CONTINUOUS BIOLOGICAL LEACHING OF COPPER

FROM A CHALCOCITE ORE AND CONCENTRATE IN A SALINE ENVIRONMENT

by

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ABSTRACT

Biological leaching of a chalcocite ore and its concentrate was studied in both batch and continuous tests in a 3 g/L chloride environment. The feed material was obtained from the Zaldívar mine in Chile. The total copper contents of the ore and concentrate were 1.56% and 17.3% respectively, and the particle sizes were P_{80} 108 µm and P_{80} 53 µm respectively. The mineralogy consisted mainly of chalcocite (Cu_2S), pyrite (FeS_2) and brochantite ($Cu_4(SO_4)(OH)_6$). A mixed bacterial culture dominated by *Thiobacillus ferrooxidans* was obtained from the mine site for use in this study.

Shake flask and batch stirred reactor tests were carried out to determine the amenability of both the ore and the concentrate to biological leaching. The effects of several variables on the rate and extent of copper extraction were investigated. In the shake flask tests and the batch reactor tests, it was determined that the bacterial culture was capable of oxidizing both the ore and the concentrate with average copper extractions of 90% and 97% obtained respectively.

Continuous biological leaching was carried out in a series of three stirred tank reactors (4 L, 2 L and 2 L) for a period of four months. The bacteria were able to adapt to progressively lower system residence times and higher pulp densities. At the lowest residence time achieved of 2⅔ days with a pulp density of 10%, the steady state copper extraction was 82% for the ore and 83% for the concentrate. The copper bioleach rate achieved in the first bioleach stage was 16 mg/L/h for the ore and 250 mg/L/h for the concentrate. This study demonstrated the feasibility of the continuous biological leaching of either a chalcocite ore or a chalcocite concentrate in a 3 g/L chloride environment. The bacteria-rich leach residue from the process would be an excellent source of inoculum for biological heap leaching.

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Nomenclature

a_i =   relative activity of component i

D_i =   impeller diameter (m)

Eh =   redox potential (mV)

N =   stirrer speed (RPM)

N_t =   number of bacteria at time t

N_o =   number of bacteria at time zero

P_{80} =   particle size 80% passing (μm)

s.g. =   specific gravity

SHE = standard hydrogen electrode reference

u =   bacterial growth rate (day^{-1})

t =   time (days)

v_t =   impeller tip speed (m/s)

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Dedicated to Beaker

1.0 INTRODUCTION

Conventional copper production involves mining of copper ores, followed by milling to extract and concentrate useful minerals, and copper recovery by pyrometallurgical smelting. A major disadvantage of smelting is the production of sulphur dioxide (SO_2) gas, which is a contributor to the production of acid rain. Modern smelters are designed to capture most of the SO_2, which is used to produce sulphuric acid. However, increasingly stringent legislated limits on SO_2 emissions to the environment have stimulated interest in hydrometallurgical processes in which sulphide is converted to elemental sulphur or sulphate. These include chloride leaching, ammonia leaching, oxygen pressure leaching and biological leaching (Biswas and Davenport, 1994; Everett, 1994; Peters, 1992). One advantage of biological leaching is potentially lower capital and operating costs, which are increasingly important as available ore grades decrease. Another advantage is the fact that copper is extracted in sulphate media. More success has been achieved in improving solvent extraction and electrowinning processes with sulphate solutions than with other media such as chloride or ammoniacal solutions (Peters, 1992).

Biological leaching is the bacterially catalysed dissolution of metals from sulphide minerals. The dominant bacteria in biological leaching processes are *Thiobacillus ferrooxidans*, which live in acidic environments, gaining energy for growth from the oxidation of reduced sulphur and iron compounds. Copper is the metal most commonly extracted by biological leaching. It has been reported that biological leaching accounts for about 5% of the world's primary copper production (Torma et al., 1979) and about 20% of copper production in the United States (Torma and Apel, 1992). The main copper sulphide minerals are chalcopyrite ($CuFeS_2$), chalcocite (Cu_2S), covellite (CuS) and bornite (Cu_5FeS_4). Chalcopyrite is the most abundant copper sulphide mineral but it is

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the least reactive to biological leaching. The secondary sulphide minerals, chalcocite, covellite and bornite, are much more reactive to biological leaching than chalcopyrite.

Bioleaching in the laboratory is accomplished in shake flasks, columns, batch reactors and continuous stirred tank reactors. Commercial biological processes currently used for copper extraction include in-situ, dump and heap methods. There are currently no commercial continuous stirred tank reactor bioleaching processes for copper extraction, although the process is used commercially for the treatment of refractory gold ores and concentrates. The potential advantages of continuous biological leaching include higher metal recovery, higher copper leach rates and more effective control of the process. The main disadvantage is the cost, which is higher than costs for in-situ, dump and heap bioleaching.

This study considered the continuous biological leaching of a copper sulphide ore and its concentrate from the Zaldívar mine in Chile. Zaldívar is an open pit mine located in the desert region of northern Chile, 196 km to the southeast of Antofagasta and 1400 km north of Santiago. The Compañia Minera Zaldívar is a 50% joint venture between Placer Dome Inc. and Outokumpu Copper Resources B.V. The deposit is porphyry copper containing about 50% oxide copper and 50% sulphide copper. There are two main rock zones, one of which is an andesite rock containing either copper oxides, or mixed copper oxide and sulphide minerals. The other is a rhyolite copper porphyry containing mainly supergene sulphide minerals. In the rhyolite zone, which was the source of the feed material for this study, chalcocite (Cu_2S) makes up about 30 to 60% of the total sulphide content, and about 85 to 90% of the total copper content. The other main sulphide mineral is pyrite (FeS_2). The main copper oxide mineral is brochantite ($Cu_4(SO_4)(OH)_6$).

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The Zaldívar copper extraction process includes ore crushing and sizing, heap leaching, and solvent extraction and electrowinning (SX-EW). A fines fraction is separated from the crushed ore, which is about 3% of the total ore mined. When fully commissioned, the annual production will be 125 000 tonnes of cathode Grade A copper. There will be an acid leaching heap for the oxide copper ore and a biological leaching heap for the sulphide copper ore. Dump leaching will contribute a small percentage of copper production. Various options are under consideration for the treatment of the ore crusher fines. One option is to send the fines to a continuous biological leaching system. Another option is to produce a flotation concentrate from the ore crusher fines, and send this concentrate to a continuous biological leaching system. Both of these options were considered in this Master's project.

This study was designed to determine the feasibility of the continuous biological leaching of the Zaldívar ore and concentrate in a saline environment of 3 g/L chloride. A 3 g/L chloride solution was used because the Zaldívar plant water was expected to have this chloride concentration. The Zaldívar ore was proposed to be an excellent application for continuous biological leaching because the main copper sulphide mineral is the secondary sulphide chalcocite, which is known to bioleach more fully and rapidly than other copper sulphide minerals. In addition, either the ore crusher fines or the fines concentrate would be an ideal feed to a continuous biological leaching circuit. The leachate could be sent to SX-EW for production of copper and the bacteria-rich solids could be used as a source of inoculum for the sulphide heap.

Figure 1.1 gives a schematic diagram that shows how a continuous system for biological leaching of the fines concentrate could fit into the flowsheet of the Zaldívar mine. For continuous biological leaching of the ore crusher fines, the flotation operation shown in Figure 1.1 would not be required.

The leaching solution could be sent to SX-EW for copper recovery. The bacteria-rich solid residue from the product slurry could be sent to sulphide heap leaching as a source of bacterial inoculum; addition of these bacteria could increase copper leaching rates in the heap. Some of the copper remaining in the continuous bioleaching system residue could be extracted by heap leaching.
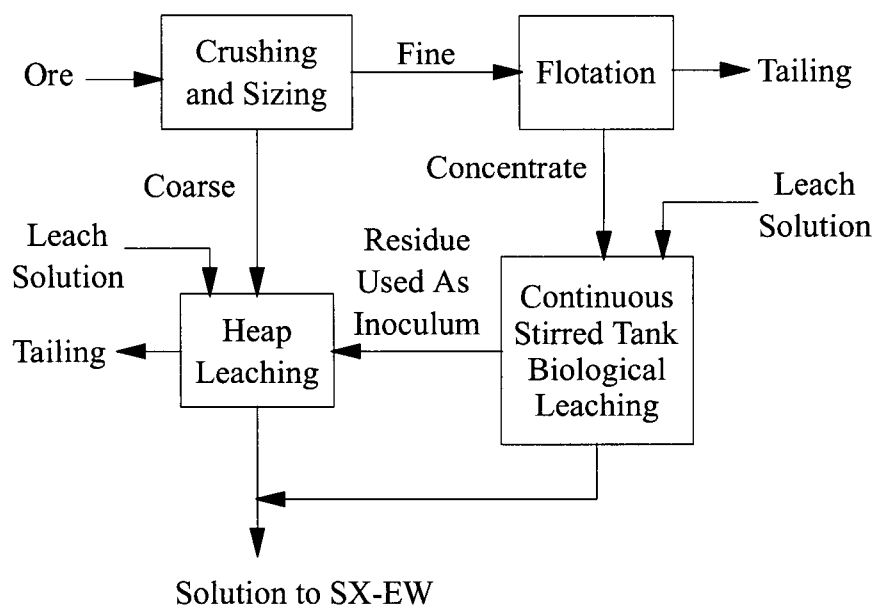
Figure 1.1 -   Proposed Flowsheet for the Continuous Biological Leaching of a Chalcocite Concentrate at the Zaldívar Mine in Chile

In this study, shake flask, batch reactor and continuous stirred tank reactor biological leaching tests were conducted on the Zaldívar ore and concentrate. The objectives of the shake flask experiments were to verify that the Zaldívar ore and concentrate were amenable to biological leaching, and to

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determine the ultimate extent of copper extraction possible. The objectives of the batch reactor experiments were to verify some of the results achieved in the shake flask experiments, and to develop an experimental method for use in the continuous bioleaching experiments. The objective of the continuous reactor experiments was to determine the feasibility of the continuous stirred tank biological leaching of the Zaldívar ore and concentrate in a 3 g/L chloride environment.

2.0   LITERATURE REVIEW

Biological leaching is the process in which metals are dissolved from sulphide minerals into aqueous solutions using bacteria as catalysts. The valuable metals can be recovered from the leachate using either conventional technologies such as cementation, or solvent extraction followed by electrowinning. Applications include pretreatment of refractory gold ores and extraction of metals from sulphide minerals. The background and theory of this topic have been covered thoroughly in several books (Barrett et al., 1993; Ehrlich and Brierley, 1990; Rossi, 1990; Karavaiko et al., 1988), and review papers (Ingledew, 1982; Torma and Bosecker, 1982; Murr, 1980). The focus of this review was literature relating to the continuous biological leaching of copper sulphide minerals, particularly chalcocite (Cu_2S).

2.1   Microbiological Concepts

The various bacteria involved in biological leaching and the mechanisms of their growth are described thoroughly in Brock and Madigan (1991) as well as several other sources (Ehrlich and Brierley, 1990; Rossi, 1990; Torma and Bosecker, 1982).

2.1.1   Microorganisms

The most commonly used bacteria in biological leaching processes are the aerobic lithoautotrophs *Thiobacillus ferrooxidans*. This bacterium was first described after being isolated from acid mine drainage waters in 1947 (Colmer and Hinkle, 1947). They are called aerobic lithoautotrophic bacteria because they require oxygen, inorganic compounds and carbon dioxide for growth. *T. ferrooxidans* are active in the temperature range of 5 to 45°C but the optimum temperature range for their growth is 35°C to 40°C. They require an acidic environment in the range pH 1.4 to pH 6.0, with optimum

growth occurring at about pH 2.35 (Temple and Colmer, 1951). These bacteria derive energy from oxidation of sulphur, reduced sulphur and reduced iron compounds. The energy source is oxidized by the bacteria, releasing electrons that are used to drive cellular reactions. *T. ferrooxidans* are rod shaped with dimensions of 0.5 x 2 microns, and they have a tail-like structure, called the flagellum, that allows motility. Like all bacteria, they have the ability to adapt to changes in their environment. *T. ferrooxidans* can readily attach to solid surfaces although the mechanism of attachment is not fully understood (Heffelfinger, 1990).

Other bacteria that require similar growth conditions to those described above for *T. ferrooxidans* are believed to be involved in biological leaching. Examples are *Thiobacillus thiooxidans*, which oxidize sulphur and reduced sulphur compounds, and *Leptospirillum ferrooxidans*, which oxidize ferrous iron and some reduced sulphur species. *T. thiooxidans* are physically similar to *T. ferrooxidans* and they are often found together in nature. However, they can survive in lower pH environments than *T. ferrooxidans* and they are better at oxidizing elemental sulphur. These bacteria have an optimum pH of 2.0 to 3.5 and grow in the range of pH 0.5 to pH 6.0 (Rossi, 1990, p.45). *Leptospirillum ferrooxidans* cannot oxidize elemental sulphur and they have a higher tolerance for the ferric ion than *T. ferrooxidans*. *L. ferrooxidans* are active in the pH range pH 1.5 to pH 5, with optimal growth at pH 3 (Pivovarova and Golovacheva, 1985). Another mesophilic bacteria that may be involved in biological leaching is the iron oxidizing strain of the filamentous bacteria of the genus *Metallogenium*. These bacteria exist in the pH range of pH 3.5 to pH 6.8 in the natural environment (Walsh and Mitchell, 1972).

The bacteria described above are known as mesophiles because they are active at temperatures below 40°C. Thermophilic bacteria, which are active in a higher temperature range, may also be present

in some biological leaching processes such as in heap or dump leaching. For example, *Sulfolobus* bacteria oxidize elemental sulphur and ferrous iron, and are active in the temperature range 55°C to 80°C. However, *Sulfolobus* bacteria have been found to be susceptible to shear effects during mixing and to date they have only tolerated pulp densities up to 5% (Torres et al., 1995). Moderately thermophilic bacteria also exist, which can oxidize reduced iron and sulphur compounds. These bacteria exhibit optimal growth in the temperature range from about 40°C to about 55°C. Many moderate thermophiles have not been classified but they are generally thought to belong to the genus *Sulfobacillus*. A moderately thermophilic bacterial culture is being used at the Youanmi mine in Australia for the biooxidation of a refractory gold concentrate in a continuous stirred tank reactor system (Rhodes et al., 1995).

2.1.2   Mixed Bacterial Cultures

Mixed cultures of bacteria can achieve higher rates of biological leaching than pure cultures of *T. ferrooxidans* (Norris and Kelly, 1978). One reason for this is thought to be the presence of heterotrophic bacteria in mixed cultures, which metabolize organic compounds that would otherwise inhibit the growth of the lithoautotrophic bacteria (Groudev, 1985; Harrison et al., 1980). Examples of heterotrophic bacteria are the *Acidiphilium* species and *Thiobacillus acidophilus*. Heffelfinger (1990) studied the adherence of *Acidiphilium cryptum* to chalcopyrite and found that there was no competition between these bacteria and *T. ferrooxidans* for sites on the mineral surface. Another reason for the improved leaching by mixed bacterial cultures may be a symbiotic relationship that exists between different lithoautotrophic bacteria. For example, it was found that none of *Leptospirillum ferrooxidans*, *T. organoparus*, *T. thiooxidans* or *T. acidophilus* could leach pyrite on their own. However, mixtures of *L. ferrooxidans* with one of the other three organisms leached pyrite at the same or higher rates than a pure culture of *T. ferrooxidans* (Norris and Kelly, 1978).

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2.1.3 Temperature

Malouf and Prater (1961) found that temperature had a significant effect on bacterial oxidation of ferrous ion by *T. ferrooxidans*, with maximum activity occurring at 35°C. They also reported that the bacteria were killed at temperatures over 50°C. Sakaguchi et al. (1976a, 1976b) found that the optimum temperature for bioleaching of chalcocite (Cu_2S) and covellite (CuS) was 35°C.

2.1.4 Nutrients

Nutrients for bioleaching are provided in the laboratory in defined media such as the 9K solution, which contains the nutrients ammonium, phosphate, magnesium, potassium, calcium and ferrous iron (Silverman and Lundgren, 1959). Such nutrients are not typically added to industrial scale bioleaching processes because they are available from gangue minerals. If necessary, ammonium and phosphate can be added in the form of fertilizers. In addition to these nutrients, oxygen and carbon dioxide must be supplied for bacterial growth.

The oxygen supply should be higher than that required for growth of *T. ferrooxidans* alone, due to the requirements from other organisms that do not contribute directly to the bioleaching process. Dissolved oxygen was found to be rate limiting below 0.3 mg/L during bioleaching, and below about 0.2 mg/L dissolved oxygen the bacteria began to die (Liu et al., 1988). However, during the bioleaching of a gold concentrate, Pinches et al. (1988) found that dissolved oxygen was not rate limiting at levels as low as 0.1 to 0.5 mg/L. In order to ensure that dissolved oxygen is not rate limiting, Gormely and Branion (1989) recommended maintaining dissolved oxygen at 1 to 2 mg/L.

In 1971, a stirred tank reactor bioleaching system was patented in which the optimal enrichment of CO_2 in air was reported to be 1% (Duncan and McGoran, 1971). Another study showed that 0.2%

CO_2 in air was adequate but 1% CO_2 in air gave enough excess that dissolved CO_2 was not rate limiting (Torma et al., 1972). Nagpal studied the effect of CO_2 enrichment of the air supply to a continuous bioreactor leaching a refractory pyrite arsenopyrite gold concentrate. He found that the optimal cell growth rate was achieved with 0.5% to 1% CO_2 in air and that cell growth rate decreased sharply with no CO_2 enrichment (Nagpal, 1993).

2.1.5   Inhibitory Substances

A substance is inhibitory to bacteria if its presence causes the lag time to increase and the bacterial growth rate to decrease. As will be discussed in Section 2.2.6, bacteria can be adapted to inhibitory substances such as copper, organic chemicals and chloride.

Malouf and Prater (1961) found that although *T. ferrooxidans* could be adapted gradually to increasing concentrations of metal ions, they were sensitive to sudden increases in metal ion concentration. Various researchers have shown that it is possible to adapt bacteria to copper concentrations of up to 55 g/L in biological leaching systems (Bruynesteyn and Duncan, 1971; Sakaguchi et al., 1976b; McElroy and Bruynesteyn, 1978; Lawrence et al., 1984; Groudev, 1985). Dave et al. (1979) were successful in adapting *T. ferrooxidans* to 64 g/L copper.

Some organic chemicals can be inhibitory or toxic to lithotrophic bacteria such as *T. ferrooxidans*. Potential sources of organic chemicals in the bioleaching process are flotation chemicals, solvent extraction reagents and waste products of bacterial metabolic reactions. Certain flotation reagents were of particular interest in this study, including Dowfroth 250 and potassium amyl xanthate. Work by Tuovinen (1978) on several flotation reagents suggested that the inhibition effect was actually caused by the products of decomposition of the flotation reagents in the low pH environment. In

particular he found that Dowfroth 250 had a low degree of inhibition, and potassium amyl xanthate had a relatively high inhibitory effect on ferrous ion oxidation (Tuovinen, 1978). The toxicity of weak organic acids to *T. ferrooxidans* may be due to their diffusion into and accumulation in the cell, which is caused by the pH gradient between the cytoplasm and the cell's environment (Ingledew, 1982). Alexander et al. (1987) found that low molecular weight organic acids accumulated in the cytoplasm of *T. ferrooxidans* and attributed the inhibitory effect to either the high concentration in the cell cytoplasm or their acidification of the cell cytoplasm. Gentina et al. (1987) studied the effect of various flotation reagents and solvent extractants on the biological leaching of copper sulphide ore using a pure strain of *T. ferrooxidans*. They did not find any inhibitory effect from the solvent extractants. However, they found that the flotation reagents, which included Dowfroth 250, significantly inhibited bacterial growth. Microbial growth rate was more inhibited than the rate of oxidation of ferrous to ferric ion, suggesting that in the presence of flotation reagents the bacteria required more energy for maintenance of cellular functions and had no extra energy for synthesis of new cells. Hackl et al. (1989) tested several flotation reagents and concluded that since many flotation reagents decompose with time in acid environments, the best way to determine effects of flotation concentrates on a commercial scale is to carry out continuous tests using fresh concentrate.

In general, *Thiobacilli* bacteria have been found to be sensitive to anions such as the chloride ion. Cameron et al. (1984) studied the effects of salinity on biological iron oxidation and found that the bacteria could be partially adapted to 9 g/L chloride but they could not be adapted to 18.2 g/L chloride (3% w/v NaCl), which is the typical chloride concentration in sea water. They were unable to isolate iron-oxidizing bacteria from sea water and could not find reports of a marine bacterial iron oxidizer in the literature. The inhibitory effects of inorganic anions such as chloride are dependent on the pH of the medium and are likely due to acidification of the bacterial cell cytoplasm. For

example, the chloride concentration required for inhibition of iron oxidation was 15 times lower at pH 0.94 than at pH 3.0 (Alexander et al., 1987). Leong et al. (1993) were able to adapt a mixed culture of *Thiobacilli* growing on a chalcocite ore to 5 g/L chloride in shake flask and column bioleaching tests.

2.2   Kinetics of Biological Leaching

The bacteria involved in biological leaching grow by using the electrons released during the oxidation of reduced iron and sulphur compounds to drive cellular reactions. In particular the electrons are used at the inner cell wall surface to reduce oxygen for the production of water. This allows the cell to maintain a neutral internal pH despite its acidic environment. Bacterial growth is typically first order according to Equation 1, where N_t is the number of bacteria at a particular time, N_o is the number of bacteria at time zero, u is the growth rate (day^{-1}) and t is time (days):

$$N_t = N_o \, e^{ut} \tag{1}$$

Torma and Apel (1992) studied the kinetics of the biological leaching of a chalcopyrite concentrate and found that the reaction was first order. Figure 2.1 gives a typical bacterial growth curve showing the various phases of bacterial growth in batch mode. Continuous biological reactors are generally operated in the exponential phase of bacterial growth.

The growth rate of a bacterial culture is dependent on environmental conditions, such as pH, temperature, nutrient concentrations and the presence of inhibitory substances. These factors were discussed in Section 2.1. Other factors can also affect the bacterial growth rate in a biological leaching process, including the following:

- nature of the substrate (eg. mineralogy, metal content),

- mineral surface area (particle size and pulp density),

- initial soluble iron concentration,

- stirring and aeration,

- adaptation.
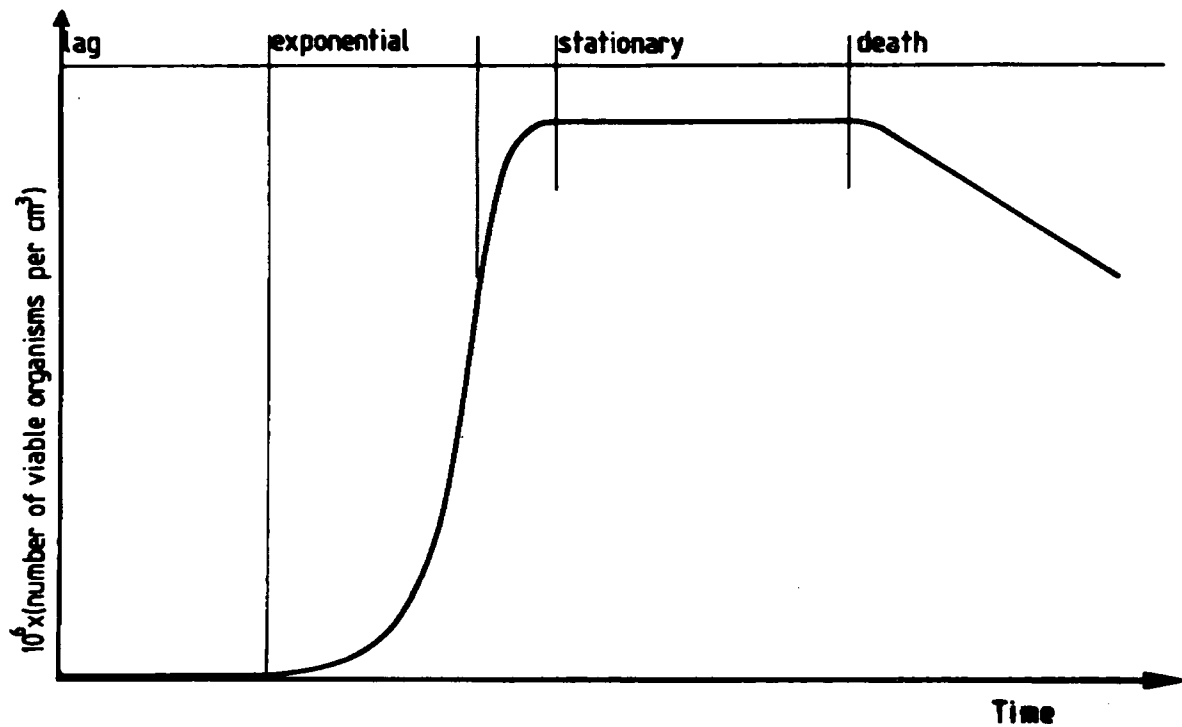
Figure 2.1 -   A Typical Bacterial Growth Curve (Rossi, 1990, Fig. 2-12)

2.2.1   Effect of Mineralogy

Secondary copper sulphide minerals such as chalcocite and covellite are more amenable to biological

leaching than primary minerals such as chalcopyrite. In a continuous biological leaching study,

copper extractions of over 90% were achieved with copper concentrates containing mainly secondary

sulphide minerals, while copper extractions of about 50% were achieved with those consisting of mainly chalcopyrite (Groudev, 1985). Low copper extractions achieved during the bioleaching of chalcopyrite are often attributed to the precipitation of jarosite on the surface of the mineral, which prevents further leaching (Groudev, 1985). A process for the silver catalysed bioleaching of chalcopyrite, which improves copper extraction and eliminates costly regrind steps, has been developed (Bruynesteyn et al., 1986; Lawrence et al., 1984; Short and Parkinson, 1983).

2.2.2   Effect of Particle Size

Both the rate and extent of metal extraction have been reported to be functions of mineral particle size (Bruynesteyn and Duncan, 1971). Particle size is particularly important during the bioleaching of chalcopyrite. As discussed in section 2.2.1, it is often necessary to regrind chalcopyrite in order to achieve reasonable metal extraction. It has been shown that decreasing the particle size increases the rate of leaching (Torma et al., 1972; Sanmugasunderam, 1981; Blancarte-Zurita, 1988).

Mineral surface area, which is a function of both mineral particle size and pulp density, is also an important parameter in biological leaching. Pinches et al. (1976) found that during the batch tank bioleaching of a chalcopyrite concentrate, the rate of bioleaching was dependent on the available mineral surface area. They suggested that to increase the leach rate one could either decrease the mineral particle size or increase the pulp density. The importance of mineral surface area was also stressed by Gormely et al. (1975).

2.2.3   Effect of Pulp Density

Pulp density is the mass of solids per unit volume of slurry. There is an optimal pulp density in biological leaching systems, which allows the highest possible throughput without a reduction in

leach rate. The leach rate has been found to decrease at high pulp densities due to limited supply of carbon dioxide to the bacteria (Pinches et al., 1976). A build-up of inhibitory factors may also limit bacterial growth at high pulp densities. Since the optimal pulp density depends on the characteristics of the particular feed material, it is determined experimentally. The optimal pulp density determined for the biological leaching of various copper sulphide concentrates and the corresponding copper leach rates are given in Table 2.1.

Table 2.1 - Optimal Pulp Density for the Bioleaching of Copper Sulphide Concentrates

Mineralogy	Type of Experiment	Optimal Pulp Density (%)	Copper Leach Rate (mg/L/h)	Reference
$CuFeS_2$	shake flask	22	215	Sakaguchi et al., 1976b
$CuFeS_2$	shake flask	4 to 5	5 to 17	Babij and Madjwick, 1983
$CuFeS_2$	shake flask	5	10 to 36	Briceño and Rossi, 1987
$Cu_2S + CuFeS_2$	shake flask	5	42	Briceño and Rossi, 1987
$CuFeS_2$	batch reactor (3.5L)	5	25	Pinches et al., 1976
$CuFeS_2$	continuous reactor (1-stage, 30L)	40	> 400	McElroy and Bruynesteyn, 1978
$CuFeS_2$	continuous reactor (5-stage, 5 L each)	25	250 to 410	Groudev, 1985
$Cu_2S + CuFeS_2$	continuous reactor (5-stage, 5 L each)	20	430 to 470	Groudev, 1985

2.2.4 Effect of Initial Soluble Iron Concentration

Since ferrous iron is a source of energy for *T. ferrooxidans*, soluble iron concentration is an important parameter in bioleaching systems. Copper bioleach rates from museum grade covellite in shake flask experiments at 6% pulp density, were found to increase in the presence of 9 g/L soluble iron relative to copper bioleach rates achieved with no soluble iron present (Groudev, 1980). The increases in copper bioleach rates ranged from 5% to 15% for various strains of *T. ferrooxidans*. Briceño and Rossi (1987) also studied the effect of varying the initial soluble iron concentration in shake flask bioleaching experiments. They found no significant effect on the lag period or the rates of copper leaching when the initial ferrous iron concentration varied from zero to 2.8 g/L.

2.2.5 Effect of Stirring and Aeration

Higher rates of biological leaching are achieved by stirred batch reactor bioleaching than by shake flask bioleaching (Groudev, 1985). Improved stirring and aeration in batch reactors results in higher rates of oxygen transfer to the mineral surface, higher carbon dioxide transfer to the bacteria, and better transport of other nutrients to the bacteria.

Increasing the rate of stirring increases particle suspension and aeration, but it also increases the shear forces acting on the cells. Bacteria are typically not considered sensitive to shear forces in stirred reactors due to their size, which is less than the microscale of turbulence in a typical bioreactor, and the strength of their cell walls (Thomas, 1993). There are two main types of stirrer impellers: radial flow and axial flow. Radial flow impellers, such as the Rushton turbine, have high shear rates and are better at air dispersion than axial flow impellers. Axial flow impellers, such as the pitched blade type, have lower shear rates and are better at particle suspension. Hackl et al. (1989) tested the shear effect of both radial and axial impellers in a bench scale continuous

bioleaching experiment and found that bacteria were sensitive to shear effects produced by radial flow impellers, but there was no observable effect when axial flow impellers were used.

Impeller tip speed (v_t) is often used as a measure of shear rates in stirred tanks because it has been found not to vary much during scale-up of bioreactors. It is given in Equation 2 in units of m/s where N is the stirrer speed in RPM and D_i is the impeller diameter in metres:

$$v_t = \pi N D_i \left(\frac{1}{60}\right) \qquad (2)$$

A typical value for v_t in industrial fermenters is 5.5 m/s (Einsele, 1978). It is important to note, however, that Equation 2 does not consider the type of impeller.

2.2.6 Adaptation

The bacterial cell can be thought of as a "super chemical reactor" because, unlike regular chemical reactors, it can improve its efficiency by genetically adapting to its environment. Bacteria have part of their genetic material on structures called plasmids, which can be transferred quickly and easily between cells. Plasmids typically contain genes for auxiliary cellular functions such as optimum temperature for growth or resistance to metals (Martin et al., 1981). Therefore, bacteria can be adapted to increased concentrations of inhibitory substances such as metal ions, organic chemicals or chloride. This adaptation process can occur naturally or it can be accomplished in the laboratory. For example, cells growing in the laboratory in media containing a certain concentration of the inhibitory substance can be transferred to new media containing a slightly higher concentration of the substance. The bacteria are then cultured at the new concentration until they are adapted. The process can then be repeated until the desired concentration is achieved. The lag phase decreases

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and the growth rate increases as bacteria become adapted to a substrate (Elzeky and Attia, 1989).

Typical bacterial growth curves resulting from an adaptation experiment are shown in Figure 2.2.

Different strains of *T. ferrooxidans* from various environmental conditions have been found to

contain different genes on their plasmids, and have varying physical and chemical characteristics

such as optimum temperature for growth and resistance to heavy metals (Leduc and Ferroni, 1993).

Therefore, the most efficient bacterial culture for bioleaching a particular ore is most likely to be

one that has already been adapted to the environmental conditions at the mine.
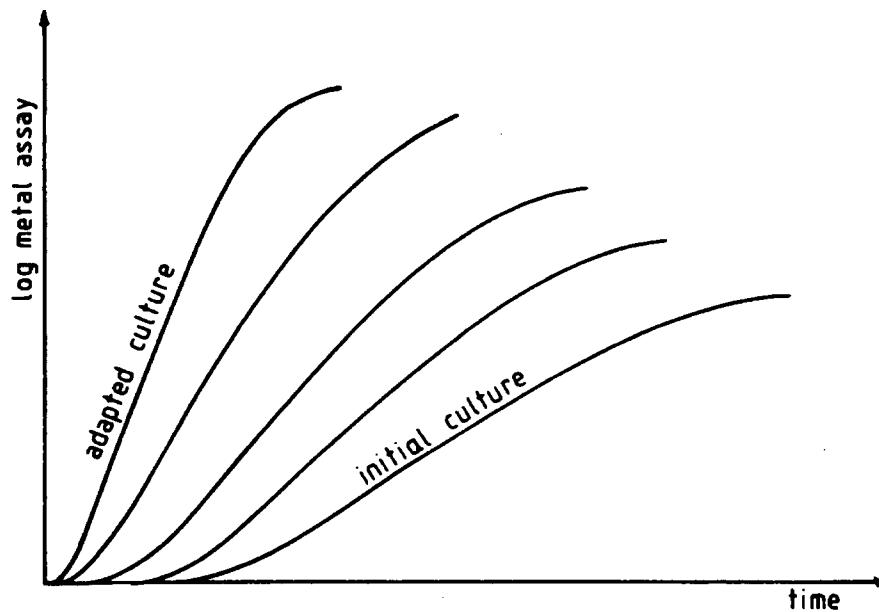
Figure 2.2 -   Typical Bacterial Growth Curves Resulting from Adaptation of the Bacterial
Culture (Rossi, 1990, Fig. 4-2)

Genetic engineering techniques may also be applied to develop bacteria that are adapted to the

desired conditions. However, a genetically engineered strain of *T. ferrooxidans* has not yet been

developed. *T. ferrooxidans* genes have been transferred to *E. coli* bacteria, but a means of returning

modified genes to *T. ferrooxidans* has not been determined (Ehrlich and Brierley, 1990, p.29).

2.3   Biological Leaching of Chalcocite

Sizable ore bodies of chalcocite have been reported to occur in Chile, Mexico, Namibia, Peru, Russia and the USA (Mottana et al., 1978). Over the past thirty years, various researchers have studied the biological leaching of chalcocite. Duncan and Trussell reported that over 90% of the copper in museum-grade chalcocite could be extracted by biological leaching for 30 days (Duncan and Trussell, 1964). Biological leaching of synthetic chalcocite using *T. ferrooxidans* was studied by Sakaguchi et al. (1976a), who found that the extent of leaching of chalcocite was 90% to 100%. The feasibility of in-situ leaching of a chalcocitic ore was studied (Johnson et al., 1988). Column biological leaching of chalcocite bearing ores has been reported (Leong et al., 1993), and a mineralogical study of the bioleach residues was carried out (Melluish et al., 1993).

The oxidation of copper sulphide minerals at ambient temperature and pressure occurs chemically but the kinetics are slow. Bacteria are known to catalyse these reactions by three mechanisms: direct, indirect and galvanic oxidation. Chalcocite oxidation, whether chemical or biological, occurs in two major steps (Nielsen and Beck, 1972). The first step is the conversion of chalcocite (Cu_2S) to covellite (CuS), and the second step, which is kinetically slower, is the oxidation of covellite (Beck, 1977). It is unclear what exactly is oxidized during the conversion of chalcocite to covellite. In studying chalcocite biooxidation, Nielsen and Beck (1972) concluded that the apparent source of energy for carbon dioxide fixation was provided by the oxidation of cuprous to cupric ions. However, there is no direct evidence that *T. ferrooxidans* oxidize cuprous copper. Most sulphide minerals are semiconductors with covalent bonds, in which electrons are shared between atoms. Therefore, it is possible that the chalcocite structure as a whole is oxidized to covellite (Beck, 1977).

2.3.1 Direct Mechanism

Direct oxidation occurs when the bacteria attach themselves to the mineral surface and oxidize the mineral using oxygen as the oxidant. Biological oxidation of chalcocite by the direct mechanism is shown by the following two reactions:

$$Cu_2S + 0.5O_2 + H_2SO_4 \rightarrow CuSO_4 + CuS + H_2O \qquad (3)$$

$$CuS + 2O_2 \rightarrow CuSO_4 \qquad (4)$$
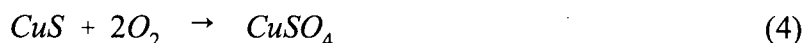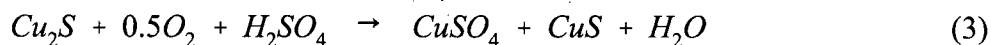
These reactions occur spontaneously, but the kinetics are slow at ambient temperature. It was found that bacterial action increases the rate of chalcocite oxidation, as shown in Equation 3, by about forty times (Beck, 1977). Chalcocite oxidation is kinetically faster than covellite oxidation (Beck, 1977).

Iron sulphide minerals such as pyrite (FeS_2) are often found with copper sulphide minerals. Biological oxidation of pyrite (FeS_2) by the direct mechanism is shown in Equation 5:

$$2FeS_2 + 7.5O_2 + H_2O \rightarrow Fe_2(SO_4)_3 + H_2SO_4 \qquad (5)$$
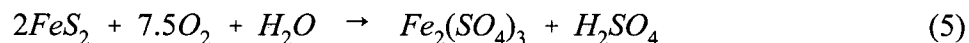
This reaction is important to the biological oxidation of copper sulphide minerals for two reasons. Firstly, since pyrite is a major component of many sulphide ore bodies, including copper sulphide ore bodies, pyrite oxidation is the prime sulphuric acid generator. Secondly, pyrite oxidation is a possible source of soluble iron, which is involved in the indirect mechanism as will be described in Section 2.3.2. It is important to note, however, that copper sulphide minerals leach preferentially to pyrite because the latter is more noble (Dutrizac and MacDonald, 1973).

2.3.2 Indirect Mechanism

In the indirect mechanism, the bacteria do not attach to the mineral surface but are suspended in the leach solution, where they catalyse the oxidation of ferrous to ferric ion. This results in the regeneration of the ferric ion, which acts as the oxidant for the leaching of chalcocite:

$$2FeSO_4 + 0.5O_2 + H_2SO_4 \rightarrow Fe_2(SO_4)_3 + H_2O \qquad (6)$$
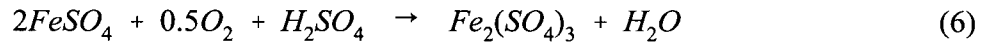
Iron is usually present in solution from the bacterially catalysed oxidation of pyrite, as shown in Equation 5 above. Chalcocite oxidation by the indirect mechanism occurs in two major steps, just as it does by the direct mechanism. The reactions for the oxidation of chalcocite to covellite by the indirect mechanism, and the oxidation of covellite by the indirect mechanism are:

$$Cu_2S + Fe_2(SO_4)_3 \rightarrow CuS + CuSO_4 + 2FeSO_4 \qquad (7)$$

$$CuS + Fe_2(SO_4)_3 \rightarrow CuSO_4 + S^o + 2FeSO_4 \qquad (8)$$
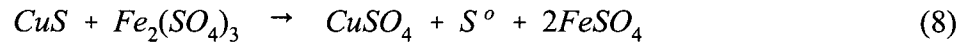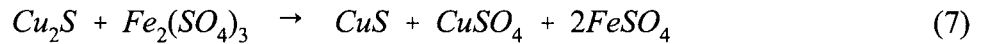
The oxidation of chalcocite is kinetically faster than the oxidation of covellite (Beck, 1977).

As shown in Equation 8, covellite is oxidized to copper sulphate and elemental sulphur by the indirect mechanism. Bacteria then catalyse the oxidation of elemental sulphur to sulphate by the direct mechanism:

$$S^o + 1.5O_2 + H_2O \rightarrow H_2SO_4 \qquad (9)$$
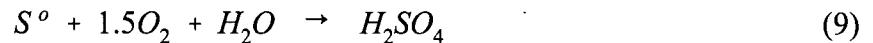
Elemental sulphur is thermodynamically unstable under the conditions of biological leaching.

However, at ambient temperature and pressure, elemental sulphur occurs as S_8, which is a very stable eight-membered sulphur atom ring (Vaughan and Craig, 1978). Therefore, the kinetics of elemental sulphur oxidation are slow at ambient temperature and pressure.

2.3.3   Galvanic Mechanism

Galvanic oxidation occurs when two sulphide minerals with different rest potentials are in contact in solution and an electrical current results between the two minerals. The less noble sulphide acts as an anode and is oxidized. The more noble sulphide acts as a cathode and receives the electrons, which are used for reduction of oxygen gas to water on the mineral surface. The galvanic mechanism results in preferential leaching of the less noble sulphide mineral because the rate of leaching of the less noble mineral is accelerated, while the rate of leaching of the more noble mineral is retarded. Galvanic oxidation occurs chemically but it is enhanced by the presence of bacteria. Bacteria act as catalysts in galvanic oxidation by oxidizing the elemental sulphur that is produced in the galvanic oxidation process. Galvanic oxidation can occur during biological leaching in a stirred tank reactor because contacts between sulphide minerals are made and broken continuously. It is expected that increasing the pulp density in a stirred tank reactor would increase the occurrence of galvanic oxidation because of increased mineral contact.

For example, the leaching rate of chalcopyrite in acid ferric sulphate solutions is accelerated in the presence of a more noble sulphide such as pyrite (Karavaiko et al., 1988; Dutrizac and MacDonald, 1973). Dave et al. (1979) found that sphalerite (ZnS) leached preferentially to covellite (CuS), and in fact there was no copper dissolution in the presence of ZnS. During the continuous bioleaching of a chalcopyrite and pyrite bearing concentrate, pyrite oxidation did not occur despite 80 to 90% chalcopyrite oxidation (Lawrence et al., 1984). Almendras et al. (1987) studied the biological

leaching of a copper sulphide concentrate consisting of mainly chalcopyrite and pyrite with some chalcocite and covellite also present. They found that chalcocite leached first, followed by covellite, chalcopyrite and then pyrite. During the batch reactor bioleaching of a gold-bearing arsenopyrite-pyrite concentrate, it was observed that arsenopyrite began leaching before pyrite and was almost fully leached before pyrite leaching commenced (Miller and Hansford, 1992).

Rest potential is a basic property of semiconducting minerals such as sulphides. It is the observed potential on an open circuit and it is a measure of the nobility of the mineral. Relative rest potentials can be used to predict the leaching behaviour of a mixture of sulphide minerals (Sato, 1966; Jyothi et al., 1988). The rest potentials of several sulphide minerals are given in Table 2.2.

Table 2.2 - Rest Potentials for Several Sulphide Minerals
(Rossi, 1990, Table 2-38)

Mineral	Rest Potential (mV, SHE)
pyrite (FeS_2)	540
chalcopyrite ($CuFeS_2$)	434
covellite (CuS)	434
chalcocite (Cu_2S)	416
sphalerite (ZnS)	181

2.3.4   Effect of Soluble Iron

The concentration of iron in solution is very important in biological leaching because of the role of the ferric ion in the indirect mechanism. At elevated pH or when the concentration of ferric iron in solution is above a certain concentration, precipitation of jarosite or ferric hydroxide can occur. The

precipitation of hydronium jarosite occurs as follows:

$$3Fe_2(SO_4)_3 + 14H_2O \rightleftharpoons 2H_3OFe_3(SO_4)_2(OH)_6 + 5H_2SO_4 \qquad (10)$$

In addition, sodium, potassium and ammonium jarosites can precipitate out of solution. Ferric hydroxide precipitates can also form, as shown in Equation 11:

$$Fe_2(SO_4)_3 + 6H_2O \rightleftharpoons 2Fe(OH)_3 + 3H_2SO_4 \qquad (11)$$
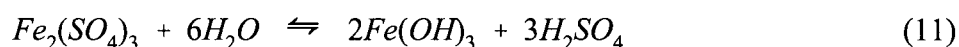
Sakaguchi et al. (1976a) found that jarosite precipitation occurred above soluble iron concentrations of 0.22 g/L to 0.56 g/L for chalcocite bioleaching at pH 1.7, and above 0.22 g/L to 1.12 g/L for covellite bioleaching at pH 2.3. During the biological oxidation of ferrous sulphate, it was found that ferric hydroxides precipitated out first, followed by jarosites, but by the end of the process the precipitates were found to be almost all jarosites (Toro et al., 1988). A study on the characterization of leach residues from column leaching of a chalcocitic ore found that dominant precipitates were jarosites and basic iron oxides (Melluish et al., 1993). Wiertz et al. (1994) found that ferric ion precipitation was enhanced by the presence of bacteria and also by the presence of solids. They found that jarosite was formed preferentially over ferric hydroxide in the presence of solids. Ferric iron may have a buffering effect in bioleach systems by maintaining the pH at the low values required for the growth of *T. ferrooxidans* (Duncan and Trussell, 1964). Wiertz et al. (1994) found that ferric ion in the leaching solution helped maintain solution pH at low values. At elevated pH

values, the equilibrium of Equations 10 and 11 would shift to the right, generating acid and causing the pH to decrease. The solubility of the ferric ion is extremely low at pH values greater than 2.5. Jarosites and ferric phosphate precipitates were found to occur at pH values as low as 1.35 to 1.5 in a packed-bed bioreactor meant for iron oxidation (Grishin et al., 1988).

