A study of the flotation characteristics of a complex copper ore

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For Gladys, Kris and Monica

Declaration

This thesis is submitted to the University of Cape Town for the degree of Master of Science (MSc). I know the meaning of plagiarism and declare that all the work in this document, save for that which is properly acknowledged, is my own.

Monica Shamvuse Kalichini

Abstract

Kansanshi copper mine is situated in the north western province of Zambia. Weathering has given rise to a vertically zoned profile comprising leached, refractory, oxide, mixed and hypogene sulphide mineralisation. As a result of the mineral variations, the processing plant treats three distinct ore types; oxide, sulphide and mixed.

The objective of this study was to investigate the floatability of a complex Kansanshi mixed copper ore comprising sulphide and oxide minerals with a view to achieving an optimal flotation performance in the treatment of the Kansanshi ore body. This required an in-depth analysis of the mineralogy of the feed as well as tailings samples after different flotation processes involving a range of reagent types and dosage procedures. The ore samples studied represented a high quality (HQ) ore dominated by sulphide minerals and low quality (LQ) ore dominated by oxide minerals. The quality of an ore at Kansanshi is defined by the acid soluble copper (ASCu) content of the ore, which is used as a proxy for oxide mineral content. An important finding in this study was that sulphide minerals are also prone to digestion during this analysis.

Chalcopyrite was the major copper mineral in the HQ ore, constituting 3.9 %, but only 1.0 % of the LQ ore. LQ ore was dominated by chrysocolla, which constituted 3.8 % of the ore. The treatment of HQ ore with 30 g/t SIBX has shown that up to 90 % of the copper can be recovered from HQ ore. On the other hand, while 30 g/t SIBX was sufficient for chalcopyrite recovery in LQ ore, the tailings mineralogy after flotation with SIBX indicated that 78.8 % of the unrecovered copper in LQ ore was present as chrysocolla, 1.3 % as malachite and 5.8 % as chalcopyrite and therefore LQ ore required alternative flotation methods for the recovery of the oxide minerals. Comparison of slug sulphidisation and controlled potential sulphidisation (CPS) of LQ ore have shown that CPS performs better than slug sulphidisation only when the correct potential range and SIBX dosage after sulphdisation are used. Tailings mineralogy of LQ ore after sulphidisation showed a copper deportment of 0.1 % cuprite, 0.6 % malachite, 0.8 % chalcopyrite and 84.8 % chrysocolla, suggesting that all oxide copper minerals present in the LQ ore, except chrysocolla, are amenable to flotation using SIBX after sulphidisation. This observation was further verified through sulphidisation in a microflotation cell, which showed malachite recovery of 18.2 % compared to only 0.5 % of the chrysocolla.

A techno-economic analysis comparing slug sulphidisation and CPS has indicated that CPS using a potential range of -300 to -400 mV performs better than slug sulphidisation from an economic stand point. At this potential, a NaHS:SIBX ratio of 7:1 was required, further highlighting the importance of using the correct collector dosage after sulphidisation.

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Publications and presentations

Kalichini, M., Corin, K.C. & O'Connor, C.T. (2013). The state of oxidised/mixed mineral recovery through flotation- A review. *SAIMM Minproc conference*. 7-8 August 2013. Cape Town.

Kalichini, M, Ngulube, C, van der Merwe, C., Peart, M. & Hoffmann L. (2013). Development and implementation of a new selective collector for copper sulphide ore at FQML, Kansanshi mine. *Proceedings of the 6th International Flotation Conference (Flotation '13)*. 18-21 November 2013. Cape Town.

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Glossary

AAS	Atomic adsorption spectroscopy
ASCu	Acid soluble copper. Determined by digesting a copper containing sample in a 10 % Sulphuric acid solution
AiCu	Acid insoluble copper. Determined as the difference between total copper and acid soluble copper
Association	Relative measure of the surface area in contact between two minerals
BSE	Back scattered electrons
Cu	Copper, Cu^{2+} as free ions
Deportment	Distribution of an element in a variety of minerals
E_h	Oxidation reduction potential. Potential value of a platinum electrode versus a saturated calomel electrode
E_S	Potential value of an Ion Selective electrode versus a saturated calomel electrode
Fe	Iron
HQ	High quality. Used to describe ore with <30 % of the total copper reporting as acid soluble copper
ISE	Ion selective electrode
LQ	Low quality. Used to describe ore with >30 % of the total copper reporting as acid soluble copper
Liberation	Area % that a mineral grain occupies in a particle, calculated by examining 2D sections of a statistically representative set of particles containing the particular mineral
Mt	Million tons
NaHS	Sodium hydrosulphide
PMA	Particle mineralogical analysis
ppm	Parts per million

PSD	Particle size distribution
XRD	X-Ray diffraction
QEMSCAN	Quantitative evaluation of minerals by scanning electron microscopy
SERS	Surface enhanced raman spectroscopy
SIBX	Sodium isobutyl xanthate
SIP	Species identification protocol
SMS	Specific mineral search
TCu	Total copper. Refers to all the copper present in sample, determined by decomposing a copper containing sample in a 10 ml mixture of 3 parts hydrochloric acid, 1 part hydrofluoric acid and 1 part nitric acid
XRF	X-Ray fluorescence spectroscopy

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1. INTRODUCTION

> "Every philosophy is tinged with the coloring of some secret imaginative background, which never emerges explicitly into its train of reasoning"
>
> \- Alfred North Whitehead

1.1. Background

The primary copper bearing mineral in the Kansanshi ore deposit is chalcopyrite ($CuFeS_2$) with chalcocite (Cu_2S) and bornite (Cu_5FeS_4) existing as secondary sulphides. There are also significant occurrences of malachite ($CuCO_3{\cdot}Cu(OH)_2$), azurite ($Cu_3(CO_3)_2(OH)_2$), chrysocolla ($(Cu,Al)_2H_2Si_2O_5(OH)_4{\cdot}nH_2O$) and tenorite ($CuO$) as a result of supergene enrichment processes. As a result of the mineral variations, the plant treats three distinct ore types; oxide, sulphide and mixed. The oxide ore contains mostly oxide minerals and is treated by first floating out any acid insoluble copper (AiCu) - which is a proxy for sulphide minerals - and then leaching the tails, which contain the bulk of the acid soluble copper (ASCu) – proxy for oxide minerals. Sulphide ore is treated only by flotation. Mixed ore is a collective term describing the transition zones between oxide and sulphide ores. It is arbitrarily regarded as either low quality or high quality depending on the ASCu content. Mixed ore is associated with calcite as the major non-sulphide gangue (NSG) and therefore cannot be leached. Table 1.1-1 and Table 1.1-2 outline the basic and detailed classification of mixed ore respectively.

Table 1.1-1: Basic classification of Kansanshi mixed ore

Quality	Minerals present	Properties
High	Chalcopyrite	• Mostly sulphide minerals with minor oxides • Low clay content • Contains some Iron Oxides • Surface tarnish
↑	Bornite	
	Chalcocite	
	Covelite	
	Digenite (minor)	
	Malachite	• Mostly oxide minerals with very minor sulphide mineral content • Very high clay content
↓	Azurite	
Low	Chrysocolla	
	Tenorite	

Table 1.1-2: Mixed ore classification (Source; Kansanshi Mining Department)

Mixed ore name	Colour code	Material type	Grade Parameters			Mineralogy	Notes
			%TCu	ASCu ratio (%)	AOR		
Mixed leach	Blue	LG	< 0.4	> 95	< 4	Tenorite, malachite, chrysocolla, chalcocite, minor bornite, diginite and covellite. Some relict chalcopyrite still remains in the center of mineralised particles.	Occurs in areas of incomplete weathering in vein hosted areas where Cu sulphide minerals have been partially weathered due to encasement in quartz. Basically there is too much ASCu for flotation, yet too little of it for leaching.
		HG	> 0.4	> 95	< 4		
Mixed float to leach	Purple	LG	< 0.4		< 4	More malachite and chrysocolla than in "Mixed leach". Higher GAC, more clays, talc and graphite (generally problematic gangue minerals).	Originally, this was pushed to to the "Mixed float" circuit and blended in, but the oxide minerals affected recovery and Cu lost to tails was more than the value of acid consumed in the leach circuit. It usually occurs in patches that cannot be mined out selectively, so it is blended in with either HG or LG "Mixed leach" (blue) blocks and processed.
		HG	> 0.4		< 4		
Mixed float to leach	Red	LG	< 1	> 30	> 5	Typically contains more oxide minerals. The ore is heavily weathered and the mineral surfaces are badly polluted, reducing the capacity for collector adsorption.	The quality of this material is too poor for efficient recovery in the mixed float circuit mainly due to a lot of clay and Iron oxide minerals further polluting the mineral surfaces and making it non ammenable to NaHS sulphidisation. Recoveries are highly variable but rarely exceed 65 %. ASCu grades are typically too low for leach and pollutants also cause major issues in the CCD
		HG	> 1	> 30	> 5		
Mixed float to float	Orange	LG	0.5 - 1	< 30		Dominantly chalcopyrite with minor bornite and often with some surface tarnish that tends to penetrate the mineral cleavage. In areas with slightly stronger weathering, mineralogy starts to change and more chalcocite, covellite and diginite appear. Iron oxides aslo starts forming.	Tarnish on the Cu mineral surface hinders reactions to collectors. NaHS is added to improve recovery. This ore type generally floats very well in the mixed float circuit and, provided it is not blended with poorer quality material, can bring recoveries up to 80 %.
		HG	> 1	< 30			

AOR = Acid to ore ratio, **GAC** = Gangue Acid Consumption, **HG** = High Grade, **LG** = Low Grade, **CCD** = Counter Current Decantation

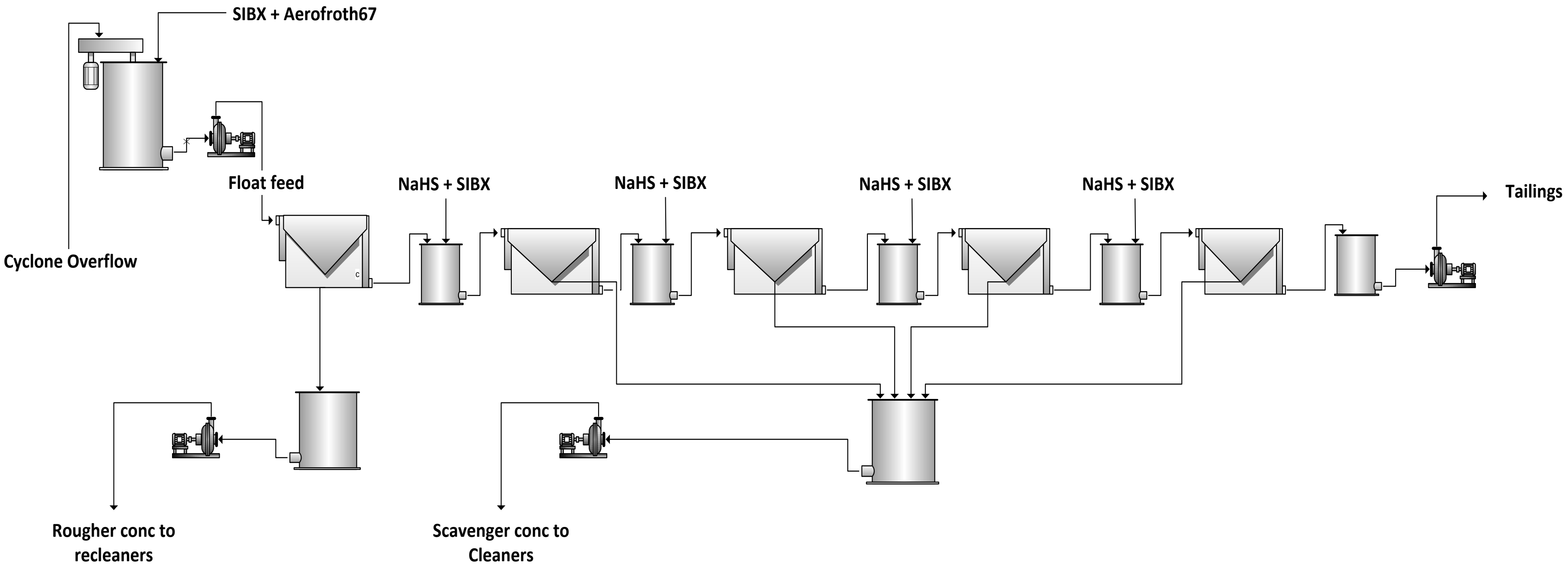


Figure 1.1-1: Schematic representation of the rougher and scavenger flotation circuit for the treatment of mixed ore at Kansanshi

Figure 1.1-1 shows the current beneficiation circuit of mixed ore at Kansanshi Mine. The mill discharge is classified using cyclones to obtain a particle size distribution of 80 % of the cyclone overflow passing 150 µm. The cyclone overflow reports to a tank where it is conditioned with collector, Sodium Iso-butyl Xanthate (SIBX), and a polyglycol frother (Cytec Aerofroth 68). The rougher stage of the mixed float circuit is used to recover only copper sulphide minerals. The rougher concentrate then proceeds to the re-cleaner section. Rougher tailings report to the rougher scavenger section, which is a four stage Controlled Potential Sulphidisation (CPS) process. Sodium hydrosulphide (NaHS) is added to the slurry to maintain a sulphidisation potential (E_S) between -500 mV and -550 mV, followed by SIBX. A silver/silver chloride Ion Selective Electrode (ISE) is used for potential control. The SIBX dosage at each CPS stage is 10 % of the NaHS dosage that maintains the potential. The sulphidised slurry is floated and the tailings move on to the next sulphidisation stage etc. All four concentrates from this part of the circuit report to the cleaning section and on to the re-cleaners. The final tailings from the sulphidisation stage report to the tailings dam. Average monthly copper recovery in the mixed ore circuit rarely exceeds 70 % (Figure 1.1-2).

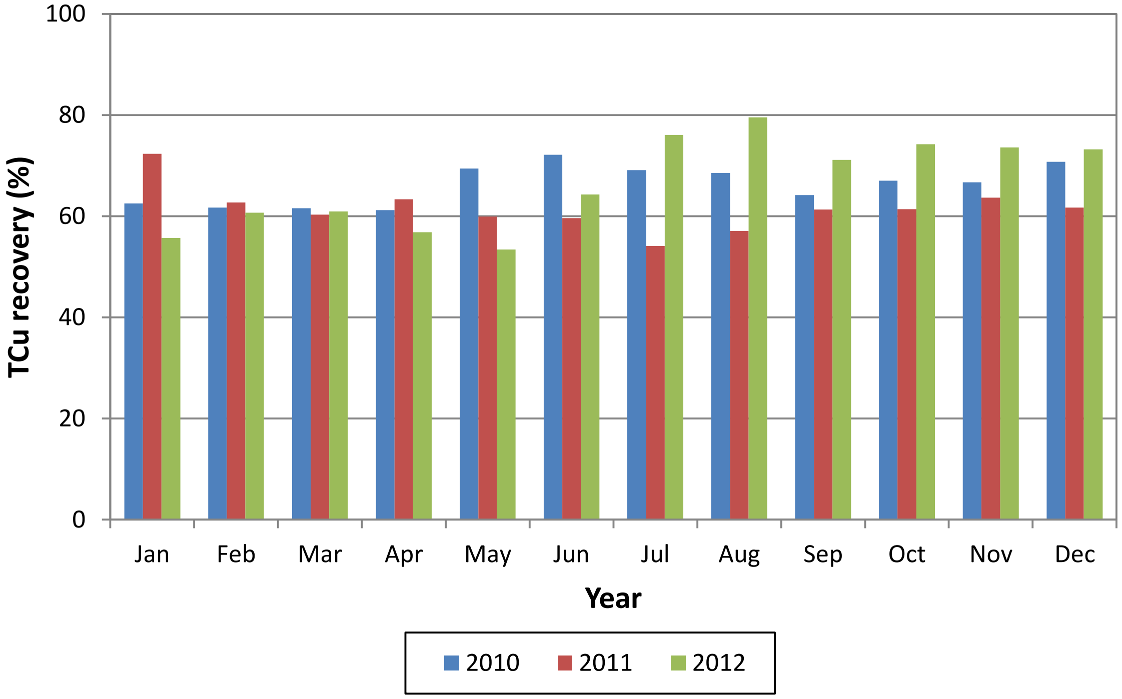
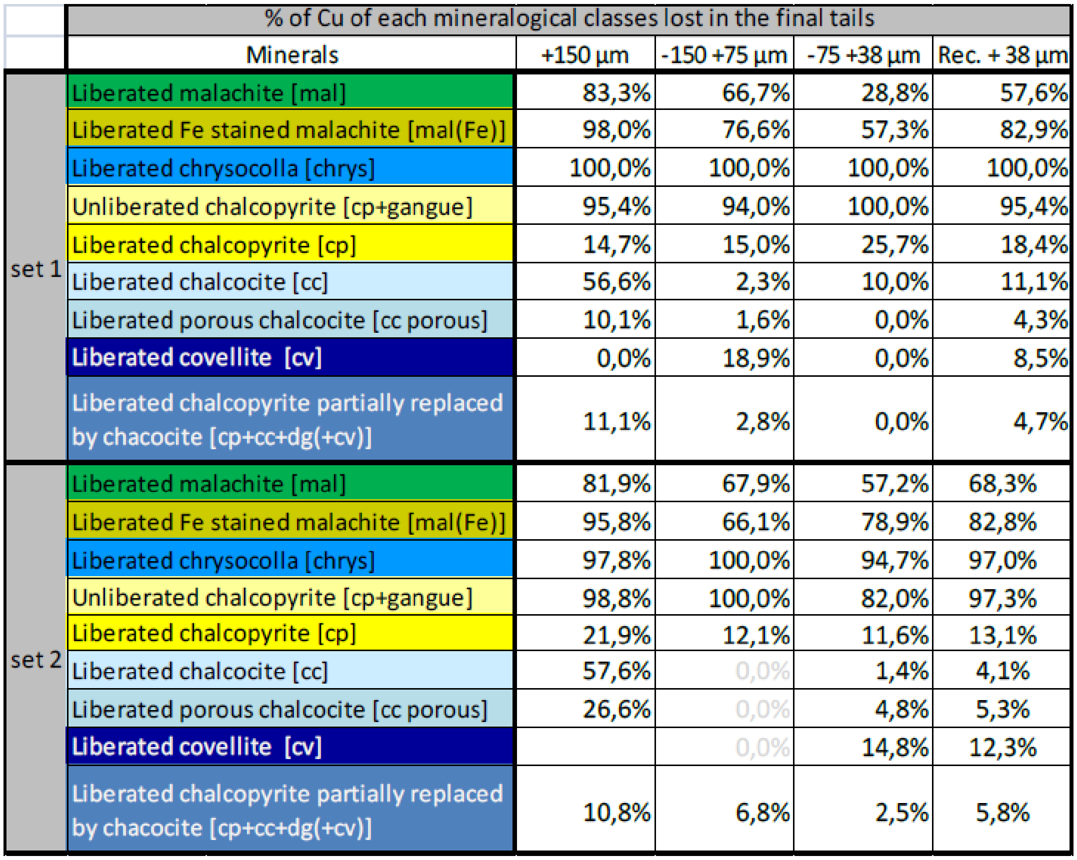


Figure 1.1-2: Mixed float recovery trends, 2010 - 2012 (Source, Kansanshi Metallurgical Accounting)

1.2. Problem statement

In treating mixed ore at Kansanshi, copper content is considered as the main ore parameter. Reagent dosage is therefore entirely dependent on throughput and the ratio of ASCu to AiCu. However, the diversity of minerals contained in the mixed ore requires a clear understanding of its mineralogy in terms of both quality and quantity as two different mixed ores with the same copper grade may vary considerably in mineralogy. It is therefore essential to understand the deportment of the copper and therefore how much of it entering the process plant is actually recoverable and to base reagent dosage on this. Initial mineralogical tests on mixed float tails have indicated that the bulk of the ASCu copper lost in the circuit is actually freely liberated malachite (Table 1.2-1). This suggests inadequate sulphidisation or poor sulphidisation techniques as malachite should be recovered quite easily if properly sulphidised.

Table 1.2-1: Kansanshi mixed float copper losses by mineralogical classes (Kottgen & Bastin, 2009)



	% of Cu of each mineralogical classes lost in the final tails				
	Minerals	+150 µm	-150 +75 µm	-75 +38 µm	Rec. + 38 µm
set 1	Liberated malachite [mal]	83,3%	66,7%	28,8%	57,6%
	Liberated Fe stained malachite [mal(Fe)]	98,0%	76,6%	57,3%	82,9%
	Liberated chrysocolla [chrys]	100,0%	100,0%	100,0%	100,0%
	Unliberated chalcopyrite [cp+gangue]	95,4%	94,0%	100,0%	95,4%
	Liberated chalcopyrite [cp]	14,7%	15,0%	25,7%	18,4%
	Liberated chalcocite [cc]	56,6%	2,3%	10,0%	11,1%
	Liberated porous chalcocite [cc porous]	10,1%	1,6%	0,0%	4,3%
	Liberated covellite [cv]	0,0%	18,9%	0,0%	8,5%
	Liberated chalcopyrite partially replaced by chacocite [cp+cc+dg(+cv)]	11,1%	2,8%	0,0%	4,7%
set 2	Liberated malachite [mal]	81,9%	67,9%	57,2%	68,3%
	Liberated Fe stained malachite [mal(Fe)]	95,8%	66,1%	78,9%	82,8%
	Liberated chrysocolla [chrys]	97,8%	100,0%	94,7%	97,0%
	Unliberated chalcopyrite [cp+gangue]	98,8%	100,0%	82,0%	97,3%
	Liberated chalcopyrite [cp]	21,9%	12,1%	11,6%	13,1%
	Liberated chalcocite [cc]	57,6%	0,0%	1,4%	4,1%
	Liberated porous chalcocite [cc porous]	26,6%	0,0%	4,8%	5,3%
	Liberated covellite [cv]		0,0%	14,8%	12,3%
	Liberated chalcopyrite partially replaced by chacocite [cp+cc+dg(+cv)]	10,8%	6,8%	2,5%	5,8%

Poor recoveries are still evident, even with the recent expansion on the circuit that added two extra CPS stages, suggesting that the reason for low recoveries is not inadequate residence time but rather a poor understanding of the chemistry of the reagents and how they interact with the mineral surfaces. Results have also shown pyrite content in the CPS concentrate was significantly higher than copper content. High pyrite content lowers the copper grade thus threatening the concentrator-smelter contract.

1.3. Scope of this work

This work focuses on evaluating the effect that reagent suites have on the flotation performance of two high grade mixed copper ores with varying quantities of oxide minerals. Batch flotation tests form the basis of the work, exploring main stream methods employed by Kansanshi Mining Plc. (controlled potential sulphidisation). Flotation performance is measured by chemical assays for copper (TCu and ASCu) and iron (Fe), coupled with mineralogical analysis by Quantitative Evaluation of Minerals by Scanning Electron Microscopy (QEMSCAN), X-Ray Fluorescence (XRF) and X-Ray Diffraction (XRD) because of the complexity arising from the variation of minerals. Pyrite depression is not included in this test work because selectivity against Fe is used as a proxy for the best reagent suite.

The results presented in the thesis are averages of duplicate and triplicate tests on which statistical analysis was done; standard error and error bars are shown on all the charts. The scope is broadly outlined in Figure 1.3-1. The green sections indicate the process, inputs and outputs; the blue sections are the fixed variables that will be considered in this work and the purple sections are the indicators that will be used to determine performance.

Microflotation studies on pure copper oxide minerals; malachite and chrysocolla are not included in the figure as they do not form the core of this work but were only undertaken to gain a fundamental understanding of the flotation process and aid in the discussion of the results obtained.

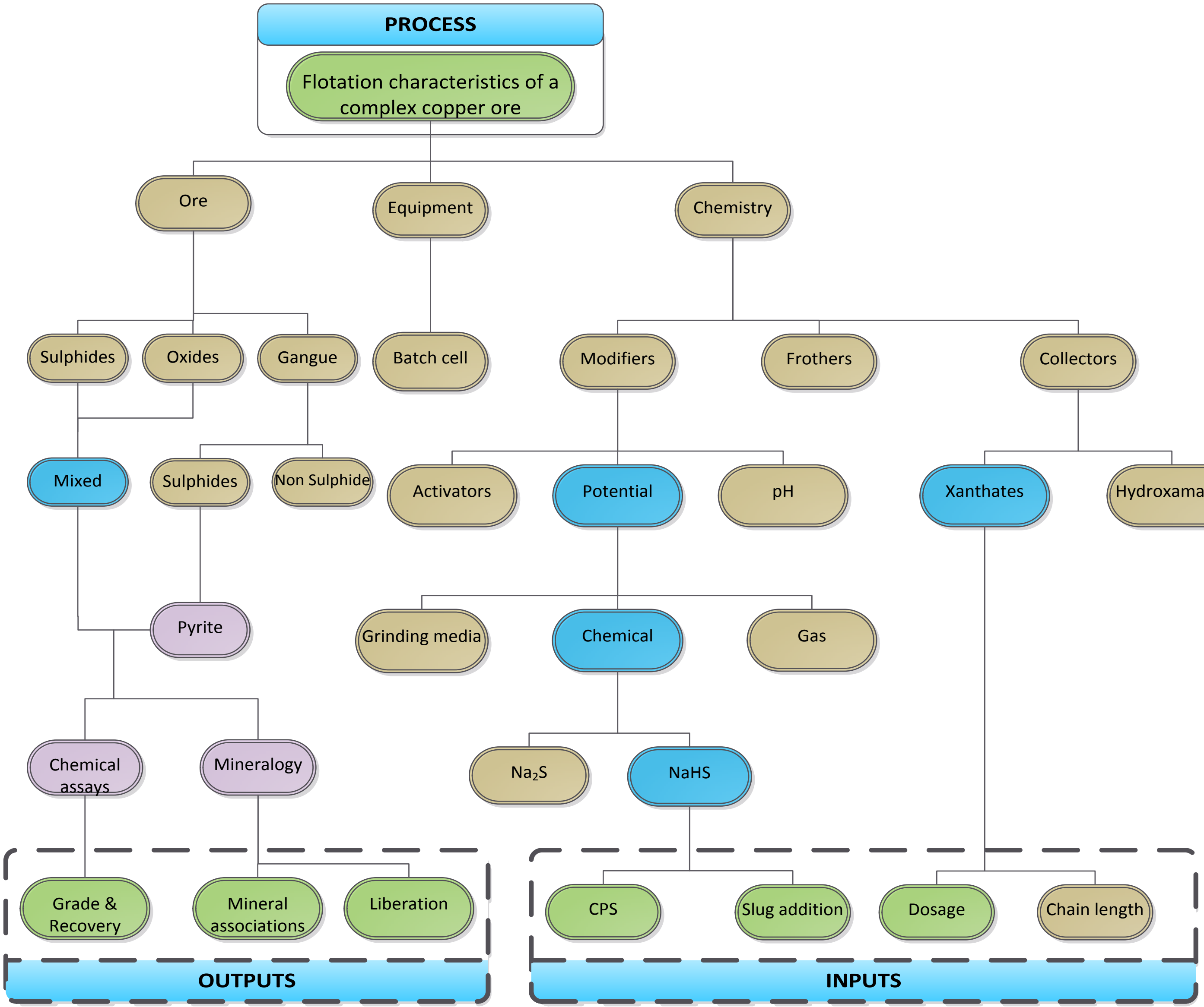


Figure 1.3-1: Schematic representation of the scope of this work

1.4. Thesis outline

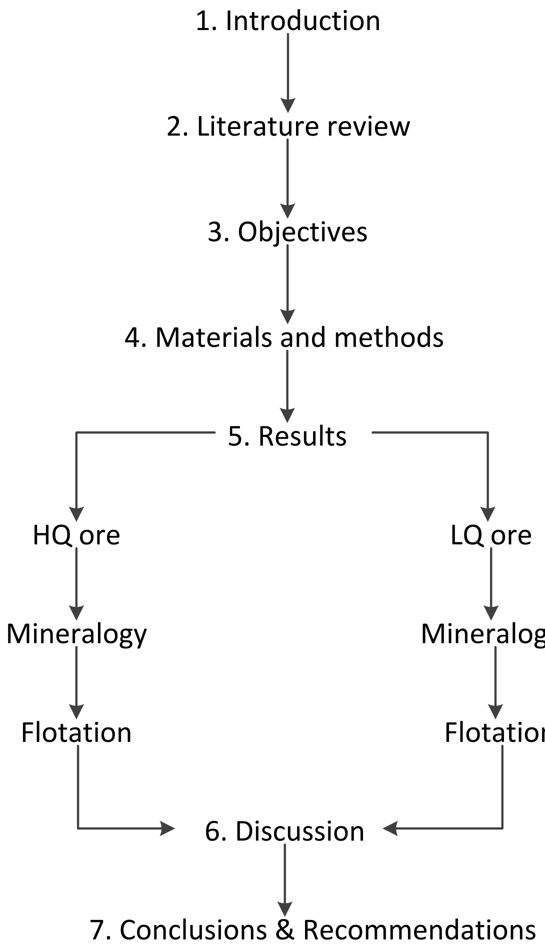


Figure 1.4-1: Thesis outline

The dissertation is divided into seven chapters as shown in Figure 1.4-1. Chapter 1 is the introduction of the study, detailing the background that necessitated it and the scope in which it is done. Chapter 2 critically reviews the literature that is relevant to the study. The objectives, key questions and hypotheses that drove the research are outlined in Chapter 3 and the materials and methods used for the experimental work are outlined in Chapter 4. Chapter 5 presents the mineralogy of the ores which justified the flotation methodology outlined in Chapter 4, forming a basis for understanding the results obtained from the test work done. The flotation results of each ore are also presented in Chapter 5 immediately after the mineralogy and a comprehensive discussion of the results and comparison to past work is given in Chapter 6. The conclusions drawn from the test work and recommendations made are outlined in Chapter 7. The last two sections of the dissertation are the bibliography, acknowledging all the previous work that is cited in this dissertation, and the appendices.

2. LITERATURE REVIEW

"The scientific man does not aim at an immediate result. He does not expect that his advanced ideas will be readily taken up. His work is like that of the planter- for the future. His duty is to lay the foundation for those who are to come, and point the way"

- Nikola Tesla

2.1. Introduction

This chapter critically synthesises the literature that is relevant to the scope outlined in Chapter 1. The Kansanshi deposit is distinguished from the Zambian copperbelt in terms of mineralogy and mineralisation prior to a discussion of the process mineralogy of the oxide minerals present and their flotation. After broadly outlining the principles of froth flotation, both sulphide and oxide mineral flotation are critically reviewed because the mixed ore in the deposit contains a wide array of copper mineral classes; sulphides, carbonates and silicates (Broughton et al., 2002) and because if well sulphidised, base metal oxides should have the same flotation behavior as the sulphides (Soto & Laskowski, 1973).

2.2. The Kansanshi deposit

Kansanshi copper mine is situated in the north western province of Zambia. Unlike the stratiform deposits common to the Zambian copperbelt, mineralisation in the Kansanshi deposit occurs in quartz-carbonate veins with the lowest exposed rock unit being the basal limestone comprising albite ($NaAlSi_3O_8$), dolomite ($CaMg(CO_3)_2$) and minor muscovite ($KAl_2(Si_3Al)O_{10}(OH,F)_2$), chlorite and dispersed sulphides (Speiser et al., 1995; Broughton et al., 2002). While the copperbelt is enriched in cobalt (Selley et al., 2005), the Kansanshi deposit has miniscule amounts of it; less than 50 ppm (Broughton et al., 2002). Figure 2.2-1 shows the location of the deposit relative to the Central African Copperbelt.

Copper oxide minerals form as a result of supergene processes (Figure 2.2-2) that occur when copper sulphide minerals are exposed to the effects of weathering (Reich et al., 2009) and encompass a series of defined assemblages that depict a variable pH, oxidizing geochemical environment known as the oxide zone in which the source rock, host rock mineralogy and iron-copper sulphide mineral abundance among other factors determine the oxide mineral formed (Chavez, 2000).

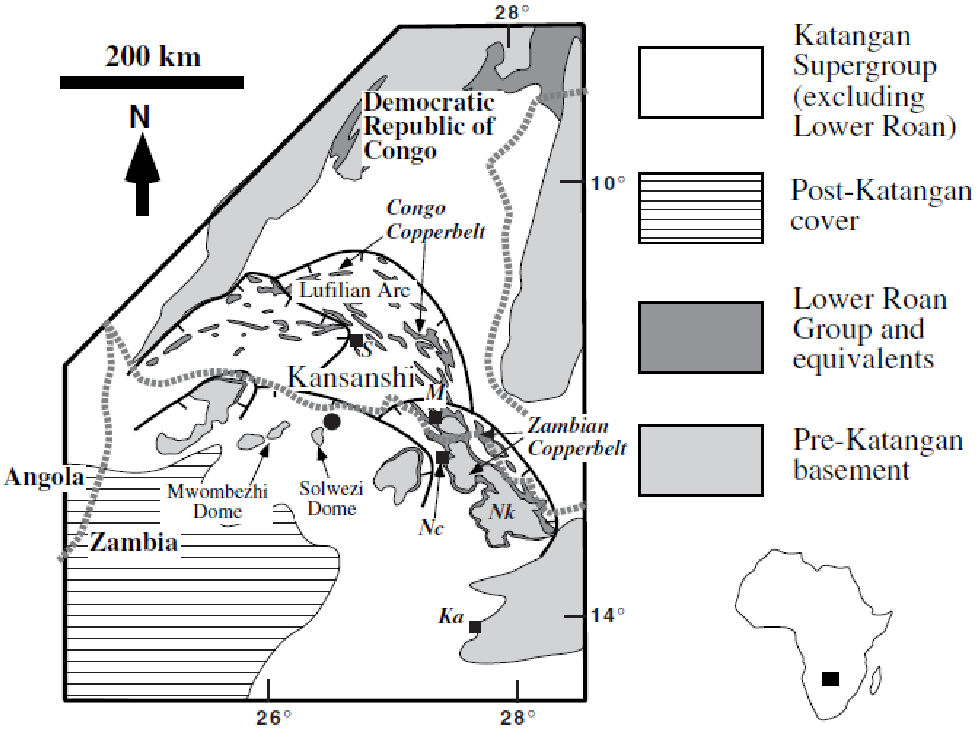


Figure 2.2-1: Location of the Kansanshi Cu (-Au) deposit relative to the Central African Copperbelt (Broughton et al., 2002)

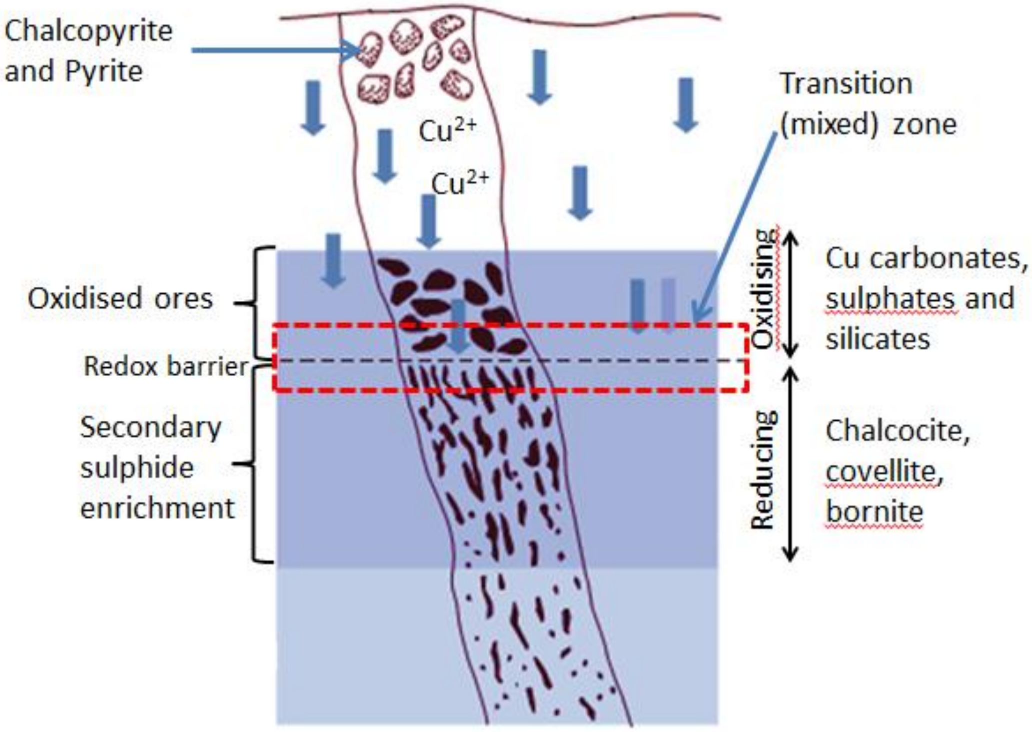


Figure 2.2-2: Supergene enrichment process (adapted from Robb, 2005)

Broughton et al., (2002) surmise that in the Kansanshi deposit, weathering has given rise to a vertically zoned profile comprising leached, refractory, oxide, mixed and hypogene sulphide mineralisation with the thickness of the various mineralisations dependent on vein abundance, quantity of sulphides to form acid and the stratigraphic level in relation to carbonate-bearing lithologies. A list of the valuable copper minerals at Kansanshi is given in Table 2.2-1.

The ore deposit contains both primary and secondary copper minerals and is therefore classed into three distinct types; sulphide, oxide and mixed. The mixed ore contains oxide, supergene oxide and hypogene sulphide copper species and the greater part of the deposit is made up of sulphide and mixed ores, with the oxide ore having a much higher grade than the two but contributing less to the resource (Broughton et al., 2002).

Table 2.2-1: Major copper bearing minerals in the Kansanshi deposit

Mineral	Type	Formula	Ideal % Cu in mineral
Chalcopyrite	Primary sulphide	$CuFeS_2$	34.6
Bornite	Secondary sulphide	Cu_5FeS_4	63.3
Digenite	Secondary sulphide	Cu_9S_5	78.1
Chalcocite	Secondary sulphide	Cu_2S	79.9
Covellite	Secondary sulphide	CuS	66.5
Cuprite	Oxide	Cu_2O	88.8
Chrysocolla	Silicate	$\sim Cu_4H_4Si_4O_{10}(OH)_8 \cdot nH_2O$	33.9
Malachite	Carbonate	$Cu_2(CO_3)(OH)_2$	57.5

Broughton et al. (2002) have identified the major gangue minerals in the deposit as dolomite, calcite, quartz, pyrite and pyrrhotite. Sulphide ore is dominated by coarse grained pyrite with the chalcopyrite set in a matrix of quartz, feldspar and muscovite mica. The mixed ore is contact zones between sulphide and oxide ore and demonstrates difficulty in liberation of chalcopyrite, chalcocite and covellite. Oxide ore is dominated by chrysocolla with lesser malachite set in a matrix of quartz, iron oxides and iron hydroxides. The veined nature of the mineralisation led to the targeted selective mining of these veins, depleting the high grades, leaving a current resource of 267 Mt (Broughton et al., 2002). First Quantum Minerals estimates that mixed ore makes up 134.6 Mt of the total reserve, at 0.75 % TCu and 0.17 % ASCu (http://www.first-quantum.com/Our-Business/operating-mines/Kansanshi/Reserves--Resources/default.aspx, Accessed 20 May, 2015).

2.3. Process Mineralogy of copper oxides

Chavez (2000) puts forward that copper oxides form either through direct precipitation resulting from supergene processes, or through the replacement of sulphide, oxide and silicate minerals. Malachite and azurite form in weathering zones where carbonate is available, but malachite exhibits a wider distribution because it is the more stable of the two (Vink, 1986). In more geochemically mature environments, near-neutral to alkaline, malachite and chrysocolla are formed with lower volumes of tenorite which progressively grades, down the weathering profile, to cuprite and native copper (Chavez, 2000).

Fuerstenau & Fuerstenau (1982) put forward that the similarity of oxide minerals to many non-metallic minerals requires a detailed appreciation of their surface chemistry in order to optimise selectivity during flotation. In the last couple of decades, research has integrated and emphasised the need for an understanding of the mineralogy of such ores in their treatment (O'meara, 1961; Lee et al., 1998; Lee et al., 2009) as this gives an indication of mineral associations, inclusions, liberation as well as gangue mineralogy, all of which have profound effects on flotation. When oxide copper minerals are hosted in a matrix of acid consuming gangue such as calcite, leaching becomes uneconomic and froth flotation is the preferred mode of treatment. Where the oxides are chemically well defined as in the case of malachite, flotation has been shown to be relatively easy (Phetla & Muzenda, 2010; Lee et al., 1998; Lee et al., 2009; Hope et al., 2010). This is not the case with chrysocolla however because of its amorphous nature.

In investigating multiple samples of chrysocolla using Powder X-Ray Diffraction (PXRD) Crane et al. (2001) observed broad reflections, indicating various elemental composition within the mineral. Different studies undertaken on the flotation of chrysocolla quote varying copper contents for the "pure" mineral; 20.8 % (Peterson et al., 1965), 24.37 % (Hope et al., 2012a) and 34.4 % (Hope et al., 2012b). This can be explained by the presence of small amounts of other metals such as Al^{3+} substituting the Cu^{2+} or by the extent of dehydration (Crane et al., 2001).

"*Mineralogical characterisation of the ore is a prerequisite to any flotation testing. Such information will allow one to estimate the amount of copper associated with well-defined oxide copper minerals and with species such as Cu-bearing goethite which are not amenable to flotation, but report as Acid Soluble Cu in chemical assays* …" (Lee et al., 1998).

Becker et al. (2009) used Quantitative Evaluation of Minerals by scanning Electron microscopy (QEMSCAN), which gives data such as bulk mineralogy and particle mineralogical analysis, to quantify the mineral species present in a Merensky ore feed and concentrates in order to identify naturally floatable gangue and established that despite the presence of talc in the ore, concentrate dilution was mostly due to orthopyroxene particles that exhibited preferential surface association to the talc. The Mineral Liberation Analyser (MLA) uses Back Scattered Electron (BSE) signals and a Scanning Electron Microscope (SEM) to identify minerals present in a sample (Gu, 2003) and has been used to demonstrate the application of mineralogy in the mill to melt approach to modelling in order to determine the distribution of nickel in a nickel-copper ore and the distribution of magnesium oxides that are detrimental to smelter operations (Evans et al., 2011).

2.4. Principles of flotation

Froth flotation is a selective physico-chemical process that utilizes the difference in properties of valuable minerals and gangue (Wills & Napier-Munn, 2006). Fuerstenau et al. (1985) surmised that the process involves the aggregation of air bubbles and mineral particles in an aqueous medium with subsequent levitation of the aggregates to the surface and transfer to a froth phase, Figure 2.4-1. Bubble attachment in flotation is dependent on hydrophobicity of the mineral in question and this property is accentuated by the addition of reagents, viz. collectors, frothers and modifiers.

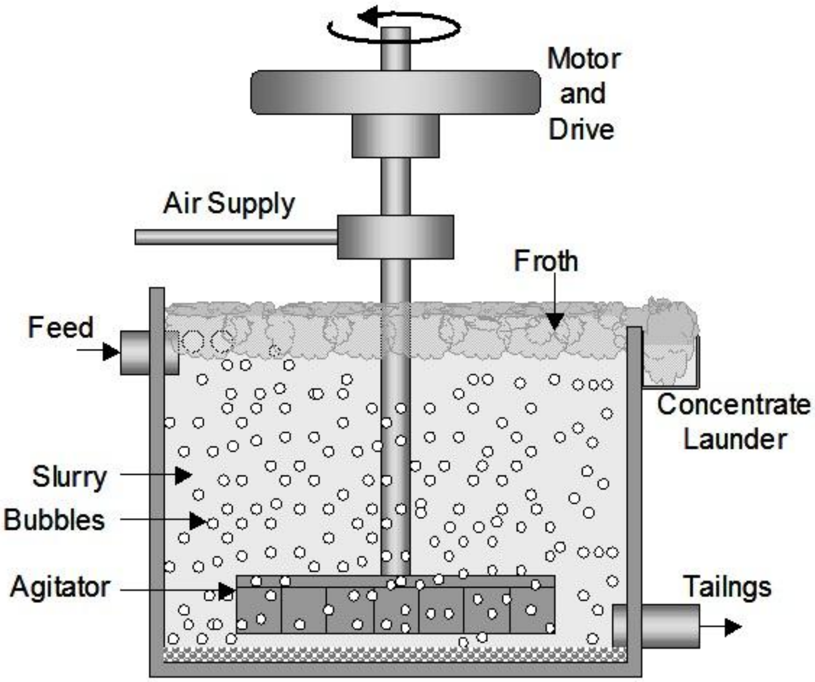


Figure 2.4-1: Elements of a conventional flotation cell (Grewal, n.d.)

2.4.1. Collectors

Collectors are heterogeneous compounds that contain an active inorganic group coupled with a hydrocarbon chain (Fuerstenau et al., 1985). They make selected minerals hydrophobic by adsorption of molecules or ions onto the mineral surface, reducing the stability of the hydrated layer separating the mineral surface from the air bubble (Wills & Napier-Munn, 2006). Collector classification depends on whether they exist as cations, anions or molecular species in solution (Fuerstenau et al., 1985).

Cationic collectors are very sensitive to the pH of the medium, being most active in slightly acid solutions and inactive in strongly alkaline and acid media and are used for floating oxides, carbonates and silicates (Wills & Napier-Munn, 2006). Amine is the only cationic collector that is used in industry (Fuerstenau et al., 1985) and ionizes in aqueous solution by protonation as follows;

$$\mathrm{RNH_2(aq) + H_2O \leftrightarrow RNH_3^+ + OH^-} \qquad \text{(Eqn. 2-1)}$$

Anionic collectors may either be oxyhydryls or sulphydryls (Wills & Napier-Munn, 2006). Oxyhydryl collectors have organic and sulpho-acid anions as their polar groups and their cation does not take part in the reagent-mineral interaction, these include fatty acids, sulphonates, alkyl sulphates and chelating agents (Fuerstenau et al., 1985). Sulphydryl collectors are the most widely used collectors and most prominent among them are xanthates and dithiophosphates (Wills & Napier-Munn, 2006). Collectors are normally used in small amounts in order to form a monolayer on the mineral surface. Increased dosages increase operation costs and may lead to reduced mineral recovery when increased dosage represents over-dosing, resulting in the formation of collector multi layers, reducing the proportion of hydrocarbon radicals oriented into the bulk of the solution (Wills & Napier-Munn, 2006). Structural formulae of common anionic collectors are given in Figure 2.4-2 where R represents an alkyl group and Me^+ is a base metal such as sodium (Na) or Potassium (K).

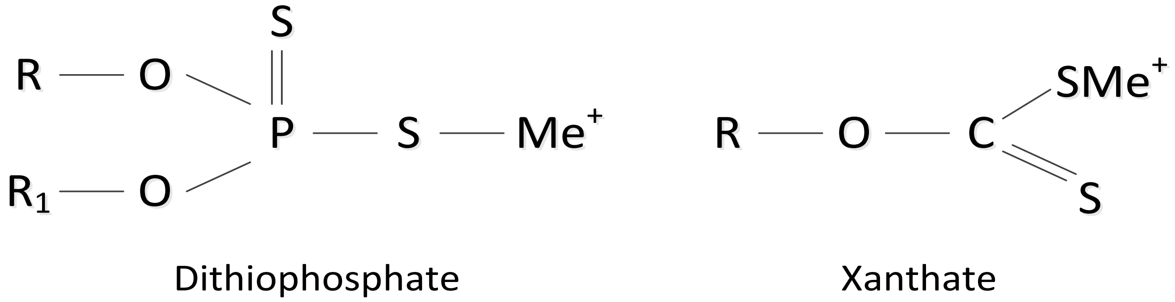


Figure 2.4-2: Examples of typical sulphydryl collectors (Bulatovic, 2007)

2.4.2. Frothers

Frothers are heteropolar surface active compounds, containing hydroxyl (OH), carboxyl (COOH), carbonyl (C=O) or sulphoxy (SO_2OH) group, whose uneven distribution of polar and nonpolar groups allows for their preferential orientation at the water-air interface (Laskowski, 1993; Bulatovic, 2007). Frothers stabilise bubble formation in the pulp phase, create a reasonably stable froth phase to allow selective drainage of entrained gangue, and increase flotation kinetics (Harris, 1982; Wills & Napier-Munn, 2006).

The acid and alcohols are the most soluble frothers while the alcohols are generally slightly soluble (Laskowski, 1993). Ekmekçi et al., (2003) investigated the influence of frother type on the flotation of chromite in UG2 ore and determined that for the same dosage of five different frothers, there were significant differences in mass, water and mineral recovery possibly due to inherent properties of the frothers such as chemical structure, solubility, surface tension and elasticity.

A number of studies have been done on the effects that frother dosage has on flotation performance. Cho & Laskowski, (2002) showed that frother concentration affected both bubble size and retention time. Bubble size and froth stability both increase with increasing frother dosage (Aldrich & Feng, 2000). Laskowski (1993) reviewed literature on frothers and deduced that good frothers have branched hydro-carbon chains that allow for the formation of condensed films. He also concluded that frother selectivity is associated with the volume of water recovered but is less significant in flotation columns.

2.4.3. Modifiers

Modifiers control the interaction of collectors between distinct minerals by magnifying or reducing the hydrophobic effect of the collector and are classified as activators, depressants or pH modifiers (Wills & Napier-Munn, 2006; Bulatovic, 2007). Bulatovic (2007) also surmised that modifiers increase or prevent the adsorption of collector on the mineral surface by changing the chemical composition of the mineral surface (activators), act as depressants by removing the collector coated on the mineral surface (depressants), or change the pulp pH on which mineral-collector interaction is dependent (pH modifiers).

Depressants ensure that no minerals other than the valuable ones float. Examples are cyanide, which depresses copper, zinc, iron and silver sulphides, and carbonates which depress salt type minerals with fatty acids (Lovell, 1982).

2.5. Flotation of sulphide minerals

The flotation of sulphide minerals is well documented and relatively straight forward when compared to non-sulphide minerals. These minerals float easily due to their natural hydrophobicity, resulting from their tendency to not form hydrogen bonds (Fuerstenau et al., 1985).

The collectorless flotation of sulphide minerals has been extensively reviewed by Hayes et al. (1987) who cited the collectorless flotation of sphalerite in the Potter process and the flotation of molybdenite in the order of 1 kt/day in many plants, without a collector. The review credited early works in collectorless flotation to Ravitz & Porter (1933) and Herd & Ure (1941) who studied the phenomenon on galena. The galena was washed with a salt and then rinsed in distilled water prior to flotation. The studies indicated that oxygen, or an oxidizing potential, is required for collectorless flotation.

Effective flotation of sulphide minerals is done in the presence of oxygen with short chain sulphydryl collectors (Fuerstenau et al., 1985). Different mechanisms have been proposed to describe the interaction between xanthates and sulphide minerals. Woods et al., (2000) used Surface Enhanced Raman Scattering (SERS) spectroscopy to prove that the adsorption of ethyl, isopropyl, isobutyl and isoamyl xanthates on sulphide mineral surfaces occurs by charge transfer chemisorption.

Heyes & Trahar (1979) investigated the effect of oxidation-reduction on the flotation of chalcocite and cuprite. In this study, they compared the flotation of chalcocite and cuprite to chalcopyrite in the absence of collector and found that cuprite recovery increased when average particle size was lower than 60 µm, probably due to entrainment. Chalcopyrite recovery was over 90 % in all size fractions and chalcocite displayed behavior intermediate to cuprite and chalcopyrite, suggesting a small degree of true flotation.

