

Acid Mine Drainage in the Bakers Creek Waste Rock Dump, Hercules, Western Tasmania

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ABSTRACT

Mining has occurred at the Hercules Ag, Au, Cu, Pb, Zn VHMS deposit in western Tasmania, almost continuously for the last 100 years. Mining activity has degraded the local environment, through clearing of vegetation, enhanced erosion and the formation of acid mine drainage (AMD). Of significant concern is the Bakers Creek waste rock dump, situated in a waterfall in the upper reaches of Bakers Creek. This steep, unstable sulfidic rock pile cannot be readily removed or engineered, and is an on-going source of AMD. To allow rehabilitation options for the waste rock dump to be devised, the contribution of AMD from the Bakers Creek waste rock dump needed to be ascertained.

Long term and intensive water sampling programs have been implemented to characterise and quantify AMD emanating from the waste rock dump, and to investigate climatic controls on water quality of Bakers Creek. Water samples were collected from ten sites along Bakers Creek, where field analyses of pH, Eh, conductivity and temperature were conducted. Water samples were collected for laboratory analyses of major and trace element concentrations, sulphate, chloride and alkalinity.

The pH of the drainage waters vary from 5.6 (upstream of mining activity) to 3.0 (foot of the waste rock dump). Maximum metal concentrations were measured at the adit, or the base of the waste rock dump. Major contaminants are Fe (43.8ppm), Pb (19.2ppm), Al (8.47ppm) and Sulfate (1440ppm) and Zn (82.2ppm). During base flow, approximately 47.6 T/yr of Zn is discharging from Bakers Creek into Ring River, (98% of which is contributed by the waste rock dump). Other contaminants contributed by the waste rock dump, represented at least 95% of the total contamination in Bakers Creek including Al (2.87 T/yr), Cu (0.77 T/yr), Fe (1.54 T/yr), Pb 1.9 T/yr, and Sulfate, (199T/yr). The maximum mass loads were recorded during storm events (measured at the base of Bakers Creek). The maximum mass load for each contaminant was 158.9 T/yr Zn, 13.9 T/yr Al, 0.33 T/yr Cd, 3.95 T/yr Cu, 65.4 T/yr Fe, 56.15 T/yr Mn, 23.75 Pb T/yr, and 756 T/yr sulphate. These metal mass loads are high enough to consider metal recovery strategies.

To remediate Bakers Creek, a combination of treatment strategies are probably required. Diversion of Bakers Creek upstream of the waste rock dump, combined with the installation of a biosulphide metal recovery plant to treat Zn, have potential to reduce metal and sulphate concentrations to acceptable levels, and to increase pH. A major

advantage would be the metals can be extracted from solution and sold to create revenue to offset costs incurred from installing and operating the plant. Diversion of Bakers Creek combined with a wetland filter system at Williamsford, could be an alternative remediation strategy if a biosulphide treatment is not viable.

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SECTION 1

INTRODUCTION AND BACKGROUND INFORMATION

Chapter 1

INTRODUCTION

1.0 PREAMBLE

Acid Rock Drainage (ARD) occurs when reactive sulfides become oxidised in the presence of water. The low pH (acidic) water that is produced is also capable of mobilising heavy metals that occur in local geological materials. The impact on the receiving environment into which this contaminated water discharges, can result in contaminated soil, groundwater and extreme degradation of aquatic ecosystems and the food chain (Taylor, 1998).

When acid drainage is produced as a result of mining activity, it is referred to as Acid Mine Drainage (AMD). It is one of the most important environmental issues facing the mining industry. Operational costs of properly managing sulfidic mine wastes have been estimated to be around \$60 million per year to the whole Australian mining industry (Harries, 1997). However this does not consider additional costs such as rehabilitating historic mine sites and unexpected rehabilitation costs (i.e. when AMD appears late in mine life or after closure). These potential financial liabilities need to be considered seriously, especially with the current trends towards increased regulatory controls supported by changing government legislation (Watson, 1996).

Western Tasmania has a more than 100 year history of mining polymetallic volcanic-hosted massive sulfide (VHMS) deposits. In the past there was little regard for the impact of mining practices on the environment. As a consequence, after almost 100 years of mining, the Hercules mine on the west coast of Tasmania has severely degraded the local environment. The hill slopes are now barren, Bakers creek (which drains the Hercules catchment), has been severely impacted by acid drainage from the mine site.

Most of the waste rock produced in the earlier part of this century was dumped directly down the western side of Mt Hamilton, into Bakers Creek (Plate 1.1). Unfortunately the waste rock is rich in sulfides, and as a result produces AMD as the oxygenated water from Bakers Creek flows through it. Exactly how much contamination the waste rock dump

contributed to Bakers Creek, was not known. This study was initiated, to investigate the impact of the waste rock dump on the water quality of Bakers Creek.



Plate 1.1 Bakers Creek as it flows through the waste rock dump during a particularly dry period.

1.1 AIMS

The aims of this study are:

- (i) Characterise and quantify the AMD emanating from the Hercules waste rock dump into Bakers Creek, by determining water compositions, mass loadings, bacterial activity, and metal speciation, at various locations along Bakers Creek.
- (ii) Investigate short-term and long term climatic controls on water quality in Bakers Creek.
- (iii) Investigate remediation and rehabilitation options and evaluate which of these are feasible options for Hercules.

1.1.1 Thesis Structure

This thesis is divided into 3 sections. The first provides introductory and background information concerning the Hercules mine, legislative requirements and previous work on the area. The characteristics and origins of AMD are also reviewed. The second section evaluates water chemistry and quality at Bakers Creek, and assess the contributions of various point sources to the overall mass loadings of trace metals in the creek. The third section investigates AMD rehabilitation and remediation options for Hercules, together with a summary of the major conclusions, and recommendations for future work.

1.3 HERCULES MINE

The Hercules mine is located on the west coast on Tasmania, 7km south of Rosebery (Figure 1.1). The deposit occurs on the western slope of Mt Hamilton (1005m), between altitudes of 700 to 850m (Pasminco, 1995, unpub.). Local geographical features include Mt Read (1124m) which is approximately 2km east of the mine, and the abandoned township of Williamsford 1.5km north east of the mine in the Ring River valley.

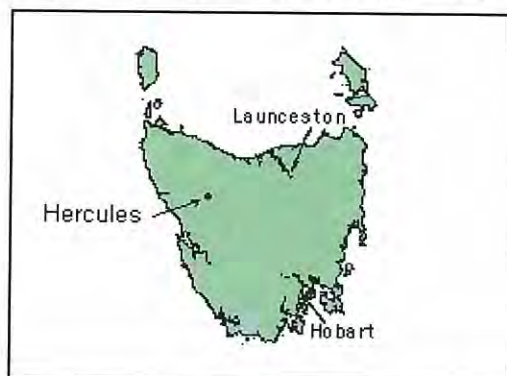


Figure 1.1 Location map of Hercules Mine

1.3.1 Environment

1.3.1.1 Physiography

Mt Hamilton and Mt Read form part of the western ranges. These ranges are parallel to the western coast of Tasmania in the Rosebery area, but swings to the NE and E further to the north. The hill-slopes reach 40 to 60° around the mine offices. The catchment is drained by Bakers Creek, which flows for approximately 2.5km, terminating at the Ring River near Williamsford (Figure 1.2; Plate 1.2). The Ring River flows for a further 10km downstream of Bakers Creek, and discharges into the Pieman River. The Pieman River is the final discharge site for all of the drainage waters on the Pasminco Mining Lease.

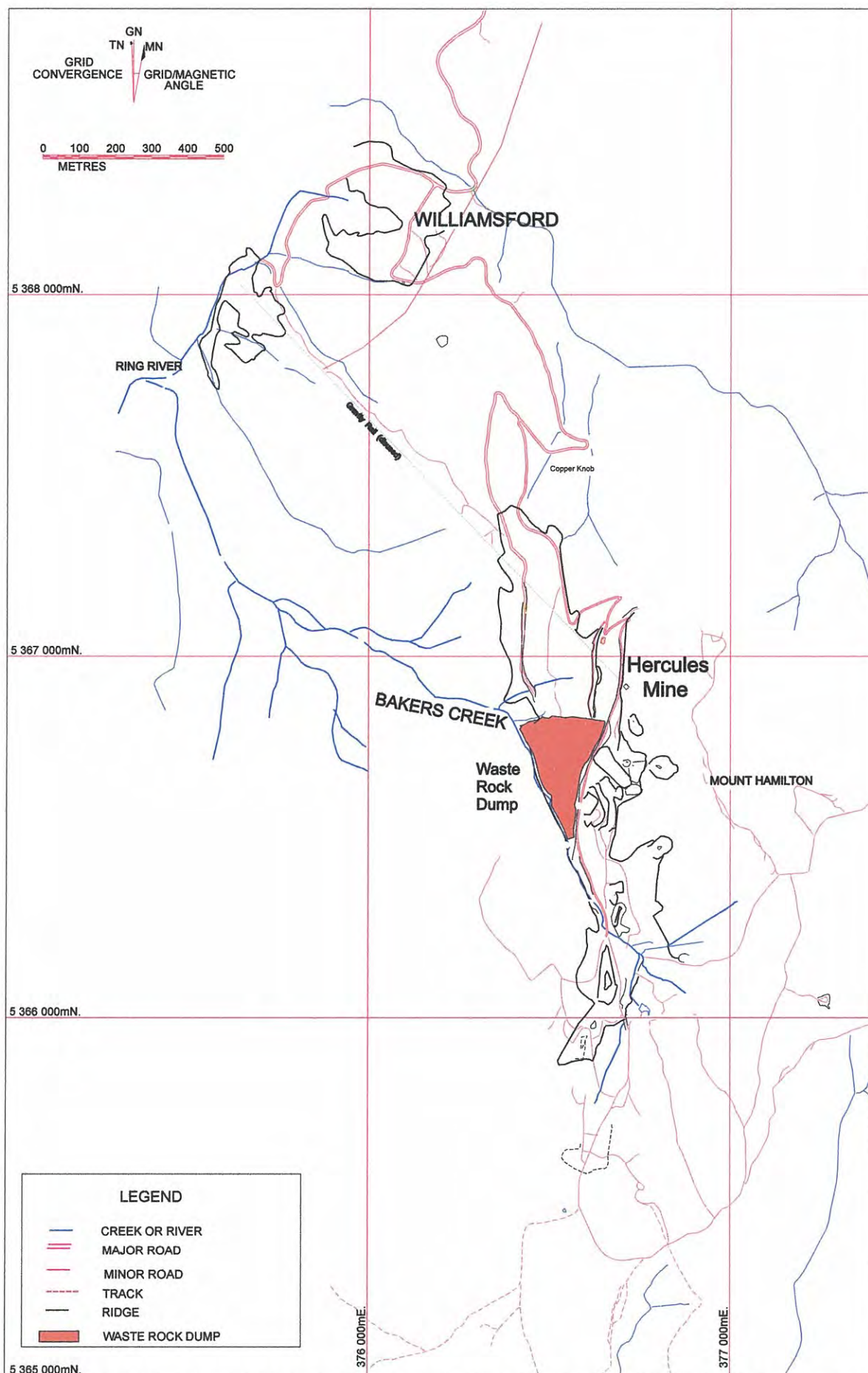


Figure 1.2 Location of Hercules Mine, Bakers Creek, Ring River, Williamsford and the Bakers Creek waste rock dump.

1.3.1.2 Climate

The west coast of Tasmania is subjected to strong winds and heavy rains, as it is located on the edge of the "Roaring Forties" wind belt. The interaction between these moist westerly winds and mountainous topography of the western half of the state, induce orographic rainfall (Pasminco, 1995, unpub.). The Hercules-Mt Read area has extremely high rainfall, averaging approximately 3,000 mm per year, with an average 234 rain days per year (Figures 1.3 and 1.4; BOM, 1998). Rosebery (7km north of Hercules), has an average rainfall of 1,953mm, this difference is due to the strong local topographical influence on precipitation. Although rainfall is the primary means of precipitation, significant snow is possible above the 600m level (Pasminco, 1995, unpub). The driest months are from January to March, although heavy rainfalls can occur in any month.

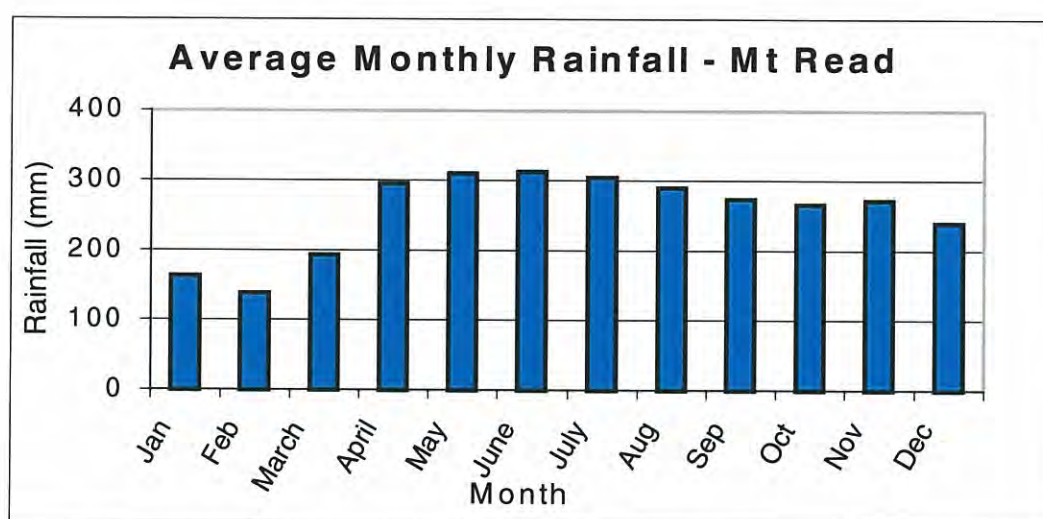


Figure 1.3. Average monthly rainfall at Mt Read from 1901 - 1994 (BOM 1998).

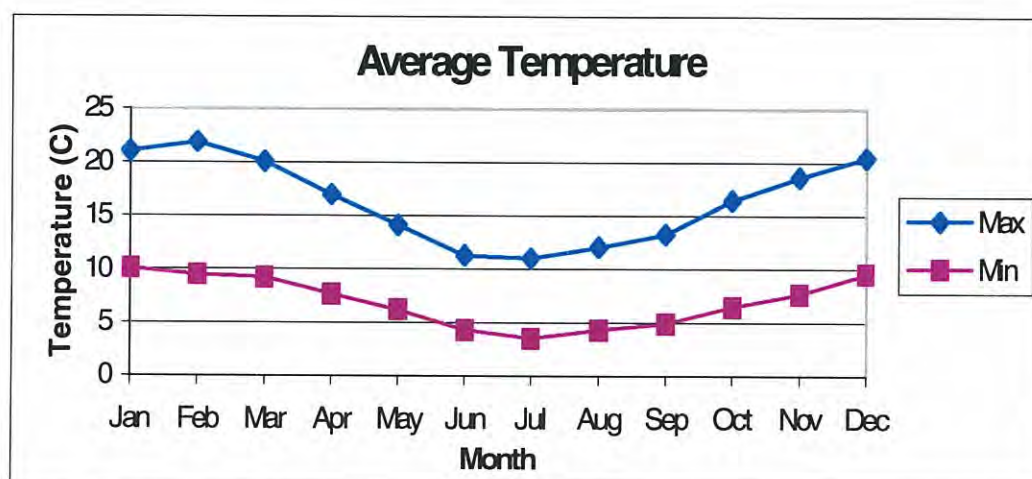


Figure 1.4. Average monthly maximum and minimum temperatures at Rosebery, (BOM 1998).

1.3.1.3 Vegetation

Before mining, Hercules would have supported mature high altitude temperate rainforest to sub-alpine forest. However, the majority of vegetation around the mine has been destroyed through fire (e.g. Plate 1.3), clearing and the impact of AMD. The distribution of plant species that are currently growing at Hercules, is largely determined by altitude and previous fire history (Pasminco, 1995, unbub.). The main vegetation groups include closed scrub, tall temperate rainforest, coniferous forest, and sub alpine-sclerophyllous scrub. Descriptions of the common species and distribution of these groups are listed in Table 1.1.

The Mt Read area contains unusual and interesting vegetation communities which are of biogeographical significance in terms of species composition (especially the ancient stands of Huon pine), biodiversity and structural forms, and has been chosen as a Recommended Area for Protection (RAP; TGSR, 1997).

Vegetation Group	Distribution	Common species
Closed Scrub	Fire induced, often replace rainforest communities.	Blackwood (<i>Acacia melanoxylon</i>), Silver wattle (<i>Acacia dealbata</i>), common wattle (<i>Acacia mucronata</i>), and tea-trees (<i>Leptospermum nitidum</i> and <i>Leptospermum lanigerum</i>). Common rainforest species include Leatherwood (<i>Eucryphia lucida</i>), and Celery-top pine (<i>Phyllocladus aspleniifolius</i>), and occasional dominants such as banksia (<i>Banksia marginata</i> and <i>Smilthton peppermint</i> (<i>Eucalyptus nitida</i>).
Temperate Rainforest	Below 900m	Myrtle (<i>Nothofagus cunninghamii</i>), dominate with leatherwood, sasfras, celery-top pines and horizontal pines also present. King Billy pines (<i>Athrotaxis selaginoides</i>) are found above 500m
Coniferous forest	Above 900m. Very sensitive to fire and can easily destroy the dominants.	Pencil pine (<i>Anthrotaxis cupressiodes</i>), tanglefoot (<i>Nothofagus gunni</i>) diselma (<i>diselma archeri</i>) comprise the taller species. Richea (<i>Richea Scoparia</i>), pineapple grass (<i>Astelia alpina</i>) and Mountain pepper (<i>Tasmannia lanceolata</i>), make up the understorey.
Sub Alpine sclerophyllous scrub	Form from to fire in coniferous forests.	Myrtle, tea-trees, lomatia (<i>Lomatia polymorpha</i>), occasional rainforest species. Bauera (<i>Bauera rubioides</i>), waratah (<i>Telopea truncata</i>) and cutting grass (<i>Gahnia grandis</i>) and snow berries (<i>Gaultheria hispida</i>), in the mid to lower storeys.

Table 1.1. Common Vegetation in the Pasminco Rosebery Mine mining lease (Pasminco, 1995, unbub.).



Plate 1.2. View of Bakers Creek with Williamsford in the Ring River Valley below.

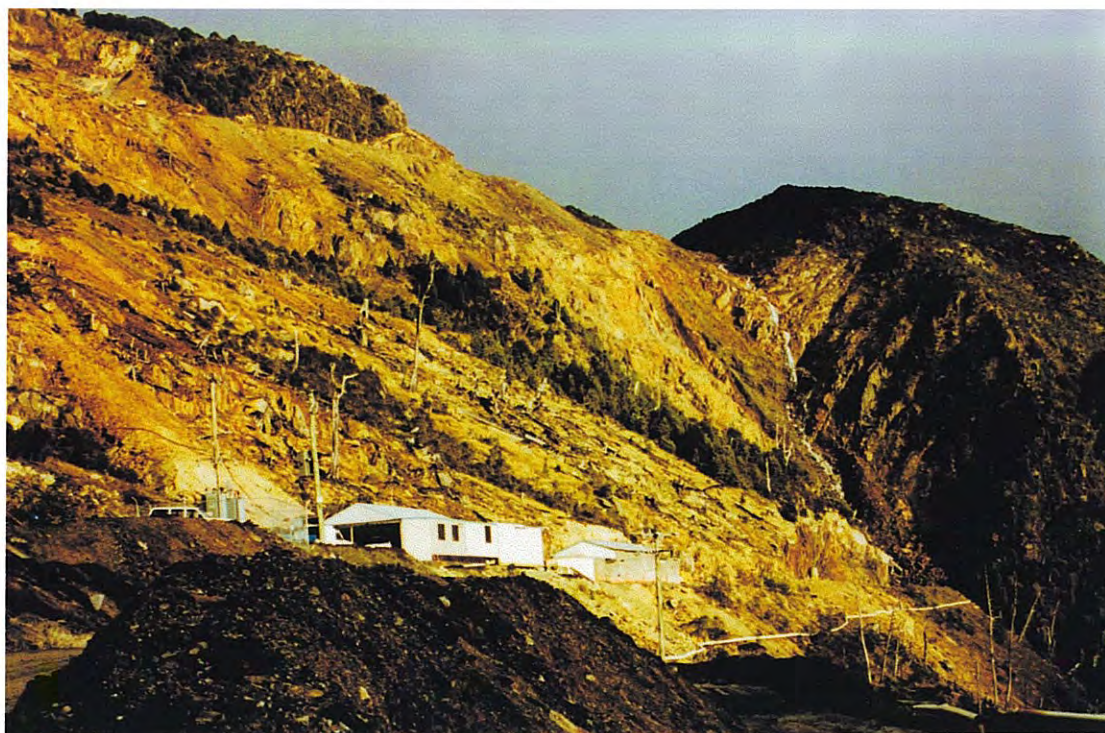


Plate 1.3. The Hercules Mine offices and Bakers Creek. Note the lack of vegetation in the vicinity of the mining area.

1.3.1.4 Geology and Soils

The Hercules orebody is a deformed stratiform polymetallic volcanically hosted massive sulfide (VHMS) deposit, located in the Mt Read Volcanics Belt (Lees *et al.*, 1990). The orebody is located in the Central Volcanic Complex (CVC). The Dundas Group (in the Western Volcano-Sedimentary Sequence; WVSS) unconformably overlies the CVC, and out-crops lower on the western slope of Mt Hamilton than the CVC.

The major rock types in the immediate mine area are feldspar-phryic and quartz-phryic volcanics, black shales and slates, and peletic to psammatic bedded tuffaceous sediments (Zaw, 1992; Lees 1986). A detailed description of the geology, mineralisation and alteration at Hercules is given in Appendix 1.

Soils in this area are primarily derived from sedimentary or volcanic rocks. Sedimentary soils develop from the Dundas group, which tend to have a thin soil and peat mantle and are easily eroded.

Volcanic soils predominate with the Mt Read Volcanics (MRV) giving rise to siliceous clay soils (due to the lava flows tuffs, volcanoclastic sandstones and shales present), and occasional orange clayey soils where dykes of basalt are present. These soils tend to be thin (up to 500mm), infertile and subject to compaction and surface puddling. They are poorly drained and are commonly sheet eroded when vegetation and surface mantle is removed (TGSR, 1997).

Soils in this area have a high organic content. The cool temperatures and high rainfall allow for the development of a peat mantle over a variety of rock types (Pasminco, 1995, unpub.). It rapidly adsorbs and retains water, making it resistant to erosion. Once removed (through fire or disturbance), the underlying soils are quickly eroded as the peat is not readily replaced. These soils are also highly acidic and low in nutrients such as nitrogen and phosphorus.

1.3.2 Recent Mining Operations

The Hercules mine closed in 1996 after 95 years of continuous mining, but reopened in 1996, (for an outline of the mining history and production at Hercules refer to Appendix I).

Hercules is currently operated by Mancala Pty Ltd, under a Sub-Lease Agreement to extract 130,000 tonnes of lead-zinc ore between 6 and 7 levels, which is being sent to the Rosebery crusher over a 2 year period. Despite the fact that Mancala are operating the mine, PRM is still responsible for maintaining environmental protection notices and certain environmental aspects and works.

1.4 HERCULES LEGISLATIVE REQUIREMENTS

The wastes which are discharged from Hercules are regulated by the *Environmental Management and Pollution Control Act 1994* (EMPCA 1994), which is administered by the Department of Environment and Land Management (DELM). However PRM is currently governed by the *Environmental Protection Act 1973* (EPA 1973). Schedule II of this act states the concentration limits for emissions into inland waters. However Schedule III, part IV of the same act, gives alternative emission concentrations for metalliferous mines, so long as the mining of these ores is in production before 1st January 1975 and remains in continuous production. Table 1.2 lists extracts from Schedule II of some of the concentrations for emissions into inland waters, and for those emissions from metalliferous mines (Schedule III). For metalliferous mines, the concentrations listed in the second column must also be adhered to if they are not listed separately in column 3.

Restricted Substance	Emission Into Inland Waters (mg/L)	Metalliferous Mines (mg/L)
Arsenic	0.05	0.1
Cadmium	0.01	0.03
Chloride	250.0	-
Chlorides and sulphates (combined total)	-	600.0
Copper	1.0	-
Cyanide	0.05	0.2
Fluoride	1.5	3.0
Iron and Manganese:	-	5.0
(combined total filterable)	1.0	0.2
Lead	0.05	0.01
Mercury	0.002	-
Sulphate	250.0	-
Zinc	5.0	-

Table 1.2. Emission concentrations into inland waters and for those from Metalliferous mines (from EPA 1973).

Water monitoring by Pasminco Rosebery Mine, has recognized that the discharge from some areas of the mine are much higher than the limits listed above, but has been exempt

from the constraints of the act via a ministerial exemption since 1977. In order to comply with the exemption, it is necessary for PRM to produce an environmental improvement programme (EIP), at least every 3 years. An EIP is a specific programme that, when approved, achieves compliance with the EMPCA 1994, either by reducing environmental harm, or by detailing the transition to a new environmental standard (EMPCA, 1994). In the next Hercules EIP however, the guidelines in the EPA 1973 will be superseded by *the State Policy on Water Quality Management 1997* (the State Water Policy; G Hawes, pers. comm, 1998).

1.5 PREVIOUS WORK

In order to comply with legislative requirements, numerous programs have been undertaken in recent years, to improve the environmental performance at Hercules. These programs include; a rehabilitation works program (1985-86), Performance Improvement Program (PIP 1992), Environmental Improvement Program (EIP 1994), and a 1996 EIP. Acid drainage, revegetation and water quality monitoring have all been key areas of focus.

Chapter 2

ACID MINE DRAINAGE

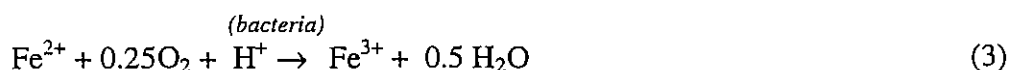
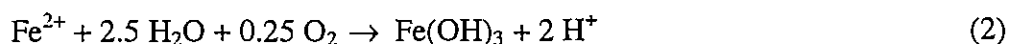
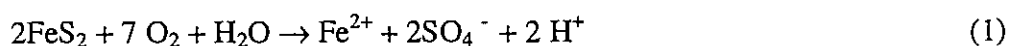
2.0 INTRODUCTION

Acid drainage is one of the largest, and most testing, environmental problems facing the minerals industry throughout the world. It can occur in both the hard rock and coal industries, whenever sulfide-bearing rocks are exposed to the environment. AMD can seriously degrade aquatic ecosystems (Harries, 1997). Major rehabilitation costs late in the life of a mine, or after mine closure can result from inappropriate management of acid-producing materials. Acid drainage is a highly complex issue, and is an area of active scientific research, as efforts are being made to develop better means of avoiding and controlling the problem.

2.1 ELEMENTS OF ACID PRODUCTION

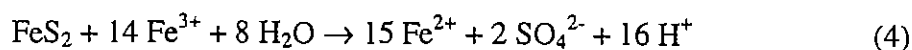
Acid drainage occurs through the oxidation of sulfide minerals and the transport of the oxidation products by water (processing water, rainfall, surface water and ground water, Taylor, 1998).

Pyrite is the most abundant sulfide mineral in the Earth's crust (Schrenk *et al.*, 1998). It is also highly reactive in the presence of free oxygen and water, and is consequently the major contributor to acid generation. Pyrite is abundant in most sulfide-bearing deposits. At Hercules it is abundant in the basal, massive to disseminated pyrite ore zone, but occurs throughout the extensive alteration halo up to 2km north and 1km south of the mine (Zaw, 1992). The oxidation of pyrite is described in the following reaction sequence (Hutchison and Ellison eds., 1992), and Sengupta (1993):



In these reactions, oxidation occurs directly from air and or water. Once reaction 1 is initiated, it sets up a self-perpetuating series of reactions (2, and 3). Reaction 1 is limited

by oxygen, however indirect oxidation is also possible with trivalent ferric iron acting as an oxidant:



Pyrite oxidation via equations 1-4 occurs in three stages, as described in Table 2.1. The rate determining step in the acidification process is the oxidation of ferrous iron to ferric iron (reaction 3). The rate at which this (and several of the other reactions) proceed is dependent on sulphur-oxidising bacteria which act as catalysts (Section 2.2.3). The formation of ferric ions allows reaction 4 to proceed, which is capable of producing substantially more acidity than air/water oxidation.

STAGE	REACTION #	CHEMICAL CONDIDITIONS
1	(1) Abiotic and bacterial oxidation (2) Abiotic and slows as pH decreases	pH approx. > 4.5 High sulphate and low iron concentrations
2	(1) Abiotic and bacterial oxidation (2) Rate determined primarily by bacterial activity	pH 2.5 - 4.5 High sulphate, total iron increasing, low $\text{Fe}^{3+}/\text{Fe}^{2+}$ ratio
3	(3) Rate totally determined by activity of bacteria. (4) Rate primarily determined by rate of reaction (3).	pH < 2.5 High Sulphate levels High total iron High $\text{Fe}^{3+}/\text{Fe}^{2+}$

Table 2.1 (Hutchison and Ellison eds., 1992) Stages of pyrite oxidation.

Pyrite morphology is an important factor in determining the rate of pyrite oxidation. Pyrite often occurs in spheroidal aggregates (framboidal texture), made up of fine-grained crystals (Deere *et al.*, 1995), rather than coarse grained euhedral crystals, or massive pyrite. Framboidal pyrite has a higher specific surface created by small particle size and high porosity (Zhang, and Evangelou, 1998).

Other common sulfide minerals capable of producing acid drainage include pyrrhotite (Fe_{1-4}S), chalcopyrite (CuFeS_2), chalcocite (Cu_2S), bornite (Cu_5FeS_4), arsenopyrite (FeAsS), tetrahedrite and tennenite ($\text{Cu}_{12}(\text{Sb,As})_4\text{S}_{13}$), molybdenite (MoS_2), sphalerite (ZnS), galena (PbS), cinnabar (HgS) and cobalite (CoAsS) (Hutchison and Ellison eds., 1992).

2.2 ENVIRONMENTAL INFLUENCES ON ACID GENERATION

Physical, chemical and biological processes can influence the rate of acid generation, and subsequent migration of acids and trace metals into the receiving environment.

2.2.1 Physical

Mining, exposes sulfides to the environment by excavating and fracturing sulfide-bearing rocks, and by production of mine wastes (tailings and waste rock). The combination of increased interaction with oxygen and the increased surface area to volume ratio of the waste products, means that AMD can be generated rapidly, and in some cases producing evidence in only a few months (Taylor, 1998).

Physical characteristics that influence metal solubility include; ambient temperature, pore space gas pressures, and waste particle size and shape. These properties are secondary to the chemical reactions that cause metals or other constituents to dissolve in AMD (Hutchison and Ellison eds., 1992).

Those characteristics that effect the rate of AMD migration include (Hutchison and Ellison eds., 1992):

- rainfall infiltration;
- waste permeability;
- pore water pressure and availability;
- transport mechanism (i.e. stream flow or diffusion).

These control the rate of movement of contaminants, amount of dilution and degree of mixing of AMD as it is transported into the receiving environment. As subsurface properties are different to waste material, several contaminant fronts may develop, each moving at different rates. The slower the rate of movement, the more opportunity there is for the chemistry of the water to change by reacting with the surrounding geological formations (Sengupta, 1993). This may improve or further degrade the quality of the water depending upon the acid-producing or acid-neutralising nature of the material.

2.2.2 Chemical

Sengupta (1993), noted that the rate of acid generation is principally controlled by chemical factors such:

- pH;
- Temperature;
- O₂ concentration, and content of the gas phase (if saturation is <100%);
- chemical activity of Fe³⁺;
- degree of water saturation;
- surface area of exposed metal sulfide; and
- chemical activation energy required to initiate acid generation.

Chemical factors are the principal factors for controlling the mobilisation of metals (Hutchison and Ellison eds., 1992), with pH being the most significant factor. Other factors include redox (reduction-oxidation potential), metal complexing, adsorption characteristics, and AMD compositions.

2.2.3 Biological

Oxidation of sulfides occurs by inorganic and biologically catalysed chemical reactions (Gould *et al.*, 1994). Microorganisms can accelerate the rate of oxidation of Fe²⁺ to Fe³⁺ (reaction 3) by a factor of 10⁶, (Ragusa and Magwick, 1990), which is the rate-determining step in the pyrite oxidation reaction scheme (Singer and Stumm *et al.*, 1970). Therefore the rate of pyrite dissolution is generally controlled by microbial activity (Schrenk *et al.*, 1998).

Some bacteria use sulphur compounds in order to obtain energy. Heterotrophic bacteria require complex organic compounds as their main carbon source, and obtain energy either by photosynthesis or oxidising organic compounds. Whereas Autotrophic bacteria can grow solely on inorganic compounds, using CO₂ for carbon, and photosynthesis or inorganic compound oxidation for energy (Gould *et al.*, 1994). Table 2.2 lists the physiological types of sulphur oxidizing bacteria.

Under oxidizing conditions, acidophilic chemolithotrophic and heterotrophic bacteria can oxidise Fe (II), with some chemolithotrophic bacteria such as *Thiobacillus ferrooxidans* capable of reducing Fe (III) under both aerobic and anaerobic conditions at pH less than 1.5 (Fortin *et al.*, 1996). At low pH, abiotic ferrous iron oxidation proceeds very slowly,

whereas iron oxidizing bacteria can catalyse the reaction by a factor of 10^5 to 10^6 , (Ragusa, and Madgwick, 1990).

PHYSIOLOGICAL TYPE	CARBON SOURCE		ENERGY SOURCE	
	Inorganic	Organic	Inorganic	Organic
Obligate chemolithoautotroph	+	-	+	-
Faculative chemolithoautotroph	+	+	+	+
Chemolithoheterotroph	-	+	+	+
Heterotroph	-	+	-	+

Table 2.2 Physiological types of bacterial able to oxidise reduced sulphur compounds. (Holt, *et al.*, 1994)

The *Thiobacillus* genus contains bacteria that are small, gram-negative, rod-shaped, and capable of oxidising reduced sulphur compounds. *T. ferrooxidans* is the most studied bacterium (in terms of AMD). It is generally assumed to be the most important species in accelerating the dissolution of metal sulfide, and is readily found in AMD environments (Schrenk *et al.*, 1998). It is capable of oxidising both iron and sulphur, where as other species such as *T. thiooxidans* can only oxidise reduced sulfides. *Leptospirillum ferrooxidans* is a Gram-negative, spiral-shaped bacterium, that oxidises iron but not reduced sulfides. Recent studies have shown that this species is capable of out-competing *T. ferrooxidans* in certain conditions, and is recognised as an important species associated with AMD (Schrenk, *et al.*, 1998). Those bacteria which are commonly associated with AMD are listed with their optimum environmental conditions in Table 2.3.

NAME	pH	TEMP. (C°)	ENERGY SOURCE
<i>Thiobacillus ferrooxidans</i>	1-3.5	30-35	Obligately Chemolithoautotrophic
<i>Thiobacillus thiooxidans</i>	2-4	25-30	Obligately Chemolithoautotrophic
<i>Thiobacillus novellus</i>	6-8	25-30	Faculatively Chemolithoautotrophic
<i>Thiobacillus denitrificans</i>	7	25-30	Obligately Chemolithoautotrophic
<i>Thiobacillus thiosparus</i>	6-8	25-30	Obligately Chemolithoautotrophic
<i>Leptospirillum ferrooxidans</i>	0.7-1	>40	Obligately Chemolithoautotrophic

Table 2.3 Bacteria commonly associated with AMD (Ragusa and Madgwick, 1990; Gould *et al.*, 1994; Shrenk *et al.*, 1998;)

There are some bacterial species which can be used to treat AMD. Sulphur reducing bacteria (SRBs), can remove metals and increase pH if anaerobic conditions exist, (such as those encountered in wetland environments; Gould *et al.*, 1994).

2.3 ENVIRONMENTAL IMPACTS

Acid drainage is a persistent and potentially severe source of pollution. It degrades the environment by increasing acidity and heavy metal concentrations in waterways. It can accelerate erosion causing high levels of suspended solids in the drainage waters and iron precipitates may form in the receiving environment. A summary of these impacts is listed in Table 2.4.

PROPERTY	TYPICAL ASSOCIATED CHEMICAL SPECIES	CONCENTRATION RANGE	ENVIRONMENTAL IMPACT
Acidity	Sulphuric acid	pH 2 - 4	mobilisation of metal ions,
Iron	Ferrous and ferric ion; Ferric oxides, hydroxides; jarosites.	100 to 3000 ppm	discolouration and turbidity in receiving waters as pH increases and ferric salts and oxy-hydroxides ppt.
Heavy metals	Co, Cu, Mg, Zn, Cd, Hg, Pb, As	1 to 200 ppm	Reduction in aquatic flora and fauna; bioaccumulation; reduced quality of drinkable ground water.
Total dissolved solids (TDS)	Ca, Mg, Al, Sulphate.	100 to 30,000 ppm	Reduction in quality of drinkable groundwater; and reduction in quality of water supplies for stock.

Table 2.4 Typical environmental impacts of AMD (modified from Ritchie, 1994).

2.4 SOURCES OF AMD

Acid drainage can form in natural, undisturbed environments, where sulfide-bearing bedrock is exposed. This process usually occurs slowly, due to the small volume of sulfides exposed. Mining however, disrupts the environment and increases oxygen availability, which is a key ingredient in acid production. Fracturing, creation of mine wastes, and lowering of the water table, are all means by which mining increases the supply of oxygen to potentially acid producing sulfidic rocks and waste.

Acid drainage can occur at many mineral deposit types, including base metal, uranium, copper and gold mines. The ore and gangue mineralogy and the disposal strategies employed, are factors which determine whether AMD occurs or not (Harries, 1998).

The major sources of acid drainage are waste rock dumps, low-grade ore, overburden and ore stockpiles (Taylor, 1998). Tailings and coal washery wastes are smaller potential sources of acid drainage compared with waste rock, however their smaller particle size and higher surface area to volume ratio increases their reactivity. Abandoned mines, adits, shafts and open-cuts, can also be significant sources of acid drainage (e.g. Prince Lyell workings in Mt Lyell in Tasmania).

2.5 AMD PREDICTION AND TESTING

Classifying acid-producing material is an important first step in managing mine wastes. There are many methods used to predict AMD, which can be divided into four groups (Hutchison and Ellison eds., 1992);

1. *Geographical, paleoenvironmental and geological models*: This method relies on extrapolating geochemistry data from mines in similar geographical areas, or assumes that areas with similar palaeoenvironments and geological formations have similar acid producing potentials.
2. *Geochemical static tests*: These are fast, relatively inexpensive screening tests, that evaluate the inorganic AMD potential in samples by comparing the acid generation and neutralisation capacity of a given material (OSS, 1997).
3. *Geochemical kinetic tests*: These laboratory tests are used to confirm static test results, to evaluate the rate of sulfide oxidation and predict acid drainage characteristics (by producing a leachate that can be analysed; OSS, 1997). However they do not involve bacteria, and as such can give an inaccurate impression of the expected rate of sulfide oxidation.
4. *Mathematical models*: Information is extrapolated from laboratory tests to predict acid generation or neutralisation, beyond the length of the tests.

The first method is useful for preliminary indications of acid generating potential, but are more reliable for prediction in coal mines rather than hard rock mines (Hutchison and Ellison eds., 1992).

Geochemical static and kinetic tests are the most reliable prediction methods available. They are often used to determine which methods will be employed to combat acid

generation. A detailed account of the various static and kinetic tests available in Australia is provided by Harries (1997).

2.5.1 Static and Kinetic Tests

Geochemical materials can be classified into four primary types (Miller, 1998);

1. *ACM*: Acid Consuming Material (i.e. limestone);
2. *NAF*: Non-Acid Forming;
3. *PAF-LC*: Potentially Acid Forming, Low Capacity (<5 kg H₂SO₄/t); and
4. *PAF*: Potentially Acid Forming.

Some of the static tests commonly used in Australia are summarised in Table 2.5.

STATIC TEST	METHOD
Acid Base Accounting	Used to determine the capacity of mine wastes to generate acid.
- Saturated paste pH and salinity	A representative sample of mine rock is crushed (<1mm) and saturated with distilled water to form a paste. the pH and EC is measured (12-24 hours afterwards to represent equilibrium).
- Maximum Potential Acidity (MPA)	This is calculated by determining the sulphur content of the sample as a percentage and multiplying the sulphur by a conversion factor that is based on the reaction stoichiometry for pyrite oxidation. If complete oxidation of sulphur occurs, the conversion factor generates kilograms of acid that can theoretically be produced from one tonne of material with that sulphur content.
-Acid Neutralising Capacity (ANC)	Quantifies the ability of the sample to buffer or neutralise acid, by reacting a sample with a known amount of HCl, and back titrating to determine the amount of acid consumed.
-Net Acid Producing Potential (NAPP)	This is the theoretical balance between the samples capacity to generate acid and its capacity to neutralise any acid that is produced, (i.e. NAPP = MPA - ANC) thus a negative value will be acid-consuming, and a positive value will be acid-producing.
Net Acid Generation (NAG)	A simple and fast estimate of the net acid potential. A strong oxidant such as hydrogen peroxide is added to a sample, and the pH is measured after the reaction is complete. A NAG pH > 4 is non acid forming; pH ≤ 4 has the potential to produce acid.

Table 2.5 Examples of static tests commonly used in Australia (OSS, 1997; Harries 1997).

Kinetic tests simulate weathering and oxidation of the samples, so they are undertaken over longer time periods than static tests. These tests give an indication of oxidation rate, lag time till the onset of acid generation, and the effectiveness of control techniques used to

limit oxidation rates (OSS, 1997). However they do not consider the influence of bacteria on the rate of acid production.

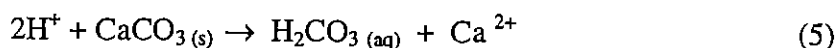
2.6 ACID-CONSUMING REACTIONS

There are three groups of reactions which inhibit, delay or prevent the formation of acidic conditions (Hutchison and Ellison eds., 1992):

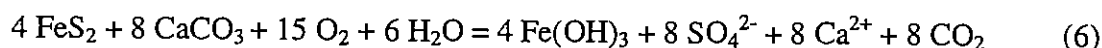
- Reactions with carbonates
- Reactions with aluminosilicates or other silicates.
- Reactions with, or formation of other compounds (e.g. gypsum and jarosite formation).

2.6.1 Reactions with Carbonates

Carbonates are regarded as the most effective minerals used in the control and prevention of AMD, as they can occur in large quantities (OSS, 1997). Calcite is the most common acid-consuming carbonate found in nature, and is the most popular choice in abating AMD (Zhang, and Evangelou, 1998). Calcium carbonate consumes acid by reaction (5):



The combined reaction of acid generation from pyrite by air/water oxidation and reaction with calcite is described by reaction (6) (Hutchison and Ellison eds., 1992):

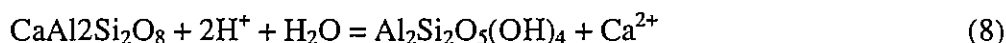


Note that no acid is produced in reaction (6). According to this reaction, two moles of calcite will neutralise the acid generated by a single mole of pyrite.

The formation of ferric hydroxide precipitates and the potential for gypsum ($\text{CaSO}_4 \cdot 2\text{H}_2\text{O}$) formation in reaction (6), can coat (or armor) the surfaces of the reacting carbonate material. This reduces the surface area of the carbonates and reduces the rate of reaction. Complete armoring can cause limestone to cease reacting with acid drainage, removing its neutralising potential. Coatings do not occur on the sulfides however, as acid generation occurs at the surface of the sulfide particles, and low pH conditions inhibits the formation of ferric hydroxides and gypsum (Hutchison and Ellison eds., 1992).

2.6.2 Reaction with Silicates

Examples of acid consuming reactions involving aluminosilicates are described in reactions (7) and (8), for plagioclase and K-feldspar respectively (Hutchison and Ellison eds., 1992);



Feldspars, olivines, pyroxenes, micas and amphiboles are all capable of consuming acid. Though they are less reactive and therefore less effective than carbonates at neutralising acid drainage, the sheer mass of silicate minerals in mine wastes or natural environment, can have a significant effect (Hutchison and Ellison eds *et al.*, 1992).

2.7 SCOPE OF THE AMD PROBLEM

AMD is a world wide problem. In Canada alone a 10 year research program (Canadian Mine Environment Neutral Drainage - MEND program) cost C\$18 million, and concluded that the total drainage liability in Canada ranged from C\$2 - C\$5 billion (C\$1 = A\$1.04 at 8 November 1998; Kleinmann, 1997). In the USA, AMD is also a major issue, with 7000 km of streams in the eastern states, considered to be seriously affected by acid drainage from coal mines, with further water quality concerns in hard rock mines in the western states. The US Forestry Service estimated 8000-16000 km of streams to be affected by both active and inactive mines and waste rock piles (Harries, 1997).

The operational cost of managing mine wastes in Australia has been estimated at \$60 million annually, for the whole Australian mining industry. While this is only a small fraction of the total annual costs of the Australian mining industry, it does represent a significant proportion of the amount spent on environmental issues (Harries, 1997 (a)). If proper strategies for minimising acid drainage are not in place from the beginning of mine operations, mining companies risk the possibility of large liabilities late in mine life and after mine closure (Harries, 1998).

In Tasmania the most well known case of AMD is from the Mt Lyell copper mine. For over a century, large volumes of acid drainage and tailings have been discharged into the King and Queen Rivers and Maquarie Harbour, with devastating environmental consequences (Koehnken, 1997). Other locations in Tasmania where historic mining practices have left a legacy of AMD problems include; Rossarden, Storys Creek and Savage River.

SECTION 2

EVALUATION OF AMD AT HERCULES

Chapter 3

SAMPLING AND ANALYTICAL TECHNIQUES

3.0 INTRODUCTION

This chapter discusses the methodology and justification of the water sampling programs used in this study. The location of each sampling site is described, together with reasons for their inclusion. Analytical techniques used in this study are summarised, including the procedures involved in water, precipitate and sediment sampling, the field measurements undertaken, and laboratory analyses performed on the samples.

3.1 WATER SAMPLING

Two water sampling programs were implemented in this study, with the aim of investigating the water chemistry under different climatic regimes (i.e. low-flow and storm events). By understanding the results at low flows and storm flows, the range of possible water chemistries can be determined, which will influence remediation and rehabilitation choices.

3.1.1 Long Term Sampling (Low Flow)

Long term sampling was undertaken to ascertain the effects of seasonal changes on the water emanating from Bakers Creek. These samples were taken at relatively low flows (non-storm events).

Four sampling runs were completed between February and August 1998. Each sampling trip involved water site analysis and water sampling at up to eleven sites (Sites A-K; Figure 3.1). Note that not every site was sampled during each trip. Reasons include; a lack of drainage water, access difficulties and certain sites being discarded as the study progressed. A list of each site and reasons for their inclusion in the sampling program is given in Table 3.1.

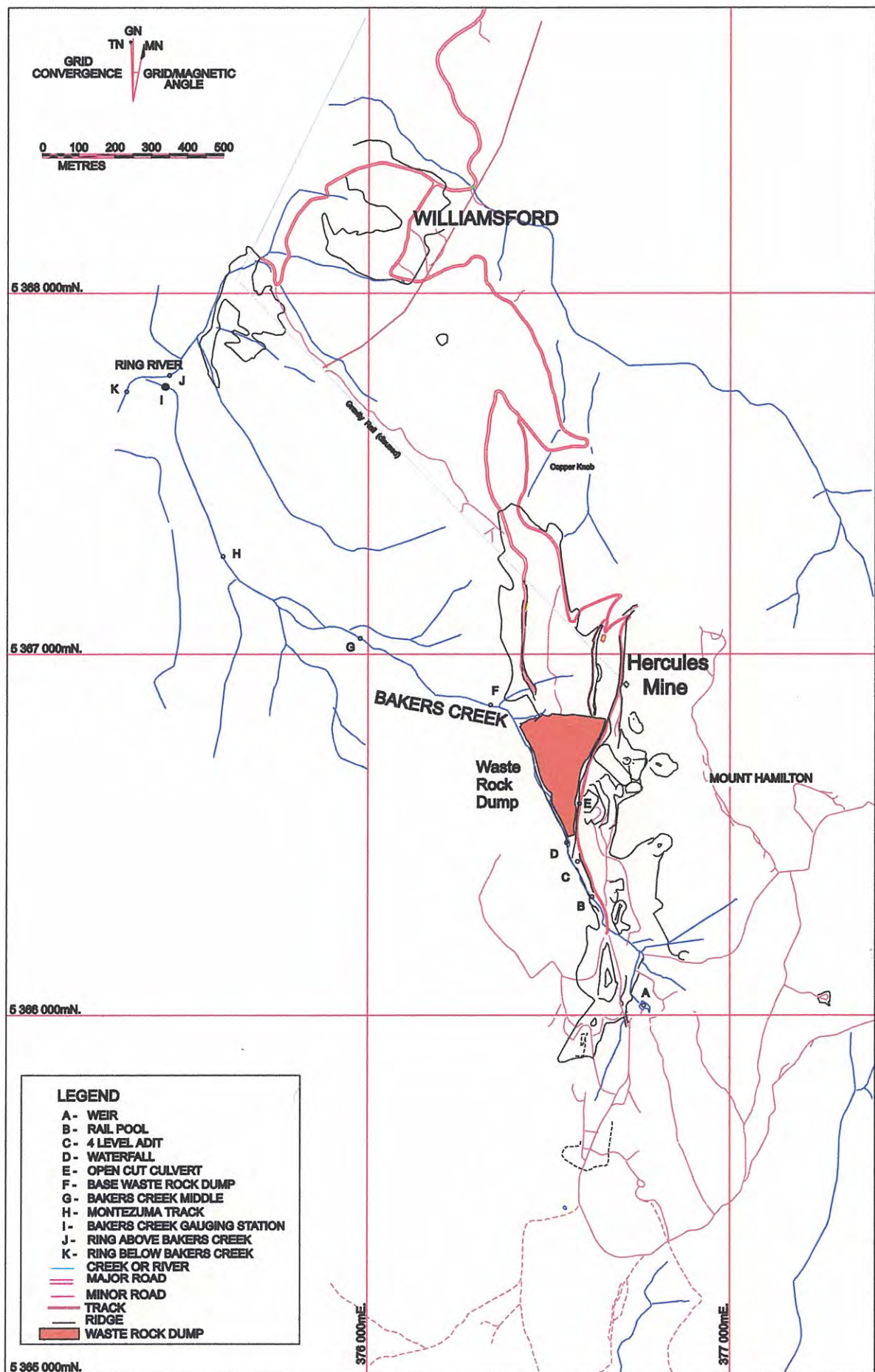


Figure 3.1 Location map of water sampling sites along Bakers Creek.

SITE	SITE NAME / ABBREVIATION	REASONS FOR INCUSION
A	Weir	Up stream of mining activity - used for base line data as it is uncontaminated by AMD (Plate 3.1).
B	Rail Pool	Located 100m downstream from the weir This site is a few metres downstream of some minor open cut workings.
C	4 Level Adit	Point source of pollution that flows directly into Bakers Creek (Plate 3.2).
D	Waterfall	400m downstream of the weir and 50m downstream of the Adit. This data can be compared with that from the rail pool to determine the impact of the Adit, and is directly before the start of the Bakers Creek waste rock dump (BC-WRD). This site constrains water quality upstream of the waste rock dump (Plate 3.3).
E	Open Cut Culvert (April only)	Drains the large open cut area adjacent to the level 4 road, and flows down the side of the mountain into the WRD.
F	Base Waste Rock Dump (August only)	Located at the base of the waste rock dump (900 m from the weir), so these measurements can be compared with those at the waterfall to see how the WRD effects the water quality. This sampling site was difficult to access, therefore was only sampled once (Plate 3.4).
G	Bakers Creek Middle	Located 1300 m downstream of Site F. This was the closest accessible point to the base of the waste rock dump, during most of the field season. Down stream from this point several small streams discharge into Bakers Creek.
H	Montezuma Track (February and April only)	Located below confluence with Tipperary Creek, 1700m downstream from the weir. This site was used at the beginning of the sampling as it was easy to access, but was later discarded when it became apparent that Tipperary Creek did not influence the water quality adversely.
I	Bakers Creek Gauging Station	Last measurable point before Bakers Creek (2400 m from the weir) discharges into the Ring River. Contains a monitoring station operated by the Hydro-Electric Corporation. The equipment at Bakers Creek consists of a UNIDATA data logger, Unidata conductivity instrument, GREENSPAN pH sensor and a DRUCK Hydrostatic depth probe. The values are logged every 15 minutes and powered by a 12vDC power supply that is backed up by a 40 watt solar panel.
J	Ring River Above Bakers Creek	This site was used to monitor the water quality of the Ring River above the confluence with Bakers Creek.
K	Ring River Below Bakers Creek	This site is below the confluence with Bakers Creek, and can be compared with Site J to determine how acid drainage from Bakers Creek impacts on the Ring River.

Table 3.1 Sampling sites and reasons for their inclusion.



Plate 3.1 Site A (Weir) located at the start of Bakers Creek used for base line data.



Plate 3.2 Site C (4 Level Adit) discharges directly into Bakers Creek (seen to the far right in this figure).



Plate 3.3 Site D (Waterfall) located directly before the start of the Bakers Creek waste rock Dump.



Plate 3.4 Site F (Base of the Waste Rock Dump)

3.1.2 Intensive Sampling (storm flow)

Intensive sampling before, during and after a storm event was undertaken in July and September to determine water quality during storm events. This data could then be compared with the long term sampling results to determine whether storms result in the discharge of contaminant plumes into Bakers Creek.

Intensive sampling was conducted at the Bakers Creek Gauging Station site (Site D). This site was used as it was easily accessed, and because continuous flow measurement data was available from the monitoring station, which allows for mass loading calculations (Chapter 4).

