AN INVESTIGATION INTO CHEMICAL FACTORS THAT AFFECT THE BEHAVIOUR OF GANGUE MINERALS IN THE FLOTATION OF PGM ORES

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Declaration

I hereby certify that the this thesis is the result of my own work and has not been submitted prior to this for any higher degree to any other university or institution

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T.D. Mailula

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Glory and praise to our God who alone gives light to our days. Many are the blessings he bears to those who trust in His ways. Thanks to my family for believing in a strange boy like me.

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ABSTRACT

Flotation is one of the key steps in the beneficiation of Platinum Group Elements (PGE) from the Bushveld Complex in South Africa. This process aims to optimise the recovery of valuable PGE and sulphide minerals and the rejection of unwanted gangue minerals. The major reefs are exploited in South Africa i.e. Merensky reef and UG-2 reef. Gangue minerals, pyroxene and feldspar constitute more than 80% of the Merensky reef and chromite up to 60% of the UG-2 reef. Although these minerals are considered to be hydrophilic and thus not floatable, these gangue minerals are known to report to the concentrate in large quantities during froth flotation. This results in lower grades and increased difficulties in downstream processing. Some Platinum Group Mineral (PGM) concentrators report that 2/3 of the gangue collected on the concentrate is by true flotation and only 1/3 by entrainment [Malysiak et al, 2001]. This also suggests that mechanisms other than entrainment are responsible for this phenomenon. The behaviour of these minerals is selected for investigation in this thesis.

During flotation of PGM ores, reagents such as copper sulphate and sodium iso-butyl xanthate (SIBX) are added to enhance the recovery of valuable minerals. Copper sulphate, although added as an activator for valuable minerals, as the conditions used has been shown to also activate gangue minerals, thus its activation is non-selective [Shackleton, 2003 and Wesseldijk et al, 1999].

Wesseldijk et al, (1999), showed that copper sulphate could activate chromite and the subsequent addition of collector resulted in high recoveries of chromite in contrast to the low recoveries obtained in the absence of copper sulphate.

Recent work by Malysiak et al, (2001) also showed that the water quality used in flotation, plays an important role in the way reagents interact with mineral surfaces and can also affect inadvertent activation of gangue minerals.

Polymeric depressants such as guar and carboxy methyl cellulose (CMC) are used in the PGM industry to reduce the recovery of floatable gangue, particularly talc. However the effect on other gangue minerals has not been studied, neither has the behaviour of depressants on activated minerals. This thesis investigates the effect copper sulphate and water quality on the recovery of pyroxene, feldspar and chromite and the ability of polymeric depressants to reverse this activation.

Microflotation tests were done to investigate the behaviour of feldspar and pyroxene. Batch flotation tests were not done with Merensky ore due to the assay problem with pyroxene and feldspar. Wesseldijk et al, (1999) previously investigated the effect of copper sulphate addition on the activation of chromite in microflotation tests and so, batch flotation studies with an ore containing chromite were done to investigate chromite. In order to evaluate mechanisms in microflotation, depressants at equilibrium conditions were added and operating conditions were in ppm (2 gram mineral: 100 ml solution: depressant in ppm). In batch flotation tests the conditions were set up to be compared to plant (1 kg ore: 2 litres water and dosages in g/ton). This has meant that the report is very much two discrete parts with an overall theme.

Microflotation tests with single mineral samples showed that pyroxene and feldspar are naturally hydrophilic in the absence of reagents. The addition of collector (SIBX) only, did not change the flotation response of these minerals. The results also indicated that activation is pH dependent. No activation was observed at pH 4 in contrast to that which occurred at pH 9. At pH 9, the addition of copper sulphate and collector in the presence of $1*10^{-3}$ M KNO_3, resulted in high recoveries of these minerals and when added in reverse order no increase

in recovery was observed. Lower recoveries were achieved with copper sulphate and SIBX at an ionic strength of $1*10^{-2}$ $Ca(NO_3)_2$.

The effect of ionic strength and type of metal ion used was investigated. It was observed that changing the ionic strength of KNO_3 from $1*10^{-3}$ to $1*10^{-2}$ did not have any effect on flotation recoveries. When $Ca(NO_3)_2$ was used as the electrolyte reduced flotation recoveries were observed. The recoveries were reduced further when ionic strength was increased from $1*10^{-3}$ to $1*10^{-2}$. thus the metal ion Ca^{2+} showed different characteristics compared to K^{+}.

The addition of a guar at low concentrations in the presence of copper sulphate and SIBX at ionic strength of $1*10^{-2}$ $Ca(NO_3)_2$ showed the reversal of activation and hence low recoveries of pyroxene and feldspar. This observation was not realised when KNO_3 as the electrolyte was added. At an ionic strength of $1*10^{-3}$ both KNO_3 and $Ca(NO_3)_2$ did not show any reversal in activation irrespective of the addition of depressants. Neither was the reversal observed when CMC was used, although lower charge CMC (Depramin C) achieved lower recoveries than high charge CMC (Depramin 186).

The batch flotation work with UG-2 are showed that in the absence of copper sulphate addition and at a dosage of 30 g/ton copper sulphate, chromite recovery was proportional to water recovery. When the copper sulphate dosage was increased to 90 g/ton, chromite recoveries were enhanced. When either carboxy methyl cellulose (CMC) or guar was added at 100 g/ton, this enhancement in chromite recovery was not reversed. However when guar dosage was increased to 200 g/ton, the chromite recoveries were reduced, the reduced chromite recovery was not observed when CMC was used at 200 g/ton.

The results of this investigation revealed that copper activation is unselective and pH dependent. Also, it showed that the gangue mineral flotation response is affected by water quality. Furthermore, the addition of guar seems more effective

than CMC in reversing the inadvertent activation of gangue minerals by copper sulphate.

The activation of gangue minerals can be attributed to the form of the copper species present in the solution and on the mineral surface at pH 9. These species enhance the collector absorption and thereby increasing hydrophobicity. The copper speciation calculations indicate that at pH 9, $Cu(OH)_2$ and lesser amounts of other copper hydroxide species are present [Wesseldijk et al, 1999]. This finding is supported by zeta potential measurements reported by Martinovic, (2002) and Shackleton, (2003). It is proposed that the mechanism of gangue activation involves the interaction of these species with the mineral surface with subsequent increased adsorption of collector. The work also shows that this can be reversed by the action of guar more effectively than that of CMCs, which has been attributed to the increased adsorption of guar onto the mineral surface. The adsorption characteristics and interaction of guars and CMCs should further be investigated and the use of mineral surface chemical analysis will be necessary.

GLOSSARY

AA	**Atomic Absorption**
APX4M	**a commercial modified guar product manufactured by African Product Exchange**
CMC	**carboxy methyl cellulose**
Depramin 186	**a commercial carboxy-methyl cellulose supplied by Akzo Nobel Functional Chemicals**
Depramin C	**a commercial carboxy-methyl cellulose supplied by Akzo Nobel Functional Chemicals**
DTP	**Dithiophosphate**
KU-5	**a commercial carboxy-methyl cellulose supplied by GM Associates**
IMP4	**a commercial guar gum product no longer commercially available**
PGM	**Platinum Group Minerals**
PGE	**Platinum Group Elements**
SIBX	**Sodium-isobutyl-xanthate supplied by Senmin**
SK 5	**A commercial reagent containing dithiophosphate supplied by Senmin**
UG-2	**Upper Group 2**

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CHAPTER 1: INTRODUCTION

The Bushveld Complex in South Africa contains the world's largest deposit of Platinum Group Minerals (PGMs). Several reefs are currently exploited for PGMs including Merensky and UG-2 [Liddell et al, 1986]. The flotation process is used to concentrate PGMs from the host reef. Merensky reef contains pyroxene, feldspar and talc as the major gangue minerals whereas the UG-2 reef contains a high proportion of chromite in addition to the pyroxene, feldspar and talc.

In flotation, the mined ore is crushed using primary and secondary crushers before being transferred into a mill. Inside the mill the ore is mixed with water to achieve a specified solids to water ratio and the ore is ground to a desired size fraction suitable for flotation. Prior to flotation, certain reagents are added to the pulp to achieve the necessary pulp-chemical environment in preparation for flotation. These reagents can be classified into the following categories: collectors, frothers, activators, depressants and dispersants. Collectors and activators are used to promote the flotation of valuables whilst depressants and dispersants are added to reduce gangue recovery. Frothers are used to create stable bubbles and a froth zone that is required to separate and collect the valuables.

Once reagents are added, flotation is achieved by transferring the pulp into flotation cells and the air is introduced into the cell to form bubbles in a turbulent zone. This facilitates the bubble-particle collision and selective attachment of hydrophobic particles. In PGM flotation, high molecular weight polymeric depressants are added to adsorb onto the surface of unwanted hydrophobic gangue minerals such as talc to render them hydrophilic. It is important that the depressants are selective and do not prevent valuable sulphide minerals from floating. The depressants currently used in the Platinum industry in South Africa are polymeric organic polysaccharides that can be classified into two categories, namely carboxy methyl cellulose (CMC) and modified guar gums (guars).

Material also reports to the concentrate unselectively by entrainment. This mechanism is dependent on different factors in flotation such as water recovery, particle size and density among others. Talc is known to be hydrophobic and its recovery leads to undesired high froth stability and subsequent recovery of other gangue minerals by entrainment. Pyroxene, feldspar and chromite are considered to be naturally hydrophilic and hence their recovery is attributed to entrainment.

Pyroxene and feldspar constitute roughly about 81% of the minerals found in the PGM bearing Merensky Reef of the Bushveld Complex whereas chromite can be up to 60% in the UG-2 ore. Although these minerals are hydrophilic they report to the concentrate during flotation [Malysiak et al, 2001 and Wesseldijk et al, 1999]. It is possible that the reagent suite currently used by platinum producers could be responsible for the inadvertent gangue activation and subsequent undesired flotation.

During the addition of reagents, activation may occur whereby some reagents can change the surface characteristics of certain minerals and hence render them to be amenable to true flotation. Copper sulphate, added as an activator for sulphide minerals, has been known to activate some gangue minerals and in the presence of collectors these minerals have been rendered floatable [Wesseldijk et al, 1999].

The current work has focused on the possible activation of naturally hydrophilic gangue minerals by activators and subsequent flotation by collector addition and has investigated whether the use of depressants can counteract this phenomenon. Typical reagents both CMCs and modified guars were used to evaluate whether this behaviour was similar for pyroxene, feldspar and chromite.

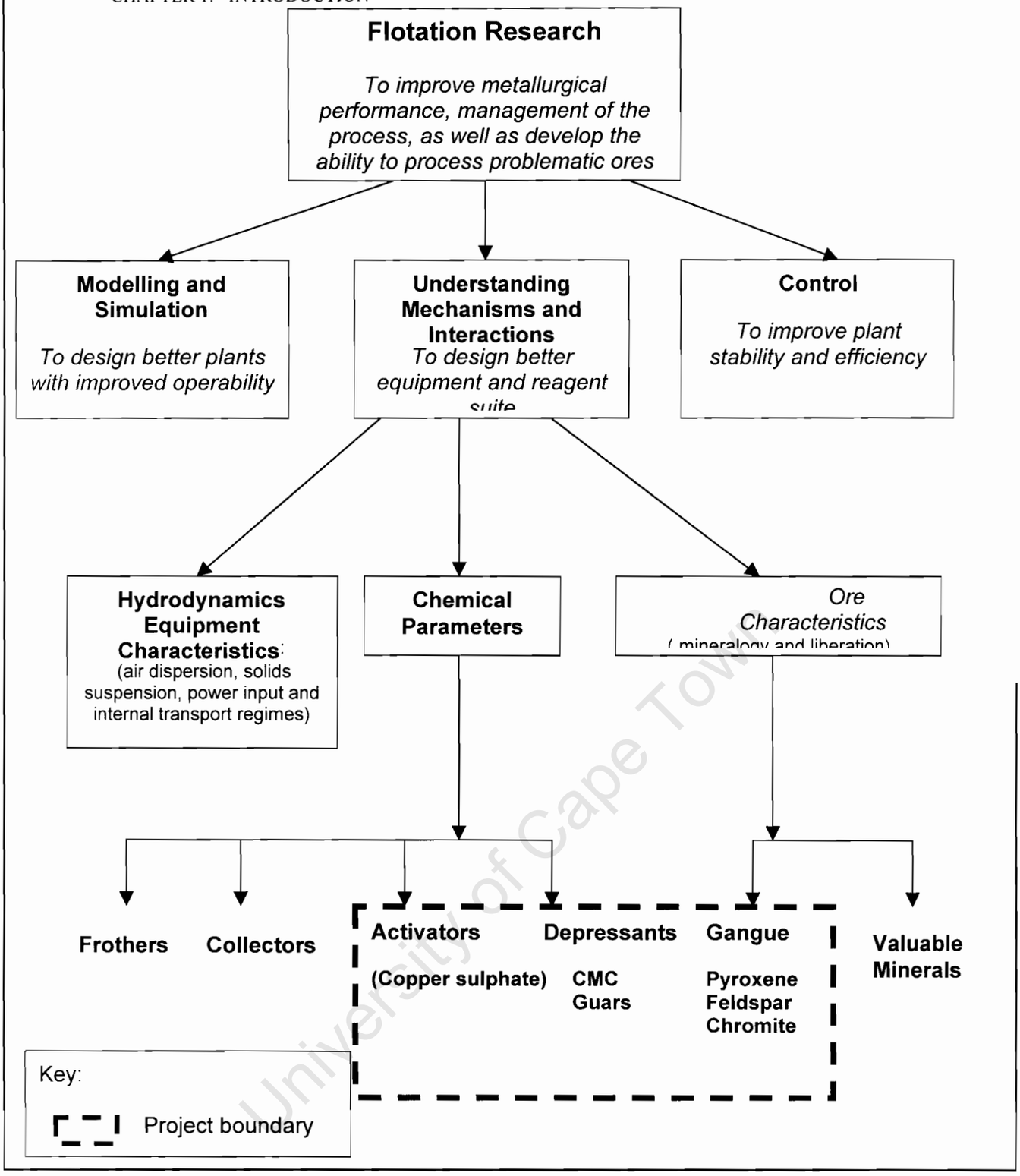

Figure 1.1: Schematic presentation of scope of this project within flotation research

CHAPTER 2: LITERATURE REVIEW

2.1. Platinum Group Minerals reef mineralogy

2.1.1 Overview

Two major Platinum Group Mineral (PGM) bearing ore bodies are exploited in South Africa: the Merensky and Upper Group 2 (UG-2) reef. The Merensky reef has been mined since 1926 [Hochreiter et al, 1985; Liddell et al, 1986], however, the exploitation of the UG-2 ore body only started in the mid-1970's, owing to difficulties in the processing of the ore [Liddell et al., 1986].

The Merensky reef is a persistent layer within the gigantic layered intrusive known as the Bushveld Igneous Complex (BIC), which underlies an area of about 40 000 km^2 in the central Gauteng and North West Province [Hochreiter et al, 1985]. It occurs near the top of what is called the Critical Zone within the BIC. It consists mainly of the chromitite-pyroxenite-norite triplet but the chromitite is poorly developed. The pyroxenite varies in width from around 60 cm to a maximum of 3 m. The Merensky Reef that is found in the Rustenburg area consists of a coarse-grained pegmatoidal feldspathic pyroxenite containing appreciable phlogopitic mica. The width varies from 15 to 45 cm [Hochreiter et al, 1989]. Thin chromitite layers usually mark the top and bottom contacts of the coarse-grained material. Immediately above the top chromitite layer is a brownish pyroxenite that is around 60 cm thick and this is known as the 'Merensky pyroxenite'.

Platinum Group Minerals (PGM) occur within the base-metal sulphides. There are three main base-metal sulphides found in the Merensky Reef, pyrrhotite ($Fe_{1-x}S$), pentlandite $(Fe,Ni)_9S_8$, and chalcopyrite ($CuFeS_2$) and are here named in the order of decreasing abundance. Due to the fact that a substantial amount of the PGM's are associated and located within these base-metal sulphides, the Merensky Reef yields substantial copper and nickel as by-products.

The base-metal sulphides and their associated PGM are not restricted to the Merensky Reef itself but are disseminated for some way into the hangingwall pyroxenite and the footwall norite [Hochreiter et al, 1985]. The Merensky horizon is not uniform throughout the Bushveld Complex. In the eastern and southern Bushveld, the Merensky pyroxenite is strongly developed, and much of the mineralisation is located in it [Hochreiter et al, 1985].

The major gangue minerals found in the UG-2 ore are talc, chlorite, phlogopite, orthopyroxene and plagioclase [Corrans et al., 1982; Liddell et al., 1986]. The major base metal sulphides present are pentlandite $(Fe,Ni)_8S_9$. Pyrrhotite ($Fe_{1-x}S$) and chalcopyrite ($CuFeS_2$) are found in subordinate amounts [Corrans et al., 1982].

2.1.2. Geology and mineralogy of Pyroxene

Pyroxenes belong to the silicate mineral class and approximately 30% of all minerals are silicates. The pyroxenes constitute one of the most important groups of rock-forming silicates. In different members of the group the elements magnesium, calcium and sodium are present in widely varying proportions, together with smaller amounts of aluminium, titanium and lithium in some species. The basic chemical unit of silicates is the SiO_4 tetrahedron shaped anionic group with a negative four charge (4-). The pyroxenites are coarse-grained, usually deep-seated rocks and normally there is neither feldspar nor olivine found in them. Pyroxenites may be monomineralic, or more than one kind of pyroxene may occur (e.g. orthopyroxene and clinopyroxene). There is a significant amount of accessories like chromite or magnetite present in the pyroxenites. In the Bushveld Complex of South Africa the chromite occurs in more abundance than the orthopyroxene, which occurs in large crystals enclosing crowds of octahedra of chromite [F.H. Hatch et al, 1949].

2.1.2.1 The structure of pyroxene

The pyroxene minerals are inosilicates of the general formula $XY(Si, Al)_2O_6$. The X, represents ions such as Ca^{2+}, Fe^{2+} and Mg^{2+} and more rarely Zn^{2+}, Mn^{2+} and Li^{2+}. The Y, represents ions of generally smaller sizes such as Cr^{3+}, Al^{3+}, Fe^{3+}, scandium, titanium, vanadium and even Fe^{2+}. The study of the X-ray structure of the pyroxenes has shown that the SiO_4 tetrahedra are linked together vertically into chains, each tetrahedron sharing two oxygens with those immediately above and below in the chain. The individual chains are joined together through the medium of the cations, Ca(II), Mg(II), Fe(II) etc., which are linked to the "free" (i.e. not shared) oxygens. In all pyroxenes the chains run parallel to the vertical crystallographic axis, and are arranged in sheets parallel to (1 0 0). The form and disposition of the chains determine the positions of the cleavage planes (illustrated in figure 2.2).

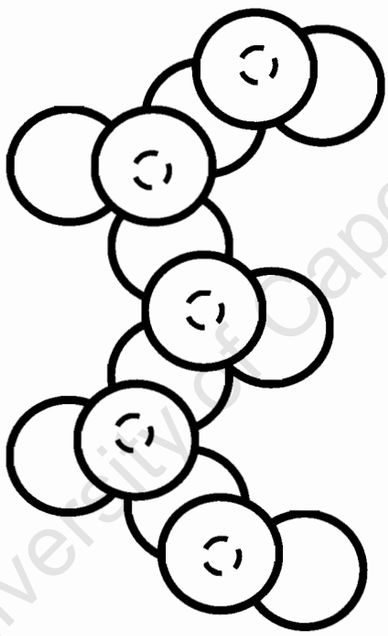

Figure 2.1: Part of a chain of SiO4 tetrahedra, as in pyroxenes. The unit of pattern contains (Si2O6). Si-atoms shown by broken circles, large circles are O-atoms. [Hatch et al, 1949].

The unit pattern in the pyroxene chains contains Si_2O_6. To balance the valency two Mg(II) ions must be added, giving $Mg_2Si_2O_6$, which may be reduced to $MgSiO_3$ – magnesium metasilicate – or the mineral enstatite. Theoretically in ferrosilite, all the cations are Fe(II), the formula being $FeSiO_3$, metasilicate of iron. Between these two extremes many combinations exists. A third essential component of the pyroxenes is the corresponding metasilicate of calcium, $CaSiO_3$, which is the formula of the mineral wollastonite.

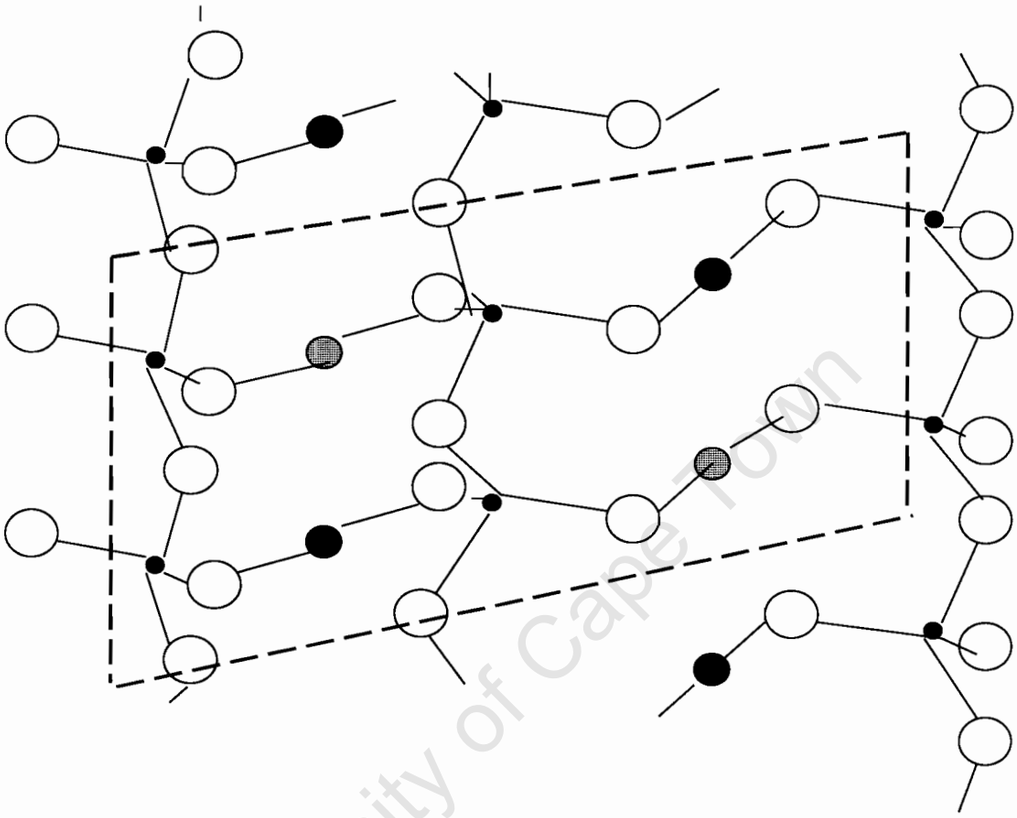

Figure 2.2: Portions of three parallel pyroxene chains projected on the plane (0 1 0). The unit cell is outlined, but only parts of its contents are shown. Si(IV) black; O(II) plain; Mg(II) dotted; Ca(II) ruled. [Hatch et al, 1949].

If certain minor constituents are ignored, the composition of the pyroxenes may be represented by points in a triangle (see figure 2.3). The aspices of the triangle represent the pure metasilicates of calcium, represented by Wo, magnesium (En), and iron (Fs). All pyroxenes may be represented by a simple formula expressing the molecular percentages of these three components. The base of the triangle embraces all possible proportions of En to FS, thus covering the compositional range of the important group of orthopyroxenes, and the chemically identical monoclinic equivalents [Hatch et al, 1949]. The pigeonites and the augites are both orthopyroxenes but they are also distinct mineral species. The pigeonites are calcium-poor and their composition can be related to those of orthopyroxenes.

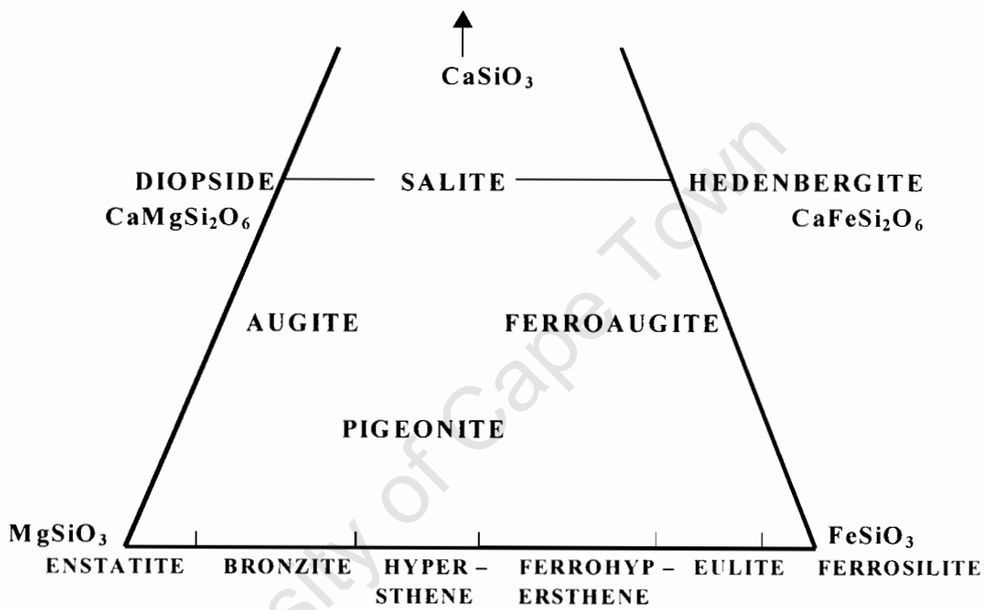

Figure 2.3: Triangular diagram showing compositional relationship between the common pyroxenes.

In this work, the ore used contains mainly enstatite and diopside.

2.1.2.2 Distinction between the orthopyroxenes and the clinopyroxenes

The orthopyroxenes belong to a series called the enstatite-ferrosilite series. It has been shown that pyroxenes within this range may occur in two forms, one orthorhombic and the other monoclinic. Orthopyroxenes that crystallise at higher temperatures are richer in Mg than those that separate at lower temperatures. Thus the shift from more basic to more acidic rocks, the pyroxene changes from magnesium-rich to iron-rich. Clinopyroxenes are calcium rich series and occupy the upper part of the triangle shown in figure 2.3.

2.1.3. Structure of the feldspars (plagioclases)

The fundamental unit of structure in crystalline silica is the positive silicon atom surrounded by four oxygen atoms in tetrahedral co-ordination. In the silica minerals, these tetrahedra are linked together by sharing corners to form interlocking oxygen-silicon chains building up a continuous three-dimensional framework. Machatschki (1928) pointed out that the feldspar minerals are characterised by a silica framework in which from one-quarter to one-half of the tetravalent Si atoms are replaced by trivalent Al atoms [Barth, 1969]. The two feldspar anions representing the ultimate ratios of substitution are:

$[AlSi_3O_8]^-$ in which every fourth Si is replaced by Al

$[AlSi_2O_8]^{2-}$ in which every second Si is replaced by Al

Feldspars are electrically balanced by the presence of positive cations like Na^+, K^+ or Ca^{2+}, which are induced into the framework. Loewenstein (1954) showed that Al could replace a maximum of 50% of the silicones in the three-dimensional frameworks. For 50% substitution, rigorous alternation between Si and Al tetrahedra becomes necessary: the unit cell of anorthite has double the size of that of albite. The unit cell or semi-unit cell of all feldspars has a volume around 700 cubic angstroms and of the following approximate dimensions:

a = 8.2 angstroms	$\alpha = 90°$
b = 13.0 angstroms	$\beta = 116^0$
c = 7.1 angstroms	$\gamma = 90^0$

The tetrahedral framework of the anorthite lattice is shown in the figure below:

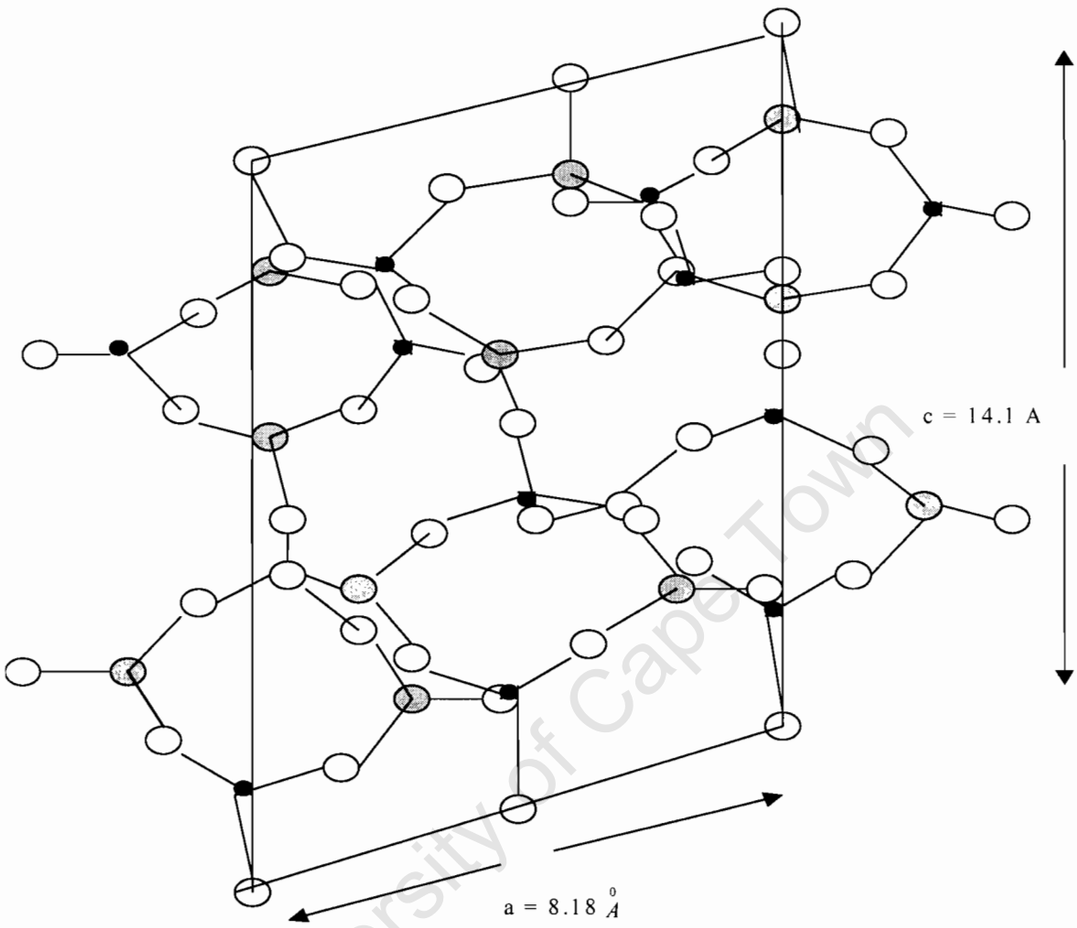

Figure 2.4: The tetrahedral framework of the anorthite lattice projected down the b axis on the plane (0 1 0). A Si-Al-oxygen chain runs parallel to the a axis. The anorthite unit cell is outlined; it has the long 14-angstrom c axis. The prominent cleavages parallel to (0 0 1) and (0 1 0) cut only the cross-linkages between the zigzag chains parallel to the a axis [Lowenstein, 1954].

2.1.4 Chromite

2.1.4.1 Chromite in PGM processing

The general formula for chromite can be written as $[(Fe,Al)O.(Cr,Mg,Fe)_2O_3$. The UG-2 ore treated in South Africa contains about 20 – 60 % chromite that has a chromium-to-iron ratio of approximately 1.35. One of the constraints to flotation of UG-2 ore is that the (Cr_2O_3) obtained in the concentrate, should be minimised, since it can cause major problems during smelting. Chromite is a member of the spinel group and forms components that may be stable up to temperatures as high as 2000 ^{0}C. The spinel is often the first phase that crystallises and forms solids [McKenzie, 1996]. The presence of these solids in the furnace reduces the efficiency of the smelting process and results in unnecessary losses of the PGMs.

