A STUDY OF FLOTATION AND LEACHING BEHAVIOUR OF BORNITE AND CHALCOPYRITE

by

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Abstract

This study was carried out to investigate the flotation and leaching behaviour of bornite and chalcopyrite when they are together as principal sulphide minerals. Bench scale flotation performance was examined both in the absence and in the presence of a collector, PEX. In the absence of a collector, chalcopyrite floated significantly better than bornite. In the presence of a collector there was a reversal in selectivity, with bornite recovery presiding over that of chalcopyrite. Chalcopyrite recovery was marginally improved from the collectorless flotation case while bornite experienced vastly improved recoveries in the presence of a collector to surpass the recovery of chalcopyrite. The gap between floatability of bornite and chalcopyrite became more prominent under oxygen-deficient conditions induced by nitrogen flotation, which significantly decreased flotation rate of both minerals, but the floatability of bornite was still much higher than that of chalcopyrite. It is suggested that galvanic interactions promote the interaction of bornite with a collector while retarding interactions of chalcopyrite with the collector. Collectorless flotation of chalcopyrite observed under oxidizing conditions using air is thought to be prevalent even in the presence of a collector obscuring the selectivity of the process.

Leaching of the mixtures of bornite and chalcopyrite at mass ratios of 1:1, 1:3 and 3:1 are performed under various conditions to investigate the possible changes in leaching behaviour in relation to galvanic interactions. Leaching at 90°C in the presence of 0.1 M ferric ions in 0.1 M sulphuric acid provided the best results with 90% copper extraction from the mixtures in 6 hours. Observed copper extractions suggested increased copper extraction from chalcopyrite with this approach. The introduction of cupric and ferrous ions into solution, together with variation of solution potential made it difficult to assess in details on the effects of galvanic interactions due to these parameters being known to improve copper extraction from chalcopyrite. Galvanic interactions together with the presence of cupric ions were held mainly responsible for the increased copper extraction.

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Statement of Originality

I hereby certify that all of the work described within this thesis is the original work of the author. Any published (or unpublished) ideas and/or techniques from the work of others are fully acknowledged in accordance with the standard referencing practices.

(Gabotswane Mosweu)

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List of Abbreviations

ORP, The redox potential in mV (Ag, AgCl) unless stated

Bo, bornite

Cp, Chalcopyrite

Sp, sphalerite

Cc, chalcocite

Co, covellite

Py, pyrite

Chapter 1

Introduction

Sulphide mineral deposits normally contain more than one sulphide mineral. Very massive sulphide deposits or the porphyry copper deposits being examples of such deposits. The presence of more than one sulphide in a system usually leads to alterations in the behaviour of those sulphide minerals in both flotation and leaching operations. This alteration in behaviour of sulphide minerals, when in contact, is brought about by galvanic effects due to the semi-conductive nature of these minerals. Under given pulp conditions, sulphide minerals exhibit different rest potentials thus rendering the mineral cathodic (noble) or anodic (active). The cathodic or anodic behaviour of a given sulphide mineral can be amplified by having another species, a sulphide mineral or grinding media, of different rest potential in contact with that mineral. This can have an adverse impact or lead to a performance improvement of a process, therefore it is important to understand the effects brought about by the interactions of these minerals.

Chalcopyrite and bornite are common copper bearing sulphide minerals. Chalcopyrite is by far the most common and accounts for a large percentage of world copper production. Even though bornite is far less abundant compared to chalcopyrite, it is a significant source of copper and has almost twice the copper content of chalcopyrite, 63% versus 35% form chalcopyrite. These minerals can often be found in the same deposit, therefore knowledge of their interactions is valuable. The interactions of these minerals were studied for both flotation and leaching processes.

1.1 Flotation

In flotation, sulphide minerals have been found to exhibit natural or induced hydrophobicity, which means that they can float without the use of collectors. This behaviour is influenced by pulp conditions. Different sulphide minerals exhibit varying degrees of this induced hydrophobicity and

it has been found out that the presence of more than one sulphide can enhance or be detrimental to the hydrophobicity of other sulphides due to increased or decreased surface reactivity. Induced hydrophobicity is often linked to the formation of elemental sulphur on the mineral surface or the formation of a sulphur-enriched metal-deficient layer on the mineral surface due to reactions taking place at the surface [1]. Oxidation and metal ion dissolution are often linked to the formation of this sulphur-enriched sulphide layer. Excessive oxidation has however found to lead to formation of hydrophilic sulphuroxy species on the mineral surface which can render the mineral unfloatable [2]. Adsorption of species from the pulp, whether ions or precipitates, onto the mineral surface can also influence hydrophobicity by either enhancing it or retarding it depending on the nature of species adsorbed. The enhancement of either cathodic or anodic reactions can lead to increased kinetics of surface reactions and lead to varying results depending on the magnitude of the galvanic interactions.

Similarly, in the presence of a collector, the floatability of sulphides can be altered by the presence of another sulphide in the system due to electrochemical reactions involved between the minerals and also the collector. A mixed potential model has been proposed to explain the electrochemical interactions of sulphide minerals with a thiol collector under different pulp conditions [3]. The electrochemical reactions resulting from galvanic interactions of sulphides can lead to alteration of the expected behaviour. Reduction or oxidation of the collector to form a variety of species can also be affected by these galvanic interactions which can lead to collector adsorption through alternative mechanisms, which affect the selectivity of the process. The presence of oxygen when galvanic interactions are involved is also known to have an influence on the resulting behaviour [4], [5].

In a bornite-chalcopyrite system, chalcopyrite is known to have the highest rest potential, thereby making it noble or cathodic while bornite with a lower rest potential is expected to be anodic and relatively more reactive [6].

1.2 Flotation Objectives

This study is aimed at investigating the differences in bench scale flotation behaviour of bornite and chalcopyrite in relation to possible galvanic interactions that might be occurring in a natural ore system. Factors that may influence the observed behaviour include: pulp pH; pulp potential and presence of oxygen; prior oxidation of samples; and liberation of mineral particles. Effects of each of these parameters were investigated on flotation with or without the use of a collector.

1.3 Leaching

Leaching of copper sulphide minerals is often characterised by lower extractions compared to that of oxides and carbonates. Under standard conditions, leaching kinetics are often slow and do not always achieve complete copper dissolution. Bornite is known to exhibit better leaching performance than chalcopyrite even though this performance decreases with increasing copper extraction from the mineral. Chalcopyrite leaching is a subject of intensive study due to the poor leaching performance of this mineral, which is also the main source of copper globally. Passivation of the mineral is usually held responsible for poor leaching performance and there are studies aimed at circumventing this issue.

Bornite is known to leach in two stages represented by the equations below:

$$Cu_5FeS_4 + 4Fe^{3+} \rightarrow Cu_3FeS_4 + 2Cu^{2+} + 4Fe^{2+} \quad \textbf{(1 - 1)}$$

Followed by:

$$Cu_3FeS_4 + 8Fe^{3+} \rightarrow 3Cu^{2+} + 9Fe^{2+}4S^0 \quad \textbf{(1 - 2)}$$

Equation (1 - 1) represents Stage I of the leaching of bornite and terminates after 28% copper extraction while equation (1 - 2) represents Stage II of the process and its rate decreases with time. Temperature and ferric ions are known to have significant effects on the leaching of bornite.

Anodic oxidation of chalcopyrite is represented by equation (1 - 3) below:

$$CuFeS_2 + 4Fe^{3+} \rightarrow Cu^{2+} + 5Fe^{2+} + 2S^0 \quad \textbf{(1 - 3)}$$

There is debate as to whether passivation in the leaching of chalcopyrite occurs and as to the cause of such passivation. Elemental sulphur, polysulphides, metal-deficient sulphide layer and jarosite formation are some of the proposed passivation layers [7]. In addition to temperature and ferric ion concentration, surface area and ORP are known to influence the leaching of this mineral.

1.4 Leaching Objectives

The leaching part of this study is aimed at investigating the effects of galvanic interactions between single minerals of bornite and chalcopyrite in leaching. Sulphuric acid was chosen as the leaching medium. Parameters that may influence the observed leaching behaviour include: acid concentration; the presence of ferric ions; temperature and particle size. The influence of each of these parameters on the copper extraction was investigated.

Chapter 2

Literature Review

2.1 Sulphide Minerals

A predominance diagram for Cu-Fe-S-H_2O system which includes both chalcopyrite and bornite is shown in Figure 1. This diagram is for the system at 298 K assuming activities of aqueous species of 10^{-1}.

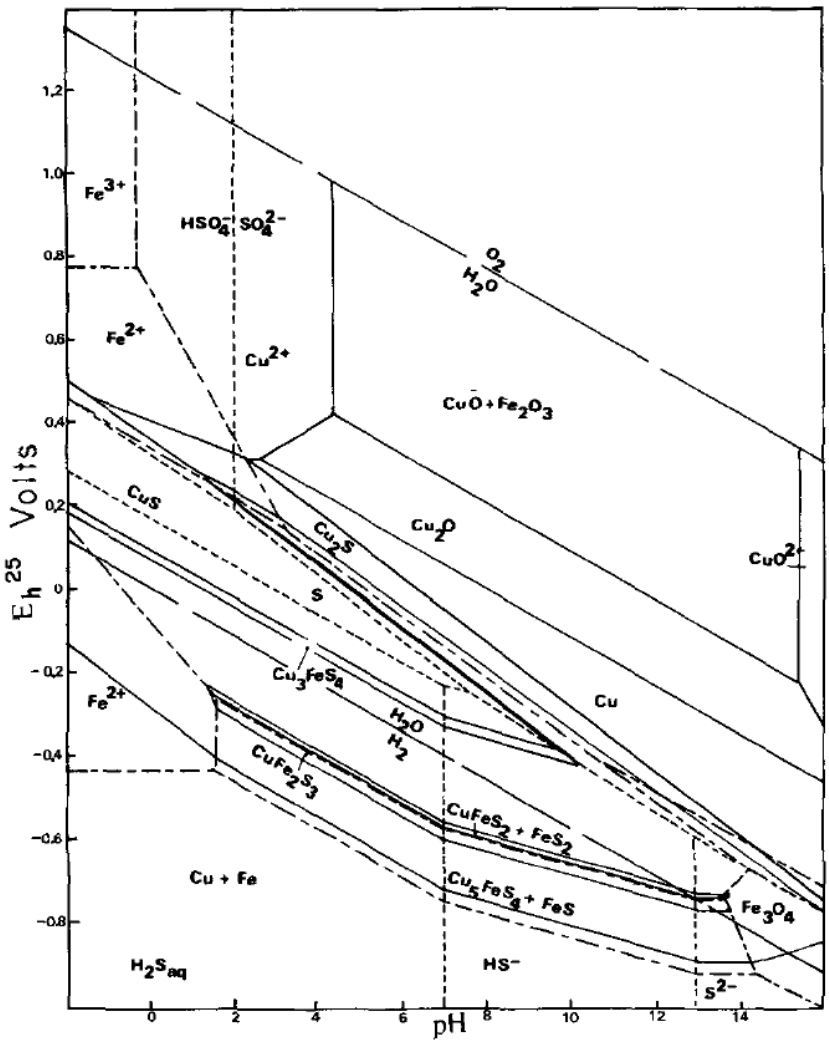

Figure 1: Eh-pH diagram for the Cu-Fe-S-H_2O system at 298 K incorporating the $CuFe_2S_3$ and Cu_3FeS_4 phases. Activity of aqueous species is 10^{-1}. [8].

This predominance diagram will be useful for understanding the chemistry involved in both flotation and leaching processes under given pulp conditions.

2.1.1 Bornite

Bornite is purplish copper mineral with a chemical formula Cu_5FeS_4. It has copper-red to pinchbeck-brown fresh surfaces which rapidly tarnishes to an iridescent purple. It is usually found associated with and disseminated in mafic igneous rocks, in pegmatites, in contact with metamorphic skarn deposits, in medium to high temperature hydrothermal deposits and in sedimentary cupriferous shales which are stable below 200^0C. Other minerals usually present in deposits containing bornite are: chalcopyrite and pyrite including other copper and iron sulfides. Garnet, calcite, wollastonite and quartz are other minerals also associated with bornite containing deposits. In these deposits bornite can be present either as a primary mineral or as a secondary mineral. Bornite deposits can be found globally, with some notable deposits in the USA, England, Austria, Kazakhstan, Zimbabwe, South Africa and Australia. [9].

The structure of bornite has been determined to be of the metastable rhombohedral form shown in Figure 2. [10].

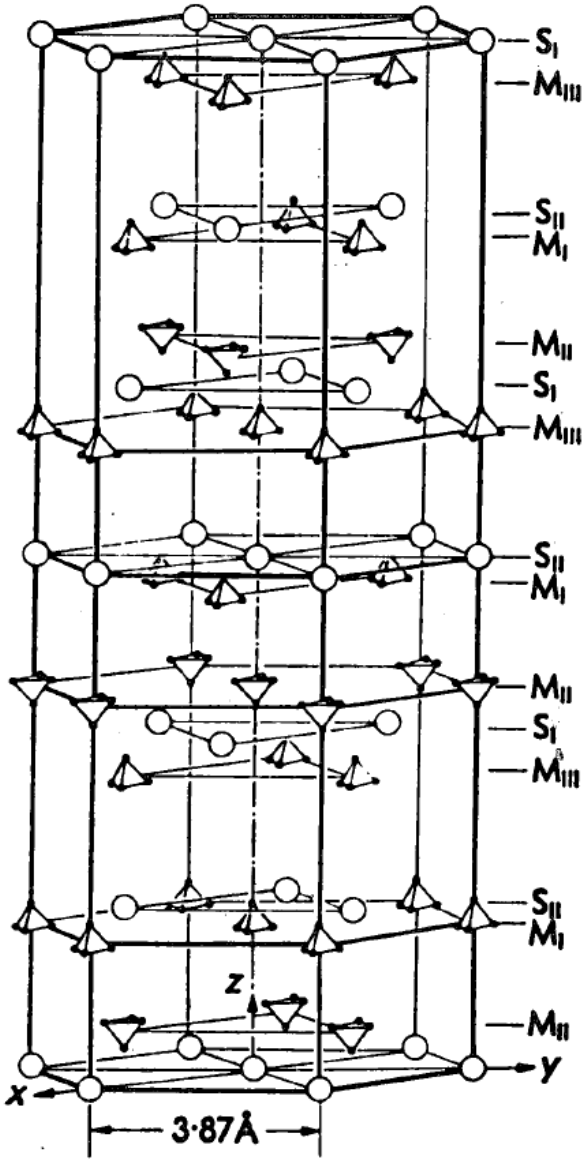

Figure 2: Structure of rhombohedral form of bornite. [10].

This type of structure is also known as the sphalerite-type structure, which in the case of bornite contains layers of ionically-bound interstitial Cu (I) atoms. The structure of bornite can therefore be written as$[Cu_3FeS_4]^{2-}.2Cu^+$.

Bornite has also been found to be a semiconductor through electrical measurements. The sphalerite type skeleton of bornite is thought to be made of Cu (I) and Fe (III) atoms. The Gibbs

free energy for bornite has been determined to be ($-715.99 + 0.00339T + 9.2550T^{0.5}$) KJ/mol between the temperature range 298 – 1200 K. [11].

2.1.2 Chalcopyrite

Chalcopyrite is a brass-yellow coloured mineral with the chemical formula $CuFeS_2$. This mineral is found in a variety of deposits and is a primary mineral in hydrothermal veins, stockworks, disseminations, and massive replacements which are exsolution products in mafic igneous rocks of sedimentary origin controlled by redox conditions. Other minerals normally associated with chalcopyrite containing deposits include sphalerite, galena, tetrahedrite, pyrite and a variety of copper sulfides. Chalcopyrite is the most common copper mineral with deposits all over the globe. This mineral accounts for the majority of copper produced globally. [9].

Chalcopyrite crystal structure is known to be similar to that of sphalerite. It is classified to be a tetragonal scalenohedral $\bar{4}2m$ crystal system. The unit cell for chalcopyrite is twice as large as that for sphalerite and is shown in Figure 3. Alternating Cu^+ and Fe^{3+} ions replace Zn^{2+} ions in this structure [12].

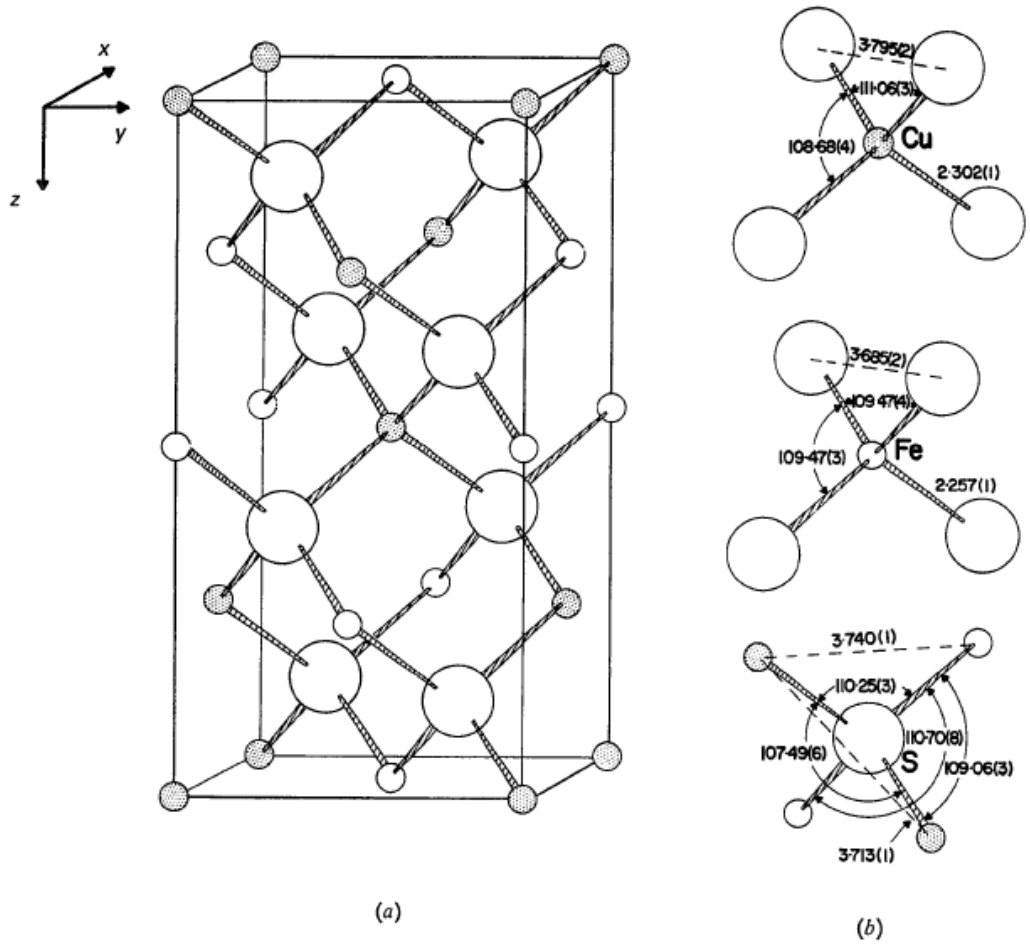

Figure 3. (a) Unit-cell model showing crystal structure of chalcopyrite. (b) Interatomic bond lengths and angles of each atom in chalcopyrite. [12].

Studies done on the structure of chalcopyrite suggest that this structure exists in a strong covalently-bonded configuration which has an effective ionic state between $Cu^{+}Fe^{3+}S_2^{2-}$ and $Cu^{2+}Fe^{2+}S_2^{2-}$. [12]. The Gibbs free energy of chalcopyrite has been determined to be (-369.64 + 0.05154T + 4.5690$T^{0.5}$) KJ/mol between the temperature range 298 – 820 K. [11].

2.2 Froth Flotation

Flotation is a versatile process used in the mining industry to concentrate minerals by using hydrophobicity to float them and recover them from the slurry into the froth (or vice versa, float the unwanted material). The ability of specific minerals (or mineral-collector) to selectively adhere

to bubbles in the ore/water slurry is what makes froth flotation an efficient method for physical separation of minerals. This method is better suited to a certain particle size range in which particles are light enough to be transported by bubbles however the particle sizes in the slurry have to be in a range where entrainment of fine particles is minimal thereby making selective flotation possible. There are various important parameters that need to be monitored for achieving a good separation using flotation. Figure 4 below shows the important components of a flotation system.

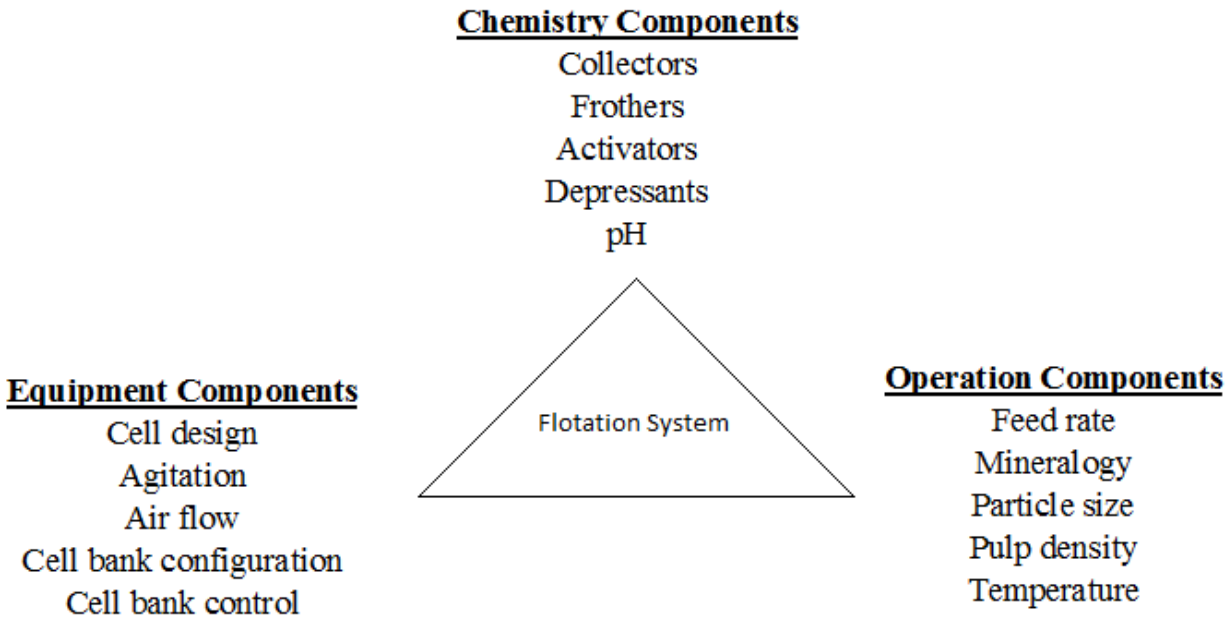

Figure 4: Components involved in the flotation system. [13].

The parameters involved in a flotation system can be diverse but still closely inter-related, with adjustments to one factor likely to lead to changes in other parts of the system. For this reason, it is difficult to isolate and study properly the effects of a single factor making it hard to develop predictive models of the system.

In a particular flotation system it is important to identify the most important factors which are likely to influence the flotation performance and determine how they can be adjusted to for the best result. Several components are discussed in detail below and their relevance to the study.

2.2.1 Flotation Theory

In the scientific study of flotation, a three phase system approach is usually employed. The three phases are: mineral particles, air and water, with the interfacial phenomenon between these phases being of paramount importance. Due to the complexity of this system in practice, laws governing this system are mostly hypothetical and based on approximations of the real process. Surface energies are held responsible for the adsorption of reagents at these interfacial phases

Of these three phases, the mineral particle phase is the most complex due to the numerous changes that occurs in the vast minerals covered in this phase. Mineral surfaces from natural ores may contain differing physiochemical properties for the same mineral and are therefore often regarded as heterogeneous. This affects how the mineral interacts with different reagents. The liquid phase is normally water. This phase affects physiochemical properties of mineral surfaces, attachment of minerals to air bubbles and response of reagents. The chemical properties of water under different conditions is key to fully understanding how the flotation process could be affected. The properties of water that are important to flotation include: hydration of ions, solubility of minerals in water, dissolved gases and the influence of impurities. The air phase is associated with aeration of the flotation pulp and dissolved gases. This includes the attachment of particles to air bubbles and also their transport to the froth.

The three phases discussed above play an important role in the determination of the degree of wettability of mineral surfaces in the flotation pulp, rending the surface hydrophilic or hydrophobic. The profile at which the three phases attach is referred to as the contact angle and it plays an important role in the hydrophobicity of the mineral. This angle is determined by the interfacial energies between the solid, liquid and gas phases. A diagram showing the contact

angle of an air bubble to a mineral surface in flotation pulp is shown by Figure 5.

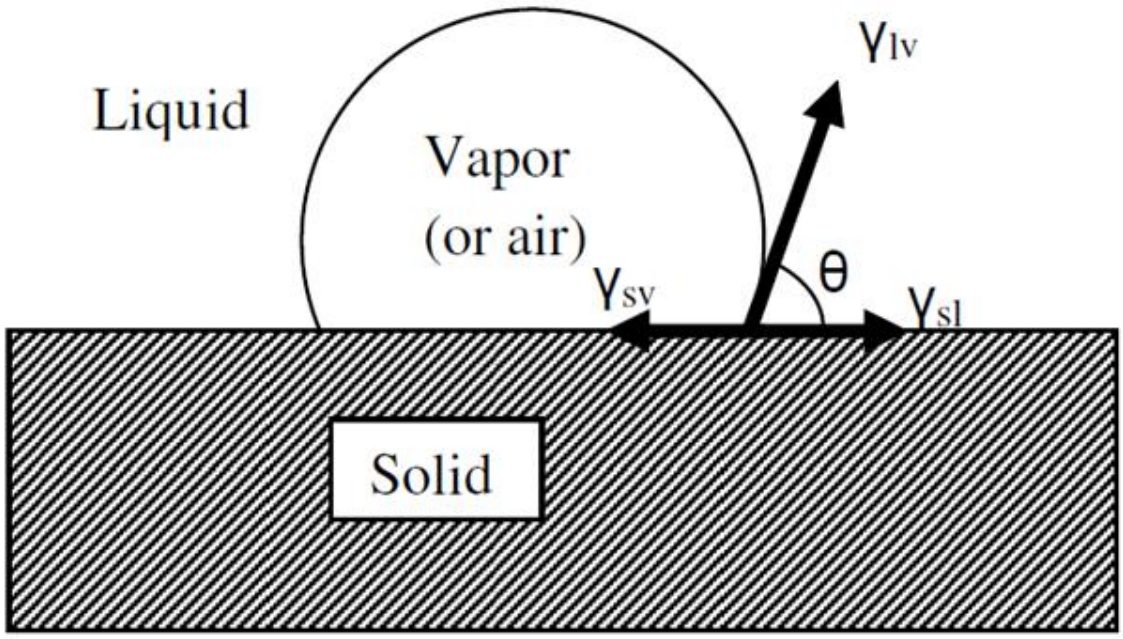

Figure 5: Contact angle between an air bubble and a solid surface immersed in liquid. [13].

The Young/Dupre equation shown below is used to determine this angle:

$$\gamma_{lv} \cos\theta = (\gamma_{sv} - \gamma_{sl}) \quad \textbf{(2 - 1)}$$

Where:

γ_{lv} is the surface energy of the liquid/vapor interface, γ_{sv} is the surface energy of the solid/vapour interface, γ_{sl} is the surface energy of the solid/liquid interface, and θ is the contact angle.

2.2.2 Adsorption Theories

Reagents in solution or reaction products between reagent and dissolved species can accumulate at the surface of mineral particles and influence its flotation performance. Understanding how this happens will help plan the best conditions for beneficiation or depression of minerals. Three basic theories are often used to describe how species adsorb to mineral surfaces. These theories are: Physisorption in which species physically adsorb to the mineral surface leading

to an increase or decrease in surface tension of water; chemisorption which involves chemical reactions in the adsorption of species with the mineral surface; and electrochemical theory adsorption which is mainly associated with the semi-conducting properties of minerals, mainly sulphides.

Electric double layer:

In flotation, electrical characteristics of the mineral surfaces are one of the most important properties for understanding the process. Electrochemical and electrokinetic potentials determine the electrical properties of mineral surfaces. Ions from the mineral surface may be transferred into the water phase causing changes to the electrical equilibrium on the mineral surface. It is believed that some of the dissolved ions concentrate near the mineral surface due to the action of the charge left on the mineral surface creating a phenomenon known as the electrical double layer [14]. This layer is believed to play a central role in the adsorption of reagents on the mineral surface thereby influencing the floatability of the mineral. A schematic showing the electrical double layer is presented in Figure 6.

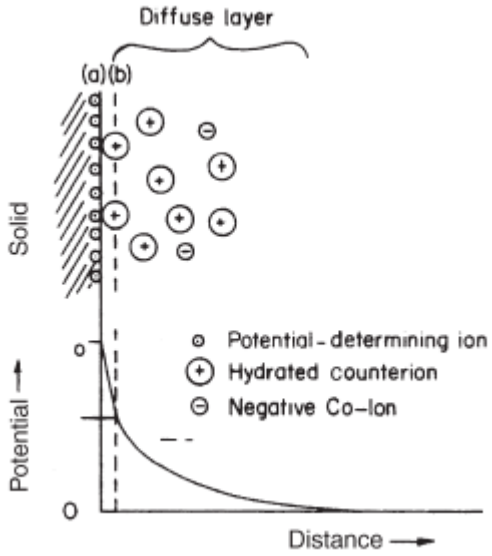

Figure 6: Schematic representation of the double layer and potential drop across the double layer; (a) surface charge, (b) stern layer and (c) diffuse layer and counter ion. [14].

During the movement of particles in the liquid, changes in the electrical double layer may occur. The electroneutrality of the mineral surface may change when the ions in the diffused layer are carried away from the surface, creating a potential difference between the moving particles and the liquid phase. This potential difference is referred to as the Electrokinetic potential or more commonly: the zeta potential. In flotation studies, zeta potential measurements are often used to determine the action of collectors on the mineral surface.

Charge development on the mineral surface generally occurs when there are mobile electrons present at the freshly ground mineral surface. If the grinding exclusively ruptures the Van der Waals bonds, with the underlying lattice occupied by covalently bonded molecules and no mobile charges, the mineral surface would remain without charge. The Stern model described below is often used to describe the action of charges between the mineral surfaces and the liquid phase. Diagrams representing components of this model are shown in Figure 7 and Figure 8.

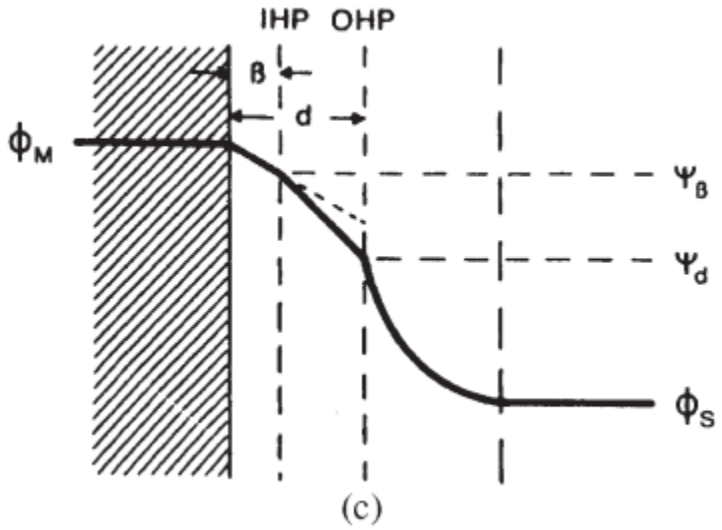

Figure 7: Stern double layer model. [13].

From this figure, OHP is the outer Helmholtz plane which represents the more weakly adsorbed hydrated ions; d is the minimum distance ions in solution can approach the solid surface; IHP represents the inner Helmholtz plane, which is the area of the adsorbed unhydrated ions; β

represents distance of the IHP from the solid phase; ϕm and ϕs represent the metallic and solution phases respectively.

Modifications to the Stern model are displayed in Figure 8 below. Figure 8 (a) Shows the occurrence of charge reversal within the compound layer while Figure 8 (b) shows the compound double layer in concentrate electrolyte solution in which the diffuse layer disappears.

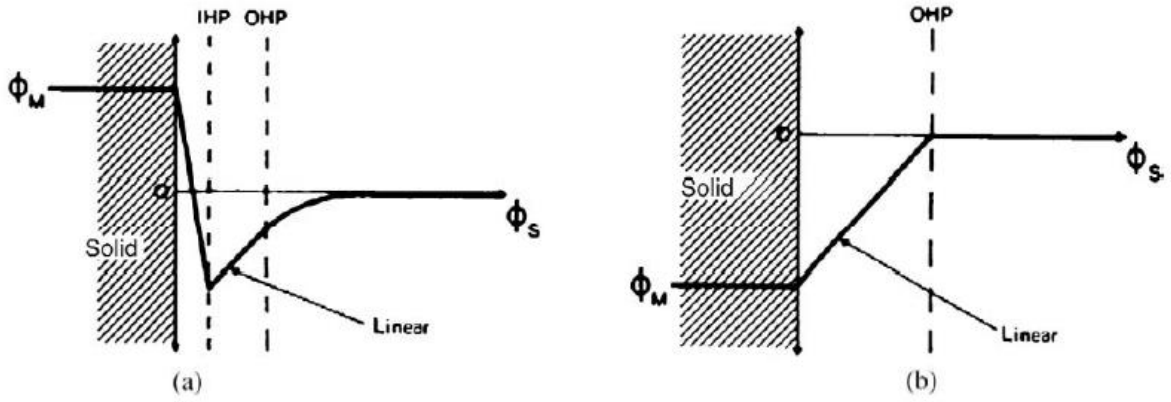

Figure 8: Types of modification of the Stern compound layer. [13].

Mechanical and electrical effects of the moving interface are very useful in surface chemistry and this Electrokinetic data is usually represented in the zeta potential. The zeta potential is then used in understanding reagent adsorption and bonding to the mineral surface.

2.3 Collectorless Flotation

Sulphide minerals may acquire a certain degree of hydrophobicity through: compositional and texture variation of the mineral; origin of formation of that mineral; or surface modification. It has been established that some minerals will float naturally without the use of collectors. There is however a debate as to whether all sulphide minerals are naturally floatable, or if their floatability is influenced by pulp conditions. Molybdenite and stibnite are some of the sulphides that have been identified to float naturally without influence by the pulp conditions [14]. Most of the regular sulphides such as pyrite, sphalerite, galena, chalcopyrite and bornite are believed to be not floatable naturally, but rather their floatability is induced by the pulp conditions, mostly the pulp potential.

Sulphide minerals normally cannot form hydrogen bonds on the surface therefore they are not very hydrophilic and any change in in their surface chemistry can induce or depress their hydrophobicity.

Some ways in which sulphides are thought to gain hydrophobicity are: through the influence by physical-chemical adsorption of oxidation products of xanthates; through the influence by electrochemical adsorption by an anionic collector assisted by the cathodic reaction of oxygen; and through influence by elemental sulphur formation on the surface of the sulphide mineral. [14].

2.3.1 Ranking

The collectorless floatability of sulphide minerals that are presumed to not possess natural floatability is often linked to the oxidation of the mineral surfaces [6]. This floatability therefore depends quite significantly on the pulp oxidation/reduction potential.

Researchers have attempted to rank sulphides according to their relative ease of collectorless flotation. This ranking has been found to be affected by pulp pH and some results from tests performed at pH 4 are shown in Table 1 below.

Table 1: Approximate order of collectorless flotation of sulphide minerals (descending order of floatability). [6]**.**

Mineral	*Chemical formula*	*Rest Potential (V vs. SHE)*
Molybdenite	MoS_2	0.11
Stibnite	Sb_2S_3	0.12
Argentite	AgS	0.28
Galena	PbS	0.40
Bornite	Cu_5FeS_4	0.42
Covellite	CuS	0.45
Sphalerite	ZnS	0.46

Chalcopyrite* (anomalous)	$CuFeS_2$	0.56
Marcasite	(Zn, Fe)S	0.63
Pyrite	FeS	0.66

Generally collectorless flotation of sulphide minerals has been found to follow along this descending order: chalcopyrite, galena, pyrrhotite, pentlandite, covellite, bornite, chalcocite, sphalerite, pyrite and arsenopyrite. [6]. From this list, the first four minerals show strong collectorless behaviour over a wide particle size range while the last four minerals in this ranking show very little collectorless behaviour. Bornite is found to be significantly less floatable, which is opposite to the behaviour of chalcopyrite.

From the rest potentials reported in Table 1 the collectorless flotation behaviour of the minerals seem to match with the descending order of their rest potentials, except for chalcopyrite. It has to be noted that the significance of the rest potential measurements is questionable due to the possibility of individual specimen of a single mineral exhibiting different rest potentials due to slightly different compositions and also due to the degree of mineral oxidation prior to rest potential measurement.

2.3.2 Oxidation Products

From the rest potential measurements, it appears that the more anodic the rest potential, which also means that the mineral is less easily oxidized, the more readily it will float without the use of a collector. It is believed that if a sulphide mineral oxidizes strongly it forms hydrophilic surface species which will prevent it from floating [15].

Oxidation products of common sulphide minerals have been investigated and the results are presented in Table 2 below.

Table 2: Reaction products formed by the oxidation of sulphide minerals. [6]**.**

Mineral	*Products from originally neutral (pH 6) suspension*		*Products from originally alkaline (pH 10 to 11) suspension*	
Galena	Present:	Cu^{2+}, Fe^{2+}, Fe^{2+}, SO_4^{2-}-	Present:	H^+, Ag^+, Pb^{2+}, SO_4^{2-}, thio-salts
	Absent:	Pb^{2+}, polythionate	Absent:	Fe^{2+}, Fe+, Cu^{2+}
Pyrite	Present:	Fe^{2+}, H^+, SO_4^{2-}	Present:	H^+, SO_4^{2-}, $S_3O_6^{2-}$, $S_4O_6^{2-}$, $S_2O_3^{2-}$, SO_3^{2-}
	Absent:	Fe^{3+}, polythionate	Absent:	Fe^{3+}, Fe^{2+}, Cu^{2+}, S^{2-},$S_5O_6^{2-}$, $S_2O_6^{2-}$
Chalcopyrite	Present:	H^+, Cu^{2+}, Fe^{2+}, Fe^{3+}, SO_4^{2-}	Present:	H^+, SO_4^{2-}, $S_2O_3^{2-}$, $S_4O_6^{2-}$
	Absent:	Polythionate	Absent:	Cu^{2+}, Fe^{2+}, Fe^{3+}, SO_3^{2-}
Sphalerite	Present:	SO_4^{2-}, thio-salts	Present:	H^+, Zn^{2+}, SO_4^{2-}, SO_3^{2-}
	Absent:	Zn^{2+}, Fe^{2+}, Fe^{3+}, Cu^{2+}	Absent:	$S_4O_6^{2-}$,$S_2O_3^{2-}$

Work has been done to assess the rate of formation and quantities of species formed under different solution conditions. This rate has been found to depend on the surface area available for reaction; the partial pressure of oxygen; the type and composition of the sulphide mineral; solution pH and temperature [6]. The presence of more than one sulphide mineral in the pulp has also been seen to change the behaviour of how these minerals respond to collectorless flotation.

2.3.3 Chalcopyrite

Studies conducted with nitrogen gas replacing air in grinding, conditioning and flotation have generally observed poor recoveries of chalcopyrite [16]. Figure 9 below shows the flotation performance of chalcopyrite after being ground in an iron mill in a nitrogen purged environment and floated under nitrogen. The remaining pulp was then conditioned for 10 minutes under oxygen and re-floated.

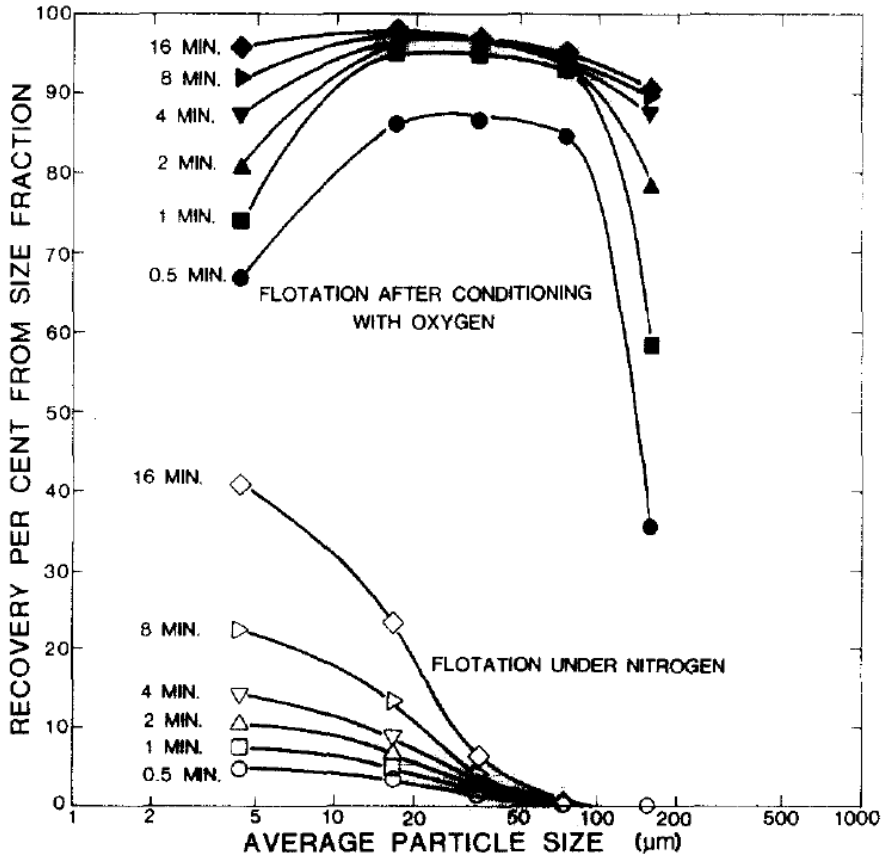

Figure 9: Recovery-size curves for flotation of chalcopyrite ground under nitrogen in iron mill before and after conditioning with oxygen. [16]**.**

The kinetics results for the test presented in Figure 9 are shown in Figure 10 below:

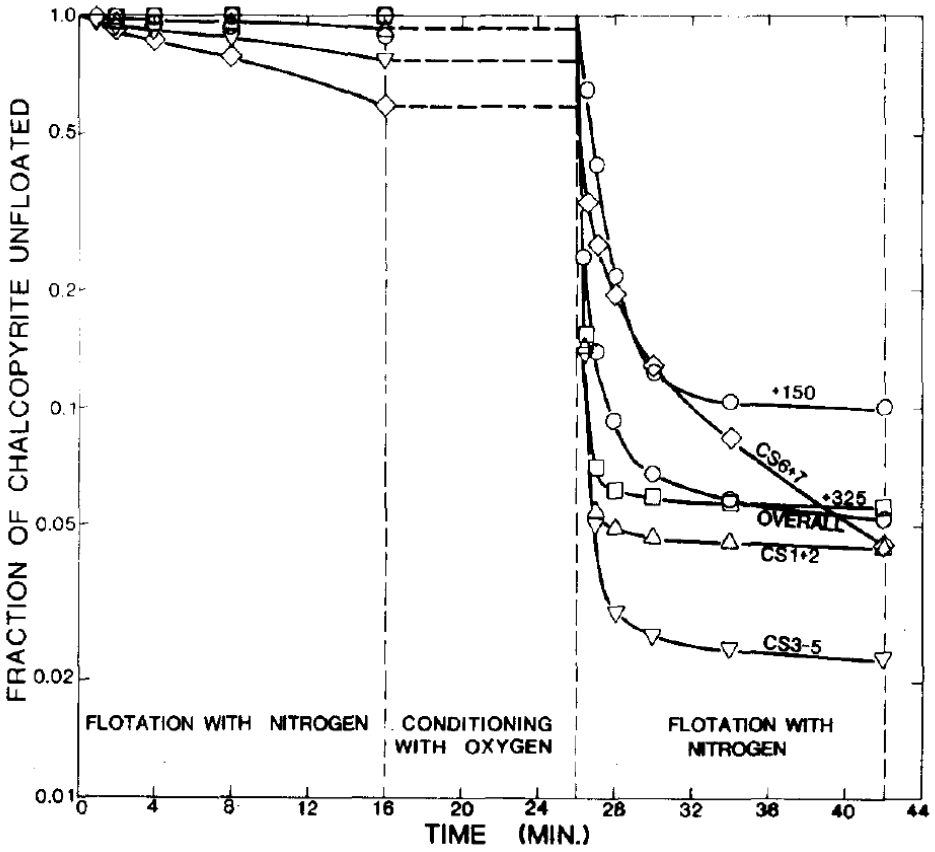

Figure 10: Rate data for flotation of chalcopyrite ground under nitrogen in iron mill before and after conditioning with oxygen. [16]**.**

From these two figures it is evident that chalcopyrite floats poorly when nitrogen gas is used during flotation. After the pulp is conditioned with oxygen, the flotation performance of chalcopyrite improves drastically showing the pronounced effect that oxygen (or oxidizing environment), has on the collectorless flotation of chalcopyrite. It has been shown that even under very low oxygen pressures, 10^{-6} mole fraction oxygen, and low dissolved oxygen (< 5 ppm), chalcopyrite flotation produced high recoveries. [17].

Pulp Potential

In studying the effects of pulp potential on chalcopyrite flotation, reducing agents such as sodium dithionite, sodium sulphite and ferrous sulphate have been used to track the flotation behavior of chalcopyrite under reducing conditions. [16]. The effects of pulp potential on the

collectorless flotation of chalcopyrite is shown in Figure 11 below, where chalcopyrite was floated over a range of solution potentials using sodium dithionite as a reducing agent and sodium hypochlorite as an oxidant.

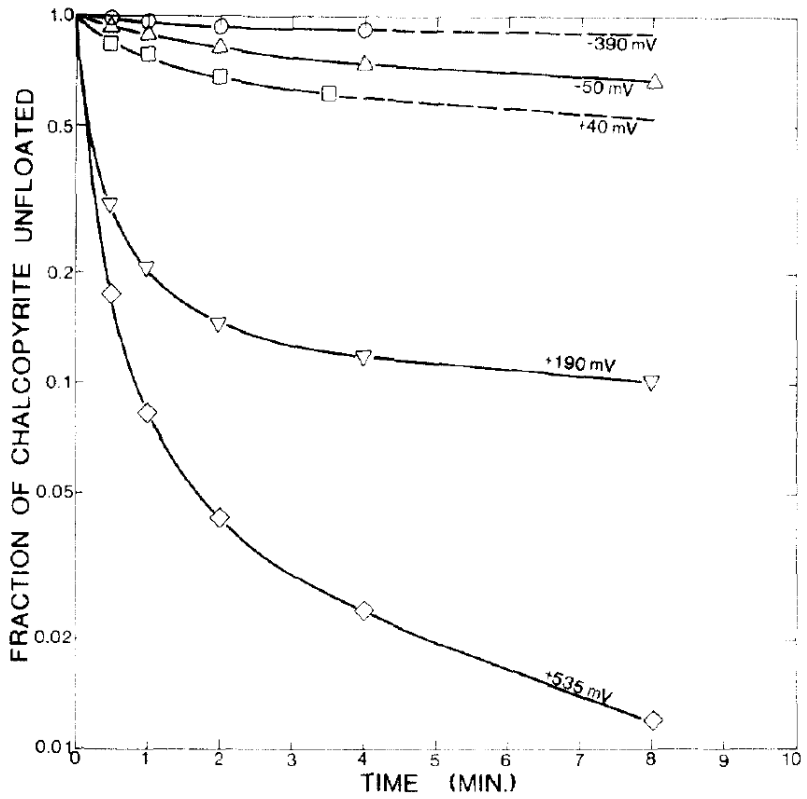

Figure 11: Effect of potential on the flotation of chalcopyrite. [16]**.**

These flotation tests were carried out using nitrogen gas to float the mineral. These tests confirm a relationship between pulp potential and collectorless flotation of chalcopyrite which shows that when the solution potential is dropped below -100 mV (vs. SHE) the mineral becomes non-floatable. When the solution potential is raised above +100 mV, chalcopyrite becomes floatable. The depression and floatability of chalcopyrite at negative and solution potential is also found to be reversible.

Another study has suggested that a mechanism of dissolution of iron and copper form chalcopyrite to leave a hydrophobic metal-deficient, sulfur rich surface followed by their

readsorption as ferric and cupric hydroxide species explain the flotation results of chalcopyrite during oxidation or under oxidizing conditions [18].

Broadly speaking, sulphide minerals are thought to go through a range of metal deficient sulphides of decreasing metal content all the way through to elemental sulphur. It is believed that further oxidation can produce thiosalts all the way to sulphates and metal ions which may readsorb onto the mineral surface as hydrolysis products. Controlled oxidation can produce elemental sulphur or produce sulphur-rich / metal-deficient zones on the mineral surface and render the mineral hydrophobic. From the results on tests performed on galena, bornite, chalcopyrite and pyrrhotite; the following equation was suggested to explain the formation of a metal deficient sulphide mineral surface [6].

$$MX + xH_2O \leftrightarrow M_{1-x}S + xMO + 2xH^+ + 2xe \qquad \textbf{(2 - 2)}$$

One condition for this hydrophobicity to occur is for the metal oxides/hydroxides that may form during this oxidation to solubilize and not coat the surface. If excessive oxidation occurs, sulphate and other sulphur oxygen species which are hydrophilic will form on the mineral surface which are normally be accompanied by metal ion hydrolysis products. This will retard the hydrophobicity of the mineral.

Using UV analysis to gather information on the surface species responsible for the collectorless flotation of chalcopyrite, an increasing amount of elemental sulfur on the chalcopyrite surface with increasing potential was detected [19]. Large amounts of elemental sulphur were also detected at reducing potentials, particularly when hydrazine was the reducing agent. From these findings elemental sulphur formation under oxidizing conditions can be attributed to high collectorless behavior of chalcopyrite under oxidizing conditions. However, poor flotation at reducing potentials cannot be attributed to the absence of elemental sulphur either.

From the same study, [19], the mass spectrometric technique detected insignificant amounts of elemental sulphur when conditioning was carried out in alkaline solutions. Possible

explanation given for this finding is that if sulphur was present as polysulphides, mass spectrometry would not have been able to detect them as they are usually strongly bound to the mineral surface. Lastly, the ESCA technique detected elemental sulphur and CuS on oxidized chalcopyrite surfaces. However these results could not be interpreted as showing the presence of polysulphides.

More recently, a study was conducted to investigate the effects of surface oxidation on the collectorless flotation of chalcopyrite by changing the mineral conditioning treatments [18]. This study proposed a mechanism of dissolution of iron and copper from chalcopyrite to leave a hydrophobic metal-deficient, sulphur-rich surface and their readsorption as ferric and cupric hydroxide species to explain the flotation of chalcopyrite during oxidation. This research found no sulphur species in solution and found that the dissolution of metal atoms from chalcopyrite left behind a metal-deficient, sulphur-rich surface rather than elemental sulphur. The iron dissolution rate was also found to be greater than or equal to that of copper. The equation below shows the proposed reaction mechanism.

$$CuFeS_2 \xrightarrow{H^+} (CuS)_{1-y}(FeS)_{1-z}(S^{2-})_{y+z} + yCu^+ + zFe^{3+} \quad \textbf{(2 - 3)}$$

Depending on the pH of the solution, the dissolved copper and iron ions could hydrolyze and form positively charged ferric and cupric hydroxide species. If these positively charged ferric and cupric hydroxide species adsorb/surface precipitate onto the negatively charged sulphur sites, a hydroxide layer covering the sulphur-rich chalcopyrite surface will form according to the reactions provided below:

$$-S^- + Fe(OH)_n^{(3-n)+} \Leftrightarrow -S^- - Fe(OH)_n^{(3-n)+} \quad \textbf{(2 - 4)}$$

and:

$$-S^- + Cu(OH)_n^{(2-n)+} \Leftrightarrow -S^- - Cu(OH)_n^{(2-n)+} \quad \textbf{(2 - 5)}$$

The adsorption of these positively charged hydroxides on a previously negatively charged surface changes the sign of the zeta potential of chalcopyrite and this agrees well with the observed correlation between the zeta potential and the pH region, where these positively charged hydroxide species are formed. This study found that chalcopyrite coverage with metal hydroxides increases with pH, the amount of oxygen and the conditioning time.

2.3.4 Bornite

Not much information has been collected on the collectorless flotation of bornite. As stated earlier, researchers have found that bornite is far less floatable when compared to chalcopyrite. From the rest potential rankings, bornite is more anodic than chalcopyrite, which translates to it being readily oxidised when compared to chalcopyrite.

A study conducted to study oxygen reduction on bornite in alkaline medium [20], found that bornite was generally stable under reducing Eh, between -0.1 V to -0.5 V as shown by Figure 12.

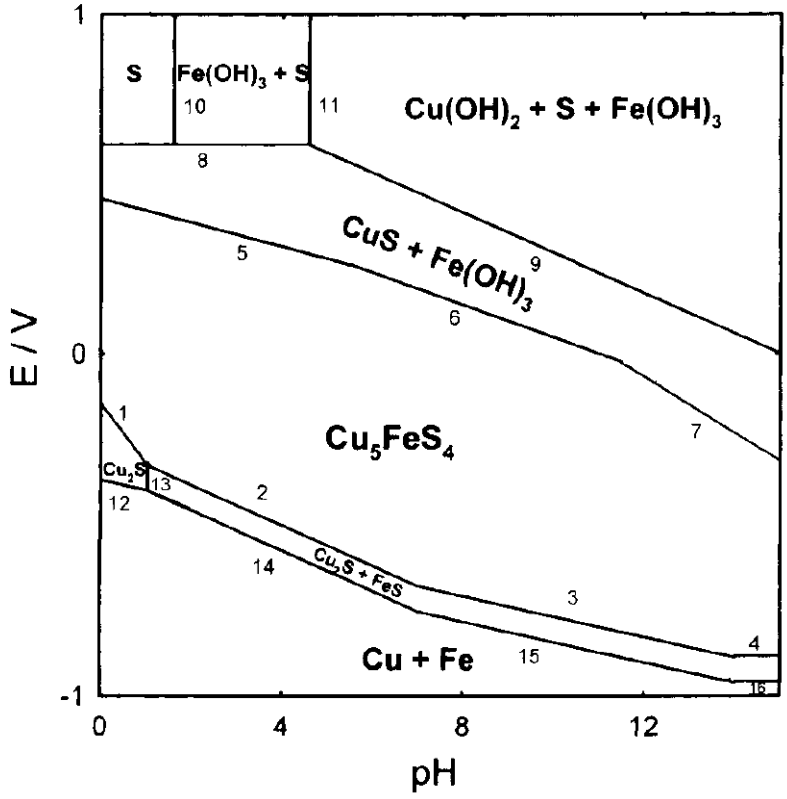

Figure 12: Eh-pH diagram of bornite. [20].

This study also found out that at E > -0.1 V, bornite can be oxidised to form CuS and $Fe(OH)_3$ or reduced to Cu_2S, FeS and S^{2-} at E < -0.5 V.

In alkaline solutions, bornite was suggested to oxidise by the following reaction.

$$Cu_5FeS_4 + 3H_2O \rightarrow Cu_5S_4 + Fe(OH)_3 + 3H^+ + 3e \qquad \textbf{(2 - 6)}$$

The formed copper sulphide further oxidises to form a sulphide with lower copper content.

$$Cu_5S_4 + 2xH_2O \rightarrow Cu_{5-x}S_4 + xCu(OH)_2 + 3xH^+ + 2xe \qquad \textbf{(2 - 7)}$$

At higher potential, covellite may form on the surface of bornite. This may improve the collectorless flotation of bornite as covellite is known to be more floatable than bornite. However, the question is whether or not this transformation can be accomplished selectively within a reasonable time period.