Ideally, an initial sample should be taken before prior to the storm event. This was achieved in the July storm event, but not in September. After the onset of rain, a sample was taken every half an hour for the first few hours, after which hourly samples were collected for several hours. A final sample was taken up to 24 hours after the onset of rain. The time at which the sampling began, the duration of the sampling and the intensity of the sampling, was ultimately governed by; time of arrival to the site before the storm event, operator fatigue and equipment availability.

3.1.3 Sampling Procedures

Samples were taken at each site in accordance with the procedures set out in *Standard Methods for the Examination of Water and Wastewater* by the American Public Health Association (APHA) et al., (1992). Polypropylene containers were used, with either one 500ml sample or two 250ml samples obtained for each site. Each bottle was rinsed and emptied three times downstream of the sampling site prior to sampling. Samples were collected as close to the centre of the stream as possible and approximately 10cm below the surface. The bottles were filled completely, to minimise interaction of the water with trapped air. Samples were filtered to remove colloidal material and stored in a refrigerator until they were ready to be analysed.

3.1.4 Field Measurements

Field measurements of pH, Eh, temperature and conductivity were taken at each site during each sampling trip. Field analyses were obtained using the equipment listed in Table 3.2.

The pH, Eh and pocket conductivity probe were calibrated prior to sampling with a number of solutions which are outlined in Appendix 2. In addition, pH, conductivity, flow and temperature measurements were collected at 15 minute intervals over the year by the gauging station at Site I.

MEASUREMENT	EQUIPMENT SPECIFICATIONS
pH	WTW pH meter with a Hanna pH probe; or WTW LF 320 pH meter and probe
Eh	Orion redox probe
Conductivity	Hanna pocket conductivity meter; or WTW LF 330 conductivity meter and probe
Temperature	WTW temperature probe

Table 3.2 Equipment types used to take field measurements.

3.1.3.1 pH

The pH scale describes the acidity of aqueous solutions, and is a direct measure of the activity of hydrogen ions in solution:

$$\text{pH} = -\log_{10} (a\text{H}^+_{aq})$$

Acids influence chemical reaction rates, chemical speciation, biological processes and contribute to corrosiveness (APHA *et al.*, 1992). pH less than 5 is lethal to fish, and acidified streams have lower macroinvertebrate densities, fewer species and an altered taxonomic composition (ANZECC/NWQMS, 1992). Low pH waters generally have a much greater metal-carrying capacity than near-neutral waters.

3.1.3.2 Redox Potential (Eh)

Eh is a measure of the reduction-oxidation (redox) potential of a solution. It represents an intensity factor of redox, but not the capacity of the system to be oxidised or reduced (APHA *et al.*, 1992). The potential is measured (in mV) as the difference between an inert indicator electrode and the standard hydrogen electrode (SHE) and units. The SHE is fragile and impractical for routine field use, therefore, a silver:silver-chloride reference electrode was used (APHA *et al.*, 1992). The difference between the potential of the reference electrode and that of the SHE was corrected with the following formula:

$$E_{(SHE)} = E_0 + C$$

Where:

$E_{(SHE)}$ = redox potential of the sample relative to the SHE.

E_0 = potential difference developed by the platinum redox electrode.

C = potential developed by the reference electrode portion relative to the SHE.

At 10°C $C = 251$, so this value was added to the redox values measured in the field.

Redox conditions strongly influence the reactivities and mobility of important biological elements (i.e. Fe, S, N, and C) and numerous other metallic elements (APHA *et al.*, 1992). Thus redox is an important parameter when predicting the behaviour of elements or compounds.

3.1.3.3 Conductivity (K)

Conductivity measures the ability of an aqueous solution to carry an electric current in μS units (APHA *et al.*, 1992). Factors which influence this ability include; the presence of charged ionic species, their total concentrations, mobility, valence, and the temperature of measurement. Conductivity can be used to estimate total dissolved solids (TDS) and salinity, in waters unaffected by heavy metal contamination (i.e. unpolluted saline groundwaters).

3.1.3.4 Temperature

Temperature influences pH, conductivity and metal solubilities, and is therefore an important parameter that must be measured to fully understand chemical speciation in drainage waters.

3.1.3.5 Flow (Q)

Flow is one of the most important parameters to be measured, as it is used to calculate mass loadings, which quantify the amount of metals being transported. Flow measurements downstream from the waste rock dump were obtained from the HEC's monitoring station. However it was also necessary to obtain flow data upstream of the waste rock dump to calculate mass loads in Bakers Creek upstream and downstream of the

waste rock dump, so that the percentage contribution of contaminants from the dump can be calculated.

Flow was only measured upstream once, between sites A and B where the stream is diverted through a long cement pipe. A jug and stopwatch technique was used, which involves placing a bucket underneath the stream-flow and measuring the time taken to fill it. In this way a direct measurement in L/sec is obtained. A 10 L bucket was used, and the exercise was repeated three times to calculate an average flow.

Ideally, flow should have been calculated at Site D, to determine the mass loads in Bakers Creek directly before the WRD. This would have included the discharge from Site C and the open cuts prior to Site B. However it was not possible to measure the flow with the techniques available as this section of the creek is filled with boulders, causing turbulent and or meandering stream flows.

An approximate flow rate for Site D was obtained. As the conditions were dry, there was no surface flow from Sites C and around B, so an assumption was made that the flow measured between Sites A and B would be approximately the same as the flow at Site D.

3.2 Analyses

3.2.1 Water Analyses

3.1.2.1 Alkalinity, Sulphate and Chloride

Samples were taken to the Renison Analytical Laboratory within 12 hours of sampling, and filtered to remove colloidal material. Once filtered, 50 ml of the sample was bottled and submitted for sulphate and chloride analyses by ion chromatography. Alkalinity titrations were performed on 200 ml of sample, where the pH measured in the field was greater than 4.5 (analytical methods are described in detail by Parr, 1997).

3.2.1.2 ICP-EOS

The remaining 250ml of sample was acidified by adding 2.5-5ml of concentrated nitric acid to preserve it for ICP-EOS analysis. Samples collected in February were analysed for Ag, Al, As, B, Ca, Cd, Co, Cr, Cu, Fe, K, Mg, Mn, Mo, Na, Ni, P, Pb, S, Si, Sn, V, and Zn. Some of these elements were found to have concentrations below detection limits at all

sample sites. This suite of elements was reduced to Al, As, Ca, Cd, Co, Cu, Fe, K, Mg, Mn, Mo, Na, Pb, S, Si, and Zn for the remaining sampling trips. The detection limits for each element analysed by the ICP-EOS at the University of Tasmania have not been determined, but are believed to be on the order of 20-50 ppb for most elements (Townsend, pers comm., 1998).

3.2.2 Other Analyses

3.2.2.1 Environmental Scanning Electron Microscope (ESEM)

Iron precipitates from Sites C and F were sampled for ESEM analyses. A small amount of the sample (3cm diameter) was dried on a watch-glass in an oven at 50 °C for at least 24 hours before taken to the ESEM laboratory. Samples were then carbon coated and placed in the ESEM for viewing of the precipitates at high resolutions (up to 10µm).

3.2.2.2 Bacteria

Sediment samples were collected from Sites C and F for bacterial identification from the *Thiobacillus* genus. The glass or plastic containers (100 - 200ml) that were used for bacterial sampling, needed to be sterilised prior to use, to ensure that they were not already contaminated with bacteria. The containers were autoclaved (a high pressure and temperature, air-free steaming apparatus; Hale *et al.*, 1995) at 121°C for 20 minutes.

While collecting samples care was taken to minimise bacterial contamination from the air. Once collected, samples were refrigerated and analysed within two weeks. Analyses were performed by Leah Hawkes in the Agricultural Science Department at the University of Tasmania.

Obligate autotrophic species do not always grow well in organic solidifying agents such as agar, as they require an inorganic energy source (Holt. *et al.*, 1994). Three types of media were used to grow the bacteria (Appendix 3.). The media used were selective, in that only inorganic media were used, with Fe, S or thiosulphate as energy sources. Thus heterotrophic bacteria were not identified in this study. *Thiobacilli* are inhibited by agar, thus for this bacteria the sample was filtered onto filter paper and placed on top of the agar, so as not to be in direct contact.

One gram of sample was added to 25 ml of medium in a conical flask, and incubated at 25° C for one week. 0.1ml of the sample was then transferred to a spread plate and incubated again for another 3 weeks at 25° C. As possible bacterial colonies appeared, sub-samples were taken and cultured separately, for identification.

Chapter 4

WATER QUALITY OF BAKERS CREEK

4.0 INTRODUCTION

Long term and intensive water sampling programs were implemented at Hercules to characterise and quantify the AMD emanating from the Bakers Creek waste rock dump (WRD). The water chemistry, mass loads, and bacterial compositions have been determined, as they are factors which influence the characteristics and quantity of AMD produced.

Six visits were made to the study area, with four used to collect samples for long-term sampling and the other two for the intensive sampling. The long-term samples were collected on February 16th-19th, April 5th, June 3rd-4th, and August 31st, 1998. The intensive samples were collected on July 27th-29th, and September 22nd-23rd, 1998. It was important to understand the climatic controls on the water chemistries, especially to identify whether there were contaminant plumes associated with storm flows, as this would have implications for choosing suitable remediation strategies.

4.1 CONTAMINANTS

The concentrations of contaminants determined in this study are compared to the *Australian Water Quality Guidelines for Fresh and Marine Waters* (ANZECC/NWQMS, 1992). Those guidelines relevant to the waters at Bakers Creek are listed in Table 4.1, with standards for protection of aquatic ecosystems and for drinking water quality.

Concentrations of the elements were analysed with ICP-EOS at University of Tasmania or Cockle Creek in NSW. The sulfate and chloride were analysed at the Renison Analytical Laboratories by ion chromatography. The results from these concentration analyses for both long term and intensive sampling are listed in Appendix 3. These suite of elements were chosen as they represent the major solutes, elements, and trace metals found in Bakers Creek.

Attribute Studied	Drinking Water (ppm)	Aquatic Ecosystems (ppm)
pH	6.5 - 8.5*	6.5 - 9.0*
Sulphate	400.00	
Chloride	400.00	
Al	0.20	0.010
As	0.05	0.050
Cd	0.01	0.002
Cu	1.00	0.005
Fe	0.30	1.000
Mn	0.10	
Na	300.00	
Ni	0.10	0.020
Pb	0.05	0.005
Zn	5.00	0.010

* These values are in pH units.

Table 4.1 Attributes covered by the ANZECC/NWQ water quality guidelines that are relevant to this study.

4.1.1 Ficklin Plots

A classification scheme was developed for AMD by Ficklin *et al.*, (1992). "Ficklin plots" summarize the major attributes of mine drainage and mineralized natural drainage by plotting pH against total trace metal concentration (Zn+Cu+Co+Ni+Cu+Pb). Several water types can be arbitrarily defined using these plots (e.g. Figure 4.1). Ficklin plots can be used to rank the degree of contamination in drainage waters.

Figure 4.1 is a Ficklin plot of all of the water samples analysed in this study (from both long term and intensive sampling programmes). The majority of the samples plot in the Acid, High Metal category, with one sample on the border line of High Acid, High Metal / Acid, High Metal. Only one sample is classified as the non-contaminated Near-Neutral, Low Acid water type.

In Figure 4.2 water analyses are discriminated on the basis of site location (the location of each site has been given in Figure 3.1). Site A (baseline sample located at the Weir), plot predominately in the Acid - Low metal region. Samples downstream of this site generally have higher metal concentrations and a lower pH, causing them to plot in the Acid High Metal category. Water located from Site F at the bottom of the waste rock dump had the highest metal concentrations (Figure 4.2) closely followed by sites E and G (open cut culvert and Bakers Creek Middle sites respectively). Waters discharging at Site C (the Adit), had the lowest pH (Figure 4.2).

Ficklin Plot of Long-term and Intensive Samples

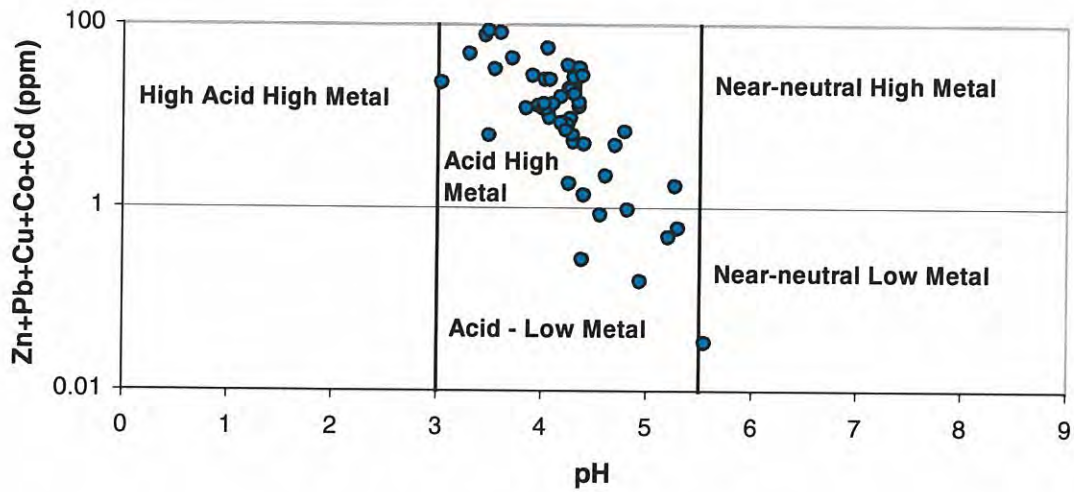


Figure 4.1 Ficklin plot of all of the water samples analysed in this study.

Ficklin Plot-Sampling Sites

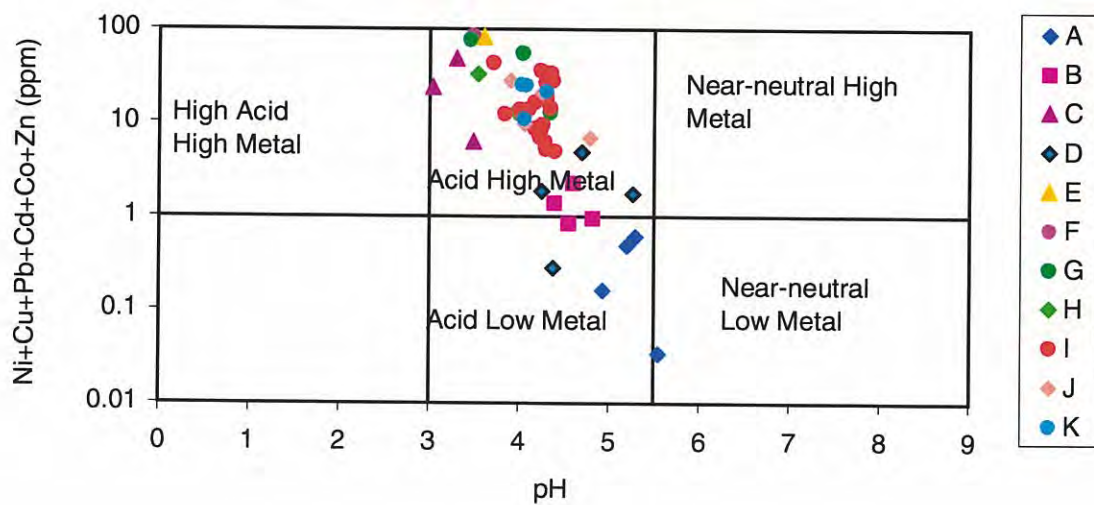


Figure 4.2 Ficklin Plot of the samples at each sampling site.

4.1.2 Long Term Sampling

Figures 4.3 and 4.4 consist of a series of graphs depicting the spatial and temporal variations in trace element compositions along Bakers Creek (where Site A (baseline) occurs at 0m and Site K (Ring River) is 2600m downstream).

The maximum and minimum values of the attributes studied in the long term sampling program are listed in Table 4.2.

Attribute Studied	Maximum Value	Minimum Value	Location of Max.
pH	5.55	3.03	Site A
Temperature	13.7	3.8	Site B
Conductivity	1200	0	Site I
Alkalinity (CaCO ₃ /L)	0.1		
Sulphate	1440	2	Site C
Chloride	29.0	3.8	Site A
Al	8.47	0.11	Site C
As	0.05	0	Site C
Ca	20.3	0	re
Cd	0.4	0	tantial
Cu	1.15	0.004	ly hiF
Fe	43.8	0.0575	Site C
K	6.31	0	Site I
Mg	9.04	0.5	Site C
Mn	47.3	0.021	Site E
Mo	0.0398	0	Site H
Na	6.56	3.03	Site E
Pb	2.40	0	Site G
S	86.8	0.96	Site F
Si	6.16	0.637	Site C
Zn	82.2	0.034	Site F

Table 4.2 Maximum and minimum values of the attributes studied in the long term sampling program.

From these graphs it is evident that the WRD has a significant impact on the AMD input into Bakers Creek, however there is also contamination entering Bakers Creek before Sites B and D. Comparing Sites J (downstream from Bakers Creek) and K (Ring River before Bakers Creek), illustrates what happens to Bakers Creek when it mixes with Ring River.

4.1.2.1 Physiochemical Parameters

pH (Figure 4.3 A)

Based on the results of the long-term sampling program, the pH of Bakers Creek decreases from Site A to Site I. The baseline values at Site A are generally around 4.9-5.5,

whereas at the end of Bakers Creek (Site I), the pH has dropped to between 3.6 and 4.5. The pH decreases by 1 unit between Sites A and B, and then stays relatively constant up to Site D (Figure 4.3 A). This implies that the Adit (Site C) is not discharging large quantities of acid water into Bakers Creek.

The pH decreased by 0.5 - 1.8 units after passing through Site D to the WRD. In August, the pH of Bakers Creek decreased from 5.3 at the top of the WRD (Site D) to 3.5 at the base (Site F). The pH then increased gradually to a value of 3.71 at Site I.

Conductivity (Figure 4.3 B)

Conductivity was fairly uniform. Values increased from Site A to B, and again at Site F. In April and August, Conductivity increased continuously to Site I, but in June dilution (after Site G) resulted in a decrease in conductivity with distance along Bakers Creek.

Temperature (Figure 4.3 C)

Temperatures were relatively consistent during each sampling trip, with a slight increase from Site A to B, and constant values from that point onwards.

Redox (Figure 4.3 D)

Except for August, Redox (Eh) values were fairly uniform at all sample sites on Bakers Creek. In August, decreased sharply downstream of Site B, then increased sharply after the waste rock dump. This increase may be due to the waters becoming more oxygenated from the high turbulence associated from flowing down the WRD in a waterfall. Eh decreases sharply after Site G, however this is most likely due to a redox probe malfunction rather than reduced conditions.

4.1.2.2 Elemental Concentrations

Sulphate

Sulphate reached a maximum concentration of 1440, at Site E (Figure 4.3 E). The guideline limit is 400 ppm but was only exceeded three times (at Sites C, E, and G). Sulphate is a particularly good indicator of AMD, as it is a by-product of acid generation, that is not reduced by subsequent acid neutralisation (Parr, 1997); i.e. it records the gross amount of acid produced.

Major Cations and Anions

This includes Ca^{2+} , Mg^{2+} , Na^+ , K^+ and Cl^- . Ca, and Mg (Figures 4.3 F, and 4.3 G respectively), remained at low concentrations until after the waste rock dump. Na^+ and Cl^- (Figures 4.3 H and 4.4 A) behaved quite differently and remained relatively constant for the length of Bakers Creek. There was a peak in the April concentrations at Site B which may have been to slightly evaporitic conditions, however there was no peak in the other salts. K^+ was below detection for most of the sampling.

Major Elements

The major elements were Mn, Al, Fe, and Si (Figures 4.4 B,C,D and E respectively). Except for Mn, the maximums for these elements were recorded at Site C, with Mn reaching its maximum at Site E. Mn was the most concentrated major element with 47.3 ppm.

Al

As the pH of Bakers Creek never reaches 6.5, the guideline limit for Al concentrations should not exceed 0.1 ppm. Even the baseline measurements for Al exceed this, with the maximum value of 14.8 at Site C. In acidic solutions aluminum is more bioavailable, and becomes toxic to fish, amphibians and phytoplankton when the guideline is exceeded within the 4.4-5.4 pH range. Unfortunately both of these conditions exist, which would help to explain the lack of aquatic life in Bakers Creek. Tadpoles are present at Site A, but there is no evidence of aquatic life downstream from this site.

Fe

Iron reached a maximum of 43.8ppm at Site C. Fe consistently exceeded the guideline limit of 1 ppm (for aquatic ecosystems) below the waste rock dump (Figure 4.4 C). Toxic effects rarely occur at concentrations below this limit, but if the ferrous iron concentration is above 0.5 ppm, floc formation may occur when the ferrous iron is oxidised (ANZECC/NWQMS, 1992). This may cause smothering effects for benthic organisms.

Trace Metals

These include As, Cd, Cu, Mo, Pb, and Zn (Figures 4.4 G, H, I, J, K, and L respectively). As, Cd, Cu, Pb, and Zn are the trace elements at Bakers Creek that are of most environmental concern in Bakers Creek as they do exceed the guidelines regularly, particularly at Site C.

As

Arsenic equaled the guideline limit of 0.05 ppm at Site C in the long term sampling, but exceeded the limit during the intensive sampling, reaching a maximum of 0.292 ppm. Though the concentrations were below the guidelines for the long term sampling, Figure 4.4 G did illustrate that As concentrations increase from Site F to Site G, which may mean that there is an As source between them (this is discussed further below). The toxicity of As is dependent on the redox potential, with the more reduced As III being much more toxic than As V. The species present in the Bakers Creek waters is investigated in Chapter 5.

Cd, Cu

Both of these elements exceeded the guidelines at the same locations. The maximum Cd value was 0.4 ppm at Site E (guideline limit of 0.002), with Cu reaching a maximum at Site F of 1.51 ppm (guideline limit of 0.005). In August these elements increased dramatically at Site F, suggesting that the waste rock dump is the major source of these contaminants.

Pb

Lead reached a maximum of 2.4 at Site G which drastically exceeded the guideline limit of 0.005 ppm, however the intensive sampling produced even higher concentrations up to 19.2 ppm. Lead exceeded the limit at every site downstream of Site A.

Zn

Zn was by far the most abundant of element measured, reaching 82.2 ppm at Site F in August. Like lead it exceeded the limit at every site downstream of Site A, but even the baseline results contained concentrations above the guideline limit of 0.05 ppm.

4.1.2.3 Spatial and Temporal Variations in Water Chemistry

The degradation of water quality along Bakers Creek can be summarised as follows. Water samples from Site A consistently exceed the guidelines for Al (and on one occasion Cd, Pb, and Zn). By the time waters have reached Site D, they now contain excessive Al, Cd, Cu, Pb and Zn. Waters discharging from Site C also exceed the guidelines for As and Fe, but these elements decrease via dilution to acceptable levels to Site D. After passing through the waste rock dump, the dissolved Fe content of Bakers Creek exceeds the

guidelines listed in Table 4.1. Downstream of the Bakers Creek waste rock dump, dilution slightly reduces trace metal concentrations, but concentrations are adversely affected in Ring River downstream of the confluence with Bakers Creek (Figures 4.3, 4.4 and Appendix 3).

There are three recurring spatial patterns observed from the results in Figure 4.3 and 4.4:

1. Al, Cd, Cu, Mg, S, SO_4^{2-} , Si, Zn have low concentrations up to Site D (at the top of the waste rock dump) followed by a dramatic increase at Site F, then gradual dilution to Site I.
2. As, Ca, Fe, Mn, and Pb have low concentrations up to Site D followed by a dramatic increase at Site F, a continued increase at Site G and dilution to Site I.
3. Cl^- , Na have constant concentrations for the entire length of Bakers Creek.

In the first and second cases, the concentration of the various elements increases sharply after passing through the waste rock dump. In the third cases, Cl and Na are not influenced by reaction with the waste rock dump. There appears to be a further source of contamination for the second group of elements between Sites F and G. The bulk of the waste rock dump extends to Site F, however with time waste rock has been washed down to at least site G, which could at least partly explain this additional source of contamination (Plate 4.1 A, C and D). Another source could be from two adits located about 20-30m upstream of Site G. One adit is to the side of Bakers Creek, and the second is cut into the face of a small waterfall and drains directly into Bakers Creek (Plate 4.1 E).

Na^+ and Cl^- concentrations are independent of sample site, and are therefore not sourced from the mine working or waste rock dump. They are probably derived from soils in the surrounding catchment.

Drainage Composition Along Bakers Creek

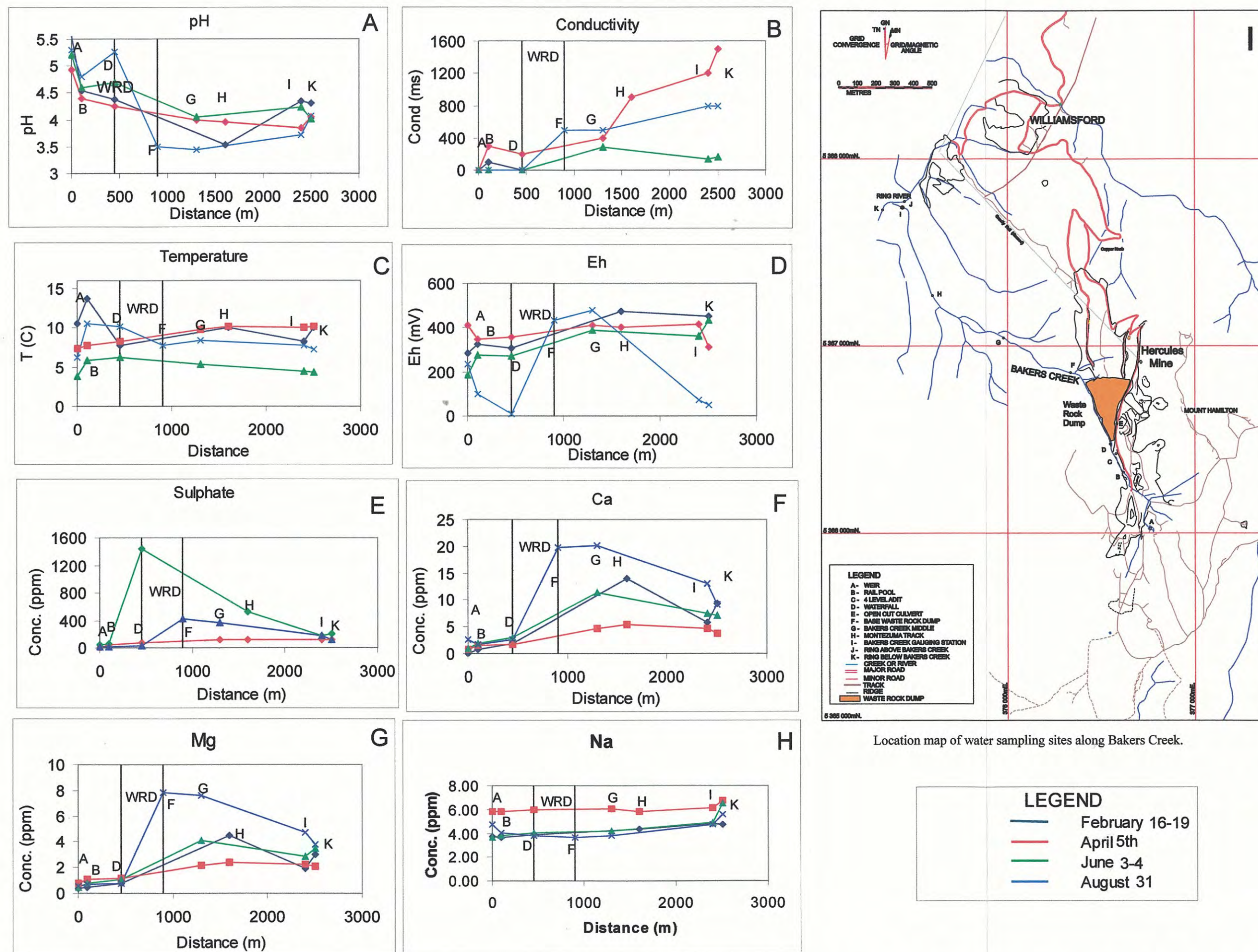


Figure 4.3 Drainage compositions as a function of distance along Bakers Creek. A : pH; B: conductivity; C: temperature; D: redox; E: Sulfate; F: Ca; G: Mg; H: Na; I: Locality Map. 43

Drainage Composition Along Bakers Creek (cont.)

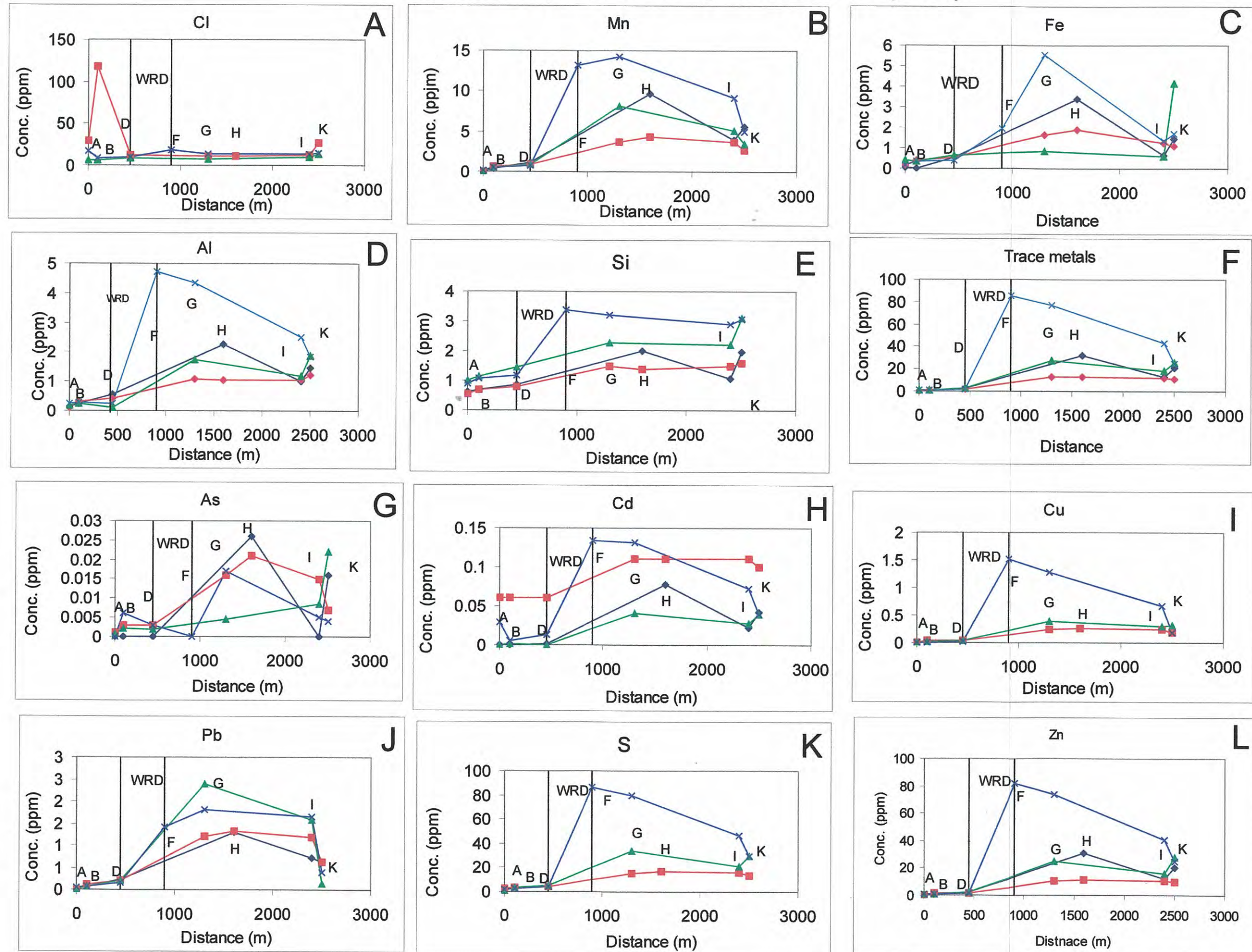
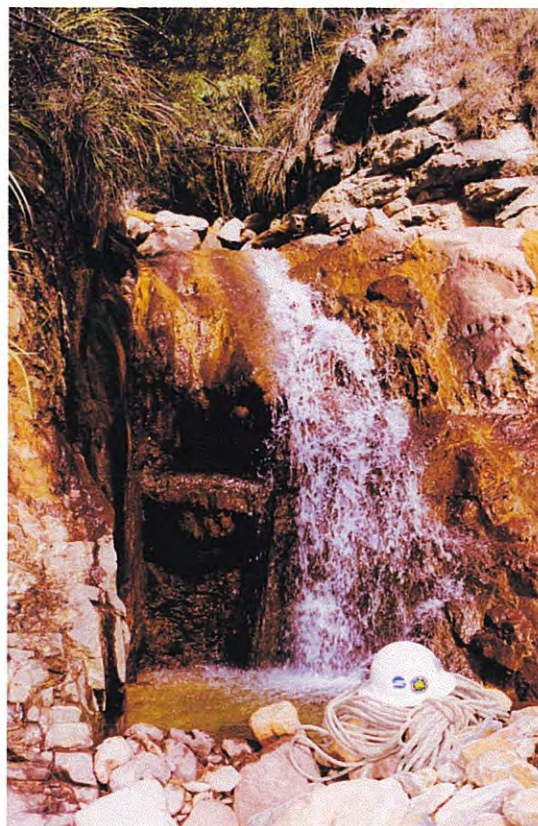


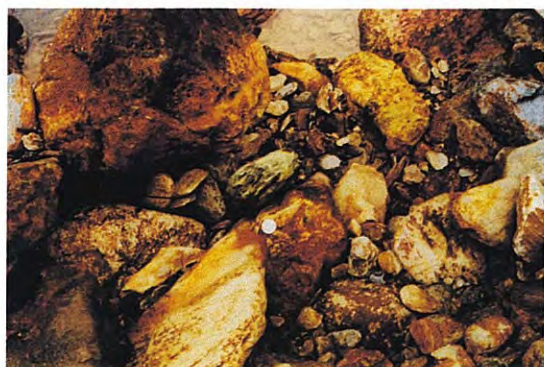
Figure 4.4 Drainage compositions along Bakers Creek A: Chloride; B: Mn; C: Fe; D: Al; E: Si; F: Trace metals; G: As; H: Cd; I: Cu; J: Pb; K: S; and L: Zn.



A



B



C



D

Plates 4.1 (A,B, C and D) (A) Bakers Creek (Site G) with Mt Hamilton in the distance. Note the amount of waste rock that has washed down from the waste rock dump upstream. (B) is the adit just upstream of Site G that was mined directly in Bakers Creek (C) and (D) are typical examples of the sulphidic waste rock at Site G.

There are temporal variations in the concentrations of individual elements. The rainfall (as measured at Mt Read by the Bureau of Meteorology) for each sampling trip is given in Figure 4.5 and in Appendix 5. February was the wettest sampling trip followed by April, June and August. In the first three sampling trips, most of the concentration data remains consistent for each element. There is a striking comparison between these samples and those collected in August. There is a significant increase in the concentration of most elements downstream of the waste rock dump in August, when conditions were relatively dry. Rain leads to dilution, while dry periods concentrate the water through evaporation, which explains the higher concentrations in August relative to the other months. June was also dry, however there was significant rain in the days leading up to the June sampling trip, as opposed to August, which had been relatively dry for weeks.

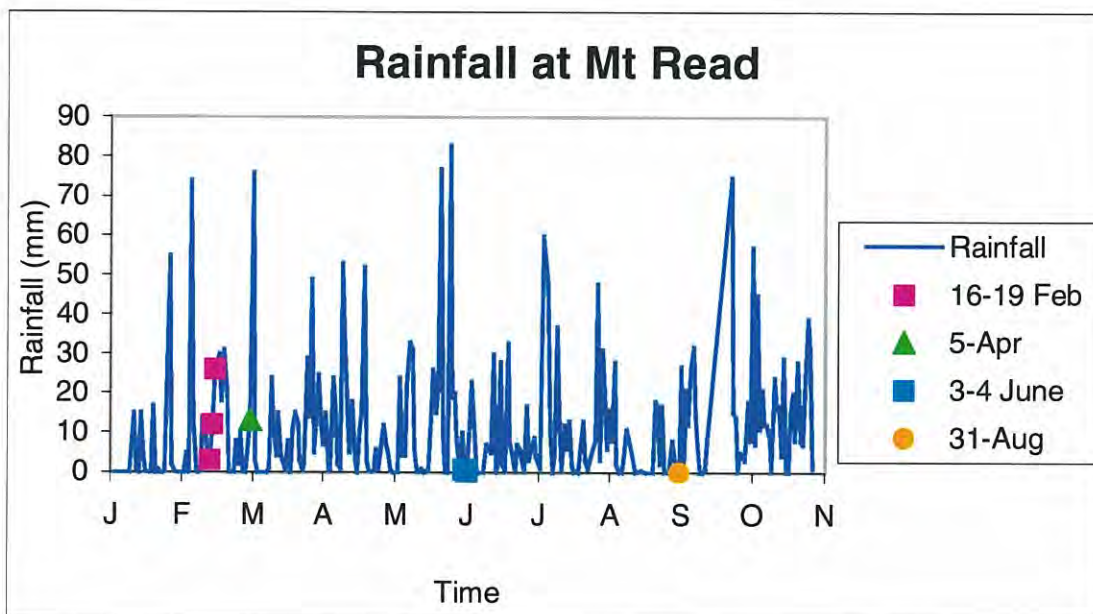


Figure 4.5 Rainfall at Mt Read during long-term sampling.

4.1.3 Intensive Sampling Results

Two intensive sampling programs were conducted, in order to determine whether storms created contaminant plumes in Bakers Creek, by flushing residual oxidation products that had built up via sulphide oxidation of the waste rock dump during dry spells. The results for the two intensive sampling trips are illustrated in Figures 4.6 and 4.7, with the maximum concentrations of specific elements listed in Table 4.3.

Intensive Sampling Results

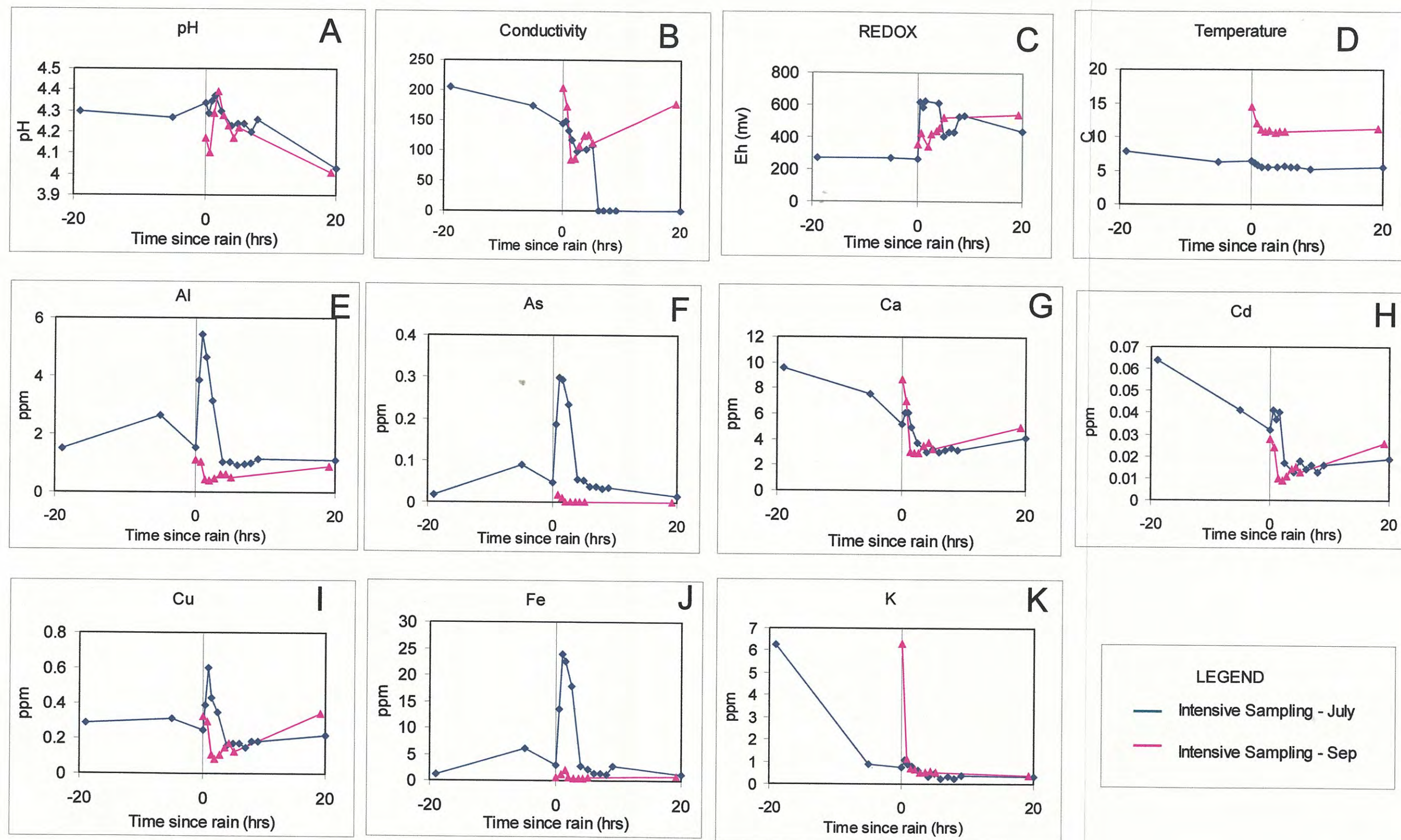


Figure 4.5 Intensive sampling results, A: pH; B: Conductivity; C: Redox; D: Temperature; E: Al, F: As, G: Ca, H: Cd, I: Cu, J: Fe, and K: K.

Intensive Sampling Results Continued

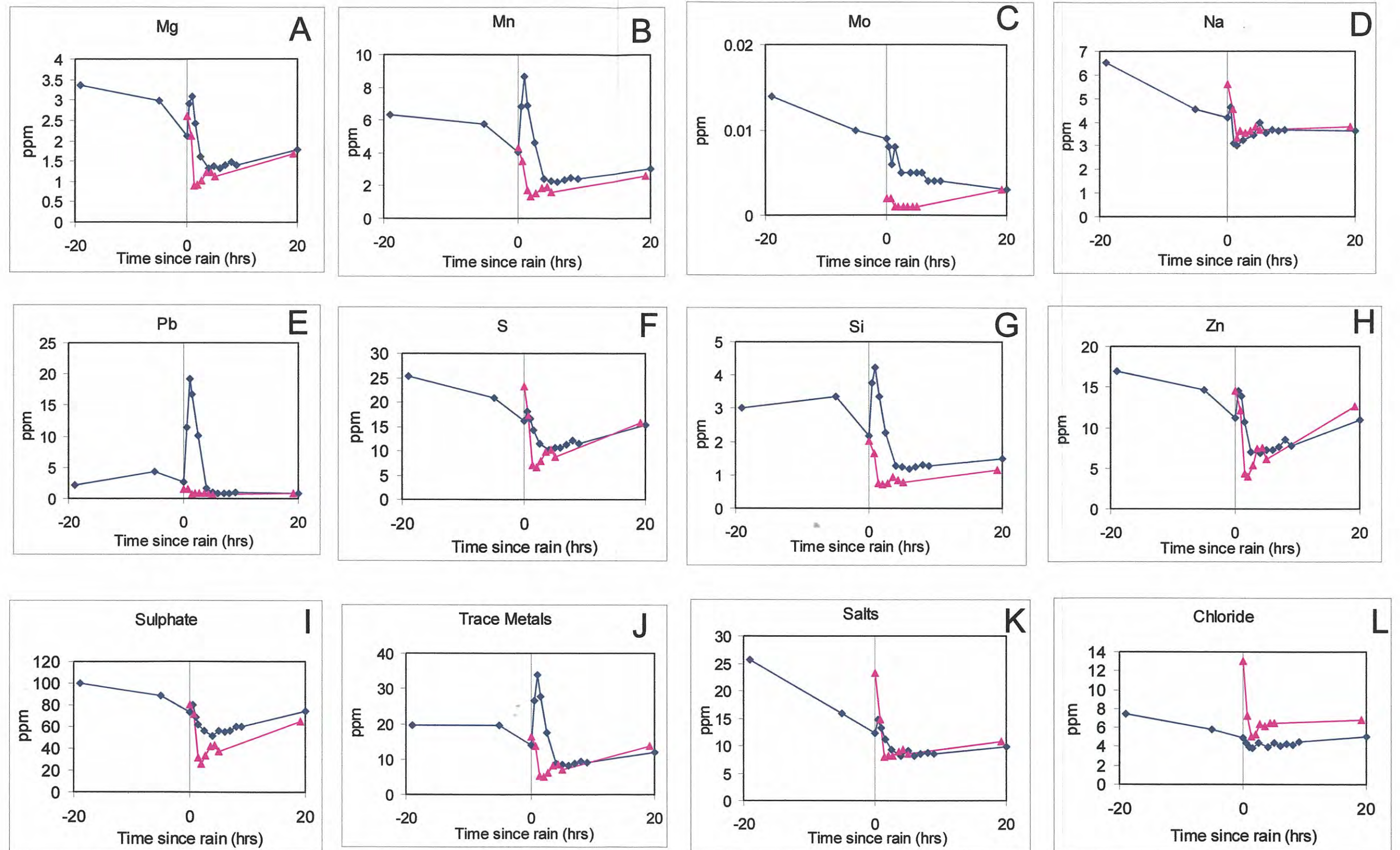


Figure 4.6 Intensive sampling results, concentration with hours since rain; A: Mg, B: Mn, C: Mo, D: Na, E: Pb, F: S, G: Si, H: Zn, I: Sulphate, J: Trace metals, K: Salts, and L: Chloride.

Parameter	Maximum	Minimum
pH	4.37	3.030
Conductivity (μ s)	206.00	0.000
Temperature (C)	14.40	5.300
Sulphate	100.00	26.000
Chloride	13.00	3.800
Al	5.44	0.380
As	0.29	<0.001
Cd	0.06	0.009
Cu	0.60	0.083
Fe	24.00	0.390
K	6.31	0.330
Mg	3.37	0.900
Mn	8.64	1.300
Mo	0.01	0.001
Na	6.53	3.030
Pb	19.20	0.670
S	25.40	6.510
Si	4.21	0.700
Zn	17.00	3.99

Table 4.3 Maximum values of physiochemical (except for pH where the minimum value is given as it is the most contaminated) and element concentrations (all units are in ppm unless otherwise stated).

Figure 4.6 and 4.7 demonstrate that contaminant plumes discharge from the Bakers Creek waste rock dump during storm events. In July, a significant spike in metal concentrations was detected 30 minutes after the onset of rain. After the initial spike, dilution occurred with continued rainfall. The concentrations of most elements rebounded after about 24 hours. Pb, Zn, Fe and Mn were the most significant contributors to the contaminant plume. As, Al, Cd, Cu, Pb, and Zn (Figure 4.6 F, E, I, and Figure 4.7 E and H, respectively), exceeded the guidelines at some stage during both intensive sampling periods. In July every sample exceeded the Al, Cd, Cu, Fe, Pb and Zn guidelines, and some of the early samples exceeded As. Concentrations were significantly lower in the September sampling, but for Cd, Cu, Pb and Zn still exceeded the guideline concentrations for every sample, and Al, and Fe were exceeded directly after rainfall. The maximum concentrations for As and Pb were higher during storm event than those recorded during the long term sampling program. This occurred despite the high flow rates during the storm events, where one may expect dilution to cause metal concentrations to decrease.

In September, there was no obvious spike in metal concentrations with the onset of rain. Figure 4.8 shows the flow rates measured at Site I (and the times at which intensive sampling was undertaken). In the days leading up to the September intensive sampling, you can see that it rained several times. Consequently dilution for the several periods of moderate rainfall in the days leading up to the storm event probably prevented accumulation.

Figures 4.6 C and illustrates how redox increased rapidly with the onset of rainfall, as the oxygenated rainwater reached the base of Bakers Creek. Conductivity decreased as metal concentrations become more diluted by continued rainfall (Figure 4.6 B). pH values are inversely proportional to Eh, (i.e. when pH increased redox decreased, and when redox increased pH decreased; Figure 4.6 A). The pH continued to decrease well after the storm had passed in both July and September (Figure 4.6 A).

4.2 MASS LOADING CALCULATIONS

If flow rates are measured, mass loadings can be calculated to determine the mass of metals carried in a stream over a given time period. Mass loadings are calculated by combining measured concentrations of specific metals obtained at a given site and the corresponding flow rate. This is explained in the following equation:

$$ML = Q \times C$$

Where: ML = Mass Load (mg / time unit)

Q = Flow (L / time)

C = Concentration (mg / L)

The final mass loads can be converted to units of kg / day or T / yr.

Mass loadings can be used to calculate the influence of seasonal changes on the degree of metal contamination. As mass loadings are partly dependent on flow rates, variations in this parameter (particularly rapid changes associated with storm events) may lead to significant variations in metal loads.

Mass loadings were calculated for both long term (low flow), and storm flow events to quantify and compare how they vary with flow rates, and to calculate the contribution of metals from the Bakers Creek waste rock dump, to the overall metal budget of Bakers Creek. These results are listed in Appendix 6. Those samples which were used to calculate mass loadings are illustrated in Figure 4.8.

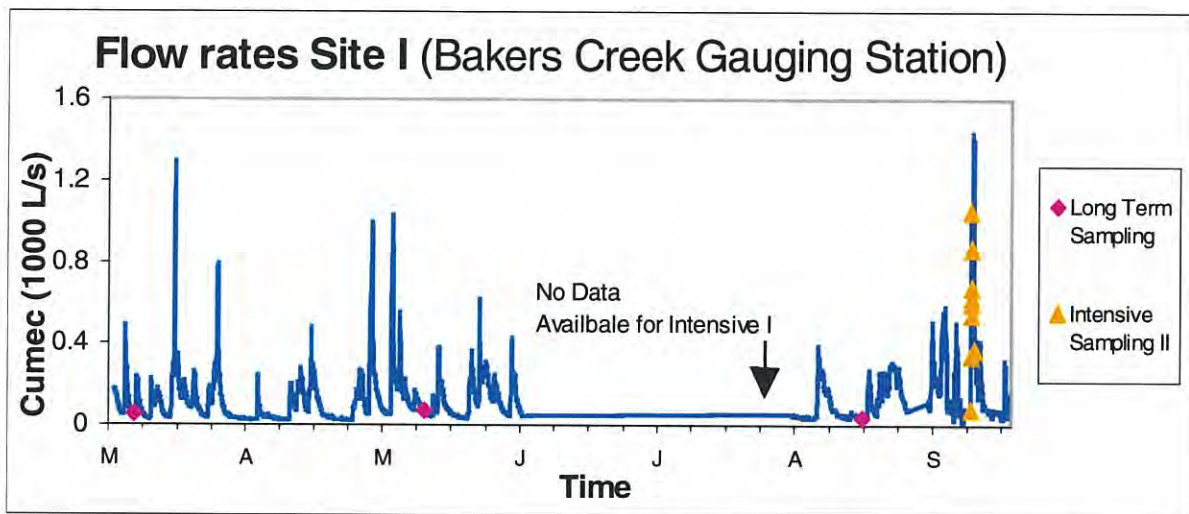


Figure 4.8 Flow of Bakers Creek at Site I (as measured by the monitoring station), and the times at which the samples used to calculate mass loads were taken.

The flow data was obtained from the monitoring station at Site I (which is operated by the Hydro Electric Commission). Due to equipment malfunction, flow rates were not measured in July which unfortunately coincided with the first intensive sampling program (27th to 29th July). Figure 4.9 illustrates the close correlation between rainfall at Mt Read and the Flow rate at Site I. This correlation is close enough to give a qualitative indication of the flow rate during the first intensive sampling program. The recorded rainfall on Mt Read during the first intensive sampling run was about as high as that measured in September during the second intensive sampling program. The flow rate can therefore be assumed to have been relatively high in July, although rainfall intensity is an important variable that may discredit this assumption.

4.2.1 Low Flow

The average values for the mass loadings calculated for Site I during the low flow (long term) sampling runs, are listed in Table 4.4 for selected elements.

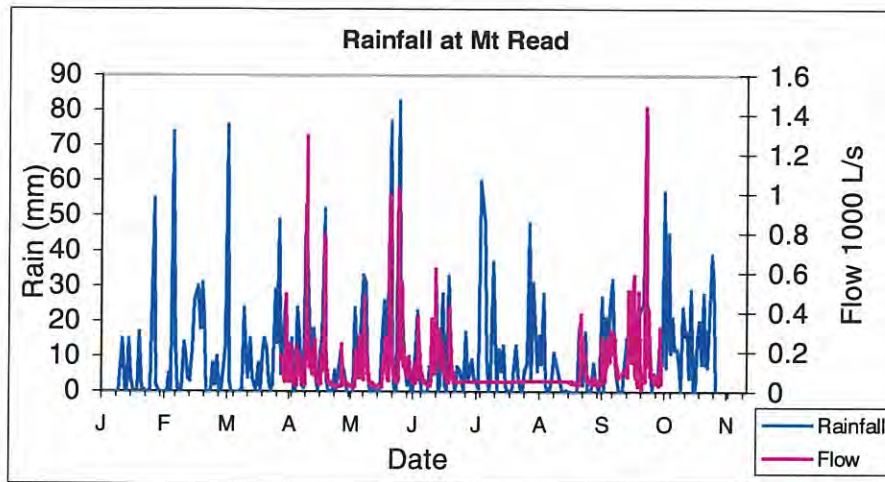


Figure 4.9 Flow at Site I and Rainfall measured at Mt Read

Element	ML (kg/day)	ML (T/yr)
Al	9.45	3.45
Cd	0.59	0.22
Cu	2.29	0.84
Fe	7.78	2.84
Mn	35.48	12.96
Pb	9.90	3.62
Zn	121.08	44.22

Table 4.4 Average calculated mass loadings for background low flow rates at Site I (Bakers Creek Gauging Station). ML = Mass Load.

