

Geochemical assessment of legacy mine sites: assigning value and seeking new opportunities

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Abstract

The closure and abandonment of mining areas is rarely due to complete consumption of a reliable resource but rather due to diminishing financial returns based on metal value, or social, political and environmental restrictions that lead to an uneconomic scenario for a resource unit.

In the assessment of a mining area, there are several potential sources of economic value, including:

- *Previously unidentified resources in the mining area.*
- *Mining of known in situ ore and stockpiled unprocessed ore.*
- *Recovery of value from previously processed waste and previously cited 'mine waste'.*
- *Recovery of value from undeveloped resource, such as the processing of mine water to recover metals and valuable salts.*
- *Recovery of new value from mining facilities such as mine sludge processed to recover ferric oxyhydroxides as a source of iron, pigments and trace metals.*
- *Development of mine water resource for agricultural, industrial, potable use and possibly for use as an energy source.*

In the case of the first two potential sources, unconventional or innovative methods of exploration may be required to identify these resources. Additionally, the development of water as a resource either for metal or salt recovery or as a source of usable water requires hydrogeochemical investigations. The successful development of these resources and value recovery often requires more-efficient metallurgical circuits or new chemical and physical extraction procedures to recover value.

In this paper, the geochemical investigation of abandoned mine lands and particularly mine waste and water are described, along with methods of metal recovery derived from an understanding of the geochemistry.

1 Introduction

The current environment of high metal demand and exhaustion of historically important metal sources is leading mining companies to seek different sources of metals (Bowell, 2004). Metal mining is often characterised by high volumes of waste material in the form of processed tailings, uneconomic low-grade ores (at prevailing metal prices) and material designated as waste rock (Bowell et al., 1999; Price, 2003; Conesa et al., 2006; Sarmiento et al., 2008). This is typical of copper extraction from sulphide ores; for example, in the Aitik Copper Mine in northern Sweden, one of the biggest copper mines in Europe, the mass of the extracted copper per ton of ground ore is only 3 kg, whereas the average ore grade for copper is approximately 0.4% (Hansen et al., 2005a, 2005b; Stjernaman-Forsberg and Ledin, 2003). In copper

recovery from such low-grade sulphide ores, it is customary to use hydrometallurgical processes (leaching, solvent extraction and electrowinning) or beneficiation is performed by froth flotation (including crushing, grinding, flotation, thickening and drying of the concentrate), which is followed by pyrometallurgical treatment of the concentrate and electrolysis (Biswas and Davenport, 1976; Dixon, 2004). More than 90% of the mined material in base metal projects typically is deposited as flotation tailings (Stjernaman-Forsberg and Ledin, 2003); for example, in the El Teniente-Codelco copper mine in Chile, 95% of mined ore is process waste by mass (Hansen et al., 2005b). Mixed in with the tailings are economic elements; for example, copper loss to tailings for most copper mines is on the order of 5–10%, and for some mines it is as much as 25% (Sievers and Meyer, 2003). With pyrometallurgical waste, poor performance in the molten metal can lead to loss of metals (Biswas and Davenport, 1976). For example, at Tsumeb in Namibia, the lead reduction furnace slag contains approximately 10% zinc by mass and is the largest known 'orebody' of germanium, gallium and indium (Dey et al., 2010).

The challenge is to identify such opportunities and find reliable methods that can be used to evaluate and recover the value. The closure and abandonment of mining areas is rarely due to complete consumption of a reliable resource but rather due to diminishing financial returns based on metal values or social, political and environmental restrictions that lead to an uneconomic scenario.

In the assessment of a mining area, there are several potential sources of economic value in an abandoned mine site, including:

- Previously unidentified primary hard rock resources.
- Mining of known in situ ore and stockpiled unprocessed ore.
- Development of the mine water resource for agricultural, industrial or even potable water use (Bowell, 2004). The potential also exists to exploit the mine water for energy storage or generation.
- Recovery of value from an undeveloped resource, such as the processing of mine water to recover metals and valuable salts (Bowell, 2004).
- Recovery of value from previously processed waste and previously cited 'mine waste' (Dey et al., 2010).
- Recovery of new value from mining facilities, such as mine sludge processed to recover ferric oxyhydroxides as a source of iron pigments and trace metals (Hedin, 2003).

The first two sources of potential value are self-evident and will not be described further, although unconventional or innovative methods of exploration may be required to identify these resources. Similarly, coal washing and recovery of fines is also relatively simple (Groppo et al., 2004). Value recovery for the other potential sources of revenue often requires more-efficient metallurgical circuits or new chemical and physical extraction procedures to recover value. It is these sources of value that are discussed further in that they often represent potentially the most significant negative impacts in a legacy mine site.

Dependent on the change in economic conditions, mine waste can become valuable either through the development of a contained commodity for a previous waste element, such as refining copper-lead mine dumps to recover zinc (Jha et al., 2001; Qin et al., 2007; Mukongo et al., 2009; Dey et al., 2010), or through the improved recovery of a commodity, such as recovery of gold from mine tailings at the Ergo operation in South Africa or recovery of coal from coal washing plants (Fisher, 2009).

This paper presents case study evaluations where value is identified in abandoned mine sites. Additionally, modified and new technologies developed in the last decade are discussed for value recovery and geochemical factors that influence the financial assessment of abandoned mine lands.

The paper presents a summary and review of information published elsewhere and as such the methodology of sampling and analysis has been different in each study; the reader is directed to the studies cited in the references for further information.

2 Recovery from mine waste

Reprocessing of tailings residues has become a profitable business for several operations, usually when incorporated into existing facilities to supplement declining primary ore supply or where sufficient infrastructure already exists (Muir et al., 2005; Antonijević et al., 2008).

In general, mine waste can generically be divided into:

- Low-grade ore left in situ.
- Mineralised material that was uneconomic at the time of initial mining and grouped with waste rock or low-grade ore stockpile.
- Tailings containing some residual value.
- Processed waste, such as a heap leach residue or slag stockpile.

