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Life cycle based water footprint of selected metal production

Assessing production processes of copper, gold and nickel

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Executive summary

Growth in water consumption has seen increased pressure placed on global water resources that are in many cases now being extracted beyond sustainable levels. The mining industry is a large consumer of water, with global withdrawals from the industry estimated to be 6 to 8 billion m3 per annum (Gunson, 2013). Many mining operations are located in regions where water is scarce and there is competition between users. Given the potential impacts of increased demand to water resources, a renewed focus on the embodied water of different metal production processes is warranted.

Life-cycle assessment (LCA) is a useful methodology to assess the environmental impacts associated metal production processes. CSIRO has previously used LCA to develop embodied water estimates of production processes for a range of metals including aluminium, copper, gold, nickel, lead, zinc and titanium (Norgate and Lovel, 2004; 2006). Since these studies were published there has been considerable progress in developing water accounting standards and reporting methods. This has led to a large increase in water related data being made available through the corporate reporting of major mining companies. Changes in the validity of the underlying assumptions of the studies, particularly relating to assumed ore grades, have also brought about a need to revisit this topic.

This study provides an assessment of the water consumed during the production of copper, gold and nickel. A pyrometallurgy and a hydrometallurgy production route are considered for both copper and nickel, while gold production from refractory ores and non-refractory ores are considered. The direct (water consumed on-site) and the indirect (water consumed in the up-stream supply chain) water footprints of these metal production processes are quantified and summed together to provide an estimate of total embodied water consumption for each metal. A summary of the results of this analysis are shown in Table 1 below.

Table 1: Estimated embodied water of several copper, gold and nickel production processes (m^3/t refined metal)

Ore Type	Ore Grade	Process	Direct (m^3/t)	Indirect (m^3/t)	Total (m^3/t)
Copper Sulfide	0.75% Cu	Pyrometallurgy	91	37	128
Copper Oxide	0.75% Cu	Hydrometallurgy	70	198	267
Gold Non-refractory	3.5 g Au/t	Carbon-in-pulp (CIP)	244,701	69,732	314,433
Gold Refractory	3.5 g Au/t	Pressure Oxidation, CIP	284,235	149,112	433,347
Nickel Sulfide	1.3% Ni	Pyrometallurgy	68	35	102
Nickel Limonite	1.3% Ni	Hydrometallurgy (HPAL)	303	1,409	1,712

The direct water consumption found by this study largely agrees with that found by Norgate and Lovel (2004; 2006) once differences in ore grades are accounted for. However the values for indirect water consumption in all cases exceed those found by Norgate and Lovel (2004; 2006), particularly for the copper and nickel hydrometallurgical processes. This is due to an increased scope of materials and energy sources included within the indirect water consumption calculations. Reagents such as sulphuric acid and sodium cyanide that were not considered by Norgate and Lovel have been shown to be significant contributors to the overall embodied water.

The factors that can contribute to reductions in direct water consumption are discussed. Reducing the water lost to tailings storage facilities through improved dewatering techniques and increased recycling of water represents a major opportunity. However any measures taken must consider the local hydrology and quality of water resources in an area.

1 Introduction

Water resources are intertwined with almost all areas of economic, environmental and social functions that society depends upon. Growth of population and the global economy are placing additional demands on water resources and it is estimated that around 39% (±10%) of groundwater use exceeds aquifer recharge rates (Wada et al. 2010). Effective management of water resources requires a comprehensive understanding of water management issues at a variety of scales and importantly the cumulative impacts of different water uses. Integrating water management across these scales has the potential to boost economic and ecosystem productivity, improve human health outcomes and contribute to a more sustainable society. This report contributes to meeting these outcomes by providing increased understanding of the demands placed on water resources during the mined production of copper, gold and nickel.

The dominant societal consumer of water in Australia is the agricultural industry that accounted for 57% of consumption in 2010/2011 (ABS, 2012). Mining contributed to only 4.1% of Australian consumption during this period. This contribution is quite variable between states and is approaching 20% for Western Australia (Table 2). On a global scale mining withdrawals have been estimated to be 6 to 8 billion m^3 per annum (Gunson, 2013). Mining often occurs in arid regions where water is already scarce and this leads to competition for access to these resources between end-uses such as agriculture, manufacturing, human consumption and maintaining environmental flows. The environment itself is also a significant water consumer and requires sufficient water flows to enable sustainable ecological productivity. Internationally the competition between different water consumers has been shown to have social and economic ramifications for the mining industry (Kemp et al., 2010). In Australia there is a water price disparity amongst users, with agricultural users paying ten times less for the water they use, despite them producing much less economic value per unit of water (Williams, 2011).

Table 2: Contribution of mining to Australia's non-environmental water consumption (ABS, 2012)

	Mining (ML)			Total (ML)			% Mining		
Year	**08/09**	**09/10**	**10/11**	**08/09**	**09/10**	**10/11**	**08/09**	**09/10**	**10/11**
ACT	0	0	0	48	47	43	0.0	0.0	0.0
NSW	67	61	78	4,555	4,323	5,041	1.5	1.4	1.5
NT	21	25	24	160	167	167	12.9	14.7	14.4
QLD	120	114	125	3,341	3,112	2,964	3.6	3.7	4.2
SA	22	22	24	1,179	1,110	1,023	1.9	2.0	2.3
TAS	18	17	16	466	464	371	4.0	3.7	4.4
VIC	6	5	10	2,951	2,904	2,359	0.2	0.2	0.4
WA	252	245	264	1,361	1,386	1,369	18.5	17.7	19.3
Australia	**506**	**489**	**540**	**14,061**	**13,515**	**13,337**	**3.6**	**3.6**	**4.1**

Life-cycle assessment (LCA) methodologies provide a framework that enables a quantitative estimate of the impacts associated with producing and using materials or products. The LCA framework covers all stages of a products life. LCA of mining and metallurgy is typically of reduced scope and is conducted in the form of a "cradle-to-gate" analysis that includes the mining and processing of ore to produce a concentrated or refined metal product, but does not include the downstream use of that product. As will be discussed in more detail later in the report, methodologies specific to estimating "water footprints" or the "embodied water" of products are continually evolving.

Water use in metal production has been assessed previously by Norgate and Lovel (2004; 2006). They identified that there was little prior work conducted to make available LCA data for the contribution of different mining stages to the overall water footprint. Since these studies were conducted there has been some progress towards making water data for the mining industry more available, particularly due to the considerable data contained within the sustainability and corporate reporting of some major companies. However there can be subtle differences in the scope of reporting for different companies that make it difficult to apply this data (for instance does the reported values for water withdrawals include precipitation landing on the site). For this reason studies into the water footprints of metal production have usually focused on a single minesite and processing operation, especially if conducted as part of a water management strategy and access to operating and site measurements for the site are available.

1.1 Water footprinting standards and guidelines

There are several differences that make water footprinting, or embodied water estimates, a distinct challenge when compared to other related assessments of LCA impact characterisation factors such as "Gross Energy Requirement" (GER) or "Global Warming Potential" (GWP). A key difference is that energy can be accounted for as though it is consumed upon use, whereas water is not consumed in the same sense and may be recycled through a system many times or be returned to catchments. Currently an international standard for water footprinting in LCA, ISO14046, is being developed to address these issues and has recently been released as a draft in March 2013.

"The Water Footprint Assessment Manual, Setting the Global Standard" (Hoekstra et al., 2011) defines a water footprint as being the sum of three sub-categories of water footprints. They are: the blue water footprint, the green water footprint and the gray water footprint. Blue water footprints are an indicator of consumptive uses of fresh surface and groundwater, where "consumptive use" refers to:

1. Water evaporation;
2. Water being incorporated in to the product;
3. Water not being returned to the same catchment area;
4. Water not being returned to the catchment in the same period.

Green water footprints are an indicator for consumptive use of rainwater. The gray water footprint is a measure of the volume of water required to dilute pollution to acceptable limits defined by environmental standards.

The boundary of analysis is an important consideration that will greatly influence the end result of any LCA based study. In general there are three different scopes for water foot-printing (Gerbens-Leenes et al. 2007):

1. Direct water footprints (or operational water footprints) refer to consumptive uses occurring within a producer's operational boundary.
2. Indirect water footprints (or supply-chain water footprints) refer to consumptive uses in a producer's upstream supply chain.
3. End-use water footprints consider the consumptive uses occurring as a result of the consumption or use of a producer's product.

Generally water foot printing of mining and metal production has only considered direct and sometimes indirect water footprints. Despite the definitions of direct, indirect and end-use water footprints providing reasonable indications of supply system boundaries, determining the appropriate geographic boundaries for mine sites is not a straightforward task. Olivares et al. (2012) determined that a major shortcoming with the methodology proposed by Hoekstra et al. (2009) (the predecessor to Hoekstra et al. 2011) is that it does not provide adequate guidelines to define geographical boundaries of analysis.

For instance should water fluxes be considered based on boundaries of individual unit processes such as through the walls of a tailings storage facility (TSF)? Or should the area of the entire mine site lease be considered the boundary for analysis, in which case water flowing from the TSF through aquifers and into the mine could be discounted. Subtle complexities such as this complicate the reporting of site water accounting. Generally considering just unit processes within the mine and processing facilities is a straightforward approach, however the analysis can quickly become complicated when considering interactions with catchments and water stores.

The *Minerals Council of Australia* and the *Sustainable Minerals Institute* released the "*Water Accounting Framework for the Minerals Industry*" in April 2012 (MCA and SMI, 2012). This framework is meant to provide a consistent methodology for the calculation and reporting of water flows within the Australian minerals industry. It is important to recognise that there are subtle, but significant, differences between this standard and other reporting standards in use. The minerals industry water accounting framework excludes rainfall from the totals used to calculate water recycling and reuse (MCA and SMI, 2012). However the *Global Reporting Initiative (GRI)* guidance notes include rainfall as gray water within the total that recycling and reuse are calculated against. The GRI approach is entirely appropriate for sectors such as agriculture where all rainfall is effectively used, however the GRI approach does not adequately reflect the way water is used within the minerals industry.

2 Life Cycle Inventory Development

A variety of production processes are available to produce copper, gold and nickel depending on the mineralogy of ore available. The production processes considered by this study for a range of ore types are shown in Table 3. All figures and data refer to these assumed ore grades unless otherwise specified. The impacts of varying these ore grades are shown in later sections. As a comparison, average milled/treated Australian ore grades for copper and nickel production in 2010 were 0.73% Cu for copper, 1.2% Ni for sulfide ores and 1.3% Ni for laterite ores (Mudd and Weng, 2012).

Table 3: Production processes considered by this study.

Ore Type	**Ore Grade**	**Process Overview**
Copper Sulfide	0.75% Cu	Pyrometallurgy: mining, beneficiation, smelting and refining.
Copper Oxide	0.75% Cu	Hydrometallurgy: mining, heap leaching, solvent extraction-electrowinning.
Gold Non-refractory	3.5 g Au/t	Mining, cyanidation, carbon-in-pulp (CIP), electrowinning, smelting and refining.
Gold Refractory	3.5 g Au/t	Mining, flotation pressure oxidation, cyanidation, CIP, electrowinning, smelting and refining.
Nickel Sulfide	1.3% Ni	Pyrometallurgy: mining, beneficiation, smelting, ammonia leaching and hydrogen reduction.
Nickel Limonite	1.3% Ni	Hydrometallurgy: mining, high pressure acid leaching (HPAL), sulfide precipitation, ammonia leaching and hydrogen reduction.

2.1 Overview of water uses in mineral production systems

Mineral processes use water because (Napier-Munn and Morrison, 2003):

- Water is an efficient medium for transporting particles, mining and supply of reactants to the site of a reaction.
- Water provides a suitable medium for gravity separation.
- Water is an essential chemical ingredient in some processes.
- The density and viscosity of water enables stronger discrimination between particles of different sizes and densities

Dust suppression estimates in Ensham Central Project EIS Supplementary Report Section 11.

2.2 Copper

Copper ores can generally be divided into sulfide ores and oxide ores. Within porphyry ore deposits (currently the most common type of copper deposit), the majority of copper is usually present as sulfide ores, although a supergene "cap" of highly oxidized ore often forms through weathering processes near the surface. Depending on the nature and depth of individual ore deposits, and the prevailing economics of the time, copper can be mined either underground or from open-pits. The subsequent copper extraction processes utilized are then dependent on the mineralogy of the ore deposit, with sulfide ores generally being processed via pyrometallurgical techniques and oxide ores being processed via hydrometallurgy. Figure 1 shows the copper production processes that have been considered. A detailed overview of copper production processes are described in "*Extractive Metallurgy of Copper*" (Schlesinger et al., 2011).