In addition to flow data from Site I, in August, the flow of Bakers Creek was measured where it passes through pipes between Sites A and B, where an average flow of 5.23 L s^{-1} was recorded (using the jug and stopwatch technique described in Chapter 3). On this sampling trip, weather conditions were extremely dry. As a result there was no surface drainage emanating from point source contributors (i.e. the adit or the nearby open cut workings found upstream of the WRD). The flow rate could not be accurately measured directly above the WRD at Site D (as discussed in Chapter 3), so it is assumed that the variation in flow between Sites A and B and Site D was negligible. Thus by using the rate of 5.23 L s^{-1} for the flow and the concentration of the waters at Site D, an estimate of the mass loadings of metals directly above the WRD was obtained (Table 4.5).

Element	Site A (No contamination)		Site D (Above WRD)		Site I (Base of Bakers Creek)	
	ML kg/day	ML T/yr	ML kg/day	ML T/yr	ML kg/day	ML T/yr
Al	0.113	0.041	0.117	0.043	7.97	2.911
Cd	0.013	0.005	0.006	0.002	0.23	0.084
Cu	0.003	0.001	0.007	0.002	2.11	0.772
Fe	0.072	0.026	0.176	0.064	4.38	1.601
Mn	0.054	0.020	0.393	0.143	29.44	10.754
Pb	0.018	0.007	0.071	0.026	5.28	1.929
Zn	0.235	0.086	2.949	1.076	130.26	47.576
SO ₄ ²⁻	3.930	1.430	10.840	3.960	556.87	203.400

Table 4.5 Mass loadings at Sites A, D and I during low flow conditions on 31st August.

The contribution from the waste rock dump to trace metal loads in Bakers Creek can be calculated by subtracting the Site D mass load from the total mass load calculated for water from Site I (Table 4.6).

Element	ML kg/day	ML T/yr	% Metal Contribution to Bakers Creek (from WRD)
Al	7.85	2.87	98.6
Cd	0.22	0.08	95.2
Cu	2.10	0.77	99.7
Fe	4.20	1.54	96.7
Mn	29.05	10.61	98.7
Pb	5.21	1.90	98.5
Zn	127.32	46.50	97.7
SO ₄ ²⁻	546.03	199.44	97.6

Table 4.6 Mass load contribution from the waste rock dump (WRD) during low flow conditions on 31st of August.

These values are only estimates, as the flow could not be measured directly above and below the WRD. Dilution occurs between Sites F and I (Figures 4.4, and 4.5), which will influence the mass loadings calculated at Site I. Nonetheless, it is apparent that the waste rock dump contributed most of the trace metals (>95%) during the low flow event in August.

4.2.2 Storm Flow

Samples were obtained from 6:20pm on the 22nd of September at the onset of rain to 1:30pm the following day after the rain had ceased. The times at which these samples were taken relative to the flow of Bakers Creek (at Site I) is shown in Figure 4.10. The

rainfall was sporadic and stopped at round 8pm, but recommenced a few hours later at around 10pm. Unfortunately no samples were gathered between the hours of 2am and 7am (where the majority of rain occurred), but the final sample was taken after the storm had passed).

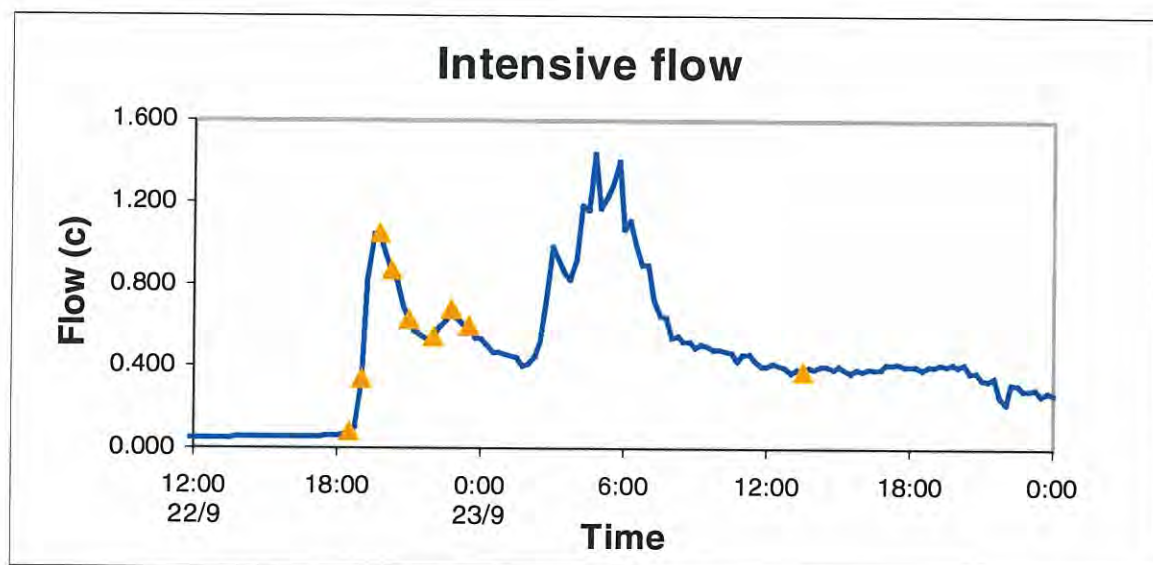


Figure 4.10 Time of intensive sampling (Sep) relative to the flow rate at Site I

The change in calculated mass loadings and metal concentrations with time is illustrated in Figure 4.11.

The mass loading values peak at the same time as maximum flow rates are recorded, rather than maximum concentration values (Figure 4.12). Increased flow by itself causes dilution. Therefore, in order to increase the mass loads at increased flow rates, more metal is being released during storm flows (so concentrations are higher than they would be if only dilution was occurring). The maximum values reached during the intensive sampling are listed in Table 4.7.

Element	ML kg/day	ML T/yr
Al	37.98	13.87
Cd	0.90	0.33
Cu	10.80	3.95
Fe	179.10	65.40
Mn	153.74	56.15
Pb	65.11	23.78
Zn	435.00	158.89
SO ₄ ²⁻	2065.96	754.59

Table 4.7 Mass Load contribution from the WRD on 22nd September (storm flow).

Mass Loading and Concentrations of Metals - Intensive Sampling (Sep)

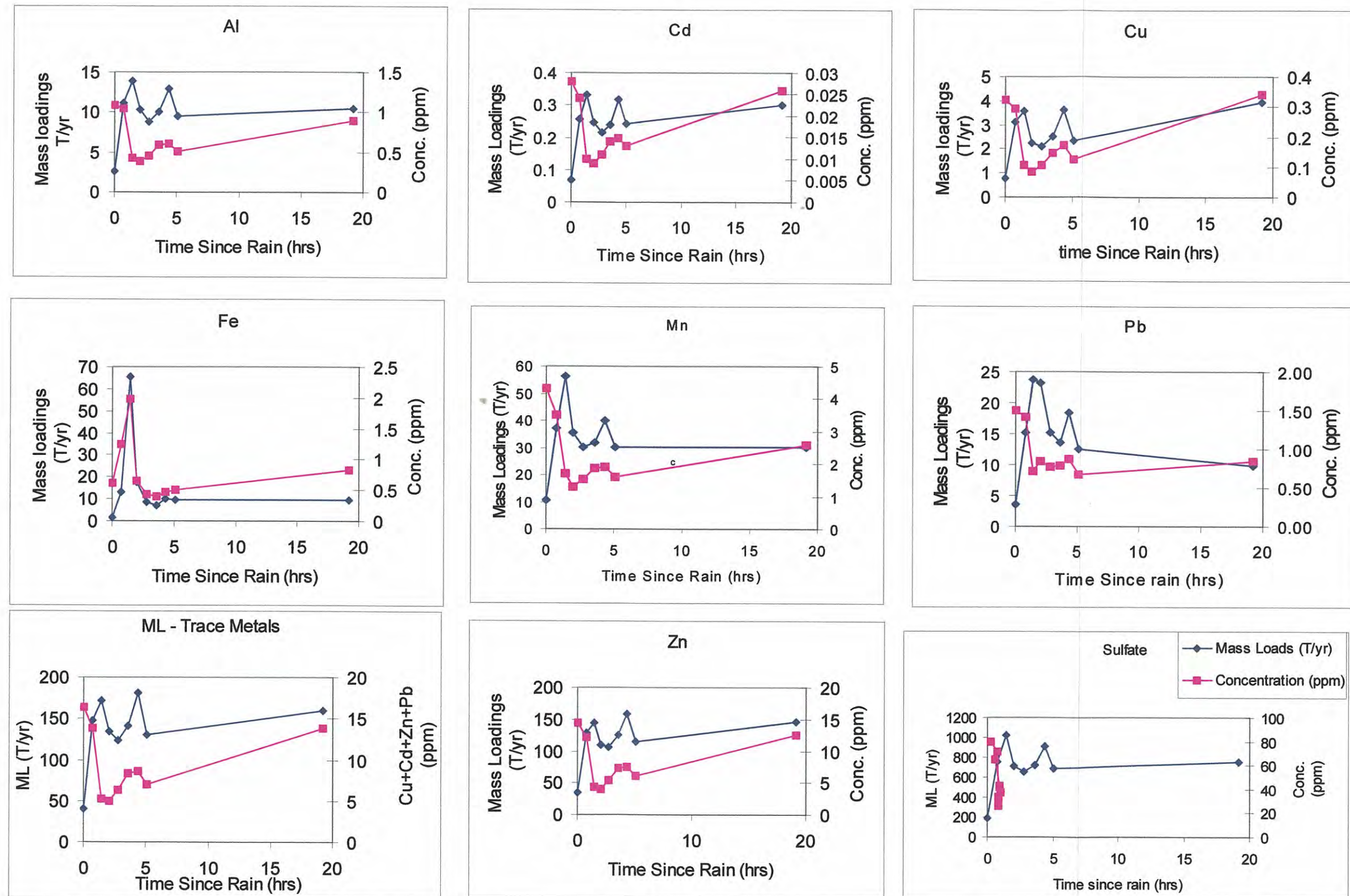


Figure 4.11 Mass loads and concentrations of metals during the September intensive sampling.

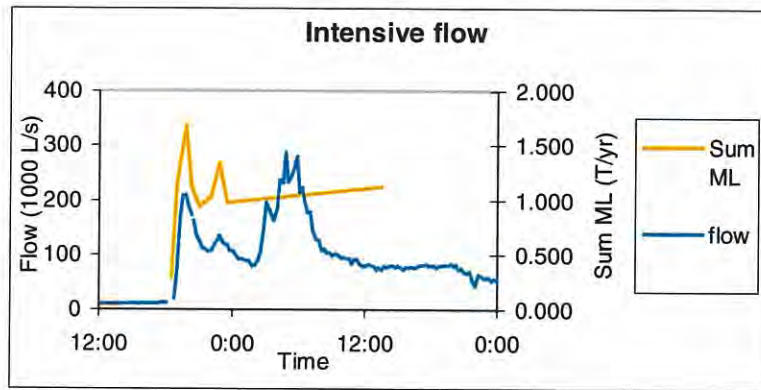


Figure 4.12 The sum of intensive metal mass loads plotted with the flow rate

Calculated peak mass loads are substantially higher than those determined for the low flow regime. The peak mass loads calculated for the intensive sampling are one order of magnitude higher than the low flow results. This implies that contaminants from the waste rock dump are flushed out in greater amounts during storm flow events, which has important implications for remediation options (Chapter 6).

It is interesting to note dilution occurs from the onset of rainfall in the samples taken in September, whereas the July intensive sampling recorded an initial spike in metal concentrations, followed by dilution. The concentrations in this spike were significantly higher than those seen in the September results, and appear to have occurred around the time of peak flow during the storm event, (based on the high rainfall at this time; Appendix 6). The combination of peak concentrations and high flow rates would have definitely resulted in high mass loadings. Rainfall reached 100mm on September (22nd-23rd) and 62mm in July (27th-28th), thus the peak flow rates would be expected to be lower in July. However, the concentrations were substantially higher in July, therefore the mass loads could still have been as high or possibly higher than those recorded in September.

4.3 BACTERIA

4.3.1 Introduction

Bacteria have an important role in controlling the rate of acid production, and are capable of catalysing the rate of reaction up to a factor of 10^6 (Chapter 2). Therefore an attempt was made to identify whether sulphur-oxidising bacteria were present in Bakers Creek. Sediment samples were collected from Sites C and F, (with the methods described in Chapter 3), and Leah Hawkes (from University of Tasmania - Agricultural Sciences), performed the analyses.

4.3.2 Results

From the two samples collected no bacteria were identified. Three mediums were used, the first to grow *Thiobacillus ferrooxidans*, and the other two for any of the *Thiobacillus* genus (these methods are described in Chapter 3). After four weeks, the only growth which occurred was of yeasts, (probably unicellular fungi known as ascomycetes; L. Hawkes, per. com. 1998).

There are several reasons which could explain the growth of yeasts rather than *Thiobacillus* bacteria (L. Hawkes, pers. com, 1998):

- Fungi and yeasts are generally more robust to adverse conditions than bacteria;
- The fungi may have occurred from sample contamination and may not have come from the original sampling site, (as fungi are good scavengers capable of surviving on minimal media).
- The site has minimal sediment and is subjected to rapid flows, thus any bacteria present in the waste rock dump may not have been capable of attaching to the sediments in the areas where the samples were taken.
- *T.ferrooxidans* is inhibited by agar, so the filtration method used to combat this (which is a complicated technique), may not have been effective, and ultimately a possible source of sample contamination.

It is possible that there may be other bacterial types present (or fungi), but the media used were selective in that the only energy sources available were Fe, S and thiosulphate. As *Thiobacilli* are chemolithotrophic, they need inorganic compounds to obtain energy for growth. These media contained a very small amount of organic material (yeast extract, which was added for vitamin value), thus any organic-degrading bacteria, present in the samples would have been incapable of growing on the plates.

4.4 SECONDARY PRECIPITATES

Precipitates were collected at Sites F and C (Catalogue No. 137944 and 137945 respectively - Appendix 10; Plate 4.2), with high magnification images (Plate 4.3), and broad spectral analyses (Figures 4.13 and 4.14) were obtained from the ESEM.

4.4.1 Site F (Base of the WRD)

Results for the spectral analyses of grab samples of secondary precipitates from Site F are illustrated in Figure 4.13.

Two bulk samples were measured at low magnification, with peaks occurring in O, Al, Si, S, K, Ba, and Fe (Figure G and H). Higher magnification made it possible to analyse individual crystals, with chlorite (Figure 4.13 E), Fe-rich white micas (Figure 4.13, A, B), Mg-Fe rich micas (Figure 4.13 F) and pyrite (Plate 4.3 A) detected. Unknown minerals with high Fe but very low O contents were also detected which was quite distinct to the remaining O-rich Fe-bearing samples.

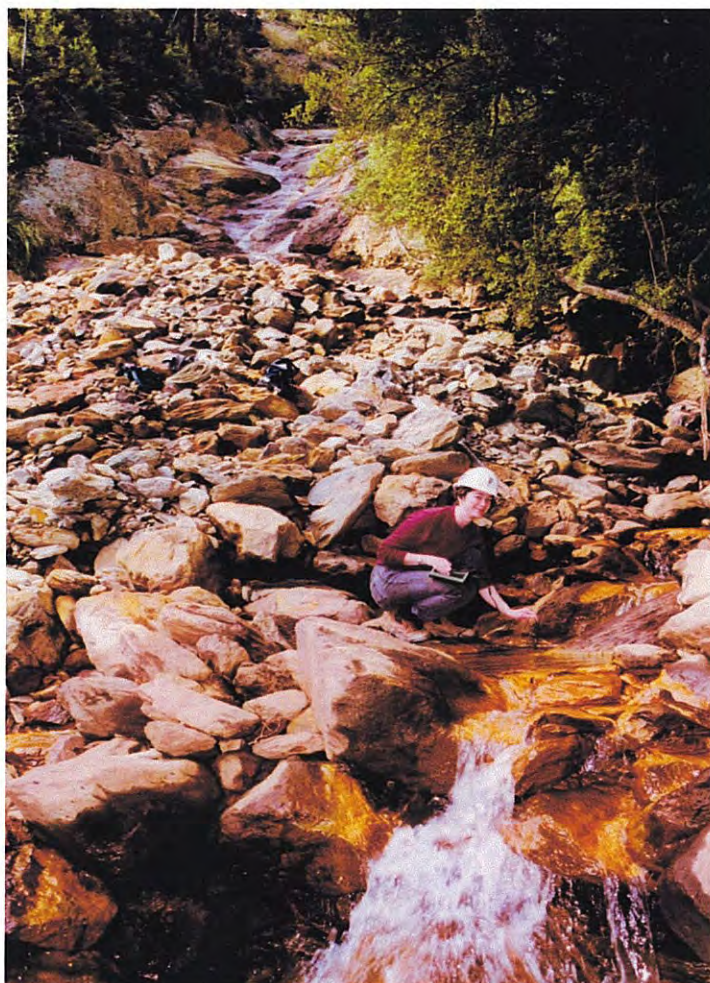
4.4.2 Site C (Adit)

Plate 4.3 B-F are images of secondary precipitates from the Acid at Site C. The precipitates were extremely fine grained ($<10\mu\text{m}$), but some aggregates of platey secondary precipitates were imaged using the ESEM. The spectra for the Adit samples (Figure 4.16) were similar to the samples from Site F, with recognition of high Fe and low oxygen spectra (Figure 4.16 A and B), Fe-white micas (Figure 4.16 D) and chlorite (Figure 4.16 C). One unknown mineral or group of minerals contained high Pb and Fe, and had low Si, and S (Figure 4.16 E).

Because secondary Fe-rich precipitates in Bakers Creek are extremely fine grained ($<10\text{m}$), nearby mineral grains can interfere with spectral readings. For a quantitative determination of the mineralogies present, XRD analyses are recommended.

4.5 SUMMARY

Bakers Creek is affected by acid drainage sourced primarily from the waste rock dump, with smaller inputs from the Adit at site C and drainage from the open cuts upstream of Site B. Acidity ranges from 3.03 to 5.55, with maximum metal concentrations of 0.2 ppm As, 0.4ppm Cd, 1.15ppm Cu, 43.8ppm Fe, 19.2 ppm Pb, and 82.2ppm Zn. The majority of the water in Bakers Creek are classified as Acid, High Metal waters using the scheme of Ficklin *et al* (1992; Figure 4.1).



A

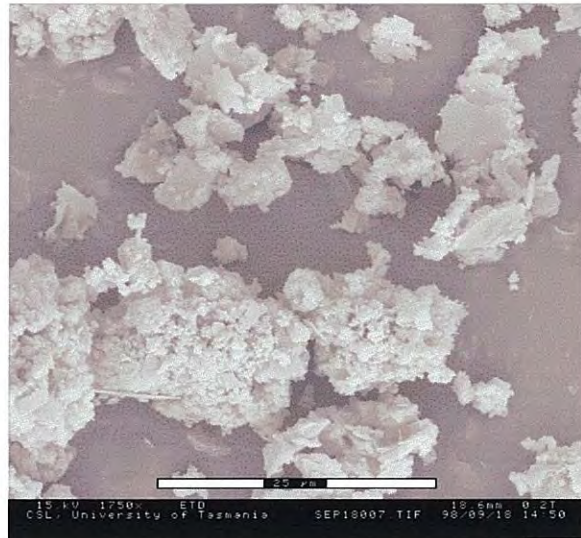


B

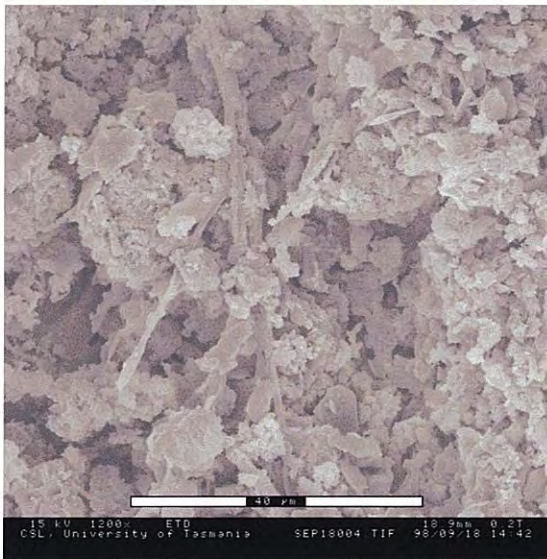
Plate 4.2 (A) Precipitates located at the base of the waste rock dump at Site F ; (B) Precipitates deposited outside the mouth of the Adit at Site C. The drainage from the adit flows directly into Bakers Creek.



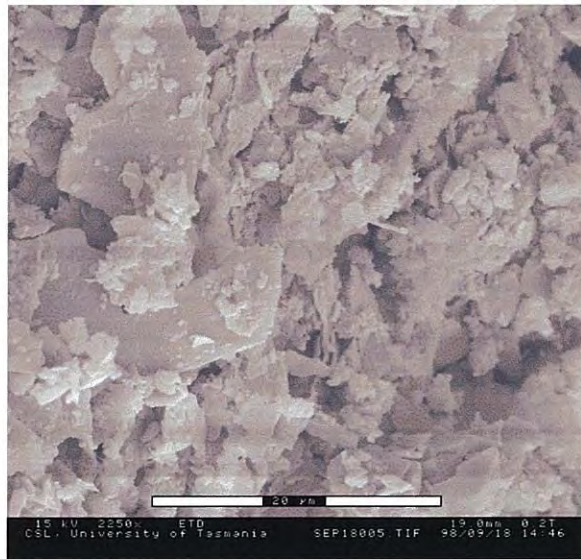
A



B



C



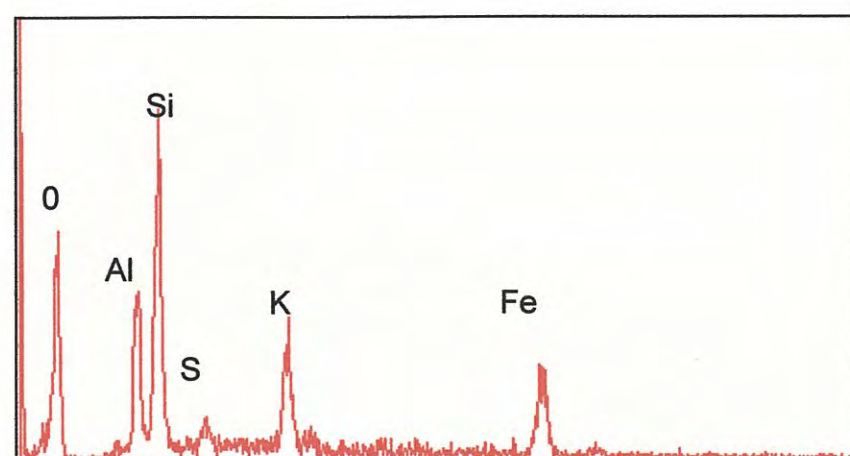
D



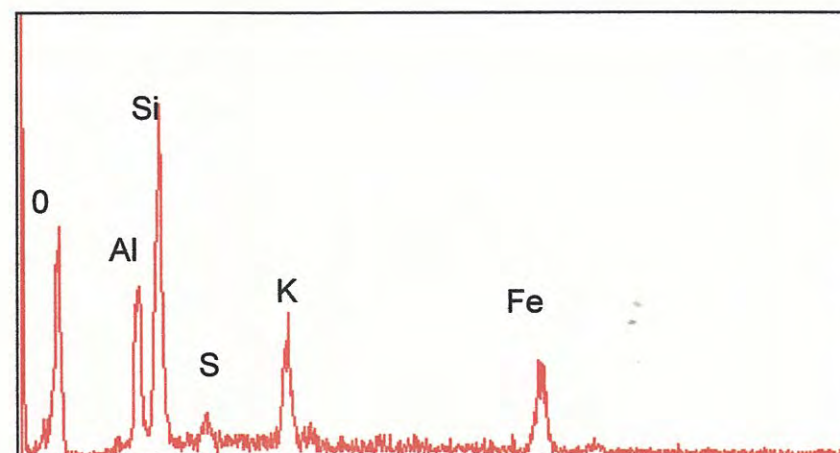
E

Plate 4.3 (a) Euhedral pyrite crystal from Site F. (B - E) are of the various fine grained platey aggregates found at Site C.

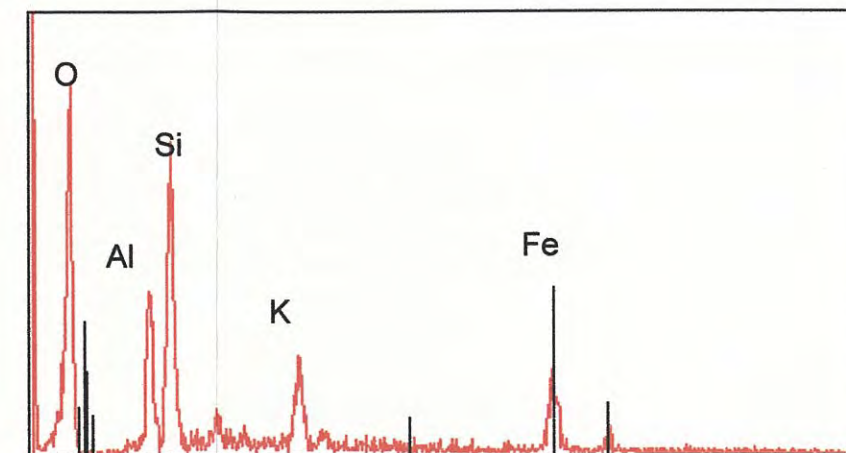
SEM Spectral Analyses of Precipitates - Site F(Aug)



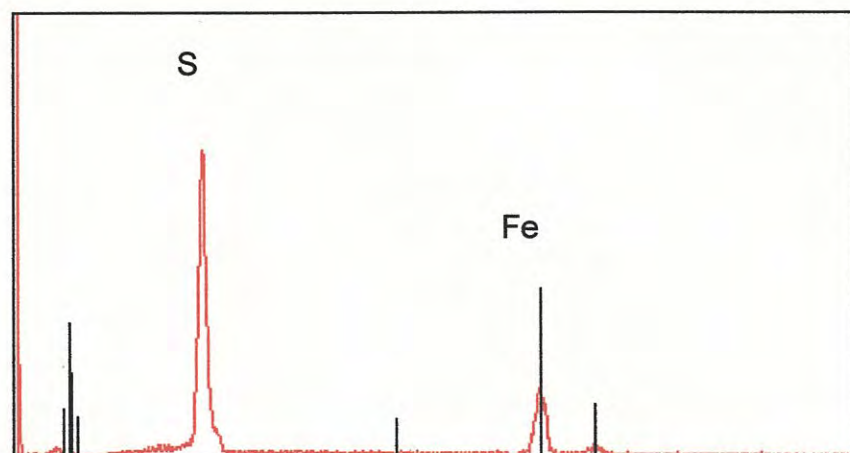
A: Fe rich white mica



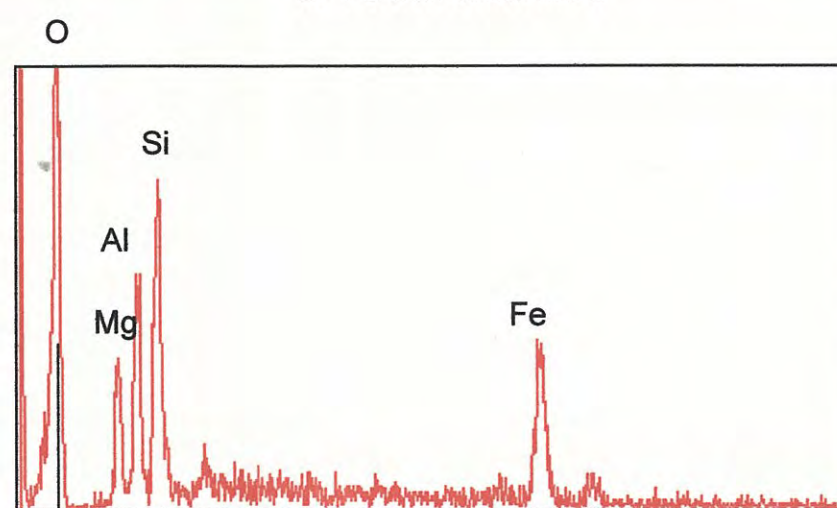
B: Fe-rich white mica



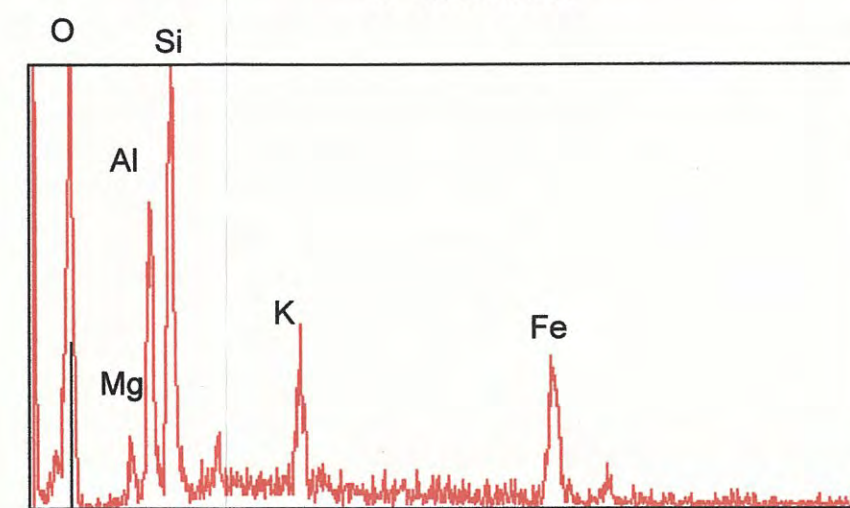
C: White mica



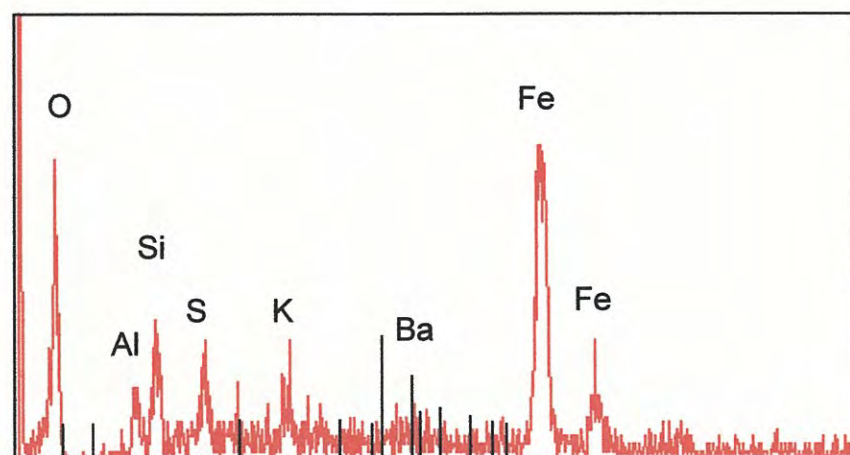
D: Pyrite



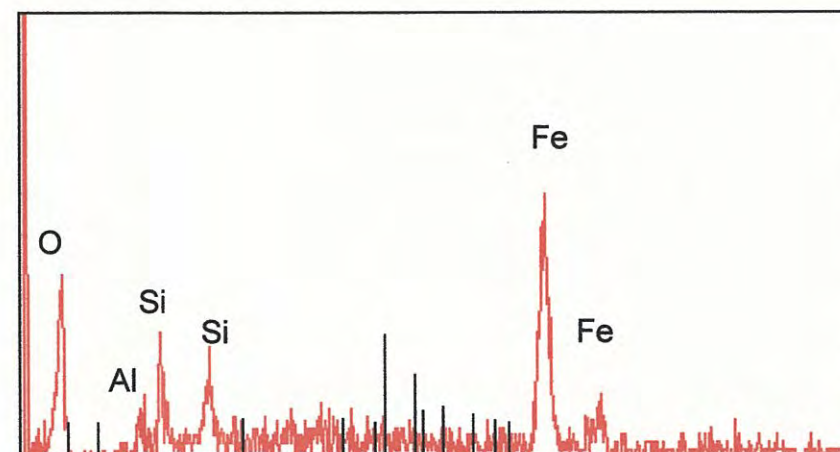
E: Chlorite



F: Mg-Fe rich mica



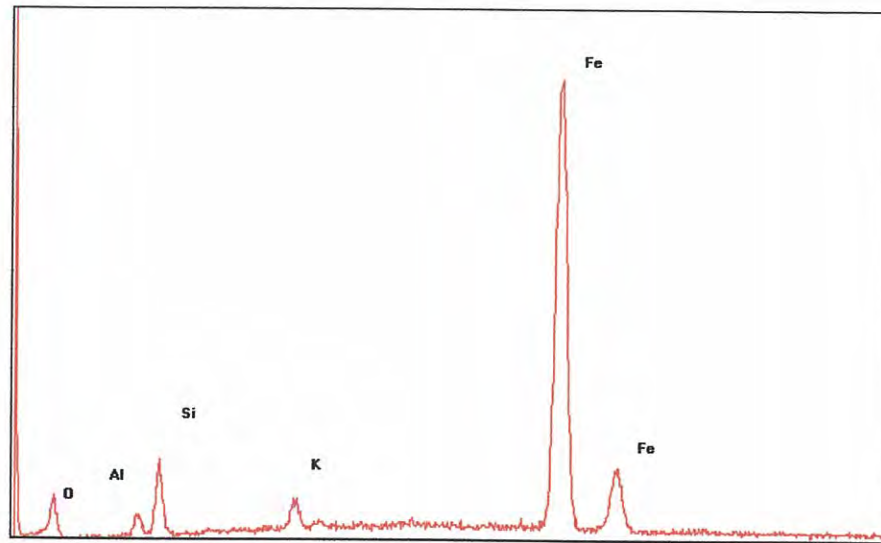
G: Bulk sample 1



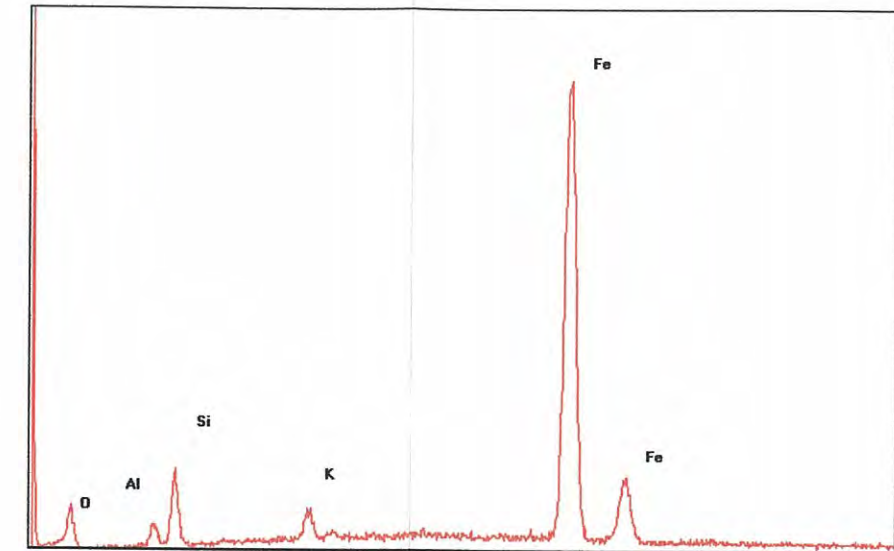
H: Bulk Sample 2

Figure 4.12 Spectral analyses of precipitates and interpreted mineralogy at Site F in August

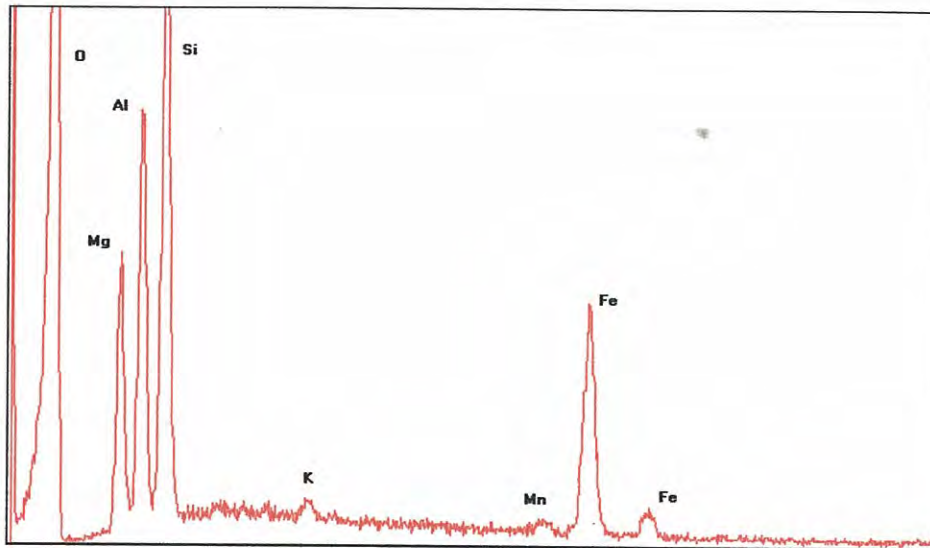
SEM Spectral Analyses of Precipitates - Site C(July)



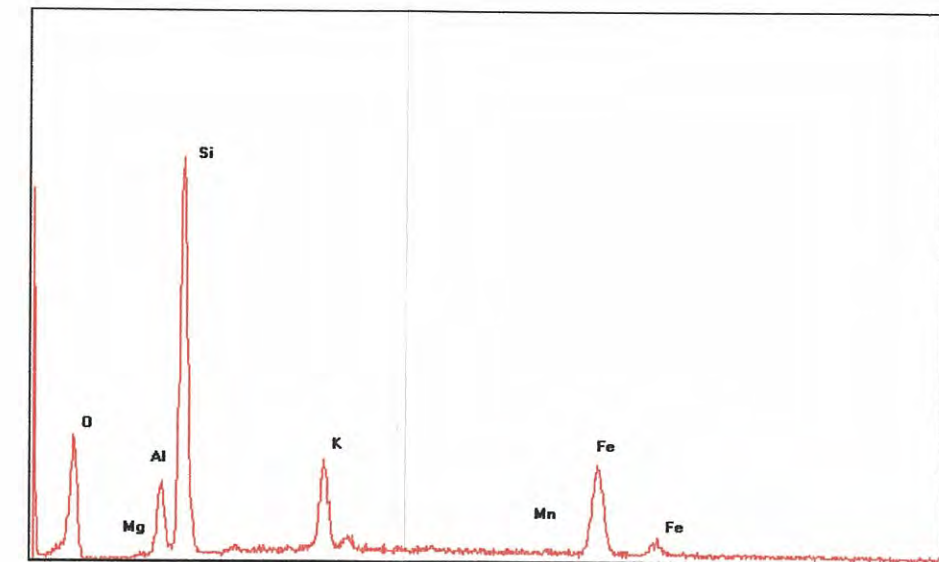
A: Hematite



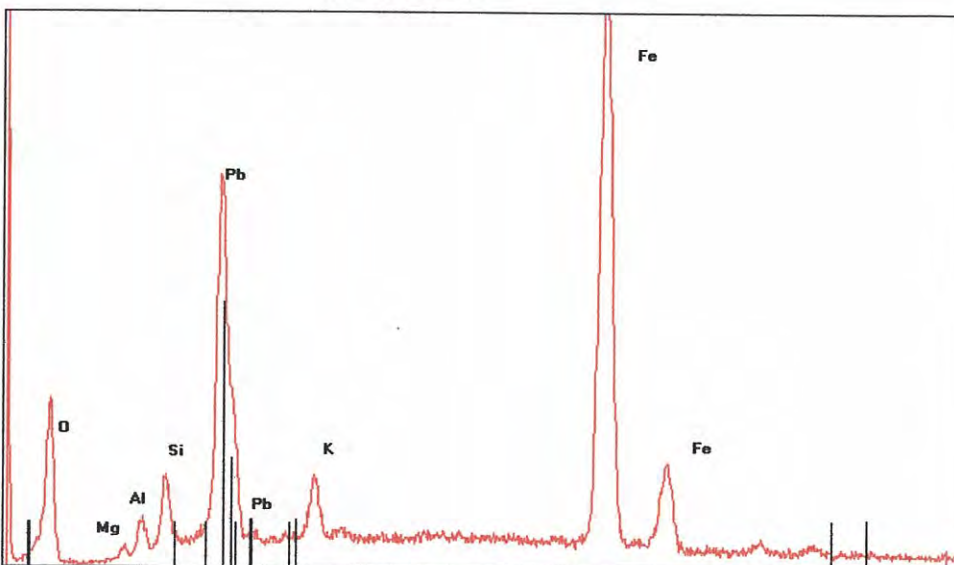
B: Hematite



C: Chlorite



D: Mg Fe rich mica



E: Pb-Fe rich Phase

Figure 4.13 continued SEM Spectral analyses of precipitates from Site C in July

The waters discharging from Bakers Creek at the weir initially exceed the guidelines for Al. After mixing with discharge from the Adit at Site C, and from the open cut workings before site B, the waters become contaminated (i.e. exceed the guidelines) with respect to Al, Cd, Cu, Pb, and Zn and pH drops at least one unit. However the most significant impact on the water quality of Bakers Creek is from the waste rock dump, with metal concentrations increasing by an order of magnitude, and Fe (as well as the previous elements) exceeding ANZECC guidelines downstream of the waste rock dump.

Intensive water sampling programs at Site I (Bakers Creek gauging station), have identified that contaminant plumes flush down Bakers Creek during storm events, with peaks in metal concentrations occurring at the gauging station 30 minutes after rainfall during a storm event.

Mass loading calculations have been used to quantify the amount of metal being discharged into Ring River during low flow and storm flow events. Storm events produced significantly higher mass loads than low flows with Zn being the major metal contaminant in both cases (158.9 T/yr and 47.6T/yr respectively at Site I). Sulphate produced the highest mass loads however (754.59 T/yr during storm flow and 3.96 T/yr during low flow). The waste rock dump was determined to be the primary source of the contamination in the long term sampling flow regimes (contributing 97.7% Zn, 98.5% Pb, 96.2 % Fe, 98.7 % Mn, 99.7% Cu, 95.2% Cd , 98.6% Al and 97.6% Sulfate).

Bacterial analyses failed to identify any sulphur-oxidising bacteria at site F below the waste rock dump. This could be due to sampling and analytical problems, or to the species of bacteria not being present. Further research is required to determine if bacteria are playing some part in the production of acid drainage from the waste rock dump.

Secondary precipitates were analysed with ESEM and determined that Al, Fe, K, Mg, O, S and Si are the predominant elements, forming hematite, chlorite, pyrite, and Mg-Fe-rich and Fe-rich micas.

Chapter 5

GEOCHEMICAL MODELLING

5.0 INTRODUCTION

Geochemical modelling of Bakers Creek was undertaken to predict the metal speciation, and saturation indices of the waters, and to compare these results with ESEM analyses.

Two modelling programs were used, PHREEQC and PHOX (version 1.3). PHREEQC is a computer program designed to perform a wide variety of aqueous geochemical calculations (Parkhurst, 1995), and was used for speciation and saturation-index calculations. The WATEQ4f database was used as it was the larger of the two database files available.

PHOX is a computer program designed to calculate and display aqueous species and predominance and (relative) mineral stability fields as a function of pH and the oxidation state parameters (e.g. Eh; Polya, 1998).

5.1 PREDOMINANT SPECIES

PHREEQC was used to calculate saturation indices, and speciation behaviour of each element analysed in selected water samples. Three representative samples were chosen for geochemical modelling. Two intensive samples from July were chosen. They had significant differences in their measured redox potentials (Eh=265 and 618 respectively). Redox can be an important influence on the speciation of certain elements (especially, Fe, Cu, and As). The low redox value sample (Eh=265) was taken at 2pm and the second sample was taken 30 minutes later, when the oxygenated waters associated with storm flows increased redox values (Eh=618). Detailed results from PHREEQC calculations are given in Appendix 7. The predicted predominant species for each water (with abundances >1%), are listed in Table 5.1.

Speciation of most elements in water samples collected during the intensive sampling program does not vary in terms of the predominant species present, but the predicted

proportions of each species does vary. For those elements affected by redox process (Fe, Cu and As), the ratios of different valences change. The ratio of As III/As V is 2.6 before the onset of rain, but changes to 2.12×10^{-13} , as the oxygenated rainwater increases the redox. As III is more toxic to biological organisms than As V, therefore flushes of more reduced groundwater from the waste rock dump are potentially more harmful to aquatic organisms. Thus there is an important predicted change in speciation behaviour.

Fe II is predicted to be predominant over Fe III species in all three examples, however, increasing Eh values cause concentrations of Fe III species to increase. The long term sample taken at the base of the waste rock dump (Site F), was relatively oxidised (Eh=685), therefore Fe^{2+} only consisted of 56.6% of the Fe species, resulting in higher Fe III proportions. Fe III (as mentioned in Chapter 2) is particularly significant in increasing the degree of acid production, thus Fe II is preferred in terms of combating acid generation.

All of the metals (except for As, and Si), occur predominantly as bare metal ions (e.g. Zn^{2+} , Pb^{2+} , etc) and with lesser concentrations of metal sulphate or hydroxide complexes. Ca, Cd, Cu, Mn, Pb, and Zn tend to form sulphate complexes, Fe occurs as hydroxide complexes, and As and Si form oxoacids (Brady and Hollum (eds), 1988). Aluminum occurs predominantly as the species Al^{3+} , but is also predicted to occur as AlSO_4^+ and as AlOH^{2+} .

Intensive (I) - High Redox July 28th 2pm			Intensive (I) - low redox July 28th 2:30pm			Long Term - Site F August 31st		
Element	Species	%	Element	Species	%	Element	Species	%
Al	Al ³⁺	73.13	Al	Al ³⁺	76.22	Al	Al ³⁺	95.04
	AlSO ₄ ⁺	23.09		AlSO ₄ ⁺	20.98		AlSO ₄ ⁺	4.38
	AlOH ²⁺	3.57		AlOH ²⁺	3.70			
As (III) (V)	HAsO ₃	72.5	As (V)	H ₂ AsO ₄ ⁻	99.19			
	H ₂ AsO ₄ ⁻	27.1						
Ca	Ca ²⁺	97.51	Ca	Ca ²⁺	98.01	Ca	Ca ²⁺	99.57
	CaSO ₄	2.52		CaSO ₄	2.31			
Cd	Cd ²⁺	95.22	Cd	Cd ²⁺	95.61	Cd	Cd ²⁺	96.22
	CdSO ₄	3.79		CdSO ₄	3.48		CdCl ⁺	3.17
Cl	Cl ⁻	99.93	Cl	Cl ⁻	100	Cl	Cl ⁻	99.80
Cu (II) (I)	Cu ²⁺	96.3	Cu (II)	Cu ²⁺	97.56	Cu (II)	Cu ²⁺	99.50
	CuSO ₄	2.6		CuSO ₄	2.46			
	Cu ⁺	1.1						
Fe (II)	Fe ²⁺	98.1	Fe (II) (III)	Fe ²⁺	60.80	Fe (II) (III)	Fe ²⁺	56.60
	FeSO ₄	1.8		Fe(OH) ₂ ⁺	26.9		FeOH ²⁺	26.00
				FeOH ²⁺	10.6		Fe(OH) ₂ ⁺	10.00
							Fe ³⁺	5.90
K	K ⁺	99.89	K	K ⁺	99.64	K	K ⁺	100
Mg	Mg ²⁺	97.84	Mg	Mg ²⁺	97.5	Mg	Mg ²⁺	99.63
				MgSO ₄	1.96			
Mn (II)	Mn ²⁺	98.10	Mn (II)	Mn ²⁺	98.39	Mn (II)	Mn ²⁺	99.54
	MnSO ₄	2.15		MnSO ₄	1.69			
Na	Na ⁺	99.95	Na	Na ⁺	100	Na	Na ⁺	100
Pb	Pb ²⁺	91.62	Pb	Pb ²⁺	92.43	Pb	Pb ²⁺	97.69
	PbSO ₄	8.02		PbSO ₄	7.41		PbSO ₄	1.4
S (VI)	SO ₄ ²⁻	89.07	S (VI)	SO ₄ ²⁻	80.94	S (VI)	SO ₄ ²⁻	71.07
	AlSO ₄ ⁺	5.13		AlSO ₄ ⁺	10.79		AlSO ₄ ⁺	11.25
	ZnSO ₄	2.05		ZnSO ₄	2.22		ZnSO ₄	9.51
	CaSO ₄	1.28		PbSO ₄	1.48		CaSO ₄	3.1
	MgSO ₄	0.74		CaSO ₄	1.26		MgSO ₄	1.77
Si	H ₄ SiO ₄	100	Si	H ₄ SiO ₄	100	Si	H ₄ SiO ₄	100
Zn	Zn ²⁺	96.96	Zn	Zn ²⁺	97.30	Zn	Zn ²⁺	99.44
	ZnSO ₄	3.04		ZnSO ₄	2.77			

Table 5.1 Speciation of intensive samples at low and high redox, and a long term from Site F (Aug).

5.2 SATURATION INDICES

The thermodynamic saturation index (SI) for a solid phase is equal to the ratio of its ion activity product (IAP) and its equilibrium solubility constant (K_{sp}; Carroll *et al*, 1998) :

$$SI = \frac{IAP}{K_{sp}}$$

where: Log SI > 0; solution is supersaturated
 Log SI = 0; solution is at equilibrium
 Log SI < 0; solution is undersaturated.

Saturation indices were calculated for the same three water samples from Section 5.1, using PHREEQC with the supersaturated minerals listed in Table 45.2. Chemical formulas and saturation indices for minerals predicted to be supersaturated in each water source, is listed in Appendix 7.

Long Term - Site F August 31st		Intensive (I) - low redox July 28th 2pm		Intensive (I) - High Redox July 28th - 2:30pm	
Phase (<i>major cation</i>)	SI	Phase (<i>major cation</i>)	SI	Phase (<i>major cation</i>)	SI
(Fe) Hematite	11.49	(Cu) Cuprous Ferrite	4.37	(Fe) Hematite	15.67
(Cu) Cupric Ferrite	3.44	(Fe) Hematite	1.88	(Cu) Cupric Ferrite	8.65
(Si) Quartz	1.41	Goethite	0.52	(Al) Pyrophyllite	2.04
(Al) Pyrophyllite	0.47	(Al) Diaspore	0.37	(Si) Quartz	0.76

Table 5.2 Saturation Indices of water samples (31/8 long term sample at Site F, and intensive samples from 28th July at 2pm and 2:30pm) calculated using PHREEQC (Parkhurst, 1995).

The water samples with higher Eh values (the long term sample, and the high redox intensive sample) are predicted to be supersaturated in Fe, Cu, Al and Si, with Hematite being the most saturated mineral phase. In contrast, the reduced sample from the intensive program was most saturated with respect to Cuprous Ferrite (CuFeO₂) rather than hematite. The Bakers Creek drainage waters are therefore supersaturated in Fe, Cu, Al and Si, and these elements are predicted to precipitate as secondary mineral coatings on gravels and boulders in Bakers Creek.

By combining the speciation data with the predicted supersaturated phases (Appendix 7), we can predict likely chemical reactions and their impact on acidity of the drainage (Table 5.3).

Supersaturated Mineral	Equation
Hematite (Fe_2O_3)	$2\text{Fe}^{2+} + 2\text{H}_2\text{O} + 0.5\text{O}_2 = \text{Fe}_2\text{O}_3 + 4\text{H}^+$ $2\text{Fe}(\text{OH})_2^+ + 0.5\text{O}_2 = \text{Fe}_2\text{O}_3 + 2\text{H}^+$ $2\text{FeOH}^{2+} + \text{H}_2\text{O} = \text{Fe}_2\text{O}_3 + 4\text{H}^+$
Goethite (FeOOH)	$\text{Fe}^{2+} + 1.5\text{H}_2\text{O} + 0.25\text{O}_2 = \text{FeOOH} + 2\text{H}^+$ $\text{Fe}(\text{OH})_2^+ = \text{FeOOH} + \text{H}^+$ $\text{FeOH}^{2+} + \text{H}_2\text{O} = \text{FeOOH} + 2\text{H}^+$
Cuprous Ferrite (CuFeO_2)	$\text{Cu}^{2+} + \text{Fe}^{2+} + 2\text{H}_2\text{O} = \text{CuFeO}_2 + 4\text{H}^+$ $\text{Cu}^{2+} + \text{Fe}(\text{OH})_2^+ + 0.5\text{H}_2\text{O} = \text{CuFeO}_2 + 0.5\text{O}_2 + 3\text{H}^+$ $\text{Cu}^{2+} + \text{FeOH}^{2+} + \text{H}_2\text{O} = \text{CuFeO}_2 + \text{O}_2 + 4\text{H}^+$
Cupric Ferrite (CuFe_2O_4)	$\text{Cu}^{2+} + 2\text{Fe}^{2+} + 3\text{H}_2\text{O} + 0.5\text{O}_2 = \text{CuFe}_2\text{O}_4 + 6\text{H}^+$ $\text{Cu}^{2+} + 2\text{Fe}(\text{OH})_2^+ = \text{CuFe}_2\text{O}_4 + \text{O}_2 + 4\text{H}^+$ $\text{Cu}^{2+} + 2\text{FeOH}^{2+} + 3\text{H}_2\text{O} = \text{CuFe}_2\text{O}_4 + 0.5\text{O}_2 + 6\text{H}^+$
Diaspore (AlOOH)	$\text{Al}^{3+} + 2\text{H}_2\text{O} = \text{AlOOH} + 3\text{H}^+$ $\text{AlSO}_4^+ + 2\text{H}_2\text{O} = \text{AlOOH} + \text{SO}_4^{2-} + 3\text{H}^+$
Pyrophyllite ($\text{Al}_2\text{Si}_2\text{O}_5(\text{OH})_4$)	$2\text{Al}^{3+} + 2\text{H}_4\text{SiO}_4 + \text{H}_2\text{O} = \text{Al}_2\text{Si}_2\text{O}_5(\text{OH})_4 + 6\text{H}^+$ $2\text{AlOH}^{2+} + 2\text{H}_4\text{SiO}_4 = \text{Al}_2\text{Si}_2\text{O}_5(\text{OH})_4 + 4\text{H}^+ + 2\text{H}_2\text{O}$

Table 5.3 Equations for supersaturated minerals from predominant species.

