QUANTIFYING, REDUCING
AND IMPROVING MINE WATER USE

by

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ABSTRACT

Water is vital to the mining industry; mines can require substantial amounts of water and are often located in some of the driest places on earth. Reducing water withdrawals and improving mine water use are key strategic requirements for moving toward a more sustainable mining industry. Mine water requirements often have significant technical, economic, environmental and political implications. This thesis quantifies global mine water withdrawals and discusses methods of improving mine water use by reducing water withdrawals and water-related energy consumption.

The thesis is composed of four main sections. First, two methods are proposed to calculate global mining water withdrawals by commodity. One method is based on the amount of water required to process a tonne of ore and the other is based on the amount of water required to produce a tonne of concentrate. A large database was created, compiling data regarding ore production, commodity production, commodity prices, and mine water withdrawals between 2006 and 2009. The study estimates that global water withdrawals range from 6 to 8 billion m^3 per annum. Second, the thesis presents a case study on the challenges faced and lessons learned during the design, start-up and modification of the water systems of a large copper mine site. Third, the thesis identifies multiple mine water reduction, reuse and recycle strategies that have been implemented around the world. A model is developed and used to show the potential impact of these strategies. The results of the modelling show how a hypothetical mine could reduce water withdrawals from 0.76 m^3/t to 0.20 m^3/t of ore processed or lower. In particular, the combination of ore pre-concentration and filtered tailings disposal reduced water consumption by over 74% of the base case. Finally, this thesis describes and demonstrates a method of determining the lowest energy option for a mine water network. The method uses a linear programming algorithm to compare options for matching water sources with consumers at mine sites. An example illustrates the method and shows how mine water system energy requirements can be reduced by over 50%.

PREFACE

Thus far the research undertaken for this dissertation has generated two journal publications and five refereed conference papers. In all cases, as the first author, I was primarily responsible for the research design, data collection and analysis, manuscript preparation and writing. Dr. Bern Klein, Dr. Marcello Veiga, Dr. Scott Dunbar, and Dr. Ward Wilson, as members of my supervisory committee, provided input, through advice and editing assistance. The publications are as follow:

Gunson, A. and K. Wood. 2008. Raw Water Supply: The Experience of Cerro Verde. In CMP 2008 Proceedings. Ottawa.

Gunson, A, Klein, B., and Veiga, M. 2008. Improving Mill Water System Design, In Proceedings WIM 2008, Santiago.

Gunson, A., Wood, K., Klein, B., and Veiga, M. 2008. Estimating Water Demand and Availability for Open Water-based Mill Cooling Systems, In Proceedings of XXIV IMPC, Beijing.

Gunson, A, Klein, B., and Veiga, M. 2010. Improving Mine/Mill Water Network Design by Reducing Water and Energy Requirements. In CMP 2010 Proceedings. Ottawa.

Gunson, A, Klein, B., and Veiga, M. 2010. Estimating Global Water Withdrawals due to Copper Mining. In Proceedings WIM 2010, Santiago.

Gunson, A. Klein, B., Veiga, M., Dunbar, S. 2010. Reducing mine water network energy requirements. Journal of Cleaner Production, Volume 18, Issue 13, September 2010, Pages 1328-1338.

Gunson, A. Klein, B., Veiga, M., Dunbar, S. 2012. Reducing mine water requirements. Journal of Cleaner Production, Volume 21, Issue 1, January 2012, Pages 71-81.

Additional assistance was provided by others as follows:

"Raw Water Supply: The Experience of Cerro Verde.", was reviewed and commented on by Ken Wood, my mentor at Fluor Canada. The paper was also reviewed, commented on and approved by John Marsdon, John Broderick, and Peter Faur, of Freeport McMoran Copper and Gold and Dave Dicaire, of Fluor Canada. In addition, Ian Orford, 1st Vice Chair of the Canadian Mineral Processing Conference, 2008, provided editing recommendations on the structure of the paper before final acceptance. The case study describes the design, pre-operations testing, start-up and upgrades to the Cerro Verde Primary Sulfide Project (CVPSP) water system. In addition to being the primary author, I worked as a process specialist with Fluor Canada and Fluor Daniel International on the CVPSP from November 2004 until March 2007, from the beginning of detailed engineering to the completion of the water system upgrade. I was the primary process designer for the CVPSP water systems, with responsibilities including:

- Creating and updating the site water balance, using data from the project mass (metallurgical) balance, existing site data and data from MWH Global on the proposed tailings storage facility;
- Compiling information about the available water quality based on SMCV environmental monitoring data;
- Compiling information from vendors about water requirements for mechanical and electrical equipment;
- Designing process flow sheets and piping and instrumentation diagrams for the plant water systems and providing process data for mechanical equipment datasheets;

- Writing the water systems process control philosophy and operating manuals;
- Leading the pre-operational testing and wet testing of the plant water systems; and
- Leading efforts to upgrade the fresh water system, including drafting evaluations of available options, completing detailed design of the process modifications, commissioning the modifications and training SMCV operators on how to operate the system.

"Improving Mill Water System Design" was reviewed and commented on by an anonymous reviewer for the Water in Mining Conference in 2008.

"Estimating Water Demand and Availability for Open Water-based Mill Cooling Systems" was reviewed, commented on, and approved by Ari Partanen of Freeport McMoran Copper and Gold. It was also reviewed and commented on by an anonymous reviewer for the International Mineral Processors Conference in 2008.

"Improving Mine/Mill Water Network Design by Reducing Water and Energy Requirements" was reviewed and commented on by Dominic Fragomeni, 1st Vice Chair of the Canadian Mineral Processing Conference, 2010.

"Estimating Global Water Withdrawals due to Copper Mining" was reviewed and commented on by an anonymous reviewer for the Water in Mining Conference in 2010.

"Reducing mine water network energy requirements" and "Reducing mine water requirements" were reviewed and commented on by anonymous peer reviewers and Gavin Hilson, Subject Editor, Journal of Cleaner Production.

Chapter 2, Literature Review, integrates the literature reviews undertaken during the research of all of the above papers in addition to further relevant publications.

Chapter 3, Methodology, integrates the methodology descriptions of all the above papers in addition to outlining the methodology used in Chapter 4, Estimating Global Mine Water Use.

Chapter 4, Estimating Global Mine Water Use, is partially based on concepts developed in “Estimating Global Water Withdrawals due to Copper Mining,” but is largely composed of material not previously published.

Chapter 5, Case Study – Cerro Verde Concentrator Fresh Water System, is primarily based on the papers, “Raw Water Supply: The Experience of Cerro Verde,” “Improving Mill Water System Design” and “Estimating Water Demand and Availability for Open Water-based Mill Cooling Systems.”

Chapter 6, Reducing Mine Water Requirements, is closely based on the paper “Reducing mine water requirements.”

Chapter 7, Mine Water Network Design, is closely based on the paper “Reducing mine water network energy requirements.” However, it also includes a previously unpublished section integrating and building on the results of the paper “Reducing mine water requirements.”

No ethics certificates were required or obtained for this dissertation.

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LIST OF ABBREVIATIONS AND SYMBOLS

a = number of mines reporting both X_{kc} and O_{kc}

aw = number of mines reporting X_{kc}, O_k, and W_k

A_1 = the flooded deposition area after 1 day of tailings discharge in a TSF, ha

A_{30} = the flooded deposition area after 30 days of discharge in a TSF, ha

$A_{flooded}$ = the flooded deposition area in a TSF, ha

A_{new} = the active deposition area contacting a dry beach in a TSF, ha

A_{POND} = a TSF pond area, ha

b_k = number of commodities produced at mine k

c = a commodity, such as a mineral or metal, produced at a mine, t/y

c_j = amount of water required by consumer j, m^3/h

C_p = slurry density reporting to the TSF in percent solids by mass, %

ΔH_c = the pressure loss across a clarifier, kPa

ΔHs_{ij} = the change in pressure in a system with a clarifier between C_j and S_i, kPa

ΔH_f = design allowable pressure drop across a filter, kPa

D_{RW} = average effective depth of rewetting on a TSF beach, metres

$e_{chiller}$ = chiller efficiency

e_f = the final void ratio of dry, consolidated tailings on inactive beaches

e_{ij} = total energy required for source i to provide water to consumer j, kWh/m^3

e_{motor} = pump motor efficiency

e_o = void ratio after completion of initial settlement in the TSF

e_{pump} = pump efficiency

e_{tower} = cooling tower fan efficiency

f_{pan} = a factor to correlate the PE to an open area (as opposed to a metal pan)

G_{ac} = the global weighted mean recovered ore grade of commodity c from the set of mines a, % mass

G_{awc} = the global weighted mean recovered ore grade of commodity c from the set of mines aw, % mass

G_S = the specific gravity of the tailings solids

GSW = gland seal water

G_{stc} = the global weighted mean recovered ore grade of commodity c, modified to determine the sensitivity of G_{tc} to additional production data, % mass

G_{tc} = the global weighted mean recovered ore grade of commodity c, % mass

g/t = grams per tonne

H_i = pressure available at s_i, kPa

H_j = pressure required by c_j, kPa

i = an individual water source on a mine site

i_p = hydraulic gradient at the recycle pond in a TSF

j = an individual water consumer on a mine site

k = an individual mine site producing one or more commodities

K_{POND} = vertical hydraulic conductivity of the tailings under the pond in a TSF, m/s

L = litre

L_{ENT} = TSF entrainment losses occurring during initial settlement, (m^3/d)

L_{EVAP} = evaporation losses on the active TSF beach, (m^3/d)

L_{POND} = TSF reclaim pond evaporation and seepage losses, (m^3/d)

L_{PONDe} = evaporation from a TSF pond, (m^3/d)

L_{PONDs} = seepage from the a TSF pond, (m^3/d)

L_{REW} = rewetting losses (seepage) on the active beach, (m^3/d)

m^3 = metres cubed

m = number of water sources

n = number of water consumers

O_k = the quantity of ore processed at mine k, t/y

O_{kc} = the quantity of ore processed at mine k producing commodity c, t/y

O_{tc} = total ore processed globally to produce commodity c, t ore/y

P_c = the average annual price of commodity c, US$/t

p_{ij} = pumping energy required for source i to provide water to consumer j, kWh/m^3

PE = pan evaporation rate, mm/day

Pow_{ij} = total power required for source i to provide water to consumer j, kW

q_{ij} = cooling energy required for source i to provide water to consumer j, kWh/m^3

Q_D = the total discharged process water in the tailings, (m^3/d)

RC_c = the mean revenue generated from water withdrawals using the concentrate method, US$/m^3

R/O = reverse osmosis water treatment

RO_c = the mean revenue generated from water withdrawals calculated using the ore production method, US$/m^3

s_i = amount of water available from source i, m^3/h

S_{dry} = the average degree of saturation of inactive tailings beach prior to re-wetting, %

SP_c = the sensitivity of W_{tc} to changes in P_c, %

S_{wmc} = the sensitivity of W_{tc} to changes in WM_{awc}, %

S_{gc} = the sensitivity of W_{tc} to changes in G_{ac}, %

S_{kG} = the percentage difference between G_{ac} and the ore grade of a hypothetical mine added to test the sensitivity of W_{tc}, %

$S_{k\%X}$ = the percentage of global production represented by a hypothetical mine added to test the sensitivity of W_{tc}, %

S_{kWM} = the percentage difference between WMO_{awc} and the amount of water require to process a tonne of ore at a hypothetical mine added to test the sensitivity of W_{tc}, %

SX/EW = Solvent extraction and electrowinning processing method

t = metric tonne

t_d = the number of days since discharge started at a given discharge point in a TSF, days

t_{ij} =treatment energy required for source i to provide water to consumer j, kWh/m^3

$t_{ij}(1)$ = energy required for filtration of source i to provide water to consumer j, kWh/m^3

$t_{ij}(2)$ = energy required for clarification of source i to provide water to consumer j, kWh/m^3

T_i = Temperature of the water available at source i, °C

T_j =Temperature of the water required at consumer j, °C

TSF = Tailings Storage Facility

VG_{tc} = the variability between the global weighted mean recovered ore grade of commodity c of the a dataset and the aw dataset, %

VX_{tc} = the variability between the global production of commodity c as reported by RMD versus the global production of commodity c as reported by the USGS and USDOE, %, where 100% indicates identical production estimates

W_e = Cooling Tower Water Evaporation, m^3/h

W_k = the total amount of water withdrawn at mine k, m^3/y

W_{kc} = the quantity of water withdrawn attributable to commodity c at mine k, m^3/y

W_{stc} = the modified value of total water withdrawals globally attributable to commodity *c*, m^3/y, based on a change to measure the sensitivity the estimate to different parameters

W_{tc} = total water withdrawals globally attributable to commodity *c*, m^3/y

WMC_{awc} = the global weighted mean quantity of water required per tonne of commodity *c* to produce commodity *c* from the set of mines *aw*, m^3/t commodity

WMC_{tc} = the global weighted mean quantity of water required per tonne of commodity *c* to produce commodity *c*, m^3/t commodity

WMO_{awc} = the global weighted mean quantity of water required per tonne of ore to produce commodity *c* from the set of mines *aw*, m^3/t ore

WMO_{stc} = the global weighted mean quantity of water required per tonne of ore to produce commodity *c*, modified to determine the sensitivity of WM_{tc} to additional production data, m^3/t ore

WMO_{tc} = the global weighted mean quantity of water required per tonne of ore to produce commodity *c*, m^3/t ore

x_{ij} = Quantity of water delivered from source *i* to consumer *j*, m^3/h

X_{atc} = total production of commodity *c* produced from the set of mines *a*, t/y

X_{awc} = total production of commodity *c* produced from the set of mines *aw*, t/y

X_{kc} = the quantity of commodity *c* produced at mine *k*, t/y

X_{kcw} = the quantity of commodity *c* produced at mine *k*, where mine *k* reports W_k, t/y

X_{tc} = total global production of commodity *c*, t/y

X_{tcRMD} = the global production of commodity c as reported by RMD in January 2011, t/y

%Solids = the percentage of solids by mass, %

$\%X_{ac}$ = percentage of X_{tc} of commodity c produced from the set of mines a, %

$\%X_{awc}$ = percentage of X_{tc} of commodity c produced from the set of mines aw, %

y = year

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First, I would like to thank my supervisors, Dr. Bern Klein and Dr. Marcello Veiga, both of whom have played critical roles in my life as mentors, supervisors, teachers, supporters, employers, and friends over the past 12 years. It is hard to exaggerate the pivotal role Dr. Klein and Dr. Veiga have played in my life.

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DEDICATION

This thesis is dedicated to Melania, Joffre, and Alec.

1. INTRODUCTION

1.1 Statement of the Problem

> *"Throughout the world, water is recognized as the most fundamental and indispensable of all natural resources and it is clear that neither social and economic development, nor environmental diversity, can be sustained without water. Today, virtually every country faces severe and growing challenges in their efforts to meet the rapidly escalating demand for water that is driven by burgeoning populations. Water supplies continue to dwindle because of resource depletion and pollution, whilst demand is rising fast because population growth is coupled with rapid industrialization, mechanization and urbanization. This situation is particularly acute in the more arid regions of the world where water scarcity, and associated increases in water pollution, limit social and economic development and are linked closely to the prevalence of poverty, hunger and disease"* (Ashton et al., 2001).

The global mining industry extracts and processes billions of tonnes of ore from the earth each year in order to produce a wide array of products that are the foundation of the global economy. Products such as coal, iron, and copper are the base of the industrial economy and products like potash and phosphate are critical to allow the agricultural industry to feed a growing population of over 7 billion people. The industry uses vast quantities of water to mine and process ores containing valuable products, often in some of the driest locations in the world, including Australia, northern Chile, southern Peru, central Asia, and the southwest United States. Water withdrawals at mine sites often have significant technical, economic, environmental and political implications.

Better water systems can reduce water and energy consumption as well as limiting or eliminating the amount of clean water required at mine sites. Improving water system design and practice are key strategic requirements in moving toward a more sustainable mining industry.

1.2 Research Objectives

The goal of this doctoral thesis was to develop a theoretical basis for better quantifying mine water use and improving mine water systems by reducing water withdrawals and water-related energy consumption.

The following questions were explored as part of this research:

- How can the amount of water withdrawn by the mining industry be better quantified?
- How can mine water use be reduced?
- How can mines better use the water they require to reduce mine energy requirements?
- How can mines take better advantage of opportunities to use wastewater or poorer quality water resources?
- What sustainability goals should mining companies pursue with respect to water use?

These questions were addressed by pursuing the following objectives:

- Undertake a thorough literature review of how the mining industry uses water and designs water systems;
- Identify methods of how to quantify global mine water withdrawals by commodity;
- Estimate global mine water withdrawals by commodity;
- Undertake a practical case study of how the mining industry uses water and designs water systems;

- Identify and describe the water system requirements of a typical mine;
- Identify options for how mines can better reduce, reuse and recycle water;
- Model the potential impact of different options to reduce, reuse and recycle water on mine site water use;
- Identify a methodology of how to better design mine and mineral processing plant water systems;
- Create a software tool to enable implementation of a methodology of how to better design mine and plant water systems;
- Model the potential impact of improved mine water system designs; and
- Identify suitable goals for mining companies to target with respect to sustainability and mine water use.

1.3 Thesis Outline

Chapter 2 summarizes the results of various literature reviews undertaken during this research. In particular, this chapter provides an overview of literature discussing the importance of water and mining, reviews previous efforts to quantify mine water use, reviews literature available on mine site water use and how mining operations have worked to better reduce, reuse and recycle water and, finally, reviews how other process industries have improved industrial water system design. In addition, key mine water system terms and concepts are defined.

Chapter 3 presents the methodologies used for estimating global mine water withdrawals, for the case study of Cerro Verde, for modeling mine water use reduction strategies and for developing a mine water network design method.

Chapter 4 details a cross-sectional study investigating global water withdrawals by the mining industry on an individual commodity basis from 2006 to 2009.

Chapter 5 presents a case study on the design, start-up and modification of the Cerro Verde Primary Sulfide Project. A reliable source of water of an acceptable quality is vital to the success of any mineral processing plant using wet methods. For Cerro Verde, a new 108,000 t/d copper mill in southern Peru, the main source of raw water was the Chili River (Río Chili). The Río Chili was heavily contaminated with raw sewage, industrial wastewater and agricultural runoff, and could also receive high sediment loads during the rainy season. The initial water treatment plant could not supply design flows of water due to the exceptionally poor water quality. Problems with treated water quality caused significant difficulties with water consuming equipment, such as the plant cooling systems. This chapter details the Cerro Verde water system and changes from design through to start up from a mineral process plant design perspective.

Chapter 6 examines past approaches used or suggested to improve mine water system performance by reducing mine water consumption and reusing or recycling water available on site. A mine water system model is developed and six scenarios are used to show the potential water and energy saving achievable using these approaches. This chapter demonstrates how to quantify the individual and combined potential impact from different water saving approaches.

Chapter 7 describes a method to estimate the energy required to supply water from available sources to specified consumers in a mineral processing plant. Linear programming is then used to minimize water system power requirements. The method, referred to as the Mine Water Network Design (MWND) method, is applied to an example to demonstrate the technique. Furthermore, the model is applied to the six water saving scenarios described in Chapter 6 in order to demonstrate the water and energy savings possible by combining the two approaches.

Chapter 8 presents an overall summary and conclusion of the thesis.

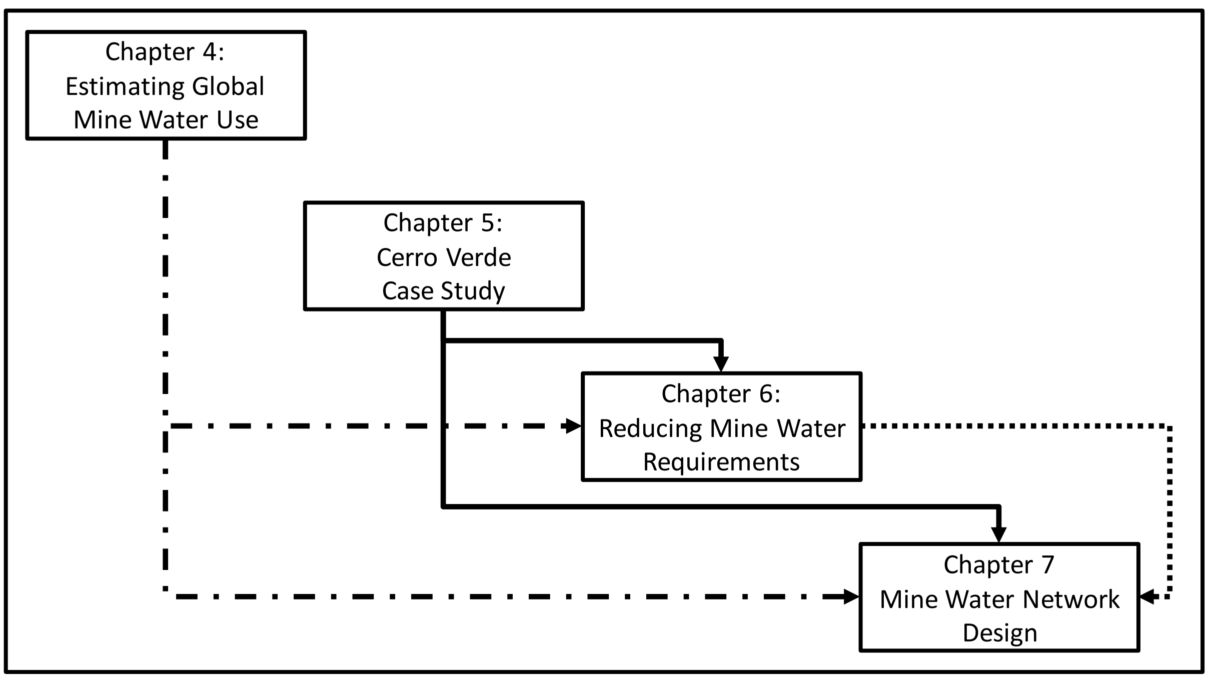
Chapter 9 presents recommendations arising from the results of the thesis.

Chapter 10 discusses claims of originality arising out of the work of this thesis.

Following Chapter 10 is the reference list for works citied within the body of the thesis, as well as an appendix providing further details of the mine water network design method.

The literature review in Chapter 2 defines and describes the context of the thesis and identifies the gaps that this research aims to fill. Chapter 3 describes the methodologies used in the research. Chapters 4, 5, 6 and 7 are the core research chapters, describing and reporting the results of this work. The links between the four research chapters are depicted in Figure 1.1.

Figure 1.1 Links Between Core Research Chapters



The results from the Chapter 4 water withdrawal estimate are used to ensure that the water withdrawals used in the Chapter 6 and Chapter 7 models are appropriate, as well as to highlight the overall importance of the research. The water sources and consumers' water quantity and water quality requirements described in the Chapter 5 case study form the basis of the models built in Chapters 6 and 7. Furthermore, the case study provides a description of the actual types of problems and resolutions that can arise during design, startup and operation of a mine and thus help to ensure that the solutions described in Chapters 6 and 7 remain practical. The models and scenarios built in Chapters 6 and 7 share a common basis. Section 7.3 brings together Chapters 6 and 7 by using the MWND method with the scenarios developed in Chapter 6 to show the water and energy reductions possible for each scenario.

2. LITERATURE REVIEW

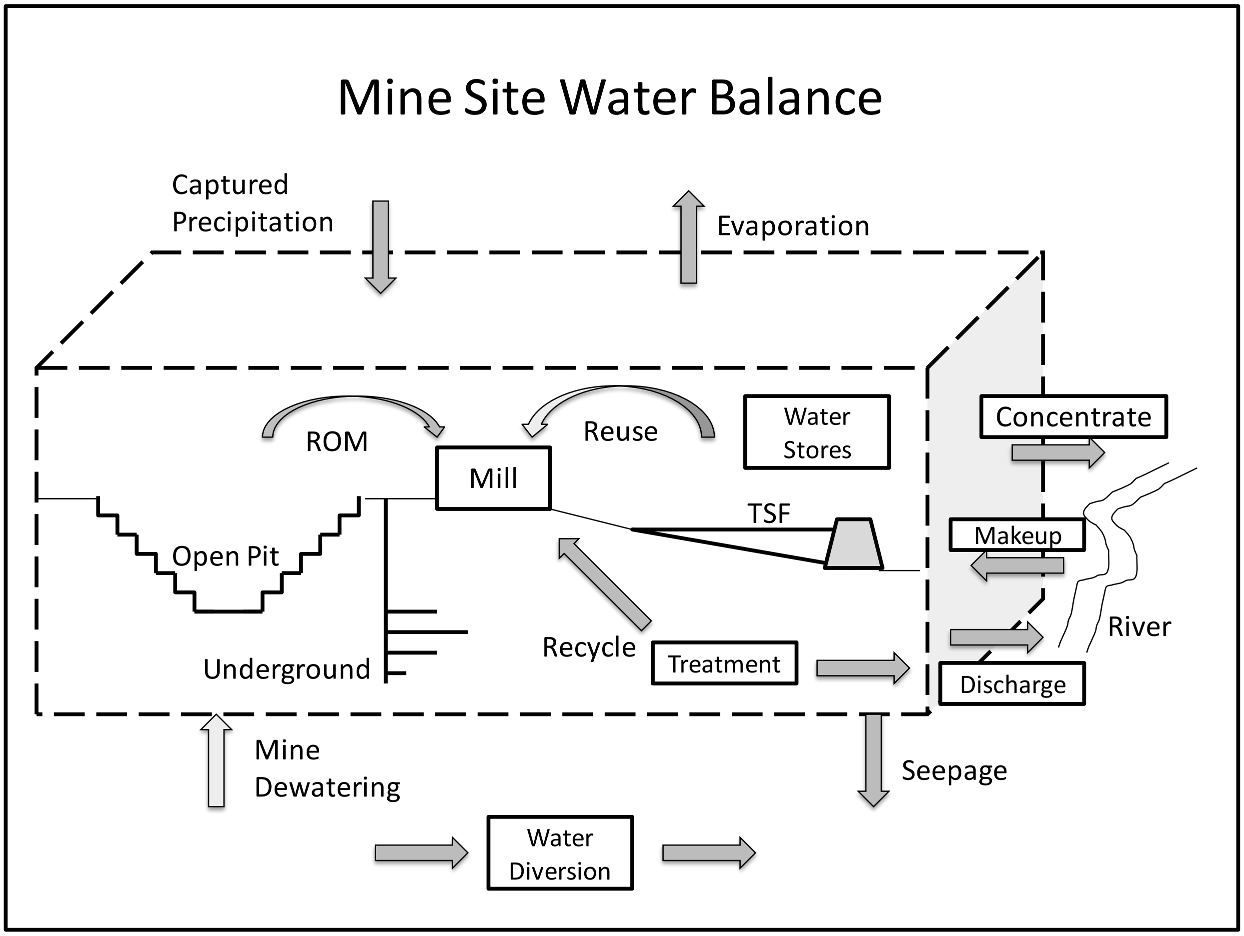
This literature review is divided into six main sections. The first provides an overview of key concepts and definitions regarding mine water use. The second gives an overview of literature discussing the importance of water and mining with respect to society, the environment and sustainable development. The third reviews previous efforts to describe how much water mines use. The forth discusses literature available on mine water use, with a subsection on water losses in tailings storage facilities (TSF). The fifth discusses literature available on how mining operations have worked to better reduce, reuse and recycle water and the final section discusses design approaches on how to improve mine water systems.

2.1 Mine Water Use: Key Concepts and Definitions

This section summarizes key concepts and definitions regarding mine water balances that are vital to this work and then discusses key relevant definitions from the literature. While the definitions generally match common industry practice, they are tailored specifically for this work.

Mine Site Water Balance or **Account**: A mine site water balance is an account of all water sources, sinks, and stores on a mine site. All mine sites fall into three general categories of water balances: positive, negative, and neutral. A positive water balance means that the site has an overall excess of water and therefore needs to discharge water to the surrounding environment. A negative water balance indicates the site has a water shortage and thus must withdraw water from the surrounding environment. A neutral water balance indicates that the site's sources and sinks are in balance. Truly neutral balances are unlikely, primarily due to seasonal weather changes, which result in sites having positive water balances at some times of the year and negative at others. Sites with positive water balances generally are required by law to treat any discharged water, whereas mines with negative balances may be able to operate as zero-discharge sites. Many mine sites both withdraw and discharge water, regardless of whether they have a positive or negative balance. Figure 2.1 depicts major elements of a mine site water balance.

Figure 2.1 Simplified Mine Water Use Sketch



Captured Precipitation: Captured precipitation refers to any rain and/or snow that needs to be accounted for in the mine site water balance. TSFs in particular may occupy a large surface area, and precipitation that is captured in a TSF can have a significant impact on mine water balances. Mine sites with negative water balances may choose to create diversions or catchment ponds to capture additional precipitation. Mines with positive water balances may need to treat and discharge any captured precipitation and may choose to instead divert precipitation away from the TSF or any other water stores. Precipitation varies widely with location and with season.

Concentrate: Concentrate is any valuable product produced by a mine or a mill that is transported off of a mine site. Water retained in concentrates is often minimal, as concentrates usually make up a small percentage of the total ore processed, and the concentrate is usually extensively dewatered prior to shipping to reduce transportation costs.

Evaporation: Water evaporates from all open water stores on the mine site, as well as wetted ore, concentrate or tailings. Water lost to evaporation within the plant itself is typically small, while evaporation in the TSF is much more significant.

External Water Sources, Fresh Water, Raw Water or **Makeup Water**: Makeup water consists of any water that is brought onto a mine site to supply site water consumers. Makeup water can come from sources such as municipal water supplies, treated waste water, ground water from wells, captured precipitation, rivers, lakes, or the ocean.

Mine: A mine is an excavation in the earth where an individual or an organization extracts ore. An open pit mine is an excavation on the surface, where as an underground mine is an underground excavation. It is common to refer to an entire mine site simply as a mine.

Mine Dewatering: Mine dewatering is the collection of ground water from an underground operation or around an open pit to prevent flooding or loss of ground stability. The collected mine water can be discharged and in some cases diverted, but is often used as a water source on the mine site.

Mine Site: A mine site is a physical location encompassing a mine, mineral processing plant, tailing storage facility (TSF), waste rock storage areas, and any other facilities required for the operation that are located close to the mine.

Mineral Processing Plant or **Mill**: A mill is a facility that converts ore into a saleable product. This could include crushing and screening plants, coal washing plants or a mineral processing concentrator. Mineral processing is formally defined as the process of economically converting mineral bearing raw material into individual mineral constituents; the minerals remaining essentially unaltered in physical and chemical form throughout. The temperature of the system normally is lower than the boiling point of water. For the purposes of this work, the term plants or mills will be used to describe facilities where run of mine (ROM) ore is processed, including crushing and screening plants and coal washing facilities. Most mills use wet processes, such as froth flotation, in order to separate valuable minerals from the waste minerals in the ore. Wet processes can use large quantities of water. Froth flotation, the most common separation process, typically takes place at 25-40% solids by mass (Wills, 2006). A flotation process would thus use 1.5 to 3.0 m^3 of water per tonne of ore processed.

Ore: Ore is defined as rocks which contain material with enough economic value to justify extracting at a mine. A rock is any naturally formed, solid material, composed of one or more minerals (and/or mineraloids, such as coal) and having some degree of chemical and mineralogical constancy. The amount of valuable commodity in an ore is referred to as **grade**, and is generally expressed as a percentage. For example, a copper ore could consist of 1% copper metal. Ore generally contains some moisture; even in a dry climate ore may contain 1 to 5% water by mass.

Reclaim Water: Reclaim water is a phrase that is used to refer to water recovered from a TSF to be recycled. Water recovered from a TSF has effectively gone through a water treatment process of settling combined with ultraviolet light from the sun. Thus, for the purposes of this work, reclaim water is considered a type of recycle water.

Recycle Water: Recycle water refers to water from one or more sources that is treated prior to reintroducing it to water consumers.

Reused Water: Water reuse refers to directly using water from one consumer in another consumer without treating the water in an intermediate step. An example would be to directly use spent cooling water as a water source to supply another water consumer.

Run of Mine (ROM) Ore: ROM Ore is ore as it comes from a mine: it has not been crushed, screened or otherwise processed through a mineral processing plant.

Seepage: Seepage consists of any water that seeps under or through the TSF or other water stores. Seepage water can be collected and recovered to the mine site, but the water is often lost representing an unrecoverable water sink.

Tailings: Tailings are any remaining waste products from the mineral processing plant after the concentrate have been removed. From a mass balance perspective, tailings equal ROM ore subtract concentrate. Tailings are generally stored on the mine site in a tailings storage facility.

Tailings Storage Facility (TSF): A TSF is a location on a mine site where tailings are stored and managed. Water entrained with the tailing in the TSF typically represents the largest water sink on a mine site, as discussed in detail in Section 2.4.2. TSFs typically also contain ponds where excess tailing water is recovered and recycled to the mill. Hence, TSFs often represent both the largest water sink and water store on a mine site.

Waste Rock: Waste rock is any rock extracted from the mine which does not have sufficient economic value to be sent off site as concentrate or sent to a mineral processing plant. Waste rock is typically placed in large storage facilities referred to as waste rock dumps.

Water Consumers, Tasks, or **Users**: A water consumer, task, or user is an operation or process on a mine site that requires water. In addition to flotation, water is used for various applications at a mine site including gravity separation processes, ore grinding dilution, screening spray water, dust scrubbing, wash water, cooling water, dust suppression, pump gland seal water (GSW), reagent mixing and dust suppression. A water consumer is not necessarily a water sink: water used in an operation or task is often available for reusing or recycling. The sum of all the water requirements at a mine site is the total mine water use; however, due to water recycling and reuse, the total amount of water a mine needs from external sources can be considerably less.

Water Consumption: Water consumption is equal to the sum of water withdrawals less any change in water stores and any discharge water. For a site practicing zero discharge, water consumption equals the sum of water sinks and any change in the volume of water stores. For a site discharging water, water consumption equals the sum of water sinks, water discharges, and any change in the volume of water stores.

Water Discharges: Water discharges are defined as any water that is not lost or stored that must be discharged in liquid form from the mine site. Evaporation is not considered a water discharge. Often discharged water must be treated in a water treatment facility to meet regulatory water quality requirements. Zero discharge refers to an operation that does not discharge any water off the mine site. For the purposes of this work seepage is considered a discharge, although this is a contentious issue in some regions. Therefore, any mine that does not capture seepage should not be considered as a zero discharge site.

Water Diversion: A mine site, particularly one with a positive water balance or strong seasonal variations in precipitation rates, may choose to divert water within or around a mine site. Typically a water diversion would consist of a ditch to capture and direct storm water to outside the mine site to avoid storage or treatment requirements or reinjecting dewatering water into a nearby aquifer. The water is managed and handled by the mining operation, but is not used by any water consumer on the mine site.

Water Sink: A water sink is an outflow of water that needs to be accounted for in a site water balance. Typical water sinks include site evaporation, water entrained in the TSF and concentrate, seepage and water discharged from the mine site. Water sinks are often described as water losses.

Water Source: A water source is an inflow of water that needs to be accounted for in a site water balance. The total amount of water lost at a mine site (the sum of the water sinks) must be replaced from water sources or by drawing down water stores. Typical water sources include captured precipitation, water recovered from mine dewatering, water moisture in the ore, and external makeup water sources. Any new water from a water source that has not been previously used by any water consumers is referred to as raw water.

Water Store: A water store is a reservoir on the mine site where water can be accumulated or discharged. The volume of water stored on a site is equal to the sum of the mine site water sources minus the sum of the water sinks over a given period of time. Typical water stores include the TSF and any water tanks or reservoirs located on a mine site.

Water Treatment: Water treatment refers to any process that removes contaminants from one or more water sources or discharges. Typical water treatment processes include straining and filtration, settling and clarification, flotation, activated carbon, anionic and cationic exchange, and reverse osmosis. Treated water can be discharged off the mine site and/or recycled back to water consumers within the mine site.

Water Withdrawals: Water withdrawal is the total amount of water taken from the surrounding environment onto a mine site. Water withdrawals include any external water sources, including surface water, ground water (including mine dewatering water), municipal water and captured precipitation. Diverted water is not included in water withdrawal calculations. It may not be obvious why precipitation and mine dewatering water are included in the calculation of mine water withdrawals, as they could be considered as internal sources of water. However, they are included in this work for the following reasons:

1. Precipitation not captured on a mine site would recharge local aquifers or otherwise join the local environment water cycle. Capturing precipitation removes water from the local environment much the same way that pumping water out of a nearby lake or river does.
2. Likewise, mine dewatering efforts can have a significant impact on lowering local aquifers, directly reducing the amount of water available for other consumers in the surrounding environment and community.
3. As described later in this section, it is standard practice by leading global reporting standards organizations and most major mining companies to include precipitation and ground water in their water withdrawal calculations.

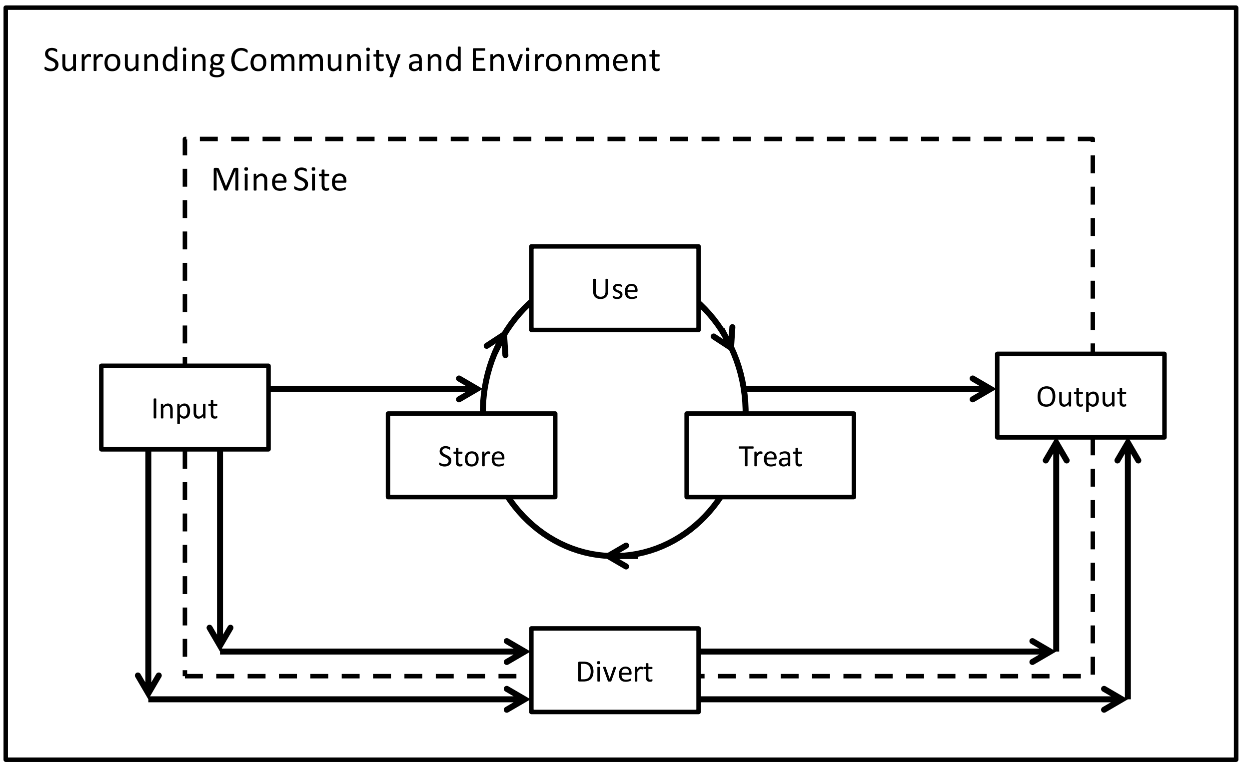
The above terms and definitions generally correspond to those used in industry and governmental standards and best practice guides published on mine water balances and use, such as those produced by the South African Department of Water Affairs and Forestry (SADWAF) (2006a; 2006b), the Australian Department of Resources, Energy and Tourism (ADRET) (2008), and the Global Reporting Initiative's (GRI) guidelines (2006).

The SADWAF approach to mine site water balances consisted of a straight forward mass balancing of water and salt contaminants. System boundaries were defined and all flows into or out of the system were balanced. A mine site water balance would consist of a series of smaller, more detailed systems aggregated together. The authors of the document recognized the importance of developing a standard format at a site to reduce complexity and confusion (Department of Water Affairs & Forestry, 2006a). The SADWAF publications define water reclamation as water treatment is defined above, and define reclaim water as recycle water as defined above. SADWAF uses the term recycling only to refer to the treatment of water from one consumer, with the treated effluent being returned to the same consumer. Water reuse is defined as above: effluent water from a process, reused without treatment (Department of Water Affairs & Forestry, 2006b).

The ADRET approach to mine water balances is to describe how site water balances should be defined to allow better comparison between sites. The approach grew out of efforts at the Centre for Water in the Minerals Industry (CWiMI) at the Sustainable Minerals Institute, University of Queensland, particularly with respect to the Bowen Basin coal operations (Cote et al., 2006). CWiMI developed a systems-based approach with the intent that it would not require the same level of detail as may be required at a scientific, engineering or operations level.

The ADRET approach, depicted in Figure 2.2, is composed of 6 main elements. Input refers to any water sources, output refers to any water sinks, store refers to any water stores, use refers to any water consumers, treat refers to any water treatment processes and divert refers to any water diversion.

Figure 2.2 Water System Map (Adapted from ADRET, 2008)



The ADRET approach was developed for the Australian Government's Leading Practice Sustainable Development Program. The working group included Australia's Department of Resources, Energy and Tourism, and the Department of the Environment and Heritage, and several major mining companies including Rio Tinto, BHPBilliton, and Newcrest, along with contributions from Xstrata (ADRET, 2008). The Minerals Council of Australia is actively advocating the adoption of the framework (Minerals Council of Australia, 2012). In addition, the framework is being evaluated by the International Council on Mining and Metals (ICMM), an international industry organization representing most of the world's major mining companies (2012). It can reasonably be considered to represent a best practice guide for defining how mine sites should report water usage.

The GRI guidelines regarding water use are also roughly consistent with the above definitions. GRI is a non-profit organization, founded in 1997, dedicated to developing and maintaining a comprehensive sustainability reporting framework across all industries (Global Reporting Initiative, 2012). Organizations can voluntarily choose to follow GRI's reporting framework by publishing a variety of their indicators ranging from economic to environmental and social factors.

The GRI requests that companies record water usage by the following indicators:

a. Quantify total water withdrawal by source (EN8);
b. Identify water sources significantly affected by withdrawal of water (EN9);
c. Quantify the percentage and total volume of water recycled and reused (EN10);
d. Identify the total water discharge by quality and destination (EN21); and
e. Indicate identity, size, protected status, and biodiversity value of water bodies and related habitats significantly affected by the reporting organization's discharges of water and runoff (Global Reporting Initiative, 2006).

Water withdrawals, as defined for indicator EN8, include groundwater, rainwater, waste water from third parties, and surface water, which include wetlands, rivers, lakes, and oceans, matching the ADRET definition.

The effectiveness of the GRI water indicators to describe a mine water system is limited. Mudd (2008) undertook a detailed evaluation of GRI water reporting, discussing the shortcomings and opportunities of the approach. Cote *et al*. (2009) also undertook a detailed review of several water accounting frameworks, including the GRI. The ADRET systems based approach was specifically developed to overcome these limitations, by including diversions, stores, and better defining how to describe water use within the mine site boundaries (ADRET, 2008). However, despite its shortcomings, the GRI indicators are still the most widely accepted protocol on of how to report mine water use.

2.2 Importance of Water in Mining

Mining often requires vast quantities of water. Chapter 4 details an estimate that the minerals industry alone (excluding coal, oil sands and aggregates), withdraws on the order of six to eight billion m^3 of water per annum. Mining also consumes enormous quantities of energy: the whole mining and metals sector consumes over 12% of the world's energy (United States Department of Energy, 2009). As explained in Section 2.3, while mine water withdrawals are large, they accounts for a small portion of overall world water withdrawals, even in mining-intensive water-poor countries such as Chile and Australia. However, when mining takes place in areas where water is scarce, mine water withdrawals can severely impact local supplies. In addition, acid rock drainage, leaks from tailings or waste rock impoundments, or direct disposal of tailings into waterways can seriously contaminate surface and groundwater (Akcil & Koldas, 2006; Cohen, 2006; Mining, Minerals and Sustainable Development, 2002; Nedved & Jansz, 2006). Mining's large demand for water and energy often has significant local and global economic, environmental and political implications.

Over the past two decades, civil society and governments have encouraged the global mining industry to move toward sustainable development, a move now largely accepted by the industry (International Council on Mining and Metals, 2003; Mining Association of Canada, 2004; Mining, Minerals and Sustainable Development, 2002; Natural Resources Canada, 1994). According to the Mining, Minerals and Sustainable Development Project, the goal of sustainable development should be to "maximize the contribution to the well-being of the current generation in a way that ensures an equitable distribution of its costs and benefits, without reducing the potential for further generations to meet their own needs." Meeting this goal requires decision makers to consider economic, social, environmental, and governance impacts (Mining, Minerals and Sustainable Development, 2002). Thus sustainable development encompasses a number of areas including corporate social responsibility, human rights, risk management, health and safety, biodiversity and the environment, recycling, community relations and transparency.

Much has been written about applying sustainable development industrial or business models to the mining industry to allow companies to address these areas (Basu & van Zyl, 2006; Damigos, 2006; Handelsman, 2009; Hilson & Murck, 2000; van Berkel, 2007a; van Berkel, 2007b). Kemp *et al.* (2010) focus specifically on the impact of mine water use on human rights.

Continual improvement of environmental performance and environmental management systems in mining is a key facet of the push toward sustainability (Driussi & Jansz, 2006a; Driussi & Jansz, 2006b; Hilson & Nayee, 2002; Hilson & Nayee, 2002; International Council on Mining and Metals, 2003; Mining Association of Canada, 2004; Mining, Minerals and Sustainable Development, 2002; Natural Resources Canada, 1994). Sustainable mining can have many definitions; however, a typical description would include the minimization of energy usage, materials consumption, environmental impacts and maximization of social satisfaction (Villas Bôas & Barreto, 1996). Hilson (2000) discussed the significant legislative, technical, and economic barriers the mining industry faces in implementing cleaner technology. Van Berkel (2007a) undertook a comprehensive review of efforts to integrate sustainability into the mining industry. He described five hierarchical levels of models for environmentally friendly sustainable business operations and categorized commonly known models such as Industrial

Ecology, The Natural Step, Green Engineering, Eco-Efficiency and Cleaner Production within these levels. A common theme in these models is the need for the mining industry to improve the efficiency of water and energy use.

Over the past decade several major mining companies have developed detailed strategies on how to address water concerns. For example, in 2005 Rio Tinto developed a water strategy aiming to improve water performance, account for the value of water and engage with others on sustainable water management (Rio Tinto, 2010). Gibson *et al.* (2003) described some of the work undertaken by Rio Tinto prior to adapting a corporate-wide water strategy. Anglo Platinum aimed to achieve zero discharge, protect water quality, not use potable water in operations, not compete with other sectors for water, and to meet their social obligations (Slatter et al., 2009). Teck adopted a water strategy in 2010, with a long term goal of keeping "clean water clean", minimizing water quality deterioration, restoring affected water resources, minimizing water use intensity, and participating in water use planning (Teck, 2012).

2.3 Global Mine Water Use

To better understand how to improve water use by the mining industry it is important to first develop an understanding of how the global mining industry uses water. Several efforts have been made to quantify and understand mine water use, with a focus on life cycle analysis and key indicators (Azapagic, 2004; Cote et al., 2009; Cote et al., 2010; Mudd, 2008; Mudd, 2009; Norgate & Lovel, 2004; Norgate & Haque, 2010; Northey et al., 2012; Suppen et al., 2006; Tejos & Proust, 2008; Worrall et al., 2009). This section discusses how mining companies report their water use as well as mine water extraction reports on a national level and academic publications estimating mine water use rates for different commodities.

Mining companies have reported on their water use in different forms for decades. Mining companies have released information on water use from at least as early 1971, when Island Copper on Vancouver Island began releasing annual environmental reports (Island Copper Mine, 1971). In the late 1980s, companies began publishing corporate environmental and sustainability reports. On Corporateregister.com, a website hosted by an organization dedicated to maintaining a database of corporate responsibility reports, the earliest report is a 1987 report by British Nuclear Fuels (Corporateregister.com, 2012). The first mining company annual report recorded is Falconbridge's 1992 "Falconbridge Limited report on the environment", which contained corporate water withdrawal and recycle rates (1994).

Mining company corporate reports have detailed mine water use through a variety of methods and parameters, as best suited the company. Mining companies and other organizations have often not been consistent with the definitions used when report water use. However, since 2000 and especially since the release of the G3 guidelines in 2006, many major international mining companies report their water use, among many other indicators, using the Global Reporting Initiative (GRI) guidelines, as described in Section 2.1 (Global Reporting Initiative, 2006). Several large diversified companies published GRI data for their organization as a whole, not by individual mines, making it difficult to extract any information on specific sites. It is effectively meaningless for a large multi-national company to report overall water withdrawal, especially as companies regularly acquire and/or sell mines within their portfolio of operations (Cote et al., 2009; Moran, 2006; Mudd, 2009).

A few papers and reports have discussed water extraction from mining on a national or industry basis. As shown in Table 2.1, Australia's mining industry withdrew 858 Mm^3 of water in 2008-2009, based on the Australian Bureau of Statistics Water Account, or less than 1.2% of overall water extraction (Pink, 2010). As shown in Table 2.2, Canada's mining industry withdrew 675

Mm^3 of water in 2009, accounting for less than 2% of industrial water extraction (Statcan, 2012). In mining-intensive and water-poor Chile, the mining industry in 2006 consumed an estimated 356 Mm^3, representing 5% of Chile's total water consumption (Tejos & Proust, 2008). Note, the Canadian and Australian statistical bureaus' definitions of water withdrawal do not match each other or the GRI's EN08 definition.

Table 2.1 Australian Water Withdrawals, 2008-2009

2008-2009 Category	**Water Withdrawals* Mm^3**	**%**
Agriculture	6,893	9.9
Aquaculture	445	0.6
Forestry & Logging	92	0.1
Fishing, Hunting & Trapping	2	0.0
Agriculture, forestry & fishing support services	61	0.1
Mining	858	1.2
Manufacturing	675	1.0
Electricity and gas	45,069	64.8
Water Supply	11,686	16.8
Waste collection, treatment and disposal services	7	0.0
Other Industries	1,264	1.8
Households	1,767	2.5
Environment	691	1.0
Total	**69,512**	**100.0**

*Includes self-extracted and distributed water withdrawals. (Pink, 2010)

Table 2.2 Canadian Industrial Water Withdrawals, 2009

2009 Category	**Water Withdrawals Mm^3**	**%**
Manufacturing	3,806	12.3
Thermal Electric Power Generation	26,346	85.3
Mining*	733	2.4
Total Industrial	**30,885**	**100**

*Includes mine dewatering. (Statcan, 2012)

Several publications have estimated average mine water use. Callow provided a list of 25 major mines operating in the 1920s and found water use (including recycle water) ranged from 0.6 to 30 m^3 water/tonne of ore (Callow, 1927). In 1932, Gaudin stated that flotation pulp density typically ranged from 18 to 35% solids, leading to a water consumption of 1.8 to 4.8 m^3 water per tonne of ore, and that incidental water consumption raised the consumption to 2 - 5 m^3 water per tonne of ore. He found the use of tailings dams, thickeners and concentrate filters allowed the reuse of water, which could have the effect of reducing water consumption to 0.75 - 1.0 m^3 water per tonne of ore (Gaudin, 1932). In 1939, he revised his estimate of mill water consumption to 0.07 to 1.5 m^3 water/tonne of ore (Gaudin, 1939). Yezzi (1985) estimated average plant water requirements to range from 0.4 to 20 m^3/t of ore processed. Brown (2003) reported that average mine water use ranged from 0.4 to 1.0 m^3/t of ore processed. Atmaca and Kuyumcu’s (2003) study of fifteen mineral processing plants found water consumption per tonne of product was found to range from 0.13 to 0.71 m^3/t for coal, 18.5 to 23.9 m^3/t for copper porphyry ores, 4 to 10 m^3/t of for chrome ores, 2 to 29 m^3/t for Cu-Pb-Zn ores and 0.4 to 5 m^3/t for potash and boron salts. Wills (2006) discussed some average process densities for grinding (~65% solids) and flotation (25-40% solids), and stated that mineral processing plants can require over 2.8 times as much water as ore. Generally these publications do not clearly define whether water consumption rates include only water withdrawals or if they account for changes in water stores and water discharges.