As described above, low-grade ore material left in situ is relatively straightforward and treated much like a primary resource. A deviation from this is with In-Situ Recovery process where residual cleaning of a field can generate further revenue through metal recovery. An example of this is the Straz and Hamr uranium mines in the Czech Republic. Over a 40 year period, approximately 4×10^6 tonnes of acid were injected into the ore-bearing sandstones at these mines (Tomas, 1995, 1996). Poor control of the leaching process had a widespread impact on the surrounding region. In 1992 reclamation work was initiated to improve groundwater quality in the Bohemian massif. As a part of this effort, a complex water treatment system was commissioned to recover sulphuric acid through filtration and uranium by ion exchange.

Low-grade ore or historic waste rock can form an important source of metals, for example in the Comstock mining district, as illustrated in Table 1 (Hudson, 1998).

Table 1 Concentration of precious metals in mining waste, Comstock district, Nevada (SRK, 2001)

Mine	Matrix	Au, ppm	Ag, ppm	Cu, wt%
Chollar	Waste rock	3.4	17.9	0.48
Potosi	Waste rock	4.2	16.3	0.36
Savage	Waste rock	2.6	15.7	0.51
Gould and curry	Waste rock	0.7	9.9	0.22
Gould and curry	Tailings	1.2	7.6	0.74
Keystone	In situ rock	4.3	3.3	0.05
Justice	Waste rock	0.9	5.4	0.02
Justice	In situ rock	4.1	8.9	0.01
America flats	Tailings	5.6	16.5	0.04

Where an initial primary ore waste contains a previously unextracted resource, this can be exploited as part of a reclamation program. This can benefit final management of the residual waste. For example, in South Africa on the Witwatersrand, many of the tailings residue piles contain not only residual gold but also uranium and pyrite (Lottering et al., 2008). Typically such ores can contain up to 10% pyrite and 0.5 kg/t U in addition to 1–5 g/t of gold (Fisher, 2009). The challenge with these ores is that much of the resource is encapsulated in inert quartz grains or quartz aggregates, and mechanical agitation and fine grinding is required to liberate the economic minerals for leaching or concentration (Lottering et al., 2008). As an example, gold production from the Ergo tailings reprocessing facility in the East Rand generated

approximately 7,000 kg of gold in 2004 at a cash cost of US\$ 389/ounce. In addition, nearly 200 tonnes of pyrite were recovered for production of sulphuric acid (Fisher, 2009).

The flotation tailings pond of the Bor Copper Mine in southern Serbia poses a great ecological problem not only for the town of Bor but also for the surrounding soils and watercourses (Antonijević et al., 2008). Since the old flotation tailings contain about 0.2–0.5% of copper, they represent potentially an economic feed material. In an evaluation of copper extraction from the tailings, it was found that the average copper and iron recovery was from 60–70% and from 2–3%, respectively. These results indicate that the old flotation tailings pond represents an important source of secondary raw material for the extraction of copper (Antonijević et al., 2008). In addition, waste rock in the region also contains potential for copper extraction, with grades up to 1% copper in the historic rock piles (Antonijević et al., 2008).

Pyrometallurgical processing of base metal ores produces a processed waste termed slag (Biswas and Davenport, 1976). Slag is a partially vitreous by-product of the process of smelting ore, which separates the desired metal fraction from the unwanted fraction. Slag is usually a mixture of metal oxides and silicon dioxide. However, slags can contain metal sulphides and metal atoms in the elemental form. While slags are generally used to remove waste in metal smelting, they can also serve other purposes, such as assisting in the temperature control of the smelting and minimising any re-oxidation of the final liquid metal product before the molten metal is removed from the furnace and used to make solid metal. Such piles can have significant potential for environmental impact (Ettler et al., 2009) and can also have potential as an economic resource (Brocchi and Moura, 2008). For example, the Tsumeb-reduced lead furnace slag pile in northern Namibia represents possibly the largest repository of germanium, gallium and indium currently known (Table 2).

Table 2 Concentration of metals in metallurgical slags (after Dey et al., 2010)

Location	Mass, Tonnes	Copper %	Zinc %	Lead %	Silver ppm	Germanium ppm	Gallium ppm	Indium ppm
Belladune, Canada	9,800,000	0.45	6.5	1.3	110	< 10	< 10	12
Bor, Serbia	14,600,000	0.34	6.4	1.2	1180	12	23	110
Penoles, Mexico	12,600,000	1.3	12.6	0.9	192	< 10	< 10	18
Swansea Valley, UK	7,400,000	1.2	13.4	1.7	54	< 10	< 10	< 10
Tsumeb, Namibia	2,200,000	3.5	9.2	1.9	12	391	173	129

Often a challenge in reprocessing of slag is the presence of highly stable spinel minerals (Mardones, 2012). One possible method to extract metals from slag is by a chlorination route (Brocchi and Moura, 2008; Dey et al., 2010). However, acid leaching, even at high temperature, can be extremely difficult even using aggressive concentrations of hydrochloric acid due to iron interference (Brocchi and Moura, 2008; Mardones, 2012). As an alternative approach, vapourisation or sublimation of metals from an inert matrix can be achieved by reaction with a chloride-based radical; even high-field-strength elements such as tin, niobium and tantalum can be mobilised by tetrachloride at high temperature as a vapour and then condensed to form metal oxide (Dey et al., 2010). The benefit of this approach is that iron, the chief contaminant in many slags or mine waste, is less mobile as a chloride complex and so can be separated from other metals of value by sublimation (Figure 1).