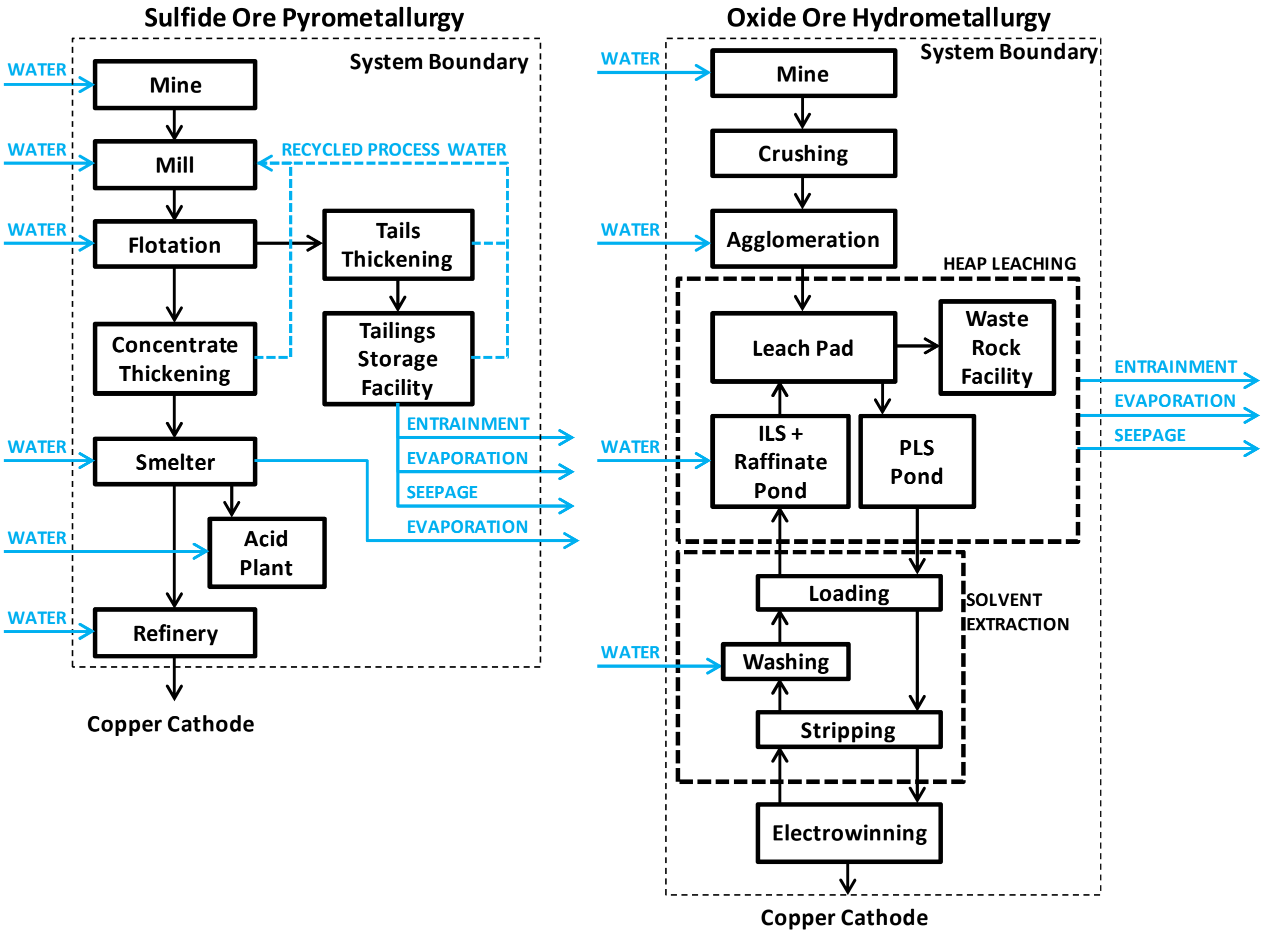

Figure 1: Copper processing routes and major water flows modelled

2.2.1 COPPER MINING

The rate of dewatering of minesites is highly geographically dependent. The porosity of surrounding rocks, intersection with both confined and unconfined aquifers, and the rate of aquifer recharge varies from site to site and so it is difficult to produce a typical flowrate for underground mine dewatering activities. Table 4 shows estimates for rates of dewatering at several Australian copper mines.

Table 4: Copper mine dewatering rates estimated from information in AMMOP (2013)

Mine	Type	kt Ore/yr	L/s	kL/yr	kL/t Ore
Degrussa	Underground	10,681	10	315,360	0.03
Ernest Henry	Openpit & Small Underground	15,000	64	2,007,500	0.13
Mt Lyell	Underground	2,000	75	2,372,500	1.19
Northparkes	Openpit & Underground	5,200	4	126,144	0.02
Total		32,881		4,821,504	0.15

Water entrainment in copper ores entering the system is very site specific. Copper ores are assumed to have a moisture content of 2%.

Water is used within underground mines primarily for dust suppression. There is little data on the water required for dust suppression activities available within the literature.

2.2.2 COPPER PYROMETALLURGY

The general process for treating sulfide copper ores is to beneficiate the ore, through milling and flotation, to produce a copper concentrate. This concentrate is then smelted to produce copper matte, the copper matte is then converted and cast to copper anodes, and the anodes are then electro-refined to produce copper cathodes of >99.99% Cu purity. Copper production processes often also produce other metal coproducts such as gold, silver and molybdenum. Sulphuric acid is often also produced as a by-product in smelter offgas scrubbing processes. This assessment has only considered the production of copper to avoid allocation issues. The main consumption of water in this process occurs in the concentration plant (i.e. milling and flotation stages).

Bleiwas (2012) describes the typical water balance for a copper concentrator and tailings facility. Mined copper sulphide ores undergo comminution processes (i.e. crushing and grinding) to liberate copper bearing minerals from the ore. Water is added to produce slurry containing approximately 35-40% solids. Additional water is also added during flotation processes as copper minerals are separated from gangue material. Tailings from this process are thickened and then discharged onto the tailings storage facility. The majority of water losses during copper production occur through the tailings storage facility as only limited amounts of water can be recycled back to the process from this water sink.

The copper concentrate leaving the flotation cells is thickened and filtered enabling additional process water to be recovered and recycled back to the mill. The filtered concentrate is then fed to dryers attached to smelting facilities. Due to the nature of water consumption in smelting operations, the processes of drying, smelting and converting of concentrate have been consolidated into a single unit process for the modelling[1]. Water is consumed in this stage primarily for cooling of furnace walls. Smelter off-gases high in sulphur dioxide are fed to an acid plant to produce sulphuric acid; a process that requires additional water inputs. The water consumption of the smelter and acid plant have been estimated to be 10.8 kL/t Cu, the average for Chagres smelter from 2003 to 2010 (Anglo-American, 2003-2010) and Altonorte Smelter from 2008 to 2010 (Xstrata, 2003-2010)

Copper anodes produced from the smelter are electrolytically refined to produce LME grade A copper cathodes (>99.99% Cu). This process occurs requires additional water inputs of around 0.51 kL/t Cu, based upon the average for Townsville Refinery from 2003 to 2010 (Xstrata, 2003-2010)

2.2.3 COPPER HYDROMETALLURGY

Copper oxide ores are processed using hydrometallurgical techniques. Various leaching methods exist to extract copper from the ore such as heap leaching, dump leaching, vat leaching, and in-situ leaching. Of these methods heap leaching is by far the most common method globally. Heaps can be operated in an on-off configuration where piles are placed on the pads and then removed again once most of the leaching copper has been extracted. Or alternatively they can be operated as a permanent heap where additional ore is just continuously stacked, with the heap becoming the ultimate waste rock dump after mining operations have concluded. Both methods are utilised by the industry and the best method for a given site is dependent on localised characteristics of the ore and surrounding geography.

Prior to leaching, mined ore undergoes comminution and subsequent agglomeration processes to ensure adequate porosity and structural integrity of the heap. Water and sulphuric acid are added during the agglomeration stage. Additional sulphuric acid solution is then added via an irrigation system to the constructed heap where it percolates and leaches copper from the ore. The pregnant leach solution (PLS) is recovered from the base of the heap and stored in (PLS) ponds prior to solvent extraction-electrowinning

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[1] Smelting and converting are essentially two stages of the same underlying physical process; that is the separation of iron and sulphur through oxidation processes.

(SX-EW) processes. Entrainment of leach solutions within the heap is a significant area of water and acid losses in the production system.

Solution from the PLS pond is pumped to solvent extraction processes where an organic solvent is loaded with copper from the PLS and the raffinate flows back to a storage pond. The copper in the organic solvent is then stripped into an electrolyte solution that flows to the electrowinning plant. The stripped organic solvent is then washed and recycled back to the loading stage.

The copper enriched electrolyte is electro-won to produce copper cathodes for sale to the market, with the depleted electrolyte from this process being recycled back to the stripping stage. The overall water balance of the SX-EW processes are controlled by the inputs of the PLS, water inputs for washing of organic solvent and the bleed rate of electrolyte. Electrolyte bleeds have been modelled to return to the "ILS + Raffinate Pond" with the wash water. Due to this the only major water sink for the process is entrainment, evaporation and seepage associated with the leach pads, waste rock facilities, "ILS+Raffinate Pond" and the "PLS Pond". Unfortunately insufficient data is available within the public literature to be able to disaggregate water losses beyond this. Therefore water losses have been modelled to occur as a whole from the heap leaching operations (see Figure 1 for the relevant process boundary), with assumed water loss to evaporation, entrainment and seepage being the same as for the tailings storage facility described by Wels and Robertson (2003).

2.3 Gold

Gold ores can generally be classed as either refractory or non-refractory. Non-refractory ores are free milling and can be processed either through gravity separation or carbon-in-pulp (CIL) or carbon-in-leach (CIP) processes. In refractory ores the gold is bound to a sulfide fraction and requires flotation and additional processes such as roasting, pressure oxidation and bio-oxidation before they can be processed via CIP or CIL processes. Following CIP/CIL or gravity separation the product stream is smelted and then refined to produce refined gold. The process routes considered by this study for gold production and the major water inputs and outputs to this are shown in Figure 2.

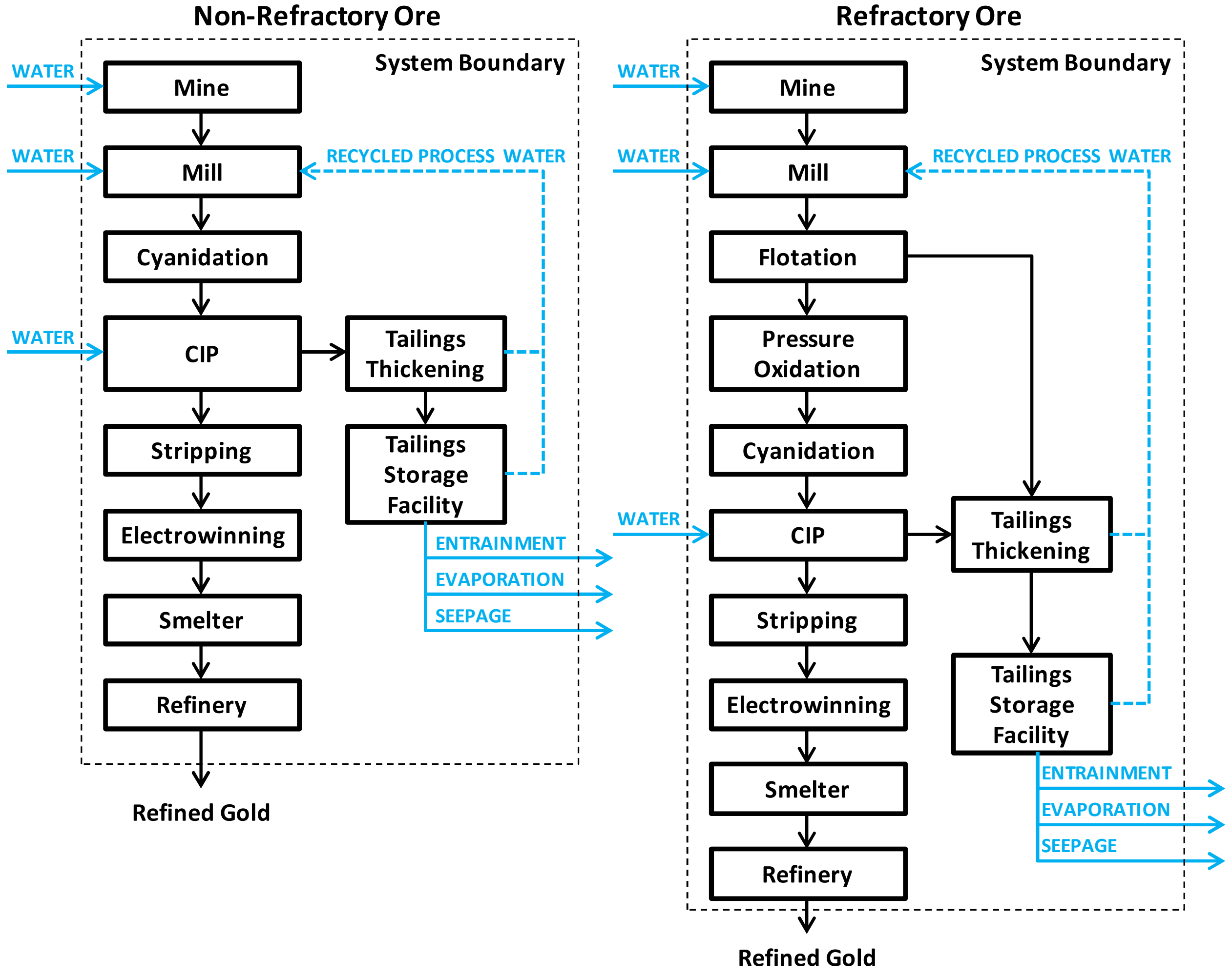


Figure 2: Gold processing routes and major water flows modelled

2.3.1 GOLD MINING

Gold only mines are generally underground operations that target low tonnages of high-grade material. As with copper mines, the rate of dewatering from gold mines is highly variable between sites due to the localized hydrology of individual mine sites. The dewatering rates for a variety of gold mines are shown in Table 5 and were aggregated to produce an estimated dewatering rate of 0.46 kL/t ore for Australian gold mines. The moisture content of gold ores has been assumed to be 2%.