2.3.5 Galvanic Interactions between Minerals

It is well known that due to the semi-conducting properties, the presence of more than one sulphide mineral in a systems affects the flotation properties of both minerals. The difference in rest potentials of sulphide minerals sets up galvanic cell. Minerals with higher rest potential act as cathodes and are considered noble, while minerals which have lower rest potential acts as anodes and are more reactive. Current flows between these minerals altering the reactivity of these minerals.

Dissolved oxygen acts as an electron acceptor and usually reacts to form OH^- ions which may further react with the less reactive mineral or metal ions in solution to form oxides or hydroxide precipitates.

Grinding media can also have reducing effect on the galvanic system depending on the rest potential of the media used, altering the reactivity of the minerals involved.

Effect of Galvanic Interactions on Collectorless Flotation

Studies have been conducted to investigate the effects of galvanic interactions in the chalcopyrite-sphalerite system, [2], and the pyrite-chalcopyrite system, [15], in the absence of a collector. In these systems, chalcopyrite and pyrite were cathodic minerals while sphalerite and chalcopyrite were anodic minerals respectively. These tests were carried out in alkaline conditions.

Generally, it was found that an increase in the amount of cathodic mineral led to a decrease in the floatability of the anodic mineral. On the other hand, an increase in the amount of anodic mineral led to an improvement in the floatability of the cathodic mineral. The degree of the reduction in floatability of the anodic mineral was generally small compared to the improvement in the floatability of the cathodic mineral.

The reduction in the floatability of the anodic mineral was thought to be a result of increased oxidation on the mineral surface leading to sulphur oxidising to form hydrophilic

sulphates or sulphuroxy products. A schematic of a pyrite-chalcopyrite system is shown in Figure 13 below:

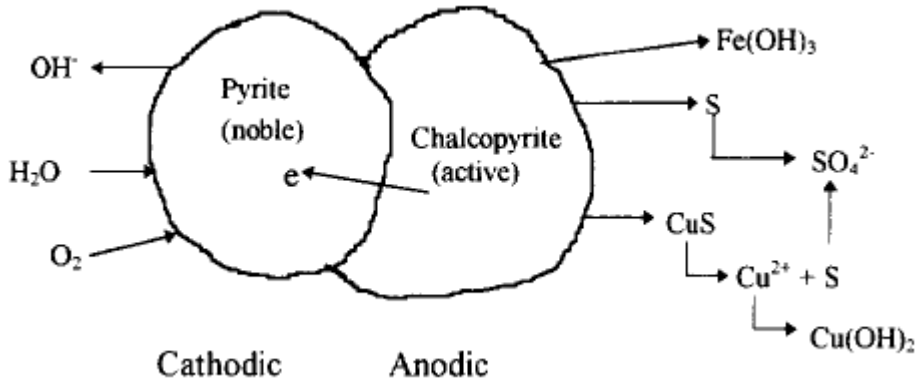

Figure 13: A model of galvanic interactions between chalcopyrite and pyrite. [15]**.**

In the presence of increased amounts of cathodic mineral, sulphur oxidation by equations **(2 - 8)** and **(2 - 9)** is thought to form hydrophilic sulphuroxy species.

$$S + 4H_2O \rightarrow SO_4^{2-} + 8H^+ + 6e \qquad \textbf{(2 - 8)}$$

and/or,

$$2S + 3H_2O \rightarrow S_2O_3^{2-} + 6H^+ + 4e \qquad \textbf{(2 - 9)}$$

In other reactions, the oxidation of the iron component of chalcopyrite is also enhanced by the increased oxidation rate leading to the formation of hydrophilic ferric hydroxide.

In the case of an increase in anodic mineral, increased reducing conditions are experienced by the cathodic mineral leading to reduction of the hydrophilic ferric hydroxide formed on the mineral surface to a more soluble ferrous hydroxide thereby leaving behind a metal-deficient sulphur-rich surface mineral surface.

The release of cupric ions into solution is also known to promote the flotation of sulphide minerals such as pyrite and sphalerite by adsorbing onto their surfaces and altering their surface chemistry.

This effect is shown in Figure 14 below in which the presence of cupric ions in solution improved the collectorless flotation of pyrite.

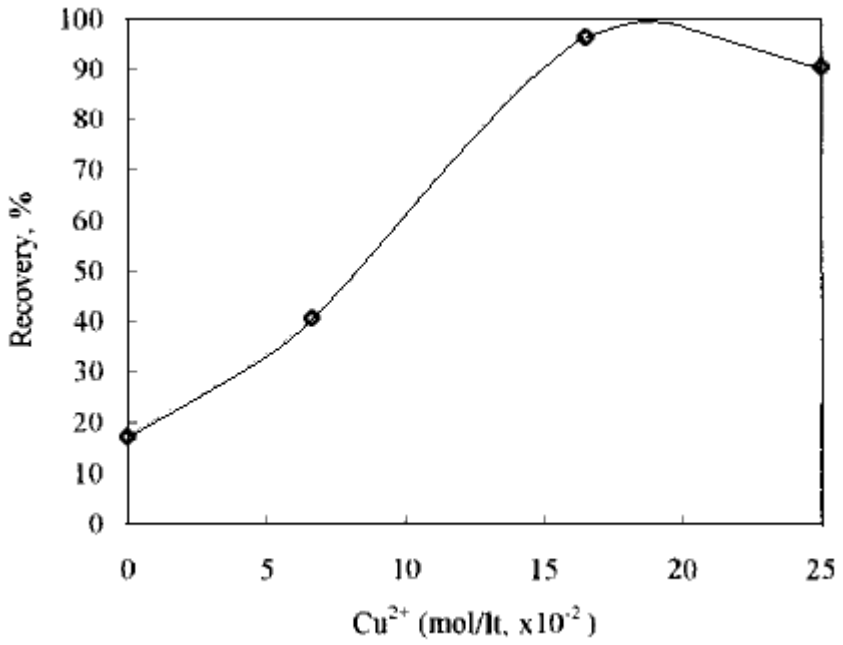

Figure 14: Collectorless flotation of pyrite in the presence of Cu^{2+} ions (pH 9.2, flotation time - 5 min). [15].

It is evident from this figure that the presence of cupric ions improve the floatability of pyrite. Similar results have been reported for sphalerite, therefore it is important to take this into consideration when interpreting the effects of galvanic interactions in systems where ions are present in solution. The general equation describing this reaction on pyrite is thought to be:

$$4Cu^{2+} + FeS_2 + 6e = Fe^{2+} + Cu_2S \qquad \textbf{(2 - 10)}$$

The formation of chalcocite on the pyrite surface is thought to promote the collectorless flotation of pyrite by the further oxidation of this mineral (chalcocite) that produces elemental

sulphur as one of the stable oxidation products. This sulphur is made stable by the slow rate of oxidation to generate sulphuroxy species.

The presence of quartz in the flotation system was also shown to improve collectorless flotation. Quartz is thought to adsorb multivalent cations, hydroxyl complexes and iron hydroxide, thereby reducing their adsorbance onto the sulphide mineral surface [15]. This leaves the sulphide mineral with a metal deficient sulphide layer which promotes collectorless flotation.

Oxygen

On the subject of the effects of oxygen in galvanic systems, ageing in the presence of oxygen was shown to have an influence the recoveries of sphalerite and chalcopyrite in cases when the minerals were floated individually or in combination [2]. Ageing sulphide minerals in the presence of oxygen is known to have adverse effects on their flotation recoveries, but it was found out that these poor recoveries were minimized in galvanic combinations depending on the relative noble or active character of the mineral in question.

2.3.6 Galvanic Interactions due to Grinding Media

In a study performed on investigation the role of grinding media-mineral interaction on the flotation response of chalcopyrite, [21], it was concluded that contact between chalcopyrite and grinding media affected the flotation of the mineral. Galvanic interactions between the grinding media (ferrous grinding media) and the mineral results in lowered floatability. The presence of oxygen during such contact is believed to further lower the floatability of chalcopyrite. Surface analysis of chalcopyrite showed the presence of oxy-hydroxide species of iron. Compared to a fresh chalcopyrite surface, an increased amount of iron concentration was detected and it is believed that the decrease in floatability could be attributed to the adsorption/deposition of iron oxy-hydroxide species formed as a result of dissolution from grinding media.

In another study conducted to gather information on the nature and extent of possible galvanic interactions between chalcopyrite and grinding media; and the influence of such interactions on the self-induced floatability of chalcopyrite, rest potential measurements for chalcopyrite and high-carbon steel were taken at various pH values and in the presence of air or nitrogen (deaerated). [22]. The results are provided in Table 3 below:

Table 3: Rest potentials of high-carbon steel and chalcopyrite in deaerated (N_2) and air-saturated simulated process water. [22]**.**

pH	Rest potential, mV (SHE)			
	High-carbon steel		Chalcopyrite	
	(N_2)	(air)	(N_2)	(air)
7.5	-527	-226	64	224
9.3	-576	-216	-36	179
11	-667	-281	-128	124

This study concluded that: Low grinding solution pH was detrimental to chalcopyrite recovery when grinding was carried out with high-carbon steel in a nitrogen-purged mill. Higher pH was associated with lower galvanic interactions, and chalcopyrite flotation was seen to improve with increasing pH. In the presence of oxygen, chalcopyrite flotation did not show significant influence by pulp pH.

Chalcopyrite flotation was observed to be higher when stainless-steel grinding media was used instead of high-carbon steel in both air-purged and nitrogen purged grinding environments. The reason for this behavior was due to the relative ease of stainless steel passivation compared to high-carbon steel which leads to less reducing conditions on chalcopyrite from galvanic current and also less ferric precipitate on the mineral surface. The results for this investigation are presented in Figure 15 below:

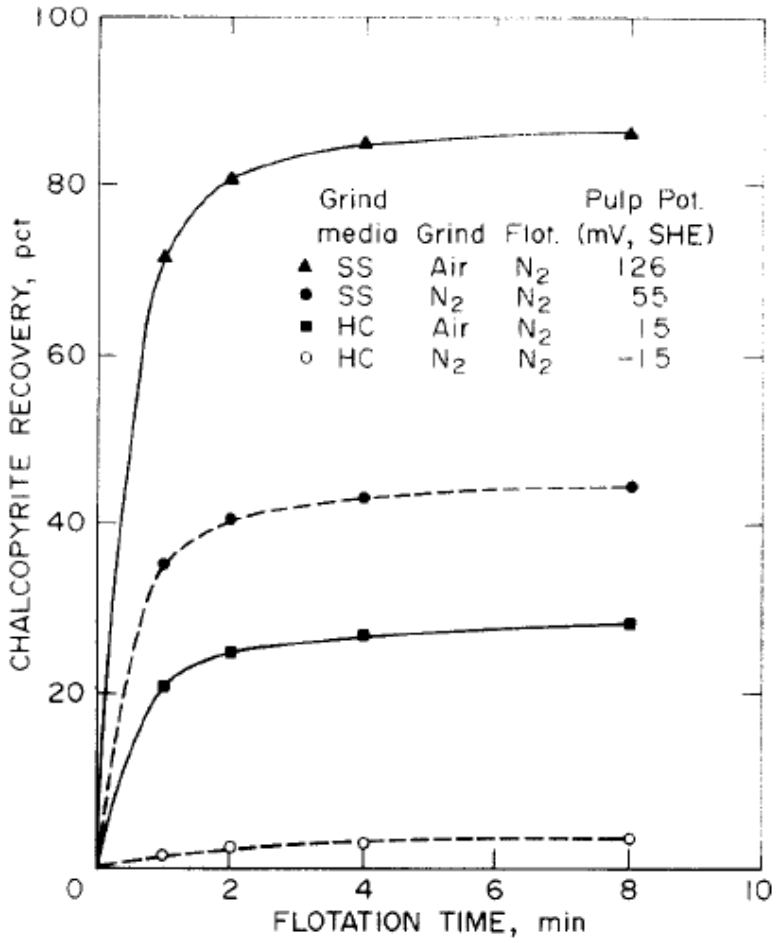

Figure 15: Effect of atmosphere (grinding, flotation) on chalcopyrite floatability after grinding with high-carbon and stainless steel media at pH 11.5 and pH 11.0 in simulated process water. [22]**.**

Recovery was found to be enhanced when 10 minute conditioning was carried out with oxygen rather than nitrogen after nitrogen-purged grinding. This could be thought to reverse the effects of reducing conditions in the mill during grinding. When investigating the effect of surface area it was found out that increasing the chalcopyrite to high-carbon steel electrode area ratio, there was an increase in galvanic current together with the an increase in the combination potential. The results for surface area tests are tabulated below:

Table 4: Combination potential (E_c, mV (SHE)) and galvanic currents (I_g, μA) for the chalcopyrite (CP)-high--carbon steel (HCS) couple in deaerated, simulated process water. [22]**.**

Electrode surface area ratio (Cp:HCS)	Actual surface area (cm^2)	pH 7.5		pH 9.3		pH 11.0	
		E_c	I_g	E_c	I_g	E_c	I_g
1	0.1:0.1	-505	0.25	-516	0.26	-338	0.08
5	0.5:0.1	-485	1.15	-311	0.42	-246	0.14
10	1.0:0.1	-471	2.05	-306	1.11	-216	0.23
20	2.0:0.1	-448	4.75	-316	3.05	-151	0.48

Increased combination potential results in less chalcopyrite polarization in the cathodic direction thereby leading to diminished effects of galvanic interaction on the flotation of chalcopyrite.

2.3.7 Flotation in the presence of a collector

Collectors are organic chemicals used to bind to the selected minerals and make them hydrophobic. They usually have a molecular structure that has a polar and a non-polar group. The non-polar part of the collector is normally a water repellent hydrocarbon chain which is responsible for the hydrophobicity of the mineral-collector complex. The length of the hydrocarbon chain of the collector molecule produces a proportional water-repelling effect. The polar portion of the molecule dissociates in water and the polar head adsorbs onto the mineral surface, binding the collector to the mineral surface. The nature of the reaction with the mineral surface; the strength of the collector attachment; and selectivity of the collector depend on the solidophilic group's composition and structure. [14]. The polar head's compositional structure is mainly responsible for

all of these effects. The bonding of the collector to the mineral can either be chemical in nature (chemisorption) or due to physical forces (physical adsorption). A molecular structure of an oleate collector is displayed in Figure 16 below, showing different parts of a collector molecule.

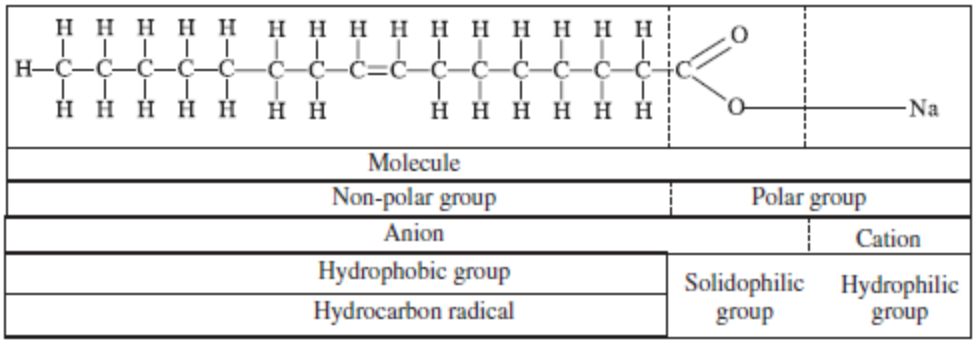

Figure 16: Molecular structure of sodium oleate. [14].

Generally collectors are classified into two main groups: ionizing and non-ionizing collectors. Ionizing collectors are those that dissociate into ions when dissolved in water while non-ionizing collectors are non-polar and generally insoluble in water. Non-ionizing collectors are mainly hydrocarbon compounds and are believed to induce hydrophobicity by covering the mineral with a thin film thus also making the mineral hydrophobic. Ionizing collectors are widely used. This group can further be divided into: anion and cation collectors. This nomenclature is based on which ion of the mineral polar group binds to the mineral surface.

In practice, anionic collectors are the most used and they too can be further broken down into oxyhydryl, sulfhydryl and organic sulfide sub-groups based on the structure of the solidophilic group. Oxyhydryl collectors have a solidophilic group based on sulfo-acid ion while the sulfhydryl collectors' solidophilic group contains bivalent sulphur and lastly, organic sulfides contain an active sulphur atom but no other electron donors. Amphoteric and chelating collectors are other groups belonging to the anionic collector group. The general classification of collectors is displayed in Figure 17 below:

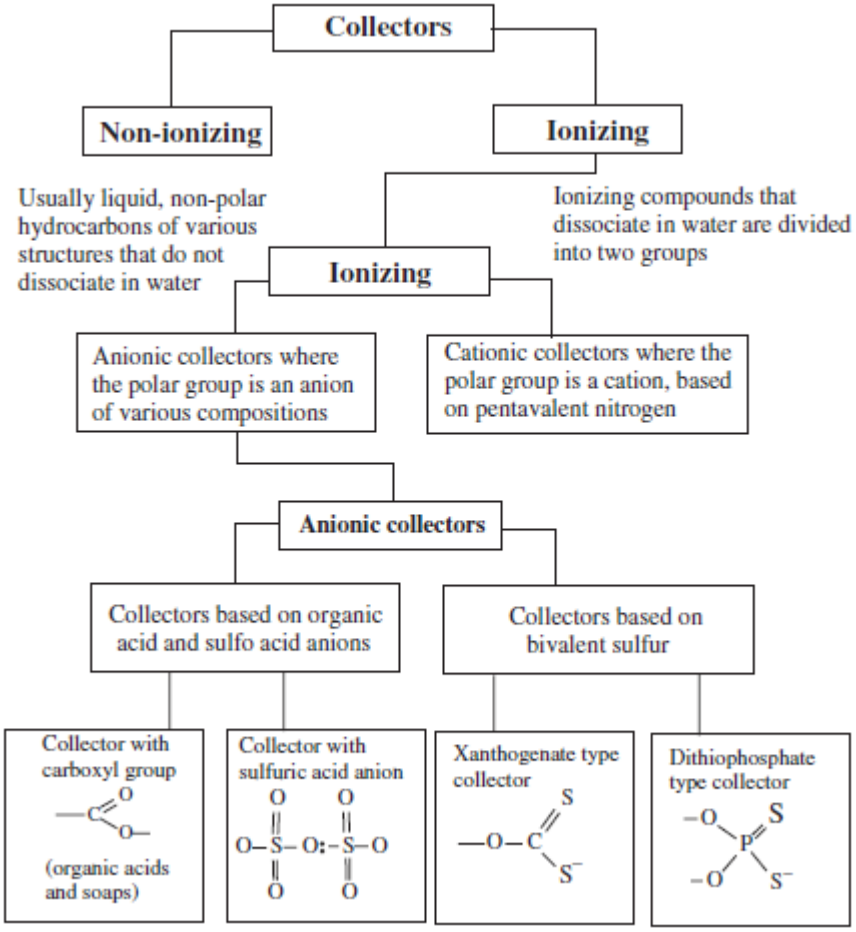

Figure 17: Classification of flotation collectors. [14].

For the purpose of this study, only sulfhydryl collectors will be considered. This group is sometimes referred to as thiols. The most studied under this class for sulphide mineral flotation are sulphur and nitrogen derivatives of carbonic acid, most importantly, xanthates. Xanthates are formed when a carbonic acid's two oxygen atoms are replaced by sulphur and an alkyl group replaces one hydrogen atom as shown below:

$$HOCOOH \rightarrow ROCSSMe \qquad \textbf{(2 - 11)}$$

Where Me = K^+, Na^+, or H^+

A more in-depth discussion of xanthates will follow in latter parts of this literature review.

2.4 Xanthates

Xanthates are the main collectors used in the flotation of sulphide minerals therefore this study will focus exclusively on them. They are anionic, sulfhydryl collectors and they are normally used as potassium or sodium salts. The water solubility of alkyl xanthates decreases with the increase in hydrocarbon chain. In the flotation pulp, xanthates normally dissociate and depending on the pulp conditions, a number of xanthate species form. These species include: xanthate ion (ROCS2), monothiocarbonate ($ROCOS^-$), xanthic acid (ROC2H), carbon disulphide (CS2), and dixanthogen (ROCS2S2COR). Several reactions involving the decomposition process of xanthates are held to be more relevant to flotation systems. These reactions are listed below:

Hydrolysis of xanthate ion:

$$K^+ + ROCS_2^- + H_2O \rightleftharpoons K^+ + OH^- + ROCS_2H\ (xanthic\ acid) \quad \textbf{(2 - 12)}$$

Decomposition of xanthic acid:

$$ROCS_2H \rightarrow ROH + CS_2 \quad \textbf{(2 - 13)}$$

Hydrolytic decomposition:

$$6ROCS_2 13H_2O \longrightarrow 6ROH 1CO_3^{2-} 13CS_2 12CS_3^{2-} \quad \textbf{(2 - 14)}$$

Oxidation to dixanthogen:

$$2ROCS_2^- \leftrightharpoons (ROCS_2)_2 + 2e \quad \textbf{(2 - 15)}$$

Or,

$$2ROCS_2^- + \frac{1}{2}O_2 + H_2O \rightleftharpoons (ROCS_2)_2 + 2OH^- \quad \textbf{(2 - 16)}$$

Oxidation to monothiocarbonate:

$$ROCSS^- + \frac{1}{2}O_2 \leftrightarrow ROCOS + S^0 \quad \textbf{(2 - 17)}$$

Oxidation to perxanthate:

$$ROCSS^- + H_2O_2 \leftrightarrow ROCSSO^- + H_2O \quad \textbf{(2 - 18)}$$

Hydrolytic decomposition occurs mostly in alkaline solutions, yielding stable decomposition products under those conditions. Since flotation is normally carried out in the pH range between 6 and 12, formation of dixanthogen through oxidation is usually an important reaction in air assisted flotation and the formation of monothiocarbonates could be of interest in cases where hydrogen peroxide is used.

Adsorption Mechanism

Sulfhydryl collectors (xanthates) are known to adsorb onto mineral surfaces through chemisorption at metal ion sites on a mineral surface and through electrostatic/physical adsorption. A model to explain the adsorption of xanthates on sulphide mineral surfaces has been developed and is known as “the mixed-potential model”. [3]. A schematic representation of this electrochemical model is shown in Figure 18.

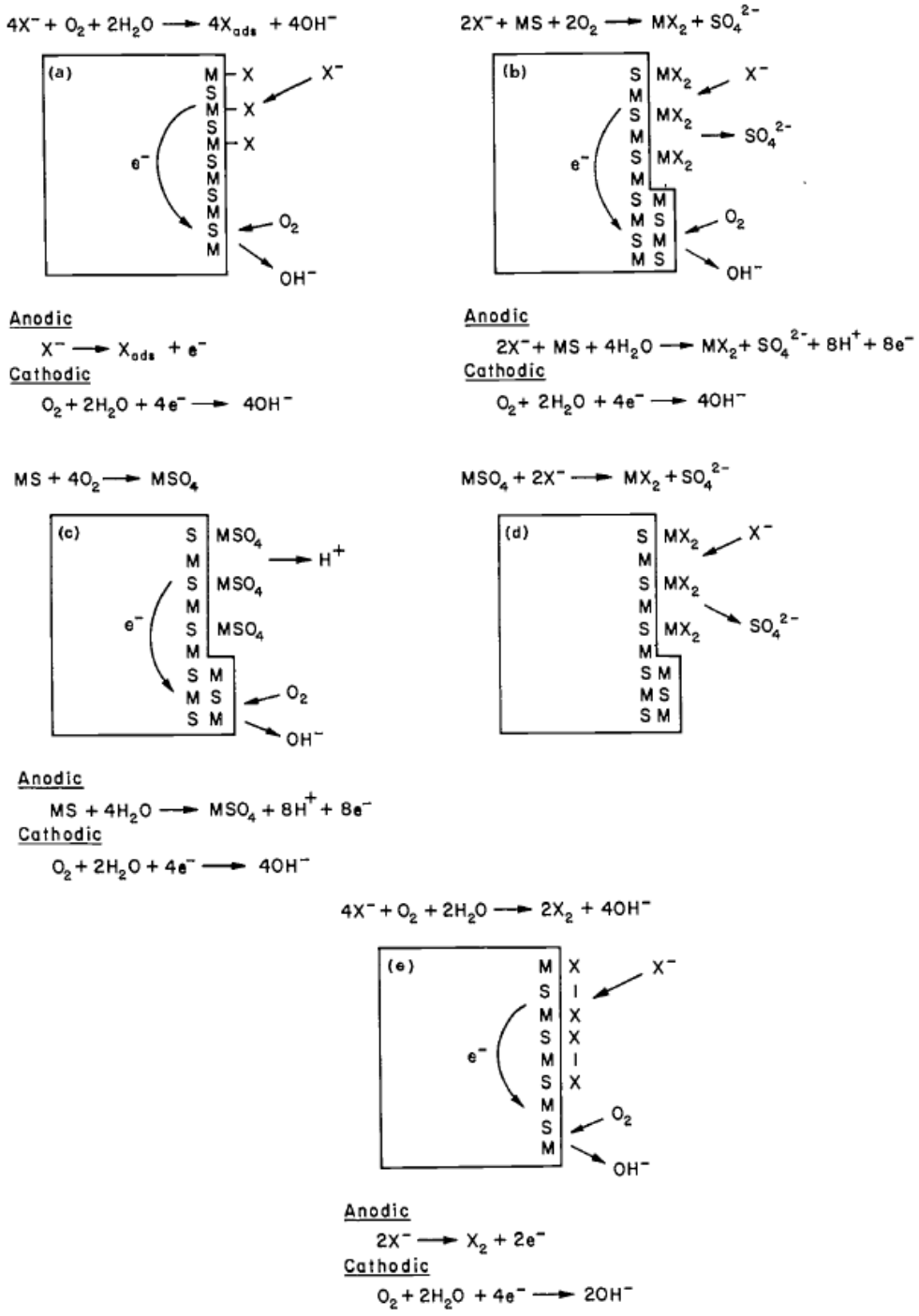

Figure 18: Schematic representation of the mixed potential mechanism for the interaction of thiol collectors with sulphide minerals in which the anodic process is (a) chemisorption, reaction (2); (b) reaction to form metal collector compound; (c & d) the two stages of (b); (c) oxidation of the mineral and (d) ion exchange with collector; and (e) formation of dithiolate. [3].

Electrochemical adsorption of xanthates on a sulphide mineral surface is thought to occur when the reversible potential (Nernst potential) that forms on the mineral surface during xanthate oxidation to dixanthogen is lower than the mineral rest potential. When the reversible potential is higher than the rest potential, xanthate oxidation does not occur and metal xanthate forms and adsorbs to the mineral surface.

In the absence of oxygen, with increasing pH, adsorption of xanthates decreases according to the length of their hydrocarbon chain. In most systems, short chain xanthates are not adsorbed in alkaline conditions. It is postulated that under these conditions, the mechanism by which the collector acts on the mineral is by dixanthogen adsorbing to the mineral surface forming a layer in which physical adsorption takes place on [14].

In the presence of oxygen, xanthates are expected to react with metal ions on the mineral surface, which would have attained a positive charge by donating their electrons to dissolved oxygen during surface oxidation. Therefore, in the presence of oxygen, metal xanthates are thought to be responsible for hydrophobicity in alkaline conditions. In the acidic pH range, xanthates decompose to xanthic acid and the resulting flotation does not rely on the action of the xanthate collector but mainly on induced collectorless behaviour [14].

2.4.1 Chalcopyrite flotation

The flotation of chalcopyrite occurs over a wide range of pH values, between 2 and 12 [15], therefore it is believed that chalcopyrite floats by different mechanism as dixanthogen is known to be generally unstable above pH 11. Industrial practice can have much higher pH values for selective recovery of chalcopyrite in column flotation corresponding to 12.5, representing saturation of slurry with lime in the presence of cyanide to ensure effective depression of nickel-iron sulphides [23]. It is believed that electrochemical oxidation of xanthate to dixanthogen together with chemisorption of xanthate are responsible for the flotation of chalcopyrite over this pH range. Self-induced flotation is still present in this pH region. A study found that dixanthogen alone is not

sufficient to render copper particles floatable [24]. CuEtX was found to form in a direct reaction between ethyl xanthate ion and chalcopyrite above -300 mV vs. SCE. This species was also found to form between -100 and 0 mV vs. SCE from oxidation products of chalcopyrite and xanthate ions. Induced hydrophobicity is held responsible for chalcopyrite flotation under acidic conditions.

2.4.2 Bornite flotation

Flotation of bornite occupies a relatively small place in the literature on flotation of sulphide minerals. Allison et al (1972), [25], included bornite in a study on the role of rest potential in the nature of hydrophobic product formed on sulphide minerals. According to their study, in the presence of a xanthate as collector, the hydrophobic product on bornite is metal xanthate. As such, it does not need formation of dixanthogen to be floatable. Later on, Zachwieja et al (1987), [26], reported that the hydrophobic entity can be cuprous xanthate or dixanthogen if bornite has an oxidized surface. Electrochemical aspects of sulphide flotation have been studied under a variety of experimental conditions. Richardson and Walker (1985), [27], reported that bornite is floatable with xanthate even under low redox potential conditions such as -200 mV (SCE) compared to chalcopyrite, which requires higher potentials, typically greater than -100 mV (SCE). As noted by Allison et al (1972), dixanthogen formation requires a much greater redox potential. For most other sulphide minerals, higher redox potentials are usually necessary. A study was carried out to investigate the electrochemical reduction of molecular oxygen on bornite between E = -0.1 V and E = -0.5 V, at pH 9.2 and 14. [20]. This study found that at pH 14 ethyl xanthate ion is oxidised by HO_2^- to perxanthate and monothiocarbonate ions. This oxidation has detrimental effects on the flotation of the mineral. At pH 9.2, a hydrophobic xanthate layer adsorbs on the bornite surface without much hindrance from the peroxide ions, leading to increased floatability of the mineral. Generally bornite flotation is known to be favoured by alkaline conditions [14].

2.4.3 Galvanic interactions

Bornite and chalcopyrite are known to perform well in flotation at different pulp potentials. As shown in Figure 19, bornite floats at a lower pulp potential than chalcopyrite.

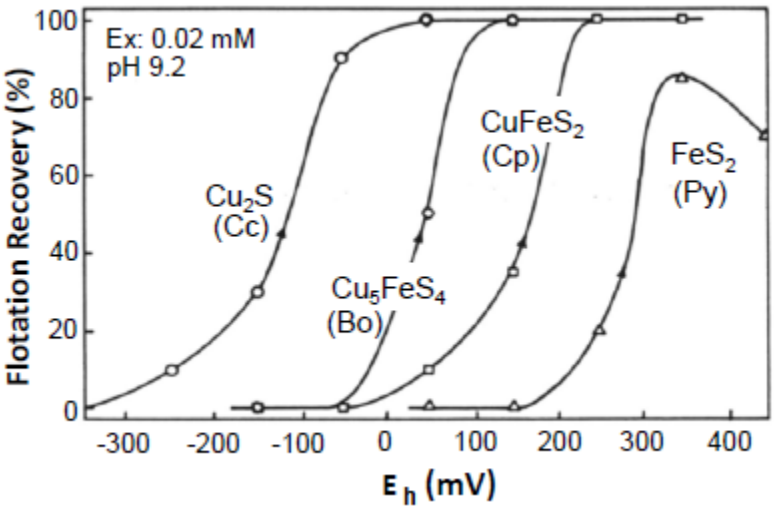

Figure 19: Recovery of sulphide minerals against pulp potential. [27].

The flotation behaviour of these minerals can be used to aid their separation during flotation. Galvanic interactions between these minerals would be expected to promote the recovery of bornite over that of chalcopyrite due to increased anodic behaviour of bornite relative to that of chalcopyrite.

When investigating the effects of having a collector in the flotation of chalcopyrite and sphalerite during ageing under different aeration conditions, it was found that the presence of collector during ageing, minimises the deleterious effects that result from surface oxidation [2]. The collector is assumed to act as an inhibitor in minimising surface oxidation of the mineral being aged.

In a study of the effects of galvanic interactions between bornite and grinding media, it was found that galvanic interactions could be weakened by deoxygenation during grinding [28]. Mild steel showed higher galvanic interactions with bornite and produced more oxidised iron species in the mill discharge leading to decreased bornite floatability.

Less thiosulphate was shown to form in sulphide-xanthate systems in the presence of iron [5]. Adsorption of xanthates on sulphides is known to parallel the amount of thiosulphate formed, which is then exchanged form xanthate by metathesis. Less xanthate adsorption can mean that other mechanisms such as sulphur formation on mineral surfaces can lead to non-selective flotation.

2.5 Leaching

The second part of this thesis focuses on the influence of galvanic interactions on the leaching behaviour of the bornite-chalcopyrite system. Due to wide coverage in the literature, using ferric ions as oxidants in acidic conditions was chosen as the leaching environment. Chloride and sulphate are the more common lixiviants studied in the literature and sulphate leaching was chosen as it had wider coverage in recent studies. Leaching was therefore carried out in sulphuric acid with ferric sulphate as the chosen oxidizing agent. A predominance diagram for a system including both minerals is already presented in Figure 1.

2.6 Bornite Leaching

Several studies have covered the oxidative leaching of bornite under different media. [29], [30], [31]. Leaching in chloride and sulphate media is more common with the choice of oxidants usually being ferric ions or oxygen. Electroleaching has also been studied [32]. The behaviour of bornite under these media is discussed below:

2.6.1 Bornite Leaching in Acidified Ferric Chloride Solutions

Pesic and Olson [30] conducted a study on the leaching behaviour of bornite in acidified ferric chloride solution and concluded that bornite leached in two stages. Temperature was found to play an important role in bornite leaching in the presence of ferric ions. The first stage of bornite leaching terminates after 28% copper extraction and generally temperatures below 40°C were found to be insufficient to carry leaching into the second stage. Figure 20 shows the copper extraction curves for temperatures less than 40°C.

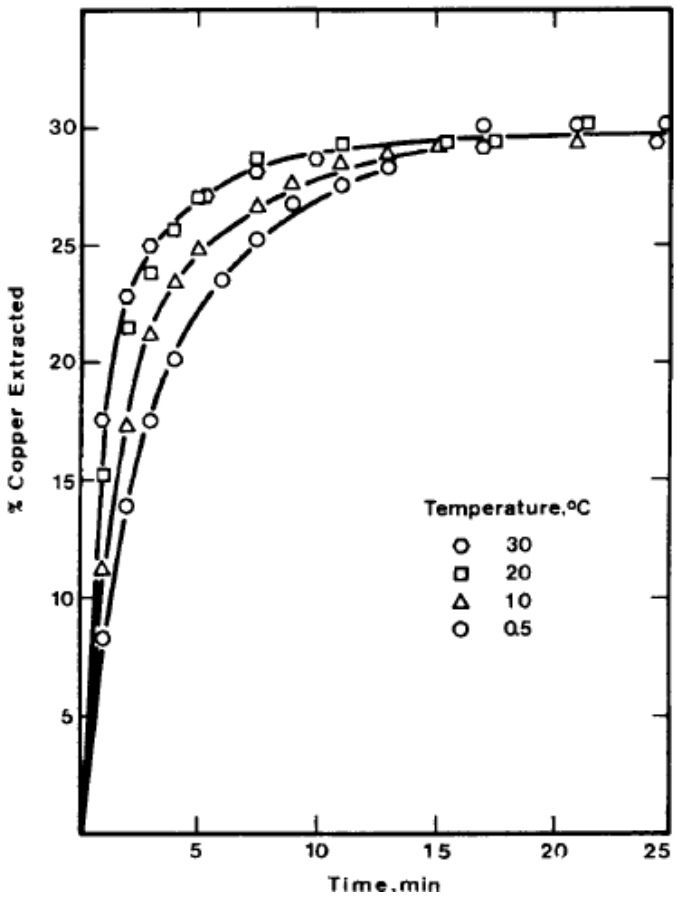

Figure 20: Effect of temperature on first-stage leaching of bornite. [30]**.**

Temperature effect is minimal in this stage and all the curves plateau at 28 % copper extraction. At temperatures above 40°C, bornite leaching proceeds beyond 28 % copper dissolution as shown by Figure 21.

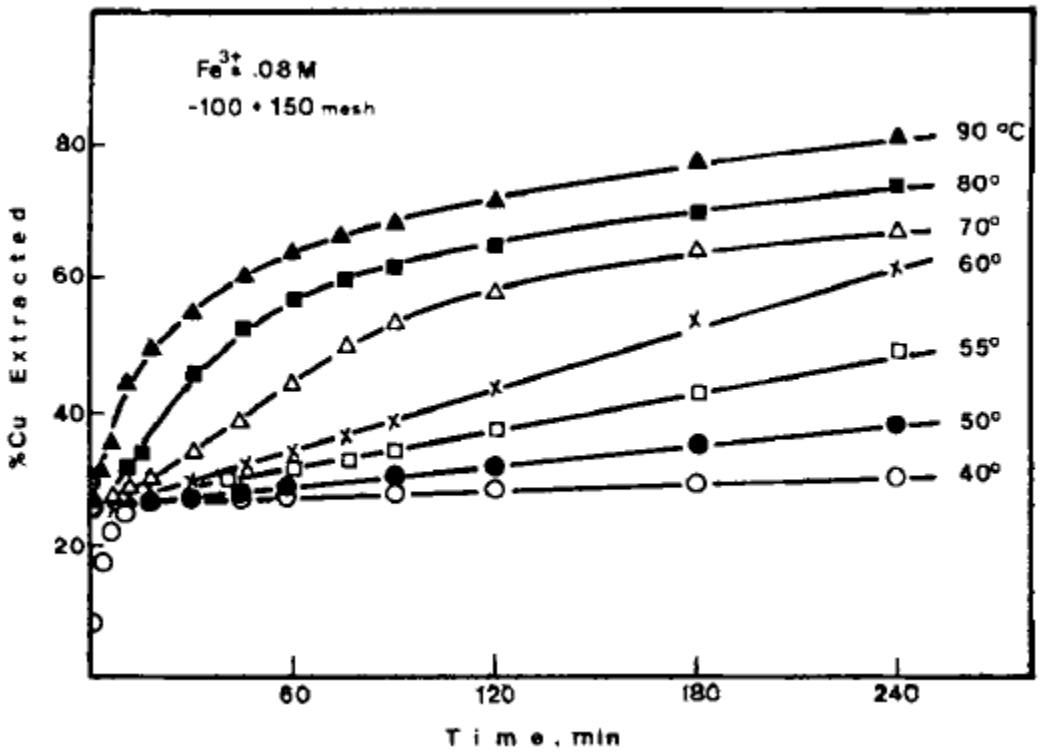

Figure 21: Bornite leaching in acidified ferric chloride at higher temperatures. [30]**.**

For these temperatures stage I leaching is significantly faster than the leaching rate that follows. Temperatures below 60°C show a linear behaviour in the second stage while temperatures above 60°C have faster initial rates which diminish with time.

When evaluating the leaching mechanisms involved in both stages of bornite leaching, the first stage was found to require an activation energy of 25 KJ/mole and showed first order dependence on ferric ion concentration. The postulated overall equation for Stage I reaction was suggested to be:

$$Cu_5FeS_4 + 4Fe^{3+} = Cu_3FeS_4 + 2Cu^{2+} + 4Fe^{2+} \qquad \textbf{(2 - 19)}$$

This reactions is believed to be controlled by diffusion of ferric ions across the solution boundary layer followed by chemical reaction on the bornite surface. Oxidation of labile Cu^{+} ions by Fe^{3+} ions is suggested to be the mechanism characterizing this reaction. This leaves behind Cu_3FeS_4 as

the reaction product and the reaction stops once all the labile Cu^{+} ions have been removed from the bornite structure.

The second stage is characterized by the dissolution of the intermediate product given below:

$$Cu_3FeS_4 + 8Fe^{3+} = 3Cu^{2+} + 9Fe^{2+} + 4S \qquad \textbf{(2 - 20)}$$

The activation energy of 76.8 KJ/mole for this stage is significantly higher than for Stage I. This high activation energy is suggested to be the reason Stage II does not proceed at low temperatures. The reaction rate is shown to depend on temperature. It was observed that the activation energy fell to approximately 30 KJ/mole after 40 % copper extraction. The 28 % to 40 % copper extraction region is taken as a transition stage to Stage II. Nucleation and growth of sulphur on the intermediate product surface is thought to be the rate determining step under these conditions. This sulphur is presumed to provide surface sites for ferric ion reduction leading to the oxidation of copper and iron from the intermediate product. Beyond 40 % copper dissolution, diffusion of species through the sulphur layer that formed in the intermediate stage, is thought to control the rate of reaction.

2.6.2 Bornite Leaching with Oxygen as the Oxidant

Pesic and Olson [29] also carried out another study using oxygen as the oxidant. This study found that acid concentration had no effect on the dissolution of bornite except that the pH had to be low enough to prevent the hydrolysis and precipitation of dissolved iron salts. However, differences in leaching behaviour were observed when comparing sulphuric to hydrochloric acid. Covellite was observed to form on the bornite particles, with an inner phase of Cu_3FeS_4, which was found to have a similar XRD pattern to chalcopyrite. In this study, the rate of iron dissolution was found to be leading the rate of copper dissolution to form CuS and Cu_3FeS_4. A schematic to illustrate the proposed mechanism is shown in Figure 22.

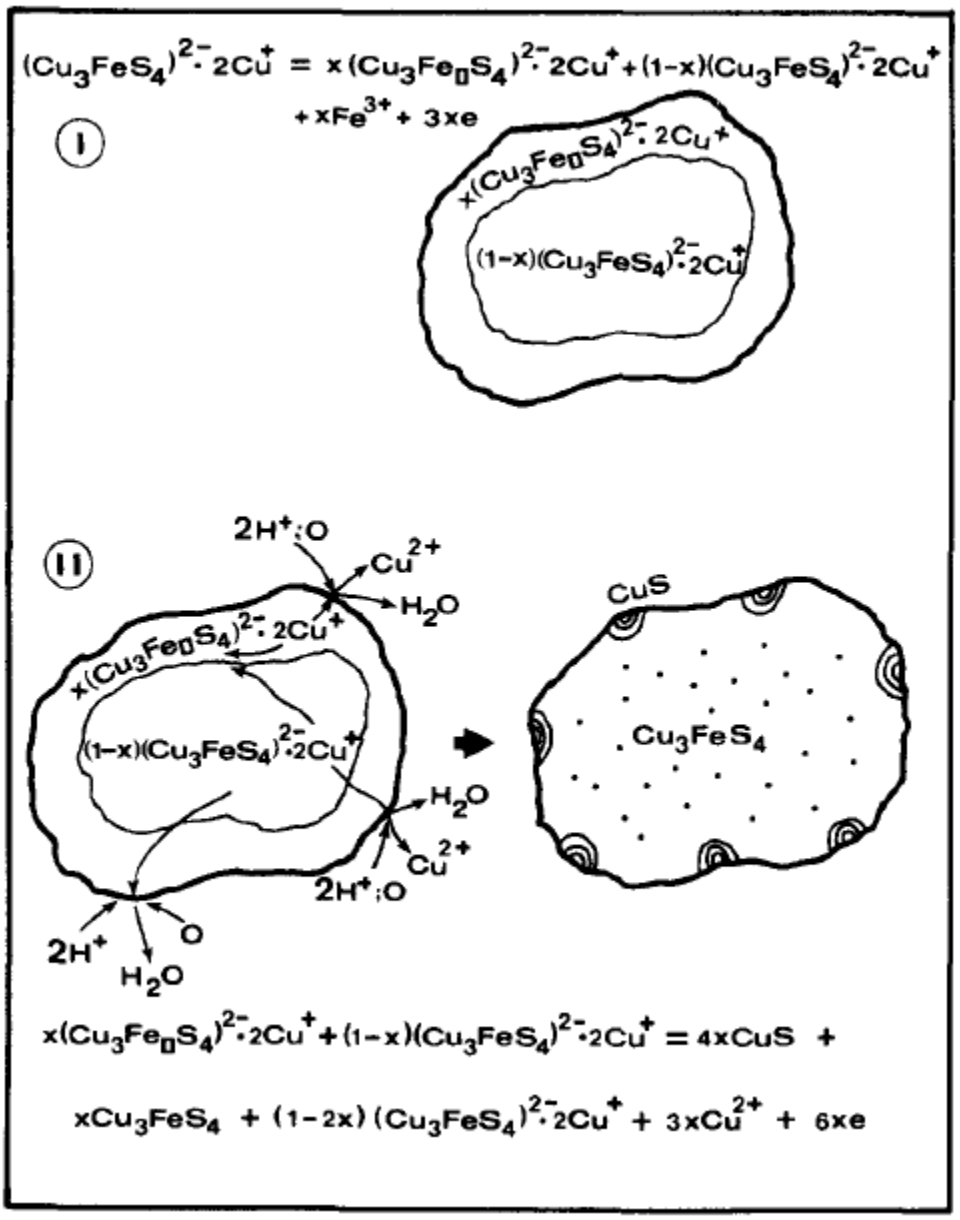

Figure 22: Reaction schematic proposed to explain the formation of covellite and idaite on bornite surface during leaching with oxygen as the oxidising agent. [29]**.**

The authors proposed that iron dissolves into solution leaving behind an iron deficient compound with the bornite structure. Labile Cu^+ then diffuses into vacancies left behind by the dissolution of iron to form Cu_3FeS_4 and covellite on the surfaces of the mineral particles. Some of this Cu^+ is expected to leach into solution. This is expected to explain the observed dissolution of iron leading that of copper and the detected covellite and Cu_3FeS_4 on the particle surfaces.

The same study also experimented with using Fe^{3+} ions as the oxidant. It was observed that when oxygen was used as the oxidant, the reaction rate was at least two orders of magnitude slower than the reaction rate in the presence of ferric ions. It was suggested that the leaching mechanisms under these oxidants were significantly different. Due to the slowness of the reactions beyond 28% copper dissolution, the results of this study were only focused on the initial stage of bornite leaching.

2.6.3 Electroleaching of Bornite

Price and Chilton [31] investigated the anodic reactions of bornite in sulphuric acid using constant current treatment. This study was carried in temperatures ranging between 20°C and 95°C, all of which produced similar potential/time plots as the one shown in Figure 23.

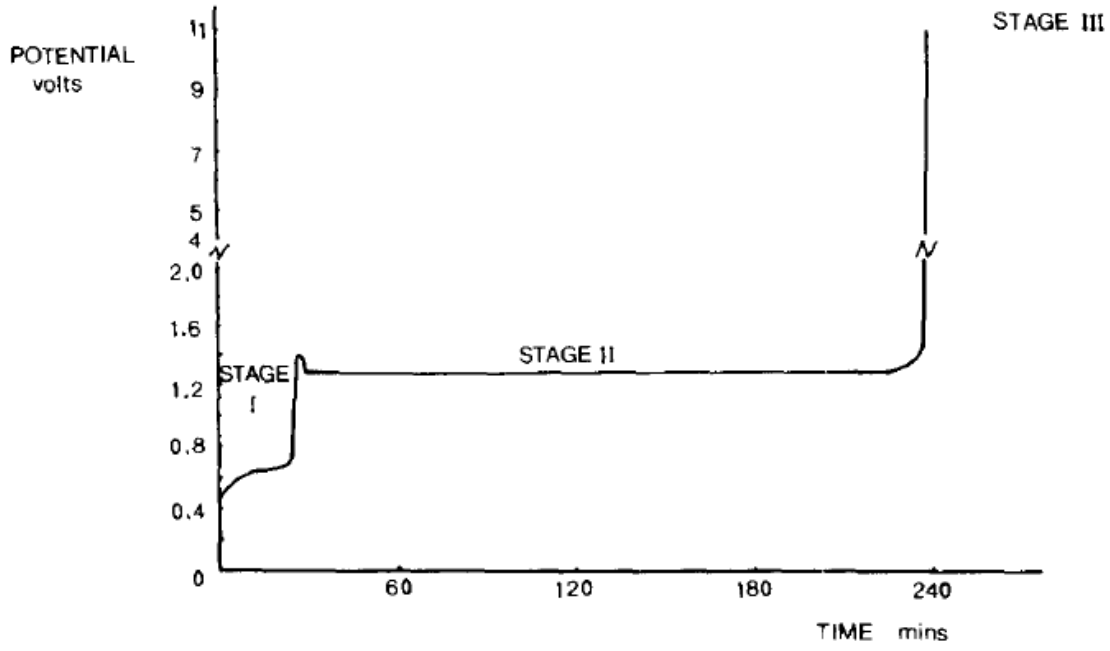

Figure 23: Potential/time relationship for the anodic treatment of bornite in sulphuric acid solution (170 A m^{-2}, 40°C). [31]**.**

Anodic leaching of bornite was seen to occur in two stages across the temperature range studied. Stage I was found to be characterized by preferential copper dissolution from the bornite lattice structure. Equation (2 - 21) was proposed to describe the reaction involved in Stage I.

$$Cu_5FeS_4 = Cu_{2.5}FeS_4 + 2.5Cu^{2+} + 5e^- \quad \textbf{(2 - 21)}$$

A solid-state diffusion process is thought to be the mechanism by which this copper leaches out. It is thought that the interstitial Cu^+ ions from the bornite lattice are the ones that diffuse and oxidise into solution. As the rate of diffusion and oxidation of electroactive species becomes insufficient to maintain the imposed current, a jump in potential occurs at the end of Stage I as shown by Figure 23. This study found that this diffusion step was the rate determining factor for temperatures between 20°C and 40°C. From 65°C to 95°C a chemical step was found to determine the transition time. The rate of Stage I reaction was found to be much greater at temperatures 65°C and above.

An intermediate stage is observed at the end of Stage I, at temperatures between 55°C and 60°C, but a direct determination of the reactions occurring at these temperatures proved challenging.

In Stage II, the diffusion of copper still occurs and depending on the rate of diffusion, the first reaction will still occur. Copper and iron ions as well as elemental sulphur were all found in solution and equation (2 - 22) is proposed to represent the Stage II reaction.

$$Cu_{2.5}FeS_4 = 2.5Cu^{2+} + Fe^{3+} + 4S^0 + 8e^- \quad \textbf{(2 - 22)}$$

This reaction maintains the imposed current at the increased potential that occurs at the end of Stage I. Diffusion is no longer the controlling factor at this stage. Stage II also shows greater reaction kinetics at temperatures above 65°C.

Even though this study provided sufficient explanation for the reaction mechanisms involved in the anodic leaching of bornite, the impact of shrinkage of the copper-deficient bornite lattice and the depletion of bulk copper concentration was not factored into this analysis.

2.7 Chalcopyrite

Leaching of chalcopyrite has been a subject of intensive study, mostly to provide an alternative to pyrometallurgical processing of chalcopyrite. Poor copper extractions from chalcopyrite leaching compared to pyrometallurgy has been an inhibiting factor for hydrometallurgical treatment of chalcopyrite. Passivation of chalcopyrite particles during leaching is mostly held responsible for poor hydrometallurgical performance of chalcopyrite. Different methods of hydrometallurgical treatment of chalcopyrite are discussed below.

2.7.1 Oxidative Leaching

Oxidative leaching of chalcopyrite has been carried out using various oxidizing agents and acid solutions. Oxidizing agents tried in the leaching of chalcopyrite include oxygen, cupric, ferric and chlorate ions in sulphate or chloride medium.

2.7.2 Electrochemical Dissolution

Virmontes-Gamboa [33] conducted a study to compare the electrochemical and oxidative leaching behaviour of chalcopyrite in an attempt to reconcile leaching behaviours observed under with each technique. Normally electrochemical techniques and oxidative leaching experiments are supposed to have similar observations on the anodic oxidation of a material. Reports in the literature of these techniques performed on chalcopyrite often show differing findings, especially on the subject of chalcopyrite passivation. Leaching experiments have often concluded that above certain solution potentials, a sulphide layer forms on the surfaces of chalcopyrite particles thus preventing further reaction by preventing mineral-solution interaction. The nature of the passivating layer in chalcopyrite leaching has been in dispute, with some authors claiming the presence of jarosite [34], [35]. Some electrochemical studies reported no observed passivation behaviour.

Polarization curves produced at different temperatures and acid concentrations showed chalcopyrite to possess an active-passive behaviour. A hypothetical polarization diagram for conductors which exhibit the active-passive behaviour is shown in Figure 24.

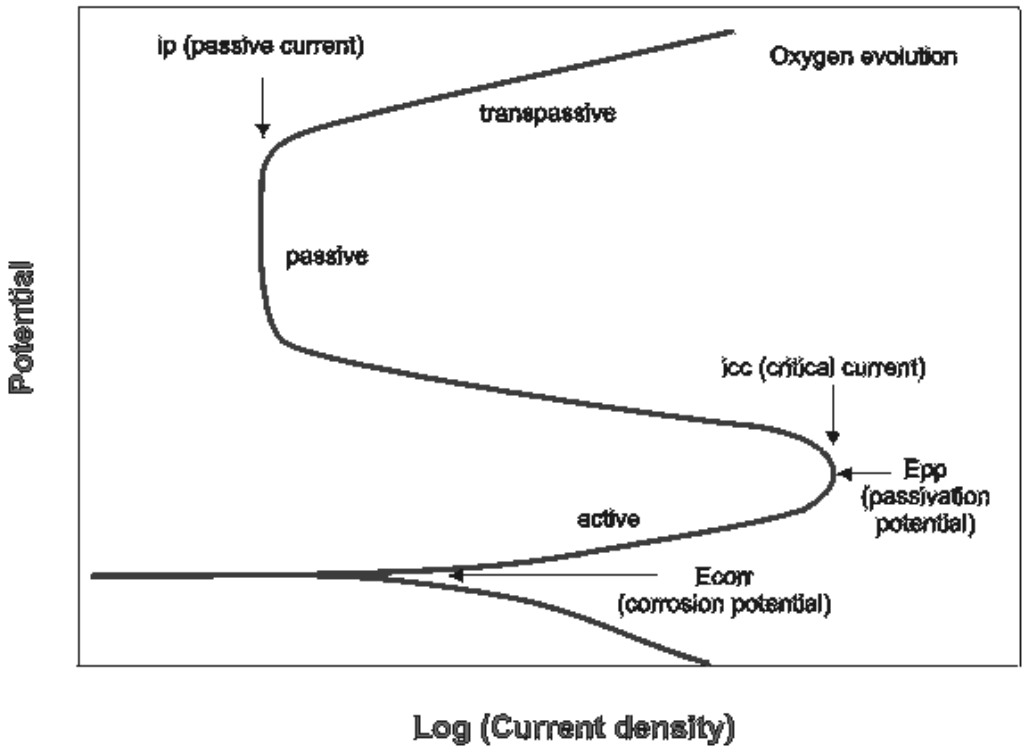

Figure 24: Hypothetical polarization diagram for a passivable system with active, passive and transpassive regions. [36].

In the active region, the rate of corrosion increases with increasing oxidizing potential, but in the passive region the rate of corrosion falls and an increase in potential does not result in a corresponding increase in corrosion. Figure 25 shows the potentiostatic anodic polarization of chalcopyrite in sulphuric acid at different temperatures.