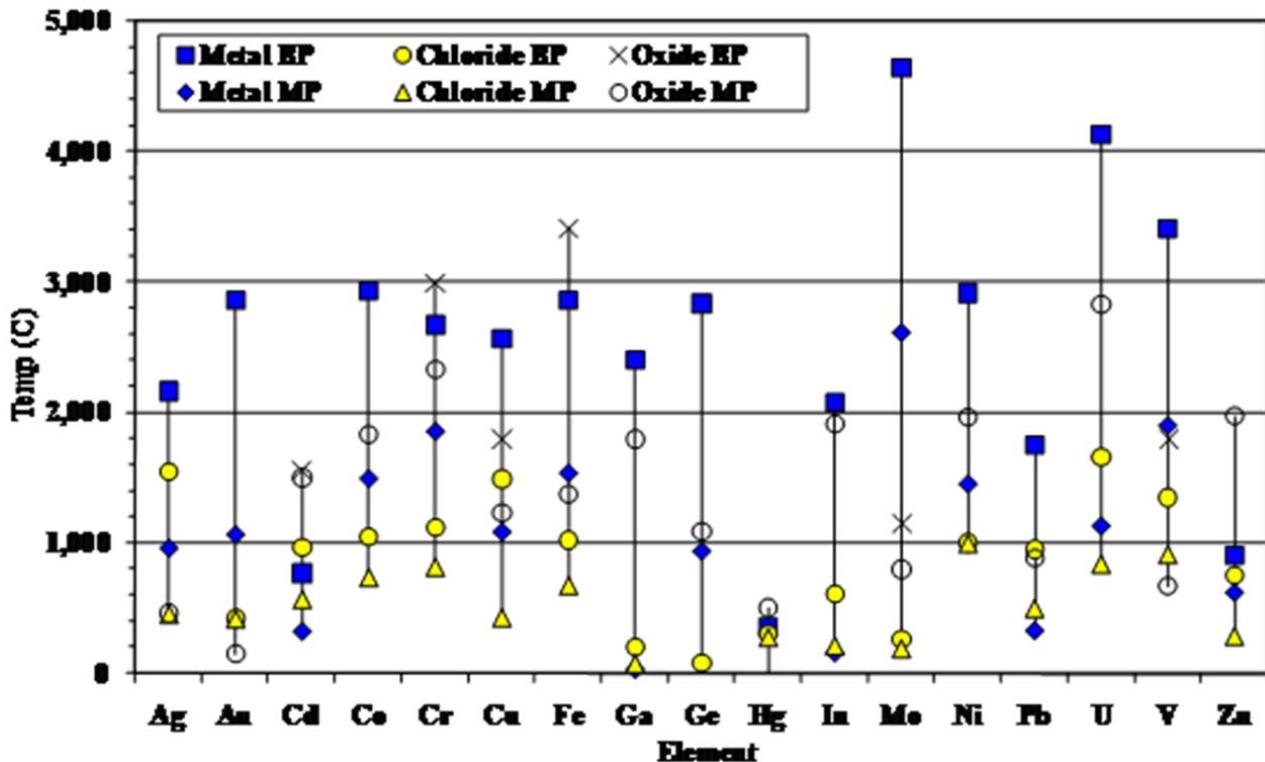


Figure 1 Range of metal, oxide and chloride boiling and melting points for selected metals (Dey et al., 2010)

Chloridisation is the process by which selected commodities are converted to metal chlorides. For this process, a number of reagents have been used as a chloride source: sodium chloride (NaCl), calcium chloride (CaCl₂), ammonium chloride (NH₄Cl), carbon tetrachloride (CCl₄), ferric chloride (FeCl₃), hydrochloric acid (HCl) and chlorine gas (Brocchi and Moura, 2008). These reagents may be utilised through wet or dry processes.

But the chloridisation process is not as simple as converting the metals to chlorides and then boiling them off; the partial pressures of the freshly formed chlorides also have a role to play. Initially the stability of the mineral phase in the slag to hydrogen chloride attack will determine whether the reaction will proceed. Then the free energy of the reaction is an indication of the driving force, although it is noted that these energies are determined under pristine laboratory conditions, and other complicating factors will have a contribution under typical operating conditions.

The application of this process to reprocessing of the Tsumeb slag has been published by Dey et al. (2010) (Figure 2). Metals in this slag are held as sulphides, oxides and silicate minerals, and the control of which metals report to the vapour phase is complex and requires a sound geochemical understanding.

