AN INVESTIGATION OF LEACHING CHALCOPYRITE ORE

By

James Schaming

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Canada

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Abstract

The abiotic leaching behavior of a chalcopyrite ore, from Asarco's Ray-Mine, was conducted in shake flasks and miniature columns at elevated temperatures. The shake flask tests, with an ore particle size of 1.18mm-2.38mm, found the highest Cu extraction was obtained using 1M NaCl in a 9.8g/L sulphuric acid solution at 60°C, with 69% Cu extracted in 16 days. The next highest extraction, 59% Cu extracted in 16 days, was achieved by adding fine pyrite at a 4:1wt ratio with the chalcopyrite content, in a 9.8g/L sulphuric acid solution at 60°C. Flask tests using other lixiviants and additions found copper extractions in the range of 30-40% Cu after 16 days. In the mini-column tests, the rates of copper extraction were similar for all test conditions. The rate of Cu extraction, even with a small particle size of 1.18mm-2.38mm and an elevated temperature of 50°C, was slow for all test conditions with an average rate of ~0.15% Cu per day.

The conceptual engineering of a hot, abiotic heap-leach for low-grade chalcopyrite ore, including hypothetical heat and mass balances was conducted. The leaching time for a commercial operation was estimated from published data on laboratory column leaching of chalcopyrite ores and extrapolated to a commercial heap-leach by analogy with known leaching times for chalcocite ores. In commercial abiotic heap-leaches of chalcopyrite ore, the partial oxidation reactions generate insignificant heat to maintain an elevated heap temperature therefore the heat required to maintain the elevated temperature must be provided externally. In commercial biotic chalcopyrite heap-leaches, the in-situ total oxidation reaction generates more heat than the abiotic reactions but is still insufficient to rapidly raise and maintain an elevated heap temperature. For a low-grade Chalcopyrite heap-leach the most practical method of providing this heat is by injecting steam into the base of the heap using current air injection pipes. An external oxidant is required and for an abiotic heap-leach external ferric generation will be required.

A comparison of the economics of a conceptual low-grade copper heap-leaching operation was made with a proven concentrator option for a hypothetical supergene heap-leaching operation that is nearing the end of the supergene ore mine life and still has a large hypogene reserve of low-grade, 0.5% copper, chalcopyrite. Based on the process and economic assumptions, it was demonstrated that the economics of the hypothetical chalcopyrite leach leaching operation were not as favourable as those for a conventional concentrator.

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Chapter 1 Introduction

1.1 Background

Most of the world's copper production comes from porphyry copper deposits that typically consist of three types of copper minerals as a function of depth, see Figure 1.1 as an example. Close to the surface is a small shallow zone consisting mainly of oxidized copper minerals. Below this is a larger supergene enrichment zone, where the copper in the ore is mainly chalcocite (Cu_2S) and other secondary copper minerals. Finally, beneath this is an even larger hypogene zone composed mainly of lower-grade chalcopyrite ($CuFeS_2$) ore.

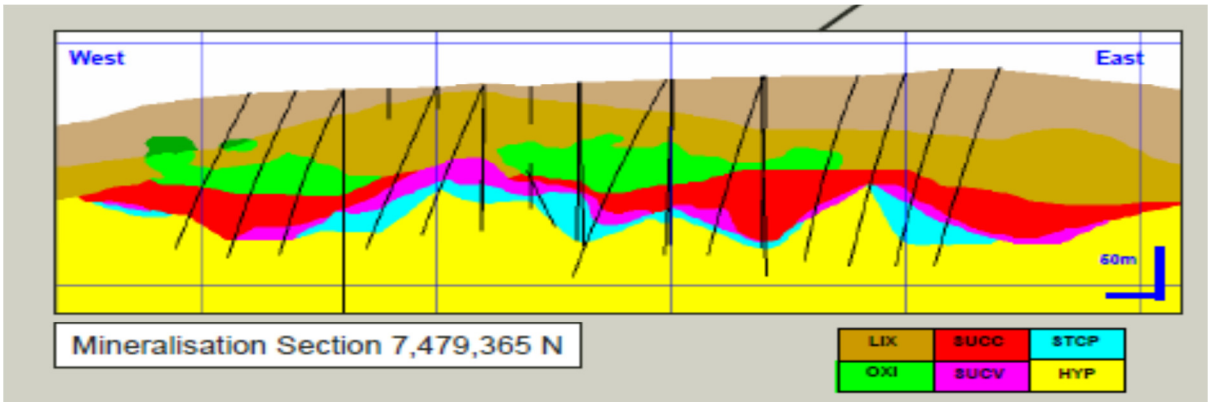

Figure 1.1: An Example of the Three Types of Copper Mineralization from the Spence Porphyry Deposit in Chile.

Chalcopyrite is the most common copper mineral accounting for about 80% of the world's copper production and approximately 70% of the world's copper reserves (Rivadeneira, 2006). Currently, all commercial chalcopyrite ore operations use grinding and flotation to recover copper. This concentrate, 25-35% Cu, is then smelted to produce copper anodes for electro-refining to copper cathode.

1.2 The History of Copper Heap Leaching

Commercial scale copper heap leaching on a small scale has been practiced for a long time but it has rapidly grown over the past forty years. One of the first copper dump leaching operations was at Rio Tinto, Spain around 1752. In the 1960s, the Stovall Copper Company was the first to develop commercial scale heap leaching of copper oxide ores, using solvent extraction and electrowinning, at the Bluebird Mine, Arizona (Scheffel, 2002).

The heap leaching process consists of crushed ore being placed in heaps that are then irrigated with a dilute acidic solution applied via either sprinklers or a drip system. The aqueous solution, containing the copper, is collected at the base of the heap. This pregnant solution is transported to the solvent extraction (SX) step where the copper is transferred from the aqueous solution to the organic extractant. The loaded organic is then stripped of its copper using spent electrolyte in the stripping stage. The electrolyte solution, enriched with copper, is then sent to the electro-winning (EW) cells to produce copper cathodes. This process is outlined in Figure 1.2.

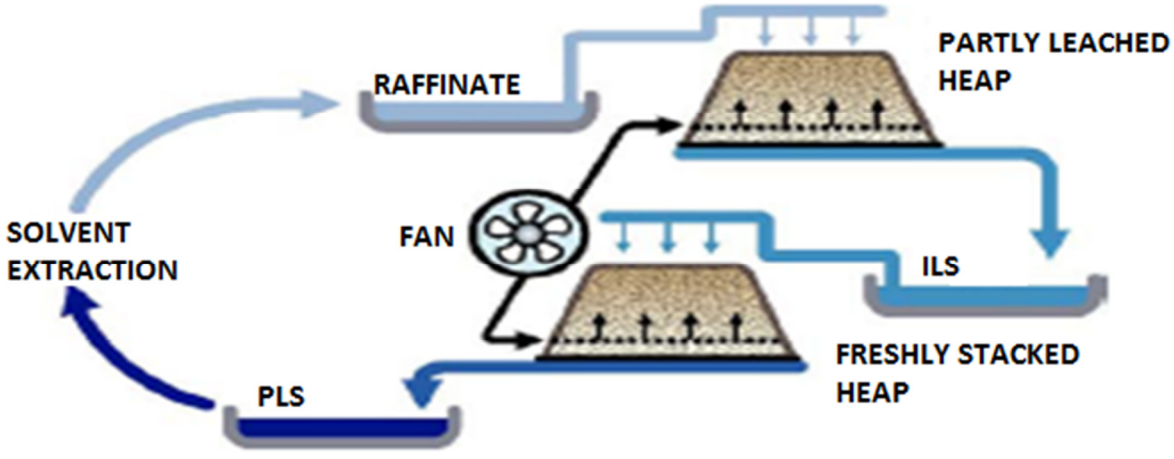

Figure 1.2: Sulphide Ore Heap Leaching Process (Watling, 2006)

1.3 Chalcocite Heap Leaching

Heap leaching/SX/EW applied to oxide copper ore is well established as a low-cost method of copper recovery. In the 1990s, this technology was successfully applied to secondary copper sulphide ores, especially chalcocite ores. In copper oxide ore leaching, the copper is dissolved using only sulphuric acid to form copper sulphate, see equation 1.

$$CuO_{(s)} + H_2SO_{4(aq)} = CuSO_{4(aq)} + H_2O_{(aq)} \quad (1)$$

Chalcocite ores, on the other hand, can only be leached under oxidizing conditions with ferric ions, see equation 2.

$$Cu_2S + 4Fe^{3+} = 2Cu^{2+} + 4Fe^{2+} + S^o \quad (2)$$

In practice the heap is aerated via fans and piping located in the base of the heap. This is to done to ensure that enough oxygen is available for the oxidation of ferrous to ferric, see equation 3.

$$4Fe^{2+} + O_2 + 4H^+ = 4Fe^{3+} + H_2O \quad (3)$$

The above reaction is catalyzed by bacteria and requires both oxygen and acid. The elemental sulphur formed, see equation 2, is also partially oxidized by bacteria to form sulphuric acid, see equation 4. This reduces the overall sulphuric acid required for the leaching process.

$$2S^o + 3O_2 + 2H_2O = 2H_2SO_4 \quad (4)$$

Crushed chalcocite ore, typically crushed to a d_{80} = 12 mm, is leached much more slowly than oxide copper ores. It typically takes a year or longer to obtain up to 80% copper extraction from chalcocite ore compared to less than 100 days to reach the same extraction from oxide copper ores (Dixon, 2007).

Dixon (2007), conducted column leaching tests on chalcocite ore that demonstrates the effect of heap height on the overall copper extraction, Figure 1.3. It was shown that, as the column height increases, the copper extraction rate decreases. This is due to the limited solution application rate and the solution percolation issues with taller heaps.

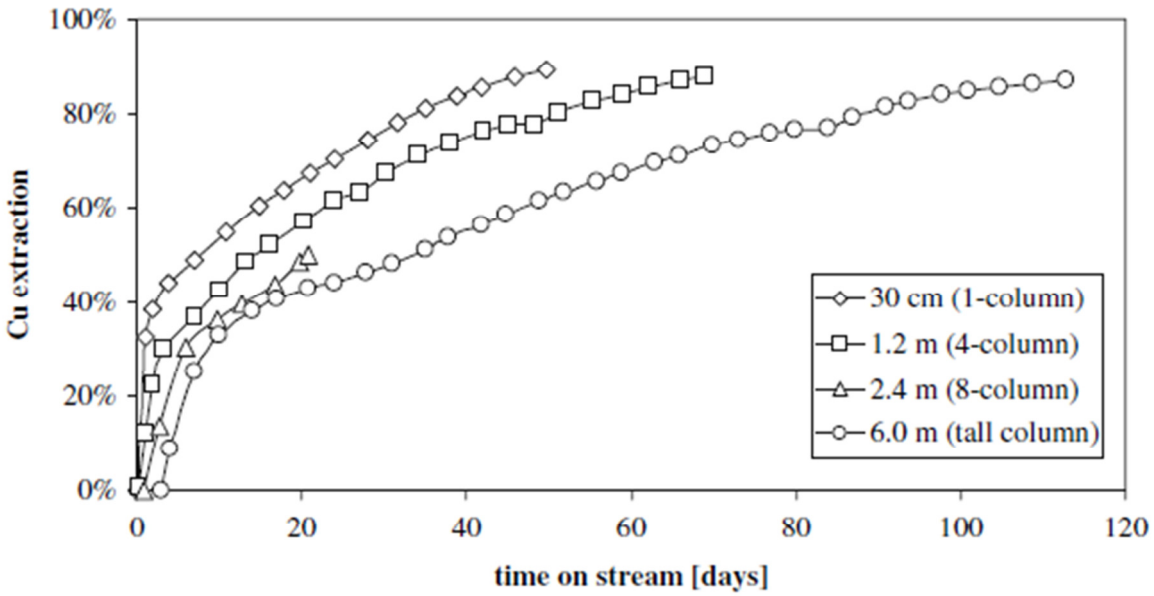

Figure 1.3: Cu Extraction vs. Time for Columns of Different Heights with the Same Width and Solution Application Rates (Dixon, 2007).

1.4 Leaching of Chalcopyrite Concentrates

Currently there are no commercial chalcopyrite leaching operations that have operated successfully on a commercial scale, though many processes for leaching chalcopyrite concentrate have been developed over the years as an alternative to smelting. High-temperature pressure oxidation (HTPOX) leaching was successfully demonstrated at a 15,000 tpa scale in 2002 by Phelps Dodge at its Bagdad, Arizona plant (Ford, 2009). The HTPOX process is conducted by reacting a chalcopyrite concentrate slurry with oxygen at a temperature of approximately 220°C in a multi-compartment stirred autoclave. Over 95% of the copper is dissolved from the concentrate with a residence time of around 1 hour.

Freeport-McMoran built an 80,000 tpa copper plant in 2008 using a similar medium temperature pressure oxidation (MTPOX) process to treat chalcopyrite concentrate. In the MTPOX process, conducted at a lower temperature of around 160°C, most of the sulphide sulphur is not oxidized thus reducing the oxygen requirements. After only a year in operation the plant was closed due to various technical and economical issues. To date it has not been reopened.

1.5 Research on Abiotic Leaching of Chalcopyrite Ore

Heap leaching of chalcocite ores with copper recovery from the aqueous leach solution by solvent extraction and electro-winning (SX/EW) has been widely implemented, especially in Chile and Arizona. Heap leaching and SX/EW has become a more economical method of processing chalcocite ores than conventional concentrator methods. Therefore there has been much research in the attempt to adapt current heap leaching technologies to chalcopyrite ores.

The extension of heap leaching to chalcopyrite ores is being extensively investigated by many researchers and most large copper companies. The main driver for this research is the fact that the average grades in copper mines have been declining rapidly in recent years, see Figure 1.4, and a more economical processing option for low-grade chalcopyrite ores than the current concentrator process is required.

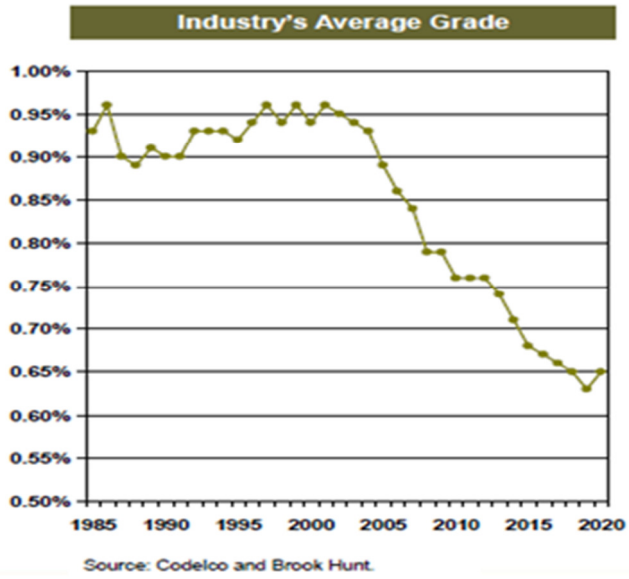

Figure 1.4: Average Grade of Treated Copper Ore: Historical and Forecast. (Hernandez, 2010)

Chalcopyrite leaching is known to be extremely slow around the ambient temperatures experienced in most commercial heap leaching operations. Much higher heap temperatures, in the range of 50 to 80°C, would be required for a commercial chalcopyrite heap leach. The majority of the research on leaching of chalcopyrite ores has been based on biological leaching using moderate and extreme thermophiles, organisms that are tolerant to the higher-temperatures, 50 to 80°C, required for leaching chalcopyrite ore. Relatively little research has been done on abiotic leaching of chalcopyrite ores even though the research done thus far suggests that it may be just as effective and easier to implement than biological leaching (Vilcaez, 2009). Bio-leaching of chalcopyrite is complex since the organisms used are very sensitive to operating temperatures. Abiotic leaching may therefore be easier to implement on a commercial scale.

The easiest possible application of heap leaching chalcopyrite ores will be at an existing chalcocite heap leach operation when the chalcocite ore reserves are exhausted and the only ore remaining is the chalcopyrite containing hypogene. In this case, the heap leach/SX/EW infrastructure is already in place and the capital costs to convert to a chalcopyrite heap leach should be considerably less than the alternative of constructing a new concentrator and tailings dam.

The objective of this research was to study:

- The abiotic leaching of a chalcopyrite ore in shake flasks and mini-columns at elevated temperatures to try to determine the effect of different lixiviants on copper dissolution;
- The conceptual engineering of a hot, abiotic heap leach for low-grade chalcopyrite ore, including hypothetical heat and mass balances.
- Comparison of the economics of a conceptual chalcopyrite heap leach with the proven concentrator option for a hypothetical supergene heap leaching operation that is nearing the end of the supergene ore mine life, yet has a large reserve of low-grade chalcopyrite ore (0.5% Cu).

Chapter 2 Literature Review

2.1 Background on Chalcopyrite

Leaching of chalcopyrite concentrate has been shown to have slow kinetics at temperatures below 100°C and atmospheric pressure. The majority of past literature has been based on the leaching of chalcopyrite concentrates and there are relatively few references on the leaching of chalcopyrite ore. Though the data found within papers on concentrates are a good starting point, it is possible that the conclusions drawn from tests on concentrate may not be applicable to the leaching of chalcopyrite ores. With copper ore grades declining, the future requirements for copper ore processing will be incorporating chalcopyrite ore grades of 0.5% Cu or less. Though the leaching of chalcopyrite ores will be dependent on their mineralogy, this chapter will review the current status on the leaching of chalcopyrite.

Dutrizac (1974) proposed that the reaction of chalcopyrite with ferric ions was as given by the following reaction.

$$CuFeS_2 + 2Fe_2(SO_4)_3 \rightarrow CuSO_4 + 5FeSO_4 + 2S^o \quad (5)$$

Lu (2000) proposed that the dominant reactions of chalcopyrite leaching are reactions (6) and (7). Reaction (6) shows the reaction of chalcopyrite with oxygen and acid, whereas reaction (7) shows the reaction of chalcopyrite with ferric ions.

$$CuFeS_2 + 4H^+ + O_2 \rightarrow Cu^{2+} + Fe^{2+} + 2S^0 + 2H_2O \quad (6)$$

$$CuFeS_2 + 4Fe^{3+} \rightarrow Cu^{2+} + 5Fe^{2+} + 2S^o \quad (7)$$

2.1.1 Passivation of Chalcopyrite Concentrate

A major issue with the leaching of chalcopyrite concentrate in an acidic solution of ferric sulphate at atmospheric pressure and temperatures below 110^{o}C, is that copper dissolution typically slows down once approximately 30% of the copper is disolved, depending on the concentrate. Many authors have concluded that this "passivation" effect is due to the formation of a film on the surface that does not allow further reaction to occur but there is no consensus to the actual composition of this layer. Munoz (1979) proposed that a layer of sulphur formed around the chalcopyrite particle, Figure 2.1.

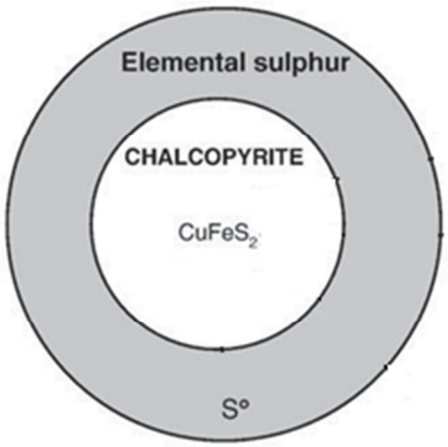

Figure 2.1: Suggested Passivating Elemental Sulphur Layer of a Chalcopyrite Particle. (Munoz, 1979)

An impervious layer of sulphur could potentially stop the leaching of a chalcopyrite particle once it reached a specific thickness. More recent research proposes that the layer is formed by an iron deficient copper sulphide thus preventing further leaching (Cordoba, 2008). Another recent paper has proposed that the passivating layer consists of jarosite that has precipitated on the surface of the particle (Dixon, 2008).

The four main candidates for the passivation of chalcopyrite concentrate that are most often proposed are; metal deficient sulphides, formation of elemental sulphur or polysulphides: XS_n , and jarosites $XFe_3(SO_4)_2(OH)_6$ (Klauber, 2007). Klauber concluded that the prime candidate for any concentrate leaching passivation is elemental sulphur. Lu (2000) suggests that a more porous layer of sulphur is formed in the presence of chloride.

Jarosite formation is also a possible inhibitor of chalcopyrite concentrate leaching. Three principal methods have been proposed by various researchers to deal with jarosite in leaching applications. The first method is to prevent jarosite formation by operating at a low pH where jarosite does not precipitate. A second suggestion is to induce jarosite to precipitate on other particles to prevent it from coating the chalcopyrite particles. A final option is to remove iron from the leaching solution circuit, thus reducing the potential for jarosite precipitation.

The passivation of chalcopyrite in atmospheric leaching of concentrate is well documented. However, there is no evidence to show that it occurs in the atmospheric leaching of chalcopyrite ore. In the leaching of chalcopyrite ore, especially with heap leaching ore particle sizes of about 10 mm, the chalcopyrite mineral particles may not be liberated and the leach solution may never contact the mineral. Within low grade ores, the chalcopyrite mineral is a minute component and the effect of the gangue minerals, particularly acid consuming gangue minerals, make controlling the local pH difficult. The gangue minerals may also increase the occurrence of iron and copper precipitation reactions.

The heap leaching of chalcopyrite ore, even at high temperatures, is considerably slower than that for finer concentrate and is measured in years rather than days. All these factors make it much more difficult to observe any of the so-called “passivation” effect.

2.1.2 The Influence of Redox Potential on Chalcopyrite Leaching

Redox potential has been shown to be a key factor in the leaching of chalcopyrite concentrate. Literature has shown that at a pH less than 2 the critical potential is between 400 to 500 mV vs Ag/AgCl. Kametani (1985) showed that at a temperature of 90°C, the leaching rate increased with increasing potential up to a critical point of 458 mV Ag/AgCl. Cordoba (2008) concluded that a potential of approximately 450 mV Ag/AgCl at the onset of chalcopyrite leaching provoked rapid surface passivation due to the precipitation of ferric ions as jarosite. Hiroyoshi (2008) found the amount of copper extracted increased markedly at potentials below 610 mV vs SHE (~410 mV Ag/AgCl), see Figure 2. 2.

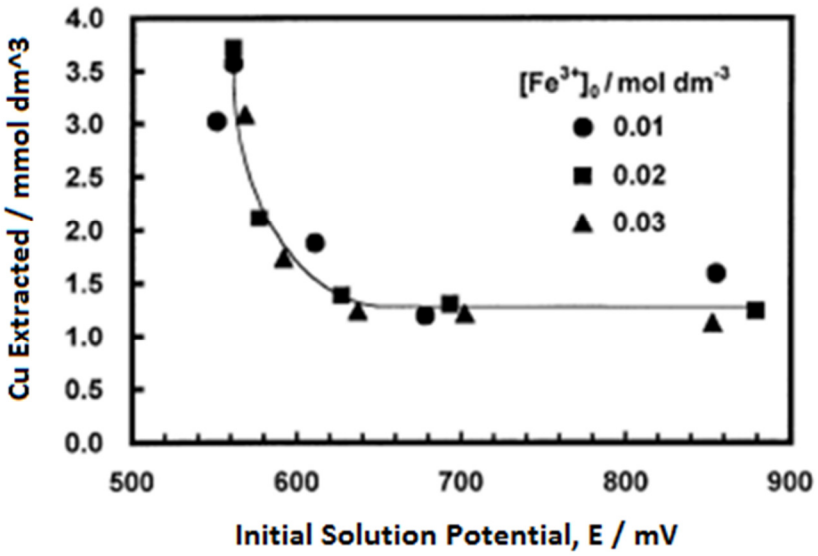

Figure 2.2: Relationship Between the Initial Redox Potential of Solution and the Amount of Copper Extracted from -200 mesh Chalcopyrite Sample after 24h in 0.1 mol dm^{-3} Sulphuric Acid Containing Various Concentrations of Ferric Ions.(Hiroyoshi, 2008)

In biological heap leaching the redox is controlled by the organisms. Most bacteria are very efficient at oxidizing ferrous to ferric and tend to operate at higher ferric to ferrous ratios. Some extreme thermophiles are less efficient at oxidizing ferrous and maintain higher ferrous to ferric ratios, which may be more suitable for leaching chalcopyrite ores.

It is generally agreed that chalcopyrite is leached prodiminantly by ferric but that leaching with only ferric ions results in rapid passivation and low copper dissolution. The use of ferrous ions has also been shown to increase copper recovery.

Hiroyoshi (1997) found that several chalcopyrite concentrate samples were more effectively leached in ferrous sulphate solution than in ferric sulphate solution. Figure 2.3 shows that the addition of ferrous sulphate to solution increases copper extraction.

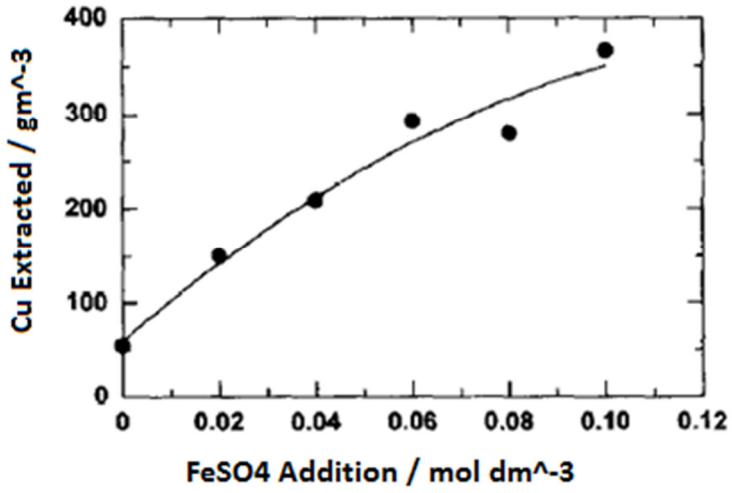

Figure 2.3: The effect of Ferrous Concentration on Copper Extraction After 168 hrs of Leaching (Hiroyoshi, 1997).

Hiroyoshi also showed that at a low pH, less than or equal to 1, the addition of ferrous iron to solution further increased copper extraction, Figure 2.4.

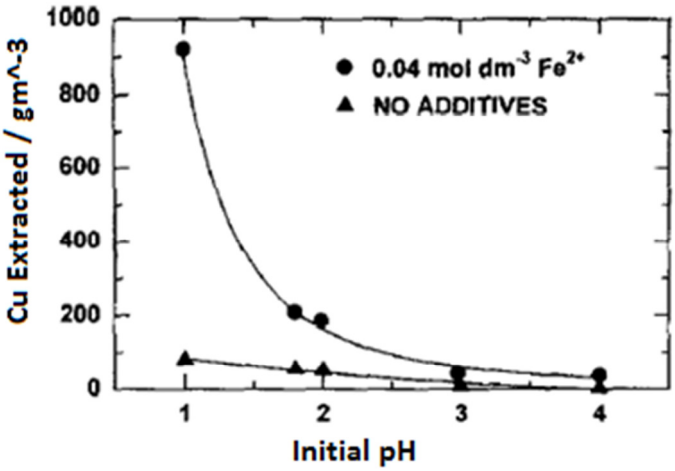

Figure 2.4: The Effect of Initial pH on Copper Extraction for 168 h, with and without Ferrous Sulphate (Hiroyoshi, 1997).

The effect of redox potential on chalcopyrite ore leaching has not been studied as extensively. However a recent patent awarded to Hollitt (2008) shows that a high ferric solution, ferric to ferrous ratio of 1000:1, at 50°C leaches copper more extensively than at a lower ORP, a ferric to ferrous ratio of 2:1, see Table 2.1. However, much less pyrite was oxidized when operating at a low ORP.

Table 2.1: Hollitt 2008 Patent: Copper Extraction Comparison Based on Ferric to Ferrous Ratios

Hollitt 2008 Patent	Example 2	Example 3
Ferric : Ferrous	1000:1	2:1
Temperature (C)	50	50
Particle Size (mm)	12	12
Cu Extraction in 140 days	75%	65%
Pyrite Oxidation	38%	10%

In Hollitt's 2008 patent it is shown that the redox potential is not important for Cu dissolution. It was shown that the redox potential is important for controlling the amount of pyrite oxidizing, in regards to acid generation and iron dissolution.

2.2 Operating Parameters of Interest

2.2.1 Acid Concentration

The acidity (pH) of the leaching solution applied to the heap is very important. The acid consumption depends on the ore mineralogy and the acidity of the PLS solution exiting the heap. The lower the pH of the PLS the higher the acid consumption. In concentrate leaching, high acid concentrations are usually employed, up to 100 g/L of acid, whereas in the heap leaching of chalcopyrite ore, lower acid concentrations are used to minimize acid consumption in order to reduce operating costs.

Most biological heap leaching operations must be operated at a pH > 1 since most organisms cannot tolerate a lower pH. The main advantage thought to come from biological leaching of chalcopyrite ores with extreme thermophiles is their ability to operate at high temperature, lower redox potential and to oxidize elemental sulphur thus removing any sulphur that may accumulate on the mineral surface, (Jordan, 2006). Vilcaez (2009) however recently showed that the abiotic leaching of a chalcopyrite concentrate at pH = 1 had significantly higher copper extractions than both abiotic and biological leaching at a pH = 1.5 at the same temperature of 80°C, see Figures 2.5 & 2.6.

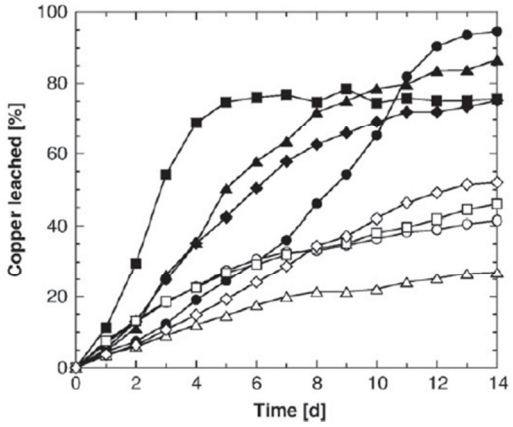

Figure 2.5: Biological Leaching Copper Extractions of Four Chaclopyrite Concentrates using Extreme Thermophilles at pH 1.5 and 80°C, Solid Symbols with Thermophiles only and the Open Symbols were with 200 mg/L of Additional Ferric Ions. (Vilcaez, 2009)

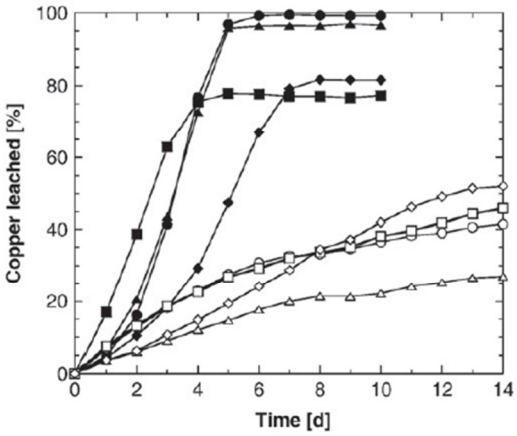

Figure 2.6: Abiotic Leaching Copper Extractions of Four Chaclopyrite Concentrates at 80°C, the Solid Symbols Tests were Run at a pH of 1 and the Open Symbols were Run at a pH of 1.5. (Vilcaez, 2009)

Maley(2009) showed that when the leaching solution pH rose above 2.3 that the copper ions previously released to solution, through the oxidation of chalcopyrite, were retained by adsorption on, or reaction with, ore minerals. It was recommended to maintain the pH within a heap at as low a pH level as is conducive to bacterial growth. The restraint imposed upon the acidity levels by relying on bacteria to create oxidizing conditions may be more detrimental to the overall recovery and economics of a full scale heap leach. Therefore this research was to be conducted by focusing on low pH of 1 or less which is not conducive to bacterial growth.

2.2.2 Temperature

It is well known that chemical reaction rates usually increase considerably at higher temperature. Figure 2.7 shows the effect of temperature on the leaching of chalcopyrite concentrate, (Lu, 2000).

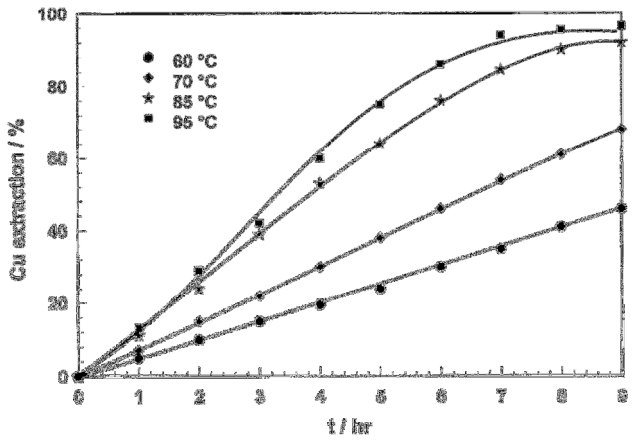

Figure 2.7: Effect of Temperature on Copper Extraction from a Chalcopyrite Concentrate. (Lu, 2000)

Van der Meer (2009) conducted laboratory column biological leaching tests and found that the effect of temperature on chalcopyrite ore leaching was even more dramatic, Table 2.2.