2.1.4.2 Structure of Chromite

Chromite, a member of the spinel group of minerals, is an oxide and has the general chemical formula $FeO.Cr_2O_3$, i.e. 68% Cr_2O_3. However, part of the ferrous iron is usually replaced by magnesium and part of the chromium by aluminium and/or ferric iron. A more realistic formula is therefore (Fe, Mg)O.(Cr, Al, $Fe)_2O_3$ with 45 to 55 % Cr_2O_3 [Sobieraj and Laskowski, 1973; Berkman, 1976; Coertze and Coetzee, 1976]. Substitution of the ferrous iron by calcium (Ca^{2+}) and manganese (Mn^{2+}) and chromium by Ti^{4+} and Si^{4+} are also possible [Aplan, 1985]. The trivalent ions such as chromium and aluminium are co-ordinated octahedrally with oxygen, while the divalent ions are tetrahedrally co-ordinated (figure 2.5) [Guney et al., 1993]. This characteristic property makes the trivalent ions more insoluble than the divalent ions, although conflicting results on the Al^{3+} species have been reported [Sobieraj and Laskowski, 1973].

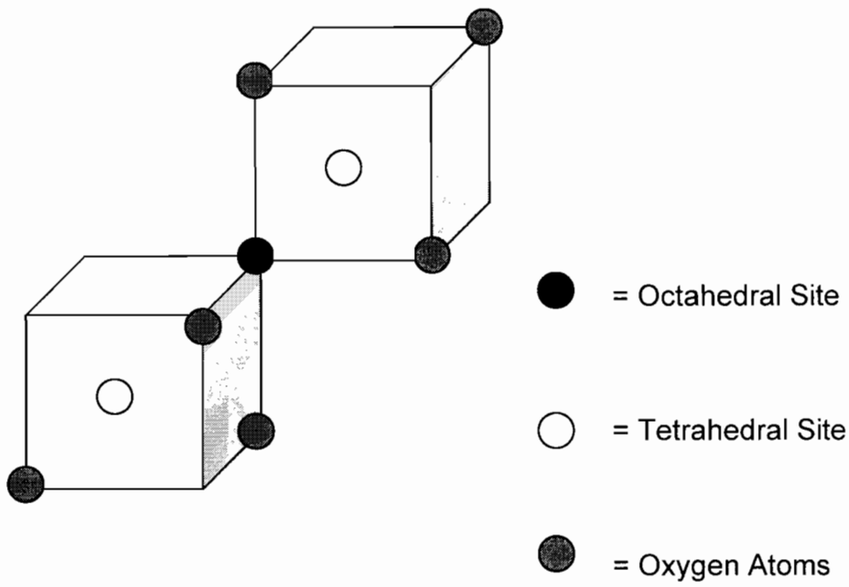

Figure 2.5 – Schematic representation of the crystal of spinel, showing the cubic arrangement of the oxygen atoms and the relative positions of the sites of tetrahedral and octahedral co-ordination [McKenzie, 1996]

2.2. Principles of froth flotation

2.2.1 Froth Flotation

Figure 2.6 shows a simplified schematic of the flotation process. There are two distinct zones: The pulp zone, in which mineral particles attach to bubbles and the froth zone, in which the concentrated mineral separated from the bulk. Particles can either reach the froth attached to bubbles or by entrainment in the water passing from the pulp zone to the froth zone. While the former process is selective and is responsible for the collection of the hydrophobic valuables, the latter is unselective and results in the unwanted gangue reporting to the flotation concentrate. It is thus desirable to maximise the recovery by true flotation and minimise the entrainment contribution. The contribution to overall flotation of the entrained material increases linearly with increase in water recovery [Smith and Warren, 1989].

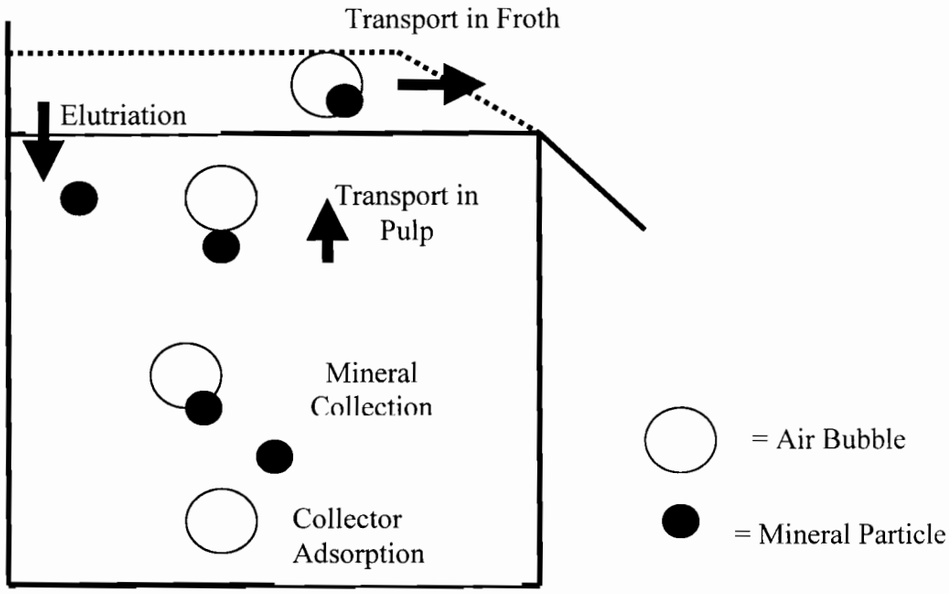

Figure 2.6 – Schematic of the flotation process [Bradshaw, 1997].

The froth zone provides the environment for the separation of the valuable mineral from the gangue, allowing drainage of the entrained material back into the pulp. When the froth is not stable enough the mineralised bubbles rupture before collection, when the froth is too stable, not enough drainage occurs and the water and gangue recoveries are too high. The nature and dosage of the frother as well as the nature of the particles in the froth affect its stability [Harris, 1982].

The overall flotation process can be divided into various consecutive sub-processes which all contribute to the success of the separation of the valuable mineral from the gangue. They can be summarised as follows: [Bradshaw, 1997]:

1. The creation of the hydrophobic mineral particle surface, which may consist of the collector adsorption onto the surface of the mineral particles and the manipulation of the pulp environment viz. pH, agitation, pulp density etc.

2. The formation of stable bubbles of a set size and distribution.

3. Mineral particle – bubble collision, resulting in possible attachment. This requires the thinning and rupture of the film between the bubble and particle and subsequent stable attachment. Detachment of the mineral from the bubble is also possible. This sub-process is also known as the particle collection.

4. The transport of the loaded bubble through the pulp phase.

5. Transfer from the pulp phase to the froth phase. There is possible elutriation of particles falling back from the froth into the pulp and entrainment of particles, not collected by bubbles, passing from the pulp into the froth.

6. The transport and collection of the loaded bubble in the froth phase.

2.2.2 Influential Parameters in Flotation

There are many factors that affect the flotation processes, both directly and indirectly. Crozier (1992) has listed over 25 clearly identifiable parameters, which can be more fully described by over 100 variables, which affect the flotation performance.

Klimpel had divided the major variables into three interactive groups: the equipment components, the operation components and the chemical components (Figure 2.7) [Heerema, 1994; Bradshaw, 1997]. The interactive nature of these components makes it difficult to analyse the effect of any particular component and careful planning and analysis of experiments is necessary to interpret the effects on performance of a particular parameter.

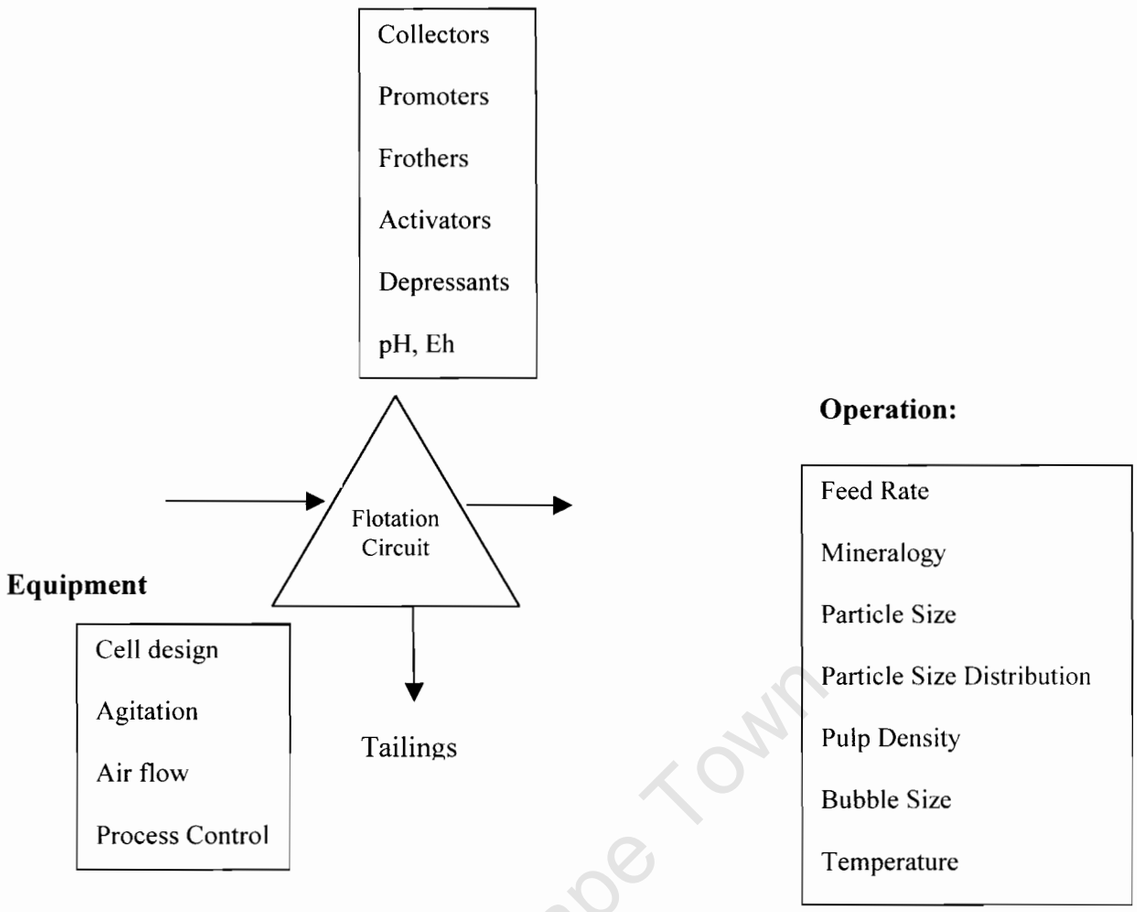

Figure 2.7: Klimpel triangular representation of flotation [Heerema, 1994; Bradshaw, 1997].

2.3 Reagents

Several types of reagents are used in flotation so as to manipulate the chemical environment and to optimise the recovery of valuable minerals. These include frothers, collectors, activators and depressants.

In PGM flotation, collectors are added to the pulp to enhance the hydrophobicity of sulphide minerals and hence recovery thereof. The most common type of collectors used in PGM flotation are thiol collectors and of these the most widely used are xanthates. In general their structures have a non-polar group that remains in liquid and a polar group that absorbs on the water mineral interface.

The role of frothers is to provide stable bubbles in the pulp phase and a suitably stable froth phase above the pulp where the separation of hydrophobic and hydrophilic minerals can be effected. The most common frothers in PGM flotation are alcohol based and vary in type. Some examples are SK 700 and SASFROTH.

Activators enhance the role of the collector in increasing the flotation of slow floating sulphide minerals. Copper sulphate is the most common activator in PGM flotation used to activate sulphide minerals like pyrrhotite. Its use is controversial and dosages vary from 0 to 90 g/ton. In some cases it is used as a froth modifier.

Depressants are added to suppress the recovery of unwanted hydrophobic gangue minerals such as talc. Although the proportion of talc in PGM ores can be small, it has a froth stabilising effect that causes increased entrainment of other gangue minerals. Polymeric depressants are generally used in PGM flotation. These can be further divided into two types i.e. guars and carboxy methyl cellulose (CMC) [Pugh, 1988].

2.3.1. Polymeric depressants

2.3.1.1 Modified Guar Gum

Guar is an agricultural product, derived from Cyamopsis tetragonolobus [Mackenzie, 1980], which is chemically modified by alkaline degradation. Guar has a branched structure and has a low degree of substitution yet is soluble in water. It is thought that the cis-configuration of the OH^- groups and its branched nature increases solubility. A monomer of guar is shown in figure 2.9 below.

Guar gum is defined as a non-ionic polysaccharide with linear chains of (1-4-β-D-mannopyranosyl units with α-D-galactopyranosyl units attached by 1-6 linkages [Whistler, 1973]. Unlike the CMC's, the guar gums are soluble in water even at low degrees of substitution. As it can be observed from the diagram that guars are non-ionic and on adsorption on mineral surfaces will not be expected to increase charge and may tend to be coagulants [Harris et al, 1999].

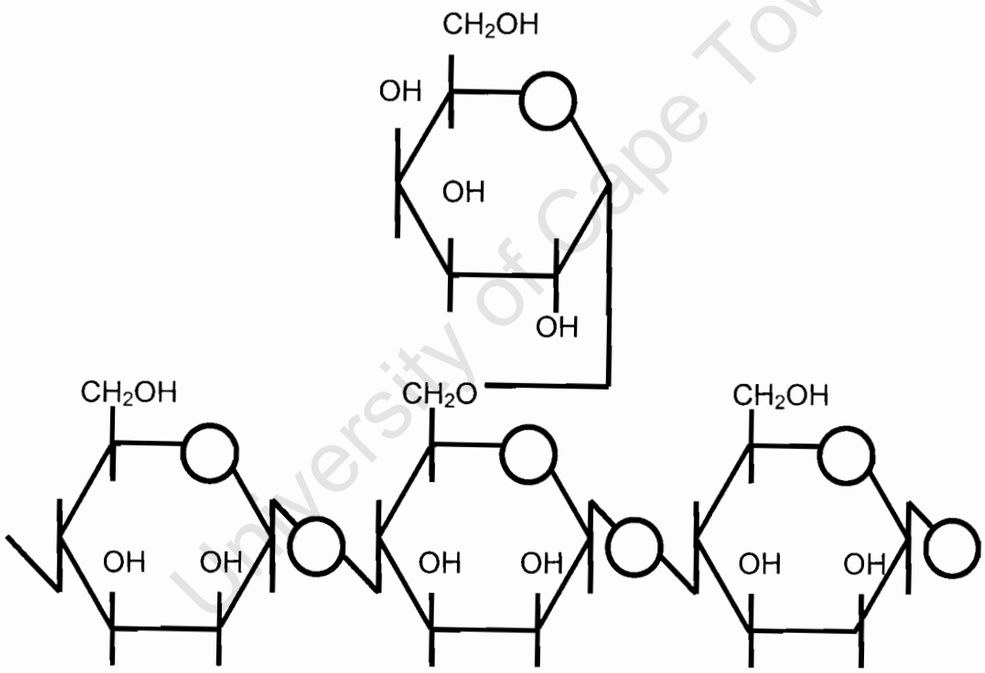

Figure 2.8: A schematic diagram of the molecular structure of guar gum

2.3.1.2 Carboxymethyl Cellulose (CMC)

Often referred to as CMC, the carboxy methyl cellulose is an ionic polysaccharide. It is prepared by steeping in sodium hydroxide solution and the alkali cellulose is then esterified with sodium monochloroacetate to form sodium carboxy methyl cellulose and sodium chloride [Batdorf and Rossman, 1973]. The equation to describe the reaction is:

$R_{cellulose}(OH) + ClCH_2COONa + NaOH \rightarrow R_{cellulose}(OH)_2(OCH_2COONa) + NaCl + H_2O$

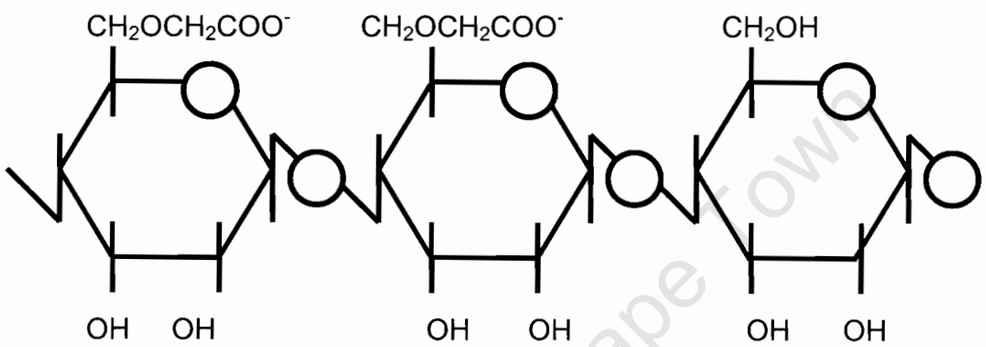

Figure 2.9: A schematic diagram of the molecular structure of Carboxy methyl cellulose (CMC).

This structure resembles the cellulose structure except that the carboxylic acid groups are substituted along the molecular chain. The degree of substitution is the number of carboxylic acid groups that are present in the cellulose molecule. These groups substitute the OH^- groups. There are three possible sites for the substitution of a carboxylic acid molecule per hexose unit (i.e. three OH sites). A possible degree of substitution would be 0.5 which implies that 1 out of every 6 sites have been substituted. Most commercial CMC's have a degree of substitution of 0.7 to 0.8 (Pugh, 1988). Cellulose is relatively insoluble in water

and a degree of substitution above 0.4 is required to make the polymeric molecule soluble in water. The ionic groups that are substituted along the chain result in the molecules being charged. CMC's have been found to be both flocculants and as dispersants [Steenberg, 1982]. Due to their high charge, on adsorption onto particles they can impact surface charge and cause particle repulsion in solution and hence dispersion.

2.4 Adsorption of guar and CMC onto talc

Guars are known to adsorb strongly and the adsorption had been shown to be independent of water quality [Steenberg and Harris, 1984; Shortridge et al, 1999b]. Gaurs adsorb more strongly onto talc than CMCs and this has been attributed to the lack of charge [Steenberg and Harris, 1984].

CMCs macromolecules ionise in water and for CMC adsorption onto a talc surface the electrostatic repulsion between its negative charge and the predominantly negative talc edge has to be overcome.

Shortridge et al, (1999b) found in microflotation tests that high ionic strength had a beneficial effect on the depression of talc by CMCs. This indicated that soluble metal hydroxide aided the adsorption of the CMC onto the talc.

Liu and Laskowski (1999) reported that both the carboxy and hydroxy groups play a role in the adsorption of the CMC onto the mineral surface. The carboxyl-group interacts with various metallic ionic species but the hydroxyl-group is only considered to react with the metal hydroxy species. This could explain why the greater depression of talc is achieved with increased ionic strength as reported by Shortridge et al, (1999b) as the greater possibility of forming metal hydroxy complexes exists [Dalvie, 2001].

The effects of molecular weight, charge and metal cations on the equilibrium adsorption of CMCs on two types of talc (New York talc and Barberton talc) have recently been investigated by Parolis et al, (2003). The work looked at adsorption of 3 CMCs (FF10, FF30 and FF150) on both talc samples in the presence of Ca^{2+}, Mg^{2+} and K^{+} ions at 10^{-3} and 10^{-2} ionic strengths. The results indicated that in the presence of divalent cations (Ca^{2+} and Mg^{2+}) the adsorption density of all the three CMCs on the talc surfaces increased in comparison with the adsorption of K^{+}. Also, what was observed was that much higher adsorption densities were obtained when the ionic strength was increased from 10^{-3} to 10^{-2}.

The molecular weight of the depressant did not have any effect on the adsorption density on New York talc but high molecular weight CMC adsorbed better on Barberton talc than low molecular weight CMCs. The work also showed that in the presence of divalent cations the adsorption density decreased with increasing CMC charge. The adsorption density of one of the CMCs (FF150) was more pronounced on the Barberton talc than on the New York talc in the presence of 10^{-2} $Ca(NO_3)_2$. This was mainly due to the difference in the plane-to-edge ratios with Barberton talc having a plane-to-edge ratio of 29:71 whereas New York talc has a ratio of 69:31.

2.5 Activation by copper sulphate

Figure 2.10 shows the species of copper ions present at various pH values.

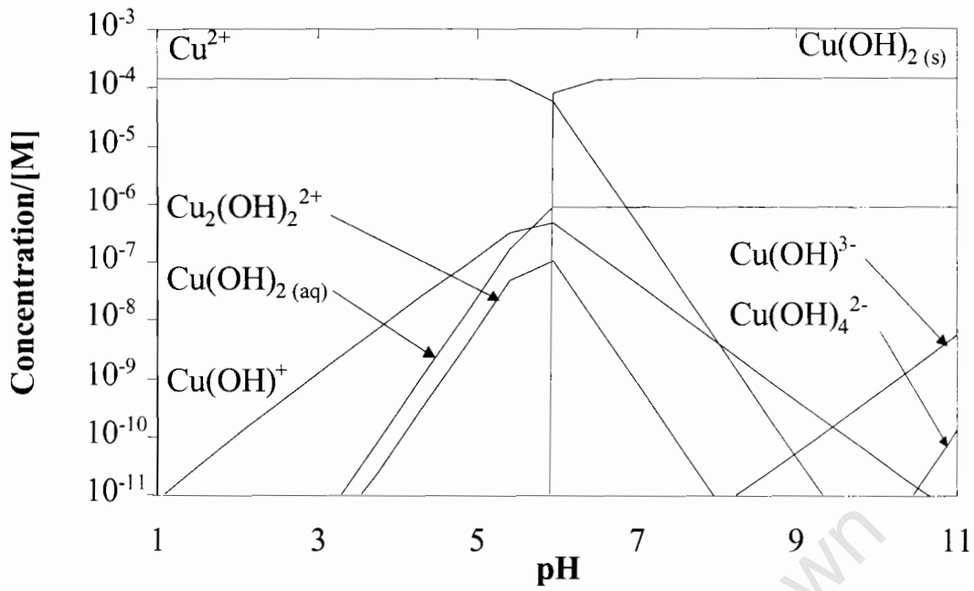

Figure 2.10: Logarithmic Concentration Diagram for $1*10^{-4}$ M Cu^{++} [Wesseldijk, 1999]

2.5.1. Classical Activation by $CuSO_4$ at low pH

As mentioned above, sulphide minerals are recovered during flotation by the use of thiol collectors. However, some of the sulphide minerals do not respond well to the utilized collectors and these minerals require the simultaneous use of chemicals called activators. Compounds such as copper sulphate are added to sulphide mineral flotation systems to improve the response of these minerals to flotation; this is a classic example of activation in flotation technology [Fuerstenau, 1982].

An important factor governing the mechanism of activation of sulphide minerals is the pH of the system. As shown in figure 2.10, depending on the pH of the system, various species can be formed.

Activation of sphalerite (ZnS) with copper sulphate has been extensively studied and the activation is controlled by the reaction:

$$ZnS(s) + Cu^{2+} \Leftrightarrow CuS(s) + Zn^{2+}$$

It has also been shown that after activation, flotation behaviour is similar to that of CuS. Gaudin et al, (1959), demonstrated that copper uptake follows the stoichiometric replacement reaction above.

However, the activation of silicate minerals does not follow the sulphide mineral activation mechanism since there are no metal ions available for substitution.

2.5.2 Activation at high pH values

During flotation of ores with high gangue content, these minerals tend to buffer the pH of the pulp at around a pH of 8.6. As a result, when copper sulphate is added, the classical activation through the mechanism of substitution does not take place. In acidic pH values when copper sulphate is added, Cu^{++} ions form in solution and hence metal substitution takes place between copper and the metal on the sulphide forming a copper sulphide surface. However, as figure 2.10 suggests, at pH values above 8.6 $Cu(OH)_2$ is the predominant species and it may precipitate unselectively on any mineral.

2.4.2.1 Proposed mechanisms of activation

From the speciation diagram of 10^{-4} M copper sulphate (figure 2.10), it is observed that the copper species dominant at pH 9 is $CuOH^+$ and $Cu(OH)_2$. The

Cu-ions will be hydrolysed to the soluble metal hydroxy species as the pH increases and at higher pH values precipitate as solid $Cu(OH)_2$, nevertheless it is likely that because of the following equilibria,

$$Cu^{2+} + H_2O \rightarrow CuOH^+ + H^+$$

And

$$CuOH^+ + H_2O \rightarrow Cu(OH)_{2(S)} + H^+$$

The reactions of the Cu^{2+} ion with the various thiol collectors, whether on the surface or in solution, will be similar. Cu^{2+} ions will react with xanthate and DTP in a similar manner.

$$Cu^{2+} + 2\ X^- \rightarrow CuX_{2\ (S)} \rightarrow CuX_{\ (S)} + \tfrac{1}{2}\ X_2$$

$$CuOH^+ + 2\ X^- \Rightarrow CuX_2 \Rightarrow CuX + \tfrac{1}{2}\ X_2 \quad \text{(Wesseldijk et al,1999)}$$

It is expected that the intermediate product will be the one that represents the copper xanthate complex in solution and that the extent of decomposition will be minimal. At pH values of around 8.4 to 10 the copper is expected to precipitate onto all surfaces of the minerals. The subsequent xanthate adsorption and formation of xanthate hydroxyl species would cause the gangue minerals to become hydrophobic and hence to float. Also at these pH values it is observed that the pyroxene surface becomes hydrated and hence more negatively charged which promotes the adsorption of copper ions. These copper ions cause the mineral surface to be positive and hence cause a further xanthate adsorption.

2.5.2.2 Reversal of activation

Ionic strength and type of cations in solution can have an influence on the extent of activation. It is expected that the presence of divalent cations in solution like Ca^{2+} and Mg^{2+} compete with metal ions like Cu^{2+} on the mineral surfaces.

This phenomenon was also observed by Lascelles et al, (2001) when they investigated the effect of Ca^{2+} (500ppm) and Mg^{2+} (500 ppm) ions on the uptake of copper and xanthate on sphalerite and flotation of Cu-activated sphalerite as a function of pH. They showed that in the presence of divalent ions the flotation recovery was significantly reduced when pH exceeded the value at which the formation of the corresponding hydroxide takes place. The Scanning Electron Microscopy and X-ray Photoelectron Spectroscopy determined that the precipitates were present as patches on the surface. Monovalent ions did not seem to compete and lower recoveries were only obtained in the presence of divalent ions.

2.5.2.3 Zeta potential measurements

Work performed by Martinovic (2003) indicated that the presence of copper sulphate can activate both sulphide and gangue minerals unselectively. Zeta measurements for pyroxene showed that pyroxene was negatively charged throughout the pH range from 4 to 10. When copper sulphate was added, the zeta potential values became less negative and at pH 10 the zeta measurement was close to zero. The addition of SIBX only did not alter the zeta potential measurements of pyroxene and were the same as obtained when no reagents were added.

The introduction of xanthate ions in the presence of copper (II) ions shifted the zeta potential – pH curve to more negative values, compared with the curve for copper sulphate on its own, showing that xanthate adsorption on copper activated mineral surfaces was occurring.

Figure 2.11 shows the zeta potential measurements obtained in the absence and presence of copper sulphate for pyroxene and feldspar (Martinovic, 2003).

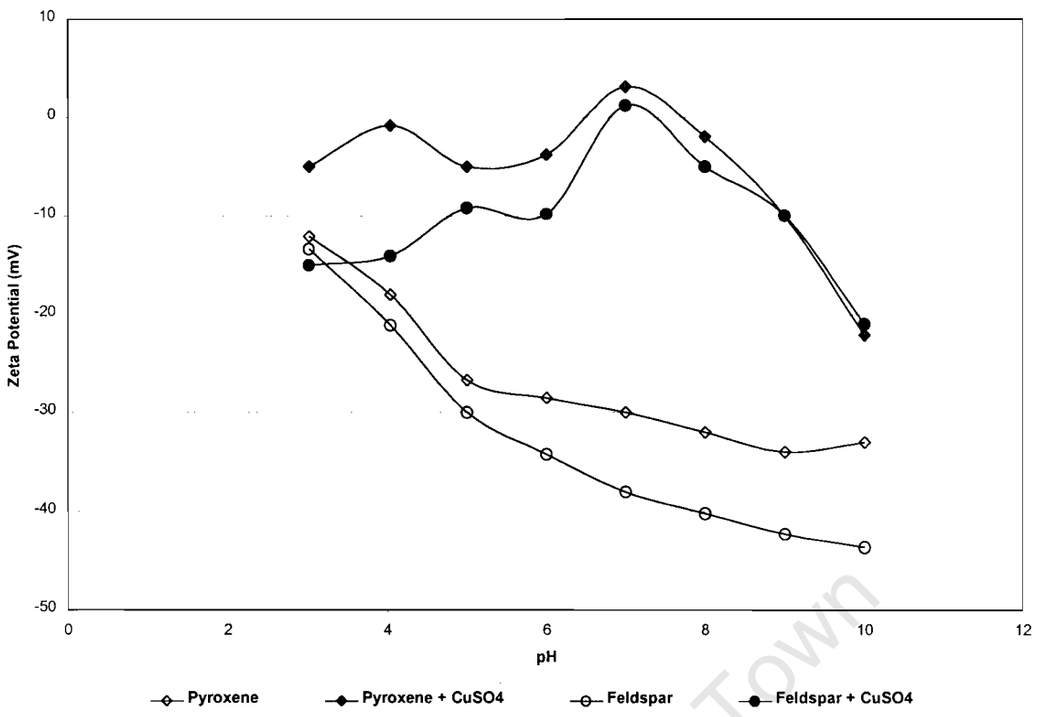

Figure 2.11: Zeta potential values for pyroxene and feldspar with copper sulphate [Martinovic, 2002]

2.6 Summary of Previous Work

Wesseldijk et al, (1999) performed microflotation tests using chromite. They investigated the effect of copper sulphate, pH and polymeric depressants on the microflotation behaviour of chromite. All the tests were performed at 10^{-3} KNO_3. It was observed that chromite was naturally hydrophilic and copper sulphate with the subsequent addition of collector resulted in enhanced flotation. Collectors alone did not have any effect on chromite behaviour. The addition of CMC at pH 4.5 resulted in slight reversal of activation shown by low recoveries. At pH 9 a slight increase in chromite recovery was observed compared to when no CMC

was added. IMP4 at pH 4.5 resulted in slight reversal in activation but complete reversal at pH 9 was obtained. No mechanisms were identified for the differences in behaviour of guars and CMCs.

Malysiak et al, (2001) performed microflotation tests on feldspar and pyroxene. Both were shown to be naturally hydrophilic over the pH range from 4 to 10. Addition of collector alone resulted in no enhancement in activation. The addition of copper sulphate and SIBX at pH 9 resulted in high recoveries of pyroxene and feldspar. No activation was observed at pH 4. It was also observed that the activation was reduced when synthetic water with a high ionic strength was used as compared to with borate solution used as a background electrolyte. The presence of divalent cations in synthetic water could be the reason for this behaviour.

Polysaccharides have been found to be effective depressants in flotation and in particular CMCs and guars have found widespread use in the processing of Platinum Group Minerals (PGM) ores in the Bushveld Complex in South Africa. The lower adsorption density of CMCs has been shown to be affected by water quality [Shortridge et al, 1999b]. Guars were found to adsorb more strongly and the adsorption had been shown to be independent of water quality [Steenberg and Harris, 1984; Shortridge et al, 1999b].

Parolis et al, (2003) have shown that metal cation assists in the adsorption of CMCs onto talc. The adsorption was shown to be more pronounced in the presence of divalent cations (i.e. Ca^{2+} and Mg^{2+}).

Thus, previous testwork provided evidence for inadvertent activation by copper sulphate of chromite, feldspar and pyroxene; the reversal of this effect by polymeric depressants has not been investigated.