Table 5.3 clearly demonstrates that the formation of the predicted supersaturated minerals is likely to be an acid producing reaction.. $\text{Fe}(\text{OH})_2^+$ is the iron species which produces the least amount of acid when forming Hematite, Magnetite, Goethite, Cuprous Ferrite and Cupric Ferrite. However Fe^{2+} is predicted to be more predominant in all of the water samples, and produces significantly more acid.

5.3 MINERAL STABILITY FIELDS

PHOX was used to plot the predominant aqueous metal species on Eh-pH diagrams, using the same water samples modelled with PHREEQC. In this way both modelling programs could be compared, through use of the saturation indices calculated with PHREEQC and the mineral stability fields predicted with PHOX.

The following results were obtained:

- The low redox intensive sampling predicted Fe^{2+} or Fe_2O_3 (Hematite) and H_3AsO_3 (As III) predominance, and saturation with respect to Cu (Figures 5.1, 5.2, and 5.3)
- The intensive high redox samples predicted H_2AsO_4^- and Cu^{2+} predominance, with Fe_2O_3 saturation (Figures 5.4, 5.5, and 5.6).
- The results for the long term sample were similar to the intensive high redox sample, with Cu^{2+} , and Fe^{2+} or saturation with respect to Fe_2O_3 predicted (Figures 5.7 and 5.8).

Most of these results are consistent with those calculated with PHREEQC, particularly the As and Cu results (for each sample), and the change from Cu saturation to Fe saturation in the low redox intensive sample. Hematite (Fe_2O_3) was predicted to be saturated for the high redox intensive sample, and long term sample in PHREEQC, and this is consistent with PHOX.

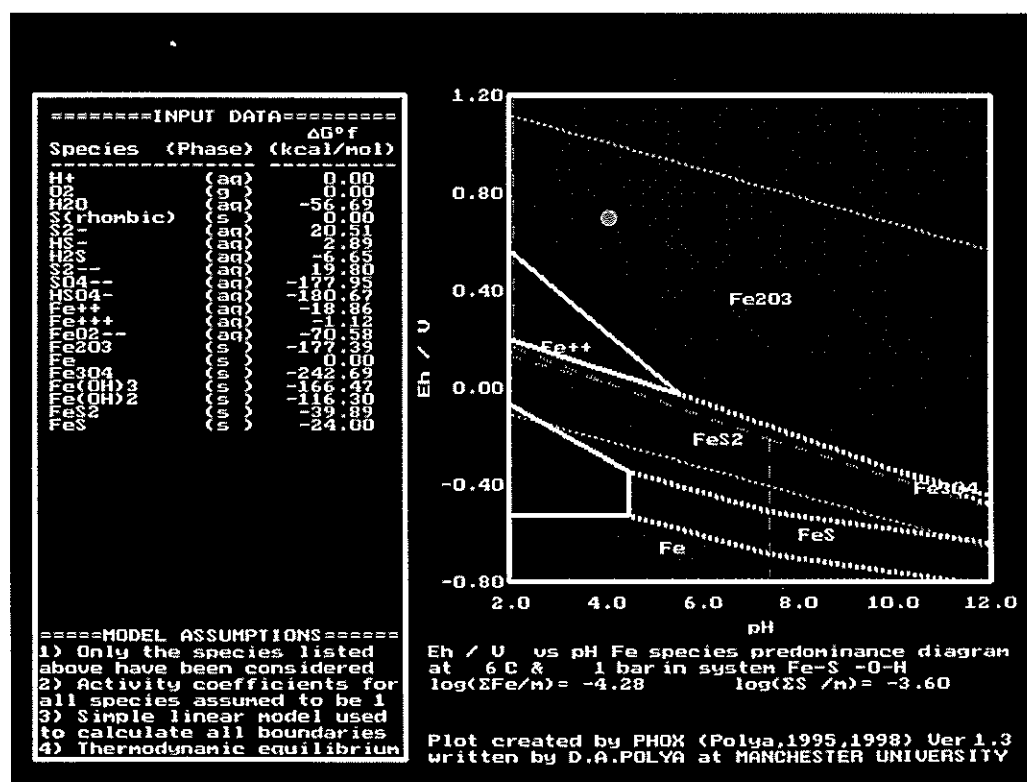


Figure 5.1 Intensive - low redox (Site I) Fe speciation.

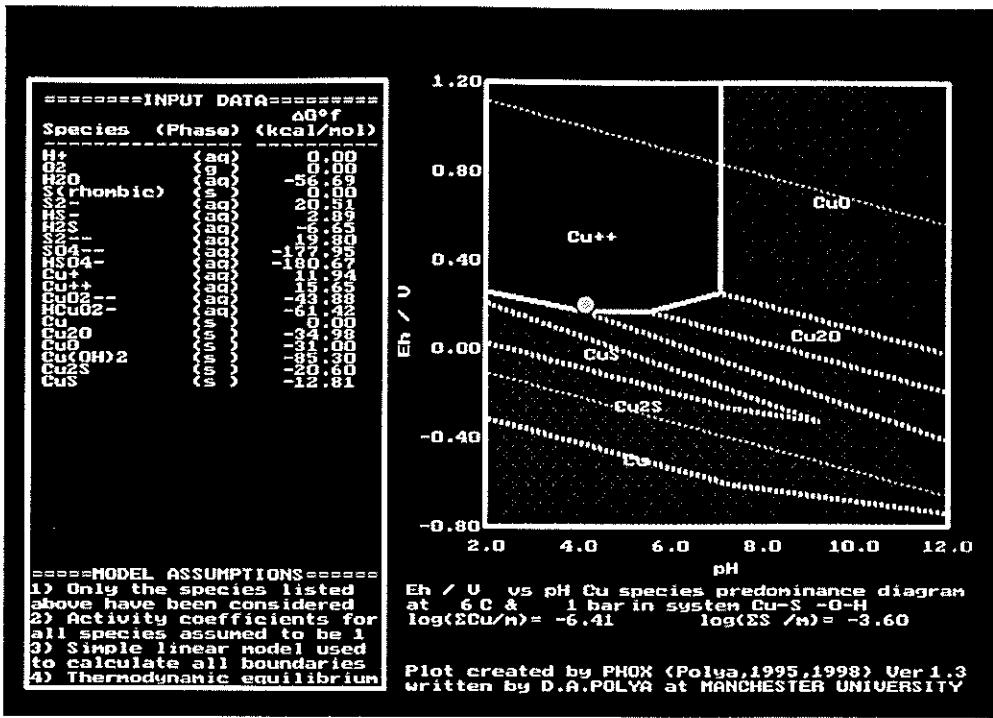


Figure 5.2 Intensive - low redox (Site I), As speciation

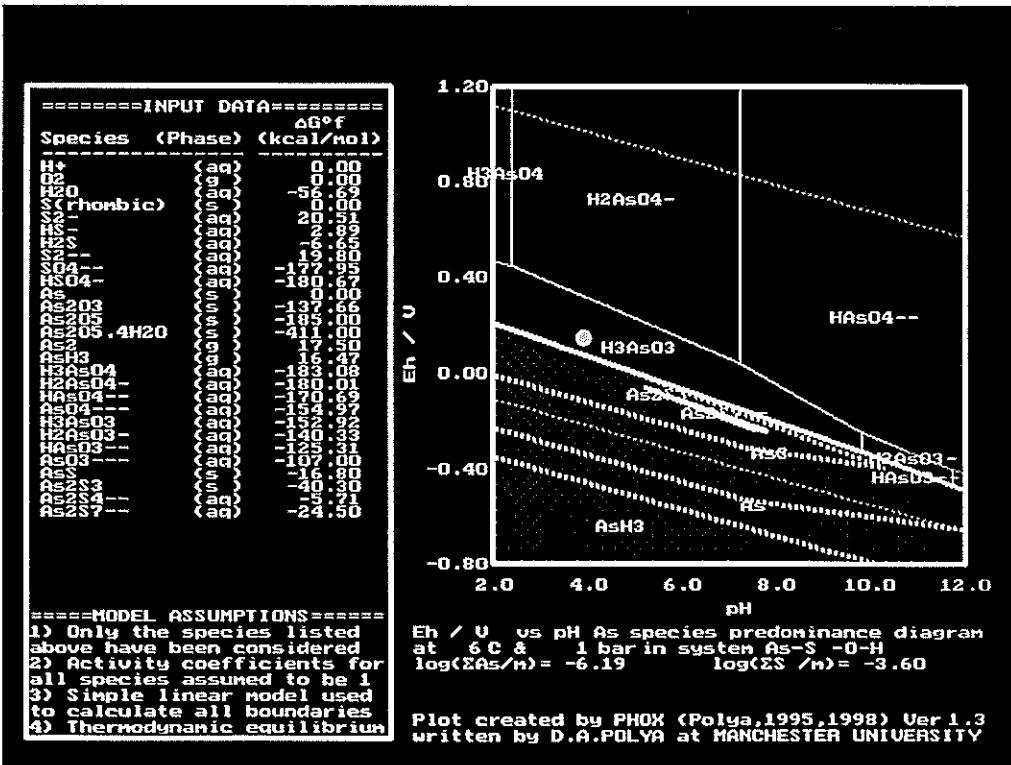


Figure 5.3 Intensive - low redox (Site 1). Cu speciation

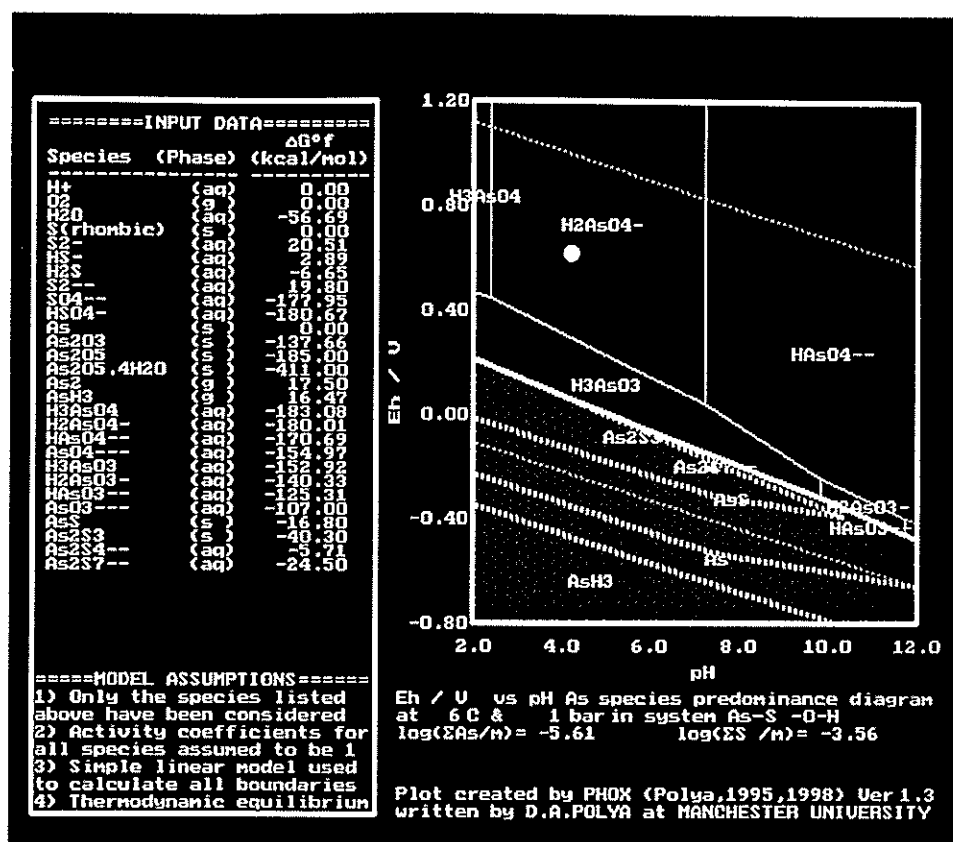


Figure 5.4 Intensive - high redox (Site I), As speciation

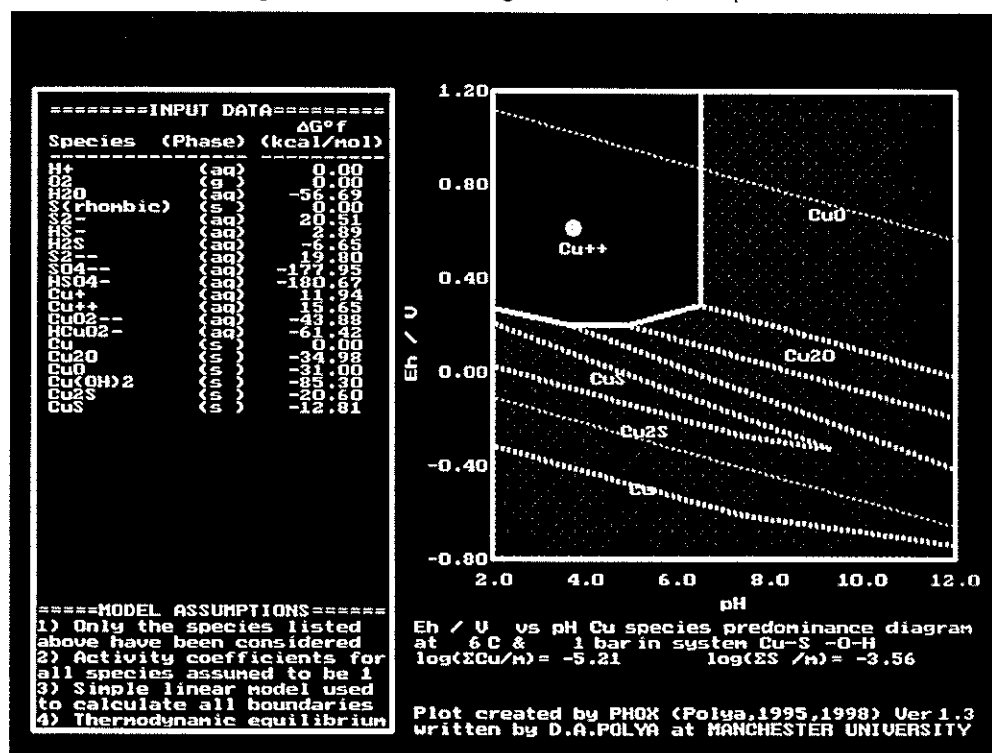


Figure 5.5 Intensive - high redox (Site I), Cu speciation.

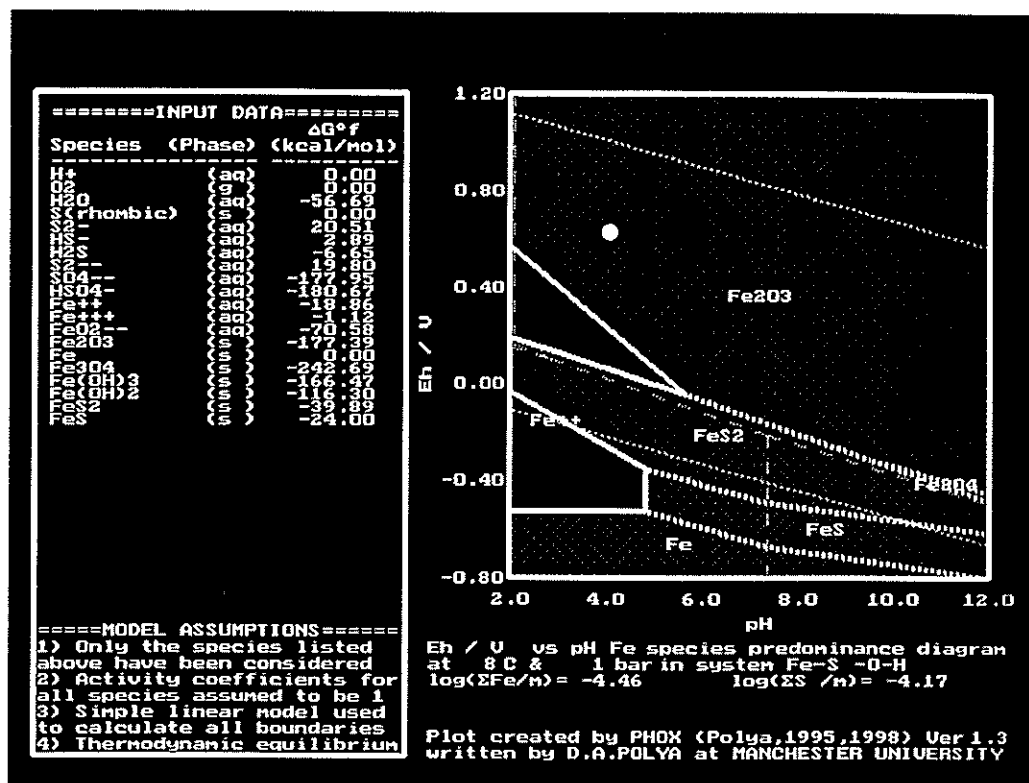


Figure 5.6 Intensive - high redox (Site I), Fe speciation.

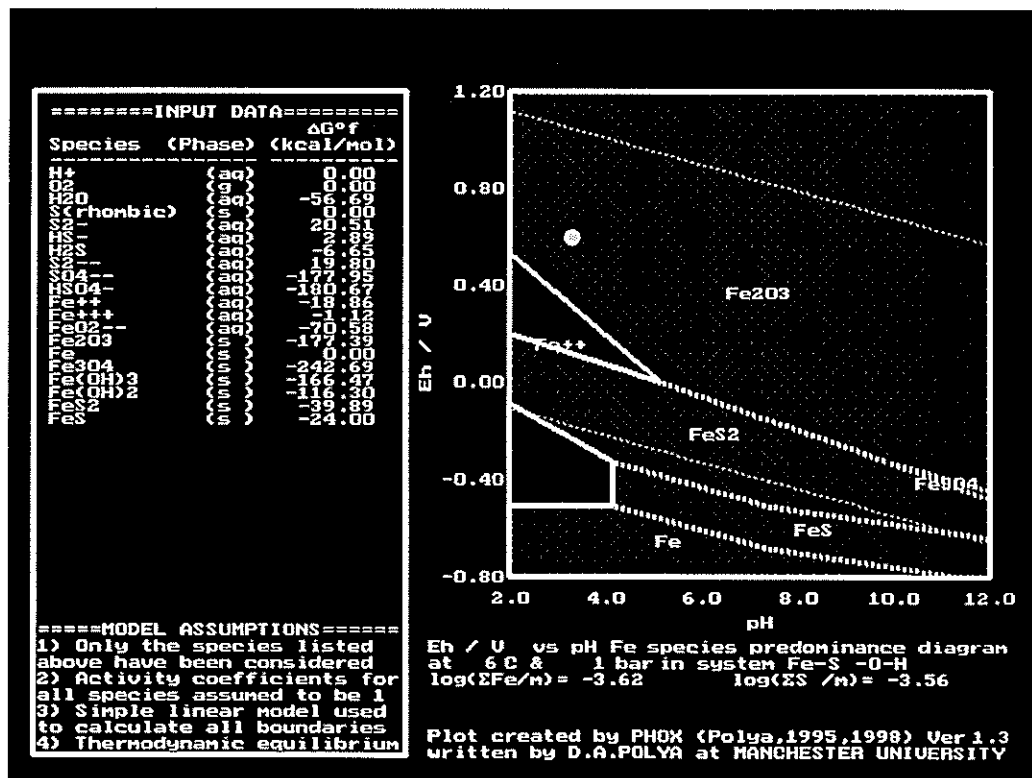


Figure 5.7 Long term - August (Site F), Fe speciation.

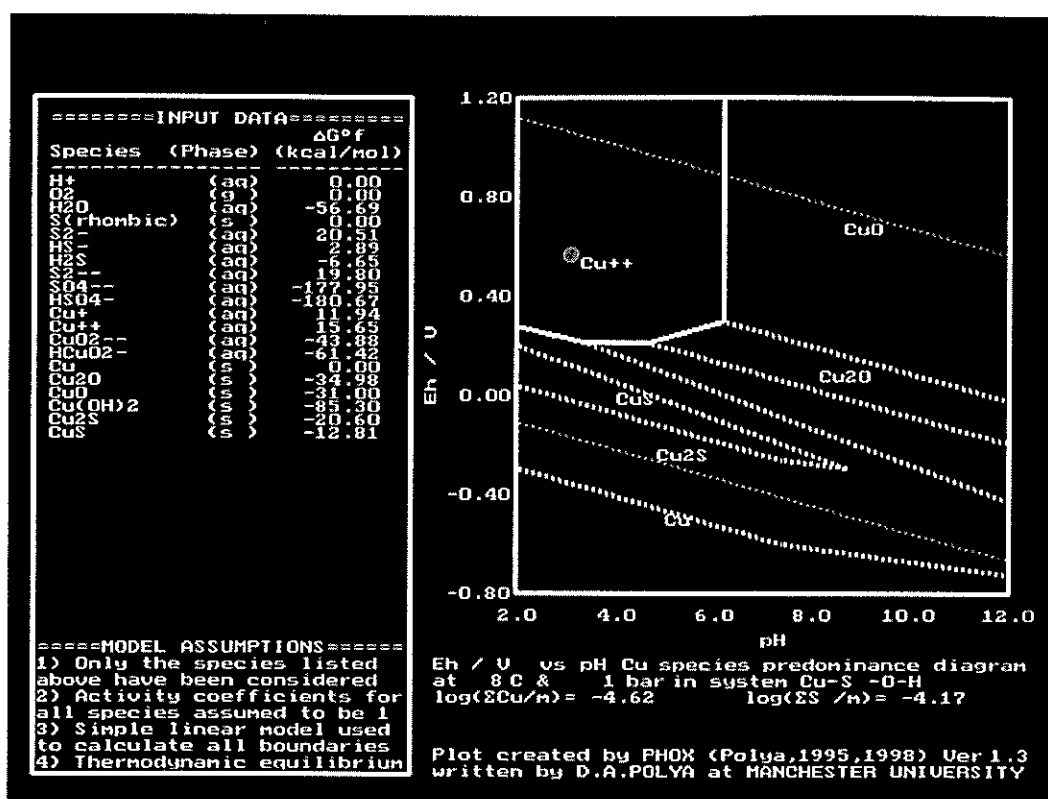


Figure 5.8 Long term, August (Site F), Cu speciation.

5.4 COMPARISONS WITH ESEM

The results from the modelling of the long term sampling at Site F, can be compared directly with the ESEM results obtained for the Site F precipitate as the data used in the modelling calculations was collected at the same time as the precipitates.

Modelling results are consistent with the iron and oxygen peaks (obtained by ESEM), which could represent hematite precipitation, and precipitation of clays or white micas predicted by PHREEQC, may correspond to Si, Al and O peaks measured by ESEM.

5.5 SUMMARY

The distribution of species calculated with PHREEQC and PHOX, determined that most of the metals in Bakers creek are transported as bare ions (e.g. Zn²⁺ and Pb²⁺). Modelling of intensive samples before and during storm flushes, determined that As changes from As III to As VI respectively (with As II being significantly more toxic). Iron occurs

predominantly as Fe^{2+} , but after storm flushes and at Site F (where redox values are higher), there is an increase in the proportion of Fe III.

Saturation index calculations predict that the Bakers Creek drainage waters are supersaturated with respect to Fe, Cu Al and Si. These elements are predicted to be the predominant components of secondary precipitates found as coatings on gravels in Bakers creek. Reconnaissance SEM analyses have partly confirmed this.

The chemical reactions involved in the formation of the predicted secondary precipitates are all acid producing, resulting in creation of additional AMD.

SECTION 3

REMEDICATION AND REHABILITATION

Chapter 6

REMEDIATION AND REHABILITATION OPTIONS

6.0 INTRODUCTION

A variety of techniques are currently employed to control and treat AMD at mine sites around the world. The choice and effectiveness of any given method is a function of:

- the status of current and potential AMD problems (sulphide content, type, distribution, reactivity, availability of neutralising material and water chemistry)
- climate, hydrology, and topography, which can in turn can affect site logistics
- sensitivity of the receiving environment to AMD
- expected time period for which the technique is to be effective
- existing natural amelioration factors (i.e. natural wetlands, high river flows)
- local natural resources
- economic resources / budgetary requirements
- regulatory standards (discharge and rehabilitation requirements)

The choice of remediation and rehabilitation techniques are site specific, and often multiple techniques are employed (Taylor *et al*, 1997). This chapter investigates remediation and rehabilitation options associated with AMD and evaluates suitable options for Bakers Creek.

6.1 REMEDIATION OBJECTIVES

Oxygen, water, and wastes containing reactive sulphide are the three primary components necessary for acid generation (Sengupta, 1993), and will typically occur when all of the following conditions are met (Hutchison and Ellison, 1992):

- Enough sulphide is contained in the waste to react chemically and biologically to form acid leachate at a faster rate than it can be neutralised by alkaline components contained within the waste.
- The waste is permeable enough to both water and oxygen, in quantities sufficient to support the chemical and biological reactions

- Transport of acidic drainage into the receiving environment is facilitated by a humid environment (which allows rainfall infiltration) and/or the waste is located in the pathway of surface water.

Most control strategies aim to *prevent* acid drainage at new facilities and to abate acid drainage where it occurs at existing sites (Hutchison and Ellison (eds.), 1992). Thus to control AMD, techniques must be designed to prevent and minimise oxidation of sulphide minerals, and the interaction between water and acid by-products *or* release of acidic water into uncontaminated surface drainage.

When AMD is well-developed, treatments must be utilized to rehabilitate the site. They are generally designed to (Taylor *et al.*, 1997:

- raise pH
- reduce metal concentrations
- reduce aqueous sulphate concentrations
- lower the toxicity / bioavailability of metals in solution
- oxidise and /or reduce the solution
- collect / dispose / isolate the metallic sludge generated.

Ideally, techniques for AMD control, rather than treatments, should be the focus of remediation (Taylor *et al.*, 1997). There are more technical options available for prevention of acid generation and leachate migration, rather than mitigating existing conditions. Treatments are limited by their need to be "retrofitted" in that they need to conform to any physical constraints at any given site (Hutchison & Ellison, 1992), which consequently tends to make them more expensive than preventative equivalents.

6.2 CONTROL AND TREATMENT TECHNIQUES

Control and treatment techniques can be classified as physical, chemical and biological. Summaries of these techniques are available in Tables 6.1 and 6.2, with more detailed descriptions in Appendix 8.

CONTROL OPTIONS	
PHYSICAL	
Diversion	<p>This minimises the volume of acidic leachate and treatment costs (as large volumes of water are generally more expensive to treat, than smaller more polluted volumes)</p> <p><i>Advantages:</i> reduces the volume of leachate to be treated</p> <p><i>Disadvantages:</i> can be expensive</p>
Covers and Seals	<p>Their primary function is to exclude oxygen and / or water.</p>
- water covers	<p>These include lakes and flooded underground and open cut workings (filled with tailings or waste rock). In dryer areas, permanent wetlands or complex covers are more appropriate</p> <p><i>Advantages:</i> Very effective as it dramatically reduces sulphide oxidation</p> <p><i>Disadvantages:</i> Only applicable in high rainfall climates; man-made water retention dams can be expensive and require maintenance; may induce unacceptable seepage.</p>
- soil covers	<p>These can range from simple clay barriers to complex, composite multi-layer organic-bearing covers. Multi-layered covers are the most effective, and saturated soil covers are more effective at reducing oxidation than dry covers.</p> <p><i>Advantages:</i> can be used in any climate;</p> <p><i>Disadvantages:</i> can be expensive; long-term effectiveness needs to be tested</p>
- micro barriers	<p>Sulphidic wastes are mixed with a phosphate rock to form insoluble iron phosphate compounds when they come into contact with pyrite oxidation products. This precipitate is believed to coat the waste rock and inhibit further oxidation.</p>
- synthetic	<p>PVC and HDPE (High-density polyethylene) are often used and are good short term measures. These are often expensive options though and degrade over time.</p> <p><i>Advantages:</i> Can contain a wide variety of fluids, high chemical and biological resistance, economical to install and maintain.</p> <p><i>Disadvantages:</i> Degrade with time limited ability to cope with stress from heavy machinery.</p>
CHEMICAL	
Selective Handling/ encapsulation	<p>AMD producing waste (or tailings) is surrounded with non-acid producing materials in an effort to reduce flow of air and water to the waste material.</p> <p><i>Advantages:</i> particularly effective if there is substantial acid-neutralising rock units</p> <p><i>Disadvantages:</i> may involve rehandling of wastes which can be expensive.</p>
Blending/Mixing and Co-Dispersal	<p>Mixtures work well for waste rock, and relies on the neutralising capacity of the non-acid producing material. Co-dispersal involves numerous small cells of AMD waste, surrounded by low permeability benign waste.</p> <p><i>Advantages and Disadvantages:</i> same as above</p>
BIOLOGICAL	
Bactericides	<p>Bactericides kill sulphur oxidising bacteria which are responsible for catalysing the conversion of ferrous iron to ferric iron.</p> <p><i>Advantages:</i> Can drastically reduce rates of acid production</p> <p><i>Disadvantages:</i> Not an effective long term strategy due to degradation of bactericides.</p>

Table 6.1 Summary of physical chemical and biological remediation strategies for control of AMD (Sengupta, 1993; OSS, 1997; Taylor et al, 1997).

TREATMENT OPTION	DESCRIPTION
PHYSICAL	
Dilution/Reaction	Only available in suitable climates and locations. Involves the release of water into peak-flowing rivers, waters with natural buffering capacities, or in marine or estuarine settings. <i>Advantages:</i> low cost <i>Disadvantages:</i> Climate dependent
Concentration	Used to reduce the volume of waters to be treated by evaporation <i>Advantages and Disadvantages:</i> same as above
CHEMICAL	
Active	These are the most common treatment strategies for acid drainage.
- Base additives	The most common alkaline reagents include limestone, hydrated lime, quick lime, caustic soda and caustic magnesia. <i>Advantages:</i> quickly increase the pH of acidic waters <i>Disadvantages:</i> Armoring may occur and reduce the ability for the Alkaline additives to react with the acidic water; expensive if local acid neutralising waste rock is not available.
- Sulphide precipitation / reduction	Toxic metal sulphides precipitated by adding soluble sulphide compounds
- Non-alkaline co-precipitation	Add BaCl_2 to precipitate BaSO_4
- Physical and or Chemical oxidation	Use oxidants to convert aqueous Fe(II) to ferric hydroxide, aqueous Mn(II) to Mn(III) hydroxides and As(III) to less toxic As(V)
- Adsorption	Coagulants and flocculants, remove metals from solution, due to their adsorption onto surfaces of other material.
- Ion Exchange	Toxic metals are substituted for harmless ions with natural or synthetic zeolites or synthetic polymeric resins.
- Electrochemistry	Solvent extraction and electrowinning provide a means by which metals can be recovered and sold to generate revenue to offset costs. <i>Advantages:</i> can generate revenue by extracting metals from the drainage <i>Disadvantages:</i> acidity and Al concentrations remain, can have high initial costs.
Passive	Designed to introduce alkalinity into drainage waters. <i>Advantages:</i> low cost and improves pH <i>Disadvantages:</i> success depends on absence of dissolved oxygen, ferric iron and Al; require maintenance; armoring effects may reduce their effectiveness.
- Open Limestone Drains	Uncovered drainage channels filled with aggregate limestone.
- Anoxic Limestone Drains (ALD)	Drainage passed through a coarse limestone gravel which excludes oxygen and minimises iron oxidation
- Successive Alkalinity Producing Systems (SAPS)	Increase the alkalinity of clean runoff with ALD's, then introduce to acid drainage streams.
- Reverse Alkalinity Producing Systems (RAPS)	Ponds are formed around AMD seeps and ground water discharges, and limestone is added to it
BIOLOGICAL	
Wetlands	Used to polish low strength AMD in conjunction with other remediation measures Anaerobic, organic rich, water saturated, thus they facilitate filtering, cation exchange, redox reactions, neutralisation and microbiological processes. <i>Advantages:</i> low maintenance and low operating costs <i>Disadvantages:</i> expensive initially; doesn't remove metals

Table 6.2 Summary of possible treatment options for AMD (Sengupta, 1992; OSS, 1997; Taylor et al, 1997).

Acid reduction using Microbiology (ACRUM)	iron reduction and oxidation then removal and anaerobic microbial reduction, and a self-sustaining organic supply.
Bioremediation in a pit lake	Under development, but investigates the role of bacteria in reducing acid and sulphate levels in static bodies of water.
Biosulphide Metal Recovery	Uses sulphur-reducing bacteria to remove Cu, Pb and Zn from drainage, increasing the pH and reducing metal loads. The metals can be sold to offset costs associated with operating the plant. Advantages: removes metals, increases pH, at low cost (even profitable) Disadvantages: initial costs may be expensive

Table 6.2 (cont.) Summary of possible treatment options for AMD (Sengupta, 1992; OSS, 1997; Taylor *et al.*, 1997).

Monitoring in and around mine sites is necessary to identify baseline conditions and to identify changes in conditions during and after mining. With this information decisions can be made regarding appropriate remediation and rehabilitation options (Sengupta, 1993).

The essential components that are needed for an AMD monitoring program include (OSS, 1997):

- Background studies identifying environmental aspects at risk (i.e. ecosystem characteristics and water catchment quality)
- Classification of all materials during the development and operational phases of mining for waste management planning
- Point source monitoring for identification of sulphide oxidation and acid generation.
- groundwater and catchment monitoring both upstream and downstream of mining operations to determine the nature and scale of any off-site impacts.
- Monitoring of control/treatment strategies to determine their effectiveness.

6.3 REMEDIATION OF BAKERS CREEK

The waste rock dump is the largest source of AMD contamination in the Hercules area, with the Adit at Site C and the open cuts before Site B the second and third most contaminated sites.

6.3.1 Waste Rock Dump Remediation

The most appropriate control or treatment methods are governed by numerous site-specific factors. The Hercules area has a high rainfall, and very steep topography, with abundant acid-producing materials. These factors exclude most of the control options described in Tables 6.1. Covers and seals are not feasible over such steep slopes. Chemical control methods are also not relevant, as the waste rock dump cannot be relocated or accessed because of the steep terrains involved. Bactericides would be unsuitable due to the high flow rates and turbulence, making it necessary to replace them at regular intervals. Furthermore bactericides are not good long term strategies for AMD control (Sengupta, 1993).

Many of the treatment options listed in Table 6.2, are also not applicable. The popular passive chemical methods are not likely to be effective, as their success depends on the absence of dissolved oxygen, ferric ion and aluminum (Bennett, 1997). Water chemistry analyses (Chapters 4 and 5) determined that iron, and aluminum are two of the major contaminants dissolved in solution, and that the waters in Bakers Creek are strongly oxidised from the turbulent flow over the waste rock dump. With this conjunction of factors, metal hydroxide precipitates are likely to armour crushed lime, which reduces the lime's ability to react and neutralise the waters. Passive treatment systems are therefore unlikely to be successful at Hercules.

To treat and control AMD emanating from the Hercules waste rock dump, a combination of techniques is most likely required in order to combat a AMD problem of this magnitude. One possible combination that may prove viable includes:

- Diversion of Bakers Creek
- Biosulphide water treatment (metal recovery)
- Wetland

Diversion of Bakers Creek before the start of the waste rock dump may improve water qualities (Figure 6.1). This would reduce the flow through the waste rock dump and thus reduce the mass loads during base flow in Bakers Creek. However this will also increase

the area of oxidation in the waste rock dump, by decreasing the immersion level, which could make the contaminant plumes from storm events even worse.

Wetlands can be an effective long term remediation strategy, given the right conditions, (particularly level ground). At Hercules, the huge volumes of contaminated water and the relatively small area of flat ground (at Williamsford), are the two major constraints. A massive leveled area would be needed to contain the large volumes of water (10 to 14000L/s), and to reduce its turbulence. Diversion could be used in conjunction with a wetland treatment system, whereby the uncontaminated water upstream of Site B could be diverted into Tipperary Creek, and the water at Site F could be collected and piped to a wetland at Williamsford (Figure 6.1).

The metal concentrations (particularly Zn), may be high enough to be treated using a biosulphide water treatment process (Koehnken, pers. comm, 1998). The biosulphide process can be divided into a biological and chemical stage (www.corp.direct.ca/ntbc ; a copy of this web page is in Appendix 9):

In the biological stage sulphate reducing bacteria is used to convert sulphate to sulphide, which produces alkalinity. The sulphide is stripped into the gas phase as hydrogen sulphide, and applied to the water in the chemical circuit. A small portion of the sulphide generated, remains in the aqueous phase and is added to the chemical circuit with the bioreactor discharge water (www.corp.direct.ca/ntbc). This also contains alkalinity (created by the bacteria), and is used to neutralise the acidity of the water in the chemical stage).

The chemical stage receives the entire flow of water to be treated, and uses the sulphide and alkalinity generated in the biological stage to neutralise the acidity and precipitate metals. Precipitation of metals as sulphides occurs sequentially, depending on the solubility of the sulphide compounds. Therefore, by controlling the process parameters selective recovery of high purity metals is possible (www.corp.direct.ca/ntbc).

Figure 6.2 outlines a biosulphide plant that recovers Cu, Zn and removes Fe and Al (as a waste sludge). The waste sludge is much less toxic than lime sludges due to the absence of Cu, Zn and metals such as Cd, Pb, Hg and Au.

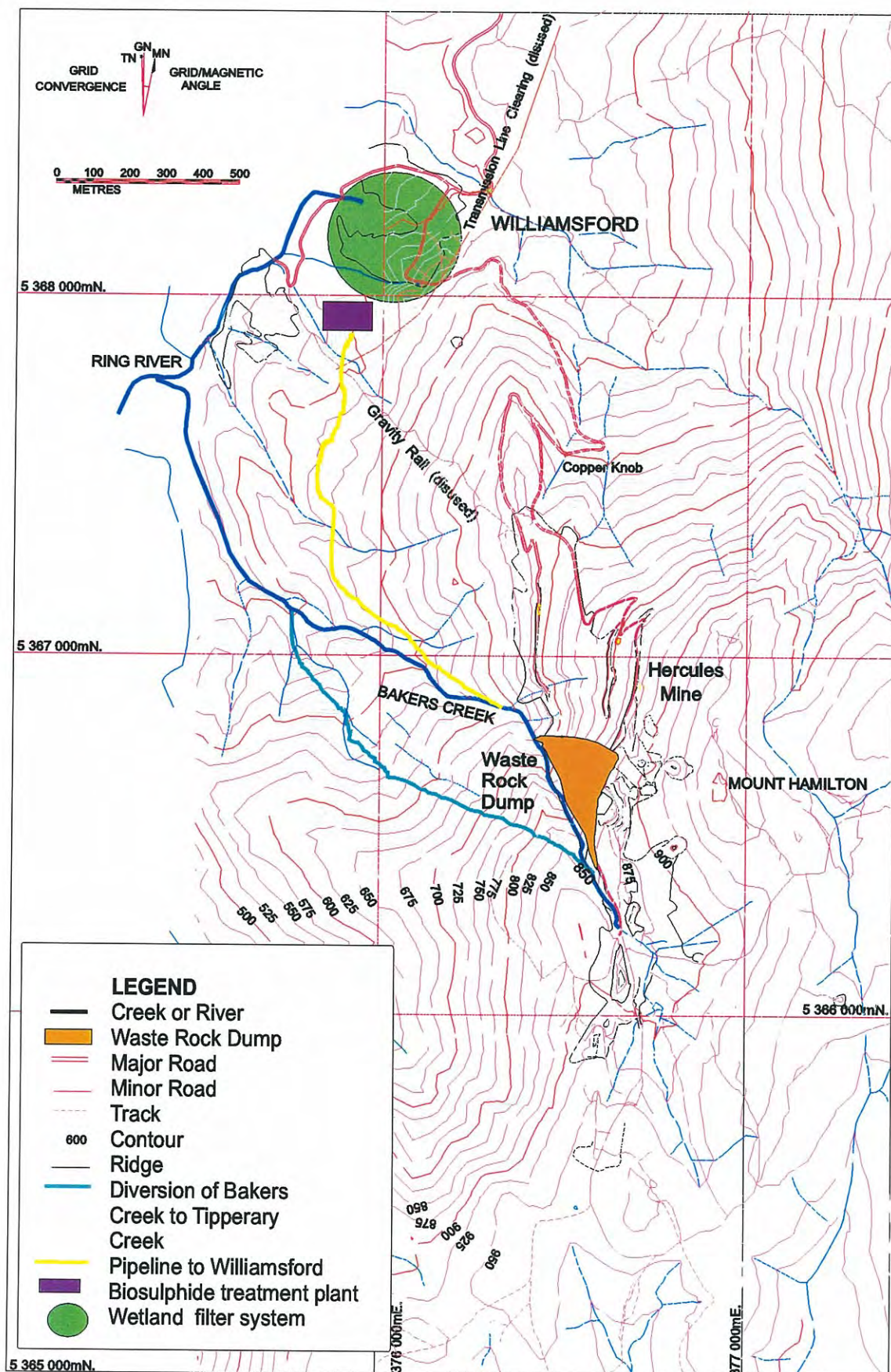


Figure 6.1 Remediation strategies for Bakers Creek including, diversion, wetland filter system and Biosulphide treatment plant.

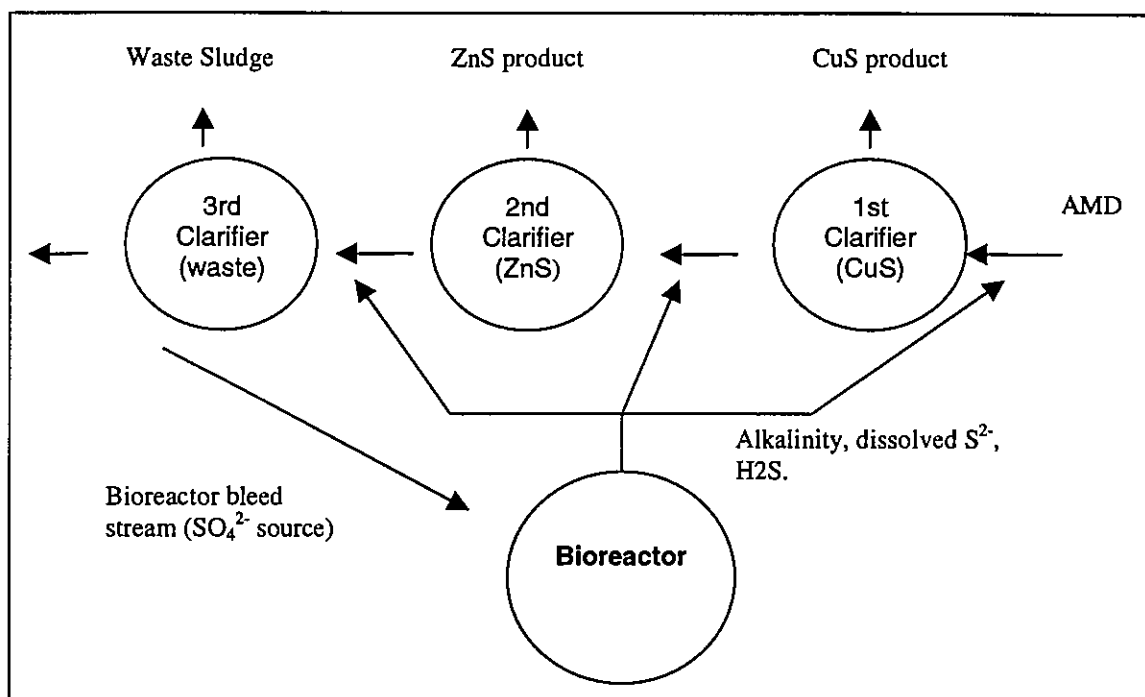


Figure 6.2 General biosulphide process configuration, modified from www.corp.direct.ca/ntbc.

The biosulphide process can be adversely affected by some of the same problems associated with wetlands (namely the large flows). Diversion of Bakers Creek could help to reduce the volume of water treated. The water could be piped from Site F, (where it has the highest metal concentrations; Figure 6.1), from a catchment at the foot of the waste rock dump. The treatment plant would be best placed at Williamsford, as it could be readily accessed and the metals extracted. This process does offer some significant advantages over a wetland system:

- Metals are recovered and removed from the environment;
- The revenue generated from the sale of the products can be used to offset operating costs, (there are some cases where the Biosulphide Process has operated at net overall profit; www.corp.direct.ca/ntbc).
- Alkalinity is produced and can be used to reduce the pH of the acidic drainage (www.corp.direct.ca/ntbc).

There are potential problems with a biosulphide treatment process. Seasonal variations in flow could limit the effectiveness of such a strategy. In Chapter 4, it was determined that the peak mass loads of metals discharge into Bakers Creek, during periods of peak flow. It

would probably not be economically viable or practical to try and treat all of the water discharged during storm flows, so an upper limit to the flow rates would need to be factored into the plant (i.e. 70 - 80 % of the peak flow may be treated). At a pilot Biosulphide project in Britannia (Canada), the bioreactor sulphide was stored during low-flow seasons, for use in times of peak flow. In this way the plant could be run at a constant optimum level year-round (Appendix 8).

If the concentration of the metals in the peak flow is sufficiently high, storing the volume of water above the cutoff point of the plan, may be feasible. A dam could be constructed in Bakers Creek, so that the optimum flow rate could then be maintained out of the dam.

Currently at Hercules, collection and diversion works have been put in place to collect all the contaminated water emerging from the mine via adit drains, so that they do not enter the waste rock dump. This water is then treated with a series of passive systems. If a treatment plant is built in Bakers Creek, the mine water could be diverted directly into the waste rock dump and treated by the biosulphide plant. This extra source of water could be useful (especially in periods of low flow), to maintain continual flow to the plant, and (as the collected water is acidic), it could be diverted into the waste rock to leach out even more metal, maximising the potential for economic operation of a treatment plant..

Other chemical treatment strategies that do not involve base additives (such as Kaolin Amorphous Derivative (KAD), and ion exchange), may also be effective at improving water quality. KAD can remove >99% of toxic metal concentrations from acid mine solutions, particularly excessive levels of Cu, Zn, Cd and or Pb (Mackinnon *et al.*, 1997). KAD is still being trialed and it is still not clear if this technique is cost-competitive with conventional reagents (Taylor *et al.*, 1997). Cost is the primary factor which inhibits the implementation of chemical remediation techniques. Conversely biosulphide metal recovery has the advantage of potentially raising revenue from the sale of the metals, making it more economically viable.

6.3.2 Remediation of minor point source contributors to Bakers Creek

The adit and open cut areas at sites C and B respectively, cannot be rehabilitated until mining ceases around the level 4 area. Site B is contaminated from the drainage from

sulphidic open cut pits. Therefore recontouring and capping may be a viable remediation strategy.

Site C is more complex. It contains highly contaminated water, which may be prevented from discharging into Bakers Creek by merely plugging the adit. This would reduce flow, and potentially decreases acid production. As this site is at a high elevation, failure of the plug from increased hydrostatic pressure is unlikely. However it would be necessary to determine where this water travels once the adit is plugged. It is likely that it would percolate downwards through fractures, and discharge from an adit further down slope. This drainage would need to be collected and treated.

The flow rates from these sites B and C during base flow and storm flow need to be measured to fully quantify their contributions to the total mass loading contribution into Bakers Creek. This information should be gathered before more comprehensive remediation strategies are formulated.

6.4 SUMMARY

A wide range of remediation and rehabilitation methods for AMD have been devised, but they are strongly site specific. At Hercules, the steep topography and rainfall are the two most significant constraints on the suitability of remediation strategies. Elevated Fe and Al concentrations in the AMD are also significant, as their presence reduces the effectiveness of lime neutralisation techniques (Taylor *et al.*, 1997).

To make a significant improvement to the water quality of Bakers Creek, the AMD emanating from the waste rock dump needs to be addressed. In the opinion of this author, a viable option is to divert the upstream segment of Bakers Creek (upstream of the waste rock dump) into Tipperary Creek, and to collect contaminated water from the base of the waste rock dump (Site F), and treat it in a biosulphide metal recovery treatment plant. In this way the metals would be removed, the pH increased, and the revenue raised from the sale of the recovered metals, could reduce, even meet the cost of installing and operating the plant.

A wetland filter system combined with diversion of Bakers Creek, may also be a suitable remediation strategy. But it does lack some of the advantages associated with Biosulphide

water treatment, such as producing alkalinity, and the revenue generated from the sale of the metals.

The other major contaminant sources in Bakers Creek occur upstream of Site B and at Site C. Further study is required to fully quantify the contribution of these sites to the contamination of Bakers Creek, before appropriate remediation strategies can be suggested. Furthermore, remediation of these sites should not take place until mining is completed in this area. For Site B, recontouring and capping would probably reduce the contamination discharging into Bakers Creek. At Site C, plugging the adit may be the most suitable remediation strategy after mining ceases.

Chapter 7

CONCLUSIONS AND RECOMMENDATIONS

7.0 CONCLUSIONS

This study was undertaken in order to characterise and quantified the AMD emanating from the Bakers Creek Waste Rock Dump. Climatic influences on the water quality were also investigated, and suitable remediation and rehabilitation options for Hercules were determined.

7.1 WATER CHEMISTRY

- Bakers Creek drainage waters ranged in pH from 3.03 to 5.55 (Table 4.2). With highest pH values measured at the adit (Site C), and the lowest pH values at the Weir (Site A).
- The maximum major and trace metal concentrations were; Al 8.47ppm, As 0.2ppm, Cd 0.4ppm, Cu 1.15ppm, Fe 43.8ppm, Pb 19.2ppm, sulfate 1440ppm and Zn 82.2ppm (Table 4.2). Al, Fe, and sulfate maximum concentrations were measured at Site C; As, and Pb at Site I; Cu, and Zn, at Site F and Cd at Site E.
- The contamination in Bakers Creek primarily emanates from the Bakers Creek waste rock dump, with additional contributions from the adit (Site C) and drainage from the open cuts up stream of Site B.
- Most of the analysed water samples from Bakers Creek are classified as Acid, high metal using the scheme of Ficklin *et al.*, (1992). This represents a degradation in water quality from the baseline samples, which have Acid, low metal to neutral-low metal compositions (Figures 4.1 and 4.2).
- The mass load contribution from the waste rock dump at times of low flow, were calculated as 2.9 T/yr Al, 0.08 T/yr Cd, 0.77 T/yr Cu, 1.54 T/yr Fe, 10.61 T/yr Mn, 1.9 T/yr Pb, 199.44 T/yr sulfate and 46.5 T/yr Zn (Table 4.6).
- The percentage contribution of contaminants from the waste rock dump to the overall metal budget of Bakers Creek is; 98.6% Al, 95.2% Cd, 99.7% Cu, 96.7 % Fe, 98.7% Mn, 98.5% Pb, 97.7 % Zn and 97.6% sulfate.

- The intensive sampling program identified that contaminant plumes are discharged from Bakers Creek during storm events, with the largest mass loads of contaminants occurring at maximum flows. The maximum mass loads (recorded at the base of Bakers Creek) are; 13.9 T/yr Al, 0.33 T/yr Cd, 3.95 T/yr Cu, 65.4 T/yr Fe, 56.15 T/yr Mn, 23.75 T/yr Pb, 754.6 T/yr Sulfate and 158.9 T/yr Zn (Table 4.7).
- Preliminary bacterial analyses failed to detect any sulphur-oxidising bacteria from the base of Bakers Creek.
- A reconnaissance survey of the chemical compositions of secondary precipitates from the base of the waste rock dump (Site F) and the Adit (Site C) identified that Al, Fe, K, Mg, O, S, and Si are principal components (Figures 4.14 and 4.15).
- Saturation indices were calculated with PHREEQC, and predicted that the Bakers Creek drainage waters are supersaturated with respect to Fe, Cu, Al and in some cases Si (Table 5.2). These calculations partly agree with the results determined from the ESEM.
- Speciation calculations determine the predominant aqueous species in a water sample. These results when combined with the saturation indices, can be used to create equations for the formation of these saturated minerals. The equations in Table 5.3 predict that the formation of the saturated minerals are all acid producing chemical reactions, and will produce additional AMD.
- Distribution of species were performed using PHREEQC, and determined that most of the metals are transported as bare ions (i.e. Zn^{2+} , Pb^{2+} ; Table 5.1).
- Fe II is predominate over Fe III in the Bakers Creek drainage waters, but that the proportion of Fe III increases rapidly after the onset of a storm event.
- As III species are predominant over As V during base flow, but As V becomes predominant as flows increase from the addition of oxygenated rainfall.

7.2 REMEDIATION AND REHABILITATION OPTIONS

- The steep topography, heavy rainfall, and high Fe, and Al concentrations, limit the remediation and rehabilitation options available for AMD emanating from the Bakers Creek waste rock dump.

- A Biosulphide metal recovery plant, combined with diversion of Bakers Creek upstream of the waste rock dump, may be a viable cost-efficient option to remediate Bakers Creek.
- A wetland could be used in place of the metal recovery plant, although it would not be capable of removing metals. The inability to generate revenue (in contrast to the biosulphide process) means that this approach has less potential to be a cost-effective strategy
- Sites B (Rail pool), and C (Adit) are additional point sources of contamination and need to be remediated once mining is complete in that area. Recontouring and covering, and plugging the Adit at Site C, appear to be the most feasible strategies, but more research is needed to fully quantify storm flow and base flow behaviour at both sites.