Table 2.2: Effect of Temperature on Copper Extraction from Chalcopyrite Ore in Biological Leaching Column Tests (van der Meer, 2009)

Test	Temp. (°C)	Temp. (K)	Cu Extraction in 230 days
1	20	293	18%
2	50	323	52%
3	70	343	74%

A typical chalcocite ore heap leach operation is run at ambient temperature and approximately 70 to 80% of the copper can be heap leached in about one year using acidified ferric sulphate solutions. Based on the current status of development, chalcopyrite ores would require the heap to be operated at elevated temperatures of 50^{o}C or higher for several years to approach similar copper recoveries. The operation of a chalcopyrite ore heap leach at high temperature will be discussed in more detail in chapter 5.

2.2.3 Particle Size

The effect of using smaller particle sizes of chalcopyrite is a well known factor in increasing overall copper dissolution. Figure 2.8 shows an example which has demonstrated the advantage of a smaller sized particle.

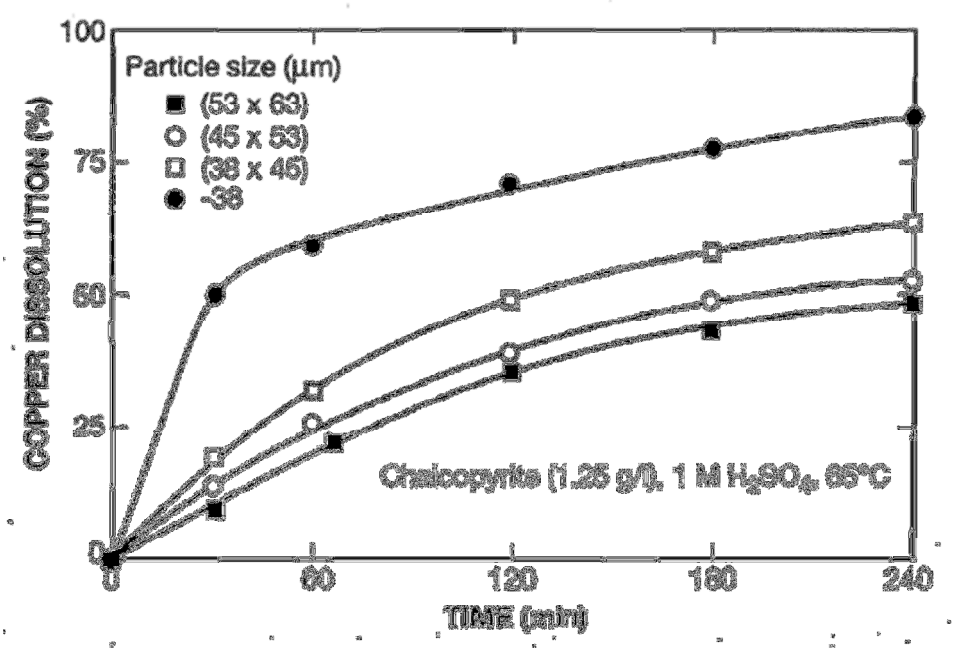

Figure 2.8: Effect of Particle Size on Copper Dissolution from a Chalcopyrite Concentrate. (Fuerstenau, 2005)

It is important to note, for a heap leaching operation, that the minimum ore size that allows for good percolation of the leaching solution is typically limited to a d_{80} particle size ranging between 5 to 10 mm, dependent upon the amount of minus 100 micron fines in the heap.

2.2.4 Chloride Additions

The leaching of chalcopyrite concentrates with chloride addition has shown faster leaching kinetics than in a sulphate only solution. Lu (2000) observed that in sodium chloride solutions the presence of sodium ions caused sodium-jarosite precipitation during leaching. Carneiro (2007) also found that copper extraction rates from a chalcopyrite concentrate with a particle size of 5.5 µm could be increased from 40% to 90% after 10 hours of leaching by adding 2M NaCl to a 50 g/L ferric in a sulphuric acid solution with a pH of 0.15 and a temperature of 95°C.

BHP Billiton patented a process in 2007 that uses an acidic ferric chloride solution at a temperature range of 35^{o}C - 100^{o}C and a pH less than one. The experimental setup used small reactors to conduct the leaching and achieved 95% Cu recovery in 33 days at 35^{o}C, (Muller, 2007).

Carneiro (2007) found that the leach residue's surface area and porosity are higher in the presence of sodium chloride. He proposed that the higher extraction in the presence of NaCl is observed because the higher porosity facilitates diffusion of reagent and products to and from the reaction sites. It was also shown that chalcopyrite in the presence of chloride showed that a more porous sulphur layer was produced on the chalcopyrite surface. Lu (2000) showed similar results when leaching chalcopyrite in the presence of chloride ions.

Qiu (2007) also found that a higher NaCl concentration increased copper recovery from chalcopyrite concentrate with a particle size of 44-53 µm at a temperature of 110°C using a 36 g/L sulphuric acid solution, see Figure 2.9.

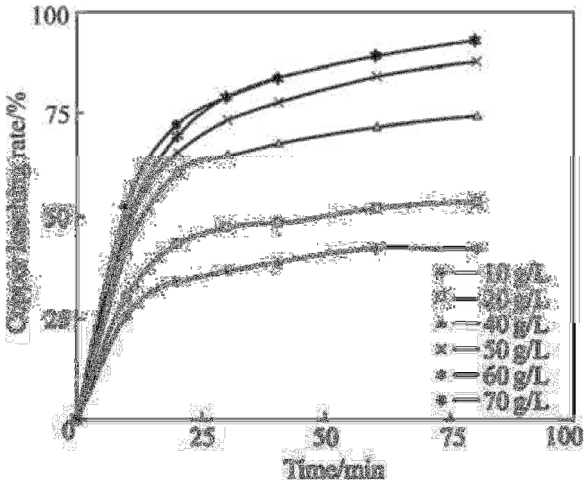

Figure 2.9: Effect of Different NaCl Concentrations on the Copper Leaching Rate of a Chalcopyrite Concentrate (Qiu, 2007).

Nicol (2010) found that, at a pH of 1, a higher chloride concentration increased the copper dissolution from a chalcopyrite ore with a particle size of 25-38 µm in hydrochloric acid at a temperature of 35°C, see Figure 2.10.

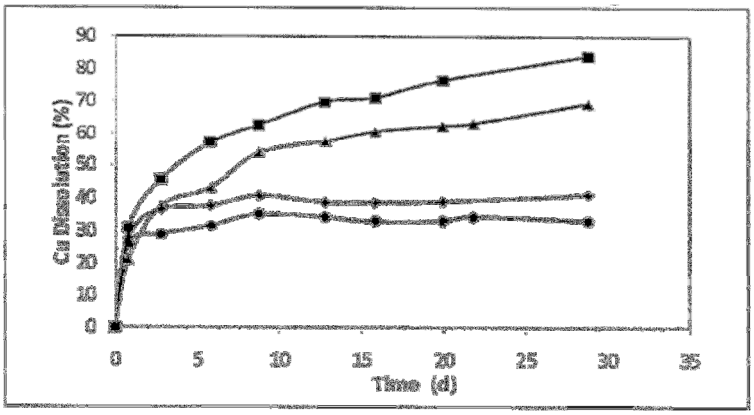

Figure 2.10: Effect of Different Chloride Concentrations on the Copper Leaching Rate of a Chalcopyrite Ore Specimen (Nicol, 2010).

2.2.5 Pyrite Additions

Berry (1978) has shown that there is a galvanic reaction between pyrite and chalcopyrite during ore leaching that increases the dissolution of copper from chalcopyrite. More recently, Dixon (2008) found that leaching chalcopyrite concentrate in the presence of ground pyrite at a level of two to four times the mass of chalcopyrite increased copper recoveries. He hypothesized that the pyrite creates an alternative, catalytic surface for ferric reduction in electrical contact with the chalcopyrite, which may alleviate the passive behaviour of chalcopyrite in a ferric sulphate solution. Pyrite is an effective and convenient provider of this alternative surface for ferric reduction. It is effective because the pyrite mass addition is roughly two to four times that of chalcopyrite which ensures rapid and complete copper extraction. This addition of pyrite for leaching of chalcopyrite concentrates has been patented as the Galvanox process.

For chalcopyrite ores that do not contain significant levels of pyrite, the addition of fine pyrite concentrate could also potentially enhance copper dissolution.

2.3 Heap Leaching of Copper Sulphide Ores

To date only chalcocite ores have been commercially heap leached. Chalcopyrite ore leaching has only been conducted in laboratory columns and pilot heaps. This section will review and compare laboratory column leaching of both chalcocite and chalcopyrite ores. It will then try to extrapolate an estimate for the commercial performance of a chalcopyrite ore leach by analogy with the relative performance of laboratory columns and commercial chalcocite heaps.

2.3.1 Chalcocite Ore Leaching

Laboratory column leaching of chalcocite ores typically achieve approximately 80% Cu extraction in less than 100 days. Figure 2.11 shows the copper recoveries for chalcocite column leach tests with a particle size of d_{80} = 20 mm, at ambient temperature and using a 7.5 g/L sulphuric acid solution and an irrigation rate of 5 L/h/m^2 , Dixon (2007).

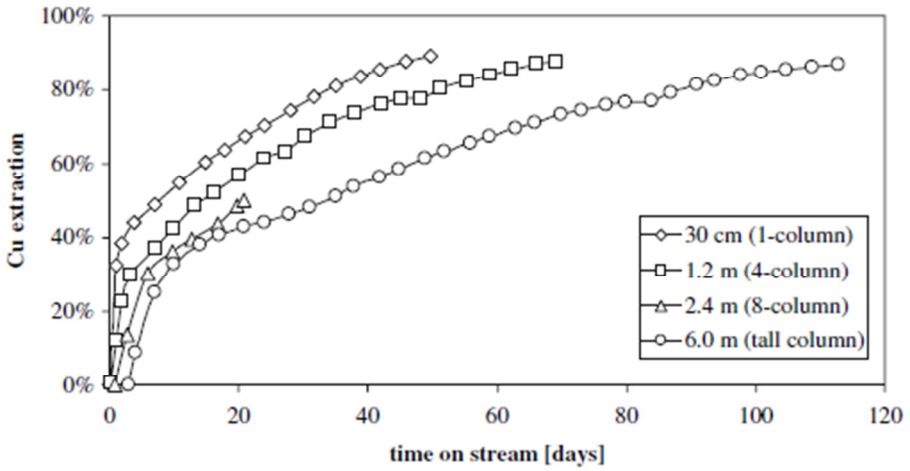

Figure 2.11: Chalcocite Laboratory Column Leach Tests Copper Recovery Plot (Dixon, 2007)

Dixon (2007) suggests that the diffusion of the leaching solution in a column is faster due to the solution confinement at the walls. In an unconfined heap, irrigated by drip emitters, the reagents and reaction products have further distance to diffuse laterally through stagnant solution.

Commercial chalcocite ore heap leaches usually take 300 to 500 days to approach 80% extraction. Figure 2.12 shows the typical commercial extraction profiles for seven different chalcocite dominated heap leaching operations, (Scheffel, 2006). The actual heap conditions were not identified by the author to protect the confidential nature of some of the information. It was noted that all but one of the operations used forced aeration and the sulphuric acid used ranged in acidity from 4 to 10 g/L.

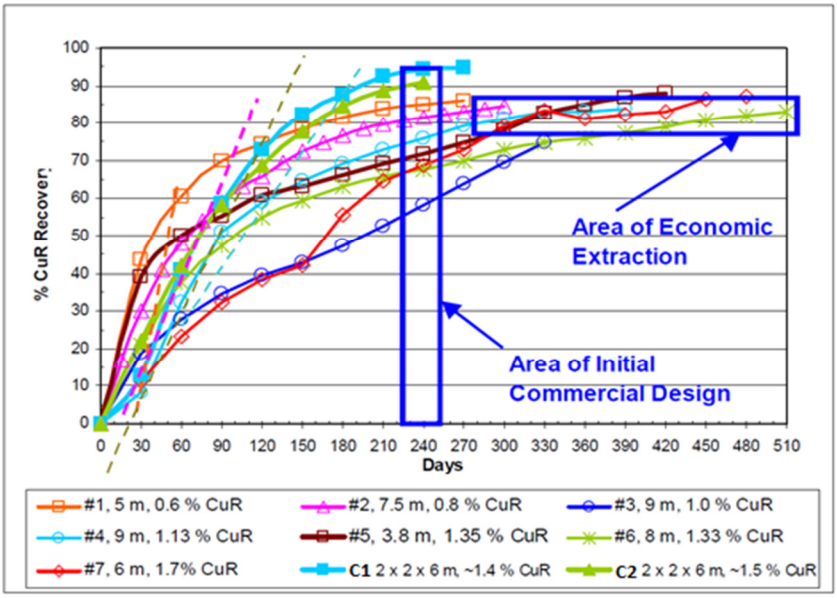

Figure 2.12: Typical Commercial Chalcocite Heap Leach Copper Recovery Plot (Scheffel, 2006)

Figure 2.12 shows that a commercial chalcocite heap takes approximately 5 times longer to achieve 80% copper extraction than typical results from a laboratory column leach test.

2.3.2 Chalcopyrite Ore Leaching

Little data has been published on laboratory column leaching of chalcopyrite ores. The data found is summarized in Table 2.3.

Table 2.3: Summary of Laboratory Column Leach Tests of Chalcopyrite Ore

	Hollitt (2008)		Muller (2007)				Van Staden (2008)		
Test	**2**	**3**	**1**	**2**	**3**	**4**	**1**	**2**	**3**
Temp. (C)	50	50	20	70	50	50	30	30	60
Bacteria	Y	N	Y	Y	Y	Y	Y	N	N
Lixivant	Ferric Sulphate	Ferric Sulphate	Ferric Sulphate	Ferric Sulphate	Ferric Sulphate	F.S. + 130 g/L NaCl	Ferric Sulphate	Ferric Sulphate	Ferric Sulphate
pH	1.5	1.5	1.7	1.7	1.7	1.7	1.5	1.5	1.5
Days Leached	140	232	229	229	230	230	338	216	294
% Cu Extracted	75%	75%	18%	74%	52%	67%	19%	17%	33%
Ore	La Granja, Peru		Aitik, Sweden				Haib, Namibia		
% Cu as Chaclopyrite	0.72%		0.25%				0.25%		
Ore Particle Size	100% < than 12 mm		Not Reported				14 to 19 mm		

The results in Table 2.3 show that the copper extraction from chalcopyrite ores is highly dependent upon temperature and ore type and that chalcopyrite ores leach much slower than chalcocite ores even at high temperatures. The effect of pH, bacteria and the lixivant is not as clear. The maximum copper extraction in the lab column tests was about 75%. In a commercial heap leach the copper recoveries are less due to the difficulty in leaching all of the ore in the heap and other conditions in a commercial sized heap. Based on these laboratory column results, the maximum copper recovery from this ore in a commercial heap leach would probably be approximately 70%.

Assuming a scale up factor of 3 to 5 times, for a chalcocite laboratory column leach test to a commercial chalcocite heap leach, would also be applicable to chalcopyrite heap leaching and it would require in the range of 750 to 1250 days to achieve about 70% Cu extraction, assuming that it is possible to maintain the heap at a high temperature.

2.3.3 Forced Aeration

Forced aeration is used in most commercial chalcocite ore heaps to provide a source of oxygen. The air is pumped by fans into porous pipes located at the bottom of the heap. The reason behind forced aeration in a chalcocite heap is to provide the oxygen required for the following reactions:

$$Cu_2S + 2Fe_2(SO_4)_3 \rightarrow 2CuSO_4 + 4FeSO_4 + S^o \quad (9)$$

$$4FeSO_4 + O_2 + 2H_2SO_4 \rightarrow 2Fe_2(SO_4)_3 + 2H_2O \quad (10)$$

$$FeS_2 + Fe_2(SO_4)_3 \rightarrow 3FeSO_4 + 2S^o \quad (11)$$

$$S^o + 1.5O_2 + H_2O \rightarrow H_2SO_4 \quad (12)$$

Oxygen requirements for oxidizing chalcocite and pyrite are shown in Table2.4.

Table 2.4: Oxygen Requirements for Oxidizing Chalcocite and Pyrite.

Reaction	Oxygen Required (mol O_2 / mol reacted)
$Cu_2S \rightarrow S^o$	1
$Cu_2S \rightarrow H_2SO_4$	2.5
$FeS_2 \rightarrow S^o$	0.5
$FeS_2 \rightarrow H_2SO_4$	3.5

The oxygen requirements for the heap reactions are relatively low if the reacted sulphide produces elemental sulphur. If the elemental sulphur is oxidized to acid then the oxygen requirements increase significantly. The oxidation of both ferrous iron and sulphur to acid are known to be catalyzed by bacterial reactions. The extent of these reactions within a chalcocite heap remains unknown.

In regards to chalcopyrite leaching, oxygen is required for the following reaction:

$$CuFeS_2 + 2Fe_2(SO_4)_3 \rightarrow CuSO_4 + 5FeSO_4 + 2S^o \quad (13)$$

Oxygen requirements for oxidizing chalcopyrite are shown in Table 2.5:

Table 2.5: Oxygen Requirements for Oxidizing Chalcopyrite

Reaction	***Oxygen Required (mol O_2 / mol reacted)***
$CuFeS_2 \rightarrow 2S^o$	1
$CuFeS_2 \rightarrow H_2SO_4$	4

The amount of oxygen required for a chalcopyrite heap is twice that of a chalcocite heap leach on an equivalent Cu basis, if elemental sulphur is the product of reaction. However, if the product of reaction is sulphuric acid, then the amount of oxygen required for the chalcopyrite reaction is just over 3 times the amount required for the chalcocite reaction.

The oxygen, from the air, utilization efficiency of a commercial heap has not been measured. Petersen (2010), conducted large column tests on chalcopyrite ore at a temperature of 40°C with a copper extraction of approximately 66% after 200 days, with a 5 g/L sulphuric acid solution at an irrigation rate of 6 L/h/m^2 and a measured oxygen utilization of approximately 30% from air at the start of the leach and only around 15% by the end.

The aeration requirements may also adversely affect both the water and heat balances in the heap, assuming that the air exiting a heap or dump is saturated air at the operating temperature of the heap. Therefore, blowing an excessive amount of air will evaporate a substantial amount of water. This will increase make-up water requirements. In addition, the evaporation will remove a significant amount of heat which could lead to significant cooling of the heap, thus retarding the leaching rate (Schlitt, 2006).

In a chalcopyrite ore heap leaching operation, the temperature within the heap must be maintained at a minimum of at least 50°C or higher. The saturated air exiting the top surface of the heap will remove a significant amount of heat that will need to be made up by the heat of reactions within the heap, namely the reaction of the chalcopyrite and pyrite minerals, or by another heat source.

For an abiotic chalcopyrite heap leach, as proposed in this thesis, the ferric containing leach solutions could be generated externally rather than in-situ. This would minimize the need for forced aeration and the amount of heat removed by the humid air. External ferric generation could be done in stirred tanks with air or oxygen injection. The ferric to ferrous ratio could also be controlled, to a certain degree, if required.

2.3.4 Heat Generation in Heap Leaching

Commercial chalcocite heap leaches typically operate at ambient temperatures, between 15°C and 25°C. The enthalpies of the reactions in a chalcocite heap are shown below:

$Cu_2S + 2Fe_2(SO_4)_3 \rightarrow 2CuSO_4 + 4FeSO_4 + S^o$ ΔH_{298} =-3.0 kJ/mol of chalcocite reacted.

$FeS_2 + Fe_2(SO_4)_3 \rightarrow 3FeSO_4 + 2S^o$ ΔH_{298} = -14.3 kJ/mol of pyrite reacted.

$4FeSO_4 + O_2 + 2H_2SO_4 \rightarrow 2Fe_2(SO_4)_3 + 2H_2O$ ΔH_{298} = -406.1 kJ/mol of oxygen reacted.

$S^o + 1.5O_2 + H_2O \rightarrow H_2SO_4$ ΔH_{298} = -603.2 kJ/mol of sulphur reacted.

The majority of heat from the above reactions is generated by the oxidation of sulphur to sulphuric acid.

The enthalpies of the reactions in a chalcopyrite heap are shown as:

$$CuFeS_2 + Fe_2(SO_4)_3 + 0.5O_2 + H_2SO_4 \rightarrow CuSO_4 + 3FeSO_4 + H_2O + 2S^o \quad (14)$$

ΔH_{298} = -221.5 kJ/mol of chalcopyrite reacted.

In a biotic leach of chalcopyrite ore some of the elemental sulphur and the ferrous ions will be oxidized generating significant heat. In an abiotic chalcopyrite ore heap leach, very little of the elemental sulphur would be oxidized to sulphate. Thus the heat generated by the oxidation reactions would be minimal.

Some studies have been conducted on the heat generation within a low grade chalcopyrite heap leach. These studies are based mainly on biotic leaching applications. Mintek conducted a pilot biological heap leaching test on 20,000 tonnes of < 10 mm low grade ore, containing 0.55% Cu with 54% occurring as chalcopyrite (van Staden, 2008). To help determine the experimental conditions Mintek's column testing was conducted using 6 m long columns. The highest temperature obtained was 45°C within the lower section of the heaps and columns, see Figure 2.13. 67% of the heap was maintained above 30°C, even during the winter when the solution temperature dropped to 10°C and the ambient temperature was between 0 °C and 10°C (van Staden, 2008).

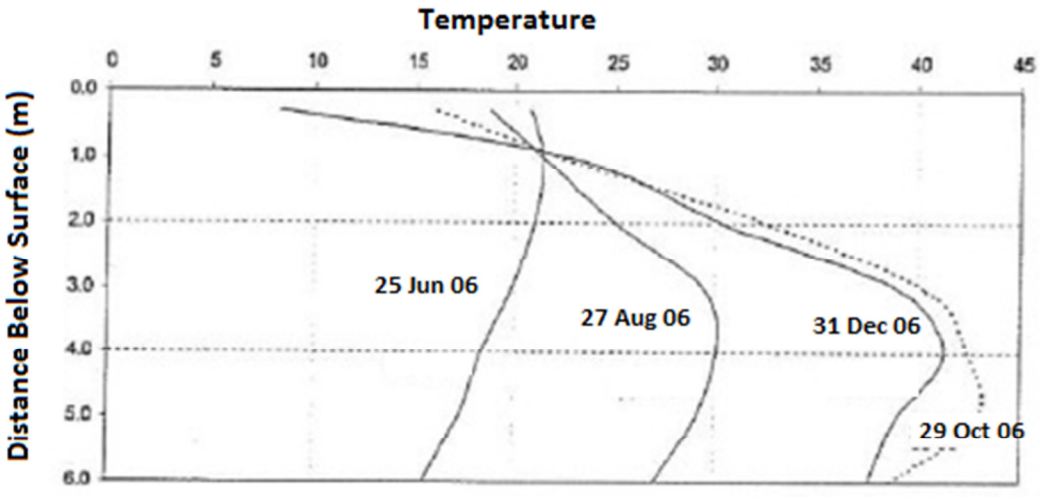

Figure 2.13: Vertical Temperature Profile from the Mintek Pilot Heap (van Staden, 2008)

Van Staden (2008) also shows the importance of initial heap temperature, Figure 2.14, for increasing the reaction rates. The initial high recovery period for the winter start, temperatures between 0 ^{o}C and 10^{o}C, compared to the summer start, temperatures > 25 ^{o}C, takes almost three times as long to extract the first 30 to 40% of the copper. This indicates that pre-heating of a heap may help increase the initial and overall extraction rates.

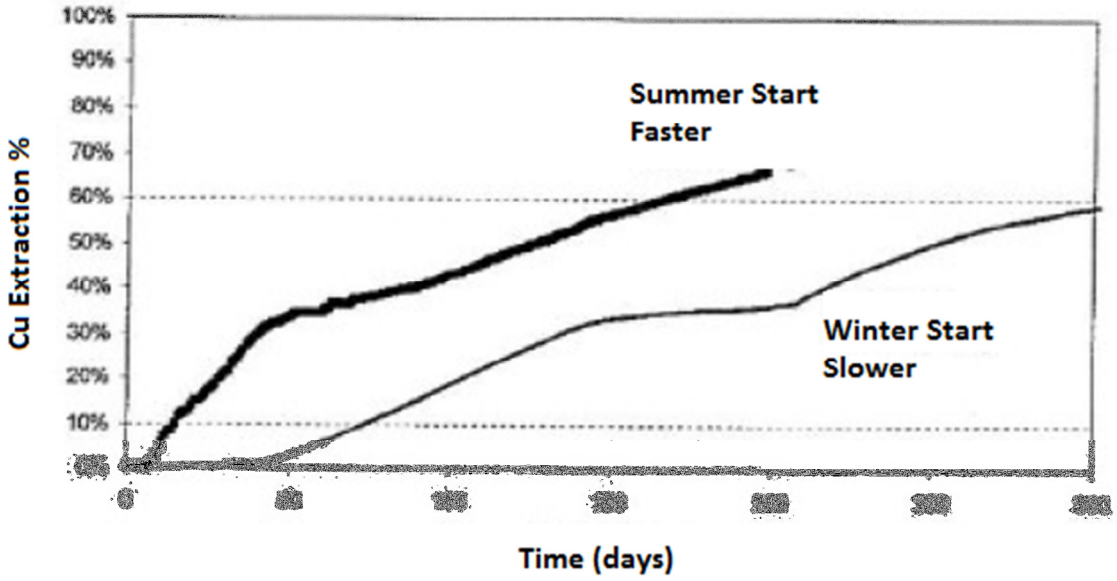

Figure 2.14 Mintek Pilot Heaps Initial Temperature Effects on Cu Extraction (van Staden, 2008).

2.4 Literature Review Summary

The majority of current literature has studied the leaching of chalcopyrite concentrates and there has been significantly less research conducted on chalcopyrite ores. Although there are some contradictions found within the literature, it seems that the effects of higher temperature, smaller particle size, acid concentration, pyrite additions and the presence of chloride ions in solution have a significant impact on the copper extraction from chalcopyrite, see Table 2.6. To date a commercially viable industrial process to heap leach chalcopyrite ore has not been developed.

Table 2.6: Literature Review Summary

Parameter	From Literature
Temperature	Maintaining a temperature above 50°C is important for a high Cu. van Staden shows that a higher initial heap temperature increases the rate of extraction significantly.
Particle size	A smaller size is important but heap size limited to about 10 mm due to percolation limits.
Acid concentration (pH)	Contradictory results from chalcopyrite concentrate tests.
Redox potential	With concentrates highest rate at low ORP equal to Fe3+:Fe2+ ratio of about 0.5. In ore column leach, no significant effect on Cu extraction except in terms of the external generation of the lixivant.
Chloride additions vs sulphate only	Higher Cu extraction with concentrates of >1 molar NaCl .
Pyrite additions	Galvanox shows high extraction for concentrates and chalcopyrite ore heap leaching works better with higher Py in ore

Chapter 3 Mineralogy and Methods

3.1 Ore Mineralogy

The mineralogy is the most important factor in determining the leachability of a chalcopyrite ore. The chalcopyrite ore and concentrate used for this thesis was provided by Asarco from their Ray Mine located in Arizona, USA.

A mineral assay was conducted by Xstrata Process Support (XPS), Figure 3.1, using QEM SCAN.

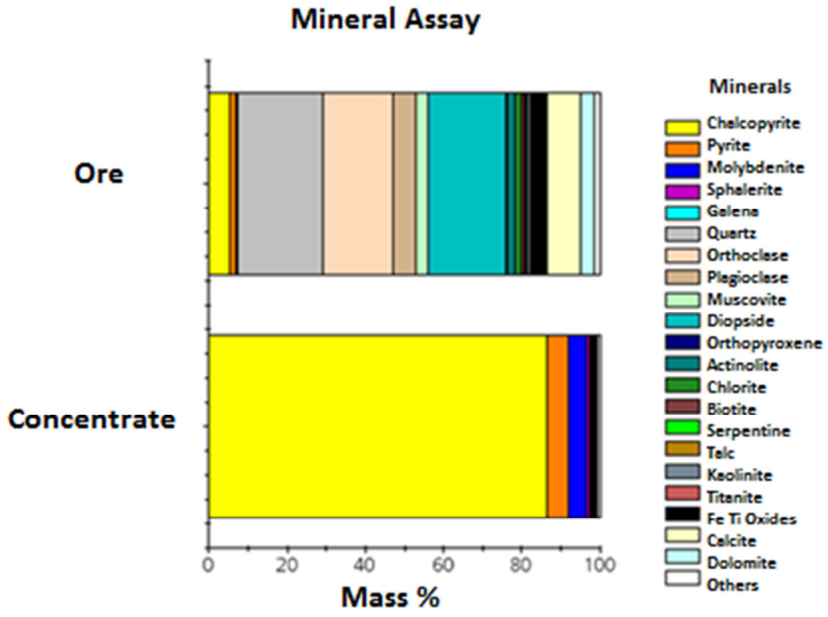

Figure 3.1: Mineral Assay of Chalcopyrite Ore and Concentrate Conducted by XPS.

The values from the QEM Scan, Figure 3.1, are summarized in Table 3.1.

Table 3.1: Chalcopyrite Concentrate and Ore mineralogical characterization by XPS using QEM SCAN

	Cu Concentrate	Feed
Chalcopyrite	77.92	5.32
Pyrite	9.10	1.55
Molybdenite	7.51	0.18
Sphalerite	1.23	0.31
Galena	0.12	0.00
Quartz	0.97	21.65
Orthoclase	0.75	17.93
Plagioclase	0.20	5.90
Muscovite	0.25	3.31
Diopside	0.39	19.71
Orthopyroxene	0.00	0.65
Actinolite	0.27	1.74
Chlorite	0.13	1.72
Biotite	0.02	0.71
Serpentine	0.00	0.42
Talc	0.03	0.40
Kaolinite	0.01	0.55
Titanite	0.02	0.27
Fe Ti Oxides	0.08	4.12
Calcite	0.05	8.72
Dolomite	0.15	3.49
Other	0.79	1.34

Note: Other category contains the following minerals:
epidote, apatite, zircon, alunite, barite, and an FeMnCa silicate.

Ore feed and copper concentrate samples used for this report were characterized mineralogically by Xstrata Process Support (XPS) with QEMSCAN. The feed measurement was completed on a size fraction in the range of 0.5 - 1 mm, in order to maintain in-situ textures, while the measurement of the copper concentrate was completed on an un-sized sample as produced by milling and flotation. The following conclusions were made based on the results of the analysis:

1. Copper mineralogy is dominated by chalcopyrite in both the ore and concentrate. No other copper sulphide minerals were measured in significant quantities.
2. Chalcopyrite in the feed shows a wide range of grain sizes with 34% of the size distribution being below 90µm and another 28% over 300µm.
3. The other sulphides are low compared to the chalcopyrite content; pyrite (1.55%), molybdenite (0.18%), and sphalerite (0.3%).
4. Gangue mineralogy in the feed is dominated by quartz, orthoclase, diopside with lesser amounts of plagioclase, muscovite, actinolite, chlorite, biotite, kaolinite, talc, as well as significant carbonates, calcite and dolomite.
5. The high level of carbonates means that this ore will have high acid requirements.

Representative samples of the Asarco ore were ground and subjected to aqua regia digestion. The amount of copper and iron was determined by atomic absorption analysis. The aqua regia dissolution was performed, in triplicate, on ore samples in the particle size range of 1.18 mm – 2.38 mm. The ore was found to contain an average of 2.1% Cu and 4.8% Fe.

3.2 Methods

3.2.1 Ore Sampling

360 kg sample of the chalcopyrite ore from the Ray Mine was coned and quartered. Two of the quarters were separated from the stock ore. These quarters were then split into 5 kg samples by means of a rotational splitter. These 5 kg samples were then screened into the following size groups: < 1.18 mm, 1.18 to 2.38 mm, 2.38 to 5 mm, and 5 to 10 mm. Ore over 10 mm was roll crushed and sorted accordingly. Each different size sample was then randomized via rotational batch distribution and mixed together.

3.2.2 Acid Consumption Analysis and Pre-Acidification Methodology

The high acid consuming gangue content suggested that the acid requirements for this ore would be very high. Therefore it was decided to pre-acidify the ore before commencing any test work, otherwise, in the initial leaching part of each test, the only reactions would involve acid neutralization rather than chalcopyrite leaching. In practice such a high acid consumption would make this ore unsuitable for heap leaching.

An initial estimate of the acid required to satisfy the acid consuming gangue in the ore was based on the XPS mineralogical analysis. Based on the calcite and dolomite content, it was estimated that the sulphuric acid requirements to react the acid consuming carbonates would be approximately 1.2 g per 10 g ore sample.

