On the selective flotation of pentlandite from pyrrhotite in Sheba's Ridge ores

Mduduzi Justice Mbonambi
BSc (Chemical Engineering), UCT

University of Cape Town

Centre for Minerals Research

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It is true indeed that, “I can do all things through Christ who strengthens me” (Philippians 4:13)

Synopsis

Metal sulphides are the raw material for most of the world's supplies of non-ferrous metals and can be considered one of the most important group of ore minerals. In the context of South Africa, the Bushveld Igneous Complex (BIC) sulphides play a very important role in that they are largely associated with the valuable platinum group elements (PGEs) and platinum group minerals (PGMs). Typically, the base metal sulphide (BMS) content in the BIC is comprised of pentlandite, pyrrhotite and chalcopyrite.

Sheba's Ridge, one of the ore deposits found in the BIC, is exploited for its PGEs and PGMs. For typical PGE processing operations like the Merensky reef, the valuable PGEs and PGMs are associated with the base metal sulphides pentlandite, pyrrhotite and chalcopyrite. Unusually, in the case of the Sheba's Ridge ore, not all the sulphides contain associated PGEs and PGMs, and pyrrhotite has been observed to show little or no association. Therefore, this study was carried out to develop a methodology for the selective flotation of pentlandite from pyrrhotite using the polysaccharide depressants that are already in use in the flotation of BIC ores to control the naturally floating gangue minerals such as talc.

Microflotation tests were conducted on a high grade Nkomati massive sulphide ore sample as a probe ore, to investigate the flotation response of pentlandite and pyrrhotite to four different chain length xanthate collectors (SEX, SNPX, SIBX and PAX) and to different types of polysaccharide depressants (guar, CMC and starch). The effect of oxygen addition and pH modification were also studied. Laboratory batch flotation tests were then carried out using the Sheba's Ridge ore, to evaluate the optimum reagent suite and flotation conditions determined in the microflotation tests.

The results showed that using the different chain length xanthates or polysaccharide depressants on their own did not produce any pentlandite selectivity over pyrrhotite, but, when these reagents were used in conjunction with one another, some pentlandite selectivity was obtained. The best combination was found to be an intermediate chain length xanthate (SNPX) with guar depressant, where a balance between collector hydrophobicity and selectivity was obtained. Adjustment of the pH to 10, using lime

instead of NaOH, together with artificial pre-oxidation showed further improvement in the selective flotation of pentlandite. This was attributed to the faster oxidation rate of pyrrhotite at these conditions, which led to selective depression of pyrrhotite while pentlandite floatability was maintained.

The optimum reagent regime for pentlandite/pyrrhotite selectivity, as determined from the microflotation testwork using the Nkomati massive sulphide probe ore, was used as the basis of the batch flotation tests. The batch flotation tests represented a scale up of the microflotation tests, in terms of sample size (1 kg vs. 2 g) and were a closer representation of real operations, as silicate gangue minerals (e.g. talc) were present, as well as a froth phase.

The variables that were found to be key in the microflotation testwork in terms of improving metallurgical performance for pentlandite selectivity were pH, collector and depressant type and dosage. These were the same variables identified as key in the batch flotation tests. However, the differences between these parameters were observed to be more subtle in the case of batch flotation tests compared to what was observed in the microflotation testwork. Nevertheless, it can be concluded that results from microflotation test work can used as a good basis to study of the interactions of different minerals in ores in a batch flotation system.

Translating the results from microflotation testwork to batch flotation testwork was not so straight forward, and further work still needs to be done to prove that this can be done successfully. It was also found that there needs to be a synergistic interpretation of the interactions present in the reagent – mineral system. Finally, the study showed that the reagents used in flotation cannot be evaluated independently but rather a holistic approach needs to be employed.

Declaration

I declare that this thesis, submitted for the degree of Master of Science in Engineering at the University of Cape Town, is my own work and has not been submitted prior to this for any degree or publication, at this university or any other institution.

Signed by candidate

Mduduzi Justice Mbonambi

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NOMENCLATURE

BET	- Brunauer, Emmett and Teller
BIC	- Bushveld Igneous Complex
BMS	- Base metal sulphides
$CaCO_3$	- Calcium carbonate (lime)
CMC	- Carboxymethyl cellulose
$CuFeS_2$	- Chalcopyrite
DEP186	- A Depramin type CMC reagent
DEP267	- A Depramin type CMC reagent
DO	- Dissolved oxygen
Dowfroth 200	- A Dowfroth frother
$Fe_{1-x}S$	- Pyrrhotite (Po)
$(Fe,Ni)_9S_8$	- Pentlandite (Pn)
FTIR-ATR	- Attenuated total reflection Fourier transform infrared
g	- grams
g/t	- Dosage in grams per ton
H^+	- Hydrogen ion
I.S.	- Ionic strength
M	- Molar concentration
µm	- micro-meter
NaOH	- Sodium hydroxide
OH^-	- Hydroxyl ion
PAX	- Potassium amyl xanthate
PGEs	- Platinum group elements
PGMs	- Platinum group minerals
pH	- Hydrogen ion concentration
ppm	- Concentration parts per million
R	- Hydrocarbon chain
SEX	- Sodium ethyl xanthate
SIBX	- Sodium iso-butyl xanthate
SNPX	- Sodium normal propyl xanthate
Stypress 504	- A Stypress type guar gum depressant
Stypress 301U	- A Stypress type starch depressant

UG2	- Upper Group 2 Chromitite (associated with PGMs and PGEs)
X^-	- Xanthate
X_2	- Dixanthogen
XPS	- X-ray Photoelectron Spectroscopy

Chapter 1: Introduction

1.1 Background

The Bushveld Igneous Complex (BIC) in South Africa is a zone of mineralization that hosts the world's largest deposits of platinum group elements and minerals (PGEs and PGMs, respectively; Holwell and McDonald, 2006). The BIC, itself, is comprised of various economic reefs including the Platreef, the Merensky reef and the UG2 reef, which host large reserves of PGEs and PGMs. Amongst the many ore bodies found within the BIC, is the Sheba's Ridge ore deposit, which is the ore body of interest in this study.

It is well known from previous studies that the PGEs and PGMs in the BIC are generally found in association with sulphide minerals. Typically, the base metal sulphide (BMS) content comprises pentlandite, pyrrhotite and chalcopyrite. For the Merensky and Platreef ores this BMS content accounts for approximately 1 % of the ore, while UG2 ores have much lower sulphide content (e.g. Schouwstra *et al.*, 2000).

The Sheba's Ridge ore body is located near the town of Groblersdal in South Africa, within the vicinity of several Merensky Reef mining operations. The sulphide mineralization at Sheba's Ridge, however, has been found to be more analogous to the Platreef than the Merensky Reef (Ridge Mining, 2008). In the Merensky reef, the PGEs and PGMs are associated with all the sulphides including pyrrhotite and therefore the recovery of all the valuable minerals is treated as a bulk sulphide float. In the Platreef deposit, the PGEs have been found to be particularly closely associated with chalcopyrite ($CuFeS_2$) and pentlandite ($(Fe,Ni)_9S_8$,) while pyrrhotite ($Fe_{1-x}S$) has been observed to have little or no PGEs associated with it (Gain and Mostert, 1982; Holwell and McDonald, 2006). Ideally, processing the Platreef and/or Sheba's Ridge ore for PGEs and PGMs should involve recovering chalcopyrite and pentlandite whilst rejecting pyrrhotite. Pyrrhotite should be rejected because, in the context of PGM extraction it may be considered to be a gangue mineral which serves to dilute the concentrate grade and increase the SO_2 emissions in downstream smelter processes, which in turn would have a detrimental impact on the environment.

This study investigates flotation which is the physico-chemical separation process used to separate the valuable minerals (i.e. PGMs and the base metal sulphides with associated solid solution PGEs) from the unwanted silicate gangue in run of mine (ROM) platinum ores to obtain a bulk sulphide concentrate. In flotation, reagents such as collectors and depressants are used to manipulate mineral surfaces to enable the separation of wanted minerals from the unwanted. This study focuses on effecting the separation of pentlandite from pyrrhotite using reagent suites already employed in the separation of sulphides from silicate gangue. The focus of this study is: *To establish a reagent suite and flotation conditions which will improve the flotation recovery of pentlandite from pyrrhotite in the Sheba's Ridge ore.*

Xanthates are the most commonly used collectors in the flotation of sulphide minerals. They have the structure ROC(=S)SH and vary in hydrocarbon chain length from ethyl to hexyl (Klimpel, 1988). An increase in the hydrocarbon chain length is known to result in an increase in the collector's hydrophobicity (Taggart, 1945; Dimou, 1986; Ackerman *et al.*, 1987), which should result in a corresponding decrease in flotation selectivity. Consequently, this thesis will investigate whether:

A decrease in the hydrocarbon chain length of the xanthate collector improves the flotation selectivity for pentlandite in Sheba's Ridge ore.

In the flotation of ores from the BIC, polysaccharide depressants such as guar gum and carboxymethyl cellulose (CMC) are used to depress the flotation of silicate gangue, particularly minerals such as talc which are the most problematic in terms of their natural floatability (Steenberg and Harris, 1984). The recent study of Bicak ***et al.*** (2007) found that guar gum and CMC can also be used to depress pyrite flotation. Like pyrite (FeS_2), pyrrhotite ($Fe_{1-x}S$) is an iron sulphide; while different behaviour may be expected, this study will also aim to investigate whether:

Polysaccharide depressants, in particular guar and CMC, which are traditionally used to depress silicate gangue minerals, can also be used to depress the flotation of pyrrhotite in Sheba's Ridge ore.

Sulphides, in general, are known to be semiconductors. They oxidise readily in the presence of water and oxygen via a coupled electrochemical reaction with the reduction of oxygen (Rand and Woods, 1984) resulting in the formation of secondary hydrophilic ferric hydroxides layers on the hydrophobic mineral surfaces, which reduces the flotation response and recovery of these minerals. In the study of Legrand *et al.* (2005) it was shown that pyrrhotite oxidises more rapidly than pentlandite. Thus, the formation of hydrophilic iron hydroxides associated with the oxidation reaction would be expected to occur more rapidly on pyrrhotite than on pentlandite; this should result in the selective depression of pyrrhotite relative to pentlandite. For this reason this study will also investigate whether:

The use of oxidation by O_2 can selectively depress the flotation of pyrrhotite in Sheba's Ridge ore.

Another important parameter that has to be considered in flotation is pH, as it is frequently the driving force for interactions between minerals and reagents in solution. Chander (1988) showed that the flotation of selected minerals can be manipulated by addition of either H^+ or OH^- ions, and that pH may also affect the degree of oxidation and the nature of oxidation products. In the study of Bicak *et al.* (2007), the flotation recovery of pyrite was found to decrease as pulp pH was increased; this may also be true of pyrrhotite. Miller et al. (2005) also showed through contact angle studies that pyrrhotite flotation was influenced by solution pH. In their study they found that pyrrhotite should not float very well, as measured contact angle of zero was obtained at normal floating pH range of 9 – 9.5. Thus the effect of pH on pentlandite/pyrrhotite flotation is worth investigating. For most flotation operations, the choice of pH modifier usually depends on economic considerations and the role that the cations or anions of the pH modifier play on the flotation behaviour of the minerals under investigation. For this reason, this study will also examine whether:

A change of pH and type of pH modifier can be used to improve the flotation selectivity of pentlandite relative to pyrrhotite in Sheba's Ridge ore.

1.2 Objective and Scope of Research

The objective of this thesis was to obtain a reagent suite and flotation conditions that would allow for selective flotation recovery of pentlandite from the Sheba's Ridge ore whilst depressing pyrrhotite. This was done through an investigation of the flotation response of pentlandite and pyrrhotite to xanthate collectors of different chain lengths; to various types of polysaccharide depressants (guar gum, CMC, starch); to oxidation; and to changes of pH and type of pH modifier; leading to a synergistic interpretation of the interactions present in the reagent-mineral system.

1.3 Research Methodology

In order to achieve the objectives of this study, a fundamental understanding of the reagent-minerals system needs to be developed through appropriate testwork. This is typically done using microflotation tests, which allow the floatability of pure minerals to be examined. Due to the low grade of the Sheba's Ridge ore (~ 2% sulphides), a sample of the massive sulphide Nkomati ore (>80% sulphide content) was used for the microflotation tests. These microflotation tests investigated the effect of different chain-length xanthate collectors, different polysaccharide depressants, pH and pH modifiers, and oxidation, on pentlandite/pyrrhotite flotation. The results from this scoping stage were then applied to Sheba's Ridge ore using a bench scale batch flotation cell. The batch flotation tests investigated if the results obtained from microflotation tests could be translated to batch tests using the more complex Sheba's Ridge ore. The batch tests also investigated the effect of different polysaccharide depressants and pH on pentlandite/pyrrhotite flotation.

Chapter 2: Literature Review

2.1 Introduction

Flotation is known worldwide as the most economical way of concentrating low grade ores that contain valuable minerals and metals such as sulphides, platinum, gold, etc. Klimpel (1988) described flotation as a three cornered process as depicted in Figure 2.1 below, with the three corners that are core to the process being the chemical, equipment and operational components.

The aim of this thesis was to optimise the selective flotation response of pentlandite from Sheba's Ridge ore, a platinum-bearing ore located in the BIC in South Africa. Due to the low sulphide content of the Sheba's Ridge ore (~ 2%), the initial microflotation tests were carried out using Nkomati ore; a massive sulphide ore (> 80% sulphides) of similar mineralogical content to the Sheba's Ridge ore. Batch tests were then carried out on Sheba's Ridge ore. The focus of this study was mainly on the chemical component of the flotation process, and the factors that were investigated are highlighted on Figure 2.1:

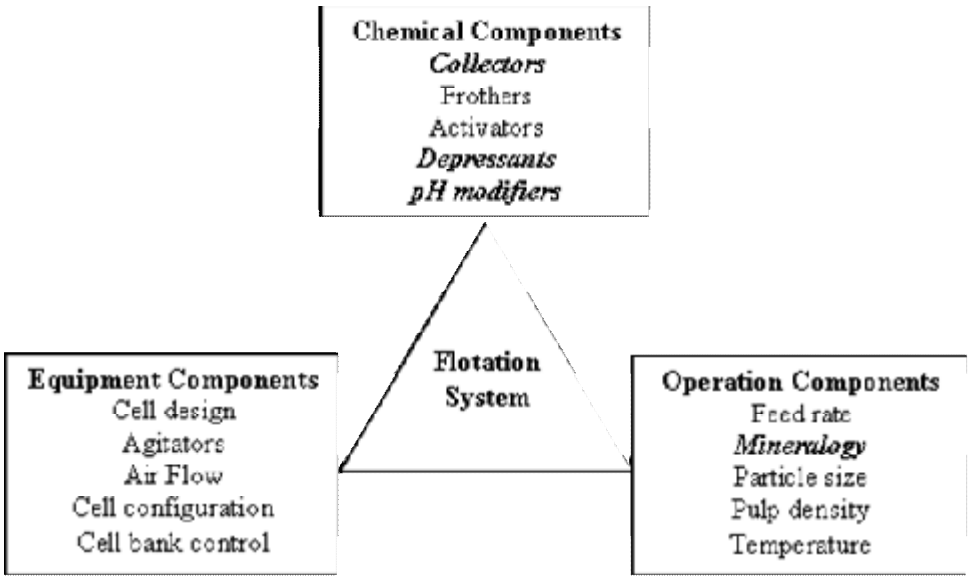


Figure 2.1: The interactive system of the flotation process (adapted from Klimpel, 1988)

As may be seen in Figure 2.1, the variables in the flotation process that were highlighted in this study are: mineralogy, collectors, depressants and pH modifiers. A critical review of the literature of these variables, as it pertains to the topic of this thesis, is presented in the section below. The chapter begins with a very brief overview of platinum ores found in South Africa.

2.2 Platinum Ores in South Africa

The BIC in South Africa is the world's largest known deposit of platinum group elements and minerals (PGEs and PGMs, respectively; Holwell and McDonald, 2006) and is comprised of various economically mineable reefs. The Merensky reef, found in the western and eastern limbs of the BIC, is characterised by a high association of the PGMs with base metal sulphides. The UG2, which is also found on the eastern and western limbs of the BIC, just below the Merensky reef, is characterised by little association of the PGMs with base metal sulphides, with most of the PGMs occurring as discrete minerals. The Platreef is similar to the Merensky reef but is much thicker, with reef thickness of up to 200 m compared to the Merensky reef which is only 0.3-3 m in thickness (Johnson Matthey, 2008).

The focus of this study is on the Sheba's Ridge platinum ore. Sheba's Ridge is located in the eastern limb of the BIC (Figure 2.2; Anglo Platinum, 2008) and is part of a Ridge Mining exploration project which commenced in April 2001. The first phase of the exploration project identified three distinct units of mineralisation, viz; a layer similar to the UG2 termed the "Platchro layer"; an upper mineralised pyroxenite (UMP) layer analogous to the Merensky Reef; and a wide sulphide zone similar to the Platreef (Ridge Mining, 2008). Since the sulphide zone (the main focus of this study) was found to be analogous to the Platreef, the Sheba's Ridge ore will be treated and considered as a Platreef ore for the purpose of this thesis.

The Nkomati ore which was used as the scoping ore in this thesis is a massive sulphide deposit, located in the Mpumalanga province, South Africa. It is similar in character and sulphide mineralisation to the BIC and is thought to be related to the BIC (Gauert *et al.* 1995).

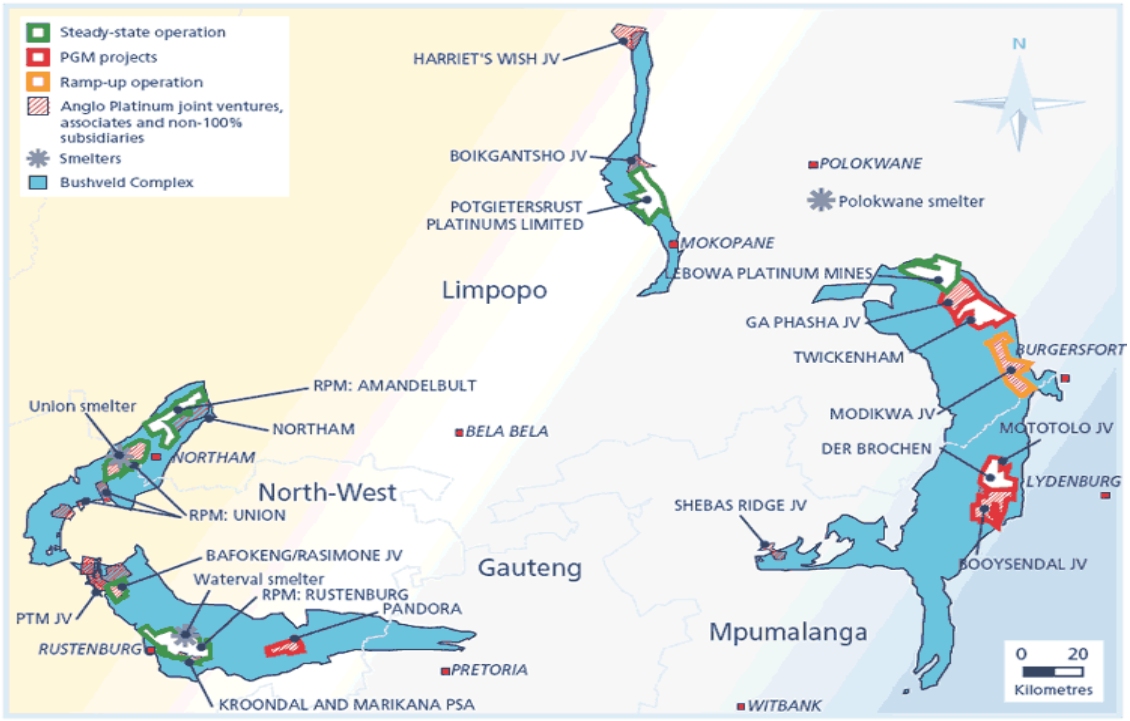


Figure 2.2: Sheba's Ridge ore location within BIC (from: Anglo Platinum, 2008)

In both the Sheba's Ridge and Nkomati ores, the main sulphides that are found are chalcopyrite, pentlandite and pyrrhotite. A brief description of these minerals, their mode of occurrence and some of their properties are presented in Section 2.3 below. A more detailed mineralogical analysis of the two ores is given in Chapter 3.

2.3 Sulphide Mineralogy

2.3.1 Pentlandite

Pentlandite is an iron nickel sulphide ($(Fe,Ni)_9S_8$) and is a well known primary source of nickel that is mined in countries like Canada, Australia and South Africa. Pentlandite is generally comprised of equal amounts of nickel and iron, although this may sometimes vary. It is brown to bronze in colour and displays a cubic crystalline habit. Pentlandite is a metallic conductor with Pauli paramagnetic magnetic character and can be considered an alloy of nickel, iron and sulphur, with a molar excess of metal (Warner *et al*, 1992).

Pentlandite is found ubiquitously in an intimate textural association with pyrrhotite. It may occur as grains (granular pentlandite) along fractures and boundaries of pyrrhotite grains or as smaller irregular, flame-like (flame pentlandite), exsolution structures within the pyrrhotite grains (Kelly and Vaughan, 1983). While granular pentlandite is likely to be liberated during comminution (the crushing and grinding stage before flotation), the flame variety is more likely to be found as locked particles in mineral processing operations. Pyrrhotite formed in association with pentlandite can carry nickel in solid solution.

Of particular economic interest, especially in samples from the BIC, is the amount of PGEs dissolved in solid solution in the pentlandite crystal structure. In a study of the noble metal enrichment in the sulphides of the Bushveld Complex, Ballhaus and Sylvester (2000) found that the concentrations of palladium dissolved in pentlandite were the most notable (summarised in Table 2.1). This provides additional motivation as to why the recovery of pentlandite during flotation is so critical.

Table 2.1: Average noble metal concentration (ppm) in pentlandite in a Merensky sample (from: Ballhaus and Sylvester, 2000)

Sample Setting	**Normal reef**	**Normal reef**	**Pothole reef**	**Pothole reef**	**Normal reef**
Number of Samples Taken	**6**	**10**	**3**	**5**	**10**
Osmium (Os)	7.2	7.6	7	7.8	4.6
Iridium (Ir)	7.2	8.4	6	8.2	4.2
Ruthenium (Ru)	26	27	30	27	10
Rhodium (Rh)	27	40	67	40	29
Platinum (Pt)	11	7	9.3	7.2	1.5
Palladium (Pd)	153	240	757	233	144

Besides being an important source of nickel, pentlandite is also a very important source of PGEs. This is the motivation for finding the optimum conditions to maximise the recovery of pentlandite at a reasonable grade (i.e. high selectivity over the unwanted material) in the Sheba's Ridge ore.

2.3.2 Pyrrhotite

Pyrrhotite is the second most abundant iron sulphide mineral after pyrite. Pyrrhotite exhibits a range of compositions represented by the general formula $Fe_{1-x}S$ (where x varies from 0 to 0.17); this formula can also be expressed as $Fe_{n-1}S_n$, with $n \geq 8$ (Belzile *et al.*, 2004). The changing amount of iron in the structure is thought to be charge balanced by the ratio of Fe(II) to Fe(III) (Ballhaus and Sylvester, 2000). Pyrrhotite is bronze to dark brown in colour.

In general, pyrrhotite is an unwanted mineral in processing operations (e.g. Sudbury in Canada; Lawson et al., 2005), unless it is associated with or hosts precious metals, such as in Merensky ores where it carries PGEs in solid solution (Ballhaus and Sylvester, 2000). In the Sheba's Ridge deposit, however, the association of PGMs with the pyrrhotite is more similar to Platreef, where it is almost negligible. Pyrrhotite occurs in magnetic and non-magnetic forms and this can sometimes be used as a method to separate out pyrrhotite (e.g. magnetic and non-magnetic pyrrhotite circuits at Sudbury) when it is present in sufficiently large amounts in an ore body. However, in the Sheba's Ridge ore, the sulphide content is minor and thus magnetic separation will not be considered.

Pyrrhotite rejection is important as it has been reported that almost all the sulphur that goes into smelting (from pyrrhotite) is emitted as SO_2 (Agar, 1991). Rejecting pyrrhotite will therefore have a direct impact on the reduction of SO_2 emission during smelting. A cost analysis study carried out by Bruce and Orr (1986), showed that the cheapest way to reduce SO_2 emission was by direct pyrrhotite rejection.

Due to the lack of association of PGMs and PGEs with pyrrhotite in the Sheba's Ridge ore, pyrrhotite is regarded as a gangue mineral with little or no value and thus needs to be rejected during flotation.

Table 2.2 below summarises some important properties of pentlandite and pyrrhotite at 25^oC.

Table 2.2: Summary of important properties of the sulphide minerals at 25^oC (adapted from Pearce *et al.*, 2006)

Sulphide Mineral	Chemical Formula	Crystal Structure	Conductivity Type	Electrical Resistivity (ohm-m)	Magnetic Property	Magnetic Susceptibility ($1x10^{-6}mol^{-1}cgs$)
Pentlandite	$(Fe,Ni)_9S_8$	Cubic	Metal	N/A	Pauli Paramagnetic	N/A
Pyrrhotite	Fe_7S_8	Monoclinic	Metal: p-type	$1x10^{-6}$ – $1x10^{-1}$	Ferromagnetic	$1.25x10^5$
Pyrrhotite	$Fe_{1-x}S$	Hexagonal	Metal	N/A	Antiferromagnetic	N/A
Troilite	FeS	Hexagonal	Metal: p-type	$1x10^{-6}$ – $1x10^{-1}$	Antiferromagnetic	5187

2.3.3 Flotation characteristics of the sulphides

The sulphides of interest in this thesis have previously been recognised to have different flotation rates with the flotation order as follows: pentlandite > pyrrhotite, based on the results of both microflotation tests (Newell *et. al.* 2006) and batch flotation tests (Wiese *et al.* 2006a). It should be noted that the order of floatability under standard conditions is found to be pentlandite > pyrrhotite, whereas in an oxygen deficient environment, the order becomes pyrrhotite > pentlandite. This change in behavior has been correlated to the galvanic interaction of the minerals during oxygen reduction (for further discussion see Section 2.5.7).

2.4 Principles of flotation

Flotation is a physico-chemical separation process that has been in use since the early 20th century to separate valuable minerals from unwanted gangue minerals by exploiting differences in the surface properties between the valuable and gangue minerals. Flotation is a complex process: Bradshaw *et al.* (2006) have argued that even though it has been known to exist for many years, the complexity and interactions of the physical and chemical parameters involved has made it difficult to characterise and control the process adequately.

Reagents are usually used to manipulate the surfaces of minerals to render them either hydrophobic or hydrophilic and hence facilitate their separation. Collectors are used to render the surfaces of the valuable minerals hydrophobic. The hydrophobic mineral surfaces can then attach to air bubbles in an agitated and aerated pulp through which

the mineral-bubble aggregates rise till they reach the top of the froth where they are recovered to the concentrate. In addition to minerals being recovered by this mechanism of true flotation, fine particles are recovered unselectively by entrainment (mechanical carry-over in the concentrate by water). Depressants are used to reduce the natural hydrophobicity of gangue minerals, i.e. to make the surfaces more hydrophilic. Additional reagents such as frothers, activators and modifiers may also be added to facilitate and optimise the flotation process. A simple schematic of the flotation process is presented in Figure 2.3.

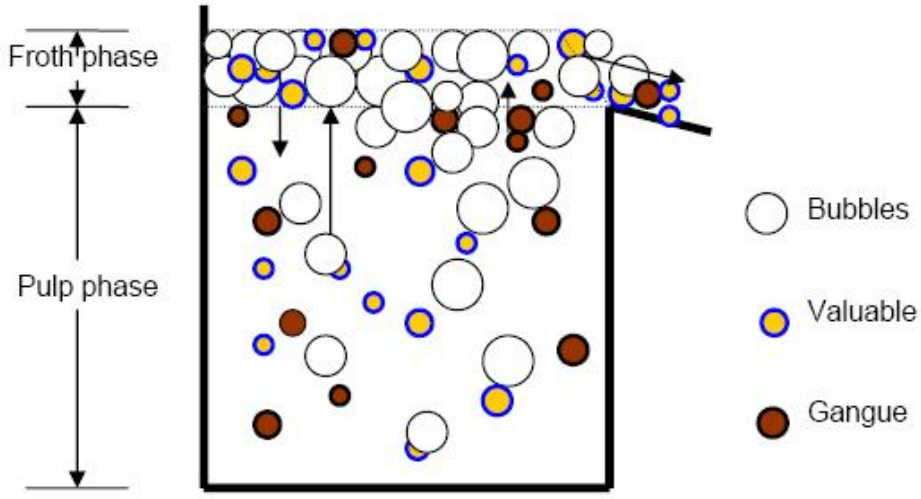


Figure 2.3: Flotation Process (from: Dhliwayo, 2005)

The action of the different types of reagents is discussed further in section 2.5 below.

2.5 Surface and Flotation Studies

Trahar (1984) argued that there are three possible ways in which sulphide minerals float, viz. through self-induced, sulphur-induced and collector-induced flotation, with each type of floatability considered to be dependent on the oxidation-reduction state of the pulp. The section below reviews two of the above mentioned methods of flotation: self-induced and collector-induced flotation, and the action of various reagents in modifying the surfaces of the minerals of interest in this thesis, pentlandite and pyrrhotite.

2.5.1 Self-induced /Collectorless flotation

The self-induced flotation of sulphide minerals has been known to exist for a number of years, with the natural floatability of sulphide minerals argued to be directly linked to their ease of oxidation as well as to the stability of the hydrophobic surfaces that are produced (Rao, 2004). Heiskanen *et al.* (1991) reported that pentlandite and pyrrhotite can sometimes exhibit self-induced (collectorless) flotation as a result of mild oxidation of the minerals themselves. Sulphide minerals are semiconductors and oxidise readily in the presence of water and oxygen via a coupled electrochemical reaction with the reduction of oxygen (Bradshaw *et al.*, 2006; Rosso and Vaughan, 2006). Comparisons between the relative oxidation rates of the sulphides have shown that pyrrhotite tends to oxidise more quickly than pentlandite (Legrand et al., 2005).

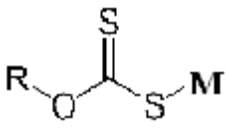
Ralston (1991) confirmed that the self-induced flotation of sulphide minerals is very sensitive to the pulp chemistry conditions, as it only takes place under specific conditions. Using batch flotation techniques, Heinskanen *et al.* (1991) examined the collectorless flotation of noritic and serpentinic nickel ores at pH values from 3-12. They proposed that the mechanism which induces natural hydrophobicity (i.e. floatability) of the sulphides is surface oxidation. They also found that pyrrhotite and pentlandite floated well at pH 3-5, but much lower recoveries were achieved at higher pH values. The results also showed that process iron in the water had a marked effect on the collectorless flotation. They found that sulphides from samples milled in a ceramic device floated better than those milled in a mild steel device. This was attributed to possible hydroxide and/or sulphate layers that formed during milling (as oxidation reactions are more prevalent in a steel mill than in a ceramic mill) thus hindering flotation. It was also noted that the oxidation of pyrrhotite progressed more quickly than that of pentlandite (Heinskanen *et al.*, 1991).

This study is aimed at investigating the effect of different reagents on the flotation of pentlandite and pyrrhotite. In order to achieve this, a base test will be carried out with no reagent addition as a bench mark for further testwork incorporating reagent addition.

2.5.2 Collectors

Collectors are heterogeneous compounds that contain an active inorganic group coupled with a hydrocarbon chain. Usually the inorganic group is the portion of the collector molecule that adsorbs onto or reacts with the mineral surface. The non-polar hydrocarbon chain is the portion that provides the hydrophobicity (i.e. the driving force leading to bubble attachment) to the mineral surface after collector adsorption. There is a wide range of commercially available collectors and they are classified on the basis of composition depending on whether they exist as cations, anions or molecular species in solution (Fuerstenau, 1982a).

This thesis will focus on the xanthate collectors, which are anionic sulphydryl type collectors, and which are typically used to float Merensky type ores. Table 2.3 lists a number of typical anionic sulphydryl type collectors and their applications. The most commonly used sulphydryl anionic collector is xanthate, though a number of other sulphur bearing surfactants have also found application. Xanthates were first invented and patented in 1925 (de Donato *et al.*, 1979), and are generally accepted as the "**workhorse of industrial-scale sulphide mineral flotation**" (Klimpel, 1988). They have been used extensively in gold, platinum and base metal sulphide flotation. Like the fatty acids, xanthates are also weak acids (Fuerstenau, 1982a). Alkali metal and alkaline xanthates are soluble in water, while heavy metal xanthate salts possess only limited solubility in aqueous solution. Xanthates have the following general structure:



where R = hydrocarbon chain

and M = cation

Table 2.3: Properties and typical applications of common anionic collectors (adapted from Lovell, 1982)

Collector	Type	Example	Formula	Properties	Stable pH Range	Application
Anionic Collectors	Sulphydryl (Thiol collectors)	Xanthate	$ROCS_2M$*	Soluble in water	8-13	Used for selective flotation of Sulphides
		Dithiophosphate	$ROHOPS_2M$*	Used undiluted	4-12	Used as collectors for Au, Ag and Cu
		Dithiocarbamate	$R_1R_2NCS_2M$*	Usually added as a liquid emulsion	5-12	Selective collectors for Cu or Cu-activated Zn over FeS_2
		Thionocarbamate	$R_1HNCSOR_2$		4-9	
		Mercaptobenzo-thiazole	C_6H_4SNCSM*	Has frothing properties	4-9	Used for treating gold (Au) and silver (Ag) ores, and applicable to flotation of FeS_2

M* = Metal
R = hydrocarbon group

The solubility of metal xanthates decreases with increase in chain length, This implies that the xanthate molecule with the longest hydrocarbon chain will have the lowest solubility in water and therefore the highest hydrophobicity. This has been confirmed by a number of authors who noted that increasing the collector chain length corresponded to increase in hydrophobicity and recovery (Taggart, 1945; Dimou, 1986; Ackerman *et al.*, 1987).

Bradshaw *et al.* (2004) compared the effect of xanthate chain length (SEX versus PAX) on the flotation of sulphides using microflotation and batch flotation tests. For the microflotation tests, they found that sulphide floatability was greater when using PAX instead of SEX. They argued that this increased hydrophobicity was due to the increased xanthate chain length. In the batch flotation testwork though, the situation was reversed: this was attributed to the destabilising effect of the froth phase caused by the increased hydrophobicity of PAX.

Xanthate collectors (ethyl xanthate, propyl xanthate, iso-butyl xanthate and amyl xanthate) will be used to investigate the effect of variation in hydrocarbon chain length (collector strength) on selectivity of pentlandite/pyrrhotite flotation of Sheba's Ridge ore.

2.5.3 Collector Induced Flotation

Much research has been carried out over the years into the mechanism of xanthate adsorption onto mineral surfaces. Fuerstenau (1982a) proposed that cupric and ferric ions oxidise xanthate in solution with oxygen to form dixanthogen as follows:

$$2X^- + ½\ O_2 + H_2O \rightarrow X_2 + 2OH^- \quad (2)$$

$$2X^- + Cu^{2+} \rightarrow CuX(s) + ½\ (X_2) \quad (3)$$

$$2X^- + 2Fe^{3+} \rightarrow 2Fe^{2+} + (X_2) \quad (4)$$

Although xanthate oxidation by dissolved oxygen is thermodynamically favourable, it is kinetically slow. As a result, the formation of dixanthogen by this mechanism (equation 2) can be assumed not to occur to any appreciable extent in flotation systems. In the presence of Cu^{2+} and Fe^{3+}, however, kinetically the reaction is said to be relatively fast (Fuerstenau, 1982a).

Xanthate is observed to oxidise completely at pH 2 with Fe^{3+} but does not oxidise at pH 6 and above when a reaction time of 10 min is used (Fuerstenau, 1982a). In the presence of Cu^{2+}, there is complete oxidation of xanthate up to pH 10, while no oxidation is possible at about pH 11 and above. Dixanthogen is not stable at pH 10.5 and above, implying that at these pH values only xanthate is stable (Fuerstenau, 1982a).

Hodgson and Agar (1989) carried out xanthate adsorption research at pH 9 on pyrrhotite and pentlandite surfaces. On the pyrrhotite surface the formation of an $Fe(OH)_2X$ product was observed upon interaction with xanthate. No electron transfer was considered to have taken place during this adsorption process of xanthate on the surface. The collector was proposed to have adsorbed through coulombic attraction

with cationic iron (III) sites generated through oxidation of the mineral surface as follows:

$$FeS + H_2O \rightarrow Fe(OH)[S]^{+} + H^{+} + 2e^{-} \quad \ldots\ldots\ldots\ldots\ldots\ldots(5)$$

$$Fe(OH)[S]^{+} + X^{-} \rightarrow Fe(OH)[S]X \quad \ldots\ldots\ldots\ldots\ldots\ldots\ldots\ldots(6)$$

Oxidation of xanthate to dixanthogen was proposed to occur through the reduction of oxygen at the pyrrhotite surface:

$$2X^{-} \rightarrow X_2 + 2e^{-} \quad \ldots\ldots\ldots\ldots\ldots\ldots(7)$$

$$O_2 + 2H_2O + 4e^{-} \rightarrow 4OH^{-} \quad \ldots\ldots\ldots\ldots(8)$$

Dixanthogen formation was considered to take place adjacent to the oxidised pyrrhotite surface; dixanthogen was then physisorbed via the alkyl groups of the Fe(OH)[S]X complex. Dixanthogen conferred the hydrophobicity on the pyrrhotite surface and oxygen was required to promote the bubble contact.

In the case of pentlandite it was proposed that the collector chemisorbs directly on to the nickel sites and that the addition of xanthate reduces the degree of oxidation of the mineral surfaces. Dixanthogen then forms from chemisorbed xanthate on the pentlandite surface according to the following reactions:

$$Ni + X^{-} \rightarrow NiX + e^{-} \quad \ldots\ldots\ldots\ldots\ldots\ldots(9)$$

$$NiX + X^{-} \rightarrow Ni(X)_2^{-} \rightarrow NiX_2 + e^{-} \quad \ldots\ldots\ldots(10)$$

Formation of dixanthogen was proposed to occur concurrently with the adsorption of xanthate on the pentlandite surface during oxidation, enhancing hydrophobicity.

Bozkurt *et al.* (1997) investigated the effect of Ni ions and the interaction of pentlandite and pyrrhotite on the adsorption of isobutyl xanthate, using FTIR-ATR spectroscopy and open circuit potential measurements. Dixanthogen was the major surface product observed under all conditions on both minerals. In single mineral studies, dixanthogen concentration was greater on pentlandite than on pyrrhotite, and Ni ions always enhanced its formation. For mineral mixtures, dixanthogen

concentration increased on pentlandite and decreased on pyrrhotite. Subsequently, in a similar study, Bozkurt *et al*. (1998a, b) reported that dixanthogen formation on pentlandite was promoted, while on pyrrhotite it was suppressed. A mixed potential model was used to explain this effect of mineral interaction on dixanthogen formation. At the potential of the mixed mineral system, the anodic reaction (xanthate oxidation to dixanthogen) occurred preferentially on pentlandite while the cathodic reaction (reduction of oxygen to hydroxyl ion) occurred on pyrrhotite

From the literature above, it can be seen that the success of xanthate collectors is dependant on the oxidation state of the pulp. This means that if these collectors are to be used, a certain degree of oxidation must take place in the pulp to facilitate the action of these collectors. To what extent the oxidation process should proceed is unclear, and this will be one of the variables that will be investigated in this thesis.

2.5.4 Regulators or Modifiers

In most cases, collectors by themselves are not adequate to optimise the flotation process. For this reason, other reagents called regulators are used in conjunction with collectors to ensure that only the valuable minerals are recovered. These regulators can be divided into the following four categories:

- *Activators*, which enhance collector attachment to the valuable mineral;
- *Depressants*, which enhance the hydrophilic character of the gangue minerals. This class includes dispersants, as they also enhance the hydrophilic nature of minerals;
- *pH modifiers*, which modify the pH of the pulp so that the optimum conditions for collection, activation or depression are achieved, and;
- *Frothers*, which are used to create a stable froth phase for the successful recovery of the valuable minerals transported from the pulp zone, for further upgrading.

It must be noted that there are limitations with regard to classifying these compounds, i.e. a reagent which is used as a depressant for one mineral can also be used as an

activator for another mineral or as a pH modifier. For example, sodium sulphide is a depressant for sulphides, an activator for oxidised minerals and a pH modifier. The same reagent can also act in more than one capacity in the same pulp depending on the level of addition (Lovell, 1982). Wiese *et al.* (2006b) have highlighted the fact that reagent-mineral interactions need a synergistic and careful interpretation.

2.5.4.1 Activators

These inorganic compounds enhance collector attachment and include the following examples:

- Copper sulphate which produces the Cu^{2+} ion for the activation of pyrrhotite (Wiese et al., 2006a)
- Sodium sulphide and hydrosulphide (Na_2S and NaSH) for the activation by sulphidisation of tarnished, oxidised minerals and carbonates (Newell et al. 2007)

Both these activators have been used extensively in the flotation of sulphide minerals, but will not be discussed further since they are beyond the scope of this study.

2.5.4.2 Depressants

According to Laskowski *et al.* (1991), the function of depressants is to inhibit the flotation of a given mineral. They proposed that this could be carried out through two mechanisms, namely:

- by preventing the collector from adsorbing onto a given mineral; or
- by imparting strong hydrophilicity onto the mineral

Evidently, both the proposed mechanisms require reagent adsorption onto the mineral surface for depression to be induced. The two main types of depressant are the organic and the inorganic depressants.

2.5.4.2.a Inorganic Depressants

The action mechanism of this group of chemicals is generally well understood and is represented by the following examples:

- *Cyanide* (as sodium or calcium cyanide) which depresses sulphides of Zn, Cu, Fe, Ag, Cd and Ni because of the greater stability of the cyano-complexes of these metals over those of the corresponding thiolates. This depressant has a detrimental environmental impact as it is highly toxic, therefore it should only be used with the outmost caution (Lovell, 1982).
- *Chromate or dichromate* which depresses PbS by the formation of the corresponding lead salt on the mineral surface.
- *Sodium silicate* (which has a complex function of dispersant, depressant and froth modifier) disperses and depresses silicate gangue slimes.

2.5.4.2.b Organic Depressants

These are usually naturally occurring products or modified natural products of high molecular weight (above 10 000 g/mol) and contain a number of strongly hydrated polar groups which are the basis of their depressing action. There are also a number of lower molecular weight organic depressants that contain anionic and cationic groups. Although the mechanisms by which these depressants operate is not well understood, it is believed that they interact with the mineral surface through electrostatic or dipole interactions of the multiple polar groups and through hydrogen bonding of the large number of hydroxyl groups present on the polymer (Steenberg and Harris, 1984). The organic depressants are sub-divided into three groups; viz. the polyglycol ethers, the polysaccharides and the polyphenols (Lovell, 1982). Although polyacrylamide depressants have been used successfully in some base metal sulphide operations (e.g. Sudbury), they will not be studied here since the depressants used for this study need to be able to sufficiently depress both the problematic gangue minerals (talc) and the unwanted sulphides (pyrrhotite). For this reason, only the polysaccharide depressants will be examined in this study.

Polysaccharide depressants are natural products that have been used successfully with little modification to their structures. Among other uses, they have found a very important function in the flotation of nickel and platinum bearing ores as depressants for minerals such as talc (Wang *et al*., 2005).

- *Starch*

Starch and pearl starches are materials that are used to flocculate haematite, disperse clay slimes and depress talcaceous minerals (Iwasaki, 1965; Frommer and Colombo, 1966; Fuerstenau, 1982b). Starch was recommended as a depressant as early as the 1930's. Industrial application has been extensive since then, but systematic work dealing with the mechanisms governing its action as a depressant has been limited. In the starch structure, the linear chain and branched chain components have molecular weights reaching millions (Fuerstenau, 1982b).

- *Cellulose*

Cellulose is usually used as a depressant in a modified form. Carboxymethylcellulose (CMC) is the most common derivative of cellulose and is used mainly as a talc depressant. This cellulose is an anionic polysaccharide molecule, with molecular weights ranging from $1x10^3$ to $1x10^6$ g/mol. Figure 2.4 shows the molecular structure of a typical CMC molecule.

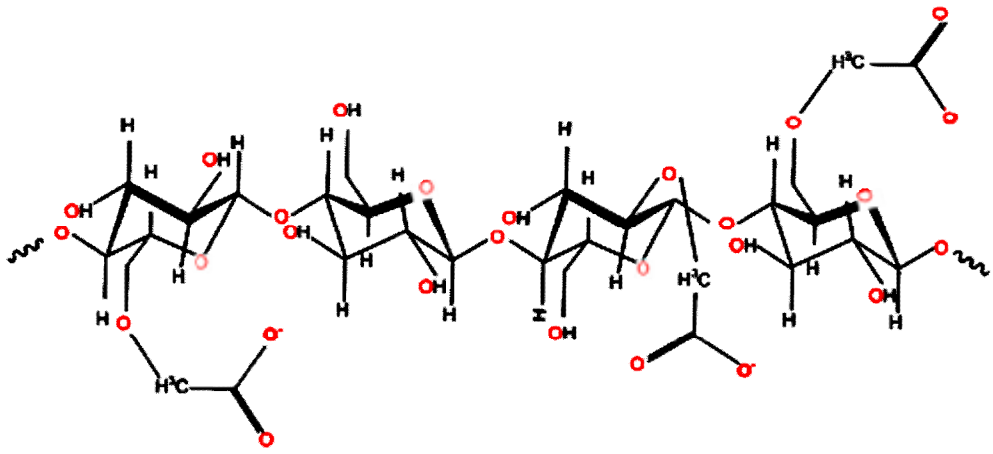


Figure 2.4: CMC Molecule Structure

In previous studies that have examined the factors influencing the action of CMCs, it was shown that ionic strength, pH, molecular weight and concentration (Morris *et al.*, 2002; Khraisheh *et al.*, 2005; Parolis *et al*, 2008) strongly influence the adsorption of CMC onto mineral surfaces. Khraisheh *et al.* (2005) and Parolis *et al*. (2008) found that an increase in the solution ionic strength (by the presence of metal cations) resulted in an increase in CMC adsorption onto the mineral surface studied (talc).