Wiese, (2009) compared the performance of four xanthates of varying chain length in the flotation of copper and nickel and found that for the same collector and guar depressant dosage, different grade-recovery trends emerged (Figure 2.5-1). The ethyl xanthates gave better results for both copper and nickel. This phenomenon has been attributed to the fact that longer hydrocarbon chains impart greater hydrophobicity on the mineral surface by controlling the interaction between collector and water molecules (Fuerstenau et al., 1985).

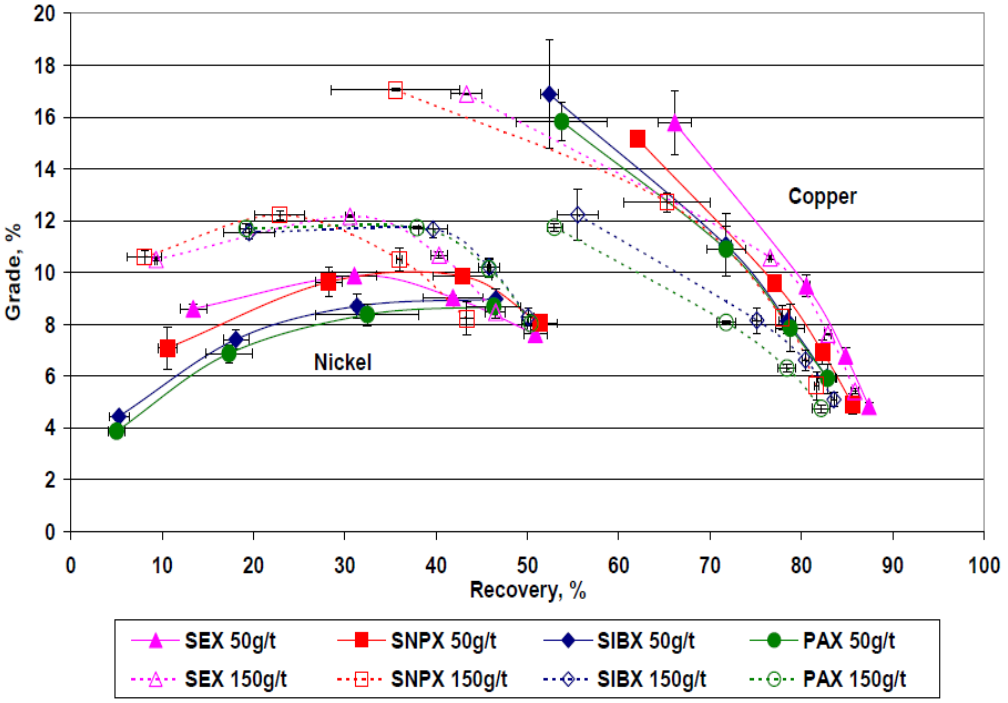


Figure 2.5-1: Copper and nickel grade-recovery curves for different xanthate chain lengths at dosages of 50 g/t and 150 g/t in the presence of guar (Wiese, 2009)

Several non-xanthate collectors have been proposed for the flotation of sulphide minerals but only a few have been commercialized, among them, thionocarbamates, dialkyl dithiophosphinates, alcoxycarbonyl, thioureas, dialkyl and diaryl monothiophosphates allyl thionocarbamates and dialkyl trithiocarbonates (Nagaraj, 1994). Fairthorne et al. (1997) compared the performance of O-isopropyl-N-ethyl thionocarbamate (IPETC), O-isobutyl-N-ethoxycarbonyl thionocarbamate (IBECTC) and ethoxycarbonyl thiourea (BECTU) in the flotation of chalcopyrite, pyrite and galena and found that all three collectors performed better in the flotation of chalcopyrite and were more selective towards chalcopyrite than pyrite (Figure 2.5-2). BECTU gave the best results overall, and this was attributed to metal ion specificity, a high acidity constant (pK_a) value and a lower solubility.

In a review of traditional and new sulphide flotation reagents, Klimpel (1994) compared xanthates, dithiophosphates and thionocarbamates and stated that:

i. Xanthate flotation is still the industry standard as the chemicals are cheap, easily manufactured and have a long shelf life in solid form,
ii. Dithiophosphates are weaker collectors than xanthates and show slower kinetics, but are more selective against pyrite at high pH, and

iii. Thionocarbamates are the most expensive of the three but are also the strongest and most selective.

Nagaraj (1994) attributes the rigidity of the industry in using xanthates, rather than alternative or new reagents, in the flotation of sulphide minerals, and the lack of successful scale up when such reagents are introduced, to such factors as a poor understanding of the reagent chemistry, inadequate testing and resistance to change within the industry.

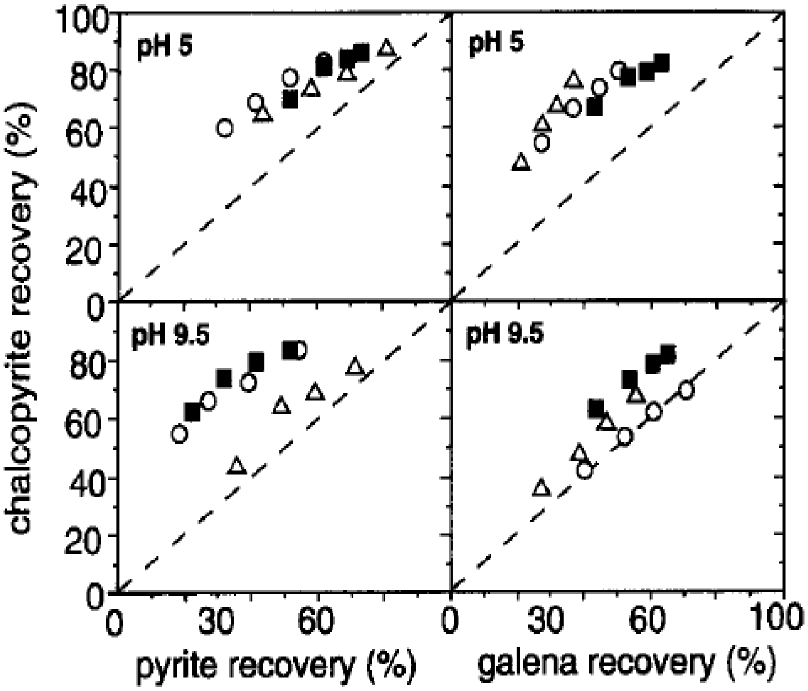


Figure 2.5-2: Flotation recovery of chalcopyrite as a function of the recovery of (left) pyrite and (right) galena at pH values of 5 and 9.5 using 2x10^{-5} mol dm^{-3} of (o) IPETC (Δ) IBECTC and (▪)BECTU. The conditioning gas is nitrogen and the total mineral concentration is 4 g dm^{-3} (Fairthorne et al., 1997)

2.6. Flotation of non-sulphide/oxide minerals

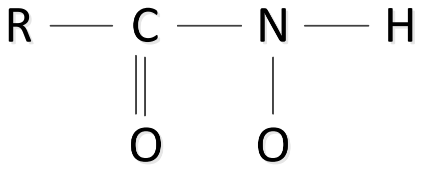
Non-sulphide minerals can be classified into three distinct groups; slightly soluble such as carbonates and halides, insoluble such as silicates and naturally floating, such as talc (Aplan, 1994). These minerals do not respond well to flotation methods used traditionally for sulphide minerals. Aplan (1994) gives a concise list of reagents that are capable of non-sulphide mineral flotation; fatty acids, sodium alkyl amines, thionocarbamates, xanthates and dithiophosphates. Although thiol collectors are used along with sulphidising agents, flotation of oxides of Ni, Cu(II) and Fe(III) by long chain xanthates has been studied and it has been shown that these oxides can be floated based on an electrostatic mechanism, where recovery decreased as pH increased (Rao & Finch 2003). Aplan & Fuerstenau (1984) have shown that while malachite is susceptible to xanthate flotation when higher xanthate homologs (hexyl, dodecyl) are used, chrysocolla is not. Anglesite ($PbSO_4$), cerrusite ($PbCO_3$) and malachite also respond well to flotation with long chain fatty acids due to the insoluble nature of heavy metal soaps (Fuerstenau et al., 1985).

There are two main methods by which non sulphide ores are treated through flotation:

i. The use of alkyl hydroxamate collectors (Yordan et al., 1994; Lee et al., 1998; Lee et al., 2009)
ii. Sulphidisation followed by xanthate flotation (Jones & Woodcock, 1979; Lee et al., 1998)

2.6.1. Hydroxamates

Hydroxamates (Figure 2.6-1), are chelating agents and as such are capable of bonding to a metal in two or more points of their molecule. Chelating reagents such as oximes have been shown to recover malachite and azurite over a narrow acidic pH range in the order of 4 to 4.8 (De Witt & von Batchelder, 1939). Chelating agents however have the inherent problem of imparting a limited degree of hydrophobicity on the metal. Marabini et al., (2007) in work carried out over thirty years showed that combined use of chelates and fuel oil can be used to achieve a double adsorption mechanism in which *"…the chelating reagent reacts with the cation of the mineral surface to form an insoluble metal ion chelate, to which the hydrocarbon chains of the oil, which are necessary to ensure hydrophobicity and hence flotation, adhere by physical adsorption."*



$$\mathrm{R - \underset{\overset{\|}{O}}{C} - \underset{\overset{|}{O}}{N} - H}$$

Figure 2.6-1: Chemical structure of a typical hydroxamate

The use of hydroxamates as collectors has been studied extensively. Peterson et al. (1965) demonstrated that potassium octyl hydroxamate could be used to successfully float chrysocolla and proposed that collection was due to the formation of an insoluble complex between surface copper ions and chelating agents. Lee et al. (2009) established that hydroxamate successfully recovered malachite and minor azurite when used in conjunction with PAX on a mixed copper oxide and sulphide blend, without any adverse effects on the recovery of sulphide copper minerals.

SERS has been used to investigate the interaction of n-octanohydroxamate and acectohydoxamate with malachite and azurite surfaces and results indicated that malachite flotation concentrates and malachite slices exposed to 10^{-3} M hydroxamate display spectra consistent with the formation of bulk cupric hydroxamate on the surface of the mineral (Hope et al., 2010).

In a study on the practical aspects of oxide copper recovery with alkyl hydroxamates, the following reasons were among those put forward by Lee et al. (1998) on why their large scale usage is not fully exploited:

i. It is generally assumed that sulphidisation-flotation is the preferred method.
ii. The use of alkyl hydroxamates has been limited to academic interest.
iii. Oxide copper is perhaps not recoverable by flotation.
iv. Insufficient efforts have been made to demonstrate the efficacy and cost benefits of using alkyl hydroxamates in a plant.

In the same study however, based on six different case studies, Lee et al. (1998) showed that alkyl hydroxamate can effectively recover “well defined” oxide copper minerals and are a simpler and more attractive substitute to sulphidisation-flotation.

Yordan et al. (1994) compared the performance of a hydroxamate against fatty acids in the flotation of anatase (TiO2) from run-of-mine kaolin clays and found that commercially available hydroxamate successfully removed anatase from clay at lower dosages. They attributed the superior performance of the hydroxamate to the fact that it did not require an activator and that it also behaved as a coagulant for the very fine particles.

2.6.2. Sulphidisation

Sulphidisation as a method of concentrating oxidised ores was patented in 1905 (Schwarz, 1905) and has since been developed and adapted as the preferred method for the flotation of oxide and mixed copper ores. The reagents used in this technique of activating base metal oxides are alkali sulphides (Soto & Laskowski, 1973) such as sodium sulphide (Na_2S) and sodium hydrosulphide (NaHS).

Slug sulphidisation, in which a lump amount of sulphidising agent is added to the pulp, has been investigated (Jones & Woodcock, 1979; Quast et al., 2005) and is generally not preferred because the lack of pulp potential control leads to poor mineral recoveries from either under-sulphidising or depression of minerals due to over-sulphidising. Controlled Potential Sulphidisation (CPS) has been applied in numerous operations around the world; Zambia

(Chabuka & Witika, 2001; Wills & Napier-Munn, 2006), the Democratic Republic of Congo (Ferron & Manu, 1994) and China (Wenbin, 1993).

When sulphidising agents are added to a pulp, dissociation of the agent occurs to produce anionic species containing sulphur (S). Sodium sulphide dissociates and then hydrolyses, releasing OH^-, S^{2-} and HS^- ions, according to equations 2-2 to 2-5, and these can react with and modify the oxide mineral surface (Wills & Napier-Munn, 2006);

$$\mathrm{Na_2S + 2H_2O \leftrightarrow 2NaOH + H_2S} \qquad \text{(Eqn. 2-2)}$$

$$\mathrm{NaOH \leftrightarrow Na^+ + OH^-} \qquad \text{(Eqn. 2-3)}$$

$$\mathrm{H_2S \leftrightarrow H^+ + HS^-} \qquad \text{(Eqn. 2-4)}$$

$$\mathrm{HS^- \leftrightarrow H^+ + S^{2-}} \qquad \text{(Eqn. 2-5)}$$

It is generally accepted that the HS^- ion is the active sulphidising species in the pulp (Clark et al., 2000); the stable species is dependent on the pulp E_h and pH, (Figure 2.6-2).

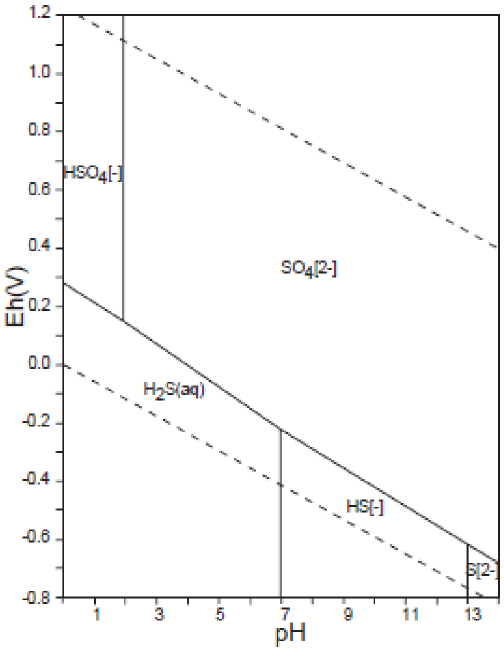


Figure 2.6-2: Sulphur Eh-pH diagram, Hatches/Flask-AQ database (adapted from Takeno, 2005)

X-ray Photoelectron Spectroscopy (XPS) has been used by Quast et al. (2005) to show mechanisms of sulphidisation. Results suggested that in the case of sodium sulphide, the HS^- ion is adsorbed on the mineral surface, whereas for polysulfide, it is the S_x^{2-} ion that is adsorbed. They proposed the following mechanism;

$$\mathrm{CuCO_3 \cdot Cu(OH)_2 + S_x^{2-} \rightarrow CuS_x + Cu^{2+} + 2OH + CO_3^-} \quad \text{(Eqn. 2-6)}$$

"Unlike elemental sulphur, polysulphide ions are held strongly to the surface by polar covalent bonds according to the following mechanism" (Quast et al., 2005);

$$\mathrm{2Cu^+ + S_x^{2-} = Cu - S_x - Cu} \quad \text{(Eqn. 2-7)}$$

In floating mixed copper ores, it is not possible to simultaneously float sulphide and oxide minerals using CPS as this would results in the depression of the sulphide minerals (Lee et al., 2009). Trahar (1984) observed that in the first minute of flotation at pH 8, at potentials more negative than 200 mV (E_{Pt}), recovery of chalcopyrite fell to less than 90 %. Guo & Yen (2003) however observed that chalcopyrite flotation was optimum within the potential range of -200 to 190 mV (E_{Pt}) using $7\text{x}10^{-5}$ M PAX, with a slight increase in the range to 205 mV at a higher PAX concentration of $7\text{x}10^{-4}$ M. In practice, sulphide minerals are floated first (Jones & Woodcock, 1979; Wills & Napier-Munn, 2006) and then the pulp potential changed by the addition of a sulphidising agent in order to float the oxide minerals.

2.6.3. Factors affecting CPS-Flotation

i. Pulp potential

A sulphide Ion Selective Electrode (ISE) has been adapted as the instrument of choice for the measurement of pulp potential in CPS. Jones & Woodcock (1979) illustrated that a sulphide ISE could be used to control the pulp potential and therefore optimize the recovery of oxide lead and permitted for reaction to variations in flotation feed. The ISE measures potential change due to ion transfer (E_S) rather than charge or electron transfer (E_h).

Nagaraj & Gorken (1991) tested the limitation of an ISE by studying its response to three representative non-hydrosulphide depressants against NaHS. They found that the ISE responded only to hydrosulphide and ISE potential did not change much even at high dosages of the non-hydrosulphide depressants- which are good depressants of copper. They concluded that this may be considered as a limitation of the ISE in systems that do not contain sulphide ions. The ISE was also insensitive to aeration of pulps in the absence of sulphide ions and to the addition of depressants such at sodium metabisulphite and sulphur dioxide.

Jones & Woodcock (1979) determined an optimum potential of -600 mV for the flotation of lead oxide and suggested that the poor recoveries observed at potentials more positive than this were either due to insufficient sulphidising time or because anglesite could not be properly sulphidised at low S^{2-} concentrations. Poor recoveries were also obtained at -700 mV, possibly due to mineral over-sulphidisation. Oxide copper recovery by CPS is optimum at potentials between – 400 and -650 mV (Nagaraj & Gorken, 1991; Ferron & Manu, 1994). CPS flotation tests on three distinct copper ores (Canadian, Indian and South East Asian) using NaHS indicated that optimum flotation occurred in the range of -400 to -600 mV and that sulphide and sulphidised oxide depression occurred in the range of -600 to -650 mV, but depended largely on the copper mineralogy (Nagaraj & Gorken, 1991).

Ferron & Manu (1994) confirmed in their work on an oxide ore from Gecamines (Democratic Republic of Congo) that pulp potential played an important role in recovery of copper oxide minerals. For their ore, optimum recovery was attained at potentials between -300 and -320 mV, and -380 mV for high silicate addition. Quast et al. (2005) compared the performance of calcium polysulphide in recovering malachite at three different potentials: -400 mV, -500 mV and -600 mV. The recoveries attained for the various potentials were 98.9 %, 99.3 % and 10.9 % respectively, indicating an optimum potential of -500 mV.

ii. Sulphidiser

Kongolo et al. (2003) compared the performance of NaHS against ammonium sulphide ($(NH_4)_2S$) in floating oxide copper-cobalt ores and determined that while $(NH_4)_2S$ gave higher metal recoveries and that the grades were lower than 8 % for copper and 3.5 % for copper cobalt. Concentrate grades when NaHS was used were much higher; greater than 12 % Cu and greater than 6 % Co.

In the 1970s the Dongchuan Copper Mining Bureau tested calcium sulphide (CaS) as an alternative sulphidising agent due to shortage of Na_2S. The CaS was added to the mill so that sulphidisation occurred during grinding and subsequent conditioning and proved to be successful, with oxide copper recoveries up to 60.2 % (Wenbin, 1993). Calcium polysulfide (CaS_x) has also been investigated as an alternative sulphidising agent and was reported to give faster kinetics, requiring only two CPS stages to reach 98.8 % copper recovery at -500 mV compared to 96.5 % for Na_2S at the same potential (Quast et al., 2005).

iii. Sulphidising/Conditioning time

In work done by Jones & Woodcock (1979), CPS flotation of lead oxide over a 1 to 5 minute range indicated minimal differences in the grade and recovery, with an apparent optimum sulphidising time of 3 minutes. Thus recovery is independent of sulphidising time provided that the optimum E_s is used.

iv. Collector type

Ferron & Manu (1994) compared the performance of potassium amyl xanthate (PAX) and sodium normal butyl xanthate (SNBX) during CPS and found that SNBX performed better than PAX. Although the general trend is to use xanthates with CPS, other collectors have been used in studies on sulphidisation; methyl isobutyl carbinol (MIBC) in conjunction with PAX (Jones & Woodcock, 1979) and dithiophosphate, thionocarbamate, dithionophosphinate and ethoxycarbonyl alkyl thiourea by Nagaraj & Gorken (1991).

v. Effect of pH

Wills & Napier-Munn (2006) explain that the pH of a pulp determines the amount of sulphidiser required as it drives the direction of equations 2-4 and 2-5 (c.f. section 2.6.2). High pH values will drive the reaction forward, producing more HS^- and S^{2-} ions. Banza & Kongolo (2001) achieved 94.8 % and 90.1 % copper and cobalt recoveries respectively at a natural pH of 9.5 treating a silicated oxide ore while Ziyadanogullari & Aydin (2005) reported up to 100 % oxidised copper recovery at a pH of 8.7 after sulphidisation. At a carefully controlled pH of 9.5, Herrera-Urbina et al. (1999) also showed improved recovery of sulphide activated cerrusite.

vi. Effect of oxygen

Soto & Laskowski (1973) established that the E_h of sulphidised pulp increased drastically when it was oxidised with air, demonstrating the consumption of sulphide ions by oxygen. This observation was also made by Ferron & Manu (1994) who noted a more rapid potential (E_S) decay (from negative to positive) when sulphidisation was carried out with an open air valve compared to a closed one. The presence of excess oxygen in the pulp can also lead to the formation of sulphate ions which, despite being capable of strong bonds with xanthates, do not form hydrophobic surfaces due to their solubility in aqueous solutions (Wills & Napier-Munn, 2006). The use of nitrogen to eliminate oxygen during sulphidisation has been shown to improve the formation of sulphide surfaces and reduce the sulphidiser consumption (Clark et al., 2000).

2.7. The role of pulp potential in sulphide mineral depression

Stauter (1975) patented the depression of pyrite in coal using sodium sulphide. Bulut et al. (2011) demonstrated that sodium metabisulphite ($Na_2S_2O_5$) depresses pyrite, with $Na_2S_2O_5$ ensuring better selectivity against pyrite compared to starch. Potential control using NaHS and hydrogen peroxide has also been used to depress pyrite at potentials more negative than -200 mV (Figure 2.7-1) (Göktepe, 2010).

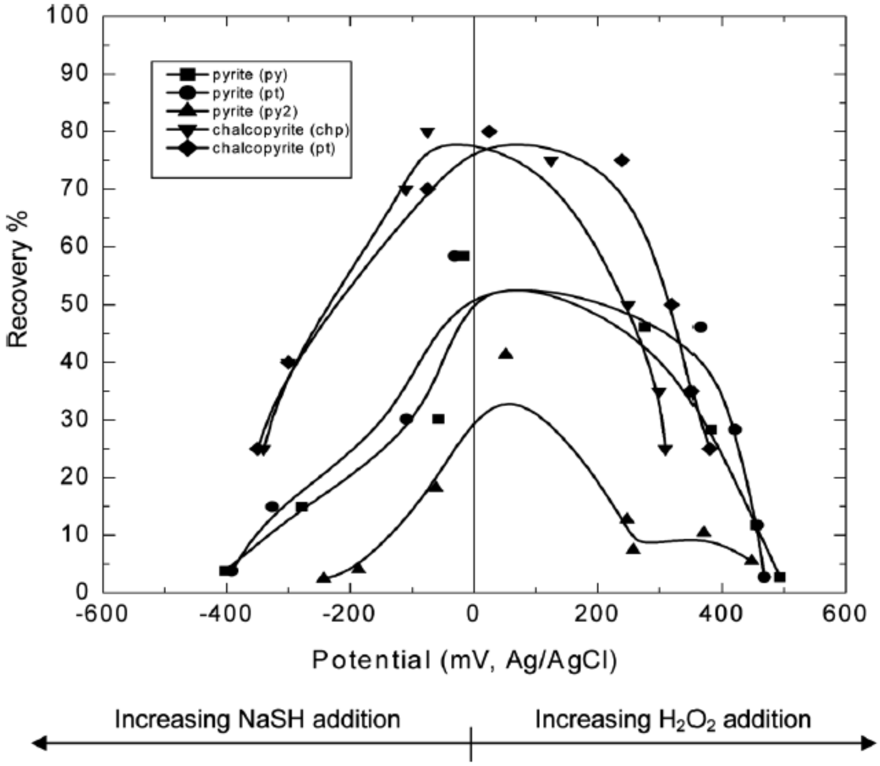


Figure 2.7-1: Potential versus recovery of chalcopyrite and pyrite in flotation, (chp) chalcopyrite electrode, (pt) platinum electrode, and (py) pyrite electrode (Göktepe, 2010).

Pyrite depression is essential in the flotation of copper sulphide minerals where it is considered gangue as it dilutes concentrate grades. Although pH control is one of the most widely used methods of pyrite depression (Janetski et al., 1977; Göktepe, 2002; Fuerstenau et al., 1985), the same alkali sulphides that are used for activation of base metal oxide minerals are well known to be sulphide mineral depressants (Soto & Laskowski, 1973).

Janetski et al. (1977) hypothesized that the depression of pyrite by sulphide was due to the creation of a mixed potential which is cathodic to the xanthate/dixanthogen potential, resulting in lack of dixanthogen formation on the mineral surface. Göktepe (2002) also attributes pyrite depression at high pH to the rapid decomposition of collector form preventing the formation of dixanthogen; *"… Under these conditions the mixed potential system becomes one of pyrite oxidation and oxygen reduction, xanthate is not oxidised, and the (mineral) surface remains hydrophilic."*

2.8. Summary of literature

It is very important to understand the mineralogy of the ore being floated in assessing the floatability of ores. Lee et al. (2009) and Phetla & Muzenda (2010) have both emphasized the need for mineralogical analysis in this respect. Lastra (2007) illustrates the need for quantitative rather than descriptive mineralogy in a case study done on a copper ore that was thought to comprise mostly chalcopyrite. In the study, mineralogy and liberation analysis proved that the bulk of the copper existed as secondary minerals, the chalcopyrite was poorly liberated and the pyrite that was diluting the concentrate grade existed as freely liberated and could therefore easily be rejected.

The recovery of non-sulphide minerals through flotation is not limited to copper. Oxidised lead (Jones & Woodcock, 1979) and cobalt (Kongolo et al., 2003; Bell, 2011) have been successfully floated using sulphidisation. Hydroxamates and fatty acids have also been shown to be capable of floating kaolin (Yordan et al., 1994). In the flotation of copper oxides, hydroxamates are generally not preferred due to their high cost (Lee et al., 1998; Aplan, 1994) and their inherent lack of long hydrocarbon chains which limits the aerophilic properties they can impart for effective flotation (Marabini, 1994). Sulphidisation is the preferred method of flotation and generally, CPS gives better results than slug sulphidisation. These results are however difficult to reproduce at plant scale. *"…The main drawback of CPS is that the optimum dose of the sulphidising agent is highly dependent on the time of conditioning, procedures of mixing and other variables, leading usually to poor reproducibility in a plant situation."* (Lee et al., 2009). Change in nature of the feed also has an effect on CPS (Jones & Woodcock, 1979). Nagaraj & Gorken (1991) suggested that the use of an on-stream analyser could eliminate the uncertainty in dosing NaHS and possibly minimise the necessity of making changes in collector dosage.

A wide range of potentials has been observed as "optimum" in the CPS of malachite rich ores; -300 to -380 (E_S) mV (Feron & Manu, 1994), - 500 (E_S) mV (Quast et al., 2005) and +250 (E_h) mV (Soto & Laskowski, 1973). The potential has been shown to be dependent on the concentration of the sulphidiser (Soto & Laskowski, 1973), the sulphidiser used (Quast et al., 2005) and the pulp chemistry (Feron & Manu, 1994).

CPS is not limited to the flotation of non-sulphide ores, but has also been used to successfully float sulphide minerals with surface tarnishing which may occur due to in-situ weathering, mining operations, stockpiling, crushing and milling and flotation (Clark et al., 2000). Newell & Bradshaw (2007) used NaHS for the CPS of tarnished pentlandite and found that flotation was possible in the presence of iron and copper ions at -600 to -700 mV.

The influence of pulp chemistry; dissolved oxygen, pH and E_h on the flotation behavior of sulphide minerals has been extensively studied over the years (Fuerstenau et al, 1985; Hintikka & Leppinen, 1995; Göktepe, 2010). Of particular interest to this work is the effect that electrochemical potential (E_h and E_S) has on the floatability of sulphide activated minerals. Pulp potential can be influenced by mineral-mineral interactions and grinding media used (Rao & Natarajan, 1989; Greet et al., 2005) as well as by the addition of modifiers such as sulphidising agents (Herrera-Urbina et al., 1999; Bulut, 2011). When oxide minerals are sulphidised, the mineral surface is modified so they behave like sulphide minerals and this calls for extreme care in the use of sulphidising agents for the flotation of mixed ores containing both sulphides and non-sulphides (Soto & Laskowski, 1973).

Wills & Napier-Munn (2006) advise that during sulphidisation, the sulphidiser must be just sufficient to produce a single layer of sulphide film to enable xanthate adsorption. Aeration during sulphidisation must be carefully controlled as it has two important effects. During sulphidisation, any oxygen present will consume sulphide ions present (Wills & Napier-Munn, 2006) leading to high sulphidiser requirements but some oxidation after sulphidisation is advantageous as it prevents the secondary effects of CPS such as the depression of the newly formed pseudo-sulphide mineral surfaces (Soto & Laskowski, 1973).

3. RESEARCH OBJECTIVES

“Plan backwards as well as forward. Set objectives and trace back to see how to achieve them. You may find that no path can get you there. Plan forward to see where your steps will take you, which may not be clear or intuitive”

- Donald Rumsfeld

3.1. Objectives

The objectives of this study were twofold:

1. To apply the basic principles of process mineralogy to characterise the two ores; high quality (HQ) and low quality (LQ).
2. To develop the best reagent suite for optimised mineral recovery for the two ores.

3.2. Key questions

Within the context of the scope outlined in Chapter 1 and the objectives outlined above, the study focused on the following key questions:

1. What is the mineralogical difference between high quality (HQ) and low quality (LQ) ore?
2. What is the extent of liberation of the different minerals in the feed to float?
3. What copper species report as acid soluble copper?
4. How do the two ores respond to flotation with xanthates?
5. What is the optimal methodology for NaHS treatment of the LQ ore?
6. Is slug sulphidisation better than controlled potential sulphidisation?
7. Is the loss of minerals to the tailings due to poor liberation or inappropriate reagent regimes?

8. How do pure samples of non-sulphide minerals respond to xanthate flotation and NaHS treatment?

3.3. Hypotheses

1. Poor copper recoveries are either due to the existence of minerals in the ore that are not amenable to flotation, or due to poor liberation of copper minerals, because these two factors have been shown to be important mineralogical attributes defining the flotation response of an ore.

2. Since the collector products that form when xanthate is added to a sphalerite pulp and a copper oxide pulp are both readily soluble in water and hence do not create a hydrophobic film around the mineral, it is possible that NaHS will play a role as an activator of oxide copper minerals similar to that of copper sulphate ($CuSO_4$) in the flotation of sphalerite due to the formation of pseudo copper sulphide molecules at the mineral interface.

3. Because it is known that collector dosage affects the recovery of value minerals, there should be optimum amount of collector required for the amount of sulphidiser used to activate the oxide minerals.

4. RESEARCH METHODOLOGY

"Execution is everything"
- Jeff Bridges

4.1. Introduction

This chapter describes the materials used and the methods that were followed in carrying out the test work. Ore preparation, assay and mineralogy techniques used are described and the flotation procedures are outlined. The key indicators for each of the techniques employed are also given in this chapter.

4.2. Ore sampling and preparation

Two high grade mixed copper ores of varying oxide copper content were obtained from Kansanshi Mining Plc. in Solwezi, Zambia. For the purpose of this work, these two ores will be referred to as High Quality (HQ) and Low Quality (LQ), the quality being an indication of the oxide mineral content as outlined in Table 1.1-1.

Sample preparation was done at the Kansanshi Mining Plc. metallurgical laboratory. 1 t of each ore type was sun and air dried on stainless steel trays on site after collection from the pit and then crushed to -1 cm using a laboratory scale TM Engineering Terminator jaw crusher. Blending and splitting were done simultaneously by using a rotary riffle splitter, as it is more robust and accurate in producing representative samples (Wills & Napier-Munn, 2006) than a stationary riffle sampler and also results in reduced grouping and segregation error. It is also quicker (Petersen et al., 2005). Each batch of crushed ore was presented to the splitter thrice so as to ensure sufficient blending and splitting. The representative 1 kg samples obtained were each packed into plastic bags and shipped to the University of Cape Town.

4.3. Grinding procedure

At the University of Cape Town Centre for Minerals Research (CMR) laboratories, an Eriez Magnetics® MASCLAB belt driven stainless steel laboratory scale rod mill was used for all milling in the test work. The mill had an internal diameter of 200 mm and a depth of 297 mm and was charged with twenty rods of three varying diameters as recommended by the University of Cape Town, Centre for Minerals research (Table 4.3-1).

Table 4.3-1: Mill charge parameters

Nominal diameter (mm)	Number of rods	Total charge weight (kg)
25	6	4.90
20	8	3.67
16	6	1.53
Total	20	10.10

The mill was charged with 1 kg ore and 1 L water, closed and operated at 256 rpm as determined by the CMR. The water used for milling prior to flotation was synthetically made up according to the CMR recipe (Wiese et al., 2005) and prepared in 20 L batches to contain specific target ions (Appendix C) as outlined in Table 4.3-2.

Table 4.3-2: The concentration of ions in synthetic plant water used for all milling and flotation procedures

Cation	Ca^{2+}	Mg^{2+}	Na^{+}	Cl^{-}	$SO_4^{2}-$	NO_3^{-}	CO_3^{2-}
Concentration (ppm)	80	70	153	287	240	176	17

4.4. Grind establishment

A grind size of 80 % passing 150 µm was chosen to replicate site conditions. Because particle size distribution (PSD) was considered a key indicator for liberation, the time required to give a finer grind size of 80 % passing 75 µm was also determined. In order to do this, milling curves were established by grinding each of the two ores at various time intervals. The grinding curves are shown in Figure 4.4-1. The grinding times obtained were 3 minutes and 18 seconds for the LQ ore and 3 minutes and 42 seconds for the HQ ore to obtain 80 % passing 150 µm. For the finer grind of 80 % passing 75 µm, the grind times determined were 7 minutes and 12 seconds for the HQ ore and 8 minutes and 54 seconds for the LQ ore. Full PSDs were done on the mill products (Figure 4.4-2) in order to confirm that the times determined from the grinding curves resulted in the target P_{80}s. All grind establishment data is given in Appendix B.

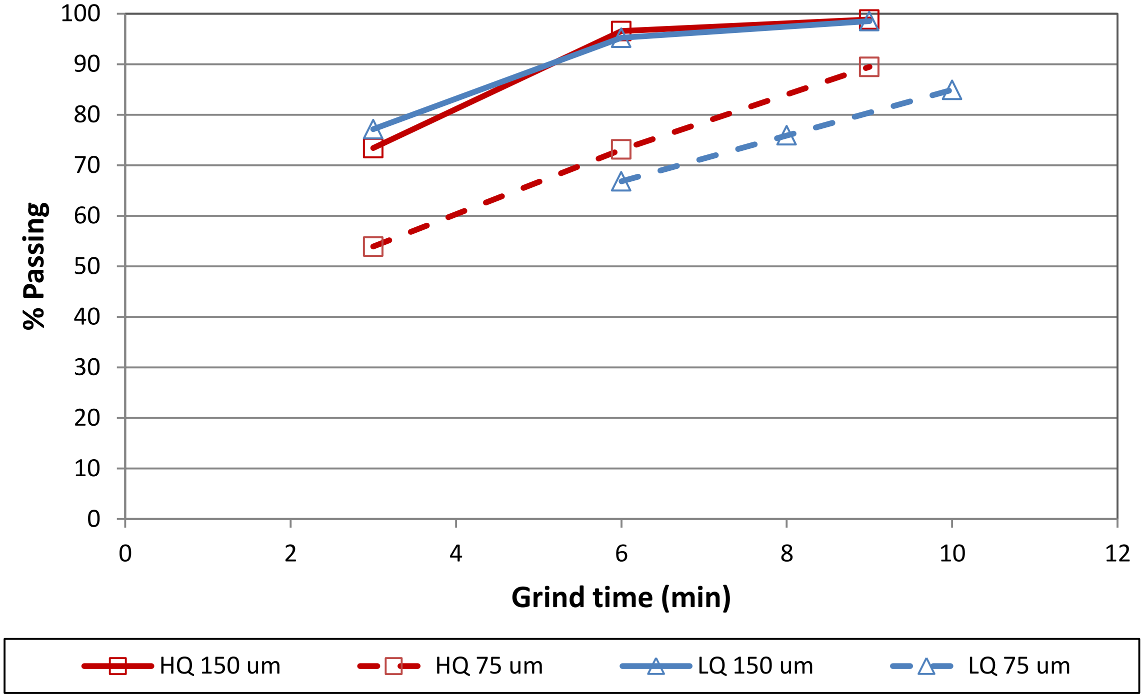


Figure 4.4-1: Milling curves for HQ and LQ ore for 80 % passing 75 µm and 150 µm

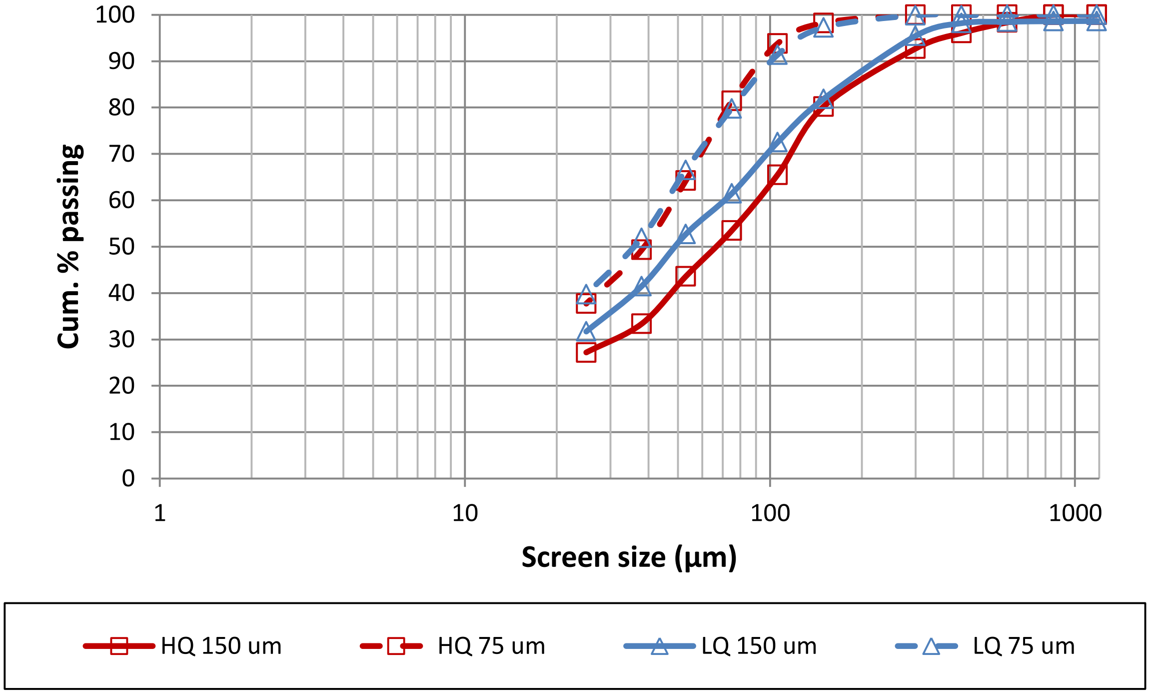


Figure 4.4-2: Full PSDs for established grind times

4.5. Ore characterisation

4.5.1. Chemistry

A 1 kg sample of each of the two ores was reduced to 10 g samples by continuous rotary sampling using a Quantachrome Instruments micro riffler and submitted to the UCT Chemical Engineering analytical chemistry laboratory to ascertain the Cu and Fe content using their standard procedure in which 0.5 g of an ore sample was decomposed by a 10 ml mixture of 3 parts hydrochloric acid, 1 part hydrofluoric acid and 1 part nitric acids for 150 minutes and then transferred to a MARS-5 microwave digester to release the metal ions into solution at 15600 W and 180 °C for 25 minutes. Cu and Fe content were then determined sequentially by Atomic Absorption Spectroscopy (AAS) using a Varian SpectraAA 110 analyser. ASCu content was also determined for both ores by digesting with 10 % Sulphuric acid (Chilumbi, 2009) followed by selective determination using AAS.

A LECO S632 analyser was used for sulphur assay. 0.2 g of the sample was heated to 1350 °C in an induction furnace while passing a stream of oxygen through the sample. The sulphur dioxide released in the process was measured by Infrared detection and gave an indication of the total sulphur content. The head grades were also reconstituted by mass balance using the weights and assays of the concentrate and tailings from the float tests carried out. Table 4.5-1 compares the reconstituted and actual assays determined by AAS.

Table 4.5-1: AAS and Leco elemental analysis of the ores

Ore	Reconstituted from mass balance (%)			Actual assay on Feed sample (%)		
	Cu	ASCu	Fe	Cu	ASCu	Fe
HQ	1.24	Not assayed	3.39	1.17	0.07	2.96
LQ	1.52	0.84	3.52	1.43	1.03	3.54

The size fractions obtained during PSD determination for each (Table 4.5-2) of the ores were prepared for X-Ray Florescence spectroscopy (XRF) in order to provide precise and accurate oxide composition, which can be applied over a wide range (0.2 % to 100 %) for major elements. The samples were micronized and 1 g of each sample was dried at 110 °C and then ashed at 950 °C. The ashed samples were mixed with 6g of flux (47 % lithium metaborate and 53 % lithium tetraborate) and then fused with a Claisse gas burner to create fusion disks (Willis, 1999). A Panalytical Axios Wavelength dispersive XRF spectrometer with a 4 kW Rh tube was used for analysis. The full XRF results are given in Appendix A and were used to validate the QEMSCAN data.

4.5.2. Mineralogy

The mineralogical analysis of the two ores was done using X-Ray Diffraction (XRD) and QEMSCAN. Two 3 g samples of each ore, obtained through micro splitting, were wet pulverized for 10 minutes in a micronizer, using ethanol. Powder XRD spectra were obtained by using a Bruker D8 Advance powder diffractometer with Vantec detector and fixed divergence and receiving slits with Co-Ka radiation. The phases were identified using Bruker Topas 4.1 software (Coelho, 2007) and the relative phase amounts (weight %) were estimated using the Rietveld method for the characterisation of crystalline materials.. The minerals identified through XRD are shown in Figure 4.5-1.

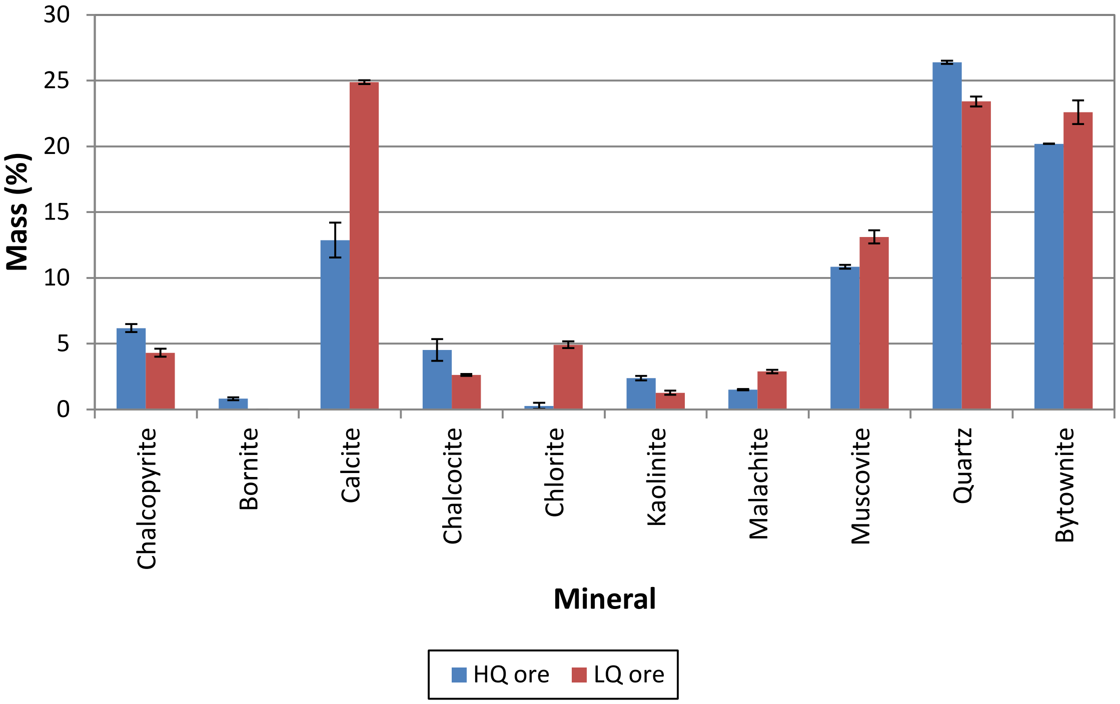


Figure 4.5-1: XRD characterisation of HQ and LQ ore

The XRD technique was however limited in determining all minerals present as some amorphous minerals such as chrysocolla are non-diffracting and hence would not give a pronounced peak in the diffractogram (Frost et al., 2012). In order to obtain more detailed mineralogical analyses, QEMSCAN was used. For each ore, the rod mill prepared samples were split into different size fractions subject to the P_{80} being investigated (Table 4.5-2). The sample preparation process for QEMSCAN analysis is shown in Figure 4.5-2.

Table 4.5-2: Size fractions used for mineralogical analysis using QEMSCAN for high quality (HQ) and low quality (LQ) ore at different grinds

Coarse grind (P_{80} = 150 µm)	Fine grind (P_{80} = 75 µm)
-25 µm	-25 µm
+25/-53 µm	+25/-53 µm
+53/-75 µm	+53/-75 µm
+75/-106 µm	+75/-1000 µm
+106/-150 µm	
+150/-1000 µm	

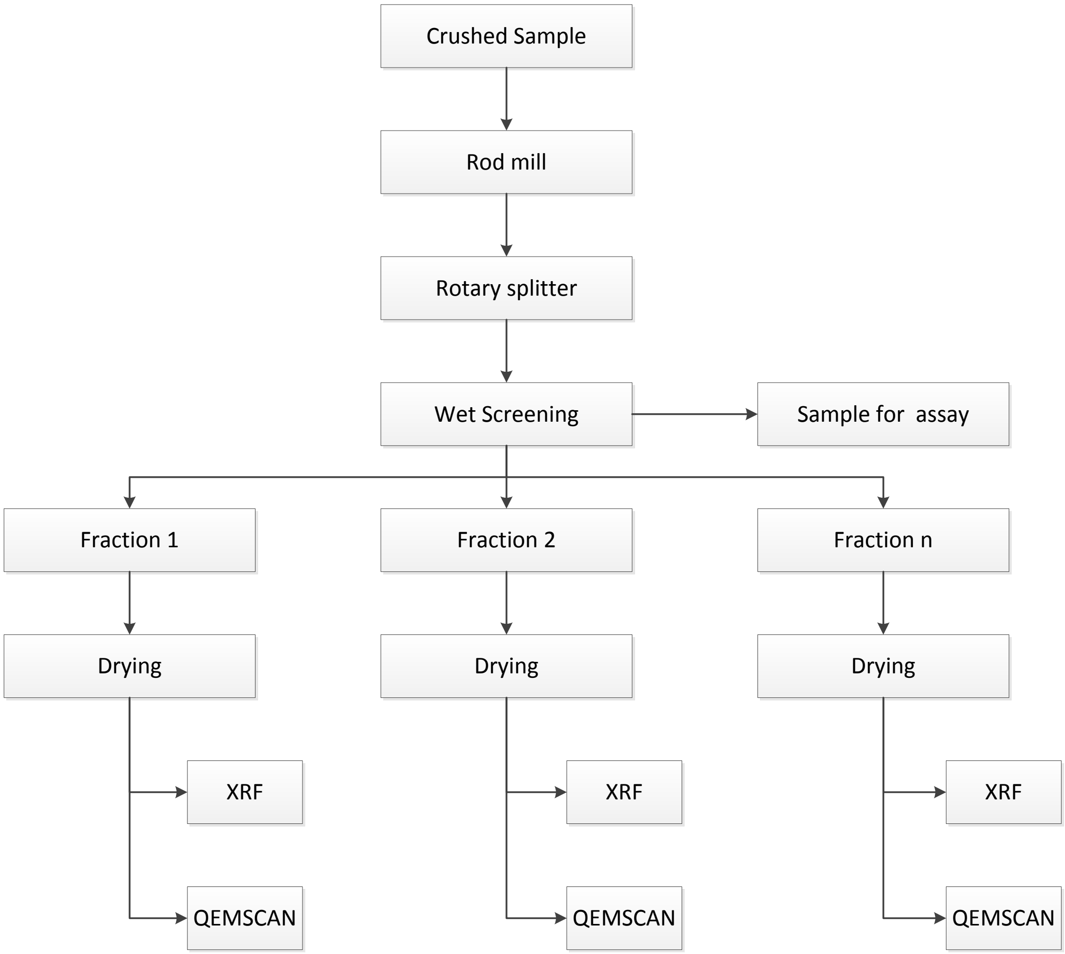


Figure 4.5-2: Bulk sample preparation for mineralogical analysis using QEMSCAN

Each size fraction was split into three 1 g aliquots using a Quantachrome micro riffler. 2 g of milled graphite one size fraction smaller than the fraction being prepared for mineralogical analysis was added in order to reduce touching particles and improve electron conductivity during analysis. The graphite/ore mixture was placed in appropriately labelled and lubricated moulds and resin was added. The samples were stirred appropriately to disperse the particles. The moulds were placed under a Struers Cito-vac vacuum for 15 minutes so as to eliminate any air bubbles and then left to cure overnight in a pressure pot. The cured moulds were labelled by placing a printed label (sample name, size and number) on the back and setting with more resin. These moulds were further cured at 30 °C until dry before polishing and washing in an ultrasonic bath and drying at 30 °C for an hour.

The blocks were removed from the moulds, and polished using a Struers TeraPol-11 polisher to obtain a mirror-like scratch free surface. Each grinding/polishing step was interjected by a rinsing stage. On completion of polishing, the blocks were placed in an ultrasonic bath for 10 minutes, cleaned with ethanol and dried for an hour. The final step in the block preparation was carbon coating in order to ensure diffusion of electrons off the sample surfaces during SEM. The LEO SEM based QEMSCAN QS18 platform equipped with two Bruker 4010 SDD detectors was used for all LQ analyses while the QEMSCAN 650 F with 6th generation Bruker 6030 Si-drift EDS X-ray detectors was used for HQ ore feed analysis as well as all tailings analyses done on LQ ore. During mineralogical analysis, it is important to have sufficient valuable mineral particles assayed in order to produce statistically sound data and ensure the elemental assay predicted by the QEMSCAN aligns with the chemical assays. All mineralogical test work done in this study ensured that at least 1000 copper-bearing particles were analysed in all samples for which liberation and mineral associations were to be determined.

Two types of measurements were used to collect mineralogical data from samples. The first measurement was Particle Mineralogical Analysis (PMA) (Figure 4.5-3), used mostly when samples were non-sulphide copper mineral rich, in order to obtain statistically sound representation of minerals present in the samples, liberation of the value minerals and their mineral associations, as well as data validation. The second measurement used was Specific Mineral Search (SMS) which was used to determine liberation and mineral associations for sulphide minerals, in this case chalcopyrite. The data obtained was processed using QEMSCAN iExplorer software and a false colour map was created for each particle analysed, each colour representing a mineral formula dictated by a manually entered Species Identification Protocol (SIP) list of the mineral groupings. The mineral list used in this study is given in Appendix A.