Table 5: Gold mine dewatering rates estimated from information in AMMOP (2013)

Mine	Type	kt Ore/yr	L/s	kL/yr	kL/t Ore
Randalls	Openpit	1,800	2	78,400	0.04
CVO-Ridgeway	Underground	5,000	22	693,500	0.14
Henty Gold Mine	Underground	260	15	473,040	1.82
Jundee	Underground	874	10	315,360	0.36
Paulsens Gold Mine	Underground	188	2	63,072	0.34
Tanami	Underground	1,612	25	788,400	0.49
Tasmania Mine	Underground	298	70	2,207,520	7.41
Total		10,032		4,619,292	0.46

2.3.2 NON-REFRACTORY ORES

The main water addition is during the crushing and grinding processes. The density of slurry leaving the mill is generally 25-40% solids. The slurry is transported to tanks and treated with cyanide to leach out the gold. Carbon-in-pulp (CIP) technology is used to separate and strip the gold into a solvent, and water is added during washing processes. Tailings are separated during the CIP process and thickened before being discharged at a TSF. Water is recovered from the thickener and TSF and recycled back to the mill. The gold dissolved in solvent is electrowon to produce gold cathodes. The cathodes are then smelted to produce dore and then further processed using chlorination and electrolysis to produce high-purity refined gold. Norgate and Haque (2012) produced a detailed LCA of gold production processes that forms the basis of this analysis. Water consumption in the CIP and stripping stage has been estimated to be 65,816 kL/t Au, based upon data for Paulsens Gold Mine (AMMOP, 2013).

2.3.3 FLOTATION AND PRESSURE OXIDATION OF REFRACTORY GOLD ORES

Refractory ores require additional processing to separate the gold particles from gangue material in the ore, as a high proportion of the gold is contained within sulphide mineralogy. Generally refractory ores have to be grinded to a smaller particle size to liberate and expose the gold containing minerals. Sulphide is removed through oxidation processes such as bioleaching, roasting or pressure oxidation. Pressure oxidation is a common process for treating refractory ores and acid inputs to this process are dependent on the sulphur content of the ore. Gold can then be separated from the oxidised using the cyanidation and CIP processes as described for non-refractory ores.

2.4 Nickel

The nickel production processes considered by this study along with major water flows are shown in Figure 3. Currently the majority of nickel production is sourced from two broad categories of ore. These are sulfide ores and laterite ores. Laterite ore deposits are formed through weather processes that create distinct layers of ore mineralogy. Each of these layers can have a distinct class of ore, with common classifications being limonite, smectite or saprolite ore. Different processes are suitable for different types of laterite ore. This study considers two production processes for nickel: A pyrometallurgical process for sulfide ore and a hydrometallurgical production process for limonite ore. The production processes for nickel are described in detail in *"Extractive Metallurgy of Nickel, Cobalt and Platinum-Group Metals"* (Crundwell et al., 2011). Norgate and

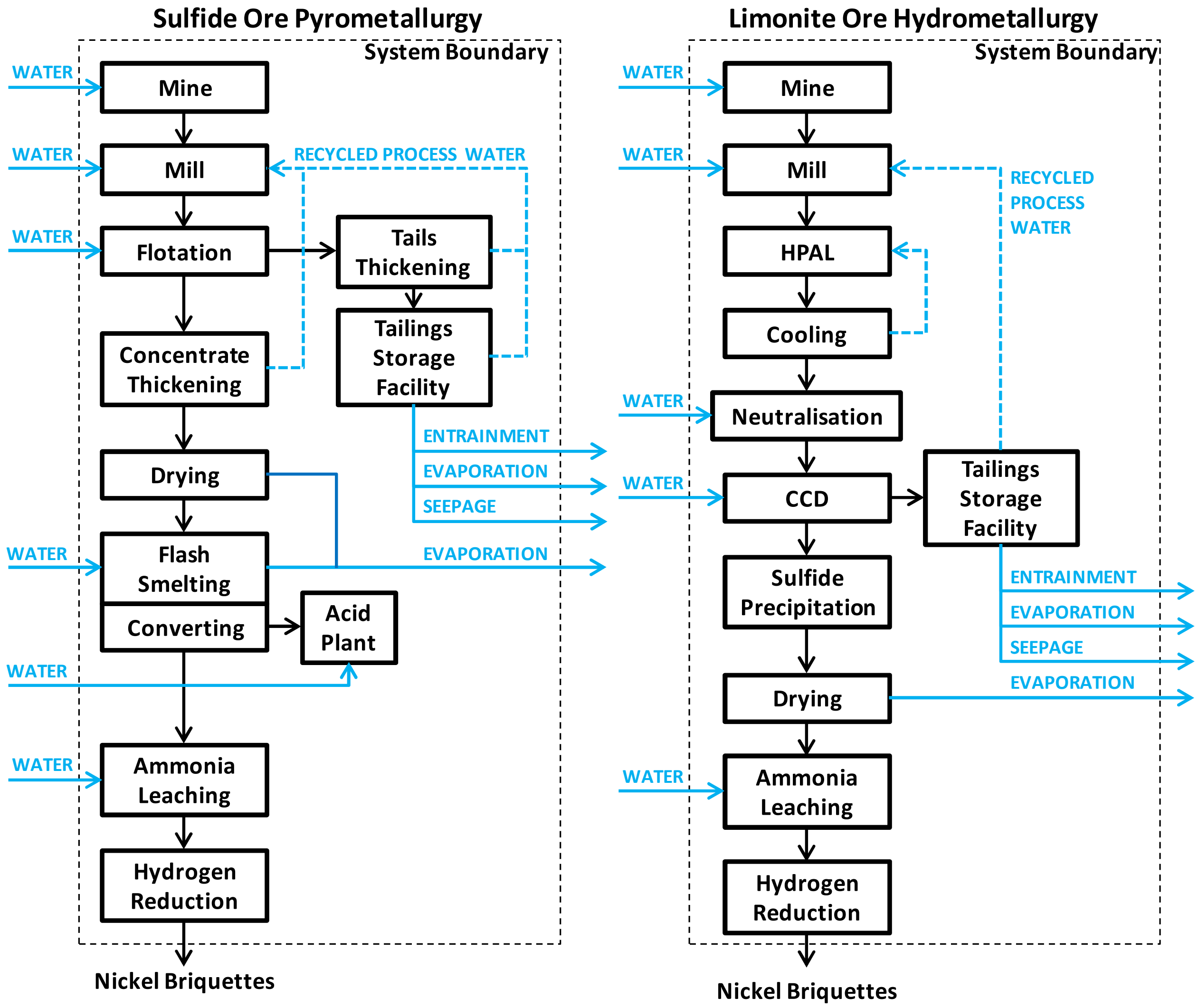


Figure 3: Nickel processing routes and major water flows modelled

2.4.1 NICKEL SULFIDES

The processing of nickel sulfides is similar to production processes for other base metals such as copper, zinc and lead.

Nickel sulfides can be mined using either underground or open pit methods. Rates of dewatering for mines are highly site specific. Table 6 shows an estimate of dewatering rates for the Flying Fox mine site in Western Australia. Due to limited data availability on the rate of dewatering in nickel mines), it has been assumed that the rate of dewatering for sulfide nickel mines is similar to the value of 0.15 kL/t Ore that was obtained for copper mines.

Table 6: Flying Fox nickel mine dewatering rate estimated from information in AMMOP (2013)

Mine	Type	kt Ore/yr	L/s	kL/yr	kL/t Ore
Flying Fox	Underground	350	30	946,080	2.70

The natural moisture content is highly regionally dependent and is affected by factors such as mine dewatering and localised aquifer hydrology. Assuming that mining occurs in an arid region experiencing, a moisture content of 2% will be applied to nickel sulphide ores.

Nickel sulphide ore is crushed and milled to liberate fine particles containing nickel minerals. Water is added to produce slurry that is about 35% solids entering the flotation plant. The flotation process generally operates at about pH 8, with additional water being added during these processes. The concentrate produced is thickened and filtered, with this water being reclaimed and recycled back to the milling stage. Overflow water from the tailings thickener is also reused in slurry preparation in the mill. Underflow from the tailings thickener is pumped to the tailings storage facility where water is lost through entrainment, evaporation and seepage processes. Some water is recovered from the TSF by decanting of wetted areas.

Filtered concentrates with a moisture content of about 7% are fed to a coal bed dryer and then flash smelted and converted to remove the iron and sulphur content via oxidation. Some water is consumed to cool the flash smelter. Converting happens in a Peirce Smith converter with slag and off gases being recycled back to the Flash Smelter. Offgases high in sulphur dioxide are sent to an acid plant to produce sulphuric acid, a process that also consumes water.

Nickel matte from the flash smelter refined using ammonia leaching and hydrogen reduction to produce nickel briquettes for sale to the market. The refining process has been estimated to consume 13.8 kL/t Ni.

2.4.2 NICKEL LATERITES

Mining of nickel laterites is almost exclusively conducted using open pit methods. Nickel laterite orebodies can contain layers of different types of mineralisations such as limonite, smectite and saprolite. The most economic process to use is heavily dependent on the proportion of different minerals in the ore. This study considers the processing of a limonite dominant ore through high pressure acid leaching (HPAL), sulphide precipitation, and ammonia leaching and hydrogen reduction.

Lateritic ores typically have high moisture contents (10-30%, see Table 7), a fact that can in part be attributed to the way in which these types of deposits form. Laterite deposits are formed from tropical weathering processes and so naturally occur in regions that lead to high ore moisture contents. Therefore groundwater brought into nickel production processes through ore entrainment can be expected to be higher for lateritic ores when compared to sulfide ores. As nickel laterite deposits are located close to the surface, dewatering rates are expected to be lower than for nickel sulfide mines. However in the absence of any reliable data on dewatering rates for nickel laterite mines, the same 0.15 kL/t ore dewatering rate obtained for copper mines has been assumed.

Table 7: Nickel laterite ore moisture contents (Whittington and Muir, 2000)

Mine	Ore Moisture Content
Bulong	< 35%
Cawse	< 10%
Murrin-Murrin	~ 30%
Syerston	13%
Moa Bay	>20%
Approx. Mean	21.6

The limonite ore that has been mined is crushed, milled and mixed with water to produce slurry. The slurry is fed to an autoclave where it is leached with sulphuric acid in a high pressure and temperature environment. Water is generated during the HPAL process as a byproduct of reactions involving sulphuric acid. Slurry leaving the autoclave is flashed cooled and the steam produced is recycled back to the HPAL to maintain optimal operating conditions. The cooled slurry is neutralised and the solids are separated using counter-current decantation to produce a clarified pregnant leach solution. Wash water is added to this process at a ratio of 1:1 with the slurry. The clarified solution is treated with hydrogen sulfide and a nickel sulfide precipitate is produced.

3 Results

3.1 Direct Water Footprint

Table 8 has been constructed in accordance with the methodology outlined in the Water Accounting Framework for the Minerals Industry (MCA and SMI, 2012). The input-output statement describes the fluxes of water entering and leaving the boundaries of the mine site as a whole. Rainfall has been excluded from this analysis. When preparing this table it was assumed that all external water withdrawals are from groundwater aquifers and that TSF and heap pad seepage is returned to these systems. The groundwater consumption provided in Table 8 is equivalent to the "direct blue water footprint" as defined by Hoekstra et al. (2011).

Table 8: Input-output statements based upon base ore grades

	Copper		Gold		Nickel	
	Pyromet.	**Hydromet.**	**Non-refrac.**	**Refrac.**	**Sulfide**	**Limonite**
Inputs	m^3/t Cu	m^3/t Cu	m^3/t Au	m^3/t Au	m^3/t Ni	m^3/t Ni
Groundwater						
- Mine Dewatering	21.6	31.1	145,758	177,971	13.6	16
- Ore Entrainment	2.94	4.24	6,337	7,738	1.9	23
- Withdrawal	79.2	41.9	121,123	133,346	59.6	272.7
Total	103.7	77.2	273,219	319,055	75.1	311.3
Outputs						
TSF/Heaps						
- Entrainment	59.9	36.39	134,812	164,606	35.2	39.1
- Evaporation	19.58	11.90	44,073	53,813	11.50	12.8
- Seepage	12.7	7.7	28,518	34,820	7.4	8.3
Task Losses	11.6	21.2	65,816	65,816	20.9	251.2
Total	103.7	77.2	273,219	319,055	75.0	311.3
Consumption						
Groundwater	91	69.5	244,701	284,235	67.6	303.0
- per tonne of ore	0.62	0.32	0.76	0.72	0.73	2.8

The main process variable that influences the direct water footprint of metal production is the grade of ore being mined. There is a general trend within the industry to mine and process lower grade ores. As ore grades decline more material has to be processed to produce the same amount of metal product. The impacts of varying ore grade on the direct water footprint for each metal production process are shown in Figure 4. Note that the models do not account for the fact that lower grade ores may require grinding to finer sizes and that this would have a negative impact on water recovery from thickeners and the tailings storage facility.