The hydrophobicity of sulphides can sometimes be impaired by the formation of slime coatings from gangue minerals that are present in the ores. CMC's have been found to be good agents of slime cleaning for the sulphides thereby improving the hydrophobicity (Wiese *et al.*, 2005; Bradshaw et al., 2005)

- *Natural Gums*

These are natural gums such as gum Arabic, gum tragacanth and guar gum. In these natural gums, the most commonly used type is guar gum, which also has strong flocculant properties and is a non-ionic long chained molecule. Guar gum is derived from a naturally occurring guar bean, and has a high molecular weight similar to CMC. The guar molecule has found successful application as a selective depressant for talc and silicate minerals. Figure 2.5 shows the molecular structure of a typical guar molecule.

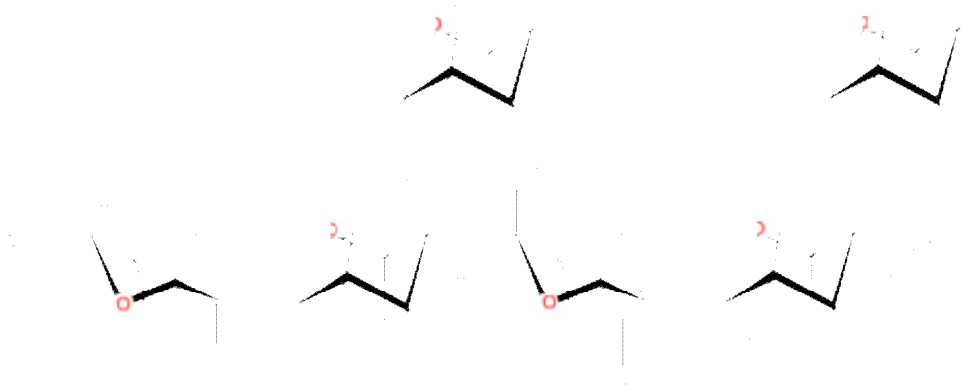


Figure 2.5: Guar Molecule Structure

Parolis *et al.* (2004) found that low-charged guar depressants adsorbed more strongly to mineral surfaces than the high-charged CMC depressants. Wang *et al.* (2005) found

that adsorption of guar onto mineral surfaces was not affected by changes in pH and ionic strength and that adsorption occurred mainly through hydrogen bonding.

Wiese *et al.* (2005) observed, when using guar gum to depress natural floating gangue in platinum ores, that depression of pyrrhotite was also possible under certain conditions. This means that a depressant that is used to depress gangue minerals may also be used to depress some sulphide minerals. Steenberg and Harris (1984) also concluded that the adsorption of the polysaccharides in flotation systems is unlikely to be selective. They suggested that the apparent selectivity is created by the much stronger specific surface interaction of the collectors which chemisorb onto the mineral surface and displace the weakly held polymer from the valuable mineral surface, thereby leaving it in a hydrophobic state and available for flotation.

In the study by Beattie *et al.* (2006) on different types of depressants in the flotation of talc and sulphide minerals, it was found that an increase in the size (molecular weight) of the depressant resulted in an increase in its strength which reduced its action in terms of selectivity (i.e. equal depression of both gangue and valuable minerals).

A similar study by Wiese *et al.* (2008) was conducted to compare the effect of a low molecular weight starch to guar and CMC in depressing the naturally floating gangue in PGM flotation. They found that the low molecular weight starch depressant was not effective in depressing the natural floating gangue, whereas both guar and CMC were.

Bicak *et al.* (2007) found that guar gum and CMC can also be used to depress pyrite flotation. Like pyrite (FeS_2), pyrrhotite ($Fe_{1-x}S$) is an iron sulphide, meaning that the polysaccharide depressants can be tested to see whether pyrrhotite responds similarly to pyrite, and if its flotation can be curtailed by these depressants.

The polysaccharide depressants were chosen for use in this study as they have already found application as efficient depressants of talc and other silicate gangue minerals in the flotation of platinum ores. These depressants have also been found to be effective in depressing pyrite, especially at higher pH values (recovery decreased

as pH was increased). These reagents are also naturally occurring, meaning that their environmental impact will be minimal.

Table 2.5: Properties, typical applications and examples of organic and inorganic depressants (adapted from Lovell, 1982)

Depressant	Type	Example	Properties	Application
Inorganic	Cyanide	sodium cyanide	Can depress most sulphides	Sulphides
	Chromate	chromate	Can depress lead sulphides	PbS
	Sodium Silicate	sodium silicate	Also acts as a dispersant and embrittles froths	disperse siliceous gangue slimes
Organic	Polyglycol Ethers	nonylphenyltetraglycol ether	they have the same structure as frothers	calcite and dolomite
	Polyphenols	quebracho	complex mixture of polyphenols	Calcite
	Polysaccharides	**starch**	used as a flocculant as well as a depressant	talcaceous minerals and calcite
		carboxymethylcellulose	used as a dispersant as well as a depressant	Talc
		guar gum	used as a flocculant as well as a depressant	talc and siliceous gangue

2.5.4.3 pH and pH modifiers

Most reagent-mineral systems are controlled by the pulp solution pH, making pH the driving force which determines the interactions between minerals and reagents in solution (Chander, 1988). Therefore, the choice of operating pH and pH modifier is important, as this may determine the success of reagent-mineral interaction.

In the study by Kolahdoozan (1996) which examined the effect of xanthate adsorption on monoclinic and hexagonal pyrrhotite at different pH values, it was found that the adsorption was dependent on pH: monoclinic pyrrhotite showed higher adsorption at pH 7 and 8.5 than hexagonal pyrrhotite. However this trend was seen to be reversed at pH 10 and above, where hexagonal pyrrhotite showed better adsorption than monoclinic pyrrhotite.

Bicak *et al.* (2007) also found that the adsorption of guar gum depressant on pyrite increased as the pulp pH was increased. They attributed this to the formation of stable hydrophilic $Fe(OH)_3$ species at higher pH values. The same hydrophilic species would be expected to form on pyrrhotite surfaces and result in the same kind of response as was observed on pyrite, which is depression of the mineral.

A number of pH modifiers are in common use, including lime, soda ash, sulphuric acid, sulphur dioxide, etc. The choice of which of these to use depends on the type of minerals present and economic factors, as well as the presence of other reagents in the pulp (like collectors, activators) that may also affect the function of the pH modifier.

In the flotation of most platinum bearing ores from the BIC, like the Sheba's Ridge ore, where there is usually a significant amount of silicate gangue minerals present, the natural flotation pH is between 9.0 and 9.5. At this pH range both pentlandite and pyrrhotite generally float well. Increasing pH generally leads to the formation of hydrophilic iron species on the surface of these sulphides, as was observed by Bicak *et al.* (2007) on the pyrite surface. The rate at which these hydrophilic species form is also linked to the rate of oxidation of the sulphide minerals.

2.5.4.4 Chelating agents

Polyamines such as triethylenetetramine (TETA) and diethylenetriamine (DETA), which are powerful chelating agents, have been found to provide good depression of hexagonal pyrrhotite in massive nickel–copper sulphide ore samples. In test work conducted by Kelebek and Tukel (1999), they found that when TETA was used in conjunction with sodium metabisulfite (SMBS), this produced an excellent separation of pentlandite from pyrrhotite resulting in a much greater concentrate grade. The focus of this project, however, was to use reagents currently in use in the ores from the BIC, and not to introduce any new reagents. For this reason, these reagents will not be considered in this study.

2.5.5 Surface Oxidation

Surface oxidation is one of the most important factors that influence the flotation performance (selectivity and recovery) of complex sulphide ores. Oxidation arises from the dissolution of minerals and grinding media. Dissolved metal ions hydrolyse and sulphide ions oxidise. These ions can be re-adsorbed on the mineral surfaces or react with each other before precipitating (Clarke *et al.*, 1995).

The alteration of pentlandite due to the oxidation of the surface has been studied by a number of researchers. Broomhead and Layers (1976) studied the factors governing the dissolution of nickel and iron in pentlandite and pyrrhotite concentrates in aqueous solutions at ambient temperatures and pressures. They showed that the direct reaction of oxygen with pyrrhotite in aerated water is much faster than that with pentlandite. The difference in reactivity was explained by the different crystal structures whereby the pyrrhotite structure has vacancies in it while pentlandite does not.

Boyd (1979) studied the alteration in the surface chemistry of pentlandite when exposed to air and found that if the mineral was cleaved in pure argon, the only oxidation product that was formed was Fe_2O_3. When the mineral was cleaved in air, Fe_2O_3 and NiO were produced. It was also found that when the mineral was in contact with SIBX, the main identifiable alteration products produced were FeO, nickel xanthate and $Ni(IBX)_2$.

Thornber (1983) studied the mineralogical and electrochemical stability of nickel-iron sulphides. In a primary assemblage composed of pentlandite, pyrrhotite and pyrite, pentlandite reacted first at the lowest oxidation potential and was altered to violarite, with iron and nickel released into solution according to the following reaction:

$$Fe_{4.5}Ni_{4.5}S_8 \rightarrow 2FeNi_2S_4 + 0.5Ni^{2+} + 2.5Fe^{2+} + 6e^- \quad \text{……(11)}$$

At the same time, the increased nickel activity caused the pyrrhotite to become unstable and it took up nickel from solution to form violarite

$$Fe_7S_8 + 4Ni^{2+} \rightarrow 2FeNi_2S_4 + 5Fe^{2+} + 2e^- \quad \text{……(12)}$$

Richardson and Vaughan (1989) studied surface alteration due to oxidation using synthetic pentlandite in contact with air, steam, ammonium hydroxide, hydrogen peroxide, or sulphuric acid as oxidants. They found that after oxidation, the pentlandite subsurface was enriched in nickel and was believed to have been restructured to violarite. The oxidised surfaces consisted of a range of iron oxides and hydroxides {Fe_3O_4, Fe_2O_3, FeOOH, $Fe(OH)_3$ }, nickel oxide (NiO) and iron sulphates {$FeSO_4$, $Fe_2(SO_4)_3$}. The proportions of the phases present in the surface layer were dependent on the strength of the oxidant employed and the thermodynamic stability of the phases.

Buckley and Woods (1991) used electrochemical techniques to investigate the surface oxidation of pentlandite when exposed to air. They found that iron was removed from the pentlandite lattice to form a hydrated iron oxide over-layer, which left metal deficient pentlandite in addition to a restructured nickel-iron sulphide. Further oxidation resulted in some nickel being included in the oxide over-layer. The study was carried under both acidic conditions, pH ~ 3 and 4.6, and alkaline conditions, pH ~ 9 and 13. The study in acetic acid showed that the oxide layer was largely soluble while oxidation in basic media indicated that virtually all the iron and most of the nickel within the outermost few nanometres were bonded to oxygen.

According to a recent study by Legrand *et al.* (2005) on pentlandite oxidation, numerous secondary products and alteration zones can form depending on their kinetics. From the XPS and Auger data presented in this study, it was shown that the violarite and FeOOH assemblage were the first to form in measurable amounts on the pentlandite surface. These were formed through the oxidation of the Fe (II) to Fe (III), with the Fe diffusing from the bulk pentlandite to the surface where it was oxidised to FeOOH. The underlying layer to FeOOH (the reacted pentlandite) was consequently Fe-depleted and residually enriched in S, leading to the formation of violarite. The other secondary layers were then deduced from these initial layers. These layers that formed and altered the surface of pentlandite were consequently hydrophilic, thereby reducing the floatability of pentlandite. They also prevented the adsorption of flotation reagents onto the mineral surface, as there was competition as to what

adsorbed on the surface first. This means that for the flotation reagents to be effective, they would need to be added prior to the oxidation of the pentlandite surface.

Nicholson and Scharer (1994) and Belzile *et al.* (2004) studied the oxidation of pyrrhotite and found that when oxygen was the primary oxidant, the oxidation of pyrrhotite proceeded as follows:

$$Fe_{1-x}S + (2 - ½x)O_2 + xH_2O \rightarrow (1-x)Fe^{2+} + SO_4^{2-} + 2xH^+ \text{ ……..(13)}$$

Depending on the pH (above 4), the ferrous iron was proposed to produce ferric iron that precipitated out of solution as ferric hydroxide according to the following reactions:

$$Fe^{2+} + ¼\ O_2 + 2H^+ \rightarrow Fe^{3+} + ½H_2O \text{ ……..(14)}$$
$$Fe^{3+} + 3\ H_2O \rightarrow Fe(OH)_3 + 3\ H^+ \text{ ……...(15)}$$

It was proposed that if enough time was allowed for the oxidation of the pyrrhotite surface, the mechanism involved the reduction of oxygen on the surface where it reacted with Fe(III) bonded to sulphur to form Fe(III) – O bonds and Fe(III) – oxyhydroxides. The Fe(III) – oxyhydroxide layer then grew thicker through the diffusion of iron from the pyrrhotite lattice resulting in a depletion of iron from the zone immediately below the Fe(III) – oxyhydroxide. This is illustrated in Figure 2.6 below:

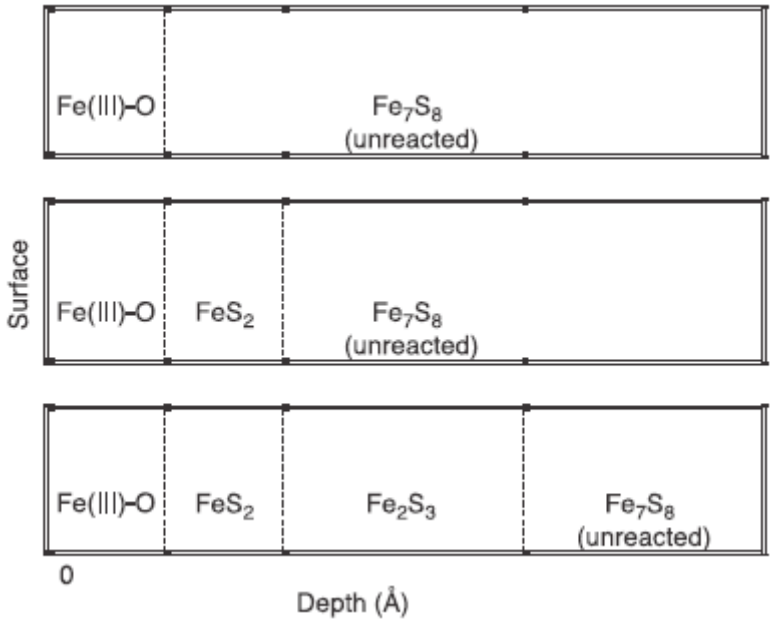


Figure 2.6: Model illustrating the sequence of oxidation products at the surface of pyrrhotite (from Pratt *et al.,* 1994)

Some form of oxidation is required for self-induced and xanthate-induced flotation of sulphides to proceed. However, excessive oxidation has a detrimental effect on the minerals as it causes the alteration of the sulphide mineral surfaces and formation of secondary products that render the surfaces hydrophilic. It has also been shown that the formation of hydrophilic iron hydroxides associated with the oxidation reaction would be expected to occur more rapidly on pyrrhotite than on pentlandite, as pyrrhotite oxidises more quickly than pentlandite resulting in the selective depression of pyrrhotite relative to pentlandite.

2.5.6 Metal Activation

The presence of metal ions in solution has a major influence on the flotation of sulphide minerals. Metal ions, which are always present from grinding media and salts dissolved in pulp solution, are argued to contaminate the surface of minerals to the detriment of the selective flotation process (Schreithofer *et al.*, 2000).

The study carried out by Hodgson and Agar (1989) to investigate the oxidation of pentlandite and pyrrhotite also investigated the electrochemical effect of Ca^{2+}, $S_2O_3^{2-}$ and SO_4^{2-} ions on pentlandite and pyrrhotite floatability and xanthate interactions. In this study, $S_2O_3^{2-}$ and Ca^{2+} ions were found to be surface active at the normal process pH and competing with xanthate for adsorption on the surface sites of pentlandite.

Hodgson and Agar (1989) proposed that the Ca^{2+} ions were adsorbed onto the surface sulphur sites, while the $S_2O_3^{2-}$ adsorbed onto the Fe sites. The $S_2O_3^{2-}$ ion was considered to be coordinated onto the surface via the oxidised Fe sites or the CaS_2 product. Iron and polysulphides were considered to be the surface-active forms, forming part of the pyrrhotite surface. Calcium (Ca^{2+}) cations were proposed to have chemisorbed onto the pentlandite surface, replacing metal ions. The initial oxidation reaction for pentlandite (pH independent) was proposed to be of the following form:

$$Fe_{4.5}Ni_{4.5}S_8 \rightarrow (Fe_{4.5-x}Ni_{4.5-x}S_8) + xFe^{2+} + xNi^{2+} + 4xe^- \text{.....(16)}$$

This reaction was proposed to have been modified by the adsorption of the Ca^{2+} on to the metal surface, resulting in the replacement of metal ions on the surface and changing the stoichiometry in the surface. The altered reaction (16) was then proposed to proceed as follows:

$$Fe_{4.5}Ni_{4.5}S_8 + Ca^{+} \rightarrow (Fe_{4.5-x}Ni_{4.5-x}Ca_{ads}S_8) + xFe^{2+} + xNi^{2+} + (4x\text{-}2)e^- \text{.....(17)}$$

From the work done by Hodgson and Agar (1989) it was also proposed that the presence of the calcium ions on the surface facilitated the formation of thiosulphates and sulphates, which precipitated out from solution on to the pentlandite surface. These precipitation reactions resulted in the promotion of pentlandite oxidation and thus reduced the hydrophobicity of the mineral surface. They concluded that these ions would influence the extent of xanthate adsorption by the sulphide minerals as well as control the hydrophobicity.

Schreithofer *et al.* (2000) also investigated the effect of Ca^{+} and $S_2O_3^{2-}$ ions on the flotation of pentlandite. They found that when using steel media in grinding, these ions had an activating effect on pentlandite flotation whereas when using ceramic media in milling the same ions had a depressing effect.

Kirjavainen *et al.* (2002) concluded that calcium and thiosulphate ions improved floatability of nickel and copper sulphides at the normal process pH (i.e. pH 9) after grinding in a steel mill. They observed that galvanic interaction between sulphides and mill iron is of major importance. When the galvanic effect of mill iron was present, calcium ions activated nickel and copper sulphides and increased the adsorption of ethyl xanthate on the sulphides. Thiosulphate ions were found to reduce the formation of hydrophilic compounds on sulphide surfaces and resulted in the improved flotation of the sulphides (Shackleton, 2003).

Malysiak *et al.* (2002) studied the pentlandite-feldspar interaction in a platinum ore and its effect on separation by flotation. It was found that when there were copper ions present in solution, they coated both the feldspar and pentlandite surface. This caused the xanthate to be subsequently adsorbed onto both the pentlandite and feldspar surface causing the feldspar to float together with the pentlandite. This resulted in decreased selectivity in the separation of pentlandite from feldspar. Electrochemical studies on the activation of pyrrhotite by copper ions (Nicol, 1984) suggested that activation of pyrrhotite could be described by the following reaction:

$$Fe_9S_{10} + 9Cu^{2+} \leftrightarrow 9Fe^{2+} + 9CuS + S^0 \quad \text{.....(18)}$$

In a separate study by Jean and Bancroft (1986) on the activation effect of heavy metals (copper, lead and zinc) on pyrrhotite, it was shown that pyrrhotite could adsorb large amounts of these ions. A strongly oxidising environment inhibited this adsorption due to the formation of a protective ferric hydroxide layer on the pyrrhotite surface.

Some metal ions will always be present in solution and so understanding the impact these ions will have on the system under investigation in this study is critical. This will therefore be considered in the interpretation of the results.

2.5.7 Galvanic interaction

As mentioned previously, sulphide minerals are semiconductors (Bradshaw *et al.*, 2006; Bozkurt, 1997) and they assume a particular rest potential when in aqueous solution. This rest potential, sometimes referred to as the mineral/solution interfacial potential, is influenced by the presence of oxidising or reducing species in solution (Bradshaw *et al.*, 2006), and is the driving force in the galvanic interactions between minerals. For example, when two sulphide minerals are in contact with each other in an aqueous solution, the mineral with the higher rest potential will act as the cathode while the other mineral, with a lower rest potential, will act as the anode (Bozkurt, 1997). These two minerals eventually reach an equilibrium where they adopt a mixed potential which is somewhere in between their two rest potentials. The electrode reactions for this mineral system were described by Kocabag and Smith (1985) to proceed as follows:

Anodic reaction:

$$MS + H_2O \leftrightarrow M(H_2O)^{2+} + S^0 + 2e^- \text{.....(19)}$$

where MS = Metal Sulphide

Cathodic reaction (oxygen reduction):

at high solution pH: $\frac{1}{2} O_2 + H_2O + 2e^- \leftrightarrow 2(OH)^-$ ……..…....(20)

at low solution pH: $\frac{1}{2} O_2 + H^+ + 2e^- \leftrightarrow H_2O$ ……..…...(21)

with the overall reaction as follows:

$$MS + \frac{1}{2} O_2 + H_2O \leftrightarrow M^{2+} + 2(OH)^- + S^0 \text{ …(22)}$$

Rand (1977) found that the order of rest potentials at pH 9.06 for oxygen reduction is as follows:

Pyrite > pentlandite > chalcocite > chalcopyrite > covellite > bornite > arsenopyrite > pyrrhotite > galena > sphalerite > steel grinding media

The above list means that pyrite will give the highest rate of oxygen reduction and sphalerite the lowest. Therefore, if pyrite is in solution with another sulphide (e.g. chalcopyrite), oxygen reduction will take place preferentially on pyrite and the oxidation of the other sulphide will be accelerated. This means that pyrite will act as the cathode and the other sulphide as an anode. The enhanced oxidation on the anodic mineral will result in the release of more metal ions from this mineral than would be the case in the absence of galvanic interactions (Bozkurt, 1997). This process is what is known as galvanic interaction and is illustrated in Figure 2.7 where pentlandite is the cathodic mineral and pyrrhotite the anodic mineral:

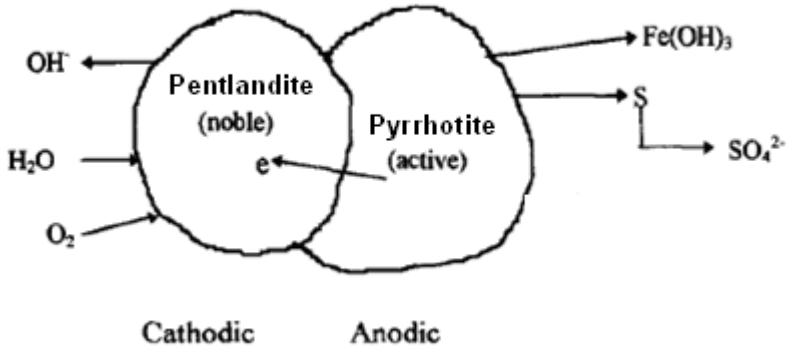


Figure 2.7: Galvanic interaction between pentlandite and pyrrhotite (adapted from Ekmekci and Demirel, 1997)

Understanding galvanic interaction is probably the most important of all the factors mentioned above, as it encompasses most of the other factors such as oxidation and metal activation. An understanding of the galvanic interaction that takes place in the pentlandite-pyrrhotite system will result in a better understanding of the sub-processes that will be investigated in this study.

2.6 Flotation of complex ores

As mentioned in Section 2.4, flotation is a physico-chemical process that exploits the differences in surface properties between the desired valuable minerals and the unwanted gangue minerals in an ore. Complex ores can be described as ore deposits that have multiple valuable minerals and a low feed grade. Examples of complex ores that are found in the BIC are the Platreef ores and the Merensky reef ores. One

example of a Platreef ore is the Sheba's Ridge ore. Firstly its mineralogy is divided into three distinct units of mineralisation, a layer similar to the UG2 termed the "Platchro layer"; an upper mineralised pyroxenite (UMP) layer analogous to the Merensky Reef; and a wide sulphide zone similar to the Platreef (Ridge Mining, 2008). What really makes this ore a complex is that it carries PGMs and valuable base metal sulphides like pentlandite (Ni) and chalcopyrite (Cu), at low feed grades. The PGM feed grade is ~2 g/t, the Ni feed grade is 0.2 % and the Cu feed grade is 0.07 %.

The problem with the flotation of complex ores is that it is usually difficult to find a flotation condition which will be suitable for optimising the recovery of all the valuable minerals and still get good upgrading. This is because the flotation response of all the targeted minerals is not the same. Therefore, special care needs to be taken when processing these ores.

2.7 Summary

Flotation is a complex process which has many sub-processes and interactions. Klimpel (1988) divided the flotation process into three main components, viz.; the chemical, equipment and operational components. The focus of this thesis is on characterising the flotation behaviour of pentlandite and pyrrhotite in a probe ore (Nkomati ore) in order to exploit the selective flotation of pentlandite from pyrrhotite in the ore of interest (Sheba's Ridge ore). In terms of the Klimpel terminology, the work will investigate the "chemical component" through an evaluation of how collectors, depressants and pH modifiers affect the process. Mineralogy was considered as it is thought to be the most significant factor that has an influence on the chemical factors.

A review of the literature on the factors that are the main focus of this thesis can be summarised as follows:

- Pentlandite is an important source of nickel and is a host mineral for the valuable PGEs and PGMs in the Bushveld Igneous Complex. Pentlandite is commonly found in close association with pyrrhotite. The pyrrhotite in the

Sheba's Ridge ore does not contain PGEs and PGMs, hence processing the base metal sulphides in this ore should reject as much of the pyrrhotite as possible.

- There are three possible mechanisms responsible for sulphide mineral flotation, viz.; through self-induced, sulphur-induced and collector-induced flotation (Trahar, 1984). Each type of flotability is considered to be dependent on the oxidation-reduction state of the pulp. The Sheba's Ridge ore is a low grade sulphide ore (~2% sulphide content) and therefore collector-induced flotation is of interest here.
- There is a wide range of commercially available collectors, classified on the basis of whether they exist as cations, anions or molecular species in solution. Sulphydryl anionic xanthate collectors, with varying chain lengths, are commonly used in the flotation of platinum ores and were therefore chosen for use in this study.
- The success of collector-induced flotation is dependant on the oxidation state of the pulp. Therefore, a certain degree of oxidation must take place in the pulp to facilitate the collector action (Bozkurt *et al*., 1998; Hodgson and Agar, 1989) but too much oxidation can impair the floatability of the valuable pentlandite. To what extent the oxidation process needs to proceed is unclear, and this will be one of the variables that will be investigated in this thesis.
- In most cases, the addition of collectors alone is not adequate to optimise the flotation process. For this reason, other reagents such as depressants, activators and pH modifiers are used in conjunction with collectors to ensure that only the right particles float. This thesis will investigate the use of the polysaccharide organic depressants; CMC, guar and starch, in conjunction with xanthate collectors, to improve the flotation of pentlandite while depressing pyrrhotite flotation.
- Sulphides are known to be semiconductors and oxidise readily in the presence of water and oxygen via a coupled electrochemical reaction with the reduction of oxygen (Bradshaw *et al*., 2006). The galvanic interaction of the sulphides with each other (Bozkurt, 1997) further accelerates their oxidation. Therefore, when considering the effect of oxidation in this thesis, consideration will also be given to the effect of galvanic interaction.

2.8 Key Questions

Based upon the literature review, the following key questions have been formulated for this study:

2.8.1 How does the xanthate chain length affect the flotation recovery of pentlandite and pyrrhotite?

2.8.2 How does the depressant type affect the selective flotation of pentlandite over pyrrhotite:

i) for collectorless flotation?
ii) in the presence of different chain-length xanthate collectors?

2.8.3 Can the flotation selectivity be further influenced by:

i) oxygen addition
ii) pH variation/modification
iii) type of pH modifier

2.8.4 Do the results (in terms of pH and depressant type) obtained in the microflotation test work done on the Nkomati ore translate to batch flotation test work done on Sheba's Ridge ore?

2.9 Hypotheses

2.9.1 An increase in the xanthate collector's hydrocarbon chain length will result in an increase in the collector's hydrophobicity and thus a decrease in the collector's selectivity.

2.9.2 A charged depressant like CMC will be more specific in its action as it will only adsorb and depress minerals with the opposite charge and thus be selective in its depression abilities unlike the uncharged depressants (guar and starch) which will adsorb on any surface irrespective of the charge present on the surface.

2.9.3 As pyrrhotite oxidises more rapidly than pentlandite, hydrophilic species will form more quickly on the surface of pyrrhotite and thus better flotation selectivity of pentlandite will be achieved during the initial stages of flotation.

2.9.4 An increase in pulp pH will result in an increase of the hydroxyl groups present in solution, which in turn will accelerate the oxidation of the different sulphides present in the pulp starting with the fastest oxidising sulphide (pyrrhotite). This will result in better flotation selectivity of pentlandite at higher pH values.

Chapter 3: Experimental Details

3.1 Introduction

This chapter describes the ores used in this thesis, and their mineral compositions and associations; the preparation of the samples for the microflotation and batch flotation testwork; the equipment used; the reagents used; the experimental techniques and the analyses carried out in this thesis.

3.2 Ore Samples

Two ores were used for this study, namely Nkomati massive sulphide ore and Sheba's Ridge ore. The Nkomati massive sulphide ore was used as a probe ore in microflotation testwork, as it has a high sulphide content (> 80% BMS). The sulphides present include chalcopyrite (4.7%) pentlandite (7.6%) and pyrrhotite (65.9%). The Sheba's Ridge ore (with ~ 2% sulphide composition) is the ore of commercial interest, used in the batch flotation test work.

The Nkomati ore was used in the microflotation test work to evaluate different reagents and conditions and to determine the best conditions for pentlandite selectivity over pyrrhotite. These results were then translated to the Sheba's Ridge ore in the more realistic environment of a laboratory batch flotation cell. A summary of the two ores used in this study is given in Table 3.1:

Table 3.1: Summary of ores used with mineral contents given by Newell *et al*., (2006) and Mintek (internal communication, 2007)

	Probe Ore	**Ore of interest**
Name of ore	Nkomati	Sheba's Ridge
Sulphide type	Massive (>80 %) sulphide	Disseminated (~2 %) sulphide
Main sulphides present	Pyrrhotite, pentlandite, chalcopyrite and pyrite	Pyrrhotite, pentlandite and chalcopyrite
Testwork programme	Microflotation	Batch flotation
Characteristics	Similar in character and sulphide mineralisation to the BIC.	Main sulphide zone said to be similar to the Platreef which is also part of the BIC
Pn/Po ratio	0.12	0.86
Weight % alteration minerals (talc, chlorite, serpentine,)	0	20.9 (13 % talc)

The minerals that were of particular interest in this study were pentlandite and pyrrhotite. The mineral composition and association for the two ores, based on QEMSCAN analysis, is summarised in Tables 3.2 and 3.3. The ore of interest, Sheba's Ridge, is a highly altered ore. Nkomati ore is a massive sulphide ore body, which has little alteration in it. Optical microscopy determinations using the magnetic colloid method indicated that the pyrrhotite in the Nkomati ore sample was a mixture of both magnetic and non-magnetic types as can be seen from Figure 3.1 below (Becker *et al.*, 2007):

Table 3.2: Modal analysis of mineral samples from Newell *et al.* (2006) and Mintek (internal communication, 2007)

	Nkomati Massive Sulphide	**Sheba's Ridge**
Mineral	**Weight %**	**Weight %**
Pentlandite	**7.62**	**0.60**
Pyrrhotite	**65.93**	**0.70**
Chalcopyrite	4.69	0.20
Pyrite	3.17	-
Other BMS	0.00	0.20
Fe-Oxides	17.13	2.80
Biotite	0.14	2.90
Plagioclase	0.79	9.30
Pyroxene	0.16	21.10
Olivine	0.15	22.30
Quartz	0.12	-
Amphibole	-	16.30
Talc	-	13.00
Serpentine	-	5.30
Chlorite	-	2.60
Other	-	2.70
Total	**100.00**	**100.00**

Table 3.3: Nkomati ore mineral associations on particles in the size fraction -106 µm and +75 µm from Newell *et al.* (2006)

PENTLANDITE		**PYRRHOTITE**	
Mineral Association	%	Mineral Association	%
Liberated	**75**	**Liberated**	**89**
Associated with pyrrhotite	**16**	**Associated with pentlandite**	**2**
Associated with chalcopyrite	**<1**	**Associated with chalcopyrite**	**1**
Associated with pyrite	**2**	**Associated with Pyrite**	**1**
Associated with Fe-oxides	**4**	**Associated with Fe-oxides**	**6**
Secondary Silicates (Chlorite, Quartz, Olivine, Amphibole)	<1	Secondary Silicates (Chlorite, Quartz, Olivine, Amphibole)	<1
Primary Silicates (Feldspar, Orthopyroxene)	<1	Primary Silicates (Feldspar, Orthopyroxene)	<1
Chromite	0	Chromite	0
Ternary / Composites	3	Ternary / Composites	1
Total	**100**	**Total**	**100**

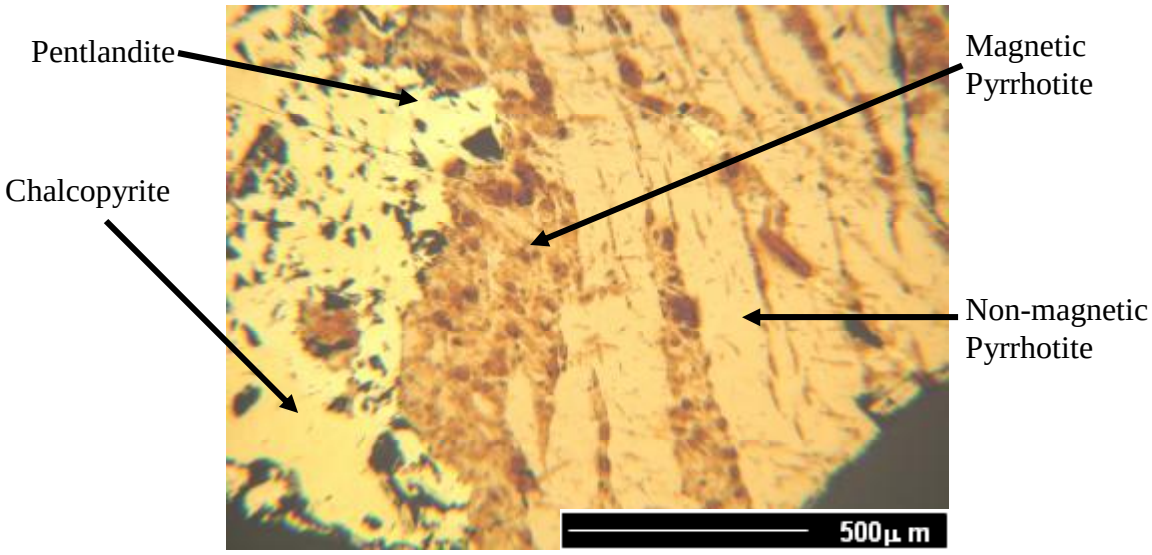


Figure 3.1: Reflected light photomicrograph of the Nkomati ore with the magnetic colloid showing the different types of pyrrhotite present within the sample (Becker *et al.*, 2007)

3.3 Sample Preparation and Characterisation

3.3.1 Microflotation Testwork

The sample used in the microflotation studies was the Nkomati massive sulphide ore. Lump samples were hand picked from the massive sulphide ore and carefully crushed and dry ground with chrome steel media. The crushed sample was dry sieved to obtain the desired size fraction for microflotation, which was -106 µm +75 µm. Using a rotary splitter, this material was split into 10 sub-samples for the microflotation tests. The sub-samples were stored in a freezer until use to minimise the effect of oxidation. The 2 g samples for microflotation were only weighed out from the sub-samples when the microflotation tests were about to commence. BET analysis was carried out on one of the sub-samples to determine the specific surface area of the particles, which was found to be:

BET Surface Area: $0.0890 \pm 0.0008\ m^2/g$

The BET machine used was the TriStar Micromeritics in the department of Chemical Engineering at the University of Cape Town which is a standard 5 point BET using nitrogen gas for degassing prior to the actual measurements.

3.3.2 Batch Flotation Testwork

The Sheba's Ridge ore sample used for this part of the study was received from Mintek in two batches of 15 kg each, bringing the total sample mass to 30 kg. The material in each batch had been crushed to - 6 mm. The two batches were blended together to make up a single homogeneous sample, which was further crushed with a cone crusher to 100% passing 2 mm. This sample was blended again and split using a riffler splitter into three sub-samples of 10 kg each. These were then further split with a rotary splitter to sub-samples of 1 kg each. The 30 sets of 1 kg samples were stored in a freezer to minimise the oxidation of the sulphides in the samples prior to the batch flotation tests.

A milling curve was established prior to the batch flotation tests to determine the milling time required to achieve the desired grind size (of 60% -75 µm). This was carried out using a 200 mm diameter Erietz laboratory stainless steel rod mill. This mill was charged with 20 stainless steel rods with varying diameters as follows: 6 rods at 25 mm, 8 rods at 20 mm and 6 rods at 16 mm. The milling was carried out on wet slurry that was made up of 1 kg of dry ore mixed it with 1 kg of synthetic plant water (Table 3.6 below) to make up a slurry of 50% solids by mass.

The target grind size for flotation tests was 60% passing 75 microns. The milling curve that was obtained for the Sheba's ridge ore sample is presented in Figure 3.2:

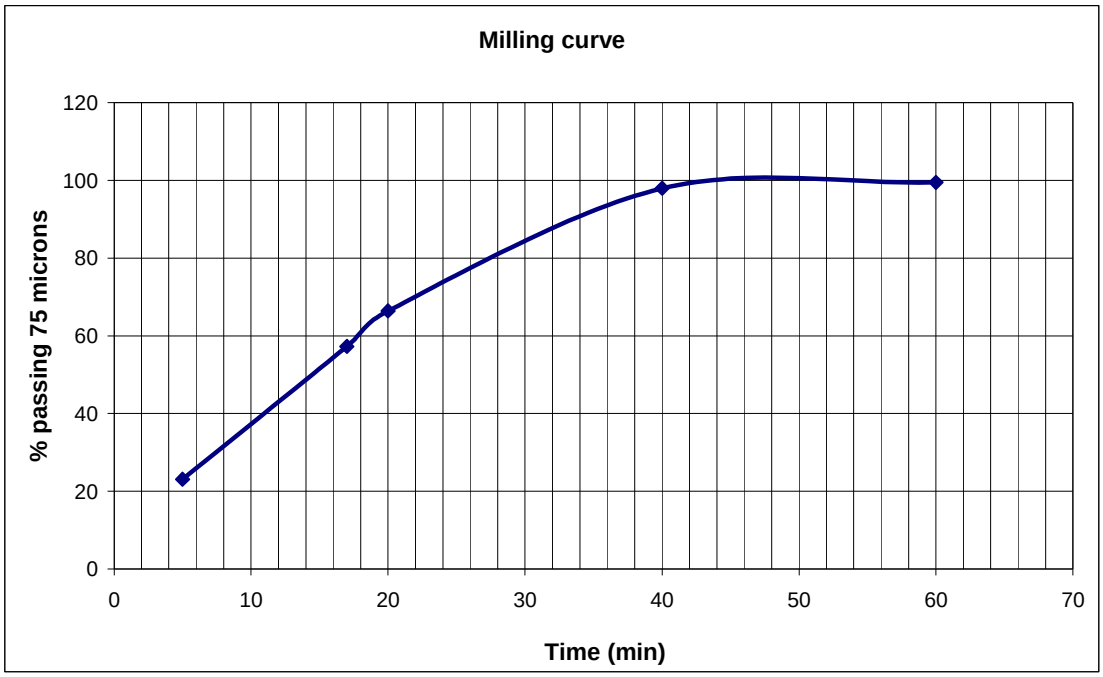


Figure 3.2: Sheba's Ridge ore sample milling curve

Once the milling curve had been established, the grind time was set at 18 minutes of milling. Batch flotation tests were conducted after milling the sample for this prescribed milling time. During the flotation tests, this milled slurry was immediately transferred into the flotation cell from the mill, as will be described in more detail in Section 3.7.

3.4 Reagents

3.4.1 Collectors

The xanthates of different chain lengths tested in this thesis were sodium ethyl xanthate (SEX), sodium normal propyl xanthate (SNPX), sodium iso-butyl xanthate (SIBX) and potassium amyl xanthate (PAX). The collectors were received as powder and were made up into solutions of 1% strength using distilled water prior to being used. For microflotation tests, the 1% stock solution was further diluted using distilled water to get the desired solution strength of $1x10^{-5}$ M. For batch flotation tests, the 1 % solution was used as is. All were supplied by SENMIN. Details of the collectors used are given in Table 3.4.

Table 3.4: Collectors used in the testwork

Collectors Used				
Name	**Abbreviation**	**Formula**	**Molecular Weight (g/gmol)**	**Concentration**
Sodium **E**thyl **X**anthate	SEX	$\mathbf{C_3H_5NaOS_2}$	144.16	$1x10^{-5}$ M
Sodium **N**ormal **P**ropyl **X**anthate	SNPX	$\mathbf{C_4H_7NaOS_2}$	158.17	$1x10^{-5}$ M
Sodium **I**so-**B**utyl **X**anthate	SIBX	$\mathbf{C_5H_9NaOS_2}$	172.18	$1x10^{-5}$ M
Potassium **A**myl **X**anthate	PAX	$\mathbf{KC_6H_{11}S_2O}$	202.30	$1x10^{-5}$ M

3.4.2 Depressants

The three polysaccharide depressants used in this study were carboxymethylcellulose (CMC), guar gum, and starch. The CMC used in the microflotation tests was Depramin 186, while the CMC used in the batch tests was Depramin 267. The guar gum used for both microflotation tests and batch tests was Stypres 504 and the starch used was Stypres 301U. These depressants were received as powder and were made up into solutions of 1% strength using distilled water prior to being used. For the microflotation tests, the 1% stock solution was further diluted using distilled water to obtain the desired solution strength of 10 ppm. For the batch flotation tests, the 1 % solution was used as is. These depressants were obtained from Cytec and Chemzyme chemicals. Details of the depressants used are given in Table 3.5.

Table 3.5: Depressants used in the testwork

Depressants Used				
Reagent	**Name**	**Molecular Weight (g/gmol)**	**Purity (%)**	**Degree of Substitution**
CMC (Cytec)	Depramin 186	260487	~ 69	0.88
CMC (Cytec)	Depramin 267	319340	~ 67	0.65
Guar Gum (Chemzyme)	Stypres 504	187691	~ 88	N/A
Starch (Chemzyme)	Stypres 301U	12464	N/A	N/A

3.4.3 pH modifiers

Two pH modifiers were used in this testwork, namely sodium hydroxide (NaOH) and lime (CaO). These were supplied by Merck chemicals. Sodium hydroxide was used because it has higher purity and does not introduce excess calcium ions into solution. Lime was also used as it is the more economically affordable option for large scale operations. These were added prior to commencing the testwork, and just after the other reagent conditioning was done.

3.4.4 Frother

The frother used in all the batch flotation tests, made up to 1% solution strength, was DowFroth 200, a polyglycol ether supplied by SENMIN. The dosage level was kept constant at 40 g/t in the rougher stage and 20 g/t in the cleaning stage. No frother was used in the microflotation tests.

3.5 Synthetic Water

3.5.1 Microflotation Tests

All the microflotation tests were performed with water that was specially prepared to have an ionic strength (I.S.) of $1x10^{-2}$ Ca^{2+} using $Ca(NO_3)_2.4H_2O$ (supplied by Merck chemicals) at 3.33×10^{-3} M dissolved in Milli-Q de-ionised water. pH adjustment to the desired value was made using NaOH or lime (0.1 M prepared with distilled water). The stock solution was prepared in 20 *l* batches.

3.5.2 Batch Flotation Tests

Since the Sheba's Ridge ore is also from the Bushveld Complex, it contains minerals similar to those that are in the Merensky reef. Therefore, in this study, synthetic Merensky plant water was used for the batch tests, using the same formula as used by Wiese *et al.* (2006). The synthetic plant water was made by adding the salts listed in Table 3.6 into distilled water.

Table 3.6: Composition of synthetic plant water for batch tests (from Wiese *et. al.,* 2006)

Chemical salt	**Formula**	**Mass(g) per 1litre used**
Magnesium sulphate	$MgSO_4.7H_2O$	0.615
Magnesium nitrate	$Mg(NO_3)_2.6H_2O$	0.107
Calcium nitrate	$Ca(NO_3)_2.4H_2O$	0.236
Calcium chloride	$CaCl_2$	0.111
Sodium chloride	NaCl	0.356
Sodium carbonate	Na_2CO_3	0.030

3.6 Microflotation equipment and procedure

In this study, microflotation was used to investigate the floatability of pentlandite and pyrrhotite in the Nkomati probe ore; how they interact with different collectors and depressants, and how their flotation is affected by oxidation and pH. The UCT microflotation cell, shown in Figure 3.3, has a volume of 365 m*l*. This apparatus is designed to measure the floatability of pure minerals by focussing on the sub-processes that takes place in the pulp zone during mineral collection (Bradshaw and O' Connor, 1996). It also allows for investigation of the reagent-mineral interaction that takes in the pulp zone during flotation.

