case analysis it was assumed that in future, the mine will sell concentrates to custom smelters in China with high SO_2 capture efficiencies (99.9%) and it was decided that if the mine integrated vertically through a smelter in Chile, it would only invest in a smelter with a high SO_2 capture efficiency (99.9%). To study the effect of SO_2 capture efficiency on the environmental performance of smelters, it is assumed in this analysis that the mine will sell concentrates to smelters in China with an average SO_2 capture efficiency of 95%. Figure 4.10 shows the resultant acidification potentials.



Key: CS – Custom smelter in China, IS - Integrated smelter in Chile, IP – Integrated POX in Chile

Figure 4.10: SO2 capture efficiency of custom smelters in China

The mine's total acidification potential would increase by 403% relative to the base case acidification potential if it sold its concentrates to custom smelters with a 95% SO_2 capture efficiency. The eco-efficiency rankings of the processes would not change in order compared to the base case rankings, i.e. integrated smelting would remain the most eco-efficient option and custom smelting the worst option, however, the differences between the overall scores of the integrated options and the custom smelting option increase (see Table 4.6 and Table 4.8).

Table 4.8: Final rankings when custom smelters capture 95% of their SO_2 emissions

Eco-efficiency category	Weight	Custom smelter – China	Integrated smelter – Chile	Integrated POX – Chile
		Normalised eco-efficiency		
Energy	0.3	0.98	1.02	0.95
GWP	0.3	0.97	1.03	0.94
AP	0.1	0.22	1.78	1.72
Water	0.3	1.03	0.97	0.97
Total	1.0			
Overall eco-efficiency score		**0.92**	**1.08**	**1.03**
Overall rank		**Worst**	**Best**	**Middle**

The differences between the overall eco-efficiency scores of the custom smelting and integrated options increase from + 6.19% and + 1.03%[77], in the base case, to + 17.39% and + 11.96%, in this case, relative to custom smelting. Ultimately, this influences the decision in favour of investing in a local smelter near the mine.

4.7.2 Sensitivity analysis: Geographical location

This section examines the effect of geographical location, within Chile, on the environmental performance of the mine described in the base case analysis. Chile distributes its electricity through four grids, SING, SIC, Aysen and Megallenes, of which SING and SIC are predominant and distribute up to 99% of Chile's electricity (Pimentel, 2007; Eggers, 2009; Mallory, 2008). The base case presented results for a mine that is located in northern Chile (Table 4.3), which has its electricity supply coming from the SING grid, whereas in this sensitivity analysis, environmental performance results are presented for the same mine, assuming it was located in central Chile, having its electricity supplied from the SIC grid. For the purposes of this illustration, it was assumed that the distances from the mine to the ports are equal from central and northern Chile; in addition, it was also assumed that the distances from the ports in Chile to the port in China are equal. Therefore, only the effects of different electricity grids are analysed here.

Table 4.9 shows the electricity generation mixes of the SIC and SING grids.

---

[77] The percentages refer to the integrated smelter and POX plants, respectively.

Table 4.9: Electricity generation mixes of SING and SIC grids (Eggers, 2009)

System	Base case analysis. Northern Chile grid (SING)	Sensitivity analysis. Central Chile grid (SIC)
	Electricity generation mix (%)	
Region	North	Central
Nat Gas	58.6	27.1
Coal	33.5	8.9
Nuclear	0.0	0.0
Hydro	0.4	52.3
Renew	0	0.2
Oil	7.5	9.7
Biomass	0	1.8
Total	100	100

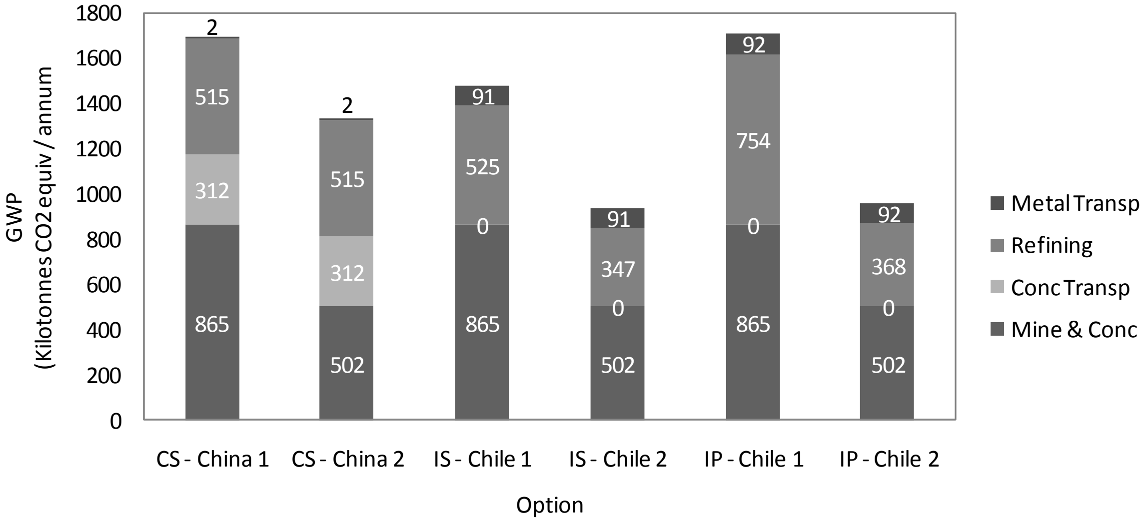
Figure 4.11 compares the global warming potentials of the mine in both locations.

If it were assumed that the mine was located in central Chile, its total global warming potential would be lower than the global warming potential that resulted from assuming that the mine was located in northern Chile, regardless of the processing option selected (Figure 4.11). This is because central Chile's electricity generation is less dependent on carbon containing fuels, compared to northern Chile, which is almost entirely dependent on carbon containing sources for electricity generation (Table 4.9).

If the mine were located in central Chile as opposed to northern Chile, the global warming potential of its mining and concentration subsystem would be equal to 41.9% of the equivalent subsystem in northern Chile. A smelter in central Chile would have a global warming potential that is equal to two thirds of the global warming potential of an equivalent smelter in northern Chile, and a POX plant located in central Chile would have a global warming potential that is about half of the global warming potential of an equivalent POX plant located in northern Chile. The differences between the POX plants are larger than the differences between the smelters because POX plants are more dependent on electricity as an energy source compared to smelters[78]. Therefore, any changes made to the electricity generation mix have a larger impact on POX plants.

---

[78] Based on the energy data used in this study, see Appendix E.

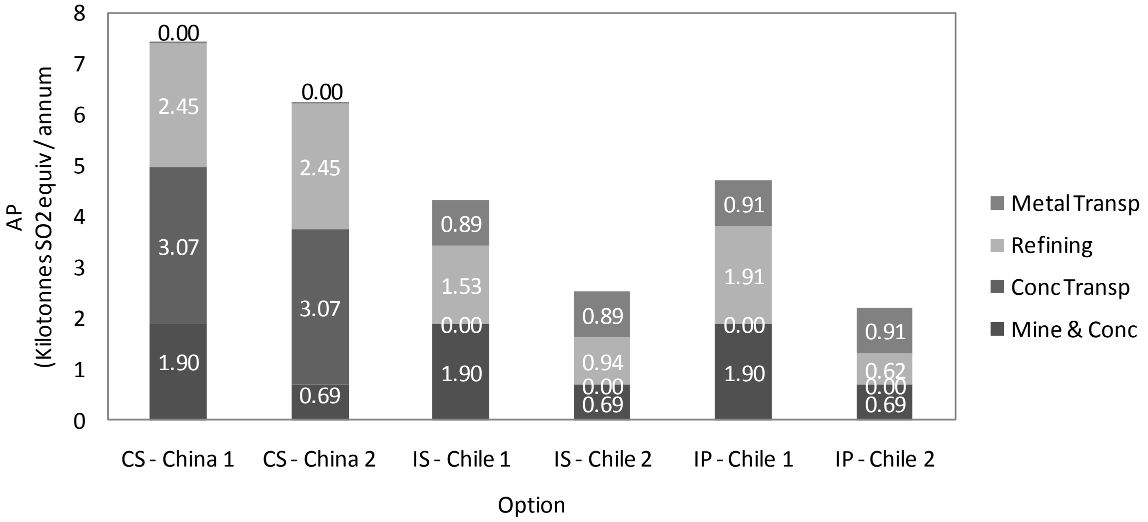


Key: CS – Custom smelter in China, IS - Integrated smelter in Chile, IP – Integrated POX in Chile

1 – Northern Chile (SING grid – Base case), 2 – Central Chile (SIC grid)

Figure 4.11: Global warming potentials of the mine in northern and central Chile

Figure 4.12 displays the acidification potentials of the mine for the cases in which it is located in northern and central Chile. Similarly to the global warming potentials, if it were assumed that the mine were located in central Chile, it would have a lower acidification potential regardless of the downstream processing option selected.



Key: CS – Custom smelter in China, IS - Integrated smelter in Chile, IP – Integrated POX in Chile

1 – Northern Chile (SING grid – Base case), 2 – Central Chile (SIC grid)

Figure 4.12: Acidification potentials of the mine in northern and central Chile

Table 4.10 shows the overall eco-efficiency scores for the each of the processes when the mine is located in central Chile.

Table 4.10: Final process rankings when the mine is located in central Chile.

Eco-efficiency category	Weight	Custom smelter – China	Integrated smelter – Chile	Integrated POX – Chile
		Normalised eco-efficiency		
Energy	0.3	0.94	1.06	1.04
GWP	0.3	0.86	1.14	1.17
AP	0.1	0.61	1.39	1.67
Water	0.3	1.03	0.97	0.97
Overall eco-efficiency score		**0.91**	**1.09**	**1.12**
Overall rank		**Worst**	**Middle**	**Best**

If the mine were located in central Chile, POX would be the best option, whereas in northern Chile an integrated smelter was the best option. An analysis of the normalised eco-efficiency indicators in Table 4.10 shows that the magnitude of the normalised acidification potential eco-efficiency of the POX option (1.67) gives it an advantage over the other processes, more specifically over custom smelting.

It is also possible that the mine could purchase electricity at a different price if it were located in central Chile. To analyse the effect of such an occurrence on the overall process rankings, electricity prices of ± 50%, relative to the base case price, were applied. However, due to a lack of a detailed break down of mining costs, the mine's operating costs were not a function of the price of electricity, therefore this analysis only affects the operating costs of the concentrator and the downstream processes[79]. A summary of the final rankings is shown in Table 4.11.

---

[79] The concentrator consumes more electricity than the mine, which depends, predominantly, on diesel as an energy source (see Marsden, 2008). Therefore, the effect of this shortcoming is likely to be minimal, furthermore, the mine is common to all the processing options therefore a change in its operating costs affects all the processing options equally.

Table 4.11: Final rankings when the mine is located in central Chile, assuming electricity prices change with location within Chile

	Custom smelter – China	**Integrated smelter – Chile**	**Integrated POX – Chile**
Price of electricity	**Overall eco-efficiency scores**		
0.045 $ / kWh	0.91	1.09	1.14
0.09 $ / kWh **(Base)**	0.91	1.09	1.12
0.135 $ / kWh	0.92	1.08	1.11
Overall rank	**Worst**	**Middle**	**Best**

Integrating vertically with a POX plant remains the best option in central Chile within the analysed electricity price range. However, as the price of electricity increases the margin between the overall scores of the POX option and the other processing options decreases. Since the POX process is predominantly dependent on electricity as an energy source, it is likely that increasing the price of electricity affects it more than a local smelter, which could explain the greater change in the eco-efficiency scores of the POX processing option compared to the smelting options. When the mine sells concentrates to custom smelters in China, only the mine and concentrator sub-systems located in Chile would be affected by the local price of electricity and not the smelter and refinery in China.

4.7.3 Sensitivity analysis: Price of copper

This analysis focuses on the sensitivity of the eco-efficiency indicators to the price of copper and it is applied on the initial base case study, as described in section 4.5. The base case price of copper was $ 7,500 per tonne of copper. Low and high values of -50% and +50%, relative to the base price were applied in this analysis, corresponding to $ 3,750 and $ 11,250 per tonne of copper, respectively.

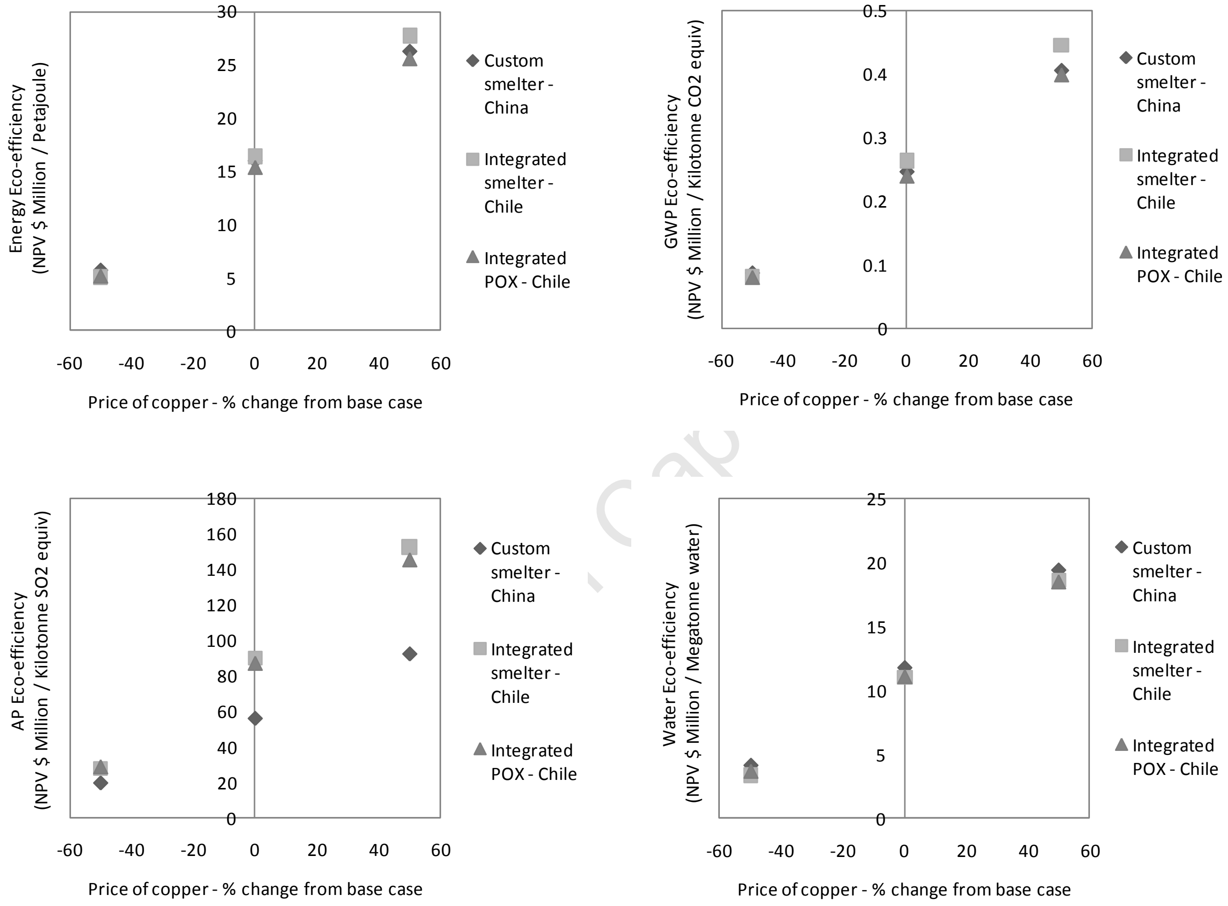


Figure 4.13: Effect of the price of copper on the mine's eco-efficiency

When the market price of copper increases, the mine's revenues increase regardless of the processing option selected, resulting in higher NPVs for all the processing options. However, the environmental performance of the mine is not directly affected by the price of copper, therefore, it remains constant. Effectively, the mine's eco-efficiency improves when the price of copper increases, and declines when the price of copper drops.

Integrating vertically through a smelter remains the best option (in terms of overall eco-efficiency rankings) at copper prices of $ 7,500 and $ 11,250 per tonne of copper, however when the price of copper is $ 3,750 per tonne of copper custom smelting is the best option. Since the environmental performance of the mine remains constant with changing copper prices, the switch in rankings is attributed to techno-economic factors. As the price of copper increases, the differences between the copper recoveries[80] of the various processing options become critical in relation to the relative revenues generated through each processing route. Since the integrated options recover a higher copper percentage than the percentage copper payable to the mine by custom smelters, it follows that as the price of copper increases, the margins between the revenues generated through the integrated options and the custom smelting option increase, in favour of the integrated options, thereby improving their (integrated options) relative economic performances.

4.7.4 Sensitivity analysis: Treatment and refining charges

Figure 4.14 depicts the sensitivity of the mine's eco-efficiency to the treatment and refining charges that are paid to custom smelters by mines. The base case treatment and refining charges were \$ 77 / dry metric tonne of concentrate and \$ 0.17 / kg of payable copper. Low and high values of -50% and +50%, relative to the base charges, were applied in this analysis.

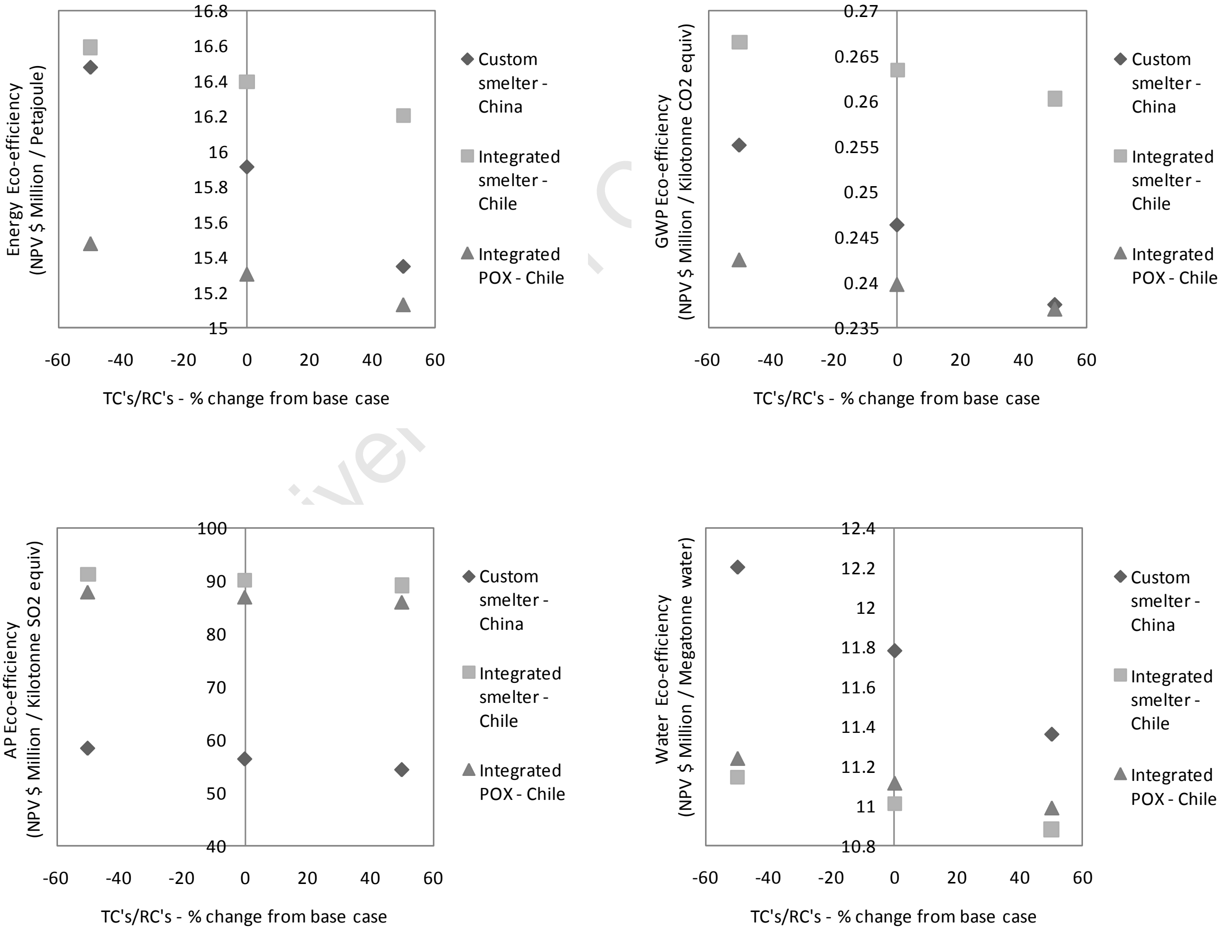


Figure 4.14: Effect of external (custom smelters') treatment and refining charges on the mine's eco-efficiency

---

[80] The copper recoveries of the integrated smelter and POX plants are 96.8% and 98.6% vs. a payable copper amount of 96.65% from the custom smelter (see Table 4.2).