2.7 Objectives of the Research

The objectives of this research were to firstly confirm that the selected gangue minerals at typical operating conditions could be inadvertently activated by copper sulphate as suggested by previous work; to evaluate whether activation by copper sulphate is mineral specific and to investigate the factors affecting activation reversal including the effect of water quality and depressant type and dosage.

The following key questions are addressed:

- Are pyroxene, feldspar and chromite naturally floatable?
- Can pyroxene, feldspar and chromite be activated by
  - Collector only?
  - Copper sulphate only?
  - Copper sulphate and collector?
- Does dosage of copper sulphate affect the extent of activation?
- Does the order of reagent addition affect activation of these minerals?
- Can the floatability of activated pyroxene and feldspar (as measured using microflotation tests) and chromite (as measured using batch flotation results) be reversed by the addition of depressants?
- Is the effect of CMC different to that of guar and is dosage a factor?
- Does the charge of CMC affect its ability to reverse activation?

- How does ionic strength and ion type affect activation and behaviour of depressant in reversing activation?

The behaviour of pyroxene and feldspar was investigated using microflotation techniques, as no analytical technique was available for evaluating concentrations of pyroxene and feldspar in the flotation concentrates. Wesseldijk et al, (1999) has investigated the microflotation of chromite and techniques are available for chromite analysis and thus, batch flotation tests were carried out to evaluate chromite behaviour.

CHAPTER 3: Experimental Details

3.1. Ore and mineral samples

3.1.1. Mineral preparation (Pyroxene and Feldspar)

The minerals used for the experiments were separated from the ore sample, which was obtained from Anglo Platinum Research Centre. The drill core sample was handpicked from Anglo Platinum's Amandelbult Section. About 30 kg of the original sample was reduced into smaller chunks using a five pound hammer before being crushed using a jaw crusher. The product was then cone crushed to – 3mm and then reduced further to the final product of +75 -150 microns by adjusting the cone crusher to give a finer product. The 20 kg product was then separated into 2 kg bags, which were magnetically separated using the Frantz Separator provided by Anglo Platinum Research Centre.

Prior to separation using the Frantz separator, magnetic minerals such as magnetite were removed from the sample using a hand magnet to prevent the magnetite from clogging the chutes in the Frantz Separator. The sample was then inspected under a microscope and it was observed that traces of chromite were present and had to be removed first since it had a higher magnetic susceptibility than feldspar and pyroxene. The Frantz separator had two big magnets, separation chute (chamber), transformer and a vibrator. The magnets were situated at the top and bottom of the chute and along the length of the chute. The sample was fed into a conical chamber which has a vibrator connected to it. The frequency of vibration was adjusted to control the feed rate into the separation chute.

The chute and the magnets can be tilted both longitudinally and sideways depending on the size, type and magnetic susceptibility of minerals to be

separated. The magnets induce a magnetic field in the chute so that magnetic and non-magnetic minerals could be separated as they moved along the chute and hence the chute split into two channels towards the end. Inducing current into the magnets when feebly magnetic minerals were to be separated can increase magnetic strength.

3.1.2. Magnetic susceptibility of chromite, feldspar and pyroxene

The following magnetic susceptibilities were obtained in the Geology Department at the University of Cape Town:

- Chromite – 0.35 - 0.45 amperes
- Pyroxene – 0.8 – 1.20 amperes
- Feldspar – 1.4 – 1.75 amperes

Thus from the readings it can be concluded that chromite was the most magnetic, followed by pyroxene and then feldspar. Chromite was separated first because of its high magnetic susceptibility and also it was present in trace amounts. Once the chromite was removed the current was increase to 0.8 A to separate pyroxene (magnetic) and feldspar (non-magnetic). The side tilt on the Frantz was kept at 15^0 and the front tilt at 25^0 as in the previous setting. Table 3.1 shows the mineral analysis obtained from ARC Mineralogy Department:

Table 3.1 – Purity of the minerals used for microflotation tests

Mineral type	Purity	Impurities/Contaminants
Pyroxene	90 – 95 %	Contains mainly chlorite (closely associated with pyroxene), feldspar and mica (mainly liberated)
Feldspar	85 – 90 %	Predominantly phlogopite (mica) and pyroxene. Both present as liberated minerals

All the mineral analysis estimates were based on X-ray diffraction and optical observation. It was agreed that the samples were pure enough to perform microflotation tests and zeta potential measurements.

The main disadvantage of most frothless flotation devices is their quite high mechanical carryover (entrainment) of small and medium-size particles, which often leads to misinterpretation of flotation data [Dryzmala et al, 1992]. It is for this reason that a reasonably coarse size fraction of the minerals was selected for microflotation purposes. In this project particle size range of +75 –106 microns was selected to be suitable for microflotation purposes.

3.1.3 Ore for Batch Flotation

A 60 kg UG-2 ore sample obtained from the Bushveld Complex with a PGE grade of 5 g/ton was crushed using a cone crusher. The ore was then homogeneously mixed and separated into 1kg samples using the splitter in the laboratory. The mineral composition of the ore used in batch flotation tests is detailed in table 3.2 below.

Table 3.2: Approximate mineral composition in the UG-2 ore

Mineral	% Composition
Chromite ($FeO.Cr_2O_3$)	46.6 (26.6 % Cr_2O_3)
Feldspar	24
Pyroxene	12
Talc	2
Chlorite	5
Phlogopite	3

3.2. Reagents and Dosages

The following reagents were used in the testwork performed in both microflotation and batch flotation tests:

- Collectors: sodium isobutyl xanthate (SIBX) and dithiophosphate (DTP – used in batch flotation only) supplied by Senmin
- Activator: Copper sulphate ($CuSO_4.5H_2O$)
- Depressants: carboxy methyl cellulose (CMC) and guar (see Table 3.3).

Microflotation experiments were based on a standard dosage of $5*10^{-5}$ M copper sulphate and $5*10^{-5}$ M SIBX. Some experiments were performed at $1*10^{-4}$ M copper sulphate and $1*10^{-4}$ M SIBX to observe whether activation was affected by the dosage and also to ascertain whether reversal of activation was affected by dosage. This ratio was chosen so that the copper ions in solution would always be in excess of xanthate ions based on the reaction stoichiometry:

$$Cu^{++} + 2\,X^{-} \Leftrightarrow CuX_2$$

Table 3.3 shows the depressants used and their characteristics. Three depressants were used in the microflotation tests, namely APX 4M (a modified guar), Dep 186 (low molecular weight, highly charged CMC) and Dep C (high molecular weight, low charged CMC). Two depressants, KU5 (CMC) and IMP 4 (Guar) were used in the batch flotation tests. The following characteristics of depressants were obtained from UCT Depressant Facility:

Table 3.3 Characterisation of depressants used in the testwork

Sample	pH 2% solution	Viscosity 2% solution (mPa.s)	Purity (wt %)	DS (CMC)
Depramin 186 (CMC)	7.9	11	68	0.86
Depramin C (CMC)	9	180	74	0.67
KU 5 (CMC)	10.5	3.85	75	0.8
IMP 4 (Guar)	6.4	11.2	95	n/a
APX 4M (Guar)	5.0	7.25	85	n/a

It is to be noted that depressant adsorption is physical and dosage was based on equivalent solution concentrations and adjusted for the % solids. In the microflotation tests, 2 g of mineral and 100 ml of solution was used (i.e. 2 % solids). In batch flotation tests 1 kg ore was used in 2.5 litres of water (i.e. 30 % solids). A depressant dosage of 40 ppm (mg/l) is equivalent to 100 g/ton in the 3 litre batch cell. It is important to note that the CMC dosages used were added based on their active content.

3.3. Flotation procedures and programmes

3.3.1. Microflotation tests

Microflotation tests were carried out with pyroxene and feldspar in the UCT flow-through microflotation cell.

The figure below shows a schematic diagram of the microflotation cell. This apparatus was developed at the University of Cape Town. It is a single mineral flotation device designed to study the mineral collection sub-process in the flotation system [Bradshaw and O'Connor, 1996]. In this system there is no froth zone and no frother is added. Thus, natural floatability of minerals and effect of the reagents and the mineral can be measured [Wesseldijk, 1999].

Procedure

The pulp circulated through the cell using a peristaltic pump. Air flowrate, adjusted with the needle valve, is measured with the bubble flowmeter and air pressure with a water manometer. Gas is introduced via the syringe and the bubbles rose through the cell and were deflected off the cone, which was situated at the top of the cell. The collected product reported to the launder. After a set time the syringe was removed and the flotation product collected, filtered, dried and weighed. The operating conditions were set such that the air flowrate was 10 cm^3/min and the peristaltic pump at 50 rpm (see Appendix A).

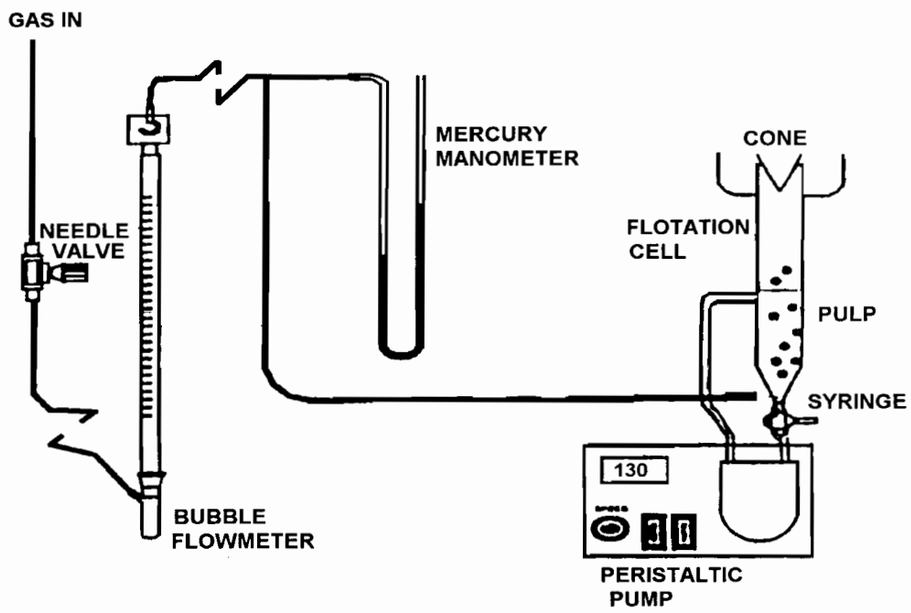

Figure 3.1 Schematic representation of the UCT flow-through microflotation cell

Experimental programme

Table 3.3 shows the experimental programme conducted to investigate the effects of collector (SIBX), activator (copper sulphate), cations (K and Ca), ionic strength and different depressant dosages. The experiments were mostly performed in duplicates except in cases of natural flotation of the minerals where tests were done in triplicate.

The microflotation results obtained with size fraction +75 –106 microns showed that recoveries of pyroxene and feldspar around 12 and 8 percent respectively were obtained when no reagents were added. Flotation time was 20 minutes and pH was 9 (Appendix C).

Table 3.3: Microflotation experimental programme

	$CuSO_4$	SIBX	Depressant Type	Depressant Dosage (ppm)	Electrolyte
Pyroxene	-	-	-	-	KNO_3
	-	$5*10^{-5}$	-	-	KNO_3
	$5*10^{-5}$	$5*10^{-5}$	-	-	KNO_3, $Ca(NO_3)_2$
	$5*10^{-5}$	$5*10^{-5}$	APX 4M	50, 100, 200	KNO_3, $Ca(NO_3)_2$
	$5*10^{-5}$	$5*10^{-5}$	Dep 186	50, 100, 200	KNO_3, $Ca(NO_3)_2$
	$5*10^{-5}$	$5*10^{-5}$	Dep C	50, 100, 200	KNO_3, $Ca(NO_3)_2$
Feldspar	-	-	-	-	KNO_3
	$5*10^{-5}$	$5*10^{-5}$	-	-	KNO_3
	$5*10^{-5}$	$5*10^{-5}$	APX 4M	50, 100, 200	KNO_3
	$5*10^{-5}$	$5*10^{-5}$	Dep 186	50, 100, 200	KNO_3
	$5*10^{-5}$	$5*10^{-5}$	-	-	$Ca(NO_3)_2$
	$5*10^{-5}$	$5*10^{-5}$	APX 4M	50, 100	$Ca(NO_3)_2$
	$5*10^{-5}$	$5*10^{-5}$	Dep 186	50, 100, 200	$Ca(NO_3)_2$
	$5*10^{-5}$	$5*10^{-5}$	Dep C	50, 100, 200	$Ca(NO_3)_2$

- $CuSO_4$ dosage = $5*10^{-5}$ M
- SIBX dosage = $5*10^{-5}$ M
- Ionic strength of KNO_3 and $Ca(NO_3)_2$ = 10^{-3} and 10^{-2}

3.3.2 Batch flotation tests

Apparatus

Milling was performed in a Sala laboratory mill. Batch flotation experiments were performed using UG-2 ore in a 3 litre Leeds batch flotation cell. The photograph below shows a Leeds batch cell.

Figure 3.2: Leeds batch flotation cell

Procedure

3.3.2.1 Milling

After separating the ore into 1kg samples, the first objective was to determine the suitable particle size for flotation and the time required for milling the ore. Three samples were milled for 5, 10 and 15 minutes respectively to plot a milling curve. After milling the samples were wet-sieved for particle size analysis and distribution. It has been recommended that a 40% passing 75 microns product would be suitable for flotation. This is a typical product of the most primary milling circuits used in UG-2 flotation in most plants in South Africa. The sieve sizes used were 103, 75 and 53 micron sieves. The milling curve was plotted using percentage passing 75-micron size as a function of time. The milling curve plot is shown in Appendix B of the thesis.

Experimental Programme

3.3.2.2. Batch Flotation experiments

The experiments were performed to observe the effect of several reagents and the effect of their dosages in the flotation of the UG-2 ore. All the experiments were performed at a pH of 9, which is the natural pH of the pulp. This is due to the buffering effect of the gangue minerals in the ore. All the tests were done in Cape Town tap water. In all the experiments frother, collectors and depressant were added. The guar gum derivative used was IMP4 and the carboxy methyl cellulose used was KU5. Standard collectors used were sodium-isobutyl xanthate (SIBX) and dithiophosphate (DTP). The collectors were added in a ratio of 1:1 (i.e. 30 g/ton each). A constant amount of collector and frother Dow 200 were added in every test. The reagents were added following a sequence and conditioning time described in Appendix D of this thesis. In the case where copper sulphate was not added, the conditioning time for it was not taken into

consideration. When the concentrates were collected, water was used to remove any particles left on the scraper and the mass of water added was recorded so as to determine the water recovered.

The parameters varied were the addition of copper sulphate and the dosages thereof and the depressant type and dosage. The flotation responses measured were mass recovery, chrome grade and recovery and water recovery. The chrome (Cr) assays were done to indicate the performance of chromite ($Cr_2O_3.FeO$).

Table 3.4: Summary of the batch flotation experimental work at pH 9

Collector dosage (g/ton)	$CuSO_4$ dosage (g/ton)	Depressant type + dosage (g/ton)
30	90	IMP4 – 100 (40 ppm)
30	0	IMP4 – 100 (40 ppm)
30	30	IMP4 – 100 (40 ppm)
30	90	IMP4 – 200 (80 ppm)
30	30	IMP4 – 200 (80 ppm)
30	90	KU5 – 100 (40 ppm)
30	30	KU5 – 100 (40 ppm)
30	90	KU5 – 200 (80 ppm)
30	30	KU5 – 200 (80 ppm)

In batch flotation tests the depressant dosages were measured as grams per tonne (g/ton) unlike in microflotation tests where the dosage was in parts per million (ppm). It is to be noted that in this case 100 g/ton is approximately equivalent to 40 ppm and 200 g/ton about 80 ppm.

At the end of each experiment, the concentrates, the feed sample and the tails were weighed and dried for further analysis.

The first three experiments were performed in triplicate and the rest in duplicate for reproducibility. For the first three experiments 5 concentrates were collected at 1, 3, 7, 13 and 23 minutes respectively. For the rest of the flotation tests four concentrates were taken at the end of 1, 3, 7 and 15 minutes respectively.

3.3.2.3. Atomic Absorption Analysis

The dried flotation samples were digested and analysed for their chrome (Cr) content using Atomic Absorption (AA). Before the AA was used, the samples were digested following the procedure outlined in Appendix G of the report. The summary of the results is shown in Appendix I of the report.

CHAPTER 4: RESULTS AND DISCUSSION

The behaviour of pyroxene and feldspar was investigated using microflotation techniques, as no suitable technique was available at UCT for evaluating concentrations of pyroxene and feldspar in the flotation concentrates. Wesseldijk (1999) investigated the microflotation of chromite and techniques are available for chromite analysis and thus, batch flotation tests were done to evaluate chromite behaviour. It is to be noted that the feldspar experimental programme was set on the outcome of pyroxene results and fewer tests were done.

4.1 Pyroxene and Feldspar (Microflotation Tests)

4.1.1 Reproducibility and statistical evaluation

Table 4.1 shows an example of reproducible results and details of all experiments can be found in appendix C Tests were done in duplicate or triplicate. Standard deviations are shown in tables 4.1, 4.2 and 4.3.

Table 4.1: Reproducibility of results

Run	Ionic Strength	pH	Total Mass (g)	Time (min)	Recovery (%)	Mean Recovery (%)	Standard Deviation
PXN 1	10^{-3} M KNO_3	9	1.88	5	1.7		
				10	6.2		
				20	11.9		
PXN 2	10^{-3} M KNO_3	9	1.84	5	1.4	1.56	0.19
				10	5.6	5.50	0.77
				20	12.4	11.99	0.42
PXN 3	10^{-3} M KNO_3	9	1.89	5	1.6		
				10	4.7		
				20	11.6		

4.1.2 Summary of the results

Table 4.2 and 4.3 show the summary of the microflotation results performed on pyroxene and feldspar respectively.

Table 4.2: Summary of the pyroxene results

Reagents	Ionic Strength	Mean Recovery (%)	Standard Deviation
none	10^{-3} KNO_3	12.0	0.46
Cu + SIBX	10^{-3} KNO_3	89.5	9.96
Cu + SIBX	10^{-3} $Ca(NO_3)_2$	74.6	5.53
Cu + SIBX	10^{-2} $Ca(NO_3)_2$	50.4	1.74
Cu + SIBX + APX 4 M (50 ppm)	10^{-3} KNO_3	79.8	3.14
Cu + SIBX + APX 4 M (50 ppm)	10^{-3} $Ca(NO_3)_2$	62.5	7.53
Cu + SIBX + APX 4 M (50 ppm)	10^{-2} $Ca(NO_3)_2$	7.3	0.75
Cu + SIBX + Dep 186 (50 ppm)	10^{-3} KNO_3	89.6	4.43
Cu + SIBX + Dep 186 (50 ppm)	10^{-3} $Ca(NO_3)_2$	75.7	0.33
Cu + SIBX + Dep 186 (50 ppm)	10^{-2} $Ca(NO_3)_2$	60.7	1.51
Cu + SIBX + Dep 186 (100 ppm)	10^{-2} $Ca(NO_3)_2$	66.7	1.23
Cu + SIBX + Dep 186 (200 ppm)	10^{-2} $Ca(NO_3)_2$	66.0	1.03
Cu + SIBX + APX 4 M (100 ppm)	10^{-3} KNO_3	76.6	2.17
Cu + SIBX + APX 4 M (100 ppm)	10^{-3} $Ca(NO_3)_2$	71.1	0.64
Cu + SIBX + APX 4 M (100 ppm)	10^{-2} KNO_3	77.6	1.45
Cu + SIBX	10^{-2} KNO_3	87.9	1.10
Cu + SIBX + APX 4 M (50 ppm)	10^{-2} KNO_3	83.5	0.83
Cu + SIBX + APX 4 M (200 ppm)	10^{-2} KNO_3	51.2	1.08
Cu + SIBX + Dep 186 (200 ppm)	10^{-2} KNO_3	82.3	1.18
Cu + SIBX (reverse order)	10^{-3} KNO_3	15.6	0.38
Cu + SIBX + APX 4 M (200 ppm)	10^{-3} KNO_3	72.7	1.79
Cu + SIBX + Dep 186 (200 ppm)	10^{-3} KNO_3	81.1	3.30
Cu + SIBX + Dep C (50 ppm)	10^{-2} $Ca(NO_3)_2$	55.9	2.74
SIBX	10^{-3} KNO_3	14.3	0.69

Cu + SIBX + Dep C (100 ppm)	10^{-2} $Ca(NO_3)_2$	51.4	3.79
Cu + SIBX + Dep C (200 ppm)	10^{-2} $Ca(NO_3)_2$	36.2	0.75

Table 4.3: Summary of the feldspar results

Reagents	Ionic Strength	Mean Recovery (%)	Standard Deviation
None	10^{-3} KNO_3	9.5	0.41
Cu + SIBX	10^{-3} KNO_3	89.1	1.96
Cu + SIBX (reverse order)	10^{-3} KNO_3	8.4	0.46
Cu + SIBX + APX 4 M (50 ppm)	10^{-3} KNO_3	83.5	0.31
Cu + SIBX + APX 4 M (100 ppm)	10^{-3} KNO_3	80.3	1.59
Cu + SIBX + APX 4 M (200 ppm)	10^{-3} KNO_3	74.1	2.28
Cu + SIBX + Dep 186 (50 ppm)	10^{-3} KNO_3	84.2	1.46
Cu + SIBX + Dep 186 (100 ppm)	10^{-3} KNO_3	81.5	1.59
Cu + SIBX + Dep 186 (200 ppm)	10^{-3} KNO_3	77.2	2.11
Cu + SIBX	10^{-2} $Ca(NO_3)_2$	53.3	1.94
Cu + SIBX + APX 4 M (50 ppm)	10^{-2} $Ca(NO_3)_2$	5.2	0.21
Cu + SIBX + APX 4 M (100 ppm)	10^{-2} $Ca(NO_3)_2$	5.0	0.30
Cu + SIBX + Dep 186 (50 ppm)	10^{-2} $Ca(NO_3)_2$	48.9	1.81
Cu + SIBX + Dep 186 (100 ppm)	10^{-2} $Ca(NO_3)_2$	49.1	0.95
Cu + SIBX + Dep 186 (200 ppm)	10^{-2} $Ca(NO_3)_2$	44.9	2.12
Cu + SIBX + Dep C (50 ppm)	10^{-2} $Ca(NO_3)_2$	54.5	1.42
Cu + SIBX + Dep C (200 ppm)	10^{-2} $Ca(NO_3)_2$	29.8	0.99

4.1.3 Natural floatability of pyroxene and feldspar

To test the natural floatability of pyroxene and feldspar, microflotation tests were carried out in the presence of KNO_3 with no other reagents added. Figure 4.1 shows the results obtained with pyroxene and feldspar under 10^{-3} KNO_3

electrolyte ionic strength. Both experiments were carried out for 20 minutes to give the samples ample time to float.

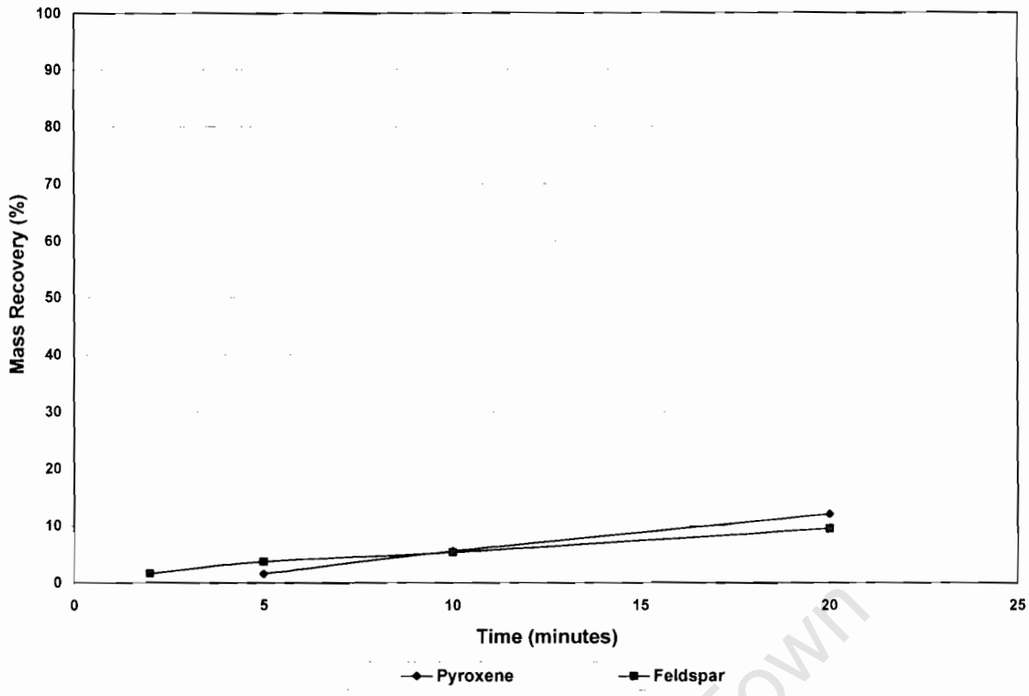

Figure 4.1: Natural floatability of pyroxene and feldspar under 10^{-3} KNO_3 ionic strength

As figure 4.1 suggests, it can be seen that pyroxene is not naturally floatable (mass recovery of about 12.0 %). The same trend was observed for feldspar (mass recovery of about 9.5 %). The results are average values for three tests each with standard deviations of 0.46 and 0.41 respectively.

4.1.4 Effect of Collector

In order to observe the effect of adding collector only, on the recovery of pyroxene and feldspar, sodium iso-butyl xanthate (SIBX) was added at $5*10^{-5}$ M in the presence of 10^{-3} KNO_3 ionic strength. Figure 4.2 shows the results of this experiment compared to the test where no xanthate was added.

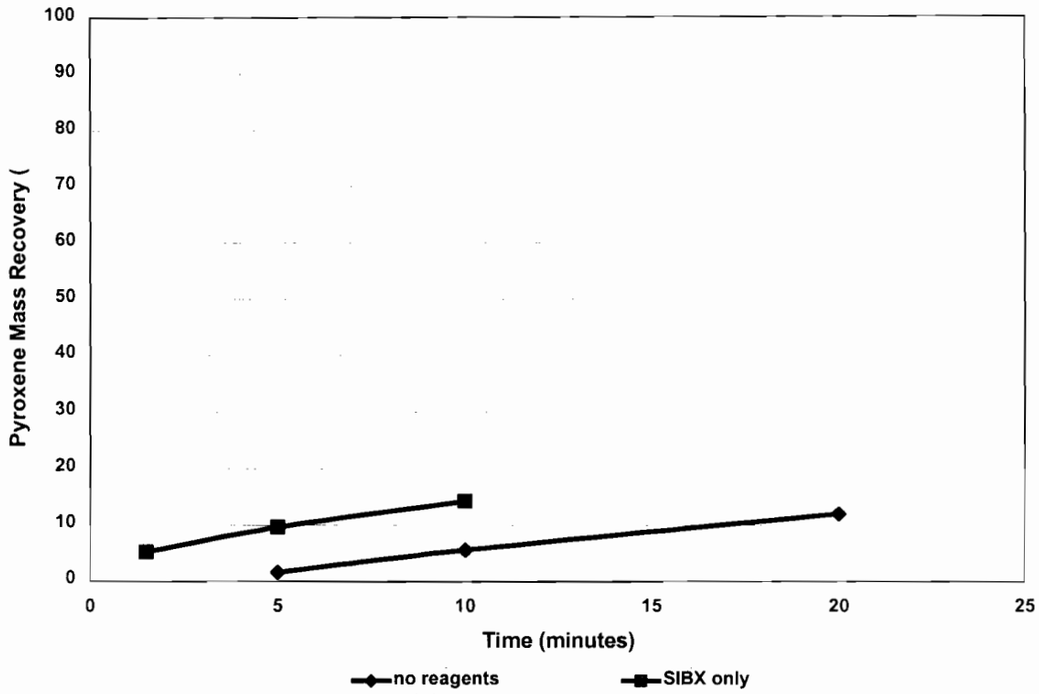

Figure 4.2: The effect of adding $5*10^{-5}$ M SIBX on the flotation response of pyroxene in the presence of 10^{-3} KNO_3

The results show that adding the collector slightly enhanced the overall flotation recovery of pyroxene and the rate of flotation was increased although final recoveries were low. The final mass recovery when collector was added was only about 14 % after 10 minutes. This result showed that the addition of collector only at pH 9 did not absorb onto the mineral surface or impart a significant hydrophobic nature onto the mineral. This was also shown by Malysiak et al, (2003), and indicates that xanthate does not absorb onto silicate minerals. Similar work was also carried out by Wesseldijk, (1999) on chromite microflotation and reported that the addition of collector only did not enhance chromite flotation.

4.1.5. Effect of copper sulphate and collector

In order to determine whether activation by copper sulphate occurred, experiments were carried out using copper sulphate and collector at pH 9 and pH 4. At pH 9 two dosages were investigated to determine the effect of reagent dosages on the flotation response of pyroxene and feldspar (i.e. $5*10^{-5}$ M copper sulphate with $5*10^{-5}$ M SIBX and $1*10^{-4}$ M copper sulphate with $1*10^{-4}$ M SIBX). The results of these tests are shown in figures 4.3, 4.4, 4.5 and 4.6.

