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THE ROLE OF PULP CHEMISTRY IN THE FLOTATION RECOVERY OF NCHANGA UNDERGROUND COPPER ORE

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SYNOPSIS

Nchanga concentrator treats various ores mined from around Chingola namely: Nchanga open pit, Luano open pit and Underground. About 30% of the copper ore (3% TCu) treated arises from the underground mine and the rest (70%) from the open pits (1.00 – 2.90% TCu). Cobalt ore is mined from the open pit at a grade of 1.30-1.80% Cu and 0.20-0.80% Co. The copper ore is a complex mixture of sulphide and oxide minerals and processing consists of sulphide flotation followed by sulphidisation and an oxide flotation step. The work reported in this study involves that of the underground copper ore only.

As the high-grade sources of ore have been depleted, the plant is increasingly treating low-grade material of varied mineralogy, which presents a variety of problems. The oxide minerals have proved difficult to recover. The reason for poor recovery has been attributed to complex mineralogical characteristics of the ore. Previous laboratory tests work carried out on the ore showed an improvement in the flotation recovery of oxide minerals after regrinding the sulphide rougher tails with mild steel media. However, further investigation showed that the increase in recovery was not due to increased liberation. This indicated that the increase was due to some other factors resulting from the regrind, such as change in the chemical environment.

This study was done in three phases. Phase I involved the work at University of Cape Town (UCT) and the following were investigated: (1) Effect of milling media (mild steel and stainless steel) on pulp chemical conditions and the resulting flotation of the ore, and (2) the effect of regrinding the sulphide rougher tails with mild steel and stainless steel media on flotation recovery of oxide minerals. Phase II of the work done at Nchanga concentrator involved the measurements of pulp chemistry parameters from milling stage to flotation stage in the Plant to find out whether they are similar to those obtained in the laboratory with mild steel or stainless steel milling media. Phase III was the work done at the University of Zambia (UNZA) and it consisted of: (1) evaluation of the effect of sulphidising in the nitrogen environment on flotation response of oxide minerals after mild steel milling, (2) determination of the nature of fast and slow floating sulphide minerals.

The results of phase I showed that the two grinding media created different chemical conditions in the pulp as evidenced by the dissolved oxygen (DO) and oxidation-reduction potential (ORP) measurements. Better overall flotation performance was obtained in pulp ground by stainless steel media than mild steel media. There was an improvement in the flotation recovery of both sulphide and oxide minerals after regrinding the sulphide rougher tails with mild steel media. Stainless steel regrind did not increase the recovery but improved the grade of oxide minerals in the oxide flotation stage. However, the flotation recovery of copper obtained after regrind by mild steel media was almost the same as that obtained without regrind by stainless steel media, showing that the increased recovery obtained in the former was not due to increased liberation. Several interesting observations concerning pulp chemistry measurements are discussed.

Results of phase II of the work at Nchanga concentrator showed that the pulp chemistry parameters obtained in the plant were almost the same as those obtained in the laboratory with mild steel milling media at UCT in Phase I. The dissolved oxygen level in the cyclone overflow was lower than that of the flotation banks due to aeration in the latter. Thus, it is possible that the effects of mild steel milling media on pulp chemical conditions and flotation performance observed in the work of phase I could be expected to occur at Nchanga concentrator.

The results of phase III of the work at UNZA showed an improvement in the flotation recovery of oxide minerals after sulphidisation in the nitrogen environment. The use of nitrogen to exclude oxygen increases the efficacy of new sulphide surface formation and improves process efficiencies by maximising the availability of sulphide ions for surface sulphidisation to occur. The recovery of acid soluble copper in the standard flotation test was higher at UNZA than at UCT (Phase I). The difference in the recoveries of acid soluble copper has been attributed to the ineffectiveness of the sulphidiser used in Phase I.

The fast-floating sulphide minerals occurred in the coarser size range and were liberated. The slow floating sulphide minerals were made up of fine particles, disseminated and partially locked mineral grains. Chalcocite was the predominant fast floating mineral in all concentrate fractions. The slow floating minerals consisted of chalcopyrite, bornite and carrollite.

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NOMENCLATURE

TCu = Total Copper
ASCu = Acid Soluble Copper
AICu = Acid Insoluble Copper
Eh = Pulp Potential
E_{pt} = Platinum Potential
DO = Dissolved Oxygen
MS = Mild Steel Grinding
MSR = Mild Steel Grinding and Regrinding
SS = Stainless Steel Grinding
SSR = Stainless Steel Grinding and Regrinding
Temp. = Temperature
ZCCM = Zambia Consolidated Copper Mines
UNZA = University of Zambia
UCT = University of Cape Town
RA = Relative Abundance
SIPX = Sodium Isopropyl Xanthate
TEB = Tri-ethoxy-Butane
PKD = Palm Kernel Oil and Diesel
NaHS = Sodium Hydrogen Sulphide
Fe = Iron
Cu = Copper
S = Sulphur
Cct = Chalcocite
Bn = Bornite
Cp = Chalcopyrite
Py = Pyrite
Crrt = Carrollite
N-Cu = Native Copper
S.G = Specific Gravity
Flot. = Flotation
Wt = Weight
G/t = Gram per Tonne
SRB = Sulphide Rougher Banks
ORB = Oxide Rougher Banks
OCB = Oxide Cleaner Banks

CHAPTER 1

Introduction and Research Objectives

1.0 INTRODUCTION AND RESEARCH OBJECTIVES

1.1 Background

Nchanga concentrator, situated in Zambia treats an average of about 24000 tonnes of copper ore per day with 70% of the material arising from Open Cast Mining and the rest from Underground Mining. The different ore sources from the Open Pits result in material blends, which exhibit marked variations on flotation response due to wide fluctuations both in the head grade and degree of mineral oxidation.