The eco-efficiency of the mine decreases across all the eco-efficiency categories when the treatment and refining charges of custom smelters in China increase. This is because the mine’s NPV decreases since it will cost more to process its concentrates, while the environmental performances remain unaffected. As expected, Figure 4.14 illustrates that the external treatment and refining charges affect the custom smelting option more than the integrated options. However, the link between the external treatment and refining charges and the integrated options in Chile is more obscure and therefore deserves further elucidation. The cashflows of the integrated smelting and POX options would be linked to the external treatment and refining charges only during the period when capital would be invested to construct each of the integrated options (assumed to be 3 years before start up of any of the integrated options). It was assumed that during that time, the mine and concentrator would not shut down, but would continue to produce and sell concentrates to custom smelters while the integrated options are being constructed next to the mine. Therefore, while money is being invested into constructing any of the integrated options, the mine still receives revenue from selling concentrates to custom smelters in China, until the selected integrated option is operating at design capacity.

Integrating vertically through a smelter remained the best option throughout the treatment and refining price-range applied in this sensitivity analysis.

4.7.5 Sensitivity analysis: The price of sulphuric acid

A sensitivity analysis on the price of sulphuric acid is displayed in Figure 4.15. The base case price was $ 100 / tonne of acid, and low and high values of -50% and +50%, relative to the base case acid price, were used in the analysis.

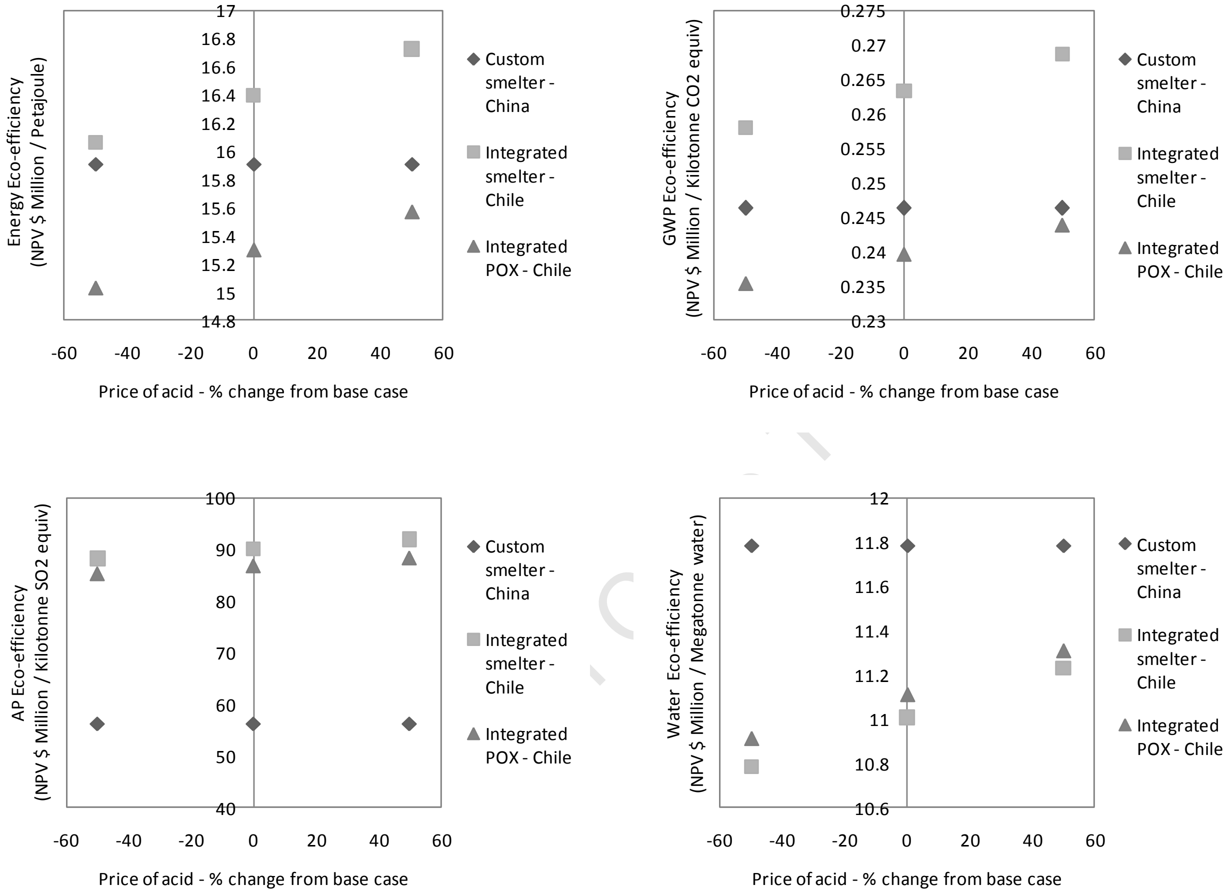


Figure 4.15: The effect of the price of acid on the mine's eco-efficiency

In this study the acid produced by the integrated smelter and POX plants was considered as a saleable by-product. Therefore, when the price of acid increases the mine's eco-efficiency increases because its NPV increases; however, this only occurs when the mine is integrated with a smelter or a POX plant and not when it sells concentrates to custom smelters. When the mine acts as a custom mine, selling concentrates to custom smelters, the acid produced from the concentrates belongs to the custom smelter and is part of the custom smelter's revenue stream (see the by-product contribution to a custom smelter's revenue in Figure 2.16), hence the mine's eco-efficiency (for the concentrate projects analysed in this study) does not change with changes in the acid price when it sells concentrates to custom smelters.

The overall rankings of the processes do not change, relative to the base case rankings, throughout the acid price range applied in this sensitivity analysis, i.e. integrating vertically through a smelter remains the best option.

4.7.6 Sensitivity analysis: Energy costs

A sensitivity analysis on the local cost of energy[81] is displayed in Figure 4.16.

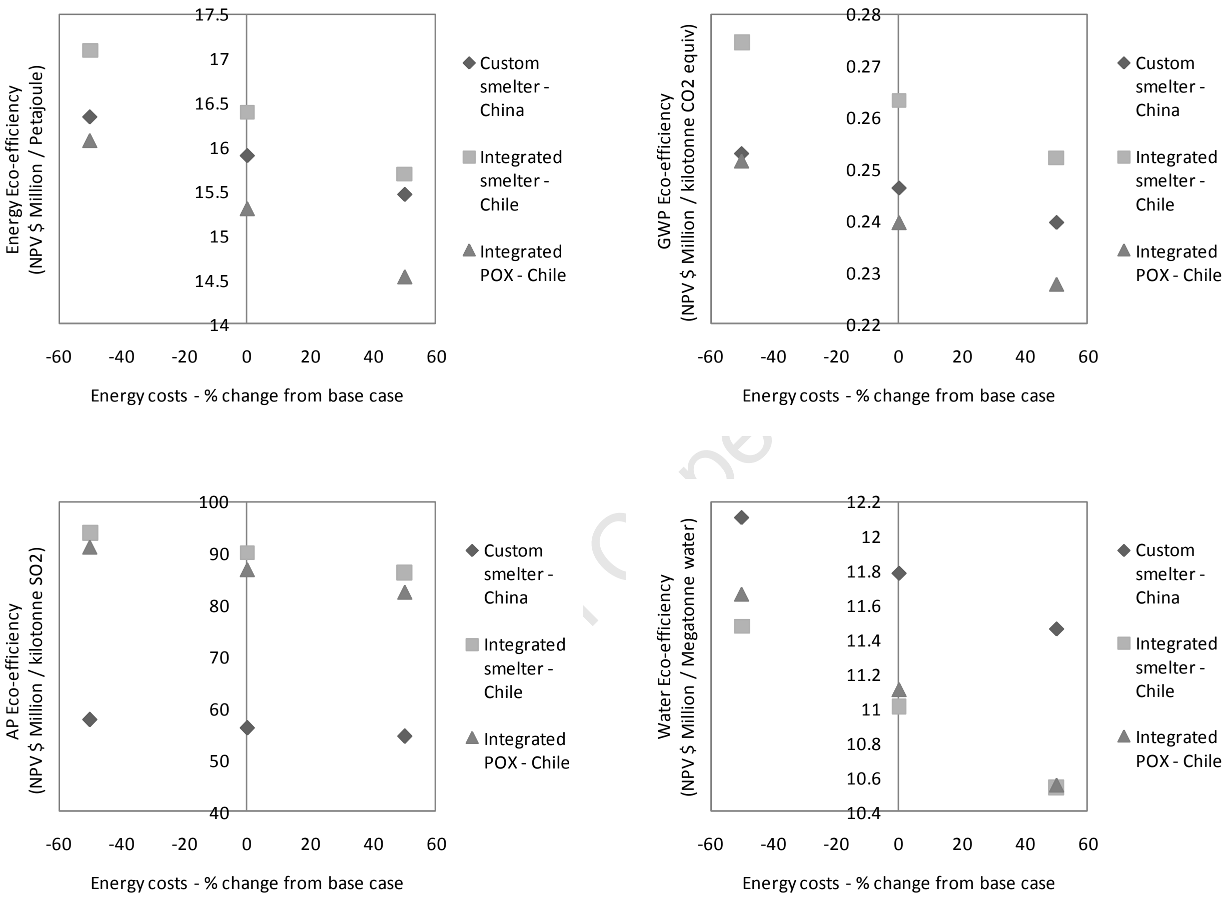


Figure 4.16: The effect of the cost of energy on the mine's eco-efficiency

The mine's eco-efficiency decreases when the cost of energy increases. The decline in the eco-efficiency of the integrated options would be more rapid than the decline in the eco-efficiency of the custom smelting option because all the processing subsystems associated with the integrated options (mine, concentrator, smelter, refinery, POX, SX/EW) would be located in Chile and would be sensitive to local energy prices, whereas with the custom smelting option, the smelter and refinery are located in China and are not sensitive to the local (Chile) energy prices.

---

[81] This refers to all the onsite energy sources, i.e. electricity, natural gas and diesel. Refer to Table 4.2 for the base prices.

Although integrating vertically with a smelter or POX plant would be more sensitive to energy costs, the relative rankings of the processes do not change when energy costs vary between -50% and +50% of the base case energy prices, i.e. integrating vertically through a smelter remains the best option.

5 CONCLUSIONS AND RECOMMENDATIONS

5.1 Summary

A methodology for assessing multiple flowsheet options and a spreadsheet simulation model that takes into account the techno-economic, environmental and logistical concerns of copper production processes were developed in this study, using simplified data and certain assumptions. Key parameters and information of significant relevance to the techno-economic and environmental performance of primary copper production processes were identified and consolidated in the development of the methodology and model. Application of the methodology and model was intended to demonstrate to the mining industry that techno-economic and environmental criteria can be used simultaneously, to inform decision making during the preliminary stages of process design and during the day-to-day management of minerals beneficiation processes.

5.2 Limitations of current study

While this study has demonstrated that early stage decision making scenarios can be influenced by techno-economic and environmental criteria simultaneously, the following limitations in the methodology and model developed are acknowledged:

- Currently the methodology and model developed do not take into account data uncertainty and distinguishability, therefore, there is limited confidence with which final decisions are taken based on the outcomes of the model.
- A detailed review of eco-efficiency weighting was beyond the scope of the current study, therefore, the weighting factors used in this thesis expressed the views of the author on the relative significance of each eco-efficiency category.
- There is a lack of data and information available in the public domain concerning the technical, economical and environmental practises of the mining industry, therefore, the model was developed and applied on the basis of currently available data.

Means of addressing these limitations are proposed in the recommendations section.

5.3 Conclusions

The methodology and model developed in this study were applied to a hypothetical case study, which involved a custom copper mine in Chile that was originally selling copper concentrates to custom smelters in China and was considering integrating vertically through a smelter or a POX plant next to the mine and concentrator in Chile. Net present value (NPV), relative / incremental net present value, internal rate of return (IRR), return on investment (ROI) and payback period (PBP) were the criteria used to differentiate between the economic performances of the three concentrate processing options. Energy consumption, global warming potential, acidification potential and water consumption were used as the environmental performance criteria, and eco-efficiency indicators were used to consolidate the economic and environmental performances of the three options.

Based on the input assumptions to the model and taking into consideration the aforementioned limitations, the following conclusions were drawn from the case study:

- Custom smelting in China had the best economic performance since it had the highest NPV.
- However, integrating vertically through a local smelter and refinery of the magnitude proposed in this study could create approximately 1000 jobs. No data was available for employment generation associated with the POX process.
- The base case assumed that there was a heap leaching operation near the mine that would utilise all the acid produced from the proposed POX plant. In that scenario the relative[82] NPV of the mine, integrated with a POX plant, was - $ 144 million. However, when it was assumed that there was no heap leaching process near the mine, implying that the acid from the POX plant has to be neutralised, the relative NPV of the mine was - $ 605 million, which led to the conclusion that the economic performance of the POX process improves when there is a market for its acidic raffinate.
- While the purely economic analysis showed that custom smelting in China was the best option, the purely environmental analysis demonstrated that integrating vertically through a local smelter was the best option available to the mine.
- Similarly, when eco-efficiency indicators were used, integrated smelting was the most eco-efficient option in all the environmental categories considered in this study, with the exception of water consumption, for which custom smelting was the best option.

---

[82] Relative to a mine that sells concentrates to custom smelters in China

- When weighting factors were applied to the various eco-efficiency categories, and an overall, single score was allocated to each processing route, integrated smelting ranked superior to custom smelting and integrating with a POX plant[83].
- A sensitivity analysis on the geographical location of the mine showed that if the mine were located in central Chile as opposed to northern Chile (base case location), POX would be the most eco-efficient option. This led to the conclusion that the combined techno-economic and environmental performance of copper production processes depends on the location of the operations.
- An increase in the price of copper improved the mine's eco-efficiency regardless of the selected processing route. However, at a low copper price custom smelting was the most eco-efficient option and at a high copper price integrated smelting was the most eco-efficient option.
- For all the processing options, an increase in the treatment and refining charges paid to custom smelters in China decreased the mine's eco-efficiency.
- When the price of acid increased, the mine's eco-efficiency increased, however, only when it was integrated through either a smelter or a POX plant that received credits for the acid, but not when it sold concentrates to custom smelters in China.
- An increase in the local price of energy decreased the mine's eco-efficiency regardless of the selected processing route, however, the integrated options were more sensitive to the cost of energy than the custom smelting option.

---

[83] In terms of eco-efficiency

5.4 Recommendations

Based on the outcomes of this thesis the following short to medium-term, step-wise recommendations are made:

- Further research should be performed to develop a methodology for valuating the various eco-efficiency indicators relative to each other.
- When a methodology for valuating eco-efficiency indicators has been developed, uncertainty and distinguishability analysis must be incorporated into future models and studies emanating from the current study.
- The methodology developed in this thesis can be used as a starting point for integrating environmental concerns into currently available and future process simulation tools that are designed specifically for the minerals industry. It is envisaged that this will add to recent efforts by the minerals industry to operate within the framework of sustainable development.
- Alternatively, the methodology and simulation tool developed in this thesis can be used as a foundation for the development of a commercial simulation tool for the copper industry and thereafter, other mineral industries. However, a multidisciplinary approach would be essential, as well as support from industrial partners who could guide model development through highlighting their specific needs and challenges, while also providing necessary information, data and expertise.

A long term recommendation is that future models should incorporate more environmental categories such as toxicity and acid mine drainage.

6 REFERENCES

Akcil, A. & Kolders, S. 2006. Acid Mine Drainage (AMD): causes, treatment and case studies. *Journal of Cleaner Production*, 14:1139 – 1145.

Alvarado, S., Maldonado, P., Barrios, A. & Jaques, L. 2002. Long term energy – related environmental issues of copper production, *Energy,* 27:183 – 196.

Anglo platinum. 2009. *Environmental management systems*. [Online]. Available: http://www.angloplatinum.com/safety/enviro/management.asp [June 2010].

Aurubis. 2010. *Pirdop plant description*. [Online]. Available: http://www.aurubis.com/en/our-business/products/cathodes/ [March 2011]

Ayres, R.U., Ayres, L.W. & Rade, I. 2002. *The life cycle of copper, its co-products and by-products*. [Online]: Available: http://www.iied.org/pubs/pdfs/G00740.pdf [March 2009]

Azapagic, A. 2005. Appendix:life cycle thinking and life cycle assessment. In Azapagic, A., Perdan, S. & Clift, C. (eds.), *Sustainable Development in Practice: Case Studies for Engineers and Scientists*. England: John Wiley and Sons Ltd, pp. 426 – 437.

Azapagic, A. 2010. *Quality Management and Safety Engineering (BSc) - MST 326: Adisa Azapagic's environmental impact classification factors*. [Online]. Available: http://www.tech.plym.ac.uk/sme/mst324/MST324-05%20Azapagic.htm [January 2010].

Barr, G., Defreyne, J. & Mayhew, K. 2005. *CESL copper process – an economic alternative to smelting.* [Online]. Available: http://www.cesl.com/Generic.aspx?PAGE=CESL+Limited/Publications&portalName=tc [November 2009].

Barratt, B. & Sherman, M. 2002. Factors which influence the selection of comminution circuits. In Mular, L.M., Halbe, D.N. & Barrat, D.J. (eds.), *Mineral processing plant design, practice, and control, Proceedings,* Volume 1, Vancouver, 20 – 24 October 2002, pp. 539 – 565. Colorado: Society for Mining, Metallurgy and Exploration, Inc.

Bartlett, R.W. 1997. Metal extraction from ores by heap leaching. *Metallurgical and Materials Transactions B*, 28b:529 – 545.

BCS Incorporated. 2002*. Energy and Environmental profile of the U.S. mining industry*. *[Online].* Available: http://www1.eere.energy.gov/industry/mining/pdfs/copper.pdf [March 2009].

BHP Billiton. 2004. *Minera Escondida: analysts visit.* [Online]. Available: http://www.bhpbilliton.com/bbContentRepository/Presentations/EscondidaSiteVisit240404.pdf [July 2009].

BHP Billiton. 2009. *Sustainability summary report: resourcing the future.* [Online]. Available: http://www.bhpbilliton.com/bbContentRepository/docs/sustainabilitySummaryReport2009.pdf [June 2010].

Biswas, A.K. & Davenport, W.G. 1994. *Extractive Metallurgy of Copper*. 3rd ed, New York: Elsevier Science Inc.