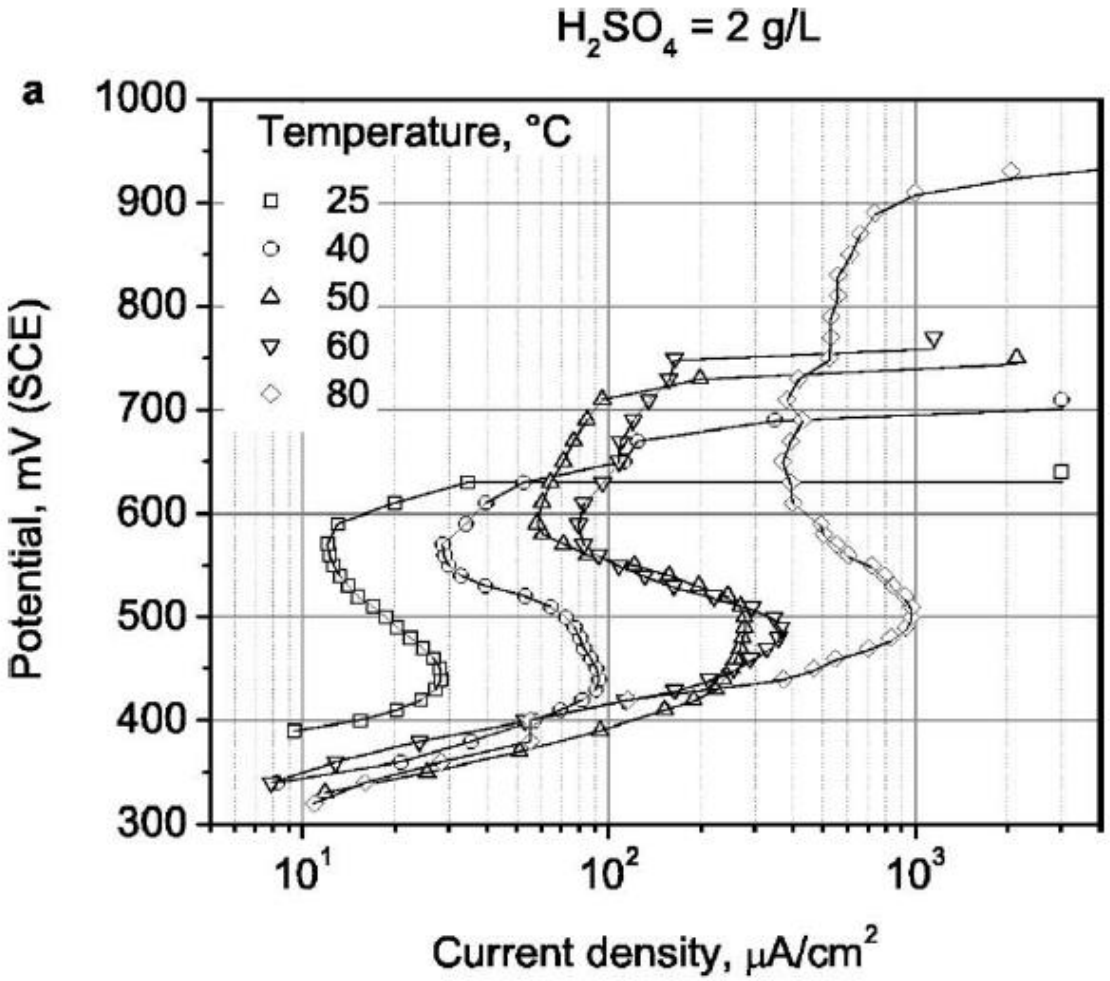

Figure 25: Potentiostatic anodic polarizations of chalcopyrite in 2 g/L sulphuric acid. [33]**.**

The passive and transpassive potentials are seen to increase with temperature from 25°C to 80°C. Current density increased by roughly two orders of magnitude. The transpassive region is assumed to indicate the oxidation of chalcopyrite to covellite. A recent study concluded that the oxidation process of chalcopyrite does not produce covellite as previously thought. [37].

2.7.3 Ferric ion leaching in Hydrochloric Acid

Several studies have been conducted on ferric ion leaching of chalcopyrite in chloride medium. Leaching in this medium is found to be relatively linear and does not show any apparent slowing down with time [38]. This behaviour is displayed in Figure 26 below.

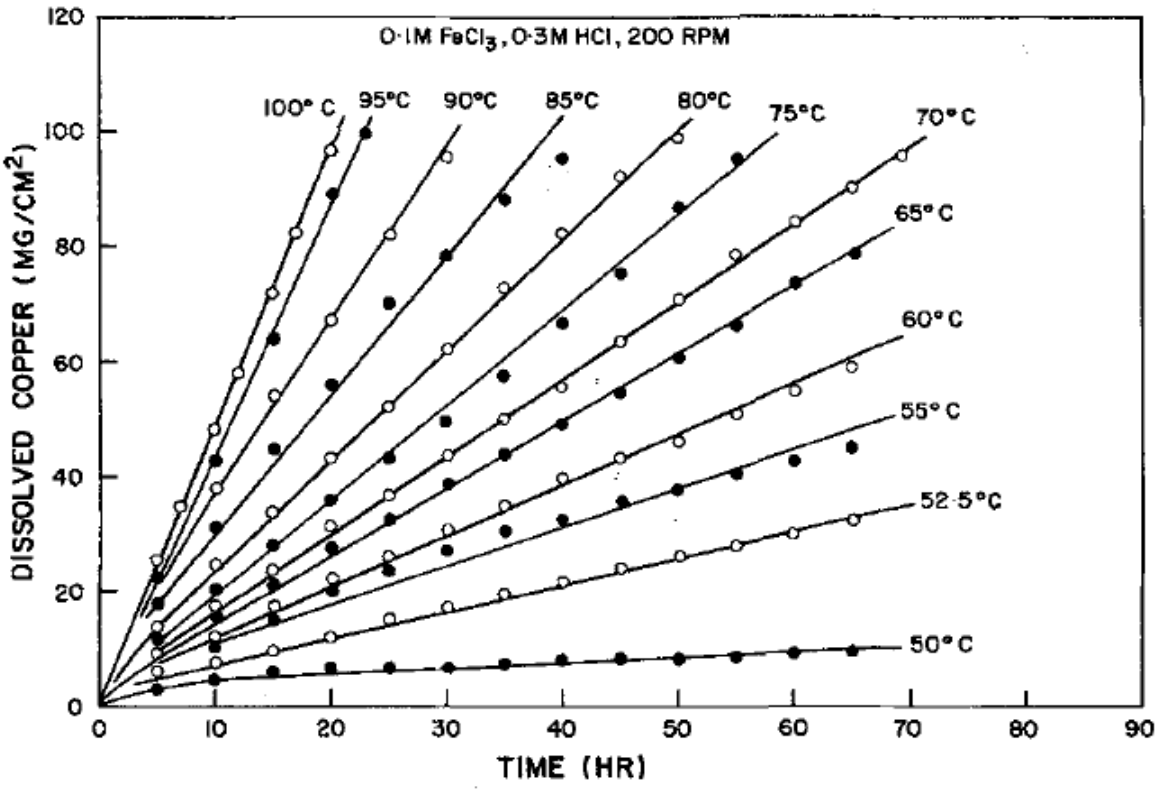

Figure 26: The amount of copper dissolved from disks of synthetic chalcopyrite as a function of time at various temperatures in chloride solutions. [38]**.**

It is quite clear from Figure 26 that the leaching rate is temperature dependent. The activation energy associated with leaching in chloride media is reported to be in the range of 10 to 12 kcal/mole range. The rate of leaching was found insensitive to hydrochloric acid concentration but was found to increase with ferric chloride concentration. The proposed reaction for the leaching reaction involved is shown by equation (2 - 23).

$$CuFeS_2 + 4Fe^{3+} \rightarrow Cu^{2+} + 5Fe^{2+} + 2S^0 \qquad \textbf{(2 - 23)}$$

The presence of sulphate ions was found to have detrimental effects on the leaching kinetics in chloride medium.

A recent study [39] on the reaction mechanisms involving the leaching of chalcopyrite in chloride medium suggests the formation of chlorine-bearing covellite as an intermediate product. The reaction mechanism is thought to involve the transformation of chalcopyrite into chlorine-

bearing covellite ($CuS_{1-0.5x}Cl_x$), Cl-covellite ($CuS_{0.5}Cl_{0.5}$) and nantokite (CuCl). The reactions proposed for the leaching mechanism are listed below.

$$3CuFeS_{2(s)} + 3Cu^{2+} \leftrightarrow 2Cu_2S.CuS_2(s, covellite) + 3Fe^{2+} \quad \textbf{(2 - 24)}$$

Covellite is then converted to chlorine-bearing covellite.

$$Cu_2S.CuS_2(s, covellite) + xCu^{2+} + 2xCl \leftrightarrow Cu_{(2+x)}SCl_xCuS_{2-x}Cl_x(s) + xS(x) \quad \textbf{(2 - 25)}$$

This chlorine-bearing covellite can also morph to nantokite by the following reaction:

$$Cu_3^{+1}S^{-2}Cl.Cu^{+2}S^{-1}Cl(Cl - covellite) + 2Cu^{2+} + 4Cl^- \leftrightarrow 6CuCl(s, nantokite) + 2S(s) \quad \textbf{(2 - 26)}$$

Which could also be expressed as:

$$Cu_2S.CuS_2(s, covellite) + 3Cu^{2+} + 6Cl^- \leftrightarrow 6CuCl(s, nantokite) + 2S(s) \quad \textbf{(2 - 27)}$$

Summing up the previous equations leads to the following overall reaction:

$$CuFeS_2(s) + 3Cu^{2+} + 4Cl^- \leftrightarrow 4CuCl(s) + 2S(s) + Fe^{2+} \quad \textbf{(2 - 28)}$$

The authors used this equation to explain the presence of nantokite and sulphur in the leaching residue. CuCl is known to be soluble in concentrated hydrochloric acid, especially at high temperature. This compound is also known to form complexes with halide ions.

This behaviour of CuCl complexes can be taken advantage of in the leaching of chalcopyrite in chloride media using cupric ions as suggested by Outokumpu HydroCopper™ technology. [40], [41]. The suggested overall reaction for the oxidation of chalcopyrite with cupric ions in chloride media is shown below:

$$CuFeS_2 + 3Cu^{2+} = 4Cu^+ + Fe^{2+} + 2S^0 \quad \textbf{(2 - 29)}$$

The ability of cuprous ions to form complexes with chloride ions helps drive the reaction forward and is also known to increase the oxidation potential of Cu^{2+}/Cu^{+} couple.

2.7.4 Ferric ion leaching of Chalcopyrite in Sulphuric Acid

Leaching of chalcopyrite using ferric sulphate in sulphuric acid is expected to occur by the following reactions:

$$CuFeS_2 + 2Fe_2(SO_4)_3 \rightarrow CuSO_4 + 5FeSO_4 + 2S^0 \quad \textbf{(2 - 30)}$$

Or,

$$CuFeS_2 + 8Fe_2(SO_4)_3 + 8H_2O \rightarrow CuSO_4 + 17FeSO_4 + 8H_2SO_4 \quad \textbf{(2 - 31)}$$

Dutrizac [42] found approximately 94% elemental sulphur formation during ferric sulphate leaching compared to roughly 6% SO_4^{2-} generation. This would imply that equation **(2 - 30)** predominates leaching in sulphate medium. Leaching curves in this medium are parabolic in shape as shown in Figure 27.

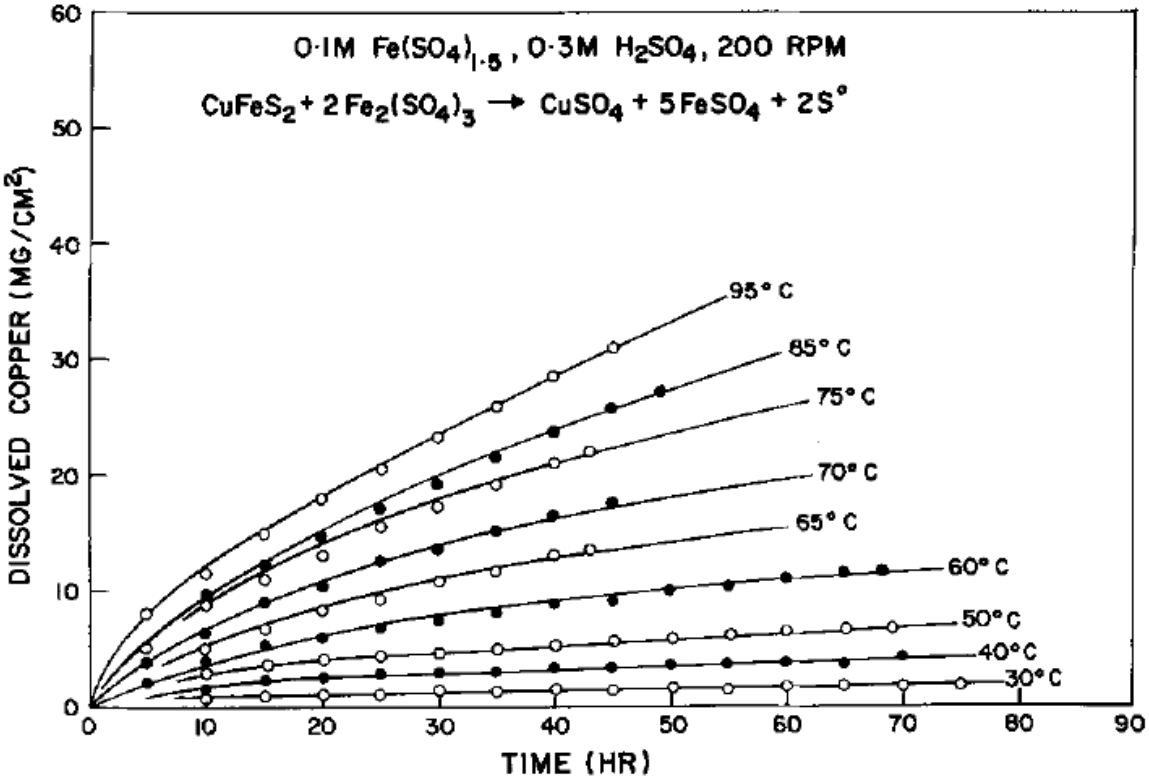

Figure 27: The amount of copper dissolved from disks of synthetic chalcopyrite as a function of time at various temperatures in sulphate solutions. [38]**.**

The rate of leaching in sulphate medium is found to be considerably slower than that of chloride medium. The leaching rate depends on temperature and an activation energy of 15 kcal/mole is associated with the reaction.

2.7.5 Chalcopyrite Leaching using chlorate ion

Chlorate ion is known to be a powerful oxidizing agent with equation (2 - 32) having a high redox potential.

$$ClO_3^- + 6H^+ + 6e^- \rightarrow 3H_2O + Cl^- \qquad \textbf{(2 - 32)}$$

Chalcopyrite leaching rate with chlorate was found to be faster than in cases of ferric chloride or ferric sulphate and is found to be influenced by both chlorate ion concentration and hydrochloric acid concentration [43]. The temperature dependence in this medium is shown by Figure 28.

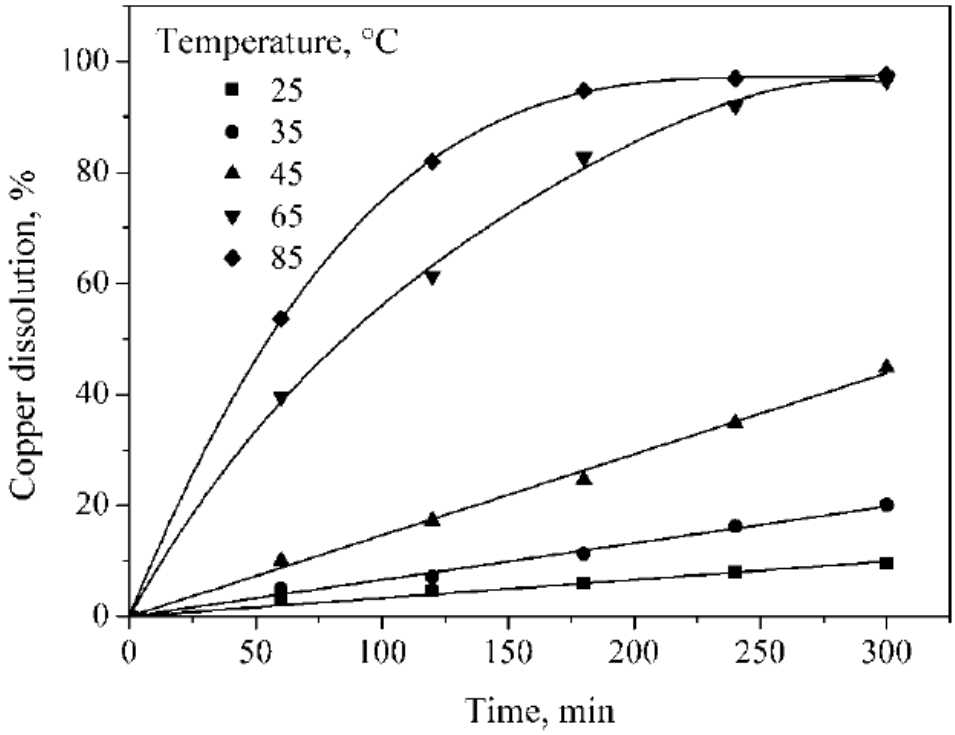

Figure 28: Effect of temperature on copper dissolution: 1M HCl, 0.5M $NaClO_3$, 1.50 gL^{-1}. [43]**.**

The leaching kinetics using chlorate as the oxidizing agent are much faster than for both ferric sulphate and ferric chloride leaching. From 0-180 minutes at 65-85°C temperature range, an activation energy of 28.17 kJ/mole was calculated , while from 180-300 minutes in the same temperature range the activation energy was found to be 0.55 kJ/mole. These results were interpreted to mean that the initial stage was both diffusion and chemically controlled while the latter stage was predominantly diffusion controlled.

2.7.6 Reduction Assisted Leaching of Chalcopyrite

Electrochemical reduction of chalcopyrite to form products that are more amenable to oxidative leaching has been researched. A study of the possibility of continuous electrolytic reduction of a chalcopyrite slurry to form a chalcocite intermediate has been proposed [44]. Equation (2 - 33) was suggested to describe the reaction for this reduction.

$$2CuFeS_2 + 6H^+ + 2e^- \rightarrow Cu_2S + 3H_2S + 2Fe^{2+} \quad \textbf{(2 - 33)}$$

The composition of the iron-free copper sulphide is thought to range between $Cu_{1.8}S$ and Cu_2S. A side reaction involving the evolution of hydrogen gas can lead to applied current inefficiencies. Transient hydrogen species (nascent or monatomic hydrogen) are postulated to form at the cathode and to assist in the reduction of chalcopyrite by the following reaction. [45]:

$$2CuFeS_2 + 2H. + 4H^+ \rightarrow Cu_2S + 3H_2S + 2Fe^{2+} \quad \textbf{(2 - 34)}$$

Reduction of chalcocite further to form elemental copper and hydrogen sulphide was also reported possible.

A model has been proposed in which chalcopyrite leaching is promoted by ferrous ions. [46]. In this model, the initial step was postulated to be the reduction of chalcopyrite by ferrous ion in the presence of cupric ions to form chalcocite, Cu_2S in the following reaction:

$$CuFeS_2 + 3Cu^{2+} + 3Fe^{2+} = 2Cu_2S + 4Fe^{3+} \quad \textbf{(2 - 35)}$$

The formed chalcocite is then oxidised to cupric ion and elemental sulphur by dissolved oxygen or ferric ions by the following reaction equations:

$$2xCu_2S + 8xH^+ + 2xO_2 = 4xCu^{2+} + 2xS + 4xH_2O \quad \textbf{(2 - 36)}$$

and/or:

$$2(1-x)Cu_2S + 8(1-x)Fe^{3+} = 4(1-x)Cu^{2+} + 8(1-x)Fe^{2+} + 2(1-x)S \quad \textbf{(2 - 37)}$$

where x is the mole ratio of chalcocite oxidised by dissolved oxygen in equations **(2 - 36)**.

Improved copper extraction is expected due to the relative ease of chalcocite oxidation compared to that of chalcopyrite. For improved leaching to occur, the solution potential must be greater than the potential at which chalcocite is oxidised to release copper ions into solution, Eox, but lower than the critical potential at which chalcopyrite is reduced to chalcocite, Ec, that is a function of ferrous and cupric ions. This study produced the diagram in Figure 29 to show regions where chalcopyrite leaching would be improved by redox potential.

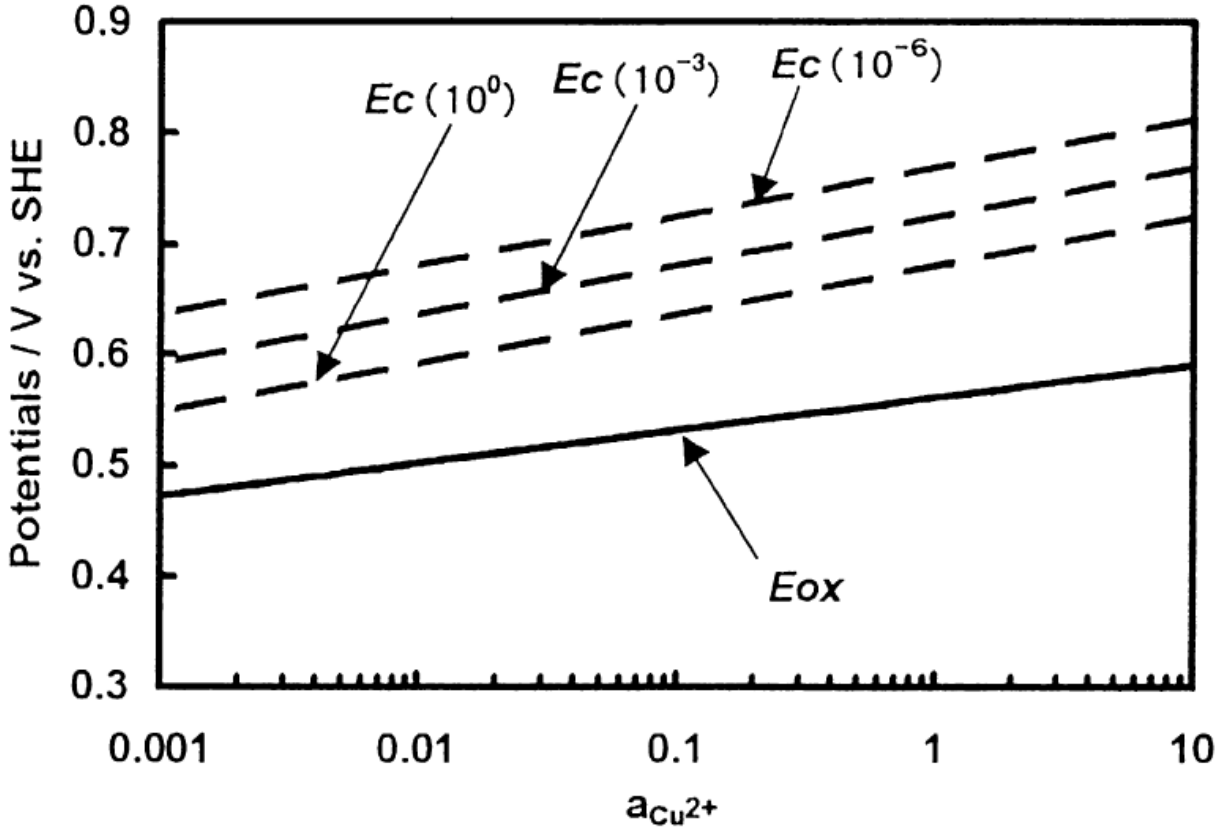

Figure 29: Critical potential (Ec) and oxidation potential of Cu_2S (Eox) as a function of cupric ion activity at 298 K under 1 atm. The numbers in the brackets of Ec indicate activities of ferrous ions. [47]**.**

Potentials above the critical potential would lead to passivation while potentials below the oxidation potential for chalcocite would lead to no copper extraction into solution.

Another study conducted by the same researchers on improving chalcopyrite leaching by redox optimization, [46], calculated the optimum solution potential for reduction assisted leaching of chalcopyrite and produced the graph shown by Figure 30 below.

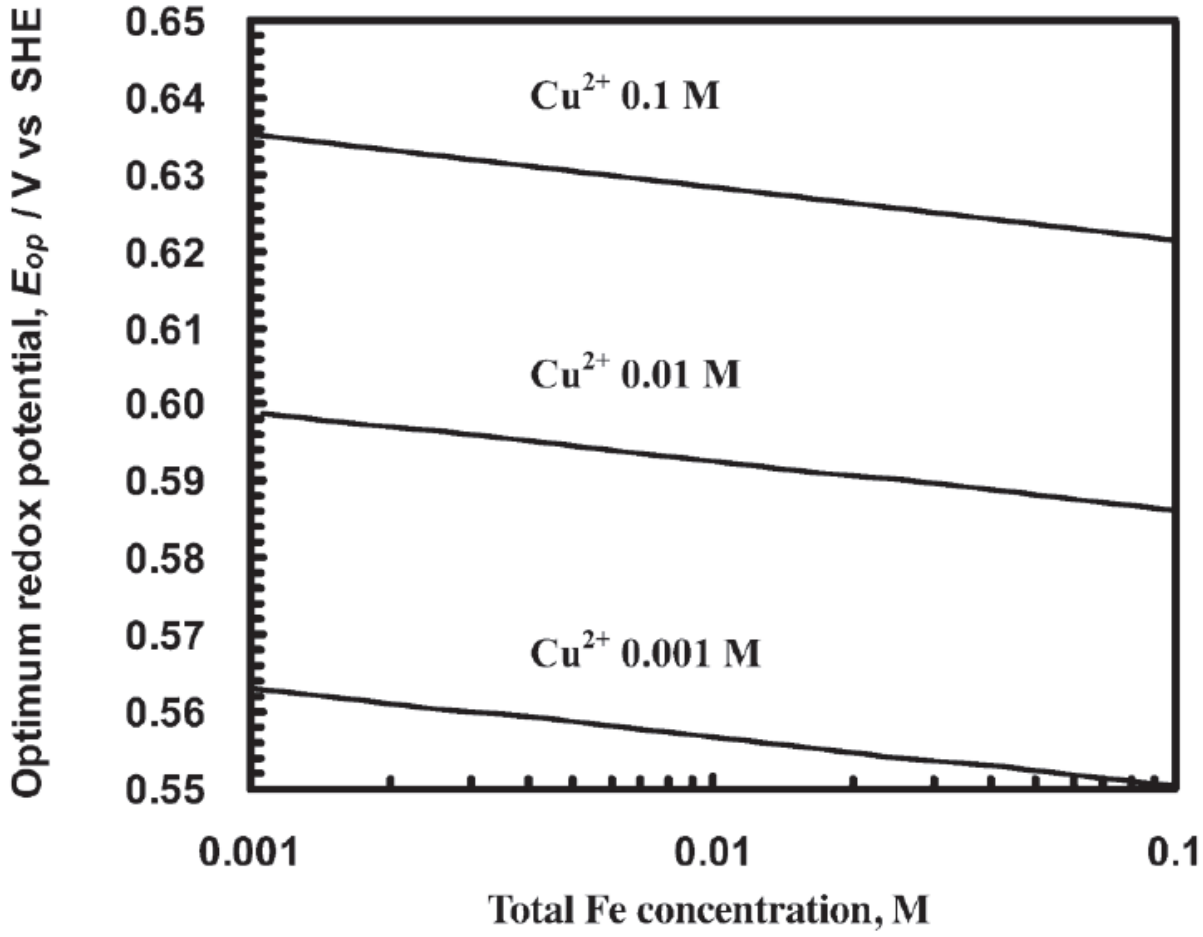

Figure 30: Effects of Cu^{2+} and total Fe concentrations on the optimum redox potential for chalcopyrite leaching. [46]**.**

This optimum redox potential is dependent on cupric ion concentration and iron concentration and was used to show that chalcopyrite passivation does not occur under correct conditions.

2.8 Galvanic Interactions

Galvanic interactions between chalcopyrite and pyrite have been studied by several authors. [48], [49], [50]. Pyrite was found to enhance the leaching kinetics of chalcopyrite by forming a galvanic cell with chalcopyrite. A schematic for this galvanic cell was shown in Figure 13 for flotation in alkaline conditions, and one for leaching in acidic conditions, [48], is shown in Figure 31 below:

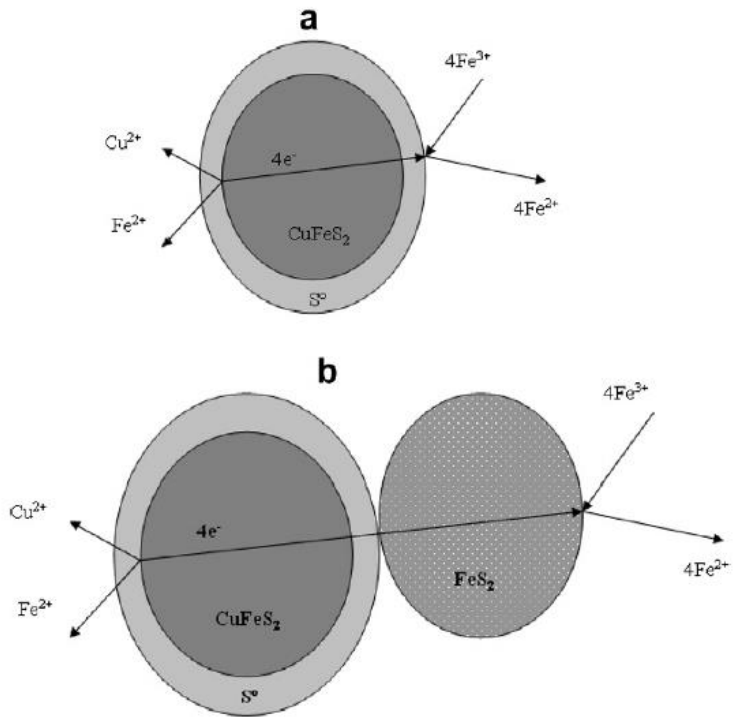

Figure 31: Chalcopyrite leaching by ferric ions: (a) without pyrite; (b) with pyrite. [48].

As shown in Figure 31, under individual leaching cases both cathodic and anodic reactions occur on the same mineral surface while in the presence of pyrite the cathodic reaction is moved to the pyrite surface. The authors who proposed this model, [51], suggest that the cathodic reduction of ferric ions is the rate limiting reaction as opposed to the anodic dissolution from chalcopyrite. Passivation due to sulphur layer formation supposedly inhibits the cathodic reduction process therefore the presence of an unpassivated pyrite surface provides a site for the cathodic reaction to take place, thereby enhancing the reaction kinetics. The galvanic assisted leaching of chalcopyrite is known to produce better results in a certain ORP range as shown by the results in Figure 32.

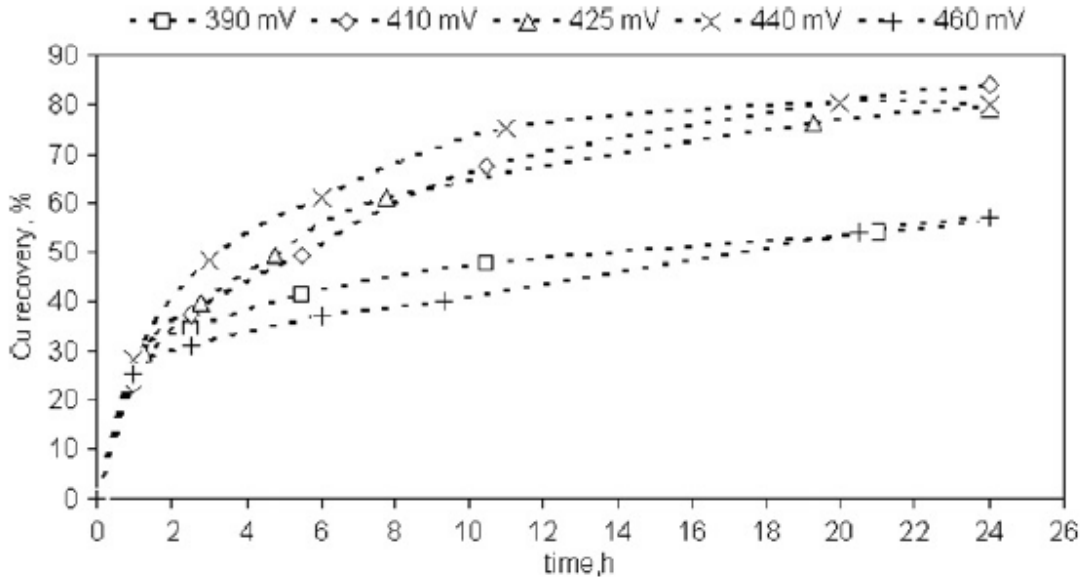

Figure 32: Copper recovery as a function of redox potential. (mV vs. Ag, AgCl) at 85°C [48].

A high ratio of pyrite to chalcopyrite produces better copper extraction but generally there is not much improvement when the ratio was greater than 2.

Different pyrite samples have shown varying results [49]. The presence of silver ions has been shown to improve the leaching results. It is postulated that the sulphur layer that forms as a reaction product insulates the chalcopyrite particles, limiting electrical contact between the two minerals. Silver ions are thought to react with this sulphur layer, increasing its conductivity thereby maintaining electrical contact between the two minerals [50].

2.9 Galvanic interactions between chalcopyrite and bornite

In a study investigating galvanic interactions between sulphide minerals at Lubin concentrator, [52], measurements of sulphide mineral rest potentials was carried out under different solution conditions. Rest potentials of mineral couples were also determined. Rest potentials of sulphide minerals in sulphuric acid purged with argon gas; oxygen gas; and with the introduction of ferric ions are shown in Figure 33 below:

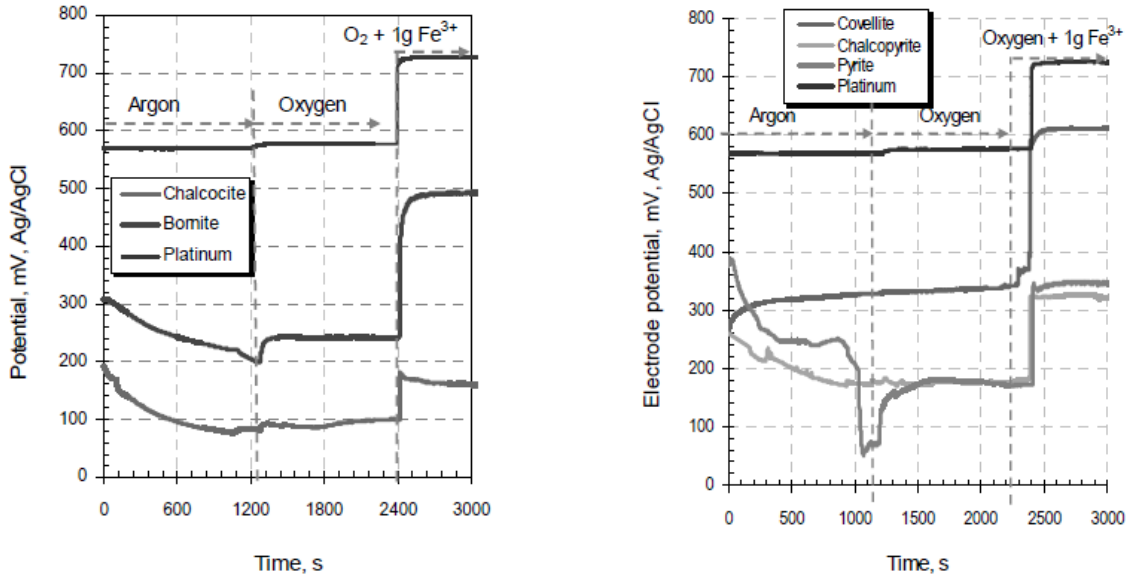

Figure 33: Potential-time plots for sulphide minerals at 70°C. Potential is measured in argon purged solution; oxygen purged solution and in the presence of ferric ions. [52].

In the presence of ferric ions, bornite rest potential is seen to rise to 500 mV while chalcopyrite rest potential is around 320 mV. This observation would mean that bornite would be the cathodic mineral in the bornite-chalcopyrite system. However, measurement of galvanic current showed current flowing from bornite to chalcopyrite. This is shown in Figure 34:

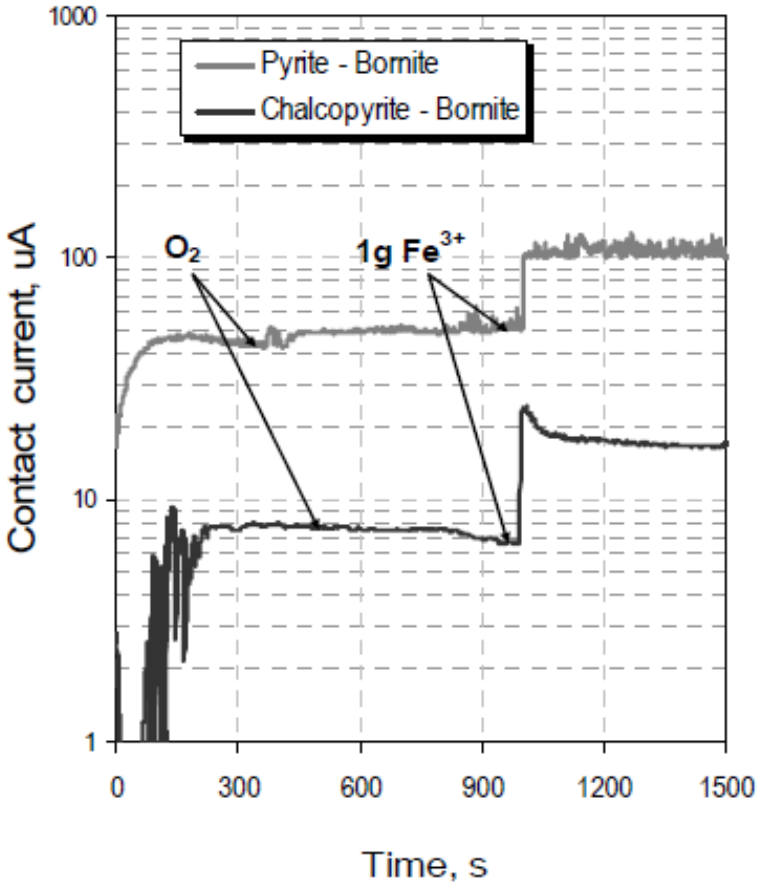

Figure 34: Contact current for galvanic systems: Pyrite-Bornite and Chalcopyrite-Bornite. [52].

The contact current in the chalcopyrite-bornite couple is significantly lower than that of the pyrite-bornite couple.

Given the relatively short time in which the measurements were taken after the addition of ferric ions, less than 10 minutes, these measurements could be only applicable to Stage I of bornite leaching while Stage II could possibly exhibit a different behaviour.

2.10 Experimental Parameters Studied

The purpose of this study was to investigate the effect of galvanic interactions between bornite and chalcopyrite on copper extraction as a result of having both minerals in the same leaching system. Parameters expected to have effects on this galvanic interactions are discussed below:

2.10.1 Temperature

Copper extraction from bornite shows a strong dependence on temperature. Temperatures below 40°C are insufficient to carry the leaching beyond Stage I. Temperature incremental from 50°C to 95°C show a corresponding increase in leaching rate. Chalcopyrite leaching also shows a dependence on temperature in the leaching media studied, with higher temperatures showing improved copper extraction. It is therefore important to observe the effects of temperature on galvanic interactions.

2.10.2 Particle Size and Mineral Ratios

Stage I of bornite leaching shows a dependence on particle size. Particle size and leaching rate have an inverse relationship, which can be interpreted to mean that leaching rate depends on the available surface area. The second stage of bornite leaching does not show dependence on initial particle size. The reason given for this observation is the formation of 1 to 2 micron subcrystallites at the end of Stage I leading to substantial increase in surface area availability for Stage II. [30].

Leaching of chalcopyrite has been shown to depend significantly on the mineral surface area with the following reaction proposed to describe controlling factors on chalcopyrite leaching rate at 75°C, [53]:

$$\frac{1}{S}\frac{dC}{dt} = (2.0 \pm 0.2)\frac{a_{Fe^{3+}}}{a_{H^+}^{0.7}} \qquad \textbf{(2 - 38)}$$

Where S is the relative surface area, C is concentration of Cu in solution, t is the time, a_{Fe3+} and a_{H+} are activities of Fe^{3+} and H^+ respectively.

Galvanic interactions are expected to be influenced by the contact area between the two minerals and the relative amount of one mineral in relation to the other. For this reasons particles size and mineral ratios are parameters of interest.

2.10.3 Acid Concentration

In the presence of an oxidant, leaching of bornite does not seem to be affected by the acid concentration. The main requirement for acid is to provide low enough pH to prevent hydrolysis and precipitation of ferric ions from solution. This is mainly because the leaching of bornite is an electrochemical reaction and relies mostly on the oxidizing agent. Hydrochloric and sulphuric acids are known to have different effects on the leaching behaviour.

Acidity in oxidative leaching of chalcopyrite is known to play a relatively small role in the overall leaching kinetics. Equation (2 - 38) suggests an inverse relation between acid strength and the leaching of chalcopyrite. However, under chlorate leaching, a high concentration of hydrochloric acid was shown to bring significant improvements to the leaching kinetics.
Sulphuric acid is the main medium used in this study and it would be worthwhile studying the effects of varying concentrations of this acid.

2.10.4 Ferric ion concentration and solution potential

The presence of ferric iron has a very significant effect on the leaching of bornite especially when compared to leaching using oxygen as the oxidant. The first stage of leaching has a first-order dependence on the concentration of ferric ion in solution. Even though the dependence of stage II on ferric concentration was not studied, it is logical to assume continued dependence on ferric ions due to the electrochemical nature of the reaction involved. Solution ORP has not been monitored in most of the studies but could have a significant influence on the leaching rate. The solution Eh is to be expected to change with progression of leaching as ferric ions are reduced to ferrous ions thus reducing the ferric to ferrous ion ratio and lowering the solution potential. Some of the ferrous ions would be produced by bornite.

The anodic leaching of chalcopyrite represented by equation **(2 - 23)** is shown to depend on ferric ion concentration. The active-passive behaviour of this mineral also means that solution ORP would be an important parameter.

The influence of ferric ion concentration together with the solution ORP would be important parameters to study and are expected to have significant influence on the galvanic effects involved in this system.

Chapter 3

Experimental

3.1 Flotation

Flotation tests were performed on high grade copper ore from the Minto mine in the Yukon, Canada. This ore was previously crushed to -6 mesh and stored in a freezer to minimise oxidation. A rotary splitter was used to divide the bulk sample into roughly 920 gram charges which were then put in airtight bags and store in freezing conditions. Due to prior crushing, this ore contained a considerable amount of fines, about 20 grams of -400 mesh fraction per ore charge.

3.1.1 Mineralogy

The copper head grade for this ore was 3.3 % and this copper content came from bornite and chalcopyrite which had the following head grades: 3.2 % bornite and 3.4 % chalcopyrite. Sphalerite assayed to around 0.05%. The main non-sulphide gangue minerals were: quartz – which made the bulk of the gangue, clinochlore, albite and orthoclase.

3.1.2 Reagents

There was little variation in the reagents used between the flotation tests. Only one collector type and frother were used.

Collector: 0.1 % solutions of potassium ethyl xanthate, PEX, were used for the collector. These solutions were prepared by dissolving weighed amounts of granulated PEX in distilled water. Fresh collector solution were prepared every time before a batch of tests. This was to minimise possible decomposition of the collector over time. The granulated PEX was stored in the freezer.

Frother: 0.1 % solution of DF250 was used as the frother of choice and was prepared in a similar manner to the PEX solution by weighing concentrated DF 250 and dissolving it in distilled water in a volumetric flask.

Regulators: Lime, CaO, was used for raising the pulp pH. Weighed amounts of lime were normally added to the mill with subsequent additions during tests to maintain the required pH. Nitrogen was used to replace oxygen and also to regulate pulp potential in one of the tests.

3.2 Design of Experiments

The design of experiments shown in Table 5 was carried out to investigate the effects of the parameters listed. In some experiments, slight deviations from procedure were undertaken if deemed necessary. Complete factorial design was not achieved and the analysis was mainly qualitative not quantitative.

Table 5: A list of tests performed and parameters studied

Test #	Fines	Collector	pH
1	1	1	1
2 & 3	1	1	-1
6 & 7 & 8	1	-1	1
9 & 4	1	-1	-1
10 & 12	-1	1	1
5 & 11 & 13	-1	1	-1
	-1	-1	1
14	-1	-1	-1

For fines:

1 - Represents conditions in which the -400 mesh fraction was not removed from the feed.

-1 - Represents tests with -400 mesh fraction removed from the feed

For Collector:

1 - Represents tests in which a collector was used for all concentrates collected

-1 - Represents test in which there was collector addition, mainly for the first three concentrates.

For pH:

1 - Represents tests in which lime was used to adjust pH to a selected value

-1 - Represents tests with no lime addition

3.2.1 Apparatus

Grinding was performed in a Denver laboratory rod mill with a mild steel rods charge of 14 kg. Between experiments, the mill was filled with water containing dissolved lime and sealed airtight to avoid oxidation of the grinding media.

A Denver laboratory flotation machine was used for flotation together with a 2 L flotation cell. The agitation speed of this machine was set to 1200 rpm.

Bench pH and ORP meters were used to monitor pulp conditions during conditioning and flotation. Recovered concentrates were vacuum filtered and dried in an oven at around 250 °C while the tailings were pressure filtered and dried in the oven. A pulveriser or mortar and pestle were used to fine grind coarse samples before assaying.

Atomic adsorption spectrometer, AAS, was used for metal ion analysis – mainly copper, iron and zinc while the ELTRA combustion analyser was used for sulphur content assaying.

X-ray diffraction, XRD, was used for both qualitative and quantitative mineralogical analysis

3.2.2 Procedure

An ore charge was loaded into the mill with a measured amount of tap water to make 65 % solids pulp. Lime was added too if needed. Normal grinding was 12 minutes unless the test required fine grinding. After grinding, the contents of the mill were transferred to a 4 L flotation cell with thorough washing of the mill to ensure no sample losses. The pulp level was increased to a suitable height by additional water and the pulp was aerated and conditioned with the pH and Eh being measured throughout. More lime was added if necessary. Collector and frother addition was

followed with a minute of conditioning before aeration and froth collection. Froth was collected at 30 strokes per minute. Concentrates and tailings were filtered and dried in the oven at 250 °C.

Dried concentrate was weighed before being analysed for metal content and sulphur. About 300 mg of concentrate was dissolved in aqua regia and analysed for metal ions by AAS. The sulphur content was determined by infrared spectroscopy using an ELTRA combustion analyser. Selected samples were analysed for either qualitative or quantitative mineralogical content by XRD spectroscopy.

Several modifications were made to the procedure to test for the effects of varying the following parameters:

3.2.3 Effect of Fines

The bulk sample had a considerable amount of fines from previous crushing which could have led to increased oxidation rate despite the samples were stored under cold conditions. To investigate the effect of the presence of these fines, some tests involved screening out the -400 mesh fraction from the feed before grinding. Wet screening was used for this purpose and the coarse sample was drained and immediately sent to the mill. The fines were recovered and analysed for mineral content to complete the metallurgical balances.

3.2.4 Effect of pH

Lime was used to alter the pulp pH in order to observe the effects of different pulp pH values on flotation. Weighed amounts of lime were added to the mill and if needed, small increments were added to the pulp during flotation. pH and ORP readings were closely monitored by having pH and ORP meter probes in the pulp.

3.2.5 Effect of Flotation with a Collector vs. Collectorless Flotation

Sulphide minerals are known to exhibit natural or induced hydrophobicity in which they float without the aid of a collector, more especially chalcopyrite. To investigate the collectorless

flotation behaviour involved in the bornite-chalcopyrite system tests were carried out in which concentrates were collected without any collector addition but only frother addition to improve frothing. Three collectorless concentrates were collected for each test followed by collector addition to recovery the remaining minerals from the pulp.

3.2.6 Effect of Collector Dosage

Varying the collector dosage in processes where there is competition for collector can aid in exposing the underlying flotation mechanism. Collector dosage was varied between 1 ml and 2 ml of collector dosage per concentrate to investigate the effects of collector dosage.

3.2.7 Effect of fine grinding

Grinding time was increased from the normal 12 minutes to 15 minutes to investigate if it leads to any improvement in selectivity of the process. Grinding time is linked to liberation which could be essential for complex ores but could also generate increased amounts of fines that would adversely the impact selectivity of the process.

3.2.8 Effect of floating with nitrogen

Nitrogen gas was purged into the pulp in the grinding mill before grinding to replace air, more specifically dissolved oxygen, from the pulp. A hose from the nitrogen cylinder was fitted to the aeration valve of the flotation machine and the pulp was also aerated with nitrogen gas at a normal air flowrate and later flotation was carried out using nitrogen in the presence of a collector. When the concentrate recovery diminished, the pulp was aerated with air and flotation with air followed. The pulp Eh was followed closely for this test.

3.3 Leaching

The second part of this research was aimed at investigating the galvanic effects on bornite and chalcopyrite interactions during leaching. The individual leaching behaviour of each mineral was studied under specific leaching conditions were studied before studying the effects of having both minerals in the same system. Experimental conditions were refined for tests following the preliminary tests and variations to original conditions were made to expand on results of interest. Beyond the original factorial design, conditions which showed poor performance were dropped and focus was shifted to tests with better copper recoveries.

3.3.1 Sample Preparation and Minerals

Mineral samples were obtained from Ward Mineral as lumps of high grade chalcopyrite and bornite ore. Chalcopyrite samples were about 96% pure while bornite was around 71% pure based on elemental balances.

3.3.2 Chalcopyrite

The chalcopyrite was ground in a pulveriser and screened into two size fractions: -100 +400 mesh; and -400 mesh fractions. Samples were stored in air tight bags in a freezer to minimize oxidation. Analytical work, including XRD, was performed on these samples to determine their composition and purity. The diagram below in Figure 35 shows the XRD scan of the chalcopyrite sample.

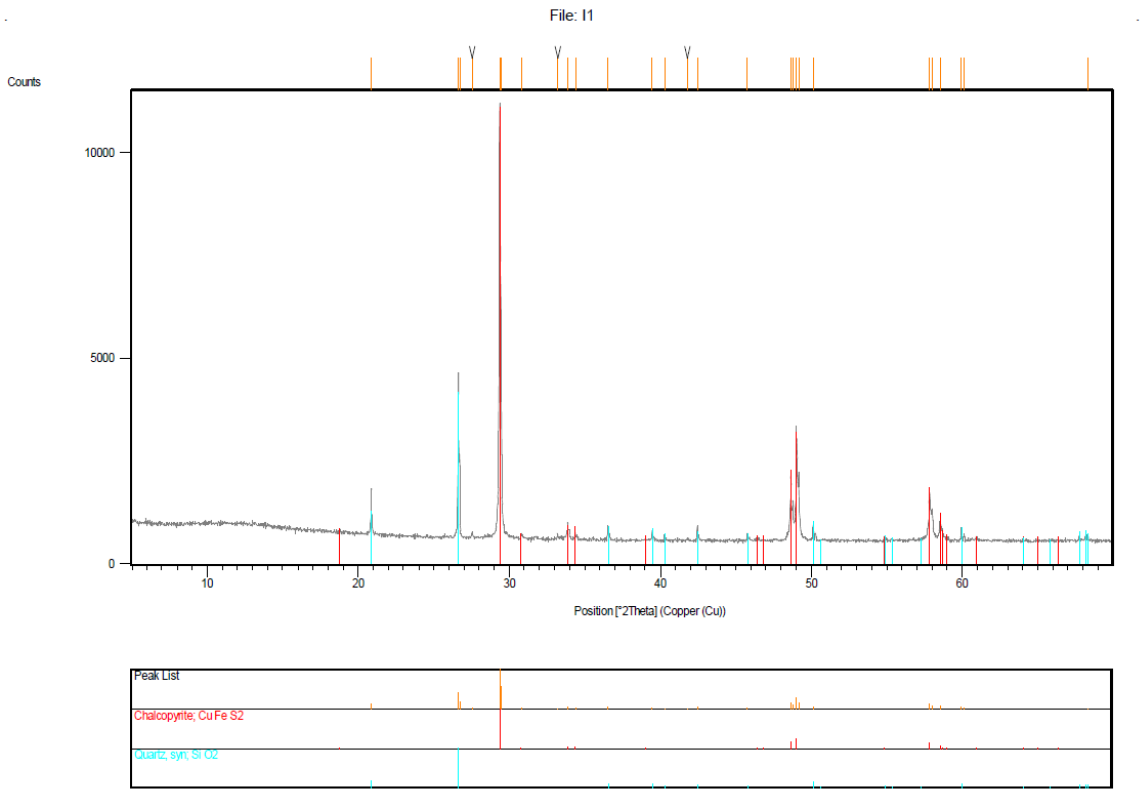

Figure 35: An XRD scan for the chalcopyrite used for leaching experiments.

As seen from Figure 35 that this sample was composed mainly of chalcopyrite which quartz being the only detectable impurity.

3.3.3 Bornite

Bornite lumps were crushed and ground in a pulveriser and screened into two size classes: -100 +400 mesh; -400 mesh samples. These samples were put in air tight bags and stored in a freezer. The composition of these samples was determined through AAS, combustion analysis and XRD. The XRD scan of the -400 mesh sample is shown in Figure 36 below:

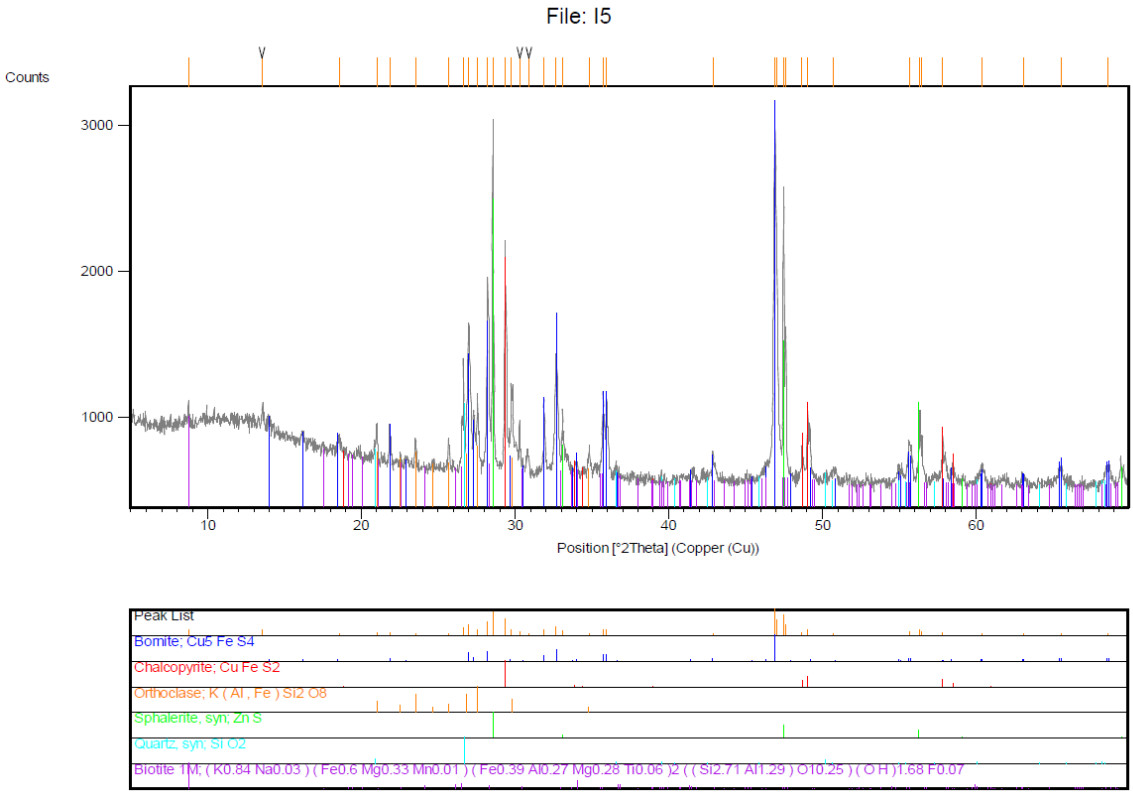

Figure 36: An XRD scan for bornite used in leaching experiments

Based on elemental balances, it was been determined that the samples contained roughly 71% bornite and small amounts of chalcopyrite, 4%, sphalerite, 2%. Quartz was the main non-sulphide mineral with small amounts of biotite and orthoclase.

Ferric (III) Sulfate

Acidified ferric sulfate solutions were used as leaching medium in some of the experiments to investigate the effects of ferric ions in solution on copper extraction. The concentration of ferric sulfate used was 0.1M prepared by dissolving 98.99% lab grade iron III sulfate granules in the warm acid solutions.