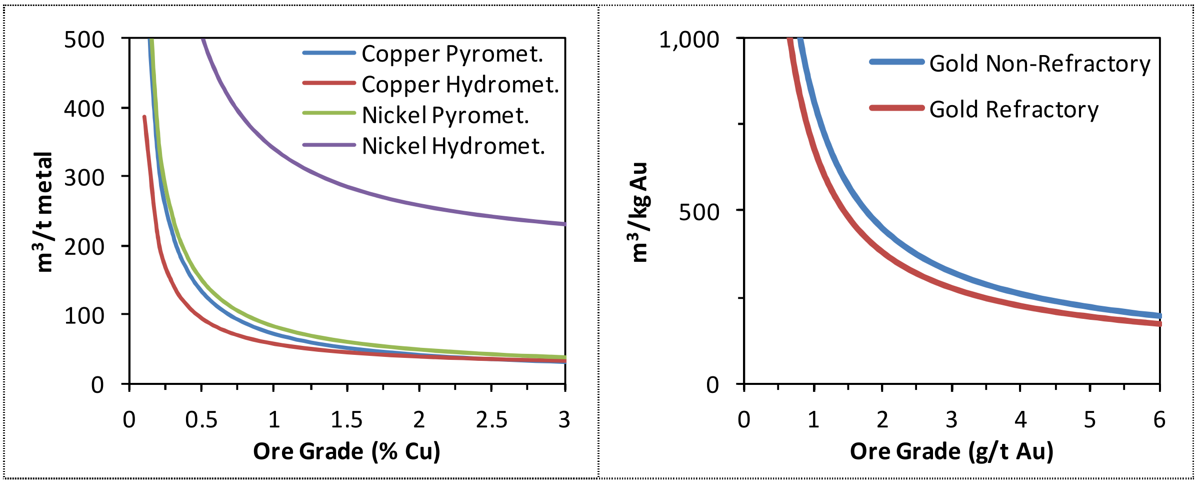


Figure 4: Estimated direct water consumption for refined metals versus ore grade

Another key factor that influences the direct water consumption of metal production is the degree of water recycling that occurs in the concentrator and tailings storage facility. The main opportunity to recover and reuse/recycle water is from the tailings dewatering processes. The base model assumes that tailings are dewatered to a solids density of 55% (w.w) before being discharged to the tailings storage facility. Figure 5 shows the impacts on direct water consumption for copper sulfide processing from dewatering tailings to different solids densities (similar results are found for gold and nickel). In general, dewatering to higher tailings solids densities will result in lower direct water consumption and as ore grades decline the differences are exacerbated.

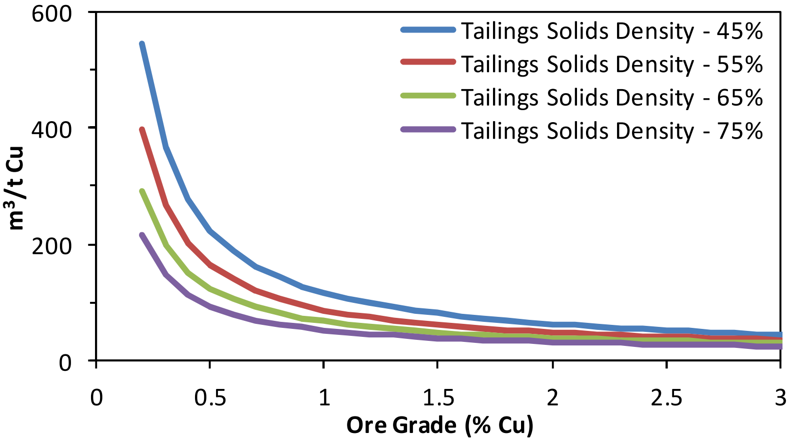


Figure 5: Variation of copper sulfide processing direct water consumption with tailings solids density

3.2 Indirect Water Footprint

The production of the metal products using the various production routes requires the consumption of materials and energy. There is an indirect water footprint associated with the supply of these materials and energy to the production processes. Life cycle inventory tables (shown in Appendix A) were developed for each metal production process to enable an estimate of the indirect water consumption. The embodied water consumption for each of the input materials and energy are shown in Table 9. These values were estimated from LCA databases using the LCA software SimaPro. For each of these materials a range of embodied water estimates are available within the databases. Estimates specific to Australia were preferentially selected, although Australian estimates were not available for all materials.

Table 9: Embodied water estimates for common materials consumed during copper, gold and nickel production (SimaPro, etc.)

Embodied water					
Acetylene	0.1016	m^3/kg	Hydrogen	0.0025	m^3/kg
Ammonia	0.0032	m^3/kg	Hydrogen sulphide	0.0377	m^3/kg
Carbon black	0.0005	m^3/kg	Lime	0.0020	m^3/kg
Chlorine	0.0614	m^3/kg	Natural gas	0.0000	m^3/kg
Cobalt	0.4793	m^3/kg	Oxygen	0.0042	m^3/kg
Diesel	0.0013	m^3/kg	Silica sand	0.0015	m^3/kg
Black coal	0.0003	m^3/kg	Sodium cyanide	0.1956	m^3/kg
Electricity	0.0021	m^3/kWh	Sodium hydroxide	0.0610	m^3/kg
Explosives	0.0338	m^3/kg	Sulphuric acid	0.0541	m^3/kg
Fuel oil	0.0014	m^3/kg	Steel	0.0027	m^3/kg
Hydrochloric acid	0.0254	m^3/kg	Vanadium	0.0010	m^3/kg

The indirect water consumption was estimated to be less than the direct water consumption for gold production, copper pyrometallurgy and nickel pyrometallurgy. Figure 6 shows that the indirect water consumption displays the same trend with ore grade as direct water consumption. As ore grades decline more materials and energy has to be processed to achieve the same output of concentrate. Indirect water consumption associated with the smelting and refining stages of production is largely a factor of concentrate grade and mineralogy rather than the original ore grade.

Indirect water consumption is higher than the direct consumption for copper and nickel hydrometallurgical processes. Figure 7 shows the contribution of different materials or energy sources to the indirect water consumption for the base case ore grades (0.75% Cu, 3.5 g/t Au and 1.3% Ni). For copper and nickel pyrometallurgy, the water associated with electricity consumption accounts for almost half of the total indirect water consumption. Interestingly sodium cyanide, which is sometimes used as a flotation reagent, is also a large source of indirect water consumption for copper pyrometallurgy. No data on the embodied water for some important flotation reagents such as xanthates was able to be found, however they may be important contributors to the embodied water of metal concentrates.

The production of acid used in hydrometallurgical processes is a water intensive process. The embodied water of sulphuric acid consumed in copper heap leaching was estimated to be approximately 189 m^3/t Cu (assuming 3.5 t H_2SO_4/t Cu), or roughly twice the processes direct water consumption. Sulphuric acid consumption for high pressure acid leaching of limonitic nickel ores was estimated to accounts for 1,353 m^3/t Ni.

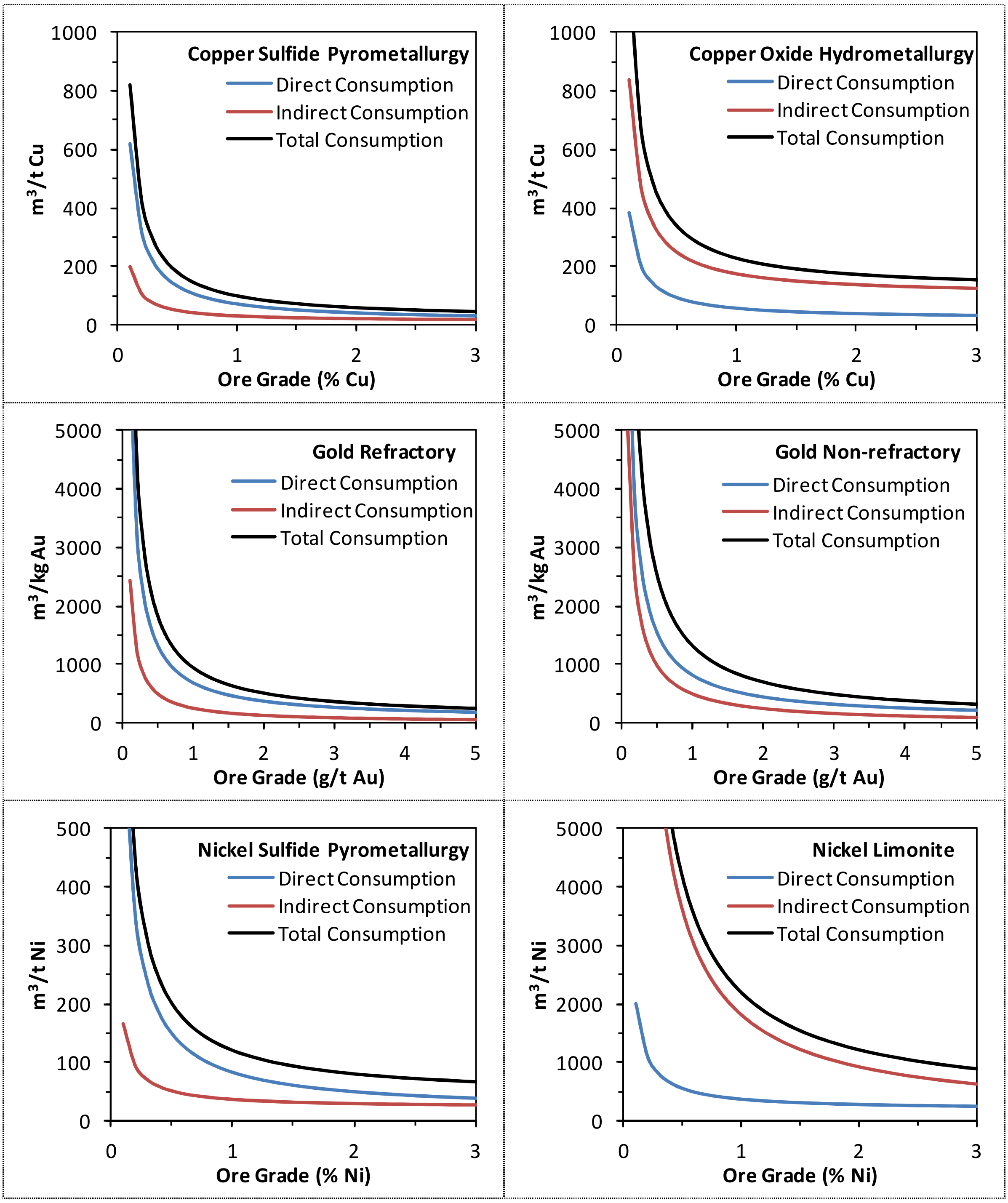
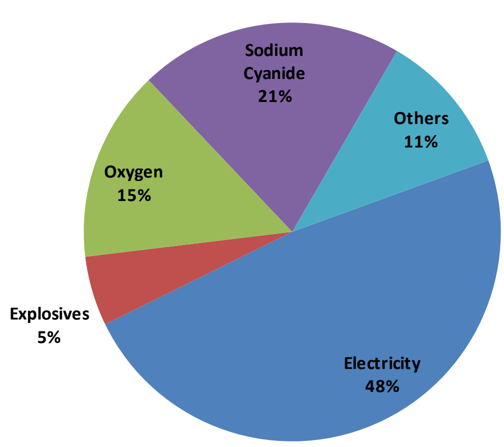
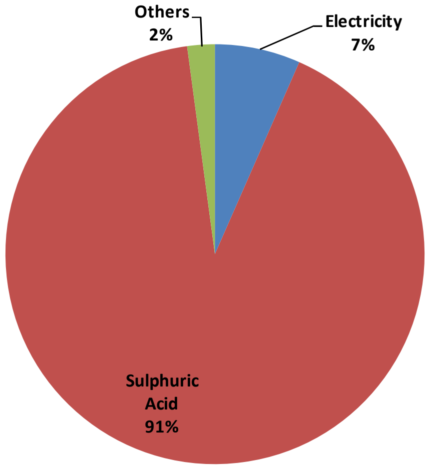


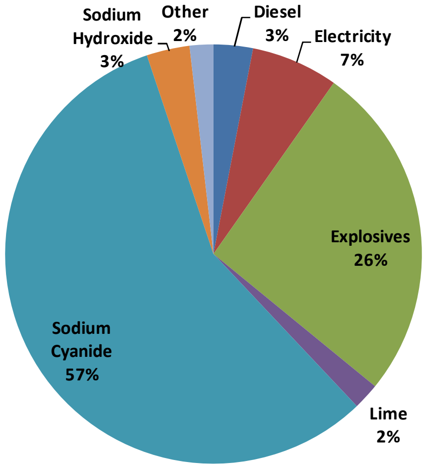
Figure 6: Estimated water consumption for refined metals versus ore grade. Note the differing axis scales and units.



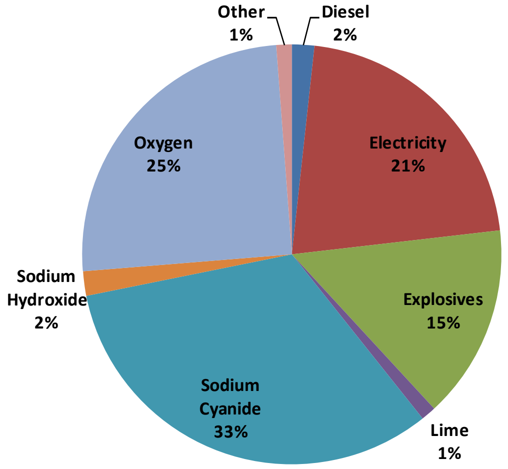
Copper Sulfide Pyrometallurgy



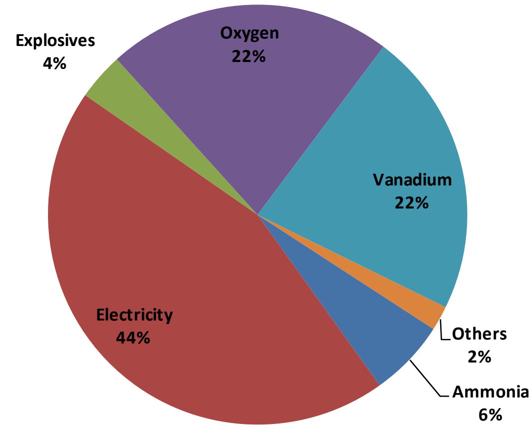
Copper Oxide Hydrometallurgy



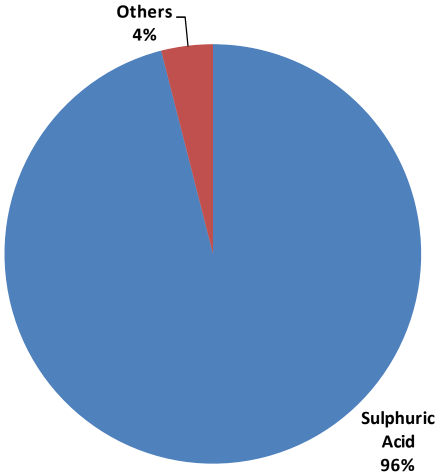
Gold Non-refractory



Gold Refractory



Nickel Sulfide Pyrometallurgy



Nickel Limonite HPAL

Figure 7: Contribution of materials and energy to indirect water consumption.