Bloomberg. 2010. *Sulphuric acid, copper smelting by-product, peaks on demand view, CRU Says*. [Online]. Available: http://www.bloomberg.com/news/2010-06-11/sulfuric-acid-copper-smelting-byproduct-peaks-on-demand-view-cru-says.html [March 2011].

Bodzin, S. 2011. Chile protests turn deadly as Latin America buckles under rising energy prices. *The Christian Science Monitor.* [Online]. Available: http://www.csmonitor.com/World/Americas/2011/0112/Chile-protests-turn-deadly-as-Latin-America-buckles-under-rising-energy-prices [March 2011].

Brantes, R. 2009. Best practices and efficient water use in the Chilean mining industry. In Wiertz, J. & Moran, C. (eds.), *Enviromine 2009 – Proceedings of the first international seminar on environmental Issues in the mining industry*. Santiago Chile, 30 September – 2 October 2009. Santiago: Gecamin.

Brierley, C.L. & Brierley, J.A. 1999. Bioheap processes: operational requirements and techniques. In Jergensen ,V. (eds.), *Copper leaching, solvent extraction and electrowinning technology*. Colorado: Society for Mining Metallurgy & Exploration, Inc. pp. 17 – 27.

Broadhurst, J. 2007. *Predicting the characteristics of solid wastes from the resource – based industries: a focus on primary copper production*. Cape Town: University of Cape Town. (PhD-thesis).

Broadhurst, J.L., Hansen, Y. & Petrie, J.G. 2007. *Waste Characterisation and Water-related Impact Predictions for Solid Mineral Wastes: A New Approach*, Water Research Commission of South Africa, Report No 1550/1/07.

Broadhurst, J. 2009. [Personal notes].

Broadhurst, J. 2010. *Introduction to environmental process engineering: acid rock drainage*. Cape Town: University of Cape Town – [CHE 4024F course notes].

Callow, I.M. & Moon, A.G. 2002. Types and characteristics of grinding equipment and circuit flowsheets. In Mular, L.M., Halbe, D.N. & Barrat, D.J. (eds.), *Mineral processing plant design, practice, and control, Proceedings,* Volume 1, Vancouver, 20 – 24 October 2002, pp. 698 – 709. Colorado: Society for Mining, Metallurgy and Exploration, Inc.

Cano-Ruiz, J.A. & McRae, G.J. Environmentally conscious chemical process design. *Energy Environ*, 23:499 – 536.

Cariola Díez Pérez-Cotapos & Cía Ltda. 2010. *Doing business in Chile*. [Online]. Available: http://www.cer.gov.cl/wp-content/uploads/downloads/2010/03/Doing-business-Cariola-firm.pdf [August 2010].

Chambers, B., Howard, P., Pottie, J., Murray, L. & Burgess, A. 2003. *Water recovery from a mine in the Atacama desert*. [Online]. Available: http://www.infomine.com/publications/docs/Chambers2003.pdf [February 2010].

Chariot Resources. 2010. *Feasibility study overview: developing Peru's next copper mine.* [Online]. Available: http://www.chariotresources.com/docs/CHDFSMay2009%20UR.pdf [April 2010].

Chevron. 2011. *Our fuel experts answer: what is S15 (ULSD)?*. [Online]. Available: http://www.chevron.com/products/tips/fuel-school/answers.aspx?aID=68&cID=16 [March 2011].

COCHILCO. 2008. *Best practices and efficient water use in the Chilean mining industry*. [Online]. Available: http://www.cochilco.cl/english/productos/doc/best_practices_and_the_efficient_use_of_water.pdf [December 2010].

CODELCO. 2009. *Chilean copper smelting and refining overview.* [Online]. Available: http://downloads.gecamin.cl/cierre_eventos/molten2009/prsntcns/00181_00808_pr.pdf [July 2009].

Commonwealth of Australia. 2008. *Handbooks in the Leading Practice Sustainable Development Program for the Mining Industry Series – Water Management*. [Online]. Available: www.ret.gov.au/resources/documents/lpsdp/lpsdp-waterhandbook.pdf [October 2010].

Copper Development Association Africa (CDAA). 2009. *Sources of copper: copper ore bodies and extraction.* [Online]. Available: http://www.copper.co.za/copper_ore.php# [December 2009].

Coulas, M. 2005. *Copper - Canadian Minerals Yearbook*. [Online]. Available: http://www.nrcan.gc.ca/smm-mms/busi-indu/cmy-amc/content/2005/25.pdf [April 2009].

Couper, J.R. 2003. *Process Engineering Economics*. New York: Marcel Decker Inc.

Crosby-Forbes, P.B. 1999. *The use of life cycle assessment in the evaluation of environmental performance in the base metal refining industry*. Cape Town: University of Cape Town. (MSc-thesis).

Crowson, P. 1998. *Inside mining: the economics of the supply and demand of minerals and metals*. London: Mining Journal Books.

Crundwell, F.K. 2003. How do bacteria interact with minerals? *Hydrometallurgy,* 71:75 – 81.

Cruz-Coke, G.L. & Aravena, L.M. 2004. *"Pay your taxes" faulty calculations or ideological prejudice? Pontificia Universidad Catolica de Chile (Catholic University of Chile) report*. [Online]. Available: http://cep.cl/UNRISD/UNRISD_CSR/References/Dimes_Diretes/PUC.doc [September 2009].

Curry hydrocarbons Inc. 2010. *Chemical Engineering Plant Cost Index*. [Online]. Available: http://www.curryhydrocarbons.ca/downloads.htm [March 2011].

Damodaran, A. 2011. *Cost of capital by sector*. [Online]. Available: http://pages.stern.nyu.edu/~adamodar/New_Home_Page/datafile/wacc.htm [March 2011].

Davenport, W.G., King, M., Schlesinger, M. & Biswas A.K. 2002*. Extractive metallurgy of copper*. 4th ed. New York: Elsevier Science Press.

de Souza, A. 2010. *The structure of the custom concentrate market*. [Online]. Available: http://www.newcrest.com.au/upload/642_3x26x201025042PM.pdf [April 2010].

Douglas, J.M.1988. *Conceptual Design of Chemical Processes.* New York: McGraw-Hill.

Dreisinger, D. 2005. Copper leaching from primary sulfides: options for biological and chemical extraction of copper. In Harrison, S.T.L. Rawlings, D.E. & Petersen, J. (eds.), *Proceedings of the 16th International Biohydrometallurgy Symposium*, Cape Town, 25 – 29 September 2005, pp. xli –li. South Africa: The 16th International Biohydrometallurgy Symposium.

Dreisinger, D. 2006 a. *Autoclave Processing of Ores and Concentrates*. Cape Town: University of Cape Town. [Short Course].

Dreisinger, D. 2006 b*. New developments in Cu and Ni hydrometallurgy: presentation to JOGMEG (Japan).* [Online]. Available: http://www.jogmec.go.jp/mric_web/koenkai/060206/060206_Dr_David_Dreisinger.pdf [May 2009].

Dresher, W.H. 2001. *Copper applications in mining and metallurgy: how hydrometallurgy and the SX/EW process made copper the "green" metal*. [Online]. Available: http://www.copper.org/publications/newsletters/innovations/2001/08/hydrometallurgy.html [March 2009].

Dunn, G.M. 2009*. Increasing the capacity of existing and new exothermic autoclave circuits.* [Online]. Available: http://www.saimm.co.za/Conferences/Hydro2009/043-056_Dunn.pdf [August 2009].

Eggers, M. 2009. *Renewable Energy Chile: sector report - UK Trade and Investment, British Embassy Santiago*. [Online]. Available: http://www.globaltrade.net/f/market-research/pdf/Chile/Energy-Coke-Oil-Gas-Electricity-Electricity-Generation-and-Distribution-Renewable-Energy-in-Chile.html [March 2011].

Elert, G. 2010. Energy density of coal. [Online]. Available: http://hypertextbook.com/facts/2003/JuliyaFisher.shtml [January 2010].

Engels, J. & Dixon-Hardy, D. 2009. *Water management considerations for conventional storage*. [Online]. Available: http://www.tailings.info/index.htm [September 2009].

Erepan, P. 2004. *Investigation of the reasons for copper and gold loss in the cleaner tail, at Ok Tedi Papua New Guinea*. Kalgoorlie: Curtin University of Technology. (MSc-thesis). [Digital].

European Sulphuric Acid Association (ESA) and European Fertilizer Manufacturers' Association (EFMA). 2000. *Best available techniques for pollution prevention and control in the European sulphuric acid and fertilizer industries: production of sulphuric acid*. [Online]. Available: http://www.efma.org/documents/file/bat/BAT%20Production%20of%20Sulphuric%20Acid.pdf [February 2011]

Farooq, S. 2010. *Ore deposits formed by oxidation and supergene enrichment.* [Online]. Available: http://www.cps-amu.org/sf/notes/b3-3-7.htm [September 2010].

Farnel Capital. 2010. *Maritime references*. [Online]. Available: http://www.searates.com/ [February 2010].

First Quantum Limited. 2010. *Kansanshi*. [Online]. Available: http://www.first-quantum.com/i/pdf/Kansanshi_Fact.pdf [August 2010].

Freeport-McMoRan Copper & Gold Inc. 2010 a. *Bagdad mine and processing facilities*. [Online]. Available: http://www.fcx.com/operations/USA_Arizona_Bagdad.htm [August 2010].

Freeport-McMoRan Copper & Gold Inc. 2010 b. *Working Toward Sustainable Development Report*. [Online]. Available: http://www.fcx.com/envir/wtsd/pdf-wtsd/2009/WTSD_Bk_2009.pdf [August 2010].

Freeport-McMoRan Copper & Gold Inc. 2010 c. *History*. [Online]. Available: http://www.fcx.com/company/history.htm [January 2011].

Ford, K.L. 2003. *Passive treatment systems for acid mine drainage*. [Online]. Available: http://www.blm.gov/nstc/library/pdf/TN409.PDF [October 2010].

Gaete, P. 2009. *COCHILCO report raises doubts about water availability for copper mining – Chile*. [Online]. Available: http://www.bnamericas.com/news/mining/Cochilco_report_raises_doubts_about_water_availability_for_copper_mining [March 2011].

Gaines, L. 1980. *Energy and material flows in the copper industry*. United States of America: US Department of Energy.

Gericke, M., Neale, J.W. & van Staden, P.J. 2009. A Mintek perspective of the past 25 years in minerals bioleaching, *The Journal of The Southern African Institute of Mining and Metallurgy*, 109:567 - 585

Giurco, D., Stewart, M., & Petrie, J. 2001. *Decision making to support sustainability in the copper industry*. [Online]. Available:

http://www.isf.uts.edu.au/publications/Giurcoetal2006sustainablecopperindustry.pdf [February 2009].

Goonan, T.G. 2004. *Flows of selected materials associated with world copper smelting - United States Geological Survey report*. [Online]. Available: http://pubs.usgs.gov/of/2004/1395/ [April 2009].

Guma, M. 2010. *Developing minerals beneficiation flowsheets for eco-efficiency: a systems approach*. Cape Town: University of Cape Town. (MSc-thesis).

Habashi, F. 2007. Abandoned but not forgotten: the recent history of copper hydrometallurgy. In Riveros, P.A., Dixon, D.G., Dreisinger, D.B. & Collins, M.J. (eds.), *Proceedings of the sixth international Copper – Cobre conference: the John E. Dutrizac symposium on copper hydrometallurgy,* Volume 4*, Book 1*, Toronto, 25 – 30 August, pp. 3 – 19. Canada: Canadian Institute of Mining, Metallurgy and Petroleum.

Hansen, Y. 2004. Impact assessment of solid wastes from primary Industries - an integrated approach. Sydney: University of Sydney. (PhD-thesis).

Harrison, S. 2008. *Sustainability in Chemical Engineering*, University of Cape Town – [CHE 5064Z course notes].

Hartman, H.L. & Mutmansky, J.M. 2002. *Introductory mining engineering*. New Jersey: John Wiley and Sons, Inc.

Hartman, H.L. 1992. *SME mining engineering handbook*. Colorado: Society for Mining, Metallurgy and Exploration, Inc.

Henriques. A. 2004. CSR, Sustainability and the triple bottom line. In Henriques, A. and Richardson, J. (eds.), *The triple bottom line: does it all add up?*. UK: Earthscan. pp. 26 – 33.

Herbst, J.A., Lo, Y.C. & Flintoff, B. 2003. Size reduction and Liberation. In Fuerstenau, M.C. and Han, N.K. (eds.), *The principles of mineral processing.* Colorado: Society for Mining, Metallurgy and Exploration, Inc. pp. 61 – 118.

Hesketh, A. 2010*. An integrated approach to AMD mitigation through sulfide removal from tailings.* Cape Town: University of Cape Town. (MSc-thesis).

Hinde, A. 2009. *Overview of rock type and mineral behaviour in comminution circuits*. Mintek Presentation.

Hrebar, M.J. & Gentry, D. W. 2003. Economics of the minerals industry. In Fuerstenau, M.C. and Han. N.K. (eds.), *The principles of mineral processing.* Colorado: Society for Mining, Metallurgy and Exploration, Inc. pp. 517 – 560.

Huo, H., Zhang, Q., Wang, M.Q., Streets, D.G. & He, K. 2010. Environmental implication of electric vehicles in China, *Environmental Science and Technology,* 44:4856-4861.

Iasillo, E. and Schlitt, W.J. 1999. Practical aspects associated with evaluation of a copper heap leach project. In Jergensen ,V. (eds.), *Copper leaching, solvent extraction and electrowinning technology*. Colorado: Society for Mining Metallurgy & Exploration, Inc. pp. 123 – 138.

Infomine Inc. 2010. *Dynamic charting tool*. [Online]. Available: http://www.infomine.com/investment/charts.aspx?mv=1&f=f&r=15y&c=ccopper.xusd.umt&w=wcopper_lme_warehouse#chart [August 2010].

International Copper Study Group (ICSG). 2009. *The world copper fact book*. [Online]. Available: http://www.icsg.org/index.php?option=com_content&task=view&id=81&Itemid=64 [May 2010].

International energy agency (IEA*). 2008. Energy efficiency indicators for public electricity production from fossil fuels.* [Online]. Available: http://www.iea.org/papers/2008/En_Efficiency_Indicators.pdf [January 2011].

International Institute for Environment and Development (IIED). 2002. *Mining, Minerals and Sustainable Development Project*. UK: Earth Scan Publications.

Intergovernmental Panel on Climate Change (IPCC). 1996. *Revised 1996 IPCC guidelines for national greenhouse gas inventories: reference manual,* Volume 3. [Online]. Available: http://www.ipcc-nggip.iges.or.jp/public/gl/invs6.html [January 2010].

Intergovernmental Panel on Climate Change (IPCC). 2006. Energy. In National greenhouse gas inventories programme, Eggleston H.S., Buendia L., Miwa K., Ngara T., & Tanabe K. (eds.), *2006 IPCC guidelines for national greenhouse gas inventories,* Volume 2. Japan: IGES.

Jansen, M. & Taylor, A. 2003. *Overview of gangue mineralogy issues in oxide copper heap leaching*. [Online] Available: http://www.altamet.com.au/Technical%20Papers%20and%20Articles/ALTA%20Copper/Overview%20of%20Gangue%20Mineralogy.pdf [August 2009].

Kamarainen, J. 2010. *Impact of new regulations in fuel pricing: TraFi.* [Online]: Available: http://www.bpoports.com/assets/files/ppk/BPO%20Seminar%202010%20-%20Jorma%20Kamarainen.pdf [March 2011].

King, M.J. 2007. The evolution of technology for extractive metallurgy over the last 50 years – Is the best yet to come?, *JOM*, 59:21-27.

Kojo, I.V. & Storch, H. 2006. *Copper production with Outokumpu Flash Smelting: an update*. [Online]. Available: http://www.outotec.com/38210.epibrw [December 2009].

Kuckshinrichs, W., Zapp, P. & Witold-Roger, P. 2007. CO_2 emissions of global metal-industries: the case of copper, *Applied Energy,* 84:842–852.

Lakernik, M.M. & Sevryukov, N.N. 1960. *Metallurgy of Copper – USSR*. [Online]. Available: http://www.dtic.mil/cgi-bin/GetTRDoc?AD=ADA357110&Location=U2&doc=GetTRDoc.pdf [November 2009].

Lange, E.K. 2010. Get the salt out, *National Geographic,* 217:32 - 35.

Lewandowski, K.A. & Kawatra, S.K. 2007. Agglomeration for copper heap leaching. In Riveros, P.A., Dixon, D.G., Dreisinger, D.B. & Collins, M.J. (eds.), *Proceedings of the sixth international Copper – Cobre conference found in The John E. Dutrizac symposium on copper hydrometallurgy*, Volume 1, Book 1, Toronto, 25 – 30 August 2007, pp. 439 – 451. Montreal: Canadian Institute of Mining, Metallurgy and Petroleum.

Lewis, P.J. & Streets, C.G. 1979. An analysis of base – metal smelter terms. *Proceedings of the eleventh Commonwealth mining and metallurgical congress,* 1978, Hong Kong, pp. 753 – 767. London: The Institute of Mining and Metalllurgy.

Long, H. & Dixon, D. 2006. Pressure oxidation kinetics of orpiment (As_2S_3) in sulfuric acid, *Hydrometallurgy,* 85:95 – 102.

Lu, J., Dreisinger, D. & West-Sells, P. 2007. Acid curing and agglomeration. In Riveros, P.A., Dixon, D.G., Dreisinger, D.B. & Collins, M.J. (eds.), *Proceedings of the sixth international Copper – Cobre conference: the John E. Dutrizac symposium on copper hydrometallurgy*. Volume 4, Book 1, Toronto, 25 – 30 August 2007, pp. 453 – 464. Montreal: Canadian Institute of Mining, Metallurgy and Petroleum.

Lynch, A.J., Johnson, N.W., Manlapig, E.V. & Thorne, C.G. 1981. *Mineral and coal flotation circuits: their simulation and control*. New York: Elsevier Scientific Publishing Company.

Major, K. 2002. Types and characteristics of crushing equipment and circuit flowsheets. In Mular, L.M., Halbe, D.N. & Barrat, D.J. (eds.), *Mineral processing plant design, practice, and control, Proceedings,* Volume 1, Vancouver, 20 – 24 October 2002, pp. 566 – 583. Colorado: Society for Mining, Metallurgy and Exploration, Inc.

Maluleke, W.J. 2006. An experimental investigation of leachate generation predictions of waste from copper sulphide ore processing. Cape Town: University of Cape Town. (PhD-thesis).

Mallory, K. 2008. *OCRI Global Marketing: doing Business in Chile.* [Online]. Available: http://www.ottawaregion.com/media_lib/Doing_Business_Archive_Presentations/EDC__Presentation.pdf [March 2011].

Marsden, J.O. 2004. *Pressure leaching of copper concentrates*. [Online]. Available: http://www.metsoc.org/conferences/hydro2004/plenary3bw.pdf [April 2009].

Marsden, J.O. & Wilmot J.C. 2007. Sulphate – based process flow sheet options for hydrometallurgical treatment of copper sulfide concentrates, In Riveros, P.A., Dixon, D.G., Dreisinger, D.B. & Collins, M.J. (eds.), *Proceedings of the sixth international Copper – Cobre conference: the John E. Dutrizac symposium on copper hydrometallurgy*. Toronto, 25 – 30 August 2007, pp. 77 – 100. Montreal: Canadian Institute of Mining, Metallurgy and Petroleum.