The disadvantages of forming jarosites are formation of insoluble coatings on mineral surfaces that reduce leaching rates, plugging of heap and dump leach beds, and removal of nutrients such as sodium, potassium and ammonium from solution. Leach residues containing jarosites are a disposal problem because they are bulkier than the slag produced by smelters, and they are less stable because they can drain acid solutions containing heavy metals (Peters, 1992).

Since _T. ferrooxidans_ oxidize ferrous iron to ferric iron, at high levels of biological activity most of the soluble iron has been oxidized to the ferric state. Thus a high potential indicates a high degree of bacterial activity. Therefore, redox potential (Eh) is often used as an indirect measure of the level of bacterial activity in biological leaching processes. The redox potential of an active bacterial culture is about 700 mV (SHE) to 900 mV (SHE), which is the potential range of the ferric/ferrous couple depending on the ferric to ferrous ratio. The Nernst equation giving the redox potential for the ferric/ferrous couple at 25°C is shown below:

$$Eh = 0.771 + 0.0591 \log \frac{a_{Fe^{+3}}}{a_{Fe^{+2}}} \tag{12}$$

2.3.5   pH

The pH will increase initially during the biological oxidation of a chalcocitic ore or concentrate

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(Nielsen and Beck, 1972; Silver and Torma, 1974; Beck, 1977; Groudev, 1985). This is due to the conversion of chalcocite to covellite by the direct mechanism and the oxidation of ferrous sulphate, which are both acid consuming reactions. Later, the pH will decrease due to the oxidation of pyrite, the oxidation of elemental sulphur and the formation of ferric ion precipitates, which are all acid generating reactions (Groudev, 1985).

2.3.6   Thermodynamics

The Eh-pH diagram for the copper-sulphur-water system is given in Figure 2.3. This diagram was generated by Outokumpu's HSC Chemistry computer program. It shows that chalcocite (Cu_2S) and covellite (CuS) are thermodynamically stable in the pH range 1 to 2 when the Eh is less than about 400 mV (Standard Hydrogen Electrode reference, SHE). However, they are thermodynamically unstable in most active bioleaching solutions, which have a pH of about 1 to 2 and an Eh of about 700 to 800 mV (SHE). The Eh-pH diagram for the copper-sulphur-iron-water system is given in Figure 2.4. The dashed box in this diagram gives the stability and activity limits for *T. ferrooxidans*. The filled and open circles in Figure 2.4 indicate measured Eh values during chalcopyrite leaching in the presence and absence of bacteria respectively (Ehrlich and Brierley, 1990, p.84). The bacteria grow in the region where the products of sulphide oxidation are thermodynamically stable and sulphide minerals are thermodynamically unstable.

2.4   Direct versus Indirect Biological Leaching Mechanisms

The relative importance of the direct versus the indirect mechanism of biological leaching is not clear. Support for the direct mechanism includes the fact that *T. ferrooxidans* can oxidize chalcocite even in the absence of iron (Nielsen and Beck, 1972; Imai, 1978; Dave et al., 1979). The leaching of a chalcopyrite concentrate was found to be significantly enhanced in the presence of bacteria as

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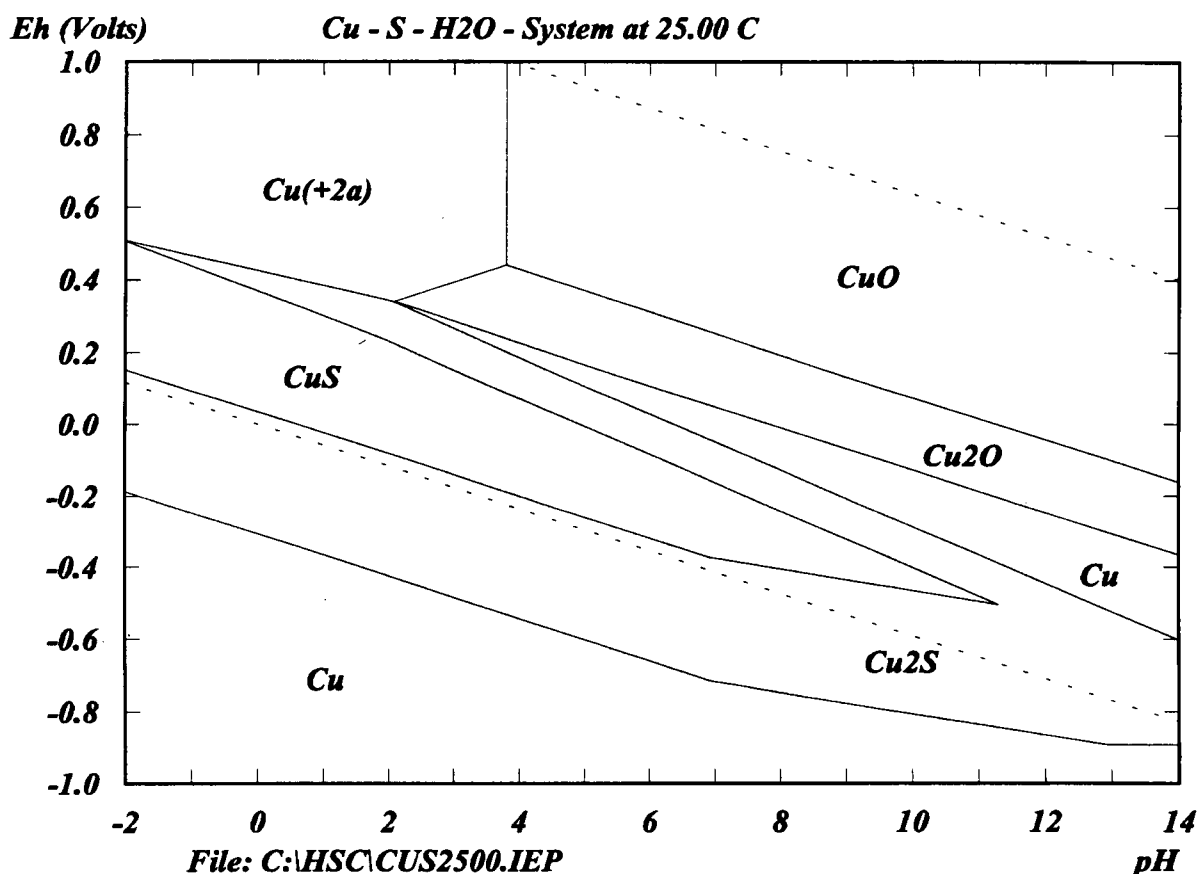
Figure 2.3 -   The Eh-pH Diagram for the Cu-S-H$_2$O System at 25°C

compared to ferric sulphate leaching, suggesting that the direct mechanism plays an important role in biological leaching (Almendras et al., 1987). Vargas et al. (1990) evaluated the numbers of attached and free *T. ferrooxidans* during the biological leaching of pure chalcopyrite and found that a significant fraction of the total bacteria in the system were attached to the mineral surface. However, Espejo and Ruiz (1987) studied the ratio of attached bacteria to bacteria in solution and found that less than 10% of the total bacterial activity, as measured by ferric iron generation, was associated with attached bacteria. Therefore, although it is clear that both direct and indirect mechanisms play a role in biological leaching, the relative importance of each is not known.

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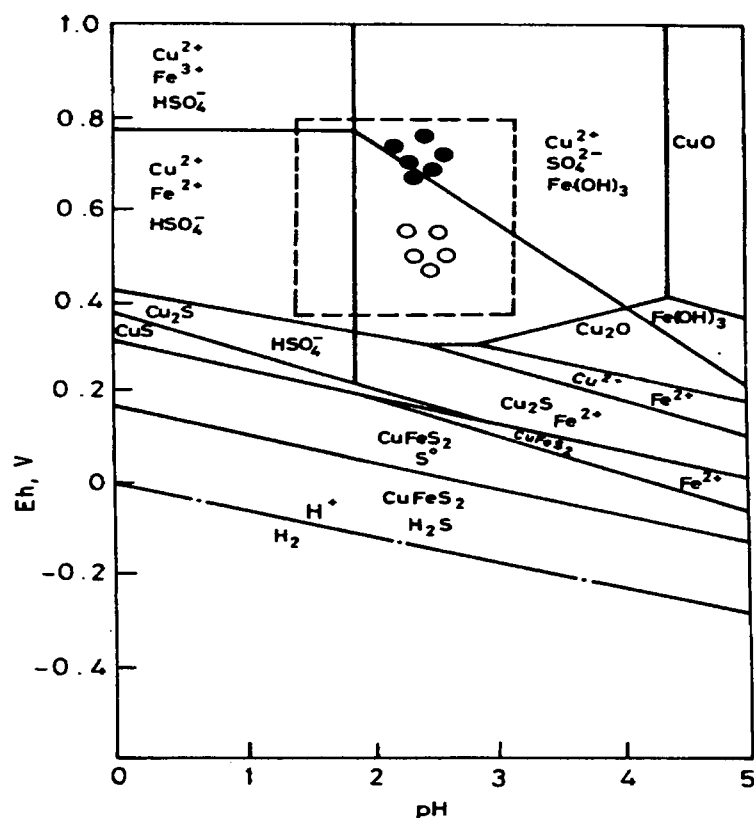
Figure 2.4 -   The Eh-pH Diagram for the Cu-S-Fe-H_2O System at 25°C
(Ehrlich and Brierley, 1990, Figure 4.2)

2.5   Bioleaching of Copper Concentrates

Much of the interest in copper sulphide biological leaching has been focussed on copper extraction

from low-grade ores (Torma and Bosecker, 1982; Murr, 1980). However, there is also much interest

in biological leaching of copper concentrates. The feasibility of biological leaching chalcopyrite

concentrates has been demonstrated (McElroy and Bruynesteyn, 1978; Torma et al., 1979; Babij and

Madgwick, 1983; Groudev, 1985; Agate and Khinvasara, 1986; Torma and Apel, 1992; Corral et

al., 1993). Bioleaching of chalcocite bearing copper concentrates has also been demonstrated

(Groudev, 1985). Biological leaching of copper concentrates can be more complex than biological

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leaching of copper ores. With concentrates, copper contents are typically higher and the presence of flotation chemicals can inhibit bacterial growth. The copper content of concentrates varies from about 15 - 40 wt% total copper and bacteria must be acclimatized to the high copper levels. Since each concentrate has a unique mineral composition, it is necessary to test individual concentrates to determine their amenability to biological leaching.

2.6   Continuous Biological Leaching

Continuous biological leaching is typically carried out in a series of stirred tanks operated at steady state. The bacterial cultures are maintained in the exponential growth phase in order to achieve the highest possible leaching rates. Therefore, the lag time associated with batch bioleaching is eliminated. Continuous bioleaching rates are typically higher than batch leaching rates achieved with the same feed materials (Groudev, 1985; Corral et al., 1993). An important design consideration is bacterial washout, which can occur if the system residence time is less than the doubling time of the bacteria. To reduce the chance of washout, the residence time of the first stage should be greater than the residence time of the subsequent tanks (Gormely and Branion, 1989). This can be accomplished by increasing the size of the first tank or by dividing the first stage into two or more equal sized tanks operating in parallel.

In designing a continuous stirred tank, it is generally assumed that the contents are perfectly mixed so that the outlet flow has the same composition as the slurry inside the tank. However, in practice, these tanks are not perfectly mixed and non-ideal flow conditions occur, such as short-circuiting and stagnant zones. Short-circuiting results in low extractions because a significant proportion of the product has been in the reactor for less than the average residence time. To increase overall extraction, the system residence time should be divided into a series of stirred tank reactors

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(Levenspiel, 1972). The greatest increase in extraction occurs if a second reactor is added, the second greatest occurs if a third reactor is added and so on. Typically in chemical engineering applications, three or four stirred reactors in series are used. There is a trade-off between maximizing conversion and minimizing capital cost. Mixing in the stirred tanks can also be improved with the use of internal baffles, which prevent swirling, and an entrance weir, which can help reduce short-circuiting effects (Gormely and Branion, 1989).

The development of bench scale systems for the continuous biological leaching of sulphide ores began in the 1960's with the use of batch stirred reactors (Duncan et al., 1967; Duncan and McGoran, 1971). Then in 1971, Bruynesteyn and Duncan proposed a continuous stirred tank leaching process for bioleaching sulphide concentrates based on the results of batch tank leaching experiments. Gormely et al. (1975) demonstrated the feasibility of bioleaching a zinc sulphide concentrate in a bench scale single-stage continuous stirred tank reactor. Several years later, a larger single stage reactor was used for the continuous bioleaching of a chalcopyrite concentrate. However, significant short-circuiting was found to occur and system upsets due to mechanical problems with solids feeding occurred (McElroy and Bruynesteyn, 1978). Sanmugasunderam (1981) operated a two-stage bench scale continuous biological leaching system for leaching zinc sulphide concentrate. The effect of the second stage was to increase the ultimate zinc extraction and the rate of zinc extraction (Sanmugasunderam et al., 1986). A continuous biological leaching system with five stages of 5 L each was used to test six different copper sulphide concentrates, which were washed in acetone to eliminate flotation chemicals (Groudev, 1985). The highest final copper extractions were achieved with the secondary sulphide concentrates. Corral et al. (1993) commissioned a 1.6 L three-stage continuous reactor system for the bioleaching of sulphide minerals. Two chalcopyrite concentrates were tested, and the pulp density and residence time were varied. Data reported for several

continuous bioleaching experiments are given in Table 2.3.

Table 2.3 - Summary of Several Continuous Bioleaching Studies

Reference:	Corral et al., 1993	Groudev, 1985	Groudev, 1985	Groudev, 1985
main copper mineral(s) in feed	$CuFeS_2$	$CuFeS_2$	CuS, Cu_2S, $CuFeS_2$	CuS, Cu_2S, $CuFeS_2$
total copper in feed (%)	26.9	21.2	21.3	35.0
number of stages	3 (1.6 L each)	5 (5 L each)	5 (5 L each)	5 (5 L each)
pulp density (%)	5	20	20	20
residence time (days)	4.0	4.3	2.7	5.7
$[Cu]_{out}$ (g/L)	6.5	26	30	58
copper leach rate (mg/L/h)	not given	250	470	430
copper extraction (%)	43	61	71	83

Although the feasibility of continuous bioleaching of chalcopyrite concentrates was demonstrated, it was found that leach residues had to be reground and releached in order to achieve reasonable copper recoveries (McElroy and Bruynesteyn, 1978; Groudev, 1985; Briceño and Rossi, 1987). A two stage continuous bioleaching system for the silver catalysed bioleaching of chalcopyrite concentrate was run for about 35 days (Lawrence et al., 1984). They found that most of the leaching occurred in the first tank, and they concluded that short-circuiting occurred because copper recoveries were lower than had been achieved in batch reactor tests.

Continuous stirred tank biological leaching for refractory gold ores and concentrates has also been carried out successfully at the bench scale (Nagpal, 1993; Pinches et al., 1988). However,

biooxidation of refractory gold ores and concentrates is further advanced than biological leaching for copper extraction. Many pilot plant projects have been successful (Lawrence, 1993; Adam et al., 1989; Hackl et al., 1989; Moore, 1989) and the process is also successful on a commercial scale.

A mathematical model is an equation or set of equations that describes the reactions occurring in a system. Mathematical models allow the prediction of system behaviour under many more conditions than it would be practical to test in the laboratory. This is particularly important for continuous bioleaching studies, which are time consuming and labour intensive. However, biological leaching systems cannot be described mathematically without some experimental data on leaching kinetics. Several mathematical models of continuous biological leaching processes have been developed (Petrova, 1985; Konishi and Asai, 1993; Nagpal, 1993). These models were all based on some form of Monod growth kinetics. Blancarte-Zurita (1988) used a shrinking particle model to describe biological leaching of sulphide mineral concentrates. A computer program was then written to solve this model for continuous stirred bioleach tanks in series. An empirical equation relating the rate of leaching to the mineral particle size is a required input to the computer program.

2.7   Commercial Processes

Several excellent reviews of the commercial application of biological leaching processes have been published (Hiskey, 1994; Marchbank, 1993; Rossi, 1990; Adam et al., 1989). Commercial biological leaching operations for extraction of copper from sulphide ores are mainly in-situ and dump leaching. Copper extraction by heap leaching was carried out for many years at the Rio Tinto mine in Spain. However, heap leaching has only reached widespread commercial application in the past few years. Continuous stirred tank bioleaching for copper extraction is still at the laboratory stage. Presently, there are no commercial scale biological leaching processes for copper sulphide

concentrates. However, there are several commercial continuous stirred tank biooxidation plants for treatment of refractory gold ores and concentrates (Rhodes et al., 1995; Lawrence, 1993).

Copper extraction by in-situ leaching is carried out at the Old Reliable mine and the Inspiration mine in the United States, the Kosaka mine in Japan and the El Teniente mine in Chile (Rossi, 1990, p.498). In-situ leaching for copper extraction is also carried out at the Blyavinsk mine in the USSR and the Avoca mine in Ireland (Karavaiko et al., 1988, p.285-292). Biological copper dump leaching is carried out at the Kennecott Bingham Canyon (Malouf and Prater, 1961) and Esperanza mines in the United States, a mine at Cananea, Mexico (Rossi, 1990, p.499), the Gibraltar mine in Canada (Bruynesteyn et al., 1987) and the Nikolayev and Kounrad mines in the USSR (Karavaiko et al., 1988, p.277-285). The Phelps-Dodge Morenci mine in Arizona, USA has been operating a copper sulphide biological heap for many years (Marchbank, 1993). Recently, a number of biological heap leaching operations have been developed for extraction of copper from sulphide ores. In Chile, the Lo Aguirre mine began bacterial heap leaching in 1988 and the Cerro Colorado and Quebrada Blanca mines opened in 1994 (Marchbank, 1993). Biological heap leaching of a copper ore will commence at the Zaldívar mine in Chile in the near future.

2.8   Summary

It is clear from the literature review that the amenability of individual ores and concentrates to biological leaching must be determined by experiment. This is particularly true for concentrates because higher metal contents and the presence of flotation chemicals can potentially inhibit bacterial growth. The results of previous studies suggested that it is necessary to carry out initial batch tests in order to ascertain the feasibility of biological leaching and to determine ultimate extraction values. Continuous biological leaching experiments were determined to be necessary to obtain scale-up data

for commercial processes and for determining the adaptability of the bacteria to a particular feed material over time. The literature review also highlighted the importance of using a mixed bacterial culture that is already adapted to environmental conditions at the site under consideration for operation of a biological leaching process. No reports were found in the literature concerning the continuous biological leaching of pure chalcocitic ores or concentrates, although there were some data for mixed chalcopyrite and chalcocite feed concentrates. Also, continuous biological leaching for copper extraction in a saline environment has apparently not yet been studied.

The present study has considered the feasibility of the continuous biological leaching a chalcocitic ore and concentrate in a saline environment. Based on the information given in the literature, it was determined that the study would be carried out in three phases, including two initial phases of batch tests followed by the continuous bioleaching experiment. A mixed bacterial culture obtained from the mine site was used. The continuous biological leaching system was designed based on systems reported in the literature. A three-stage system was selected to achieve a high conversion with a reasonable amount of work involved in system maintenance.

The optimal values of several key variables were obtained from the literature, such as pH, temperature, oxygen, carbon dioxide and nutrient concentrations. It had been reported in the literature that the bacteria could be adapted to growth in the presence of chloride, flotation chemicals and high copper concentrations. Therefore, it was anticipated that it would be possible to biologically leach the Zaldívar ore and concentrate under these conditions, and no attempt was made to wash the concentrate to remove flotation chemicals. The literature review revealed that pulp density and initial soluble iron concentration were important variables in biological leaching. Therefore, they were selected as key variables for this study. Stirring and aeration were determined

to be critical parameters. An axial impeller was selected for this study in order to achieve adequate solids suspension and air dispersion with low shear rates.

There have been several attempts to mathematically model continuous bioleaching processes. Apparently no mathematical models have yet been applied to the continuous bioleaching of copper sulphide ores. Mathematical modelling was not undertaken in this study due to time constraints.

Continuous stirred tank biological leaching has been successful commercially for the biooxidation of refractory gold ores. Copper extraction is another potential application for this process. The success of continuous bioleaching for the treatment of refractory gold ores will likely be extended to copper extraction when an economically viable system is developed.

3.0 EXPERIMENTAL METHODS

3.1 Feed Materials

The two feed materials used for this study were a rhyolite sulphide ore from the Zaldívar mine and a flotation concentrate produced from the ore by Placer Dome Inc. (PDI). The mineralogy included mainly the sulphide minerals chalcocite (Cu_2S) and pyrite (FeS_2), and the copper oxide brochantite ($Cu_4(SO_4)(OH)_6$). Brochantite is soluble in very dilute acids (Mottana et al., 1978). The other main iron containing mineral was hematite (Fe_2O_3). Analyses of the feed materials are given in Table 3.1.

Table 3.1 - Characteristics of the Zaldívar Ore and Concentrate

Feed	Cu_T (%)	$Cu_{Acid Soluble}$ (%)	Fe_T (%)	S_T (%)	S^{2-} (%)	P_{80} (µm)	s.g.
Ore	1.56	0.50	2.66	0.91	0.81	108	2.70
Concentrate	17.30	1.60	11.60	11.70	11.62	53	3.32

The feed materials were supplied by PDI in slurry form at pulp densities of 47% and 33 % for the ore and concentrate respectively. Portions of each slurry were filtered to obtain dry ore and concentrate for use in the shake flask and batch reactor experiments. The filtration residues were dried at ambient temperature, rolled and separated into representative sections. Dried samples of each feed material were ground in a rod mill to obtain various particle size distributions for testing in the shake flask experiments. Fresh ore and concentrate slurries were used as the feed materials for the continuous bioleaching systems.

PDI attempted to produce a flotation concentrate with a total copper content of about 30%, which

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was the expected copper content of the concentrate to be produced at the Zaldívar mine site. Unfortunately, the highest total copper content attainable was 17.3%. The poor copper grade of the concentrate was likely attributable to the Zaldívar ore having been stored outside in drums for over one year. As the drums were not tightly sealed, the ore was exposed to air and water during this time and was likely partially oxidized. The chemicals used in the flotation process are given in Table 3.2 along with their concentrations.

Table 3.2 - Flotation Chemicals Used in the Production of the Zaldívar Concentrate

Product Name	Chemical Name	Concentration (ppm)
Dowfroth (R) 250-D Frother	polypropylene glycol methyl ethers	15-20
sodium sulphide	sodium sulphide	250
T-22	isopropyl ethylthionocarbamate (85%) methyl ethyl-dithiocarbamate (7%) isopropyl methyl-xanthate (2%) dimethyltrithiocarbonate (2%) unknown organic impurities (4%)	30
Aero (R) 350	potassium amyl xanthate	60-75

3.2   Bacterial Culture and Nutrient Media

The bacterial culture used was a mixed culture dominated by *Thiobacillus ferrooxidans*. The culture was originally obtained from the Zaldívar site for a previous study at the University of British Columbia. It had been acclimatized to the Zaldívar ore and a saline environment of 9 g/L chloride before being used in this study. The bacterial nutrient solution used in the shake flask and batch reactor experiments was the 9K solution developed by Silverman and Lundgren (1959) containing 9 g/L ferrous iron. Similar solutions containing lower concentrations of ferrous iron were used in

shake flask and batch reactor experiments to test the effect of the initial soluble iron concentration. The bacterial nutrient solution used in the continuous reactor experiments was the Silverman and Lundgren solution (1959) containing 3 g/L ferrous iron. It was expected that the Zaldívar plant water would have this soluble iron concentration. A 3 g/L chloride solution was used because the Zaldívar plant water was expected to have this chloride concentration. The required chloride concentration was achieved by adding the appropriate amount of a 211 g/L NaCl stock solution.

Stock bacterial cultures were grown in shake flasks on the Zaldívar ore at a pulp density of 10% in a 3 g/L chloride environment. They were maintained by successive transfer of 5 mL of slurry containing cells in the late exponential phase of growth. This is known as the serial transfer method. During the continuous reactor experiment, stock bacterial cultures were maintained on both the ore and concentrate in 2 L batch reactors. When the cultures reached the late exponential phase of growth, 500 mL of slurry was removed from the batch reactor and replaced with fresh slurry. These cultures were available for reinoculation of the continuous bioleaching systems in case washout conditions occurred.

3.3   Shake Flask Experiments

3.3.1   Apparatus and Procedures

The shake flask experiments were carried out in 250 mL bottom-baffled Erlenmeyer flasks. Each flask contained a specific amount of feed material (depending on the required pulp density), 70 mL of bacterial nutrient solution, 3 g/L chloride and 5 mL of bacterial inoculum from a stock culture. Sterile control experiments were also carried out in which 5 mL of a bactericide (2 g/L thymol in methanol) was substituted for the bacterial inoculum. The flasks were incubated in a rotary shaker

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at a stirring speed of 250 RPM and a temperature of 35 °C.

The shake flasks were inoculated approximately twenty-four hours after starting the experiment. Time zero was considered to be the time of inoculation. Acid consuming reactions occurred during the twenty-four hours before inoculation and caused the pH to increase. Therefore, it was necessary to stabilize the pH in the optimal range for bacterial growth before adding the bacterial inoculum. The pH was adjusted to pH 2 by adding 6M H_2SO_4. Next, 5 mL of bacterial inoculum was added from a stock culture in the late exponential phase of growth. Bacteria on solids as well as in solution were transferred from the stock culture to the experimental flask. The shake flasks were maintained at a constant weight by addition of deionized water to compensate for evaporative losses. The pH was maintained at about pH 2 by adding 6M H_2SO_4 as required. When pH was an experimental variable, it was controlled by adding either 6M H_2SO_4 or 6M NaOH as required.

Dissolved copper and iron, as well as the pH and the Eh of the slurry were determined on a regular basis. In order to minimize loss of slurry, the pH and Eh probes were rinsed back into the shake flasks after each measurement. Upon completion of each experiment, the shake flask contents were filtered using the repulp method (refer to Section 3.7). The filtrate and wash solutions were analysed for copper and iron. The leach residue was analysed for copper, iron, total sulphur and sulphate sulphur.

3.3.2   Experimental Design

The variables tested in the shake flask experiments for the ore and concentrate are given in Table 3.3. Only one of each experiment was carried out. Sterile control experiments were also carried out for each variable tested. In addition, the bacterial culture was adapted to growth on the concentrate

using the serial transfer method and by gradually increasing the pulp density over a four month period.

Table 3.3 - Shake Flask Experimental Variables

Variable	Ore	Concentrate
P_{80} particle size (μm)	47 / 60 / 108	37 / 53
pH	1.25 / 1.5 / 1.75 / 2.0	1.25 / 1.5 / 1.75 / 2.0
pulp density (% w/v)	5 / 10 / 15 / 20	2.5 / 5 / 7.5 / 10
soluble iron concentration (g/L)	0 / 3 / 6 / 9	0 / 3 / 6 / 9
temperature (°C)	30 / 35 / 40	30 / 35 / 40
chloride concentration (g/L)	not tested	0/3

In tests where particle size was not a variable, particles sizes of P_{80} 108 μm and P_{80} 53 μm were used for the ore and concentrate respectively. Unless soluble iron concentration was a variable, 9K bacterial nutrient solution was used containing 9 g/L soluble iron.

3.4   Batch Reactor Experiments

3.4.1   Apparatus and Procedures

Upon completion of the shake flask experiments, larger scale batch reactor experiments were carried out in 2 L stirred tanks. The tanks and tank lids were manufactured of acrylic. A tank diameter to tank height ratio of approximately 0.7 was used. The tanks were vertically baffled with the baffle width being 0.1 times the tank diameter. The baffles were raised above the tank bottom to prevent stagnant zones from forming. Two impellers per tank were used. They were the four blade, forty-

five degree pitch down type with a diameter of 0.4 times the tank diameter. Titanium impellers and shafts were used due to the corrosive environment (chloride and ferric iron). An attempt was made to use Teflon coated stainless steel impellers but this was not successful. The Teflon coating wore rapidly and flaked off due to abrasion from the solid particles. Variable speed mixers were used to provide constant stirring at approximately 800 RPM. This was the minimum speed required for complete particle suspension. At this stirring rate, the impeller tip speed was calculated to be approximately 2.4 m/s. Air was sparged into the tank below the bottom impeller. The air supply came from a compressor and was CO_2-enriched to 1% v/v CO_2 using a CO_2 gas cylinder and a gas proportioner. A gas flowmeter was located at the inlet of each tank, and was used to regulate the CO_2-enriched air flowrate at 0.25 L air/ L pulp/minute. The temperature in each tank was controlled to 35°C using an immersion heater.

Operation and maintenance of the batch reactors was similar to the procedure used for the shake flask experiments. The batch reactors were inoculated approximately twenty-four hours after starting the experiment. Time zero was considered to be the time of inoculation. The inoculum consisted of 130 mL of a stock shake flask culture in the late exponential growth phase. The batch reactors were maintained at a constant volume by addition of deionized water to compensate for evaporative losses. The pH was maintained at about pH 2 by addition of 6M H_2SO_4 as required.

Dissolved copper and iron, as well as solution pH and Eh were determined on a regular basis. Upon completion of the experiment, the slurry was filtered using the repulp method (refer to Section 3.7). The filtrate and wash solutions were analysed for copper and iron. The leach residue was analysed for copper, iron, total sulphur and sulphate sulphur.

3.4.2 Experimental Design

The variables tested were pulp density and soluble iron concentration, as detailed in Table 3.4. Otherwise, conditions were the same as those used for the shake flask experiments. When soluble iron was a variable, the ore pulp density was fixed at 15%.

Table 3.4 - Batch Reactor Experimental Variables

Variable	Ore	Concentrate
pulp density (%)	10 / 15 / 20	5 / 7.5 / 10
soluble iron concentration (g/L)	0 / 3 / 6 / 9	not tested

Each experiment was carried out only once. Sterile control tests were not carried out for the batch reactor experiments.

3.5   Continuous Reactor Experiments

3.5.1   Apparatus

Two identical bench scale continuous reactor biological leaching systems were operated simultaneously for a period of four months; one system tested the ore and the other tested the concentrate. Each system consisted of three continuously stirred biological leaching reactors in series. The volume of the first stage reactor was 4 L, and the second and third stage reactor volumes were each 2 L. A schematic diagram of the system is given in Figure 3.1.

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Feed Tank
8 L

M

Timer

Feed
Pump

Acid Pump

Acid
Supply

M

pHC

TC

Bioleach
Stage 1
4 L

M

TC

Bioleach
Stage 2
2 L

M

TC

Bioleach
Stage 3
2L

Product
Tank
3 L

Air (1% CO$_2$)

Legend:
M   =   mixer
TC   =   temperature control
pHC =   pH control
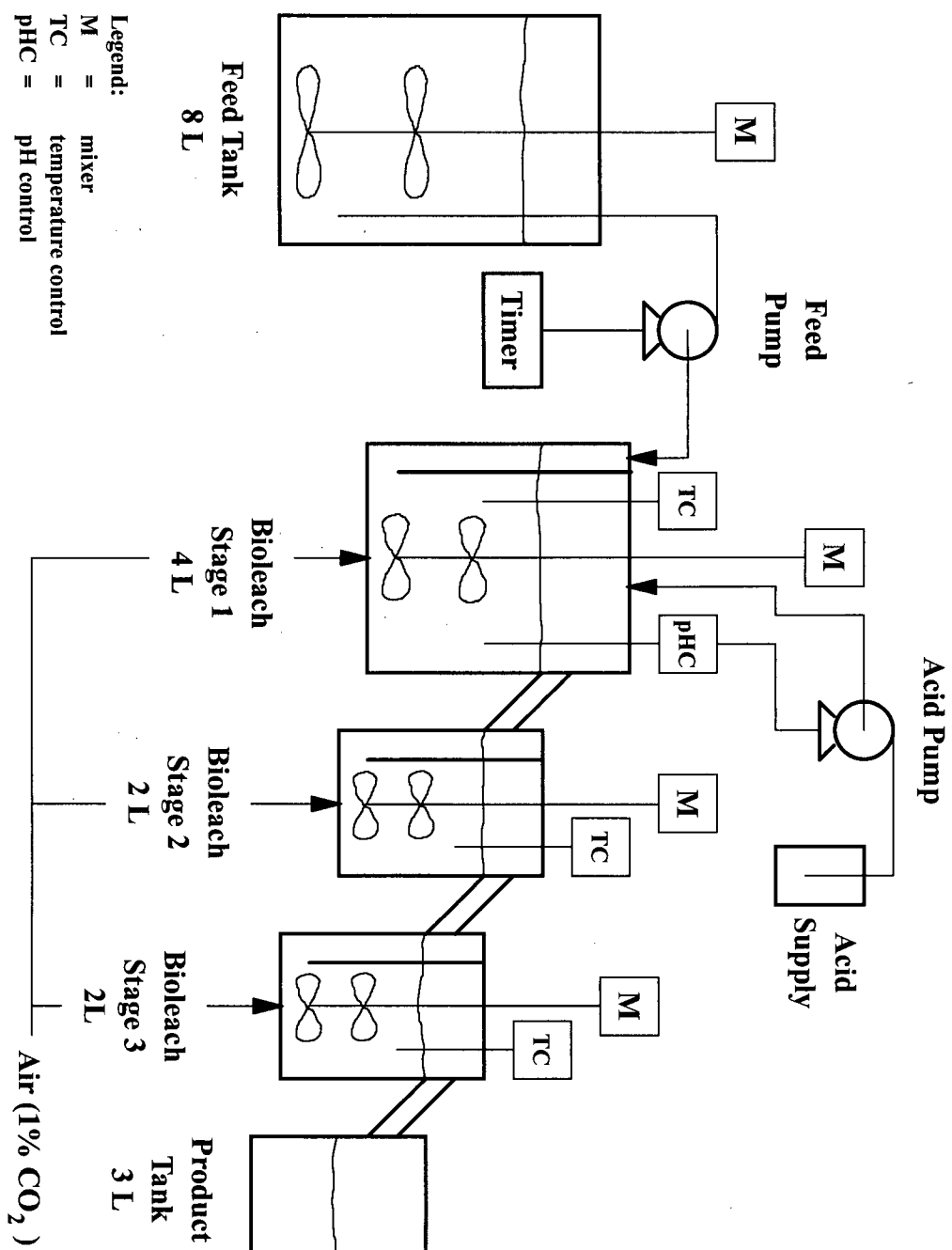
Figure 3.1 -   Schematic Diagram of the Continuous Biological Leaching System

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The feed tanks were very similar to the batch reactor tanks described above except that they were not supplied with air and they were not heated. Continuous flow was simulated by pumping the feed intermittently two or three times per hour for less than 10 seconds each cycle, depending on the desired residence time. Intermittent pumping was used to maximize the feed flow velocity and ensure representative transferring of slurry.

The continuous biological leaching tanks were also very similar to the batch reactor tanks described above. In addition, they contained an entrance weir that forced the incoming feed to the bottom of the tank before being mixed in the bulk solution; this was to reduce short-circuiting of the feed solids. Gravity overflow and inflow tubes were attached to each tank at the appropriate height to achieve the desired tank volume. The overflow and inflow tubes between tanks were joined by flexible PVC tubing. A controller maintained the pH in the first stage reactor at about pH 2 by addition of 6M H_2SO_4. Acid was not added to the feed tank or the second and third bioleach stages. The stirring rate was set at the minimum speed required for complete particle suspension. This was a range from about 600 RPM to about 1100 RPM. At these stirring rates, the impeller tip speeds were calculated to range from 2.3 to 4.2 m/s in the first stage bioleach reactor, and from 1.8 to 3.3 m/s in the second and third stage bioleach reactors.

Dissolved oxygen was measured using a dissolved oxygen meter with salinity compensation. A programmable timer was used to provide intermittent feed pumping. A pH probe was placed in each of the first stage reactors for continuous pH measurement. All pumps were variable speed peristaltic pumps. Tygon tubing was used to transfer acid, and silicone tubing was used to transfer the feed slurry. As in the batch reactor experiments, the impellers were the four blade, forty-five degree pitch down type. In the feed tanks, where the environment was not as corrosive as in the biological

leaching tanks, 316 stainless steel impellers were used. Due to the difficulty of obtaining titanium impellers with the correct diameter for the first stage reactor (2-7/8"), acrylic impellers were used. Titanium impellers were used in the second and third stage bioleach reactors.

3.5.2 System Operation

The system was commissioned in batch mode. When the Eh in all three tanks reached 800 mV (Standard Hydrogen Electrode reference, SHE), the feed pump was turned on to begin continuous mode operation. The dissolved oxygen concentration, pH and Eh in each bioleach tank were measured daily. Solution samples were taken daily to analyse for copper and iron. Product slurry generation was recorded daily in terms of volume and weight. The volume of sulphuric acid added to the first bioleach stage was recorded daily.

The criterion for determining when the system had reached steady state was a constant dissolved copper concentration for three consecutive days. On three different days at steady state, a slurry sample of approximately 100 mL was taken from each of the three bioleach tanks. At least one 100 mL slurry sample was taken from the feed tank during each steady state period. These samples were filtered using the repulp method (refer to Section 3.7). The leach residues were analysed for copper, iron, total sulphur and sulphate sulphur. The results were used to determine average percent copper extraction and average percent sulphate formation at that steady state condition. Next, one variable was changed and the process was repeated.

3.5.3 Experimental Design

Four steady state conditions were tested, as given in Table 3.5.

Table 3.5 - Design of the Continuous Biological Leaching Experiments

Variables	Test 1	Test 2	Test 3	Test 4
Zaldívar Ore				
pulp density (%)	10	10	10	20
flowrate (L/day)	1	2	3	3
total residence time (days)	8	4	2.67	2.67
Zaldívar Concentrate				
pulp density (%)	5	5	5	10
flowrate (L/day)	1	2	3	3
total residence time (days)	8	4	2.67	2.67

3.6   pH and Eh Measurements

Gel-filled combination pH probes with a silver/silver chloride reference electrode were used to measure pH. A flat-bottomed pH probe was used for the shake flask experiments because this type of probe minimized adherence of slurry to the probe.

Eh was measured using a platinum redox electrode combined with a silver/silver chloride reference electrode. A constant value of 199 mV was added to all measured Eh values. This was the redox potential developed by the reference electrode relative to the standard hydrogen electrode at a temperature of $25\,^{\circ}C$.

3.7   Repulp Filtration Method

Slurries were filtered using the repulp method. This method involved filtering the solids twice. After the first filtration, the filtrate volume was recorded and a sample of the filtrate solution was taken. Next the solids were washed in a volume of deionized water that was about four times the

filtrate volume. The purpose of this repulping step was to wash dissolved copper containing solution out of the first filter cake. The solids were suspended in this solution by stirring for about 10 minutes before being filtered again. After the second filtration, the wash volume was recorded and the wash solution was sampled. The filtrate and wash solutions were analysed for copper and iron concentrations. The final filtration residue was dried at ambient temperature for two or three days. Since most sulphide minerals do not oxidize appreciably below about 100°C (Vaughan and Craig, 1978), drying at less than 100°C would likely not cause significant errors in the total sulphur results. However, to minimize the tendency for oxidation of sulphide sulphur or elemental sulphur to sulphur dioxide, ambient temperature was used.

3.8   Sample Analysis

Samples were analysed by personnel at either the Placer Dome Research Centre or International Plasma Laboratory (IPL) of Vancouver. Solution samples were analysed for copper and iron by atomic absorption spectrophotometry (AAS). The filtration residues were split into several parts for various analyses. Part of the residue was analysed for copper and iron by acid digestion and AAS analysis of the resulting solution. The rest of the residue was analysed for percent total sulphur and percent sulphate sulphur. Total sulphur was determined by the Leco method. Sulphate sulphur was determined by boiling in 15% HCl to selectively dissolve sulphate, followed by gravimetric determination of sulphur as $BaSO_4$. The difference between percent total sulphur and percent sulphate sulphur included both sulphide sulphur and elemental sulphur. This difference was used to calculate percent sulphate formation.

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4.0 RESULTS AND DISCUSSION

The results of this study are presented in the following order: shake flask experimental results, batch reactor experimental results and continuous reactor experimental results. The acid consumption results for all three phases are presented and discussed in a separate section.

4.1 Shake Flask Experiments

4.1.1 Comparison of Ore and Concentrate Bioleaching

The results of the shake flask experiments showed that the bacterial lag phase was much longer with the Zaldívar concentrate than with the ore, particularly in the presence of chloride. This is illustrated in Figure 4.1, which gives the Eh as a function of time for both the ore and the concentrate in 3 g/L chloride media, and for the concentrate in 0 g/L chloride media. A significant effect of chloride on bacterial growth was not observed with the ore. These results suggest that the higher copper content of the concentrate and the presence of flotation chemicals had an inhibitory effect on bacterial growth. This effect was magnified in the presence of 3 g/L chloride. As will be shown, this inhibition of bacterial growth was overcome in the concentrate continuous bioleaching experiment. The stock bacterial cultures used in this study were previously adapted to growth on the Zaldívar ore. However, they had not previously been grown on the Zaldívar concentrate. It was anticipated that the culture could also be adapted to the concentrate.

4.1.2 Bacterial Adaptation to the Concentrate

Figure 4.2 illustrates the effect of bacterial adaptation to the concentrate on the solution redox potential. The bacteria were adapted to growth on the concentrate in a 3 g/L chloride environment
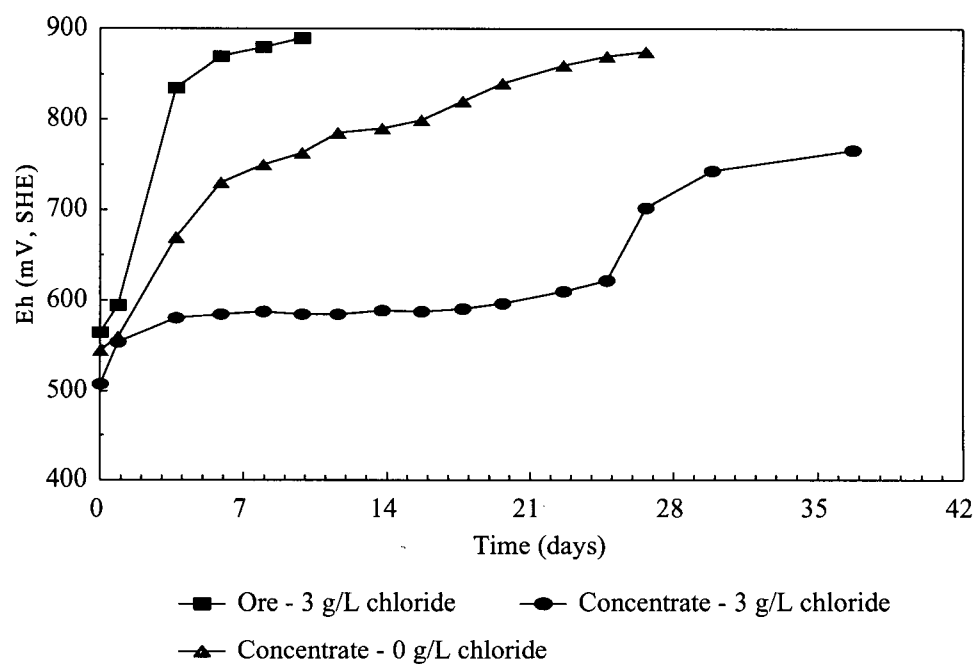
Figure 4.1 -   Effect of Feed Material and Chloride Concentration on Bacterial Growth in Shake
                Flasks at a Pulp Density of 10% as Measured by Eh
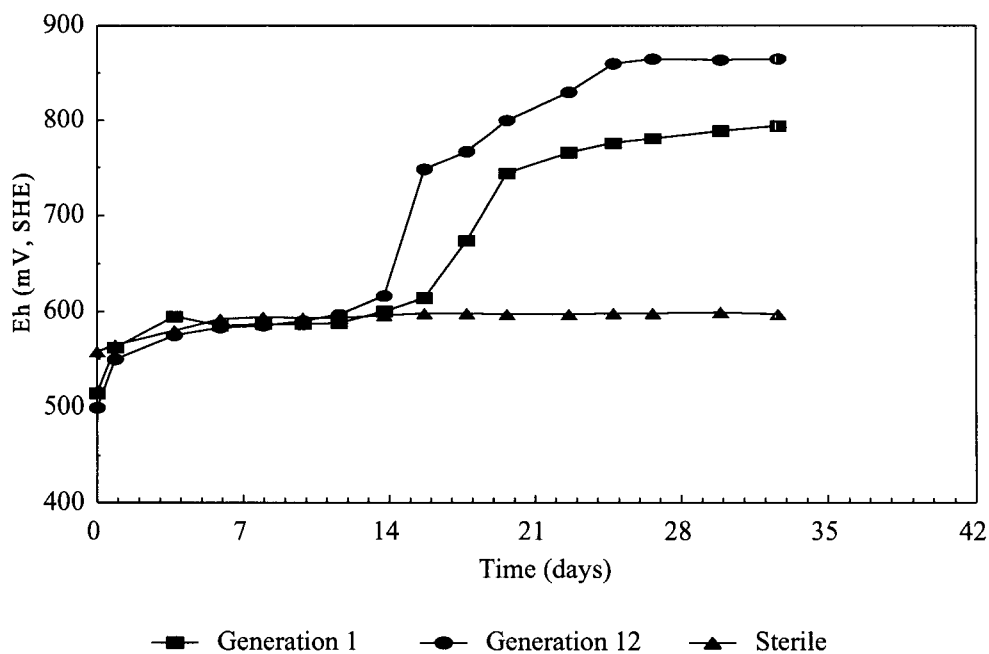
Figure 4.2 -   Adaptation of Bacteria to the Zaldívar Concentrate at a Pulp Density of 7.5% in
                Shake Flask Tests in a 3 g/L Chloride Environment as Measured by Eh

using the serial transfer method and by gradually increasing the pulp density. Curves of Eh versus time for experiments in the first and twelfth generations of bacteria growing on the Zaldívar concentrate at a pulp density of 7.5% are shown. The Eh profile for a sterile control experiment at the same pulp density is also given. The twelfth generation was reached in about four months.