A pre-acidification experiment was conducted to determine the amount of acid required to eliminate the acid consuming minerals. Representative samples of ore, in the particle size range of 1.18 to 2.38 mm, were pre-leached with a 31 g/L sulphuric acid solution until a stable pH around 1 was attained. It was found at 60°C that 1.55 g of free acid per 10 g of ore was required to react the majority of the acid consuming gangue. According to Scheffel (2002), an economic acid consumption is generally less than 50 kg of acid/t of ore.

3.2.4 Shake Flask Experiments

The preliminary leach tests were performed in an orbital shaker at 200 rpm using 250 mL Erlenmeyer flasks covered with sponge stoppers to allow the circulation of air while reducing water loss through evaporation, see Figure 3.2. The desired amount of concentrate or pre-acidified ore was placed in the flasks. 100 milliliters of dilute sulphuric acid was added to each sample. The acid solution was prepared differently depending on the specific experimental conditions. The flasks were then shaken at 200 RPM and at either room temperature or at 60°C. Evaporation losses were replaced with de-ionized water. From this solution samples of 15 to 20 ml were taken at specific intervals and filtered. The pH and ORP of each sample was measured in-situ and then the acid was added to reach the target pH level. The total copper and iron in solution was measured using atomic adsorption analysis. The ferrous iron concentration was determined using potassium permanganate titration and verified using spectrophotometrics.

Figure 3.2: Orbital Environmentally Controlled Shaker Table.

3.2.5 Mini-Column Leach Experiments

A second set of leach tests was conducted using miniature leaching columns. These mini-column leach tests were performed on the same sized ore sample to determine how the ore reacts under simulated heap leach conditions of unsaturated flow. The experimental setup was designed to maintain the ore at a temperature of 50°C, using a hot PVC water jacket placed around the columns. The pre-acidified ore was placed in the 2.54 cm diameter heated glass columns. A volume of 200 ml for each of the four different sulphuric acid based lixivants was put into separate 250 ml Erlenmeyer catch flasks. A peristaltic pump was used to precisely control the irrigation drip rate at a constant flow rate of 10 $L/m^2/h$. The solution was re-circulated and the flow was only stopped to take samples and to adjust the pH. Each of the lixivants was run in two separate columns. The pregnant leach solution was then collected and the pH and ORP of each sample was measured. The total copper and iron in solution was measured using atomic absorption analysis. A schematic and pictures of the mini-column apparatus are shown in section 3.2.5.1.

3.2.5.1 Column Design and Setup

A column leaching apparatus was designed and built for this experimental work based on the results of previous work and the initial flask test results.

The base plate was machined out of acid resistant PVC with a tapered acid drainage funnel, a $\frac{1}{4}$" acid drainage port, a screen holder, and finally locating slots for the PVC water jacket and the glass ore leaching column. See Figure 3.3.

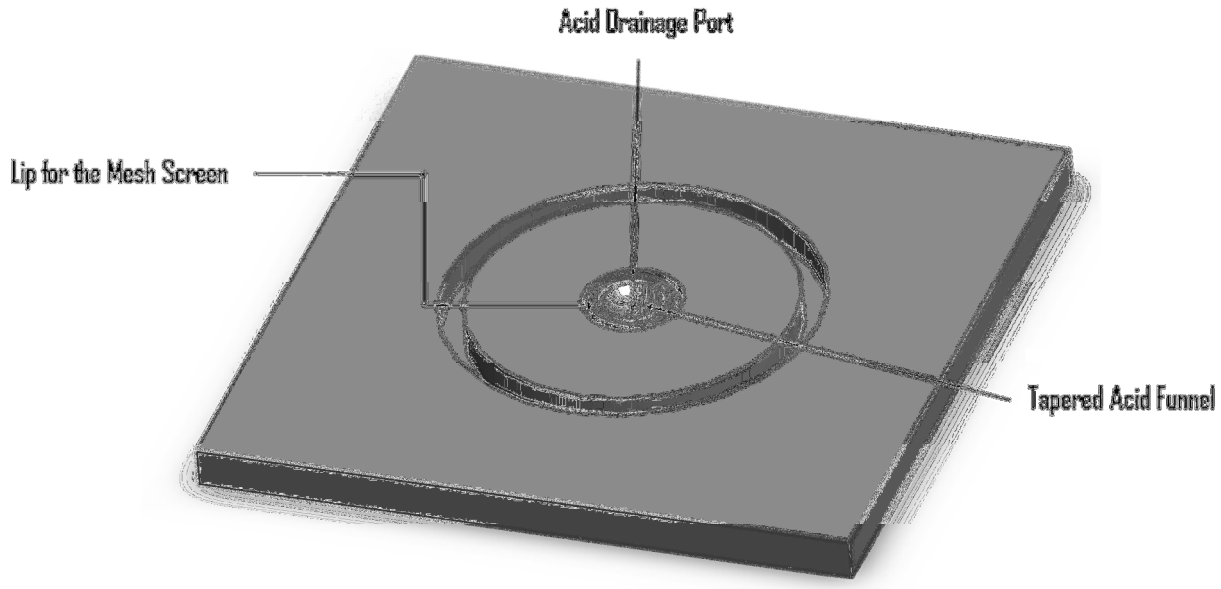

Figure 3. 3: Base Plate Design

The complete column assemblies are depicted in Figures 3.4 and 3.5.

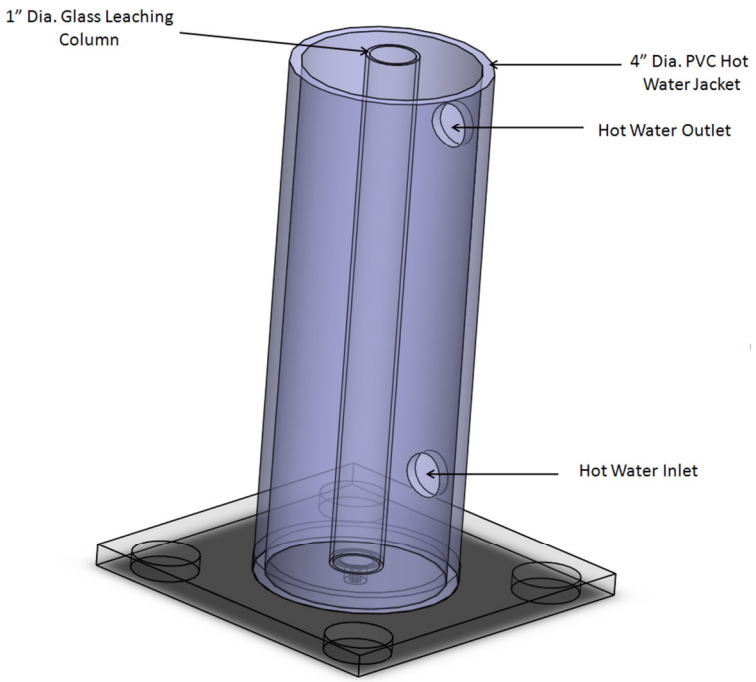

Figure 3.4: Transparent Assembly View of the Miniature Leaching Columns

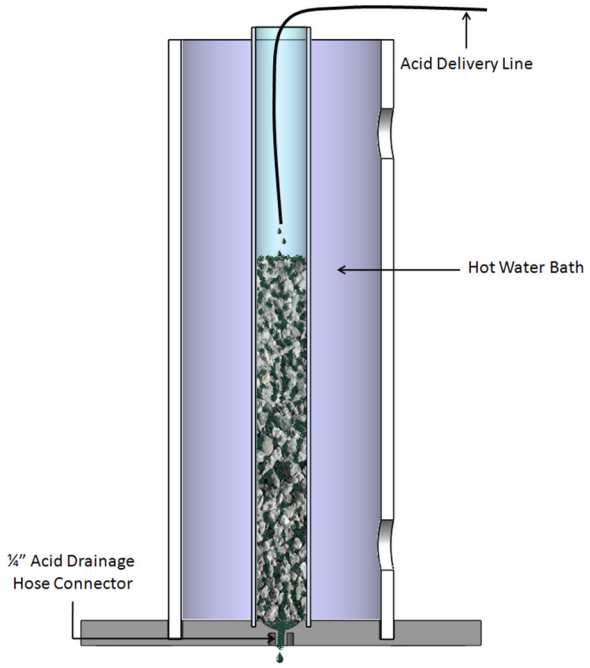

Figure 3.5: Miniature Leaching Column Apparatus Cross-Sectional View

The fully assembled leaching apparatus is shown in Figures 3.6 & 3.7.

Figure 3.6: Miniature Leaching Column Apparatus Setup

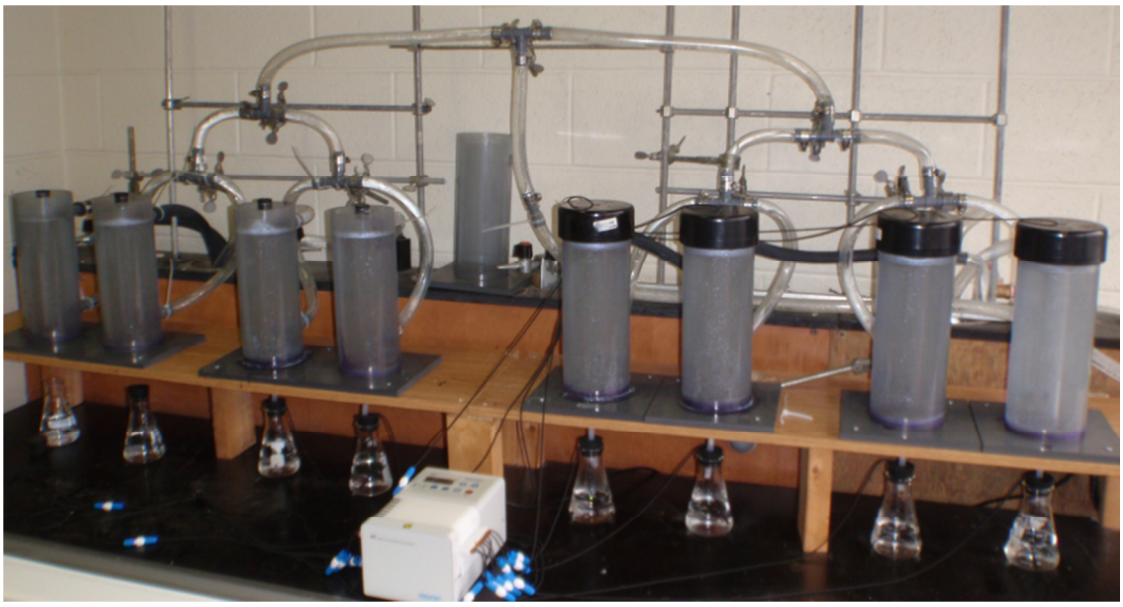

Figure 3.7: Miniature Leaching Column Apparatus Full Setup

Chapter 4 Results and Discussion

4.1 Concentrate Shake Flask Experiments

The initial shake flask experiments were conducted using chalcopyrite concentrate obtained from Asarco's Ray Mine in Arizona. The key parameters observed were the particle size, pH, temperature, pyrite and chloride addition, see Table 4.1. The following conditions were kept the same for each sample: 100ml of sulphuric acid and 2 g of chalcopyrite concentrate. For the samples with pyrite, the pyrite was sized at -200 mesh and was added at a one to one ratio with the chalcopyrite. The A# & B# tests were conducted at 25 degrees Celsius and the AA# & BB# tests were conducted at 60 degrees Celsius.

Table 4.1: Test Conditions for Initial Concentrate Shake Flask Experiments.

Sample	Cp Size (Mesh)	pH	Py:Cp
A1/AA1	200	0.5	1:1
A1/AA2	200	0.5	1:1
A1/AA3	200	0.5	0
A1/AA4	200	0.5	0
A1/AA5	200	1	1:1
A1/AA6	200	1	1:1
A1/AA7	200	1	0
A1/AA8	200	1	0
B1/BB1	400	0.5	1:1
B1/BB2	400	0.5	1:1
B1/BB3	400	0.5	0
B1/BB4	400	0.5	0
B1/BB5	400	1	1:1
B1/BB6	400	1	1:1
B1/BB7	400	1	0
B1/BB8	400	1	0

4.1.1 Effect of Particle Size and Acidity on Leaching Chalcopyrite Concentrate

The effect of particle size on copper recoveries from a chalcopyrite concentrate was examined. These tests showed that a smaller particle size and a stronger acid concentration increased the copper recovery of chalcopyrite, see Figure 4.1. The decreased particle size and stronger acid concentration have both been shown within the literature to be potentially beneficial to the leaching of chalcopyrite.

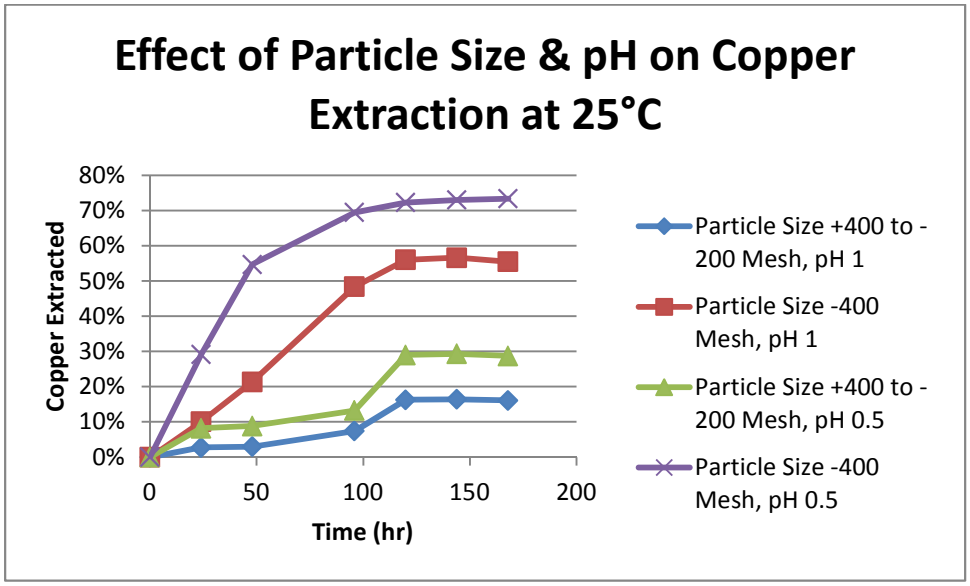

Figure 4.1: The Effect of Particle Size and pH on Cu Extraction at 25°C.

Detailed results of the leaching tests conducted with chalcopyrite concentrate are given in the appendix. The factorial design indicated that the most significant parameters were particle size, pH and temperature. The effect of sodium chloride and pyrite additions were not significant. It was believed that this was because the levels used were not high enough to produce the desired reactions.

No further concentrate leaching tests were conducted once the Asarco ore sample was received. All subsequent leaching tests were performed using only the Asarco chalcopyrite ore.

4.2 Ore Shake Flask Experiments

4.2.1 Effect of Temperature and Particle Size

The smallest ore size used in a commercial heap has a d_{80} size of about 10 mm. At this size, all of the copper minerals are not necessarily liberated such that they are all exposed to the lixiviant. Even with long reaction times of over a year, the copper minerals may still not be totally leached. Therefore, to increase the ore mineral liberation for leaching and to reduce the time required for testing, a smaller size fraction was desirable. An initial experiment was conducted, with different sized ore fractions ranging from 1 to 10 mm, to determine an appropriate ore size to be used in the shake flask and column tests. The following test conditions were selected. Each 250 ml flask contained 20 grams of ore and 100 ml of 9.8 g/L sulphuric acid. The pH was adjusted to pH = 1 after each sample was taken, for the duration of the tests. The tests were conducted for a total of 120 hours with samples taken every 24 to 48 hours.

As expected, the smaller particle size fraction of 1.18 mm to 2.38 mm had a faster Cu extraction rate and higher overall Cu extraction than the 9.85 mm to 10.3 mm size fraction, see Figure 4.2. Therefore a particle size fraction of 1.18 mm to 2.38 mm was chosen for the remaining shake flask tests.

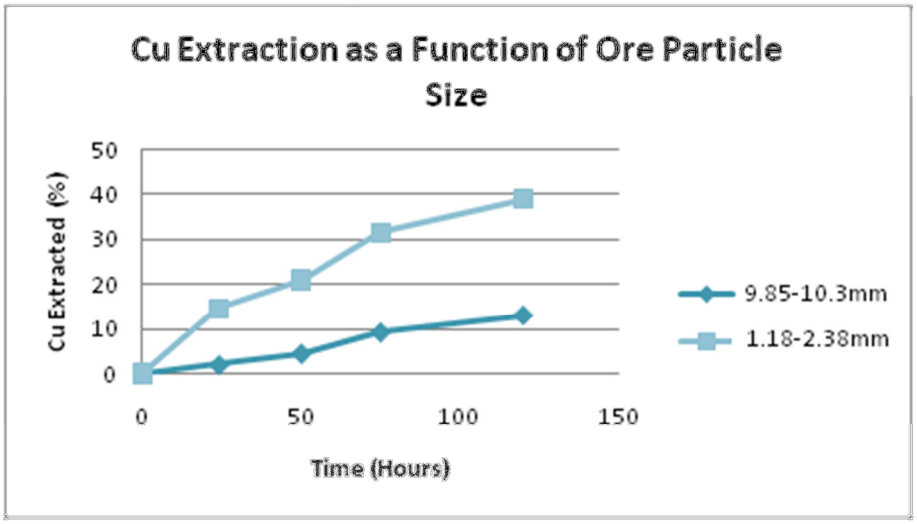

Figure 4.2: Rate of Copper Extraction as a Function of Ore Particle Size.

4.2.2 Effect of Temperature

It is well known that reactions usually proceed at a higher rate at a higher temperature. An initial experiment was conducted with 1.18 to 2.38 mm sized ore. One set of tests was done at an ambient temperature of 25°C and another was done at an elevated temperature of 60°C. The following test conditions were selected. Each 250 ml flask contained 20 grams of ore and 100 ml of 9.8 g/L sulphuric acid solution with 1 M of NaCl. The pH was adjusted to pH = 1 after each sample was taken for the duration of the tests. The tests were conducted for a total of 12 days with samples taken every 24 to 48 hours.

The test results are shown in Figure 4.3. The rate of copper extraction was significantly increased at the elevated temperature of 60°C. The increased rate of Cu extraction towards the end of the test at 60°C may be due to the higher Fe levels in solution.

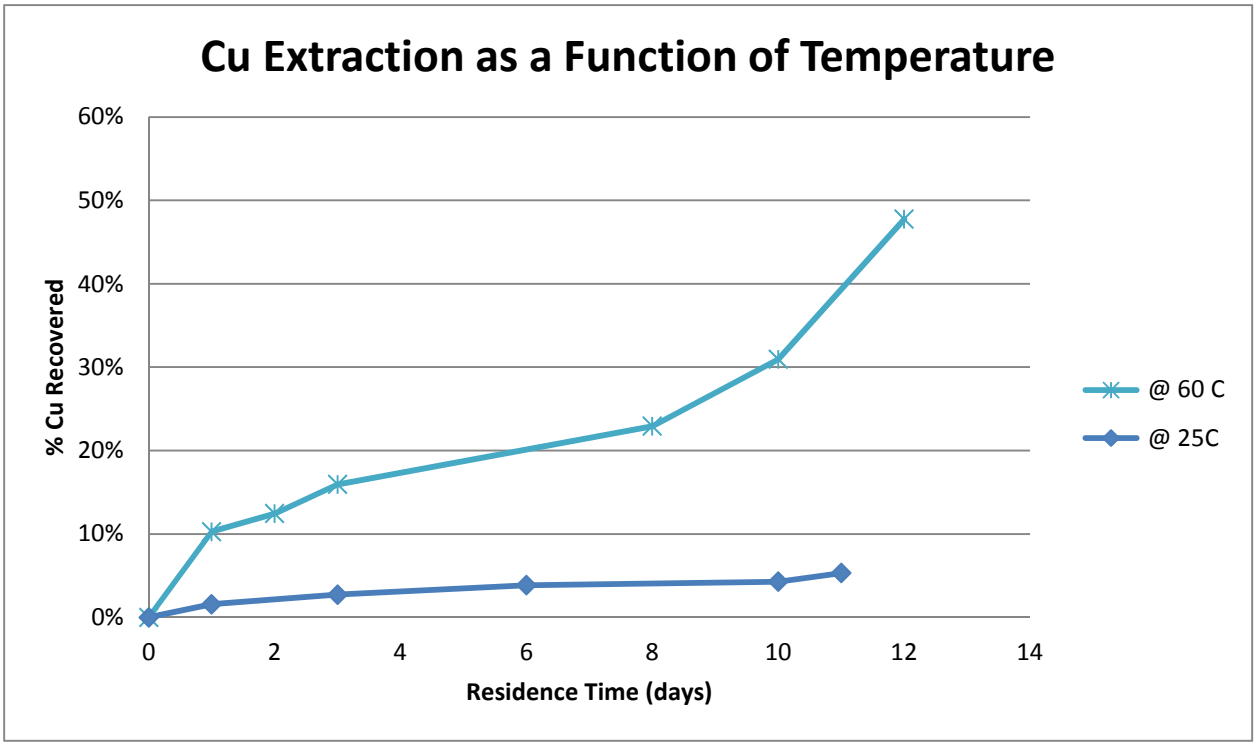

Figure 4.3: Rate of Copper Extraction from a Chalcopyrite Ore Leach as a Function of Temperature.

4.2.3 Ore Shake Flask Conditions

The chalcopyrite ore sample was pre-acidified to minimize the pH adjustments required during the tests. Less than 7,5% of the Cu was dissolved during the pre-acidification step. The first set of ore leach tests were performed in 250 ml shake flasks containing 10 grams of ore with a size fraction of 1.18 to 2.38 mm and 9.8 g/L sulphuric acid plus other additions to the lixivant.

Based on a recent paper by Vilcaez, 2009, discussed in the literature review, the effect of various ferrous and ferric ion concentrations were studied to try to evaluate the effect of redox potential. The "Galvanox" concept was explored by adding 4 grams of – 200 mesh pyrite concentrate to the pre-acidified ore. The effect of chloride addition was evaluated by adding 1M of NaCl to the lixivant. These tests were conducted in a shaker table incubator using 200 RPM and a temperature of 60°C. The pH was adjusted to pH = 1 after each sample was taken, for the duration of the tests.

Below is the description for the Figure legends in this section:

Ferrous: The initial solution contained 5 gpL of ferrous ions in a 9.8 g/L sulphuric acid solution.

Ferric: The initial solution contained 5 gpL of ferric ions in a 9.8 g/L sulphuric acid solution.

H_2SO_4: **9.**8 g/L sulphuric acid only.

H_2SO_4 + pyrite: The samples with pyrite contained 4 grams of -200 mesh pyrite concentrate was added after pre-acidification.

Sodium chloride: 1 molar NaCl with 9.8 g/L sulphuric acid.

These experiments were run in duplicate and the resulting Cu extractions were similar for each duplicate experiment with less than a 10% variation in the measured values. The data from individual experiments is given in the Appendix.

4.2.4 Copper Extraction in Ore Shake Flask Test

The ore shake flask tests were conducted using pre-acidified ore provided by Asarco. The tests were carried out in duplicate and the average copper extraction rates for the tests are plotted in Figure 4.4.

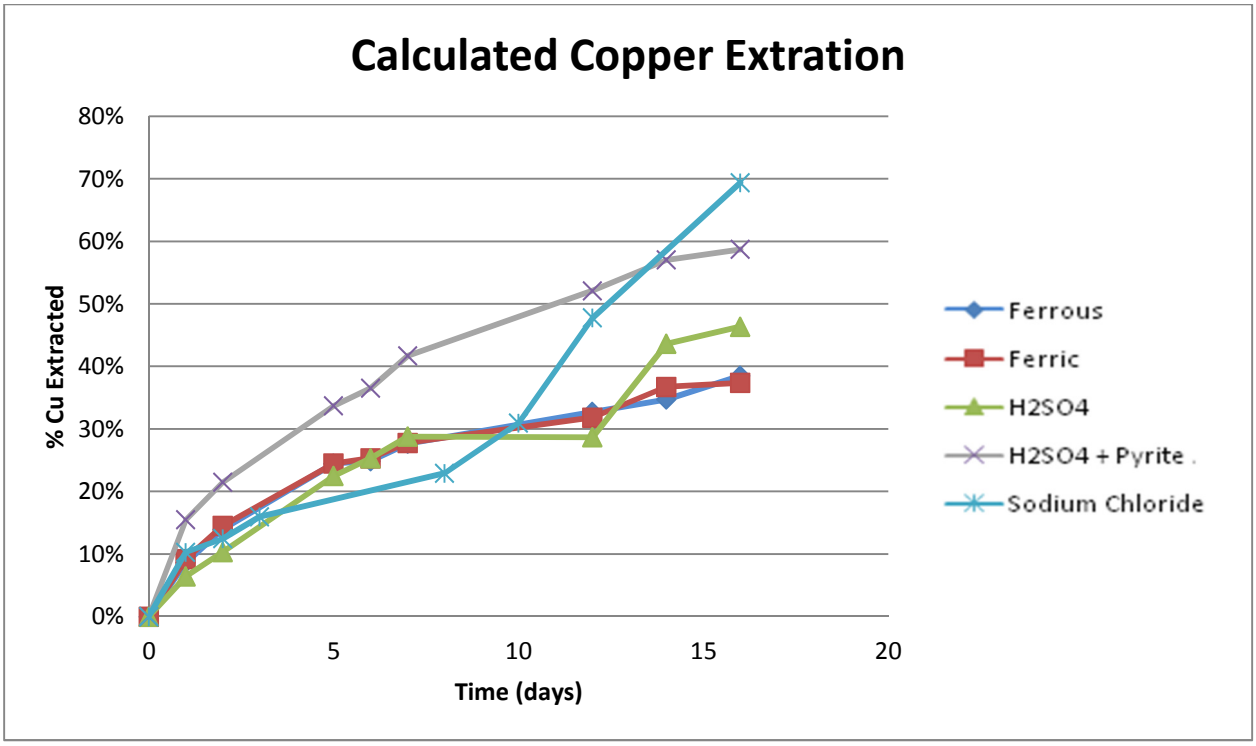

Figure 4.4: Measured Copper Extracted for Ore Shake Flask Tests.

It can be seen from Figure 4.4 that initially the sulphuric acid with pyrite sample had the highest initial copper extraction rate and reached approximately 60% after 16 days. The sodium chloride addition sample initially leached more slowly but reached the highest Cu extraction of about 70% after 16 days. The rate of copper extraction in the sodium chloride sample increased significantly after day 8. No explanation for this behaviour could be developed.

The ore shake flask tests with the additions of ferrous and ferric ions both demonstrated similar extraction rates. Neither the additional ferrous or ferric ions were found to improve the overall copper extraction. The sample with only sulphuric acid had a similar leach curve to the ferrous and ferric samples.

In all of the ore leach tests some copper extraction was still occurring at the end of the test period. The copper extraction rates, for the tests with iron addition in particular, slowed significantly. The copper extraction rate, using chloride addition was the highest at the end of the test period suggesting that it could continue and extract most of the copper within another 10 days of leaching. Due to a temporary lab closure the tests had to be stopped after 16 days.

4.2.5 Iron Extraction in Ore Shake Flask Test

The iron extraction for each of the tests is plotted against time in Figure 4.5. This showed that the lowest amount of iron was extracted in the ferric addition and sodium chloride addition tests. In these tests the iron was probably precipitated, which would reduce the iron level in solution and the apparent extracted during leaching.

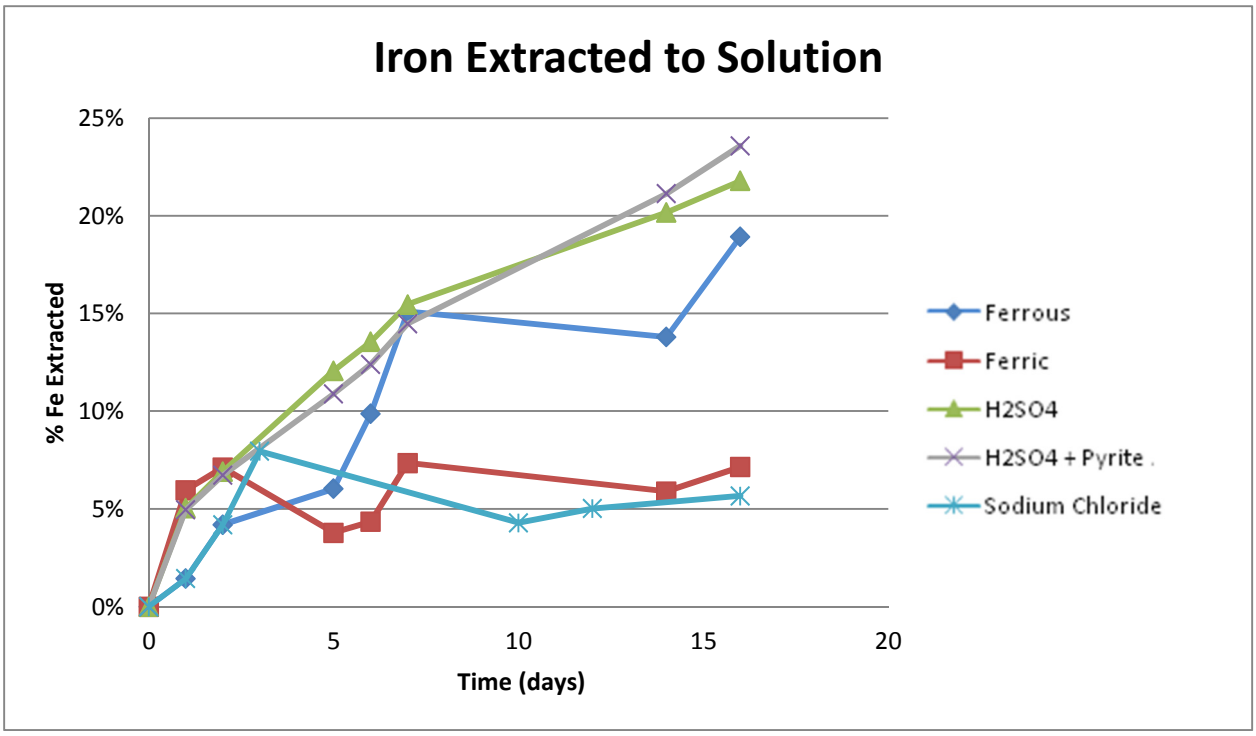

Figure 4.5: Calculated Iron Extracted from Ore Shake Flask Tests.

4.2.6 Acidity of Ore Shake Flask Tests

The pH measurements shown in Figure 4.6 were taken at the end of each test interval and before the sample pH was adjusted back to pH = 1.

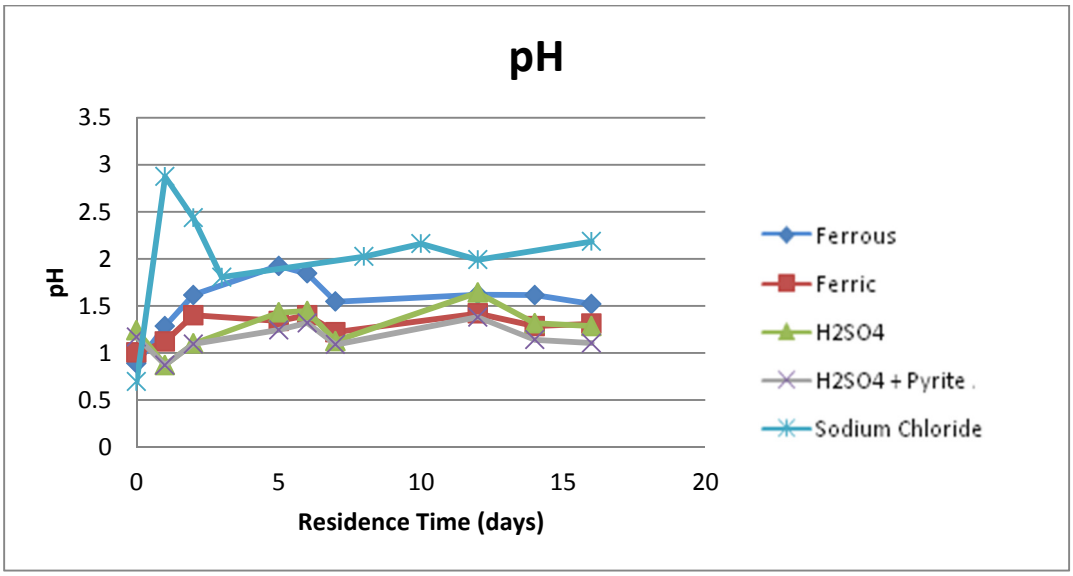

Figure 4.6: Average pH Levels Measured Throughout the Ore Shake Flask Experiment.

Figure 4.6 shows that the most acid was consumed in the tests with chloride and ferrous addition. The apparent acid consumption was lower in the pyrite, ferric and sulphuric acid only tests. The high acid consumption of the ferrous solutions is due to the ferrous reacting with the acid to produce ferric, as shown by the following reaction:

$$2Fe^{2+} + \frac{1}{2}O_2 + 2H^+ = 2Fe^{3+} + H_2O \quad (10)$$

The sodium chloride solution resulted in the highest copper extractions, thus using the most acid to produce the high amount of ferric ions required.