Marsden, J.O. 2008. Energy efficiency and copper hydrometallurgy. In Young, C.A., Taylor, P.R., Anderson, C.G. & Choi, Y (eds.) *Hydrometallurgy 2008: Proceedings of the sixth international symposium*, Phoenix, Arizona, 17 – 20 August 2008, pp 29 – 42. Colorado: Society for Mining, Metallurgy and Exploration, Inc.

Mbendi. 2009. *Phelps Dodge Mining Company - Phoenix, Maricopa County, Arizona, United States*. [Online]. Available: http://www.mbendi.com/orgs/cbik.htm [May 2010].

McElroy, R. & Young, W. 1999. Pressure oxidation of complex copper ores and concentrates. In Jergensen ,V. (eds.), *Copper leaching, solvent extraction and electrowinning technology*. Colorado: Society for Mining Metallurgy & Exploration. pp. 29 – 40.

Metalprices. 2010. *Nickel Prices and News*. [Online]. Available: http://www.metalprices.com/FreeSite/metals/nickelalloy/nickelalloy.asp [January 2010].

MIM Holdings. 2001. *2001 Environmental report*. [Online]. Available: http://archive.xstrata.com/mim/www.mim.com.au/downloads/01_ER.pdf [December 2010].

Mining Basics. 2010 a. *Skarns*. [Online]. Available: http://www.miningbasics.com/skarn-deposits [September 2010].

Mining Basics. 2010 b. *Primary and Secondary Vein Minerals - Secondary Enrichment*. [Online]. Available : http://www.miningbasics.com/primary-and-secondary-vein-minerals-secondary-enrichment [September 2010].

Mining Media International. (MMI) 2010. *Finding energy-effective solutions for mine pumping systems*. [Online]. Available: http://www.e-mj.com/index.php/features/580-finding-energy-effective-solutions-for-mine-pumping-systems.html [March 2010].

Moskalyk, R.R. & Alfantazi, A.M. 2003. Review of copper pyrometallurgical practice: today and tomorrow, *Minerals Engineering*, 16:893–919.

Mwale, A.H., Musonge, P. & Fraser, D.M. 2005. The influence of particle size on energy consumption and water recovery in comminution and dewatering systems, *Minerals Engineering*. 18:915 – 926.

Napier-Munn, T.J., Morrell, S., Morrison, R.D. & Kojovic, T. 1999. *Mineral comminution circuits, their operation and optimisation*. Queensland: Julius Kruttshnitt Mineral Research Centre.

Natural Resources Canada (NRC). 2009. *Media release*. [Online]. Available: http://www.nrcan-rncan.gc.ca/com/index-eng.php [April 2009].

Net Industries. 2010. Coal: composition of coal. [Online]. Available: http://science.jrank.org/pages/1533/Coal-Composition-coal.html [January 2010].

Norgate, T. E. 2001. *A comparative life cycle assessment of copper production processes - CSIRO report*. [Online]. Available: http://www.intec.com.au/docs/media/2001%20Oct-CSIRO%20Life%20Cycle%20Analysis.pdf [April 2009].

Norgate, T.E. and Lovel. R.R. 2004. *Water use in metal production: a life cycle perspective*. [Online]. Available: http://www.minerals.csiro.au/sd/pubs/waterrev1.pdf [October 2010].

Nyrstar. 2009. *Introduction to zinc and lead smelting business*. [Online]. Available: http://www.nyrstar.com/nyrstar/en/investors/investor-materials/introduction.pdf [January 2010].

O'Bryan, K. 2002. Selection, sizing and special considerations for pebble crushers. In Mular, L.M., Halbe, D.N. & Barrat, D.J. (eds.), *Mineral processing plant design, practice, and control, Proceedings,* Volume 1, Vancouver, 20 – 24 October 2002*,* pp. 628 – 635. Colorado: Society for Mining, Metallurgy and Exploration, Inc.

Omalu, K. & Waelde, T.W. 2000. *Key issues of mining law: a brief comparative survey as a background study for the reform of mining law*. [Online]. Available: http://www.dundee.ac.uk/cepmlp/journal/html/vol3/article3-13.html [August 2010].

Outotec. 2007. *HydroCopper*. [Online]. Available: http://www.outotec.com/36286.epibrw [September 2010]

Peacey, J., Guo, J.X. & Robles, E. 2003. *Copper hydrometallurgy – current status, preliminary economics, future direction and positioning versus smelting*. [Online]. Available: http://www.hatch.ca/Non_Ferrous/articles/copper_hydrometallurgy.pdfHatch [March 2009].

Peters, M.S. & Timmerhaus. K.D. 1991. *Plant design and economics for engineers. Singapore: McGraw Hill.*

Petersen, J. & Dixon, D.G. 2007. Principles, mechanisms and dynamics of chalcocite heap bioleaching, In.Donati, E. & Sand, W. (eds.), *Microbial processing of metal sulfides*. Berlin: Springer Verlag. pp. 193-218.

Pimentel, S. 2009. The Chilean copper mining sector: energy consumption and greenhouse gas emission profile (2001 – 2007). In Wiertz, J. & Moran, C. (eds.), *Enviromine 2009 – Proceedings of the first international seminar on environmental Issues in the mining industry*. Santiago Chile, 30 September – 2 October 2009. Santiago: Gecamin.

PKF. 2010. *Chile tax guide 2010*. [Online]. Available: http://www.pkf.com/media/135158/chile%20pkf%20tax%20guide%202010.pdf [March 2010].

Press, F., Siever, R., Grotzinger, J. & Jordan, T.H. 2004. *Understanding earth*. 4th ed. New York: W.H. Freeman and Company.

Ramachandran, V., Diaz, C., Eltringham, T., Jiang, C.Y., Lehner, T., McKenzie, P. J., Newman, C.J. & Tarasov, A. 2003. Primary copper production – a survey of operating world copper smelters. In Diaz, C., Kapusta, J. & Newman, C.J. (eds.), *Proceedings of the fifth international Copper – Cobre conference: pyrometallurgy of copper - Hermann Schwarze symposium*, Volume 4, Book 1, Santiago, November 2003, pp. 3 – 106. Montreal: Canadian Institute of Mining, Metallurgy and Petroleum.

Reuters. 2011 a. *Update 2 – CODELCO ups Japan, Korea, copper premiums by 31%.* [Online]. Available: http://www.reuters.com/article/2010/11/16/copper-premiums-codelco-idUSSGE6AF02Y20101116 [January 2011].

Reuters. 2011 b. *Analysis: Costly Chile power may jolt renewable energy* [Online]. Available: http://www.reuters.com/article/2011/02/10/us-chile-energy-wind-idUSTRE7193DH20110210 [March 2011].

Rico, M., Benito, G., Salgueiro, A.R., Diez-Herrero, A. & Pereira, H.G. 2007. Reported tailings dam failures: a review of the European incidents in the worldwide context. *Journal of Hazardous Materials*, 152:846–852.

Robertson, A.W., Vercuil, A., Van Staden, P.J. & Craven, P. 2005. *A bacterial heap leaching approach for the treatment of low grade primary copper sulphide ore*. [Online]. Available: http://www.basemetals.org/Kitwe2005/471-484_Robertson.pdf [December 2009].

Robles, E. 2009. *Economical considerations in the analysis of alternative processes for the treatment of copper sulphides*. [Online]. Available: http://downloads.gecamin.cl/cierre_eventos/hydrocopper2009/prsntcns/00215_00947_pr.pdf [January, 2010].

Sanchez-Corrales, V.M., Valera-Gonzalez, J.A., Flores-Perez, P. & Perez-Tello, M. 2004. Mass balance calculations in copper flash smelting by means of genetic algorithms, *JOM*, 56:29 – 33.

Savitz, A.W. & Weber, K. 2006. *How today's best-run companies are achieving economic, social, and environmental success – and how you can too*. San Francisco: John Wiley and Sons, Inc.

Seitz, J. 2011 a. Rising energy costs force Chilean institutions, companies to run more efficiently. *The Santiago Times*. [Online]. Available: http://www.santiagotimes.cl/news/environmental/20806-rising-energy-costs-force-chilean-institutions-companies-to-run-more-efficiently [March, 2011].

Seitz, J. 2011 b. Energy prices for Chilean businesses highest in three years. *The Santiago Times*. [Online]. Available: http://www.santiagotimes.cl/business/other/20882-energy-prices-for-chilean-businesses-highest-in-three-years [March, 2011].

Seyedbagheri, A., Van Staden, P. & Mclaren,C. 2009. *A study of the acid – gangue reactions in heap leach operations*. [Online]. Available: http://downloads.gecamin.cl/cierre_eventos/hydrocopper2009/prsntcns/00213_00937_pr.pdf [February 2010].

Singer, D.A., Berger, V.I. & Moring, B.C. 2002. *Porphyry copper deposits of the world: database, map, and grade and tonnage models*. [Online]. Available: http://pubs.usgs.gov/of/2005/1060/ [July 2009].

Smith, L. D. 2002. Financial analysis and economic optimisation. In Mular, L.M., Halbe, D.N. & Barrat, D.J. (eds.), *Mineral processing plant design, practice, and control, Proceedings,* Volume 1, Vancouver, 20 – 24 October 2002, pp. 347 – 369. Colorado: Society for Mining, Metallurgy and Exploration, Inc.

Soderstrom, U. 2008. *Boliden Capital Markets Day presentation: copper smelter revenue stream*. [Online]. Available: http://www.boliden.se/ [November 2009].

Sterlite Industries India Limited. 2008. *National award for excellence in water management*. [Online]. Available: http://www.slideshare.net/indiawaterportal/sterlite-industries-india-ltd-tuticorin [March 2010]

Stewart, M. & Petrie, J. 2004. A process systems approach to life cycle inventories for minerals: South African and Australian case studies. *Journal of cleaner production*, 14:1042-1056.

Sylwestrzak, L., Baxter, K., Turney, J. & Dixon, D. 2009. Galvanox™ - An opportunity for existing copper flotation concentrators to improve overall project economics. In Domic, E. & Casa, J. (eds.), *Proceedings of the V international copper hydrometallurgy workshop*, Santiago, 13 – 15 May 2009, pp. 17 – 25. Santiago: Gecamin.

Takeuchi, T. & Nambu, M. 1952. Secondary enrichment and microscopic intergrowth of copper ores from the Akayama mine, Yamagata prefecture, Japan. In The Publication Committee (eds.), *Science reports of the Research Institutes, Tohoku University. Series. A. Physics, chemistry and metallurgy*. Japan: Tohoku University. pp. 63 – 70.

The Engineering Tool Box. 2009. Fuels combustion: nitrogen (NOx) emission. [Online]. Available: http://www.engineeringtoolbox.com/nox-emission-combustion-fuels-d_1086.html [November 2009].

Todd, J. & Reddick, K. 1997. *Acid mine drainage* [Online]. Available: http://www.cee.vt.edu/ewr/environmental/teach/gwprimer/acidmine/acidmine.html [October 2010].

Trading Markets. 2011. *Chinese copper smelters settle higher 2011 copper charges with BHP, report*. [Online]. Available: http://www.tradingmarkets.com/news/stock-alert/bhp_chinese-copper-smelters-settle-higher-2011-copper-charges-with-bhp-report-1485451.html [March 2011].

United Nations. 2009. *Division for sustainable development*. [Online]. Available: http://www.un.org/esa/dsd/index.shtml?utm_source=OldRedirect&utm_medium=redirect&utm_content=dsd&utm_campaign=OldRedirect [January 2011].

United Nations. 2010. *Greater efforts needed to curb global warming - UN report*. [Online]. Available: http://www.un.org/apps/news/story.asp?NewsID=33859&Cr=unep&Cr1= [March 2010].

United Nations Conference on Trade and Development (UNCTAD). 2004. *Manual for the preparers and users of Eco-Efficiency Indicators*. [Online]. Available: http://www.unctad.org/en/docs/iteipc20037_en.pdf [June 2010].

United Nations Economic and Social Commission for Asia and the Pacific (UNESCAP). 2009. *Eco-efficiency Indicators: measuring resource - use efficiency and the impact of economic activities on the environment.* [Online]. Available: http://www.unescap.org/esd/environment/publications/EEI/ESCAP_EEI%20Publication%202561.pdf [October 2010].

University of Pretoria. 2010. *Surface mining methods*. [Online]: Available: http://blogs.up.ac.za/weblog/media/file/estelle/adobe%20files/ch2.pdf [October 2010].

U.S. Congress, Office of Technology Assessment (USCOTA). 1988. *Copper: technology and competitiveness.* [Online]. Available: http://govinfo.library.unt.edu/ota/Ota_3/DATA/1988/8808.PDF [August 2009].

U.S. Department of State (USDS). 2010. *Background note: Chile*. [Online] Available: http://www.state.gov/r/pa/ei/bgn/1981.htm [January 2011].

U.S. Energy Information Administration (USEIA). 2010. *Natural gas weekly update*. [Online]: Available: http://www.eia.doe.gov/oog/info/ngw/ngupdate.asp [March 2010].

U.S. Gold Coporation. 2007. *Custom smelter*. [Online]. Available: http://www.usgold.com/glossary-term/153/ [August 2010].

Utley, W. 2002. Selection and sizing of primary crushers. In Mular, L.M., Halbe, D.N. & Barrat, D.J. (eds.), *Mineral processing plant design, practice, and control, Proceedings,* Volume 1, Vancouver, 20 – 24 October 2002, pp. 584 – 604. Colorado: Society for Mining, Metallurgy and Exploration, Inc.

Watling, H.R. 2006. The bioleaching of sulphide minerals with emphasis on copper sulphides: a review. *Hydrometallurgy*. 84: 81 – 108.

Wenzl, C., Filzweiser, A. & Antrekowitsch, H., 2007, Review of anode casting: part 1, *World Of Metallurgy - Erzmetall,* 60:77 – 82.

Wels, C. & Robertson, A.M. 2003. *Conceptual model for estimating water recovery in tailings impoundments.* [Online]. Available: http://www.rgc.ca/publications/welcmf.pdf [August 2010].

Wills, B. A. & Napier-Munn, T.J. 2006. *Wills' mineral processing technology: an introduction to the practical aspects of ore treatment and mineral recovery.* 7th ed. Oxford: Butterworth-Heinemann.

Woodbank Communication Ltd. 2005. *Energy efficiency*. [Online]. Available. http://www.mpoweruk.com/energy_efficiency.htm [January 2011].

World Business Council for Sustainable Development (WBCSD). 2000. *Creating more value with less impact*. [Online]. Available: http://www.wbcsd.org/web/publications/eco_efficiency_creating_more_value.pdf

World Commission on Environment and Development (WCED). 1987. *Our common future.* [Online]. Available: http://www.un.org/documents/ga/res/42/ares42-187.htm [June 2010].

Xstrata. 2006. *Isasmelt.* [Online]. Available: http://www.isasmelt.com/index.cfm?action=dsp_content&contentID=35 [September 2010].

Zipper, C., Skousen, J. & Jage, C. 2009. *Passive treatment of acid mine drainage.* [Online]. Available: http://www.cses.vt.edu/PRP/Update-PrePubs/460-133-update-prepub.pdf [March 2011]

7 APPENDICES

7.1 Appendix A: Mineralogy considered

Table 7.1: Minerals considered in the mass balances

Substance	***M***[84] **(g/mol)**	***M*** **(tonnes/mol)**	**Name**
Cu	63.546	0.000063546	(Cu Metal)
Sulphides			
Cu_5FeS_4	501.833	0.000501833	Bornite
Cu_2S	159.156	0.000159156	Chalcocite
$CuFeS_2$	183.521	0.000183521	Chalcopyrite
CuS	95.610	0.000095610	Covelite
Cu_3AsS_4	393.814	0.000393814	Enargite
Oxides			
$CuSO_4.2Cu(OH)_2$	354.742	0.000354742	Antlerite
$Cu_3(OH)_2(CO3)_2$	344.678	0.000344678	Azurite
$Cu_4(OH)_6SO_4$	452.308	0.000452308	Bronchanite
$CuSiO_3.2H_2O$	175.676	0.000175676	Chrysocolla
$CuSO_4$	159.610	0.000159610	Copper sulphate
Cu_2O	143.092	0.000143092	Cuprite
$CU_2(OH)_2(CO_3)$	221.122	0.000221122	Malachite
CuO	79.546	0.000079546	Tenorite
Common gangue Minerals			
$CaCO_3$	100.090	0.000100090	Calcite
$CaMg(CO_3)_2$	184.410	0.000184410	Dolomite
$KAlSiO_3O_8$	278.350	0.000278350	Feldspar
FeO(OH)	88.857	0.000088857	Goethite
Fe_2O_3	159.694	0.000159694	Hematite
$Fe(OH)_3$	106.877	0.000106877	iron(III) Hydroxide or Ferric Hydroxide
$MgCO_3$	84.320	0.000084320	Magnesite
FeS_2	119.975	0.000119975	Pyrite
FeS	87.911	0.000087911	Pyrrhotite
SiO_2	60.090	0.000060090	Quartz
$FeCO_3$	115.857	0.000115857	Siderite

---

[84] *M* - Molar Mass

Table 7.1 continued…

Substance	*M* (g/mol)	*M* (tonnes/mol)	Name
Al_2O_3	101.960	0.000101960	Alumina
$Al_2(SO_4)_3$	342.152	0.000342152	Aluminum sulfate
$KAl_3(OH)_6(SO_4)_2$	414.228	0.000414228	Alunite
$PbSO_4$	303.264	0.000303264	Anglesite
Sb_2O_3	291.400	0.000291400	Antimony trioxide
H_3AsO_4	141.950	0.000141950	Arsenic acid
As_2O_3	197.840	0.000197840	Arsenic trioxide
FeAsS	162.831	0.000162831	Arsenopyryte
Bi_2O_3	465.960	0.000465960	Bismuth oxide
Bi_2S_3	514.152	0.000514152	Bismuth(III) sulfide
$CaSO_4$	136.144	0.000136144	Calcium sulphate
CO_2	44.010	0.000044010	Carbon dioxide
CO	28.010	0.000028010	Carbon monoxide
CoAsS	165.914	0.000165914	Cobaltite
Fe_2SiO_4	203.784	0.000203784	Fayalite
$FeAsO_4$	194.767	0.000194767	Ferric arsenate
$Fe_2(SO_4)_3$	399.886	0.000399886	Ferric sulphate
$Fe(SO_4)$	151.911	0.000151911	Ferrous sulphate
PbS	239.264	0.000239264	Galena
FeO(OH)	88.857	0.000088857	Goethite
Ag	107.897	0.000107897	Silver
$CaSO_4.2H_2O$	172.184	0.000172184	Gypsum
$H_3OFe_3(SO_4)_2(OH)_6$	480.759	0.000480759	Hydronium jarosite
FeO	71.847	0.000071847	Iron (II) oxide
$Fe_2O_3.H_2O$	177.714	0.000177714	Iron(III) oxide hydrate
$KFe_3(SO_4)_2(OH)_6$	500.829	0.000500829	Jarosite
$Al_2O_3.2SiO_2.2H_2O$	258.180	0.000258180	Kaolinite
PbO	223.200	0.000223200	Lead oxide
$MgSO_4$	120.374	0.000120374	Magnesium sulphate
Fe_3O_4	231.541	0.000231541	Magnetite
MoS_2	160.068	0.000160068	Molybdenum
H_2MoO_4	161.960	0.000161960	Molybdic Acid
$NiSO_4$	154.764	0.000154764	Nickel sulphate
As_2S_3	246.032	0.000246032	Orpiment
$KAlSi_3O_8$	278.350	0.000278350	Orthoclase feldspar
O_2	32.000	0.000032000	Oxygen
$(Fe,Ni)_9S_8$	771.974	0.000771974	Pentlandite
K_2SO_4	174.264	0.000174264	Pottasium sulphate
As_2S_2	213.968	0.000213968	Realgar
H_4SiO_4	96.130	0.000096130	Silicic acid