4.1.3 Typical Bioleaching Results

Figure 4.3 gives a typical copper extraction profile resulting from the biological leaching of the Zaldívar ore in a 3 g/L chloride environment. This experiment was carried out at a pulp density of 10%. The results of a sterile control experiment are also given. Profiles of dissolved copper and iron concentrations for the same experiment are given in Figure 4.4, and the corresponding solution Eh and pH profiles are given in Figure 4.5. Similar results for the biological leaching of the Zaldívar concentrate are given in Figure 4.6, Figure 4.7 and Figure 4.8. The apparent decrease in the final data point of most copper extraction and dissolved copper profiles was due to dilution of the filtrate during repulp filtration on the final day of the experiment. This apparent decrease was not observed for experiments that were terminated before copper extraction was complete.

In the shake flask experiments, the ultimate copper extractions were consistently very high for all variables tested. The average ultimate copper extractions were found to be 90% for the Zaldívar ore and 97% for the Zaldívar concentrate. As shown in Figures 4.3 and 4.6, a significant proportion of the copper extraction occurred before inoculation of the bioleach experiments. As will be discussed in Section 4.1.4, part of this non-biological copper extraction is attributable to the dissolution of the oxide copper mineral brochantite ($Cu_4(SO_4)(OH)_6$). Also, as will be discussed in Section 4.1.4, part of the copper extraction that occurred before inoculation is attributable to chalcocite oxidation to covellite.
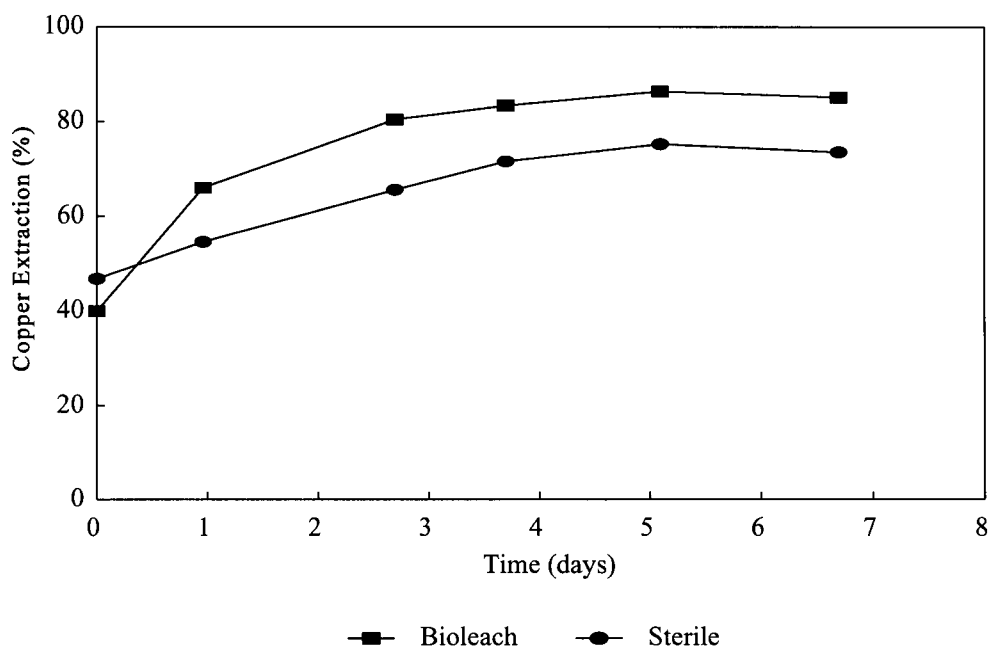
Figure 4.3 -   Copper Extraction During the Shake Flask Biological Leaching of the Zaldívar
                Ore at a Pulp Density of 10% in a 3 g/L Chloride Environment
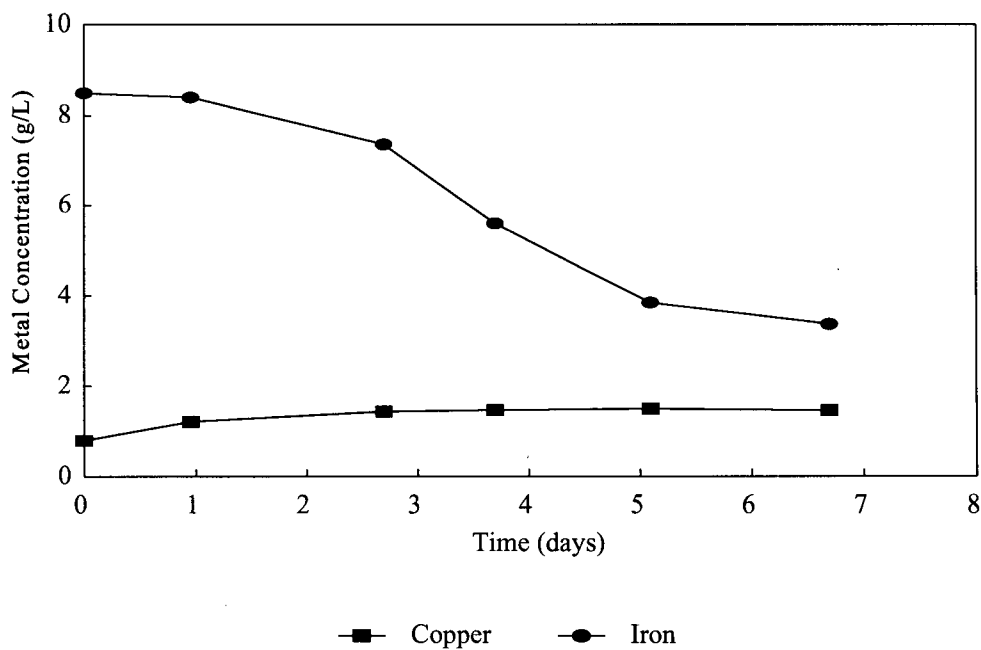
Figure 4.4 -   Copper and Iron Concentrations in Solution During the Shake Flask Biological
                Leaching of the Zaldívar Ore at a Pulp Density of 10% in a 3 g/L Chloride
                Environment
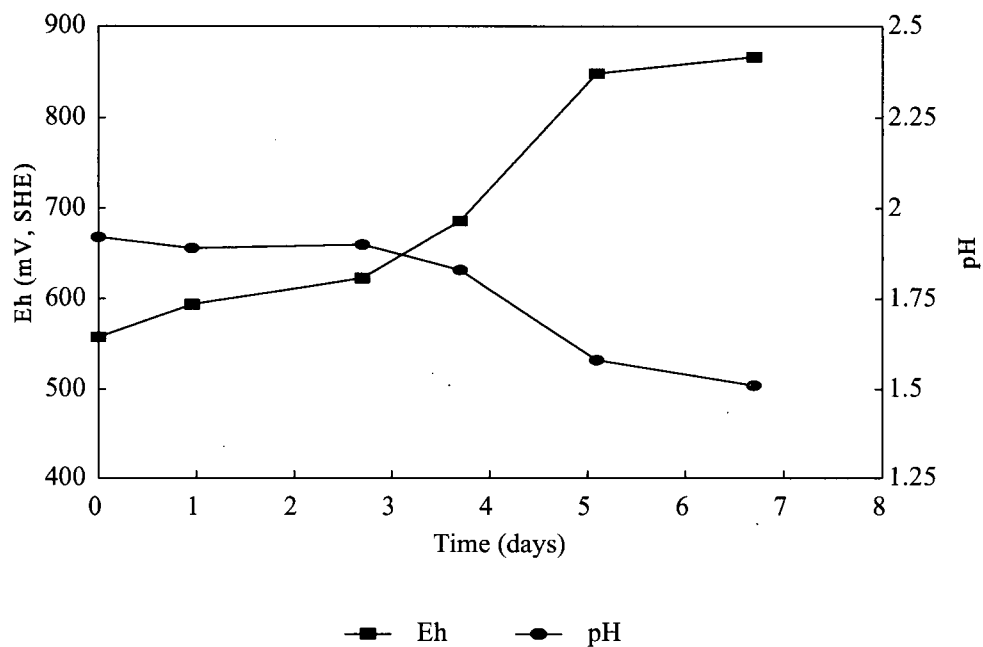
Figure 4.5 -   Solution Eh and pH During the Shake Flask Biological Leaching of the Zaldívar
                Ore at a Pulp Density of 10% in a 3 g/L Chloride Environment
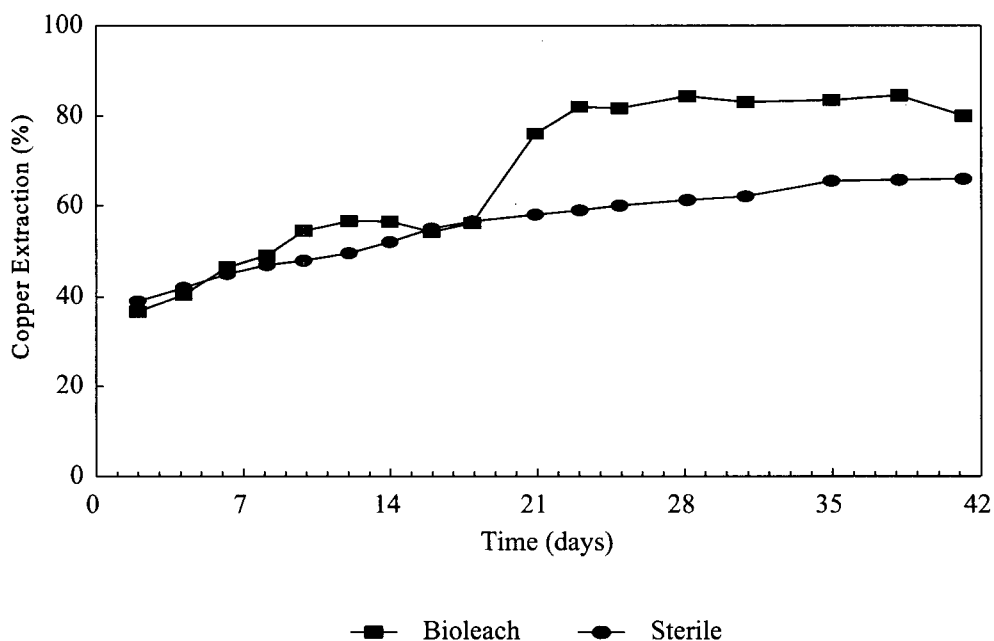
Figure 4.6 -   Copper Extraction During the Shake Flask Biological Leaching of the Zaldívar
                Concentrate at a Pulp Density of 10% in a 3 g/L Chloride Environment
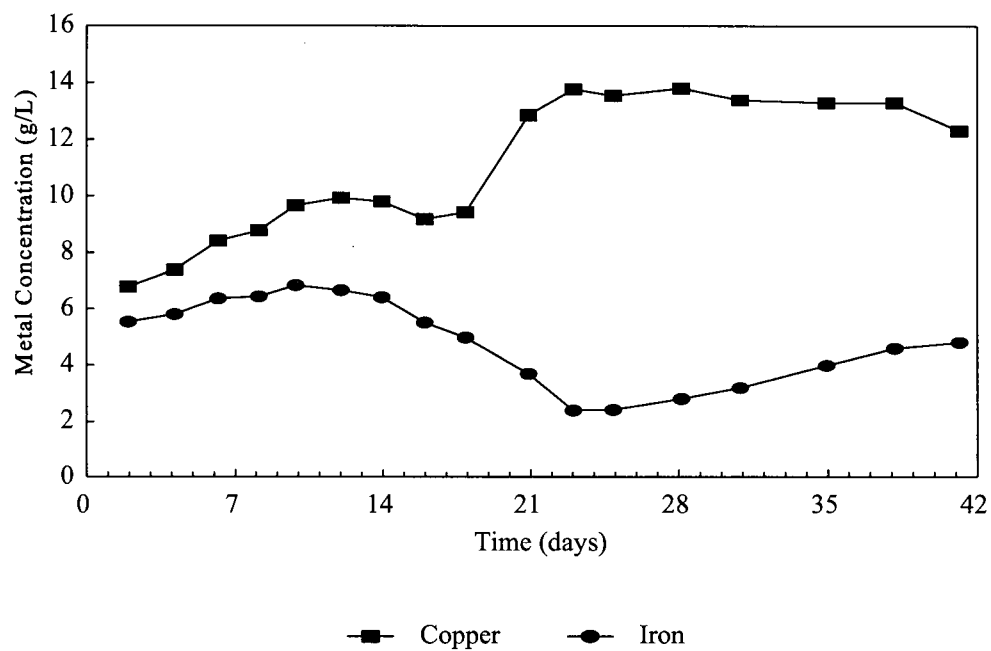
Figure 4.7 -   Copper and Iron Concentrations in Solution During the Shake Flask Biological Leaching of the Zaldívar Concentrate at a Pulp Density of 10% in a 3 g/L Chloride Environment
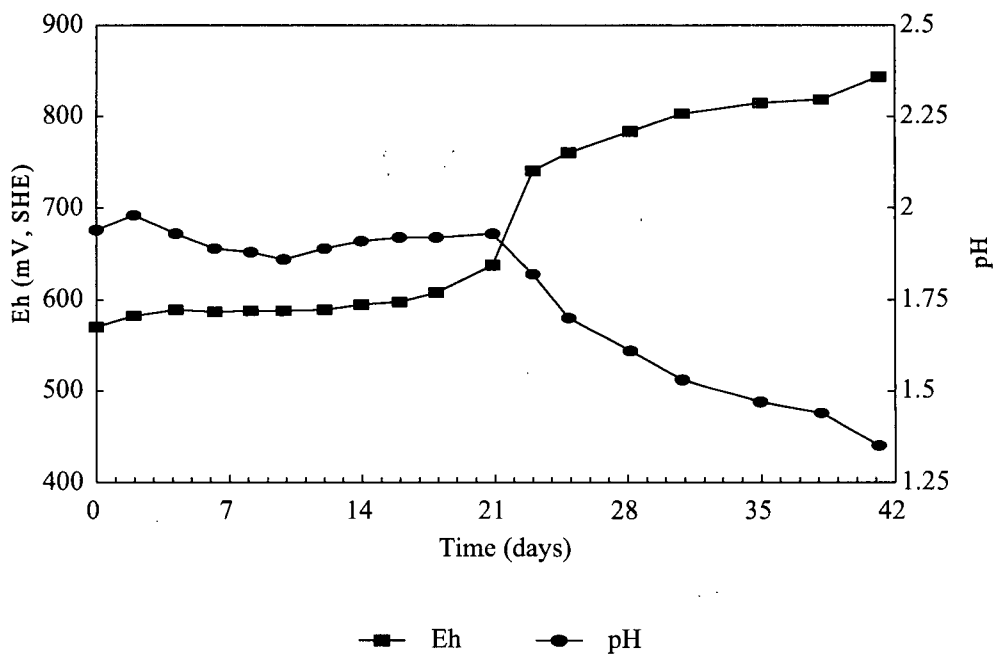
Figure 4.8 -   Solution Eh and pH During the Shake Flask Biological Leaching of the Zaldívar Concentrate at a Pulp Density of 10% in a 3 g/L Chloride Environment

The dissolved iron profile shown in Figure 4.7 is of particular interest. During the third week of the experiment, a significant decrease in dissolved iron concentration was observed, which coincided with an increase in dissolved copper concentration. The decrease in dissolved iron concentration was most likely due to jarosite precipitation. Once the dissolved copper concentration had reached its ultimate value, the dissolved iron concentration began to increase again. This phenomenon is also illustrated very clearly in Figure 4.9, which gives dissolved copper and iron profiles for the biological leaching of the Zaldívar concentrate in solutions with no added chloride. The increase in dissolved iron concentration was most likely related to pyrite oxidation. The dissolved iron profiles suggest that the majority of pyrite was oxidized after most of the copper leaching had occurred. This confirms that chalcocite and covellite leach preferentially to pyrite.
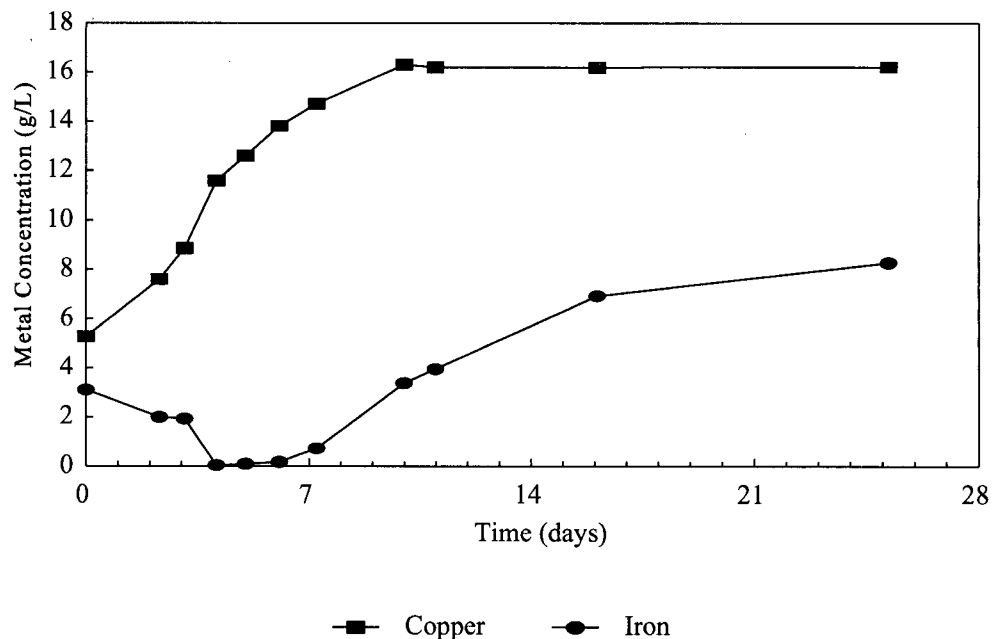
Figure 4.9 - Copper and Iron Concentrations in Solution During the Shake Flask Biological Leaching of the Zaldívar Concentrate at a Pulp Density of 10% in a 0 g/L Chloride Environment

The shake flask experiments were terminated when copper leaching was complete, not necessarily when the maximum sulfate formation was reached. Therefore, most experiments resulted in very high copper extraction but much lower sulfate formation. As discussed above, incomplete sulfate formation was partly attributable to incomplete oxidation of pyrite. As will be discussed later, another likely factor was incomplete oxidation of elemental sulphur.

4.1.4 Sterile Control Experiments

The average copper extractions for the sterile control experiments were found to be 77% for the ore and 68% for the concentrate. Part of the copper extraction can be attributed to the dissolution of the oxide copper mineral brochantite:

$$Cu_4(SO_4)(OH)_6 + 3H_2SO_4 \rightarrow 4CuSO_4 + 6H_2O \tag{13}$$
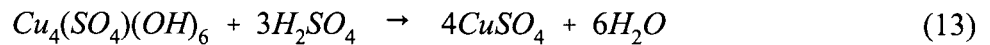
However, based on the ratio of acid soluble copper to total copper in each of the feed materials, brochantite dissolution can only account for 32% and 9% of copper extraction from the ore and concentrate respectively. Therefore, some chemical oxidation of chalcocite must have occurred in the sterile control experiments. Chalcocite oxidation to covellite can account for a further 34% and 46% of the copper extraction observed in sterile control experiments on the ore and concentrate respectively:
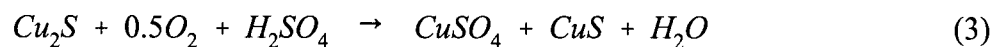
$$Cu_2S + 0.5O_2 + H_2SO_4 \rightarrow CuSO_4 + CuS + H_2O \tag{3}$$

Chalcocite oxidation occurs spontaneously but the kinetics are slow at ambient temperature and pressure. However, over the long leach times of these experiments, it was expected that some

chemical oxidation of chalcocite would occur. In order to maintain a pH of 2, it was only necessary to add acid to the biological leaching shake flasks at the beginning of each experiment. However, it was necessary to add acid to the sterile control shake flasks throughout the experiment. Since chalcocite oxidation is acid consuming, as shown in Equation 3, this is further evidence that chalcocite oxidation occurred in the sterile control experiments.

The remaining 11% and 13% of the copper extractions that were observed in the sterile control experiments on the ore and concentrate respectively, can be attributed to covellite oxidation:

$$CuS + 2O_2 \rightarrow CuSO_4 \qquad (4)$$

However, less than 5% sulphate formation was found to occur in the ore sterile control experiments, and no sulphate formation was found to occur in the concentrate sterile control experiments. These results suggest that the covellite-sulphide was not oxidized to sulphate but to elemental sulphur, as shown in Equation 8:

$$CuS + Fe_2(SO_4)_3 \rightarrow CuSO_4 + S^o + 2FeSO_4 \qquad (8)$$
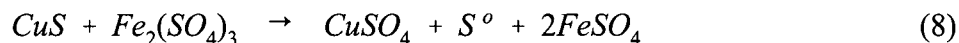
In order for the above reaction to occur, there must have been some ferric iron in solution in the sterile control flasks. Some ferrous iron present in the bacterial nutrient media was likely oxidized to ferric iron as follows:

$$2FeSO_4 + 0.5O_2 + H_2SO_4 \rightarrow Fe_2(SO_4)_3 + H_2O \qquad (6)$$
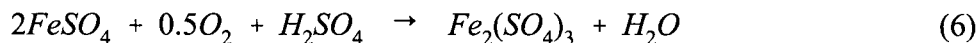
This reaction is catalysed by bacteria but it also occurs chemically at much lower rates. Ferric iron must have been consumed as quickly as it was produced because no Eh increase was observed.

4.1.5 Effect of Particle Size

The effect of ore particle size on sulphate formation after five days of biological leaching is given in Figure 4.10. Particle size had no noticeable effect on ultimate copper extraction, copper leach rate or acid consumption. However, the results showed that for both the ore and concentrate, as the particle size increased, the sulphate formation at the end of the experiment was lower. As copper extractions were not affected by particle size, and were over 95% in these experiments, the incomplete sulphate formation was likely due to incomplete oxidation of pyrite or elemental sulphur. Sulphate formation for the two concentrate particle sizes tested, which were P_{80} 37 microns and P_{80} 53 microns, were 87% and 74% respectively after eleven days of biological leaching.
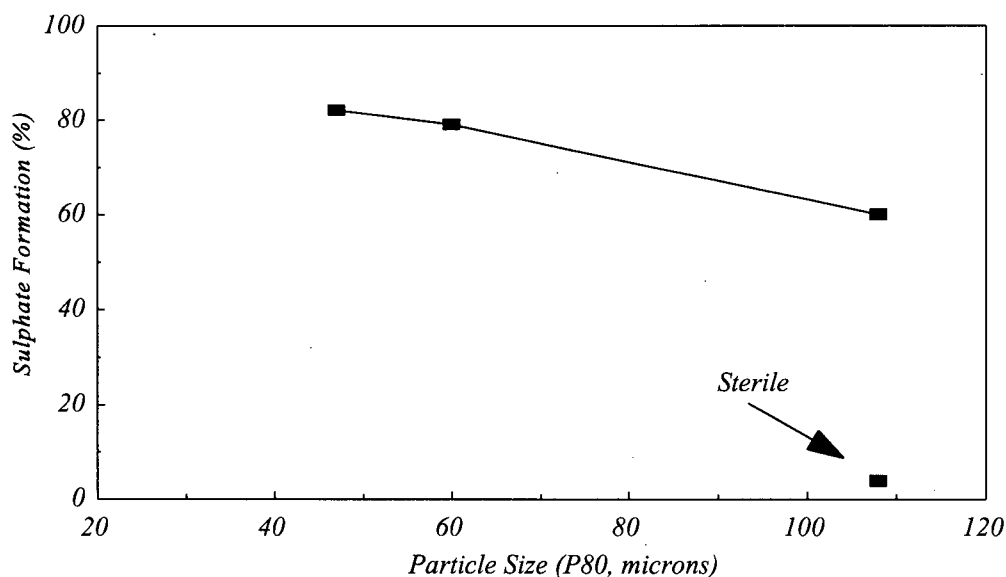
Figure 4.10 - Effect of Ore Particle Size on Sulphate Formation After 5 Days of Shake Flask Biological Leaching in a 3 g/L Chloride Environment

4.1.6   Effect of pH

There was no noticeable effect of the controlled pH value on ultimate copper extraction or copper leach rate. The controlled pH value was observed to effect the sulphate formation at the end of the experiment. As the controlled pH value decreased, the sulphate formation at the end of the experiment decreased. For example, in an experiment on the concentrate at 10% pulp density, the pH was controlled to pH 2 and the sulphate formation was found to be 81% after 26 days of biological leaching. However, when the pH was controlled to pH 1.25, there was found to be no sulphate formation after 54 days of biological leaching. The lower sulphate formation was most likely related to lower oxidation of pyrite or elemental sulphur. Since pyrite oxidation is an acid generating reaction, it was inhibited in more acidic environments:

$$2FeS_2 + 7.5O_2 + H_2O \rightarrow Fe_2(SO_4)_3 + H_2SO_4 \tag{5}$$
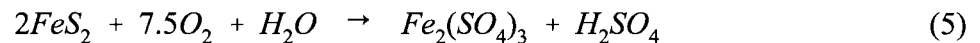
Another possibility is that the lower pH values inhibited the oxidation of elemental sulphur, which is also an acid generating reaction:

$$S^o + 1.5O_2 + H_2O \rightarrow H_2SO_4 \tag{9}$$
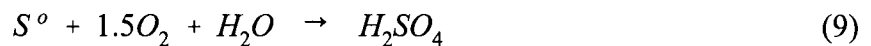
4.1.7   Effect of Pulp Density

Figure 4.11 and Figure 4.12 give the Eh profiles for the bioleaching of various pulp densities of the Zaldívar ore and the Zaldívar concentrate respectively in a 3 g/L chloride environment. The results from sterile control experiments are also given. The length of the bacterial lag phase was greater for

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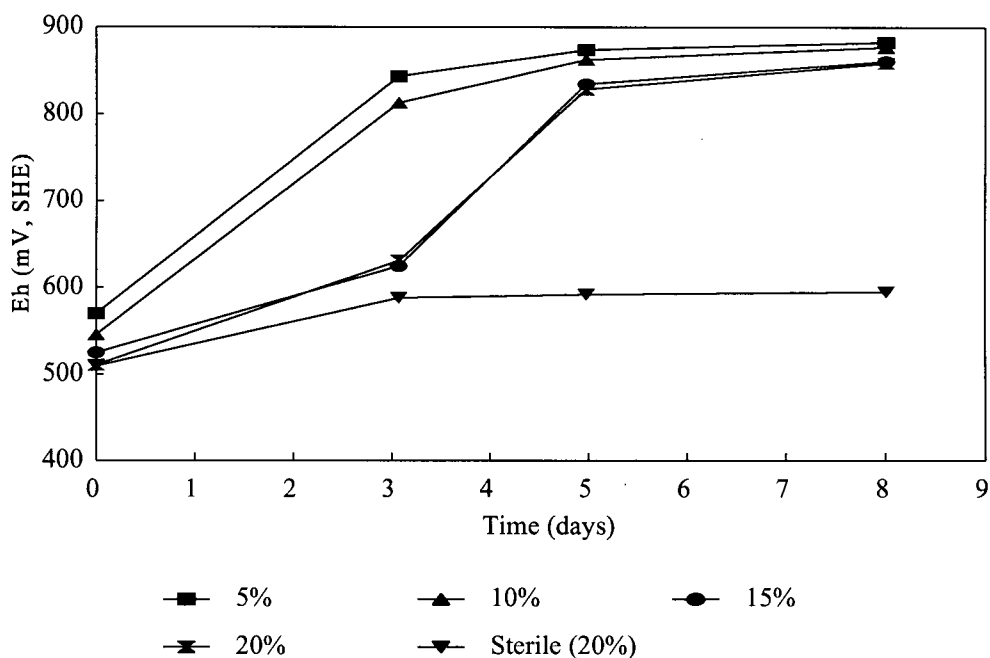
Figure 4.11 -   Effect of Ore Pulp Density on Biological Leaching in Shake Flask Tests in a 3 g/L
Chloride Environment as Measured by Eh
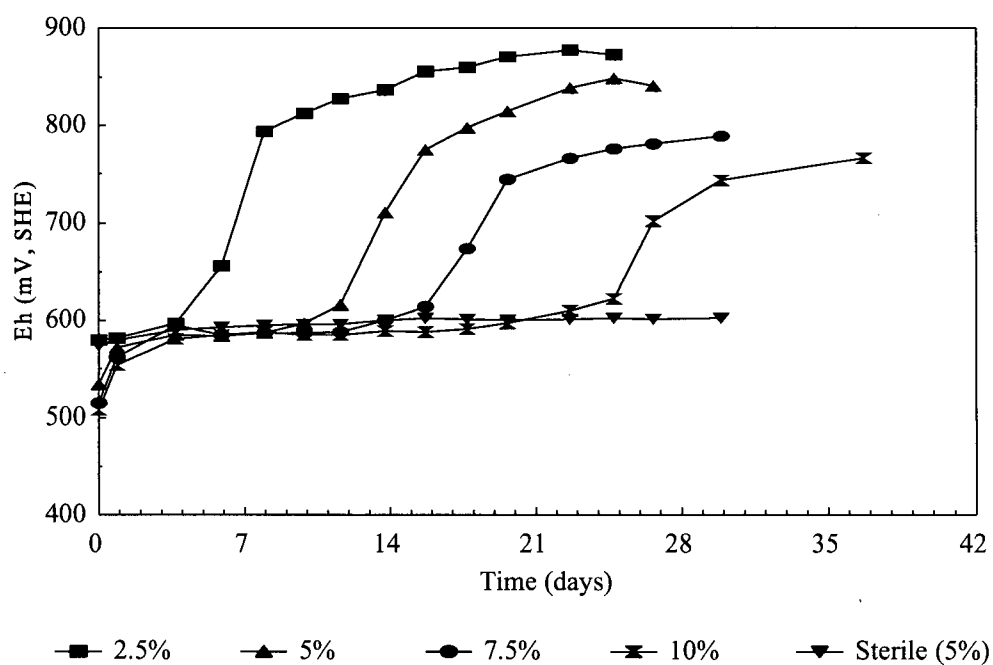
Figure 4.12 -   Effect of Concentrate Pulp Density on Biological Leaching in Shake Flask Tests
in a 3 g/L Chloride Environment as Measured by Eh

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ore pulp densities over 10%, and increased with increasing pulp density for the concentrate. The increased lag time with increasing pulp density suggests that it took longer to establish a bacterial population at higher pulp densities. This may have been due to a limited supply of carbon dioxide at higher pulp densities in the shake flask experiments. Another reason may have been adherence of bacteria to solids. As the pulp density increased, the initial ratio of bacteria in solution to bacteria on solids may have decreased. Then, there would have been fewer bacteria in solution to oxidize ferrous to ferric iron, resulting in the longer observed bacterial lag phase at higher pulp densities.

There was no noticeable effect of pulp density on the ultimate copper extraction or on acid consumption. The copper leach rate was observed to be directly proportional to pulp density for both feed materials; it doubled with a doubling of the pulp density. The copper leach rate ranged from 4 to 18 mg/L/h for ore pulp densities ranging from 5 to 20%, and from 13 to 42 mg/L/h for concentrate pulp densities ranging from 2.5 to 10%. The sulphate formation at the end of the experiment was observed to decrease with increasing pulp density. This effect of pulp density on sulphate formation was likely due to incomplete oxidation of pyrite or elemental sulphur.

4.1.8 Effect of Initial Soluble Iron Concentration

Figure 4.13 shows the effect of initial soluble iron concentration on the bioleaching of the Zaldívar concentrate at a pulp density of 10%. The length of the lag phase appeared to be related to the initial soluble iron concentration, for both the Zaldívar ore and concentrate. Shorter lag phases were observed with higher initial soluble iron concentrations. Apparently, as shown in Figure 4.13, bacterial growth was inhibited during the experiment in which the initial soluble iron concentration was 6 g/L. This may have been due to insufficient transfer of bacteria when the shake flask experiment was inoculated. No significant effect of initial soluble iron concentration was observed

on the acid consumption, ultimate copper extraction, rate of copper leaching or sulphate formation.
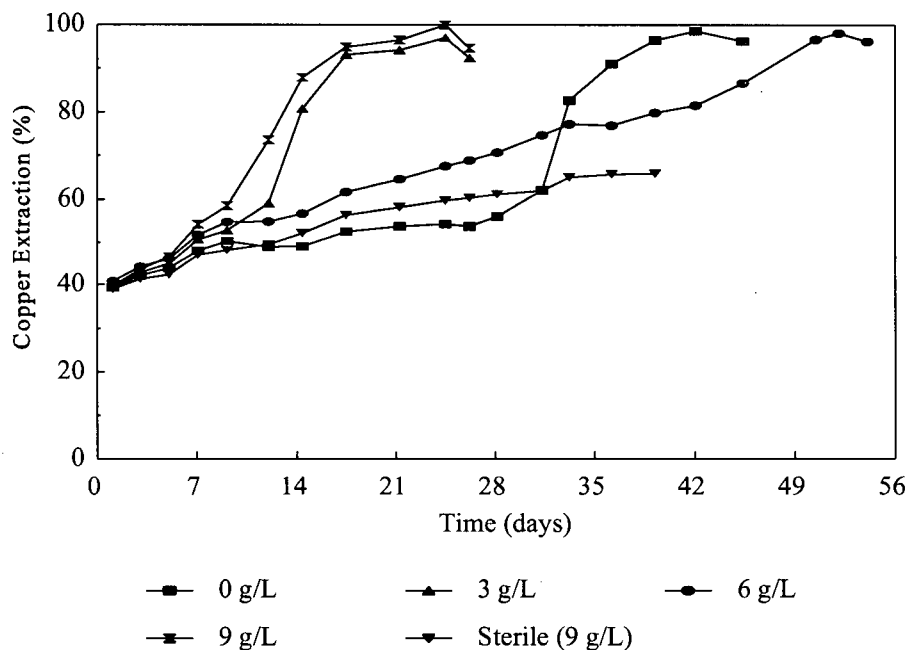
Figure 4.13 - Effect of Initial Soluble Iron Concentration on Copper Extraction from the Zaldívar Concentrate at 10% Pulp Density in Shake Flask Tests in a 3 g/L Chloride Environment

The final solution pH was observed to be lower with higher initial soluble iron concentrations for both feed materials, as illustrated in Figure 4.14 for the biological leaching of the Zaldívar ore at 10% pulp density. This was likely due to increased jarosite precipitation at higher iron concentrations. Since sulphuric acid is generated when jarosite precipitates, the more this reaction occurred, the more acidic the solution became:

$$3Fe_2(SO_4)_3 + 14H_2O \rightleftharpoons 2H_3OFe_3(SO_4)_2(OH)_6 + 5H_2SO_4 \qquad (10)$$

4.1.9 Effect of Temperature

No effects of temperature on acid consumption, ultimate copper extraction or length of the bacterial lag phase were observed in the range tested of 30°C to 40°C. Both the copper leach rate and the sulphate formation at the end of the experiment were observed to increase with increasing temperature. The higher sulphate formation values were most likely due to higher rates of oxidation of pyrite or elemental sulphur. These effects were observed in the sterile control flasks, as well as in the bioleaching shake flask experiments. For example, the copper leach rate from the ore at 10% pulp density was found to increase from 9 to 23 mg/L/h in the temperature range 30 to 40°C. In the corresponding sterile control experiments, the rate was found to increase from 2 to 6 mg/L/h in the temperature range 30 to 40°C.
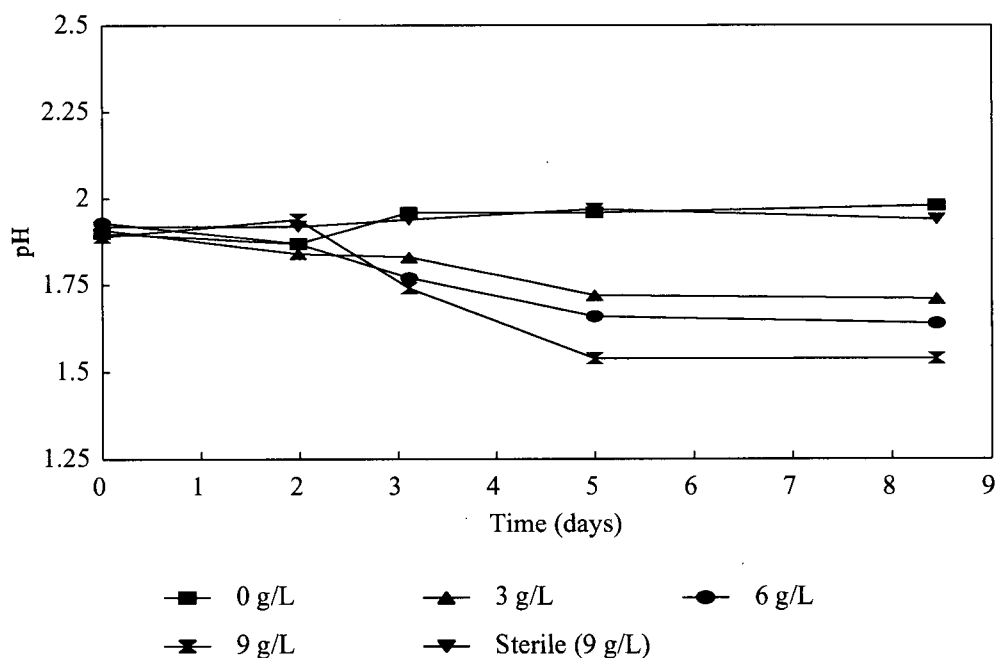
Figure 4.14 -   Effect of Initial Soluble Iron Concentration on the Solution pH During the Shake Flask Biological Leaching of the Zaldívar Ore at 10% Pulp Density in a 3 g/L Chloride Environment

4.1.10 Summary

The shake flask experiments demonstrated that the bacterial culture was successfully grown on both the Zaldívar ore and concentrate. The average copper extractions achieved in the shake flask tests were 90% and 97% for the ore and concentrate respectively. The higher copper extractions achieved for the concentrate were most likely due to the fact that the concentrate experiments were allowed to proceed longer than the ore experiments; the ore experiments were terminated before copper extraction was complete. The average copper leach rates were 12 mg/L/h for the ore and 50 mg/L/h for the concentrate, for pulp densities of 10%.

Average acid consumptions were 20 kg/t for the ore and 170 kg/t for the concentrate. The Eh ranged from initial values of about 500 mV to 900 mV (SHE) in active bacterial cultures. Sulphate formation, which was determined at the end of the experiment, was found to be significantly lower than the final copper extraction values achieved. Incomplete sulphate formation was thought to be due to incomplete oxidation of pyrite or elemental sulphur. Sample calculations for copper extraction, copper leach rate and sulphate formation are given in Appendix A. Selected shake flask experimental data is given in Appendix B.

4.2   Batch Reactor Experiments

4.2.1   Effect of Pulp Density

Figure 4.15 and Figure 4.16 give the copper extraction versus time for various pulp densities of the Zaldívar ore and the Zaldívar concentrate respectively in a 3 g/L chloride environment. As was observed in the shake flask experiments, the copper leach rate was directly proportional to the pulp density for both the ore and concentrate. There was an increase in the length of the lag phase at pulp
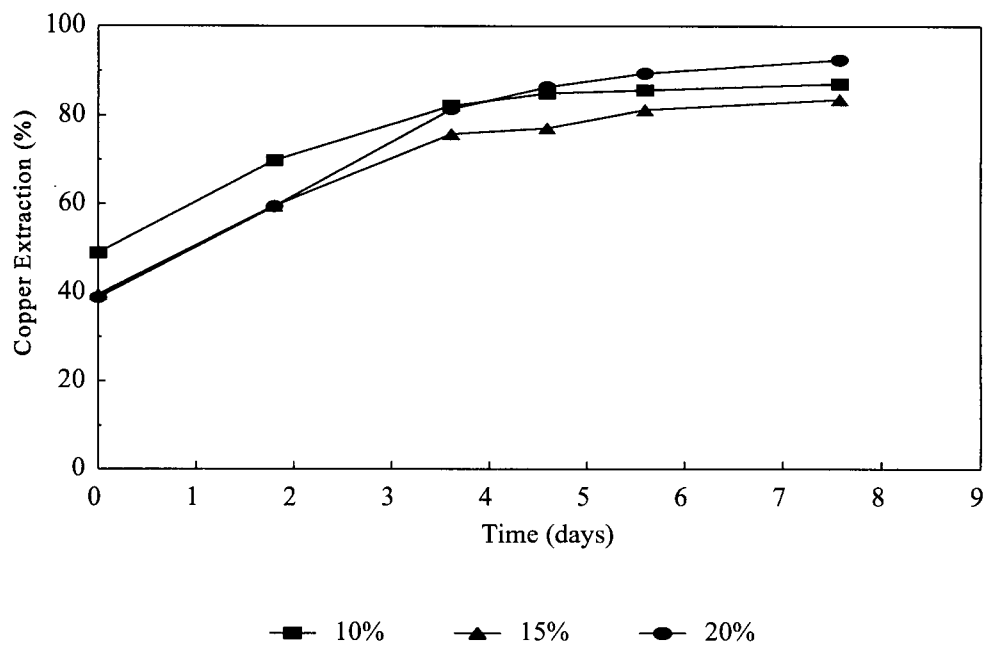
Figure 4.15 - Effect of Ore Pulp Density on Copper Extraction During Batch Reactor Tests in a 3 g/L Chloride Environment
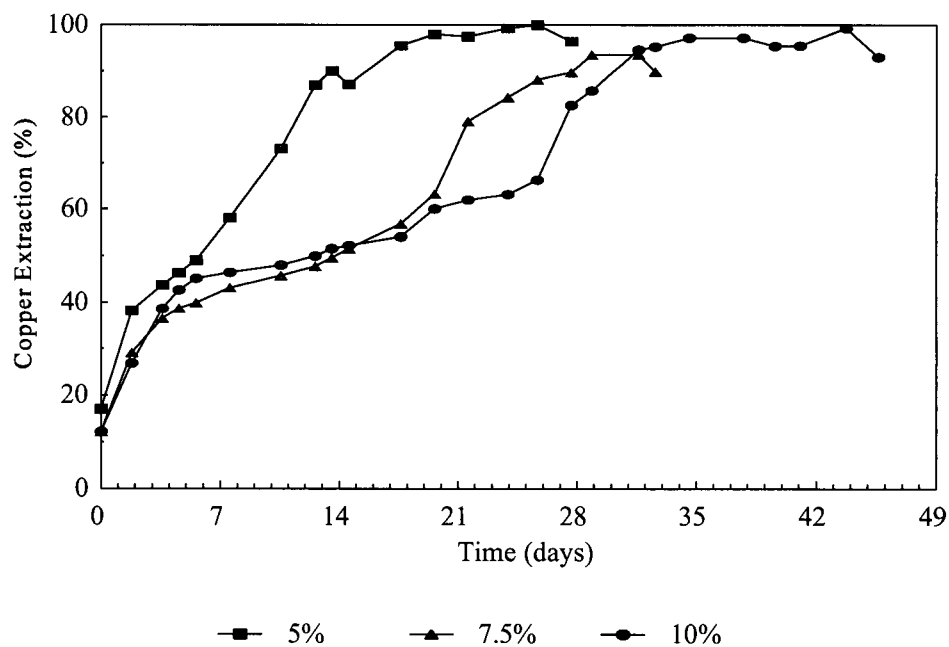
Figure 4.16 - Effect of Concentrate Pulp Density on Copper Extraction During Batch Reactor Tests in a 3 g/L Chloride Environment

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densities over 10% for the ore, and with increasing pulp density for the concentrate, as was observed in the shake flask experiments. Pulp density had no noticeable effect on ultimate copper extraction, sulphate formation at the end of the experiment, or acid consumption. The increased lag phase at higher pulp densities may have been due to limited availability of carbon dioxide, which is necessary for bacterial growth. In the case of the concentrate, longer lag phases may have been partly due to the inhibitory effects of higher dissolved copper and higher concentrations of flotation chemicals. The fact that there was no noticeable effect of pulp density on ultimate copper extraction suggests that the bacteria were able to adapt to these conditions at higher concentrate pulp densities.