The cell consists of a conical tapered cylindrical tube with air introduced through a hypodermic needle at the base. Mineral loaded bubbles rise through the cell and are deflected by the cone at the top of the cell, after which they burst, resulting in the particles falling into the concentrate launder (see Figure 3.3). After a set time the

needle is removed and the solids in the launder are collected as the concentrate. The solids inside the microflotation cell are kept suspended using a Watson Marlow 520S peristaltic pump.

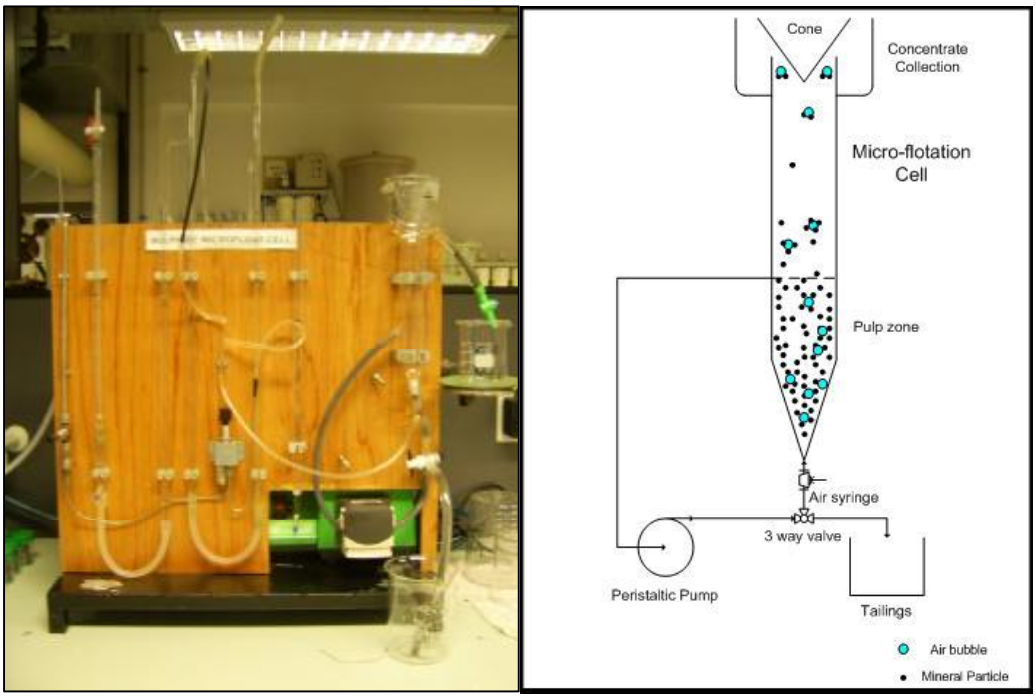


Figure 3.3: The UCT microflotation cell apparatus used in this study

The flotation tests conducted in the UCT microflotation cell were performed using 2 g samples. These samples were taken from one of the 10 sub-samples stored in the freezer. Once the 2 g sample was taken, the rest of the sample was returned to the freezer for storage to minimise oxidation of the sample. Each flotation sample was treated ultrasonically prior to flotation in an Elma Transonic 310 ultrasonic bath (0.05 W/cm^3) for 5 min. This was followed by decantation and removal of fines by wet sieving using a 75 µm sieve. The sample was then transferred to a 100 m*l* beaker for conditioning.

If oxidation was employed, the slurry was first sparged with medical oxygen at 6 m*l*/min for 5 min. After oxidation, reagent addition was performed by first adding the collector and mixing for 5 min with a glass impeller attached to an electrically driven shaft; and then adding the depressant and conditioning for another 5 min. After conditioning, the pulp pH was checked and adjusted to the desired pH using one of the pH modifiers.

The conditioned slurry was then transferred into the micro-flotation cell and the cell was filled up to the desired level with the specially prepared pH adjusted water (with $1x10^{-2}$ Ca^{2+} I.S.; refer to Section 3.5.1). The peristaltic pump was turned on and set at 80 rpm (equivalent to 803 m*l*/min) to keep the solids suspended in the cell. The flotation process was commenced by introducing synthetic air at the base of the cell through the hypodermal needle at an air flow-rate of 7 m*l*/min. Each test was continued for 20 minutes, with concentrates collected after 2, 5, 10 and 20 minutes.

Once the test was completed, the collected concentrates were filtered, dried and weighed to calculate mass recoveries. Elemental recoveries were determined by AAS (see Section 3.9.1). The microflotation test procedure is summarised in Table 3.7. All tests were conducted in duplicate.

3.7 Batch Flotation equipment and techniques

Two modified Leeds laboratory batch flotation cells were used in this testwork, namely a 3 *l* cell (for rougher tests) and a 650 m*l* cell (for the cleaner tests). The Leeds batch flotation cell is designed to simulate all the flotation sub-processes that take place in an industrialised full scale flotation system, but on a laboratory bench scale. The modified Leeds laboratory batch flotation cells that were used in this study are depicted in Figure 3.4.

Table 3.7: Microflotation test procedure

Equipment/Parameter	Experimental Detail
Synthetic Water	$1x10^{-2}$ Ca^{2+} I.S. water made up using $Ca(NO3)_2.4H_2O$
Filter Papers	Clearly labelled, weighed and dried in oven
Sample	2 g of -106 µm +75 µm sample
Ultrasonic Bath	To clean the surface of the minerals by treatment for 5 min in distilled water
Wet screening	75 µm sieve used to remove fines after ultrasonic treatment
Conditioning	In a 100 ml glass beaker with $1x10^{-5}$ M collector and 10 ppm depressant for 5 min each*; agitation was by means of a glass impeller
pH modifier	Lime or sodium hydroxide
Microflotation Cell	365 ml volume
Peristaltic Pump	80 rpm
Air Flow Rate	7 m*l*/min
Air Source	Synthetic air (with no H_2O and CO_2)
Concentrates	Collect concentrates after 2, 5, 10 and 20 min
Tailings	Collect the tailings when the test is finished
Filtering	Filter the concentrates and tailings samples
Drying and weighing	Dry the samples and weigh them
Masses	Weigh the mass of concentrates and tailings to complete the mass balance
Analysis	Acid digest the samples to prepare them for AA analyses

*When oxidation was used, this was done by sparging medical oxygen at 6 m*l*/min for 5 min prior to slurry conditioning.

It would have been better to use factorially designed experiments to conduct the microflotation tests, but this was however beyond the scope of this thesis.

Figure 3.4: Modified Leeds flotation cells used during this study (the 3 *l* cell on the left and the 650 m*l* cell on the right)

The milled slurry which already had been dosed with the desired amount of collector (see Section 3.3.2) was transferred into the 3 *l* flotation cell and diluted to between 30 and 35% solids, using synthetic plant water (see Section 3.5.2). The impeller was switched on and set to a speed of 1200 rpm. The slurry was then conditioned for 5 min with the depressant. Lastly, the frother was added and conditioned for 1 min (all the reagents were prepared to 1% solution strength). Concentrates were collected at timed intervals as shown in Table 3.8. During concentrate collection, froth scraping was done manually every 15 seconds. In the rougher stage, two concentrates were collected, the first one after 10 min (for fast floating material) and the second one after 30 min (i.e. 20 more min flotation time, for the slow floating material).

The rougher concentrates were subjected to cleaning tests which were conducted in the 650 m*l* flotation cell. The same basic procedure was followed as in the rougher stage, and the impeller speed was also run at 1200 rpm. The schematic representation of this rougher-cleaner test configuration is presented in Figure 3.5.

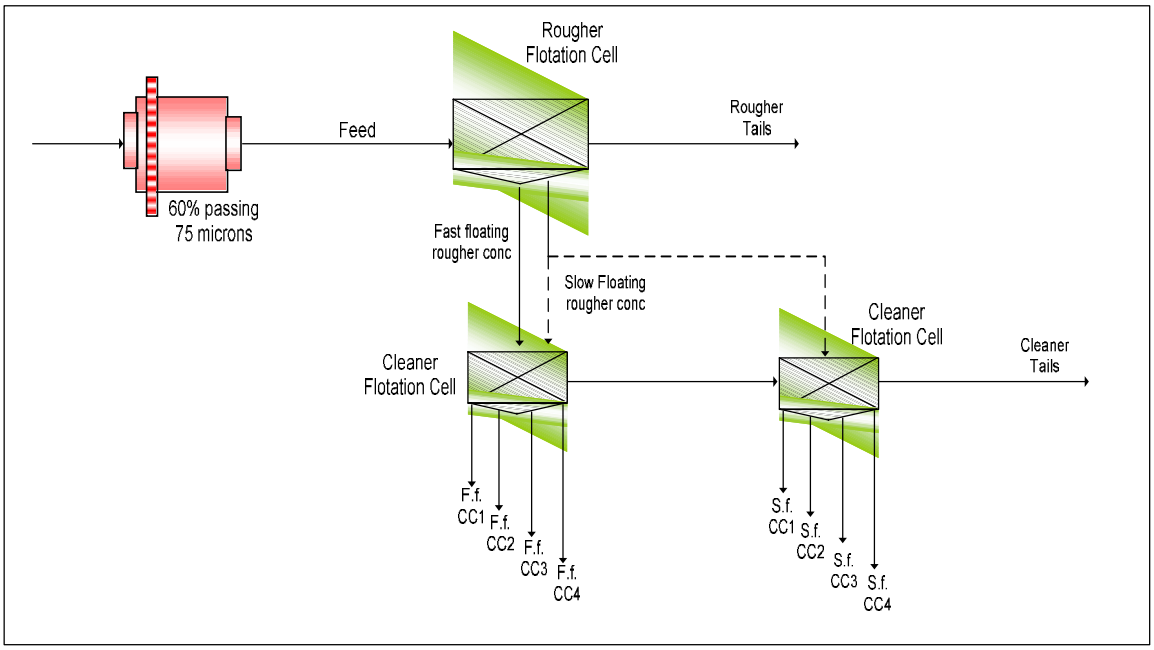


Figure 3.5: Cleaner Rate tests configuration

As shown in Figure 3.5, the slow floating rougher concentrate was either floated together with the fast floating fraction, or with the tails from the cleaner float of the fast floating fraction. In most tests the fast and slow floating fractions were added together and cleaned at the same time. The exact procedure of the batch flotation tests is given in Table 3.8.

A typical rougher concentrate has a higher concentration of the valuable minerals than the feed. However, this product is frequently not of sufficiently high grade to be sold as final product and thus needs further cleaning. This rougher concentrate then becomes the feed to a cleaner circuit. In this thesis the rougher concentrate from a laboratory batch test on Sheba's Ridge ore is considered (because of its higher grade than the original feed) to be the equivalent to the feed used in the microflotation tests (i.e. the Nkomati ore). This therefore means that the conditions tested in the microflotation tests will be tested in the cleaner stages of the lab batch flotation tests.

Table 3.8: Batch flotation test procedure

Rougher float				
Stage	**Reagent/Product**	**Dosage (g/t)**	**Conditioning/Float Time (min)**	**Cum Time (min)**
Mill	Collector	25	18	0
Float cell	Depressant	40	5	0 - 5
	Frother	40	1	5 - 6
Flotation	Fast floating fraction	-	10	6 - 16
	Slow floating fraction	-	20	16 - 36
Cleaner float				
Stage	**Reagent/Product**	**Dosage (g/t)**	**Conditioning/Float Time (min)**	**Cum Time (min)**
Float cell	pH modifier	adjust pH to 10.5*	as required	0
	Collector	10	2	0 - 2
	Depressant	10 - 100	5	2 - 7
	Frother	20	1	7 - 8
Flotation	Conc 1	-	2	8 - 10
	Conc 2	-	5	10 - 13
	Conc 3	-	10	13 - 18
	Conc 4	-	20	18 - 28

*Not all the cleaner tests were done at pH 10.5, some were done at natural pH, and for those tests the lime addition stage was skipped.

All the batch flotation products were filtered, dried, weighed and packaged and sent to Mintek for assaying for Cu, Ni and S (refer to Section 3.9.2 and 3.9.3). Rougher tailings from each test were assayed three times to ensure accuracy of the results.

3.8 Dissolved oxygen (DO) measurements

For measuring DO levels in the pulp, a YSI 5739 dissolved oxygen probe was used. The probe is fitted with YSI high speed membrane connected to a TPS meter.

3.9 Chemical Analysis

3.9.1 Atomic Absorption Spectroscopy (AAS)

AAS was used to analyse the concentrates and tailings from the microflotation testwork for the elements of interest, viz, copper, nickel and iron. Acid digestion was

needed prior to the measurement of samples by AAS. The acid digestion preparation procedure used can be found in Appendix A.

3.9.2 Inductively Coupled Plasma Optical Emission Spectroscopy

This method of chemical analysis was used for the samples emanating from the batch flotation testwork and was carried out at Mintek. The feeds, concentrates and tailings samples that were collected after each test were analysed for the elements of interest, viz, copper and nickel. The samples from the batch test work were first pulverised and then bagged and sent to Mintek where the analyses were done using standard procedures. The analysis of copper and nickel was done through an acid digestion preparation, using brominated HCl/HNO_3.

3.9.3 Sulphur (LECO) analysis

Sulphur analysis was done using the Leco method at Mintek on the batch flotation samples. This method uses combustion of oxygen to produce SO_2 to quantify for the amount of sulphur present in the sample.

3.10 Mineral recovery calculations for Nkomati Ore

Previous mineralogical work done on the Nkomati ore showed that the three main sulphides present in the sample had the following formulae and percent distributions:

- Pentlandite : $(Fe_{4.5}Ni_{4.5})S_8$
- Chalcopyrite : $CuFeS_2$
- Pyrrhotite : FeS

The sample was analysed for copper, nickel and iron. These proposed formulas could have been used to estimate mineral recoveries. However, these would still be estimate and not exact values. For this reason it was decided not to convert the elements to minerals and the data was analysed as individual elements.

Chapter 4: Microflotation test results

4.1 Introduction

This chapter presents the results of the first phase of work performed in this study, the microflotation testwork. Microflotation tests were used as a scoping tool in order to determine the optimum operating conditions for pentlandite/pyrrhotite selectivity using the Nkomati massive sulphide ore. The test work programme began with an evaluation of four xanthates and three depressants, and this was narrowed down to a single xanthate collector and a single depressant which were found to be best for pentlandite flotation and pyrrhotite depression. The effect of oxygen addition and pH modification were also investigated for the single xanthate collector and single depressant. In the analysis of the results, pentlandite/pyrrhotite selectivity was inferred from the Ni/Fe ratio, as Ni represents pentlandite and Fe pyrrhotite. A Ni/Fe ratio $\geq$ 1.3 was taken as meaning good pentlandite/pyrrhotite selectivity, and Ni/Fe ratio of < 1.3 was interpreted as poor pentlandite selectivity. This ratio of 1.3 was recommended by Dr Mike Bryson, a specialist consultant in the flotation group in Mintek.

Table 4.1 shows the conditions employed in each of the microflotation tests. Detailed experimental results from all the testwork done are presented in Appendix B. All the tests were performed in duplicate and the error bars shown in the figures in the rest of this chapter represent a two sigma standard deviation.

Table 4.1: Testwork conducted in this phase of the study

	Polysaccharide depressants used						pH
Collectors	**Guar: Stypress 504**		**CMC: DEP 186**		**Starch: Stypress 301U**		(Lime +
@ $1x10^{-5}$M	0 ppm	10 ppm	0 ppm	10 ppm	0 ppm	10 ppm	NaOH)
No Collector	-	√	-	√	-	√	9
SEX	√	√	√	√	√	√	9
SNPX	√	√	√	√	√	√	9; 9.5; 10; 10.5
SIBX	√	√	√	√	√	√	9
PAX	√	√	√	√	√	√	9

M = Molar concentration, ppm = parts per million

4.2 Ageing of the Nkomati Ore

The microflotation testwork was performed over a period of several months as a series of mini-investigations (effect of different collectors, effect of different depressants, effect of oxidation, etc). Consequently, it was decided to check whether the Nkomati ore suffered from the effects associated with ageing during the course of the study. Six replicate tests at the original base test condition (pH 9 and no oxidation) were conducted on a sample that had been in the freezer for 12 months. Six replicate tests were also performed at pH 10 with 5 min oxidation (see Section 4.7) using a sample that was allowed to age for 1 month.

The mass recovery data (Figure 4.1) shows that the test carried out at pH 9 after 12 months of sample ageing differed significantly from the original test, and achieved a mass recovery of only 64% after 20 min whereas the original test achieved 78%. This shows a 14% drop in recovery from the original test, which is significant. The pH 10 test results on the other hand, were much closer together, with the original test achieving a mass recovery of 57% and the test done a month later achieving 60%, only deviating from each other by ~ 3% mass recovery.

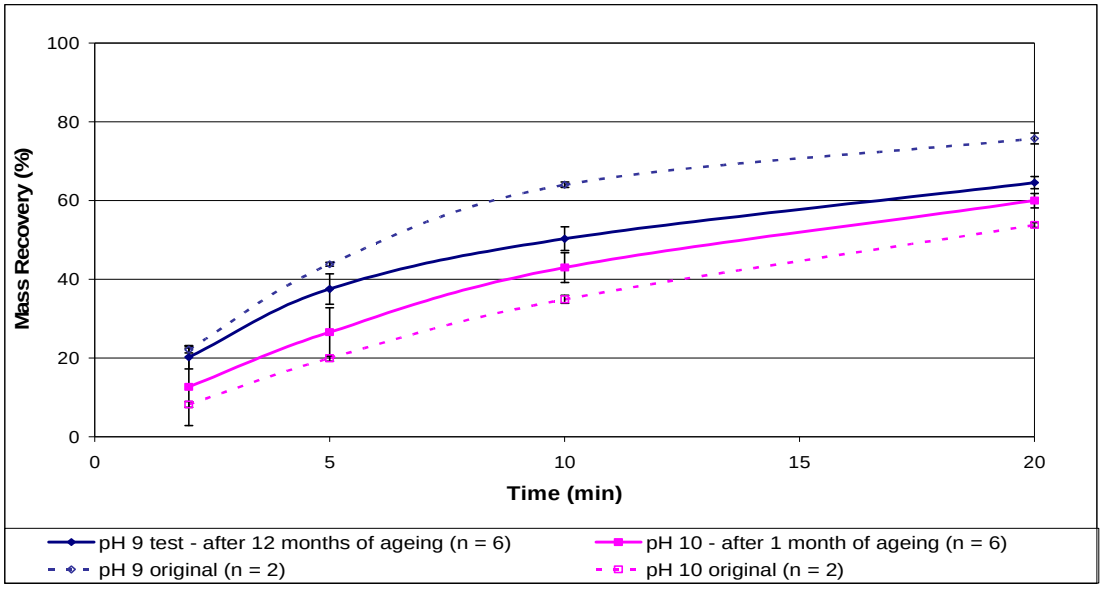


Figure 4.1: Graph of flotation kinetics curves for microflotation tests with Nkomati ore at pH 9 and pH 10, comparing the effect of ageing of the sample at short (1 month) and long (12 months) exposure time. All collector concentrations were $1x10^{-5}M$

From these results, it is evident that the sample deteriorates if it is allowed to age for too long. This was particularly evident for the pH 9 test which was allowed to age for 12 months (compared to the pH 10 test which only aged for one month). It is clear then that samples must be used immediately or within a short period of time if results are to be compared, as results may change after a long period of time has elapsed.

When the selectivity data for these tests are compared (Figure 4.2), the base test condition (pH 9) shows the worst deterioration, as for the kinetic data (Figure 4.1). The pH 10 condition shows a much better agreement between the initial test and after the sample had aged for 1 month, which is consistent with the observations seen in Figure 4.1.

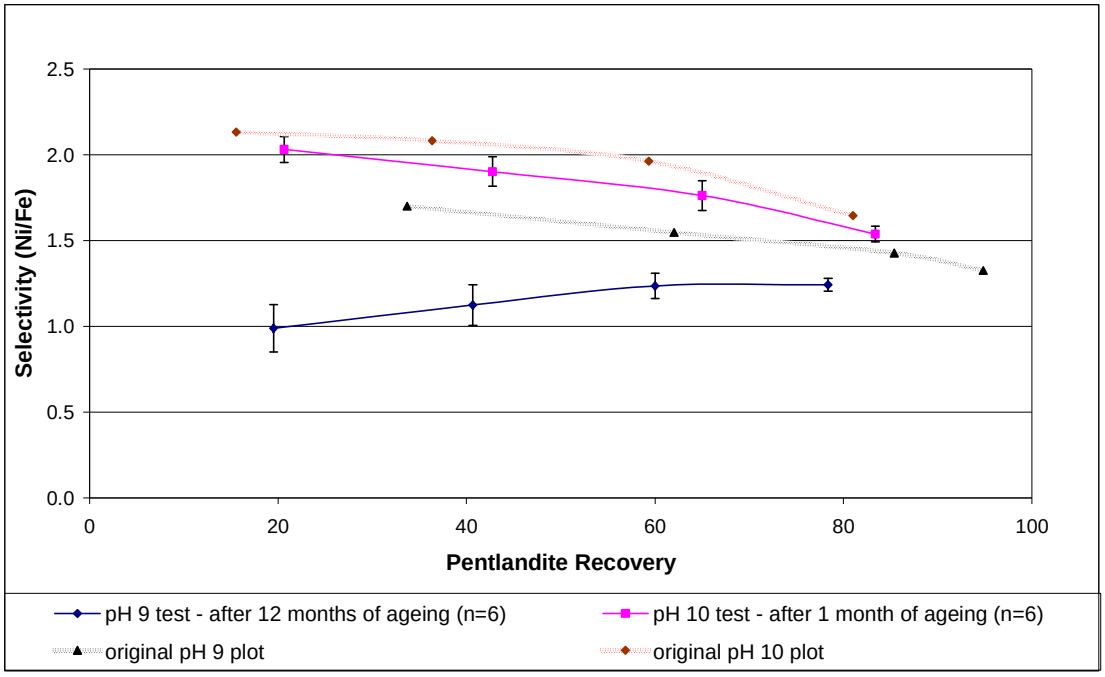


Figure 4.2: Selectivity versus pentlandite recovery plots for microflotation tests with Nkomati ore at pH 9 and pH 10, comparing the effect of ageing of the sample at short (1 month) and long (12 months) exposure time. All collector concentrations were $1x10^{-5}$M

Based on these results, it can be seen that care needs to be taken when comparing the results of tests carried out several months apart. This, however, does not disqualify the integrity of data obtained over a short period (i.e. in the mini investigations), but does

show that after a long time has elapsed (greater than a month) the data may need to be reassessed as it would likely have changed as shown in Figure 4.2.

The only treatment that was undertaken to minimise or try to reverse oxidation with time was ultrasonic bath treatment. Mild acid wash treatment would have been interesting to investigate as a method of reversing the oxidation effect on the samples, but this was however beyond the scope of this thesis. The observation made after ultrasonic bath treatment was that there were some surface species that were removed from the surface of the minerals, as the water discarded after the treatment contained fine material which was not there before. Acid wash treatment was not attempted, though it may have been worthwhile to try this treatment and observe if this could have resulted in the reversing of the oxidation effects.

4.3 Comparison of different xanthate collectors (no depressant)

Figure 4.3 shows the cumulative mass recoveries obtained with the four different collectors when no depressant was used. SEX, SNPX, SIBX and PAX are compared against a ‘no reagent’ test. From Figure 4.3 it can be seen that for all cases the final mass recovery achieved was above 80%. The rate of flotation was greatest for the longest chained xanthates, PAX and SIBX, which produced a mass recovery of ~ 85% in about 3 min of flotation.

As the xanthate chain length decreases from SIBX to SNPX to SEX, the flotation kinetics show a corresponding decrease, with ~ 82% mass recovered obtained after 5 min for SNPX and ~ 80% mass recovered after 6 min for SEX The slowest kinetics were obtained in the ‘no reagent’ test. From this it may be concluded that although the xanthate collector chain length has little effect on the final recovery achieved, it has a significant impact on the flotation kinetics, especially in the initial stages of each flotation test.

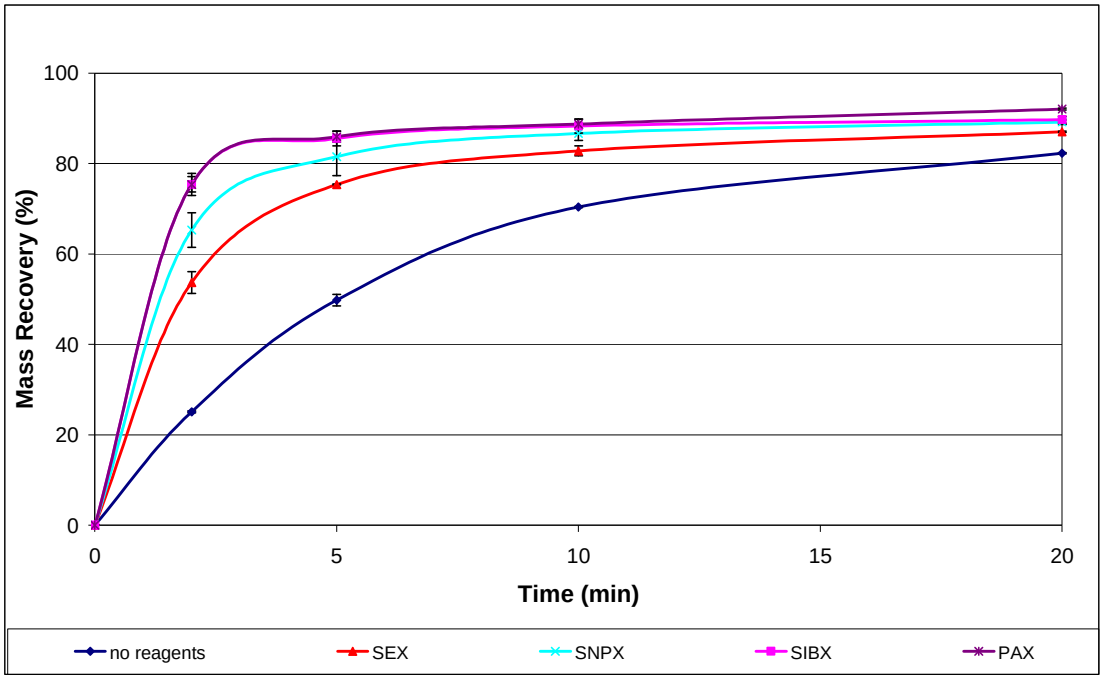


Figure 4.3: Mass recovery versus time for microflotation tests with Nkomati ore at pH 9, comparing the performance of xanthate collectors of different chain length in the absence of depressant. All collector concentrations were at $1x10^{-5}$M

Simulation of the data shown on Figure 4.3 of the recovery versus time was also performed by applying the Klimpel and Kelsall. The Klimpel Model is of the form:

$$R = R_u\left[1 - \frac{1-\exp(-k_{\max}t)}{k_{\max}t}\right]$$

Where R is the recovery at time t, R_u is the ultimate recovery for the given species for an infinite flotation time, $k_{\max}$ is a constant and t is time.

The Kelsall Model characterises the mineral species into two rate constants corresponding to slow and fast floating components. The model is of the form:

$$R = \Phi_s[1-\exp(-k_s t)] + (1-\Phi)[1-\exp(-k_f t)]$$

Where R is the recovery at time t, Φ_s is the mass fraction of the slow floating component and k_s and k_f are the rate constants for the slow and fast floating components respectively.

These equations were fitted to the measured recovery data using the Solver Routine in Microsoft Excel. The Klimpel and Kelsall parameters are summarised in table 4.2 below.

Table 4.2: Klimpel and Kelsall parameters

		Φ_s	k_f	k_s	R	k_{max}
No reagent	Ni	0.10	0.13	0.13	100.00	0.53
	Fe	0.10	0.12	0.11	100.00	0.47
SEX	Ni	0.11	0.59	0.09	100.00	2.58
	Fe	0.25	0.57	0.03	86.65	2.24
SNPX	Ni	0.13	0.76	0.11	100.00	3.23
	Fe	0.21	0.75	0.03	88.42	3.22
SIBX	Ni	0.07	1.03	0.10	100.00	5.76
	Fe	0.17	1.03	0.02	90.34	4.91
PAX	Ni	0.08	1.06	0.08	99.40	5.83
	Fe	0.19	1.06	0.04	90.34	4.91

From this data, it is clearly seen that as the xanthate chain length was increased, the kinetics of flotation also increased as indicated by the increasing values of 'k_{max}' and 'k_f' in Table 4.2.

Figure 4.4 shows the pentlandite and pyrrhotite recoveries obtained in these same tests after 5 min and 20 min of flotation. In all the tests, except for the 'no reagent' test, the pentlandite recovery obtained after 5 min of flotation was greater than 80% which suggests that most of the pentlandite that is present in this sample is fast floating. For pyrrhotite, only when SIBX and PAX were used were the recoveries obtained after 5 min greater than 80%. The fact that all the final recoveries, after 20 min, were greater than 80%, even for the test done with no reagents, suggests that this is a readily floatable ore sample.

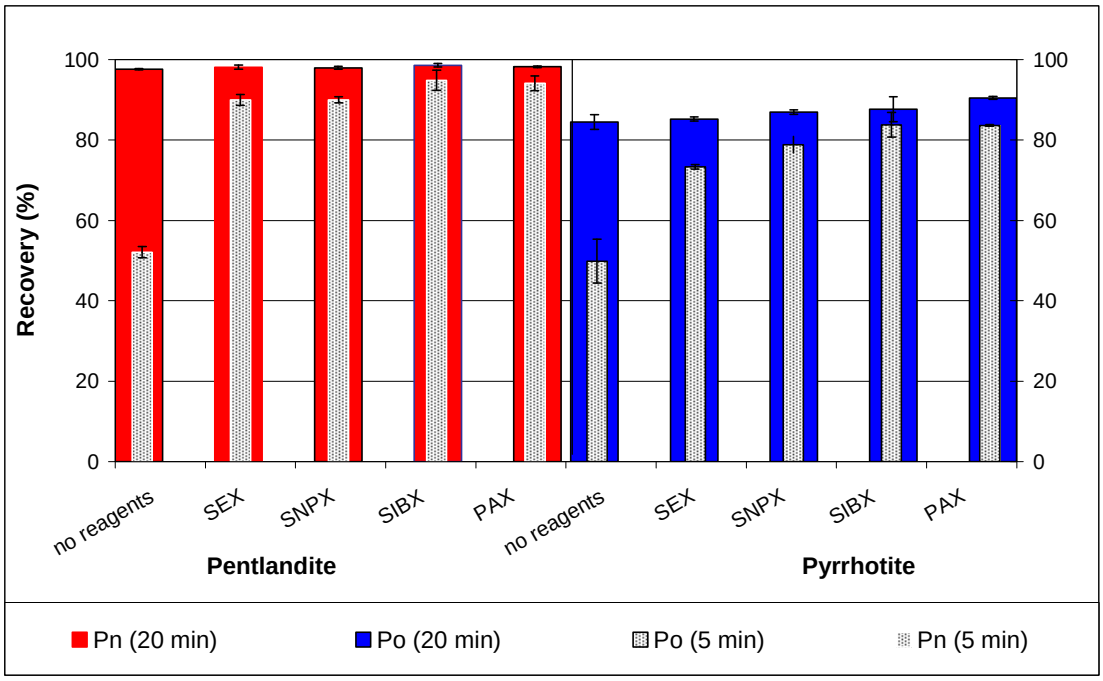


Figure 4.4: Recovery of pentlandite and pyrrhotite in microflotation tests with Nkomati ore at pH 9, comparing the performance of xanthate collectors of different chain length in the absence of depressant. All collector concentrations were $1x10^{-5}$M

The Ni/Fe ratios plotted in Figure 4.5 for the different chain length xanthate collectors all lie between 1.0 and 1.2. As indicated in section 4.1 above, these ratios represent poor selectivity. This is because at these conditions there is no meaningful separation of pentlandite and pyrrhotite and the two minerals both float well. As was shown in Figure 4.4, both pentlandite and pyrrhotite recoveries were greater than 80% in these tests, after 20 min of microflotation. Nevertheless, selectivity with SEX is clearly the best (even though it is low).

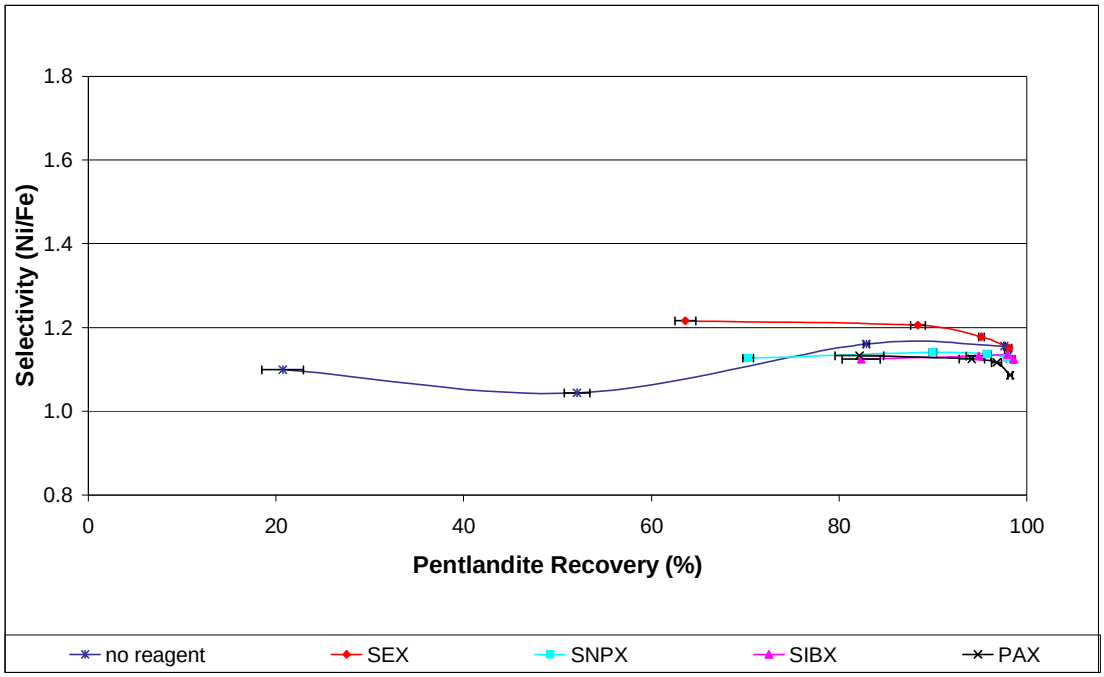


Figure 4.5: Ni/Fe ratio versus pentlandite recovery for microflotation tests with Nkomati ore at pH 9, comparing the performance of xanthate collectors of different chain length in the absence of depressant. All collector concentrations were $1x10^{-5}$M

4.4 Comparison of the different depressants

Figure 4.6 shows the cumulative mass recoveries obtained in the microflotation tests with the three polymeric depressants when no collector was used. Guar and starch appear to be the most effective depressants with mass recoveries reduced to 26% and 30%, respectively after 20 minutes, from > 80% in the test with no reagent. CMC is the weakest depressant of this sulphide sample, as shown by the high mass recovery of 60%, after 20 minutes of flotation time.

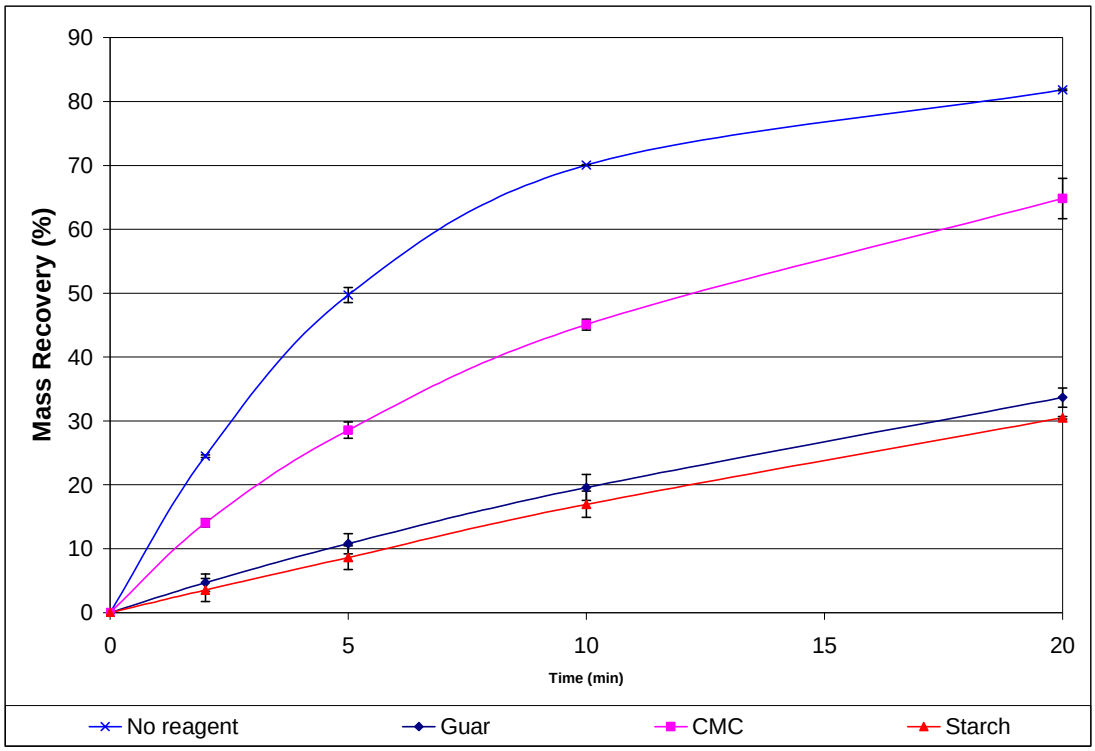


Figure 4.6: Mass recovery versus time for microflotation tests with Nkomati ore at pH 9, comparing the performance of three polymeric depressants in the absence of collector.

Figure 4.7 shows the pentlandite and pyrrhotite recoveries obtained in the same tests as in Figure 4.6, after 5 min and 20 min of flotation. The results are also compared against the test done with no reagents. All three depressants depressed both pentlandite and pyrrhotite to some degree, with CMC being the least effective. Guar gum and starch are seen to depress the sulphides strongly with final recoveries lower than 50% for both pentlandite and pyrrhotite, whereas when CMC was used the final recovery obtained was greater than 60% for both pentlandite and pyrrhotite. All the depressants slowed down the flotation of the sulphides in the first few minutes of flotation.

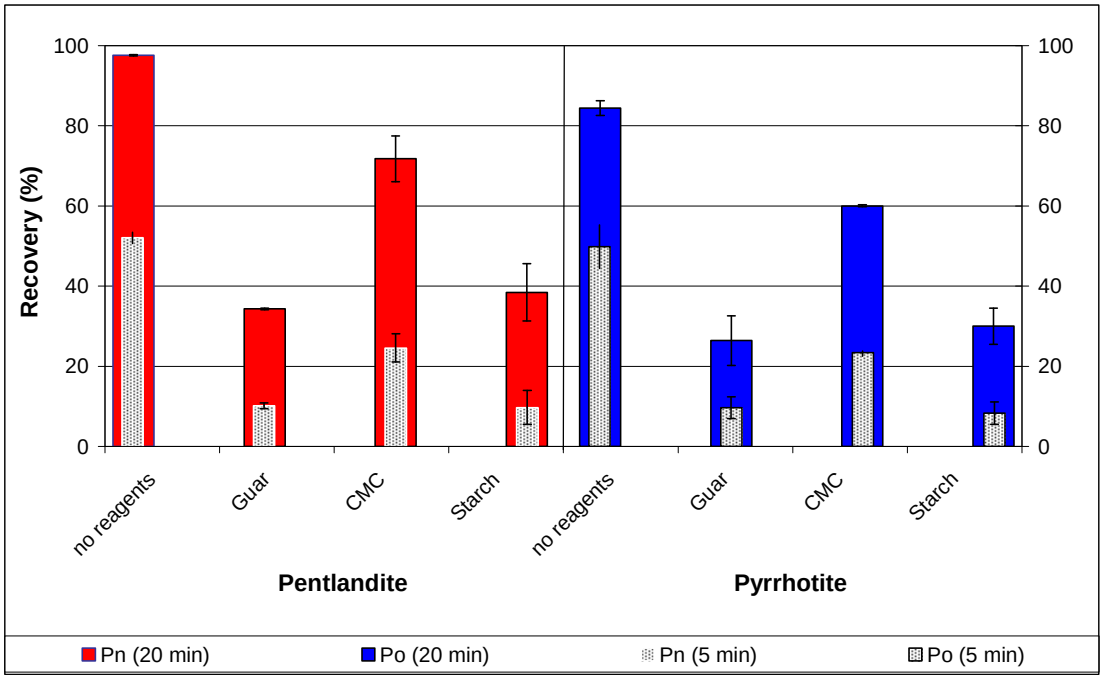


Figure 4.7: Recovery of pentlandite and pyrrhotite in microflotation tests with Nkomati ore at pH 9, comparing the performance of three polymeric depressants in the absence of collector. All depressant concentrations were 10 ppm

The Ni/Fe ratios for the different polymeric depressants are plotted in Figure 4.8. The CMC and the 'no reagent' test have the lowest ratios, between 1.0 and 1.2, indicating poor selectivity. Tests with guar and starch have ratios with a wider range, between 0.9 and 1.3. For all these depressants, however, none of the Ni/Fe ratios are within the desired range, as was the case for the collectors. The collectors allowed for good flotation of both pentlandite and pyrrhotite, whereas the depressants are seen to depress both the minerals equally, with guar and starch imparting stronger depression than CMC.

The next section investigates the joint action of collector and depressant in the microflotation of the Nkomati massive sulphide ore.

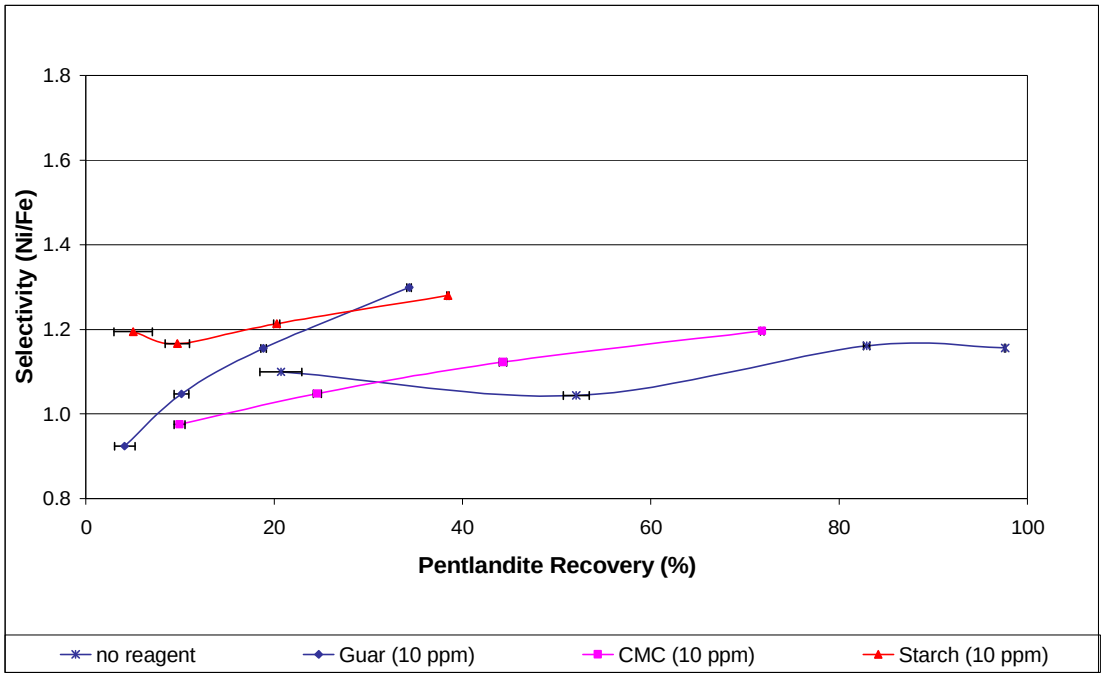


Figure 4.8: Ni/Fe ratio versus pentlandite recovery for microflotation tests with Nkomati ore at pH 9, comparing the performance of three polymeric depressants in the absence of collector.