Table 7.1 continued…

Substance	*M* (g/mol)	*M* (tonnes/mol)	Name
Au	196.967	0.000196967	Gold
ZnS	97.444	0.000097444	Sphalerite
Sb_2S_3	339.592	0.000339592	Stibnite
SO_2	64.064	0.000064064	Sulphur dioxide
H_2SO_4	98.084	0.000098084	Sulphuric acid
Sb_4O_6	582.800	0.000582800	Tetrantimony hexaoxide
As_4O_6	395.680	0.000395680	Tetrarsenic hexaoxide
ZnO	81.380	0.000081380	Zinc oxide
$ZnSO_4$	161.444	0.000161444	Zinc sulphate

7.2 Appendix B: Smelter

7.2.1 Reactions considered in smelting and converting.

1. $As_2S_3(s) + {}^{9}/_{2}\ O_2(g) \rightarrow As_2O_3(l) + 3\,SO_2(g)$
2. $Sb_2S_3(s) + {}^{9}/_{2}\ O_2(g) \rightarrow Sb_2O_3(l) + 3\,SO_2(g)$
3. $As_2S_3(s) + {}^{9}/_{2}\ O_2(g) \rightarrow {}^{1}/_{2}\ As_4O_6(g) + 3\,SO_2(g)$
4. $Sb_2S_3(s) + {}^{9}/_{2}\ O_2(g) \rightarrow {}^{1}/_{2}\ Sb_4O_6(g) + 3\,SO_2(g)$
5. $Bi_2S_3(s) + {}^{9}/_{2}\ O_2(g) \rightarrow Bi_2O_3(l) + 3\,SO_2(g)$
6. $2\,CuSO_4(s) + 3\,FeS_2(s) \rightarrow Cu_2S\,(l) + 3\,FeS\,(l) + 4\,SO_2(g)$
7. $FeAsS(s) + {}^{9}/_{4}\ O_2\,(g) \rightarrow {}^{1}/_{2}\ As_2O_3\,(l) + FeO\,(l) + SO_2(g)$
8. $FeAsS(s) + {}^{9}/_{4}\ O_2\,(g) \rightarrow {}^{1}/_{4}\ As_4O_6\,(g) + FeO\,(l) + SO_2(g)$
9. $FeS\,(s) + {}^{3}/_{2}\ O_2\,(g) \rightarrow FeO\,(l) + SO_2\,(g)$
10. $ZnS\,(s) + {}^{3}/_{2}\ O_2(g) \rightarrow ZnO\,(l) + SO_2(g)$
11. $FeS_2\,(s) + O_2(g) \rightarrow FeS\,(l) + SO_2\,(g)$
12. $PbS\,(s) + {}^{3}/_{2}\ O_2\,(g) \rightarrow PbO\,(l) + SO_2\,(g)$
13. $As_2S_3\,(s) + O_2\,(g) \rightarrow As_2S_2\,(l) + SO_2\,(g)$
14. $CuS\,(s) + {}^{1}/_{2}\ O_2\,(g) \rightarrow {}^{1}/_{2}\ Cu_2S\,(l) + {}^{1}/_{2}\ SO_2\,(g)$
15. $CuFeS_2\,(s) + {}^{1}/_{2}\ O_2\,(g) \rightarrow {}^{1}/_{2}\ Cu_2S\,(l) + FeS\,(l) + {}^{1}/_{2}\ SO_2\,(g)$
16. $FeAsS\,(s) + {}^{1}/_{2}\ O_2\,(g) \rightarrow {}^{1}/_{2}\ As_2S_2\,(l) + FeO\,(l)$
17. $FeO\,(s) + {}^{1}/_{6}\ O_2\,(g) \rightarrow {}^{1}/_{3}\ Fe_3O_4\,(l)$
18. $Cu\,(s) + {}^{1}/_{4}\ O_2\,(g) \rightarrow {}^{1}/_{2}\ Cu_2O\,(l)$
19. $FeO\,(s) + SiO_2\,(s) \rightarrow FeO \cdot SiO_2\,(l)$
20. $4Cu_3AsS_4\,(s) + 13O_2 \rightarrow 6Cu_2S(l) + 10SO_2(g) + As_4O_6(g)$
21. $Ca(CO_3)\,(\mathrm{s}) \rightarrow CaO\,(l) + CO_2\,(g)$
22. $Mg(CO_3)\,(s) \rightarrow MgO\,(l) + CO_2\,(g)$
23. $Fe(CO_3)\,(\mathrm{s}) \rightarrow FeO\,(l) + CO_2\,(g)$
24. $CaMg(CO_3)_2\,(\mathrm{s}) \rightarrow CaO\,(l) + MgO\,(l) + 2CO_2\,(g)$
25. $2\,FeO(OH)\,(\mathrm{s}) \rightarrow Fe_2O_3\,(l) + H_2O\,(g)$

Please refer to the literature review (Chapter 2) for the main reactions considered in each stage of the pyrometallurgical processing route.

7.2.2 The distribution of elements in the smelter and refinery

Table 7.2: Assumed distribution of elements in an Isasmelt furnace. Davenport et al.'s (2002) Flash and Isasmelt data used as guideline

Element	Matte	Slag	Gas
Copper	97	2	1
Al, Ti	0	100	0
Ag, Au, Pt	95	2	3
Sb	60	20	20
As	15	25	60
Bi	30	20	50
Co	45	45	10
Pb	55	15	30
Ni	75	20	5
Se	85	10	5
Zn	40	50	10

Table 7.3: The distribution of elements in a Flash furnace. Data from Davenport et al. (2002)

Element	Matte	Slag	Gas
Copper	97	2	1
Al, Ti	0	100	0
Ag, Au, Pt	99	1	0
Sb	60-70	5-35	5-25
As	15-40	5-25	35-80
Bi	30-75	5-30	15-60
Co	45-55	45-55	0-5
Pb	45-80	15-20	5-40
Ni	70-80	20-25	0-5
Se	85	5-15	0-5
Te	60-80	10-30	0-10
Zn	40	55	5

Table 7.4: The distribution of elements during Pierce Smith converting. Davenport et al.'s (2002) data used as guideline

Element	Blister	Slag	Gas
Copper	99	2	1
Al, Ti	0	100	0
Ag, Au, Pt	95	2	3
Sb	59	26	15
As	50	32	18
Bi	55	23	22
Co	40	55	5
Pb	5	49	46
Ni	75	20	5
Se	70	5	25
Zn	8	79	13

Table 7.5: The distribution of anode elements during electrorefining (Davenport et al., 2002)

	Slimes	Electrolyte
Cu	0.1	99.9
Au	100	0
Ag	99	1
Se	98	2
Te	98	2
Pb	98	2
Bi	60	40
Sb	60	40
As	25	75
S	1	99
Ni	1	99
Co	1	99
Fe	0	100
Zn	0	100

7.3 Appendix C: POX

7.3.1 Reactions considered in the leach autoclave

1. $Cu_2S + O_2 + 2\,H_2SO_4 \rightarrow 2\,CuSO_4 + S° + 2H_2O$
2. $CuFeS_2 + O_2 + 2\,H_2SO_4 \rightarrow CuSO_4 + FeSO_4 + 2\,H_2O + 2\,S°$
3. $FeS_2 + 0.5\,O_2 + H_2SO_4 \rightarrow FeSO_4 + 2\,S° + H_2O$
4. $FeSO_4 + {}^{1}/_{4}\,O_2 + {}^{1}/_{2}\,H_2SO_4 \rightarrow {}^{1}/_{2}\,Fe_2(SO_4)_3 + {}^{1}/_{2}\,H_2O$
5. $Fe_2(SO_4)_3 + 3\,H_2O \rightarrow Fe_2O_3 + 3\,H_2SO_4$
6. $S° + {}^{3}/_{2}\,O_2 + H_2O \rightarrow H_2SO_4$
7. $H_2SO_4 + CaCO_3 + H_2O \rightarrow CaSO_4 \cdot 2H_2O + CO_2$
8. $H_2SO_4 + MgCO_3 \rightarrow MgSO_4 + H_2O + CO_2$
9. $H_2SO_4 + FeCO_3 \rightarrow FeSO_4 + H_2O + CO_2$
10. $Cu_3AsS_4 + 2.75\,O_2 + 3\,H_2SO_4 \rightarrow 3\,CuSO_4 + 4S° + H_3AsO_4 + 1.5\,H_2O$
11. $Fe_2(SO_4)_3 + 2\,H_3AsO_4 \rightarrow 2\,FeAsO_4 + 3\,H_2SO_4$
12. $As_2S_3 + 3.75\,O_2 + 4.5\,H_2O \rightarrow 2\,H_3AsO_3 + 1.5\,H_2SO_4 + 1.5\,S$
13. $FeAsS + 1.75\,O_2 + H_2SO_4 + 0.5\,H_2O \rightarrow H_3AsO_4 + FeSO_4 + S$
14. $ZnS + {}^{1}/_{2}\,O_2 + H_2SO_4 \rightarrow ZnSO_4 + S° + H_2O$
15. $PbS + H_2SO_4 + {}^{1}/_{2}\,O_2 \rightarrow PbSO_4 + S° + H_2O$
16. $CaMg(CO_3)_2 + 2H_2SO_4 \rightarrow CaSO_4 + MgSO_4 + 2CO_2 + 2H_2O$
17. $Fe_2O_3 + 3\,H_2SO_4 \rightarrow Fe_2(SO_4)_3 + 3H_2O$
18. $2\,FeO(OH) + 3\,H_2SO_4 \rightarrow Fe_2(SO_4)_3 + 4H_2O$
19. $2\,KAlSi_3O_8 + 4H_2SO_4 + 8H_2O \rightarrow K_2SO_4 + Al_2(SO_4)_3 + 6H_4SiO_4$
20. $3\,Fe_2(SO_4)_3 + K_2SO_4 + 12\,H_2O \rightarrow 2KFe_3(SO_4)_2(OH)_6 + 6\,H_2SO_4$
21. $3\,Al_2(SO_4)_3 + K_2SO_4 + 12H_2O \rightarrow 2\,KAl_3OH_6(SO_4)_2 + 6\,H_2SO_4$
22. $H_2O + Al_2(SO_4)_3 + 2H_4SiO_4 \rightarrow Al_2O_3 \cdot 2SiO_2 \cdot 2H_2O + 3\,H_2SO_4$
23. $2\,KAl_3Si_3O_{10}(OH)_2 + 10\,H_2SO_4 \rightarrow K_2SO_4 + 3\,Al_2(SO_4)_3 + 6\,H_4SiO_4$
24. $H_3AsO_3\,(aq) + {}^{3}/_{2}\,Fe_2(SO_4)_3\,(aq) + H_2O\,(l)$

    $\rightarrow FeAsO_4\,(s) + 2\,FeSO_4\,(aq) + {}^{5}/_{2}\,H_2SO_4\,(aq)$

7.3.2 Reactions considered in acid bleed and solids neutralization

24. $FeSO_4 + CaCO_3 + {}^{5}/_{2}\ H_2O + {}^{1}/_{4}\ O_2 \rightarrow {}^{1}/_{2}\ Fe_2O_3 \cdot H_2O + CaSO_4 \cdot 2H_2O + CO_2$
25. $Fe_2(SO_4)_3 + 3\ CaCO_3 + 2\ H_3AsO_4 + 3\ H_2O \rightarrow 2\ FeAsO_4 + 3\ CaSO_4 \cdot 2H_2O + 3\ CO_2$
26. $Fe_2(SO_4)_3 + 3\ CaCO_3 + 7\ H_2O \rightarrow Fe_2O_3 \cdot H_2O + 3\ CaSO_4 \cdot 2H_2O + 3\ CO_2$
27. $H_2SO_4 + CaCO_3 + H_2O \rightarrow CaSO_4 \cdot 2H_2O + CO_2$
28. $CuSO_4 + 2\ CaCO_3 + 6\ H_2O \rightarrow CuSO_4.2Cu(OH)_2 + 2\ CaSO_4 \cdot 2H_2O + 2\ CO_2$

7.4 Appendix D: Heap leaching

7.4.1 Reactions considered in heap and ROM leaching

1. $Cu_2S + O_2 + 2\,H_2SO_4 \rightarrow 2\,CuSO_4 + S° + 2H_2O$
2. $Cu_5FeS_4 + 3\,O_2 + 6\,H_2SO_4 \rightarrow 5\,CuSO_4 + 4\,S° + FeSO_4 + 6\,H_2O$
3. $CuFeS_2 + O_2 + 2\,H_2SO_4 \rightarrow CuSO_4 + FeSO_4 + 2\,H_2O + 2\,S°$
4. $CuS + {}^1/_2\ O_2 + H_2SO_4 \rightarrow CuSO_4 + H_2O + S°$
5. $FeS_2 + 0.5\,O_2 + H_2SO_4 \rightarrow FeSO_4 + 2\,S° + H_2O$
6. $FeS + {}^1/_2\ O_2 + H_2SO_4 \rightarrow FeSO_4 + S° + H_2O$
7. $(Fe,Ni)9S8 + 17.625O2 + 3.25H2SO4 \rightarrow 4.5NiSO4 + 2.25Fe2(SO4)3 + 3.25H2O$
8. $PbS + Fe_2(SO_4)_3 \rightarrow PbSO_4 + 2\,FeSO_4 + S°$
9. $ZnS + {}^1/_2\,O_2 + H_2SO_4 \rightarrow ZnSO_4 + S° + H_2O$
10. $2\,MoS_2 + 9\,O_2 + 6\,H_2O \rightarrow 2\,H_2MoO_4 + 4\,H_2SO_4$
11. $FeAsS + 3.5\,O_2 + H_2O \rightarrow FeAsO_4 + H_2SO_4$
12. $Cu_3(OH)_2(CO_3)_2 + 3H_2SO_4 \rightarrow 3\,CuSO_4 + 2CO_2 + 4\,H_2O$
13. $Cu_2(OH)_2(CO_3) + 2\,H_2SO_4 \rightarrow 2\,CuSO_4 + CO_2 + 3\,H_2O$
14. $CuO + H_2SO_4 \rightarrow CuSO_4 + H_2O$
15. $Cu_2O + H_2SO_4 \rightarrow Cu + CuSO_4 + H_2O$
16. $CuSiO_3 \cdot 2H_2O + H_2SO_4 \rightarrow CuSO_4 + SiO_2 + 3\,H_2O$
17. $Cu_4(OH)_6SO_4 + 3\,H_2SO_4 \rightarrow 4\,CuSO_4 + 6\,H_2O$
18. $Cu_3AsS_4 + 2.75\,O_2 + 3\,H_2SO_4 \rightarrow 3\,CuSO_4 + 4S° + H_3AsO_4 + 1.5\,H_2O$
19. $CaMg(CO_3)_2 + 2H_2SO_4 \rightarrow CaSO_4 + MgSO_4 + 2CO_2 + 2H_2O$
20. $CaCO_3 + H_2SO_4 + H_2O \rightarrow CaSO_4 \cdot 2H_2O + CO_2$
21. $Fe_2(SO_4)_3 + 2\,H_3AsO_4 \rightarrow 2\,FeAsO_4 + 3\,H_2SO_4$
22. $FeSO_4 + {}^1/_4\ O_2 + {}^1/_2\ H_2SO_4 \rightarrow {}^1/_2\ Fe_2(SO_4)_3 + {}^1/_2\ H_2O$
23. $Fe_2(SO_4)_3 + 3\,H_2O \rightarrow Fe_2O_3 + 3\,H_2SO_4$
24. $Fe_2(SO_4)_3 + 4\,H_2O \rightarrow Fe_2O_3 \cdot H_2O + 3\,H_2SO_4$
25. ${}^3/_2\,Fe_2(SO_4)_3 + 7\,H_2O \rightarrow H_3OFe_3(SO_4)_2(OH)_6 + {}^5/_2\ H_2SO_4$
26. $S° + {}^3/_2\ O_2 + H_2O \rightarrow H_2SO_4$
27. $MgCO_3 + H_2SO_4 \rightarrow MgSO_4 + H_2O + CO_2$
28. $Fe_2O_3 + 3\,H_2SO_4 \rightarrow Fe_2(SO_4)_3 + 3H_2O$
29. $2\,FeO(OH) + 3\,H_2SO_4 \rightarrow Fe_2(SO_4)_3 + 4H_2O$
30. $2\,KAlSi_3O_8 + 4H_2SO_4 + 8H_2O \rightarrow K_2SO_4 + Al_2(SO_4)_3 + 6H_4SiO_4$
31. $3\,Fe_2(SO_4)_3 + K_2SO_4 + 12\,H_2O \rightarrow 2KFe_3(SO_4)_2(OH)_6 + 6\,H_2SO_4$
32. $3\,Al_2(SO_4)_3 + K_2SO_4 + 12H_2O \rightarrow 2\,KAl_3(OH)_6(SO_4)_2 + 6\,H_2SO_4$

33. $H_2O + Al_2(SO_4)_3 + 2H_4SiO_4 \rightarrow Al_2O_3 \cdot 2SiO_2 \cdot 2H_2O + 3\ H_2SO_4$

34. $H_2SO_4 + FeCO_3 \rightarrow FeSO_4 + H_2O + CO_2$

35. $As_2S_3 + 3.75\ O_2 + 4.5\ H_2O \rightarrow 2\ H_3AsO_3 + 1.5\ H_2SO_4 + 1.5\ S°$

36. $2\ KAl_3Si_3O_{10}(OH)_2 + 10\ H_2SO_4 \rightarrow K_2SO_4 + 3\ Al_2(SO_4)_3 + 6\ H_4SiO_4$

37. $H_3AsO_3\ (aq) + {}^3/_2\ Fe_2(SO_4)_3\ (aq) + H_2O\ (l) \rightarrow FeAsO_4\ (s) + 2\ FeSO_4\ (aq) + {}^5/_2\ H_2SO_4\ (aq)$

7.5 Appendix E: Energy Model (Marsden, 2008)

7.5.1 Unit Conversions

Table 7.6: Unit conversions

1 KWh	=	3600	Kj
1 Kj	=	0.000278	KWh
1 gal	=	3.785	L
1 L	=	0.264	gal
1 lb	=	0.454	kg
1 kg	=	2.205	lb
1 mile	=	1.609	km
1 km	=	0.621	miles
1 short ton	=	0.907	tonnes
1 nautical mile	=	1.150779	miles

7.5.2 *Marsden's (2008) key assumptions*

Table 7.7: Key assumptions used in the development of the model (Marsden, 2008)

		Units
Stripping ratio or all ores except ROM	3.00	
Stripping ratio for ROM	1.50	
Mine fragmentation power	0.02	kWh/ton
Mine transportation power	0.30	kWh/ton
Mine fragmentation diesel	0.002	gal/ton
Mine transportation diesel	0.08	gal/ton
Ore grade, %Cu except low grade ROM	0.50	
Ore grade, %Cu for low grade ROM	0.25	
Concentrate grade, %Cu	30.00	
ROM leach recovery	50.00	
Crushed ore heap leach recovery	75.00	
Mill recovery	90.00	
Smelter recovery	97.00	
Concentrate leach MT recovery	97.50	
Concentrate leach HT recovery	98.50	
Bond work index (BWI)	13.50	kWh/st
SAG power index (SPI)	150.00	minutes
Power requirements:		
Smelting	370.00	kWh/t-con
Concentrate leach MT oxygen plant	200.00	kWh/t-con
Concentrate leach MT - process	160.00	kWh/t-con
Concentrate leach HT oxygen plant	390.00	kWh/t-con
Concentrate leach HT - process	100.00	kWh/t-con
Concentrate leach EW	0.90	kWh/lb-Cu
Concentrate freight by truck	200.00	miles round trip
Concentrate freight by truck	22.50	st/truck
Concentrate freight by truck	0.20	gal-diesel/mile
Concentrate freight by truck	0.009	gal-diesel/ton-mile
Concentrate freight by truck	1.80	gal-diesel/t-con for trip
Concentrate freight by ship	6000.00	miles one way
Concentrate freight by ship	0.002	gal-oil/ton-mile
Concentrate freight by ship	12.00	gal-oil/t-con for trip