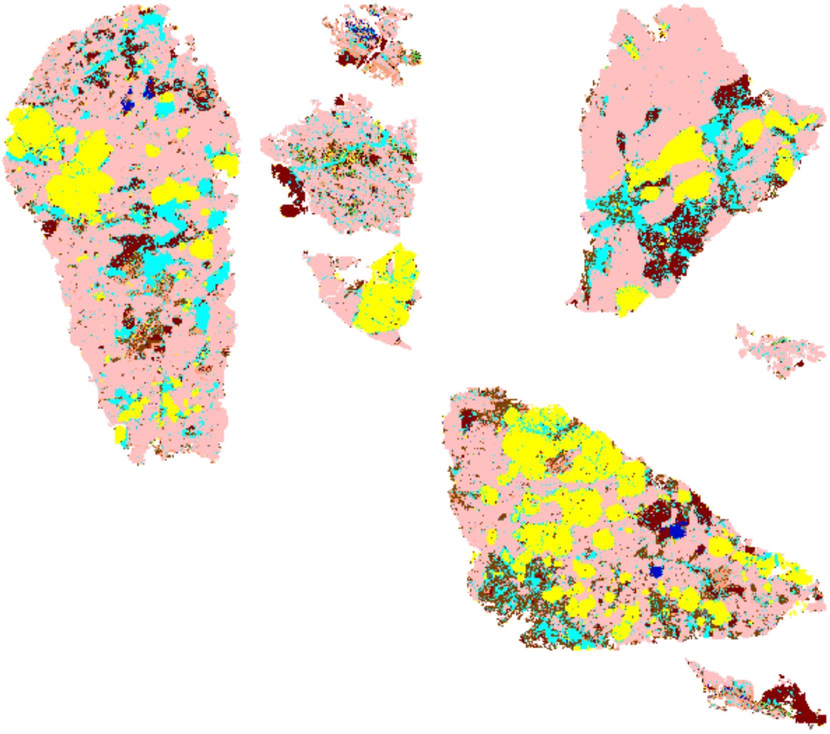


Figure 4.5-3: Generic image illustrating Particle Mineralogical analysis of grains. Each colour represents a different mineral (as dictated by the operator) and associations are readily discernible

The extent of liberation was determined by analyzing the percentage of particle surface area occupied by chalcopyrite (Becker et al., 2009 & Lee et al., 2009) in relation to particle perimeter and classed according to the criteria outlined in Table 4.5-3. Data validation within iExplorer was done by comparing QEMSCAN assays with XRF assays. The data validation tables are given in Appendix A.

Table 4.5-3: Liberation criteria used in mineralogical analysis

Liberated	Area percent chalcopyrite $\geq$ 90 %
Middlings	90 % $\geq$Area percent chalcopyrite $\geq$ 30 %
Locked	Area percent chalcopyrite < 30 %

4.6. Batch flotation

Flotation tests were only done on the coarse grind (80 % passing 150 µm) for both ores. All flotation tests were carried out in a 3 L UCT modified Leeds flotation cell (Figure 4.6-1). The cell is made of clear Perspex to enable froth depth control and is fitted with a top driven variable speed impeller. In each flotation test, the milled slurry was transferred to the flotation cell and topped up to the 3 L mark using synthetic plant water to make up a solids concentration of 35 %. The froth depth was maintained at 1 cm for all tests and for both ores. SIBX was used as the collector of choice because it is used on site. Various impeller speeds and frothers were tested in the initial scoping tests because it was observed that there was loaded material draining back into the pulp. While the HQ ore required 40 g/t of DOW 200 frother, the LQ ore only required 30 g/t DOW 200 for sufficient froth to form. The float cell was operated at 1500 rpm because the standard 1200 rpm recommended in the CMR methodology also resulted in backflow for this particular ore.

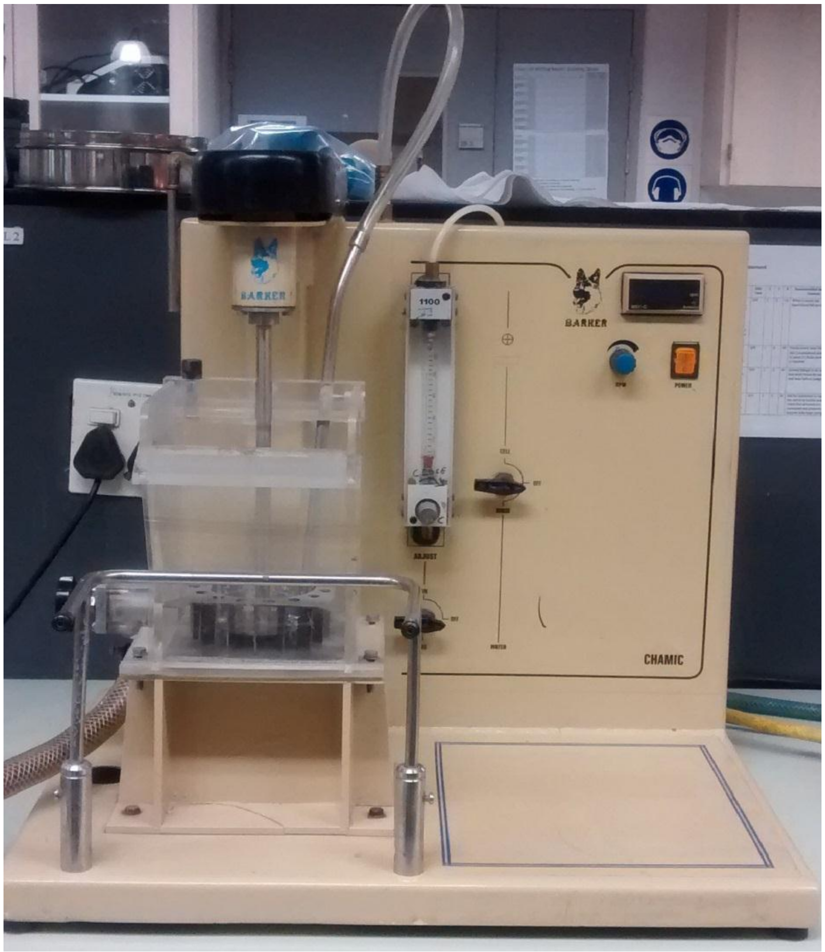


Figure 4.6-1: UCT modified Leeds 3 L flotation cell used for batch flotation tests

Prior to the introduction of reagents, a syringe-full of the feed was drawn, while agitating, for head grade assays. After reagent conditioning, 7 L/s of compressed air was introduced into the cell via a Wilkerson 0.0065 m (¼ in) 0-800 KPa regulator. Throughout the flotation period, concentrate was scrapped from the top of the cell every 15 seconds into weighed concentrate trays that were labeled for each concentrate stage. Each concentrate stage had a corresponding wash bottle used which was weighed before and after the flotation tests for water recovery calculations. At the end of the test the air was turned off and two samples of tailings were drawn using syringes while the pulp was still under agitation. The remainder of the tailings was emptied into a bucket and filtered using an Eriez Magnetics® MASCLAB filter press, dried and weighed. Each of the concentrate trays was weighed and the contents filtered using a vacuum filter and dried at 80 °C. The feed, concentrate and tailings were all submitted for TCu, ASCu and Fe assay using the methods outlined in section 4.5.1. The masses and assays were used to determine the solids and water recovery as well as the value element recovery. Table 4.6-1 gives the reagents used for batch flotation tests.

Table 4.6-1: Reagents used for batch flotation tests and their concentrations

Reagent type	Reagent	Purity (%)	Concentration when made up (wt%)
Collector	SIBX	100	1
Sulphidiser	NaHS	90	10
Frother	DOW 200	100	Neat

4.6.1. Batch flotation procedure for High Quality (HQ) ore

Based on the mineralogy of HQ ore (c.f. Section 5.2), HQ ore flotation tests were done to optimise chalcopyrite recovery. The flotation procedure for HQ ore is shown in Figure 4.6-2 and Table 4.6-2. 30 g/t and 50 g/t of SIBX were investigated for HQ ore.

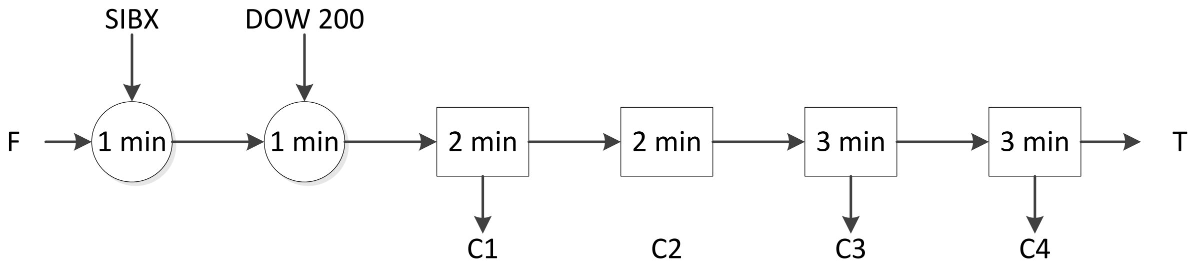


Figure 4.6-2: Batch flotation procedure for HQ ore

Table 4.6-2: Batch flotation procedure for HQ ore

Stage	Float	Conditioning	SIBX	Dow 200
	min	min	g/t	g/t
Grind		3 min 42 sec		
Conditioning		1	30 & 50	
		1		40
Concentrate 1	2			
Concentrate 2	2			
Concentrate 3	3			
Concentrate 4	3			

4.6.2. Batch flotation procedure for Low Quality (LQ) ore

Bearing in mind that this ore contained both sulphide and oxide copper minerals, the flotation procedures developed first focused on optimizing sulphide mineral recovery and then the oxide minerals by investigating both slug and controlled potential sulphidisation (Figure 4.6-3). Sulphide mineral optimisation was done over 10 minutes and a collection of four concentrate (Table 4.6-3). The collector dosages tested were 20 g/t, 30 g/t and 50 g/t of SIBX.

Table 4.6-3: Flotation procedure for sulphide copper recovery in LQ ore

Stage	Float	Conditioning	SIBX	Dow 200
	min	min	g/t	g/t
Grind		3 min 18 sec		
Conditioning		1	20, 30 & 50	
		1		30
Concentrate 1	2			
Concentrate 2	2			
Concentrate 3	3			
Concentrate 4	3			

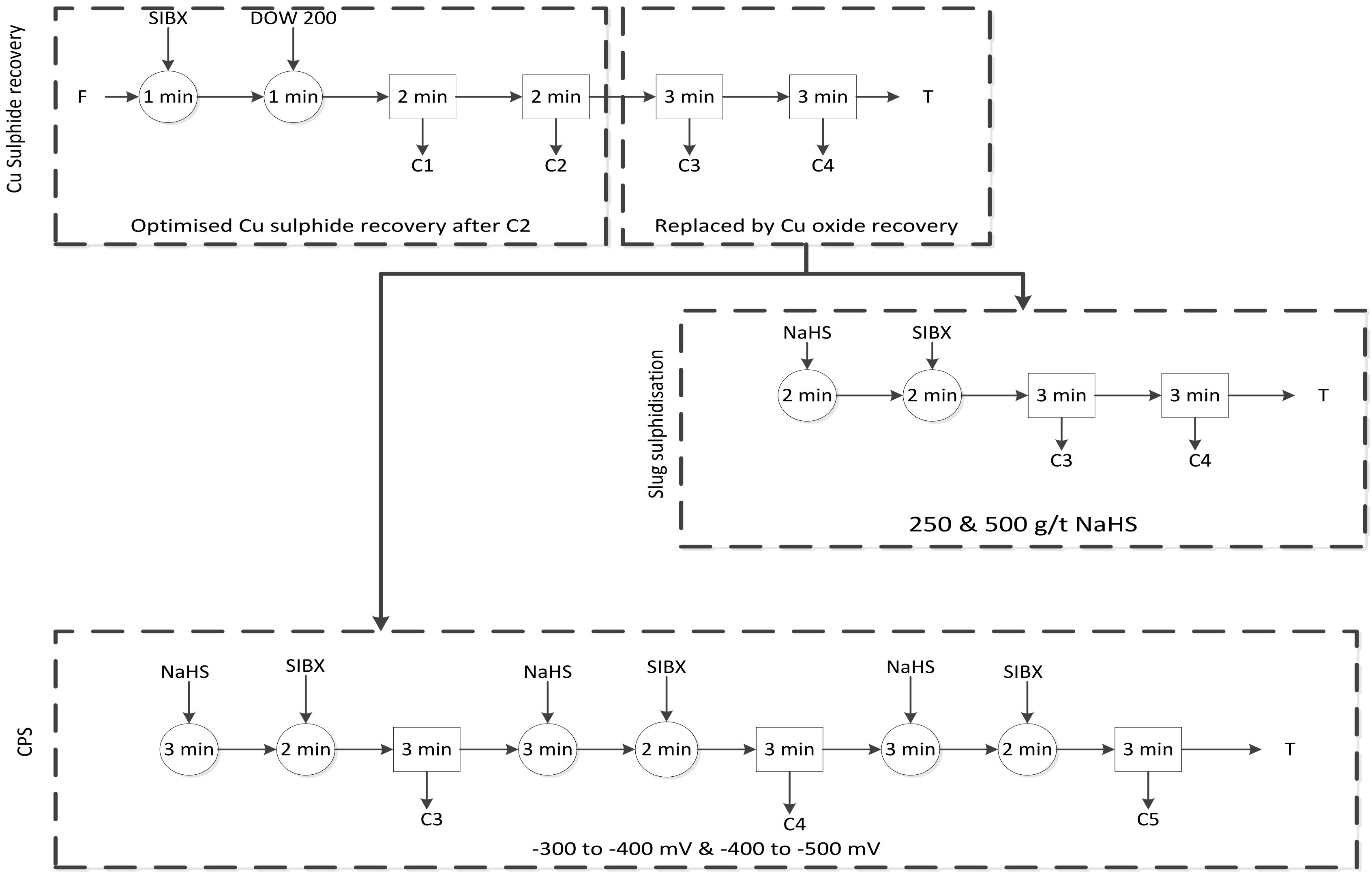


Figure 4.6-3: Summary illustration of sulphidisation and xanthate tests carried out on LQ ore showing Cu sulphide recovery optimisation, slug sulphidisation and CPS

After determining the optimum dosage and time for sulphide copper mineral recovery, a slug sulphidisation stage was added to the flotation procedure (Table 4.6-4) in which 250 g/t and 500 g/t of NaHS were both investigated. The SIBX dosage after sulphidisation was 25 g/t for 250 g/t NaHS and 50 g/t for 500 g/t NaHS to reproduce the 10:1 (g/t) NaHS:SIBX ratio used on the plant.

Table 4.6-4: Slug sulphidisation procedure for LQ ore

Stage	Float	Conditioning	SIBX	Dow 200	NaHS
	min	min	g/t	g/t	g/t
Grind		3 min 18 sec			
Conditioning		1	30		
		1		30	
Concentrate 1	2				
Concentrate 2	2				
		2			250 & 500
		2	25 &50		
Concentrate 3	3				
Concentrate 4	3				

In order to monitor the pulp chemistry during sulphidisation, various electrode systems were used. A YSI 556 MPS multi probe system was used to measure and log the temperature, pH, dissolved oxygen and Redox potential (E_h) of the system and the E_S was measured using a HANNA silver/silver-chloride ISE and HANNA HI 8424 meter. The key difference between the two meters being that the YSI is data logging capable. The reason two meters and two pulp potential probes were used was to determine the relationship, if any, between E_h and E_S. E_h and E_S potentials of a continuously stirred LQ ore were monitored after slug addition of 250 g/t and 500 g/t of NaHS. Figure 4.6-4 shows the relationship between the two. The data used to plot this graph is given in Appendix E. The comparison showed linear relationships between the two expressions within defined potential ranges; -300 to -400 mV and -400 to -500 mV. These two potential ranges were investigated during all CPS test work as they fell within the wide potential ranges that have been prescribed by various studies. Table 4.6-5 shows the flotation procedure for CPS of LQ ore. Both E_h and E_S were monitored during CPS in order to validate the relationship between them further. The NaHS dosage was determined by the potential targeted and three NaHS:SIBX ratios were investigated during CPS, viz. 20:1, 10:1 and 7:1.

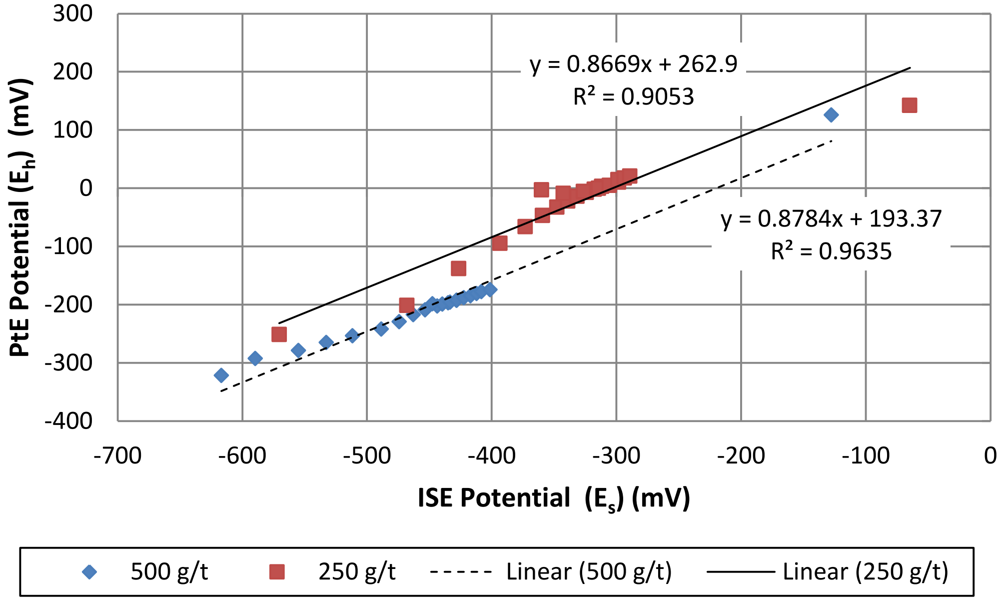


Figure 4.6-4: Relationship between E_h and E_S

Table 4.6-5: CPS flotation procedure for LQ ore

Stage	Float	Conditioning	SIBX	Dow 200	NaHS	Potential
	min	min	g/t	g/t	g/t	mV
Grind		3 min 18 sec				
Conditioning		1	30			
		1		30		
Concentrate 1	2					
Concentrate 2	2					
		3			X	-300 to -400 -400 to -500
		2	0.05X, 0.1X & 0.15X			
Concentrate 3	3					
		3			Y	-300 to -400 -400 to -500
		2	0.05Y, 0.1Y & 0.15Y			
Concentrate 4	3					
		3			Z	-300 to -400 -400 to -500
		2	0.05Z, 0.1Z & 0.15Z			
Concentrate 5	3					

4.7. Microflotation

Microflotation tests were done on pure malachite and chrysocolla samples in order to answer fundamental questions on the ability of SIBX to collect non-sulphide copper minerals. The UCT microflotation cell (Figure 4.7-1) was used. The cell comprises a 365 ml columnar glass cell and launder and a U-tube system for air flow rate and pressure control. A Hamilton syringe is used to inject 7 ml/s of synthetic air at the base of the cell in order to produce a single stream of bubbles to which hydrophobic particles attach and rise to the top of the cell. The particles fall into the collection launder when the bubbles burst after hitting the cone.

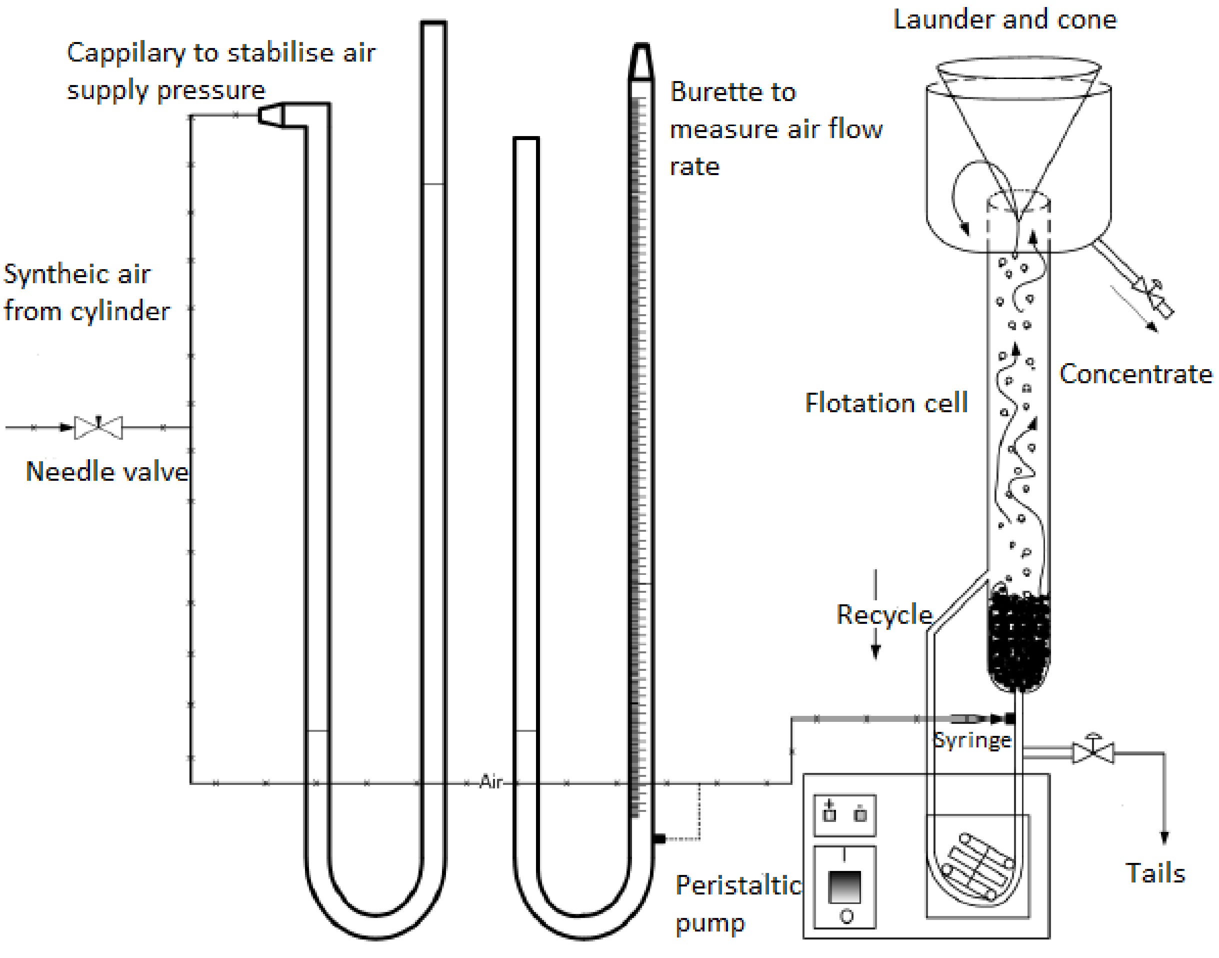


Figure 4.7-1: Schematic representation of the UCT microflotation cell (adapted from Castelyn, 2012)

The malachite (from Arizona) and chrysocolla (from Shaba in Zaire, now the Democratic republic of Congo) used were Ward's study pack samples that had been supplied to the CMR by

Wards Natural Science Establishment Inc. Each of the mineral samples was crushed, pulverized and screened to obtain a -150/+38 µm size fraction on which microflotation would be carried out. 2 g of the sample was carefully weighed out and placed into a 50 ml beaker to which synthetic water (Table 4.3-2) was added to form slurry. The slurry was treated in an ultrasonic bath to loosen any partcles that may have agglomerated. The sonicated slurry was transferred to the microflotation cell and the level made up with additional synthetic water. The reagents under investigation were added and flotation carried out as specified in Table 4.7-1 along with conditioning and flotation times. Each of the concentrates obtained and the tailings were filtered and dried. No assays were done because recoveries were only determined on a mass basis.

Table 4.7-1: Microflotation procedure for pure malachite and chrysocolla using only SIBX

Stage	Float	Conditioning	SIBX
	min	min	M
Conditioning		6	10^{-5}
Concentrate 1	2		
Concentrate 2	2		
Concentrate 3	3		
Concentrate 4	3		

An additional set of tests was done to investigate sulphidisation on a microflotation scale (Table 4.7-2). Only 10^{-5} M concentrations of reagents were investigated as the objective of these tests was only to understand whether or not these reagents are capable of recovering non sulphide copper minerals.

Table 4.7-2: Microflotation procedure for pure malachite and chrysocolla using SIBX and incorporating sulphidisation with NaHS

Stage	Float	Conditioning	NaHS	SIBX
	min	min	M	M
Conditioning		3	10^{-5}	10^{-5}
Conditioning		3		
Concentrate 1	2			
Concentrate 2	2			
Concentrate 3	3			
Concentrate 4	3			

5. RESULTS

"The ability to simplify means to eliminate the unnecessary so that the necessary may speak"
- Hans Hoffmann

5.1. Introduction

This chapter presents the results obtained from mineralogical analyses and batch flotation test work carried out on the High Quality (HQ) and Low Quality (LQ) ores. The mineralogical features presented are; bulk mineralogy, copper deportment, mineral liberation and mineral associations. The flotation response of the ore to varying reagent dosages is given in terms of recovery kinetics and grade versus recovery curves. HQ ore results are presented first, and then LQ ore. In both cases, the mineralogy of each ore is presented first, followed by the flotation results.

5.1.1. Precision of tests

All error bars in the charts in this chapter indicate standard error calculated from triplicate tests (or duplicate tests where stated). Table 5.1-1 shows AAS assays of three HQ and LQ ore samples (A, B and C) after splitting and blending. The standard error between samples was minimal and gave confidence in the AAS technique used.

Table 5.1-1: HQ and LQ AAS analysis precision for TCu, ASCu and Fe

Feed Sample	HQ ore			LQ ore		
	TCu	ASCu	Fe	TCu	ASCu	Fe
A	1.14	0.07	2.99	1.47	1.06	3.40
B	1.15	0.07	2.93	1.26	0.92	3.73
C	1.22	0.07	2.95	1.57	1.10	3.49
Average	1.17	0.07	2.96	1.43	1.03	3.54
Standard error	±0.03	±0.00	±0.02	±0.09	±0.05	±0.10

In order to indicate the reproducibility and confidence in the flotation technique, cumulative solids and water recoveries are compared for triplicate 50 g/t SIBX flotation test done on HQ ore. These are shown in Table 5.1-2 and Figure 5.1-1.

Table 5.1-2: Cumulative solids and water recoveries, their averages and standard error for triplicate 50 g/t SIBX dosages on HQ ore

Conc.	Cum. Time (min)	Test 1		Test 2		Test 3		Average		Standard error	
		Solids (g)	Water (g)	Solids (g)	Water (g)	Solids (g)	Water (g)	Solids (g)	Water (g)	Solids	Water
C1	2	48.12	54.16	49.61	61.47	50.76	61.49	49.50	59.04	±0.76	±2.44
C2	4	60.39	79.89	61.37	84.16	65.90	96.85	62.55	86.97	±0.70	±5.09
C3	7	71.02	100.13	71.33	109.44	77.17	123.09	73.17	110.89	±2.00	±6.67
C4	10	77.03	111.02	76.82	119.37	82.79	138.63	78.88	123.01	±1.96	±8.18

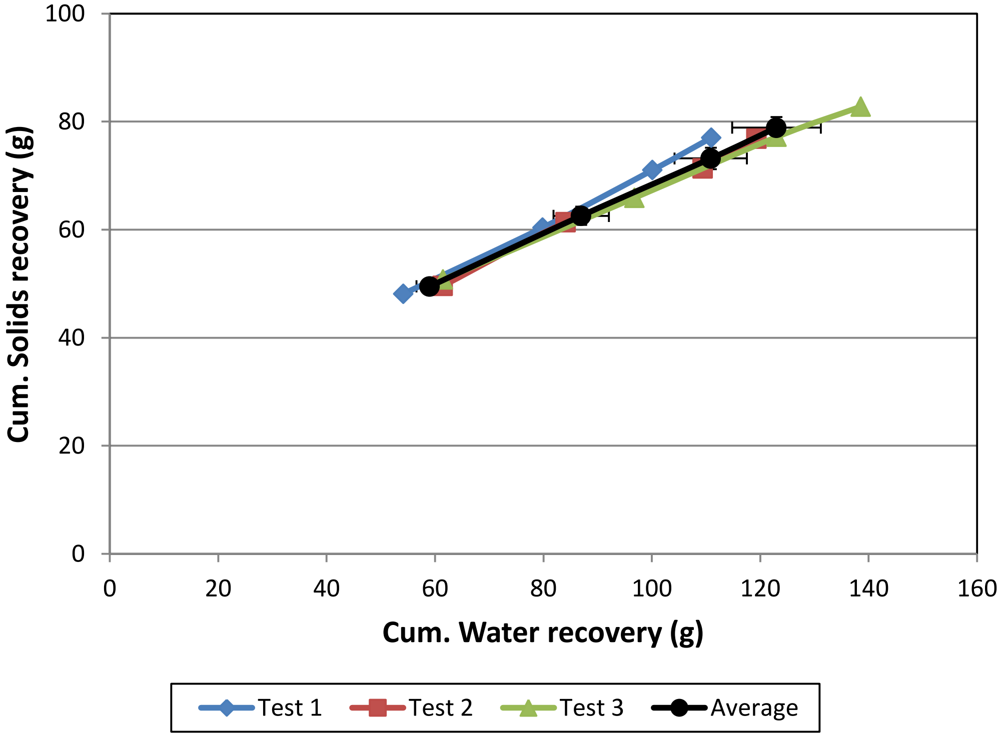


Figure 5.1-1: Degree of reproducibility of flotation tests; indicated by cumulative solids recovery versus cumulative water recovery of three 50 g/t SIBX tests (on HQ ore), their average and standard error

All the figures cited in this chapter are reported to 1 decimal place because the standard error between the assay results was minor.

5.2. Mineralogy and batch flotation of High Quality (HQ) ore

5.2.1. Bulk mineralogy of HQ ore

Table 5.2-1 summarises the average bulk mineralogy of HQ ore. The ore contained about 3.9 % chalcopyrite, 0.1 % chrysocolla and trace amounts of other copper minerals, viz. malachite, bornite, covellite and chalcocite/digenite. The bulk of the gangue minerals comprised mica, plagioclase-feldspar and quartz. Since the most abundant copper mineral was chalcopyrite, liberation and mineral associations for this ore, determined using iExplorer software, was done using only the Specific Mineral Search option as explained in section 4.5.2.

Table 5.2-1: Bulk mineralogy of HQ ore as determined by QEMSCAN

Mineral	Mineral content (%)		
	P_{80} = 75 µm	P_{80} = 150 µm	Average
Pyrite	2.6	3.4	3.0
Pyrrhotite	0.9	0.8	0.8
Chalcopyrite	3.6	4.2	3.9
Bornite	<0.1	<0.1	<0.1
Chalcocite / Digenite	<0.1	<0.1	<0.1
Covellite	<0.1	<0.1	<0.1
Other sulphides	<0.1	<0.1	<0.1
Cuprite	0.0	<0.1	<0.1
Malachite/Azurite	<0.1	<0.1	<0.1
Chrysocolla	0.1	0.1	0.1
Amphibole	0.8	0.6	0.7
Mica	23.6	23.0	23.3
Kaolinite	0.3	0.2	0.3
Plagioclase-Feldspar	32.0	33.4	32.7
Quartz	25.9	25.0	25.4
Calcite	4.4	4.0	4.2
Fe-Ti minerals	1.7	1.7	1.7
Limonite	3.0	2.4	2.7
Others	1.1	1.1	1.1

5.2.2. Copper deportment in HQ ore

The copper in the HQ ore was mostly hosted in sulphide minerals (Figure 5.2-1). About 93.6 % of the Cu was present in the form of chalcopyrite, 1.3 % as covellite and 0.7 % as bornite and 1.7 % as chrysocolla. The full copper deportment is given in Appendix A. The empirical formulae of the copper minerals present in the ore have been given in Table 2.2-1.

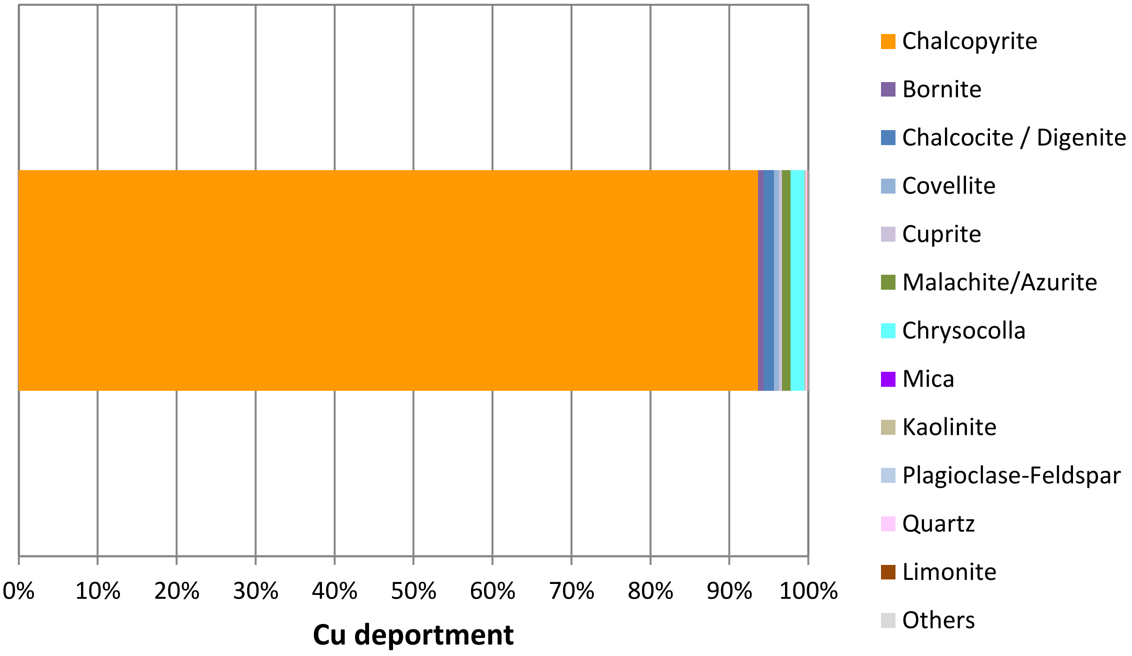


Figure 5.2-1: Copper deportment in HQ ore as determined by QEMSCAN

The copper grades in the size fractions obtained after grinding the ore to a P_{80} of 150 µm are given in Table 5.2-2. 18.8 % of the copper reported to the -25 µm fraction and 25.0 % of the copper was in the +150 µm fraction.

Table 5.2-2: Copper distribution by size fraction in HQ ore for a P_{80} of 150 µm as determined by QEMSCAN

Size fraction (µm)	**+150**	**+106**	**+75**	**+53**	**+25**	**-25**
Copper grade (%)	0.4	0.3	0.2	0.2	0.2	0.3
% of the total Cu in size fraction	25.0	18.8	12.5	12.5	12.5	18.8

Figure 5.2-2 shows the normalized copper deportment in the size fractions. The chalcocite/digenite mineral grouping only existed in the +150 µm fraction while the chrysocolla, malachite and covellite occurred mostly in the -25 µm fraction. The deportment also showed that the chalcopyrite made up more than 85.0 % of the copper minerals across all the size fractions, viz. 87.2 %, 94.7 %, 96.0 %, 96.3 %, 97.0 % and 90.1 % in the, -25 µm, -53/+25 µm, -75/+53 µm, -106/+75 µm, -150/+106 µm and +150 µm size fractions respectively.

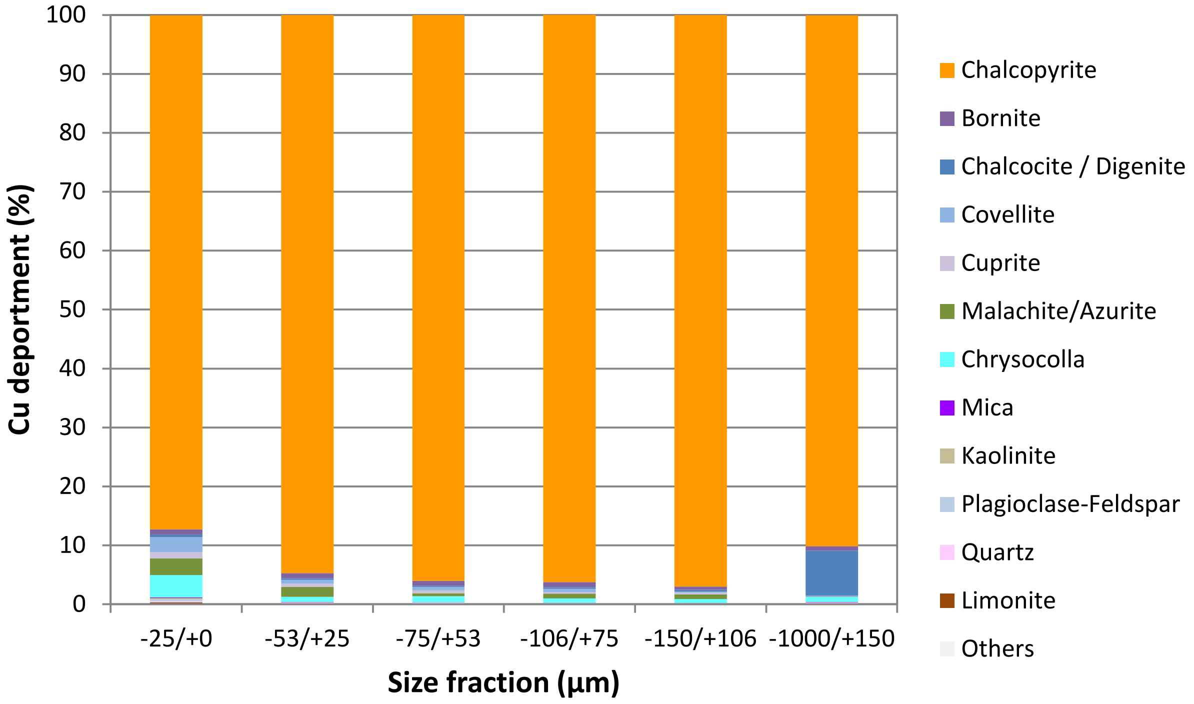


Figure 5.2-2: Normalised copper deportment by size fraction in HQ ore, P_{80} = 150 µm

Table 5.2-3 shows that grinding finer to 80 % passing 75 µm increased the amount of copper reporting to the -25 µm fraction from 18.8 % to 28.6 %. The Normalised copper deportment per size fraction of the fine grind (P_{80} = 75 µm) is shown in Figure 5.2-3. The chalcopyrite made up 89.8 % of the -25 µm size fraction, 95.2 % of the -53/+25 µm fraction, 94.1 % of the -75/+53 µm fraction and 97.2 % of the +75 µm fraction.

Table 5.2-3: Copper distribution by size fraction in HQ ore for a P_{80} of 75 µm as determined by QEMSCAN

Size fraction (µm)	**+75**	**+53**	**+25**	**-25**
Copper grade (%)	0.4	0.3	0.3	0.4
% of the total Cu in size fraction	28.6	21.4	21.4	28.6

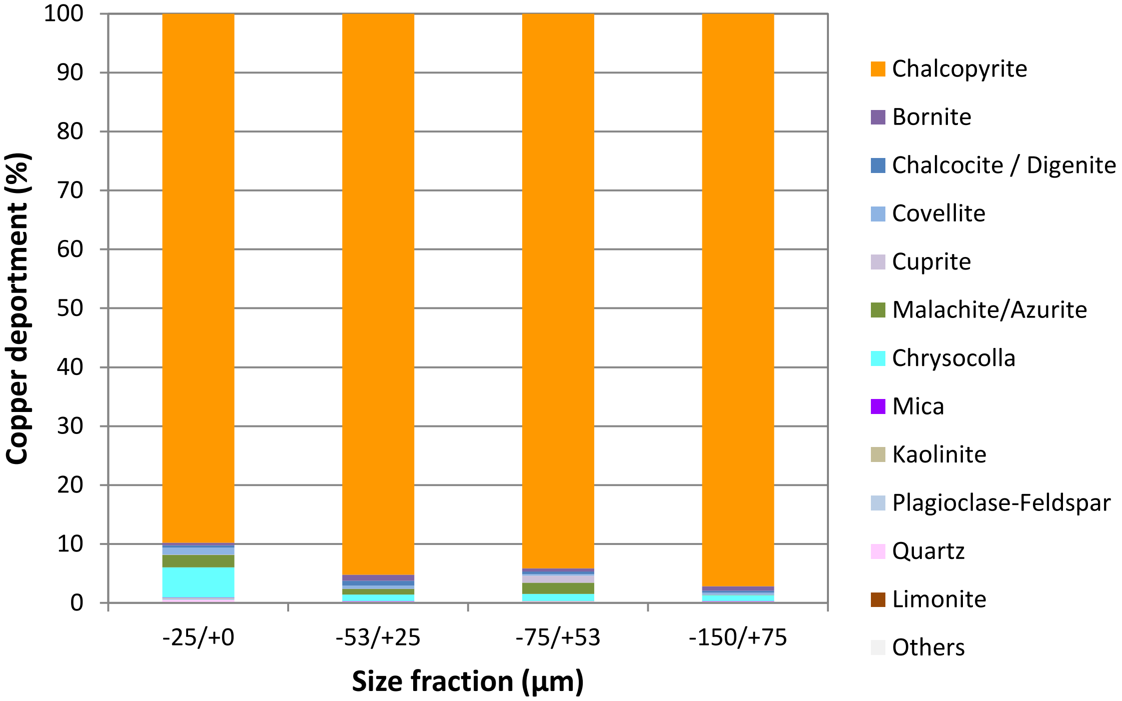


Figure 5.2-3: Normalised copper deportment by size fraction in HQ ore, P_{80} = 75 µm

5.2.3. Liberation and mineral associations in HQ ore

Since the bulk of the copper in HQ ore existed in the form of chalcopyrite, all the mineral properties investigated for HQ ore focused only on this mineral. The two P_{80}s investigated gave similar extents of liberation (Figure 5.2-4). The meaning of liberation has been defined in section 4.5.2. In the case of 80 % passing 150 µm, 90.6 % of the chalcopyrite (corresponding to 3.5 % out of the 3.9 % present in the ore) was fully liberated. This increased slightly to 90.8 % (3.5 % out of the 3.9 % present in the ore) when the ore was ground finer to 80 % passing 75 µm. This very slight increase in liberation implied that there was probably no advantage to be gained by grinding to 80 % passing 75 µm.

Since the two P_{80}s gave similar extents of liberation, only the coarse grind (80 % passing 150 µm) associations were considered. These are presented in Figure 5.2-5. 4.5 % of the chalcopyrite in LQ ore was associated with chrysocolla and 2.6 % with malachite. A negligible amount of the chalcopyrite was associated with other copper sulphides (bornite, covellite and digenite) and iron sulphides.

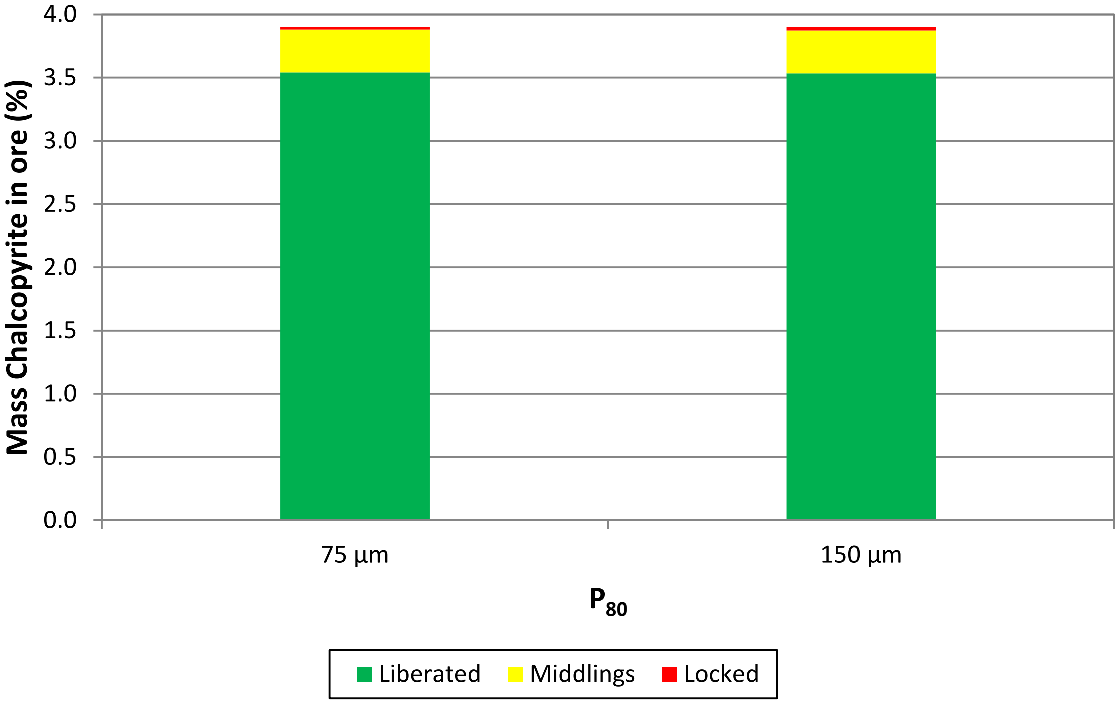


Figure 5.2-4: Chalcopyrite liberation for coarse (P_{80} = 150 µm) and fine (P_{80} = 75 µm) grind of HQ ore

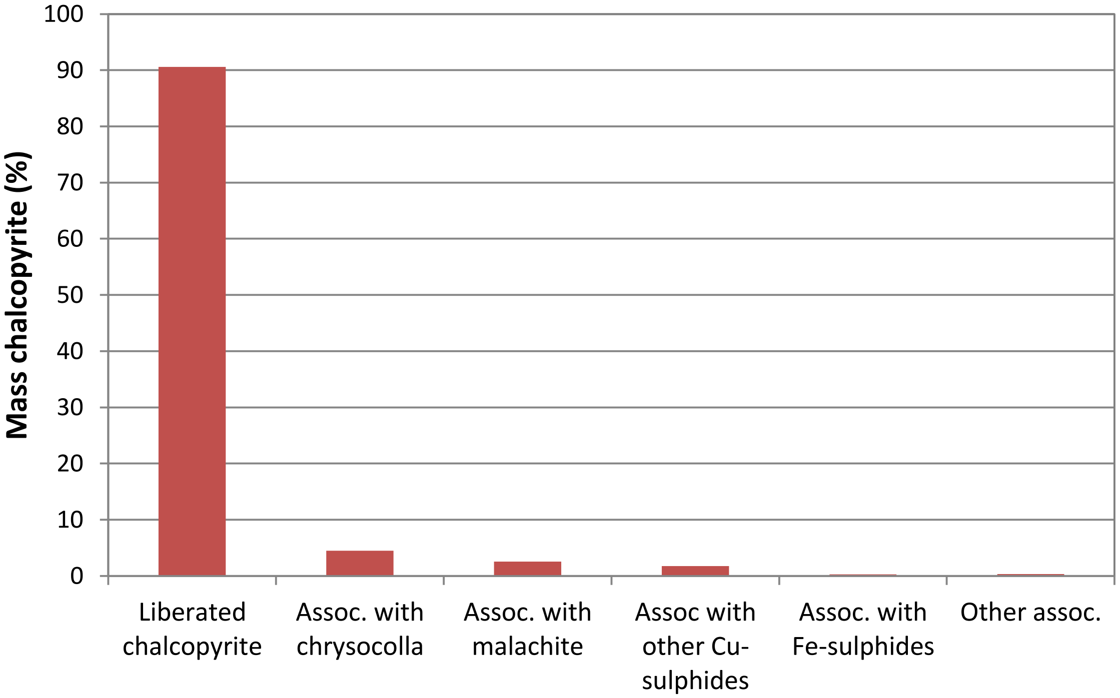


Figure 5.2-5: Associations between chalcopyrite and other minerals in HQ ore, P_{80} = 150 µm

5.2.4. Batch flotation of HQ ore using SIBX

Based on the mineralogy, the targeted value constituent of HQ ore was chalcopyrite (which made up about 93.6 % of the TCu) and other sulphides that were present in minor amounts, viz. covellite, digenite, chalcocite and bornite. Figure 5.2-6 shows the cumulative solids and water recoveries when 30 g/t and 50 g/t of SIBX were used. 90.7 g and 78.9 g of solids per kg of ore treated were recovered at dosages of 30 g/t and 50 g/t of SIBX respectively. 30 g/t SIBX gave higher water recovery than 50 g/t at equivalent solids recoveries.

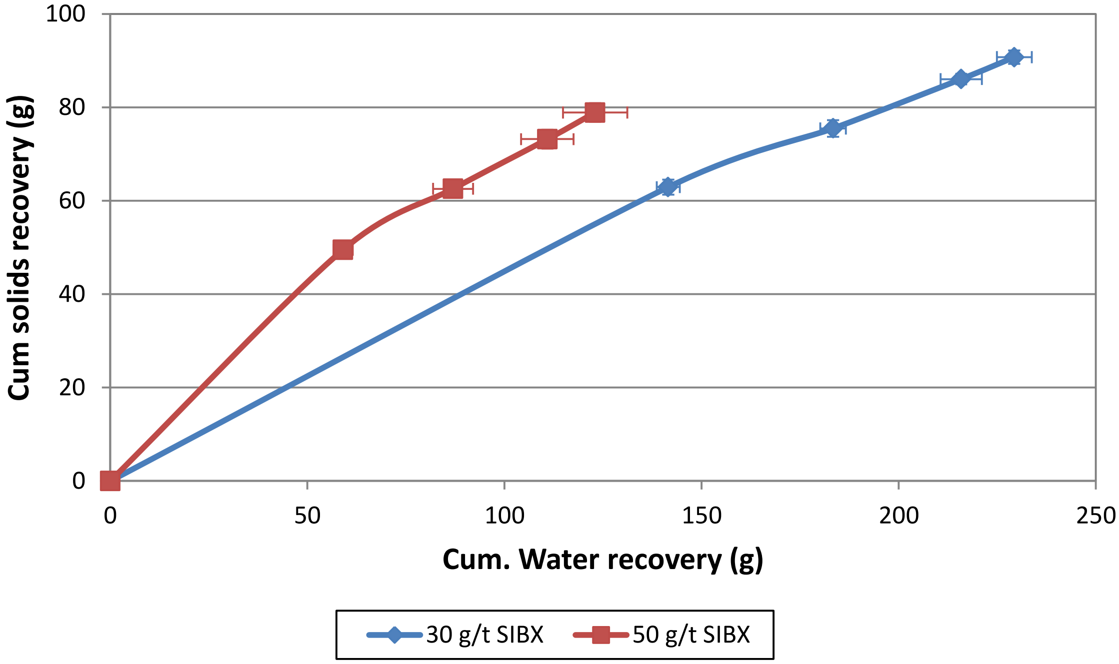


Figure 5.2-6: Cumulative solids versus cumulative water recovery for dosages of 30 g/t and 50 g/t SIBX on HQ ore

Figure 5.2-7 shows cumulative recovery vs. time for both TCu and Fe as determined by AAS. Despite the differences in solids and water recovery, the two dosages resulted in almost equivalent cumulative copper recovery. The cumulative copper recovery at the end of 10 minutes was 89.6 % for 30 g/t and 89.7 % for 50 g/t of SIBX. Figure 5.2-8 presents the grade-recovery data for both SIBX dosages. Fe concentrate grade at the 30 g/t SIBX dosage was slightly higher than that of 50 g/t and this corresponded with the slight decrease in Cu grade from 13.4 % at 50 g/t SIBX to 12.9 % at 30 g/t despite similar TCu recoveries.