Sulphuric Acid

Two acid concentrations were used for the experiments; 0.1 M and 0.5 M sulfuric acid prepared from a 18M stock solution. The acid was prepared in 500 ml or 1 L volumetric flasks depending on the amount of acid needed for the test.

Hydrogen Peroxide

In the tests where redox control was needed, hydrogen peroxide solutions were used to adjust the ORP. These solutions were prepared from a 50 % stock solution.

Ferrous Sulphate

Ferrous sulphate was used together with ferric sulphate to manipulate the initial ORP in tests carried out at lower redox potential.

3.3.4 Experimental Design

A factorial design for the leaching experiments is shown in Table 6. High and low values for the parameters studied are given.

Table 6. High and low leaching parameters

	Low	High
Particle size	-400	-100 + 400
Acid concentration (H_2SO_4)	0.1M	0.5 M
Presence of Fe^{3+}	0 M	0.1 M
Temperature	25°C	90°C

This resulted in a total of 16 tests to be performed on each individual mineral and on mixtures of bornite-Chalcopyrite under the following sample mass ratios: 3:1; 1:1; 1:3.

Follow up experiments to this experimental design included maintaining the solution ORP at the initial value of 750 mV for the entire experiment. This procedure was performed for individual bornite and chalcopyrite leaching together. It was then repeated on the 3:1 and 1:3 mineral mixtures. Chalcopyrite was leached in a leachate solution in which bornite was previously leached for 1 hour to investigate the effect of the changes in solution composition caused by the progression of bornite leaching. 2 g of chalcopyrite was used and the leachate solution was prepared by leaching 1 g and 3 g of bornite samples respectively.

3.3.5 Leaching Apparatus and Procedure

500 ml and 1L conical flasks sealed with sponge stoppers were used as reactors for the experiments. These flasks were immersed in water baths set up by heating water in a large beaker on a hot plate with a magnetic stirrer. Mineral samples were administered once the temperature was steady and stirring was carried out at 750 rpm using a magnetic stirrer. Sample weight ranged from 1.5 g for individual minerals to 4 g for mineral mixtures. A thermometer was always present in solution to monitor the temperature. 5 ml aliquots were drawn at set time intervals to be analysed for copper content.

A pH meter was used to measure the pH at the beginning and end of the experiment while an ORP meter was used to measure the solution potential at both the beginning and end of the experiment. There were experiments where redox potential was controlled which required continuous monitoring of the solution potential. These experiments required addition of hydrogen peroxide to control the solution potential. A micropipette was used to administer hydrogen peroxide solution while an ORP meter monitored the solution potential.

At the end of the experiments, residues were filtered, washed and dried while the pregnant solution was transferred into a volumetric flask and brought to the mark by addition of distilled water. This solution was then analysed for copper content and used as a check for reported copper content from samples drawn during the experiment.

3.3.6 Analysis of Leaching Products

In selected tests the dried residues were weighed and analyzed for their metal and sulfur contents to close the elemental balances. Sulphur determination was carried out on an ELTRA combustion analyser while copper and iron content were determined through AAS. Some solids were also analyzed by SEM to determine the nature of the leaching residue species. Some of the leaching residue was sent for XRD analysis. Residues which were high in elemental sulphur were washed with carbon disulphide to get rid of the elemental sulphur before running an XRD scan.

Determination of Copper Extraction

The pregnant solution was only analysed for copper content only as high background amounts of iron and sulphate ions in solutions made it difficult to measure the dissolved iron and sulphate formation with any degree of accuracy.

Chapter 4

Results and Analysis

Flotation

Results from the flotation tests are presented in this section. Only selected recovery data is presented here. Raw data from flotation tests, metallurgical balances and grade curves are shown in Appendices A and B. The effects of selected parameters are presented below:

4.1 Effect of Fines

The ore used in these experiments had been in storage for a considerable amount of time, in which oxidation of the fine fraction could have occurred despite the ore being stored in freezing conditions. As such it was important to test for the effects of removing these fines from the feed. Tests in which the -400 mesh fraction was removed from the feed and those in which this fraction was not removed were performed under similar conditions to test for the influence of the fine fraction on flotation performance of each mineral. The tests for flotation with a collector and collectorless flotation are presented in Figure 37 and Figure 38 below.

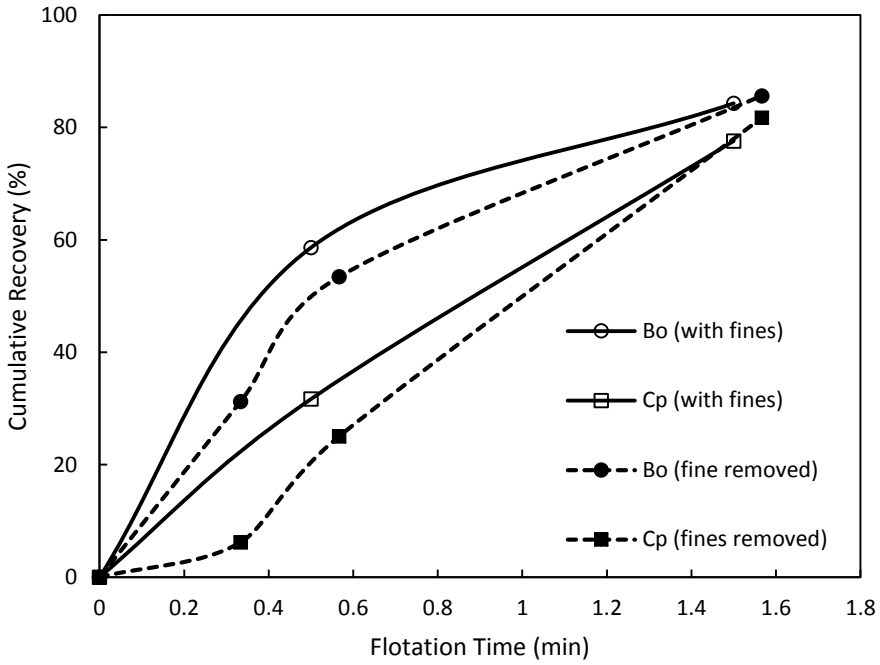

Figure 37: Effect of removing the fines from the feed before flotation with a collector and no lime addition

From the graphs in Figure 37, there is a slight downward shift in the recovery curves for both minerals when fines are removed from the feed before grinding. This downward shift seems to be more prompt before 0.6 minutes, especially for chalcopyrite. After 1.5 minutes there not difference in the recoveries from the two tests.

Results in which collectorless flotation was performed to test the effect of fines are presented in Figure 38. In one of the test, collectorless flotation was carried out for only 1.5 minutes as compared to 2.5 minutes in the other two tests.

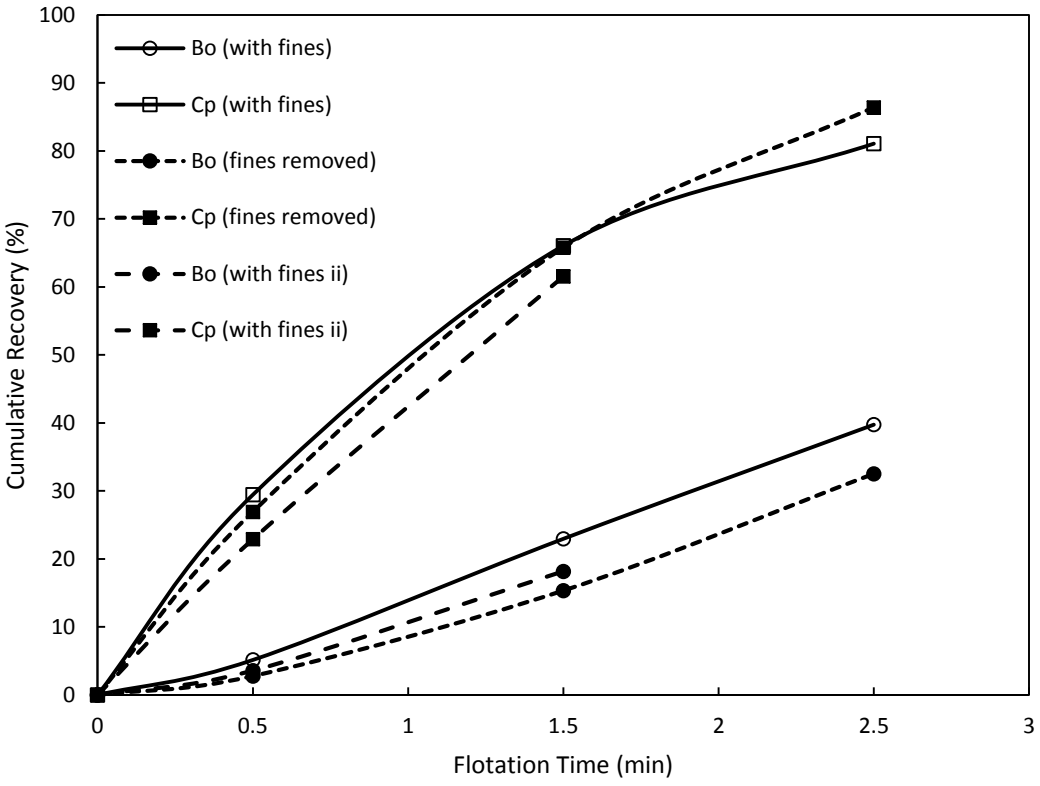

Figure 38: Effect of oxidised fines on collectorless flotation with no lime addition

From the results shown in Figure 38, the difference in collectorless flotation with or without fines in the feed is minimal. There is less variance in chalcopyrite recoveries. Bornite recovery from feed with fines seem to be slightly less in tests in which fines were removed. This discrepancy, however small, is worth mentioning.

4.2 Effect of a Collector

Flotation performance between tests in which a collector was used and those in which flotation was performed without the addition of a collector are compared in Figure 39 and Figure 40 below. Figure 39 shows the results between collector and collectorless flotation carried out with no lime addition with the -400 mesh fraction removed from the feed.

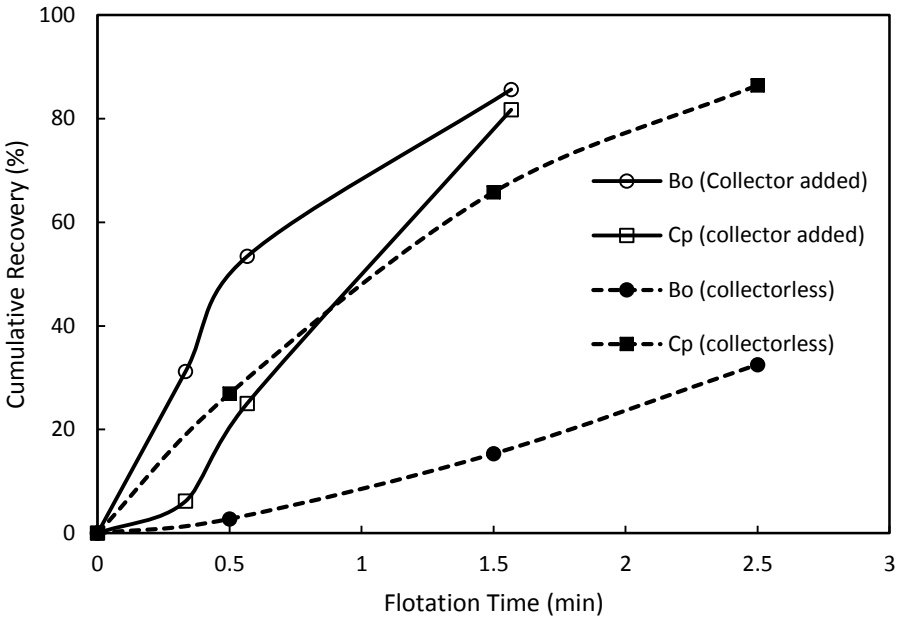

Figure 39: Effect of collector vs. collectorless flotation with no lime addition and with fines removed

It is quite visible that flotation kinetics when a collector is used are much faster than in the collectorless case. There is notable decrease in chalcopyrite recovery for the first two concentrates while bornite recovery experiences a marked improvement. Another important observation from these tests is that the order of floatability is reversed. Flotation in the presence of a collector favours bornite recovery while collectorless flotation favours chalcopyrite recovery. Collectorless flotation shows an overall higher degree of separation between chalcopyrite and bornite.

Results from similar tests as the ones described earlier but with a lower collector dosage, 1 ml/conc. vs. 2 ml/ conc. are presented in Figure 40. Fines were not removed from the feed.

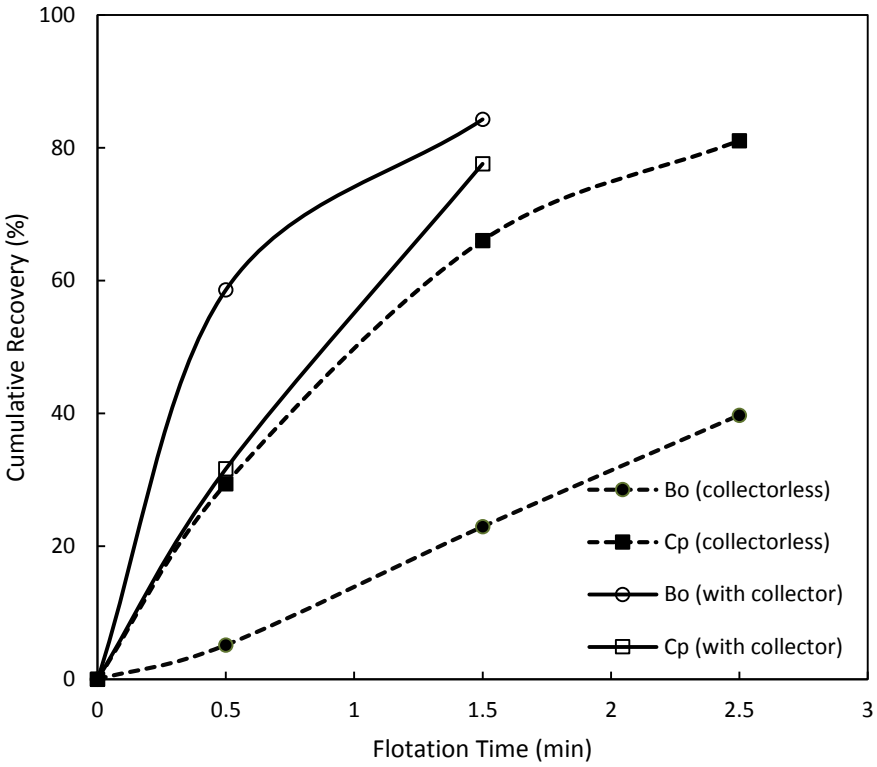

Figure 40: Effect of collector vs. collectorless flotation at pH 8 without the removal of fines

The trend observed here is similar to that observed when fines were removed. There is general increase in recoveries more especially in the case of flotation of chalcopyrite in the presence of a collector.

4.3 Effect of Collector Dosage

Collector dosage can give an insight into interactions in a system if selectivity is brought about as a result from competition for collector. Figure 41 shows results from tests comparing the effect of collector dosage. There was no lime addition in these tests and the feed was not screened to remove the fine fraction. Dosages compared were 2 ml versus 1 ml 0.1 % potassium ethyl xanthate additions per concentrate recovered and these tests were compared against collectorless flotation results.

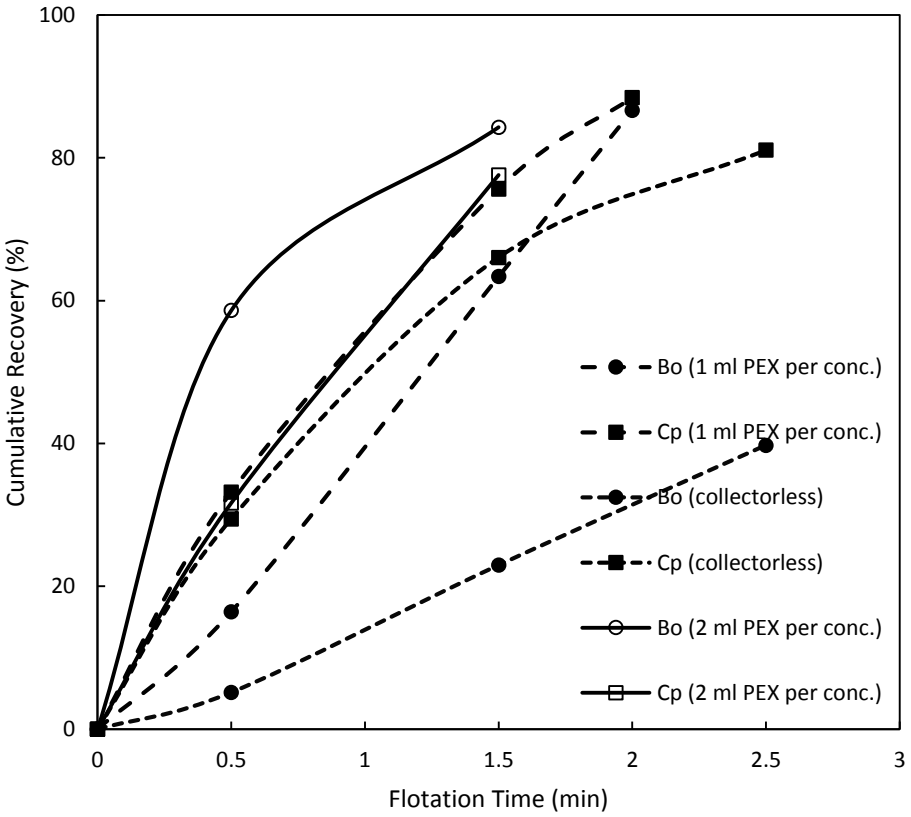

Figure 41: Effect of collector dosage with no lime addition and no fines removal

It is clear from Figure 41 that higher collector dosage produced faster kinetics, especially for bornite flotation in the first concentrate. An important observation from these tests has to do with the reversal of flotation order of the two minerals. Chalcopyrite recovery is only slightly affected by the collector dosage amount. The chalcopyrite recoveries from both test are close to each other and show virtually no variation. On the contrary, the floatability of bornite is significantly affected. When less xanthate was used, the recovery of bornite was less than that of chalcopyrite but increased to more than that of chalcopyrite with higher dosage

4.4 Effect of pH

pH is often an important factor in the flotation behaviour of most minerals and is known to affect interactions between minerals and collectors. All of the experiments were carried out in neutral to alkaline conditions, pH 7.7 to pH 11. No tests were carried out in acidic conditions.

Figure 42 to Figure 44 show the results from tests which were carried out to investigate the effects of pH on flotation behaviour of minerals in this system.

Results from collectorless flotation without removing the fine fraction from the feed are shown in Figure 42. These tests were carried at pH 7.7 (no lime addition) and pH 9.

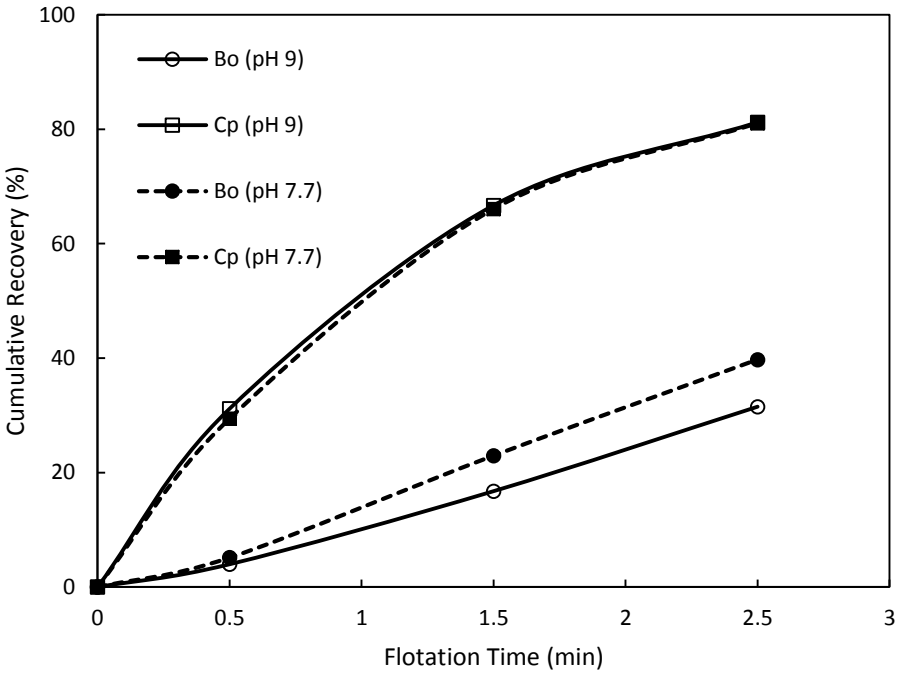

Figure 42: Effect of pH under collectorless flotation without the removal of fines

The effects of pH between the chosen pH values is quite minimal, with the chalcopyrite recoveries mirroring each other almost perfectly. There is a slight discrepancy in the bornite recoveries with the lower pH showing slightly better recoveries.

Effects of pH on collectorless flotation are shown in Figure 43. The fine fraction in the feed was not removed and this time the highest pH value was pH 11, compared to pH 9 in the previous case.

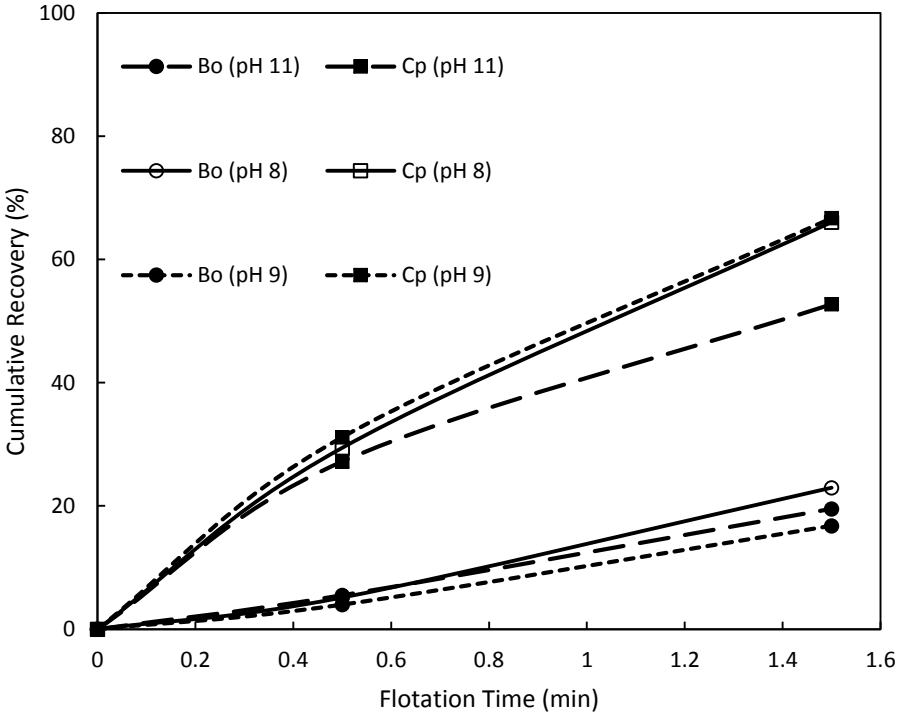

Figure 43: Effect of pH on collectorless flotation without the removal of fines

Results from these tests follow each other very closely especially at lower pH values. Chalcopyrite recoveries for pH 8 and 9 are identical while the overall chalcopyrite recovery at pH 11 after 1.5 minutes is lower than that for low pH values. Bornite recoveries are much lower than those for chalcopyrite and there is no notable departure in behaviour from any of the tests.

Figure 44 shows the results from tests performed with the use of a collector and with the feed screened for the -400 mesh fraction. These tests used similar pH values as the ones in Figure 42 except there was no test performed at pH 11.

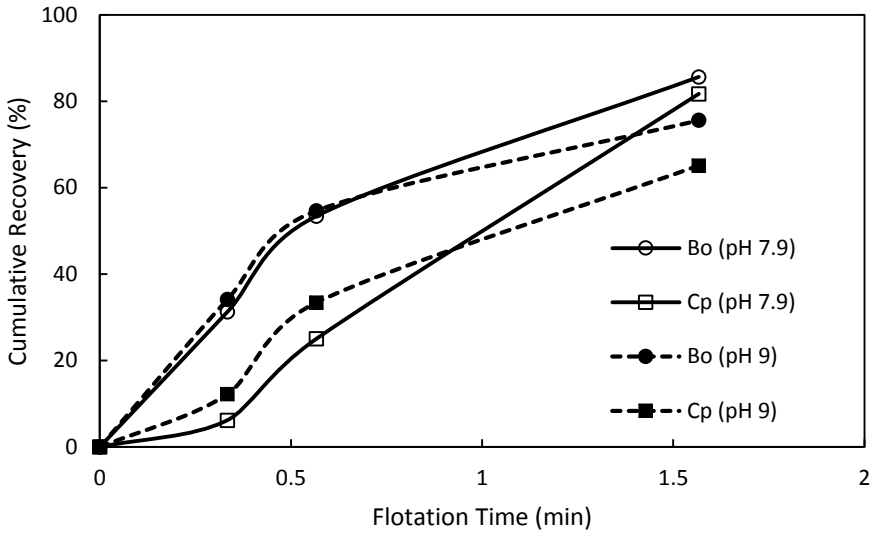

Figure 44: Effect of pH on xanthate flotation with fines removed

With the collector addition there is a reversal of floatability, with bornite flotation taking precedence over chalcopyrite flotation. This behaviour has been seen before when comparing flotation with a collector against collectorless flotation as shown in Figure 40. The pH effect seems to be minimal on bornite recovery while chalcopyrite flotation seems lower at low pH for the first two concentrates leading to slightly better selectivity for pH 7.9. The selectivity for both tests is diminished after 1.5 minutes and low pH shows higher overall recoveries.

4.5 Effect of Extending Grinding Time (Fine Grinding)

Extended grinding can be related to improved liberation, which could improve the selectivity in complex ore flotation. Grinding time was extended by three minutes from 12 minutes to 15 minutes and the results are presented below.

The grinding effects in tests where the fine fraction from the feed was removed and no lime was added are shown in Figure 45. Xanthate was used in all concentrate collected.

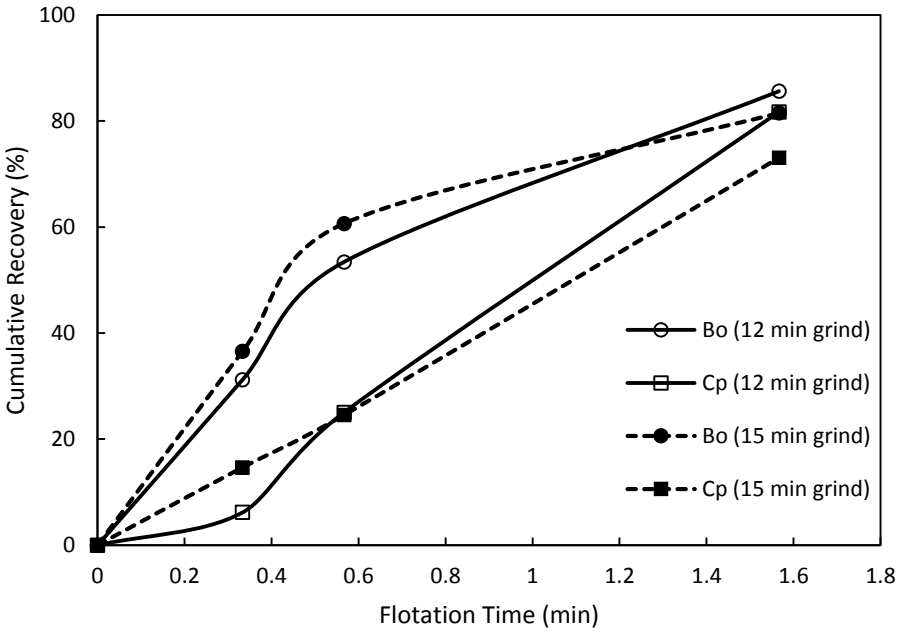

Figure 45: Effect of grinding with xanthate flotation, no lime addition and with fines removed

As seen from Figure 45 the first concentrate favoured 12 minute grinding time in terms of selectivity, but this was reversed in the second concentrate with fine grinding producing slightly better selectivity. Considering that the results are cumulative, the individual improvement in selectivity for the second concentrate is significantly higher as shown by the grade data from Figure AA - 11 and Figure AA - 13 in the appendix.

Similar test to the ones presented in Figure 45 were performed under collectorless conditions. Due to relatively low recoveries of bornite under collectorless flotation, the collectorless flotation stage was followed with addition of xanthate to recovery the remaining bornite. It was therefore important to monitor the effect of fine grinding for this "scavenging" stage.

Results from collectorless flotation are presented in Figure 46. Fines were removed in the 15 minute grind and no lime was added.

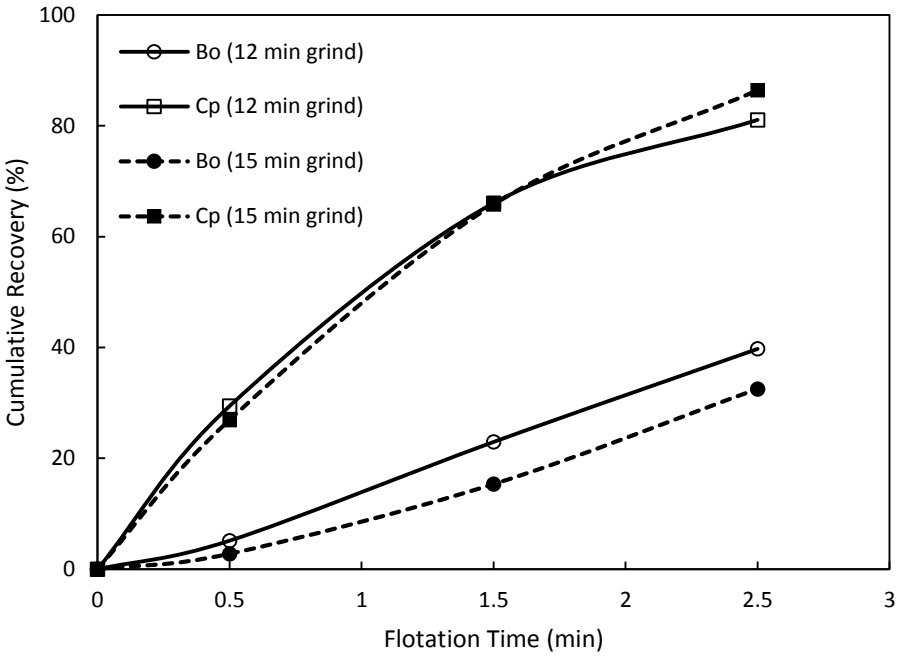

Figure 46: Effect of fine grinding in collectorless flotation with no lime addition

There seems to be no standing out behaviour from these two tests and the relatively lower bornite recovery could be linked to fines removal from the feed.

Figure 47 shows results from the flotation of collectorless flotation (Figure 46) tailings with xanthate additions. 1 ml PEX was added per concentrate.

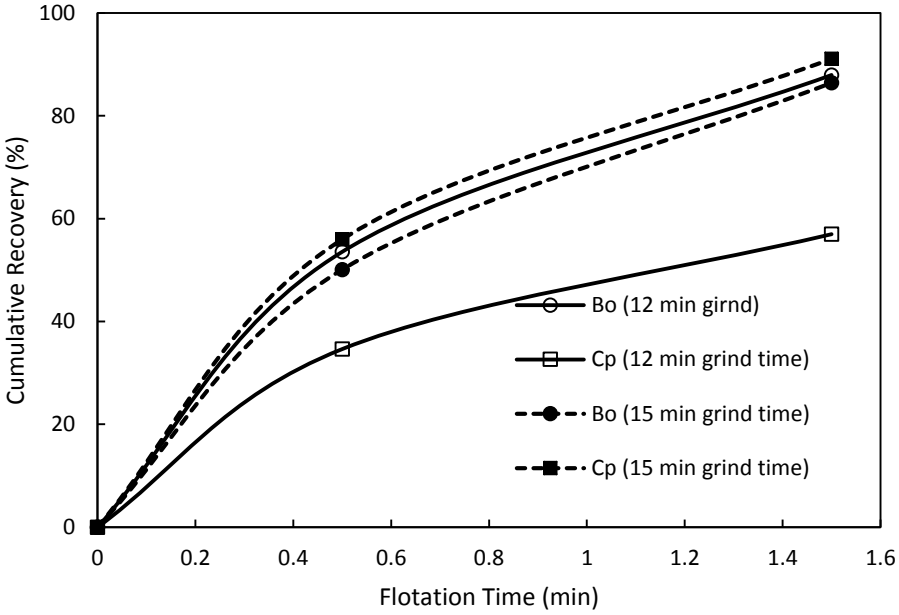

Figure 47: Effect of fine grinding on "scavenging" of collectorless flotation tails

From Figure 47, recovery of Chalcopyrite is improved when grinding time is increased while the recovery of bornite is almost identical as when grinding was performed at the standard 12 minutes. The recovery of chalcopyrite after fine grinding almost matches that of bornite. Fine grinding even see the recovery of chalcopyrite exceed that of bornite, which was not the case previously.

4.6 Effect of Nitrogen Flotation

Nitrogen was used to replace air during flotation. It was also sparged into the slurry mixture in the mill before grinding to replace air and dissolved oxygen from water. Nitrogen gas was also used during flotation. The results comparing nitrogen flotation with normal air flotation are presented in Figure 48. There was no lime addition in these test and the fine fraction from the feed was removed prior to these tests.

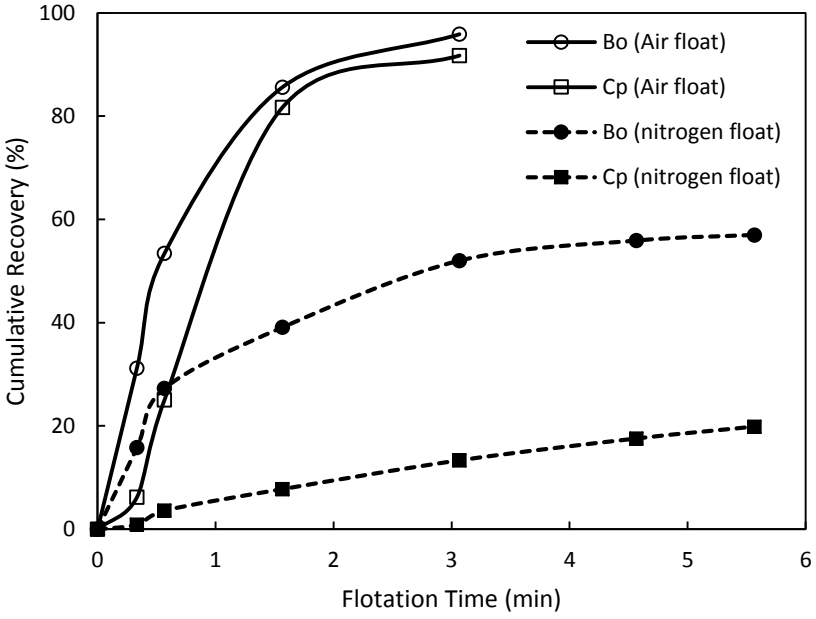

Figure 48: Effect of flotation with nitrogen gas versus air. (Eh = mV with nitrogen and Eh = mV with air)

It is quite clear from Figure 48 that flotation kinetics under normal air are much faster than when nitrogen was used. None of the mineral recoveries exceeds 60 % even after extending flotation time to over 5 minutes. In both cases bornite flotation is favoured over chalcopyrite flotation, however, chalcopyrite recovery is dramatically poor in nitrogen flotation compared to flotation with normal air leading to better separation efficiency in favour of bornite.

Leaching

Results from leaching test are presented in this section. Not all result from the experimental design are presented. Leaching data for the experiments can be found in Appendix C. Most tests showed less variation due to acid concentration and particle size, for this reason some of those result were not present in this section to avoid unnecessary repetitions.

4.7 Leaching of Bornite

Results of leaching of bornite at the two temperatures studied are presented below. There was less variation when acid concentration was varied, therefore only results for 0.1 M acid concentration are presented. Copper extraction as a function of time for bornite leaching at 25°C are shown by Figure 49 below:

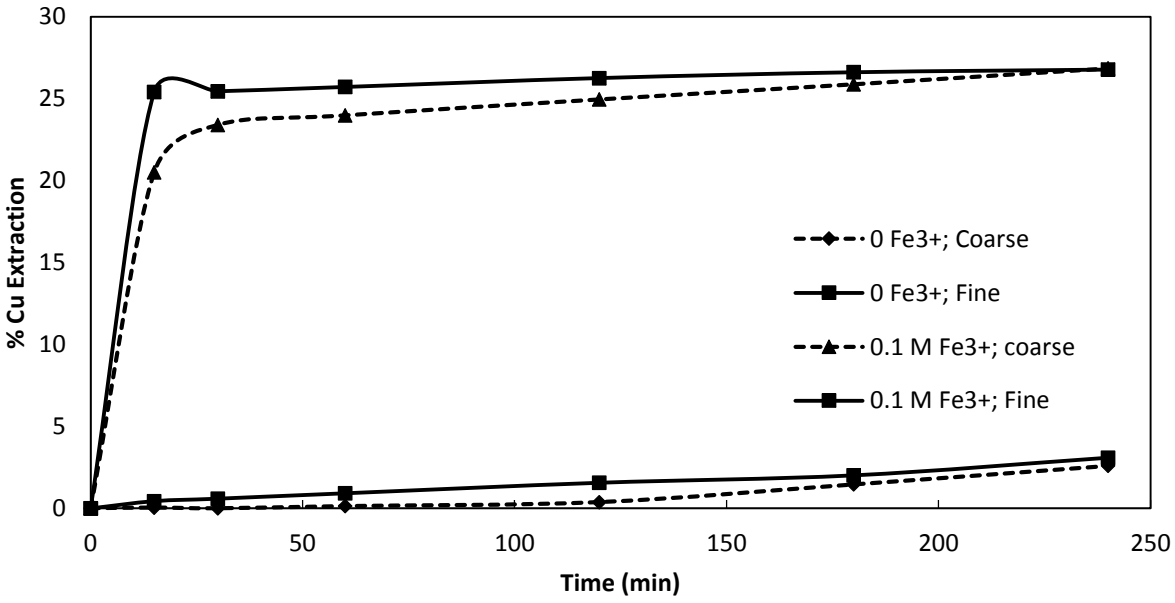

Figure 49: Leaching results of bornite at 25°C in the presence and absence of ferric ions in 0.1 M and 0.5 M acid concentrations.

In the absence of ferric ions in the original solution, the leaching rate is very minimal with less variation between the size fractions. In the presence of ferric ions, the initial leaching rate is

very rapid but levels off at around 26 % copper extraction. There is still less variation between the size fractions.

Results for bornite leaching at 90°C are shown in Figure 50. Due to less variation in the leaching behaviour between the acid concentrations and size fractions, only results for leaching in 0.1 M acid using the -400 mesh fraction are presented.

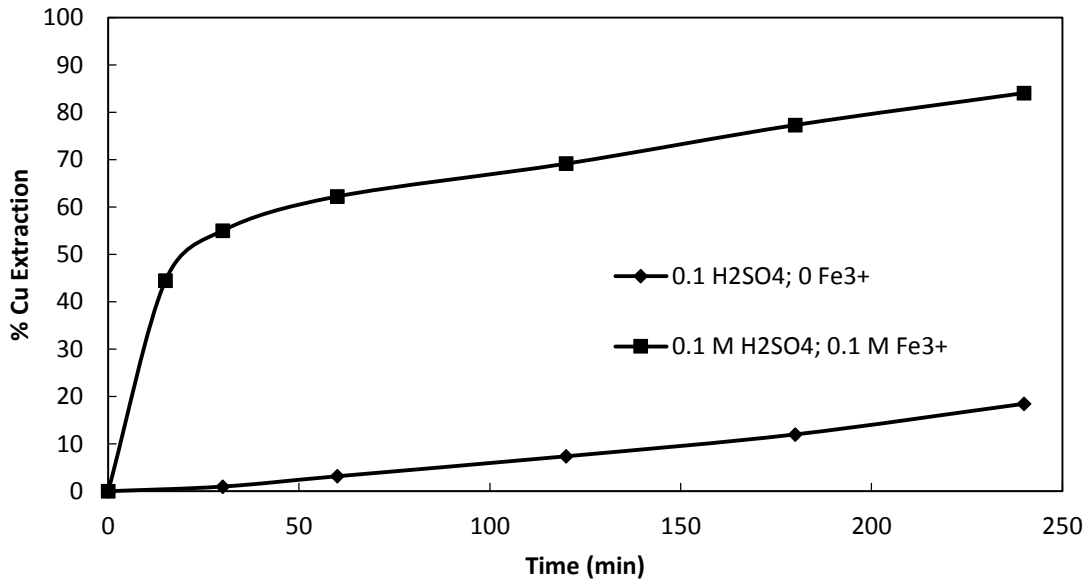

Figure 50: Leaching of -400 mesh bornite at 90°C in the presence and absence of ferric ions at 0.1M sulphuric acid.

From Figure 50, leaching in the absence of ferric ion is still low but more than double that at 25°C. The rate of leaching seems to be improving with time. Leaching in the presence of ferric ions has a very rapid initial rate but starts to slow down around 40 % copper extraction. The rate is significantly reduced after 50 % copper extraction and continues at this slowed rate.

4.8 Leaching of Chalcopyrite

Leaching of chalcopyrite was performed under similar conditions to those in which bornite was leached. Leaching results for chalcopyrite were much slower compared to bornite leaching performance. Less variation was observed between acid concentration and size fractions, so not all

results are presented in this section. Room temperature leaching results are presented in Figure 51. These are only -400 mesh chalcopyrite leaching results.

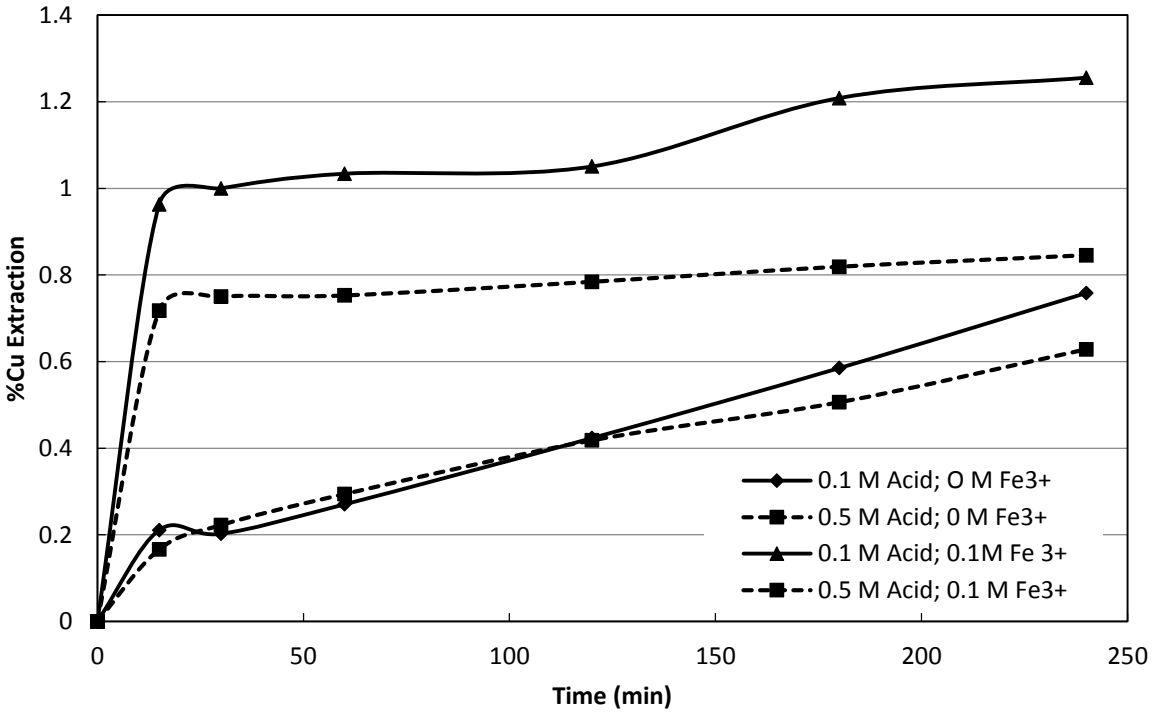

Figure 51: Leaching of -400 mesh chalcopyrite at 25°C in the presence and absence of ferric ions; at 0.1 M and 0.5M sulphuric acid.

At 25°C leaching barely goes past 1 % copper extraction for all conditions studied. The presence of ferric ions show a relatively improved initial rate but levels off to a much lower rate after this stage. In the absence of ferric ions, the rate is slow but climbs at a steady pace.

Results for experiments carried at 90°C to observe the temperature effects on leaching of chalcopyrite are shown in Figure 52. Results for the size fractions studied were similar so only -400 mesh results are presented.

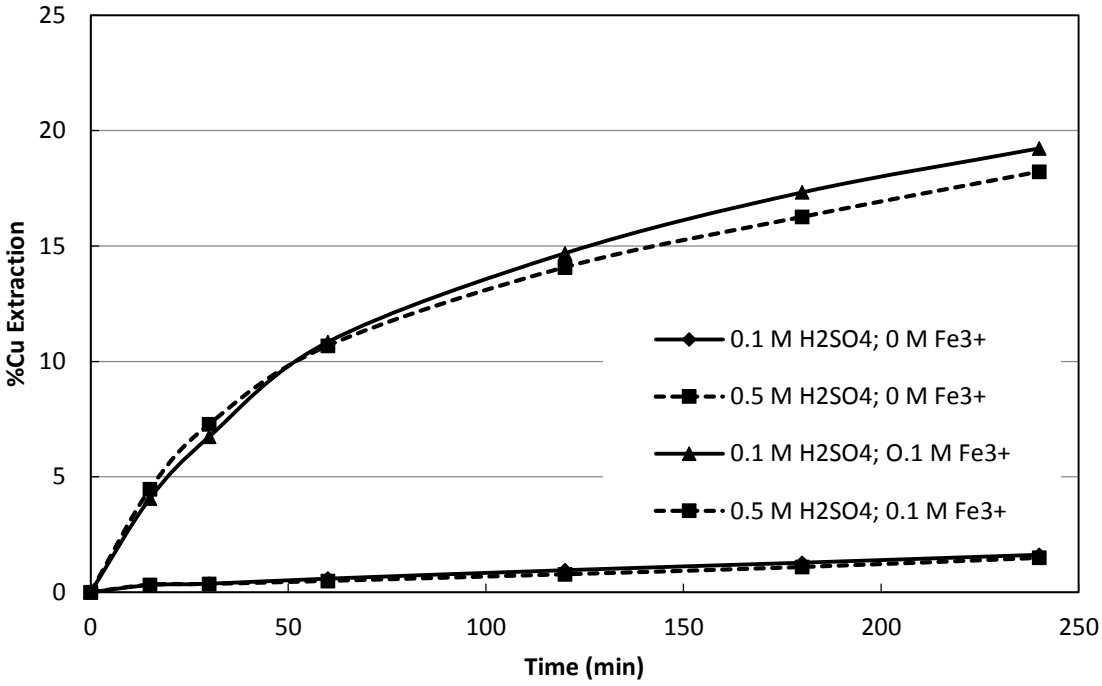

Figure 52: Leaching of -400 mesh chalcopyrite at 90°C in the presence and absence of ferric ions; at 0.1 M and 0.5 M acid concentrations.

The absence of ferric ions still lead to low extractions, even though the copper extraction is now a little improved compared to low temperature results. In the presence of ferric ions the leaching rate is much improved resulting in close to 20 % copper extraction after 4 hours. The rate of leaching in the first hour seems to be a bit higher than the leaching rate that follows. In both cases the leaching result under the two acid concentrations are almost the same. For comparison, chalcopyrite leaching under all conditions studied was significantly lower that the leaching performance of bornite under similar conditions

4.9 Leaching of combined minerals

The effects of galvanic interactions between the two minerals were investigated by leaching of mixtures of these minerals under the same conditions in which these individual minerals were previously leached in. To study the surface area effect that can result from having more of

one mineral type, different mineral ratios were used for the tests. Mineral sample mass ratios were used and chosen ratios were 1:1; 1:3; and 3:1 all being mass of bornite sample to mass of chalcopyrite sample ratios. These ratios were rough estimates due to the presence of impurities in the mineral samples, especially bornite. Results from these leaching tests are presented in the sections that follow.

4.9.1 1:1 Bo-Cp Mixture

This subsection presents the results from leaching of a 1:1 mineral mixture. Only -400 mesh samples results are presented. The result for room temperature leaching are presented in Figure 53, showing the effects of acid strength and presence of ferric ions.

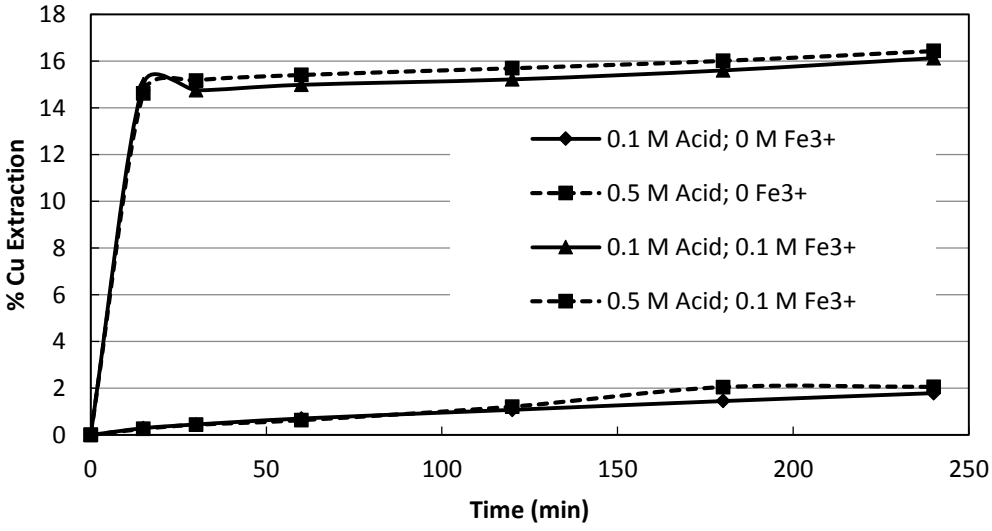

Figure 53: Leaching of 1:1 bornite to chalcopyrite (wt. %) at 25°C. -400 mesh samples used at 0.1 M and a0.5 M acid concentrations; in the presence and absence of ferric ions.

From Figure 53, the leaching behaviour of bornite is predominant in the leaching curves for this mixture. The leaching curves follow closely to the expected summation of the leaching curves for individual minerals: 13.5% in the presence of ferric ions and 2% in their absence. The leaching rate is rapid and levels off at around 16 % for the case with ferric ions while the case with

no ferric ions, which is slightly higher than expected, has a slow steady increase in copper extraction.

Copper extraction results for leaching at 90°C in the presence of ferric ions are shown in Figure 54. For bornite, two size fractions were investigated, -100 +400 mesh and -400 mesh, while only -400 mesh fraction of chalcopyrite was used. The results were compared to the individual leaching of bornite together with the expected behaviour from the summation of individual leaching of both minerals.

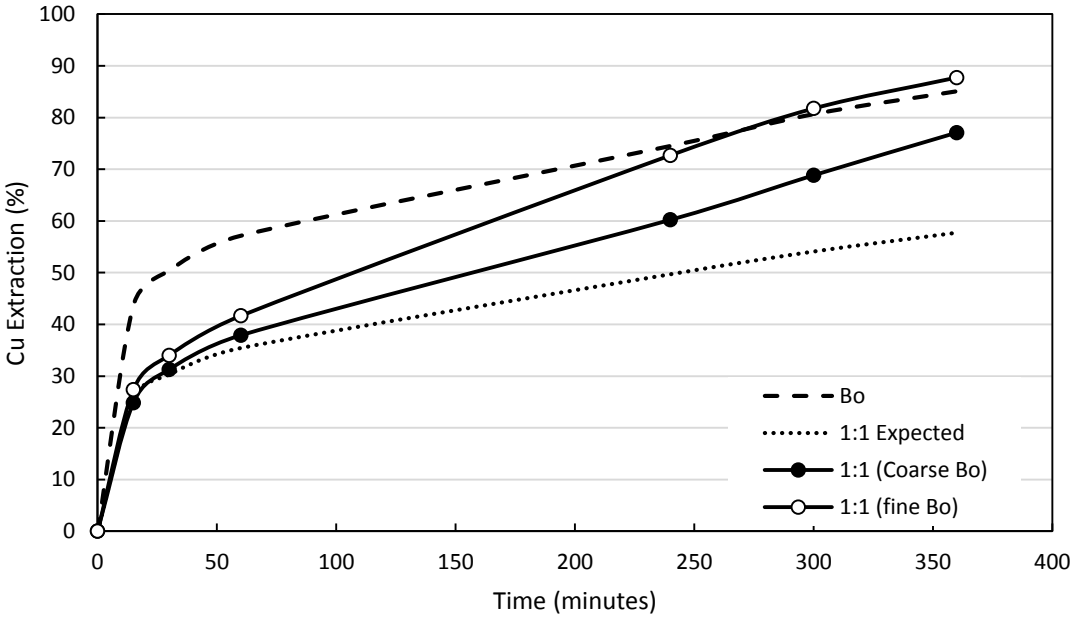

Figure 54: Leaching of 1:1 Bo-Cp (wt. %) mixture at 90°C in the presence of ferric ions in 0.1 M acid. -400 mesh Cp used.

The initial leaching rate follows closely to the expected behaviour but starts to depart from this behaviour after a point which normally sees a considerable reduction in the leaching of bornite. The rate of leaching experienced by this mixture in the second stage looks significantly higher than expected. There is variation in the leaching results of the mixtures, with fine bornite showing better results, but both curves shows increased rate of leaching.

4.9.2 1:3 Bo-Cp Mixture

This section explores the leaching behaviour of the 1:3 mineral mixture. Only -400 mesh chalcopyrite was used for these mixtures as well. Room temperature results varying the acid strength and ferric ion presence are presented in Figure 55 below:

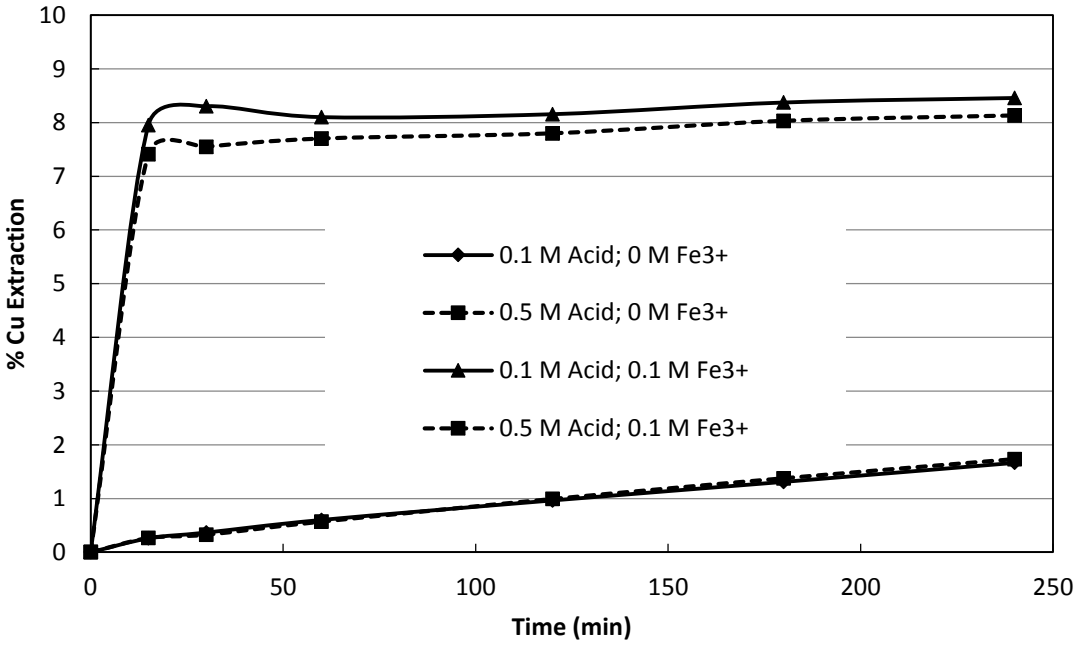

Figure 55: Leaching of 1:3 Bo-Cp (wt. %) mixture at 25°C in the presence and absence of ferric ions in 0.1 M and 0.5 M acid concentrations. -400 mesh samples used.