Table 7.7 continued …

		Units
Tertiary crushing and ball mill T80	8.00	mm
SAG/SABC and ball mill T80	2.00	mm
AG and ball mill	1.50	mm
HPGR and ball mill T80	2.00	mm
Final ball mill product size	150.00	µm
Bond Work Index after SAG/AG/Crush	13.50	kWh/ton
Bond Work Index after HPGR	12.60	kWh/ton
Ball mill specific power after HPGR	7.47	kWh/ton
Ball mill specific power after Tertiary crush	9.51	kWh/ton
Ball mill specific power after SAG w/pebble crush	8.00	kWh/ton
Ball mill specific power after SAG no pebble crush	8.00	kWh/ton
Ball miill specific power after AG	7.54	kWh/ton
SAG specific power	7.00	kWh/ton
AG specific power	7.00	kWh/ton
Pebble crusher after SAG	N/A	kWh/ton
Wear steel consumption:		
Secondary & Tertiary crusher liner wear	0.002	lb/t-ore
AG liner wear	0.050	lb/t-ore
SAG liner wear	0.045	lb/t-ore
Ball mill liner wear, primary (following AG or SAG)	0.020	lb/t-ore
Ball mill liner wear, primary (without AG or SAG)	0.040	lb/t-ore
SAG ball consumption	0.900	lb/t-ore
Pebble crusher liner wear	0.001	lb/t-ore
Ball mill consumption (following AG or SAG)	0.600	lb/t-ore
Ball mill consumption (without AG or SAG)	1.000	lb/t-ore
Regrind mill ball consumption	0.100	lb/t-ore
HPGR liner wear	0.002	lb/t-ore
Energy factor for steel	5.230	lb/t-ore

Table 7.8: Specific energy consumption by unit step by unit source (Marsden, 2008)

Energy Source	**Electricity**		**Natural Gas**	**Diesel/Oil**
Units	kWh/ton	kJ/ton	kJ/ton	kJ/ton
Mining (to feed ROM leach)	0.800	2882.000	0.000	28181.000
Mining (to feed all other processes)	1.280	4611.000	0.000	45089.000
ROM Leaching	1.000	3600.000	0.000	0.000
Primary Crushing & Conveying	1.990	7180.000	0.000	0.000
Secondary Crushing	0.990	3580.000	0.000	0.000
Tertiary Crushing	0.990	3580.000	0.000	0.000
Ball Milling (after tertiary crushing)	9.520	34275.000	0.000	0.000
SAG Milling (SABC)	7.010	25220.000	0.000	0.000
Ball Milling (after SAG)	8.010	28836.000	0.000	0.000
HPGR	2.400	8646.000	0.000	0.000
Ball Milling (after HPGR)	7.480	26914.000	0.000	0.000
Flotation & Regrinding	4.000	14411.000	0.000	0.000
Units	kWh/lb Cu	kJ/lb Cu	kJ/lb Cu	kJ/lb Cu
Transportation to Smelter	0.000	0.000	0.000	3247.000
Smelting	0.640	2289.000	2856.000	0.000
Refining	0.200	722.000	1976.000	0.000
MT Concentrate Leaching	0.620	2216.000	93.000	21.000
HT Concentrate Leaching	0.830	2986.000	93.000	21.000
SX	0.550	1975.000	0.000	0.000
Increamental SX (concentrate leach)	0.060	200.000	0.000	0.000
EW	0.900	3241.000	601.000	0.000
EW (Alternative Anode)	0.770	2754.000	601.000	0.000
EW (Ferrous/Ferric)	0.600	2171.000	53.000	0.000
Transportation to Market	0.000	0.000	0.000	122.000
Super Fine grinding	0.110	396.000		
Neutralisation (Stand Alone process HT POX estimate)	0.040			

Notes

- The energy data with units of per ton refers to energy consumption per ton of ore.
- Discussions with Dr J. Gediga[85] (Personal communication, 2010) seemed to suggest that shipping fuel oil consumption is about 0.000213536 gal oil / ton km (calculated from data obtained during the discussion). This value is nearly 10 times less than the value reported by Marsden (2008). These data sets can be confirmed in future studies. Marsden's (2008) data was used without any changes in the case study applied in this thesis.

---

[85] PE International

7.5.3 Verification of the application of Marsden's 2008 model

Prior to adapting Marsden's (2008) energy model in this study, the accuracy of its application was verified using model results published by Marsden (2008) for selected processing options (Figure 7.1 to Figure 7.5). Refer to Marsden (2008) and Table 7.7 for details regarding the configurations and assumptions used to derive the data presented in Figure 7.1 to Figure 7.5.

Mining / ROM leaching / SX / EW / Metal transportation to market

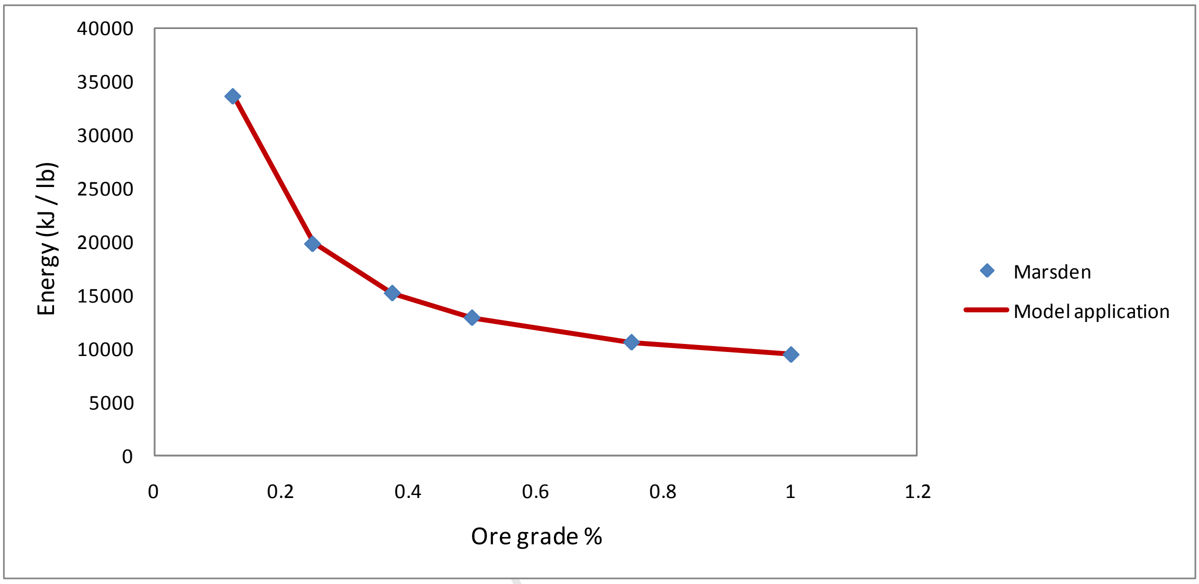
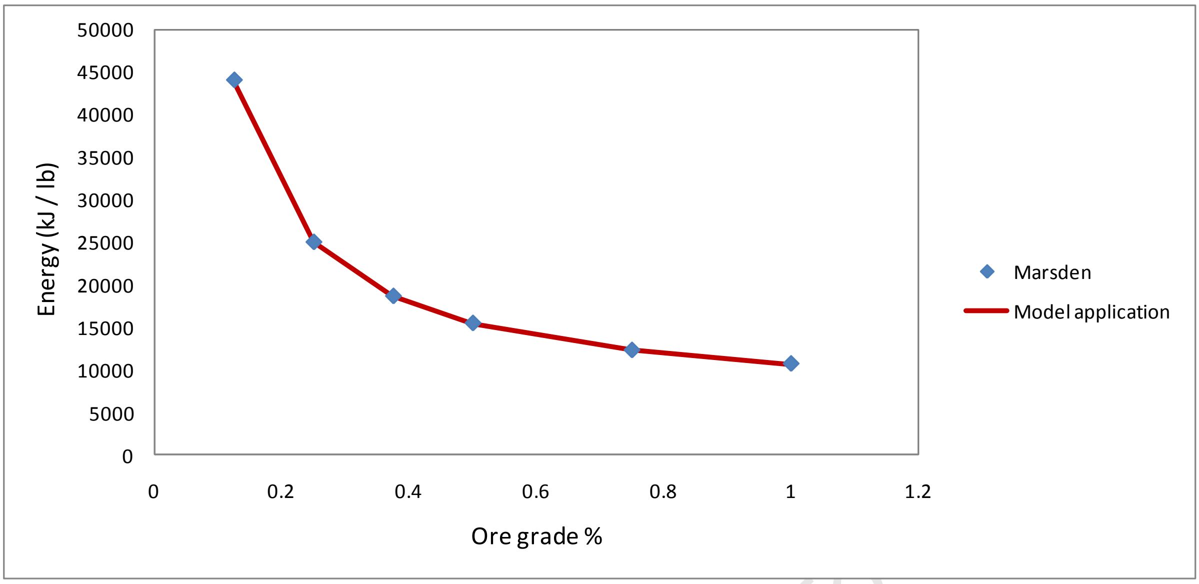


Figure 7.1: Total energy consumption for Mining / ROM leaching / SX / EW / Metal transportation to market

Mining / Primary / Secondary / Tertiary crushing / Heap leaching / SX / EW / Metal transportation to market



7.2: Total energy consumption for Mining / Primary / Secondary / Tertiary crushing / Heap leaching / SX / EW / Metal transportation to market

Mining / Primary crushing / AG milling / Ball milling / Flotation / Concentrate transportation / Smelting and refining / Metal transportation to market

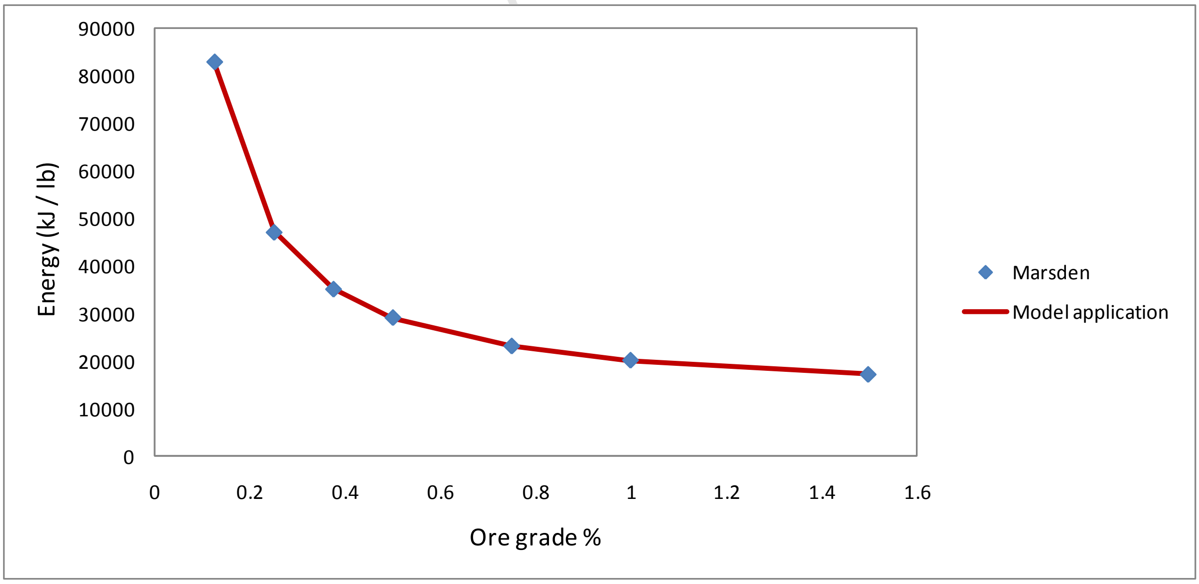


Figure 7.3: Total energy consumption for Mining / Primary crushing / AG milling / Ball milling / Flotation / Concentrate transportation / Smelting and refining / Metal transportation to market

Mining / Primary crushing / Secondary crushing/ HPGR / Ball milling / Concentrate transportation / Smelting and refining / Metal transportation to market

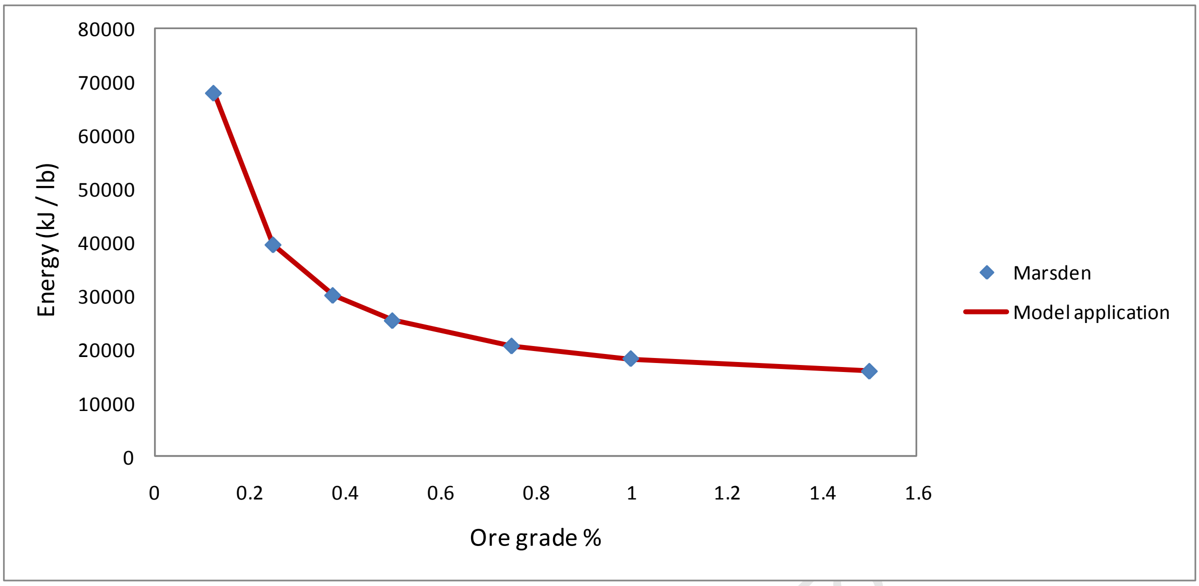


Figure 7.4: Total energy consumption for Mining / Primary crushing / Secondary crushing/ HPGR / Ball milling / Concentrate transportation / Smelting and refining / Metal transportation to market

Mining / Primary crushing / AG milling / Ball milling / Flotation / Concentrate transportation / POX / SX / EW / Metal transportation to market

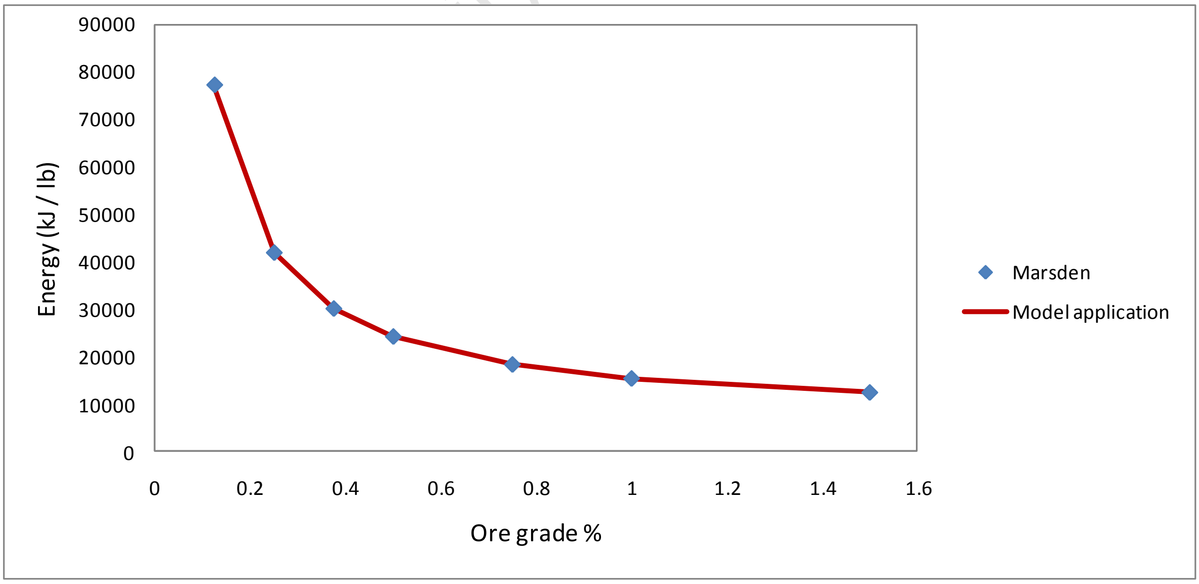


Figure 7.5: Total energy consumption for Mining / Primary crushing / AG milling / Ball milling / Flotation / Concentrate transportation / POX / SX / EW / Metal transportation to market

Appendix F: Gaseous Emissions

7.5.4 IPCC data used to calculate gaseous emissions (IPCC, 1996; IPCC, 2006)

Table 7.9: IPCC gaseous emissions data (kg emissions / TJ energy)

Fuel	CO_2 (kg)/TJ	CH_4 (kg)/TJ	N_2O (kg)/TJ	NMVOC (kg/TJ)	CO (kg/TJ)
Coal (bitum-av)	95000.0	1.0	1.5	20.0	150.0
Gasoline	69300.0	3.0	0.6		
Light Oil	74100.0	3.0	0.6	5.0	10.0
Diesel	74100.0	3.0	0.6	5.0	10.0
LPG	63100.0	1.0	0.1		
Natural Gas, Methane	56100.0	1.0	0.1	5.0	30.0
Crude oil	73000.0	3.0	0.6		
Kerosene	71900.0	3.0	0.6		
Wood	112000.0	30.0	4.0	50.0	2000.0
Peat	106000.0	1.0	1.5	20.0	20.0
Lignite	101000.0	1.0	1.5	20.0	20.0
Bio Energy	70800.0	3.0	0.6		
Anthracite	98300.0	1.0	3.0	20.0	20.0

Table 7.10: Gaseous emissions data (kg emissions / KWh energy)

Fuel	CO_2 (kg)/kWh	CH_4 (kg)/kWh	N_2O (kg)/kWh	NMVOC (kg/kWh)	CO (kg)/kWh
Coal (bitum-av)	3.4200E-01	3.6000E-06	5.4000E-06	7.2000E-05	5.4000E-04
Gasoline	2.4948E-01	1.0800E-05	2.1600E-06	0.0000E+00	0.0000E+00
Light Oil	2.6676E-01	1.0800E-05	2.1600E-06	1.8000E-05	3.6000E-05
Diesel	2.6676E-01	1.0800E-05	2.1600E-06	1.8000E-05	3.6000E-05
LPG	2.2716E-01	3.6000E-06	3.6000E-07	0.0000E+00	0.0000E+00
Natural Gas, Methane	2.0196E-01	3.6000E-06	3.6000E-07	1.8000E-05	1.0800E-04
Crude oil	2.6280E-01	1.0800E-05	2.1600E-06	0.0000E+00	0.0000E+00
Kerosene	2.5884E-01	1.0800E-05	2.1600E-06	0.0000E+00	0.0000E+00
Wood	4.0320E-01	1.0800E-04	1.4400E-05	1.8000E-04	7.2000E-03
Peat	3.8160E-01	3.6000E-06	5.4000E-06	7.2000E-05	7.2000E-05
Lignite	3.6360E-01	3.6000E-06	5.4000E-06	7.2000E-05	7.2000E-05
Bio Energy	2.5488E-01	1.0800E-05	2.1600E-06	0.0000E+00	0.0000E+00
Anthracite	3.5388E-01	3.6000E-06	1.0800E-05	7.2000E-05	7.2000E-05