Norgate and Lovel (2004) estimated water consumption for several commodities in an effort undertake life cycle analysis of different minerals, as shown in Table 2.3. For example, they used a base feed grade of 3% copper for conventional flotation/smelting operations and 2% for leaching/solvent extraction and electrowinning (SX/EW) operations. Their data was based on a mine/mill consuming 0.37 m^3 water/t ore, representing the arithmetic mean of four copper mining sites (Ernest Henry, McArthur River, Valenzuela, and Kidd Mine). Leaching consumption was estimated at 27.4 m^3 water/t of cathode copper produced, but appears to be based on theoretical calculations, not actual mine usage. Based on their results, Norgate and Lovel (2004) estimated that the amount of water consumed was primarily a function of the initial ore grade and estimated the amount of water embodied per tonne of refined metal was equal to 167.7 multiplied by the grade of the ore to the power of -0.9039. Key limitations to this study are as follows: the mine ore grades and water consumption rates are not necessarily representative of the mining industry; the study double counts water use for ore with significant by-products; and the term water consumption is not clearly defined.

Table 2.3 Norgate and Lovel's Estimate of Water Consumption

Metal	Process	Ore Grade	Stage	Water	Consumption Units
Copper	Smelting/converting and electrorefining	3% Cu	Mine & mill	0.37	m^3/t ore
			Smelting	7.8	m^3/t Cu
			Refining	0.6	m^3/t Cu
	Heap acid leaching and SX/EW	2% Cu	Mine & heap	23	m^3/t Cu
			SX/EW	6.4	m^3/t Cu
Nickel	Flash furnace smelting and Sherritt-Gordon refining	2.3% Ni	Mine & mill	0.93	m^3/t Ore
			Smelting	0.81	m^3/t conc
			Refining	7.16	m^3/t matte
	Pressure acid leaching and SX/EW	1% Ni	Total†	3.4	m^3/t ore
Lead	Blast furnace	8.6% Zn, 5.5% Pb	Mine & mill	0.64	m^3/t ore
			Smelting	4.85	m^3/t Pb
			Refining	0.47	m^3/t Pb
	Imperial smelting process	8.6% Zn, 5.5% Pb	Mine & mill	0.64	m^3/t ore
			Smelting	12.73	m^3/t Pb
			Refining	0.47	m^3/t Pb
Zinc	Imperial smelting process	8.6% Zn, 5.5% Pb	Mine & mill	0.64	m^3/t ore
			Smelting	12.73	m^3/t Zn
			Refining	0.54	m^3/t Zn
	Electrolytic process	8.6% Zn, 5.5% Pb	Mine & mill	0.64	m^3/t
			Refining	12.33	m^3/t Zn
Aluminium	Bayer/Hall-Heroult processes	17.4% Al	Mine	0.03	m^3/t bauxite
			Refining	2.9	m^3/t alumina
			Smelting	1.5	m^3/t Al
Titanium	Becher/Kroll processes	9.8% Ti	Mine & mill	5.16	m^3/t ilmenite
			Becher	6	m^3/t S rutile
			Kroll	40	m^3/t Ti
Iron/steel	Blast furnace and basic oxygen furnace	64% Fe	Mine & mill	0.21	m^3/t ore
			Sintering	0.15	m^3/t sinter
			BF and BOF	1.94	m^3/t steel
Gold	CIL cyanidation and EW/smelt	3.6 g Au/t	Total†	0.74	m^3/t ore

† Stage by stage water useage data not available. (Norgate & Lovel, 2004)

Mudd (2007) undertook a comprehensive review of available data for gold mines and found annual water consumption (sometimes including recycled water) ranged from 0.72 to 2.87 m^3/t of ore processed over the period from 1991 to 2006, with an average use of 1.42 m^3/t. Alternatively, the amount of water consumed ranged from 224 to 1783 m^3/kg of Au produced over the period from 1991 to 2006, with an average consumption of 691 m^3/kg of Au produced (Mudd, 2007). The following year Mudd (2008) published a more comprehensive review of mining sustainability indicators, including water. He tried to only include mines reporting total water used (including recycled water). Mudd found that generally water consumption per tonne of ore processed was lower for large tonnage operations than smaller tonnage operations and attributed this relationship due to economies of scale. He also found that generally water consumption per tonne of product increased with decreasing ore grades. Table 2.4 details Mudd's results.

Table 2.4 Mudd's Estimate of Water Consumption, Including Recycled Water

	Sample Size	**Ore Processed (e.g. m^3/t ore)**		**Concentrate (e.g. m^3/t metal)**	
Commodity		**Average**	**SD**	**Average**	**SD**
Bauxite (m^3/t bauxite)	17	1.09	0.44	–	–
Black coal (m^3/t coal)	18	0.3	0.26	–	–
Copper (m^3/t ore; m^3/t Cu)	48	1.27	1.03	172	154
Copper–gold (m^3/t ore; m^3/t Cu)	42	1.22	0.49	116	114
Diamonds (m^3/t ore; m^3/carat)	11	1.32	0.32	0.477	0.17
Gold (m^3/t ore; m^3/kg Au) - total	311	1.96	5.03	716	1,417
Gold (m^3/t ore; m^3/kg Au) - outlier removed	306	1.372	1.755	609	1,136
Zinc ± lead ± silver ± copper ± gold (kL/t ore; kL/t Zn ± Pb ± Cu)	28	2.67	2.81	29.2	28.1
Nickel (sulfide) (m^3/t ore; m^3/t Ni)	33	1.01	0.26	107	87
Platinum group (m^3/t ore; m^3/kg PGM)	30	0.94	0.66	260	162
Uranium (m^3/t ore; m^3/t U3O8)	24	1.36	2.47	505	387

Modified from (Mudd, 2008)

Mudd's (2008) figures did not consider by-product production, the relative importance of the mine (a small mine would have equal weighting on the average water consumption as a large mine). The figures also do not differentiate between the types of processing methods used, such as between SX/EW and copper concentrate operations. Furthermore, Mudd's analysis did not differentiate between different mines meaning that the same mine reporting for several years could be included multiple times.

Northey *et al.* (2012) addressed some of these concerns in a later paper on copper life cycle assessment. They used the ISO 14044 Life Cycle Analysis standard to weight the proportion of copper produced at an operation relative to other products and identified different types of copper processing methods in their results. They found that as water scarcity at a site increases, the amount of energy required to supply water typically increases as well, due to increased pumping and/or treatment requirements. They calculated water withdrawals averaged 74 m^3/t Cu produced, but ranged up to 350 m^3/t Cu produced. They also found green house gas emissions and energy intensity per unit of copper produced generally increased with lower copper grades, but did not find as strong a correlation with water.

Menacho and Hurtado-Guzman (2004) estimated that SX-EW plants consumed 40.5 Mm^3 of water in Chile and 125.5 Mm^3 globally. They assume a mean copper ore grade of 0.85% Cu with recoveries of 78%, with a water consumption rate of 0.16 m^3/t of ore leached. Tejos and Proust (2008) provided an excellent dataset showing production and water consumption at several key copper mines in Chile. Water consumption averaged 0.79 m^3 water /t of ore for flotation concentrators, ranging from 0.3 to 2.1 m^3/t, and averaged 0.13 m^3 water/t of ore and ranged from 0.08 to 0.25 m^3/t for leach operations.

As demonstrated by the above examples, current literature show a large range between estimates of average mine water use and does not describe any systematic effort to estimate global mine water use rates.

2.4 Mine Site Water Systems

Where Section 2.2 described the literature available on the impact of mine water use in the overall context of sustainability and Section 2.3 described efforts to quantify how the mining industry uses water, this section describes how and why water is used on mine sites and the available literature on the topic. It primarily addresses conventional base-metal or precious-metal mines, which usually consist of an underground and/or open pit mine and a mineral processing plant. In addition, typically the mineral processing plant is where most water is required on a site and the TSF is where most water is lost, and these areas will be the primary focus of this section. However, most of the details apply to the mining industry as a whole.

2.4.1 Mine and Mineral Processing Plant Water Systems

There are relatively few prominent mining and mineral processing design texts that have been published throughout history. The earliest is Agricola's *De Re Metallica*, published in 1556. Agricola clearly felt water in mining was a key issue of concern and water features prominently in his work, where he discussed concerns with mine water pollution, the importance and potential cost of a secure water supply and the potential social impacts of water use. He also included numerous references to mine dewatering, water diversion, water vessels, pipes and launders, the use of water power and the use of water in ore processing and smelting (Agricola, 1556).

Taggart's seminal *Handbook of Ore Dressing* (1927) contains a section by John Callow (Callow, 1927) which identifies most of the basic concepts of mill water system design. Callow stated that wet mills require large volume of water and that "securing an adequate supply often becomes an important engineering problem." Access to water and water pumping costs were described as key factors in determining a suitable mill site location. He found water requirements depend on the processing technology use, with gravity methods using the most and leaching the least. His preference was for gravity water supply systems, and he discussed different water transport options. He stated that water recycling was most often practiced at locations where water was scarce, but mines could also recycle water to recover soluble flotation reagents or to minimize pumping requirements. Water conservation could be utilized to minimize water supply costs, to save water pumping costs, to avoid losses of valuable materials and to "escape prosecution for polluting streams." Callow described key water conservation technologies, including tailings ponds, settlers, filters and clarifiers. He described different potential sources of water and highlighted the importance of water quality to flotation chemistry, scaling and corrosion. He included a table of 25 operating mines, detailing the key processing methods used, the daily ore production and the water use per ton of ore, as well as indicating whether any water recovery was practiced. For example, in the 1920s the Chino Mill (still in operation in 2013) had a daily ore production of 12,700 tonnes per day and used 7.1 m^3 of water per tonne of ore, using tables, vanners and flotation. Chino recovered 48% of its water from thickeners and another 34% from its tailings pond, with a total water system power requirement of 1380 kW. Of the 25 mills he stated 13 practiced no water recycling. In addition, Callow provided further details of several mill water systems, including pipe sizes, pumping power requirements and cost and treatment technologies used. Taggart's *Elements of Ore Dressing* (1951) also contained a small section on the impact of water supply to a flotation plant, where he argued that water should be considered "as much as part of the raw feed as the ore" due to its impact on flotation. He highlighted the potential impact of slimes, dissolved salts and organic materials such as tannins, and believed that these impacts could be more important at mills that recycled water.

Gaudin's *Flotation* (1932) also discussed the importance of water supply in selecting a mill site to reduce water supply costs. Gaudin described the impact of water quality on flotation and the need to treat water prior to flotation in some cases. He stated the importance of organic impurities, such as tannic acid, glue, starch, and gelatin, and the requirement to treat water to remove organic contamination. He also raised the importance of controlling the amount of soluble minerals, such as carbonates and sulfates and as well as the impact of adjusting mill water pH. He highlighted the potential impact due to seasonal variations in water supply quality. Recycling mill water could reduce reagent requirements, such as frothers, but Gaudin argued that water recycling was generally more expensive than the potential savings to be gained unless it was urgent to save water. This statement is in disagreement with Taggart's description of the potential savings from water recycling. Gaudin also dismissed the environmental impact of water discharged from flotation plants. Like Taggart, Gaudin argued for the use of gravity based water systems where possible, highlighting the high cost of pumping water in a plant recycling water. Gaudin's *Principles of Mineral Dressing* (1939) also contained a short section on water storage, where he stated the critical importance of water if water availability was inadequate, alternatively abundant or scarce, or of poor quality. He found that water storage and purification could become a large cost item for mines.

Pryor's *Mineral Processing* (1965) included a small section on process water, stating that mill water is usually drawn from rivers and lakes. He discussed the impact of climate on surface water quality and its potential impact on flotation and states that process water may depress or activate minerals if over-charged with various ions. Pryor also discussed the importance of keeping oils and grease from the plant, such as from floor sumps, out of the process water.

The SME's *Mineral Processing Plant Design* (Mular & Bhappu, 1980) contained a chapter which discussed the importance of water quality and quantity on metallurgical testing (Coleman, 1978). The SME's *Mineral Processing Plant Design, Practice, and Control* (Mular et al., 2002) did not contain sections specifically addressing mine and mill water system design. However, the *SME Mineral Processing Handbook* contained a small but valuable section on water by Yezzi (1985). As with Agricola, Callow, and Gaudin, he stated that a dependable source of water was a key determinate for locating a mill. Yezzi argued that increasing governmental regulations controlling effluent quality would lead to more costly and complex water system designs. His section reviewed common water sources available to mines, and discussed how to develop site water balances and the importance of water quality and storage. Yezzi included a table outlining water use and storage data for 34 mills and summarized the water systems at five mines. He discussed how to initially design a plant water system by first considering a once-through system based on the process flow sheet and comparing process water requirements with the available water supply. If there was a shortage, the designer should take steps to decrease water losses and increase water reuse to match requirements with supply. Yezzi discussed the importance of water quality and that each process step could require a different water quality. He stated that if water required treatment prior to use, designing several separate systems to reduce treatment costs should be considered. The discharge of one system could be used by another requiring lower water quality. As an example, the Fontana steel plant had seven separate and distinct water systems. Yezzi also discussed the importance of water quality with respect to corrosion, fouling, and scaling and listed some common water treatment technologies. He suggested that if a plant had strict water quality requirements, it may be valuable to use a systems approach to the water system design. Yezzi argued that the need to reduce pollution made it "almost mandatory for process plants to introduce an effective water management program," and suggested that, depending on water availability, water systems could be designed to recover nearly all process water to reduce makeup water requirements to a minimum.

Fuerstenau and Han's *Principles of mineral processing* (2003) included a section on plant metallurgical balances and efficiency, but did not deal with water balances specifically. The chapter by Smith and Groudev discussed how mill water is generally re-circulated, but primarily from a water treatment perspective. They discussed how tailings ponds are designed to remove suspended solids and should be designed to provide sufficient surface area and retention time. They also briefly discussed thickening and other water removal technologies and water recycling within a plant.

Wills' *Mineral Processing Technology* (2006) discussed the important role of water in mineral processing operations, including its use as both a transportation and separation medium. He primarily introduced water in the context of metallurgical accounting. Wills briefly discussed the reuse of thickener overflow water to feed screen spray washes, and the practice of recycling tailings pond water to the plant, claiming it was mainly due to pressures from governments and environmentalists. He argued that as much water as possible should be recycled from the tailings pond for re-use in the mill and that the volume of makeup water used should be minimized. However, he provided no indication of how this could be achieved.

Turcotte's (1986) chapter in Mular and Anderson's *Design and Installation of Concentration and Dewatering Circuits* reviewed water use in the minerals industry in the USA and described the basics of mine water balances, the importance of testing recycled water in bench scale flotation tests, key water consumers in mineral processing plants and the importance of mine quality. He provided descriptions and figures for basic recycle water flow sheets and a comprehensive literature review of mine water recycling in the 1960s, 1970s and early 1980s.

Turcotte's (1986) key findings were: the importance of bench-scale test work to determine the quantity of both fresh and recycle water required; the importance of water quality in water recycling; his belief that the trend of increased water recycling will continue; his opinion that the importance of water chemistry in processing will continue to increase; and finally that recycled water must be viewed as both a "raw material" and a "reagent".

Turcotte drew on a set of earlier publications and reports which addressed water use in mills. In 1966, the United Nations held a seminar on ore concentration in water-short areas. Gaudin (1968) provided an introduction to dry and wet mineral processing and estimated the amount of water required per unit of ore for different processing options. He concluded his session by suggesting that processing with water is generally more technically advantageous than dry processing and that "under proper management, water-suspension processes, such as flotation and leaching, may be more advantageous than any dry processes, even in water-short areas." The UN seminar primarily focused on dry processing methods; however, two papers discussed the use of sea and saline water, including the impact of saline water on flotation and other separation methods and the importance of the material selection for plant equipment (Rey & Raffinot, 1968).

The UN seminar also had a detailed paper on methods for recovery of process water by Sutulov (1968). Sutulov discussed efforts in large copper porphyry mines in Chile to reduce water requirements during processing, to recover and treat process water both within the plant and from the TSF, and to conserve water where possible. Sutulov initially discussed the cost of water and the cost of lost production if adequate water was not available, highlighting the importance of carefully managing the available water supply at a mine. He then described how water is used in a plant and how hydrocyclones' improved efficiency allowed for lower water requirements than older rake and spiral classifiers, thus allowing for flotation at higher solids densities. Interestingly, he stated that flotation scavenger circuits can be as important for water recovery as they are for metallurgical improvements; the water in the low density scavenger concentrate is returned back to the process rather than being discharged to the TSF. He presented the key aspects of a site water balance and highlighted the importance of using treated sewage effluent from the mining camp as a source of water for the plant. The workforce in mines during the 1960s was higher than is common in current mining operations and camp water requirements could constitute up to 20% of the fresh water consumption at a mine. Sutulov discussed the potential negative impact of using recycle water in flotation, but stated that typically recycle water did not interfere with flotation and sometimes reduced regent requirements. However, he mentioned the negative impact of organic impurities on flotation and the use of digestion tanks and biofilters to remove organic contaminates. Sutulov highlighted the importance of large tailings thickeners for recycling process water and the key role of flocculants to increase water quality and reduce impurities. He identified the potential to use hydrocyclones to scalp coarse sands and reduce the feed to tailings thickeners, thereby increasing thickener capacity. He described the construction and operation of large tailings embankments built with coarse tailings sand, now common at large mining operations. He also discussed the use of an organic monolayer on water surfaces to reduce evaporation, and the difficulty in maintaining the layer in windy conditions. Sutulov also provided examples of typical water quality from different water sources.

Turcotte (1986) also referenced Pickett and Joe’s (1974) survey of water recycling in Canadian mills, which reported the following findings: water recycling in gravity and magnetic circuits and in flotation circuits targeting a bulk concentrate or one mineral appears straightforward and beneficial; water recycling in circuits with selective flotation or gold cyanidation circuits could have significant complications; and uranium processing plants tended to have complex and expensive, but effective, recycling systems. They found that cascade pond systems seemed to be effective to treat recycle water to an acceptable quality for reuse. An alternative possibility was to match recycle water with the part of the plant it came from, achieving the best possible combination of chemical properties between the water, the ore and the reagents. Like Yezzi (1985), this matching of water sources and consumers based on water quality hinted at the later development of water allocation planning in other industries, discussed further in Section 2.6. The importance of reducing water consumption in mills was discussed in the Italian context by Alfano *et al*. (1983). The paper argued that waste water should be recycled as much as possible to reduce water consumption and discussed treatment options for the recycled water. The primary benefits cited were the reduction of potential environmental pollution, flotation reagent savings, and the reduction in fresh water consumption. The article considered in detail the opportunities and problems that are associated with recycling process water in an oxide operation. Glumov *et al*. (1983) discussed the importance of thorough calculation of water balances and reuse of water, although the topic is not discussed in detail.

There appears to be a considerable gap in common mineral processing literature between Turcotte (1986) and the start of the commodities boom in the mid-2000s. Recent mining journal publications have not tended to discuss mine water systems and the few articles about water tend to focus on waste water treatment and tailings facilities. For example, SME's *Minerals & Metallurgical Processing* journal had only 16 articles with "water" in the title from its inception in 1984 to 2012, and none of them were strictly relevant to mine water system design. *Minerals Engineering* had 47 articles with the word "water" in their titles from its inception in 1988 to 2012, with five articles relevant to mine water systems, including an article on the impact of using recycle water on flotation (Rao & Finch, 1989), a method of improving water recovery from coal tailings (Duong et al., 2000), a method of improving water recovery from oil sand tailings (Madge et al., 2005), an article on the impact of particle size on water recovery (Mwale et al.,2005) and an article on the use of sea water at a small copper concentrator in Chile (Moreno et al., 2011). *Mine Water and the Environment* would be an obvious source of articles relating to mine water system design; however, it has been mostly dedicated to mine drainage and treatment concerns and contained only a handful of articles relevant to this thesis.

An exception is an article in *Erzmetall* by Atmaca and Kuyumcu (2003), which summarized a report on water consumption in mineral processing plants produced by Germany's Federal Institute for Geosciences and Natural Resources. Fifteen plants were examined, including coal, porphyry copper, complex copper-lead-zinc, chrome, potash and borate processing plants, with coal plants as the main focus of the paper. Atmaca and Kuyumcu (2003) described a systems-based approach to reducing mill water consumption. They categorized unit operations as elements which could be used to build up a hierarchy of subsystems to create an overall water system. The water quantity and quality required for each element was described, with the definition of water quality based on solids content. They divided plant water systems into three general systems: closed systems with tailings filtration, closed systems without tailings filtration, and open systems, with the first system requiring the least amount of water and the last the most. Atmaca and Kuyumcu argued that mines must transition to the first type of water system if they want to see a significant reduction in mine water consumption. They recognized that fully closed water systems can lead to pollutant buildups in the recirculating water streams. They stated that filtration of tailings may not cost more than deposition in a settling pond, but recognized that the tailings ponds may work to remove contaminants such as lime. They also mentioned the possibility of treating a bleed stream of 20% or more to improve the quality of recirculating water. They highlighted the need to reduce evaporation in arid locations and therefore advocated the use of deep cone tailings thickeners over traditional thickeners, to reduce surface evaporation.

A few general process journals have published articles specific to mine water system design. Stewart *et al.*(2003) published an interesting paper exploring a more systematic approach to process design in *Process Safety and Environmental Design*, looking at life cycle analysis and environmental impact assessment for mines. They discussed problem analysis with respect to mine design, or what process should be used to reach the best decisions regarding tradeoffs (economic, social, or environmental). Progress indicators were chosen, based on general principles such as minimizing the input of materials and energy and limiting the use of toxic, persistent and bio-accumulative materials. Stewart *et al.* (2003) recommended using these principles early on in the design process to "ensure that alternatives selected for more detailed development in subsequent design stages are consistent with a transition to more sustainable process." Multi-Criteria Decision Analysis (MCDA) was recommended to evaluate design options, usually by weighting the different options using both qualitative and quantitative measurements. Stewart *et al.* also discussed how mineral processing plants are fundamentally based on heuristics and are far more variable than chemical plants. Their methodology is described with a zinc smelter case study, which showed that including environmental concerns modified the optimum selection of technology. *Process Safety and Environmental Design* did not have many other articles related to mine water system design with the exception of a paper on filtering oil sand tailings (Xu et al., 2008) and a paper on integrating sustainable development into design (Petrie, 2007).

Since 2003, there have been a number of industry conferences dedicated specifically to water in mining, in particular the Australasian Institute of Mining and Metallurgy (AusIMM) Water in Mining conferences (2003, 2006, and 2009) and Gecamin's International Congresses on Water

Management in the Mining Industry (2008, 2010, and 2012). These conferences covered a large range of topics and are integrated throughout this literature review and thesis more generally. In addition, the Society for Mining, Metallurgy & Exploration (SME)'s held an International Symposium on Water in Mineral Processing in 2012. The proceedings were published as a book entitled *Water in Mineral Processing* (Drelich, 2012), which focused on the use of salt water, water treatment, water quality, and tailings management.

Other groups focused on mine water include the International Mine Water Association (IMWA) and the International Symposiums on Environmental Issues and Waste Management in Energy and Mineral Production (SWEMP). The IWMA, which was founded in 1979, has primarily focused on mine drainage and groundwater modeling and not on mine water design or supply. However, their conferences and journal have a handful of relevant articles integrated into the rest of this chapter. SWEMP proceedings have had a few papers relevant to mine water use. Shevtsov (1996) pointed out that treated mine waters should be used for water supply, feeding water-cooled systems and tailings reservoirs. He argued that thickeners and filters should be used in water treatment of mine waters. McQuade and Riley (1996) discussed water management systems in Australia, and argued that mining companies rarely considered water issues in the past, except where water was scarce or neighbours especially sensitive. They highlighted the need to prevent or at least minimize environmental impacts, and that a good water plan begins prior to development and extends after mine closure. They stated:

> *"Each mining operation is unique in the shape and extent of its resource, rate of development, hydrologic setting, climate and location with respect to its neighbours. A mine water management system is specifically designed for each mine and must meet the needs of water supply and disposal while conserving and protecting flora and fauna habitat,*

culturally significant sites, and protecting the quality and total resource of downstream water resources.(McQuade & Riley, 1996)*"*

They discussed the process for environmental impact investigation and risk assessment and suggested that due to ever more restrictive water release standards, companies are looking more closely at how they can minimize water pollution.

More recently, some government/organization-led work has been done on how to systematically design water systems specifically related to the mining industry. The best available efforts include the Australian Department of Resources, Energy and Tourism's *Leading Practice Sustainable Development Program for the Mining Industry* program (ADRET, 2008), and the South African Department of Water Affairs and Forestry's series of *Best Practice Guidelines for Water Resource Protection* (Department of Water Affairs & Forestry, 2006b). Other efforts include the IFC's *Environmental, Health and Safety Guidelines for Mining* (International Finance Corporation, 2007) and the World Resource Institute's summary on water-related risks associated with the mining industry (Miranda et al., 2010).

The Australian effort borrows heavily from the Centre for Water in the Minerals Industry at the Sustainable Minerals Institute, University of Queensland, which has developed a "hierarchical conceptual systems approach" to analyze mine water management (Cote et al., 2007), as discussed in Section 2.1. De Kretser *et al.* (2009) investigated holistic approaches to improving plant water efficiency, looking at options like ore pre-concentration and minimization of clay production.

The series of best practice guidelines published by the department of Water Affairs and Forestry, South Africa (2006b) represents a comprehensive effort towards improving the mining industry's use of water. While the guidelines are specific to South Africa, they can be applied throughout the global mining industry. The *H3 Water Reuse and Reclamation* guideline had several concrete suggestions on developing a water reuse and reclamation plan at a mine, largely pulled from other processing industries (Department of Water Affairs & Forestry, 2006b). The guide calls for mines to "optimally match uses with the required water, taking cognisance of economic and practical restraints with regard to having different water reticulation systems," and recommends water pinch analysis as a best practice method. This type of approach is discussed in more detail in Section 2.6.

Other countries with large and established mining industries have generally not matched Australia's and South Africa's efforts. Chile's COCHILCO has published a handful of relevant reports addressing mine water use, discussed elsewhere in this work. The United States Environmental Protection Agency Metal Mining Sector Notebook identified methods of minimizing mine waste, but did not talk about reducing energy consumption or water consumption (USEPA, 1995). The US government does not appear to have published recent relevant information on mine water use. Canada is moving toward a more targeted approach to addressing mine water use. In 2007, Canadian governmental, industrial and educational institutions launched the Canadian Mining Innovation Council (CMIC) and identified water as a key targeted area (CMIC, 2008). In 2011 the Processing group of CMIC listed "Cost Effective Closed Circuit Water Systems" as a research priority (Kondos, 2012), but as of late 2012 has not yet published any findings.

2.4.2 Tailings Storage Facility Water Systems

The largest water sink, or source of water losses, at most mine sites is the TSF (E. Brown, 2003; Wels & Robertson, 2003). Available literature on tailings facilities water losses is fairly extensive and detailed. This section does not aim to present a comprehensive review of the literature available on tailings facilities, but to provide an overview of the literature used in this thesis for creating simplified models of TSF water losses.

Wels and Robertson (2003) discussed how to create a simplified TSF water balance model using 16 parameters, described in detail in Section 2.4.3. Engels and Dixon (2007) describe in detail the relationship between tailings facilities and the water balance and discuss the possibilities of dewatering tailings material to reduce water consumption, minimize effluent treatment problems, and produce a stable waste deposit. Dewatered tailings disposal, or dry stack disposal, typically has a moisture content of less than 20%, achieved by using a combination of belt, drum, horizontal and vertical stacked pressure plates and vacuum filtration systems (Engels & Dixon, 2007).

Davies and Rice (2001) also discussed the use of dry stack filtered tailings, for use in arid areas, in situations where recovery is enhanced by tailings filtration (SX/EW), where there are high seismic conditions, or where temperatures are cold. Benefits include smaller footprints, and lower tailings liabilities. They cited the size distribution of the tailings as key, as fine material is filtered less effectively and they point out that stacked tailings must be designed to withstand any contact with water. Similarly, Fall *et al.* (2005) discussed the influence of tailings density and particle distribution on paste backfill.

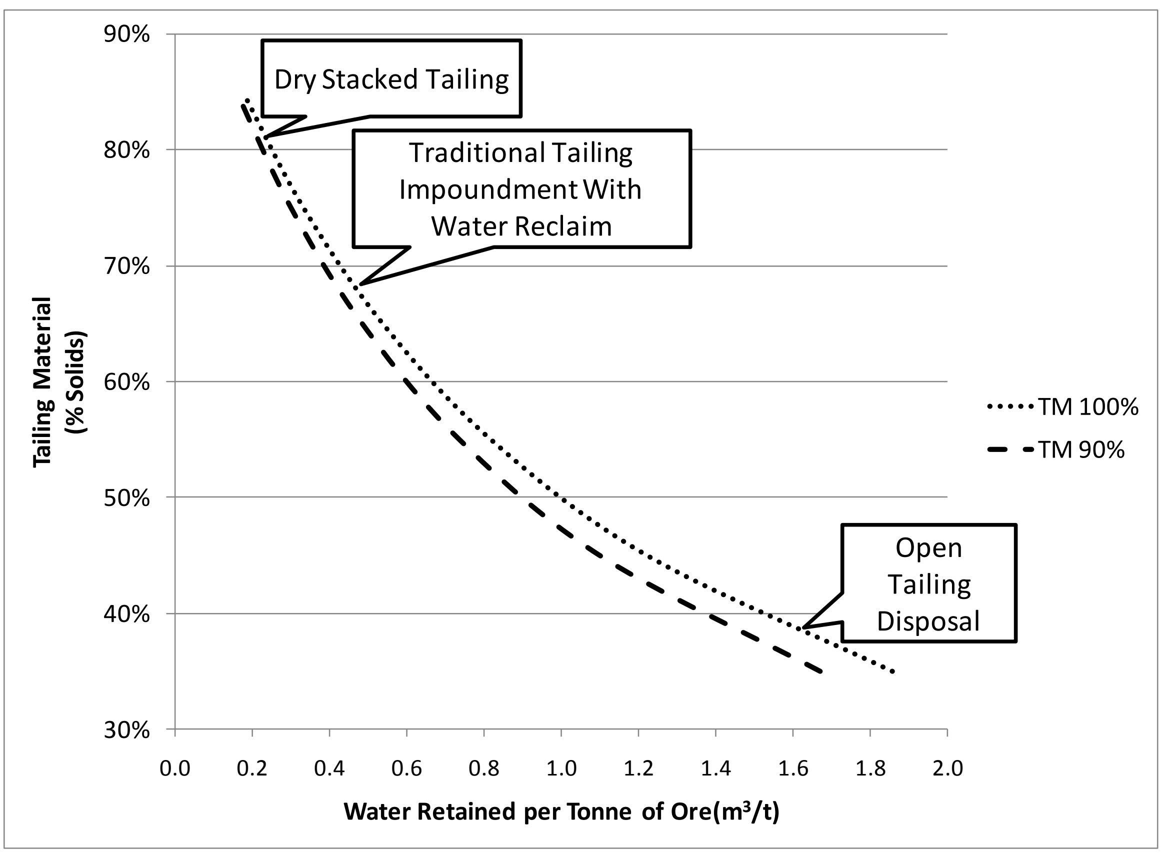
Figure 2.3 shows the impact of different tailing disposal methods on the amount of water retained in the tailing, based on the solid content by mass of the final tailings. The basic identity of percent solids by mass is shown in Equation 2.1.

$$\%Solids = \frac{Solids}{Solids + Water} \cdot 100 \qquad [2.1]$$

where %Solids = the percentage of solids by mass, [%]; Solids is the total mass of solids considered [t]; and Water is the total mass of water considered, [t, where 1 t water = 1 m^3].

Water lost can range from over 2 m^3/t for an open tailings systems to less than 0.25 m^3/t for dry stacked tailings. For most base or precious metal mines, the mass of concentrate recovered is small, usually consisting of less than 10% of the feed; thus, the amount of water lost per tonne of tailing is approximately the same as the amount lost per tonne of ore. However, for higher grade deposits, the amount of water retained in the tailings is reduced, as the total mass of tailings stored is a smaller portion of the plant feed. TM is the amount of solids reporting to the TSF as a percentage of the feed ore tonnage. Figure 2.3 indicates the amount of water lost per tonne of ore for a TM of 90% and one of 100%.

Figure 2.3 Water Retained in Tailings



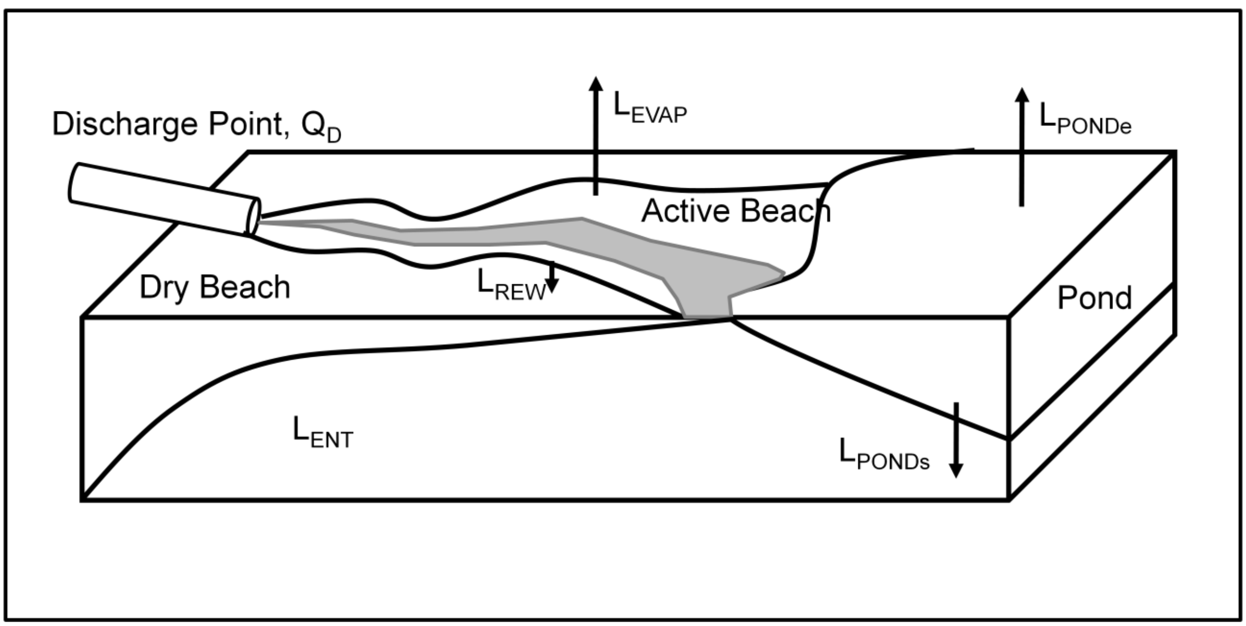
Mwale *et al.* (2005) presented an interesting study on the interaction between energy consumption, particle size and water recovery. They created a model incorporating particle size, energy requirements for grinding, and water recovery from thickeners, filters, and tailings impoundments. The conclusion was that finer grind size led to higher energy costs and lower water recovery. Mwale *et al.* argued that comparing grind size to water recoveries should be included in the plant design phase in order to maximize benefits.

2.4.3 Tailings Storage Facility Water Loss Model Review

In order to model a mine water system it is necessary to use a TSF water balance model. Chapter 6 uses a water model developed by Wels and Robertson (2003; 2004) to estimate water loss. The model was initially developed for the Collahuasi copper mine in Chile by Robertson GeoConsultants and AMEC, and was validated at Collahuasi and Chuquicamata, two large, low grade, open pit copper mines. The Wels and Robertson model is used because it was developed for large copper porphyry mines, the main focus of this thesis. Furthermore, the model is relatively straightforward and concentrates on the amount of water available to return to the plant, which is the most relevant aspect when targeting reduced raw water losses. Typically TSF water balances try to account for all water sources and sinks and are more complex than required for the purposes of this paper (Truby et al., 2011). The Wels and Robertson model is reviewed and explained in detail below.

Figure 2.4 presents the basic components of the Wels and Robertson Model, modified from Wels and Robertson (2003). A typical TSF consists of an impoundment with several tailings discharge points located around the perimeter. Water is primarily lost through evaporation and seepage from the tailings as they are deposited on the beach of the TSF, through evaporation and seepage from the pond that forms from run-off water as the tailings are deposited, and through water entrainment in the pore spaces of the tailings. Water collected in the pond can be pumped back to the plant and reused or recycled. Not all discharge points are active at the same time - operators will switch discharge points every few weeks or months to keep the tailings evenly deposited across the impoundment. When tailings are deposited from the discharge point, the tailings form a fan-shaped wetted area which grows over time until the discharge point becomes inactive. The Wels and Robertson Model focuses on water losses from the active discharge points; while water retained in tailings at inactive discharge fans continues to evaporate or seep into the ground after deposition, this water is assumed to be non-recoverable from a practical standpoint.

Figure 2.4 Tailings Storage Facility Water Balance



The basic concept of the Wels and Robertson Model is that the water available for recovery to the mine equals the total discharged water in the tailings minus the total water losses in the TSF.

This can be expressed as shown in Equation 2.2:

$$Return\ Water = Q_D - (L_{ENT} + L_{EVAP} + L_{REW} + L_{POND}) \qquad [2.2]$$

where *Return Water* = the amount of water available to return to the plant, [m^3/d]; Q_D = the total amount of water discharged from the plant to the TSF, [m^3/d]; L_{ENT} = entrainment losses occurring during initial settlement, [m^3/d]; L_{EVAP} = evaporation losses on the active beach, [m^3/d]; L_{REW} = rewetting losses (seepage) on the active beach, [m^3/d]; and L_{POND} = reclaim pond evaporation and seepage losses, [m^3/d]. Wels and Robertson (2004) found repeated rewetting losses to be negligible at the smaller Collahuasi mine and modified their model to only account for the initial rewetting losses. The modified model is used in this research, as Collahuasi is closer in scale to the examples used in Chapters 6 and 7.

Q_D is given by:

$$Q_D = Tailings \times \left(\frac{1}{C_p} - 1\right) \qquad [2.3]$$

where Q_D = the total discharged process water in the tailings, [m^3/d]; *Tailings* is the total mass of tailings discharged per day, [tpd]; and C_p is the slurry density in percent solids by mass, [%].

L_{ENT} is given by:

$$L_{ENT} = e_o \times \frac{Tailings}{G_S} \qquad [2.4]$$

where L_{ENT} = entrainment losses occurring during initial settlement, [m^3/d]; e_o is the void ratio after completion of initial settlement; *Tailings* is the total mass of tailings discharged per day, [tpd]; and G_S is the specific gravity of the tailings solids.

L_{EVAP} is given by:

$$L_{EVAP} = PE \times f_{PAN} \times A_{flooded} \qquad [2.5]$$

where L_{EVAP} = evaporation losses on the active beach, [m^3/d]; PE is the pan evaporation rate, [mm/day]; f_{pan} = a factor to correlate the PE to an open area (as opposed to a metal pan); and $A_{flooded}$ = the flooded deposition area [ha].

L_{REW} consists of the initial rewetting losses which occur when tailings are first placed on a dry beach. L_{REW} is given by:

$$L_{REW} = D_{RW} \times (1 - S_{dry}) \times \frac{e_f}{1 + e_f} \times A_{new} \qquad [2.6]$$

where L_{REW} = rewetting losses (seepage) on the active beach, [m^3/d]; D_{RW} is the average effective depth of rewetting, [m]; S_{dry} = the average degree of saturation of inactive tailings beach prior to re-wetting, [%]; e_f = the final void ratio of dry, consolidated tailings on inactive beaches; and A_{new} is the active deposition area contacting a dry beach, [ha].

L_{POND} is also composed of two factors: L_{PONDe}, the evaporation from the pond, and L_{PONDs}, the seepage from the pond.

L_{PONDe} is given by:

$$L_{PONDe} = PE \times f_{PAN} \times A_{POND} \qquad [2.7]$$

where L_{PONDe} = evaporation from a TSF pond, [m^3/d]; PE = pan evaporation rate, [mm/day]; f_{pan} = a factor to correlate the PE to an open area; and A_{POND} is the pond area [ha].

L_{PONDs} is given by:

$$L_{PONDs} = K_{POND} \times i_p \times A_{POND} \qquad [2.8]$$

where L_{PONDs} = seepage from the a TSF pond, [m^3/d]; K_{POND} is the vertical hydraulic conductivity of the tailings under the pond, [m/s]; i_p = hydraulic gradient at the recycle pond; and A_{POND} is the pond area [ha]. A_{POND} is assumed to be the same for the calculations of both L_{PONDe} and L_{PONDs}.

$A_{flooded}$ and A_{new} are highly dependent on the deposition method and geography of the TSF. Wels and Robertson (2004) estimated the flooded deposition area for Collahuasi as:

$$A_{flooded} = \frac{(A_{30} - A_1)}{\ln(30)} \times \ln(t_d) + A_1 \qquad [2.9]$$

where $A_{flooded}$ = the flooded deposition area in a TSF, [ha]; A_1 is the flooded deposition area after 1 day of discharge, [ha]; A_{30} is the flooded deposition area after 30 days of discharge, [ha]; and t_d is the number of days since discharge started at a given discharge point. A_{new} for any given day i thus equals $A_i - A_{(i-1)}$.

The modified model, which is used in Chapter 6, only accounts for the initial rewetting losses, not the repeated rewetting losses that Wels and Robertson found to be negligible at the similarly-sized Collahuasi mine.

2.5 Reducing Mine Water Requirements

This section discusses available literature about efforts to reduce, reuse and recycle water at mine sites. Data from the literature cited in this section is used in Chapter 6 to model the effect of efforts to reduce mine water use.

Past efforts to improve mine water system performance by reducing mine water consumption and reusing or recycling water available on site can be categorized into the three components of the decades-old waste management hierarchy: reduce, reuse, and recycle. This section details the literature on efforts at mine sites to implement these concepts. In the context of this thesis, “reduce” means the effort to reduce the overall amount of water required at a site, including both fresh and process water. “Reuse” means the effort to directly use water from one consumer in another without treating the water in an intermediate step. “Recycle” means treating water to some extent prior to reintroducing it to consumers.

This thesis does not focus on water treatment system design. However, there are a number of good references for water treatment systems, including Degrémont’s Water Treatment Handbook (1991) and Domestic Wastewater Treatment in Developing Countries (Mara, 2003).

The first step toward improving or designing mine water systems is to develop a good understanding of the mine’s existing system. No effort should be undertaken to improve a mine water system until a reasonably accurate site water balance has been completed. Putting effective water metering technologies or methods in place and having an accurate site water model is critical (Department of Water Affairs & Forestry, 2006a; Department of Water Affairs & Forestry, 2006b; International Council on Mining and Metals, 2009; Mayer et al., 2008; Newmont, 2008). Mining companies should consider implementing a comprehensive water management strategy. Any water management plan should ensure that existing facilities are well run and maintained. Basic steps such as fixing leaky pipes and valves, replacing undersized or worn-out pumps, and improving thickener or clarifier operation can lead to inexpensive and impressive improvements (Chambers et al., 2003; Stegink et al., 2003; Thompson & Minns, 2003).

2.5.1 Efforts to Reduce Mine Water Use

The current high rate of water recycling at most current mining operations means water consumption usually depends not on the amount of water required for individual unit operations, but the amount of water lost to permanent water sinks, such as evaporation, seepage or retention in the concentrate or tailings material. Efforts to reduce mine water use include:

- Reducing wet area/open area in the TSF
- Reducing fines generation during grinding to lower tailings water retention
- Improving tailings thickener performance
- Reducing water losses through thickened tailings or paste tailings disposal
- Reducing water losses through the installation of drains in the TSF
- Reducing water losses through tailings compaction
- Reducing water losses through selective tailings size classification
- Reducing water losses by using tailings filtration
- Reducing the concentrate moisture content
- Reducing evaporation through covers (tanks/thickeners/tailings pond)
- Reducing evaporation through alternative dust suppressants
- Ensuring no unplanned pipeline water losses
- Eliminating evaporative cooling systems
- Reducing pump GSW usage
- Reducing site employee/contractor water use
- Reducing water consumption through ore pre-concentration
- Reducing water use through dry processing

The largest water sink at most mine sites is the TSF; water is lost to evaporation or seepage, or is entrained within the tailings (E. Brown, 2003; Wels & Robertson, 2003). Significant water savings can be achieved by reducing the available wet and/or open water area, which can be done by carefully managing the placement of the tailings (Chambers et al., 2003; Soto, 2008).

Water entrained within the tailings can be reduced by a number of methods, including lowering the amount of fines (clay-sized particles) generated during processing (de Kretser et al., 2009; Mwale et al., 2005). Many mines have tailings thickeners to recycle water and reagents – increasing the solids density of the thickener underflow can reduce the amount of water sent to the TSF, thus potentially reducing the amount of water lost to evaporation, retention or seepage (Chambers et al., 2003; International Council on Mining and Metals, 2009; Mayer et al., 2008; Soto, 2008). Furthermore, increasing the solids density of the thickener underflow reduces the amount of water pumped back from the TSF to the plant, and hence the amount of energy consumed by the reclaim water system. Some mines, such as Barrick's Bulyanhulu Mine in Tanzania and Breakwater's Myra Falls Mine in British Columbia, have installed high density thickeners to dispose of tailings as a low-water paste (Engels & Dixon-Hardy, 2009).

In cases with high water costs or the need for tailings storage stability, filtered tailings or dry stack tailings storage has been successfully implemented (E. Brown, 2003; Davies, 2004). Davies reported that 55 metal or industrial mineral operations used dry stack tailings storage by 2004. Reducing the water content of the concentrate by improving filter performance also lowers water losses, and reduces concentrate shipping costs (Soto, 2008). Davies and Rice (2001) stated that filtered tailings can save up to 75% of the water required for a conventional tailings storage facility. Davies and Rice claimed that dry stack is limited to operations under 15,000 t/d, and is most attractive for mines under 2,000 t/d. Costs are higher than traditional facilities, around 1 to 10$/tonne, averaging 1.5 to 3$/tonne, including filtration, transportation, placement, and compaction. Dry cake can be as dry as 70 to 85% solids (Davies & Rice, 2001).

Barrera and Ortiz (2010) described a number of other options to reduce water losses in tailings, including installing drains underneath the TSF and compacting tailings through co-disposal with waste rock or even explosives. They also highlighted the effect of size classification on reducing water entrainment in tailings. Hydro-cyclones are often used on mine sites to separate the coarse fraction of the tailings for use in impoundment construction. As water in the coarse sand drains quickly, the overall amount of water retained in the tailings may be lower than the amount in the coarse and fine tailings deposited together. To further reduce evaporation at tailings facilities, some mines have placed floating covers on open water in tailings ponds, in addition to covering water storage tanks and thickeners (Mayer et al., 2008; Slatter et al., 2009; Soto, 2008). Alternatives such as floating plastic balls may work as well (AWTT, 2010; Slatter et al., 2009).

Typically, mining operations require a significant focus on handling dust, both on mine roads and in the plant. Mines generally have a number of water trucks which spray the haul roads and other transportation routes to minimize dust generation. Some mines have experimented with additives or alternative dust suppressants to reduce water consumption (General Electric, 2010; Mayer et al., 2008; McIntosh & Cronin, 2003; Soto, 2008; Xstrata, 2007). There are several dust suppressant alternatives, including salts, surfactants, soil cements, bitumens, and films (Organiscak et al., 2003). GE (2006; 2010) reported dust suppression water savings of 67% to 90% by using organic binders to harden road surfaces. For crushing, screening and conveying dust suppression, there are a number of common and widely available alternatives to traditional water sprays. These include fogging systems and foam, which can significantly reduce water consumption (Kissell, 2003).

Many operations have long pipelines delivering water to the mine site. These pipelines may have significant water losses due to leaks. In addition, impoverished communities along pipeline corridors have been known to extract significant quantities of water. Where possible, pipelines should be repaired to reduce leaks. In areas where communities are using the water, a formalized water distribution system should be implemented to minimize waste and improve community relations.

Many mines use evaporative cooling methods, such as cooling towers, to cool major mill equipment. However, due to the large volume of makeup water required in a conventional mine, most sites have enough cool water to meet all of their cooling needs with open water-based heat exchangers. Available cooling technologies are discussed in Section 5.6.1.

Slurry pumps are often fitted with GSW systems to flush abrasive particles from the pump packing, while also cooling and lubricating the packing. Often GSW is ineffectively used, leading to leaking and excessive water consumption. Mechanical seals can be used to replace gland seals, or systems can be put in place to monitor and limit GSW consumption (Savage, 2007). As GSW is not lost to the system, but flows into the slurry, mechanical seals present an opportunity to reduce clean water requirements, not overall water requirements.

Mines have also implemented common off-the-shelf water saving techniques such as low water showers and low or zero water toilets (Newmont, 2008; Thompson & Minns, 2003).

A more radical opportunity to reduce water use is to pre-concentrate ore at coarser particle sizes using such methods as dense media separation or sorting technologies. For some ores, over half of the ore can be rejected prior to grinding with a minimum loss of valuable mineral (Bamber, 2008). Test work reports on copper porphyry ores from several mines found conductivity sorting achieved up to 50% waste rejection by mass from the ores, at recoveries from 85 – 92% (Miller et al., 1978). Reducing the amount of ore sent to the mill can reduce the size of the mineral processing plant and associated energy requirements, the amount of water required for processing, the amount of cooling water required, and the corresponding evaporation and water retained in the tailings. Slatter *et al.* (2009) also mentioned the use of pre-concentration to reduce water consumption. In addition, pre-concentration can lead to a reduction of fines generated, thereby lowering water losses in the tailings (de Kretser et al., 2009). Pre-concentration of gold ores may also lead to a reduction in cyanide consumption and associated cyanide destruction water treatment requirements.

An extreme option to reduce water consumption would be to implement more dry processes or an entirely dry processing plant (E. Brown, 2003). Earlier Jancovich (1971) discussed dry processes and claimed that the water requirement of 1 to 3 m^3/tonne of ore processed in wet mills make wet processes either extremely expensive or impossible in arid environments. Dry processes currently have significant limitations and wet processes are typically more efficient, consume less energy, and create fewer dust-related health concerns than dry processes. Furthermore, there are no real dry alternatives to flotation or hydrometallurgical processes. However, further research and development may make dry processes more feasible in the future (Napier-Munn & Morrison, 2003).

2.5.2 Efforts to Reuse Mine Water

Efforts to reuse mine water include:

- Collecting and reusing surface runoff water
- Reusing mine dewatering water
- Reusing cooling water
- Reusing dust scrubber water
- Reusing grey water
- Reusing waste water from off-site locations and alternative sources of water

Any site with precipitation can investigate the usefulness of implementing a watershed management plan to collect site precipitation and any other water runoff for mine use. Storing precipitation in reservoirs, or in the TSF, can significantly reduce offsite water requirements (DWAF, 2007). Mount Isa Mines recently built a 20,000 m^3 reservoir to collect rainwater (Lévy et al., 2006). Barrick's Buzwagi gold mine in Tanzania built a 75 ha high-density polyethylene lined rainfall harvesting area, which drains into a storage pond with a floating cover system to reduce evaporation (Mayer et al., 2008). Mines with excavations below the water table are required to dewater around their pits or underground. This water is generally suitable for use, and is commonly used for process water or by other water consumers. Similarly, seepage water from a TSF may be reused in the plant or mine.

Some mines use water-based cooling systems, such as "tube and shell" or plate heat exchangers to cool major equipments' electrical, gear or lubrication systems. This water is generally clean and can be used for a wide variety of mill water consumers, including dust scrubbers, pump

GSW, spray water, reagent mixing water, and flotation or grinding dilution water. This is described in more detail in Section 5.6. In the oil sands industry, Syncrude uses heat exchangers to allow the spent cooling water to pre-heat the process water required for bitumen separation (Matte & Velden, 2005). Brunswick Mine similarly reuses spent cooling water (Roberts et al., 2008). Mine sites also have the opportunity to reuse grey water from showers or washing utilities in toilets or for watering green space around the site (Thompson & Minns, 2003).

Reusing waste water from an offsite location, such as waste water from an external industrial facility or treated effluent from a municipal sewage plant, may be a valuable option to reduce water withdrawals from lakes, rivers or the ground. Other alternative sources for water available may include sea, saline and industrial waste water. Nearby municipal or industrial sites may have waste water available that is less expensive or more responsible/politically astute to use than local freshwater or groundwater sources. In Australia's Hunter Valley, Xstrata built a 16 km pipeline linking several mines together to enable water sharing (Lévy et al., 2006). Queensland Alumina, Cadia Hill mine and the Commodore Coal Mine in Australia all use treated effluent from nearby sewage treatment plants for their water supply (Kent & McCreath, 2003; Schumann et al., 2003; Stegink et al., 2003). The use of sea water as a replacement for fresh water has been practiced for decades (Rey & Raffinot, 1968). Queensland Alumina also planned on replacing freshwater evaporative cooling with seawater cooling towers paired with heat exchangers during a period of intense water restrictions (Stegink et al., 2003).