Bornite leaching behaviour still predominates low temperature leaching result with the copper extraction levelling off at around 8 % for the case with ferric ions while no ferric ions case sees slow consistent rate rising with time. The extraction curves fall closely to their expected behaviour which was expected to be 7% in the presence of ferric ions and 2% in their absence.

90°C leaching results in the presence of ferric ions is presented in Figure 56. Expected leaching curve and bornite leaching curve are shown form reference. Mixture included coarse (-100 +400 mesh) and fine (-400 mesh) bornite samples mixed with -400 mesh chalcopyrite sample.

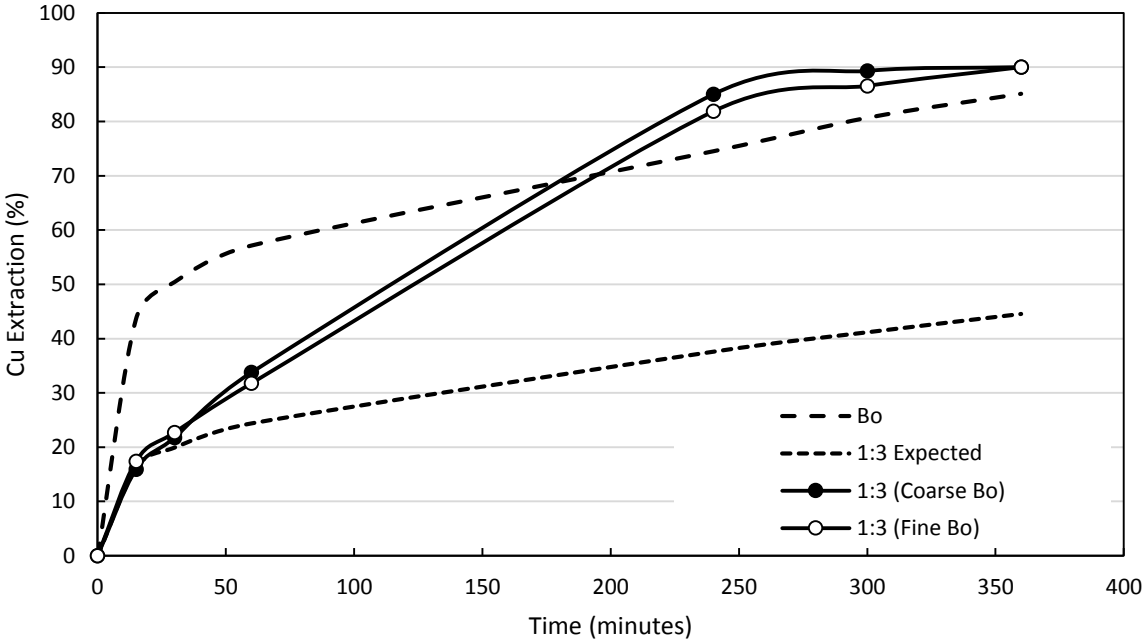

Figure 56: Leaching of 1:3 bo-cp (wt. %) mixture at 90°C in the presence of ferric ions in 0.1 M acid. -400 mesh chalcopyrite used.

The initial copper extraction follows that of the expected results but there is a significant deviation from the expected results for both mixtures similar to the observation in the 1:1 case. The leaching results go on to exceed the results for the individual leaching of bornite but levels off at around 90 % copper extraction.

4.9.3 3:1 Bo-Cp Mixture

Results for leaching of the 3:1 mineral mixture are presented in this section. No room temperature leaching was performed for this mixture but only leaching at 90°C was performed. Only -400 mesh chalcopyrite was used together with coarse and fine bornite fractions. Leaching results at this temperature are shown in Figure 57. The expected and bornite leaching curves are shown for reference.

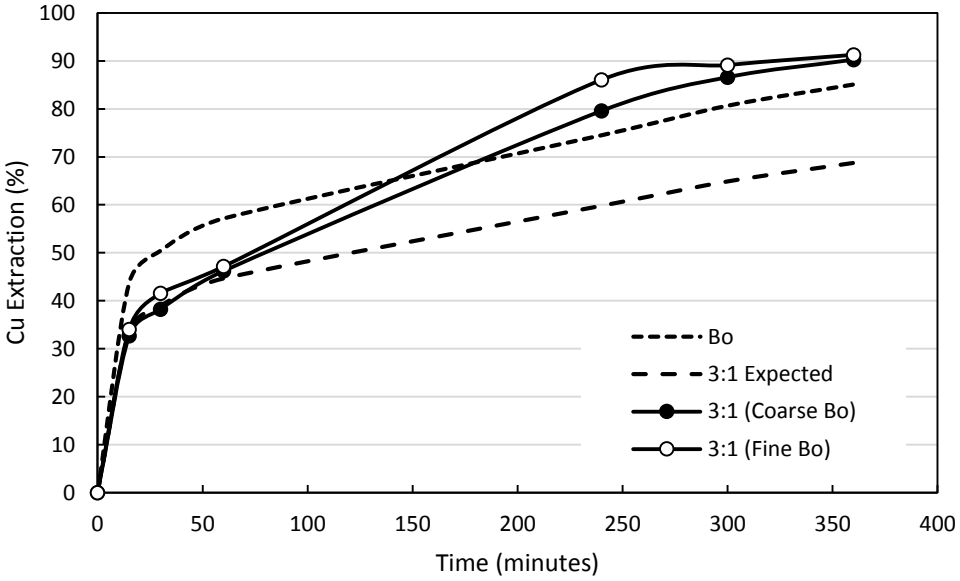

Figure 57: Leaching of 3:1 bo-cp (wt. %) mixtures at 90°C in the presence of ferric ions at 0.1 M acid concentration. -400 mesh chalcopyrite used.

Similar behaviour to that already observed in the 1:1 and 1:3 cases is observed here. The initial leaching follows the expected pattern but deviates as soon as the second stage of bornite leaching ensues. There is plateauing at around 90 % copper extraction in this case too.

4.10 Residue Analysis

Leaching residue from selected tests was sent for SEM and XRD analysis to identify the nature of species formed during leaching and provide an insight into the possible reaction mechanisms involved during the leaching process. The results from the SEM and XRD scans are presented in the following subsections;

Bornite

An SEM scan of the leaching residue of bornite is shown in Figure 58 below. This residue was from leaching in the presence of ferric ions at 90°C.

Figure 58: SEM scan of bornite leaching residue showing the intermediate idaite with sulphur matrix in the background.

No bornite was detected by SEM scans, but rather intermediate phases with similar copper and iron content to that for idaite (Cu_3FeS_4). XRD spot analysis detected this mineral phase to have roughly the following elemental composition: 51%Cu, 15%Fe, and 34%S. Elemental sulphur was predominant in this leaching residue.

Chalcopyrite

SEM scans from the high temperature leaching of chalcopyrite in the presence of ferric ions is shown in Figure 59 below. This residue was taken after 30% copper dissolution from the chalcopyrite sample during a steady leaching rate.

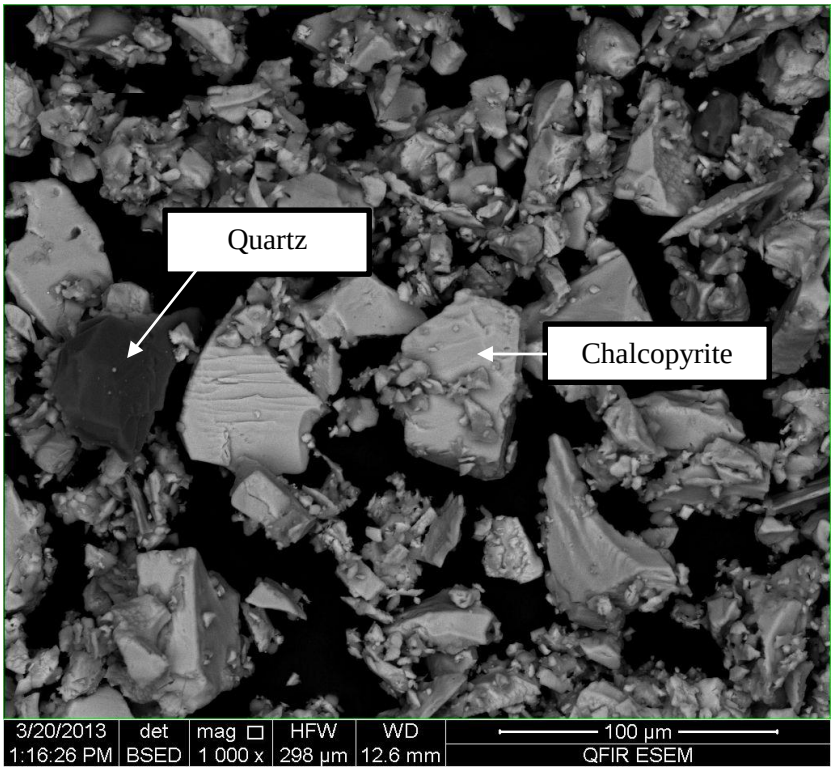

Figure 59: SEM scan of chalcopyrite leaching residue showing chalcopyrite mineral particles after 30% Cu extraction at 90°C.

XRD spot analysis during SEM scans did not detect any sulphur layer on the chalcopyrite particles however XRD analysis of the residue detected elemental sulphur which was further supported by increased sulphur assay from combustion analysis.

Mixtures

Residues from leaching residues of mineral mixtures was analysed by both XRD and SEM. For all mixtures, bornite was not detected but rather intermediate phases with varying copper-iron ratios were detected by XRD spot analysis during SEM scans. A low magnification SEM scan of the 3:1 mixture is shown in Figure 60 below:

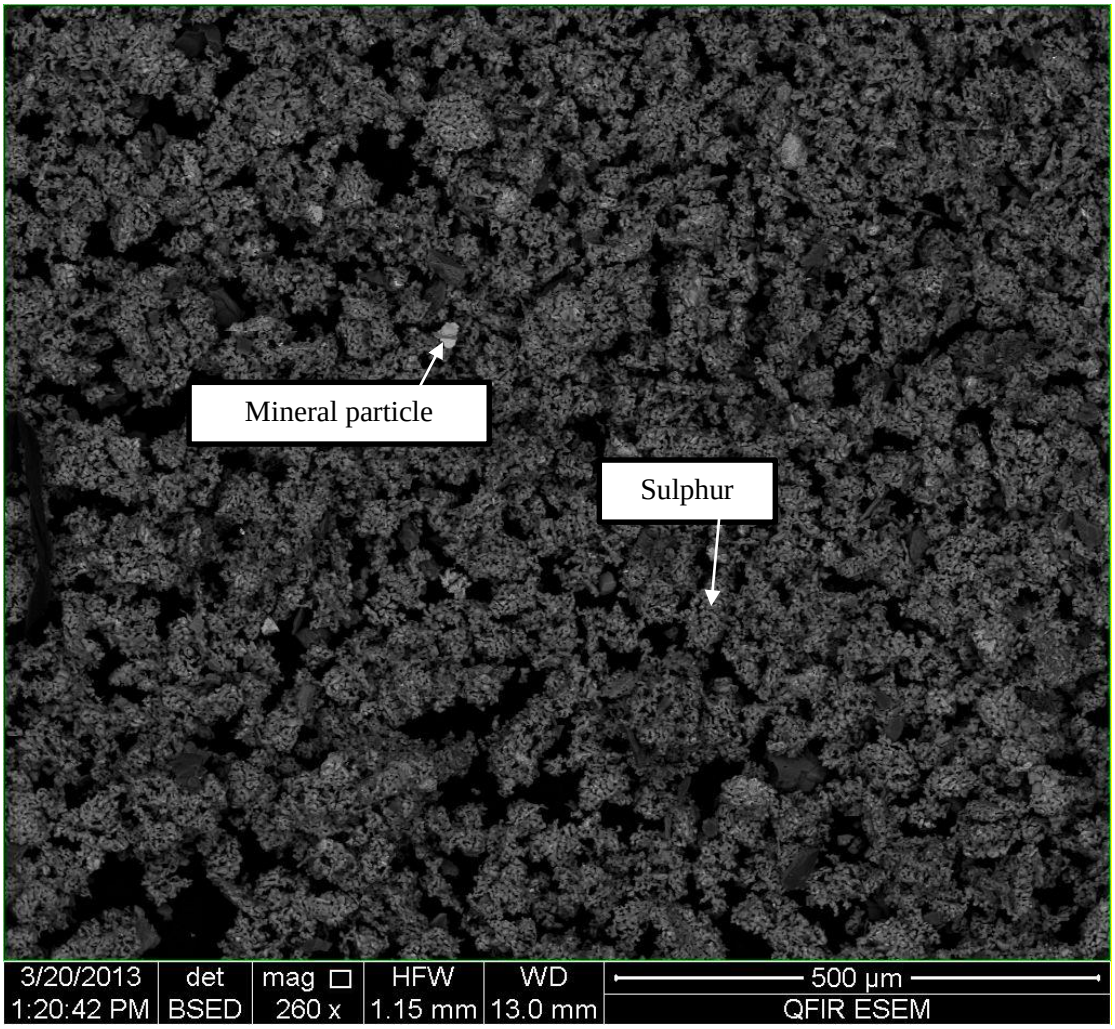

Figure 60: Low magnification SEM scan of a 3:1 bo-cp mixture from leaching in ferric solution at 90^0C. This scan was dominated by sulphur from leached out mineral particles with a few remaining mineral particles scattered around.

No bornite was detected in these samples but there were copper-iron sulphide phases with varying copper/iron ratios. Chalcopyrite was present in all residues. Sulphur growth was observed on all sulphide species. Figure 61 shows sulphur growth on chalcopyrite.

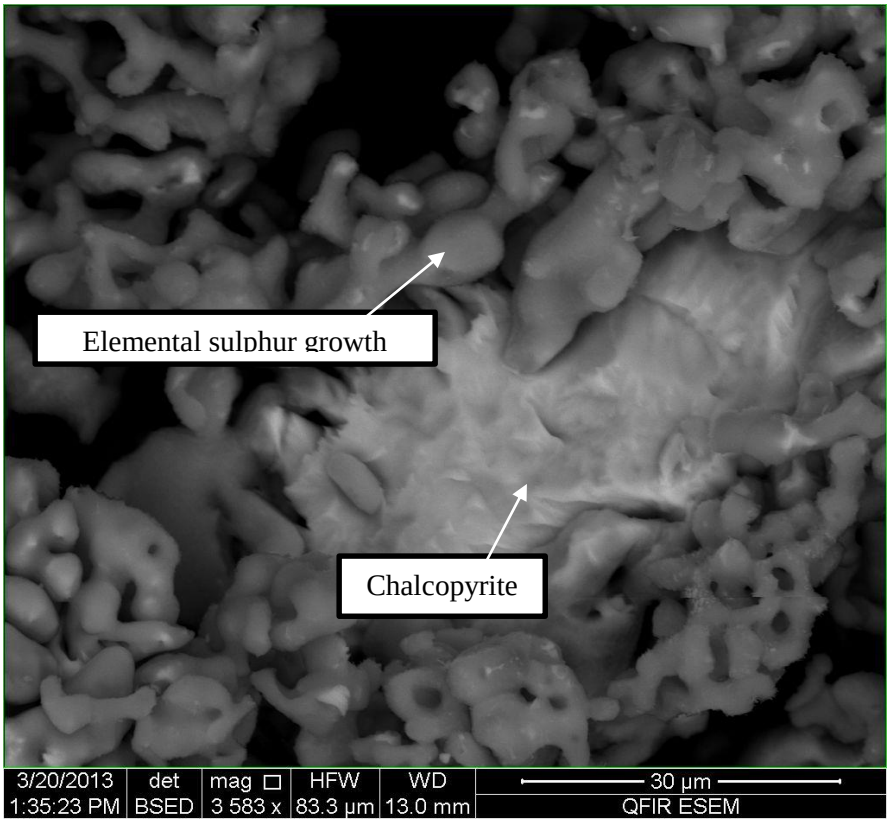

Figure 61: High magnification SEM scan showing sulphur growth on chalcopyrite from a 1:3 bo-cp mixture leached at 90°C in the presence of ferric ions.

Unlike the case, where chalcopyrite was individually leached, sulphur growth was observed on leaching residues of mineral mixtures. The sulphur formation has a porous looking structure for chalcopyrite, while idaite particles had platy-looking sulphur formation as shown in Figure 62 below:

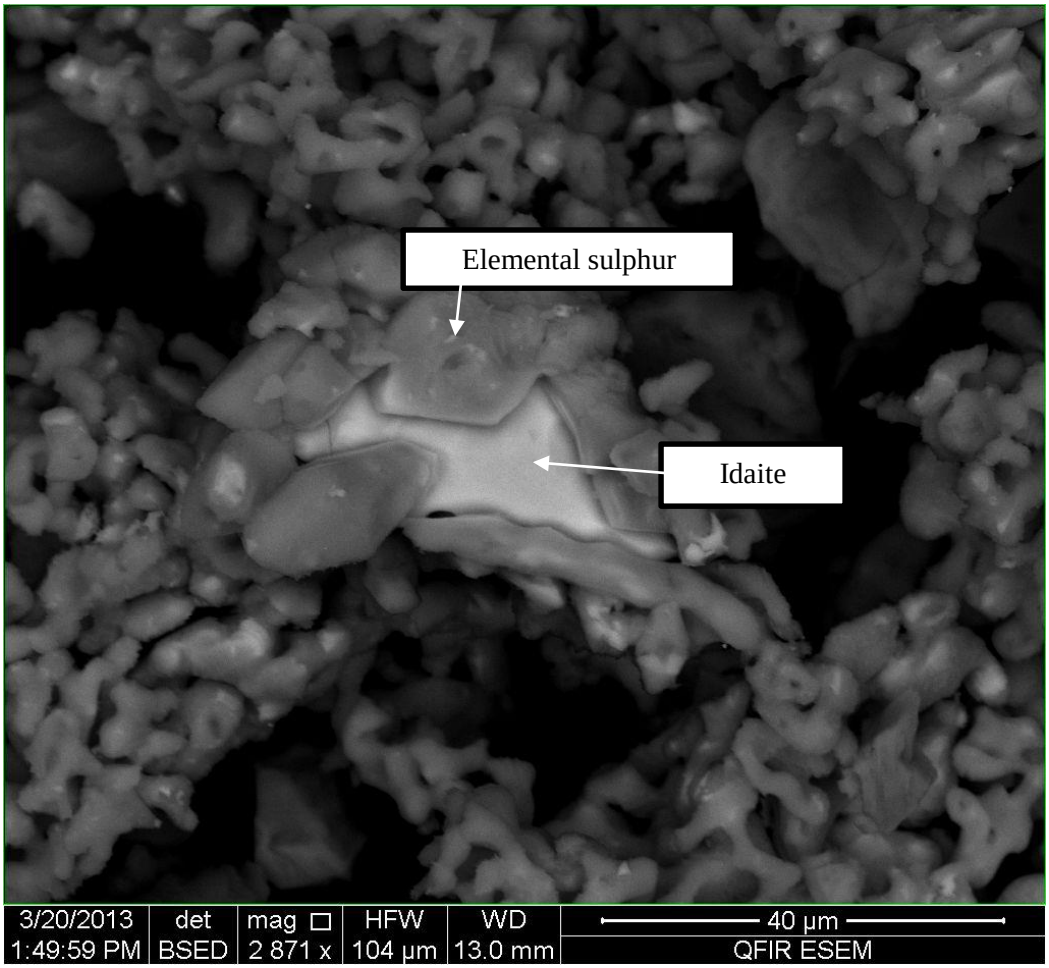

Figure 62: SEM scan showing crystalline-looking sulphur growth from an intermediate bornite leaching specie (Cu_3FeS_4).

The sulphur formation shown in Figure 62 does not look porous and tends to engulf the mineral particle. In addition to the sulphide species, silver particles were also detected by SEM scans but were not detected by XRD scans. Figure 63 is an SEM scan showing a silver particle in the 3:1 leach residue.

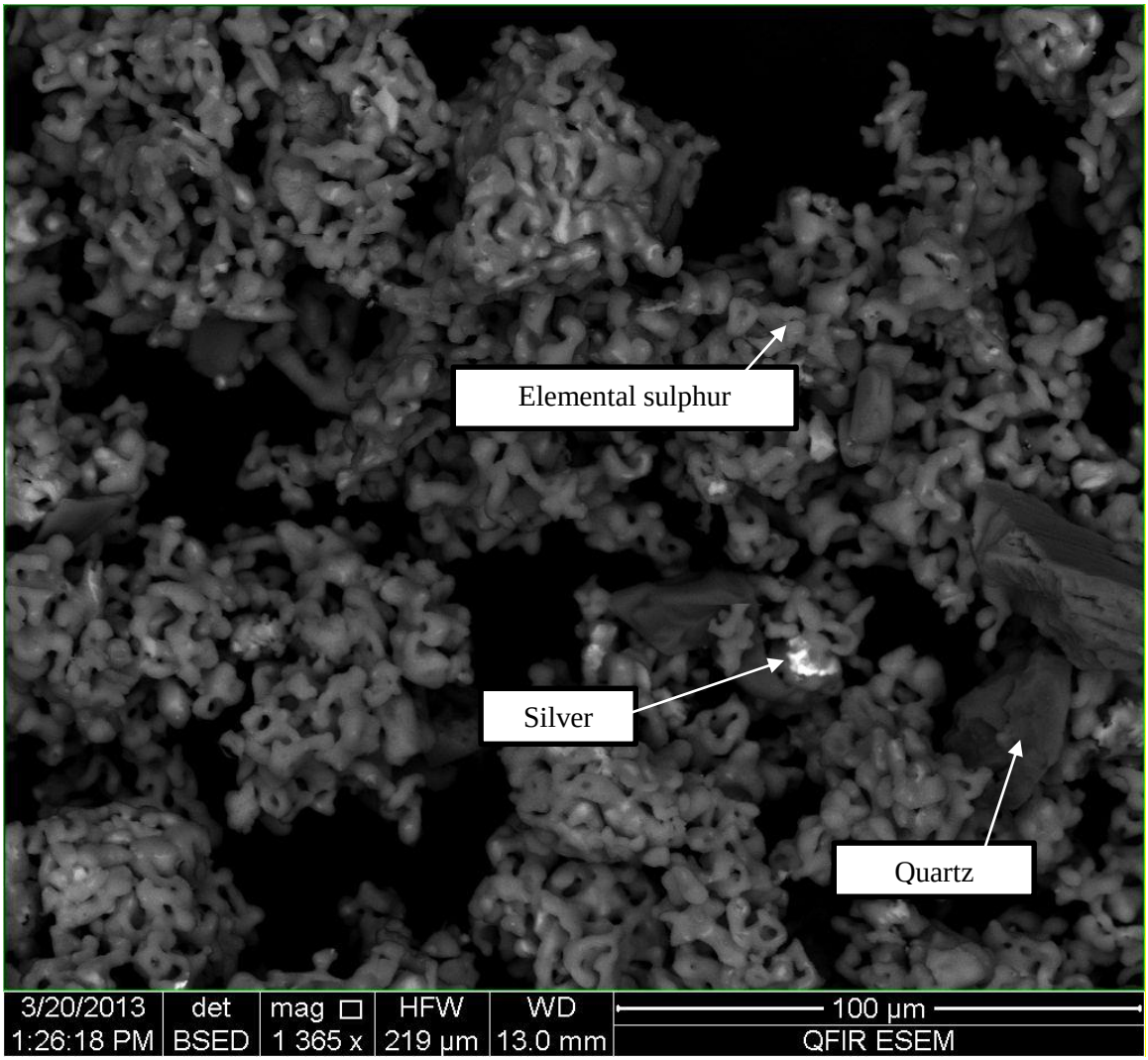

Figure 63: An SEM scan of the 3:1 mixture residue showing a silver particle in the residue.

Silver is known to have catalysing effect on the leaching of chalcopyrite. The amount of silver present in solution or a solids was not determined but was presumed to be negligible.

4.11 ORP control

The individual and mixed leaching tests carried out so far were done without any redox potential control. Due to the redox nature of the reactions involved, the solution ORP is expected to be affected by the introduction of oxidation products into solution and reduction of ferric ions. The redox potential was seen to fall with the progression of the experiments. The falling of the ORP can have an effect on the leaching rate due to the electrochemical nature of the reactions. The

initial redox potential for the leaching solutions used was usually around 750 mV vs. AgCl at 90°C but fell within the 400-600 mV range during the progression of leaching in most cases tested.

4.11.1 Bornite

Hydrogen peroxide solution was used to hold the redox potential at around 750 mV for the duration of the experiments. There was considerable variation in this redox value as it kept falling and had to be raised to the set value. The ORP often fell down to 720 mV for brief periods before being raised back to 750 mV by hydrogen peroxide additions. Attempts were made to keep the variations minimal. Bornite leaching results at 750 mV solution potential, at 90°C are shown in Figure 64.

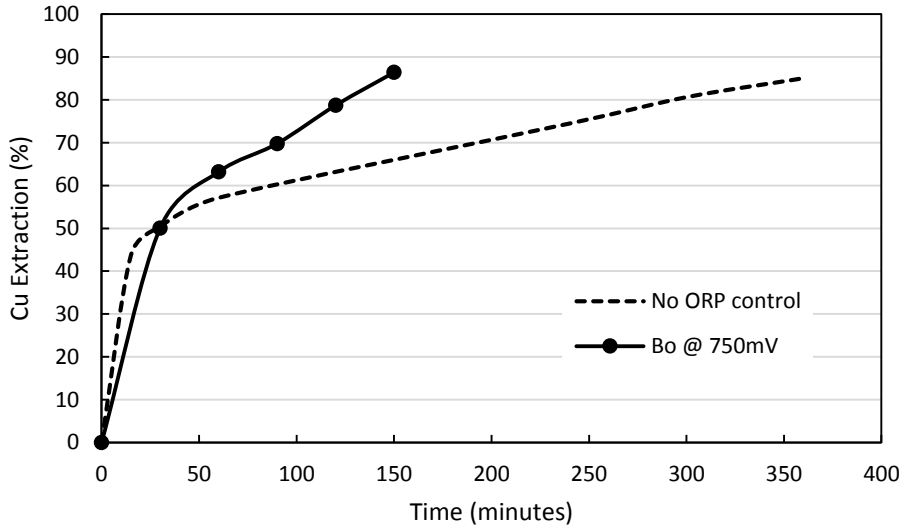

Figure 64: -400 mesh bornite leaching at 90°C with the redox potential held at 750 mV vs. AgCl

Even though no reading was taken at 15 minutes, it is assumed that the initial rate for both tests are similar up to 30 minutes of leaching, at which they both have about 50 % copper extraction. After this point the leaching rate at 750 mV progresses at a steeper rate than that with no ORP control.

4.11.2 Chalcopyrite

Chalcopyrite was also leached at 750 mV in the same manner used for bornite. The results for the comparison of the tests with or without ORP control in the leaching of chalcopyrite are shown in Figure 65.

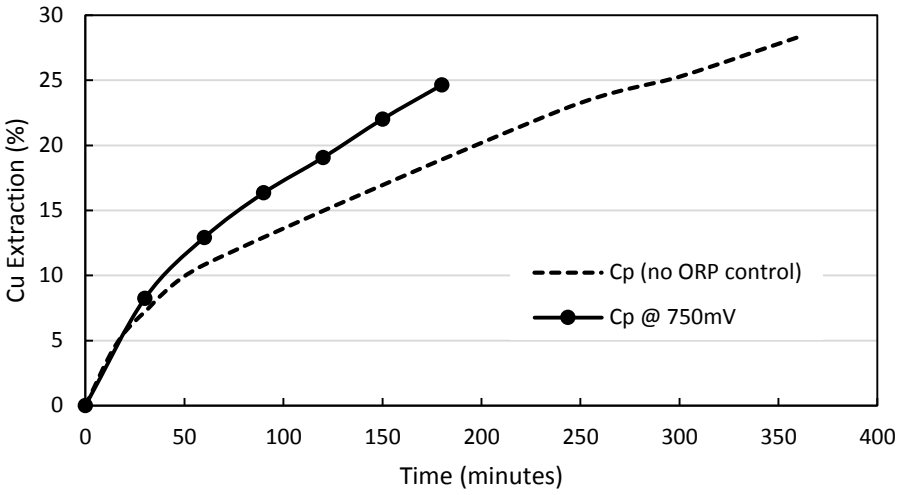

Figure 65: Chalcopyrite leaching at 90°C with the redox potential held at 750 mV vs. AgCl

From Figure 65, the initial leaching rate for both cases is roughly the same followed by deviation as time progressed. Copper extraction in the case with no ORP control falls off from the initial rate quicker than when ORP was fixed at 750 mV.

4.11.3 Mineral Mixtures

ORP control was investigated for the leaching of the mineral mixtures too. 3:1 and 1:3 were the chosen ratios for these tests. The results for this investigation are shown in Figure 66. The leaching results for these mixtures were compared to their expected leaching, calculated from the individual copper extraction curves of both mineral at 750 mV. Bornite and chalcopyrite leaching curves at 750 mV are provided for reference.

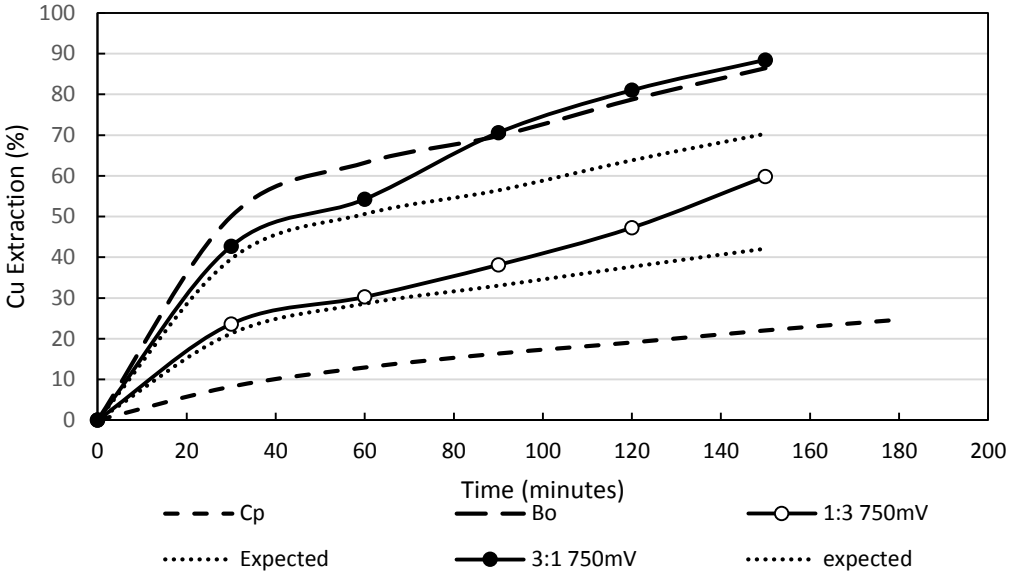

Figure 66: Comparison of leaching at 750 mV, 90°C, for 1:3 and 3:1 mixtures.

Leaching in the first hour follows closely to the expected behaviour but starts to deviate thereafter. The deviations are not as dramatic as those observed in the tests without ORP control but given the short span of current experiments, the total copper extraction after two and a half hours is relatively high.

4.11.4 Chalcopyrite leaching in bornite leachate solutions

Experiments carried out by leaching chalcopyrite in a pregnant solution in which bornite samples were previously leached in for an hour at 90°C in the presence of ferric ions showed improved copper extractions from chalcopyrite by those solutions. The results of those tests are shown in Figure 67 below:

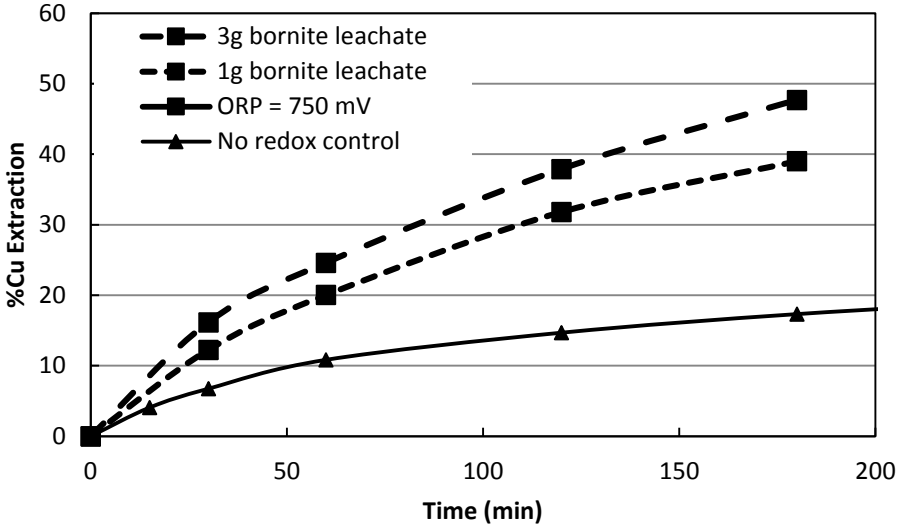

Figure 67: Copper extractions from chalcopyrite leached in different solutions at 90°C.

A solution in which 3 g of bornite was previously leached in produced relatively higher extraction compared to that in which 1 g of bornite was leached in, both of which were considerably higher than cases with no prior bornite leaching.

Chapter 5

Discussion

Flotation

The results from flotation tests performed on a system that contained bornite and chalcopyrite are discussed in this section. The effects of investigated parameters are explored, including how they could have influenced the results of these tests.

5.1 Effect of Fines

Due to prolonged storage, despite it being in cold conditions, it is quite possible that the fine fraction in the feed experienced considerable oxidation. The fines would be the fraction that had been most affected primarily due to its relatively large surface area. Surface oxidation on this fraction would most likely result in altered behaviour from that of freshly ground mineral surfaces.

For collectorless flotation, the recoveries between the tests were similar and had considerable variation in bornite recovery, making it hard to reach any strong conclusion on the presence of any trend, Figure 38. From the grade plots for Test 9 and 14 shown in Figure AA - 25 and Figure AA - 27, grade comparison for the first three concentrates shows that the removal of fines results in slight increase of chalcopyrite grade and a slight reduction in bornite grade for all collectorless flotation concentrates. From these observations it can be concluded that oxidised fines promote collectorless flotation of bornite even though the process favours the recovery of chalcopyrite. This behaviour leads to poor selectivity of the process.

Tests in which the feed was not screened to remove the -400 mesh fraction had slightly higher recoveries for both minerals than in a test for which a collector was present, Figure 37. Chalcopyrite recovery seemed slightly more affected. Cumulative grade comparison between Test 2 and 5, Figure AA - 3 and Figure AA - 11 in the appendix, showed about 5% reduction in

chalcopyrite grade together with 5% increase in bornite after the first 30 seconds of flotation. This led to about 10% improvement in selectivity in favour of bornite. It can be therefore be suggested that the presence of oxidised fines adversely affect the selectivity of flotation by promoting chalcopyrite recovery.

It can be concluded that oxidised fines promote collectorless flotation, possibly due to oxidation of mineral surfaces and generation of activating metal ions (i.e., Cu^{2+}). This collectorless flotation would then impact selectivity by improving the flotation of a mineral that would otherwise not experience increased flotation.

5.2 Effect of Collector vs. Collectorless Flotation

As seen from Figure 39 and Figure 40, flotation without the use of a collector heavily favours the recovery of chalcopyrite over that of bornite both in the presence and absence of fines in the feed. On the contrary, the use of a collector greatly improves the recovery of bornite while having minimal effect on the recovery of chalcopyrite. The improvement in the recovery of bornite in the presence of a collector is very pronounced such that it leads to the reversal of the selectivity of the process compared to that of collectorless flotation. The presence of a collector heavily favours the recovery of bornite over that of chalcopyrite.

Chalcopyrite is known to exhibit strong collectorless behaviour of a wide pH range while bornite is known to float poorly in the absence of a collector, therefore the collectorless flotation results fit expectations. The lack of individual flotation data for each mineral under similar conditions makes it difficult to assess the alteration in behaviour of each mineral brought about by having both minerals in the same system. Due to bornite being anodic and chalcopyrite being cathodic in this system, it would be expected that anodic oxidation of bornite would be enhanced, which is expected lead to formation of a metal-deficient sulphur-rich layer thereby improving its floatability. In the case of chalcopyrite being the cathodic mineral, it is expected to experience

reducing conditions, which are known to adversely affect its flotation, therefore its recovery is expected to be slightly reduced.

In the presence of a collector, the more anodic mineral is expected to be more likely to react with the anionic collector. Galvanic contact between the minerals is expected to increase anodic reactions on the anodic mineral generating metal ions on the mineral surface that would react with the anionic collector molecules and bind the collector to the mineral surface. Reducing conditions experienced by the cathodic mineral would discourage reaction with anionic collector molecules hence leading to decreased adsorption of the collector to the cathodic mineral. For these reasons, bornite is expected to preferentially bond to the collector over chalcopyrite.

This behaviour is supported by the flotation results with bornite experiencing marked improvement in recovery while chalcopyrite recovery is marginally improved in the latter stages of flotation after much of the bornite had been removed from the pulp. The relatively high recoveries of chalcopyrite could be explained by the strong collectorless flotation of this mineral. Chalcopyrite flotation in the presence of a collector still mirrors its collectorless flotation behaviour and only starts to improve towards the end of flotation. It can be suggested that chalcopyrite recovery in the early stage of flotation is mainly by a collectorless mechanism and only starts to be affected by the collector once most of the bornite has been removed from the system. This leads to relatively high dilution and poor selectivity in the process.

5.3 Effect of Collector Dosage

Competition for collector can assist in exposing the effects of galvanic interactions between sulphide minerals. The anodic mineral will be favoured to bind with the collector electrochemically, depending on the collector species present at the given pulp conditions. It is clear from previous results that collectorless flotation is heavily in favour of chalcopyrite flotation. As shown in Figure 41, when compared to the collectorless case, addition of small amounts of a collector (1ml/conc. PEX) greatly improve the recovery of bornite while having little effect on the recovery of

chalcopyrite. Further increase in collector dosage to 2 ml/conc. results in further increases the recovery of bornite while there is no similar improvement in chalcopyrite flotation. This improvement is large enough to result in reversal of flotation behaviour in favour of bornite. Chalcopyrite recoveries stayed pretty much the same with varying dosages of a collector which would suggest that the chalcopyrite flotation at this stage does not rely on the collector flotation but rather it is through induced hydrophobicity from the pulp conditions. From these observations it can be suggested that galvanic interactions promote the reaction of bornite with a collector over that for chalcopyrite leading to a marked improvement in the recovery bornite while chalcopyrite recovery is pretty much unchanged until considerable amounts of bornite have been removed from the system. Chalcopyrite reaction with the collector comes secondly after the ratio of bornite in the system has been reduced. Grade data for Test 2, 3 and 14, shown in Figure AA - 3, Figure AA - 5 and Figure AA - 27 in the appendix, could be used to further support this analysis

5.4 Effect of pH

Chalcopyrite is known to float over a wide range of pH, while bornite is known to float better in alkaline conditions with the use of a xanthate collector [14]. Most of the pH values used in the tests to investigate the effect of pH, between pH 8 and 11, where these variations could not be observed between the current tests. All of these pH values were in the alkaline range therefore the investigation of the effect of pH was limited to alkaline range.

In the case of collectorless flotation tests carried at pH 8, 9 and 11 showed a reduction in chalcopyrite recovery at pH 11, Figure 43, while no noticeable effect was observed on bornite. High pH is normally linked with decreased pulp potential which is known to adversely affect the collectorless floatability of chalcopyrite [15]. Also, increased hydroxyl concentration together with continued generation of these ions at cathodic sites could lead to deposition of hydrophilic metal oxides/hydroxides on the mineral surface decreasing its floatability.

When flotation was carried out in the presence of a collector, lower pH showed better selectivity in the first two concentrates, which came mainly from apparent depression of chalcopyrite, Figure 44. Overall recoveries were higher for pH 7.9 compared to pH 9.

Since bornite recoveries for the first two concentrates was unchanged, it can be assumed that all of the collector added was used up in the flotation of bornite while chalcopyrite was under collectorless flotation. For this reason pH 9 therefore favoured this collectorless behaviour more than pH 7.9. Collectorless flotation of chalcopyrite has been shown to dip around neutral pH and improve with increasing or decreasing pH [22].

As for the overall recoveries, very high pH could lead to the competition between hydroxyl ion and the collector for adsorption on mineral surfaces by reaction with metal ions formed. Less competition between xanthate ions and hydroxyl ions at slightly alkaline pH could promote increased adsorption on both minerals, possibly explaining high overall recoveries and reduced selectivity at latter stages of pH 7.9 flotation. Oxidation of xanthate by peroxide ions to perxanthate at very high pH has been reported, which could lead to reduced flotation performance. [20].

Dixanthogen formation under alkaline pH conditions has been reported in literature, [14], which could also have an impact in the selectivity of the process at high pH values as its absorption to the mineral surface is not necessarily electrochemical.

5.5 Effect of Grinding Time

When investigating selectivity of a process using natural ore, mineral liberation is an important parameter to consider. This is because unliberated minerals can be carried into the froth or held in solution as a result of the nature of the minerals they are bound to.

Extending the grinding time can improve liberation but care has to be taken as to not produce an increased amount of fines, which could also adversely impact selectivity through entrainment of

fines resulting in fine material reporting to the froth regardless of whether the mineral would normally float or not.

From Figure 45and Figure 46, In the presence of a collector fine grinding results in small improvements in the recovery of bornite for the first two concentrates. Chalcopyrite experienced improved recovery in the first concentrate after fine grinding. The recovery in the second concentrate dropped while bornite recovery stayed high.

Possible explanation for this behaviour is that fine grinding improves liberation therefore leading to increased amounts of free bornite being available to be floated by the collector. Since chalcopyrite is under collectorless flotation, an improved liberation would free up more chalcopyrite to float leading to increased recovery in the first concentrate. However, once the free floating chalcopyrite is removed in the first concentrate, recoveries of chalcopyrite in the second concentrate would fall, while bornite recovery stays high leading to improved separation.

From the grades data of Test 5 and Test 11, Figure AA - 11 and Figure AA - 13, the grades of both mineral are improved by fine grinding in the first concentrate. However for the second concentrate, the grade of bornite is improved while that of chalcopyrite is reduced which supports the case that fine grinding improves liberation leading to most of the collectorless floating chalcopyrite reporting mainly to the first concentrate.

Fine grinding would also increase the surface area available for the collector to adsorb onto. This would mean finely ground ore would require higher dosage of a collector to achieve high recoveries. This reasoning could help explain the slightly lower overall recoveries observed in the fine grinding case.

In the collectorless flotation case, fine grinding does not affect chalcopyrite flotation in the first two concentrates. However, there is reduction of bornite recovery. This could mean that some of the bornite that was normally bound to chalcopyrite was liberated and stayed in solution while chalcopyrite floated collectorlessly. Grade data for Test 9 and 14, Figure AA - 25 and Figure AA -

27, shows increased chalcopyrite grades accompanied by reduced bornite grades for all collectorless flotation grades resulting in improved selectivity.

During the addition of a collector after collectorless flotation, the recovery of chalcopyrite showed increased recovery while bornite recovery was pretty much unchanged, Figure 47. This could mean that some of the chalcopyrite that would normally be bound to gangue minerals and held in the pulp was liberated and floated through collector action.

5.6 Effect of Nitrogen Flotation

Sulphide minerals are known to perform poorly under reducing conditions. The replacement of air with nitrogen in the system often leads to reducing pulp conditions.

It is very clear from Figure 48 that flotation kinetics are significantly affected by the replacement of air with nitrogen. Chalcopyrite flotation seems to be the most impacted by nitrogen flotation with very low recovery in the first two concentrates. Bornite is clearly floating preferentially to chalcopyrite despite general poor flotation performance. It is known in literature that nitrogen flotation retards the recovery of chalcopyrite [22]. This poor collectorless flotation of chalcopyrite when nitrogen is used is often linked to reducing conditions that result from removal of oxygen. From the flotation data sheet for Test 13 shown in Figure AA - 15, the pulp potential during flotation ranges from -180 to -303 mV vs. Ag, AgCl In the case of nitrogen flotation, flotation can be assumed to be predominantly collector induced since collectorless flotation is minimised under these conditions. Bornite has been shown to float at a relatively lower pulp potential than chalcopyrite, [27], which could explain its relatively higher recovery. The presence of chalcopyrite would also increase the anodic behaviour of bornite .This would lead to bornite being able to interact with the collector at an even lower pulp potential while interactions of chalcopyrite with the collector would be limited.

From the grade data of Test 13 and Test 5, Figure AA - 15 and Figure AA - 11, nitrogen flotation leads to increased selectivity between the grades of these minerals in favour of bornite.

The cumulative grade and recovery data suggests that the overall recovery versus grade for the air and nitrogen flotation almost the same. The main difference is the slowed down flotation kinetics, making it possible to observe incremental recoveries versus grade more clearly under the slow nitrogen flotation case.

Conditioning the pulp with air for five minutes before continuing flotation increases the pulp redox potential and greatly improves the flotation kinetics. Since most of the bornite was removed during nitrogen flotation, chalcopyrite recoveries dominate the air flotation stage. Recoveries of both minerals are improved by introduction of air.

5.7 Possible Sources of Error

Experimental Errors

A rotary splitter was used to generate representative ore charges from the bulk sample. The presence of rust and residual reagents (lime) in the mill could also lead to variations between the tests. Thorough cleaning of the mill before tests was undertaken. The mill was filled with lime water between the tests to minimise grinding media oxidation. Fresh reagent solutions were prepared before each batch of tests as some of these reagents are known to deteriorate with time.

Analytical errors

Representative samples from concentrates were dissolved in aqua regia before analysing for metal content using atomic absorption spectrometer while sulphur was analysed directly using ELTRA combustion analyser. Measures were taken to minimise sampling and analytical instrument calibration errors. Mineral balances were calculated from elemental assays by linear algebra.

Leaching

Results from leaching experiments are discussed in this section. Individual leaching behaviour is compared against existing information and possible mechanisms are explored. The leaching results for mineral mixtures are evaluated with reference to results from individual mineral test.

5.8 **Bornite**

Overall bornite leaching behaviour was in agreement with the reported behaviour in literature [30]. In temperatures below 40C, bornite leaching is not expected to progress past the first stage of leaching. Copper extraction from bornite is supposed to stop at 28 %. For experiments conducted at 25°C shown in Figure 49, copper leaching levelled off at around 26 %, which is close to the expected value. Discrepancies in the extraction values could be brought about by the presence of chalcopyrite in bornite samples. Equation (2 - 19) can therefore be assumed to describe the reaction taking place at this stage.

In the absence of ferric ions the leaching rate was significantly low. From bornite leaching experiments conducted using oxygen as an oxidant, [29], the leaching of bornite is known to experience an induction stage which can last for over 2 hours before the rate increases considerably. In the absence of ferric ions, dissolved oxygen is expected to be the main oxidant, therefore leaching behaviour of these tests is expected to resemble that from tests conducted with oxygen as the oxidising agent. Taking into consideration that no air or oxygen was introduced to the leaching solution, activities of oxygen are expected to be low in these solution leading to an even slower oxidation of bornite as observed in these experiments. Non-oxidative leaching of bornite would also be prevalent under these conditions.

High temperature leaching of bornite also follows the expected behaviour in which the sharp initial rate starts to fall off with time especially after 50% copper extraction. Leaching of bornite at this stage is thought to be that described by equation (2 - 20). Temperature in this case is

enough to overcome the higher activation energy required by the intermediate and second stage. Effects of acid strength and particle size were minimal.

Maintaining ORP at 750 mV resulted in sustained higher leaching rate for the second stage which had a linear trend. High ORP did not show improvement in the early stages of bornite leaching. Faster kinetics of this stage and relatively high ORP at the beginning of the experiment could be the reason for no observable variations. Increased leaching of Stage II is in agreement with the leaching equation of this stage, in which ferric ions are on the reactant side of the equation.

Analysis of leaching residue showed mineral particles covered by elemental sulphur. SEM scan combined with X-ray analysis detected a phase with a composition closely matching that of Cu_3FeS_4. In the residue, chalcopyrite particles were also detected by SEM. From literature [30], it is known that the X-ray diffraction pattern of idaite closely resembles that of chalcopyrite. Due to the unavailability of idaite in the XRD database, XRD scan of the residue reported only chalcopyrite as the copper sulphide mineral even though the presence of idaite was shown by SEM scans and inferred by elemental balance of the residue.

5.9 **Chalcopyrite**

Chalcopyrite leaching in sulphate media is known to have a slow kinetics. Experiments performed on the leaching of chalcopyrite at room temperature are normally in the order of days for significant leaching results to be observed [38].

Given that experiments conducted at 25°C were only for 4 hours, not much extraction was expected from chalcopyrite leaching. Low temperature results were poor as expected. Leaching in the absence of ferric ions has been proposed to be mainly non-oxidative, [54], and the proposed reaction is shown by equation (5 – 1) below:

$$CuFeS_{2(s)} + 4H^{+}_{(aq)} = Cu^{2+}_{(aq)} + Fe^{2+}_{(aq)} + 2H_2S_{(aq)} \quad \textbf{(5 - 1)}$$

In the presence of ferric ions, at the same temperature equation (2 - 23) is suggested to describe the reaction taking place under these conditions.

Leaching at 90°C in the presence of ferric ion produced the best results. The -400 mesh fraction had about 20% copper extraction after 4 hours or 30% copper extraction after 6 hours Figure 52. Even though the rate after the first hour seemed to reduce, the leaching was going at a steady rate when the experiment was stopped.

SEM scans on randomly selected samples (Figure 59) did not show sulphur growth on mineral particles as observed in the case of bornite, but analysis of total sulphur by combustion on leached residues showed increased sulphur content from that of the original sample (39% vs. 30%), which becomes even pronounced in terms of sulphur to copper ratios when factoring in the reduced metal content in the residue. XRD analysis detected chalcopyrite, sulphur and quartz as the only mineral phases in the residue. Equation (2 - 23) is still expected to describe leaching of chalcopyrite under these conditions, while equation (5 – 2) and equation (5 – 1) would be expected to describe reactions in the absence of ferric ions.

$$CuFeS_{2(s)} + 4O_{2(aq)} = Cu^{2+}_{(aq)} + Fe^{2+}_{(aq)} + 2SO^{2-}_{4(aq)} \qquad (5 - 2)$$

In the absence of ferric ions, decreased amounts of dissolved oxygen concentration due to high temperatures would most likely limit oxidation by oxygen leaving non-oxidative leaching as the main leaching mechanism.

Maintaining ORP at 750 mV (against Ag/AgCl) resulted in a sustained linear leaching rate that was steeper than the case with no ORP control Figure 65. Leaching rate in the first 30 minutes seems a bit steeper than the rate that followed but overall the leaching rate was faster. Chalcopyrite is known to exhibit trans-passive behaviour at high redox potentials [33]. This reasoning could be used to explain the lack of observed passivation in the leaching behaviour. However, leaching was stopped at relatively low copper extraction which could also be the reason for no observed

passivation due to lack sulphur formation. No observable passivation was seen in the case without ORP control either, although the rate was lower.

5.10 **Galvanic Interactions in Mixtures**

Leaching results from tests performed on mineral mixtures at 25°C in the presence of ferric ions closely follow the expected behaviour depending on the ratios of the minerals in the system. Due to the copper extraction from bornite and chalcopyrite under similar conditions being 26% and 1% respectively, mineral mixtures of 1:1 and 1:3 ratios were expected to have copper extractions of about 13.5%, and 7% respectively. The observed 16% and 8% extractions from those mixtures are slightly above the expected values, Figure 53 and Figure 55. These values would not be too far from normal variations but the positive bias of these values could be interpreted to reflect slight effect of galvanic interactions on copper extraction.

In the absence of ferric ions at 25°C copper extractions were around 4% and 0.7% for bornite and chalcopyrite respectively. The expected extraction from 1:1 and 1:3 mineral ratios would be around 2% and 2% respectively which is in agreement with the leaching results of these mixtures. It can therefore be concluded that galvanic interactions did not show any observable effects at 25°C in the absence of ferric ions. Lack of oxidant and low temperatures could be the reason for slowed reactions thus low extractions, making it difficult to observe galvanic interactions effects if they occurred. Low extraction percentages result in considerable error in the results making it difficult to observe small effects.

From individual mineral leaching, the expected copper extractions for mineral mixtures at 90°C in the absence of ferric ions were to be around 11% and 6% for 1:1 and 1:3 bo-cp ratios. The observed copper extractions for these cases were 11% and 7% as shown in Table AC - 7 and Table AC - 9. From these values it can be concluded that no significant galvanic effects are observable. Lack of oxidant in the system due to dissolved oxygen concentrations being low at high temperature could be the reason for slowed oxidation reactions thereby suppressing the effects of galvanic

interactions. Non-oxidative leaching together with limited oxygen assisted leaching can be assumed to describe leaching reactions taking place under these conditions. If non-oxidative leaching was the prevalent mechanism as suggested, galvanic effects would be minimised.

High temperature, 90°C, and the presence of ferric ions produced pronounced departures from expected behaviour, Figure 54, Figure 56 and Figure 57. Initial leaching rates followed expected for periods corresponding to first and intermediate stage of bornite leaching: up to around 50% copper extraction from bornite.

Due to the rapid leaching rate in periods corresponding to first stage of bornite leaching, galvanic effects could not be adequately analysed for. Equations (2 - 19), (2 - 20) and (2 - 23) can be assumed to describe bornite and chalcopyrite reactions during this period. There are no observed effects of galvanic interactions at this stage.

Deviations from expected behaviour were observed in periods corresponding to second stage of bornite leaching. Normal leaching equations for individual minerals at this stage are given by equations (2 - 20) and (2 - 23) for bornite and chalcopyrite respectively. In all of the three cases studied, the resulting copper extraction was considerably higher than the expected value, meaning the presence of both minerals in one system increased the leaching rate of at least one of the minerals.

From the leaching result of these mixtures the 1:3 showed a steeper slope followed by 3:1 and 1:1. Extraction seemed to plateau around 90% copper extraction. The 1:1 mixture did not reach 90% copper extraction. Residue analysis by SEM showed sulphur growing from chalcopyrite particles Figure 61 and “idaite” covered with platy-looking, crystalline form of sulphur layer, Figure 62. XRD analysis did not detect any bornite, but mainly sulphur and chalcopyrite together with gangue minerals. Sphalerite was also detected.

Experiments carried out with redox potential maintained at 750 mV also showed similar behaviour of increased extraction even though they were ran for a shorter time. This is shown in

Figure 66. Departures from expected behaviour were still delayed until the period corresponding to the second stage of bornite leaching. Possible explanations for this behaviour are explored below.

1. **Accelerated Bornite Oxidation due to galvanic contact with chalcopyrite**

Anodic species are expected to experience increased oxidative leaching due to galvanic currents while cathodic species are expected to act as reducing sites and experience less tendency to dissolve. This is similar to the explanation given in the pyrite - chalcopyrite system [51], [48]. Taking into consideration that the most observable departure from expected leaching behaviour occurs in what could be thought of as Stage II of bornite leaching, it would then be fitting that the galvanic interaction present would be between chalcopyrite and the newly formed intermediate product form bornite leaching. This intermediate product will be regarded as idaite or Cu_3FeS_4 for the purposes of this discussion even though no additional work was done to ensure that the intermediate product exactly fits this description.