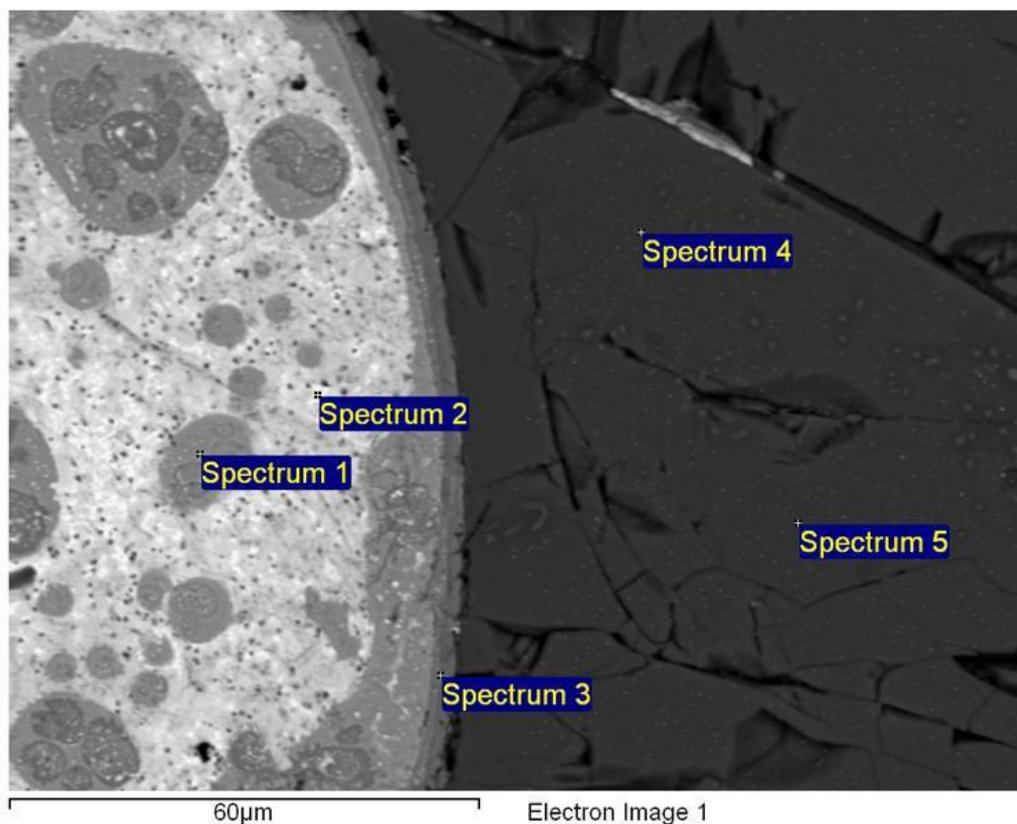


Figure 2 Electron micrograph of Tsumeb slag minerals. Spectrum 1 is copper arsenide (Cu_2As), spectrum 2 is lead oxide and Spectrum 3 is copper oxide. Spectra 4 and 5 are silicates (stoichiometrically near $(\text{Ca}, \text{Zn}, \text{Ge})(\text{Fe}, \text{Mg})(\text{Si}, \text{Al})\text{O}_4$). It can be seen that the refined lead and copper are occurring as inclusions in the silica slag matrix and that the zinc is distributed through this matrix (Dey et al., 2010).

In the development trials, a simple quench system was used to capture the pregnant off-gas. This pregnant solution was then subjected to a typical hydrometallurgical process of precipitation and cementation in order to generate 'saleable' products from the process. However, with the quenching route a proportion of the extraction hydrogen chloride gas is lost to the quench solution and requires neutralisation before the extracted metals can be recovered. Recoveries from the pregnant quench solution were greater than 90% for zinc, 80% for lead and 70% for copper. It is felt that further refining of the recovery methods should improve these recoveries (Figure 3).

The proposed flowsheet for such a process, based on three fluidised bed reactors for the extraction stage, is shown in Figure 4.

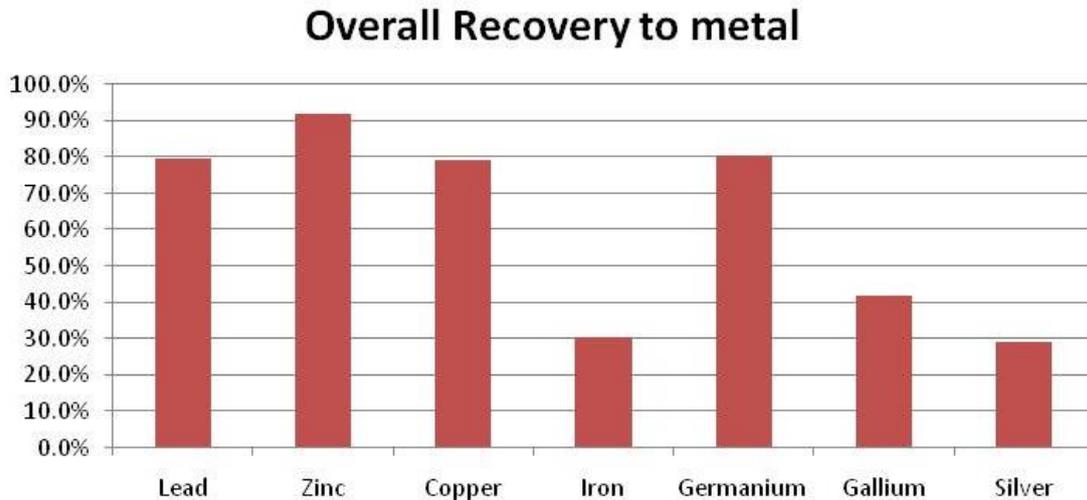


Figure 3 Metal recovery by chloridation and quenching of metal chlorides derived from reaction of chloride gas with Tsumeb slag (Dey et al., 2010)