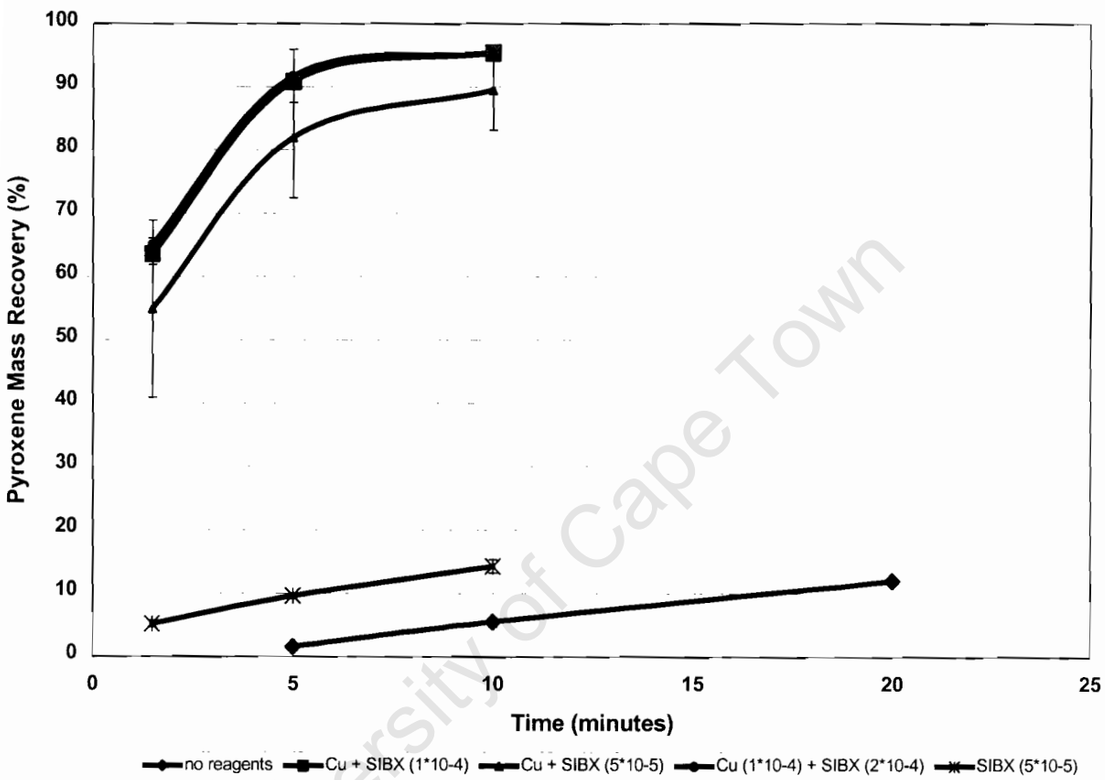

Figure 4.3: Effect of varying copper sulphate dosage and collector at pH 9 on pyroxene recovery in the presence of 10^{-3} KNO_3

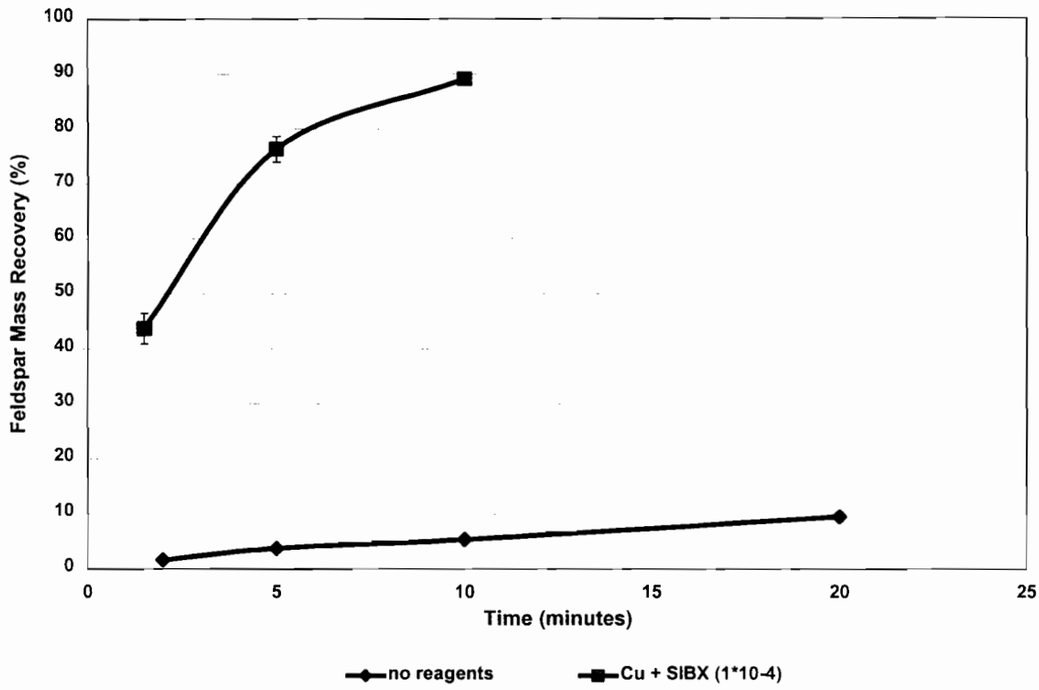

Figure 4.4: The effect of copper sulphate and collector at pH on the recovery of feldspar in the presence of 10^{-3} KNO_3

Both figures 4.3 and 4.4 show that the addition of copper sulphate before collector greatly enhanced the recovery of both pyroxene and feldspar at pH 9 (greater than 85 % in both cases). Furthermore, experiments on pyroxene where the copper sulphate and collector dosage were increased showed that pyroxene recovery was not affected by dosage even when stoichiometric amounts of copper sulphate and collector were added. This can be attributed to the copper species present at this pH. Figure 4.15 shows the logarithmic concentration diagram for $1*10^{-4}$ mol/l Cu^{++} obtained from Wesseldijk et al, (1999). At pH 9 $Cu(OH)_2$ is the only species present in solution and mechanisms of this species precipitating onto the mineral surface have been suggested [Wesseldijk et al, 1999]. This indicates that $Cu(OH)_2$ precipitates on the surface by adsorption e.g. via hydrogen bonding with some MOH^+ (M = metal atom from mineral surface) that could be present on the mineral surface and/or that the freshly formed $Cu(OH)_2$ in solution might adsorb some of the collectors [Harris, 1997].

4.1.6. Effect of pH on activation

Figure 4.5 shows the effect of pH in the presence of copper sulphate and collector. The results suggest that no activation of silicate minerals was possible at pH 4 since Cu^{2+} ions cannot directly absorb onto the mineral surface unlike copper species present at pH 9.

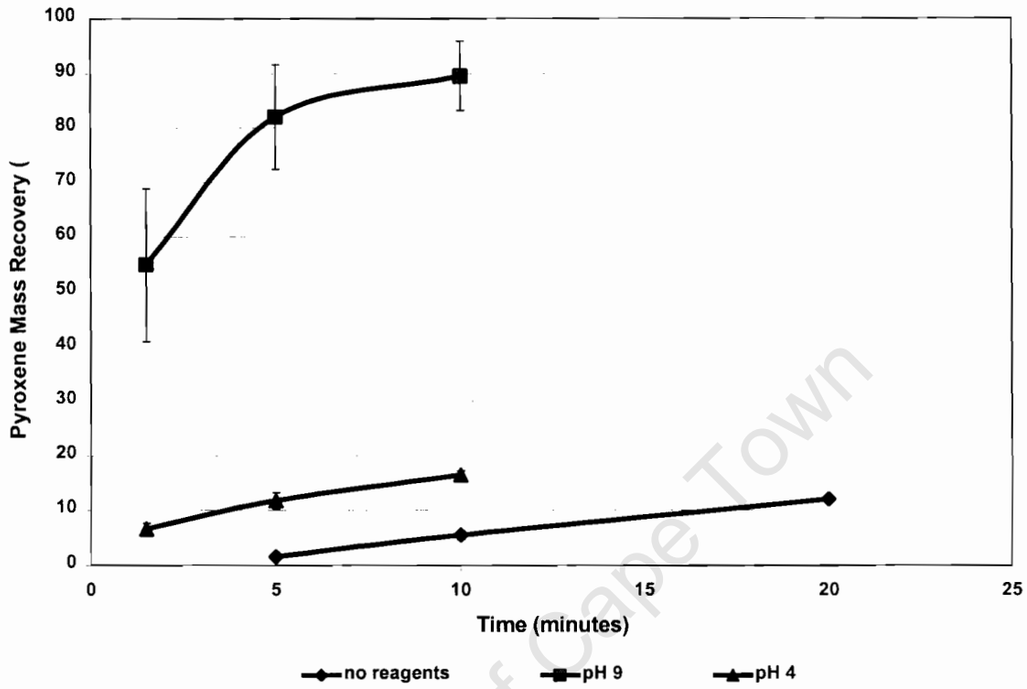

Figure 4.5: The effect of pH on the recovery of pyroxene under $5*10^{-5}$ M $CuSO_4$ and $5*10^{-5}$ M SIBX in the presence of 10^{-3} KNO_3

The difference between the activation at pH 4 and pH 9 is quite significant. This showed that there was less activation of pyroxene at pH 4 and that activation at pH 9 caused by the addition of copper sulphate and collector enhanced significantly pyroxene recovery.

4.1.7. Effect of type of metal ion and ionic strength

Ionic strength and the type of metal ion used can adversely affect the extent of activation of minerals by copper sulphate. Experiments were performed with two types of electrolytes (i.e. KNO_3 and $Ca(NO_3)_2$. The aim was to evaluate the effect of monovalent and divalent metal ions present in the electrolyte and also their ionic strength.

4.1.7.1. Effect of KNO_3

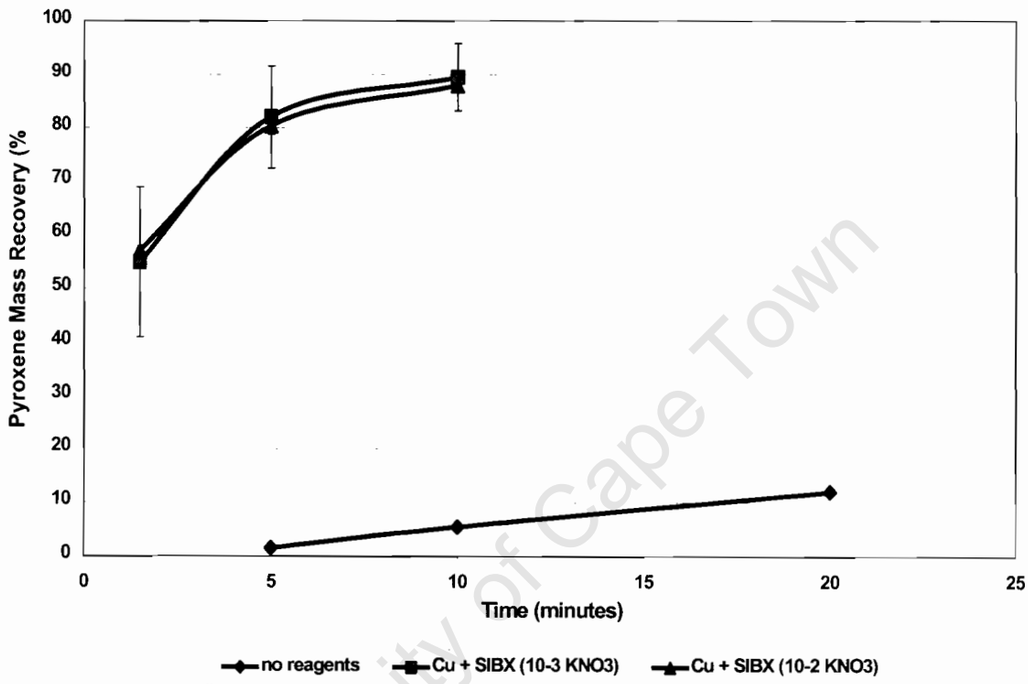

Figure 4.6: Effect of varying KNO_3 strength at $5*10^{-5}$ M $CuSO_4$ and $5*10^{-5}$ M SIBX on the recovery of Pyroxene at pH 9

Figure 4.6 shows the effect of ionic strength of KNO_3 on the recovery of pyroxene in the presence of copper sulphate and SIBX at $5*10^{-5}$ mol/l dosages at pH 9. The results show that increasing the ionic strength of KNO_3 from 10^{-3} to 10^{-2} has

no significant effect on the reduction of the pyroxene recovery. Thus activation is still strong and the error bars of both are overlapping. It can be concluded from these results that ionic strength of KNO_3 does not interfere or compete with the activation of pyroxene. It would seem that the potassium species at this pH would not adsorb onto the pyroxene surface, as would the $CuOH^+$ or $Cu(OH)_2$ species. The reason can be attributed to the fact that potassium is a monovalent cation and copper is a divalent cation and thus it can form a link between the mineral surface and the collector. This exercise was limited to pyroxene as feldspar experiments were already carried out by Malysiak et al, (2003).

4.1.7.2. Effect of $Ca(NO_3)_2$ compared to KNO_3

Figure 4.7 shows the effect of ionic strength of $Ca(NO_3)_2$ on the recovery of pyroxene in the presence of copper sulphate and SIBX each added at $5*10^{-5}$ mol/l. The results show that the presence of $Ca(NO_3)_2$ in solution reduced the extent of activation. The reduction observed at an ionic strength of 10^{-3} $Ca(NO_3)_2$ ($3.33*10^{-4}$ M) was further reduced at 10^{-2} $Ca(NO_3)_2$ ($3.33*10^{-3}$ M). The same reduction is observed with 10^{-2} $Ca(NO_3)_2$ with feldspar in the presence of $1*10^{-4}$ M copper sulphate and SIBX in figure 4.8. It is to be noted that changing the metal ion from K^+ to Ca^{2+} the effects observed in flotation properties will not be only based on ionic strength but also the metal ion itself since these two metal ions have different properties. K^+ is a monovalent cation and Ca^{2+} is a divalent cation. Thus the reduction in activation when $Ca(NO_3)_2$ was added was mainly due to the change in metal ion properties and the ionic strength.

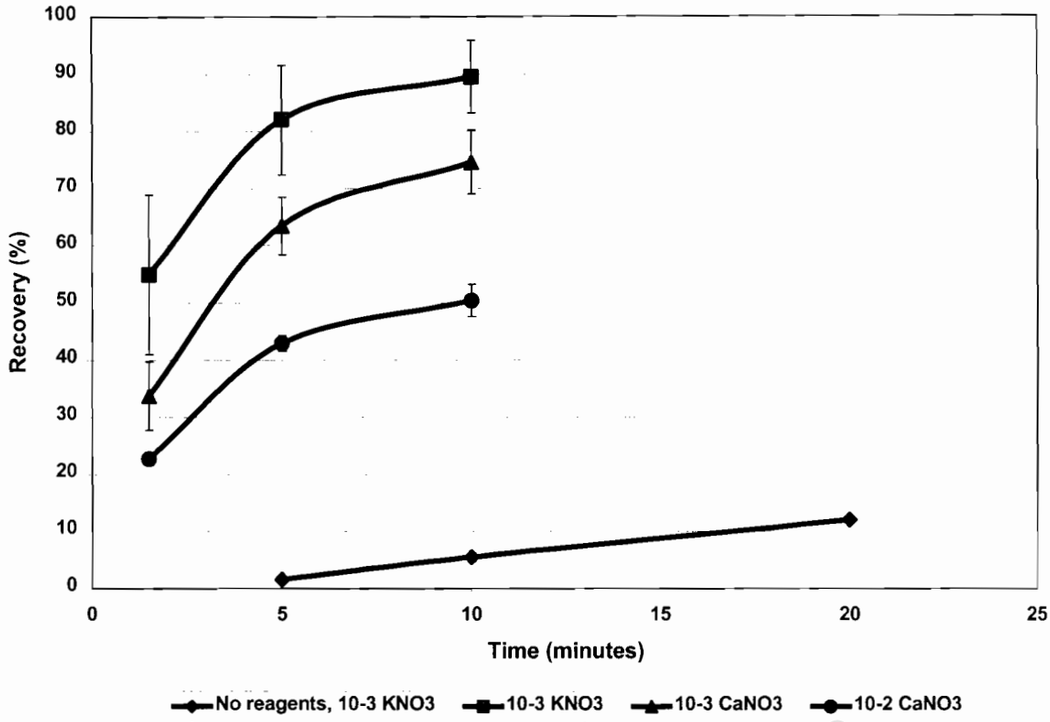

Figure 4.7: The effect of ionic strength and change in metal ion on the recovery of pyroxene in the presence of $5*10^{-5}$ M copper sulphate and $5*10^{-5}$ M SIBX at pH 9

Figures 4.7 and 4.8 clearly show that $Ca(NO_3)_2$ has reduced the extent of activation on both pyroxene and feldspar which shows that this effect is not mineral specific.

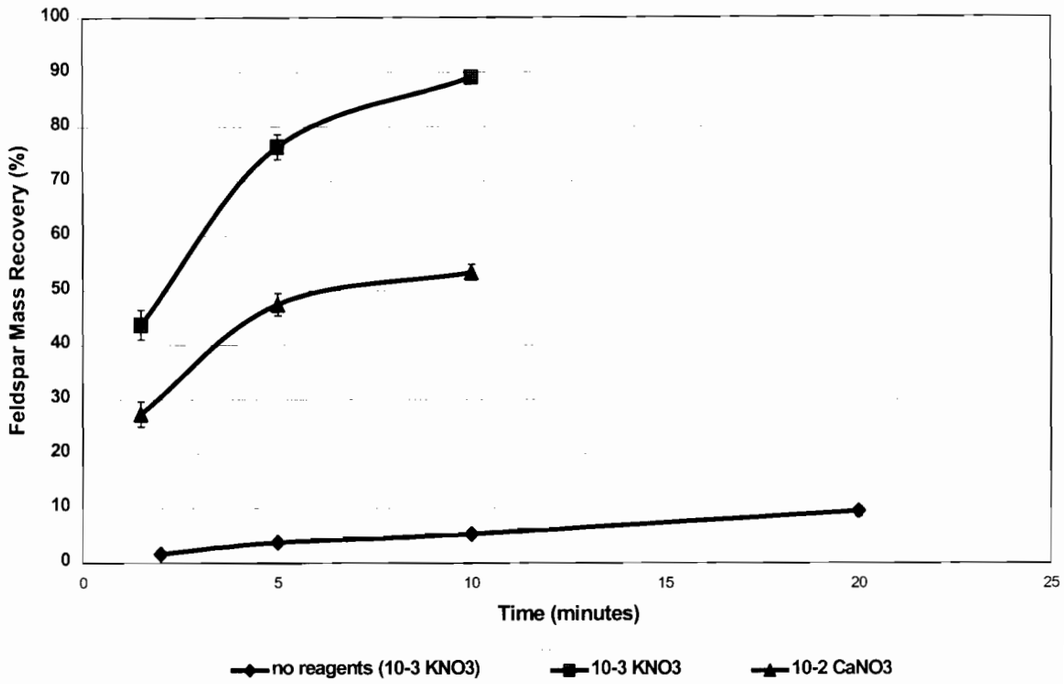

Figure 4.8: The effect of ionic strength and change in metal ion $Ca(NO_3)$ on the recovery of feldspar in the presence of $1*10^{-4}$ M copper sulphate and $1*10^{-4}$ M SIBX at pH 9

4.1.8 Effect of depressants on the recovery of pyroxene and feldspar

Figures 4.3 and 4.4 have shown that significant recoveries of pyroxene and feldspar were obtained in the presence of copper sulphate and collector at pH 9. Since the improved recovery of these minerals is undesirable for operation in a plant, the effectiveness of depressants to reduce this activation has been investigated under various ionic conditions. As discussed in chapter 3, two types of polymers were chosen (i.e. guar gum and CMC). One guar derivative was chosen (i.e. APX 4M) and two CMC's were used (i.e. Depramin 186 and Depramin C). Depramin 186 is a low molecular weight CMC with a high degree of substitution. This means that it is highly charged whereas Depramin C is a

high molecular weight CMC with a lower degree of substitution. The dosage of the polymers varied from 50 to 200 ppm. The minimum of 50 ppm was chosen as the activation of these minerals was very strong and lower dosages may not show any significant reversal of activation.

4.1.8.1 The effect of guar (APX 4M) on pyroxene recovery various metal ions and ionic conditions

High recoveries of pyroxene were observed in the presence of KNO_3 and lower recoveries in the presence of $Ca(NO_3)_2$. The change in the metal ion plays a significant role in the flotation response in conjunction with the ionic strength used. The effect of guar was investigated under both these conditions. Figure 4.9 illustrates the pyroxene recovery in the presence of APX 4M under various ionic conditions. All these results were performed at pH 9 since high recoveries were observed in the presence of copper sulphate and SIBX. It is known that guars are relatively uncharged polymers and adsorb readily [Steenberg and Harris, 1984]. However, the type and strength of the electrolyte may play a significant role on how these polymers adsorb onto the mineral surface.

The results in figure 4.9 show that under both KNO_3 concentrations, no significant reversal of copper activation of pyroxene occurred at dosages less than 100 mg/l APX 4M. This suggests that activation is still strong and the addition of guar did not have much impact on the reversal of activation. At 200 ppm guar in 10^{-2} KNO_3 ionic strength recovery of pyroxene was reduced to about 50%.

The addition of $Ca(NO_3)_2$ alone reduced the recovery of pyroxene from 90% to 75% thus changing the metal cation affected flotation recovery and at the higher ionic strength of 10^{-2}, reduced to 50%. When the ionic strength was increased to 10^{-2} the addition of 50 ppm guar was sufficient to completely reverse activation and recovery of less than 10% was achieved. However at the lower ionic strength

of 10^{-3} $Ca(NO_3)_2$ no significant reduction in recovery was observed with guar addition.

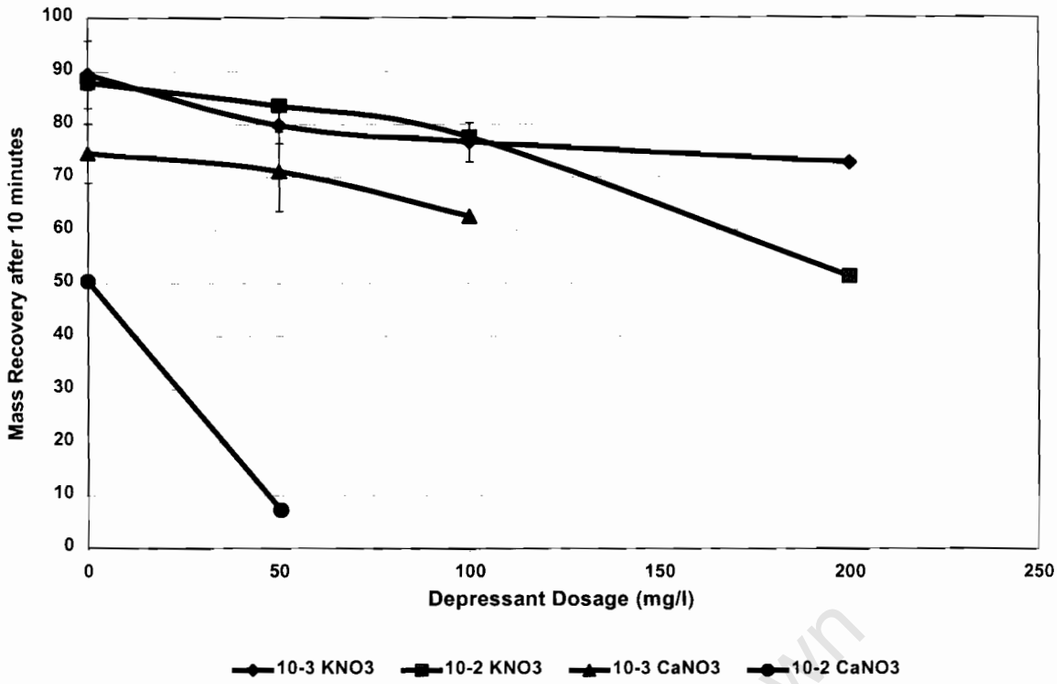

Figure 4.9: The effect of guar (APX 4M) on the recovery of pyroxene under various ionic conditions in the presence of $5*10^{-5}$ M copper sulphate and $5*10^{-5}$ SIBX

4.1.8.2 The effect of guar (APX 4M) on the recovery of feldspar under various ionic conditions and metal ions

Based on the experiments performed on pyroxene, several conditions were selected for evaluation of feldspar. Two ionic conditions of ionic strength were investigated i.e. 10^{-3} KNO_3 and 10^{-2} $Ca(NO_3)_2$. These conditions were selected as 10^{-3} KNO_3 is a standard condition and 10^{-2} $Ca(NO_3)_2$ showed significant reduction in copper activation.

Figure 4.10 shows the feldspar results at the various ionic conditions in the presence of guar (APX 4M). As observed with the pyroxene results, the feldspar results showed that the presence of guar using 10^{-3} KNO_3 ionic strength was not effective in the reversal of activation and hence high feldspar recoveries were obtained (greater than 70 %) under all dosages used in this scope of work.

However, at an ionic strength of 10^{-2} $Ca(NO_3)_2$, the guar completely reverses activation even at low dosages (50 ppm). The same observation was seen with the pyroxene results.

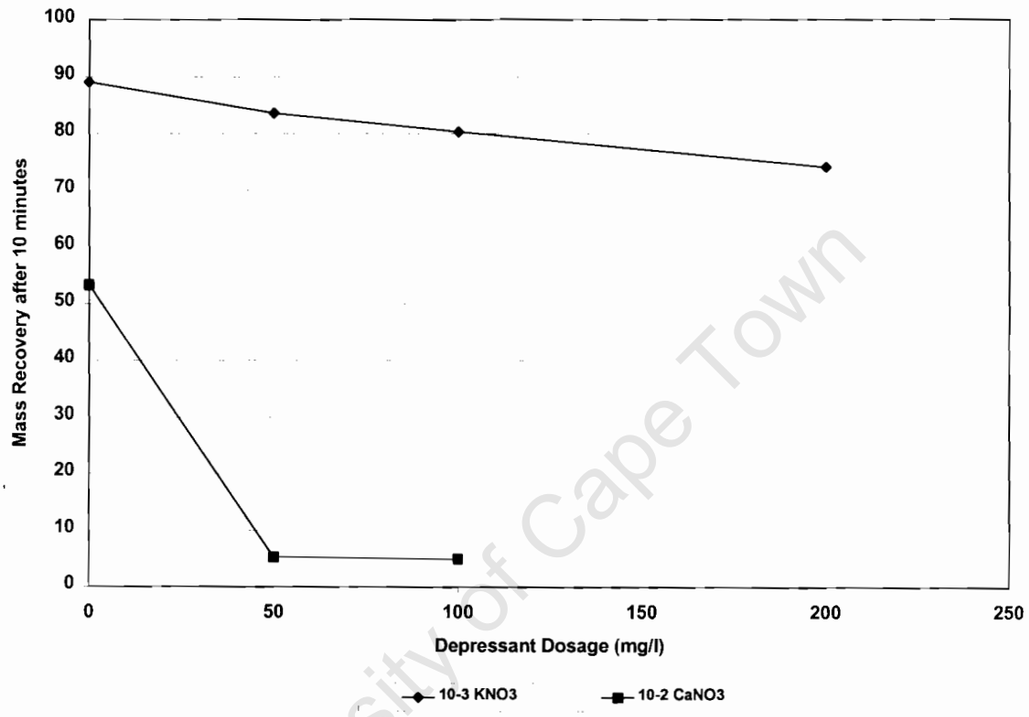

Figure 4.10: The effect of guar (APX 4M) on the recovery of feldspar under various ionic conditions in the presence of $1*10^{-4}$ M copper sulphate and $1*10^{-4}$ SIBX

The feldspar experiments were performed at a higher copper sulphate and SIBX dosages (i.e. $1*10^{-4}$ mol/l). The results show that the guar is successful in

reversing the activation irrespective of the dosage of copper sulphate and SIBX used.

4.1.8.3 The effect of CMCs on the recovery of pyroxene under various ionic conditions and metal ions

Two CMC's were selected for the investigation and in comparison with the guar. Depramin 186 was tested under the same conditions as APX 4M and Depramin C was only used to compare its effectiveness relative to Depramin 186. Figure 4.12 illustrates the results obtained with Dep186 under various ionic conditions.

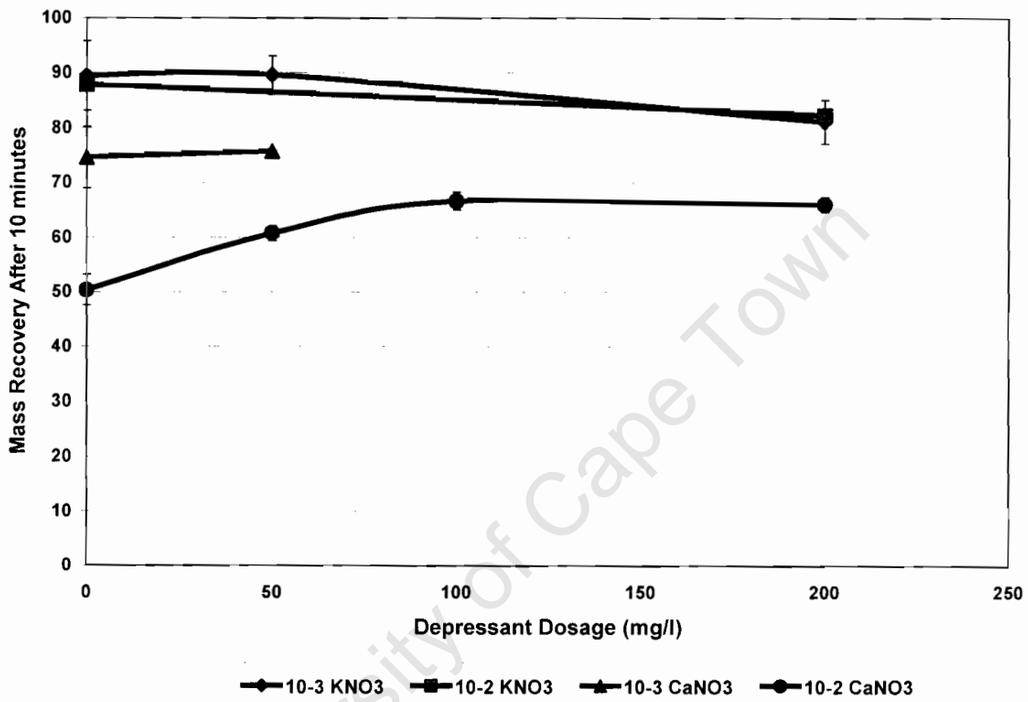

Figure 4.11: The effect of CMC (Depramin 186) on the recovery of pyroxene under various ionic conditions in the presence of $5*10^{-5}$ M copper sulphate and $5*10^{-5}$ SIBX

As mentioned in chapter three of this report, the addition of CMC's was based on their active content. The results suggest that Depramin 186 is not an effective

depressant under all the ionic conditions investigated. In some instances a slight increase in pyroxene recovery was observed. The reason for Depramin 186 being such an ineffective depressant may lie with the fact that it is a highly charged CMC and that adsorption onto the mineral surface is not achieved.

Figure 4.12 compares the behaviour obtained with guars and CMCs. The results suggest that Depramin C is a better depressant than Depramin 186. This may be based on the fact that Depramin C is not as highly charged as Depramin 186 and may adsorb onto the mineral surface and hence reduce activation by copper. However, Depramin C is not as effective as the guar since recoveries of pyroxene are still over 30%. The only condition that was investigated for Depramin C was at 10^{-2} $Ca(NO_3)_2$ ionic strength.

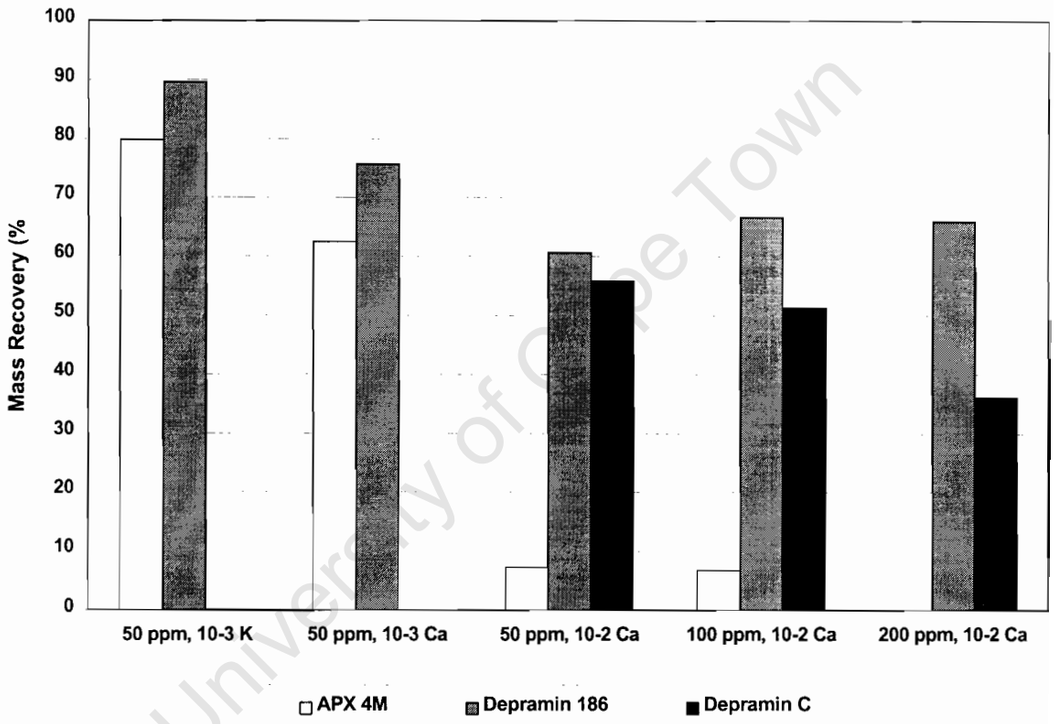

Figure 4.12: The effect of Depramin 186 and Depramin C on the recovery of pyroxene under various conditions in the presence of $5*10^{-5}$ M $CuSO_4$ and $5*10^{-5}$ M SIBX

4.1.9. The effect of order of reagent addition on the activation of pyroxene and feldspar

In the standard experiments, copper sulphate was added before the SIBX. From this procedure it was shown that activation of hydrophilic minerals was possible at pH 9. However, a few experiments were performed where SIBX was added before the copper sulphate to observe if the same activation would be obtained. Figure 4.13 shows the results of this procedure.