Rest potential for this intermediate product was not determined but for the purposes of this reasoning it is assumed to be lower than that for chalcopyrite. The anodic decomposition of this product to release copper and iron ions into solution, together with elemental sulphur occurs during oxidation. Contact with the cathodic chalcopyrite would increase the rate of this anodic dissolution leading to faster kinetics which could explain the increased leaching rate of these mineral mixtures. Anodic leaching of chalcopyrite on the other hand would be negatively affected by increasingly reducing conditions caused by galvanic current from the oxidation of idaite.

This reasoning falls short of providing adequate explanation when taking into consideration the observed total copper extraction at the end of the experiment, especially in the 1:3 and 1:1 cases. Being the cathodic mineral in this case, chalcopyrite is expected to experience decreased anodic leaching according to this galvanic leaching model. If the leaching rate of chalcopyrite was to be lowered or stay unchanged, 30% or lower, the total copper extraction from the mixtures would not be expected to exceed 50% or 65% for the 1:3 and 1:1 cases respectively even if 100% copper

extraction from bornite was achieved. The observed copper extraction of 90% and 80% for these cases respectively is significantly greater than these projected amounts. This would definitely mean that copper extraction from chalcopyrite did increase despite it being the supposedly cathodic species in this system.

2. **Accelerated chalcopyrite anodic oxidation due to contact with idaite (Cu_3FeS_4)**

Due to the lack of knowledge of the rest potential for the intermediate product formed in the oxidation of bornite, it could be possible for this product to have a higher rest potential than chalcopyrite. Bornite has been shown to have a rest potential higher than that of chalcopyrite in the presence of ferric ions [52].

As a cathodic mineral, idaite leaching would be expected to negatively affect depending on the current density while chalcopyrite on the other hand would leach at an increased rate. Assuming an unaffected copper extraction from bornite, the observed leaching results would suggest a total copper extraction from chalcopyrite of 90%, 70% and 90% for 3:1, 1:1, and 1:3 mixtures.
Although this is a plausible explanation, in a study to investigate effects of galvanic interaction between sulphide minerals, galvanic current (admittedly low) was seen to be flowing from bornite to chalcopyrite [52].

ORP assisted copper extraction

In a study to investigate the active-passive nature of chalcopyrite leaching [33], the active range of chalcopyrite leaching ranged between 400 mV and 600 mV (SCE). The monitoring of redox potential during experimentation was not continuous. Even though the ORP was not adequately followed for these experiments, the initial solution ORP at 90°C was around 750 mV and fell between 500 mV and 600 mV range after an hour of leaching depending on the amount of bornite – more bornite in the sample led to lower ORP after the first hour of leaching. It is therefore plausible to attribute the increased leaching rate in this mixtures to the solution ORP falling in the active range of chalcopyrite anodic leaching.

The occurrence of increased leaching even when the solution ORP was held at 750 mV however would suggest that this reasoning is inadequate.

Improved leaching due to the presence of cupric and ferrous ions in solution.

The presence of ferrous and cupric ions have already been proposed to improve copper extraction from chalcopyrite by forming chalcocite as an intermediate product as shown in equation **(2 - 35)**. Ferrous ions could also be replaced by supply of electrons to carry out the reduction.

The chalcocite formed as the intermediate product would then be oxidised by ferric ions in solution or by dissolved oxygen as suggested in equations (2 - 36) and (2 - 37).

Figure 67 showed that experiments carried out by leaching chalcopyrite in a pregnant solution, in which bornite samples were leached in for an hour at 90°C in the presence of ferric ions showed improved copper extractions from chalcopyrite by those solutions

A solution, in which 3 g of bornite was previously leached, resulted in relatively higher extraction compared to that in which 1 g of bornite was leached in, both of which were considerably higher than the cases with no prior bornite leaching. This observation could be used to support the idea that enhanced copper extraction was a result of the presence of cupric and ferric ions in solution as higher concentration leads to higher copper extraction percentages, Figure 67.

The observed improved copper extractions from chalcopyrite are still somewhat lower than what the results from leaching of the mineral mixtures would suggest. Also, enhanced leaching had been observed when ORP was held high at 750 mV. Even though cupric ions are present in solution when the ORP is maintained high, this observation would question the necessity of ferrous ions in acceleration of chalcopyrite leaching as their concentration would be quite low. Further possible explanation is explored below:

Anodic leaching of bornite (and idaite) would generate electrons which would be conducted through to chalcopyrite by galvanic contact. From equations 4, the dissolution of "idaite"

to cupric, ferrous ions and sulphur generates 8 moles of electrons compared to 4 in the first stage of bornite leaching. The electrons generated are consumed by the reduction of ferric ions but another suggestion would be that these electrons may drive the reduction of chalcopyrite and increase the rate of formation of chalcocite. The presence of cupric ions in solution would also promote this reaction. The overall suggested reactions for the generation of chalcocite could then be:

Anodic reaction:

$$Cu_3FeS_4 = 3Cu^{2+} + Fe^{2+} + 4S^0 + 8e^- \quad \textbf{(5 - 3)}$$

Cathodic reactions:

$$2CuFeS_2 + 6H^+ + 2e^- \rightarrow Cu_2S + 3H_2S + 2Fe^{2+} \quad \textbf{(5 - 4)}$$

Or,

$$CuFeS_2 + 3Cu^{2+} + 3e^- = 2Cu_2S + Fe^{3+} \quad \textbf{(5 - 5)}$$

These reactions would also include:

$$CuFeS_2 + 3Cu^{2+} + 3Fe^{2+} = 2Cu_2S + 4Fe^{3+} \quad \textbf{(5 - 6)}$$

Chalcocite would then be oxidised relatively easily leading to improved copper extraction from chalcopyrite. This anodic reaction of bornite (idaite) would not be inhibited, in fact it would be promoted by the reduction taking place on chalcopyrite. Overall the proposed mechanism would lead to increased copper extractions from both minerals and could be a possible explanation for the observed result.

Possible sources of error

Experimental Errors

Care was taken when drawing aliquots for assaying to avoid removing solids and changing the mass of the sample in the test. Minerals tended to float and sometimes deposit on

flask wall, therefore not being subjected to the leaching solution. Flask wall were washed periodically with the leaching solution to ensure the samples were subjected to leaching.

There was variation in the reproducibility of leaching results for mineral mixtures. However, enhanced copper extraction was always observed with varying magnitudes. Possible causes for this observation include: The degree of contact between the minerals during leaching; temperature consistency; redox potential. Stirring was maintained at a steady rate to ensure consistency in mixing, hence, maintaining relatively similar level of contact between the minerals in each test. Temperature was maintained within +/- 2°C.

Redox potential is an important parameter in this system, which was not monitored in all experiments. Different amounts of sample used would affect the redox potential differently during leaching especially in relation to different ratios of bornite and chalcopyrite use. This potential was not monitored or controlled for most of the experiments. When the redox potential was set to 750 mV, it was observed to fluctuate between 720 mV and 760 mV although an attempt was made to control it at set point by hydrogen peroxide addition.

Silver particles were detected by SEM scans in some of the bornite and mixtures residue. XRD scans of the same samples did not detect any silver, which possibly meant that this silver was only available in very low quantities. Silver ions have been shown to improve chalcopyrite leaching [49], therefore the presence of this silver could possibly influence the result. The pH and ORP probes contain silver chloride which, although unlikely, could leak into and contaminate the leaching solution.

Experimental Errors

Analytical Errors

Representative mineral samples were analytically weighed from respective bulk samples of known elemental content. These samples were leached in known volumes of lixiviants and copper

extraction was determined by analysing for copper concentration in solution. Care was taken during sampling to minimise metal content variation between tests.

The final solution at the end of the tests was transferred into a volumetric flask to make a standard solution which was analysed for copper content to validate the copper content determined from samples taken during the experiment. In some tests the residue was also analysed for metal content to further validate the extraction percentages.

Chapter 6

Summary and Conclusions, and Recommendations

6.1 Flotation

Copper ore of 3.3% copper head grade containing roughly equal amount of bornite and chalcopyrite, (3.2% and 3.4% respectively) was floated in the presence and also in the absence of a collector to investigate galvanic effects on the flotation of the two minerals.

Collectorless flotation favoured the recovery of chalcopyrite over that of bornite. In such cases, the mineral recoveries after 1.5 minutes of flotation at pH 9 were about 66% for chalcopyrite and 20% for bornite.

Flotation in the presence of a collector resulted in a large increase in the recovery of bornite while chalcopyrite recovery stayed relatively unchanged, especially during the initial stages and slightly improved with time. Increase in collector dosage resulted in corresponding increase in bornite recovery while chalcopyrite was marginally affected. It was concluded that the recovery of chalcopyrite was still mainly through collectorless flotation mechanism, thereby obscuring the results of galvanic effects on competition for collector.

In this galvanic system, chalcopyrite is expected to be the cathodic component while bornite is anodic. Galvanic contact between the two minerals is expected to increase the rate of anodic reactions on bornite while chalcopyrite experiences reducing conditions from galvanic current. This alteration in reactivity of the minerals would affect their flotation performance under both collectorless flotation and flotation with a collector. The effect of galvanic interactions on each mineral would also be affected by the relative amounts of each species. The study on a real ore could have benefitted from a simplified system having bornite and chalcopyrite as single minerals, and exploring galvanic interactions on a smaller scale under more easily controllable conditions. Due to the lack of individual flotation behaviour of the minerals involved under similar

conditions, it is difficult to assess how their recoveries can be affected in natural complex ore, which might have additional electroactive components.

Collectorless flotation

Chalcopyrite exhibited a strong collectorless flotation behaviour, as expected also based on flotation literature. However, without individual flotation performance data, it was difficult to judge whether this behaviour was enhanced or retarded by having bornite in the system. Due to the collectorless flotation of chalcopyrite being known to be negatively affected by reducing conditions, it could be expected that the flotation performance of chalcopyrite in this case could have been lower than what it would normally be in the absence of bornite. Another aspect of this flotation system is the grinding media affect, which may have had a masking effect on the galvanic interaction between the two sulphide minerals.

Bornite showed a relatively poor collectorless flotation behaviour compared to chalcopyrite. The presence of chalcopyrite in this system would be expected to increase anodic activity of bornite, e.g., to leave behind a metal-deficient sulphur-rich layer which would improve its collectorless flotation. Oxidation of bornite by oxygen generally results in preferential iron dissolution which could form hydrophilic ferric hydroxide species on bornite. This can also be a galvanic mechanism leading to a poor collectorless flotation behavior of bornite. Bornite collectorless recovery was seen to increase with time as more chalcopyrite was removed from the pulp, suggesting that the presence of chalcopyrite (and its associated galvanic effect) could have retarded the flotation of bornite in the initial stages.

Flotation with a collector

Flotation in the presence of xanthate as the collector is expected to favour the collection of the anodic mineral due to the presence of greater cationic activity on the mineral surface. For this

reason, galvanic interactions between these two minerals were expected to favour the recovery of bornite over that of chalcopyrite. This was the observation from the experiments carried out with the use of a collector. Galvanic interaction would also be expected to cause reducing conditions on chalcopyrite surface thereby discouraging interactions with the same collector. Chalcopyrite flotation in the presence of a collector did not experience much improvement but due to its strong collectorless behaviour, its recovery stayed relatively unchanged and increased as more bornite was removed from the system.

The presence of oxidised fines in the feed resulted in a slight impairment of selectivity by introducing dilution in both collectorless and flotation with a collector cases. This was attributed to the effect of a greater activity of their copper ions generated during the size reduction steps.

Increasing pH from 7.8 to 9 improved the floatability of chalcopyrite, but a further increase to pH 11 showed a fall in the chalcopyrite recoveries. This was explained by the higher concentration of hydroxyl ions.

Flotation using nitrogen in the presence of a collector significantly slowed down the flotation kinetics for both minerals. It also lowered the pulp potential to around -200 mV. Sulphide minerals are known to float poorly under reducing conditions or when nitrogen is used to replace air during flotation. Chalcopyrite flotation (whether under collector-induced or collectorless case) was more significantly depressed while bornite floated relatively better leading to improved separation in favour of bornite. Residual galvanic effects could have been responsible for bornite's relatively higher flotation response because, as an anodic component, it can still react with xanthate relatively better due to its greater metal ion activity.

Recommendations

A lot was learnt from these experiments, but due to multiple variables inherent to the system, it was difficult to draw hard conclusion to the observation from these experiments. To

improve the understanding of the effects of galvanic interactions on this system several improvement could be made including:

Studies using high grade samples of each mineral could be performed to minimise the effects of parameters inherent to natural ore such as active gangue material and liberation.

These studies would also allow for investigation of individual mineral flotation behaviour for which results from a system containing both minerals can be compared to.

Variation of mineral ratios would be useful for evaluating the extent of galvanic interactions in these systems.

Extensive manipulation of pulp potential by the use of different modifying agents would also help clarify the effects of galvanic interactions. This can include rest potential measurements.

Fundamental studies linking the mineral flotation behaviour to surface chemistry would be helpful. Surface reactions involving each mineral under similar conditions should be conducted to increase the confidence level when analysing observations on more complex flotation systems. Much of the analysis in the current study is based on inferences than hard data.

6.2 Leaching

Leaching of a mixture of -400 mesh fractions of bornite and chalcopyrite in 0.1M sulphuric acid at 90°C in the presence of ferric ions produced higher copper extractions from mineral mixtures than that expected from the sum of their individual extraction percentages.

The 1:3 and 3:1 bornite-to-chalcopyrite mixtures achieved 90% copper extraction at these conditions after 6 hours while the 1:1 mixture achieved 80%. Taking into consideration that the individual performance of each mineral under similar conditions is 90% and 26% for bornite and chalcopyrite respectively, the observed extraction from the mineral mixtures are significantly higher than their expected values and suggests increased copper extraction from chalcopyrite.

Individually, chalcopyrite leached at more or less steady rate for the given leaching period while bornite experienced rapid initial rate followed by a considerable fall in leaching rate after about 50% copper extraction. Bornite is known to leach in two stages described by different equations. At high temperatures Stage I has very fast kinetics and usually does not last long while Stage II is relatively slower and continues to slow down with time.

The observed increase in leaching rate for the mineral mixtures occurred in times corresponding to Stage II of bornite leaching.

Under acidic conditions, in general, bornite has been reported to have lower rest potential than chalcopyrite, but another study performed in the presence of ferric ions showed a large increase in the rest potential of bornite, which significantly exceeded that of chalcopyrite. However the same study reported current flowing from bornite to chalcopyrite. This makes determination of cathodic and anodic species of this system difficult. Bornite is also known to form a stable intermediate product during Stage I, whose rest potential has not been reported.

Galvanic interactions are expected to accelerate the anodic leaching of the anodic mineral at the same time leading to reducing conditions on the cathodic mineral depending on their magnitude. Explaining the role of galvanic interactions on copper extraction in this system proved

challenging due to inherent conditions to the system that could promote the leaching of copper from chalcopyrite, and are not necessarily connected to galvanic effects.

Initial rapid leaching rate of bornite introduces cupric and ferrous ions into solution at the same time resulting in reduction of ferric ions in solution to ferrous ions. The presence of cupric ions and ferrous ions is known to enhance the dissolution of chalcopyrite.

To investigate the effect of the change in solution composition with the progression of leaching, chalcopyrite was leached in solutions in which 1 g and 3 g samples of bornite were previously leached in for 1 hour. Both solutions resulted in increased copper extraction from chalcopyrite. The ability of the solution composition to increase the leaching of chalcopyrite makes it difficult to evaluate the effects of galvanic interactions in this system.

The reduction of ferric ions in solution to ferrous ions and the introduction of ferrous ions from the minerals during initial stages of leaching also reduce the solution potential to below 600 mV. Chalcopyrite is known to exhibit active leaching behaviour in the region between 400 and 600 mV. For this reason it is not possible to differentiate how much of the increase in copper extraction was a result of favourable solution potential and how much was a result of galvanic interactions.

The observation of increased extraction even when the solution potential was held at one value, 750 mV, would suggest favourable solution potential if at all played a role, was not the only parameter that had an effect on the increase in copper extraction.

Even though cupric ions were present in solution at 750 mV, the concentration of ferrous ion would be very low. This would leave the presence of cupric ions and the galvanic interactions as the main parameters which would be responsible for increase in copper extraction from the mineral mixture. Through galvanic interactions, it is possible that the galvanic current flowing from bornite to chalcopyrite together with the presence of cupric ions influenced the formation of intermediate chalcocite, which then leached relatively easily resulting in increased copper extractions from the mineral mixtures.

Recommendations

Future research to expand on the current work and provide more insight into interpreting the current result would include:

Detailed rest potential and galvanic current measurements for bornite, its intermediate – idaite, and chalcopyrite in the presence of ferric ions to determine the nature of these species in this galvanic system.

Monitoring of residue composition from leaching of mineral mixtures at different time intervals to determine the effects of possible galvanic interactions on the leaching rate of each mineral.

Leaching under controlled redox potentials at different potentials would help determine the role of varying redox potentials present in this experiment in the enhanced copper extraction.

Investigation of the formation of chalcocite on chalcopyrite under different conditions to support or oppose the suggested mechanism of intermediate chalcocite formation being the mechanism of enhanced copper extraction from chalcopyrite.

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[99] R. H. Yoon, "Collectorless flotation of chalcopyrite and sphalerite ores by using sodium sulfide," *International Journal of Mineral Processing, 8,* pp. 31-48, 1981.

[100] M. Lundstrom, J. Aromaa, O. Forsen, O. Hyvarinen and M. H. Barker, "Leaching of chalcopyrite in cupric chloride solution," *Hydrometallurgy,* vol. 77, pp. 89-95, 2005.

Appendix A

Flotation Tests Performance Graphs

The experimental design for flotation experiments together with grade versus time plots are presented in this appendix.

Table AA - 1: Flotation experimental design

Test #	Fines	Collector	pH
1	1	1	1
2 & 3	1	1	-1
6 & 7 & 8	1	-1	1
9 & 4	1	-1	-1
10 & 12	-1	1	1
5 & 11 & 13	-1	1	-1
	-1	-1	1
14	-1	-1	-1

Parameters:

Fines: 1 represents a case in which fines were not removed from the feed.

-1 represents cases with -400 mesh fraction removed from the feed

Collector: 1 represents cases in which a collector was used in all concentrates

-1 represents collectorless flotation test for the first 3 concentrates

pH: 1 represents cases in which lime was used to adjust the pH.

-1 represents cases with no lime addition (natural pH)

Table AA - 2: Test conditions for all flotation tests

Test #	Lime/ pH	Fines	Collector (PEX)	Grinding Time(min)
Test 1	0.1g - 8.2		2ml/conc.	12
Test 2	0g - 8.0		2ml/conc.	12
Test 3	0g - 7.6		1ml/conc.	12
Test 4	0g - 7.7		0 - 1	12
Test 5	0g - 7.85	removed	2.5	12
Test 6	0.4g - 9		0 - 1	12
Test 7	0.8g - 10		0 - 1	12
Test 8	2.3g - 11		0 - 1	12
Test 9	0 - 7.7		0 - 1	12
Test 10	0.4g - 9	removed	2.5	12
Test 11	0g	removed	2.5	15
Test 12	10	removed	2.5	12
Test 13	0	removed	2.5	15
Test 14	0	removed	0 - 1	15

Test Conditions: Feed not screened for fines; Collector used for all concentrates; pH adjusted

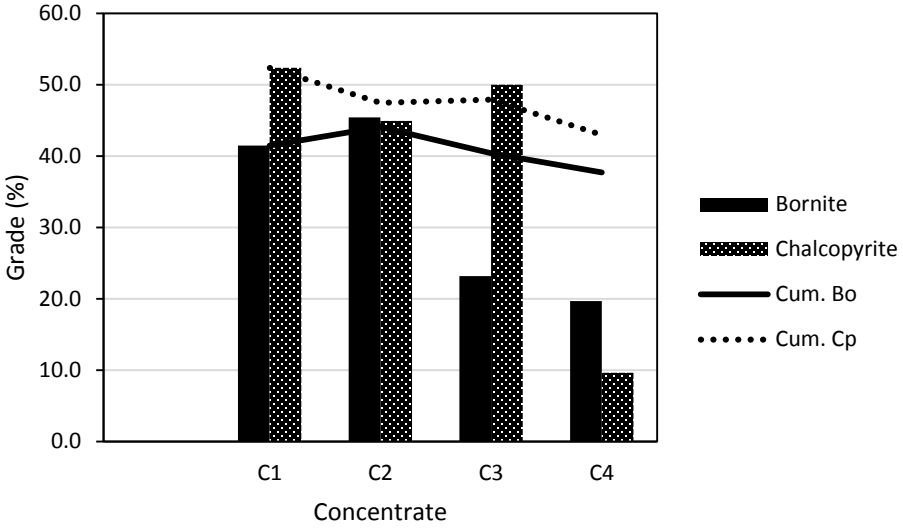

Figure AA - 1: Individual and cumulative grade data for Test 1. 0.1 grams of lime: pH 8.2

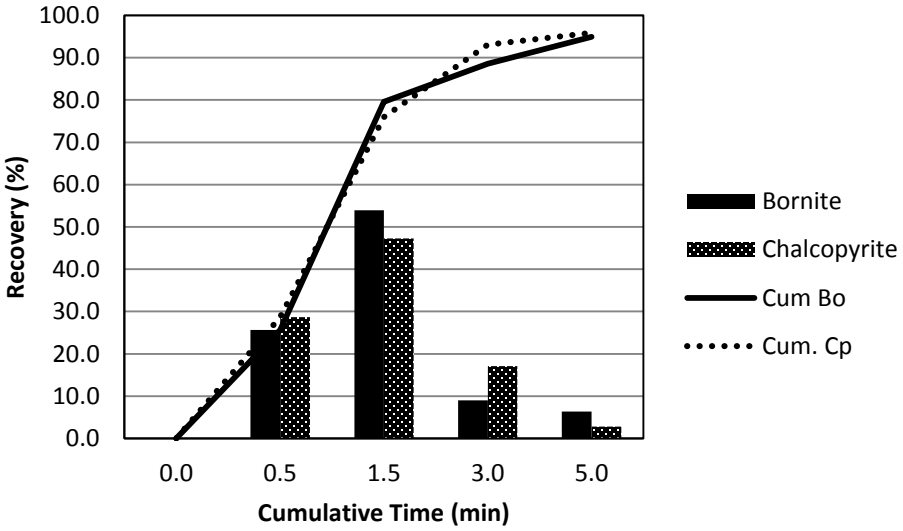

Figure AA - 2: Individual and cumulative recovery data for Test 1. 0.1 grams of lime: pH 8.2

Test Conditions: Feed not screened for fines; Collector used in all concentrates; pH not adjusted

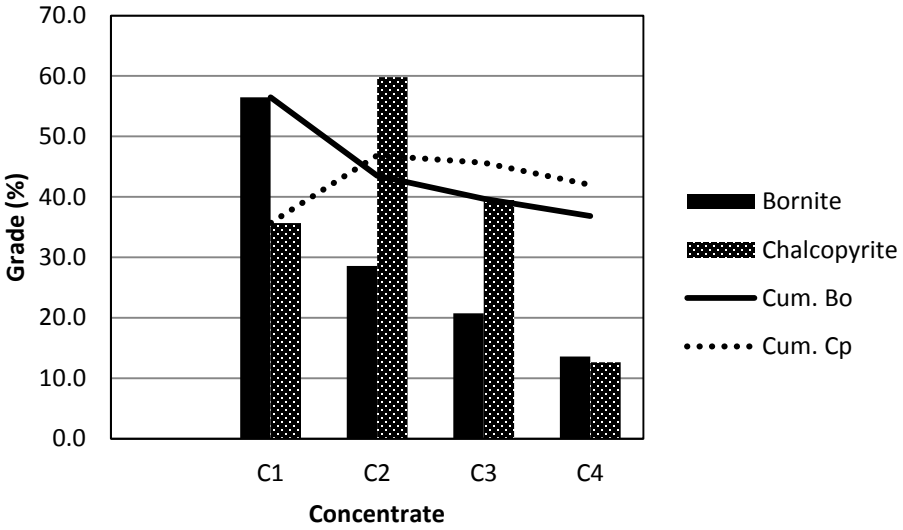

Figure AA - 3: Individual and cumulative grade data for Test 2. No lime added: pH 8.0

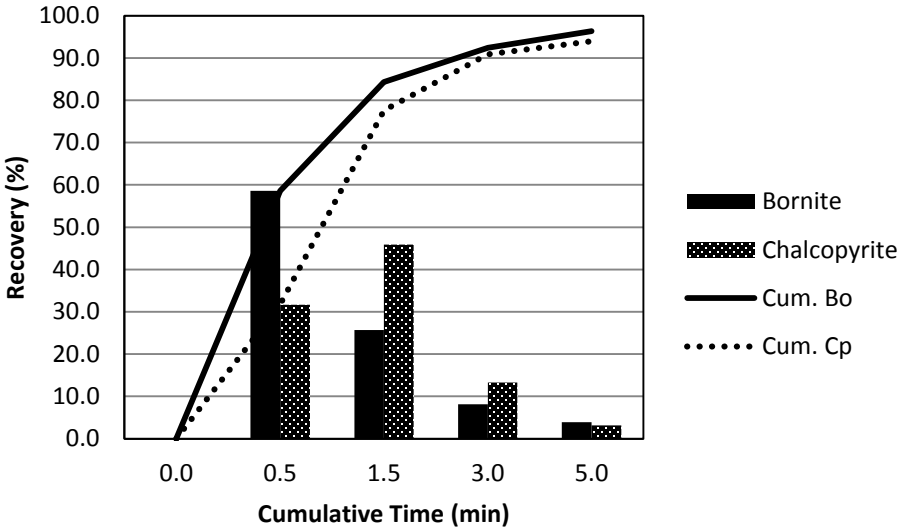

Figure AA - 4: Individual and cumulative recovery data for Test 2. No lime added: pH 8.0

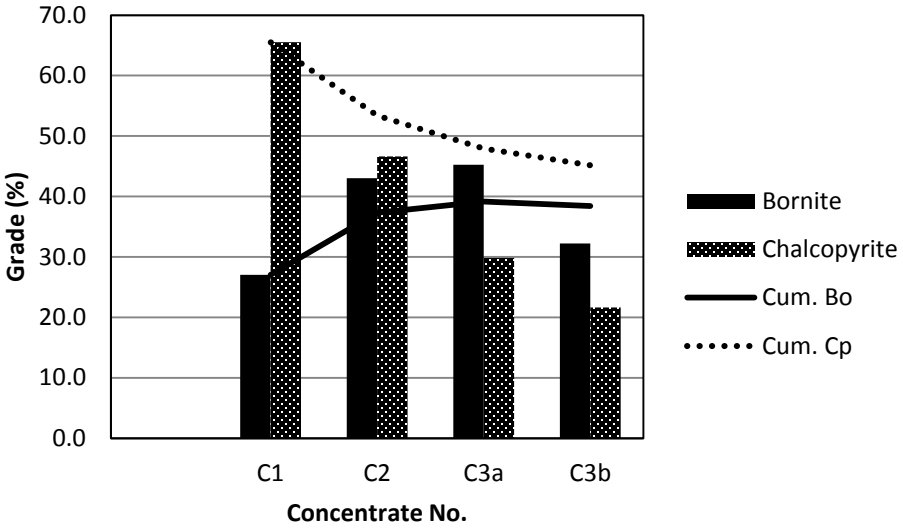

Figure AA - 5: Individual and cumulative grade data for Test 3. No lime added: pH 7.6

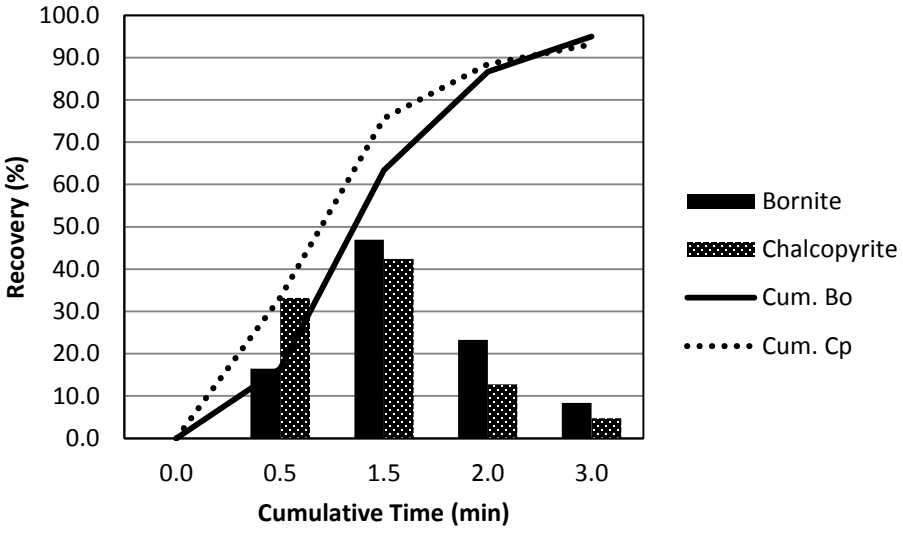

Figure AA - 6: Individual and cumulative recovery data for Test 3. No lime added: pH 7.6

Test Conditions: Feed not screened for fines; Collectorless first 3 concentrates; pH adjusted

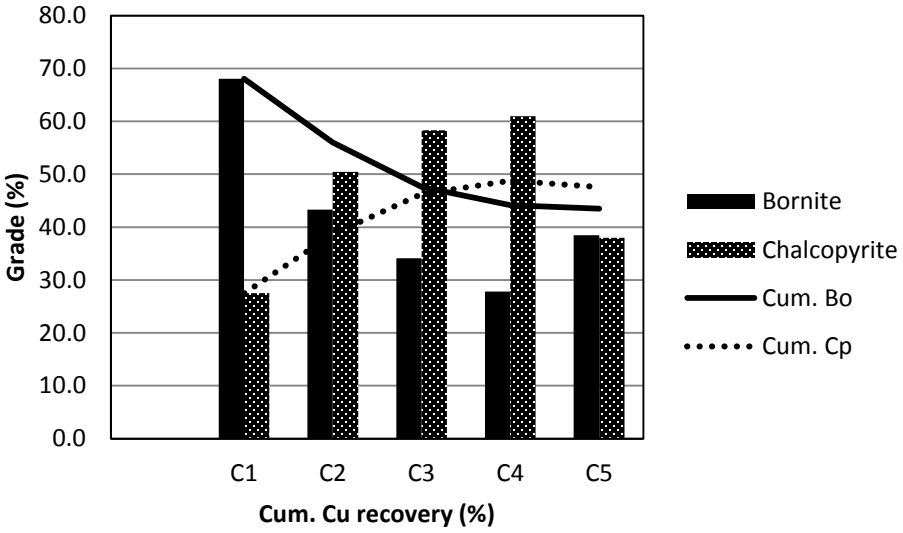

Figure AA - 7: Individual and cumulative grade data for Test 10. 0.4 gram lime added: pH 9

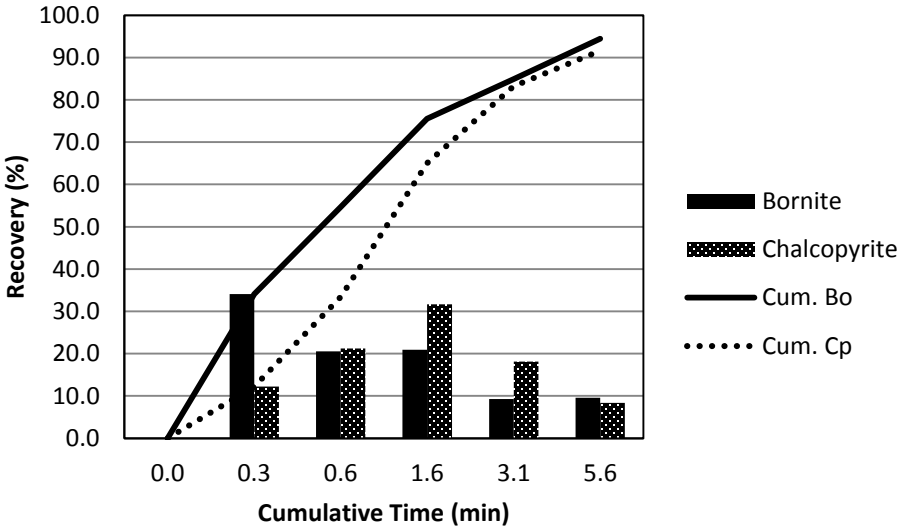

Figure AA - 8: Individual and cumulative recovery data for Test 10. 0.4 gram lime added: pH 9

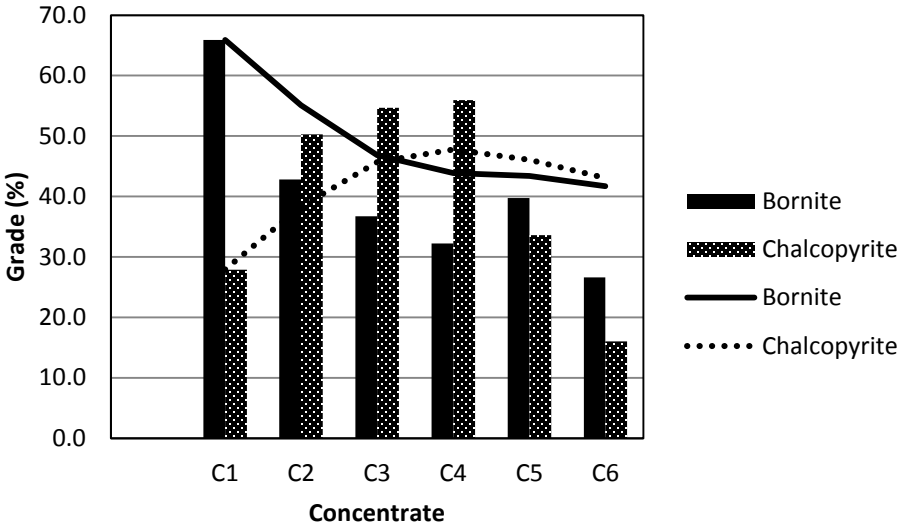

Figure AA - 9: Individual and cumulative grade data for Test 12. 0.8 gram lime added: pH 10

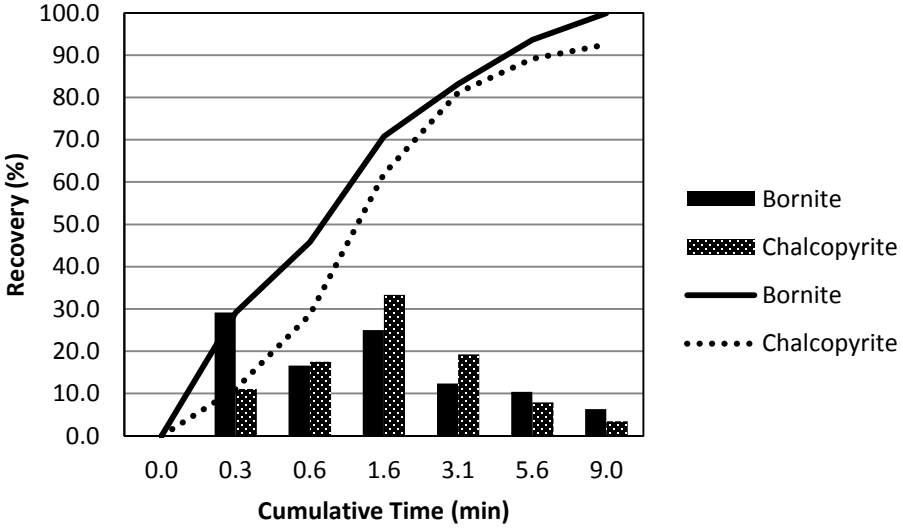

Figure AA - 10: Individual and cumulative recovery data for Test 12. 0.8 gram lime added: pH 10

Test Conditions: Feed not screened for fines; Collectorless first 3 concentrates; pH not adjusted

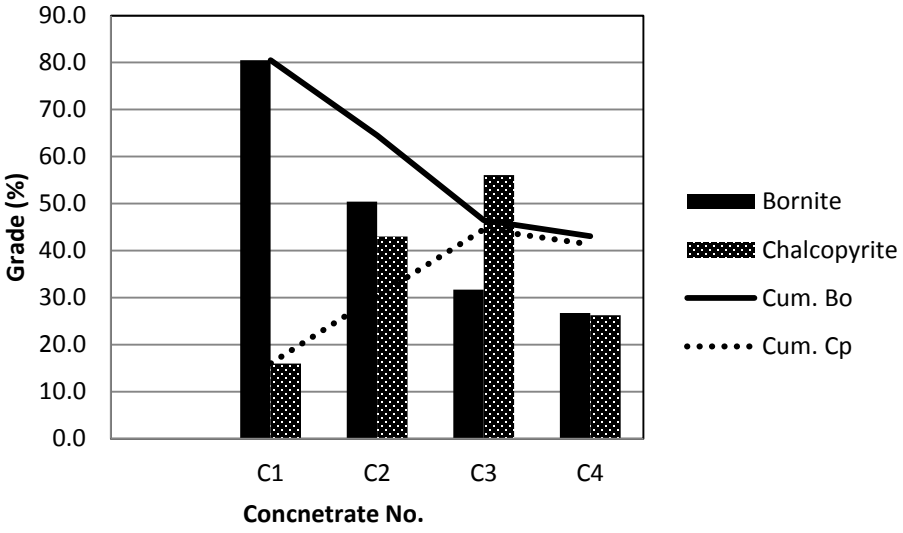

Figure AA - 11: Individual and cumulative grade data for Test 5. No lime added: pH 7.85

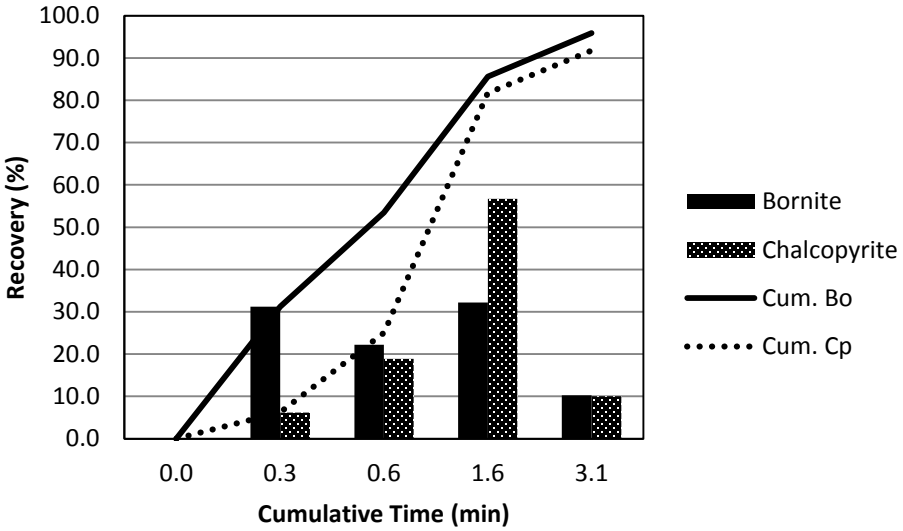

Figure AA - 12: Individual and cumulative recovery data for Test 5. No lime added: pH 7.85

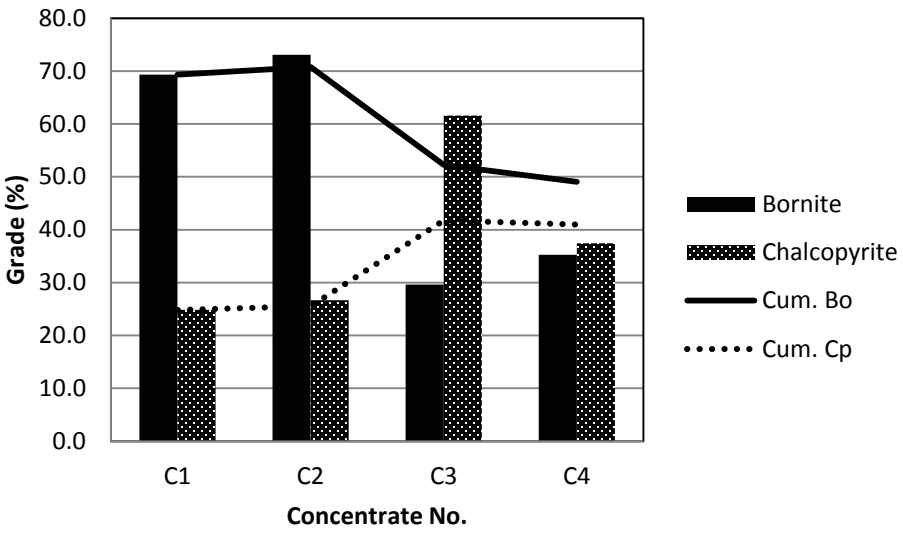

Figure AA - 13: Individual and cumulative grade data for Test 11. No lime added: pH 7.85; 15 minute grinding.

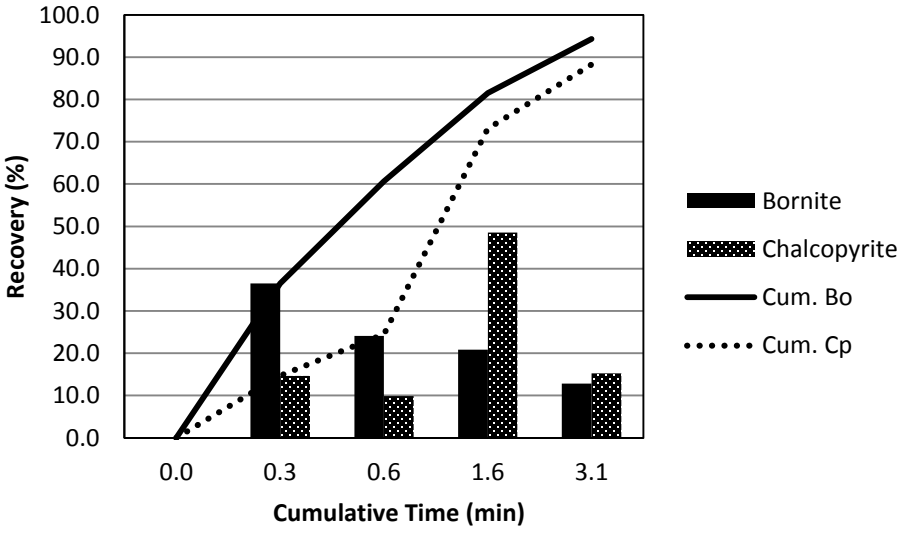

Figure AA - 14: Individual and cumulative recovery data for Test 11. No lime added: pH 7.85; 15 minute grinding.

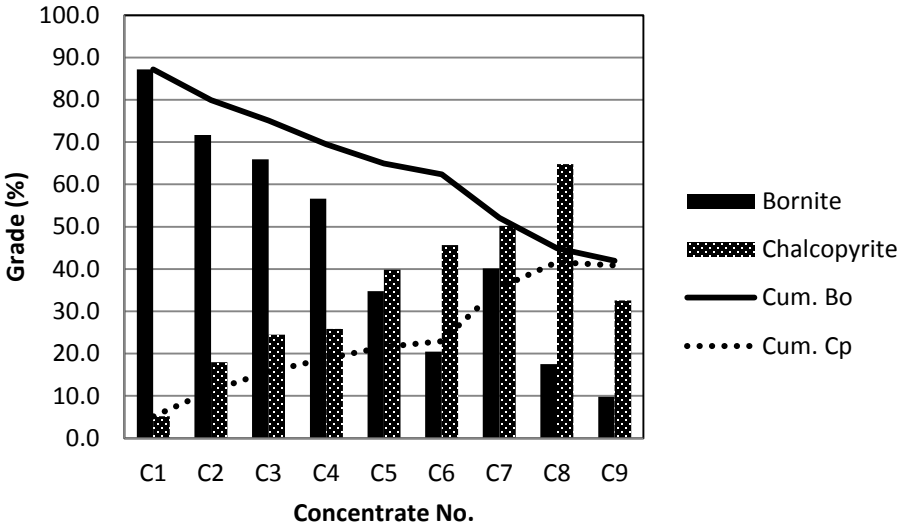

Figure AA - 15: Individual and cumulative grade data for Test 13. No lime added: pH 7.85; 15 minute grinding.

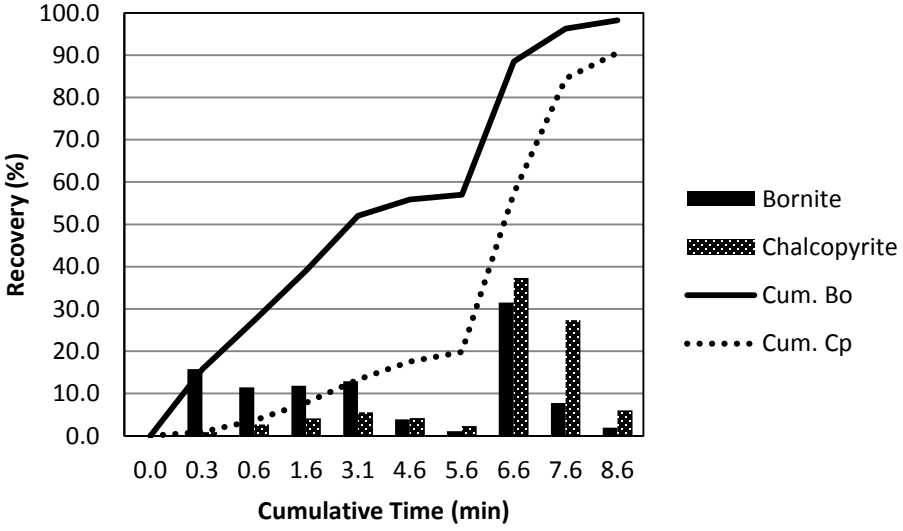

Figure AA - 16: Individual and cumulative recovery data for Test 13. No lime added: pH 7.85; 15 minute grinding.

Test Conditions: Feed screened for -400 mesh fraction; Collector used in all concentrates; pH adjusted

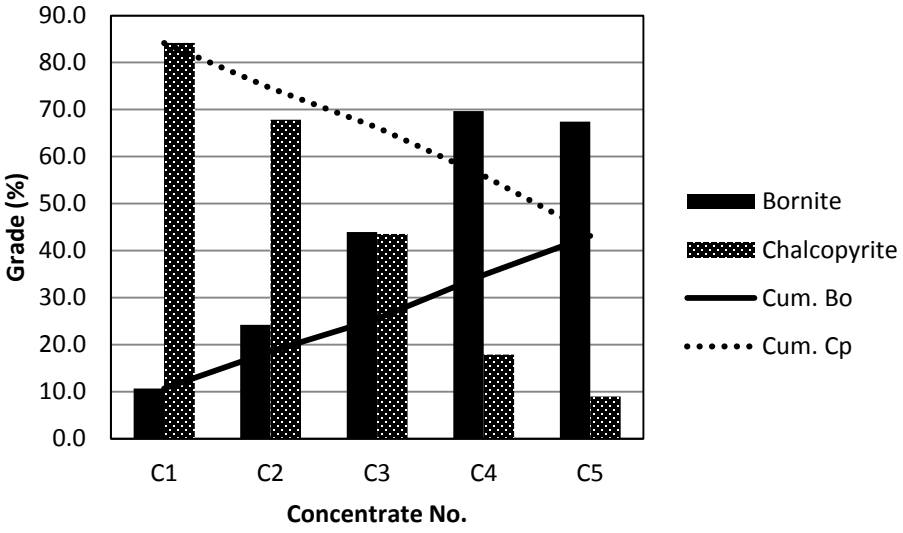

Figure AA - 17: Individual and cumulative grade data for Test 6. 0.4 grams of lime added: pH 9

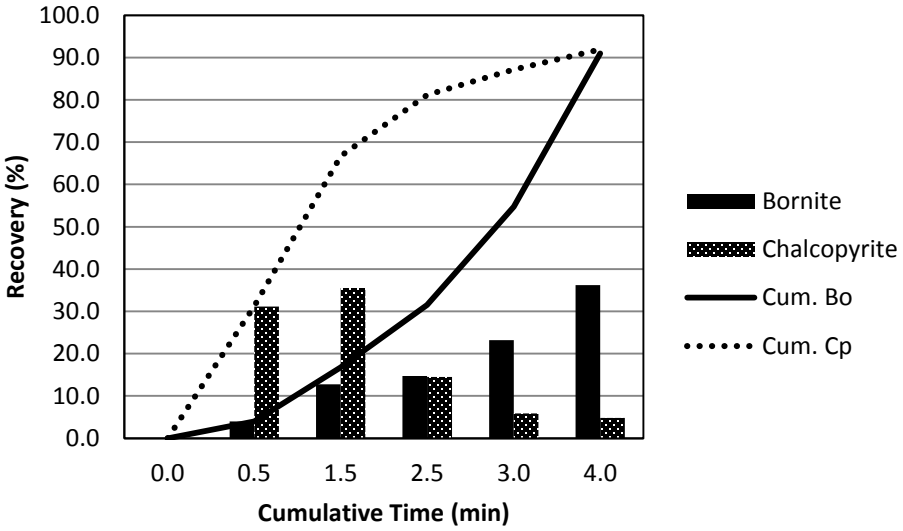

Figure AA - 18: Individual and cumulative recovery data for Test 6. 0.4 grams of lime added: pH 9

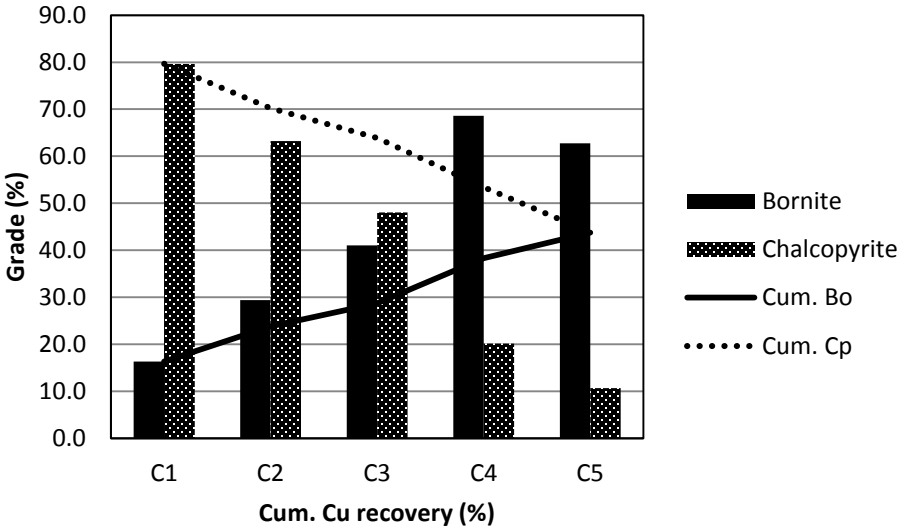

Figure AA - 19: Individual and cumulative grade data for Test 7. 0.8 grams of lime added: pH 10

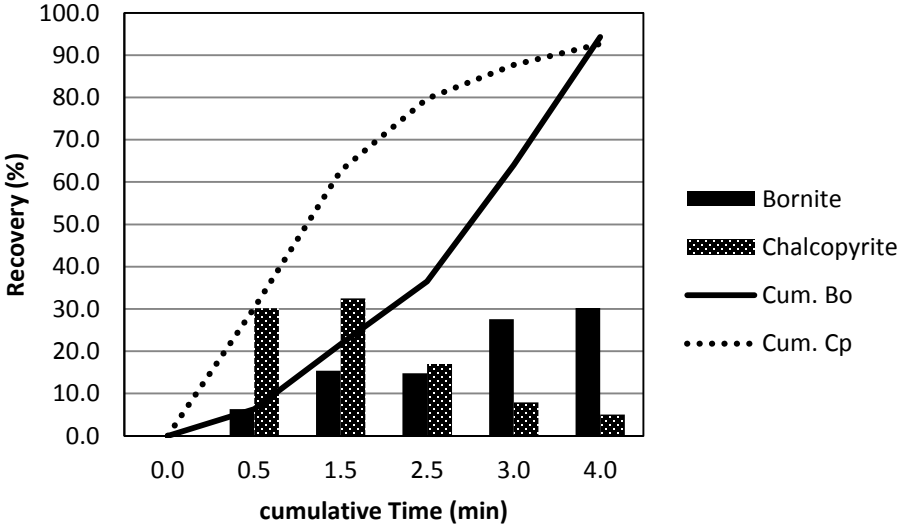

Figure AA - 20: Individual and cumulative recovery data for Test 7. 0.8 grams of lime added: pH 10

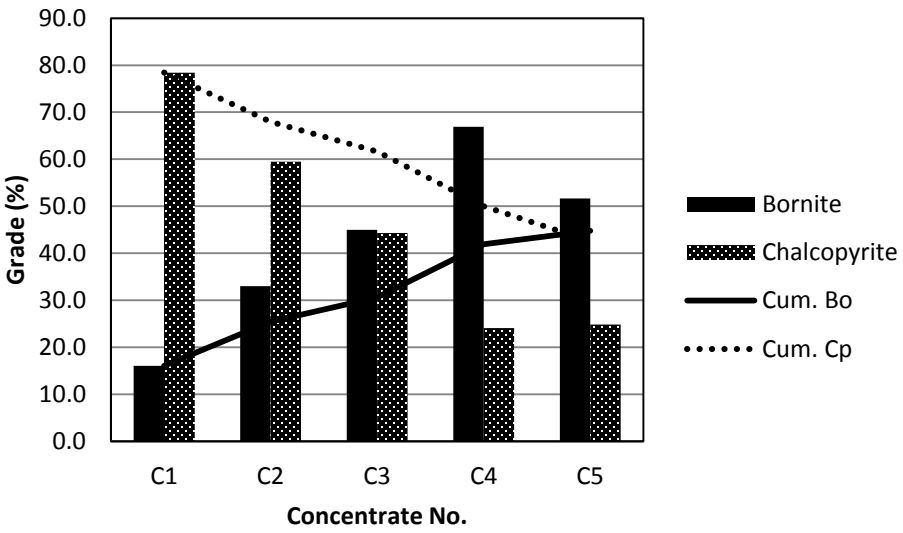

Figure AA - 21: Individual and cumulative grade data for Test 8. 2.3 grams of lime added: pH 11.

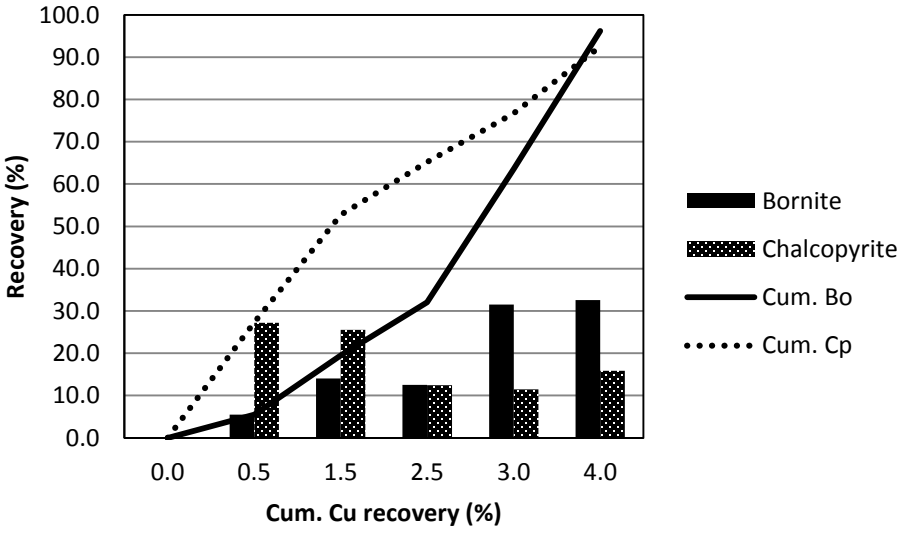

Figure AA - 22: Individual and cumulative recovery data for Test 8. 2.3 grams of lime added: pH 11.