While the use of alternative water sources presents many excellent opportunities to reduce clean water consumption, caution must be used due to the potential negative impact of water quality on processing. Inspiration Copper tried to switch to using treated sewage effluent for process make-up water and quickly experienced a sharp reduction in concentrate recovery and grade. Flotation test work was undertaken with different water treatment technologies, but it was found the cost of treatment was too high to justify use. They mixed sludge from a lime-based mine water neutralization clarifier with the treated sewage effluent prior to addition to the TSF and then found flotation results satisfactory (Riggs et al., 1977). Similar test work was undertaken by Fisher and Rudy (1978) on ore from Cyprus Prima, who found that copper recoveries dropped by 2% and molybdenum recovers by 32% when undertaking flotation in treated sewage effluent versus de-mineralized water. Problems with froth generation where also noted. Fisher and Rudy (1978) tested several secondary water treatment methods to reduce the impact on flotation, including mixing the effluent with mill tailings, treating the effluent with activated carbon, anionic exchange, cationic exchange, and flotation of the effluent. Treating the effluent through anionic exchange eliminated the negative flotation impact and activated carbon significantly reduced the negative impact. Zhang and Zhang (2012) undertook similar research and found treating sewage effluent with slaked lime eliminated the negative impact of copper and molybdenum recovery. Lui *et al.* (2011) investigated the impact of *E. coli* on copper flotation to better understand the exact mechanisms for the negative impact of using sewage effluent as makeup water. Muzenda (2010) also undertook test work on the impact of different water sources on flotation. Anglo Platinum aims to replace all potable water with water from alternative sources, primarily treated sewage (Slatter et al., 2009). The shortage of potable water at some of Anglo's operations led to efforts to replace potable water use, primarily used for reagent mixing and pump GSW, with treated sewage effluent. Test work found that dissolved organics had a negative impact on flotation and, in addition, that ion-exchange was a better treatment option than the use of activated carbon. Several others have found negative impacts from using treated sewage effluent (Muzenda, 2010).

A corollary to the concept of using waste water is for mines with a positive water balance to sell/give any mine discharge water to a nearby industrial or municipal consumer, rather than discharging the water directly to the environment, thus offsetting another user's freshwater requirements. The Phosphate mining operations in Jordan have successfully used waste water for irrigation of fodder crops with no reported ill effects (Rimawi et al., 2009). In Nevada, Newmont owns the Elko Land & Livestock Co., which manages the 450,000-acre TS Ranch outside of Carlin. Water discharged from their nearby mining operation is used at the ranch to irrigate crops (Newmont, 2008). Xstrata's Ulan coal mine in Australia uses its waste water to irrigate 242 hectares of perennial pasture land (Lévy et al., 2006). Newmont's Yanacocha mine in Peru converted a mined-out pit to a treated water reservoir for regional agricultural use (Newmont, 2009). Barrett *et al*. (2010) described a method for evaluating the potential for trading excess water between nearby mines.

2.5.3 Efforts to Recycle Mine Water

Most major mines now practice at least some water reuse and recycling and many mines recycle all available water. By 1973 an estimated 69% of water use in the minerals industry in the USA was recycled (Turcotte, 1986). A Canadian Mineral Processing study in 1973 found most Canadian mines were practicing recycling (Pickett & Joe, 1974). As shown in Table 2.5, in 2009 mining recirculated, or reused and recycled, a higher percentage of water than any other industrial category in Canada (Statcan, 2012).

Table 2.5 Canadian Industrial Water Recirculation Rates, 2009

2009 Category	**Water Recirculation Mm^3**	**%***
Manufacturing	2,003	52.6
Thermal Electric Power Generation	4,220	16.0
Mining	1,548	311.3
Total Industrial	**7,771**	**100**

*Amount of water recirculated as a percentage of water withdrawn. (Statcan, 2012)

The key advantages for recycling water at mine sites can include: reduced pumping costs, increased heat recovery, recovery of residual reagents, reduced biological activity, reduced water consumption (Alfano et al., 1983; Turcotte, 1986).

Efforts to recycle mine water include:

- Recycling TSF surface water
- Recycling TSF seepage water
- Recycling tailings thickener overflow
- Recycling concentrate or intermediate thickener overflow
- Recycling potential mine effluent water

Water is commonly recycled when it drains out of deposited tailings and collects in TSF collection ponds. Likewise, any collected seepage or drainage from a TSF is commonly returned to the plant. Tailings thickeners are often used in mines around the world, and the effluent is recycled as process water. Callow (1927) detailed multiple mines recycling water over 80 years ago and Turcotte (1986) and Pickett and Joe (1974) described more operations recycling water several decades ago. Hamilton and Woodcock's (1993) review of the Bougainville copper mine water balance was an early example showing the advantages of tailings thickeners: to process 135 000 t/d of ore required 2.2 m^3/s of water and a further 0.5 m^3/s for gland seals, lubrication, cooling, crusher dust seals, and belt washing. Water came from a nearby river, but pumping costs were significant. Internal studies pointed toward better flotation at lower pulp densities, so high rate tailings thickeners were installed. Concentrate thickeners and intermediate thickeners or clarifiers are also common within plants and the overflow is available to recycle. Frommer (1970) detailed recycling water in the taconite flotation whereas Dahstrom (1986) focused on coal processing.

Mines with a positive water balance by definition discharge excess water into the local environment. However, some mines with a positive balance may still import raw water onto site for clean water users, such as pump GSW. Due to regulatory requirements, the discharge water may be of higher quality than the raw water available to the mine. Recycling the water to the mine reduces the overall water consumption of the site and may also reduce overall water treatment costs.

Some mineral processing plants have reported negative process impacts from using recycle water for process water. Rao and Finch (1989) discussed the reasons of why recycle water can have a negative impact on flotation chemistry, in addition to providing potential solutions. Negative examples are typically found in multi-stage flotation circuits (Forssberg & Hallin, 1989; Johnson, 2003; Pickett & Joe, 1974; Sharp & Clifford, 1973). Coal mines in Australia can also experience negative impacts from using recycle water due to the buildup of salts (Cote et al., 2007; Moran & Moore, 2005). Turcotte (1986) references how some mines are required to have separate recycle water systems due to the significant differences in each water system and potential negative process impacts. This is similar to Alaska's Pogo mine, which provides an example of the advantage of combining, reducing and recycling plant water. Teck Cominco undertook a detailed study of the water system at the Pogo project during the design phase (Konigsmann, 2002). The Pogo layout has several re-circulating recycle water streams to separate water contacted with cyanide from other streams. The plant produces two tailings streams. The cyanide-contaminated, sulfide tailings, from the flotation concentrate, are disposed of underground through paste backfill. The non-contaminated, quartz-only, flotation tailings material is filtered and dry stacked on the surface.

Slatter *et al.* (2009) described the experience of Anglo Platinum, which found it could take several weeks for a plant to achieve equilibrium if there was a significant change in water supply quality. They also described the difference in water quality between water reclaimed from the TSF and water recovered from thickeners, due to the impact of time, oxidation and the impact of evaporation and or precipitation.

If recycle water needs to be treated, often treating a small side stream will achieve acceptable results (Johnson, 2003). Breakwater Tunisia reported on a small mine with zero discharge and high water recycling rates where excessive salinity of the recycle water was having a negative process impact (Henchiri & McDonald, 2003). Breakwater undertook flotation test work and decided on a reverse osmosis (R/O) plant to clean the water. They installed an R/O plant with a capacity of 85 m^3/hr, or 45% of the total process water requirement. The salinity of the process water decreased from 10-12 g/L to less than 3 g/L after installation, leading to a significant increase in zinc concentrate grade.

Often, however, there are no major process impediments to using recycle water and there may be significant benefits, such as the recycling of reagents. Johnson (2003) undertook a review of past literature regarding the process implications of using recycle water and documented how to determine the impact of this use. Schumann *et al.* (2003) studied the impact of changing the blend of recycle and makeup water on flotation recovery at the Cadia and Ridgeway concentrators in Australia. Bahrami *et al.* (2007) investigated water treatment to eliminate the negative process impacts of recycle water at a gold plant in Iran.

Levay & Schumann (2006) argued for the need for mines to have a systematic approach to managing water systems at a mine. They highlight the importance of water quality between different fresh and recycle water streams, and its impact on process unit operations, including flotation. They discuss how to systematically determine the key elements of a mine site water system. However, Levay & Schumann (2006) do not describe how to use this information to improve mine water system design.

2.6 General Water Network Design

"The reuse of process water will become a standard practice in the future for reasons like environmental constraints, economic factors and above all, the utilization of energy and available water"(Forssberg & Hallin, 1989).

Pickett and Joe (1974), Yezzi (1985), Stewart *et al.* (2003), Atmaca and Kuyumcu (2003) and the South African best practices guidelines are among the only mining literature which argues for a more systematic approach to mine water system design. All five draw heavily from methods developed for other process industries.

Cano-Ruiz and McRae (1998) undertook a large review of process design literature from an environmental view point. Cano-Ruiz and McRae described how the system boundaries for process design have expanded from just looking at the raw materials, utilities and products in the 1960s, to including utilities production and heat recovery in the 1980s, to the need to consider waste treatment and the supply of raw materials in the 21st century. Incorporating a more complete system in the formation of the design problem allows optimization of a wider range of options. Systems engineering methods present one approach to how to improve water system management. Mar (1998) provided a comprehensive overview of the use of systems engineering for water resources. Some efforts at improving mine water systems, such as Cote *et al.* (2007) and Atmaca and Kuyumcu (2003), use systems engineering based approaches.

Cano-Ruiz and McRae (1998) discussed several methods for optimizing waste generation, minimizing chemical release and categorizing the impact of releases. They identified pinch methodology as a key tool for minimizing waste. This method was initially intended to reduce capital and energy costs in processing plants through the better design of heat exchanger networks, but was extended to water system analysis by Wang and Smith in 1994.

Water pinch analysis is one of several water allocation planning (WAP) techniques developed out of mass exchanger network design and heat exchanger design (Bagajewicz, 2000; Cano-Ruiz & McRae, 1998; Castro et al., 1999; Manan et al., 2004; Olesen & Polley, 1997; Sorin & Bédard, 1999; Tripathi, 1996; Wang & Smith, 1994). In other process industries much research has been devoted to developing approaches to optimize plant utility networks, particularly regarding heat exchanger, water treatment and water recycling optimization. The main goal of WAP research is to find optimal wastewater reuse solutions to minimize or completely eliminate water consumption and/or discharge (Bagajewicz, 2000; Koppol et al., 2004). Over the past 15 years, several other methods have been proposed to minimize plant water requirements, in some cases also minimizing energy utilities, operating costs or overall costs. These methods include linear programming, nonlinear programming and algorithmic methods, and remove some of the constraints of the graphical water pinch methods (Aly et al., 2005; Bagajewicz et al., 2000; Bagajewicz, 2000; Bagajewicz et al., 2002; Campos de Faria et al., 2009; Savelski & Bagajewicz, 2001).

None of the WAP papers specifically addressed mining. Chapter 7 presents an attempt to demonstrate how WAP can be adopted to the mining industry, particularly with respect to plant water system design.

Better water system design revolves around two key concepts: first, running all processes at the highest solids density possible without negatively impacting the process and, second, supplying all processes with the poorest acceptable quality water, that does not impact process performance (Bagajewicz et al., 2000; Department of Water Affairs & Forestry, 2006b).

2.7 Conclusion

This literature review aimed to highlight literature in the following areas:

- First, the importance of water and energy use to discussions of mining and sustainability;
- Second, past efforts to quantify mine water use;
- Third, past efforts to describe mine site water systems, including within mineral processing plants and TSFs;
- Fourth, past efforts to reduce, reuse and recycle water on a mine site; and
- Fifth, how other process industries have worked to improve water system design.

The findings of this literature review were as follows:

- Mine water use has been a key concern for hundreds of years and its importance has increased in recent years as the mining industry strives toward becoming more sustainable;
- Water use has consistently been raised as a key issue in literature about mining and sustainability;
- Past efforts to quantify mine water use on an industry wide level have been hampered by a lack of consistency in reporting standards and have often relied on anecdotal case studies and/or potentially non-representative samples;
- Past literature often found that access to water, water quality, and the energy associated with water use were key concerns facing the mining industry;
- There have been considerable efforts over the past century to address concerns raised by mine water use, including efforts to reduce water use and to deal with challenges faced by the mining industry with respect to mine water systems.
- In recent years, there have been several calls to improve mine water use in a systematic manner, often referring to methodologies developed by other process industries; and

- There may be opportunities to apply water allocation planning approaches from other process industries to the mining industry.

3. METHODOLOGY

3.1 Estimating Global Water Withdrawals Due to Mining

The aim of this section is to describe the methodology used to estimate the total global water withdrawals for most major mining commodities based on publically available data. This was a cross-sectional study investigating water withdrawals for the years 2006-2009. This period was selected because 2009 was the most recent year that water withdrawal data was readily available at the time this research was initiated, and widespread water withdrawal data becomes increasingly scarce prior to 2006.

Data was collected for mines producing bauxite, chromite, cobalt, copper from concentrators, copper from SX/EW operations, diamonds, gold, iron, lead, manganese, molybdenum, nickel, palladium, phosphate, platinum, potash, rhodium, silver, tantalum, tin, titanium, tungsten, uranium and zinc. The results of this study are presented in Chapter 4.

Aggregates (sand and gravel), coal and oil sands are excluded from the scope of this study, although all can withdraw significant amounts of water. Aggregates and coal were excluded due to the combination of the large number of mine sites and the extremely poor quality of publically available data. Although much of the production from oil sands is mined and processed in a similar fashion to the rest of the mining industry, oil sands were excluded from this analysis as they would be better included in an oil, or perhaps oil and gas, industry study.

This estimate focuses on mine water withdrawals rather than mine water consumption due to the following reasons:

1. Most of the publically available data from mining companies was reported using the GRI indicators. As mentioned in Section 2.1, the GRI protocols does not include water

consumption, but rather water withdrawals (EN08) and water discharges (EN21). Water consumption could be calculated by subtracting water discharges from water withdrawals. However, this would add to uncertainty of the results, as changes in mine site water stores volumes would not be accounted for. Water stores are not included in the GRI indicators and are not typically reported by mining companies;

2. Calculating water discharges is more challenging than water withdrawals and thus the data is less reliable. EN21 includes the sum of water effluents discharged over the course of the reporting period to subsurface waters, surface waters, sewers that lead to rivers, oceans, lakes, wetlands, treatment facilities, and ground water either through a defined discharge point or over land in a dispersed or undefined manner (Global Reporting Initiative, 2006). Determining dispersed water losses, such as seepage, to any level of accuracy often requires sophisticated efforts to model ground water flows;
3. Other water sinks, such as evaporation, are not included, even though the water also reports back to the local environment. Mining companies regularly utilize large evaporation ponds to eliminate excess water. Furthermore, several mining companies reduce water discharges by using large scale water evaporators (COLDmist, 2010; Sanders, 2011). Mining companies often need to choose between treating excess water to an acceptable standard for discharge or evaporating excess water. It is not clear why one method would be considered a discharge and the other is not; and
4. Excluding water discharges from water consumption calculations does not fully account for the impact of water withdrawals on a local environment, such as changes in local aquifers. Furthermore, single point discharges into rivers and streams can have a negative impact due to increased flow or altering seasonal flow variations, even when discharging clean water, although these impacts can be controlled and minimized

through careful design (ADRET, 2008). Oddly, if a mining company was to target reducing water consumption instead of reducing water withdrawals, it could focus on discharging more water rather than reducing water withdrawals.

Data was only collected on water withdrawals (EN8) rather than the other three GRI indicators due to several reasons:

1. EN8, or withdrawal data, is relatively comparable between sites and provides a strong indication of water consumption;
2. EN9, or water sources affected by water withdrawals, is entirely site specific and not widely reported, and so is not useful for comparison;
3. EN10, or the volume of water recycled or reused, has less direct impact on the surrounding environment, as discussed below; and
4. EN21, or water discharged, is limited in its usefulness in describing mine water use as described above.

As discussed in Section 2.3, recycling and/or reusing water is practiced at most mine sites and can significantly reduce water withdrawals in addition to reducing reagent and energy requirements. Regardless, water recycling and reuse data, as reported by mining companies, were not included in this global estimate of mine water use. Because water withdrawals have a more direct impact on the surrounding environment, this estimate focused on global water withdrawals. The amount of water recycled and reused on the mine site has less impact on the surrounding environment than water withdrawals do, other than the amount of energy required for recycling and reuse.

To summarize, with the mine water use information publically available, mine water withdrawal rates were chosen rather than water consumption or recycle rates in order to provided a more complete and more accurate estimate of global mine water use, and to better reflected the full impact of water use on the surrounding environment.

Estimating global water withdrawals for a mined commodity has a few key challenges. First, there is no single standard used to define what to include or exclude in water withdrawal calculations. For example, it is necessary to decide if rainfall on a mine site should be included. Second, many mines produce multiple products. When determining the amount of water used to produce different commodities, it can be challenging not to double count water use. Third, a relatively small fraction of the world's mines report their water use. Thus, any estimate must take information from a subset of mines and extrapolate the data to estimate global water withdrawals.

This section addressed these challenges as follows:

Only water withdrawals specifically occurring on mine sites, as defined by GRI indicator EN08, were included in this study, excluding smelters and refineries. Once a saleable product has been produced on a mine site it can be shipped anywhere in the world to be further processed, thus downstream process water withdrawals are not mine site specific. However, the initial processing generally must occur on site due to the high economic costs of transporting low value ore significant distances.

The second challenge identified in estimating global mine water withdrawals by commodity was how to apportion water withdrawals at mines producing multiple commodities. Mining operations typically calculate their ore reserves and cut-off grades by including revenue from by-products. Thus an operation may mine an area that had a copper grade that is too low to be economically viable on its own, provided that the ore has a high enough gold grade that the combined copper and gold revenue is profitable. If the ore did not have the higher gold grade, the mine may choose not to extract the ore in that area. Thus one approach to attributing water use to a particular commodity would be based on the portion of revenue the mine derives from the commodity. For example, if a mine processes 2,000,000 tonnes of ore to produce 1 tonne of gold and 15,000 tonnes of copper, while withdrawing 1,000,000 m^3 of water, it would be double counting water withdrawals to record that it took 1,000,000 m^3 of water to produce 1 tonne of gold and an additional 1,000,000 m^3 of water to produce 15,000 tonnes of copper. If gold is worth $30,000,000/tonne and copper is worth $6,000/tonne, then the mine's total revenue would be $120,000,000. Gold would have contributed $30,000,000, or one quarter of the revenue at the mine. Thus the amount of water withdrawn attributable to gold, based on revenue, could be calculated as one quarter of the total, or 250,000 m^3.

There are some limitations to this approach. For example, compare the mine in the previous example, processing 2,000,000 tonnes of ore per year to produce 1 tonne of gold and 15,000 tonnes of copper, with a similar 2,000,000 tonnes of ore per year mine to produce 15,000 tonnes of copper with no gold by-products, both withdrawing 1,000,000 m^3 of water. The first mine would have withdrawn 750,000 m^3 of water attributable to copper while the second mine would have 1,000,000 m^3 of water attributable to copper, even though the two mines were using similar processes and were withdrawing the same amount of water. Furthermore, if the price of copper relative to gold increased, the amount of water withdrawn attributable to copper would increase as well. These two limitations make it difficult to benchmark individual mines by using this method. However, there are limited alternatives to estimating global water withdrawals otherwise. If by-products are ignored, then double counting becomes unavoidable. It is not clear how one would choose the primary product at many mines, as a change in the relative commodity price could change what was considered the primary product from year to year. For benchmarking individual mines against each other, it may be more reasonable to compare mines using similar production methods, without accounting for commodity production. However, using such an approach would not allow for estimates of global withdrawals.

The third challenge identified in estimating global mine water withdrawals by commodity was how to use the limited information available from individual mines to extrapolate total global water withdrawals. There are several ways that global water withdrawals can be estimated. If all mines around the world reported water withdrawals, total global water withdrawal could be calculated by summing all of the data from the mines. However, only a small portion of mines report this data. Two methods can be used to estimate global withdrawals by extrapolating data from reporting mines: estimating water withdrawals based on commodity production or estimating water withdrawals based on ore production. Estimates of annual global production of different commodities are available from a handful of sources. If an average water withdrawal per unit of commodity production from a representative sample of mines is calculated, this average can be multiplied by the global annual production to estimate global water withdrawals for a specific commodity. However, in many cases the amount of water withdrawn by a mine is driven more by the amount of ore processed than the amount of commodity produced. For example, a low-grade copper porphyry mine processing 2,000,000 tonnes of ore per annum and producing 15,000 tonnes of copper, with an annual water withdrawal of 1,000,000 m^3, would have an average recovered copper ore grade of 0.75%. As discussed in Chapter 2, the amount of water withdrawn would be 66.7 m^3/tonne of copper produced. Most of the water lost on the site would be through evaporation and water retained in the tailings storage facility. If a similar sized operation mined ore with a lower recovered grade of 0.60% Cu, the amount of water required would most likely remain similar, at 0.5 m^3/tonne of ore. However, the amount of copper produced would be 12,000 tonnes and the water withdrawn would be significantly higher at 83.3 m^3/tonne of copper produced. Thus using the average ore grade and the average water withdrawn per tonne of ore processed may be a more accurate tool for estimating global water withdrawals than using the average water withdrawn per tonne of product.

The methodologies used for estimating global water withdrawals are described in detail below. The annual global water withdrawal attributable to commodity c due to mining (W_{tc}, m^3/y) was estimated using two methods: the ore production estimate and the concentrate production estimate. The key assumption of the ore production estimate method is that water withdrawals are closely associated with the amount of ore processed at a mine site, whereas the key assumption of the concentrate production estimate method is that water withdrawals are closely associated with the amount of commodity produced at a mine.

3.1.1 Data Collection

Annual ore tonnage and production by commodity data collection

The amount of ore processed and concentrate produced at individual mines from around the world was collected for the years 2006-2009. O_k is the quantity of ore processed at mine k (t/y) and X_{kc} is the quantity of commodity c produced at mine k (t/y). Where possible all individual products produced from a mine were recorded. For example, many copper mines produce by-products such as gold, silver and molybdenum. The total annual global production of each commodity, or X_{tc}, was also recorded (t/y).

Annual water withdrawal data collection

Sustainability reports and secondary sources were used to determine annual water withdrawals at mining operations for the years 2006-2009. W_k is the amount of water withdrawn from mine k. Data was collected where possible from various companies and organizations. When possible, EN8 figures from GRI reports were used.

Water withdrawals from precipitation (the amount of rain or snow that fell on a mine site) were not included, as few mines included precipitation in their water withdrawal calculations, even though EN8 recommends including precipitation. In some cases mine sites only reported their water withdrawal per tonne of ore processed or concentrate produced. The mine's water use was then calculated by multiplying the production by the water consumption rate. In some cases mines reported water withdrawals in graphical charts. In these cases, the data was extracted by pasting the chart into a spreadsheet and extrapolating data to replicate the charts as closely as possible.

In virtually all cases, mine production data (ore processed and products produced) were available for all sites reporting water consumption. As a rule, any organization which is transparent enough to publish water withdraw data also likely publishes their production data.

Annual Average Commodity Prices Data Collection

The average commodity prices (P_c) from 2006 to 2009 for all relevant commodities were collected from a variety of sources.

3.1.2 Ore Production Method

With the ore production method, the total amount of water withdrawn attributable to a commodity was calculated by estimating the total amount of ore processed to produce the commodity multiplied by the weighted mean (average) amount of water required to process a given quantity of ore, or:

$$W_{tc} = O_{tc} \cdot WMO_{tc} \qquad [5.1]$$

where W_{tc} = total water withdrawals globally attributable to commodity c, [m^3/y]; O_{tc} = total ore processed globally to produce commodity c, [t/y]; and WMO_{tc} = the global weighted mean quantity of water required to produce commodity c, [m^3/t ore].

O_{tc} was estimated by dividing the sum of the global production of commodity *c* (X_{tc}, t/y) by the weighted mean ore grade of the commodity (G_{tc}, % mass), or:

$$O_{tc} = \frac{X_{tc}}{G_{tc}} \qquad [5.2]$$

where O_{tc} = total ore processed globally to produce commodity *c*, [t/y]; X_{tc} = total global production of commodity *c*, [t/y]; and G_{tc} = the global weighted mean recovered ore grade of commodity *c*, [% mass].

The global mean recovered ore grade of commodity *c* (G_{tc}) was assumed to be equal to the mean recovered ore grade of commodity *c* from the set of mines *a* (G_{ac}). G_{ac} was estimated by dividing the sum of the total quantity of commodity c produced by the sum of the total quantity of ore processed from mines reporting both ore production and production for commodity *c* (*a*), or:

$$G_{ac} = \frac{\sum_{k=1}^{a} X_{kc}}{\sum_{k=1}^{a} O_{kc}} \qquad [3.3]$$

where G_{ac} =the global weighted mean recovered ore grade of commodity *c* from the set of mines *a*, [% mass]; X_{kc} = the quantity of commodity c produced at mine *k*, [t/y]; O_{kc} = the quantity of ore processed at mine *k* while producing commodity *c*, [t/y]; and *a* = the total number of mines reporting both X_{kc} and O_{kc}, $1 \leq k \leq a$. G_{ac} is the average recovered metal grade, not the average metal grade of the ore, allowing the mine's beneficiation plant efficiency of recovery to be ignored.

The global weighted mean quantity of water required per tonne of ore to produce commodity *c* (WMO_{tc}) was assumed to equal the global weighted mean quantity of water required per tonne of ore to produce commodity *c* from the set of mines *aw* (WMO_{awc}). WMO_{awc} was determined by dividing the sum of the quantity of water withdrawn attributably to a particular commodity (W_{kc}, m^3/y) by the amount of ore processed from mines reporting ore production, production for commodity c and water withdrawals (aw), or:

$$WMO_{awc} = \frac{\sum_{k=1}^{aw} W_{kc}}{\sum_{k=1}^{aw} O_{kc}} \quad [3.4]$$

where WMO_{awc} = the global weighted mean quantity of water required to produce commodity c from the set of mines aw, [m^3/t ore]; W_{kc} = the quantity of water withdrawn attributable to commodity c at mine k, [m^3/y]; O_{kc} = the quantity of ore processed from mine k producing commodity c, [t/y]; and aw = the total number of mines reporting X_{kc}, O_k, and W_k, $1 \leq k \leq aw$.

The amount of water withdrawn attributable to commodity c at a given mine was calculated as follows:

$$W_{kc} = W_k \cdot \frac{X_{kc} \cdot P_c}{\sum_{c=1}^{b_k} X_{kc} \cdot P_c} \quad [3.5]$$

where W_{kc} = the quantity of water withdrawn attributable to commodity c at mine k, [m^3/y]; W_k = the total amount of water withdrawn at mine k, [m^3/y]; X_{kc} = the quantity of commodity c produced at mine k, [t/y]; P_c = the average annual price of commodity c, [US$/t]; and where mine k produces b_k different types of commodities, $1 \leq c \leq b_k$. The actual amount of revenue received at the mine will be less than the product of X_{kc} and P_c, as annual price does not include any transportation, smelter or other fees. However, the only purpose of calculating the mine revenue for this exercise was to approximate the relative values of the different commodities produced at a mine.

3.1.3 Concentrate Production Method

The estimate of global water withdrawals based on the concentrate production method was based on the assumption that water withdrawals are proportional to the amount of concentrate produced at a mine site. The total amount of water withdrawn attributable to commodity was calculated by multiplying the total amount of commodity produced in a year by the weighted mean (average) amount of water required to produce a given quantity of concentrate, or:

$$W_{tc} = X_{tc} \cdot WMC_{tc} \qquad [3.6]$$

where W_{tc} = total water withdrawals globally attributable to commodity c, [m^3/y]; X_{tc} = total global production of commodity c, [t/y] and WMC_{tc} = the global weighted mean quantity of water required per tonne of commodity c to produce commodity c, [m^3/t].

The global weighted mean quantity of water per tonne of commodity c required to produce commodity c (WMC_{tc}) was assumed to equal the global weighted mean quantity of water per tonne of commodity c required to produce commodity c from the set of mines aw (WMC_{awc}). WMC_{awc} was determined by dividing the sum of the quantity of water withdrawn attributably to a particular commodity (W_{kc}, m^3/y) by the amount of commodity produced from mines reporting production for commodity c and water withdrawals (aw), or:

$$WMC_{awc} = \frac{\sum_{k=1}^{aw} W_{kc}}{\sum_{k=1}^{aw} X_{kc}} \qquad [3.7]$$

where WMC_{awc} = the global weighted mean quantity of water per tonne of commodity c required to produce commodity c from the set of mines aw, [m^3/t]; W_{kc} = the quantity of water withdrawn attributable to commodity c at mine k, [m^3/y]; X_{kc} = the quantity of commodity c produced from mine k, [t/y]; and aw = the total number of mines reporting X_{kc} and W_k, $1 \leq k \leq aw$. W_{kc} was calculated as per Equation 3.5.

3.1.4 Summary

To summarize, the two different methods described were used here to estimate global water withdrawals due to mining. Both methods required that ore production and commodity production values were collected for as many mines as possible. Mine water withdrawal data were then also collected for as many mines as possible; however, water withdrawal data are less widely available, so the water dataset is considerably smaller than the ore and production dataset. In addition, global production of different commodities was collected, as well as annual commodity prices.

For the ore production method, Equation 3.3 was used to calculate the average ore grade of different commodities produced from the mine ore and commodity production dataset. The global production of a commodity was then divided by the average ore grade in order to estimate the total amount of ore processed to produce the commodity, using Equation 3.2. Equation 3.5 was used to calculate the amount of water used to produce the commodity at each mine reporting water data. Equation 3.4 was used to calculate the weighted average amount of water used per tonne of ore to produce a commodity. The results from Equation 3.2 and 3.4 are used in Equation 3.1 to estimate the global amount of water used to produce a commodity.

For the concentrate production method, Equation 3.5 was used to calculate the amount of water used to produce a commodity at each mine reporting water data. Then Equation 3.7 was used to calculate the weighted average amount of water used to produce a commodity. The results from Equation 3.7 are used in Equation 3.6 to estimate the global amount of water used to produce a commodity.

3.2 Case Study – Cerro Verde Concentrator

The aim of this case study was to describe the design, startup and optimization of a mine site and plant water system. The case study outlines some of the opportunities and constraints faced by mining and engineering companies during the design of mine water systems. The purpose of this description is to provide context and realistic data for the remaining chapters of the thesis, which focus on methods to reduce mine water usage and improve mine water system design. The results of the case study are presented in Chapter 5.

The case study methodology is as follows: First, the case study location and project are presented, along with the preliminary design concepts of the plant water system developed during the project feasibility study. Second, the detailed design phase is described, outlining the fresh water system requirements, the quantity and quality of the available water source, and the water treatment and pumping system developed to meet the plant requirements. Third, the challenges faced during the pre-operational testing of the fresh water system are presented. These challenges were sufficient to jeopardize a successful startup of the plant. Fourth, the various options available to allow a successful startup of the plant are detailed and the choice of a temporary solution is described. Finally, the details of how to define and design a more permanent solution are described.

The description and data were taken from internal Fluor engineering documents produced by the author or from the author’s personal experience. The engineering documents included the Process Design Criteria, the Mass, Water, and Heat Balances, Process Flow Diagrams, Piping and Instrumentation Diagrams, and various reports and trade off studies. The documents were based on calculations, vendor and client data, and industry rules of thumb.

3.3 Mine Water Requirements

The aim of this section is to describe the methodology used to create a mine water system model and test different options to reduce mine water consumption and reuse or recycle water available on a mine site. Section 2.5 describes several different options identified in past literature on how mines can better reduce, reuse or recycle water. A base case mine water system model was developed and six scenarios were used to test the potential water saving achievable using these approaches. These scenarios demonstrate how to quantify the individual and combined potential impact from different water saving approaches.

The mine water system model and scenarios were built around a simple mine water balance of a hypothetical 50,000 tpd copper mine. The potential water savings that could be gained by different water saving strategies were estimated and applied to the water balance model in each of the scenarios. The Wels and Robertson model, described in Section 2.4.3, is the basis for the water balance of the TSF. The six scenarios include the base case, the base case with water conservation efforts to reduce site evaporation, a paste tailings case, a filtered tailings case, an ore pre-sorting case and a case which combines water saving features of the other scenarios. Chapter 6 presents the model and the results of the scenario in detail.

3.4 Mine Water Network Design

Section 2.6 described several different approaches used in other process industries to solve water allocation problems. Better water system design can reduce energy needs, reduce water treatment requirements, and in some cases reduce overall water consumption.

The aim of this section is to describe a method of reducing energy requirements in a mine water system by matching water sources and consumers in low energy combinations. The method, referred to as the Mine Water Network Design (MWND) method, estimates the energy required to supply water from available sources to specified consumers and then identifies potential combinations to lower overall energy requirements.

The MWND approach is based on the assumption that any water source can be treated sufficiently to reach the quality requirements of any water consumer; however, there is a cost associated with meeting those requirements. A water consumer may have several requirements, including minimum water quantity, pressure, temperature and purity. Likewise, any water source will have a particular quantity and quality of water available. A water source can meet the pressure, temperature and purity requirements of a consumer by pumping, cooling/heating, and water treatment. A combination of water sources may be required to meet the specific water quantity requirements of a consumer. If there is a large discrepancy between the water consumer requirements and the water source, the energy necessary to meet the consumer requirements will be large as well.

The method consists of five steps: a description of the water balance, a description of all potential water sources, a description of all major water consumers, the construction of energy requirement matrices, and the use of linear programming to minimize energy requirements.

Energy consumption can be reduced by improving the water network design by taking advantage of opportunities to reuse water discharged from one operation in another operation, and by analyzing options to minimize pumping, cooling and water treatment requirements. Figure 3.1 and Figure 3.2 depict a simple flow sheet of how to reduce the quantity of higher quality water required. Figure 3.1 is a block diagram of water requirements for some water consumers common in mineral processing plants: ball mill cooling systems, air compressor cooling water and gland seal water for slurry pumps. All require water of relatively high pressure and high purity. Figure 3.2 provides an example of what a MWND analysis may produce. Ball mill cooling water generally requires cooler water than air compressors, so the waste water from the ball mill cooling systems can feed the air compressors. Gland seal water is not generally temperature dependant, so could be fed by the air compressors' spent cooling water. Such a solution would not reduce the overall plant water quantity requirement, but could significantly reduce the quantity of higher quality water required, as indicated by the required purity, pressure, and temperature.

Figure 3.1 Traditional Water Network

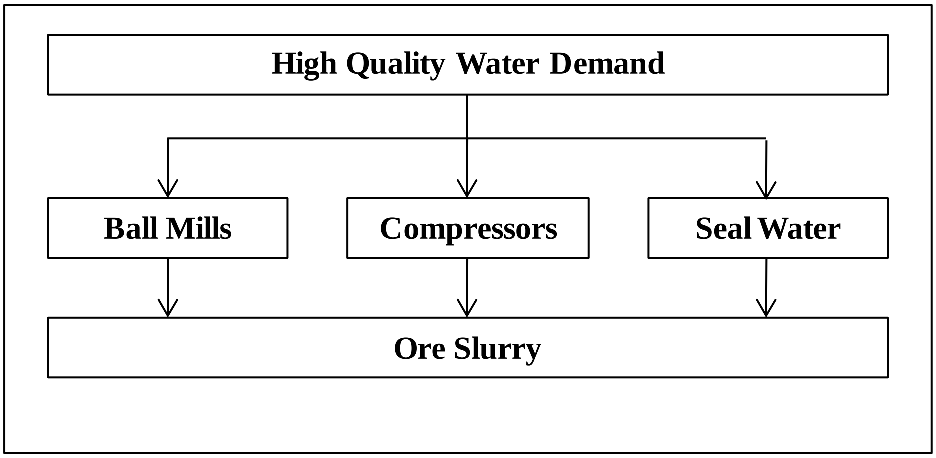
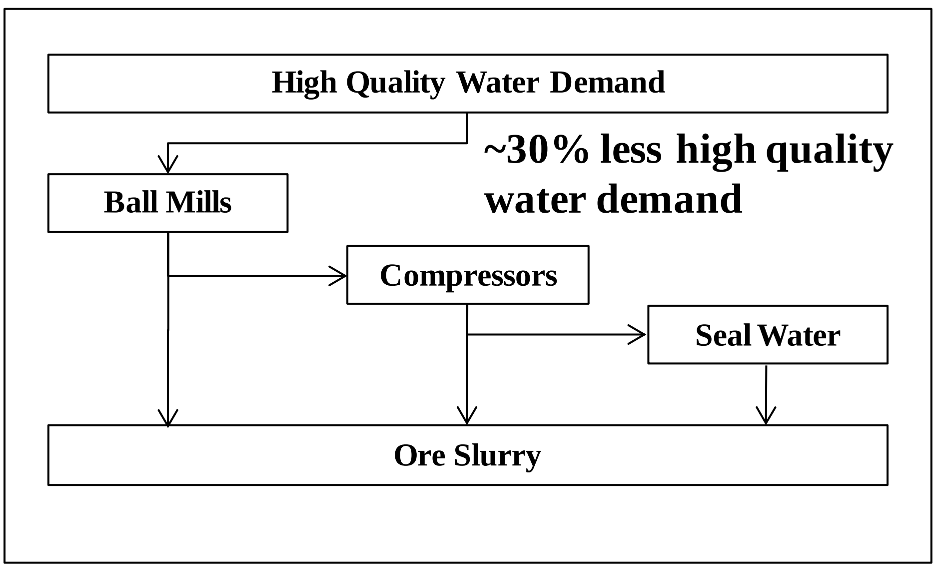


Figure 3.2 Improved Water Network



Chapter 7 details the calculations used in the MWND method, in addition to applying the method to the model described in Chapter 6. Furthermore, the model is applied to the six water saving scenarios described in Chapter 6 in order to demonstrate the water and energy savings possible by combining the two approaches.

4. ESTIMATING GLOBAL MINE WATER USE

The objective of this chapter is to estimate global mine water withdrawals by commodity. The chapter applies two novel methodologies; the Ore Production Method and the Concentrate Production Method. The methodologies are described in Section 3.1. Later sections of the chapter are dedicated to analyzing the inputs and results of the estimates.

4.1 Data Collection Results

Annual ore tonnage and production by commodity dataset

The Raw Materials Database (Raw Materials Group, 2011), a commercially available mining database, was used to create a preliminary database. This database was substantially supplemented and corrected from a number of documents made publically available by companies, governments and other organizations. Documents included annual, monthly and quarterly reports, sustainability reports and other related reports or documents. The US Geological Survey (USGS) also publishes estimates on global annual mining production by commodity on a national and global basis, although it does not include production figures for rhodium and includes titanium as ilmenite (USGS, 2008; USGS, 2009; USGS, 2010; USGS, 2011). Global uranium production data, not included in the USGS studies, was collected from the World Nuclear Association (WNA) (2011). Table 4.1 lists companies and organizations from which information was taken to compile the data for this study.

Table 4.1 Water Withdrawal Estimate Data Sources

Mining Companies		
African Rainbow	Gold Fields	Pan American Silver
Agnico-Eagle	Goldcorp	Phelps Dodge
Agrium	Grange Resources	Polyus
Alcoa	Grupo Mexico	Potash Corp. of Sask.
Alumina	Implats	OZ Minerals
Anglo Platinum	Inmet	PT Timah
AngloGold Ashanti	Kinross	Rio Tinto
Antamina	Kumba Iron Ore	Richards Bay Minerals
Antofagasta Minerals	Lihir Gold	Rusal
ArcelorMittal	Lonmin	Sesa Goa
Barrick	Lundin	Sherritt International
BHP-Billiton	Metals X	Southern Copper
Cantung	Minera Alumbrera	Sterlite
Cameco	Minara Resources	Sumitomo
Cliffs Natural Resources	Mineracao Rio do Norte	Teck
Codelco	Minsur	Vale
Collahuasi	Mosaic	Vendanta Resources
Debeers	Newcrest	Xstrata
Escondida	Newmont	Yamana
Fortescue	Norilsk	Yunnan Tin
Freeport-McMoRan	North American Tungsten	Yanacocha
Gem Diamonds	Northam Platinum	
Additional Organizations		
Australian Aluminium Council Ltd		
Chilean Copper Commission (Cochilco)		
Geraldton Iron Ore Alliance		
International Council on Mining and Metals		
International Aluminium Institute		
Mackenzie Valley Land and Water Board		
National Roundtable on the Environment and the Economy – Canada		
United States Geological Survey		
World Nuclear Association		
World Resources Institute		

The annual global production of each commodity reviewed (X_{tc}) is shown in Table 4.2. Production data was recorded from the RMD and was cross checked with data from the USGS and the WNA. In the case of discrepancies, the figures from the USGS and the WNA were used.

Most of the production figures in Table 4.2 and the following discussion are in tonnes of metal contained in the concentrate produced, including cobalt, copper, gold, lead, molybdenum, nickel, palladium, platinum, rhodium, silver, tantalum, tin, titanium, tungsten, and zinc. The following commodities are reported as total mass of the ores or concentrates produced by the mine: bauxite (mainly $Al(OH)_3$); chromite (typically containing less than 50% Cr_20_3); diamonds; iron ore (typically containing less than 69% iron); manganese (typically less than 54% Mn); phosphate rock (typically containing around 33% P_2O_5); potash (containing K_20 equivalent); and (uranium (U_3O_8 , or yellow cake).

Table 4.2 Annual Global Commodity Production

	Total Global Production (X_{tc}), t/y			
Commodity	**2006**	**2007**	**2008**	**2009**
Bauxite	193,000,000	204,000,000	211,000,000	199,000,000
Chromite	19,700,000	22,900,000	24,100,000	19,300,000
Cobalt	53,800	53,300	57,200	59,800
Copper (Concentrators)	12,300,000	12,600,000	12,400,000	12,600,000
Copper (SX-EW)	2,820,000	2,970,000	3,090,000	3,250,000
Copper (Total)	15,060,000	15,510,000	15,440,000	15,950,000
Diamonds	35	34	33	25
Gold	2,370	2,360	2,290	2,450
Iron	1,475,000,000	1,585,000,000	1,675,000,000	1,695,000,000
Lead	3,590,000	3,690,000	3,860,000	3,860,000
Manganese	33,200,000	35,100,000	38,100,000	33,600,000
Molybdenum	186,000	213,000	218,000	221,000
Nickel	1,570,000	1,670,000	1,560,000	1,400,000
Palladium	222	219	199	192
Phosphate	151,000,000	160,000,000	165,000,000	166,000,000
Platinum	217	208	189	181
Potash	31,200,000	35,800,000	34,700,000	20,800,000
Rhodium	26	28	24	22
Silver	20,300	21,000	21,300	21,800
Tantalum	870	872	1,190	670
Tin	291,000	301,000	257,000	260,000
Titanium	3,200,000	3,300,000	3,300,000	2,900,000
Tungsten	56,400	54,100	62,200	61,300
Uranium	46,516	48,683	51,716	59,875
Zinc	10,300,000	11,000,000	11,600,000	11,200,000

The number of mines (a) reporting O_{kc} and X_{kc} are shown in Table 4.3. In addition, Table 4.3 shows the percentage of global production for each commodity c included in dataset a ($\%X_{ac}$). Dataset a includes production figures from 1,076 mines over a four year period (2006 to 2009) and includes 6,723 commodity production results. For example, the dataset includes the amount of ore processed and gold produced for 358 mines in 2006, which combined account for 74% of the world's gold production.

Table 4.3 Mine Ore and Commodity Production

	Number of Mines (*a*)				**%X_{ac}, %**			
Commodity	**2006**	**2007**	**2008**	**2009**	**2006**	**2007**	**2008**	**2009**
Bauxite	36	37	44	42	79	76	79	78
Chromite	14	16	11	7	59	54	43	36
Cobalt	26	28	24	24	40	38	30	39
Copper (Concentrators)	196	209	202	183	84	84	84	82
Copper (SX-EW)	29	28	25	24	78	72	58	53
Copper (Total)	225	237	227	207	83	82	79	76
Diamonds	18	23	23	22	56	54	55	40
Gold	358	384	366	334	74	73	70	67
Iron	156	162	199	109	60	58	65	46
Lead	86	83	78	78	59	52	47	47
Manganese	17	16	5	4	26	25	19	14
Molybdenum	29	25	28	26	77	66	56	54
Nickel	68	71	66	55	64	67	67	63
Palladium	38	39	39	36	102	102	97	94
Phosphate	3	3	3	2	5	5	5	4
Platinum	38	39	39	35	104	102	101	91
Potash	5	10	10	10	12	27	27	23
Rhodium	29	31	31	29	103	93	102	104
Silver	186	192	181	171	68	69	67	68
Tantalum	0	0	1	0	0	0	47	0
Tin	1	1	3	3	1	1	4	6
Titanium	1	1	1	0	8	8	9	0
Tungsten	20	20	1	1	49	49	4	5
Uranium	16	17	17	14	70	69	66	36
Zinc	107	112	101	96	66	64	59	59

For palladium, platinum and rhodium, %X_{ac} is a little over 100% of the reported global production for some years. There are a couple of possible reasons for this discrepancy - this report focuses on the amount of metal produced at the mine and mill site, where the global figure may report the amount produced at smelters. The difference, which is at most 3%, may also be explained by rounding global production figures.

For some commodities little public information was available on mine production, especially with respect to phosphate, tantalum, tin, and titanium. Tantalum, tin and titanium are generally not produced by public companies or the companies are not based in countries with strong financial reporting requirements. Phosphate is a special case - although there is a large public phosphate mining industry in the US and Canada, the companies do not report ore processed or ore grade, but only phosphate rock produced.

Annual water withdrawal data collection

Water withdrawal data was collected when available from the 65 companies shown in Table 4.1 and the data is summarized in Table 4.4. The number of mines (*aw*) reporting O_k, X_k and W_k was 123 in 2006 and increased to 155 in 2009, for a total of 541 water withdrawal data points over the four year period. In addition, Table 4.4 shows the percentage of global production for each commodity *c* included in dataset *aw* ($\%X_{awc}$). As many of the mines produced multiple commodities, these data points link water withdrawals to 1,174 production figures. For example, the dataset includes the amount of water withdrawn at 92 mines producing gold in 2006, which combined account for 36% of the world's gold production.

Table 4.4 Number of Mines Reporting Water Use

	Number of Mines Reporting Water Use (*aw*)				$\%X_{awc}$, %			
Commodity	**2006**	**2007**	**2008**	**2009**	**2006**	**2007**	**2008**	**2009**
Bauxite	3	3	3	2	29	28	27	25
Chromite	0	0	6	5	0	0	17	17
Cobalt	1	3	5	6	0	4	6	7
Copper (Concentrators)	38	34	26	34	41	21	24	31
Copper (SX-EW)	7	4	5	6	38	12	14	24
Copper (Total)	45	38	31	40				
Diamonds	1	1	5	9	6	7	27	19
Gold	92	101	87	103	36	37	36	33
Iron	4	4	3	5	11	11	2	5
Lead	2	4	6	6	9	11	12	12
Manganese	0	0	0	0	0	0	0	0
Molybdenum	8	3	5	7	31	8	10	16
Nickel	12	17	9	12	1	4	9	9
Palladium	12	16	5	8	18	25	11	16
Phosphate	2	2	2	2	5	5	4	4
Platinum	12	16	5	7	35	50	23	26
Potash	5	5	5	5	12	13	13	9
Rhodium	12	16	5	7	33	43	23	31
Silver	29	26	27	31	13	13	14	15
Tantalum	0	0	0	0	0	0	0	0
Tin	0	0	0	0	0	0	0	0
Titanium	0	0	0	0	0	0	0	0
Tungsten	1	1	1	1	4	5	4	5
Uranium	6	6	6	6	10	11	11	7
Zinc	4	8	10	11	3	5	9	11

No water withdrawal figures were found for manganese, tantalum, tin and titanium. For tungsten, water withdrawals were only found for one mine, and for phosphate, only two mines. As can be seen in Table 4.2, mines producing these commodities also tended not to report detailed production data.

Annual Average Commodity Prices Data Collection

Annual metal prices for most commodities were available from the USGS (2008; 2009; 2010; 2011). Diamond prices were estimated from carat production and revenues reported in annual reports from a few key producers, including: Alrosa, Anglo American, BHP, Debeers, Gem Diamonds, Ponahalo and Rio Tinto. Prices for uranium were collected from the US Department of Energy (2010). P_c values for 2006 to 2009 are shown in Table 4.5. All prices are reported in US dollars per metric tonne ($US/t). For example, the USGS reported that the average annual price of gold in 2006 was US$606 per troy ounce or US$19,483,355/t.

Table 4.5 Average Annual Commodity Prices, P_c

	2006	**2007**	**2008**	**2009**
Commodity	**$US/t**	**$US/t**	**$US/t**	**$US/t**
Bauxite	$28	$31	$26	$30
Chromite	$141	$156	$227	$227
Cobalt	$38,964	$67,351	$86,002	$39,242
Copper	$6,722	$7,117	$6,945	$5,150
Diamonds	$339,986,022	$365,832,785	$410,885,366	$370,889,675
Gold	$19,483,355	$22,473,374	$28,099,756	$31,346,981
Iron	$54	$60	$70	$93
Lead	$1,279	$2,579	$2,090	$1,720
Manganese	$151	$146	$571	$310
Molybdenum	$54,620	$66,790	$62,990	$25,800
Nickel	$24,244	$37,216	$21,104	$14,649
Palladium	$10,382,442	$11,488,749	$11,417,374	$8,540,847
Phosphate	$30	$51	$77	$127
Platinum	$36,793,961	$42,067,327	$50,742,243	$38,823,638
Potash	$375	$400	$675	$835
Rhodium	$146,641,500	$199,433,996	$210,059,176	$51,162,131
Silver	$373,270	$431,785	$482,904	$472,295
Tantalum	$70,548	$79,366	$85,980	$59,525
Tin	$8,774	$14,528	$18,453	$13,558
Titanium	$475	$488	$525	$533
Tungsten	$20,933	$20,807	$20,681	$18,916
Uranium	$41,028	$72,268	$101,148	$101,104
Zinc	$3,274	$3,241	$1,874	$1,656

4.2 Ore Production Method Estimate

As per Equation 1, the total global water withdrawal for each commodity is estimated by multiplying the total amount of ore mined to produce a commodity by the weighted average water withdrawal per tonne of ore used to produce the commodity.

Using Equation 3, the sum of the amount of each commodity produced at each reporting mine was divided by the sum of all ore production at reporting mines producing the commodity in order to calculate the G_{ac}, which was estimated to equal G_{tc}. The mean commodity grades and the resulting estimate of the amount of ore processed to produce the commodities are shown in Table 4.6. The G_{tc} results are shown as percentages, with the exception of diamonds, gold, palladium, platinum, rhodium, and silver, which are shown in grams per tonne (g/t). As mentioned at the start of this section, bauxite, chromite, iron, manganese, phosphate, potash and uranium are based on the amount of concentrate produced, where as the others are based on the amount of metal contained in the concentrate. O_{tc} results are presented in millions of metric tonnes. For example, for the dataset available in 2006 an average of 0.908 g/t of gold were recovered from each tonne of ore processed at the reporting mine site. In 2006, as shown in Table 4.2, 2,370 tonnes of gold was produced from mines globally. Thus, as per Equation 3.2, an estimated 2,610 million tonnes of ore containing gold were processed, including at mines where gold is produced as a by-product. Insufficient data was available to estimate the mean grade of tantalum and titanium for some years.