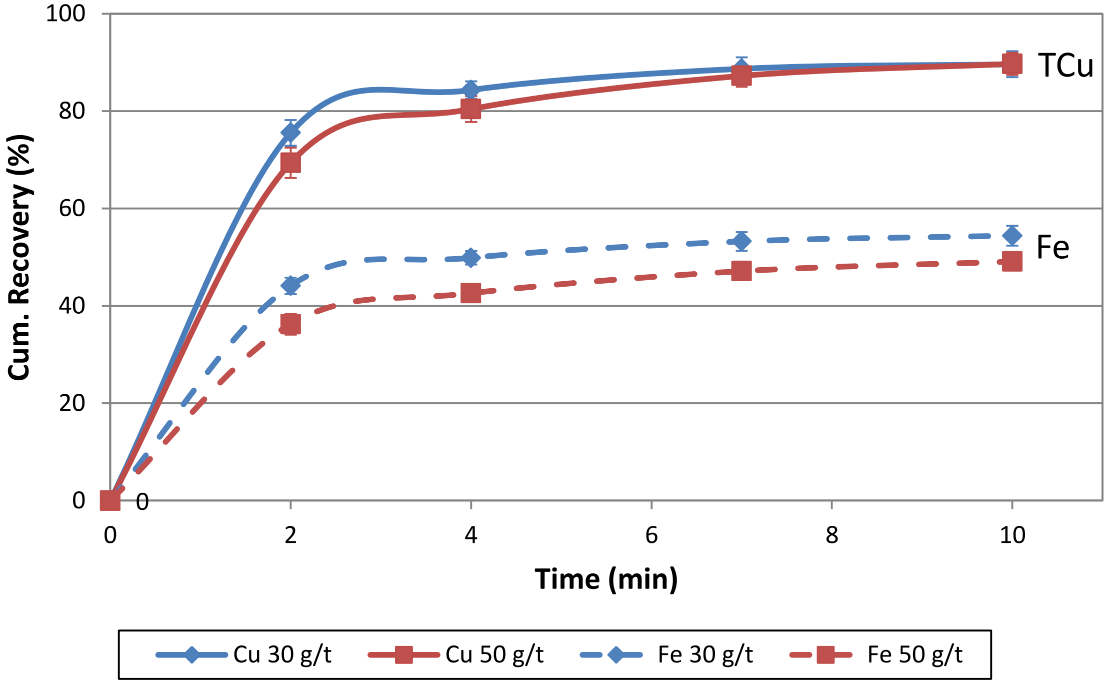


Figure 5.2-7: TCu and Fe recovery-time plots for the flotation of HQ ore at dosages of 30 g/t and 50 g/t SIBX

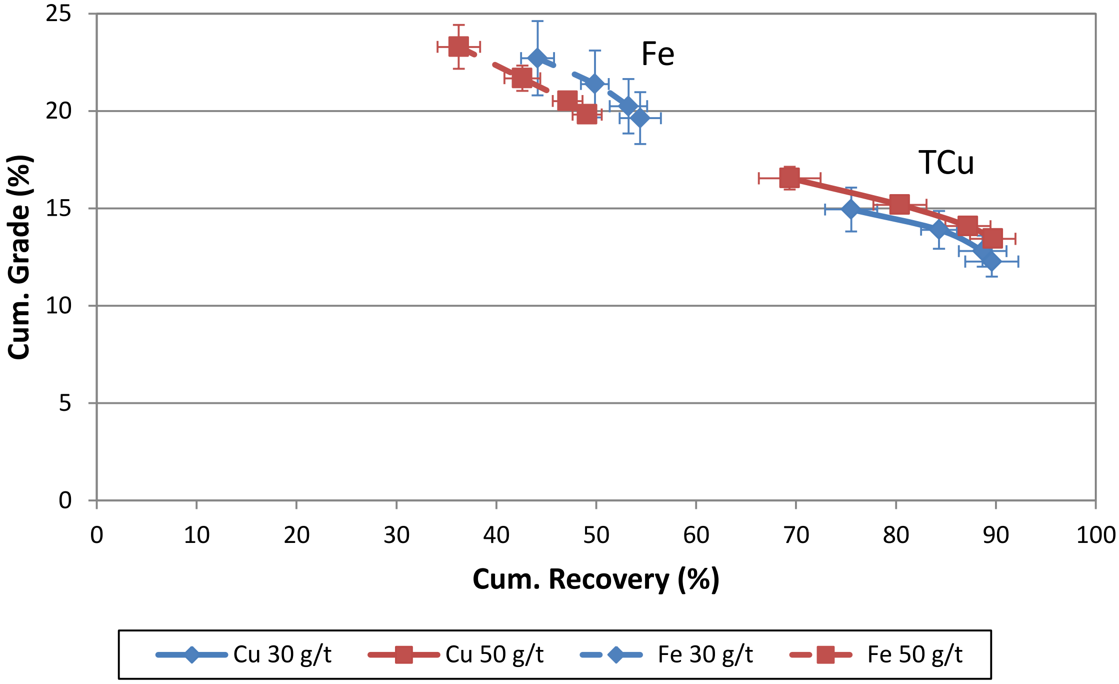


Figure 5.2-8: TCu and Fe grade versus recovery for flotation of HQ ore at dosages of 30 g/t and 50 g/t SIBX

5.3. Mineralogy and batch flotation of Low Quality (LQ) ore

5.3.1. Bulk mineralogy of LQ ore

The bulk mineralogy of LQ (Table 5.3-1) ore was significantly different from that of HQ ore both in terms of value mineral and gangue mineral content. While HQ ore contained mostly chalcopyrite, the copper mineral that dominated LQ ore was chrysocolla (3.8 %). Chalcopyrite and malachite made up 1.0 % 0.1 % the LQ ore respectively. The major gangue minerals in LQ ore were calcite (29.1 %), plagioclase-Feldspar (23.8 %) and quartz (16.4 %).

Table 5.3-1: Bulk mineralogy of LQ ore as determined by QEMSCAN

Mineral	Mineral content (%)		
	P_{80} = 75 µm	P_{80} = 150 µm	Average
Pyrite	1.1	0.8	1.0
Pyrrhotite	0.7	0.6	0.7
Chalcopyrite	1.1	0.8	1.0
Bornite	<0.1	<0.1	<0.1
Covellite	<0.1	<0.1	<0.1
Other sulphides	<0.1	0.1	0.1
Cuprite	<0.1	<0.1	<0.1
Malachite/Azurite	0.1	0.1	0.1
Chrysocolla	3.7	3.9	3.8
Amphibole	3.8	3.8	3.8
Mica	9.4	9.0	9.2
Kaolinite	1.3	2.0	1.7
Plagioclase-Feldspar	24.0	23.6	23.8
Quartz	17.3	15.5	16.4
Calcite	27.8	30.4	29.1
Fe-Ti minerals	1.1	1.1	1.1
Limonite	6.9	6.6	6.7
Others	1.7	1.7	1.7

5.3.2. Copper deportment in LQ ore

LQ ore had a wide array of both sulphide and oxide copper minerals (Figure 5.3-1). 63.1 % of the copper reported as chrysocolla, 22.2 % as chalcopyrite, 1.5 % as cuprite and 2.9 % as malachite/azurite. The full copper deportment of LQ ore is given in Appendix A.

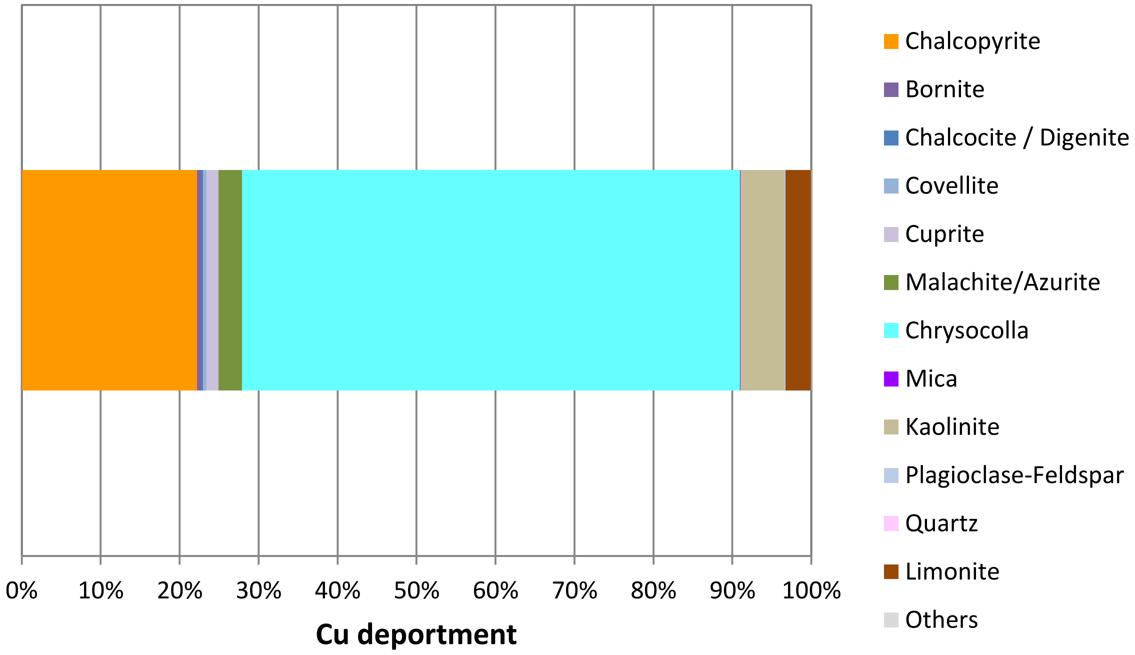


Figure 5.3-1: Copper deportment in LQ ore as determined by QEMSCAN

Table 5.3-2 shows the copper distribution by size fraction when the ore was ground to P_{80} of 150 µm. 33.3 % of all the copper reported to the -25 µm size fraction and 22.2 % to the +150 µm fraction.

Table 5.3-2: Copper distribution by size fraction in LQ ore for a P_{80} of 150 µm as determined by QEMSCAN

Size fraction (µm)	+150	+106	+75	+53	+25	-25
Copper grade (%)	0.4	0.3	0.2	0.1	0.2	0.6
% of the total Cu in size fraction	22.2	16.7	11.1	5.6	11.1	33.3

Figure 5.3-2 shows the normalized copper deportment in all the size fractions. About 77.0 % of the copper in the -25 µm fraction existed as chrysocolla. This was followed by the +150 /-1000 µm fraction in which 65.4 % of the copper also reported as chrysocolla. Chalcopyrite existed

mostly in the -75/+53, -106/+75 and -150/+106 µm fractions, making up 29.7 %, 34.1 % and 26.7 % of the deportment respectively.

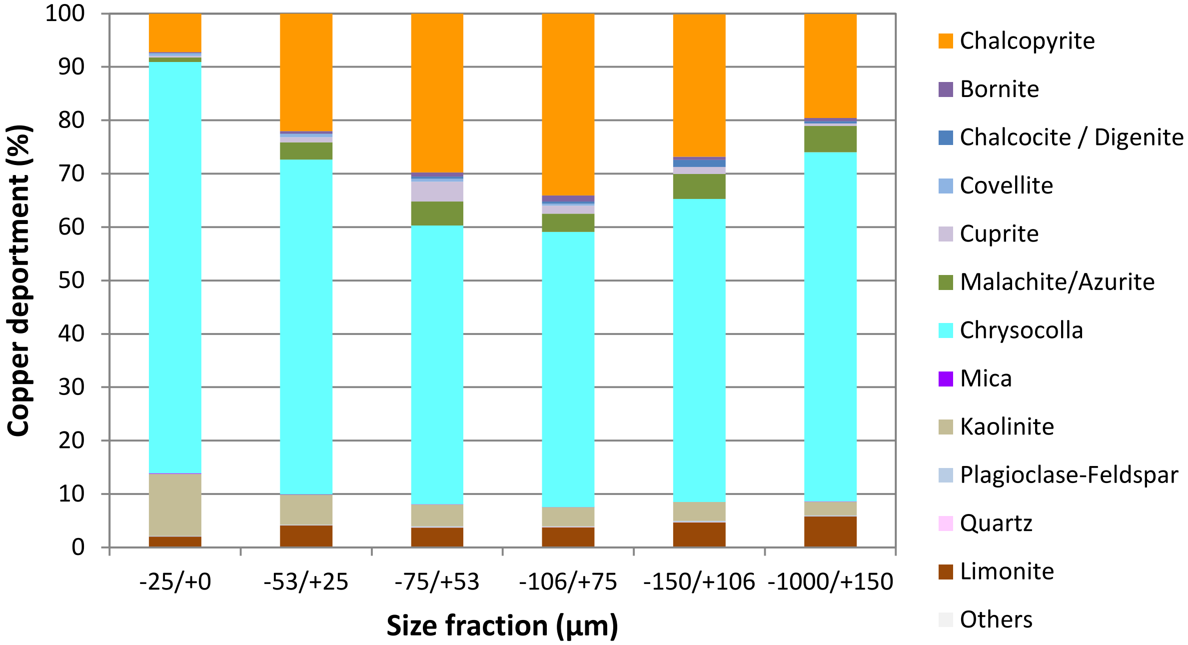


Figure 5.3-2: Normalised copper deportment by size fraction in LQ ore, P_{80} = 150 µm

When LQ ore was ground finer to 80 % passing 75 µm, the copper distribution across the size fractions changed (Table 5.3-3). About 43.8 % of all the copper reported to the -25 µm fraction, increasing from 33.3 % when the P_{80} was 150 µm.

Table 5.3-3: Copper distribution by size fraction in LQ ore for a P_{80} of 75 µm as determined by QEMSCAN

Size fraction (µm)	**+75**	**+53**	**+25**	**-25**
Copper grade (%)	0.2	0.3	0.4	0.7
% of the total Cu in size fraction	12.5	18.8	25.0	43.8

Figure 5.3-3 shows the normalized copper deportment by size fraction when LQ ore was ground to 80 % passing 75 µm. Chrysocolla made up the bulk of the copper deportment hosting 71.2 % of all the copper in the -25 µm fraction, 61.0 % in the -53/+25 µm fraction, 34.7 % in the -75/+53 µm fraction and 56.4 % in the +75 µm fraction. Chalcopyrite made up 16.9 %, 25.3 %, 47.7 % and 30.1 % of the -25 µm, -53/+25 µm, -75/+53 µm and +75 µm fractions respectively.

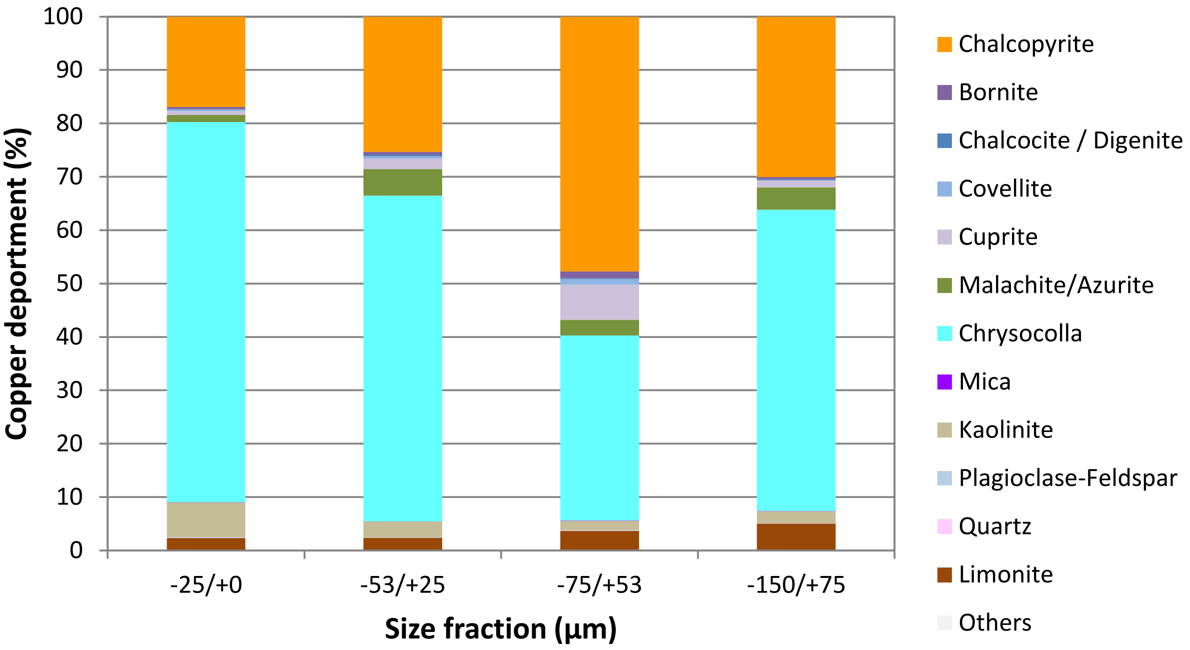


Figure 5.3-3: Normalised copper deportment by size fraction in LQ ore, P_{80} = 75 µm

5.3.3. Chalcopyrite and chrysocolla liberation and mineral associations in LQ ore

The focus for LQ ore was on chalcopyrite and chrysocolla because they were the two most abundant copper minerals. Liberation and mineral association analyses for chalcopyrite were done using Specific Mineral Search data while Particle Mineral Analysis data was used for chrysocolla. Figure 5.3-4 shows the extent of liberation of both minerals at the coarse (P_{80} = 150 µm) and fine (P_{80} = 75 µm) grind. Grinding finer did not seem to increase the liberated fraction of either mineral; About 29.8 % of the chrysocolla (1.1 % out of the 3.8 % in the ore) and 51.5 % of the chalcopyrite (0.5 % out of the 1.0 % in the ore) were fully liberated at a P_{80} of 150 µm compared to 38.9 % chrysocolla and 55.2 % chalcopyrite at 75 µm. No locked chrysocolla particles were identified in LQ ore for both 80 % passing 75 µm and 150 µm.

The middling and locked fractions of the LQ ore ground to 80 % passing 150 µm were defined by their mineral associations. Figure 5.3-5 shows that the chalcopyrite was mostly associated with chrysocolla, viz. 42.9 %. 3.9 % of the chalcopyrite was associated with malachite. Figure 5.3-6 shows the associations between chrysocolla and other minerals at 80 % passing 150 µm. Most of the chrysocolla was associated with limonite, viz. 39.4 %. 1.26 % of the chrysocolla was associated with malachite and 4.0 % with chalcopyrite.

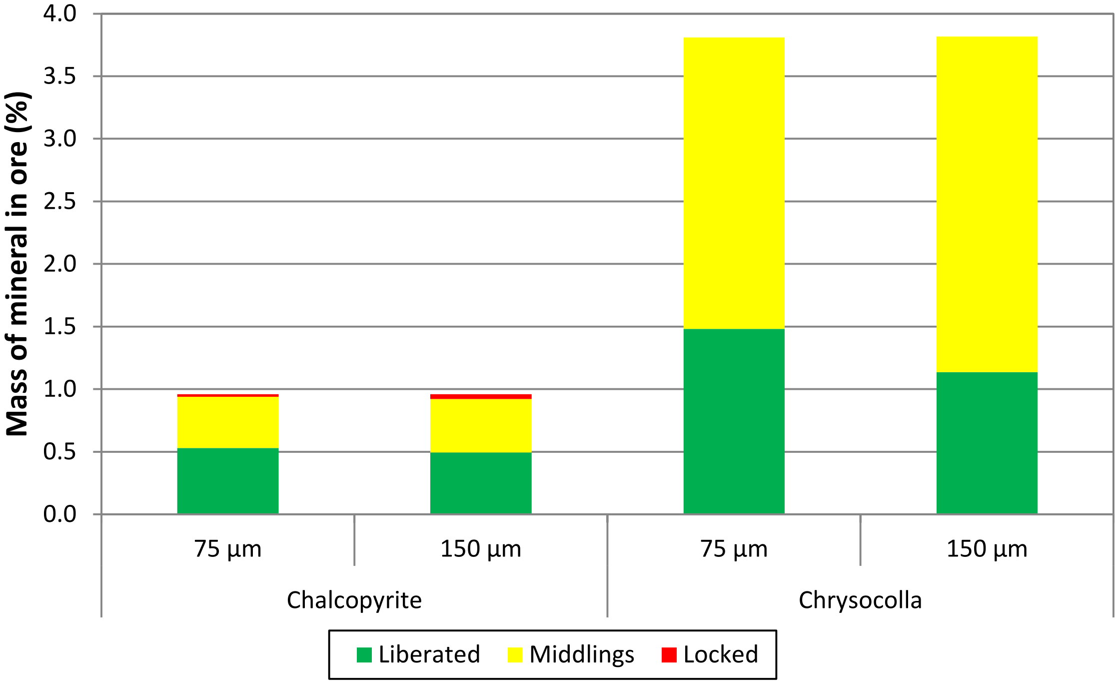


Figure 5.3-4: Chalcopyrite and chrysocolla liberation for coarse (P_{80} = 150 µm) and fine grind (P_{80} =75 µm) of LQ ore

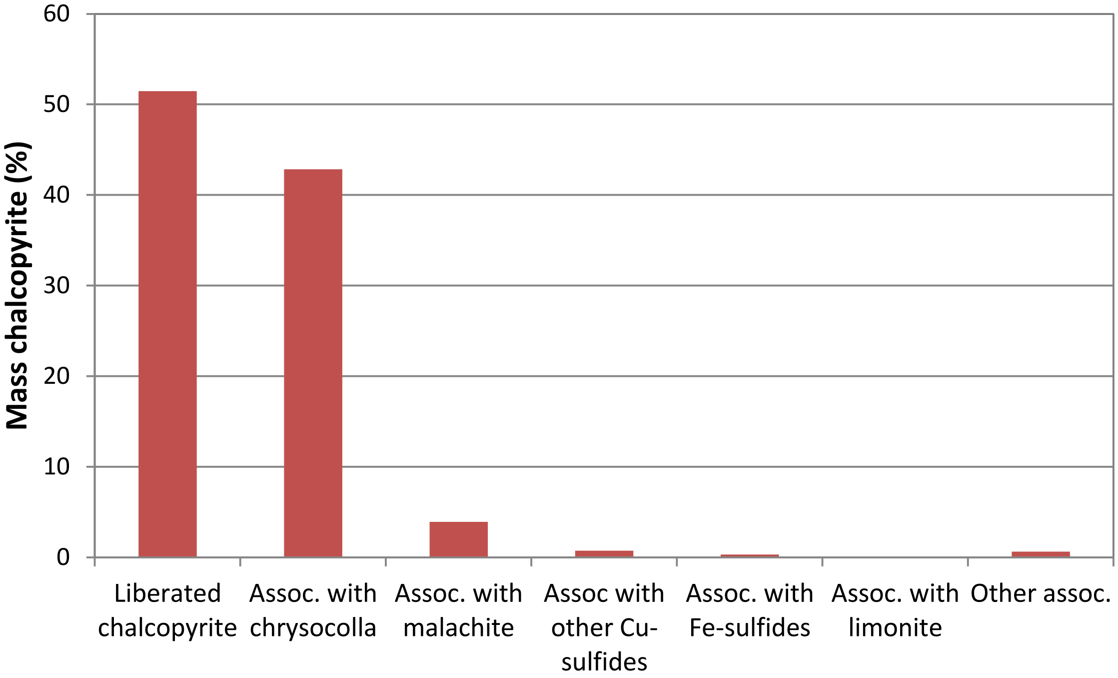


Figure 5.3-5: Associations between chalcopyrite and other minerals in LQ ore, P_{80} = 150 µm

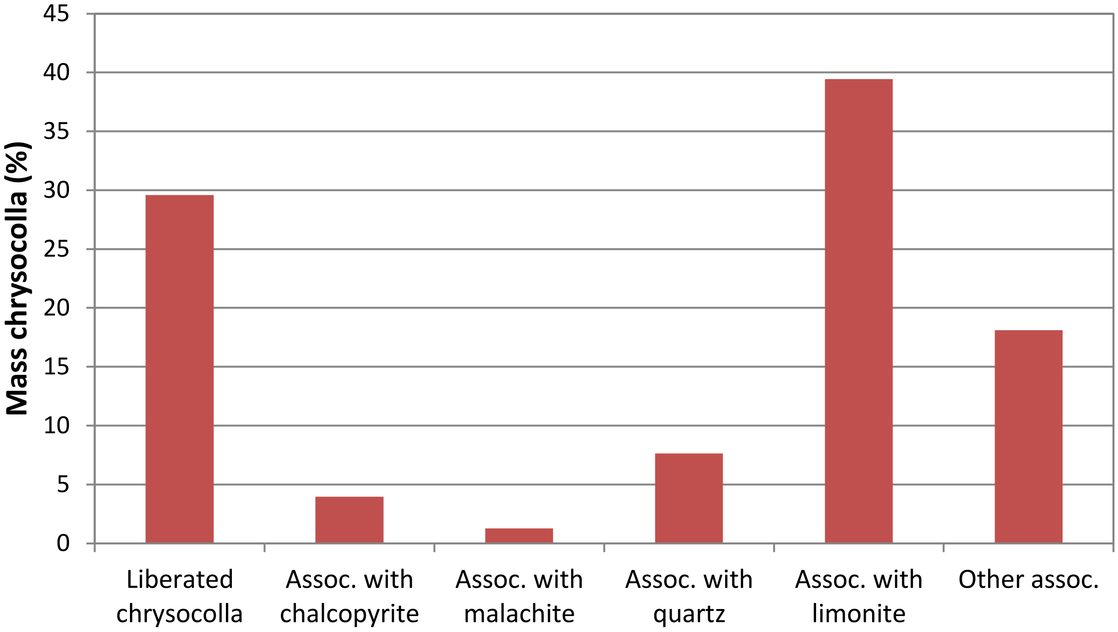


Figure 5.3-6: Associations between chrysocolla and other minerals in LQ ore, P_{80} = 150 µm

5.3.4. Mineralogical definition of Acid Soluble Copper (ASCu)

In order to determine which copper minerals responded to digestion with sulphuric acid when determining the ASCu content of an ore (the procedure has been outlined in section 4.5.1), 6 g of LQ ore was digested. The digestion residue (2.5 g) was subjected to rapid mineralogical analysis. Table 5.3-4 shows the change in copper mineral content by comparing the bulk mineralogy of the feed and residue. The covellite results were disregarded in this case because the mass balance did not close, possibly due to inaccuracy in the mineral assay.

Table 5.3-4: The change in mineral assay in LQ ore after digestion with sulphuric acid to determine ASCu content

Mineral	Feed		ASCu digestion residue		Mass Digested (g)	% digested
	%	Mass (g)	%	Mass (g)		
Chalcopyrite	1.08	0.07	0.68	0.017	0.048	73.8
Bornite	0.02	0.001	0.02	0.001	0.001	58.3
Cuprite	0.06	0.004	0.00	0.000	0.004	100.0
Malachite/Azurite	0.09	0.005	0.01	0.000	0.005	95.3
Chrysocolla	3.89	0.233	0.09	0.002	0.231	99.0

Figure 5.3-7 shows the extent of digestion, determined by expressing the amount of a mineral digested as a percentage of the amount that was contained in the feed. The oxide minerals (cuprite, malachite/azurite and chrysocolla) showed a much greater extent of digestion than the sulphide minerals, with cuprite being completely digested. The fact that at least 58 % of the bornite and 73.8 % of the chalcopyrite also responded to sulphuric acid digestion indicates that perhaps using ASCu as a proxy for the amount of oxide minerals present in the ore is an inappropriate assumption.

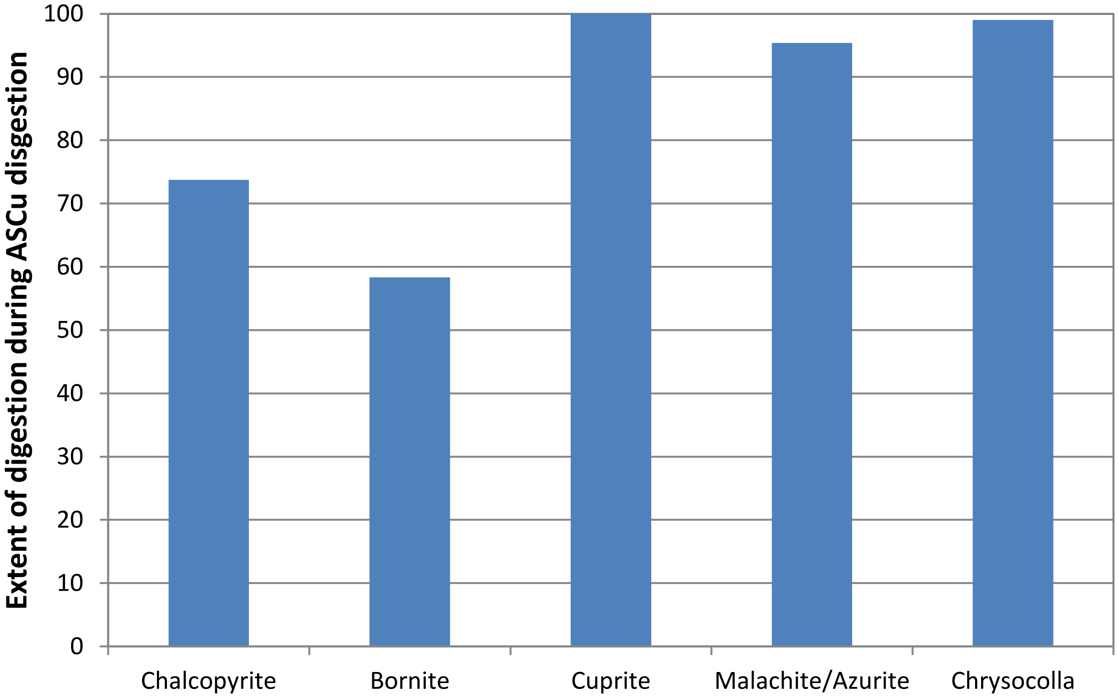


Figure 5.3-7: The extent of copper mineral change when LQ ore was digested with sulphuric acid to determine ASCu

5.3.5. Batch flotation of LQ ore using SIBX for the recovery of chalcopyrite

Initial tests were done to optimise chalcopyrite and secondary copper sulphide mineral recovery using SIBX. 20 g/t, 30 g/t and 50 g/t collector dosages were investigated. Solids and water recoveries for the various dosages are shown in Figure 5.3-8. 30 g/t SIBX gave the highest solids recoveries (24.0 g), followed by 20 g/t (21.7 g) and 50 g/t (20.8 g) per kg of ore floated. The highest water recoveries were observed at 20 g/t and 30 g/t. 50 g/t SIBX resulted in the lowest water recoveries as also observed in HQ ore. It should be noted that the non-linearity of the plots indicates that the recovery of the value mineral is not due to entrainment.

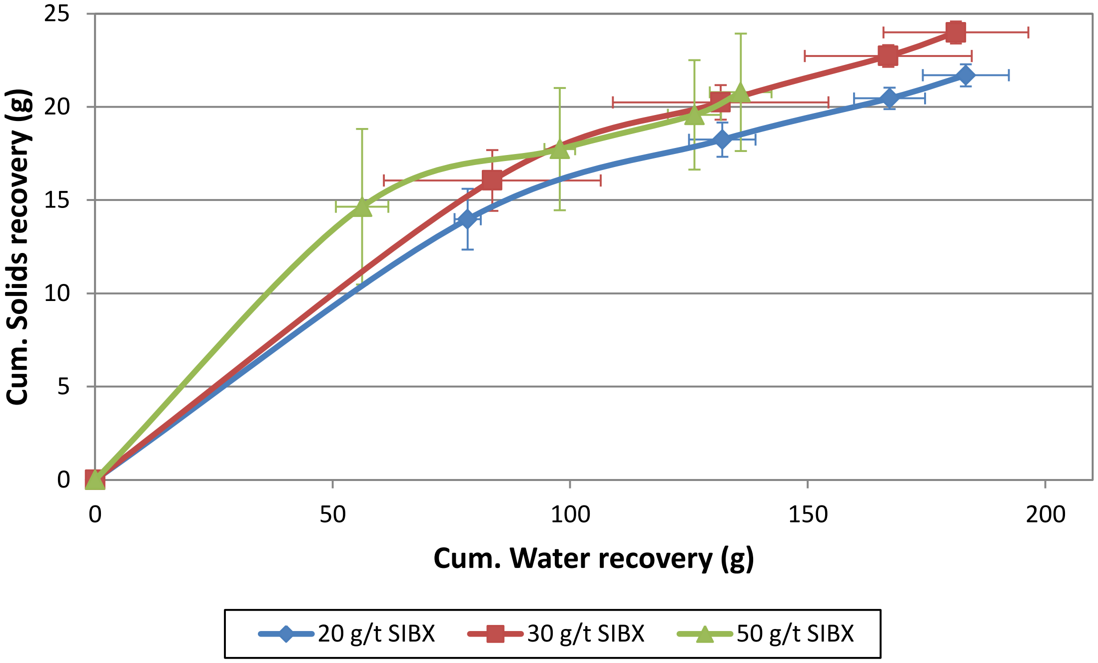


Figure 5.3-8: Cumulative solids versus cumulative water recovery at dosages of 20 g/t, 30 g/t and 50 g/t SIBX on LQ ore. Error bars indicate standard error for duplicate tests

The TCu and Fe recovery-time plots are shown in Figure 5.3-9. 30 g/t SIBX gave the highest recovery and fastest kinetics of the three dosages, recovering 24.7 % of the TCu at the end of 10 minutes. 20.1 % and 15.0 % TCu was recovered for 20 g/t and 50 g/t SIBX respectively. The Fe recovery followed the same trends as TCu recovery, with 30 g/t SIBX giving 14.3 % Fe recovery at the end of 10 minutes. Although SIBX is probably only capable of recovering the sulphide minerals, it was relevant to this study to investigate whether any oxide copper minerals (reported as ASCu) were recovered by SIBX. Figure 5.3-10 shows the ASCu recovery-time plots for 20 g/t, 30 g/t and 50 g/t SIBX. The ASCu recovered also followed the TCu trends. 6.5 % of the ASCu was recovered at 30 g/t SIBX. 20 g/t SIBX recovered 4.8 % and 50 g/t SIBX recovered 3.7 % of the ASCu.

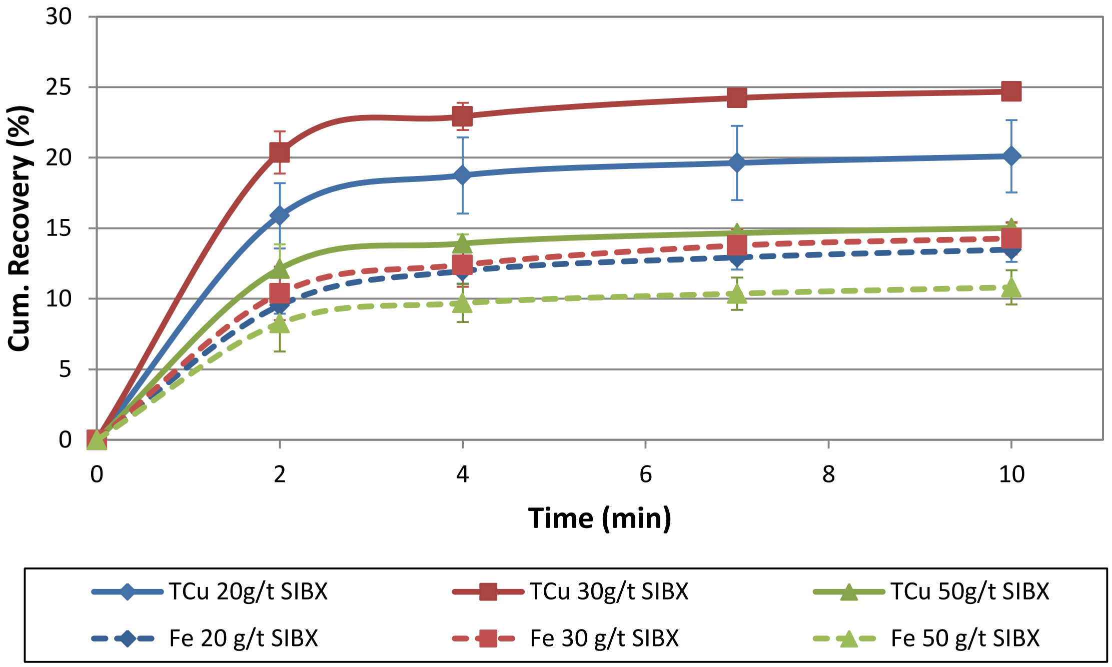


Figure 5.3-9: TCu and Fe recovery-time plots for sulphide mineral recovery optimisation in LQ ore, using 20 g/t, 30 g/t and 50 g/t SIBX. Error bars indicate standard deviation of duplicate tests

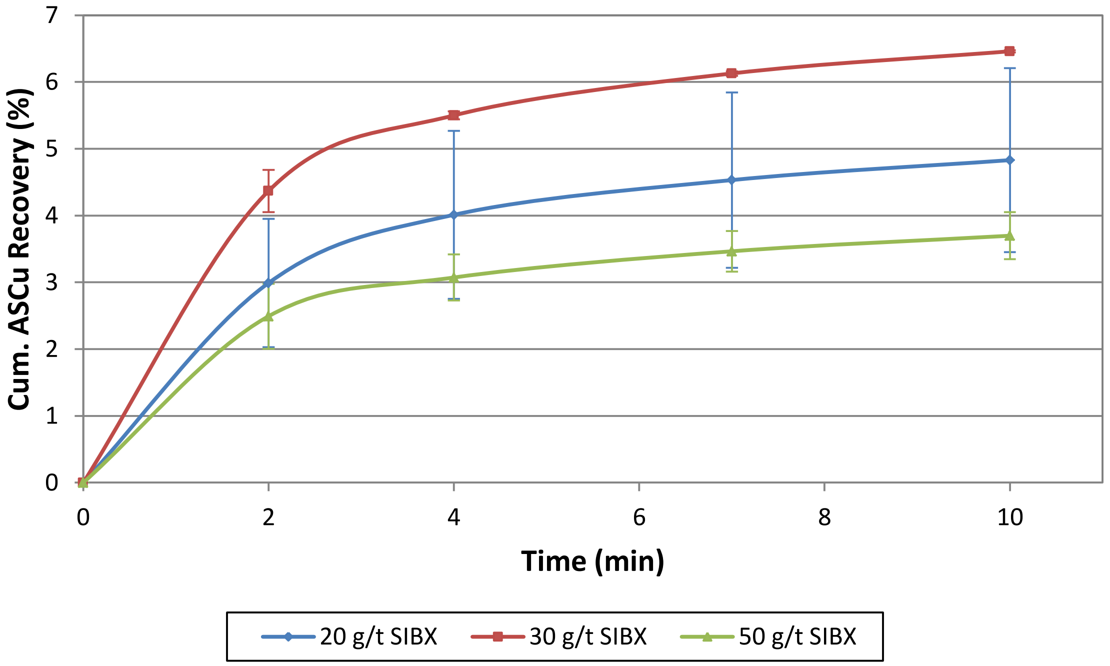


Figure 5.3-10: ASCu recovery-time plots for sulphide mineral recovery optimisation in LQ ore using 20 g/t, 30 g/t and 50 g/t SIBX. Error bars indicate standard deviation for duplicate tests

Figure 5.3-11 shows the grade-recovery trends for LQ ore using SIBX only. Both TCu recovery and grade increased with an increase in collector dosage from 20 g/t to 30 g/t and dropped off when collector dosage was increased to 50 g/t. 30 g/t SIBX gave the highest TCu grade of 14.5 %. Increasing the collector dosage resulted in reduced concentrate Fe grades. Final Fe concentrate grades at the end of 10 minutes were 21.5 %, 20.5 % and 19.5 % for 20 g/t, 30 g/t and 50 g/t SIBX respectively.

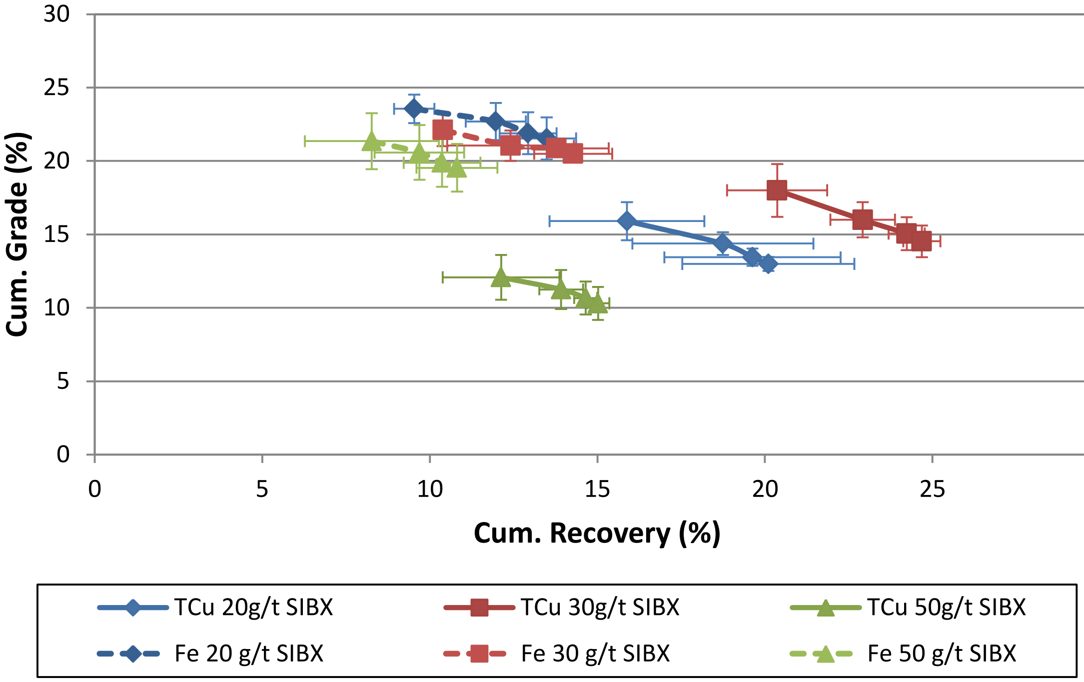


Figure 5.3-11: TCu grade versus recovery for flotation of LQ ore with 20 g/t, 30 g/t and 50 g/t SIBX. Error bars indicate standard deviation for duplicate tests

30 g/t SIBX was selected as the best dosage for sulphide mineral recovery. The tailings of the 30 g/t SIBX dosage were subjected to mineralogical analysis using QEMSCAN. Table 5.3-5 compares the bulk mineralogy of the feed to that of the tailings after flotation with 30 g/t SIBX. Only 80.8 % of the chalcopyrite was recovered. 0.9 % of the chrysocolla and 66.7 % of the malachite were recovered by the SIBX. The gangue minerals that were recovered included 75.8 % of the pyrite, 32.6 % pyrrhotite, 14.2 % mica and 18.7 % quartz. Figure 5.3-12 shows the copper deportment in the LQ ore tailings after flotation with 30 g/t SIBX. 5.8 % of the copper that was not recovered was chalcopyrite and 1.2 % was malachite, but the bulk of the copper in the tailings was chrysocolla. The associations between chalcopyrite and other minerals in the tailings are shown in Figure 5.3-13. 74.8 % of the chalcopyrite was fully liberated and should have easily been recovered but was not. 18.6 % and 3.3 % of the unrecovered chalcopyrite was associated with chrysocolla and malachite respectively.

Table 5.3-5: Comparison of LQ ore feed and tailings bulk mineralogy after flotation with 30 g/t SIBX

Mineral	LQ ore mineral content (%)		% Recovered
	Feed	Tailings after flotation with 30 g/t SIBX	
Pyrite	1.0	0.3	75.8
Pyrrhotite	0.7	0.5	32.6
Chalcopyrite	1.0	0.2	80.8
Bornite	<0.1	<0.1	65.7
Chalcocite / Digenite	<0.1	<0.1	68.6
Covellite	<0.1	<0.1	66.2
Other sulphides	0.1	0.1	0
Cuprite	<0.1	<0.1	72.9
Malachite/Azurite	0.1	<0.1	66.8
Chrysocolla	3.8	3.9	0
Amphibole	3.8	6.2	0
Mica	9.2	7.4	14.2
Kaolinite	1.7	2.1	0
Plagioclase-Feldspar	23.8	24.0	0
Quartz	16.4	13.4	18.7
Calcite	29.1	30.3	0
Fe-Ti minerals	1.1	3.6	0
Limonite	6.7	6.6	0
Others	1.7	1.5	36.9

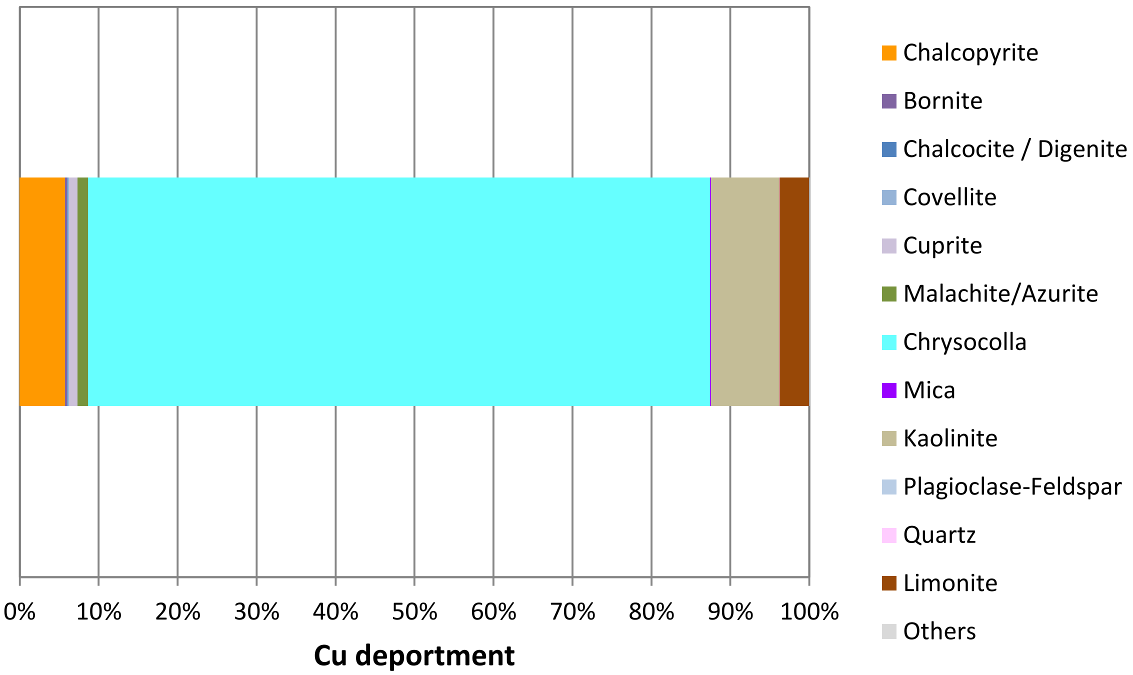


Figure 5.3-12: Copper deportment in LQ ore tailings after flotation with 30 g/t SIBX

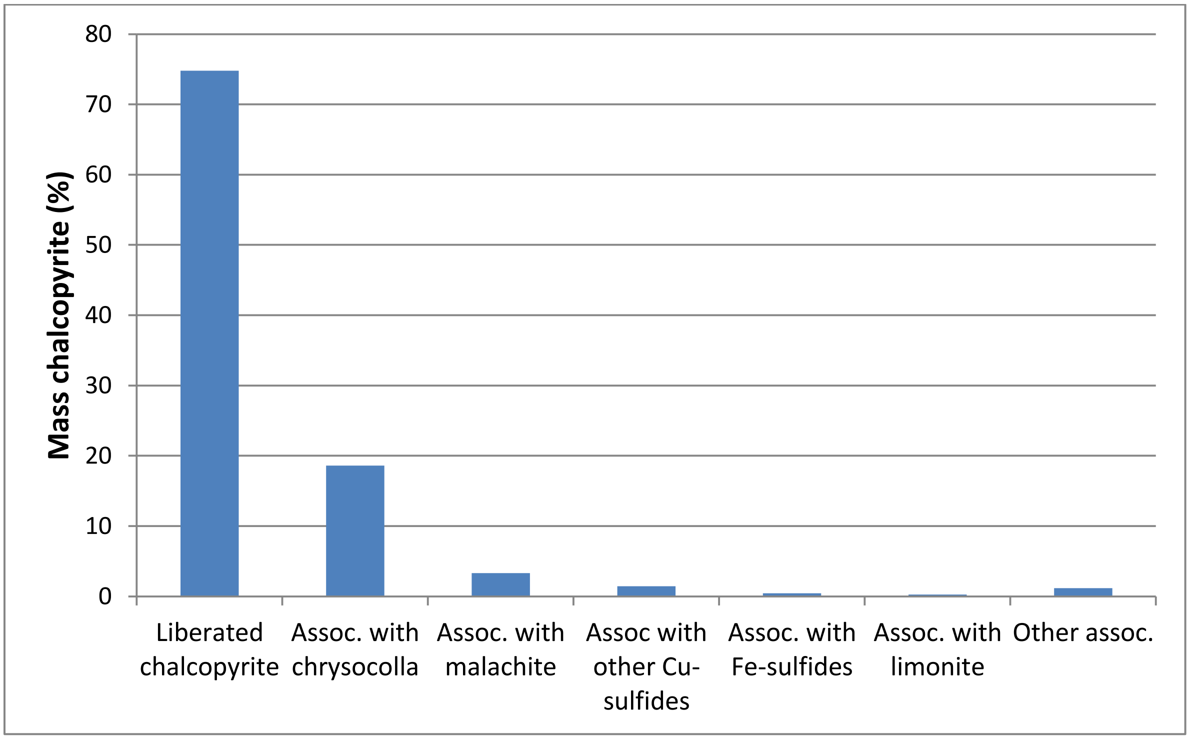


Figure 5.3-13: Associations between chalcopyrite and other minerals in LQ ore tailings after flotation with 30 g/t SIBX

5.3.6. Slug sulphidisation of LQ ore

The results in this section are presented such that the sulphide copper recovery stage is averaged across all the tests done and the sulphidisation stages are separated to emphasise the step change observed for different sulphidisation conditions (Figure 5.3-14). The error bars in the “sulphide Cu recovery” section therefore represent standard error between four tests, while those in the “Oxide Cu recovery” section represent standard error between duplicate tests.