There is also the possibility of higher gangue acid consumption with the different lixivant solutions.

4.2.7 Effect of Redox Potential

The resulting oxygen redox potential (ORP) was measured during the ore shake flask tests using Ag/AgCl with a platinum based electrode and the ORP measurements are plotted against time in Figure 4.7. After the initial leaching period, around 2 to 3 days, the average ORP of all the tests was in the range of 420 to 480 mV. The measured ferric to ferrous ratio in solution for all tests is shown in Figure 4.8.

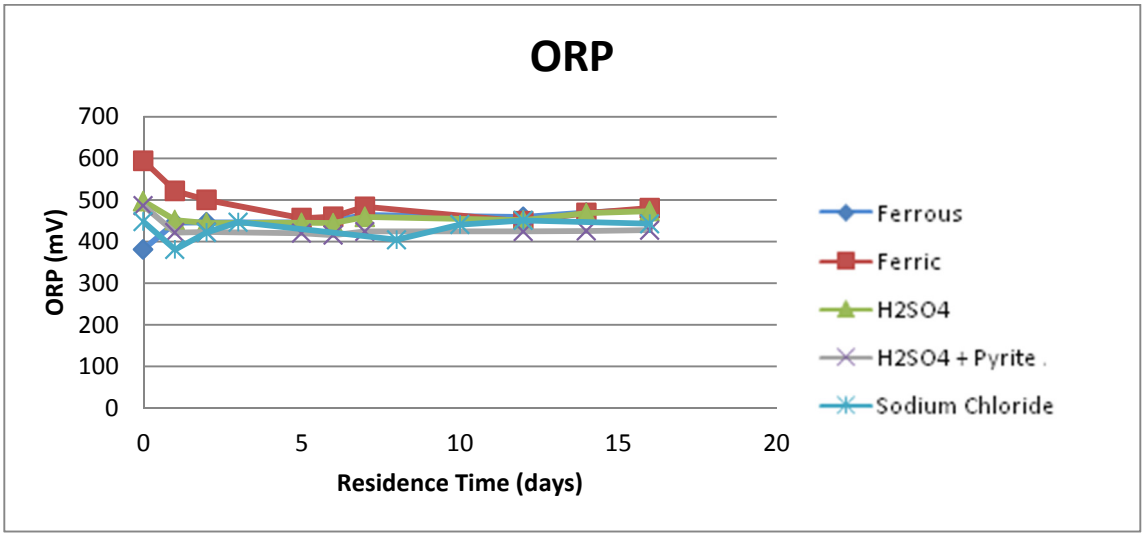

Figure 4.7: The Oxygen Redox Potential Measured Throughout the Ore Shake Flask Experiment

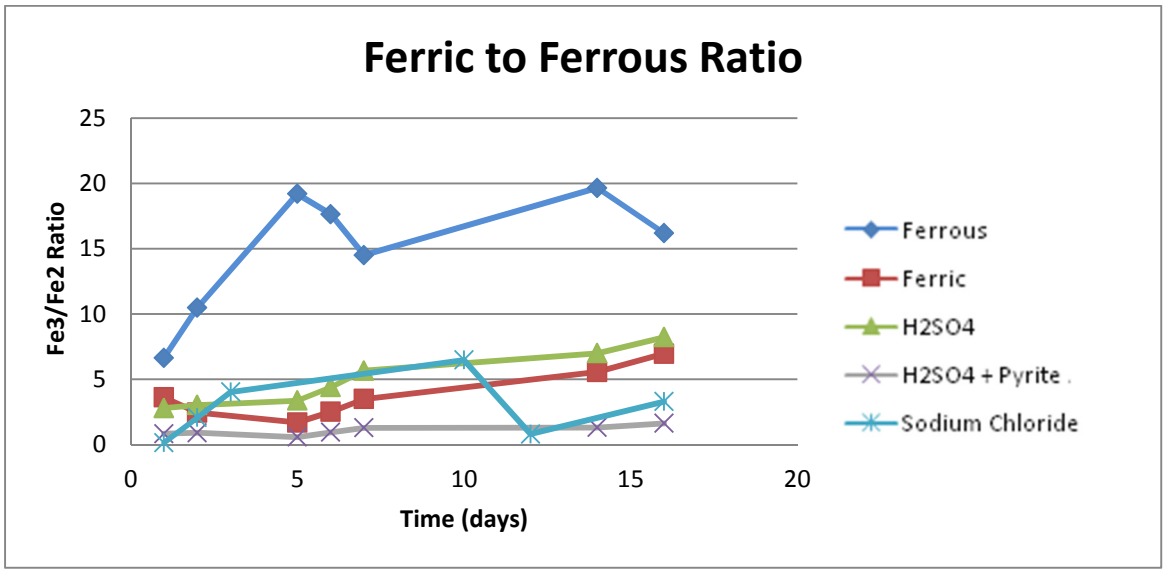

Figure 4.8: Ferric to Ferrous Ratio for the Ore Shake Flask Tests.

Figure 4.8 shows that most of the tests operated with a ferric to ferrous ratio in the 1 to 5 range. Surprisingly the test that started with only ferrous ions had the highest ferric to ferrous ratio. The lowest ferric to ferrous ratio was found with sodium chloride addition that also gave the highest copper extraction.

4. 2.8 Ore Shake Flask Leach Residue Analysis

X-ray diffraction analysis (XRD) of the leach residue was conducted in an attempt to determine if any iron precipitates were formed during leaching. Figure 4.9 shows an XRD analysis of the leach residue from the sulphuric acid only test. In this example and all of the other tests the majority of the remaining leach residue is quartz and gypsum. Due to the large amount of gangue material in the ore no iron precipitates were found in any of the leached ore samples. The XRD images for each test can be found in the appendix.

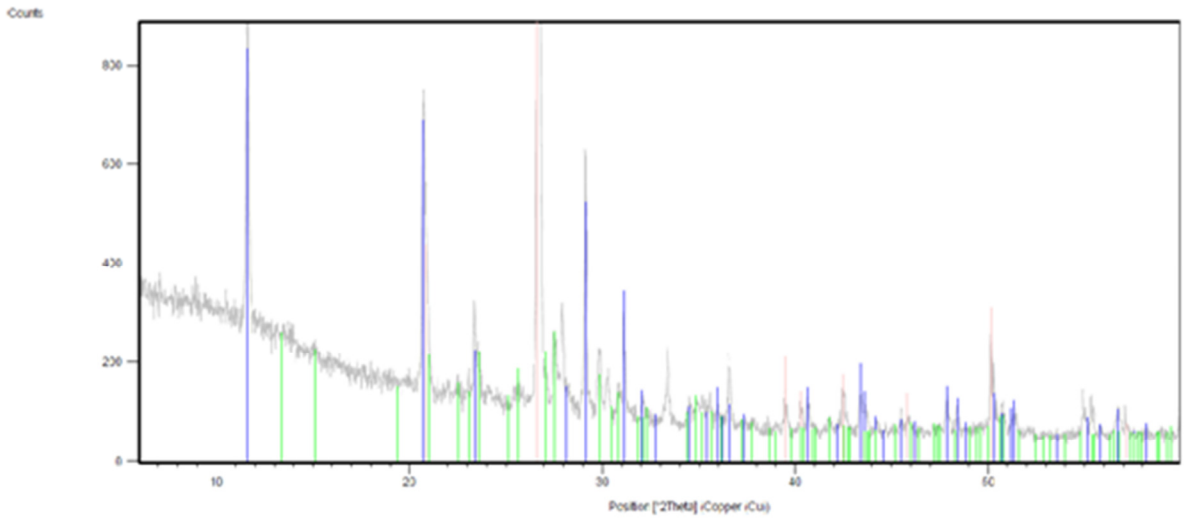

Figure 4.9: XRD Analysis of an Ore Shake Flask Leached Residue, where Blue is Gypsum and Pink is Quartz.

4. 2.9 Ore Shake Flask Test Summary

The ore shake flask tests were conducted to try to identify the preffered lixivants to use in the mini-column tests. The shake flask test results showed, with this specific ore, that the acidified NaCl solution extracted the most copper with nearly 70% Cu extracted in 16 days at a temperature of 60°C, with a particle size in the range of 1.18 mm to 2.38 mm. This was followed by the ore with pyrite addition with nearly 60% Cu extracted in 16 days, at a temperature of 60°C, with a particle size in the range of 1.18 mm to 2.38 mm. The other tests extracted lower amounts of copper, in the range of 30 to 40% copper extracted. Therefore, based on these results, the sodium chloride and pyrite solutions were chosen for further testing. It was also decided to test the effect of ferric to ferrous ion ratio in a solution containing 5 g/L of iron, one with a low (1 to 1) ferric to ferrous ratio and the other with a higher (5 to 1) ferric to ferrous ratio, were used in the mini-column tests.

4.3 Ore Mini-Column Tests

4.3.1 Ore Mini-Column Leaching Tests

The samples of chalcopyrite ore used in the leaching tests were pre-acidified to minimize the pH adjustments required during the tests. The leach tests were performed in 2.54 cm diameter glass columns, 25.4 cm in length containing 100 grams of ore with a size fraction of 1.18 to 2.38 mm and enclosed in a hot water bath that maintained the ore at a temperature of 50°C. The pH was adjusted to pH = 1 after each 20 ml sample was taken, for the duration of the tests. The operating parameters for the column tests were based on the ore shake flask tests and literature to try to maximize the copper extraction. Based on these results four different experimental arrangements were tested in the mini-column experiments using three different lixiviant solutions. The following are the test conditions that were chosen:

1. 5 gpl of iron, with a ferric to ferrous ratio of 5 to 1, in a 9.8 g/L sulphuric acid lixiviant.
2. 5 gpl of iron, with a ferric to ferrous ratio of 1 to 1, in a 9.8 g/L sulphuric acid lixiviant.
3. 5 gpl of iron, with a ferric to ferrous ratio of 5 to 1 and 1 M NaCl, in a 9.8 g/L sulphuric acid lixiviant.
4. 5 gpl of iron, with a ferric to ferrous ratio of 5 to 1, in a 9.8 g/L sulphuric acid lixiviant, and with 40 g of the pure pyrite mineral added to the pre-acidified ore.

The lixivants were re-circulated to the columns at an irrigation rate equivalent to 10 $L/m^2/h$ using a multi-channel peristaltic pump and acid resistant plastic drip tubing. The pregnant leach solution from each column was collected in a flask and analysed periodically for copper and iron content using atomic absorption analysis.

These experiments were run in duplicate, except for the simulated 1:1 ferric to ferrous test, and the results were relatively analogous for each experiment with less than a 25% variation in the measured values. The following results are based on the average of the duplicate experiments. The data regarding the individual experiments can be found in the appendices.

4.3.2 Copper Extraction in Ore Mini-Column Tests

The average copper extraction rates for the ore mini-column tests are found in Figure 4.10. The rate of copper extraction was very slow for all test cases.

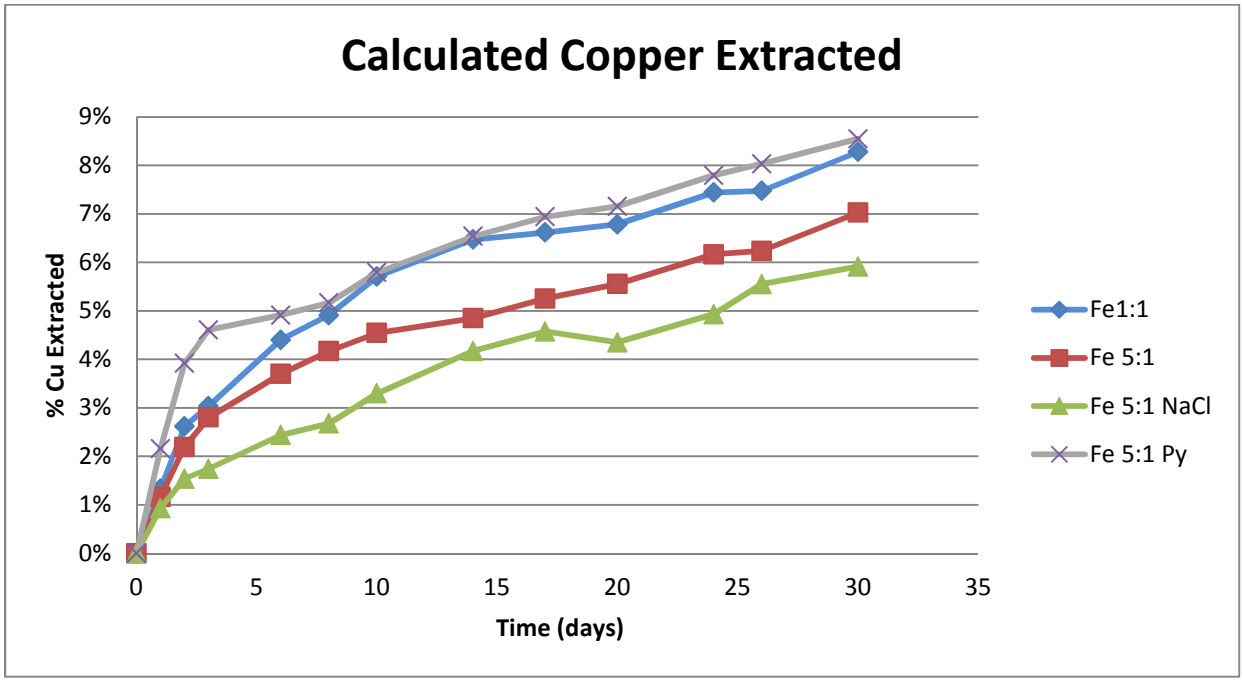

Figure 4.10: Calculated Copper Extracted for Ore Mini-Column Tests.

It is difficult to make any conclusions based on such a short test period and the low copper extractions but it can be seen that, over the test period, the test with pyrite and the lowest ferric to ferrous ratio had the highest copper extractions. The lowest extraction was with the sodium chloride test.

All of the tests has similar copper extraction slopes of approximately 1.5% Cu / 10 days. Extrapolating this to achieve 70% Cu extraction would require about 500 days of leaching. In view of this, and due to the considerable effort required to operate the heating and solution irrigation system and the sampling/analysis, it was decided that the tests be stopped after 30 days and to focus on the preliminary techno-economic evaluation.

4.3.3 Total Iron in Solution for the Ore Mini-Column Tests

The total iron extracted from solution was found using atomic absorption analysis and is shown in Figure 4.11.

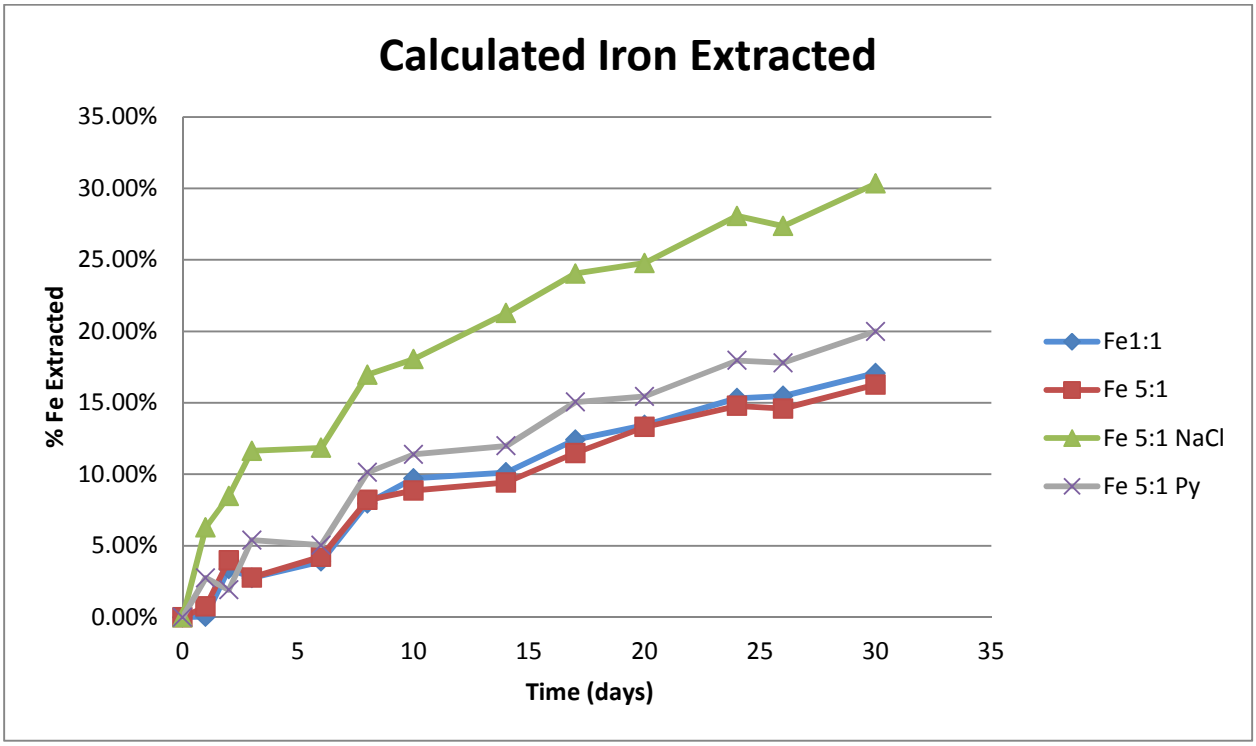

Figure 4.11: Calculated Iron Extracted for the Ore Mini-Column Tests.

The iron extraction levels were much higher than the copper extractions, and may be due to soluble gangue iron minerals in the ore.

4.2.4 Acidity of Ore Mini-Column Tests

The pH measurements shown in Figure 4.12 were taken at the end of each test interval before the sample pH was adjusted back to pH = 1.

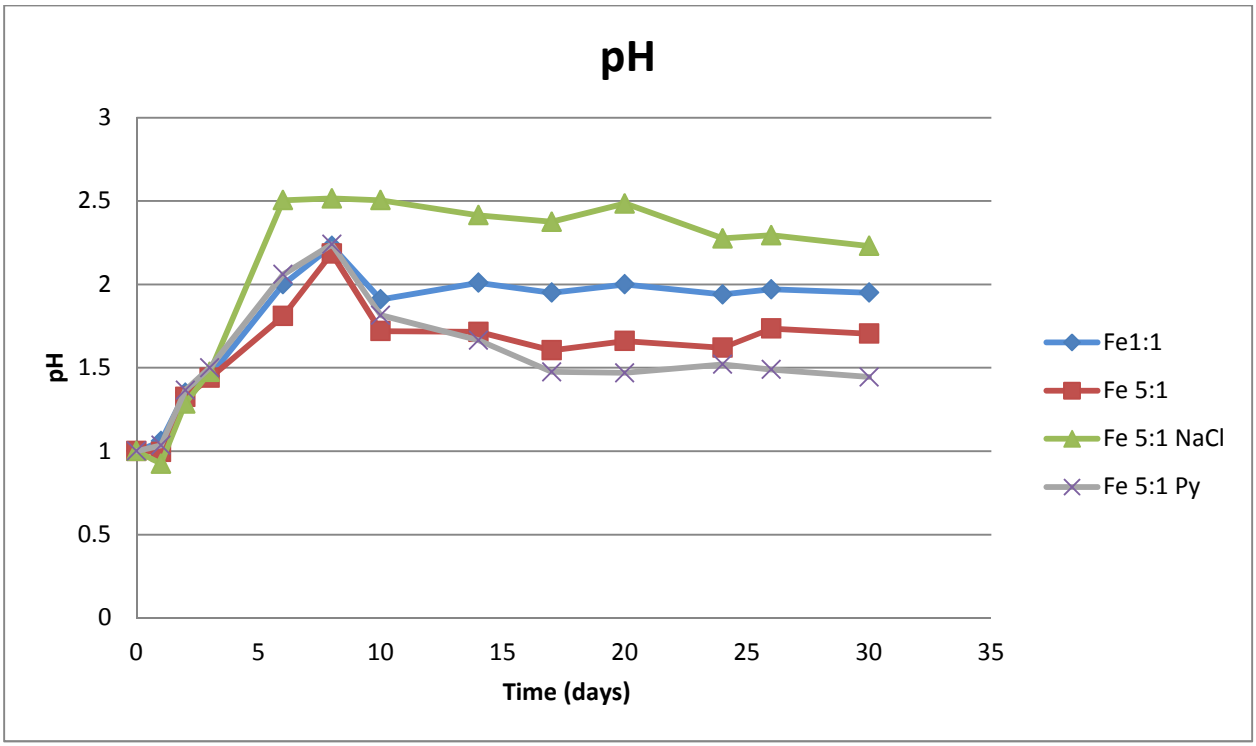

Figure 4.12: Average pH Levels Measured Throughout the Ore Mini-Column Tests.

Figure 4.12 shows similar results to those found in the ore shake flask tests. The highest acid consumer was the sodium chloride test, as found in the shake flask tests. All of these tests seem to be relatively high acid consumers, which indicates that the gangue in the ore was still consuming acid.

4.3.5 Redox Potential of Ore Mini-Column Tests

The resulting oxygen redox potential (ORP) trend of the ore mini-column experiments all stabilize around 425 mV Ag/AgCl with the exception of the chloride test which stabilized around 350 mV Ag/AgCl, Figure 4.13. Further experimentation on the electro-chemistry of ore leaching is required to determine if there is an optimal range for ORP or to substantiate whether or not a critical potential exists for chalcopyrite ores.

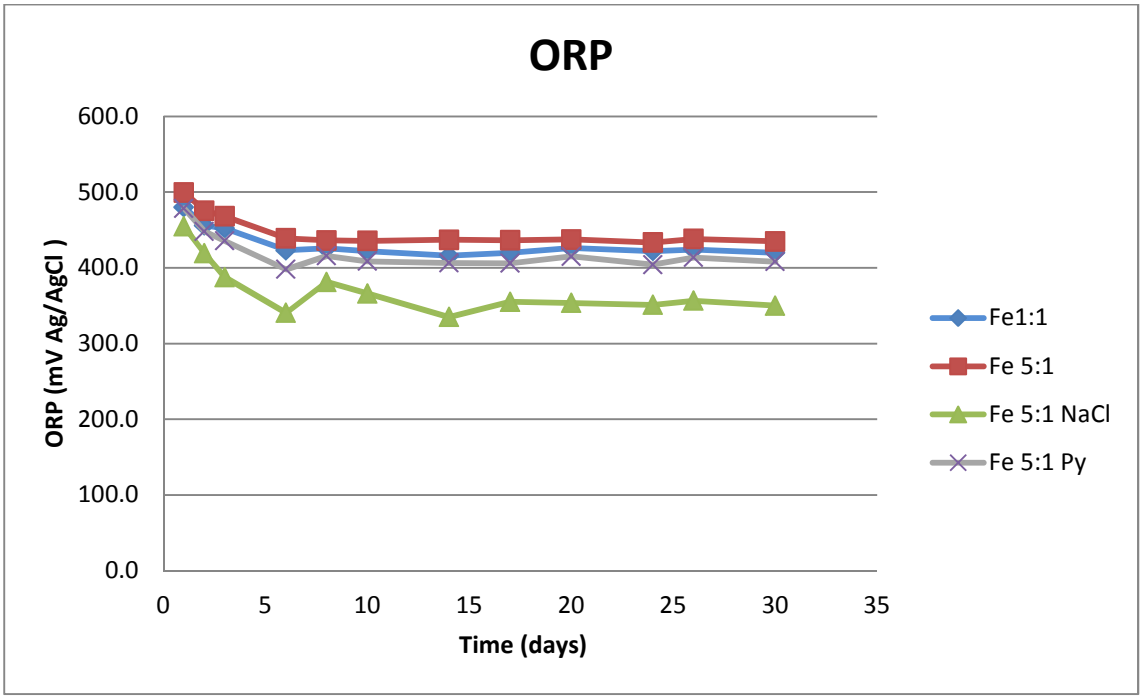

Figure 4.13: The Oxygen Redox Potential Measured Throughout the Ore Mini-Column Experiment.

4.3.6 Ore Mini-Column Leach Residue Analysis

Figure 4.14 shows an x-ray diffraction (XRD) analysis of the leach residue from the 5:1 ferric to ferrous, 5 gpl Fe, test. In this example, and all of the other tests, the majority of the remaining leach residue is quartz, gypsum and the majority of chalcopyrite has not been leached. The XRD images for each test can be found in the appendix.

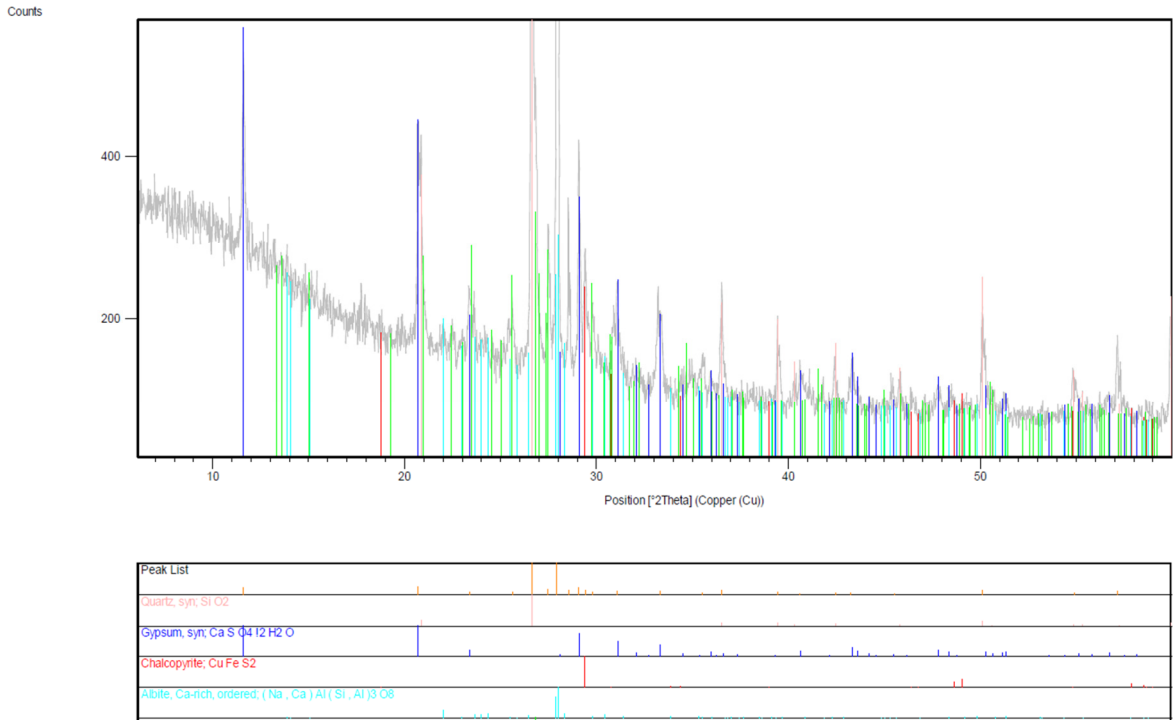

Figure 4.14: XRD Analysis of an Ore Shake Flask Leached Residue.

4.4 Chalcopyrite Ore Leaching Tests Summary

In the ore shake flask tests the highest copper extraction was achieved using 1 M NaCl addition to 9.8 g/L of sulphuric acid. The next highest was achieved with the addition of fine pyrite concentrate to the ore with a ratio of 4 to 1 pyrite to chalcopyrite.

In the mini-column tests, the rates of copper extraction were similar for all test conditions. Even using a small 1.18 to 2.38 mm size fraction and a temperature of 50°C, the rate of Cu extraction from the ore was very slow, only about 0.15% Cu/day. This slow rate may have been due to the fact that the ore was still consuming significant amounts of acid. In view of the slow rate of Cu extraction and the significant effort required in maintaining and analyzing the solutions, it was decided to curtail the mini-column tests after 30 days.

Chapter 5 Preliminary Evaluation of a Chalcopyrite Ore Heap Leach

The following chapter uses the experience gained from leaching chalcocite ores in laboratory columns and in commercial heap leaching operations, see Section 2.3.1, and from the limited published chalcopyrite ore laboratory column data, see Section 2.3.2, to estimate the time required to a achieve a given copper extraction from a hypothetical chalcopyrite ore heap leach. The ranges of time required to achieve 80% and 70% copper extraction, for chalcocite and chalcopyrite ores respectively, in laboratory columns and in practice are summarized in Table 5.1.

Table 5.1: Extrapolated Chalcopyrite Commercial Heap Recoveries and Leaching Times

Parameters	Chalcocite	Chalcopyrite
Target Cu Extraction	80%	70%
Lab Column Leaching Time	50 to 100 days	150 to 250 days
Commercial Heap Leaching Time	350 to 500 days	750 to 1250 days

The target Cu extraction for the chalcopyrite ore is based on the fact that none of the chalcopyrite laboratory column tests extracted significantly more than 70% Cu and the copper recovery from commercialized chalcocite heap leaching operations is typically at least 5% less than column test results. The chalcopyrite commercial heap leaching time found in the above table assumes that the commercial chalcopyrite heap can be operated at a temperature above 50°C, preferably 60°C to 70°C. This has never been done in practice and therefore the heat balance for a hypothetical low-grade chalcopyrite ore heap leach was investigated as part of this study.

5.1 Heap Heat Balances

Simplified steady-state heat balances were conducted for both an abiotic heap leach, using external ferric generation at a low ferric to ferrous ion ratio, with no forced air injection into the heap and only partial oxidation reactions, and a biotic heap leach, using forced aeration and total oxidation reactions. These balances were done using the chalcopyrite and pyrite reaction rates extrapolated from Hollitt's (2008) data for a hypothetical commercial heap leach.

It is important to maintain a chalcopyrite ore heap leach at a temperature above 50°C, preferably 60°C to 70°C, to achieve an economic level of copper extraction within a viable leaching time. It is also one of the few potentially controllable factors within a heap leaching operation through the use of steam injection into the heap. The other main factors have been discussed in the previous chapter: the acidity/solution make up and particle size. Each of these factors has specific constraints which limit the operator's ability to control the heap conditions. This section will examine a simplified heat balance for a chalcopyrite ore heap leach at a temperature of 60°C and discuss the most significant parameters.

The most complete data to date in the literature on the column leaching of chalcopyrite ore is given by Hollitt (2008). Figure 5.1 shows two of the copper leach curves, in terms of days, found in this source. The top curve is for the bioleaching of a chalcopyrite ore from the La Granja deposit in Peru, with a high ferric to ferrous ratio of 1000 to 1 and with air injected directly into the column to simulate in-situ forced aeration. The bottom curve is for the abiotic leaching of the same ore with a controlled lower ferric to ferrous ratio of 2 to 1 using external ferric oxidization with no forced aeration into the base of the column. Both curves reached similar final copper extractions of about 75% but the biotic leach achieved this level of copper extraction much quicker, in approximately 140 days, versus 240 days for the abiotic leach. Both curves show an initial high rate of copper extraction up to around 50% copper extraction and then a much slower rate of extraction for the remainder of the testing period.

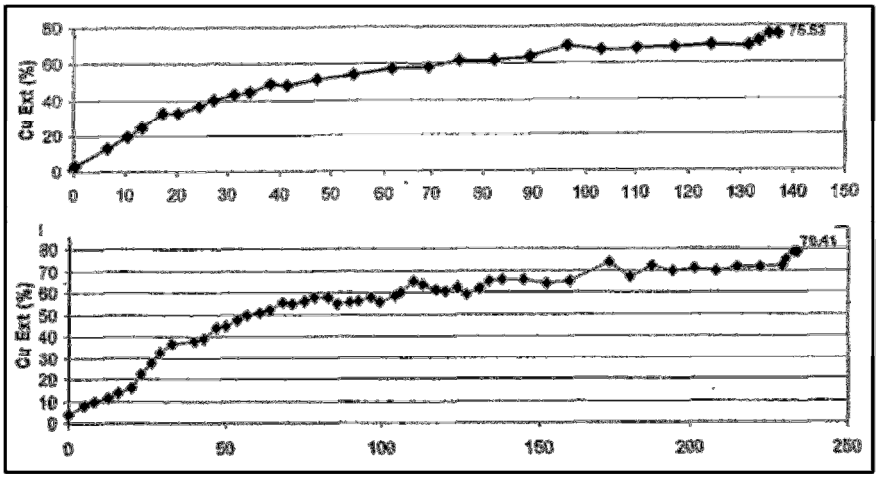

Figure 5.1: Comparison of the Copper Extraction Rate, in days, of the Aerated Biotic Column Leaching Test, top, vs. the Non-Aerated Abiotic Column Leaching Test, bottom, both were run with T = 50°C and d_{80} = 10mm (Hollitt, 2008)

Based on Hollitt (2008), the rates of copper extraction from chalcopyrite and of iron from pyrite were estimated and are summarized in Table 5.2.