Table 4.6 Weighted Mean Grade and Total Ore Processed – Ore Production Method

	Weighted Mean Grade (G_{tc})				**Total Global Ore Processed (O_{tc}), Mt**			
Commodity	**2006**	**2007**	**2008**	**2009**	**2006**	**2007**	**2008**	**2009**
Bauxite, %	100	100	100	100	193	204	211	199
Chromite, %	92.8	76.1	76.4	77.6	21	30	32	25
Cobalt, %	0.024	0.020	0.017	0.026	221	261	328	231
Copper (Concentrators), %	0.70	0.69	0.67	0.58	1,757	1,817	1,844	2,177
Copper (SX-EW), %	0.29	0.28	0.26	0.28	958	1,059	1,199	1,165
Diamonds, g/t	0.21	0.15	0.14	0.16	164	223	214	161
Gold, g/t	0.91	0.87	0.84	0.85	2,610	2,713	2,716	2,880
Iron Concentrate, %	76	74	72	84	1,948	2,134	2,340	2,023
Lead, %	1.37	1.23	1.16	1.07	263	300	333	360
Manganese, %	91.8	90.1	92.7	99.9	36	39	41	34
Molybdenum, %	0.022	0.023	0.019	0.019	843	943	1,177	1,177
Nickel, %	0.55	0.57	0.55	0.48	284	291	286	292
Palladium, g/t	1.68	1.64	1.46	1.56	132	134	136	123
Phosphate, %	86.1	88.3	92.3	100.0	175	181	179	166
Platinum, g/t	1.68	1.56	1.44	1.46	129	133	132	124
Potash, %	19.0	17.2	17.9	14.7	164	208	194	141
Rhodium, g/t	0.22	0.21	0.20	0.21	117	132	118	104
Silver, g/t	10.6	10.9	11.0	11.4	1,913	1,928	1,930	1,913
Tantalum, %			0.04				2.99	
Tin, %	1.00	0.76	1.17	1.58	29	39	22	16
Titanium, %	8.39	8.37	8.56		38	39	39	
Tungsten, %	0.27	0.26	0.68	0.79	21.2	21.2	9.1	7.8
Uranium, %	0.08	0.08	0.08	0.05	59	59	63	116
Zinc, %	4.47	4.53	4.27	3.55	231	243	272	316

Using Equations 3.4 and 3.5, the WMO_{awc} for each commodity for each year was calculated and is shown in Table 4.7. WMO_{awc} was estimated to equal WMO_{tc}. Then, using Equation 3.1 and also shown in Table 4.7, the WMO_{tc} results were multiplied by the O_{tc} results from Table 4.6 to calculate the total water withdrawals attributable to each commodity for each year. For example, the WMO_{tc} for gold in 2006 was 0.252 m^3/t of ore processed for all gold producing mines reporting both water and production data. As per Table 4.6, an estimated 2,610 million tonnes of ore were processed to produce 2,370 tonnes of gold in 2006. Thus, an estimated 657 million m^3 of water was withdrawn from mines in 2006 to produce gold. As can be seen in Table 4.7, insufficient data was available to calculate the WMO_{tc} and the W_{tc} for manganese, tantalum, tin, and titanium in all four years and for chromite for 2006 and 2007.

Table 4.7 Weighted Mean Water Withdrawal and Total Global Water Withdrawal Estimate – Ore Production Method

	Mean Water Withdrawal (WMO_{tc}), m^3/t				**Total Global Water Withdrawal (W_{tc}), Mm^3**			
Commodity	**2006**	**2007**	**2008**	**2009**	**2006**	**2007**	**2008**	**2009**
Bauxite	0.347	0.320	0.359	0.395	67	65	76	79
Chromite			8.115	1.340			256	33
Cobalt	0.014	0.060	0.079	0.080	3	16	26	19
Copper (Concentrators)	0.669	0.578	0.594	0.538	1,176	1,051	1,096	1,171
Copper (SX-EW)	0.168	0.172	0.171	0.164	161	182	205	192
Copper (Total)					1,337	1,233	1,301	1,363
Diamonds	3.330	3.191	0.963	1.894	546	713	206	305
Gold	0.252	0.347	0.367	0.396	657	942	997	1,141
Iron	0.303	0.345	0.237	0.436	589	737	555	883
Lead	0.168	0.392	0.089	0.080	44	118	30	29
Manganese								
Molybdenum	0.153	0.108	0.073	0.049	129	102	86	58
Nickel	0.069	0.242	0.515	0.567	20	71	147	165
Palladium	0.057	0.064	0.078	0.113	8	8	11	14
Phosphate	17.372	17.985	17.835	18.388	3,046	3,258	3,187	3,052
Platinum	0.385	0.447	0.738	0.754	50	59	97	94
Potash	0.377	0.292	0.337	0.563	62	61	66	80
Rhodium	0.177	0.236	0.383	0.151	21	31	45	16
Silver	0.011	0.011	0.015	0.021	22	21	28	40
Tantalum								
Tin								
Titanium								
Tungsten	3.056	3.599	3.878	3.463	65	76	35	27
Uranium	0.149	0.210	0.288	0.218	9	12	18	25
Zinc	0.846	1.000	0.300	0.302	195	243	82	95

As shown in Table 4.7, the commodities with the largest water withdrawals were estimated to be phosphate, copper, gold, iron and diamonds. The total amount of water withdrawal estimated for commodities with relevant data available, was from 6.9 to 7.8 billion m^3 per annum.

4.3 Concentrate Production Method Estimate

As per Equation 3.6, the total global water withdrawal for each commodity for the concentrate production method is estimated by multiplying the total amount of commodity *c* produced by the weighted average water withdrawal per tonne of concentrate or metal produced.

Using Equation 3.7, WMC_{awc} for each commodity for each year was calculated and is shown in Table 4.8. WMC_{awc} was estimated to equal WMC_{tc}. Then, using Equation 3.6, the WMC_{tc} results were multiplied by the X_{tc} (shown in Table 4.2) to calculate the total water withdrawals attributable to each commodity for each year. For example, the WMC_{tgold} in 2006 was 309,110 m^3/t of concentrate produced for all gold producing mines reporting both water and production data. As per Table 4.2, approximately 2,370 tonnes of gold was produced in 2006. Thus, an estimated 540 million m^3 of water was withdrawn from mines in 2006 to produce gold. As can be seen in Table 4.8, insufficient data were available to calculate the WMC_{tc} and the W_{tc} for manganese, tantalum, tin, and titanium in all four years, and for chromite for 2006 and 2007.

Table 4.8 Weighted Mean Water Withdrawal and Total Global Water Withdrawal Estimate – Concentrate Production Method

	Mean Water Withdrawal (WMC_{tc}), m^3/t				**Total Global Water Withdrawal (W_{tc}), Mm^3**			
Commodity	**2006**	**2007**	**2008**	**2009**	**2006**	**2007**	**2008**	**2009**
Bauxite	0.347	0.320	0.359	0.395	67	65	76	79
Chromite			8.115	1.652			196	32
Cobalt	364.0	562.9	802.1	598.8	20	30	46	36
Copper (Concentrators)	94.79	99.55	88.51	74.32	1,166	1,254	1,097	936
Copper (SX-EW)	35.12	42.26	45.99	37.90	99	126	142	123
Copper (Total)					1,265	1,380	1,240	1,060
Diamonds	3,948,515	3,206,808	5,037,935	12,928,323	138	109	155	334
Gold	309,110	338,547	395,793	453,305	733	799	906	1,111
Iron	0.307	0.364	0.307	0.452	453	577	514	766
Lead	3.995	8.353	8.129	8.485	14	31	31	33
Manganese								
Molybdenum	797	717	658	382	148	153	144	84
Nickel	138	240	156	214	217	402	243	300
Palladium	56,779	66,243	86,051	108,162	13	15	17	21
Phosphate	17.37	17.99	17.84	18.39	2,623	2,878	2,943	3,052
Platinum	200,400	244,978	389,573	437,462	43	51	74	79
Potash	1.982	1.615	1.745	2.899	62	58	61	60
Rhodium	801,771	1,109,839	1,584,936	601,578	21	31	38	13
Silver	3,107	1,740	2,558	3,128	63	37	54	68
Tantalum								
Tin								
Titanium								
Tungsten	436	463	568	440	25	25	35	27
Uranium	698	872	1,183	1,256	32	42	61	75
Zinc	24.65	27.41	13.37	13.10	254	301	155	147

As shown in Table 4.8, the largest water withdrawals were estimated to be from phosphate, copper, gold, iron and nickel. The total amount of water withdrawals estimated, for commodities with relevant data available, was from 6.2 to 7.4 billion m^3 per annum. This is a somewhat lower range than was estimated using the ore production method.

Table 4.9 Total Global Water Withdrawal Estimate – Concentrate Production Method versus Ore Production Method

	Difference between methods (OPM - CPM), Mm^3				Percentage Difference between methods, %			
Commodity	**2006**	**2007**	**2008**	**2009**	**2006**	**2007**	**2008**	**2009**
Bauxite	0	0	0	0	0	0	0	0
Chromite			61	1			24	4
Cobalt	-16	-14	-20	-17	-521	-91	-78	-93
Copper (Concentrators)	10	-203	-1	235	1	-19	0	20
Copper (SX-EW)	61	57	63	68	38	31	31	36
Copper (Total)	72	-146	61	303	5	-12	5	22
Diamonds	408	604	51	-28	75	85	25	-9
Gold	-76	143	91	30	-12	15	9	3
Iron	136	160	41	117	23	22	7	13
Lead	30	87	-2	-4	67	74	-6	-14
Manganese								
Molybdenum	-19	-51	-57	-27	-15	-50	-66	-47
Nickel	-197	-331	-95	-135	-1,005	-469	-65	-82
Palladium	-5	-6	-6	-7	-67	-71	-61	-49
Phosphate	423	380	244	0	14	12	8	0
Platinum	6	9	23	14	13	14	24	15
Potash	0	3	5	19	0	5	8	24
Rhodium	0	0	7	2	-1	0	16	14
Silver	-41	-15	-26	-28	-189	-70	-92	-71
Tantalum								
Tin								
Titanium								
Tungsten	40	51	0	0	62	67	0	0
Uranium	-24	-30	-43	-50	-271	-242	-236	-198
Zinc	-59	-58	-74	-51	-30	-24	-90	-54

Table 4.9 shows the differences between the W_{tc} calculated by the Ore Production Method and the Concentrate Production Method, in addition to the percentage difference. The first set of data is the W_{tc} results from the Ore Production Method less the *Wtc* results from the Concentrate Production Method (Mm^3). The second set calculates the percentage difference by dividing the

difference with the W_{tc} results from the Ore Production Method. As can be seen, the differences range from zero to hundreds of millions of m^3 per annum.

4.4 Water Withdrawal Data Summary

This section provides an un-weighted statistical summary of water withdrawal datasets by commodity for both the ore and concentrate production methods. The global water withdrawal estimates for both methods are based on weighting water withdrawals by the amount of commodity produced at mine sites. A summary of the un-weighted data is provided in Table 4.10 and Table 4.11, along with the weighted data shown in Table 4.7 and Table 4.8 respectively. As can be seen, the arithmetic mean is often different from the weighted mean, indicating that a number of smaller producers have water withdrawals that differ significantly from those of the major producers. This difference shows the importance of weighting the data when estimating global water withdrawals.

Table 4.10 Summary of Water Withdrawal by Ore Production Dataset

	Weighted Water Withdrawals (WMO), m^3/t				**Water Withdrawal Datasets (WMO), m^3/t**				
Commodity	**2006**	**2007**	**2008**	**2009**	**#**	**Nom.**	**Min.**	**Max.**	**St. Dev.**
Bauxite	0.347	0.320	0.359	0.395	11	0.404	0.022	1.154	0.521
Chromite			8.115	1.340	11	10.958	0.150	75.260	22.336
Cobalt	0.014	0.060	0.079	0.080	10	0.028	0.006	0.093	0.004
Copper (Concentrators)	0.669	0.578	0.594	0.538	132	0.521	0.000	3.065	0.661
Copper (SX-EW)	0.168	0.172	0.171	0.164	22	0.220	0.100	0.432	0.100
Diamonds	3.330	3.191	0.963	1.894	16	9.926	0.174	113.833	27.922
Gold	0.252	0.347	0.367	0.396	383	0.745	0.003	10.900	1.206
Iron	0.303	0.345	0.237	0.436	16	0.588	0.094	3.000	0.889
Lead	0.168	0.392	0.089	0.080	18	0.693	0.002	5.330	1.242
Manganese					0				
Moly	0.153	0.108	0.073	0.049	23	0.103	0.005	0.373	0.088
Nickel	0.069	0.242	0.515	0.567	45	0.162	0.000	1.490	0.085
Palladium	0.057	0.064	0.078	0.113	41	0.083	0.007	0.371	0.040
Phosphate	17.372	17.985	17.835	18.388	8	17.414	13.160	21.795	3.430
Platinum	0.385	0.447	0.738	0.754	40	0.498	0.017	1.446	0.273
Potash	0.377	0.292	0.337	0.563	20	0.445	0.105	1.250	0.280
Rhodium	0.177	0.236	0.383	0.151	40	0.231	0.049	0.749	0.156
Silver	0.011	0.011	0.015	0.021	113	0.052	0.000	0.331	0.080
Tantalum					0				
Tin					0				
Titanium					0				
Tungsten	3.056	3.599	3.878	3.463	4	3.499	3.056	3.878	0.342
Uranium	0.149	0.210	0.288	0.218	24	0.113	0.015	0.449	0.127
Zinc	0.846	1.000	0.300	0.302	33	1.681	0.003	19.928	3.703

Table 4.11 Summary of Water Withdrawal by Concentrate Production Dataset Data Accuracy

	Weighted Water Withdrawal (WMC), m^3/t				**Water Withdrawal Datasets (WMC), m^3/t**				
Commodity	**2006**	**2007**	**2008**	**2009**	**#**	**Nom.**	**Min.**	**Max.**	**St. Dev.**
Bauxite	0.347	0.320	0.359	0.395	11	0.404	0.022	1.15	0.521
Chromite			8.115	1.652	11	10.99	0.15	75.26	22.32
Cobalt	364.0	562.9	802.1	598.8	15	548	71	1,099	258
Copper (Concentrators)	94.79	99.55	88.51	74.32	128	88.03	0.013	402.61	79.76
Copper (SX-EW)	35.12	42.26	45.99	37.90	22	48.01	27.77	96.18	19.40
Diamonds (10^6)	3.949	3.207	5.038	12.928	16	78.21	2.01	632.41	159.55
Gold (10^3)	309	339	396	453	379	400	0.61	4,742	603
Iron	0.307	0.364	0.307	0.452	16	0.598	0.094	3.00	0.880
Lead	3.995	8.353	8.129	8.485	18	21.85	2.93	131.53	32.70
Manganese									
Moly	797	717	658	382	23	706	151	1,812	463
Nickel	138	240	156	214	46	189	0	827	149
Palladium (10^3)	56.8	66.2	86.1	108	37	81.9	15.5	255.3	58.3
Phosphate	17.37	17.99	17.84	18.39	8	17.41	13.16	21.79	3.43
Platinum (10^3)	200	245	390	437	36	302	70.7	935	184
Potash	1.982	1.615	1.745	2.899	20	2.35	0.653	5.42	1.38
Rhodium (10^3)	801	1,109	1,584	601	36	1,168	307	4,432	865
Silver	3,107	1,740	2,558	3,128	113	5,440	44	55,569	7,516
Tantalum									
Tin									
Titanium									
Tungsten	436	463	568	440	4	477	436	568	61.79
Uranium	698	872	1,183	1,256	24	1,451	226	7,690	2,076
Zinc	24.65	27.41	13.37	13.10	33	26.76	2.82	126.64	29.65

Northey *et al.* (2012) published a comparable dataset on copper producers, weighted to account for the production of byproducts. Their database includes 31 copper producing operations from 1991 to 2010. Figure 4.1 summarizes their results by comparing water intensity (kL H_20/t Cu, or m^3 water per tonne of Cu metal) by copper production (kilotonnes (1,000 t) of copper metal).

Figure 4.1 Water Intensity as a Function of Copper Production (Northey et al., 2012)

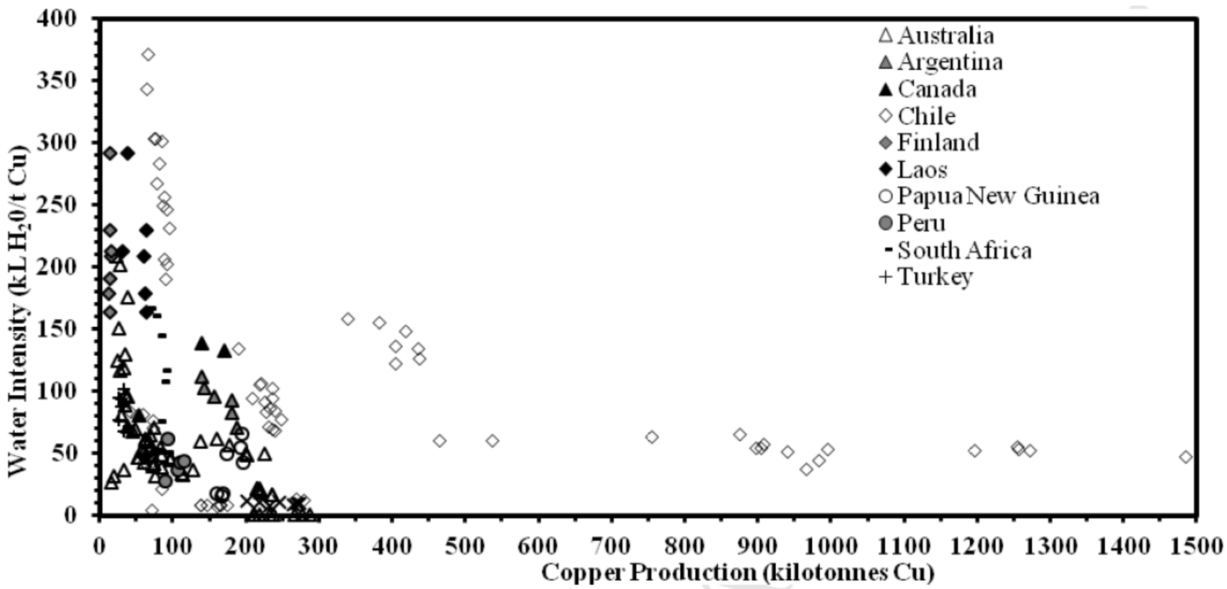


Figure 4.2 Water Intensity as a Function of Copper Production

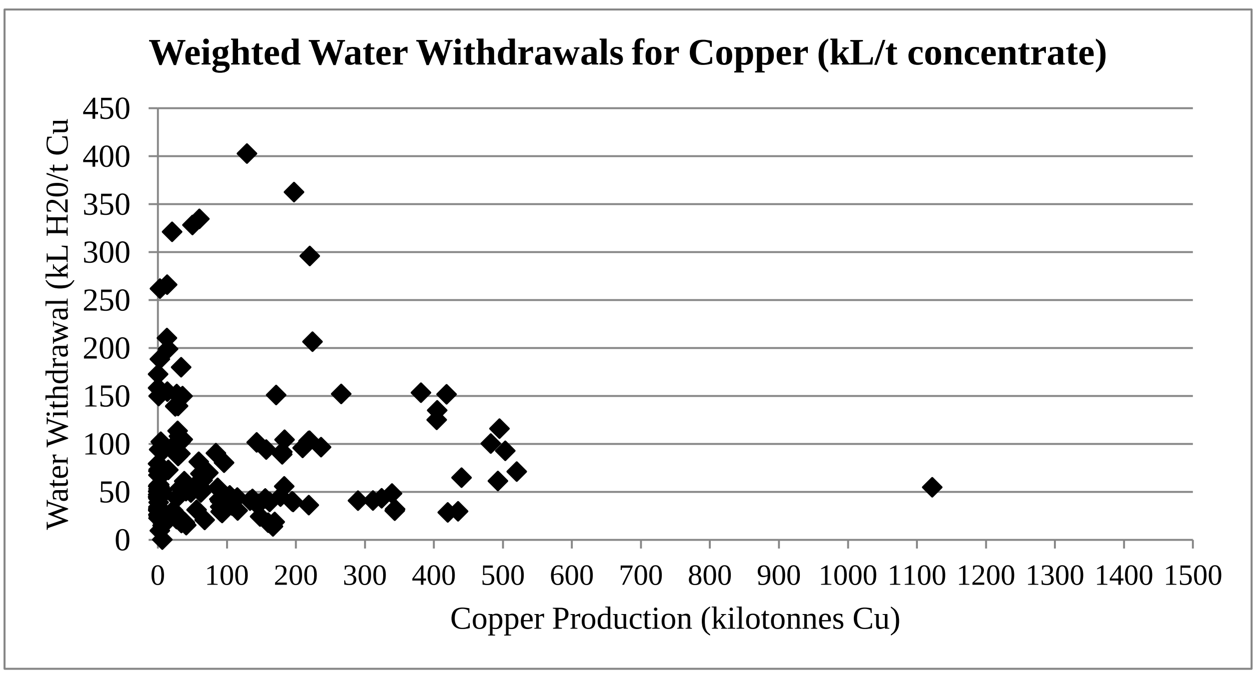


Figure 4.2 reports the data from this study in the same format. This data is sourced from 31 to 45 mines per year from 2006 to 2009. As can be seen, the two graphs are fairly similar. Northey *et al.* (2012) have a larger number of data points beyond 500,000 t/y copper production, as their

database has data from the world's two largest copper producers (Escondita and Chuquicamata) over a longer time period.

4.5 Data Accuracy, Representativity and Analysis

The amount, accuracy and representativity of the data available vary significantly from commodity to commodity and from year to year. The accuracy of the Ore Production and Concentrate Production methods for estimating global water withdrawals depends on the accuracy and representativeness of the inputs into the estimate, specifically:

1. The global production of commodity c (X_{tc});
2. The amount of commodity c produced for each mine (X_{kc});
3. The amount of ore processed at each mine (O_k);
4. The water withdrawal from each mine (W_k); and
5. The price of commodity c (P_c).

Section 4.5.1 discusses the accuracy of the data, Section 4.5.2, discusses the representativity of the data, and Section 4.5.3 describes a qualitative approach to determining the representativity and thus the accuracy of the estimates.

4.5.1 Data Accuracy

This section describes the accuracy of the data used in the water withdrawal estimates. The accuracy is categorized based on a subjective evaluation of the source of the data and ranges from relatively high to very high. The reasoning behind the categorization is discussed for each variable.

The accuracy of the total global production of commodity c per year (X_{tc}) is likely relatively high to high, depending on the commodity (±5% or ±10%). The USGS and the USDOE have been following global production for most commodities for decades and appear to be the most reliable source of global commodity production available. However, these agencies are limited in their ability to estimate production from non-transparent nations, such as North Korea, or from diffuse and/or illegal mining operations, particularly artisanal and small-scale mining activities.

One limited indication of the accuracy of X_{tc} is to compare the USGS and USDOE data for 2009 to the RMD data (2011), as shown in Table 4.12. Much of the information used by the USGS, USDOE and RMD likely comes from the same or similar sources; therefore, it should be no surprise if there is little variability between the results. The variability (VX_{tc}) is calculated as shown in Equation 4.1, where X_{tcRMD} is the global production of commodity *c* as reported by RMD in January 2011.

$$VX_{tc} = \frac{|X_{tcRMD} - X_{tc}|}{X_{tc}} \cdot 100 \qquad [4.1]$$

where VX_{tc} = the variability between the global production of commodity *c* as reported by RMD and the global production of commodity *c* as reported by the USGS and USDOE, [%]; X_{tcRMD} = the global production of commodity *c* as reported by RMD in January 2011, [t/y]; and X_{tc} = total global production of commodity *c*, [t/y] as reported by the USGS and USDOE. A VX_{tc} value of 0.0% indicates the RMD and USGS or USDOE commodity production estimates are identical.

The accuracy of the amount of concentrate or commodity produced for each mine (X_{kc}) is likely extremely high (±1%). Most of these data are provided by publically traded companies who are legally required to accurately report their production and revenues. In addition, mines' products are usually purchased by second parties and both the producers and the purchasers have significant interest in ensuring the accuracy of the metal and mineral content in concentrates sold.

The accuracy of the amount of ore processed at each mine (O_k) is likely high (±5%). Mines have considerable interest in knowing the exact amount of ore mined and processed on an hourly and daily basis in order to estimate their mining and processing costs and ore reserve depletion. However, the accuracy of weightometers, flow meters and other instrumentation that account for ore tonnages is not perfect. While mines regularly reconcile this data to get the most accurate estimates possible, the data accuracy will be less than that of the concentrate produced. However, for the purposes of this estimate, the level of inaccuracy is likely negligible.

Table 4.12 Variability of Global Production Data, VX_{tc}

	VX_{tc}, %			
Commodity	**2006**	**2007**	**2008**	**2009**
Bauxite	0.5	2.0	2.8	3.0
Chromite	0.0	0.4	0.4	19.2
Cobalt	0.4	0.6	32.9	47.2
Copper (Concentrators)	0.0	0.0	0.0	0.0
Copper (SX-EW)	2.1	2.0	1.6	0.0
Copper (Total)	2.1	2.0	1.6	0.0
Diamonds	0.6	1.2	5.8	3.1
Gold	0.0	0.4	1.3	2.0
Iron	0.0	0.0	0.0	0.0
Lead	2.5	2.4	1.0	3.6
Manganese	0.6	2.6	0.3	10.7
Molybdenum	0.5	0.0	0.0	0.9
Nickel	4.5	4.5	3.8	4.3
Palladium	1.4	2.7	0.5	1.0
Phosphate	0.7	1.9	2.4	4.8
Platinum	3.2	3.4	3.2	2.2
Potash	0.6	2.2	0.9	20.2
Rhodium	0.0	0.0	0.0	7.6
Silver	0.5	1.4	0.0	0.9
Tantalum	0.0	0.0	0.0	0.0
Tin	15.1	16.3	23.3	21.2
Titanium	0.0	0.0	0.0	0.0
Tungsten	1.1	1.7	11.6	10.3
Uranium	1.0	0.7	0.5	0.2
Zinc	1.0	0.9	0.9	1.8

The accuracy of the water withdrawal from each mine (W_k) is likely relatively high (±10%) (Department of Water Affairs & Forestry, 2006a). Water withdrawals are determined by flow meters on water supply pipelines where available and/or water balance estimates calculated by mines. Modern flow meters can be extremely accurate (±1% or less) under good conditions (Emerson Process Management, 2007). However, the actual accuracy is typically lower due to operational factors, such as maintaining full pipe flow through the meter. Many mines, especially at older operations, do not have any flow meters installed. Fundamentally, most mines have little financial incentive to ensure the accuracy of their water withdrawal and these figures are not typically independently verified by third parties to a high level of accuracy, such as in the case of X_k. Furthermore, as discussed in Section 2.1 and Section 2.3, there can be inconsistencies in the definitions used by mines to report water withdrawals.

The accuracy of P_c is likely relatively high to very high, depending on the commodity (±1% or ±10%). For a commodity such as gold or copper, trading is open and pricing is transparent. The USGS annual prices for publically traded commodities are likely highly accurate (±1% or better). For bulk or specialty commodities pricing is far more variable and less transparent. For bulk commodities, such as iron or bauxite, shipping costs have a significant impact on pricing and thus a global average price is less meaningful. For a specialty commodity, such as tantalum, purchase contracts are confidential and price estimates are less reliable (±10%).

4.5.2 Data Representativity

While the accuracy of the collected data subjectively ranges from relatively high to extremely high, as discussed in Section 4.5.1, the accuracy of the calculated water withdrawal estimates is likely considerably lower, particularly with respect to X_{kc}, O_k, and W_k. As discussed in, X_{tc} and P_c are estimates of global production and price by institutions with dedicated staff and long histories for estimating these values. For the purposes of this study they are considered to be highly representative.

As discussed in Section 3.1, the ore production method estimates water withdrawals by multiplying the amount of ore processed globally to produce a commodity (O_{tc}) by the weighted global mean water withdrawal recorded per unit of ore (WMO_{tc}). O_{tc} is calculated as per Equation 3.2, by dividing X_{tc} by G_{tc}. The weighted mean grade with respect to the mines reporting production data, or G_{ac}, is directly calculated from the X_{kc} and O_k and is likely highly accurate (±5%). However, in Equation 3.2, G_{ac} is assumed to estimate average global commodity grade G_{tc}. This assumption introduces the potential for significant uncertainty, as it assumes that the mean grade of all the mines reporting production data is equal to the mean grade of all producing mines globally.

For some commodities, such as the platinum group metals, essentially all of global production is included in the *a* dataset and G_{ac} can be assumed to equal G_{tc}. For other commodities, such as tin or titanium, the dataset *a* includes less than 10% of global production, and thus X_{atc} may be significantly different than X_{tc}. Thus the accuracy of the estimate depends on the representativity of the *a* dataset.

Similarly, as the accuracy for W_k and P_c is likely relatively to very highly accurate (±1-10%), the accuracy of W_{kc} for any mine *k* should also be relatively accurate, as should be the accuracy for WMO_{awc}. However, in Equation 1 WMO_{tc} is assumed to equal WMO_{awc}, so that the accuracy of the estimate depends on the representativeness of the *aw* dataset.

Consequently, the representativeness of X_{kc}, O_k , and W_k depends on the representativeness of the *a* and *aw* datasets. Organizations that report their mines' production data (the *a* dataset) are most likely large public companies in open markets with significant incentives to be profitable and transparent. These companies typically run larger operations, which means that the *a* dataset is likely not representative of smaller mines, mines from more closed countries (such as China), or mines producing commodities in less transparent markets. Smaller mines are likely to have higher grades, as it is especially challenging to profitably run a small tonnage, low-grade operations, and so the *a* dataset is likely biased toward larger lower grade mines. Mines reporting water withdrawals (the *aw* dataset) are more likely to be owned by public companies than have water management programs and actively work to reduce their water withdrawals. Thus, the *aw* dataset likely has a bias toward larger, lower grade mines that are actively working to reduce their water withdrawals, and may have lower water withdrawals per tonne of concentrate or ore produced than non-reporting mines. Furthermore, there is little to no production or water withdrawal information available on mines that are not in these datasets.

As the total global production of commodity c (X_{tc}) is known to a reasonable degree of accuracy, one approach to determining the representativeness of the datasets is by evaluating the percentage of global production represented by the dataset. Table 4.3 and Table 4.4 indicate this percentage on a commodity-by-commodity basis. Even if the dataset is not representative, if it accounts for a large portion of global production of a commodity, the accuracy of the estimate of global water withdrawal due to a commodity is likely high. For example, if the *aw* dataset for a commodity contained 10 mines producing 95% of the global commodity production, one can assume the missing mines producing the remaining 5% of global production use proportionately the same amount of water, and thus estimate global water withdrawals for the commodity. However, the 10 mines may not to be representative of all the mines producing the commodity. It is possible that there would be a larger number of small mines withdrawing more water per tonne of ore or concentrate produced than the 10 major mines. However, the amount of water used by the small mines would have to be staggeringly large in proportion to the 10 reporting mines to create a significant shift in the estimate. It is also possible that all of the remaining production comes from one or two large mines using less water than the other 10; however, even if they withdrew no water they would not have a large impact on the estimate. It is reasonable to assume that mines producing the remaining 5% of commodity will not consume so much more or less water as to significantly change the estimate of the global amount of water withdrawn to produce the commodity.

Three indirect methods of measuring the representativeness of the *a* and *aw* dataset are to compare the difference between the weighted average grades of the *a* and *aw* datasets, to compare the variation from year to year within the two datasets, and to compare the revenue generated per unit of water withdrawn per commodity per year.

The differences between the weighted average grades are calculated as follows:

$$VG_{tc} = \frac{|G_{ac} - G_{awc}|}{G_{ac}} \cdot 100 \qquad [4.3]$$

where VG_{tc} = the variability between the global weighted mean recovered ore grade of commodity *c* of the *a* dataset and the *aw* dataset [%]; G_{ac} = the global weighted mean recovered ore grade of commodity *c* from the set of mines *a* [% mass]; and G_{awc} = the global weighted mean recovered ore grade of commodity *c* from the set of mines *aw* [% mass]. A VG_{tc} value of 0.0% indicates G_{ac} and G_{awc} are equal.

The variability of the grade (VG_{ac}, in %) within G_{ac} for each year from 2006 to 2009 is calculated by taking the difference between the largest and smallest grade and dividing it by the mean grade. The variability of the grade (VG_{awc}, in %) within G_{awc} for each year from 2006 to 2009 is calculated by taking the difference between the largest and smallest grade and dividing it by the mean grade. A VG_{ac} or VG_{awc} value of 0.0% indicates that the ore grades were equal for each year from 2006 to 2009. The results are shown in Table 4.13.

Table 4.13 Variability of Grades, VG_{tc}, VG_{ac} *and* VG_{awc}

	VG_{tc}					
Commodity	**2006**	**2007**	**2008**	**2009**	VG_{ac}	VG_{awc}
Bauxite	0.0	0.0	0.0	0.0	0.0	0.0
Chromite			31.0	4.5	20.7	
Cobalt	83.9	47.7	43.7	48.3	38.4	100.2
Copper (Concentrators)	0.9	16.2	0.1	25.0	18.3	21.3
Copper (SX-EW)	62.1	45.1	44.2	55.5	13.1	24.9
Diamonds	295.1	553.8	32.7	8.5	41.5	156.0
Gold	10.4	17.9	10.0	2.7	7.5	23.3
Iron	30.1	27.6	8.1	15.3	16.0	23.0
Lead	207.2	281.4	5.8	12.3	24.2	137.2
Manganese					10.5	
Molybdenum	13.0	33.2	39.8	31.9	19.8	55.3
Nickel	90.9	82.4	39.2	44.9	17.4	150.5
Palladium	40.0	41.4	37.9	33.1	13.9	14.0
Phosphate	16.1	13.2	8.3	0.0	15.2	0.0
Platinum	14.5	16.7	31.9	18.3	15.7	10.6
Potash	0.0	5.2	8.2	31.9	24.8	7.2
Rhodium	0.7	0.3	19.3	16.8	9.1	16.2
Silver	65.4	41.2	47.9	41.6	7.1	53.1
Tantalum					0.0	
Tin					72.2	
Titanium					2.3	
Tungsten	162.6	205.1	0.0	0.0	107.0	14.2
Uranium	73.1	70.8	70.2	66.4	41.7	32.2
Zinc	23.2	19.4	47.4	35.0	23.3	48.4

As can be seen, there is often considerable variation between the grades in the *a* and *aw* datasets. The most significant example is the diamond grade in 2007, which is over 550% higher in the *aw* dataset than in the *a* dataset, then drops to 32% in 2008. This indicates that the *aw* dataset is not representative of the *a* dataset and is likely not a good representation of diamond production or water withdrawals in 2007. The $W_{tdiamonds}$ drops from 713 million m^3 in 2007 to

206 million m^3 in 2008. Thus the 2008 estimate is likely more accurate than the 2007 estimate. However, in 2009 the VG_{tc} drops from 8.5% in 2009 and the $W_{tdiamonds}$ rises to 305 million m^3. Thus, while the amount of production has dropped in the *a* and *aw* datasets, there is little difference between the weighted grades of the ore, and the water estimate is not necessarily less accurate than in 2008.

An additional method of analyzing the variability of the data is to review the amount of revenue generated per m^3 of water withdrawn. The amount of revenue generated per unit of water withdrawn may be high or low at any given mine, varying with the ore grade, water availability and process used. However, the average amount of revenue generated per unit of water withdrawn for a commodity on a global basis should tend to be in the same order of magnitude as for other mined commodities. Total revenue per mine was determined using Equation 3.5 to estimate the WMO_{tc}. Applying the equation, the amount of revenue associated with each m^3 of water withdrawn can be calculated by dividing the total annual revenue generated per commodity by the W_{tc}. As shown in Table 4.14, the range of revenue generated per m^3 of water withdrawn is large, swinging from 0.07 \$/ m^3 to 1,246.62 \$/ m^3. However, most of the RO_c and RC_c values are in the range of 20 US\$/m^3 to 300 US\$/m^3; values significantly above or below this range may indicate the estimate is less accurate. Similarly, large swings in RO_c and RC_c for the same commodity in different years may indicate inaccuracy in the Wt_c estimate.

Table 4.14 Revenue Associated With Water Withdrawal

	Average Revenue \$/m^3 - RO_c				Average Revenue \$/m^3 - RC_c			
Commodity	**2006**	**2007**	**2008**	**2009**	**2006**	**2007**	**2008**	**2009**
Bauxite	63.67	73.96	57.30	59.23	63.67	73.96	57.30	59.23
Chromite			9.25	46.74			12.11	48.85
Cobalt	259.34	85.92	56.25	48.90	41.74	44.97	31.66	25.27
Copper (Concentrators)	59.29	71.79	66.19	45.30	59.82	60.17	66.10	56.65
Copper (SX-EW)	91.70	83.76	60.35	46.34	148.60	121.58	87.00	72.05
Copper (Total)								
Diamonds	12.17	9.50	33.96	12.60	48.08	62.09	45.06	11.53
Gold	51.82	41.00	45.15	45.00	46.44	48.34	49.68	46.23
Iron	80.41	74.94	138.32	82.63	104.61	95.65	149.46	95.23
Lead	61.31	41.88	128.86	107.85	188.36	159.72	121.39	94.62
Manganese								
Molybdenum	60.74	92.79	89.59	53.62	52.85	61.97	53.95	36.50
Nickel	1,246.62	591.42	150.02	78.51	112.83	104.00	91.13	43.25
Palladium	311.77	300.84	208.04	110.31	187.13	176.20	129.25	73.84
Phosphate	0.08	0.13	0.19	0.07	0.10	0.14	0.21	0.07
Platinum	167.37	150.10	99.50	68.15	191.60	175.15	131.21	80.61
Potash	21.77	64.28	97.02	49.46	21.77	67.60	104.98	65.22
Rhodium	189.28	166.91	113.66	75.39	187.90	167.34	135.56	88.07
Silver	237.67	292.52	243.12	175.25	82.25	172.08	126.62	102.41
Tantalum								
Tin								
Titanium								
Tungsten	8.94	7.17	1.60	2.21	23.48	21.87	1.60	2.21
Uranium	153.00	196.73	188.65	85.89	41.22	57.49	56.18	28.83
Zinc	113.45	94.56	156.20	114.46	87.10	76.24	82.14	74.37

For example, gold, which has one of the largest datasets, has a fairly tight range of revenue from \$41/m^3 to \$52/m^3. However, the RO_{Nickel} was \$1247/m^3 in 2006 and \$591/m^3 in 2007 before dropping to \$150/m^3 and \$79/m^3. The RC_{Nickel} ranged from \$112.83/m^3 to \$43.25/m^3. Table 4.4 indicated that the $\%X_{awnickel}$, or the percentage of global nickel production represented by the *aw* dataset in 2006 was 1%. From 2007 to 2009 the $\%X_{awnickel}$ increased to 9%. The extremely high RO_{Nickel} in 2006 thus indicates that the data for nickel in 2006 was likely not representative.

4.5.3 Qualitative Review of the Representativity of the Water Withdrawal Estimates

The objective of this section is to determine the representativity, and thus potential accuracy, of the estimates of global water withdrawals. For any given commodity, if the *aw* dataset was complete (contained water withdrawal data from all mines producing the commodity) or if it was representative of the mines, then the estimate could be considered relatively accurate. The only errors would be from inaccuracies in the mine's measurements of water withdrawals, which as discussed in Section 4.5.1, are expected to have a relatively high level of accuracy.

However, as discussed in Section 4.5.2, the representativity of the water withdrawal estimate is primarily based on the amount of missing data (mines not reporting water withdrawals) from the *aw* dataset. The key difference between the Ore Production Method and the Concentrate Production Method is that the Ore Production Method takes into consideration the ore grade of the dataset reporting water withdrawals and the ore grade of a larger, more representative dataset to help account for a potential lack of representativeness in the first set. For the Concentrate Production Method the accuracy only depends on the representativity of the *aw* dataset. In the Ore Production Method, the more complete *a* dataset is used to modify the estimate in an attempt to increase its accuracy. For less complete datasets the confidence in the accuracy of the estimate will be smaller.

Statistically, both the Ore Production Method and the Concentrate Production Methods of estimating global water withdrawals presents a problem of missing data. It is possible to calculate the variance, standard deviation or other factors for *a* and *aw* datasets, as was shown in Section 4.4. However, both the *a* and *aw* datasets are known to be incomplete and biased, and in most cases it is unknown how incomplete or biased the datasets are. The water withdrawal estimate problem is relatively complex due to the biased nature of the *a* and *aw* sample groups and the unknown number of missing mines.

There is a large body of literature available on statistical methods of accounting for missing data by using various algorithms and assumptions (Honaker & King, 2010; Osborne, 2012). Honaker and King (2010) describe a methodology using predictive models that incorporates any available information in the observed data together along with any prior knowledge to fill in holes in datasets. Arguably this is similar to the approach used in the Ore Production Method, where the observed data from the *a* dataset was used to modify the *aw* dataset in an effort to make the estimate more realistic.

A simpler, mainly qualitative method was chosen to provide an indication of the accuracy of the two estimation methods. Due to the level of uncertainty in many of the variables, the purpose of the qualitative method is to determine if the water withdrawal estimate is in the right order of magnitude. Four subjective indicators were chosen to evaluate each commodity: Each variable was evaluated using an assigned qualitative ranking from 0-4, with 0 representing no confidence in accuracy of the estimate and 4 representing a high level of confidence in the accuracy.

The variables for the qualitative ranking are as follows:

1. The percentage of global production of commodity *c* accounted for in the dataset of mines reporting ore and commodity production and water withdrawals (dataset *aw*), $\%X_{awc}$. For both methods, this is the most important variable with respect to representativeness;
2. The percentage of global production of commodity *c* accounted for in the dataset of mines reporting ore and commodity production (dataset *a*), $\%X_{ac}$. For the Ore Production Method, this variable compensates for the less complete *aw* dataset;

3. The variability between the global weighted mean recovered ore grade of commodity *c* of the *a* dataset and the *aw* dataset, VG_{tc}. If this variable is high, it indicates a significant difference between the mines in dataset *a* and the smaller group of mines in *aw*. This suggests that the *aw* dataset is not representative of most of the mines producing commodity *c*. It is an indicator of the suitability of using the ore concentration method.
4. The mean revenue generated from water withdrawals using the ore production or concentrate method, RO_c or RC_c respectively. In most cases, RO_c and RC_c range from 30 to 200 $\$/m^3$. If the RO_c or RC_c value is significantly different from this range, it suggests that the calculated WMC_{awc} or the WMO_{awc} is not reasonable.

The confidence ranges for the variables are shown in Table 4.15 below.

Table 4.15 Qualitative Ranking of Representativity of Variables

$\%X_{ac}$	$\%X_{awc}$	VG_{tc}	$R0_c$ **or** RC_c	**Description**	**Value**
50+	50+	0-25	30-200	High	4
35-50	35-50	25-50	10-30, 200-400	Moderate	3
15-35	15-35	50-100	1-10, 500-1000	Low	2
<15	<15	100-200	0.1-1, 1000-2000	Poor	1
0	0	>200	<0.1, >2000	None	0

The weighting of the importance of the different variables is shown below in Table 4.16. The value of the weights has been subjectively assigned. For the Ore Production Method, $\%X_{awc}$ has been assigned a weighting of 50%, as the completeness of the *aw* dataset is clearly the most important variable. However, $\%X_{ac}$ has been assigned a weighting of 30%, indicating that even if the *aw* dataset comprises a relatively small portion of the production of a commodity, if the *a* dataset is large, the estimate may still be accurate. Likewise, VG_{tc} and RO_c have been assigned a value of 10% each. For example, if the $\%X_{awc}$ equaled only 20% of global production for a commodity, if $\%X_{ac}$ was over 50%, VG_{tc} was below 25%, and RO_c was between \$30/m^3 and \$200/m^3, qualitatively, the estimate may still be representative. In other words, even if the portion of production from mines reporting water withdrawal is small, if the average grade of mines that did report appears to be representative of the majority of mines producing the commodity and the revenue generated per m^3 of water withdrawn is in the normal range, it is reasonable to assume the estimate may be representative. For the Concentrate Production Method, $\%X_{ac}$ and VG_{tc} are not relevant to the estimate and are thus assigned a weighting of 0%. $\%X_{awc}$ is assigned a weighting of 90%, as the method overwhelmingly depends on the representativity of the *aw* dataset for the commodity. RC_c is assigned a value of 10%; if the revenue generated per m^3 of water withdrawn is found to be in the normal range, it is possible that even if the *aw* dataset is small, it may still be representative. Any commodity with no data in the *aw* dataset is assumed to have a value of 0, indicating there is no confidence in the estimate of water withdrawn due to the commodity.

Table 4.16 Qualitative Weighting of Representativity of Variables

Ore Production Method		**Concentrate Method**	
Variable	Weighting	Variable	Weighting
$\%X_{ac}$	30%	$\%X_{ac}$	0%
$\%X_{awc}$	50%	$\%X_{awc}$	90%
VG_{tc}	10%	VG_{tc}	0%
$R0_c$	10%	RC_c	10%

An example of how Table 4.15 and Table 4.16 are used is as follows: 92 mines producing gold reported water withdrawals in 2006, representing the *aw* dataset. These mines accounted for 36% of global gold production in 2006 ($\%X_{awgold}$). For the purposes of this evaluation, a $\%X_{awc}$ value of 36% is a moderately accurate estimate of water withdrawals due to gold mining in 2006, and is assigned a qualitative value of 3. The average revenue generated by water withdrawn was \$46/m^3, within the high range of representativity, and is thus assigned a value of 4. For the Concentrate Method, weighting the variables as per Table 4.16 leads to the following calculation: 3 x 90% + 4 x 10% = 3.1. A value of 3 indicates a moderate degree of confidence that the estimate of water withdrawals due to gold mining in 2006 is in the right order of magnitude.

Table 4.17 presents the results of the qualitative analysis of the representativity of the water withdrawal estimate by commodity. As can be seen, the confidence in the Ore Production Method is typically somewhat higher than for the Concentrate Production Method. The qualitative analysis places more confidence in the representativity of the estimate calculated from the Ore Production Method, as the smaller *aw* dataset is compensated for by the larger and likely more representative X_{ac} dataset.

Table 4.17 Qualitative Analysis of the Representativity of the Water Withdrawal Estimates

	Ore Method				**Concentrate Method**			
Commodity	**2006**	**2007**	**2008**	**2009**	**2006**	**2007**	**2008**	**2009**
Bauxite	3	3	3	3	2	2	2	2
Chromite	0	0	2	3	0	0	2	2
Cobalt	2	2	2	2	1	1	1	1
Copper (Concentrators)	4	3	3	3	3	2	2	2
Copper (SX-EW)	3	2	2	3	3	1	1	2
Diamonds	2	2	3	3	1	1	2	2
Gold	4	4	4	3	3	3	3	2
Iron	2	2	3	2	1	1	1	1
Lead	2	2	2	2	1	1	1	1
Manganese	0	0	0	0	0	0	0	0
Moly	3	2	2	3	2	1	1	2
Nickel	2	2	2	2	1	1	1	1
Palladium	3	3	2	3	2	2	1	2
Phosphate	1	1	1	1	1	1	1	1
Platinum	3	4	3	3	2	3	2	2
Potash	2	2	2	2	1	1	1	1
Rhodium	3	4	3	3	2	3	2	2
Silver	2	2	2	2	1	1	1	1
Tantalum	0	0	0	0	0	0	0	0
Tin	0	0	0	0	0	0	0	0
Titanium	0	0	0	0	0	0	0	0
Tungsten	2	2	1	1	1	1	1	1
Uranium	2	2	2	2	1	1	1	1
Zinc	3	3	2	2	1	1	1	1

As can be seen in Table 4.17, the qualitative analysis found that representativity of the water withdrawal estimates for commodities such as gold and copper, with relatively large *a* and *aw* datasets, were generally high or moderate. Thus it may be reasonable to assume the withdrawal estimate was relatively accurate. However, water withdrawal estimate for commodities such as phosphate and tungsten, which had low *a* and *aw* datasets, was found to have poor representativity. While the estimate for phosphate and tungsten may still be fairly accurate, there is a reasonable probability they are not.

4.6 Data Sensitivity

Section 4.5 estimated the accuracy of the datasets used for the water withdrawal estimate, the certainty that the data was representative, and undertook a qualitative analysis of the representativity of the estimate. This section measures sensitivity of the estimate to changes in the concentrate production, ore grade, mean water withdrawal, additional mine data and price.

4.6.1 Sensitivity to Concentrate Production, Grade, and Mean Water Withdrawal Estimates

The sensitivity of the W_{tc} to changes in estimates of X_{tc}, WMC_{tc} and WMO_{tc} are linear, as per Equations 3.1, 3.2, and 3.6. Thus, if the global production of bauxite or the water withdrawn per tonne of bauxite ore processed was 10% higher than estimated in 2006, $W_{tbauxite}$ would also be 10% higher at 74 million m^3. For the Ore Production Method, W_{tc} is inversely related to G_{tc}. If the estimated grade of lead was 50% higher than estimated in 2006, then W_{tlead} would be 33% lower at 29 million m^3. However, G_{tc} has no impact on the Concentrate Production Method.

4.6.2 Sensitivity to Additional Mine Data

The sensitivity of the W_{tc} to changes in X_{tc}, G_{tc}, WMC_{tc} and WMO_{tc} are linear and uniform across all the commodities studied. However, as shown in Section 4.5, the quality of different commodity datasets varies considerably. If the $\%X_{ac}$ or the $\%X_{awc}$ is low, then the estimates of overall water withdrawals is significantly more sensitive to new data than a dataset representing most of the production for a commodity. The extent of this sensitivity can be measured by calculating the impact of adding a hypothetical mine to the commodity dataset that represents both a significant portion of global production for the commodity and has a considerably different grade or water withdrawal rate than the rest of the dataset.

The sensitivity of the W_{tc} to different G_{kc} data (S_{gc}) is calculated as follows:

$$S_{gc} = \frac{W_{stc} - W_{tc}}{W_{tc}} \cdot 100 \quad [4.4]$$

where S_{gc} = the sensitivity of W_{tc} to changes in G_{ac}, [%]; W_{tc} = total water withdrawals globally attributable to commodity c, [m^3/y]; and W_{stc} is the global water withdrawal attributed to commodity c, [m^3/y] calculated as per Equations 3.1 and 3.2, but with G_{tc} replaced with a global ore grade adjusted to indicate the sensitivity of the commodity to additional production data (G_{stc}). G_{stc} is calculated as follows:

$$G_{stc} = \frac{G_{ac} \cdot \%X_{ac} + G_{ac} \cdot S_{kG} \cdot S_{k\%X}}{\%X_{ac} + S_{k\%X}} \quad [4.5]$$

where G_{stc} = the global weighted mean recovered ore grade of commodity c, modified to determine the sensitivity of G_{tc} to additional production data, [% mass]; G_{ac} = the global weighted mean recovered ore grade of commodity c from the set of mines a, [% mass]; $\%X_{ac}$ = percentage of X_{tc} of commodity c produced from the set of mines a, [%]; S_{kG} = the percentage difference between G_{ac} and the ore grade of the hypothetical mine added to test the sensitivity of W_{tc}, [%];and $S_{k\%X}$ = the percentage of global production represented by the hypothetical mine added to test the sensitivity of W_{tc}, [%]. For this sensitivity test, S_{kG} is set at 50% and $S_{k\%X}$ is set at 5%. For example, with regard to zinc, X_{tzinc} was 10.3 million t and G_{azinc} was 4.468% in 2006, based on a $\%X_{azinc}$ of 66% of the world zinc production. To test the sensitivity of G_{tzinc}, a hypothetical mine k with a grade 50% of G_{atzinc}, or a G_{kzinc} of 2.234%, which produced 5% of X_{tzinc}, or 515,000 t, is added to dataset a. The resultant G_{stzinc} is 4.309% for the modified dataset, which results in an increase in the estimated W_{tzinc} of 3.7%. The results of the sensitivity tests on W_{tc} are shown in Table 4.18. Note this sensitivity test only applies to the Ore Production Method.

Table 4.18 W_{tc} Sensitivity to Additional Mine Production Data, S_{gc}

	S_{gc}, %			
Commodity	**2006**	**2007**	**2008**	**2009**
Bauxite	3.1	3.2	3.1	3.1
Chromite			5.5	6.6
Cobalt	5.9	6.2	7.8	6.1
Copper (Concentrators)	2.9	2.9	2.9	3.0
Copper (SX-EW)	3.1	3.3	4.2	4.5
Copper (Total)				
Diamonds	4.3	4.4	4.3	5.9
Gold	3.3	3.3	3.4	3.6
Iron	4.0	4.1	3.7	5.1
Lead	4.1	4.6	5.0	5.1
Manganese				
Molybdenum	3.1	3.6	4.2	4.4
Nickel	3.7	3.6	3.6	3.8
Palladium	2.4	2.4	2.5	2.6
Phosphate	31.5	33.4	33.9	38.3
Platinum	2.3	2.4	2.4	2.7
Potash	17.8	8.4	8.4	9.9
Rhodium	2.4	2.6	2.4	2.4
Silver	3.5	3.5	3.6	3.6
Tantalum				
Tin				
Titanium				
Tungsten	4.9	4.9	36.3	32.7
Uranium	3.4	3.5	3.7	6.5
Zinc	3.7	3.7	4.1	4.1

To test the sensitivity of W_{tc} calculated from the Ore Production Model, the sensitivity of the WMO_{tc} to different W_k data (S_{wmc}) is calculated as per Equation 4.5, with S_{wmc} replacing S_{gc}. W_{stc} is the global water withdrawal attributed to commodity c, calculated as per Equation 3.1 but with WM_{tc} replaced with a global weighted mean quantity of water required to produce a commodity, adjusted to indicate the sensitivity of the commodity to additional production data (WMO_{stc}).