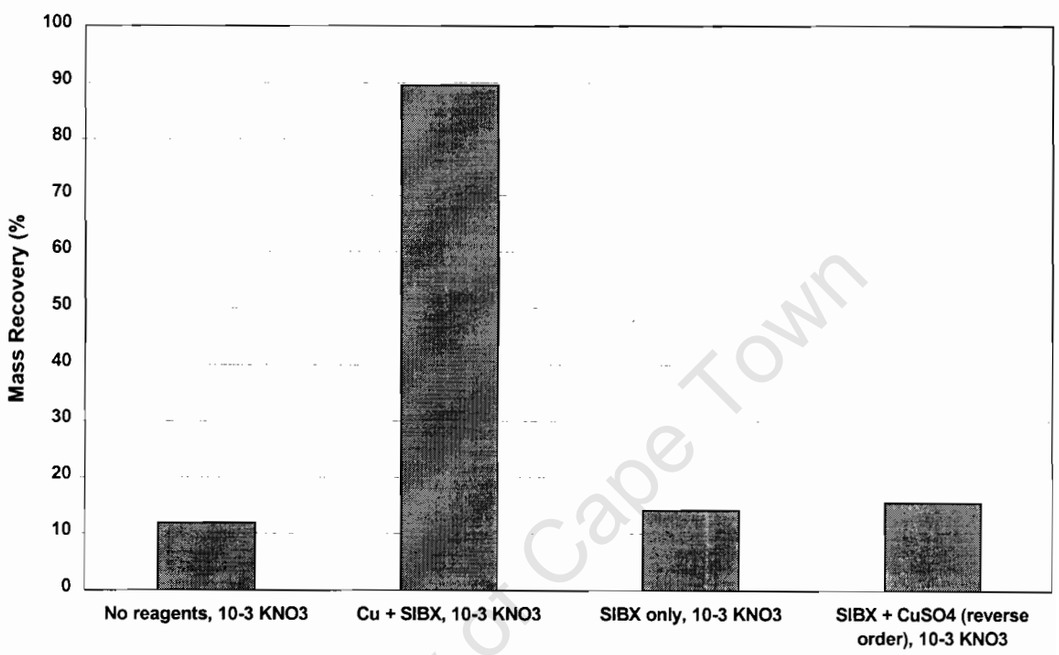

Figure 4.13: The effect of order of reagent addition on the activation of pyroxene under 10^{-3} KNO_3 ionic strength

It can clearly be seen that order of addition was very important in activating mineral surfaces. When SIBX was added before copper sulphate, no activation was observed. The recovery was almost the same as that obtained when no reagents are added or when only SIBX was added. This shows that copper

sulphate did not adsorb onto the mineral surface after SIBX has been added. Another possibility is that they react in solution and not on the surface and do not impart hydrophobicity onto the mineral. The observation that SIBX on its own does not enhance flotation of pyroxene suggests that neither SIBX nor a SIBX-copper complex adsorb onto the mineral surface.

Figure 4.14 shows the results obtained with feldspar. This demonstrates that the mechanisms of activation are similar for both minerals.

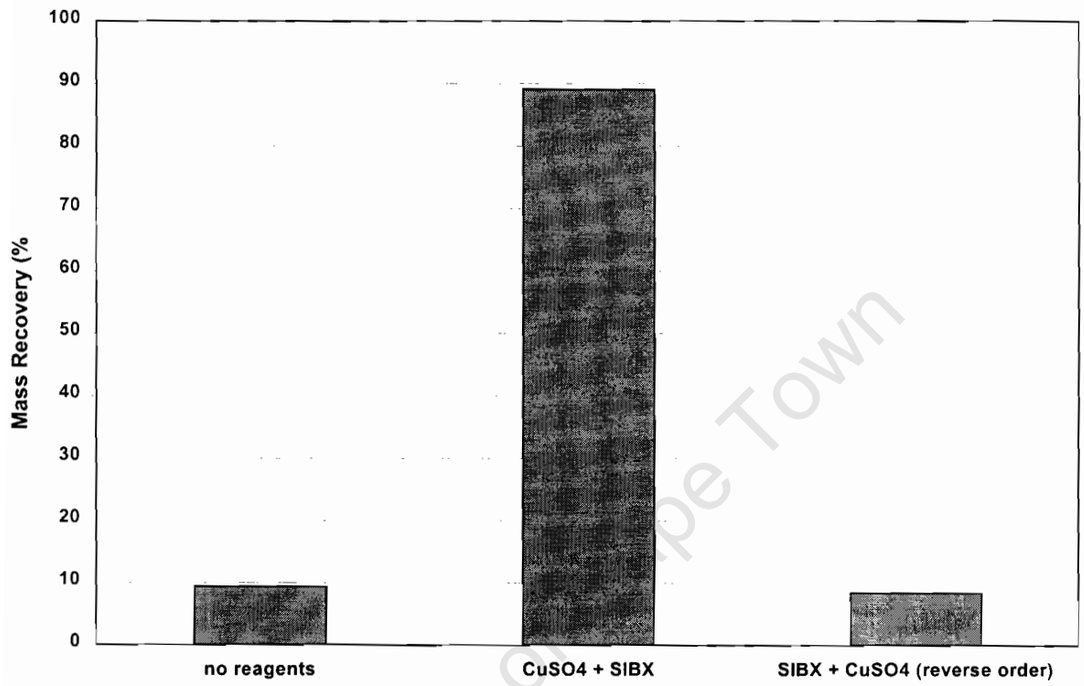

Figure 4.14: The effect of order of reagent addition on the activation of feldspar under 10^{-3} KNO_3 ionic strength

Zeta potential measurements performed by Shackleton, (2003) show that the adsorption of copper species at high pH values is evident and the zeta shifted to a less negative value when copper sulphate was added.

4.2. Chromite (Batch Tests)

The microflotation test results obtained with pyroxene and feldspar and those by Wesseldijk, (1999) showed that activation by copper sulphate was not mineral specific. Batch flotation tests were done with UG-2 ore to investigate the effect of copper sulphate dosage and depressant type and dosage on chromite recovery. The chrome (Cr) assays were performed to indicate the performance of chromite (Cr_2O_3.FeO).

4.2.1. Reproducibility of the results

Flotation experiments were performed either in triplicate or duplicate and the table below shows the summary of the results of the tests and the associated standard deviations.

Table 4.4: Summary of experimental results.

Reagent Added	Water Rec. (g)	Chrome Grade (%)	Chrome Rec. (%)	Mass Recovery (g)	Standard deviation
90 g/ton $CuSO_4$ + 100 g/ton IMP4	179.8	2.42	0.28	15.96	0.03
100 g/ton IMP4	261.4	2.22	0.24	15.45	0.04
30 g/ton $CuSO_4$ + 100 g/ton IMP4	279.0	2.98	0.26	18.18	0.08
90 g/ton $CuSO_4$ + 200 g/ton IMP4	228.4	4.54	0.21	10.61	0.03
30 g/ton $CuSO_4$ + 200 g/ton IMP4	279.3	4.54	0.23	10.08	0.02
90 g/ton $CuSO_4$ + 100 g/ton KU5	216.4	3.08	0.39	25.20	0.07
30 g/ton $CuSO_4$ + 100 g/ton KU5	275.8	2.81	0.33	24.40	0.03
90 g/ton $CuSO_4$ + 200 g/ton KU5	182.0	3.57	0.19	12.45	0.12
30 g/ton $CuSO_4$ + 200 g/ton KU5	242.6	4.12	0.20	10.95	0.03

- Frother (Dow 200 @ 30 g/ton)
- Collectors (SIBX and DTP @ 30 g/ton each)
- pH = 8.7 (natural)

4.2.2 Effect of copper sulphate dosage on chromite recovery

Three dosages of copper sulphate were used to determine the effect of varying copper sulphate dosage i.e. 0 g/ton, 30 g/ton and 90 g/ton with constant depressant dosage of 100 g/ton IMP4 or KU5. The 30-g/ton dosage is the stoichiometric requirement to react with thiol collectors used in UG-2 flotation. The 90-g/ton dosage was selected as typical of that used in industry. The results were represented in terms of chrome (Cr) recovery plotted against water recovery and are shown figure 4.15 and 4.16.

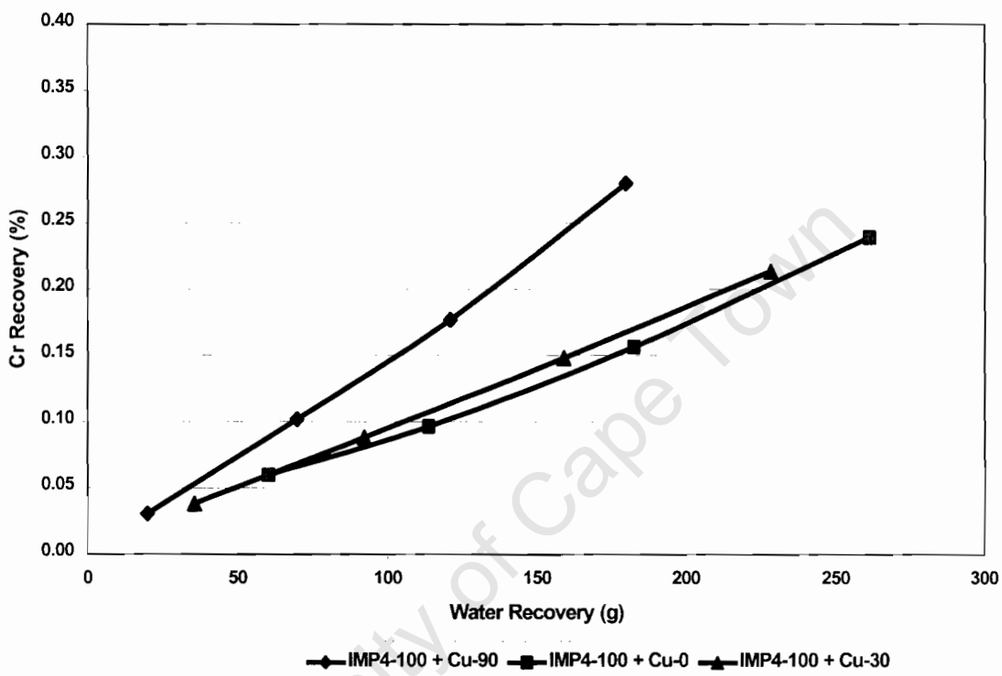

Figure 4.15: Effect of copper sulphate dosage on chrome recovery at 100 g/ton guar (IMP4)

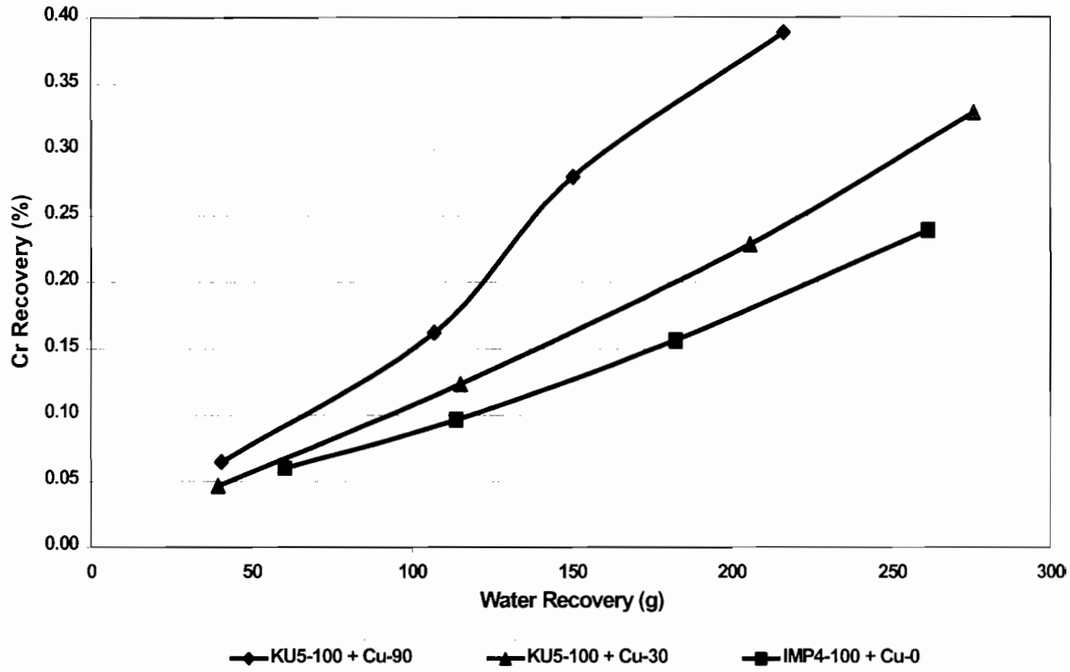

Figure 4.16: The effect of copper sulphate dosage on chrome recovery with the addition of 100 g/ton CMC (KU5) and guar (IMP4) as depressants

In the absence of copper sulphate, the chrome recovery can be attributed to entrainment. Increasing the copper sulphate dosage to 30 g/ton resulted in no significant increase in chrome recovered compared to that when no copper sulphate was added. The confidence level was below 90%. When the copper sulphate dosage was increased to 90 g/ton the enhancement in chrome recovery was observed and statistically was 99% significant. This suggests that some chromite was recovered by true flotation. This is based on the testwork by Wesseldijk et al, (1999) who observed activation of coarse chromite in microflotation work (i.e. +75 microns). It has been shown that the copper species present at pH 9 can activate the chrome and subsequent addition of collector will render the activated chromite hydrophobic and amenable to true flotation. It was observed that neither IMP4 nor KU5 at 100 g/ton reduced the recovery of chrome demonstrating that the activation was reversed.

4.2.3 The effect of depressant type and dosage

To determine the effectiveness in reversal of activation by depressants IMP4 (guar) and KU5 (CMC), experiments were carried out with these reagents at different dosages (100 and 200 g/ton). During all the tests copper sulphate was kept constant at (90g/ton). Other reagents were also kept constant at their standard dosages. Figure 4.17 shows the results from the experiments performed to determine the effect of changing the IMP4 dosages on chrome recovery. In this graph it can be clearly seen that increasing the IMP4 dosage decreases the chrome recovery. The chrome recovery at increased IMP4 dosage was not significantly different from the one obtained at zero addition of copper sulphate as validated by the statistical analysis (F – test). It can be concluded that the addition of IMP4 was successful in depressing the copper activated chromite as shown by chrome recoveries.

Figure 4.18 shows the results from the experiments performed to determine the effect of changing the KU5 dosages. It can be seen in this graph that increasing the KU5 dosage has no noticeable effect in decreasing the chrome recovery. The chrome recovery at 200 g/ton KU5 dosage is as significantly different from the one at zero copper sulphate as the chrome recovery obtained with 100 g/ton addition of KU5. This is validated by the statistical analysis as depicted in table 4.4. It can be concluded that increasing the KU5 dosage was not successful in depressing the chromite that has been inadvertently activated by the addition of copper sulphate.

The difference in the behaviour of these depressants can be attributed to the fact that the CMCs are negatively charged compared to guar. Copper ions would be expected to react with the negatively charged CMCs and result in formation of a precipitate and affect the adsorption behaviour.

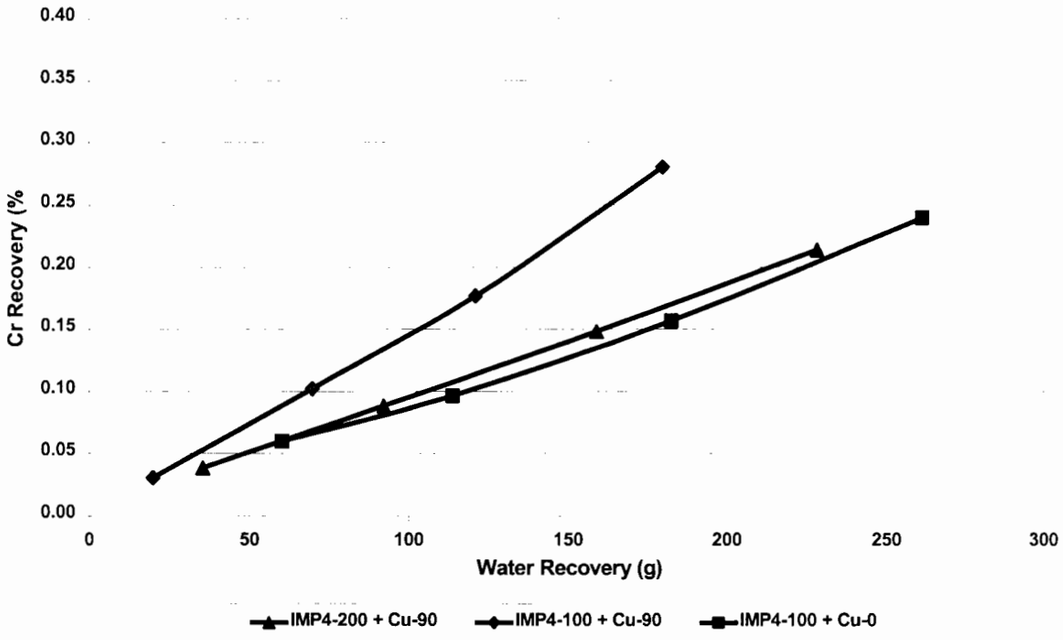

Figure 4.17: The effect of varying guar (IMP4) dosage at 90 g/ton copper sulphate on chrome recovery

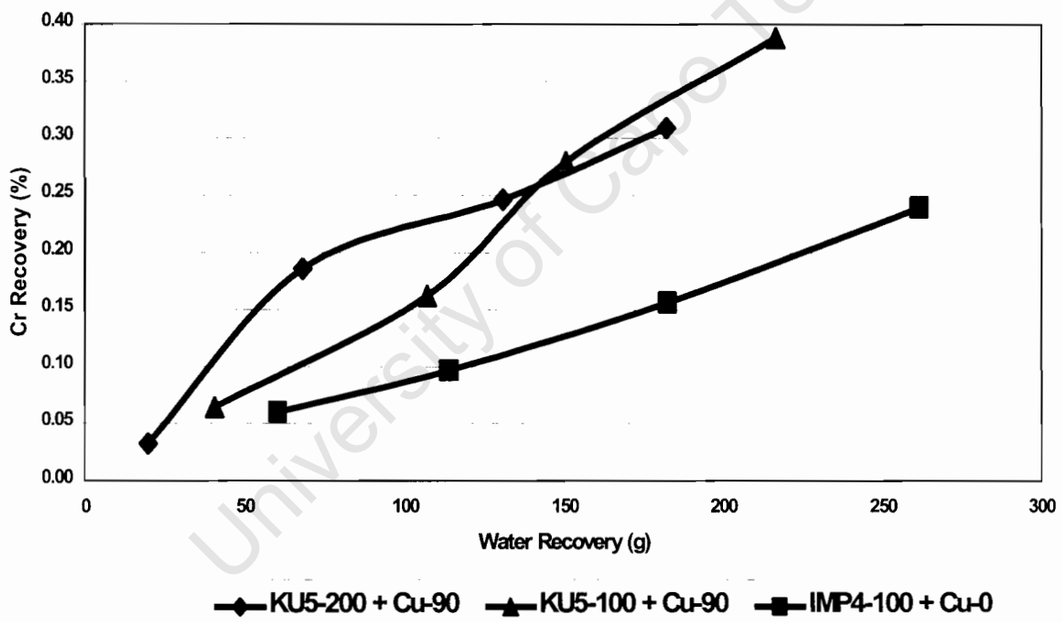

Figure 4.18: The effect of varying CMC (KU5) dosage at a dosage of 90 g/t copper sulphate with guar (IMP4) addition of 100 g/ton as the standard

4.2.4 Effect of copper sulphate on water recovery

The froth zone generally provides the environment for the separation of valuable minerals from the gangue material, allowing drainage of the entrained material back into the pulp. There is optimum froth stability. When the froth is not stable enough the mineralised bubbles rupture before collection. When the froth is too stable, not enough drainage occurs and the water and gangue recoveries are too high. It is generally accepted that the nature and dosage of the frother as well as the nature of the particles in the froth affect the froth stability.

Figures 4.19 and 4.20 below show that in the case where excess copper sulphate (90 /ton) was added in the solution, lower water recoveries were obtained compared to the water recoveries obtained at the copper sulphate dosages of 0 g/ton and 30 g/ton. This suggests that the froth has been rendered unstable by the addition of excess copper sulphate. At 90 g/ton dosage of copper sulphate, it was observed that the water recovery was reduced showing instability of the froth, however chrome recoveries were higher than when high water recoveries were obtained in the case where no copper sulphate was added. This showed that other mechanisms other than entrainment were responsible for high chrome recoveries.

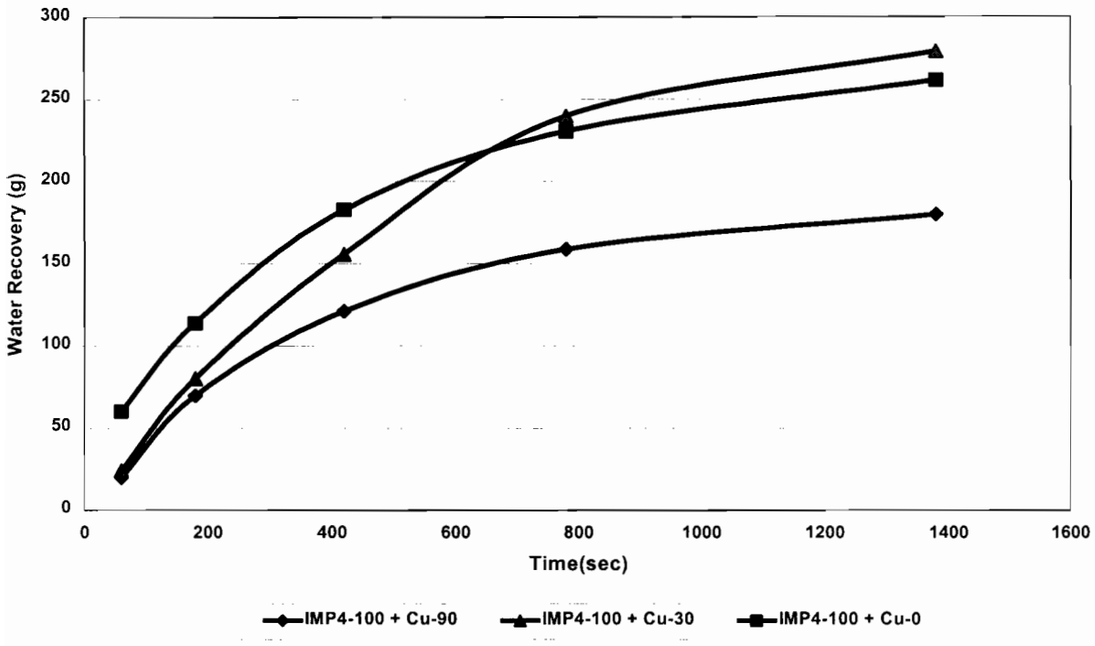

Figure 4.19: Effect of copper sulphate on water recovery at constant guar (IMP4) dosage as a function of time

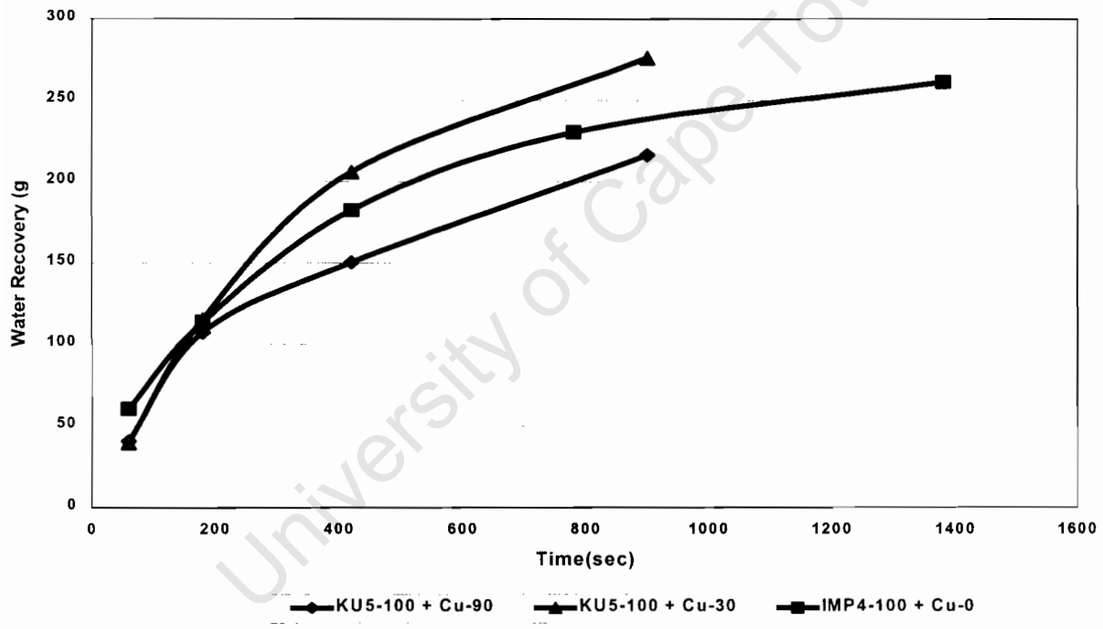

Figure 4.20: Effect of copper sulphate on water recovery at constant CMC (KU5) dosage as a function of time

4.2.5 Stoichiometric addition of copper sulphate with the thiol collectors

The activation of chromite is believed to be brought about by the presence of excess copper ions in solution. Figure 4.21 and 4.22 show the results of chrome recoveries obtained when 30 g/ton of copper sulphate was added. Given that this is the stoichiometric amount required to react with thiol collectors, it is expected that the chrome recovery would not increase significantly higher than that obtained when no copper sulphate was added. The results obtained using 200 g/ton of IMP4 in the presence of 30 g/ton copper sulphate show that the chrome recoveries were similar to those achieved when no copper sulphate was added. The results at 100 g/ton IMP4 were also not statistically different from the results when no copper sulphate was added.

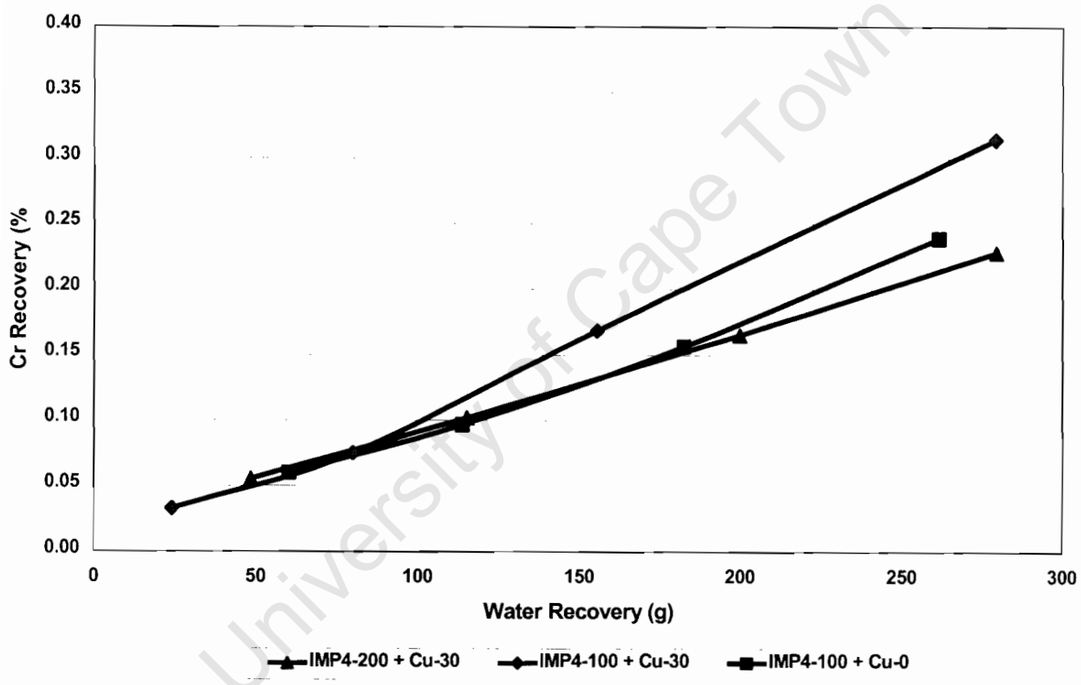

Figure 4.21: The effect of adding 30 g/ton copper sulphate on the recovery of chrome under various guar (IMP4) dosages

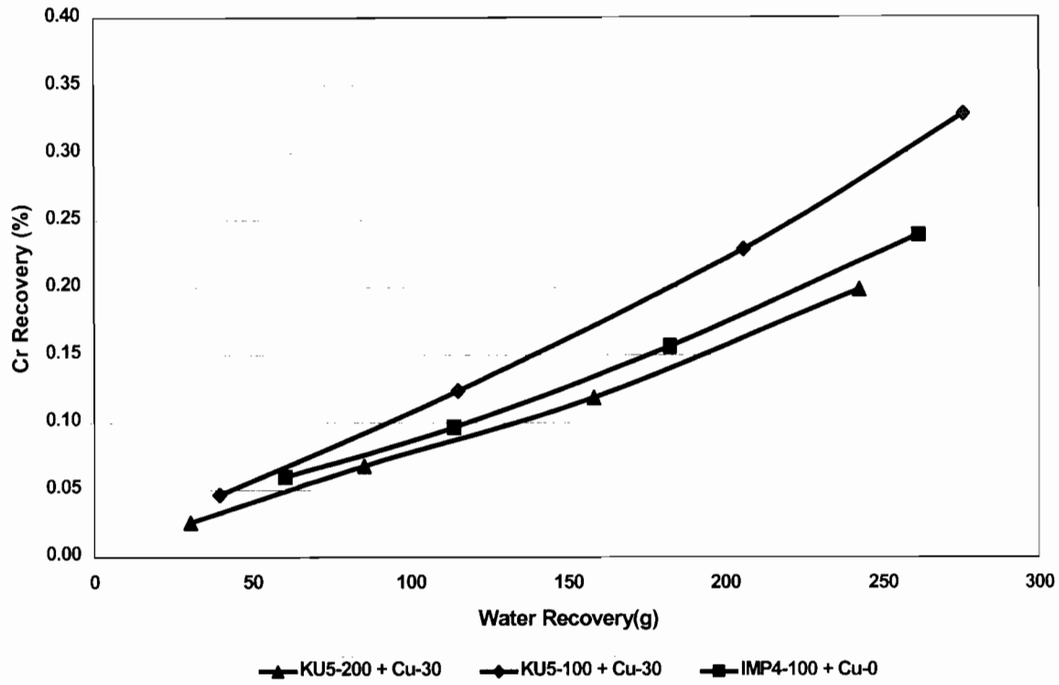

Figure 4.22: The effect of adding 30 g/ton copper sulphate on the recovery of chrome under various CMC (KU5) dosages

Both figures 4.21 and 4.22 show that the addition of 30 g/ton copper sulphate does not significantly increase the chrome recovery (90% confidence interval). Although the results obtained with 100 g/ton of depressant showed a slight increase in the recovery, this was not statistically different. The confidence interval of 90 % which is generally accepted as significant.

Thus based on these results it can be said that activation of chromite required excess copper ions in solution.

4.3 Summary of the Results

4.3.1 Pyroxene and Feldspar

The microflotation tests showed that both pyroxene and feldspar are naturally hydrophilic. The addition of $5*10^{-5}$ M $CuSO_4$ and SIBX $5*10^{-5}$ resulted in the increase in pyroxene and feldspar recovery at pH 9 in the presence of 10^{-3} KNO_3 ionic strength (89.5% and 89.1% respectively).

However the addition of copper sulphate ($5*10^{-5}$ M) and SIBX ($5*10^{-5}$ M) in the presence of 10^{-2} $Ca(NO_3)_2$ ionic strength resulted in lower increase in recovery (50.4% and 53.3% respectively). This suggested that calcium ions were competing with copper ions on the surface of pyroxene and feldspar.