Table 5.2: Copper and Iron Extraction Rates from the 2008 RTZ Patent on Chalcopyrite Column Heap Leaching Test Recoveries (Hollitt, 2008)

	Hollitt (2008)			
Test	**Biotic Leach**		**Abiotic Leach**	
Ferric to Ferrous Ratio	1000 to 1		2 to 1	
Extraction Rate	**Cu**	**Fe**	**Cu**	**Fe**
0 - 50% of Cu	1.25 %/day	0.35 %/day	0.75 %/day	0.05 %/day
50-70% of Cu	0.25 %/day	0.35 %/day	0.15 %/day	0.05 %/day

5.1.1 Possible Modes for Operating a Low-Grade Chalcopyrite Ore Heap Leach

The mode which has been the focus of the majority of research to date is a biotic heap leach with forced aeration. This mode occurs when all ferric and elemental sulphur oxidation reactions occur in-situ in the heap and generate the reaction heat for the heap. The other mode is an abiotic heap leach with external ferric generation and no forced aeration. This means that there is little heat of reaction occurring within the heap as all ferric and any elemental sulphur oxidation reactions occur outside of the heap. The partial oxidation reactions of chalcopyrite and pyrite reacting with the acidified ferric sulphate solution in the abiotic heap are shown together with the enthalpy of reactions at 298K in kJ/g.mol. Both generate very little heat of reactions, as indicated.

Cu$FeS_{2(s)} + 2Fe_2(SO_4)_{3(aq)} \rightarrow CuSO_{4(aq)} + 5FeSO_{4(aq)} + 2S^o_{(s)}$ ΔH_{298} = -7.7 kJ/mol of Cp reacted.

$FeS_{2(s)} + Fe_2(SO_4)_{3(aq)} \rightarrow 3FeSO_{4(aq)} + 2S^o_{(s)}$ ΔH_{298} = -6.1 kJ/mol of pyrite reacted.

Sulphides are largely oxidized to sulphur rather than to sulphuric acid thus an insignificant amount of heat is generated from the reactions. This means that all the heat required to keep a heap at temperature must be supplied from an external source. Steam injection appears to be the most practical source. Apart from the small amount of air to carry the saturated steam, there is minimal aeration. All ferric oxidation would have to be done outside the heap in either stirred tanks with air/oxygen injection.

In a biotic chalcopyrite ore heap leach, the initial stage of heap leaching occurs with the mesophile bacteria reactions at low temperatures between 25°C to 40°C. The mesophiles mainly oxidize pyrite, first to ferrous sulphate and elemental sulphur, then to ferric and sulphuric acid. These reactions generate heat to increase the heap temperature, as shown in the following reactions. At this stage the pyrite reacts much faster than the chalcopyrite in the ore.

$CuFeS_{2(s)} + 4O_{2(g)} \rightarrow CuSO_{4(aq)} + FeSO_{4(aq)}$ ΔH_{298} = -1652.7 kJ/mol of chalcopyrite reacted.

$FeS_{2(s)} + 3.5O_{2(g)} + H_2O_{(aq)} \rightarrow FeSO_{4(aq)} + H_2SO_{4(aq)}$ ΔH_{298} = -1430.2 kJ/mol of pyrite reacted.

$2FeSO_{4(aq)} + 0.5O_{(g)} + H_2SO_{4(aq)} \rightarrow Fe_2(SO_4)_{3(aq)}$ ΔH_{298} = -112.4.2 kJ/mol of $FeSO_4$ reacted.

$S^o_{(s)} + 1.5O_{2(g)} + H_2O_{(g)} \rightarrow H_2SO_{4(aq)}$ ΔH_{298} = -603.2 kJ/mol of sulphur reacted.

Once the heap reaches about 40°C, moderate thermophiles take over as the dominant oxidizing organisms and the chalcopyrite in the ore begins to oxidize along with pyrite. Once the temperature rises to around 60°C, thermophiles become active and the chalcopyrite is oxidized at a higher rate than the pyrite. Ensuring a successful transition from one microbe-dominant zone to another higher temperature microbe dominant zone may be very difficult and has never been demonstrated outside a laboratory environment.

Of the two modes, the biotic heap is much more difficult to control for transitions from one dominant organism to another as the heap heats up. This raises the question: is the heat generated from the biotic leaching reactions sufficient to raise the heap temperature 60°C and keep it there through the whole heap cycle? The overall heap mineralogy, and the question as to whether or not the organisms will survive, are major factors in determining the answer. Unfortunately, the operator of the heap leach has little to no control over either of these factors. The acid requirements for a biotic leach should be minimal and, once the heap is operating at steady-state, there will also be an additional cost for

neutralizing the excess iron bleed produced from the chalcopyrite and pyrite oxidation reactions for both modes of heap operation.

The main questions concerning the abiotic heap revolve around how much steam is required to keep the heap at the required temperature and what cost is associated with this operation. Since the elemental sulphur is not oxidized significantly within the heap to sulphate, the acid requirements will be higher than for a biotic leach. With a controlled low Fe^{3+}:Fe^{2+} ratio, less pyrite should be reacted thus reducing the iron bleed and neutralization costs. A sulphur burning acid plant generates about 1.5 t steam/t acid produced and depending on the acid requirements, it is possible that it could provide the bulk of the steam requirements for the heap, which in turn will reduce the heating costs.

5.1.2 Options to Pre-Heat a Low-Grade Chalcopyrite Ore Heap to Operating Temperature

Pre-heating a heap, using the following methods, was examined for:

1. Hot air injection: The time and energy requirements for air to pre-heat a heap were found to be prohibitive. This examination included air heated by waste heat sources.
2. Ore pre-heating during agglomeration: This method is practical in theory but due to the fact that there is usually a long period of time between stacking the ore and starting irrigation, typically several weeks, there is no efficient method available to maintain the ore heat during this period.

The only practical method of raising the temperature of the heap would appear to be through steam injection into the base of the heap using air as a carrier gas. In practice the top of the heap would be covered with an insulating layer of ore above the drip system and covered with a plastic thermo-film to prevent heat loss and evaporation.

Simplified steady-state heat balances at 333K were conducted for both abiotic and biotic commercial heap leaches using the reaction rate data for the initial leaching period found in Table 5.1. The amount of steam required to pre-heat the ambient ore from 298K to 333K prior to the start of irrigation and leaching was also calculated. The calculations assumed a 10 m high, 1 m^2 section in the middle of the heap with no heat losses. The section of the heap used in the following calculations assumes that no heat is lost to the surface or surroundings. The total energy required to pre-heat a 10 m section of ore with a surface area of 1 m^2 with a bulk density of 1.5 t/ m^3 from 298K to 333K is calculated below.

$$\text{Heat Required } \mathrm{Q_r} = 15{,}000\mathrm{kg_{ore}} \times 0.8\frac{kj}{kg_{ore}K} \times (333K - 298K) = 420{,}000\ kJ$$

Assuming that the steam is at an initial temperature of 373K and exits the heap at a temperature of 333K, with values for the specific enthalpy taken from the steam tables, then the total volume of steam required was calculated based on the total energy required to pre-heat the ore:

$$M_{in} = m_{steam} + m_{Air}$$

$$Q_{Air} = m_{Air} \int_{373K}^{333K} C_{p\ (air)}\, \partial T = 40.36\, m_{Air}$$

$$Q_{Steam} = m_{steam} \int_{373K}^{333K} C_{p\ (steam)}\, \partial T + m_{steam\ (cond)} \Delta H_{g \rightarrow l} = 1791 \frac{kJ}{kg}\ m_{steam}$$

$$Q_r = Q_{Air} + Q_{Steam}$$

$$\text{Therefore } M_{in} = 229 \text{ kg of fully saturated air.}$$

Therefore 0.036 m^3 of steam, at 373K, is required per kg of ore.

Based on a section of ore, 1 m^2 by 10 m, with a 10 day pre-heating period:

A flow rate of 54 m^3 of steam a day at 373K for 10 days, per section containing 15,000 kg of ore, would be required to increase the initial heap temperature from ambient, 298K, to the required temperature of 333K.

At a steam cost of $20 per ton, the steam cost to heat a heap initially from ambient to 333K for the start up would only be approximately $0.3 per ton of ore. In practise this cost will likely be higher.

5.1.2 Heat Balance for a Hypothetical Low-Grade Chalcopyrite Heap

To determine the viability of maintaining an elevated temperature for a low-grade chalcopyrite heap leach operation a heat balance analysis was conducted.

This analysis was conducted based on the following assumptions:

1. Chalcopyrite is the only source of copper in the ore.
2. A chalcopyrite grade of 1.44% exists which equates to $0.079\ \frac{gmol_{Cp}}{kg_{ore}}$ and a copper grade of 0.5%.
3. A pyrite grade of 3% exists which equates to $0.242\ \frac{gmol_{Py}}{kg_{ore}}$
4. Ore is pre-heated to 333K unless otherwise stated.
5. An acid solution flow rate of 10 $\frac{L}{h}$ for the initial leaching stage.
6. The chalcopyrite and pyrite were reacted at rates of 0.2% per day and 0.1% per day respectively.
7. Partial oxidation reactions only for the abiotic leach and total oxidation for the biotic leach.
8. The dimensions of the section of the heap analyzed: Width 1m, Length 1m, Lift Height 10 m.
9. No heat loss to surrounding ore.
10. An ore bulk density of 1.5 t/m^3.
11. Ambient air temperature of 298K.

Steady-state heat balances were estimated based on a section of the heap which was assumed to be fully insulated by the rest of the heap thus having no heat loss to the surroundings, Figure5.2.

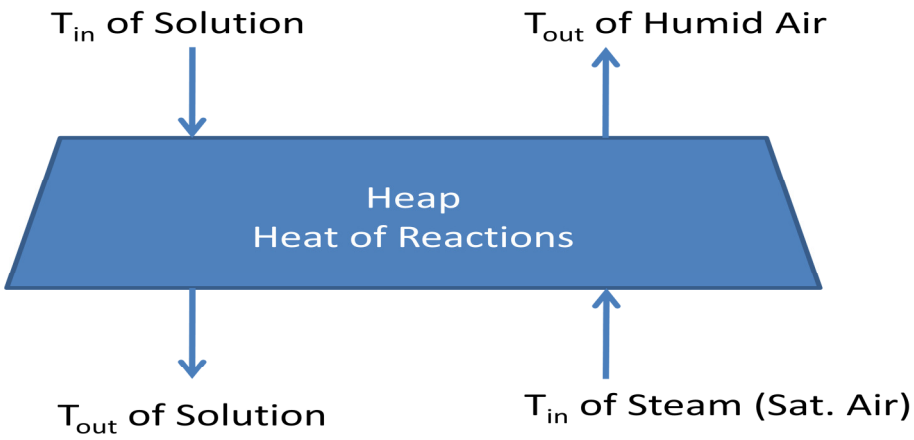

Figure 5.2: Steady-State Heap Heat Leach Balance.

Simple steady-state heat balances were calculated for both an abiotic and a biotic chalcopyrite heap leach at 333K for the initial reaction stage (0 to 50% Cu extraction) when the chalcopyrite and pyrite oxidation rates are highest and for the final stage (50 to 70% Cu extraction) when the reaction rates are much slower. It was assumed that only partial oxidation of sulphides occurred in the abiotic leach compared to total oxidation of sulphides in the biotic leach. The steady-state heat balance was calculated assuming an ore with 0.5% Cu ore as chalcopyrite and 3% pyrite ore, on a per day basis for a 1 m^2 by 10 m high heap section.

For a commercial biotic heap, all ferric and sulphur oxidation occurs within the heap at a temperature of 333K with the leach solution and air at 298K. It was also assumed that the solution temperature was 323K and the irrigation rate was 10 L/m^2/h in the initial stage and 5 L/m^2/h in the final stage. The forced air rate was calculated from the extent of the total oxidation reactions for chalcopyrite and pyrite, assuming a 30% O_2 utilization in the initial stage and 15% in the final stage as measured by Petersen (2007) in a pilot chalcopyrite ore column leach. It was assumed that the solution and the humid air exited at the heap temperature, as shown by the following calculations.

Biotic heap heat of reactions:

$$Chalcopyrite = \frac{15{,}000kg \times 0.0144}{183.5} \times 0.2\frac{\%}{day} \times -1652.7\frac{kJ}{gmol} = 3.30\frac{MJ}{day}$$

$$Pyrite = \frac{15{,}000kg \times 0.03}{120} \times 0.1\frac{\%}{day} \times -1430.2\frac{kJ}{gmol} = 5.36\frac{MJ}{day}$$

Biotic heap oxygen requirements assuming 30% efficiency:

$$Chalcopyrite = \frac{15{,}000kg \times 0.0144}{183.5} \times 0.2\frac{\%}{day} = 2\frac{gmol}{day}$$

$$Pyrite = \frac{15{,}000kg \times 0.03}{120} \times 0.1\frac{\%}{day} = 3.75\frac{gmol}{day}$$

$$Oxygen = 2\frac{gmol_{Cp}}{day} \times 4\frac{mol_{O_2}}{mol_{Cp}} + 3.75\frac{gmol_{Py}}{day} \times 3.5\frac{mol_{O_2}}{mol_{Py}} = \frac{18.02}{30\%}\frac{gmol}{day} = 60.07\frac{gmol}{day}$$

Biotic heap air requirements based on the above oxygen requirements:

$$Air = 6.47\frac{m^3}{day}$$

Biotic heat of content of steam in the exit air:

$$Heat\ out - humid\ air = 6.47\frac{m^3}{day} \times \frac{0.2\ atm}{0.8\ atm} \times 2600\frac{kJ}{m^3} = 4.20\frac{MJ}{day}$$

For a commercial abiotic heap leach it was assumed that the chalcopyrite and pyrite are only partially oxidized.

Abiotic heap heat of reactions:

$$Chalcopyrite = \frac{15{,}000kg \times 0.0144}{183.5} \times 0.2\frac{\%}{day} \times -7.7\frac{kJ}{gmol} = 0.01\frac{MJ}{day}$$

$$Pyrite = \frac{15{,}000kg \times 0.03}{120} \times 0.1\frac{\%}{day} \times -6.1\frac{kJ}{gmol} = 0.02\frac{MJ}{day}$$

It can be seen from the above reactions that the heat of the partial oxidation reactions is only slightly exothermic and little heat will be generated within the heap from these reactions. Therefore virtually all of the heat required for a high temperature abiotic heap leach must be supplied externally.

In practice there will be some heat lost in both the biotic and abiotic operations from the solution as the heat exchanger and the solution distribution system will not be 100% efficient. Assuming a temperature drop of 10°C between the PLS at 60°C and the irrigation solution at 50°C, the make-up heat required was calculated as shown:

$$Make\ up\ Heat\ Required = 0.24\frac{m^3_{soln}}{day} \times \Delta T \times Cp_{soln} = 5.04\frac{MJ}{day}$$

The heat balance for the hypothetical commercial abiotic heap leach is based on the same assumptions and are compared with the biotic heap leach results in Tables 5.2 and 5.3.

Table 5.2: Steady-State Heat Balance of an Abiotic Low-Grade Chalcopyrite Heap Leach

Parameter	Abiotic Heap Initial Stage	Abiotic Heap Final Stage
Cp reaction rate	0.2%/day	0.04%/day
Py reaction rate	0.1%/day	0.02%/day
Heat of reaction, MJ/d	0	0
Heat out – solution, MJ/d	-5.0	-2.5
Heat out – humid air, MJ/d	0	0
Total heat out, MJ/d	-5.0	-2.5
Heat required, MJ/d	**-5.0**	**-2.5**
Steam Required, $m^3/d/t_{ore}$	**0.22**	**0.11**

Table 5.3: Steady-State Heat Balance of a Biotic, Low-Grade Chalcopyrite Heap Leach

Parameter	Biotic Heap Initial Stage	Biotic Heap Final Stage
Cp reaction rate	0.2%/day	0.04%/day
Py reaction rate	0.1%/day	0.02%/day
Heat of reaction, MJ/d	8.6	1.76
Heat out – solution, MJ/d	-5	-2.5
Heat out – humid air, MJ/d	-4.2	-1.65
Total heat out, MJ/d	-0.5	-2.39
Heat required, MJ/d	**-0.5**	**-2.39**
Steam Required, $m^3/d/t_{ore}$	**0.02**	**0.096**

The abiotic heap leach requires 10 times more steam than the biotic heap leach in the initial 250 day stage. For the final stage the heat required to maintain the heap temperature was similar for both the abiotic (2.5 MJ/day) and biotic (2.39 MJ/day) heap leaches since the amount of heat generated by the oxidation reactions is very small. Assuming a steam cost of $20 per ton of steam, a cost per ton of ore during the initial stage for an abiotic leach would be approximately $2.75 and $0.25 for a biotic leach. The cost for steam heating could be reduced by the waste steam generated from a sulphur burning acid plant.

5.1.3 Heap Heat Balance Analysis Summary

A summary of the previous heat balance analyses to maintain optimal leaching conditions with the ore at 333K indicate that:

- The partial oxidation reactions for chalcopyrite and pyrite generate insignificant amounts of heat and significant external heat input is required to maintain the ore at the target temperature of 333K.
- Although the biotic leach total oxidation reactions generate significant amounts of heat some external heat will still be required.
- Steam is considered to be the only practical method of supplying heat to a commercial heap leaching operation.
- The ore can be brought to the initial temperature of 333K using 0.036 m^3 of steam at 373K per kg of ore within the heap section.

Based on the simplified steady-state heat balances and the assumptions used, the steam requirements for an abiotic leach would be significantly higher than for a biotic heap leach. This difference in heat requirements equates to about $2.50 more per ton of ore for the abiotic heap at a steam cost of $20 per ton of steam. The economics of a hypothetical abiotic heap leach are evaluated in Section 5.3 based on the assumption that a hot heap is technically feasible.

5.2 Proposed Heap Setup

As stated in Section 5.1.1 there are two ways to operate a heap. The first is a biotic heap leach with forced aeration where all ferric and elemental sulphur oxidation reactions occur in-situ in the heap and generate the reaction heat for the heap. The other method is an abiotic heap leach with steam injection for heating, external ferric generation and no aeration. This means that essentially zero heat of reaction occurs within the heap as all ferric and any elemental sulphur oxidation reactions occur outside the heap. Biotic leaching is likely to be more difficult to implement effectively in the near future due to the sequential nature of the following setup, Figure 5.3. The figure shows a process flow sheet of an abiotic, external ferric generating chalcopyrite ore heap leaching operation, where (1) is the sulphuric acid solution, (2) is the leaching solution containing the required ferric concentration, (3) is the copper containing pregnant leach solution, (4) is the iron bleed system to control the iron content of the leaching solution and sends the excess iron to be neutralized, (5) is the re-acidified raffinate from the SX plant, and (6) is the waste steam used to maintain the heap temperature.

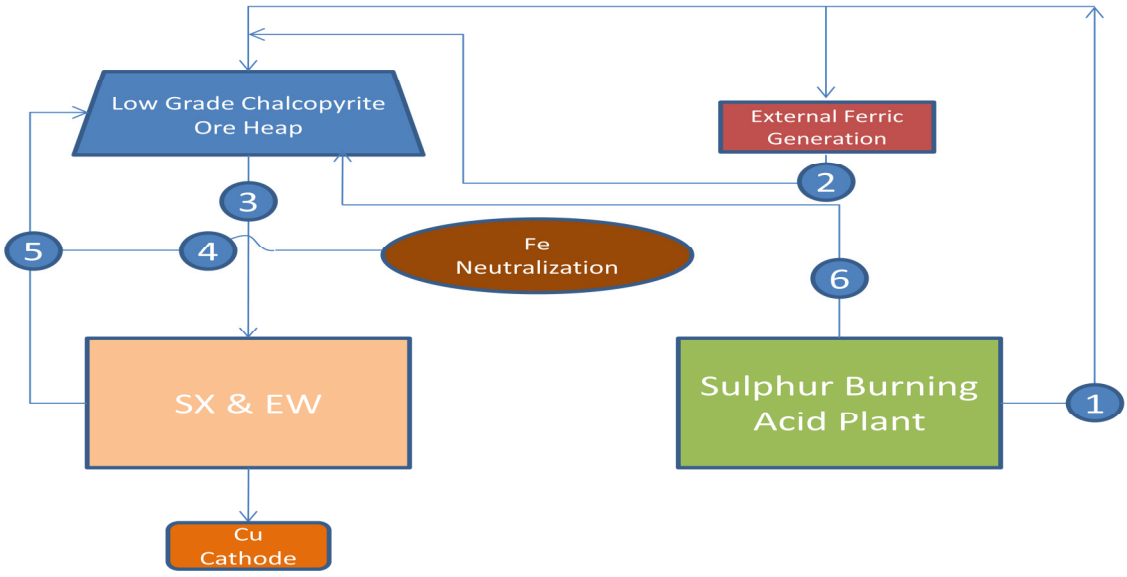

Figure 5.3: Proposed External Ferric Generation Low Grade Heap Leaching Circuit.

5.3 Economics of a Low-Grade Chalcopyrite Heap Leach vs. a Concentrator

As with any mineral processing the ability to achieve high recoveries is important only if these recoveries allow the overall operation to be economically viable. The following sections will analyse and compare the economics of a low-grade chalcopyrite heap leaching operation and a standard milling operation. Detailed cash flow sheets are found in the Appendix.

5.3.1 Future Copper Price

The copper price trends over the past 30 years have been directly correlated to energy costs, and the global economy. The price of copper performs like other base metals in a cyclical manner making future predictions difficult. Even with this instability the future copper price is not likely to ever drop back to the $1 (2008 USD) per pound level due to continued demand from Asia for copper and the increasing cost of complex copper ore processing. Based on future copper production costs, energy prices and global demand it is reasonable to conclude that a copper price between $2-$4 USD per pound can be expected over the next 20 years. An estimated long term copper price of $2.25 USD per pound will be used for the following economic analysis.

5.3.2 Economic Analysis of a Low Grade Chalcopyrite Leach

An economic feasibility analysis was conducted for a hypothetical chalcopyrite heap leaching operation. It was assumed that an existing 100,000 tpa copper heap leaching/SX/EW facility that has completed mining operations of a chalcocite ore will be used for chalcopyrite leaching now that the chalcocite ore body is depleted. It is assumed that the chalcopyrite ore body, below the depleted chalcocite ore body, has a minimum mine life of 20 years at 0.5% Cu in chalcopyrite ore. Two industry examples of this type of mine are BHP Billiton's Spence Mine and Teck's Quebrada Blanca Mine which both have very large chalcopyrite reserves.

This analysis was conducted based on the following assumptions:

1. Three year construction period with the CAPEX divided evenly over the three years.
2. Three leaching cycle with a total of 70% of the copper extracted from the chalcopyrite ore. Cu recoveries by year were expected to be - Yr1: 50%, Yr2: 10%, Yr3: 10%.
3. Copper grade of 0.5%.
4. Strip ratio of 1:1.
5. Lift height of 10 meters.
6. The yearly ambient temperature is 298K
7. Acid requirements are 20 kg acid/t ore
8. The heap pad, SX/Electro-winning plant and mining equipment are already in place.
9. A discount rate of 5% was used.
10. Long term copper price of $2.25 US/lb of Cu.

5.3.2.1 Heap Leach Capital Cost Estimate

The capital expenditures are all based on the assumption that the pre-existing infrastructure is in place from a chalcocite leaching operation that has reached the end of its ore reserve. Since the time required for the chalcopyrite ore leaching is at least 3 years, compared to only 1 year for the previous chalcocite heap leach, the capacity of the heap leach pad will need to be increased and, as such, the crushing plant, irrigation system and stacking equipment will need to be upgraded to handle the higher load. It is also expected that the copper grade in the PLS will be reduced by almost 50%, thus the SX plant will need to be roughly doubled in size. For the abiotic heap leach, an external ferric generating facility is required. The capital and operating costs were estimated from a detailed cost spreadsheet for copper ore crushing, heap leaching, and purchase of a SX&EW plant provided from recent industry sources by Dr. Peacey, 2010 and is summarized in Table 5.4.

Table 5.4: Summary of Heap Leaching Process Capital Expenditures (±30%)

Direct Costs	**US$ M**
Crushing Plant	$ 36.4
Agglomeration & Stacking Equipment	$ 35.2
Expansion of Pad Construction	$ 96.1
Heap Leaching Circuit	$ 62.8
Upgrading of SX Plant	$ 85.2
EW Plant	Existing
Sulphuric Acid Plant	$ 91.2
Ferric Generation Facility	$ 20.0
Direct Cost Break Down	
Equipment	$ 170.8
Materials	$ 192.2
Construction	$ 64.0
Total Direct Costs	$ 427.1
Field Costs	
Indirect	$ 128.1
Total Field Costs	$ 555.3
Other Costs	
Engineering and Services	$ 61.6
Contingency	$ 175.0
Total Other Costs	$ 236.6
Total CAPEX	**$ 792.0**

5.3.2.2 Heap Leaching Operating Costs

The estimated operating costs (±25%) are based on the proposed heap leach and are summarized in Table 5.5.

Table 5.5: Summary of Heap Leaching Process Operating Expenditures(±25%)

Operating Costs	Unit Cost (US$)	Annual Cost (US$ M)
Mined Ore (US$/t ore)	$1.50	$42.8
Mined Waste (US$/t ore)	$1.50	$42.8
Crushing (US$/t ore)	$0.6	$17.1
Steam (US$/t ore)	$1.83	$52.2
Agglomeration & Stacking (US$/t ore)	$0.75	$21.4
Concentrated Acid Cost (US$/t ore)	$0.75	$64.2
Other Heap Leaching Costs (US$/t ore)	$0.15	$12.8
SX & EW (US$/t Cu)	$330	$36.2
Ferric Generation (US$/t ore)	$0.2	$5.7
Iron Bleed and Fixing (US$/L soln)	$0.8	$22.8
G&A Cost (US$/t ore)	$0.5	$14.2
Total OPEX	**$11.65 US/t ore**	**$332.8**

5.3.2.3 Heap Leach Pre-Tax Cash Flow Analysis

Table 5.6 shows the pre-tax cash flows from the leach cycle on a yearly basis once it has reached 100% capacity. A complete cash flow analysis is found in the appendix. The three year leaching cycle recovers 50% in the first year and 10% for the second and third years.

Table 5.6: Estimated Annual Pre-Tax Cash Flow for a Chalcopyrite Heap Leach

Cash Flow	
Cu Revenues (US$ M/yr)	$493.9
OPEX (US$ M/yr)	$332.7
Yearly Cash Flow Before Tax(US$ M/yr)	**$161.1**

The estimated net present value, at a 5% discount rate, and the pay-back period for the hypothetical chalcopyrite heap leach is summarized in Table 5.7. The full analysis is given in the Appendix, Table 8.13.

Table 5.7: NPV and Payback Period for a Chalcopyrite Heap Leach

Item	
NPV @5% (US$ M)	$1,068
Payback Period (yrs)	7.1

5.3.2.4 Heap Leaching Sensitivity Analysis

A sensitivity analysis was done to investigate the impact of copper price, recovery, operating costs and capital cost variations on the NPV, plotted in Figure 5.4.

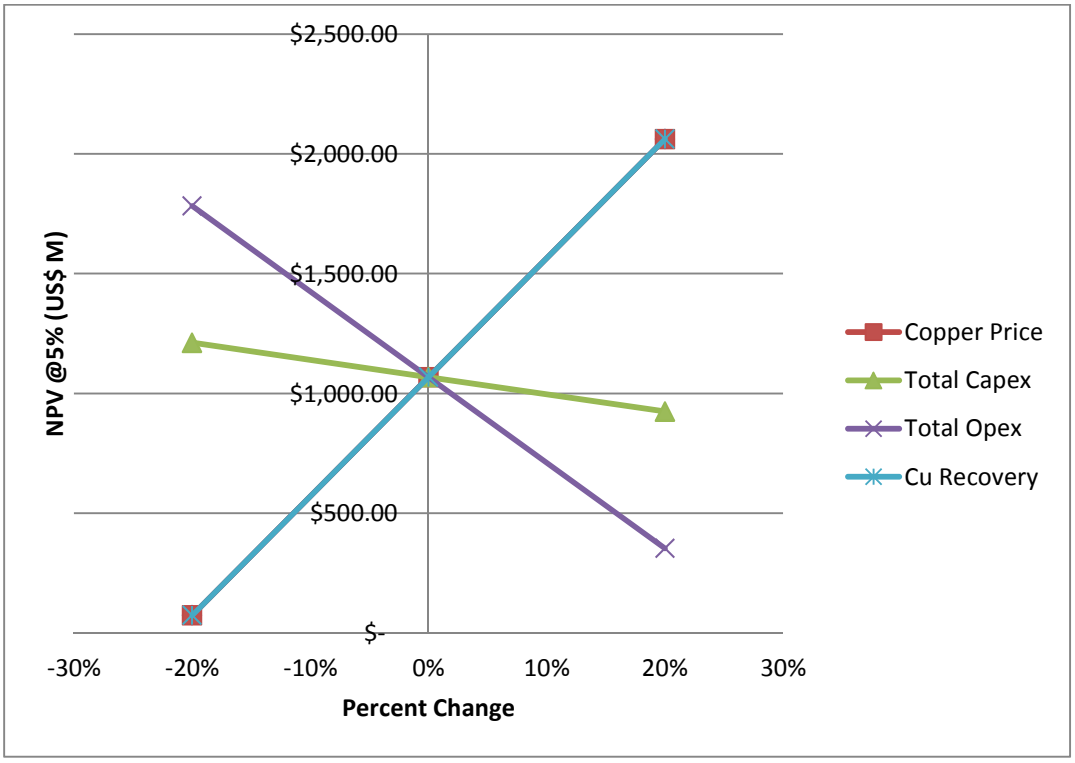

Figure 5.4: Sensitivity Analysis of a Heap Leaching Operation's NVP

Figure 5.4 shows that thepre-tax NPV of the leaching operation is most sensitive to changes in either the copper price or recovery, and then the operating costs. It is least sensitive to capital costs. The impact of a 20% change in either the copper price or recovery is the same.

5.3.3 Economic Analysis of a Standard Milling Operation

The method for milling chalcopyrite ore is proven and has been shown to be economical for various commercial projects. The following analysis will show the economics behind a current day milling operation option for a low grade chalcopyrite ore in comparison against the low-grade heap leaching operation that was described in the previous section.

This analysis was conducted based on the following assumptions:

1. Three year construction period with the capital expenditures divided evenly over the three years.
2. The first year ramp up reaches 75% mill capacity.
3. 90% of the copper is recovered from the chalcopyrite ore.
4. Copper grade is 0.5%.
5. Long term copper price of $2.25 US/lb of Cu.
6. 20 year life of mine with 28.57 million tonnes of ore mined per year.
7. Mining equipment is already in place.
8. Strip ratio of 1:1.
9. A discount rate of 5% was used.
10. Capital and operating costs for the concentrator option are based on a recent copper project.

5.3.3.1 Concentrator Option - Process Capital Cost Estimates

The initial capital cost estimates used in the analysis are shown in Table 5.8. The capital and operating costs are based on a recent NI 43-101 pre-feasibility study conducted by Augusta Resource Corporation in 2009 for a 75,000 tpd concentrator. (Augusta, 2009)

Table 5.8: Estimated Capital Costs for Concentrator Option

Direct Costs	**US$ M**
TDC Mill	$457.3
Field Costs	
Indirect	$137.2
Total Field Costs	$594.5
Other Costs	
Engineering and Services	$89.2
Contingency	$205.1
Total Other Costs	$294.3
Total CAPEX	**$ 888.8**

5.3.3.2 Concentrator Option - Operating Cost Estimates

The operational costs estimated for the concentrator option are shown in Table 5.9

Table 5.9: Estimated Concentrate Operating Costs

OPEX	**Cost (US$)**	**Annual Costs (US$ M)**
Mined Ore (US$/t ore)	$1.50	$42.9
Mined Waste (US$/t ore)	$1.50	$42.9
Mill Operating Costs (US$/t ore)	$5.00	$142.9
Concentrate Freight (US$/t conc)	$60.00	$25.7
Transport Cost (US$/t conc)	$80.00	$34.3
Refinery Charge (US$/lb payable Cu)	$0.08	$23.0
G&A Cost (US$/t ore)	$0.25	$7.1
Total OPEX (US$ M/yr)		**$318.8**

5.3.3.3 Mill Process Cash Flow Analysis

In the first year of operation it was assumed, based on previous industry experience, that the mill was running at only 75% capacity. Table 5.10 shows the cash flow of the standard milling operation on a yearly basis once it reaches 100% capacity. The full analysis is given in the Appendix, Table 8.14.

Table 5.10: A Cash Flow Comparison of Heap Leaching Process Leach Cycle Times

Cash Flow	**Mill**
Cu Production Cost (US $/lb)	$1.23
Cu Revenues (USM $)	$583.80
OPEX (USM $)	$318.75
Yearly Cash Flow Before Tax(USM $)	**$264.44**

Table 5.11 shows the before tax discounted NPV and payback period for a standard milling process.