4.5 Comparison of different depressants in the presence of collectors

In this section, each of the three depressants tested previously (Section 4.4) was evaluated in conjunction with each of the four different chain length xanthate collectors. The objective of this was to investigate whether the collector-depressant combinations gave better pentlandite/pyrrhotite flotation selectivity, and, if so, which combination gave the best. The results from the microflotation tests are presented in Figure 4.9, which shows the Ni/Fe ratios versus pentlandite recoveries obtained for all combinations of the polysaccharide depressants and the different xanthate collectors (see Table 4.1). Each line in Figure 4.9 represents a test with four timed concentrates collected over a total time of 20 min. The target Ni/Fe ratio for pentlandite/pyrrhotite flotation selectivity is ≥ 1.3

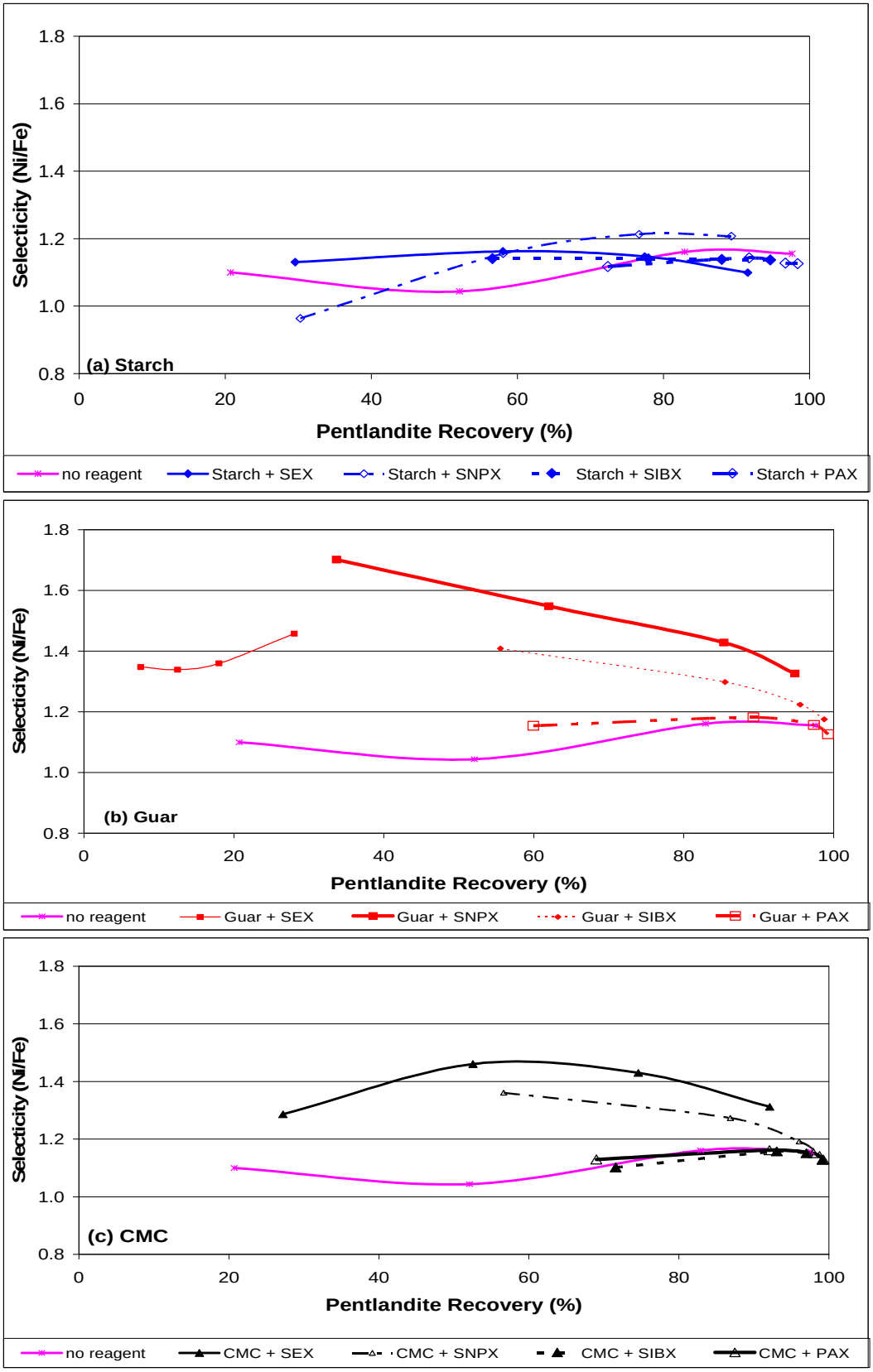


Figure 4.9: Ni/Fe ratio versus pentlandite recovery for microflotation tests with Nkomati ore at pH 9, comparing the performance of the four xanthate collectors of different chain lengths used in conjunction with the three polymeric depressants. All collector concentrations were $1x10^{-5}$M; all depressant dosages were 10 ppm.

Figure 4.9 shows that the combination that produced the best selectivity was SNPX used in conjunction with guar, with Ni/Fe ratios ranging from 1.7 to 1.3 and pentlandite recoveries ranging from 38% to 92%. The test performed with guar and SEX showed the lowest recovery, with pentlandite recovery of ~ 30% whereas the average recovery obtained for all the other tests was ≥ 90%. Figure 4.10 shows the Ni/Fe ratios versus pentlandite recoveries for SNPX in conjunction with all of the polymeric depressants tested. It can be noted from Figure 4.10 that the selectivity index for starch and SNPX is low in the beginning of the test. This is probably due to higher rate of pyrrhotite recovery which was not expected to happen. This is not what was expected and further testwork, which is outside the scope of this thesis, would need to be conducted to investigate why this was the case. It can be seen that the best combination to use if good selectivity is desired (i.e. high Ni/Fe ratio) is SNPX and guar. If higher recoveries are desired the best combination to use is SNPX and CMC. For this study, higher pentlandite selectivity was desired, while maintaining good recovery and the SNPX-guar combination achieved this with Ni/Fe ratio of ≥ 1.3 and pentlandite recovery of ≥ 90%.

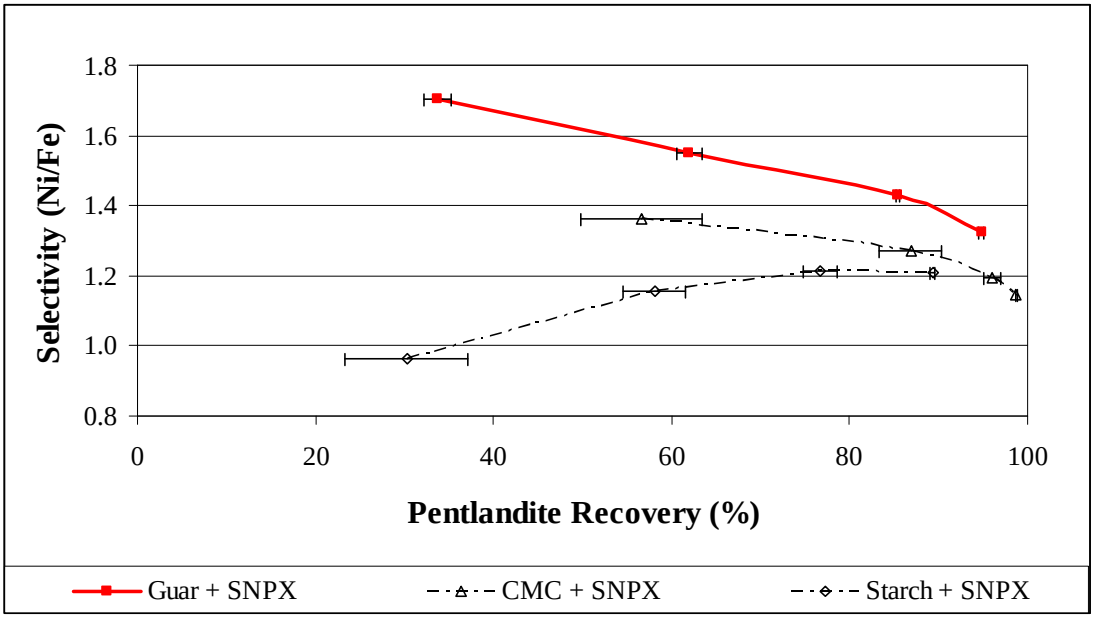


Figure 4.10: Ni/Fe ratio versus pentlandite recovery obtained in microflotation tests obtained with Nkomati ore at pH 9 comparing the performance of SNPX in the presence of the three polymeric depressants. All depressant dosages were 10 ppm

4.6 The effect of oxygen addition

The next set of microflotation tests was performed to investigate the effect of oxygen addition on the flotation kinetics and pentlandite/pyrrhotite selectivity for Nkomati massive sulphide ore. The best reagent combination determined in Sections 4.3 to 4.5 above was used, i.e. SNPX collector (at $1x10^{-5}$ M) and guar depressant (at 10 ppm) at pH 9 adjusted using NaOH.

Figure 4.11 shows the cumulative mass recoveries obtained in the microflotation tests after oxygen addition for various times in the conditioning stage. For all the tests, the final mass recovery achieved reached was ≥ 80%, except for the test done with no oxidation. It can be seen that an increase in oxidation time led to an increase in recovery and flotation rate.

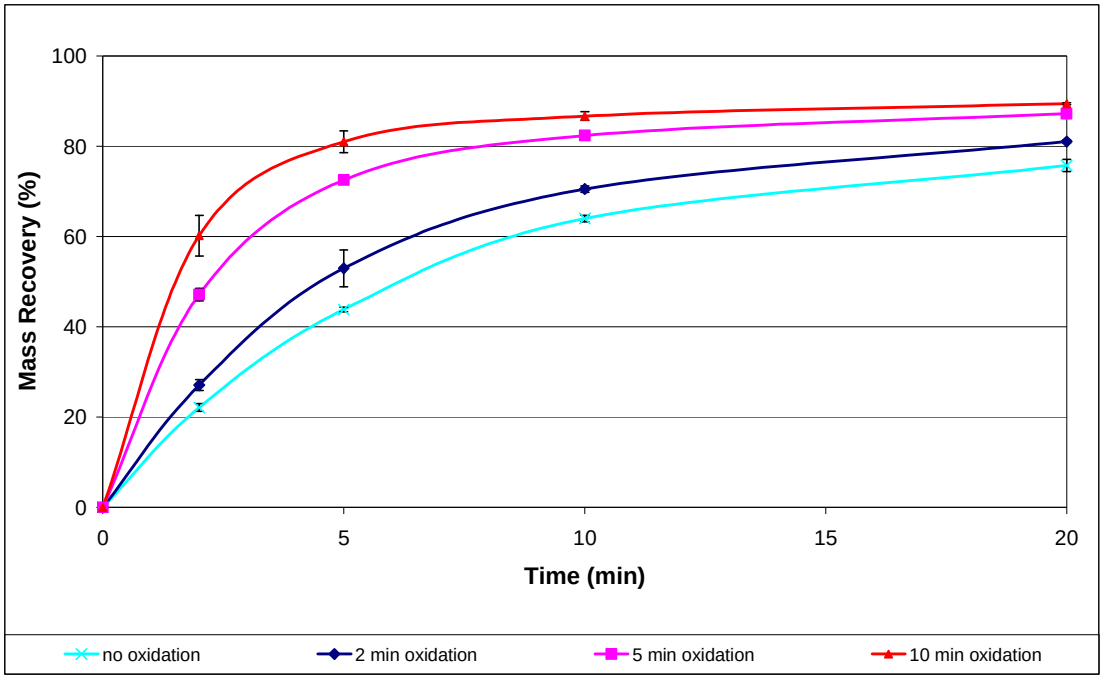


Figure 4.11: Mass recovery versus time for microflotation tests with Nkomati ore at pH 9, comparing the performance after various pulp oxidation times. Collector = $1x10^{-5}$M SNPX; depressant = 10 ppm guar.

Figure 4.12 shows the recoveries of pentlandite and pyrrhotite obtained in these tests after 5 and 20 min of flotation. Clearly the test done at no oxidation is still the best test under these conditions, in terms of pentlandite/pyrrhotite selectivity. With no oxidation, the pentlandite recovery obtained after 5 min was > 60% while the

corresponding pyrrhotite recovery was ~ 40%. After 20 min of flotation, the pentlandite recovery, with no oxidation, was > 90% while the corresponding pyrrhotite recovery was ~ 75%. As oxidation time was increased, recovery of both pentlandite and pyrrhotite increased, which is undesired as this consequently leads to a decrease in selectivity.

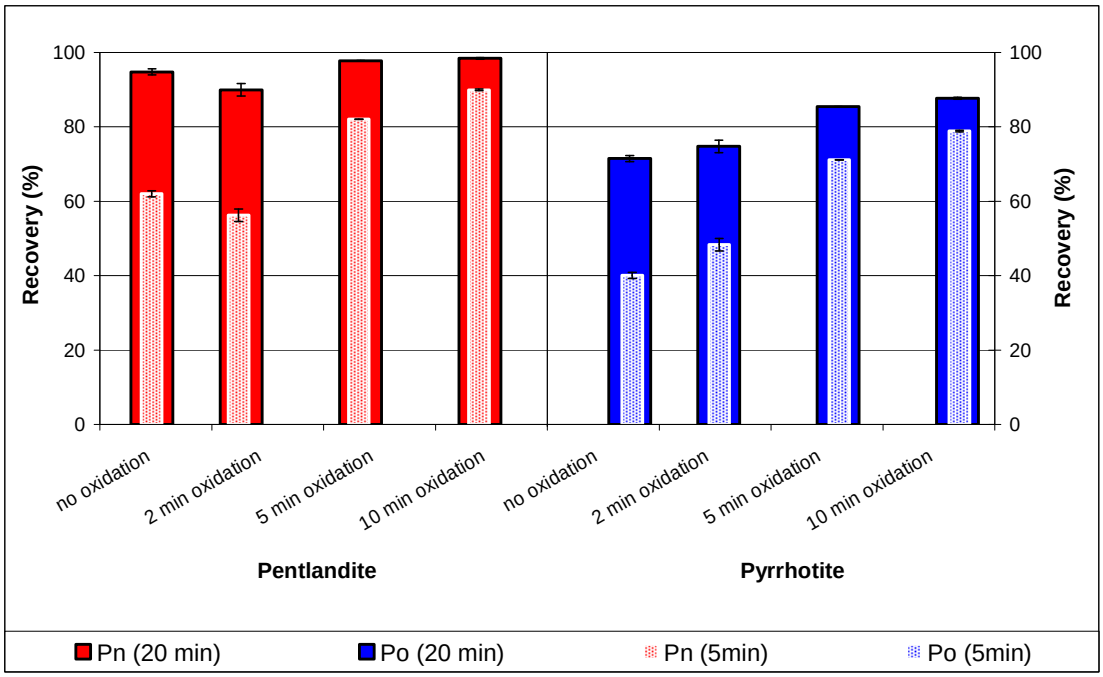


Figure 4.12: Recovery of pentlandite and pyrrhotite in microflotation tests with Nkomati ore at pH 9 comparing the performance after various pulp oxidation times. Collector = $1x10^{-5}$M SNPX; depressant = 10 ppm guar.

This is shown more clearly in Figure 4.13, which presents the Ni/Fe ratio versus pentlandite recovery obtained in these tests. It can be seen that the results obtained at no oxidation are far better than in the other tests, in which oxidation was carried out. The Ni/Fe ratios obtained when oxidation was carried out range between 1.0 and 1.2, which represents poor selectivity. The test done with no oxidation, however, resulted in Ni/Fe ratios between 1.7 and 1.3.

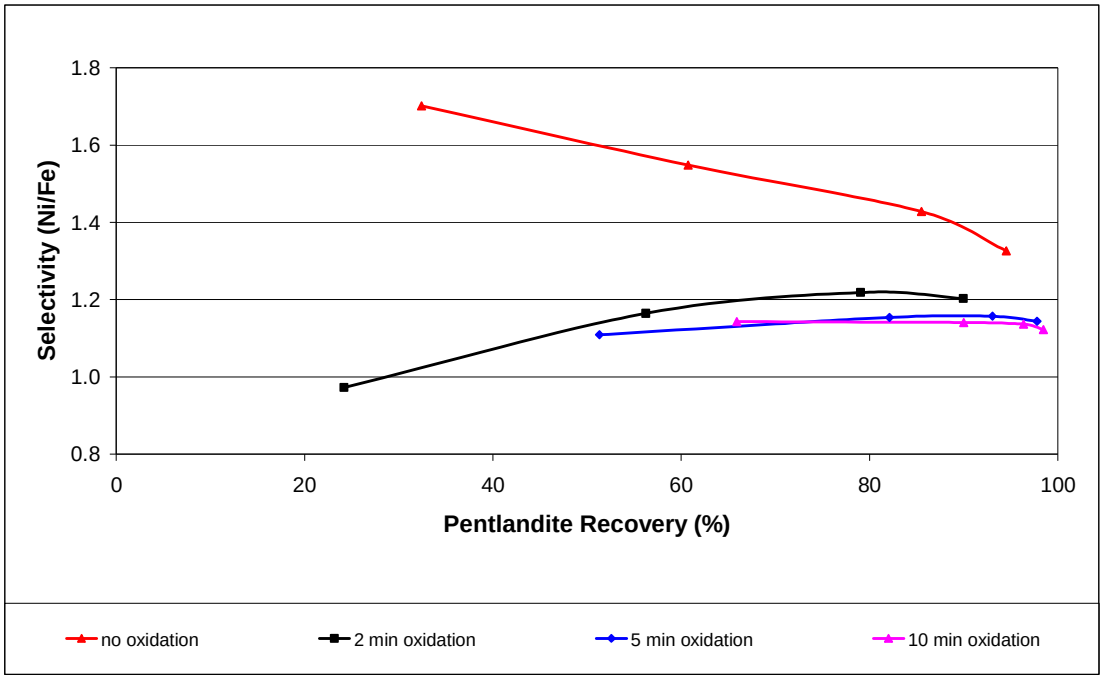


Figure 4.13: Ni/Fe ratio versus pentlandite recovery for microflotation tests with Nkomati ore at pH 9, comparing the performance after various pulp oxidation times. Collector = $1x10^{-5}$M SNPX; depressant = 10 ppm guar.

The oxidation conditions tested here, i.e. sparging the pulp with medical oxygen prior to microflotation, clearly favour the flotation of both pentlandite and pyrrhotite. It is also known that given enough oxidation, depression of these sulphides minerals can be achieved. Based on this it was decided to use a different method of oxidation, by varying pH in conjunction with oxidation (5 min). This is described in the following section.

4.7 The effect of pH and pH modifier

The effect of pH was investigated in the microflotation test, by varying the pH in increments of 0.5 between pH 9 and pH 10.5, using 10^{-5} M SNPX and 10 ppm guar as standard reagents. The effect of pH modifier was investigated at the same time, by using two different pH modifiers, viz. lime ($Ca(OH)_2$) and NaOH. The results are presented in Figure 4.14 and Figure 4.15.

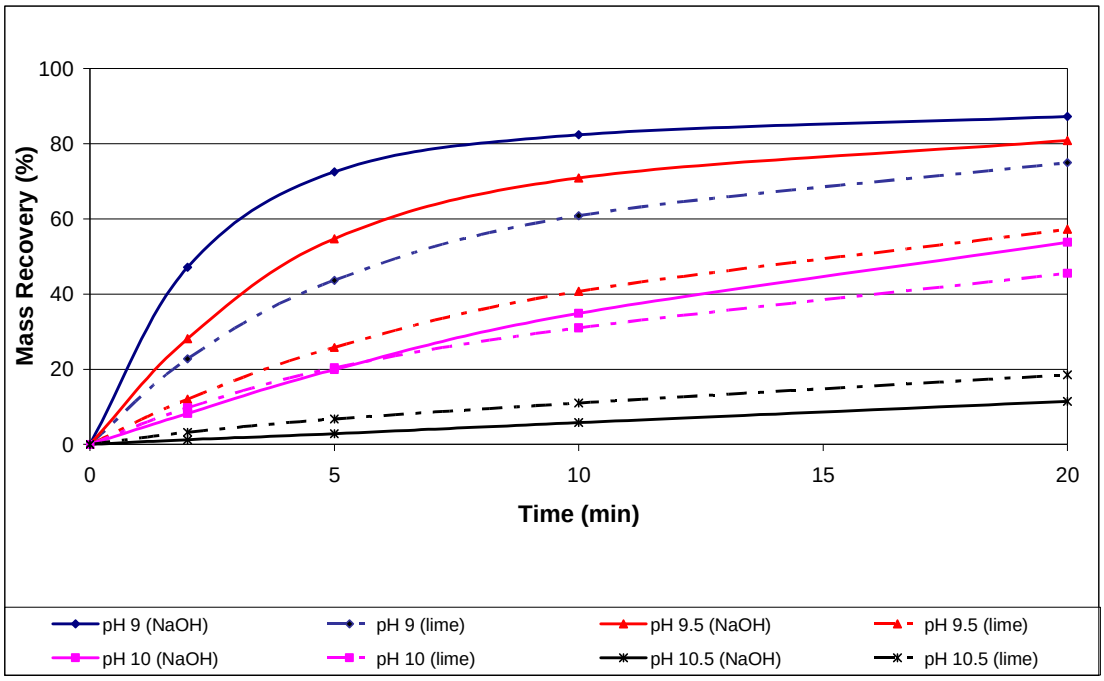


Figure 4.14: Mass recovery versus time for microflotation tests with Nkomati ore, comparing the performance at various pH values, using different pH modifiers. Collector = $1x10^{-5}$M SNPX; depressant = 10 ppm guar, oxidation = 5 min

Figure 4.14 shows that the mass recoveries obtained when using lime as a pH modifier were uniformly less than those obtained when NaOH was used, at pH values between 9 and 10. At pH 9, the final mass recovery obtained when using NaOH was 85%, whereas when lime was used, 78% recovery was obtained. At pH 9.5, when NaOH was used, 80% mass recovery was obtained whereas when lime was used, only 58% mass recovery was obtained. At pH 10, 57% mass recovery was obtained when using NaOH, while 53% mass recovery was obtained when lime was used. At pH 10.5 the trend is reversed, with greater mass recovery obtained when using lime than when using NaOH. Figure 4.14 also shows that an increase in pH results in the decrease of both the flotation kinetics and cumulative mass recoveries obtained at all flotation times.

Figure 4.15 shows the pentlandite recoveries obtained in these tests, after 5 and 20 min of flotation. The graph shows clearly that as pH was increased, there was a corresponding decrease in pentlandite recovery, irrespective of the pH modifier used.

The tests done when using lime as a pH modifier showed more sensitivity than the tests done with NaOH and resulted in higher losses in pentlandite recovery.

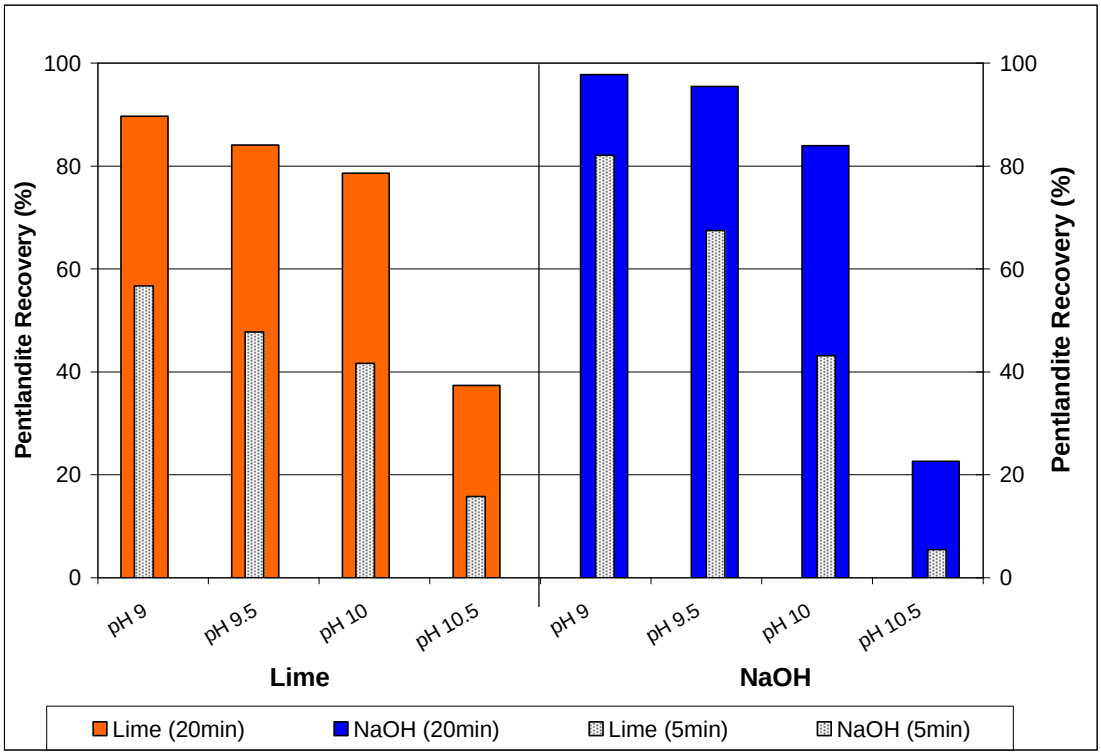


Figure 4.15: Recovery of pentlandite in microflotation tests with Nkomati ore, comparing the performance at various pH values, using different pH modifiers. Collector = $1x10^{-5}$M SNPX; depressant = 10 ppm guar; oxidation = 5 min

Figure 4.16 shows the Ni/Fe ratios obtained in these tests. The dramatic effect of pH on the Ni/Fe ratio obtained is evident. This suggests that oxidation carried out in conjunction with pH modification is much more selective in terms of which mineral is affected first. The good Ni/Fe ratios seen in Figure 4.16 suggest that pyrrhotite was more severely affected than pentlandite. Comparing lime with NaOH as pH modifier, it can also be seen that lime was more favourable than NaOH for achieving the desired pentlandite/pyrrhotite selectivity ratios.

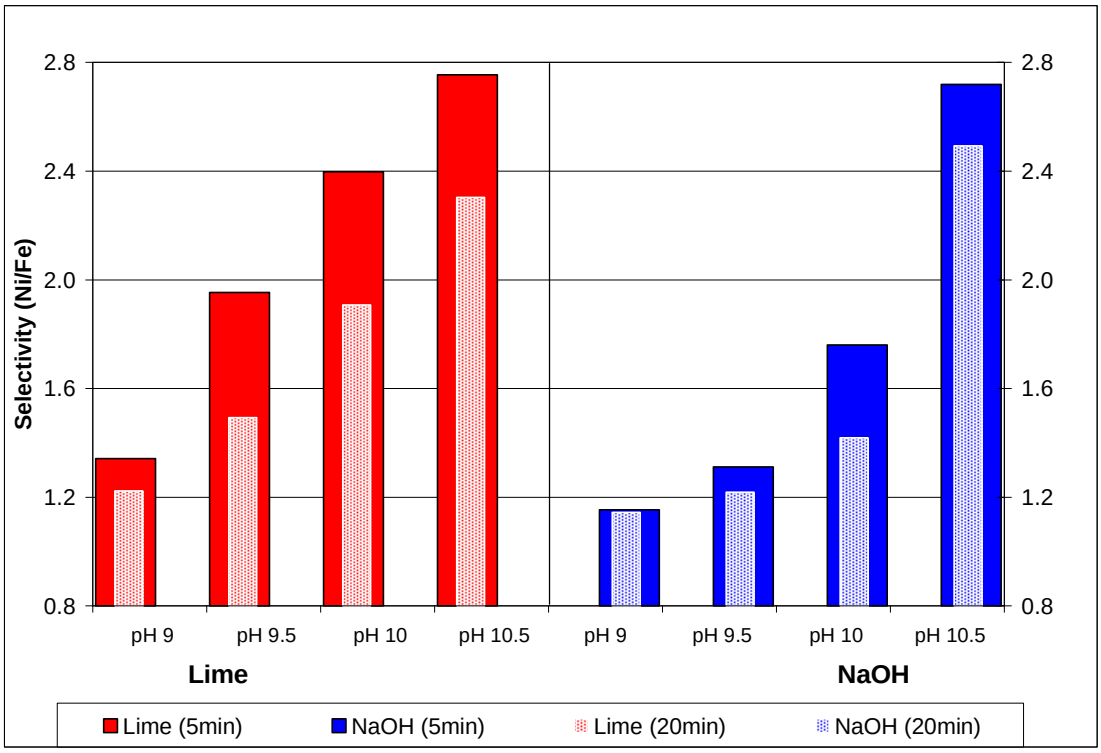


Figure 4.16: Ni/Fe ratio obtained in microflotation tests with Nkomati ore, comparing the performance at various pH values, using different pH modifiers.
Collector = $1x10^{-5}$M SNPX; depressant = 10 ppm guar, oxidation = 5 min

Figure 4.17 shows the Ni/Fe ratio versus pentlandite recovery obtained in the same tests. From Figure 4.17 it can be seen that with the improved Ni/Fe ratios came a penalty of slightly lower pentlandite recoveries. As pH was increased from pH 9 to pH 10, the Ni/Fe ratio improved from 1.2 to 1.9 when lime was used and from 1.2 to 1.4 when NaOH was used. The corresponding recoveries dropped from 90 to 79% for lime, and from 96 to 84% for NaOH. This means that there is a trade off between grade and recovery. A balance would need to be struck when deciding where to operate, so as to minimise pentlandite losses while achieving the best selectivities possible.

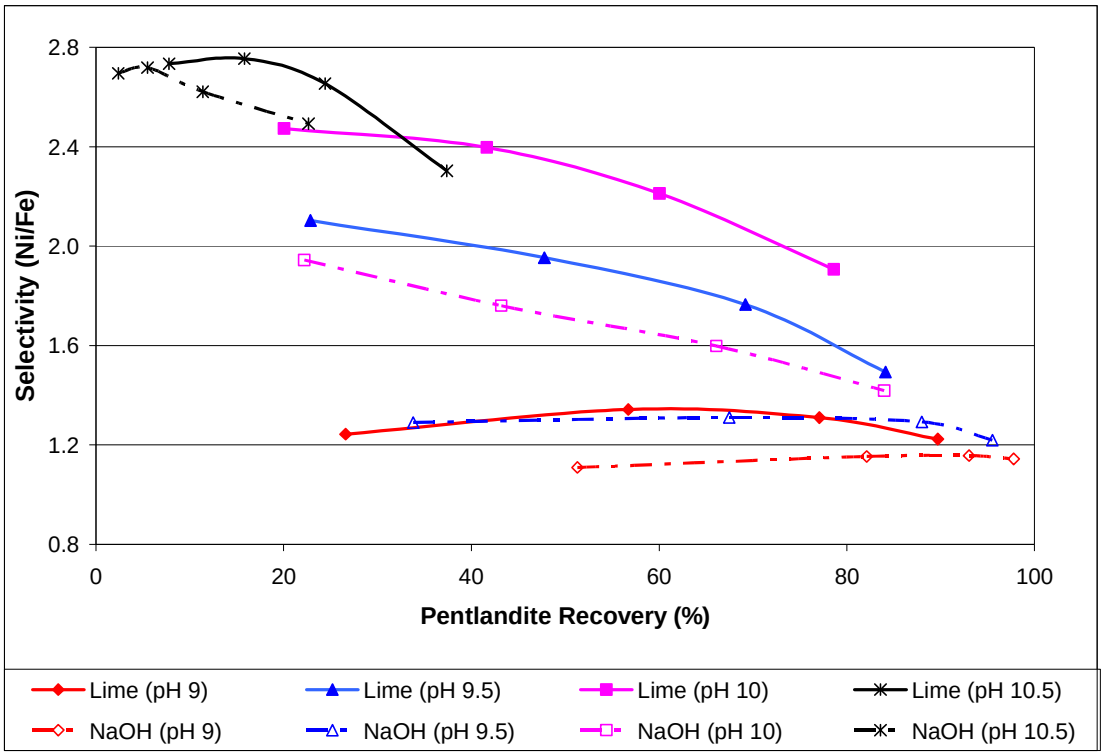


Figure 4.17: Ni/Fe ratio versus pentlandite recovery for microflotation tests with Nkomati ore, comparing the performance at various pH values, using different pH modifiers. Collector = $1x10^{-5}$M SNPX; depressant = 10 ppm guar, oxidation = 5 min

4.8 The effect of dissolved oxygen level

The dissolved oxygen (D.O.) content was measured in the tests described in the previous section in order to determine the amount of oxygen that was present in solution under the different pH conditions tested. The measurements were only done on samples sparged for 0 min and 5min. It would have been interesting to measure D.O. levels at varying sparging times but this was not done and beyond the scope of this thesis. Figure 4.18 shows that the tests in which oxidation was applied resulted in higher dissolved oxygen levels in the pulp, as was expected.

The other observation that can be made is that at the same pH, the use of lime resulted in much higher D.O. levels than the use of NaOH. When pH was adjusted to 10 using lime, the D.O. level peaked at 15.8 ppm, whereas when pH was adjusted to 10 using NaOH, D.O. level peaked at 12.2 ppm. The same trend is seen when pH was adjusted to 9. When using lime, the D.O. level peaked at 16 ppm, whereas when pH was adjusted to 9 using NaOH, D.O. level peaked at 9.8 ppm. This means that when lime

was used, more oxygen was retained in solution than when NaOH was used, thus allowing for a greater extent of oxidation to take place. When no oxidation was applied, the D.O. levels remained at ~ 8 ppm for both lime and NaOH pH adjusted conditions. The possible reason for higher DO levels when lime was used as opposed to the lower levels when NaOH was used is possibly due to the molecule structures. Lime dissociates to Ca^{2+} and O^{2-} while NaOH would dissociate to Na^{+} and OH^{-}. It is clear that when lime dissociates, it adds more dissolved oxygen to the pulp as opposed to NaOH which adds the OH ions to the pulp. This would explain why lime resulted in higher DO levels than NaOH.

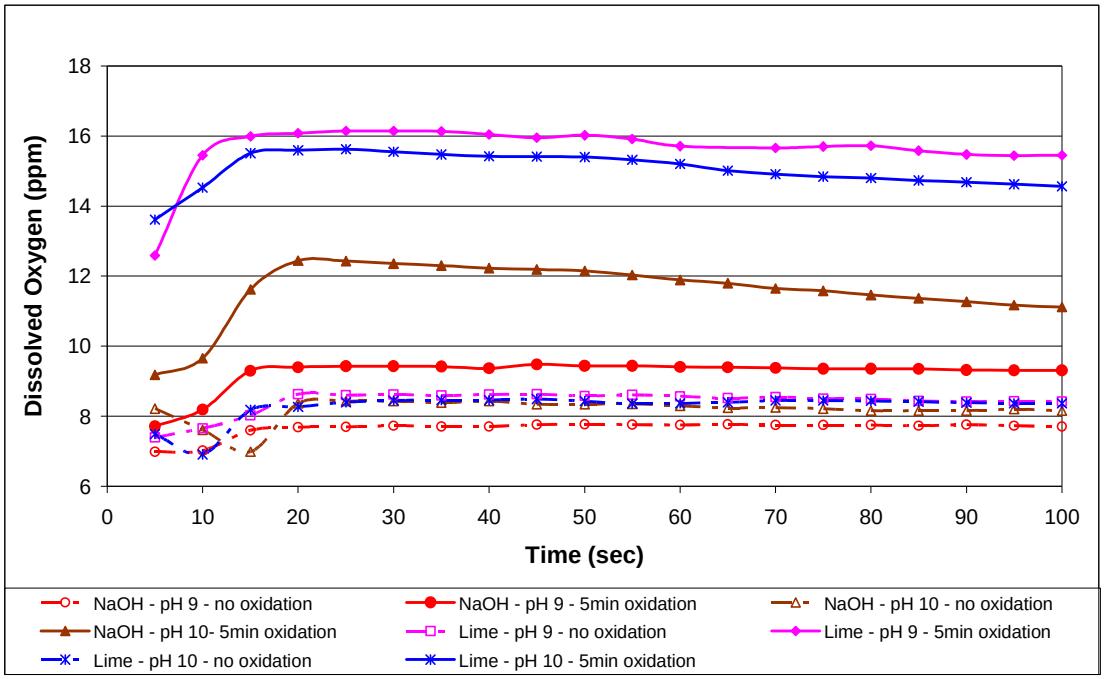


Figure 4.18: Dissolved oxygen levels obtained in the microflotation tests with Nkomati ore when using lime and NaOH as pH modifiers.

From the results above, it is clear that to achieve pentlandite selectivity over pyrrhotite, the optimum conditions for use are guar and SNPX operated at a pH of 10 with lime as the pH modifier and with oxidation (by sparging oxygen into the pulp).

4.9 Summary of results

i) Flotation using collectors in the absence of depressants

- Greater than 80% mass recovery was achieved for all collectors.

- Flotation kinetics increased as the chain length increased, i.e. PAX > SIBX > SNPX > SEX.
- High Ni/Fe ratio, and therefore pentlandite selectivity (> 1.3) was not obtained for any collectors used on their own, since good flotation of both pentlandite and pyrrhotite was achieved.

ii) Flotation using depressants in the absence of collectors

- CMC was (by far) the weakest sulphide depressant (highest mass recoveries were achieved in its presence) while the guar was the strongest (lowest mass recoveries were achieved in its presence).
- The lowest Ni/Fe ratios and hence pentlandite selectivities were achieved for all depressants in the absence of collector. This was due to the fact that they either depressed both pentlandite and pyrrhotite too much or they weakly depressed both minerals. Starch and guar depressed all the minerals strongly while CMC depressed all the minerals weakly.

iii) Flotation using both collectors and depressants

- When the depressants were tested in conjunction with the collectors, it was found that the guar worked best, in terms of good pyrrhotite depression, while maintaining good pentlandite recovery, when used with an intermediate chain length SNPX collector
- The guar/SNPX combination gave the highest Ni/Fe ratio, and hence good pentlandite selectivity, of between 1.3 and 1.7.

iv) Flotation with O_2 addition

- The addition of oxygen improved the recovery of both pentlandite and pyrrhotite and consequently reduced the Ni/Fe ratio obtained, and hence the selectivity obtained.

v) pH and pH modifier

- Lime proved to be a better pH modifier that NaOH, in the context of pyrrhotite depression. Mass recoveries obtained with NaOH as the pH modifier always lagged behind those obtained when using lime by a factor of 0.5 pH units, i.e. the

results achieved using lime were only achieved by NaOH at a pH value 0.5 higher than with lime

- The highest Ni/Fe ratio and hence pentlandite selectivity was found when operating at pH 10.5, using lime and 5 min oxygen sparging. At this high Ni/Fe ratio (~ 2.8), however, there were also high losses in pentlandite recovery (~ 38% pentlandite recovery). In order to limit these losses and still maintain a high Ni/Fe ratio it was better to operate at pH 10 instead where the Ni/Fe ratio obtained was 2.4 at ~ 79% pentlandite recovery.

Based on these microflotation results, the batch flotation tests described in the next chapter were conducted at pH 10 with lime using the ore of interest, Sheba's Ridge ore. As the ore was first milled under normal conditions (i.e. in the presence of air) the oxidation of the ore was omitted. It was inferred that the time that the ore would be exposed to air during milling would be sufficient to substitute for the oxidation that was done in the microflotation testwork.

Chapter 5: Batch flotation test results

5.1 Introduction

This chapter reports the results of the batch flotation tests conducted on Sheba's Ridge ore. The optimum reagent regime for pentlandite/pyrrhotite selectivity, as determined from the microflotation testwork using the Nkomati massive sulphide probe ore, was used as the basis of these tests. The batch flotation tests represent a scale up from the previous microflotation testwork, in terms of sample size (1 kg vs. 2 g) and are a closer representation of real operations, as silicate gangue minerals (e.g. talc) were present as well as a froth phase.

As with the microflotation testwork, pentlandite selectivity is of key importance. In this phase of work, pentlandite selectivity is measured in terms of solids mass recovery and nickel grade. A mass recovery of 1% was targeted as the Sheba's Ridge ore has about 2% sulphide content (refer to Table 3.2 in Chapter 3). The Sheba's ridge sample only contained 0.6% pentlandite and 0.2% chalcopyrite, implying that if all the pentlandite and chalcopyrite was recovered this will only amount to 0.8%. A 1% mass pull was then targeted as this would only account for the targeted minerals only. A nickel grade of greater than 9 wt % is taken to imply good pentlandite selectivity. Iron (Fe) can not be used as a measure of selectivity in this phase of the testwork as it is present in both the sulphides and gangue minerals, especially in the pyroxene which accounts for 21% of the total ore (refer to Section 3.2). Thus pentlandite/pyrrhotite selectivity will be inferred from overall Ni grade.

Microflotation tests on pure minerals may be considered to be closer to the cleaner stage in a full scale (industrial) or batch flotation test set-up. Therefore, the test conditions identified in the microflotation tests were tested in cleaner stages of the laboratory scale batch flotation tests (refer to Section 3.7 for flowsheet – rougher vs. cleaner). The configuration of the cleaner stages and details of the experimental procedures used were represented in Figure 3.5 and Table 3.8 and are reproduced below for convenience.

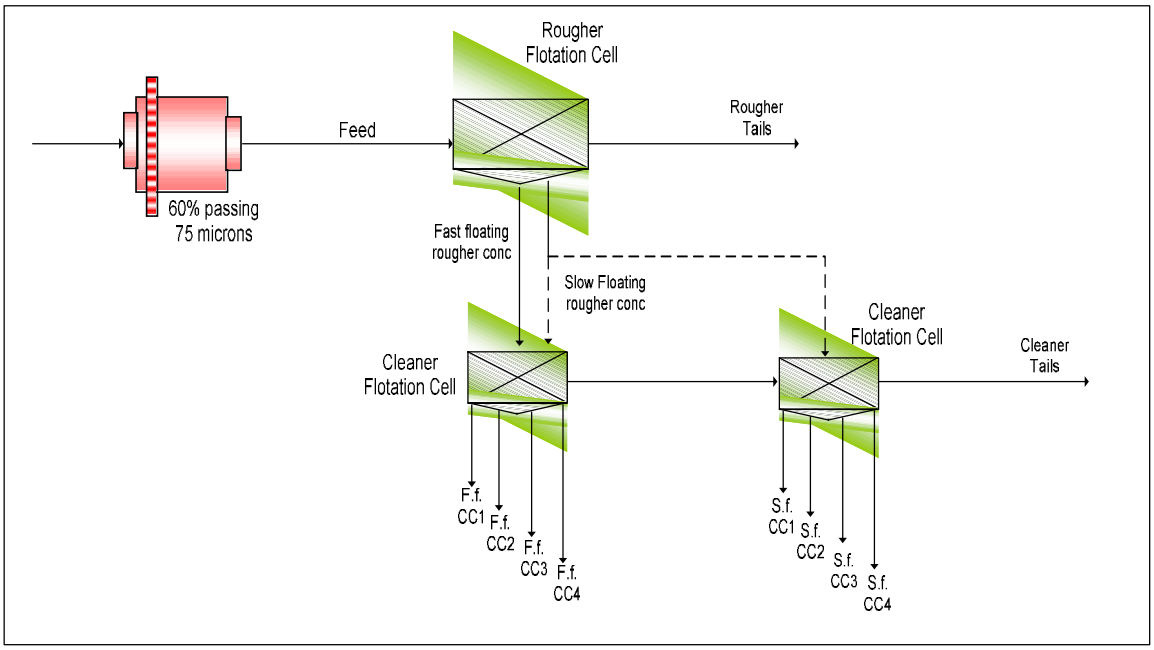


Figure 3.5: Configuration of laboratory batch flotation test

Table 3.8: Batch flotation test procedures

Rougher float				
Stage	**Reagent/Product**	**Dosage (g/t)**	**Conditioning/Float Time (min)**	**Cum Time (min)**
Mill	Collector	25	18	0
Float cell	Depressant	40	5	0 - 5
	Frother	40	1	5 - 6
Flotation	Fast floating fraction	-	10	6 - 16
	Slow floating fraction	-	20	16 - 36
Cleaner float				
Stage	**Reagent/Product**	**Dosage (g/t)**	**Conditioning/Float Time (min)**	**Cum Time (min)**
Float cell	pH modifier	adjust pH to 10.5*	as required	0
	Collector	10	2	0 - 2
	Depressant	10 – 100	5	2 - 7
	Frother	20	1	7 - 8
Flotation	Conc 1	-	2	8 - 10
	Conc 2	-	5	10 - 13
	Conc 3	-	10	13 - 18
	Conc 4	-	20	18 - 28

*Not all the cleaner tests were done at pH 10.5; some were done at natural pH of 9.5. For those tests the lime addition stage was skipped.

The rougher flotation conditions (reagent dosages and targeted grind) used were based on the recommended Mintek standard conditions (Dhliwayo *et al.*, 2007).

Microflotation tests on the Nkomati probe ore showed that SNPX and guar, at a pH raised to 10 with lime addition and 5 min of pre-oxidation, represent the best test conditions for pentlandite selectivity. It was also observed that of the three depressants tested, CMC was the weakest (see section 4.2). However, in other tests, both guar and CMC have proven to be effective in the depression of gangue minerals in ores from the BIC (Wiese *et al.*, 2006a; Beattie *et al.*, 2006; Wang *et .al.*, 2005; Parolis *et al.*, 2004; Steenberg and Harris, 1984). This is due to the fact that the ores from the BIC are dominated by silicate gangue minerals (especially talc) for which these organic depressants were specifically developed. Moreover, in addition to its depressant action, CMC has also been reported to be a slime cleaner of sulphides, resulting in better sulphide flotation performance (refer to section 2.3.4.2). Based on this, it was decided to conduct initial batch tests on Sheba's Ridge ore using both guar and CMC.

5.2 Statistical evaluation of the key results

When microflotation tests were conducted, the differences observed when different conditions were tested, were generally clear. However, for laboratory batch flotation tests this was not the case. Generally the differences observed in the batch flotation tests were small and a statistical analysis was done to check whether those differences observed were significantly different or not. This was done by conducting t-tests to verify this.