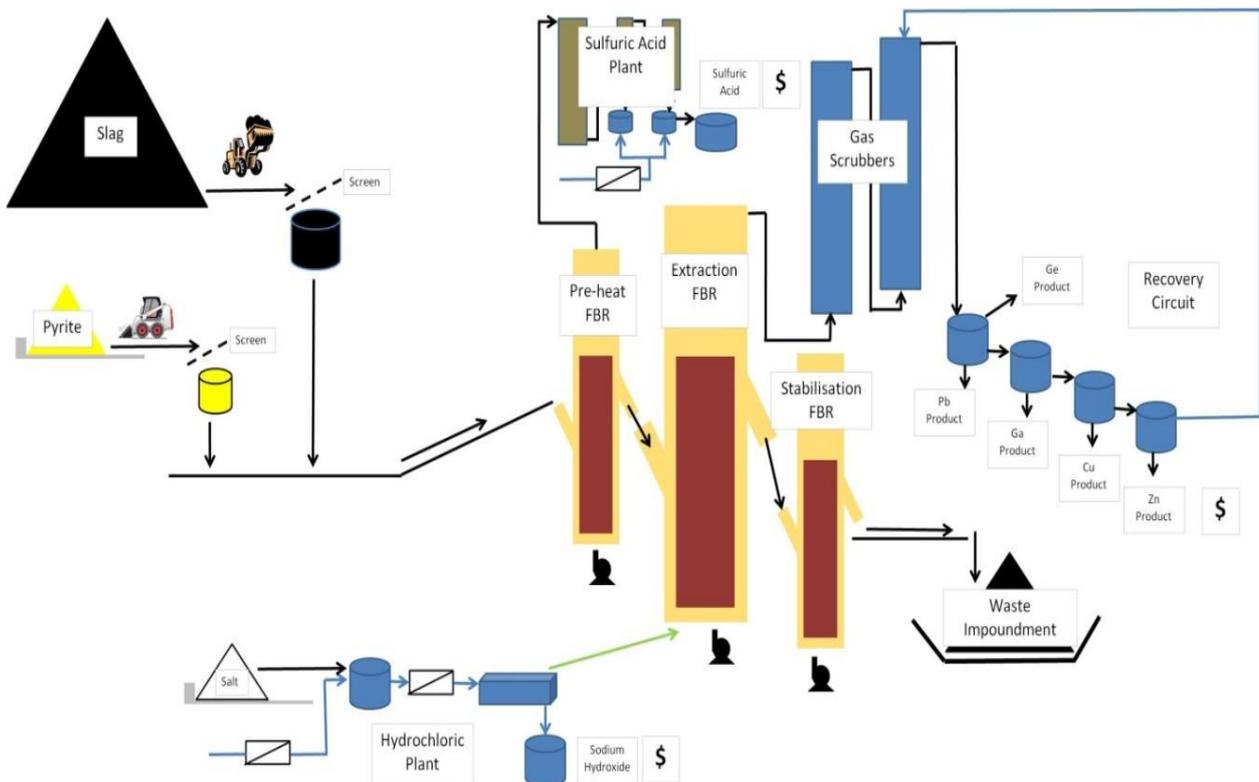


Figure 4 Schematic flowsheet for chloridation extraction of metals from Tsumeb slag (Dey et al., 2010)

3 Recovery of value from mine water

Several commodities can be recovered from mine water, not least clean water (Bowell, 2004). Metal- and sulphate-laden mine water is a common by-product of mining activities (Lottermoser, 2010). This water requires treatment to remove the dissolved metals and sulphate salts in order for it to be reused or safely released to the environment.

Mine water chemistry is highly dependent on many factors, including geology, ore deposit composition and mineralogy, mining methods, climate and many others. Mine waters are typically Ca-Mg-SO₄±Al±Fe±HCO₃, but many other metals can be present, including Mn. Decreasing pH generally results in higher metal concentrations, as depicted in Figure 5 for metal mines.

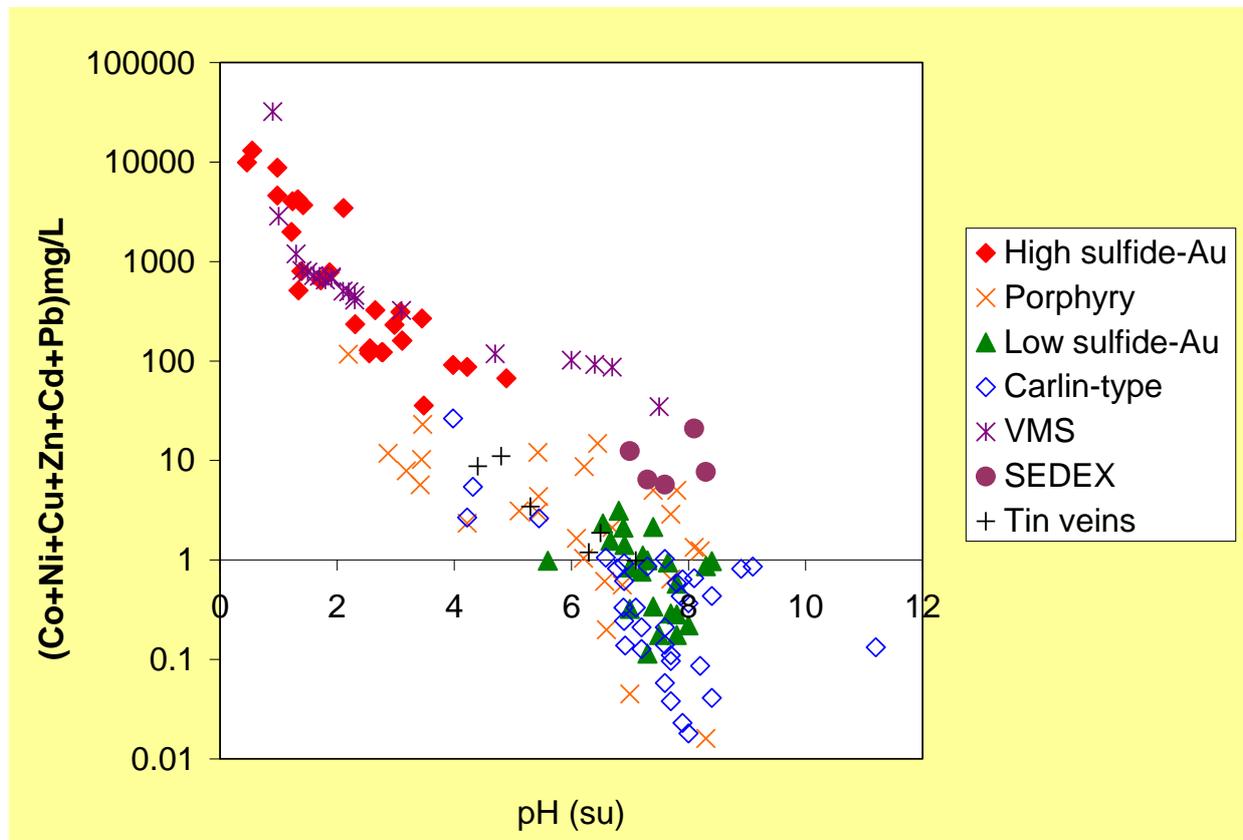


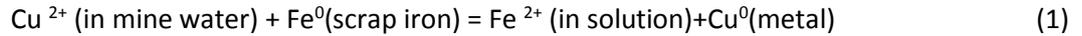
Figure 5 Ficklin plot for different mine waters associated with metallic mineral deposits (Bowell, 2004). VMS = volcanogenic massive sulphide ores. SEDEX = sedimentary exhalative deposits

In many cases these waters contain metals that exceed trigger limits that indicate a potential to impact human health or cause an impact to flora or fauna (Table 3). However, what is clear is that these mine waters have a chemistry similar to process water pregnant leach solutions, for example an acid leach for solvent process derived from heap leaching of a porphyry copper deposit (Table 3).