4.2.2   Effect of Leach Time on the Extent of Sulphate Formation

The extent of sulphate formation was found to increase with time after copper leaching was complete, all other experimental variables being equal. In duplicate batch reactor experiments on the Zaldívar concentrate, sulphate formations of 68% and 82% were achieved after 15 and 27 days respectively; corresponding copper extraction values were 95% and 97% respectively. As discussed in Section 4.1.3, incomplete sulfate formation is partly attributable to incomplete oxidation of pyrite. In addition, longer leaching times may have resulted in further oxidation of the elemental sulphur formed by the indirect biological leaching of covellite:

$$CuS + Fe_2(SO_4)_3 \rightarrow CuSO_4 + S^o + 2FeSO_4 \qquad (8)$$
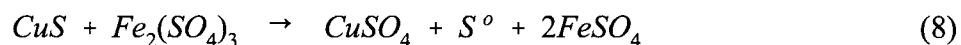
Figure 4.17 gives the dissolved copper and iron concentration profiles resulting from the duplicate batch reactor experiments carried out on Zaldívar concentrate. This graph clearly illustrates the increase in dissolved iron concentration between day 15 and day 27 of the longer experiment, which was likely due to pyrite oxidation. After approximately 15 days, the bacteria had oxidized all the

chalcocite and were left with only pyrite and elemental sulphur as a sources of energy.
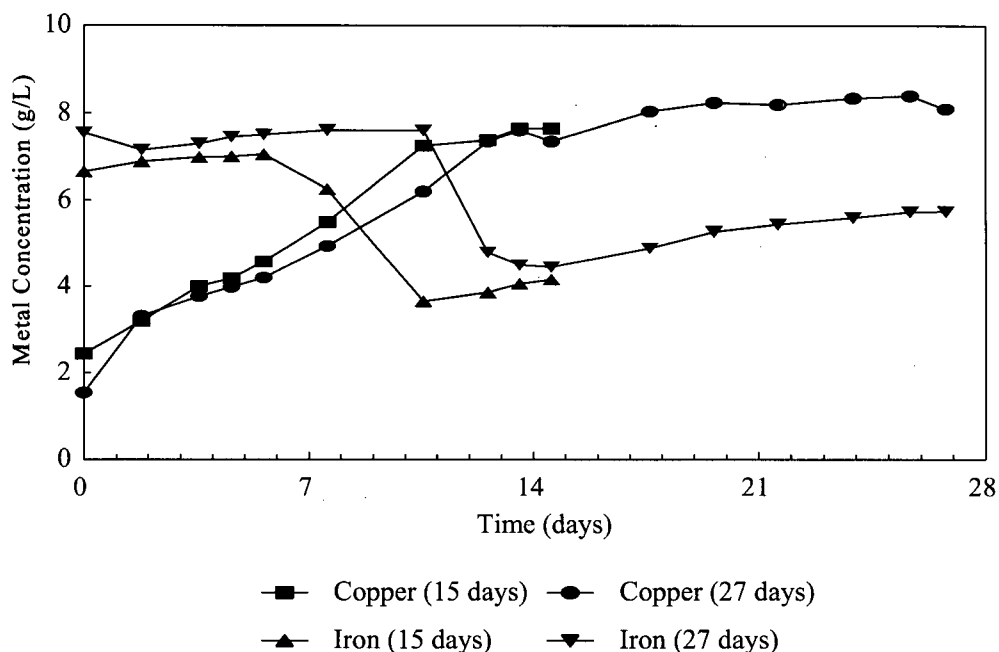
Figure 4.17 - Copper and Iron Concentrations in Solution in Duplicate Batch Reactor Tests at a Pulp Density of 5% in a 3 g/L Chloride Environment

4.2.3 Effect of Initial Soluble Iron Concentration

Figure 4.18 gives copper extraction versus time for the bioleaching of the Zaldívar ore at 15% pulp density with several initial soluble iron concentrations. As shown in Figure 4.18, the initial soluble iron concentration had no observable effect on the copper leach rate or the ultimate copper extraction. There was also no noticeable effect of initial soluble iron concentration on sulphate formation at the end of the experiment, or on acid consumption.

Figure 4.19 shows the effect of initial soluble iron concentration on the pH profiles for batch reactor bioleaching experiments on the ore at 15% pulp density. As was observed in the shake flask

experiments, higher initial soluble iron concentrations resulted in lower final pH values in the batch reactors. This was likely due to increased jarosite precipitation and sulphuric acid generation when the soluble iron concentration was higher, as shown in Equation 10:

$$3Fe_2(SO_4)_3 + 14H_2O \rightleftharpoons 2H_3OFe_3(SO_4)_2(OH)_6 + 5H_2SO_4 \qquad (10)$$
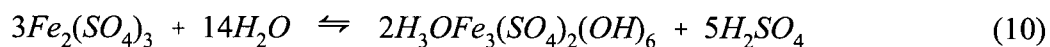
4.2.4 Summary

The average percent copper extractions achieved in the batch reactor experiments were 90% and 97% for the ore and concentrate respectively. The higher copper extractions achieved for the concentrate were most likely due to the fact that the concentrate experiments were allowed to proceed longer than the ore experiments; the ore experiments were terminated before copper extraction was complete. The average copper leach rates were found to be 7 mg/L/h for the ore and 55 mg/L/h for the concentrate, when the pulp density was 10%. The average acid consumptions were 20 kg/t and 160 kg/t for the ore and concentrate respectively. The batch reactor results verified the shake flask results concerning the effects of pulp density and initial soluble iron concentration. The batch stirred tank reactor experimental method was developed successfully for use in the continuous experiments. Sample calculations for copper extraction, copper leach rate and sulphate formation are given in Appendix A. Selected batch reactor data are given in Appendix C.
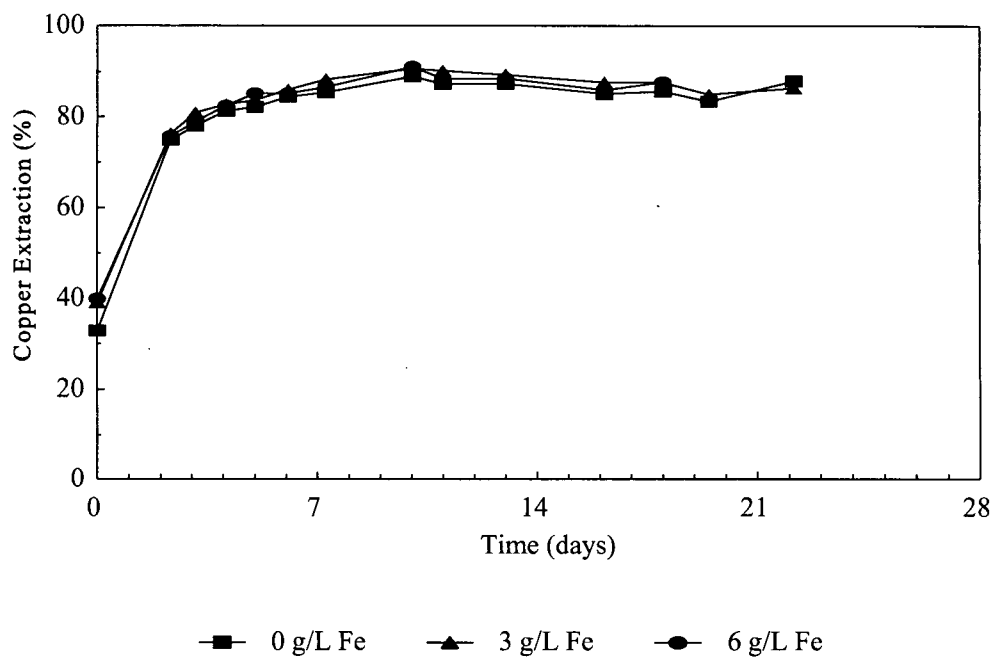
Figure 4.18 - Effect of Initial Soluble Iron Concentration on Copper Extraction from the Zaldívar Ore at 15% Pulp Density During Batch Reactor Tests in a 3 g/L Chloride Environment
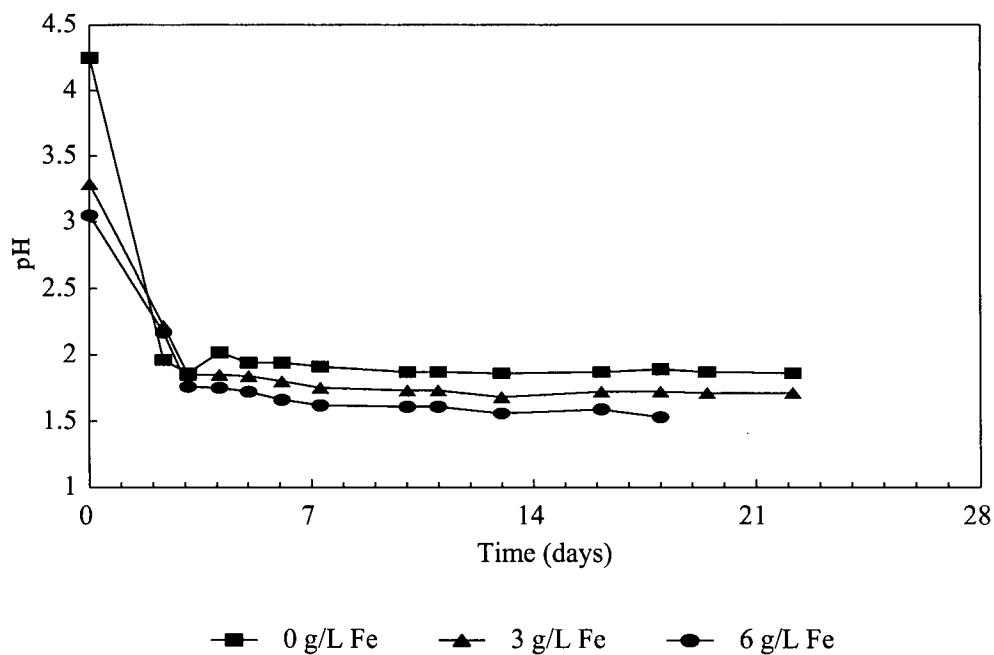
Figure 4.19 - Effect of Initial Soluble Iron Concentration on pH During the Batch Reactor Biological Leaching of the Zaldívar Ore at 15% Pulp Density in a 3 g/L Chloride Environment

4.3   Continuous Reactor Experiments

4.3.1   General Results and System Operation

A graph of soluble copper concentration versus time for the feed tank and the three stages of biological leaching for the ore run is given in Figure 4.20. A similar graph for the concentrate run is given in Figure 4.21. For approximately the first ten days of each experiment, the bacteria were grown in batch mode. After commencement of continuous mode operation, the copper level rose initially, partly because of excessive water loss due to evaporation. Water addition to compensate for evaporative losses was begun on approximately the twentieth day of each experiment. The amount of water added to the bioleach tanks was 40 mL water/L slurry for the ore run and 50 mL water/L slurry for the concentrate run. The evaporative losses were higher for the concentrate run because of the lower pulp density.

Foam was observed to form in all the bioleach tanks and solids were observed to coat the tank lids and tank walls above the liquid level. These solids were washed back into the slurry on a daily basis during the addition of water to compensate for evaporative losses. In addition to foaming, the accumulation of solids at the slurry surface may have been partly due to the presence of flotation chemicals in the concentrate slurry. In that case, the solids coating the tank lids and tank walls may have been sulphide-rich. As will be discussed in Section 4.3.4, this accumulation of sulphide-rich solids at the slurry surface was likely the cause of an accelerated short-circuiting effect observed in the concentrate continuous bioleaching system.

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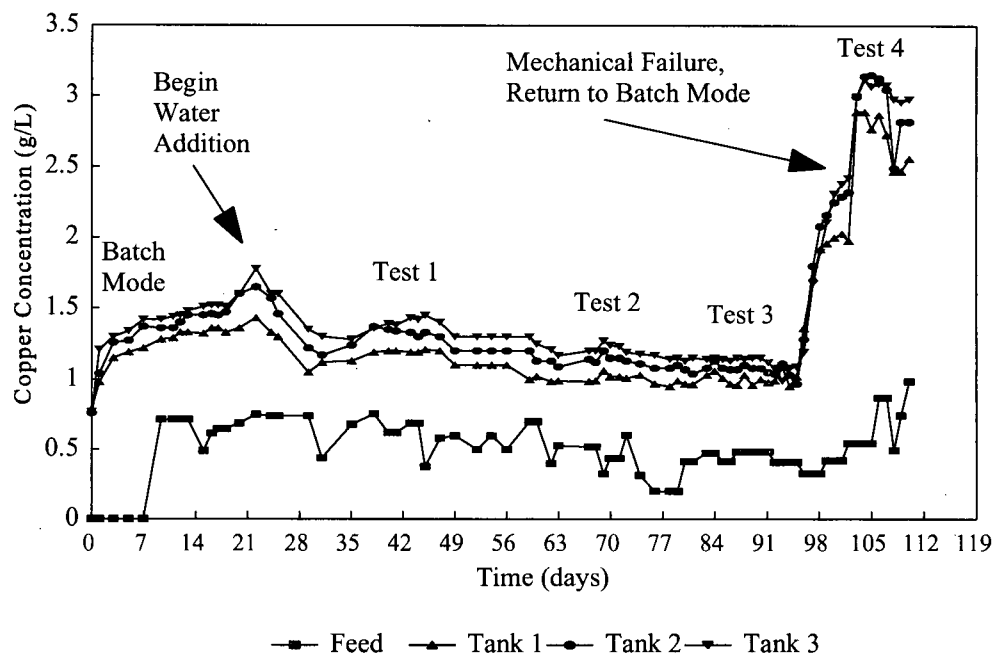
Figure 4.20 - Copper Concentration versus Time for the Ore Continuous Biological Leaching Run in a 3 g/L Chloride Environment
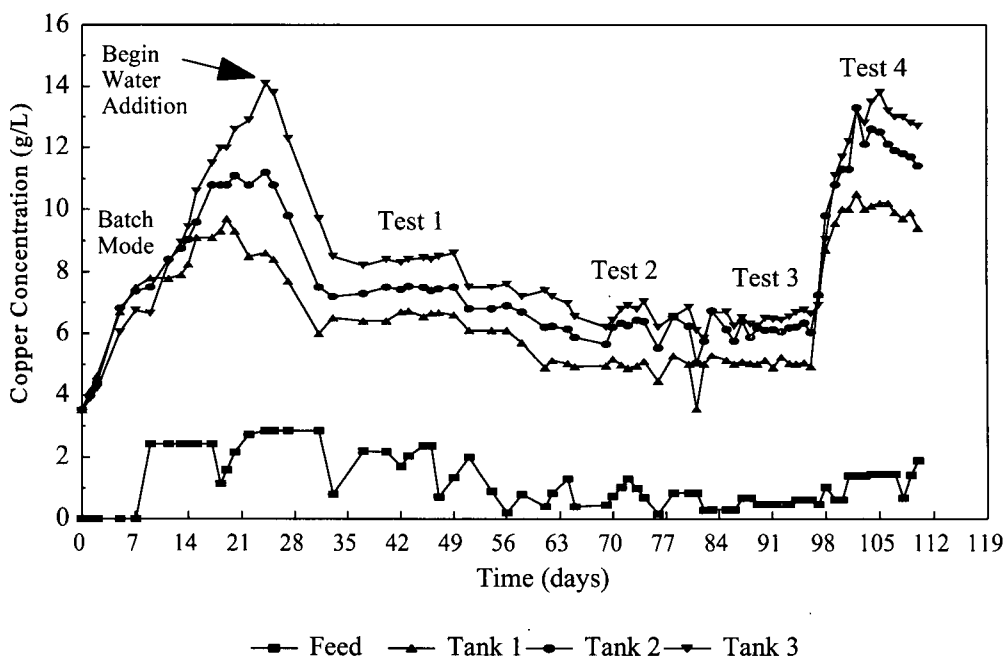
Figure 4.21 - Copper Concentration versus Time for the Concentrate Continuous Biological Leaching Run in a 3 g/L Chloride Environment

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During Test 4 of the ore run, mechanical failure caused simultaneous interruption of both the air supply and the acid supply to the bioleaching system. This resulted in a significant decrease in the activity of the bacterial culture. The feed pumps were turned off and the ore system was run in batch mode for several days. The bacterial culture recovered but unfortunately time constraints prevented the system from reaching steady state before completion of the experiment. However, as shown in Figure 4.20, before the mechanical failure occurred, a doubling of the pulp density resulted in a doubling of the copper concentration in the bioleaching tanks. This indicated that the system would likely have successfully reached steady state in the absence of mechanical problems.

It was difficult to continuously achieve the target flowrates for each test during the continuous bioleaching experiments, because of solid feeding problems. The feed tube often plugged during the night, particularly with the ore feed slurry since the P_{80} particle size was larger than that of the concentrate feed slurry. McElroy and Bruynesteyn (1978) also reported difficulties with solids feeding in their continuous bioleaching system. The actual flowrate profiles for the ore and concentrate runs are given in Figure 4.22 and Figure 4.23 respectively.

Acrylic impellers were used in the first stage of both continuous bioleaching systems because titanium impellers of the correct diameter were difficult to obtain. Titanium impellers were used in the subsequent stages of the continuous bioleaching system. The blades of the acrylic impellers used in the ore system wore away over time because the P_{80} particle size of the ore was about double that of the concentrate, making the ore more abrasive than the concentrate. In fact, the impeller blades used in the concentrate system wore only slightly during the four month experiment, while the ore blades wore away completely and required replacement. This decreased the amount of solids suspension in the first stage of the ore bioleaching system and reduced the amount of solids flowing

71
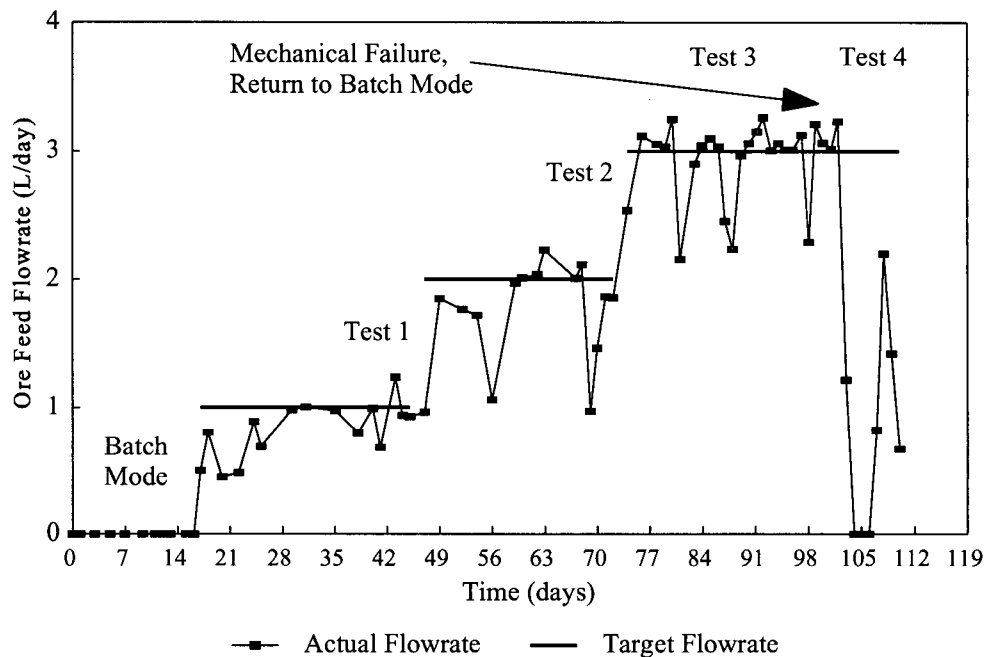
Figure 4.22 - Actual Flowrate for the Ore Continuous Biological Leaching Run in a 3 g/L
Chloride Environment
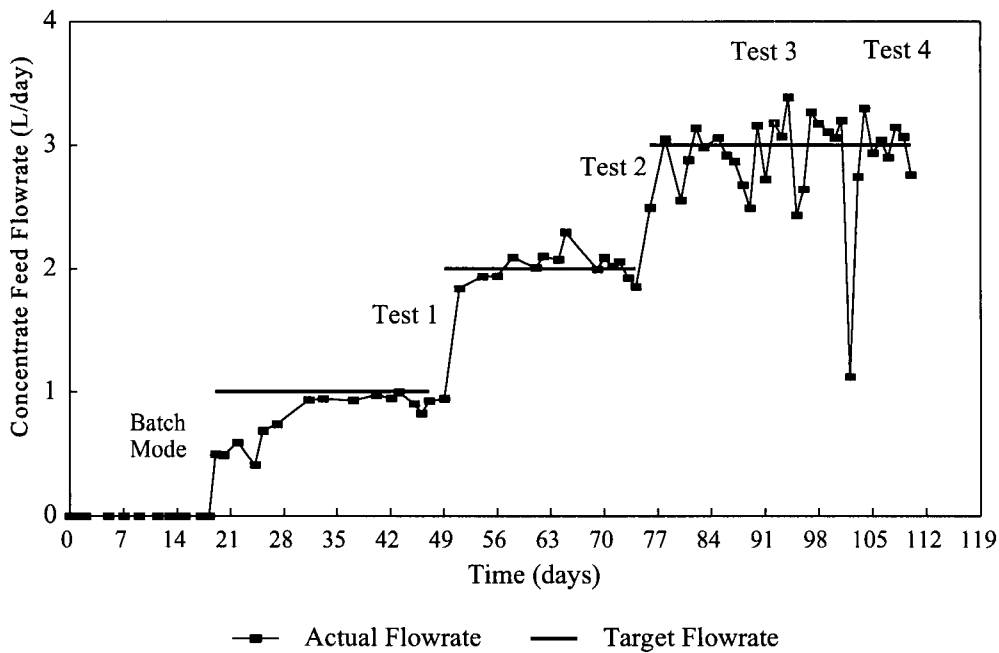
Figure 4.23 - Actual Flowrate for the Concentrate Continuous Biological Leaching Run in a 3
g/L Chloride Environment

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from the first to the second ore bioleach stage. Acrylic was found to be an unsatisfactory material for production of continuous bioleaching system impellers. Titanium impellers are recommended for future use.

4.3.2 Results at Steady State

Summaries of the continuous biological leaching results for the Zaldívar ore and concentrate are given in Table 4.1. For each steady state condition, the table gives the pulp density, total system residence time, redox potential in the third bioleach stage, copper leach rate in the first bioleach stage, cumulative sulphate formation in the third bioleach stage and cumulative copper extraction in the third bioleach stage. As mentioned above, due to mechanical failure near the end of the ore run, steady state conditions were not achieved for Test 4. Therefore, no results are given in Table 4.1 for continuous biological leaching of the ore at a pulp density of 20% and a total residence time of $2\frac{2}{3}$ days. The values for copper extraction and sulphate formation given in Table 4.1 are average values for three slurry samples taken during the steady state period. Three samples allowed confirmation that steady state had been achieved. The values for Eh and copper leach rate given in Table 4.1 are averages achieved during each steady state period. Sample calculations for copper extraction, copper leach rate and sulphate formation are given in Appendix A. Daily monitoring data for both the ore and concentrate continuous bioleaching systems are given in Appendix D.

The amounts of sulphuric acid required to maintain the pH below 2 in the first stage of continuous bioleaching were found to be approximately 50 kg per tonne of ore and 200 kg per tonne of concentrate. The net sulphuric acid requirements on an industrial scale would be lower than these values because some sulphuric acid would be recycled from the SX-EW circuit in an industrial process. This is discussed further in Section 4.4.

Table 4.1 - Continuous Biological Leaching Steady State Results

Variables	Test 1	Test 2	Test 3	Test 4
Zaldívar Ore				
pulp density (%)	10	10	10	20
total residence time (days)	8	4	2⅔	2⅔
stage 3 Eh (mV, SHE)	820	800	800	no results
stage 1 copper leach rate (mg/L/h)	6	10	16	no results
cumulative sulphate formation (%)	63	47	50	no results
cumulative copper extraction (%)	84	83	82	no results
Zaldívar Concentrate				
pulp density (%)	5	5	5	10
total residence time (days)	8	4	2⅔	2⅔
stage 3 Eh (mV, SHE)	840	820	800	770
stage 1 copper leach rate (mg/L/h)	50	83	120	250
cumulative sulphate formation (%)	71	28	7	0
cumulative copper extraction (%)	91	89	87	83

Profiles of the copper leach rate in the first bioleach stage are given in Figures 4.24 and 4.25 for the ore and concentrate runs respectively. As these figures and the data in Table 4.1 show, the copper leach rate was found to increase as the residence time decreased, while the overall copper extraction remained relatively constant. In order to maintain the same copper extraction with less residence time, the bacterial leach rate increased proportionately. Similarly, the leach rate was observed to double when the pulp density was doubled, while the copper extraction remained approximately constant. These results confirmed that chalcocite was readily amenable to oxidation by bacterial activity. In order to achieve copper extractions of 82% and 83% for the ore and concentrate respectively at 10% pulp density, a residence time of no more than 2⅔ days was required.
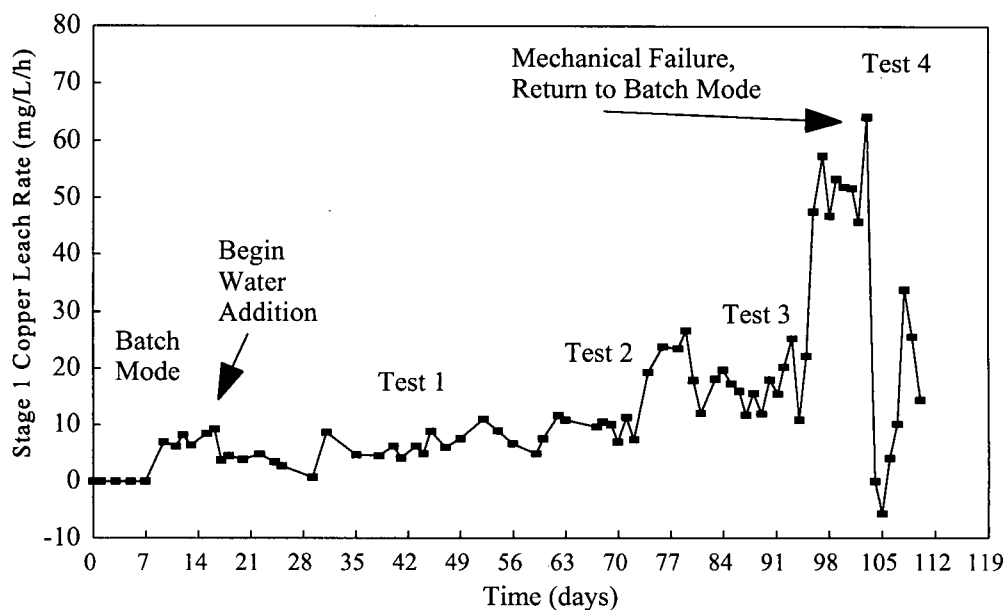
Figure 4.24 -   Copper Leach Rate versus Time in the First Bioleach Stage of the Ore Continuous
Biological Leaching Run in a 3 g/L Chloride Environment
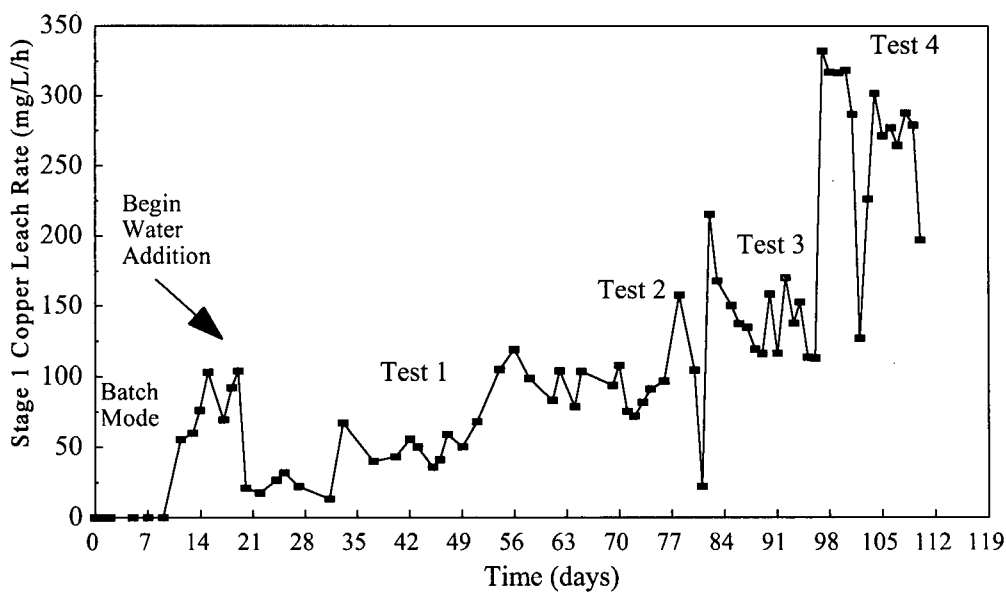
Figure 4.25 -   Copper Leach Rate versus Time in the First Bioleach Stage of the Concentrate
Continuous Biological Leaching Run in a 3 g/L Chloride Environment

The stage 1 copper leach rates given in Table 4.1 were calculated based on the difference between the dissolved copper concentrations in the first bioleach stage and the feed tank. A check of these results was carried out as shown in Table 4.2. The stage 1 copper leach rate was used to calculate a theoretical copper extraction, which was then compared with the actual copper extraction observed in the first bioleach stage. This theoretical copper extraction is defined as the copper extraction that would have occurred at the stage 1 copper leach rate, for a given feed material, pulp density and residence time. A sample calculation of the theoretical copper extraction is given in Appendix A.

Table 4.2 - Check of Stage 1 Copper Leach Rates

Test No. (Residence Time, Pulp Density)	Stage 1 Copper Leach Rate (mg/L/h)	Theoretical Stage 1 Copper Extraction (%)	Actual Stage 1 Copper Extraction (%)
Zaldívar Ore			
Test 1 (8 days, 10%)	6	36	30
Test 2 (4 days, 10%)	10	30	47
Test 3 (2⅔ days, 10%)	16	32	40
Zaldívar Concentrate			
Test 1 (8 days, 5%)	50	55	49
Test 2 (4 days, 5%)	83	45	73
Test 3 (2⅔ days, 5%)	120	44	72
Test 4 (2⅔ days, 10%)	250	45	75

As shown in Table 4.2, the theoretical copper extraction was found to be in good agreement with the actual stage 1 copper extraction for the tests on the ore, and for Test 1 on the concentrate when the system residence time was 8 days. However, the theoretical copper extraction was found to be much

lower than the actual stage 1 copper extraction for subsequent tests on the concentrate. As will be discussed in Section 4.3.4, this was likely due to an accelerated short-circuiting effect observed in the first stage of the concentrate continuous bioleaching system when the residence time was 4 days or 2⅔ days. The accelerated short-circuiting effect resulted in elevated actual copper extractions in the first bioleach stage. When this effect was not observed, the stage 1 copper leach rates were in good agreement with the actual copper extractions observed in the first bioleach stage.

Percent sulphate formation was found to be lower than percent copper extraction under all test conditions, as shown in Table 4.1. In addition, as the residence time decreased, copper extraction remained relatively constant while sulphate formation tended to decrease. As suggested by the results of the shake flask and batch reactor experiments, incomplete sulphate formation was likely due to incomplete oxidation of pyrite or elemental sulphur. In addition, according to the chemistry, it is possible to have the majority of the copper extracted and still have much of the sulphide unoxidized. Chalcocite oxidation involves the intermediate production of covellite:

$$Cu_2S + 0.5O_2 + H_2SO_4 \rightarrow CuSO_4 + CuS + H_2O \qquad (3)$$
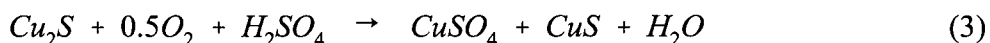
Covellite is then oxidized at a slower rate:

$$CuS + 2O_2 \rightarrow CuSO_4 \qquad (4)$$

If most of the chalcocite is converted to covellite before any covellite oxidation occurs, then about half of the sulphide copper in a chalcocitic ore or concentrate is extracted before any sulphide

oxidation occurs. Based on the chemistry, it is possible to estimate the sulphate formation for a given copper extraction; a sample calculation is given in Appendix A. For example, the results of concentrate Test 4 given in Table 4.1, showed that there was 83% copper extraction but no sulphate formation. It was estimated based on the chemistry that in order to achieve 83% copper extraction, 22% sulphate formation would occur, even if there was no pyrite oxidation. Since no sulphate formation was observed, the covellite-sulphide must have been oxidized by the indirect mechanism to elemental sulphur, as shown in Equation 8:

$$CuS + Fe_2(SO_4)_3 \rightarrow CuSO_4 + S^o + 2FeSO_4 \tag{8}$$
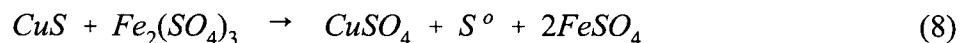
Since Table 4.1 gives only the cumulative stage 3 copper extraction and sulphate formation, charts are included showing the copper extraction and sulphate formation by stage of the continuous biological leaching systems for each steady state condition tested. Figure 4.26, Figure 4.27 and Figure 4.28 give this data for the three steady state conditions tested in the ore run. Similarly, Figure 4.29, Figure 4.30, Figure 4.31 and Figure 4.32 are included for the concentrate run. In general, the largest proportion of the copper extraction occurred in the first stage of biological leaching. Correspondingly, the highest copper leach rate was observed in the first bioleach stage. For most test conditions, some sulphate formation occurred in the feed tank, and additional sulphate formation was observed in all three bioleach stages. An interesting phenomenon was observed in the concentrate system under Test 2, 3 and 4 conditions. Lower values for copper extraction and sulphate formation were achieved in the second bioleach stage than in the first bioleach stage. This was most likely due to an accelerated short-circuiting effect in the first bioleach stage, as will be discussed in Section 4.3.4.
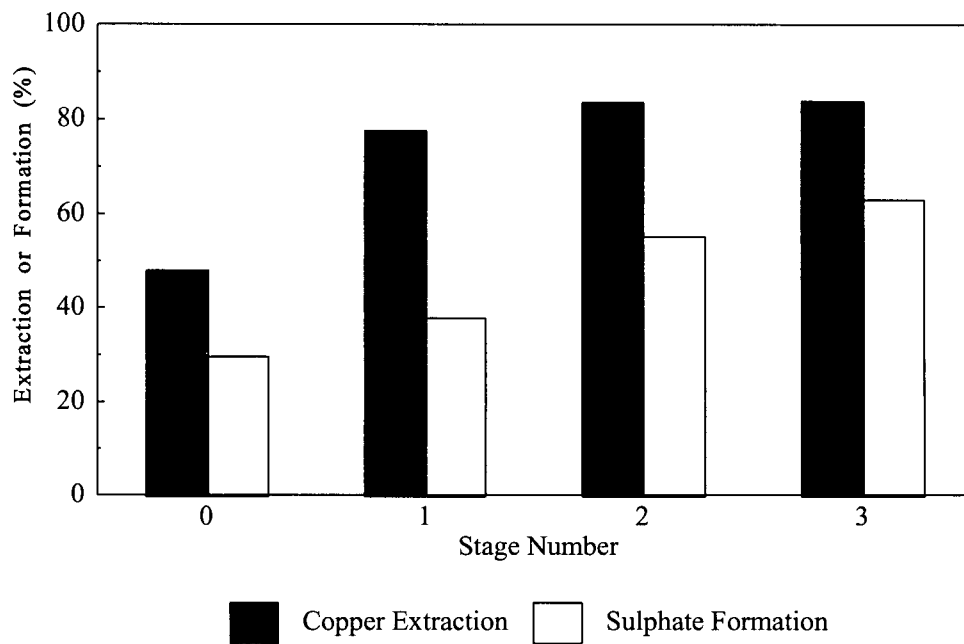
Figure 4.26 - Copper Extraction and Sulphate Formation for the Ore Continuous Biological Leaching Run in a 3 g/L Chloride Environment with a Total Residence Time of 8 Days and a Pulp Density of 10%
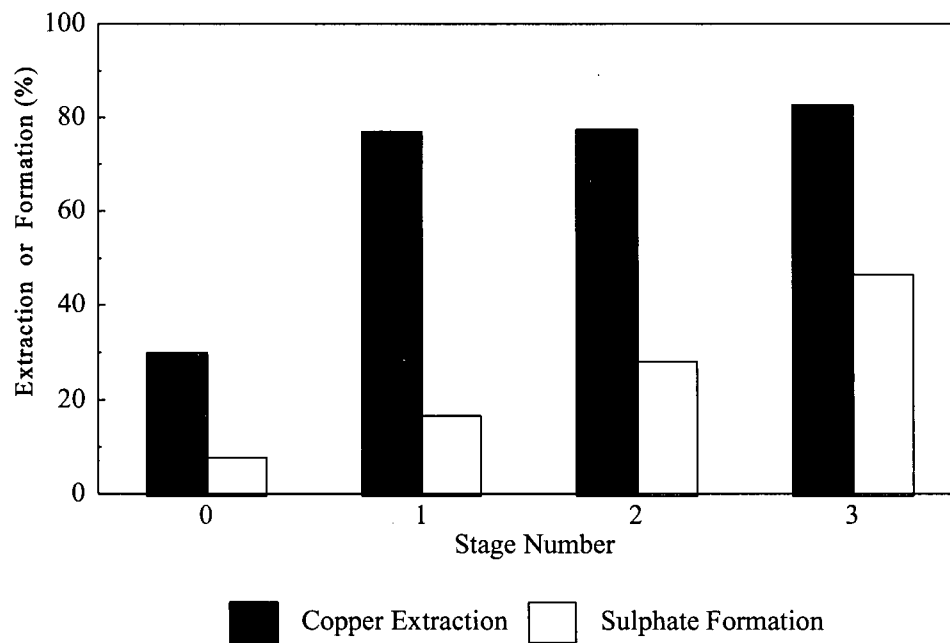
Figure 4.27 - Copper Extraction and Sulphate Formation for the Ore Continuous Biological Leaching Run in a 3 g/L Chloride Environment with a Total Residence Time of 4 Days and a Pulp Density of 10%
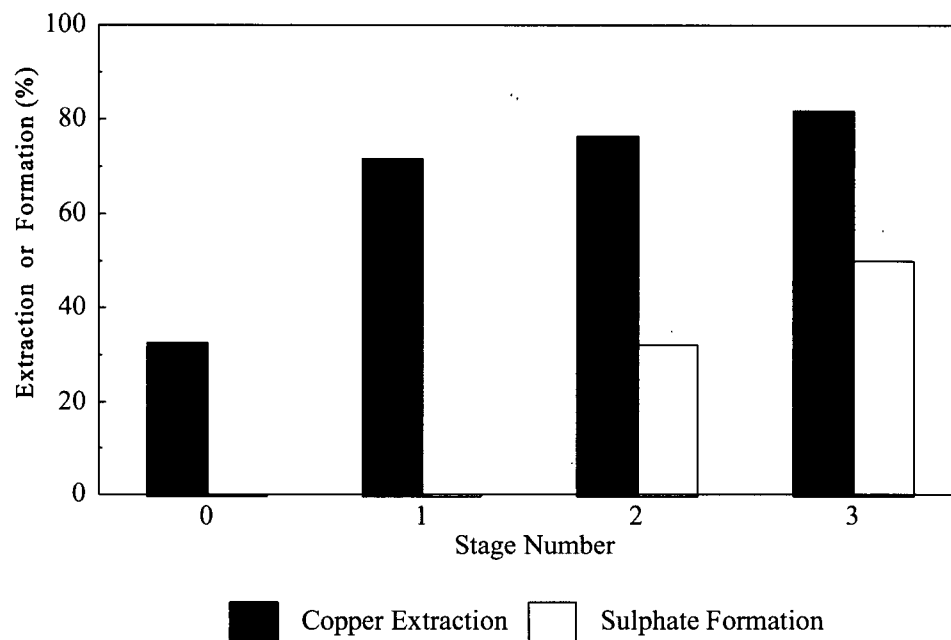
Figure 4.28 - Copper Extraction and Sulphate Formation for the Ore Continuous Biological
Leaching Run in a 3 g/L Chloride Environment with a Total Residence Time of
2⅔ Days and a Pulp Density of 10%
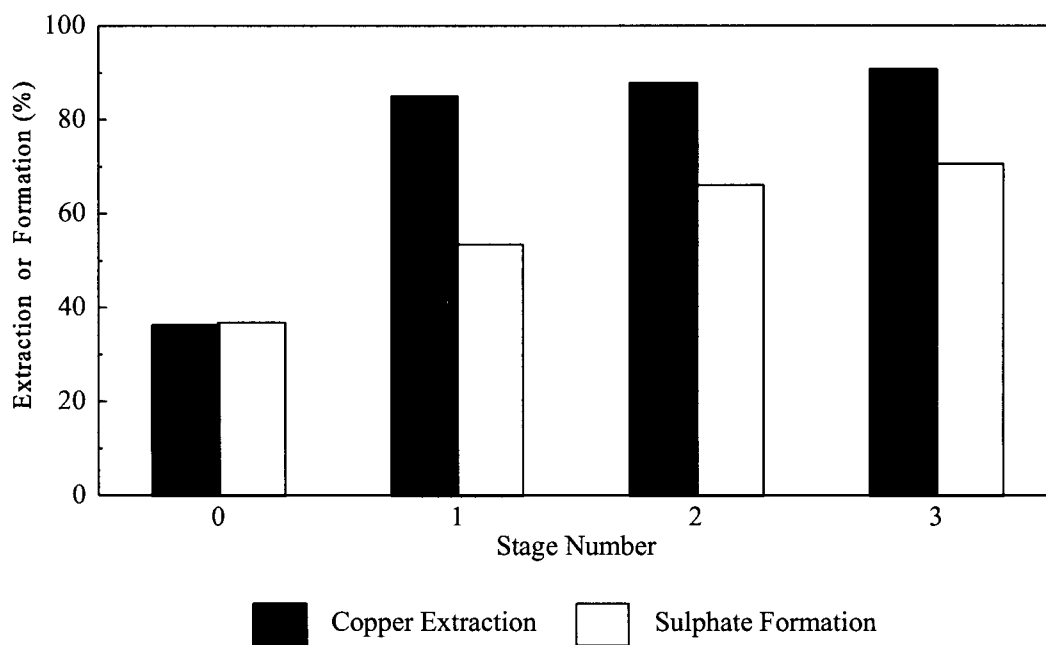
Figure 4.29 - Copper Extraction and Sulphate Formation for the Concentrate Continuous
Biological Leaching Run in a 3 g/L Chloride Environment with a Total Residence
Time of 8 Days and a Pulp Density of 5%
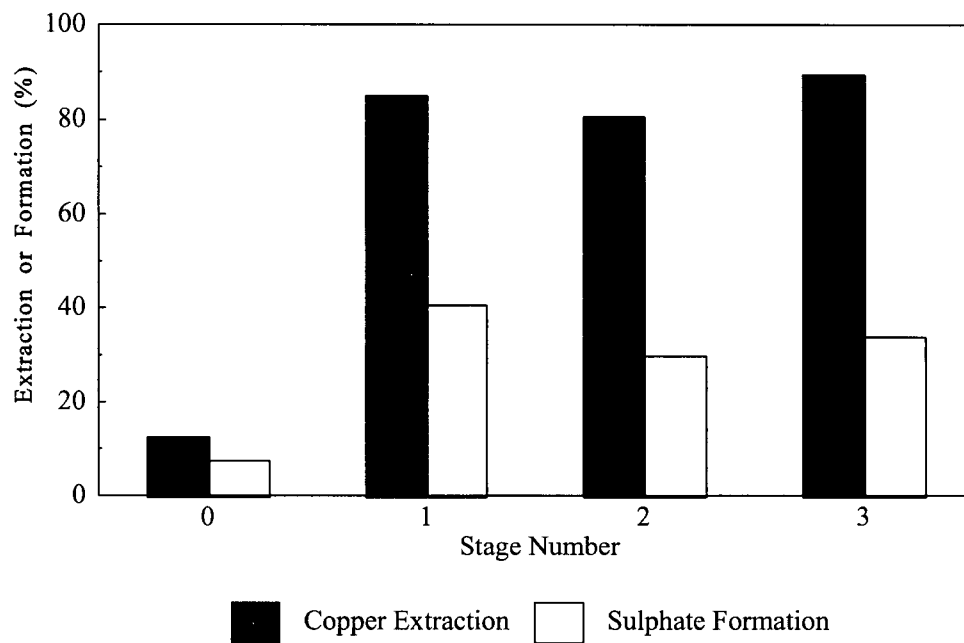
Figure 4.30 -   Copper Extraction and Sulphate Formation for the Concentrate Continuous
Biological Leaching Run in a 3 g/L Chloride Environment with a Total Residence
Time of 4 Days and a Pulp Density of 5%
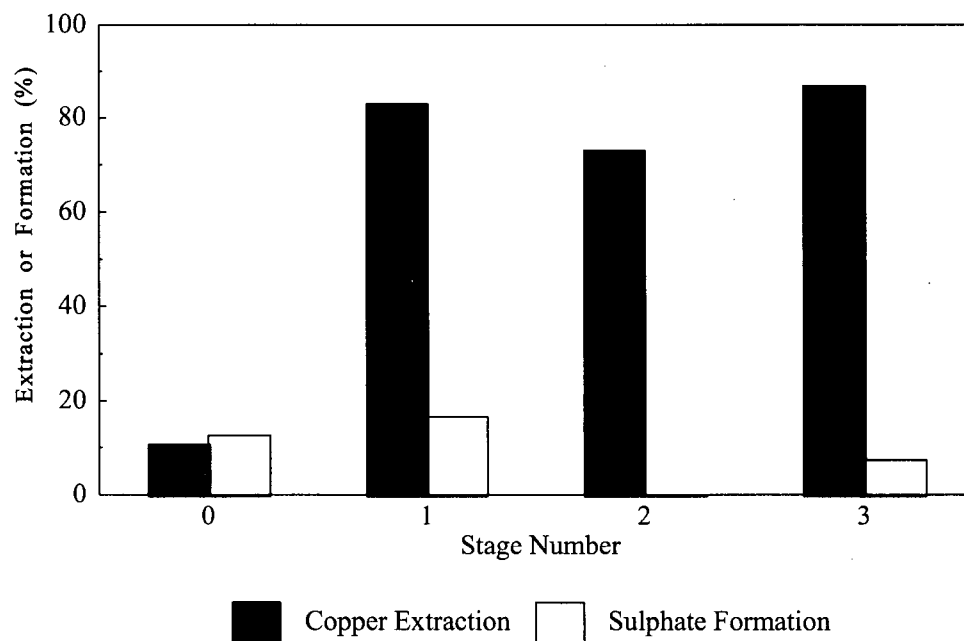
Figure 4.31 -   Copper Extraction and Sulphate Formation for the Concentrate Continuous
Biological Leaching Run in a 3 g/L Chloride Environment with a Total Residence
Time of 2⅔ Days and a Pulp Density of 5%
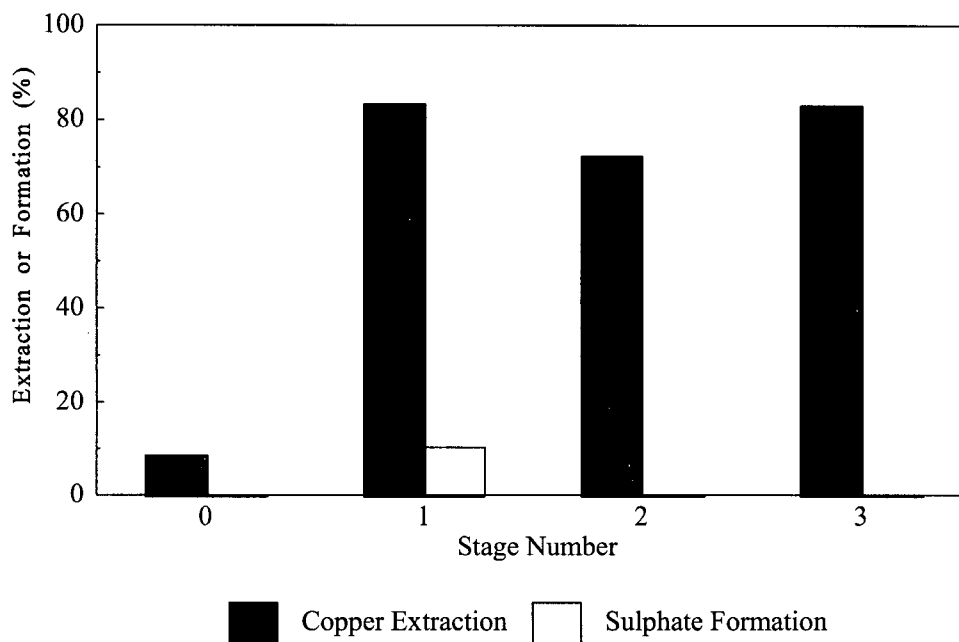
Figure 4.32 - Copper Extraction and Sulphate Formation for the Concentrate Continuous
Biological Leaching Run in a 3 g/L Chloride Environment with a Total Residence
Time of 2⅔ Days and a Pulp Density of 10%