The order of addition of copper sulphate and SIBX during conditioning stage plays an important role in activation of these minerals. When copper sulphate was added first, activation occurred but when SIBX was added first, no activation was observed. This suggested that copper ions facilitated the attachment of the collector onto the surface. No activation was observed at pH 4 suggesting that the copper species present at that pH did not adsorb onto the mineral surface.

The addition of polymeric depressants i.e. guar on the activated mineral surfaces resulted in the reversal of activation indicated by low recoveries of pyroxene and feldspar which was equivalent to when no copper sulphate was added. This was only observed in the presence of 10^{-2} $Ca(NO_3)_2$ ionic strength. The addition of a highly charged CMC (Depramin 186) did not show any reversal in activation under all the conditions studied. This suggests that no CMC adsorption onto the mineral surface was observed and thus no reduction in recovery occurred. The addition of a lower charge CMC (Depramin C) resulted in slight reversal of activation although recoveries were still above 30%.

Zeta potential measurements showed reversal in charge at pH values above 7. This shows that depending on the pH, different copper species are formed and may be responsible for activation although copper sulphate alone did not result in increased flotation

4.3.2 Chromite

Chrome assays were done to indicate the performance of chromite (Cr_2O_3.FeO). The results obtained to investigate the effect of copper sulphate dosage on the recovery of chrome have shown that at a copper sulphate dosage of 90 g/ton and depressant dosage of 100 g/ton, increased chrome recoveries were achieved. The results with the system where no copper sulphate was added and the system with 30 g/ton showed no significant difference in chrome recoveries. This can be explained by the fact that 30 g/ton is the stoichiometric amount required to react with the thiol collectors. The increase in chrome recoveries can be attributed to the excess copper sulphate in solution that is responsible for activating the chromite.

When observing the effects of increasing the depressant dosage to 200 g/ton, it was observed that guar (IMP4) was able to depress the activated chromite at 90 g/ton of copper sulphate dosage. CMC (KU5) at 200 g/ton however, was unable to depress the activated chromite.

In addition, the increase in copper sulphate dosage (i.e. at 90 g/ton) resulted in lower water recoveries compared with other systems at 30 g/ton and 0 g/ton copper sulphate. This is an indication that excess copper sulphate causes the destabilisation of the froth phase, resulting in lower contribution to recovery from entrainment.

The link between microflotation work and batch testwork is difficult since the reagent addition rates were not the same. The aim was to confirm previous studies on copper sulphate activation and the use of depressant to counter the

activation. Microflotation tests were done to investigate the behaviour of feldspar and pyroxene. This has meant that the report is very much two discrete parts with an overall theme. Although this works reasonably well, there are some instances where it is difficult to relate the operating conditions – the major example being the dosages of the reagents added. In the microflotation tests where 2 g of mineral was added to 100 ml solution, dosages are ppm in solution, whereas in the batch flotation tests 1 kg of ore was added to 2 l water to make up to 3 l and it more appropriate to use dosages in g/t ore [Bradshaw, 2003].

4.4 Discussion of proposed mechanisms

4.4.1 Activation by copper sulphate

Both the batch and microflotation tests have shown that pyroxene, feldspar and chromite are not naturally floatable. Their recoveries in the absence of reagents can be attributed to entrainment. The pyroxene and feldspar microflotation recoveries in the absence of reagents were found to be 12.0 and 9.5 % respectively, showing low flotation response.

It has been shown that activation of various minerals is a function of pH. For example a classical activation case of sulphide minerals at acidic pH values involves Cu^{++} as the dominating species. Activation of sphalerite (ZnS) with copper sulphate is controlled by the reaction [Mao et al, (1959)]:

$$ZnS(s) + Cu^{2+} \Leftrightarrow CuS(s) + Zn^{2+}$$

It has also been shown that after activation, flotation behaviour is similar to that of the activator metal sulphide mineral. Thus, stoichiometric replacement occurred. However, the activation of silicate minerals does not follow the sulphide mineral activation mechanism since the activator metal can directly substitute no metal on the silicate mineral (Fuerstenau, 1982). Figure 4.6 shows the flotation of

pyroxene in the presence of $5*10^{-5}$ M copper sulphate and $5*10^{-5}$ M SIBX at pH 4. The results suggest that no activation of silicate minerals is possible at this low pH since Cu^{2+} ions cannot directly adsorb onto the mineral surface unlike species occurring at higher pH values.

Malysiak, (2003) did extensive work on feldspar and pentlandite and showed that at pH 9, activation by copper sulphate was evident for both pyroxene and pentlandite. The copper speciation diagram (Figure 2.10) suggests that $Cu(OH)_2$ precipitates on the mineral surfaces and the subsequent addition of collector enhances the flotation recovery. This finding is supported by zeta potential measurements performed by Martinovic, 2002 and Shackleton, 2003. This thesis also reports that pyroxene and feldspar are activated by copper sulphate and the subsequent addition of SIBX enhances flotation recovery. When SIBX was added before copper sulphate no enhancement in recovery was observed indicating that the xanthate reacted with the copper sulphate in solution and that there was no mineral surface activity.

As is the case for pyroxene and feldspar, the copper species diagram suggests that depending on the pH, various copper species are likely to occur and these can be responsible for activation of minerals. Wesseldijk reported that in the conditioning stage of the microfloats, copper sulphate was added first, followed by the addition of the xanthate (SIBX). Two copper sulphate dosages were used i.e. $5*10^{-5}$ M and $1*10^{-4}$ M and the same concentrations of SIBX were added. This meant that the copper sulphate concentration was always greater than the collector concentration and that there will always be excess copper ions in solution. This is based on the fact that one mole of copper will react with two moles of xanthate according to the reaction:

$$Cu^{++} + 2\,X^{-} \Leftrightarrow CuX_2 \Leftrightarrow CuX + 1/2\,X_2$$

The first mechanism suggests that depending on the pH of the pulp, the copper ions in solution will be hydrolysed to the metal hydroxy species according to the reaction:

$$Cu^{++} + H_2O \Rightarrow CuOH^{+} + H^{+}$$

And

$$CuOH^{+} + H_2O \Rightarrow Cu(OH)_{2\,(s)} + H^{+}$$

Since the experiments were conducted at pH 9, $Cu(OH)_2$ will be the dominant species. Zeta potential measurements show a decline in its positive value, which is close to neutral at pH 9 due to the presence of this [Wesseldijk et al, 1999]. The mechanism suggests that the $Cu(OH)_2$ precipitates on the mineral surface.

Microflotation experiments carried out by Wesseldijk [1999] on chromite showed that substantial chromite recoveries were observed when copper sulphate was added, since copper ions react strongly with thiol collectors. The results also suggested that there was some interaction between the copper ions and the mineral surface. The logarithmic concentration diagram of $1*10^{-4}$ M of copper ions as a function of pH shows that below pH 4, Cu^{2+} ions is the only species present. Also interesting is the fact that at pH around 7, the copper hydroxide ($Cu(OH)_2$) starts to precipitate. From pH around 8.3 onwards it is the only species left. Zeta potential measurements by Martinovic [2002] also show that during this change of copper species i.e. pH 7 there is a charge reversal in the zeta-potential and becomes negative in the pH 9 to pH 10 region. This is consistent with what is reported in literature.

4.4.2 Effect of water quality and metal ions

Activation was shown to be more enhanced in the presence of KNO_3. The pyroxene and feldspar recoveries were found to be greater than 80%. The presence of $Ca(NO_3)_2$ showed reduction of activation of pyroxene. The

mechanism involved is not clear but there is evidence that calcium ions interfere with the adsorption of copper species onto the mineral and hence less copper is adsorbed and this will reduce flotation response. The difference in metal ions used (i.e. K^+ and Ca^{2+}) has shown that monovalent and divalent metal ions have different properties and these change the flotation behaviour. Shortridge (2002) also showed that the presence of calcium and magnesium ions in solution reduced the natural flotation of talc. He suggests that the presence of divalent ions have a greater degree adsorption onto the mineral surface. The investigation of speciation of $Ca(NO_3)_2$ indicated that at pH 9, the predominant form of the calcium ions in solution was Ca^{2+} with less than 1% of these species being in the form $CaOH^+$. It lead to the conclusion that it was unlikely that these hydrated monovalent ions played any significant role in adsorption at the talc surface at the pH values used.

Two possible mechanisms are that the calcium species present at that pH adsorb onto the mineral surface and hence reduce the amount of copper that will adsorb or there maybe a reaction in solution. No reaction between xanthate and the calcium species is expected thus there would be no increased hydrophobicity of the mineral surface. Further research is recommended to confirm which of these occurs.

4.4.3 Effect of depressants

The use of depressants showed that activation could be reversed. The guar depressants showed to be more effective in the reversal of activation. For the microflotation results this was shown in the presence of $Ca(NO_3)_2$ ions. The low charge CMC (Depramin C) slightly reversed activation under these conditions whereas the high charge CMC (Depramin 186) showed no reversal in activation under all conditions. This can be attributed to the increased adsorption of guar compared to CMCs as reported by Steenberg and Harris (1984) and Shortridge et al, (1999b).

The work carried out by Parolis et al, (2003) and Shortridge (2002) demonstrated that the adsorption of CMCs on talc was affected by the type of ions in solution. Divalent cations improved adsorption of CMCs onto talc and this made the act of depressants more effective.

The effectiveness of the ability of depressants to reverse activation has thus been attributed to the extent of adsorption of the depressants on the mineral surface. This should be further investigated and the use of mineral surface techniques will be necessary.

CHAPTER 5: CONCLUSIONS

- The thesis has shown that pyroxene, feldspar and chromite are not naturally floatable and the levels recovered in both microflotation and batch tests could be attributed to entrainment.

- All the gangue minerals investigated were equally activated by copper sulphate at pH 9 and this is shown by the flotation tests and supported by zeta potential measurements performed by Martinovic, (2002) and Shackleton et al, (2003).

    - The addition of collector (SIBX) only, did not enhance the flotation of pyroxene, feldspar and chromite.
    - Although copper sulphate alone adsorbed onto the gangue minerals as shown by the zeta potential measurements performed by Martinovic (2002) and Shackleton (2003), no enhancement in flotation recovery was observed.
    - The subsequent addition of SIBX to the copper activated minerals resulted in enhanced flotation recoveries. This enhancement was observed at pH 9 and not at pH 4.
    - When SIBX was added prior to copper sulphate, no enhancement in flotation recoveries were observed.

- Microflotation tests showed that flotation enhancement was not affected by the copper sulphate dosages used. The batch flotation results showed that additions of the stoichiometric amount of copper sulphate required to react with collectors did not enhance chromite recovery but dosages above the stoichiometric level enhanced chromite recoveries.

- The tests performed showed that under certain conditions, depressants could reverse the enhanced recoveries of pyroxene, feldspar and chromite but this depended on depressant type and dosage.

- The effect of guar was shown to be different from that of CMC. In both the microflotation tests and batch tests, guar was effective in reversing the activation of pyroxene, feldspar and chromite. The CMCs were not effective in reversing activation. In batch flotation tests it was shown that depressant dosage does have an effect in the reversal of activation. In microflotation tests even low guar dosages were effective in reversing activation.

- The low charge CMC (Depramin C) was more effective in reversing the activation of pyroxene and feldspar as compared to the high charge CMC (Depramin 186). This was observed in the presence of 10^{-3} and 10^{-2} ionic strength conditions of $Ca(NO_3)_2$. This can be attributed to increased adsorption which is consistent with the findings of Parolis et al, (2004) who reported increased adsorption onto talc with decreased charge of CMC.

- Ionic strength and type of type of cation played a significant role on the extent of activation of pyroxene and feldspar. In the presence of KNO_3 at 10^{-3} and 10^{-2} ionic strength the activation was enhanced. In the presence of 10^{-3} $Ca(NO_3)_2$ activation was still enhanced but when the ionic strength was increased to 10^{-2}, reduction of activation of both pyroxene and feldspar was observed.

- The batch flotation tests have shown qualitatively that copper sulphate, inadvertently activated chromite at dosages higher than the stoichiometric requirement and thus results in higher chromite recoveries. This was shown by the graphs based on the chrome (Cr) assays that were used to study the performance of chromite (Cr_2O_3.FeO). It has also been shown that guar (IMP4) is more effective at reducing this activation at higher copper sulphate

dosages that CMC (KU5). The behaviour of pyroxene and feldspar in batch flotation was not investigated as no techniques were available but work from microflotation suggests that behaviour would be the same.

CHAPTER 6: RECOMMENDATIONS

- From all the testwork done, it can be seen that copper sulphate addition can inadvertently activate gangue minerals and it is non-selective. Batch flotation or pilot plant testwork on Merensky ore would be the next step to evaluate the effect of copper sulphate on pyroxene and feldspar if mineralogical techniques were available. Microflotation experiments only showed the fundamental behaviour of reagents added during flotation. The presence of frother was ignored and this would not give a true indication of results obtained in plants.

- Valuable minerals should also be analysed to get an indication of how the copper sulphate addition and dosage impact the recovery of these minerals. UG-2 ore has high chromite content and the base metals are only present in subordinate amounts except for pentlandite. Thus pilot plant work could reveal the indication and impact on valuable minerals since enough samples can be generated. Size by size analysis of chrome would give a better identification of the material recovered by true flotation and entrainment.

- More testwork needs to be done on depressant type to identify conditions in which guars and CMCs perform effectively.

- Microflotation tests also suggest that water quality is an important factor in flotation. Plant water should be used when flotation is performed to get a realistic interpretation of plant operation. More work should be done with various CMCs that have a variety of characteristics.

- Surface chemical analysis techniques should be used for the investigation of interaction of cations and polymers on the mineral surfaces.

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APPENDICES

APPENDIX A: MICROFLOTATION PROCEDURE

1. Weigh 2 grams of the sample and ultrasound for 30 seconds with pure deionised water.
2. Decant the ultrafines with the supernatant liquid and ultrasound again for 30 seconds. Decant any ultrafines.
3. All the standard experiments are to be done in 10^{-3} M KNO_3. Add exactly 100 ml.

4. Reagent conditioning is to be done in a 100 ml beaker before transferring the pulp into the microfloat cell. The reagent conditioning is to be done in the following way:

    - **5 minutes for $CuSO_4$**
    - **2 minutes for SIBX**
    - **2 minutes for depressant**

5. The conditioning is to be done using an overhead stirrer since magnetic bars will tend to grind the particles to a finer size.

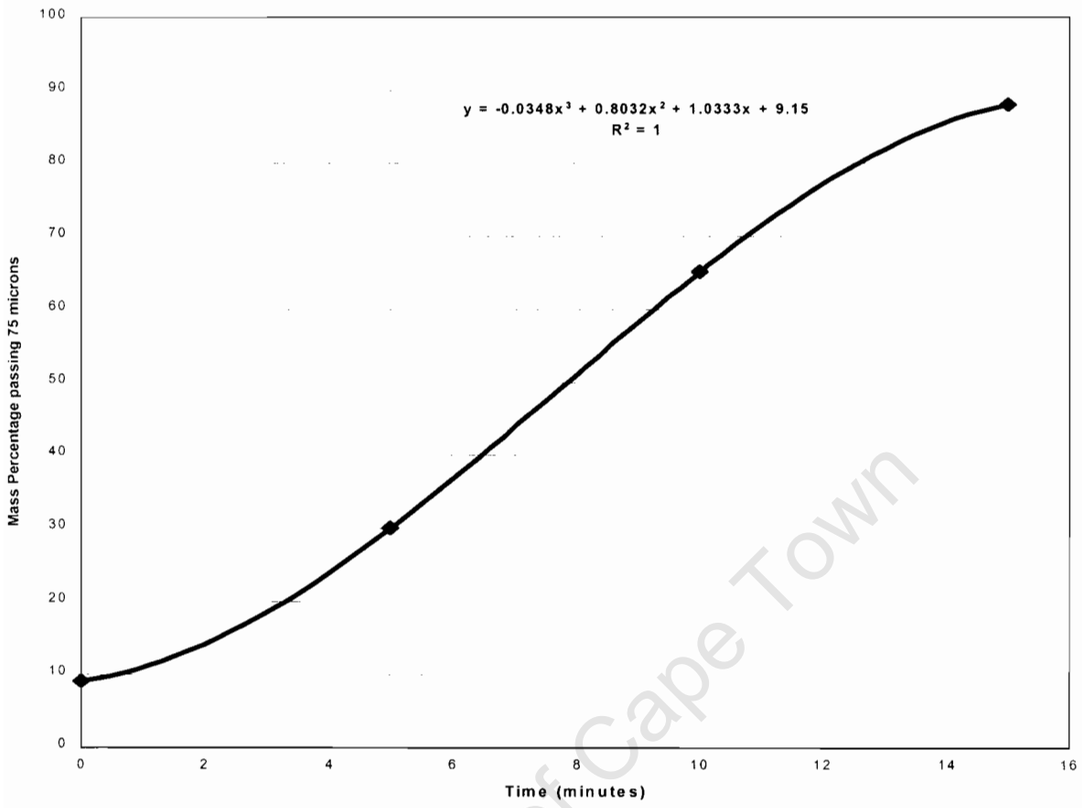
6. The filter papers are to be desiccated 12 hours before using and weighing. After filtering and drying, the samples are to be desiccated.

7. Flotation time is to last 10 minutes with three concentrates taken at 1.5, 5 and 10 minutes in the concentrate launder. When collecting the concentrate a wash bottle is used to wash out the solids in the launder.

The ultrafines to be discarded because they tend to consume reagents during conditioning.

APPENDIX B: MILLING CURVE FOR THE UG-2 BATCH FLOTATION TESTS

The equation displayed on the graph can be used to determine the time required obtaining a product with 40 % passing 75 microns. A time of 6.38 minutes was obtained. For experimental purposes, a time of 6 minutes and 30 seconds was chosen. Using curve it shows that at this time 40.9 % of the particles are below 75 microns.



APPENDIX C: SUMMARY OF THE MICROFLOTATION DATA

Run	Reagents added	pH	Total Mass (g)	Time (min)	Recovery (%)	Mean Recovery (%)	Standard Deviation
PXN 1	10^{-3} M KNO_3	9	1.8844	5	1.69		
				10	6.22		
				20	11.92		
PXN 2	10^{-3} M KNO_3	9	1.8438	5	1.35	1.56	0.19
				10	5.57	5.50	0.77
				20	12.43	11.99	0.42
PXN 3	10^{-3} M KNO_3	9	1.8908	5	1.63		
				10	4.70		
				20	11.61		
PXN 4	$CuSO_4$ - $1*10^{-4}$ M	9	1.8333	1.5	63.52		
	SIBX - $1*10^{-4}$ M			5	89.76		
	10-3 M KNO3			10	95.11		
PXN 5	$CuSO_4$ - $1*10^{-4}$ M	9	1.8659	1.5	61.92	63.54	1.63
	SIBX - $1*10^{-4}$ M			5	91.29	90.89	0.99
	10-3 M KNO3			10	95.77	95.39	0.34
PXN 6	$CuSO_4$ - $1*10^{-4}$ M	9	1.9144	1.5	65.19		
	SIBX - $1*10^{-4}$ M			5	91.61		
	10-3 M KNO3			10	95.30		
PXN 7	$CuSO_4$ - $1*10^{-4}$ M	9	1.8758	1.5	65.74	65.01	1.03
	SIBX - $2*10^{-4}$ M			5	94.70	91.71	4.22
	10-3 M KNO3			10	95.15	95.46	0.44
PXN 8	$CuSO_4$ - $1*10^{-4}$ M	9	1.9490	1.5	64.28		
	SIBX - $2*10^{-4}$ M			5	88.73		
	10-3 M KNO3			10	95.78		
PXN 9	$CuSO_4$ - $1*10^{-4}$ M	9					
	SIBX - $1*10^{-4}$ M			1.5	60.31	61.61	1.84
	APX 4 M @ 50 mg/l			5	91.56	91.48	0.12
	10-3 M KNO3			10	95.31	94.99	0.45
PXN 10	$CuSO_4$ - $1*10^{-4}$ M	9					
	SIBX - $1*10^{-4}$ M			1.5	62.91		
	APX 4 M @ 50 mg/l			5	91.40		
	10-3 M KNO3			10	94.68		

PXN 11	$CuSO_4$ - $1*10^{-4}$ M	9					
	SIBX - $1*10^{-4}$ M			1.5	58.24	59.35	1.56
	APX 4 M @ 100mg/l			5	86.12	86.72	0.85
	10-3 M KNO3			10	92.31	91.88	0.61
PXN 12	$CuSO_4$ - $1*10^{-4}$ M	9					
	SIBX - $1*10^{-4}$ M			1.5	60.45		
	APX 4 M @ 100mg/l			5	87.32		
	10-3 M KNO3			10	91.45		
PXN 13	$CuSO_4$ - $5*10^{-5}$ M	9	1.8664	1.5	64.79	54.92	13.96
	SIBX - $5*10^{-5}$ M			5	88.80	82.02	9.59
	10-3 M KNO3			10	93.98	89.50	6.34
PXN 14	$CuSO_4$ - $5*10^{-5}$ M	9	1.9276		45.05		
	SIBX - $5*10^{-5}$ M				75.23		
	10-3 M KNO3				85.02		
PXN 15	$CuSO_4$ - $5*10^{-5}$ M	9	1.952	1.5	29.54	33.75	5.95
	SIBX - $5*10^{-5}$ M			5	59.90	63.45	5.03
	$Ca(NO_3)_2$ @ $3.33*10^{-4}$ M			10	70.60	74.58	5.62
PXN 16	$CuSO_4$ - $5*10^{-5}$ M	9	1.973	1.5	37.96		
	SIBX - $5*10^{-5}$ M			5	67.00		
	$Ca(NO_3)_2$ @ $3.33*10^{-4}$ M			10	78.56		
PXN 17	$CuSO_4$ - $5*10^{-5}$ M	9	1.9123	1.5	21.94	22.67	1.02
	SIBX - $5*10^{-5}$ M			5	41.92	42.92	1.40
	$Ca(NO_3)_2$ @ $3.33*10^{-3}$ M			10	48.38	50.36	2.80
PXN 18	$CuSO_4$ - $5*10^{-5}$ M	9	1.9375	1.5	23.39		
	SIBX - $5*10^{-5}$ M			5	43.91		
	$Ca(NO_3)_2$ @ $3.33*10^{-3}$ M			10	52.34		
PXN 19	$CuSO_4$ - $5*10^{-5}$ M	9	1.8747				
	SIBX - $5*10^{-5}$ M			1.5	41.42	40.35	1.52
	(10^{-3} M KNO_3)			5	72.30	69.10	4.52
	APX 4M @ 50 mg/l			10	82.16	79.77	3.37
PXN 20	$CuSO_4$ - $5*10^{-5}$ M	9	1.8856				
	SIBX - $5*10^{-5}$ M			1.5	39.27		
	(10^{-3} M KNO_3)			5	65.90		
	APX 4M @ 50 mg/l			10	77.39		

PXN 21	$CuSO_4$ - $5*10^{-5}$ M	9	1.9277				
	SIBX - $5*10^{-5}$ M			1.5	32.47	29.58	4.09
	$3.33*10^{-4}$ M $Ca(NO_3)_2$			5	57.70	49.96	10.95
	APX 4M @ 50 mg/l			10	67.87	62.54	7.54
PXN 22	$CuSO_4$ - $5*10^{-5}$ M	9	1.9144				
	SIBX - $5*10^{-5}$ M			1.5	26.68		
	$3.33*10^{-4}$ M $Ca(NO_3)_2$			5	42.22		
	APX 4M @ 50 mg/l			10	57.21		
PXN 23	$CuSO_4$ - $5*10^{-5}$ M	9	1.9066				
	SIBX - $5*10^{-5}$ M			1.5	3.59	4.28	0.98
	$3.33*10^{-3}$ M $Ca(NO_3)_2$			5	5.49	5.93	0.62
	APX 4M @ 50 mg/l			10	6.80	7.25	0.64
PXN 24	$CuSO_4$ - $5*10^{-5}$ M	9	1.9498				
	SIBX - $5*10^{-5}$ M			1.5	4.98		
	$3.33*10^{-3}$ M $Ca(NO_3)_2$			5	6.38		
	APX 4M @ 50 mg/l			10	7.71		
	$CuSO_4$ - $5*10^{-5}$ M	9	1.9207				
PXN 25	SIBX - $5*10^{-5}$ M			1.5	54.82	58.86	5.71
	10^{-3} M KNO_3			5	80.35	83.22	4.07
	DEP 186 @ 50 mg/l (active content)			10	87.13	89.61	3.50
	$CuSO_4$ - $5*10^{-5}$ M	9	1.9816				
PXN 26	SIBX - $5*10^{-5}$ M			1.5	62.89		
	10^{-3} M KNO_3			5	86.10		
	DEP 186 @ 50 mg/l (active content)			10	92.08		
PXN 27	$CuSO_4$ - $5*10^{-5}$ M	9	1.9307				
	SIBX - $5*10^{-5}$ M			1.5	37.33	36.91	0.59
	$3.33*10^{-4}$ M $Ca(NO_3)_2$			4.96	66.12	66.28	0.22
	DEP 186 @ 50 mg/l (active content)			10	75.58	75.70	0.17
PXN 28	$CuSO_4$ - $5*10^{-5}$ M	9	1.9147				
	SIBX - $5*10^{-5}$ M			1.5	36.49		
	$3.33*10^{-4}$ M $Ca(NO_3)_2$			5	66.44		
	DEP 186 @ 50 mg/l (active content)			10	75.81		
PXN 29	$CuSO_4$ - $5*10^{-5}$ M	9					
	SIBX - $5*10^{-5}$ M			1.5	32.19	31.03	1.65
	$3.33*10^{-3}$ M $Ca(NO_3)_2$			5	51.34	52.31	1.37
	DEP 186 @ 50 mg/l (active content)			10	59.73	60.70	1.37
PXN 30	$CuSO_4$ - $5*10^{-5}$ M	9					
	SIBX - $5*10^{-5}$ M			1.5	29.86		
	$3.33*10^{-3}$ M $Ca(NO_3)_2$			5	53.28		
	DEP 186 @ 50 mg/l (active content)			10	61.67		

PXN 31	$CuSO_4$ - $5*10^{-5}$ M	9					
	SIBX - $5*10^{-5}$ M			1.5	37.33	37.17	0.22
	$3.33*10^{-3}$ M $Ca(NO_3)_2$			5	62.70	61.42	1.81
	DEP 186 @ 100 mg/l (active content)			10	67.82	66.65	1.65
PXN 32	$CuSO_4$ - $5*10^{-5}$ M	9					
	SIBX - $5*10^{-5}$ M			1.5	37.01		
	$3.33*10^{-3}$ M $Ca(NO_3)_2$			5	60.14		
	DEP 186 @ 100 mg/l (active content)			10	65.49		
PXN 33	$CuSO_4$ - $5*10^{-5}$ M	9					
	SIBX - $5*10^{-5}$ M			1.5	38.61	37.61	1.41
	$3.33*10^{-3}$ M $Ca(NO_3)_2$			5	60.59	60.34	0.36
	DEP 186 @ 200 mg/l (active content)			10	66.97	66.03	1.33
PXN 34	$CuSO_4$ - $5*10^{-5}$ M	9					
	SIBX - $5*10^{-5}$ M			1.5	36.61		
	$3.33*10^{-3}$ M $Ca(NO_3)_2$			5	60.09		
	DEP 186 @ 200 mg/l (active content)			10	65.09		
PXN 35	$CuSO_4$ - $5*10^{-5}$ M	9					
	SIBX - $5*10^{-5}$ M			1.5	47.61	47.37	0.34
	$1*10^{-3}$ M KNO_3			5	73.19	71.48	2.42
	APX 4M @ 100 mg/l			10	79.28	76.62	3.76
PXN 36	$CuSO_4$ - $5*10^{-5}$ M	9					
	SIBX - $5*10^{-5}$ M			1.5	47.13		
	$1*10^{-3}$ M KNO_3			5	69.77		
	APX 4M @ 100 mg/l			10	73.97		
PXN 37	$CuSO_4$ - $5*10^{-5}$ M	9					
	SIBX - $5*10^{-5}$ M			1.5	42.97	42.41	0.80
	$3.33*10^{-4}$ M $Ca(NO_3)_2$			5	63.18	62.55	0.90
	APX 4M @ 100 mg/l			10	71.24	71.10	0.21
PXN 38	$CuSO_4$ - $5*10^{-5}$ M	9					
	SIBX - $5*10^{-5}$ M			1.5	41.84		
	$3.33*10^{-4}$ M $Ca(NO_3)_2$			5	61.91		
	APX 4M @ 100 mg/l			10	70.95		
PXN 39	$CuSO_4$ - $5*10^{-5}$ M	9					
	SIBX - $5*10^{-5}$ M			1.5	45.99	44.63	1.92
	$1*10^{-2}$ M KNO_3			5	72.36	71.22	1.62
	APX 4M @ 100 mg/l			10	78.12	77.56	0.80
PXN 40	$CuSO_4$ - $5*10^{-5}$ M	9					
	SIBX - $5*10^{-5}$ M			1.5	43.27		
	$1*10^{-2}$ M KNO_3			5	70.07		
	APX 4M @ 100 mg/l			10	76.99		

PXN 41	$CuSO_4$ - $1*10^{-4}$ M	9					
	SIBX - $1*10^{-4}$ M			1.5	2.93	3.04	0.16
	3.33*10-3 M Ca(NO3)2			5	5.63	5.64	0.01
	APX 4M @ 100 mg/l			10	8.29	8.60	0.44
PXN 42	$CuSO_4$ - $1*10^{-4}$ M	9					
	SIBX - $1*10^{-4}$ M			1.5	3.15		
	3.33*10-3 M Ca(NO3)2			5	5.64		
	APX 4M @ 100 mg/l			10	8.91		
PXN 43	CuSO4 - 1*10-4 M	9					
	SIBX - 1*10-4 M			1.5	43.52	44.70	1.66
	3.33*10-3 M Ca(NO3)2			5	61.39	61.86	0.66
				10	72.14	73.44	1.84
PXN 44	CuSO4 - 1*10-4 M	9					
	SIBX - 1*10-4 M			1.5	45.87		
	3.33*10-3 M Ca(NO3)2			5	62.33		
				10	74.74		
PXN 45	CuSO4 - 1*10-4 M	9					
	SIBX - 1*10-4 M			1.5	42.21	42.79	0.82
	3.33*10-3 M Ca(NO3)2			5	56.14	55.95	0.27
	Dep186 @ 50 mg/l (active content)			10	68.69	67.06	2.31
PXN 46	CuSO4 - 1*10-4 M	9					
	SIBX - 1*10-4 M			1.5	43.37		
	3.33*10-3 M Ca(NO3)2			5	55.76		
	Dep186 @ 50 mg/l (active content)			10	65.42		
PXN 47	CuSO4 - 1*10-4 M	9					
	SIBX - 1*10-4 M			1.5	38.41	37.31	1.56
	3.33*10-3 M Ca(NO3)2			5	51.32	50.48	1.20
	Dep186 @ 200 mg/l (active content)			10	61.49	59.21	3.23
PXN 48	CuSO4 - 1*10-4 M	9					
	SIBX - 1*10-4 M			1.5	36.21		
	3.33*10-3 M Ca(NO3)2			5	49.63		
	Dep186 @ 200 mg/l (active content)			10	56.92		
PXN 49	$CuSO_4$ - $5*10^{-5}$ M	9					
	SIBX - $5*10^{-5}$ M			1.5	56.78	57.06	0.40
	$1*10^{-2}$ M KNO_3			5	79.21	80.33	1.58
				10	88.84	87.91	1.32
PXN 50	$CuSO_4$ - $5*10^{-5}$ M	9					
	SIBX - $5*10^{-5}$ M			1.5	57.34		
	$1*10^{-2}$ M KNO_3			5	81.45		
				10	86.98		