Table 3 Typical composition of some mine waters and process waters (parameter concentrations in mg/L except pH) (Bowell, 2004))

Parameter	Volcanogenic Massive Sulfide	High Sulfidation Epithermal	Mantos deposit	Porphyry	Copper SXEW (porphyry)
pH	< 1–6	2–4	< 2–6	2–8	< 2
Cu	< 0.1–6,800	< 0.01–5,400	< 0.01–790	< 0.01–2,100	~6,000
Zn	< 0.1 – > 10,000	< 0.1–3,900	< 0.01–4,300	< 0.01–80	< 500
Fe	10 – > 10,000	< 1–28,000	< 1–5,500	< 0.01–1,700	~2,000
Pb	< 1–165	< 0.1–12	< 1–210	< 6	< 100
Ag	< 1–630	< 1–90	< 1–580	< 2	~5

This has long been recognised, and Phoenician miners in the Iberian peninsula, almost 3,000 years ago, would build pits and load these with roasted goethite-rich boulders (pig-iron ore) and then divert copper-rich mine water through the pits to precipitate copper by the reaction:



Such processes were used throughout the following four millennia including in places such as Parys Mountain in North Wales. Copper obtained in this fashion is fairly pure, with the precipitate obtaining a purity of 90% and better. Evaluation on mine waters in southern Spain, Chile and North Wales has identified that mine-impacted waters hosted significant 'ore-grade' concentrations of several metals, with often quite noticeable discolouration of surface water.

Mine water treatment options typically employed to mitigate environmental cleanup tend to precipitate or remove metals as one or two products from which it is generally not economic to recover the value. In lime treatment, lime ($\text{Ca}(\text{OH})_2$) is added to contaminated water. The dissolved metals precipitate into solids, forming a metal-laden sludge. The treated water is separated from the sludge; the sludge is typically stored in large sludge ponds (Biswas and Davenport, 1976). Although very common in the mining industry, the disadvantage of this treatment method is that the sludge can create a waste disposal problem, and possibly an ongoing environmental liability (depending on the type of metals in the sludge). Because the sludge contains the residual metals from the water, it requires careful storage and management to prevent the metals from re-dissolving and re-entering the environment. This method is most suited for waters that contain high levels of iron or aluminium, as these metals are relatively benign and do not usually pose a threat to human health.

However, options do exist for selective removal of certain metals, particularly copper, zinc, silver and uranium, based on existing metallurgical recovery processes. The most commonly applied of these are copper cementation (as described above), electrowinning and biological reduction of sulphate or sulphur to produce a metal sulphide concentrate.

The options for metal recovery from mine drainage include:

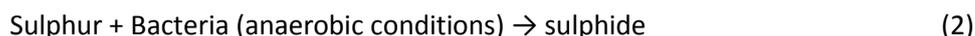
- Direct electrowinning from mine water.
- Direct solvent extraction and electrowinning. In this approach the metal is extracted using conventional hydroxyoxime extractants, stripped using a spent electrolyte solution and then electrowon in conventional cells.
- Solvent impregnation recovery of copper using the Umatilla process developed in Australia followed by stripping with spent electrolyte and electrowinning to produce copper metal.
- Ion exchange recovery using IBC Superlig products. Metal recovery would be by stripping of the Superlig followed by electrowinning; this process is in general the only one feasible for zinc removal in the presence of ferric iron. Zinc could be stripped using weak acid followed by either electrowinning, precipitation of a zinc intermediate product or even shipment of a zinc sulphate strip solution for tertiary refining.
- Hybrid IX/SX. Utilisation of a commercially available chelating ion exchange resin to semi-selectively remove metals and then further process these solutions using conventional SX and electrowinning.
- Metal precipitation using biogenically produced hydrogen sulphide.
- For some mine drainages, the SART process (sulphidisation-acidification-recycle-thickening), developed jointly by SGS Lakefield and Teck Corporation is also an option.
- Ion exchange for metal recovery is most suited for large flows of water that contain low concentrations of dissolved metals. Most of these use cation exchange resins to recover metals.

These technologies are appropriate for low concentrations of dissolved metals and can produce a high-purity saleable product (Jha et al., 2001; Gaikwad et al., 2010).

Solvent extraction coupled with electrowinning is the standard technique for copper cathode production from heap leach operations. Typically, feed solutions for such a plant contain from 0.5–6 g/L Cu, although a low tenor feed of 300 mg/L can achieve an extraction efficiency of 70%. However, the solvent extraction is not without difficulties, and significant clay content in the copper loaded or pregnant solutions can reduce the SX process efficiency. The recovery of zinc through acid pressure leaching and electrolysis is also fairly standard and applied widely throughout the world (Jha et al., 2001; Qin et al., 2007). A benefit for mine water recovery from this process is that SX has been applied to a very wide range of acidic solutions; typical feed solutions have a pH of 1–2.

Removal and recovery of dissolved metals from wastewater are now possible using innovative sulphide precipitation processes that produce a solid metal product that can be recycled into useful products and clean water that can be safely discharged to the environment. Sulphide (in the form of H₂S or NaHS) is added to contaminated water (Buisman et al., 1999). The conditions are adjusted so that individual metals can be selectively precipitated, forming high-grade solid metal-sulphides. The treated water is separated from the metal solids and can be released directly to the environment or reused in the mining process.