Table 5.11: NPV and Payback Period Comparison of Heap Leaching Process Leach Cycle Times

Item	**Mill**
NPV @5% (US$ M)	$1920.13
Pay Back Period (yrs)	4.51

5.3.3.4 Mill Option Sensitivity Analysis

A sensitivity analysis was done to investigate the impact of copper price, recovery, operation cost and capital cost variations on the NPV of a standard milling operation, Figure 5.5.

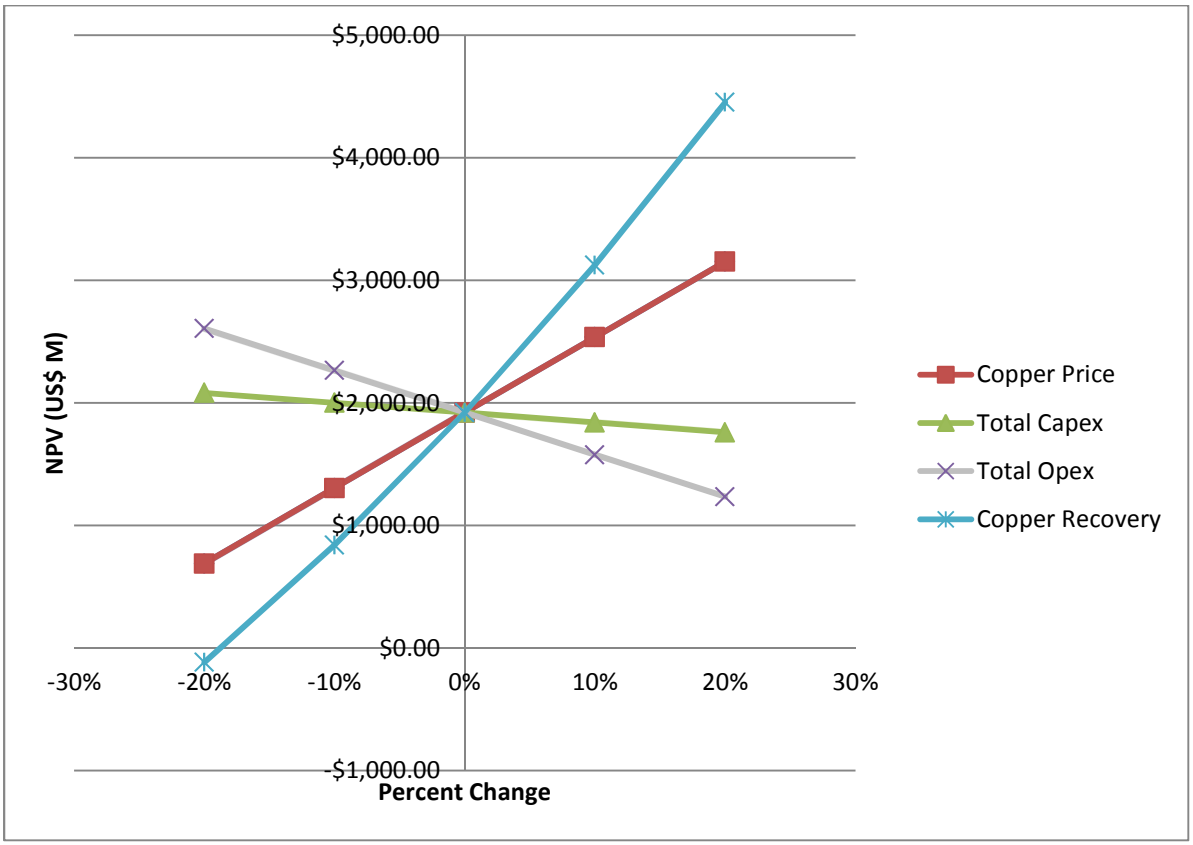

Figure 5.5: Sensitivity Analysis of a Standard Mill Operation's NPV

Figure 5.5 shows that the NPV of the milling operation is most sensitive to changes in the copper recovery rate and price.

5.3.6 Comparison of the Mill Operation and Low Grade Chalcopyrite Heap Leach Options

The pre-tax cash flow of the two processing options gives a solid overview of the economic benefits of the standard milling operation. Figure 5.6 shows that the cash flow for a mill operation is more favourable than for the 3 year leach cycle heap operation with the exception of the final 3 years of leaching where the physical mining operation is complete and there are no mining costs. Note that the initial capital expenses are not shown in this figure.

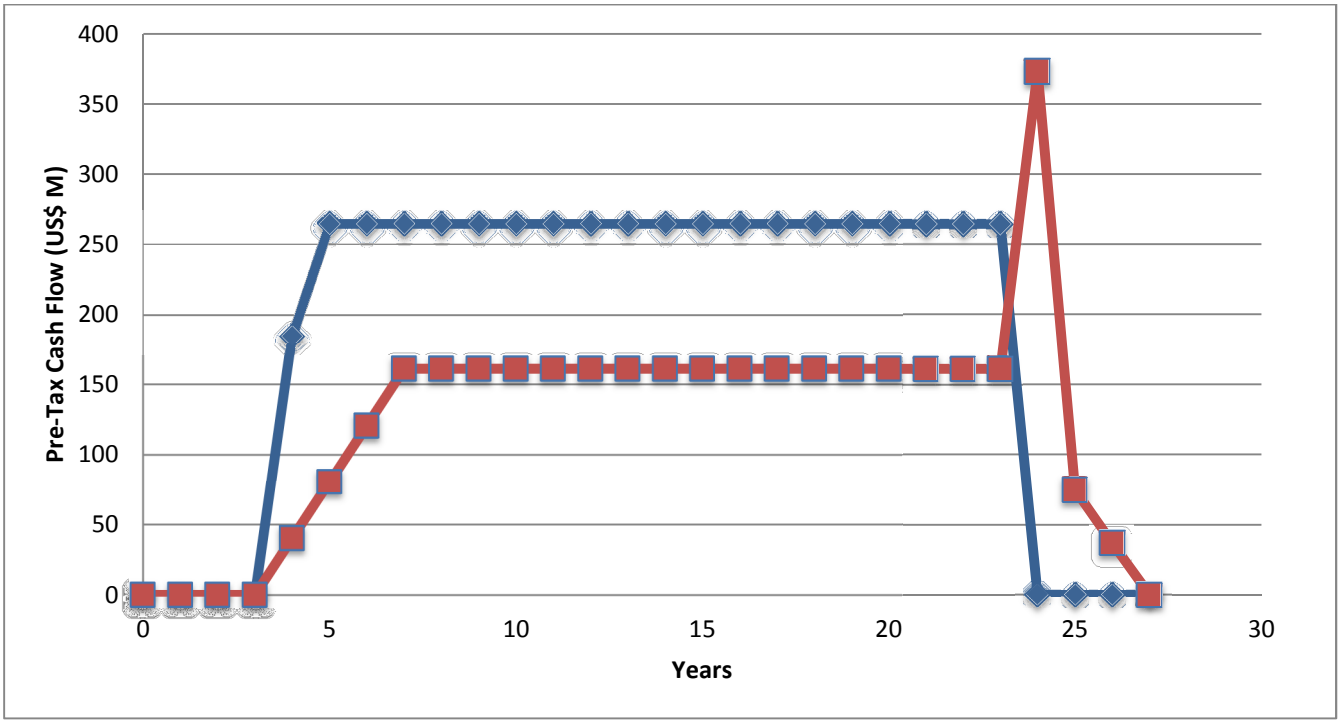

Figure 5.6: Cash Flow Comparison of a Chalcopyrite Heap Leach and a Standard Mill Options.

The key economic considerations from the above evaluation of a low-grade chalcopyrite heap leaching operation and a standard milling operation are summarized in Table 5.12.

Table 5.12: Economic Comparison of a Standard Mill Operation and a Heap Leaching Operation.

	Standard Mill Operation	**Heap Leaching Operation 3 Year Cycle**
Capital Expenditures (US$ M)	$888.7	$792.0
Operational Expenditures (US$ M)	$318.7	$332.7
Copper Revenues (US$ M)	$583.8	$493.92
Annual Pre-Tax Profit (US$ M)	$264.4	$161.1
Net Present Value @5% (US$ M)	$1,920.1	$1,068.0
Payback Period (Years)	4.51	7.1

In terms of the copper revenues, annual pre-tax profits, annual operating costs and net present value the milling option is the most economically viable option.

This preliminary conceptual economic comparison shows that, even under the most favourable conditions, an ideal low-grade chalcopyrite heap leaching operation does not appear to have better economics than the proven concentrator option.

Chapter 6 Conclusions and Future Work

6.1 Conclusions

The shake flask tests using the specified ore, with a particle size of 1.18 mm to 2.38 mm, found that the lixiviant employing 1 M NaCl in a 9.8 g/L sulphuric acid solution at 60°C extracted the highest amount of copper, this being 69% Cu extraction in 16 days. The next highest extraction was from the pre-acidified ore with a fine pyrite concentrate, added at a 4 to 1 ratio of pyrite to chalcopyrite, in a 9.8 g/L sulphuric acid solution at 60°C that achieved 59% Cu extraction in 16 days. The other flask tests extracted lower amounts of copper and were all in the range of 30 to 40% copper extracted. In the mini-column tests, the rates of copper extraction were similar for all test conditions. The rate of Cu extraction, even with a small particle size of 1.18 mm to 2.38 mm and at an elevated temperature of 50°C, was slow for all test conditions. The mini-column ore leaching tests showed that, with this specific ore and under these conditions, the test using ore with 40 g of pure pyrite mineral addition in a 9.8 g/L sulphuric acid solution extracted the highest amount of copper, this being 8.6% Cu extracted in 30 days. This was followed closely by the ore with a 5 gpl iron in a 9.8 g/L sulphuric acid solution using a 1:1 ferric to ferrous ion ratio that achieved 8.3% Cu extracted in 30 days. The chloride test using 1M NaCl in a 5 gpl iron in a 9.8 g/L sulphuric acid solution, using a 5:1 ferric to ferrous ion ratio, extracted the lowest amount of copper, this being 5.9 % Cu extracted in 30 days. Based on extrapolating the average rate of reaction for the last 20 days, it was estimated that it would take approximately 500 days to achieve 70% copper extraction.

The effects of key parameters are summarized in Table 6.1.

Table 6.1: Summary of Key Operating Parameters

Parameter	From Literature	From Test Results
Temperature	Maintaining a temperature above 50°C is important for a high Cu. van Staden shows that a higher initial heap temperature increases the rate of extraction significantly.	A temperature of 60oC had a significant increase in Cu extraction when compared with a test at 25oC under the same conditions in both ore and concentrate tests.
Particle size	A smaller size is important but heap size limited to about 10 mm due to percolation limits.	A smaller ore size increases the extraction rate of Cu from chalcopyrite ore tests.
Acid concentration (pH)	Contradictory results from chalcopyrite concentrate tests.	The lower overall pH of the pyrite tests may have helped increase Cu extraction.
Redox potential	With concentrates highest rate at low ORP equal to Fe3+:Fe2+ ratio of about 0.5. In ore column leach, no significant effect on Cu extraction except in terms of the external generation of the lixivant.	Low Fe3+:Fe2+ did not show significant effect with ore.
Chloride additions vs sulphate only	Higher Cu extraction with concentrates of >1 molar NaCl .	Higher Cu extraction rates in shake flask tests.
Pyrite additions	Galvanox shows high extraction for concentrates and chalcopyrite ore heap leaching works better with higher Py in ore	Higher Cu extraction rates in mini-columns.

The conceptual engineering of a hot, abiotic heap leach for low-grade chalcopyrite ore, including hypothetical heat and mass balances, was conducted. It was found that the partial oxidation reactions, during an abiotic heap leaching operation, for chalcopyrite and pyrite, generate insignificant amounts of heat and that external heat input is required to bring the ore from ambient temperature to 333K. The reactions from a biotic heap were found to produce significantly more heat than an abiotic leach but will still require external heat input. The ore can be brought to a temperature of 333K using forced saturated air injection of 0.036 m^3 of steam, at 373K, per kg of ore over 10 days. To maintain the heap at this temperature a significant amount of steam is required over the leaching cycle. This steam can be generated by using the waste heat from a sulphur burning acid plant.

It was also found that aeration of a heap has a potentially negative effect on maintaining a high heap temperature and that forced aeration at high rates may be impractical. Instead of aeration an external oxidant will need to be generated.

A comparison of the economics of a conceptual low-grade copper heap leaching operation was made with a proven concentrator option for a hypothetical supergene heap leaching operation that is nearing the end of the supergene ore mine life, which has a large hypogene reserve of low-grade chalcopyrite ore bearing 0.5% copper. The preliminary conceptual economic comparison showed that, even under the most favourable conditions, an ideal low-grade chalcopyrite heap leaching operation does not have better economics than the proven concentrator option.

6.2 Areas of Future Work

The leaching of chalcopyrite will be difficult to control outside of a laboratory setting and scaling up to a commercial heap will present more difficulties in creating ideal bacteria reaction conditions and high temperature heap operations. This means that future research should also focus on the engineering of a heap leach to ensure a reliable high temperature operation.

Based on the low cost of sodium chloride and its apparent positive effect on leaving chalcopyrite ore while reducing the overall iron in solution, meaning that less iron neutralization would be required, it would be worthwhile pursuing future research based on leaching chalcopyrite ore using chloride as an additive. The addition of silver to experimental chalcopyrite ore leach tests has shown high copper extraction even at ambient temperature. Research into exploring lower cost metal ion additives may be a worthwhile endeavour.

7.0 References

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8.0 Appendices

8.1 Concentrate Shaker Flask Raw Data

Factorial design experiments were conducted on chalcopyrite concentrate but the results lead to no substantial conclusions. The test results are given in Tables 8.1 and 8.2.

Table 8.1: Copper Recoveries (Temperature = 60°C)

% Rec	0	24	48	96	120
AA1	0%	10.90%	20.78%	29.29%	40.88%
AA2	0%	3.76%	7.86%	7.33%	14.88%
AA3	0%	10.55%	14.36%	26.33%	26.46%
AA4	0%	3.38%	5.63%	28.32%	37.82%
AA5	0%	8.87%	14.48%	22.59%	37.67%
AA6	0%	5.32%	8.12%	15.16%	37.53%
AA7	0%	4.10%	8.84%	19.42%	34.38%
AA8	0%	1.88%	3.19%	7.49%	17.34%
BB1	0%	9.42%	21.60%	50.84%	69.96%
BB2	0%	10.27%	25.66%	62.08%	63.64%
BB3	0%	34.32%	45.33%	57.34%	67.20%
BB4	0%	22.18%	47.31%	59.12%	83.58%
BB5	0%	3.54%	4.87%	16.49%	25.39%
BB6	0%	2.92%	3.95%	8.25%	19.32%
BB7	0%	2.72%	9.61%	33.03%	75.16%
BB8	0%	8.95%	20.12%	40.97%	72.27%

Table 8.2: Copper Recoveries (Temperature = 25°C)

% Rec	0	24	48	96	120
A1	0%	13.08%	25.58%	32.67%	33.19%
A2	0%	6.52%	9.61%	9.45%	16.43%
A3	0%	16.58%	15.82%	29.66%	30.90%
A4	0%	8.22%	12.93%	23.50%	29.02%
A5	0%	18.30%	16.74%	25.39%	35.29%
A6	0%	6.88%	8.64%	17.01%	31.65%
A7	0%	5.06%	8.63%	21.79%	31.12%
A8	0%	2.76%	2.97%	7.41%	16.27%
B1	0%	12.01%	23.53%	61.21%	62.74%
B2	0%	11.88%	28.54%	72.49%	73.36%
B3	0%	32.14%	52.68%	67.24%	67.91%
B4	0%	29.13%	54.68%	69.49%	72.96%
B5	0%	7.79%	8.12%	24.93%	25.29%
B6	0%	8.33%	8.53%	9.30%	19.55%
B7	0%	5.83%	15.26%	38.93%	49.33%
B8	0%	10.07%	21.36%	48.37%	56.03%

8.2 Ore Tests Raw Data

Table 8.3: Particle Size Comparison 333K, 20g Ore & H2SO4 @ pH 1

Time	24	48	72	120
1 mm	14.50	20.55	32.70	39.20
10 mm	2.24	4.29	8.65	14.45

Table 8.4: Calculated average values taken from the duplicate experimental data from the ore shake flask tests.

Ferrous Avg Days	ORP	pH	% Cu Rec	% Fe Rec	Ferrous in Soln (mg)	Ferric in Soln (mg)	g/L Ferric in Soln	Fe3/Fe2	Total Fe In Solution
0	381.15	0.89	0%	0%	500.00	0.00			500.00
1	445.2	1.285	9%	21%	70.19	465.70	4.65702556	6.63477	535.89
2	446.4	1.615	14%	24%	52.75	552.24	5.522426119	10.4686	605.00
5	446.3	1.925	24%	26%	32.26	618.90	6.18903959	19.1838	651.17
6	451.3	1.845	25%	30%	40.11	707.29	7.072899963	17.6341	747.40
7	463.8	1.545	28%	35%	56.68	821.66	8.216638806	14.4975	878.34
14	469.8	1.615	35%	34%	40.98	804.89	8.04889306	19.6405	845.87
16	476.65	1.52	39%	39%	56.68	917.76	9.177598806	16.1931	974.44
Ferric Avg Days	ORP	pH	% Cu Rec	% Fe Rec	Ferrous in Soln	Ferric in Soln (mg)	g/L Ferric in Soln	Fe3/Fe2	0.00 Total Fe In Solution
0	593.85	1.005	0%	0%	0.00	500			500.00
1	521.8	1.1	9%	25%	138.64	499.37	4.993692658	3.60195	638.01
2	500.05	1.4	14%	27%	193.13	471.60	4.71603672	2.44184	664.74
5	456.2	1.34	24%	23%	219.29	368.30	3.683023497	1.6795	587.60
6	459.55	1.4	25%	24%	171.77	428.89	4.288868437	2.49683	600.66
7	483.55	1.22	28%	27%	149.54	520.66	5.20659069	3.48179	670.20
14	468.65	1.285	37%	25%	97.22	539.31	5.393070301	5.54721	636.53
16	480.3	1.31	37%	27%	83.71	581.78	5.817763444	6.95021	665.48
H2SO4 Avg Days	ORP	pH	% Cu Rec	% Fe Rec	Ferrous in Soln	Ferric in Soln (mg)	g/L Ferric in Soln	Fe3/Fe2	Total Fe In Solution
0	497	1.24	0%	0%	0.00	0.00			0.00
1	451.8	0.9	6%	5%	58.42	67.95	0.679452937	1.16305	126.37
2	444.75	1.1	10%	7%	75.42	97.68	0.976780226	1.29507	173.10
5	445.05	1.43	22%	12%	121.64	180.85	1.808471075	1.48679	302.48
6	444.2	1.445	25%	14%	110.30	229.36	2.293628226	2.07944	339.66
7	459.15	1.125	29%	15%	102.02	285.54	2.855371164	2.79892	387.55
14	468.2	1.315	44%	20%	110.74	394.84	3.94842066	3.5656	505.58
16	473.05	1.285	46%	22%	103.76	442.45	4.4245162	4.26415	546.21
H2SO4 & Pyrite Avg Days	ORP	pH	% Cu Rec	% Fe Rec	Ferrous in Soln	Ferric in Soln (mg)	g/L Ferric in Soln	Fe3/Fe2	Total Fe In Solution
0	486.25	1.17	0%	0%	0.00	0.00			0.00
1	421.6	0.9	15%	5%	68.45	55.91	0.55905678	0.81677	124.35
2	422.85	1.095	21%	7%	88.94	79.49	0.794946577	0.89382	168.43
5	420.25	1.245	34%	11%	174.39	98.55	0.985509606	0.56512	272.94
6	415.65	1.32	37%	12%	160.87	149.84	1.498392117	0.93141	310.71
7	424.25	1.09	42%	14%	159.57	203.29	2.032862047	1.274	362.85
14	425.6	1.14	57%	21%	230.19	299.59	2.995941747	1.3015	529.79
16	427.35	1.105	59%	24%	225.83	365.34	3.65343939	1.61776	591.18
Sodium Chloride Avg Days	ORP	pH	% Cu Rec	% Fe Rec	Ferrous in Soln	Ferric in Soln (mg)	g/L Ferric in Soln	Fe3/Fe2	Total Fe In Solution (mg)
0	447.6	0.695	0%	0%	0.00	0.00			0.00
1	380.8	2.9	10%	1%	31.39	4.42	0.044185086	0.14076	35.81
2	422.05	2.435	12%	4%	34.22	70.55	0.705505663	2.06146	104.77
3	446.8	1.805	16%	8%	39.67	159.50	1.594964654	4.02025	199.17
10	441	2.16	31%	4%	14.39	93.06	0.93057699	6.46817	107.44
12	450.85	1.99	48%	5%	69.76	55.93	0.559337536	0.80186	125.69
16	442.8	2.185	69%	6%	33.13	108.75	1.08749188	3.28213	141.88

Table 8.5: Calculated average values taken from the duplicate experimental data from the ore mini-column tests.

Fe1:1	Days	ORP	pH	% Cu Rec	% Iron ext.
	0		1	0.00%	0%
	1	480.0	1.1	1.34%	0%
	2	456.0	1.4	2.62%	3%
	3	452.0	1.5	3.04%	3%
	6	423.0	2.0	4.40%	4%
	8	425.7	2.2	4.91%	8%
	10	422.0	1.9	5.71%	10%
	14	416.0	2.0	6.47%	10%
	17	420.0	2.0	6.62%	12%
	20	426.0	2.0	6.79%	13%
	24	422.0	1.9	7.44%	15%
	26	424.0	2.0	7.47%	15%
	30	420.0	2.0	8.28%	17%
Fe 5:1	**Days**	**ORP**	**pH**	**% Cu Rec**	**% Iron ext.**
	0		1	0.00%	0%
	1	499.5	1.0	1.16%	1%
	2	475.5	1.325	2.19%	4%
	3	468	1.44	2.80%	3%
	6	439	1.81	3.70%	4%
	8	436.2	2.185	4.17%	8%
	10	435.5	1.72	4.54%	9%
	14	437	1.715	4.85%	9%
	17	436.5	1.605	5.25%	11%
	20	437.5	1.66	5.55%	13%
	24	433.5	1.62	6.16%	15%
	26	438	1.735	6.24%	15%
	30	435	1.705	7.03%	16%
Fe 5:1 NaCl	**Days**	**ORP**	**pH**	**% Cu Rec**	**% Iron ext.**
	0		1	0.00%	0%
	1	454.5	0.9	0.93%	6%
	2	419	1.285	1.54%	8%
	3	387.5	1.475	1.74%	12%
	6	340.5	2.505	2.44%	12%
	8	381.2	2.515	2.68%	17%
	10	366	2.505	3.30%	18%
	14	335	2.415	4.17%	21%
	17	355	2.375	4.58%	24%
	20	353.5	2.485	4.35%	25%
	24	351	2.275	4.93%	28%
	26	356.5	2.295	5.55%	27%
	30	350	2.23	5.92%	30%
Fe 5:1 Py	**Days**	**ORP**	**pH**	**% Cu Rec**	**% Iron ext.**
	0		1	0.00%	0%
	1	478.5	1.0	2.16%	3%
	2	448	1.365	3.92%	2%
	3	435.5	1.5	4.61%	5%
	6	398	2.06	4.91%	5%
	8	415.65	2.24	5.17%	10%
	10	408.5	1.815	5.79%	11%
	14	406.5	1.665	6.54%	12%
	17	406	1.475	6.94%	15%
	20	415	1.47	7.15%	15%
	24	404	1.52	7.80%	18%
	26	413.5	1.49	8.03%	18%
	30	408	1.445	8.55%	20%

The following Figures summarize the Iron analysis for the ore shake flask tests.

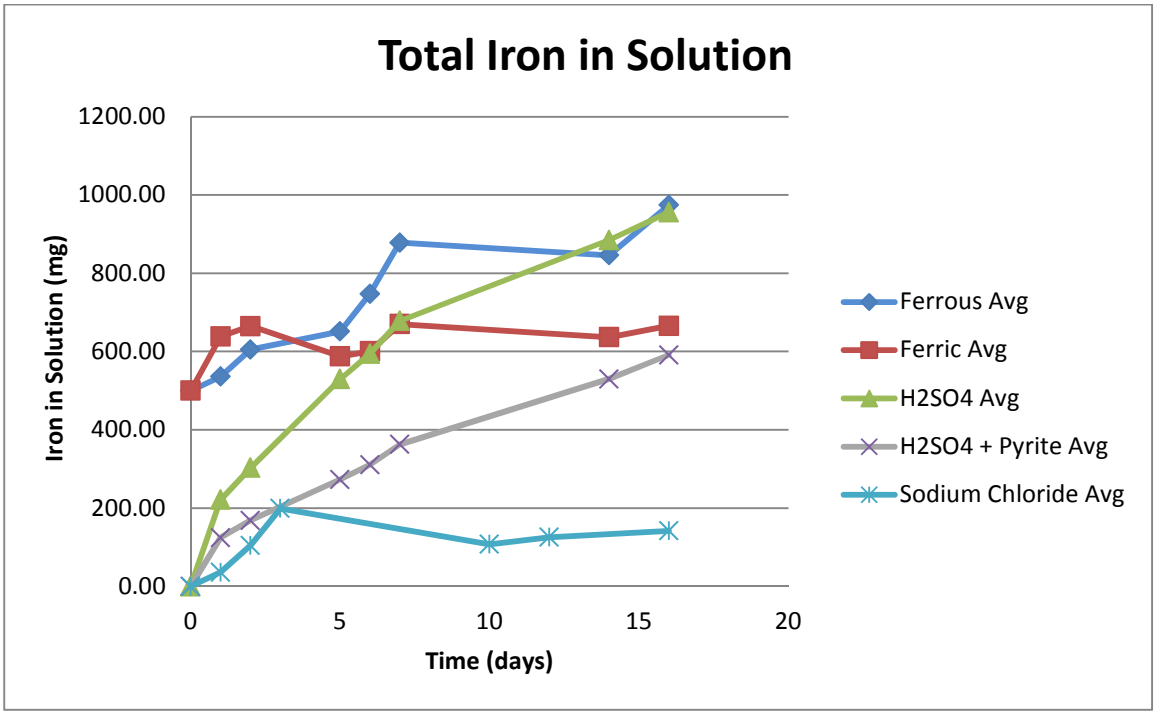

Figure 8.1: Total Iron in Solution for the Ore Shake Flask Tests.

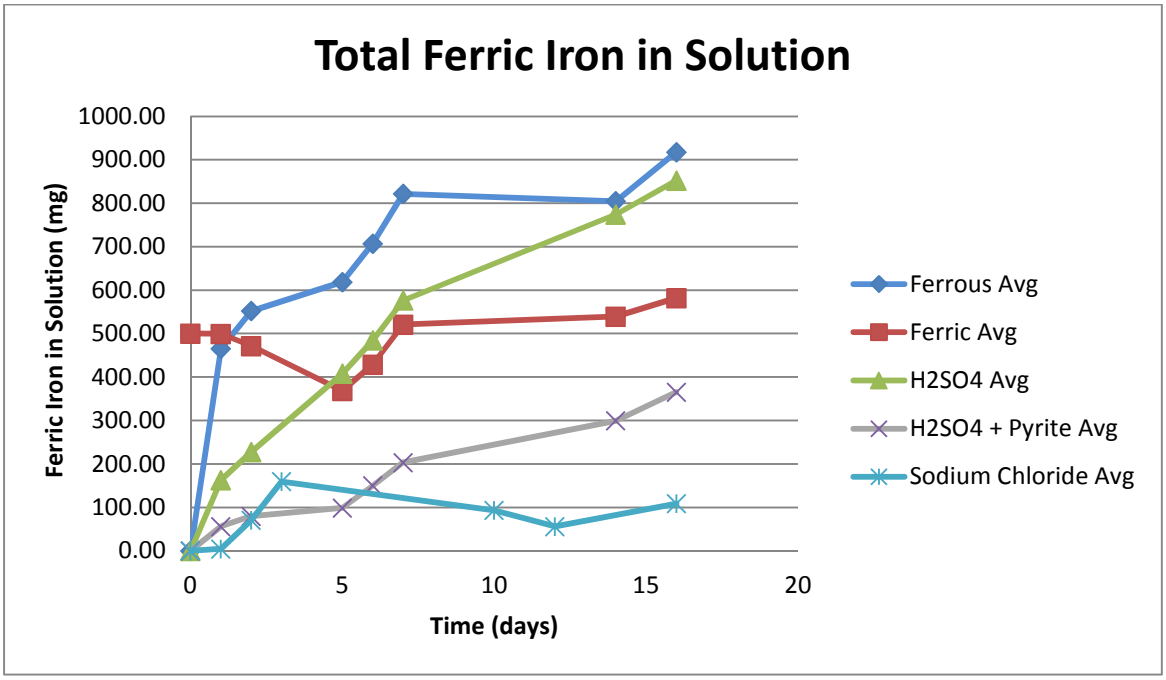

Figure 8.2: Total Ferric Iron in Solution for the Ore Shake Flask Tests.

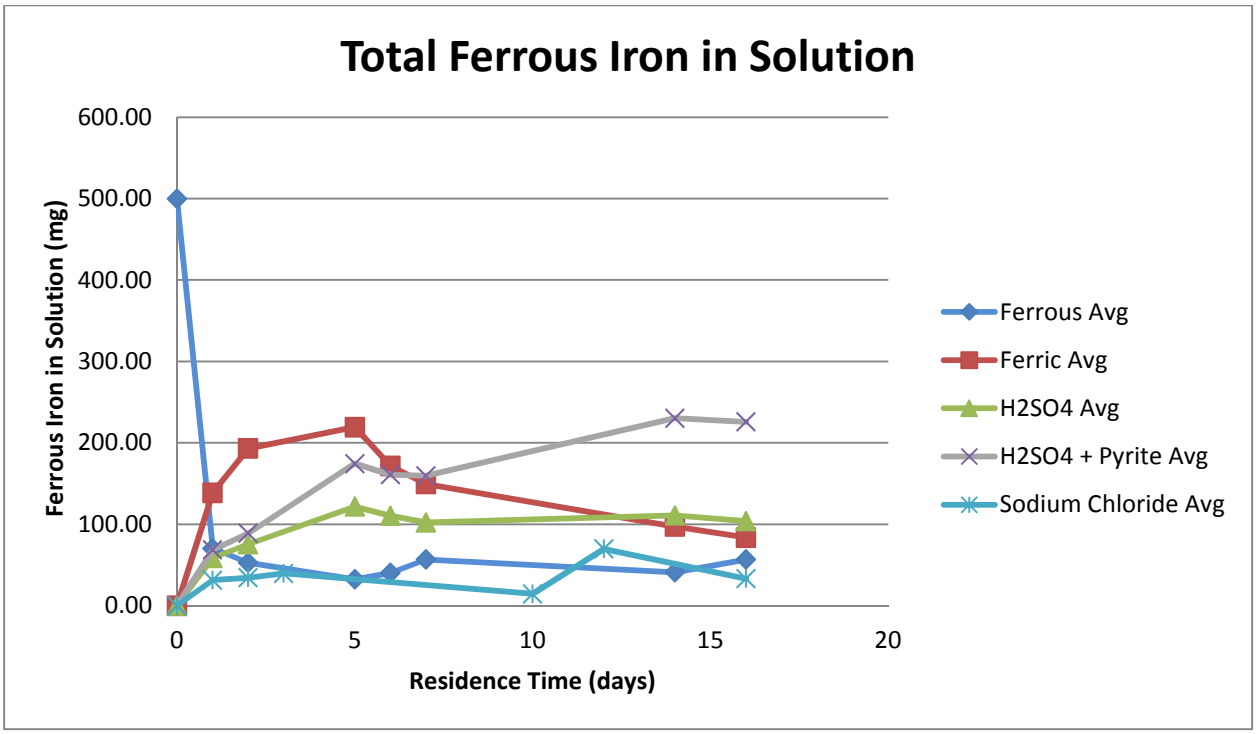

Figure 8.3: Total Ferrous Iron in Solution for the Ore Shake Flask Tests.

8.3 Heat Balance Assumptions

The assumptions used for the estimation of the steady state heat balance are summarized in Table 8.6, 8.7 and 8.8.

Table 8.6: Steady-State Heat Balance Fluid Flow Assumptions.