4.3.3   Feed Tank Results

Copper extraction and sulphate formation were observed to occur in both the ore and concentrate

feed tanks, particularly when the residence time was 8 days. Since the feed tank volume was 8 L and

the total system volume was 8 L, the feed tank was refilled with fresh slurry every 8 days for a

system residence time of 8 days, every 4 days for a system residence time of 4 days, and so on. The

longer the residence time, the more copper extraction and sulphate formation occurred in the feed

tank. In order to better understand the reactions that occurred in the feed tank, copper extraction and

sulphate formation in the feed tank were plotted versus the time since the feed tank was refilled. As

shown in Figure 4.33 and Figure 4.34, the copper extraction and sulphate formation in the ore feed

tank increased as the age of the feed tank slurry increased. A similar correlation was observed in the

concentrate feed tank, as shown in Figure 4.35 and Figure 4.36.
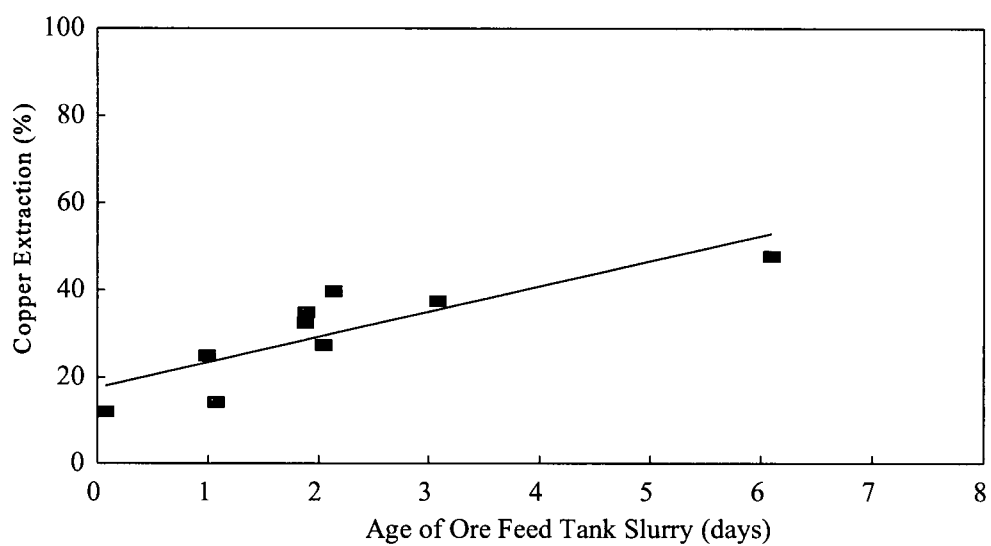
Figure 4.33 -   Copper Extraction in the Ore Feed Tank versus the Age of the Feed Tank Slurry
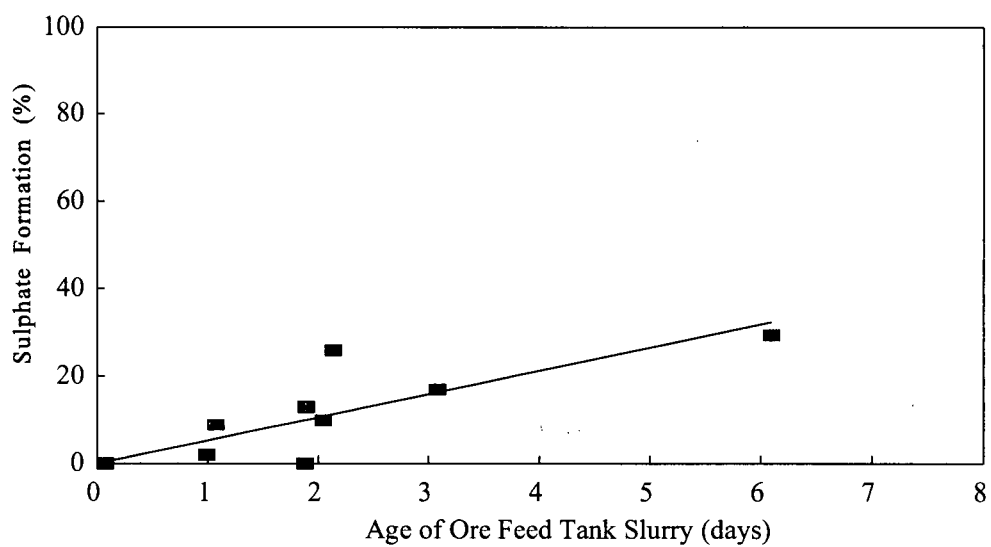
Figure 4.34 -   Sulphate Formation in the Ore Feed Tank versus the Age of the Feed Tank Slurry
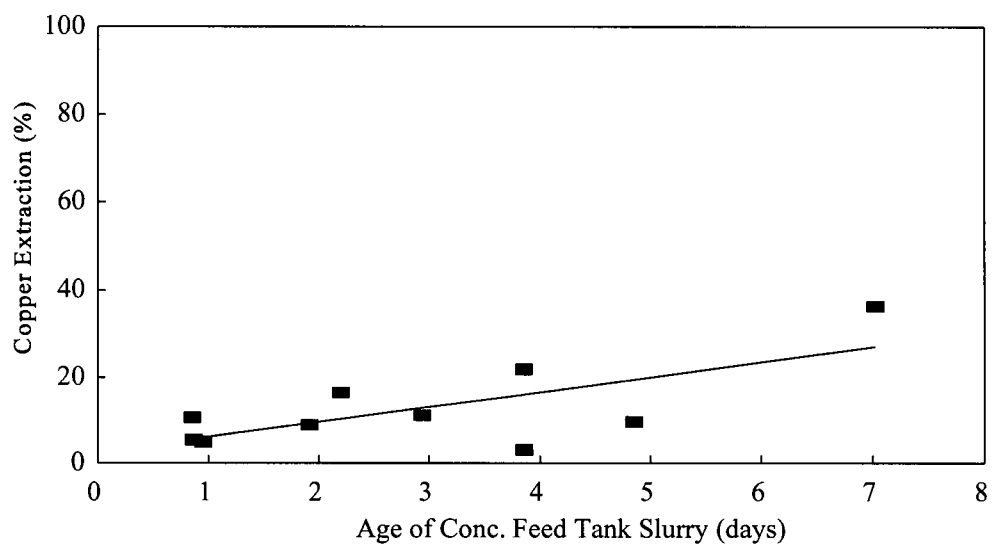
Figure 4.35 -   Copper Extraction in the Concentrate Feed Tank versus the Age of the Feed Tank Slurry
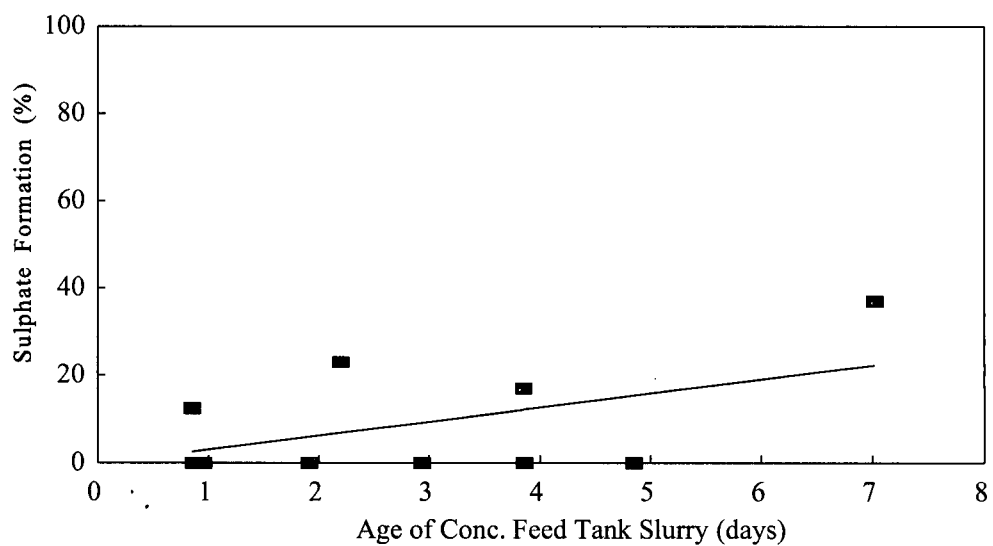
Figure 4.36 -   Sulphate Formation in the Concentrate Feed Tank versus the Age of the Feed Tank Slurry

Part of the copper extraction observed in the feed tank is attributable to dissolution of the copper mineral brochantite ($Cu_4(SO_4)(OH)_6$):

$$Cu_4(SO_4)(OH)_6 + 3H_2SO_4 \rightarrow 4CuSO_4 + 6H_2O \qquad (13)$$
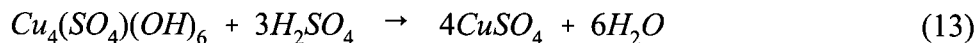
If it is assumed that all the acid soluble copper in the feed materials was brochantite and that all the brochantite dissolved in the feed tanks, then the resulting feed tank copper extractions would be 32% for the ore and 9% for the concentrate. The actual feed tank copper extractions were close to these values when the residence time was 4 days or 2⅔ days, as shown in Figures 4.27 and 4.28 for the ore and Figures 4.30, 4.31 and 4.32 for the concentrate. However, the feed tank copper extractions were significantly higher than the copper extractions based on dissolution of brochantite when the residence time was 8 days, as shown in Figure 4.26 for the ore and Figure 4.29 for the concentrate.

The proportion of the copper extraction greater than that expected solely based on brochantite dissolution was most likely due to the conversion of chalcocite to covellite. Chalcocite oxidation occurred either by direct or indirect oxidation, as shown in Equations 3 and 7 respectively:

$$Cu_2S + 0.5O_2 + H_2SO_4 \rightarrow CuSO_4 + CuS + H_2O \qquad (3)$$

$$Cu_2S + Fe_2(SO_4)_3 \rightarrow CuS + CuSO_4 + 2FeSO_4 \qquad (7)$$
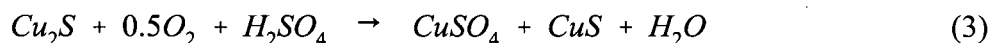
It is unlikely that there was any significant covellite oxidation in the feed tank because covellite oxidation is kinetically slower than chalcocite oxidation (Beck, 1977). Even if there was some covellite oxidation, it would still not explain the observed sulphate formation in the feed tanks. Since pyrite was the other main sulphide mineral, the sulfate formation was likely attributable to

pyrite oxidation:

$$2FeS_2 + 7.5O_2 + H_2O \rightarrow Fe_2(SO_4)_3 + H_2SO_4 \qquad (5)$$
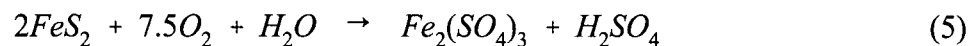
Figure 4.37 shows the proportion of overall sulphate formation that is attributable chalcocite-sulphide and pyrite-sulphide by stage of the ore continuous bioleaching system for an 8 day residence time and a pulp density of 10%.
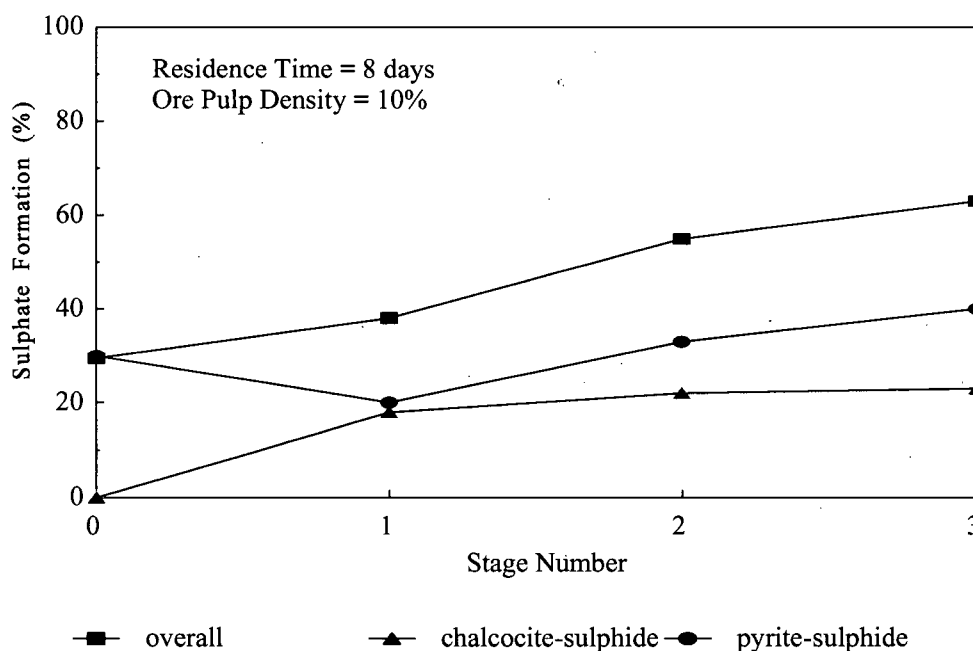
Figure 4.37 - Sulphate Formation and the Proportion of Sulphate Formation Attributable to Chalcocite-Sulphide and Pyrite-Sulphide by Stage of the Ore Continuous Bioleaching System for Test 1

The observed copper extraction and sulphate formation results, which occurred primarily when the residence time was 8 days, were likely due to the development of a bacterial population in the feed

tanks. Bacterial populations are associated with most ores, but they only become measurably active under certain conditions. Although no air was supplied to the feed tanks, some dissolved oxygen would have been available for biological activity due to the vigourous stirring required to maintain adequate solids suspension.

The average and range of pH and Eh observed in the ore and concentrate feed tanks are given in Table 4.3. Although pH 2.9 to 4.5 is out of the optimal range for *T. ferrooxidans*, these bacteria are still active in this pH range. There are also other mesophilic iron-oxidizing bacteria that are active in this pH range such as *T. thiooxidans*, *L. ferrooxidans* and the *Metallogenium* genus. It is possible that all of these bacteria were associated with the Zaldívar ore. Beck (1977) found that biologically catalysed chalcocite oxidation occurred at pH values less than about 4.6.

Table 4.3 - Average and Range of pH and Eh in the Ore and Concentrate Feed Tanks

Feed Material	pH	Eh (mV, SHE)
Zaldívar Ore	3.5 (2.9 to 3.8)	498 (465 to 544)
Zaldívar Concentrate	3.8 (3.6 to 4.5)	496 (446 to 563)

The pH and Eh in the feed tanks were both observed to increase with time after the tank was refilled with fresh feed slurry. One possible reason for the initially low feed tank pH was the acidity of the deionized water used for dilution of the feed materials. Also, acid may have been generated in the feed tanks by precipitation of ferric hydroxides; ferric sulphates may have been present in the feed materials due to partial oxidation during storage. Another possible source of acid may have been

the oxidation of other iron sulphide minerals such as marcasite. The acid in the feed tank was then available for the dissolution of brochantite:

$$Cu_4(SO_4)(OH)_6 + 3H_2SO_4 \rightarrow 4CuSO_4 + 6H_2O \tag{13}$$
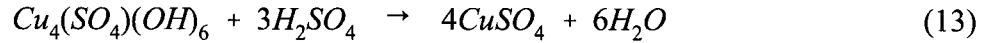
The observed pH increase with time in the feed tanks was most likely due to chalcocite oxidation, as shown in Equation 3 or Equation 7:

$$Cu_2S + 0.5O_2 + H_2SO_4 \rightarrow CuSO_4 + CuS + H_2O \tag{3}$$

$$Cu_2S + Fe_2(SO_4)_3 \rightarrow CuS + CuSO_4 + 2FeSO_4 \tag{7}$$
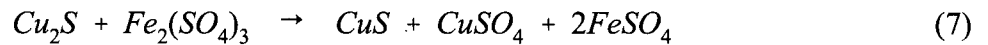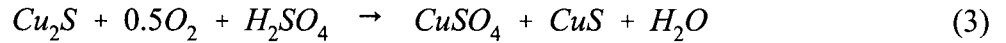
Ferric iron may have been available from the oxidation of pyrite, shown in Equation 5, or the oxidation of ferrous iron, shown in Equation 6:

$$2FeS_2 + 7.5O_2 + H_2O \rightarrow Fe_2(SO_4)_3 + H_2SO_4 \tag{5}$$

$$2FeSO_4 + 0.5O_2 + H_2SO_4 \rightarrow Fe_2(SO_4)_3 + H_2O \tag{6}$$
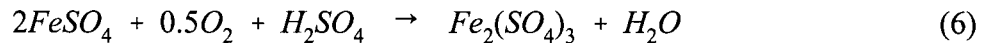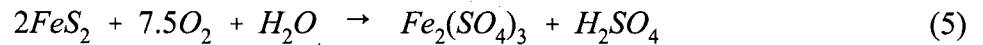
Pyrite oxidation also generates acid, which would then be available for brochantite dissolution (Equation 13), chalcocite oxidation (Equation 3) or ferrous iron oxidation (Equation 6). Jarosite or iron hydroxides may have precipitated in the feed tank to generate acid as shown below:

$$3Fe_2(SO_4)_3 + 14H_2O \rightleftharpoons 2H_3OFe_3(SO_4)_2(OH)_6 + 5H_2SO_4 \qquad (10)$$

$$Fe_2(SO_4)_3 + 6H_2O \rightleftharpoons 2Fe(OH)_3 + 3H_2SO_4 \qquad (11)$$
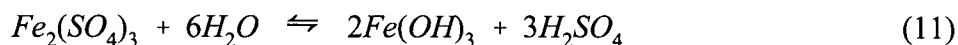
The observed increase in Eh with time may have been due to the generation of ferric iron by pyrite oxidation (Equation 5) or the oxidation of ferrous iron (Equation 6). Ferrous iron oxidation occurs spontaneously but the kinetics are much faster when a bacterial catalyst is available, as may have been the case in the feed tank when the residence time was 8 days.

4.3.4 Accelerated Short-Circuiting Effects

As illustrated in Figures 4.30, 4.31 and 4.32, a lower copper extraction was observed in the second bioleach stage of the concentrate continuous bioleaching system than in the first bioleach stage, when the residence time was 4 days or 2⅔ days. This was thought to have been caused by an accelerated short-circuiting effect in the first bioleach stage, which was due to foaming and the presence of flotation chemicals in the concentrate feed. The flotation effect in the first bioleach stage created a sulphide-rich overflow to the second bioleach stage, reducing the observed copper extraction in the second bioleach stage. Pinches et al. (1976) reported a similar mineral flotation effect during the batch reactor biological leaching of a chalcopyrite concentrate.

Copper extractions in the third bioleach stage were higher than in the second bioleach stage, as shown in Figures 4.30, 4.31 and 4.32. This suggests that the accelerated short-circuiting effect did not occur in bioleach stage 2. In other words, the flotation effect did not occur to the same extent, and consequently there was improved mixing of the slurry in the second bioleach stage. In the

second bioleach stage, there had been more time for flotation chemicals to decompose. Also, bacteria excrete wetting agents that tend to minimize the effect of flotation chemicals. Either of these factors could have reduced the flotation effect in the second bioleach stage.

Therefore, the accelerated short-circuiting effect observed in this study was likely not due to poor tank design but to a flotation effect caused by residual flotation chemicals in the concentrate. This led to a preferential transfer of a sulphide-rich slurry from the first to the second bioleach stage. These results suggest that four stages of bioleaching would probably be required to achieve adequate copper extraction in a full scale plant. Also, an outlet weir may help to retard the preferential transfer of a sulphide-rich froth from one stage to the next. In a larger scale system, transfer of slurry between stages could be accomplished by pumping from the mid-height of the tank, instead of by gravity overflow.

4.3.5   Ore versus Concentrate Continuous Biological Leaching

For a pulp density of 10% and a residence time of 2⅔ days, the ore and concentrate steady state copper extractions were found to be 82% and 83% respectively. It is important to note, however, that some copper loss results from flotation. Therefore, the overall copper recovery for the concentrate was actually somewhat less than 83%, depending on the efficiency of the flotation process. A comparison of the costs of ore versus concentrate continuous biological leaching is critical to the selection of one feed material over the other for use in a larger scale operation. Continuous biological leaching of the ore would require larger bioleach tanks due to the presence of gangue minerals in the ore. However, as mentioned above, higher overall copper recoveries would likely be achieved in an ore system. One advantage of concentrate continuous biological leaching is a smaller plant size, since most of the gangue minerals are rejected during flotation.

Also, net acid consumption is typically lower for concentrates than for ores since concentrates contain a much lower proportion of gangue minerals. The main disadvantage of concentrate continuous biological leaching is copper loss during flotation.

4.3.6 Summary

A flowsheet of the continuous bioleaching system is given in Figure 4.38. A mass balance over the system is given to summarize the results achieved at steady state during the continuous bioleaching of the Zaldívar concentrate, for a residence time of 2⅔ days and a pulp density of 10%. The oxygen consumption was determined to be 0.024 kg/day for a concentrate feed rate of 0.3 kg/day and an ultimate copper extraction of 83%. For the ore, the oxygen consumption was determined to be 0.003 kg/day for an ultimate copper extraction of 82% and a feed rate of 0.3 kg/day. This feed rate corresponds to a residence time of 2⅔ days and a pulp density of 10%. Oxygen consuming reactions were the oxidation of chalcocite, covellite and pyrite as shown in Equations 3, 4 and 5 respectively:

$$Cu_2S + 0.5O_2 + H_2SO_4 \rightarrow CuSO_4 + CuS + H_2O \tag{3}$$

$$CuS + 2O_2 \rightarrow CuSO_4 \tag{4}$$

$$2FeS_2 + 7.5O_2 + H_2O \rightarrow Fe_2(SO_4)_3 + H_2SO_4 \tag{5}$$
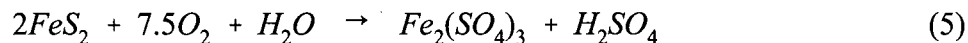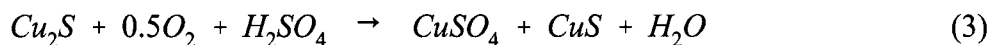
The gross consumption of sulphuric acid was found to be 200 kg/t for continuous bioleaching of the concentrate, or 0.06 kg/day for a concentrate feed rate of 0.3 kg/day and according to the conditions given in Figure 4.38. The Zaldívar concentrate was found to be a net acid generator, as will be discussed in Section 4.4.

concentrate (kg/day)	copper (kg/day)
0.300	0.052

Feed Tank

oxygen consumption (kg/day)	gross acid consumption (kg/day)	net acid generation (kg/day)
0.024	0.060	0.006

Continuous Stirred Tank Biological Leaching

copper leach rate = 250 mg/L/h

copper extraction (%)	copper in solution (kg/day)	copper in solids (kg/day)
8	0.005	0.047
83	0.043	0.009

Product to SX-EW
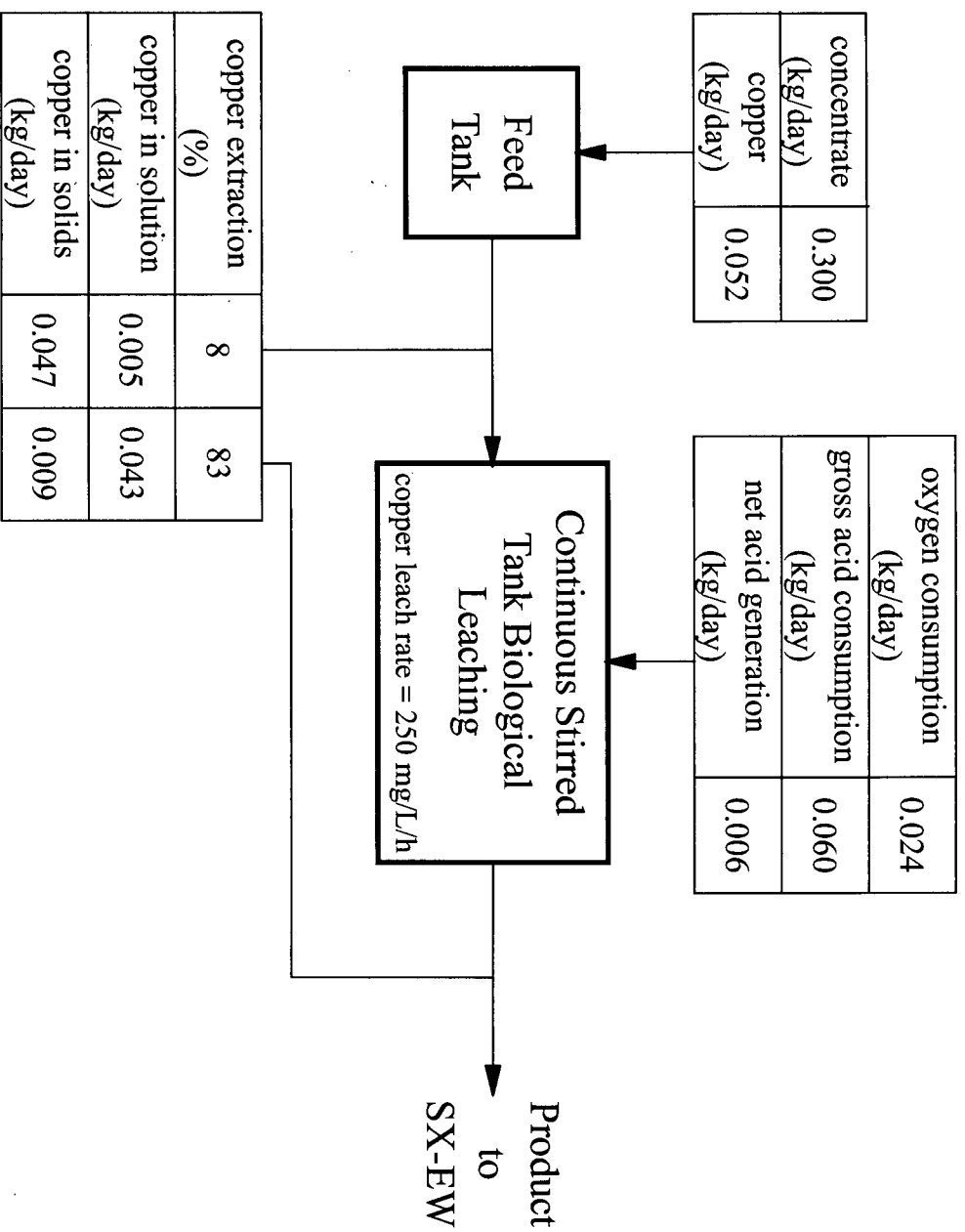
Figure 4.38 - Concentrate Continuous Biological Leaching Mass Balance at Steady State with a Residence Time of 2⅔ Days and a 10% Pulp Density

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The continuous biological leaching of both the Zaldívar chalcocite ore and concentrate in a 3 g/L chloride environment was successfully demonstrated. The bacteria were able to adapt to progressively lower system residence times and higher pulp densities. At the lowest residence time achieved of 2⅔ days with a pulp density of 10%, the steady state copper extraction was 82% for the ore and 83% for the concentrate. These copper extraction results are higher than the copper extractions estimated for the heap leaching operation at the Zaldívar mine. However, they are lower than the copper extractions typically achieved at copper smelters. Addition of a fourth stage of continuous biological leaching may improve copper recovery, particularly in the case of the concentrate, for which an accelerated short-circuiting effect was observed. Also, heap leaching of the continuous bioleaching system residue, as shown in Figure 1.1, would increase the ultimate copper extraction. Finer grinding of the crusher ore fines or the flotation concentrate could potentially result in improved copper recovery for the same residence time. A mineralogical examination of the experimental bioleach residues would provide information as to the nature of the unleached copper minerals.

There appeared to be no difference in the time required for the ore and concentrate systems to reach steady state. Since bacterial washout was not experienced, shorter residence times and/or higher pulp densities than those tested may be possible. Also, copper leach rates are likely to improve as bacteria adapt to conditions in a continuous biological leaching system. In comparing the ore and concentrate runs, no inhibitory effect of copper content in the presence of chloride was observed in the case of the concentrate. Continuous biological leaching is worth further investigation as an alternative method of processing the ore crusher fines or the flotation concentrate at the copper mine of the Compañia Minera Zaldívar in Chile. Other copper extraction operations with a secondary copper sulphide feed may also benefit from this continuous biological leaching system.

4.4   Acid Consumption

Gross acid consumption is defined as the amount of acid required to neutralize acid consuming minerals and dissolve copper minerals in the ore. Gross acid consumptions were calculated as the total amount of sulphuric acid added during an experiment to keep the pH at about 2, divided by the amount of substrate added. The average gross acid consumptions determined for each phase of the experimental work are given in Table 4.4.

Table 4.4 - Summary of Gross Acid Consumption

Experiment	Zaldívar Ore (kg H₂SO₄/t)	Zaldívar Concentrate (kg H₂SO₄/t)
Shake Flask	20	170
Batch Reactor	20	160
Continuous Reactor	50	200

The acid consumption in continuous mode was found to be higher than in batch mode for both the ore and the concentrate. For the shake flask and batch reactor experiments, the pH was maintained between approximately pH 1.95 and pH 2. For the continuous reactor experiments, the gross acid consumptions were calculated as the amount of sulphuric acid added to the first stage of the continuous biological leaching system during a particular steady state period, divided by the amount of concentrate processed during that period. The average pH values for the ore and concentrate continuous reactor experiments were 1.78 and 1.83 respectively; it was difficult to control the pH to exactly 2. The lower pH accounts for the higher acid consumptions observed in the continuous reactor experiments as compared with the batch experiments. The difference between the acid consumptions observed in batch versus continuous experiments was much greater for the ore than

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for the concentrate because a much lower proportion of the acid consumption was due to copper extraction in the case of the ore. The pH is a very important factor in considering the acid consumption results obtained for this continuous bioleaching experiment. In an industrial plant, the acid consumption would likely be lower than the results for this experiment since pH control would be more tightly controlled to pH 2, and acid would be recycled from SX-EW.

A breakdown of the calculated acid consumption for the ore and concentrate continuous reactor experiments by type of mineral is given in Table 4.5. Acid consuming gangue minerals in an ore include carbonates, such as calcite, and silicates. The gangue minerals in the Zaldívar ore were mainly quartz (SiO_2) and muscovite ($KAl_2(AlSi_3)O_{10}(OH)_2$). Other gangue minerals in the oxide zone of the rhyolite ore were kaolinite ($Al_2Si_2O_5(OH)_4$), rutile (TiO_2), calcite ($CaCO_3$), jarosite, gypsum ($CaSO_4.2H_2O$) and hematite (Fe_2O_3).

Table 4.5 - Percentage of Gross Acid Consumption by Type of Mineral

Description	Zaldívar Ore (kg H_2SO_4/t)	Zaldívar Concentrate (kg H_2SO_4/t)
brochantite dissolution	5.8 (12%)	19 (10%)
chalcocite oxidation	8.2 (16%)	121 (61%)
ferrous ion oxidation	14.0 (28%)	23 (11%)
gangue minerals	22.0 (44%)	37 (18%)
TOTAL	50.0 (100%)	200 (100%)
average pH	1.78	1.83
percent attributable to copper extraction	28%	71%

The percentage of gross acid consumption that is attributable to copper extraction is important because it gives an indication of the amount of acid that can be recycled from the SX-EW circuit. Acid is regenerated during copper electrowinning as follows:

$$CuSO_4 + H_2O \rightarrow Cu + H_2SO_4 + 0.5O_2 \tag{14}$$
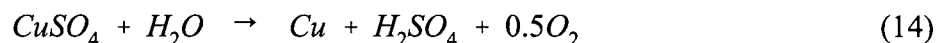
According to Equation 14, one mole of sulphuric acid is generated per mole of cathode copper produced. Using the molecular weights of copper and sulphuric acid, it can be shown that 1.54 kilograms of sulphuric acid are generated per kilogram of cathode copper produced. Therefore, equations for net acid consumption can be written as follows:

$$net\ acid\ consumption\ =\ gross\ acid\ consumption\ -\ 1.54\left(\frac{Cu\ extracted}{tonne\ ore}\right) \tag{15}$$

$$net\ acid\ consumption\ =\ gross\ acid\ consumption\ -\ 1.54\left(\frac{Cu\ extracted}{tonne\ concentrate}\right) \tag{16}$$

Using Equation 15, the net acid consumption for the Zaldívar ore was found to be about 30 kg/t. This was calculated based on a feed total copper content of 1.56% and a copper extraction of 82%.

Using Equation 16, the Zaldívar concentrate was found to be a net acid generator of about 20 kg/t. This was calculated based on a feed total copper content of 17.3% and a copper extraction of 83%.

Several reactions in the system contributed to the overall acid consumption. Chalcocite oxidation by the direct mechanism (Equation 3), bacterial oxidation of ferrous to ferric iron (Equation 6) and brochantite dissolution (Equation 13) were all acid consuming reactions:

$$Cu_2S + 0.5O_2 + H_2SO_4 \rightarrow CuSO_4 + CuS + H_2O \tag{3}$$

$$2FeSO_4 + 0.5O_2 + H_2SO_4 \rightarrow Fe_2(SO_4)_3 + H_2O \tag{6}$$

$$Cu_4(SO_4)(OH)_6 + 3H_2SO_4 \rightarrow 4CuSO_4 + 6H_2O \tag{13}$$
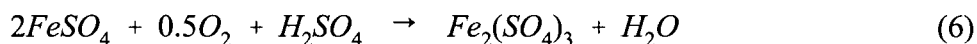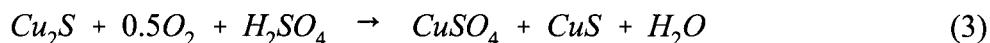
Acid generating reactions included pyrite oxidation by the direct mechanism (Equation 5), biooxidation of elemental sulphur (Equation 9) and jarosite precipitation (Equation 10):

$$2FeS_2 + 7.5O_2 + H_2O \rightarrow Fe_2(SO_4)_3 + H_2SO_4 \tag{5}$$

$$S^o + 1.5O_2 + H_2O \rightarrow H_2SO_4 \tag{9}$$

$$3Fe_2(SO_4)_3 + 14H_2O \rightleftharpoons 2H_3OFe_3(SO_4)_2(OH)_6 + 5H_2SO_4 \tag{10}$$
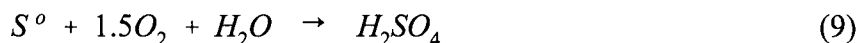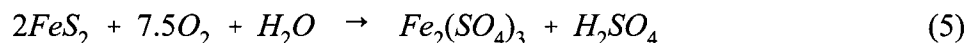
Due to the complex chemistry, it was difficult to predict theoretically the acid requirements of the continuous biological leaching systems. However, for both the ore and concentrate runs, the acid consumption reached a constant value when the system was operating at steady state. Also, as expected, the acid consumption was approximately equal for tests in which the final copper extraction was approximately equal.

5.0 CONCLUSIONS AND RECOMMENDATIONS

5.1 Conclusions

1. Shake flask and batch reactor experiments demonstrated that the bacteria were capable of oxidizing both the Zaldívar ore and concentrate in a 3 g/L chloride environment, and adapting to growth on these feed materials at various pulp densities.

2. Under batch conditions, the higher copper content of the Zaldívar concentrate as compared to the ore appeared to have an inhibitory effect on bacterial growth, which was magnified in the presence of 3 g/L chloride. This inhibitory effect was overcome in the continuous biological leaching experiment.

3. The Zaldívar ore and concentrate were found to be readily amenable to slow chemical oxidation, with 77% and 68% copper extractions respectively occurring in sterile control shake flask experiments.

4. The residues from the biological leaching of both the ore and concentrate were thought to contain elemental sulphur, in addition to sulphide sulphur and sulphate sulphur. This suggests that covellite oxidation by the indirect ferric oxidation mechanism of biological leaching (Equation 8) was more important than covellite oxidation by the direct mechanism of biological leaching (Equation 4).

5. Continuous biological leaching of a chalcocite ore and concentrate in a saline environment of 3 g/L chloride was successful.

6. Continuous biological leaching is worth further investigation as a method of processing the ore crusher fines or the flotation concentrate at the Zaldívar mine in Chile. The solid residue from the continuous biological leaching process would be an excellent source of bacterial inoculum for biological heap leaching. Some of the copper remaining in the residue could be extracted by heap leaching. Higher copper extractions and leaching rates than those observed in this study are possible as bacteria adapt to system conditions over time. Other copper extraction operations with a secondary copper sulphide feed may also benefit from this continuous biological leaching system.

7. The results of the ore continuous biological leaching experiment showed that the bacteria were able to adapt to progressively lower system residence times and higher pulp densities. The lowest residence time achieved was $2\frac{2}{3}$ days at a pulp density of 10%. The ultimate copper extraction at this steady state condition was 82% and the first stage copper leach rate was 16 mg/L/h.

8. The results of the concentrate continuous biological leaching experiment showed that the bacteria were able to adapt to progressively lower system residence times and higher pulp densities. The lowest residence time achieved was $2\frac{2}{3}$ days and the highest pulp density achieved at this residence time was 10%. The ultimate copper extraction at this steady state condition was 83% and the first stage copper leach rate was 250 mg/L/h.

9. Chalcocite was apparently biologically leached preferentially to pyrite from both the Zaldívar ore and concentrate. The main reasons for incomplete sulphate formation were thought to be incomplete oxidation of pyrite or elemental sulphur.

10.   An accelerated short-circuiting effect was observed in the first bioleach stage of the concentrate continuous bioleaching system due to flotation of a sulphide-rich froth, and transfer of this froth from the first to the second bioleach stage. Four stages of biological leaching are probably required to achieve adequate copper extraction in a full scale plant.

Warning: The results achieved in this continuous bioleaching experiment were partly due to leaching in the feed tank, particularly for the longest residence times tested. At a larger scale, it is not likely that leaching will occur in the feed tank because relatively low residence times will be used, and the feed material will be fresh and unoxidized.

5.2   Recommendations for Further Work at Zaldívar

The bench scale continuous biological leaching experiments carried out in this study are currently being continued at the Zaldívar site. The feed material being used is a flotation concentrate produced on-site. Following are several recommendations for these experiments at Zaldívar.

1.   The possibility of improved copper extraction could be investigated by studying the effect of finer grinding of the concentrate. In addition, a mineralogical examination of the bioleach residues would provide information as to the nature of the unleached copper minerals.

2.   Cost estimates of ore and concentrate continuous biological leaching could be used to compare the two systems.

3.   The amenability of the bacterial culture to biological leaching of the Zaldívar ore and concentrate without addition of the 9K nutrient media should be tested. The effects of

addition of the individual component nutrients in the 9K media could also be tested. The objective would be to determine what nutrients would be required in a larger scale continuous bioleaching plant. A full analysis of the feed materials would provide information about nutrients already present in the gangue minerals.

4. The tests at the Zaldívar site are being conducted without CO_2-enrichment of the air supply. The results from the on-site tests should be compared with the results of this study to determine the effect of CO_2-enrichment on the rate of bioleaching of the Zaldívar ore and concentrate. It may not be necessary to provide carbon dioxide to a full scale bioleaching process. However, if the lack of CO_2-enrichment of the air supply is shown to have a detrimental effect, a cheap source of CO_2 should be found, such as carbonate minerals or combustion gases.

5.3 General Recommendations

1. Titanium was found to be the best material for the impellers and impeller shafts in stirred reactor biological leaching experiments carried out in a saline environment.

2. Elemental sulphur analysis of bioleach residues would provide further information about the chemistry of bioleaching. The results of the elemental sulphur analysis could be used to evaluate the relative importance of the direct and indirect mechanisms of bioleaching.

3. A mathematical model of the continuous bioleaching system used in this study could be developed and compared with the experimental results. Copper leach rates achieved in this study could be used as input to the model.