PXN 41	$CuSO_4$ - $1*10^{-4}$ M	9					
	SIBX - $1*10^{-4}$ M			1.5	2.93	3.04	0.16
	3.33*10-3 M Ca(NO3)2			5	5.63	5.64	0.01
	APX 4M @ 100 mg/l			10	8.29	8.60	0.44
PXN 42	$CuSO_4$ - $1*10^{-4}$ M	9					
	SIBX - $1*10^{-4}$ M			1.5	3.15		
	3.33*10-3 M Ca(NO3)2			5	5.64		
	APX 4M @ 100 mg/l			10	8.91		
PXN 43	CuSO4 - 1*10-4 M	9					
	SIBX - 1*10-4 M			1.5	43.52	44.70	1.66
	3.33*10-3 M Ca(NO3)2			5	61.39	61.86	0.66
				10	72.14	73.44	1.84
PXN 44	CuSO4 - 1*10-4 M	9					
	SIBX - 1*10-4 M			1.5	45.87		
	3.33*10-3 M Ca(NO3)2			5	62.33		
				10	74.74		
PXN 45	CuSO4 - 1*10-4 M	9					
	SIBX - 1*10-4 M			1.5	42.21	42.79	0.82
	3.33*10-3 M Ca(NO3)2			5	56.14	55.95	0.27
	Dep186 @ 50 mg/l (active content)			10	68.69	67.06	2.31
PXN 46	CuSO4 - 1*10-4 M	9					
	SIBX - 1*10-4 M			1.5	43.37		
	3.33*10-3 M Ca(NO3)2			5	55.76		
	Dep186 @ 50 mg/l (active content)			10	65.42		
PXN 47	CuSO4 - 1*10-4 M	9					
	SIBX - 1*10-4 M			1.5	38.41	37.31	1.56
	3.33*10-3 M Ca(NO3)2			5	51.32	50.48	1.20
	Dep186 @ 200 mg/l (active content)			10	61.49	59.21	3.23
PXN 48	CuSO4 - 1*10-4 M	9					
	SIBX - 1*10-4 M			1.5	36.21		
	3.33*10-3 M Ca(NO3)2			5	49.63		
	Dep186 @ 200 mg/l (active content)			10	56.92		
PXN 49	$CuSO_4$ - $5*10^{-5}$ M	9					
	SIBX - $5*10^{-5}$ M			1.5	56.78	57.06	0.40
	$1*10^{-2}$ M KNO_3			5	79.21	80.33	1.58
				10	88.84	87.91	1.32
PXN 50	$CuSO_4$ - $5*10^{-5}$ M	9					
	SIBX - $5*10^{-5}$ M			1.5	57.34		
	$1*10^{-2}$ M KNO_3			5	81.45		
				10	86.98		

PXN 51	$CuSO_4$ - $5*10^{-5}$ M	9					
	SIBX - $5*10^{-5}$ M			1.5	55.28	56.32	1.47
	$1*10^{-2}$ M KNO_3			5	76.14	75.74	0.57
	APX 4M @ 50 mg/l			10	83.15	83.47	0.45
PXN 52	$CuSO_4$ - $5*10^{-5}$ M	9					
	SIBX - $5*10^{-5}$ M			1.5	57.36		
	$1*10^{-2}$ M KNO_3			5	75.34		
	APX 4M @ 50 mg/l			10	83.79		
PXN 53	$CuSO_4$ - $5*10^{-5}$ M	9					
	SIBX - $5*10^{-5}$ M			1.5	28.45	27.80	0.93
	$1*10^{-2}$ M KNO_3			5	43.78	44.73	1.34
	APX 4M @ 200 mg/l			10	51.92	51.23	0.98
PXN 54	$CuSO_4$ - $5*10^{-5}$ M	9					
	SIBX - $5*10^{-5}$ M			1.5	27.14		
	$1*10^{-2}$ M KNO_3			5	45.68		
	APX 4M @ 200 mg/l			10	50.53		
PXN 55	$CuSO_4$ - $5*10^{-5}$ M	9					
	SIBX - $5*10^{-5}$ M			1.5	61.6	60.72	1.25
	$1*10^{-2}$ M KNO_3			5	76.74	76.04	0.99
	DEP 186 @ 200 mg/l			10	83.23	82.31	1.31
PXN 56	$CuSO_4$ - $5*10^{-5}$ M	9					
	SIBX - $5*10^{-5}$ M			1.5	59.83		
	$1*10^{-2}$ M KNO_3			5	75.34		
	DEP 186 @ 200 mg/l			10	81.38		
PXN 57	SIBX - $5*10^{-5}$ M	9					
	$CuSO_4$ - $5*10^{-5}$ M			1.5	4.32	4.75	0.61
	$1*10^{-3}$ M KNO_3			5	8.83	9.05	0.31
	Reverse reagent addition			10	15.74	15.58	0.23
PXN 58	SIBX - $5*10^{-5}$ M	9					
	$CuSO_4$ - $5*10^{-5}$ M			1.5	5.18		
	$1*10^{-3}$ M KNO_3			5	9.27		
	Reverse reagent addition			10	15.42		
PXN 59	SIBX - $5*10^{-5}$ M	9					
	$CuSO_4$ - $5*10^{-5}$ M			1.5	39.21	40.79	2.23
	$1*10^{-3}$ M KNO_3			5	64.71	63.00	2.42
	APX 4M @ 200 mg/l			10	72.16	72.67	0.72
PXN 60	SIBX - $5*10^{-5}$ M	9					
	$CuSO_4$ - $5*10^{-5}$ M			1.5	42.36		
	$1*10^{-3}$ M KNO_3			5	61.29		
	APX 4M @ 200 mg/l			10	73.18		

PXN 61	SIBX - 5 *10^{-5} M	9					
	$CuSO_4$ - 5*10^{-5} M			1.5	47.61	44.95	3.77
	1*10^{-3} M KNO_3			5	71.52	70.03	2.11
	DEP 186 @ 200 mg/l			10	83.93	81.08	4.03
PXN 62	SIBX - 5 *10^{-5} M	9					
	$CuSO_4$ - 5*10^{-5} M			1.5	42.28		
	1*10^{-3} M KNO_3			5	68.53		
	DEP 186 @ 200 mg/l			10	78.23		
PXN 63	SIBX - 5 *10^{-5} M	9					
	$CuSO_4$ - 5*10^{-5} M			1.5	31.54	30.17	1.94
	3.33*10^{-3} M $Ca(NO_3)_2$			5	52.00	50.23	2.50
	Dep C @ 50 mg/l			10	58.55	55.88	3.78
PXN 64	SIBX - 5 *10^{-5} M	9					
	$CuSO_4$ - 5*10^{-5} M			1.5	28.79		
	3.33*10^{-3} M $Ca(NO_3)_2$			5	48.46		
	Dep C @ 50 mg/l			10	53.21		
PXN 65	SIBX - 5 *10^{-5} M	9		1.5	4.81	5.18	0.52
	1*10^{-3} M KNO3			5	9.24	9.58	0.48
				10	13.50	14.25	1.06
PXN 66	SIBX - 5 *10^{-5} M	9		1.5	5.54		
	1*10^{-3} M KNO3			5	9.92		
				10	15.00		
PXN 67	SIBX - 5 *10^{-5} M	9					
	$CuSO_4$ - 5*10^{-5} M			1.5	28.38	30.26	2.66
	3.33*10^{-3} M $Ca(NO_3)_2$			5	41.30	44.58	4.63
	Dep C @ 100 mg/l			10	48.47	51.35	4.07
PXN 68	SIBX - 5 *10^{-5} M	9					
	$CuSO_4$ - 5*10^{-5} M			1.5	32.14		
	3.33*10^{-3} M $Ca(NO_3)_2$			5	47.85		
	Dep C @ 100 mg/l			10	54.22		
PXN 69	SIBX - 5 *10^{-5} M	9					
	$CuSO_4$ - 5*10^{-5} M			1.5	22.58	23.07	0.69
	3.33*10^{-3} M $Ca(NO_3)_2$			5	32.81	32.57	0.34
	Dep C @ 200 mg/l			10	37.01	36.15	1.22
PXN 70	SIBX - 5 *10^{-5} M	9					
	$CuSO_4$ - 5*10^{-5} M			1.5	23.56		
	3.33*10^{-3} M $Ca(NO_3)_2$			5	32.33		
	Dep C @ 200 mg/l			10	35.28		
	SIBX - 5 *10^{-5} M	4					
	$CuSO_4$ - 5*10^{-5} M			1.5	7.38	6.67	1.00
	1*10^{-3} M KNO3			5	12.78	11.77	1.44
	reverse addition			10	17.00	16.50	0.71
	SIBX - 5 *10^{-5} M	4					
	$CuSO_4$ - 5*10^{-5} M			1.5	5.95		
	1*10^{-3} M KNO3			5	10.75		
	reverse addition			10	15.99		
	$CuSO_4$ - 5*10^{-5} M	9					
	SIBX - 5 *10-5 M			10	91.83	90.71	1.59

$1*10^{-3}$ M KNO3						
$CuSO_4$ - $5*10^{-5}$ M SIBX - 5 *10-5 M $1*10^{-3}$ M KNO3	9		10	89.58		
$CuSO_4$ - $5*10^{-5}$ M SIBX - 5 *10-5 M $3.33*10^{-3}$ M $Ca(NO_3)_2$	9		10	55.86	57.83	2.79
$CuSO_4$ - $5*10^{-5}$ M SIBX - 5 *10-5 M $3.33*10^{-3}$ M $Ca(NO_3)_2$	9		10	59.80		
$CuSO_4$ - $5*10^{-5}$ M SIBX - 5 *10-5 M $3.33*10^{-3}$ M $Ca(NO_3)_2$ APX4M @ 50 mg/l	9		10	11.80	10.85	1.35
$CuSO_4$ - $5*10^{-5}$ M SIBX - 5 *10-5 M $3.33*10^{-3}$ M $Ca(NO_3)_2$ APX4M @ 50 mg/l	9		10	9.90		

Run	Reagents added	pH	Total Mass	Time (min)	Recovery (%)	Mean Recovery (%)	Standard Deviation
FSP 1	10-3 M KNO3	9		2	1.32	1.61	0.41
				5	3.89	3.72	0.25
				10	5.24	5.31	0.11
				20	8.87	9.47	0.85
FSP 2	10-3 M KNO3	9		2	1.90		
				5	3.54		
				10	5.39		
				20	10.08		
FSP 3	$CuSO_4$ - $1*10^{-4}$ M	9		1.5	46.16		
	SIBX - $1*10^{-4}$ M			5	74.28		
	10-3 M KNO3			10	88.16		
FSP 4	$CuSO_4$ - $1*10^{-4}$ M	9		1.5	40.71	43.70	2.76
	SIBX - $1*10^{-4}$ M			5	75.67	76.26	2.32
	10-3 M KNO3			10	89.72	89.06	0.80
FSP 5	$CuSO_4$ - $1*10^{-4}$ M	9		1.5	44.23		
	SIBX - $1*10^{-4}$ M			5	78.82		
	10-3 M KNO3			10	89.31		
FSP 6	$CuSO_4$ - $5*10^{-5}$ M	9		1.5	4.02	3.76	0.35
	SIBX - $5*10^{-5}$ M			5	6.72	6.40	0.45
	10-3 M KNO3			10	8.83	8.42	0.58
	Reverse addition						
FSP 7	$CuSO_4$ - $1*10^{-4}$ M	9		1.5	3.51		
	SIBX - $1*10^{-4}$ M			5	6.08		
	10-3 M KNO3			10	8.01		
	Reverse addition						
FSP 8	$CuSO_4$ - $1*10^{-4}$ M	9		1.5	44.28	44.43	0.21
	SIBX - $1*10^{-4}$ M			5	77.27	77.58	0.43
	10-3 M KNO3			10	83.32	83.54	0.30
	APX 4M @ 50 mg/l						
FSP 9	$CuSO_4$ - $1*10^{-4}$ M	9		1.5	44.57		
	SIBX - $1*10^{-4}$ M			5	77.88		
	10-3 M KNO3			10	83.75		
	APX 4M @ 50 mg/l						
FSP 10	$CuSO_4$ - $1*10^{-4}$ M	9		1.5	40.14	41.55	2.00
	SIBX - $1*10^{-4}$ M			5	71.43	72.65	1.73
	10-3 M KNO3			10	79.54	80.27	1.03
	APX 4M @ 100 mg/l						
FSP 11	$CuSO_4$ - $1*10^{-4}$ M	9		1.5	42.96		
	SIBX - $1*10^{-4}$ M			5	73.87		
	10-3 M KNO3			10	81.00		
	APX 4M @ 100 mg/l						

FSP 12	$CuSO_4$ - $1*10^{-4}$ M	9		1.5	39.14	39.72	0.82
	SIBX - $1*10^{-4}$ M			5	65.09	67.86	3.92
	10-3 M KNO3			10	72.65	74.12	2.09
	APX 4M @ 200 mg/l						
FSP 13	$CuSO_4$ - $1*10^{-4}$ M	9		1.5	40.30		
	SIBX - $1*10^{-4}$ M			5	70.63		
	10-3 M KNO3			10	75.60		
	APX 4M @ 200 mg/l						
FSP 14	$CuSO_4$ - $1*10^{-4}$ M	9		1.5	43.21	42.93	0.40
	SIBX - $1*10^{-4}$ M			5	77.11	75.92	1.69
	10-3 M KNO3			10	85.84	84.22	2.29
	Dep 186 @ 50 mg/l						
FSP 15	$CuSO_4$ - $1*10^{-4}$ M	9		1.5	42.65		
	SIBX - $1*10^{-4}$ M			5	74.72		
	10-3 M KNO3			10	82.60		
	Dep 186 @ 50 mg/l						
FSP 16	$CuSO_4$ - $1*10^{-4}$ M	9		1.5	42.26	43.66	1.97
	SIBX - $1*10^{-4}$ M			5	72.90	74.11	1.71
	10-3 M KNO3			10	80.71	81.48	1.09
	Dep 186 @ 100 mg/l						
FSP 17	$CuSO_4$ - $1*10^{-4}$ M	9		1.5	45.05		
	SIBX - $1*10^{-4}$ M			5	75.32		
	10-3 M KNO3			10	82.25		
	Dep 186 @ 100 mg/l						
FSP 18	$CuSO_4$ - $1*10^{-4}$ M	9		1.5	41.12	42.31	1.68
	SIBX - $1*10^{-4}$ M			5	68.46	70.47	2.84
	10-3 M KNO3			10	75.90	77.18	1.82
	Dep 186 @ 200 mg/l						
FSP 19	$CuSO_4$ - $1*10^{-4}$ M	9		1.5	43.50		
	SIBX - $1*10^{-4}$ M			5	72.47		
	10-3 M KNO3			10	78.47		
	Dep 186 @ 200 mg/l						
FSP 20	$CuSO_4$ - $1*10^{-4}$ M	9		1.5	28.95	27.29	2.34
	SIBX - $1*10^{-4}$ M			5	48.84	47.40	2.04
	$Ca(NO_3)_2$ @ $3.33*10^{-3}$ M			10	54.30	53.29	1.43
FSP 21	$CuSO_4$ - $1*10^{-4}$ M	9		1.5	25.64		
	SIBX - $1*10^{-4}$ M			5	45.96		
	$Ca(NO_3)_2$ @ $3.33*10^{-3}$ M			10	52.27		
FSP 22	$CuSO_4$ - $1*10^{-4}$ M	9		1.5	2.58	2.43	0.22
	SIBX - $1*10^{-4}$ M			5	4.25	4.07	0.26
	$Ca(NO_3)_2$ @ $3.33*10^{-3}$ M			10	5.31	5.21	0.14
	APX 4M @ 50 mg/l						
FSP 23	$CuSO_4$ - $1*10^{-4}$ M	9		1.5	2.27		
	SIBX - $1*10^{-4}$ M			5	3.68		
	$Ca(NO_3)_2$ @ $3.33*10^{-3}$ M			10	5.11		
	APX 4M @ 50 mg/l						

FSP 24	$CuSO_4$ - $1*10^{-4}$ M	9		1.5	2.34	2.15	0.27
	SIBX - $1*10^{-4}$ M			5	3.85	3.66	0.27
	$Ca(NO_3)_2$ @ $3.33*10^{-3}$ M			10	5.24	4.98	0.37
	APX 4M @ 100 mg/l						
FSP 25	$CuSO_4$ - $1*10^{-4}$ M	9		1.5	1.96		
	SIBX - $1*10^{-4}$ M			5	3.47		
	$Ca(NO_3)_2$ @ $3.33*10^{-3}$ M			10	4.71		
	APX 4M @ 100 mg/l						
FSP 26	$CuSO_4$ - $1*10^{-4}$ M	9		1.5	28.61	27.35	1.79
	SIBX - $1*10^{-4}$ M			5	43.20	42.73	0.66
	$Ca(NO_3)_2$ @ $3.33*10^{-3}$ M			10	51.01	48.91	2.97
	Dep 186 @ 50 mg/l						
FSP 27	$CuSO_4$ - $1*10^{-4}$ M	9		1.5	26.09		
	SIBX - $1*10^{-4}$ M			5	42.26		
	$Ca(NO_3)_2$ @ $3.33*10^{-3}$ M			10	46.81		
	Dep 186 @ 50 mg/l						
FSP 28	$CuSO_4$ - $1*10^{-4}$ M	9		1.5	26.66	27.88	1.72
	SIBX - $1*10^{-4}$ M			5	43.62	43.49	0.19
	$Ca(NO_3)_2$ @ $3.33*10^{-3}$ M			10	48.48	49.14	0.94
	Dep 186 @ 100 mg/l						
FSP 29	$CuSO_4$ - $1*10^{-4}$ M	9		1.5	29.10		
	SIBX - $1*10^{-4}$ M			5	43.36		
	$Ca(NO_3)_2$ @ $3.33*10^{-3}$ M			10	49.81		
	Dep 186 @ 100 mg/l						
FSP 30	$CuSO_4$ - $1*10^{-4}$ M	9		1.5	24.06	25.81	2.46
	SIBX - $1*10^{-4}$ M			5	37.86	39.09	1.73
	$Ca(NO_3)_2$ @ $3.33*10^{-3}$ M			10	43.38	44.92	2.18
	Dep 186 @ 200 mg/l						
FSP 31	$CuSO_4$ - $1*10^{-4}$ M	9		1.5	27.55		
	SIBX - $1*10^{-4}$ M			5	40.32		
	$Ca(NO_3)_2$ @ $3.33*10^{-3}$ M			10	46.46		
	Dep 186 @ 200 mg/l						
FSP 32	$CuSO_4$ - $1*10^{-4}$ M	9		1.5	29.36	28.20	1.63
	SIBX - $1*10^{-4}$ M			5	47.96	46.79	1.65
	$Ca(NO_3)_2$ @ $3.33*10^{-3}$ M			10	55.17	54.48	0.98
	Dep C @ 50 mg/l						
FSP 33	$CuSO_4$ - $1*10^{-4}$ M	9		1.5	27.05		
	SIBX - $1*10^{-4}$ M			5	45.62		
	$Ca(NO_3)_2$ @ $3.33*10^{-3}$ M			10	53.78		
	Dep C @ 50 mg/l						
FSP 34	$CuSO_4$ - $1*10^{-4}$ M	9		1.5	17.78	16.72	1.49
	SIBX - $1*10^{-4}$ M			5	25.06	24.07	1.41
	$Ca(NO_3)_2$ @ $3.33*10^{-3}$ M			10	29.69	29.75	0.08
	Dep C @ 200 mg/l						
FSP 35	$CuSO_4$ - $1*10^{-4}$ M	9		1.5	15.66		
	SIBX - $1*10^{-4}$ M			5	23.07		
	$Ca(NO_3)_2$ @ $3.33*10^{-3}$ M			10	29.81		
	Dep C @ 200 mg/l						

APPENDIX D: EXPERIMENTAL PROCEDURE OF UG-2 BATCH FLOTATION

The following details and procedures were followed in the flotation process.

The experiments were performed using a 3 litre Leeds Batch Cell. For every run, 1 kg of ore was used. The depressant type, dosage together with the $CuSO_4$ dosages were changed during the each float depending on what was investigated. The frother and collector dosages were kept constant.

1. Operating Conditions.

Ore	Upper Ground 2 (UG-2)
Air flowrate	6 litres/minute
Impeller Speed	1200 rev/minute
pH	9
Frother dosage	40 µl (DOW 200)
Collector dosage	30 g/ton SIBX
	30 µl DTP
Temperature	Room temperature

2. Determined parameters.

- The mass concentrate collected
- The water recovered in the concentrate
- The change in pH during flotation
- The chromite grade in the concentrate
- The amount of chromite recovery

3. Flotation procedure.

- Mill 1-kg ore for 6.5 minutes.
- Add the milled ore into the flotation cell (the impeller is turned on while the ore is added) and add water to make up the volume.
- Take the feed sample- about 30 ml.
- Measure the pH and record. Then adjust to pH 9 using either HCl or NaOH depending on the pH of the ore.
- 0 min: Add the required amount of $CuSO_4$ dosage and condition.
- 5 min: Add the collectors.
- 8 min: Add the required amount and type of depressant required.
- 9 min: Add the frother and condition.
- 10 min: Turn the air on.
- 10.15 – 11.15: Collect concentrate 1.
- 11.15 – 13.15: Collect concentrate 2.
- 13.15 – 17.15: Collect concentrate 3.
- 17.15 – 25.15: Collect concentrate 4.
- Turn air off.
- Measure and record the final pH.

The concentrate is scraped every 15 seconds. The different concentrates are dried and weighed. When dry they are taken for the chromite analysis.

APPENDIX E: SUMMARY OF THE RESULTS OF UG-2 BATCH FLOTATION TESTS

0 g/ton $CuSO_4$
100 g/ton IMP4

	conc(g)	water(g)	Cr grade	Cr rec
C1	3.46	76.43	3.31	0.08
C2	2.17	54.05	2.28	0.04
C3	3.47	62.91	1.96	0.05
C4	6.36	86.10	1.82	0.08

	conc(g)	water(g)	Cr grade	Cr rec
C1	2.19	44.05	2.53	0.04
C2	2.55	53.01	2.18	0.04
C3	4.59	74.72	2.29	0.07
C4	6.11	71.47	1.98	0.08

90 g/ton $CuSO_4$
100 g/ton IMP4

	conc(g)	water(g)	Cr grade (%)	Cr rec
Conc 1	1.57	18.74	2.41	0.03
Conc 2	3.92	50.13	2.54	0.08
Conc 3	3.46	39.76	2.42	0.06
Conc 4	6.62	67.34	2.22	0.11

	conc(g)	water(g)	Cr grade (%)	Cr rec
Conc 1	1.83	21.17	2.53	0.03
Conc 2	3.86	49.38	2.51	0.07
Conc 3	5.06	62.85	2.45	0.09
Conc 4	5.59	50.30	2.44	0.09

30 g/ton $CuSO_4$
100 g/ton IMP4

	conc(g)	water(g)	Cr grade	Cr rec
C1	1.170	13.680	1.708	0.009
C2	3.120	51.090	2.511	0.033
C3	5.430	85.430	3.339	0.077
C4	7.750	120.380	2.211	0.073

	conc(g)	water(g)	Cr grade	Cr rec
C1	2.220	34.850	3.767	0.057
C2	3.180	60.430	2.333	0.051
C3	4.170	65.780	3.847	0.110
C4	9.310	126.420	3.436	0.219

90 g/ton $CuSO_4$
200 g/ton IMP4

	conc(g)	water(g)	Cr grade	Cr rec
C1	1.370	24.930	3.754	0.023
C2	1.970	49.000	4.797	0.042
C3	3.680	67.480	3.851	0.063
C4	3.000	65.970	4.514	0.060

	conc(g)	water(g)	Cr grade	Cr rec
C1	2.000	46.190	6.084	0.054
C2	2.320	64.250	5.684	0.058
C3	2.950	66.300	4.364	0.057
C4	3.920	72.620	4.103	0.071

90 g/ton $CuSO_4$
100 g/ton KU5

	conc(g)	water(g)	Cr grade	Cr rec
C1	5.040	49.320	3.198	0.088
C2	6.770	71.000	3.075	0.114
C3	7.460	16.140	3.238	0.132
C4	6.740	71.660	3.493	0.128

	conc(g)	water(g)	Cr grade	Cr rec
C1	3.910	31.840	2.362	0.041
C2	6.770	61.940	2.721	0.082
C3	7.640	70.550	3.018	0.102
C4	6.070	60.330	3.340	0.090

30 g/ton $CuSO_4$
100 g/ton KU5

	conc(g)	water(g)	Cr grade	Cr rec
C1	4.070	48.920	2.876	0.053
C2	6.530	86.350	2.838	0.084
C3	7.220	85.820	3.041	0.099
C4	6.560	72.700	2.936	0.087

	conc(g)	water(g)	Cr grade	Cr rec
C1	3.000	30.380	2.606	0.039
C2	5.480	64.700	2.552	0.071
C3	8.330	95.720	2.624	0.110
C4	7.610	66.930	2.914	0.112

90 g/ton $CuSO_4$
200 g/ton KU5

	conc(g)	water(g)	Cr grade	Cr rec
C1	1.040	20.140	3.218	0.015
C2	2.300	53.570	23.047	0.235
C3	4.140	62.850	3.868	0.071
C4	5.810	54.780	3.049	0.079

	conc(g)	water(g)	Cr grade	Cr rec
C1	3.410	19.510	3.457	0.049
C2	3.390	42.820	5.004	0.071
C3	2.960	62.480	4.078	0.051
C4	3.330	47.900	3.352	0.047

30 g/ton $CuSO_4$
200 g/ton IMP4

	conc(g)	water(g)	Cr grade	Cr rec
C1	2.130	47.700	5.876	0.062
C2	1.740	69.390	5.538	0.048
C3	3.410	99.520	4.602	0.078
C4	3.490	86.080	3.638	0.063

	conc(g)	water(g)	Cr grade	Cr rec
C1	1.740	49.400	5.520	0.048
C2	1.690	64.110	5.444	0.046
C3	2.350	69.530	4.150	0.049
C4	3.610	72.870	3.481	0.063

30 g/ton $CuSO_4$
200 g/ton KU5

	conc(g)	water(g)	Cr grade	Cr rec
C1	0.710	14.350	2.868	0.008
C2	1.670	48.830	6.083	0.041
C3	2.520	70.920	5.120	0.052
C4	5.510	70.060	3.172	0.071

	conc(g)	water(g)	Cr grade	Cr rec
C1	1.580	46.670	5.834	0.043
C2	1.360	60.860	6.763	0.043
C3	2.100	75.040	4.885	0.048
C4	6.450	98.550	2.937	0.089

30 g/ton $CuSO_4$
100 g/ton KU5

	conc(g)	water(g)	Cr grade	Cr rec
C1	4.070	48.920	2.876	0.053
C2	6.530	86.350	2.838	0.084
C3	7.220	85.820	3.041	0.099
C4	6.560	72.700	2.936	0.087

	conc(g)	water(g)	Cr grade	Cr rec
C1	3.000	30.380	2.606	0.039
C2	5.480	64.700	2.552	0.071
C3	8.330	95.720	2.624	0.110
C4	7.610	66.930	2.914	0.112

90 g/ton $CuSO_4$
200 g/ton KU5

	conc(g)	water(g)	Cr grade	Cr rec
C1	1.040	20.140	3.218	0.015
C2	2.300	53.570	23.047	0.235
C3	4.140	62.850	3.868	0.071
C4	5.810	54.780	3.049	0.079

	conc(g)	water(g)	Cr grade	Cr rec
C1	3.410	19.510	3.457	0.049
C2	3.390	42.820	5.004	0.071
C3	2.960	62.480	4.078	0.051
C4	3.330	47.900	3.352	0.047

30 g/ton $CuSO_4$
200 g/ton IMP4

	conc(g)	water(g)	Cr grade	Cr rec
C1	2.130	47.700	5.876	0.062
C2	1.740	69.390	5.538	0.048
C3	3.410	99.520	4.602	0.078
C4	3.490	86.080	3.638	0.063

	conc(g)	water(g)	Cr grade	Cr rec
C1	1.740	49.400	5.520	0.048
C2	1.690	64.110	5.444	0.046
C3	2.350	69.530	4.150	0.049
C4	3.610	72.870	3.481	0.063

30 g/ton $CuSO_4$
200 g/ton KU5

	conc(g)	water(g)	Cr grade	Cr rec
C1	0.710	14.350	2.868	0.008
C2	1.670	48.830	6.083	0.041
C3	2.520	70.920	5.120	0.052
C4	5.510	70.060	3.172	0.071

	conc(g)	water(g)	Cr grade	Cr rec
C1	1.580	46.670	5.834	0.043
C2	1.360	60.860	6.763	0.043
C3	2.100	75.040	4.885	0.048
C4	6.450	98.550	2.937	0.089

APPENDIX F: STATISTICAL ANALYSIS OF BATCH FLOTATION RESULTS (ANOVA)

Analysis of Variance (ANOVA)

The analysis of variance is used to determine whether several means are significantly different to one another or as a group by using the F-test to assess the significance of the variance due to the different means.

The variance is expressed by using the sum of squares. This sum of squares can be either between samples (S_1) or within samples (S_0). Both sum of squares will be performed. The following general equations will be considered:

$$S_1 = n\sum_{i=1}^{k}\left(\overline{x_i} - \overline{\overline{x}}\right)^2$$

$$S_0 = \sum_{i=1}^{k}\sum_{j=1}^{n}\left(x_{ij} - \overline{\overline{x}}\right)^2$$

$\overline{\overline{x}}$ = Overall data mean
$\overline{x_1}$ = mean of the first system
$\overline{x_2}$ = mean of the second system

$Df_{treatment}$ = number of treatments – 1 = 1
Df_{total} = total number of results –1 = 3
$Df_{error} = Df_{total} - Df_{treatment} = 2$

All the experiments were performed in duplicates and the degrees of freedom will remain the same for each experiment.

$$F = \frac{MS_{treatment}}{MS_{error}}$$

1. ANOVA for the Systems of Varying Copper Sulphate Dosages

1.1. Comparing the System at $CuSO_4$ of 90 g/t with the one without $CuSO_4$

Run	Cu – 90	Cu – 0
1	0.2799	0.220
2	0.2799	0.210

$\bar{x} = 0.2475$

$\bar{x_1} = 0.2150$

$\bar{x_2} = 0.2799$

$S_1 = 0.0042$

$S_2 = 0.0001$

Source of Variation	Sum of Squares	Degree of Freedom	Mean Square	F
Between Samples	0.0042	1	0.0042	168.6
Within Samples	0.0001	2	0.00005	-
Total	0.0043	3	-	-

1.2. Comparing the System at $CuSO_4$ of 30 g/t with the one without $CuSO_4$

Run	Cu-0	Cu-30
1	0.220	0.205
2	0.210	0.354

$\bar{x} = 0.2471$

$\bar{x_1} = 0.2150$

$\bar{x_2} = 0.2792$

$S_1 = 0.0041$

$S_2 = 0.0111$

Source of Variation	Sum of Squares	Degree of Freedom	Mean Square	F
Between Samples	0.0041	1	0.0041	0.743
Within Samples	0.0111	2	0.00555	
Total	0.0152	3	-	

2. ANOVA for the Systems of Varying Depressant Dosage

2.1. Comparing the System at IMP4 of 100 g/t with the one at IMP4 at 200 g/ton at $CuSO_4$ dosage of 30 g/ton

Run	IMP4 – 100	IMP4 – 200
1	0.205	0.251
2	0.354	0.205

$\bar{x} = 0.2537$

$\overline{x_1} = 0.2792$

$\overline{x_2} = 0.2282$

$S_1 = 0.0026$

$S_2 = 0.0121$

Source of Variation	Sum of Squares	Degree of Freedom	Mean Square	F
Between Samples	0.0026	1	0.0026	0.4291
Within Samples	0.0121	2	0.0061	
Total	0.0147	3	-	

2.2. Comparing the System of IMP4 at 100 g/t with the one with IMP4 of 200 g/t at $CuSO_4$ dosage of 90 g/t

Run	IMP4 – 100	IMP4 – 200
1	0.280	0.187
2	0.280	0.240

$\bar{x} = 0.2467$

$\overline{x_1} = 0.2799$

$\overline{x_2} = 0.2135$

$S_1 = 0.0044$

$S_2 = 0.0014$

Source of Variation	Sum of Squares	Degree of Freedom	Mean Square	F
Between Samples	0.0044	1	0.0044	6.39
Within Samples	0.0014	2	0.0007	-
Total	0.0058	3	-	-

2.3. Comparing the System at IMP4 at 100 g/t and no $CuSO_4$, with the system of IMP4 at 200 g/t and $CuSO_4$ at 90 g/t.

Run	IMP4 – 100	IMP4 – 200
1	0.220	0.187
2	0.210	0.240

$\bar{x} = 0.2143$

$\bar{x}_1 = 0.215$

$\bar{x}_2 = 0.2135$

$S_1 = 0.0000021$

$S_2 = 0.0014$

Source of Variation	Sum of Squares	Degree of Freedom	Mean Square	F
Between Samples	0.0000021	1	0.0000021	0.0030
Within Samples	0.0014	2	0.0007	
Total	0.0014021	3	-	

2.4. Comparing the System with KU5 at 100 g/t and the system with KU5 at 200 g/t with $CuSO_4$ at 30 g/t

Run	KU5 – 100	KU5 – 200
1	0.324	0.172
2	0.333	0.225

$\bar{x} = 0.2633$

$\bar{x}_1 = 0.3282$

$\bar{x}_2 = 0.1984$

$S_1 = 0.0168$

$S_2 = 0.0014$

Source of Variation	Sum of Squares	Degree of Freedom	Mean Square	F
Between Samples	0.0168	1	0.0168	23.72
Within Samples	0.0014	2	0.0007	-
Total	0.0182	3	-	-

2.5. Comparing the System at IMP4 at 100 g/t and no $CuSO_4$, with the system of KU5 at 200 g/t and $CuSO_4$ at 90 g/t.

Run	IMP4 – 100	KU5 – 200
1	0.220	0.189
2	0.210	0.186

$\bar{x} = 0.2011$

$\overline{x_1} = 0.215$

$\overline{x_2} = 0.1873$

$S_1 = 0.0008$

$S_2 = 0.0001$

Source of Variation	Sum of Squares	Degree of Freedom	Mean Square	F
Between Samples	0.0008	1	0.0008	27.76
Within Samples	0.0001	2	0.00005	-
Total	0.0009	3	-	-

APPENDIX G: DETERMINATON OF CHROME BY ATOMIC ANALYSIS PROCEDURE

Determination of Chrome by Atomic Analysis Procedure.