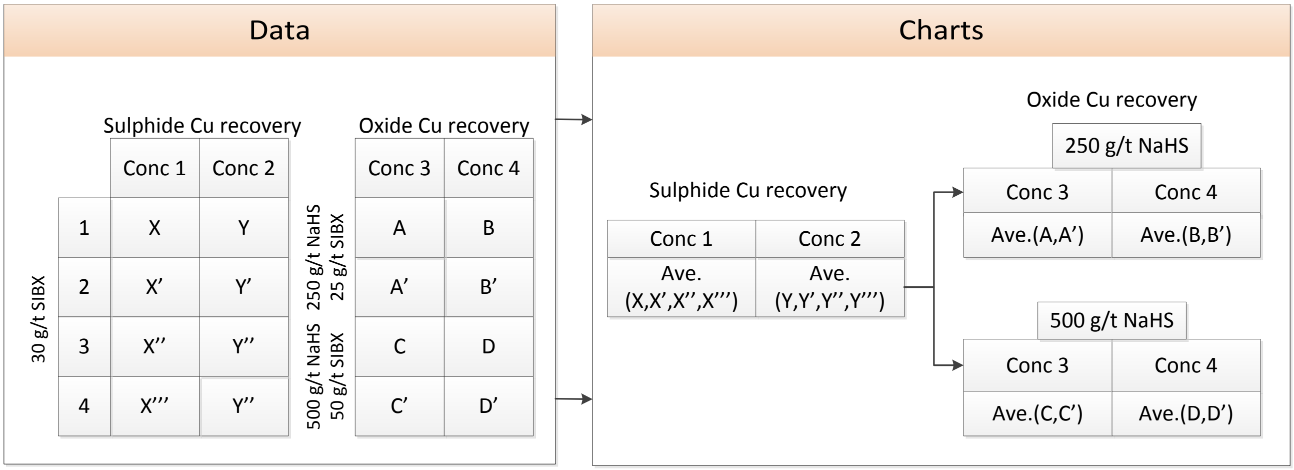


Figure 5.3-14: Data analysis model for slug sulphidisation tests on LQ ore

Based on the copper flotation results, SIBX dosage of 30 g/t and flotation time of four minutes were selected as the conditions for the pre-sulphidisation procedure. All the Cu oxide mineral recovery tests were done by first floating the Cu sulphides (Conc. 1 and Conc. 2) and then floating the oxides. 250 g/t or 500 g/t of NaHS was added to the pulp after sulphide Cu recovery, viz. after 4 minutes. The pulp was then conditioned for 2 minutes after which SIBX dosage amounting to 10 % of the NaHS used (25 g/t SIBX for 250 g/t NaHS or 50 g/t SIBX for 500 g/t NaHS) was added. Figure 5.3-15 shows the solids and water recovery profiles for the two test conditions. In both cases, the introduction of NaHS destabilized the froth, leading to greater variations in the amount of water recovered. The two NaHS dosages and their respective SIBX dosages gave the same solids recoveries (about 31 g) at the end of the flotation period but slug sulphidisation with 500 g/t NaHS followed by flotation with 50 g/t SIBX gave higher water recoveries overall.

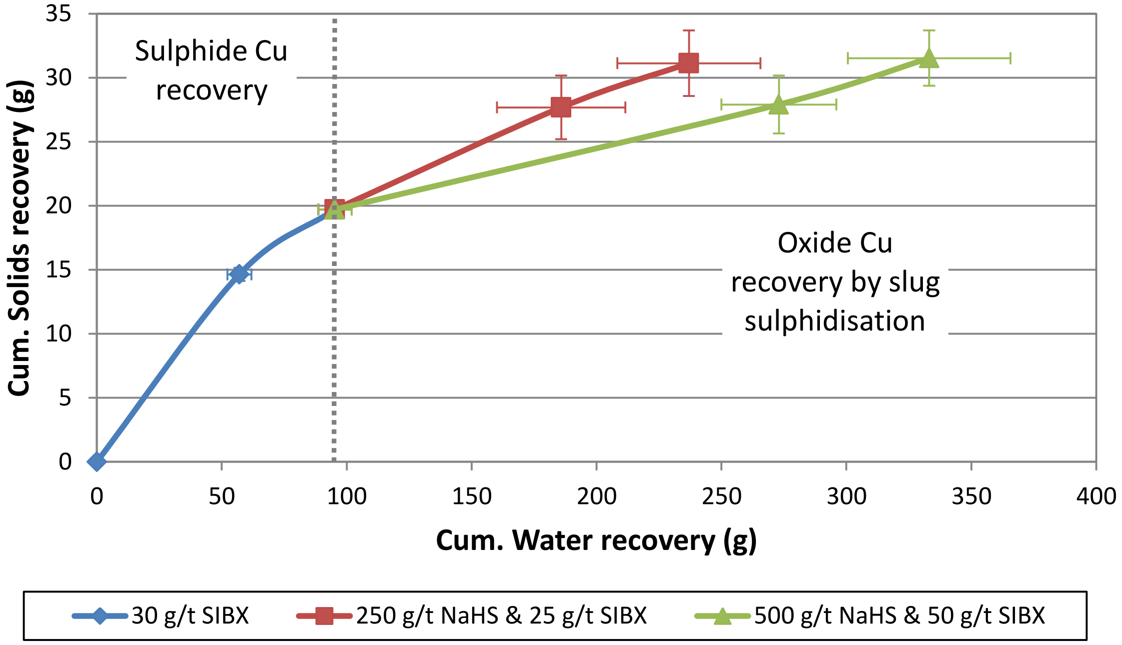


Figure 5.3-15: Cumulative solids versus cumulative water recovery for slug sulphidisation of LQ ore using 250 g/t NaHS (and 25 g/t SIBX) and 500 g/t NaHS (and 50 g/t SIBX). Error bars indicate standard error for duplicate tests. The vertical dotted line represents the point at which NaHS was slug added and conditioned for 2 min followed by the addition of SIBX (also conditioned for 2 min) before continuing with flotation

The TCu and Fe recovery-time plots for the two conditions are shown in Figure 5.3-16. There was an increase in both copper and Fe recovery after sulphidisation. TCu recovery at the end of 14 minutes was 28.5 % when 250 g/t NaHS and 25 g/t SIBX was used and increased to 31.2 % when 500 g/t NaHS and 50 g/t SIBX was used. In both cases, sulphidisation led to an increase in Fe recovery as well. The Fe recovery followed the same trends as the TCu recovery with 500 g/t NaHS and 50 g/t SIBX giving 16.4 % Fe recovery while 250 g/t NaHS and 25 g/t SIBX giving 12.7 % Fe recovery.

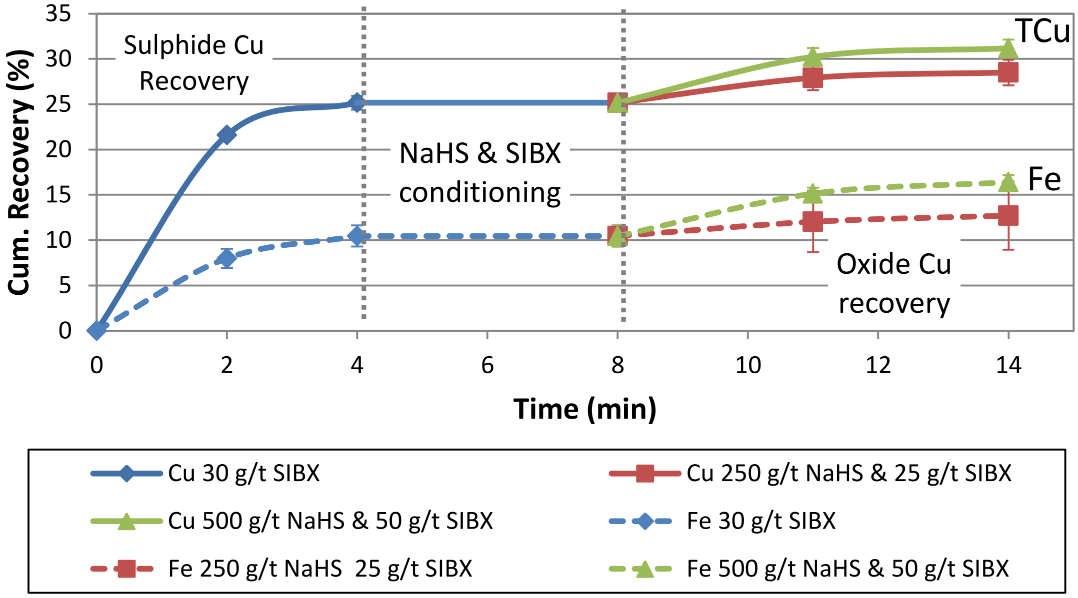


Figure 5.3-16: TCu and Fe recovery-time plots for slug sulphidisation of LQ ore using 250 g/t NaHS (and 25 g/t SIBX) and 500 g/t NaHS (and 50 g/t SIBX)

The same observation was made for ASCu recovery (Figure 5.3-17). ASCu recovery however only increased from 5.3 % pre-sulphidisation to 10.2 % after sulphidisation in the best case, viz. 500 g/t NaHS and 50 g/t SIBX. Since ASCu is a component of TCu, this implies that at the end of 14 minutes, 10.2 % of the 31.2 % (i.e. 32.8 %) TCu recovered was in fact ASCu.

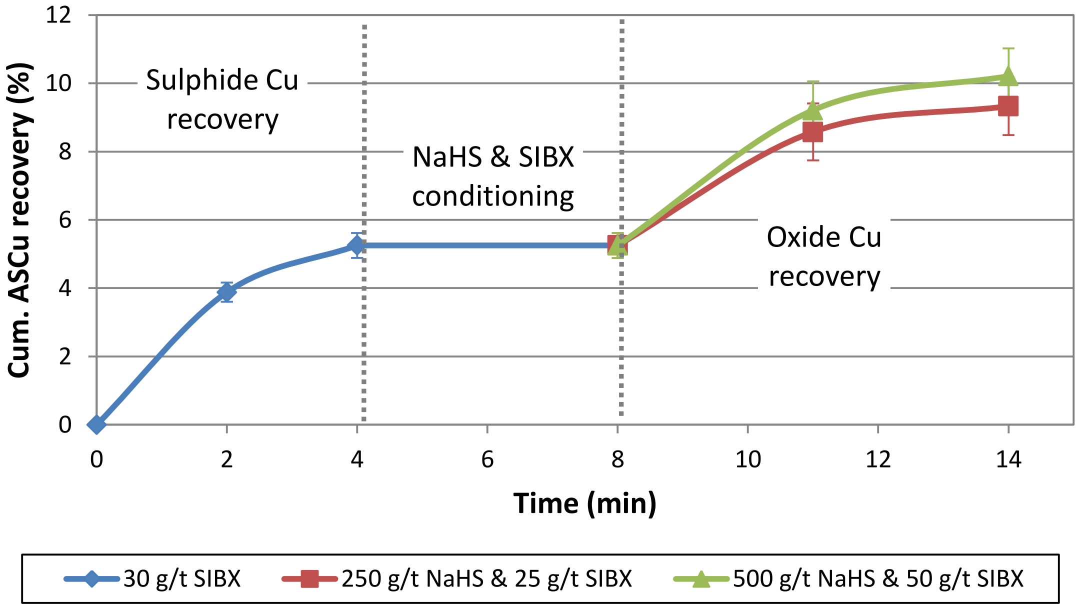


Figure 5.3-17: ASCu recovery-time plots for slug sulphidisation of LQ ore using 250 g/t NaHS (and 25 g/t SIBX) and 500 g/t NaHS (and 50 g/t SIBX)

Figure 5.3-18 shows the grade vs. recovery trends for Fe and TCu. Slug sulphidisation using 500 g/t NaHS and 50 g/t resulted in a higher TCu recovery (31.2 %). The two dosages however gave comparable final copper grades, viz. 14.7 % for 250 g/t NaHS and 25 g/t SIBX and 15.5 % for 500 g/t NaHS and 50 g/t SIBX. Higher NaHS and SIBX dosage during sulphidisation also led to an increase in Fe recovery and grade, viz. 16.4 % and 17.3 % respectively.

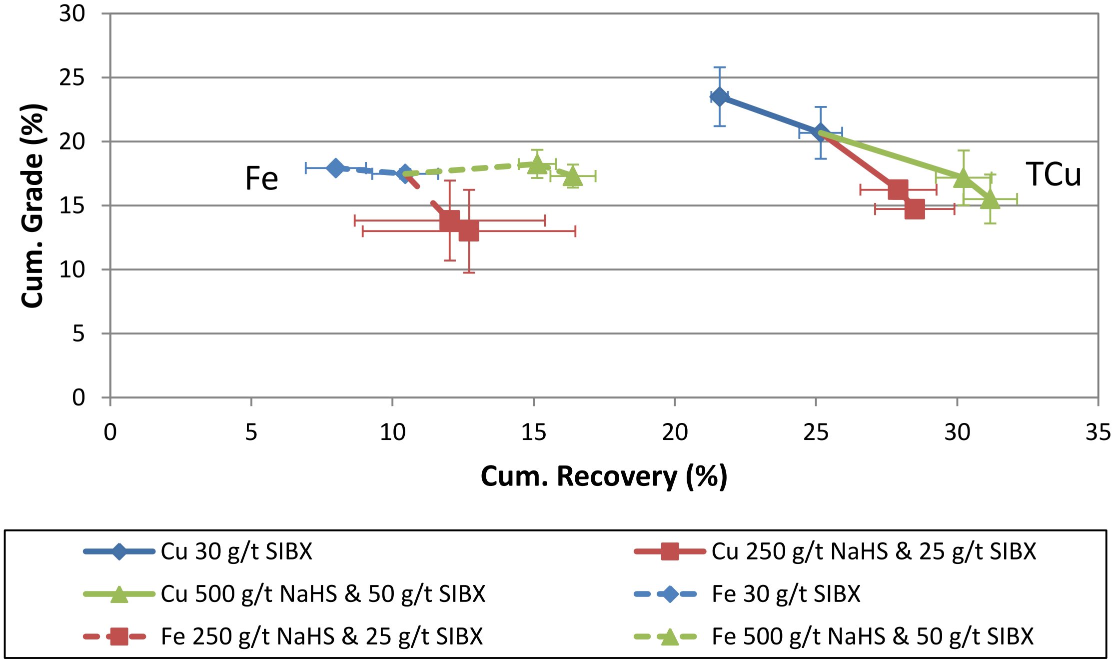


Figure 5.3-18: TCu and Fe grade versus recovery for slug sulphidisation of LQ ore using 250 g/t NaHS (and 25 g/t SIBX) and 500 g/t NaHS (and 50 g/t SIBX). The blue lines are common to both NaHS dosages as they indicate the pre-sulphidisation stage of the grade-recovery curve

5.3.7. Controlled Potential Sulphidisation (CPS) of LQ ore

The results in this section present the data obtained in the case of CPS (c.f. Table 4.6-5 for the procedure). As with the slug sulphidisation, the results from the pre-sulphidisation stage are averaged into one series and the NaHS:SIBX ratios investigated during CPS branch out from it in order to emphasise the changes. The CPS potential ranges are cited as E_S, which is different from E_h (c.f. glossary)

i. CPS at E_S range of -400 to -500 mV

Figure 5.3-19 shows the cumulative solids vs. water recovered for the different SIBX collector dosages after CPS. The three ratios had an average gradient of 0.063 g (±0.001) solids per g water recovered. The 20 NaHS:1 SIBX ratio gave the highest solids and water recovery at the end of flotation.

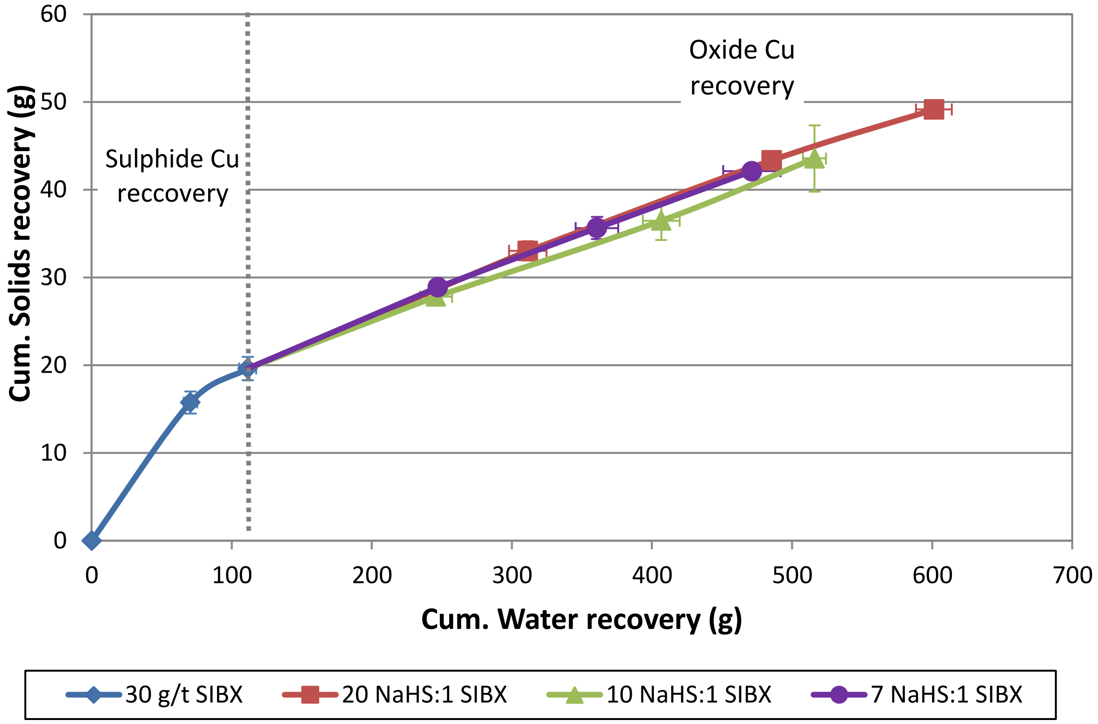


Figure 5.3-19: Cumulative solids versus cumulative water recovery at the E_S potential range of -400 mV to -500 mV on LQ ore using the following NaHS:SIBX ratios: 20:1, 10:1 and 7:1. The vertical dotted line represents the first of three CPS stages at which NaHS was slowly added to the pulp for 3 min followed by the addition of SIBX (conditioned for 2 min) before continuing with flotation

Figure 5.3-20 shows that CPS resulted in an increase in both TCu and Fe recoveries. The highest copper recovery of 32.8 % was achieved for a NaHS:SIBX ratio of 20:1. 10 NaHS:1 SIBX and 7 NaHS:1 SIBX gave 28.5 % and 29.8 % TCu recovery respectively. All three ratios however gave very similar Fe recovery profiles. Average Fe recovery at the end of the test across all three ratios was 18.2 %. ASCu recovery-time plots are shown in Figure 5.3-21. As with the TCu, the 20 NaHS:1SIBX ratio resulted in the most significant increase in ASCu recovery after CPS. Final ASCu recoveries were 14.2 %, 10.8 % and 10.7 % for 20 NaHS:1 SIBX, 10 NaHS:1 SIBX and 7 NaHS:1 SIBX respectively. CPS with a NaHS:SIBX ratio of 20:1 resulted in a higher amount of ASCu recovery compared to slug sulphidisation with 500 g/t NaHS (c.f. Figure 5.3-17). The ASCu recovery in the latter case being 10.21% compared to 14.2 % in the case of CPS.

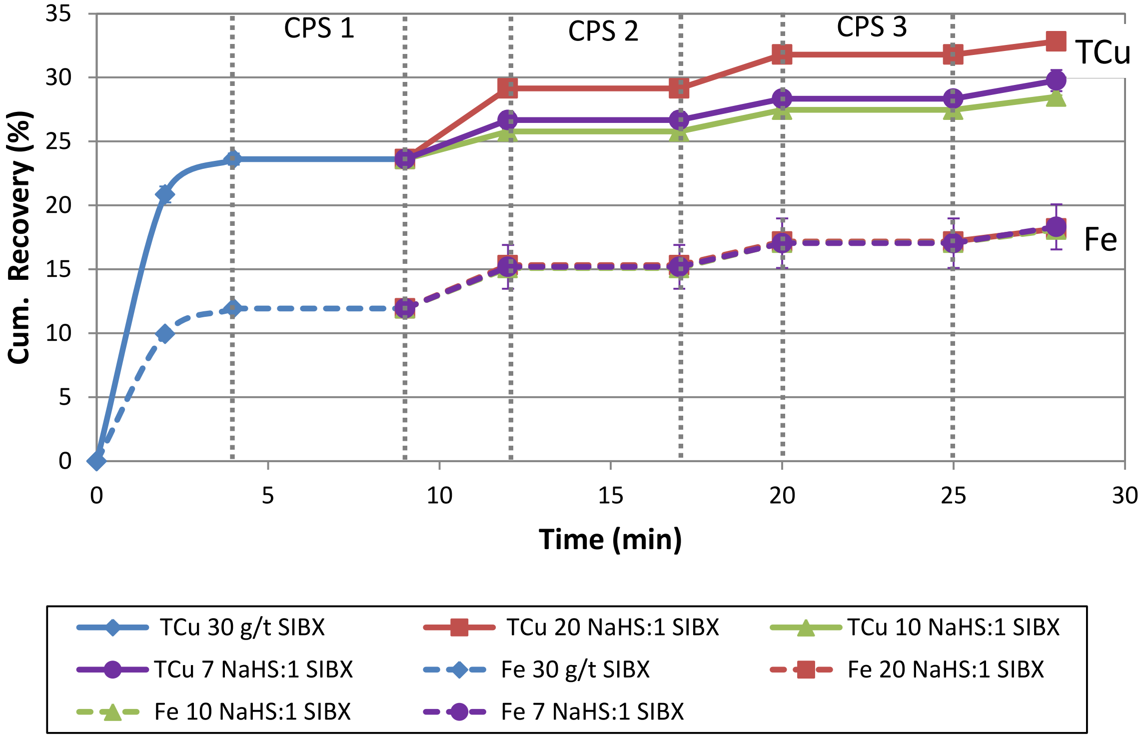


Figure 5.3-20: TCu and Fe recovery-time plots for CPS on LQ ore at the E_S potential range of -400 mV to -500 mV and the following NaHS:SIBX ratios; 20:1, 10:1 and 7:1

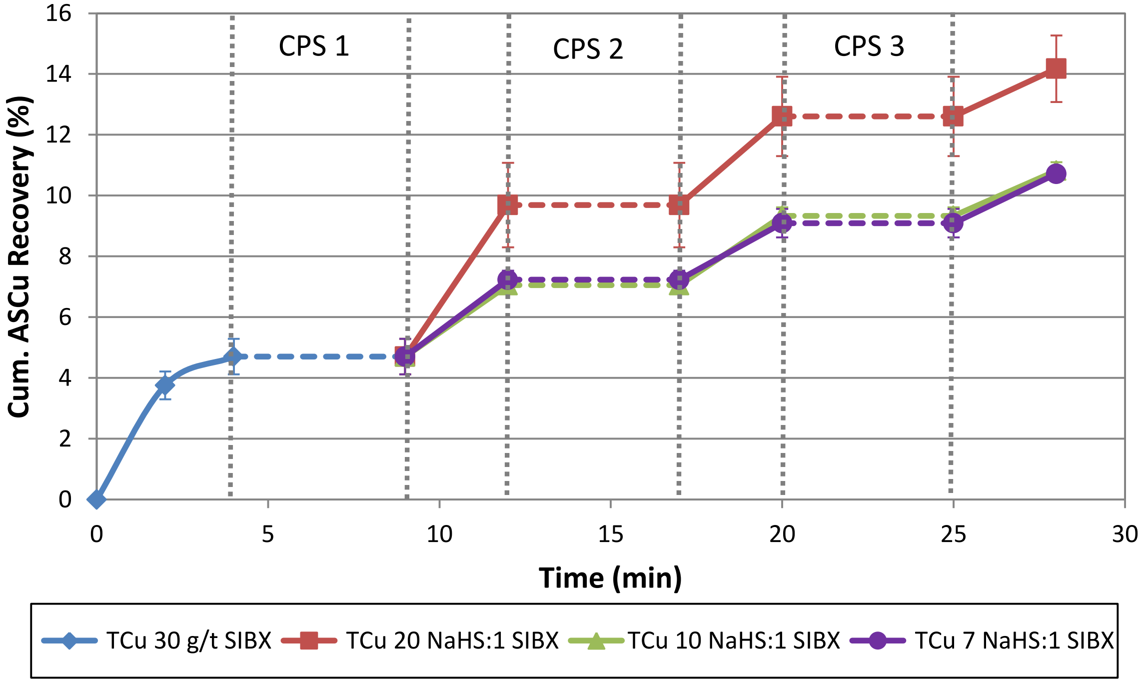


Figure 5.3-21: ASCu recovery-time plots for CPS on LQ ore at the Es potential range of -400 mV to -500 mV and the following NaHS:SIBX ratios; 20:1, 10:1 and 7:1

TCu and Fe grade versus recovery curves for the potential range (-400 to -500 mV) are shown in Figure 5.3-22. The final TCu grade attained for all three ratios was about 10 %. This was less than the 15.5 % observed for slug sulphidisation with 500 g/t NaHS and 50 g/t SIBX (c.f. Figure 5.3-18). The 20 NaHS:1 SIBX ratio resulted in the lowest Fe grade of 12.6 % while 10 NaHS:1 SIBX and 7 NaHS:1 SIBX had 14.4 % and 14.6 % Fe respectively.

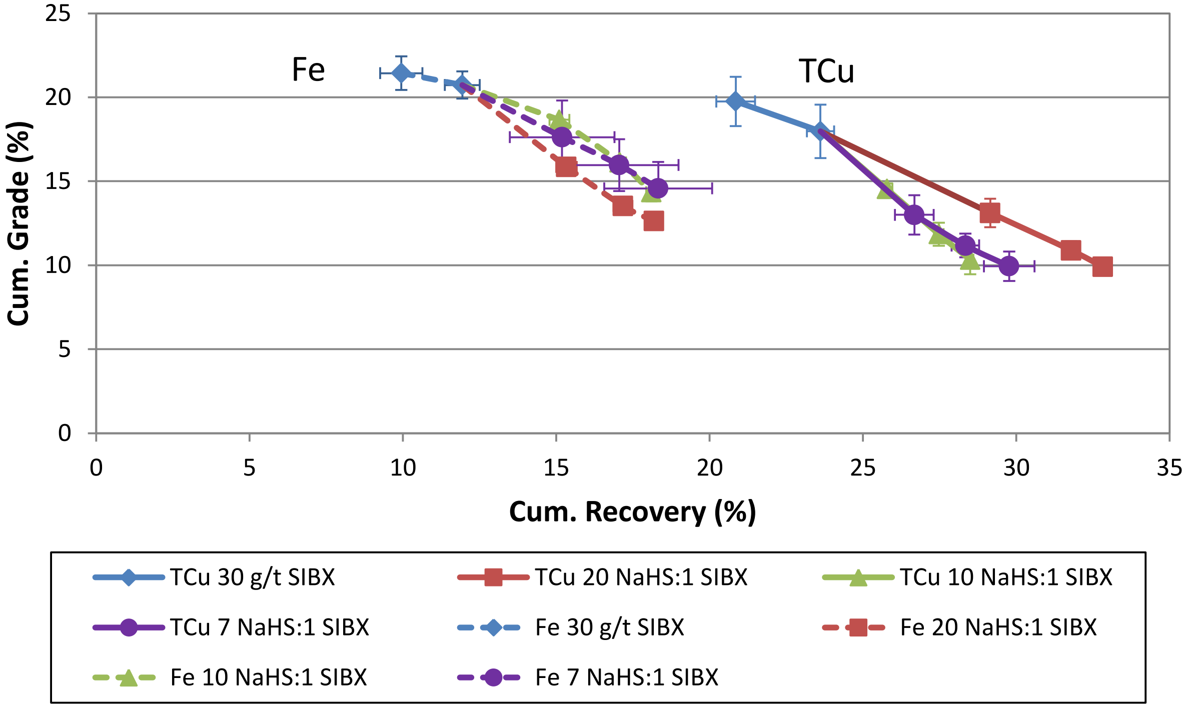


Figure 5.3-22: TCu and Fe grade versus recovery for CPS on LQ ore at the E_S potential range of -400 mV to -500 mV using the following NaHS:SIBX ratios: 20:1, 10:1 and 7:1. The blue lines are common to all NaHS:SIBX ratios as they indicate the pre-sulphidisation stage of the grade-recovery curve

ii. CPS at E_S range of -300 to -400 mV

Figure 5.3-23 shows the solids and water recovery obtained for the different NaHS:SIBX ratios at the CPS potential range of -300 to -400 mV. The highest water recovery was observed for 7 NaHS:1 SIBX. Cumulative solids recovery at the end of the flotation tests was 39.6 g, 38.8 g and 43.9 g for 20 NaHS: 1 SIBX, 10 NaHS:1 SIBX and 7 NaHS:1 SIBX respectively.

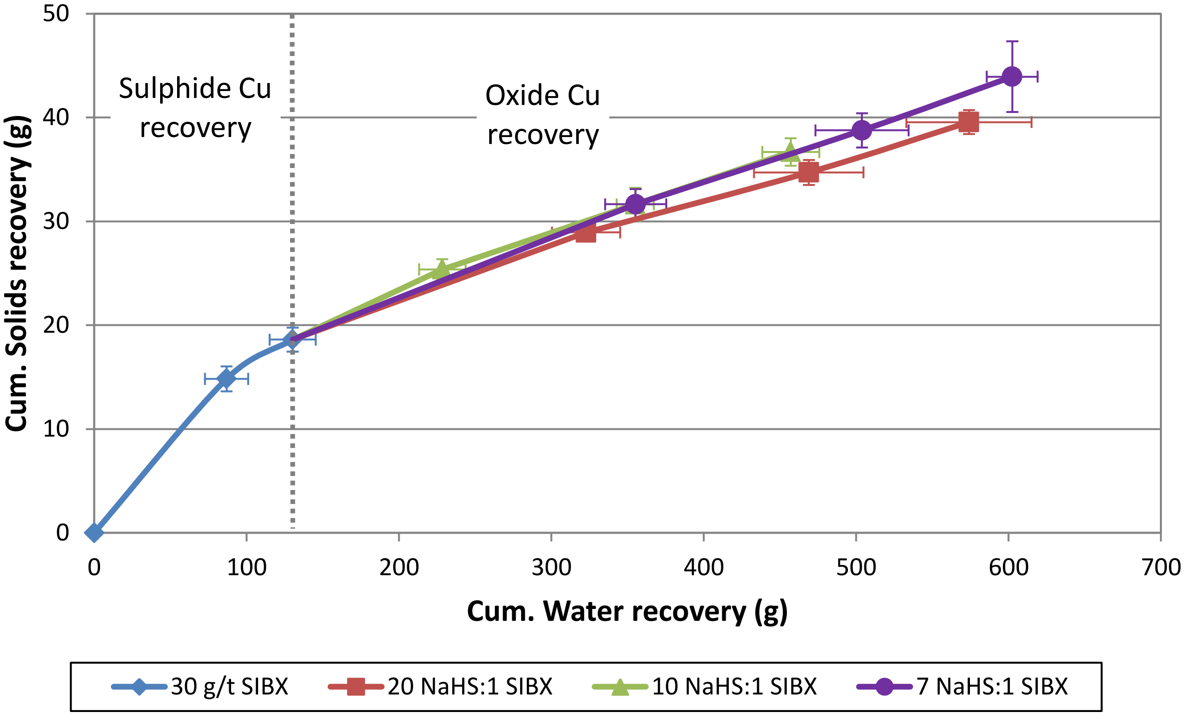


Figure 5.3-23: Cumulative solids versus cumulative water recovery at the E_S potential range of -300 mV to -400 mV on LQ ore using the following NaHS:SIBX ratios: 20:1, 10:1 and 7:1. The vertical dotted line represents the first of three CPS stages at which NaHS was slowly added to the pulp for 3 min followed by the addition of SIBX (conditioned for 2 min) before continuing with flotation

Figure 5.3-24 shows that there was no significant difference in TCu and Fe recovery across the three ratios tested. 30.1 %, 29.3 % and 29.3 % TCu was recovered for 20 NaHS: 1 SIBX, 10 NaHS:1 SIBX and 7 NaHS:1 SIBX respectively. An average of 17.3 % (±0.29) Fe was recovered. Figure 5.3-25 shows that the ASCu recovery increased from 4 % before sulphidisation to 9.6 %, 8.1 % and 8.6 % for 20:1, 10:1 and 7:1 NaHS:SIBX ratios respectively after CPS.

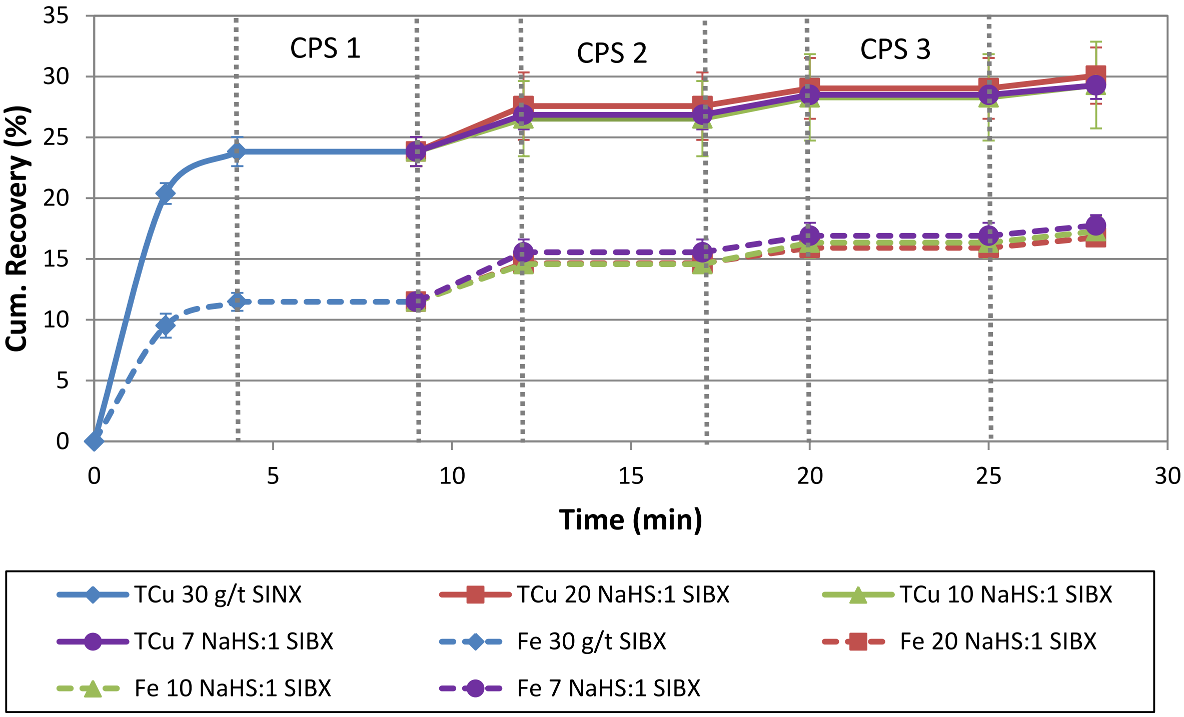


Figure 5.3-24: TCu and Fe recovery-time plots for CPS on LQ ore at the E_S potential range of -300 mV to -400 mV and the following NaHS:SIBX ratios; 20:1, 10:1 and 7:1

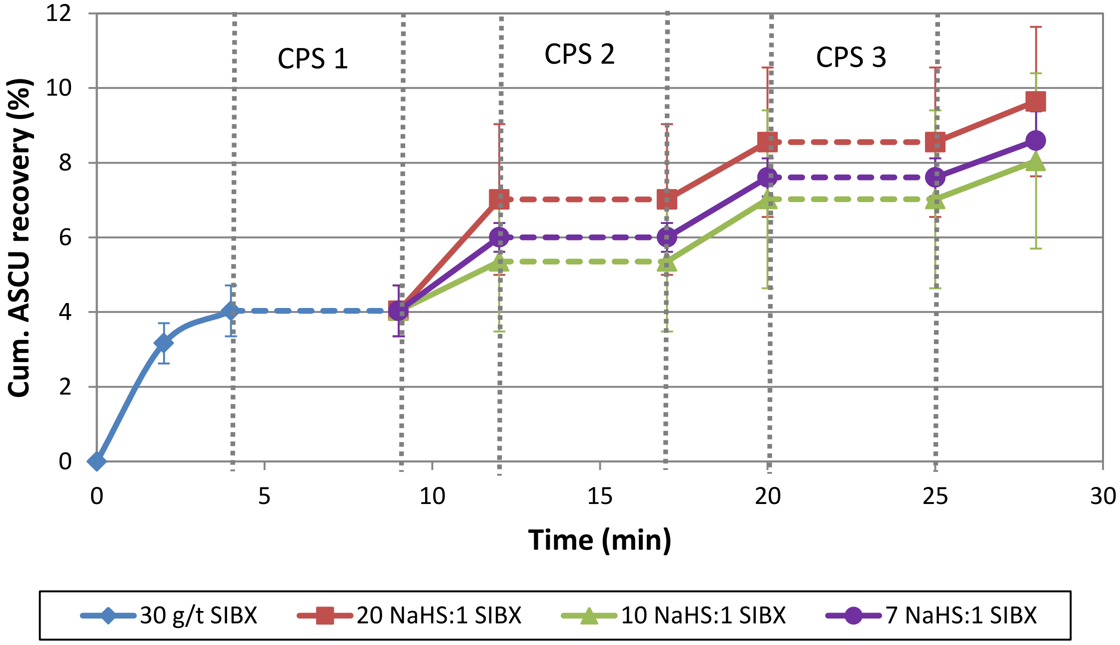


Figure 5.3-25: ASCu recovery-time plots for CPS on LQ ore at the E_S potential range of -300 mV to -400 mV and the following NaHS:SIBX ratios; 20:1, 10:1 and 7:1

The Final TCu and Fe grades did not differ across the NaHS:SIBX ratios (Figure 5.3-26). Final concentrate grade was about 10.0 % TCu and 15.0 % Fe. The final TCu concentrate grade for CPS at -300 to -400 mV was similar to that observed at the more negative potential range of -400 to -500 mV (c.f. Figure 5.3-22) but also proved lower than the grade observed for slug sulphidisation using 500 g/t NaHS (c.f. Figure 5.3-18).

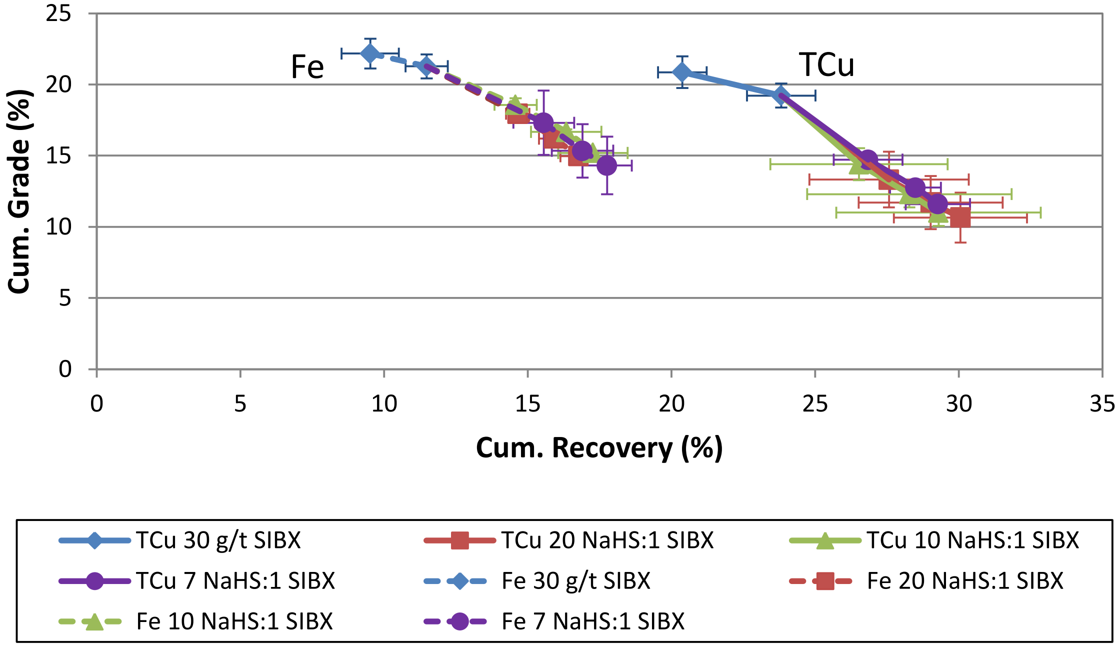


Figure 5.3-26: TCu and Fe grade versus recovery for CPS on LQ ore at the E_S potential range of -300 mV to -400 mV using the following NaHS:SIBX ratios: 20:1, 10:1 and 7:1. The blue lines are common to all NaHS:SIBX ratios as they indicate the pre-sulphidisation stage of the grade-recovery curve

Since the two sulphidisation techniques investigated (slug and CPS) gave comparable copper recoveries, tailings mineralogy for sulphidisation tails was done on the basis of which technique resulted in the best concentrate grade. Slug sulphidisation with 500g/t NaHS gave the highest concentrate grade of 15.5 % Cu. The bulk mineralogy of LQ ore tailings after sulphidisation with 500 g/t NaHS is compared to that of the tailings after flotation with only 30 g/t SIBX (i.e. the feed into the slug sulphidisation) in Table 5.3-6. 88.5 % of the chalcopyrite that had not been recovered by the 30 g/t SIBX was recovered when the tailings were sulphidised, along with 93.3 % and 33.3 % of the cuprite and malachite respectively. Most notably however is the fact that no chrysocolla was recovered by sulphidisation. Slug sulphidisation also led to the recovery of some gangue minerals, viz. 76.4 % pyrite, 52.6 % pyrrhotite and 8.9 % calcite.

Table 5.3-6: LQ ore tailings mineralogy after flotation with 30 g/t SIBX compared to tailings after an extra stage of slug sulphidisation with 500 g/t NaHS and 50 g/t SIBX

Mineral	Tailings mineral content (%)		% Recovered
	30 g/t SIBX	500 g/t NaHS + 50 g/t SIBX	
Pyrite	0.3	0.1	76.4
Pyrrhotite	0.5	0.2	52.6
Chalcopyrite	0.2	<0.1	88.5
Bornite	<0.1	<0.1	93.5
Chalcocite / Digenite	<0.1	<0.1	98.8
Covellite	<0.1	<0.1	8.2
Other sulphides	0.1	<0.1	88.8
Cuprite	<0.1	<0.1	93.3
Malachite/Azurite	<0.1	<0.1	33.3
Chrysocolla	3.9	4.0	0
Amphibole	6.2	5.5	7.5
Mica	7.4	10.4	0
Kaolinite	2.1	2.0	4.0
Plagioclase-Feldspar	24.0	25.5	0
Quartz	13.4	15.9	0
Calcite	30.3	27.8	8.9
Fe-Ti minerals	3.6	1.0	73.3
Limonite	6.6	6.4	7.1
Others	1.5	1.2	39.0

The copper deportment of the tailings after slug sulphidisation with 500 g/t NaHS is shown in Figure 5.3-27. 84.8 % of the copper that was not recovered was hosted in chrysocolla, 9.2 % in kaolinite and 4.2 % in limonite.

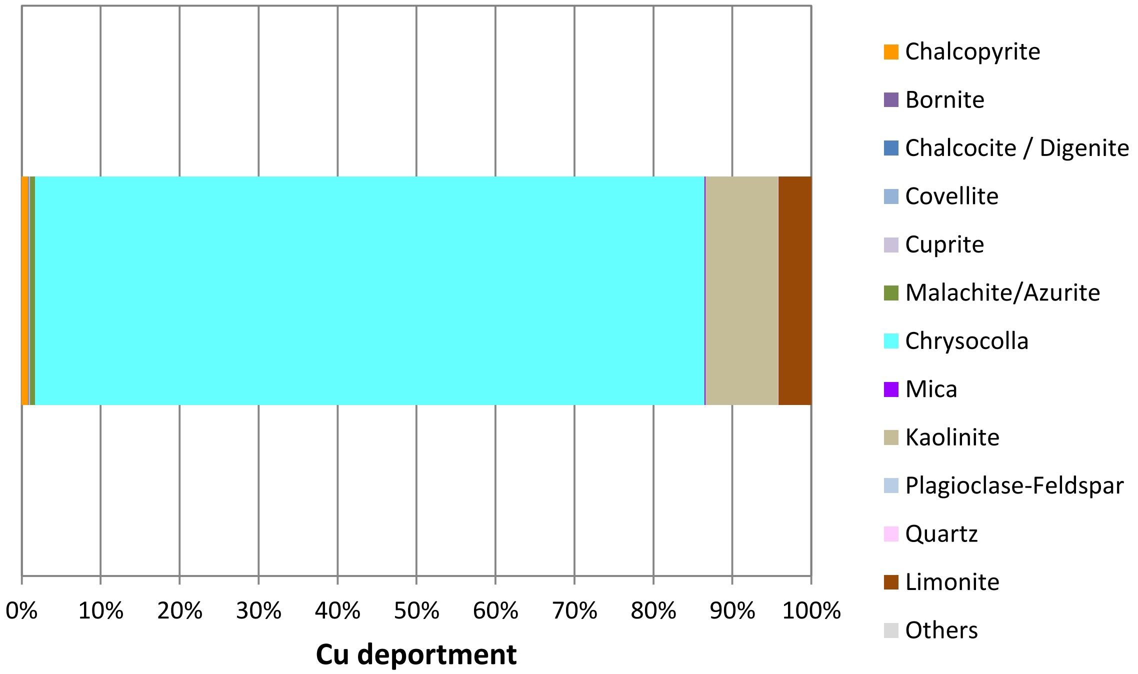


Figure 5.3-27: Copper deportment in LQ ore tailings after flotation with 30 g/t SIBX followed by slug sulphidisation with 500 g/t NaHS and 50 g/t SIBX

The reagent split for the two potential ranges used for CPS investigation is summarised in Table 5.3-7. The more negative potential range (-400 to -500 mV) resulted in a higher NaHS consumption, and consequently higher total SIBX dosage. Stage 1 represents the first part of the tests, focusing on copper sulphide recovery and was constant at 30 g/t SIBX across all the tests. It is important to note that the NaHS consumption at the potential range of -300 to -400 mV was significantly lower than that of the -400 to -500 mV range without any compromise to the recovery and concentrate grade. Moreover, when compared to slug addition of 500 g/t NaHS, the recoveries observed at the CPS range of -300 to -400 mV are similar but the grades are slightly lower.

Table 5.3-7: NaHS and SIBX consumption during CPS of LQ ore at potential ranges of -300 to -400 mV and -400 to -500 mV

Stage	-300 to -400 mV				-400 to -500 mV			
	NaHS consumption (g/t)	SIBX Consumption (g/t)			NaHS consumption (g/t)	SIBX Consumption (g/t)		
		NaHS:SIBX Ratio at CPS				NaHS: SIBX ratio at CPS		
		20:1	10:1	7:1		20:1	10:1	7:1
1	0	30	30	30	0	30	30	30
CPS 1	119	5.97	11.94	17.92	236	11.81	23.61	35.42
CPS 2	64	3.19	6.39	9.58	100	5.00	10.00	15.00
CPS 3	36	1.81	3.61	5.42	61	3.06	6.11	9.17
Total	**219**	41	52	63	**397**	50	70	90

5.3.8. The effect of CPS on the pulp potential of LQ ore

Figure 5.3-28 shows the E_S and E_h profiles of LQ ore when slug sulphidised with 250 g/t and 500 g/t NaHS. For both dosages, the addition of NaHS led to a sharp drop in potential (both E_s and E_h) with 500g/t NaHS leading to a more negative potential than 250 g/t. The potential became more positive as the NaHS was consumed. The profiles also showed that the potential measured by an ISE (E_S) was more negative than that measured using a PtE (E_h).

Figure 5.3-29 shows the average E_h profiles during CPS. For both potential ranges, the Potential became more negative with the introduction of NaHS into the system. For the potential range (E_S) of -300 to -400 mV, it took 30 seconds for the target pulp E_h (determined in section 4.6.1) to be reached after which the controlled potential addition of NaHS was successfully held within the target range. It took 40 seconds for the same level of control to be attained for the more negative E_S range of -400 to -500 mV. Generally however, CPS using an ISE to target a given E_S was matched and confirmed by the fact that the pulp was held at the equivalent E_h ranges. It is acknowledged that the starting readings (at 0 sec) for the two E_S ranges were not always equal because these readings started after a flotation stage (copper sulphide recovery) had already been done.

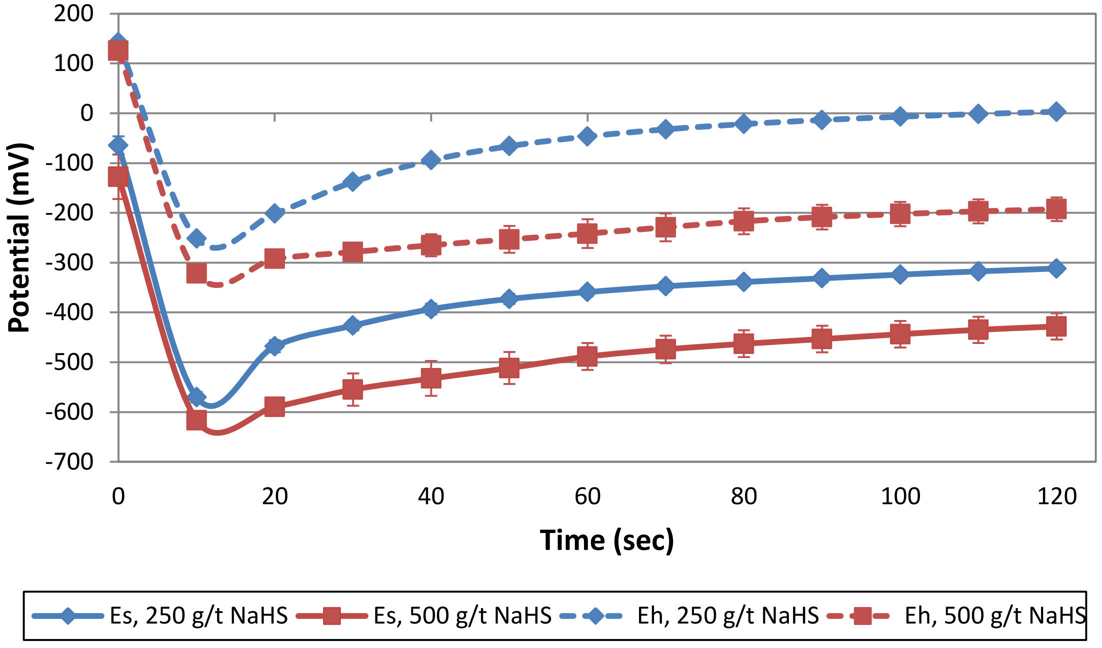


Figure 5.3-28: Pulp E_S and E_h profiles of LQ ore during slug sulphidisation with 250 g/t and 500 g/t NaHS. NaHS was introduced into the system at the 0 sec mark. Error bars indicate standard error between 6 tests for 250 g/t NaHS and 3 tests for 500 g/t NaHS

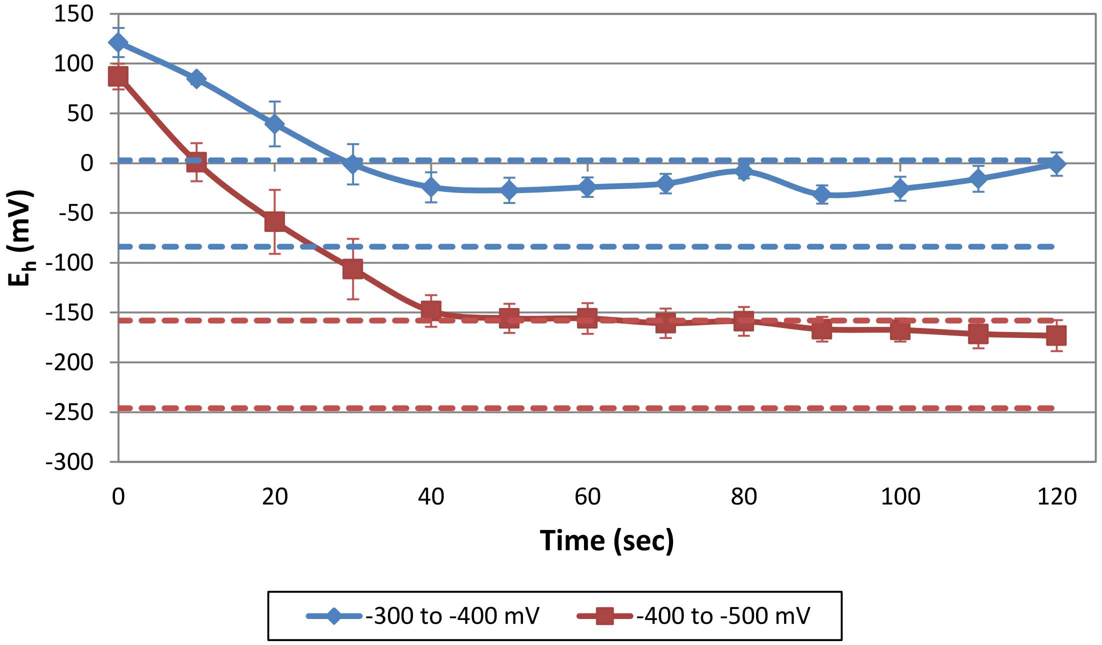


Figure 5.3-29: E_h profiles of LQ ore pulp during CPS. NaHS was introduced into the system at the 0 sec mark. Broken lines represent the E_h ranges that were predicted when E_h was plotted against E_S. Error bars represent standard error between 8 tests

5.3.9. Microflotation of pure malachite and chrysocolla

In order to get a greater understanding of whether or not the observations that were made are related to froth phase behavior, a selected set of tests were carried out on two oxide minerals using the microflotation technique outlined in section 0. The two minerals chosen were malachite and chrysocolla because they represented significant amounts of oxide copper minerals in the LQ ore being studied. Microflotation also provides good indications of the extent to which a reagent is rendering a mineral hydrophobic.