Test Conditions: Feed screened for -400 mesh fraction; Collectorless first 3 concentrates; pH not adjusted

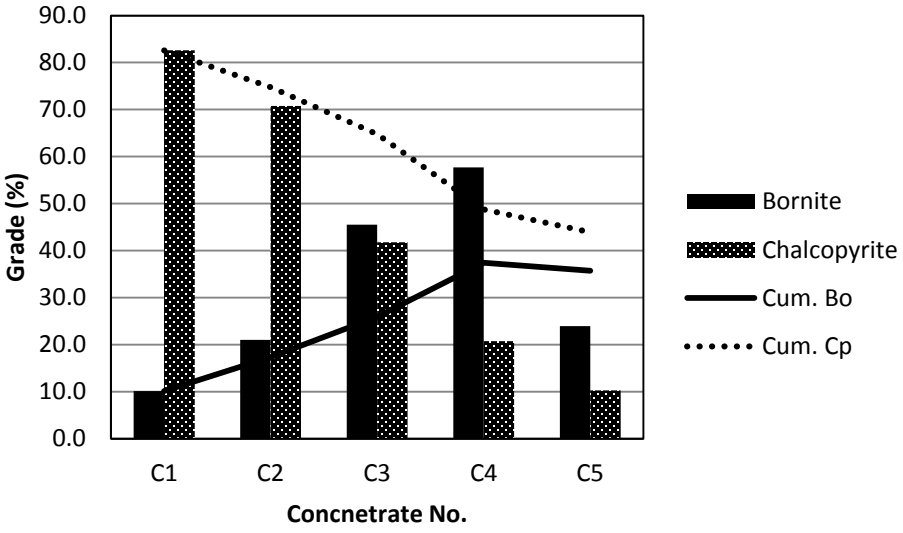

Figure AA - 23: Individual and cumulative grade data for Test 4. No lime added: pH 7.7.

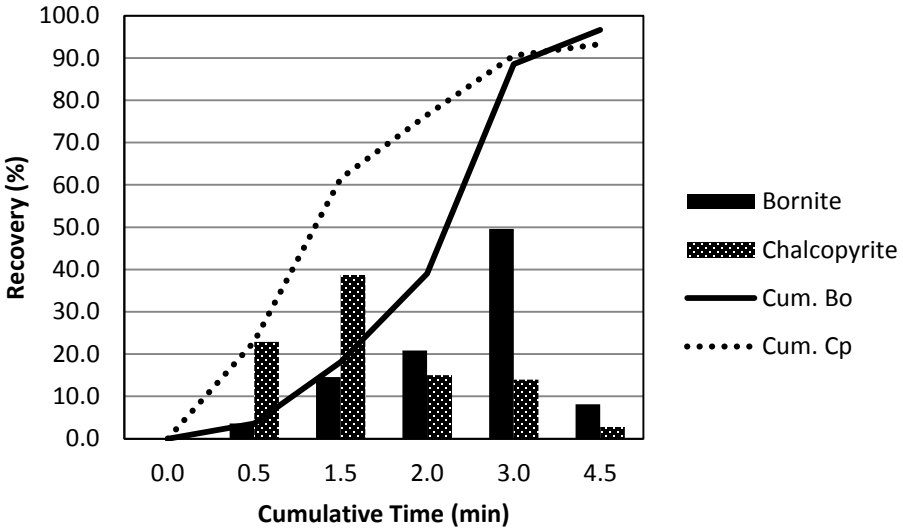

Figure AA - 24: Individual and cumulative recovery data for Test 4. No lime added: pH 7.7.

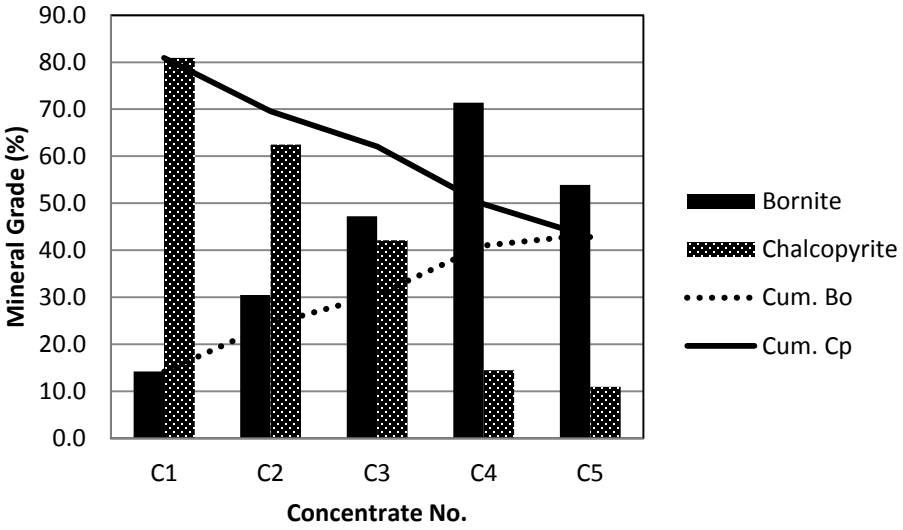

Figure AA - 25: Individual and cumulative grade data for Test 9. No lime added: pH 7.7

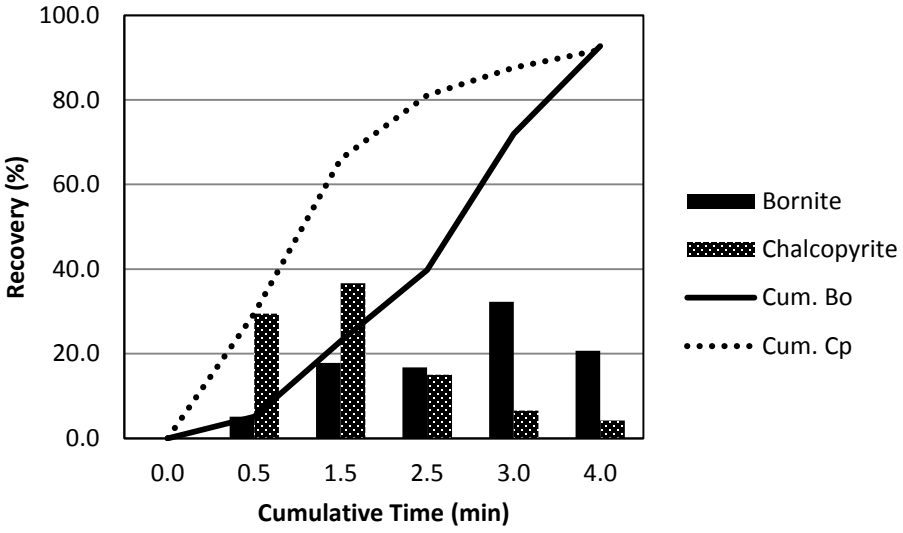

Figure AA - 26: Individual and cumulative recovery data for Test 9. No lime added: pH 7.7

Test Conditions: Feed screened for -400 mesh fraction; Collectorless first 3 concentrates; pH not adjusted

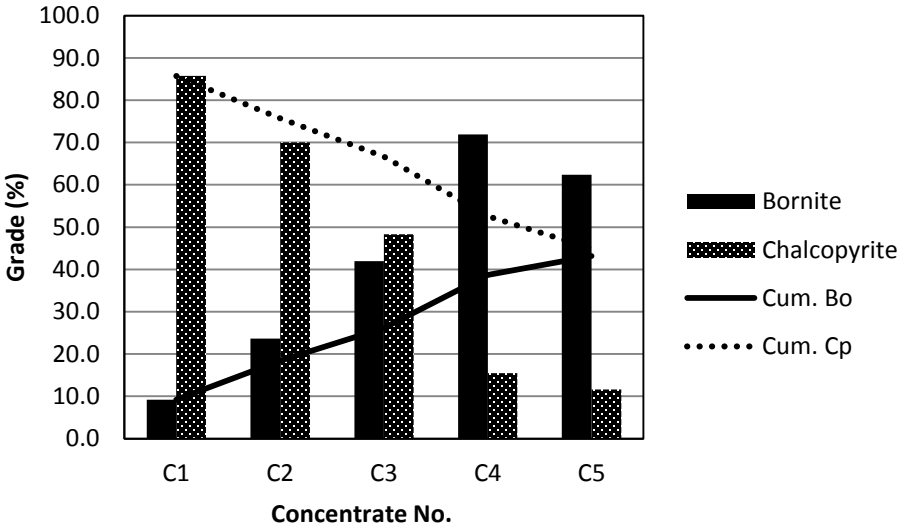

Figure AA - 27: Individual and cumulative grade data for Test 14. No lime added: pH 7.7; 15 minute grinding.

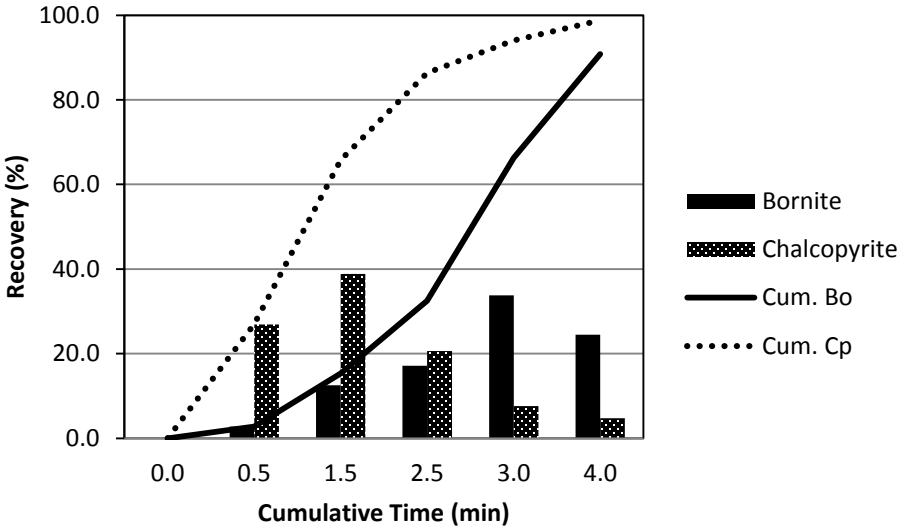

Figure AA - 28: Individual and cumulative recovery data for Test 14. No lime added: pH 7.7; 15 minute grinding.

Appendix B

Flotation Reports and Metallurgical Balances

Flotation report sheets and metallurgical balances for the flotation tests are presented in this appendix. The reports are presented in chronological order in which the tests were performed. A summary of tests conditions is presented in appendix A.

Table AB - 1: Test 1 flotation report. Feed with fines; pH adjusted to 8.2; collector present

TEST 1				DATE: March 17, 2011				
OBJECTIVE:	Flotation selectivity					Feed:	bornite-chalcopyrite ore	
GRINDING CONDITIONS (Denver Mill):					FLOAT CONDITIONS:		FLOATED BY:	
MILL TYPE:	ROD TYPE/Charge :	*As is*			CELL TYPE & VOLUME:		*2 L*	GAS: *Air*
CHARGE (g): *920*	65%	solids			IMPELLER RPM:	*1200*	REPULP WATE	*tap*
WATER: *495*					Ro GAS RATE:		by air valve	
GRIND: 12 min					# of STROKES: Ro:	*30/min*		
Lime 0.10 g/ml		STAGE	COND	FLOAT	pH	DOX / pH: as	REDOX	DO
REAGENTS			TIME	TIME	End		mV	
Redox					After 1 min stirring			
			1		8.31		44	
KAX 0.1%	2							
DF 250 0.1%	5							
		Cond	1		8.3		5	
		Float #1		0.5	8.25		71	
KAX 0.1%	2							
DF 250 0.1%	5							
		Cond	1					
		Float #2		1	8.23		147	
KAX 0.1%	2							
DF 250 0.1%	5							
		Cond.	1		8.22		96	
		Float #3		1.5	8.2			
KAX 0.1%	2							
DF 250 0.1%	5							
		Cond.	1		8.18		160	
		Float #4		2	8.16		177	
KAX 0.1%	2							
DF 250 0.1%	5							
		Cond.	1		8.15		90	
		Float #5		2	8.13		161	
KAX 0.1%	2							
DF 250 0.1%	5							
		Cond.	1		8.11		74	
		Float #6		2	8.09		139	
Totals								
Lime			9	Minutes				
KAX	12							
DF 250	30							

Table AB - 2: Metallurgical table for Test 1

			Cu				Fe			S
Products	Anlytic Wt. (g)	Vol.	ppm	Dilution	mg	%	ppm	Dilution	mg	%	%
Conc 1	0.4085	250	14.5	50	181.3	44.37	6.59	50	82.4	20.17	28.90
Conc 2	0.4022	250	28.5	25	178.1	44.29	12.50	25	78.1	19.42	27.30
Conc 3	0.4098	250	31.5	17	131.3	32.03	19.70	17	82.1	20.03	23.40
Conc 4	0.4052	250	30.7	8	63.96	15.78	21.70	8	45.2	11.16	8.40
Conc 5	0.4034	250	20	5	25.00	6.20	28.70	5	35.9	8.89	3.80
Conc 6	0.4036	100	19.5	8	16.25	4.03	12.10	25	30.3	7.50	2.50
Tails	1.5061	100	11.9	1	1.19	0.079	29.80	25	74.50	4.95	0.06
Acual Head											

			Grades (%)				Recoveries (%)				Unit factors			
Products	Time (Min)	Wt. g	Wt.%	Cu	Fe	S	Bo	Cp	Cu	Fe	S	Bo	Cp	Cu	Fe	S	Bo	Cp
Conc.1	0.5	17.7	1.93	44.4	20.2	28.9	41.5	52.4	26.9	6.4	27.5	25.6	28.7	85.7	38.9	55.8	80.1	101.1
Conc.2	1	34.1	3.72	44.3	19.4	27.3	45.4	44.9	51.7	11.9	49.9	53.9	47.3	164.6	72.2	101.4	168.7	166.9
Conc.3	1.5	11.1	1.21	32.0	20.0	23.4	23.2	50.0	12.2	4.0	13.9	9.0	17.1	38.7	24.2	28.3	28.0	60.4
Conc.4	2	9.3	1.01	15.8	11.2	8.4	19.7	9.6	5.0	1.9	4.2	6.4	2.8	16.0	11.3	8.5	19.9	9.8
Conc.5	2	6.4	0.69	6.2	8.9	3.8	6.4	6.2	1.4	1.0	1.3	1.4	1.2	4.3	6.2	2.6	4.4	4.3
Conc.6	2	4.0	0.44	4.0	7.5	2.5	4.0	4.2	0.6	0.5	0.5	0.6	0.5	1.8	3.3	1.1	1.8	1.9
Tails		835.8	91.00	0.1	4.9	0.1	0.1	0.1	2.3	74.3	2.7	3.1	2.4	7.2	450.1	5.5	9.7	8.5
Calc. Head		918.52	100.0	3.2	6.1	2.0	3.1	3.5	100.0	100.0	100.0	100.0	100.0	318.2	606.2	203.2	312.7	352.9
Actual Head		~920																

Products Cum. & Time (Min)		Wt. g	Wt.%	Cum. Grades (%)					Cum. Rec. (%)					Cum. Unit factors				
	0			Cu	Fe	S	Bo	Cp	Cu	Fe	S	Bo	Cp	Cu	Fe	S	Bo	Cp
Conc.1	0.5	17.74	1.93	44.4	20.2	28.9	41.5	52.4	26.9	6.4	27.5	25.6	28.7	85.7	38.9	55.8	80.1	101.1
Conc.2	1.5	51.87	5.65	44.3	19.7	27.8	44.1	47.5	78.6	18.3	77.4	79.6	76.0	250.3	111.1	157.3	248.8	268.0
Conc.3	3.0	62.97	6.86	42.1	19.7	27.1	40.4	47.9	90.8	22.3	91.3	88.5	93.1	289.0	135.3	185.5	276.9	328.5
Conc.4	5.0	72.27	7.87	38.8	18.6	24.7	37.7	43.0	95.8	24.2	95.5	94.9	95.8	304.9	146.6	194.0	296.8	338.2
Conc.5	7.0	78.64	8.56	36.1	17.8	23.0	35.2	40.0	97.2	25.2	96.8	96.3	97.1	309.2	152.8	196.7	301.2	342.5
Conc.6	9.0	82.68	9.00	34.6	17.3	22.0	33.7	38.3	97.7	25.7	97.3	96.9	97.6	311.0	156.1	197.8	303.0	344.4
Calc. Head		918.52	100.00	3.2	6.1	2.0	3.1	3.5	100.0	100.0	100.0	100.0	100.0	318.2	606.2	203.2	312.7	352.9

Table AB - 3: Test 2 flotation report. Feed with fines; no lime addition; collector used.

TEST 2			DATE: March 17, 2011						
OBJECTIVE:	Flotation selectivity					Feed:	bornite-chalcopyrite ore		
GRINDING CONDITIONS (Denver Mill):					FLOAT CONDITIONS:		FLOATED BY:		
MILL TYPE:	*Pulverizer*	ROD TYPE/Charge :	*As is*		CELL TYPE & VOLUME:		*2 L*	GAS:	*Air*
CHARGE (g):	*920*	65%	solids		IMPELLER RPM:	*1200*	REPULP WATER:	*tap*	
WATER:	*495*				Ro GAS RATE:		by air valve		
GRIND:	12 min				# of STROKES: Ro:	*30/min*			
Lime	0g	ml	STAGE	COND	FLOAT	pH	DOX / pH: as	REDOX	DO
REAGENTS				TIME	TIME	End		mV	
Redox									
						8.06	-28		
KAX		2							
DF250		5							
			Cond.	1					
			Float #1		0.5	8.06	125		
KAX		2							
DF250		5							
			Cond.	1		8.05	82		
			Float #2		1	8.05	180		
KAX		2							
DF250		5							
			Cond.	1		8.04	100		
			Float #3		1.5	8.04	150		
KAX		2							
DF250		5							
			Cond.	4		8.02	47		
			Float #4		2	8.02	150		
KAX		2							
DF250		5							
			Cond.	1		8	72		
			Float #5		2	8	115		
KAX		2							
DF250		5							
			Cond.	1		7.99	51		
			Float #6		2	7.99	82		
Totals									
Lime				9	Minutes				
PEX		12							
DF 250		30							

Table AB - 4: Metallurgical table for Test 2

			Cu				Fe			S
Products	Analytic Wt. (g)	Vol.	ppm	Dilution	mg	%	ppm	Dilution	mg	%	%
Conc 1	0.4155	250	16	50	200.0	48.13	5.70	50	71.3	17.15	26.90
Conc 2	0.4016	250	24.9	25	155.6	38.75	14.20	25	88.8	22.10	28.20
Conc 3	0.4086	250	26.3	17	109.6	26.82	17.40	17	72.5	17.74	19.10
Conc 4	0.4018	250	25.1	8	52.29	13.01	21.50	8	44.8	11.15	7.90
Conc 5	0.4062	250	22.1	5	27.63	6.80	26.50	5	33.1	8.15	3.80
Conc 6	0.4024	100	20.6	8	17.17	4.27	12.90	25	32.3	8.01	2.50
Tails	1.5333	100	17.9	1	1.79	0.117	30.70	25	76.75	5.01	0.08
Acual Head											

			Grades (%)				Recoveries (%)				Unit factors			
Products	Time (Min)	Wt. g	Wt.%	Cu	Fe	S	Bo	Cp	Cu	Fe	S	Bo	Cp	Cu	Fe	S	Bo	Cp
Conc.1	0.5	28.8	3.14	48.1	17.1	26.9	56.5	35.7	47.9	8.8	42.0	58.6	31.7	151.0	53.8	84.4	177.1	112.0
Conc.2	1	24.9	2.71	38.8	22.1	28.2	28.6	59.8	33.4	9.9	38.1	25.7	45.9	105.1	60.0	76.5	77.5	162.3
Conc.3	1.5	10.9	1.19	26.8	17.7	19.1	20.7	39.5	10.1	3.5	11.3	8.1	13.3	31.8	21.1	22.7	24.6	46.9
Conc.4	2	8.0	0.87	13.0	11.1	7.9	13.6	12.7	3.6	1.6	3.4	3.9	3.1	11.3	9.7	6.9	11.8	11.0
Conc.5	2	4.4	0.48	6.8	8.2	3.8	8.0	5.0	1.0	0.6	0.9	1.3	0.7	3.3	3.9	1.8	3.9	2.4
Conc.6	2	4.3	0.46	4.3	8.0	2.5	4.8	3.6	0.6	0.6	0.6	0.7	0.5	2.0	3.7	1.2	2.2	1.7
Tails		837.1	91.14	0.1	5.0	0.1	0.1	0.2	3.4	75.0	3.6	1.6	4.9	10.6	456.2	7.3	5.0	17.2
Calc. Head		918.43	100.0	3.2	6.1	2.0	3.0	3.5	100.0	100.0	100.0	100.0	100.0	315.2	608.4	200.7	302.1	353.5
Actual Head		~920																

Products Cum. & Time (Min)		Wt. g	Wt.%	Cum. Grades (%)					Cum. Rec. (%)					Cum. Unit factors				
	0			Cu	Fe	S	Bo	Cp	Cu	Fe	S	Bo	Cp	Cu	Fe	S	Bo	Cp
Conc.1	0.5	28.81	3.14	48.1	17.1	26.9	56.5	35.7	47.9	8.8	42.0	58.6	31.7	151.0	53.8	84.4	177.1	112.0
Conc.2	1.5	53.73	5.85	43.8	19.4	27.5	43.5	46.9	81.3	18.7	80.2	84.3	77.6	256.1	113.8	160.9	254.6	274.2
Conc.3	3.0	64.63	7.04	40.9	19.2	26.1	39.7	45.6	91.4	22.2	91.4	92.4	90.8	288.0	134.8	183.6	279.2	321.1
Conc.4	5.0	72.62	7.91	37.9	18.3	24.1	36.8	42.0	95.0	23.8	94.9	96.3	94.0	299.3	144.5	190.4	291.0	332.1
Conc.5	7.0	77.06	8.39	36.1	17.7	22.9	35.1	39.9	96.0	24.4	95.8	97.6	94.6	302.6	148.5	192.3	294.9	334.6
Conc.6	9.0	81.33	8.86	34.4	17.2	21.8	33.6	38.0	96.6	25.0	96.4	98.4	95.1	304.6	152.2	193.4	297.1	336.3
Calc. Head		918.43	100.00	3.2	6.1	2.0	3.0	3.5	100.0	100.0	100.0	100.0	100.0	315.2	608.4	200.7	302.1	353.5

Table AB - 5: Test 3 flotation report. Fines present; no lime addition; collector used.

TEST 3				DATE: March 17, 2011				
OBJECTIVE:	Flotation selectivity					Feed:	bornite-chalcopyrite ore	
GRINDING CONDITIONS (Denver Mill):					FLOAT CONDITIONS:		FLOATED BY:	
MILL TYPE:	*Pulverizer*	ROD TYPE/Charge :	*As is*		CELL TYPE & VOLUME:		*2 L*	GAS: *Air*
CHARGE (g):	*919.14*	65%	solids		IMPELLER RPM:	*1200*	REPULP WATER:	*tap*
WATER:	*495*				Ro GAS RATE:		by air valve	
GRIND:	12 min				# of STROKES: Ro:	*30/min*		
Lime	0g	ml	STAGE	COND	FLOAT	pH	DOX / pH: as	REDOX
REAGENTS				TIME	TIME	End		mV
Redox						After 1 min stirring		
						7.58		23
KAX 0.1%		1						
DF 250 0.1%		5						
			Cond	1				
			Conc. 1		0.5	7.59		171
KAX 0.1%		1						
DF 250 0.1%		5						
			Cond	1		7.6		108
			Conc. 2		1	7.61		260
KAX 0.1%		1						
DF 250 0.1%		5						
			Cond	1		7.62		196
			Conc. 3		1.5	7.63		300
KAX 0.1%		1						
DF 250 0.1%		5						
			Cond	1		7.62		239
			Conc. 4		2	7.64		310
KAX 0.1%		1						
DF 250 0.1%		5						
			Cond	1		7.63		259
			Conc. 5		2	7.64		305
KAX 0.1%		1						
DF 250 0.1%		5						
			Cond	1		7.64		249
			Conc. 6		2	7.65		310
Totals								
Lime				9	Minutes			
PEX		6						
DF 250		30						

Conc. 3 was collected in 2 stages:
1st stage was 30 seconds
2nd stage was 1 min

Table AB - 6: Metallurgical table for Test 3.

			Cu				Fe			S
Products	Analytic Wt. (g)	Vol.	ppm	Dilution	mg	%	ppm	Dilution	mg	%	%
Conc 1	0.4056	250	12.9	50	161.3	39.76	7.59	50	94.9	23.39	29.8
Conc 2	0.4093	250	14.2	50	177.5	43.37	6.50	50	81.3	19.85	27.3
Conc 3	0.4085	250	12.75	50	159.4	39.01	5.35	50	66.9	16.37	22.0
Conc 4	0.4057	250	27.2	17	113.33	27.94	14.01	17	58.4	14.39	15.8
Conc 5	0.4101	250	29.6	5	37.00	9.02	30.28	5	37.9	9.23	4.8
Conc 6	0.4060	100	7.18	25	17.95	4.42	12.82	25	32.1	7.89	2.8
Conc.7	0.5015	100	6.24	25	15.60	3.11	14.95	25	37.4	7.45	2.14
Tails	1.5248	100	16.1	1	1.61	0.106	15.40	50	77.00	5.05	0.08
Acual Head											

			Grades (%)				Recoveries (%)				Unit factors			
Products	Time (Min)	Wt. g	Wt.%	Cu	Fe	S	Bo	Cp	Cu	Fe	S	Bo	Cp	Cu	Fe	S	Bo	Cp
Conc.1	0.5	16.7	1.82	39.8	23.4	29.8	27.0	65.5	23.0	6.9	26.8	16.4	33.2	72.2	42.5	54.2	49.1	119.1
Conc.2	1	30.0	3.26	43.4	19.9	27.3	43.0	46.7	45.1	10.5	44.1	46.9	42.4	141.5	64.8	89.1	140.4	152.2
Conc.3	0.5	14.1	1.54	39.0	16.4	22.0	45.3	29.8	19.1	4.1	16.7	23.2	12.8	59.9	25.1	33.8	69.5	45.8
Conc.4	1	7.2	0.78	27.9	14.4	15.8	32.2	21.6	6.9	1.8	6.1	8.4	4.7	21.7	11.2	12.3	25.1	16.8
Conc.5	2	5.7	0.62	9.0	9.2	4.8	11.1	5.7	1.8	0.9	1.5	2.3	1.0	5.6	5.7	3.0	6.9	3.5
Conc.6	2	4.6	0.50	4.4	7.9	2.8	4.3	4.9	0.7	0.6	0.7	0.7	0.7	2.2	4.0	1.4	2.1	2.4
Conc.7	2	3.3	0.36	3.1	7.5	2.1	2.6	4.2	0.4	0.4	0.4	0.3	0.4	1.1	2.7	0.8	0.9	1.5
Tails		836.9	91.13	0.1	5.0	0.1	0.1	0.2	3.1	74.7	3.6	1.7	4.8	9.6	460.2	7.3	5.0	17.2
Calc. Head		918.39	100.0	3.1	6.2	2.0	3.0	3.6	100.0	100.0	100.0	100.0	100.0	313.9	616.1	201.7	299.0	358.6
Actual Head		~920																

			Cum. Grades (%)				Cum. Rec. (%)				Cum. Unit factors			
Products Cum. & Time (Min)		Wt. g	Wt.%	Cu	Fe	S	Bo	Cp	Cu	Fe	S	Bo	Cp	Cu	Fe	S	Bo	Cp
	0																	
Conc.1	0.5	16.69	1.82	39.8	23.4	29.8	27.0	65.5	23.0	6.9	26.8	16.4	33.2	72.2	42.5	54.2	49.1	119.1
Conc.2	1.5	46.65	5.08	42.1	21.1	28.2	37.3	53.4	68.1	17.4	71.0	63.4	75.6	213.7	107.3	143.2	189.5	271.3
Conc.3	2.0	60.75	6.61	41.4	20.0	26.8	39.2	47.9	87.2	21.5	87.7	86.6	88.4	273.6	132.4	177.0	259.0	317.1
Conc.4	3.0	67.90	7.39	40.0	19.4	25.6	38.4	45.2	94.1	23.3	93.8	95.0	93.1	295.4	143.6	189.3	284.1	333.9
Conc.5	5.0	73.59	8.01	37.6	18.6	24.0	36.3	42.1	95.9	24.2	95.3	97.3	94.1	301.0	149.3	192.3	291.0	337.4
Conc.6	7.0	78.19	8.51	35.6	18.0	22.7	34.4	39.9	96.6	24.9	96.0	98.0	94.8	303.2	153.3	193.7	293.1	339.9
Conc.7	9.0	81.48	8.87	34.3	17.6	21.9	33.1	38.5	96.9	25.3	96.4	98.3	95.2	304.3	155.9	194.4	294.1	341.4
Calc. Head		918.39	100.00	3.1	6.2	2.0	3.0	3.6	100.0	100.0	100.0	100.0	100.0	313.9	616.1	201.7	299.0	358.6

Table AB - 7: Test 4 flotation report. Fines present; no lime addition; collectorless first 2 concentrates.

TEST 4			DATE: March 17, 2011					
OBJECTIVE:	Flotation selectivity				Feed:	bornite-chalcopyrite ore		
GRINDING CONDITIONS (Denver Mill):					FLOAT CONDITIONS:		FLOATED BY:	
MILL TYPI *Pulverizer*	ROD TYPE/Charge : *As is*				CELL TYPE & VOLUME:	*2 L*	GAS:	*Air*
CHARGE (g *920*	65%	solids			IMPELLER RPM: *1200*		REPULP W *tap*	
WATER: *495*					Ro GAS RATE:		by air valve	
GRIND: 12 min					# of STROKES: Ro: *30/min*			
Lime 0g	ml	STAGE	COND	FLOAT	pH	DOX / pH: as	REDOX	DO
REAGENTS			TIME	TIME	End		mV	
Redox					After 1 min stirring			
					7.76		130	
DF 250	5							
		Cond.	1		7.74		105	
		Conc. 1		0.5	7.74		237	
DF 250	5							
		Cond.	1					
		Conc. 2		1	7.72		300	
KAX	1							
DF 250	5							
		Cond.	1		7.72		220	
		Conc. 3		0.5	7.72		320	
KAX	1							
DF 250	5							
		Cond.	1		7.72		250	
		Conc. 4		1	7.71		300	
KAX	1							
DF 250	5							
		Cond.	1		7.71		243	
		Conc. 5		1.5	7.69		350	
KAX	1							
DF 250	5							
		Cond.	1		7.69		275	
		Conc. 6		2	7.67		350	
KAX	2							
DF250	2.5							
		Cond.	1		7.69		241	
		Conc. 7		2.5	7.68		350	
Totals								
Lime			9	Minutes				
PEX	6							
DF 250	33							

Table AB - 8: Metallurgical table for Test 4

			Cu				Fe			S
Products	Anlytic Wt. (g)	Vol.	ppm	Dilution	mg	%	ppm	Dilution	mg	%	%
Conc 1	0.4111	250	11.5	50	143.8	34.97	8.80	50	110.0	26.76	31.4
Conc 2	0.4064	250	12.3	50	153.8	37.83	7.94	50	99.3	24.42	30.1
Conc 3	0.4184	250	14.5	50	181.3	43.32	6.35	50	79.4	18.97	26.2
Conc 4	0.4065	250	14.2	50	177.50	43.67	4.75	50	59.4	14.61	22.0
Conc 5	0.4243	250	12.7	25	79.38	18.71	7.42	25	46.4	10.93	9.7
Conc 6	0.4034	100	12.5	25	31.25	7.75	13.60	25	34.0	8.43	5.5
Conc.7	0.5129	100	10.3	25	25.75	5.02	16.23	25	40.6	7.91	3.14
Tails	1.5054	100	15.15	1	1.52	0.101	14.80	50	74.00	4.92	0.08
Acual Head											

			Grades (%)				Recoveries (%)				Unit factors		
Products	Time (Min)	Wt. g	Wt.%	Cu	Fe	S	Bo	Cp	Cu	Fe	S	Bo	Cp	Cu	Fe	S	Bo	Cp
Conc.1	0.5	9.4	1.03	35.0	26.8	31.4	10.1	82.6	11.5	4.5	15.8	3.6	22.9	35.9	27.5	32.3	10.4	84.8
Conc.2	1	18.6	2.02	37.8	24.4	30.1	21.0	70.8	24.5	8.2	29.9	14.6	38.7	76.5	49.4	60.9	42.4	143.2
Conc.3	0.5	12.2	1.33	43.3	19.0	26.2	45.5	41.8	18.4	4.2	17.1	20.8	15.0	57.6	25.2	34.9	60.5	55.5
Conc.4	1	23.0	2.50	43.7	14.6	22.0	57.7	20.7	34.9	6.0	27.0	49.6	14.0	109.1	36.5	55.0	144.1	51.8
Conc.5	1.5	9.1	0.99	18.7	10.9	9.7	23.9	10.3	5.9	1.8	4.7	8.1	2.7	18.5	10.8	9.6	23.6	10.1
Conc.6	2	4.5	0.49	7.7	8.4	5.5	5.8	11.6	1.2	0.7	1.3	1.0	1.5	3.8	4.1	2.7	2.8	5.6
Conc.7	2.5	3.5	0.38	5.0	7.9	3.1	5.0	5.4	0.6	0.5	0.6	0.7	0.6	1.9	3.0	1.2	1.9	2.0
Tails		838.5	91.27	0.1	4.9	0.1	0.1	0.2	2.9	74.1	3.6	1.7	4.7	9.2	448.6	7.3	5.0	17.3
Calc. Head		918.75	100.0	3.1	6.1	2.0	2.9	3.7	100.0	100.0	100.0	100.0	100.0	312.4	605.1	203.7	290.8	370.4
Actual Head		~920																

Products Cum. & Time (Min)		Wt. g	Wt.%	Cum. Grades (%)					Cum. Rec. (%)					Cum. Unit factors				
	0			Cu	Fe	S	Bo	Cp	Cu	Fe	S	Bo	Cp	Cu	Fe	S	Bo	Cp
Conc.1	0.5	9.43	1.03	35.0	26.8	31.4	10.1	82.6	11.5	4.5	15.8	3.6	22.9	35.9	27.5	32.3	10.4	84.8
Conc.2	1.5	28.01	3.05	36.9	25.2	30.6	17.3	74.8	36.0	12.7	45.7	18.2	61.5	112.4	76.9	93.1	52.8	227.9
Conc.3	2.0	40.22	4.38	38.8	23.3	29.2	25.9	64.8	54.4	16.9	62.8	39.0	76.5	170.0	102.1	128.0	113.3	283.5
Conc.4	3.0	63.18	6.88	40.6	20.2	26.6	37.4	48.8	89.3	22.9	89.8	88.5	90.5	279.1	138.6	183.0	257.5	335.3
Conc.5	4.5	72.26	7.87	37.8	19.0	24.5	35.7	43.9	95.2	24.7	94.5	96.7	93.3	297.6	149.4	192.6	281.1	345.5
Conc.6	6.5	76.73	8.35	36.1	18.4	23.4	34.0	42.0	96.4	25.4	95.8	97.6	94.8	301.3	153.5	195.2	283.9	351.1
Conc.7	9.0	80.23	8.73	34.7	17.9	22.5	32.7	40.4	97.1	25.9	96.4	98.3	95.3	303.3	156.5	196.4	285.8	353.1
Calc. Head		918.75	100.00	3.1	6.1	2.0	2.9	3.7	100.0	100.0	100.0	100.0	100.0	312.4	605.1	203.7	290.8	370.4

Table AB - 9: Test 5 flotation report. Fines removed; no lime addition; collector used

TEST 5			DATE: March 17, 2011					
OBJECTIVE:	Flotation selectivity				Feed:	bornite-chalcopyrite ore		
GRINDING CONDITIONS (Denver Mill):					FLOAT CONDITIONS:		FLOATED BY:	
MILL TYPE	*Pulverizer*	ROD TYPE/Charge :	*As is*		CELL TYPE & VOLUME:	*2 L*	GAS:	*Air*
CHARGE (g	*920*	65%	solids		IMPELLER RPM:	*1200*	REPULP WA	*tap*
WATER:	*495*				Ro GAS RATE:		by air valve	
GRIND:	12 min				# of STROKES: Ro:	*30/min*		
Lime	0g	ml	STAGE	COND	FLOAT	pH	DOX / pH: as	REDOX	DO
REAGENTS				TIME	TIME	End		mV	
Redox						After 1 min stirring			
						7.85		29	
KAX (0.1%)		2.5	Cond.	2					
DF 250 (0.1%)		4		0.5		7.81		-13	
						7.81		96	
			Conc. 1		0.3				
KAX (0.1%)		2.5	Cond.	1					
DF 250 (0.1%)		4		0.5		7.82		126	
			Conc. 2		0.2				
KAX (0.1%)		2.5	Cond.	1					
DF 250 (0.1%)		4		0.5					
			Conc. 3		1				
KAX (0.1%)		2.5	Cond.	1					
DF 250 (0.1%)		4		0.5		7.85		140	
			Conc. 4		1.5	7.79		163	
KAX (0.1%)		2.5	Cond.	1					
DF 250 (0.1%)		4		0.5					
			Conc. 5		2.5	7.75		160	
KAX (0.1%)		2.5	Cond.	1					
DF 250 (0.1%)		4		0.5		7.72		110	
			Conc. 6		3.5	7.73		170	
Totals									
Lime				9.1	Minutes				
PEX		15							
DF 250		24							

Table AB - 10: Metallurgical table for Test 5.

			Cu				Fe			S
Products	Anlytic Wt. (g)	Vol.	ppm	Dilution	mg	%	ppm	Dilution	mg	%	%	%
Conc 1	0.4105	250	18.54	50	231.8	56.46	5.14	50	64.2	15.65		26.2
Conc 2	0.4017	250	15.05	50	188.1	46.83	6.12	50	76.5	19.04		27.9
Conc 3	0.4048	250	12.8	50	160.0	39.53	6.96	50	87.0	21.48		27.7
Conc 4	0.4298	250	8.927	50	111.59	25.96	5.20	25	32.5	7.56		16.0
Conc 5	0.4104	250	7.574	25	47.34	11.53	6.84	25	42.8	10.42		6.6
Conc 6	0.4116	100	10.93	25	27.33	6.64	14.54	25	36.4	8.83		3.7
-400F	1.5022	100	9.837	50	49.19	3.27	23.60	5	11.8	0.79		2.3967
Tails	1.5261	100	14.53	1	1.45	0.095	31.43	50	157.15	10.30		0.19129
Acual Head												

				Grades (%)				Recoveries (%)					Unit factors			
Products	Time (Min)	Wt. g	Wt.%	Cu	Fe	S	Bo	Cp	Cu	Fe	S	Bo	Cp	Cu	Fe	S	Bo	Cp
Conc.1	0.3	11.8	1.31	56.5	15.6	26.2	80.5	16.0	22.2	1.9	15.9	31.2	6.2	74.1	20.5	34.3	105.6	21.0
Conc.2	0.2	13.4	1.49	46.8	19.0	27.9	50.4	43.0	20.9	2.6	19.2	22.2	18.9	69.8	28.4	41.6	75.1	64.1
Conc.3	1	30.9	3.44	39.5	21.5	27.7	31.7	56.1	40.6	6.8	44.0	32.2	56.7	135.8	73.8	95.2	109.0	192.6
Conc.4	1.5	11.7	1.30	26.0	7.6	16.0	26.7	26.2	10.1	0.9	9.6	10.3	10.0	33.7	9.8	20.8	34.7	34.1
Conc.5	2.5	6.9	0.77	11.5	10.4	6.6	12.9	9.6	2.7	0.7	2.4	2.9	2.2	8.9	8.0	5.1	10.0	7.4
Conc.6	3.4	4.6	0.51	6.6	8.8	3.7	7.8	4.8	1.0	0.4	0.9	1.2	0.7	3.4	4.5	1.9	4.0	2.5
Tails		819.0	91.18	0.1	10.3	0.2	0.0	0.2	2.6	86.6	8.1	0.0	5.4	8.7	938.9	17.4	0.0	18.2
-400F		22.2		3.3	0.8	2.4	2.4	5.1										
Calc. Head		898.28	100.0	3.3	10.8	2.2	3.4	3.4	100.0	100.0	100.0	100.0	100.0	334.4	1084.0	216.3	338.5	339.9
Actual Head		~900	920.52															

Products Cum. & Time (Min)		Wt. g	Wt.%	Cum. Grades (%)					Cum. Rec. (%)					Cum. Unit factors				
	0			Cu	Fe	S	Bo	Cp	Cu	Fe	S	Bo	Cp	Cu	Fe	S	Bo	Cp
Conc.1	0.3	11.79	1.31	56.5	15.6	26.2	80.5	16.0	22.2	1.9	15.9	31.2	6.2	74.1	20.5	34.3	105.6	21.0
Conc.2	0.6	25.18	2.80	51.3	17.4	27.1	64.5	30.4	43.0	4.5	35.1	53.4	25.0	143.9	48.9	75.9	180.8	85.1
Conc.3	1.6	56.04	6.24	44.8	19.7	27.4	46.4	44.5	83.6	11.3	79.1	85.6	81.7	279.7	122.7	171.1	289.8	277.7
Conc.4	3.1	67.71	7.54	41.6	17.6	25.5	43.0	41.4	93.7	12.2	88.7	95.9	91.7	313.4	132.5	191.9	324.5	311.8
Conc.5	5.6	74.64	8.31	38.8	16.9	23.7	40.3	38.4	96.4	13.0	91.1	98.8	93.9	322.3	140.6	197.0	334.5	319.1
Conc.6	9.0	79.26	8.82	36.9	16.4	22.5	38.4	36.5	97.4	13.4	91.9	100.0	94.6	325.7	145.1	198.9	338.5	321.6
Calc. Head		898.28	100.00	3.3	10.8	2.2	3.4	3.4	100.0	100.0	100.0	100.0	100.0	334.4	1084.0	216.3	338.5	339.9

Table AB - 11: Test 6 flotation report. Fines present; pH 9; collectorless first 3 concentrates.

TEST 6			DATE: March 17, 2011							
OBJECTIVE:	Flotation selectivity						Feed:	bornite-chalcopyrite ore		
GRINDING CONDITIONS (Denver Mill):					FLOAT CONDITIONS:			FLOATED BY:		
MILL TYPI *Pulverizer*	ROD TYPE/Charge :	*As is*			CELL TYPE & VOLUME:		*2 L*	GAS:	*Air*	
CHARGE (g *920*	65%	solids			IMPELLER RPM:	*1200*	REPULP W/	*tap*		
WATER: *495*					Ro GAS RATE:		by air valve			
GRIND: 12 min					# of STROKES: Ro:	*30/min*				
Lime 0.4 g	ml	g	STAGE	COND	FLOAT	pH	pH	DOX / pH: as	REDOX	DO
REAGENTS				TIME	TIME	Start	End		mV	
Redox							After 1 min stirring			
						9.23	9.16	38		
			Aeration		3		9.12	36		
DF 250 (0.1%)	5									
			Conc.1		0.5		8.99	78		
Lime		0.03					9.09	55		
DF 250 (0.1%)	5									
			Conc. 2		1		8.97	145		
Lime										
DF 250 (0.1%)	5						9.04	125		
			Conc. 3		1		8.93	200		
Lime		0.01								
KAX (0.1%)	1									
DF 250 (0.1%)	5						9.09	153		
			Conc. 4a		0.2					
			Conc. 4b		0.3		8.98	204		
KAX (0.1%)	1									
DF 250 (0.1%)	5									
			Conc. 5		1		9.06			
KAX (0.1%)	2.5									
DF 250 (0.1%)	5						9.02	132		
			Conc. 6		2.5		8.77			
Lime		0.01								
KAX (0.1%)	3									
DF 250 (0.1%)	5						8.97	161		
			Conc. 7		2.5		8.77	270		
Totals										
Lime		0.05		12	Minutes					
PEX	8									
DF 250	35									

Table AB - 12: Metallurgical table for Test 6.

			Cu				Fe		S
Products	Analytic Wt.(g)	Vol.	ppm	Dilution	mg	%	ppm	Dilution	mg	%	%	%
Conc 1	0.419	250	12.02	50	150.3	35.86	9.37	50	117.2	27.96		32.1
Conc 2	0.401	250	12.46	50	155.8	38.84	7.25	50	90.6	22.59		29.9
Conc 3	0.4172	250	14.32	50	179.0	42.91	6.22	50	77.8	18.65		26.4
Conc 4	0.4032	250	16.24	50	203.00	50.35	4.71	50	58.9	14.61		24.0
Conc 5	0.4119	250	30.17	25	188.56	45.78	8.61	25	53.8	13.06		20.4
Conc 6	0.407	100	19.48	50	97.40	23.93	8.75	50	43.7	10.74		11.5
Conc.7	0.4036	100	8.68	25	21.69	5.37	12.45	25	31.1	7.71		2.9819
Tails	1.5306	100	13.90	1	1.39	0.091	31.85	25	79.63	5.20		0.14704
Acual Head												

			Grades (%)				Recoveries (%)				Unit factors			
Products	Time (Min)	Wt. g	Wt.%	Cu	Fe	S	Bo	Cp	Cu	Fe	S	Bo	Cp	Cu	Fe	S	Bo	Cp
Conc.1	0.5	11.5	1.25	35.9	28.0	32.1	10.7	84.2	13.5	5.6	19.1	4.0	31.2	44.9	35.0	40.3	13.4	105.4
Conc.2	1	16.3	1.77	38.8	22.6	29.9	24.2	67.8	20.8	6.4	25.1	12.8	35.5	68.9	40.1	53.0	42.9	120.3
Conc.3	1	10.4	1.13	42.9	18.6	26.4	43.9	43.6	14.6	3.4	14.1	14.8	14.5	48.4	21.0	29.8	49.6	49.1
Conc.4	0.5	10.3	1.12	50.3	14.6	24.0	69.7	17.9	17.0	2.6	12.7	23.2	5.9	56.3	16.3	26.9	78.0	20.0
Conc.5	1	16.6	1.80	45.8	13.1	20.4	67.4	9.0	24.9	3.8	17.4	36.2	4.8	82.6	23.6	36.7	121.6	16.2
Conc.6	2.5	7.9	0.85	23.9	10.7	11.5	32.9	9.0	6.2	1.5	4.7	8.3	2.3	20.4	9.2	9.8	28.0	7.6
Conc.7	2.5	3.4	0.37	5.4	7.7	3.0	6.4	3.8	0.6	0.5	0.5	0.7	0.4	2.0	2.9	1.1	2.4	1.4
Tails		843.9	91.70	0.1	5.2	0.1	0.0	0.2	2.5	76.3	6.4	0.0	5.4	8.3	477.0	13.5	0.0	18.3
Calc. Head		920.29	100.0	3.3	6.3	2.1	3.4	3.4	100.0	100.0	100.0	100.0	100.0	331.9	625.1	211.2	335.9	338.4
Actual Head		~920																

Products Cum. & Time (Min)		Wt. g	Wt.%	Cum. Grades (%)					Cum. Rec. (%)					Cum. Unit factors				
	0			Cu	Fe	S	Bo	Cp	Cu	Fe	S	Bo	Cp	Cu	Fe	S	Bo	Cp
Conc.1	0.5	11.53	1.25	35.9	28.0	32.1	10.7	84.2	13.5	5.6	19.1	4.0	31.2	44.9	35.0	40.3	13.4	105.4
Conc.2	1.5	27.85	3.03	37.6	24.8	30.8	18.6	74.6	34.3	12.0	44.2	16.8	66.7	113.8	75.1	93.2	56.3	225.7
Conc.3	2.5	38.23	4.15	39.0	23.1	29.6	25.5	66.2	48.9	15.4	58.3	31.5	81.2	162.2	96.1	123.1	105.8	274.8
Conc.4	3.0	48.53	5.27	41.4	21.3	28.4	34.9	55.9	65.9	18.0	71.0	54.7	87.1	218.5	112.5	150.0	183.8	294.8
Conc.5	4.0	65.13	7.08	42.5	19.2	26.4	43.2	43.9	90.7	21.8	88.4	90.9	91.9	301.1	136.0	186.7	305.4	311.0
Conc.6	6.5	72.98	7.93	40.5	18.3	24.8	42.1	40.2	96.9	23.2	93.1	99.3	94.2	321.5	145.2	196.6	333.5	318.6
Conc.7	9.0	76.40	8.30	39.0	17.8	23.8	40.5	38.6	97.5	23.7	93.6	100.0	94.6	323.5	148.1	197.7	335.9	320.1
Calc. Head		920.29	100.00	3.3	6.3	2.1	3.4	3.4	100.0	100.0	100.0	100.0	100.0	331.9	625.1	211.2	335.9	338.4

Table AB - 13: Test 7 flotation report. Fines present; pH 10; collectorless first 3 concentrates.

TEST 7			DATE: March 17, 2011							
OBJECTIVE:	Flotation selectivity					Feed:	bornite-chalcopyrite ore			
GRINDING CONDITIONS (Denver Mill):					FLOAT CONDITIONS:		FLOATED BY:			
MILL TYPI *Pulverizer*	ROD TYPE/Charge :	*As is*			CELL TYPE & VOLUME:		*2 L*	GAS:	*Air*	
CHARGE (g *920*	65%	solids			IMPELLER RPM:	*1200*	REPULP W/	*tap*		
WATER: *495*					Ro GAS RATE:		by air valve			
GRIND: 12 min					# of STROKES: Ro:	*30/min*				
Lime 0.8 g	ml	g	STAGE	COND	FLOAT	pH	pH	DOX / pH: as	REDOX	DO
REAGENTS		0.0005106		TIME	TIME	Start	End		mV	
Redox							After 1 min stirring			
							10.17	47		
DF250 (0.1%)	5									
			Conc. 1		0.5		10.11	52		
DF250 (0.1%)	5									
			Conc. 2		1		10.08	74		
DF250 (0.1%)	5									
							10.07	65		
			Conc. 3		1		10.04	80		
KAX (0.1%)	1									
DF250 (0.1%)	5									
							10.02	51		
			Conc. 4		0.5		9.99	80		
KAX (0.1%)	1									
DF250 (0.1%)	5									
							9.99	69		
			Conc. 5		1		9.96	90		
Lime		0.01								
KAX (0.1%)	2.5									
DF250 (0.1%)	5									
							9.99	63		
			Conc. 6		2.5		9.93	115		
Lime		0.03								
KAX (0.1%)	3									
DF250 (0.1%)	5									
							9.98	69		
			Conc. 7		2.5		9.91	110		
Totals										
Lime		0.04		9	Minutes					
PEX	8									
DF 250	35									

Table AB - 14: Metallurgical table for Test 7.

			Cu				Fe			S
Products	Anlytic Wt.(g)	Vol.	ppm	Dilution	mg	%	ppm	Dilution	mg	%	%
Conc 1	0.4041	250	12.26	50	153.3	37.92	8.63	50	107.8	26.68	32.0
Conc 2	0.4072	250	13.20	50	165.0	40.52	7.35	50	91.9	22.57	29.6
Conc 3	0.4045	250	13.80	50	172.5	42.65	6.38	50	79.8	19.73	27.3
Conc 4	0.4201	250	16.95	50	211.88	50.43	4.97	50	62.2	14.80	24.6
Conc 5	0.4119	250	28.59	25	178.69	43.38	9.11	25	57.0	13.83	19.8
Conc 6	0.4027	100	22.59	50	112.95	28.05	9.87	50	49.3	12.25	13.2
Conc. 7	0.402	100	12.19	25	30.48	7.58	13.44	25	33.6	8.36	4.6848
Tails	1.556	100	12.84	1	1.28	0.083	32.45	25	81.13	5.21	0.15143
Acual Head											

			Grades (%)					Recoveries (%)					Unit factors				
Products	Time (Min)	Wt. g	Wt.%	Cu	Fe	S	Bo	Cp	Cu	Fe	S	Bo	Cp	Cu	Fe	S	Bo	Cp
Conc.1	0.5	11.9	1.29	37.9	26.7	32.0	16.3	79.6	14.8	5.5	19.5	6.3	30.2	48.9	34.4	41.3	21.0	102.7
Conc.2	1	16.1	1.75	40.5	22.6	29.6	29.4	63.2	21.4	6.3	24.5	15.4	32.5	70.9	39.5	51.8	51.4	110.6
Conc.3	1	11.1	1.21	42.6	19.7	27.3	41.0	48.0	15.6	3.8	15.5	14.8	17.0	51.5	23.8	32.9	49.5	58.0
Conc.4	0.5	12.4	1.34	50.4	14.8	24.6	68.6	20.2	20.5	3.2	15.6	27.6	7.9	67.7	19.9	33.0	92.0	27.0
Conc.5	1	14.8	1.61	43.4	13.8	19.8	62.7	10.7	21.1	3.5	15.0	30.2	5.0	69.8	22.2	31.8	100.9	17.1
Conc.6	2.5	3.8	0.42	28.0	12.3	13.2	39.4	8.8	3.5	0.8	2.6	4.9	1.1	11.6	5.1	5.5	16.3	3.7
Conc.7	2.5	3.3	0.35	7.6	8.4	4.7	7.8	7.7	0.8	0.5	0.8	0.8	0.8	2.7	3.0	1.7	2.8	2.7
Tails		847.1	92.03	0.1	5.2	0.2	0.0	0.2	2.3	76.4	6.6	0.0	5.4	7.6	479.8	13.9	0.0	18.4
Calc. Head		920.37	100.0	3.3	6.3	2.1	3.3	3.4	100.0	100.0	100.0	100.0	100.0	330.6	627.7	211.8	333.9	340.3
Actual Head		~920																

Products Cum. & Time (Min)	Wt. g	Wt.%	Cum. Grades (%)					Cum. Rec. (%)					Cum. Unit factors				
0			Cu	Fe	S	Bo	Cp	Cu	Fe	S	Bo	Cp	Cu	Fe	S	Bo	Cp
Conc.1	11.87	1.29	37.9	26.7	32.0	16.3	79.6	14.8	5.5	19.5	6.3	30.2	48.9	34.4	41.3	21.0	102.7
0.5	27.97	3.04	39.4	24.3	30.6	23.8	70.2	36.2	11.8	43.9	21.7	62.7	119.8	73.9	93.1	72.4	213.3
Conc.2	39.08	4.25	40.3	23.0	29.7	28.7	63.9	51.8	15.6	59.5	36.5	79.7	171.3	97.7	126.0	121.9	271.3
1.5	51.43	5.59	42.8	21.0	28.4	38.3	53.4	72.3	18.7	75.1	64.1	87.7	238.9	117.6	158.9	214.0	298.3
Conc.3	66.23	7.20	42.9	19.4	26.5	43.7	43.8	93.4	22.3	90.1	94.3	92.7	308.7	139.8	190.7	314.8	315.5
2.5	70.05	7.61	42.1	19.0	25.8	43.5	41.9	96.9	23.1	92.6	99.2	93.8	320.3	144.9	196.2	331.2	319.2
Conc.4	73.31	7.97	40.6	18.6	24.8	41.9	40.4	97.7	23.6	93.4	100.0	94.6	323.0	147.8	197.8	333.9	321.9
3.0																	
Conc.5																	
4.0																	
Conc.6																	
6.5																	
Conc.7																	
9.0																	
Calc. Head	920.37	100.00	3.3	6.3	2.1	3.3	3.4	100.0	100.0	100.0	100.0	100.0	330.6	627.7	211.8	333.9	340.3

Table AB - 15: Test 8 flotation report. Fines present; pH 11; collectorless first 3 concentrates.