The dried samples from flotation are then analysed for the chrome content using a technique called Atomic Absorption. These analysis were performed in the Main Laboratory in the Chemical Engineering Department at UCT. The zirconium crucibles were used for the experimental part. The following procedure was followed:

- Weigh out ± 0.5 g of the concentrate into the zirconium crucible.
- Add ± 5 g of sodium peroxide granules (Na_2O_2) into the crucible and mix thoroughly using a glass rod.
- Fuse the mixture over an LP gas flame, first by using a low flame until the mixture is a black slurry. Then increase the flame by increasing the LP gas flow until the mixture is molten red. Ensure that all the solids are molten.
- Allow the melt to cool at room temperature and immerse the crucible into 100 ml of de-ionised water in a 250-ml beaker. Allow the fuse to leach into the water until the reaction is complete.
- Add 1-ml hydrogen peroxide (H_2O_2) and wait for the reaction to go to completion.
- Add 30-ml concentrated hydrochloric acid (HCl) and wait for the reaction to go to completion.
- Rinse the crucible with de-ionised water and remove it from the beaker using a glass rod.
- Pour the solution into the 250-ml volumetric flask and make up to volume using de-ionised water.
- Mix the volumetric flask thoroughly and add some of the solution into the sample bottles for chrome analysis.

The Atomic Absorption (AA) machine reports the chrome results using mass of chrome per volume of the solution (i.e. mg/l). These results are then used to calculate the percentage of chrome in the sample and hence the chromite content in each concentrate. The following formula was used to calculate the percentage of chrome in the concentrate:

$$\text{Chrome \%} = \frac{\text{AA reading (mg/l)}}{40 * \text{mass of sample (g)}}$$

The chrome results are reported in Appendix G of this report.

APPENDIX H: HAZARDS INVOLVED IN MILLING, FILTERING AND SODIUMPEROXIDE USAGE

MILLING.

During milling care must be taken since the equipment used for milling can be dangerous. A brief note of the following equipment is considered:

- Milling rods and lid: These are heavy and must be put in a safe place to avoid accidental falling. As a prevention measure, closed shoes must be worn at all times.

- Noise: during milling high levels of noise are emitted as the milling rods rotate in the mill. These levels of noise can lead to ear damage and noise protection equipment provided should be used when milling.

FILTERING.

- The filter press used in the Mineral Processing Laboratory is a high-pressure vessel and over-pressuring could result in the vessel rupturing or malfunctioning. When the pressure is applied to effect filtration of the pulp, it must be released after the cake is formed.

SODIUM PEROXIDE.

- The sodium peroxide used in the experiments is in granular form and it is a very unstable chemical. It is a very strong irritant to the skin, eyes and mucous membranes. Materials to be avoided for contact with Na_2O_2 are combustible materials, organic matter and acetic acid. It will react with water or steam to produce heat and toxic fumes. In the case of fire use CO_2 or dry chemical powder fire extinguishers. Combustible materials ignited by Na_2O_2 should be smothered with soda ash.

APPENDIX I : ATOMIC ABSORPTION RESULTS FOR CHROME CONTENT

Batch 1	Name	AA reading(mg/l)	Mass (g)	Cr grade(%)
1	A1C1	20.75	0.3814	1.360
2	A1C2	82.74	0.5112	4.046
3	A1C3	61.23	0.5046	3.034
4	A1C4	54.66	0.5159	2.649
5	A1Feed	243.06	0.5157	11.783
6	A1Tails	348	0.4946	17.590
7	A2C1	47.64	0.4951	2.406
8	A2C2	51.87	0.5099	2.543
9	A2C3	47.64	0.4923	2.419
10	A2C4	43.77	0.4923	2.223
11	A2Feed	269.11	0.5113	13.158
12	A2Tails	340.55	0.4895	17.393
13	A3C1	50.67	0.5011	2.528
14	A3C2	49.24	0.4896	2.514
15	A3C3	48.81	0.4976	2.452
16	A3C4	48.4	0.4963	2.438
17	A3Feed	283.75	0.4919	14.421

Key: A1C1 = Run A1 Concentrate 1

Batch 2	Name	AA reading(mg/l)	Mass (g)	Cr grade(%)
1	A3Tails	360	0.4996	18.014
2	B1C1	66.98	0.5059	3.310
3	B1C2	46.14	0.5062	2.279
4	B1C3	38.42	0.4905	1.958
5	B1C4	36.78	0.5062	1.816
6	B1Feed	275	0.4897	14.039
7	B1Tails	345	0.4905	17.584
8	B2C1	49.98	0.4933	2.533
9	B2C2	44.41	0.5087	2.183
10	B2C3	46.95	0.5131	2.288
11	B2C4	40.37	0.5089	1.983
12	B2Feed	287	0.4903	14.634
13	B2Tails	334.4	0.5017	16.663
14	C1C1	33.54	0.4908	1.708
15	C1C2	50.61	0.5038	2.511
16	C2C2	46.09	0.4938	2.333
17	C1C4	44.6	0.5044	2.211

Batch 3	Name	AA reading(mg/l)	Mass (g)	Cr grade(%)
1	C1C3	65.53	0.4907	3.339
2	C1Feed	476.33	0.5073	23.474
3	C1Tails	507.83	0.4923	25.789
4	C2C1	75.69	0.5023	3.767
5	C2C3	75.87	0.493	3.847
6	C2C4	67.78	0.4932	3.436
7	C2feed	408.77	0.5092	20.069
8	C2Tails	506.45	0.5067	24.988
9	D1C1	74.65	0.4971	3.754
10	D1C2	94.81	0.4941	4.797
11	D1C3	77.23	0.5014	3.851
12	D1C4	90.74	0.5026	4.514
13	D1Feed	450.2	0.4983	22.587
14	D1Tails	483.87	0.5058	23.916
15	D2C1	122.85	0.5048	6.084
16	D2C2	114.45	0.5034	5.684
17	D2C3	84.47	0.4839	4.364

Batch 4	Name	AA reading(mg/l)	Mass (g)	Cr grade(%)
1	D2C4	81.16	0.4945	4.103
2	D2Feed	453.48	0.5005	22.651
3	D2Tails	450.09	0.5064	22.220
4	E1C1	115.19	0.4901	5.876
5	E1C2	109.52	0.4944	5.538
6	E1C3	91.09	0.4948	4.602
7	E1C4	71.66	0.4924	3.638
8	E1Feed	400.8	0.4981	20.116
9	E1Tails	510.57	0.5034	25.356
10	E2C1	108.69	0.4923	5.520
11	E2C2	110.64	0.5081	5.444
12	E2C3	82.94	0.4996	4.150
13	E2C4	68.76	0.4938	3.481
14	E2Feed	402.27	0.5018	20.041
15	E2Tails	445.76	0.5008	22.252
16	F1C1	63.83	0.499	3.198

Batch 5	Name	AA reading(mg/l)	Mass (g)	Cr grade(%)
1	F1C2	62	0.5041	3.075
2	F1C3	64.66	0.4992	3.238
3	F1C4	69.31	0.4961	3.493
4	F1Feed	370.98	0.5059	18.333
5	F1Tails	428.47	0.5034	21.279
6	F2C1	47.23	0.4999	2.362
7	F2C2	54.71	0.5026	2.721
8	F2C3	61.32	0.5079	3.018
9	F2C4	65.95	0.4936	3.340
10	F2Feed	446.33	0.4953	22.528
11	F2Tails	446.43	0.4972	22.447
12	G1C1	57.26	0.4978	2.876
13	G1C2	56.27	0.4957	2.838
14	G1C3	61.58	0.5063	3.041
15	G1C4	58.47	0.4979	2.936
16	G1Feed	443.47	0.5024	22.068

Batch 6	Name	AA reading(mg/l)	Mass (g)	Cr grade(%)
1	G1Tails	446.06	0.4989	22.352
2	G2C1	52.62	0.5048	2.606
3	G2C2	51.51	0.5046	2.552
4	G2C3	52.39	0.4991	2.624
5	G2C4	57.6	0.4942	2.914
6	G2Feed	397.54	0.5021	19.794
7	G2Tails	85.54	0.4991	4.285
8	H1C1	64.54	0.5014	3.218
9	H1C2	460.76	0.4998	23.047
10	H1C3	75.58	0.4885	3.868
11	H1C4	61.02	0.5003	3.049
12	H1Feed	415.93	0.4995	20.817
13	H1Tails	456.7	0.5072	22.511
14	H2C1	68.12	0.4926	3.457
15	H2C2	101.79	0.5085	5.004
16	H2C3	81.78	0.5014	4.078

Batch 7	Name	AA reading(mg/l)	Mass (g)	Cr grade(%)
1	H2C4	67.34	0.5022	3.352
2	H2Feed	480.12	0.503	23.863
3	H2Tails	479.29	0.5002	23.955
4	I1C1	44.02	0.3837	2.868
5	I1C2	121.57	0.4996	6.083
6	I1C3	102.79	0.5019	5.120
7	I1C4	62.91	0.4959	3.172
8	I1Feed	490.97	0.4964	24.727
9	I1Tails	486.05	0.4989	24.356
10	I2C1	115.04	0.493	5.834
11	I2C2	133.97	0.4952	6.763
12	I2C3	97.74	0.5002	4.885
13	I2C4	59.85	0.5094	2.937
14	I2Feed	423.56	0.4997	21.191
15	I2Tails	447.45	0.4964	22.535

APPENDIX J: RAW DATA FROM THE UG-2 BATCH FLOTATION EXPERIMENTS

FLOAT 1.

Run A1 **Standard Conditions** **90 g/ton CuSO4** **100 g/ton IMP4**

Mass	Conc 1	Conc 2	Conc 3	Conc 4	Conc 5	Feed	Tails
Conc+ paper	3.59	7.43	7.14	7.24	9.29	4.51	
Paper	3.16	3.18	3.12	2.97	3.04	3.51	8.22
Conc	0.43	4.25	4.02	4.27	6.25	1	
Cr Grade(%)	1.36	4.046	3.034	2.649		11.783	17.59
B + H2O	538.29	546.89	550.22	531.75	543.37		
Bottle	454.51	461.28	395.52	293.25	178.01		
H2O	83.78	85.61	154.7	238.5	365.36		
D + C + H2O	453.23	498.75	584.51	657.79	795.18		
Dish	365.15	361.36	354.64	368.08	382.42		
C + H2O	88.08	137.39	229.87	289.71	412.76		
H2O recovery	3.87	47.53	71.15	46.94	41.15		

Key: **Conc = concentrate** **Cr = chromite**
B = bottle **D = dish**

FLOAT 2.

Run A2

Mass	Conc 1	Conc 2	Conc 3	Conc 4	Conc 5	Feed	Tails
C + paper	5.26	6.99	6.81	7.3	5.96	5.86	
Paper	3.69	3.07	3.35	3.32	3.32	3.31	
Conc	1.57	3.92	3.46	3.98	2.64	2.55	
Cr Grade(%)	2.41	2.54	2.42	2.22		13.16	17.39
B + H2O	549.54	533.61	536.53	530.58	551.79		
Bottle	517.98	444.67	374.15	333.94	213.41		
H2O	31.56	88.94	162.38	196.64	338.38		
D + C + H2O	417.02	504.35	560.24	611.4	748.08		
Dish	365.15	361.36	354.64	368.08	382.42		
C + H2O	51.87	142.99	205.6	243.32	365.66		
H2O recovery	18.74	50.13	39.76	42.7	24.64		

FLOAT 1.

Run A1 Standard Conditions — 90 g/ton $CuSO_4$ — 100 g/ton IMP4

Mass	Conc 1	Conc 2	Conc 3	Conc 4	Conc 5	Feed	Tails
Conc+ paper	3.59	7.43	7.14	7.24	9.29	4.51	
Paper	3.16	3.18	3.12	2.97	3.04	3.51	8.22
Conc	0.43	4.25	4.02	4.27	6.25	1	
Cr Grade(%)	1.36	4.046	3.034	2.649		11.783	17.59
B + H2O	538.29	546.89	550.22	531.75	543.37		
Bottle	454.51	461.28	395.52	293.25	178.01		
H2O	83.78	85.61	154.7	238.5	365.36		
D + C + H2O	453.23	498.75	584.51	657.79	795.18		
Dish	365.15	361.36	354.64	368.08	382.42		
C + H2O	88.08	137.39	229.87	289.71	412.76		
H2O recovery	3.87	47.53	71.15	46.94	41.15		

Key: Conc = concentrate; Cr = chromite; B = bottle; D = dish

FLOAT 2.

Run A2

Mass	Conc 1	Conc 2	Conc 3	Conc 4	Conc 5	Feed	Tails
C + paper	5.26	6.99	6.81	7.3	5.96	5.86	
Paper	3.69	3.07	3.35	3.32	3.32	3.31	
Conc	1.57	3.92	3.46	3.98	2.64	2.55	
Cr Grade(%)	2.41	2.54	2.42	2.22		13.16	17.39
B + H2O	549.54	533.61	536.53	530.58	551.79		
Bottle	517.98	444.67	374.15	333.94	213.41		
H2O	31.56	88.94	162.38	196.64	338.38		
D + C + H2O	417.02	504.35	560.24	611.4	748.08		
Dish	365.15	361.36	354.64	368.08	382.42		
C + H2O	51.87	142.99	205.6	243.32	365.66		
H2O recovery	18.74	50.13	39.76	42.7	24.64		

FLOAT 3.

Run A3

Mass	Conc 1	Conc 2	Conc 3	Conc 4	Conc 5	Feed	Tails
C + paper	5.33	7.44	8.53	6.51	5.86	8.53	
Paper	3.5	3.58	3.47	3.49	3.29	3.33	
Conc	1.83	3.86	5.06	3.02	2.57	5.2	
Cr Grade(%)	2.53	2.51	2.45	2.44		14.42	18.01
B + H2O	517.95	527.46	528.33	549.28	557.8		
Bottle	485.75	443.5	354.73	333.68	179.88		
H2O	32.2	83.96	173.6	215.6	377.92		
D + C + H2O	420.35	498.56	596.15	619.35	780.56		
Dish	365.15	361.36	354.64	368.08	382.42		
C + H2O	55.2	137.2	241.51	251.27	398.14		
H2O recovery	21.17	49.38	62.85	32.65	17.65		

FLOAT 4.

Run B1 **Base Case** 0 g/ton CuSO4 100 g/ton IMP4

Mass	Conc 1	Conc 2	Conc 3	Conc 4	Conc 5	Feed	Tails
C + paper	6.63	5.54	6.93	7.36	5.77	7.48	
Paper	3.17	3.37	3.46	3.53	3.24	3.19	
Conc	3.46	2.17	3.47	3.83	2.53	4.29	
B + H2O	550.1	538.85	555.46	550.93	558.7		
Bottle	508.15	449.71	366.36	329.4	164.34		
H2O	41.95	89.14	189.1	221.53	394.36		
D + C + H2O	486.99	506.72	610.12	650.1	808.75		
Dish	365.15	361.36	354.64	368.08	382.42		
C + H2O	121.84	145.36	255.48	282.02	426.33		
H2O recovery	76.43	54.05	62.91	56.66	29.44		

FLOAT 5.

Run B2

Mass	Conc 1	Conc 2	Conc 3	Conc 4	Conc 5	Feed	Tails
C + paper	5.46	5.78	7.98	6.52	6.27	8.12	
Paper	3.27	3.23	3.39	3.34	3.34	3.28	
Conc	2.19	2.55	4.59	3.18	2.93	4.84	
B + H2O	508.1	537	543.72	549.62	545.53		
Bottle	469.28	422.97	357.77	282.91	153.52		
H2O	38.82	114.03	185.95	266.71	392.01		
D + C + H2O	450.21	530.95	619.9	677.28	809.52		
Dish	365.15	361.36	354.64	368.08	382.42		
C + H2O	85.06	169.59	265.26	309.2	427.1		
H2O recovery	44.05	53.01	74.72	39.31	32.16		

FLOAT 6.

Run B3

Mass	Conc 1	Conc 2	Conc 3	Conc 4	Conc 5	Feed	Tails
C + paper	4.66	5.4	7.61	7.33	6.25	6.13	
Paper	3.34	3.27	3.19	3.23	3.13	3.14	
Conc	1.32	2.13	4.42	4.1	3.12	2.99	
B + H2O	550.44	545.26	553.16	536.05	546.35		
Bottle	511.6	445.47	393.81	343.01	209.61		
H2O	38.84	99.79	159.35	193.04	336.74		
D + C + H2O	438.68	518.49	606.53	632.09	767.47		
Dish	365.15	361.36	354.64	368.08	382.42		
C + H2O	73.53	157.13	251.89	264.01	385.05		
H2O recovery	33.37	55.21	88.12	66.87	45.19		

FLOAT 7.

100 g/t
IMP4
Run C1 30 g/ton
CuSO4

Mass	Conc 1	Conc 2	Conc 3	Conc 4	Conc 5	Feed	Tails
C + paper	4.42	6.09	8.45	7.48	6.29	11.24	
Paper	3.25	2.97	3.02	2.9	3.12	3.18	
Conc	1.17	3.12	5.43	4.58	3.17	8.06	
B + H2O	556.28	541.46	538.06	546.71	557.01		
Bottle	528.82	426.8	355.96	344.09	162.68		
H2O	27.46	114.66	182.1	202.62	394.33		
D + C + H2O	407.46	530.23	627.6	671.2	804.38		
Dish	365.15	361.36	354.64	368.08	382.42		
C + H2O	42.31	168.87	272.96	303.12	421.96		
H2O recovery	13.68	51.09	85.43	95.92	24.46		

FLOAT 8.

Run C2

Mass	Conc 1	Conc 2	Conc 3	Conc 4	Conc 5	Feed	Tails
C + paper	5.64	6.6	7.6	8.51	7.43	8.47	
Paper	3.42	3.42	3.43	3.39	3.24	3.24	
Conc	2.22	3.18	4.17	5.12	4.19	5.23	
B + H2O	540.93	550.24	552.21	552.6	553.22		
Bottle	494.24	440.9	337.41	307.95	168.52		
H2O	46.69	109.34	214.8	244.65	384.7		
D + C + H2O	448.91	534.31	639.39	690.26	825.32		
Dish	365.15	361.36	354.64	368.08	382.42		
C + H2O	83.76	172.95	284.75	322.18	442.9		
H2O recovery	34.85	60.43	65.78	72.41	54.01		

FLOAT 7.

100 g/t
IMP4
Run C1 30 g/ton
CuSO4

Mass	Conc 1	Conc 2	Conc 3	Conc 4	Conc 5	Feed	Tails
C + paper	4.42	6.09	8.45	7.48	6.29	11.24	
Paper	3.25	2.97	3.02	2.9	3.12	3.18	
Conc	1.17	3.12	5.43	4.58	3.17	8.06	
B + H2O	556.28	541.46	538.06	546.71	557.01		
Bottle	528.82	426.8	355.96	344.09	162.68		
H2O	27.46	114.66	182.1	202.62	394.33		
D + C + H2O	407.46	530.23	627.6	671.2	804.38		
Dish	365.15	361.36	354.64	368.08	382.42		
C + H2O	42.31	168.87	272.96	303.12	421.96		
H2O recovery	13.68	51.09	85.43	95.92	24.46		

FLOAT 8.

Run C2

Mass	Conc 1	Conc 2	Conc 3	Conc 4	Conc 5	Feed	Tails
C + paper	5.64	6.6	7.6	8.51	7.43	8.47	
Paper	3.42	3.42	3.43	3.39	3.24	3.24	
Conc	2.22	3.18	4.17	5.12	4.19	5.23	
B + H2O	540.93	550.24	552.21	552.6	553.22		
Bottle	494.24	440.9	337.41	307.95	168.52		
H2O	46.69	109.34	214.8	244.65	384.7		
D + C + H2O	448.91	534.31	639.39	690.26	825.32		
Dish	365.15	361.36	354.64	368.08	382.42		
C + H2O	83.76	172.95	284.75	322.18	442.9		
H2O recovery	34.85	60.43	65.78	72.41	54.01		

FLOAT 9.

Run C3

Mass	Conc 1	Conc 2	Conc 3	Conc 4	Conc 5	Feed	Tails
C + paper	4.71	6.87	9.25	8.1	6.4	10.47	
Paper	3.09	4.02	3.37	3.52	3.4	3.41	
Conc	1.62	2.85	5.88	4.58	3	7.06	
B + H2O	554.52	554.41	556.69	550.29	538.12		
Bottle	506.79	430.5	356.34	323.18	135.93		
H2O	47.73	123.91	200.35	227.11	402.19		
D + C + H2O	437.47	537.86	655.18	661.67	819.59		
Dish	365.15	361.36	354.64	368.08	382.42		
C + H2O	72.32	176.5	300.54	293.59	437.17		
H2O recovery	22.97	49.74	94.31	61.9	31.98		

FLOAT 10.

Run D1 **90 g/t Cu** **200 g/ton IMP4**

Mass	Conc 1	Conc 2	Conc 3	Conc 4	Feed	Tails
C + paper	4.33	5.49	7.05	6.48	10.54	972.69
Paper	2.96	3.52	3.37	3.48	3.62	7.14
Conc	1.37	1.97	3.68	3	6.92	965.55
B + H2O	545.24	539.54	544.87	560.51		
Bottle	495.43	421.53	362.79	252.25		
H2O	49.81	118.01	182.08	308.26		
D + C + H2O	441.26	530.34	607.88	745.31		
Dish	365.15	361.36	354.64	368.08		
C + H2O	76.11	168.98	253.24	377.23		
H2O recovery	24.93	49	67.48	65.97		

FLOAT 11.

Run D2

Mass	Conc 1	Conc 2	Conc 3	Conc 4	Feed	Tails
C + paper	5.4	5.8	6.45	7.25	10.66	967.41
Paper	3.4	3.48	3.5	3.33	3.39	7.14
Conc	2	2.32	2.95	3.92	7.27	960.27
B + H2O	549.78	536.12	544.81	548.51		
Bottle	493.14	407.89	339.15	163.48		
H2O	56.64	128.23	205.66	385.03		
D + C + H2O	469.98	556.16	629.55	829.65		
Dish	365.15	361.36	354.64	368.08		
C + H2O	104.83	194.8	274.91	461.57		
H2O recovery	46.19	64.25	66.3	72.62		

FLOAT 12.

Run E1 **30 g/t Cu** **200 g/t IMP4**

Mass	Conc 1	Conc 2	Conc 3	Conc 4	Feed	Tails
C + paper	5.34	5.11	6.44	6.97	10.01	977.75
Paper	3.21	3.37	3.03	3.48	3.57	7.44
Conc	2.13	1.74	3.41	3.49	6.44	970.31
B + H2O	542.64	535.25	546.33	556.64		
Bottle	494.76	390.54	339.29	171.77		
H2O	47.88	144.71	207.04	384.87		
D + C + H2O	462.86	577.2	664.61	842.52		
Dish	365.15	361.36	354.64	368.08		
C + H2O	97.71	215.84	309.97	474.44		
H2O recovery	47.7	69.39	99.52	86.08		

FLOAT 13.

Run E2

Mass	Conc 1	Conc 2	Conc 3	Conc 4	Feed	Tails
C + paper	5.13	5.19	5.88	7.16	10.44	986.24
Paper	3.39	3.5	3.53	3.55	3.45	7.23
Conc	1.74	1.69	2.35	3.61	6.99	979.01
B + H2O	565.05	551.65	565.69	557.5		
Bottle	506.11	410.6	296.27	165.14		
H2O	58.94	141.05	269.42	392.36		
D + C + H2O	475.23	568.21	695.94	836.92		
Dish	365.15	361.36	354.64	368.08		
C + H2O	110.08	206.85	341.3	468.84		
H2O recovery	49.4	64.11	69.53	72.87		

FLOAT 14.

Run F1 **90 g/t Cu** **100 g/t KU5**

Mass	Conc 1	Conc 2	Conc 3	Conc 4	Feed	Tails
C + paper	8.64	10.42	11.14	10.19	8.31	973.77
Paper	3.6	3.65	3.68	3.45	3.4	7.25
Conc	5.04	6.77	7.46	6.74	4.91	966.52
B + H2O	559.6	524.88	540.69	543.09		
Bottle	504.72	372.81	226.49	140.46		
H2O	54.88	152.07	314.2	402.63		
D + C + H2O	474.39	591.2	692.44	849.11		
Dish	365.15	361.36	354.64	368.08		
C + H2O	109.24	229.84	337.8	481.03		
H2O recovery	49.32	71	16.14	71.66		

FLOAT 15.

Run F2

Mass	Conc 1	Conc 2	Conc 3	Conc 4	Feed	Tails
C + paper	7.38	10.42	11.14	9.6	13.59	965.35
Paper	3.47	3.65	3.5	3.53	3.46	7.31
Conc	3.91	6.77	7.64	6.07	10.13	958.04
B + H2O	566.5	568.15	564.58	558.87		
Bottle	509.94	406.56	290.97	161.93		
H2O	56.56	161.59	273.61	396.94		
D + C + H2O	457.46	591.66	706.44	831.42		
Dish	365.15	361.36	354.64	368.08		
C + H2O	92.31	230.3	351.8	463.34		
H2O recovery	31.84	61.94	70.55	60.33		

FLOAT 16.

Run G1 **30 g/t Cu** **100 g/t KU5**

Mass	Conc 1	Conc 2	Conc 3	Conc 4	Feed	Tails
C + paper	7.65	10.02	10.67	10.03	14.31	971.76
Paper	3.58	3.49	3.45	3.47	3.42	7.43
Conc	4.07	6.53	7.22	6.56	10.89	964.33
B + H2O	548.11	535.58	541.74	545.51		
Bottle	503.11	403.95	315.39	161.67		
H2O	45	131.63	226.35	383.84		
D + C + H2O	463.14	585.87	674.03	831.18		
Dish	365.15	361.36	354.64	368.08		
C + H2O	97.99	224.51	319.39	463.1		
H2O recovery	48.92	86.35	85.82	72.7		

FLOAT 17.

Run G2

Mass	Conc 1	Conc 2	Conc 3	Conc 4	Feed	Tails
C + paper	6.61	9.02	11.86	10.99	13.29	973.84
Paper	3.61	3.54	3.53	3.38	3.34	7.38
Conc	3	5.48	8.33	7.61	9.95	966.46
B + H2O	563.24	559.35	565.31	564.56		
Bottle	493.61	402.58	292.3	180.95		
H2O	69.63	156.77	273.01	383.61		
D + C + H2O	468.16	588.31	731.7	826.23		
Dish	365.15	361.36	354.64	368.08		
C + H2O	103.01	226.95	377.06	458.15		
H2O recovery	30.38	64.7	95.72	66.93		

FLOAT 18.

Run H1 90 g/t Cu 200 g/t KU5

Mass	Conc 1	Conc 2	Conc 3	Conc 4	Feed	Tails
C + paper	4.37	5.69	7.28	9.37	15.2	986.28
Paper	3.33	3.39	3.14	3.56	3.52	7.26
Conc	1.04	2.3	4.14	5.81	11.68	979.02
B + H2O	545.55	546	543.32	551.07		
Bottle	487.64	429.99	306.61	171.68		
H2O	57.91	116.01	236.71	379.39		
D + C + H2O	444.24	533.24	658.34	808.06		
Dish	365.15	361.36	354.64	368.08		
C + H2O	79.09	171.88	303.7	439.98		
H2O recovery	20.14	53.57	62.85	54.78		

FLOAT 19.

Run H2

Mass	Conc 1	Conc 2	Conc 3	Conc 4	Feed	Tails
C + paper	4.36	5.06	6.44	8.84	16.48	982.73
Paper	3.41	3.39	2.96	3.33	3.3	7.25
Conc	0.95	1.67	3.48	5.51	13.18	975.48
B + H2O	539.39	552.26	551.33	548.57		
Bottle	480.5	403.17	312.84	143.75		
H2O	58.89	149.09	238.49	404.82		
D + C + H2O	444.5	554.94	659.09	826.31		
Dish	365.15	361.36	354.64	368.08		
C + H2O	79.35	193.58	304.45	458.23		
H2O recovery	19.51	42.82	62.48	47.9		

FLOAT 20.

Run I1 **30 g/t Cu** **200 g/t KU5**

Mass	Conc 1	Conc 2	Conc 3	Conc 4	Feed	Tails
C + paper	3.86	4.9	5.62	8.72	17.3	973.84
Paper	3.15	3.23	3.1	3.21	3.2	7.41
Conc	0.71	1.67	2.52	5.51	14.1	966.43
B + H2O	559.36	544.39	552.35	558.75		
Bottle	504.64	424.17	298.9	140.77		
H2O	54.72	120.22	253.45	417.98		
D + C + H2O	434.93	532.08	681.53	861.63		
Dish	365.15	361.36	354.64	368.08		
C + H2O	69.78	170.72	326.89	493.55		
H2O recovery	14.35	48.83	70.92	70.06		

FLOAT 21.

Run I2

Mass	Conc 1	Conc 2	Conc 3	Conc 4	Feed	Tails
C + paper	4.82	4.65	5.41	9.67	14.91	981.8
Paper	3.24	3.29	3.31	3.22	3.18	7.53
Conc	1.58	1.36	2.1	6.45	11.73	974.27
B + H2O	557.56	553.39	565.18	568.26		
Bottle	503.28	422.15	313.84	190.32		
H2O	54.28	131.24	251.34	377.94		
D + C + H2O	467.68	554.82	683.12	851.02		
Dish	365.15	361.36	354.64	368.08		
C + H2O	102.53	193.46	328.48	482.94		
H2O recovery	46.67	60.86	75.04	98.55		