Figure 5.3-30 shows the results obtained when chrysocolla and malachite were recovered by microflotation using SIBX as the collector and NaHS a sulphidiser. For the flotation time of 10 minutes, there was virtually no recovery of chrysocolla, whether it was sulphidised with NaHS or not. Flotation of pure malachite with pure SIBX led to a 12.1 % recovery of the mineral. Sulphidising malachite prior to addition of SIBX increased its recovery to 18.2 %. The recovery data is given in Appendix D.

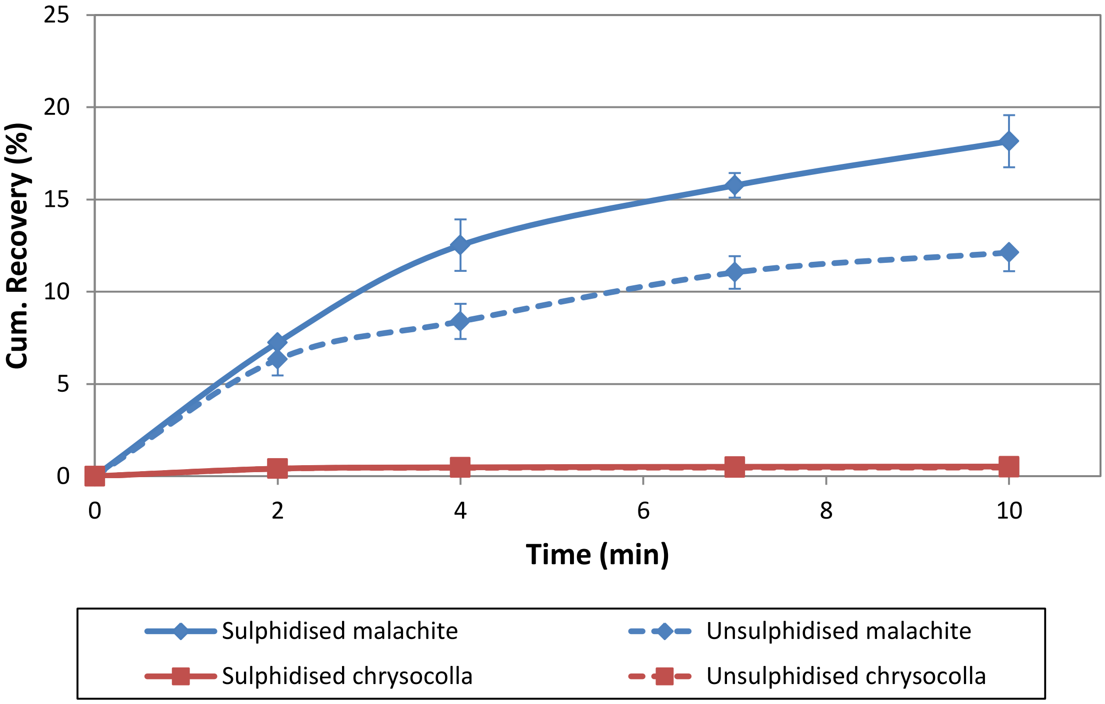


Figure 5.3-30: Recovery-time plots for the microflotation of pure malachite and chrysocolla. Error bars indicate standard error between duplicate tests

6. DISCUSSION

"I welcome new words, or old words used in new ways, provided the result is more precision, added colour or greater expressiveness"

- William Safire

The objective of this study was to investigate the floatability of a complex Kansanshi mixed copper ore comprising sulphide and oxide minerals with a view to achieving an optimal flotation performance in the treatment of the Kansanshi ore body. This required an in-depth analysis of the mineralogy of the feed as well as tailings samples after different flotation processes involving a range of reagent types and dosage procedures. The focus on the mineralogical study of tailings rather than concentrate was to identify minerals that did not respond favourably to any of the flotation procedures used. In this chapter, the results obtained in Chapter 5 are linked to the key questions that were listed under each objective in Chapter 3 and the findings are compared to other studies that have been done on the use of sulphidisation in the flotation of complex copper ores.

The ore samples studied represented a high quality (HQ) ore dominated by sulphide minerals and low quality (LQ) ore dominated by oxide minerals. The quality of an ore at Kansanshi is determined by the acid soluble copper (ASCu) content of the ore, which is used as a proxy for oxide mineral content. An important finding in this study was that sulphide minerals are also prone to digestion during this analysis. The digestion of LQ ore to determine the fraction of copper that reports as acid soluble copper showed greater than 95 % digestion of chrysocolla and malachite/azurite and 100 % digestion of cuprite. However, over 70 % of the chalcopyrite and 58 % of the bornite were also digested. The implication of this finding is that perhaps using ASCu as a proxy for the amount of oxide minerals present could lead to an overestimation of the oxide minerals and an underestimation of the sulphide minerals present in the ore and this could influence decision making with respect to the flotation process to be adopted. For example, this could result in inappropriate reagent dosages being used. Lee et al. (1998) also highlighted the importance of mineralogy in describing an oxide ore, referring to the fact that the chemical assay methods used to determine copper occurrences as acid soluble have not been standardized, give different values for copper oxide content and *"leave no room for the possibility that much of this acid soluble copper may not be present as well defined oxide minerals"* and therefore is not amenable to sulphidisation-xanthate flotation. Therefore it is of critical importance in such a study to carry out mineralogical analyses in combination with chemical assays.

HQ and LQ ore contained the same copper minerals but varied significantly in their copper deportment. Chalcopyrite was the major copper mineral in the HQ ore, constituting 3.9 %, but only 1.0 % of the LQ ore. LQ ore was dominated by chrysocolla which constituted 3.8 % of the ore. The bulk mineralogy of the HQ and LQ ore has been presented in Table 5.2-1 and Table 5.3-1, respectively. In the HQ ore sample, 93.6 % of the copper in the ore was present as chalcopyrite, 2.7 % as secondary copper sulphides (bornite, covellite and chalcocite) and 1.7 % as chrysocolla. The implication of this deportment was that the copper from HQ ore could be mainly recovered by standard xanthate flotation procedures provided it was sufficiently liberated. In the case of LQ ore however, only 22.2 % of the copper was hosted in chalcopyrite. 63.1 % of the copper in LQ ore was in chrysocolla, 1.5 % in cuprite and 2.9 % in malachite. The deportment in both sulphide and oxide minerals implied that LQ ore flotation with xanthates alone would lead to poor recoveries since non-sulphide minerals do not respond to simple xanthate flotation (Fuerstenau et al., 1985; Aplan, 1994).

The flotation of chalcopyrite and other sulphide minerals has been extensively studied and shown to be dependent on a number of factors including pulp potential (Heyes & Trahar, 1979; Guo & Yen, 2003; Greet et al., 2005) and pH as long as the optimum reagent dosage is used (Göktepe, 2002). Evans et al. (2011) emphasised the need to model liberation in order to predict flotation performance. The liberation characteristics of the chalcopyrite in the two ores showed significant variation. In the case of HQ ore, the extent of liberation between the two P_{80}s investigated (75 µm and 150 µm) was about 91 %. The chalcopyrite in LQ ore was poorly liberated at both P_{80}s, viz. 51.4 % at 150 µm and 55.2 % at 75 µm, with the bulk of the unliberated chalcopyrite associated with chrysocolla. This naturally has significant ramifications with respect to the operation of the communition circuit since the purpose of the communition circuit is to optimise liberation without overgrinding. Based on the liberation characteristics referred to above, all the flotation tests in this study were carried out at 80 % passing 150 µm since it was shown that no greater degree of liberation was achieved by finer grinding.

Both ores were first only treated with 30 g/t and 50 g/t of SIBX to investigate their response to xanthate flotation but 20 g/t SIBX was also investigated for LQ ore because of the lower copper deportment in chalcopyrite. Figure 6-1 shows the cumulative solids and water recovery for the various SIBX dosages used. Solids recovery was higher in HQ ore than LQ ore. The difference in solids recovery between the two ores was expected due to the difference in chalcopyrite content. The higher chalcopyrite content in the HQ ore would inevitably result in higher solids recovery. For both HQ and LQ ore however, increasing the collector dosage from 30 g/t to 50 g/t led to a significant reduction in water recovery.

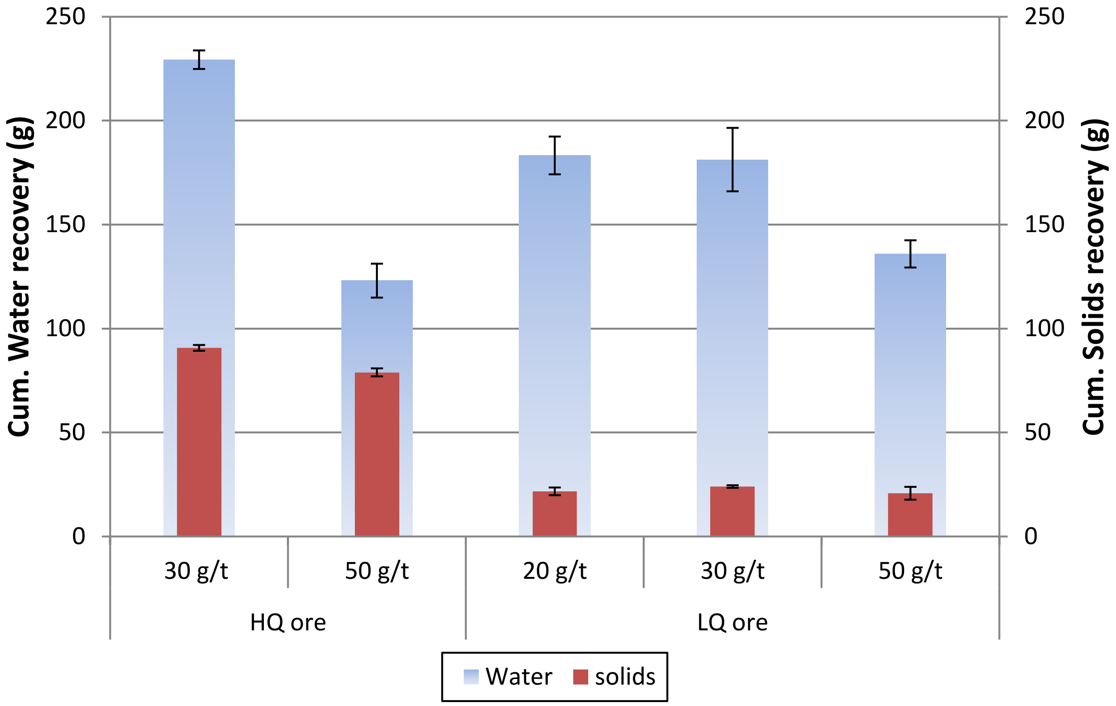


Figure 6-1: Cumulative solids and water recovery for flotation of HQ and LQ ore to recover chalcopyrite

Figure 6-2 shows the froth structure when air was introduced into HQ ore slurry conditioned with 30 g/t SIBX (a) and 50 g/t SIBX (b). At a dosage of 50 g/t SIBX, the bubbles in the froth were much larger than at 30 g/t and the froth did not flow easily over the lip of the cell. At elevated collector dosages, the hydrophobicity of minerals present in the pulp would be enhanced to a greater degree. It is possible that at 50 g/t SIBX, the particles in the froth are at their most hydrophobic. Harris (1982) and Ata et al., (2004) have proposed that an increased presence of hydrophobic species in the froth phase destabilizes the froth, leading to increased water drainage rates and drier froths.

Figure 6-2: Froth structure when air was introduced into HQ ore pulp with 30 g/t SIBX (a) and 50 g/t SIBX (b). The froth structure at a dosage of 50 g/t SIBX comprised bigger bubbles than 30 g/t and resulted in relatively low water recovery without any significant loss of solids

Figure 6-3 shows the cumulative copper recovery and grades when HQ and LQ ore were floated with SIBX. 30 g/t and 50 g/t resulted in similar copper recoveries for HQ ore. Since 90.6 % of the chalcopyrite in HQ ore was fully liberated, a recovery of almost 90 % for both 30 g/t and 50 g/t of SIBX represents a loss of possibly locked chalcopyrite. Clearly, given that chalcopyrite is easily floatable, these results indicate that it is not necessary to use dosages greater than 30 g/t for HQ ore. 30 g/t SIBX also gave the highest copper recovery for LQ, viz. 24.7 %. This relatively poor result was expected because of the low copper deportment in chalcopyrite. LQ ore contained mostly oxide copper minerals which are not amenable to xanthate flotation. This poor flotation response was clearly illustrated in microflotation cell studies where it was observed that virtually no chrysocolla was recovered but about 12.1 % of the malachite was recovered by SIBX (c.f. section 5.3.9). This result suggests that to a certain extent, malachite responds to xanthate flotation, but chrysocolla does not, and is consistent with what has been shown about the recovery of oxide copper minerals in literature (Aplan & Fuerstenau, 1984; Lee et al., 2009) .

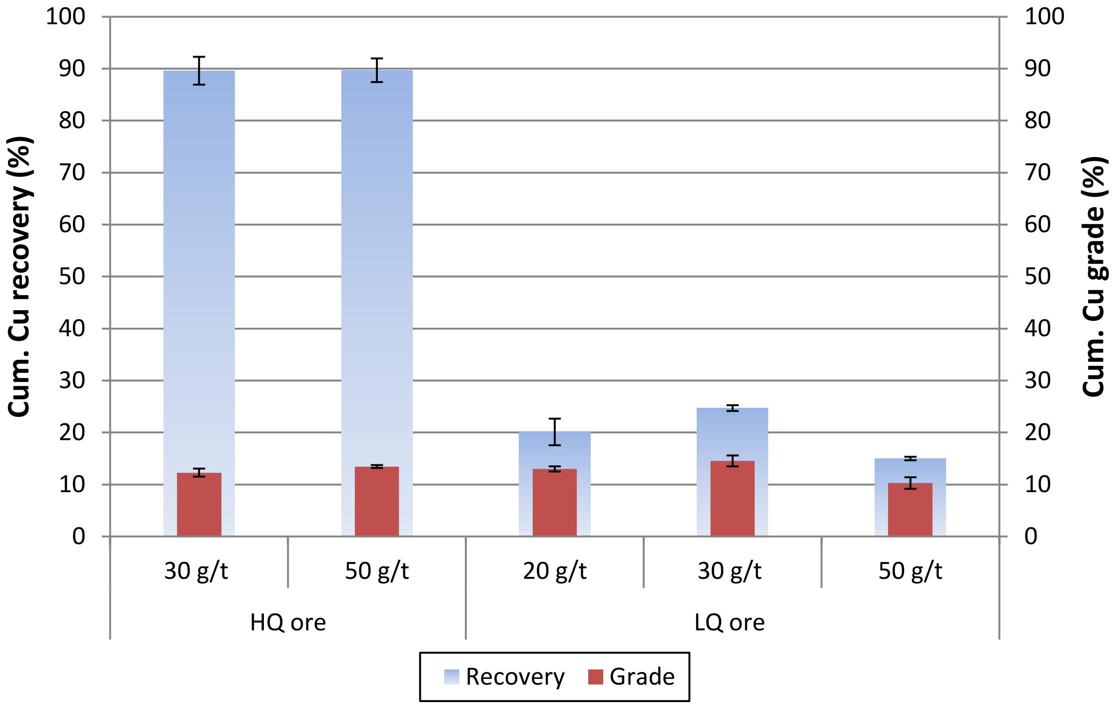


Figure 6-3: Cumulative grade-recovery trends for the flotation of HQ and LQ ore with SIBX

In the case of LQ ore, there was a marked decrease in concentrate grade when collector dosage was increased from 30 g/t to 50 g/t. This is consistent with the reduced water recovery observed for the same increase in collector dosage and suggests that the increased recovery of hydrophobic particles in the froth also increased the recovery of hydrophilic particles in the froth by entrainment (Ata et al., 2004), leading to the dilution of the concentrate.

In conclusion, the treatment of HQ ore with SIBX has shown that HQ ore responds well to xanthates and requires no further treatment after xanthate flotation. On the other hand, while 30 g/t SIBX was sufficient for chalcopyrite recovery in LQ ore, the tailings mineralogy after flotation with SIBX indicated that 78.8 % of the unrecovered copper was present as chrysocolla, 1.3 % as malachite and 5.8 % as chalcopyrite and therefore LQ ore required alternative flotation methods for the recovery of the oxide minerals.

Two sulphidisation techniques were investigated for the recovery of the oxide copper minerals in the LQ ore, viz. slug sulphidisation and controlled potential sulphidisation (CPS). 250 g/t and 500 g/t NaHS were added to the pulp during slug sulphidisation and two potential ranges were investigated during CPS. The reason for this is that literature has indicated that optimum sulphidisation of copper oxides takes place over a wide potential range (c.f. section 2.6.3; Nagaraj & Gorken, 1991; Ferron & Manu, 1994; Quast et al., 2005) and because the sensitivity of an ISE made it impossible to target one particular potential but could be controlled within the margins of two potentials if the difference between them was about 100 mV. Initial scoping tests comparing E_h and E_S indicated that there existed linear relationships between the two measurements in the two E_S potential ranges investigated. The potential ranges of -300 to -400 mV and -400 to -500 mV measured with an ISE can be converted to E_h using equations 6-1 and 6-2 respectively:

$$\mathrm{E_h} = 0.8669\mathrm{E_S} + 262.9 \qquad \text{(Eqn. 6-1)}$$

$$\mathrm{E_h} = 0.8784\mathrm{E_S} + 193.37 \qquad \text{(Eqn. 6-2)}$$

It can be seen in Figure 6-4 that introducing NaHS into the pulp, regardless of technique led to increased solids and water recovery. This phenomenon was also observed by Becker et al., (2014) who noted that the addition of NaHS in the absence of a hydroxanmate collector (AM28) increased water recovery when compared to a test in which no NaHS was added, thus indicating greater froth stability. For slug sulphidisation in particular, increasing the NaHS dosage from 250 g/t to 500 g/t led to a 41 % increase in water recovery (viz. from 237 g to 333 g) with no significant increase in the amount of solids recovered.

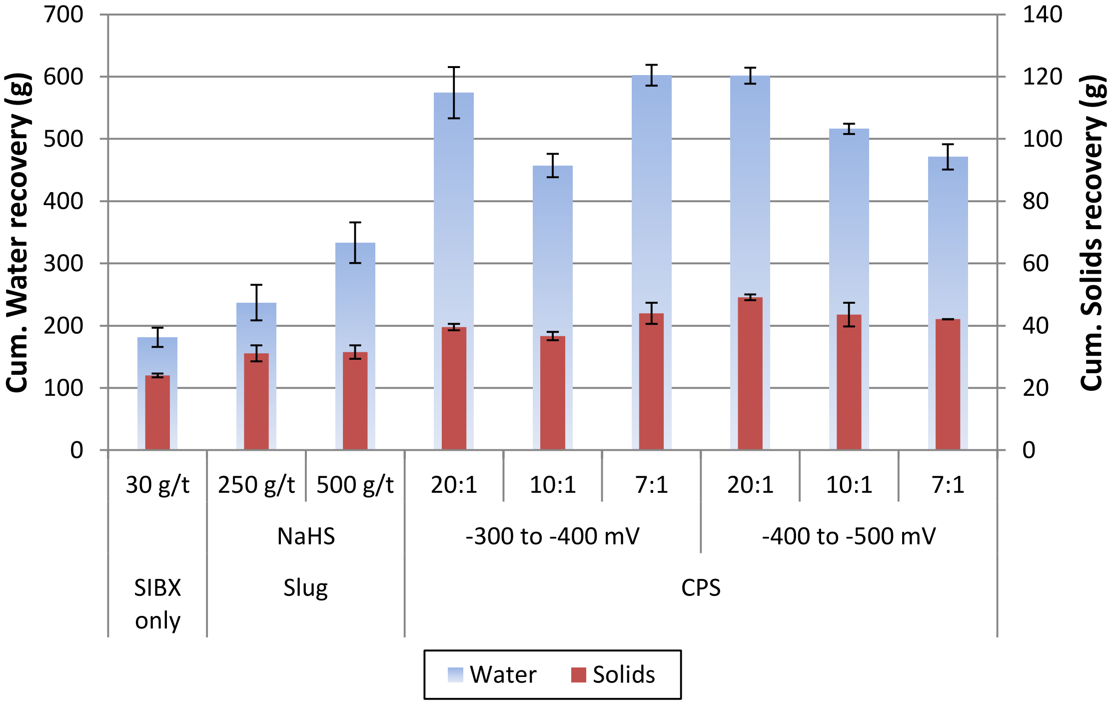


Figure 6-4: Cumulative solids and water recovery for the flotation of LQ ore with 30 g/t SIBX followed by slug sulphidisation with 250 g/t and 500 g/t NaHS and CPS at -300 to -400 mV and -400 to -500 mV using the following NaHS:SIBX ratios: 20:1, 10:1 and 7:1

The difference in water recoveries seems to imply that the NaHS is having a stabilising effect on the froth behavior and results in greater water recovery without any enhanced solids recovery. This behavior is evident in Figure 6-5 which shows the difference in froth structure between 250 g/t (a) and 500 g/t (b) of NaHS. For a NaHS dosage of 500 g/t, the froth comprised very small bubbles and flowed easily into the concentrate launder. The bigger bubbles observed for the NaHS dosage of 250 g/t resulted in lower water recoveries, just as it was in the initial flotation tests with SIBX only, where the bigger bubbles observed at 50 g/t SIBX also resulted in lower water recoveries. A contradictory observation is made in the case of CPS however, because despite using less NaHS than slug sulphidisation, CPS resulted in higher water recoveries than slug sulphidisation.

Figure 6-5: Froth structure of LQ ore after slug sulphidisation with 250 g/t (a) and 500 g/t (b) NaHS. The froth at 500 g/t NaHS was made up of smaller bubbles than at 250 g/t NaHS and flowed much easily, accounting for higher water recoveries

The increase in solids recovery after the introduction of NaHS, whether by slug sulphidisation or CPS, was expected and can be partially attributed to the recovery of oxide minerals which had not been recovered during flotation with SIBX only. The grade-recovery trends are summarised in Figure 6-6. In the case of slug sulphidisation, TCu recovery increased with increasing NaHS addition. The same observation has been made in slug sulphidisation of oxidised lead-zinc-silver material with Na_2S (Jones & Woodcock, 1979) and malachite with calcium polysulphide (Quast et al., 2005). Slug addition of NaHS also increased TCu recovery, when compared to the base case (30 g/t SIBX) without compromising the grade, thus illustrating the selective role that NaHS played in recovering the oxides of copper. The concentrate grades between the two slug sulphidisation dosages were comparable consistent with the fact that solids recoveries were also similar.

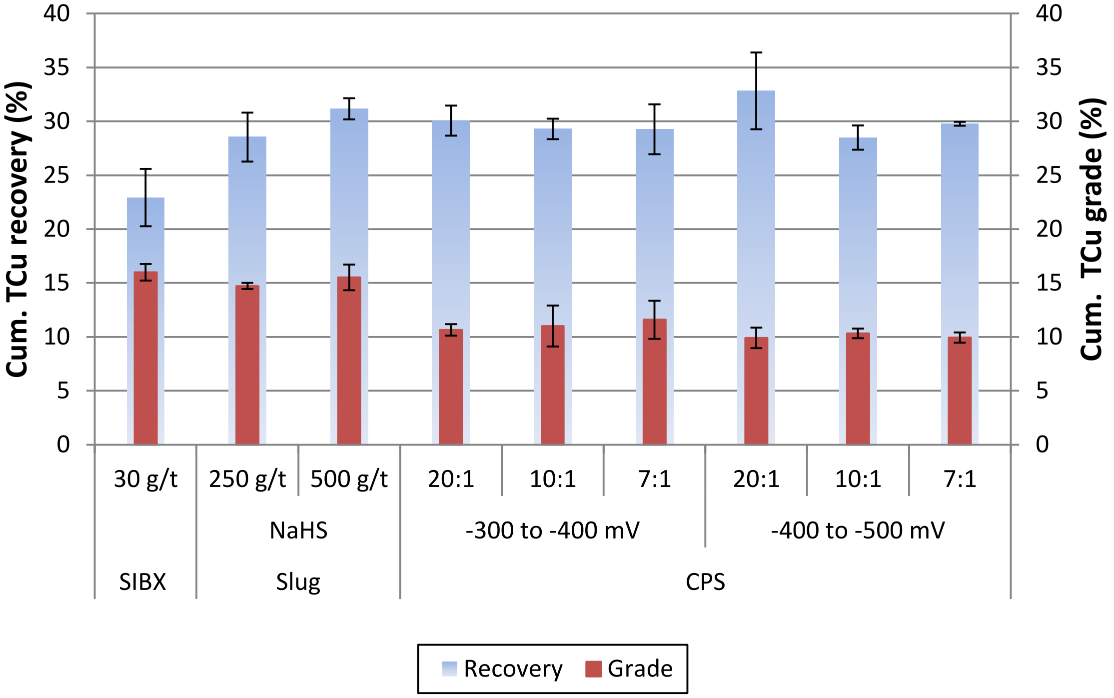


Figure 6-6: Cumulative grade-recovery trends for the sulphidisation-flotation of LQ ore with 30 g/t SIBX followed by slug sulphidisation with 250 g/t and 500 g/t NaHS and CPS at -300 to -400 mV and -400 to -500 mV using the following NaHS:SIBX ratios: 20:1, 10:1 and 7:1

In the case of CPS, the highest solids recoveries were paired with the highest water recoveries for both potential ranges. CPS addition of NaHS also resulted in a significant increase in TCu recovery from the 22.9 % observed for the base case (30 g/t SIBX) to an average of about 30 %. However, CPS treatment resulted in a general decrease in grades from approximately 15 % to 10 % when compared to slug sulphidisation. This indicates a greater recovery into the concentrate of gangue minerals thus diluting the copper grade. This is an interesting result since the highest water recoveries were also observed in CPS. This coupled with lower concentrate grades implies that as more water is carried into the concentrate, more gangue is carried as well (Buswell et al., 2002), hence the lower grades. The statement is confirmed by the slightly higher solids recovery during CPS (c.f. Figure 6-4).

Only the tailings after slug sulphidisation with 500 g/t NaHS were subjected to mineralogical analysis because the test gave the highest concentrate grade for about the same recovery as all the other sulphidisation tests. Two important observations were made from the mineralogy of the tailings:

i. The chalcopyrite that had not been recovered in the pre-sulphidisation stage was recovered after sulphidisation. Since the chalcopyrite in the feed to the sulphidisation stage had been shown to be mostly liberated, it is possible that some of it was tarnished due to in-situ weathering or during storage and therefore required sulphidisation. Sulphidisation followed by xanthate flotation has been shown to recover tarnished chalcopyrite (Clark et al., 2000) and other sulphide minerals such as pentlandite (Newell & Bradshaw, 2007).

ii. All the other oxide copper minerals except chrysocolla were susceptible to xanthate flotation after sulphidisation. These findings are consistent with observations made by Kottgen & Bastin (2009). Microflotation tests with NaHS and SIBX further proved that malachite recovery is possible with sulphidisation-flotation but NaHS treatment was not beneficial to the recovery of chrysocolla. Thus the hypothesis that NaHS will play a role similar to copper sulphate in sulphide flotation and act as an activator for oxide minerals holds true for malachite but not for chrysocolla. This also suggests that the sulphidisation mechanisms suggested in literature (Wills & Napier-Munn, 2006; Quast et al., 2005) do not promote the recovery of chrysocolla, possibly due to its amorphous nature.

In evaluating the relative benefit of the two sulphidisation processes, it is useful to carry out a comparative techno-economic evaluation. Table 6-1 shows a scenario for a possible techno-economic analysis comparing the slug and CPS procedures from a cost-benefit point of view i.e. based on the ratio of revenue to reagent cost per ton of LQ ore treated. The analysis does not take into account other operational costs but highlights the potential economic benefit that could be achieved using CPS at the correct potential range and NaHS:SIBX ratio. It should also be noted that this analysis is limited by the fact that only the roughing stage is considered. While this study has indicated that slug sulphidisation can be used to recover some oxide copper minerals and give superior concentrate grades to CPS, the cost benefit analysis indicates that CPS at the correct potential range results in better flotation performance from an economic stand point. The economic benefits highlighted are in line with observations made by Quast et al. (2005) who showed that although slug sulphidisation gave acceptable flotation results, it also resulted in higher reagent consumption than CPS. The analysis also outlines the importance of using the right collector dosage at the potential range selected. CPS at the potential range of -300 to -400 mV required 219 g/t NaHS and a total SIBX dosage of 63 g/t for optimised flotation economic performance, viz. US$ 82 per US$ used on reagents to treat 1 t of ore. It would therefore be of value to the operations to pay careful attention to the potential benefits of implementing a well-planned CPS process.

Table 6-1: Techno-economic analysis comparing slug sulphidisation of LQ ore to CPS

Basis	1	t LQ ore
Cu	6465	US$/t (11 Dec, 2014)
SIBX	1710	US$/t
NaHS	1300	US$/t

	Slug sulphidisation		**CPS: -300 to -400 mV**			**CPS: -400 to -500 mV**		
NaHS (g/t)	250	500	219	219	219	397	397	397
NaHS:SIBX	10:1	10:1	20:1	10:1	7:1	20:1	10:1	7:1
SIBX (including pre sulphidisation) (g/t)	55	80	41	52	63	50	70	90
TCu recovered (%)	28.55	31.17	30.06	29.30	29.27	32.82	28.50	29.77
Reagent cost (US$/t ore treated)	0.43	0.8	0.36	0.38	0.40	0.61	0.65	0.68
Solids recovered per kg ore treated (g)	31.13	31.52	39.55	36.68	43.93	49.13	43.56	42.11
Solids recovered per t ore treated (t)	0.03	0.03	0.04	0.04	0.04	0.05	0.04	0.04
Concentrate grade (% TCu)	14.73	15.5	10.65	11	11.59	9.91	10.32	9.94
Cu in concentrate (t)	0.005	0.005	0.004	0.004	0.005	0.005	0.004	0.004
Cu value in concentrate (US$)	29.64	31.59	27.23	26.08	32.92	31.48	29.06	27.06
Revenue on Cu/Reagent cost (per ton ore)	**69.50**	**39.39**	**75.35**	**68.61**	**82.50**	**51.31**	**44.87**	**39.68**

7. CONCLUSIONS AND RECOMMENDATIONS

"I think that's the best piece of advice; constantly think about how you could be doing things better and questioning yourself"

- Elon Musk

7.1. Conclusions

The conclusions are presented as answers to the key questions outlined in Chapter 3.

1. What is the mineralogical difference between high quality (HQ) and low quality (LQ) ore?

While the same copper minerals were present in the ore, the bulk mineralogy indicated significant differences in grade and deportment. HQ ore was dominated by chalcopyrite, which constituted 3.9 % of the ore while the most abundant copper mineral in LQ ore was chrysocolla, viz. 3.8 % with chalcopyrite only making up 1 %. Gangue mineralogy was also different for the two ores. HQ ore was dominated by quartz (25.4 %), plagioclase-feldspar (32.7 %) and mica (23.3 %) while LQ ore contained mostly calcite (29.1 %) and quartz (16.4 %).

2. What is the extent of liberation of the different minerals in the feed to float?

At the target P_{80} of 150 µm that is used at Kansanshi, 90.6 % of the chalcopyrite in HQ ore was fully liberated, with 4.5 % and 2.6 % of the unliberated chalcopyrite associated with chrysocolla and malachite respectively. In the case of LQ ore however, a PSD with 80 % passing 150 µm resulted in only 51.5 % of the chalcopyrite and 29.8 % of the chrysocolla being fully liberated. Grinding finer to 80 % passing 75 µm did not significantly increase the liberation of either mineral in the LQ ore.

3. What copper species report as acid soluble copper?

When LQ ore was digested in sulphuric acid in order to determine its ASCu content, over 90 % of each of the oxide minerals, viz. cuprite, malachite and chrysocolla, was digested. The mineralogy of the residue also indicated that 73.8 % of the chalcopyrite and 58.3 % of the bornite in the LQ ore were digested by sulphuric acid. These results clearly showed that ASCu is an inaccurate proxy for the amount of copper oxides present in the ore.

4. How do the two ores respond to flotation with xanthates?

This study has shown that HQ and LQ ore respond very differently to flotation with SIBX. Since 93.6 % of all the copper in HQ ore was hosted in chalcopyrite and the extent of liberation was very high, it was expected that the ore would respond very well to flotation with SIBX. 89.6 % and 89.7 % of the copper in HQ ore was recovered at SIBX dosages of 30 g/t and 50 g/t, respectively. In the case of LQ ore, SIBX dosages of 20 g/t, 30 g/t and 50 g/t were investigated. The highest copper recovery of 22.9 % was observed at 30 g/t of SIBX. These results imply that while 30 g/t of SIBX is sufficient for the recovery of chalcopyrite from both HQ and LQ ore, LQ ore requires additional flotation techniques to recover the oxide minerals present.

5. What is the optimal methodology for NaHS treatment of the LQ ore?

The optimal methodology for the treatment of LQ ore is twofold. Firstly and most importantly, it is important to choose the sulphidisation technique to use between slug and CPS after the recovery of sulphide minerals with xanthates. Secondly, it is a well-established fact that collector dosage is key in froth flotation as it directly impacts the extent of value mineral recovery, therefore this study aimed to determine the best technique and variables within that technique, as well as the correct collector dosage to use after sulphidisation. At Kansanshi, a NaHS:SIBX ratio of 10:1 is used with CPS. In this study, the amount of collector added after CPS was varied using various NaHS:SIBX ratios, viz. 1:20, 1:10 and 1:7 and compared to slug sulphidisation at a NaHS:SIBX ratio of 10:1. This resulted in different copper recoveries for the different combinations of variables, viz. technique (slug vs. CPS), NaHS dosage or potential range and collector dosage after sulphidisation. Based on the revenue on copper/reagent cost per ton of ore treated, the optimal methodology for the NaHS treatment of LQ ore is CPS at a potential range of -300 to -400 mV and a NaHS:SIBX ratio of 7:1 as it gave the highest revenue, i.e. US$82.5/US$ of reagents used to treat 1 t of ore.

6. Is slug sulphidisation better than controlled potential sulphidisation?

This study has shown that CPS performs better than slug sulphidisation only when the correct potential range and SIBX dosage are used. Slug sulphidisation at 250 g/t and a NaHS:SIBX ratio of 10:1 performed better than CPS at -400 to -500 mV, regardless of the NaHS:SIBX ratio used at that potential range. CPS at -300 to -400 mV performed better than all the other conditions tested. Although the revenue on copper/reagent cost for slug sulphidisation using 250 g/t NaHS and CPS at -400 to -500 mv for the NaHS:SIBX ratio of 10:1 were comparable, viz. 69.5 and 68.6 respectively, CPS required less NaHS than slug sulphidisation, viz. 219 g/t was used for CPS and 250 g/t for slug. Further to this, reducing the amount of SIBX used during CPS, i.e. using a NaHS:SIBX ratio of 20:1 increased the revenue on copper/reagent cost to 75.4 (US$/US$). This value increased to 82.5 when a NaHS:SIBX ratio of 7:1 was used. Therefore as addressed in question 5 above, it is important to use the correct sulphidisation technique as well as the correct amount of collector after sulphidisation.

7. Is the loss of value minerals to the tailings due to poor liberation or inappropriate reagent regimes?

This question is answered in the context of LQ ore since the highest copper recovery observed in the flotation of LQ ore was 32.8 % in comparison to HQ ore in which 89.7 % was recovered. Mineralogical analysis of the tailings before sulphidisation indicated that 1.1 % of the unrecovered copper was present in cuprite, 1.3 % in malachite, 5.8 % in chalcopyrite and 78.9 % in chrysocolla. After slug sulphidisation the tailings had a copper deportment of 0.1 % cuprite, 0.6 % malachite, 0.8 % chalcopyrite and 84.8 % chrysocolla. The conclusion drawn from this copper deportment was that all the copper minerals present in the ore except chrysocolla are amenable to flotation using SIBX coupled with sulphidisation. Therefore the loss of value minerals, i.e. chrysocolla, to the tailings is due to the use of an inappropriate reagent regime.

8. How do pure samples of non-sulphide minerals respond to xanthate flotation and NaHS treatment?

Not all oxide minerals respond the same to xanthate and sulphidisation flotation. The microflotation of pure malachite and chrysocolla samples indicated that malachite responds to flotation with SIBX, with recoveries of up to 12.1 %. Sulphidisation with NaHS increased malachite recovery to 18.2 %. In comparison, only 0.5 % chrysocolla was recovered by both techniques, viz. SIBX flotation and sulphidisation. This justifies the conclusion addressed in question 7 above that chrysocolla is not amenable to recovery by flotation with either xanthates or sulphidisation coupled with xanthates.

7.2. Implications of the study

An important consideration in defining any ore is the identification and quantification of the minerals present; both value and gangue. This study has highlighted the need for process mineralogy in defining an ore in terms of bulk mineralogy, copper deportment and liberation. Two ores classified as mixed ore were shown to be mineralogically different and therefore required different reagent suites for optimised copper recovery. The study has further highlighted the flaw in using acid soluble copper (ASCu) as a proxy for the amount of oxide copper minerals in an ore due to copper sulphides also being susceptible to digestion during ASCu determination.

Sulphidisation-flotation has been shown to recover tarnished chalcopyrite as well as non-sulphide copper minerals such as cuprite and malachite. The flotation tailings of LQ ore after sulphidisation comprised 84.7 % copper deportment in chrysocolla, which proves that at ambient conditions, chrysocolla cannot be recovered through sulphidisation. Although slug sulphidisation is the easier of the two sulphidisation techniques, the study has shown that CPS is the more cost effective of the two processes. Further to this it is important to use the right potential range and collector dosage after the CPS stage. It is acknowledged that potential control is difficult when an ISE is used but this study has shown that a platinum electrode can be used for control. The implication of this finding is that if used in an automated control system and linked to a reagent dosage system, CPS using E_h rather than E_S can be implemented successfully on the plant.

7.3. Recommendations

Within the context of the findings of this study and the implications outlined above, the following recommendations are made:

1. Routine and careful mineralogical characterisation of mixed ore at Kansanshi in order to determine whether xanthate flotation will suffice for optimal recovery of copper in a mixed ore from a particular part of the pit or if the ore will require sulphidisation-flotation.

2. Controlled potential sulphidisation linked to an automated reagent dosage system will allow for the correct potential range and NaHS and collector dosage to be used. This will possibly result in a reduction in waste (of reagents) and therefore has cost saving benefits.

3. A study into the mechanisms of CPS on pure oxide copper minerals could give insight on why certain oxide minerals respond to sulphidisation while others, such as chrysocolla, do not.

4. Literature has indicated that hydroxamates can be used as an alternative to sulphidisation-flotation and can recover chrysocolla. Since virtually no chrysocolla was recovered from the sulphidisation of LQ ore (and microflotaion of the pure chrysocolla), an investigation into the possibility of improving copper recovery in LQ ore by using hydroxamates is recommended. Preliminary tests with hydroxamates have shown that up to 25 % copper recovery in LQ ore is possible with hydroxamates (Mphepya & Radingwana, 2014). In the study only three dosages of one hydroxamate were

investigated. Perhaps a wider range of hydroxamates and dosages would give better results.

5. Further mineralogical studies of pure chrysocolla are recommended in order to increase the fundamental knowledge base of the mineral. This could possibly lead to novel techniques, not limited to froth flotation, through which the mineral can be recovered.

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Appendix A: Ore characterisation

XRF determination of HQ ore elemental composition

Sample	P_{80} = 75 µm				P_{80} = 150 µm					
	A/+75	A/+53	A/+25	A/-25	B/+150	B/+106	B/+75	B/+53	B/+25	B/-25
Major Oxides wt%										
SiO2	59.76	62.61	61.31	56.02	58.24	59.61	62.01	64.37	64.70	56.76
TiO2	1.15	1.14	1.05	1.16	1.02	1.24	1.29	1.18	1.01	1.15
Al2O3	13.19	11.21	14.40	20.40	14.31	10.62	10.78	11.82	15.47	21.39
Fe2O3	6.31	5.81	4.33	4.47	6.40	7.44	6.43	5.13	3.75	4.21
MnO	0.03	0.03	0.02	0.03	0.03	0.04	0.03	0.03	0.02	0.03
MgO	1.91	1.44	1.25	1.45	1.69	1.50	1.49	1.36	1.25	1.47
CaO	3.67	3.60	2.59	2.43	3.05	3.68	3.50	2.92	2.15	2.07
Na2O	2.21	3.19	3.78	2.60	2.21	2.55	3.07	3.82	4.17	2.50
K2O	2.44	1.40	1.91	3.97	2.65	1.55	1.38	1.27	2.03	4.23
P2O5	0.13	0.18	0.19	0.15	0.14	0.15	0.17	0.21	0.19	0.15
SO3	3.29	3.82	2.50	1.73	3.52	5.26	4.46	3.26	1.86	1.39
Cr2O3	0.02	0.06	0.04	0.14	0.02	0.02	0.02	0.09	0.08	0.11
NiO	0.01	0.01	0.01	0.02	0.01	0.01	0.01	0.01	0.01	0.02
CuO	1.45	1.43	0.90	0.74	1.63	2.19	1.73	1.26	0.71	0.70
Total	95.56	95.92	94.27	95.31	94.92	95.85	96.35	96.71	97.38	96.17

Sample	+75	+53	+25	-25	Element	Conversion factor	+75	+53	+25	-25
	Normalised major oxide wt %						Element %			
SiO_2	62.54	65.27	65.04	58.78	**Si**	0.47	29.23	30.51	30.40	27.48
TiO_2	1.21	1.19	1.11	1.21	**Ti**	0.60	0.72	0.71	0.67	0.73
Al_2O_3	13.80	11.69	15.27	21.40	**Al**	0.53	7.31	6.18	8.08	11.33
Fe_2O_3	6.60	6.05	4.59	4.69	**Fe**	0.70	4.62	4.23	3.21	3.28
MnO	0.03	0.03	0.03	0.03	**Mn**	0.77	0.03	0.03	0.02	0.03
MgO	2.00	1.50	1.33	1.52	**Mg**	0.60	1.21	0.90	0.80	0.91
CaO	3.84	3.76	2.74	2.55	**Ca**	0.71	2.75	2.68	1.96	1.82
Na_2O	2.31	3.33	4.01	2.73	**Na**	0.74	1.71	2.47	2.97	2.03
K_2O	2.55	1.46	2.02	4.16	**K**	0.83	2.12	1.21	1.68	3.46
P_2O_5	0.13	0.19	0.20	0.16	**P**	0.44	0.06	0.08	0.09	0.07
SO_3	3.45	3.98	2.65	1.81	**S**	0.40	1.38	1.59	1.06	0.73
Cr_2O_3	0.02	0.06	0.04	0.15	**Cr**	0.68	0.02	0.04	0.03	0.10
NiO	0.01	0.01	0.01	0.02	**Ni**	0.79	0.00	0.01	0.01	0.02
CuO	1.51	1.49	0.96	0.78	**Cu**	0.80	1.21	1.19	0.77	0.62

Sample	+150	+106	+75	+53	+25	-25	Element	Conversion factor	+150	+106	+75	+53	+25	-25
	Normalised major oxide wt %								Element %					
SiO_2	61.36	62.19	64.36	66.55	66.44	59.02	**Si**	0.47	28.68	29.07	30.08	31.11	31.06	27.59
TiO_2	1.07	1.29	1.33	1.22	1.03	1.19	**Ti**	0.60	0.64	0.77	0.80	0.73	0.62	0.72
Al_2O_3	15.08	11.08	11.19	12.22	15.89	22.24	**Al**	0.53	7.98	5.86	5.92	6.47	8.41	11.77
Fe_2O_3	6.74	7.76	6.67	5.30	3.85	4.37	**Fe**	0.70	4.72	5.43	4.67	3.71	2.69	3.06
MnO	0.03	0.04	0.03	0.03	0.02	0.03	**Mn**	0.77	0.03	0.03	0.03	0.02	0.02	0.02
MgO	1.78	1.57	1.54	1.40	1.29	1.53	**Mg**	0.60	1.07	0.95	0.93	0.85	0.78	0.92
CaO	3.21	3.84	3.63	3.02	2.20	2.15	**Ca**	0.71	2.30	2.75	2.59	2.16	1.57	1.54
Na_2O	2.33	2.66	3.19	3.95	4.29	2.60	**Na**	0.74	1.73	1.97	2.36	2.93	3.18	1.93
K_2O	2.79	1.62	1.43	1.31	2.08	4.39	**K**	0.83	2.32	1.34	1.19	1.09	1.73	3.65
P_2O_5	0.14	0.15	0.17	0.22	0.19	0.15	**P**	0.44	0.06	0.07	0.08	0.10	0.08	0.07
SO_3	3.71	5.49	4.63	3.37	1.91	1.44	**S**	0.40	1.49	2.20	1.85	1.35	0.76	0.58
Cr_2O_3	0.02	0.02	0.02	0.09	0.08	0.12	**Cr**	0.68	0.02	0.01	0.01	0.06	0.06	0.08
NiO	0.01	0.01	0.01	0.01	0.01	0.02	**Ni**	0.79	0.01	0.01	0.01	0.01	0.01	0.01
CuO	1.72	2.28	1.79	1.30	0.73	0.73	**Cu**	0.80	1.37	1.83	1.43	1.04	0.58	0.58

XRF determination of LQ ore elemental composition

Sample	P_{80} = 75 µm				P_{80} = 150 µm					
	A/+75	A/+53	A/+25	A/-25	B/+150	B/+106	B/+75	B/+53	B/+25	B/-25
Major Oxides wt%										
SiO2	36.00	38.02	41.65	36.84	29.78	38.13	42.88	45.82	45.86	37.60
TiO2	0.61	1.00	1.29	0.92	0.56	0.59	0.90	1.21	1.48	0.97
Al2O3	10.30	8.04	9.79	10.47	6.90	6.81	8.42	9.85	11.98	12.30
Fe2O3	5.52	5.90	5.18	8.94	6.87	6.07	5.27	4.72	4.66	9.81
MnO	0.09	0.13	0.13	0.28	0.10	0.11	0.11	0.12	0.12	0.34
MgO	1.21	0.96	0.87	0.70	0.90	0.92	1.03	0.95	0.92	0.83
CaO	24.76	28.77	22.58	19.22	34.51	29.52	25.00	19.83	17.56	15.64
Na2O	1.09	1.76	2.39	1.53	0.79	1.72	2.33	2.82	3.34	1.27
K2O	2.17	0.94	0.99	1.00	1.30	0.82	0.99	1.10	1.27	1.30
P2O5	0.09	0.22	0.27	0.13	0.06	0.11	0.25	0.38	0.36	0.13
SO3	0.58	0.74	0.64	0.49	0.64	0.93	0.87	0.67	0.54	0.36
Cr2O3	0.02	0.03	0.04	0.19	0.02	0.02	0.03	0.04	0.05	0.09
NiO	0.01	0.01	0.01	0.04	0.01	0.01	0.01	0.01	0.01	0.03
CuO	1.27	1.40	1.30	2.57	1.45	1.27	1.06	1.02	0.98	2.85
Total	83.72	87.90	87.13	83.31	83.88	87.02	89.14	88.54	89.12	83.51

Sample	+75	+53	+25	-25	Element	Conversion factor	+75	+53	+25	-25
	Normalised major oxides wt %						Element %			
SiO_2	42.99	43.25	47.80	44.22	**Si**	0.47	20.10	20.22	22.35	20.67
TiO_2	0.73	1.13	1.48	1.10	**Ti**	0.60	0.43	0.68	0.89	0.66
Al_2O_3	12.30	9.15	11.24	12.57	**Al**	0.53	6.51	4.84	5.95	6.65
Fe_2O_3	6.60	6.71	5.94	10.73	**Fe**	0.70	4.61	4.69	4.16	7.50
MnO	0.11	0.15	0.14	0.34	**Mn**	0.77	0.09	0.12	0.11	0.26
MgO	1.44	1.09	0.99	0.84	**Mg**	0.60	0.87	0.66	0.60	0.51
CaO	29.58	32.73	25.91	23.07	**Ca**	0.71	21.14	23.39	18.52	16.49
Na_2O	1.30	2.01	2.75	1.84	**Na**	0.74	0.96	1.49	2.04	1.37
K_2O	2.59	1.06	1.13	1.19	**K**	0.83	2.15	0.88	0.94	0.99
P_2O_5	0.11	0.24	0.31	0.16	**P**	0.44	0.05	0.11	0.13	0.07
SO_3	0.69	0.84	0.74	0.59	**S**	0.40	0.28	0.34	0.30	0.24
Cr_2O_3	0.03	0.03	0.05	0.22	**Cr**	0.68	0.02	0.02	0.04	0.15
NiO	0.01	0.01	0.01	0.04	**Ni**	0.79	0.01	0.01	0.01	0.03
CuO	1.52	1.59	1.50	3.08	**Cu**	0.80	1.22	1.27	1.19	2.46