6.0 REFERENCES

Adam, K., Stefanakis, M. and Kontopoulos, A., Bacterial Oxidation of the Olympias Arsenical Pyrite Concentrate, In: Biotechnology in Minerals and Metal Processing, B.J. Scheiner, F.M. Doyle and S.K. Kawatra (Eds), Society of Mining Engineers, Inc., Cushing-Malloy, Inc., Ann Arbor, MI, USA, 131-138, (1989)

Agate, A.D. and Khinvasara, N.J., Bioleaching of Copper Ores and Concentrate of Malanjkhand Area, India, In: Workshop on Biotechnology for the Mining, Metal-Refining and Fossil Fuel Processing Industries, Biotechnol. Bioeng. Symp., No.16, H.L. Ehrlich and D.S. Holmes (Eds), John Wiley and Sons, NY, 83-90, (1986)

Alexander, B., Leach, S. and Ingledew, W.J., The Relationship Between Chemiosmotic Parameters and Sensitivity to Anions and Organic Acids in the Acidophile *Thiobacillus ferrooxidans*, Journal of General Microbiology, Vol. 133, 1171-1179, (1987)

Almendras, E., Bustos, S., Ruiz, P., Wiertz, J. and Badilla-Ohlbaum, R., Behavior of Copper Sulfide Minerals Under Bacterial Leaching, In: Hydrometallurgy and Electrometallurgy of Copper, W.C. Cooper, G.E. Lagos and G. Ugarte (Eds), Copper 87, Volume 3, University of Chile, Chile, 119-130, (1987)

Babij, T. and Madgwick, J.C., High Yield Bacterial Leaching of Copper Concentrates, Proc. Australas. Inst. Min. Metall., No.287, September, 61-64, (1983)

Barrett, J., Hughes, M.N., Karavaiko, G.I. and Spencer, P.A., Metal Extraction by Bacterial Oxidation of Minerals, Ellis Horwood Limited, London, England, (1993)

Beck, J.V., Chalcocite Oxidation by Concentrated Cell Suspensions of *Thiobacillus ferrooxidans*, In: Conference Bacterial Leaching 1977, Schwartz, W. (Ed), Verlag Chemie, New York, 119, (1977)

Biswas, A.K. and Davenport, W.G., Extractive Metallurgy of Copper, 3rd edition, Pergamon Press, U.K., (1994)

Blancarte-Zurita, M.A. and Branion, R.M.R., Computer Simulation of Bioleaching in Continuous Reactors in Series, In: Biohydrometallurgy, Proc. Int'l Symp., Warwick, 1987, P.R. Norris and D.P. Kelly (Eds), Science and Technology Letters, 517-520, (1988)

Briceño, H. and Rossi, M.C., Bacterial Leaching of Copper Concentrates: A Preliminary Feasibility Study, In: Hydrometallurgy and Electrometallurgy of Copper, W.C. Cooper, G.E. Lagos and G. Ugarte (Eds), Copper 87, Volume 3, University of Chile, Chile, 131-147, (1987)

Brock, T.D. and Madigan, M.T., Biology of Microorganisms, 6th edition, Prentice Hall, (1991)

Bruynesteyn, A. and Duncan, D.W., Microbiological Leaching of Sulphide Concentrates, Canadian Metallurgical Quarterly, Vol. 10, No. 1, 57-63, (1971)

Bruynesteyn, A., Hackl, R.P., Lawrence, R.W. and Vizsolyi, A.I., Biological-Acid Leach Process,

United States Patent No. 4 571 387, February 18, 1986, Canadian Patent No. 1 214 043, November 18, 1986, (1986)

Bruynesteyn, A., Lindsay, D. and O'Rourke, J., Development of Dump Leach Operations at Gibraltar Mines Ltd., In: Hydrometallurgy and Electrometallurgy of Copper, W.C. Cooper, G.E. Lagos and G. Ugarte (Eds), Copper 87, Volume 3, University of Chile, Chile, 39-47, (1987)

Cameron, F.J., Jones, M.V. and Edwards, C., Effects of Salinity on Bacterial Iron Oxidation, Current Microbiology, Vol. 10, 353-356, (1984)

Colmer, A.R. and Hinkle, M.E.,The Role of Microorganisms in Acid Mine Drainage: A Preliminary Report, Science, Vol. 106, 253-256, (1947)

Corral, M.T., González, F., Blázquez, M.L., Ballester, A. and Muñoz, J.A., Continuous Bioleaching of Mineral Sulphides, In: Biohydrometallurgical Technologies, Volume I: Bioleaching Processes, A.E. Torma, J.E. Wey and V.L. Lakshmanan (Eds), The Minerals, Metals & Materials Society, Warrendale, PA, USA, 65-75, (1993)

Dave, S.R., Natarajan, K.A. and Bhat, J.V., Bio-oxidation Studies with Thiobacillus ferrooxidans in the Presence of Copper and Zinc, Trans. Inst. Min. Metall., Vol. 88, C234-C237, (1979)

Duncan, D.W. and Trussell, P.C., Advances in the Microbiological Leaching of Sulphide Ores, Canadian Metallurgical Quarterly, Vol. 3, No. 1, 43-55, (1964)

Duncan, D.W., Walden, C.C. and Trussell, P.C., Accelerated Microbiological Ore Extraction Process, United States Patent No. 3,305,353, Feb. 21, 1967, (1967)

Duncan, D.W. and McGoran, C.J.M., Rapid Bacteriological Metal Extraction Method, United States Patent No. 3,607,235, September, 21, 1971, Canadian Patent No. 869,470, April 27, 1971, (1971)

Dutrizac, J.E. and MacDonald, R.J.C., The Effect of Some Impurities on the Rate of Chalcopyrite Dissolution, Canadian Metallurgical Quarterly, Vol. 12, No. 4, 409-420, (1973)

Ehrlich, H.L. and Brierley, C.L., Microbial Mineral Recovery, McGraw Hill Publishing Company, USA, (1990)

Einsele, A., Scaling Up Bioreactors, Process Biochemistry, July, (1978)

Elzeky, M. and Attia, Y.A., Effects of Bacterial Adaptation on Bioleaching of Ferrous Sulfides, In: Biotechnology in Minerals and Metal Processing, B.J. Scheiner, F.M. Doyle and S.K. Kawatra (Eds), Society of Mining Engineers, Inc., Cushing-Malloy, Inc., Ann Arbor, MI, USA, 139-149, (1989)

Espejo, R.T. and Ruiz, P., Growth of Free and Attached Thiobacillus ferrooxidans in Ore Suspension, Biotechnology and Bioengineering, Vol. 30, 586-592, (1987)

Everett, P.K., Development of Intec Copper Process by an International Consortium, In: Hydrometallurgy '94, Institution of Mining and Metallurgy (IMM) and Society of Chem. Industry

(SCI), Chapman & Hall, London, U.K., 913-922, (1994)

Gentina, J.C., Rivera, R., Acevedo, F., Retamal, J. and Schaffeld, G., Influence of Organic Agents on the Kinetics of Bacterial Leaching of Copper Ores, In: Hydrometallurgy and Electrometallurgy of Copper, W.C. Cooper, G.E. Lagos and G. Ugarte (Eds), Copper 87, Volume 3, University of Chile, Chile, 107-117, (1987)

Gormely, L.S., Duncan, D.W., Branion, R.M.R. and Pinder, K.L., Continuous Culture of *Thiobacillus ferrooxidans* on a Zinc Sulfide Concentrate, Biotechnol. and Bioeng., Vol. 17, 31-49, (1975)

Gormely, L.S. and Branion, R.M.R., Engineering Design of Microbiological Leaching Reactors, In: Biohydrometallurgy - 1989, Proceedings of the International Biohydrometallurgy Symposium, Jackson Hole, WY, USA, Aug 13-18, 1989, J. Salley, R.G.L. McCready and P.L. Wichlacz (Eds), 499-518, (1989)

Grishin, S.I., Bigham, J.M. and Tuovinen, O.H., Characterization of Jarosite Formed Upon Bacterial Oxidation of Ferrous Sulfate in a Packed-Bed Reactor, Applied and Environmental Microbiology, Vol. 54, No. 12, 3101-3106, (1988)

Groudev, S.N., Differences Between *Thiobacillus ferrooxidans* Strains With Respect to Their Ability to Oxidize Covellite, Comptes Rendus de l'Académie Bulgare des Sciences, Vol. 33, No. 5, 659-662, (1980)

Groudev, S.N., Continuous Bacterial Leaching of Copper Sulphide Concentrates, In: Fundamental and Applied Biohydrometallurgy, Proc. of the 6th Internat. Symp. on Biohyd., Vancouver, Aug 21-24, 1985, R.W. Lawrence, R.M.R. Branion and H.G. Ebner (Eds), Elsevier, New York, 43-50, (1985)

Hackl, R.P., Wright, F.R. and Gormely, L.S., Bioleaching of Refractory Gold Ores - Out of the Lab and Into the Plant, In: Biohydrometallurgy - 1989, Proceedings of the International Biohydrometallurgy Symposium, Jackson Hole, WY, USA, Aug 13-18, 1989, J. Salley, R.G.L. McCready and P.L. Wichlacz (Eds), 533-549, (1989)

Harrison, A.P. Jr., Jarvis, B. and Johnson, J.L., Heterotrophic Bacteria From Cultures of Autotrophic *Thiobacillus ferrooxidans*: Relationships as Studied by Means of Deoxyribonucleic Acid Homology, J. Bacteriol., Vol. 143, 448-454, (1980)

Heffelfinger, B., The Adherence of *Acidiphilium Cryptum* to Chalcopyrite, M.Sc. Thesis, The University of British Columbia, (1990)

Hiskey, J.B., In-situ Leaching Recovery of Copper: What's Next?, In: Hydrometallurgy '94, Institution of Mining and Metallurgy (IMM) and Society of Chem. Industry (SCI), Chapman & Hall, London, U.K., 43-67, (1994)

Imai, K., On the Mechanism of Bacterial Leaching, In: Metallurgical Applications of Bacterial Leaching and Related Microbiological Phenomena, L.E. Murr, A.E. Torma and J.A. Brierley (Eds),

Academic Press, New York, 275-295, (1978)

Ingledew, W.J., Thiobacillus ferrooxidans - The Bioenergetics of an Acidophilic Chemolithotroph, Biochimica et Biophysica Acta, Vol. 683, 89-117, (1982)

Johnson, A.M., Carlson, D.G., Bagley, S.T. and Johnson, D.L., In-situ Bioleaching Investigations of Michigan Chalcocite Ores, In: Proc. of the SME Annual Meeting, Jan 25-28, Phoenix, Arizona, (1988)

Jyothi, N., Sudha, K.N., Brahmaprakash, G.P., Natarajan, K.A. and Ramananda Rao, G., Electrochemical Aspects of Bioleaching of Mixed Sulphides, In: Biotechnology in Minerals and Metal Processing, B.J. Scheiner, F.M. Doyle and S.K. Kawatra (Eds), Society of Mining Engineers, Inc., Cushing-Malloy, Inc., Ann Arbor, MI, USA, 9-16, (1989)

Karavaiko, G.I., Rossi, G., Agate, A.D., Groudev, S.N. and Avakyan, Z.A. (Eds), Biogeotechnology of Metals - Manual, Centre for International Projects, Moscow, USSR, (1988)

Konishi, Y. and Asai, S., A Biochemical Engineering Approach to the Bioleaching of Pyrite in Batch Continuous Tank Reactors, In: Biohydrometallurgical Technologies, A.E. Torma, J.E. Wey and V.L. Lakshmanan (Eds), The Minerals, Metals & Materials Society, 259-268, (1993)

Lawrence, R.W., Vizsolyi, A., Vos R.J. and Bruynesteyn, A., Continuous Bioleaching of Copper Concentrates, Paper presented at the A.I.Ch.E. National Meeting, March 11-15, 1984, Atlanta, Georgia, (1984)

Lawrence, R.W., The Application of Stirred-Tank Bioleaching for the Enhancement of Gold Recovery from Refractory Ores and Concentrates, In: Proc. of the Tenth Annual General Meeting of BIOMINET, W.D. Gould, L. Lortie and D. Rodrigue (Eds), October 28, 1993, Mississauga, Ontario, 1-23, (1993)

Leduc, L.G. and Ferroni, G.D., The Need for Thiobacillus ferrooxidans Strain Selection in Applications of Bioleaching, In: Proc. of the Tenth Annual General Meeting of BIOMINET, W.D. Gould, L. Lortie and D. Rodrigue (Eds), October 28, 1993, Mississauga, Ontario, 25-42, (1993)

Leong, B.J.Y., Dreisinger, D.B., Lo, M., Branion, R., Hackl, R.P., Gormely, L.S. and Crombie, D.R., The Microbiological Leaching of a Sulphidic Copper Ore in a Strongly Saline Medium (I): Shakeflask and Column Studies, In: Biohydrometallurgical Technologies, A.E. Torma, J.E. Wey and V.L. Lakshmanan (Eds), The Minerals, Metals & Materials Society, 117-126, (1993)

Levenspiel, O., Chemical Reaction Engineering, Second Edition, John Wiley and Sons, New York, (1972)

Liu, M.S., Branion, R.M.R. and Duncan, D.W., Oxygen Transfer to Thiobacillus Cultures, In: Biohydrometallurgy, Proc. Int'l Symp., Warwick, 1987, P.R. Norris and D.P. Kelly (Eds), Science and Technology Letters, 375-384, (1988)

Malouf, E.E. and Prater, J.D., Role of Bacteria in the Alteration of Sulfide Minerals, Journal of

Metals, May, 353-356, (1961)

Marchbank, A.R., Applications of Biotechnology in Metals Recovery (Technical Review), In: Proc. of the Tenth Annual General Meeting of BIOMINET, W.D. Gould, L. Lortie and D. Rodrigue (Eds), October 28, 1993, Mississauga, Ontario, 43-52, (1993)

Martin, P.A.W., Dugan, P.R. and Tuovinen, O.H., Plasmid DNA in Acidophilic, Chemolithotrophic thiobacilli, Canadian Journal of Microbiology, Vol. 27, 850-853, (1981)

McElroy, R.O. and Bruynesteyn, A., Continuous Biological Leaching of Chalcopyrite Concentrates: Demonstration and Economic Analysis, In: Metallurgical Applications of Bacterial Leaching and Related Microbiological Phenomena, L.E. Murr, A.E. Torma and J.A. Brierley (Eds), Academic Press Publishers, New York, 441-462, (1978)

Melluish, J.M., Groat, L.A., Dreisinger, D.B., Branion, R., Leong, B.J.Y. and Crombie, D.R., Mineralogical Characterization of Residues from a Microbiologically Leached Copper Sulphide Ore, In: Biohydrometallurgical Technologies, A.E. Torma, J.E. Wey and V.L. Lakshmanan (Eds), The Minerals, Metals & Materials Society, 127-136, (1993)

Miller, D.M. and Hansford, G.S., Batch Biooxidation of a Gold-Bearing Pyrite-Arsenopyrite Concentrate, Minerals Engineering, Vol. 5, No. 6, 613-629, (1992)

Moore, D.C., Development of a Biooxidation Leach Process at Asamera Minerals (U.S.) Cannon Mine, In: Biotechnology in Minerals and Metal Processing, B.J. Scheiner, F.M. Doyle and S.K. Kawatra (Eds), Society of Mining Engineers, Inc., Cushing-Malloy, Inc., Ann Arbor, MI, USA, 123-130, (1989)

Mottana, A., Crespi, R. and Liborio, G., Guide to Rocks and Minerals, M. Prinz, G. Harlow and J. Peters (Eds), Simon and Schuster Inc., New York, (1978)

Murr, L.E., Theory and Practice of Copper Sulphide Leaching in Dumps and In-Situ, Minerals Sci. Engng, Vol. 12, No.3 , 121-189, (1980)

Nagpal, S., Microbial Leaching of a Mixed Sulphide Ore Concentrate in Continuous-Flow Reactors: Development of a Mathematical Model for the Process, Ph.D. Thesis, University of Utah, (1993)

Nielsen, A.M. and Beck, J.V., Chalcocite Oxidation and Coupled Carbon Dioxide Fixation by *Thiobacillus ferrooxidans*, Science, Vol. 175, p. 1124-1126, (1972)

Norris, P.R. and Kelly, D.P., Dissolution of Pyrite (FeS2) by Pure and Mixed Cultures of Some Acidophilic Bacteria, FEMS Microbiol. Lett., Vol. 4, 143-146, (1978)

Peters, E., Hydrometallurgical Process Innovation, Hydrometallurgy, Vol. 29, 431-459, (1992)

Petrova, T.A., A Study of Growth and Development of *Thiobacillus ferrooxidans* with Regard to Oxidation of Fe^{2+} and Sulfide Minerals by Means of Mathematical Modelling, In: Biogeotechnology of Metals, G.I. Karavaiko and S.N. Groudev (Eds), Proc. of Int'l Sem. on Modern Aspects of

Microbio. Hydromet., Centre of Int'l Projects, G.K.N.T., Moscow, 64-82, (1985)

Pinches, A., Al-Jaid, F.O., Williams, D.J.A. and Atkinson, B., Leaching of Chalcopyrite Concentrates with *Thiobacillus ferrooxidans* in Batch Culture, Hydrometallurgy, Vol. 2, 87-103, (1976)

Pinches, A., Chapman, J.T., De Riele, W.A.M. and Van Staden, M., The Performance of Bacterial Leach Reactors for the Pre-Oxidation of Refractory Gold-Bearing Sulphide Concentrates, In: Biohydrometallurgy, Proc. Int'l Symp., Warwick, U.K., P.R. Norris and D.P. Kelly (Eds), Science and Technology Letters, 329-344, (1988)

Pivovarova, T.A. and Golovacheva, R.S., Microorganisms Important for Hydrometallurgy: Cytology, Physiology, and Biochemistry, In: Biogeotechnology of Metals, G.I. Karavaiko and S.N. Groudev (Eds), Centre of International Projects, G.K.N.T., Moscow, 27-55, (1985)

Rhodes, M.K., Budden, J.R. and Bunyard, M., Bactech Bacterial Oxidation Plant at the Youanmi Mine: Pilot Testing, Engineering Design and Commissioning, EPD Congress 1995, G.W. Warren (Ed), The Minerals, Metals & Materials Society, 145-153, (1995)

Rossi, G., Biohydrometallurgy, McGraw Hill Book Company, (1990)

Sakaguchi, H., Torma, A.E. and Silver, M., Microbiological Oxidation of Synthetic Chalcocite and Covellite by *Thiobacillus ferrooxidans*, Applied and Environmental Microbiol., Vol. 31, No. 1, 7-10, (1976a)

Sakaguchi, H., Silver, M. and Torma, A.E., Microbiological Leaching of a Chalcopyrite Concentrate by *Thiobacillus ferrooxidans*, Biotechnol. and Bioeng., Vol. 18, 1091-1101, (1976b)

Sanmugasunderam, V., Kinetic Studies on the Biological Leaching of a Zinc Sulphide Concentrate in Two Stage Continuous Stirred Tank Reactors, Ph.D. Thesis, The University of British Columbia, (1981)

Sanmugasunderam, V., Branion, R.M.R. and Duncan, D.W., Two Stage Microbiological Leaching, With and Without Recycle to the First Stage, of a Zinc Sulfide Concentrate by *Thiobacillus ferrooxidans*, In: Fundamental and Applied Biohydrometallurgy, R.W. Lawrence, R.M.R. Branion and H.G. Ebner (Eds), Elsevier, Amsterdam, 263-275, (1986)

Sato, M., Half-Cell Potentials of Semiconductive Simple Binary Sulphides in Aqueous Solution, Electrochimica Acta, Vol. 11, 361-373, (1966)

Short, H. and Parkinson, G., Rx for Leaching Copper: Go the Biological Way, Chemical Engineering, July 11, 1983, 26-29, (1983)

Silver, M. and Torma, A.E., Oxidation of Metal Sulfides by *Thiobacillus ferrooxidans* Grown on Different Substrates, Canadian Journal of Microbiology, Vol. 20, 141-147, (1974)

Silverman, M.P. and Lundgren, D.G., Studies on the Chemoautotrophic Iron Bacterium

Ferrobacillus Ferrooxidans I. An Improved Medium and a Harvesting Procedure for Securing High Cell Yields, J. Bacteriol., Vol. 77, 642-647, (1959)

Temple, K.L. and Colmer, A.R., The Autotrophic Oxidation of Iron by a New Bacterium: *Thiobacillus ferrooxidans*, J. Bacteriol., Vol. 62, 605-611, (1951)

Thomas, C.R., Shear Effects On Cells In Bioreactors, In: Processing of Solid-Liquid Suspensions, P.A. Shamlou (Ed), Butterworth-Heinemann Ltd., Oxford, U.K., 158-191, (1993)

Torma, A.E., Walden, C.C., Duncan, D.W. and Branion, R.M.R., The Effect of Carbon Dioxide and Particle Surface Area on the Microbiological Leaching of a Zinc Sulfide Concentrate, Biotechnol. and Bioeng., Vol. 14, 777-786, (1972)

Torma, A.E., Ashman, P.R., Olsen, T.M. and Bosecker, K., Microbiological Leaching of a Chalcopyrite Concentrate and Recovery of Copper by Solvent Extraction and Electrowinning, Metall (Berlin), Vol. 33, 479 - 484, (1979)

Torma, A.E. and Bosecker, K., Bacterial Leaching, In: Progress in Industrial Microbiology, Vol. 16, 77-118, (1982)

Torma, A.E. and Apel, W.A., Bioleaching of Chalcopyrite Samples, In: EPD Congress 1992, J.P. Hager (Ed), The Minerals, Metals & Materials Society, 3-15, (1992)

Toro, L., Paponetti, B. and Cantalini, C., Precipitate Formation in the Oxidation of Ferrous Ions in the Presence of *Thiobacillus ferrooxidans*, Hydrometallurgy, Vol. 20, 1-9, (1988)

Torres, F., Blázquez, M.L., González, F., Ballester, A. and Mier, J.L., The Bioleaching of Different Sulfide Concentrates Using Thermophilic Bacteria, Metallurgical and Materials Transactions B, Vol. 26B, 455-465, (1995)

Tuovinen, O.H., Inhibition of *Thiobacillus ferrooxidans* by Mineral Flotation Reagents, European J. Applied Microbiol. Biotechnol., Vol. 5, 301-304, (1978)

Vargas, T., Wiertz, J.V., Escobar, B. and Badilla-Ohlbaum, R., The Catalytic Role of *Thiobacillus ferrooxidans* in the Leaching of Pure Natural Chalcopyrite, In: EPD Congress 1990, D.R. Gaskell (Ed), The Minerals, Metals & Materials Society, 229-238, (1990)

Vaughan, D.J. and Craig, J.R., Mineral Chemistry of Metal Sulphides, Cambridge University Press, New York, (1978)

Walsh, F. and Mitchell, R., An Acid-Tolerant Iron-Oxidizing *Metallogenium*, J. Gen. Microbiol., Vol. 72, 369-376, (1972)

Wiertz, J.V., Godoy Ríos, I. and Escobar Miguel, B., Dissolved Iron Equilibrium in Bacterial Leaching Systems, In: Hydrometallurgy '94, Institution of Mining and Metallurgy (IMM) and Society of Chem. Industry (SCI), Chapman & Hall, London, U.K., 385-393, (1994)

SHAKE FLASK AND BATCH REACTOR EXPERIMENTS

Copper Extraction Based on Copper in Solution and Calculated Head

This sample calculation is based on the results for batch reactor experiment BC11 on 94/9/7. The experiment tested the bioleaching of the concentrate at a pulp density of 5%. First the copper extraction based on copper in solution must be determined.

Data:
$$[Cu_{solution}] = 8.25 \text{ g/L}$$
$$V_{solution} = 1.944 \text{ L}$$
$$[Cu_{inoc}] = 1.79 \text{ g/L}$$
$$V_{inoc} = 0.130 \text{ L}$$
$$[Cu_{media}] = 0.001 \text{ g/L}$$
$$V_{media} = 1.820 \text{ L}$$
$$\sum Cu_{sample} V_{sample} = 0.0583 \text{ g}$$
$$Mass_{feed} = 99.7 \text{ g}$$

$$Copper\ Extraction_1 = 100 \left[\frac{[Cu_{solution}]V_{solution} - \left([Cu_{inoc}]V_{inoc} + [Cu_{media}]V_{media}\right) + \sum Cu_{sample}V_{sample}}{(\frac{17.3}{100})\ Mass_{feed}(g)} \right]$$

$$Copper\ Extraction_1 = 100 \left[\frac{(8.25)(1.944) - \left((1.79)(0.130) + (0.001)(1.820)\right) + 0.0583}{(\frac{17.3}{100})\ (99.7)} \right]$$

$$Copper\ Extraction_1 = 92\%$$

Next, the copper calculated head must be determined.

Data:
$$Cu_{out} = 16.391 \text{ g}$$
$$Cu_{inoc} = 0.233 \text{ g}$$
$$Cu_{media} = 0.002 \text{ g}$$
$$Mass_{feed} = 99.7 \text{ g}$$

$$Calculated\ Head = 100 \left[\frac{Cu_{out}(g) - \left(Cu_{inoc}(g) + Cu_{media}(g)\right)}{Mass_{feed}(g)} \right]$$

$$Calculated\ Head\ =\ 100 \left[\frac{16.391\ -\ \left(0.233\ +\ 0.002\right)}{99.7}\right]$$

$$Calculated\ Head\ =\ 16.2\%$$

Finally, the copper extraction based on copper in solution and calculated head can be determined.

$$Copper\ Extraction_2\ =\ Copper\ Extraction_1 \left(\frac{17.3}{Calculated\ Head}\right)$$

$$Copper\ Extraction_2\ =\ 92\% \left(\frac{17.3}{16.2}\right)$$

$$Copper\ Extraction_2\ =\ 98\%$$

Therefore, copper extraction based on copper in solution and calculated head was found to be 98%.

Copper Extraction Based on Outputs

This sample calculation is based on the results for batch reactor experiment BC11, which tested the bioleaching of the concentrate at a pulp density of 5%.

Data:

$Cu_{out,\ solids}$	= 0.414 g
Cu_{out}	= 16.391 g
Cu_{inoc}	= 0.233 g
Cu_{media}	= 0.002 g

$$Copper\ Extraction\ =\ 100 \left[1\ -\ \frac{Cu_{out,\ solids}(g)}{Cu_{out}(g)\ -\ \left(Cu_{inoc}(g)\ +\ Cu_{media}(g)\right)}\right]$$

$$Copper\ Extraction\ =\ 100 \left[1\ -\ \frac{0.414}{16.391\ -\ (0.233\ +\ 0.002)} \right]$$

$$Copper\ Extraction\ =\ 97\%$$

Therefore, the copper extraction based on outputs was found to be 97%.

Copper Leach Rate

This sample calculation is based on results from the batch reactor experiment BC11 on 94/8/31.

Data:
$[Cu]_t$ = 7.35 g/L
$[Cu]_{t-1}$ = 6.20 g/L
δt = 48 h

$$Rate\ =\ \left[\frac{[Cu]_t\ (g/L)\ -\ [Cu]_{t-1}\ (g/L)}{\delta t\ (h)} \right] \frac{1000\ mg}{g}$$

$$Rate\ =\ \left[\frac{7.35\ -\ 6.20}{48} \right] 1000$$

$$Rate\ =\ 24\ mg/L/h$$

Therefore, the copper leach rate was determined to be 24 mg/L/h.

Sulphate Formation

This sample calculation is based on the results for batch reactor experiment BC11, which tested the bioleaching of the concentrate at a pulp density of 5%.

Data: Solids $(S^{2-} + S^o)_{out}$ = 2.038 g
          Solids Sulphide$_{feed}$ = 11.585 g

$$Sulphate\ Formation\ =\ 100 \left[1 - \frac{Solids\ (S^{2-} + S^o)_{out}\ (g)}{Solids\ Sulphide_{feed}\ (g)} \right]$$

$$Sulphate\ Formation\ =\ 100 \left[1 - \frac{2.038}{11.585} \right]$$

$$Sulphate\ Formation\ =\ 82\%$$

Therefore, the sulphate formation was determined to be 82%.

CONTINUOUS REACTOR EXPERIMENTS

Copper Extraction

This is an example of the calculated head method of determining percent copper extraction. The sample calculation uses data from a 100 ml slurry sample taken from the third concentrate bioleach stage on 94/12/5. This was during Test 1, when the residence time was eight days and the pulp density was 5%.

Data: Filtrate   94.5 ml   8.35 g/L Cu   0.789 g Cu
          Wash   445 ml   0.104 g/L Cu   0.046 g Cu
          Solids   4.38 g   1.95% Cu   0.085 g Cu
          Total   0.920 g Cu

Copper Extraction   =   100 * (1 - 0.085/0.920)
                    =   91%

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Therefore, the copper extraction was determined to be 91%. The average copper extraction for the three 100 ml slurry samples taken during this steady state period was also determined to be 91%.

Copper Leach Rate

This is a sample calculation for the first concentrate bioleach stage on 94/12/5. This was during Test 1, when the residence time was eight days and the pulp density was 5%.

Data:

$$[Cu]_N = 6.65 \text{ g/L}$$
$$[Cu]_{N-1} = 1.27 \text{ g/L}$$
$$V_N = 4.0 \text{ L}$$
$$\text{feed flowrate} = 0.824 \text{ L/day}$$
$$[Cu]_{N,t} = 6.65 \text{ g/L}$$
$$[Cu]_{N,t-1} = 6.55 \text{ g/L}$$
$$\delta t = 23.4 \text{ hr}$$

$$Rate = \left[\frac{[Cu]_N - [Cu]_{N-1}}{\dfrac{V_N}{feed\ flowrate}} + \frac{[Cu]_{N,t} - [Cu]_{N,t-1}}{\dfrac{\delta t}{24}} \right] \frac{1000}{24}$$

$$Rate = \left[\frac{6.65 - 1.27}{\dfrac{4.0}{0.824}} + \frac{6.65 - 6.55}{\dfrac{23.4}{24}} \right] \frac{1000}{24}$$

$$Rate = 50 \ mg/L/h$$

Therefore, the copper leach rate for the first concentrate bioleach stage on 94/12/5 was determined to be 50 mg/L/h.

Check of Bioleach Stage 1 Copper Leach Rate

This is a sample calculation for Test 1 of the concentrate continuous reactor bioleaching, when the overall residence time was 8 days and the pulp density was 5%. The overall stage 1 copper extraction based on solids analysis was found to be 85%. The copper extraction in the feed tank was found to be 36%. The actual copper extraction in bioleach stage 1 was the difference or 49%. Now, the theoretical copper extraction can be calculated based on the stage 1 copper leach rate.

Data: Stage 1 Copper Leach Rate   = 50 mg/L/h
Stage 1 Residence Time   = 4 days
Pulp Density   = 50 g/Lpulp
Head Assay   = 17.3% total copper
Concentrate Solids s.g.   = 3.32
x $L_{solution}$ / L_{pulp} = 0.985

$$Copper\ Extraction = 100 \left[\frac{Rate(\frac{mg_{Cu}}{L_{solution}\ h})\ Residence\ Time(days)\ (\frac{24h}{day})\ (\frac{g_{Cu}}{1000mg_{Cu}})}{Pulp\ Density(\frac{g_{concentrate}}{L_{pulp}})\ Head\ Assay(\frac{g_{Cu}}{g_{concentrate}})(\frac{\%}{100\%})\ (\frac{L_{pulp}}{x\ L_{solution}})} \right]$$

$$Copper\ Extraction = 100 \left[\frac{(50)\ (4)\ (24)\ (\frac{1}{1000})}{50\ (\frac{17.3}{100})\ (\frac{1}{0.985})} \right]$$

$$Copper\ Extraction = 55\%$$

Therefore, the theoretical copper extraction was calculated to be 55%.

Sulphate formation

This sample calculation uses data from a 100 ml concentrate slurry sample taken from the third bioleach stage on 94/12/5. This was during Test 1, when the residence time was eight days and the pulp density was 5%. The feed sulphide-sulphur assay was 11.62%.

Data:	Feed	5.00 g	17.3% Cu	0.865 g Cu
	Feed	5.00 g	11.62% S^{2-}	0.581 g sulphide
	Filtrate	94.5 ml	8.35 g/L Cu	0.789 g Cu
	Wash	445 ml	0.104 g/L Cu	0.046 g Cu
	Solids	4.38 g	1.95% Cu	0.085 g Cu
	Total			0.920 g Cu
	Solids	4.38 g	3.45% (S^{-2} + $S°$)	0.151 g (S^{-2} + $S°$)

$$Sulphate\ Formation = 100\ [1 - \frac{(S^{2-} + S^{o})_{solids}(g)}{(Sulphide_{feed}(g))\ (\frac{Cu_{actual}(g)}{Cu_{feed}(g)})}]$$

$$Sulphate\ Formation = 100\ [1 - \frac{0.151}{(0.581)\ (\frac{0.920}{0.865})}]$$

$$Sulphate\ Formation = 76\%$$

Therefore, the sulphate formation was determined to be 76%. The average of three 100 ml slurry samples analyzed during this steady state period was 71% sulphate formation.

Overall Sulphate Formation Based on Chemistry

This is a sample calculation to check the validity of the sulphate formation results. This calculation uses data from Test 4 of the concentrate continuous reactor bioleaching. The copper extraction based on mass balance analyses of the 100 mL slurry samples was found to be 83% and there was found to be no sulphate formation.

Data:	Total copper assay	= 17.3%
	Acid soluble copper assay	= 1.60%
	Sulphide sulphur assay	= 11.62%
	Total iron assay	= 11.6%
	Copper molecular weight	= 63.546 g/mol
	Sulphur molecular weight	= 32.06 g/mol

Assume the difference between total copper and acid soluble copper (15.7%) is chalcocite-copper.

For a 100 g sample of dry concentrate feed:

mass copper	= 15.7 g
moles copper	= 15.7/63.546
	= 0.247 mol
moles chalcocite	= 0.247/2
	= 0.1235 mol
moles sulphide as chalcocite	= 0.1235 mol
mass sulphide as chalcocite	= 0.1235 (32.06)
	= 3.96 g

total mass sulphide in 100 g   = 11.62 g
percent of total sulphide as chalcocite   = 100 (3.96/11.62)
                                                = 34%

From the chemistry:

$$Cu_2S + 0.5O_2 + H_2SO_4 \rightarrow CuS + CuSO_4 + H_2O \tag{3}$$

$$CuS + 2O_2 \rightarrow CuSO_4 \tag{4}$$
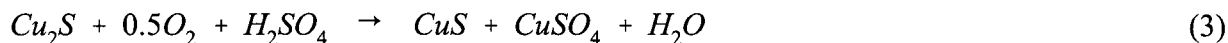
Assume that all the chalcocite oxidizes before any covellite oxidation occurs. Therefore, half the total copper is extracted before any sulphide oxidation occurs. Then, to achieve overall 83% copper extraction, a further 33% copper extraction is required. This corresponds to 66% covellite oxidation. If 66% covellite oxidation occurs, then 66% of the chalcocite-sulphide must be oxidized to elemental sulphur or sulphate. As calculated above, the chalcocite-sulphide is 34% of the total sulphide in the feed. Assume that no pyrite oxidation occurs. Let X represent the overall sulphide oxidation.

pyrite-sulphide   +   chalcocite-sulphide   =   total sulphide

(0%) (66%)   +   (66%) (34%)   =   X (100%)

X   =   22%

Therefore, it is possible to have 83% copper extraction and only 22% total sulphide oxidation, assuming no pyrite oxidation occurs. It was found that there was no overall sulphate formation for Test 4 of the concentrate continuous bioleaching experiment. Therefore, the sulphide must have been oxidized to elemental sulphur.

Summary of Shake Flask Bioleach Experiment

Expt. Name: R24
Expt Type: Bioleach - 9K
Ore: Ore
Cl- Level: 3 gpl
Fe(2+) Level: ~9 gpl
Revision Date: April 16/95

INPUTS:

Solids	Weight (g)	Wt% Cu	Wt% Fe	Wt% S-2		Wt Cu (g)	Wt Fe (g)	Wt S-2 (g)
Ore	8.02	1.56	2.66	0.81		0.125	0.213	0.065

Solutions	Volume(mL)	gpl Cu	gpl Fe		Wt Cu (g)	Wt Fe (g)
Inoculum	5	1.72	3.75		0.009	0.019
Media	70	0	9.35		0.000	0.655
Acid	0.25	NA	NA		NA	NA
211 gpl NaCl	1.726	NA	NA		NA	NA
System Volume (neglecting acid):	76.726			Total IN (g):	0.134	0.887	0.065

OUTPUTS:

Solids	Weight (g)	Wt% Cu	Wt% Fe	Wt% (S-2+S°)		Wt Cu (g)	Wt Fe (g)	Wt% (S-2+S°) (g)
Leach Residue	9.6300	0.109	6.840	0.11		0.010	0.659	0.011

Solutions Sample #	Volume(mL)	gpl Cu	gpl Fe		Wt Cu (g)	Wt Fe (g)
1	1	0.91	8.5		0.0009	0.00850
2	1	1.32	7.05		0.0013	0.00705
3	1	1.34	4.18		0.0013	0.00418
4	1	1.44	3.55		0.0014	0.00355
Filtrate	75	1.47	3.06		0.1103	0.22950
Wash	265	0.1000	0.083		0.0265	0.02200
				Total OUT (g):	0.152	0.933	0.011

	Input - Output:	-0.019	-0.047
	% Lost of input:	-13.87%	-5.29%

Acid Addition Summary
Volume 6M H2SO4 added (ml):   0.25
Acid Consumption (kg H2SO4/tonne conc):   18.34

Copper Extraction Summary
Copper Extraction based on Outputs (%):   92.7
Calculated Head Copper (%):   1.79

Sulphate Formation Summary
Sulphate Formation (%):   83.7

APPENDIX B - Selected Shake Flask Data

SHAKEFLASK BIOLEACH SAMPLE LOG:

R24, 8g ore, ~9K medium, 3 g/L Cl-
Zaldivar ORE [particle size = P80 108 microns]
T = 35 °C

* Eh value includes +199 mV [E(Pt combination electrode) vs. SHE] under saturated Cl- conditions

System Weight (g) = 205.11
Weight of Ore (g) = 8.02
Inoc Vol (ml): 5
Media Vol (ml): 70

	Cu	Fe
Innoculum (g/L)	1.72	3.75
Media (g/L)	0	9.35
Ore (wt %)	1.56	2.66

Calculated Head (%): 1.79

daily weight flask (g)	total vol. soln. (ml)	Corrected for Dilution Cu in soln. (g/l)	Fe in soln. (g/l)	% Cu extracted	% Fe in soln.	Calc. Head % Cu extracted	Rate of Cu Extract. (mg/L/h)	date	Delta Time (hrs)	Total Leach Time (days)	pH	Eh* (mV)	Sample gpl Cu	Sample gpl Fe	6 M H2SO4 added (ml)	211 g/l NaCl added (ml)
205.11	76.73	0.91	8.50	48.9	73.6	42.6		June 1/94	0	0.00	1.89	565	0.91	8.5	0.20	1.726
206.66	78.28	1.35	7.19	76.4	62.2	66.6	9.1	June 3/94	47.75	1.99	1.94	629	1.32	7.05	0.05	0.00
206.86	78.48	1.37	4.28	79.0	37.0	68.8	0.9	June 4/94	27.00	3.11	1.74	825	1.34	4.18	0.00	0.00
205.11	76.73	1.44	3.55	84.3	30.7	73.4	1.5	June 6/94	45.25	5.00	1.54	865	1.44	3.55	0.00	0.00
205.11	76.73	1.47	3.06	87.3	26.5	76.0	0.4	June 9/94	82.75	8.45	1.54	885	1.47	3.06	0.00	0.00
												Totals:			0.25	1.726

Summary of Shake Flask Bioleach Experiment

Expt. Name: C52
Expt Type: Bioleach (35°C)
Substrate: Concentrate
Cl- Level: 3 gpl
Fe(2+) Level: ~9 gpl
Revision Date: April 15/95

INPUTS:

Solids	Weight (g)	Wt% Cu	Wt% Fe	Wt% S-2		Wt Cu (g)	Wt Fe (g)	Wt S-2 (g)
Ore	2.04	17.30	11.60	11.62		0.353	0.237	0.237

Solutions	Volume(mL)	gpl Cu	gpl Fe		Wt Cu (g)	Wt Fe (g)	
Inoculum	5	1.71	3.88		0.009	0.019	
Media	70	0.030	9.65		0.002	0.676	
Acid	0.55	NA	NA		NA	NA	
211 gpl NaCl	1.726	NA	NA		NA	NA	

System Volume (neglecting acid):	76.726			Total IN (g):	0.364	0.932	0.237

OUTPUTS:

Solids	Weight (g)	Wt% Cu	Wt% Fe	Wt% (S-2+S°)		Wt Cu (g)	Wt Fe (g)	Wt% (S-2+S°) (g)
Leach Residue	3.0500	0.483	17.400	2.37		0.015	0.531	0.072

Solutions Sample #	Volume(mL)	gpl Cu	gpl Fe		Wt Cu (g)	Wt Fe (g)
1	1	1.85	8.85		0.0019	0.00885
2	1	3.88	6.75		0.0039	0.00675
3	1	4.2	4.8		0.0042	0.00480
4	1	4.3	4.73		0.0043	0.00473
5	1	4.3	4.68		0.0043	0.00468
Filtrate	75	4.23	4.63		0.3173	0.34725
Wash	300	0.0720	0.0656		0.0216	0.01968

Total OUT (g):		0.372	0.927	0.072

Input - Output:	-0.008542	0.0041	
% Lost of input:	-2.35%	0.44%	

Acid Addition Summary
Volume 6M H2SO4 added (ml):   0.55
Acid Consumption (kg H2SO4/tonne conc):   158.66

Copper Extraction Summary
Copper Extraction based on Outputs (%):   95.9
Calculated Head Copper (%):   17.7

Sulphate Formation Summary
Sulphate Formation (%):   69.5

Appendix B - Selected Shake Flask Data

SHAKE FLASK BIOLEACH SAMPLE LOG:

C52, 2g conc, ~9K medium, 3 g/L Cl-
Zaldivar ORE [particle size = P80 108 microns]
T = 35 °C

System Weight (g): 167.4
Substrate Weight (g): 2.04
Inoculum Volume (ml): 5
Media Volume (ml): 70

* Eh value includes +199 mV [E(Pt combination electrode) vs. SHE] under saturated Cl- conditions

	Cu	Fe
Inoculum Conc'n (g/L):	1.71	3.88
Media Conc'n (g/L):	0.03	9.65
Substrate (wt %):	17.3	11.6

Calculated Head (%): 17.7

daily weight flask (g)	total vol. soln. (ml)	Corrected for Dilution Cu in soln. (g/l)	Fe in soln. (g/l)	% Cu extracted	% Fe in soln.	Calc. Head % Cu extracted	Rate of Cu Extrac. (mg/L/h)	date	Delta Time (hrs)	Total Leach Time (days)	pH	Eh* (mV)	Sample gpl Fe	6 M H2SO4 added (ml)	211 g/l NaCl added (ml)
167.40	76.73	1.85	8.85	37.2	72.9	36.3		May 18/94	0	0.00	1.75	551	8.85	0.45	1.726
168.33	77.66	3.93	6.83	82.9	56.3	80.9	26.9	May 21/94	77.25	3.22	1.85	653	6.75	0.10	0.00
167.40	76.73	4.20	4.80	89.9	39.5	87.8	6.2	May 23/94	44.25	5.06	1.64	796	4.8	0.00	0.00
167.40	76.73	4.30	4.73	93.3	39.0	91.1	4.3	May 24/94	23.00	6.02	1.56	820	4.73	0.00	0.00
167.40	76.73	4.30	4.68	94.5	38.5	92.3	0.0	May 26/94	41.75	7.76	1.53	830	4.68	0.00	0.00
167.40	76.73	4.23	4.63	94.2	38.1	92.0	-1.2	May 28/94	57.50	10.16	1.46	850	4.63	0.00	0.00
													Totals:	0.55	1.726

Summary of Shake Flask Bioleach Experiment

Expt. Name: C52ST
Expt Type: Sterile (35°C)
Substrate Concentrate
Cl- Level: 3 gpl
Fe(2+) Level: ~9 gpl
Revision Date: April 15/95

INPUTS:

Solids	Weight (g)	Wt% Cu	Wt% Fe	Wt% S-2		Wt Cu (g)	Wt Fe (g)	Wt S-2 (g)
Ore	2.03	17.30	11.60	11.62		0.351	0.235	0.236

Solutions	Volume(mL)	gpl Cu	gpl Fe		Wt Cu (g)	Wt Fe (g)	
Inoculum	5	0	0		0.000	0.000	
Media	70	0.03	9.65		0.002	0.676	
Acid	0.7	NA	NA		NA	NA	
211 gpl NaCl	1.726	NA	NA		NA	NA	
System Volume (neglecting acid):	76.726			Total IN (g):	0.353	0.911	0.236

OUTPUTS:

Solids	Weight (g)	Wt% Cu	Wt% Fe	Wt% (S-2+S°)		Wt Cu (g)	Wt Fe (g)	Wt% (S-2+S°) (g)
Leach Residue	1.7300	6.030	13.400	14.8		0.104	0.232	0.256

Solutions Sample #	Volume(mL)	gpl Cu	gpl Fe		Wt Cu (g)	Wt Fe (g)	
1	1	1.8	8.68		0.0018	0.00868	
2	1	2.3	8.45		0.0023	0.00845	
3	1	2.53	8.3		0.0025	0.00830	
4	1	2.68	8.3		0.0027	0.00830	
5	1	2.89	8.25		0.0029	0.00825	
Filtrate	76	3.05	7.85		0.2318	0.59660	
Wash	300	0.0410	0.103		0.0123	0.03090	
Total OUT (g):					0.361	0.901	0.256
Input - Output:					-0.007329	0.00968	
% Lost of input:					-2.07%	1.06%	

Acid Addition Summary
Volume 6M H2SO4 added (ml): 0.70
Acid Consumption (kg H2SO4/tonne conc): 202.92

Copper Extraction Summary
Copper Extraction based on Outputs (%): 70.9
Calculated Head Copper (%): 17.7

Sulphate Formation Summary
Sulphate Formation (%): 0

Appendix B - Selected Shake Flask Data

SHAKE FLASK BIOLEACH SAMPLE LOG:

C52ST, 2g conc, ~9K medium, 3 g/L Cl-
Zaldivar ORE [particle size = P80 108 microns]
T = 35 °C

* Eh value includes +199 mV [E(Pt combination electrode) vs. SHE] under saturated Cl- conditions

System Weight (g):	204.18
Substrate Weight (g):	2.03
Inoculum Volume (ml):	5
Media Volume (ml):	70

	Cu	Fe
	0	0
Inoculum Conc'n (g/L):		
Media Conc'n (g/L):	0.03	9.65
Substrate (wt %):	17.3	11.6

Calculated Head (%): 17.7

daily weight flask (g)	total vol. soln. (ml)	Corrected for Dilution Cu in soln. (g/l)	Fe in soln. (g/l)	% Cu extracted	% Fe in soln.	Calc. Head % Cu extracted	Rate of Cu Extract. (mg/L/h)	date	Delta Time (hrs)	Total Leach Time (days)	pH	(mV) Eh *	Sample gpl Cu	Sample gpl Fe	6 M H2SO4 added (ml)	211 g/l NaCl added (ml)
204.18	76.73	1.80	8.68	38.7	73.1	37.9		May 18/94	0	0.00	1.75	550	1.8	8.68	0.45	1.726
206.07	78.62	2.36	8.66	51.4	72.9	50.4	7.2	May 21/94	77.25	3.22	1.94	587	2.3	8.45	0.10	0.00
206.39	78.94	2.60	8.54	57.4	71.9	56.3	5.6	May 23/94	44.25	5.06	1.92	591	2.53	8.3	0.10	0.00
204.18	76.73	2.68	8.30	59.8	69.9	58.6	3.4	May 24/94	23.00	6.02	1.94	594	2.68	8.3	0.00	0.00
204.18	76.73	2.89	8.25	65.2	69.5	63.9	5.0	May 26/94	41.75	7.76	1.94	596	2.89	8.25	0.05	0.00
204.18	76.73	3.05	7.85	69.5	66.1	68.1	2.8	May 28/94	57.50	10.16	2.05	598	3.05	7.85	0.00	0.00
													Totals:		0.7	1.726

Summary of Batch Reactor Bioleach Experiment

Expt. Name: BO11
Expt Type: Bioleach - pulp density 10%
Ore: Ore
Cl- Level: 3 gpl
Fe(2+) Level: ~9 gpl
Revision Date: April 15/95

INPUTS:

Solids	Weight (g)	Wt% Cu	Wt% Fe	Wt% S-2		Wt Cu (g)	Wt Fe (g)	Wt S-2 (g)
Ore	199.40	1.56	2.66	0.81		3.111	5.304	1.615

Solutions	Volume(mL)	gpl Cu	gpl Fe			Wt Cu (g)	Wt Fe (g)	
Inoculum	130	1.81	4.6			0.235	0.598	
Media	1820	0	9.06			0.000	16.489	
Acid	3.9	NA	NA			NA	NA	
211 gpl NaCl	44	NA	NA			NA	NA	
System Volume (neglecting acid):	1994				Total IN (g):	3.346	22.391	1.615