3.3 Total embodied water

The results of the study for the base case ore grades are shown in Table 10. The direct water consumption represents 65-80% of the total embodied water for copper and nickel pyrometallurgy and the gold production processes. Whereas for the copper and nickel hydrometallurgy processes the indirect water consumption represents 75-82% of the total embodied water.

Table 10: Summary of the embodied water of copper, gold and nickel production processes

Ore Type	Ore Grade	Process	Direct (m^3/t)	Indirect (m^3/t)	Total(m^3/t)
Copper Sulfide	0.75% Cu	Pyrometallurgy	91	37	128
Copper Oxide	0.75% Cu	Hydrometallurgy	70	198	267
Gold Non-refractory	3.5 g Au/t	Carbon-in-pulp (CIP)	244,701	69,732	314,433
Gold Refractory	3.5 g Au/t	Pressure Oxidation, CIP	284,235	149,112	433,347
Nickel Sulfide	1.3% Ni	Pyrometallurgy	68	35	102
Nickel Limonite	1.3% Ni	Hydrometallurgy (HPAL)	303	1,409	1,712

A breakdown of the contribution of individual unit processes to the embodied water is shown for each production process in Figure 8. The embodied water is mostly associated with the milling and flotation stages of production for all processes, excluding the copper and nickel hydrometallurgy routes. For the hydrometallurgy routes, the indirect water associated with acid consumption is by far the largest contributor to the overall embodied water.

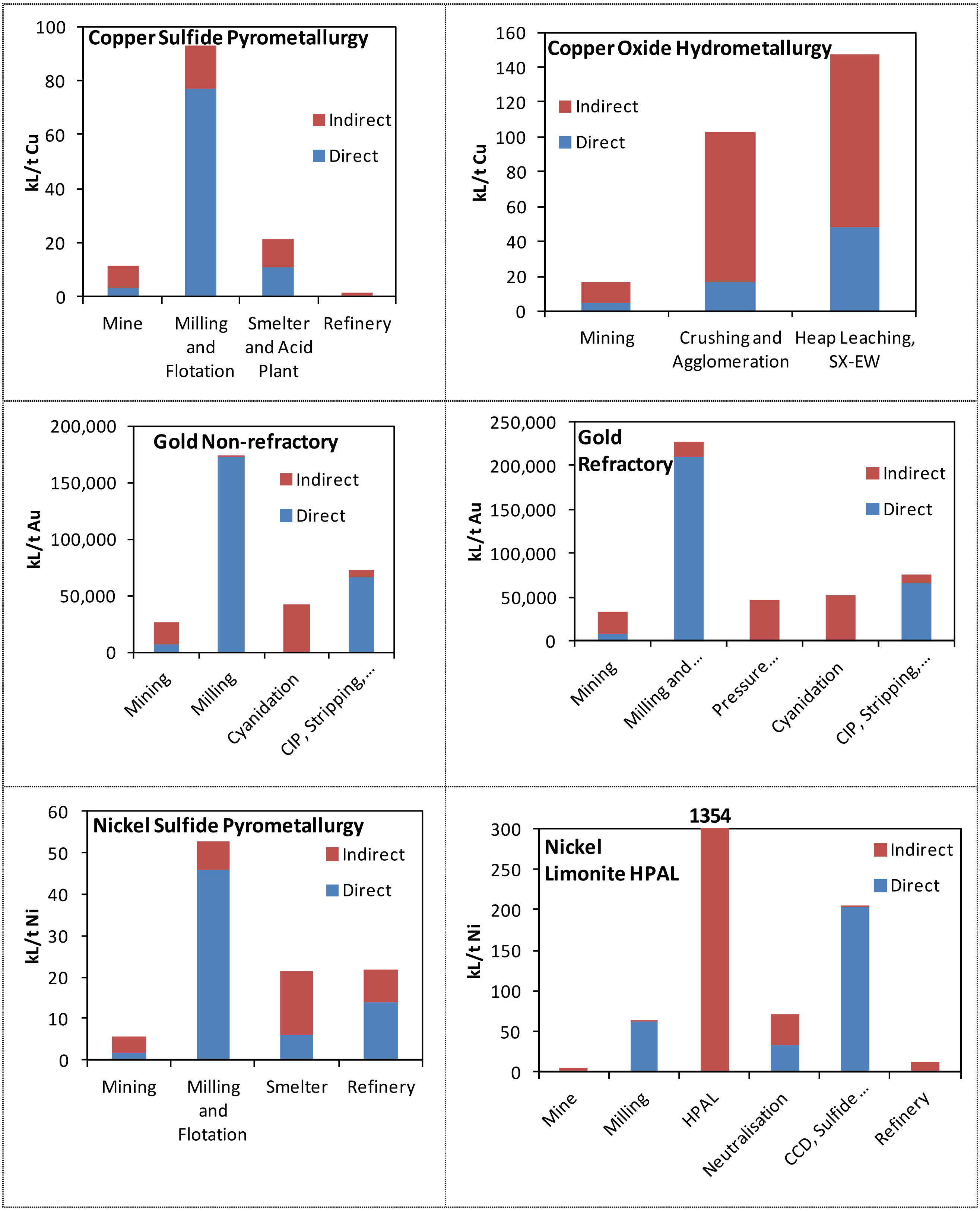


Figure 8: Direct and indirect water consumption by unit processes.

4 Discussion

4.1 Comparisons with other studies

CSIRO has previously conducted work into the embodied water of various metal production processes (Norgate and Lovel, 2004; 2006; Norgate and Haque, 2012). The results of these studies are plotted alongside this study for comparison in Figure 9.

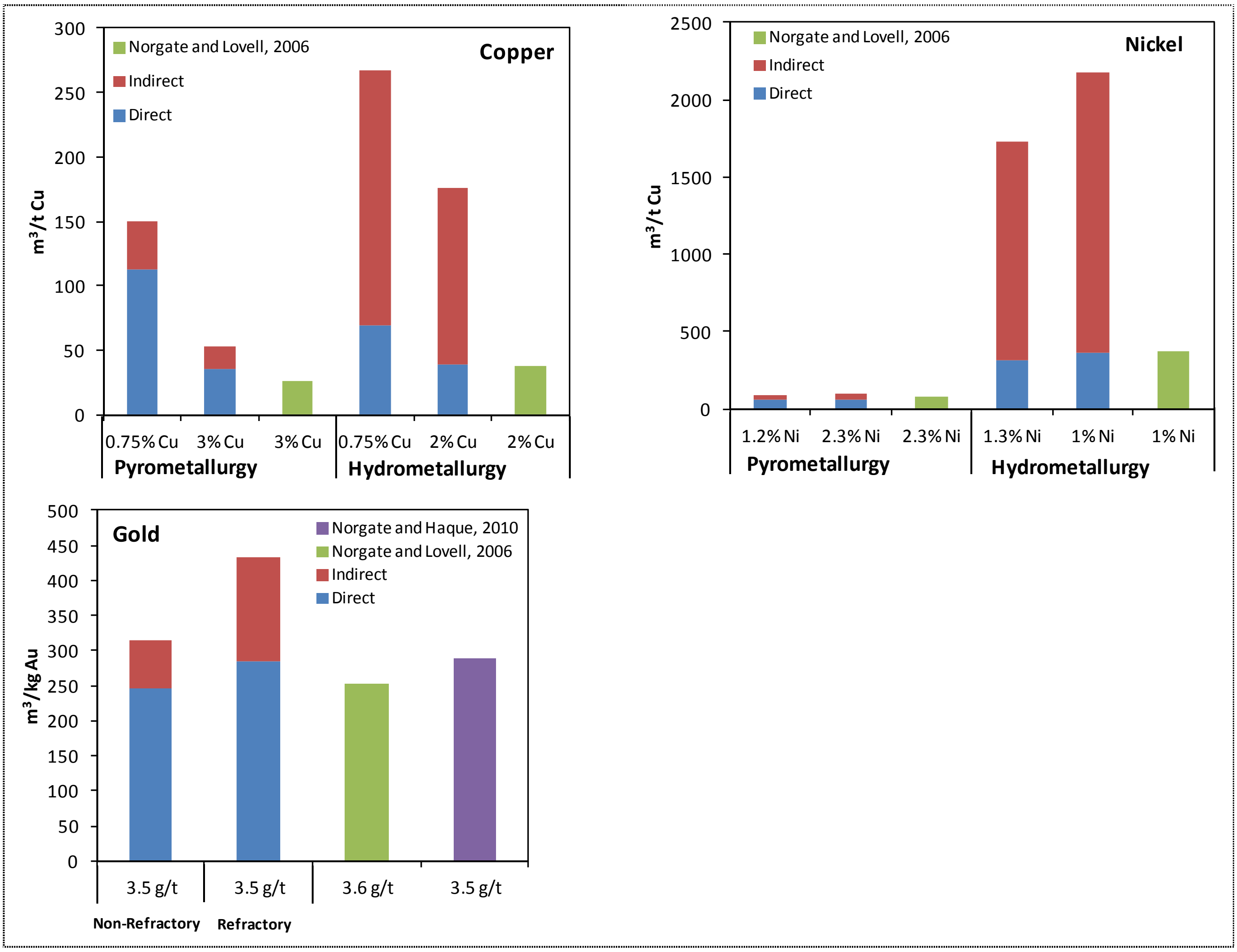


Figure 9: Comparison between this study and others

When differences in ore grades are taken into account, the total water consumption in all cases exceeds the estimates provided by Norgate and Lovel (2006). The direct water consumption found in this study is similar to the total water consumption found in other studies. Indirect water consumption is significantly higher for this study and this is particularly evident for the copper and nickel hydrometallurgical processes. The main source of indirect water consumption in Norgate and Lovel (2006) was associated with electricity production from black coal and the study did not account for many materials consumed such as sulphuric acid.

The indirect water consumption for the hydrometallurgical processes is highly sensitive to the unit acid consumption and the embodied water of this acid. In Chile it has been shown that the acid consumption required to produce 1 tonne of electrowon copper has grown from 2.86 t H_2SO_4/t Cu in 2004 to 3.65 t

H_2SO_4/t Cu in 2010 (Chilean Copper Commission, 2011). This is expected to increase further to 4.5 t H2SO4/t Cu in 2020 as leachable copper ore grades decline.

This study assumes that sulphuric acid consumed by processes has been produced by a stand-alone acid plant with an embodied water consumption of 54.1 L/kg H_2SO_4. Sulphuric acid produced from smelter off-gases are likely to have lower embodied water consumption as the majority of the water consumption are allocated to the primary metal products. For many regions the production of sulphuric acid is largely a by-product of these metallurgical processes. In the Chilean sulphuric acid market 96% of acid is produced as a by-product of smelting and 96% of acid is consumed by copper hydrometallurgy (Chilean Copper Commission, 2011). The supply and demand of acid between copper pyrometallurgy and hydrometallurgy is essentially balanced in this case. This study has assumed no allocation of impacts to the acid produced as a by-product of smelting processes.

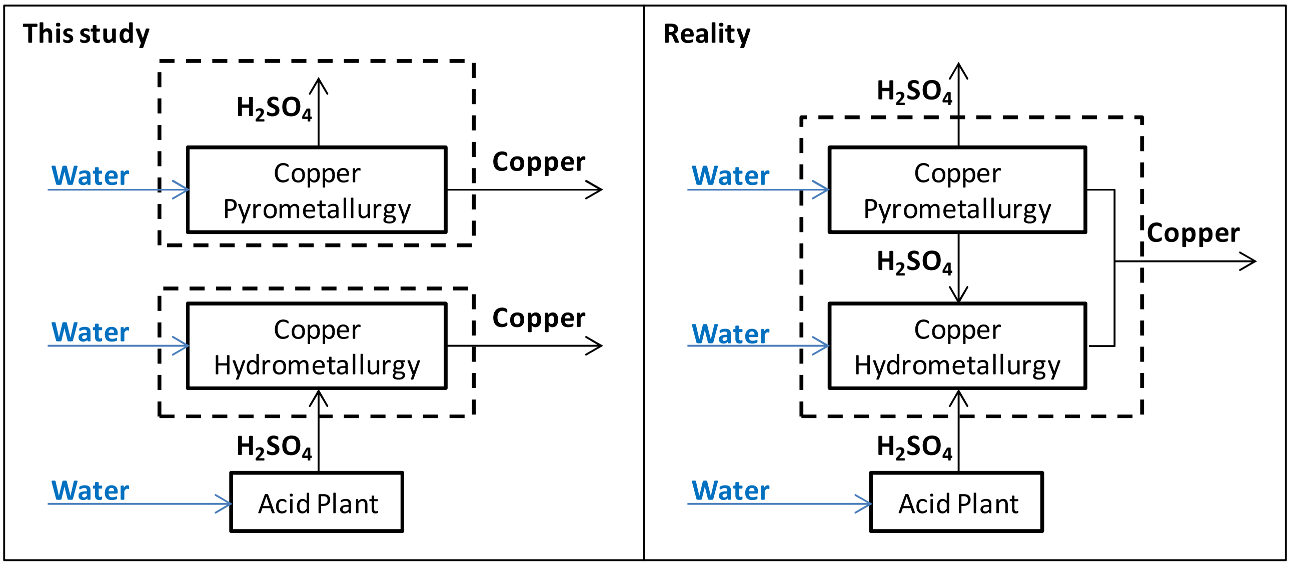


Figure 10: Flows of sulphuric acid for copper production processes assumed by this study (left) compared with reality (right).