Solution flow Initial, L/m2/h	10	**0.016**	L/kg ore per day		
Solution flow Final, L/m2/h	5	**0.008**	L/kg ore per day		
Specific Heat of Solution	1.46	kj/kg soln			
Density of H2SO4	1.84	kg soln/L			
Total Solution Per Day Initial	441.6	kg soln/d			
Total Solution Per Day Final	220.8	kg soln/d			
Delta T Solution (298 to 333)	35	K			
Humid Air flow Initial, L/m^2/h	8	**0.013**	L/kg ore per day		
Humid Air flow Final L/m^2/h	4	**0.006**	L/kg ore per day		
Specific Heat of Sat. Air	1.639	kj/kgK			
Humidity Ratio	0.15	kg H2O / kg Air			
Density of Sat. Air	0.94	kg/L			
Total Sat. Air Per Day Initial	180.01	kg sat. air/d			
Total Sat. Air Per Day Final	90.00	kg sat. air/d			
Delta T Sat Air (373 to 333)	40	K			
Cp	1.44%	0.078	gmole/kg ore		
Py	2.9%	0.242	gmole/kg ore		
S	2.1%	0.156	gmole/kg ore		
	Initial	**Final**			
Cp reaction	1.0%	0.04%	%/d	1.44%	% Cp in ore
Py reaction rate	0.5%	0.02%	%/d	2.9%	% Py in ore
O2 Utilization	30%	15%			
Cp reacted	21.6	0.864	kg/d		
Py reacted	21.75	0.87	kg/d		
S Reacted to Sulphuric Acid	19.13	0.77	kg/d		
Heat of Reactions (Biotic)	12.12	0.48	Mj/d		
Heat out - solution	22.57	11.28	Mj/d		
Heat out - humid air	11.80	5.90	Mj/d		
Heat from Heat Exchanger	12.89	6.45	Mj/d		
Height	10	m			
Area	1	m^2			
kg Cp in 10x1x1 section	2160				
kg Py in 10x1x1 section	4350				
Heat Exchanger					
Assume Flow In = Flow Out					
Delta T Solution (333K to 323K)	10	K			

Table 8.7: Steady-State Heat Balance Operating Parameters.

Heap Section			
	Length	1	m
	Width	1	m
	Lift Height	10	m
	Ambient Temperature	298	K
	Top surface Leaching Area	1	m^2
	Total Leaching Surface Area	4.1	m^2
Ore			
	Density	1.5	t/m^3
	Cu Grade	0.5%	
	Cp	1.44%	0.079 gmole/kg ore
	Py	2.9%	0.242 gmole/kg ore
	Ore Size	0.01	m dia
	Cp of Ore	0.8	kJ/kgK
	Ore Tonnage	15000.00	t
	Initial/Required Temperature	333	K
	Estimate Thermal Conductivity of ore	3	W/m*oC
H2SO4 Solution			
	Irrigation Rate	10/5	L/h/m^2
	Mass of Solution per kg Ore	0.0108	kg H2SO4 / kg ore
	Temperature In	298	K
	Heat transfer co-eff of Liquid H2SO4	250	W/m^2*K
	Cp of Sulphuric Acid Solution	1.46538	kJ/kgK
	Assume Temperature Out	333	K
Air			
	Heat transfer co-eff of air	15	W/m^2*K
	Density of Air	1.225	kg/m^3
	% O2 in Dry Air	24%	
	Density of O2 in Dry Air	0.336	kg/m^3
	Total Air per year	35063.24	m^3/yr
	Total Air per year	2.34	m^3/kg ore
	Mass of Air per kg Ore	2.86	kg air/kg ore
	Flow Rate of O2	1.344	kg/m^2/h
	Cp of Air	1.012	kJ/kgK
	Moisture Carring Capacity of Air	0.02	kg H2O / kg Air
	Heat transfer co-eff of H2O (g)	15	W/m^2*K

Table 8.8: Specific Species Thermodynamic Variables.

Species	**H298**	**H(T-298) kJ/gmol/ K**	**H333**	**Mol wt (g/mol)**	p (g/cm^3)	kJ/gmol /K	kJ/gm ol/K
	kJ/g. Mol		**kJ/g.Mol**				
CuFeS2	190.4	0.09977	193.89	183.5		0.065	0.069
FeS2	171.5	0.06481	173.77	120		0.024	0.033
S	0	0.02349	0.82	32			
H2SO4 (100%)	814	0.14293	819.00	98		0.065	0.069
H2SO4 (aq)	889	0.12021	893.21			0.065	0.069
Dilute H2SO4				9.8	1.84		
FeSO4(aq)	998.5	0.15085	1003.78	152		0.065	0.069
Fe2(SO4)3(aq)	2,825	0.50716	2842.75	400		0.065	0.069
CuSO4(aq)	844.6	0.15551	850.04	159.5		0.075	0.055
H2O	285.8	0.07259	288.34	18		0.065	0.034
H2O (g)	241.8	0.03220	242.93	18		0.030	0.030
O2	0	0.02980	1.04	32		0.029	0.029
N2	0	0.02883	1.01	28			
Air				38.76			
Heat Req to Raise Ore T by 1 degree K @ 333K		**0.81**	**kJ/kg/K**				
Heat Req to Raise SiO2 T by 1 degree K @ 333K		**0.82**	kJ/kg/K				
Temperature	**pH2O, atm**						
293K	0.042						
333K	0.195						

8.4 XRD Analysis

8.4.1 XRD for Ore Shake Flask Test Residues

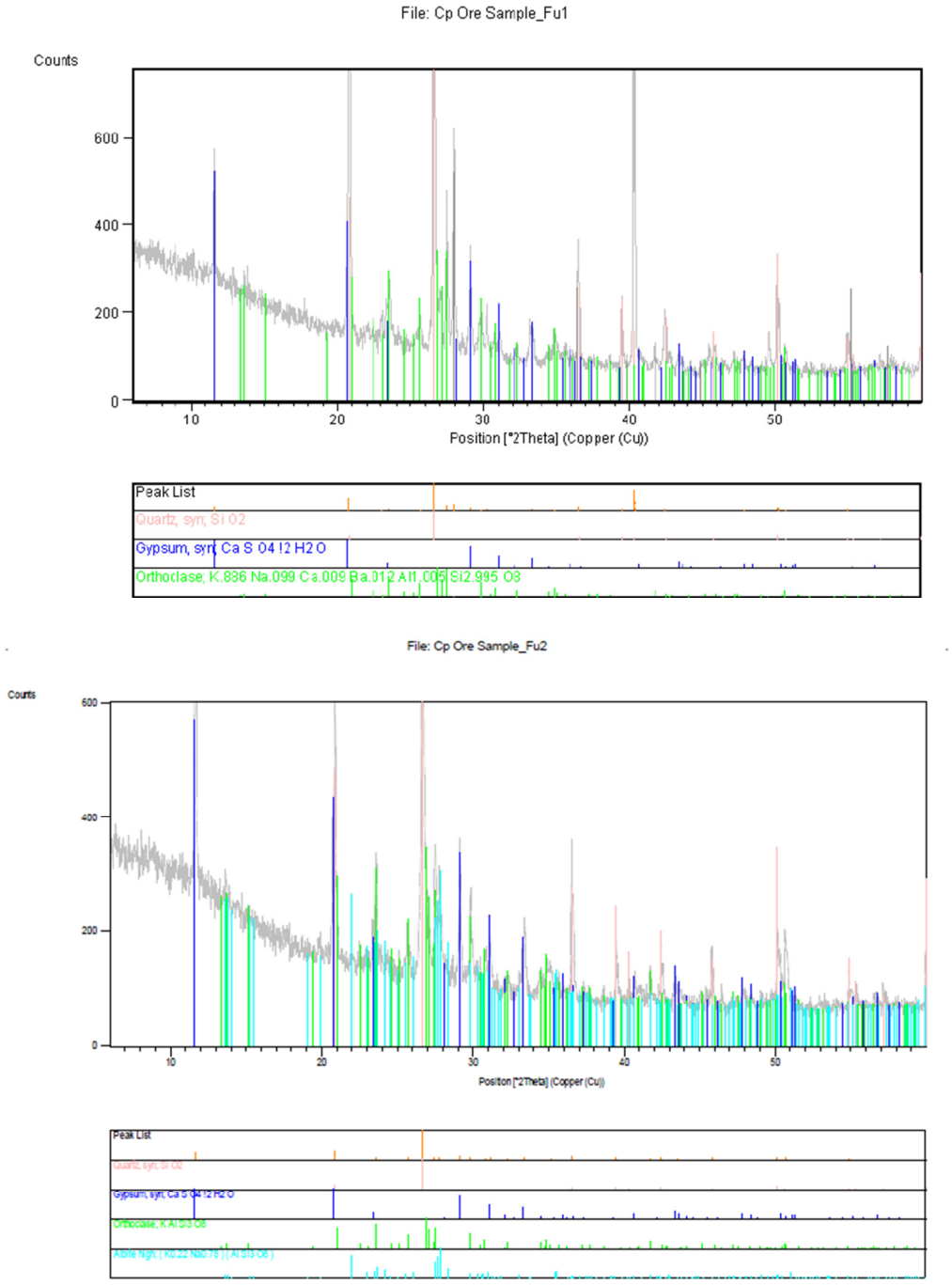

Figure 8.4: X-ray Diffraction Analysis of the Ore Shake Flask Ferrous Test Residues

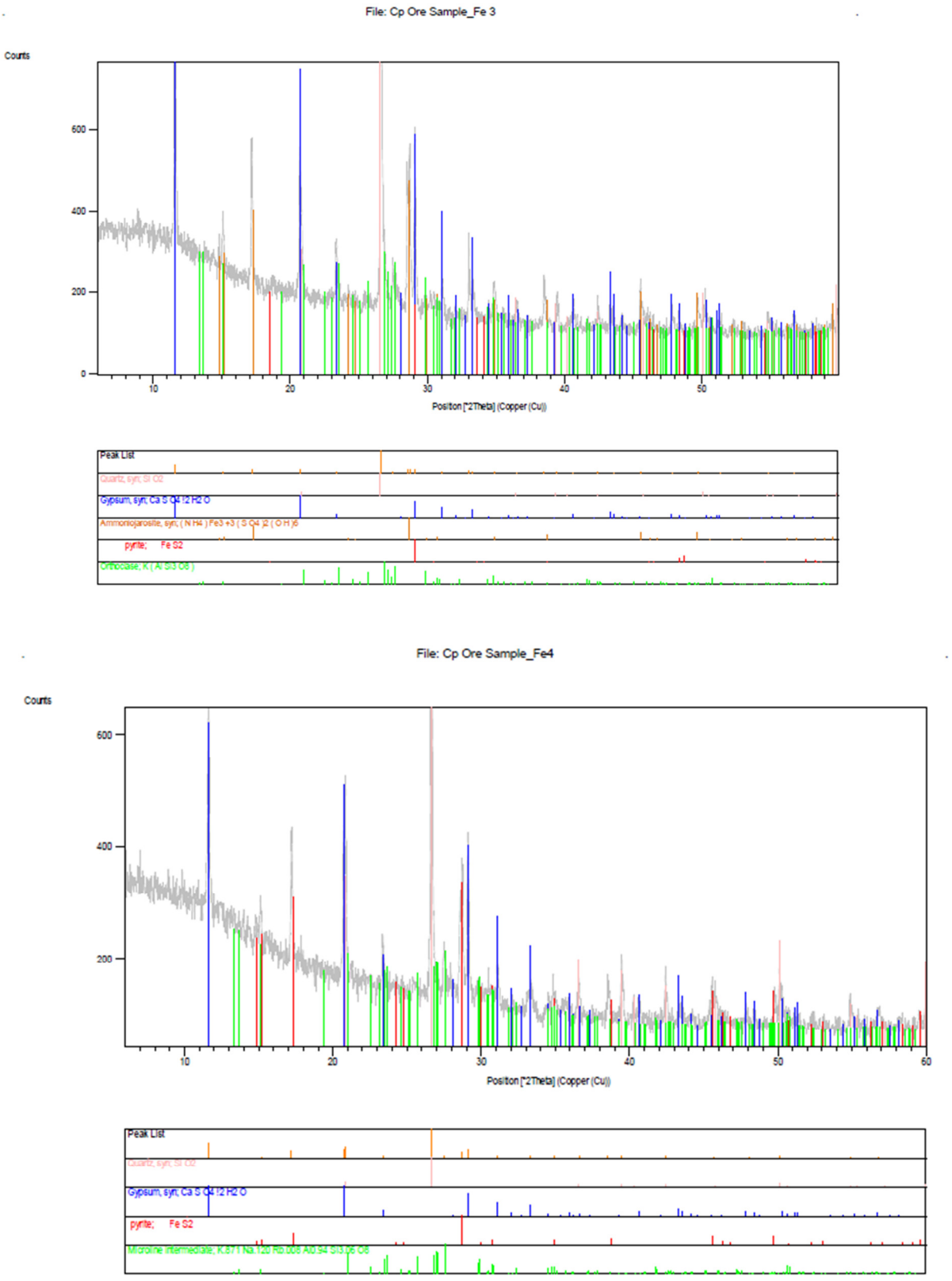

Figure 8.5: X-ray Diffraction Analysis of the Ore Shake Flask Ferric Test Residues

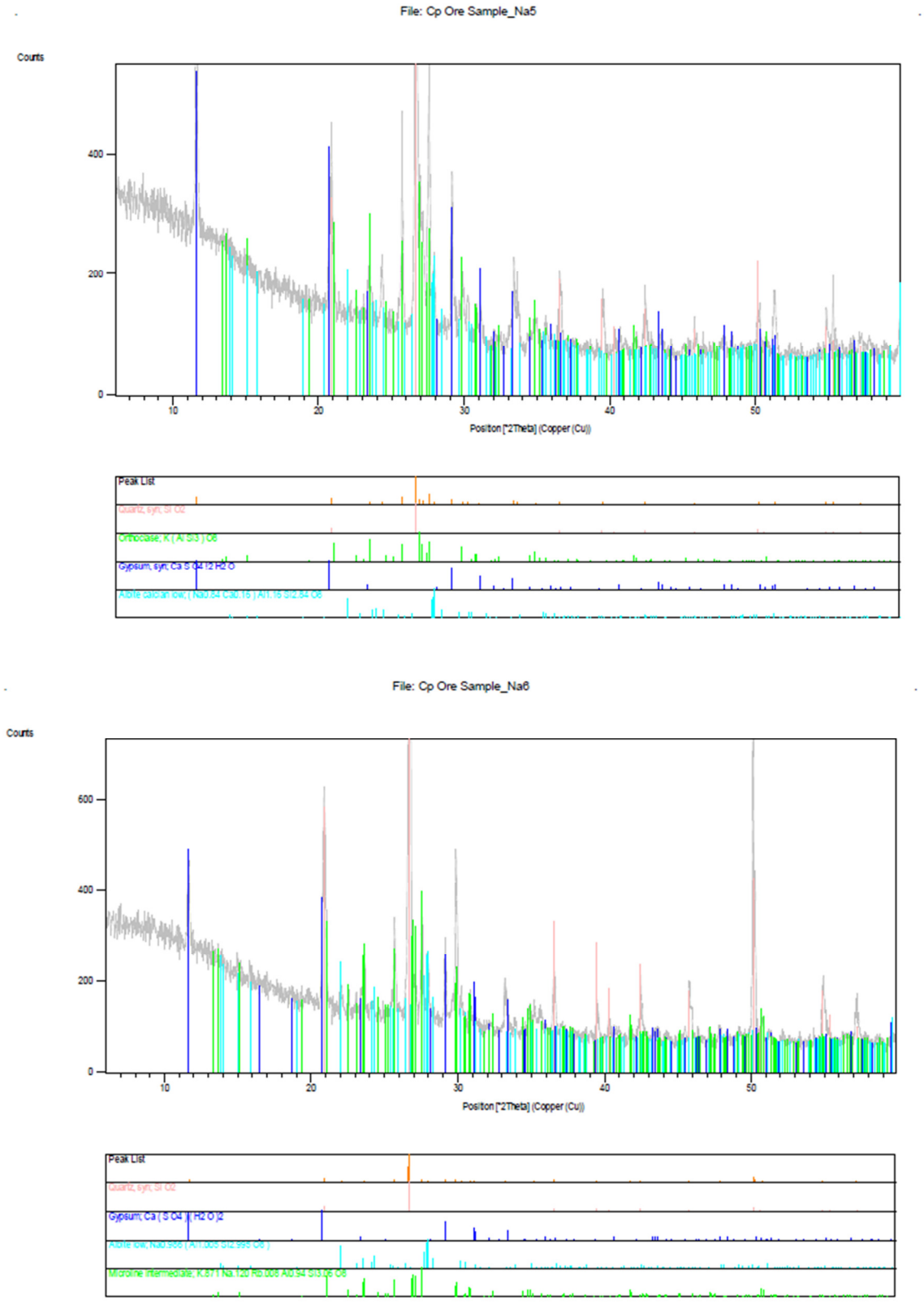

Figure 8.6: X-ray Diffraction Analysis of the Ore Shake Flask NaCl Test Residues

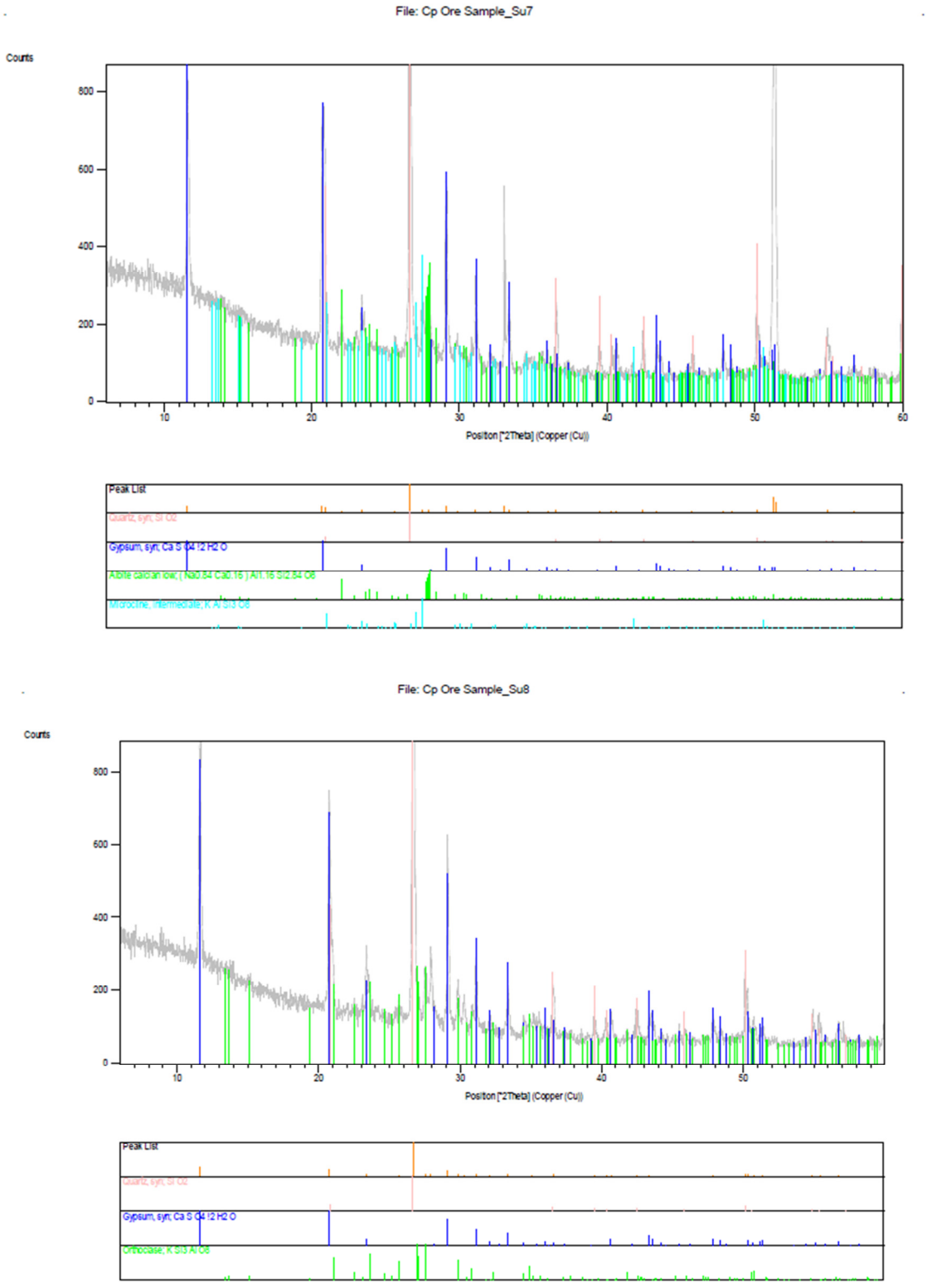

Figure 8.7: X-ray Diffraction Analysis of the Ore Shake Flask Sulphuric Acid Only Test Residues

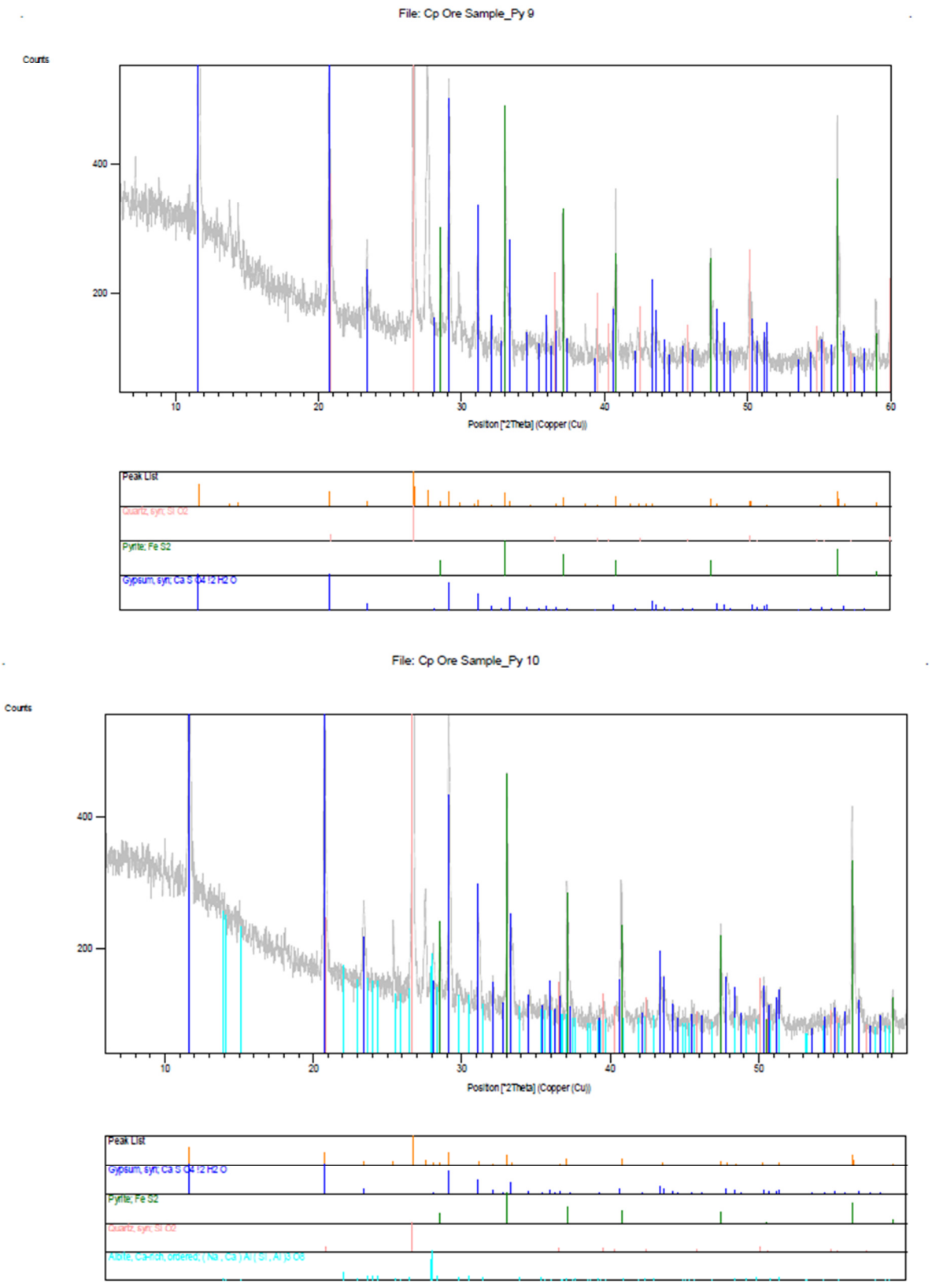

Figure 8.8: X-ray Diffraction Analysis of the Ore Shake Flask Pyrite Test Residues

8.4.2 XRD for Ore Mini-Column Test Residues

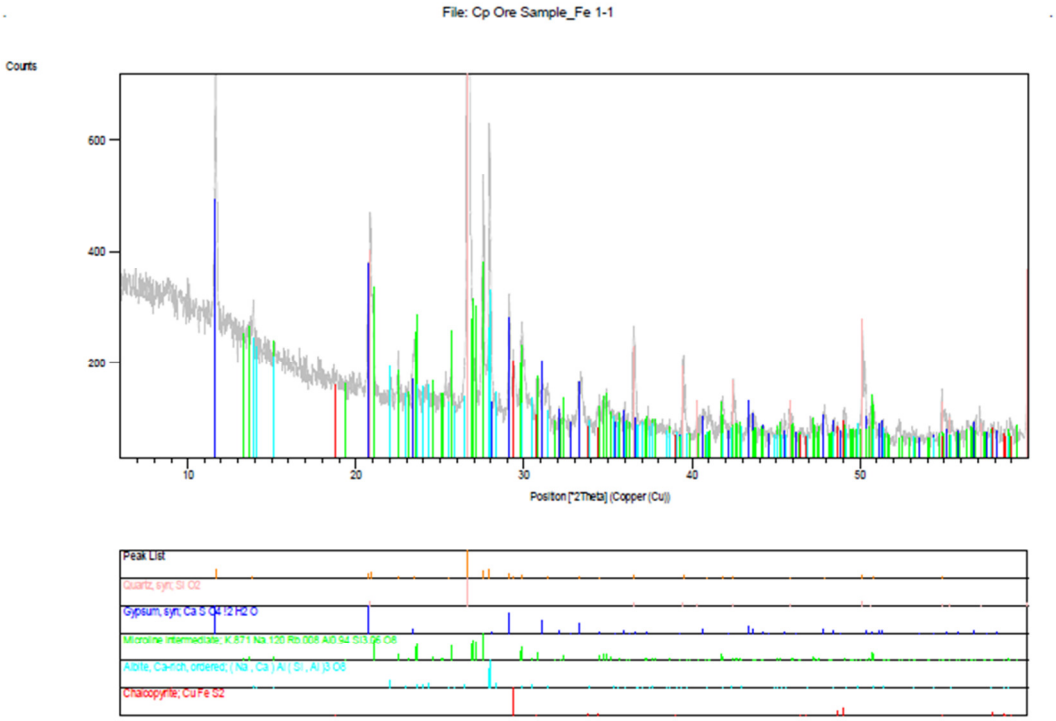

Figure 8.9: X-ray Diffraction Analysis of the Ore Mini-Column 1:1 Ferric to Ferrous Test Residue

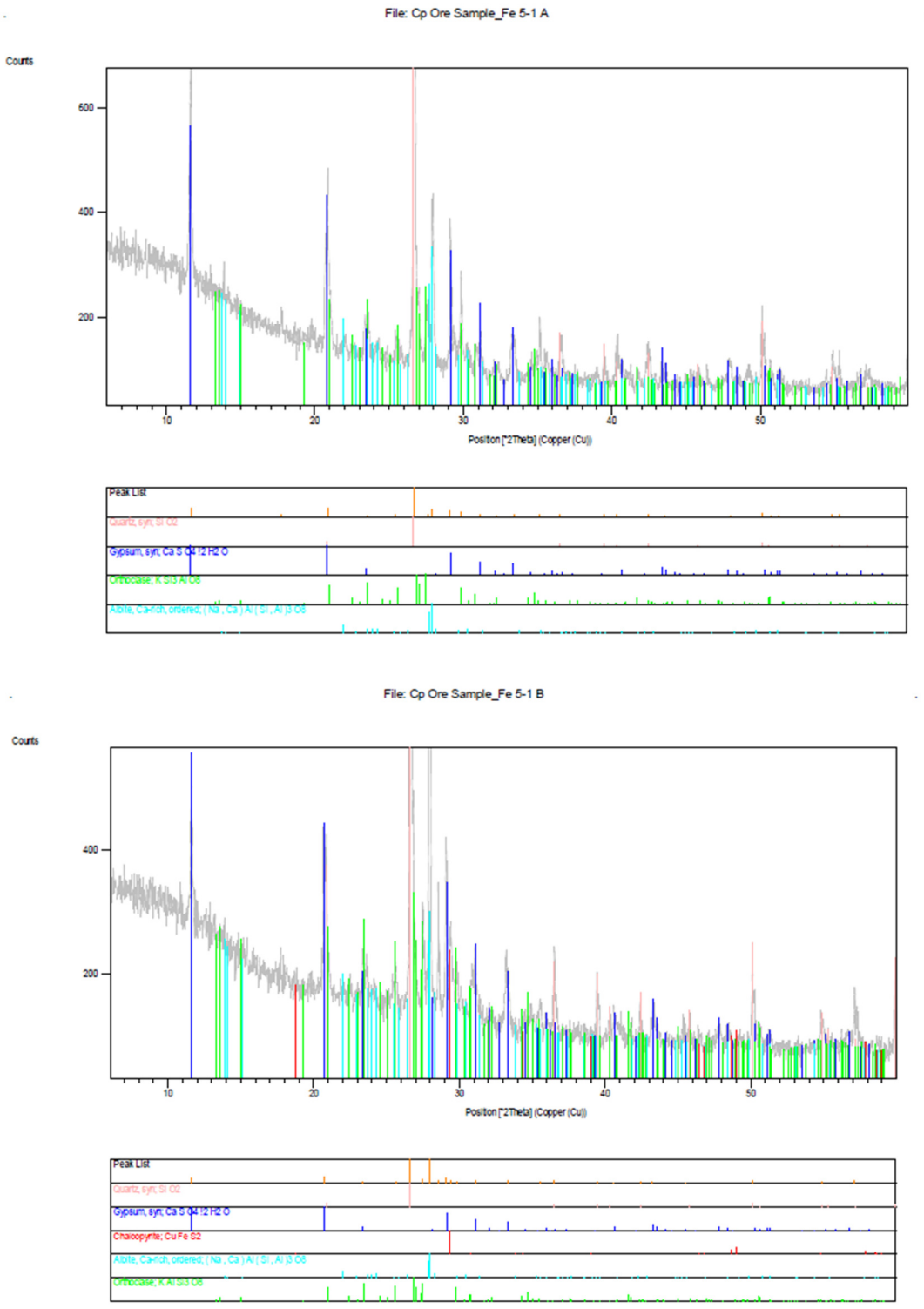

Figure 8.10: X-ray Diffraction Analysis of the Ore Mini-Column 5:1 Ferric to Ferrous Test Residues

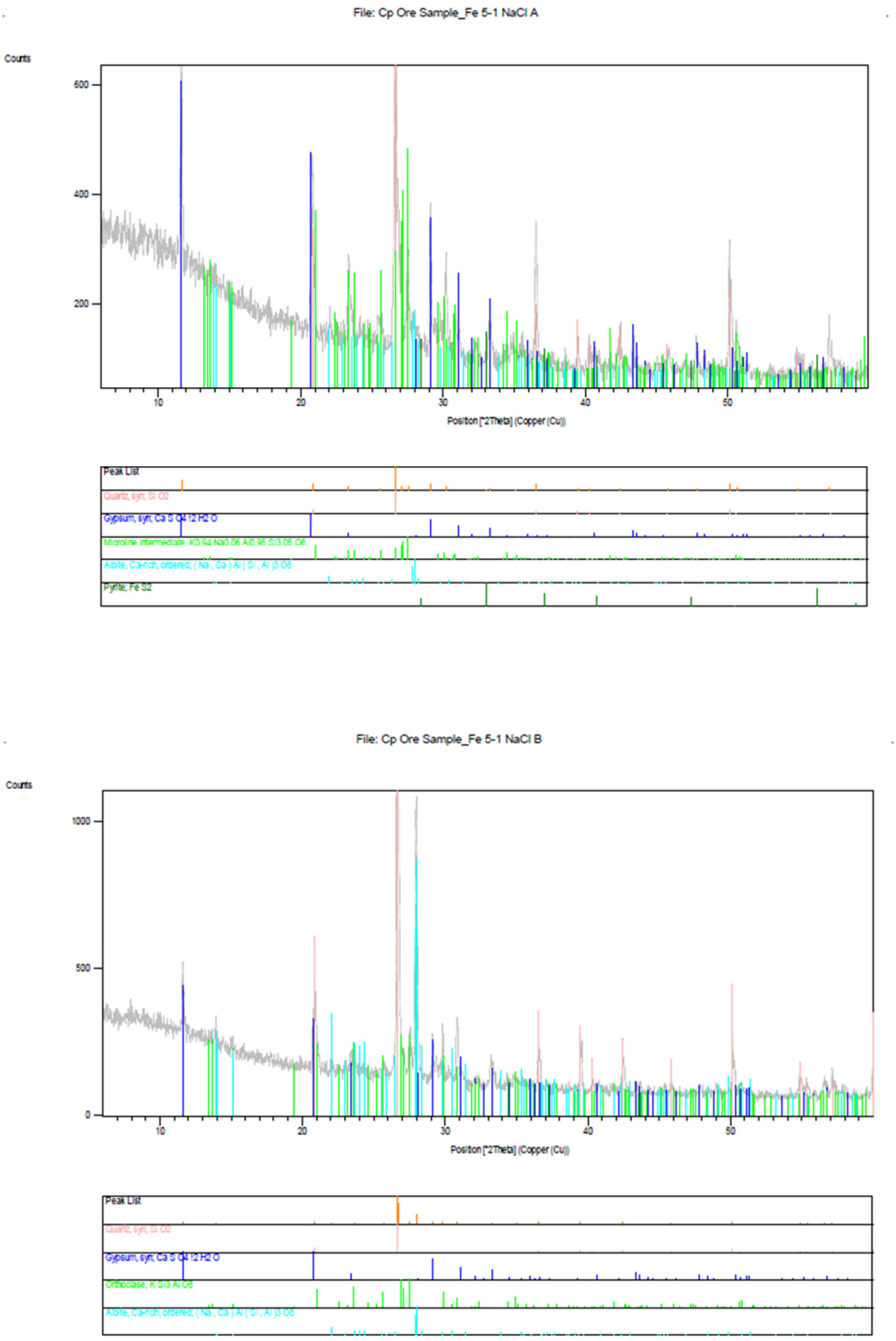

Figure 8.11: X-ray Diffraction Analysis of the Ore Mini-Column 1M NaCL with 5:1 Ferric to Ferrous Test Residues