A t-test is used to determine whether two samples are likely to have come from the same underlying population. This implies that this test can be used to determine whether two data sets are statistically different or not. In this case it was used to check whether the results obtained from the different conditions tested were statistically different, or attributable to experimental error.

The results from the t-tests done on the key batch flotation tests are presented in Table 5.1 and Table 5.2.

Table 5.1: T-test of key batch flotation results, analysing recoveries, with √√√ = significant experimental difference with P ≤ 59%, √√ = experimental difference with P≤ 69%, √ = minimum experimental difference with P ≤ 79%, difference X= no experimental difference with P > 80%.

Referrence Section in the chapter	Depressant Type	Depressant dosage g/t (of rougher feed)	Collector Dosage (g/t) Rougher	Collector Dosage (g/t) Cleaner	Cleaner pH	Final Recovery Ni	Final Recovery S		Statistical t-test analysis of recoveries P	t	Standard error of difference	Statistical significance	Experimental significance in differences
Section 5.3	CMC	10	25	10	9.5	47.96	63.11	Ni	0.50	0.82	1.42	no	√√√
	Guar	10	25	10	9.5	46.90	63.69	S	0.45	0.93	1.54	no	√√√
	CMC	50	25	10	9.5	48.31	57.52	Ni	0.50	0.82	2.23	no	√√√
	Guar	50	25	10	9.5	46.61	61.15	S	0.59	0.63	1.97	no	√√√
	CMC	100	25	10	9.5	40.74	53.19	Ni	0.70	0.45	5.45	no	√
	Guar	100	25	10	9.5	38.95	50.99	S	0.97	0.04	2.52	no	X
Section 5.4	Guar	100	25	10	9.5	38.95	50.99	Ni	0.51	0.80	4.84	no	√√√
	Guar	100	25	10	10.5	41.15	51.75	S	0.13	2.53	1.21	no	√√√
Section 5.5	Guar	100	25	10	10.5	41.15	51.75	Ni	0.93	0.10	3.91	no	X
	Guar	100	50	10	10.5	42.08	55.55	S	0.26	1.54	1.09	no	√√√

The P value is a probability, with a value ranging from zero to one. If the P value is small, than the difference between sample means is unlikely to be a coincidence, and hence there the difference would be deemed an experimental difference.

Table 5.2: T-test of key batch flotation results, analysing grades, with √√√ = significant experimental difference with P ≤ 59%, √√ = experimental difference with P≤ 69%, √ = minimum experimental difference with P ≤ 79%, difference X= no experimental difference with P > 80%.

Referrence Section in the chapter	Depressant Type	Depressant dosage g/t (of rougher feed)	Collector Dosage (g/t) Rougher	Collector Dosage (g/t) Cleaner	Cleaner pH	Final Grade Ni	Final Grade S		Statistical t-test analysis of grades P	t	Standard error of difference	Statistical significance	Experimental significance in differences
Section 5.3	CMC	10	25	10	9.5	3.79	7.68	Ni	0.46	0.92	0.82	no	√√√
	Guar	10	25	10	9.5	4.36	8.76	S	0.87	0.19	2.49	no	X
	CMC	50	25	10	9.5	6.11	11.35	Ni	0.79	0.30	1.13	no	√
	Guar	50	25	10	9.5	6.18	12.20	S	0.56	0.69	3.51	no	√√√
	CMC	100	25	10	9.5	10.72	19.06	Ni	0.40	1.06	0.43	no	√√√
	Guar	100	25	10	9.5	10.65	19.09	S	0.67	0.49	7.50	no	√√
Section 5.4	Guar	100	25	10	9.5	10.65	19.09	Ni	0.62	0.58	0.84	no	√√
	Guar	100	25	10	10.5	11.24	22.18	S	0.69	0.46	6.95	no	√√
Section 5.5	Guar	100	25	10	10.5	11.24	22.18	Ni	0.63	0.56	0.71	no	√√
	Guar	100	50	10	10.5	10.92	23.99	S	0.68	0.48	5.77	no	√√

5.3 Effect of depressant type and dosage

The initial objective in this part of testwork was to achieve the desired solids mass recovery (1%) in the cleaner circuit. This was done by varying the dosage of the depressant. All the tests reported in this section and in section 5.4 were done at identical rougher flotation conditions as presented in Table 3.8. All the flotation test conditions described in this section and section 5.4 refer to the cleaner flotation conditions. All the depressant dosages that will be quoted in this chapter and subsequent chapters are the depressant dosages based on the rougher feed and not the cleaner feed.

Figure 5.1 shows the solids mass recoveries obtained at various dosages (10, 50 and 100 g/t) of the two depressants tested (guar and CMC), at a constant SNPX collector dosage of 10 g/t. SNPX collector was chosen based on the results of the microflotation tests which showed that this was the best collector to use. As may be seen in Figure 5.1, a higher recovery was always obtained when using CMC in comparison to guar. This indicates that CMC was a less effective depressant than guar, which is consistent with the observations from the microflotation testwork. However, Figure 5.1 also shows that there was little difference between the two depressants, in terms of the final solids mass recoveries obtained, at higher dosages.

For tests conducted using 50 g/t and 100 g/t of depressant, the final mass recoveries obtained when using guar and CMC were almost equal, at about 2% and 1% respectively. The only distinct difference in the results obtained, in terms of mass recovery, was observed when a low depressant dosage of 10 g/t was used. At this dosage the mass recovery obtained when using guar was 2.8% whereas with CMC the mass recovery was 3.2%. This low dosage of 10 g/t is equivalent to the dosage used in the microflotation testwork of 10 ppm. This means a direct comparison can be made at this dosage, and the results obtained from both the microflotation and batch flotation testwork are consistent, in that guar is a more effective depressant than CMC.

Another key thing to note from Figure 5.1 is that if more than 100 g/t depressant dosage is used, the targeted mass recovery will not be achieved. This means that a depressant dosage of more than 100 g/t cannot be used.

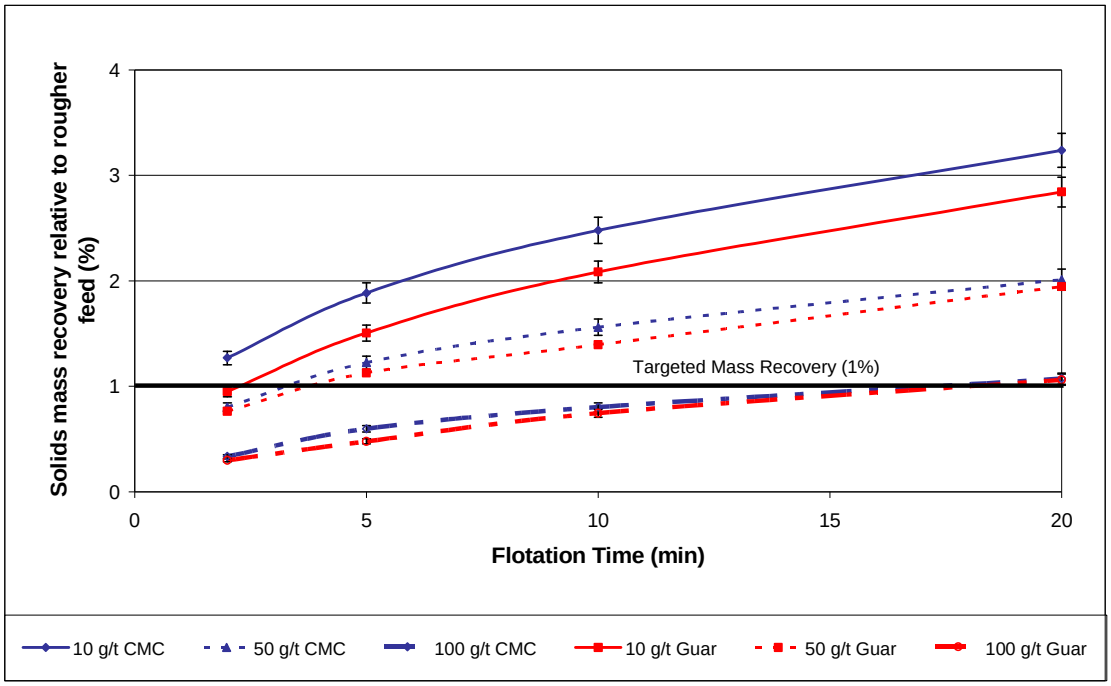


Figure 5.1: Solids mass recovery versus time for cleaner batch flotation tests with Sheba's Ridge ore at pH 9.5 at various dosages of CMC and guar depressant. Collector = 10 g/t SNPX

The solids mass recovery data, however, do not give a complete understanding of the system, since selectivity also needs to be considered. The role of entrainment needs to be taken into account, and since entrainment is controlled by water recovery, one can investigate whether a direct correlation between mass recovery and water recovery exists. The graph of solids versus water recovery is presented in Figure 5.2. This graph shows that at 10 g/t dosage of both CMC and guar the maximum mass recovery was obtained at the same water recovery (~ 170 g). As the dosage of the depressant was increased, there was a corresponding decrease in the amount of water and solids recovered suggesting that floatable gangue, which would tend to stabilise froth (and lead to higher water recovery) was being depressed.

Initially, at 10 g/t, there was ~ 3% solids recovered (relative to the solids in the original rougher feed) and ~ 170 g of water. At 50 g/t, there was a sharp drop in both water and solids recovery to 80 g and 2%, respectively. As the depressant dosage was increased even further to 100 g/t, only solids recovery showed a marked decrease to 1% (from 2%), while the water recovery only dropped from 80 to about 55 g.

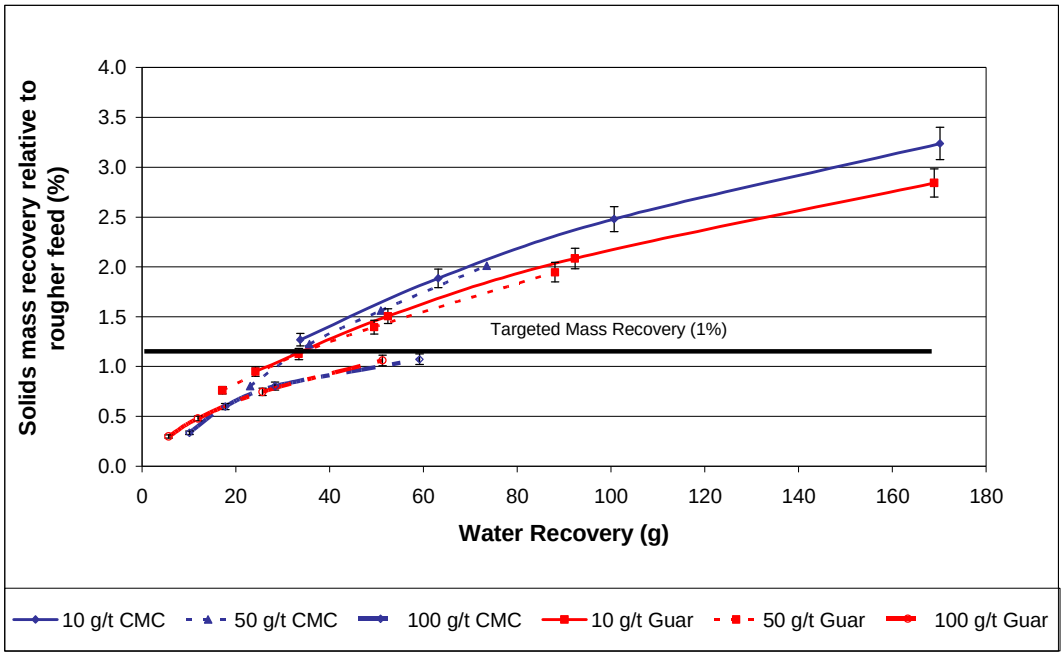


Figure 5.2: Solids and water recoveries for varying depressant type and dosage in the cleaning stage with Sheba's Ridge ore at pH 9.5 at various dosages of CMC and guar depressant. Collector = 10 g/t SNPX

Based on the water recovery data in Figure 5.2, it can be assumed that at 100 g/t guar or CMC dosage, most of the floatable gangue has been depressed and the concentrate should contain mostly the valuable mineral at a good grade. This needs to be confirmed by chemical analysis of the concentrate, by checking the recovery and grade of the valuable minerals obtained, which is presented in Figures 5.3 and 5.4.

Figure 5.3 shows that Ni grades of 9% and above (this is the target for good pentlandite selectivity, see Section 5.1 above) were obtainable, mostly at high depressant dosages. At this targeted Ni grade of 9%, the recoveries obtained ranged from 41% when guar was used at 50 g/t to about 44% when CMC was used at 100 g/t.

When guar was used at 100 g/t, the recovery obtained at the targeted Ni grade was ~43% (refer to Figure 5.3).

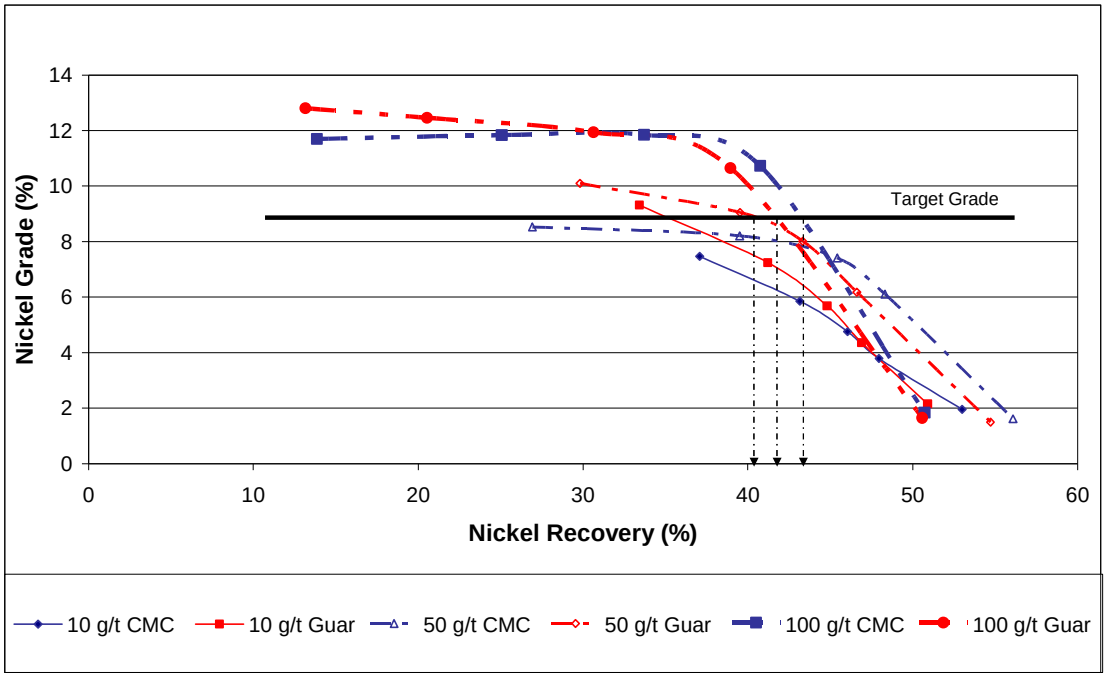


Figure 5.3: Cumulative nickel grade versus recovery for cleaner batch flotation tests for Sheba's Ridge ore at pH 9.5 at various CMC and guar depressant dosages. Collector = 10 g/t SNPX

A general trend that can be seen in Figure 5.3 is that in the initial stages of the tests, guar depressant works better than the CMC, in terms of higher nickel grades and recoveries. However, as the test proceeds, there is a crossover point where the trend is reversed and the CMC performs better than the guar depressant.

Another point to note from Figure 5.3 is that to obtain the desired nickel grade, higher depressant dosages have to be used, i.e. at least 50 g/t when using guar, and 100 g/t when using CMC. Operating at 100 g/t depressant proved to be the best as the highest nickel recoveries were obtained at these dosages. Consequently, it was decided that 100 g/t dosage would be used in all the subsequent testwork.

The different conditions tested here were also analysed statistically to check whether the differences were statistically significant, experimentally different or due to experimental error. Table 3.9 showed that comparing these two depressants, at lower

dosages of 10 and 50 g/t, there were significant differences. However, as the dosage was increased to 100 g/t, the differences became less significant meaning that the two depressants started to behave similarly. The differences observed in the results obtained at lower depressants dosages of 10 and 50 g/t were not statistically significant (refer to Table 5.1 and Table 5.2) but were experimentally different. However, at the highest depressant dosage of 100 g/t the differences observed were not statistically or experimentally different. This means that at this dosage, the two depressants are interchangeable.

Figure 5.4 shows that at the lower depressant concentrations (10 and 50 g/t), better metallurgical performance was obtained when using guar rather than CMC in terms of higher sulphur grades and recoveries. Even though the solids mass recovery data showed that lower masses were obtained when using guar rather than CMC, the recovery data here proves just how much more effective guar was compared to CMC in terms of depressing unwanted gangue minerals, at low dosages. At 10 g/t and 50 g/t, when the sulphur recovery obtained was at ~ 60%, the sulphur grade obtained was ~9% when using CMC, and ~12% when using guar. At the highest concentration (100 g/t) of both depressants, the metallurgical performance obtained was similar, with the test carried out using CMC producing a concentrate with 55% sulphur recovery at a grade of 19% and the test carried out using guar depressant resulting in a concentrate with a sulphur recovery of 53% at a grade of 19%.

Statistically, the differences in the sulphur recoveries obtained at 10 g/t and 50 g/t depressant dosage were not significant but were experimentally different. As for grade, there was no difference obtained at the lowest dosage used, 10 g/t. However, as the dosage was increased from 50 g/t to 100 g/t there were experimental differences observed in the grades obtained, which however had no statistical significance. At 100 g/t there was no statistical or experimental difference in the recoveries obtained. Based on the nickel data, however, which showed that the best results were obtained at 100 g/t, this was the depressant dosage used for all the remaining testwork.

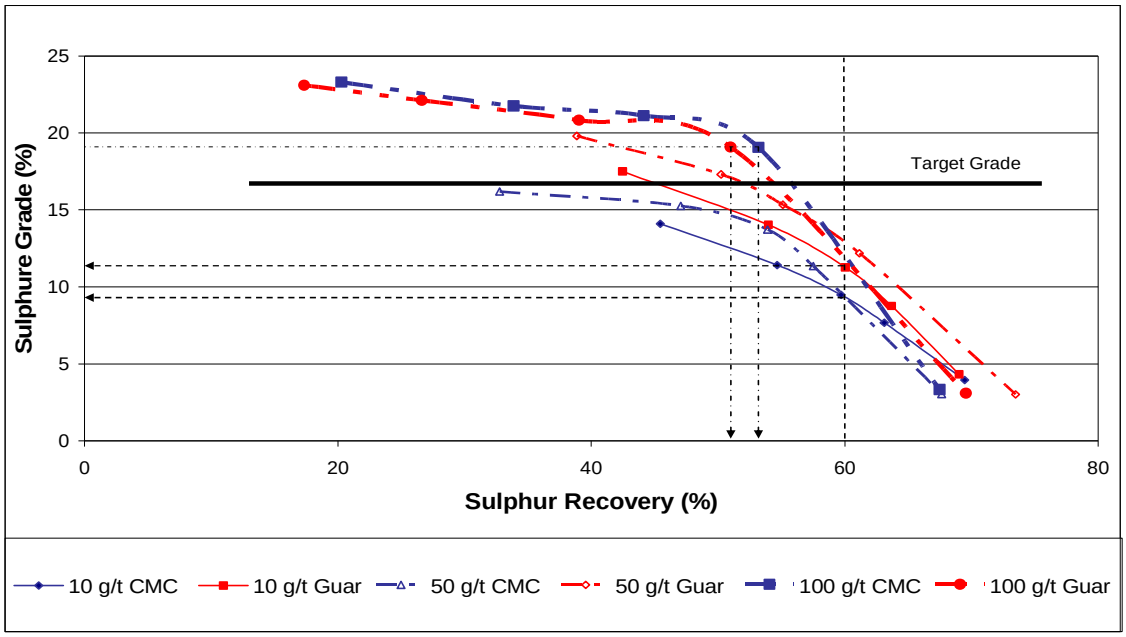


Figure 5.4: Cumulative sulphur grade versus recovery for cleaner batch flotation tests of Sheba's Ridge ore at pH 9.5 at various dosages of CMC and guar depressant. Collector = 10 g/t SNPX

5.4 Effect of pH

After examination of the results reported in the previous section, investigating depressant type and dosage, it is apparent that at the laboratory scale, guar and CMC had generally similar metallurgical performances, especially at higher depressant dosages, in terms of achieving the desired cumulative nickel grade of 9% or above. However, the findings and recommendations from the microflotation testwork (Chapter 4) showed that for pyrrhotite depression, guar was the better depressant to use. Thus, it was then decided that the investigation of the effect of pH would be done using guar depressant at 100 g/t as a baseline.

In Section 4.7, a pH greater than 10 was determined to be optimum for pentlandite/pyrrhotite selectivity for Nkomati ore which has a natural pH of 9. It was decided to investigate the effect of pH for Sheba's Ridge ore at natural pH (9.5) and a pH of 10.5, which is 1.0 pH units above natural pH as was the case for the Nkomati ore. Since the microflotation testwork campaign showed that the better pH modifier to use is lime, all the pH modifications done in this section were performed using lime as

a pH modifier. pH variation was done after the pulp was made up to the desired level and the required reagent conditioning had been carried out (refer to Section 3.4). The results are presented in Figure 5.5 and Figure 5.6.

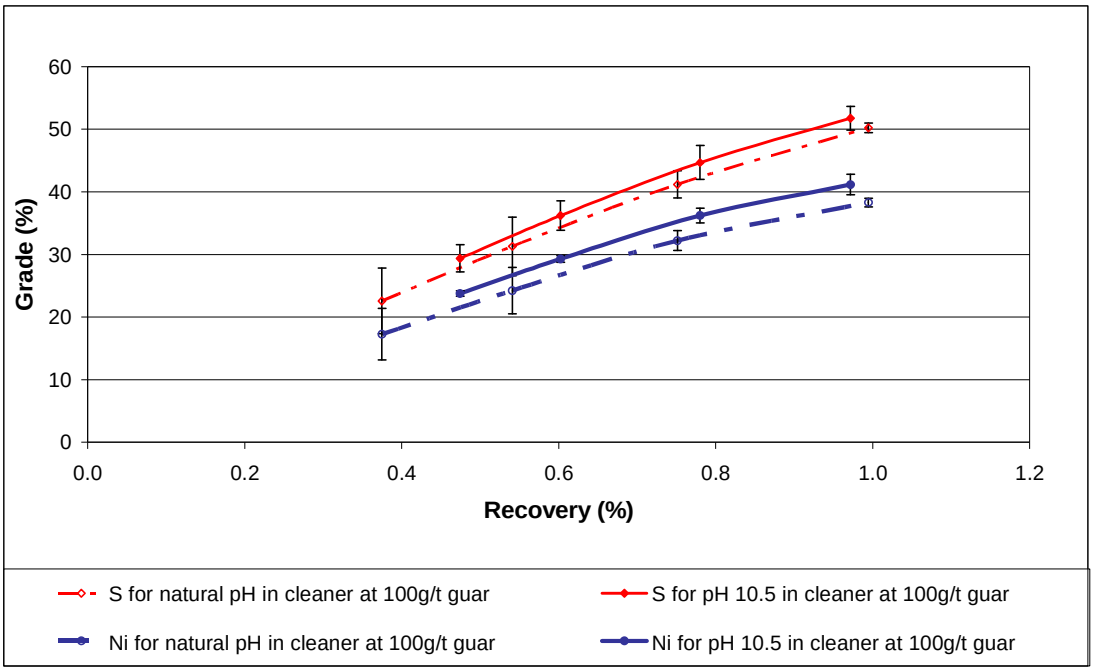


Figure 5.5: Sulphur and nickel recoveries versus mass recovery for cleaner batch flotation tests at collector dosage of 10 g/t with Sheba's Ridge ore at different pH values in the cleaner

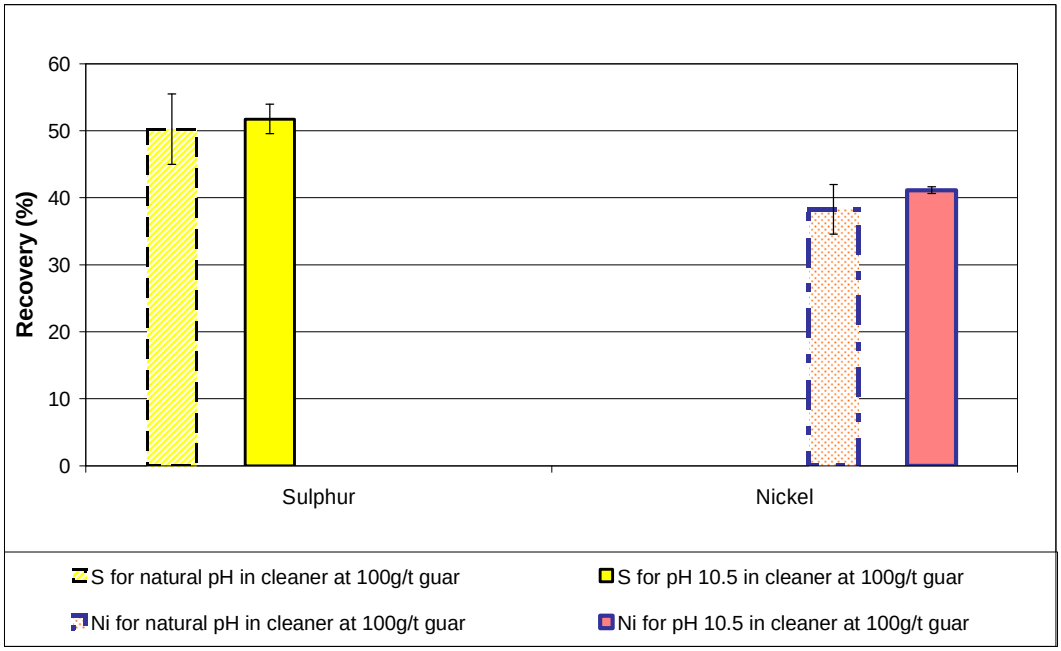


Figure 5.6: Sulphur and nickel recoveries for cleaner batch flotation tests at collector dosage of 10 g/t with Sheba's Ridge ore at different pH values in the cleaner

Figure 5.5 shows that there was a small decrease in the mass recovery obtained between operating at pH 10.5 and the natural pH of 9.5. However, the elemental analyses of Ni and S recoveries showed that there was a slight increase in recovery of both Ni and S at the higher pH. The Ni recovery improved from 38% to 41% and S recovery improved from 50% to 52%.

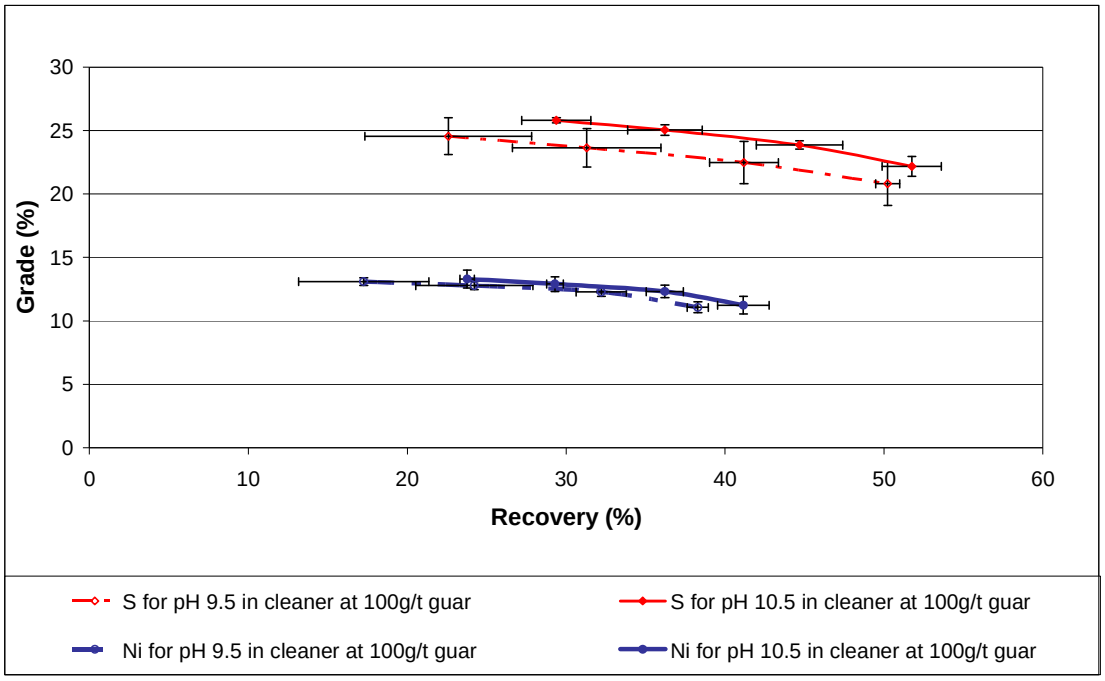


Figure 5.7: Grade versus recovery for cleaner batch flotation tests at collector dosage of 10 g/t with Sheba's Ridge ore, comparing the effect of pH in the cleaner float

In terms of overall metallurgical performance, Figure 5.7 shows the sulphur and nickel grade vs. recovery curves at two different pH values tested, indicating a shift to the right with increasing pH. The shift to the right indicates an increase in recovery and grade and hence better performance. The changes however were observed to be statistically not significant but experimentally different. This was the desired shift as it resulted in an increase in both recovery and grade of sulphur (the grade improved from 21% to 23%, and recovery improved from 50% to 52%). For nickel, the high nickel grade of 10.5% was not compromised when the recovery improved from 38% to 41%, indicating that the increase in pentlandite recovery did not compromise the good selectivity that was obtained at pH of 10.5.

Thus conducting the batch flotation tests at pH 10.5 instead of natural pH (9.5) has brought about slight improvements in the metallurgical performance, as was observed in the microflotation results. However, the improvements that were observed in the microflotation tests were much more noticeable than those in the batch flotation test. Based on these observations, it was decided to change the operating conditions in the batch rougher flotation tests in an attempt to further improve the results in the cleaner stages. The next section describes how this was carried out.

When operating at pH 10.5 as opposed to pH 9.5, showed that the nickel recoveries obtained were of no statistical significant difference but were experimentally different. Similarly for nickel grade, there was no statistical significance in the results obtained, but the differences observed were experimentally different. The differences in sulphur recoveries and grades were also observed to have no statistical difference but were experimentally different. This means that the overall sulphur content of the concentrate was the same, but there was slightly more nickel available. This shows that only pentlandite was affected and not pyrrhotite and hence operating at higher pH does have a slight benefit to selective pentlandite flotation over pyrrhotite.

5.5 Effect of higher collector addition in rougher stage

It has been shown that operating at higher pH (Section 4.7 and Section 5.3) resulted in an improvement of the metallurgical performance obtained. However, the improvements observed in Section 5.3 were not considered sufficient, as the mass recoveries obtained were still lower than targeted. To increase the recovery obtained in the cleaner stage, it was therefore decided to change the operating conditions in the rougher stage by increasing the amount of collector added at this stage. This was done so that higher recoveries of the valuable minerals (especially pentlandite) would be obtained in the rougher concentrate and thereby increase the amount going to the cleaner stage. SNPX was increased from 25 g/t to 50 g/t (i.e. doubled).

This test, with higher collector addition in the rougher, was done at pH 10.5 and 100 g/t guar depressant in the cleaner, and the results are presented in Figure 5.8 and Figure 5.9.

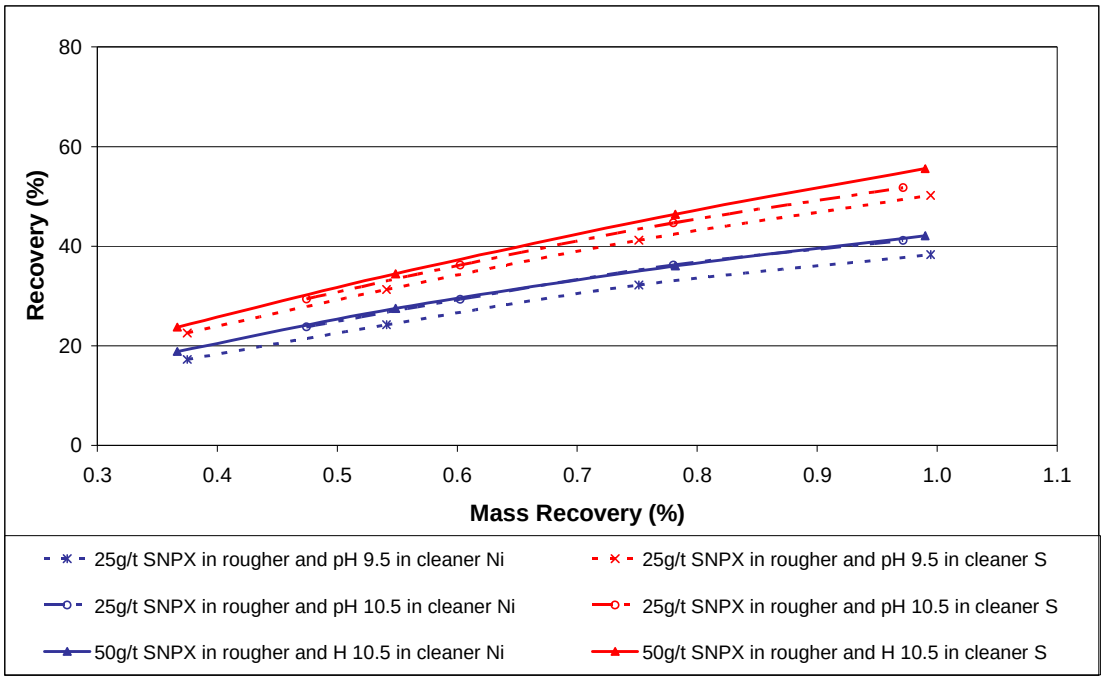


Figure 5.8: Sulphur and nickel recovery versus mass recovery for cleaner batch flotation tests obtained with Sheba's Ridge ore, comparing the effect of collector dosage in the rougher float and pH in the cleaner float at 100 g/t guar depressant dosage in the cleaner stage

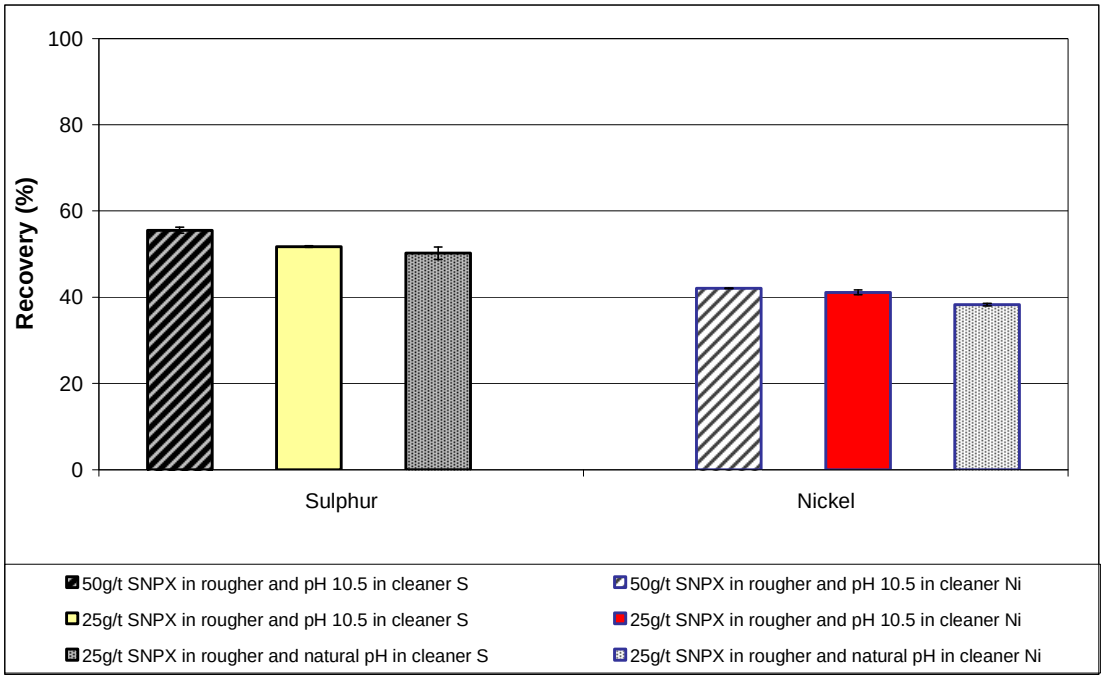


Figure 5.9: Sulphur and nickel recoveries of batch flotation tests obtained with Sheba's Ridge ore for the effect of pH in the cleaner and collector dosage in rougher at 100 g/t guar depressant dosage in the cleaner stage

Figure 5.8 shows that there was a small but significant improvement in the total mass recovery when the collector dosage in the rougher was increased. There was a minimal improvement in nickel recovery from 42% to 42.5%. Sulphur showed much better improvement in recovery, from 52% to 56%. This possibly implies that the increase in collector dosage in the rougher resulted only in higher pyrrhotite recovery in the cleaner circuit.

Figure 5.10 supports what was seen in Figure 5.9, in that nickel performance showed minimal response to the higher dosage of collector added in the rougher. The cumulative nickel grade at higher collector dosage (50 g/t) was the same as that seen at normal collector dosage (25 g/t) at around 11%, while the recovery was still around 42% for all cases. In contrast, sulphur grade improved from 22% to 24%, and the recovery improved from 52% to 56%.

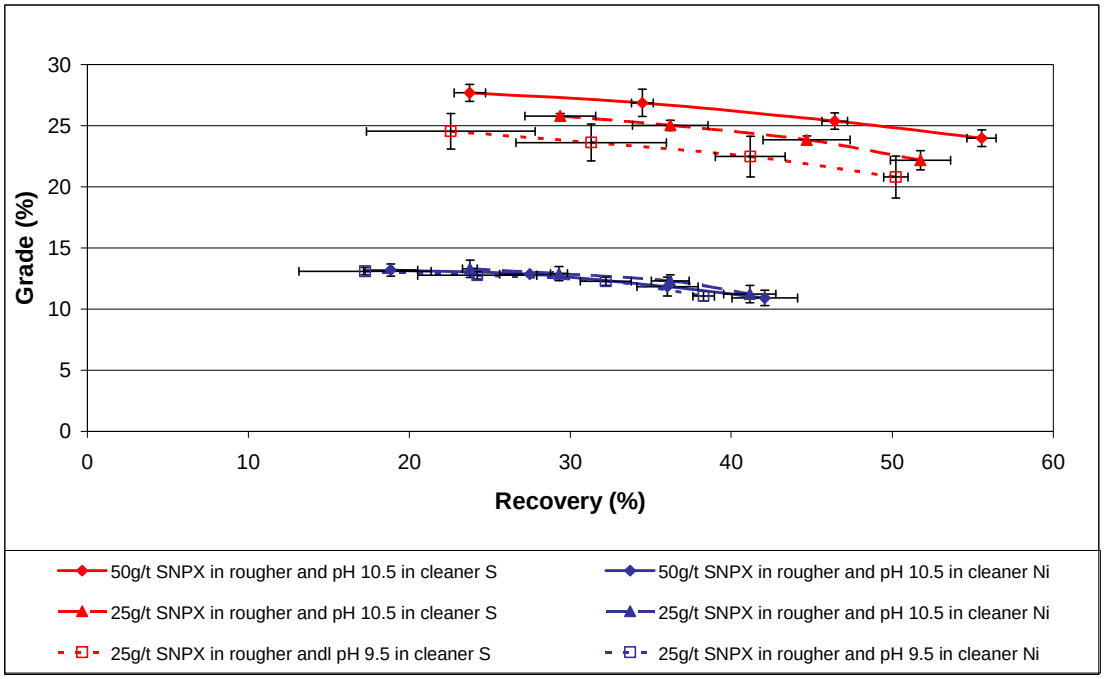


Figure 5.10: Grade versus recovery of batch flotation tests obtained with Sheba's Ridge ore for the effect of pH in the cleaner and collector dosage in the rougher

The results in Figure 5.9 and 5.10 show that operating at a higher collector dosage in the rougher resulted in a very small improvement in nickel (i.e. pentlandite) performance. In terms of sulphur metallurgical performance, however, there was a definite improvement. Higher grades and recovery were obtained. This implies that higher sulphide content was now available in the concentrate. This implies that the improved sulphur performance is probably due to higher amount of pyrrhotite present in the concentrate and not pentlandite. This would have to be confirmed by mineralogical analysis of the concentrate which was not carried out because of cost and time considerations. The statistical analysis supports this observation as the differences in nickel recoveries was not statistically or experimentally different. However, the differences in sulphur recoveries were statistically significant, further supporting that higher collector dosage in the rougher did not result in higher pentlandite recovery, but in better pyrrhotite recovery.

5.6 Summary

i) Interchangeability of CMC vs guar depressant in the cleaning stage

The differences observed when comparing these two depressants were seen to have no statistical significance especially at lower dosages (10 and 50 g/t of rougher feed) but were experimentally different. The experimental differences, however, became less apparent as the dosage was increased to 100g/t (of rougher feed) and at this point either one of the depressants, CMC or guar, could be used. However based on the microflotation test results and recommendations, it was decided to use guar depressant as the depressant of choice and all the subsequent testwork was then done using guar depressant.

ii) Effect of depressant concentration in the cleaning stage

Depressant dosage is a good variable to use to control the concentrate grade while maintaining good recovery. It was also shown that to achieve the desired nickel grade and solids mass recovery, a depressant concentration of 100 g/t (of rougher feed) has to be used. This was evident as the following improvements were seen:

- Sulphur grade improved from ~ 7% to ~ 19% while the recovery changed from ~ 63% to ~ 55% when the dosage was increased from 10 g/t to 100 g/t
- Nickel grade improved from ~ 4% to ~ 11% while the recovery changed from ~ 48% to ~ 41% when the dosage was increased from 10 g/t to 100 g/t

iii) Effect of pH in the cleaning stage

Increasing the operating pH from the natural pH of 9.5 to pH 10.5 caused a slight increase in the nickel and sulphur recoveries, but differences in the sulphur recoveries and grades were shown to be statistically not significant. However, the differences observed in the nickel recoveries obtained were deemed to have no statistical significance but were experimentally different. Similarly, the changes observed in grades obtained when operating at the different pH values were experimentally different but statistically not significant.

iv) Effect of collector dosage on sulphur and nickel in the rougher stage

Increasing the amount of collector used in the rougher stage (25 to 50 g/t) resulted in no statistical improvement in nickel performance but a significant improvement in sulphur recovery and grade. This means that operating at higher collector dosage in the rougher resulted in higher iron recovery in the cleaner stage, and not nickel as was targeted.

The key variables that were found to improve metallurgical performance were depressant and pH. However, the differences between these parameters were more subtle in the case of batch flotation tests compared to what was observed in the microflotation testwork. Similar variables were also found to be key in the microflotation testwork in terms of improving metallurgical performance for better pentlandite selectivity. These were pH, collector and depressant type and dosage. From this work, it can be concluded that results from microflotation test work can used as a starting point to studying the interactions of different minerals in ores in a batch flotation system.

Chapter 6: Discussion

6.1 Introduction

In this chapter, the results obtained from Chapter 4 and 5 are discussed in more detail. The findings from the previous chapters are put into perspective and their implications are discussed in the context of this thesis, and its objectives.

6.2 Microflotation testwork

6.2.1 Comparison of different xanthate collectors in microflotation of Nkomati ore

Comparison of the four xanthate collectors used in this study, SEX, SNPX, SIBX and PAX, showed that mass recoveries greater than 80% could be achieved for pentlandite and pyrrhotite when using these collectors. As shown in Figure 4.3, the rate of flotation increased as the xanthate chain length increased (PAX > SIBX > SNPX > SEX) which is consistent with the fact that the hydrocarbon chain is the portion of the collector that provides the hydrophobicity, and hence the driving force leading to bubble attachment (see Section 2.5.2.1). Figure 4.5, however, showed that the increase in flotation response was not a selective mechanism as both pentlandite and pyrrhotite flotation was improved. This was not expected; because pyrrhotite tends to oxidise faster than pentlandite, and the fact that the shorter chained xanthates are expected to be more selective than the longer chained xanthates, it was anticipated that some selectivity would be obtained for some of the xanthate collectors tested.

The increase in hydrophobicity with increased xanthate chain length was expected to lead to a decrease in selectivity, which was not the case. The shorter chained collectors are less hydrophobic than the longer chained collectors, implying that they have less affinity to attach to air bubbles, after attaching to mineral surface, and would have longer retention time within the pulp and hence better chance of attaching to the most hydrophobic mineral surface, which is pentlandite in this case.

These results showed clearly that the desired selectivity could not be achieved through the use of collectors. It was for this reason that it was decided to seek the desired flotation selectivities that could be obtained by using different depressants.

6.2.2 Comparison of different depressants (with and without collectors) in microflotation of Nkomati ore

In the flotation of platinum bearing ores, polysaccharides such as guar gum and carbomethylcellulose, CMC, are commonly used as depressants. These depressants have proven to be effective in minimising the flotation of problematic silicate gangue minerals, such as talc, a naturally floatable layered silicate (see Section 2.5.4.2 above). There is however, some indication that these depressants also affect the flotation of sulphides (Wiese *et al.*, 2005).