Norgate and Lovel (2006) also provided a generic equation that can be used to produce a rough first estimate of the embodied water of refined metals based on ore grades. This equation is shown below:

$$W=167.7\ G^{-0.9039}$$

where: W = embodied water of (m^3/t refined metal), G = grade of ore used to produce metal (%)

The equation has been plotted alongside the results of this study for comparison in Figure 11. At higher ore grades (>2 % metal) the equation produces similar results for copper and nickel pyrometallurgy, but underestimates the hydrometallurgical processes. At lower copper ore grades (< 0.5% Cu) the equation provides reasonable estimates for copper hydrometallurgy and overestimates the pyrometallurgical processes. For gold production the equation underestimates the embodied water, particularly for non-refractory ores. Despite these differences the equation provides a reasonable order of magnitude estimate of the embodied water of refined metals.

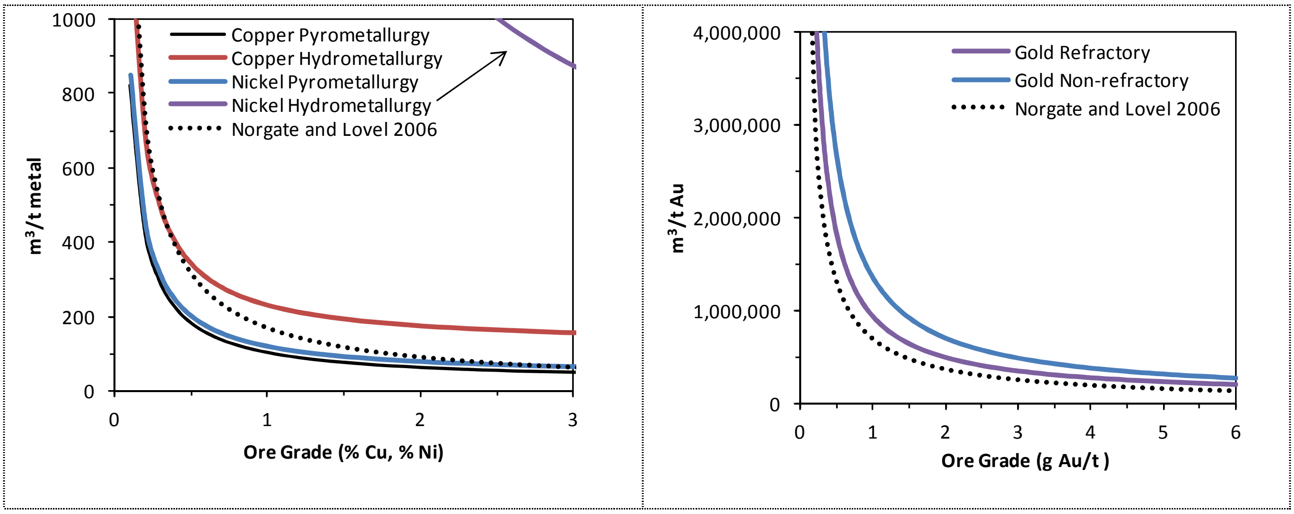


Figure 11: Comparison with Norgate and Lovel's (2006) embodied water equation.

4.2 Reported water data

The overall water balance for mining operations is in large part quite variable depending on the local climate and hydrology of the area. A site operating a positive water balance will accumulate water on site as water inputs through precipitation and groundwater infiltration will exceed water losses through evaporation and seepage. For most sites this accumulation will occur within tailings storage facilities or process water ponds. Due to this sites operating a positive water balance need to do controlled discharges to the environment, to prevent water stores exceeding capacity. Many mines in Australia are located in arid regions and experience a negative site water balance. For these mines water has to be actively extracted from the environment to supply water to processing facilities. Often mining operations compete with other industries such as agriculture for the limited water available in these areas.

Figure 12 shows water withdrawals plotted against rainfall for 10 Australian copper operations based on data collated by Northey *et al.* (2013). It is important to note that these sites produce a variety of products ranging from copper concentrates to refined copper and may also produce co-products such as gold, silver and nickel.

As rainfall increases reported withdrawals generally appear to decrease. The reporting methodology used by individual mine sites can be unclear. Despite many reporting according to the GRI protocol they generally appear to discount rainfall from their calculations of water withdrawals. However a notable exception is the reported data for Rosebery that does appear to include rainfall within its calculations, given that Rosebery's water withdrawals per tonne of ore are an order of magnitude different compared to other sites considered. It is unclear whether the site highlighted in red Figure 12 includes rainfall in their estimates of water withdrawals, and so this site and Rosebery have not been included during the production of the trend line. There appears to be a moderate correlation between rainfall and water withdrawals. As rainfall increases, reported water withdrawals appear to decrease. This could be due to a variety of factors such as rainfall on tailings dams being included in "reused" water that returns to concentrators, or possibly lower rates of evapo-transpiration in high rainfall climates resulting in reduced water losses from TSFs. Due to the small sample size it is difficult to draw any strong conclusions from the data apart from it being evident that the recycling rates also shown in Figure 12 are a large determining factor on the overall water withdrawals of individual sites.

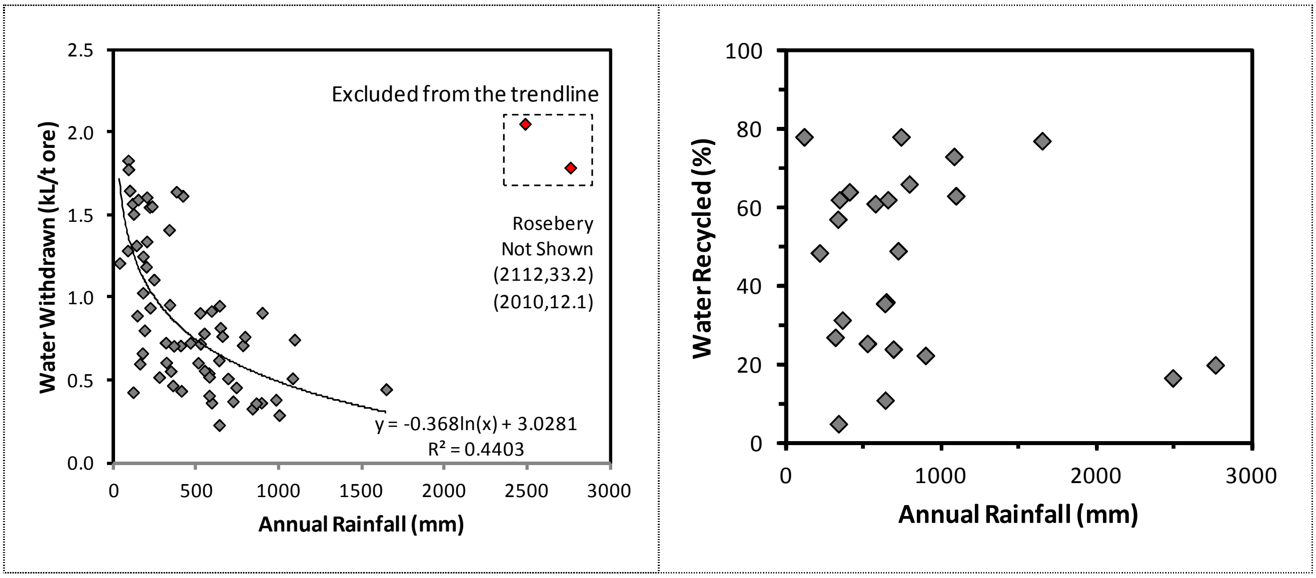


Figure 12: Annually reported water withdrawals and recycling rates versus rainfall for 10 Australian mine sites (Northey et al., 2013, with some updates; BOM, 2013).

Figure 13 shows water recycling data plotted against withdrawals and ore throughput for 31 international copper mines. As expected there is a moderate correlation (R^2=0.409) between water recycling and water withdrawals. Increases in the rate of recycling will result in lowered external water demands and vice-versa. The reported data also shows a weak correlation between water recycling rates and milled ore throughput. However this data alone is insufficient to provide a clear causal link between these two variables, as the high recycling rates of mines with high throughputs is likely due to them being located in arid desert regions in Chile, rather than high throughputs leading to efficiencies of scale for water reuse/recycling.

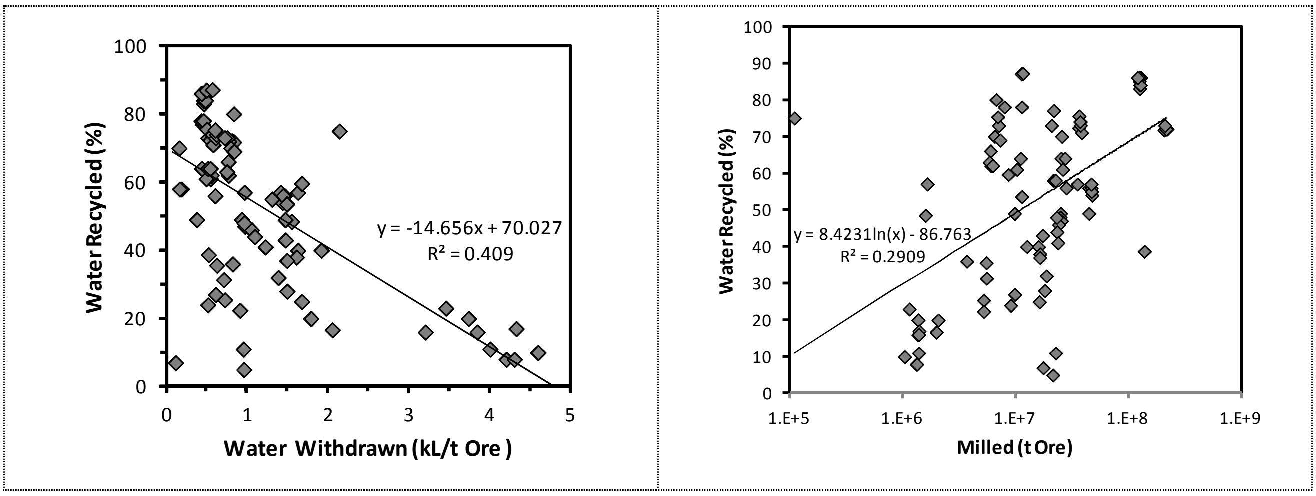


Figure 13: Annually reported water recycling plotted against water withdrawals and ore throughput for 31 global copper mines (Northey et al., 2013, with some updates).

Northey et al. (2013) found that for 31 copper mines, representing about 40% of annual production, the weighted average water withdrawal was 74 kL/t Cu when co-products were allocated based on their economic value. Mudd (2008) presented reported water data for a range of commodities including copper, gold and nickel. Mines producing copper only reported an average water consumption of 172 kL/t Cu or 1.27 kL/t ore. For mines producing gold only the average was 716 kL/t Au or 1.96 kL/t ore, while nickel sulfide mines averaged 107 kL/t Ni or 1.01 kL/t ore. The boundaries of this reported data are in many cases unclear and the sites produce products ranging from metal concentrates to refined metal. These differences limit their direct comparison with the study, however, they do indicate that our results are of the correct order of magnitude.

4.3 Mine site water management

Legislative requirements and societal pressure have required mines to adopt water management systems (Brown, 2003). These provide a framework for the companies to assess their own performance and make changes to how they interact with and place demand on local water assets. Water quality and water quantity are inextricably linked, as quality is a constraining factor on the uses of a water source. An effective water management strategy that provides strong outcomes will actively manage both water quality and quantity. Effective mine site water management revolves around two key concepts (Bagajewicz, 2000):

1. Processes should operate using the highest solids density possible
2. Processes should be supplied with the poorest quality water that does not impact on performance.

Gunson et al. (2012) described various strategies to reduce the water consumption at mine sites. Generally the tailings thickener underflow solids density is a key factor in determining the amount of water available to be reclaimed for the process. Increasing underflow solids densities increases the overflow available to be reclaimed and reused within the process. Water can also be recovered from tailings filtration processes when these are present on-site.

Despite large differences and variability in individual processes for different metals, the largest sink of water at most mine sites is the TSF. Water is lost from the TSF through the processes of entrainment, evaporation and seepage. The rate of water loss through these processes is a function of a diverse range of factors that include the construction of the dam, ambient temperature, humidity, precipitation rates, groundwater interactions and the tailings solids particle size. Developing a full water model that accounts for all physical processes occurring in a tailings dam is a difficult task (Wels and Robertson, 2003).