Sample	+150	+106	+75	+53	+25	-25	Element	Conversion Factor	+150	+106	+75	+53	+25	-25
	Normailised major oxides wt %								Elements %					
SiO_2	35.50	43.82	48.10	51.75	51.46	45.03	**Si**	0.47	16.60	20.48	22.49	24.19	24.06	21.05
TiO_2	0.66	0.68	1.01	1.37	1.66	1.16	**Ti**	0.60	0.40	0.41	0.61	0.82	0.99	0.69
Al_2O_3	8.22	7.82	9.44	11.13	13.45	14.73	**Al**	0.53	4.35	4.14	5.00	5.89	7.12	7.79
Fe_2O_3	8.19	6.97	5.91	5.33	5.23	11.75	**Fe**	0.70	5.73	4.88	4.14	3.73	3.66	8.22
MnO	0.12	0.13	0.12	0.14	0.14	0.41	**Mn**	0.77	0.10	0.10	0.09	0.11	0.11	0.32
MgO	1.08	1.05	1.15	1.07	1.03	0.99	**Mg**	0.60	0.65	0.64	0.70	0.64	0.62	0.60
CaO	41.15	33.92	28.04	22.40	19.70	18.73	**Ca**	0.71	29.41	24.24	20.04	16.01	14.08	13.39
Na_2O	0.94	1.97	2.61	3.18	3.74	1.52	**Na**	0.74	0.70	1.46	1.94	2.36	2.78	1.13
K_2O	1.54	0.94	1.11	1.24	1.43	1.55	**K**	0.83	1.28	0.78	0.92	1.03	1.18	1.29
P_2O_5	0.07	0.13	0.28	0.43	0.40	0.15	**P**	0.44	0.03	0.06	0.12	0.19	0.17	0.07
SO_3	0.76	1.07	0.97	0.75	0.60	0.43	**S**	0.40	0.31	0.43	0.39	0.30	0.24	0.17
Cr_2O_3	0.03	0.02	0.03	0.05	0.05	0.10	**Cr**	0.68	0.02	0.01	0.02	0.03	0.04	0.07
NiO	0.01	0.01	0.01	0.01	0.01	0.03	**Ni**	0.79	0.01	0.01	0.00	0.01	0.01	0.02
CuO	1.73	1.46	1.19	1.16	1.10	3.41	**Cu**	0.80	1.38	1.17	0.95	0.92	0.88	2.73

Bulk mineralogy of HQ and LQ ore as determined by XRD

Mineral	HQ ore		LQ ore	
	P_{80} = 150 µm	P_{80} = 75 µm	P_{80} = 150 µm	P_{80} = 75 µm
Chalcopyrite	6.48	5.87	4.61	4
Bornite	0.92	0.71	0	0
Calcite	11.54	14.2	24.74	25.03
Chalcocite	5.35	3.69	2.69	2.55
Chlorite	0.04	0.5	5.17	4.66
Kaolinite	2.56	2.21	1.42	1.12
Malachite	1.45	1.56	3.01	2.75
Muscovite	10.69	10.98	13.62	12.61
Quartz	26.51	26.26	23.04	23.79
Bytownite	20.18	20.2	21.7	23.49
Pyrite	0.13	0.28		
Pyrrhotite	0.98	1.27		
Covellite	0.56	0.35		
Cuprite	0.26	0.2		
Diopside	9.97	9.33		
Enstatite	1.14	1.16		
Hornblende magnesian iron	0	0		
Lizardite	1.06	0.61		
Talc	0.19	0.63		

QEMSCAN SIP list and mineral groupings

Mineral group	Minerals Included in the group
Pyrite	Pyrite, sulphide trap
Pyrrhotite	Pyrrhotite
Chalcopyrite	Chalcopyrite, Cu.Fe Sulphide trap, Cu sulphide trap, Chalcopyrite-TiO2 interface, Cu sulphide alunite interface, Cu sulphide-Al.Si interface, Cu.Ag.Zn.Fe sulphide, Cu.As-sulphide (Enargite), Cu.Zn-sulphide (sulfosalt), Cu.Sb sulphide,
Bornite	Bornite, Bornite rim
Chalcocite/Digenite	Chalcocite, Chalcocite/digenite-pyrite interface
Covellite	Covellite
Other sulphides	Galena, Galena rim, Pb-sulohide trap, Molybdenite, Molybdenite rim, Shalerite, Sphalerite rim, Zn-sulphide trap, Pentlandite, Conalite, Sb-sulphide, Bismuthinite, Arsenopyrite, Aspy-silicate interface, Mn-sulphide, Ag-sulphide, As-sulphide (Realgar)
Cuprite	Cuprite, Cu metal
Malachite/Azurite	Malachite/Azurite, Pseudomalachite, Cu-sulphate
Chrysocolla	Cu-sulphide incl in silicate, Chrysocolla, Chrysocolla-sulphate interface, Chrysocolla-CuClay interface, Chrysocolla-FeOx/Hydr interface, Cu trap, Chrysocolla interface with Mn

Amphibole	Serpentine, Talc, Olivine, Garnet, Pyroxine (Diopside, En-Fs, Augite), Amphibole, Wollastonite, olivine sulphide intergrowth, Pyroxene sulpide intergrowth, Talc sulphide intergrowth, Serpentine-olivine intergrowth, Diopside-calcite intergrowth, Ni-sulphide serpentine intergrowth, Enstatite, MgSi trap, Enstatite rim
Mica	Cu-Mica, Altered Cu-Mica, Muscovite, Muscovite-Pyrite interface, Biotite/Phlogopite, Biotite (Fe-Mica), Cu-Biotite/Phlogopite, K.Al-silicate trap, Chlorite, Cu-chlorite, Chlorite-Quartz interface, AlSilicate trap, Silicate trap, TiO2-Muscovite interface, Biotite-silicate intergrowth
Kaolinite	Kaolinite (clay), Kaolinite (clay_Fe), Cu-Kaolinite (clay), Kaolinite-Cu-sulphide interface, Pyrophyllite
Plagioclase-Feldpar	Plagioclase Feldspar, Feldspar-Anorthite, Feldspar-Albite, Albite-chalcocite interface, Albite-chalcopyrite interface, K-Feldspar, Alkali Feldspar, Epidote, TiO2-Albite interface, TiO2_AlkaliFeldspar interface, Plagioclase rim, CaSi trap
Quartz	Quartz, Quartz-AlSilicate interface, Quartz-Cu mineral interface, Quartz sulphide interface, TiO2-silicate interface, Ti mineral in Quartz, Ti-mineral trap, Quartz-CaSulphate interface
Calcite	Calcite, Dolomite, Mn-Carbonate, Magnesite, Sidenite, Ankerite
Fe-Ti minerals	Rutile, Ilmenite, Mn-Ilmenite, Ti-Hematite, Magnetite, Cr-Spinel, Cr trap
Limonite	FeOx/Hydr infiltrated Muscovite, Goethite, Limonite, Cu-Limonite, Limonite-clay interface, FeOx/Hydrox trap, FeOx/hydrox-silicate trap, FeOx/hydrox-silicate interface, FeOx/hydrox-carbonate interface

Others	Idaite, Cu-chloride (Atacamite), Turquoise, Fe-Olivine, Mg-Olivine, Pyrope, Spessartine, Grosular, Andradite, Gross-Spess-Alm, Andr-Gross, Spess-Alm-Gross, Alm-Gross-Spess, Alm-Gross, Alm-Pyro-Gross, Alm-Pyro Spess, Alm-Goss, Alm-Spess, Spess-Gross, Uvarovite, Toumaline, Staurolite, Sphene, Zircon, Andalusite/Ky/Sill, Mn-Silicate (Fe,Ca), Mn.FeSilicate-sulphide interface, Silicate-sulphate interface, Silicate-Cusulphate interface, Apatite, Apatite-Quartz interface, Apatite-Albite interface, Apatite-KFeldspar interface, Apatite-FeSulphide interface, Apatite-sulphide interface, Apatite in Ilmenite, Ce-phosphate (Monazite), Al-phosphate, Ca-phosphate trap, Zn-Spinel, Spinel, Cassiterite, Zr-oxide (Baddeleyite), Fluorite, Ca-Sulphate, CaSulphate-Quartz interface, Alunite, Alunite-Fe, Alunite-clay interface, Cu-Alunite-clay interface, Alunite-silicate interface, Alunite-Pb, Hydronium-Jarrosite, Jarosite, Barite, Gold, Electrum, Complex Ag-sulphide trap, Sb-mineral, Pb(Sb.Te)-mineral, Bi.Sb-sulphide, Pb.Sb.Cu-sulphide, U/Th oxide, Steel, HiBSE

Bulk mineralogy of HQ and LQ ore as determined by QEMSCAN

Mineral	HQ ore			LQ ore		
	P_{80} = 75 µm	P_{80} = 150 µm	Average	P_{80} = 75 µm	P_{80} = 150 µm	Average
Pyrite	2.55	3.35	2.95	1.10	0.83	0.96
Pyrrhotite	0.89	0.75	0.82	0.67	0.63	0.65
Chalcopyrite	3.60	4.24	3.92	1.14	0.78	0.96
Bornite	0.01	0.02	0.02	0.02	0.01	0.01
Chalcocite / Digenite	0.01	0.05	0.03	0.00	0.01	0.01
Covellite	0.02	0.02	0.02	0.02	0.01	0.01
Other sulphides	0.03	0.03	0.03	0.00	0.09	0.05
Cuprite	0.00	0.01	0.01	0.04	0.02	0.03
Malachite/Azurite	0.03	0.03	0.03	0.08	0.07	0.08
Chrysocolla	0.14	0.09	0.11	3.69	3.93	3.81
Amphibole	0.82	0.65	0.74	3.80	3.79	3.80
Mica	23.61	22.99	23.30	9.41	8.97	9.19
Kaolinite	0.28	0.22	0.25	1.29	2.02	1.66
Plagioclase-Feldspar	31.95	33.40	32.68	24.04	23.57	23.80
Quartz	25.86	24.98	25.42	17.29	15.48	16.39
Calcite	4.42	3.99	4.20	27.77	30.44	29.11
Fe-Ti minerals	1.66	1.69	1.67	1.09	1.10	1.09
Limonite	3.03	2.40	2.71	6.86	6.58	6.72
Others	1.10	1.09	1.09	1.71	1.66	1.69

Copper deportment in HQ and LQ ore as determined by QEMSCAN

Mineral	HQ ore			LQ ore		
	P_{80} = 75 µm	P_{80} = 150 µm	Average	P_{80} = 75 µm	P_{80} = 150 µm	Average
Chalcopyrite	93.96	93.27	93.62	26.23	18.23	22.23
Bornite	0.67	0.77	0.72	0.63	0.48	0.55
Chalcocite / Digenite	0.46	2.19	1.32	0.11	0.35	0.23
Covellite	0.62	0.70	0.66	0.44	0.38	0.41
Cuprite	0.32	0.39	0.35	2.11	0.95	1.53
Malachite/Azurite	1.27	0.96	1.12	2.98	2.88	2.93
Chrysocolla	2.22	1.27	1.74	60.20	66.01	63.10
Mica	0.04	0.04	0.04	0.05	0.08	0.06
Kaolinite	0.04	0.05	0.05	4.20	6.93	5.56
Plagioclase-Feldspar	0.08	0.06	0.07	0.07	0.15	0.11
Quartz	0.22	0.19	0.21	0.05	0.04	0.04
Limonite	0.05	0.07	0.06	2.91	3.52	3.22
Others	0.06	0.04	0.05	0.02	0.02	0.02

Normalised copper deportment by size fraction in HQ ore as determined by QEMSCAN

Mineral	P_{80} = 150 µm						P_{80} = 75 µm			
	-1000/+150	-150/+106	-106/+75	-75/+53	-53/+25	-25/+0	-150/+75	-75/+53	-53/+25	-25/+0
Chalcopyrite	90.19	96.98	96.28	96.03	94.71	87.21	97.18	94.12	95.19	89.79
Bornite	0.75	0.57	0.85	0.82	0.90	0.86	0.74	0.57	1.04	0.40
Chalcocite / Digenite	7.58	0.30	0.28	0.21	0.26	0.43	0.32	0.39	0.81	0.38
Covellite	0.18	0.18	0.52	0.62	0.63	2.55	0.37	0.30	0.53	1.21
Cuprite	0.00	0.28	0.29	0.44	0.58	1.11	0.00	1.20	0.01	0.06
Malachite/Azurite	0.04	0.82	0.80	0.49	1.68	2.82	0.05	1.87	0.96	2.14
Chrysocolla	0.87	0.66	0.70	1.04	0.83	3.80	1.02	1.22	1.12	5.09
Mica	0.04	0.01	0.03	0.02	0.07	0.08	0.04	0.02	0.04	0.06
Kaolinite	0.01	0.01	0.04	0.02	0.04	0.24	0.01	0.01	0.02	0.10
Plagioclase-Feldspar	0.00	0.01	0.03	0.04	0.03	0.31	0.02	0.02	0.02	0.25
Quartz	0.29	0.12	0.15	0.20	0.18	0.17	0.17	0.19	0.15	0.35
Limonite	0.02	0.04	0.03	0.04	0.04	0.30	0.05	0.06	0.06	0.05
Others	0.03	0.02	0.01	0.03	0.05	0.13	0.03	0.04	0.03	0.11

Normalised copper deportment by size fraction in LQ ore as determined by QEMSCAN

Mineral	P_{80} = 150 µm						P_{80} = 75 µm			
	-1000/+150	**-150/+106**	**-106/+75**	**-75/+53**	**-53/+25**	**-25/+0**	**-1000/+75**	**-75/+53**	**-53/+25**	**-25/+0**
Chalcopyrite	19.51	26.70	34.09	29.71	22.04	7.22	30.06	47.70	25.34	16.93
Bornite	0.56	0.62	1.14	0.81	0.47	0.17	0.44	1.28	0.61	0.43
Chalcocite / Digenite	0.45	1.31	0.38	0.43	0.08	0.04	0.13	0.18	0.19	0.03
Covellite	0.13	0.12	0.39	0.50	0.57	0.48	0.10	1.01	0.46	0.32
Cuprite	0.48	1.29	1.48	3.74	0.97	0.30	1.27	6.59	1.95	0.69
Malachite/Azurite	4.93	4.70	3.45	4.49	3.22	0.86	4.15	2.93	4.96	1.37
Chrysocolla	65.36	56.73	51.50	52.22	62.72	77.02	56.41	34.68	61.02	71.16
Mica	0.05	0.03	0.04	0.05	0.08	0.11	0.05	0.05	0.02	0.07
Kaolinite	2.58	3.56	3.56	4.11	5.58	11.64	2.32	1.83	3.02	6.49
Plagioclase-Feldspar	0.19	0.21	0.17	0.15	0.12	0.10	0.01	0.02	0.04	0.12
Quartz	0.04	0.04	0.05	0.07	0.05	0.02	0.03	0.05	0.02	0.08
Limonite	5.70	4.68	3.73	3.68	4.06	2.02	5.03	3.66	2.35	2.28
Others	0.01	0.01	0.02	0.02	0.04	0.01	0.00	0.01	0.02	0.03

Liberation and mineral associations of copper minerals in HQ and LQ ore

Ore	**Mineral**	**P_{80} = 75 µm**			**P_{80} = 150 µm**		
		Liberated	**Middling**	**Locked**	**Liberated**	**Middling**	**Locked**
HQ	Chalcopyrite	90.81	8.70	0.50	90.61	8.67	0.72
LQ	Chalcopyrite	55.19	42.73	2.08	51.48	44.55	3.96
	Chrysocolla	38.85	61.15	0	29.8	70.42	0

Ore	**Mineral**	**Associations**						
		Liberated	**Chrysocolla**	**Malachite**	**Cu-sulphides**	**Fe-sulphides**	**Others**	
HQ	Chalcopyrite	90.61	4.46	2.55	1.75	0.28	0.35	
LQ	Chalcopyrite	**Liberated**	**Chrysocolla**	**Malachite**	**Cu-sulphides**	**Fe-sulphides**	**Limonite**	**Others**
		51.48	42.85	3.91	0.75	0.30	0.07	0.63
	Chrysocolla	**Liberated**	**Chalcopyrite**	**Malachite**	**Quartz**	**Limonite**	**Others**	
		29.58	3.97	1.26	7.64	39.43	18.11	

Bulk mineralogy and copper deportment of LQ ore tailings pre and post sulphidisation

Bulk Mineralogy			Copper deportment		
Mineral	**SIBX tailings**	**Slug sulphidisation Tailings**	**Mineral**	**SIBX tailings**	**Slug sulphidisation Tailings**
Pyrite	0.27	0.05	Chalcopyrite	5.75	0.80
Pyrrhotite	0.47	0.19	Bornite	0.28	0.06
Chalcopyrite	0.21	0.03	Chalcocite / Digenite	0.04	0.01
Bornite	0.01	0.00	Covellite	0.16	0.10
Chalcocite / Digenite	0.00	0.00	Cuprite	1.14	0.09
Covellite	0.00	0.00	Malachite/Azurite	1.31	0.63
Other sulphides	0.06	0.05	Chrysocolla	78.77	84.79
Cuprite	0.02	0.00	Mica	0.09	0.14
Malachite/Azurite	0.03	0.01	Kaolinite	8.62	9.15
Chrysocolla	3.86	4.00	Plagioclase-Feldspar	0.02	0.02
Amphibole	6.17	5.54	Quartz	0.02	0.03
Mica	7.41	10.43	Limonite	3.79	4.18
Kaolinite	2.07	2.00	Others	0.00	0.00
Plagioclase-Feldspar	24.02	25.51			
Quartz	13.38	15.92			
Calcite	30.30	27.75			
Fe-Ti minerals	3.63	1.02			
Limonite	6.58	6.35			
Others	1.53	1.15			

Associations of chalcopyrite in LQ tailings pre-sulphidisation

Associations						
Liberated	**Chrysocolla**	**Malachite**	**Cu-Sulphides**	**Fe-sulphides**	**Limonite**	**Others**
74.79	18.61	3.31	1.43	0.45	0.25	1.17

Bulk mineralogy of LQ ASCu digestion residue

Mineral	**ASCu residue**
Pyrite	0.95
Pyrrhotite	0.87
Chalcopyrite	0.68
Bornite	0.02
Chalcocite / Digenite	0.02
Covellite	0.33
Other sulphides	0.03
Cuprite	0.00
Malachite/Azurite	0.01
Chrysocolla	0.09
Amphibole	0.87
Mica	18.97
Kaolinite	0.8
Plagioclase-Feldspar	39.64
Quartz	27.85
Calcite	0.10
Fe-Ti minerals	1.59
Limonite	7.63
Others	0.54

QEMSCAN data validation

Element	XRF assay (%)	QEMSCAN Assay (%)	Element	XRF assay (%)	QEMSCAN Assay (%)	Element	XRF assay (%)	QEMSCAN Assay (%)
Al	4.35	3.25	**K**	1.28	0.88	**P**	0.03	0.02
	4.14	2.98		0.78	0.38		0.06	0.05
	5.00	4.03		0.92	0.51		0.12	0.08
	5.89	4.79		1.03	0.58		0.19	0.14
	7.12	6.17		1.18	0.83		0.17	0.15
	7.79	6.24		1.29	1.05		0.07	0.05
Ca	29.41	18.66	**Mg**	0.65	0.20	**S**	0.31	1.00
	24.24	15.24		0.64	0.22		0.43	1.64
	20.04	12.47		0.70	0.27		0.39	1.56
	16.01	10.81		0.64	0.25		0.30	1.43
	14.08	8.96		0.62	0.25		0.24	0.87
	13.39	8.69		0.60	1.53		0.17	0.73
Cr	0.02	0.02	**Mn**	0.10	0.00	**Si**	16.60	15.05
	0.01	0.03		0.10	0.00		20.48	17.13
	0.02	0.04		0.09	0.00		22.49	20.56
	0.03	0.07		0.11	0.00		24.19	21.14
	0.04	0.07		0.11	0.01		24.06	22.00
	0.07	0.08		0.32	0.00		21.05	19.66
Cu	1.38	1.16	**Na**	0.70	0.89	**Ti**	0.40	0.26
	1.17	1.43		1.46	1.40		0.41	0.33
	0.95	1.24		1.94	2.04		0.61	0.53
	0.92	1.24		2.36	2.44		0.82	0.79
	0.88	1.09		2.78	3.08		0.99	1.03
	2.73	2.19		1.13	2.06		0.69	0.45
Fe	5.73	4.57	**Ni**	0.01	0.00			
	4.88	5.45		0.01	0.00			
	4.14	4.46		0.01	0.00			
	3.73	4.68		0.01	0.00			
	3.66	4.15		0.02	0.00			
	8.22	6.91						

Appendix B: Grind establishment

HQ ore grinding curves and PSD data

GRINDING CURVES									
75 µm					150 µm				
Time	Mass (g)	mass ret. (g)	%ret	%passing	Time	Mass (g)	mass ret. (g)	%ret	%passing
3	96.39	44.43	46.09	53.91	3	96.35	25.65	26.62	73.38
6	102.4	27.55	26.90	73.10	6	100.41	3.48	3.47	96.53
9	106.36	11.23	10.56	89.44	9	100.74	1.19	1.18	98.82
7 min 12 sec					3 min 42 sec				

Screen Size (µm)	**150 µm**			**75 µm**		
	Mass ret. (g)	**% ret.**	**cum. % Pass.**	**Mass ret. (g)**	**% ret.**	**cum. % Pass.**
1180	2.45	1.34	98.66	0.00	0.00	100.00
850	0.10	0.05	98.61	0.00	0.00	100.00
600	0.19	0.10	98.51	0.00	0.00	100.00
425	0.57	0.31	98.20	0.16	0.09	99.91
300	5.05	2.75	95.44	0.02	0.01	99.90
150	24.87	13.56	81.88	4.68	2.62	97.28
106	17.19	9.37	72.51	10.50	5.88	91.40
75	20.27	11.05	61.46	20.68	11.58	79.82
53	16.08	8.77	52.69	23.71	13.28	66.54
38	20.53	11.19	41.50	26.13	14.63	51.90
25	17.97	9.80	31.70	21.71	12.16	39.75
-25	58.14	31.70		70.97	39.75	
Total	183.41			178.56		

LQ ore grinding curves and PSD data

GRINDING CURVES									
75 µm					150 µm				
Time	Mass (g)	mass ret (g)	%ret	%passing	Time	Mass (g)	mass ret (g)	%ret	%passing
0	196.91	142.65	72.44	27.56	0	194.68	121.99	62.66	37.34
6	157.5	52.26	33.18	66.82	3	185.13	42.37	22.89	77.11
8	145.96	35.19	24.11	75.89	6	196.38	9.43	4.80	95.20
10	141.95	21.39	15.07	84.93	9	194.76	2.93	1.50	98.50
9 min					3 min 18 sec				

Screen Size (µm)	**150 µm**			**75 µm**		
	Mass ret. (g)	**% ret.**	**cum. % Pass.**	**Mass ret. (g)**	**% ret.**	**cum. % Pass.**
1180	0.00	0.00	100.00	0.00	0.00	100.00
850	0.00	0.00	100.00	0.00	0.00	100.00
600	3.63	1.64	98.36	0.00	0.00	100.00
425	5.04	2.28	96.08	0.00	0.00	100.00
300	7.47	3.38	92.70	0.00	0.00	100.00
150	27.59	12.48	80.22	3.22	1.75	98.25
106	32.50	14.70	65.52	8.14	4.43	93.82
75	26.60	12.03	53.49	22.75	12.38	81.44
53	21.90	9.91	43.58	31.63	17.21	64.23
38	22.56	10.20	33.37	27.40	14.91	49.31
25	13.67	6.18	27.19	21.32	11.60	37.71
-25	60.11	27.19		69.30	37.71	
Total	221.07			183.76		

LQ pre-sulphidisation and post-sulphidisation tailings PSDs

Screen (µm)	SIBX tailings			Slug sulphidisation tailings		
	Mass ret. (g)	**% ret.**	**Cum. % Pass.**	**Mass ret. (g)**	**% ret.**	**Cum. % Pass.**
+150	40.77	23.38	76.62	25.93	19.75	80.25
+106	30.01	17.21	59.42	16.12	12.28	67.97
+75	20.84	11.95	47.47	15.11	11.51	56.46
+53	15.98	9.16	38.30	10.33	7.87	48.59
+25	24.22	13.89	24.42	13.56	10.33	38.26
-25	42.58	24.42		50.23	38.26	
Total	174.4			131.28		

Appendix C: Synthetic plant water and flotation reagent make up

Synthetic plant water recipe

The chemically pure salts in the table below are added in the order in which they are listed to 20 L distilled water, stirring well after each addition to ensure complete mixing.

Salt	**Mass (g)**
$MgSO_4 \cdot 7H_2O$	12.3
$Mg(NO_3)_2 \cdot 6H_2O$	2.14
$Ca(NO_3)_2 \cdot 4H_2O$	4.72
$CaCl_2 \cdot 2H_2O$	2.94
NaCl	4.12
Na_2CO_3	0.6

Flotation reagent make up

Reagent	**state**	**Purity**	**Concenration**	**Make up**
SIBX	solid (powder)	100%	1 wt%	dissolve 1 g in distilled water, transfer to a 100 ml volumetric flask and make up the volume to the mark with distilled water
NaHS	Solid (pellets)	90%	10 wt%	dissolve 14.7 g in distilled water, transfer to a 100 ml volumetric flask and make up the volume to the mark with distilled water
Dow 200	Liquid	100%	100 wt%	Use neat

Appendix D: Batch and microflotation data

HQ ore solids and water recovery

SIBX dosage (g/t)	Run	Sample	Sample Wt (g)	Bottle + H_2O (g)	Bottle Wt (g)	Wash H_2O added to dish (g)	D + C +H_2O (g)	Dish (g)	H_2O recovered (g)	Total cons wt (g)	Solids recovery (%)	Water recovery (g)
30	1	Feed	9.62							88.23	9.10	220.85
		C1	62.65	554.57	376.73	177.84	591.44	214.78	136.17			
		C2	9.36	563.71	379.53	184.18	452.98	218.69	40.75			
		C3	11.87	556.13	304.26	251.87	506.7	214.5	28.46			
		C4	4.35	566.71	330.82	235.89	470.61	214.9	15.47			
		T1	10.33									
		T2	10.04									
		T3	861.36									
	2	Feed	10.28							90.73	9.30	235.75
		C1	65.8	523.02	343.43	179.59	602.3	214.78	142.13			
		C2	12.22	542.93	351.65	191.28	466.53	218.69	44.34			
		C3	8.52	537.03	284.76	252.27	509.18	214.5	33.89			
		C4	4.19	560.12	367.03	193.09	427.57	214.9	15.39			
		T1	11.8									
		T2	13.06									
		T3	859.88									
	3	Feed	13.5							93.2	9.54	231.34
		C1	60.26	524.42	327.44	196.98	618.32	214.78	146.3			
		C2	16.13	539.2	379.6	159.6	434.84	218.69	40.42			
		C3	11.3	567.3	366.88	200.42	461.35	214.5	35.13			
		C4	5.51	559.55	351.01	208.54	438.44	214.9	9.49			
		T1	10									
		T2	11.2									
		T3	862.67									

SIBX dosage (g/t)	Run	Sample	Sample Wt (g)	Bottle + H_2O (g)	Bottle Wt (g)	Wash H_2O added to dish (g)	D + C +H_2O (g)	Dish (g)	H_2O recovered (g)	Total cons wt (g)	Solids recovery (%)	Water recovery (g)
50	1	Feed	12.45							77.03	7.86	111.02
		C1	48.12	388.65	242.48	146.17	463.23	214.78	54.16			
		C2	12.27	398.53	246.79	151.74	408.43	218.69	25.73			
		C3	10.63	324.23	116.69	207.54	452.91	214.5	20.24			
		C4	6.01	348.46	166.7	181.76	413.56	214.9	10.89			
		T1	8.85									
		T2	10.86									
		T3	883.46									
	2	Feed	10.05							76.82	7.97	119.37
		C1	49.61	343.43	169.81	173.62	499.48	214.78	61.47			
		C2	11.76	351.65	180.15	171.5	424.64	218.69	22.69			
		C3	9.96	534.09	318.13	215.96	465.7	214.5	25.28			
		C4	5.49	367.03	150.95	216.08	446.4	214.9	9.93			
		T1	10.16									
		T2	10.62									
		T3	866.47									
	3	Feed	10.38							82.79	8.39	138.63
		C1	50.76	327.44	138.85	188.59	515.62	214.78	61.49			
		C2	15.14	379.6	179.52	200.08	469.27	218.69	35.36			
		C3	11.27	366.88	144.19	222.69	474.7	214.5	26.24			
		C4	5.62	351.01	146.49	204.52	440.58	214.9	15.54			
		T1	12.03									
		T2	12.98									
		T3	879.17									

Grade/recovery data (30 g/t SIBX on HQ ore)

RUN 1

Date	Flotation Stage	Concentrate			Assay (%)		Wt (g)		Cumulative Wt. (g)		Recovery %		Cumulative Recovery		Cum Grade %		Upgrading	
		Wt (g)	Wt %	Cumm Wt (g)	TCu	Fe	TCu	Fe	TCu	Fe	TCu	Fe	TCu	Fe	TCu	Fe	TCu	Fe
9-Jun-14	Conc 1	62.65	6.46	62.65	16.10	27.05	10.09	16.95	10.09	16.95	80.70	47.29	80.70	47.29	16.10	27.05	12.49	7.32
	Conc 2	9.36	0.96	72.01	8.43	14.97	0.79	1.40	10.88	18.35	6.31	3.91	87.01	51.20	15.10	25.48	11.72	6.90
	Conc 3	11.87	1.22	83.88	4.21	9.89	0.50	1.17	11.38	19.52	4.00	3.28	91.01	54.47	13.56	23.27	10.52	6.30
	Conc 4	4.35	0.45	88.23	2.52	8.17	0.11	0.36	11.49	19.88	0.88	0.99	91.89	55.47	13.02	22.53	10.10	6.10
	Tails	881.73	9.10		0.12	1.81	1.01	15.96			8.11	44.53						
	Head	969.96			1.29	3.69	12.50	35.84										

RUN 2

Date	Flotation Stage	Concentrate			Assay (%)		Wt (g)		Cumulative Wt. (g)		Recovery %		Cumulative Recovery		Cum Grade %		Upgrading	
		Wt (g)	Wt %	Cumm Wt (g)	TCu	Fe	TCu	Fe	TCu	Fe	TCu	Fe	TCu	Fe	TCu	Fe	TCu	Fe
13-Jun-14	Conc 1	65.80	6.78	65.8	12.69	20.56	8.35	13.53	8.35	13.53	72.27	41.66	72.27	41.66	12.69	20.56	10.71	6.18
	Conc 2	12.22	1.26	78.02	8.17	14.40	1.00	1.76	9.35	15.29	8.64	5.42	80.91	47.08	11.98	19.60	10.12	5.89
	Conc 3	8.52	0.88	86.54	4.06	9.27	0.35	0.79	9.69	16.08	2.99	2.43	83.91	49.51	11.20	18.58	9.46	5.58
	Conc 4	4.19	0.43	90.73	1.09	6.94	0.05	0.29	9.74	16.37	0.40	0.90	84.30	50.41	10.74	18.04	9.06	5.42
	Tails	884.74	9.35		0.21	1.82	1.81	16.10			15.70	49.59						
	Head	975.47			1.18	3.33	11.55	32.47										

RUN 3

Date	Flotation Stage	Concentrate			Assay (%)		Wt (g)		Cumulative Wt. (g)		Recovery %		Cumulative Recovery		Cum Grade %		Upgrading	
		Wt (g)	Wt %	Cumm Wt (g)	TCu	Fe	TCu	Fe	TCu	Fe	TCu	Fe	TCu	Fe	TCu	Fe	TCu	Fe
18-Jun-14	Conc 1	60.26	6.21	60.26	16.04	24.85	9.67	14.97	9.67	14.97	73.57	43.42	73.57	43.42	16.04	24.85	11.93	7.04
	Conc 2	16.13	1.66	76.39	9.31	16.90	1.50	2.73	11.17	17.70	11.43	7.90	85.00	51.32	14.62	23.17	10.87	6.56
	Conc 3	11.30	1.16	87.69	7.09	13.29	0.80	1.50	11.97	19.20	6.10	4.35	91.10	55.67	13.65	21.90	10.15	6.20
	Conc 4	5.51	0.57	93.2	3.57	10.37	0.20	0.57	12.17	19.77	1.50	1.66	92.60	57.33	13.05	21.22	9.71	6.01
	Tails	883.87	9.61		0.11	1.67	0.97	14.72			7.40	42.67						
	Head	977.07			1.34	3.53	13.14	34.49										

Grade/recovery data (50 g/t SIBX on HQ ore)

RUN 1

Date	Flotation Stage	Concentrate			Assay (%)		Wt (g)		Cumulative Wt. (g)		Recovery %		Cumulative Recovery		Cum Grade %		Upgrading	
		Wt (g)	Wt %	Cumm Wt (g)	TCu	Fe	TCu	Fe	TCu	Fe	TCu	Fe	TCu	Fe	TCu	Fe	TCu	Fe
9-Jun-14	Conc 1	48.12	4.91	48.12	16.43	21.07	7.91	10.14	7.91	10.14	63.16	32.21	63.16	32.21	16.43	21.07	12.87	6.56
	Conc 2	12.27	1.25	60.39	12.47	18.05	1.53	2.21	9.44	12.35	12.22	7.04	75.38	39.25	15.63	20.46	12.24	6.37
	Conc 3	10.63	1.08	71.02	9.25	15.85	0.98	1.68	10.42	14.04	7.86	5.35	83.24	44.60	14.67	19.77	11.49	6.16
	Conc 4	6.01	0.61	77.03	5.61	11.35	0.34	0.68	10.76	14.72	2.69	2.17	85.93	46.77	13.96	19.11	10.93	5.95
	Tails	903.17	7.86		0.20	1.86	1.76	16.75			14.07	53.23						
	Head	980.2			1.28	3.21	12.52	31.47										

RUN 2

Date	Flotation Stage	Concentrate			Assay (%)		Wt (g)		Cumulative Wt. (g)		Recovery %		Cumulative Recovery		Cum Grade %		Upgrading	
		Wt (g)	Wt %	Cumm Wt (g)	TCu	Fe	TCu	Fe	TCu	Fe	TCu	Fe	TCu	Fe	TCu	Fe	TCu	Fe
13-Jun-14	Conc 1	49.61	5.06	49.61	15.60	24.10	7.74	11.96	7.74	11.96	72.39	37.04	72.39	37.04	15.60	24.10	14.07	7.20
	Conc 2	11.76	1.20	61.37	10.95	16.75	1.29	1.97	9.03	13.93	12.04	6.10	84.43	43.14	14.71	22.69	13.26	6.78
	Conc 3	9.96	1.02	71.33	7.10	12.56	0.71	1.25	9.73	15.18	6.61	3.88	91.05	47.02	13.65	21.28	12.31	6.35
	Conc 4	5.49	0.56	76.82	5.31	10.13	0.29	0.56	10.03	15.73	2.73	1.72	93.78	48.74	13.05	20.48	11.77	6.12
	Tails	887.25	7.84		0.08	1.87	0.67	16.55			6.22	51.26						
	Head	964.07			1.11	3.35	10.69	32.28										

RUN 3

Date	Flotation Stage	Concentrate			Assay (%)		Wt (g)		Cumulative Wt. (g)		Recovery %		Cumulative Recovery		Cum Grade %		Upgrading	
		Wt (g)	Wt %	Cumm Wt (g)	TCu	Fe	TCu	Fe	TCu	Fe	TCu	Fe	TCu	Fe	TCu	Fe	TCu	Fe
18-Jun-14	Conc 1	50.76	5.18	50.76	17.61	24.70	8.94	12.54	8.94	12.54	72.53	39.47	72.53	39.47	17.61	24.70	14.10	7.67
	Conc 2	15.14	1.54	65.9	7.19	12.43	1.09	1.88	10.03	14.42	8.83	5.92	81.36	45.39	15.22	21.88	12.19	6.80
	Conc 3	11.27	1.15	77.17	6.56	12.32	0.74	1.39	10.77	15.81	6.00	4.37	87.36	49.76	13.95	20.48	11.17	6.36
	Conc 4	5.62	0.57	82.79	4.39	11.30	0.25	0.64	11.01	16.44	2.00	2.00	89.36	51.76	13.30	19.86	10.65	6.17
	Tails	904.18	8.45		0.15	1.70	1.31	15.33			10.64	48.24						
	Head	986.97			1.25	3.22	12.32	31.77										

LQ ore solids and water recovery- batch flotation with SIBX only

SIBX Dosage (g/t)	Sample	Sample Wt (g)	Bottle + H_2O (g)	Bottle Wt (g)	Wash H_2O added to dish (g)	D + C +H_2O (g)	Dish (g)	H_2O recovered	Total cons wt (g)	Solids recovery (%)	Water recovery (g)
20	Feed	8.39							23.59	2.45	192.32
	C1	14.81	389.66	289.82	99.84	405.09	214.78	75.66			
	C2	5.01	496.87	416.79	80.08	367.16	218.69	63.38			
	C3	2.4	548.72	410.96	137.76	390.34	214.5	35.68			
	C4	1.37	541.06	405.54	135.52	369.39	214.9	17.6			
	T1	8.39									
	T2	9.53									
	T3	919.58									
30	Feed	7.88							24.58	2.53	196.43
	C1	17.69	289.82	184.07	105.75	444.62	214.78	106.4			
	C2	3.47	416.79	323.71	93.08	363.23	218.69	47.99			
	C3	2.15	410.96	267.42	143.54	390.3	214.5	30.11			
	C4	1.27	405.54	249.94	155.6	383.7	214.9	11.93			
	T1	9.64									
	T2	10.7									
	T3	926.67									
50	Feed	14.98							23.93	2.42	129.42
	C1	18.82	184.07	106.02	78.05	362.36	214.78	50.71			
	C2	2.19	323.71	236.75	86.96	351.69	218.69	43.85			
	C3	1.49	267.42	131.3	136.12	378.15	214.5	26.04			
	C4	1.43	249.94	118.49	131.45	356.6	214.9	8.82			
	T1	14.05									
	T2	13.79									
	T3	937.43									

SIBX Dosage (g/t)	Sample	Sample Wt (g)	Bottle + H_2O (g)	Bottle Wt (g)	Wash H_2O added to dish (g)	D + C +H_2O (g)	Dish (g)	H_2O recovered	Total mass in conc	Solids recovery (%)	Water recovery (g)
20	Feed	9.53							19.8	2.00	174.23
	C1	13.14	557.68	434.98	122.7	431.79	214.78	81.17			
	C2	3.53	544.71	457.79	86.92	352.99	218.69	43.85			
	C3	2.02	566.07	480.17	85.9	337.13	214.5	34.71			
	C4	1.11	545.18	428.72	116.46	346.97	214.9	14.5			
	T1	8.84									
	T2	7.9									
	T3	954.32									
30	Feed	7.35							23.4	2.42	165.94
	C1	14.42	434.98	314.98	120	410.01	214.78	60.81			
	C2	4.9	457.79	368.5	89.29	361.03	218.69	48.15			
	C3	2.83	480.17	387.48	92.69	350.46	214.5	40.44			
	C4	1.25	428.72	293.19	135.53	368.22	214.9	16.54			
	T1	8.76									
	T2	8.49									
	T3	927.38									
50	Feed	10.12							17.63	1.77	142.41
	C1	10.47	314.98	168.35	146.63	433.59	214.78	61.71			
	C2	3.99	368.5	268.68	99.82	361.86	218.69	39.36			
	C3	2.18	387.48	307.49	79.99	327.33	214.5	30.66			
	C4	0.99	293.19	163.02	130.17	356.74	214.9	10.68			
	T1	9.87									
	T2	10.88									
	T3	958.67									

Grade/Recovery data (20 g/t SIBX on LQ ore)

RUN 1

Date	Flotation Stage	Concentrate			Assay (%)			Wt (g)			Cumulative Wt. (g)			Recovery %			Cumulative Recovery			Cum Grade %			Upgrading		
		Wt (g)	Wt %	Cumm W	TCu	ASCu	Fe	TCu	ASCu	Fe	TCu	ASCu	Fe	TCu	ASCu	Fe	TCu	ASCu	Fe	TCu	ASCu	Fe	TCu	ASCu	Fe
14-Oct-13	Conc 1	14.81	1.54	14.81	17.20	1.36	22.58	2.55	0.20	3.34	2.55	0.20	3.34	18.19	3.95	10.13	18.19	3.95	10.13	17.20	1.36	22.58	11.81	2.56	6.57
	Conc 2	5.01	0.52	19.82	9.10	1.34	18.01	0.46	0.07	0.90	3.00	0.27	4.25	3.26	1.32	2.73	21.45	5.27	12.87	15.15	1.35	21.42	10.40	2.55	6.24
	Conc 3	2.40	0.25	22.22	4.76	1.22	12.58	0.11	0.03	0.30	3.12	0.30	4.55	0.82	0.57	0.91	22.26	5.84	13.78	14.03	1.34	20.47	9.63	2.53	5.96
	Conc 4	1.37	0.14	23.59	4.19	1.35	14.09	0.06	0.02	0.19	3.17	0.32	4.74	0.41	0.36	0.58	22.67	6.21	14.36	13.46	1.34	20.10	9.24	2.53	5.85
	Tails	937.50	2.45		1.16	0.51	3.015	10.83	4.78	28.27				77.33	93.79	85.64									
	Head	961.09			1.46	0.53	3.43	14.00	5.10	33.01															

RUN 2

Date	Flotation Stage	Concentrate			Assay (%)			Wt (g)			Cumulative Wt. (g)			Recovery %			Cumulative Recovery			Cum Grade %			Upgrading		
		Wt (g)	Wt %	umm Wt (	TCu	ASCu	Fe	TCu	ASCu	Fe	TCu	ASCu	Fe	TCu	ASCu	Fe	TCu	ASCu	Fe	TCu	ASCu	Fe	TCu	ASCu	Fe
23-Oct-13	Conc 1	13.14	1.33	13.14	14.60	0.73	24.52	1.92	0.10	3.22	1.92	0.10	3.22	13.58	2.03	8.94	13.58	2.03	8.94	14.60	0.73	24.52	10.24	1.53	6.74
	Conc 2	3.53	0.36	16.67	9.88	0.97	21.83	0.35	0.03	0.77	2.27	0.13	3.99	2.47	0.72	2.14	16.04	2.75	11.07	13.60	0.78	23.95	9.54	1.64	6.58
	Conc 3	2.02	0.20	18.69	6.65	1.09	18.03	0.13	0.02	0.36	2.40	0.15	4.36	0.95	0.47	1.01	16.99	3.22	12.08	12.85	0.81	23.31	9.01	1.71	6.41
	Conc 4	1.11	0.11	19.8	6.91	1.00	17.16	0.08	0.01	0.19	2.48	0.16	4.55	0.54	0.23	0.53	17.54	3.45	12.61	12.52	0.82	22.97	8.78	1.73	6.31
	Tails	971.06	2.00		1.20	0.47	3.245	11.65	4.56	31.51				82.46	96.55	87.39									
	Head	990.86			1.43	0.48	3.64	14.13	4.73	36.06															

Grade/Recovery data (30 g/t SIBX on LQ ore)

RUN 1

Date	Flotation Stage	Concentrate			Assay (%)			Wt (g)			Cumulative Wt. (g)			Recovery %			Cumulative Recovery			Cum Grade %			Upgrading		
		Wt (g)	Wt %	Cumm Wt (g)	TCu	ASCu	Fe	TCu	ASCu	Fe	TCu	ASCu	Fe	TCu	ASCu	Fe	TCu	ASCu	Fe	TCu	ASCu	Fe	TCu	ASCu	Fe
14-Oct-13	Conc 1	17.69	1.82	17.69	16.20	1.26	23.2	2.87	0.22	4.10	2.87	0.22	4.10	21.86	4.69	12.28	21.86	4.69	12.28	16.20	1.26	23.20	12.01	2.57	6.74
	Conc 2	3.47	0.36	21.16	7.65	1.20	16.32	0.27	0.04	0.57	3.13	0.26	4.67	2.02	0.88	1.69	23.89	5.56	13.97	14.80	1.25	22.07	10.97	2.55	6.42
	Conc 3	2.15	0.22	23.31	5.41	1.18	15.03	0.12	0.03	0.32	3.25	0.29	4.99	0.89	0.53	0.97	24.77	6.09	14.94	13.93	1.24	21.42	10.33	2.54	6.23
	Conc 4	1.27	0.13	24.58	4.75	1.31	12.64	0.06	0.02	0.16	3.31	0.31	5.15	0.46	0.35	0.48	25.23	6.44	15.42	13.46	1.25	20.97	9.97	2.55	6.10
	Tails	947.01	2.53		1.04	0.47	2.985	9.80	4.45	28.27				74.77	93.56	84.58									
	Head	971.6			1.35	0.49	3.44	13.11	4.76	33.42															

RUN 2

Date	Flotation Stage	Concentrate			Assay (%)			Wt (g)			Cumulative Wt. (g)			Recovery %			Cumulative Recovery			Cum Grade %			Upgrading		
		Wt (g)	Wt %	Cumm Wt (g)	TCu	ASCu	Fe	TCu	ASCu	Fe	TCu	ASCu	Fe	TCu	ASCu	Fe	TCu	ASCu	Fe	TCu	ASCu	Fe	TCu	ASCu	Fe
23-Oct-13	Conc 1	14.42	1.49	14.42	19.80	1.22	21.01	2.86	0.18	3.03	2.86	0.18	3.03	18.87	4.05	8.49	18.87	4.05	8.49	19.80	1.22	21.01	12.67	2.72	5.70
	Conc 2	4.90	0.51	19.32	9.50	1.23	17.1	0.47	0.06	0.84	3.32	0.24	3.87	3.08	1.39	2.35	21.95	5.44	10.84	17.19	1.22	20.02	11.00	2.72	5.43
	Conc 3	2.83	0.29	22.15	9.29	1.11	22.23	0.26	0.03	0.63	3.58	0.27	4.50	1.74	0.72	1.76	23.69	6.16	12.61	16.18	1.21	20.30	10.35	2.69	5.51
	Conc 4	1.25	0.13	23.4	5.36	1.09	15	0.07	0.01	0.19	3.65	0.28	4.68	0.44	0.31	0.53	24.13	6.48	13.13	15.60	1.20	20.02	9.98	2.68	5.43
	Tails	944.63	2.42		1.22	0.43	3.28	11.48	4.06	30.98				75.87	93.52	86.87									
	Head	968			1.56	0.45	3.68	15.13	4.34	35.67															

Grade/Recovery data (50 g/t SIBX on LQ ore)

RUN 1

Date	Flotation	Concentrate			Assay (%)			Wt (g)			Cumulative Wt. (g)			Recovery %			Cumulative Recovery			Cum Grade %			Upgrading		
	Stage	Wt (g)	Wt %	Cumm Wt (g)	TCu	ASCu	Fe	TCu	ASCu	Fe	TCu	ASCu	Fe	TCu	ASCu	Fe	TCu	ASCu	Fe	TCu	ASCu	Fe	TCu	ASCu	Fe
14-Oct-13	Conc 1	18.82	1.90	18.82	10.55	0.74	19.45	1.99	0.14	3.66	1.99	0.14	3.66	13.87	2.98	10.27	13.87	2.98	10.27	10.55	0.74	19.45	7.29	1.56	5.40
	Conc 2	2.19	0.22	21.01	4.56	0.94	12.41	0.10	0.02	0.27	2.09	0.16	3.93	0.70	0.44	0.76	14.57	3.42	11.03	9.93	0.76	18.72	6.86	1.61	5.19
	Conc 3	1.49	0.15	22.5	4.07	1.10	11.48	0.06	0.02	0.17	2.15	0.18	4.10	0.42	0.35	0.48	14.99	3.77	11.51	9.54	0.78	18.24	6.59	1.66	5.06
	Conc 4	1.43	0.14	23.93	3.64	0.92	12.81	0.05	0.01	0.18	2.20	0.19	4.29	0.36	0.28	0.51	15.36	4.05	12.02	9.19	0.79	17.91	6.35	1.67	4.97
	Tails	965.27	2.42		1.26	0.47	3.25	12.11	4.49	31.37				84.64	95.95	87.98									
	Head	989.2			1.45	0.47	3.60	14.31	4.68	35.66															

RUN 2

Date	Flotation	Concentrate			Assay (%)			Wt (g)			Cumulative Wt. (g)			Recovery %			Cumulative Recovery			Cum Grade %			Upgrading		
	Stage	Wt (g)	Wt %	Cumm Wt (g)	TCu	ASCu	Fe	TCu	ASCu	Fe	TCu	ASCu	Fe	TCu	ASCu	Fe	TCu	ASCu	Fe	TCu	ASCu	Fe	TCu	ASCu	Fe
23-Oct-13	Conc 1	10.47	1.05	10.47	13.60	1.02	23.25	1.42	0.11	2.43	1.42	0.11	2.43	10.38	2.01	6.27	10.38	2.01	6.27	13.60	1.02	23.25	9.88	1.91	5.968
	Conc 2	3.99	0.40	14.46	9.92	0.96	20.32	0.40	0.04	0.81	1.82	0.15	3.25	2.89	0.72	2.09	13.27	2.73	8.35	12.58	1.00	22.44	9.15	1.88	5.761
	Conc 3	2.18	0.22	16.64	6.58	1.06	15.45	0.14	0.02	0.34	1.96	0.17	3.58	1.05	0.43	0.87	14.31	3.16	9.22	11.80	1.01	21.53	8.58	1.89	5.525
	Conc 4	0.99	0.10	17.63	5.11	0.99	15	0.05	0.01	0.15	2.01	0.18	3.73	0.37	0.18	0.38	14.68	3.35	9.60	11.42	1.01	21.16	8.30	1.89	5.431
	Tails	979.42	1.77		1.20	0.53	3.59	11.70	5.14	35.11				85.32	96.65	90.40									
	Head	997.1			1.38	0.53	3.90	13.72	5.32	38.84															

LQ ore solids and water recovery- batch flotation with SIBX and slug sulphidisation

NaHS (g/t)	Sample	Sample Wt (g)	Bottle + H_2O (g)	Bottle Wt (g)	Wash H_2O added to dish (g)	D + C +H_2O (g)	Dish (g)	H_2O recovered	Total cons wt (g)	Solids recovery (%)	Water recovery (g)
250	Feed	11.53							28.57	2.92	208.32
	C1	13.32	396.81	261.78	135.03	407.18	214.78	44.05			
	C2	5.4	437.42	328.36	109.06	373.2	218.69	40.05			
	C3	6.47	341.79	194.01	147.78	444.86	214.5	76.11			
	C4	3.38	380.89	223.42	157.47	423.86	214.9	48.11			
	T1	10.88									
	T2	10.74									
	T3	929.17									
250	Feed	12.25							33.68	3.48	265.61
	C1	15.66	547.83	406.86	140.97	427.24	214.78	55.83			
	C2	5.41	531.43	411.39	120.04	380.21	218.69	36.07			
	C3	9.09	523.95	354.94	169.01	512.19	214.5	119.59			
	C4	3.52	537.17	399.93	137.24	409.78	214.9	54.12			
	T1	13.14									
	T2	14.2									
	T3	907.47									
500	Feed	11.31							34.52	3.54	365.68
	C1	14.41	519.76	376.68	143.08	436.83	214.78	64.56			
	C2	5	549.2	444.65	104.55	378.29	218.69	50.05			
	C3	10.93	535.66	400.43	135.23	542.03	214.5	181.37			
	C4	4.18	537.33	397.77	139.56	428.34	214.9	69.7			
	T1	8.31									
	T2	15.03									
	T3	917.3									
500	Feed	8.42							29.36	3.05	300.59
	C1	15.15	410.94	302.03	108.91	402.63	214.78	63.79			
	C2	4.48	408.92	282.62	126.3	376.27	218.69	26.8			
	C3	6.01	543.11	365.87	177.24	557.12	214.5	159.37			
	C4	3.72	569.13	429.82	139.31	408.56	214.9	50.63			
	T1	8.15									
	T2	7.85									
	T3	917.5									