7.5.5 NOx emissions

Table 7.11: NOx emissions data (The Engineering Tool Box, 2009)

Fuel	NOx Emissions (*g/kg fuel*)
Oil	3.00
Kerosene	3.00
Coal	4.50
Propane	2.30
Gasoline	27.00
Hydrogen	0.00
Natural Gas	1.00
Butane	2.30

7.5.6 Coal energy densities and compositions

Table 7.12: The energy densities and compositions of coal (Net Industries, 2010; Elert, 2010)

ASTM class	ASTM group	MJ/kg	C content	Average C %	S content %	kW/kg
Lignite	Lignite A	<14.6	46 - 60	53	0.48	
	Lignite B	14.6-19.3			0.48	5
Sub-bituminous	Sub-bituminous C	19.3-22.1	46 - 86	66	0.52	
	Sub-bituminous B	22.1-24.4			0.52	6.666667
	Sub-bituminous A	24.4-26.7			0.52	
	High volatile C	24.4-30.2			0.52	
Bituminous	High Volatile B	30.2-32.5	46 - 86	66	0.5	8.388889
	High Volatile A	>32.5			0.5	
	Medium volatile	>32.5			0.5	
	Low volatile	>32.5			0.5	
Anthracite	Semi-anthracite	>32.5	86 - 98	92	2.4	9.027778
	Anthracite	>32.5			2.4	
	Meta-anthracite	>32.5			2.4	

7.5.7 Environmental impact factors

Table 7.13: Global warming potential impact factors (Azapagic, 2010)

Burden	GWP 100 years vs CO_2
CO_2	1
CH_4	21
N_2O	310
HFCs	400
$CHCl_3$	100
PFCs	5000
SF_6	3200
Other volatile organic compounds	11

Table 7.14: Acidification potential impact factors (Azapagic, 2010)

Burden	AP vs SO_2
SO_2	1
NOx	0.7
HCl	0.88
HF	1.6
NH_3	1.88

7.6 Appendix G: Economics

7.6.1 Capital Costs

Chemical engineering plant cost index - CEPCI

Table 7.15: Chemical engineering plant cost indices (Curry hydrocarbons Inc., 2010)

Year	Index
2002	395.6
2004	444.2
2010 (June)	556.4

Smelter & refinery total capital investment

Table 7.16: Smelter total capital investment (Davenport et al., 2002; Peacey et al., 2003)

Facility	Fixed investment cost – US$/annual tonne of Cu	Fixed investment cost – US$/annual tonne of Cu updated
Smelter	2500	3516
Electrolytic refinery (excluding precious metals refinery)	500	703
Fixed Capital Investment	3000	4219
Other - % of fixed or total capital investment		
Land (% of fixed capital investment)	0	0.00
Start up (% of fixed capital investment)	6	253
Working Capital (% of total capital investment)	15	789
Total capital investment		**5262**

POX total capital costs

Table 7.17: POX Total capital investment (Marsden, 2004)

Facility	**Fixed investment cost – US$/annual tonne of Cu**	**Fixed investment cost – US$/annual tonne of Cu updated (Stand alone)**	**Fixed investment cost – US$/annual tonne of Cu updated (Operated with heap leaching)**
POX plant	1100	1378	1240
SX	350	438	438
EW	550	689	689
Fixed Capital Investment	2000	2505	2367
Other - % of fixed or total capital investment			
Land (% of fixed capital investment)	0	0	0
Start up (% of fixed capital investment)	6	150	142
Working Capital (% of total capital investment)	15	469	443
Total Investment		**3124**	**2952**

Notes

- There was no data available to the author regarding typical technology royalties for a smelter or a POX plant. Therefore, it was assumed, for the purposes of this study that the licensing / royalty fees for the technologies were included in the capital costs quoted in the literature, i.e. a “paid up royalty” as opposed to a “running royalty”.
- It was assumed that the land for expansions was already available and was not capitalized in these new projects (Couper, 2003).

7.6.2 Operating costs

Concentrator only - operating costs

Table 7.18: Operating costs of the concentrator (Mass and energy balances; Davenport et al., 2002)

Component	US$/tonne of ore updated
Energy**[86]	2.08
Operating labour	0.16
Maintenance labour	0.16
Maintenance and operating supplies	0.95
Reagents and grinding balls	0.79
Total	4.14

Smelter and refinery only – operating costs

Table 7.19: Smelter operating costs (Mass and energy balances; Peacey et al., 2003)

Component	US$/tonne Cu
Manpower	100
Power**	167
Fuel**	40
Maintenance	85
Consumables**	75
Total	467

---

[86] **Calculated from mass and energy balances. See Appendix A – E. Davenport et al's. (2002) and Peacey et al's. (2003) original data was updated using 3% inflation from 2003 to 2010.

Overall mine to smelter / refinery operating costs (updated)

Table 7.20: Mine and refining operating costs (Mass and energy balances; Davenport et al., 2002)

Mine & Smelter operating costs	US$/tonne Cu	US$/tonne ore
Open pit mining		2.03
Concentrator		4.14
Smelting / refinery	467	
Sales and distribution	63	
Local management and overhead	63	
Total	**593**	**6.17**

Overall mine to POX / SX / EW operating costs

Table 7.21: Mine and POX operating costs (Mass and energy balances; McElroy and Young, 1999; Davenport et al., 2002)

Mine and POX operating costs	US$/tonne	US$/tonne ore
Mine & Concentrator		
Open pit mining		2.03
Concentrator		4.14
Leaching		
Reagents**	167	
Power**	165	
Other fuel**	3	
Maintenance supplies	40	
Salaries and labour	18	
SX / EW		
Reagents**	38	
Power**	191	
Labour	27	
Maintenance supplies	24	
Other (Within SX/EW)	24	
Waste Management	61	
Cathode Marketing/Freight	92	
Local management and overhead	63	
Total	**913**	**6.17**

7.6.3 Reagent costs

Table 7.22: POX reagent costs (G. Dunn, personal communication, 2009)

	Cost – US$ /tonne copper	Cost – US$/tonne reagent
Lime		146
Limestone		39
Oxygen (Non energy component)		16
Biocide Water treatment	0.13	
Anti-scalant Water treatment	0.06	
Sodium Hydroxide Water treatment	0.02	
Other reagents & consumables (Allowance)	4.92	

Table 7.23: Smelting reagent costs (Public media; G. Dunn, personal communication, 2009; J. Bezuidenhout, personal communication, 2010).

	Cost - US$/tonne reagent
O2 (Non labour and power component)	16
Limestone	39
Lime	146
Silica	100

Notes

- The costs for external refining are a function of the copper contract terms. Refer to Chapter 4 for the base case terms used in this study.
- All the other assumptions made in this study were presented in Chapters 3 - 4.

7.7 Appendix H: Sample calculations

The calculations detailed here show the general methodology used to calculate:

- Energy requirements in kJ
- Fuel and electricity requirements in kg and kWh
- CO_2, CH_4, N_2O, NMVOC, CO, SO_2 and NOx emissions
- Global warming and Acidification potential

The sample calculations are shown for the POX process only and exclude mining, concentration, solvent extraction and electrowinning which are system units associated with the POX process. The methodology shown here was applied on all the system units considered in this study.

Inputs

Energy consumption - $E_i\left(\frac{kJ}{lb}\right)$; where *i* is an energy source e.g. electricity, diesel, coal etc.

Copper production rate after mass balance computation (Cu) = 300000 tonnes / annum

See Appendix E for POX energy consumption data and Appendix F for calorific values and gaseous emission factors.

Calculation of energy requirements excluding energy losses associated with electricity generation

General formula

$$Total\ energy\ requirements\ for\ each\ process\ unit\ \left(\frac{kJ}{annum}\right)$$

$$= \frac{\left(\sum_{i=1}^{N} E_i\left(\frac{kJ}{lb}\right)\right)}{0.454\left(\frac{kg}{lb}\right)} * Cu\ \left(\frac{tonnes}{annum}\right) * 1000\left(\frac{kg}{tonne}\right)$$

Example: Applied on the onsite energy requirements of the POX plant.

$$Total\ energy\ consumption\ for\ POX\ unit\ \left(\frac{kJ}{annum}\right)$$

$$= \frac{(2986 + 93 + 21 + 0 + 0)\left(\frac{kJ}{lb}\right)}{0.454\left(\frac{kg}{lb}\right)} * 300000\ \left(\frac{tonnes}{annum}\right) * 1000\left(\frac{kg}{tonne}\right)$$

$$= 2048458149780\left(\frac{KJ}{annum}\right) = 2.05\ \left(\frac{\mathrm{Pj}}{annum}\right)$$

Calculation of energy requirements including energy losses associated with electricity generation

General formula

$$Total\ energy\ consumption\ including\ electricity\ generation\left(\frac{kJ}{annum}\right)$$

$$=\left(\sum_{i=1}^{N}\frac{Onsite\ electricity\ consumption\ EE\left(\frac{kJ}{annum}\right)}{\frac{\%\ Electricity\ generation\ efficiency_i}{100}}\frac{*\ \%\ Energy\ source_i\ (\ in\ electricty\ mix)}{100}\right)$$

$$+\ Total\ non\ electricity\ energy\ requirements\ \left(\frac{kJ}{annum}\right)$$

Where EE is the total electric energy required by a unit and *i* is the source of electric energy, e.g. coal, hydroelectricity, natural gas etc biomass in the power plant.

Example: Applied on the energy consumed by the POX plant including energy consumed to generate electricity.

$$Total\ energy\ consumption\ including\ electricity\ generation\ for\ POX\ plant\left(\frac{kJ}{annum}\right)$$

$$=\left(\frac{(2986)\left(\frac{kJ}{lb}\right)}{0.454\left(\frac{kg}{lb}\right)}*\left(\frac{\frac{58.6}{100}}{\frac{40}{100}}+\frac{\frac{33.5}{100}}{\frac{35}{100}}+\frac{\frac{0}{100}}{\frac{35}{100}}+\frac{\frac{0.4}{100}}{\frac{85}{100}}+\frac{\frac{0}{100}}{\frac{30}{100}}+\frac{\frac{7.5}{100}}{\frac{35}{100}}+\frac{\frac{0}{100}}{\frac{35}{100}}\right)\right.$$

$$\left.*\ 300000\ \left(\frac{tonnes}{annum}\right)*1000\left(\frac{kg}{tonne}\right)\right)+\frac{(93+21+0+0)\left(\frac{kJ}{lb}\right)}{0.454\left(\frac{kg}{lb}\right)}$$

$$*\ 300000\ \left(\frac{tonnes}{annum}\right)*1000\left(\frac{kg}{tonne}\right)=5286626087439\left(\frac{kJ}{annum}\right)$$

$$=5.29\left(\frac{pJ}{annum}\right)$$

Calculation of onsite fuel requirements $\mathbf{F_i}$ $(\mathbf{kg/annum})$

General formula

$$Onsite\ fuel\ consumption\ \ F_i\left(\frac{kg\ fuel}{annum}\right) = \frac{\left(\dfrac{E_i\left(\frac{kJ}{lb}\right) * Cu\ \left(\frac{tonnes}{annum}\right) * 1000\left(\frac{kg}{tonne}\right)}{3600\,\frac{kJ}{kWh} * 0.454\left(\frac{kg}{lb}\right)}\right)}{Calorific\ content\ of\ fuel\ (\frac{kWh}{kg\ fuel})}$$

Where *i* is fuel type e.g. coal, diesel, natural gas.

Example: Applied on the fuel required for the POX plant operations.

$$Onsite\ fuel\ consumption\ \ F_{Diesel}\left(\frac{kg\ fuel}{annum}\right) = \frac{\left(\dfrac{21\left(\frac{kJ}{lb}\right) * 300000\ \left(\frac{tonnes}{annum}\right) * 1000\left(\frac{kg}{tonne}\right)}{3600\left(\frac{kJ}{kWh}\right) * 0.454\left(\frac{kg}{lb}\right)}\right)}{11.944\ \left(\frac{kWh}{kg\ fuel}\right)}$$

$$= 322725\ kg/annum$$

$$Onsite\ fuel\ consumption\ \ F_{Natural\ gas}\left(\frac{kg\ fuel}{annum}\right)$$

$$= \frac{\left(\dfrac{93\left(\frac{kJ}{lb}\right) * 300000\ \left(\frac{tonnes}{annum}\right) * 1000\left(\frac{kg}{tonne}\right)}{3600\left(\frac{kJ}{kWh}\right) * 0.454\left(\frac{kg}{lb}\right)}\right)}{13.333\ \left(\frac{kWh}{kg\ fuel}\right)} = 1280318\ kg/annum$$

Volume consumption can be obtained by dividing the mass flows of each fuel by their respective densities.

Calculation of onsite emissions due to fuel use in the plant (non electricity energy sources onsite)

General formula

$$Gaseous\ emissions\ \left(\frac{kg}{annum}\right) = \sum_{i=1}^{N}\left(\frac{\frac{E_i\left(\frac{kJ}{lb}\right)}{0.454\left(\frac{kg}{lb}\right)} * Cu\ (tonnes/annum) * 1000\left(\frac{kg}{tonne}\right) *\ IPCC\ gas\ emission\ factor_i\ \left(\frac{kg\ gas}{kWh}\right)}{3600\left(\frac{kJ}{kWh}\right)}\right)$$

Where $E_i(\frac{kJ}{lb})$ is energy (E) from source *i*, e.g. natural gas, diesel and coal (excluding electricity in this case since emissions due to fuel use only, in the plant, are being calculated).

Example: Applied on the CO_2 emissions from the fuel used during POX plant operations

$$CO_2\ emissions\ \left(\frac{kg}{annum}\right) = \sum_{i=1}^{N}\left(\frac{\frac{E_i\left(\frac{kJ}{lb}\right)}{0.454\left(\frac{kg}{lb}\right)} * Cu\ (tonnes/annum) * 1000\left(\frac{kg}{tonne}\right) *\ IPCC\ CO_2\ emission\ factor_i\ \left(\frac{kg\ CO_2}{kWh}\right)}{3600\left(\frac{kJ}{kWh}\right)}\right)$$

$$\begin{aligned} CO_2\ emissions\ \left(\frac{kg}{annum}\right) &= \frac{\frac{93\left(\frac{kJ}{lb}\right)}{0.454\left(\frac{kg}{lb}\right)} * 300000\ (tonnes/annum) * 1000\left(\frac{kg}{tonne}\right) *\ 0.20196\ \left(\frac{kg\ CO_2}{kWh}\right)}{3600\left(\frac{kJ}{kWh}\right)} \\ &+ \frac{\frac{21\left(\frac{kJ}{lb}\right)}{0.454\left(\frac{kg}{lb}\right)} * 300000\ (tonnes/annum) * 1000\left(\frac{kg}{tonne}\right) *\ 0.26676\ \left(\frac{kg\ CO_2}{kWh}\right)}{3600\left(\frac{kJ}{kWh}\right)} \\ &= 4475815\ \left(\frac{kg}{annum}\right) \end{aligned}$$

The methodology used to calculate the CO_2 emissions was used to calculate the CH_4, N_2O, NMVOC and CO emissions. The emission factors for CO_2, CH_4, N_2O, NMVOC and CO where available in units of $\left(\frac{\text{kg gas}}{\text{kWh}_\text{i}}\right)$, whereas the emission factors for SO_2 and NOx were available in units of $\left(\frac{\text{kg SO}_2}{\text{kg fuel}_\text{i}}\right)$ and $\left(\frac{\text{g NO}_\text{x}}{\text{kg fuel}_\text{i}}\right)$ respectively, where i is a fuel type e.g. coal, diesel and natural gas. Adjustments for SO_2 and NOx emissions were incorporated as shown below.

$$CH_4\ emissions\ \left(\frac{kg}{annum}\right)$$

$$= \sum_{i=1}^{N}\left(\frac{\frac{E_i\left(\frac{kJ}{lb}\right)}{0.454\left(\frac{kg}{lb}\right)} * Cu\ (tonnes/annum) * 1000\left(\frac{kg}{tonne}\right) *\ IPCC\ CH_4\ emission\ factor_i\ \left(\frac{kg\ CH_4}{kWh}\right)}{3600\left(\frac{kJ}{kWh}\right)}\right)$$

$$= 103\ \left(\frac{kg}{annum}\right)$$

$$N_2O\ emissions\ \left(\frac{kg}{annum}\right)$$

$$= \sum_{i=1}^{N}\left(\frac{\frac{E_i\left(\frac{kJ}{lb}\right)}{0.454\left(\frac{kg}{lb}\right)} * Cu\ (tonnes/annum) * 1000\left(\frac{kg}{tonne}\right) *\ IPCC\ N_2O\ emission\ factor_i\ \left(\frac{kg\ N_2O}{kWh}\right)}{3600\left(\frac{kJ}{kWh}\right)}\right)$$

$$=\ 14.5\ \left(\frac{kg}{annum}\right)$$

$$NMVOC\ emissions\ \left(\frac{kg}{annum}\right)$$

$$= \sum_{i=1}^{N}\left(\frac{\frac{E_i\left(\frac{kJ}{lb}\right)}{0.454\left(\frac{kg}{lb}\right)} * Cu\ (tonnes/annum) * 1000\left(\frac{kg}{tonne}\right) *\ IPCC\ NMVOC\ emission\ factor_i\ \left(\frac{kg\ NMVOC}{kWh}\right)}{3600\left(\frac{kJ}{kWh}\right)}\right)$$

$$= 377\ \left(\frac{kg}{annum}\right)$$

$$CO\ emissions\ \left(\frac{kg}{annum}\right)$$

$$= \sum_{i=1}^{N}\left(\frac{\dfrac{E_i\left(\frac{kJ}{lb}\right)}{0.454\left(\frac{kg}{lb}\right)} * Cu\ \left(\frac{tonnes}{annum}\right) * 1000\left(\frac{kg}{tonne}\right) * \ IPCC\ CO\ emission\ factor_i\ \left(\frac{kg\ CO}{kWh}\right)}{3600\left(\frac{kJ}{kWh}\right)}\right)$$

$$= 1982\ \left(\frac{kg}{annum}\right)$$

$$SO_2 emissions\ \left(\frac{kg}{annum}\right)$$

$$= \sum_{i=1}^{N}\left(\frac{\dfrac{\dfrac{E_i\left(\frac{kJ}{lb}\right)}{0.454\left(\frac{kg}{lb}\right)} * Cu\ \left(\frac{tonnes}{annum}\right) * 1000\left(\frac{kg}{tonne}\right)}{3600\frac{kJ}{kWh} *} * SO_2 emission\ factor_i\left(\frac{kg\ SO_2}{kg\ fuel}\right)}{Calorific\ content\ of\ fuel_i\ \left(\frac{kWh}{kg\ fuel}\right)}\right)$$

$$= 12.1\ \left(\frac{kg}{annum}\right)$$

It was assumed that the sulphur contents of the various fuels are as follows: diesel – 15ppm; natural gas – 1 ppm; coal – 0.5%, heavy fuel oil for ships – 1.5.%.

$$NO_x emissions\ \left(\frac{kg}{annum}\right)$$

$$= \sum_{i=1}^{N}\left(\frac{\dfrac{\dfrac{E_i\left(\frac{kJ}{lb}\right)}{0.454\left(\frac{kg}{lb}\right)} * Cu\ \left(\frac{tonnes}{annum}\right) * 1000\left(\frac{kg}{tonne}\right)}{3600\frac{kJ}{kWh}} * \dfrac{NO_X emission\ factor_i\left(\frac{g\ NO_x}{kg\ fuel}\right)}{\left(\frac{1000g}{kg}\right)}}{Calorific\ content\ of\ fuel_i\ \left(\frac{kWh}{kg\ fuel}\right)}\right)$$

$$= 2248\left(\frac{kg}{annum}\right)$$

Offsite emissions due to electricity generation

General formula

$$Gaseous\ emissions\ \left(\frac{kg}{annum}\right)$$
$$= \sum_{i=1}^{N} \frac{Onsite\ electricity\ consumption\ EE\ \left(\frac{kWh}{annum}\right)}{Electricity\ generation\ efficiency_i}$$
$$*\ \%\ Energy\ source_i\ (\ in\ electricty\ mix)$$
$$*\ IPCC\ gas\ emission\ factor_i\ \left(\frac{kg\ gas}{kWh}\right)$$

Where i = is energy source in a country's electricity mix, i.e. coal, natural gas, nuclear, hydro, renewable and diesel. Nuclear, hydro, and renewable energy sources are considered clean energy sources in the model. However, any possible emissions associated with these energy sources can be easily incorporated into the model when reliable emission factors are available.