WMO_{stc} is calculated as follows:

$$WMO_{stc} = \frac{WMO_{awc} \cdot \%X_{awc} + WMO_{awc} \cdot S_{kWM} \cdot S_{k\%X}}{\%X_{awc} + S_{k\%X}} \qquad [4.6]$$

where WMO_{stc} = the global weighted mean quantity of water required to produce commodity *c* per tonne of ore, modified to determine the sensitivity of WM_{tc} to additional production data, [m^3/t]; WMO_{awc} = the global weighted mean quantity of water per tonne of ore required to produce commodity *c* from the set of mines *aw*, [m^3/t]; $\%X_{awc}$ = percentage of X_{tc} of commodity *c* produced from the set of mines *aw*, [%]; S_{kWM} = the percentage difference between WMO_{awc} and the amount of water required to process a tonne of ore at the hypothetical mine added to test the sensitivity of W_{tc}, [%]; and $S_{k\%X}$ = the percentage of global production represented by a hypothetical mine added to test the sensitivity of W_{tc}, [%]. For this sensitivity test, S_{kWM} is set at 150% and $S_{k\%X}$ is set at 5%. For example, with regard to zinc, X_{tzinc} was 10.3 million t and WMO_{awzinc} was 0.846 m^3/t in 2006, based on a $\%X_{awtzinc}$ of 3% of the world zinc production. To test the sensitivity of G_{tzinc}, a hypothetical mine *k*, which produced 5% of X_{tzinc}, or 515,000 t, at a water withdrawal of 150% of WMO_{awzinc}, or a WMO_{kzinc} of 1.268 m^3/t is added to dataset *aw*. The resultant WMO_{stzinc} is 1.100 m^3/t for the modified dataset, which results in an increase in the estimated W_{tzinc} of 30.1%. The results of the sensitivity tests on W_{tc} are shown in Table 4.19. The sensitivity of W_{tc} for the Concentrate Production Model is identical to that of the Ore Production Model.

Table 4.19 W_{tc} Sensitivity to Additional Mine Water Withdrawal Production Data, S_{wmc}

	S_{wmc}, %			
Commodity	**2006**	**2007**	**2008**	**2009**
Bauxite	7.3	7.5	7.8	8.4
Chromite			11.2	11.5
Cobalt	48.9	27.7	23.3	20.4
Copper (Concentrators)	5.4	9.7	8.6	7.0
Copper (SX-EW)	5.8	15.1	13.1	8.5
Copper (Total)	5.5	10.5	9.3	7.2
Diamonds	23.5	20.8	7.8	10.3
Gold	6.2	5.9	6.1	6.6
Iron	15.6	15.4	34.4	25.2
Lead	18.0	15.6	14.6	15.1
Manganese				
Molybdenum	6.9	18.8	17.2	12.2
Nickel	40.0	29.0	18.3	18.5
Palladium	11.0	8.4	16.0	11.9
Phosphate	24.8	26.1	27.0	27.7
Platinum	6.3	4.6	8.8	8.1
Potash	15.1	13.5	14.0	17.9
Rhodium	6.5	5.2	8.8	6.9
Silver	14.3	14.1	13.1	12.5
Tantalum				
Tin				
Titanium				
Tungsten	27.1	24.3	26.7	24.7
Uranium	16.1	15.7	16.1	20.9
Zinc	30.1	26.0	17.4	15.2

4.6.3 Sensitivity to Price

The total amount of water withdrawn for the commodities studied is not sensitive to changes in the P_c. However, as per Equation 3.5, P_c changes the amount of water attributed to individual commodities at mines producing multiple commodities. The sensitivity to this change, or SP_c, is shown in Table 4.20, which indicates the impact on W_{tc} if the P_c for a commodity is changed, assuming no other commodity prices change. The largest change in W_{tc} over the four years studied was recorded. For example, if the $P_{bauxite}$ increased from 28 to 42 US$/t there would be no impact on the W_{tc} of any commodity, as none of the mines reporting bauxite production produced any by-products. However, if the P_{lead} increased by 50%, the $W_{tcopper\text{-}conc}$ would decrease by up to 0.02%, the W_{tlead} would increase by 34%, the $W_{tmolybdenum}$ would decrease by 0.03%, the $W_{tsilver}$ would decrease by 5% and the W_{tzinc} would decrease by 6%. The sensitivity of W_{tc} for the Concentrate Production Model is identical to that of the Ore Production Model.

Table 4.20 W_{tc} Sensitivity to a 50% Increase in P_c, SP_c

Commodity	**Sensitivity P_c +50%**
Bauxite	No Impact
Chromite	No Impact
Cobalt	<.2% Ch, <50% Co, <.001% Cu Conc, Au, <5% Ni, <.3% for PGM
Copper (Concentrators)	<3% Co, <8% Cu Conc, <6% Au, <1% Pb, <31% Mo, <2% Ni, PGM, <22% Ag, <14% U, <11% Zn
Copper (SX-EW)	No Impact
Diamonds	No Impact
Gold	<2% Co, <7% Cu Conc, < 8%Au, <3% Mo, <1% Ni, <3% PGM, <10% Ag, <20% U, <1% Zn
Iron	No Impact
Lead	Cu Conc <.02%, Pb <34%, Mo <.03%, Ag <5%, Zn <6%
Manganese	Insufficient Data
Molybdenum	Cu Conc <4%, Au <.4%, Mo <44%, Ag <4%, Zn <2%
Nickel	<1% Cr, <30% Co, <.03% Cu Conc, Au, <40% Ni, <6% PGM
Palladium	<.05% Cr, <.02% Cu Conc, <.03% Au, <5% Ni, <46% Pd, <4% Pt, <5% Rh,
Phosphate	No Impact
Platinum	<.02% Cr, <20% Co, <.06 Cu Conc, <.06% Au, <20% Ni, <22% Pd, <20% Pt, <24% Rh
Potash	No Impact
Rhodium	< 7% Co, < .03% Cu Conc, <.03% Au, <10% Ni, <13% Pd, < 13% Pt, <41% Rh
Silver	<.9% Cu Conc, <.4% Au, <11% Pb, <2% Mo, <45% Ag, <.3% U, <4% Zn
Tantalum	Insufficient Data
Tin	Insufficient Data
Titanium	Insufficient Data
Tungsten	No Impact
Uranium	<.4% Cu Conc, Au, <.2% Ag, <39% U
Zinc	<.8% Cu, <.7% Au, <21% Pb, <2% Mo, <6% Ag, <23% Zn

As can be seen, most of the bulk or specialty commodities included come from mines without significant by-products (bauxite, diamonds, iron, phosphate, potash), hence P_c has no impact on their water withdrawal calculations. As discussed in the data accuracy section, commodities that come from mines producing multiple products tend to be the commodities with the most transparent pricing (copper, gold, lead, nickel, etc). Thus, while W_{tc} values are sensitive to changes in P_c, in practice it is a minor source of error for W_{tc} calculations.

4.7 Commodity Analysis

In the preceding sections, W_{tc} is estimated where possible for the commodities studied and then the accuracy and variability of the data used is assessed and the sensitivity of the results is analyzed. This section reviews the results on a commodity-by-commodity basis to discuss the significance of the results.

4.7.1 Bauxite

The dataset *a* for bauxite is robust, accounting for 76 to 79% of global bauxite production. The $\%X_{aw}$ is also comparatively high, ranging from 25% to 29%. All of the mines reporting data indicate a bauxite grade of 100%, removing considerable uncertainty from the estimates. The $WMO_{tbauxite}$ has little variability and the S_{gc} and S_{wmc} are relatively small, indicating the estimate of a $WMO_{tbauxite}$ of 0.320 to 0.395 m^3/t is robust and a $W_{tbauxite}$ of 65 to 79 million m^3 per year is reasonable. There is no difference in $W_{tbauxite}$ between the Ore Production Method and the Concentrate Production Method. The key uncertainly is the water withdrawals from bauxite mines in China, especially if they are mining ore with a grade less than 100%. The qualitative ranking of the representivity of the estimates was 3 for the Ore Production Method and 2 for the Concentrate Production Method.

4.7.2 Chromite

The dataset $\%X_{achromite}$ ranges from 36 to 59% and $\%X_{awchromite}$ ranges from 0 to 17%. The $G_{tchromite}$ range is significant, changing from 76 to 93%. The $WMO_{tchromite}$ could only be calculated for 2008 and 2009 and dropped from 8.115 m^3/t to 1.340 m^3/t, resulting in a $W_{tchromite}$ change from 256 million m^3 in 2008 to 33 m^3 in 2009. The $WMC_{tchromite}$ was similar, dropping from 8.115 m^3/t to 1.652 m^3/t, leading to a $W_{tchromite}$ change from 196 million m^3 in 2008 to 32 million m^3 in 2009. While the S_{gc} and S_{wmc} are relatively small, the RO_{chromite} rises from a low 9.25 to a more common 46.74 US$/m^3 from 2008 to 2009 ($12.11 to $48.85 for the Concentrate Production Method). The change can largely be attributed to a water consumption of around 24 million m^3 in 2008 at the Kroondal Mine in South Africa in 2008, dropping to 3.4 million m^3 in 2009. While the data show considerable uncertainty, the 2009 data appear to be more representative of the industry and a $WMO_{tchromite}$ of around 1.3 m^3/t with an annual $W_{tchromite}$ of approximately 33 million m^3 may be considered a more typical withdrawal rate for chromite. The 2009 qualitative ranking of the representivity of the estimates was 3 for the Ore Production Method and 2 for the Concentrate Production Method.

4.7.3 Cobalt

The dataset $\%X_{acobalt}$ ranges from 30 to 40% and $\%X_{awcobalt}$ ranges from 0.1 to 7%. The $G_{tcobalt}$ range is relatively constant, ranging from 0.017 to 0.026%. In 2006 the *aw* dataset consisted of only one mine producing cobalt as a by-product, and the results can safely be ignored. The Ore Production Method found the $WMO_{tcobalt}$ is relatively constant, ranging from 0.060 to 0.080 m^3/t and $W_{tcobalt}$ ranges from 16 to 26 million m^3. The RO_{cobalt} is reasonable, ranging from 48.90 to 85.92 US$/m^3. The Concentrate Production Method found the $WMC_{tcobalt}$ ranging from 364 to 802 m^3/t concentrate and $W_{tcobalt}$ ranging from 20 to 46 million m^3, with an RC_{cobalt} ranging from 25.27 to 44.97 US$/m^3. The G_{ac} was considerably higher than the G_{awc}, meaning that mines reporting water use generally had lower grades than the average global grade. Thus the Ore Production Method may be more reliable in this case. The qualitative ranking of the representivity of the estimates was 2 for the Ore Production Method and 1 for the Concentrate Production Method.

4.7.4 Copper

Copper production is separated into two subsets, one for concentrators and another for SX/EW operations. The datasets *a* and *aw* are robust, with the $\%X_{acopper\text{-}conc}$ ranges from 82 to 84% and $\%X_{awcopper\text{-}conc}$ ranges from 21-41%. The $G_{tcopper\text{-}conc}$ range is relatively constant, ranging from 0.579 to 0.700 %, the $WMO_{tcopper\text{-}conc}$ is relatively constant, declining from 0.700 to 0.538 m^3/t, and $W_{tcopper\text{-}conc}$ ranges from 1,051 to 1,176 million m^3. Similarly, the $WMC_{tcopper\text{-}conc}$ is relatively constant, ranging from 99.55 to 74.32 m^3/t concentrate and $W_{tcopper\text{-}conc}$ ranges from 936 to 1,254 million m^3. The $RO_{copper\text{-}conc}$ and $RC_{copper\text{-}conc}$ are reasonable, ranging from 45.30 to 71.79 US$/$m^3$, and the S_{gc} and S_{wmc} are comparatively low, indicating that the sensitivity to additions in the database is low. Overall, the copper concentrator data and estimates are among the most robust of the commodities studied. The quality of the data may be high enough to assume that the declining trend visible in $WMO_{tcopper\text{-}conc}$ reflects a decline in the intensity of water use by copper concentrators over the period of 2006-2009, although the declining $G_{copper\text{-}conc}$ means that $W_{tcopper\text{-}conc}$ has not significantly dropped. Another partial explanation is the increase in the price of gold relative to copper, leading to a decrease in the water withdrawals attributed to copper, as many of these mines produce gold as a by-product. With respect to copper concentrators, the Ore Production Method appears to be more reliable than the concentrate production method. The Concentrate Production Method estimates water withdrawals have dropped from 2006 to 2009. However, the Ore Production Method highlights the dropping ore grade in dataset *a* versus the mines reporting water withdrawal in dataset *aw*, indicating that total ore processed has risen due to the requirement to process more ore to maintain concentrate production. Thus, the Ore Production Method more reasonably estimates that water withdrawals have remained steady, even if water requirements per tonne of ore or concentrate have decreased. The qualitative ranking of the representivity of the estimates was 4 for the Ore Production Method in 2006, dropping to 3 after 2007. For the Concentrate Production Method, the ranking dropped from 3 to 2 after 2007.

While not quite as robust as the copper concentrate datasets, the SX/EW datasets *a* and *aw* are also strong with the $\%X_{acopper\text{-}SX/EW}$ ranging from 53 to 78% and $\%X_{awcopper\text{-}SX/EW}$ ranges from 12-38%. The $G_{tcopper\text{-}SX/EW}$ range is relatively constant, ranging from 0.258 to 0.294 %. The $WMO_{tcopper\text{-}SX/EW}$ is relatively constant, ranging from 0.164 to 0.172 m^3/t and $W_{tcopper\text{-}SX/EW}$ ranges from 161 to 205 million m^3. The $WMC_{tcopper\text{-}SX/EW}$ is relatively constant, ranging from 35.12 to 45.99 m^3/t concentrate and $W_{tcopper\text{-}SX/EW}$ ranges from 99 to 142 million m^3. The Concentrate Production Method estimates are higher than the Ore Production Method estimates, as ore grades in the $G_{awcopper\text{-}SX/EW}$ dataset are consistently 40 to 60% higher than in the $G_{acopper\text{-}SX/EW}$ dataset. The $RO_{copper\text{-}SX/EW}$ for ore ranges from 46.34 to 91.70 US\$/m^3, quite similar to $RO_{copper\text{-}conc}$. This is interesting, as there is often a perception that SX/EW operations are more water efficient than concentrators. However, the $RC_{copper\text{-}SX/EW}$ values are higher than the $RC_{copper\text{-}conc}$ values. The S_{gc} and S_{wmc} are comparatively low, indicating that the sensitivity to additions in the database is low. The qualitative ranking of the representivity of the estimates ranged between 2 and 3 for the Ore Production Method and for the Concentrate Production Method, ranged between 1 and 3. Overall, the copper SX/EW data and estimates are fairly complete.

Combining the concentrator and SX/EW estimates leads to a $W_{tcopper}$ ranging from 1,233 to 1,363 million m^3, among the most significant sources of water withdrawals in the mining industry.

4.7.5 Diamonds

The dataset $\%X_{adiamond}$ ranges from 40 to 56% and $\%X_{awdiamond}$ ranges from 6 to 27%. The $G_{tdiamond}$ range is relatively constant, ranging from 0.144 to 0.213 g/t. The $WMO_{tdiamond}$ varies significantly, ranging from 0.963 to 3.330 m^3/t and $W_{tdiamond}$ ranges from 206 to 713 million m^3. The $WMC_{tdiamond}$ also varies significantly, ranging from 3.2 million m^3/t to 12.9 million m^3/t of concentrate and $W_{tdiamond}$ ranges from 109 to 334 million m^3. In 2006 and 2007 there is only one mine in dataset *aw*, the Diavik Diamond Mine in northern Canada. As the Diavik ore deposit is located under a large lake and requires constant dewatering, the W_k is comparatively high. Similarly, the $RO_{diamond}$ is 12.17 and 9.50 US$/$m^3$ in 2006 and 2007 respectively. In 2008 it rises to 33.96 US$/$m^3$, but drops to 12.60 US$/m^3 in 2009, largely due to the start up of another mine in northern Canada. $RC_{diamond}$ also is quite variable, ranging 11.53 to 62.09 US$/$m^3$.While the sensitivity measures are relatively low, overall, the northern Canadian diamond mines appear to skew the water withdrawal estimates. However, there are some significant diamond dredging operations as well, which may mean that diamond extraction on a global basis has a high overall water consumption. The qualitative ranking of the representivity of the estimates ranged between 2 and 3 for the Ore Production Method and for the Concentrate Production Method, ranged between 1 and 2.

4.7.6 Gold

The dataset $\%X_{agold}$ ranges from 67 to 74% and $\%X_{awgold}$ ranges from 33 to 37%. The *a* and *aw* datasets contain the largest number of mines of any of the commodities studied. The G_{tgold} range is consistent, ranging from 0.843 to 0.908 g/t. The WMO_{tgold} grew from 0.252 to 0.396 m^3/t and W_{tgold} grew consistently from 657 to 1,141 million m^3. The WMC_{tgold} grew from 309,110 to 453,305 m^3/t concentrate and the Concentrate Production Method estimated W_{tgold} growing from 733 to 1,111 million m^3. The RO_{gold} is fairly consistent and ranges from 41.00 to 51.82 $US/$m^3$

and the RC_{gold} ranged from 46.23 to 49.68 $US/m^3, well within the average range found for commodities in this study. The S_{gc} and S_{wmc} values are low and the sensitivity of W_{tgold} to a 50% chance in P_{gold} was less than 8%. The qualitative ranking of the representivity of the estimates was 4 for the Ore Production Method, dropping to 3 in 2009. For the Concentrate Production Method, the ranking dropped from 3 to 2 in 2009.

Overall, the quality of the data may be high enough to assume that the increasing trend visible in WMO_{tgold} and W_{tgold} may reflect an increase in the intensity of water use by gold over the period of 2006-2009. However, the increasing value of gold during this period also led to an increase in the water withdrawals attributed to gold, as many of mines produce gold as a by-product. Furthermore, gold, like copper, is processed by several different methods, including heap leaching, agitated leaching, flotation, gravity, and as by product of production of other mines. There may be value in separating the different methods in the database, as the processes all have different water requirements.

4.7.7 Iron

Iron ore, is primarily produced from two sources – mines that can direct ship their ore with no little to no mineral processing, and mines that produce concentrate ores that require mineral processing. The former often consists of solely crushing and screening plants and can be considered dry processes. Water withdrawals for these operations are primarily for mine dewatering and dust suppression. The latter generally includes wet grinding, spirals, magnetic separation, flotation and/or other wet processes. Ideally this water withdrawal estimate would separate the two major sources of iron ore and would use iron content instead of concentrate tonnage. However, iron mines typically only report the concentrate or ore shipped, not the ore mined, making it challenging to determine overall plant tonnages or if they use wet processes. In addition, few mines report water usage and some mining operations have both direct shipping ore and ore requiring beneficiation. Thus, iron concentrate is reported as a single commodity and the *a* and *aw* datasets are inherently limited.

The dataset $\%X_{airon}$ ranges from 46 to 60% based on 109 or more mines and $\%X_{awiron}$ ranges from 2 to 11% based on 3 to 5 mines. The G_{tiron}, reported in % concentrate not % iron, ranges from 71.6 to 83.8%, which is a large range for a major bulk commodity like iron. The RO_{iron} is volatile, ranging from 74.94 to 138.32 \$US/m^3, and the RC_{iron} ranged from 95.23 to 149.46 \$US/m^3. The P_{iron} rose substantially from 2006 to 2009, but as iron mines typically have no by-products, the SP_{iron} is negligible. The mines in the *aw* dataset are major operations in Australia and South Africa and are in no way representative of iron producers in Brazil, China, India or North America. The WMO_{tiron} is relatively consistent, ranging from 0.237 to 0.436 m^3/t and W_{tiron} ranges from 555 to a peak 883 million m^3. The WMC_{tiron} is also relatively consistent, ranging from 0.307 to 0.452 m^3/t and W_{tiron} for the Concentrate Production Method estimate ranges from 453 to a peak 776 million m^3. The S_{wmiron} is moderate, ranging from 15.4 to 34.4%. The qualitative ranking of the representivity of the estimates ranged from 2 to 3 for the Ore Production Method and 1 for the Concentrate Production Method.

4.7.8 Lead

The dataset $\%X_{alead}$ ranges from 47 to 59% based on 78 or more mines and $\%X_{awlead}$ ranges from 9 to 12% based on 2 mines in 2006, 4 mines in 2007 and 6 mines in 2008 and 2009. The G_{tlead} range is relatively constant, ranging from 1.073 to 1.366%. The WMO_{tlead} ranges from 0.080 to 0.392 m^3/t and W_{tlead} ranges from 44 to a peak 118 million m^3 in 2007 before backing down to 30 and 29 million m^3 in 2008 and 2009. The WMC_{tlead} ranges from 3.995 to 8.485 m^3/t and W_{tlead} estimates for the Concentrate Production Method ranges from 14 to a peak of 33 million m^3 in 2009. As the $\%X_{awlead}$ increases the W_{tlead} estimates from the two methods converged. The RO_{lead} is volatile, rising from 61.31 and 41.88 US\$/m^3 in 2006 and 2007 to 128.86 and 107.85

US$/m^3 in 2008 and 2009. The RC_{lead} ranged from 95.25 to 149.46 US$/m^3. The price of lead rose substantially from 1,279 to 2,579 $US/t from 2006 to 2007 before settling at 2,090 and 1,720 $US/t in 2008 and 2009. The maximum SP_{lead} was 34% for lead, although lead was also sensitive to changes in silver (11%) and zinc (21%). The VG_{tlead} is very high, at 207.2 and 281.4% in 2006 and 2007 before dropping to 5.8 and 12.3% in 2008 and 2009, indicating a significant disparity between the average lead grades in the *a* and *aw* datasets. The high W_{tlead} in 2007 can thus be partly explained by the spike in the lead price and the relatively small number of mines in the *aw* dataset. In addition, Mount Isa Mine in Australia, one of the major lead/zinc mines in the *aw* dataset, had an increased water withdrawal of about 20% in 2007, before dropping to back to approximately 2006 levels in 2008 and 2009. The qualitative ranking of the representivity of the estimates was 2 for the Ore Production Method and 1 for the Concentrate Production Method.

4.7.9 Manganese

The $\%X_{amanganese}$ ranges from 14 to 26% based on 4 to 17 mines. No manganese producing mines were found that reported water data, leading to $\%X_{awmanganese}$ of zero. It is uncertain why Samacor, the operator of several major manganese producing mines, or BHP, Samacor's majority owner, reports so little data on its operations.

In 2004, a life cycle analysis of Samacor's South African manganese mines found approximately 11 litres of water consumed per kilogram of SiMn produced and 10.5 litres of water consumed per kilogram of FeMn produced during mining (Cronje, 2004). If a ratio of 10 m^3/t of concentrate produced is used, the $W_{tmanganese}$ would be fairly high at 332 to 381 million m^3.

4.7.10 Molybdenum

The dataset $\%X_{amolybdenum}$ ranges from 54 to 77% based on 25 or more mines and $\%X_{awmolybdenum}$ ranges from 8 to 31% based on 3 to 8 mines. The $G_{tmolybdenum}$ range is relatively constant, ranging from 0.019 to 0.023%. The $WMO_{tmolybdenum}$ ranges from 0.049 to 0.153 m^3/t and $W_{tmolybdenum}$ ranges from 58 to 129 million m^3. The $WMC_{tmolybdenum}$ ranges from 382 to 797 m^3/t and $W_{tmolybdenum}$ estimate from the Concentrate Production Method ranges from 152 to 84 million m^3. The $RO_{molybdenum}$ ranges from 53.62 to 89.59 US$/m^3 and the $RC_{molybdenum}$ ranges from 36.50 to 61.97 US$/m^3. The price of molybdenum dropped from around 60,000 to 25,800 $US/t in 2009. The maximum $SP_{molybdenum}$ was 44% for molybdenum, although it was also sensitive to changes in copper (31%), gold (3%), silver (2%) and zinc (2%). The high $W_{tmolybdenum}$ in 2006 can thus be partly explained by the higher $P_{molybdenum}$. The qualitative ranking of the representivity of the estimates ranged from 2 to 3 for the Ore Production Method and 1 to 2 for the Concentrate Production Method.

4.7.11 Nickel

The dataset $\%X_{anickel}$ ranges from 63 to 67% based on 55 or more mines. The $\%X_{awnickel}$ rises from 1 to 9% based on 12 mines in 2006, 17 mines in 2007, 9 mines in 2008 and 12 mines 2009. The $G_{tnickel}$ range is relatively constant, ranging from 0.553 to 0.480%. The $WMO_{tnickel}$ rises from 0.069 to 0.567 m^3/t and $W_{tnickel}$ rises from 20 to 165 million m^3 from 2006 to 2009. The $WMC_{tnickel}$ ranges from 138 to 240 m^3/t and $W_{tnickel}$ ranges from 217 to 402 million m^3. The RO_{nickel} in 2006 is 1,246.62 \$/m^3, but drops to 78.51 \$/m^3 in 2009. RC_{nickel} is lower, dropping from 112.83 to 43.25 \$/m^3 from 2006 to 2009. Thus, it appears likely that the 2006 data for nickel was not representative and the 2009 water withdrawal estimate is more reasonable. $G_{awnickel}$ is significantly lower than $G_{anickel}$, leading to a higher water withdrawal estimate using the Ore Production Method. The qualitative ranking of the representivity of the estimates was 2 for the Ore Production Method and 1 for the Concentrate Production Method.

A review of the 2006 dataset indicates that most of the water withdrawal information was from mines primarily producing platinum group metals in South Africa, where as the 2009 dataset also includes some more significant primary nickel producers from Australia. Unfortunately, the major nickel producers from Canada and Russia did not provide water withdrawals on a mine by mine basis over these years. Furthermore, mines processing sulfides use considerably different methods than mines processing nickel laterites – insufficient data was available to differentiate water withdrawals from these methods.

4.7.12 Palladium

The dataset $\%X_{apalladium}$ ranges from 94 to 102% based on 36 or more mines. The $\%X_{awpalladium}$ rises from 11 to 25% based on data from 5 to 12 mines. The $G_{tpalladium}$ range is relatively constant, ranging from 1.462 to 1.682 g/t. The $WMO_{tpalladium}$ steadily rises from 0.057 to 0.113 m^3/t and $W_{tpalladium}$ rises from 8 to 14 million m^3 from 2006 to 2009. The $WMC_{tpalladium}$ steadily rises from 56,779 to 108,162 m^3/t and $W_{tpalladium}$ rises from 13 to 21 million m^3 from 2006 to 2009. The $RO_{palladium}$ is high, but drops from 311.77 \$/m^3 in 2006 to 110.31 \$/m^3 in 2009. The $RC_{palladium}$ drops from 187.13 \$/m^3 in 2006 to 73.84 \$/m^3 in 2009. $G_{awpalladium}$ is about 20% higher than $G_{apalladium}$, leading to a lower water withdrawal estimate using the Ore Production Method.

Overall, the quality of the data may be high enough to assume that the increasing trend visible in $W_{tpalladium}$ may reflect an increase in the intensity of water use by palladium producers over the period of 2006-2009. The qualitative ranking of the representivity of the estimates ranged from 2 to 3 for the Ore Production Method and from 1 to 2 for the Concentrate Production Method.

4.7.13 Phosphate

The Ore Production Method estimate $W_{phosphate}$ ranged from 3,046 to 3,187 million m^3 and the Concentrate Production Method estimate ranged from 2,623 to 3,052 million m^3, which is the largest water withdrawal for any of the commodities studied. However, this estimate likely erroneous for several reasons. The dataset $\%X_{aphosphate}$ ranges from 4 to 5% based on 2 or 3 mines and $\%X_{awphosphate}$ ranges from 4 to 5%, also based on 2 mines. Even more so than iron, most producers do not report the amount of ore moved, only the amount of concentrate produced. The $G_{tphosphate}$, reported in % concentrate not % phosphate, ranges from 86.1 to 100%, and likely does not represent the actual grade of phosphate ore. The $RO_{phosphate}$ is extremely low, ranging from 0.07 to 0.19 \$/m^3 and $RC_{phosphate}$ ranges from 0.07 to 0.14 \$/m^3. While the $P_{phosphate}$ rose substantially from 2006 to 2009, but as phosphate mines typically have no by-products, the $SP_{phosphate}$ is negligible. The qualitative ranking of the representivity of the estimates was 1 for both the Ore Production Method and Concentrate Production Method, indicating a low degree of confidence in the estimate.

The two mines reporting water withdrawals are PCS's Aurora and White Springs mines, each of which withdrew from 37,000,000 m^3/y to 86,000,000 m^3/y. The PCS reported Aurora operation water withdrawal is larger than any of the other mining operations studied, including the largest copper and iron mines. While ore volumes are not included in PCS annual reports, a publically available report from the Aurora Chief Geologist and Superintendant of Mine Planning reported that 11 million tons of ore produced 5 million tons of concentrate, which would indicate an effective ore grade of 45% (Gilmore, 2011). Ore is transported by slurry from the mine to the plant, requiring significant volumes of water, with much of the water recycled back to the mine (US Army Corps of Engineers, 2008). PCS is required to continuously dewater an area with a radius of about 20 miles to keep mining areas dry, providing a considerable source of water for transporting slurry to the plant. The remaining dewatering water is diverted to a nearby river (US Army Corps of Engineers, 2008). As per the GRI protocols, PCS is assumed not to include diversion in their calculation of indicator EN08. Withdrawals at White Springs are on similar scale. Chen *et al.* (2006) state that Florida phosphate mines waste slurry ranges from 20,000 to 60,000 gallons per minute (4,542 to 13,630 m^3/h), indicating that the Florida phosphate mines also require large water withdrawals.

The mines in the *a* and *aw* dataset are major operations in the USA and are in no way representative of other major phosphate producers in Brazil, China, North Africa, the Mid-East, or Russia. The USA represents about 20% of world phosphate production. However, the sheer scale of water withdrawals from Aurora and White Springs indicate that phosphate water withdrawals on a global basis will still be impacted by the high withdrawals in the USA.

4.7.14 Platinum

The dataset $\%X_{aplatinum}$ ranges from 91 to 104% based on 35 or more mines. The $\%X_{awplatinum}$ rises from 23 to 50% based on data from 5 to 16 mines. The $G_{tplatinum}$ range is relatively constant, ranging from 1.436 to 1.677 g/t. The $WMO_{tplatinum}$ steadily rises from 0.385 to 0.754 m^3/t and $W_{tplatinum}$ rises from 50 to 94 million m^3 from 2006 to 2009. The $WMC_{tplatinum}$ steadily rises from 200,4000 to 437,462 m^3/t and $W_{tplatinum}$ rises from 43 to 79 million m^3 from 2006 to 2009. The $RO_{platinum}$ drops from 167.37 $/m^3$ in 2006 to 68.15 $/m^3$ in 2009 and the $RC_{platinum}$ is similar, dropping from 191.60 to 80.61 $/m^3$. The qualitative ranking of the representivity of the estimates ranged from 3 to 4 for the Ore Production Method and from 2 to 3 for the Concentrate Production Method.

Overall, the quality of the data may be high enough to assume that the increasing trend visible in $WMO_{tplatinum}$ and $W_{tplatinum}$ may reflect an increase in the intensity of water use by platinum producers over the period of 2006-2009.

4.7.15 Potash

The dataset $\%X_{apotash}$ ranges from 12 to 27% based on 5 to 10 mines and $\%X_{awpotash}$ ranges from 9 to 13%, also based on 5 mines. The $G_{tpotash}$, reported in % concentrate not % potassium, dropped from 19.0 to 14.7%. The P_{potash} rose substantially from 2006 to 2009, but as potash mines typically have no by-products, the SP_{potash} is negligible. However, the change in price is a significant contributor to the increase in RO_{potash} 21.77 to 97.02 \$/m^3. The Ore Production Method estimated W_{potash} ranged from 58 to 80 million m^3 and for the Concentrate Production Method, 60 to 62 million m^3. The $G_{apotash}$ decreases substantially in 2009 as the price rose, but the $G_{awpotash}$ remained the same, indicating that mines with lower grade deposits that do not report water withdrawals ramped up production to take advantage of higher prices. The qualitative ranking of the representivity of the estimates was 2 for the Ore Production Method and1 for the Concentrate Production Method.

Overall, all of the *aw* dataset comes from underground potash mines in Canada. While Canada produces almost a third of world's potash, the dataset may not be representative of the other major potash producing locations, including Belarus, Germany, and Russia. However, most of the other major potash mines appear to also be underground mines with similar grades and processing. Provided that none of them have significant dewatering requirements, the W_{potash} estimates may be reasonable. There are also some operations which practice *in situ* or solution mining of potash, pumping hot water into a potash deposit and extracting a potash rich brine. Water withdrawal data is available from the Patience Lake mine which uses solution mining to extract potash. The mine is not included in the *a* or *aw* datasets as there is no information available on the ore grade, only the potash production. While it is not possible to calculate the water withdrawal per tonne of ore at Patience Lake, the water withdrawal per tonne of concentrate ranges from 2.50 to 5.03 m^3/t, which is similar to the global weighted average withdrawal of 1.615 to 2.899 m^3/t concentrate. It is reasonable to conclude that the Ore Production Method W_{potash} estimates are in the right order of magnitude and may be fairly accurate.

4.7.16 Rhodium

The dataset $\%X_{arhodium}$ ranges from 93 to 104% based on 29 or more mines. The $\%X_{awrhodium}$ rises from 23 to 43% based on data from 5 to 16 mines. The $G_{trhodium}$ range is relatively constant, ranging from 0.203 to 0.222 g/t. The $WMO_{trhodium}$ steadily rises from 0.117 to 0.383 m^3/t from 2006 to 2008, but drops to 0.151 when the $P_{rhodium}$ dropped by 75% in 2009. This price drop was relatively larger than palladium or platinum. The $W_{trhodium}$ rose from 21 to 45 million m^3 from 2006 to 2008, then dropped to 16 million m^3 in 2009. The $R_{rhodium}$ drops from 189.28 \$/m^3 in 2006 to 75.39 \$/m^3 in 2009. Likewise, the $WMC_{trhodium}$ steadily rises from 802,000 to 1,585,000 m^3/t from 2006 to 2008, but drops to 602,000 m^3/t in 2009. The Concentrate Production Method estimate of $W_{trhodium}$ rose from 21 to 38 million m^3 from 2006 to 2008, then dropped to 13 million m^3 in 2009. The $RC_{rhodium}$ drops from 187.90 \$/m^3 in 2006 to 88.07 \$/m^3 in 2009. The qualitative ranking of the representivity of the estimates ranged from 3 to 4 for the Ore Production Method and from 2 to 3 for the Concentrate Production Method.

Overall, the quality of the data may be high enough to assume that the increasing trend visible in $WMO_{trhodium}$ and $W_{trhodium}$ may reflect an increase in the intensity of water use by rhodium producers over the period of 2006-2008 and the dramatic price drop in 2009 reduced the water withdrawals attributable to rhodium. Since palladium, platinum and rhodium all tend to be produced from the same mines, overall PGMs water withdrawals seem to be trending up.

4.7.17 Silver

The dataset $\%X_{asilver}$ ranges from 67 to 69% from 171 or more mines and $\%X_{awsilver}$ ranges from 13 to 15% from 26 or more mines. The $G_{tsilver}$ range is consistent, ranging from 10.613 to 11.393 g/t. The $WMO_{tsilver}$ grew from 0.011 to 0.021 m^3/t and $W_{tsilver}$ ranged from 21 to 40 million m^3. The $WMC_{tsilver}$ ranges from 1,740 to 3,128 m^3/t and for the Concentrate Production Method estimate $W_{tsilver}$ ranged from 37 to 68 million m^3. The V_{gtc} is somewhat high, ranging from 41.2 to 65.4%, as the $G_{asilver}$ was consistently higher than the $G_{awsilver}$. The sensitivity of $W_{tsilver}$ to a 50% chance in P_{silver} was fairly high at 45%. The RO_{silver} ranged from 175.25 to 292.52 \$US/m^3, on the high end of the range found for commodities in this study, where as the RC_{silver} ranged from 82.25 to 172.08 \$US/m^3. The S_{gc} and S_{wmc} are low. The qualitative ranking of the representivity of the estimates was 2 for the Ore Production Method and 1 for the Concentrate Production Method. Overall, the quality of the data indicates that the Ore Production Method estimate of $W_{tsilver}$ may not be highly accurate, but is in the right order of magnitude.

4.7.18 Tantalum

The $\%X_{atantalum}$ ranges from 0 to 47% based on one mine reporting data in 2008. No tantalum producing mines were found that reported water data, leading to $\%X_{awtantalum}$ of zero. The bulk of tantalum production comes from 7 mines located in Australia, Brazil, Canada, Ethiopia and Mozambique. There is also artisanal and small-scale mining production from locations such as the Democratic Republic of the Congo. By far the largest mine is the Wodgina Pan West mine in Australia, owned by Global Advanced. In 2008, Wodgina reported ore production, allowing for the ore grade to be calculated at 0.040% tantalum. Wodgina appears to be a open pit hard rock mine in an arid environment, running a gravity concentration circuit, and likely does not have a higher than average water withdrawal per tonne of ore processed. The overall amount of ore processed globally, estimated with a grade of 0.040% tantalum, is 2.99 Mt in 2008, by far the smallest ore production of the commodity investigated in this report. The amount of water withdrawal per tonne of ore processed is likely significantly less than 1 m^3/t, which would make tantalum the lowest source of water withdrawals out of the commodities studied. Global tantalum sale revenues are estimated at $40 to 100 million $US/year, also the smallest of any of the commodities studied.

4.7.19 Tin

The $\%X_{atin}$ ranges from 1 to 6% based on 1 to 3 mines, based in Australia and Bolivia. The G_{ttin} ranged from 0.763 to 1.577%, but due to the small size of the *a* dataset, these values cannot be considered representative. No tin producing mines were found that reported water data, leading to $\%X_{awtin}$ of zero. The bulk of the world's tin production comes from Bolivia, Brazil, China, Indonesia, and Peru, with China and Indonesia producing almost two-thirds of global production. Production is dominated by state-owned companies with little production information available. Much of Indonesia's production comes from placer or offshore production, which likely leads to a high volume of water withdrawals. There are likely many smaller placer operations as well. In summary, no reliable information could be found on water withdrawals, but due to the likelihood of significant production coming from placer operations, it is possible that global water withdrawals are significant.

4.7.20 Titanium

The $\%X_{atitanium}$ ranges from 0 to 9% based on the Tellnes Ilmenite Mine in Norway. $G_{ttitanium}$ ranged from 8.368 to 8.562%, but due to the small size of the *a* dataset, these values cannot be considered representative. No titanium producing mines reported both water withdrawals and ore production, leading to $\%X_{awtitanium}$ of zero.

The largest titanium mine in the world is Rio Tinto's Tio mine in Quebec, but the operation does not report on water withdrawals or ore grades. The ore is crushed on the mine site, then sent directly to a metallurgical complex, where the water consumption is high (QIT, 2011). Overall, the $W_{ttitanium}$ is expected to be low compared to other commodities, due to the relatively low production, relatively high grades, and the practice of direct shipping ore.

4.7.21 Tungsten

The $\%X_{atungsten}$ ranges from 49% in 2006 and 2007 based on 20 mines, to 4 or 5% in 2008 and 2009 based on one mine. The $\%X_{awtungsten}$ ranges from 4 to 5% from one mine, the Cantung Mine in the Yukon. The $G_{atungsten}$ ranged from 0.267 to 0.255 % in 2006 and 2007, rising to 0.683 and 0.788 in 2008 and 2009, whereas $G_{awtungsten}$ remained fairly constant at 0.683 to 0.788%. The $O_{ttungsten}$ dropped from 21.2 Mt in 2006 to 7.8 Mt in 2009, but this drop is solely a function of the higher than average grade at Cantung. The $WMO_{ttungsten}$, based on Cantung, ranged from 3.056 to 3.878 m^3/t, however, due to the decreasing $O_{ttungsten}$, the $W_{ttungsten}$ dropped from 65 to 27 million m^3. The $WMC_{ttungsten}$ ranged from 436 to 463 m^3/t and the Concentrate Production Method estimate of $W_{ttungsten}$ ranged only from 27 to 35 million m^3. The V_{gtc} went from 162.5% to zero. The $RO_{tungsten}$ in 2006 was low at \$8.94, but dropped to \$2.21 in 2009. The S_{gc} is low for 2006 and 2007, rising sharply in 2008 and the S_{wmc} is high accross all four years. Overall, the quality of the *a* dataset in 2006 and 2007 is fair, but the dataset in 2008 and 2009 and the entire *aw* dataset is too reliant on one mine. The Cantung Mine water withdrawals are also likely higher than the global average – almost all of the other tungsten mines are located in South East China and are likely in areas with significant competition for water resources. The qualitative ranking of the representivity of the estimates ranged from 1 to 2 for the Ore Production Method and was 1 for the Concentrate Production Method.

4.7.22 Uranium

The dataset $\%X_{auranium}$ drops from 70 to 36% from 13 or more mines and $\%X_{awuranium}$ ranges from 7 to 11% from 6 mines. The $G_{auranium}$ ranges from 0.052 to 0.082 and $G_{awuranium}$ ranges from 0.017 to 0.024 %. The $WMO_{turanium}$ ranges from 0.149 to 0.288 m^3/t and the $WMC_{turanium}$ ranges from 698 to 1,256 m^3/t concentrate. The dropping average grade and rising water consumption lead the $W_{turanium}$ to rise from 9 to 25 million m^3 for the Ore Production Method and 32 to 75 million m^3 for the Concentrate Production Method, due to the lower grade of the *aw* dataset. One of the reasons for the drop is because the 2009 production data was unavailable for the McArthur River uranium mine, the highest grade and highest uranium producer in the world. The $RO_{uranium}$ ranged from 85.89 to 196.73 $US/$m^3$, on the high end of the range found for commodities in this study, but $RC_{uranium}$ ranged from 28.83 to 56.18 $US/$m^3$. The qualitative ranking of the representivity of the estimates was 2 for the Ore Production Method and 1 for the Concentrate Production Method. Unfortunately none of the Canadian or Kazakh uranium mines reported water withdrawals, limiting the overall representatives of the water data and throwing into question the value of the withdrawal estimates.

4.7.23 Zinc

The dataset $\%X_{azinc}$ ranged from 59 to 66% based on 96 or more mines and $\%X_{awzinc}$ ranges from 3 to 11% based on 4 mines in 2006, 8 in 2007,10 in 2008 and 11 in 2009. The G_{tzinc} values drop from 4.468 to 3.549% in 2009. The WMO_{tzinc} drops from 0.846 in 2006 to 0.302 m^3/t in 2009 and W_{tzinc} drops from 195 in 2006 and 243 in 2007 to 95 million m^3 in 2009. The WMC_{tzinc} drops from 24.65 in 2006 to 13.10 m^3/t in 2009 and the Concentrate Production Method estimate of W_{tzinc} drops from 254 in 2006 and 301 in 2007 to 147 million m^3 in 2009. The RO_{zinc} ranges from 94.56 to 156.20 US$/$m^3$ and the RC_{zinc} ranges from 87.10 to 72.37 US$/$m^3$. The price of zinc steadily dropped from 3,274 to 1,656 $US/t from 2006 to 2009. The maximum SP_{zinc} was 23% for zinc, although zinc was also sensitive to changes in copper (11%), gold (1%), lead (6%), and molybdenum (2%). The qualitative ranking of the representivity of the estimates ranged from 2 to 3 for the Ore Production Method and was 1 for the Concentrate Production

Method. The dropping W_{tzinc} can thus be partly explained by the drop in the zinc price and partly due to the lowering of the grade due to the relatively small number of mines in the *aw* dataset in 2006 and 2007. Overall the 2008 data appears to be the most reliable.

4.8 Other Commodities

This study did not include water withdrawals for aggregates, coal or oil sands mining operations, or several more minor industries.

Aggregates are dispersed around the world and no estimates are available on their water withdrawals for dewatering, washing and dust suppression, but total consumption is likely significant.

The coal extraction industry is massive and dispersed, with little water withdrawal information available. The World Coal Association estimates total coal production in 2009 to be 6.823 billion tonnes (World Coal Association, 2012). Mudd estimated black coal average water use at 0.3 m^3/t of coal processed (Mudd, 2008), which would lead to a total water withdrawal in the range of 2 billion m^3 per year. A large portion of coal production is not washed, so water withdrawals may be lower. However, Circle of Blue, an international network of water experts, estimated that coal washing in China alone uses an estimated 178 to 238 million m^3 per year (Schneider, 2011). Assuming total water withdrawals for coal is in the range of a billion m^3 per annum or higher is not unreasonable.

Oil sands extraction is limited to Fort McMurray, in Alberta, Canada, but oil sands extraction rates are high and rising. Water extraction permits in 2006 totaled 370 million m^3 per year and could rise to 529 million m^3 per year (National Energy Board, 2006).

4.9 Conclusion

As stated in Section 1.2, the first question this research aimed to answer was: How can the amount of water withdrawn by the mining industry be better quantified? This question was addressed by pursuing the following objectives:

- Identifying methods for quantifying global mine water withdrawals by commodity; and
- Estimating global mine water withdrawals by commodity.

In the literature review, Section 2.1 defined the term water withdrawal and associated concepts. Then Section 2.3 described past efforts to describe water use in the mining industry and indicated that the available literature showed a large range between estimates of mine water use. The literature review found there had been no systematic effort or models available to estimate global mine water use or withdrawals.

Section 3.1 described two novel methodologies to estimate global mine water withdrawal rates, the Ore Production Method and the Concentrate Production Method, and Chapter 4 undertook an estimate of global mine water withdrawals by commodity.

For the commodities included in this study, total global water withdrawals due to mining were estimated to range from six to eight billion m^3 per annum over the period of 2006 to 2009. The methodologies extrapolated these estimates using production data from 1,076 mines over a four-year period (2006 to 2009) and included 6,723 commodity production figures and 541 water withdrawal figures, accounting for a large portion of the global mining industry. As shown in Table 4.3, the production datasets accounted for between none and all of global production, depending on the commodity. As shown in Table 4.4, the water withdrawal datasets accounted for between zero and 50% of all global production, depending on the commodity.

While water withdrawal estimates vary with the two methods, they are generally in the same order of magnitude. However, the amount, accuracy and representativity of the data available varied significantly from commodity to commodity and from year to year. The publically available production and water withdrawal data was incomplete and assumed to be biased, presenting a statistical problem of missing data. Section 4.5 assessed the accuracy and representativity of the variables used in the datasets and qualitatively analyzed the accuracy and representativity of the estimate. The analysis found that for some commodities, such as copper and gold, the datasets should be representative of the global mining industry, whereas for other commodities, such as phosphate and tungsten, the datasets may not be representative. Section 4.6 undertook sensitivity tests of the estimate, and found similar conclusions to the qualitative review described in Section 4.5.3. Section 4.7 discussed the significance of the results of the preceding sections on a commodity-by-commodity basis and Section 4.8 discussed commodities not included in the study.

Overall, these estimates are likely conservative, as mines which report their water usage are most likely to be owned by companies than have water management programmes and work to reduce their water withdrawals.

To put these water withdrawal figures into perspective, the United Nations states that people need at least 20 L of water/day to meet the most basic human needs, although many people in the world live on less (United Nations Development Programme, 2006). Thus, the mining water withdrawals from 2006-2009 identified in this paper equals the minimum water requirements of 0.8 to 1.1 billion people. However, water use by mining must be kept in perspective. Mining takes a small percentage of overall water withdrawals. As discussed in Section 2.2, even in mining intensive countries such as Australia, Canada and Chile, mining constitutes less than 5% of national water withdrawals. As shown in Table 2.1 and Table 2.2, global water withdrawals due to mining are of the same order of magnitude as agriculture in Australia in 2008-09 and only twice as large as water withdrawals by manufacturing in Canada in 2009. It is also important to recognize that mining provides an essential material to modern civilization and the value of the production of the commodities included in this study ranged from $386 to $471 billion dollars per annum.

Caution needs to be used when comparing individual mines' water consumption with industry averages or best practices. The realities of local climate conditions, ground water conditions, process requirements and so on may mean that a mine has no option but to withdraw comparatively high rates of water. Sound water practices can mitigate high withdrawal requirements. It is also important to strike a balance between efforts to improve water efficiency with energy efficiency and capital expenditure. For example, it may not be reasonable for a mine in a rainforest to spend significant amounts of capital on reducing evaporation rates.

This study represents the first systematic effort to quantify water global withdrawals due to mining. As efforts such as the GRI become more widespread, it should be possible to increase the accuracy of these estimates in coming years. This approach could be extended to energy use as well as other key indicators relevant to the mining industry.

5. CASE STUDY – CERRO VERDE CONCENTRATOR FRESH WATER SYSTEM

The objectives of this chapter are to undertake a practical case study of how the mining industry uses water and designs water systems, and to describe the water system requirements of a typical mine.

5.1 Introduction

Cerro Verde is a low-grade copper porphyry mine, located near the city of Arequipa in southern Peru, at an elevation approaching 2,800 m. In 1972, one of the world's first copper Solvent Extraction – Electrowinning (SX/EW) facilities was built on the site, processing oxide ore from two main open pits. In 2004, the operator, Sociedad Minera Cerro Verde S.A.A. (SMCV), approved a 108,000 t/d expansion project to process sulfide ore reserves through a new concentrator plant. The plant was designed under extremely tight schedule and capital constraints, with a mandate to intentionally avoid any overdesign.

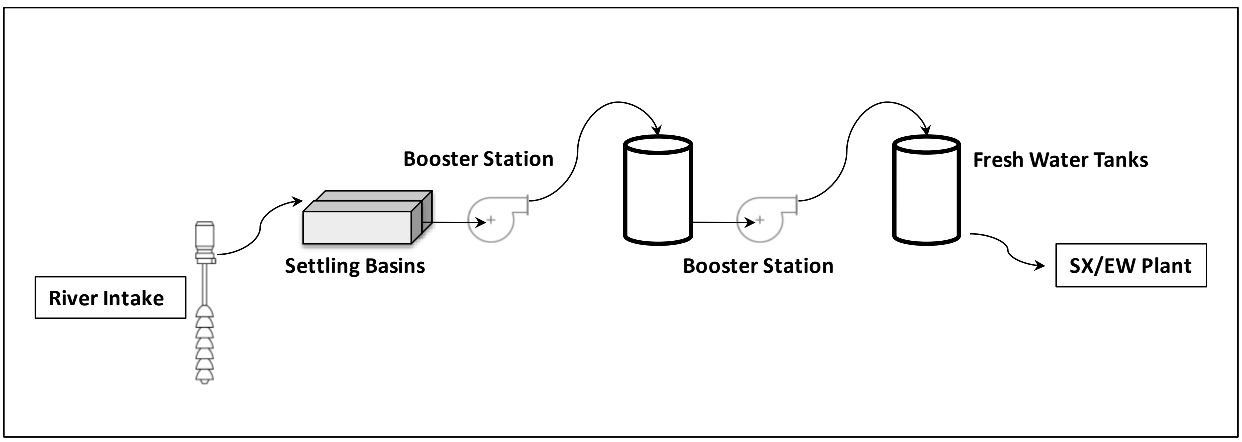
The concentrator required the site to significantly increase water withdrawals, although the site would reuse or recycle all available water and continue to practice zero discharge. Additional water rights were secured through participation in a reservoir project upstream of Arequipa. Water came from the Chili River (Río Chili), also the main source of water for the SX/EW plant. The river intake was located downstream from Arequipa, a city with close to a million residents. The city had little sewage treatment facilities and most of the agriculture in the area was supported by irrigation water, with runoff water returning to the river. Furthermore, Arequipa supported a range of industries, and water treatment of industrial effluents was known to be weak. Thus the water supply of the proposed plant was in essence untreated domestic, agricultural and industrial wastewater.

5.2 Feasibility Study

5.2.1 Pre-existing SX/EW Fresh Water System

SMCV's SX/EW fresh water system consisted of an intake channel and pumping station on the Río Chili, followed by two small settling basins, feeding two booster pump stations in series. The system discharged approximately 360 m^3/h of river water into 5 holding tanks located close to the SX/EW plant, over a distance of more than 10 km and with an elevation gain of over 700 m. Figure 5.1 shows a simple schematic of the system. The system had several ongoing operational problems, including the need to shut down occasionally due to excessive solids levels in the river during the rainy season and high wear due to abrasive fine suspended solids.