From the results of the microflotation tests carried out using depressants only as reagents, it was seen that the greatest depression of both pentlandite and pyrrhotite was obtained when guar gum was used, followed by starch, and then CMC. The focus of this study, however, is on pentlandite *flotation* and pyrrhotite *depression*: the depressant that produced the highest recovery of pentlandite for the flotation test with no collector was CMC (Figure 4.6). The fact that the recoveries obtained when using the different depressants were lower than the recovery in the flotation test with no reagents at all indicates that there is some interaction between the sulphides and these polysaccharide depressants.

The higher recoveries obtained when using CMC may be attributed to the slime cleaning effects. Robertson (2003) proposed that CMCs, which are dispersing depressants, increase the hydrophobicity of sulphides by slime cleaning. This means that instead of depressing sulphides, it is possible for CMCs to improve their flotation, as seen in Figure 4.7 where CMC, the only depressant, resulted in recoveries greater than 60% for both pentlandite and pyrrhotite.

Guar and starch, however, are effective in depressing both pyrrhotite and pentlandite as seen from the recoveries of lower than 40 % for both pyrrhotite and pentlandite. This is in agreement with the findings of Wiese *et al.* (2006, refer to Section 2.5.4.2

on depressants) who observed that low-charged guar depressants appear to adsorb more strongly to mineral surfaces than the high-charged CMC, resulting in greater depression. Rath *et al.* (2001) also observed that a guar depressant interacted with the hydroxylated mineral surface of chalcopyrite thereby depressing it.

However, as in the case where only collectors were used, the microflotation tests using only depressants did not show selectivity for pentlandite over pyrrhotite. When only depressants were used, both pyrrhotite and pentlandite were depressed. When only collectors were used, both pentlandite and pyrrhotite were floated. Hence it was decided to carry out microflotation tests using collectors and depressants together.

Figure 4.9 shows that when collectors were used in conjunction with depressants, the combination that gave the highest pentlandite selectivity (inferred from the Ni/Fe ratio), was guar/SNPX. It is interesting to note that this reagent combination is between guar, the strongest depressant of both pentlandite and pyrrhotite when used with no collector, and SNPX, which is an intermediate chain length collector. SEX, with the shortest chain length, seemed to be largely ineffective when used in conjunction with guar; the guar was most likely too strong for it.

These results confirm that the shorter chained xanthate collectors are more selective than the longer chained xanthates. However, if the xanthate chain length is too short (e.g. SEX) then in the presence of strong depressants like guar, such collectors become ineffective in their collecting ability.

6.2.2.1 The effect of oxygen addition on the microflotation of Nkomati ore in the presence of SNPX collector and guar depressant

The effect of oxygen addition was tested by sparging oxygen prior to conditioning the pulp with the different reagents. The addition of oxygen probably resulted in some oxidation taking place in the pulp. The presence of oxygen in solution also promotes the conversion of xanthate to dixanthogen, and dixanthogen promotes the hydrophobicity of pyrrhotite by formation of iron hydroxide dixanthogen species on the surface of pyrrhotite. Such observations were made by Hodgson and Agar (1989)

for xanthate adsorption at pH 9 on pyrrhotite and pentlandite surfaces. On the pyrrhotite surface, they observed the formation of an $Fe(OH)_2X$ product upon interaction with xanthate. A similar process was proposed to have taken place on the pentlandite surface, where the presence of oxygen in solution facilitates the chemisorption of xanthate on the surface thereby inducing hydrophobicity.

Other authors (Bozkurt, 1997; Kocabag and Smith, 1985) have pointed out that when two sulphide minerals are in contact with each other in an aqueous solution, galvanic interaction will take place resulting in enhanced oxidation of the anodic mineral. In this study pentlandite is the cathodic mineral and pyrrhotite the anodic mineral. Thus pyrrhotite would be expected to oxidise first upon addition of oxygen in the pulp.

Figure 4.11 shows that when oxygen addition was used on the Nkomati sample, prior to microflotation, there was an increase in the mass recovery observed. This was probably due to higher rate of pyrrhotite flotation, due to the dixanthogen that was present on the pyrrhotite surface, which was not available in conditions where there was no prior oxidation of the sample. Where there was little dissolved oxygen in solution, this only facilitated the presence of xanthate.

Excessive oxygen addition has been shown to have an adverse effect on the flotation of sulphides, in that it can result in alteration of the original mineral surfaces. According to Legrand *et al.* (2005) and Richardson and Vaughan (1989), pentlandite can be altered to violarite. The resulting mineral surface can be covered by a number of secondary layers including hydrophilic FeOOH which may prevent the adsorption of flotation reagents (i.e. collectors). This would result in lower flotation recovery of the desired mineral and lower than desired selectivity.

In the work of Newell *et al.*, (2006), on the same Nkomati sample, they found that of the two sulphides of interest (pentlandite and pyrrhotite) pyrrhotite was less responsive to flotation than pentlandite; they attributed this to reactivity. They proposed that since pyrrhotite was more reactive than pentlandite, this would result in pyrrhotite oxidising more readily than pentlandite. This is in agreement with the

findings of Legrand *et al.* (2006), who also proposed that pyrrhotite oxidation occurred more readily on pyrrhotite surface than on pentlandite.

However, results obtained in this study show that pyrrhotite and pentlandite flotation was not adversely affected by oxygen addition, but rather facilitated as the recoveries obtained were better with oxygen addition. This suggests that the higher dissolved oxygen level facilitated the formation of dixanthogen and hence improved the flotation of pyrrhotite and pentlandite.

It has nevertheless been shown by Newell *et al.* (2006) and Legrand *et. al.* (2006) that under the specific oxidising conditions, the flotation response of pyrrhotite and pentlandite can be adversely affected. For this reason, it was decided to test the effect of pH modification in conjunction with oxygen addition in an attempt to obtain optimum conditions for selective pentlandite flotation.

6.2.2.2 The effect of pH and pH modifier

As noted in the previous sections, oxygen addition resulted in an increase in the amount of dissolved oxygen in solution, which was intended to facilitate the oxidation of the sulphides present in solution. Nevertheless, the flotation of pentlandite and pyrrhotite were not observed to be affected adversely by oxygen addition.

Another means of increasing the rate of oxidation used was by increasing the pH, as this would result in more hydroxyl groups present in solution. Of the sulphides present in the Nkomati ore, pyrrhotite is known to be more reactive, meaning it would also be expected to be the first to be affected by oxidation and thus the first sulphide to be depressed by oxidation due to increase of pH. This should lead to improved selectivity of pentlandite over pyrrhotite.

Figure 4.15 shows that when 5 min oxygen addition was used in conjunction with an increase in pH, pentlandite recovery remained mostly high (≥ 79%), except for the test done at pH 10.5 in which the recovery dropped to below 40% for both pH modifiers. The selectivity of pentlandite (inferred from Ni/Fe ratio) increased sharply,

as seen in Figure 4.16, increasing from 1.3 at pH 9 to 2.4 at pH 10, when lime was used as pH modifier. This Ni/Fe selectivity also implies pyrrhotite depression.

However, when NaOH was used as pH modifier, the selectivity only increased from 1.1 at pH 9 to 1.7 at pH 10. This suggests that when lime was used there was greater pyrrhotite depression while pentlandite recovery was maintained. In fact it is clear from Figure 4.16 that lime consistently performed better than NaOH as a pH modifier by achieving better selectivities. The same selectivities that were obtained using NaOH were always achieved using lime at 0.5 pH units lower. This is counter intuitive as the NaOH used in this research was of higher purity than lime, and would thus be expected to perform better than lime.

When pyrrhotite oxidises, it consumes the dissolved oxygen that is in the pulp resulting in a decrease in the amount of measured DO. During this process, the surface of pyrrhotite is progressively passivated due to the formation of hydrophilic ferric hydroxides that inhibit any further oxidation. When no further interaction of the pyrrhotite surface with oxygen is possible, there would be a corresponding increase in the DO level in the pulp as there would be no reactive sulphide phase to consume the oxygen. Alternatively, it could be that there was an analytical effect which resulted in incorrect calibration of the meter resulting in incorrect reading measured.

Figure 4.18 also shows that lime always achieved higher dissolved oxygen content than NaOH. This means that under lime pH modified conditions there would be a higher degree of pyrrhotite oxidation. This explains why lime always performed better than NaOH, as it was able to facilitate a higher degree of oxidation, which was directed to the fast oxidising mineral pyrrhotite, and allowed for better adsorption of guar onto pyrrhotite and consequently better selective flotation of pentlandite.

It should be noted that pentlandite also plays a role in the higher oxidation and depression of pyrrhotite through galvanic interaction. Oxygen reduction would take place preferentially on pentlandite, and the oxidation of pyrrhotite would be correspondingly accelerated (Bozkurt, 1997). This also means that the objective of this study, of selective pentlandite flotation and pyrrhotite depression, was better

achieved under the conditions where lime was used as a pH modifier at pH 10, and using SNPX and guar, with oxygen addition for 5 min.

6.3 Translation of the microflotation results to laboratory batch flotation tests

The conditions in which the microflotation tests were carried out can be summarised as follows:

- The mass of sample used was 2 g
- The sample used was a massive sulphide ore with a low gangue mineral content (Nkomati ore)
- No froth phase was present
- The depressants used were selected to interact and depress mostly sulphide minerals
- The collectors used interacted mostly with sulphide minerals
- Only one size fraction was considered (between 106 µm and 75 µm)
- The pulp environment was quiescent

In the case of the batch flotation tests, the conditions can be summarised as follows:

- The mass of sample used was 1 kg
- The sample used was a disseminated sulphide ore with low sulphide content (~ 2%) and a high gangue mineral content (Sheba's Ridge ore)
- A froth phase was present
- The depressants used were selected to interact and depress mostly gangue minerals present in the ore
- The collectors used interacted almost solely with the sulphides minerals
- A wide particle size distribution was considered
- The pulp environment was turbulent

As can be seen there were many differences between the microflotation tests and the laboratory batch flotation tests. With this in mind, the first relationship that was tested

to see if observations made in the microflotation tests could be translated to the batch flotation tests was the interaction of the sulphide minerals with the depressants in the presence of high gangue mineral content. Instead of evaluating all the depressants tested in the microflotation testwork, only CMC and guar gum were used, as the mineralogical analysis showed that the major problematic silicate mineral in the Sheba's Ridge ore was talc (Table 3.2). Previous work (Wiese *et al.*, 2008) showed poor depression and selectivity obtained when using starch, so it was decided not to test starch as it was already proven to be an ineffective depressant.

When comparing the performance of guar and CMC depressants (Figure 5.2 to Figure 5.4) better metallurgical performance was achieved when using guar gum especially at lower concentrations. However, when the concentrations were increased, the performances of the two depressants were almost comparable. It is well known that CMC is negatively charged and that guar gum is an uncharged molecule. It has also been stated in literature that CMC interacts with the cations in solution more strongly with the divalent cations like Ca^{2+} and Mg^{2+} (Parolis *et al.*, 2008; Khraisheh *et al.*, 2005), to cause the overall charge of the molecule to be neutral. The interaction with the cations also causes the CMC molecule to coil up, which results in lower area of adsorption that the depressant affects (Burdukova, 2007).

From coiling up of CMC, it can be seen that in the presence of cations which are always present in the batch flotation pulp (refer to Table 3.6 for batch flotation plant water composition) CMC can adopt a similar chemistry to that of guar gum (low charge). This may explain why CMC achieved lower depression than guar gum at low concentrations as there was insufficient surface coverage of CMC as it had coiled up.

For the batch flotation tests at higher pH values it was decided to use guar, as it was clear from the microflotation work that guar was the better depressant and the only one proposed to be able to also depress both pyrrhotite and the naturally floating gangue. In addition, Wang *et al.* (2005) showed that the adsorption of guar onto talc was not affected by changes in pH, whereas Burdukova (2007) showed that the adsorption of CMC was affected by pH. Wang *et al.* (2005) also found that guar was not affected by ionic strength of solution and the presence of cations, which Parolis *et al.* (2008) showed does affect the functioning of CMC. Guar can be adsorbed

sufficiently to a mineral surface that has a net positive/negative charge on it as it is an uncharged molecule.

It is seen in Figure 5.7 that when pH was increased from natural pH of 9.5 to 10.5, there was a corresponding increase in the nickel recovery while the sulphur recovery stayed constant. This essentially means that the metallurgical performance was not compromised due to the corresponding increase in recovery, as high grades were still maintained. Similar trends were seen in Figure 5.9 when the collector concentration used in the rougher stage was increased as higher sulphur recoveries were obtained, while the nickel recovery stayed constant.

The results also show that good responses at higher pH were also achieved in batch flotation tests, as was seen in the microflotation tests, indicating that the results from microflotation tests were translated to batch flotation tests. As was discussed for the microflotation testwork, the influence of operating at higher pH resulted in selective depression of pyrrhotite, hence the high nickel grades achieved. As in the work done by Wiese *et al.* (2006a and b), higher nickel grades were achieved when using guar as guar probably depressed pyrrhotite, particularly when operating at higher pH.

From this work it can be seen that the depressant work done in the microflotation tests were translated to the batch system, confirming that guar is the preferred depressant to use in terms of achieving the desired metallurgical performance. The pH studies also proved to be translatable in that operating at higher pH provided improved metallurgical performance just as was seen in the microflotation testwork. However, the differences between these parameters were observed to be subtle and less apparent compared to what was observed in the microflotation testwork.

7 Conclusions and Recommendations

Pentlandite from the Nkomati and Sheba's Ridge ore can be selectively floated while depressing pyrrhotite by operating at pH 10.5, using lime as pH modifier, and SNPX collector with guar gum depressant.

1. Xanthate hydrocarbon chain length does affect the selectivity of flotation for pentlandite and pyrrhotite. However, the difference observed was very subtle, lying within a narrow selectivity range. The difference was attributed to the kinetics of flotation. Better selectivities were achieved for sulphide flotation with the shorter chain length collectors due to their slower flotation rates, whereas the longer chained collectors resulted in rapid flotation kinetics that were too fast for any selectivity to occur.

2. The depressant type does affect the selective flotation of pentlandite and pyrrhotite
   i. For collectorless flotation, the low-charged depressants (guar gum and starch) achieved much better depression of pyrrhotite and pentlandite than the high-charged CMC. In terms of selectivities achieved, the lower molecular weight starch achieved the best selectivity (high vs low), followed by guar gum and CMC which induced the lowest and weakest selectivity.
   ii. For collector induced flotation (in the presence of SEX, SNPX, SIBX and PAX), use of guar gum achieved the best selectivity (in the presence of SNPX) in terms of the greatest amount of pyrrhotite depression while maintaining good pentlandite flotation.

3. The good selectivity achieved by the combination of guar gum and SNPX can be further influenced by:
   i. The presence of oxygen which facilitates the conversion of xanthate to dixanthogen (at pH 10 and above, Legrand *et al.*, 2005). During this process, dixanthogen promotes the hydrophobicity of pyrrhotite by formation of iron hydroxide dixanthogen species on the surface of pyrrhotite. This resulted in lower selectivities.

ii. Increasing the pH when used with moderate oxidation of 5 min using lime, resulted in higher and faster rates of oxidation of the sulphides by introducing more hydrophilic hydroxyl groups at the surface. Pyrrhotite being the most severely affected since it has fastest oxidation rate of the sulphides present in solution. This led to even better selectivities achieved in that more pyrrhotite was depressed while the same amount of pentlandite was still floating.

iii. Changing the pH modifier allowed the greatest amount of oxygen to be retained in solution when lime was used. Lime was also proved to show consistently better selectivities relative to NaOH, as it allowed the highest pyrrhotite depression.

4. The depressant type results proved translatable from microflotation work to the batch flotation work in that just as it was seen in the microflotation work that guar gum used with SNPX gave better selectivities (hence better metallurgical performance) than CMC, the same results were seen for the batch tests in that better grades and recoveries were obtained when using guar than when using CMC .The pH work proved to be translatable from microflotation work to the batch flotation work as operating at higher pH provided resulted in slightly improved metallurgical performance (in batch test work) just like it was seen for the microflotation testwork that it improved the selectivities achieved in the desired direction. Statistically, the differences observed in the batch flotation testwork were not considered significant, but were experimentally different. This showed that though the results were translated to a certain degree, more work still needs to be done in the future.

It was also shown that there needs to be a synergistic interpretation of the interactions present in the reagent – mineral system. The study has also shown that the reagents used in flotation cannot be evaluated independently but rather a holistic approach needs to be employed.

In the light of the above, the following recommendations can be made:

The suggested reagent suite should be tested in a pilot plant to see if the same effectiveness that was seen in the laboratory scale tests can be reproduced in a pilot plant scale. The objective of this thesis was to find a solution that has viable commercial application. The results shown in Chapter 4 and Chapter 5 showed that the results obtained in a microflotation test campaign was successfully scaled up to batch flotation. To validate this scaling up applicability, it is recommended that the same conditions used in the batch flotation tests be tested on a continuous pilot plant scale.

Mineralogical analysis should be carried out on metallurgical samples of all the flotation tests, to confirm pentlandite selectivity since it was based on a chemical assay and not actual mineralogical results.

A full characterisation of the tailings sample should be made to confirm that there are indeed no PGM's lost with pyrrhotite. The rejection of pyrrhotite, from Sheba's Ridge ore, was targeted as it was assumed that there were no valuable minerals associated with it. However this assumption was not validated and it is recommended that a full characterisation of the tailing streams from the batch flotation tests be carried out to check if there is any significant loss of valuable material from this stream.

8 References

Ackerman, P.K., Harris, G.H., Klimpel, R.R., Aplan, F.F., 1987, "**Evaluation of flotation collectors for copper sulphides and pyrite, I. Common sulphydryl collectors,**" International Journal of Mineral Processing, Vol. 21, pp. 105-127.

Agar, G.E., 1991, "**Flotation of chalcopyrite, pentlandite and pyrrhotite ores,**" International Journal of Mineral Processing, Vol. 33, Issue 1, pp. 1-19.

Arnold, R.G., 1966, "**Mixtures of hexagonal and monoclinic pyrrhotite and the measurement of the metal content of pyrrhotite by X-ray diffraction,**" American Mineralogist, Vol. 51, pp.1221–1227.

Ballhaus, C. and Sylvester, P., 2000, "**Noble Metal Enrichment Process in the Merensky Reef, Bushveld Complex,"** Journal of Petrology, Vol 41, No. 4, pg. 545–561.

Beattie, D.A., Huynh, L., Kaggwa, G.B.N., Ralston, J., 2006, "**The Effect of Polysaccharides and Polyacrylamides on the Depression of Talc and the Flotation of Sulphide Minerals,**" Minerals Engineering, Vol 19, pp. 598-608.

Becker M., Butcher A.R., Botha P.W.S.K., Cropp A.F.R., De Villiers J.P.R., Bradshaw D.J., 2007, "**Development of a measurement technique for the quantitative determination of hexagonal and monoclinic pyrrhotite: Evaluation using examples from the Bushveld Complex using QEMSCAN®,"** Presented at Automated Mineralogy, Brisbane.

Becker, U., Munz, A.W., Lennie, A.R., Thornton, G. and Vaughan, D.J., 1997, "**The atomic and electronic structure of the (001) surface of monoclinic pyrrhotite (Fe_7S_8) as studied using STM, LEED and quantum mechanical calculations,"** Surface Science, Vol. 389, pp. 66–87.

References

Belzile, N., Chen, Y., Cai, M., and Li, Y, 2004, **"Review on pyrrhotite oxidation,**" Journal of Geochemical Exploration, Vol. 84, pp. 65-76.

Bicak, O., Ekmekci, Z., Bradshaw, D.J. and Harris, P.J., 2007, "**Adsorption of guar gum and CMC on pyrite,**" Minerals Engineering, Vol. 20, pp. 996-1002.

Boyd, D.A., 1979, "**An investigation into electron spectroscopy as a means of studying the chemistry of froth flotation**," Johnson Matthey Report No. 78J1.

Bozkurt, V., 1997, "**Pentlandite/pyrrhotite interaction and xanthate adsorption,"** PhD Thesis, Department of Mining and Metallurgical Engineering, McGill University, Montreal, Canada.

Bozkurt, V., Xu, Z., and Finch, J.A., 1997, "**Pentlandite/pyrrhotite interaction: xanthate adsorption in presence of Ni ions in Processing of complex ores,"** Ed. J.A. Finch, S.R. Rao and I. Holubec, CIM, Montreal, pp. 101-114.

Bozkurt, V., Xu, Z., and Finch, J.A., 1998a, "**Xanthate adsorption on pentlandite and pyrrhotite: Effect of mineral interactions."** Innovations in mineral and coal processing Ed. Atak, Onal and Celik, Rotterdam, pp. 93-98.

Bozkurt, V., Xu, Z., Finch, J.A., 1998b, "**Pentlandite/Pyrrhotite Interaction and Xanthate Adsorption,**" International Journal of Minerals Processing, Vol. 52, pp. 203-214.

Bradshaw, D.J, Buswell, M.A., Harris, P.J, Ekmekc¸i, Z., 2006, "**Interactive Effects of the Type of Milling Media and Copper Sulphate Addition on the Flotation Performance of Sulphide Minerals from Merensky Ore Part I: Pulp Chemistry,"** Minerals Engineering, Vol. 78, pp. 153–163.

Bradshaw, D.J, Oostendorp, B., Harris, P.J, 2005, "**Development of methodologies to improve the assessment of reagent behaviour in flotation with particular reference to collectors and depressants"** Minerals Engineering, Vol. 18, pp. 239–246.

References

Bradshaw, D.J, Allie, M., Dawood, N., Hatfield, D. and Harris, P.J, 2004, "**The effect of solid hydrophobicity on froth stability and the overall effect on flotation performance,"** SME Annual Meeting 23-25 February, Denver.

Bradshaw, D J and O'Connor, C T, 1996, "**Measurement of the sub-process of bubble loading in flotation**," Minerals Engineering, Vol. 9, pp. 443-448.

Broomhead, J.A. and Layers, G., 1976, "**Leaching studies with pentlandite and pyrrhotite"** Proc. Australias Inst. Min. Metall., No. 259, pp. 19-22.

Bruce, T.J. and Orr, R.G., 1986, "**Pyrrhotite rejection at INCO Ltd.'s Manitoba Division,**" In E. Ozbek and S.W. Marcuson (Editors), Nickel Metallurgy, Proc. 25th Annual Conference of Metal., CIM, Montreal, Que., pp. 57-94.

Buckley, A.N. and Woods, R., 1991, "**Surface composition of pentlandite under flotation-related conditions**," Surface and Interface Analysis, Vol. 17, pp. 675-680.

Burdukova, E., 2007, "**Surface Properties of New York Talc as a Function of pH, Polymer Adsorption and Electrolyte Concentration"** pHD thesis, Department of Chemical Engineering, University of Cape Town, South Africa.

Chander, S., 1988, "**Inorganic depressants for sulfide minerals,**" In: P. Somasundaran and B.M. Moudgil (Editors), Reagents in Mineral Technology, Surfactant Science Series Vol 27, Marcel Dekker, New York, pp.429-469.

Clarke, P., Fornasiero, D., Ralston, J. and Smart R.St.C, 1995, "**A study of the removal of oxidation products from sulphide mineral surfaces**," Minerals Engineering, Vol. 8, No.11, pp. 1347-1357.

de Donato, P., Cases, J.M., Kongolo, M., 1979, "**Stability of the amyl xanthate ion as a function of pH: modelling and comparison with the ethyl xanthate ion,**" International Journal of Mineral Processing, Vol. 25, pp. 1-28.

Dhliwayo, E. C., Maharaj, D., Hinde, E., Mohlabane, K., Ndlovu, L. and Nyuliwe, A, 2007, "**Pilot plant work on samples from Sheba's Ridge Deposit**" Mintek External Report 4637, Randburg, South Africa.

Dhliwayo, E. C.,2005, "**The interactive effect of depressant type and dosage with frother dosage in the flotation of a PGE ore,**" MSc Thesis, Department of Chemical Engineering, University of Cape Town, South Africa.

Dimou, A., 1986, "**The flotation of pyrite using xanthate collectors,**" MSc Thesis, Department of Chemical Engineering, University of Cape Town, South Africa.

Ekmekci, Z., Buswell, M.A., Bradshaw, D.J. and Harris, P.J., 2005, "**The value and limitations of electrochemical measurements in flotation of precious metal ores,**" Minerals Engineering, Vol. 18, pp. 825 – 831.

Ekmekci, Z. and Demirel, H., 1997, "**Effect of galvanic interaction on collectorless flotation behaviour of chalcopyrite and pyrite,**" International Journal of Mineral Processing, Vol. 52, pp. 31-48.

Frommer, D.W. and Colombo, A.F., 1966, "**Process for improved flotation treatment of iron ores by selective flotation,**" U.S. Patent No. 3,292,780.

Fuerstenau, M.C., 1982, "**Chemistry of Collectors in Solution**," In: R.P, King (Editor), Principles of Flotation, South African Institute of Mining and Metallurgy Monograph Series No. 3, Chapter 1, pp. 1-16.

Fuerstenau, M.C.,1982, "**Semi-Soluble Salt Flotation**." In: R.P. King (Editor), Principles of Flotation, South African Institute of Mining and Metallurgy Monograph Series No. 3, Chapter 7, pp. 199-213.

Gain, S.B. and Mostert, A.B., 1982, "**The geological setting of the platinoid and base metal sulphides in the Platreef of the Bushveld Complex in Drethe, North of Portgietersrus,**" Economic Geology, Vol. 77, pp. 1395-1404.

Goh, S.W., Buckley, A.N., Lamb, R.N., Fan, L.J., Jang and L.Y., Yang, Y, 2006, "**Pentlandite sulfur core electron binding energies,"** Phys Chem. Minerals, Vol. 33, pp. 445–456.

Gauert, C. D. K., De Waal, S. A. and Wallmach, T., 1995, **"Geology of the ultrabasic to basic Uitkomst complex, eastern Transvaal, South Africa: an overview,"** Journal of African Sciences, Vol. 21, pp. 553-570.

Heiskanen, K., Kirjavainen V., and Laapas, H., 1991, "**Possibilities of collectorless flotation in the treatment of pentlandite ores**," International Journal of Mineral Processing, 33, pp. 263-274.

Hodgson, M., Agar, G.E., 1989, "**Electrochemical Investigations Into the Flotation Chemistry of Pentlandite and Pyrrhotite: Process Water and Xanthate Interactions,**" Canadian Metallurgical Quarterly, Vol. 28, no. 3, pp. 189-198. July-Sept..

Holwell, D.A. and McDonald, I., 2006, "**Petrology, geochemistry and the mechanisms determining the distribution of platinum group element and base metal sulphide mineralisation in the Platreef at Overysel, northern Bushveld Complex, South Africa,**" Mineral Deposita, Vol. 41, pp. 575-598.

Ishihara, T., and Kagami, Y., 1964, "**Nippon Kogyo Kaishi,**" 80:881.

Iwasaki, I., 1965, "**Interaction of starch and calcium in soap flotation of activated silica from iron ores,**" Trans. AIME, Vol. 232, pp. 383-387.

Jean, E.G. and Bancroft, G.M., 1986, "**Heavy metal adsorption by sulphide mineral surfaces,**" Geochimi Cosmochimi Acta, pp. 1455-1463.

Kelebek, S. and Tukel, C., 1999, "**The effect of sodium metabisulfite and Triethylenetetramine system on pentlandite-pyrrhotite separation,**" International Journal of Mineral Processing, Vol. 57, pp. 135-152.

Kelly, D.P. and Vaughan, D.J., 1983, "**Pyrrhotine - Pentlandite ore textures: a mechanistic approach,**" Mineralogical Magazine, vol. 47, pp. 453-463.

Khraisheh, M., Holland, C., Creany, C., Harris, P. and Parolis, L., 2005, "**Effect of molecular weight and concentration on the adsorption of CMC onto talc at different ionic strengths,**" International Journal of Mineral Processing, Vo. 75, pp. 197-206.

Kirjavainen, V., Schreithofer, N. and Heiskanen, K., 2002, "**Effect of calcium and thiosulfate ions on flotation selectivity of nickel-copper ores**," Minerals Engineering, Vol. 15, pp. 1-5.

Klimpel, R.R., 1988, "**The industrial practice of sulfide mineral collectors**," In: P. Somasundaran and B.M. Moudgil (Editors), Reagents in Mineral Technology, Surfactant Science Series Vol 27, Marcel Dekker, New York, pp.663-681.

Kocabag, D. and Smith, M.R., 1985, "**The effect of grinding media and galvanic interactions upon the flotation of sulphide minerals,**" Complex Sulphide, Processing of Ores, Concentrates and By-Products, A.D. Zunkel, R.S. Boorman and R.J. Wesely, pp. 55-81.

Kolahdoozan, M., 1996, "**Adsorption and flotation characteristics of hexagonal and monoclinic pyrrhotite**," PhD thesis, Queen's University, Kingston, Ontario, Canada.

Laskowski, J.S., Liu, Q. and Bolin, N.J., 1991, "**Polysaccharides in flotation of sulphides Part I: Adsorption of polysaccharides onto mineral surfaces,**" International Journal of Mineral Processing, Vol. 33, Issue 1, pp. 223-234.

Lawson V, Kerr AN, Shields Y, Wells PF, Xu M, Dai Z, 2005, **"Improving pentlandite pyrrhotite separation at INCO's Clarabelle Mill."** In: Centenary of

Flotation Symposium, Australian Institute of Mining and Metallurgy, Brisbane, Australia, pp. 875-885.

Legrand, D.L., Bancroft, G.M., Nesbitt, H.W., 2005, "**Oxidation/Alteration of Pentlandite and Pyrrhotite Surfaces at pH 9.3: Part 1. Assignment of XPS Spectra and Chemical Trends,**" American Mineralogy, Volume 90, pp 1042-1054.

Lovell, V.M., 1982, "**Industrial Flotation Reagents,**" In: R.P. King (Editor), Principles of Flotation, South African Institute of Mining and Metallurgy Monograph Series No. 3, pp. 73-89.

Lotter, N.O., Kowal, D.L., Tuzun, M.A., Whittaker, P.J. and Kormos, L., 2003, "**Sampling and flotation testing of Sudbury Basin drill core for process mineralogy modelling,**" Minerals Engineering, Vol. 16, pp. 857-864.

Lui, Q., Zhang, Y. and Laskowski, J.S., 2000, "**The adsorption of polysaccharides onto mineral surfaces: an acid/base interaction,**" International Journal of Mineral Processing, Vol. 60, pp. 229-245.

Malysiak, V., O'Connor, C.T., Ralston, J., Gerson, A.R., Coetzer, L.P. Bradshaw, D.J., 2002, "**Pentlandite-feldspar interaction and its effect on separation by flotation,**" International Journal of Minerals Processing, Vol. 66, pp. 89-106.

Miller, J.D., Davidtz, J.C. and Vos, F., 2005, "**A review of pyrrhotite flotation chemistry in the processing of PGM ores,**" Minerals Engineering, Vol. 18, pp. 855-865.

Morris, G.E., Fornasiero, D. and Ralston, J., 2002, "**Polymer depressants at the talc-water interface: adsorption isotherm, microflotation and electrokinetic studies,**" International Journal of Mineral Processing, Vol. 67, pp. 211-227.

Newell, A.J.H., Bradshaw, D.J. and Harris, P.J., 2006, "**The effect of heavy oxidation upon flotation and potential remedies for Merensky type sulphides,**" Minerals Engineering, Vol. 19, pp. 675-686.

Newell, A.J.H., Skinner, W. M. and Bradshaw, D.J., 2007, "**Restoring the floatability of oxidised sulfides using sulphidisation,**" International Journal of Mineral Processing, Vol. 84, pp. 108 -117.

Nicholson, R.V. and Scharer, J.M., 1994, "**Laboratory studies of pyrrhotite oxidation kinetics. In: Alpers, C.N., Blowes, D.W. (Eds.), Environmental Geochemistry of Sulfide Oxidation,**" ACS Symposium Series, vol. 550, pp. 14–30. Washington, DC.

Nicol, M.J., 1984, "**Electrochemistry in Mineral and Metal Processing,**" P.E. Richardson, S. Srinivasan and R Woods, Eds., pp. 153-164

Parolis, L.A.S., van der Merve, R., Groenmeyer, G.V. and Harris, P., 2008, "**The influence of metal cations on the behaviour of carboxymethyl cellulose as talc depressants,**" Colloids and Surfaces A: Physicochem. Eng. Aspects, Vol. 317, pp. 109-115.

Parolis, L.A.S., R., Groenmeyer, G.V. and Harris, P., 2004, "**Equilibrium adsorption studies of polysaccharides on talc: The effect of molecular weight, charge and influence of metal cations,**" Presented at SME meeting, Denver Colorado, USA.

Pearce, C I, Patrick, R A D and Vaughan, D J, 2006, **"Electrical and Magnetic Properties of Sulfides,**" In: D.J. Vaughan (Editor), Sulfide Mineralogy and Geochemistry, Reviews in Mineralogy and Geochemistry, Vol. 61, Chapter 3, Mineralogical Society of America, Chantilly, Virginia, pp 127-180.

Poling, G. W. 1976, "**Reactions Between Thiol Reagents and Sulphide Minerals in Flotation," In: M.C. Fuerstenau (Editor),** A.M. Gaudin Memorial Volume, AIME,1, pp. 334.

Pratt, A.R., Muir, I.J., Nesbitt, H.W., 1994, **"X-ray photoelectron and Auger electron spectroscopic studies of pyrrhotite and mechanism of air oxidation,"** Geochim. Cosmochim. Vol. 58, pp. 827– 841.

Ralston, J., 1991, "**E_h and its consequences in sulphide mineral flotation,**" Minerals Engineering, Vol. 4, Nos. 7-11, pp. 859-878.

Rand, D.A.J., 1977, "**Oxygen Reduction on Sulphide Minerals. Part III Comparison of activities of various copper, iron, lead and nickel mineral electrodes,**" Journal of Electroanalytical Chemistry, Vol. 83, pp. 19-32.

Rand, D.A.J., Woods, R., 1984, "**Eh measurements in sulphide mineral slurries,**" International Journal of Mineral Processing 13, 29– 42.

Rao, S.R., 2004, "**Surface chemistry of froth flotation,**" Second Edition, Vol. 2: Reagents and mechanisms, Chapter 8, pp. 351-384.

Rath, R.K., Subramanian, S., Sivanandam, V., and Pradeep, T., 2001**, "Studies on the interaction of Guar Gum with chalcopyrite,"** Canadian Metallurgical Quarterly,Vol. 40, pp. 1-12.

Richardson, S., and Vaughan, D.J., 1989, "**Surface alteration of pentlandite and spectroscopic evidence for secondary violarite formation**," Mineralogical Magazine, Vol. 53, pp. 213-22.

Robertson, C., 2003, "**Development of a Methodology to decouple the effects of Dispersion and Depression in batch flotation,**" MSc Thesis, Department of Chemical Engineering, University of Cape Town, South Africa

Rosso, K M and Vaughan, D J, 2006."**Sulfide Mineral Surfaces**." In: D J Vaughan (Editor), Sulfide Mineralogy and Geochemistry, Reviews in Mineralogy and

Geochemistry, Vol. 61, Chapter 9, Mineralogical Society of America, Chantilly, Virginia, pp 505-556.

Schouwstra, R.P., Kinloch, E.D.and C. A. Lee, 2000. "**A short geological review of the Bushveld complex,"** Platinum Metals Review 44, pp. 33–39.

Schreithofer, N., Kirjavainen, V., Heiskanen, K., Ahveninen, R., 2000, "**Effect Of Ca^{+} And $S_2O_3^{2-}$ On The Flotation Of Pentlandite In Real Ore System,**" Proceedings of the XXI International Mineral Processing Congress, pp. B8b – 40-46.

Shackleton, N. J., 2003, "**The Role Of Complexing Agents In The Floatation Of Pentlandite-Pyroxene Mixtures,**" MSc Thesis, Department of Chemical Engineering, University of Cape Town, South Africa

Steenberg, E. and Harris, P.J., 1984, "**Adsorption of carboxymethylcellulose, guar gum and starch onto talc, sulphides, oxides and salt-type minerals,**" South African Journal of Chemistry, Vol. 37, pp. 85-90.

Taggart, A.F., 1945, "**Handbook of Mineral Dressing**," Wiley, New York

Thornber, M.R., 1983, "**Mineralogical and electrochemical stability of the nickel-iron sulphides-pentlandite and violarite**," Journal of Applied Electrochemistry, Vol. 13, pp. 253-267.

Trahar, J. W., 1984, "**The influence of pulp potential in sulphide flotation**." In M.H. Jones and J.T. Woodcock (Editors), Principles of Mineral Flotation, The Wark Symposium, Australasian Institute of Mining and Metallurgy, Symposia Series No. 40, Melbourne, pp. 117-135.

Wang, J., Somasundaran, P. and Nagaraj, D.R., 2005, "**Adsorption mechanism of guar gum at solid-liquid interfaces,**" Minerals Engineering, Vol. 18, pp. 77-81.

Wang, X., Forssberg, K.S.E., and Bolin, N.J., 1989a, "**Thermodynamic calculations on iron-containing sulphide mineral flotation systems I: The stability of iron-xanthate."** International Journal of Mineral Processing, 27, pp. 1-19.

Warner, T E, Rice, N M and Taylor, N, 1992. "**Electrochemical study of the dissolution of pentlandite,**" Hydrometallurgy, Vol. 31, pg. 55-90.

Wiese, J., Harris, P., Bradshaw, D., 2008, "**The use of very low molecular weight polysaccharides as depressants in PGM flotation,**" in press

Wiese, J., Harris, P., Bradshaw, D., 2006a, "**The Role of the Reagent Suite in Optimising Pentlandite Recoveries from the Merensky Reef,**" Minerals Engineering, Vol. 19, pp. 1290-1300.

Wiese, J., Becker, M., Harris, P., and Bradshaw, D., 2006b, "**Interpreting the role of reagents in the flotation of platinum-bearing Merensky ores,**" International Platinum Conference 'Platinum Surges Ahead', The Southern African Institute of Mining and Metallurgy, 8-12 October.

Wiese, J., Harris, P., Bradshaw, D., 2005, "**The Influence of the reagent suite in the flotation of ores from the Merensky reef,**" Minerals Engineering, Vol. 18, pp. 189-198.

Wills, B.A. and Napier-Munn, T., 2006, "**Froth flotation,**" In Wills' Mineral Processing Technology: An introduction to the practical aspects of ore treatment and mineral recovery, Seventh Edition, Chapter 12, pp 267- 352.

http://www.lsbu.ac.uk/water/hycmc.html (last visited 30/05/2007)
http://www.lsbu.ac.uk/water/hygua.html (last visited 30/05/2007)
http://www.lsbu.ac.uk/water/hysta.html (last visited 30/05/2007)
Ridge Mining, 2008, website:
http://www.ridgemining.com/02-shebasridge.html, last accessed 12-Februsry-2008

References

Johnson Matthey, 2008, website: http://www.platinum.matthey.com/production/1048863442.html (last accessed 16-July-08)

Anglo Platinum, 2008, website: http://www.angloplatinum.com/def_main.asp?Id=about_us/au_com_profile/au_cp_location/au_cp_location.asp&Related=true (last accessed 16-July-08)

Appendix A: Acid Digestion procedure

Acid Digestion procedure:

1 Four parts of hydrochloric acid was to be mixed with one part of hydrofluoric acid per volume, to make up an HCl/HF acid mixture. The acids used were of the following compositions:

   a. Hydrofluoric acid, HF (approx. 40 %)
   b. Nitric Acid, HNO_3 (approx 60 %)
   c. Perchloric acid, $HClO_4$ (concentrated)
   d. Hydrochloric acid, HCl (approx 30 %)

2 Prepare the solid samples to be digested by weighing out the following amounts for digestion into a 250 m*l* Erlenmeyer flask:

   a. Weigh 0.1 g of concentrate sample into Erlenmeyer flask
   b. Weigh 0.5 g of tailing sample into Erlenmeyer flask

   * Carefully record the exact mass of each sample that is used

3 Digest the solid samples as follows:

   a. Add 10 m*l* of HCl/HF acid mixture into the flask containing sample and heat up the mixture to boiling on the hot plate
   b. Add 10 m*l* of HNO_3 into the flask and boil until the sample volume is approx. 2 m*l*
   c. Add 5 m*l* of $HClO_4$ to flask and boil until sample volume is approx. 2 m*l* (a white cloud will form which will rise once the reaction has taken place)

4 Prepare/Make up volume and filter sample as follows:

   a. Transfer the digested sample quantitatively from Erlenmeyer flask to a 100 m*l* volumetric flask and make up the volume to 100ml using distilled water
   b. Filter the sample into a 100 m*l* sample bottle and submit for chemical analysis.

The samples were analysed for the different elements using the Varian AA110 at UCT. From this elemental chemical analysis, elemental recoveries could then be

calculated. The Nkomati sample also contained a reasonable amount of magnetite which could interfere with the iron recovery data but the flotation recovery of magnetite is assumed to be negligible and thus the iron data from the analysis is assumed to be sufficient to approximate only the sulphides.