7.3 RECOMMENDATIONS

Recommendations for further work at Hercules include:

1. Identifying the additional source of contamination that occurs between Site F and G. Levels of As, Fe, Mn and Pb, increase between these sites (Figures 4.5 and 4.6). The adit upstream of Site G would be one site of particular interest. If it is a significant source of contamination it may be better for the water to be piped downstream of Site G rather than F to the treatment plant.
2. A V-notch weir should be placed at Site C, so the flow rate of discharge from the adit can be monitored. Percentage mass load contribution from the waste rock dump to the total load in Bakers Creek, have only been determined at a time when waterways not flowing from the adit. It is therefore important to determine what proportion of contaminants the adit is contributing to the metal budget of Bakers Creek during storm flow, so that a suitable remediation strategy can be proposed.
3. Quantifying contamination levels from the adit also has important implications for locating where Bakers Creek should be diverted from. It could be diverted upstream of Site B, and the contaminated water between Sites B and C could flow into the waste rock dump. Alternatively the creek could be diverted from Site D,

once the contamination from Sites B and C is isolated and controlled. The creek could also be diverted before Site B, and discharge from Sites B and C could be isolated and controlled.

4. On a larger scale, the hydrogeology of the Hercules mine still needs to be established. This would allow a greater understanding of subsurface flow, and would allow the effects of plugging adits to be predicted.
5. Assess economic viability and potential application of biosulphide process at Bakers Creek.

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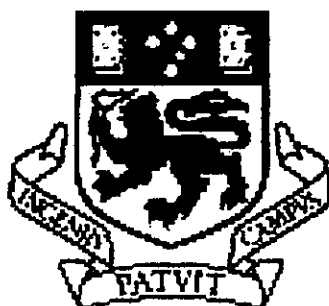
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Appendix 1

Geology, Mineralisation and Alteration of the Hercules Orebody.

Geology, Mineralisation and Alteration of the Hercules Orebody

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A literature review submitted in partial fulfilment of the requirements of
the degree of Bachelor of Science with Honours



Centre for Ore Deposit Research (CODES)

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ABSTRACT

Hercules and South Hercules are deformed stratiform polymetallic VHMS orebodies and disseminated to semi-massive base metal sulphide deposits respectively, located south of Rosebery on the west coast of Tasmania. The deposits are located in the Cambrian Mount Read Volcanic Belt, and are found within the Central Volcanic Complex. VHMS deposits have a high potential for producing acid mine drainage (AMD). Thus, wall-rock mineralogy plays a critical role in determining the severity of AMD.

The stratigraphy (from bottom to top) consists of Footwall pyroclastics, Host rocks, Black shales and slates, Quartz-phyric volcanics and Feldspar-phyric volcanics. The Hercules area contains doubling plunging parasitic folds, with strong axial planar cleavage. Beds strike approximately N-S, dipping east.

The ore lenses vary from sheets to pods and strike close to north, with a constant easterly dip. The lenses are zoned with a massive to disseminated pyrite zone at the base, then becoming a more massive sphalerite - galena \pm pyrite, to a spotty sphalerite - galena \pm pyrite, and finally on top is a carbonate \pm barite zone. Tennantite-tetrahedrite, and bismuth sulphosalts are distributed through several of these zones.

A large alteration zone extends 2km north and at least 1 km south of the ore bodies. Sericite, K-feldspar, chlorite, pyrite and carbonate are common alteration minerals.

In general the lithologies in the Hercules area have low buffering capacities, due to the relatively low quantities of carbonates, and the high abundance of pyrite. Also it contains hazardous trace metals such as Cu, Pb, Zn, Bi, Fe, Cd and As. Mining has worsened the AMD problem at Hercules, and one can predict that if the mine site is left in its current state, AMD will continue to be a large problem for many years to come.

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Chapter 1

INTRODUCTION

The Hercules orebody is located 7 km South of Rosebery on the west coast of Tasmania. The mine is found on the western slope of Mt Hamilton (1,075m) (Burton, 1975). The South Hercules deposit lies 1 km south along strike of the main Hercules ore lenses (Zaw and Large, 1992).

Hercules and South Hercules are deformed stratiform polymetallic volcanically hosted massive sulphide (VHMS) ore bodies (Lees *etal*, 1990), and disseminated to semi-massive base metal sulphide deposits respectively (Zaw and Large, 1992).

VHMS deposits have a high potential for producing acid mine drainage (AMD). Thus, wall-rock mineralogy plays a critical role in determining the severity of AMD (Alpers and Zierenberg, 1998). The mineralogy of wall rocks is controlled by the original composition and the integrated effects of sea-floor hydrothermal alteration/metasomatism, and any subsequent metamorphism. By understanding the geology, mineralogy and the alteration types in the Hercules-South Hercules area it is possible to explain why the AMD, problems are occurring, and help predict the scope of AMD in the future.

Chapter 2

HISTORY AND PRODUCTION

2.1 HISTORY

In 1891 A.E. Concliffe discovered gossan from the southern part of the Hercules orebody (Lees, 1987). Later in that year Woolands and Rooke found alluvial gold in Ring River, Bakers Creek, and Tipperary Creek, and these were subsequently worked for gold. Joseph Wills discovered more gossanous outcrop in 1894 further north where the number 1 tunnel was placed. In 1900 large scale ore output was established, through the connection of the Hercules haulage to the northeast Dundas tramway. In February 1905 to December 1906 a strike forced the mine to close when miners wages were tried to be reduced due to low metal values (1915). In 1916, the Mount Lyell Co. purchased the mine, and it was subsequently acquired in 1920 by the E.Z.Co., who discovered a way to extract the zinc from the ore, and built an aerial ropeway to Rosebery. This wasn't commissioned until after the depression in 1936.

Production was nearly continuous until 1986, but was interrupted in 1960 by a bushfire destroyed most of the surface workings

2.1 PRODUCTION

The mine closed in August 1986 after 95 years of production. In this time Hercules produced at least 2.21 Mt of sulphide ore at an average grade of 5.6% Pb, 17.6% Zn, 142% Cu, 176 g/t Ag, 2.9 g/t Au (Lees et al 1990). The mine reopened in 1996 and is operational at present.

South Hercules has an estimated ore resource of 0.7 Mt at 1.9%pb, 3.5% Zn, 0.1% Cu, 147 g/t Ag and 2.7 g/t Au (Zaw and Large, 1992).

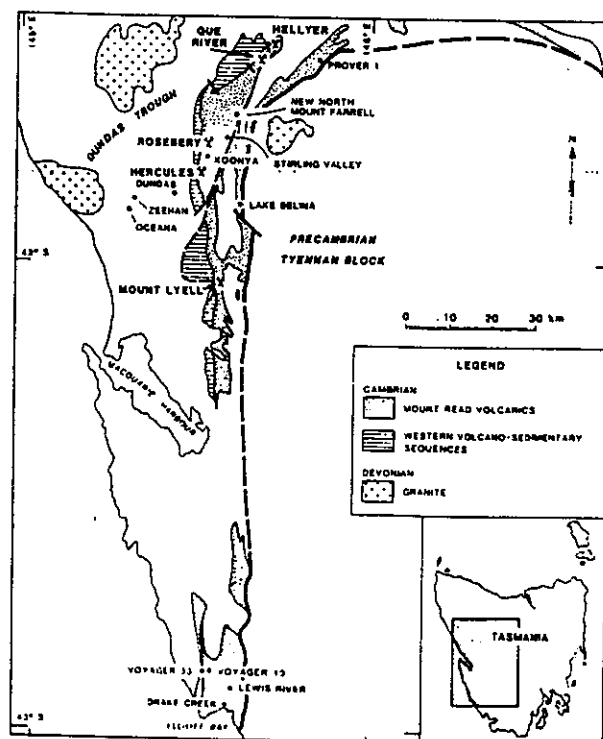
Chapter 3

REGIONAL GEOLOGY

3.1 INTRODUCTION

The Cambrian Mt Read Volcanics (MRV), provide the host for the many western Tasmania VHMS deposits, i.e. Hellyer, Que River, Rosebery, Mt Lyell and Hercules (Corbett, 1992). This 200 by 20 km belt occurs on the eastern side of the Dundas Trough (see Figure 1), which lies between the NW Precambrian Rocky Cape Region, and the eastern Precambrian Tyennan Region (White 1996).

Figure 1. (Huston and Large, 1987)
Geologic map of Tasmanian west coast showing locations of deposits discussed in text.



These volcanics contain middle to late Cambrian volcanic and Volcanogenic sedimentary rocks, interbedded with non-volcanic sedimentary rocks, and are both compositionally and texturally diverse. Rhyolitic and dacitic lavas, and syn volcanic intrusions predominate, and high to medium-K calc-alkaline andesites and basalts are locally abundant (White, 1996).

Most of the MRV were emplaced in a submarine environment, below wave base (McPhie and Allen, 1992). The evidence for this includes the presence of black pyritic mudstone, fossils, pillow lavas, hyaloclastite breccias, combined with massive sulphide mineralogy.

This volcanic belt contains rocks that have been strongly altered (both hydrothermally and diagenetically), intruded by Cambrian and Devonian granites, undergone at least 2 deformations, and were regionally metamorphosed to greenschist facies during the Devonian (White, 1996).

A number of major faults disrupt the MRV. The Henty fault (between Mt Read and Mt Charter, trends N-NE dipping west and bisects the belt obliquely, causing severe stratigraphic disruption. The Rosebery fault dips east and forms the western margin of the Rosebery and Pinnacles area (Corbett, 1992).

The major lithostratigraphic units of the MRV include the Sticht Range formation, Eastern quartz-pyritic sequence, Central Volcanic Complex, Western volcano-sedimentary sequences, and Tyndall group. There are three other locally important lithological associations, including tholeiitic mafic rocks, andesitic to basaltic volcanics, and granites and associated porphyries (White, 1992).

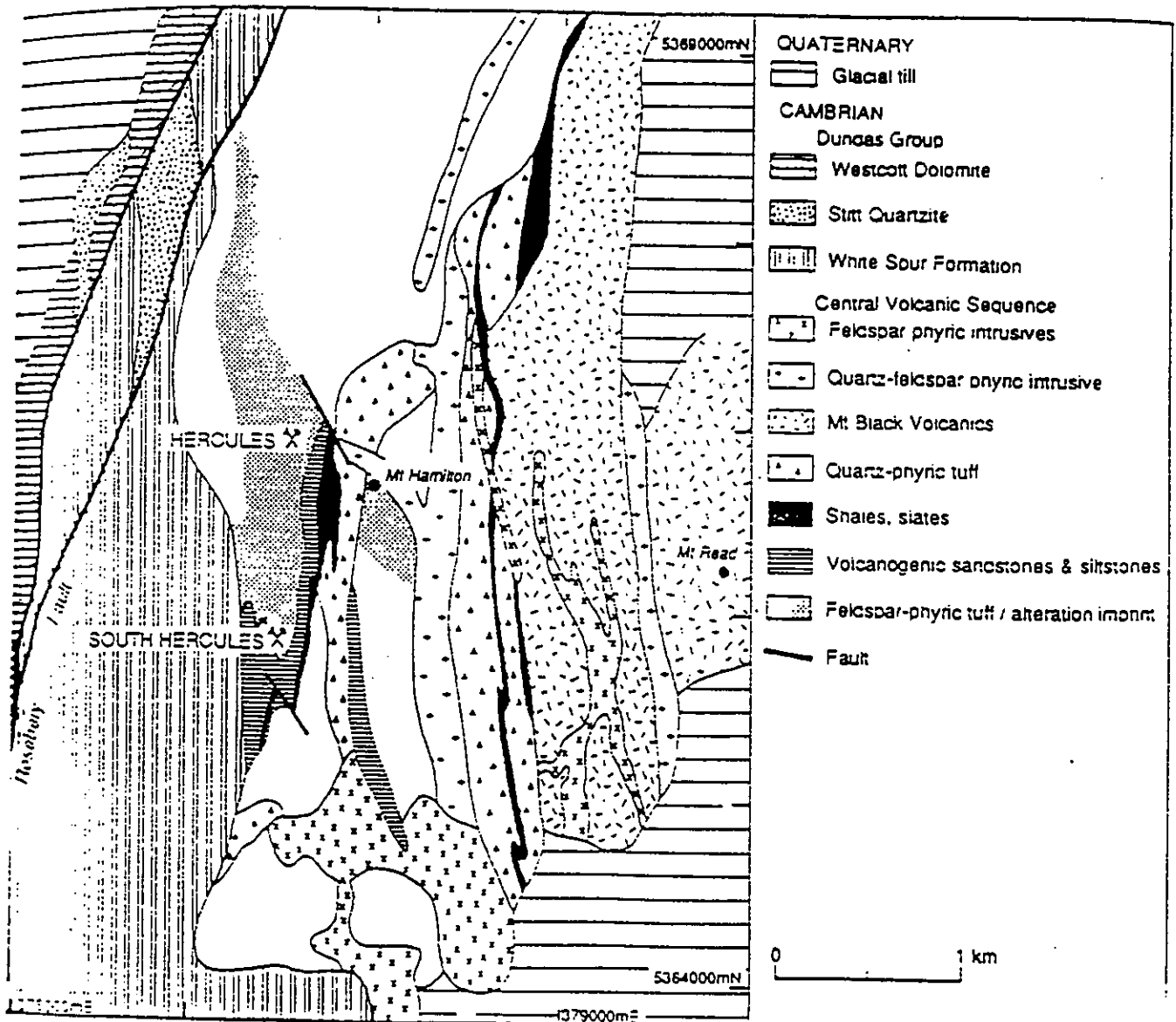
3.2 HERCULES - SOUTH HERCULES LITHOSTRATIGRAPHY

The Hercules - South Hercules ore deposits are located within the Central Volcanic Complex. Part of the Western volcano-sedimentary sequence (the Dundas Group), is found just west of the orebodies. Figure 2 (Zaw, 1992), shows the geology of the Hercules-South Hercules area.

3.2.1 Central Volcanic Complex

The Hercules and South Hercules ore deposits are located within the Central Volcanic Complex (CVC). The CVC is made up of dominantly feldspar porphyritic to lava rich volcanics. These include abundant Rhyolitic to dacitic lavas, pumice bearing volcanoclastics and massive dome like lava bodies (Corbett 1992).

Figure 2. Geology of the Hercules - South Hercules area, Western Tasmania (Zaw, 1992).



Rare bedded sedimentary rock and the abundance of pumice rich units and lavas (interpreted as ignimbrites), suggested a subaerial emplacement for the CVC during early interpretations. However, recent studies suggest that the welded textures in the massive pumice breccia units are false, produced by compaction and metamorphism. Thus they were probably submarine mass flow deposits with initially unwelded pumiceous material (Corbett, 1992).

The major divisions of the CVC in the Hercules area, are listed in Figure 2., and described in more detail in Chapter 4.

3.2.2 Western volcano-sedimentary sequence

The Western volcano-sedimentary sequence (WVSS), is made up of the Yolande River Sequence, the Dundas Group, Mt Charter Group and the Henty Fault Wedge Sequence. These interfinger with the CVC in the central part of the MRV (i.e. around the Hercules area), and overlie older Cambrian units on the west side of the Dundas Trough (White 1996). There is a wide variety of lithologies including volcanoclastic and Precambrian basement derived mass flow units, fine grained turbidities and andesitic to felsic lavas, syn-volcanic intrusions and associated breccias (White, 1996).

The Dundas Group overlies or is faulted against the NW of the CVC in the Hercules area and consists of several sub-units. White Spur formation (quartz-phyric epiclastics), Stitt quartzite, and the Westcott dolomite (Lees *et al*, 1990).

Chapter 4

LOCAL GEOLOGY

4.1 STRATIGRAPHY

The stratigraphy in the immediate mine area may be summarized as follows (top to base) (Zaw 1992, and Lees 1986).

Feldspar-phyric Volcanics	>100m	} Hanging wall "pyroclastics" (McPhie and Allen, 1992).
Quartz-phyric Volcanics	0-50m	
Black Shales and Slates	0-50m	Thinly bedded black slates, with thin lithic wacke bands and minor other clastics. Pyrite is locally abundant.
Host Rocks	20-100m	Pelitic to psammatic bedded tuffaceous sediments
Footwall Pyroclastics	>300m	fiamme bearing feldspar-phyric volcanics

4.1.1 Footwall Pyroclastics

The significant differences between this sequence and the sequence at Rosebery includes the coarse, silicified volcanic breccia immediately underlying most of the host rocks, and the ashy tuffaceous nature of the host rocks (Lees *et al*, 1990). The silicified footwall volcanics contain abundant pumice fragments which have been replaced by pyrite - chlorite assemblages. Feldspars have been totally altered by sericite and quartz (Zaw and Large 1992).

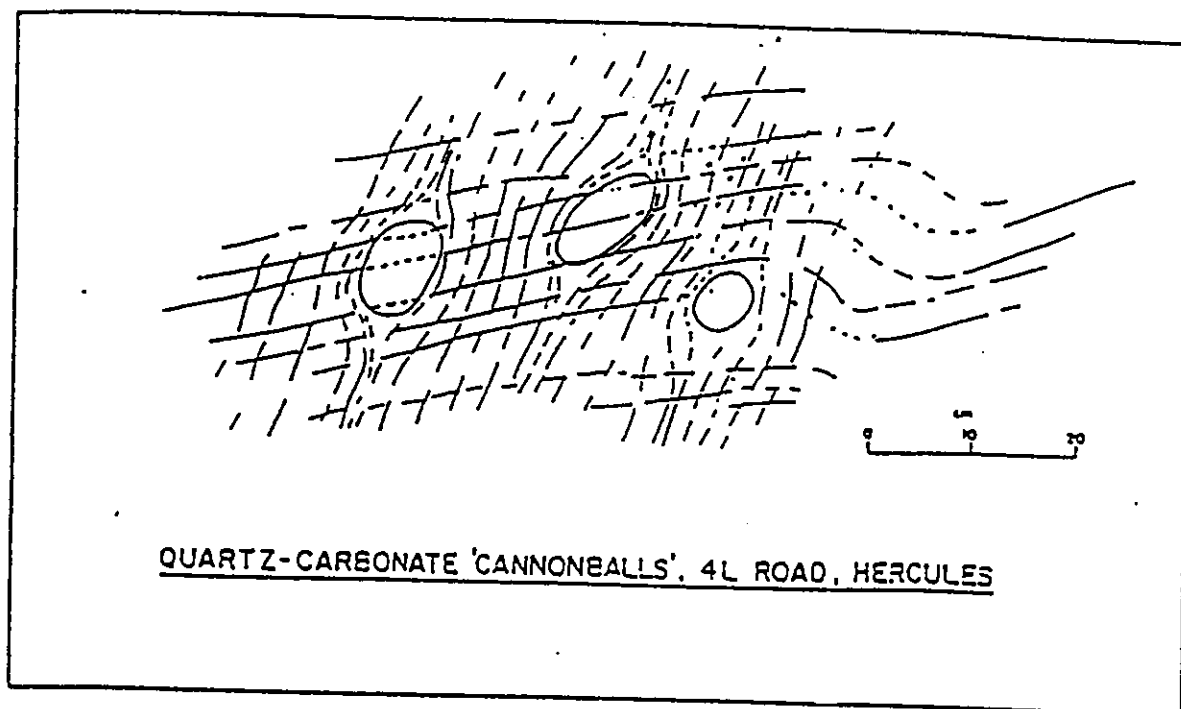
4.1.2 Host Rocks

The host rocks at the main Hercules ore lense consist of poorly bedded, psammitic to pelitic ashy and tuffaceous sedimentary rocks up to 100m thick. Quartz and variably carbonated feldspar in a sericite/chlorite matrix, makes up the main components (Zaw, 1992). The host

rocks contain the massive sulphide ore. South Hercules host rocks are similar, consisting of grey fine grained siliceous, tuffaceous sediments and sericitic feldspar-phyric volcanics (Zaw and Large, 1992).

Carbonate occurs as diagenetic nodules, and as 'pisolites', and are common within this horizon. The nodules can be 1-15 cm across with cannonball shapes. Bedding can sometimes be traced through them, although cleavage often wraps around them (see Figure 3). The pisolites form a halo around the ore lenses. They are usually composed of concentric layers of manganiferous or Fe-bearing carbonate, (although some have quartz layers within the structure), forming 1-10mm diameter spheroids and are often tightly packed (Lees 1987). Dolomite rhombs occur in limited extent. "Fire works" textures of radiating carbonate needles and colloform and botryoidal textures have been noted. Massive recrystallised carbonate also occurs close to the ores. South Hercules has similar variations in carbonate textures (Zaw 1992).

Figure 3 Quartz-carbonate "cannonballs" structure (Akerman and Barrell 1992)



4.1.3 Black Shale

This unit overlies the host rock, and consists of thinly bedded black shales with thin lithic wacke bands, and other minor clastics. Pyrite is locally abundant as disseminations, nodules and bands.

4.1.4 Hanging Wall Pyroclastics

After the shale dominated sequence, massive to thickly bedded quartz and feldspar, crystal -, lithic-, and pumice rich volcanoclastic sandstone breccias dominate (McPhie and Allen 1992). The basal beds are commonly more lithic rich (Zaw and Large 1992).

4.2 HERCULES GEOCHEMISTRY

Table 1 shows the whole rock analyses by Lees (1987) for the footwall, host rock, and hangingwall units. The three groups are chemically distinct, and can be distinguished based on the following characteristics.

Footwall:	Low Na ₂ O, High SiO ₂ and Low Na ₂ O/K ₂ O and Sr/Rb ratios.
Host Rocks:	High MnO + CaO + MgO, also high CO ₂ and BaO; low SiO ₂ .
Unaltered Hangingwall :	High Na ₂ O and Na ₂ O/K ₂ O, Sr/Rb ratios.

4.3 STRUCTURE

The Hercules area contains doubling plunging parasitic folds in a dominantly east dipping host rock sequence (Aerden, 1993). They are open to moderately tight with strong axial plane cleavage at 60 - 75° E, that is subparallel to the strike of the host rocks (Zaw and Large 1992). The strongly deformed zones contain the ore, enclosed by either sericite - chlorite schists, disrupted pisolitic or massive carbonate zones, or strongly deformed host rocks (Zaw and Large, 1992). The overall dip of the host rocks are 20 - 30° E.

There are several faults. A N-S trending steeply dipping fault is present producing a fault breccia or shear zone near the black slates/hangingwall contact. It only has a small displacement and isn't persistent. A thin fault gauge or breccia occurs in a set of NW-SE

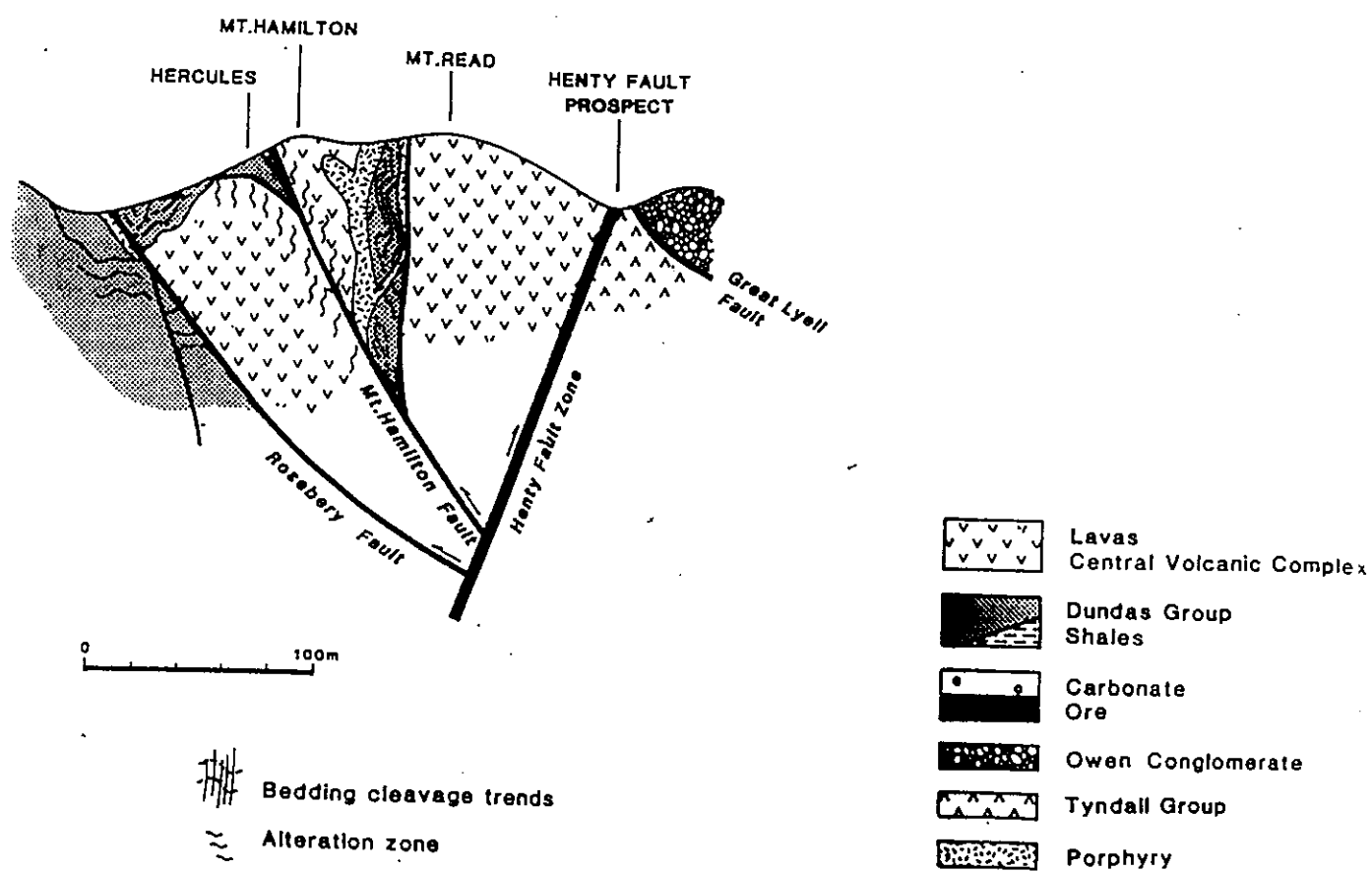
	Average Footwall	Average Host Rock	Average Hangingwall
Al ₂ O ₃	11.5	15.95	13.9
SiO ₂	74.3	62.0	73.5
TiO ₂	0.29	0.53	0.35
FeO	2.08	1.31	1.26
Fe ₂ O ₃	1.81	1.13	1.15
MnO	0.25	0.22	0.05
CaO	0.20	3.12	0.28
K ₂ O	3.70	3.90	2.40
MgO	1.24	4.20	0.85
Na ₂ O	0.06	0.36	3.74
P ₂ O ₅	0.03	0.12	0.035
SO ₃	1.56	1.35	0.03
CO ₂	0.45	2.90	0.15
H ₂ O -	0.14	0.12	0.09
H ₂ O _t	2.59	3.70	1.65
Pb+Zn+Cu	0.08	0.01	0.01
Rb (ppm)	210	185	120
Sr "	15	100	190
Ba "	1100	3325	670
Na ₂ O/K ₂ O	0.045	0.09	1.56
Sr/Rb	0.07	0.54	1.58
MnO+CaO+MgO	1.69	7.54	1.10

Table 1 (Lees, 1987) Averaged whole rock analyses of Footwall, Host and Hangingwall rocks in the Hercules area.

trending faults, but with little or no displacement. The host rock and the overlying massive units are seen in part to be a major reverse fault as seen in Figure 4.

South Hercules contains sedimentary rocks with bedding of 010 and dips 40 - 45° E. Cleavage varies from 170 - 185°, strikes and dips are steeper than the bedding at 65-70° E. The footwall contains a shear zone often with pug zones and or quartz veins. It strikes parallel with bedding. An east west shearing component has also been recognized, and the host and sedimentary rocks are offset 100m by an apparent cross fault (Zaw 1992).

Figure 4 E-W section through the Hercules area. Note the thrust faulting (Wright *et al.* 1991).



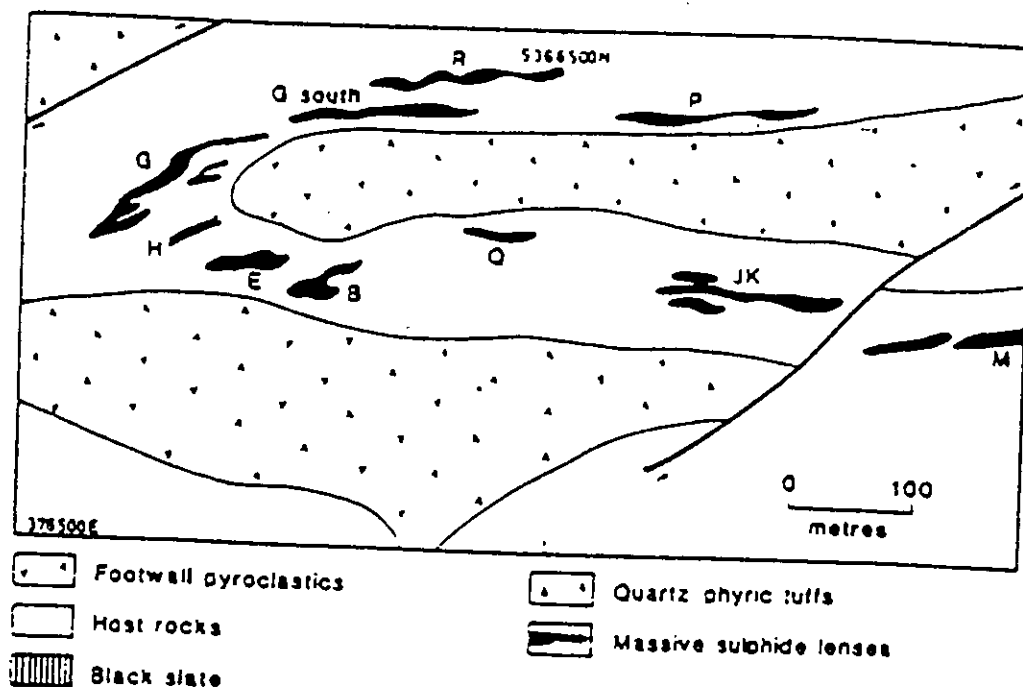
Chapter 5

ORE LENSE MORPHOLOGY AND MINERALISATION

5.1 ORE LENSE STRUCTURE

The Hercules orebody consists of numerous small lenses in the order of 100 by 100 m by 5-10m. They are generally joined at some point to a neighboring lense and in plan they form a disrupted fold (see Figure 5 and 6), (Lees *et al*, 1990). Post cleavage shearing has made the exact nature of the fold indeterminable. The lenses vary from sheets to pods, they strike close to north and consistently dip east at 60-70°. Most lie parallel to the bedding (E and M lenses are exceptions).

Figure 5 Simplified geological plan of No. 5 level. Hercules mine, showing the distribution of the ore lenses (Lees *et al*, 1992).



It was thought that the ore lenses were originally stratiform, but were realigned or transposed along cleavage. More recent suggestions conclude that the ore lenses are not stratiform but grossly stratabound within the host rock. Figure 7 (Zaw, 1992) shows how deformation has resulted in the present day arrangement of the orebodies.

Figure 6. (a) East-West cross-section of the Hercules mine (b) East west cross-section of the South Hercules deposit (Zaw, 1992).

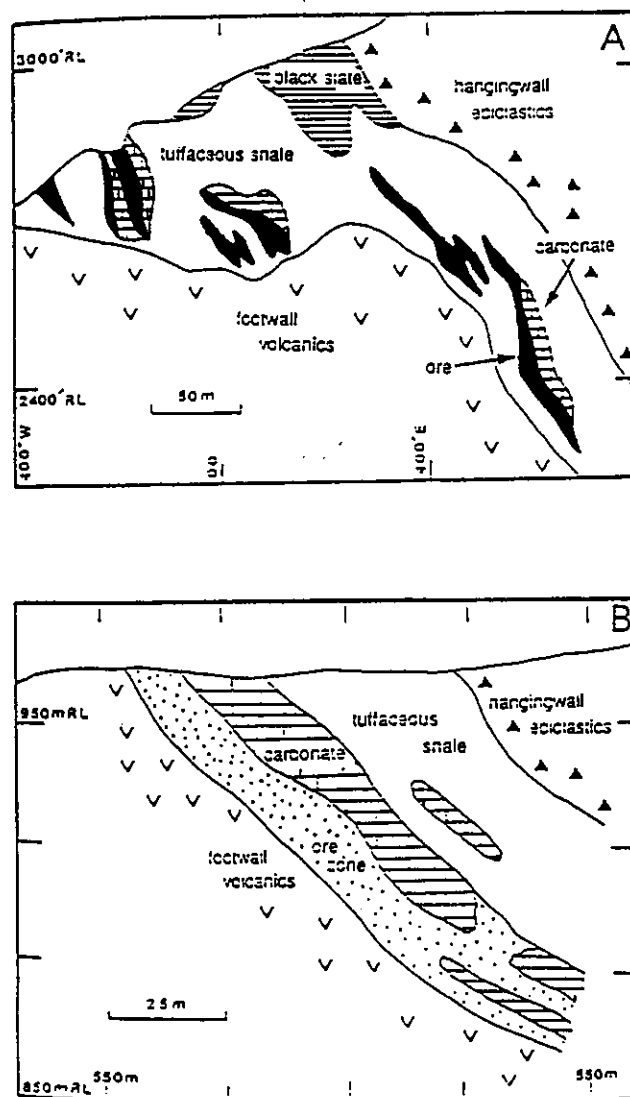
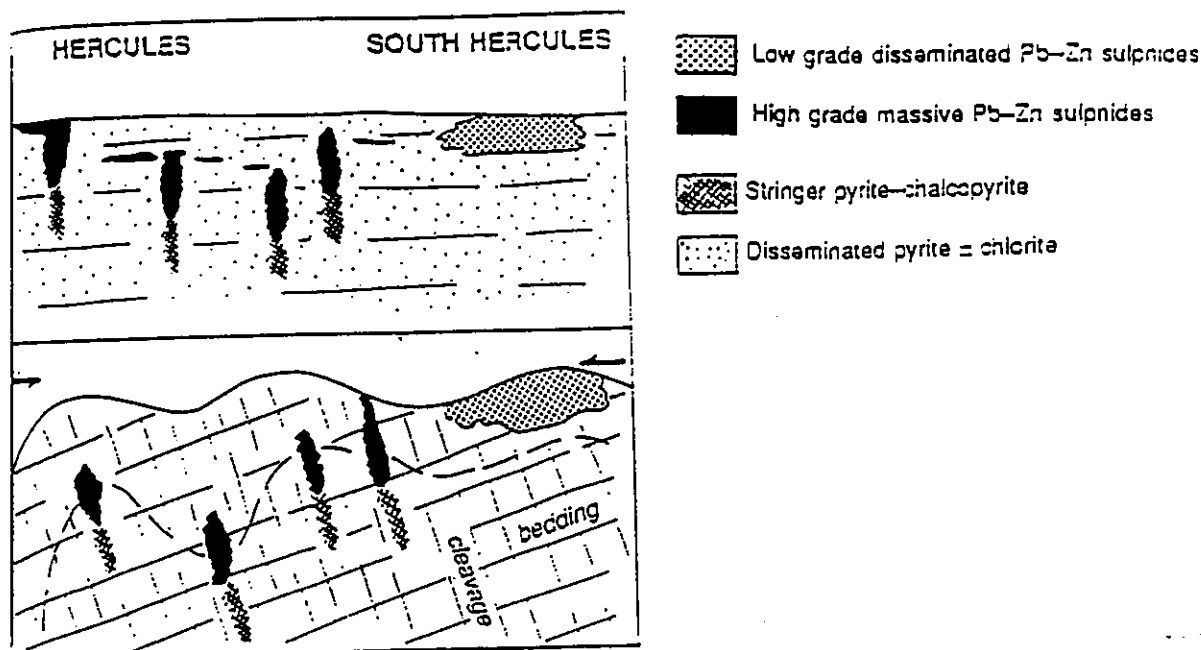


Figure 7 Folding and localization of stratabound ore lenses along cleavage at Hercules-South Hercules(Zaw 1992)



5.2 MINERALISATION

Mineralisation is localized in the short limbs of parasitic F2 folds from the micro to macroscopic scale (Aerden, 1993). The G lense is the only lense that is not zoned both vertically and longitudinally at Hercules. The bulk of the mineable lenses consist of massive to disseminated sphalerite - galena - pyrite ore. This progresses to massive pyrite, with or without a siliceous pyrite - chalcopyrite tail that pinches out with depth. Near the host rock - footwall contact the massive pyrite may become chalcopyrite rich or vein like (Zaw, 1992). The G lense at Hercules is the only barite rich lense with banded sulphides similar to Rosebery (Lees *etal*, 1990).

Detailed metal zonation of J(K)-P lense at Hercules has been determined by Zaw (1992).

Top	Carbonate \pm barite zone
	Spotty sphalerite - galena \pm pyrite
	Massive sphalerite - galena \pm pyrite
Base	Massive to disseminated pyrite zone (up to 4% Cu)

5.2.1 Massive to disseminated pyrite zone

This forms the base of the ore lenses and is dominated by pyrite with lesser chalcopyrite and variable sphalerite and galena. Minor minerals include tetrahedrite-tennantite and bismuth sulphosalt minerals. Gangue mineralogy is dominated by chlorite and quartz with lesser sericite and carbonate.

5.2.2 Massive sphalerite - galena \pm pyrite

This zone is above the massive to disseminated pyrite zone. It merges into low grade ore at the edges or spotty sphalerite - galena with or without pyrite, with minor chalcopyrite, tetrahedrite-tennantite. Free electrum is in trace amounts. Gangue includes chlorite, carbonate and sericite. Fluorite is observed in this zone, but is quite rare.

5.2.3 Spotty sphalerite - galena \pm pyrite

Spotty ores are not found at Rosebery. The spots or blebs are of sphalerite, rimmed by sericite, galena and chalcopyrite, giving a macro-zonal texture. At South Hercules the blebs are 1-20mm across and commonly elongated and stretched. This zone also contains sphalerite, galena, pyrite and minor chalcopyrite, tetrahedrite-tennantite, and like the zone below it, has trace free electrum.

5.2.4 Carbonate \pm barite zone

This zone is found at the top of the sulphide lenses. Pisolitic, colloform, and botryoidal aggregates as well as massive recrystallised tightly packed aggregates occur. Quartz carbonate veins are found with minor Pb-Zn Mineralisation. Fluorite as gangue, is located in this zone with various textures, from rounded discrete grains to thin gash veins.

Chapter 6

ALTERATION

All volcanic rocks and volcanoclastic sediments in the Rosebery-Hercules area have been texturally and geochemically modified by strong hydrothermal alteration and cleavage development (Allen, 1991). The geology is essentially similar to that at Rosebery except that Hercules' sediments are more intensely deformed (Eastoe *et al*, 1987). The assemblages of secondary mineralogies are typical of lower to middle greenschist facies metamorphism (Table 2).

Table 2 Mineralogy of the Mount Read Volcanics (from Eastoe *et al*, 1987)

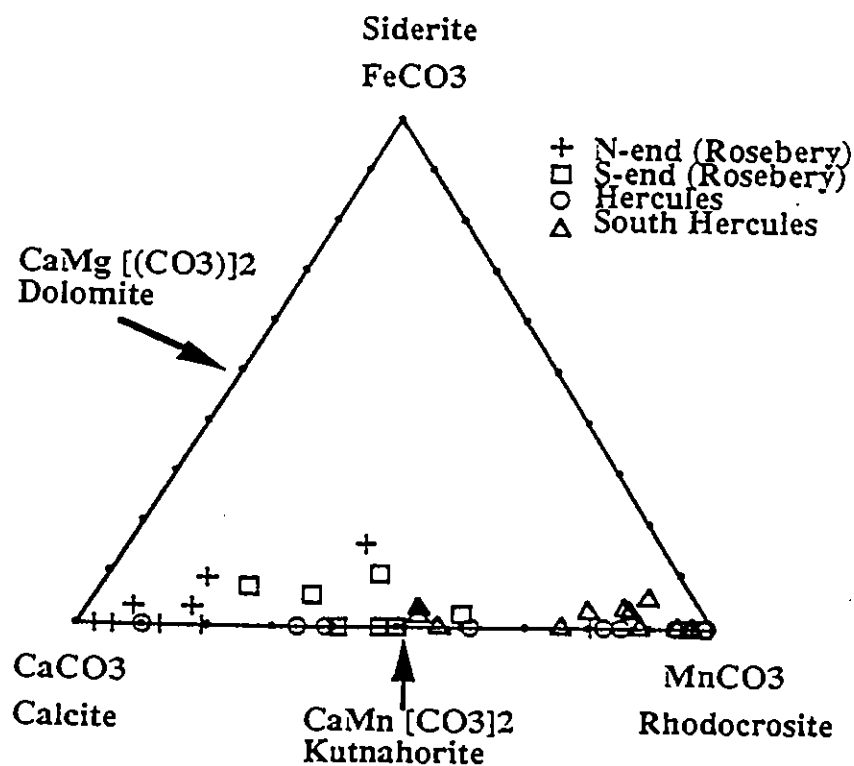
Primary	Secondary
<i>Phenocrysts:</i> Plagioclase, quartz, hornblende, clinopyroxene, biotite, ilmenite, magnetite, apatite, K-feldspar.	<i>Widespread, major:</i> Quartz, albite, chlorite, sericite, calcite, epidote, K-feldspar
<i>Groundmass:</i> Glass (now devitrified), plagioclase, apatite, zircon.	<i>Widespread minor:</i> Biotite, sphene, pyrite, hematite, magnetite, rutile-leucoxene
	<i>Local minor:</i> Apatite, allanite, tourmaline, barite, garnet, monazite.

A zone of alteration extends 2km north and at least 1km south of the mine, resulting in a zone of chloritized, pyritic, pumice in a silicified groundmass. These chlorites have anomalously high Mg and Mn. In the immediate footwall of the ore lenses, intense alteration has obliterated primary textures producing an augen textured quartz sericite schist (Zaw 1992). The proportion of chlorite decreases outward from a dark green, pyritic feeder zone located beneath the centre of the deposit. K-feldspar also occurs, both below and above the host horizon, at Hercules.

Carbonate alteration haloes occur around the ore lenses, either as 1-10mm diameter concretions and spheroids or tightly packed when close to the ore. South Hercules contains spectacular pervasive carbonate alteration assemblages ranging from bladed, disseminated, spheroid, recrystallised and massive to vein carbonate. The carbonate alteration zones have no precious metals and low base metal grades. The footwall has intense quartz sericite alteration with a schistose fabric (Zaw, 1992).

The most common carbonate assemblages can be seen from Figure 8 (Zaw, 1992)

Figure 8 Ternary plot of CaCO_3 - MnCO_3 - FeCO_3 (Mole %), for the compositional variation of carbonates from the Rosebery, Hercules and South Hercules deposits (Zaw, 1992).



Chapter 7

IMPLICATIONS FOR WATER QUALITY

7.1 ACID PRODUCTION VS ACID CONSUMPTION

Only sulphide minerals with a low metal to sulphur ratio are considered to be acid producing. Table 3 shows the influence of the metal to sulphur ratio on acid production for sulphide oxidation. Therefore a sulphide with a metal to sulphur ratio $< 1:1$ will produce acid if oxidative dissolution occurs. Thus pyrite and pyrrhotite are particularly acid producing when abundant. Disseminated pyrite is more reactive than massive pyrite (as it has a higher surface area to volume ratio).

Table 3 Influence of Metal/sulfur ratio on acid generation in sulphide oxidation reactions (from Innes, 1993).

Metal/Sulphur Ratio	Sample Reaction	Observation
1:2	$\text{FeS}_2 + 3.75\text{O}_2 + 3.5\text{H}_2\text{O} \rightarrow \text{Fe}(\text{OH})_3 + 2\text{SO}_4^{2-} + 4\text{H}^+$	Acid Generated
1:1	$\text{CuS} + 2\text{O}_2 \rightarrow \text{Cu}^{2+} + \text{SO}_4^{2-}$	Neutral
2:1	$\text{Cu}_2\text{S} + 2.5\text{O}_2 + 2\text{H}^+ \rightarrow 2\text{Cu}^{2+} + \text{SO}_4^{2-} + \text{H}_2\text{O}$	Acid Consumed

Carbonates, silicates and exchangeable cations on clays and other aluminosilicates are all capable of acid neutralisation (Innes, 1993). At Hercules, the carbonates have virtually no iron content and only minor amounts at south Hercules. The carbonates tend to be more Mn rich with up to 38.2 wt % Mn (Zaw, 1992). Rhodochrosite (MnCO_3), Calcite (CaCO_3) and Kutnahorite ($\text{CaMn}(\text{CO}_3)_2$), are the most common assemblages (see Figure 8). The low iron content is good for the buffering capacity of the carbonates as the ability for Fe bearing

carbonates to neutralize acid is moderated by their Fe content. Oxidation and hydrolysis of the iron leads to a net production of hydrogen ions (Alpers and Zierenberg, 1998). K feldspars can also neutralize acid, but not nearly as effectively as carbonates.

The rocks in the Hercules are have a large amount of pyrite. The footwall pyroclastics, host rocks and black slates, all contain pyrite. The alteration zone discussed in chapter 6 (which extends 2km N and 1km S) has undergone chlorite and pyrite alteration, much of which is disseminated. It is these characteristics that are responsible for the acidity of the waters emanating from the Hercules catchment. Although there are carbonates associated with the ore zones, the sheer magnitude of the acid producing material located at Hercules, would make the effectiveness of their neutralizing capabilities negligible giving the Hercules area a low buffering capacity.

7.2 HAZARDOUS TRACE ELEMENTS

The trace elements of environmental concern most commonly associated with VMS deposits are Cu, Zn, Pb, and Cd; Some deposits have anomalous As, Bi, Co, Hg, Mo, and or Sn. These trace elements generally occur in primary ores as Sulphide and sulfosalt minerals including chalcopyrite, sphalerite, galena, arsenopyrite tennantite-tetrahedrite, cobalite, and others as well as in solid solution and or mineral inclusions in pyrite and pyrrhotite (Alpers and Zierenberg, 1998).

Hercules contains chalcopyrite (CuFeS_2), galena (PbS), pyrite (FeS_2), tennantite-tetrahedrite ($(\text{Cu,Fe})_{12}\text{As}_4\text{S}_{13}$, sphalerite (Zn,FeS) and bismuth sulphosalt minerals (Lapidus, 1990). Data from Pasminco's 1997 Interim Environmental Management Plan Review Report, also shows elevated Cd, and not surprisingly, high Pb and Zn, from water samples taken from Bakers Creek (which drains the Hercules catchment). So not only is the acid producing nature of the wall - rock a problem, but they also contain particularly hazardous trace metals.

Chapter 8

CONCLUSIONS

The lithologies in the Hercules area, have a low buffering capacity, due to the relatively low quantities of carbonates, the high concentrations of iron sulphides (especially pyrite), and their wide distribution within the zone of alteration associated with the Hercules deposit.

There are many minerals at Hercules which have dangerous trace elements. Chalcopyrite, galena, tennantite-tetrahedrite, sphalerite and bismuth sulphosalts, are all common to the area, thus Cu, Pb, Zn, Fe, Bi, and As, are all present in the Hercules catchment, and can (and do) enter the nearby creeks and rivers, (Cd is also present in elevated quantities).

VHMS deposits by there very nature, have the potential to be environmentally hazardous. Hercules is no exception, and combined with 95 + years of mining, it has lead to the water quality problems that we see today. One can predict, that if the mine site is left in its current state, AMD will still continue to be a large problem for many years to come.

Appendix 2

Calibration Solutions for Field Equipment

Preparation of Standards and Electrolytes for Probes

pH probe

3M KCl solution, made up by dissolving 111.75g KCl into 5000mL of distilled water.

pH standards

Hanna buffer solutions pH 4 and 7. Each solution has a 2 month life expectancy once the bottle was opened

Eh probe Electrolyte

Saturated Ag/AgCl solution purchased from Orion

Eh Standard

Lights solution (Eh standard hydrogen electrode 645)

39.21g Ferrous Ammonium Sulphate $\text{Fe}(\text{NH}_4)_2(\text{SO}_4)_2 \cdot 6\text{H}_2\text{O}$

48.22g Ferric Ammonium Sulfate $\text{Fe}(\text{NH}_4)(\text{SO}_4)_2 \cdot 12\text{H}_2\text{O}$

56.2ml Sulfuric Acid H_2SO_4 , spgr 1.84

Solution must be kept in dark container as it degrades on exposure to light

Conductivity Standard

Hanna KCl Solution

Appendix 3

Summary of Thiobacillus Enrichment Media

Summary of Thiobacillus Enrichment Media Used

All measurements in g/L

For solid media, recommended 0.4% agar be used to create semi-solid gel for placement of filters. (*Thio ferrooxidans*)
Agar can inhibit Thio ferroo.

Ingredients	T & K	MS-S	Prac Book	A & P (#1)	A & P (#2)
H ₂ SO ₄	0.11 N	1N		(Thio ferro)	(Solid)
	6.1 mL	adjust pH		0.09	
K ₂ HPO ₄	0.4	0.25	3		0.6
KH ₂ PO ₄				0.4	0.4
MgSO ₄ ·7H ₂ O	0.4	0.25		0.1	
(NH ₄) ₂ SO ₄	0.4	2		0.1	
FeSO ₄ ·7H ₂ O	33.3			10	
Ca(NO ₃) ₂		0.01			
KCl		1			
NH ₄ Cl			0.1		1
MgCl ₂			0.1		0.5
CaCl ₂			0.1	0.03	
NaCl				1	
Na ₂ S ₂ O ₃ ·5H ₂ O			10		10
Al ₂ (SO ₄) ₃ ·12H ₂ O				1.4	
MnSO ₄ ·4H ₂ O				0.02	
sulfur (sublimed)		10 (1%)			
Agar	4 (0.4%)				15
Yeast extract					1
Chlorophenol Red					0.08
FeCl ₃					0.02
pH	1.3	2-4	6.8		
Extra info				follow	follow
				directions	directions

References:

T & K

Tuovinen, O.H and Kelly, D.P. 1973

Studies on the Growth of *Thiobacillus ferrooxidans*. I Use of Membrane filters and ferrous iron agar to determine viable number, and comparison with ¹⁴CO₂-fixation and iron oxidation as measures of growth.
Arch. Mikrobiol 88: 285-298

MS-S

Harrison, A.P (1984)

The Acidophilic Thiobacilli and Other Acidophilic Bacteria That Share Their Habitat.
Ann. Rev. Microbiol. 38: 265-292

Prac Book

Ummmmmm

A & P

Atlas, R.M and Parks, L.C (1993)
Handbook of Microbiological Media
CRC Press, London

Appendix 4

Water sampling concentration data.