Figure 5.1 SX/EW River Water System



5.2.2 Primary Sulfide Project Feasibility System

The main sources of water for the proposed plant were to be fresh water from the Río Chili, recycle water from the tailings thickener overflow, and recycle water from the TSF (referred to as reclaim water). The feasibility study outlined the necessary changes required to the SX/EW fresh water system in order for the system to provide enough water for the proposed concentrator. The second booster station of the SX/EW system had to be relocated due to the location of the TSF; instead of building a new pump station for the SX/EW plant and a separate system for the new plant, the systems were twinned. In the feasibility study it was recognized that there would not be enough area for new settling basins that could handle the increase of flow required by the concentrator; however, settling basins were included in the cost estimate to ensure some funds would be available for the chosen treatment system. The feasibility study left the specification of the water treatment system to be defined during detailed engineering.

The plant water system was to be a traditional design, with the fresh water tank overflowing into the process water tank and all water consumers being fed by gravity. The water systems of the plant were assumed to be open flow systems, with all water, including cooling water, passing to the process slurry, and then being recycled through the reclaim and recycle water systems.

5.3 Detailed Design

Once detailed engineering began, a more thorough evaluation was undertaken to design the plant water systems. The plant's fresh water requirements needed to be defined in detail as well as the actual fresh water quality available in order to allow the detailed design of the plant water systems, including any required water treatment.

5.3.1 Fresh Water System Requirements

The fresh water requirements were defined as follows:

- The system must provide 3,600 m^3/h of water;
- The system must provide water of an acceptable quality for the plant;
- The system must fit in the available area;
- The system must not require extensive capital or operating costs; and
- The system must not use any chemical addition.

The design flow of 3,600 m^3/h was largely determined by the makeup water required to replace the estimated amount of water lost to evaporation or retained in solids in the TSF.

Acceptable water quality requirements were determined by reviewing all of the vendor requirements of water consuming equipment. Key fresh water consumers were as follows:

- Process makeup water
- Mills, crushers and air compressors' cooling water
- Pump GSW
- Reagent mixing and dilution water
- Spray and wash water
- ADS dust suppression
- Road dust suppression
- HVAC cooling water

The design philosophy was to compile the strictest fresh water quality criteria out of all the plant mechanical and electrical equipment that required water to set the overall fresh water quality requirements. The strictest water quality criteria were as follows:

- Maximum particle size: 25 microns
- pH range: 6.8 – 8.0
- Total Dissolved Solids (TDS): <750 mg/L (<960 μS/cm)
- Total Suspended Solids (TSS): <10 ppm
- Chlorides: <150 ppm; Free Chloride (Cl_2) max: 0.5 ppm continuous
- Sulfates (SO_4): <50ppm
- Carbonate Hardness: 50 – 200 ppm $CaCO_3$; Calcium Carbonate ($CaCO_3$): 10 ppm

- Ammonia: <0.5 ppm
- Copper, Iron, and Manganese: Each <1 ppm
- Organic Contaminants: No algae or oil allowed
- Any residual reagents (especially coagulants) added for treatment must be minimized in the treated water discharge to minimize impact on downstream users.

The fresh water supply also needed to be large enough and cool enough to supply the major plant cooling systems, detailed in Table 5.1. For the Cerro Verde system, 8881 kW of heat was removed with 1014 t/h or m^3/h of water, increasing the temperature on average by 7°C. The allowable temperature range varies with the design of the heat exchanger system, but for mineral processing equipment typically ranges from 5°C to 10°C.

Table 5.1 Major Cooling Systems at Cerro Verde

Equipment	**Installed Power (kW)**	**Cooling Requirements (kW)**	**Water Requirements (m^3/h)**
Tertiary Crushing Bearing Systems	20 000	279	24
Tertiary Crusher Lubrication Systems		605	52
Tertiary Crushing Electrical System		930	80
Ball Mill Lubrication Systems	48 000	800	172
Ball Mill Electrical Systems		632	54
Ball Mill Gearless Drive Systems		2093	360
Air Compressors Cooling	3000	3062	176
Auxiliary Cooling Systems	N/A	480	96
Total	71 000	8881	1014

The original cost estimate was based on settling basins as per the old system. The estimated operating and capital costs for the feasibility settling basins was quite low, which led to significant pressure to design a low cost system for water treatment.

The available areas for a fresh water treatment plant consisted of a narrow band of land directly beside the river and a somewhat larger area on the cliffs above by the SX/EW settling basins.

The area beside the river was extremely constricted, bounded on one side on by the river and the other by 35 metre cliffs. The area available above was bounded by the cliffs on two sides and by agricultural areas on the other sides.

The requirement that there be no chemical addition was due to two main considerations:

- The SX/EW system, which returned the settled water sludge to the river, as permitted, was originally expected to continue. Any reagent addition to a water treatment system could require modification to the existing permit.
- The SX/EW plant operators did not want any chemical additions to the water, such as coagulants, believing reagents could negatively impact the operation of the SX/EW plant.

5.3.2 Fresh Water System Feed Water Quality

Due to the presence of raw sewage and other contaminants from Arequipa, the river water quality was known to be poor, and to change day by day, if not more frequently. River fecal coliform levels were measured in the hundreds of thousands to millions per 100 mL. For comparison, maximum coliform levels for drinking water is less than 1 per 100 mL, and treated sewage discharge should be under 2000 per 100 mL. The river had a noticeable odor and visually was not clear or clean.

Getting representative Río Chili water samples for water system design was difficult. SMCV took monthly water samples from the river, measuring water chemical composition, total dissolved solids, total suspended solids, and a few other parameters. However, there was no detailed analysis of the characteristics of the solids in the water, with the exception of one grab sample.

Table 5.2 shows basic information obtained from the monthly water samples. These ranges of concentrations were generated from water sampling data obtained between January 2003 and January 2005 at two sampling locations on the Río Chili and one sampling location at the river pump station. The numbers do not reflect the water quality at any one time, but the extreme and average values for different variables over time. Fecal coliforms were not measured during this sampling period.

Table 5.2 Chili River Water Quality Data

		Min	Mean	Median	Max
pH		6.46	7.38	7.27	10.15
Bicarbonates	mg/L	0	70	69	240
Calcium	mg/L	12	25.1	27.9	57
Carbonates	mg/L	0	73	69	226
Copper	mg/L	0	0.066	0.033	1.28
Chloride	mg/L	38	89	84	225
Hardness	mg/L	150	474	390	2500
Iron	mg/L	0.02	0.331	0.31	1.75
Manganese	mg/L	0	0.107	0.08	1.4
Sulphates	mg/L	12	77	77	183
TDS	mg/L	98	609	470	11,467
TSS-Filtered to 50µm	mg/L	1	11	9	99
Conductivity	uS/cm	480	666	660	910

Generally, the quality of water with regard to the chemical composition, pH, and dissolved solids was found to be acceptable. There were some high peak values that were assumed to be outliers. However, it was clear that most of the suspended solids would need to be removed, especially as the river was known to have significant sand loading during the rainy season.

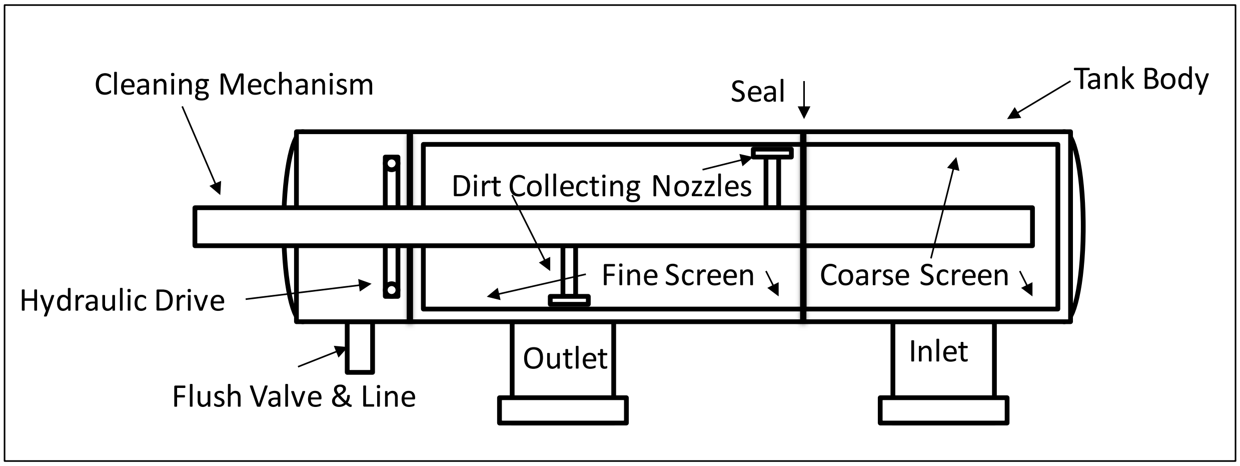
5.3.3 Water Treatment System Design

Common water treatment technologies were reviewed alongside the water quality of the source river water. Finding an acceptable technology that met all the criteria proved to be difficult. Options studied included settling basins, sand filters, clarifiers, hollow fibre micro-filtration, and in-line mechanical filters or strainers. The study estimated that there would not be enough area, given the new flow, for settling basins or sand filters. Clarifiers were dismissed, because for effective operation they required chemical addition. Micro-filtration was dismissed as being far too expensive. Mechanical self-cleaning filters were chosen as the only option which seemed to meet the criteria. Site visits were made to facilities using different self-cleaning filters in Washington State, and other users were contacted for interviews. Users stated that after the filters were correctly installed, they were generally effective and required little maintenance.

A recommendation for a trial with a test filter on the Río Chili was made, but was rejected as it was known that the river quality varied widely over the year and that any short test would not have been representative. In retrospect, this was a mistake. A test at any time of year, with a small test filter, would have shown the problems with the high level of organics in the water, and alternative technologies would have needed to be selected.

The chosen self-cleaning filter worked as follows. Each of the strainers consisted of an approximately 2ft diameter tank body, an internal cylindrical screen made up of a coarse pre-screen section on the inlet side and a fine section on the outlet side, inlet/outlet flanges, a cleaning mechanism and a flush valve. The cleaning mechanism was a pipe running down the center of the tank body with dirt collecting nozzles attached perpendicular to the pipe, leading to the face of the fine screen. Water entered the inlet flange into the body of the filter and passed through the coarse screen, which provided protection to the fine screen. The dirty water flowed through the inside of the cylindrical screen to the fine side, and then passed through the fine screen from the inside out. A seal prevented dirty water from bypassing the screens. Dirt was collected on the inside surface of the fine screen and the clean water passed through the outlet. A schematic drawing of the filter is shown in Figure 5.2.

Figure 5.2 Self-Cleaning Filter Schematic



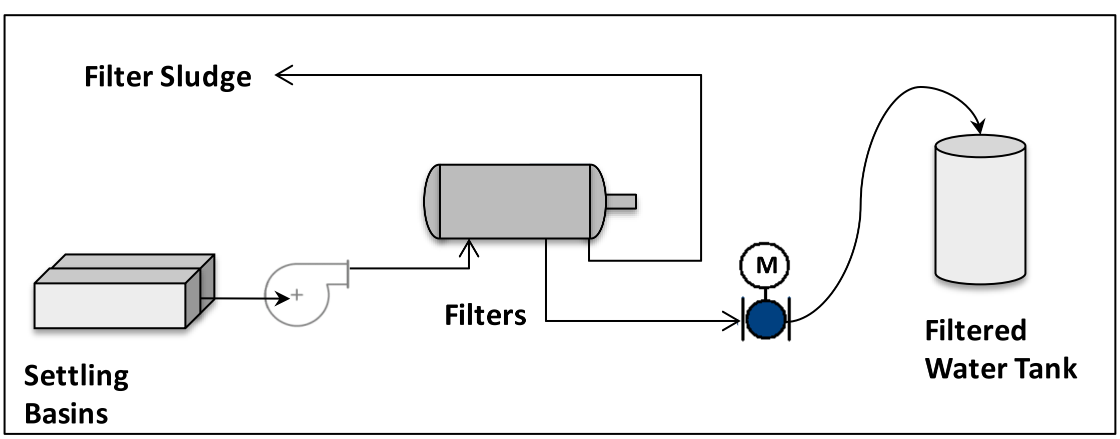
As the screened material built up on the surface of the fine screen, a pressure drop developed across the screen. When this differential pressure reached 7 psi, as measured by a pressure differential transmitter, the flushing cycle was initiated by the control system. Since the pressure in the discharge line was at atmosphere, when the flush valve opened a differential pressure was created which drove the cleaning mechanism. The dirt collector nozzles removed the dirt from the fine screen, also drawing water in to create a slurry discharge. As long as the system pressure was above 35 psi, sufficient pressure would be applied to clean the screen. This discharge flowed through the cleaning mechanism, through the hydraulic drive and out the flush line, causing the cleaning mechanism to spin. The reverser at the end of the cleaning mechanism ensured that the mechanism moved back and forth across the fine screen. A rotation governor attached to the cleaning mechanism ensured that the mechanism did not move too quickly. The reverser indicator provided visual indication that the cleaning mechanism was working properly. The balance piston line balanced the transverse pressure on the assembly and powered the reverser indicator. The filters would continue to supply water during a flush cycle.

With the selection of the technology, detailed design commenced. A key problem was obtaining the 35 psi system pressure required to effectively flush the filters. The filters needed to be located before the first main booster pump, which was to be located near the SX/EW system settling basins, or the booster pumps and pipes would need to be modified to handle high suspended solids levels. In addition, space was extremely limited at the main plant site. Locating the filters immediately beside the river would have provided adequate head pressure; however the space and access constraints made this unworkable. Thus, the filters were to be located by the settling basins, above the river.

A holding tank for the filtered water was included to provide a break in the system, ensuring better system control (see Figure 5.3). The SX/EW settling basins were incorporated into the system to provide limited pretreatment for the filters, by allowing the coarsest particles to settle.

In order to provide back pressure for the filters, an automated control valve was added before the tank. When the pressure differential built up across the filters, indicating the filters needed to be cleaned, the control valve would partially close and the variable speed filter feed pumps would speed up until the system pressure reached 35 psi. When the differential pressure reached 7 psi, the filters would flush. Based on the water quality information available and past vendor experience, flush times were estimated to be on the order of once every 20 minutes. Filtration was provided to 50 µm, with addition filtration occurring at select locations in the plant.

Figure 5.3 Water Treatment System



After a review, returning the sludge to the river was determined to be an unacceptable practice. A thickener was included to collect the filter and settling basins' sludge and recirculate the sludge until a higher density was reached. Once adequate sludge had been collected, a tanker truck would haul the material to the tailings storage facility for disposal, ensuring a zero discharge system.

5.4 Pre-Operations Testing

The concentrator water system needed to begin operation a few months before plant startup, as the TSF pond needed to be filled before plant startup could commence. Once construction and installation of the system was completed, pre-operation and start-up testing began.

Problems arose almost immediately. The differential pressure across the filters was far higher than the 7 psi expected and the filter pumps immediately jumped above the pump curve. The automated valve closed more slowly than anticipated, taking around five minutes to open or close. Water flow was well below the expected rate. Light material, such as plastic and corn, was not removed in the settling basins and quickly built up on the pre-screens (see Figure 5.4).

Figure 5.4 Dirty Filter Pre-Screen

Testing soon found that with the automated valve mostly closed, enough back pressure could be created to “power flush” the filter fine screens clean. However, the power flush required the filter pumps to pump low flows at high speeds, off their curves. The control valve actuator was changed to speed up the open/close time to increase the system availability. The filter vendor was brought to the site for consultation and, in addition to recommending test work with a lab-scale filter, the vendor suggested additional prescreening before the filters to remove the light coarse material. To fill the TSF pond, two of the filter bodies were removed, and untreated water was pumped to the tailings facility.

Work began immediately to procure a travelling screen on the river intake channel. The vendor returned with a test filter to complete the test work that had been proposed during detailed design. Based on the test work, and the number of filters available, the vendor recommended 300 μm screens be installed, not the 50 μm screens originally specified. The test results effectively meant that the system should have had at least 3 times as much screen area as was originally specified. The problems that arose with the filter technology were likely not related specifically to the vendor used. Any similar straining technology would have had the same problem of the apertures getting blocked almost immediately by organics.

A traveling screen was manufactured and installed on the inlet channel to the pump station on the river (see Figure 5.5), and the 300 µm screens were ordered. By this time, the concentrator plant needed water for pre-operations and commissioning. However, even with the 300 µm screens, flow was still well below the design specification.

Figure 5.5 Traveling Screen on Intake Structure

A series of tests with the filters were undertaken to maximize available flow. The tests involved increasing the filter pump speed and finding the optimum open position for the control valve to achieve the highest flow rate, yet still be able to relatively quickly close the valve and power flush the filters clean. Peak flows of over 4000 m^3/h were achieved; however, overall flows, including reduced flows during flushes, were only a little over 2000 m^3/h. Operations, on average, achieved even less flow. In addition, fine organics were building up on the inside of the pre-screen (see Figure 5.6), requiring the filters to be disassembled and cleaned at least once per day.

Figure 5.6 Organic Buildup on Clean Side Of Pre-Screen

The filter sludge thickener was commissioned and it was found that residual flocculent in the overflow, which returned to the settling basins feeding the filters, further reduced the capacity of the filters. The overflow was redirected to the filtered water tank, effectively increasing the amount of treated water produced by close to 360 m^3/h, or 10% of the requirement.

With further improvements to the actuator on the control valve, combined with the thickener overflow redirection and filter operation optimization, it was estimated that flows may have approached 75% of design flow. This maximum flow would have impacted plant production. Furthermore, the water filtered to 300 µm was not of high enough quality. Fresh water consumers in the concentrator, such as the cooling systems for the ball mills and tertiary crushers, were already blocking due to remaining organics in the water.

Temporary solutions included installing additional duplex and cartridge filters on critical lines within the plant, and redirecting process water to the cooling systems, instead of the river water. The tailings thickeners overflow water made up the bulk of the process water, and while the overflow water had a high pH level, it was likely the cleanest water available, especially during startup. However, high sand levels in the process water would occasionally cause difficulty in the cooling equipment, and it was clearly not a sustainable solution.

5.5 Plant Startup

With the plant close to startup, a timely and more effective solution was required. Several options were studied including:

- Installing another bank of filters;
- Moving the filters closer to the plant to only treat the fresh water, leaving the makeup water to the process water system and the SX/EW water untreated;
- Pumping the river water to the TSF and using reclaim water. The water which would be treated by the lime in the tailings and any sediment would have time to settle in the pond. The SX/EW plant would still need river water pumped to its tanks;
- Using different types of filters, such as sand filters;
- Using groundwater from mine dewatering wells;
- Replacing the filter station with a clarifier system or a flotation system; and
- Putting the cooling systems on closed loops, removing heat with cooling towers or other cooling technologies.

The temporary solution agreed upon consisted of relocating the filters closer to the plant and rearranging the piping so that the filters could treat either reclaim water or river water as necessary. Figure 5.7 and Figure 5.8 show the original and modified designs. The filters were moved to an elevation where the system pressure was high enough not to require a control valve. A separate line took water from upstream of the relocated filters to the SX/EW plant. Water from the reclaim system or the fresh water system could be directed to either the fresh water tank or the process water tanks. Studies were undertaken to ensure the relocation would not negatively impact the hydraulics of the system and it was found to be acceptable. Some river water would also need to be sent directly to the TSF to make-up the water balance.

The other options were deemed unsatisfactory for the following reasons:

- Another bank of filters may have allowed enough water flow, but did not solve the remaining water quality problem. Additionally, the lead time for delivery of new filters was significant.
- Pumping all of the river water to the TSF would require a far larger reclaim water pumping system, some river water would still need to be pumped to the SX/EW plant, and the final water quality still may not have been acceptable.
- Different types of filters were deemed likely to have similar problems to the current filters. As studied originally during the design phase, other filter types would require too much area or too much capital, or both.
- An adequate flow of groundwater was not available and the lead time was too great.
- Replacing the filters with a clarifier or flotation system would take to long too install immediately, but was evaluated as a long term solution.
- Closed or partially closed cooling loops would take too long to install immediately, but were also evaluated as a long term solution.

Figure 5.7 Original Water System Design

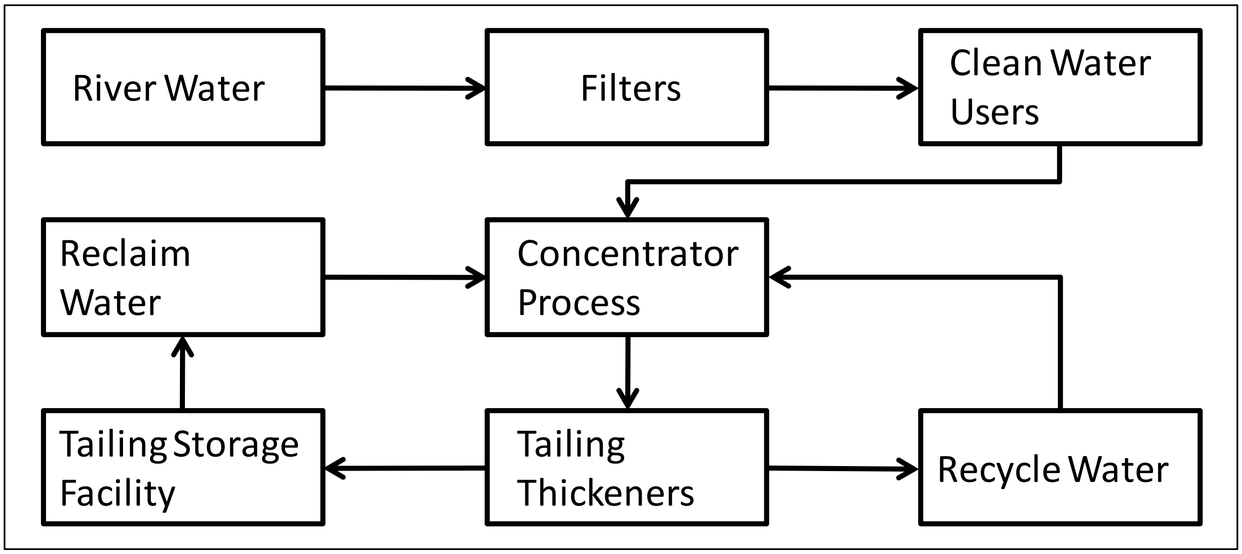
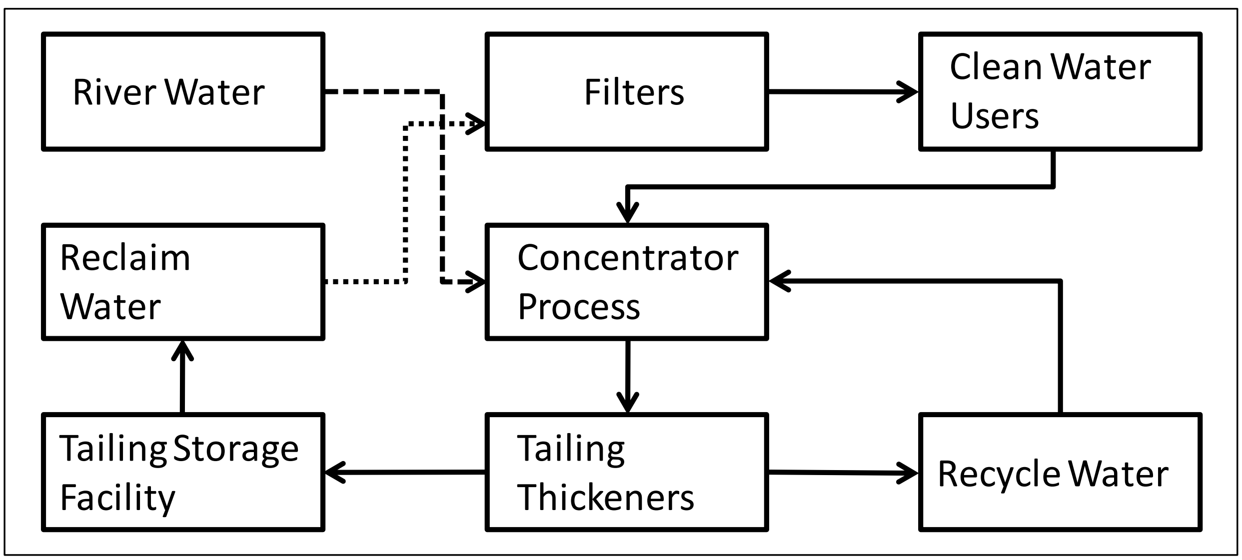


Figure 5.8 Upgrade Water System Design



The relocated filters were re-commissioned Figure 5.9 and found to generally perform as designed. The reclaim pumps had some deficiencies which did not allow them to pump at capacity, but this was remedied by the vendor. The reclaim water occasionally had very high solids loading, but improved management practices of the TSF generally ensured a higher water quality. The filters still required regular maintenance, but with fewer difficulties than in the previous location. Significantly fewer problems were found in the cooling systems.

Figure 5.9 Re-Located Filter Station

5.6 A Longer Term Solution

The concentrator soon achieved full production, with the upgraded system providing an adequate supply of water. However, the upgrade was determined to be inadequate for a plant with an expected 30 year life. First, the filter relocation meant that river water with high solids content was being pumped through the two booster stations, which would lead to high wear on the pipe lines and high maintenance on the pumps over time. In addition, water quality still did not meet the vendor specification, especially for the cooling systems, and would thus likely require more maintenance than was desirable.

The full water treatment and closed loop cooling options evaluated prior to relocating the filters were studied in considerably more detail. Several different cooling technologies and treatment systems were evaluated. The studies concluded that a high efficiency clarifier system or a closed cooling tower system were the most optimal for the concentrator. The clarifier option was recommended, as it addressed the root problem of poor water quality. It reduced the risk of needing to replace parts of the fresh water system due to wear, addressed the water quality for other clean water consumers in addition to the cooling consumers, and it required the least amount of redesign within the plant.

Section 5.6.1 describes the cooling system options evaluated in more detail.

5.6.1 Cooling System Options

A cooling system must eliminate heat from the mill equipment. Cooling systems on major equipment usually have tube-based heat exchangers which remove heat from hot air, lubrication oil or water, and transfer that heat to a cooling medium such as water, or a water/glycol combination. After the heat from the major equipment is transferred to the cooling medium, the added heat must be removed. There are several types of heat removal methods for cooling systems, five of which are discussed here: cooling towers (closed or open), chillers, dry air coolers/fans, hybrid systems, and open cooling systems.

Wet cooling methods, such as cooling towers, remove heat through evaporative heat transfer by applying water and air to the outside of finned tube heat exchangers. Open cooling towers use the evaporative water as the cooling medium, where closed systems separate the evaporative water from the cooling medium. Evaporated water must be replaced regularly. The key advantage of wet cooling methods is significantly reduced operating and capital costs compared to chillers and air coolers. In addition, the method requires relatively small amounts of makeup water, and thus is largely independent of the water system. The main disadvantages include the need for water treatment, the need to add bactericides and an overall increase in plant water consumption due to evaporation. Open cooling towers are inexpensive, but the re-circulating water can potentially be fouled by dust or contamination build-up. Closed cooling towers reduce the likelihood of fouling, but have higher capital and operating costs.

Chillers also work through evaporative heat transfer, using a refrigerant in a closed loop as the cooling medium. When the refrigerant evaporates, it can remove much more heat from the cooling medium than water or glycol systems. After evaporation, the gas must be compressed and condensed, which requires additional air or water cooling. Air-cooled chillers have a small footprint and enable the use of a cooling system that is independent from the plant water system and largely independent of ambient temperature. The main disadvantages include significant capital and operating costs, the need for specialized maintenance, and the use of refrigerants that are often toxic and can have negative environmental impacts.

Fans can be used to directly cool equipment, such as motors, or to blow or suck air across finned heat exchangers, as is the case for dry air coolers. Dry air-cooled heat exchangers reject heat from the cooling medium to the surrounding atmosphere by convective heat transfer. The key advantage of dry air coolers is that the cooling system is independent of the water system. The main disadvantages are significant capital and operating costs; high operating costs are primarily due to electrical consumption from the fans. Other disadvantages include the large footprint required for the coolers, limited cooling with high ambient air temperatures, and intensive maintenance, particularly in dusty areas.

Hybrid systems combine two or more of the above methods. For example, above a specified temperature, air coolers can be sprayed with water to combine convective with evaporative heat transfer. Alternatively, a small set of chillers could be included on an air cooled system to deal with peak periods. Hybrid systems can lower capital and/or operating costs and can increase the range of operating conditions for a cooling system. However, they are still significantly more expensive than cooling towers and are relatively complex systems to run and maintain.

Open cooling systems, or once-through water systems, pass water through the equipment heat exchangers, and then discharge the water, either into other processes or into the outside environment. Key advantages to once-through water systems are simplicity and low capital and operating costs. The main disadvantages are reliance on adequate water quality, water treatment costs, water system reliability, and high water consumption rates.

The capital and operating costs of the different types of cooling systems are highly site dependent. Table 5.3 shows a ratio of capital and operating costs for different options, based on the study completed for Cerro Verde. This table shows that the capital cost of an air-cooled chiller-based system is roughly five times that of a cooling tower based system, and the electrical costs are sixteen times that of an open cooling water system. At Cerro Verde, the installed power for a chiller system would have been 3300 kW, or almost 5% of the installed comminution power. Hybrid systems are not included in the table, as there were several possible combinations and the operating and capital costs were considerably higher than for cooling towers or open systems.

Table 5.3 Ratio of Capital and Electrical Operating Costs for Different Cooling Systems

	Cooling Towers	Dry Air Coolers	Chillers	Open Water
Equipment Capital Costs	1	4	5	<1
Electrical Operating Costs	2	13	16	1

Open water-based cooling systems are easily the simplest, cheapest and most energy efficient cooling method, provided there is a low-cost, adequate and reliable supply of suitably clean and cool water.

5.7 Conclusion

In order to answer questions on how to reduce and improve mine water use, it is critical to have detailed information on typical mine water system requirements as well as an understanding of the practical considerations faced during mine water system design. Thus, the objectives of this chapter were to undertake a practical case study of how the mining industry uses water and designs water systems, and to describe the water system requirements of a typical mine.

During detailed engineering the design requirements of all water consumers on the site were specified. A water treatment system was chosen for a required flow and water quality. The system was required to have a small footprint, to have a low capital and operating cost, and to be chemical-free. Mechanical self-cleaning water filters, or strainers, were chosen as the main water treatment method.

Upon startup of the water system, water quality was worse than expected. Organic material blocked the filters, leading to low flows, even when larger screen openings were installed. The water quality data did not adequately characterize the makeup of the solids, which was the source of the problems rather than the actual quantity of solids.

In order to ensure a supply of water prior to and during commissioning, the following steps were undertaken:

- A travelling screen was installed upstream from the water treatment system;
- The treatment system was temporarily bypassed in order to fill the tailings storage facility starter pond;
- A test filter was installed and, after a series of tests, the original 50 µm screens were replaced with 300 µm screens;

- The filter control strategy was improved to maximize flow, in conjunction with replacing a slow-acting actuator;
- The sludge clarifier overflow was redirected to the filtered water tank to reduce the load on the filters;
- Additional filters were installed on critical lines in the plant; and
- In the concentrator, process water was diverted to some of the plant raw water consumers.

As an intermediate solution, the filters were relocated to the reclaim water, allowing the system to meet design flow rates. In addition, a clarifier-based full water treatment plant as well as maintaining the existing open cooling system was recommended as a long term solution.

Lessons learnt from the experience of starting the Cerro Verde water system include:

- If the plant water supply may be a problem or is known to be of poor quality, ensure that the water treatment system is addressed as early as possible in the project;
- Undertake laboratory or pilot testing of any key technology in the water system. A water treatment system should be treated as a key unit operation in a plant;
- If possible, avoid integrating an old water system with a new one. In a quote attributed to mineral processor Bob Shoemaker: "Based on my years of experience in the mining industry, it's my opinion that used equipment is only slightly more expensive than new equipment (Keane, 1998)." Reusing as much as possible of the existing SX/EW water system saved a significant amount of capital on paper, specifically several kilometers of 12" pipe. In reality, the reuse and twinning of the system led to compromises which reduced design and startup flexibility. Specifically, it led to the selection of reagent-free mechanical treatment methods rather than more appropriate clarifier systems, and it

required the new system to be running efficiently almost immediately after startup to ensure the SX/EW plant was not negatively impacted. Maintaining two separate systems, perhaps with an option on a tie-over line, would have given much more flexibility;

- Filtered reclaim water was found to be adequate as a short term measure to supply clean water to the concentrator; and
- When choosing a water source for a concentrator, look at all sources of water available. Using the Río Chili downstream of Arequipa, essentially a wastewater stream, for the raw water supply of the concentrator was a responsible decision given that there was no attractive alternative. As long as adequate water treatment is provided, using wastewater or water of poor quality in areas where there is a lack of water resources may be a sound design choice to limit mining's impact on a region's water systems.

SMCV committed to assisting with the construction of a potable water plant, which was completed in 2012 at a cost of $US 99 million (Andina, 2012). This investment was a good, long-term approach to problem solving with the community, by working jointly with the local government to solve a real health and environmental problem that impacted much of the region's population. SMCV demonstrated an understanding of community issues and the investment seems to be an effective use of funds. In addition, SMCV reached an agreement to build a wastewater treatment plant in Arequipa should it proceed with plans for a large-scale concentrator expansion, which would be an important step toward helping Arequipa to improve the water quality of the Río Chili (Freeport-McMoRan, 2011).

The literature review undertaken in Chapter 2 found there was little public literature available on the design requirements of a typical mine water system. This chapter described a practical case study of how the mining industry uses water and designs water systems and also provided the water system requirements of a typical mine.

6. REDUCING MINE WATER REQUIREMENTS

The objective of this chapter is to model the potential impact of different options to reduce, reuse and recycle water on mine site water use. Section 2.5 reviewed past efforts to reduce, reuse and recycle water and Section 3.3 describes the methodology used in this chapter. Section 6.1 introduces the model and Sections 6.2 to 6.7 describe the different scenarios and then calculate the amount of potential water savings. Section 6.8 describes the limitations of the approach and Section 6.9 concludes the chapter.

The plant water withdrawals are in line with typical copper concentrators, as recorded in Chapter 4. The site water sources and consumers are comparable to that described in the case study, as recorded in Chapter 5.

6.1 Mine Water Reduction Model

The following model is used to quantify the potential individual and combined impacts of several of the key water reduction options discussed in Section 2.5. The model describes potential water savings for a hypothetical 50,000 tonne per day (tpd) low grade copper deposit in an arid region. The mineral processing plant uses conventional froth flotation separation to process 50,000 tpd of copper sulfide ore, with a grade of 0.5% Cu. The final concentrate grade is 28% copper, with a 90% copper recovery. Figure 6.1 shows a simplified flow sheet describing the process. The ROM ore is crushed and ground with a gyratory crusher and a SAG Mill circuit with two ball mills. The copper sulfide is then separated from the gangue in a flotation circuit at 30% solids. The flotation circuit consists of two lines of ten 160 m^3 rougher/scavenger cells, a line of ten 80 m^3 cleaner/scavenger cells, two 5 m diameter cleaner columns and a regrind mill. The copper concentrate is dewatered in a 15 m diameter thickener, followed by a pressure filter. and then shipped off site. ROM ore typically has a moisture content of 2% to 5%, even in arid locations. In this model, the ROM moisture content is assumed to be 2%. The major process ore and slurry flows are shown in Table 6.1. The plant flow sheet is loosely based on a SAG mill design described by Vanderbeek *et al.* (2006).

Figure 6.1 Basic Process Flow Sheet

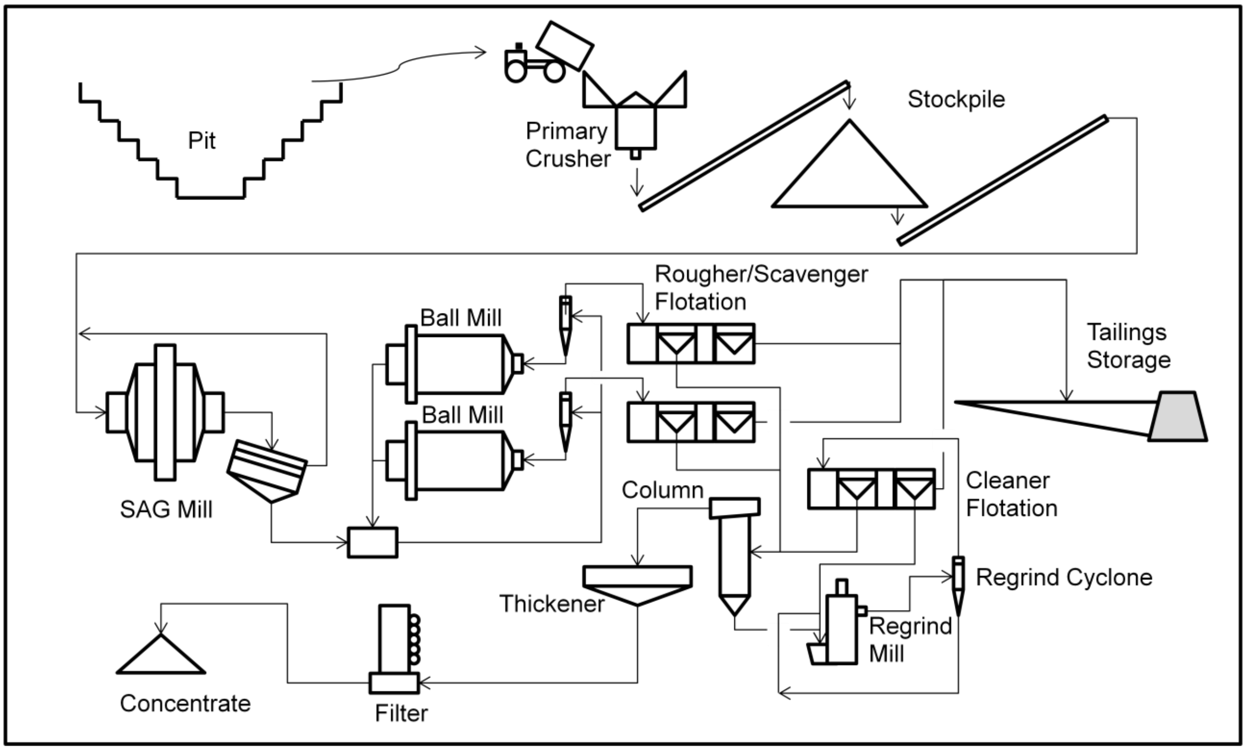


Table 6.1 Major Ore Flows in Mill Model

Process Slurry Flows	Solids Content (tpd)	Solids Content (%)	Water Content (m^3/day)
ROM → Comminution	50,000	98	1,020
Comminution → Flotation	50,000	30	116,667
Flotation Concentrate → Concentrate Dewatering	804	30	1,875
Concentrate Dewatering → Final Concentrate	804	90	89
Flotation Tailings → Tailings Storage Facility	49,196	30	114,792

Figure 6.2 and Table 6.2 detail the major water consumers at the site, including: the flotation process water, SAG mill, ball mills, air compressor cooling water, froth wash water, pump

GSW, reagent mixing water, dust suppression, and hose stations for mill cleanups. In the mine, water is primarily used for haul road dust suppression and in the truck maintenance shop. In the office, water is used for washrooms, showers, food preparation, and drinking. For this model, the maintenance shop water consumption is assumed to be negligible, as it is assumed that the use is contained and water is reused after being treated by removing waste sand and oil. Wash water from the hose stations is assumed to be negligible. All process areas with hose stations are assumed to be contained, with sump pumps returning any water and material back to the process where the spill originated. The only available sources of water are assumed to be the moisture in the ROM ore and a nearby river. Precipitation and mine dewatering flows are assumed to be negligible.

Figure 6.2 Mine Site Water Sources and Consumers

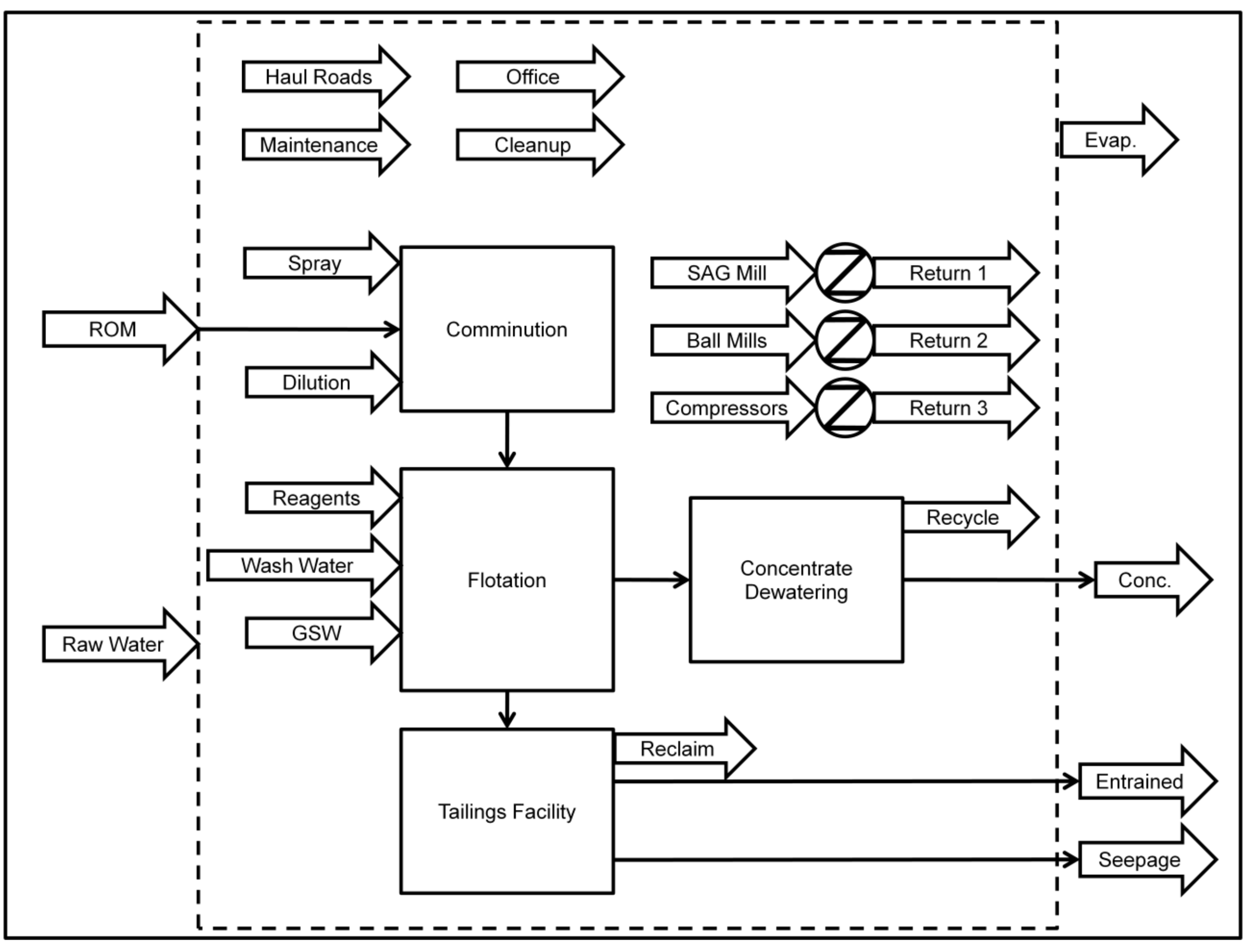


Table 6.2 Major Water Consumers

Major Water Consumers	Flow (m^3/d)
Flotation Process Water (30% solids by mass)	116,667
SAG Mill Cooling Water	4,100
Ball Mill Cooling Water	4,100
Compressor Cooling Water	4,100
Road - Dust Suppression	3,520
Froth Wash Water	2,880
Pump GSW	1,440
Reagent Dilution Water	720
Primary Crusher Dump Pocket - Dust Suppression	358
Coarse Ore Stockpile - Dust Suppression	121
Mine/Mill/Office Staff - Domestic Water	58.1
Maintenance Shop	0.0
Hose stations - clean up	0.0

Table 6.3 outlines the six scenarios used to quantify the impact of different options to reduce mine water withdrawals. The scenarios are described in detail in the following sections.

Table 6.3 Water Reduction Scenarios

#	Title	Scenario Description
1	Base Case	Conventional 50 kT/d Cu Concentrator
2	Base Case with Water Conservation	Conventional 50 kT/d Cu Concentrator Reducing Evaporation Losses.
3	Paste tailings Case	Conventional 50 kT/d Cu Concentrator with Paste Tailing Disposal
4	Filtered Tailings Case	Conventional 50 kT/d Cu Concentrator with Filtered Tailing Disposal
5	Ore Pre-sorting Case	50 kT/d Cu Concentrator with 20% Ore Rejection
6	Combined Water Reduction Case	50 kT/d Cu Concentrator with Reduced Evaporation Losses, 20% Ore Rejection and Filtered Tailing Disposal

6.2 Scenario 1 - Base Case

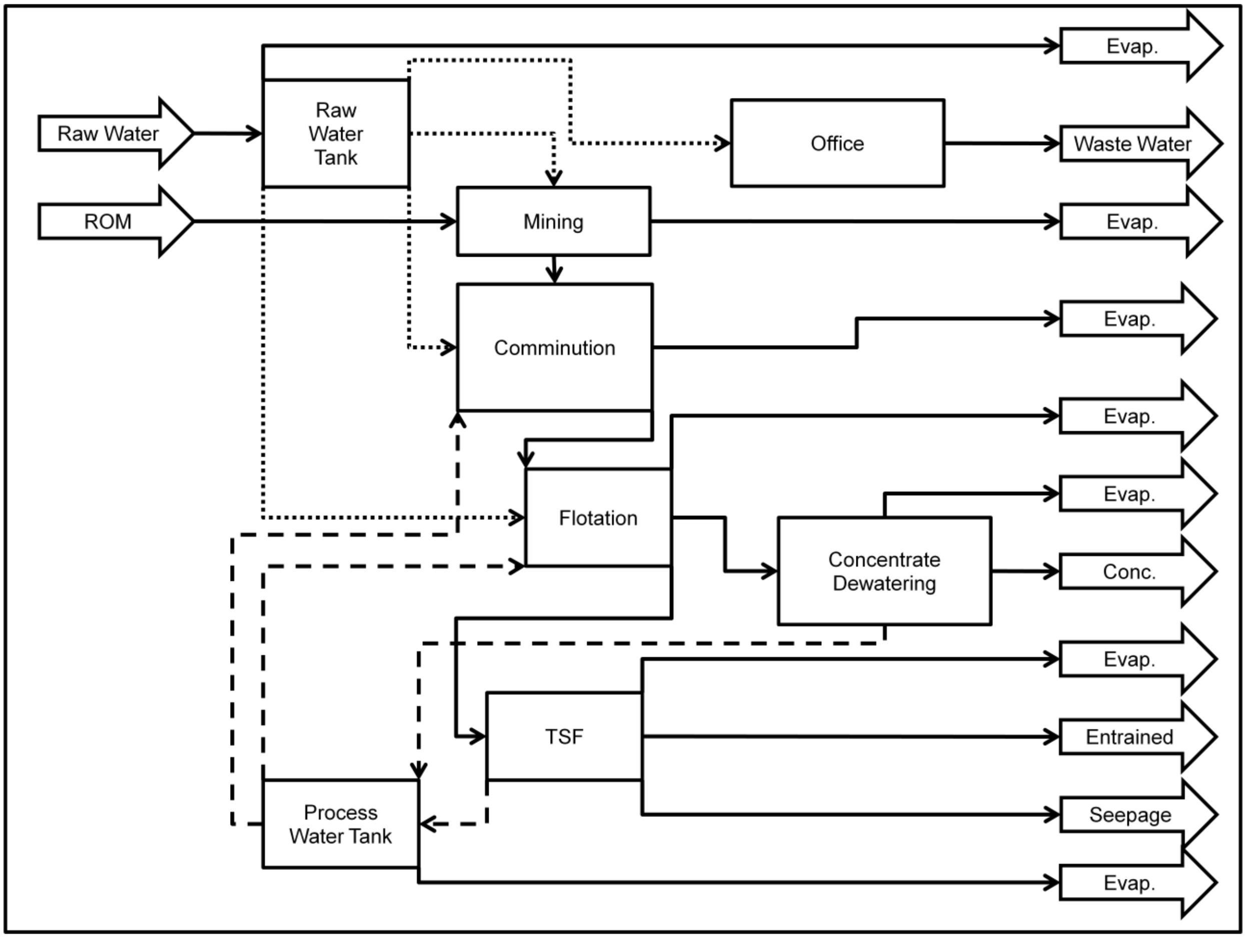
In Scenario 1, the base case, no effort is undertaken to reduce water evaporation or otherwise lower water consumption beyond reclaiming water from the TSF and the concentrate dewatering processes. Table 6.4 outlines the water sinks and the makeup water required to balance the losses.

Table 6.4 Base Case Mine Water Balance Scenario

Water Withdrawals		**Water Sinks**	
Name	**m^3/day**	**Name**	**m^3/day**
Run of Mine Ore	1,020	Road - Dust Suppression	3,520
Raw water	37,935	Human Consumption	58
		Raw Water Tank – Evap.	2.2
		Process Water Tank – Evap.	3.4
		Primary Crusher - Dust Supp.	360
		Stockpile - Dust Supp.	120
		Flotation Cell – Evap.	6.7
		Conc. Thickener – Evap.	1.2
		Final Concentrate	89
		Tailings Retained, L_{ENT}	20,792
		Beach Evaporation, L_{EVAP}	9,312
		Pond Evaporation, L_{POND}	1,890
		Beach Rewetting, L_{REW}	2,800
Total	**38,955**	**Total**	**38,955**

As shown in Table 6.2, the process water requirement for flotation is 116,667 m^3/d, based on a flotation density of 30% solids by mass. This water requirement is met by a mixture of raw water, reused raw water discharged from primary uses such as cooling systems, and recycle water from the TSF or thickeners, shown schematically in Figure 6.3. In scenario 1, the recycle water requirement is assumed to be equal to the flotation process water requirement less the water initially present in the ROM ore, and the other water consumers in the plant.

Figure 6.3 Water Flow Block Diagram



Road-dust suppression water use is determined by the area of road to be treated, the amount of water used per application, and the number of applications per time period. The application requirement is typically 1 to 2 L/m^2 and each application can be effective for a period of one hour (Organiscak et al., 2003; Tannant & Regensburg, 2001). However, dust suppression is only needed during some seasons and at some times of the day. For the purposes of the model, the mine is assumed to have 10 km of 32 m wide haul roads and 10 km of 8 m wide service roads (Tannant & Regensburg, 2001). For the base case the haul roads are watered an average of 10 times per day and the service roads are watered an average of 4 times per day. The application requirement is assumed to be 1 L/m^2 leading to a total water use of 3520 m^3/day.

On a mine site, employees and contractors use water for drinking, sanitation, bathing and food preparation. As mines are generally located outside of urban areas, they typically install water and sewage systems on site. During the design phase of a mine, typically an engineering firm will estimate water demand and design appropriately sized systems to accommodate the estimated demand. Water demand estimates vary by project and location. An environmental review for a proposed mine in Canada indicated water use of 250 liters per person per day (L/p/d) (Taseko Mines Limited, 2009), whereas the Washington State Department of Health Water System Design Manual suggests average construction site water consumption is 189 L/p/d (2009). For the base case scenario, the site is assumed to have 500 people on site, 355 of whom require access to showers. Water use is assumed to be between 120 L/p/d and 200 L/p/d for people needing access to shower facilities. Each person works an average of 240 days per year, leading to a total water use of 58 m^3/d. All waste water is assumed to be treated and disposed of through a septic system and is not available for reuse or recycling.

Any exposed body of water, such as thickeners, water tanks, flotation cells or ponds, will experience evaporation. Basic site climate information typically includes average evaporation rates, either on an annual basis or a more detailed time period basis. For this model, the average annual evaporation rate is 7 mm/d and evaporation quantities are estimated by multiplying the surface area of a body of water by the evaporation rate. The site is assumed to have a 20 m diameter raw water tank, a 25 m diameter process water tank, and flotation cells and a concentrate thickener as detailed above. The evaporation rates are thus 2.2 m^3/d, 3.4 m^3/d, 6.7 m^3/d, and 1.2 m^3/d respectively.

Dust suppression at the primary crusher and ore stockpiles is critical to reduce dust inhalation, to increase visibility, to improve maintenance and to reduce product lost to dust. A traditional dust control system consists of numerous water spray nozzles surrounding a dust generating area such as a primary crusher dump pocket. For this scenario, the primary crusher is assumed to have 30 nozzles and the coarse ore stockpile is assumed to have 10 nozzles, each spraying 1 m^3/h 50% of the time, for a total water use of 360 m^3/d and 120 m^3/d respectively. All of the water used is assumed to evaporate off the ore on the coarse ore stockpile.

The final copper concentrate is dewatered to 90% solids by mass and shipped off site, leading to a water loss of 89 m^3/day.