TEST 8					DATE: March 17, 2011					
OBJECTIVE:	Flotation selectivity						Feed:	bornite-chalcopyrite ore		
GRINDING CONDITIONS (Denver Mill):					FLOAT CONDITIONS:			FLOATED BY:		
MILL TYPE *Pulverizer*	ROD TYPE/Charge :	*As is*			CELL TYPE & VOLUME:			*2 L*	GAS:	*Air*
CHARGE (*920*	65%	solids			IMPELLER RPM:	*1200*		REPULP W *tap*		
WATER: *495*					Ro GAS RATE:			by air valve		
GRIND: 10 min					# of STROKES: Ro:	*30/min*				
Lime 0.10 g	ml	g	STAGE	COND	FLOAT	pH	pH	DOX / pH: as	REDOX	DO
REAGENTS		0.000511		TIME	TIME	Start	End		mV	
Redox							After 1 min stirring			

Table AB - 16: Metallurgical table for Test 8.

			Cu				Fe			S
Products	Anlytic Wt. (g)	Vol.	ppm	Dilution	mg	%	ppm	Dilution	mg	%	%	%
Conc 1	0.4043	250	12.05	50	150.6	37.26	8.47	50	105.9	26.19		31.5
Conc 2	0.4049	250	13.44	50	168.0	41.49	7.28	50	91.0	22.46		29.2
Conc 3	0.4136	250	14.50	50	181.3	43.82	6.408	50	80.1	19.37		27.0
Conc 4	0.4017	250	16.28	50	203.50	50.66	5.071	50	63.4	15.78		25.5
Conc 5	0.4010	250	26.49	25	165.56	41.29	10.41	25	65.1	16.23		21.9
Conc 6	0.4061	100	25.08	25	62.70	15.44	17.79	25	44.5	10.95		8.1
Conc 7	0.403	100	7.834	25	19.59	4.86	12.99	25	32.5	8.06		3.111
Tails	1.5983	100	13.41	1	1.34	0.084	32.75	25	81.88	5.12		0.13894
Actual Head												

			Grades (%)				Recoveries (%)				Unit factors			
Products	Time (Min)	Wt. g	Wt.%	Cu	Fe	S	Bo	Cp	Cu	Fe	S	Bo	Cp	Cu	Fe	S	Bo	Cp
Conc.1	0.5	10.8	1.17	37.3	26.2	31.5	16.0	78.4	13.1	4.9	17.4	5.5	27.2	43.6	30.7	36.9	18.8	91.8
Conc.2	1	13.4	1.45	41.5	22.5	29.2	33.0	59.5	18.0	5.2	20.0	14.0	25.5	60.1	32.5	42.3	47.8	86.1
Conc.3	1	8.8	0.95	43.8	19.4	27.0	45.0	44.3	12.5	3.0	12.1	12.5	12.5	41.7	18.4	25.6	42.8	42.1
Conc.4	0.5	14.9	1.61	50.7	15.8	25.5	66.9	24.1	24.4	4.1	19.4	31.5	11.5	81.5	25.4	41.0	107.6	38.7
Conc.5	1	19.9	2.15	41.3	16.2	21.9	51.7	24.8	26.6	5.6	22.3	32.6	15.8	89.0	35.0	47.1	111.3	53.5
Conc.6	2.5	5.6	0.61	15.4	11.0	8.1	19.4	9.0	2.8	1.1	2.3	3.4	1.6	9.4	6.6	4.9	11.8	5.5
Conc.7	2.5	2.4	0.26	4.9	8.1	3.1	4.8	5.4	0.4	0.3	0.4	0.4	0.4	1.3	2.1	0.8	1.2	1.4
Tails		847.4	91.80	0.1	5.1	0.1	0.0	0.2	2.3	75.7	6.0	0.0	5.4	7.7	470.3	12.8	0.0	18.4
Calc. Head		923.08	100.0	3.3	6.2	2.1	3.4	3.4	100.0	100.0	100.0	100.0	100.0	334.2	621.0	211.5	341.3	337.5
Actual Head		~920																

Products Cum. & Time (Min)		Wt. g	Wt.%	Cum. Grades (%)					Cum. Rec. (%)					Cum. Unit factors				
	0			Cu	Fe	S	Bo	Cp	Cu	Fe	S	Bo	Cp	Cu	Fe	S	Bo	Cp
Conc.1	0.5	10.81	1.17	37.3	26.2	31.5	16.0	78.4	13.1	4.9	17.4	5.5	27.2	43.6	30.7	36.9	18.8	91.8
Conc.2	1.5	24.18	2.62	39.6	24.1	30.2	25.4	67.9	31.0	10.2	37.4	19.5	52.7	103.7	63.2	79.2	66.6	178.0
Conc.3	2.5	32.96	3.57	40.7	22.9	29.4	30.6	61.6	43.5	13.1	49.6	32.0	65.2	145.4	81.6	104.8	109.4	220.1
Conc.4	3.0	47.81	5.18	43.8	20.7	28.2	41.9	50.0	67.9	17.2	69.0	63.6	76.7	226.9	107.0	145.9	217.0	258.8
Conc.5	4.0	67.70	7.33	43.1	19.4	26.3	44.8	42.6	94.5	22.9	91.3	96.2	92.5	315.9	142.0	193.0	328.3	312.3
Conc.6	6.5	73.30	7.94	41.0	18.7	24.9	42.8	40.0	97.3	23.9	93.6	99.6	94.1	325.2	148.6	197.9	340.1	317.8
Conc.7	9.0	75.69	8.20	39.8	18.4	24.2	41.6	38.9	97.7	24.3	94.0	100.0	94.6	326.5	150.7	198.8	341.3	319.2
Calc. Head		923.08	100.00	3.3	6.2	2.1	3.4	3.4	100.0	100.0	100.0	100.0	100.0	334.2	621.0	211.5	341.3	337.5

Table AB - 17: Test 9 flotation report. Fines present; no lime addition; collectorless first 3 concentrates.

TEST 9			DATE: March 17, 2011							
OBJECTIVE:	Flotation selectivity						Feed:	bornite-chalcopyrite ore		
GRINDING CONDITIONS (Denver Mill):					FLOAT CONDITIONS:		FLOATED BY:			
MILL TYPI *Pulverizer*	ROD TYPE/Charge : *As is*				CELL TYPE & VOLUME:		*2 L*	GAS:	*Air*	
CHARGE (g *920*	65%	solids			IMPELLER RPM:	*1200*	REPULP W/	*tap*		
WATER: *495*					Ro GAS RATE:		by air valve			
GRIND: 12 min					# of STROKES: Ro:	*30/min*				
Lime 0g	ml	g	STAGE	COND	FLOAT	pH	pH	ORP / pH: as	REDOX	DO
REAGENTS		0.0005106		TIME	TIME	Start	End		mV	
Redox							After 1 min stirring			
							7.67			
			Aeration	3			7.69			
DF 250	5									
			Cond.	1						
			Conc. 1		0.5		7.66			
DF 250	5									
			Cond.	1			7.64			
			Conc. 2		1		7.64			
KAX	1									
DF 250	5									
			Cond.	1			7.63			
			Conc. 3		1		7.64			
KAX	1									
DF 250	5									
			Cond.	1			7.62			
			Conc. 4		0.5		7.61			
KAX	1									
DF 250	5									
			Cond.	1			7.61			
			Conc. 5		1		7.61			
KAX	2.5									
DF 250	5									
			Cond.	1			7.61			
			Conc. 6		2.5		7.62			
KAX	3									
DF250	5									
			Cond.	1			7.62			
			Conc. 7		2.5		7.62			
Totals										
Lime				9	Minutes					
PEX	9									
DF 250	35									

Table AB - 18: Metallurgical table for Test 9.

			Cu				Fe			S
Products	Analytic Wt. (g)	Vol.	ppm	Dilution	mg	%	ppm	Dilution	mg	%	%	%
Conc 1	0.4075	250	12.06	50	150.8	36.99	8.56	50	107.1	26.27		31.9
Conc 2	0.4064	250	13.31	50	166.4	40.94	7.45	50	93.2	22.92		29.6
Conc 3	0.4032	250	14.34	50	179.3	44.46	6.335	50	79.2	19.64		26.8
Conc 4	0.4047	250	16.25	50	203.13	50.19	4.679	50	58.5	14.45		23.3
Conc 5	0.4021	250	24.38	25	152.38	37.89	8.4	25	52.5	13.06		17.6
Conc 6	0.4080	100	26.15	25	65.38	16.02	16.50	25	41.3	10.11		8.0
Conc 7	0.4172	100	25.41	25	63.53	15.23	17.85	25	44.6	10.70	7.9705	
Tails	1.5516	100	16.13	1	1.61	0.104	30.8	25	77.00	4.96	0.31761	
Actual Head												

			Grades (%)				Recoveries (%)				Unit factors			
Products	Time (Min)	Wt. g	Cu	Fe	S	Bo	Cp	Cu	Fe	S	Bo	Cp	Cu	Fe	S	Bo	Cp
Conc.1	0.5	11.33	37.0	26.3	31.9	14.2	80.9	13.6	5.3	17.3	5.1	29.4	45.6	32.4	39.4	17.5	99.7
Conc.2	1.0	18.27	40.9	22.9	29.6	30.4	62.4	24.2	7.5	25.8	17.8	36.6	81.3	45.6	58.8	60.5	124.1
Conc.3	1.0	11.10	44.5	19.6	26.8	47.2	42.1	16.0	3.9	14.2	16.8	15.0	53.7	23.7	32.4	57.0	50.9
Conc.4	0.5	14.12	50.2	14.5	23.3	71.4	14.5	23.0	3.7	15.7	32.3	6.6	77.1	22.2	35.7	109.6	22.2
Conc.5	1.0	12.01	37.9	13.1	17.6	53.9	11.0	14.7	2.8	10.1	20.7	4.2	49.5	17.1	23.0	70.4	14.3
Conc.6	2.5	7.94	16.0	10.1	8.0	21.3	7.3	4.1	1.4	3.0	5.4	1.9	13.8	8.7	6.9	18.4	6.3
Conc.7	2.5	3.05	15.2	10.7	8.0	19.1	8.9	1.5	0.6	1.2	1.9	0.9	5.1	3.5	2.6	6.4	2.9
Tails		841.6	0.1	5.0	0.3	0.0	0.2	2.8	74.8	12.8	0.0	5.4	9.5	454.3	29.1	0.0	18.3
Calc. Head		919.46	3.4	6.1	2.3	3.4	3.4	100.0	100.0	100.0	100.0	100.0	335.6	607.4	227.8	339.7	338.8
Actual Head		~920															

Products Cum. & Time (Min)		Wt. g	Wt.%	Cum. Grades (%)						Cum. Rec. (%)						Cum. Unit factors					
	0			Cu	Fe	S	Bo	Cp	Cu	Fe	S	Bo	Cp	Cu	Fe	S	Bo	Cp
Conc.1	0.5	11.33	1.23	37.0	26.3	31.9	14.2	80.9	13.6	5.3	17.3	5.1	29.4	45.6	32.4	39.4	17.5	99.7
Conc.2	1.5	29.60	3.22	39.4	24.2	30.5	24.2	69.5	37.8	12.8	43.1	23.0	66.1	126.9	77.9	98.1	78.0	223.8
Conc.3	2.5	40.70	4.43	40.8	23.0	29.5	30.5	62.1	53.8	16.7	57.3	39.7	81.1	180.6	101.6	130.5	135.0	274.7
Conc.4	3.0	54.82	5.96	43.2	20.8	27.9	41.0	49.8	76.8	20.4	73.0	72.0	87.6	257.7	123.8	166.2	244.6	296.9
Conc.5	4.0	66.83	7.27	42.3	19.4	26.0	43.3	42.8	91.5	23.2	83.0	92.7	91.9	307.2	140.9	189.2	315.0	311.2
Conc.6	6.5	74.77	8.13	39.5	18.4	24.1	41.0	39.1	95.7	24.6	86.1	98.1	93.7	321.0	149.6	196.1	333.3	317.6
Conc.7	9.0	77.82	8.46	38.5	18.1	23.5	40.1	37.9	97.2	25.2	87.2	100.0	94.6	326.1	153.2	198.8	339.7	320.5
Calc. Head		919.46	100.00	3.4	6.1	2.3	3.4	3.4	100.0	100.0	100.0	100.0	100.0	335.6	607.4	227.8	339.7	338.8

Table AB - 19: Test 10 flotation report. Fines removed; pH 9; collector used.

TEST 10			DATE: March 17, 2011						
OBJECTIVE:	Flotation selectivity					Feed:	bornite-chalcopyrite ore		
GRINDING CONDITIONS (Denver Mill):					FLOAT CONDITIONS:		FLOATED BY:		
MILL TYPI *Pulverizer*	ROD TYPE/Charge :	*As is*			CELL TYPE & VOLUME:	*2 L*	GAS:	*Air*	
CHARGE (; *920*	65%	solids			IMPELLER RPM:	*1200*	REPULP W/	*tap*	
WATER: *495*					Ro GAS RATE:		by air valve		
GRIND: 10 min					# of STROKES: Ro:	*30/min*			
Lime 0.37	ml	g	STAGE	COND	FLOAT	pH	DOX / pH: as	REDOX	
REAGENTS		0.0005106		TIME	TIME	End		mV	
Redox						After 1 min stirring			
						9.23		-82	
			Aeration	3					
KAX (0.1%)	2.5								
DF 250 (0.1%)	5								
						9.14		-115	
			Conc. 1		0.3	8.95		-31	
Lime		0.03							
KAX (0.1%)	2.5								
DF 250 (0.1%)	4								
						9.01		-131	
			Conc. 2		0.2				
			Conc. 3		1	9.01		-4	lack of froth
KAX (0.1%)	2.5								
DF 250 (0.1%)	4								
						8.99			
			Conc. 4		1.5	8.88		45	Poor froth
KAX (0.1%)	2.5								
DF 250 (0.1%)	4								
						8.99		-27	
			Conc. 5		2.5	8.92			No stable froth
Lime		0.04							
KAX (0.1%)	2.5								
DF 250 (0.1%)	13								
						9.01			
			Conc. 6		3.5	8.84		-29	Poor frothing particles agglomerated in the pulp
Lime		0.08							
DF 250 (0.1%)									
						9.06		-40	
			Conc. 7		3	8.91		40	More bubbles formed but not loaded with mineral particles
Totals									
Lime		0.15		12.1	Minutes				
PEX	13								
DF 250	30								

Table AB - 20: Metallurgical table for Test 10.

			Cu				Fe			S
Products	Anlytic Wt. (g)	Vol.	ppm	Dilution	mg	%	ppm	Dilution	mg	%	%
Conc 1	0.4101	250	17.27	50	215.9	52.64	5.48	50	68.5	16.71	27.0
Conc 2	0.4021	250	14.44	50	180.5	44.89	6.51	50	81.3	20.23	28.7
Conc 3	0.4116	250	13.77	50	172.1	41.82	7.035	50	87.9	21.36	29.1
Conc 4	0.4006	250	12.41	50	155.13	38.72	6.774	50	84.7	21.14	28.4
Conc 5	0.4074	250	24.46	25	152.88	37.52	11.54	25	72.1	17.70	23.1
Conc 6	0.4002	100	9.029	25	22.57	5.64	12.97	25	32.4	8.10	3.4
Conc 7	0.4029	100	13.55	25	33.88	8.41	14.44	25	36.1	8.96	4.6
-400F	1.5377	100	10.43	50	52.15	3.39	24.37	50	121.9	7.92	2.4982
Tails	1.5214	100	10.8	1	1.08	0.071	30.12	25	75.30	4.95	0.24178
Acual Head											

				Grades (%)				Recoveries (%)				Unit factors			
Products	Time (Min)	Wt. g	Wt.%	Cu	Fe	S	Bo	Cp	Cu	Fe	S	Bo	Cp	Cu	Fe	S	Bo	Cp
Conc.1	0.3	13.82	1.54	52.6	16.7	27.0	68.0	27.5	25.7	4.3	19.3	34.1	12.2	81.2	25.8	41.6	104.9	42.4
Conc.2	0.2	13.08	1.46	44.9	20.2	28.7	43.3	50.4	20.8	4.9	19.5	20.6	21.2	65.5	29.5	41.9	63.2	73.6
Conc.3	1.0	16.89	1.88	41.8	21.4	29.1	34.1	58.3	25.0	6.7	25.4	20.9	31.7	78.8	40.3	54.7	64.3	109.9
Conc.4	1.5	9.24	1.03	38.7	21.1	28.4	27.8	61.0	12.7	3.6	13.6	9.3	18.1	39.9	21.8	29.3	28.6	62.8
Conc.5	2.5	6.86	0.77	37.5	17.7	23.1	38.5	38.0	9.1	2.3	8.2	9.6	8.4	28.7	13.5	17.7	29.4	29.1
Conc.6	3.4	5.27	0.59	5.6	8.1	3.4	5.9	5.4	1.1	0.8	0.9	1.1	0.9	3.3	4.8	2.0	3.5	3.2
Conc.7	3.0	12.06	1.35	8.4	9.0	4.6	10.1	5.8	3.6	2.0	2.9	4.4	2.2	11.3	12.1	6.2	13.6	7.8
Tails		819.19	91.39	0.1	4.9	0.2	0.0	0.2	2.1	75.4	10.3	0.0	5.3	6.5	452.3	22.1	0.0	18.3
-400F		18.89		3.4	7.9	2.5	2.4	5.4										
Calc. Head		896.41	100.00	3.2	6.0	2.2	3.1	3.5	100.0	100.0	100.0	100.0	100.0	315.2	600.0	215.4	307.5	347.0
Actual Head		~900	915.30															

Products Cum. & Time (Min)		Wt. g	Wt.%	Cum. Grades (%)					Cum. Rec. (%)					Cum. Unit factors				
	0			Cu	Fe	S	Bo	Cp	Cu	Fe	S	Bo	Cp	Cu	Fe	S	Bo	Cp
Conc.1	0.3	13.82	1.54	52.6	16.7	27.0	68.0	27.5	25.7	4.3	19.3	34.1	12.2	81.2	25.8	41.6	104.9	42.4
Conc.2	0.6	26.90	3.00	48.9	18.4	27.8	56.0	38.7	46.5	9.2	38.8	54.7	33.4	146.7	55.3	83.5	168.1	116.0
Conc.3	1.6	43.79	4.89	46.2	19.6	28.3	47.6	46.2	71.5	15.9	64.2	75.6	65.1	225.4	95.5	138.2	232.4	225.9
Conc.4	3.1	53.03	5.92	44.9	19.8	28.3	44.1	48.8	84.2	19.6	77.8	84.9	83.2	265.4	117.3	167.5	261.1	288.7
Conc.5	5.6	59.89	6.68	44.0	19.6	27.7	43.5	47.6	93.3	21.8	85.9	94.5	91.6	294.1	130.9	185.2	290.5	317.8
Conc.6	9.0	65.16	7.27	40.9	18.7	25.8	40.4	44.2	94.4	22.6	86.9	95.6	92.5	297.4	135.6	187.2	293.9	321.0
Conc.7	12.0	77.22	8.61	35.8	17.1	22.4	35.7	38.2	97.9	24.6	89.7	100.0	94.7	308.7	147.7	193.3	307.5	328.8
Calc. Head		896.41	100.00	3.2	6.0	2.2	3.1	3.5	100.0	100.0	100.0	100.0	100.0	315.2	600.0	215.4	307.5	347.0

Table AB - 21: Test 11 flotation report. Fines removed; no lime addition; collector used.

Test 11			DATE: March 17, 2011							
OBJECTIVE:	Flotation selectivity					Feed:	bornite-chalcopyrite ore			
GRINDING CONDITIONS (Denver Mill):					FLOAT CONDITIONS:		FLOATED BY:			
MILL TYPE *Pulverizer*	ROD TYPE/Charge : *As is*				CELL TYPE & VOLUME:		*2 L*	GAS:	*Air*	
CHARGE (g *920*	65% solids				IMPELLER RPM:	*1200*	REPULP WA *tap*			
WATER: *495*					Ro GAS RATE:		by air valve			
GRIND: 15 min					# of STROKES: Ro:	*30/min*				
Lime 0g	ml	g	STAGE	COND	FLOAT	pH	pH	DOX / pH: as	REDOX	DO
REAGENTS		0.0005106		TIME	TIME	Start	End		mV	
Redox							After 1 min stirring			
							7.61		110	
KAX (0.1%)	2.5									
DF 250 (0.1%)	4									
							7.62		47	
			Conc. 1		0.3		7.64		215	
KAX (0.1%)	2.5									
DF 250 (0.1%)	4									
							7.64		185	
			Conc. 2		0.2					
KAX (0.1%)	2.5									
DF 250 (0.1%)	4									
			Conc. 3		1		7.65		270	
KAX (0.1%)	2.5									
DF 250 (0.1%)	4									
							7.65		180	
			Conc. 4		1.5		7.65		240	
KAX (0.1%)	2.5									
DF 250 (0.1%)	4									
							7.64		186	
			Conc. 5		2.5		7.64		250	
KAX (0.1%)	2.5									
DF 250 (0.1%)	4									
							7.64		174	
			Conc. 6		3.5		7.64		250	
Totals										
Lime				9.1	Minutes					
PEX	15									
DF 250										

Table AB - 22: Metallurgical table for Test 11.

			Cu			Fe		S
Products	Anlytic Wt.(g)	Vol.	ppm	Dilution	mg	ppm	Dilution	mg	%	%
Conc 1	0.3025	250	12.71	50	158.9	52.52	3.94	50	49.2	16.27		26.4
Conc 2	0.3034	250	13.48	50	168.5	55.54	3.34	50	41.8	13.77		28.0
Conc 3	0.3013	250	9.67	50	120.9	40.13	5.392	50	67.4	22.37		29.1
Conc 4	0.3090	250	8.726	50	109.08	35.30	4.587	50	57.3	18.56		22.1
Conc 5	0.3098	250	4.202	50	52.53	16.95	2.949	50	36.9	11.90		9.2
Conc 6	0.3025	100	10.44	25	26.10	8.63	11.86	25	29.7	9.80		4.8
-400 Feed	0.5382	250	7.191	10	17.98	3.34	17.28	10	43.2	8.03		2.4
Tails	1.503	100	14.55	1	1.46	0.097	7.84	100	78.40	5.22		0.0671
Acual Head												

			Grades (%)				Recoveries (%)				Unit factors			
Products	Time (Min)	Wt. g	Wt.%	Cu	Fe	S	Bo	Cp	Cu	Fe	S	Bo	Cp	Cu	Fe	S	Bo	Cp
Conc.1	0.3	16.99	1.89	52.5	16.3	26.4	69.3	24.8	29.4	5.0	24.9	36.5	14.7	99.3	30.8	49.9	131.1	47.0
Conc.2	0.2	10.64	1.18	55.5	13.8	28.0	73.1	26.7	19.5	2.6	16.6	24.1	9.9	65.8	16.3	33.2	86.5	31.6
Conc.3	1.0	22.66	2.52	40.1	22.4	29.1	29.6	61.6	30.0	9.1	36.6	20.8	48.5	101.2	56.4	73.4	74.8	155.3
Conc.4	1.5	11.71	1.30	35.3	18.6	22.1	35.3	37.4	13.6	3.9	14.4	12.8	15.2	46.0	24.2	28.8	46.0	48.8
Conc.5	2.5	6.53	0.73	17.0	11.9	9.2	20.8	11.1	3.6	1.4	3.3	4.2	2.5	12.3	8.6	6.7	15.1	8.1
Conc.6	3.4	4.55	0.51	8.6	9.8	4.8	10.1	6.3	1.3	0.8	1.2	1.4	1.0	4.4	5.0	2.4	5.1	3.2
Tails		825.48	91.87	0.1	5.2	0.1	0.0	0.3	2.6	77.2	3.1	0.1	8.2	8.9	479.2	6.2	0.2	26.1
-400 Feed		22.85		3.3	8.0	2.4	2.4	5.1	0.0	0.0	0.0	0.0	0.0					
Calc. Head		898.56	100.0	3.4	6.2	2.0	3.6	3.2	100.0	100.0	100.0	100.0	100.0	337.9	620.5	200.6	358.8	320.1
Actual Head		~920	921.41															

Products Cum. & Time (Min)		Wt. g	Wt.%	Cum. Grades (%)					Cum. Rec. (%)					Cum. Unit factors				
	0			Cu	Fe	S	Bo	Cp	Cu	Fe	S	Bo	Cp	Cu	Fe	S	Bo	Cp
Conc.1	0.3	16.99	1.89	52.5	16.3	26.4	69.3	24.8	29.4	5.0	24.9	36.5	14.7	99.3	30.8	49.9	131.1	47.0
Conc.2	0.6	27.63	3.07	53.7	15.3	27.0	70.8	25.5	48.9	7.6	41.4	60.7	24.5	165.1	47.1	83.1	217.6	78.5
Conc.3	1.6	50.29	5.60	47.6	18.5	28.0	52.2	41.8	78.8	16.7	78.0	81.5	73.1	266.3	103.5	156.5	292.4	233.9
Conc.4	3.1	62.00	6.90	45.3	18.5	26.9	49.0	41.0	92.4	20.6	92.4	94.3	88.3	312.3	127.7	185.3	338.4	282.7
Conc.5	5.6	68.53	7.63	42.6	17.9	25.2	46.3	38.1	96.1	22.0	95.7	98.5	90.8	324.6	136.3	192.0	353.4	290.8
Conc.6	9.0	73.08	8.13	40.4	17.4	23.9	44.1	36.1	97.4	22.8	96.9	99.9	91.8	329.0	141.3	194.4	358.6	294.0
Calc. Head		898.56	100.00	3.4	6.2	2.0	3.6	3.2	100.0	100.0	100.0	100.0	100.0	337.9	620.5	200.6	358.8	320.1

Table AB - 23: Fines removed; pH 10; collector used.

TEST 12		DATE: March 17, 2011									
OBJECTIVE:	Flotation selectivity						Feed:	bornite-chalcopyrite ore			
GRINDING CONDITIONS (Denver Mill):					FLOAT CONDITIONS:			FLOATED BY:			
MILL TYPI *Pulverizer*	ROD TYPE/Charge :		*As is*		CELL TYPE & VOLUME:		*2 L*	GAS:	*Air*		
CHARGE (g *920*	65%	solids			IMPELLER RPM:	*1200*	REPULP W.	*tap*			
WATER: *495*					Ro GAS RATE:		by air valve				
GRIND: 10 min					# of STROKES: Ro:	*30/min*					
Lime 0.72 g	ml	g	STAGE	COND	FLOAT	pH	pH	DOX / pH: as	REDOX	DO	
REAGENTS		0.0005106		TIME	TIME	Start	End		mV		
Redox							After 1 min stirring				
						10.16			-133		
KAX (0.1%)	2.5										
DF250 (0.1%)	4										
						10.14			-158		
			Conc. 1		0.3	10.04			-100		
KAX (0.1%)	2.5										
DF250 (0.1%)	4										
						10.03			-140		
			Conc. 2		0.2						
			Conc. 3		1	10			-40		
KAX (0.1%)	2.5										
DF250 (0.1%)	4										
						9.99			-93		
			Conc. 4		1.5	9.96			-40		Poor froth
Lime		0.01									
KAX (0.1%)	2.5										
DF250 (0.1%)	4										
						9.99			-102		Poor froth
			Conc. 5		2.5	9.94			-40		
Lime		0.04									
KAX (0.1%)	2.5										
DF250 (0.1%)	4										
						10			-98		
			Conc. 6		3.5	9.93			-40		
Totals											
Lime		0.05		9.1	Minutes						
PEX	13										
DF 250	20										

Table AB - 24: Metallurgical table for Test 12.

			Cu				Fe			S
Products	Anlytic Wt. (g)	Vol.	ppm	Dilution	mg	%	ppm	Dilution	mg	%	%
Conc 1	0.3276	250	13.47	50	168.4	51.40	4.45	50	55.7	16.99	26.6
Conc 2	0.3105	250	11.05	50	138.1	44.48	4.94	50	61.8	19.89	28.5
Conc 3	0.3088	250	10.42	50	130.3	42.18	5.227	50	65.3	21.16	28.5
Conc 4	0.3044	250	9.703	50	121.29	39.84	5	50	62.5	20.53	27.8
Conc 5	0.3015	250	8.883	50	111.04	36.83	4.015	50	50.2	16.65	21.9
Conc 6	0.3056	100	27.36	25	68.40	22.38	15.30	25	38.3	12.52	12.4
-400 Feed	1.5104	100	9.716	50	48.58	3.22	24.3	50	121.5	8.04	
Tails	1.5214	100	18.98	1	1.90	0.125	7.504	100	75.04	4.93	0.07179
Acual Head											

			Grades (%)				Recoveries (%)				Unit factors			
Products	Time (Min)	Wt. g	Wt.%	Cu	Fe	S	Bo	Cp	Cu	Fe	S	Bo	Cp	Cu	Fe	S	Bo	Cp
Conc.1	0.3	12.53	1.39	51.4	17.0	26.6	65.9	27.9	22.2	4.0	18.5	29.2	11.1	71.7	23.7	37.1	92.0	38.9
Conc.2	0.2	10.99	1.22	44.5	19.9	28.5	42.8	50.3	16.8	4.1	17.4	16.6	17.5	54.4	24.3	34.8	52.4	61.5
Conc.3	1.0	19.24	2.14	42.2	21.2	28.5	36.7	54.7	27.9	7.6	30.5	25.0	33.3	90.3	45.3	61.1	78.7	117.1
Conc.4	1.5	10.85	1.21	39.8	20.5	27.8	32.2	56.0	14.9	4.2	16.7	12.4	19.2	48.1	24.8	33.5	39.0	67.6
Conc.5	2.5	7.43	0.83	36.8	16.6	21.9	39.7	33.6	9.4	2.3	9.0	10.4	7.9	30.5	13.8	18.1	32.9	27.8
Conc.6	3.4	6.74	0.75	22.4	12.5	12.4	26.6	16.0	5.2	1.6	4.6	6.3	3.4	16.8	9.4	9.3	20.0	12.0
Tails		830.52	92.45	0.1	4.9	0.1	0.0	0.3	3.6	76.3	3.3	0.1	7.5	11.5	456.0	6.6	0.2	26.3
-400 Feed		23.15		3.2	8.0	0.0	8.4	-6.2										
Calc. Head		898.30	100.0	3.2	6.0	2.0	3.2	3.5	100.0	100.0	100.0	100.0	100.0	323.4	597.3	200.5	315.0	351.3
Actual Head		~900	921.45															

Products Cum. & Time (Min)		Wt. g	Wt.%	Cum. Grades (%)					Cum. Rec. (%)					Cum. Unit factors				
	0			Cu	Fe	S	Bo	Cp	Cu	Fe	S	Bo	Cp	Cu	Fe	S	Bo	Cp
Conc.1	0.3	12.53	1.39	51.4	17.0	26.6	65.9	27.9	22.2	4.0	18.5	29.2	11.1	71.7	23.7	37.1	92.0	38.9
Conc.2	0.6	23.52	2.62	48.2	18.3	27.5	55.1	38.4	39.0	8.0	35.8	45.8	28.6	126.1	48.0	71.9	144.3	100.5
Conc.3	1.6	42.76	4.76	45.5	19.6	27.9	46.8	45.7	66.9	15.6	66.3	70.8	61.9	216.5	93.3	133.0	223.0	217.6
Conc.4	3.1	53.61	5.97	44.3	19.8	27.9	43.9	47.8	81.8	19.8	83.0	83.2	81.2	264.6	118.1	166.5	262.0	285.2
Conc.5	5.6	61.04	6.80	43.4	19.4	27.2	43.4	46.1	91.2	22.1	92.1	93.6	89.1	295.0	131.9	184.6	294.8	313.0
Conc.6	9.0	67.78	7.55	41.3	18.7	25.7	41.7	43.1	96.4	23.7	96.7	99.9	92.5	311.8	141.3	193.9	314.8	325.0
Calc. Head		898.30	100.00	3.2	6.0	2.0	3.2	3.5	100.0	100.0	100.0	100.0	100.0	323.4	597.3	200.5	315.0	351.3

Table AB - 25: Fines removed; no lime addition; collector used; 15 minutes grinding; Nitrogen used in grinding pulp, conditioning and flotation.

TEST 13		DATE: March 17, 2011								
OBJECTIVE:	Flotation selectivity				Feed:	bornite ore				
GRINDING CONDITIONS (Denver Mill):					FLOAT CONDITIONS:		FLOATED BY:			
MILL TYPI *Pulverizer*	ROD TYPE/Charge :	*As is*			CELL TYPE & VOLUME:	*2 L*	GAS:	*Air*		
CHARGE (; *510.6*	100%	solids			IMPELLER RPM:	*1200*	REPULP W/	*tap*		
WATER:					Ro GAS RATE:		by air valve			
GRIND: 15 min					# of STROKES: Ro:	*30/min*				
0 lime	ml	g	STAGE	COND	FLOAT	pH	pH	DOX / pH: as	REDOX	DO
REAGENTS		0.0005106		TIME	TIME	Start	End		mV	
Redox							After 1 min stirring			
						7.87			-86	
KAX	2.5								-140	
DF 250	4			1		7.84			-180	
			CON. 1		20sec	7.84			-190	
KAX	2.5									
DF 250	4					7.84			-195	
			CON. 2		14sec	7.84			-202	
			CON. 3		1	7.85			-211	
KAX	2.5									
DF 250	4									
				1		7.85			-253	
			CON. 4		1.5	7.85				
KAX	2.5									
DF 250	4					7.86			-255	
			CON. 5		1.5	7.88				
KAX	2.5									
DF 250	4					7.88			-303	
			CON.6		1	7.91			-298	
		NITROGEN STOPPED (5 min conditioning)								

Table AB - 26: Continued Test 13 flotation with aeration using normal air and flotation in the presence of a collector.

Second part of nitrogen float: Aeration with normal air for 5 minutes before continuing flotation.

Reagent	*Dosage*	*Stage*	*float time*	*pH*	*Redox mV*
					-320
					-310
					-277
					-262
					-110
					-54
					-37
					-30
					-30
					-35
					-38
KAX	2.5				
DF 250	4			7.92	-65
		CON.7	1	7.89	-33
KAX	2.5				
DF 250	4			7.88	-69
		CON. 8	1	7.86	-49
KAX	2.5				
DF 250	4			7.86	-67
		CON. 9	1.0	7.82	-65
KAX	2.5				
DF 250	4			7.8	-70
		CON. 10	2	7.78	-61
TOTAL					
KAX	22.5		10.5		
DF 250	36				

Table AB - 27: Metallurgical table for Test 13.

				Grades (%)					Recoveries (%)					Unit factors				
Products	Time (Min)	Wt. g	Wt.%	Cu	Fe	S	Bo	Cp	Cu	Fe	S	Bo	Cp	Cu	Fe	S	Bo	Cp
Conc.1	0.3	5.19	0.58	57.0	12.6	24.1	87.2	5.2	10.5	1.2	7.1	15.8	0.9	33.2	7.4	14.1	50.9	3.0
Conc.2	0.2	4.59	0.52	44.8	20.0	24.6	71.7	18.0	7.3	1.7	6.4	11.5	2.7	23.1	10.3	12.7	37.0	9.3
Conc.3	1.0	5.14	0.58	50.2	14.0	25.4	65.9	24.5	9.2	1.3	7.4	11.8	4.2	29.0	8.1	14.7	38.1	14.2
Conc.4	1.5	6.53	0.73	44.8	15.9	23.5	56.6	25.8	10.4	1.9	8.7	12.9	5.6	32.9	11.7	17.2	41.6	19.0
Conc.5	1.5	3.21	0.36	35.8	18.3	22.8	34.8	39.8	4.1	1.1	4.2	3.9	4.2	12.9	6.6	8.2	12.5	14.4
Conc.6	1.0	1.52	0.17	28.8	19.3	21.2	20.5	45.7	1.6	0.5	1.8	1.1	2.3	4.9	3.3	3.6	3.5	7.8
Conc.7	1.0	22.5	2.53	42.8	20.3	27.8	40.1	50.2	34.2	8.3	35.7	31.5	37.3	108.4	51.4	70.3	101.6	127.0
Conc.8	1.0	12.77	1.44	33.5	23.9	27.1	17.5	64.8	15.2	5.5	19.8	7.8	27.3	48.1	34.3	38.9	25.1	93.0
Conc.9	1.0	5.62	0.63	17.5	16.2	13.9	9.8	32.6	3.5	1.6	4.4	1.9	6.1	11.1	10.2	8.8	6.2	20.6
Conc.10	1.9	5.79	0.65	8.6	11.5	5.4	8.6	9.2	1.8	1.2	1.8	1.7	1.8	5.6	7.5	3.5	5.6	6.0
Tails		816.64	91.81	0.1	5.1	0.1	0.0	0.3	2.4	75.7	2.6	0.1	7.7	7.6	468.5	5.1	0.2	26.1
-400 Feed		22.02		3.2	7.9	2.4	2.2	5.3										
Calc. Head		889.50	100.0	3.2	6.2	2.0	3.2	3.4	100.0	100.0	100.0	100.0	100.0	316.8	619.2	197.2	322.2	340.3
Actual Head		~890	911.52															

				Cum. Grades (%)					Cum. Rec. (%)					Cum. Unit factors				
Products Cum. & Time (Min)		Wt. g	Wt.%	Cu	Fe	S	Bo	Cp	Cu	Fe	S	Bo	Cp	Cu	Fe	S	Bo	Cp
	0																	
Conc.1	0.3	5.19	0.58	57.0	12.6	24.1	87.2	5.2	10.5	1.2	7.1	15.8	0.9	33.2	7.4	14.1	50.9	3.0
Conc.2	0.6	9.78	1.10	51.2	16.1	24.3	79.9	11.2	17.8	2.9	13.6	27.3	3.6	56.3	17.7	26.7	87.9	12.3
Conc.3	1.6	14.92	1.68	50.9	15.3	24.7	75.1	15.8	26.9	4.2	21.0	39.1	7.8	85.3	25.7	41.4	125.9	26.5
Conc.4	3.1	21.45	2.41	49.0	15.5	24.3	69.5	18.8	37.3	6.0	29.7	52.0	13.3	118.2	37.4	58.6	167.5	45.4
Conc.5	4.6	24.66	2.77	47.3	15.9	24.1	64.9	21.6	41.4	7.1	33.9	55.9	17.6	131.2	44.0	66.9	180.1	59.8
Conc.6	5.6	26.18	2.94	46.2	16.1	23.9	62.4	23.0	43.0	7.6	35.7	57.0	19.9	136.1	47.3	70.5	183.6	67.6
Conc.7	6.6	48.68	5.47	44.7	18.0	25.7	52.1	35.6	77.2	15.9	71.4	88.5	57.2	244.5	98.7	140.8	285.1	194.6
Conc.8	7.6	61.45	6.91	42.3	19.3	26.0	44.9	41.6	92.3	21.5	91.2	96.3	84.5	292.5	133.1	179.7	310.2	287.5
Conc.9	8.6	67.07	7.54	40.3	19.0	25.0	42.0	40.9	95.8	23.1	95.6	98.2	90.6	303.6	143.3	188.5	316.4	308.2
Conc.10	10.5	72.86	8.19	37.8	18.4	23.4	39.3	38.4	97.6	24.3	97.4	99.9	92.3	309.2	150.8	192.0	322.0	314.1
Calc. Head		889.50	100.00	3.2	6.2	2.0	3.2	3.4	100.0	100.0	100.0	100.0	100.0	316.8	619.2	197.2	322.2	340.3

Table AB - 28: Test 14 flotation report. Fines removed; no lime addition; collectorless first 3 concentrates.

TEST 14			DATE: March 17, 2011							
OBJECTIVE:	Flotation selectivity						Feed:	bornite-chalcopyrite ore		
GRINDING CONDITIONS (Denver Mill):					FLOAT CONDITIONS:		FLOATED BY:			
MILL TYPI *Pulverizer*	ROD TYPE/Charge :	*As is*			CELL TYPE & VOLUME:		*2 L*	GAS:	*Air*	
CHARGE (: *920*	65%	solids			IMPELLER RPM:	*1200*	REPULP W/	*tap*		
WATER: *495*					Ro GAS RATE:		by air valve			
GRIND: 10 min					# of STROKES: Ro:	*30/min*				
Lime 0.10 g	ml	g	STAGE	COND	FLOAT	pH	pH	DOX / pH: as	REDOX	DO
REAGENTS		0.0005106		TIME	TIME	Start	End		mV	
Redox							After 1 min stirring			
							9.02		-163	
DF 250 (0.1%)	5						9.02		-163	
			Conc. 1		0.5		8.95		-110	
Lime		0.01								
DF 250 (0.1%)	5									
							9.04		-139	
			Conc. 2		1		8.96		-110	
Lime		0.01								
DF 250 (0.1%)	5									
							9.19		-138	
			Conc. 3		1		9.1		-100	
KAX (0.1%)	1									
DF 250 (0.1%)	5									
							9.06		-97	
			Conc. 4		0.5		8.99		-28	
Lime		0.005								
KAX (0.1%)	1									
DF 250 (0.1%)	5									
							9		-66	
			Conc. 5		1		8.91		-10	
Lime		0.015								
KAX (0.1%)	2.5									
DF 250 (0.1%)	5									
							9.01		-35	
			Conc. 6		2.5		8.84		50	
Lime		0.01								
KAX (0.1%)	3									
DF 250 (0.1%)	5									
							9.01		4	
			Conc. 7		2.5		8.82		40	
Totals										
Lime		0.050		9	Minutes					
PEX	8									
DF 250	35									

Table AB - 29: Metallurgical table for Test 14.

			Cu			Fe			S
Products	Anlytic Wt. (g)	Vol.	ppm	Dilution	mg	%	ppm	Dilution	mg	%	%
Conc 1	0.3047	250	8.66	50	108.3	35.53	6.57	50	82.1	26.94	32.3
Conc 2	0.3017	250	9.461	50	118.3	39.20	5.91	50	73.9	24.50	30.5
Conc 3	0.3018	250	10.46	50	130.8	43.32	4.876	50	61.0	20.20	27.6
Conc 4	0.3016	250	12.28	50	153.50	50.90	3.347	50	41.8	13.87	23.8
Conc 5	0.3013	250	10.49	50	131.13	43.52	3.11	50	38.9	12.90	20.0
Conc 6	0.3026	100	21.62	25	54.05	17.86	12.30	25	30.8	10.16	8.8
Conc 7	0.3014	100	9.121	25	22.80	7.57	9.84	25	24.6	8.16	3.7
-400 Feed	0.5341	250	7.024	10	17.56	3.29	17.03	10	42.6	7.97	2.5
Tails	1.5282	100	11.23	1	1.12	0.073	7.6	100	76.00	4.97	0.04189
Acual Head											

			Grades (%)			Recoveries (%)			Unit factors		
Products	Time (Min)	Wt. g	Wt.%	Cu	Fe	S	Bo	Cp	Cu	Fe	S	Bo	Cp	Cu	Fe	S	Bo	Cp
Conc.1	0.5	9.34	1.03	35.5	26.9	32.3	9.2	85.7	11.2	4.6	16.5	2.8	26.9	36.6	27.8	33.3	9.5	88.4
Conc.2	1.0	16.51	1.82	39.2	24.5	30.5	23.6	70.0	21.9	7.4	27.6	12.6	38.9	71.4	44.7	55.5	43.0	127.6
Conc.3	1.0	12.70	1.40	43.3	20.2	27.6	42.0	48.3	18.6	4.7	19.2	17.2	20.6	60.7	28.3	38.7	58.9	67.7
Conc.4	0.5	14.59	1.61	50.9	13.9	23.8	71.9	15.5	25.2	3.7	19.0	33.8	7.6	82.0	22.3	38.3	115.8	25.0
Conc.5	1.0	12.2	1.35	43.5	12.9	20.0	62.3	11.6	18.0	2.9	13.4	24.5	4.8	58.6	17.4	26.9	84.0	15.7
Conc.6	2.5	3.59	0.40	17.9	10.2	8.8	24.2	7.5	2.2	0.7	1.7	2.8	0.9	7.1	4.0	3.5	9.6	3.0
Conc.7	2.5	3.04	0.34	7.6	8.2	3.7	10.4	3.0	0.8	0.5	0.6	1.0	0.3	2.5	2.7	1.2	3.5	1.0
Tails		833.94	92.06	0.1	5.0	0.0	0.2	0.0	2.1	75.7	1.9	5.4	0.0	6.8	457.8	3.9	18.4	0.0
-400 Feed		22.74		3.3	8.0	2.5	2.2	5.6										
Calc. Head		905.91	100.0	3.3	6.1	2.0	3.4	3.3	100.0	100.0	100.0	100.0	100.0	325.8	605.0	201.3	342.6	328.3
Actual Head		~900	928.65															

				Cum. Grades (%)					Cum. Rec. (%)					Cum. Unit factors				
Products Cum. & Time (Min)		Wt. g	Wt.%	Cu	Fe	S	Bo	Cp	Cu	Fe	S	Bo	Cp	Cu	Fe	S	Bo	Cp
	0																	
Conc.1	0.5	9.34	1.03	35.5	26.9	32.3	9.2	85.7	11.2	4.6	16.5	2.8	26.9	36.6	27.8	33.3	9.5	88.4
Conc.2	1.5	25.85	2.85	37.9	25.4	31.1	18.4	75.7	33.2	12.0	44.1	15.3	65.8	108.1	72.4	88.8	52.5	216.0
Conc.3	2.5	38.55	4.26	39.7	23.7	30.0	26.2	66.7	51.8	16.7	63.3	32.5	86.4	168.8	100.7	127.5	111.4	283.7
Conc.4	3.0	53.14	5.87	42.8	21.0	28.3	38.7	52.6	77.0	20.3	82.4	66.3	94.0	250.8	123.1	165.8	227.2	308.6
Conc.5	4.0	65.34	7.21	42.9	19.5	26.7	43.1	45.0	95.0	23.2	95.8	90.8	98.8	309.4	140.5	192.7	311.2	324.3
Conc.6	6.5	68.93	7.61	41.6	19.0	25.8	42.2	43.0	97.1	23.9	97.5	93.6	99.7	316.5	144.5	196.2	320.7	327.3
Conc.7	9.0	71.97	7.94	40.2	18.5	24.9	40.8	41.3	97.9	24.3	98.1	94.6	100.0	319.0	147.2	197.4	324.2	328.3
Calc. Head		905.91	100.0	3.3	6.1	2.0	3.4	3.3	100.0	100.0	100.0	100.0	100.0	325.8	605.0	201.3	342.6	328.3

Appendix C

Leaching Balances

Experimental conditions for preliminary experiments (Factorial design) are presented in this appendix. Repeat test are then presented with residue analysis.

Table AC - 1: Sample data for preliminary leaching tests.

	Individual		1:1		1:3		3:1	
Mineral	Bo	cp	Bo	Cp	Bo	Cp	Bo	Cp
Sample weight (g)	1.0	1.0	1.0	1.0	1.0	3.0	3.0	1.0
Cu content (mg)	450	330						

Table AC - 2: Factorial design parameter high and low values.

Parameters	Value	Binary
Acid Concentration (M)	0.5	1
	0.1	-1
Ferric ions (M)	0.1	1
	0	-1
Particle size (mesh)	-100 +400	1
	-400	-1

Volume used for preliminary experiments: 500 ml.

Volume used for mineral mixture leaching and corresponding individual leaching: 1L

Table AC - 3: Factorial design for leaching experiment - individual and mixed mineral leaching test sets.

Test #.	P.S.	Temp.	Acid conc.	Fe3+
1	-1	-1	-1	-1
2	-1	-1	-1	1
3	-1	-1	1	-1
4	-1	-1	1	1
5	-1	1	-1	-1
6	-1	1	-1	1
7	-1	1	1	-1
8	-1	1	1	1
9	1	-1	-1	-1

10	1	-1	-1	1
11	1	-1	1	-1
12	1	-1	1	1
13	1	1	-1	-1
14	1	1	-1	1
15	1	1	1	-1
16	1	1	1	1

Conditions presented in Table AC - 3 above were used in leaching tests for bornite, chalcopyrite and mineral mixtures. The data for these leaching tests is presented in tables that follow. The tests numbers correspond to the test conditions given in Table AC - 3.

Table AC - 4: leaching results for leaching of -400 mesh bornite.

Time (min)	Bo. 1	Bo. 2	Bo. 3	Bo. 4	Bo. 5	Bo. 6	Bo. 7	Bo. 8
0	0	0	0	0	0	0	0	0
15	0.0	20.5	0.4	24.4	1.7	44.4	0.3	43.1
30	0.0	23.4	0.5	23.8	1.7	55.0	1.0	51.1
60	0.1	24.0	0.6	24.8	2.8	62.2	2.8	60.8
120	0.4	25.0	1.3	25.9	6.7	69.1	6.9	68.2
180	1.5	25.9	1.6	25.2	11.2	77.3	11.8	77.4
240	2.6	26.8	1.7	26.1	19.6	84.1	15.3	80.6

Table AC - 5: leaching results for leaching of -100 +400 mesh bornite.

Time (min)	Bo. 9	Bo. 10	Bo. 11	Bo. 12	Bo. 13	Bo. 14	Bo. 15	Bo. 16
0	0	0	0	0	0	0	0	0
15	0.4	25.4	1.0	23.3		46.4	0.2	44.8
30	0.6	25.4	1.6	24.3	1.0	36.5	0.4	53.4
60	0.9	25.7	1.7	24.9	3.2	52.7	4.0	59.4
120	1.6	26.3	2.6	25.2	7.4	56.5	10.2	68.8
180	2.0	26.6	4.0	26.7	12.0	63.5	15.3	74.8
240	3.1	26.8	4.8	26.9	18.5	66.7	17.9	79.3

Table AC - 6: leaching results for leaching of -400 mesh chalcopyrite.

Time (min)	Cp. 1	Cp. 2	Cp. 3	Cp. 4	Cp. 5	Cp. 6	Cp. 7	Cp. 8
0	0	0	0	0	0	0	0	0
15	0.2	1.0	0.2	0.7	0.3	4.1	0.3	4.5
30	0.2	1.0	0.2	0.7	0.4	6.7	0.4	7.3
60	0.3	1.0	0.3	0.8	0.6	10.8	0.5	10.7
120	0.4	1.1	0.4	0.8	1.0	14.7	0.8	14.1
180	0.6	1.2	0.5	0.8	1.3	17.3	1.1	16.3
240	0.8	1.3	0.6	0.8	1.6	19.2	1.5	18.2

Table AC - 7: leaching results for leaching of -400 mesh 1:1 bo-cp mixture.

Time (min)	1:1. 1	1:1. 2	1:1. 3	1:1. 4	1:1. 5	1:1. 6	1:1. 7	1:1. 8
0	0	0	0	0	0	0	0	0
15	0.1	13.4	0.1	14.2	0.2	28.2	0.3	29.7
30	0.2	14.9	0.1	15.4	0.3	36.6	0.5	38.3
60	0.4	16.1	0.2	16.1	0.8	43.0	1.2	45.8
120	0.9	17.0	0.3	16.8	2.1	52.2	3.7	56.6
180	1.4	17.0	0.4	17.5	3.8	61.1	6.8	65.5
240	2.0	17.6	0.5	18.3	6.3	70.0	10.8	72.8

Table AC - 8: leaching results for leaching of -100 +400 mesh 1:1 bo-cp mixture.

Time (min)	1:1. 9	1:1. 10	1:1. 11	1:1. 12	1:1. 13	1:1. 14	1:1. 15	1:1. 16
0	0	0	0	0	0	0	0	0
15	0.3	15.0	0.3	14.6	0.2	31.4	0.6	31.4
30	0.4	14.8	0.4	15.2	0.7	37.5	1.8	38.1
60	0.7	15.0	0.6	15.4	2.5	45.1	5.1	45.2
120	1.1	15.2	1.2	15.7	6.3	55.5	10.2	55.8
180	1.5	15.6	2.1	16.0	10.6	63.0	13.4	65.0
240	1.8	16.1	2.1	16.4	13.3	70.5	16.0	72.2

Table AC - 9: leaching results for leaching of -400 mesh 1:3 bo-cp mixture.

Time (min)	1:3. 1	1:3. 2	1:3. 3	1:3. 4	1:3. 5	1:3. 6	1:3. 7	1:3. 8
0	0	0	0	0	0	0		
15	0.1	7.9	0.1	6.7	0.2	16.6		
30	0.2	7.9	0.2	7.4	0.3	23.0		
60	0.4	8.7	0.8	7.7	0.6	30.7		
120	0.7	9.1	0.8	8.0	1.4	43.1		
180	1.1	9.6	1.2	8.2	2.2	53.5		
240	1.4	9.9	1.4	8.4	3.0	61.9		

Table AC - 10: leaching results for leaching of -100 +400 mesh 1:3 bo-cp mixtures.

Time (min)	1:3. 9	1:3. 10	1:3. 11	1:3. 12	1:3. 13	1:3. 14	1:3. 15	1:3. 16
0	0	0	0	0	0	0		
15	0.3	8.0	0.3	7.4	0.3	16.3		
30	0.4	8.3	0.3	7.5	0.8	21.0		
60	0.6	8.1	0.6	7.7	2.1	30.1		
120	1.0	8.2	1.0	7.8	4.2	48.6		
180	1.3	8.4	1.4	8.0	5.7	61.6		
240	1.7	8.5	1.7	8.1	7.0	69.8		

Table AC - 11: leaching results for chalcopyrite and bornite in 1000 ml solution.

	Bornite				Chalcopyrite			
Mesh	Bo -100	Bo -100	Bo-400	Bo -400	Cp -100	Cp -100	Cp -100	Cp -400
Weight	1.50	1.50	1.64	1.60	2.00	1.53	1.52	1.63
Wt. Cu	0.713	0.718	0.723	0.704	0.540	0.460	0.440	0.555
Time	Bo -100	Bo -100	Bo-400	Bo -400	Cp -100	Cp -100	Cp -100	Cp -400
0	0	0	0	0	0	0	0	0
15	36	35	46	43	1	1	1	5
30	42	43	52	50	2	1	1	7
60	48	51	60	56	3	2	1	11
240	61	67	84	74	7	3	2	23
300	65	74	89	80	9	3	2	25
360	71	78	92	84	11	3	2	28
	71	78	92	84	11	3	2	28
Residue	27	24	8	19	88	98	99	72

Table AC - 12: leaching results for 1:1 bo-cp mixture in 1000 ml solution.

	2 g		-100 Bo & Cp		-100 Bo		-400 Bo	
Mesh	Bo -100	Cp -100	Bo-100	Cp -100	Bo -100	Cp -400	Bo -400	Cp -400
Weight	1.24	1.00	1.0559	1.0042	1.5044	1.4182	1.5087	1.4038
Wt. Cu	0.875056		0.788008		1.147572		1.126033	
0	0		0		0		0	
15	24		19		25		27	
30	30		24		31		34	

60	34	30	38	42
240	47	49	60	73
300	52	52	69	82
360	57	57	77	88
Extracted	57	57	77	88
Residue	42	44	23	14

Table AC - 13: Leaching results for 3:1 bo-cp mixture in 1000 ml solution.

	2:1		-100 Bo & Cp		-100 Bo		-400 Bo	
Mesh	Bo -100	Cp -100	Bo-100	Cp -100	Bo -100	Cp -400	Bo -400	Cp -400
Weight	2.0974	0.9999	3.1592	1.0009	3.1265	1.0003	3.1055	1.0011
Wt. Cu	1.29		1.80		1.83		1.76	
0	0		0		0		0	
15	29		30		33		34	
30	34		36		38		42	
60	39		41		46		47	
240	69		68		80		86	
300	81		74		87		89	
360	90		79		90		91	
Extracted	90		79		90		91	
Residue	10		21		12		10	

Table AC - 14: Leaching results for 1:3 bo-cp mixture in 1000 ml solution.

	2 g		-100 Bo & Cp		-100 Bo		-400 Bo	
Mesh	Bo -100	Cp -100	Bo-100	Cp -100	Bo -100	Cp -400	Bo -400	Cp -400
Weight	0.60	1.94	1.0024	2.8577	1.0009	3.0001	1.0463	3.0011
Wt. Cu	0.83		1.31		1.50		1.47	
0	0		0		0		0	
15	13		5		16		17	
30	16		6		22		23	
60	18		7		34		32	
240	25		36		85		82	
300	27		46		89		87	
360	30		62		90		90	

Extracted	30	62	90	90
Residue	67	39	11	11