In recent years, companies such as Paques in the Netherlands and BioteQ in North America have developed mine water treatment plants that reduce metal loadings by reacting sulphide gas with the mine water so as to form metal sulphides (Howell, 2004). The basic principles behind this process are the production of hydrogen sulphide (H₂S) from elemental sulphur via the action of anaerobic bacteria.



The sulphide is reacted with a solution containing target metals, and the metals are precipitated from solution as the insoluble sulphide. Dependent upon the solution chemistry and the target metals, pH adjustment of the solution may be required. In the case of copper, the metal sulphide will form at a wide pH range, so typically no pH adjustment is required.

Such plants have been installed in several places around the world, including the BioteQ plants at the Caribou and Raglan mines in Canada and the Bisbee mine in Arizona. In these locations the metal sulphide precipitate produced can be processed by conventional smelting operations, and metals that would be otherwise lost are recovered. The operations have proved to have positive cash flows. The economics are adversely affected by the presence of ferric. Ferric will also precipitate as a metal sulphide and so will consume sulphide, increasing costs (both capital and operational). Additional reagents such as ethanol or acetic acid as well as nutrients are required. The bacteria require to be kept within an optimal temperature range typically 20–35°C.

At the Bisbee project in southern Arizona, the BioteQ plant has a design capacity of 10,900 m³/day with a solution composition of Cu = 350 mg/L; Fe³⁺ = 550 mg/L. The project capital costs are of the order of US\$ 2.5 million and quoted operating costs are US\$ 0.20/lb (US\$ 0.44/kg) Cu plus US\$ 0.18/lb (US\$ 0.40/kg) transport/smelting costs (R. Lawrence, pers. comm. 2002). In addition, ferric hydroxide precipitates, or sludges, may also prove to be a commodity that can be reused in cements and construction materials. The plant has been operating since 2004, harnessing BioteQ's BioSulfide[®] process, which uses an anaerobic bioreactor to biologically generate hydrogen sulphide gas (H₂S) on demand. The plant has a treatment capacity of 500 m³/h, and can produce up to 3.8 tonnes per day of sulphide. The general flowsheet for such a process is shown in Figure 6.

An extension of this process is the SART process, developed jointly by SGS Lakefield and Teck Corporation (Cuenca et al., 2012). This process aims to remove the metallurgical interference of leachable copper and zinc and to regenerate cyanide so that it can be recycled to the gold operation. The metals are recovered in a high-grade form that can be sold to a refinery and recycled into useful products. This reduces copper- and zinc-cyanide complexes and the associated environmental impact, maximises resource recovery at the site and creates an additional revenue source.

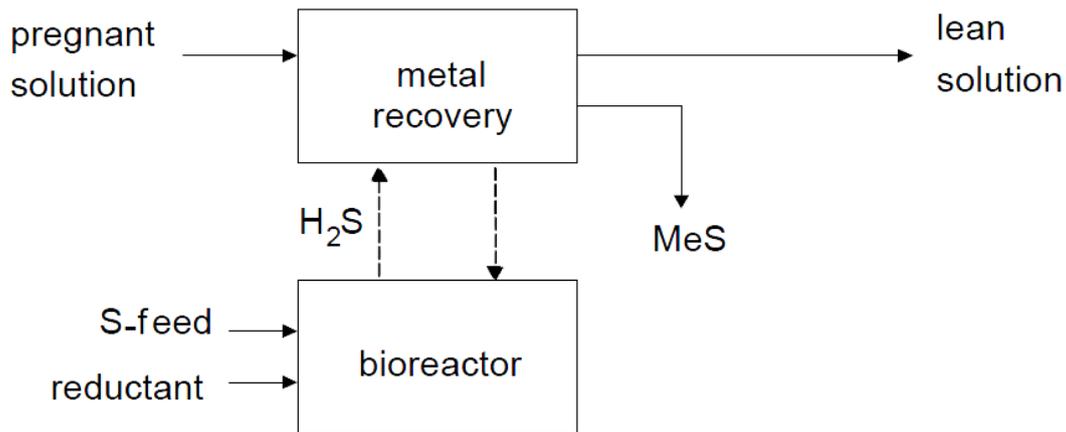


Figure 6 Schematic flowsheet for biological sulphate reduction reactor for precipitation of metals from mine water (Bowell, 2004)

The SART process has been designed to recycle cyanide and recover copper from pregnant solutions containing gold. The name of the process makes reference to the unit operations of the process: sulphidisation, acidification, recycling of precipitates and thickening of precipitates. Basically, the process starts with the precipitation of copper sulphide by adding sulphuric acid and sodium hydrosulphide at pH 5. The reactions of sulphidisation and acidification promote the formation of HCN gas, which is kept in solution. The reactions of the process are as follows:



Acidification of the solution destroys the weak metallic cyanide compounds formed by the reaction of cyanide with copper, zinc, nickel, silver and mercury. The addition of sodium hydrosulphide promotes the precipitation of metallic ions as metallic sulphides such as copper sulphide. The process is quoted as being highly efficient for copper removal (up to 92% of effluent copper) that is precipitated as a copper sulphide similar to chalcocite (Cu_2S). The solids formed during the precipitation process are removed by several stages of thickening, filtration and drying. The reactor feeds a thickener where the solids content is increased from 1% to 15%. The thickener underflow is dosed with sodium hydroxide to pH 12 to avoid the formation of HCN gas and goes to the filtration stage where the precipitate is dewatered.

The thickener overflow and the filtration solution are mixed and the pH maintained at 11. In this case, the lime effects the conversion of HCN to calcium cyanide and the precipitation of calcium sulphate (gypsum), which is separated by further thickening and filtration. The reactions involved in this stage are:



The gypsum thickener operates with recirculation of 85–90% of the underflow. The other fraction of the underflow is sent to filtration in order to recover the gold and cyanide contained in the filtrate. The overflow of the gypsum thickener and the residual filtrate are the final solution of the SART process and are recycled to the leaching process with the remaining content of cyanide. A very important aspect of the process is the scrubbing system to absorb any potential release of HCN gas or H_2S . A solution of sodium hydroxide is added to the scrubber to neutralise acid gases.