Sample Results

Date	Location	pH	Eh	Cond	Temp	Al	As	Ca	Cd	Co	Cu	Fe	K	Mg	Mn	Mo	Na	Pb	S	S 180.7 S 182.0	Si	Zn	Cl	SO42-
16/02/98	weir	5.55	536	0	10.5		0	0	0	0		0.0575	0	0.363	0.021	0	3.71	0.00	1.758	0.50	0.637	0.034		
16	bad adit	3.03	772	700	12.7	0.006	17.4	0.7	0	0.042				5.667	10.1	0.0115	3.70		320.1	291	3.709			
16	railway pool	4.55	574	100	13.7	0	0	0	0	0		0.0127	0	0.493	0.41	0	3.65		8.247	6.25	0.682			
17	Waterfall	4.38	559	0	7.8	0	1.834	0.0012	0			0.5577	0	0.802	1.13	0	3.88		19.06	15.84	0.862			
17	m.track	3.54	726		10	0.026	14.05		0.003					4.483	9.62	0.0398	4.34		181.1	160	1.987			
19	bogs	4.34	251		8.2	0	5.786		0			0.6538	0	1.970	3.91	0.0046	4.83		59.36	52.65	1.063			
17	rbcc	4.3	702		10	0.016	9.337		0.001				0	3.014	5.63	0.0108	4.74		103.6	90.83	1.951			
17	rbcc	4.78	600		9.8	0	3.494		0			0.5664	0	1.558	1.90	0	5.11		35.14	29.87	1.855			
Date	Location	pH	Eh	Cond	Temp	Al	As	Ca	Cd	Co	Cu	Fe	K	Mg	Mn	Mo	Na	Pb	S	S 180.7 S 182.0	Si	Zn	Cl	SO42-
5/04/98	Weir	4.93	663	0	7.4	0.001	0.72	0.72				0.14	<0.5	0.79	0.06	<0.01	5.86	<0.01	2.84		0.54		29.0	2.0
5/04/98	Rail Pool	4.39	597	300	7.8	0.003	1.44	1.44				0.38	<0.5	1.06	0.69	<0.01	5.83		3.78		0.69		118	47.7
5/04/98	Bad adit	3.49	725	500	8.1	0.007	5.22	5.22					<0.5	2.66	3.07	<0.01	6.34		22.3		2.13		17.7	143
5/04/98	Waterfall	4.25	606	200	8.3	0.003	1.70	1.70				0.53	<0.5	1.16	0.88	<0.01	5.97		4.28		0.80		12.3	70.3
5/04/98	Occ	3.6	691	1100	8.9	0.006	56.8	56.8					0.70	14.5	47.3	<0.01	6.56		129		5.83		12.9	1010
5/04/98	BC middle	4	662	392	9.8	0.016	4.63	4.63					<0.5	2.18	3.72	<0.01	6.04		15.6		1.47		11.6	121
5/04/98	M track	3.96	651	900	10.2	0.021	5.33	5.33					<0.5	2.38	4.30	<0.01	5.79		17.2		1.39		11.2	115
5/04/98	bogs	3.84	666	1200	10	0.015	4.58	4.58					<0.5	2.22	3.68	<0.01	6.13		15.9		1.48		11.9	116
5/04/98	rbcc	4.06	628	700	9.6	0.003	1.81	1.81					<0.5	1.99	1.78	<0.01	7.28		11.7		1.71		25.7	101
5/04/98	rbcc	4.05	562	1500	10.2	0.007	3.81	3.81					<0.5	2.07	2.67	<0.01	6.77		13.4		1.59		27.4	122
Date	Location	pH	Eh	Cond	Temp	Al	As	Ca	Cd	Co	Cu	Fe	K	Mg	Mn	Mo	Na	Pb	S	S 180.7 S 182.0	Si	Zn	Cl	SO42-
4/06/98	Weir	5.2	435	0	3.8	8E-04	0.9	0.9	0.0001	<0.0001	0.004	0.41	<0.5	0.50	0.028	<0.0001	3.62	0.0046	0.96		0.99	0.47	6.3	50.6
4/06/98	Rail Pool	4.6	527	0	5.8	0.002	1.9	1.9	0.0018	0.0007		0.33	<0.5	0.77	0.47	<0.0001	3.77		2.98		1.15		6.6	52.9
4/06/98	Bad Adit	3.3	749	600	6		20.3	20.3		0.045			<0.5	9.04	10.2	<0.0001	4.79		1.04		6.16		10.2	1440
4/06/98	Waterfall	4.59	522	0	6.2	0.002	3.0	3.0	0.0003	0.0014		0.66	<0.5	1.06	0.95	<0.0001	4.02		5.39		1.44		8.1	60.5
3/06/98	BC middle	4.04	639	289	5.3	0.004	11.4	11.4		0.0050		0.85	<0.5	4.08	8.13	<0.0001	4.23		33.5		2.29		6.8	523
3/06/98	BCGS	4.24 or 5.3	610	134.7	4.4	0.009	7.5	7.5		0.0038		0.58	<0.5	2.89	5.07	<0.0001	4.86		21.4		2.20		9.8	178
3/06/98	rbcc	4.02	684	161.3	4.3	0.033	7.0	7.0		0.0039			<0.5	3.4	3.82	<0.0001	6.03		27.8		2.94		10.2	207
3/06/98	rbcc	3.9 or 5.1	671	167.6	4.5	0.022	7.0	7.0		0.0038			0.57	3.52	3.49	<0.0001	6.56		29.5		3.11		13.0	217

Above guideline (ANZECC/NWQMS, 199)

Suspect result

Date	Location	pH	Eh	Cond	Temp	Al	As	Ca	Cd	Co	Cu	Fe	K	Mg	Mn	Mo	Na	Pb	S	Si	Zn	Cl	SO42-
31/08/98	weir	5.29	487	0	6.2		<0.001	2.55		<0.05	0.006	0.16	1.14	0.53	0.12		4.77	0.000	1.69	0.89		17	8.7
	railpool	4.81	349	0	10.5		0.006	1.64		<0.05	0.008	0.34	0.64	0.69	0.47		4.04	0.000	2.89	1.07		8.5	15
	waterfall	5.26	258	0	10.1		0.003	2.59		<0.05		0.39	0.44	0.81	0.82		3.82	0.000	4.32	1.17		9.7	24
	bwrdr	3.48	685	500	7.7		<0.001	19.7		<0.05				7.82	13.2		3.66	0.000	86.8	3.38		18	420
	bcmiddle	3.45	729	5	8.4		0.017	20.2		<0.05			0.46	7.56	14.2		3.82	0.000	79.8	3.20		13	357
	bccgs	3.71	321	800	7.8		0.005	13.1		<0.05			0.61	4.89	9.20		4.76	0.000	46.2	2.89		13	174
	rbbc	4.07	301	800	7.2		0.004	9.17		<0.05			0.78	3.80	5.02		5.59	0.000	29.6	3.08		15	117
	rbac	4.25	271	800	7.1		0.004	7.96		<0.05			0.82	3.50	3.76		5.67	0.000	24.9	3.03		15	97

Intensive Sampling

Date	Time	pH	Eh	Cond	Temp	Al	As	Ca	Cd	Co	Cu	Fe	K	Mg	Mn	Mo	Na	Pb	S	Si	Zn	Cl	SO42-
27/07/98 28/798	13:50	4.3	269	206	7.8		0.018	9.57					6.25	3.37	6.31	0.014	6.53	2.1	25.4	3.02		7.4	100
	9:00	4.27	271	174.6	6.3			7.51					0.90	2.99	5.74	0.010	4.54	1.37	20.8	3.34		5.8	89
	14:00	4.34	265	144.4	6.4			5.15					0.74	2.12	4.04	0.009	4.21		16.2	2.16		4.9	73
	14:30	4.29	618	148.3	6.1			6.06					1.07	2.91	6.81	0.008	4.64		18.1	3.77		4.4	80
	15:00	4.35	584	132.2	5.8		0.298	6.08					0.87	3.08	8.64	0.006	3.09		16.7	4.21		3.9	69
	15:30	4.37	626	116.7	5.5			4.94					0.80	2.42	6.90	0.008	3.03		14.3	3.35		3.8	62
	16:30	4.3	251	98.7	5.5			3.76					0.63	1.61	4.60	0.005	3.25		11.4	2.26		4.4	56
	18:00	4.23	611	102.1	5.5			3.00					0.32	1.32	2.38	0.005	3.47		10.2	1.26		3.9	51
	19:00	4.24	405	108.5	5.7			3.24					0.47	1.37	2.31	0.005	3.96		10.7	1.24		4.4	56
	20:00	4.24	435	0	5.6		0.038	3.02					0.23	1.33	2.23	0.005	3.55		10.6	1.19		4.1	55
	21:00	4.2	435	0	5.6		0.038	3.13					0.32	1.39	2.33	0.004	3.66		11.2	1.25		4.3	56
	22:00	4.26	527	0	10.8		0.033	3.35					0.25	1.48	2.47	0.004	3.62		12.2	1.29		4.2	60
29/07/98	23:00		538	0	5.3		0.036	3.14					0.38	1.39	2.43	0.004	3.67		11.4	1.28		4.5	60
	10:00	4.03	441	0	5.5		0.016	4.15					0.33	1.78	3.04	0.003	3.64		15.4	1.49		5	74
Date	Time	pH	Eh	Cond	Temp	Al	As	Ca	Cd	Co	Cu	Fe	K	Mg	Mn	Mo	Na	Pb	S	Si	Zn	Cl	SO42-
22/09/98	6:20 AM	4.17	352	203	14.4		0.008	8.67		<0.05			6.31	2.59	4.32	0.002	5.60		23.1	2.01		13	80
	19:05	4.1	425	173	12		0.017	7.00		<0.05			1.13	2.11	3.50	0.002	4.54		17.3	1.64		7.2	71
	7:45 PM	4.29	251	85	11.1	0.42	0.011	2.99		<0.05			0.71	0.90	1.70	0.001	3.32		7.07	0.74		5	31
	20:20	4.39	339	85.4	10.8	0.38	0.001	2.88		<0.05		0.65	0.67	0.91	1.30	0.001	3.65		6.51	0.70		5.3	26
	21:05	4.28	419	107	10.9	0.45	0.001	2.95		<0.05		0.44	0.51	1.03	1.54	0.001	3.56		7.86	0.75		6.3	33
	21:55	4.23	438	124.1	10.6		<0.001	3.49		<0.05		0.40	0.53	1.22	1.86	0.001	3.65		9.88	0.93		6.1	42
	22:40	4.17	459	125.8	10.7		0.001	3.72		<0.05		0.47	0.56	1.22	1.89	0.001	3.80		10.3	0.84		6.4	43
	23:25	4.22	525	112.5	10.8		<0.001	3.24		<0.05		0.51	0.50	1.11	1.61	0.001	3.67		8.67	0.79		6.5	37
	13:30	4.01	545	177.4	11.3		<0.001	4.94		<0.05		0.83	0.39	1.68	2.60	0.003	3.82		15.8	1.15		6.8	65

Appendix 5

Rainfall at Mt Read

Rainfall at Mt Read Monitoring Station - (Bureau of Meteorology (BOM), 1998)

Date	Rainfall	Date	Rainfall	Date	Rainfall	Date	Rainfall	Date	Rainfall
January		18-Mar-98	4	24-May-98	77	August		Intensive Data	
07-Jan-98	0	19-Mar-98	2	25-May-98	9	02-Aug-98	6	7/26/98 0:00	
08-Jan-98	0	20-Mar-98	0.4	26-May-98	0.2	03-Aug-98	18	7/26/98 3:00	
09-Jan-98	0	21-Mar-98	8	27-May-98	0	04-Aug-98	8	7/26/98 6:00	
10-Jan-98	0	22-Mar-98	0	28-May-98	83	05-Aug-98	28	7/26/98 12:00	
11-Jan-98	0	23-Mar-98	11	29-May-98	19	06-Aug-98	2	7/26/98 15:00	
12-Jan-98	0	24-Mar-98	15	30-May-98	20	07-Aug-98	0	7/26/98 18:00	0.2
13-Jan-98	0	25-Mar-98	12	31-May-98	8	08-Aug-98	0	7/26/98 21:00	
15-Jan-98	15	26-Mar-98	3	June		09-Aug-98	4	7/27/98 0:00	0.2
16-Jan-98	1	27-Mar-98	0.4	01-Jun-98	2	10-Aug-98	11	7/27/98 3:00	2
17-Jan-98	0	28-Mar-98	2	02-Jun-98	10	12-Aug-98	6	7/27/98 6:00	4
18-Jan-98	15	29-Mar-98	29	03-Jun-98	0.6	14-Aug-98	0	7/27/98 9:00	
19-Jan-98	4	30-Mar-98	14	04-Jun-98	0	16-Aug-98	0.4	7/27/98 15:00	2
20-Jan-98	0.2	31-Mar-98	49	05-Jun-98	12	18-Aug-98	0	7/27/98 18:00	0.2
21-Jan-98	0	April		06-Jun-98	23	19-Aug-98	0	7/27/98 21:00	0.2
22-Jan-98	0	01-Apr-98	5	07-Jun-98	12	21-Aug-98	0	7/28/98 3:00	3
23-Jan-98	17	02-Apr-98	15	08-Jun-98	0.2	22-Aug-98	18	7/28/98 6:00	0.6
24-Jan-98	0	03-Apr-98	25	09-Jun-98	0	23-Aug-98	17	7/28/98 9:00	2
25-Jan-98	0.6	04-Apr-98	12	10-Jun-98	0	24-Aug-98	2	7/28/98 12:00	2
26-Jan-98	0	05-Apr-98	7	11-Jun-98	0	25-Aug-98	17	7/28/98 15:00	2
27-Jan-98	0	06-Apr-98	15	12-Jun-98	7	26-Aug-98	0	7/28/98 18:00	4
28-Jan-98	0.2	07-Apr-98	0.4	14-Jun-98	5	27-Aug-98	0	7/29/98 0:00	6
30-Jan-98	55	08-Apr-98	0.2	15-Jun-98	30	28-Aug-98	0	7/29/98 3:00	15
31-Jan-98	2	09-Apr-98	24	16-Jun-98	0	29-Aug-98	8	7/29/98 6:00	11
February		10-Apr-98	14	17-Jun-98	2	30-Aug-98	3	7/29/98 9:00	7
01-Feb-98	0.2	11-Apr-98	2	18-Jun-98	28	September		7/29/98 12:00	2
02-Feb-98	0	12-Apr-98	1	19-Jun-98	2	01-Sep-98	0	7/29/98 18:00	0.2
04-Feb-98	0	13-Apr-98	53	20-Jun-98	0.2	02-Sep-98	27	7/29/98 21:00	
05-Feb-98	0	14-Apr-98	32	21-Jun-98	33	03-Sep-98	0	7/30/98 0:00	
06-Feb-98	5	15-Apr-98	16	22-Jun-98	8	04-Sep-98	21	7/30/98 3:00	
07-Feb-98	0	16-Apr-98	5	23-Jun-98	4	05-Sep-98	12	7/30/98 6:00	
08-Feb-98	74	17-Apr-98	18	24-Jun-98	0.2	06-Sep-98	26	7/30/98 9:00	
09-Feb-98	13	18-Apr-98	3	25-Jun-98	7	07-Sep-98	32	7/30/98 12:00	4
10-Feb-98	1	19-Apr-98	0	26-Jun-98	6	08-Sep-98	14	7/30/98 15:00	7
11-Feb-98	0	20-Apr-98	11	28-Jun-98	0.2	09-Sep-98	5	7/30/98 21:00	
12-Feb-98	0.4	21-Apr-98	15	29-Jun-98	17	10-Sep-98	0	7/31/98 0:00	
13-Feb-98	14	22-Apr-98	52	30-Jun-98	3	11-Sep-98	0	9/14/98 3:00	2
14-Feb-98	10	23-Apr-98	3	July		12-Sep-98	0.6	9/14/98 6:00	0.8
15-Feb-98	4	24-Apr-98	0.4	01-Jul-98	6	13-Sep-98	8	9/14/98 9:00	1
16-Feb-98	3	25-Apr-98	0.2	02-Jul-98	9	14-Sep-98	15	9/14/98 12:00	2
17-Feb-98	12	26-Apr-98	0.4	03-Jul-98	4	23-Sep-98	75	9/14/98 15:00	10
18-Feb-98	26	27-Apr-98	6	05-Jul-98	1	24-Sep-98	15	9/14/98 18:00	1
20-Feb-98	30	28-Apr-98	0.8	06-Jul-98	60	25-Sep-98	14	9/14/98 21:00	0.2
21-Feb-98	18	30-Apr-98	12	08-Jul-98	48	26-Sep-98	0.2	9/22/98 12:00	2
22-Feb-98	31	May		09-Jul-98	10	27-Sep-98	5	9/22/98 15:00	4
23-Feb-98	21	01-May-98	7	10-Jul-98	0	28-Sep-98	4	9/22/98 18:00	5
24-Feb-98	0	02-May-98	5	11-Jul-98	4	29-Sep-98	3	9/22/98 21:00	14
25-Feb-98	0	03-May-98	0.4	12-Jul-98	37	30-Sep-98	18	9/23/98 0:00	9
26-Feb-98	0	04-May-98	0	13-Jul-98	16	October		9/23/98 3:00	16
27-Feb-98	8	05-May-98	0	14-Jul-98	0.4	01-Oct-98	8	9/23/98 9:00	5
28-Feb-98	2	06-May-98	0	15-Jul-98	12	02-Oct-98	57	9/23/98 12:00	3
March		07-May-98	24	16-Jul-98	6	03-Oct-98	7	9/23/98 15:00	3
01-Mar-98	10	08-May-98	4	17-Jul-98	13	04-Oct-98	45	9/23/98 18:00	5
02-Mar-98	0.2	09-May-98	4	18-Jul-98	2	05-Oct-98	11	9/23/98 21:00	4
03-Mar-98	0	10-May-98	22	19-Jul-98	0	06-Oct-98	21	9/24/98 0:00	0.4
04-Mar-98	5	11-May-98	33	20-Jul-98	0.2	07-Oct-98	12	9/24/98 3:00	
05-Mar-98	13	12-May-98	31	21-Jul-98	0	08-Oct-98	12	9/24/98 6:00	
06-Mar-98	76	13-May-98	6	22-Jul-98	7	09-Oct-98	8	9/24/98 9:00	
07-Mar-98	3	14-May-98	0.2	23-Jul-98	13	10-Oct-98	0.2	9/24/98 12:00	9
08-Mar-98	0	15-May-98	0.2	24-Jul-98	3	11-Oct-98	24	9/24/98 15:00	5
10-Mar-98	0	16-May-98	1	25-Jul-98	1	12-Oct-98	16	9/24/98 18:00	0.4
11-Mar-98	0	17-May-98	0	27-Jul-98	6	13-Oct-98	17	9/24/98 21:00	
12-Mar-98	0	18-May-98	0.2	28-Jul-98	8	14-Oct-98	4	9/25/98 0:00	
13-Mar-98	0.8	19-May-98	0.4	29-Jul-98	48	15-Oct-98	29		
14-Mar-98	24	20-May-98	17	30-Jul-98	3	16-Oct-98	0		
15-Mar-98	11	21-May-98	26	31-Jul-98	31	17-Oct-98	0		
16-Mar-98	4	22-May-98	15			18-Oct-98	15		
17-Mar-98	15	23-May-98	21			19-Oct-98	20		

Rainfall measured in mm.

Appendix 6

Mass Loading Calculations

Intensive Sampling - (22/9/98 - 23/9/98), Mass Loadings

Time	Al	Flow @	Flow L/s	ML mg/s	ML kg/s	ML kg/day	ML kg/y	ML t/y
6:20 PM	1.09	0.077	76.79	83.70	0.00	7.23	2641.24	2.64
19:05	1.05	0.337	336.80	353.64	0.00	30.55	11160.02	11.16
7:45 PM	0.42	1.047	1046.69	439.61	0.00	37.98	13873.08	13.87
20:20	0.38	0.864	864.47	328.50	0.00	28.38	10366.64	10.37
21:05	0.45	0.625	624.83	281.17	0.00	24.29	8873.21	8.87
21:55	0.59	0.541	540.96	319.17	0.00	27.58	10072.18	10.07
22:40	0.61	0.673	673.11	410.59	0.00	35.48	12957.38	12.96
23:25	0.51	0.593	592.79	302.32	0.00	26.12	9540.59	9.54
13:30	0.90	0.368	367.87	331.08	0.00	28.61	10448.18	10.45
Cd								
6:20 PM	0.028	0.077	76.79	2.15	0.00	0.19	67.85	0.07
19:05	0.024	0.337	336.80	8.08	0.00	0.70	255.09	0.26
7:45 PM	0.010	1.047	1046.69	10.47	0.00	0.90	330.31	0.33
20:20	0.009	0.864	864.47	7.78	0.00	0.67	245.53	0.25
21:05	0.011	0.625	624.83	6.87	0.00	0.59	216.90	0.22
21:55	0.014	0.541	540.96	7.57	0.00	0.65	239.00	0.24
22:40	0.015	0.673	673.11	10.10	0.00	0.87	318.62	0.32
23:25	0.013	0.593	592.79	7.71	0.00	0.67	243.19	0.24
13:30	0.026	0.368	367.87	9.56	0.00	0.83	301.84	0.30
Cu								
6:20 PM	0.324	0.077	76.79	24.88	0.00	2.15	785.10	0.79
19:05	0.294	0.337	336.80	99.02	0.00	8.56	3124.80	3.12
7:45 PM	0.108	1.047	1046.69	113.04	0.00	9.77	3567.36	3.57
20:20	0.083	0.864	864.47	71.75	0.00	6.20	2264.29	2.26
21:05	0.108	0.625	624.83	67.48	0.00	5.83	2129.57	2.13
21:55	0.147	0.541	540.96	79.52	0.00	6.87	2509.51	2.51
22:40	0.171	0.673	673.11	115.10	0.00	9.94	3632.32	3.63
23:25	0.126	0.593	592.79	74.69	0.00	6.45	2357.09	2.36
13:30	0.340	0.368	367.87	125.08	0.00	10.81	3947.09	3.95
Fe								
6:20 PM	0.61	0.077	76.79	46.84	0.00	4.05	1478.13	1.48
19:05	1.25	0.337	336.80	421.00	0.00	36.37	13285.73	13.29
7:45 PM	1.98	1.047	1046.69	2072.45	0.00	179.06	65401.67	65.40
20:20	0.65	0.864	864.47	561.91	0.00	48.55	17732.41	17.73
21:05	0.44	0.625	624.83	274.93	0.00	23.75	8676.03	8.68
21:55	0.40	0.541	540.96	216.39	0.00	18.70	6828.60	6.83
22:40	0.47	0.673	673.11	316.36	0.00	27.33	9983.56	9.98
23:25	0.51	0.593	592.79	302.32	0.00	26.12	9540.59	9.54
13:30	0.83	0.368	367.87	305.33	0.00	26.38	9635.55	9.64
Mn								
6:20 PM	4.32	0.077	76.79	331.71	0.00	28.66	10468.04	10.47
19:05	3.50	0.337	336.80	1178.80	0.00	101.85	37200.05	37.20
7:45 PM	1.70	1.047	1046.69	1779.38	0.00	153.74	56152.95	56.15
20:20	1.30	0.864	864.47	1123.81	0.00	97.10	35464.81	35.46
21:05	1.54	0.625	624.83	962.24	0.00	83.14	30366.09	30.37
21:55	1.86	0.541	540.96	1006.19	0.00	86.93	31752.99	31.75
22:40	1.89	0.673	673.11	1272.17	0.00	109.92	40146.64	40.15
23:25	1.61	0.593	592.79	954.39	0.00	82.46	30118.35	30.12
13:30	2.60	0.368	367.87	956.46	0.00	82.64	30183.64	30.18

	Pb							
6:20 PM	1.50	0.077	76.79	115.18	0.00	9.95	3634.74	3.63
19:05	1.42	0.337	336.80	478.26	0.00	41.32	15092.59	15.09
7:45 PM	0.72	1.047	1046.69	753.62	0.00	65.11	23782.43	23.78
20:20	0.85	0.864	864.47	734.80	0.00	63.49	23188.53	23.19
21:05	0.77	0.625	624.83	481.12	0.00	41.57	15183.04	15.18
21:55	0.79	0.541	540.96	427.36	0.00	36.92	13486.48	13.49
22:40	0.87	0.673	673.11	585.60	0.00	50.60	18480.20	18.48
23:25	0.67	0.593	592.79	397.17	0.00	34.32	12533.72	12.53
13:30	0.85	0.368	367.87	312.69	0.00	27.02	9867.73	9.87
	Sulphate							
6:20 PM	80	0.077	76.79	6142.82	0.01	530.74	193852.60	193.85
19:05	71	0.337	336.80	23912.77	0.02	2066.06	754629.66	754.63
7:45 PM	31	1.047	1046.69	32447.51	0.03	2803.46	1023965.59	1023.97
20:20	26	0.864	864.47	22476.24	0.02	1941.95	709296.25	709.30
21:05	33	0.625	624.83	20619.50	0.02	1781.52	650701.88	650.70
21:55	42	0.541	540.96	22720.45	0.02	1963.05	717002.98	717.00
22:40	43	0.673	673.11	28943.56	0.03	2500.72	913389.17	913.39
23:25	37	0.593	592.79	21933.25	0.02	1895.03	692160.80	692.16
13:30	65	0.368	367.87	23911.54	0.02	2065.96	754590.95	754.59
	Zn							
6:20 PM	14.5	0.077	76.79	1113.39	0.00	96.20	35135.78	35.14
19:05	12.1	0.337	336.80	4075.28	0.00	352.10	128605.90	128.61
7:45 PM	4.38	1.047	1046.69	4584.52	0.00	396.10	144676.43	144.68
20:20	3.99	0.864	864.47	3449.24	0.00	298.01	108849.69	108.85
21:05	5.37	0.625	624.83	3355.35	0.00	289.90	105886.94	105.89
21:55	7.33	0.541	540.96	3965.26	0.00	342.60	125134.09	125.13
22:40	7.48	0.673	673.11	5034.83	0.01	435.01	158887.23	158.89
23:25	6.17	0.593	592.79	3657.52	0.00	316.01	115422.49	115.42
13:30	12.6	0.368	367.87	4635.16	0.00	400.48	146274.55	146.27
Trace metals Cu +Pb+Zn+Cd		Flow @	Flow L/s	ML mg/s	ML kg/s	ML kg/day	ML kg/y	ML t/y
6:20 AM	16.352	0.076785229	76.79	1255.59	0.00	108.48	39623.47	39.62
19:05	13.838	0.336799592	336.80	4660.63	0.00	402.68	147078.38	147.08
7:45 PM	5.218	1.046693921	1046.69	5461.65	0.01	471.89	172356.53	172.36
20:20	4.932	0.86447084	864.47	4263.57	0.00	368.37	134548.04	134.55
21:05	6.259	0.624833286	624.83	3910.83	0.00	337.90	123416.46	123.42
21:55	8.281	0.540963173	540.96	4479.72	0.00	387.05	141369.09	141.37
22:40	8.536	0.673105955	673.11	5745.63	0.01	496.42	181318.37	181.32
23:25	6.979	0.592790604	592.79	4137.09	0.00	357.44	130556.49	130.56
13:30	13.816	0.367869914	367.87	5082.49	0.01	439.13	160391.21	160.39

Long Term Sampling - Mass Loads

Date	Location	Element	Flow @	Flow L/s	ML mg/s	ML kg/s	ML kg/day	ML kg/y	ML t/y
		Al							
5/04/98	bcgs	1.05	0.14	143.38	150.55	0.0002	13.01	4751.01	4.75
3/06/98	BCGS	1.19	0.07	71.75	85.38	0.0001	7.38	2694.33	2.69
31/08/98	bcgs	2.49	0.04	37.04	92.23	0.0001	7.97	2910.67	2.91
31/08/98	weir	0.25	0.01	5.23	1.31	0.0000	0.11	41.23	0.04
31/08/98	wfall	0.26	0.01	5.23	1.36	0.0000	0.12	42.88	0.04
		Cd							
5/04/98	bcgs	0.11	0.14	143.38	15.77	0.0000	1.36	497.72	0.50
3/06/98	BCGS	0.028	0.07	71.75	2.01	0.0000	0.17	63.40	0.06
31/08/98	bcgs	0.072	0.04	37.04	2.67	0.0000	0.23	84.16	0.08
31/08/98	weir	0.029	0.01	5.23	0.15	0.0000	0.01	4.78	0.00
31/08/98	wfall	0.013	0.01	5.23	0.07	0.0000	0.01	2.14	0.00
		Cu							
5/04/98	bcgs	0.24	0.14	143.38	34.41	0.0000	2.97	1085.94	1.09
3/06/98	BCGS	0.29	0.07	71.75	20.81	0.0000	1.80	656.60	0.66
31/08/98	bcgs	0.66	0.04	37.04	24.45	0.0000	2.11	771.50	0.77
31/08/98	weir	0.006	0.01	5.23	0.03	0.0000	0.00	0.99	0.00
31/08/98	wfall	0.015	0.01	5.23	0.08	0.0000	0.01	2.47	0.00
		Fe							
5/04/98	bcgs	1.24	0.14	143.38	177.79	0.0002	15.36	5610.71	5.61
3/06/98	BCGS	0.58	0.07	71.75	41.61	0.0000	3.60	1313.20	1.31
31/08/98	bcgs	1.37	0.04	37.04	50.75	0.0001	4.38	1601.45	1.60
31/08/98	weir	0.16	0.01	5.23	0.84	0.0000	0.07	26.39	0.03
31/08/98	wfall	0.39	0.01	5.23	2.04	0.0000	0.18	64.32	0.06
		Mn							
5/04/98	bcgs	3.68	0.14	143.38	527.64	0.0005	45.59	16651.15	16.65
3/06/98	BCGS	5.07	0.07	71.75	363.75	0.0004	31.43	11479.21	11.48
31/08/98	bcgs	9.20	0.04	37.04	340.78	0.0003	29.44	10754.27	10.75
31/08/98	weir	0.12	0.01	5.23	0.63	0.0000	0.05	19.79	0.02
31/08/98	wfall	0.87	0.01	5.23	4.55	0.0000	0.39	143.48	0.14
		Pb							
5/04/98	bcgs	1.18	0.14	143.38	169.19	0.0002	14.62	5339.23	5.34
3/06/98	BCGS	1.58	0.07	71.75	113.36	0.0001	9.79	3577.35	3.58
31/08/98	bcgs	1.65	0.04	37.04	61.12	0.0001	5.28	1928.76	1.93
31/08/98	weir	0.04	0.01	5.23	0.21	0.0000	0.02	6.60	0.01
31/08/98	wfall	0.158	0.01	5.23	0.83	0.0000	0.07	26.06	0.03
Sulphate									
5/04/98	bcgs	116	0.14	143.38	16632.23	0.0166	1437.02	524873.15	524.87
3/06/98	BCGS	178	0.07	71.75	12770.86	0.0128	1103.40	403017.60	403.02
31/08/98	bcgs	174	0.04	37.04	6445.23	0.0064	556.87	203396.01	203.40
31/08/98	weir	8.7	0.01	5.23	45.47	0.0000	3.93	1434.84	1.43
31/08/98	wfall	24	0.01	5.23	125.43	0.0001	10.84	3958.17	3.96

		Zn							
5/04/98	bcgs	10.8	0.14	143.38	1548.52	0.0015	133.79	48867.50	48.87
3/06/98	BCGS	16.0	0.07	71.75	1147.94	0.0011	99.18	36228.30	36.23
31/08/98	bcgs	40.7	0.04	37.04	1507.59	0.0015	130.26	47575.96	47.58
31/08/98	weir	0.52	0.01	5.23	2.72	0.0000	0.23	85.76	0.09
31/08/98	wfall	6.53	0.01	5.23	34.13	0.0000	2.95	1076.95	1.08

Appendix 7

Geochemical Modelling results with PHREEQC

Intensive Sampling Low Redox (July 28th)

Reading data base.

SOLUTION_MASTER_SPECIES
SOLUTION_SPECIES
PHASES
EXCHANGE_MASTER_SPECIES
EXCHANGE_SPECIES
SURFACE_MASTER_SPECIES
SURFACE_SPECIES
END

Reading input data for simulation 1

TITLE Ex1a - Bakers Creek Intensive sampling program - Jul-98
SOLUTION 1 Start of rainfall event: T = 2PM
units ppm
pH 4.34
pe 4.527
density 1.023
temp 8.4
Ca 5.15
Mg 2.12
Na 4.21
K 0.74
Fe 2.98
Mn 4.04
Si 2.18
Cl 4.9
As 0.048
Al 1.52
Cd 0.032
Cu 0.025
Pb 2.62
Zn 11.2
S(6) 24.34
O(0) 1 O2(g) -0.7
SOLUTION_MASTER_SPECIES
SOLUTION_SPECIES
PHASES
END

TITLE

Ex1a - Bakers Creek Intensive sampling program - July 1998

Beginning of initial solution calculations.

Initial solution 1 Start of rainfall event: T = 2PM

#NAME? composition

Elements	Molality	Moles
Al	5.83E-05	5.83E-05
As	6.41E-07	6.41E-07
Ca	1.29E-04	1.29E-04
Cd	2.85E-07	2.85E-07
Cl	1.38E-04	1.38E-04
Cu	3.93E-07	3.93E-07
Fe	5.30E-05	5.30E-05
K	1.89E-05	1.89E-05
Mg	8.72E-05	8.72E-05
Mn	7.35E-05	7.35E-05
Na	1.83E-04	1.83E-04
O(0)	5.38E-04	5.38E-04 Equilibrium with O2(g)
Pb	1.27E-05	1.27E-05
S(6)	2.53E-04	2.53E-04
Si	3.60E-05	3.60E-05
Zn	1.71E-04	1.71E-04

#NAME? of solution-----

pH = 4.34

pe = 4.527

Activity of water = 1

Ionic strength = 1.87E-03

Mass of water (kg) = 1.00E+00

Total alkalinity (eq/kg) = -4.63E-05

Total carbon (mol/kg) = 0.00E+00

Total CO2 (mol/kg) = 0.00E+00

Temperature (deg C) = 8.4

Electrical balance (eq) = 8.28E-04

Iterations = 4

Total H = 1.11E+02

Total O = 5.55E+01

#NAME? couples-----

Redox couple pe Eh (volts)

O(-2)/O(0) 17.9309 0.9946

#NAME? of species-----

	Log Species	Log Molality	Log Activity	Molality	Activity	Gamma
	H+	4.78E-05	4.57E-05	-4.321	-4.34	-0.019
	OH-	4.84E-11	4.62E-11	-10.315	-10.338	-0.02
	H2O	5.55E+01	1.00E+00	0	0	0
Al	5.83E-05					
	Al+3	4.12E-05	2.71E-05	-4.385	-4.568	-0.183
	AlSO4+	1.30E-05	1.24E-05	-4.886	-4.908	-0.02
	AlOH+2	2.01E-06	1.87E-06	-5.697	-5.778	-0.081
	Al(SO4)2-	7.04E-08	6.72E-08	-7.152	-7.173	-0.02
	Al(OH)2+	4.86E-08	4.64E-08	-7.313	-7.334	-0.02
	AlHSO4+2	5.41E-11	4.49E-11	-10.267	-10.348	-0.081
	Al(OH)3	3.15E-11	3.15E-11	-10.502	-10.501	0
	Al(OH)4-	1.18E-12	1.12E-12	-11.929	-11.949	-0.02
As(3)	4.65E-07					
	H3AsO3	4.65E-07	4.65E-07	-6.333	-6.332	0
	H4AsO3+	1.10E-11	1.05E-11	-10.957	-10.977	-0.02
	H2AsO3-	3.02E-12	2.88E-12	-11.52	-11.54	-0.02
	HAsO3-2	2.55E-20	2.11E-20	-19.594	-19.675	-0.081
	AsO3-3	1.38E-29	9.04E-30	-28.861	-29.044	-0.183
As(5)	1.76E-07					
	H2AsO4-	1.74E-07	1.66E-07	-8.76	-8.78	-0.02
	H3AsO4	1.10E-09	1.10E-09	-8.959	-8.959	0
	HAsO4-2	7.01E-10	5.82E-10	-9.154	-9.235	-0.081
	AsO4-3	3.01E-17	1.98E-17	-16.521	-16.704	-0.183
Ca	1.29E-04					
	Ca+2	1.25E-04	1.04E-04	-3.902	-3.982	-0.08
	CaSO4	3.24E-08	3.24E-08	-5.489	-5.489	0
	CaHSO4+	7.55E-10	7.20E-10	-9.122	-9.143	-0.02
	CaOH+	3.97E-13	3.78E-13	-12.402	-12.422	-0.02
Cd	2.85E-07					
	Cd+2	2.71E-07	2.25E-07	-6.567	-6.848	-0.081
	CdSO4	1.08E-08	1.08E-08	-7.987	-7.987	0
	CdCl+	2.78E-09	2.65E-09	-8.557	-8.577	-0.02
	Cd(SO4)2-2	3.02E-11	2.51E-11	-10.52	-10.601	-0.081
	CdCl2	1.35E-12	1.35E-12	-11.869	-11.869	0
	CdOH+	9.85E-14	9.40E-14	-13.007	-13.027	-0.02
	CdOHCl	1.57E-14	1.57E-14	-13.805	-13.804	0
	CdCl3-	8.74E-17	8.35E-17	-16.058	-16.079	-0.02
	Cd(OH)2	4.81E-19	4.81E-19	-18.318	-18.318	0
	Cd2OH+3	2.02E-19	1.33E-19	-18.695	-18.877	-0.183
	Cd(OH)3-	1.24E-27	1.18E-27	-26.908	-26.928	-0.02
	Cd(OH)4-2	2.77E-37	2.30E-37	-36.557	-36.638	-0.081
Cl	1.38E-04					
	Cl-	1.38E-04	1.32E-04	-3.86	-3.88	-0.02
	MnCl+	3.37E-08	3.21E-08	-7.473	-7.493	-0.02
	PbCl+	3.23E-08	3.09E-08	-7.49	-7.511	-0.02
	ZnCl+	2.14E-08	2.04E-08	-7.671	-7.691	-0.02
	FeCl+	8.22E-09	7.85E-09	-8.085	-8.105	-0.02
	CdCl+	2.78E-09	2.65E-09	-8.557	-8.577	-0.02

	CuCl+	4.42E-11	4.22E-11	-10.355	-10.375	-0.02
	CuCl2-	2.48E-11	2.34E-11	-10.81	-10.83	-0.02
	ZnOHCl	1.32E-11	1.32E-11	-10.881	-10.881	0
	PbCl2	9.33E-12	9.34E-12	-11.03	-11.03	0
	ZnCl2	2.80E-12	2.80E-12	-11.588	-11.588	0
	MnCl2	1.85E-12	1.85E-12	-11.733	-11.733	0
	CdCl2	1.35E-12	1.35E-12	-11.889	-11.889	0
	CdOHCl	1.57E-14	1.57E-14	-13.805	-13.804	0
	CuCl3-2	5.48E-15	4.53E-15	-14.282	-14.344	-0.081
	CuCl2	2.41E-15	2.41E-15	-14.818	-14.818	0
	PbCl3-	9.08E-16	8.85E-16	-15.043	-15.063	-0.02
	ZnCl3-	3.57E-16	3.41E-16	-15.447	-15.467	-0.02
	FeCl+2	1.20E-16	9.92E-17	-15.922	-16.003	-0.081
	CdCl3-	8.74E-17	8.35E-17	-16.058	-16.079	-0.02
	MnCl3-	7.03E-17	6.71E-17	-16.153	-16.173	-0.02
	FeCl2+	1.15E-19	1.10E-19	-18.94	-18.981	-0.02
	PbCl4-2	5.64E-20	4.88E-20	-19.249	-19.33	-0.081
	ZnCl4-2	2.32E-20	1.92E-20	-19.635	-19.716	-0.081
	CuCl3-	8.30E-22	7.92E-22	-21.081	-21.101	-0.02
	FeCl3	1.44E-24	1.44E-24	-23.841	-23.841	0
	CuCl4-2	3.99E-28	3.31E-28	-27.4	-27.481	-0.081
Cu(1)	4.29E-09					
	Cu+	4.28E-09	4.07E-09	-8.37	-8.39	-0.02
	CuCl2-	2.48E-11	2.34E-11	-10.81	-10.83	-0.02
Cu(2)	3.89E-07					
	Cu+2	3.79E-07	3.14E-07	-6.422	-6.503	-0.081
	CuSO4	1.05E-08	1.05E-08	-7.979	-7.979	0
	CuOH+	7.20E-11	6.87E-11	-10.143	-10.163	-0.02
	CuCl+	4.42E-11	4.22E-11	-10.355	-10.375	-0.02
	Cu(OH)2	3.14E-12	3.14E-12	-11.503	-11.503	0
	CuCl2	2.41E-15	2.41E-15	-14.818	-14.818	0
	Cu2(OH)2+2	3.47E-16	2.88E-16	-15.459	-15.54	-0.081
	Cu(OH)3-	4.34E-21	4.14E-21	-20.363	-20.383	-0.02
	CuCl3-	8.30E-22	7.92E-22	-21.081	-21.101	-0.02
	CuCl4-2	3.99E-28	3.31E-28	-27.4	-27.481	-0.081
	Cu(OH)4-2	2.18E-29	1.81E-29	-28.662	-28.743	-0.081
Fe(2)	5.30E-05					
	Fe+2	5.20E-05	4.31E-05	-4.284	-4.365	-0.081
	FeSO4	1.00E-06	1.00E-06	-6	-5.999	0
	FeCl+	8.22E-09	7.85E-09	-8.085	-8.105	-0.02
	FeHSO4+	3.12E-10	2.98E-10	-9.505	-9.528	-0.02
	FeOH+	7.10E-11	6.78E-11	-10.149	-10.169	-0.02
	Fe(OH)2	2.25E-18	2.25E-18	-17.648	-17.648	0
	Fe(OH)3-	1.58E-24	1.50E-24	-23.803	-23.823	-0.02
Fe(3)	9.97E-12					
	Fe(OH)2+	7.35E-12	7.01E-12	-11.134	-11.154	-0.02
	FeOH+2	2.48E-12	2.05E-12	-11.608	-11.887	-0.081
	Fe+3	7.12E-14	4.88E-14	-13.148	-13.33	-0.183
	FeSO4+	6.50E-14	6.20E-14	-13.187	-13.208	-0.02
	Fe(OH)3	8.32E-15	8.32E-15	-14.08	-14.08	0
	Fe(SO4)2-	2.47E-16	2.38E-16	-15.608	-15.628	-0.02
	FeCl+2	1.20E-16	9.92E-17	-15.922	-16.003	-0.081
	FeHSO4+2	9.78E-18	8.11E-18	-17.01	-17.091	-0.081
	FeCl2+	1.15E-19	1.10E-19	-18.94	-18.981	-0.02
	Fe(OH)4-	7.84E-20	7.48E-20	-19.108	-19.128	-0.02
	Fe2(OH)2+4	5.44E-22	2.58E-22	-21.264	-21.589	-0.324
	FeCl3	1.44E-24	1.44E-24	-23.841	-23.841	0
	Fe3(OH)4+5	7.57E-30	2.35E-30	-29.121	-29.628	-0.507
H(0)	2.74E-21					
	H2	1.37E-21	1.37E-21	-20.864	-20.863	0
K	1.89E-05					
	K+	1.89E-05	1.80E-05	-4.723	-4.744	-0.02
	KSO4-	1.78E-08	1.68E-08	-7.754	-7.774	-0.02
Mg	8.72E-05					
	Mg+2	8.53E-05	7.11E-05	-4.069	-4.148	-0.079
	MgSO4	1.88E-06	1.88E-06	-5.727	-5.727	0
	MgOH+	9.87E-13	9.42E-13	-12.008	-12.028	-0.02
Mn(2)	7.35E-05					
	Mn+2	7.21E-05	5.99E-05	-4.142	-4.223	-0.081
	MnSO4	1.37E-06	1.37E-06	-5.864	-5.864	0
	MnCl+	3.37E-08	3.21E-08	-7.473	-7.493	-0.02
	MnOH+	7.00E-12	6.68E-12	-11.155	-11.175	-0.02
	MnCl2	1.85E-12	1.85E-12	-11.733	-11.733	0
	MnCl3-	7.03E-17	6.71E-17	-16.153	-16.173	-0.02
	Mn(OH)3-	1.04E-26	9.93E-27	-25.983	-26.003	-0.02

Mn(3)	5.23E-27					
Mn(6)	Mn+3	5.23E-27	3.43E-27	-26.282	-26.484	-0.183
	MnO4-2	0.00E+00	0.00E+00	-77.07	-77.151	-0.081
Mn(7)	0.00E+00					
	MnO4-	0.00E+00	0.00E+00	-83.285	-83.308	-0.02
Na	1.83E-04					
	Na+	1.83E-04	1.75E-04	-3.738	-3.758	-0.02
	NaSO4-	1.52E-07	1.45E-07	-8.819	-8.839	-0.02
O(0)	5.38E-04					
	O2	2.69E-04	2.69E-04	-3.57	-3.57	0
Pb	1.27E-05					
	Pb+2	1.16E-05	9.62E-06	-4.936	-5.017	-0.081
	PbSO4	1.02E-06	1.02E-06	-5.994	-5.994	0
	PbCl+	3.23E-08	3.09E-08	-7.49	-7.511	-0.02
	PbOH+	4.30E-09	4.10E-09	-8.387	-8.387	-0.02
	Pb(SO4)2-2	1.21E-09	1.00E-09	-8.919	-9	-0.081
	PbCl2	9.33E-12	9.34E-12	-11.03	-11.03	0
	Pb2OH+3	1.35E-12	8.84E-13	-11.871	-12.054	-0.183
	Pb(OH)2	3.49E-14	3.49E-14	-13.457	-13.457	0
	PbCl3-	9.08E-16	8.65E-16	-15.043	-15.083	-0.02
	PbCl4-2	5.64E-20	4.68E-20	-19.249	-19.33	-0.081
	Pb(OH)3-	9.19E-21	8.77E-21	-20.037	-20.057	-0.02
	Pb3(OH)4+2	1.65E-23	1.37E-23	-22.782	-22.863	-0.081
	Pb(OH)4-2	5.30E-28	4.40E-28	-27.276	-27.357	-0.081
S(8)	2.53E-04					
	SO4-2	2.28E-04	1.88E-04	-3.646	-3.727	-0.08
	AlSO4+	1.30E-05	1.24E-05	-4.886	-4.906	-0.02
	ZnSO4	5.20E-06	5.20E-06	-5.284	-5.284	0
	CaSO4	3.24E-06	3.24E-06	-5.489	-5.489	0
	MgSO4	1.88E-06	1.88E-06	-5.727	-5.727	0
	MnSO4	1.37E-06	1.37E-06	-5.884	-5.884	0
	PbSO4	1.02E-06	1.02E-06	-5.994	-5.994	0
	FeSO4	1.00E-06	1.00E-06	-6	-5.999	0
	HSO4-	6.02E-07	5.75E-07	-6.22	-6.241	-0.02
	NaSO4-	1.52E-07	1.45E-07	-8.819	-8.839	-0.02
	Al(SO4)2-	7.04E-08	6.72E-08	-7.152	-7.173	-0.02
	KSO4-	1.76E-08	1.68E-08	-7.754	-7.774	-0.02
	Zn(SO4)2-2	1.12E-08	9.25E-09	-7.953	-8.034	-0.081
	CdSO4	1.08E-08	1.08E-08	-7.967	-7.967	0
	CuSO4	1.05E-08	1.05E-08	-7.979	-7.979	0
	Pb(SO4)2-2	1.21E-09	1.00E-09	-8.919	-9	-0.081
	CaHSO4+	7.55E-10	7.20E-10	-9.122	-9.143	-0.02
	FeHSO4+	3.12E-10	2.98E-10	-9.505	-9.526	-0.02
	AlHSO4+2	5.41E-11	4.49E-11	-10.287	-10.348	-0.081
	Cd(SO4)2-2	3.02E-11	2.51E-11	-10.52	-10.601	-0.081
	FeSO4+	6.50E-14	6.20E-14	-13.187	-13.208	-0.02
	Fe(SO4)2-	2.47E-16	2.36E-16	-15.608	-15.628	-0.02
	FeHSO4+2	9.78E-18	8.11E-18	-17.01	-17.091	-0.081
Si	3.60E-05					
	H4SiO4	3.60E-05	3.60E-05	-4.444	-4.444	0
	H3SiO4-	5.84E-11	5.57E-11	-10.234	-10.254	-0.02
	H2SiO4-2	2.58E-20	2.14E-20	-19.588	-19.669	-0.081
Zn	1.71E-04					
	Zn+2	1.66E-04	1.38E-04	-3.78	-3.861	-0.081
	ZnSO4	5.20E-06	5.20E-06	-5.284	-5.284	0
	ZnCl+	2.14E-08	2.04E-08	-7.871	-7.891	-0.02
	Zn(SO4)2-2	1.12E-08	9.25E-09	-7.953	-8.034	-0.081
	ZnOH+	7.69E-10	7.34E-10	-9.114	-9.134	-0.02
	ZnOHCl	1.32E-11	1.32E-11	-10.881	-10.881	0
	ZnCl2	2.60E-12	2.60E-12	-11.586	-11.586	0
	Zn(OH)2	8.30E-13	8.30E-13	-12.081	-12.081	0
	ZnCl3-	3.57E-16	3.41E-16	-15.447	-15.487	-0.02
	Zn(OH)3-	8.02E-20	5.74E-20	-19.22	-19.241	-0.02
	ZnCl4-2	2.32E-20	1.92E-20	-19.635	-19.716	-0.081
	Zn(OH)4-2	2.40E-28	1.99E-28	-27.62	-27.701	-0.081

#NAME? indices-----

Phase	SI	log	IAP	log	KT
CuprousFermi		4.37	6.49	2.12 CuFeO2	
Hematite		1.88	28.36	24.48 Fe2O3	
Goethite		0.52	13.18	12.68 FeOOH	
Diaspore		0.37	8.45	8.08 AlOOH	
Fe(OH)2.7Cl:		0.26	10.72	10.45 Fe(OH)2.7ClO.3	
Quartz		-0.17	-4.44	-4.27 SiO2	

Cristobalite	-0.59	-4.44	-3.86 SiO2
Chalcedony	-0.66	-4.44	-3.78 SiO2
O2(g)	-0.7	-3.57	-2.87 O2
Jarosite	-0.72	-3.95	-3.23 Al(OH)SO4
Gibbsite	-0.77	8.45	9.22 Al(OH)3
Anglesite	-0.85	-8.74	-7.89 PbSO4
Alunite	-0.91	0.14	1.05 KAl3(SO4)2(OH)8
Kaolinite	-1.14	8.02	9.18 Al2Si2O5(OH)4
Silicagel	-1.21	-4.44	-3.23 SiO2
Basaluminite	-1.3	21.4	22.7 Al(OH)10SO4
Boehmite	-1.51	8.45	9.96 AlOOH
SiO2(a)	-1.57	-4.44	-2.87 SiO2
Pyrophyllite	-2.04	-0.87	1.17 Al2Si4O10(OH)2
Magnetite	-2.5	30.68	33.18 Fe3O4
Gypsum	-3.11	-7.71	-4.6 CaSO4:2H2O
CuMetal	-3.32	-15.56	-12.23 Cu
Anhydrite	-3.37	-7.71	-4.34 CaSO4
ZnSiO3	-3.45	0.38	3.82 ZnSiO3
Al(OH)3(a)	-3.84	8.45	12.09 Al(OH)3
Kmica	-3.98	11.62	15.6 KAl3Si3O10(OH)2
Nantokite	-5.02	-14.91	-9.89 CuCl
Lamarkite	-5.11	-5.08	0.03 PbO:PbSO4
Pb(OH)2	-5.17	3.66	8.83 Pb(OH)2
Laurionite	-5.18	-4.56	0.62 PbOHCl
Fe(OH)3(a)	-5.2	13.18	18.38 Fe(OH)3
AlAsO4:2H2O	-5.43	-0.51	4.93 AlAsO4:2H2O
Goslarite	-5.47	-7.59	-2.12 ZnSO4:7H2O
Epsomite	-5.6	-7.87	-2.28 MgSO4:7H2O
Montmorillonit	-5.62	4.16	9.77 Ca0.165Al2.33Si3.67O10(OH)2
Melanterite	-5.63	-8.09	-2.46 FeSO4:7H2O
Beidelite	-5.78	3.65	9.43 (NaKMg0.5)0.11Al2.33Si3.67O10(OH)2
Blanchite	-5.83	-7.59	-1.76 ZnSO4:6H2O
Tenorite	-6.19	2.18	8.36 CuO
CupricFerriite	-6.21	28.54	34.75 CuFe2O4
Cuprite	-6.25	-13.36	-7.13 Cu2O
Halkosite	-6.43	8.02	14.44 Al2Si2O5(OH)4
ZnO(a)	-6.49	4.82	11.31 ZnO
Zn(OH)2-a	-6.68	4.82	11.5 Zn(OH)2
Montmorillonit	-6.87	0.3	7.17 (HNaK)0.09Mg0.29Fe0.24Al1.57Si3.93O10(OH)2
Zn(OH)2-g	-6.89	4.82	11.71 Zn(OH)2
Zn(OH)2-b	-6.93	4.82	11.75 Zn(OH)2
Maghemite	-7.01	26.36	33.37 Fe2O3
Cu(OH)2	-7.21	2.18	9.38 Cu(OH)2
Zn(OH)2-c	-7.38	4.82	12.2 Zn(OH)2
Zincite(c)	-7.39	4.82	12.21 ZnO
Chalcantite	-7.52	-10.23	-2.71 CuSO4:5H2O
ZnSO4:H2O	-7.54	-7.59	-0.05 ZnSO4:H2O
Zn(OH)2-a	-7.63	4.82	12.45 Zn(OH)2
Cotunnite	-7.73	-12.78	-5.04 PbCl2
Adularia	-7.95	-5.28	2.67 KAlSi3O8
PbSiO3	-8.55	-0.78	7.77 PbSiO3
CdSO4:2.7H2	-8.71	-10.37	-1.66 CdSO4:2.67H2O
CdSO4:H2O	-9.08	-10.37	-1.29 CdSO4:H2O
Halite	-9.18	-7.64	1.54 NaCl
Diopside	-9.2	-2.27	8.94 CuSiO3:H2O
Illite	-9.2	4.78	13.97 K0.6Mg0.25Al2.3Si3.5O10(OH)2
Mirabilite	-9.2	-11.24	-2.04 Na2SO4:10H2O
Montmorillonit	-9.27	1.86	11.13 (HNaK)0.14Mg0.45Fe0.33Al1.47Si3.82O10(OH)2
PbO:0.3H2O	-9.32	3.68	12.98 PbO:0.33H2O
Phillipsite	-9.68	-4.79	4.87 Na0.5K0.5AlSi3O8:H2O
Pb2(OH)3Cl	-9.89	-0.89	8.79 Pb2(OH)3Cl
Albite	-9.77	-4.3	5.48 NaAlSi3O8
Scorodite	-9.78	4.22	14.01 FeAsO4:2H2O
Litharge	-9.88	3.68	13.52 PbO
CdOHCl	-10.07	-6.19	3.88 CdOHCl
Massicot	-10.07	3.68	13.73 PbO
Zn2(OH)2SO4	-10.27	-2.77	7.5 Zn2(OH)2SO4
Pyrochroite	-10.74	4.46	15.2 Mn(OH)2
CdSO4	-10.99	-10.37	0.62 CdSO4
Analcime	-11.01	0.15	11.15 NaAlSi2O8:H2O
Thénardite	-11.09	-11.24	-0.15 Na2SO4
AlumK	-11.24	-16.78	-5.52 KAl(SO4)2:12H2O
MnSO4	-11.37	-7.95	3.42 MnSO4
Zincosite	-11.53	-7.59	3.95 ZnSO4
Cd(OH)2	-11.62	2.03	13.65 Cd(OH)2
Willemite	-11.76	5.19	16.96 Zn2SiO4

Manganite	-12.02	13.32	25.34 MnOOH
Atacamite	-12.12	-3.87	8.25 Cu ₂ (OH)3Cl
Clinoesstatite	-12.23	0.09	12.32 MgSiO ₃
CdSiO ₃	-12.28	-2.41	9.87 CdSiO ₃
CdCl ₂ ·2.5H ₂ O	-12.38	-14.41	-2.02 CdCl ₂ ·2.5H ₂ O
Cd(OH) ₂ (a)	-12.71	2.03	14.74 Cd(OH) ₂
Laumontite	-12.77	3.83	16.59 CaAl ₂ Si ₄ O ₁₂ ·4H ₂ O
CdCl ₂ ·H ₂ O	-12.79	-14.41	-1.82 CdCl ₂ ·H ₂ O
Pb ₃ O ₂ SO ₄	-12.83	-1.42	11.41 PbSO ₄ ·2PbO
Montepontite	-12.95	2.03	14.98 CdO
Pb ₃ (AsO ₄) ₂	-13.05	-6.93	6.13 Pb ₃ (AsO ₄) ₂
Brucite	-13.83	4.53	18.18 Mg(OH) ₂
Zn ₂ (OH) ₂ 3Cl	-13.78	1.42	15.2 Zn ₂ (OH) ₂ 3Cl
MnCl ₂ ·4H ₂ O	-13.85	-11.98	1.68 MnCl ₂ ·4H ₂ O
CdCl ₂	-13.95	-14.41	-0.48 CdCl ₂
CuSO ₄	-14.12	-10.23	3.89 CuSO ₄
Antlerite	-14.17	-5.88	8.29 Cu ₃ (OH) ₄ SO ₄
Jarosite(ss)	-16.21	14.44	30.65 (K _{0.77} Na _{0.03} H _{0.2})Fe ₃ (SO ₄) ₂ (OH) ₆
Magadite	-16.23	-30.53	-14.3 NaSi ₇ O ₁₃ (OH) ₃ ·3H ₂ O
Annite	-16.44	7.66	24.1 KFe ₃ AlSi ₃ O ₁₀ (OH) ₂
Anorthite	-16.49	12.71	29.2 CaAl ₂ Si ₂ O ₈
Fe ₃ (OH) ₈	-16.53	30.68	47.21 Fe ₃ (OH) ₈
Greenalite	-16.75	4.06	20.81 Fe ₃ Si ₂ O ₅ (OH) ₄
Leonhardite	-17.17	7.65	24.82 Ca ₂ Al ₄ Si ₈ O ₂₄ ·7H ₂ O
Mn ₃ (AsO ₄) ₂ ·8H ₂ O	-17.37	-4.55	12.82 Mn ₃ (AsO ₄) ₂ ·8H ₂ O
Zn ₃ (AsO ₄) ₂ ·2H ₂ O	-17.44	-3.48	13.98 Zn ₃ (AsO ₄) ₂ ·2H ₂ O
Wairakite	-17.67	3.83	21.5 CaAl ₂ Si ₄ O ₁₂ ·2H ₂ O
Cu ₃ (AsO ₄) ₂ ·f	-17.79	-11.39	6.41 Cu ₃ (AsO ₄) ₂ ·6H ₂ O
H ₂ (g)	-17.8	-17.73	0.07 H ₂
Pb ₂ SiO ₄	-18.15	2.88	21.03 Pb ₂ SiO ₄
PbMetal	-18.32	-14.07	4.25 Pb
Jarosite-K	-18.46	14.33	32.79 KFe ₃ (SO ₄) ₂ (OH) ₆
Melanconite	-18.59	-14.26	4.33 CuCl ₂
Cu ₂ SO ₄	-18.78	-25.79	-7.01 Cu ₂ SO ₄
Pb ₄ (OH) ₆ SO ₄	-18.85	2.25	21.1 Pb ₄ (OH) ₆ SO ₄
Pb ₂ O(OH) ₂	-18.87	7.33	26.2 PbO·Pb(OH) ₂
Brochantite	-19.04	-3.7	15.34 Cu ₄ (OH) ₆ SO ₄
ZnCl ₂	-19.5	-11.62	7.88 ZnCl ₂
Portlandite	-19.61	4.7	24.31 Ca(OH) ₂
Nsutite	-20.37	22.19	42.56 MnO ₂
Sepiolite	-20.55	-4.27	16.28 Mg ₂ Si ₃ O ₇ ·5OH·3H ₂ O
Diopside	-21.13	0.34	21.47 CaMgSi ₂ O ₆
CuO·CuSO ₄	-21.32	-8.05	13.26 CuO·CuSO ₄
Bimessite	-21.41	22.19	43.6 MnO ₂
Zn ₄ (OH) ₆ SO ₄	-21.53	6.87	28.4 Zn ₄ (OH) ₆ SO ₄
Pb ₄ O ₃ SO ₄	-21.56	2.25	23.81 PbSO ₄ ·3PbO
Claudelite	-21.62	-106.77	-85.16 As ₄ O ₆
Jarosite-Na	-21.65	15.31	36.96 NaFe ₃ (SO ₄) ₂ (OH) ₆
Arsenolite	-21.83	-106.77	-84.94 As ₄ O ₆
Pyrolusite	-22.36	22.19	44.56 MnO ₂
Langite	-22.42	-3.7	18.72 Cu ₄ (OH) ₆ SO ₄ ·H ₂ O
Sepiolite(f)	-22.93	-4.27	16.68 Mg ₂ Si ₃ O ₇ ·5OH·3H ₂ O
JarositeH	-23.04	14.73	37.78 (H ₃ O)Fe ₃ (SO ₄) ₂ (OH) ₆
Prehnite	-24.31	12.97	37.28 Ca ₂ Al ₂ Si ₃ O ₁₀ (OH) ₂
As ₂ O ₅	-24.88	-17.92	6.96 As ₂ O ₅
Cd ₃ (OH) ₂ (SC	-25.43	-18.72	6.71 Cd ₃ (OH) ₂ (SO ₄) ₂
Forsterite	-26.06	4.62	30.66 Mg ₂ SiO ₄
Ca ₃ (AsO ₄) ₂ ·x	-26.45	-3.82	22.62 Ca ₃ (AsO ₄) ₂ ·4H ₂ O
Bixbyite	-27.02	26.65	53.67 Mn ₂ O ₃
Talc	-27.84	-4.18	23.68 Mg ₃ Si ₄ O ₁₀ (OH) ₂
Cd ₃ (OH) ₄ SO ₄	-28.87	-8.31	22.56 Cd ₃ (OH) ₄ SO ₄
Chrysotile	-29.95	4.71	34.65 Mg ₃ Si ₂ O ₅ (OH) ₄
CdMetal	-30.07	-15.7	14.37 Cd
Cd(gamma)	-30.18	-15.7	14.47 Cd
Zn ₅ (OH) ₈ Cl ₂	-30.84	7.66	38.5 Zn ₅ (OH) ₈ Cl ₂
Platnerite	-31.35	21.4	52.75 PbO ₂
Zn ₃ O(SO ₄) ₂	-32.4	-10.36	22.04 ZnO·2ZnSO ₄
Cd ₄ (OH) ₆ SO ₄	-32.68	-4.28	28.4 Cd ₄ (OH) ₆ SO ₄
Hausmannite	-34.83	31.11	65.94 Mn ₃ O ₄
Pb ₂ O ₃	-35.98	25.06	61.04 Pb ₂ O ₃
Phlogopite	-37.05	8.31	45.36 KMg ₃ AlSi ₃ O ₁₀ (OH) ₂
ZnMetal	-40.47	-12.91	27.55 Zn
Chionite14A	-49.54	26.23	75.77 Mg ₅ Al ₂ Si ₃ O ₁₀ (OH) ₈
Minium	-49.98	28.72	78.7 Pb ₃ O ₄
Chionite7A	-53.09	26.23	79.32 Mg ₅ Al ₂ Si ₃ O ₁₀ (OH) ₈
Mn ₂ (SO ₄) ₃	-60.3	-10.57	49.73 Mn ₂ (SO ₄) ₃

Tremolite -64.79 -3.5 61.3 $\text{Ca}_2\text{Mg}_5\text{Si}_8\text{O}_{22}(\text{OH})_2$

End of simulation.