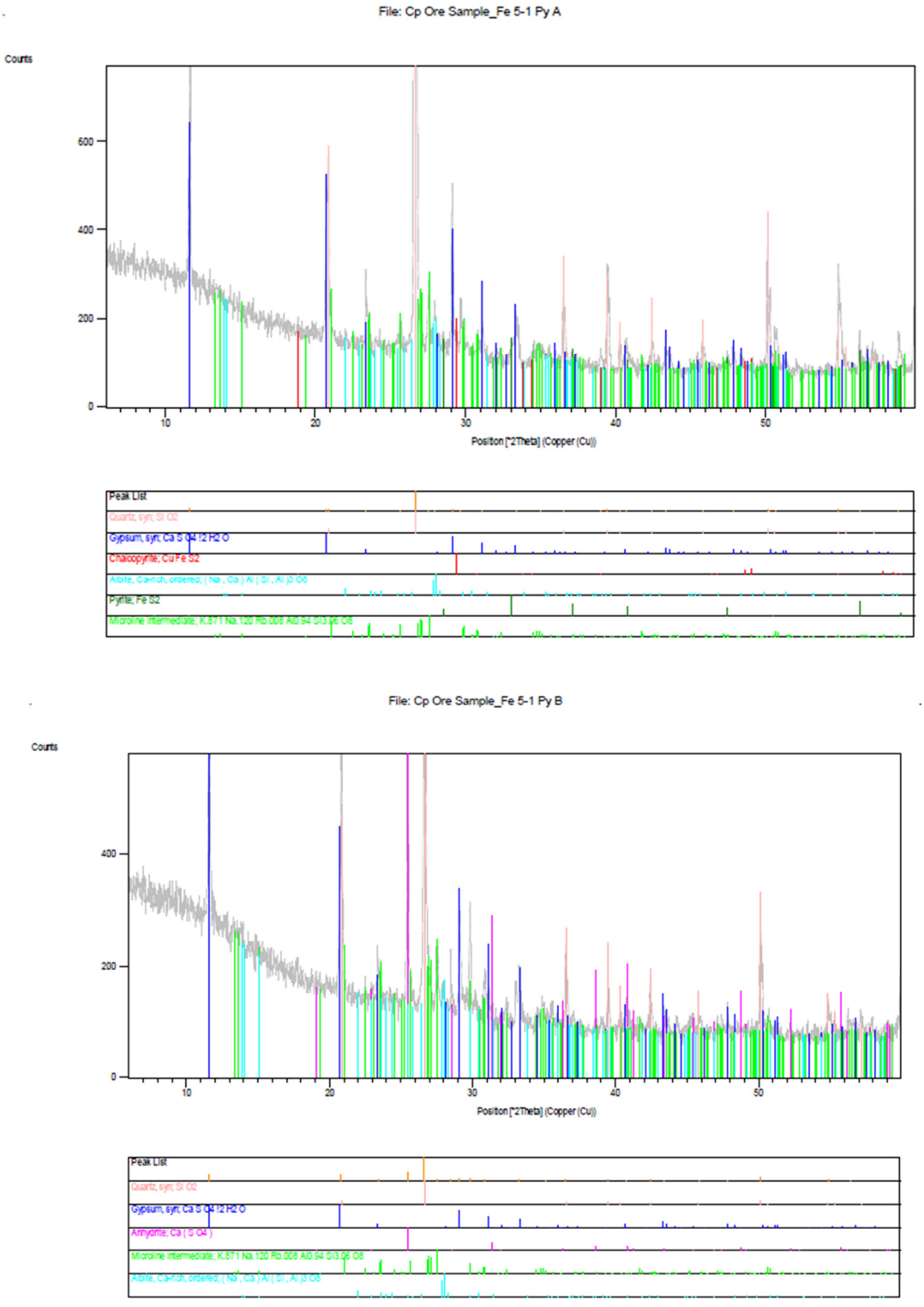

Figure 8.12: X-ray Diffraction Analysis of the Ore Mini-Column Pyrite 5:1 Ferric to Ferrous Test Residues

8.5 Economics Analysis Details

The assumptions made for the economic analysis of the low-grade chalcopyrite heap leach are given in Tables 8.9, 8.10, 8.11 and 8.12.

Table 8.9: Heap Leach Capital Expenditure Estimate.

CAPEX over a 3 yr Construction Period			**Capacity**	
Direct Costs			**Capacity**	
Crushing Plant	$	36,430,000.00	10	Millions t/y
Agglomeration & Stacking Equipement	$	35,270,000.00	10	Millions t/y
Pad Construction	$	96,180,000.00	2.2	Million m^2
Heap Leaching Circuit	$	62,860,000.00	5,500	m^3/h
SX Plant	$	85,210,000.00	2,740	m^3/h
EW Plant	$	-	**Existing**	
Acid Plant	$	91,220,000.00		
Oxygen Plant	$	20,000,000.00		
Direct Cost Break Down				
Equipment	$	170,868,000.00	40%	of Total Direct Costs
Materials	$	192,226,500.00	45%	of Total Direct Costs
Construction	$	64,075,500.00	15%	of Total Direct Costs
Total Direct Costs	$	427,170,000.00		
Field Costs				
Indirects	$	128,151,000.00	30%	of Direct Construction Costs
Total Field Costs	$	555,321,000.00		
Other Costs				
Engineering and Services	$	61,610,000.00	7.5%	of Field Costs
Contingency	$	175,070,000.00	20%	of Field Costs + Engineering
Total Other Costs	$	236,680,000.00		
Total CAPEX	$	792,001,000.00		

Table 8.10: Heap Leach Operating Conditions Estimate.

Conditions		
Long Term Cu price	2.25	$/lb
Average Grade	0.50%	Cu
Three year leaching cycle		
First Year Leached Ore Recovery	50%	Cu
Second Year Leached Ore Recovery	10%	Cu
Third Year Leached Ore Recovery	10%	Cu
Tonnes of Ore on Heap	28,571,429	t/yr
Lift Hei*ght of Heap*	10	m
Est. Ore Density	1.5	mt/m^3
Irrigation Top of Heap Area	2,380,952.42	m^2
leach pad availability for 2.2 Million m^2	9.24	years
Acid Flow Rate	10	L/hm^2
Dilute Acid Concentration	9.808	g/L
Dilute Acid Usage	203,428,574,480.00	L/year/ 10 million tonnes of ore
Concentrated Acid Usage	1,271,428,590.50	L/year/ 10 million tonnes of ore
Water Usage to Dilute Concentrated Acid	202,157,145,889.50	L/year/ 10 million tonnes of ore
Discount Rate	5%	
SX & EW Recovery	98%	
Tax Rate	15%	

Table 8.11: Heap Leach Operating Expenditure Estimate

OPEX			
Mined Ore	1.5$/t		
Waste Ore	1.5$/t		
Concentrated Acid (16 Mol)	50$/t		0.092$/L
Water	0.25$/m^3		0.00025$/L
G&A	0.25$/t		
SX & EW	0.13$/lb		SX = 0.03$
Power Cost	0.3$/kWh		
Energy Req For Soln Temp Increase to 365K From 298K	0.0055169	kj/L	From Thermo
Energy Req For Soln Temp Increase to 365K From 298K	0.00000153	kWh/L	
Acid Heating Cost	0.00000046	$/L	
Energy Req For Initial Ore Pre-Heat during Agglomeration to 333K From 298K	0.017283204	kj/kg ore	From Thermo
Energy Req For Initial Ore Pre-Heat during Agglomeration to 333K From 298K	0.00000480	kWh/kg ore	
Pre-Heat Cost	0.0014414	$/t	
Strip Ratio	1	Waste to Ore	
Ferric Addition Cost	0.00002	$/L	
Iron Bleed and Fixing Cost	0.00003	$/L	
Crushing	0.59$/t		
Agglomeration & Stacking	0.38$/t		
Other Heap Leaching Costs	0.28$/t		

Table 8.12:External Ferric and Neutralization Requirements

Assume all Fe in Cp is leached
Assume equivalent amount of Fe in Py is extracted
Assume Fe precipitated in heap ripios & ROM heap

$CuFeS_2 + 4Fe^{3+} = Cu^{2+} + 5Fe^{2+} + 2S$
$FeS_2 + 2Fe^{2+} = 3Fe^{2+} + 2S$

$Fe_2(SO_4)_3 + 3CaCO_3 + 3H_2O = 2Fe(OH)_3 + 3CaSO_4$

$2FeSO_4 + 0.5O_2 + H_2SO_4 = Fe_2(SO_4)_3 + H_2O$

Fe precipitated with $CaCO_3$, tpa	88,189.0
$CaCO_3$, tpa	291,630.2
% $CaCO_3$	90%
% $CaCO_3$ utilization	90%
Cost ground $CaCO_3$, US$/t	$25.0
$CaCO_3$ cost, US$/y	$7,290,755
O_2 requirements for ext Ferric	
Fe oxidized, tpa	529,133.9
O_2 requirements for ext Ferric, tpa	167,979
% O_2	90%
O_2 utilization	50%
O_2 costs, US$/t	$30.0
O_2 costs, US$/yr	$5,039,370

Table 8.13: Heap Leach Option Economic Cash flow Analysis

Year	0	1	2	3	4	5	6	7	8	9
Mined Ore (t)	-	-		28,571,429.00	28,571,429.00	28,571,429.00	28,571,429.00	28,571,429.00	28,571,429.00	28,571,429.00
First Year Leached Ore 50% Rec				28,571,429.00	28,571,429.00	28,571,429.00	28,571,429.00	28,571,429.00	28,571,429.00	28,571,429.00
Second Year Leached Ore 10% Rec			-		28,571,429.00	28,571,429.00	28,571,429.00	28,571,429.00	28,571,429.00	28,571,429.00
Third Year Leached Ore 10% Rec						28,571,429.00	28,571,429.00	28,571,429.00	28,571,429.00	28,571,429.00
CAPEX										
Total CAPEX	$264,000,333.33	$264,000,333.33	$264,000,333.33							
OPEX										
Mined Ore	$0.00	$0.00	$0.00	$42,857,143.50	$42,857,143.50	$42,857,143.50	$42,857,143.50	$42,857,143.50	$42,857,143.50	$42,857,143.50
Mined Waste	$0.00	$0.00	$0.00	$42,857,143.50	$42,857,143.50	$42,857,143.50	$42,857,143.50	$42,857,143.50	$42,857,143.50	$42,857,143.50
Crushing	$0.00	$0.00	$0.00	$17,142,857.40	$17,142,857.40	$17,142,857.40	$17,142,857.40	$17,142,857.40	$17,142,857.40	$17,142,857.40
Steam	$0.00	$0.00	$0.00	$52,285,715.07	$52,285,715.07	$52,285,715.07	$52,285,715.07	$52,285,715.07	$52,285,715.07	$52,285,715.07
Agglomeration & Stacking	$0.00	$0.00	$0.00	$21,428,571.75	$21,428,571.75	$21,428,571.75	$21,428,571.75	$21,428,571.75	$21,428,571.75	$21,428,571.75
Concentrated Acid Cost	$0.00	$0.00	$0.00	$21,428,571.75	$42,857,143.50	$64,285,715.25	$64,285,715.25	$64,285,715.25	$64,285,715.25	$64,285,715.25
Other Heap Leaching Costs	$0.00	$0.00	$0.00	$4,285,714.35	$8,571,428.70	$12,857,143.05	$12,857,143.05	$12,857,143.05	$12,857,143.05	$12,857,143.05
SX & EW	$0.00	$0.00	$0.00	$25,872,000.39	$31,046,400.47	$36,220,800.54	$36,220,800.54	$36,220,800.54	$36,220,800.54	$36,220,800.54
Ferric Generation	$0.00	$0.00	$0.00	$5,714,285.80	$5,714,285.80	$5,714,285.80	$5,714,285.80	$5,714,285.80	$5,714,285.80	$5,714,285.80
Iron Bleed and Fixing	$0.00	$0.00	$0.00	$22,857,143.20	$22,857,143.20	$22,857,143.20	$22,857,143.20	$22,857,143.20	$22,857,143.20	$22,857,143.20
G&A Cost	$0.00	$0.00	$0.00	$14,285,714.50	$14,285,714.50	$14,285,714.50	$14,285,714.50	$14,285,714.50	$14,285,714.50	$14,285,714.50
Total OPEX	$0.00	$0.00	$0.00	$271,014,861.21	$301,903,547.39	$332,792,233.56	$332,792,233.56	$332,792,233.56	$332,792,233.56	$332,792,233.56
Cashflow										
Copper Produced (lb)	0.00	0.00	0.00	156,800,002.35	188,160,002.82	219,520,003.29	219,520,003.29	219,520,003.29	219,520,003.29	219,520,003.29
Cu Production Cost ($/lb)				$1.73	$1.60	$1.52	$1.52	$1.52	$1.52	$1.52
Cu Revenues	$0.00	$0.00	$0.00	$352,800,005.29	$423,360,006.35	$493,920,007.41	$493,920,007.41	$493,920,007.41	$493,920,007.41	$493,920,007.41
CAPEX	$264,000,333.33	$264,000,333.33	$264,000,333.33	$0.00	$0.00	$0.00	$0.00	$0.00	$0.00	$0.00
OPEX	$0.00	$0.00	$0.00	$271,014,861.21	$301,903,547.39	$332,792,233.56	$332,792,233.56	$332,792,233.56	$332,792,233.56	$332,792,233.56
Cashflow Before Tax	$264,000,333.33	$264,000,333.33	$264,000,333.33	$81,785,144.08	$121,456,458.96	$161,127,773.85	$161,127,773.85	$161,127,773.85	$161,127,773.85	$161,127,773.85
NPV Factor	0.95	0.91	0.86	0.82	0.78	0.75	0.7:	0.68	0.64	0.61
DCFBT	$251,428,888.89	$239,456,084.66	$228,033,413.96	$67,284,840.44	$95,164,313.69	$120,236,025.67	$114,510,500.64	$109,057,619.65	$103,864,399.67	$98,918,475.88
Cumulative DCFBT	$251,428,888.89	$490,884,973.54	$718,918,387.50	$551,653,547.06	$556,489,233.38	$436,253,207.71	$321,742,707.07	$212,685,087.42	$108,820,687.75	$9,902,211.87

10	11	12	13	14	15	16	17	18	19	20	21	22	23	24	25
28,571,429.00	28,571,429.00	28,571,429.00	28,571,429.00	28,571,429.00	28,571,429.00	28,571,429.00	28,571,429.00	28,571,429.00	28,571,429.00	28,571,429.00	28,571,429.00	28,571,429.00			
28,571,429.00	28,571,429.00	28,571,429.00	28,571,429.00	28,571,429.00	28,571,429.00	28,571,429.00	28,571,429.00	28,571,429.00	28,571,429.00	28,571,429.00	28,571,429.00	28,571,429.00	28,571,429.00		-
28,571,429.00	28,571,429.00	28,571,429.00	28,571,429.00	28,571,429.00	28,571,429.00	28,571,429.00	28,571,429.00	28,571,429.00	28,571,429.00	28,571,429.00	28,571,429.00	28,571,429.00	28,571,429.00	28,571,429.00	-
28,571,429.00	28,571,429.00	28,571,429.00	28,571,429.00	28,571,429.00	28,571,429.00	28,571,429.00	28,571,429.00	28,571,429.00	28,571,429.00	28,571,429.00	28,571,429.00	28,571,429.00	28,571,429.00	28,571,429.00	28,571,429.00

$42,857,143.50	$42,857,143.50	$42,857,143.50	$42,857,143.50	$42,857,143.50	$42,857,143.50	$42,857,143.50	$42,857,143.50	$42,857,143.50	$42,857,143.50	$42,857,143.50	$42,857,143.50	$42,857,143.50	$0.00	$0.00	$0.00
$42,857,143.50	$42,857,143.50	$42,857,143.50	$42,857,143.50	$42,857,143.50	$42,857,143.50	$42,857,143.50	$42,857,143.50	$42,857,143.50	$42,857,143.50	$42,857,143.50	$42,857,143.50	$42,857,143.50	$0.00	$0.00	$0.00
$17,142,857.40	$17,142,857.40	$17,142,857.40	$17,142,857.40	$17,142,857.40	$17,142,857.40	$17,142,857.40	$17,142,857.40	$17,142,857.40	$17,142,857.40	$17,142,857.40	$17,142,857.40	$17,142,857.40	$0.00	$0.00	$0.00
$52,285,715.07	$52,285,715.07	$52,285,715.07	$52,285,715.07	$52,285,715.07	$52,285,715.07	$52,285,715.07	$52,285,715.07	$52,285,715.07	$52,285,715.07	$52,285,715.07	$52,285,715.07	$52,285,715.07	$0.00	$0.00	$0.00
$21,428,571.75	$21,428,571.75	$21,428,571.75	$21,428,571.75	$21,428,571.75	$21,428,571.75	$21,428,571.75	$21,428,571.75	$21,428,571.75	$21,428,571.75	$21,428,571.75	$21,428,571.75	$21,428,571.75	$0.00	$0.00	$0.00
$64,285,715.25	$64,285,715.25	$64,285,715.25	$64,285,715.25	$64,285,715.25	$64,285,715.25	$64,285,715.25	$64,285,715.25	$64,285,715.25	$64,285,715.25	$64,285,715.25	$64,285,715.25	$64,285,715.25	$64,285,715.25	$42,857,143.50	$21,428,571.75
$12,857,143.05	$12,857,143.05	$12,857,143.05	$12,857,143.05	$12,857,143.05	$12,857,143.05	$12,857,143.05	$12,857,143.05	$12,857,143.05	$12,857,143.05	$12,857,143.05	$12,857,143.05	$12,857,143.05	$12,857,143.05	$8,571,428.70	$4,285,714.35
$36,220,800.54	$36,220,800.54	$36,220,800.54	$36,220,800.54	$36,220,800.54	$36,220,800.54	$36,220,800.54	$36,220,800.54	$36,220,800.54	$36,220,800.54	$36,220,800.54	$36,220,800.54	$36,220,800.54	$36,220,800.54	$10,348,800.16	$5,174,400.08
$5,714,285.80	$5,714,285.80	$5,714,285.80	$5,714,285.80	$5,714,285.80	$5,714,285.80	$5,714,285.80	$5,714,285.80	$5,714,285.80	$5,714,285.80	$5,714,285.80	$5,714,285.80	$5,714,285.80	$0.00	$0.00	$0.00
$22,857,143.20	$22,857,143.20	$22,857,143.20	$22,857,143.20	$22,857,143.20	$22,857,143.20	$22,857,143.20	$22,857,143.20	$22,857,143.20	$22,857,143.20	$22,857,143.20	$22,857,143.20	$22,857,143.20	$0.00	$0.00	$0.00
$14,285,714.50	$14,285,714.50	$14,285,714.50	$14,285,714.50	$14,285,714.50	$14,285,714.50	$14,285,714.50	$14,285,714.50	$14,285,714.50	$14,285,714.50	$14,285,714.50	$14,285,714.50	$14,285,714.50	$7,142,857.25	$3,571,428.63	$1,785,714.31
$332,792,233.56	$332,792,233.56	$332,792,233.56	$332,792,233.56	$332,792,233.56	$332,792,233.56	$332,792,233.56	$332,792,233.56	$332,792,233.56	$332,792,233.56	$332,792,233.56	$332,792,233.56	$332,792,233.56	$120,506,516.09	$65,348,800.98	$32,674,400.49

219,520,003.29	219,520,003.29	219,520,003.29	219,520,003.29	219,520,003.29	219,520,003.29	219,520,003.29	219,520,003.29	219,520,003.29	219,520,003.29	219,520,003.29	219,520,003.29	219,520,003.29	219,520,003.29	62,720,000.94	31,360,000.47
$1.52	$1.52	$1.52	$1.52	$1.52	$1.52	$1.52	$1.52	$1.52	$1.52	$1.52	$1.52	$1.52	$0.55	$1.04	$1.04
$493,920,007.41	$493,920,007.41	$493,920,007.41	$493,920,007.41	$493,920,007.41	$493,920,007.41	$493,920,007.41	$493,920,007.41	$493,920,007.41	$493,920,007.41	$493,920,007.41	$493,920,007.41	$493,920,007.41	$493,920,007.41	$141,120,002.12	$70,560,001.06
$0.00	$0.00	$0.00	$0.00	$0.00	$0.00	$0.00	$0.00	$0.00	$0.00	$0.00	$0.00	$0.00	$0.00	$0.00	$0.00
$332,792,233.56	$332,792,233.56	$332,792,233.56	$332,792,233.56	$332,792,233.56	$332,792,233.56	$332,792,233.56	$332,792,233.56	$332,792,233.56	$332,792,233.56	$332,792,233.56	$332,792,233.56	$332,792,233.56	$120,506,516.09	$65,348,800.98	$32,674,400.49
$161,127,773.85	$161,127,773.85	$161,127,773.85	$161,127,773.85	$161,127,773.85	$161,127,773.85	$161,127,773.85	$161,127,773.85	$161,127,773.85	$161,127,773.85	$161,127,773.85	$161,127,773.85	$161,127,773.85	$373,413,491.32	$75,771,201.14	$37,885,600.57
0.58	0.56	0.53	0.51	0.48	0.46	0.44	0.42	0.40	0.38	0.36	0.34	0.33	0.31	0.30	0.28
$94,208,072.26	$89,721,973.58	$85,449,498.65	$81,380,474.91	$77,505,214.20	$73,814,489.71	$70,299,514.01	$66,951,918.11	$63,763,731.53	$60,727,363.36	$57,835,584.15	$55,081,508.72	$52,458,579.73	$115,783,540.92	$22,375,445.71	$10,654,974.15
$84,305,860.40	$174,027,833.98	$259,477,332.63	$340,857,807.54	$418,363,021.74	$492,177,511.45	$562,477,025.46	$629,428,943.56	$693,192,675.09	$753,920,038.45	$811,755,622.61	$866,837,131.33	$919,295,711.06	$1,035,079,251.98	$1,057,454,697.69	$1,068,109,671.84

Table 8.14: Mill Option Economic Cash flow Analysis

Year	0	1	2	3	4	5	6	7	8	9	10	11	12
Mined Ore (t)	-	-		28,571,428.57	28,571,428.57	28,571,428.57	28,571,428.57	28,571,428.57	28,571,428.57	28,571,428.57	28,571,428.57	28,571,428.57	28,571,428.57
CAPEX													
Total CAPEX	$296,247,705.00	$296,247,705.00	$296,247,705.00										
OPEX													
Mined Ore	$0.00	$0.00	$0.00	$42,857,142.86	$42,857,142.86	$42,857,142.86	$42,857,142.86	$42,857,142.86	$42,857,142.86	$42,857,142.86	$42,857,142.86	$42,857,142.86	$42,857,142.86
Mined Waste	$0.00	$0.00	$0.00	$42,857,142.86	$42,857,142.86	$42,857,142.86	$42,857,142.86	$42,857,142.86	$42,857,142.86	$42,857,142.86	$42,857,142.86	$42,857,142.86	$42,857,142.86
Mill operating costs	$0.00	$0.00	$0.00	$142,857,142.86	$142,857,142.86	$142,857,142.86	$142,857,142.86	$142,857,142.86	$142,857,142.86	$142,857,142.86	$142,857,142.86	$142,857,142.86	$142,857,142.86
Conc. freight	$0.00	$0.00	$0.00	$25,714,285.71	$25,714,285.71	$25,714,285.71	$25,714,285.71	$25,714,285.71	$25,714,285.71	$25,714,285.71	$25,714,285.71	$25,714,285.71	$25,714,285.71
TC	$0.00	$0.00	$0.00	$34,285,714.29	$34,285,714.29	$34,285,714.29	$34,285,714.29	$34,285,714.29	$34,285,714.29	$34,285,714.29	$34,285,714.29	$34,285,714.29	$34,285,714.29
RC, US$/lb payable Cu	$0.00	$0.00	$0.00	$23,040,000.00	$23,040,000.00	$23,040,000.00	$23,040,000.00	$23,040,000.00	$23,040,000.00	$23,040,000.00	$23,040,000.00	$23,040,000.00	$23,040,000.00
G&A	$0.00	$0.00	$0.00	$7,142,857.14	$7,142,857.14	$7,142,857.14	$7,142,857.14	$7,142,857.14	$7,142,857.14	$7,142,857.14	$7,142,857.14	$7,142,857.14	$7,142,857.14
Cashflow													
Copper Produced (lb)	0.00	0.00	0.00	194,400,000.00	259,200,000.00	259,200,000.00	259,200,000.00	259,200,000.00	259,200,000.00	259,200,000.00	259,200,000.00	259,200,000.00	259,200,000.00
Cu Production Cost ($/lb)				$1.64	$1.23	$1.23	$1.23	$1.23	$1.23	$1.23	$1.23	$1.23	$1.23
Cu Revenues	$0.00	$0.00	$0.00	$437,400,000.00	$583,200,000.00	$583,200,000.00	$583,200,000.00	$583,200,000.00	$583,200,000.00	$583,200,000.00	$583,200,000.00	$583,200,000.00	$583,200,000.00
CAPEX	$296,247,705.00	$296,247,705.00	$296,247,705.00	$0.00	$0.00	$0.00	$0.00	$0.00	$0.00	$0.00	$0.00	$0.00	$0.00
OPEX	$0.00	$0.00	$0.00	$318,754,285.71	$318,754,285.71	$318,754,285.71	$318,754,285.71	$318,754,285.71	$318,754,285.71	$318,754,285.71	$318,754,285.71	$318,754,285.71	$318,754,285.71
Cashflow Before Tax	$296,247,705.00	$296,247,705.00	$296,247,705.00	$118,645,714.29	$264,445,714.29	$264,445,714.29	$264,445,714.29	$264,445,714.29	$264,445,714.29	$264,445,714.29	$264,445,714.29	$264,445,714.29	$264,445,714.29
NPV Factor	0.95	0.91	0.86	0.82	0.78	0.75	0.71	0.68	0.64	0.61	0.58	0.56	0.53
DCFBT	$282,140,671.43	$268,705,401.36	$255,909,906.06	$97,610,122.77	$207,200,136.75	$197,333,463.57	$187,936,831.98	$178,987,268.55	$170,464,065.28	$162,346,728.84	$154,615,932.23	$147,253,268.79	$140,241,208.37
Cumulative DCFBT	$282,140,671.43	$550,846,072.79	$806,755,978.85	$709,145,856.08	$501,945,719.33	$304,612,255.75	$116,675,423.78	$62,311,644.77	$232,775,710.06	$395,122,438.90	$549,738,371.13	$696,991,639.92	$837,232,848.29
Taxes	$0.00	$0.00	$0.00	$17,796,857.14	$39,666,857.14	$39,666,857.14	$39,666,857.14	$39,666,857.14	$39,666,857.14	$39,666,857.14	$39,666,857.14	$39,666,857.14	$39,666,857.14
Cashflow After Tax	$296,247,705.00	$296,247,705.00	$296,247,705.00	$100,848,857.14	$224,778,857.14	$224,778,857.14	$224,778,857.14	$224,778,857.14	$224,778,857.14	$224,778,857.14	$224,778,857.14	$224,778,857.14	$224,778,857.14
Cumulative CFAT	$296,247,705.00	$592,495,410.00	$888,743,115.00	$787,894,257.86	$563,115,400.71	$338,336,543.57	$113,557,686.43	$111,221,170.71	$336,000,027.86	$560,778,885.00	$785,557,742.14	$1,010,336,599.29	$1,235,115,456.43
Payback Period (Yrs)	-	-	-	-	-	4.51 yrs							

13	14	15	16	17	18	19	20	21	22
28,571,428.57	28,571,428.57	28,571,428.57	28,571,428.57	28,571,428.57	28,571,428.57	28,571,428.57	28,571,428.57	28,571,428.57	28,571,428.57
$42,857,142.86	$42,857,142.86	$42,857,142.86	$42,857,142.86	$42,857,142.86	$42,857,142.86	$42,857,142.86	$42,857,142.86	$42,857,142.86	$42,857,142.86
$42,857,142.86	$42,857,142.86	$42,857,142.86	$42,857,142.86	$42,857,142.86	$42,857,142.86	$42,857,142.86	$42,857,142.86	$42,857,142.86	$42,857,142.86
$142,857,142.86	$142,857,142.86	$142,857,142.86	$142,857,142.86	$142,857,142.86	$142,857,142.86	$142,857,142.86	$142,857,142.86	$142,857,142.86	$142,857,142.86
$25,714,285.71	$25,714,285.71	$25,714,285.71	$25,714,285.71	$25,714,285.71	$25,714,285.71	$25,714,285.71	$25,714,285.71	$25,714,285.71	$25,714,285.71
$34,285,714.29	$34,285,714.29	$34,285,714.29	$34,285,714.29	$34,285,714.29	$34,285,714.29	$34,285,714.29	$34,285,714.29	$34,285,714.29	$34,285,714.29
$23,040,000.00	$23,040,000.00	$23,040,000.00	$23,040,000.00	$23,040,000.00	$23,040,000.00	$23,040,000.00	$23,040,000.00	$23,040,000.00	$23,040,000.00
$7,142,857.14	$7,142,857.14	$7,142,857.14	$7,142,857.14	$7,142,857.14	$7,142,857.14	$7,142,857.14	$7,142,857.14	$7,142,857.14	$7,142,857.14
259,200,000.00	259,200,000.00	259,200,000.00	259,200,000.00	259,200,000.00	259,200,000.00	259,200,000.00	259,200,000.00	259,200,000.00	259,200,000.00
$1.23	$1.23	$1.23	$1.23	$1.23	$1.23	$1.23	$1.23	$1.23	$1.23
$583,200,000.00	$583,200,000.00	$583,200,000.00	$583,200,000.00	$583,200,000.00	$583,200,000.00	$583,200,000.00	$583,200,000.00	$583,200,000.00	$583,200,000.00
$0.00	$0.00	$0.00	$0.00	$0.00	$0.00	$0.00	$0.00	$0.00	$0.00
$318,754,285.71	$318,754,285.71	$318,754,285.71	$318,754,285.71	$318,754,285.71	$318,754,285.71	$318,754,285.71	$318,754,285.71	$318,754,285.71	$318,754,285.71
$264,445,714.29	$264,445,714.29	$264,445,714.29	$264,445,714.29	$264,445,714.29	$264,445,714.29	$254,445,714.29	$264,445,714.29	$264,445,714.29	$264,445,714.29
0.51	0.48	0.46	0.44	0.42	0.40	0.38	0.36	0.34	0.33
$133,563,055.59	$127,202,910.09	$121,145,628.66	$115,376,789.20	$109,882,656.38	$104,650,148.93	$99,666,808.51	$94,920,770.00	$90,400,733.34	$86,095,936.51
$970,795,903.88	$1,097,998,813.97	$1,219,144,442.63	$1,334,521,231.82	$1,444,403,888.20	$1,549,054,037.13	$1,648,720,845.64	$1,743,641,615.64	$1,834,042,348.98	$1,920,138,285.49
$39,666,857.14	$39,666,857.14	$39,666,857.14	$39,666,857.14	$39,666,857.14	$39,666,857.14	$39,666,857.14	$39,666,857.14	$39,666,857.14	$39,666,857.14
$224,778,857.14	$224,778,857.14	$224,778,857.14	$224,778,857.14	$224,778,857.14	$224,778,857.14	$224,778,857.14	$224,778,857.14	$224,778,857.14	$224,778,857.14
$1,459,894,313.57	$1,684,673,170.71	$1,909,452,027.86	$2,134,230,885.00	$2,359,009,742.14	$2,583,788,599.29	$2,808,567,456.43	$3,033,346,313.57	$3,258,125,170.71	$3,482,904,027.86