OUTPUTS:

Solids	Weight (g)	Wt% Cu	Wt% Fe	Wt% (S-2+S°)		Wt Cu (g)	Wt Fe (g)	Wt% (S-2+S°) (g)
Leach Residue	236.4900	0.150	6.120	0.24		0.355	14.473	0.568

Solutions Sample #	Volume(mL)	gpl Cu	gpl Fe			Wt Cu (g)	Wt Fe (g)	
1	1	0.835	8.94			0.0008	0.00894	
2	1	1.14	8.63			0.0011	0.00863	
3	1	1.32	5.65			0.0013	0.00565	
4	1	1.36	4.65			0.0014	0.00465	
5	1	1.37	4.28			0.0014	0.00428	
Filtrate	1970	1.39	4.075			2.7383	8.02775	
Wash	2090	0.0264	0.113			0.0551	0.23617	
					Total OUT (g):	3.154	22.769	0.568
					Input - Output:	0.192	-0.378	
					% Lost of input:	5.73%	-1.69%	

Acid Addition Summary
Volume 6M H2SO4 added (ml):   3.90
Acid Consumption (kg H2SO4/tonne conc):   11.51

Copper Extraction Summary
Copper Extraction based on Outputs (%):   87.8
Calculated Head Copper (%):   1.46

Sulphate Formation Summary
Sulphate Formation (%):   64.9

Appendix C - Selected Batch Reactor Data

BATCH REACTOR BIOLEACH SAMPLE LOG:

BO11, 100 g/L ore, ~9K medium, 3 g/L Cl-
Zaldivar ORE [particle size = P80 108 microns]
T = 35 °C

* Eh value includes +199 mV [E(Pt combination electrode) vs. SHE] under saturated Cl- conditions

System Volume (ml) = 1994
Weight of Ore (g) = 199.40
Inoc. Volume (ml) = 130
Media Volume (ml) = 1820

	Cu	Fe
Inoc. Conc'n (g/L) =	1.81	4.6
Media Conc'n (g/L) =	0	9.06
Ore (wt %) =	1.56	2.66

Calculated Head (%) = 1.46

daily volume tank (ml)	total vol. soln. (ml)	Corrected for Dilution Cu in soln. (g/l)	Fe in soln. (g/l)	% Cu extracted	% Fe in soln.	Calc. Head % Cu extracted	Rate of Cu Extrac. (mg/L/hr)	date	Delta Time (hrs)	Total Leach Time (days)	pH	(mV) Eh *	Sample gpl Cu	Sample gpl Fe	6 M H2SO4 added (ml)	211 g/l NaCl added (ml)
1994.00	1994.00	0.84	8.94	46.0	79.6	49.0		Aug 18/94	0	0.00	2.12	567	0.835	8.94	3.90	44.000
1994.00	1994.00	1.14	8.63	65.5	76.9	69.8	7.1	Aug 20/94	43.25	1.80	1.91	606	1.14	8.63	0.00	0.00
1994.00	1994.00	1.32	5.65	77.1	50.3	82.2	4.1	Aug 22/94	43.50	3.61	1.89	819	1.32	5.65	0.00	0.00
1994.00	1994.00	1.36	4.65	79.7	41.4	85.0	1.7	Aug 23/94	23.50	4.59	1.65	861	1.36	4.65	0.00	0.00
1994.00	1994.00	1.37	4.28	80.4	38.1	85.7	0.4	Aug 24/94	24.00	5.59	1.58	873	1.37	4.28	0.00	0.00
1994.00	1994.00	1.39	4.08	81.7	36.3	87.1	0.8	Aug 25/94	24.00	6.59	1.56	880	1.39	4.075	0.00	0.00
													Totals:		3.90	44.000

Appendix C - Selected Batch Reactor Data

Summary of Bioleach Batch Experiment

Expt. Name: BC12
Expt. Type: pulp density 7.5%
Ore: Concentrate
Cl- Level: 3 gpl
Fe(2+) Level: ~9 gpl
Revision Date: April 14/95

INPUTS:

Solids

	Weight (g)	Wt% Cu	Wt% Fe	Wt% S-2	Wt Cu (g)	Wt Fe (g)	Wt S-2 (g)
Ore	149.60	17.30	11.60	11.62	25.881	17.354	17.384

Solutions

	Volume(mL)	gpl Cu	gpl Fe		Wt Cu	Wt Fe	Wt S-2
Inoculum	130.00	1.77	3.88		0.230	0.504	
Media	1820.00	0	9.03		0.000	16.435	
Acid	41.4	NA	NA		NA	NA	
211 gpl NaCl	44	NA	NA		NA	NA	

System Volume (neglecting acid): 1994

Total IN (g): 26.111 | 34.293 | 17.384

OUTPUTS:

Solids

	Weight (g)	Wt% Cu	Wt% Fe	Wt% (S-2+S°)	Wt Cu (g)	Wt Fe (g)	Wt (S-2+S°) (g)
Leach Residue	152.8600	0.456	14.700	2.15	0.697	22.470	3.286

Solutions

Sample #	Volume(mL)	gpl Cu	gpl Fe			
1	1	1.74	6.9		0.0017	0.00690
2	1	3.98	5.8		0.0040	0.00580
3	1	4.98	5.95		0.0050	0.00595
4	1	5.26	6.15		0.0053	0.00615
5	1	5.42	6.35		0.0054	0.00635
6	1	5.85	6.35		0.0059	0.00635
7	1	6.2	6.5		0.0062	0.00650
8	1	6.45	6.9		0.0065	0.00690
9	1	6.7	6.8		0.0067	0.00680
10	1	6.95	6.85		0.0070	0.00685
11	1	7.65	6.95		0.0077	0.00695
12	1	8.5	6.78		0.0085	0.00678
13	1	10.6	3.55		0.0106	0.00355
14	1	11.3	3.98		0.0113	0.00398
15	1	11.8	4.7		0.0118	0.00470
16	1	12	5.33		0.0120	0.00533
17	1	12.5	5.7		0.0125	0.00570
18	1	12.5	6.15		0.0125	0.00615
Filtrate	1995	12	6.35		23.9400	12.66825
Wash1	3680	0.187	0.065		0.6882	0.24472
Wash2	1905	0.174	0.0603		0.3315	0.11487
Wash3	1897	0.15	0.0485		0.2846	0.09200

Total OUT (g): 26.082 | 35.698 | 3.286

Input - Output: 0.029298 | -1.405356

% Lost of input: 0.11% | -4.10%

Acid Addition Summary
Volume 6M H2SO4 added (ml): 41.40
Acid Consumption (kg H2SO4/tonne conc): 162.85

Copper Extraction Summary
Copper Extraction based on Outputs (%): 97.3
Calculated Head Copper (%): 17.28

Sulphate Formation Summary
Sulphate Formation (%): 81.1

Appendix C - Selected Batch Reactor Data

BIOLEACH SAMPLE LOG:
BC12, 75 g/L concentrate, ~9K medium, 3 g/L Cl-
Zaldivar Concentrate [particle size = P80 53 microns]
T = 35 °C

* Eh value includes +199 mV [E(Pt combination electrode) vs. SHE] under saturated Cl- conditions

Constant Volume (ml): 1944
Substrate Weight (g): 149.60
Inoculum Volume (ml): 130
Media Volume (ml): 1820

	Cu	Fe
Inoculum Conc'n (g/L):	1.77	3.88
Media Conc'n (g/L):	0	9.03
Substrate (wt %):	17.3	11.6

Calculated Head (%): 17.28

daily volume tank (ml)	total vol. soln. (ml)	Corrected for Dilution Cu in soln. (g/l)	Fe in soln. (g/l)	% Cu extracted	% Fe in soln.	Calc. Head % Cu extracted	Rate of Cu Extrac. (mg/L/hr)	date	Delta Time (hrs)	Total Leach Time (days)	pH	(mV) Eh *	Sample gpl Cu	Sample gpl Fe	6 M H2SO4 added (ml)	211 g/l NaCl added (ml)
1944	1944	1.74	6.90	12.2	39.1	12.2		Aug 18/94	0	0.00	3.42	498	1.74	6.9	10.40	44.000
1944	1944	3.98	5.80	29.0	32.9	29.0	51.8	Aug 20/94	43.25	1.80	3.08	517	3.98	5.8	8.00	0.00
1944	1944	4.98	5.95	36.5	33.7	36.6	23.3	Aug 22/94	43.00	3.59	2.63	554	4.98	5.95	5.00	0.00
1944	1944	5.26	6.15	38.7	34.9	38.7	11.7	Aug 23/94	24.00	4.59	2.29	569	5.26	6.15	3.00	0.00
1944	1944	5.42	6.35	39.9	36.0	39.9	6.7	Aug 24/94	24.00	5.59	2.20	577	5.42	6.35	4.00	0.00
1944	1944	5.85	6.35	43.1	36.0	43.2	9.1	Aug 26/94	47.50	7.57	2.17	582	5.85	6.35	4.00	0.00
1944	1944	6.20	6.50	45.8	36.8	45.8	4.8	Aug 29/94	72.25	10.58	2.14	586	6.2	6.5	5	0.00
1944	1944	6.45	6.90	47.7	39.1	47.7	5.2	Aug 31/94	48.00	12.58	1.76	588	6.45	6.9	0.00	0.00
1944	1944	6.70	6.80	49.6	38.5	49.6	10.5	Sept 1/94	23.75	13.57	1.84	591	6.7	6.8	0	0
1944	1944	6.95	6.85	51.5	38.8	51.6	10.4	Sept 2/94	24.00	14.57	1.82	589	6.95	6.85	0	0
1944	1944	7.65	6.95	56.8	39.4	56.8	9.6	Sept 5/94	73.25	17.63	1.89	596	7.65	6.95	0	0
1944	1944	8.50	6.78	63.2	38.4	63.3	17.8	Sept 7/94	47.75	19.61	2.02	612	8.5	6.78	2	0
1944	1944	10.60	3.55	79.0	20.1	79.1	44.2	Sept 9/94	47.50	21.59	1.77	763	10.6	3.55	0	0
1944	1944	11.30	3.98	84.3	22.6	84.4	12.5	Sept 11/94	56.00	23.93	1.53	820	11.3	3.98	0	0
1944	1944	11.80	4.70	88.1	26.6	88.2	11.8	Sept 13/94	42.25	25.69	1.52	833	11.8	4.7	0	0
1944	1944	12.00	5.33	89.6	30.2	89.7	4.2	Sept 15/94	47.50	27.67	1.47	840	12	5.33	0	0
1944	1944	12.50	5.70	93.4	32.3	93.6	17.2	Sept 16/94	29.00	28.88	1.44	828	12.5	5.7	0	0
1944	1944	12.50	6.15	93.5	34.9	93.6	0.0	Sept 19/94	66.00	31.63	1.42	866	12.5	6.15	0	0
1944	1944	12.00	6.35	89.8	36.0	89.9	-21.5	Sept 20/94	23.25	32.59	1.4	868	12	6.35	0	0
												Totals:			41.40	44.00

Appendix D - Continuous Reactor Data

ORE CONTINUOUS BIOLOGICAL LEACHING - FEED TANK

Day	Date	Time	Hours	Cumul. Hours	Cumul. Days	Target Flowrate (L/day)	pH	Eh (mV)	[Cu] (g/L)	[Fe] (g/L)	Comments
0	21-Oct-94	11:15 AM	0.0	0.0	0.0						batch mode;
1	22-Oct-94	01:00 PM	25.8	25.8	1.1						pulp density 10%
3	24-Oct-94	11:00 AM	46.0	71.8	3.0						
5	26-Oct-94	11:30 AM	48.5	120.3	5.0						
7	28-Oct-94	11:00 AM	47.5	167.8	7.0	1					continuous mode 1L/day
9	30-Oct-94	08:30 PM	57.5	225.3	9.4	1			0.715	2.530	
11	1-Nov-94	11:30 AM	39.0	264.3	11.0	1			0.715	2.530	
12	2-Nov-94	09:45 AM	22.3	286.5	11.9	1	3.07	544	0.715	2.530	
13	3-Nov-94	11:00 AM	25.3	311.8	13.0	1	3.07	544	0.715	2.530	Nov 4 FILL
15	5-Nov-94	01:00 PM	50.0	361.8	15.1	1	3.73	488	0.490	2.270	
16	6-Nov-94	03:40 PM	26.7	388.4	16.2	1	3.46	503	0.615	2.130	
17	7-Nov-94	11:55 AM	20.3	408.7	17.0	1	3.32	516	0.650	2.070	
18	8-Nov-94	01:15 PM	25.3	434.0	18.1	1	3.32	516	0.650	2.070	
20	10-Nov-94	09:35 AM	44.3	478.3	19.9	1	3.05	543	0.690	1.880	
22	12-Nov-94	03:15 PM	53.7	532.0	22.2	1	3.00	535	0.750	1.910	
24	14-Nov-94	03:35 PM	48.3	580.3	24.2	1	2.92	501	0.740	1.860	FILL
25	15-Nov-94	02:05 PM	22.5	602.8	25.1	1	2.92	501	0.740	1.860	Nov 17 FILL
29	19-Nov-94	04:35 PM	98.5	701.3	29.2	1	2.92	501	0.740	1.860	FILL, 11/16 feed tube plug
31	21-Nov-94	01:10 PM	44.6	745.9	31.1	1	3.63	486	0.440	2.040	11/20 feed tube plugged
35	25-Nov-94	01:15 PM	96.1	842.0	35.1	1	3.28	515	0.680	1.730	Nov 27 FILL
38	28-Nov-94	01:35 PM	72.3	914.3	38.1	1	3.64	498	0.755	4.800	11/26 feed tube plugged
40	30-Nov-94	12:45 PM	47.2	961.5	40.1	1	3.43	502	0.620	1.960	
41	1-Dec-94	12:45 PM	24.0	985.5	41.1	1	3.43	502	0.620	1.960	
43	3-Dec-94	12:45 PM	48.0	1033.5	43.1	1	3.25	518	0.690	1.710	slurry sample at 3:20pm
44	4-Dec-94	12:10 PM	23.4	1056.9	44.0	1	3.25	518	0.690	1.710	FILL
45	5-Dec-94	12:05 PM	23.9	1080.8	45.0	1	3.72	486	0.375	2.180	
47	7-Dec-94	11:25 AM	47.3	1128.2	47.0	1	3.57	501	0.580	1.990	new flowrate 2L/day
49	9-Dec-94	11:40 AM	48.3	1176.4	49.0	2	3.44	514	0.600	2.000	Dec 10 FILL, 12/12 feed tube plug
52	12-Dec-94	12:15 PM	72.6	1249.0	52.0	2	3.66	507	0.500	2.400	slurry sample at 2:15pm
54	14-Dec-94	11:25 AM	47.2	1296.2	54.0	2	3.40	516	0.600	2.100	FILL, 12/13 feed tube plug
56	16-Dec-94	11:25 AM	48.0	1344.2	56.0	2	3.66	509	0.500	2.400	12/15 feed tube plugged
59	19-Dec-94	01:10 PM	73.8	1417.9	59.1	2	3.30	527	0.700	2.000	
60	20-Dec-94	11:40 AM	22.5	1440.4	60.0	2	3.30	527	0.700	2.000	FILL
62	22-Dec-94	12:20 PM	48.7	1489.1	62.0	2	3.66	509	0.400	2.300	
63	23-Dec-94	10:30 AM	22.2	1511.3	63.0	2	3.67	483	0.528	2.030	Dec 24 FILL, 12/25 stirrers off
67	27-Dec-94	02:35 PM	100.1	1611.3	67.1	2	3.58	508	0.520	2.040	12/25 compressor down
68	28-Dec-94	11:15 AM	20.7	1632.0	68.0	2	3.58	508	0.520	2.040	FILL
69	29-Dec-94	01:40 PM	26.4	1658.4	69.1	2	3.72	497	0.323	2.280	

Appendix D - Continuous Reactor Data

ORE CONTINUOUS BIOLOGICAL LEACHING - FEED TANK

Day	Date	Time	Hours	Cumul. Hours	Cumul. Days	Target Flowrate (L/day)	pH	Eh (mV)	[Cu] (g/L)	[Fe] (g/L)	Comments
70	30-Dec-94	10:50 AM	21.2	1679.6	70.0	2	3.69	504	0.438	2.210	slurry sample at 11:20am
71	31-Dec-94	03:20 PM	28.5	1708.1	71.2	2	3.69	504	0.438	2.210	
72	1-Jan-95	02:40 PM	23.3	1731.4	72.1	2	3.45	515	0.605	1.950	Jan 2 FILL
74	3-Jan-95	11:30 AM	44.8	1776.3	74.0	3	3.45	515	0.315	2.200	new flowrate 3L/day
76	5-Jan-95	10:50 AM	47.3	1823.6	76.0	3	3.98	465	0.198	2.480	FILL, 01/03 slurry at 2:05pm
78	7-Jan-95	11:50 AM	49.0	1872.6	78.0	3	3.98	465	0.198	2.480	FILL
79	8-Jan-95	12:30 PM	24.7	1897.3	79.1	3	3.98	465	0.198	2.480	
80	9-Jan-95	12:10 PM	23.7	1920.9	80.0	3	3.64	498	0.415	2.160	
81	10-Jan-95	12:15 PM	24.1	1945.0	81.0	3	3.64	498	0.415	2.160	FILL
83	12-Jan-95	10:10 AM	45.9	1990.9	83.0	3	3.65	500	0.475	2.030	FILL, slurry sample at 2:20pm
84	13-Jan-95	11:05 AM	24.9	2015.8	84.0	3	3.65	500	0.475	2.030	
85	14-Jan-95	12:50 PM	25.8	2041.6	85.1	3	3.65	496	0.415	2.210	FILL
86	15-Jan-95	02:05 PM	25.3	2066.8	86.1	3	3.65	496	0.415	2.210	
87	16-Jan-95	12:10 PM	22.1	2088.9	87.0	3	3.47	493	0.483	2.170	FILL, slurry sample at 1:25pm
88	17-Jan-95	12:15 PM	24.1	2113.0	88.0	3	3.47	493	0.483	2.170	feed tube plugged
89	18-Jan-95	01:30 PM	25.3	2138.3	89.1	3	3.47	493	0.483	2.170	
90	19-Jan-95	03:30 PM	26.0	2164.3	90.2	3	3.47	493	0.483	2.170	FILL
91	20-Jan-95	04:45 PM	25.3	2189.5	91.2	3	3.47	493	0.483	2.170	
92	21-Jan-95	01:10 PM	20.4	2209.9	92.1	3	3.59	498	0.408	2.100	
93	22-Jan-95	02:30 PM	25.3	2235.3	93.1	3	3.59	498	0.408	2.100	FILL
94	23-Jan-95	12:30 PM	22.0	2257.3	94.1	3	3.59	498	0.408	2.100	
95	24-Jan-95	11:50 AM	23.3	2280.6	95.0	3	3.59	498	0.408	2.100	FILL, new pulp density 10%
96	25-Jan-95	10:05 AM	22.3	2302.8	96.0	3	3.77	470	0.328	2.200	
97	26-Jan-95	03:50 PM	29.8	2332.6	97.2	3	3.77	470	0.328	2.200	
98	27-Jan-95	02:15 PM	22.4	2355.0	98.1	3	3.77	470	0.328	2.200	FILL, feed tube plugged
99	28-Jan-95	11:20 AM	21.1	2376.1	99.0	3	3.75	467	0.420	2.450	slurry sample at 1:45pm
100	29-Jan-95	11:55 AM	24.6	2400.7	100.0	3	3.75	467	0.420	2.450	
101	30-Jan-95	01:30 PM	25.6	2426.3	101.1	3	3.75	467	0.420	2.450	FILL
102	31-Jan-95	10:05 AM	20.6	2446.8	102.0	3	3.69	469	0.545	1.930	
103	1-Feb-95	12:30 PM	26.4	2473.3	103.1	3	3.69	469	0.545	1.930	feed tube plugged
104	2-Feb-95	02:15 PM	25.8	2499.0	104.1	3	3.69	469	0.545	1.930	
105	3-Feb-95	11:30 AM	21.3	2520.3	105.0	3	3.69	469	0.545	1.930	
106	4-Feb-95	12:10 PM	24.7	2544.9	106.0	3	3.65	490	0.870	1.430	
107	5-Feb-95	12:50 PM	24.7	2569.6	107.1	3	3.65	490	0.870	1.430	FILL
108	6-Feb-95	11:45 AM	22.9	2592.5	108.0	3	3.66	491	0.493	2.030	
109	7-Feb-95	11:00 AM	23.3	2615.8	109.0	3	3.71	484	0.745	1.720	slurry sample at 2:05pm
110	8-Feb-95	12:55 PM	25.9	2641.7	110.1	3	3.59	492	0.985	1.440	slurry sample at 2:55pm

Appendix D - Continuous Reactor Data

ORE CONTINUOUS BIOLOGICAL LEACHING - BIOLEACH STAGE 1

Day	Date	Cumul. Days	Actual Flowrate (L/day)	pH	Eh (mV)	[Cu] (g/L)	[Fe] (g/L)	Rate Cu Ext. (mg/L/h)	6M H2SO4 Daily (ml)	Ore Daily (kg)	H2SO4 Daily (kg/t)	Comments
0	21-Oct-94	0.0		1.92	560	0.79	2.75			0.380	15	batch mode;
1	22-Oct-94	1.1		2.04	614	0.98	2.75			0.380	3	pulp density 10%
3	24-Oct-94	3.0		1.90	820	1.15	1.54			0.380	0	
5	26-Oct-94	5.0		1.82	834	1.19	1.40			0.380	0	
7	28-Oct-94	7.0		1.89	741	1.22	1.59			0.198	0	continuous mode 1 L/day
9	30-Oct-94	9.4		1.97	838	1.28	1.49	6.9		0.240	0	
11	1-Nov-94	11.0		1.98	836	1.29	1.42	6.2		0.163	0	
12	2-Nov-94	11.9		1.96	832	1.33	1.42	8.2		0.093	0	
13	3-Nov-94	13.0		1.96	832	1.33	1.37	6.4		0.105	0	
15	5-Nov-94	15.1		2.02	830	1.32	1.27	8.4		0.208	0	
16	6-Nov-94	16.2		2.00	828	1.36	1.16	9.3		0.111	0	
17	7-Nov-94	17.0	0.500	2.01	833	1.36	1.08	3.7	0	0.084	0	
18	8-Nov-94	18.1	0.801	2.09	829	1.33	1.02	4.5	1	0.085	7	
20	10-Nov-94	19.9	0.447	1.93	835	1.36	1.23	3.8	8	0.083	57	ore tube plugged
22	12-Nov-94	22.2	0.484	1.89	828	1.43	1.23	4.7	3	0.108	16	
24	14-Nov-94	24.2	0.888	1.95	824	1.33	1.33	3.4	4	0.179	13	
25	15-Nov-94	25.1	0.690	1.91	824	1.30	1.16	2.7	2	0.065	18	
29	19-Nov-94	29.2	0.981	1.80	825	1.05	1.48	0.6	26	0.403	38	
31	21-Nov-94	31.1	1.005	1.96	695	1.12	1.74	8.7	15	0.187	47	ore tube plugged
35	25-Nov-94	35.1	0.975	1.95	795	1.13	1.30	4.7	15	0.390	23	
38	28-Nov-94	38.1	0.798	2.05	772	1.19	1.01	4.4	6	0.241	15	
40	30-Nov-94	40.1	0.993	1.76	759	1.20	1.49	6.2	25	0.195	75	slurry sample at 1:15pm
41	1-Dec-94	41.1	0.682	1.80	796	1.20	1.61	4.1	8	0.068	69	
43	3-Dec-94	43.1	1.240	1.78	799	1.19	1.66	6.3	17	0.248	40	
44	4-Dec-94	44.0	0.939	1.82	783	1.19	1.57	4.9	2	0.092	13	slurry sample at 2:05pm
45	5-Dec-94	45.0	0.928	1.82	779	1.21	1.64	8.9	12	0.093	76	12/07 slurry at 2:05 pm
47	7-Dec-94	47.0	0.961	1.82	780	1.20	1.80	6.0	11	0.189	34	new flowrate 2L/day
49	9-Dec-94	49.0	1.850	1.82	763	1.10	2.00	7.6	30	0.372	47	
52	12-Dec-94	52.0	1.769	1.83	749	1.10	2.10	11.1	47	0.535	52	
54	14-Dec-94	54.0	1.722	1.81	778	1.10	2.10	9.0	25	0.339	43	
56	16-Dec-94	56.0	1.061	1.76	776	1.10	1.80	6.6	15	0.212	42	slurry sample at 3:50pm
59	19-Dec-94	59.1	1.970	1.73	757	1.00	2.00	4.8	45	0.605	44	
60	20-Dec-94	60.0	2.010	1.77	741	1.02	1.89	7.6	11	0.188	34	
62	22-Dec-94	62.0	2.036	1.87	750	0.99	1.81	11.7	29	0.413	41	
63	23-Dec-94	63.0	2.228	1.85	744	0.99	1.78	10.9	11	0.206	31	slurry sample at 10:55am
67	27-Dec-94	67.1	2.007	1.93	723	0.99	1.29	9.7	45	0.837	32	Dec. 25 no air, no stirring
68	28-Dec-94	68.0	2.112	1.79	732	0.99	1.62	10.6	18	0.182	58	slurry sample at 12:55pm
69	29-Dec-94	69.1	0.970	1.83	743	1.06	1.39	10.1	2	0.107	11	

Appendix D - Continuous Reactor Data

ORE CONTINUOUS BIOLOGICAL LEACHING - BIOLEACH STAGE 1

Day	Date	Cumul. Days	Actual Flowrate (L/day)	pH	Eh (mV)	[Cu] (g/L)	[Fe] (g/L)	Rate Cu Ext. (mg/L/h)	6M H2SO4 Daily (ml)	Ore Daily (kg)	H2SO4 Daily (kg/t)	Comments
70	30-Dec-94	70.0	1.465	1.84	727	1.02	1.55	7.0	12	0.129	55	slurry sample at 11:20am
71	31-Dec-94	71.2	1.867	1.80	732	1.02	1.64	11.3	16	0.222	42	
72	1-Jan-95	72.1	1.859	1.79	735	1.01	1.77	7.4	14	0.181	46	slurry sample at 3:05pm
74	3-Jan-95	74.0	2.538	1.77	671	1.03	2.17	19.3	35	0.474	43	new flowrate 3L/day
76	5-Jan-95	76.0	3.120	1.77	670	0.97	2.47	23.8	47	0.615	45	
78	7-Jan-95	78.0	3.054	1.82	657	0.95	2.54	23.5	45	0.623	42	
79	8-Jan-95	79.1	3.037	1.81	651	0.99	2.59	26.7	0	0.312	0	
80	9-Jan-95	80.0	3.250	1.82	657	0.97	2.62	17.9	47	0.320	86	
81	10-Jan-95	81.0	2.153	1.81	677	0.97	2.61	12.1	20	0.216	54	
83	12-Jan-95	83.0	2.899	1.75	685	1.03	2.47	18.2	46	0.555	49	
84	13-Jan-95	84.0	3.044	1.78	681	1.06	2.40	19.8	22	0.316	41	
85	14-Jan-95	85.1	3.098	1.76	697	1.01	2.22	17.3	27	0.332	48	slurry sample at 3:15pm
86	15-Jan-95	86.1	3.039	1.74	702	0.97	2.20	16.0	23	0.320	42	
87	16-Jan-95	87.0	2.455	1.77	730	0.96	2.02	11.7	19	0.226	50	
88	17-Jan-95	88.0	2.232	1.68	730	1.03	1.84	15.6	12	0.224	32	
89	18-Jan-95	89.1	2.965	1.66	744	0.96	1.79	12.0	21	0.312	40	slurry sample at 1:15pm
90	19-Jan-95	90.2	3.062	1.76	749	1.00	1.92	18.0	25	0.332	44	
91	20-Jan-95	91.2	3.154	1.80	743	0.98	2.07	15.5	26	0.332	46	
92	21-Jan-95	92.1	3.261	1.79	749	0.99	1.58	20.3	10	0.277	21	slurry sample at 4:10pm
93	22-Jan-95	93.1	3.005	1.71	748	1.09	1.63	25.3	37	0.317	69	
94	23-Jan-95	94.1	3.061	1.67	752	0.95	1.94	10.9	19	0.281	40	
95	24-Jan-95	95.0	3.009	1.76	756	1.02	1.69	22.2	20	0.585	20	new pulp density 20%
96	25-Jan-95	96.0	3.010	1.75	732	1.36	1.88	47.6	25	0.558	26	replaced acrylic impeller
97	26-Jan-95	97.2	3.127	1.75	708	1.72	2.04	57.4	33	0.775	25	
98	27-Jan-95	98.1	2.289	1.73	792	1.92	1.90	46.9	32	0.428	44	
99	28-Jan-95	99.0	3.211	1.81	640	1.96	1.89	53.4	22	0.564	23	air plugged, acid pump off
100	29-Jan-95	100.0	3.064	1.75	627	2.00	2.41	52.1	33	0.628	31	
101	30-Jan-95	101.1	3.017	1.79	609	2.03	2.64	51.8	36	0.643	33	
102	31-Jan-95	102.0	3.232	1.79	591	1.98	2.79	45.9	39	0.554	41	batch mode
103	1-Feb-95	103.1	1.216	1.83	647	2.89	3.09	64.2	11	0.268	24	
104	2-Feb-95	104.1	0.000	1.83	682	2.89	2.80	0.0	3	0.000		
105	3-Feb-95	105.0	0.000	1.73	752	2.77	2.14	-5.6	0	0.000		continuous mode 3L/day
106	4-Feb-95	106.0	0.000	1.76	732	2.87	1.90	4.1	0	0.000		
107	5-Feb-95	107.1	0.817	1.89	722	2.73	1.78	10.2	9	0.168	32	
108	6-Feb-95	108.0	2.198	1.80	595	2.47	2.38	33.9	31	0.420	43	continuous mode 1L/day
109	7-Feb-95	109.0	1.424	1.86	629	2.47	2.48	25.6	16	0.276	34	slurry sample at 2:05pm
110	8-Feb-95	110.1	0.669	1.83	656	2.56	2.67	14.5	15	0.145	61	slurry sample at 2:55pm

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Appendix D - Continuous Reactor Data

ORE CONTINUOUS BIOLOGICAL LEACHING - BIOLEACH STAGE 2

Day	Date	Time	Hours	Cumul. Days	Actual Flowrate (L/day)	pH	Eh (mV)	[Cu] (g/L)	[Fe] (g/L)	Rate Cu Ext. (mg/L/h)	Comments
0	21-Oct-94	11:15 AM	0.0	0.0		1.90	556	0.76	2.75		batch mode;
1	22-Oct-94	01:00 PM	25.8	1.1		2.07	621	1.04	2.73		pulp density 10%
3	24-Oct-94	11:00 AM	46.0	3.0		1.87	820	1.26	1.49		
5	26-Oct-94	11:30 AM	48.5	5.0		1.80	832	1.27	1.36		
7	28-Oct-94	11:00 AM	47.5	7.0		1.79	845	1.37	1.40		continuous mode 1 L/day
9	30-Oct-94	08:30 PM	57.5	9.4		1.83	847	1.36	1.40	-0.2	
11	1-Nov-94	11:30 AM	39.0	11.0		1.84	848	1.36	1.34	0.0	
12	2-Nov-94	09:45 AM	22.3	11.9		1.86	847	1.40	1.34	1.8	
13	3-Nov-94	11:00 AM	25.3	13.0		1.84	850	1.45	1.31	2.0	
15	5-Nov-94	01:00 PM	50.0	15.1		1.88	841	1.45	1.25	0.0	
16	6-Nov-94	03:40 PM	26.7	16.2		1.86	845	1.46	1.17	0.4	
17	7-Nov-94	11:55 AM	20.3	17.0	0.500	1.87	845	1.45	1.12	0.4	
18	8-Nov-94	01:15 PM	25.3	18.1	0.801	1.94	843	1.47	1.02	3.1	
20	10-Nov-94	09:35 AM	44.3	19.9	0.447	1.82	845	1.60	1.06	5.2	ore tube plugged
22	12-Nov-94	03:15 PM	53.7	22.2	0.484	1.84	845	1.65	1.16	3.2	
24	14-Nov-94	03:35 PM	48.3	24.2	0.888	1.88	840	1.57	1.16	2.8	
25	15-Nov-94	02:05 PM	22.5	25.1	0.690	1.84	840	1.46	1.08	-2.6	
29	19-Nov-94	04:35 PM	98.5	29.2	0.981	1.78	840	1.22	1.29	1.0	
31	21-Nov-94	01:10 PM	44.6	31.1	1.005	1.82	803	1.17	1.54	-0.1	ore tube plugged
35	25-Nov-94	01:15 PM	96.1	35.1	0.975	1.83	830	1.24	1.25	3.0	
38	28-Nov-94	01:35 PM	72.3	38.1	0.798	1.86	821	1.37	1.07	4.8	
40	30-Nov-94	12:45 PM	47.2	40.1	0.993	1.76	811	1.35	1.20	2.7	slurry sample at 1:15pm
41	1-Dec-94	12:45 PM	24.0	41.1	0.682	1.78	824	1.34	1.31	1.6	
43	3-Dec-94	12:45 PM	48.0	43.1	1.240	1.78	830	1.33	1.50	3.4	
44	4-Dec-94	12:10 PM	23.4	44.0	0.939	1.77	819	1.30	1.52	0.9	slurry sample at 2:05pm
45	5-Dec-94	12:05 PM	23.9	45.0	0.928	1.78	816	1.33	1.52	3.6	12/07 slurry at 2:05 pm
47	7-Dec-94	11:25 AM	47.3	47.0	0.961	1.80	818	1.30	1.70	1.4	new flowrate 2L/day
49	9-Dec-94	11:40 AM	48.3	49.0	1.850	1.78	801	1.20	1.80	1.8	
52	12-Dec-94	12:15 PM	72.6	52.0	1.769	1.77	805	1.20	1.90	3.7	
54	14-Dec-94	11:25 AM	47.2	54.0	1.722	1.76	803	1.20	2.00	3.6	
56	16-Dec-94	11:25 AM	48.0	56.0	1.061	1.76	809	1.20	1.80	2.2	
59	19-Dec-94	01:10 PM	73.8	59.1	1.970	1.75	789	1.20	1.90	8.2	
60	20-Dec-94	11:40 AM	22.5	60.0	2.010	1.75	789	1.13	1.70	1.5	
62	22-Dec-94	12:20 PM	48.7	62.0	2.036	1.82	789	1.13	1.69	6.2	
63	23-Dec-94	10:30 AM	22.2	63.0	2.228	1.78	786	1.09	1.67	2.8	slurry sample at 10:55am
67	27-Dec-94	02:35 PM	100.1	67.1	2.007	1.88	774	1.14	1.36	7.0	Dec. 25 no air, no stirring
68	28-Dec-94	11:15 AM	20.7	68.0	2.112	1.84	776	1.12	1.41	4.8	slurry sample at 12:55pm
69	29-Dec-94	01:40 PM	26.4	69.1	0.970	1.82	795	1.20	1.28	5.9	

ORE CONTINUOUS BIOLOGICAL LEACHING - BIOLEACH STAGE 2

Day	Date	Time	Hours	Cumul. Days	Actual Flowrate (L/day)	pH	Eh (mV)	[Cu] (g/L)	[Fe] (g/L)	Rate Cu Ext. (mg/L/h)	Comments
70	30-Dec-94	10:50 AM	21.2	70.0	1.465	1.89	784	1.15	1.34	1.6	slurry sample at 11:20am
71	31-Dec-94	03:20 PM	28.5	71.2	1.867	1.78	782	1.15	1.44	5.1	
72	1-Jan-95	02:40 PM	23.3	72.1	1.859	1.79	784	1.13	1.51	3.8	slurry sample at 3:05pm
74	3-Jan-95	11:30 AM	44.8	74.0	2.538	1.82	719	1.11	1.81	3.8	new flowrate 3L/day
76	5-Jan-95	10:50 AM	47.3	76.0	3.120	1.84	717	1.08	2.10	6.5	
78	7-Jan-95	11:50 AM	49.0	78.0	3.054	1.85	709	1.08	2.16	8.3	
79	8-Jan-95	12:30 PM	24.7	79.1	3.037	1.87	715	1.10	2.16	7.8	
80	9-Jan-95	12:10 PM	23.7	80.0	3.250	1.85	714	1.07	2.28	5.5	
81	10-Jan-95	12:15 PM	24.1	81.0	2.153	1.79	740	1.04	2.08	2.1	
83	12-Jan-95	10:10 AM	45.9	83.0	2.899	1.79	750	1.08	2.18	3.9	
84	13-Jan-95	11:05 AM	24.9	84.0	3.044	1.78	758	1.13	2.10	6.4	
85	14-Jan-95	12:50 PM	25.8	85.1	3.098	1.78	758	1.08	1.92	2.6	slurry sample at 3:15pm
86	15-Jan-95	02:05 PM	25.3	86.1	3.039	1.72	757	1.07	1.98	5.9	
87	16-Jan-95	12:10 PM	22.1	87.0	2.455	1.74	779	1.07	1.86	5.6	
88	17-Jan-95	12:15 PM	24.1	88.0	2.232	1.63	786	1.10	1.79	4.5	
89	18-Jan-95	01:30 PM	25.3	89.1	2.965	1.71	779	1.08	1.73	6.6	slurry sample at 1:15pm
90	19-Jan-95	03:30 PM	26.0	90.2	3.062	1.74	775	1.08	1.77	5.1	
91	20-Jan-95	04:45 PM	25.3	91.2	3.154	1.74	773	1.05	1.87	3.4	
92	21-Jan-95	01:10 PM	20.4	92.1	3.261	1.83	770	1.04	1.53	2.9	slurry sample at 4:10pm
93	22-Jan-95	02:30 PM	25.3	93.1	3.005	1.75	774	1.11	1.48	4.0	
94	23-Jan-95	12:30 PM	22.0	94.1	3.061	1.73	775	1.03	1.74	1.5	
95	24-Jan-95	11:50 AM	23.3	95.0	3.009	1.75	773	0.97	1.74	-5.7	new pulp density 20%
96	25-Jan-95	10:05 AM	22.3	96.0	3.010	1.75	769	1.28	1.66	8.9	replaced acrylic impeller
97	26-Jan-95	03:50 PM	29.8	97.2	3.127	1.79	761	1.80	1.73	22.7	
98	27-Jan-95	02:15 PM	22.4	98.1	2.289	1.71	773	2.08	1.63	20.1	
99	28-Jan-95	11:20 AM	21.1	99.0	3.211	1.92	695	2.16	1.54	17.2	air plugged, acid pump off
100	29-Jan-95	11:55 AM	24.6	100.0	3.064	1.82	668	2.25	1.84	19.6	
101	30-Jan-95	01:30 PM	25.6	101.1	3.017	1.88	642	2.29	2.33	17.9	
102	31-Jan-95	10:05 AM	20.6	102.0	3.232	1.92	623	2.32	2.58	24.4	batch mode
103	1-Feb-95	12:30 PM	26.4	103.1	1.216	1.94	691	3.00	2.21	28.5	
104	2-Feb-95	02:15 PM	25.8	104.1	0.000	1.75	817	3.14	1.55	5.4	
105	3-Feb-95	11:30 AM	21.3	105.0	0.000	1.71	828	3.15	1.56	0.5	continuous mode 3L/day
106	4-Feb-95	12:10 PM	24.7	106.0	0.000	1.69	835	3.13	1.59	-0.8	
107	5-Feb-95	12:50 PM	24.7	107.1	0.817	1.77	819	3.05	1.69	2.2	
108	6-Feb-95	11:45 AM	22.9	108.0	2.198	1.87	681	2.49	1.89	-23.5	continuous mode 1L/day
109	7-Feb-95	11:00 AM	23.3	109.0	1.424	1.86	756	2.82	1.60	24.6	slurry sample at 2:05pm
110	8-Feb-95	12:55 PM	25.9	110.1	0.669	1.83	784	2.82	1.73	3.6	slurry sample at 2:55pm

ORE CONTINUOUS BIOLOGICAL LEACHING - BIOLEACH STAGE 3

Day	Date	Time	Hours	Cumul. Days	Actual Flowrate (L/day)	pH	Eh (mV)	[Cu] (g/L)	[Fe] (g/L)	Rate Cu Ext. (mg/L/h)	Comments
0	21-Oct-94	11:15 AM	0.0	0.0		1.94	550	0.77	2.75		batch mode;
1	22-Oct-94	01:00 PM	25.8	1.1		2.23	638	1.21	2.68		pulp density 10%
3	24-Oct-94	11:00 AM	46.0	3.0		1.83	823	1.30	1.98		
5	26-Oct-94	11:30 AM	48.5	5.0		1.82	831	1.34	1.85		
7	28-Oct-94	11:00 AM	47.5	7.0		1.71	850	1.42	1.64		continuous mode 1 L/day
9	30-Oct-94	08:30 PM	57.5	9.4		1.76	853	1.42	1.51	0.0	
11	1-Nov-94	11:30 AM	39.0	11.0		1.75	855	1.44	1.42	0.5	
12	2-Nov-94	09:45 AM	22.3	11.9		1.77	852	1.45	1.41	0.4	
13	3-Nov-94	11:00 AM	25.3	13.0		1.75	858	1.48	1.35	1.2	
15	5-Nov-94	01:00 PM	50.0	15.1		1.79	849	1.51	1.29	0.6	
16	6-Nov-94	03:40 PM	26.7	16.2		1.78	852	1.52	1.23	0.4	
17	7-Nov-94	11:55 AM	20.3	17.0	0.500	1.80	852	1.52	1.21	0.7	
18	8-Nov-94	01:15 PM	25.3	18.1	0.801	1.89	853	1.51	1.13	0.3	
20	10-Nov-94	09:35 AM	44.3	19.9	0.447	1.79	856	1.60	1.06	2.0	ore tube plugged
22	12-Nov-94	03:15 PM	53.7	22.2	0.484	1.79	852	1.78	1.13	4.7	
24	14-Nov-94	03:35 PM	48.3	24.2	0.888	1.83	848	1.60	1.05	-3.2	
25	15-Nov-94	02:05 PM	22.5	25.1	0.690	1.80	849	1.60	1.05	2.0	
29	19-Nov-94	04:35 PM	98.5	29.2	0.981	1.76	846	1.35	1.16	0.1	
31	21-Nov-94	01:10 PM	44.6	31.1	1.005	1.75	842	1.30	1.31	1.6	ore tube plugged
35	25-Nov-94	01:15 PM	96.1	35.1	0.975	1.78	835	1.28	1.25	0.6	
38	28-Nov-94	01:35 PM	72.3	38.1	0.798	1.79	830	1.36	1.13	0.9	
40	30-Nov-94	12:45 PM	47.2	40.1	0.993	1.78	809	1.39	1.08	1.5	slurry sample at 1:15pm
41	1-Dec-94	12:45 PM	24.0	41.1	0.682	1.78	829	1.38	1.15	0.2	
43	3-Dec-94	12:45 PM	48.0	43.1	1.240	1.78	834	1.43	1.33	3.6	
44	4-Dec-94	12:10 PM	23.4	44.0	0.939	1.74	826	1.42	1.38	1.9	slurry sample at 2:05pm
45	5-Dec-94	12:05 PM	23.9	45.0	0.928	1.73	823	1.45	1.47	3.6	12/07 slurry at 2:05 pm
47	7-Dec-94	11:25 AM	47.3	47.0	0.961	1.78	824	1.40	1.60	0.9	new flowrate 2L/day
49	9-Dec-94	11:40 AM	48.3	49.0	1.850	1.78	810	1.30	1.70	1.8	
52	12-Dec-94	12:15 PM	72.6	52.0	1.769	1.75	816	1.30	1.90	3.7	
54	14-Dec-94	11:25 AM	47.2	54.0	1.722	1.74	819	1.30	1.90	3.6	
56	16-Dec-94	11:25 AM	48.0	56.0	1.061	1.73	822	1.30	1.80	2.2	
59	19-Dec-94	01:10 PM	73.8	59.1	1.970	1.73	811	1.30	1.70	4.1	
60	20-Dec-94	11:40 AM	22.5	60.0	2.010	1.75	811	1.25	1.68	2.8	
62	22-Dec-94	12:20 PM	48.7	62.0	2.036	1.82	807	1.21	1.63	2.6	
63	23-Dec-94	10:30 AM	22.2	63.0	2.228	1.76	803	1.17	1.57	1.9	slurry sample at 10:55am
67	27-Dec-94	02:35 PM	100.1	67.1	2.007	1.82	790	1.20	1.40	2.8	Dec. 25 no air, no stirring
68	28-Dec-94	11:15 AM	20.7	68.0	2.112	1.84	788	1.20	1.32	3.5	slurry sample at 12:55pm
69	29-Dec-94	01:40 PM	26.4	69.1	0.970	1.80	803	1.27	1.22	4.1	