Example: Applied on the CO_2 emissions from power plants that generate electricity for the POX plant.

$$CO_2 emissions\ \left(\frac{kg}{annum}\right)$$
$$= \sum_{i=1}^{N} \frac{Offsite\ electricity\ consumption\ EE\ \left(\frac{kWh}{annum}\right)}{Electricity\ generation\ efficiency_i} *\ \%\ Energy\ source_i$$
$$*\ \ IPCC\ CO_2\ emission\ factor_i\ \left(\frac{kg\ gas}{kWh}\right)$$

Assuming electricity mix is 58.6% natural gas, 33.5% coal, 0% nuclear, 0.4% hydro, 0% renewable, 7.5% diesel / oil and 0% biomass. Assuming coal type is bituminous (Analysis – Carbon 66%, Sulphur 0.5%, 8.38889 kWh / kg).

$$CO_2 emissions \left(\frac{kg}{annum}\right)$$

$$= \left(\frac{\frac{(2986)\left(\frac{kJ}{lb}\right)}{0.454\left(\frac{kg}{lb}\right)} * 300000 \left(\frac{tonnes}{annum}\right) * 1000 \left(\frac{kg}{tonne}\right)}{3600 \frac{kJ}{kWh}} \right)$$

$$* \left(\frac{\frac{58.6}{100}}{\frac{40}{100}} * 0.20196 \left(\frac{kg\ gas}{kWh}\right) + \frac{\left(\frac{\frac{\frac{33.5}{100}}{\frac{35}{100}}}{8.38889 \left(\frac{kWh}{kg\ coal}\right)} * \frac{66}{100} \right) * 44 (Mr\ CO_2)}{12\ (Mr\ carbon)} \right.$$

$$+ \frac{\frac{0}{100}}{\frac{35}{100}} * 0 \left(\frac{kg\ gas}{kWh}\right) + \frac{\frac{0.4}{100}}{\frac{85}{100}} * 0 \left(\frac{kg\ gas}{kWh}\right) + \frac{\frac{0}{100}}{\frac{30}{100}} * 0 \left(\frac{kg\ gas}{kWh}\right) + \frac{\frac{7.5}{100}}{\frac{35}{100}}$$

$$\left. * 0.26676 \left(\frac{kg\ gas}{kWh}\right) + \frac{\frac{0}{100}}{\frac{35}{100}} * 0.36000 \left(\frac{kg\ gas}{kWh}\right) \right) = 344830246 \left(\frac{kg}{annum}\right)$$

$$CH_4 emissions \left(\frac{kg}{annum}\right)$$

$$= \sum_{i=1}^{N} \frac{Offsite\ electricity\ consumption\ EE \left(\frac{kWh}{annum}\right)}{Electricity\ generation\ efficiency_{\ i}} * \%\ Energy\ source_i$$

$$* \ IPCC\ CH_4\ emission\ factor_i \left(\frac{kg\ gas}{kWh}\right) = 6048 \left(\frac{kg}{annum}\right)$$

$$
\begin{aligned}
&{}_2O\ emissions\ \left(\frac{kg}{annum}\right)\\
&\quad = \sum_{i=1}^{N} \frac{Offsite\ electricity\ consumption\ EE\ \left(\frac{kWh}{annum}\right)}{Electricity\ generation\ efficiency_i} * \%\ Energy\ source_i\\
&\quad * \ IPCC\ N_2O\ emission\ factor_i\ \left(\frac{kg\ gas}{kWh}\right)\ = 3376\ \left(\frac{kg}{annum}\right)
\end{aligned}
$$

$$
\begin{aligned}
&NMVOC\ emissions\ \left(\frac{kg}{annum}\right)\\
&\quad = \sum_{i=1}^{N} \frac{Offsite\ electricity\ consumption\ EE\ \left(\frac{kWh}{annum}\right)}{Electricity\ generation\ efficiency_i} * \%\ Energy\ source_i\\
&\quad * \ IPCC\ NMVOC\ emission\ factor_i\ \left(\frac{kg\ gas}{kWh}\right)\ = 54339\left(\frac{kg}{annum}\right)
\end{aligned}
$$

$$
\begin{aligned}
&CO\ emissions\ \left(\frac{kg}{annum}\right)\\
&\quad = \sum_{i=1}^{N} \frac{Offsite\ electricity\ consumption\ EE\ \left(\frac{kWh}{annum}\right)}{Electricity\ generation\ efficiency_i} * \%\ Energy\ source_i\\
&\quad * \ IPCC\ CO\ emission\ factor_i\left(\frac{kg\ gas}{kWh}\right)\ = 374232\ \left(\frac{kg}{annum}\right)
\end{aligned}
$$

$$
\begin{aligned}
&SO_2\ emissions\ \left(\frac{kg}{annum}\right)\\
&\quad = \sum_{i=1}^{N} \frac{\frac{Offsite\ electricity\ consumption\ EE\ \left(\frac{kWh}{annum}\right)}{Electricity\ generation\ efficiency\ _i} * \%\ Energy\ source_i}{Calorific\ content\ of\ fuel_i\left(\frac{kWh}{kg\ fuel}\right)}\\
&\quad * \ \ SO_2\ emission\ factor_i\ \left(\frac{kg\ gas}{kg\ fuel}\right) = 625761\ \left(\frac{kg}{annum}\right)
\end{aligned}
$$

$$
\begin{aligned}
&NOx\ emissions\ \left(\frac{kg}{annum}\right)\\
&\quad = \sum_{i=1}^{N} \frac{\frac{Offsite\ electricity\ consumption\ EE\ \left(\frac{kWh}{annum}\right)}{Electricity\ generation\ efficiency\ _i} * \%\ Energy\ source_i}{Calorific\ content\ of\ fuel_i\left(\frac{kWh}{kg\ fuel}\right)}\\
&\quad * \ \ NOx\ emission\ factor_i\ \left(\frac{kg\ gas}{kg\ fuel}\right) = 371129\left(\frac{kg}{annum}\right)
\end{aligned}
$$

Classification and characterisation

Global warming

The burdens used in this study, that contribute towards global warming potential and their equivalence factors are:

CO_2 $\quad 1\frac{kg\ CO_2\ equivalents}{kg\ CO_2}$

CH_4 $\quad 21\frac{kg\ CO_2\ equivalents}{kg\ CH_4}$

N_2O $\quad 310\frac{kg\ CO_2\ equivalents}{kg\ N_2O}$

Acidification potential

The burdens used in this study that contribute towards acidification potential and their equivalence factors are:

SO_2 $\quad 1\frac{kg\ SO_2\ equivalents}{kg\ SO_2}$

NOx $\quad 0.7\ \frac{kg\ SO_2\ equivalents}{kg\ NOx}$

Calculation of global warming and acidification potentials

Global warming

General formula

$$Global\ warming\ potential\ of\ POX\ plant\ from\ fuels\ combustion\ \left(GWP_f\right) = \sum_{k=1}^{N} B_k E_{k,GWP}$$

$$Global\ warming\ potential\ of\ POX\ plant\ from\ electricity\ consumption\ (GWP_e)$$
$$= \sum_{E_{k,G}k=1}^{N} B_k\, E_{k,GWP}$$

Where B_k is an environmental burden (k) e.g. (N_2O in kg) and $E_{k,GWP}$ is the equivalency factor of that burden for global warming potential (GWP) (i.e. $310\frac{kg\ CO_2\ equivalents}{kg\ N_2O}$)

Example 1: Applied on the global warming potential of the POX plant (due to onsite fuel use).

$$\begin{aligned} GWP_f &= \left(4475815\ \frac{kg\ CO_2}{annum} * 1\frac{kg\ CO_2\ equivalents}{kg\ CO_2} + 103\frac{kg\ CH_4}{annum} * 21\frac{kg\ CO_2\ equivalents}{kg\ CH_4}\right. \\ &\quad \left. + 14.5\frac{kg\ N_2O}{annum} * 310\frac{kg\ CO_2\ equivalents}{kg\ N_2O}\right) \\ &= 4482473\ \left(\frac{kg\ CO_2 equivalents}{annum}\right) \end{aligned}$$

Example 2: Applied on the global warming potential of the POX plant (due to electricity generated).

$$\begin{aligned} GWP_e &= \left(344830246\ \frac{kg\ CO_2}{annum} * 1\frac{kg\ CO_2\ equivalents}{kg\ CO_2} + 6048\frac{kg\ CH_4}{annum} * 21\frac{kg\ CO_2\ equivalents}{kg\ CH_4}\right. \\ &\quad \left. + 3376\frac{kg\ N_2O}{annum} * 310\frac{kg\ CO_2\ equivalents}{kg\ N_2O}\right) \\ &= 346003814\ \left(\frac{kg\ CO_2 equivalents}{annum}\right) \end{aligned}$$

Total global warming due to energy use in POX plant (fuels and electricity)

$$\begin{aligned} GWP\ of\ POX\ plant &= GWP_f + GWP_e \\ &= 4482473\left(\frac{kg\ CO_2\ equivalents}{annum}\right) + 346003814\left(\frac{kg\ CO_2\ equivalents}{annum}\right) \\ &= 350486287\left(\frac{kg\ CO_2\ equivalents}{annum}\right) \end{aligned}$$

Acidification Potential

General formula

$$Acidification\ potential\ of\ POX\ plant\ from\ fuels\ combustion\ (AP_f) = \sum_{k=1}^{N} B_k E_{k,AP}$$

$$Acidification\ potential\ of\ POX\ plant\ from\ electricity\ consumption\ (AP_e) = \sum_{E_{k,G}k=1}^{N} B_k\ E_{k,AP}$$

Where B_k is an environmental burden (k) e.g. (SO_2 in kg) and $E_{k,AP}$ is the equivalency factor of that burden for acidification potential (AP) (i.e. $1\frac{kg\ SO_2\ equivalents}{kg\ SO_2}$)

Example 1: Applied on the acidification potential of the POX plant (due to onsite fuel use).

$$AP_f = \left(12.1\ \frac{kg\ SO_2}{annum} * 1\frac{kg\ SO_2\ equivalents}{kg\ SO_2} + 2248\frac{kg\ NOx}{annum} * 0.7\ \frac{kg\ SO_2\ equivalents}{kg\ NOx}\right)$$

$$= 1586\ \left(\frac{kg\ SO_2\ equivalents}{annum}\right)$$

Example 2: Applied on the acidification potential of the POX plant (due to electricity generated)

$$AP_f = \left(625761\ \frac{kg\ SO_2}{annum} * 1\ \frac{kg\ SO_2\ equivalents}{kg\ SO_2} + 371129\frac{kg\ NO_x}{annum} * 0.7\frac{kg\ SO_2\ equivalents}{kg\ NOx}\right)$$

$$= 885551\ \left(\frac{kg\ SO_2 equivalents}{annum}\right)$$

$$AP\ of\ POX\ plant = AP_f + AP_e$$

$$= 1586\ \left(\frac{kg\ SO_2 equivalents}{annum}\right)\ + 885551\left(\frac{kg\ SO_2 equivalents}{annum}\right)$$

$$= 887137\ \frac{kg\ SO2\ equivalents}{annum}$$

Concentrate contract terms modelled

Table 7.24: Terms of copper concentrate contract modelled.

Clause	***Clauses are a function of (Inventory model variables)***
Copper pricing clause	
Smelter accountability (%)	Cu metallurgical recovery ~ 96.65% for high grade concentrates
Acceptable concentrate cu (%)	Cu content > 30% - No deduction and 96.65% of the metals value[87] received before other deductions
Subject to a unit deduction of (%)	1% for low grade concentrates
Treatment and refining charges	
Treatment charge ($/DMT)[88]	Dry metric tonnes of concentrate treated
Refining charge ($/kg Cu)	kg of payable copper
Price participation	
Basis Treatment charge ($/tonne Cu)	Determined by market
Price participation amount ($/tonne Cu)	Function of the difference between the current price of copper and pre determined price. A ±% is used and this can be an advantage / disadvantage.
As	
Penalty($/DMT/% in excess)	Applied on % in excess of maximum allowed content
Max content allowed	Typically < 0.2% - No penalty
% above max content	% As in concentrate less Max As content allowed
Pb	
Penalty ($/DMT/% in excess)	% in excess of maximum allowed content
Max content allowed	Typically < 0.5% - No penalty
% above max content	% Pb in concentrate less Max Pb content allowed

---

[87] The London Metal Exchange price of copper was used (figure 13).

[88] Dry metric tonne, typically refers to concentrate.

Table 7 continued ...

Zn	
Penalty ($/DMT/% in excess)	% in excess of maximum allowed content
Max content allowed	Typically < 3% - No penalty
% above max content	% Zn in concentrate less Max Zn content allowed
Bi	
Penalty ($/DMT/% in excess)	% in excess of maximum allowed content
Max content allowed	Typically < N/A[89]% - No penalty
% above max content	% Bi in concentrate less Max Bi content allowed
Sb	
Penalty ($/DMT/% in excess)	% in excess of maximum allowed content
Max content allowed	Typically < N/A% - No penalty
% above max content	% Sb in concentrate less Max Sb content allowed
Moisture	
Penalty ($/DMT/% in excess)	Typically < 8% - No penalty
Max content allowed	% H_2O in concentrate less Max H_2O allowed
% above max content	% H_2O in concentrate less Max H_2O content allowed

---

[89] Not available

Excel logic was used to automatically detect the clauses in the copper contract and to implement the required action. To calculate the payment received by the mine for the copper in the concentrate before treatment and refining charges are deducted the following logic was used:

$$
\begin{aligned}
&Mine's\ Cu\ revenue\left(\frac{\$}{annum}\right) = If\ (\%Cu\ \ in\ dry\ concentrate \\
&\geq 30, Out; \frac{\%Smelter\ accountability}{100} * Cu\ in\ dry\ concentrate\ \left(\frac{tonnes\ Cu}{annum}\right) \\
&* LME\ Price\ of\ copper\ \left(\frac{\$}{tonne\ Cu}\right), Else\ out; \frac{\%Cu\ in\ dry\ concentrate - \%\ Unit\ deduction}{\%Cu\ in\ dry\ concentrate} \\
&* \frac{\%Cu\ in\ dry\ concentrate}{100} * dry\ concentrate\ \left(\frac{tonnes\ of\ dry\ concentrate}{annum}\right) \\
&* LME\ Price\ of\ Copper\left(\frac{\$}{tonne\ Cu}\right)
\end{aligned}
$$

- If the copper content in the concentrate is equal to, or greater than 30% no penalty is applied except for an allowance copper smelting companies are entitled to since they do not recover all the copper in the concentrate (i.e. metallurgical losses).

- If the copper content in the concentrate is less than 30%, 1% of the copper content is deducted from the actual copper content before payment is awarded to the mining company.

Treatment and refining charges were calculated as follows

$$
\begin{aligned}
&Treatment\ charges\left(\frac{\$}{annum}\right) \\
&\qquad = \ Concentrate\ \left(\frac{tonnes\ of\ dry\ concentrate}{annum}\right) * Treatment\ charges\ \left(\frac{\$}{DMT}\right)
\end{aligned}
$$

$$
\begin{aligned}
&Refining\ charges\left(\frac{\$}{annum}\right) \\
&\qquad = \ Copper\left(\frac{kg\ of\ payable\ copper}{annum}\right) \\
&\qquad * Treatment\ charges\ \left(\frac{\$}{kg\ of\ payable\ copper}\right)
\end{aligned}
$$

The penalty deductions were set up as follows:

$$\begin{aligned}
&\mathit{Penalty\ for\ excess\ element\ y}\left(\frac{\$}{\mathit{annum}}\right) = \mathit{If}\ (\%\ \mathit{Excess\ of\ element\ y}\\
&\qquad > 0, \mathit{Out}; \%\ \mathit{Excess\ of\ element\ y} * \mathit{Penalty}\left(\frac{\frac{\$}{\mathit{DMT}}}{\%\ \mathit{excess}}\right)\\
&\qquad * \mathit{concentrate}\left(\frac{\mathit{tonnes\ of\ dry\ concentrate}}{\mathit{annum}}\right), \mathit{Else\ out}, 0
\end{aligned}$$

This formula is used in the model to calculate the financial penalty for deleterious element (y) that could be in the concentrate in amounts that are in excess of the allowable content. Similarly for excess moisture, the formula below was used

$$\begin{aligned}
&\mathit{Penalty\ for\ excess\ moisture}\left(\frac{\$}{\mathit{annum}}\right) = \mathit{If}\ (\%\ \mathit{Excess\ moisture}\\
&\qquad > 0, \mathit{Out}; \%\ \mathit{Excess\ moisture} * \mathit{Penalty}\left(\frac{\frac{\$}{\mathit{DMT}}}{\%}\right)\\
&\qquad * \mathit{Concentrate}\left(\frac{\mathit{tonnes\ of\ dry\ concentrate}}{\mathit{annum}}\right), \mathit{Else\ out}, 0
\end{aligned}$$

The calculation for price participation was set up as follows:

$$\begin{aligned}
&\mathit{Price\ participation}\ \frac{\$}{\mathit{annum}} = \mathit{If\ LME\ price\ of\ copper}\left(\frac{\$}{\mathit{tonne\ Cu}}\right)\\
&< \mathit{Basis\ price}\left(\frac{\$}{\mathit{tonne\ Cu}}\right), \mathit{Out}; (\mathit{LME\ price\ of\ copper} - \mathit{Basis\ price})\left(\frac{\$}{\mathit{tonne\ Cu}}\right)\\
&* \frac{\%\mathit{Price\ participation\ amount}}{100}\\
&* \mathit{Payable\ copper\ in\ concentrate}\left(\frac{\mathit{tonnes\ Cu}}{\mathit{annum}}\right), \mathit{Else\ out}; (\mathit{LME\ price\ of\ copper}\\
&- \mathit{Basis\ price}) * \frac{\%\mathit{Price\ participation\ amount}}{100}\\
&* \mathit{Payable\ copper\ in\ concentrate}\left(\frac{\mathit{tonnes\ Cu}}{\mathit{annum}}\right)
\end{aligned}$$

This formula is used to calculate the increase / decrease in refining charges due to copper price changes from basis price.