The bulk of water lost is in the TSF. TSF water balances can be extremely complex, with contributing variables including the climate, the geography and geology of the TSF site, the tailings mineralogy, size distribution and water content, the process plant reagents, and the tailings disposal method chosen.

Scenarios 1, 2, and 5 use a water model developed by Wels and Robertson (2003; 2004) to estimate water loss, detailed in Section 2.4.3. Table 6.5 outlines the Wels and Robertson parameters used.

Table 6.5 Modified Wels and Robertson Model Parameters

Parameter	**Units**	**Value**
Specific Gravity of Tailings Solids, Gs		2.65
Void Ratio of Tailings after Completion of Initial Settlement, e_0		1.12
Pan Evaporation, PE	mm/day	7
Pan factor, f_{PAN}		0.9
Initial Flooded Area, A_1	Hectares	30
Final Flooded Area, A_{30}	Hectares	90
Average effective depth of rewetting, D_{rw}	Meters	0.5
Final void ratio of Tailings (after consolidation), e_f		0.79
Average saturation of inactive tailings beach prior to rewetting, S_{dry}	Percent	80%
Surface Area of Recycle Pond, A_{POND}	Hectares	30
Vertical Permeability of Slimes underlying pond, K_{POND}	Meters/second	2.0E-09

Overall, the site described in Scenario 1 withdraws 37,935 m^3/d of water from the local river, using 0.76 m^3/t of ore processed, below the Chilean average 0.79 m^3/t of ore calculated by Tejos and Proust (2008). Note, this figure is the total water withdrawal from the mine, not the water withdrawal attributed to a specific commodity, as detailed in Chapter 4.

6.3 Scenario 2 - Base Case with Water Conservation

In Scenario 2, the mine has the same processes and equipment as the base case, but has eliminated as many sources of water loss as possible, with a focus on evaporation. Table 6.6 details the water savings achieved in Scenario 2.

Table 6.6 Base Case with Conservation - Scenario 2 Savings

	Scenario 2	**Scenario 1**	**Savings**
Name	**m^3/day**	**m^3/day**	**m^3/day**
Road - Dust Suppression	757	3,520	2,763
Human Consumption	3.3	58	55
Raw Water Tank – Evap.	0.0	2.2	2.2
Process Water Tank – Evap.	0.0	3.4	3.4
Primary Crusher - Dust Supp.	50	360	310
Stockpile - Dust Supp.	0.0	120	120
Flotation Cell – Evap.	0.0	6.7	6.7
Conc. Thickener – Evap.	0.0	1.2	1.2
Final Concentrate	60	89	29
Tailings Retained, L_{ENT}	20,792	20,792	0
Beach Evaporation, L_{EVAP}	9,312	9,312	0
Pond Evaporation, L_{POND}	47	1,890	1,843
Beach Rewetting, L_{REW}	2,800	2,800	0
Total	**33,822**	**38,955**	**5,133**

To reduce water losses for dust suppression on the mine roads, the mine has applied an organic binder to the site roads. The binder is assumed to reduce the water requirements by 78.5%, the median of the water savings reported by GE (2006; 2010). This effort reduces road dust suppression water requirements to 757 m^3/d, a saving of 2,763 m^3/d.

To reduce water lost due to human consumption, the mine has redirected all available waste water, or grey water, after treatment, directly to the process water tank. Gleick (1996) recommends a basic water requirement of 50 L/p/d, based on 5 L/p/d for water consumption, 20 L/p/d for sanitation, 15 L/p/d for bathing and 10 L/p/d for food preparation. Of this water, a relatively small portion is lost through perspiration and breathing. For example, in a hot environment, a 70 kg person will sweat 4-6 L/d (Gleick, 1996). The water required for showers and toilets can also be reduced by using off-the-shelf low water use toilets and shower heads. This scenario assumes that 10 L/p/d is irretrievably lost to the mine site due to perspiration, breathing, and evaporation, for a total of 3.3 m^3/d, a savings of 55 m^3/d.

To reduce water lost to evaporation due to dust suppression at the primary crusher dump pocket, the mine has installed a fog dust suppression system. Fog dust suppression systems produce fine water particles in the less than 30 micron size range, are effective at capturing dust, and consume little water. Fog dust suppression systems have proven effective in dump pocket installations, provided that the pocket can be protected from the wind (TRC Group, 2010). Water addition rates are in the range of 1 L/t of material (VSR, 2010). Thus the water requirement is reduced to 50 m^3/d, a savings of 310 m^3/d.

It is difficult to contain the dust from the discharge conveyor to the coarse ore stockpile, limiting the usefulness of a fog dust suppression system. However, the entire coarse ore stockpile can be covered to eliminate dust emissions, as has been done at Teck's Highland Valley Copper Mine (Teck, 2008). A covered coarse ore stockpile would eliminate the need for a separate dust control system, providing a water savings of 120 m^3/d. If the copper in the dust can be recovered, reduced dust losses may also increase the mine's copper production.

Evaporation from the concentrate thickener, the water tanks, the flotation cells and the TSF pond can also be eliminated by covering the open areas. The covers can either be created by placing a cover or roof over the open area or in the case of thickeners or ponds by placing a floating cover on the surface. Floating covers can take the form of balls or tiles and can provide coverage of up to 95% of the surface area (AWTT, 2010). Mines also can manage the size of the TSF pond by careful management of the tailings deposition plan (Wels & Robertson, 2004), allowing reduced evaporation and reduced coverage requirements. This scenario assumes that the flotation cells and the concentrate thickener are completely covered, that the TSF pond is covered with tiles and that the pond size has been reduced to 15 ha, providing additional savings of 6.7 m^3/d, 1.2 m^3/d and 1,843 m^3/d respectively. The remaining assumptions regarding the TSF are unchanged from Scenario 1.

The concentrate filter performance can be improved to reduce the water content of the final concentrate. This could be achieved by upgrading filtration equipment, increasing filtration time or improving filter operation. Candelaria mine in Chile, among others, has reduced water consumption by improving filter performance (Soto, 2008). An additional benefit to improved filter performance is that lower water content reduces the concentrate weight and thus the concentrate shipping costs. This scenario assumes that the concentrate filter performance was improved, increasing the solids content from 90% solids to 93% solids, providing additional water savings of 29 m^3/d.

The combined water-saving efforts reduce the water loss from 38,955 m^3/d to 33,822 m^3/d - a savings of 5,133 m^3/d. This allows the raw water requirements to be reduced from 37,935 m^3/d to 32,801 m^3/d, a savings of 13.5%. 54% of the savings are due to reducing evaporation from the mine roads and 36% of the savings are due to covering and reducing the size of the TSF pond.

6.4 Scenario 3 - Paste Tailings Case

Thickening the tailings prior to deposition can reduce the evaporation and rewetting losses in the TSF (L_{EVAP}, L_{POND} and L_{REW}) by wetting a smaller area. The solids content of the tailings can be increased to the point of forming paste, thus eliminating any TSF pond. A thickener producing lower solids content non-paste slurry would still reduce water requirements in comparison to the base case. However, paste disposal represents the most extreme reduction likely through using thickeners.

In Scenario 3, the mine has installed a 75 m diameter tailings thickener to produce paste, but left everything else as per Scenario 1. All of the flotation tailings flow to the new thickener – the overflow water is pumped to the process water tank and the paste tailings, at 65% solids by mass, is pumped to the TSF.

Installing the paste thickener reduces the water discharged with the slurry to the TSF from 114,792 m^3/d to 26,490 m^3/d. However, thickening does not change the properties of the tailings material, and so does not change the amount of water retained in the tailings. This scenario assumes that no reclaim water is available to return to the plant, as the paste tailings is already close to its final moisture content of 70% solids. Any excess water would evaporate from the paste beach. No pond is formed in the TSF. The paste thickener creates a new open area of 4,418 m^2, leading to an evaporation of 31 m^3/d.

Table 6.7 Paste Tailings - Scenario 3 Savings

Name	Scenario 3 m^3/day	Scenario 1 m^3/day	Savings m^3/day
Road - Dust Suppression	3,520	3,520	0
Human Consumption	58	58.0	0
Raw Water Tank – Evap.	2.2	2.2	0.0
Process Water Tank – Evap.	3.4	3.4	0.0
Primary Crusher - Dust Supp.	360	360	0
Stockpile - Dust Supp.	120	120.0	0.0
Flotation Cell – Evap.	6.7	6.7	0.0
Conc. Thickener – Evap.	1.2	1.2	0.0
Final Concentrate	89	89	0
Tailings Thickener – Evap.	30.9	0	-31
Tailings Retained, L_{ENT}	20,792	20,792	0
Beach Evaporation, L_{EVAP}	5,698	9,312	3,614
Pond Evaporation, L_{POND}	0	1,890	1,890
Beach Rewetting, L_{REW}	0	2,800	2,800
Total	**30,682**	**38,955**	**8,273**

As shown in Table 6.7, installing the tailings thickener thus reduces the water lost from 38,955 m^3/d to 30,682 m^3/d, a savings of 8,273 m^3/d. This allows the raw water requirements to be reduced from 37,935 m^3/d to 29,662 m^3/d, a savings of 21.8%, due to the loss of the pond in the TSF and reduced rewetting losses and beach evaporation.

6.5 Scenario 4 - Filtered Tailings Case

In Scenario 4, the mine has installed a 75 m diameter tailings thickener to feed a bank of tailings filters, but with no further change to Scenario 1. All of the flotation tailings flow to the new thickener with the overflow water pumped to the process water tank. The thickened tailings are then filtered to 80% by mass, and deposited by a stacker conveyor in the TSF. The filtrate can be reused or recycled.

Installing the tailings filter reduces the water placed in the TSF from 114,792 m^3/d to 12,299 m^3/d. As the tailings are below their saturation limit, no reclaim water is available to return to the plant. No pond is formed in the TSF. The thickener creates a new open area of 4,418 m^2, leading to an evaporation of 31 m^3/d, as in Scenario 3.

Table 6.8 Filtered Tailings Scenario - Scenario 4 Savings

	Scenario 4	**Scenario 1**	**Savings**
Name	**m^3/day**	**m^3/day**	**m^3/day**
Road - Dust Suppression	3,520	3,520	0
Human Consumption	58	58.0	0
Raw Water Tank – Evap.	2.2	2.2	0.0
Process Water Tank – Evap.	3.4	3.4	0.0
Primary Crusher - Dust Supp.	360	360	0
Stockpile - Dust Supp.	120	120.0	0.0
Flotation Cell – Evap.	6.7	6.7	0.0
Conc. Thickener – Evap.	1.2	1.2	0.0
Final Concentrate	89	89	0
Tailings Thickener – Evap.	30.9	0	-31
Tailings Retained, L_{ENT}	12,299	20,792	8,493
Beach Evaporation, L_{EVAP}	0	9,312	9,312
Pond Evaporation, L_{POND}	0	1,890	1,890
Beach Rewetting, L_{REW}	0	2,800	2,800
Total	**16,491**	**38,955**	**22,464**

As shown in Table 6.8, installing the filtered tailings system thus reduces the water lost from 38,955 m^3/d to 16,491 m^3/d, a savings of 22,464 m^3/d. This allows the raw water requirements to be reduced from 37,935 m^3/d to 15,471 m^3/d, a savings of 59.2%, due to filtering the tailings to below their saturation limit and the elimination any water losses in the TSF.

6.6 Scenario 5 - Ore Pre-sorting Case

In Scenario 5, the mine has installed an ore pre-sorting system after the primary crusher with the purpose of rejecting any ore below a certain grade. This model conservatively assumes that 20%

of the ore can be rejected while retaining 98% of the copper. While the mine could increase production to keep the mill running at 50,000 tpd, for the purposes of comparison, the mine production in the model will remain unchanged, dropping the post-sorter mill feed rate to 40,000 tpd. This would allow much of the mill equipment to be reduced in size. However, for simplicity, the surface area of the flotation cells and the fresh and process water tanks will be unchanged. Pre-sorting may also require an additional crushing step not addressed here.

As a result of the reduced mill feed, the tailings production drops from 49,196 tpd to 39,213 tpd and reduces the water discharged with the slurry to the TSF from 114,792 m^3/d to 106,293 m^3/d. This model does assume that the reduced volume of slurry being deposed in the TSF reduces the size of the flooded deposition areas by a proportional amount. Thus the flooded deposition area after 1 day of discharge is lowered from 30 ha to 23.9 ha and the flooded deposition area after 30 days of discharge is lowered from 90 ha to 71.7 ha (variables A_1 and A_{30} respectively). The smaller wetter area reduces the beach evaporation to 7,422 m^3/d and the beach rewetting to 2,232 m^3/d. The TSF pond area is also assumed to be reduced from 30 ha to 23.9 ha, lowering the pond evaporation to 1,506 m^3/d. Some moisture is also contained in the rejected ore, leading to an additional loss of 204 m^3/d.

As shown in Table 6.9, installing the pre-sorting system thus reduces the water lost from 38,955 m^3/d to 32,096 m^3/d, a savings of 6,859 m^3/d. This allows the raw water requirements to be reduced from 37,935 m^3/d to 31,076 m^3/d, a savings of 18.1%, due to the reduced tailings production.

Table 6.9 Ore Pre-Sorting Scenario - Scenario 5 Savings

Name	**Scenario 5 m^3/day**	**Scenario 1 m^3/day**	**Savings m^3/day**
Road - Dust Suppression	3,520	3,520	0
Human Consumption	58	58.0	0
Raw Water Tank – Evap.	2.2	2.2	0.0
Process Water Tank – Evap.	3.4	3.4	0.0
Primary Crusher - Dust Supp.	360	360	0
Stockpile - Dust Supp.	120	120.0	0.0
Flotation Cell – Evap.	6.7	6.7	0.0
Conc. Thickener – Evap.	1.2	1.2	0.0
Final Concentrate	88	89	2
Tailings Thickener – Evap.	0.0	0	0
Tailings Retained, L_{ENT}	16,573	20,792	4,219
Beach Evaporation, L_{EVAP}	7,422	9,312	1,890
Pond Evaporation, L_{POND}	1,506	1,890	384
Beach Rewetting, L_{REW}	2,232	2,800	568
Pre-Sorting Rejects	204	0	-204
Total	**32,096**	**38,955**	**6,859**

6.7 Scenario 6 - Combined Water Reduction Case

In this final scenario, the water savings options which most reduced water withdrawals in the previous scenarios are combined. The scenario includes the water conservation methods included in scenario 2, the filtered tailings system described in scenario 4, and the ore pre-sorting system described in scenario 5. To summarize the process, the ore is pre-sorted, rejecting 20% of the ore while retaining 98% of the copper, and the flotation tailings are filtered to a solids content of 80% by mass. In addition, an organic binder is applied to the site roads, all site grey water is directed to the process water tank, a fog dust suppression system is installed on the primary crusher dump pocket and the coarse ore stockpile is covered. The concentrate thickener, the water tanks and the flotation cells are covered. Finally, tiles are placed on the tailings thickener to reduce evaporation by 95% and the final concentrate is filtered to 93% solids by mass.

Table 6.10 details the savings achieved through combining the different water reduction options. Installing the combined system thus reduces the water lost from 38,955 m^3/d to 10,878 m^3/d, a savings of 28,077 m^3/d. This allows the raw water requirements to be reduced from 37,935 m^3/d to 9,858 m^3/d, a savings of 74.0%. Figure 6.4 shows the savings achieved in the different water sinks at the mine. The combination of pre-sorting and filtered tailings led to 88.4% of the water savings. Reducing water lost on road dust control contributed 9.8% of the savings and the other efforts reduced water losses by 1.9%.

Figure 6.4 Scenario 6 Water Savings

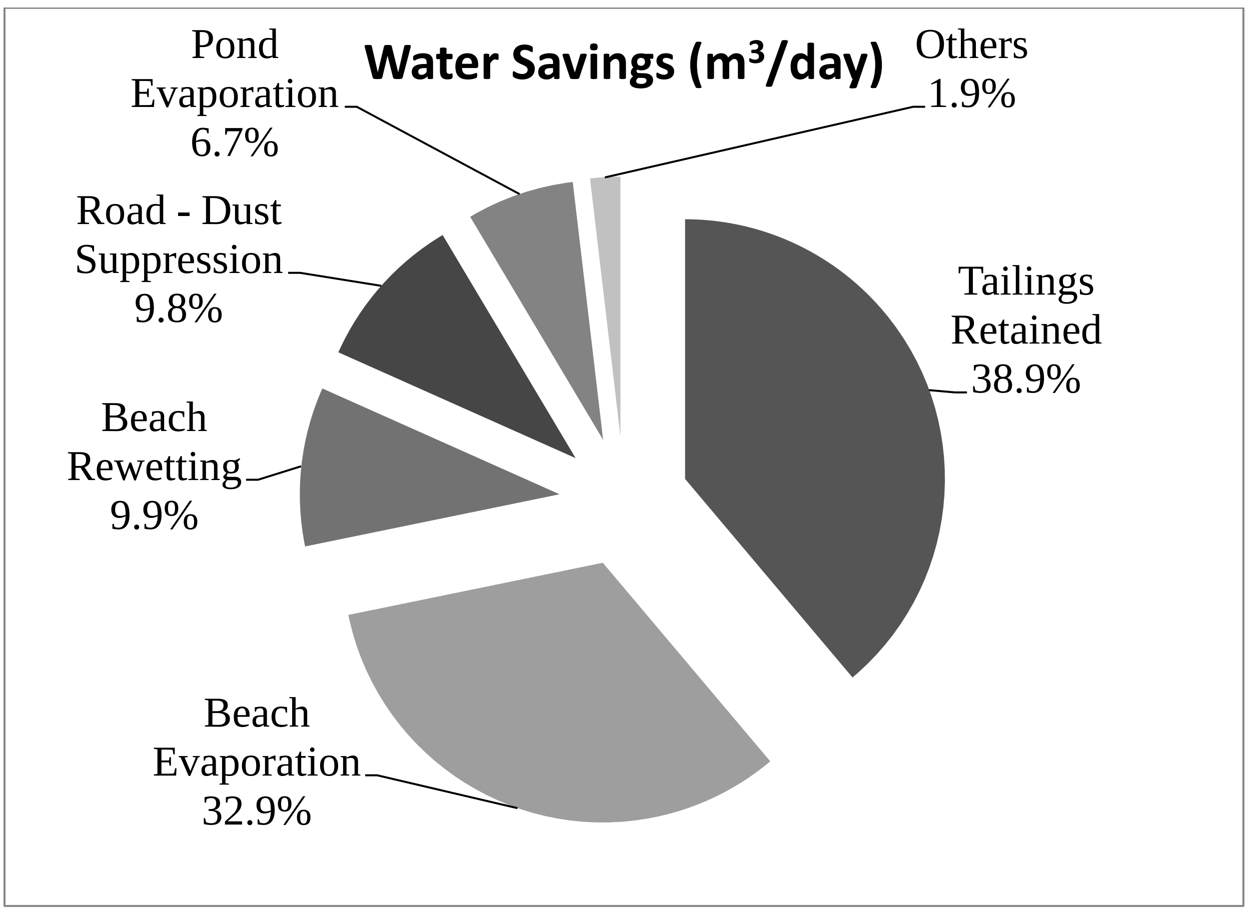


Table 6.10 Combined Water Reduction - Scenario 6 Savings

Name	Scenario 6 m^3/day	Scenario 1 m^3/day	Savings m^3/day
Road - Dust Suppression	757	3,520	2,763
Human Consumption	3	58	55
Raw Water Tank – Evap.	0.0	2.2	2.2
Process Water Tank – Evap.	0.0	3.4	3.4
Primary Crusher - Dust Supp.	50	360	310
Stockpile - Dust Supp.	0	120	120
Flotation Cell – Evap.	0.0	6.7	6.7
Conc. Thickener – Evap.	0.0	1.2	1.2
Final Concentrate	59	89	30
Tailings Thickener – Evap.	1.5	0	-1.5
Tailings Retained, L_{ENT}	9,803	20,792	10,989
Beach Evaporation, L_{EVAP}	0	9,312	9,312
Pond Evaporation, L_{POND}	0	1,890	1,890
Beach Rewetting, L_{REW}	0	2,800	2,800
Pre-Sorting Rejects	204	0	-204
Total	**10,878**	**38,955**	**28,077**

Furthermore, if the percentage of ore rejected could rise above 20% or if the filtered tailings solids content could reach above 80% solids, the water withdrawals could be further reduced significantly. Table 6.11 shows how the water consumption, in m^3/t of ore, changes when varying the percentage of ore rejected and the filtered tailings solids content, with all other variables unchanged from Scenario 6. Improving the filtered tailings solids density or the percent of ore rejected by pre-sorting alone reduces water consumption linearly, but combining the two leads to dramatic reductions in water requirements. If the mine could achieve a filtered tailings solids density of 90% solids by mass while rejecting 50% of the ore through pre-sorting, water consumption could reach as low as 0.06 m^3/t of ore mined, allowing the 50,000 tpd mine to withdrawal only 3,051 m^3/day of raw water.

Table 6.11 Impact of Combining Sorting with Filtered Tailings on Water Withdrawals (m^3/t ore)

Ore Pre-Sorting (% Rejected)	**Filtered Tailings Solids Density (%)**			
	75%	**80%**	**85%**	**90%**
10%	0.29	0.22	0.16	0.10
20%	0.26	0.20	0.14	0.09
30%	0.23	0.17	0.12	0.08
40%	0.20	0.15	0.11	0.07
50%	0.17	0.13	0.09	0.06

6.8 Limitations

The model and scenarios presented contain a number of limitations, including:

- Dry stack tailing disposal methods have high capital and operating costs and have not yet been applied to operations processing more than 20,000 tpd (Engels, 2012);
- Ore presorting is not suitable for all ores and can also have significant costs. The technology has not yet been applied to a full scale copper porphyry operation. In addition, equipment manufacturers do not currently produce equipment with the same tonnage capacity as other equipment typically used in large porphyry operations, which could lead to a large number of lower capacity units;
- Reduced water losses would lead to an increase in water reuse and recycling, which could have negative process impacts, as described in Section 2.5.3; and
- All of the evaporation saving techniques described in Section 6.3 have costs and risks associated with them and would need to be implemented with caution.

It is important to note that all the scenarios are using simplified water balance models. A comprehensive site water balance requires hydro-geological techniques to model water flow in the mine, tailings impoundments, waste rock facilities and other major site features, in addition to climatic data. The water balance may also have significant seasonal variations.

6.9 Conclusion

As stated in Section 1.2, a key question this research aimed to answer was: how can mine water use be reduced? This question was addressed by pursuing the following objectives:

- Identifying options for how mines can better reduce, reuse and recycle water; and
- Modeling the potential impact of different options to reduce, reuse and recycle water on mine site water use.

In the literature review, Section 2.1 defined key terms and concepts for mine water balances. Section 2.4 reported on mine water use and included a subsection on TSF water losses. Section 2.5 discussed strategies of how mining operations have worked to better reduce, reuse and recycle water. The literature review found there had been no systematic effort or models available to evaluate the combined potential impact of these strategies.

Section 3.3 described a novel methodology to model and test different options to reduce mine water consumption and Chapter 6 modeled six scenarios to evaluate the potential water savings that can be gained through the identified water reduction strategies.

The results from all six scenarios are summarized in Table 6.12. The base case scenario, representing common practice at open pit copper mines, achieved a water withdrawal of 0.76 m^3/t of ore processed. Scenario 2 implemented a water saving strategy of reducing evaporation throughout the mine site, lowering the water withdrawal to 0.66 m^3/t. Scenario 3 implemented tailings paste thickening, approximating the maximum water savings achievable through implementing tailings thickening technologies, and reduced water withdrawals to 0.59 m^3/t. Scenario 4 introduced filtered tailings disposal and achieved a water withdrawal of 0.31 m^3/t by filtering the tailings to 80% solids by mass. Scenario 5 introduced ore sorting to reject 20% of the ore, reducing the water withdrawals to 0.62 m^3/t. Scenario 6 combined the efforts of scenarios 2, 4, and 5 to achieve a water withdrawal of 0.20 m^3/t of ore. Further increasing the portion of ore rejected in ore sorting and/or the final solids density achieved by tailings filtration could bring water withdrawals significantly below any existing conventional major copper mine and into the range of the water consumption of the best performing SX/EW operations, as shown in Chapter 4.

Table 6.12 Scenario Summary

	Scenarios					
Water	**1**	**2**	**3**	**4**	**5**	**6**
Withdrawal	**Base Case**	**Conservation**	**Paste**	**Filtered**	**Sorting**	**Combined**
m^3/day	37,935	32,801	29,662	15,471	31,076	9,858
m^3/t ore	0.76	0.66	0.59	0.31	0.62	0.20
m^3/t Cu	157.4	136.1	123.0	64.2	140.9	44.7
% reduction	0	13.5	21.8	59.2	18.2	74.0

The results of the different scenarios highlight the obvious importance of tailings disposal in mine water management – most other options have comparatively minor impacts on overall water consumption. However, the key finding of the study is the massive potential water savings from combining ore pre-concentration and filtered tailings disposal. It is still comparatively rare to test ore for amenability to pre-concentration, but in areas with high water costs, evaluating pre-concentration as an option may be reasonable. Filtered and paste tailings disposal options are relatively expensive and difficult technically. They would be significantly more attractive if a large portion of the ore could be rejected prior to comminution. There may also be considerable plant capital and operating cost reductions if pre-concentration is viable.

However, as discussed in Section 2.3, reducing water and energy consumption is only one facet of sustainable development. Filtered and paste tailings disposal can have additional benefits in the social, environmental and governance spheres of consideration including improved geotechnical stability, improved reclamation, and better meeting regulatory requirements (Davies, 2004).

The low levels of water consumption discussed in this chapter may seem unrealistic to many in the mining industry today. However, these scenarios were developed by combining currently available off-the-shelf mining technology. While not all of these options may be suitable for many mine sites, these scenarios outline how the mining industry can work toward dramatically reducing its water requirements.

7. MINE WATER NETWORK DESIGN

The objectives of this chapter are to:

- Identify a methodology of how to better design mine and mineral processing plant water systems;
- Describe a software tool to enable implementation of a methodology of how to better design mine and plant water systems;
- Model the potential impact of improved mine water system designs; and
- What sustainability goals should mining companies pursue with respect to water use.

This chapter describes and demonstrates the MWND method of reducing energy requirements in a mine water system by matching water sources and consumers in low energy combinations. The method estimates the energy required to supply water from available sources to specified consumers and then identifies potential combinations to lower overall energy requirements. Section 3.4 introduced the methodology for the approach. Section 7.1 provides a detailed description of the MWND method. Section 7.2 presents an example of how to implement the MWND method, using the model presented in Chapter 6 as a base. The plant water withdrawals are in line with typical copper concentrators, as recorded in Chapter 4. The site water sources and consumers are comparable to those described in the case study, as recorded in Chapter 5. Section 7.3 calculates the water and energy reductions from each of 6 scenarios described in Chapter 6. Section 7.4 outlines the limitations of the MWND method and Section 7.5 concludes the chapter.

7.1 Model Description

The MWND method consists of five steps:

- Description of site water balance;
- Description of all potential water sources;
- Description of all major water consumers;
- Construction of energy requirement matrices; and
- Use of linear programming to minimize energy requirements.

7.1.1 Site Water Balance

In order to set the overall problem constraints, it is necessary to describe the process mass flow and site water balance. At a typical site this includes determining the ROM tonnage and ore water content, the percent solids at which mineral separation takes place, the final concentrate mass and water content, and the final tailings mass and water content. Based on this information and some further site details, it is possible to calculate a simplified water balance for a mine, as described in Section 2.1.

7.1.2 Major Water Sources

Once an overall water balance is created, all the major site water sources must be identified, which includes a description of the quantity, quality, pressure and temperature available for each major water source. There are a number of external and internal water sources available to the mine and plant including the raw water, the recycle water, the reclaim water from the tailings and the discharge water of heat exchangers.

The mass and water balances provide information about the quantity of water available from the different sources. Where possible water quality should be determined by water testing, preferably from frequent sampling campaigns taken over a long period of time to allow for changes resulting from seasonal variations. If detailed data is not available, data from other similar sites can be used as estimates of water quality.

Discharged spent cooling water is treated as a water source: it can be recycled to the heat exchanger system in a closed-loop system or reused by other consumers in an open system. The water discharged from heat exchangers is assumed to have the same flow and water quality as the heat exchanger supply water. The maximum allowable heat transfer is assumed and defines the maximum water temperature. The pressure is reduced by the estimated pressure drop across the heat exchanger system.

7.1.3 Major Water Consumers

All of the major water consumers at a site must be identified, which includes describing the required water quantity, quality, pressure and temperature. A water consumer is a unit that requires water – the water may or may not be available for reuse. Mine water consumers can include cooling systems, process makeup water, pump GSW, reagent mixing and dilution water, spray and wash water and dust suppression water. While process makeup water and wash water can be of relatively poor quality, the other consumers listed typically require a higher water quality. Water quantity, quality, pressure and temperature requirements are generally specified by equipment vendors. However, vendors can modify these requirements significantly by changing piping materials and nozzle designs or altering the heat exchanger design. Water specifications can also be calculated from engineering handbooks, such as Perry (1997).

7.1.4 Energy Requirement Matrices

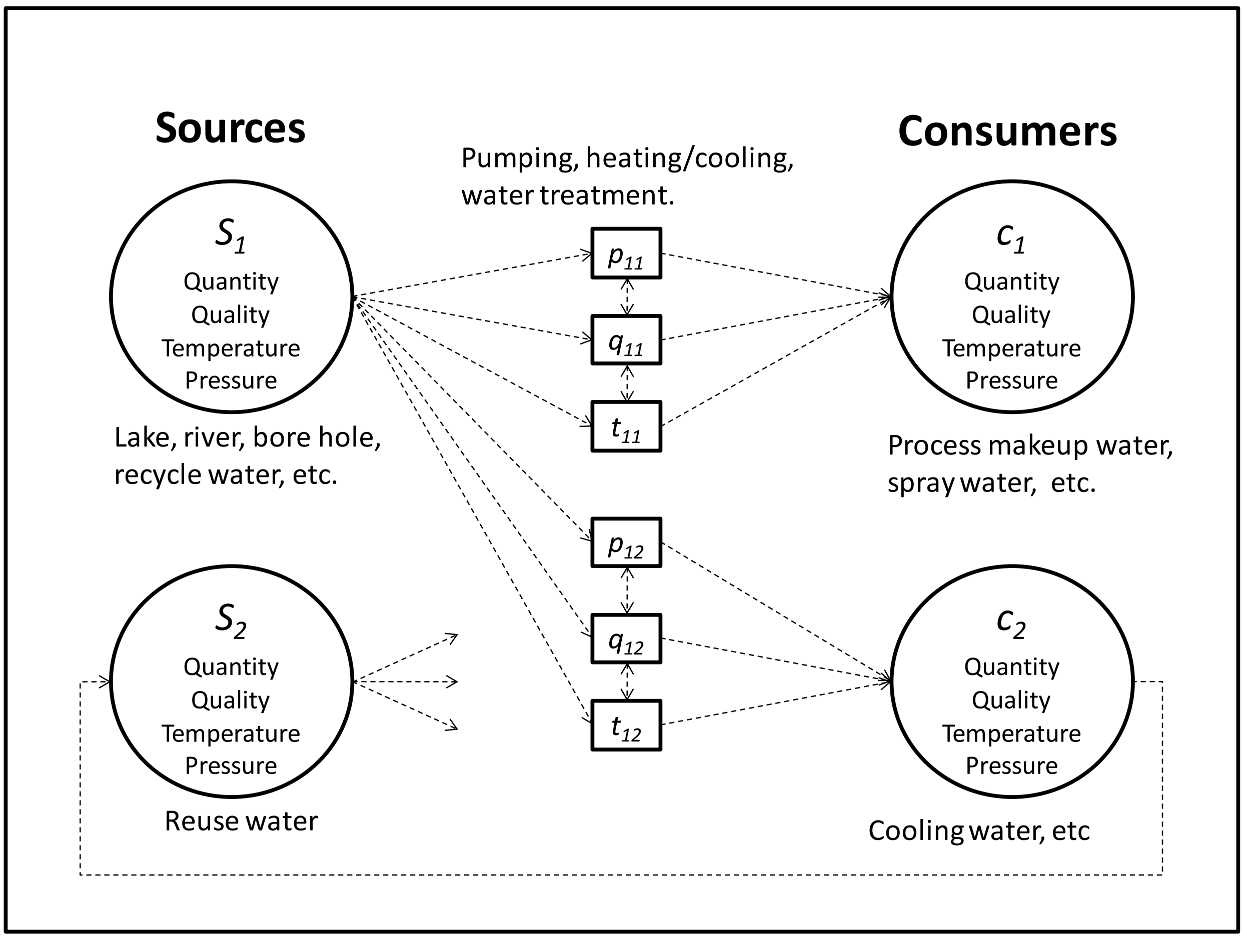
This section explains how to construct matrices describing the amount of energy required for each water source to supply each consumer. If there are m water sources and n water consumers, then there are m x n possible connections in a water network. Each source i, where $1 \leq \mathrm{i} \leq m$, can supply a quantity of water s_i and each consumer j, where $1 \leq j \leq n$, can consume a quantity c_j.

Pumping, cooling, and treatment each have an associated energy requirement. The amount of energy required to allow 1 m^3 of water from i to meet the requirements of consumer j can be described as e_{ij}, and can be calculated as follows:

$$e_{ij} = p_{ij} + q_{ij} + t_{ij} \tag{7.1}$$

where e_{ij} = total energy required, [kWh/m^3]; p_{ij} = pumping energy required, [kWh/m^3]; q_{ij} = cooling energy required, [kWh/m^3]; and t_{ij} = treatment energy required, [kWh/m^3]. Figure 7.1 shows this concept graphically.

Figure 7.1 Construction of Energy Requirement Matrices



The pumping energy p_{ij} can be estimated as follows. Pump power output can be estimated by:

$$\text{Pump Power} = \text{Head} \cdot \text{Q} \cdot \rho / 3.670 \times 10^5 \qquad [7.2]$$

where Pump Power is in [kW]; Head = total dynamic head, [N·m/kg] (column of liquid); Q = capacity, [m^3/h]; and ρ = liquid density, [kg/m^3] (Perry & Green, 1997).

The total dynamic head is the difference between the pressure required by c_j (H_j) and the pressure available at s_i (H_i). Water consumer pressures are given in kPa, thus the total dynamic head can be determined as shown by Equation 7.3.

$$\text{Head} = \frac{H_j - H_i}{9.83} \qquad [7.3]$$

where Head = total dynamic head, [N·m/kg]; H_j = pressure required by c_j, [kPa]; H_i = pressure available at s_i, [kPa]; and 9.83 kPa equals 1 N·m/kg.

Pump efficiency and motor efficiency can be calculated as shown [16]:

$$\text{Pump efficiency } (e_{pump}) = (\text{power output})/(\text{power input}) \qquad [7.4]$$

$$\text{Motor efficiency } (e_{motor}) = (\text{power output})/(\text{power input}) \qquad [7.5]$$

By combining Equations 7.2, 7.3, 7.4 and 7.5, p_{ij}, or the energy required for pumping 1 m^3 of water from i to j, can be determined from Equation 7.6, assuming a water density of 1000 kg/m^3 and ignoring pipe friction loss:

$$p_{ij} = \left(\frac{H_j - H_i}{9.83} \cdot 1000/3.670 \times 10^5\right) \cdot e_{pump} \cdot e_{motor} \qquad [7.6]$$

where p_{ij} = pumping energy required, [kWh/m^3]; H_j = pressure required by c_j, [kPa]; H_i = pressure available at s_i, [kPa]; e_{pump} = pump efficiency; and e_{motor} = motor efficiency.

Pipe friction loss is a function of a number of variables including the type of fluid, the fluid velocity, the pipe diameter, length and roughness, and the number of bends in the pipe. As the MWND method evaluates different flow combinations, including pipe friction would make the optimization calculation non-linear. However, generally pipe friction losses are a small component of overall system head requirements within a plant. For example, using the Darcy-Weisbach equation and a friction coefficient of 0.015, the head loss for water flowing at 2 m/s in a 1 meter long section of 8" (0.2 m) diameter pipe new carbon steel pipe, would be approximately 0.015 m. The Darcy-Weisbach equation is one of the main accepted methods for calculating piping losses (G. Brown, 2002).

The energy required for cooling, q_{ij}, depends on the cooling method used. The two options described here are air-cooled chiller systems and mechanical draft cooling towers. There are many other options available, most of which are either less energy intensive than air-cooled chillers or more energy intensive than cooling towers, as discussed in Section 5.6.1. Thus these two options define the range of energy requirements for cooling applications.

Chillers work through evaporative heat transfer, using a refrigerant in a closed loop as the cooling medium. When the refrigerant evaporates, it removes heat from the heated medium. After evaporation, the gas must be compressed and condensed, which requires additional air or water cooling. Air-cooled chillers have a small footprint and enable the use of a cooling system that is independent from the plant water system and largely independent of ambient temperature. The main disadvantages include high energy consumption, significant capital and operating costs, the need for specialized maintenance, and the use of refrigerants that are often toxic and can have negative environmental impacts. Generally chillers cool the medium to below the recommended vendor specifications for mill equipment; therefore the medium needs to be blended or passed through a separate heat exchanger to achieve the required temperature.

Mechanical draft cooling towers remove heat through evaporative heat transfer. Warm water is mixed with cooler air in the tower's fill area, which contains material with a high surface area. Open cooling towers use the evaporative water as the cooling medium, whereas closed systems separate the evaporative water from the cooling medium. Evaporated water must be replaced regularly. The key advantage of wet cooling is significantly lower operating and capital costs compared to chillers. In addition, the method requires relatively small amounts of makeup water, and thus is somewhat independent of the water system. The main disadvantages include the need for water treatment, the need to add bactericides, and an overall increase in plant water consumption due to evaporation. Open cooling towers are inexpensive, but the re-circulating water may be fouled by dust or contamination build-up. Closed cooling towers reduce the likelihood of fouling, but have somewhat higher capital and operating costs.

The cooling energy q_{ij} can be estimated for a chiller system as follows. Modern air-cooled chillers require at least 0.3 kWh for every kWh of heat removed (Alliant Energy, 2009; Carrier, 2009). This factor is referred to as the chiller efficiency, or $e_{chiller}$. As the specific heat of water is 1.163 kWh, meaning 1.163 kWh can cool or heat 1 m^3 by 1°C in an hour, q_{ij} for an air-cooled chiller can be estimated as follows:

$$q_{ij} = (T_i - T_j) \cdot c_p \cdot \mathrm{e}_{\mathrm{chiller}} \qquad [7.7]$$

where q_{ij} = cooling energy required, [kWh/m^3]; T_i = temperature of the water available at source i, [°C]; T_j = temperature of the water required at consumer j, [°C]; c_p = the specific heat of water, [kWh/°C/t]; and $\mathrm{e}_{\mathrm{chiller}}$ = the chiller efficiency. In addition, chiller systems require some small pumps to re-circulate the fluid in the closed loop.

The cooling energy q_{ij} for an open cooling tower system depends on a number of factors. Cooling tower selection and performance are dependent on the required cooling range, the approach to the wet-bulb temperature, the quantity of water to be cooled, the wet-bulb temperature, the air velocity through the tower and the tower height (Perry & Green, 1997). The wet-bulb temperature and the required approach, or how close the temperature of the cooled water needs to be to the wet-bulb temperature, are site specific, whereas the other variables depend on the system design. If the wet-bulb temperature is assumed to be low enough to allow for a reasonable approach (≥4°C), standard vendor specifications may allow for an estimate of energy requirements.

Cooling towers require energy for 3 primary functions: running fans to force air into or out of the cooling towers (q_f), pumping the re-circulating cooling water (q_p), and replacing the amount of water evaporated (q_e). Thus, the energy requirement for q_{ij} is shown by Equation 7.8:

$$q_{ij} = q_f + q_p + q_e \qquad [7.8]$$

where q_{ij} = cooling energy required, [kWh/m^3]; q_f = cooling tower fan energy required, [kWh/m^3]; q_p = cooling tower re-circulating pump energy required, [kWh/m^3]; and q_p = energy required to replace the water evaporating in the cooling tower, [kWh/m^3].

Cooling towers also have additional costs associated with operation, such as water treatment reagents, that need to be accounted for in a study based on cost rather than on energy requirements.

Assuming a T_i of 35°C, a T_j of 29.4°C, and a wet-bulb temperature of 25.6°C, a typical cooling tower requires 0.01 kWh of fan energy per kWh of cooling energy required (Baltimore Aircoil, 2009). This efficiency factor, or e_{tower}, can be used to calculate q_f in the same manner as air-cooled chillers.

$$q_f = (T_i - T_j) \cdot c_p \cdot \mathrm{e}_{\mathrm{tower}} \qquad [7.9]$$

where q_f = cooling tower fan energy required, [kWh/m^3]; T_i = temperature of the water available at source i, [°C]; T_j = temperature of the water required at consumer j, [°C]; c_p = the specific heat of water, [kWh/°C/t]; and e_{tower} = the cooling tower efficiency.

Note, q_f represents the maximum energy consumption during periods of high wet-bulb temperatures. Cooling towers with two-speed or variable speed fans can significantly reduce fan power consumption during low temperature periods.

The amount of pumping energy required, q_p, is primarily dependent on the height of the tower, in addition to any nozzle pressure requirement. The pumping energy can be determined in a manner similar to Equation 7.6 for p_{ij}.

$$q_p = (\text{tower height} \cdot 1 \cdot 1000/3.670 \times 10^5) \cdot \mathrm{e_{pump}} \cdot \mathrm{e_{motor}} \qquad [7.10]$$

where q_p = cooling tower re-circulating pump energy required, [kWh/m^3]; tower height = the height of the cooling tower, [m]; e_{pump} = pump efficiency; and e_{motor} = motor efficiency.

For a cooling tower with a wide temperature approach, or the difference between the cooled water temperature and the wet bulb temperature, a tower height of 4.6 to 6.1 m is generally adequate (Perry & Green, 1997).

The amount of energy required to replace the evaporated water, q_e, can be determined as follows. Makeup water requirements of a cooling tower include evaporative loss, drift loss, and blowdown. Evaporative loss is the water directly evaporated in the process of removing heat from the cooling water. Drift loss is water entrained in the tower discharge vapors, and blowdown is the portion of the re-circulated water that is discharged to lower the system solids concentration (Perry & Green, 1997). Blowdown water is assumed to report to the process water, reducing makeup water requirements elsewhere, and is thus ignored in this model. Drift loss is a function of the drift-eliminator design. In the past, drift losses varied between 0.1 and 0.2 percent of the water supplied to the tower (Perry & Green, 1997). However, new drift eliminators reduce drift loss to negligible amounts and thus it is also ignored in this paper.

The latent heat of evaporation is 2260 kJ/kg of water, or 628 kWh/m^3 of water (Perry & Green, 1997). Therefore, to remove 1 kWh of heat, 1.59 kg of water needs to be evaporated. Evaporation loss can thus be estimated by:

$$W_e = 0.00159 \cdot c_j \cdot (T_i - T_j) \qquad [7.11]$$

where W_e = Cooling Tower Water Evaporation, [m^3/h]; c_j = the amount of water required by a consumer j, [m^3/h]; T_i = temperature of the water available at source i, [°C]; and T_j = temperature of the water required by consumer j, [°C]. Typically when estimating evaporation, c_j would refer to the total water entering the cooling tower, including evaporation, but not the water required by the consumer (Perry & Green, 1997). However, as can be seen from Equation 7.11, the difference in water volume for small temperature changes is small, and can be ignored here.

Any evaporated water increases the mine's overall water consumption, requiring an equal increase in makeup water consumption. Thus the total energy required to provide makeup water to the cooling tower, $e_{makeup,plant}$ must be calculated. The pressure required depends on the cooling tower location, which is likely at the same general elevation as the plant. The makeup water does not need to be cooled, although it may need to be filtered and treated. Thus q_e can be estimated by Equation 7.12:

$$q_e = W_e \cdot \mathrm{e}_{\mathrm{makeup,plant}} \qquad [7.12]$$

The treatment energy t_{ij} depends on the water quality available at the source, the water quality required by the consumer, the water flow, and the treatment selected. Many treatment processes are available to remove or render harmless the impurities in water, ranging from simple strainers to evaporation followed by condensation. Often two or more treatment methods are used to achieve the desired water quality. Common treatment methods include:

- Filtration or clarification to remove suspended solids;
- Addition of antiscalants or corrosion inhibitors to reduce scale formation or corrosion in pipelines and/or equipment;
- Cold-lime clarification, hot-lime softening, ion exchange and zeolite treaters to reduce dissolved solids;
- R/O and electrodialysis for partial removal of both dissolved and suspended solids, including desalination; and
- Evaporation or ion exchange to meet high water purity requirements.

In general, the dirtier the source water and the purer the water required, the higher the capital and operating costs of the treatment system. The capital cost for treatment varies widely and the most appropriate option can depend on the quantity of water to be treated. Operating costs can be broken down into energy requirements, reagent requirements, and labor requirements. For several treatment options, such as R/O and filtering, energy requirements represent the most significant cost. For others options, such as the addition of antiscalants or corrosion inhibitors and chemical clarification methods, reagent costs can be more significant. Overall, the least sophisticated treatment option that can meet the quality requirements is often also the lowest capital and operating cost option.

As a water source may require several stages of treatment, t_{ij} is the sum of the energy requirements for each stage:

$$t_{ij} = \sum_{k} t_{ij}(k) \qquad [7.13]$$

where t_{ij} = treatment energy required, [kWh/m^3]; k = the number of possible stages of treatment, and $t_{ij}(1)$ and $t_{ij}(2)$ = specific energy required for the first and second stages of treatment, [kWh/m^3].

For the purposes of this work, water must be treated by a filter when the maximum particle size or the total suspended solids (TSS) in a water source exceeds the requirements of the water consumer. In addition, a clarifier treatment stage is required when the maximum total dissolved solids (TDS) or the pH range of the water source does not meet the requirements of the water consumer. The specific energy requirements for filtration and clarification are referred to as $t_{ij}(1)$ and $t_{ij}(2)$ respectively.

A filter typically has a filter medium, such as silica sand, through which water is forced, whereas a strainer consists of a metal or plastic screen across the flow of water. However, this work uses the term filter for both filters and strainers. The amount of treatment energy required is primarily a function of the pressure drop across the filter, although there can also be a temporary energy requirement associated with cleaning the filter. Selecting a filter requires balancing the cost and size of the installed screen with the pressure drop across the screen. An infinitely large filter would have a pressure drop approaching zero, whereas an undersized filter requires a large pressure drop. In addition, an undersized filter clogs more quickly and needs more frequent screen cleaning. An engineer must choose an appropriate, design-allowable pressure drop and select and size a filter to match the design drop. For a given design pressure drop, the filter energy requirement can determined in a similar manner to p_{ij}, as shown in Equation 7.14.

$$t_{ij}(1) = \left(\frac{\Delta H_f}{9.83} \cdot 1000/3.670 \times 10^5\right) \cdot \mathrm{e_{pump}} \cdot \mathrm{e_{motor}} \qquad [7.14]$$

Where $t_{ij}(1)$ = filter energy required, [kWh/m^3]; ΔH_f = design allowable filter pressure drop, [kPa] ; e_{pump} = pump efficiency; and e_{motor} = motor efficiency.

Clarification systems require energy for reagent addition and preparation equipment, agitators, reactor turbines, rakes and sludge pumps, however, the unit energy requirements of this equipment (kWh/m^3) are practically negligible. The production of a reagent, such as the crushing and grinding required for preparing limestone, can also require energy. However, often the only significant direct energy requirement for a clarifier system is the pumping requirement. The amount of pressure needed, assuming only pumping and clarification, is:

$$\Delta Hs_{ij} = H_j + \Delta H_c - H_i \qquad [7.15]$$

where ΔHs_{ij} = the change in pressure in a system with a clarifier between C_j and S_i, [kPa]; H_j = pressure required by c_j, [kPa]; H_i = pressure available at s_i, [kPa]; and ΔH_c = the pressure loss across the clarifier, [kPa].

Clarification takes place at atmospheric pressure, usually in open tanks. Water must enter the clarifier and then the clarified water must be pumped back to the required pressure for the consumer. If the source water pressure, P_{si}, is greater than the pressure loss across the clarifier system, any excess pressure is lost. Equation 7.6, solving for p_{ij}, determines how much pumping energy is required to pump from s_i to c_j, but if clarification is necessary, the pumping requirement may increase due to the loss of pressure across a clarifier. If the pressure available at s_i is less than the pressure required for the clarifier, then the system must have additional pumping capacity to meet the pressure loss across the clarifier. However, if the pressure at s_i is greater than the clarifier pressure drop, the additional pressure, accounting for the pumping requirement already included in p_{ij}, will equal the source pressure. This relationship is shown as:

$$\text{if } H_i - \Delta H_c \leq 0 \text{ then } \Delta Hcs_{ij} = \Delta H_c \text{ else } \Delta Hcs_{ij} = H_i \qquad [7.16]$$

where H_i = pressure available at s_i, [kPa]; ΔH_c = the pressure loss across the clarifier, [kPa]; and ΔHs_{ij} = the change in pressure in a system with a clarifier between C_j and S_i, [kPa].

Once the pressure loss across the clarifier is determined, the energy requirement can be determined:

$$t_{ij}(2) = \left(\frac{\Delta Hs_{ij}}{9.83} \cdot 1000/3.670 \times 10^5\right) \cdot \mathrm{e_{pump}} \cdot \mathrm{e_{motor}} \qquad [7.17]$$

where $t_{ij}(2)$ = clarifier energy required, [kWh/m^3]; ΔHs_{ij} = the change in pressure in a system with a clarifier between C_j and S_i, [kPa]; $\mathrm{e_{pump}}$ = pump efficiency; and $\mathrm{e_{motor}}$ = motor efficiency.

As p_{ij}, q_{ij}, and t_{ij} are calculated for each possible combination, matrices P_{mxn}, Q_{mxn}, and T_{mxn} are thus built. As per Equation 7.6, these matrices are then added to determine E_{mxn}, a matrix describing e_{ij}.

7.1.5 Linear Programming to Minimize Energy

Once matrix E_{mxn} has been created, detailing the unit energy requirements for e_{ij}, the total amount of power required can be determined by:

$$Pow_{ij} = e_{ij} \cdot x_{ij} \qquad [7.18]$$

where Pow_{ij} = power required for source *i* to provide water to consumer *j*, [kW]; e_{ij} = total energy required, [kWh/m^3]; and x_{ij} = the quantity of water delivered from source *i* to consumer *j*, [m^3/h].

The minimum amount of power required can be calculated by means of the following linear program:

$$\text{Minimize Pow} = \sum_{i=1}^{m}\sum_{j=1}^{n} e_{ij}x_{ij}$$

where: $$\sum_{j=1}^{n} x_{ij} \le s_i \qquad [7.19]$$

and $$\sum_{i=1}^{m} x_{ij} \le c_j$$

This type of problem is referred to as an assignment problem (Carpaneto et al., 1988). A linear programming algorithm, such as the MS Excel solver, can then be used to solve Equation 7.19. Once a solution is found, it needs to be reviewed to ensure practicality, constructability and safety, and to identify opportunities for simplification.

7.2 Case Study

The following simplified example is used to demonstrate the MWND method. The goal is to minimize the water system energy requirements for a 50,000 tpd low grade copper deposit in an arid region. The mine uses dry stacked tailings deposition to minimize water consumption and reduce tailings stability concerns.

7.2.1 Site Water Balance

The plant is a conventional froth flotation separation process, similar to the example given in Figure 6.1. As shown in Table 7.1, the plant processes 50,000 tpd of copper sulfide ore, grading 0.5% Cu, with a moisture content of 2%. The final concentrate grades 28% copper, with a 90% copper recovery. The ROM ore is crushed and ground with a gyratory crusher and a SAG Mill/Ball Mill circuit. The copper sulfide is then separated from the gangue in a flotation circuit at 30% solids and the copper concentrate is dewatered to 90% solids. The tailings are filtered and stored in a tailings impoundment, with a final moisture content of 80% solids.

Table 7.1 Major Process Flows

Process Slurry Flows	**Solids Content (tpd)**	**Solids Content (%)**	**Water Content (m^3/day)**
ROM → Comminution	50,000	98	1,020
Comminution → Flotation	50,000	30	116,667
Flotation Concentrate → Concentrate Dewatering	804	30	1,875
Concentrate Dewatering → Final Concentrate	804	90	89
Flotation Tailings → Tailings Dewatering	49,196	30	114,792
Tailings Dewatering → Final Retained Tailings	49,196	80	12,299

Table 7.2 shows the simplified water balance. The average evaporation rate, including precipitation, is assumed to be 931 m^3/d, and no mine water is assumed to be available. Thus, the minimum amount of makeup water required is shown to be 12 300 m^3/d.