One such plant in operation is at the Mastra gold mine in Turkey developed by Bioteq (Bioteq, 2012). This plant is designed to treat $120 \text{ m}^3/\text{hr}$ of process solution containing up to 1,500 parts per million of dissolved copper, of which approximately 98% is recovered as a sulphide precipitate. In addition, almost 90% of the cyanide is recycled in the operation.

Salts are another water component that can have potential value. Lithium is one such salt and it has a variety of uses in the production of aluminium, glass, ceramics, lubricants and chemicals. Lithium batteries accounted for 20% of annual global consumption in 2005 (approximately 135,000 t). South Crofty mine in Cornwall has a high lithium content in mine water. This has led to several studies on the potential to extract lithium from mine water and saline brines in the mine (Beer et al., 1978; Jones and Phillips, 1987a, 1987b).

The composition of brines analysed from South Crofty is shown in Table 4, and this shows lithium concentrations from 11.8 to 198 mg/L. Currently the production costs are such that any operation would break even, but should the mine be reopened, synergies with other forms of value recovery for mineral products could reduce these costs and make lithium extraction viable. As the UK does not have any other domestic source of lithium, and as even within the European Union resources are low, the potential to extract lithium not just at South Crofty but throughout Cornwall does have regional strategic importance.

In addition to lithium, two associated elements, rubidium and caesium, also commonly occur in such brines. These have a more specialised use but if present add considerable value to the brines. Both have been identified in water analyses of South Crofty brines with up to 3 mg/l Rb and 7.7 mg/l Cs (Table 4). Recovery of other components from mine sludge, such as recovery of ferric oxyhydroxides as a source of iron, pigments and trace metals (such as chromium, vanadium and titanium), has been evaluated but as yet no commercial operation exists (Hedin, 2003).

4 Conclusions

Based on this review it is clear that the potential exists to recover value from several different sources in legacy mine sites. The closure and abandonment of mining areas is rarely due to complete consumption of a reliable resource but rather due to diminishing financial returns based on metal values or social, political and environmental restrictions that lead to an uneconomic scenario for a resource unit. However, in the closure and environmental management of a site, the continued recovery of economic components is an essential part of closure, primarily to reduce environmental impact (and thus ongoing management) and possibly also to offset the cost of long-term mitigation and management.

In the assessment of a mining area, there are several potential sources of economic value, including:

- Previously unidentified resources in the mining area.
- Mining of known in situ ore and stockpiled unprocessed ore.
- Recovery of value from previously processed waste and previously cited 'mine waste'.
- Recovery of value from undeveloped resource, such as the processing of mine water to recover metals and valuable salts.
- Recovery of new value from mining facilities, such as mine sludge processed to recover ferric oxyhydroxides as a source of iron, pigments and trace metals.
- Development of mine water resource for agricultural and industrial (or even potable) water and potentially for energy storage or generation.

In the case of the first two potential sources, unconventional or innovative methods of exploration may be required to identify these resources. Additionally, the development of water as a resource either for metal or salt recovery or as a source of usable water requires hydrogeochemical investigations. The successful development of these resources and value recovery often requires more efficient metallurgical circuits or new chemical and physical extraction procedures to recover value.

A caveat exists, however, that even if the 'ore potential' can be proven and the technology will recover economic amounts of metal, there may still be little incentive to 're-mine' many old mining districts. New mining ventures may be held responsible for all past mining legacy as well as any new disturbance, and the mere mention of metal value from these old districts could result in legal action from property owners or bankruptcy trustees who will lay claim to any recovered value.

Table 4 Hydrogeochemistry of South Crofty Mine (Beer et al., 1978)

Sample	34 380- Level 4	35 380- Level 4	36 380- Level	37 380- Level	38 380-Level 4E Hot Dry	39 380-Level 4E Hot Dry	380 DN	SC2
Depth, m	690	690	690	690	690	690	690	690
Flow rate, L/s	3.0	3.0	4.0	3.5	3.5	3.5	4.5	8.7
Temp°C	37.0	37.0	36.8	43.0	41.5	41.5	48	6.75
pH	6.80	6.00	6.30	6.00	7.21	6.50	6.62	7
Na, mg/L	465	–	910	1,875	4,440	4,300	5,540	608
K, mg/L	22.0	–	64.0	54.0	135	180	210	26
Li, mg/L	13.0	–	11.8	54.0	120	125	198	18
Ca, mg/L	365	–	800	1,940	2,325	2,470	3,550	150
Mg, mg/L	36.0	–	108	139	72.8	73.0	–	–
Sr, mg/L	5.40	–	14.3	33.3	41.2	40.0	56	1.2
Rb, mg/L	–	–	–	–	–	–	3	0.2
Cs, mg/L	–	–	–	–	–	–	7.7	0.1
HCO ₃ , mg/L	117	121	57	53	64	68	–	–
SO ₄ , mg/L	81	–	175	102	173	145	150	84
Cl, mg/L	1,500	–	3,300	7,100	11,890	11,500	17,000	2,110
F, mg/L	2.00	–	4.50	2.80	2.70	2.70	5	–
Br, mg/L	5.0	4.3	12.0	23.0	–	43.0	42	–
B, mg/L	1.5	–	1.3	2.6	–	11	12.3	–
SiO ₂ , mg/L	25.7	–	23.5	25.7	3.5	34.2	26	–
Fe, mg/L	2.22	–	22.6	13.0	4.75	4.75	16.5	–
Cu, mg/L	0.003	–	0.005	0.014	0.022	0.023	0.019	–

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