Appendix B: Microflotation Data

Appendix

		Cum Mass	Recovery (%)			Average Recovery (%)			Std Dev		
		g	Fe	Cu	Ni	Fe	Cu	Ni	Fe	Cu	Ni
C1	no reagents	0.5	17.5	22.8	19.9	18.8	23.4	20.7	1.30	0.65	0.80
C2	no reagents	0.97	54.7	51.7	50.8	49.9	54.4	52.1	4.80	2.75	1.25
C3	no reagents	1.38	74.6	81.2	82.5	71.4	82.6	82.9	3.20	1.35	0.45
C4	no reagents	1.61	86.1	97	97.5	84.5	97.4	97.6	1.65	0.45	0.15
Tails	no reagents	1.94									

		Cum Mass	Recovery (%)			Selectivity			Std Error		
		g	Fe	Cu	Ni	Cu/Fe	Cu/Ni	Ni/Fe	Fe	Cu	Ni
C1	no reagents	0.49	20.1	24.1	21.5	1.2	1.1	1.1	0.75	0.38	0.46
C2	no reagents	1.00	45.1	57.2	53.3	1.1	1	1	2.77	1.59	0.72
C3	no reagents	1.41	68.2	83.9	83.4	1.2	1	1.2	1.85	0.78	0.26
C4	no reagents	1.65	82.8	97.9	97.8	1.2	1	1.2	0.95	0.26	0.09
Tails	no reagents	2.01									

		Cum Mass	Recovery (%)			Average Recovery (%)			Std Dev		
		g	Fe	Cu	Ni	Fe	Cu	Ni	Fe	Cu	Ni
C1	SIBX	1.55	76.7	94.3	85.2	73.2	91.4	82.3	3.50	2.85	2.90
C2	SIBX	1.73	86.9	99.1	97	83.8	97.8	94.9	3.10	1.25	2.15
C3	SIBX	1.76	88.4	99.5	98.5	86.3	99.1	97.9	2.15	0.35	0.55
C4	SIBX	1.78	89.4	99.6	98.8	87.7	99.5	98.6	1.75	0.10	0.20
Tails	SIBX	1.96									

		Cum Mass	Recovery (%)			Selectivity			Std Error		
		g	Fe	Cu	Ni	Cu/Fe	Cu/Ni	Ni/Fe	Fe	Cu	Ni
C1	SIBX	1.5	69.7	88.6	79.4	1.2	1.1	1.1	2.02	1.65	1.67
C2	SIBX	1.73	80.7	96.6	92.7	1.2	1	1.1	1.79	0.72	1.24
C3	SIBX	1.8	84.1	98.8	97.4	1.1	1	1.1	1.24	0.20	0.32
C4	SIBX	1.84	85.9	99.4	98.4	1.1	1	1.1	1.01	0.06	0.12
Tails	SIBX	2.08									

		Cum Mass	Recovery (%)			Average Recovery (%)			Std Dev		
		g	Fe	Cu	Ni	Fe	Cu	Ni	Fe	Cu	Ni
C1	SEX	1.03	51.7	78.1	62.4	52.3	76.7	63.6	0.60	1.45	1.20
C2	SEX	1.47	72.8	95.7	87.2	73.3	95.4	88.4	0.50	0.30	1.20
C3	SEX	1.63	81.4	98.3	94.7	80.8	98.3	95.1	0.65	0.06	0.45
C4	SEX	1.71	85.7	99.3	97.7	85.2	99.4	98.1	0.45	0.10	0.45
Tails	SEX	1.97									

		Cum Mass	Recovery (%)			Selectivity			Std Error		
		g	Fe	Cu	Ni	Cu/Fe	Cu/Ni	Ni/Fe	Fe	Cu	Ni
C1	SEX	1.1	52.9	75.2	64.8	1.5	1.2	1.2	0.35	0.84	0.69
C2	SEX	1.53	73.8	95.1	89.6	1.3	1.1	1.2	0.29	0.17	0.69
C3	SEX	1.66	80.1	98.4	95.6	1.2	1	1.2	0.38	0.03	0.26
C4	SEX	1.75	84.8	99.5	98.6	1.2	1	1.2	0.26	0.06	0.26
Tails	SEX	2.01									

		Cum Mass	Recovery (%)			Average Recovery (%)			Std Dev		
		g	Fe	Cu	Ni	Fe	Cu	Ni	Fe	Cu	Ni
C1	SNPX	1.33	66.6	82.7	71	62.4	84.9	70.3	4.20	2.25	0.70
C2	SNPX	1.59	80.7	95	89.3	78.9	96.1	90	1.80	1.10	0.65
C3	SNPX	1.67	85	96.9	95.1	84.3	97.7	95.8	0.70	0.85	0.65
C4	SNPX	1.71	87.4	98.1	97.7	86.9	98.6	98	0.50	0.50	0.30
Tails	SNPX	1.96									

		Cum Mass	Recovery (%)			Selectivity			Std Error		
		g	Fe	Cu	Ni	Cu/Fe	Cu/Ni	Ni/Fe	Fe	Cu	Ni
C1	SNPX	1.22	58.2	87.2	69.6	1.4	1.2	1.1	2.42	1.30	0.40
C2	SNPX	1.6	77.1	97.2	90.6	1.2	1.1	1.1	1.04	0.64	0.38
C3	SNPX	1.72	83.6	98.6	96.4	1.2	1	1.1	0.40	0.49	0.38
C4	SNPX	1.78	86.4	99.1	98.3	1.1	1	1.1	0.29	0.29	0.17
Tails	SNPX	2.00									

		Cum Mass	Recovery (%)			Average Recovery (%)			Std Dev		
		g	Fe	Cu	Ni	Fe	Cu	Ni	Fe	Cu	Ni
C1	PAX	1.44	68.6	88.5	78.5	72.5	90.3	82.2	3.85	1.80	3.65
C2	PAX	1.66	80.4	95.3	91.9	83.6	96.7	94.1	3.20	1.40	2.20
C3	PAX	1.73	84.3	96.7	95.9	86.7	97.8	96.8	2.40	1.05	0.90
C4	PAX	1.79	87.7	97.7	97.9	90.5	98.5	98.2	2.75	0.80	0.35
Tails	PAX	1.98									

		Cum Mass	Recovery (%)			Selectivity			Std Error		
		g	Fe	Cu	Ni	Cu/Fe	Cu/Ni	Ni/Fe	Fe	Cu	Ni
C1	PAX	1.51	76.3	92.1	85.8	1.2	1.1	1.1	2.22	1.04	2.11
C2	PAX	1.7	86.8	98.1	96.3	1.2	1	1.1	1.85	0.81	1.27
C3	PAX	1.74	89.1	98.8	97.7	1.1	1	1.1	1.39	0.61	0.52
C4	PAX	1.81	93.2	99.3	98.6	1.1	1	1.1	1.59	0.46	0.20
Tails	PAX	1.99									

		Cum Mass	Recovery (%)			Average Recovery (%)			Std Dev		
		g	Fe	Cu	Ni	Fe	Cu	Ni	Fe	Cu	Ni
C1	SIBX + Guar	0.87	40.1	66	55.6	39.4	65.9	55.5	0.65	0.10	0.06
C2	SIBX + Guar	1.4	65.7	89.5	83.9	65.9	91.3	85.5	0.20	1.80	1.65
C3	SIBX + Guar	1.65	78.4	97.3	95.4	78.1	97.5	95.5	0.30	0.20	0.15
C4	SIBX + Guar	1.76	84.3	99.2	98.9	84	99.2	98.7	0.30	0.06	0.15
Tails	SIBX + Guar	2.04									

		Cum Mass	Recovery (%)			Selectivity			Std Error		
		g	Fe	Cu	Ni	Cu/Fe	Cu/Ni	Ni/Fe	Fe	Cu	Ni
C1	SIBX + Guar	0.86	38.8	65.8	55.5	1.7	1.2	1.4	0.38	0.06	0.03
C2	SIBX + Guar	1.44	66.1	93.1	87.2	1.4	1.1	1.3	0.12	1.04	0.95
C3	SIBX + Guar	1.67	77.8	97.7	95.7	1.2	1	1.2	0.17	0.12	0.09
C4	SIBX + Guar	1.78	83.7	99.1	98.6	1.2	1	1.2	0.17	0.03	0.09
Tails	SIBX + Guar	2.06									

		Cum Mass g	Recovery (%) Fe	Cu	Ni	Average Recovery (%) Fe	Cu	Ni	Std Dev Fe	Cu	Ni
C1	SEX + Guar	0.14	5.8	15.5	8.9	5.6	13.1	7.6	0.15	2.40	1.30
C2	SEX + Guar	0.25	10	28.1	15.4	9.3	22.7	12.5	0.65	5.45	2.90
C3	SEX + Guar	0.36	14.3	42	22.5	13.3	33.3	18	1.05	8.65	4.45
C4	SEX + Guar	0.53	21.2	59.9	35	19.3	47.7	28.1	1.95	12.15	6.95
Tails	SEX + Guar	2.06									

		Cum Mass g	Recovery (%) Fe	Cu	Ni	Selectivity Cu/Fe	Cu/Ni	Ni/Fe	Std Error Fe	Cu	Ni
C1	SEX + Guar	0.13	5.5	10.7	6.3	2.3	1.7	1.3	0.09	1.39	0.75
C2	SEX + Guar	0.2	8.7	17.2	9.6	2.4	1.8	1.3	0.38	3.15	1.67
C3	SEX + Guar	0.29	12.2	24.7	13.6	2.5	1.8	1.4	0.61	4.99	2.57
C4	SEX + Guar	0.42	17.3	35.6	21.1	2.5	1.7	1.5	1.13	7.01	4.01
Tails	SEX + Guar	2.04									

		Cum Mass g	Recovery (%) Fe	Cu	Ni	Average Recovery (%) Fe	Cu	Ni	Std Dev Fe	Cu	Ni
C1	SNPX + Guar	0.43	18.9	42.6	32.4	19.8	43.4	33.7	0.95	0.80	1.30
C2	SNPX + Guar	0.87	40.1	71.1	60.7	40.1	72.6	62	0.06	1.45	1.30
C3	SNPX + Guar	1.29	60.3	89.2	85.5	59.8	89.7	85.4	0.50	0.55	0.15
C4	SNPX + Guar	1.5	70.4	96.3	94.5	71.5	96.9	94.8	1.15	0.65	0.30
Tails	SNPX + Guar	1.99									

		Cum Mass g	Recovery (%) Fe	Cu	Ni	Selectivity Cu/Fe	Cu/Ni	Ni/Fe	Std Error Fe	Cu	Ni
C1	SNPX + Guar	0.46	20.8	44.2	35	2.2	1.3	1.7	0.55	0.46	0.75
C2	SNPX + Guar	0.88	40	74	63.3	1.8	1.2	1.5	0.03	0.84	0.75
C3	SNPX + Guar	1.27	59.3	90.3	85.2	1.5	1.1	1.4	0.29	0.32	0.09
C4	SNPX + Guar	1.54	72.7	97.6	95.1	1.4	1	1.3	0.66	0.38	0.17
Tails	SNPX + Guar	2.01									

		Cum Mass g	Recovery (%) Fe	Cu	Ni	Average Recovery (%) Fe	Cu	Ni	Std Dev Fe	Cu	Ni
C1	PAX + Guar	1.13	56.3	73.1	61.8	51.9	69	59.9	4.35	4.15	1.85
C2	PAX + Guar	1.59	78.4	94.4	91.7	75.6	91.6	89.3	2.85	2.75	2.40
C3	PAX + Guar	1.73	85.5	98.1	98.2	84.2	97.6	97.4	1.30	0.45	0.85
C4	PAX + Guar	1.78	88.3	99.2	99.3	88.1	99.2	99.2	0.20	0.06	0.10
Tails	PAX + Guar	1.98									

		Cum Mass g	Recovery (%) Fe	Cu	Ni	Selectivity Cu/Fe	Cu/Ni	Ni/Fe	Std Error Fe	Cu	Ni
C1	PAX + Guar	1.02	47.6	64.8	58.1	1.3	1.2	1.2	2.51	2.40	1.07
C2	PAX + Guar	1.55	72.7	88.9	86.9	1.2	1	1.2	1.65	1.59	1.39
C3	PAX + Guar	1.77	82.9	97.2	96.5	1.2	1	1.2	0.75	0.26	0.49
C4	PAX + Guar	1.87	87.9	99.3	99.1	1.1	1	1.1	0.12	0.03	0.06
Tails	PAX + Guar	2.1									

Appendix

		Cum Mass	Recovery (%)			Average Recovery (%)			Std Dev		
		g	Fe	Cu	Ni	Fe	Cu	Ni	Fe	Cu	Ni
C1	SIBX + CMC	1.36	67.8	87.1	71.7	65	86.6	71.6	2.85	0.50	0.15
C2	SIBX + CMC	1.65	82.5	97.1	93	80.4	96.7	93	2.15	0.35	0.06
C3	SIBX + CMC	1.74	86.9	98.6	97.6	84.3	98.1	97	2.60	0.50	0.60
C4	SIBX + CMC	1.77	89	99.3	99.1	87.8	99.2	99.1	1.15	0.10	0.06
Tails	SIBX + CMC	1.98									

		Cum Mass	Recovery (%)			Selectivity			Std Error		
		g	Fe	Cu	Ni	Cu/Fe	Cu/Ni	Ni/Fe	Fe	Cu	Ni
C1	SIBX + CMC	1.32	62.1	86.1	71.4	1.3	1.2	1.1	1.65	0.29	0.09
C2	SIBX + CMC	1.67	78.2	96.4	93.1	1.2	1	1.2	1.24	0.20	0.03
C3	SIBX + CMC	1.74	81.7	97.6	96.4	1.2	1	1.2	1.50	0.29	0.35
C4	SIBX + CMC	1.84	86.7	99.1	99.2	1.1	1	1.1	0.66	0.06	0.03
Tails	SIBX + CMC	2.07									

		Cum Mass	Recovery (%)			Average Recovery (%)			Std Dev		
		g	Fe	Cu	Ni	Fe	Cu	Ni	Fe	Cu	Ni
C1	SEX + CMC	0.5	23.1	57.9	29.2	21.2	55.5	27.2	1.95	2.45	2.00
C2	SEX + CMC	0.82	38.5	78.3	54.2	36	78.7	52.5	2.55	0.40	1.65
C3	SEX + CMC	1.14	55.1	90.4	75.2	52.2	91.4	74.6	2.95	1.05	0.60
C4	SEX + CMC	1.45	71.2	97.7	92.2	70.2	97.9	92.1	1.00	0.25	0.10
Tails	SEX + CMC	1.97									

		Cum Mass	Recovery (%)			Selectivity			Std Error		
		g	Fe	Cu	Ni	Cu/Fe	Cu/Ni	Ni/Fe	Fe	Cu	Ni
C1	SEX + CMC	0.43	19.2	53	25.2	2.6	2	1.3	1.13	1.41	1.15
C2	SEX + CMC	0.73	33.4	79.1	50.9	2.2	1.5	1.5	1.47	0.23	0.95
C3	SEX + CMC	1.05	49.2	92.5	74	1.8	1.2	1.4	1.70	0.61	0.35
C4	SEX + CMC	1.43	69.2	98.2	92	1.4	1.1	1.3	0.58	0.15	0.06
Tails	SEX + CMC	1.99									

		Cum Mass	Recovery (%)			Average Recovery (%)			Std Dev		
		g	Fe	Cu	Ni	Fe	Cu	Ni	Fe	Cu	Ni
C1	SNPX + CMC	0.8	37.9	74.4	50.7	41.6	74.5	56.6	3.70	0.10	5.95
C2	SNPX + CMC	1.37	67.6	94.3	83.7	68.2	93.7	86.8	0.65	0.55	3.10
C3	SNPX + CMC	1.62	81.4	98.3	95.3	80.6	97.9	96	0.80	0.35	0.75
C4	SNPX + CMC	1.74	87.4	99.3	98.6	86	99.2	98.7	1.40	0.15	0.15
Tails	SNPX + CMC	2.01									

		Cum Mass	Recovery (%)			Selectivity			Std Error		
		g	Fe	Cu	Ni	Cu/Fe	Cu/Ni	Ni/Fe	Fe	Cu	Ni
C1	SNPX + CMC	0.95	45.3	74.6	62.6	1.8	1.3	1.4	2.14	0.06	3.44
C2	SNPX + CMC	1.43	68.9	93.2	89.9	1.4	1.1	1.3	0.38	0.32	1.79
C3	SNPX + CMC	164	79.8	97.6	96.8	1.2	1	1.2	0.46	0.20	0.43
C4	SNPX + CMC	1.73	84.6	99	98.9	1.2	1	1.1	0.81	0.09	0.09
Tails	SNPX + CMC	1.99									

Appendix

		Cum Mass	Recovery (%)			Average Recovery (%)			Std Dev		
		g	Fe	Cu	Ni	Fe	Cu	Ni	Fe	Cu	Ni
C1	PAX + CMC	1.27	62.7	85.7	67.8	61.1	87.4	69	1.60	1.75	1.20
C2	PAX + CMC	1.65	80.5	97.3	92.6	79.2	97.1	92.1	1.25	0.20	0.55
C3	PAX + CMC	1.76	85.9	99.1	98.3	85.1	98.7	97.9	0.75	0.40	0.35
C4	PAX + CMC	1.8	88.2	99.5	99.3	87.9	99.3	99.3	0.25	0.15	0.06
Tails	PAX + CMC	1.99									

		Cum Mass	Recovery (%)			Selectivity			Std Error		
		g	Fe	Cu	Ni	Cu/Fe	Cu/Ni	Ni/Fe	Fe	Cu	Ni
C1	PAX + CMC	1.25	59.5	89.2	70.2	1.4	1.3	1.1	0.92	1.01	0.69
C2	PAX + CMC	1.62	78	96.9	91.5	1.2	1.1	1.2	0.72	0.12	0.32
C3	PAX + CMC	1.75	84.4	98.3	97.6	1.2	1	1.2	0.43	0.23	0.20
C4	PAX + CMC	1.81	87.7	99.2	99.2	1.1	1	1.1	0.15	0.09	0.03
Tails	PAX + CMC	2.03									

		Cum Mass	Recovery (%)			Average Recovery (%)			Std Dev		
		g	Fe	Cu	Ni	Fe	Cu	Ni	Fe	Cu	Ni
C1	SIBX + Starch	1.11	54.7	71.5	64.5	49.6	67.6	56.6	5.10	3.90	7.95
C2	SIBX + Starch	1.43	71.6	88.3	81.8	68.3	86.7	77.9	3.30	1.55	3.90
C3	SIBX + Starch	1.56	78.5	94.2	89.1	77.2	93.8	88	1.30	0.35	1.10
C4	SIBX + Starch	1.66	83.9	97.4	95.2	83.2	97.2	94.6	0.70	0.15	0.60
Tails	SIBX + Starch	1.96									

		Cum Mass	Recovery (%)			Selectivity			Std Error		
		g	Fe	Cu	Ni	Cu/Fe	Cu/Ni	Ni/Fe	Fe	Cu	Ni
C1	SIBX + Starch	0.85	44.5	63.7	48.6	1.4	1.2	1.1	2.94	2.25	4.59
C2	SIBX + Starch	1.28	65	85.2	74	1.3	1.1	1.1	1.91	0.90	2.25
C3	SIBX + Starch	1.5	75.9	93.5	86.9	1.2	1.1	1.1	0.75	0.20	0.64
C4	SIBX + Starch	1.63	82.5	97.1	94	1.2	1	1.1	0.40	0.09	0.35
Tails	SIBX + Starch	1.98									

		Cum Mass	Recovery (%)			Average Recovery (%)			Std Dev		
		g	Fe	Cu	Ni	Fe	Cu	Ni	Fe	Cu	Ni
C1	SEX + Starch	0.44	26.3	33.4	28.7	26.1	42.7	29.6	0.15	9.25	0.85
C2	SEX + Starch	0.85	51.1	60.5	55	49.9	69.1	58	1.20	8.65	3.00
C3	SEX + Starch	1.14	69	73.9	71.7	67.5	82	77.4	1.50	8.05	5.75
C4	SEX + Starch	1.45	88.3	87.4	88.9	83.2	91.4	91.5	5.05	4.00	2.65
Tails	SEX + Starch	1.96									

		Cum Mass	Recovery (%)			Selectivity			Std Error		
		g	Fe	Cu	Ni	Cu/Fe	Cu/Ni	Ni/Fe	Fe	Cu	Ni
C1	SEX + Starch	0.55	26	51.9	30.4	1.6	1.4	1.1	0.09	5.34	0.49
C2	SEX + Starch	1.01	48.7	77.8	61	1.4	1.2	1.2	0.69	4.99	1.73
C3	SEX + Starch	1.35	66	90	83.2	1.2	1.1	1.1	0.87	4.65	3.32
C4	SEX + Starch	1.58	78.2	95.4	94.2	1.1	1	1.1	2.92	2.31	1.53
Tails	SEX + Starch	1.99									

		Cum Mass	Recovery (%)			Average Recovery (%)			Std Dev		
		g	Fe	Cu	Ni	Fe	Cu	Ni	Fe	Cu	Ni
C1	SNPX + Starch	0.63	31	41.1	36.4	31.4	38.5	30.3	0.45	2.60	6.10
C2	SNPX + Starch	1.01	49.6	62.4	61.2	50.2	61.7	58.1	0.65	0.70	3.10
C3	SNPX + Starch	1.27	62.3	76.1	78.4	63.2	75.6	76.7	0.90	0.50	1.70
C4	SNPX + Starch	1.48	72.9	88.6	89.5	73.9	88.6	89.3	1.05	0.00	0.20
Tails	SNPX + Starch	1.99									

		Cum Mass	Recovery (%)			Selectivity			Std Error		
		g	Fe	Cu	Ni	Cu/Fe	Cu/Ni	Ni/Fe	Fe	Cu	Ni
C1	SNPX + Starch	0.62	31.9	35.9	24.2	1.2	1.3	1	0.26	1.50	3.52
C2	SNPX + Starch	1.02	50.9	61	55	1.2	1.1	1.2	0.38	0.40	1.79
C3	SNPX + Starch	1.29	64.1	75.1	75	1.2	1	1.2	0.52	0.29	0.98
C4	SNPX + Starch	1.51	75	88.6	89.1	1.2	1	1.2	0.61	0.00	0.12
Tails	SNPX + Starch	1.98									

		Cum Mass	Recovery (%)			Average Recovery (%)			Std Dev		
		g	Fe	Cu	Ni	Fe	Cu	Ni	Fe	Cu	Ni
C1	PAX + Starch	1.37	67.3	82.5	73.6	64.8	83.6	72.4	2.55	1.10	1.20
C2	PAX + Starch	1.66	81.5	95.3	92.8	80.3	95.4	91.8	1.25	0.10	1.05
C3	PAX + Starch	1.75	86.9	98.2	97.5	85.7	98.4	96.7	1.20	0.15	0.80
C4	PAX + Starch	1.78	88.3	99	98.7	87.3	99.2	98.4	0.95	0.15	0.35
Tails	PAX + Starch	2									

		Cum Mass	Recovery (%)			Selectivity			Std Error		
		g	Fe	Cu	Ni	Cu/Fe	Cu/Ni	Ni/Fe	Fe	Cu	Ni
C1	PAX + Starch	1.27	62.2	84.7	71.2	1.3	1.2	1.1	1.47	0.64	0.69
C2	PAX + Starch	1.6	79	95.5	90.7	1.2	1	1.1	0.72	0.06	0.61
C3	PAX + Starch	1.7	84.5	98.5	95.9	1.1	1	1.1	0.69	0.09	0.46
C4	PAX + Starch	1.74	86.4	99.3	98	1.1	1	1.1	0.55	0.09	0.20
Tails	PAX + Starch	1.99									

		Cum Mass	Recovery (%)			Average Recovery (%)			Std Dev		
		g	Fe	Cu	Ni	Fe	Cu	Ni	Fe	Cu	Ni
C1	(SNPX $^{(10\text{-}5M)}$ Guar $^{(10ppm)}$) 2min oxidation	0.57	27.9	38.34	26.88	24.88	38.58	24.2	3.02	0.25	2.68
C2	(SNPX $^{(10\text{-}5M)}$ Guar $^{(10ppm)}$) 2min oxidation	1.01	49.81	64.12	54.81	48.3	68.4	56.24	1.51	4.29	1.44
C3	(SNPX $^{(10\text{-}5M)}$ Guar $^{(10ppm)}$) 2min oxidation	1.38	68.51	82.98	79.98	64.85	84.44	79.03	3.66	1.46	0.96
C4	(SNPX $^{(10\text{-}5M)}$ Guar $^{(10ppm)}$) 2min oxidation	1.60	79.8	92.44	91.44	74.76	92.29	89.95	5.04	0.15	1.49
Tails	(SNPX $^{(10\text{-}5M)}$ Guar $^{(10ppm)}$) 2min oxidation	2.01									

		Cum Mass	Recovery (%)			Selectivity			Std Error		
		g	Fe	Cu	Ni	Ni/Fe	Cu/Ni	Cu/Fe	Fe	Cu	Ni
C1	(SNPX $^{(10\text{-}5M)}$ Guar $^{(10ppm)}$) 2min oxidation	0.52	21.87	38.83	21.52	0.97	1.59	1.55	1.74	0.14	1.55
C2	(SNPX $^{(10\text{-}5M)}$ Guar $^{(10ppm)}$) 2min oxidation	1.12	46.79	72.69	57.68	1.16	1.22	1.42	0.87	2.47	0.83
C3	(SNPX $^{(10\text{-}5M)}$ Guar $^{(10ppm)}$) 2min oxidation	1.46	61.2	85.9	78.07	1.22	1.07	1.3	2.11	0.84	0.55
C4	(SNPX $^{(10\text{-}5M)}$ Guar $^{(10ppm)}$) 2min oxidation	1.66	69.72	92.14	88.46	1.2	1.03	1.23	2.91	0.09	0.86
Tails	(SNPX $^{(10\text{-}5M)}$ Guar $^{(10ppm)}$) 2min oxidation	2.01									

		Cum Mass	Recovery (%)			Average Recovery (%)			Std Dev		
		g	Fe	Cu	Ni	Fe	Cu	Ni	Fe	Cu	Ni
C1	(SNPX $^{(10\text{-}5M)}$ Guar $^{(10ppm)}$) 5min oxidation	0.92	45.87	59.12	49.26	46.24	61.32	51.29	0.38	2.20	2.03
C2	(SNPX $^{(10\text{-}5M)}$ Guar $^{(10ppm)}$) 5min oxidation	1.44	71.44	87.91	82.21	71.16	87.91	82.11	0.28	0.00	0.10
C3	(SNPX $^{(10\text{-}5M)}$ Guar $^{(10ppm)}$) 5min oxidation	1.65	80.41	94.82	92.89	80.4	95.14	93.03	0.01	0.32	0.14
C4	(SNPX $^{(10\text{-}5M)}$ Guar $^{(10ppm)}$) 5min oxidation	1.75	85.41	97.99	97.74	85.47	98.19	97.79	0.06	0.20	0.05
Tails	(SNPX $^{(10\text{-}5M)}$ Guar $^{(10ppm)}$) 5min oxidation	2.02									

		Cum Mass	Recovery (%)			Selectivity			Std Error		
		g	Fe	Cu	Ni	Ni/Fe	Cu/Ni	Cu/Fe	Fe	Cu	Ni
C1	(SNPX $^{(10\text{-}5M)}$ Guar $^{(10ppm)}$) 5min oxidation	0.98	46.62	63.52	53.32	1.11	1.2	1.33	0.22	1.27	1.17
C2	(SNPX $^{(10\text{-}5M)}$ Guar $^{(10ppm)}$) 5min oxidation	1.48	70.88	87.91	82.01	1.15	1.07	1.24	0.16	0.00	0.06
C3	(SNPX $^{(10\text{-}5M)}$ Guar $^{(10ppm)}$) 5min oxidation	1.67	80.4	95.45	93.17	1.16	1.02	1.18	0.00	0.18	0.08
C4	(SNPX $^{(10\text{-}5M)}$ Guar $^{(10ppm)}$) 5min oxidation	1.77	85.53	98.39	97.84	1.14	1	1.15	0.03	0.12	0.03
Tails	(SNPX $^{(10\text{-}5M)}$ Guar $^{(10ppm)}$) 5min oxidation	2.02									

		Cum Mass	Recovery (%)			Average Recovery (%)			Std Dev		
		g	Fe	Cu	Ni	Fe	Cu	Ni	Fe	Cu	Ni
C1	(SNPX $^{(10\text{-}5M)}$ Guar $^{(10ppm)}$) 10min oxidation	1.28	61.55	82.91	70.99	57.66	77.41	65.89	3.89	5.50	5.10
C2	(SNPX $^{(10\text{-}5M)}$ Guar $^{(10ppm)}$) 10min oxidation	1.64	80.36	94.98	91.28	78.88	94.01	90.00	1.48	0.98	1.28
C3	(SNPX $^{(10\text{-}5M)}$ Guar $^{(10ppm)}$) 10min oxidation	1.73	85.11	97.59	96.53	84.73	97.25	96.34	0.38	0.34	0.19
C4	(SNPX $^{(10\text{-}5M)}$ Guar $^{(10ppm)}$) 10min oxidation	1.78	87.76	98.59	98.68	87.7	98.48	98.47	0.06	0.11	0.21
Tails	(SNPX $^{(10\text{-}5M)}$ Guar $^{(10ppm)}$) 10min oxidation	1.98									

		Cum Mass	Recovery (%)			Selectivity			Std Error		
		g	Fe	Cu	Ni	Ni/Fe	Cu/Ni	Cu/Fe	Fe	Cu	Ni
C1	(SNPX $^{(10\text{-}5M)}$ Guar $^{(10ppm)}$) 10min oxidation	1.10	53.77	71.92	60.79	1.14	1.17	1.34	2.25	3.17	2.94
C2	(SNPX $^{(10\text{-}5M)}$ Guar $^{(10ppm)}$) 10min oxidation	1.56	77.41	93.03	88.72	1.14	1.04	1.19	0.85	0.56	0.74
C3	(SNPX $^{(10\text{-}5M)}$ Guar $^{(10ppm)}$) 10min oxidation	1.69	84.36	96.92	96.15	1.14	1.01	1.15	0.22	0.19	0.11
C4	(SNPX $^{(10\text{-}5M)}$ Guar $^{(10ppm)}$) 10min oxidation	1.75	87.64	98.37	98.26	1.12	1	1.12	0.03	0.06	0.12
Tails	(SNPX $^{(10\text{-}5M)}$ Guar $^{(10ppm)}$) 10min oxidation	1.97									

		Cum Mass g	Recovery (%) Fe	Cu	Ni	Average Recovery (%) Fe	Cu	Ni	Std Dev Fe	Cu	Ni
C1	SNPX (5min) Guar (5min), repeat	0.43	16.53	30	16.46	21.1	32.12	21.02	4.57	2.13	4.56
C2	SNPX (5min) Guar (5min), repeat	0.86	37.65	56.62	43.25	42.65	58.74	49.1	5.00	2.12	5.85
C3	SNPX (5min) Guar (5min), repeat	1.29	57.92	78.47	71.44	61.36	78.13	73.66	3.44	0.34	2.22
C4	SNPX (5min) Guar (5min), repeat	1.50	74.01	92.62	91.12	74.8	92.1	91.87	0.79	0.52	0.75
Tails	SNPX (5min) Guar (5min), repeat	1.99									

		Cum Mass g	Recovery (%) Fe	Cu	Ni	Selectivity Cu/Fe	Cu/Ni	Ni/Fe	Std Error Fe	Cu	Ni
C1	SNPX (5min) Guar (5min), repeat	0.46	25.67	34.25	25.57	1.52	1.53	1	2.64	1.23	2.63
C2	SNPX (5min) Guar (5min), repeat	0.88	47.65	60.85	54.95	1.38	1.2	1.15	2.89	1.22	3.38
C3	SNPX (5min) Guar (5min), repeat	1.27	64.8	77.79	75.88	1.27	1.06	1.2	1.99	0.20	1.28
C4	SNPX (5min) Guar (5min), repeat	1.54	75.59	91.58	92.61	1.23	1	1.23	0.46	0.30	0.43
Tails	SNPX (5min) Guar (5min), repeat	2.01									

		Cum Mass g	Recovery (%) Fe	Cu	Ni	Average Recovery (%) Fe	Cu	Ni	Std Dev Fe	Cu	Ni
C1	SNPX (5min) Guar (5min) 5min oxidation (pH 10)	0.18	7.93	21.11	15.96	7.3	19.53	15.57	0.63	1.59	0.39
C2	SNPX (5min) Guar (5min) 5min oxidation (pH 10)	0.40	17.66	44.56	35.23	17.47	42.44	36.37	0.19	2.12	1.14
C3	SNPX (5min) Guar (5min) 5min oxidation (pH 10)	0.67	30.2	68.23	58.1	30.24	65.86	59.35	0.05	2.37	1.26
C4	SNPX (5min) Guar (5min) 5min oxidation (pH 10)	1.05	49.16	88.1	81.15	49.2	86.01	80.99	0.05	2.09	0.16
Tails	SNPX (5min) Guar (5min) 5min oxidation (pH 10)	1.98									

		Cum Mass g	Recovery (%) Fe	Cu	Ni	Selectivity Cu/Fe	Cu/Ni	Ni/Fe	Std Error Fe	Cu	Ni
C1	SNPX (5min) Guar (5min) 5min oxidation (pH 10)	0.15	6.68	17.94	15.19	2.67	1.25	2.13	0.36	0.92	0.22
C2	SNPX (5min) Guar (5min) 5min oxidation (pH 10)	0.39	17.28	40.32	37.5	2.43	1.17	2.08	0.11	1.22	0.66
C3	SNPX (5min) Guar (5min) 5min oxidation (pH 10)	0.71	30.29	63.49	60.61	2.18	1.11	1.96	0.03	1.37	0.72
C4	SNPX (5min) Guar (5min) 5min oxidation (pH 10)	1.07	49.25	83.93	80.83	1.75	1.06	1.65	0.03	1.20	0.09
Tails	SNPX (5min) Guar (5min) 5min oxidation (pH 10)	1.97									

		Cum Mass	Recovery (%)			Average Recovery (%)			Std Dev		
		g	Fe	Cu	Ni	Fe	Cu	Ni	Fe	Cu	Ni
C1	no reagents (10 min oxid.)	0.29	14.07	22.53	16.83	15.83	33.91	23.28	1.77	11.38	6.45
C2	no reagents (10 min oxid.)	0.68	32.73	52.98	43.75	34.47	62.57	49.26	1.74	9.59	5.51
C3	no reagents (10 min oxid.)	1.13	54.99	79.29	73.49	57.31	83.41	75.47	2.33	4.12	1.99
C4	no reagents (10 min oxid.)	1.50	74.09	94.68	93.98	75.23	95.8	93.61	1.15	1.12	0.38
Tails	no reagents (10 min oxid.)	1.95									

		Cum Mass	Recovery (%)			Selectivity			Std Error		
		g	Fe	Cu	Ni	Cu/Fe	Cu/Ni	Ni/Fe	Fe	Cu	Ni
C1	no reagents (10 min oxid.)	0.39	17.6	45.29	29.72	2.14	1.46	1.47	1.02	6.57	3.72
C2	no reagents (10 min oxid.)	0.78	36.21	72.16	54.77	1.82	1.27	1.43	1.00	5.54	3.18
C3	no reagents (10 min oxid.)	1.24	59.64	87.53	77.46	1.46	1.11	1.32	1.34	2.38	1.15
C4	no reagents (10 min oxid.)	1.57	76.38	96.92	93.23	1.27	1.02	1.24	0.66	0.65	0.22
Tails	no reagents (10 min oxid.)	1.99									

		Cum Mass	Recovery (%)			Average Recovery (%)			Std Dev		
		g	Fe	Cu	Ni	Fe	Cu	Ni	Fe	Cu	Ni
C1	no reagents (5 min oxid.)	0.55	26.01	35.66	24.47	23.03	35.27	22.7	2.98	0.40	1.77
C2	no reagents (5 min oxid.)	0.96	46.56	63.77	52.68	42.9	65.26	50.64	3.66	1.49	2.04
C3	no reagents (5 min oxid.)	1.38	66.53	83.11	77.54	58.95	82.23	72.71	7.58	0.88	4.84
C4	no reagents (5 min oxid.)	1.65	80.09	94.82	92.85	76.99	94.79	92.65	3.11	0.03	0.20
Tails	no reagents (5 min oxid.)	2.01									

		Cum Mass	Recovery (%)			Selectivity			Std Error		
		g	Fe	Cu	Ni	Cu/Fe	Cu/Ni	Ni/Fe	Fe	Cu	Ni
C1	no reagents (5 min oxid.)	0.42	20.05	34.87	20.93	1.53	1.55	0.99	1.72	0.23	1.02
C2	no reagents (5 min oxid.)	0.85	39.24	66.75	48.6	1.52	1.29	1.18	2.11	0.86	1.18
C3	no reagents (5 min oxid.)	1.27	51.37	81.36	67.87	1.39	1.13	1.23	4.38	0.51	2.79
C4	no reagents (5 min oxid.)	1.84	73.88	94.76	92.45	1.23	1.02	1.2	1.79	0.02	0.12
Tails	no reagents (5 min oxid.)	2.27									

		Cum Mass	Recovery (%)			Average Recovery (%)			Std Dev		
		g	Fe	Cu	Ni	Fe	Cu	Ni	Fe	Cu	Ni
C1	no reagents (2 min oxid.)	0.60	31.33	39.69	36.26	28.99	36.81	31.52	2.34	2.89	4.75
C2	no reagents (2 min oxid.)	1.01	50.46	62.1	61.41	48.58	58.61	55.13	1.88	3.49	6.28
C3	no reagents (2 min oxid.)	1.36	70.85	82.74	85.55	66.74	80.54	80.89	4.11	2.20	4.66
C4	no reagents (2 min oxid.)	1.61	81.81	92.9	94.93	78.44	92.4	94.35	3.37	0.50	0.58
Tails	no reagents (2 min oxid.)	1.98									

		Cum Mass	Recovery (%)			Selectivity			Std Error		
		g	Fe	Cu	Ni	Cu/Fe	Cu/Ni	Ni/Fe	Fe	Cu	Ni
C1	no reagents (2 min oxid.)	0.51	26.65	33.92	26.77	1.27	1.17	1.09	1.35	1.67	2.74
C2	no reagents (2 min oxid.)	0.86	46.7	55.13	48.85	1.21	1.06	1.13	1.09	2.01	3.63
C3	no reagents (2 min oxid.)	1.23	62.63	78.34	76.23	1.21	1	1.21	2.37	1.27	2.69
C4	no reagents (2 min oxid.)	1.47	75.07	91.9	93.77	1.18	0.98	1.2	1.95	0.29	0.33
Tails	no reagents (2 min oxid.)	1.97									

		Cum Mass	Recovery (%)			Average Recovery (%)			Std Dev		
		g	Fe	Cu	Ni	Fe	Cu	Ni	Fe	Cu	Ni
C1	SNPX (5 min) + Guar (5 min), 5min oxidation, pH 10 (NaOH)	0.31	7.93	21.11	15.96	7.3	19.53	15.57	0.63	1.59	0.39
C2	SNPX (5 min) + Guar (5 min), 5min oxidation, pH 10 (NaOH)	0.59	17.66	44.56	35.23	17.47	42.44	36.37	0.19	2.12	1.14
C3	SNPX (5 min) + Guar (5 min), 5min oxidation, pH 10 (NaOH)	0.93	30.2	68.23	58.1	30.24	65.86	59.35	0.05	2.37	1.26
C4	SNPX (5 min) + Guar (5 min), 5min oxidation, pH 10 (NaOH)	1.25	49.16	88.1	81.15	49.2	86.01	80.99	0.05	2.09	0.16
Tails	SNPX (5 min) + Guar (5 min), 5min oxidation, pH 10 (NaOH)	1.97									

		Cum Mass	Recovery (%)			Selectivity			Std Error		
		g	Fe	Cu	Ni	Ni/Fe	Cu/Ni	Cu/Fe	Fe	Cu	Ni
C1	SNPX (5 min) + Guar (5 min), 5min oxidation, pH 10 (NaOH)	0.14	6.68	17.94	15.19	2.13	1.25	2.67	0.36	0.92	0.22
C2	SNPX (5 min) + Guar (5 min), 5min oxidation, pH 10 (NaOH)	0.39	17.28	40.32	37.5	2.08	1.17	2.43	0.11	1.22	0.66
C3	SNPX (5 min) + Guar (5 min), 5min oxidation, pH 10 (NaOH)	0.74	30.29	63.49	60.61	1.96	1.11	2.18	0.03	1.37	0.72
C4	SNPX (5 min) + Guar (5 min), 5min oxidation, pH 10 (NaOH)	1.17	49.25	83.93	80.83	1.65	1.06	1.75	0.03	1.20	0.09
Tails	SNPX (5 min) + Guar (5 min), 5min oxidation, pH 10 (NaOH)	2.01									

		Cum Mass	Recovery (%)			Average Recovery (%)			Std Dev		
		g	Fe	Cu	Ni	Fe	Cu	Ni	Fe	Cu	Ni
C1	SNPX (5 min) + Guar (5 min), 5min oxidation, pH 10 (NaOH)	0.27	12.32	27.12	23.14	12.08	29.23	24.19	0.24	2.11	1.05
C2	SNPX (5 min) + Guar (5 min), 5min oxidation, pH 10 (NaOH)	0.55	25.36	49.76	45.26	26	54.54	48.79	0.64	4.78	3.53
C3	SNPX (5 min) + Guar (5 min), 5min oxidation, pH 10 (NaOH)	0.82	38.41	68.93	65.6	40.17	74.29	69.54	1.76	5.36	3.95
C4	SNPX (5 min) + Guar (5 min), 5min oxidation, pH 10 (NaOH)	1.13	53.39	88.97	84.32	54.93	89.93	85.13	1.54	0.96	0.81
Tails	SNPX (5 min) + Guar (5 min), 5min oxidation, pH 10 (NaOH)	1.99									

		Cum Mass	Recovery (%)			Selectivity			Std Error		
		g	Fe	Cu	Ni	Ni/Fe	Cu/Ni	Cu/Fe	Fe	Cu	Ni
C1	SNPX (5 min) + Guar (5 min), 5min oxidation, pH 10 (NaOH)	0.27	11.84	31.34	25.24	2	1.21	2.42	0.14	1.22	0.61
C2	SNPX (5 min) + Guar (5 min), 5min oxidation, pH 10 (NaOH)	0.59	26.63	59.32	52.31	1.88	1.12	2.1	0.37	2.76	2.04
C3	SNPX (5 min) + Guar (5 min), 5min oxidation, pH 10 (NaOH)	0.90	41.93	79.64	73.49	1.73	1.07	1.85	1.02	3.09	2.28
C4	SNPX (5 min) + Guar (5 min), 5min oxidation, pH 10 (NaOH)	1.18	56.47	90.88	85.94	1.55	1.06	1.64	0.89	0.55	0.47
Tails	SNPX (5 min) + Guar (5 min), 5min oxidation, pH 10 (NaOH)	1.98									

		Cum Mass	Recovery (%)			Average Recovery (%)			Std Dev		
		g	Fe	Cu	Ni	Fe	Cu	Ni	Fe	Cu	Ni
C1	SNPX (5 min) + Guar (5 min), 5min oxidation, pH 10 (NaOH)	0.25	11.45	28.41	24.63	11.42	28.16	22.2	0.04	0.25	2.43
C2	SNPX (5 min) + Guar (5 min), 5min oxidation, pH 10 (NaOH)	0.54	26.27	53.42	49.88	24.52	50.43	43.17	1.75	3.00	6.72
C3	SNPX (5 min) + Guar (5 min), 5min oxidation, pH 10 (NaOH)	0.88	43.18	74.9	72.31	41.37	72.08	66.1	1.82	2.82	6.21
C4	SNPX (5 min) + Guar (5 min), 5min oxidation, pH 10 (NaOH)	1.21	60.7	91.41	88.27	59.19	89.34	83.98	1.51	2.08	4.29
Tails	SNPX (5 min) + Guar (5 min), 5min oxidation, pH 10 (NaOH)	1.95									

		Cum Mass	Recovery (%)			Selectivity			Std Error		
		g	Fe	Cu	Ni	Ni/Fe	Cu/Ni	Cu/Fe	Fe	Cu	Ni
C1	SNPX (5 min) + Guar (5 min), 5min oxidation, pH 10 (NaOH)	0.27	11.38	27.91	19.77	1.94	1.27	2.47	0.02	0.14	1.40
C2	SNPX (5 min) + Guar (5 min), 5min oxidation, pH 10 (NaOH)	0.51	22.77	47.43	36.45	1.76	1.17	2.06	1.01	1.73	3.88
C3	SNPX (5 min) + Guar (5 min), 5min oxidation, pH 10 (NaOH)	0.85	39.55	69.27	59.9	1.6	1.09	1.74	1.05	1.63	3.58
C4	SNPX (5 min) + Guar (5 min), 5min oxidation, pH 10 (NaOH)	1.21	57.69	87.26	79.69	1.42	1.06	1.51	0.87	1.20	2.48
Tails	SNPX (5 min) + Guar (5 min), 5min oxidation, pH 10 (NaOH)	2.03									

Appendix

		Cum Mass	Recovery (%)			Average Recovery (%)			Std Dev		
		g	Fe	Cu	Ni	Fe	Cu	Ni	Fe	Cu	Ni
C1	SNPX (5 min) + Guar (5 min), 5min oxidation, pH 9.5 (Lime)	0.24	10.89	30.86	24.64	10.86	33.03	22.86	0.03	2.18	1.78
C2	SNPX (5 min) + Guar (5 min), 5min oxidation, pH 9.5 (Lime)	0.50	24.3	55.94	48.87	24.45	58.29	47.77	0.15	2.35	1.10
C3	SNPX (5 min) + Guar (5 min), 5min oxidation, pH 9.5 (Lime)	0.79	38.47	74.72	69.18	39.2	77.45	69.21	0.73	2.74	0.03
C4	SNPX (5 min) + Guar (5 min), 5min oxidation, pH 9.5 (Lime)	1.11	55.94	87.59	83.1	56.34	88.89	84.12	0.40	1.30	1.02
Tails	SNPX (5 min) + Guar (5 min), 5min oxidation, pH 9.5 (Lime)	1.98									

		Cum Mass	Recovery (%)			Selectivity			Std Error		
		g	Fe	Cu	Ni	Ni/Fe	Cu/Ni	Cu/Fe	Fe	Cu	Ni
C1	SNPX (5 min) + Guar (5 min), 5min oxidation, pH 9.5 (Lime)	0.24	10.84	35.21	21.08	2.1	1.45	3.04	0.01	1.26	1.03
C2	SNPX (5 min) + Guar (5 min), 5min oxidation, pH 9.5 (Lime)	0.53	24.59	60.64	46.67	1.95	1.22	2.38	0.08	1.36	0.64
C3	SNPX (5 min) + Guar (5 min), 5min oxidation, pH 9.5 (Lime)	0.84	39.93	80.19	69.23	1.77	1.12	1.98	0.42	1.58	0.01
C4	SNPX (5 min) + Guar (5 min), 5min oxidation, pH 9.5 (Lime)	1.17	56.73	90.18	85.14	1.49	1.06	1.58	0.23	0.75	0.59
Tails	SNPX (5 min) + Guar (5 min), 5min oxidation, pH 9.5 (Lime)	2.00									

		Cum Mass	Recovery (%)			Average Recovery (%)			Std Dev		
		g	Fe	Cu	Ni	Fe	Cu	Ni	Fe	Cu	Ni
C1	SNPX (5 min) + Guar (5 min), 5min oxidation, pH 10 (Lime)	0.21	8.74	41.4	22.75	8.1	32.58	20.04	0.64	8.83	2.72
C2	SNPX (5 min) + Guar (5 min), 5min oxidation, pH 10 (Lime)	0.42	17.83	65.84	46.04	17.38	56.43	41.65	0.46	9.42	4.39
C3	SNPX (5 min) + Guar (5 min), 5min oxidation, pH 10 (Lime)	0.62	26.74	81.71	61.73	27.15	74.37	60.05	0.41	7.34	1.68
C4	SNPX (5 min) + Guar (5 min), 5min oxidation, pH 10 (Lime)	0.90	39.77	88.57	78.61	41.22	87.93	78.62	1.45	0.64	0.01
Tails	SNPX (5 min) + Guar (5 min), 5min oxidation, pH 10 (Lime)	2.00									

		Cum Mass	Recovery (%)			Selectivity			Std Error		
		g	Fe	Cu	Ni	Ni/Fe	Cu/Ni	Cu/Fe	Fe	Cu	Ni
C1	SNPX (5 min) + Guar (5 min), 5min oxidation, pH 10 (Lime)	0.18	7.46	23.75	17.32	2.47	1.63	4.02	0.37	5.10	1.57
C2	SNPX (5 min) + Guar (5 min), 5min oxidation, pH 10 (Lime)	0.39	16.92	47.01	37.27	2.4	1.35	3.25	0.26	5.44	2.53
C3	SNPX (5 min) + Guar (5 min), 5min oxidation, pH 10 (Lime)	0.62	27.56	67.03	58.38	2.21	1.24	2.74	0.24	4.24	0.97
C4	SNPX (5 min) + Guar (5 min), 5min oxidation, pH 10 (Lime)	0.93	42.67	87.29	78.62	1.91	1.12	2.13	0.84	0.37	0.00
Tails	SNPX (5 min) + Guar (5 min), 5min oxidation, pH 10 (Lime)	1.99									