Table 7.2 Mine Water Balance

Sources					**Sinks**				
Name	**m^3/h**	**m^3/day**	**Solids t/d**	**% Solids**	**Name**	**m^3/h**	**m^3/day**	**Solids t/d**	**% Solids**
ROM	43	1020	50,000	98	Retained Tailings	512	12,300	49 196	80
Makeup	512	12,300			Concentrate	4	89	804	90
Precipitation		0			Evaporation	39	931		
Mine Water		0			Discharge Water		0		
Total	**555**	**13,320**	**50,000**		**Total**	**555**	**13,320**	**50 000**	

7.2.2 Potential Water Sources

Descriptions of the water sources are shown in Table 7.3. The raw water quantity is determined from the overall water balance. The cooling water sources, or the water discharged from heat exchangers, have the same flow and water quality as the cooling water quality requirements. The maximum allowable heat transfer is assumed and defines the maximum water temperature. The pressure is reduced by the estimated pressure drop across the heat exchanger system.

Table 7.3 Major Water Sources

Water Sources / **Name**	**S_i** **m³/h**	**T_i** **(°C)**	**Quality[a]** **(µm)**	**Quality** **(TDS)**	**Hs_i** **(kPa)**	**pH** **Low**	**pH** **High**
Raw Water	512	25	2000	1000	-980	6.5	7.5
SAG Mill Cooling Water Discharge	171	40	50	750	200	6.8	8.0
Ball Mill Cooling Water Discharge	171	40	50	750	400	6.8	8.0
Compressor Cooling Water Discharge	171	50	25	750	200	6.8	8.0
Recycle Water	74	40	100	2000	0	9.0	11.0
Reclaim Water	4232	25	125	1500	-490	8.0	10.0

[a]Maximum particle size in source water.

The recycle water quantity is the difference between the water content of the feed to the concentrate dewatering stage and the water content of the final concentrate. The water temperature is elevated, since much grinding energy is converted to heat, increasing the slurry temperature. The recycle water is assumed to have fine particles, retained during dewatering, and high dissolved solids and pH, primarily due to the lime added to the process. The dewatering stage is assumed to be at the same elevation as the main processing plant, thus the initial supply pressure is zero.

The reclaim water quantity is the difference between the water content of the flotation tailings, the amount of evaporation from the tailings area, and the amount of water retained in the tailings. The reclaim water temperature is assumed to be at ambient temperature (25 °C). Some tailings particles may remain in the reclaim water. The water is assumed to have high dissolved solids and pH, primarily due to lime from flotation. The TSF is assumed to be 100 m below the main site, thus the reclaim water must be pumped back to the plant.

7.2.3 Major Water Consumers

Major water consumers and the water quantity and quality required are specified in Table 7.4. The water quality requirements are taken from vendor specifications shown in Section 5.3.1.

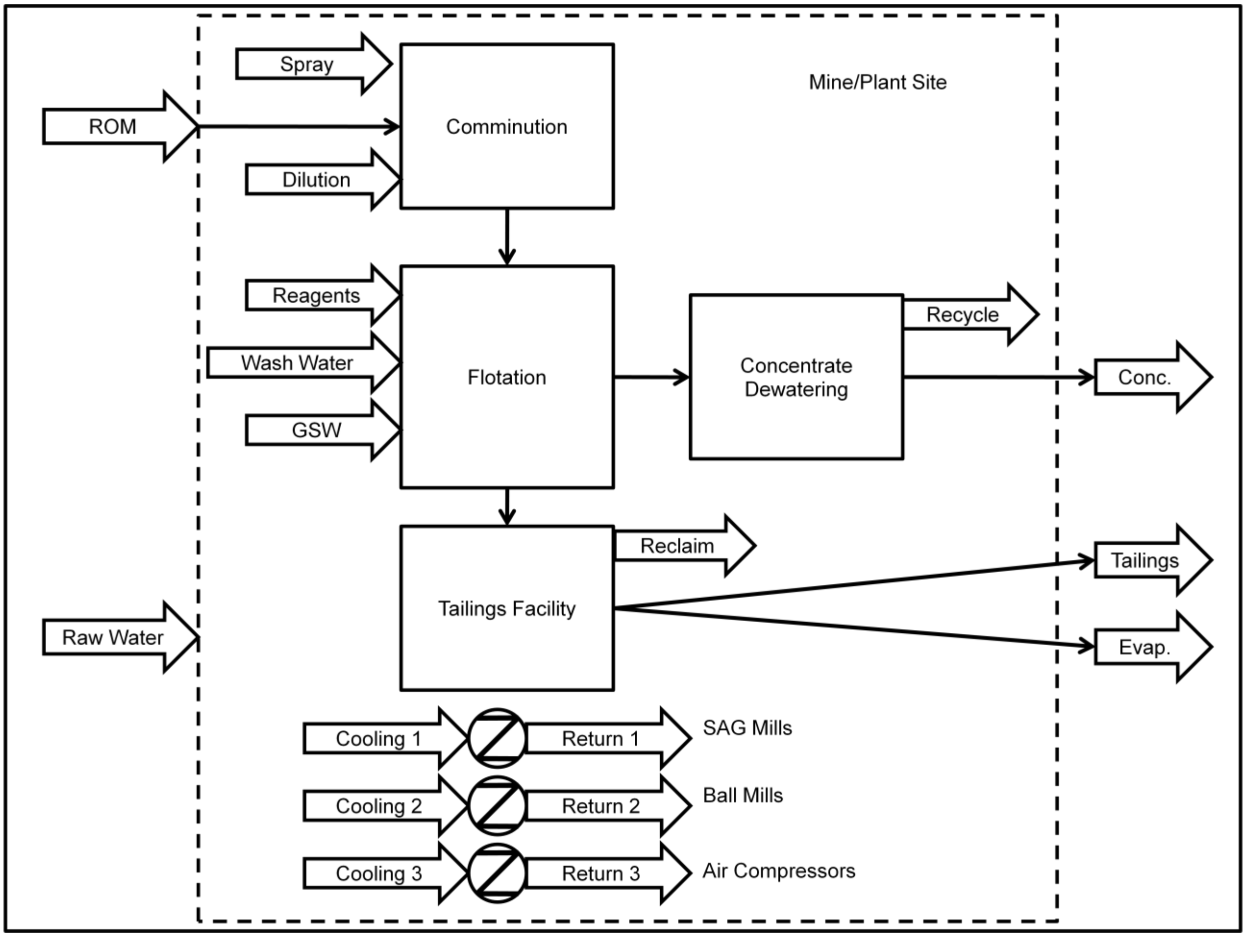
Table 7.4 Major Water Consumers

Water Consumers Name	**S_i m^3/h**	**T_i (°C)**	**Quality[a] (µm)**	**Quality (TDS)**	**Hc_j (kPa)**	**Pres. Loss Max (kPa)**	**pH Low**	**pH High**
SAG Mill Cooling Water	171	30	50	750	300	100	6.8	8.0
Ball Mill Cooling Water	171	30	50	750	500	100	6.8	8.0
Compressor Cooling Water	171	40	25	750	300	100	6.8	8.0
Flotation Froth Wash Water	120	60	1000	10000	300	0	6.5	11.0
Pump GSW	60	60	50	1000	800	0	6.8	8.0
Reagent Dilution Water	30	60	50	750	300	0	6.5	8.0
Dust Suppression Spray Water	10	60	50	750	400	0	6.8	8.0
Comminution Dilution Water	4599	60	1000	10000	300	0	6.5	11.0

[a]Maximum allowable particle size.

The comminution dilution water is determined by taking the amount of water required for flotation less the ROM moisture, the froth wash water, the pump GSW and the dust suppression water. The quality of this water does not need to be high as it is added directly to the ore on screens and in pump boxes and launders. Figure 7.2 shows a flow sheet illustrating the flows between water sources and consumers.

Figure 7.2 Flow Sheet Showing Water Sources and Consumers



7.2.4 Energy Requirement Matrices

As shown in Table 7.3 and Table 7.4, there are six major water sources (m = 6) and eight major water consumers (n = 8). Given adequate pumping, cooling, and treatment any sources can be used to supply the water consumers. Multiple sources can be used to provide water to any consumer, up to the maximum amount of water available at the source and the maximum required by the consumer. Each linkage requires varying amounts of pumping (p_{ij}), cooling (q_{ij}), or treatment (t_{ij}), allowing the development of matrices P_{mxn}, Q_{mxn}, and T_{mxn}.

Table 7.5 shows the matrix P_{mxn}. The first entry, $p_{Raw\ Water\ x\ SAG\ Cooling}$, can be determined using Equation 7.6 as follows. As shown in Table 4, $Pc_{SAG\ Cooling}$, or the SAG cooling circuit minimum water pressure, is 300 kPa. The raw water source is at an elevation 100 m below the plant site, thus $Ps_{Raw\ Water}$ is 980 kPa (1 m water head = 9.8 kPa). Assuming e_{pump} = 80% (pump efficiency) and e_{motor} = 90% (pump motor efficiency), then $p_{Raw\ Water\ x\ SAG\ Cooling}$ = 0.4928 kWh/m^3. If there is a negative pressure (more pressure available from the sources than required by the consumer), for final consumers, such as the Pump GSW, this excess pressure is lost (0 kW). However, for water that can be directly reused, without additional pressure loss, such as the water discharged from the heat exchangers, this energy can be credited.

Table 7.5 P_{mxn} – Pumping Energy Requirements

Energy (kWh/m^3)	Consumers							
Sources	SAG Cooling C	BM Cooling C	Comp Cooling C	Froth WW	Pump GSW	Reagent DW	DS Spray	Com. Dilution
Raw Water	0.4928	0.5698	0.4928	0.6853	0.6853	0.4928	0.5313	0.4928
SAG Cooling S	0.0385	0.1155	0.0385	0.2310	0.2310	0.0385	0.0770	0.0385
BM Cooling S	-0.0385	0.0385	-0.0385	0.1540	0.1540	0.0000	0.0000	0.0000
Comp Cooling S	0.0385	0.1155	0.0385	0.2310	0.2310	0.0385	0.0770	0.0385
Recycle Water	0.1155	0.1925	0.1155	0.3080	0.3080	0.1155	0.1540	0.1155
Reclaim Water	0.3042	0.3812	0.3042	0.4967	0.4967	0.3042	0.3427	0.3042

Q_{mxn}, and T_{mxn} are then created for cooling and treatment energy requirements, using Equations 6.7 and 6.13. The following assumptions are made: $e_{chiller}$ = 0.3; ΔP_d = 100 kPa; and ΔP_c = 100 kPa. E_{mxn} is calculated by adding P_{mxn}, Q_{mxn}, and T_{mxn}, shown in Table 7.6. Table 7.6 assumes water is cooled by a chiller system when necessary, but can also be calculated for cooling towers with the following assumptions: e_{tower} = 0.01; tower height = 5 m; and $e_{makeup,plant}$ = 0.4543 kWh/m^3.

Table 7.6 E_{mxn} - Total Energy Requirements

Energy (kWh/m^3)	Consumers							
Sources	SAG Cooling C	BM Cooling C	Comp Cooling C	Froth WW	Pump GSW	Reagent DW	DS Spray	Com. Dilution
Raw Water	0.5698	0.6468	0.5698	0.7238	0.7623	0.5698	0.6083	0.5313
SAG Cooling S	3.5275	3.6045	0.0770	0.2310	0.2310	0.0385	0.0770	0.0385
BM Cooling S	3.4505	3.5275	0.0000	0.1540	0.1540	0.0000	0.0000	0.0000
Comp Cooling S	7.0165	7.0935	3.5275	0.2310	0.2310	0.0385	0.0770	0.0385
Recycle Water	3.6815	3.7585	0.1925	0.3080	0.3850	0.1925	0.2310	0.1155
Reclaim Water	0.3812	0.4582	0.3812	0.4967	0.5737	0.3812	0.4197	0.3042

7.2.5 Linear Programming to Minimize Energy

A linear programming algorithm is then used to minimize the total power requirements, as per Equation 7.19. Constraints s_i and c_j are taken from the flow figures in Table 7.3 and Table 7.4, to ensure each source supplies no more than is available and each consumer receives the minimum amount of water required.

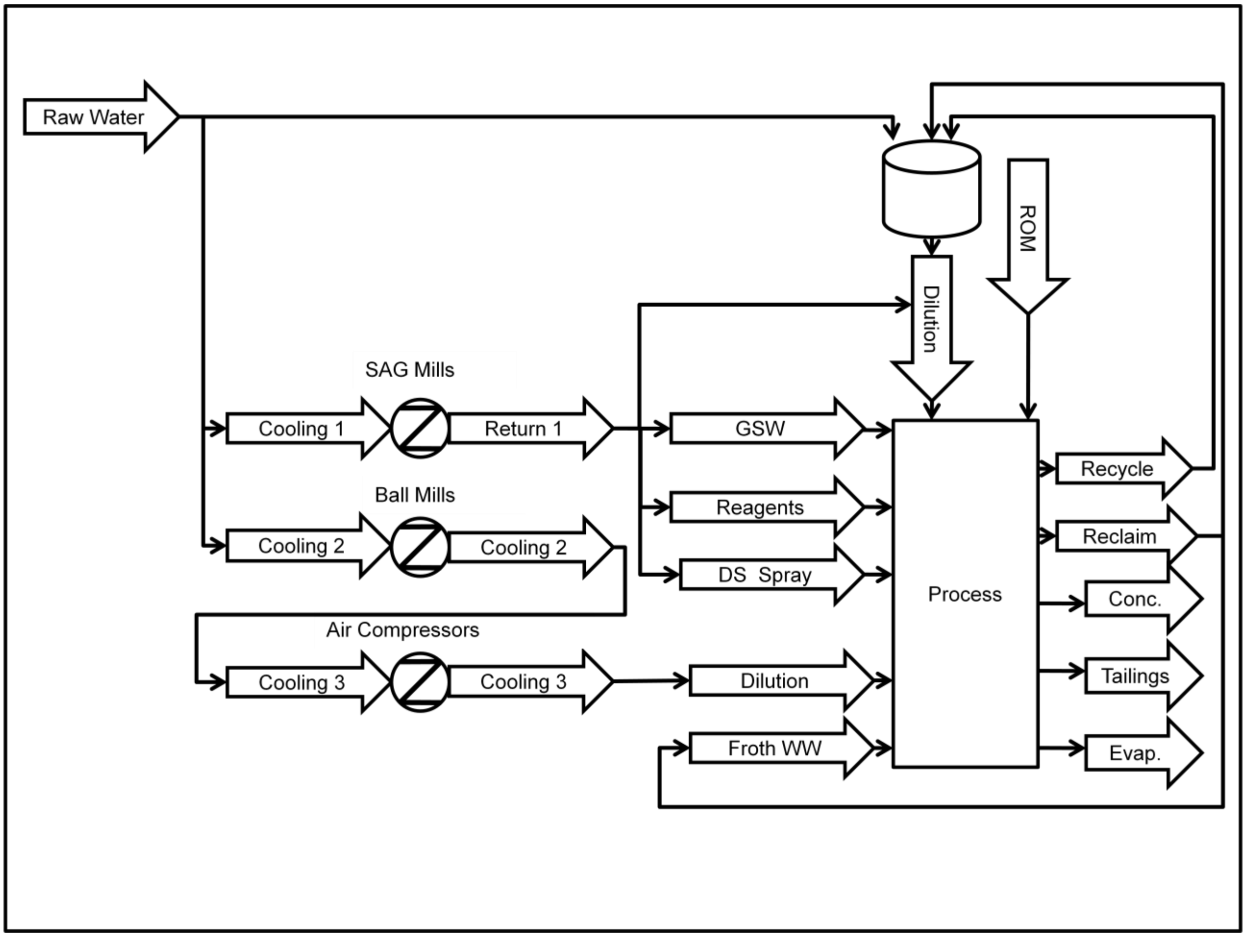
Solving for the example data gives a system power consumption of 1643 kW. A suggested water distribution network is shown in Table 7.7, with zero showing where no connection is required. For this example there are multiple solutions.

Table 7.7 Network Solution Water Distribution

(m^3/h)	Consumers							
Sources	SAG Cooling C	BM Cooling C	Comp Cooling C	Froth WW	Pump GSW	Reagent DW	DS Spray	Com. Dilution
Raw Water	171	171	0	0	0	0	0	171
SAG Cooling S	0	0	0	0	60	30	10	71
BM Cooling S	0	0	171	0	0	0	0	0
Comp Cooling S	0	0	0	0	0	0	0	171
Recycle Water	0	0	0	0	0	0	0	74
Reclaim Water	0	0	0	120	0	0	0	4112

Figure 7.3 shows the solution graphically. Required pumping, cooling, filtering and treatment are not shown. As can be seen, the low power solution involves the staged use of water. For example, the ball mill cooling water is used to cool the air compressors and the SAG mill cooling water is used for the pump GSW, reducing water pumping, cooling, filtering, and treatment requirements. A third of the raw water supply goes directly to the process water for dilution, giving some additional operational flexibility. If the raw water supply is disrupted the plant can likely run temporarily on two-thirds the requirement, using extra reclaim water or by operating flotation at a higher solids content.

Figure 7.3 Low Energy Solution



Any solution must be reviewed to ensure the system would be operable, and that the piping arrangements are reasonable. Actual plant design is an iterative exercise and as design progresses, the model can be developed further. Connections between sources and consumers which are impractical can be accommodated by adding constraints to the model.

The model can also be used to identify high power requirement connections in order to examine options for further optimization. Table 7.8 details the amount of power required for the network model solution, which is determined by multiplying e_{ij}, shown in Table 7.6, with the network solution flow rates x_{ij} shown in Table 7.7. Table 7.8 indicates that in the solved problem most of the power is consumed through returning reclaim water from the tailings storage facility to the plant. The high power consumption indicates an area to investigate design alternatives. One option would be to further reduce pumping costs by installing tailings thickeners at the site, thus sending less water to the tailings storage facility. These thickeners would likely be installed to feed the tailings filters, as this would also likely improve filter operation, lead to some reagent savings and may reduce evaporation losses.

Table 7.8 Minimized Power Requirements

Power (kW)	Consumers							
Sources	SAG Cooling C	BM Cooling C	Comp Cooling C	Froth WW	Pump GSW	Reagent DW	DS Spray	Com. Dilution
Raw Water	97	110	0	0	0	0	0	91
SAG Cooling S	0	0	0	0	14	1	1	3
BM Cooling S	0	0	0	0	0	0	0	0
Comp Cooling S	0	0	0	0	0	0	0	7
Recycle Water	0	0	0	0	0	0	0	9
Reclaim Water	0	0	0	60	0	0	0	1,251

Solving Equation 7.19 for the maximum power, instead of the minimum, shows the saving that can be achieved using the MWND method. In this case the maximum achievable result is 3,421 kW, or over twice the minimum requirement. While this maximum would involve some unlikely arrangements, such as using recycle water from concentrate dewatering for cooling water, the total power requirement is similar to the estimate for a closed cooling system (~3,389 kW). Using cooling towers instead of chillers would significantly reduce the potential savings, while marginally increasing the overall plant water consumption due to increased evaporation.

7.3 Water and Energy Reduction Estimate

The example described in Section 7.2 can be integrated with the example described in Chapter 5 to demonstrate the significant energy savings possible when combining the MWND with water use reduction strategies. The energy matrix shown in Table 7.6 is used to calculate the energy savings possible from the six scenarios described in Chapter 5.

Table 7.9 shows the energy requirements to supply water energy differences between the different scenarios. The table was calculated by taking the energy requirements shown in Table 7.6 and multiplying the water requirements calculated for each consumer in each scenario. The energy requirements for supplying raw water to the road dust suppression, the primary crusher, the coarse ore stockpile, and the domestic water supply are assumed to be the same as supplying water to the SAG cooling circuit. The water network is per Figure 7.3, modified slightly for the different scenarios. The water system energy requirements are shown to reduce substantially as the water requirements decrease. Scenario 6 would require only 36% of the energy required in the base scenario. Note these calculations only take into consideration the water pumping requirements, not the energy requirements of running the tailings filters, thickeners or ore-sorting devices, or the large energy savings that would be achieved by reducing comminution requirements.

Table 7.9 - Energy Requirements

	Scenarios					
Major Water Users	**1**	**2**	**3**	**4**	**5**	**6**
(kWh/day)	**Base Case**	**Conservation**	**Paste**	**Filtered**	**Sorting**	**Combined**
Process Water – Raw	8,786	7,785	4,390	1,706	5,141	451
Process Water - Con Recycle	206	210	206	206	202	205
Process Water - Tail Recycle	-	-	10,195	11,834	-	9,436
Process Water – Reclaim	24,330	24,893	-	-	19,392	-
SAG Cooling Water	2,336	2,336	2,336	2,336	2,336	2,336
BM Cooling Water	2,652	2,652	2,652	2,652	2,652	2,652
Comp Cooling Water	2,336	2,336	2,336	-	2,336	-
Road - Dust Suppression	2,006	431	2,006	2,006	2,006	431
Froth Wash Water	2,085	2,085	2,085	887	2,085	887
Pump GSW	1,098	1,098	1,098	333	1,098	333
Reagent Dilution Water	410	410	410	28	410	28
Primary Crusher Dump Pocket - Dust Suppression	205	28	205	205	205	28
Coarse Ore Stockpile - Dust Suppression	68	-	68	68	68	-
Mine/Mill/Office Staff - Domestic Water	33	2	33	33	33	2
Water Energy (kWh/d)	**46,551**	**44,266**	**28,020**	**22,294**	**37,965**	**16,789**

COCHILCO calculated the 2006 total electrical energy consumption by open pit copper mines and concentrators in Chile to be 8039 MJ/t of copper produced (Pimentel, 2009). As the mine described in the model above produces 225 tpd of copper, the mine energy consumption could be estimated at 502,431 kWh/d and thus the water system in Scenario 1 would consist of 9.3%

of the total electrical energy consumption. An operation pumping desalinated sea water to a mine at a high elevation could save far more energy.

The water savings achieved were mirrored by water system power savings. Water system pumping requirements were estimated to account for 9.3% of the mine's total electrical demand in Scenario 1. The water system described in Scenario 6 required only 36% of energy required in Scenario 1, a saving of 30 MWh/day.

7.4 Discussion and Limitations

The MWND method applies the principles of WAP to mine and mill water network design. As discussed in Section 2.1, there are several other approaches available to model mine water systems. In particular, the ADRET method (2008) was identified as a best practice guide for defining how mine sites should report water usage. The MWND method provides an alternative, focused specifically on water system optimization for the purposes of design. The ADRET method was specifically built to provide systems level models to allow accurate longer-term water balances. It is able to take a complicated mine water network and simplify the components into key categories to allow comparison between different sites. It is also able to deal with a range of time-series data such as climatic input and changes in water stores. The MWND method, on the other hand, is focused specifically on the design and optimization of water systems at an engineering level. It focuses on defining and separating key water sources and consumers in order to identify opportunities to reduce the overall system water and energy requirements. It is more focused on individual sites rather than providing comparable outputs between different sites. As a design tool it is focused on instantaneous design cases rather than longer-term water balances. Thus, the MWND method can accommodate a considerable amount of detail on an engineering level, but is less suited to modeling changes over time, such as changing climatic data.

The MWND method presented above contains a number of limitations, including:

- The unit amount of energy to pump, cool or treat water in the model is assumed to increase linearly. In reality larger pumps, motors, cooling equipment and treatment systems are often more efficient than smaller ones. Thus the model may favor smaller, less practical connections.
- The distance between sources and consumers is not accounted for, nor are pipe friction losses.
- The choice to focus on energy, rather than overall operating, capital, or net present values biases the model, especially with respect to reagent costs. For an actual project, operating costs or net present value calculations (NPV$/$m^3$) are recommended. However, at a study level, energy requirements may be sufficient.
- The energy estimates are somewhat simplistic. Vendor numbers should be used where possible.
- There are dedicated methods for optimizing heat exchanger network and mass exchange network design, such as pinch theory or linear programming. If a plant requires an extensive heat exchange network, it could be more appropriate to first use a dedicated model.

These limitations can be overcome during the detailed design or optimization of a mine water system by undertaking the following efforts:

- Closely review connections recommended by the model to insure they are as simple and practical as possible and that the system provides some flexibility. Often the MWND output has multiple solutions, which means the designer or design team can review the output to ensure consumers use water from the closest sources with the simplest arrangements in order to reduce piping costs and pipe friction losses;

- Ensure that capital and operating costs are taken into consideration when reviewing the model outputs. This can be done when reviewing the outputs of the model or if sufficient information is available, the model can solve for minimum capital costs, operating costs or net present value. A simple step is to multiply the energy requirements (kWh/m^3) by the cost of power to calculate the energy cost per m^3; and
- Ensure certified vendor data are used wherever possible.

7.5 Conclusion

As the mining industry moves toward sustainable development, mine sites must generally become more energy and water efficient. As stated in Section 1.2, key questions this research aimed to answer included: how can mines better use the water they require to reduce energy requirements, how can mines take better advantage of opportunities to use poorer quality water resources, and what are suitable sustainability goals for mining companies to pursue with respect to water use? These questions were addressed by pursuing the following objectives:

- Identifying a methodology of how to better design mine and mineral processing plant water systems;
- Creating a software tool to enable implementation of a methodology of how to better design mine and plant water systems;
- Modeling the potential impact of improved mine water system designs; and
- Identifying what sustainability goals mining companies should pursue with respect to water use.

In the literature review, Section 2.1 defined key terms and concepts for mine water balances and Section 2.2 reported the importance of water in mining and corporate sustainability goals. Section 2.6 discussed WAP methodologies that have been developed for other industries. The literature review found there had been no detailed effort or models available to apply the principles of WAP to the mining industry.

Section 3.4 described the principles of the Mine Water Network Design method, a novel method of applying the principles of WAP to improve mine water system design by reducing energy needs, reducing water treatment requirements, and in some cases reducing overall water consumption.

This chapter described in detail the MWND approach, including the creation of a water balance model, the description of all site water sources and consumers, the calculation of energy requirements and the energy minimization methodology. An Excel-based software tool was developed and an example of a 50,000 tpd copper porphyry mine was used to show how water system power requirements could be reduced by more than 50%. While this chapter focused on reducing site power requirements, the MWND method can also be used to reduce operating costs, capital costs, or the net present value of mine water systems.

The MWND method highlights key potential goals for mining companies to target with respect to sustainability and mine water use. Reducing mine water withdrawals is obviously a major target. Theoretical and potential minimum water consumption levels can be determined through analysis of mines' water systems, as described in Chapter 6, to set practical targets. However, reducing water withdrawals does not necessarily mean reducing energy use. For example, eliminating evaporative cooling with the use of chillers would reduce water losses but also significantly increase energy requirements. Increasing water reuse or water recycling may appear to be a valuable goal. However, it is easy to recycle more water than is necessary, thus incurring greater treatment and energy costs. WAP principles and the MWND method can be used to determine potential minimum water system energy requirements. In addition, the MWND method can determine potential minimum high quality water requirements to allow a mine site to target reducing high quality water withdrawals where possible.

8. SUMMARY AND CONCLUSION

As Section 1.2 set out, this thesis undertook to explore the following questions:

- How can the amount of water withdrawn by the mining industry be better quantified?
- How can mine water use be reduced?
- How can mines better use the water they require to reduce mine energy requirements?
- How can mines take better advantage of opportunities to use wastewater or poorer quality water resources?
- What sustainability goals should mining companies pursue with respect to water use?

These questions were pursued in the following ways:

- Chapter 2 undertook a thorough literature review of how the mining industry uses water and designs water systems;
- Chapters 3 and 4 identified methods of quantifying global mine water withdrawals by commodity ;
- Chapter 4 estimated global mine water withdrawals by commodity;
- Chapter 5 described a practical case study of how the mining industry uses water and designs water systems;
- Chapter 5 identified and described the water system requirements of a typical mine;
- Chapters 2 and 6 identified options for how mines can better reduce, reuse and recycle water;
- Chapter 6 modeled the potential impact of different options to reduce, reuse and recycle water on mine site water use;
- Chapters 2, 3 and 7 identified a methodology of how to better design mine and mineral processing plant water systems;

- Chapter 7 described the creation of a software tool to enable implementation of a methodology of how to better design mine and plant water systems;
- Chapter 7 also modeled the potential impact of improved mine water system designs; and
- Chapter 7 identified suitable goals for mining companies to target with respect to sustainability and mine water use.

The literature review undertaken in Chapter 2 shows that mine water use has been a key concern for centuries, and its importance has increased as the mining industry strives toward becoming more sustainable. Past efforts to quantify mine water use have been hampered by a lack of consistency in reporting and have often relied on potentially non-representative samples. It was found that limited literature was available on the design requirements of a typical mine water system. While efforts to reduce mine water use have been both considerable and diverse there has been limited concerted effort to organize and evaluate these efforts. Finally, there is a significant opportunity to apply lessons from other process industries to mine plant design in order to systematically improve mine water use.

Chapter 3 described and explained methodologies developed over the course of this work which allowed the estimation of global mine water withdrawals by commodity, the examination and modeling of past efforts to reduce mine water consumption, and the minimization of mine water network energy requirements.

Chapter 4 detailed the results of an estimate of global mine water withdrawals by commodity. Water withdrawals were estimated to range from six to eight billion m^3 per annum over the period of 2006 to 2009. The study included production data from 1,076 mines and included 6,723 commodity production figures and 541 water withdrawal figures, accounting for a large portion of the global mining industry. The estimates were based on two methods, referred to as the Ore Production Method and the Concentrate Production Method. While water withdrawal estimates vary with the two methods, they are generally in the same order of magnitude. The accuracy and representativity of the variables used in the datasets were analyzed as well as the accuracy and representativity of the estimate. Sensitivity tests were undertaken to evaluate the robustness of the estimate and then each commodity was reviewed to better understand the results and the potential accuracy of the estimate. The estimates were found to likely be most accurate for copper, gold, precious metals and bauxite and least accurate for cobalt, phosphate and potash. No water data was found for manganese, tantalum, tin and titanium. Overall, the estimates are likely conservative, as mines which report their water usage are more likely to be owned by companies than have water management programs and actively work to reduce their water withdrawals.

Chapter 5 detailed a case study of the design, startup and modification of the Cerro Verde Primary Sulfide Project water system. This chapter undertook a practical case study of how the mining industry uses water and designs water systems, and described the water system requirements of a typical mine. Key lessons learned included the importance of ensuring that the design of the water supply and treatment system are addressed as early as possible in a project, and the importance of undertaking laboratory or pilot testing of any key technology in the water system. This chapter also underscored the importance of having a clear understanding of the quality of water available at a mine site. Advance knowledge of the actual required water quantities and qualities within the mine allow for better design of plant water systems.

Chapter 6 examined options for reducing mine water consumption and reusing or recycling water available on site. A mine water system model was run with six scenarios showing the potential water saving that could be achieved on a theoretical mine site. The key finding of the study was the massive potential water savings from combining ore pre-concentration and filtered tailings disposal. The application of the water reduction options to the model detailed how water withdrawals could be reduced from the base case of 0.76 m^3/tonne of ore processed to 0.20 m^3/tonne of ore processed, or even down to 0.06 m^3/tonne of ore processed depending on the amount of ore that could be rejected using ore sorting technology.

Chapter 7 described the MWND method for minimizing the amount of energy required to supply water from available sources to specified consumers in a mineral processing plant. A software tool was developed and applied to an example to demonstrate how an improved mine water network could significantly reduce energy requirements. Furthermore, the method was applied to the six water saving scenarios described in Chapter 6, in order to demonstrate the water and energy savings possible by combining the two approaches. The example showed how energy required for the water system could be reduced from 46,551 kWh/d down to 16,789 kWh/d or lower.

Some major mining companies set targets for reduction of energy consumption and water withdrawals. These targets can be arbitrary and are generally based on reductions from past consumption (i.e. a 10% reduction in water withdrawals from a set date). Theoretical and practical minimum water consumption levels, minimum water system energy requirements, and minimum high quality water requirements can be determined through analysis of mines' water systems, as described in Chapters 6 and 7. Mining companies can determine how much they can actually reduce their water withdrawals, energy requirements and high quality water requirements and set their targets accordingly.

In addition, the mining industry must help civil society to recognize that mine water use is relatively limited and can produce economic returns well in advance of most other industries. Furthermore, the mining industry should help communities interacting with mines to understand that while mines generally need water, there are real options to limit the amount of water they require and they often do not require clean water.

9. RECOMMENDATIONS

There are several opportunities for further work based on the outcomes of this thesis.

The effort to quantify global mine water withdrawals faced several limitations, including poor quality data, especially for commodities such as cobalt, manganese, phosphate, potash, tantalum, tin and titanium. The database developed should be expanded past the year 2009 as more data becomes available. Efforts to directly contact mining companies to fill in data gaps could make the database much more accurate and useful. In addition, the database should be split more finely, to better define the types of mining and processing used at mine sites; this in turn will enable better comparison between sites. Furthermore, there is a significant opportunity to expand the database to include other indicators, with energy being the obvious first choice.

The effort to determine the best options to reduce mine water withdrawals has significant opportunities for further development. This thesis highlighted the massive water and energy savings possible by combining ore pre-concentration and tailings filtration. The potential savings justify more work in the lab and on mine sites. This combination clearly offers an opportunity to make a technological step change in mine water and energy use.

The effort undertaken in this thesis to adopt water allocation planning (WAP) analysis to the mining industry is only a first step. Other processing industries have developed more sophisticated methodologies and algorithms to better identify optimum water networks.

Finally, over the past decade more and more mining companies have worked to develop water and energy strategies aimed at better implementing sustainability principles in the mining industry. It would be a worthwhile exercise to evaluate their decision-making motives and track the success of these strategies to highlight best practices that could be adopted across the industry.

10. CLAIMS OF ORIGINALITY

The research undertaken for this thesis was novel and significant in the following areas:

Global Water Estimate:

This work was the first to define a methodology of how to determine global water withdraws by commodity. It included a comprehensive estimate of global water withdrawals from 2006 to 2009 based on 6,723 data points for mine commodity production and 541 data points for mine water withdrawals. The total amount of water withdrawals estimated per year, for commodities with relevant data available, was between 6.2 and 7.8 billion m^3 per annum. This methodology provides a basis for improving the quality of efforts to track global mine water withdrawals and for improving the accuracy of mine water withdrawal indicators by commodity. The methodology and results can be useful for efforts to determine life cycle assessments of commodities and for mining companies, policy makers and communities wanting information on mine water withdrawals.

Case Study- Cerro Verde Concentrator Fresh Water System

This case study was the first published description of all significant water quantity and quality parameters of sources and consumers at a major copper mine. The case study provided a description of the practical considerations that must be measured during the design, startup, and optimization of a large mine water system. In addition, the case study described the implementation of a significant water network change that involved switching several key plant water consumers from fresh water to recycle water. The parameters and practical considerations described in this case study provided the basis for modeling options to reduce mine water use and for applying WAP principles to mine water system design.

Reducing Mine Water Use

This work was the first to compile a comprehensive list of strategies on how to reduce mine water use and how to evaluate the combined potential impact of these strategies. Scenarios were developed to demonstrate the potential water savings that can be achieved through implementing these strategies and the results indicated that water withdrawals can be dramatically reduced from current practices. In particular, this work highlighted for the first time the enormous potential water savings achievable by combining ore pre-concentration with tailing filtration.

Mine Water Network Design

This work was the first to apply the principles of water allocation problems to mine and mill water system design. While systematic approaches to water network analysis are common in other process industries and others have suggested the use of water allocation techniques for use in the mining industry, this work is the first to detail and demonstrate how to apply these principles to a mining site and the potential improvements that can be gained by using these principles. This work will help make water allocation planning more accessible and useful to the mining industry.

In addition, a software tool was developed to allow the practical implementation of water network analysis at mines and on mining projects. This software tool is an Excel-based platform which takes as inputs: water consumers and sinks, water sources, the water quality required or supplied, and the cooling energy required. The software tool output indicates which streams can be recycled and/or cleaned to reduce mine water network energy requirements. Details of the inputs, calculations and outputs were shown in the appendix.

Furthermore, the model highlights key potential goals for mining companies to target with respect to sustainability and mine water use. Mine sites can model their systems in order to determine practical targets for reducing mine water withdrawals, reducing mine water system energy requirements and reducing the quantity of high quality water required.

Combining Water Reduction Strategies with Mine Water Network Design

This work was the first to combine water reduction strategies with water allocation strategies. This combination can provide significant water and energy savings on mine sites. The potential impact is demonstrated through the use of models and scenarios describing in detail how to implement these strategies.

While all the water saving options described here have been described elsewhere, this is the first effort to integrate these concepts and methods into a common framework that allows a quantitative estimate of the individual and combined effects of a variety of different water saving options. A key finding is that the combination of ore pre-concentration and filtered tailings disposal can dramatically reduce mine site water consumption.

Table 10.1 summarizes the key objectives the models used, the existing material that was used, and the novel contributions each chapter makes.

Table 10.1 Summary of Key Objectives, Methodology and Novel Contributions

	Chapter 4	**Chapter 5**	**Chapter 6**	**Chapter 7**
Title	Estimating Global Mine Water Use	Case Study: Cerro Verde	Reducing Mine Water Requirements	Mine Water Network Design
Key Objectives	Estimate Global Mine Water Withdrawals by Commodity.	Describe the design, startup and optimization of a mine water system; Describe water system requirements of a typical mine.	Create a mine water system model and test options to reduce, reuse and recycle water on mine site water use.	Develop a method of reducing energy requirements in a mine water system; Model the potential impact of improved mine water system designs.
Methodology	Ore Production Model and Concentrate Production Model.	Case study description.	Mine water balance model created and difference scenarios tested.	Mine Water Network Design.
Application of Existing Material	Data taken from multiple secondary sources.	Case study is based on previous design documents completely or partially produced and written by the author.	Wells and Robertson's model (2003, 2004) is used to model TSF water losses. Individual techniques to reduce, reuse and recycle water are from secondary sources.	WAP methodologies have been developed for other industries. Pumping, evaporation, and heat loss calculations from Perry and Green (1997). Linear programming is a well established optimization technique.
Novel Contributions	Development of a unique large database of mine water withdrawals. First effort to define a methodology to determine global water and to complete comprehensive estimate of global water withdrawals by commodity.	First published description of all major parameters of sources and consumers at a major copper mine. Case study included details of changing a major water source due to water quality concerns.	First work to compile a comprehensive list of strategies on how to reduce mine water use and how to model and evaluate the combined potential impact of these strategies.	First effort to apply the principles of WAP to mine and mill water system design. First to develop a software tool to apply these principles for design purposes in mining industry.

To summarize, this work was the first to describe methods and examples to allow the systematic evaluation of mine water withdrawal data, mine water reduction options, and mine water network design. These methods provide a practical basis to help the mining industry implement the principles of sustainability with respect to reducing and improving mine water and energy use. Better design and management of mine water systems can reduce overall water consumption, reduce the use of high quality water, reduce energy requirements, minimize mine effluent, and maximize social satisfaction by reducing impacts on communities and watersheds.

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APPENDIX

This appendix details the software platform used for Mine Water Network Design method detailed in Chapter 7. The example developed in the chapter is used for illustration.

Table A.1 Developing the Description of the Site Water Balance

Step 1 - Description of Site Water Balance									
Sources					Sinks				
Name	m3/h	m3/day	Solids t/d	% Solids	Name	m3/h	m3/day	Solids t/d	% Solids
Run of Mine	43	1,020	50,000	98.0%	Tailing Storage	512	12,299	49,196	80.0%
Raw Water	512	12,299			Concentrate	4	89	804	90.0%
					Evaporation	39	931		
					Discharge Water		0		
Total	555	13,319	50,000	79.0%	Total	555	13,319	50,000	79.0%
Difference		0.0000	0.00%						

Table A.2 Developing the Description of All Major Water Sources*

Step 2 - Description of All Major Water Sources							
Water Sources	Flow	Max Temp	Quality	Quality	Pressure	pH	pH
Name	m3/h*	(°C)	(um)	(TDS)	(kPa)	Low	High
Raw Water	512	25	2000	1000	-980	6.5	7.5
SAG Cooling S	171	40	50	750	200	6.8	8.0
BM Cooling S	171	40	50	750	400	6.8	8.0
Comp Cooling S	171	50	25	750	200	6.8	8.0
Recycle Water	74	40	100	2000	0	9.0	11.0
Reclaim Water	4232	25	125	1500	-490	8.0	10.0

Table A.3 Developing the Description of All Major Water Consumers*

Step 3 - Description of All Major Water Consumers									
Water Consumers	Flow	Max Inlet	Quality	Quality	Pressure	Pressure	Pres. Loss	pH	pH
Name	m3/h*	Temp (°C)	(um)	(TDS)	Min (kPa)	Max (kPa)	Max (kPa)	Low	High
SAG Cooling C	171	30	50	750	300	700	100	6.8	8.0
BM Cooling C	171	30	50	750	500	1000	100	6.8	8.0
Comp Cooling C	171	40	25	750	300	700	100	6.8	8.0
Froth WW	120	60	1000	10000	300	300	0	6.5	11.0
Pump GSW	60	60	50	1000	800	800	0	6.8	8.0
Reagent DW	30	60	50	750	300	300	0	6.5	8.0
DS Spray	10	60	50	750	400	400	0	6.8	8.0
Com. Dilution	4599	60	1000	10000	300	1000	0	6.5	11.0

*Quality: um = μm (top sized particle in water); TDS = total dissolved solids, in ppm.

Table A.4 Developing the Construction of Energy Requirement Matrices

Step 4 - Construction of energy requirement matrices

Total Energy (kWh/m3)	Consumers							
Sources	SAG Cooling C	BM Cooling C	Comp Cooling C	Froth WW	Pump GSW	Reagent DW	DS Spray	Com. Dilution
Raw Water	0.5698	0.6468	0.5698	0.7238	0.7623	0.5698	0.6083	0.5313
SAG Cooling S	3.5275	3.6045	0.0770	0.2310	0.2310	0.0385	0.0770	0.0385
BM Cooling S	3.4505	3.5275	0.0000	0.1540	0.1540	0.0000	0.0000	0.0000
Comp Cooling S	7.0165	7.0935	3.5275	0.2310	0.2310	0.0385	0.0770	0.0385
Recycle Water	3.6815	3.7585	0.1925	0.3080	0.3850	0.1925	0.2310	0.1155
Reclaim Water	0.3812	0.4582	0.3812	0.4967	0.5737	0.3812	0.4197	0.3042

Pumping Pressure kPa	Consumers							
Sources	SAG Cooling C	BM Cooling C	Comp Cooling C	Froth WW	Pump GSW	Reagent DW	DS Spray	Com. Dilution
Raw Water	1280	1480	1280	1780	1780	1280	1380	1280
SAG Cooling S	100	300	100	600	600	100	200	100
BM Cooling S	-100	100	-100	400	400	-100	0	-100
Comp Cooling S	100	300	100	600	600	100	200	100
Recycle Water	300	500	300	800	800	300	400	300
Reclaim Water	790	990	790	1290	1290	790	890	790

Pumping Power Energy (kWh/m3)	Consumers							
Sources	SAG Cooling C	BM Cooling C	Comp Cooling C	Froth WW	Pump GSW	Reagent DW	DS Spray	Com. Dilution
Raw Water	0.4928	0.5698	0.4928	0.6853	0.6853	0.4928	0.5313	0.4928
SAG Cooling S	0.0385	0.1155	0.0385	0.2310	0.2310	0.0385	0.0770	0.0385
BM Cooling S	-0.0385	0.0385	-0.0385	0.1540	0.1540	0.0000	0.0000	0.0000
Comp Cooling S	0.0385	0.1155	0.0385	0.2310	0.2310	0.0385	0.0770	0.0385
Recycle Water	0.1155	0.1925	0.1155	0.3080	0.3080	0.1155	0.1540	0.1155
Reclaim Water	0.3042	0.3812	0.3042	0.4967	0.4967	0.3042	0.3427	0.3042

Cooling Required (°C)	Consumers							
Sources	SAG Cooling C	BM Cooling C	Comp Cooling C	Froth WW	Pump GSW	Reagent DW	DS Spray	Com. Dilution
Raw Water	-5	-5	-15	-35	-35	-35	-35	-35
SAG Cooling S	10	10	0	-20	-20	-20	-20	-20
BM Cooling S	10	10	0	-20	-20	-20	-20	-20
Comp Cooling S	20	20	10	-10	-10	-10	-10	-10
Recycle Water	10	10	0	-20	-20	-20	-20	-20
Reclaim Water	-5	-5	-15	-35	-35	-35	-35	-35

Cooling Energy (kWh/m3)	Consumers							
Sources	SAG Cooling C	BM Cooling C	Comp Cooling C	Froth WW	Pump GSW	Reagent DW	DS Spray	Com. Dilution
Raw Water	0.0000	0.0000	0.0000	0.0000	0.0000	0.0000	0.0000	0.0000
SAG Cooling S	3.4890	3.4890	0.0000	0.0000	0.0000	0.0000	0.0000	0.0000
BM Cooling S	3.4890	3.4890	0.0000	0.0000	0.0000	0.0000	0.0000	0.0000
Comp Cooling S	6.9780	6.9780	3.4890	0.0000	0.0000	0.0000	0.0000	0.0000
Recycle Water	3.4890	3.4890	0.0000	0.0000	0.0000	0.0000	0.0000	0.0000
Reclaim Water	0.0000	0.0000	0.0000	0.0000	0.0000	0.0000	0.0000	0.0000

Filtering Pressure Drop								
(kPa)	Consumers							
Sources	SAG Cooling C	BM Cooling C	Comp Cooling C	Froth WW	Pump GSW	Reagent DW	DS Spray	Com. Dilution
Raw Water	100	100	100	100	100	100	100	100
SAG Cooling S	0	0	100	0	0	0	0	0
BM Cooling S	0	0	100	0	0	0	0	0
Comp Cooling S	0	0	0	0	0	0	0	0
Recycle Water	100	100	100	0	100	100	100	0
Reclaim Water	100	100	100	0	100	100	100	0

Filtering								
Energy (kWh/m3)	Consumers							
Sources	SAG Cooling C	BM Cooling C	Comp Cooling C	Froth WW	Pump GSW	Reagent DW	DS Spray	Com. Dilution
Raw Water	0.0385	0.0385	0.0385	0.0385	0.0385	0.0385	0.0385	0.0385
SAG Cooling S	0.0000	0.0000	0.0385	0.0000	0.0000	0.0000	0.0000	0.0000
BM Cooling S	0.0000	0.0000	0.0385	0.0000	0.0000	0.0000	0.0000	0.0000
Comp Cooling S	0.0000	0.0000	0.0000	0.0000	0.0000	0.0000	0.0000	0.0000
Recycle Water	0.0385	0.0385	0.0385	0.0000	0.0385	0.0385	0.0385	0.0000
Reclaim Water	0.0385	0.0385	0.0385	0.0000	0.0385	0.0385	0.0385	0.0000

Treatment Pressure Drop								
kPa	Consumers							
Sources	SAG Cooling C	BM Cooling C	Comp Cooling C	Froth WW	Pump GSW	Reagent DW	DS Spray	Com. Dilution
Raw Water	100	100	100	0	100	100	100	0
SAG Cooling S	0	0	0	0	0	0	0	0
BM Cooling S	0	0	0	0	0	0	0	0
Comp Cooling S	0	0	0	0	0	0	0	0
Recycle Water	100	100	100	0	100	100	100	0
Reclaim Water	100	100	100	0	100	100	100	0

Treatment Pressure Drop								
Energy (kWh/m3)	Consumers							
Sources	SAG Cooling C	BM Cooling C	Comp Cooling C	Froth WW	Pump GSW	Reagent DW	DS Spray	Com. Dilution
Raw Water	0.0385	0.0385	0.0385	0.0000	0.0385	0.0385	0.0385	0.0000
SAG Cooling S	0.0000	0.0000	0.0000	0.0000	0.0000	0.0000	0.0000	0.0000
BM Cooling S	0.0000	0.0000	0.0000	0.0000	0.0000	0.0000	0.0000	0.0000
Comp Cooling S	0.0000	0.0000	0.0000	0.0000	0.0000	0.0000	0.0000	0.0000
Recycle Water	0.0385	0.0385	0.0385	0.0000	0.0385	0.0385	0.0385	0.0000
Reclaim Water	0.0385	0.0385	0.0385	0.0000	0.0385	0.0385	0.0385	0.0000

Table A.5 Inputting the System Limits for the Linear Program

Nodes	Flow Constraint		Supply/ Demand
Raw Water	-512	>=	-512
SAG Cooling S	-171	>=	-171
BM Cooling S	-171	>=	-171
Comp Cooling S	-171	>=	-171
Recycle Water	-74	>=	-74
Reclaim Water	-4232	>=	-4232
SAG Cooling C	171	>=	171
BM Cooling C	171	>=	171
Comp Cooling C	171	>=	171
Froth WW	120	>=	120
Pump GSW	60	>=	60
Reagent DW	30	>=	30
DS Spray	10	>=	10
Com. Dilution	4599	>=	4599

Table A.6 The Results of the Linear Program

Flow	From	To	Unit Cost
171	Raw Water	SAG Cooling C	0.570
0	SAG Cooling S	SAG Cooling C	3.528
0	BM Cooling S	SAG Cooling C	3.450
0	Comp Cooling S	SAG Cooling C	7.017
0	Recycle Water	SAG Cooling C	3.682
0	Reclaim Water	SAG Cooling C	0.381
171	Raw Water	BM Cooling C	0.647
0	SAG Cooling S	BM Cooling C	3.605
0	BM Cooling S	BM Cooling C	3.528
0	Comp Cooling S	BM Cooling C	7.094
0	Recycle Water	BM Cooling C	3.759
0	Reclaim Water	BM Cooling C	0.458
0	Raw Water	Comp Cooling C	0.570
0	SAG Cooling S	Comp Cooling C	0.077
171	BM Cooling S	Comp Cooling C	0.000
0	Comp Cooling S	Comp Cooling C	3.528
0	Recycle Water	Comp Cooling C	0.193
0	Reclaim Water	Comp Cooling C	0.381
0	Raw Water	Froth WW	0.724
0	SAG Cooling S	Froth WW	0.231
0	BM Cooling S	Froth WW	0.154
0	Comp Cooling S	Froth WW	0.231
0	Recycle Water	Froth WW	0.308
120	Reclaim Water	Froth WW	0.497
0	Raw Water	Pump GSW	0.762
60	SAG Cooling S	Pump GSW	0.231
0	BM Cooling S	Pump GSW	0.154
0	Comp Cooling S	Pump GSW	0.231
0	Recycle Water	Pump GSW	0.385
0	Reclaim Water	Pump GSW	0.574
0	Raw Water	Reagent DW	0.570
30	SAG Cooling S	Reagent DW	0.039
0	BM Cooling S	Reagent DW	0.000
0	Comp Cooling S	Reagent DW	0.039
0	Recycle Water	Reagent DW	0.193
0	Reclaim Water	Reagent DW	0.381
0	Raw Water	DS Spray	0.608
10	SAG Cooling S	DS Spray	0.077
0	BM Cooling S	DS Spray	0.000
0	Comp Cooling S	DS Spray	0.077
0	Recycle Water	DS Spray	0.231
0	Reclaim Water	DS Spray	0.420
171	Raw Water	Com. Dilution	0.531
71	SAG Cooling S	Com. Dilution	0.039
0	BM Cooling S	Com. Dilution	0.000
171	Comp Cooling S	Com. Dilution	0.039
74	Recycle Water	Com. Dilution	0.116
4112	Reclaim Water	Com. Dilution	0.304

Table A.7 Formatting the Results of the Linear Program

MWND Results								
Minimum Power	1643	kW						
Power (kW)	Consumers							
Sources	SAG Cooling C	BM Cooling C	Comp Cooling C	Froth WW	Pump GSW	Reagent DW	DS Spray	Com. Dilution
Raw Water	97	110	0	0	0	0	0	91
SAG Cooling S	0	0	0	0	14	1	1	3
BM Cooling S	0	0	0	0	0	0	0	0
Comp Cooling S	0	0	0	0	0	0	0	7
Recycle Water	0	0	0	0	0	0	0	9
Reclaim Water	0	0	0	60	0	0	0	1251
Water (m3/h)	Consumers							
Sources	SAG Cooling C	BM Cooling C	Comp Cooling C	Froth WW	Pump GSW	Reagent DW	DS Spray	Com. Dilution
Raw Water	171	171	0	0	0	0	0	171
SAG Cooling S	0	0	0	0	60	30	10	71
BM Cooling S	0	0	171	0	0	0	0	0
Comp Cooling S	0	0	0	0	0	0	0	171
Recycle Water	0	0	0	0	0	0	0	74
Reclaim Water	0	0	0	120	0	0	0	4112