Grade/Recovery data (250 g/t NaHS & 25 g/t SIBX on LQ ore)

Run 1

Flotation Stage	Concentrate			Assay (%)			Wt (g)			Cumulative Wt. (g)			Recovery %			Cumulative Recovery			Cum Grade %			Upgrading		
	Wt (g)	Wt %	Cumm Wt (g)	TCu	ASCu	Fe	TCu	ASCu	Fe	TCu	ASCu	Fe	TCu	ASCu	Fe	TCu	ASCu	Fe	TCu	ASCu	Fe	TCu	ASCu	Fe
																0	0	0						
Conc 1	13.32	1.36	13.32	23.90	2.54	11.58	3.18	0.34	1.54	3.18	0.34	1.54	21.83	3.66	4.95	21.83	3.66	4.95	23.90	2.54	11.58	16.05	2.69	3.64
Conc 2	5.40	0.55	18.72	12.79	2.68	11.85	0.69	0.14	0.64	3.87	0.48	2.18	4.74	1.57	2.05	26.57	5.22	7.00	20.70	2.58	11.66	13.90	2.73	3.66
Conc 3	6.47	0.66	25.19	6.09	3.60	7.98	0.39	0.23	0.52	4.27	0.72	2.70	2.70	2.52	1.66	29.27	7.74	8.66	16.94	2.84	10.71	11.38	3.01	3.37
Conc 4	3.38	0.35	28.57	2.75	2.02	2.64	0.09	0.07	0.09	4.36	0.78	2.79	0.64	0.74	0.29	29.91	8.48	8.94	15.26	2.74	9.76	10.25	2.91	3.07
Tails	950.79	2.92		1.08	0.89	2.99	10.22	8.46	28.38				70.09	91.52	91.06									
Head	979.4			1.49	0.94	3.18	14.58	9.25	31.17															

Run 2

Flotation Stage	Concentrate			Assay (%)			Wt (g)			Cumulative Wt. (g)			Recovery %			Cumulative Recovery			Cum Grade %			Upgrading		
	Wt (g)	Wt %	Cumm Wt (g)	TCu	ASCu	Fe	TCu	ASCu	Fe	TCu	ASCu	Fe	TCu	ASCu	Fe	TCu	ASCu	Fe	TCu	ASCu	Fe	TCu	ASCu	Fe
																0	0	0						
Conc 1	15.66	1.62	15.66	23.77	2.60	18.07	3.72	0.41	2.83	3.72	0.41	2.83	21.11	4.71	8.53	21.11	4.71	8.53	23.77	2.60	18.07	13.06	2.92	5.27
Conc 2	5.41	0.56	21.07	9.29	2.47	18.82	0.50	0.13	1.02	4.22	0.54	3.85	2.85	1.55	3.07	23.96	6.26	11.60	20.05	2.57	18.26	11.01	2.88	5.33
Conc 3	9.09	0.94	30.16	5.05	2.99	13.9	0.46	0.27	1.26	4.68	0.81	5.11	2.60	3.15	3.81	26.57	9.41	15.40	15.53	2.69	16.95	8.53	3.02	4.95
Conc 4	3.52	0.36	33.68	2.66	1.88	10.14	0.09	0.07	0.36	4.78	0.88	5.47	0.53	0.77	1.08	27.10	10.17	16.48	14.19	2.61	16.24	7.79	2.93	4.74
Tails	934.81	3.48		1.38	0.83	2.965	12.85	7.76	27.72				72.90	89.83	83.52									
Head	968.5			1.82	0.89	3.43	17.63	8.64	33.19															

Grade/Recovery data (500 g/t NaHS & 50 g/t SIBX on LQ ore)

Run 1

Flotation	Concentrate			Assay (%)			Wt (g)			Cumulative Wt. (g)			Recovery %			Cumulative Recovery			Cum Grade %			Upgrading		
Stage	Wt (g)	Wt %	Cumm Wt (g)	TCu	ASCu	Fe	TCu	ASCu	Fe	TCu	ASCu	Fe	TCu	ASCu	Fe	TCu	ASCu	Fe	TCu	ASCu	Fe	TCu	ASCu	Fe
																0	0	0						
Conc 1	14.41	1.48	14.41	21.70	2.08	19.61	3.13	0.30	2.83	3.13	0.30	2.83	21.39	3.68	8.58	21.39	3.68	8.58	21.70	2.08	19.61	14.47	2.49	5.80
Conc 2	5.00	0.51	19.41	10.43	2.09	17.22	0.52	0.10	0.86	3.65	0.40	3.69	3.57	1.28	2.61	24.96	4.97	11.19	18.80	2.08	18.99	12.54	2.49	5.62
Conc 3	10.93	1.12	30.34	8.37	3.79	13.86	0.91	0.41	1.51	4.56	0.82	5.20	6.26	5.09	4.60	31.21	10.05	15.79	15.04	2.70	17.14	10.03	3.23	5.08
Conc 4	4.18	0.43	34.52	3.17	1.88	11.06	0.13	0.08	0.46	4.70	0.90	5.66	0.91	0.97	1.40	32.12	11.02	17.19	13.60	2.60	16.41	9.07	3.11	4.86
Tails	940.64	3.54		1.06	0.77	2.90	9.92	7.24	27.28				67.88	88.98	82.81									
Head	975.2			1.50	0.83	3.38	14.62	8.14	32.94															

Run 2

Flotation	Concentrate			Assay (%)			Wt (g)			Cumulative Wt. (g)			Recovery %			Cumulative Recovery			Cum Grade %			Upgrading		
Stage	Wt (g)	Wt %	Cumm Wt (g)	TCu	ASCu	Fe	TCu	ASCu	Fe	TCu	ASCu	Fe	TCu	ASCu	Fe	TCu	ASCu	Fe	TCu	ASCu	Fe	TCu	ASCu	Fe
																0	0	0						
Conc 1	15.15	1.57	15.15	24.60	2.36	22.45	3.73	0.36	3.40	3.73	0.36	3.40	22.02	3.47	9.92	22.02	3.47	9.92	24.60	2.36	22.45	14.00	2.21	6.31
Conc 2	4.48	0.47	19.63	11.94	2.50	16.01	0.53	0.11	0.72	4.26	0.47	4.12	3.16	1.09	2.09	25.18	4.56	12.01	21.71	2.39	20.98	12.35	2.24	5.89
Conc 3	6.01	0.62	25.64	11.45	6.53	14.01	0.69	0.39	0.84	4.95	0.86	4.96	4.07	3.81	2.46	29.25	8.37	14.47	19.31	3.36	19.35	10.98	3.14	5.43
Conc 4	3.72	0.39	29.36	4.44	2.86	10.32	0.17	0.11	0.38	5.12	0.97	5.34	0.98	1.03	1.12	30.22	9.40	15.59	17.42	3.30	18.20	9.91	3.08	5.11
Tails	933.50	3.05		1.27	1.00	3.1	11.81	9.34	28.94				69.78	90.60	84.41									
Head	962.9			1.76	1.07	3.56	16.92	10.30	34.28															

LQ ore solids and water recovery- batch flotation with SIBX and CPS at -300 to -400 mV

Run	NaHS:SIBX	Sample	Sample Wt (g)	Bottle + H_2O (g)	Bottle Wt (g)	Wash H_2O added to dish (g)	D + C +H_2O (g)	Dish (g)	H_2O recovered	Total mass in conc	Solids recovery (%)	Water recovery (g)
1	20:1	Feed	10.41							38.5	3.918296	615.22
		C1	13.17	522.45	376.14	146.31	465.01	214.78	90.75			
		C2	2.93	541.39	431.62	109.77	376.92	218.69	45.53			
		C3	10.85	523.14	346.28	176.86	611.19	214.5	208.98			
		C4	6.31	515.18	345.47	169.71	550.6	214.9	159.68			
		C5	5.24	521.8	304.15	217.65	548.74	215.57	110.28			
		T1	11.21									
		T2	9.08									
		T3	923.78									
2	10:1	Feed	9.99							38.01	4.013304	438.35
		C1	13.85	376.14	236.15	139.99	421.7	214.78	53.08			
		C2	4.97	431.62	322.99	108.63	373.96	218.69	41.67			
		C3	7.52	346.28	199.92	146.36	486.89	214.5	118.51			
		C4	6.87	345.47	192.71	152.76	504.2	214.9	129.67			
		C5	4.8	304.15	97.16	206.99	522.78	215.57	95.42			
		T1	11									
		T2	12.22									
		T3	885.87									
3	7:1	Feed	11.03							47.32	4.742576	585.77
		C1	17.54	513.24	381.06	132.18	466.85	214.78	102.35			
		C2	3.06	534.48	428.9	105.58	365.51	218.69	38.18			
		C3	12.51	497.82	325.75	172.07	593.88	214.5	194.8			
		C4	7.3	508.06	339.79	168.27	528.36	214.9	137.89			
		C5	6.91	509.6	330.91	178.69	513.72	215.57	112.55			
		T1	12.18									
		T2	12.66									
		T3	925.61									

Run	NaHS:SIBX	Sample	Sample Wt (g)	Bottle + H_2O (g)	Bottle Wt (g)	Wash H_2O added to dish (g)	D + C +H_2O (g)	Dish (g)	H_2O recovered	Total mass in conc	Solids recovery (%)	Water recovery (g)
4	20:1	Feed	9.65							40.6	4.248195	532.99
		C1	16.62	561.46	419.54	141.92	451.34	214.78	78.02			
		C2	4.81	564.52	458.76	105.76	377.1	218.69	47.84			
		C3	9.47	552.19	400.65	151.54	550	214.5	174.49			
		C4	5.26	550.74	417.55	133.19	486.07	214.9	132.72			
		C5	4.44	559.07	396.53	162.54	482.47	215.57	99.92			
		T1	11									
		T2	11.82									
		T3	892.28									
5	10:1	Feed	9.89							35.34	3.647849	475.79
		C1	10.22	453.16	300.69	152.47	429.65	214.78	52.18			
		C2	4.13	452.64	330.54	122.1	384.08	218.69	39.16			
		C3	10.04	357.61	180.44	177.17	554.12	214.5	152.41			
		C4	5.67	385.52	232.4	153.12	497.18	214.9	123.49			
		C5	5.28	414.56	244.94	169.62	499.02	215.57	108.55			
		T1	0									
		T2	0									
		T3	933.45									
6	7:1	Feed	0							40.53	4.071403	619.2
		C1	17.56	300.69	192.76	107.93	485.3	214.78	145.03			
		C2	2.76	330.54	226.68	103.86	372.76	218.69	47.45			
		C3	9.85	573.05	420.91	152.14	559.35	214.5	182.86			
		C4	6.94	577.25	407.77	169.48	550.47	214.9	159.15			
		C5	3.42	563.77	412.77	151	454.7	215.57	84.71			
		T1	9.21									
		T2	9.09									
		T3	936.65									

Grade/Recovery data (-300 to -400 mV, 20:1)

Run 1

Flotation Stage	Concentrate			Assay (%)			Wt (g)			Cumulative Wt. (g)			Recovery %			Cumulative Recovery			Cum Grade %			Upgrading		
	Wt (g)	Wt %	Cumm Wt (g)	TCu	ASCu	Fe	TCu	ASCu	Fe	TCu	ASCu	Fe	TCu	ASCu	Fe	TCu	ASCu	Fe	TCu	ASCu	Fe	TCu	ASCu	Fe
Conc 1	13.17	1.34	13.17	17.86	1.57	24.17	2.35	0.21	3.18	2.35	0.21	3.18	19.05	2.21	9.52	19.05	2.21	9.52	17.86	1.57	24.17	14.21	1.65	7.10
Conc 2	2.93	0.30	16.1	9.50	2.09	19.09	0.28	0.06	0.56	2.63	0.27	3.74	2.25	0.65	1.67	21.31	2.87	11.19	16.34	1.66	23.25	13.00	1.75	6.83
Conc 3	10.85	1.10	26.95	3.97	1.84	11.89	0.43	0.20	1.29	3.06	0.47	5.03	3.49	2.13	3.86	24.79	5.00	15.05	11.36	1.74	18.67	9.04	1.82	5.49
Conc 4	6.31	0.64	33.26	3.37	2.30	7.34	0.21	0.15	0.46	3.27	0.61	5.50	1.72	1.55	1.39	26.52	6.55	16.44	9.84	1.84	16.52	7.83	1.94	4.86
Conc 5	5.24	0.53	38.5	2.89	1.94	6.12	0.15	0.10	0.32	3.43	0.71	5.82	1.23	1.09	0.96	27.74	7.64	17.40	8.90	1.86	15.11	7.08	1.95	4.44
Tails	944.07	3.92		0.95	0.92	2.925	8.92	8.64	27.61				72.26	92.36	82.60									
Head	982.57			1.26	0.95	3.40	12.35	9.35	33.43															

Run 2

Flotation Stage	Concentrate			Assay (%)			Wt (g)			Cumulative Wt. (g)			Recovery %			Cumulative Recovery			Cum Grade %			Upgrading		
	Wt (g)	Wt %	Cumm Wt (g)	TCu	ASCu	Fe	TCu	ASCu	Fe	TCu	ASCu	Fe	TCu	ASCu	Fe	TCu	ASCu	Fe	TCu	ASCu	Fe	TCu	ASCu	Fe
Conc 1	16.62	1.74	16.62	22.07	2.71	20.02	3.67	0.45	3.33	3.67	0.45	3.33	23.57	4.89	8.92	23.57	4.89	8.92	22.07	2.71	20.02	13.55	2.81	5.13
Conc 2	4.81	0.50	21.43	14.00	2.73	18.63	0.67	0.13	0.90	4.34	0.58	4.22	4.33	1.42	2.40	27.90	6.31	11.32	20.26	2.71	19.71	12.44	2.81	5.05
Conc 3	9.47	0.99	30.9	4.01	2.65	11.54	0.38	0.25	1.09	4.72	0.83	5.32	2.44	2.72	2.93	30.34	9.03	14.25	15.28	2.69	17.20	9.38	2.79	4.41
Conc 4	5.26	0.55	36.16	3.50	2.66	8.1	0.18	0.14	0.43	4.91	0.97	5.74	1.18	1.52	1.14	31.52	10.55	15.39	13.57	2.69	15.88	8.33	2.79	4.07
Conc 5	4.44	0.46	40.6	2.98	2.26	6.24	0.13	0.10	0.28	5.04	1.07	6.02	0.85	1.09	0.74	32.37	11.64	16.13	12.41	2.64	14.83	7.62	2.74	3.80
Tails	915.10	4.25		1.15	0.89	3.42	10.52	8.14	31.30				67.63	88.36	83.87									
Head	955.70			1.63	0.96	3.90	15.56	9.22	37.32															

Grade/Recovery data (-300 to -400 mV, 10:1)

Run 1

Flotation	Concentrate			Assay (%)			Wt (g)			Cumulative Wt. (g)			Recovery %			Cumulative Recovery			Cum Grade %			Upgrading		
Stage	Wt (g)	Wt %	Cumm Wt (g)	TCu	ASCu	Fe	TCu	ASCu	Fe	TCu	ASCu	Fe	TCu	ASCu	Fe	TCu	ASCu	Fe	TCu	ASCu	Fe	TCu	ASCu	Fe
Conc 1	13.85	1.46	13.85	20.94	2.39	19.93	2.90	0.33	2.76	2.90	0.33	2.76	21.00	3.98	8.86	21.00	3.98	8.86	20.94	2.39	19.93	14.36	2.72	6.06
Conc 2	4.97	0.52	18.82	15.33	2.39	18.31	0.76	0.12	0.91	3.66	0.45	3.67	5.52	1.43	2.92	26.52	5.41	11.78	19.46	2.39	19.50	13.35	2.72	5.93
Conc 3	7.52	0.79	26.34	5.68	2.01	14.64	0.43	0.15	1.10	4.09	0.60	4.77	3.09	1.82	3.53	29.61	7.22	15.31	15.52	2.28	18.11	10.65	2.60	5.50
Conc 4	6.87	0.73	33.21	4.47	2.64	10.23	0.31	0.18	0.70	4.40	0.78	5.47	2.22	2.18	2.25	31.84	9.40	17.56	13.24	2.36	16.48	9.08	2.68	5.01
Conc 5	4.80	0.51	38.01	2.93	1.72	5.95	0.14	0.08	0.29	4.54	0.86	5.76	1.02	0.99	0.92	32.85	10.40	18.48	11.94	2.28	15.15	8.19	2.59	4.60
Tails	909.09	4.01		1.02	0.82	2.795	9.27	7.45	25.41				67.15	89.60	81.52									
Head	947.1			1.46	0.88	3.29	13.81	8.32	31.17															

Run 2

Flotation	Concentrate			Assay (%)			Wt (g)			Cumulative Wt. (g)			Recovery %			Cumulative Recovery			Cum Grade %			Upgrading		
Stage	Wt (g)	Wt %	Cumm Wt (g)	TCu	ASCu	Fe	TCu	ASCu	Fe	TCu	ASCu	Fe	TCu	ASCu	Fe	TCu	ASCu	Fe	TCu	ASCu	Fe	TCu	ASCu	Fe
Conc 1	10.22	1.05	10.22	23.54	1.08	23.96	2.41	0.11	2.45	2.41	0.11	2.45	17.42	1.15	7.30	17.42	1.15	7.30	23.54	1.08	23.96	14.80	1.09	6.92
Conc 2	4.13	0.43	14.35	10.12	1.34	18.52	0.42	0.06	0.76	2.82	0.17	3.21	3.03	0.58	2.28	20.45	1.73	9.59	19.68	1.15	22.39	12.37	1.17	6.47
Conc 3	10.04	1.04	24.39	4.12	1.68	14.21	0.41	0.17	1.43	3.24	0.33	4.64	3.00	1.76	4.26	23.44	3.48	13.84	13.27	1.37	19.03	8.35	1.38	5.50
Conc 4	5.67	0.59	30.06	3.11	1.96	7.51	0.18	0.11	0.43	3.41	0.45	5.07	1.28	1.16	1.27	24.72	4.64	15.11	11.36	1.48	16.85	7.14	1.50	4.87
Conc 5	5.28	0.55	35.34	2.66	1.92	6.01	0.14	0.10	0.32	3.55	0.55	5.38	1.02	1.06	0.95	25.74	5.70	16.06	10.06	1.55	15.23	6.32	1.56	4.40
Tails	933.45	3.65		1.27	0.97	3.015	11.85	9.05	28.14				76.93	94.30	83.94									
Head	968.79			1.59	0.99	3.46	15.41	9.60	33.53															

Grade/Recovery data (-300 to -400 mV, 7:1)

Run 1

Flotation Stage	Concentrate			Assay (%)			Wt (g)			Cumulative Wt. (g)			Recovery %			Cumulative Recovery			Cum Grade %			Upgrading		
	Wt (g)	Wt %	Cumm Wt (g)	TCu	ASCu	Fe	TCu	ASCu	Fe	TCu	ASCu	Fe	TCu	ASCu	Fe	TCu	ASCu	Fe	TCu	ASCu	Fe	TCu	ASCu	Fe
Conc 1	17.54	1.76	17.54	20.85	1.85	19.62	3.66	0.32	3.44	3.66	0.32	3.44	21.09	3.56	10.01	21.09	3.56	10.01	20.85	1.85	19.62	12.00	2.02	5.70
Conc 2	3.06	0.31	20.6	16.09	1.69	16.16	0.49	0.05	0.49	4.15	0.38	3.94	2.84	0.57	1.44	23.93	4.12	11.45	20.14	1.83	19.11	11.59	2.00	5.55
Conc 3	12.51	1.25	33.11	5.70	1.65	8.36	0.71	0.21	1.05	4.86	0.58	4.98	4.11	2.26	3.04	28.04	6.39	14.49	14.69	1.76	15.05	8.45	1.92	4.37
Conc 4	7.30	0.73	40.41	3.16	2.16	6.31	0.23	0.16	0.46	5.09	0.74	5.44	1.33	1.73	1.34	29.37	8.11	15.83	12.60	1.83	13.47	7.25	2.00	3.91
Conc 5	6.91	0.69	47.32	2.56	1.67	5.34	0.18	0.12	0.37	5.27	0.86	5.81	1.02	1.26	1.07	30.39	9.38	16.91	11.14	1.81	12.28	6.41	1.98	3.56
Tails	950.45	4.74		1.27	0.87	3.005	12.07	8.27	28.56				69.61	90.62	83.09									
Head	997.77			1.74	0.91	3.44	17.34	9.12	34.37															

Run 2

Flotation Stage	Concentrate			Assay (%)			Wt (g)			Cumulative Wt. (g)			Recovery %			Cumulative Recovery			Cum Grade %			Upgrading		
	Wt (g)	Wt %	Cumm Wt (g)	TCu	ASCu	Fe	TCu	ASCu	Fe	TCu	ASCu	Fe	TCu	ASCu	Fe	TCu	ASCu	Fe	TCu	ASCu	Fe	TCu	ASCu	Fe
Conc 1	17.56	1.76	17.56	19.88	1.96	25.32	3.49	0.34	4.45	3.49	0.34	4.45	20.13	3.21	12.51	20.13	3.21	12.51	19.88	1.96	25.32	11.26	1.82	7.09
Conc 2	2.76	0.28	20.32	16.75	2.18	13.41	0.46	0.06	0.37	3.95	0.40	4.82	2.67	0.56	1.04	22.80	3.77	13.55	19.45	1.99	23.70	11.01	1.85	6.64
Conc 3	9.85	0.99	30.17	5.02	2.01	11.05	0.49	0.20	1.09	4.45	0.60	5.90	2.85	1.85	3.06	25.65	5.62	16.61	14.74	2.00	19.57	8.35	1.85	5.48
Conc 4	6.94	0.70	37.11	4.89	2.29	6.98	0.34	0.16	0.48	4.79	0.76	6.39	1.96	1.48	1.36	27.61	7.10	17.97	12.90	2.05	17.22	7.30	1.90	4.82
Conc 5	3.42	0.34	40.53	2.78	2.23	6.68	0.10	0.08	0.23	4.88	0.84	6.62	0.55	0.71	0.64	28.15	7.81	18.61	12.05	2.07	16.33	6.82	1.92	4.57
Tails	954.95	4.07		1.33	1.04	3.03	12.70	9.88	28.93				72.23	92.19	81.39									
Head	995.48			1.77	1.08	3.57	17.58	10.72	35.55															

LQ ore solids and water recovery- batch flotation with SIBX and CPS at -400 to -500 mV

Run	NaHS:SIBX	Sample	Sample Wt (g)	Bottle + H_2O (g)	Bottle Wt (g)	Wash H_2O added to dish (g)	D + C +H_2O (g)	Dish (g)	H_2O recovered	Total mass in conc	Solids recovery (%)	Water recovery (g)
1	20:1	Feed	11.21							48.22	4.904194	588.53
		C1	17.46	558.64	428.59	130.05	440.35	214.78	78.06			
		C2	4.84	571.18	466.21	104.97	366.57	218.69	38.07			
		C3	9.62	563.25	404.34	158.91	564.85	214.5	181.82			
		C4	10.44	524.53	355.9	168.63	575.65	214.9	181.68			
		C5	5.86	508.1	175.93	332.17	662.5	215.57	108.9			
		T1	0									
		T2	0									
		T3	935.02									
2	10:1	Feed	0							47.34	4.898745	507.81
		C1	15.1	428.59	306.64	121.95	423.49	214.78	71.66			
		C2	3.76	466.21	357.71	108.5	369.78	218.69	38.83			
		C3	9.62	404.34	243.35	160.99	508.97	214.5	123.86			
		C4	10.16	355.9	208.61	147.29	531.47	214.9	159.12			
		C5	8.7	536.9	325.57	211.33	549.94	215.57	114.34			
		T1	12.3									
		T2	13.17									
		T3	893.56									
3	7:1	Feed	10.97							42.16	4.389975	450.86
		C1	13.1	529.59	378.73	150.86	441.03	214.78	62.29			
		C2	4.1	565.06	443.76	121.3	376.66	218.69	32.57			
		C3	11.66	518.03	326.85	191.18	575.48	214.5	158.14			
		C4	5.52	547.7	361.2	186.5	499.53	214.9	92.61			
		C5	7.78	554.86	321.19	233.67	562.27	215.57	105.25			
		T1	20.04									
		T2	10.91									
		T3	887.26									

Run	NaHS:SIBX	Sample	Sample Wt (g)	Bottle + H_2O (g)	Bottle Wt (g)	Wash H_2O added to dish (g)	D + C +H_2O (g)	Dish (g)	H_2O recovered	Total mass in conc	Solids recovery (%)	Water recovery (g)
4	20:1	Feed	10.75							50.04	5.077316	614.12
		C1	18.92	562.65	425.26	137.39	444.98	214.78	73.89			
		C2	3.28	553.47	447.24	106.23	375.2	218.69	47			
		C3	11.91	530.5	358.79	171.71	602	214.5	203.88			
		C4	10.19	529.78	372.82	156.96	548.48	214.9	166.43			
		C5	5.74	502.11	316.8	185.31	529.54	215.57	122.92			
		T1	12.28									
		T2	9.5									
		T3	913.74									
5	10:1	Feed	7.38							39.77	4.089797	524.32
		C1	14.93	556.54	424.9	131.64	439.15	214.78	77.8			
		C2	2.69	564.44	447.97	116.47	382.68	218.69	44.83			
		C3	9.52	562.39	386.2	176.19	534.79	214.5	134.58			
		C4	7.13	552.24	373.14	179.1	563.75	214.9	162.62			
		C5	5.5	564.32	334.04	230.28	555.84	215.57	104.49			
		T1	9.1									
		T2	11.7									
		T3	911.85									
6	7:1	Feed	10.93							42.05	4.256461	491.68
		C1	15.01	424.9	278.72	146.18	435.65	214.78	59.68			
		C2	4.5	447.97	335.01	112.96	380.27	218.69	44.12			
		C3	9.37	386.2	172.07	214.13	575.55	214.5	137.55			
		C4	8	373.14	179.38	193.76	551.3	214.9	134.64			
		C5	5.17	334.04	124.25	209.79	546.22	215.57	115.69			
		T1	11.12									
		T2	11.34									
		T3	923.4									

Grade/Recovery data (-400 to -500 mV, 20:1)

Run 1

Flotation	Concentrate			Assay (%)			Wt (g)			Cumulative Wt. (g)			Recovery %			Cumulative Recovery			Cum Grade %			Upgrading		
Stage	Wt (g)	Wt %	Cumm Wt (g)	TCu	ASCu	Fe	TCu	ASCu	Fe	TCu	ASCu	Fe	TCu	ASCu	Fe	TCu	ASCu	Fe	TCu	ASCu	Fe	TCu	ASCu	Fe
Conc 1	17.46	1.78	17.46	19.81	3.54	20.5	3.46	0.62	3.58	3.46	0.62	3.58	22.55	5.88	10.31	22.55	5.88	10.31	19.81	3.54	20.50	12.70	3.31	5.81
Conc 2	4.84	0.49	22.3	8.75	3.40	16.29	0.42	0.16	0.79	3.88	0.78	4.37	2.76	1.57	2.27	25.31	7.44	12.58	17.41	3.51	19.59	11.16	3.28	5.55
Conc 3	9.62	0.98	31.92	5.99	4.32	8.42	0.58	0.42	0.81	4.46	1.20	5.18	3.76	3.95	2.33	29.07	11.40	14.92	13.97	3.75	16.22	8.95	3.51	4.60
Conc 4	10.44	1.06	42.36	3.63	2.83	6.24	0.38	0.30	0.65	4.84	1.49	5.83	2.47	2.81	1.88	31.54	14.21	16.80	11.42	3.53	13.76	7.32	3.30	3.90
Conc 5	5.86	0.60	48.22	2.88	2.35	6.12	0.17	0.14	0.36	5.01	1.63	6.19	1.10	1.31	1.03	32.64	15.52	17.83	10.38	3.38	12.83	6.66	3.16	3.64
Tails	935.02	4.90		1.11	0.95	3.05	10.33	8.88	28.52				67.36	84.48	82.17									
Head	983.24			1.56	1.07	3.53	15.34	10.51	34.71															

Run 2

Flotation	Concentrate			Assay (%)			Wt (g)			Cumulative Wt. (g)			Recovery %			Cumulative Recovery			Cum Grade %			Upgrading		
Stage	Wt (g)	Wt %	Cumm Wt (g)	TCu	ASCu	Fe	TCu	ASCu	Fe	TCu	ASCu	Fe	TCu	ASCu	Fe	TCu	ASCu	Fe	TCu	ASCu	Fe	TCu	ASCu	Fe
Conc 1	18.92	1.92	18.92	17.20	1.65	20.62	3.25	0.31	3.90	3.25	0.31	3.90	22.74	3.96	11.63	22.74	3.96	11.63	17.20	1.65	20.62	11.84	2.06	6.06
Conc 2	3.28	0.33	22.20	7.21	1.62	13.32	0.24	0.05	0.44	3.49	0.37	4.34	1.65	0.67	1.30	24.39	4.63	12.94	15.72	1.65	19.54	10.83	2.06	5.74
Conc 3	11.91	1.21	34.11	5.82	2.22	7.91	0.69	0.26	0.94	4.18	0.63	5.28	4.84	3.35	2.81	29.23	7.98	15.74	12.27	1.85	15.48	8.45	2.31	4.55
Conc 4	10.19	1.03	44.3	3.95	2.34	5.97	0.40	0.24	0.61	4.59	0.87	5.89	2.81	3.02	1.81	32.04	11.01	17.56	10.35	1.96	13.29	7.13	2.45	3.91
Conc 5	5.74	0.58	50.04	2.40	2.51	5.77	0.14	0.14	0.33	4.72	1.01	6.22	0.96	1.83	0.99	33.01	12.83	18.55	9.44	2.02	12.43	6.50	2.53	3.65
Tails	935.52	5.08		1.03	0.74	2.92	9.59	6.88	27.32				66.99	87.17	81.45									
Head	985.56			1.45	0.80	3.40	14.31	7.89	33.54															

Grade/Recovery data (-400 to -500 mV, 10:1)

Run 1

Flotation	Concentrate			Assay (%)			Wt (g)			Cumulative Wt. (g)			Recovery %			Cumulative Recovery			Cum Grade %			Upgrading		
Stage	Wt (g)	Wt %	Cumm Wt (g)	TCu	ASCu	Fe	TCu	ASCu	Fe	TCu	ASCu	Fe	TCu	ASCu	Fe	TCu	ASCu	Fe	TCu	ASCu	Fe	TCu	ASCu	Fe
Conc 1	15.10	1.56	15.1	20.22	2.00	22	3.05	0.30	3.32	3.05	0.30	3.32	19.34	3.41	9.17	19.34	3.41	9.17	20.22	2.00	22.00	12.38	2.19	5.87
Conc 2	3.76	0.39	18.86	12.58	1.88	17.76	0.47	0.07	0.67	3.53	0.37	3.99	3.00	0.80	1.84	22.34	4.21	11.02	18.70	1.98	21.15	11.45	2.16	5.65
Conc 3	9.62	1.00	28.48	5.37	2.92	13.87	0.52	0.28	1.33	4.04	0.65	5.32	3.27	3.18	3.68	25.61	7.39	14.70	14.20	2.29	18.69	8.69	2.51	4.99
Conc 4	10.16	1.05	38.64	2.66	1.99	7.53	0.27	0.20	0.77	4.31	0.86	6.09	1.71	2.29	2.11	27.32	9.68	16.82	11.16	2.21	15.76	6.83	2.42	4.21
Conc 5	8.70	0.90	47.34	1.95	1.51	4.48	0.17	0.13	0.39	4.48	0.99	6.48	1.07	1.49	1.08	28.40	11.16	17.89	9.47	2.09	13.69	5.80	2.28	3.65
Tails	919.03	4.90		1.23	0.86	3.235	11.30	7.86	29.73				71.60	88.84	82.11									
Head	966.37			1.63	0.92	3.75	15.79	8.84	36.21															

Run 2

Flotation	Concentrate			Assay (%)			Wt (g)			Cumulative Wt. (g)			Recovery %			Cumulative Recovery			Cum Grade %			Upgrading		
Stage	Wt (g)	Wt %	Cumm Wt (g)	TCu	ASCu	Fe	TCu	ASCu	Fe	TCu	ASCu	Fe	TCu	ASCu	Fe	TCu	ASCu	Fe	TCu	ASCu	Fe	TCu	ASCu	Fe
Conc 1	14.93	1.54	14.93	21.72	2.05	24.72	3.24	0.31	3.69	3.24	0.31	3.69	20.86	2.75	11.32	20.86	2.75	11.32	21.72	2.05	24.72	13.59	1.79	7.37
Conc 2	2.69	0.28	17.62	12.99	2.27	16.38	0.35	0.06	0.44	3.59	0.37	4.13	2.25	0.55	1.35	23.11	3.29	12.67	20.39	2.08	23.45	12.75	1.82	6.99
Conc 3	9.52	0.98	27.14	4.65	4.01	9.69	0.44	0.38	0.92	4.03	0.75	5.05	2.85	3.42	2.83	25.96	6.72	15.50	14.87	2.76	18.62	9.30	2.41	5.55
Conc 4	7.13	0.73	34.27	3.64	3.55	8.33	0.26	0.25	0.59	4.29	1.00	5.65	1.67	2.27	1.82	27.63	8.99	17.32	12.53	2.92	16.48	7.84	2.55	4.92
Conc 5	5.50	0.57	39.77	2.75	3.01	5.9	0.15	0.17	0.32	4.45	1.17	5.97	0.97	1.49	1.00	28.60	10.47	18.32	11.18	2.94	15.02	6.99	2.56	4.48
Tails	932.65	4.09		1.19	1.07	2.855	11.10	9.98	26.63				71.40	89.53	81.68									
Head	972.42			1.60	1.15	3.35	15.54	11.15	32.60															

Grade/Recovery data (-400 to -500 mV, 7:1)

Run 1																									
Flotation	Concentrate			Assay (%)			Wt (g)				Cumulative Wt. (g)			Recovery %			Cumulative Recovery			Cum Grade %			Upgrading		
Stage	Wt (g)	Wt %	Cumm Wt (g)	TCu	ASCu	Fe	TCu	ASCu	AlCu	Fe	TCu	ASCu	Fe	TCu	ASCu	Fe	TCu	ASCu	Fe	TCu	ASCu	Fe	TCu	ASCu	Fe
Conc 1	13.10	1.36	13.1	19.55	2.04	17.63	2.56	0.27	2.29	2.31	2.56	0.27	2.31	20.49	2.99	7.01	20.49	2.99	7.01	19.55	2.04	17.63	15.02	2.19	5.14
Conc 2	4.10	0.43	17.2	9.50	2.01	20.12	0.39	0.08	0.31	0.82	2.95	0.35	3.13	3.12	0.92	2.50	23.60	3.92	9.51	17.15	2.03	18.22	13.18	2.19	5.31
Conc 3	11.66	1.21	28.86	3.97	2.26	10.15	0.46	0.26	0.20	1.18	3.41	0.61	4.32	3.70	2.95	3.59	27.30	6.87	13.10	11.83	2.12	14.96	9.09	2.29	4.36
Conc 4	5.52	0.57	34.38	3.37	2.67	9.45	0.19	0.15	0.04	0.52	3.60	0.76	4.84	1.49	1.65	1.58	28.79	8.52	14.68	10.47	2.21	14.08	8.04	2.38	4.10
Conc 5	7.78	0.81	42.16	2.89	2.27	6.29	0.22	0.18	0.05	0.49	3.82	0.94	5.33	1.80	1.98	1.48	30.59	10.50	16.16	9.07	2.22	12.64	6.97	2.39	3.68
Tails	918.21	4.39		0.95	0.87	3.01	8.68	7.99	0.69	27.64				69.41	89.50	83.84									
Head	960.37			1.30	0.93	3.43	12.50	8.93	3.58	32.97															
Run 2																									
Flotation	Concentrate			Assay (%)			Wt (g)				Cumulative Wt. (g)			Recovery %			Cumulative Recovery			Cum Grade %			Upgrading		
Stage	Wt (g)	Wt %	Cumm Wt (g)	TCu	ASCu	Fe	TCu	ASCu	AlCu	Fe	TCu	ASCu	Fe	TCu	ASCu	Fe	TCu	ASCu	Fe	TCu	ASCu	Fe	TCu	ASCu	Fe
Conc 1	15.01	1.52	15.01	20.02	2.46	23.15	3.01	0.37	2.64	3.47	3.01	0.37	3.47	19.13	3.57	10.25	19.13	3.57	10.25	20.02	2.46	23.15	12.59	2.35	6.75
Conc 2	4.50	0.46	19.51	13.31	2.66	19.98	0.60	0.12	0.48	0.90	3.60	0.49	4.37	3.81	1.16	2.65	22.94	4.72	12.90	18.47	2.51	22.42	11.62	2.39	6.53
Conc 3	9.37	0.95	28.88	5.20	3.18	15.86	0.49	0.30	0.19	1.49	4.09	0.79	5.86	3.10	2.88	4.38	26.05	7.60	17.29	14.17	2.72	20.29	8.91	2.60	5.91
Conc 4	8.00	0.81	36.88	3.62	2.67	9.05	0.29	0.21	0.08	0.72	4.38	1.00	6.58	1.84	2.06	2.14	27.89	9.66	19.42	11.88	2.71	17.85	7.47	2.59	5.20
Conc 5	5.17	0.52	42.05	3.20	2.56	6.97	0.17	0.13	0.03	0.36	4.55	1.13	6.94	1.05	1.28	1.06	28.94	10.94	20.48	10.81	2.69	16.51	6.80	2.57	4.81
Tails	945.86	4.26		1.18	0.98	2.85	11.16	9.22	1.94	26.96				71.06	89.06	79.52									
Head	987.91			1.59	1.05	3.43	15.71	10.35	5.35	33.90															

Microflotation of pure malachite and chrysocolla

Sulphidised malachite											
		Run 1			Run 2			Recovery			
time		Filter paper	paper + sample	sample	Filter paper	paper + sample	sample	Run 1	Run 2	Average	Error
0										0	0
2	C1	1.0885	1.2126	0.1241	1.0651	1.2035	0.1384	7.45	7.04	7.25	0.20
4	C2	1.048	1.1556	0.1076	1.0317	1.112	0.0803	13.91	11.13	12.52	1.39
7	C3	1.0288	1.0485	0.0197	1.0511	1.1552	0.1041	15.09	16.42	15.76	0.66
10	C4	1.0317	1.1061	0.0744	1.032	1.0382	0.0062	19.56	16.74	18.15	1.41
	Tails	1.0381	2.3778	1.3397	1.0287	2.6651	1.6364				
	Feed			1.6655			1.9654				
Unsulphidised malachite											
		Run 1			Run 2			Recovery			
time		Filter paper	paper + sample	sample	Filter paper	paper + sample	sample	Run 1	Run 2	Average	Error
0										0	0
2	C1	4.4455	4.5511	0.1056	0.3	0.435	0.135	5.45	7.21	6.33	0.88
4	C2	4.2234	4.2619	0.0385	0.3	0.3399	0.0399	7.44	9.34	8.39	0.95
7	C3	2.437	2.4897	0.0527	0.3	0.3484	0.0484	10.17	11.92	11.04	0.88
10	C4	2.2742	2.2924	0.0182	0.3	0.3228	0.0228	11.11	13.14	12.12	1.02
	Tails	4.2465	5.9675	1.721	0.3	1.9271	1.6271				
	Feed			1.936			1.8732				
Sulphidised chrysocolla											
		Run 1			Run 2			Recovery			
time		Filter paper	paper + sample	sample	Filter paper	paper + sample	sample	Run 1	Run 2	Average	Error
0										0	0
2	C1	1.0617	1.07	0.0083	1.0228	1.0299	0.0071	0.45	0.38	0.41	0.03
4	C2	1.0481	1.0492	0.0011	1.0356	1.0371	0.0015	0.51	0.46	0.48	0.02
7	C3	1.0601	1.0605	0.0004	1.0814	1.0821	0.0007	0.53	0.50	0.51	0.02
10	C4	1.0308	1.031	0.0002	1.0364	1.0367	0.0003	0.54	0.51	0.53	0.01
	Tails	1.0306	2.8778	1.8472	1.0348	2.8951	1.8603				
	Feed			1.8572			1.8699				
Unsulphidised chrysocolla											
		Run 1			Run 2			Recovery			
time		Filter paper	paper + sample	sample	Filter paper	paper + sample	sample	Run 1	Run 2	Average	Error
0										0	0
2	C1	1.126	1.1335	0.0075	1.038	1.046	0.008	0.40	0.42	0.41	0.01
4	C2	1.048	1.0489	0.0009	1.077	1.078	0.001	0.44	0.48	0.46	0.02
7	C3	1.066	1.0663	0.0003	1.034	1.034	0	0.46	0.48	0.47	0.01
10	C4	1.007	1.0072	0.0002	1.128	1.128	0	0.47	0.48	0.47	0.00
	Tails	1.117	3.005	1.888	1.057	2.937	1.88				
	Feed			1.8969			1.889				

Appendix E: Pulp E_h/E_S profiles and reagent splits during sulphidisation

E_h and E_S profiles during slug sulphidisation of LQ ore with 250 g/t NaHS

	Es								Eh							
Time (s)	1	2	3	4	5	6	Ave	Std. Error	1	2	3	4	5	6	Ave	Std. Error
0	1.9	-72.5	-33.3	-82.5	-130.5	-71	-64.65	18.41	154	146	150	139	151	113	142.17	6.20
10	-577.5	-534.7	-555.5	-604.5	-558	-592	-570.37	10.54	-253	-236	-259	-248	-258	-254	-251.33	3.46
20	-447.9	-447.5	-441.1	-517.7	-490	-463.2	-467.90	12.28	-193	-195	-200	-211	-212	-197	-201.33	3.35
30	-408.6	-417.2	-400.9	-451.8	-460.7	-418.6	-426.30	9.89	-131	-133	-135	-152	-147	-130	-138.00	3.76
40	-375.8	-396.3	-360	-422.1	-426.2	-379.6	-393.33	10.84	-92	-90	-93	-106	-101	-85	-94.50	3.13
50	-357.6	-382.3	-341.7	-395.6	-402	-358.2	-372.90	9.79	-67	-62	-66	-75	-72	-55	-66.17	2.91
60	-347.5	-372.3	-329.6	-376.4	-383.7	-344.9	-359.07	8.73	-50	-42	-47	-54	-52	-35	-46.67	2.89
70	-337.6	-364	-318.6	-362.2	-370.5	-332	-347.48	8.55	-37	-27	-33	-39	-37	-21	-32.33	2.86
80	-329.4	-357.7	-310.6	-350.2	-360.8	-324.5	-338.87	8.28	-27	-17	-23	-27	-26	-11	-21.83	2.66
90	-323.3	-352.4	-302.1	-342.3	-350.8	-318	-331.48	8.25	-19	-9	-15	-18	-17	-4	-13.67	2.42
100	-315	-347.3	-294.3	-334.6	-342.1	-310.5	-323.97	8.42	-12	-3	-8	-11	-10	2	-7.00	2.22
110	-310.3	-340.5	-287.1	-328.6	-334	-304.5	-317.50	8.31	-6	2	-3	-5	-4	6	-1.67	1.91
120	-304.7	-336.7	-280.9	-322.2	-327.3	-298.9	-311.78	8.45	-1	7	2	0	1	9	3.00	1.65
130			-347.8	-355.5	-367	-369.3	-359.90	4.11			0	-1	-6	-4	-2.75	1.12
140			-327.4	-337.1	-351.3	-353.8	-342.40	5.07			-12	0	-9	-14	-8.75	2.52
150			-309.6	-324.1	-335.7	-336.1	-326.38	5.10			-10	3	-4	-12	-5.75	2.76
160			-295.3	-317.2	-323.3	-321.6	-314.35	5.29			-3	7	2	-7	-0.25	2.48
170			-285.9	-311.8	-315.4	-309	-305.53	5.45			3	10	7	0	5.00	1.80
180			-279.2	-307.1	-307.2	-299.1	-298.15	5.39			9	14	12	7	10.50	1.27
190				-302.9	-301.5	-291.7	-298.70	2.49				17	16	11	14.67	1.31
200				-298.2	-294.8	-287.3	-293.43	2.28				19	19	15	17.67	0.94
210				-295.2	-290.5	-281.7	-289.13	2.80				22	22	19	21.00	0.71

E_h and E_S profiles during slug sulphidisation of LQ ore with 500 g/t NaHS

Time (s)	Es					Eh				
	1	2	3	Ave	Std. Error	1	2	3	Ave	Std. Error
0	-54.5	-209.1	-119.3	-127.63	44.82	118	132	127	125.67	4.10
10	-621.9	-625.2	-603.2	-616.77	6.85	-349	-315	-301	-321.67	14.25
20	-600.2	-607.8	-560.6	-589.53	14.63	-299	-295	-283	-292.33	4.81
30	-581.6	-592.5	-490.5	-554.87	32.34	-292	-290	-254	-278.67	12.35
40	-560.7	-574	-463.1	-532.60	34.96	-287	-287	-221	-265.00	22.00
50	-533.1	-553.2	-448.4	-511.57	32.11	-279	-282	-199	-253.33	27.18
60	-504.5	-525.3	-435.7	-488.50	27.07	-268	-273	-184	-241.67	28.87
70	-488.2	-513.1	-420.9	-474.07	27.54	-253	-261	-173	-229.00	28.10
80	-475.1	-502.4	-411.2	-462.90	27.02	-239	-247	-165	-217.00	26.10
90	-464.8	-492.6	-402.8	-453.40	26.54	-230	-237	-159	-208.67	24.92
100	-453.6	-483.6	-393.7	-443.63	26.43	-223	-230	-154	-202.33	24.25
110	-443.3	-475.6	-385.4	-434.77	26.39	-218	-224	-149	-197.00	24.06
120	-435.5	-469.6	-379.1	-428.07	26.39	-213	-220	-145	-192.67	23.92
130	-440.8	-473.1	-428.3	-447.40	13.35	-210	-216	-170	-198.67	14.44
140	-433.4	-467.3	-417.7	-439.47	14.64	-210	-216	-171	-199.00	14.11
150	-427.9	-462.5	-409.6	-433.33	15.51	-207	-214	-167	-196.00	14.64
160	-422.8	-458.1	-403.1	-428.00	16.09	-204	-211	-162	-192.33	15.30
170	-417.1	-453.9	-395.4	-422.13	17.07	-201	-207	-156	-188.00	16.09
180	-413.5	-449.9	-387.5	-416.97	18.10	-198	-204	-152	-184.67	16.42
190	-409.1	-445.9	-380.7	-411.90	18.87	-195	-201	-147	-181.00	17.09
200	-405.6	-442	-376.5	-408.03	18.95	-192	-199	-142	-177.67	17.95
210	-401.5	-438.2	-363.7	-401.13	21.51	-189	-196	-138	-174.33	18.28

E_h profiles during CPS at E_S range of -300 to -400 mV

E_S	-300 to -400 mV									
	E_h									
Run	1	2	3	4	5	6	7	8	Ave.	std error
0 sec	215	111	89	103	104	98	104	145	121.125	14.62
10 sec	79	78	75	100	86	104	59	94	84.375	5.22
20 sec	46	80	-27	11	86	97	-74	95	39.25	22.50
30 sec	-10	-6	-39	-34	87	59	-92	25	-1.25	20.24
40 sec	-10	-61	-19	-74	46	-6	-75	6	-24.125	15.13
50 sec	-22	-42	2	-56	22	-62	-72	12	-27.25	12.75
60 sec	-41	-25	16	-21	-30	-62	-45	15	-24.125	9.77
70 sec	-18	-7	16	2	-42	-69	-39	-7	-20.5	9.79
80 sec	1	9	6	13	-19	-45	-24	-8	-8.375	7.01
90 sec	-23	-58	-36	13	-3	-43	-57	-45	-31.5	9.02
100 sec	-6	-55	-18	22	11	-47	-76	-37	-25.75	11.98
110 sec	17	-31	-1	11	23	-52	-81	-12	-15.75	12.90
120 sec	35	-10	8	20	29	-42	-56	9	-0.875	11.65

E_h profiles during CPS at E_S range of -400 to -500 mV

E_S	-400 to -500 mV									
	E_h									
Run	1	2	3	4	5	6	7	8	Ave.	std error
0 sec	129	46	43	111	45	94	120	108	87	12.89
10 sec	-13	-108	8	-37	15	33	61	48	1	19.18
20 sec	-82	-215	11	-124	-125	34	18	12	-59	32.17
30 sec	-131	-237	14	-161	-150	-51	-2	-132	-106	30.37
40 sec	-140	-232	-142	-164	-171	-129	-72	-138	-149	15.91
50 sec	-153	-213	-188	-167	-185	-132	-85	-123	-156	14.61
60 sec	-154	-221	-177	-167	-195	-127	-86	-120	-156	15.44
70 sec	-171	-221	-199	-173	-174	-127	-98	-123	-161	14.65
80 sec	-194	-212	-188	-177	-158	-123	-102	-116	-159	14.39
90 sec	-205	-211	-183	-177	-174	-133	-116	-135	-167	12.37
100 sec	-194	-213	-181	-180	-172	-156	-116	-128	-168	11.55
110 sec	-181	-217	-224	-200	-161	-145	-119	-125	-172	14.31
120 sec	-191	-220	-240	-193	-152	-139	-120	-131	-173	15.58

Reagent (NaHS and SIBX) splits during CPS

-300 to -400 mV

		NaHS (g/t)			
SIBX/NAHS (%)	**Test**	**CPS 1**	**CPS 2**	**CPS 3**	**Total**
5	1	100	50	50	200
	2	150	50	50	250
	3	100	50	25	175
10	1	125	125	25	275
	2	100	50	50	200
	3	100	50	25	175
15	1	200	100	50	350
	2	100	50	25	175
	3	100	50	25	175
Average		119	64	36	**219**
Split		54.4%	29.1%	16.5%	100.0%

Reagent split				
	NaHS	**SIBX**		
		NaHS:SIBX at CPS (g/t)		
		20:1	**10:1**	**5:1**
Sulphide float	0	30	30	30
CPS 1	119	5.97	11.94	17.92
CPS 2	64	3.19	6.39	9.58
CPS 3	36	1.81	3.61	5.42
Total	219	41	52	63

-400 to -500 mV

		NaHS (g/t)			
SIBX/NAHS (%)	**Test**	**CPS 1**	**CPS 2**	**CPS 3**	**Total**
5	1	400	200	100	700
	2	200	100	50	350
	3	200	100	75	375
10	1	350	100	50	500
	2	150	100	50	300
	3	200	50	50	300
15	1	250	100	75	425
	2	225	100	50	375
	3	150	50	50	250
Average		236	100	61	**397**
Split		59.4%	25.2%	15.4%	100.0%

Reagent split				
	NaHS	**SIBX**		
		SIBX:NaHS at CPS (g/t)		
		1:20	**1:10**	**1:7**
Sulphide float	0	30	30	30
CPS 1	236	11.81	23.61	35.42
CPS 2	100	5.00	10.00	15.00
CPS 3	61	3.06	6.11	9.17
Total	397	50	70	90