To provide estimates of water recovery from TSFs, an accurate estimation of the size of the wetted is essential to be able to determine water losses (Wels and Robertson, 2003). Seepage and evaporation are maximised within the wetted area of tailings dams, although seepage losses may only be a significant factor when the bedrock or underlying surface has a high effective permeability. Declines in seepage and evaporation happen over a very short timescale for coarse tailings particles (within days), and much more gradually (within weeks) for fine grained tailings particles (RGC, 2003 in Wels and Robertson, 2003). Wels and Robertson (2003) provide a case study for a tailings dam located in an arid region. They determined that entrainment accounts for around 52% of water discharged with the slurry, with evaporation and rewetting losses accounting for 15% and 11% respectively. The remainder of the water (~0.156 m^3/t Ore milled) is recycled back to the process via a recycle pond. There are several opportunities available to reduce the water lost from TSF facilities. Significant water savings can be made by reducing the wet/open water area by managing the placement of tailings (Chambers et al., 2003). Water entrainment in tailings can be reduced by lowering clays generated (Mwale et al., 2005; de Kretser et al., 2009)

Johnson (2003) described the main benefits of recycling water at mine sites as being as being: The requirements for external water withdrawals are reduced, and the quantity of water requiring discharge into the surrounding environment is also lowered greatly. If there are beneficial chemical species present in discharge water than recycling can keep them in the processing system. Likewise if deleterious (or pollutants) are present in discharge water than recycling them to the processing plant where they can be separated and stored in a site facility (eg. the TSF) can be beneficial. The quality of recycled water should be matched to meet the requirements of individual processes. There is a time lag between water input to the tailings pond and recycling of that water to the plant. Maximisation of recycling is achieved when removal of water from product streams is maximised and all of this water is returned to the process. There are cases where the TSF water is not recovered because this can cause accumulation of deleterious chemical species within the processing system. The quality of external water withdrawals can range from very high to very poor quality hard water. Water quality and recycling can have a significant impact on recovery rates in concentration plants (Schumann et al 2003). Johnson (2003) provides examples of recycling altering plant performance, both beneficially and negatively.

There is a relationship between the particle size that ores are grinded to, energy consumption and water consumption (Mwale et al., 2005). Decreases in ore grind size have several adverse effects on the availability of water for recycling. Lower grind sizes result in a decrease in thickener performance due to decreases in the settling velocity of particles. Unless detention times are increased (through building larger thickeners) the water available to be recycled from tailings and concentrate thickeners will decrease as a result. Lower particle sizes also reduce the performance of filtration systems as they are more susceptible to blocking the pores of the filter media. Reduced particle sizes also increase entrainment in filtered tailings and concentrates due to higher void ratios. As concentrate and tailings particle sizes decrease the amount of water able to be recovered through dewatering processes decrease. Pressure filtration at some sites is being employed to separate water to counteract trends such as the increasing fineness of concentrates (Johnson, 2003).

Dewatering aspects of mine water management plans (WMP) require consideration of both groundwater and surface water interactions to be understood (Hair, 2003). Reasons for mine-site dewatering can be grouped into geotechnical and mining considerations (Hair, 2003). Mining considerations include the need to reduce acid drainage and limit corrosion to mine infrastructure and equipment, reducing drilling, blasting and hauling costs, workplace safety and mine-site trafficability. Geotechnical considerations are also very important to ensure the structural integrity of the mine as reducing water inflows will reduce shear strengths, hydrostatic pressures, the weight of porous material and prevent the development of seepage forces.

Management of catchments and water sources is essential to ensure the long term viability of the industry. Groundwater demand in Western Australia increased dramatically between 1986 and 2000 (Johnson and Commander, 2003). Groundwater usage by the WA gold industry is in excess of 80 GL/year. Groundwater abstractions from individual borefields decrease over the life of a mine as other water sources become available such as dewatering bores and the recovery of tails and open pit lake water. Nickel laterite ores require little dewatering as they are generally above the groundwater table.

There is a large energy cost associated with supplying water to various minesite processes. Gunson et al. (2010) assessed various strategies to reduce the energy requirements associated with mine water networks. The strategies considered included tailings paste thickening to reduce the energy costs associated with the pumping of reclaimed TSF water back to the mine site. Using water for individual processes that is the minimal quality possible without affecting performance is an important consideration, as it significantly reduces the energy consumed at water treatment facilities. Ihle (2013) suggests that the greenhouse gas emissions associated with supplying water to mine sites through long distance pipelines are significant and warrant inclusion within mining LCA studies.

Paterson (2012) describes the adoption of high density tailings slurries for long pipelines, a practice that has very high energy requirements. The adoption of high density slurry pipelines is perceived as difficult by the industry, however where TSFs are located far from processing facilities it can be appropriate. Several examples of this technology exist. For instance the Esperanza project in Chile is thickening to slurries densities of 60-70%, albeit a very high pressure head is required to prevent settling occurring within the pipeline.

Napier-Munn and Morrison (2003) discussed the potential for dry separation processes to significantly reduce the water requirements of mineral processing. They concluded that more research is required to determine how to replace wet processes with dry. The advantages of dry processing would be: no tailings effluent, no pumping costs or aqueous chemicals, no dewatering processes, smaller plant footprint and capital costs, reduced bulk freight costs from ~0% product moisture. Major disadvantages of dry separation are due to the production of dust and their inherently low throughput.

Finally, mining requires large amounts of water over long timeframes and the water management issues post closure are different to those during the operational period (Johnson, 2003; Hair, 2003). Post closure impacts include: aquifer recovery, re-establishment of groundwater equilibrium, the development of pit lakes in mine voids, and the isolation of waste products and contaminants from seepage. Following mining the evaporation of open pit lakes effectively becomes a permanent groundwater sink, depressing groundwater flows in the surrounding area. The changing impacts over the full life cycle of a mine should be considered during the environmental approval processes prior to mining, as they can potentially have adverse consequences on the longer term water security for an area.

"Water Neutrality" is a relatively recent concept that has emerged over the past few years. Muñoz and Guzmán (2012) considered water neutrality as it applies to the mining industry using the same definition as the United Nations Environment Programme; that is "water neutrality refers to the reduction and offsetting of the negative externalities resulting from the water footprint of a unit of analysis, such as a product, service, individual consumer, business or other organisation." Although there are issues with quantifying what water neutrality actually means in practice, on the surface it is an intuitively appealing concept that provides a general direction or aspirational target for the mining and mineral processing industries to move towards. The TSF is the area with the highest potential for reducing make-up water required, with high rate tailings thickeners and tailings classification being the best available technology to achieve this (Muñoz and Guzmán, 2012). Areas with less potential for minimising site-wide consumption are camps, roads and cleaning facilities, as they are typically small consumers of water. Achieving water neutrality for the mining industry is not entirely possible because additional water will always be required to make up for losses from the processes. Therefore water neutrality for the mining industry can only be achieved through the use of offsets that result in a more sustainable use of water in other industries. These offsets need to demonstrate some capability to compensate for the effects of negative environmental, social and economic externalities.

4.4 Legacy water quality issues

The impacts to water quality are important consideration at all stages of mining and processing. Local conditions are large determinants of the quality of water available for consumption by the minerals industry. Mismanagement of water resources can lead to degradation of regional water quality through time. Companies should use the lowest quality water available for a given purpose, so that high quality resources are maintained for more critical needs.

Many mines are heavily reliant on groundwater sources and in Australia there is large variability in groundwater quality between regions (Johnson 2003). Mining companies have the potential to degrade water quality in surrounding catchments if they employ poor wastewater management systems. The need to manage discharges is well recognised by the industry in Australia (Nevad and Jansz, 2006) and significant investments are made to manage pollution appropriately (Driussi and Jansz, 2006). Risks posed by acid mine drainage are considerable for not only their immediate impacts but also legacy impacts once mines close (Akcil and Koldas, 2006).

The impacts of poor mine water management on water quality, caused by inadequate regulation and practices, can lead to a variety of consequences. These have the potential to cause conflicts between different groups of water consumers. Kemp et al. (2010) argue that the right of "access to water", although not currently enshrined in international law, is an important consideration for the mining industry as it can result in conflict between communities and industries. There is a division between engineering and social science backgrounds in the mining industry. This lack of integration hinders sustainable water management because it reduces the ability for companies to incorporate the views of all stakeholders into their decision making framework (Kemp, 2010). South Africa is a prominent example of this where more than a century of poor water management has led to significant legacy AMD problems, drawdown in aquifers and social unrest (Adler et al., 2007).

4.5 Case studies of copper concentrators

During March 2012 the authors visited an Australian mine and copper concentrator that receives more than 2,000mm of rainfall annually. The biggest focus from an environmental point of view has been on managing water quality issues and also assessing energy and greenhouse gas emissions. Because the site operates a positive water balance there has been little incentive to improve water use efficiency and recycle water on-site. A broad objective of the parent company's sustainability policy is to reduce water consumption across all sites and so there is an aim to better understand water use within their processes. A process flow diagram for the site's concentrator is shown in Figure 14.

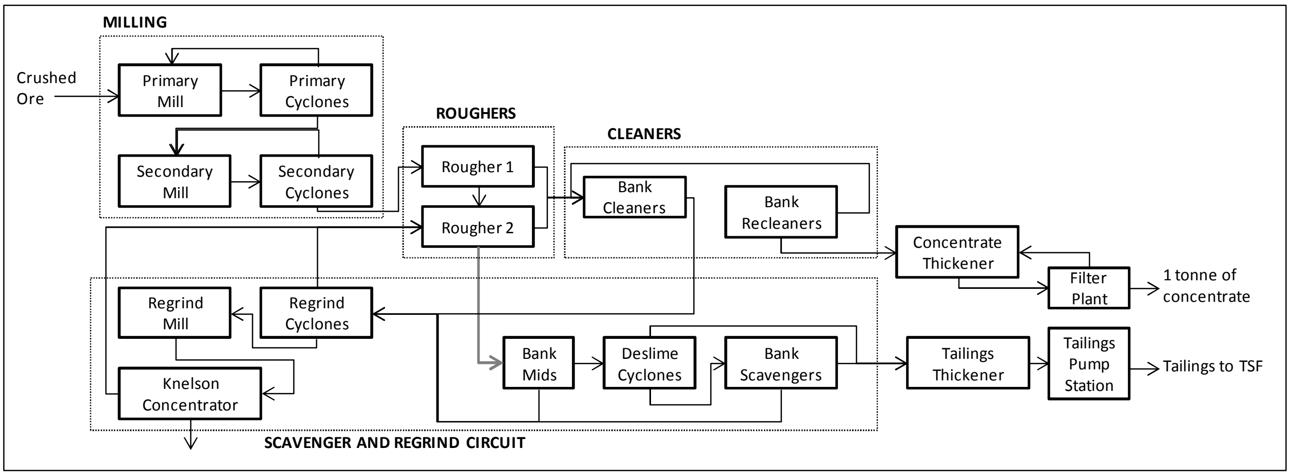


Figure 14: Process flow diagram for an actual Australian copper concentrator

The inputs and outputs of water were estimated for each of the major process areas within the concentrator (Table 11). The majority of water inputs occur during the milling stage where ore of about 2.4% moisture content is converted to slurry containing 38.5% solids (w.w). Within the flotation circuit, water additions are highest in the roughing and cleaning flotation cells, with only a minor addition of ~4.8 m3/t concentrate occurring in the scavenger and regrind circuits. Concentrates are thickened to 50-57% solids and then filtered to a moisture content of 7.5-8% for export to an international copper smelter. Tailings slurry containing ~43% solids are pumped and discharged to the site's TSF.

Table 11: Estimated water inputs and outputs per tonne of concentrate.

Water Inputs		
Ore Entrainment	0.6	m^3/t Conc.
Milling	35.6	m^3/t Conc.
Roughers	12.1	m^3/t Conc.
Cleaners	9.8	m^3/t Conc.
Scavenger & Regrind circuit	1.9	m^3/t Conc.
Tailings Pump Station	4.8	m^3/t Conc.
Total	**64.8**	m^3/t Conc.
Water Outputs		
Concentrate Entrainment	0.1	m^3/t Conc.
Concentrate Thickener (Recycled)	2.7	m^3/t Conc.
Tailings Entrainment	26.5	m^3/t Conc.
Tailings Thickener (Recycled to Mill)	35.5	m^3/t Conc.
Total	**64.8**	m^3/t Conc.
Makeup water required	26.6	m^3/t Conc.
- per tonne of ore milled	1.2	m^3/t Ore
- Per tonne of Cu in concentrate	102.2	m^3/t Cu

The concentrator's water usage has been highly variable through time as shown in Figure 15. The exact reasons for this are poorly understood by site staff but are likely to be a combination of a number of factors. These factors could include errors in measurement of water flowrates, variability in thickener and pump performances, changes in water levels in the tank that gravity feeds the concentrator, maintenance taking equipment offline and rainfall occurring on the thickeners.

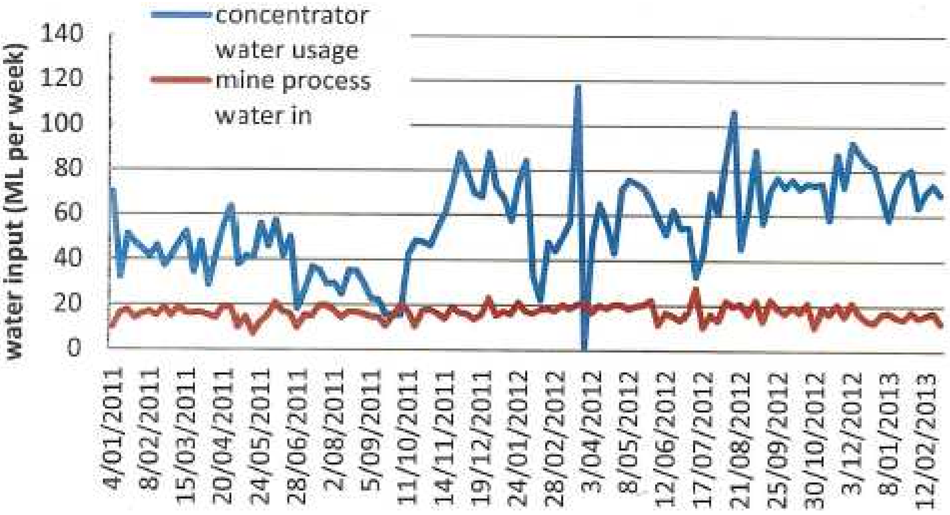


Figure 15: Variability in water inputs through time for an Australian copper concentrator

Case studies exist in the literature for other copper concentrators. Olivares et al. (2012) used the methodology presented by Hoekstra *et al.* (2009) to estimate the direct blue and grey water footprint of copper concentrate produced by Codelco's El Teniente division in central Chile. The average blue water footprint of the site was found to be 29.5 m^3/t concentrate when the TSF was excluded from the system boundary, or only 8.7 m^3/t concentrate when the TSF was included. The grey water footprint of the site (the water required to dilute discharges to below certain standards) was found to be 31.5 m^3/t concentrate.