Reading input data for simulation 2

End of run.

Intensive Sampling - High Redox (July 28th)

Reading data base.

SOLUTION_MASTER_SPECIES
SOLUTION_SPECIES
PHASES
EXCHANGE_MASTER_SPECIES
EXCHANGE_SPECIES
SURFACE_MASTER_SPECIES
SURFACE_SPECIES
END

Reading input data for simulation 1

TITLE	Example	Bakers	Creek	Intensive	Samplig	program	-	July	1998
SOLUTION	1	T=2:30							
temp	6.1								
pH	4.29								
pe	11.16								
units	ppm								
density	1.023								
Ca	8.06								
Al	3.87								
As	0.186								
Cd	0.041								
Cu	0.39								
Pb	11.5								
Zn	14.5								
Mg	2.91								
Na	6.81								
K	1.07								
Fe	13.5								
Mn	6.81								
Si	18.1								
Cl	4.4								
S(6)	26.7								
SOLUTION_MASTER_SPECIES									
SOLUTION_SPECIES									
PHASES									
END									

TITLE

Example 1.-Add uranium and speciate seawater.

Beginning of initial solution calculations.

Initial solution 1

#NAME? composition-----

Elements	Molality	Moles
Al	1.43E-04	1.43E-04
As	2.48E-06	2.48E-06
Ca	1.51E-04	1.51E-04

Cd	3.85E-07	3.85E-07
Cl	1.24E-04	1.24E-04
Cu	8.14E-08	8.14E-08
Fe	2.42E-04	2.42E-04
K	2.74E-05	2.74E-05
Mg	1.20E-04	1.20E-04
Mn	1.24E-04	1.24E-04
Na	2.98E-04	2.96E-04
Pb	5.55E-05	5.55E-05
S(6)	2.78E-04	2.78E-04
Si	3.01E-04	3.01E-04
Zn	2.22E-04	2.22E-04

#NAME? of solution-----

pH = 4.29
 pe = 11.16
 Activity of water = 1
 Ionic strength = 2.92E-03
 Mass of water (kg) = 1.00E+00
 Total alkalinity (eq/kg) = -7.92E-05
 Total carbon (mol/kg) = 0.00E+00
 Total CO2 (mol/kg) = 0.00E+00
 Temperature (deg C) = 6.1
 Electrical balance (eq) = 1.90E-03
 Iterations = 4
 Total H = 1.11E+02
 Total O = 5.55E+01

#NAME? of species-----

	Log Species	Log Molality	Log Activity	Molality	Activity	Gamma	%
	H+	5.41E-05	5.13E-05	-4.267	-4.29	-0.023	
	OH-	4.24E-11	4.00E-11	-10.373	-10.398	-0.025	
	H2O	5.55E+01	1.00E+00	0	0	0	
Al	1.43E-04						
	Al+3	1.09E-04	6.48E-05	-3.963	-4.188	-0.225	76.2238
	AlSO4+	3.00E-05	2.83E-05	-4.523	-4.548	-0.025	20.979
	AlOH+2	4.39E-06	3.49E-06	-5.358	-5.458	-0.1	3.0699
	Al(SO4)2-	1.55E-07	1.46E-07	-6.81	-6.835	-0.025	0.1084
	Al(OH)2+	8.86E-08	8.37E-08	-7.052	-7.077	-0.025	0.062
	AlHSO4+2	1.44E-10	1.15E-10	-9.841	-9.941	-0.1	
	Al(OH)3	4.92E-11	4.92E-11	-10.308	-10.308	0	
	Al(OH)4-	1.66E-12	1.56E-12	-11.781	-11.806	-0.025	
As(3)	5.25E-19						
	H3AsO3	5.25E-19	5.28E-19	-18.28	-18.279	0	100
	H4AsO3+	1.41E-23	1.34E-23	-22.849	-22.874	-0.025	
	H2AsO3-	3.04E-24	2.87E-24	-23.518	-23.543	-0.025	
	HAsO3-2	2.32E-32	1.85E-32	-31.634	-31.734	-0.1	
	AsO3-3	0.00E+00	0.00E+00	-40.933	-41.158	-0.225	
As(5)	2.48E-06						
	H2AsO4-	2.46E-06	2.32E-06	-5.61	-5.635	-0.025	99.1935
	H3AsO4	1.72E-08	1.72E-08	-7.766	-7.765	0	0.6935
	HAsO4-2	9.10E-09	7.23E-09	-8.041	-8.141	-0.1	
	AsO4-3	3.65E-16	2.17E-16	-15.438	-15.663	-0.225	
Ca	1.51E-04						
	Ca+2	1.48E-04	1.18E-04	-3.831	-3.929	-0.098	98.0132
	CaSO4	3.49E-06	3.49E-06	-5.457	-5.457	0	2.3113
	CaHSO4+	9.20E-10	8.69E-10	-9.036	-9.061	-0.025	0.0006
	CaOH+	4.04E-13	3.81E-13	-12.394	-12.419	-0.025	

Cd	3.85E-07					
	Cd+2	3.49E-07	2.77E-07	-6.457	-6.557	-0.1 95.6164
	CdSO4	1.27E-08	1.27E-08	-7.897	-7.897	0 3.4795
	CdCl+	3.07E-09	2.89E-09	-8.513	-8.538	-0.025 0.8411
	Cd(SO4)2-2	3.55E-11	2.82E-11	-10.45	-10.55	-0.1
	CdCl2	1.31E-12	1.31E-12	-11.883	-11.883	0
	CdOH+	1.07E-13	1.01E-13	-12.972	-12.997	-0.025
	CdOHCl	1.52E-14	1.52E-14	-13.82	-13.819	0
	CdCl3-	7.55E-17	7.13E-17	-16.122	-16.147	-0.025
	Cd(OH)2	4.71E-19	4.71E-19	-18.327	-18.327	0
	Cd2OH+3	2.95E-19	1.76E-19	-18.53	-18.755	-0.225
	Cd(OH)3-	1.09E-27	1.03E-27	-26.962	-26.987	-0.025
	Cd(OH)4-2	2.25E-37	1.79E-37	-36.647	-36.747	-0.1
Cl	1.24E-04					
	Cl-	1.24E-04	1.17E-04	-3.907	-3.932	-0.025 100
	PbCl+	1.22E-07	1.15E-07	-6.915	-6.94	-0.025
	MnCl+	4.88E-08	4.61E-08	-7.311	-7.336	-0.025
	ZnCl+	2.35E-08	2.21E-08	-7.63	-7.655	-0.025
	FeCl+	1.99E-08	1.88E-08	-7.701	-7.726	-0.025
	CdCl+	3.07E-09	2.89E-09	-8.513	-8.538	-0.025
	FeCl+2	1.25E-09	9.91E-10	-8.904	-9.004	-0.1
	CuCl+	5.90E-10	5.57E-10	-9.229	-9.254	-0.025
	PbCl2	3.10E-11	3.11E-11	-10.508	-10.508	0
	ZnOHCl	1.29E-11	1.29E-11	-10.889	-10.888	0
	ZnCl2	2.50E-12	2.50E-12	-11.602	-11.602	0
	MnCl2	2.35E-12	2.35E-12	-11.629	-11.628	0
	CdCl2	1.31E-12	1.31E-12	-11.883	-11.883	0
	FeCl2+	1.04E-12	9.82E-13	-11.983	-12.008	-0.025
	CuCl2	2.81E-14	2.81E-14	-13.551	-13.551	0
	CdOHCl	1.52E-14	1.52E-14	-13.82	-13.819	0
	PbCl3-	2.70E-15	2.55E-15	-14.569	-14.594	-0.025
	ZnCl3-	3.08E-16	2.91E-16	-15.512	-15.537	-0.025
	MnCl3-	8.03E-17	7.58E-17	-16.096	-16.121	-0.025
	CdCl3-	7.55E-17	7.13E-17	-16.122	-16.147	-0.025
	CuCl2-	6.87E-17	6.48E-17	-16.163	-16.188	-0.025
	FeCl3	1.15E-17	1.15E-17	-16.941	-16.94	0
	PbCl4-2	1.54E-19	1.22E-19	-18.814	-18.914	-0.1
	ZnCl4-2	1.83E-20	1.45E-20	-19.738	-19.838	-0.1
	CuCl3-2	1.40E-20	1.11E-20	-19.854	-19.954	-0.1
	CuCl3-	8.64E-21	8.16E-21	-20.063	-20.088	-0.025
	CuCl4-2	3.77E-27	3.00E-27	-26.423	-26.523	-0.1
Cu(1)	1.52E-14					
	Cu+	1.51E-14	1.43E-14	-13.82	-13.845	-0.025 99.3421
	CuCl2-	6.87E-17	6.48E-17	-16.163	-16.188	-0.025
Cu(2)	6.14E-06					
	Cu+2	5.99E-06	4.76E-06	-5.223	-5.323	-0.1 97.557
	CuSO4	1.51E-07	1.52E-07	-6.82	-6.82	0 2.4593
	CuOH+	9.82E-10	9.27E-10	-9.008	-9.033	-0.025 0.016
	CuCl+	5.90E-10	5.57E-10	-9.229	-9.254	-0.025
	Cu(OH)2	3.77E-11	3.78E-11	-10.423	-10.423	0
	Cu2(OH)2+;	6.38E-14	5.07E-14	-13.195	-13.295	-0.1
	CuCl2	2.81E-14	2.81E-14	-13.551	-13.551	0
	Cu(OH)3-	4.70E-20	4.44E-20	-19.328	-19.353	-0.025
	CuCl3-	8.64E-21	8.16E-21	-20.063	-20.088	-0.025
	CuCl4-2	3.77E-27	3.00E-27	-26.423	-26.523	-0.1
	Cu(OH)4-2	2.17E-28	1.73E-28	-27.663	-27.763	-0.1
Fe(2)	1.49E-04 % of SFe(2)					
	Fe+2	1.47E-04	1.17E-04	-3.834	-3.934	-0.1 98.32
	FeSO4	2.57E-06	2.57E-06	-5.59	-5.59	0 1.72
	FeCl+	1.99E-08	1.88E-08	-7.701	-7.726	-0.025 0.01
	FeHSO4+	9.10E-10	8.59E-10	-9.041	-9.066	-0.025 0

Fe(3)	FeOH+	1.69E-10	1.59E-10	-9.774	-9.799	-0.025	0
	Fe(OH)2	4.56E-18	4.56E-18	-17.341	-17.341	0	0
	Fe(OH)3-	2.87E-24	2.71E-24	-23.542	-23.567	-0.025	0
	9.25E-05						
	Fe(OH)2+	6.50E-05	6.13E-05	-4.187	-4.212	-0.025	70.27
	FeOH+2	2.57E-05	2.04E-05	-4.59	-4.69	-0.1	27.7838
	Fe+3	8.93E-07	5.32E-07	-6.049	-6.274	-0.225	0.97
	FeSO4+	7.09E-07	6.70E-07	-6.149	-6.174	-0.025	0.7665
	Fe2(OH)2+	6.49E-08	2.59E-08	-7.188	-7.588	-0.4	0.07
	Fe(OH)3	6.39E-08	6.39E-08	-7.195	-7.194	0	0.0691
	Fe3(OH)4+	8.99E-09	2.13E-09	-8.046	-8.671	-0.625	
	Fe(SO4)2-	2.57E-09	2.43E-09	-6.59	-8.615	-0.025	
	FeCl+2	1.25E-09	9.91E-10	-8.904	-9.004	-0.1	
	FeHSO4+2	1.24E-10	9.85E-11	-9.906	-10.006	-0.1	
	FeCl2+	1.04E-12	9.82E-13	-11.983	-12.008	-0.025	
	Fe(OH)4-	5.35E-13	5.05E-13	-12.271	-12.296	-0.025	
H(0)	FeCl3	1.15E-17	1.15E-17	-16.941	-16.94	0	
	1.87E-34						
K	H2	9.35E-35	9.35E-35	-34.029	-34.029	0	
	2.74E-05						
Mg	K+	2.73E-05	2.58E-05	-4.563	-4.588	-0.025	99.635
	KSO4-	2.42E-08	2.29E-08	-7.616	-7.641	-0.025	
	1.20E-04						
Mn(2)	Mg+2	1.17E-04	9.38E-05	-3.93	-4.028	-0.097	97.5
	MgSO4	2.35E-06	2.35E-06	-5.63	-5.63	0	1.9583
	MgOH+	1.14E-12	1.07E-12	-11.944	-11.969	-0.025	
	1.24E-04						
	Mn+2	1.22E-04	9.68E-05	-3.914	-4.014	-0.1	98.3871
	MnSO4	2.10E-06	2.10E-06	-5.678	-5.678	0	1.6935
	MnCl+	4.88E-08	4.61E-08	-7.311	-7.336	-0.025	0.0394
	MnOH+	9.92E-12	9.36E-12	-11.004	-11.029	-0.025	
	MnCl2	2.35E-12	2.35E-12	-11.629	-11.628	0	
	MnCl3-	8.03E-17	7.58E-17	-16.096	-16.121	-0.025	
Mn(3)	Mn(OH)3-	1.20E-26	1.14E-26	-25.919	-25.944	-0.025	
	3.81E-20						
Mn(6)	Mn+3	3.81E-20	2.27E-20	-19.419	-19.644	-0.225	
	0.00E+00						
Mn(7)	MnO4-2	0.00E+00	0.00E+00	-50.837	-50.937	-0.1	
	0.00E+00						
Na	MnO4-	0.00E+00	0.00E+00	-50.455	-50.48	-0.025	
	2.96E-04						
	Na+	2.96E-04	2.80E-04	-3.529	-3.553	-0.025	100
O(0)	NaSO4-	2.34E-07	2.21E-07	-6.63	-6.655	-0.025	
	2.16E-31						
Pb	O2	1.08E-31	1.08E-31	-30.967	-30.967	0	
	5.55E-05						
	Pb+2	5.13E-05	4.07E-05	-4.29	-4.39	-0.1	92.4324
	PbSO4	4.11E-06	4.11E-06	-5.387	-5.388	0	7.4054
	PbCl+	1.22E-07	1.15E-07	-6.915	-6.94	-0.025	0.2198
	PbOH+	1.64E-08	1.55E-08	-7.785	-7.81	-0.025	
	Pb(SO4)2-2	4.87E-09	3.87E-09	-8.313	-8.413	-0.1	
	PbCl2	3.10E-11	3.11E-11	-10.508	-10.508	0	
	Pb2OH+3	2.37E-11	1.41E-11	-10.625	-10.85	-0.225	
	Pb(OH)2	1.17E-13	1.17E-13	-12.93	-12.93	0	
	PbCl3-	2.70E-15	2.55E-15	-14.569	-14.594	-0.025	
	PbCl4-2	1.54E-19	1.22E-19	-18.814	-18.914	-0.1	
	Pb(OH)3-	2.79E-20	2.63E-20	-19.555	-19.58	-0.025	
	Pb3(OH)4+	7.85E-22	6.24E-22	-21.105	-21.205	-0.1	
	Pb(OH)4-2	1.48E-27	1.17E-27	-26.83	-26.93	-0.1	
	2.78E-04						
S(6)	SO4-2	2.25E-04	1.79E-04	-3.648	-3.746	-0.099	80.9353
	AlSO4+	3.00E-05	2.83E-05	-4.523	-4.548	-0.025	10.7914

	ZnSO4	6.18E-06	6.17E-06	-5.21	-5.21	0	2.2158
	PbSO4	4.11E-06	4.11E-06	-5.387	-5.388	0	1.4784
	CaSO4	3.49E-06	3.49E-06	-5.457	-5.457	0	1.2554
	FeSO4	2.57E-06	2.57E-06	-5.59	-5.59	0	0.9245
	MgSO4	2.35E-06	2.35E-06	-5.63	-5.63	0	
	MnSO4	2.10E-06	2.10E-06	-5.878	-5.678	0	
	FeSO4+	7.09E-07	6.70E-07	-6.149	-6.174	-0.025	
	HSO4-	6.49E-07	6.13E-07	-6.188	-6.213	-0.025	
	NaSO4-	2.34E-07	2.21E-07	-6.63	-6.655	-0.025	
	Al(SO4)2-	1.55E-07	1.46E-07	-6.81	-6.835	-0.025	
	CuSO4	1.51E-07	1.52E-07	-6.82	-6.82	0	
	KSO4-	2.42E-08	2.29E-08	-7.616	-7.641	-0.025	
	Zn(SO4)2-2	1.32E-08	1.05E-08	-7.879	-7.979	-0.1	
	CdSO4	1.27E-08	1.27E-08	-7.897	-7.897	0	
	Pb(SO4)2-2	4.87E-09	3.87E-09	-8.313	-8.413	-0.1	
	Fe(SO4)2-	2.57E-09	2.43E-09	-8.59	-8.615	-0.025	
	CaHSO4+	9.20E-10	8.69E-10	-9.036	-9.061	-0.025	
	FeHSO4+	9.10E-10	8.59E-10	-9.041	-9.066	-0.025	
	AlHSO4+2	1.44E-10	1.15E-10	-9.841	-9.941	-0.1	
	FeHSO4+2	1.24E-10	9.85E-11	-9.906	-10.006	-0.1	
	Cd(SO4)2-2	3.55E-11	2.82E-11	-10.45	-10.55	-0.1	
Si	3.01E-04						
	H4SiO4	3.01E-04	3.02E-04	-3.521	-3.521	0	
	H3SiO4-	4.35E-10	4.11E-10	-9.361	-9.386	-0.025	
	H2SiO4-2	1.73E-19	1.37E-19	-18.763	-18.863	-0.1	
Zn	2.22E-04						
	Zn+2	2.16E-04	1.71E-04	-3.866	-3.766	-0.1	97.2973
	ZnSO4	6.16E-06	6.17E-06	-5.21	-5.21	0	2.7748
	ZnCl+	2.35E-08	2.21E-08	-7.63	-7.655	-0.025	0.0106
	Zn(SO4)2-2	1.32E-08	1.05E-08	-7.879	-7.979	-0.1	
	ZnOH+	8.39E-10	7.93E-10	-9.076	-9.101	-0.025	
	ZnOHCl	1.29E-11	1.29E-11	-10.889	-10.888	0	
	ZnCl2	2.50E-12	2.50E-12	-11.602	-11.602	0	
	Zn(OH)2	8.19E-13	8.20E-13	-12.087	-12.086	0	
	ZnCl3-	3.08E-16	2.91E-18	-15.512	-15.537	-0.025	
	Zn(OH)3-	5.36E-20	5.06E-20	-19.271	-19.296	-0.025	
	ZnCl4-2	1.83E-20	1.45E-20	-19.738	-19.838	-0.1	
	Zn(OH)4-2	1.97E-28	1.56E-28	-27.706	-27.806	-0.1	

#NAME? indices-----

Phase	Si	log	IAP	log	KT
Hematite	15.67	40.19	24.52	Fe2O3	
Magnetite	11.6	44.84	33.24	Fe3O4	
CupricFerrit	8.65	43.45	34.8	CuFe2O4	
Goethite	7.42	20.1	12.67	FeOOH	
Fe(OH)2.7C	7.17	17.63	10.48	Fe(OH)2.7Cl0.3	
Maghemite	6.81	40.19	33.39	Fe2O3	
CuprousFer	5.77	7.9	2.13	CuFeO2	
Jarosite(ss)	4.78	35.43	30.87	(K0.77Na0.03H0.2)Fe3(SO4)2(OH)6	
Jarosite-K	2.5	35.34	32.84	KFe3(SO4)2(OH)6	
Pyrophyllite	2.04	3.28	1.24	Al2Si4O10(OH)2	
Fe(OH)3(a)	1.71	20.1	18.39	Fe(OH)3	
Kaolinite	1.14	10.32	9.19	Al2Si2O5(OH)4	
Quartz	0.76	-3.52	-4.28	SiO2	
Diaspore	0.58	8.68	8.1	AlOOH	
Cristobalite	0.34	-3.52	-3.86	SiO2	
Chalcedony	0.26	-3.52	-3.79	SiO2	
Alunite	0	1.09	1.09	KAl3(SO4)2(OH)6	
Anglesite	-0.24	-8.14	-7.9	PbSO4	
Silicagel	-0.28	-3.52	-3.24	SiO2	

Basaluminit	-0.3	22.4	22.7 Al ₄ (OH) ₁₀ SO ₄
Jurbanite	-0.41	-3.64	-3.23 AlOHSO ₄
Kmica	-0.46	15.18	15.65 KAl ₃ Si ₃ O ₁₀ (OH) ₂
Gibbsite	-0.58	8.68	9.24 Al(OH) ₃
Jarosite-Na	-0.64	38.37	37.02 NaFe ₃ (SO ₄) ₂ (OH) ₈
SiO ₂ (a)	-0.64	-3.52	-2.88 SiO ₂
Montmorillo	-1.25	5.98	7.23 (HNaK)0.09Mg0.29Fe0.24Al1.57Si ₃ .93O ₁₀ (OH) ₂
Boehmite	-1.3	8.68	9.98 AlOOH
Scorodite	-1.69	12.33	14.02 FeAsO ₄ ·2H ₂ O
Montmorillo	-1.73	8.08	9.81 Ca _{0.165} Al _{2.33} Si _{3.67} O ₁₀ (OH) ₂
Beidellite	-1.86	7.61	9.47 (NaKMg0.5)0.11Al _{2.33} Si _{3.67} O ₁₀ (OH) ₂
JarositeH	-2.21	35.64	37.85 (H ₃ O)Fe ₃ (SO ₄) ₂ (OH) ₈
Fe ₃ (OH) ₈	-2.38	44.84	47.22 Fe ₃ (OH) ₈
ZnSiO ₃	-2.54	1.29	3.84 ZnSiO ₃
Gypsum	-3.08	-7.67	-4.6 CaSO ₄ ·2H ₂ O
Montmorillo	-3.14	8.05	11.19 (HNaK)0.14Mg0.45Fe0.33Al1.47Si ₃ .82O ₁₀ (OH) ₂
Anhydrite	-3.33	-7.67	-4.34 CaSO ₄
Al(OH) ₃ (a)	-3.43	8.68	12.11 Al(OH) ₃
Lamakite	-3.99	-3.95	0.04 PbO:PbSO ₄
AlAsO ₄ ·2H ₂ O	-4.01	0.92	4.93 AlAsO ₄ ·2H ₂ O
Halloysite	-4.16	10.32	14.48 Al ₂ Si ₂ O ₅ (OH) ₄
Pb(OH) ₂	-4.65	4.19	8.84 Pb(OH) ₂
Laurionite	-4.86	-4.03	0.62 PbOHCl
Adularia	-4.85	-2.18	2.68 KAlSi ₃ O ₈
Tenorite	-5.12	3.26	8.38 CuO
Melanterite	-5.21	-7.68	-2.47 FeSO ₄ ·7H ₂ O
Manganite	-5.32	20.02	25.34 MnOOH
Goslarite	-5.39	-7.51	-2.12 ZnSO ₄ ·7H ₂ O
Illite	-5.4	8.6	14.01 K _{0.6} Mg _{0.25} Al _{2.3} Si _{3.5} O ₁₀ (OH) ₂
Epsomite	-5.49	-7.77	-2.28 MgSO ₄ ·7H ₂ O
Bianchite	-5.76	-7.51	-1.76 ZnSO ₄ ·6H ₂ O
Cu(OH) ₂	-6.14	3.26	9.4 Cu(OH) ₂
Chalcanthite	-6.36	-9.07	-2.71 CuSO ₄ ·5H ₂ O
ZnO(a)	-6.5	4.81	11.31 ZnO
Phillipsite	-6.57	-1.66	4.9 Na _{0.5} K _{0.5} AlSi ₃ O ₈ ·H ₂ O
Albite	-6.63	-1.14	5.49 NaAlSi ₃ O ₈
Zn(OH) ₂ -e	-6.69	4.81	11.5 Zn(OH) ₂
Zn(OH) ₂ -g	-6.9	4.81	11.71 Zn(OH) ₂
Zn(OH) ₂ -b	-6.94	4.81	11.75 Zn(OH) ₂
Nsutite	-7.1	35.47	42.56 MnO ₂
PbSiO ₃	-7.11	0.67	7.78 PbSiO ₃
Colunnite	-7.21	-12.25	-5.05 PbCl ₂
Diopase	-7.21	-0.26	6.94 CuSiO ₃ ·H ₂ O
Zn(OH) ₂ -c	-7.39	4.81	12.2 Zn(OH) ₂
Zincite(c)	-7.41	4.81	12.22 ZnO
ZnSO ₄ ·H ₂ O	-7.47	-7.51	-0.04 ZnSO ₄ ·H ₂ O
Zn(OH) ₂ -a	-7.64	4.81	12.45 Zn(OH) ₂
Birnessite	-8.14	35.47	43.6 MnO ₂
CdSO ₄ ·2.7H ₂ O	-8.64	-10.3	-1.86 CdSO ₄ ·2.87H ₂ O
Pb ₂ (OH) ₃ Cl	-8.64	0.16	8.79 Pb ₂ (OH) ₃ Cl
Laumontite	-8.7	7.93	16.63 CaAl ₂ Si ₄ O ₁₂ ·4H ₂ O
PbO·0.3H ₂ O	-8.79	4.19	12.98 PbO·0.33H ₂ O
Analcime	-8.8	2.38	11.17 NaAlSi ₂ O ₆ ·H ₂ O
Mirabilite	-8.8	-10.85	-2.06 Na ₂ SO ₄ ·10H ₂ O
CdSO ₄ ·H ₂ O	-9.02	-10.3	-1.28 CdSO ₄ ·H ₂ O
Halite	-9.02	-7.49	1.54 NaCl
Leonhardite	-9.02	15.86	24.89 Ca ₂ Al ₄ Si ₈ O ₂₄ ·7H ₂ O
Pb ₃ (AsO ₄) ₂	-9.09	-2.96	6.13 Pb ₃ (AsO ₄) ₂
Pyrolusite	-9.14	35.47	44.61 MnO ₂
Litharge	-9.34	4.19	13.53 PbO
Massicot	-9.55	4.19	13.74 PbO
Magadiite	-9.61	-23.91	-14.3 NaSi ₇ O ₁₃ (OH) ₃ ·3H ₂ O

Atacamite	-9.98	-1.71	8.27 Cu ₂ (OH) ₃ Cl
CdOHCl	-10.09	-8.2	3.89 CdOHCl
Zn ₂ (OH) ₂ Si	-10.2	-2.7	7.5 Zn ₂ (OH) ₂ SO ₄
Nantokite	-10.52	-20.42	-9.89 CuCl
Pyrochroite	-10.83	4.57	15.2 Mn(OH) ₂
Thenardite	-10.7	-10.85	-0.15 Na ₂ SO ₄
AlumK	-10.74	-16.27	-5.53 KAl(SO ₄) ₂ ·12H ₂ O
Antlerite	-10.85	-2.58	8.29 Cu ₃ (OH) ₄ SO ₄
Willemite	-10.88	6.11	16.99 Zn ₂ SiO ₄
CdSO ₄	-10.93	-10.3	0.63 CdSO ₄
Pb ₃ O ₂ SO ₄	-11.19	0.24	11.43 PbSO ₄ ·2PbO
MnSO ₄	-11.2	-7.76	3.44 MnSO ₄
Clinoenstatite	-11.3	1.03	12.34 MgSiO ₃
CdSiO ₃	-11.38	-1.5	9.89 CdSiO ₃
Zincosite	-11.48	-7.51	3.96 ZnSO ₄
Cd(OH) ₂	-11.63	2.02	13.65 Cd(OH) ₂
Cu ₃ (AsO ₄) ₂	-12.17	-5.76	6.41 Cu ₃ (AsO ₄) ₂ ·6H ₂ O
Annite	-12.4	11.76	24.16 KFe ₃ AlSi ₃ O ₁₀ (OH) ₂
CdCl ₂ ·2.5H ₂ O	-12.4	-14.42	-2.02 CdCl ₂ ·2.5H ₂ O
Cd(OH) ₂ (a)	-12.74	2.02	14.76 Cd(OH) ₂
CdCl ₂ ·H ₂ O	-12.8	-14.42	-1.62 CdCl ₂ ·H ₂ O
CuSO ₄	-12.98	-9.07	3.91 CuSO ₄
Monteponite	-12.98	2.02	15 CdO
Wairakite	-13.62	7.93	21.55 CaAl ₂ Si ₄ O ₁₂ ·2H ₂ O
Brucite	-13.63	4.55	18.18 Mg(OH) ₂
Bixbyite	-13.69	40.03	53.73 Mn ₂ O ₃
MnCl ₂ ·4H ₂ O	-13.73	-11.88	1.85 MnCl ₂ ·4H ₂ O
Zn ₂ (OH) ₃ Cl	-13.79	1.41	15.2 Zn ₂ (OH) ₃ Cl
Greenalite	-13.91	6.9	20.81 Fe ₃ Si ₂ O ₅ (OH) ₄
CdCl ₂	-13.96	-14.42	-0.46 CdCl ₂
Anorthite	-14.29	14.97	29.27 CaAl ₂ Si ₂ O ₈
Brochantite	-14.64	0.7	15.34 Cu ₄ (OH) ₆ SO ₄
Mn ₃ (AsO ₄) ₂	-14.66	-1.83	12.83 Mn ₃ (AsO ₄) ₂ ·8H ₂ O
Zn ₃ (AsO ₄) ₂	-15.08	-1.09	13.99 Zn ₃ (AsO ₄) ₂ ·2.5H ₂ O
CuMetal	-15.39	-27.64	-12.25 Cu
Pb ₂ SiO ₄	-16.19	4.86	21.05 Pb ₂ SiO ₄
Pb ₄ (OH) ₆ SO ₄	-16.67	4.43	21.1 Pb ₄ (OH) ₆ SO ₄
Cuprite	-17.25	-24.39	-7.14 Cu ₂ O
Melanothallite	-17.53	-13.19	4.34 CuCl ₂
Plattnerite	-17.72	35.09	52.81 PbO ₂
Sepiolite	-17.75	-1.46	16.29 Mg ₂ Si ₃ O ₇ ·5OH·3H ₂ O
Pb ₂ O(OH) ₂	-17.82	8.38	26.2 PbO·Pb(OH) ₂
Langite	-18.05	0.7	18.76 Cu ₄ (OH) ₆ SO ₄ ·H ₂ O
CuOCuSO ₄	-19.11	-5.81	13.29 CuO·CuSO ₄
Diopside	-19.34	2.16	21.5 CaMgSi ₂ O ₆
Pb ₄ O ₃ SO ₄	-19.41	4.43	23.84 PbSO ₄ ·3PbO
ZnCl ₂	-19.53	-11.63	7.9 ZnCl ₂
Portlandite	-19.69	4.65	24.34 Ca(OH) ₂
Sepiolite(d)	-20.12	-1.46	18.66 Mg ₂ Si ₃ O ₇ ·5OH·3H ₂ O
Prehnite	-21.24	16.1	37.35 Ca ₂ Al ₂ Si ₃ O ₁₀ (OH) ₂
Hausmannite	-21.43	44.6	66.02 Mn ₃ O ₄
Zn ₄ (OH) ₆ SO ₄	-21.47	6.93	28.4 Zn ₄ (OH) ₆ SO ₄
Pb ₂ O ₃	-21.76	39.28	61.04 Pb ₂ O ₃
As ₂ O ₅	-22.5	-15.53	6.97 As ₂ O ₅
Talc	-24.12	-0.43	23.7 Mg ₃ Si ₄ O ₁₀ (OH) ₂
Ca ₃ (AsO ₄) ₂	-24.21	-1.58	22.63 Ca ₃ (AsO ₄) ₂ ·4H ₂ O
Forsterite	-25.13	5.58	30.72 Mg ₂ SiO ₄
Cd ₃ (OH) ₂ (S)	-25.29	-18.58	6.71 Cd ₃ (OH) ₂ (SO ₄) ₂
Chrysotile	-28.08	6.62	34.7 Mg ₃ Si ₂ O ₅ (OH) ₄
O ₂ (g)	-28.1	61.8	89.9 O ₂
Cd ₃ (OH) ₄ SO ₄	-28.82	-6.26	22.56 Cd ₃ (OH) ₄ SO ₄
Cu ₂ SO ₄	-29.71	-36.71	-7 Cu ₂ SO ₄

Zn5(OH)8Cl	-30.88	7.62	38.5 Zn5(OH)8Cl2
PbMetal	-30.98	-28.71	4.25 Pb
H2(g)	-30.97	-30.9	0.07 H2
Zn3O(SO4)3	-32.31	-10.21	22.1 ZnO:2ZnSO4
Cd4(OH)6S	-32.63	-4.23	28.4 Cd4(OH)6SO4
Phlogopite	-33.92	11.48	45.4 KMg3AlSi3O10(OH)2
Minium	-35.32	43.47	78.79 Pb3O4
CdMetal	-43.26	-28.88	14.38 Cd
Cd(gamma)	-43.37	-28.88	14.49 Cd
Chlorite14A	-46.33	29.56	75.9 Mg5Al2Si3O10(OH)8
Mn2(SO4)3	-46.75	3.05	49.81 Mn2(SO4)3
Chlorite7A	-49.89	29.56	79.45 Mg5Al2Si3O10(OH)8
ZnMetal	-53.67	-26.09	27.58 Zn
Tremolite	-57.48	3.9	61.38 Ca2Mg5Si8O22(OH)2
Claudetite	-69.39	-154.66	-85.27 As4O6
Arsenolite	-69.61	-154.66	-85.06 As4O6

End of simulation.

Reading input data for simulation 2

End of run.

Long Term Sampling - (31Aug)

 Reading data base.

SOLUTION_MASTER_SPECIES
 SOLUTION_SPECIES
 PHASES
 EXCHANGE_MASTER_SPECIES
 EXCHANGE_SPECIES
 SURFACE_MASTER_SPECIES
 SURFACE_SPECIES
 END

 Reading input data for simulation 1

TITLE	Example	1.--Add	uranium	and	speciate	seawater.	
SOLUTION		1 SEAWATER FROM		NORDSTR(ET	AL.		-1979
units	ppm						
pH	3.48						
pe	12.3						
density	1.023						
temp	7.7						
Ca	19.7						
Al	4.73						
Cd	0.134						
Cu	1.51						
Pb	1.41						
Zn	82.2						
Mg	7.82						
Na	3.66						
K	0.54						
Fe	1.94						
Mn	13.2						
Si	86.8						
Cl	18						
S(6)	6.56						
SOLUTION_MASTER_SPECIES							
SOLUTION_SPECIES							
PHASES							
END							

 TITLE

Example 1.--Add uranium and speciate seawater.

 Beginning of initial solution calculations.

Initial solution 1 SEAWATER FROM NORDSTR(ET AL. -1979

#NAME? composition-----

Elements	Molality	Moles
Al	1.75E-04	1.75E-04
Ca	4.92E-04	4.92E-04
Cd	1.19E-06	1.19E-06
Cl	5.08E-04	5.08E-04
Cu	2.38E-05	2.38E-05
Fe	3.48E-05	3.48E-05
K	1.38E-05	1.38E-05
Mg	3.22E-04	3.22E-04
Mn	2.40E-04	2.40E-04
Na	1.59E-04	1.59E-04
Pb	6.81E-06	6.81E-06
S(6)	6.83E-05	6.83E-05
Si	1.45E-03	1.45E-03
Zn	1.26E-03	1.26E-03

#NAME? of solution-----

pH	=	3.48		
pe	=	12.3		
Activity	of	water	=	1
Ionic	strength	=	6.10E-03	
Mass	of	water	(kg)	= 1.00E+00
Total	alkalinity	(eq/kg)	=	-3.70E-04
Total	carbon	(mol/kg)	=	0.00E+00
Total	CO2	(mol/kg)	=	0.00E+00
Temperatur	(deg	C)	=	7.7
Electrical	balance	(eq)	=	5.17E-03
Iterations	=	4		
Total	H	=	1.11E+02	
Total	O	=	5.55E+01	

#NAME? of species-----

	Log Species	Log Molality	Log Activity	Molality	Activity	Gamma
	H+	3.56E-04	3.31E-04	-3.448	-3.48	-0.032
	OH-	7.79E-12	7.18E-12	-11.109	-11.144	-0.035
	H2O	5.55E+01	1.00E+00	0	0	0
Al	1.75E-04					
	Al+3	1.67E-04	8.05E-05	-3.778	-4.094	-0.316
	AlSO4+	7.68E-06	7.09E-06	-5.114	-5.15	-0.035
	AlOH+2	1.04E-06	7.50E-07	-5.984	-6.125	-0.14
	Al(SO4)2-	7.88E-09	7.27E-09	-8.103	-8.138	-0.035
	Al(OH)2+	3.60E-09	3.32E-09	-8.444	-8.479	-0.035
	AlHSO4+2	2.57E-10	1.86E-10	-9.589	-9.73	-0.14
	Al(OH)3	3.51E-13	3.51E-13	-12.455	-12.454	0.001
	Al(OH)4-	1.88E-15	1.73E-15	-14.726	-14.761	-0.035
Ca	4.92E-04					
	Ca+2	4.90E-04	3.57E-04	-3.31	-3.447	-0.137
	CaSO4	2.12E-06	2.12E-06	-5.674	-5.673	0.001
	CaHSO4+	3.74E-09	3.45E-09	-8.427	-8.463	-0.035
	CaOH+	1.94E-13	1.79E-13	-12.712	-12.747	-0.035
Cd	1.19E-06					

Cl	Cd+2	1.15E-06	8.30E-07	-5.94	-6.081	-0.14
	CdCl+	3.78E-08	3.48E-08	-7.423	-7.458	-0.035
	CdSO4	7.55E-09	7.56E-09	-8.122	-8.121	0.001
	CdCl2	6.33E-11	6.33E-11	-10.199	-10.198	0.001
	Cd(SO4)2-2	4.53E-12	3.28E-12	-11.344	-11.485	-0.14
	CdOH+	5.79E-14	5.34E-14	-13.237	-13.272	-0.035
	CdOHCl	2.93E-14	2.94E-14	-13.533	-13.532	0.001
	CdCl3-	1.54E-14	1.42E-14	-13.814	-13.849	-0.035
	Cd2OH+3	5.65E-19	2.73E-19	-18.248	-18.564	-0.316
	Cd(OH)2	3.38E-20	3.38E-20	-19.472	-19.471	0.001
	Cd(OH)3-	1.24E-29	1.15E-29	-28.906	-28.941	-0.035
	Cd(OH)4-2	4.26E-40	3.08E-40	-39.371	-39.511	-0.14
	5.08E-04					
	Cl-	5.07E-04	4.67E-04	-3.295	-3.331	-0.036
	ZnCl+	5.49E-07	5.06E-07	-6.261	-6.296	-0.035
	MnCl+	3.57E-07	3.29E-07	-6.447	-6.482	-0.035
	PbCl+	6.15E-08	5.67E-08	-7.211	-7.246	-0.035
	CdCl+	3.78E-08	3.48E-08	-7.423	-7.458	-0.035
	FeCl+2	1.08E-08	7.82E-09	-7.966	-8.107	-0.14
	FeCl+	9.96E-09	9.19E-09	-8.002	-8.037	-0.035
	CuCl+	9.49E-09	8.75E-09	-8.023	-8.058	-0.035
	ZnCl2	2.30E-10	2.30E-10	-9.639	-9.639	0.001
	MnCl2	6.70E-11	6.71E-11	-10.174	-10.173	0.001
	CdCl2	6.33E-11	6.33E-11	-10.199	-10.198	0.001
	PbCl2	5.91E-11	5.92E-11	-10.228	-10.228	0.001
	ZnOHCl	4.22E-11	4.23E-11	-10.375	-10.374	0.001
	FeCl2+	3.17E-11	2.92E-11	-10.5	-10.535	-0.035
	CuCl2	1.80E-12	1.80E-12	-11.745	-11.745	0.001
Cu(1)	ZnCl3-	1.17E-13	1.08E-13	-12.932	-12.967	-0.035
	CdOHCl	2.93E-14	2.94E-14	-13.533	-13.532	0.001
	PbCl3-	2.13E-14	1.96E-14	-13.673	-13.708	-0.035
	CdCl3-	1.54E-14	1.42E-14	-13.814	-13.849	-0.035
	MnCl3-	9.36E-15	8.63E-15	-14.029	-14.064	-0.035
	FeCl3	1.36E-15	1.36E-15	-14.866	-14.865	0.001
	CuCl2-	2.96E-16	2.73E-16	-15.529	-15.564	-0.035
	ZnCl4-2	3.02E-17	2.18E-17	-16.521	-16.661	-0.14
	PbCl4-2	5.25E-18	3.80E-18	-17.279	-17.42	-0.14
	CuCl3-	2.33E-18	2.15E-18	-17.632	-17.667	-0.035
	CuCl3-2	2.60E-19	1.88E-19	-18.585	-18.725	-0.14
	CuCl4-2	4.55E-24	3.29E-24	-23.342	-23.482	-0.14
	4.41E-15					
	Cu+	4.11E-15	3.79E-15	-14.386	-14.421	-0.035
	CuCl2-	2.96E-16	2.73E-16	-15.529	-15.564	-0.035
	CuCl3-2	2.60E-19	1.88E-19	-18.585	-18.725	-0.14
Cu(2)	2.38E-05					
	Cu+2	2.37E-05	1.71E-05	-4.626	-4.767	-0.14
	CuSO4	1.09E-07	1.09E-07	-6.964	-6.964	0.001
	CuCl+	9.49E-09	8.75E-09	-8.023	-8.058	-0.035
	CuOH+	5.60E-10	5.17E-10	-9.252	-9.287	-0.035
	Cu(OH)2	3.26E-12	3.26E-12	-11.487	-11.487	0.001
	CuCl2	1.80E-12	1.80E-12	-11.745	-11.745	0.001
	Cu2(OH)2+:	2.61E-14	1.89E-14	-13.584	-13.724	-0.14
	CuCl3-	2.33E-18	2.15E-18	-17.632	-17.667	-0.035
	Cu(OH)3-	6.43E-22	5.93E-22	-21.192	-21.227	-0.035
	CuCl4-2	4.55E-24	3.29E-24	-23.342	-23.482	-0.14
	Cu(OH)4-2	4.94E-31	3.58E-31	-30.306	-30.447	-0.14

Fe(2)	1.98E-05					
	Fe+2	1.97E-05	1.43E-05	-4.706	-4.846	-0.14
	FeSO4	6.39E-08	6.40E-08	-7.194	-7.194	0.001
	FeCl+	9.98E-09	9.19E-09	-8.002	-8.037	-0.035
	FeHSO4+	1.49E-10	1.38E-10	-9.827	-9.862	-0.035
	FeOH+	3.74E-12	3.45E-12	-11.427	-11.462	-0.035
	Fe(OH)2	1.79E-20	1.80E-20	-19.747	-19.746	0.001
	Fe(OH)3-	1.82E-27	1.68E-27	-26.739	-26.774	-0.035
Fe(3)	1.50E-05					
	FeOH+2	9.07E-06	6.56E-06	-5.042	-5.183	-0.14
	Fe(OH)2+	3.55E-06	3.27E-06	-5.45	-5.485	-0.035
	Fe+3	2.06E-06	9.93E-07	-5.687	-6.003	-0.316
	FeSO4+	2.78E-07	2.56E-07	-6.556	-6.592	-0.035
	FeCl+2	1.08E-08	7.82E-09	-7.966	-8.107	-0.14
	Fe2(OH)2+	9.03E-09	2.48E-09	-8.044	-8.606	-0.562
	Fe(OH)3	5.71E-10	5.71E-10	-9.244	-9.243	0.001
	FeHSO4+2	3.33E-10	2.41E-10	-9.478	-9.619	-0.14
	Fe(SO4)2-	2.00E-10	1.84E-10	-9.699	-9.735	-0.035
	Fe3(OH)4+	6.96E-11	9.22E-12	-10.158	-11.035	-0.878
	FeCl2+	3.17E-11	2.92E-11	-10.5	-10.535	-0.035
	FeCl3	1.36E-15	1.36E-15	-14.866	-14.865	0.001
	Fe(OH)4-	8.16E-16	7.52E-16	-15.089	-15.124	-0.035
H(0)	4.07E-35					
	H2	2.03E-35	2.04E-35	-34.692	-34.691	0.001
K	1.38E-05					
	K+	1.38E-05	1.27E-05	-4.86	-4.895	-0.036
	KSO4-	2.49E-09	2.29E-09	-8.605	-8.64	-0.035
Mg	3.22E-04					
	Mg+2	3.21E-04	2.35E-04	-3.494	-3.629	-0.135
	MgSO4	1.21E-06	1.21E-06	-5.917	-5.916	0.001
	MgOH+	5.32E-13	4.90E-13	-12.274	-12.309	-0.035
Mn(2)	2.40E-04					
	Mn+2	2.39E-04	1.73E-04	-3.621	-3.762	-0.14
	MnSO4	7.65E-07	7.66E-07	-6.116	-6.116	0.001
	MnCl+	3.57E-07	3.29E-07	-6.447	-6.482	-0.035
	MnCl2	6.70E-11	6.71E-11	-10.174	-10.173	0.001
	MnOH+	3.26E-12	3.01E-12	-11.487	-11.522	-0.035
	MnCl3-	9.36E-15	8.63E-15	-14.029	-14.064	-0.035
	Mn(OH)3-	8.19E-29	7.56E-29	-28.087	-28.122	-0.035
Mn(3)	1.51E-18					
	Mn+3	1.51E-18	7.30E-19	-17.821	-18.137	-0.316
Mn(6)	0.00E+00					
	MnO4-2	0.00E+00	0.00E+00	-51.795	-51.935	-0.14
Mn(7)	0.00E+00					
	MnO4-	0.00E+00	0.00E+00	-50.185	-50.22	-0.035
Na	1.59E-04					
	Na+	1.59E-04	1.47E-04	-3.798	-3.833	-0.035
	NaSO4-	2.51E-08	2.32E-08	-7.6	-7.635	-0.035
O(0)	1.80E-29					
	O2	8.98E-30	9.00E-30	-29.047	-29.046	0.001
Pb	6.81E-06					
	Pb+2	6.65E-06	4.81E-06	-5.177	-5.318	-0.14
	PbSO4	9.55E-08	9.56E-08	-7.02	-7.019	0.001
	PbCl+	6.15E-08	5.67E-08	-7.211	-7.246	-0.035
	PbOH+	3.07E-10	2.83E-10	-9.513	-9.548	-0.035
	PbCl2	5.91E-11	5.92E-11	-10.228	-10.228	0.001