		Cum Mass	Recovery (%)			Average Recovery (%)			Std Dev		
		g	Fe	Cu	Ni	Fe	Cu	Ni	Fe	Cu	Ni
C1	SNPX (5 min) + Guar (5 min), 5min oxidation, pH 10.5 (Lime)	0.05	2.51	11.17	6.49	2.85	14.38	7.79	0.34	3.21	1.30
C2	SNPX (5 min) + Guar (5 min), 5min oxidation, pH 10.5 (Lime)	0.11	4.97	20.43	13.17	5.74	27.05	15.82	0.78	6.63	2.66
C3	SNPX (5 min) + Guar (5 min), 5min oxidation, pH 10.5 (Lime)	0.19	8.41	30.38	21.07	9.19	37.73	24.4	0.79	7.35	3.33
C4	SNPX (5 min) + Guar (5 min), 5min oxidation, pH 10.5 (Lime)	0.32	15.05	43.22	33.17	16.23	51.96	37.37	1.18	8.75	4.20
Tails	SNPX (5 min) + Guar (5 min), 5min oxidation, pH 10.5 (Lime)	1.97									

		Cum Mass	Recovery (%)			Selectivity			Std Error		
		g	Fe	Cu	Ni	Ni/Fe	Cu/Ni	Cu/Fe	Fe	Cu	Ni
C1	SNPX (5 min) + Guar (5 min), 5min oxidation, pH 10.5 (Lime)	0.08	3.19	17.59	9.09	2.73	1.85	5.05	0.20	1.85	0.75
C2	SNPX (5 min) + Guar (5 min), 5min oxidation, pH 10.5 (Lime)	0.16	6.52	33.68	18.48	2.75	1.71	4.71	0.45	3.82	1.53
C3	SNPX (5 min) + Guar (5 min), 5min oxidation, pH 10.5 (Lime)	0.25	9.98	45.08	27.73	2.65	1.55	4.11	0.45	4.24	1.92
C4	SNPX (5 min) + Guar (5 min), 5min oxidation, pH 10.5 (Lime)	0.42	17.4	60.71	41.57	2.3	1.39	3.2	0.68	5.05	2.42
Tails	SNPX (5 min) + Guar (5 min), 5min oxidation, pH 10.5 (Lime)	2.00									

		Cum Mass	Recovery (%)			Average Recovery (%)			Std Dev		
		g	Fe	Cu	Ni	Fe	Cu	Ni	Fe	Cu	Ni
C1	SNPX (5 min) + Guar (5 min), 5min oxidation, pH 9 (Lime)	0.47	22.22	40.52	33.25	21.39	37.61	26.6	0.83	2.91	6.65
C2	SNPX (5 min) + Guar (5 min), 5min oxidation, pH 9 (Lime)	0.94	44.87	72.55	65.56	42.24	69.36	56.71	2.63	3.19	8.85
C3	SNPX (5 min) + Guar (5 min), 5min oxidation, pH 9 (Lime)	1.29	62.27	89.59	85.22	58.85	85.84	77.09	3.43	3.75	8.13
C4	SNPX (5 min) + Guar (5 min), 5min oxidation, pH 9 (Lime)	1.54	75.62	95.61	94.07	73.29	93.77	89.69	2.33	1.84	4.38
Tails	SNPX (5 min) + Guar (5 min), 5min oxidation, pH 9 (Lime)	1.99									

		Cum Mass	Recovery (%)			Selectivity			Std Error		
		g	Fe	Cu	Ni	Ni/Fe	Cu/Ni	Cu/Fe	Fe	Cu	Ni
C1	SNPX (5 min) + Guar (5 min), 5min oxidation, pH 9 (Lime)	0.44	20.57	34.7	19.96	1.24	1.41	1.76	0.48	1.68	3.84
C2	SNPX (5 min) + Guar (5 min), 5min oxidation, pH 9 (Lime)	0.80	39.61	66.17	47.87	1.34	1.22	1.64	1.52	1.84	5.11
C3	SNPX (5 min) + Guar (5 min), 5min oxidation, pH 9 (Lime)	1.13	55.42	82.09	68.96	1.31	1.11	1.46	1.98	2.17	4.69
C4	SNPX (5 min) + Guar (5 min), 5min oxidation, pH 9 (Lime)	1.44	70.96	91.93	85.31	1.22	1.05	1.28	1.35	1.06	2.53
Tails	SNPX (5 min) + Guar (5 min), 5min oxidation, pH 9 (Lime)	1.99									

		Cum Mass	Recovery (%)			Average Recovery (%)			Std Dev		
		g	Fe	Cu	Ni	Fe	Cu	Ni	Fe	Cu	Ni
C1	SNPX (5 min) + Guar (5 min), 5min oxidation, pH 9.5 (NaOH)	0.57	27.16	38.06	28.96	26.18	41.92	33.8	0.98	3.86	4.84
C2	SNPX (5 min) + Guar (5 min), 5min oxidation, pH 9.5 (NaOH)	1.11	54.02	70.6	63.89	51.47	74.39	67.48	2.55	3.79	3.59
C3	SNPX (5 min) + Guar (5 min), 5min oxidation, pH 9.5 (NaOH)	1.47	71.7	93.04	89.81	68.09	91.86	88	3.62	1.18	1.81
C4	SNPX (5 min) + Guar (5 min), 5min oxidation, pH 9.5 (NaOH)	1.67	81.64	97.88	96.83	78.36	97.18	95.5	3.28	0.71	1.33
Tails	SNPX (5 min) + Guar (5 min), 5min oxidation, pH 9.5 (NaOH)	1.97									

		Cum Mass	Recovery (%)			Selectivity			Std Error		
		g	Fe	Cu	Ni	Ni/Fe	Cu/Ni	Cu/Fe	Fe	Cu	Ni
C1	SNPX (5 min) + Guar (5 min), 5min oxidation, pH 9.5 (NaOH)	0.55	25.21	45.78	38.63	1.29	1.24	1.6	0.56	2.23	2.79
C2	SNPX (5 min) + Guar (5 min), 5min oxidation, pH 9.5 (NaOH)	1.06	48.92	78.18	71.07	1.31	1.1	1.45	1.47	2.19	2.07
C3	SNPX (5 min) + Guar (5 min), 5min oxidation, pH 9.5 (NaOH)	1.34	64.47	90.69	86.2	1.29	1.04	1.35	2.09	0.68	1.04
C4	SNPX (5 min) + Guar (5 min), 5min oxidation, pH 9.5 (NaOH)	1.54	75.08	96.47	94.18	1.22	1.02	1.24	1.89	0.41	0.76
Tails	SNPX (5 min) + Guar (5 min), 5min oxidation, pH 9.5 (NaOH)	2.00									

		Cum Mass	Recovery (%)			Average Recovery (%)			Std Dev		
		g	Fe	Cu	Ni	Fe	Cu	Ni	Fe	Cu	Ni
C1	SNPX (5 min) + Guar (5 min), 5min oxidation, pH 10.5 (NaOH)	0.03	0.9	2.39	2.35	0.88	2.93	2.37	0.02	0.54	0.03
C2	SNPX (5 min) + Guar (5 min), 5min oxidation, pH 10.5 (NaOH)	0.06	2.03	6.53	5.34	2.02	7.44	5.49	0.01	0.91	0.16
C3	SNPX (5 min) + Guar (5 min), 5min oxidation, pH 10.5 (NaOH)	0.13	4.78	14.81	11.58	4.35	15.16	11.39	0.44	0.35	0.19
C4	SNPX (5 min) + Guar (5 min), 5min oxidation, pH 10.5 (NaOH)	0.26	10.35	30.77	23.31	9.08	30.42	22.65	1.27	0.35	0.66
Tails	SNPX (5 min) + Guar (5 min), 5min oxidation, pH 10.5 (NaOH)	2.03									

		Cum Mass	Recovery (%)			Selectivity			Std Error		
		g	Fe	Cu	Ni	Ni/Fe	Cu/Ni	Cu/Fe	Fe	Cu	Ni
C1	SNPX (5 min) + Guar (5 min), 5min oxidation, pH 10.5 (NaOH)	0.02	0.86	3.46	2.4	2.7	1.23	3.33	0.01	0.31	0.01
C2	SNPX (5 min) + Guar (5 min), 5min oxidation, pH 10.5 (NaOH)	0.05	2.01	8.34	5.65	2.72	1.35	3.68	0.01	0.52	0.09
C3	SNPX (5 min) + Guar (5 min), 5min oxidation, pH 10.5 (NaOH)	0.10	3.91	15.5	11.21	2.62	1.33	3.49	0.25	0.20	0.11
C4	SNPX (5 min) + Guar (5 min), 5min oxidation, pH 10.5 (NaOH)	0.20	7.82	30.07	21.99	2.49	1.34	3.35	0.73	0.20	0.38
Tails	SNPX (5 min) + Guar (5 min), 5min oxidation, pH 10.5 (NaOH)	2.00									

Reproducibility data for first base case at pH 9

	Average				STD Dev				Std Error			
	C1	C2	C3	C4	C1	C2	C3	C4	C1	C2	C3	C4
	2	5	10	20	2	5	10	20	2	5	10	20
Mass	0.40	0.34	0.25	0.28	0.03	0.06	0.01	0.04	0.01	0.02	0.01	0.02
Ni	19.53	40.67	60.00	78.35	2.58	5.83	7.41	5.17	1.05	2.38	3.03	2.11
Fe	19.77	36.30	48.60	63.09	1.67	4.83	5.16	2.44	0.68	1.97	2.11	1.00
Ni/Fe	0.99	1.12	1.24	1.24	0.11	0.11	0.10	0.07	0.04	0.05	0.04	0.03

Reproducibility data for second base case at pH 10

	Average				STD Dev				Std Error			
Conc	C1	C2	C3	C4	C1	C2	C3	C4	C1	C2	C3	C4
Time (min)	2	5	10	20	2	5	10	20	2	5	10	20
Mass	0.25	0.53	0.86	1.20	0.06	0.03	0.03	0.05	0.02	0.01	0.01	0.02
Ni	20.7	42.8	65.0	83.4	4.4	7.4	6.6	3.4	1.8	3.0	2.7	1.4
Fe	10.3	22.7	37.3	54.4	2.4	4.2	5.7	4.7	1.0	1.7	2.3	1.9
Ni/Fe	2.0	1.9	1.8	1.5	0.2	0.2	0.2	0.1	0.1	0.1	0.1	0.0

Appendix C: Batch flotation data

Date	28-11-07
Ore	Sheba's Ridge
pH	natural

Reagents	**Type**	**Dosage**		
Frother	Dow 200	40, 20	3	1min
Collector	SNPX	25, 10	1	in mill, 2min
Depressant 1	Dep 267	40, 10	2	5min
Depressant 2				
pH modifier				

Product	**Time**	**Water**			**Solids**	
	min	**Total in Conc**	**Recovered**	**Cum Rec**	**Rec**	**Cum Rec**
		g	**g**	**g**	**g**	**g**
FF Cl C1	2	68.3	33.74	33.74	12.76	12.76
FF Cl C2	5	70.16	29.41	63.15	6.19	18.95
FF Cl C3	10	89.86	37.51	100.66	5.97	24.92
FF Cl C4	20	177.29	69.47	170.13	7.63	32.55
Combined FF Conc		**405.61**	**170.13**		**32.55**	

Product	**Time**	**Mass**	**Mass**	**Grade (%)**			**Recovery (%)**		
	min	**g**	**%**	**Cu**	**Ni**	**S**	**Cu**	**Ni**	**S**
FF Cl C1	2	12.76	1.27	3.35	7.47	14.1	50.15	37.07	45.45
FF Cl C2	5	6.19	0.62	0.87	2.53	5.9	6.32	6.09	9.23
FF Cl C3	10	5.97	0.59	0.5	1.25	3.33	3.5	2.9	5.02
FF Cl C4	20	7.63	0.76	0.3	0.64	1.77	2.69	1.9	3.41
Combined FF Conc		32.55	3.24	1.64	3.79	7.68	62.66	47.96	63.11
Cleaner Tails		37.01	3.68	0.16	0.35	0.68	6.95	5.04	6.36
Rougher Conc		69.56	6.92	0.85	1.96	3.95	69.61	52.99	69.46
Rougher Tails		863.46	85.89	0.03	0.14	0.14	30.39	47.01	30.54
Other samples		72.25	7.19						
Head (meas)		1005.27	100	0.08	0.26	0.45	100	100	100
Head (calc)			100	0.08	0.26	0.39			

		Mass	**Mass**	**Grade (%)**			**Recovery (%)**		
		g	**%**	**Cu**	**Ni**	**S**	**Cu**	**Ni**	**S**
FFCC1	2	12.76	1.27	3.35	7.47	14.1	50.15	37.07	45.45
FFCC1 + CC2	5	18.95	1.89	2.54	5.86	11.42	56.47	43.15	54.67
FFCC1 + CC2 + CC3	10	24.92	2.48	2.05	4.75	9.48	59.97	46.06	59.7
FFCC1 + CC2 + CC3 + CC4	20	32.55	3.24	1.64	3.79	7.68	62.66	47.96	63.11
FFCC1 + CC2 + CC3 + CC4 + CT		69.56	6.92	0.85	1.96	3.95	69.61	52.99	69.46
FFCC1 + CC2 + CC3 + CC4 + CT + RT		1005.27	100	0.08	0.26	0.45	100	100	100

Date	22-11-07
Ore	Sheba's Ridge
pH	natural

Reagents	**Type**	**Dosage**	**Add. Seq**	
Frother	Dow 200	40, 20, 20	3	1min
Collector	SNPX	25, 10, 10	1	n mill, 2min
Depressant 1	Dep 267	40	2	5min
Depressant 2	Sty 504	10	2	5min
pH modifier				

Product	**Time**	**Water**			**Solids**	
		Total in Conc	**Recovered**	**Cum Rec**	**Rec**	**Cum Rec**
	min	**g**	**g**	**g**	**g**	**g**
FF Cl C1	2	57.83	24.2	24.2	9.53	9.53
FF Cl C2	5	62.24	28.24	52.44	5.58	15.11
FF Cl C3	10	113.52	39.87	92.31	5.82	20.93
FF Cl C4	20	178.53	76.61	168.92	7.61	28.54
Combined FF Conc		**412.12**	**168.92**		**28.54**	

Product	**Time**	**Mass**	**Mass**	**Grade (%)**			**Recovery (%)**		
	min	**g**	**%**	**Cu**	**Ni**	**S**	**Cu**	**Ni**	**S**
FF Cl C1	2	9.53	0.95	3.97	9.31	17.5	60.14	33.41	42.46
FF Cl C2	5	5.58	0.56	1.16	3.71	8.1	10.29	7.8	11.51
FF Cl C3	10	5.82	0.58	0.59	1.64	4.1	5.46	3.59	6.08
FF Cl C4	20	7.61	0.76	0.31	0.73	1.88	3.75	2.09	3.64
Combined FF Conc		28.54	2.84	1.76	4.36	8.76	79.64	46.9	63.69
Cleaner Tails		34.34	3.42	0.12	0.31	0.61	6.55	4.01	5.33
Rougher Conc		62.88	6.26	0.86	2.15	4.31	86.19	50.91	69.02
Rougher Tails		869.09	86.56	0.01	0.15	0.14	13.81	49.09	30.98
Other samples		72.02	7.17						
Head (meas)		1003.99	100	0.08	0.26	0.45	100	100	100
Head (calc)			100	0.06	0.26	0.39			

		Mass	**Mass**	**Grade (%)**			**Recovery (%)**		
		g	**%**	**Cu**	**Ni**	**S**	**Cu**	**Ni**	**S**
FFCC1	2	9.53	0.95	3.97	9.31	17.5	60.14	33.41	42.46
FFCC1 + CC2	5	15.11	1.5	2.93	7.24	14.03	70.43	41.21	53.97
FFCC1 + CC2 + CC3	10	20.93	2.08	2.28	5.68	11.27	75.89	44.8	60.05
FFCC1 + CC2 + CC3 + CC4	20	28.54	2.84	1.76	4.36	8.76	79.64	46.9	63.69
FFCC1 + CC2 + CC3 + CC4 + CT		62.88	6.26	0.86	2.15	4.31	86.19	50.91	69.02
FFCC1 + CC2 + CC3 + CC4 + CT + RT		1003.99	100	0.08	0.26	0.45	100	100	100

Date	22-01-08
Ore	Sheba's Ridge
pH	natural

Reagents	**Type**	**Dosage**	**Add. Seq**	**Cond. Time**
Frother	Dow 200	40, 20	3	1min
Collector	SNPX	25, 10	1	in mill, 2min
Depressant 1	Dep 267	40, 50	2	5min
Depressant 2				
pH modifier				

Product	**Time**	**Water**			**Solids**	
	min	**Total in Conc**	**Recovered**	**Cum Rec**	**Rec**	**Cum Rec**
		g	**g**	**g**	**g**	**g**
FF Cl C1	2	48.06	23.05	23.05	8.04	8.04
FF Cl C2	5	52.22	12.66	35.71	4.21	12.25
FF Cl C3	10	71.95	15.25	50.96	3.37	15.62
FF Cl C4	20	123.8	22.53	73.49	4.52	20.14
Combined FF Conc		**296.03**	**73.49**		**20.14**	

Product	**Time**	**Mass**	**Mass %**	**Grade (%)**			**Recovery (%)**		
	min	**g**		**Cu**	**Ni**	**S**	**Cu**	**Ni**	**S**
FF Cl C1	2	8.04	0.8	4.32	8.53	16.2	47.96	26.93	32.76
FF Cl C2	5	4.21	0.42	2.96	7.6	13.5	17.21	12.56	14.3
FF Cl C3	10	3.37	0.34	1.43	4.48	8.09	6.65	5.93	6.86
FF Cl C4	20	4.52	0.45	0.5	1.63	3.17	3.12	2.88	3.6
Combined FF Conc		20.14	2.01	2.69	6.11	11.35	74.94	48.31	57.52
Cleaner Tails		68.21	6.81	0.1	0.29	0.59	9.23	7.77	10.12
Rougher Conc		88.35	8.83	0.69	1.62	3.04	84.17	56.08	67.64
Rougher Tails		838.87	83.8	0.01	0.13	0.15	15.83	43.92	32.36
Other samples		73.77	7.37						
Head (meas)		1000.99	100	0.08	0.27	0.41	100	100	100
Head (calc)			100	0.07	0.25	0.4			

		Mass	**Mass %**	**Grade (%)**			**Recovery (%)**		
		g		**Cu**	**Ni**	**S**	**Cu**	**Ni**	**S**
FFCC1	2	8.04	0.8	4.32	8.53	16.2	47.96	26.93	32.76
FFCC1 + CC2	5	12.25	1.22	3.85	8.21	15.27	65.17	39.5	47.06
FFCC1 + CC2 + CC3	10	15.62	1.56	3.33	7.41	13.72	71.82	45.43	53.92
FFCC1 + CC2 + CC3 + CC4	20	20.14	2.01	2.69	6.11	11.35	74.94	48.31	57.52
FFCC1 + CC2 + CC3 + CC4 + CT		88.35	8.83	0.69	1.62	3.04	84.17	56.08	67.64
FFCC1 + CC2 + CC3 + CC4 + CT + RT		1000.99	100	0.08	0.27	0.41	100	100	100

Date	22-01-08
Ore	Sheba's Ridge
pH	natural

Reagents	**Type**	**Dosage**	**Add. Seq**	**Cond. Time**
Frother	Dow 200	40, 20	3	1min
Collector	SNPX	25, 10	1	in mill, 2min
Depressant 1	Dep 267	40	2	5min
Depressant 2	Sty 504	50	2	5min
pH modifier				

Product	**Time**	**Water**			**Solids**	
	min	**Total in Conc g**	**Recovered g**	**Cum Rec g**	**Rec g**	**Cum Rec g**
FF Cl C1	2	49.93	17.15	17.15	7.64	7.64
FF Cl C2	5	44.81	16.28	33.43	3.66	11.3
FF Cl C3	10	61.87	16.08	49.51	2.69	13.99
FF Cl C4	20	124.49	38.58	88.09	5.53	19.52
Combined FF Conc		**281.1**	**88.09**		**19.52**	

Product	**Time**	**Mass**	**Mass**	**Grade (%)**			**Recovery (%)**		
	min	**g**	**%**	**Cu**	**Ni**	**S**	**Cu**	**Ni**	**S**
FF Cl C1	2	7.64	0.76	5.22	10.1	19.8	57.69	29.81	38.86
FF Cl C2	5	3.66	0.36	2.48	6.87	12.1	13.13	9.71	11.38
FF Cl C3	10	2.69	0.27	1.11	3.66	7.09	4.32	3.8	4.9
FF Cl C4	20	5.53	0.55	0.51	1.54	4.24	4.08	3.28	6.02
Combined FF Conc		19.52	1.95	2.81	6.18	12.2	79.21	46.61	61.15
Cleaner Tails		74.97	7.47	0.08	0.28	0.64	8.68	8.11	12.32
Rougher Conc		94.49	9.42	0.64	1.5	3.03	87.89	54.72	73.48
Rougher Tails		837.22	83.46	0.01	0.14	0.12	12.11	45.28	26.52
Other samples		71.38	7.12						
Head (meas)		1003.09	100	0.07	0.27	0.44	100	100	100
Head (calc)			100	0.07	0.26	0.39			

		Mass	**Mass**	**Grade (%)**			**Recovery (%)**		
		g	**%**	**Cu**	**Ni**	**S**	**Cu**	**Ni**	**S**
FFCC1	2	7.64	0.76	5.22	10.1	19.8	57.69	29.81	38.86
FFCC1 + CC2	5	11.3	1.13	4.33	9.05	17.31	70.82	39.52	50.23
FFCC1 + CC2 + CC3	10	13.99	1.39	3.71	8.02	15.34	75.13	43.33	55.13
FFCC1 + CC2 + CC3 + CC4	20	19.52	1.95	2.81	6.18	12.2	79.21	46.61	61.15
FFCC1 + CC2 + CC3 + CC4 + CT		94.49	9.42	0.64	1.5	3.03	87.89	54.72	73.48
FFCC1 + CC2 + CC3 + CC4 + CT + RT		1003.09	100	0.07	0.27	0.44	100	100	100

Date	24-01-08
Ore	Sheba's Ridge
pH	natural

Reagents	**Type**	**Dosage**	**Add. Seq**	**Cond. Time**
Frother	Dow 200	40, 20	3	1min
Collector	SNPX	25, 10	1	in mill, 2min
Depressant 1	Dep 267	40	2	5min
Depressant 2	Sty 504	100	2	5min
pH modifier				

Product	**Time**	**Water**			**Solids**	
	min	**Total in Conc g**	**Recovered g**	**Cum Rec g**	**Rec g**	**Cum Rec g**
FF Cl C1	2	26.32	5.7	5.7	3	3
FF Cl C2	5	36.38	6.23	11.93	1.81	4.81
FF Cl C3	10	64.17	13.76	25.69	2.68	7.49
FF Cl C4	20	120.16	25.59	51.28	3.19	10.68
Combined FF Conc		**247.03**	**51.28**		**10.68**	

Product	**Time**	**Mass**	**Mass**	**Grade (%)**			**Recovery (%)**		
	min	**g**	**%**	**Cu**	**Ni**	**S**	**Cu**	**Ni**	**S**
FF Cl C1	2	3	0.3	7.94	12.8	23.1	28.02	13.15	17.34
FF Cl C2	5	1.81	0.18	5.53	11.9	20.5	11.77	7.38	9.28
FF Cl C3	10	2.68	0.27	3.87	11	18.5	12.2	10.1	12.4
FF Cl C4	20	3.19	0.32	1.88	7.62	15	7.05	8.32	11.97
Combined FF Conc		10.68	1.06	4.7	10.65	19.09	59.04	38.95	50.99
Cleaner Tails		78.99	7.85	0.14	0.43	0.94	13.01	11.63	18.57
Rougher Conc		89.67	8.92	0.68	1.65	3.1	72.05	50.58	69.56
Rougher Tails		848.85	84.4	0.03	0.17	0.14	27.95	49.42	30.44
Other samples		67.18	6.68						
Head (meas)		1005.7	100	0.1	0.31	0.44	100	100	100
Head (calc)			100	0.08	0.29	0.4			

		Mass	**Mass**	**Grade (%)**			**Recovery (%)**		
		g	**%**	**Cu**	**Ni**	**S**	**Cu**	**Ni**	**S**
FFCC1	2	3	0.3	7.94	12.8	23.1	28.02	13.15	17.34
FFCC1 + CC2	5	4.81	0.48	7.03	12.46	22.12	39.79	20.53	26.62
FFCC1 + CC2 + CC3	10	7.49	0.74	5.9	11.94	20.83	51.99	30.62	39.02
FFCC1 + CC2 + CC3 + CC4	20	10.68	1.06	4.7	10.65	19.09	59.04	38.95	50.99
FFCC1 + CC2 + CC3 + CC4 + CT		89.67	8.92	0.68	1.65	3.1	72.05	50.58	69.56
FFCC1 + CC2 + CC3 + CC4 + CT + RT		1005.7	100	0.1	0.31	0.44	100	100	100

Date	24-01-08
Ore	Sheba's Ridge
pH	natural

Reagents	**Type**	**Dosage**	**Add. Seq**	**Cond. Time**
Frother	Dow 200	40, 20	3	1min
Collector	SNPX	25, 10	1	in mill, 2min
Depressant 1	Dep 267	40, 100	2	5min
Depressant 2				5min
pH modifier				

Product	**Time**	**Water**			**Solids**	
	min	**Total in Conc g**	**Recovered g**	**Cum Rec g**	**Rec g**	**Cum Rec g**
FF Cl C1	2	36.19	10.11	10.11	3.35	3.35
FF Cl C2	5	42.74	7.62	17.73	2.64	5.99
FF Cl C3	10	76.6	10.66	28.39	2.05	8.04
FF Cl C4	20	129.49	30.78	59.17	2.7	10.74
Combined FF Conc		**285.02**	**59.17**		**10.74**	

Product	**Time**	**Mass**	**Mass**	**Grade (%)**			**Recovery (%)**		
	min	**g**	**%**	**Cu**	**Ni**	**S**	**Cu**	**Ni**	**S**
FF Cl C1	2	3.35	0.33	6.48	11.7	23.3	32.24	13.87	20.28
FF Cl C2	5	2.64	0.26	4.25	12	19.8	16.67	11.21	13.58
FF Cl C3	10	2.05	0.2	3.03	11.9	19.3	9.23	8.63	10.28
FF Cl C4	20	2.7	0.27	1.39	7.37	12.9	5.55	7.04	9.05
Combined FF Conc		10.74	1.07	3.99	10.72	19.06	63.69	40.74	53.19
Cleaner Tails		67.18	6.71	0.11	0.42	0.82	10.98	9.98	14.31
Rougher Conc		77.92	7.78	0.65	1.84	3.33	74.66	50.72	67.5
Rougher Tails		852.86	85.17	0.02	0.16	0.15	25.34	49.28	32.5
Other samples		70.6	7.05						
Head (meas)		1001.38	100	0.07	0.3	0.43	100	100	100
Head (calc)			100	0.07	0.28	0.38			

		Mass	**Mass**	**Grade (%)**			**Recovery (%)**		
		g	**%**	**Cu**	**Ni**	**S**	**Cu**	**Ni**	**S**
FFCC1	2	3.35	0.33	6.48	11.7	23.3	32.24	13.87	20.28
FFCC1 + CC2	5	5.99	0.6	5.5	11.83	21.76	48.91	25.07	33.86
FFCC1 + CC2 + CC3	10	8.04	0.8	4.87	11.85	21.13	58.13	33.7	44.14
FFCC1 + CC2 + CC3 + CC4	20	10.74	1.07	3.99	10.72	19.06	63.69	40.74	53.19
FFCC1 + CC2 + CC3 + CC4 + CT		77.92	7.78	0.65	1.84	3.33	74.66	50.72	67.5
FFCC1 + CC2 + CC3 + CC4 + CT + RT		1001.38	100	0.07	0.3	0.43	100	100	100

Date	24-01-08
Ore	Sheba's Ridge
pH	natural + 10.5

Reagents	**Type**	**Dosage**	**Add. Seq**	**Cond. Time**
Frother	Dow 200	40, 20	3	1min
Collector	SNPX	25, 10	1	in mill, 2min
Depressant 1	Dep 267	40	2	5min
Depressant 2	Sty 504	100		5min
pH modifier	lime			

Product	**Time**	**Water**			**Solids**	
		Total in Conc	**Recovered**	**Cum Rec**	**Rec**	**Cum Rec**
	min	**g**	**g**	**g**	**g**	**g**
FF Cl C1	2	35.12	11.67	11.67	4.99	4.99
FF Cl C2	5	37.91	5.24	16.91	1.22	6.21
FF Cl C3	10	54.16	10.98	27.89	1.73	7.94
FF Cl C4	20	63.73	12.86	40.75	1.63	9.57
Combined FF Conc		**190.92**	**40.75**		**9.57**	

Product	**Time**	**Mass**	**Mass**	**Grade (%)**			**Recovery (%)**		
	min	**g**	**%**	**Cu**	**Ni**	**S**	**Cu**	**Ni**	**S**
FF Cl C1	2	4.99	0.49	5.93	12.6	26	40.19	24.03	30.48
FF Cl C2	5	1.22	0.12	4.29	10.9	22.7	7.11	5.08	6.51
FF Cl C3	10	1.73	0.17	3.06	9.83	20.3	7.19	6.5	8.25
FF Cl C4	20	1.63	0.16	1.91	7.29	17.1	4.23	4.54	6.55
Combined FF Conc		9.57	0.93	4.52	10.98	23.03	58.72	40.15	51.79
Cleaner Tails		64.23	6.25	0.18	0.52	1.18	15.7	12.64	17.81
Rougher Conc		73.8	7.19	0.74	1.87	4.01	74.43	52.79	69.59
Rougher Tails		882.53	85.94	0.02	0.14	0.15	25.57	47.21	30.41
Other samples		70.54	6.87						
Head (meas)		1026.87	100	0.07	0.26	0.41	100	100	100
Head (calc)			100	0.07	0.25	0.41			

		Mass	**Mass**	**Grade (%)**			**Recovery (%)**		
		g	**%**	**Cu**	**Ni**	**S**	**Cu**	**Ni**	**S**
FFCC1	2	4.99	0.49	5.93	12.6	26	40.19	24.03	30.48
FFCC1 + CC2	5	6.21	0.6	5.61	12.27	25.35	47.3	29.11	36.99
FFCC1 + CC2 + CC3	10	7.94	0.77	5.05	11.74	24.25	54.49	35.61	45.24
FFCC1 + CC2 + CC3 + CC4	20	9.57	0.93	4.52	10.98	23.03	58.72	40.15	51.79
FFCC1 + CC2 + CC3 + CC4 + CT		73.8	7.19	0.74	1.87	4.01	74.43	52.79	69.59
FFCC1 + CC2 + CC3 + CC4 + CT + RT		1026.87	100	0.07	0.26	0.41	100	100	100

Date	24-01-08
Ore	Sheba's Ridge
pH	natural

Reagents	**Type**	**Dosage**	**Add. Seq**	**Cond. Time**
Frother	Dow 200	40, 20	3	1min
Collector	SNPX	50, 10	1	in mill, 2min
Depressant 1	Dep 267	40	2	5min
Depressant 2	Sty 504	100		5min
pH modifier	lime			

Product	**Time**	**Water**			**Solids**	
	min	**Total in Conc g**	**Recovered g**	**Cum Rec g**	**Rec g**	**Cum Rec g**
FF Cl C1	2	35.04	7.63	7.63	3.77	3.77
FF Cl C2	5	41.48	4.54	12.17	1.87	5.64
FF Cl C3	10	62.09	12.39	24.56	2.4	8.04
FF Cl C4	20	75.48	19.59	44.15	2.14	10.18
Combined FF Conc		**214.09**	**44.15**		**10.18**	

Product	**Time**	**Mass**	**Mass**	**Grade (%)**			**Recovery (%)**		
	min	**g**	**%**	**Cu**	**Ni**	**S**	**Cu**	**Ni**	**S**
FF Cl C1	2	3.77	0.37	6.53	13.2	27.7	33.34	18.84	23.75
FF Cl C2	5	1.87	0.18	5.15	12.2	25.2	13.04	8.64	10.72
FF Cl C3	10	2.4	0.23	3.54	9.41	21.9	11.51	8.55	11.95
FF Cl C4	20	2.14	0.21	2	7.46	18.75	5.8	6.05	9.13
Combined FF Conc		10.18	0.99	4.62	10.92	23.99	63.68	42.08	55.55
Cleaner Tails		56.05	5.45	0.2	0.58	1.49	15.18	12.2	19
Rougher Conc		66.23	6.44	0.88	2.16	4.95	78.86	54.28	74.55
Rougher Tails		883.42	85.91	0.02	0.14	0.13	21.14	45.72	25.45
Other samples		78.6	7.64						
Head (meas)		1028.25	100	0.07	0.26	0.42	100	100	100
Head (calc)			100	0.07	0.26	0.43			

		Mass	**Mass**	**Grade (%)**			**Recovery (%)**		
		g	**%**	**Cu**	**Ni**	**S**	**Cu**	**Ni**	**S**
FFCC1	2	3.77	0.37	6.53	13.2	27.7	33.34	18.84	23.75
FFCC1 + CC2	5	5.64	0.55	6.07	12.87	26.87	46.38	27.48	34.47
FFCC1 + CC2 + CC3	10	8.04	0.78	5.32	11.84	25.39	57.89	36.03	46.43
FFCC1 + CC2 + CC3 + CC4	20	10.18	0.99	4.62	10.92	23.99	63.68	42.08	55.55
FFCC1 + CC2 + CC3 + CC4 + CT		66.23	6.44	0.88	2.16	4.95	78.86	54.28	74.55
FFCC1 + CC2 + CC3 + CC4 + CT + RT		1028.25	100	0.07	0.26	0.42	100	100	100

Date	24-01-08
Ore	Sheba's Ridge
pH	natural

Reagents	**Type**	**Dosage**	**Add. Seq**	**Cond. Time**
Frother	Dow 200	40, 20	3	1min
Collector	SNPX	25, 10	1	in mill, 2min
Depressant 1	Dep 267	40	2	5min
Depressant 2	Sty 504	100		5min
pH modifier				

Product	**Time**	**Water**			**Solids**	
		Total in Conc	**Recovered**	**Cum Rec**	**Rec**	**Cum Rec**
	min	**g**	**g**	**g**	**g**	**g**
FF Cl C1	2	32.6	11.06	11.06	4.51	4.51
FF Cl C2	5	32.73	6.86	17.92	1.52	6.03
FF Cl C3	10	32.22	9.9	27.82	1.54	7.57
FF Cl C4	20	59.4	17.6	45.42	1.69	9.26
Combined FF Conc		**156.95**	**45.42**		**9.26**	

Product	**Time**	**Mass**	**Mass**	**Grade (%)**			**Recovery (%)**		
	min	**g**	**%**	**Cu**	**Ni**	**S**	**Cu**	**Ni**	**S**
FF Cl C1	2	4.51	0.45	6.51	13.4	26	35.84	21.36	27.82
FF Cl C2	5	1.52	0.15	4.46	12.2	22.6	8.28	6.55	8.15
FF Cl C3	10	1.54	0.15	2.72	10.8	20.2	5.11	5.88	7.38
FF Cl C4	20	1.69	0.17	1.78	6.41	15.25	3.66	3.83	6.11
Combined FF Conc		9.26	0.93	4.68	11.49	22.52	52.89	37.62	49.47
Cleaner Tails		64.23	6.43	0.16	0.5	0.98	12.55	11.35	14.93
Rougher Conc		73.49	7.36	0.73	1.89	3.69	65.44	48.97	64.4
Rougher Tails		849.35	85.05	0.03	0.17	0.18	34.56	51.03	35.6
Other samples		75.86	7.6						
Head (meas)		998.7	100	0.08	0.28	0.42	100	100	100
Head (calc)			100	0.08	0.28	0.42			

		Mass	**Mass**	**Grade (%)**			**Recovery (%)**		
		g	**%**	**Cu**	**Ni**	**S**	**Cu**	**Ni**	**S**
FFCC1	2	4.51	0.45	6.51	13.4	26	35.84	21.36	27.82
FFCC1 + CC2	5	6.03	0.6	5.99	13.1	25.14	44.12	27.91	35.97
FFCC1 + CC2 + CC3	10	7.57	0.76	5.33	12.63	24.14	49.23	33.79	43.35
FFCC1 + CC2 + CC3 + CC4	20	9.26	0.93	4.68	11.49	22.52	52.89	37.62	49.47
FFCC1 + CC2 + CC3 + CC4 + CT		73.49	7.36	0.73	1.89	3.69	65.44	48.97	64.4
FFCC1 + CC2 + CC3 + CC4 + CT + RT		998.7	100	0.08	0.28	0.42	100	100	100

Date	24-01-08
Ore	Sheba's Ridge
pH	natural

Reagents	**Type**	**Dosage**	**Add. Seq**	**Cond. Time**
Frother	Dow 200	40, 20	3	1min
Collector	SNPX	25, 10	1	in mill, 2min
Depressant 1	Dep 267	40, 100	2	5min
Depressant 2				
pH modifier				

Product	**Time**	**Water**			**Solids**	
		Total in Conc	**Recovered**	**Cum Rec**	**Rec**	**Cum Rec**
	min	**g**	**g**	**g**	**g**	**g**
FF Cl C1	2	33.92	11.09	11.09	4.67	4.67
FF Cl C2	5	25.01	5.88	16.97	1.39	6.06
FF Cl C3	10	52.2	12.98	29.95	1.89	7.95
FF Cl C4	20	106.53	27.09	57.04	2.27	10.22
Combined FF Conc		**217.66**	**57.04**		**10.22**	

Product	**Time**	**Mass**	**Mass**	**Grade (%)**			**Recovery (%)**		
	min	**g**	**%**	**Cu**	**Ni**	**S**	**Cu**	**Ni**	**S**
FF Cl C1	2	4.67	0.46	7	14	25.6	42.57	23.5	28.41
FF Cl C2	5	1.39	0.14	3.9	12	21.7	7.06	6	7.17
FF Cl C3	10	1.89	0.19	2.49	10.8	19.4	6.13	7.34	8.71
FF Cl C4	20	2.27	0.22	1.54	6.41	13.65	4.55	5.23	7.36
Combined FF Conc		10.22	1.01	4.53	11.45	21.27	60.3	42.07	51.65
Cleaner Tails		65.34	6.47	0.14	0.42	0.85	11.91	9.86	13.2
Rougher Conc		75.56	7.48	0.73	1.91	3.61	72.22	51.93	64.85
Rougher Tails		853.52	84.53	0.03	0.16	0.17	27.78	48.07	35.15
Other samples		80.69	7.99						
Head (meas)		1009.77	100	0.08	0.27	0.42	100	100	100
Head (calc)			100	0.08	0.28	0.42			

		Mass	**Mass**	**Grade (%)**			**Recovery (%)**		
		g	**%**	**Cu**	**Ni**	**S**	**Cu**	**Ni**	**S**
FFCC1	2	4.67	0.46	7	14	25.6	42.57	23.5	28.41
FFCC1 + CC2	5	6.06	0.6	6.29	13.54	24.71	49.62	29.5	35.57
FFCC1 + CC2 + CC3	10	7.95	0.79	5.39	12.89	23.44	55.75	36.84	44.29
FFCC1 + CC2 + CC3 + CC4	20	10.22	1.01	4.53	11.45	21.27	60.3	42.07	51.65
FFCC1 + CC2 + CC3 + CC4 + CT		75.56	7.48	0.73	1.91	3.61	72.22	51.93	64.85
FFCC1 + CC2 + CC3 + CC4 + CT + RT		1009.77	100	0.08	0.27	0.42	100	100	100

Date	24-01-08
Ore	Sheba's Ridge
pH	natural

Reagents	**Type**	**Dosage**	**Add. Seq**	**Cond. Time**
Frother	Dow 200	40, 20	3	1min
Collector	SNPX	25, 10	1	in mill, 2min
Depressant 1	Dep 267	40, 100	2	5min
Depressant 2				5min
pH modifier				

Product	**Time**	**Water**			**Solids**	
		Total in Conc	**Recovered**	**Cum Rec**	**Rec**	**Cum Rec**
	min	**g**	**g**	**g**	**g**	**g**
FF Cl C1	2	35.63	8.86	8.86	4.53	4.53
FF Cl C2	5	45.75	5.13	13.99	1.69	6.22
FF Cl C3	10	75.81	7.54	21.53	1.61	7.83
FF Cl C4	20	114.15	16.63	38.16	2.15	9.98
Combined FF Conc		**271.34**	**38.16**		**9.98**	

Product	**Time**	**Mass**	**Mass**	**Grade (%)**			**Recovery (%)**		
	min	**g**	**%**	**Cu**	**Ni**	**S**	**Cu**	**Ni**	**S**
FF Cl C1	2	4.53	0.45	6.97	12.2	26.3	28	33.45	27.54
FF Cl C2	5	1.69	0.17	4.71	11.6	22.7	7.06	11.87	8.87
FF Cl C3	10	1.61	0.16	3.33	9.55	20.7	4.75	9.31	7.7
FF Cl C4	20	2.15	0.21	2	6.76	16.9	3.81	8.8	8.4
Combined FF Conc		9.98	0.99	4.93	10.5	22.76	43.63	63.42	52.51
Cleaner Tails		66.01	6.58	0.19	0.31	1.18	11.12	12.39	18
Rougher Conc		75.99	7.57	0.81	1.65	4.01	54.75	75.81	70.51
Rougher Tails		850.47	84.72	0.06	0.05	0.15	45.25	24.19	29.49
Other samples		77.35	7.71						
Head (meas)		1003.81	100	0.12	0.16	0.43	100	100	100
Head (calc)			100	0.11	0.16	0.43			

		Mass	**Mass**	**Grade (%)**			**Recovery (%)**		
		g	**%**	**Cu**	**Ni**	**S**	**Cu**	**Ni**	**S**
FFCC1	2	4.53	0.45	6.97	12.2	26.3	28	33.45	27.54
FFCC1 + CC2	5	6.22	0.62	6.36	12.04	25.32	35.06	45.32	36.41
FFCC1 + CC2 + CC3	10	7.83	0.78	5.73	11.53	24.37	39.81	54.62	44.11
FFCC1 + CC2 + CC3 + CC4	20	9.98	0.99	4.93	10.5	22.76	43.63	63.42	52.51
FFCC1 + CC2 + CC3 + CC4 + CT		75.99	7.57	0.81	1.65	4.01	54.75	75.81	70.51
FFCC1 + CC2 + CC3 + CC4 + CT + RT		1003.81	100	0.12	0.16	0.43	100	100	100

Date	7/3/2008
Ore	Sheba's Ridge
pH	natural and 10.5

Reagents	**Type**	**Dosage**	**Add. Seq**	**Cond. Time**
Frother	Dow 200	40, 20	3	1min
Collector	SNPX	50+50, 10	1	in mill, 2min
Depressant 1	Dep 267	40	2	5min
Depressant 2	Sty 504	150	2	5min
pH modifier	lime			

Product	**Time**	**Water**			**Solids**	
	min	**Total in Conc**	**Recovered**	**Cum Rec**	**Rec**	**Cum Rec**
		g	**g**	**g**	**g**	**g**
FF Cl C1	2	41.93	19.09	19.09	7.63	7.63
FF Cl C2	5	36.49	8.5	27.59	1.61	9.24
FF Cl C3	10	54.32	13.43	41.02	1.52	10.76
FF Cl C4	20	88.9	26.95	67.97	2.16	12.92
Combined FF Conc		**221.64**	**67.97**		**12.92**	

Product	**Time**	**Mass**	**Mass %**	**Grade (%)**			**Recovery (%)**		
	min	**g**		**Cu**	**Ni**	**S**	**Cu**	**Ni**	**S**
FF Cl C1	2	7.63	0.76	5.07	10.9	22.5	57.76	33.27	35.84
FF Cl C2	5	1.61	0.16	3.03	7.94	19.3	7.28	5.11	6.49
FF Cl C3	10	1.52	0.15	2.19	5.78	16.1	4.97	3.51	5.11
FF Cl C4	20	2.16	0.22	1.41	3.85	12.6	4.55	3.33	5.68
Combined FF Conc		12.92	1.29	3.87	8.75	19.69	74.57	45.23	53.12
Cleaner Tails		68.16	6.78	0.13	0.39	1.26	12.72	10.5	17.86
Rougher Conc		81.08	8.07	0.72	1.72	4.19	87.29	55.73	70.98
Rougher Tails		851.18	84.73	0.01	0.13	0.16	12.71	44.27	29.02
Other samples		72.36	7.2						
Head (meas)		1004.62	100	0.07	0.27	0.48	100	100	100
Head (calc)			100	0.07	0.25	0.48			

		Mass	**Mass %**	**Grade (%)**			**Recovery (%)**		
		g		**Cu**	**Ni**	**S**	**Cu**	**Ni**	**S**
FFCC1	2	7.63	0.76	5.07	10.9	22.5	57.76	33.27	35.84
FFCC1 + CC2	5	9.24	0.92	4.71	10.38	21.94	65.05	38.39	42.33
FFCC1 + CC2 + CC3	10	10.76	1.07	4.36	9.73	21.12	70.02	41.9	47.44
FFCC1 + CC2 + CC3 + CC4	20	12.92	1.29	3.87	8.75	19.69	74.57	45.23	53.12
FFCC1 + CC2 + CC3 + CC4 + CT		81.08	8.07	0.72	1.72	4.19	87.29	55.73	70.98
FFCC1 + CC2 + CC3 + CC4 + CT + RT		1004.62	100	0.07	0.27	0.48	100	100	100