The U.S. Geological Survey released a report describing the typical water requirements for the conventional flotation of copper ores (Bleiwas, 2012). Water requirements for the process were shown to range from 1.5 to 3.5 m^3/t ore, with makeup water constituting anywhere from 10% to 100% of this. The study found that for a 50,000 t ore/day copper concentrator operating in a semi-arid environment, processing an ore grade of 0.5% Cu and discharging tails at 50% solids, raw water requirements are approximately 38 m^3/t concentrate.

Conclusions

The general growth of the mining and mineral processing industries coupled with declining ore grades will continue to place pressure on water resources. Integrated management approaches to water use within the industry is essential to maintain the long-term sustainability of these water resources.

The water consumption at mine-sites is highly variable due to a range of factors including climate, process configuration and the competition between water users in an area. The key variables that influence direct water consumption for metal production processes are the ore grade being processed, the tailings solids density, and the rate of reuse/recycling within concentration facilities. In all cases the overall site water balance is highly dependent on the climatic conditions. These affect not only water availability and inflows into the site, but also the ability to reuse and recycle water within process facilities.

Prior studies were found to underestimate the direct and indirect water footprints of the metal production processes considered. This is particularly the case for the copper and nickel hydrometallurgy processes, where acid consumption was found to be a significant contributor to the indirect and total embodied water. The results for these two processes are therefore highly sensitive to the assumed embodied water of the sulphuric acid.

Reported data and the case study provided for a copper concentrator highlight the significant variability in direct water consumption, both within and between mine sites. Improved assessment of the potential for individual sites to reduce their water consumption is essential for reducing the overall impact of the industry.

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Appendix A - Indirect Water Inventory Data

Table 12: Underground copper mining material and energy consumption

Underground Copper Mine			
Diesel	2.8	kg/t Ore	Norgate and Haque, 2010
Explosives	0.4	kg/t Ore	Norgate and Haque, 2010
Electricity	20.4	kWh/t Ore	Norgate and Haque, 2010

Table 13: Copper sulfide pyrometallurgy inventory data

Copper Pyrometallurgy			
Milling			
Electricity	18.5	kWh/t Ore	Norgate and Haque, 2010a
Grinding Media	1.4	kg/t Ore	Norgate and Haque, 2010a
Flotation			
Recovery	93.7	%	
Reagents	1.7	kg/t Ore	Norgate and Haque, 2010a
Tailings			Dependent on ore grade
Smelter, converter and casting			
Concentrate	3.60	t Conc./t Anode	Recovery 98.7%, Concentrate grade 27.3% Cu
Natural Gas	2.5	GJ/t Anode	El Teniente LCA
Fuel Oil	0.7	GJ/t Anode	USGS, 2004. "Mixed fuel" assumed fuel oil.
Silica	312	kg/t Anode	El Teniente, Schlesinger et al. 2010
Refractories	6.3	kg/t Anode	El Teniente, Schlesinger et al. 2010
Oxygen	0.35	t/t Conc.	Mass balance. El Teniente's consumption is 320kg/t Conc.
Slag Generated	2.6	t Slag/t Conc.	Mass balance. Concentrate 27.3% Cu, Anode 97% Cu, Slag 0.5% Cu.
Acid Plant			
Sulphuric Acid	3.90	t H_2SO_4/t Anode	Concentrate grade 35% S, 98% SO2 recovery and 98% H2SO4 concentration
Electricity	1.25	GJ/t H_2SO_4	Outotec Brochure
Vanadium Oxide	230	L/t H_2SO_4	Outotec Brochure
SO2 Emissions	13	kg/t H_2SO_4	98% SO_2 recovery
Refinery			
Cu Anode	1.03	t Anode/t Cu	Anode 97% Cu, Cathode 99.99% Cu.
Electricity	410	kWh/t Cu	Townsville Refinery (Xstrata, 2008-2010 Sustainability Reports)
Natural Gas	0.97	GJ/t Cu	Townsville Refinery (Xstrata, 2008-2010 Sustainability Reports)
Gelatin	72	g/t Cu	Townsville Refinery (Xstrata, 2007-2010 Sustainability Reports)
Acetylene	0.4	kL/t Cu	Townsville Refinery (Xstrata, 2010 Sustainability Reports)
Lime	0.09	kg/t Cu	Townsville Refinery (Xstrata, 2010 Sustainability Reports)
Sulphuric Acid	5.13	kg/t Cu	Townsville Refinery (Xstrata, 2007-2010 Sustainability Reports)
Thiourea	9.6	g/t Cu	Townsville Refinery (Xstrata, 2007-2010 Sustainability Reports)

Table 14: Copper oxide hydrometallurgy inventory data

Copper Hydrometallurgy			
Comminution			
Ore	265.0	t Ore/t Cu	1% Cu grade, 85% Cu leachable (Schlesinger et al. 2010), Leachable Cu Recovered to PLS 80% (Schlesinger et al. 2010)
Grinding Media	0.39	kg/t Ore	Anglo-American Chile Operations (Milled ore: 61% oxide) (AA, 2006-2008 Sustainability Reports)
Agglomeration			
Sulphuric Acid	7.5	kg/t Ore	Mass balance (derived from Schlesinger et al. 2010)
Water	80	L/t Ore	Bouffard 2004
Heap Leaching			
Sulphuric Acid	1.75	t H_2SO_4/t Cu	Mass balance (derived from Schlesinger et al. 2010.)
Solvent Extraction			
Organic Solvent	1.1	kg/t Cu	Tintayya (Xstrata, 2006-2009 Sustainability Reports)
Electrowinning			
Cobalt	800	g Co^{2+}/t Cu	Schlesinger et al. 2010
Electricity	2000	kWh/t Cu	Schlesinger et al. 2010

Table 15: Gold CIP processes inventory data

Gold Inventory				**Reference**
Mining	Diesel Fuel	5.3	kg/t Ore	Norgate and Haque, 2012
	Explosives	1.7	kg/t Ore	Norgate and Haque, 2012
	Waste Rock	3	t/t Ore	Norgate and Haque, 2012
Comminution	Electricity	17.7	kWh/t Ore	Norgate and Haque, 2012
	Steel Balls	0.71	kg/t Ore	Norgate and Haque, 2012
Extraction and Recovery				
Flotation - Refractory Ore	Electricity	3	kWh/t Ore	Norgate and Haque, 2012
	Reagents	154	g/t Ore	Norgate and Haque, 2012
	Concentrate	0.1	t/t Ore	Norgate and Haque, 2012
	Tailings	0.9	t/t Ore	Norgate and Haque, 2012
Pressure Oxidation-Refractory Ore	Electricity	121	kWh/t Conc.	Norgate and Haque, 2012
	Oxygen	0.23	t/t Conc.	Norgate and Haque, 2012
	Fuel	68	MJ/t Conc.	Norgate and Haque, 2012, Natural gas 53 MJ/kg
		1.3	kg/t Conc.	Norgate and Haque, 2012
	Sulphuric Acid	98	g/t Conc.	Norgate and Haque, 2012
Cyanidation	Electricity	1.4	kWh/t Ore	Norgate and Haque, 2012
	Lime	2.2	kg/t Ore	Norgate and Haque, 2012
	Sodium cyanide	0.64	kg/t Ore	Norgate and Haque, 2012
	Tailings (Non-refrac)	1	t/t Ore	Norgate and Haque, 2012
	Tailings (Refrac)	0.1	t/t Ore	Norgate and Haque, 2012
CIP and Stripping	Electricity	5.8	kWh/t Ore	Norgate and Haque, 2012
	Carbon	24	g/t Ore	Norgate and Haque, 2012
	Sodium Hydroxide	0.12	kg/t Ore	Norgate and Haque, 2012

	Hydrochloric Acid	83	g/t Ore	Norgate and Haque, 2012
Electrowinning	Electricity	3100	kWh/t Au	Norgate and Haque, 2012
	Steel wool cathodes	0.25	t/t Au	Norgate and Haque, 2012
	Hydrochloric Acid	0.49	g/t Ore	Norgate and Haque, 2012
Smelting	Natural Gas	0.35	GJ/t Au	Norgate and Haque, 2012
		6.6	kg/t Au	Norgate and Haque, 2012
Miscellaneous	Fuel	0.3	kg/t Ore	Norgate and Haque, 2012
	Electricity	8.6	kWh/t Ore	Norgate and Haque, 2012
Gold Refining				
Chlorination Process	Electricity	480	kWh/t dore	Norgate and Haque, 2012
	Chlorine	0.07	t/t dore	Norgate and Haque, 2012
Electrolytic Process	Electricity	325	kWh/t gold	Norgate and Haque, 2012
Silver Refining				
Electrolytic Process	Electricity	630	kWh/t silver	Norgate and Haque, 2012

Table 16: Nickel sulfide pyrometallurgy inventory data

Nickel Sulfides				
Mine	Diesel	2	kg/t Ore	Norgate and Rankin, 2000
	Explosives	0.4	kg/t Ore	Norgate and Haque 2010a, Copper sulfide value
	Electricity	13	kWh/t Ore	Norgate and Rankin, 2000
Concentrator	Electricity	35	kWh/t Ore	Norgate and Rankin, 2000
	Nickel Concentrate	0.20	t Conc./t Ore	Mass balance, Flotation nickel recovery = 90%
Comminution	Grinding Media	0.8	kg/t Ore	Extractive Metallurgy of Nickel & Cobalt
Flotation	Reagents	175	g/t Ore	
	Dextrin	125	g/t Ore	Kambalda, Handbook of Flotation Reagents, 2007
	R407	20	g/t Ore	Kambalda, Handbook of Flotation Reagents, 2007
	SIPX	15	g/t Ore	Kambalda, Handbook of Flotation Reagents, 2007
	MIBC	15	g/t Ore	Kambalda, Handbook of Flotation Reagents, 2007
Smelter				
Drying	Coal	16	kg/t Conc.	Gansu, China, Jinchuan Group Ltd. (Warner et al. 2007)
Flash Smelter	Recovery	93.7	%	Kalgoorlie, (Warner et al. 2007)
	Natural Gas	11	L/t Conc.	Kalgoorlie, (Warner et al. 2007)
	Oxygen	0.2	t/t Conc.	MEAN Table 18.1 (excluding Harjavalta) , Extractive Metallurgy of Nickel & Cobalt. Kalgoorlie is 5.6 t/t Conc.
	Slag	0.7	t/t Conc.	Includes converter slag
Acid Plant	Sulphuric Acid	2.08	t H_2SO_4/t Conc.	Estimated for Kalgoorlie (Warner et al. 2007)
	Vanadium Oxide	230	L/t H_2SO_4	Outotec Brochure
	Electricity	1.25	GJ/t H_2SO_4	Outotec Brochure
	Water	0.37	kL/t H_2SO_4	Mass balance
Slag Cleaning	Electricity	117	kWh/t Slag	MEAN Nickel Sulfide Smelters (Warner et al. 2007)
	Electrodes	1.5	kg/t Slag	MEAN, Nickel Sulfide Smelters (Warner et al. 2007)
Refinery	Natural Gas	0.37	t/t Ni	Norgate and Rankin, 2000
		17.2	GJ/t Ni	46.5GJ/t NG, DCEE factor
	Electricity	2900	kWh/t Ni	Norgate and Rankin, 2000
Ammonia	Ammonia	2.2	mol NH_3/mol	Saarinen et al. 1998

Leaching			Ni	
		0.64	t/ t Ni	Mass balance, same value as Norgate and Rankin, 2000
Hydrogen Reduction	Hydrogen	0.07	t/t Ni	Norgate and Rankin, 2000

Table 17: Nickel limonite HPAL inventory data

Nickel Laterite - Limonite Ore				
Mining	Diesel	1.4	kg/t Ore	Norgate and Haque, 2010b.
	Electricity	3.7	kWh/t Ore	Norgate and Haque, 2010b.
	Explosives	0.5	kg/t Ore	Open-cut iron ore value, Norgate and Haque, 2010a
Beneficiation	Electricity	9.078	kWh/t Ore	Norgate and Haque, 2010b.
HPAL	Electricity	42.81	kWh/t Ore	Norgate and Haque, 2010b.
	Natural Gas	171	kg/t Ore	Norgate and Haque, 2010b.
	Sulphuric Acid	0.3	t/t Ore	Extractive Metallurgy of Ni & Co, p 131. Norgate et al. Has 0.4 t/t ore
Neutralisation	Limestone	19.4	t/t Ni	Norgate and Haque, 2010b. Ni Laterites
CCD	Flocculants	150	g/t solids residue	Extractive Metallurgy of Ni & Co, Ap D (Usually Polyacrylamide)
Sulfide Precipitation	Hydrogen Sulfide	0.637	t/t Ni	Mass balance, assumed stoichiometric consumption
Refinery	Electricity	3880	kWh/t Ni	Norgate and Haque, 2010b.
Ammonia Leaching	Ammonia	2.4	mol NH_3/mol Ni	Extractive Met of Ni & Co, Fig 23.2
		0.7	t/t Ni	Mass balance, same value as Norgate and Rankin, 2000
Hydrogen Reduction	Hydrogen	0.07	t/t Ni	Norgate and Rankin, 2000

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