	Pb(SO4)2-2	2.45E-11	1.77E-11	-10.811	-10.751	-0.14
	Pb2OH+3	6.32E-14	3.05E-14	-13.199	-13.515	-0.316
	PbCl3-	2.13E-14	1.96E-14	-13.673	-13.708	-0.035
	Pb(OH)2	3.32E-16	3.33E-16	-15.478	-15.478	0.001
	PbCl4-2	5.25E-18	3.80E-18	-17.279	-17.42	-0.14
	Pb(OH)3-	1.25E-23	1.15E-23	-22.903	-22.938	-0.035
	Pb3(OH)4+;	1.07E-27	7.77E-28	-26.969	-27.11	-0.14
	Pb(OH)4-2	1.10E-31	7.98E-32	-30.957	-31.098	-0.14
S(6)	6.83E-05					
	SO4-2	4.86E-05	3.53E-05	-4.314	-4.452	-0.138
	AlSO4+	7.68E-06	7.09E-06	-5.114	-5.15	-0.035
	ZnSO4	6.50E-06	6.51E-06	-5.187	-5.187	0.001
	CaSO4	2.12E-06	2.12E-06	-5.674	-5.673	0.001
	MgSO4	1.21E-06	1.21E-06	-5.917	-5.916	0.001
	HSO4-	8.70E-07	8.03E-07	-6.06	-6.095	-0.035
	MnSO4	7.65E-07	7.66E-07	-6.116	-6.116	0.001
	FeSO4+	2.78E-07	2.56E-07	-6.556	-6.592	-0.035
	CuSO4	1.09E-07	1.09E-07	-6.964	-6.964	0.001
	PbSO4	9.55E-08	9.56E-08	-7.02	-7.019	0.001
	FeSO4	6.39E-08	6.40E-08	-7.194	-7.194	0.001
	NaSO4-	2.51E-08	2.32E-08	-7.6	-7.635	-0.035
	Al(SO4)2-	7.88E-09	7.27E-09	-8.103	-8.138	-0.035
	CdSO4	7.55E-09	7.56E-09	-8.122	-8.121	0.001
	CaHSO4+	3.74E-09	3.45E-09	-8.427	-8.463	-0.035
	Zn(SO4)2-2	2.98E-09	2.15E-09	-8.527	-8.667	-0.14
	KSO4-	2.49E-09	2.29E-09	-8.605	-8.64	-0.035
	FeHSO4+2	3.33E-10	2.41E-10	-9.478	-9.619	-0.14
	AlHSO4+2	2.57E-10	1.86E-10	-9.589	-9.73	-0.14
	Fe(SO4)2-	2.00E-10	1.84E-10	-9.699	-9.735	-0.035
	FeHSO4+	1.49E-10	1.38E-10	-9.827	-9.862	-0.035
	Pb(SO4)2-2	2.45E-11	1.77E-11	-10.611	-10.751	-0.14
	Cd(SO4)2-2	4.53E-12	3.28E-12	-11.344	-11.485	-0.14
Si	1.45E-03					
	H4SiO4	1.45E-03	1.45E-03	-2.84	-2.84	0.001
	H3SiO4-	3.55E-10	3.28E-10	-9.449	-9.484	-0.035
	H2SiO4-2	2.67E-20	1.93E-20	-19.574	-19.715	-0.14
Zn	1.26E-03					
	Zn+2	1.25E-03	9.05E-04	-2.903	-3.043	-0.14
	ZnSO4	6.50E-06	6.51E-06	-5.187	-5.187	0.001
	ZnCl+	5.49E-07	5.06E-07	-6.261	-6.296	-0.035
	Zn(SO4)2-2	2.98E-09	2.15E-09	-8.527	-8.667	-0.14
	ZnOH+	8.07E-10	7.44E-10	-9.093	-9.128	-0.035
	ZnCl2	2.30E-10	2.30E-10	-9.639	-9.639	0.001
	ZnOHCl	4.22E-11	4.23E-11	-10.375	-10.374	0.001
	ZnCl3-	1.17E-13	1.08E-13	-12.932	-12.967	-0.035
	Zn(OH)2	1.04E-13	1.04E-13	-12.984	-12.983	0.001
	ZnCl4-2	3.02E-17	2.18E-17	-16.521	-16.661	-0.14
	Zn(OH)3-	1.08E-21	9.92E-22	-20.968	-21.003	-0.035
	Zn(OH)4-2	6.56E-31	4.75E-31	-30.183	-30.323	-0.14

#NAME? indices-----

Phase	SI	log	IAP	log	KT
Adularia	-6.21	-3.59	2.62 KAISI3O8		
Al(OH)3(a)	-5.65	6.35	12 Al(OH)3		

Albite	-7.94	-2.53	5.42 NaAlSi3O8
AlumK	-12.4	-17.89	-5.5 KAl(SO4)2·12H2O
Alunite	-6.07	-5.2	0.87 KAl3(SO4)2(OH)6
Analcime	-10.75	0.31	11.06 NaAlSi2O6·H2O
Anglesite	-1.88	-9.77	-7.89 PbSO4
Anhydrite	-3.56	-7.9	-4.34 CaSO4
Annite	-21.11	2.75	23.86 KFe3AlSi3O10(OH)2
Anorthite	-18.41	10.53	28.94 CaAl2Si2O8
Antlerite	-13.12	-4.83	8.29 Cu3(OH)4SO4
Atacamite	-10.61	-2.42	8.18 Cu2(OH)3Cl
Basaluminite	-8.73	13.97	22.7 Al4(OH)10SO4
Beidellite	-4.94	4.35	9.29 (NaKMg0.5)0.11Al2.33Si3.67O10(OH)2
Bianchite	-5.74	-7.5	-1.76 ZnSO4·6H2O
Birnessite	-8.84	34.76	43.6 MnO2
Bixbyite	-15.47	37.96	53.43 Mn2O3
Boehmite	-3.51	6.35	9.86 AlOOH
Brochantite	-17.98	-2.64	15.34 Cu4(OH)6SO4
Brucite	-14.73	3.33	18.06 Mg(OH)2
Cd(gamma)	-45.09	-30.68	14.41 Cd
Cd(OH)2	-12.77	0.88	13.65 Cd(OH)2
Cd(OH)2(a)	-13.79	0.88	14.67 Cd(OH)2
Cd3(OH)2(ε)	-26.9	-20.19	6.71 Cd3(OH)2(SO4)2
Cd3(OH)4S	-31.33	-8.77	22.56 Cd3(OH)4SO4
Cd4(OH)6S	-36.3	-7.9	28.4 Cd4(OH)6SO4
CdCl2	-12.26	-12.74	-0.48 CdCl2
CdCl2·2.5H2O	-10.73	-12.74	-2.02 CdCl2·2.5H2O
CdCl2·H2O	-11.11	-12.74	-1.63 CdCl2·H2O
CdMetal	-44.98	-30.68	14.3 Cd
CdOHCl	-9.79	-5.93	3.85 CdOHCl
CdSiO3	-11.77	-1.96	9.81 CdSiO3
CdSO4	-11.1	-10.53	0.57 CdSO4
CdSO4·2.7H2O	-8.85	-10.53	-1.68 CdSO4·2.67H2O
CdSO4·H2O	-9.22	-10.53	-1.32 CdSO4·H2O
Chalcanthite	-6.51	-9.22	-2.71 CuSO4·5H2O
Chalcedony	0.93	-2.84	-3.76 SiO2
Chlorite14A	-54.39	20.83	75.22 Mg5Al2Si3O10(OH)8
Chlorite7A	-57.94	20.83	78.76 Mg5Al2Si3O10(OH)8
Chrysotile	-30.16	4.31	34.47 Mg3Si2O5(OH)4
Clinoenstatite	-11.76	0.49	12.25 MgSiO3
Cotunnite	-6.96	-11.98	-5.02 PbCl2
Cristobalite	1	-2.84	-3.84 SiO2
Cu(OH)2	-7.14	2.19	9.33 Cu(OH)2
Cu2SO4	-31.55	-38.59	-7.04 Cu2SO4
CuMetal	-17.19	-29.37	-12.18 Cu
CuOCuSO4	-20.16	-7.03	13.14 CuO·CuSO4
CupricFerri	3.44	37.98	34.54 CuFe2O4
Cuprite	-20.05	-27.17	-7.12 Cu2O
CuprousFer	2.24	4.31	2.06 CuFeO2
CuSO4	-13.05	-9.22	3.83 CuSO4
Diaspore	-1.65	6.35	7.99 AlOOH
Diopside	-20.19	1.16	21.35 CaMgSi2O6
Diopase	-7.55	-0.65	6.9 CuSiO3·H2O
Epsomite	-5.81	-8.08	-2.27 MgSO4·7H2O
Fe(OH)2.7C	5.43	15.85	10.42 Fe(OH)2.7ClO.3
Fe(OH)3(a)	-0.45	17.89	18.35 Fe(OH)3
Fe3(OH)8	-9.23	37.9	47.14 Fe3(OH)8

Forsterite	-26.68	3.82	30.5 Mg2SiO4
Gibbsite	-2.79	6.35	9.14 Al(OH)3
Goethite	5.28	17.69	12.61 FeOOH
Goslarite	-5.39	-7.5	-2.11 ZnSO4:7H2O
Greenalite	-20.15	0.66	20.81 Fe3Si2O5(OH)4
Gypsum	-3.3	-7.9	-4.6 CaSO4:2H2O
H2(g)	-31.62	-31.56	0.06 H2
Halite	-8.7	-7.16	1.54 NaCl
Halloysite	-7.29	7.01	14.3 Al2Si2O5(OH)4
Hausmannite	-24.42	41.15	65.57 Mn3O4
Hematite	11.49	35.79	24.3 Fe2O3
Illite	-9.17	4.64	13.81 K0.6Mg0.25Al2.3Si3.5O10(OH)2
Jarosite(ss)	-0.78	29.76	30.54 (K0.77Na0.03H0.2)Fe3(SO4)2(OH)6
Jarosite-K	-3.13	29.44	32.57 KFe3(SO4)2(OH)6
Jarosite-Na	-6.22	30.5	36.72 NaFe3(SO4)2(OH)6
JarositeH	-6.61	30.86	37.47 (H3O)Fe3(SO4)2(OH)6
Jurbanite	-1.84	-5.07	-3.23 AlOHSO4
Kaolinite	-2.02	7.01	9.03 Al2Si2O5(OH)4
Kmica	-6.28	9.1	15.38 KAl3Si3O10(OH)2
Langite	-21.22	-2.64	18.58 Cu4(OH)6SO4:H2O
Lamakite	-8.14	-8.13	0.01 PbO:PbSO4
Laumontite	-11.58	4.85	16.43 CaAl2Si4O12:4H2O
Laurionite	-5.79	-5.17	0.62 PbOHCl
Leonhardite	-14.83	9.69	24.52 Ca2Al4Si8O24:7H2O
Litharge	-11.82	1.64	13.46 PbO
Magadiite	-5.93	-20.23	-14.3 NaSi7O13(OH)3:3H2O
Maghemite	2.49	35.79	33.3 Fe2O3
Magnetite	4.97	37.9	32.93 Fe3O4
Manganite	-6.36	18.98	25.34 MnOOH
Massicot	-12.03	1.64	13.67 PbO
Melanothallite	-15.71	-11.43	4.29 CuCl2
Melanterite	-6.85	-9.3	-2.44 FeSO4:7H2O
Minium	-41.84	36.49	78.33 Pb3O4
Mirabilite	-10.15	-12.12	-1.97 Na2SO4:10H2O
Mn2(SO4)3	-45.68	3.72	49.4 Mn2(SO4)3
MnCl2:4H2O	-12.35	-10.42	1.93 MnCl2:4H2O
MnSO4	-11.58	-8.21	3.37 MnSO4
Monteponite	-14.01	0.88	14.89 CdO
Montmorillonite	-5.26	5.64	10.9 (HNaK)0.14Mg0.45Fe0.33Al1.47Si3.82O10(OH)2
Montmorillonite	-3.01	3.9	6.92 (HNaK)0.09Mg0.29Fe0.24Al1.57Si3.93O10(OH)2
Montmorillonite	-4.68	4.94	9.62 Ca0.165Al2.33Si3.67O10(OH)2
Nantokite	-10.54	-20.4	-9.86 CuCl
Nsutite	-7.81	34.76	42.56 MnO2
O2(g)	-26.17	63.12	89.29 O2
Pb(OH)2	-7.14	1.64	8.78 Pb(OH)2
Pb2(OH)3Cl	-12.32	-3.53	8.79 Pb2(OH)3Cl
Pb2O(OH)2	-22.92	3.28	26.2 PbO:Pb(OH)2
Pb2O3	-26.2	34.84	61.04 Pb2O3
Pb2SiO4	-20.49	0.45	20.93 Pb2SiO4
Pb3O2SO4	-17.82	-6.48	11.34 PbSO4:2PbO
Pb4(OH)6SO4	-25.94	-4.84	21.1 Pb4(OH)6SO4
Pb4O3SO4	-28.53	-4.84	23.68 PbSO4:3PbO
PbMetal	-34.17	-29.92	4.25 Pb
PbO:0.3H2O	-11.34	1.64	12.98 PbO:0.33H2O
PbSiO3	-8.94	-1.2	7.74 PbSiO3
Phillipsite	-7.77	-3.06	4.71 Na0.5K0.5AlSi3O8:H2O

Phlogopite	-38.81	8.4	45.21 $\text{KMg}_3\text{AlSi}_3\text{O}_{10}(\text{OH})_2$
Plattnerite	-19.29	33.2	52.49 PbO_2
Portlandite	-20.89	3.51	24.2 $\text{Ca}(\text{OH})_2$
Prehnite	-25.81	11.2	37.01 $\text{Ca}_2\text{Al}_2\text{Si}_3\text{O}_{10}(\text{OH})_2$
Pyrochroite	-12	3.2	15.2 $\text{Mn}(\text{OH})_2$
Pyrolusite	-9.56	34.76	44.32 MnO_2
Pyrophyllite	0.47	1.33	0.86 $\text{Al}_2\text{Si}_4\text{O}_{10}(\text{OH})_2$
Quartz	1.41	-2.84	-4.25 SiO_2
Sepiolite	-18.1	-1.86	16.24 $\text{Mg}_2\text{Si}_3\text{O}_7 \cdot 5\text{OH} \cdot 3\text{H}_2\text{O}$
Sepiolite(d)	-20.52	-1.86	18.66 $\text{Mg}_2\text{Si}_3\text{O}_7 \cdot 5\text{OH} \cdot 3\text{H}_2\text{O}$
Silicagel	0.38	-2.84	-3.22 SiO_2
$\text{SiO}_2(\text{a})$	0.02	-2.84	-2.86 SiO_2
Talc	-24.86	-1.37	23.49 $\text{Mg}_3\text{Si}_4\text{O}_{10}(\text{OH})_2$
Tenorite	-6.11	2.19	8.31 CuO
Thenardite	-11.96	-12.12	-0.15 Na_2SO_4
Tremolite	-59.98	0.96	60.95 $\text{Ca}_2\text{Mg}_5\text{Si}_8\text{O}_{22}(\text{OH})_2$
Wairakite	-16.44	4.85	21.29 $\text{CaAl}_2\text{Si}_4\text{O}_{12} \cdot 2\text{H}_2\text{O}$
Willemite	-11.84	4.99	16.84 Zn_2SiO_4
Zincite(c)	-8.21	3.92	12.13 ZnO
Zincosite	-11.37	-7.5	3.88 ZnSO_4
$\text{Zn}(\text{OH})_2\text{-a}$	-8.53	3.92	12.45 $\text{Zn}(\text{OH})_2$
$\text{Zn}(\text{OH})_2\text{-b}$	-7.83	3.92	11.75 $\text{Zn}(\text{OH})_2$
$\text{Zn}(\text{OH})_2\text{-c}$	-8.28	3.92	12.2 $\text{Zn}(\text{OH})_2$
$\text{Zn}(\text{OH})_2\text{-e}$	-7.58	3.92	11.5 $\text{Zn}(\text{OH})_2$
$\text{Zn}(\text{OH})_2\text{-g}$	-7.79	3.92	11.71 $\text{Zn}(\text{OH})_2$
$\text{Zn}_2(\text{OH})_2\text{SO}_4$	-11.08	-3.58	7.5 $\text{Zn}_2(\text{OH})_2\text{SO}_4$
$\text{Zn}_2(\text{OH})_3\text{Cl}$	-14.18	1.02	15.2 $\text{Zn}_2(\text{OH})_3\text{Cl}$
$\text{Zn}_3\text{O}(\text{SO}_4)_2$	-32.89	-11.07	21.82 $\text{ZnO} \cdot 2\text{ZnSO}_4$
$\text{Zn}_4(\text{OH})_6\text{Si}$	-24.15	4.25	28.4 $\text{Zn}_4(\text{OH})_6\text{SO}_4$
$\text{Zn}_5(\text{OH})_8\text{Cl}$	-32.54	5.96	38.5 $\text{Zn}_5(\text{OH})_8\text{Cl}_2$
ZnCl_2	-17.52	-9.7	7.82 ZnCl_2
ZnMetal	-55.06	-27.64	27.42 Zn
$\text{ZnO}(\text{a})$	-7.39	3.92	11.31 ZnO
ZnSiO_3	-2.68	1.08	3.75 ZnSiO_3
$\text{ZnSO}_4 \cdot \text{H}_2\text{O}$	-7.41	-7.5	-0.09 $\text{ZnSO}_4 \cdot \text{H}_2\text{O}$

End of simulation.

Reading input data for simulation 2

End of run.

Appendix 8

Control and Treatment Options

Appendix 8

CONTROL AND TREATMENT TECHNIQUES

CONTROL

Diversión

The diversion of surface water from acid-producing material or ponded drainage, minimises the volume of acidic leachate (Taylor *et al*, 1997). This prevents dilution of the AMD and means that the leachate will be at a higher strength, but dilution to acceptable discharge standards is rare, and the cost of treating large volumes of water can exceed the cost of treating highly polluted drainage.

Covers and Seals

The prime function of covers and seals is to exclude oxygen and/or water. The cover needs to have a low permeability to either air or water. Cover types include:

- (i) Water covers
- (ii) Soil covers
- (iii) Micro barriers
- (iv) Synthetic barriers

(i) Water Covers

A cover of water stops oxidation of sulphidic mine wastes, leaving them largely chemically unreactive. Both the solubility of oxygen in water and the diffusion rate of oxygen through water are low. Therefore in the absence of convective transport, acid generation is negligible due to the low rate of oxygen transport through water (Sengupta, 1993).

Water covers are the most effective cover for prevention of sulphide oxidation, but are only feasible in areas where precipitation exceeds evaporation (i.e. in Tasmania and NE Victoria).

Underground and open-cut workings can be backfilled with waste rock and/or tailings in conjunction with flooding. Quartz sands and binders (i.e. flyash, cements or lime), can be added to help stabilize the backfilled material (Taylor *etal*, 1997). In drier climates however, permanent wetlands or complex layered covers (designed to trap precipitation on tailings surface), rather than lakes are more appropriate (OSS, 1997).

Detailed knowledge of the hydrology and water balance of the local of area must be taken into account when mine flooding and engineered water covers are being considered. It is important for the mineralogy, chemistry and reactivity of the waste-rock and/or tailings under local conditions to be detailed as the suitability of water covers, needs to be assessed on a site-by site basis (Taylor *etal*, 1997).

(ii) Soil Covers

There are a wide range of soil covers, ranging from simple clay barriers to complex, composite, multi-layer organic-bearing covers. Saturated soil covers are more effective at reducing oxidation than dry soil covers, due to the low diffusivity of oxygen in water, and lack of cracking (Sengupta, 1993).

Wet site composite soil covers usually consist of a compacted layer covered by layers designed to support vegetation and reduce erosion. Capillary break layers are sometimes installed below the compacted layer, these limit the upward migration of soluble constituents to the surface and subsequent damage to the vegetative cover (OSS, 1997). These layers can also reduce evaporation from the compacted material when placed above, by acting as a moisture retention layer. Effective wet site covers need to maintain a high degree of saturation in the compacted layer and promote runoff (Taylor *etal*, 1997).

Dry site covers aim to store relatively large volumes of infiltrating water long enough for the majority to be removed by evapotranspiration, before percolating into the underlying waste material. Alluvium, topsoil and oxide rock are commonly used, and often include vegetation support layers to enhance the store/release capacity of the cover (Taylor *etal*, 1997). An example of the typical design of a multi-layered soil cover is illustrated in Figure 1.

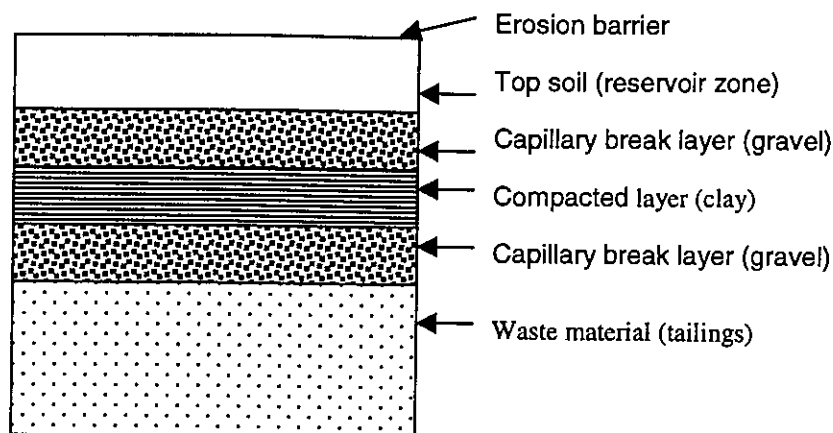


Figure 1. Multi-layered soil cover incorporating an infiltration barrier.

Soil covers still need to be tested for their long term performance in the presence of disruptive forces such as erosion, cracking, root action, burrowing animals and frost action (Sengupta, 1993). There is recent evidence to suggest that the covers currently used in Australia may not prevent long-term AMD (Taylor, 1997).

(iii) Micro Barriers

Sulphidic mine wastes can be mixed with a phosphate rock containing soluble hydroxyapatite (Taylor *et al*, 1997). Phosphate ions are released into solution under acidic conditions, and form insoluble iron phosphate compounds when they come into contact with pyrite oxidation products. Some studies suggest that this precipitate forms a passive surface coating over the waste rock fragments that inhibit further oxidation (OSS, 1997). But others believe that the hydroxyapatite gives no long term benefit, and merely provides some neutralising capacity at a very high cost (Taylor *et al*, 1997).

(iv) Synthetic Barriers

Synthetic barriers such as polyvinyl chloride (PVC) and high-density polyethylene (HDPE), have extremely low conductivities to air, thus they are an effective means in which to control sulphide oxidation and water infiltration (Sengupta, 1993). These barriers provide good medium term control, but degrade over time, lose plasticity and ultimately crack, and are commonly quite an expensive option (Taylor *et al*,

1997). Other synthetic materials include geopolymers, asphalts, and polymer-modified clay and grout barriers (e.g. flyash Portland cement mixture). These covers often suffer cracking and disruption over long periods of time, and can be prohibitively expensive (Sengupta, 1993).

Chemical

There are several control strategies that have been designed to act upon the chemistry of the waste material. These include:

- (i) Selective handling / encapsulation
- (ii) Blending / mixing and co-disposal

(i) Selective Handling / Encapsulation

AMD producing waste (or tailings) is surrounded with non acid producing (benign) materials in an effort to reduce flow of air and water to the waste material, and subsequent AMD outflow. The benign material is used for the base and sides with the surface covered with a compacted low permeable layer (usually clay). See Figure 2 (OSS, 1997).

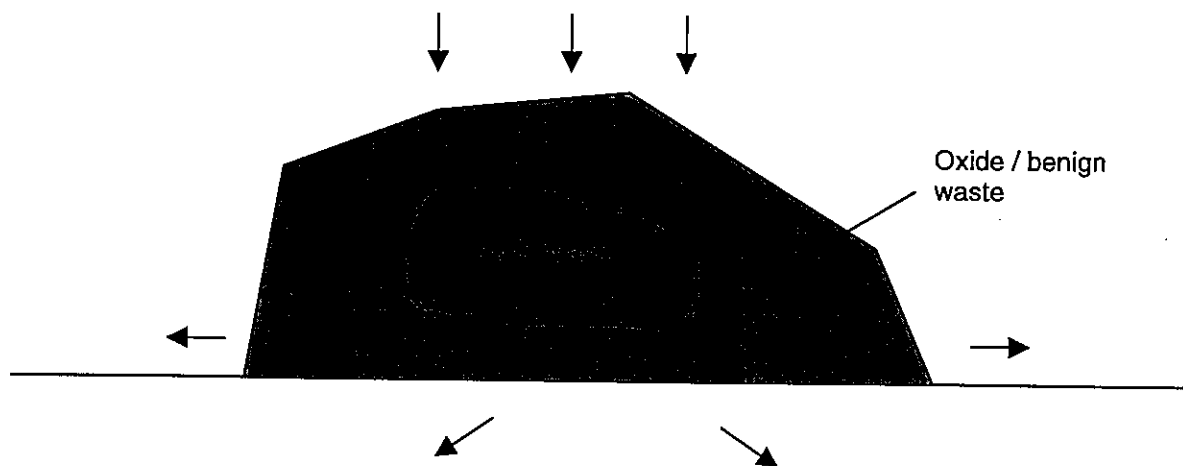


Figure 2. Encapsulation (after OSS, 1997)

(ii) Blending / Mixing and co-dispersal

This involves the blending/mixing and co-dispersal of AMD wastes/tailings, with non acid producing or acid neutralising materials (OSS, 1997). Co-dispersal involves

numerous small cells of AMD waste is surrounded by low permeability benign waste. Mixtures are more effective for waste rock rather than tailings and relies on the neutralising capacity of the non acid producing material rather than encapsulation. See Figures 3 (a) and 3 (b) (OSS, 1997).

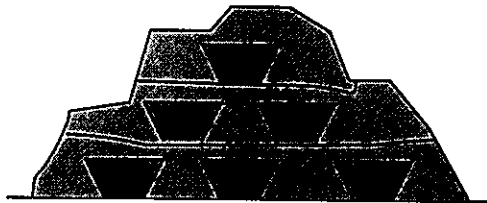


Figure 3 (a)
Co-disposal (after OSS, 1997)

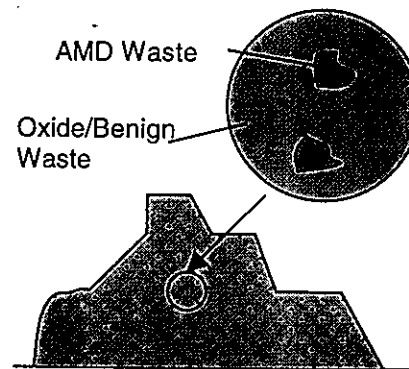


Figure 3 (b)
Mixture (after OSS, 1997)

Biological

Bactericides kill the sulphur oxidising bacteria which catalyse the conversion of ferrous iron to ferric iron under acid conditions (where ferric iron is the principal oxidant) (OSS, 1997). Popular bactericides include benzoate compounds, sorbate compounds, anionic surfactants such as sodium lauryl sulfate and phosphate compounds, and generally give a 50 - 90% reduction of acidity, and or sulfate (Sengupta, 1993). They do not provide permanent solutions to AMD, as they are relatively expensive, and easily degraded by infiltrating and percolating water. Slow-release rubber-based surfactant pellets have been developed to extend the effective lifetime of these agents, but they are only adequate for short term control of oxidation. Bactericides can also be applied with sprayers or hydroseeders to surfaces of piles and fields (Sengupta, 1993). They are often used to delay the onset of acid conditions or to reduce the secondary treatment costs (i.e. addition of lime) on temporary ore stockpiles or waste rock stockpiles, while more permanent remediation options are implemented (OSS, 1997).

TREATMENT

Dilution / Reaction

In suitable climates and locations, discharge and/or reaction with nearby water resources is regarded as a suitable treatment option (Taylor *et al*, 1997). Discharge standards can be achieved by:

- (i) Only releasing water during high rainfall or peak river flow
- (ii) Releasing water into waters with adequate natural buffering capacities
- (iii) Releasing water into marine or estuarine settings where AMD is treated by dilution, bicarbonate buffering and gypsum saturation.

Concentration

Evaporative concentration of acidic drainage can reduce the volumes of water to treat, which in turn can lower the cost of subsequent treatment reagents (e.g. limestone), which are used for at least partial treatment. (Taylor *et al*, 1997)

Chemical

Active Treatment

The most common treatment strategy for acid drainage is collection and neutralisation with alkaline reagents, followed by a settlement stage to recover fine metal precipitates (hydroxides). (OSS, 1997). There are a wide range of reagents available but cost, local availability and the target of the final pH all govern which are utilized. Possible reagents include (Taylor *et al*, 1997):

- Local acid neutralising host rocks (by disposing acidic drainage down abandoned underground workings)
- Acid neutralising waste rock (AMD can be irrigated over the waste rock pile)
- Acid neutralising tailings
- Alkaline tailings liquor (to treat some AMD)
- Kiln dust (comprises of several oxides but mostly CaO)
- Fluidised bed combustion ash (multiple metal oxides)
- Limestone (CaCO_3)
- Quicklime (CaO)
- Hydrated lime (Ca(OH)_2)
- Calcium peroxide (CaO_2 , trapzene; neutralises and oxidises)
- Dolomite ($\text{CaMg(CO}_3)_2$)
- Magnesite (MgCO_3)

- Caustic magnesia (MgO)
- Sodium carbonate (Na_2CO_3 , soda-ash)
- Sodium hydroxide (NaOH)
- Potassium hydroxide (KOH)
- Ammonia (NH_3 or $\text{NH}_4(\text{OH})$)
- Barium Carbonate (BaCO_3 , witherite)
- Hydroxyapatite ($\text{Ca}_5(\text{PO}_4)_3\text{OH}$), phosphate rock)
- Sodium orthosilicate (Na_4SiO_4).

Active treatments do encounter some problems. Several of these reagents form rinds or blankets of precipitates which coat (or armor) the reagent particles inhibiting complete reaction. Another problem is the formation of strong chemical gradients in thermally stratified treated water bodies (i.e. deeper than 3-5m)

Other chemical methods used to treat acid drainage without the addition of alkaline materials are:

- (i) *Sulphide precipitation / reduction* : soluble sulphide compounds (i.e. NaS) are used to facilitate toxic metal sulphide precipitation.
- (ii) *Non-alkaline co-precipitation* : Toxic metal concentrations are reduced by adding BaCl_2 to precipitate BaSO_4 .
- (iii) *Physical and / or Chemical oxidation* : Oxidation causes aqueous Fe(II) to convert to ferric hydroxide, aqueous Mn(II) to Mn(III) hydroxides and oxides and As(III) to less toxic As(V) . Oxidants include O_2 , Cl_2 , Ca(ClO)_2 , NaOCl , CaCl_2 , NaCl , FeCl_3 , CaO_2 , H_2O_2 and KMnO_4 .
- (iv) *Adsorption* : Coagulants and flocculants can be used to remove an array of metals from solution, due to their adsorption onto surfaces of other material. Coagulants include $\text{Al}_2(\text{SO}_4)_3$, FeSO_4 , $\text{Fe}_2(\text{SO}_4)_3$, FeCl_3 , NaAlO_2 , MgCl_2 , AlCl_3 and CaCl_2 . Common flocculants include Na_4SiO_4 , bentonite, metal hydroxides, starch derivatives, polysaccharides, sodium alginates, and synthetic reagents.
- (v) *Ion Exchange*: Natural or synthetic zeolites or a variety of synthetic polymeric resins can be used to substitute toxic metals for harmless ions.

(vi) *Electrochemistry* : Electrowinning techniques can now be used to extract solvents from typical acid drainage.

Passive Treatment

These systems are designed to introduce alkalinity into drainage waters. Some of these methods include:

(i) *Open Limestone Drains* : uncovered drainage channels filled with aggregate limestone. Armoring reactions are combated by scaling up the volume of reagent as it is assumed that very low proportions of limestone will be available for dissolution.

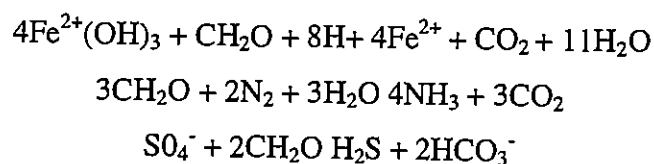
ii) *Anoxic Limestone Drains (ALD)* : Acid drainage is passed through a coarse limestone gravel which excludes oxygen, thus keeping iron in a reduced (ferrous) state which is soluble. This minimises metal hydroxide precipitation which would reduce the efficiency of the drain. Once the pH has been adjusted and drained from the channel, aeration and ponding of the discharge allows oxidation, hydrolysis and precipitation to occur. ALD's are only effective as short term remediation as it requires ongoing maintenance as the alkaline materials are consumed.

(iii) *Successive Alkalinity Producing Systems (SAPS)*: These avoid some of the ALD problems by increasing alkalinity levels in 'clean' runoff/drainage streams, and then introducing these streams into acid drainage streams. Limestone drains provide the alkalinity, and armoring is avoided by maintaining anoxic conditions. These systems are followed by aerobic settling ponds and sometimes wetlands which help 'polish' the effluents (OSS, 1997).

(iv) *Limestone Ponds (LSP) and Reverse Alkalinity Producing Systems (RAPS)*: AMD seeps or groundwater discharges are treated by creating a pond around the zone, and adding limestone and/or organic matter to it.

Biological

Biological AMD treatment methods are commonly used for final polishing of low strength acidic drainage, in conjunction with other remediation measures (Taylor *et al*, 1997). When anaerobic conditions are maintained sulphur reducing bacteria thrive and as a consequence remove metals and consume H^+ ions. Examples of typical microbially mediated reactions in the anaerobic zone include (Sengupta, 1993):



Biological treatment methods include:

- (i) Wetlands
- (ii) Acid reduction using microbiology (ARUM Cells)
- (iii) Bioremediation in a pit lake
- (iv) Biosulphidation Metal recovery (Appendix *

(i) Wetlands

(This applies to both natural and constructed systems). Wetlands are generally anaerobic, organic-rich, water-saturated environments with a large water/air surface. They are quite effective in remediating low-medium flows and medium -low strength acid drainage. This is achieved through processes such as filtering, cation exchange, redox reactions, neutralisation reactions and complex microbiological processes (Taylor *et al*, 1997). Limitations to wetlands establishment include difficulties in treating large flows, physical relief and availability of land for large wetland ponds. The performance of wetlands can degrade over time if the substrate becomes clogged with precipitate sludges or saturated with metals.

(ii) ARUM Cells

This involves iron reduction or oxidation, and a removal stage, followed by anaerobic microbial reduction and the establishment of a self sustaining organic

supply to the anaerobic cell (Taylor et al, 1997). This approach is still under development at this stage.

(iii) Bioremediation in a pit lake

This is also in the development stage, with trials being conducted to investigate the role of bacteria in reducing acid and sulfate levels in largely static bodies of water (Taylor et al, 1997).

RECENT ADVANCES

Numerous AMD control and treatment strategies are currently under development. Some of these are outlined below.

Self Healing / Self Sealing Barriers

These are based on key reactions that naturally generate hardpans in some tailings dams. Oxygen diffusion rates can be decreased with the formation of low permeability calcite \pm gypsum \pm metal hydroxide cement layers generated by the reaction of lime-based reagents and acid water (Taylor et al 1997).

Porous Reactive Walls

Some early results have indicated that porous walls comprised of reactive organic matter (e.g. leaf mulch, sawdust, sewage sludge, wood chips) installed into subsurface porous media have the capacity to remediate acidic groundwater

Non-Reactive Treatment Sludge

A product developed by a US organisation called Keeko, is reportedly capable of generating highly alkaline conditions and then encapsulating the resulting metal precipitates in a silica cement.

Neutral Barrier Technology (NBT)

A technique is under development to control the flow of liquid and gas within porous media. Oxygen diffusion and water infiltration into sulphidic wastes is minimised through controlled precipitation of carbonate minerals in pore spaces. It is hoped that

NBT will be able to control AMD in waste rock piles, tailings dams and polluted groundwater systems.

Kaolinite Amorphous Derivative (KAD)

This strategy removes a range of metals from acid drainage by enhancing the ion exchange capabilities of a modified aluminosilicate.

Appendix 9

Biosulphidation Metal Recovery - (internet data)

The Biosulphide Process

Water Treatment - Metal Recovery

Technical Summary

Introduction Biological Stage Chemical Stage Pilot Results Cost Comparison

Process Background

The Biosulphide Process is an integrated chemical-biological process designed to treat metal-contaminated, sulphate-rich, waste streams. The Biosulphide Process has been developed over the past 10 years to solve the increasing problem of treatment and disposal of solutions, gases, and waste streams that contain heavy metals, thiosalts, and sulphate species at low pH. The Biosulphide Process was developed with the objective of treating acidic waste streams and concurrently recovering saleable metal and sulphide-based co-products.

Process Objectives

The objective in the development of the Biosulphide Process was to provide a cost effective process that could be easily engineered and operated, utilized conventional equipment, integrated known chemical and biological process steps, and had a stable biological stage. An additional objective was that the process would not only remove dissolved metals to very low levels, but would also isolate metal concentrate products from the water for sale to offset or eliminate the costs of treatment. It also was very important that the process could be easily integrated into existing operations, without special personnel or training.

Process Description

The Biosulphide Process can be divided into two stages: a chemical circuit in which treatment of the drainage occurs, and a biological circuit in which inexpensive reagents are generated for use in the chemical circuit. In this manner each stage can operate at maximum efficiency independent of the other. The process can be generalized as follows:

The Biosulphide Process

Water Treatment - Metal Recovery

The Biosulphide Process provides highly effective metals removal using "off-the-shelf" components at greatly reduced operating costs with capital costs comparable to those of lime treatment. From the sale of recovered metals, some treatment plants can operate at a profit.

The Biosulphide Process is a process for the treatment of waters contaminated with metals, including:

- Acid Rock Drainage (ARD, AMD)
- electroplating waste solutions
- metal-laden process streams
- groundwater

By utilizing sulphide precipitation technology, the Biosulphide Process can:

- achieve effluent metal levels far below conventional technologies
- selectively recover metals as high-grade sulphide products
- precipitate metals "in pipe", eliminating reaction vessels
- form product sludges more rapidly, reducing clarifier requirements
- form denser product sludges, reducing associated costs
- lower capital costs due to higher settling rates, reduction of reaction vessel requirements

By recovering and selling valuable metals the Biosulphide Process removes toxic metals from waste sludges, with the following benefits:

- reduction of operating expenses
- potential to operate at a profit at many sites during treatment
- reduction or elimination of waste sludge production

At sites where waste sludges are produced there will be:

- reduced waste sludge volume
- increased waste sludge density
- reduced waste sludge toxicity
- reduced sludge disposal and monitoring costs

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The Biosulphide Process

Water Treatment - Metal Recovery

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Biological Stage



The Biosulphide Process utilizes naturally-occurring anaerobic bacteria called sulphate reducing bacteria (or SRB) such as *Desulphovibrio desulphuricans* (at left, bar = 10um) to convert sulphate present in the water to sulphide, with the simultaneous generation of alkalinity. Sulphide is stripped into the gas phase as hydrogen sulphide and applied to the water in the chemical circuit. Some portion of the sulphide generated remains in the aqueous phase and is added to the chemical circuit with the bioreactor discharge water. The bioreactor discharge water also contains alkalinity generated by the bacteria which is used to neutralize the acidity of the water in the chemical stage. In some applications the biological circuit receives as little as 4% of the total flow of water in the chemical circuit, depending on metal removal and alkalinity requirements.

The bacteria need a supply of nutrients to survive. More specifically, they require sources of sulphate, carbon, energy, along with lesser quantities of nitrogen, phosphorous and potassium. Sulphate is generally provided by the water being treated. Carbon and energy requirements are met with a system developed NTBC to provide a reliable, inexpensive supply from natural gas, propane, or other low cost fuel. Nitrogen, phosphorous and potassium are met with small additions of inexpensive fertilizer. Reagent costs are usually in the order of \$0.11/kg sulphate metabolized by the bacteria.

A photograph of the 10m³ Britannia Biosulphide pilot plant showing the biological circuit.



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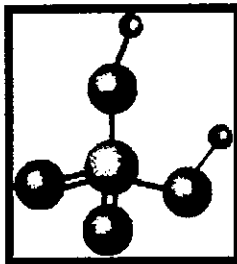
The Biosulphide Process

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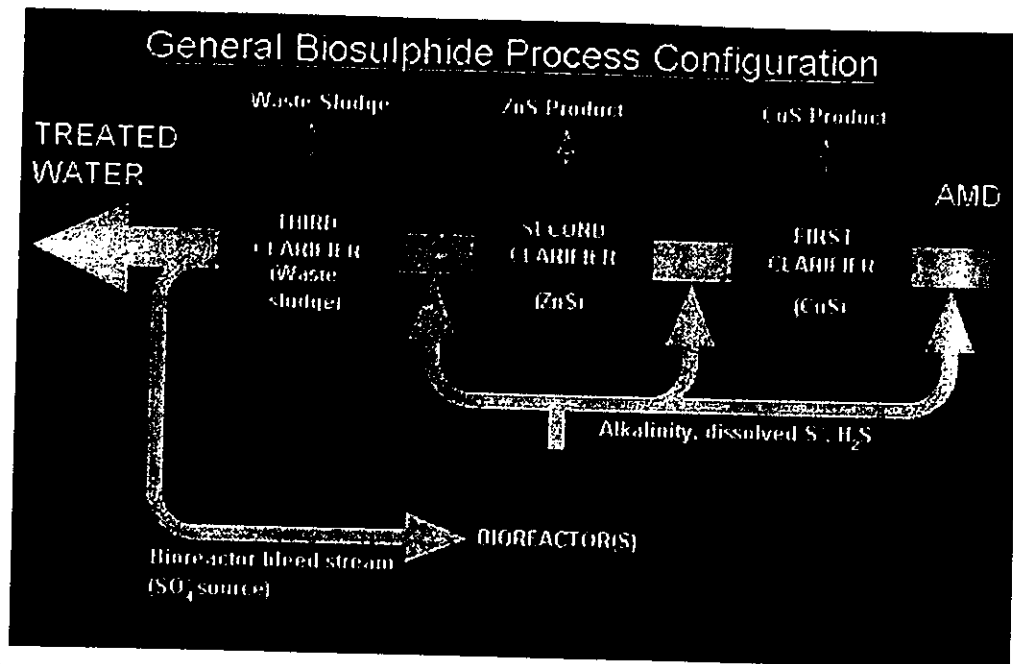
Chemical Stage



The chemical stage of the Biosulphide Process receives the entire flow of water to be treated, applying the sulphide and alkalinity generated in the biological stage for the precipitation of metals and neutralization of acidity. Because the precipitation of metals as sulphides occurs sequentially based on the solubility of the sulphide compounds, control of process parameters can result in a highly selective recovery. The selective recovery of metals is used to produce high-purity metal sulphide products for the generation of revenues through on-site processing or sale to smelters.

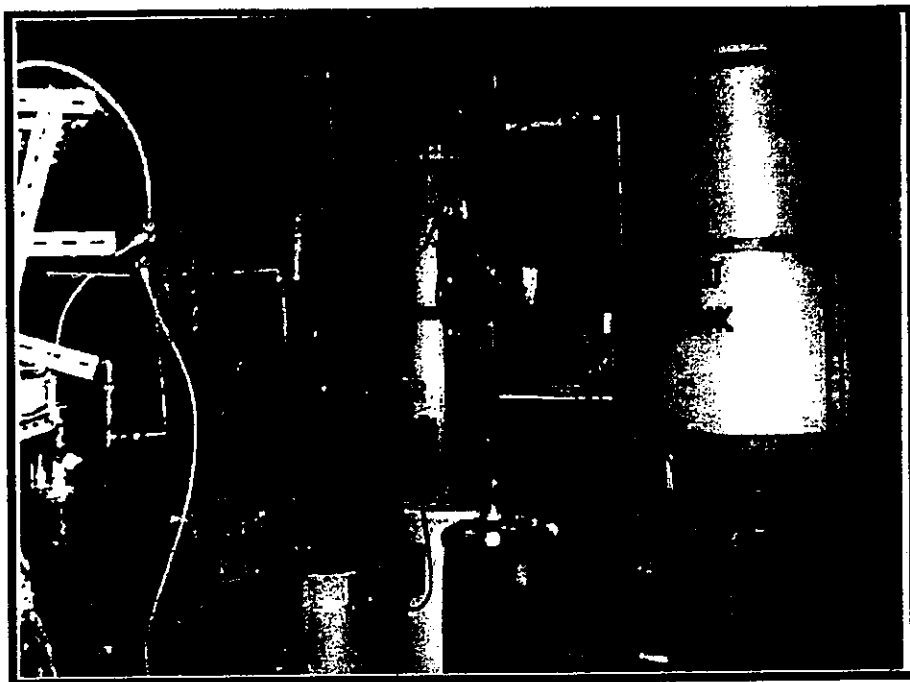
As shown in the [generalized process schematic](#), the chemical circuit consists of a series of precipitation stages (three are shown). Each precipitation stage consists of a clarifier and a means of introducing sulphide (and alkalinity, if required), together with some process control equipment (not shown). Typically there will be one precipitation stage for each metal that is deemed to be recoverable for economic or technical reasons, with a final stage for the removal of any remaining metals that are economically less attractive or inseparable with current technology. A common Biosulphide schematic from previous pilot projects includes a precipitation stage for copper recovery, a second precipitation stage for zinc recovery, and a final stage for the removal of iron and aluminum as a waste sludge during final neutralization of the water. Although a waste sludge is produced in this type of Biosulphide system, the sludge is much less toxic than other sludges (eg. lime sludge) due to the absence of copper, zinc, and other metals that may be present in smaller quantities (eg. cadmium, lead, mercury, silver). The waste sludge is also of reduced volume due in part to the absence of the above-mentioned metals, but also due to the formation of iron sulphides and the absence of gypsum sludge produced with lime neutralization. [A more detailed comparison with lime treatment](#) is also available.

A photograph of the 10m³ Britannia Biosulphide pilot plant showing the chemical circuit.



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Summary of Results of the 10m³ Britannia Pilot Project

The Biosulphide Process was piloted for over a year in a 10m³ pilot system at the former Britannia mine, north of Vancouver, B.C.. The results of this project demonstrate the process to be highly effective at removing copper, cadmium, and zinc to part-per-billion levels. In addition, the process is projected to operate at a net overall profit of about \$130,000 per year during the treatment through the sale of copper and zinc sulphide concentrates. Capital costs of a full-scale plant were estimated to be at least \$1 million less than a lime plant due primarily to higher settling rates, the elimination of reaction vessels, and the absence of waste sludge requiring disposal.

During the pilot project, test runs were conducted to address the following areas of concern:

- **106 Hour Continuous Operation** The plant was operated continuously for 5 days to demonstrate the sustainable nature of the treatment. The run was highly successful; discharge metal levels were consistently low, typically less than 0.01 mg/L copper, less than 0.001 mg/L cadmium, and less than 0.2 mg/L zinc.
- **Seasonal Flow Changes** To accommodate the highly variable drainage flows at Britannia, NTBC has developed a system of economically storing bioreactor sulphide during low-flow seasons for use in times of peak flow, thereby permitting the bioreactors to operate at a constant optimum level year-round. Several runs were conducted where the flow through the chemical circuit was increased substantially while the bioreactors maintained a constant, normal rate of operation. The results demonstrate the sulphide storage system to be highly effective, with no compromise in treatment during the flow increase.
- **Selective Metal Recovery. Product Concentrate Grades** The process has demonstrated a highly selective removal of copper and zinc; more than 99.9% of the copper is removed in the copper stage, with the zinc remaining for removal in the zinc stage. This selective precipitation has resulted in very good product concentrate grades of 41% copper in the copper product, and 26% zinc in the zinc product.

Removal of Dissolved Metals in the Britannia Biosulphide Pilot Plant Showing Selective Metal Recovery



- 47 Hour Total Shut-Down** To demonstrate the resiliency of the bacteria to operational upsets, everything was turned off for nearly two days. After 47 hours the supply of heat and nutrients was restored, at which time the bioreactors displayed no measurable loss of performance as a result of the shut-down. From this test it is concluded that equipment failures that result in 2 day interruptions of bioreactor operation can be tolerated without a loss of performance on repair and start-up.
- Operational Upsets** During the 47 hour shut-down, the chemical circuit was operated using only the stored sulphide produced on-site previously. The process functioned very well, providing complete drainage treatment in the absence of active bioreactors. This test served to demonstrate that substantial operational upsets can be accommodated without compromising the degree of metal removal from the water, assuming that non-power intensive means of sulphide introduction are employed.



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Cost Comparison With Lime Treatment

Although a comprehensive comparison of costs is difficult due to the highly site-specific nature of important cost factors, some generalities can be made.

The great strength of the Biosulphide Process is its ability to reduce or eliminate the net operating cost during the treatment by recovering metals instead of disposing of them. Some sites can even operate at a substantial profit. Metal recovery not only generates revenues from the sale of products, but also greatly improves the nature of sludge for disposal (if a waste sludge is produced at all). Increased density, with reduced toxicity and volume, are among the potential benefits in waste sludge nature.

No unusual or highly specialized equipment is required in the construction or operation of a Biosulphide facility. As a result, the capital costs associated with large-scale Biosulphide treatment plants are generally comparable to those of similarly-sized lime plants. A substantial reduction in capital costs can be realized at some sites, however, depending on the drainage and treatment requirements. At Britannia, for example, copper and zinc removal constitutes a complete treatment of the drainage. Because of the nature of sulphides, the Biosulphide Process removes copper and zinc to below target levels in an acidic environment. In contrast, a lime plant requires a pH of over 10 to achieve target zinc discharge levels due to the relatively soluble nature of the zinc hydroxide precipitate formed. The result is that there is little difference in lime plant capital and operating costs between a "zinc removal only" plant and a full treatment plant. A "zinc removal only" Biosulphide plant, however incurs lower capital costs due to the smaller overall size of the plant, and the simpler plant design. At Britannia this results in an estimated reduction in full-size plant capital cost of over \$1,000,000.

A more detailed comparison with lime treatment is also available.



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The Biosulphide Process

Water Treatment - Metal Recovery

Comparison with Lime Treatment

Lime neutralization is the standard, accepted method for treating acidic, metal-contaminated water. Increasingly, however, the process is recognized as a "non-solution" to the pollution problem presented by these effluents. The production of toxic sludges that are difficult to settle and prone to redissolving, as well as the cost of adding lime "in perpetuity" are among the primary reasons to look for a new, better method.

The metal sulphides produced in the Biosulphide Process have several advantages over the metal hydroxides produced in lime plants. One key point is that sulphides are orders of magnitude more insoluble, making it much easier to achieve extremely low levels of metals in the treated water (see graph comparing sulphide and hydroxide solubilities).

COST COMPARISON

Although an accurate comparison of costs is difficult due to the highly site specific nature of many cost factors (power, labour, and reagent expenses, sludge disposal costs, etc...), some important generalities can be made.

Capital Costs

Metal sulphides are extremely rapid to form, permitting a substantial reduction or elimination of the main reaction vessels in a Biosulphide plant. Metal sulphide particles are also faster to settle, and will form a denser sludge, than their hydroxide counterparts, reducing clarifier requirements.

The capital cost of a Biosulphide plant will generally be similar to that of a comparable lime plant, but can be dramatically lower for certain applications. No exotic or unusual equipment is required in the construction or operation of a Biosulphide Process treatment facility.

Operating Costs

Because lime treatment precipitates metals simultaneously into one waste product, there is no option for recovery of metal values during treatment. In a Biosulphide plant, however, metals are isolated into high-grade concentrates for sale to smelters or on-site processing. In terms of operating costs, this selective recovery achieves two principal benefits. Firstly, operating costs can be offset with the revenue generated through the sale of products. Some sites can actually operate at a profit as a result, depending on treatment requirements and metal quantities. Secondly, waste sludges will be of reduced quantity, increased density, and reduced or eliminated toxicity (if waste sludges are produced at all during treatment). The net result of this

is a potentially huge reduction in the costs of sludge transportation, disposal, and long-term monitoring.

NTBC Research has developed a system of providing inexpensive bacterial nutrients from hydrocarbon fuels, resulting in substantial reagent savings. In general, reagent expenses in a Biosulphide plant will be in the range of \$0.11/kg SO₄ metabolized by the bacteria, which may be as little as 4% of the total SO₄. In contrast, lime plant reagent costs are typically \$0.15 to \$0.20/kg SO₄ treated, and may react essentially all of the SO₄ present. Additional savings are achieved in the Biosulphide Process through the reduced power requirements resulting from smaller reaction vessels and reduced mixing requirements. Overall power requirements will be lower in a Biosulphide plant than in a comparable lime plant.

At sites where a bond is required at closure, the reduction of long-term treatment operating costs can translate into multi-million dollar savings in the required bond amount at closure.

Comparison of Estimated Costs. Full-Scale Plant at Britannia:

The following comparison of capital and operating costs was prepared for the Britannia Biosulphide pilot project. Lime process costs were estimated by Steffen, Robertson, and Kirsten in their 1991 report on the Britannia pollution.

	Lime Process	Biosulphide
Projected Capital Expense	\$3.4 million	\$2.2 million
Projected Overall Annual Operating Profit	-\$910,000	+\$130,000

The Lime amounts in the above table are in 1991 dollars.

The Lime plant operating cost does not include a full projection of sludge disposal costs.

While not operating at a large profit, the Biosulphide Process is shown to be the only option for the Britannia site that is not a long term financial burden to the owner. This is the primary objective of NTBC, to provide sustainable, economically attractive long-term treatment of acidic drainage through metal recovery.

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Appendix 10

Rock Catalogue

**Precipitates: Acid Mine Drainage in the Bakers Creek Waste Rock Dump,
Hercules, W Tas.**

Catalogue #	137944	137945
Field #	Site F	Site C
Rock Name	precipitate	precipitate
Rock Description	Chlorite, Pyrite Hematite, White mica	Chlorite, Pyrite Hematite, White mica
AMG Northing	5366870N	5366420N
AMG Easting	376340E	376580E
Preps	S	S