Appendix D - Continuous Reactor Data

ORE CONTINUOUS BIOLOGICAL LEACHING - BIOLEACH STAGE 3

Day	Date	Time	Hours	Cumul. Days	Actual Flowrate (L/day)	pH	Eh (mV)	[Cu] (g/L)	[Fe] (g/L)	Rate Cu Ext. (mg/L/h)	Comments
70	30-Dec-94	10:50 AM	21.2	70.0	1.465	1.87	803	1.24	1.24	1.3	slurry sample at 11:20am
71	31-Dec-94	03:20 PM	28.5	71.2	1.867	1.77	804	1.23	1.31	2.8	
72	1-Jan-95	02:40 PM	23.3	72.1	1.859	1.77	805	1.19	1.35	0.6	slurry sample at 3:05pm
74	3-Jan-95	11:30 AM	44.8	74.0	2.538	1.79	774	1.18	1.60	3.5	new flowrate 3L/day
76	5-Jan-95	10:50 AM	47.3	76.0	3.120	1.83	768	1.17	1.84	5.6	
78	7-Jan-95	11:50 AM	49.0	78.0	3.054	1.82	774	1.14	1.93	3.2	
79	8-Jan-95	12:30 PM	24.7	79.1	3.037	1.80	777	1.15	1.91	3.6	
80	9-Jan-95	12:10 PM	23.7	80.0	3.250	1.80	793	1.13	1.95	3.2	
81	10-Jan-95	12:15 PM	24.1	81.0	2.153	1.74	796	1.15	1.90	5.8	
83	12-Jan-95	10:10 AM	45.9	83.0	2.899	1.75	800	1.14	2.00	3.4	
84	13-Jan-95	11:05 AM	24.9	84.0	3.044	1.75	799	1.15	1.99	1.7	
85	14-Jan-95	12:50 PM	25.8	85.1	3.098	1.74	797	1.14	1.80	3.5	slurry sample at 3:15pm
86	15-Jan-95	02:05 PM	25.3	86.1	3.039	1.70	794	1.13	1.82	3.4	
87	16-Jan-95	12:10 PM	22.1	87.0	2.455	1.68	803	1.15	1.77	5.0	
88	17-Jan-95	12:15 PM	24.1	88.0	2.232	1.61	807	1.14	1.76	1.4	
89	18-Jan-95	01:30 PM	25.3	89.1	2.965	1.70	799	1.15	1.70	4.7	slurry sample at 1:15pm
90	19-Jan-95	03:30 PM	26.0	90.2	3.062	1.71	794	1.15	1.67	4.5	
91	20-Jan-95	04:45 PM	25.3	91.2	3.154	1.72	792	1.12	1.74	3.4	
92	21-Jan-95	01:10 PM	20.4	92.1	3.261	1.78	792	1.08	1.58	0.8	slurry sample at 4:10pm
93	22-Jan-95	02:30 PM	25.3	93.1	3.005	1.76	792	0.99	1.89	-11.1	
94	23-Jan-95	12:30 PM	22.0	94.1	3.061	1.73	787	1.08	1.62	7.3	
95	24-Jan-95	11:50 AM	23.3	95.0	3.009	1.75	792	1.09	1.63	7.9	new pulp density 20%
96	25-Jan-95	10:05 AM	22.3	96.0	3.010	1.74	789	1.19	1.58	-1.1	replaced acrylic impeller
97	26-Jan-95	03:50 PM	29.8	97.2	3.127	1.74	781	1.68	1.60	8.7	
98	27-Jan-95	02:15 PM	22.4	98.1	2.289	1.70	779	1.90	1.54	1.2	
99	28-Jan-95	11:20 AM	21.1	99.0	3.211	1.83	749	2.11	1.50	6.6	air plugged, acid pump off
100	29-Jan-95	11:55 AM	24.6	100.0	3.064	1.82	721	2.31	1.49	12.0	
101	30-Jan-95	01:30 PM	25.6	101.1	3.017	1.88	685	2.38	1.74	8.4	
102	31-Jan-95	10:05 AM	20.6	102.0	3.232	1.92	656	2.42	2.09	8.7	batch mode
103	1-Feb-95	12:30 PM	26.4	103.1	1.216	1.87	724	3.00	1.73	22.0	
104	2-Feb-95	02:15 PM	25.8	104.1	0.000	1.78	773	3.12	1.42	4.7	
105	3-Feb-95	11:30 AM	21.3	105.0	0.000	1.76	782	3.07	1.36	-2.4	continuous mode 3L/day
106	4-Feb-95	12:10 PM	24.7	106.0	0.000	1.75	787	3.09	1.34	0.8	
107	5-Feb-95	12:50 PM	24.7	107.1	0.817	1.75	809	3.08	1.46	0.1	
108	6-Feb-95	11:45 AM	22.9	108.0	2.198	1.80	757	2.98	1.64	18.1	continuous mode 1L/day
109	7-Feb-95	11:00 AM	23.3	109.0	1.424	1.80	802	2.96	1.50	3.3	slurry sample at 2:05pm
110	8-Feb-95	12:55 PM	25.9	110.1	0.669	1.78	809	2.98	1.51	3.0	slurry sample at 2:55pm

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Appendix D - Continuous Reactor Data

CONCENTRATE CONTINUOUS BIOLOGICAL LEACHING - FEED TANK

Day	Date	Time	Hours	Cumul. Hours	Cumul. Days	Target Flowrate (L/day)	pH	Eh (mV)	[Cu] (g/L)	[Fe] (g/L)	Comments
0	19-Oct-94	10:00 AM	0.0	0.0	0.00						batch mode;
1	20-Oct-94	11:45 AM	25.8	25.8	1.07						pulp density 5%
2	21-Oct-94	10:00 AM	22.3	48.0	2.00						
5	24-Oct-94	11:00 AM	73.0	121.0	5.04						
7	26-Oct-94	11:30 AM	48.5	169.5	7.06						
9	28-Oct-94	11:00 AM	47.5	217.0	9.04	1	4.5	518	2.43	0.26	continuous mode 1 L/day
11	30-Oct-94	08:30 PM	57.5	274.5	11.44	1	4.5	518	2.43	0.26	
13	1-Nov-94	11:30 AM	39.0	313.5	13.06	1	4.5	518	2.43	0.26	
14	2-Nov-94	09:45 AM	22.3	335.8	13.99	1	3.78	518	2.43	0.26	
15	3-Nov-94	11:00 AM	25.3	361.0	15.04	1	3.78	518	2.43	0.26	
17	5-Nov-94	01:00 PM	50.0	411.0	17.13	1	3.78	518	2.43	0.26	FILL
18	6-Nov-94	03:40 PM	26.7	437.7	18.24	1	3.63	498	1.16	1.56	
19	7-Nov-94	11:55 AM	20.3	457.9	19.08	1	3.6	506	1.6	1.05	
20	8-Nov-94	01:15 PM	25.3	483.3	20.14	1	3.6	506	2.165	0.5315	
22	10-Nov-94	09:35 AM	44.3	527.6	21.98	1	4.2	563	2.73	0.013	
24	12-Nov-94	03:15 PM	53.7	581.3	24.22	1	4.4	550	2.85	0.005	
25	13-Nov-94	04:15 PM	25.0	606.3	25.26	1	4.45	551	2.85	0.005	Nov 14 FILL
27	15-Nov-94	02:05 PM	45.8	652.1	27.17	1	3.8	489	2.85	0.005	Nov 17 FILL
31	19-Nov-94	04:35 PM	98.5	750.6	31.27	1	3.8	489	2.85	0.005	FILL
33	21-Nov-94	01:10 PM	44.6	795.2	33.13	1	3.65	476	0.81	1.5	
37	25-Nov-94	01:15 PM	96.1	891.3	37.14	1	4.07	499	2.19	0.055	Nov 26 FILL
40	28-Nov-94	01:35 PM	72.3	963.6	40.15	1	3.6	507	2.17	2.53	
42	30-Nov-94	12:45 PM	47.2	1010.8	42.11	1	3.62	507	1.71	0.573	
43	1-Dec-94	12:45 PM	24.0	1034.8	43.11	1	3.62	507	2.04	0.2865	
45	3-Dec-94	12:45 PM	48.0	1082.8	45.11	1	4.51	534	2.37	0	slurry sample at 3:25pm
46	4-Dec-94	12:10 PM	23.4	1106.2	46.09	1	4.51	534	2.37	0	FILL
47	5-Dec-94	12:05 PM	23.9	1130.1	47.09	1	3.77	479	0.713	1.65	
49	7-Dec-94	11:25 AM	47.3	1177.4	49.06	2	3.69	501	1.34	1.05	new flowrate 2L/day
51	9-Dec-94	11:40 AM	48.3	1225.7	51.07	2	3.81	516	2	0.4	Dec 10 FILL
54	12-Dec-94	12:15 PM	72.6	1298.3	54.09	2	3.71	507	0.9	1.6	slurry sample at 2:15pm
56	14-Dec-94	11:25 AM	47.2	1345.4	56.06	2	4.02	446	0.2	2.4	FILL
58	16-Dec-94	11:25 AM	48.0	1393.4	58.06	2	3.73	502	0.8	1.7	Dec 18 FILL
61	19-Dec-94	01:10 PM	73.8	1467.2	61.13	2	3.8	492	0.4	2.1	
62	20-Dec-94	11:45 AM	22.6	1489.8	62.07	2	3.8	492	0.85	1.6	
64	22-Dec-94	12:20 PM	48.6	1538.3	64.10	2	3.7	516	1.3	1.1	FILL
65	23-Dec-94	10:35 AM	22.3	1560.6	65.02	2	3.95	503	0.405	2.02	Dec 26 FILL
69	27-Dec-94	02:35 PM	100.0	1660.6	69.19	2	3.74	487	0.445	1.93	
70	28-Dec-94	11:15 AM	20.7	1681.3	70.05	2	3.74	487	0.7425	1.605	

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Appendix D - Continuous Reactor Data

CONCENTRATE CONTINUOUS BIOLOGICAL LEACHING - FEED TANK

Day	Date	Time	Hours	Cumul. Hours	Cumul. Days	Target Flowrate (L/day)	pH	Eh (mV)	[Cu] (g/L)	[Fe] (g/L)	Comments
71	29-Dec-94	01:45 PM	26.5	1707.8	71.16	2	3.67	501	1.04	1.28	Dec 30 FILL
72	30-Dec-94	10:50 AM	21.1	1728.8	72.03	2	3.76	509	1.3	0.935	slurry sample at 11:25am
73	31-Dec-94	03:20 PM	28.5	1757.3	73.22	2	3.76	509	1	1.3025	
74	1-Jan-95	02:40 PM	23.3	1780.7	74.19	2	3.57	503	0.7	1.67	01/03 slurry at 2:05pm
76	3-Jan-95	11:30 AM	44.8	1825.5	76.06	3	3.57	503	0.163	2.17	new flowrate 3L/day
78	5-Jan-95	10:50 AM	47.3	1872.8	78.03	3	3.65	497	0.85	1.59	Jan 3 FILL
80	7-Jan-95	11:50 AM	49.0	1921.8	80.08	3	3.65	497	0.85	1.59	Jan 6 FILL
81	8-Jan-95	12:30 PM	24.7	1946.5	81.10	3	3.65	497	0.85	1.59	FILL
82	9-Jan-95	12:10 PM	23.7	1970.2	82.09	3	3.79	475	0.29	2.1	
83	10-Jan-95	12:15 PM	24.1	1994.2	83.09	3	3.79	475	0.2925	2.115	Jan 11 FILL
85	12-Jan-95	10:10 AM	45.9	2040.2	85.01	3	3.78	475	0.295	2.13	slurry sample at 2:20pm
86	13-Jan-95	11:05 AM	24.9	2065.1	86.05	3	3.78	475	0.295	2.13	FILL
87	14-Jan-95	12:50 PM	25.8	2090.8	87.12	3	3.68	488	0.678	1.69	
88	15-Jan-95	01:55 PM	25.1	2115.9	88.16	3	3.68	488	0.678	1.69	FILL
89	16-Jan-95	12:10 PM	22.3	2138.2	89.09	3	3.56	475	0.473	1.97	slurry sample at 1:25pm
90	17-Jan-95	12:15 PM	24.1	2162.3	90.09	3	3.56	475	0.473	1.97	
91	18-Jan-95	01:30 PM	25.3	2187.5	91.15	3	3.56	475	0.473	1.97	FILL
92	19-Jan-95	03:30 PM	26.0	2213.5	92.23	3	3.56	475	0.473	1.97	
93	20-Jan-95	04:45 PM	25.3	2238.8	93.28	3	3.56	475	0.473	1.97	FILL
94	21-Jan-95	01:10 PM	20.4	2259.2	94.13	3	3.61	492	0.63	1.79	
95	22-Jan-95	02:30 PM	25.3	2284.5	95.19	3	3.61	492	0.63	1.79	Jan 23 FILL
96	23-Jan-95	12:30 PM	22.0	2306.5	96.10	3	3.61	492	0.63	1.79	new pulp density 10%
97	24-Jan-95	11:50 AM	23.3	2329.8	97.08	3	3.73	471	0.473	1.64	
98	25-Jan-95	10:05 AM	22.3	2352.1	98.00	3	3.72	479	1.04	0.97	FILL
99	26-Jan-95	03:50 PM	29.8	2381.8	99.24	3	3.72	479	0.63	1.79	
100	27-Jan-95	02:15 PM	22.4	2404.3	100.18	3	3.72	479	0.63	1.79	Jan 28 FILL
101	28-Jan-95	11:20 AM	21.1	2425.3	101.06	3	3.78	482	1.4	0.505	slurry sample at 1:45pm
102	29-Jan-95	11:55 AM	24.6	2449.9	102.08	3	3.78	482	1.4	0.505	
103	30-Jan-95	01:30 PM	25.6	2475.5	103.15	3	3.78	482	1.4	0.505	
104	31-Jan-95	10:05 AM	20.6	2496.1	104.00	3	3.87	480	1.45	0.425	
105	1-Feb-95	12:30 PM	26.4	2522.5	105.10	3	3.87	480	1.45	0.425	FILL
106	2-Feb-95	02:15 PM	25.8	2548.3	106.18	3	3.87	480	1.45	0.425	
107	3-Feb-95	11:30 AM	21.3	2569.5	107.06	3	3.87	480	0.673	0.425	FILL
108	4-Feb-95	12:10 PM	24.7	2594.2	108.09	3	3.84	478	0.673	1.33	slurry sample at 2:05pm
109	5-Feb-95	12:50 PM	24.7	2618.8	109.12	3	3.83	493	1.42	0.585	slurry sample at 3:10pm
110	6-Feb-95	11:45 AM	22.9	2641.8	110.07	3	4.01	502	1.88	0.11	slurry sample at 3:45pm

CONCENTRATE CONTINUOUS BIOLOGICAL LEACHING - BIOLEACH STAGE 1

Appendix D - Continuous Reactor Data

Day	Date	Cumul. Days	Actual Flowrate (L/day)	pH	Eh (mV)	[Cu] (g/L)	[Fe] (g/L)	Rate Cu Ext. (mg/L/h)	6M H2SO4 Daily (ml)	Concentrate Daily (kg)	H2SO4 Daily (kg/t)	Comments
0	19-Oct-94	0.00		2.85	550	3.5	1.4		36	0.190	112	batch mode;
1	20-Oct-94	1.07		2.23	585	4.1	2.4		10	0.190	31	pulp density 5%
2	21-Oct-94	2.00		1.94	589	4.6	2.6		0	0.190	0	
5	24-Oct-94	5.04		2.01	785	6.7	1.3		2	0.190	6	
7	26-Oct-94	7.06		1.78	795	7.5	1.6		0	0.190	0	
9	28-Oct-94	9.04		1.68	845	7.8	2.0		0	0.049	0	continuous mode 1 L/day
11	30-Oct-94	11.44		1.95	821	7.8	1.9	55	11	0.117	55	
13	1-Nov-94	13.06		1.96	815	7.9	1.6	60	11	0.080	81	
14	2-Nov-94	13.99		2.00	791	8.3	1.4	76	4	0.045	52	
15	3-Nov-94	15.04		2.01	793	9.1	1.2	103	3	0.052	34	
17	5-Nov-94	17.13		2.14	790	9.1	1.0	69	8	0.102	46	
18	6-Nov-94	18.24		2.11	765	9.3	1.2	92	13	0.054	141	
19	7-Nov-94	19.08	0.500	2.06	784	9.7	1.0	104	4	0.041	57	
20	8-Nov-94	20.14	0.496	2.18	792	9.3	0.8	21	0	0.026	0	
22	10-Nov-94	21.98	0.596	1.77	818	8.5	1.8	18	39	0.055	417	
24	12-Nov-94	24.22	0.416	1.88	805	8.6	2.2	27	16	0.046	203	
25	13-Nov-94	25.26	0.691	1.91	806	8.4	2.2	32	13	0.036	213	
27	15-Nov-94	27.17	0.742	1.85	805	7.7	2.3	22	21	0.071	174	
31	19-Nov-94	31.27	0.938	1.92	821	6.0	2.0	14	37	0.192	113	
33	21-Nov-94	33.13	0.948	1.91	833	6.5	1.8	67	24	0.088	160	
37	25-Nov-94	37.14	0.933	1.92	805	6.4	2.5	40	74	0.187	233	
40	28-Nov-94	40.15	0.977	2.07	780	6.4	1.3	43	20	0.147	80	slurry sample at 2:15pm
42	30-Nov-94	42.11	0.951	1.85	805	6.7	2.0	56	39	0.093	246	
43	1-Dec-94	43.11	1.000	1.88	810	6.7	2.2	50	16	0.050	188	slurry sample at 2:45pm
45	3-Dec-94	45.11	0.906	1.87	821	6.6	2.3	36	29	0.091	188	
46	4-Dec-94	46.09	0.824	1.88	816	6.7	2.3	41	12	0.040	176	
47	5-Dec-94	47.09	0.928	1.89	798	6.7	2.5	59	17	0.046	216	slurry sample at 2:15pm
49	7-Dec-94	49.06	0.948	1.89	810	6.6	2.7	50	31	0.094	195	new flowrate 2L/day
51	9-Dec-94	51.07	1.838	1.94	794	6.1	2.6	68	64	0.185	204	
54	12-Dec-94	54.09	1.939	1.86	802	6.1	2.9	105	118	0.293	237	
56	14-Dec-94	56.06	1.941	1.85	782	6.1	3.0	119	76	0.191	234	
58	16-Dec-94	58.06	2.097	1.85	769	5.7	2.9	99	77	0.210	216	
61	19-Dec-94	61.13	2.008	1.86	791	4.9	2.5	83	105	0.309	200	
62	20-Dec-94	62.07	2.105	1.78	800	5.1	2.3	104	38	0.099	226	
64	22-Dec-94	64.10	2.078	1.85	793	5.0	2.5	79	74	0.210	207	
65	23-Dec-94	65.02	2.293	1.75	779	4.9	2.5	104	40	0.106	221	
69	27-Dec-94	69.19	1.998	1.63	763	5.0	2.5	94	160	0.416	226	slurry sample at 4:50pm
70	28-Dec-94	70.05	2.094	1.86	794	5.2	2.5	108	31	0.090	202	

CONCENTRATE CONTINUOUS BIOLOGICAL LEACHING - BIOLEACH STAGE 1

Day	Date	Cumul. Days	Actual Flowrate (L/day)	pH	Eh (mV)	[Cu] (g/L)	[Fe] (g/L)	Rate Cu Ext. (mg/L/h)	6M H2SO4 Daily (ml)	Concentrate Daily (kg)	H2SO4 Daily (kg/t)	Comments
71	29-Dec-94	71.16	2.021	1.85	801	5.0	2.4	75	39	0.112	206	slurry sample at 2:05pm
72	30-Dec-94	72.03	2.062	1.82	795	4.9	2.3	72	35	0.091	227	
73	31-Dec-94	73.22	1.926	1.78	797	5.0	2.4	82	40	0.114	206	slurry sample at 5:15pm
74	1-Jan-95	74.19	1.854	1.86	790	5.1	2.2	91	40	0.090	261	
76	3-Jan-95	76.06	2.490	1.8	739	4.5	2.3	97	79	0.233	200	new flowrate 3L/day
78	5-Jan-95	78.03	3.046	1.81	790	5.3	2.3	158	124	0.300	243	
80	7-Jan-95	80.08	2.550	2.06	734	5.0	1.4	105	63	0.260	142	
81	8-Jan-95	81.10	2.880	1.97	705	3.6	1.3	22	17	0.148	68	
82	9-Jan-95	82.09	3.138	1.89	768	5.0	2.1	215	76	0.155	289	
83	10-Jan-95	83.09	2.981	1.78	784	5.3	2.2	168	63	0.150	248	
85	12-Jan-95	85.01	3.058	1.9	785	5.1	2.2	150	105	0.293	211	
86	13-Jan-95	86.05	2.916	1.78	764	5.0	2.2	138	59	0.151	229	
87	14-Jan-95	87.12	2.870	1.86	790	5.1	2.2	135	60	0.154	229	
88	15-Jan-95	88.16	2.677	1.76	798	5.0	2.3	119	49	0.140	206	slurry sample at 3:45pm
89	16-Jan-95	89.09	2.487	1.79	722	5.0	2.4	116	46	0.115	235	
90	17-Jan-95	90.09	3.158	1.63	763	5.1	2.4	159	57	0.158	212	slurry sample at 12:45pm
91	18-Jan-95	91.15	2.720	1.71	775	4.9	2.3	116	54	0.143	222	
92	19-Jan-95	92.23	3.180	1.71	739	5.2	2.5	170	73	0.172	249	slurry sample at 3:55pm
93	20-Jan-95	93.28	3.069	1.83	752	5.0	2.6	138	62	0.161	226	
94	21-Jan-95	94.13	3.387	1.88	774	5.0	2.4	153	66	0.144	270	
95	22-Jan-95	95.19	2.427	1.75	799	5.1	2.4	114	42	0.128	193	feed tube plugged
96	23-Jan-95	96.10	2.641	1.74	791	4.9	2.3	113	44	0.242	107	new pulp density 10%
97	24-Jan-95	97.08	3.266	1.82	695	7.3	2.7	332	100	0.318	185	
98	25-Jan-95	98.00	3.174	1.88	750	8.7	2.7	317	100	0.294	200	
99	26-Jan-95	99.24	3.104	1.79	764	9.6	2.7	317	110	0.385	168	
100	27-Jan-95	100.18	3.058	1.75	768	10.0	2.5	319	116	0.286	239	
101	28-Jan-95	101.06	3.200	1.92	739	10.0	2.5	287	90	0.281	188	
102	29-Jan-95	102.08	1.124	1.7	691	10.5	2.7	127	20	0.115	102	
103	30-Jan-95	103.15	2.743	1.79	717	10.0	2.6	226	110	0.292	221	
104	31-Jan-95	104.00	3.295	1.9	747	10.1	2.6	302	95	0.283	198	
105	1-Feb-95	105.10	2.934	1.9	713	10.2	2.5	271	105	0.323	191	
106	2-Feb-95	106.18	3.040	1.85	736	10.2	2.6	277	116	0.326	209	
107	3-Feb-95	107.06	2.897	1.8	765	9.9	2.7	264	80	0.257	184	slurry sample at 1:35pm
108	4-Feb-95	108.09	3.145	1.92	762	9.7	2.7	288	106	0.323	193	slurry sample at 2:05pm
109	5-Feb-95	109.12	3.065	1.94	756	9.9	2.7	279	100	0.316	186	slurry sample at 3:10pm
110	6-Feb-95	110.07	2.758	1.85	769	9.4	2.6	197	78	0.262	175	slurry sample at 3:35pm

Appendix D - Continuous Reactor Data

CONCENTRATE CONTINUOUS BIOLOGICAL LEACHING - BIOLEACH STAGE 2

Day	Date	Time	Hours	Cumul. Days	Actual Flowrate (L/day)	pH	Eh (mV)	[Cu] (g/L)	[Fe] (g/L)	Rate Cu Ext. (mg/L/h)	Comments
0	19-Oct-94	10:00 AM	0.0	0.00		2.87	548	3.53	1.36		batch mode; pulp density 5%
1	20-Oct-94	11:45 AM	25.8	1.07		2.23	582	3.98	2.25		
2	21-Oct-94	10:00 AM	22.3	2.00		1.92	586	4.40	2.48		
5	24-Oct-94	11:00 AM	73.0	5.04		1.97	792	6.83	1.26		
7	26-Oct-94	11:30 AM	48.5	7.06		1.80	806	7.38	1.60		
9	28-Oct-94	11:00 AM	47.5	9.04		1.71	841	7.50	1.98		continuous mode 1L/day
11	30-Oct-94	08:30 PM	57.5	11.44		1.76	839	8.40	2.35	16	
13	1-Nov-94	11:30 AM	39.0	13.06		1.77	841	8.75	2.30	9	
14	2-Nov-94	09:45 AM	22.3	13.99		1.77	832	9.05	2.25	13	
15	3-Nov-94	11:00 AM	25.3	15.04		1.77	839	9.60	2.23	22	
17	5-Nov-94	01:00 PM	50.0	17.13		1.90	827	10.80	2.00	24	
18	6-Nov-94	03:40 PM	26.7	18.24		1.92	822	10.80	1.94	0	
19	7-Nov-94	11:55 AM	20.3	19.08	0.500	1.89	824	10.80	1.85	11	
20	8-Nov-94	01:15 PM	25.3	20.14	0.496	1.99	822	11.10	1.80	30	
22	10-Nov-94	09:35 AM	44.3	21.98	0.596	1.89	823	10.80	1.80	22	
24	12-Nov-94	03:15 PM	53.7	24.22	0.416	1.77	823	11.20	2.50	30	
25	13-Nov-94	04:15 PM	25.0	25.26	0.691	1.80	825	10.80	2.70	19	
27	15-Nov-94	02:05 PM	45.8	27.17	0.742	1.77	832	9.80	2.75	11	
31	19-Nov-94	04:35 PM	98.5	31.27	0.938	1.83	836	7.50	2.50	6	
33	21-Nov-94	01:10 PM	44.6	33.13	0.948	1.85	843	7.20	2.20	7	
37	25-Nov-94	01:15 PM	96.1	37.14	0.933	1.79	828	7.30	2.70	19	
40	28-Nov-94	01:35 PM	72.3	40.15	0.977	1.93	795	7.50	2.01	25	slurry sample at 2:45pm
42	30-Nov-94	12:45 PM	47.2	42.11	0.951	1.83	819	7.43	2.13	13	
43	1-Dec-94	12:45 PM	24.0	43.11	1.000	1.83	827	7.53	2.20	21	slurry sample at 2:45pm
45	3-Dec-94	12:45 PM	48.0	45.11	0.906	1.82	829	7.50	2.46	17	
46	4-Dec-94	12:10 PM	23.4	46.09	0.824	1.84	837	7.40	2.65	9	
47	5-Dec-94	12:05 PM	23.9	47.09	0.928	1.84	824	7.45	2.73	17	slurry sample at 2:15pm
49	7-Dec-94	11:25 AM	47.3	49.06	0.948	1.83	825	7.50	3.00	19	new flowrate 2 L/day
51	9-Dec-94	11:40 AM	48.3	51.07	1.838	1.91	803	6.80	2.80	12	
54	12-Dec-94	12:15 PM	72.6	54.09	1.939	1.84	809	6.80	3.00	28	
56	14-Dec-94	11:25 AM	47.2	56.06	1.941	1.81	803	6.90	3.10	34	slurry sample at 1:45pm
58	16-Dec-94	11:25 AM	48.0	58.06	2.097	1.85	795	6.70	2.90	40	
61	19-Dec-94	01:10 PM	73.8	61.13	2.008	1.82	803	6.20	2.70	48	
62	20-Dec-94	11:45 AM	22.6	62.07	2.105	1.85	800	6.23	2.50	50	slurry sample at 2:15pm
64	22-Dec-94	12:20 PM	48.6	64.10	2.078	1.89	802	6.15	2.50	47	
65	23-Dec-94	10:35 AM	22.3	65.02	2.293	1.84	820	5.88	2.48	33	12/25 compressor down
69	27-Dec-94	02:35 PM	100.0	69.19	1.998	1.81	801	5.65	2.60	27	slurry sample at 4:50pm
70	28-Dec-94	11:15 AM	20.7	70.05	2.094	1.86	795	6.20	2.53	71	

Appendix D - Continuous Reactor Data

CONCENTRATE CONTINUOUS BIOLOGICAL LEACHING - BIOLEACH STAGE 2

Day	Date	Time	Hours	Cumul. Days	Actual Flowrate (L/day)	pH	Eh (mV)	[Cu] (g/L)	[Fe] (g/L)	Rate Cu Ext. (mg/L/h)	Comments
71	29-Dec-94	01:45 PM	26.5	71.16	2.021	1.89	802	6.33	2.48	62	slurry sample at 2:05pm
72	30-Dec-94	10:50 AM	21.1	72.03	2.062	1.93	793	6.25	2.31	55	
73	31-Dec-94	03:20 PM	28.5	73.22	1.926	1.82	802	6.43	2.36	66	slurry sample at 5:15pm
74	1-Jan-95	02:40 PM	23.3	74.19	1.854	1.9	790	6.38	2.33	47	
76	3-Jan-95	11:30 AM	44.8	76.06	2.490	1.93	802	5.53	2.23	37	new flowrate 3L/day
78	5-Jan-95	10:50 AM	47.3	78.03	3.046	1.96	787	6.53	2.25	100	
80	7-Jan-95	11:50 AM	49.0	80.08	2.550	2.08	774	6.23	1.36	59	
81	8-Jan-95	12:30 PM	24.7	81.10	2.880	1.97	790	5.05	1.26	42	
82	9-Jan-95	12:10 PM	23.7	82.09	3.138	1.99	779	5.75	1.63	79	
83	10-Jan-95	12:15 PM	24.1	83.09	2.981	1.86	785	6.73	1.74	130	
85	12-Jan-95	10:10 AM	45.9	85.01	3.058	1.95	788	6.13	2.08	51	
86	13-Jan-95	11:05 AM	24.9	86.05	2.916	1.91	803	5.75	2.14	30	
87	14-Jan-95	12:50 PM	25.8	87.12	2.870	1.95	784	6.38	2.10	102	
88	15-Jan-95	01:55 PM	25.1	88.16	2.677	1.86	798	5.88	2.25	27	slurry sample at 3:45pm
89	16-Jan-95	12:10 PM	22.3	89.09	2.487	1.84	793	6.18	2.28	75	
90	17-Jan-95	12:15 PM	24.1	90.09	3.158	1.72	785	6.10	2.28	60	slurry sample at 12:45pm
91	18-Jan-95	01:30 PM	25.3	91.15	2.720	1.79	794	6.13	2.27	71	
92	19-Jan-95	03:30 PM	26.0	92.23	3.180	1.81	792	6.05	2.44	51	slurry sample at 3:55pm
93	20-Jan-95	04:45 PM	25.3	93.28	3.069	1.9	789	6.18	2.42	79	
94	21-Jan-95	01:10 PM	20.4	94.13	3.387	1.9	779	6.20	2.37	86	
95	22-Jan-95	02:30 PM	25.3	95.19	2.427	1.85	790	6.33	2.28	70	
96	23-Jan-95	12:30 PM	22.0	96.10	2.641	1.81	790	6.03	2.29	47	new pulp density 10%
97	24-Jan-95	11:50 AM	23.3	97.08	3.266	1.96	769	7.25	2.29	50	
98	25-Jan-95	10:05 AM	22.3	98.00	3.174	1.98	752	9.80	2.30	187	
99	26-Jan-95	03:50 PM	29.8	99.24	3.104	1.95	773	10.80	2.52	114	
100	27-Jan-95	02:15 PM	22.4	100.18	3.058	1.93	765	11.30	2.32	105	
101	28-Jan-95	11:20 AM	21.1	101.06	3.200	1.97	754	11.30	2.39	87	
102	29-Jan-95	11:55 AM	24.6	102.08	1.124	1.87	764	13.30	2.18	147	
103	30-Jan-95	01:30 PM	25.6	103.15	2.743	1.98	758	12.10	2.41	73	
104	31-Jan-95	10:05 AM	20.6	104.00	3.295	2.06	752	12.60	2.31	196	
105	1-Feb-95	12:30 PM	26.4	105.10	2.934	2.01	763	12.50	2.28	137	
106	2-Feb-95	02:15 PM	25.8	106.18	3.040	1.99	759	12.10	2.43	105	
107	3-Feb-95	11:30 AM	21.3	107.06	2.897	1.97	754	11.90	2.44	111	slurry sample at 1:35pm
108	4-Feb-95	12:10 PM	24.7	108.09	3.145	2.06	756	11.80	2.40	134	slurry sample at 2:05pm
109	5-Feb-95	12:55 PM	24.8	109.12	3.065	2.08	753	11.70	2.47	111	slurry sample at 3:10pm
110	6-Feb-95	11:45 AM	22.8	110.07	2.758	2.03	760	11.40	2.36	99	

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Appendix D - Continuous Reactor Data

CONCENTRATE CONTINUOUS BIOLOGICAL LEACHING - BIOLEACH STAGE 3

Day	Date	Time	Hours	Cumul. Days	Actual Flowrate (L/day)	pH	Eh (mV)	[Cu] (g/L)	[Fe] (g/L)	Rate Cu Ext. (mg/L/h)	Comments
0	19-Oct-94	10:00 AM	0.0	0.00		2.85	550	3.50	1.25		batch mode; pulp density 5%
1	20-Oct-94	11:45 AM	25.8	1.07		2.30	581	3.90	2.13		
2	21-Oct-94	10:00 AM	22.3	2.00		1.94	584	4.23	2.45		
5	24-Oct-94	11:00 AM	73.0	5.04		2.11	771	6.05	1.15		
7	26-Oct-94	11:30 AM	48.5	7.06		1.83	800	6.75	1.40		
9	28-Oct-94	11:00 AM	47.5	9.04		1.75	834	6.65	1.65		continuous mode 1L/day
11	30-Oct-94	08:30 PM	57.5	11.44		1.67	845	8.30	2.45	29	
13	1-Nov-94	11:30 AM	39.0	13.06		1.63	845	8.95	2.75	17	
14	2-Nov-94	09:45 AM	22.3	13.99		1.65	846	9.45	2.93	22	
15	3-Nov-94	11:00 AM	25.3	15.04		1.56	856	10.60	3.20	46	
17	5-Nov-94	01:00 PM	50.0	17.13		1.62	848	11.50	3.20	18	
18	6-Nov-94	03:40 PM	26.7	18.24		1.61	851	12.00	3.18	19	
19	7-Nov-94	11:55 AM	20.3	19.08	0.500	1.63	851	12.00	3.13	12	
20	8-Nov-94	01:15 PM	25.3	20.14	0.496	1.72	849	12.60	3.03	39	
22	10-Nov-94	09:35 AM	44.3	21.98	0.596	1.70	849	12.90	2.88	33	
24	12-Nov-94	03:15 PM	53.7	24.22	0.416	1.63	839	14.10	3.20	47	
25	13-Nov-94	04:15 PM	25.0	25.26	0.691	1.65	846	13.80	3.33	31	
27	15-Nov-94	02:05 PM	45.8	27.17	0.742	1.65	847	12.30	3.18	6	
31	19-Nov-94	04:35 PM	98.5	31.27	0.938	1.71	845	9.70	3.00	17	
33	21-Nov-94	01:10 PM	44.6	33.13	0.948	1.75	853	8.50	2.78	-1	
37	25-Nov-94	01:15 PM	96.1	37.14	0.933	1.70	843	8.20	2.98	14	
40	28-Nov-94	01:35 PM	72.3	40.15	0.977	1.75	813	8.40	2.90	21	slurry sample at 2:45pm
42	30-Nov-94	12:45 PM	47.2	42.11	0.951	1.73	826	8.30	2.50	15	
43	1-Dec-94	12:45 PM	24.0	43.11	1.000	1.76	841	8.40	2.68	22	slurry sample at 2:45pm
45	3-Dec-94	12:45 PM	48.0	45.11	0.906	1.77	830	8.45	2.83	19	
46	4-Dec-94	12:10 PM	23.4	46.09	0.824	1.72	847	8.40	2.93	15	
47	5-Dec-94	12:05 PM	23.9	47.09	0.928	1.73	830	8.50	3.10	24	slurry sample at 2:15pm
49	7-Dec-94	11:25 AM	47.3	49.06	0.948	1.73	841	8.60	3.40	24	new flowrate 2L/day
51	9-Dec-94	11:40 AM	48.3	51.07	1.838	1.80	810	7.50	3.10	4	
54	12-Dec-94	12:15 PM	72.6	54.09	1.939	1.79	817	7.50	3.30	28	
56	14-Dec-94	11:25 AM	47.2	56.06	1.941	1.72	813	7.60	3.40	30	slurry sample at 1:50pm
58	16-Dec-94	11:25 AM	48.0	58.06	2.097	1.76	808	7.20	3.30	14	
61	19-Dec-94	01:10 PM	73.8	61.13	2.008	1.71	805	7.40	3.50	53	
62	20-Dec-94	11:45 AM	22.6	62.07	2.105	1.73	803	7.20	3.18	34	slurry sample at 2:15pm
64	22-Dec-94	12:20 PM	48.6	64.10	2.078	1.78	798	6.98	3.10	31	
65	23-Dec-94	10:35 AM	22.3	65.02	2.293	1.73	825	6.55	3.03	13	
69	27-Dec-94	02:35 PM	100.0	69.19	1.998	1.72	802	6.20	3.03	19	slurry sample at 4:50pm
70	28-Dec-94	11:15 AM	20.7	70.05	2.094	1.74	803	6.43	3.25	21	

Appendix D - Continuous Reactor Data

CONCENTRATE CONTINUOUS BIOLOGICAL LEACHING - BIOLEACH STAGE 3

Day	Date	Time	Hours	Cumul. Days	Actual Flowrate (L/day)	pH	Eh (mV)	[Cu] (g/L)	[Fe] (g/L)	Rate Cu Ext. (mg/L/h)	Comments
71	29-Dec-94	01:45 PM	26.5	71.16	2.021	1.76	811	6.78	2.95	32	slurry sample at 2:05pm
72	30-Dec-94	10:50 AM	21.1	72.03	2.062	1.80	806	6.90	2.98	34	
73	31-Dec-94	03:20 PM	28.5	73.22	1.926	1.75	812	6.78	2.88	10	slurry sample at 5:15pm
74	1-Jan-95	02:40 PM	23.3	74.19	1.854	1.77	792	7.03	2.98	36	
76	3-Jan-95	11:30 AM	44.8	76.06	2.490	1.82	820	6.20	2.55	16	new flowrate 3L/day
78	5-Jan-95	10:50 AM	47.3	78.03	3.046	1.86	806	6.55	2.33	9	
80	7-Jan-95	11:50 AM	49.0	80.08	2.550	1.94	797	6.85	1.85	39	
81	8-Jan-95	12:30 PM	24.7	81.10	2.880	1.95	804	6.10	1.52	33	
82	9-Jan-95	12:10 PM	23.7	82.09	3.138	1.96	788	5.85	1.58	-4	
83	10-Jan-95	12:15 PM	24.1	83.09	2.981	1.85	801	6.68	1.71	31	
85	12-Jan-95	10:10 AM	45.9	85.01	3.058	1.88	793	6.70	2.13	37	
86	13-Jan-95	11:05 AM	24.9	86.05	2.916	1.85	816	6.23	2.18	10	
87	14-Jan-95	12:50 PM	25.8	87.12	2.870	1.86	792	6.50	2.25	18	
88	15-Jan-95	01:55 PM	25.1	88.16	2.677	1.81	810	6.30	2.27	15	slurry sample at 3:45pm
89	16-Jan-95	12:10 PM	22.3	89.09	2.487	1.81	811	6.23	2.41	-1	
90	17-Jan-95	12:15 PM	24.1	90.09	3.158	1.66	797	6.48	2.40	35	slurry sample at 12:45pm
91	18-Jan-95	01:30 PM	25.3	91.15	2.720	1.75	810	6.45	2.43	17	
92	19-Jan-95	03:30 PM	26.0	92.23	3.180	1.76	805	6.43	2.53	24	slurry sample at 3:55pm
93	20-Jan-95	04:45 PM	25.3	93.28	3.069	1.81	802	6.53	2.73	26	
94	21-Jan-95	01:10 PM	20.4	94.13	3.387	1.85	790	6.68	2.53	41	
95	22-Jan-95	02:30 PM	25.3	95.19	2.427	1.76	807	6.75	2.57	24	
96	23-Jan-95	12:30 PM	22.0	96.10	2.641	1.74	799	6.63	2.51	28	new pulp density 10%
97	24-Jan-95	11:50 AM	23.3	97.08	3.266	1.85	803	6.90	2.44	-12	
98	25-Jan-95	10:05 AM	22.3	98.00	3.174	1.88	767	9.05	2.42	47	
99	26-Jan-95	03:50 PM	29.8	99.24	3.104	1.86	782	11.10	2.60	88	
100	27-Jan-95	02:15 PM	22.4	100.18	3.058	1.84	772	11.70	2.52	52	
101	28-Jan-95	11:20 AM	21.1	101.06	3.200	1.93	759	12.20	2.54	84	
102	29-Jan-95	11:55 AM	24.6	102.08	1.124	1.73	781	13.20	2.78	38	
103	30-Jan-95	01:30 PM	25.6	103.15	2.743	1.89	769	12.80	2.64	24	
104	31-Jan-95	10:05 AM	20.6	104.00	3.295	1.95	766	13.50	2.54	96	
105	1-Feb-95	12:30 PM	26.4	105.10	2.934	1.90	771	13.80	2.48	91	
106	2-Feb-95	02:15 PM	25.8	106.18	3.040	1.92	767	13.20	2.58	46	
107	3-Feb-95	11:30 AM	21.3	107.06	2.897	1.88	764	13.00	2.63	57	slurry sample at 1:35pm
108	4-Feb-95	12:10 PM	24.7	108.09	3.145	1.95	765	13.00	2.56	79	slurry sample at 2:05pm
109	5-Feb-95	12:55 PM	24.8	109.12	3.065	1.98	762	12.80	2.65	62	slurry sample at 3:10pm
110	6-Feb-95	11:45 AM	22.8	110.07	2.758	1.91	767	12.70	2.53	70	

Appendix D - Continuous Reactor Data

CONCENTRATE CONTINUOUS BIOLEACH TEST 1

Flowrate (L/day):	1	1	1	1	1	1	1	1	1	1
Description of State:	steady	steady	steady	steady	steady	steady	steady	steady	steady	steady
Sample Tank (C#):	0	1	2	3	1	2	3	1	2	3
Sample Date:	03-Dec-94	28-Nov-94	28-Nov-94	28-Nov-94	01-Dec-94	01-Dec-94	01-Dec-94	05-Dec-94	05-Dec-94	05-Dec-94
Sample Time:	03:25 PM	02:45 PM	02:45 PM	02:45 PM	02:45 PM	02:45 PM	02:45 PM	02:15 PM	02:15 PM	02:15 PM
Sample Volume (ml):	101.000	101.000	102.000	100.000	102.000	100.000	101.000	100.500	101.000	101.000
Sample Weight (g):	105.660	107.220	106.950	105.160	108.010	106.060	108.110	106.980	106.860	106.410
Sample Slurry Density (g/L):	1046.139	1061.584	1048.529	1051.600	1058.922	1060.600	1070.396	1064.478	1058.020	1053.564
AVG. SLURRY DENSITY (g/L):	1046.1	1061.7	1055.7	1058.5						
Feed Slurry Volume (ml):	101.000	101.000	102.000	100.000	102.000	100.000	101.000	100.500	101.000	101.000
Mass Solids per Volume (g/100ml):	5.000	5.000	5.000	5.000	5.000	5.000	5.000	5.000	5.000	5.000
Mass Solids in Feed (g):	5.050	5.050	5.100	5.000	5.100	5.000	5.050	5.025	5.050	5.050
% Cu in Feed (%):	17.300	17.300	17.300	17.300	17.300	17.300	17.300	17.300	17.300	17.300
Mass Copper in Feed (g):	0.874	0.874	0.882	0.865	0.882	0.865	0.874	0.869	0.874	0.874
% Sulphide in Feed (%):	11.620	11.620	11.620	11.620	11.620	11.620	11.620	11.620	11.620	11.620
Mass Sulphide in Feed (g):	0.587	0.587	0.593	0.581	0.593	0.581	0.587	0.584	0.587	0.587
Filtrate Volume (ml):	92.000	93.000	93.000	93.000	96.000	95.000	98.000	96.000	97.000	94.500
Wash Volume (ml):	423.000	420.000	455.000	465.000	398.000	534.000	666.000	425.000	448.000	445.000
Solid Residue (g):	4.620	5.460	4.240	4.810	4.750	4.060	4.300	4.380	4.070	4.380
Cu Conc'n Filtrate (g/L):	2.320	6.400	7.400	8.200	6.500	7.300	8.100	6.650	7.450	8.350
Cu Conc'n Wash (g/L):	0.062	0.086	0.091	0.102	0.096	0.078	0.064	0.084	0.085	0.104
Solid Residue % Cu (%):	9.100	2.360	2.610	1.990	2.380	2.180	1.600	2.410	2.700	1.950
Mass Cu Filtrate (g):	0.213	0.595	0.688	0.763	0.624	0.694	0.794	0.638	0.723	0.789
Mass Cu Wash (g):	0.026	0.036	0.041	0.047	0.038	0.042	0.043	0.036	0.038	0.046
Mass Cu Solids (g):	0.420	0.129	0.111	0.096	0.113	0.089	0.069	0.106	0.110	0.085
Total Mass Cu (g):	0.660	0.760	0.840	0.906	0.775	0.824	0.905	0.780	0.871	0.921
Overall Cu Extraction (%):	36.3	83.0	86.8	89.4	85.4	89.3	92.4	86.5	87.4	90.7
AVG. OVERALL Cu EXTRACTION (%):	36.3	85.0	87.8	90.9						
Solids Cu Actual/Feed Ratio:	0.755	0.870	0.952	1.047	0.879	0.952	1.036	0.897	0.997	1.054
Solid Residue % (S^{-2} + S$^\circ$) (%):	6.07	5.66	4.73	5.2	5.2	4.9	3.23	3.79	4.34	3.45
Overall Sulphate Formation (%):	36.7	39.5	64.5	58.9	52.6	64.0	77.2	68.3	69.8	75.6
AVG. OVERALL SULPHATE FORMATION (%):	36.7	53.4	66.1	70.5						

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