The plant is increasingly treating low-grade material of varied mineralogy because of the continuous depletion of high-grade sources of copper ore from the mines. Oxide minerals have proved difficult to recover. The reason for poor recovery has been attributed to complex mineralogical characteristics of the ore.

In trying to improve the flotation performance of the copper ore, various changes were adopted by Nchanga concentrator between 1988 and 1994. These include the following:

(i) Commissioning of the new sulphide plant to replace small cells of 45 cubic ft with bigger Wemco cells of 1500 cubic ft.

(ii) The replacement of 36 inch diameter hydrocyclones with 20 inch Krebs hydrocyclones to improve the classification efficiency in grinding plant.

(iii) Introduction of column flotation cells and separate flotation of washed slimes from the ground product.

These process innovations did not entirely resolve the low flotation circuit recoveries. As such in August 1995 an extensive plant trial was conducted on the separate flotation of Underground and Open Pit copper ores at sulphide roughing and cleaning stage to investigate further the findings obtained from laboratory test work on the different characteristics of the two materials.

The complete separate treatment of the Underground and Open Pit copper ores on 10th October 1995 resulted in the creation of two copper flotation circuits at Nchanga concentrator (Underground and Open Pit flotation circuits) (Beene et al., 1996). The overall total copper recovery on Underground circuit now averages 65% while the Open Pit circuit recovery is about 25% to 50% TCu. Also the ease with which the Underground rougher concentrate upgrades through one stage cleaning in the 72 inch flotation columns results in low copper content in the circulating load to the mechanical scavenger banks. However the overall flotation recovery of valuables still remains marginal.

Previous laboratory test work carried out on underground copper ore showed an improvement in the flotation recovery of oxide minerals after regrinding the sulphide rougher tails with mild steel media (Tepa, 1998). However, the investigation showed that the increase in recovery was not due to increased liberation. This indicated that the increase was due to some other factors resulting from the regrind, such as the effect of the change in pulp chemical environment.

1.2 The Context of this Research in Flotation Research

Flotation research is a wide and complex field of study. It is well known to mineral processing scientists and engineers, that flotation involves the selective capture of small valuable mineral particles by bubbles in slurry, followed by their levitation and collection in a froth layer leaving the unwanted gangue in the pulp. Successful flotation separation depends on the interrelation among the various physical, chemical and mechanical factors involved in the system (Fuerstenau, 1988).

The very basis of flotation is a physical process involving the relative interaction of three phases: one solid and two fluids. These interactions control which of the two fluids will wet the solid phase. Physical phenomena control bubble-particle interaction (microscopic flotation kinetics) and particle –particle interactions.

Chemical manipulation is the key to flotation processing, in that the concentration of minerals by flotation is achieved by modifying the various mineral surfaces, which affect the interaction between water molecules and the mineral surfaces. The success of this process rests in the judicious use of a wide variety of chemical reagents that, upon their adsorption at the solid-liquid, solid-gas or liquid-gas interfaces, modify the physical-chemical characteristics of the system.

Mechanical factors, such as the design of flotation cells, control bubble generation, mineral-particle dispersion and, consequently, the macroscopic kinetics of flotation. It is in this area that the practising flotation engineer attempts to optimise the process in order to make the best use of the chemistry and physics of the system for effecting separations between hydrophilic and hydrophobic mineral particles.

The scope of this research within flotation science and engineering is shown in Figure 1.2 Examples are the effect of milling media on pulp chemical conditions and flotation performance of valuables, and the effect of sulphidising in the nitrogen environment on the floatability of oxide minerals. Both the milling media (process variable) and nitrogen (chemical variable) affect the chemical conditions in the pulp to influence the floatability of valuables in one way or another.

1.3 Research Objectives

The reasons for the poor flotation performance of copper from the underground ore at Nchanga have not been fully established. The hypotheses of this research are:

(a) The recovery of acid soluble copper can be improved by regrinding the sulphide rougher tails, but the grinding media may influence the effect.
(b) Nitrogen conditioning can enhance the efficiency of sulphidisation to improve the recovery of acid soluble copper.
(c) Pulp chemistry measurements can be used to characterise the pulp and predict flotation performance, so that if similar pulp chemistries are achieved, then the same flotation performance can be expected.
(d) The Nchanga copper ore feed is made up of fast- and slow-floating copper bearing particles with different floatabilities depending on liberation and mineral type.

This study has been initiated to extend the understanding of the flotation behaviour by addressing the following:

(i) To evaluate the differences in pulp chemistry and flotation performance between ore milled with Stainless Steel (SS) and Mild Steel Media (MS).

(ii) To determine the cause of the benefit of Regrinding [with Stainless Steel (SSR) and Mild Steel (MSR)] the Sulphide Rougher Tails in the Flotation Recovery of Oxide Minerals.

(iii) To evaluate the effect of Nitrogen Conditioning on Sulphidisation and Flotation Response of Oxide Minerals.

(iv) To determine the Nature of Fast- and Slow-Floating Sulphide Minerals based on mineralogy and flotation kinetics in order to recommend how to improve the slow floating minerals.

(v) To compare the pulp chemistry measurements of Nchanga Concentrator Plant with that of laboratory tests in order to establish whether there is a direct relationship between pulp chemistry measurements and flotation performance.

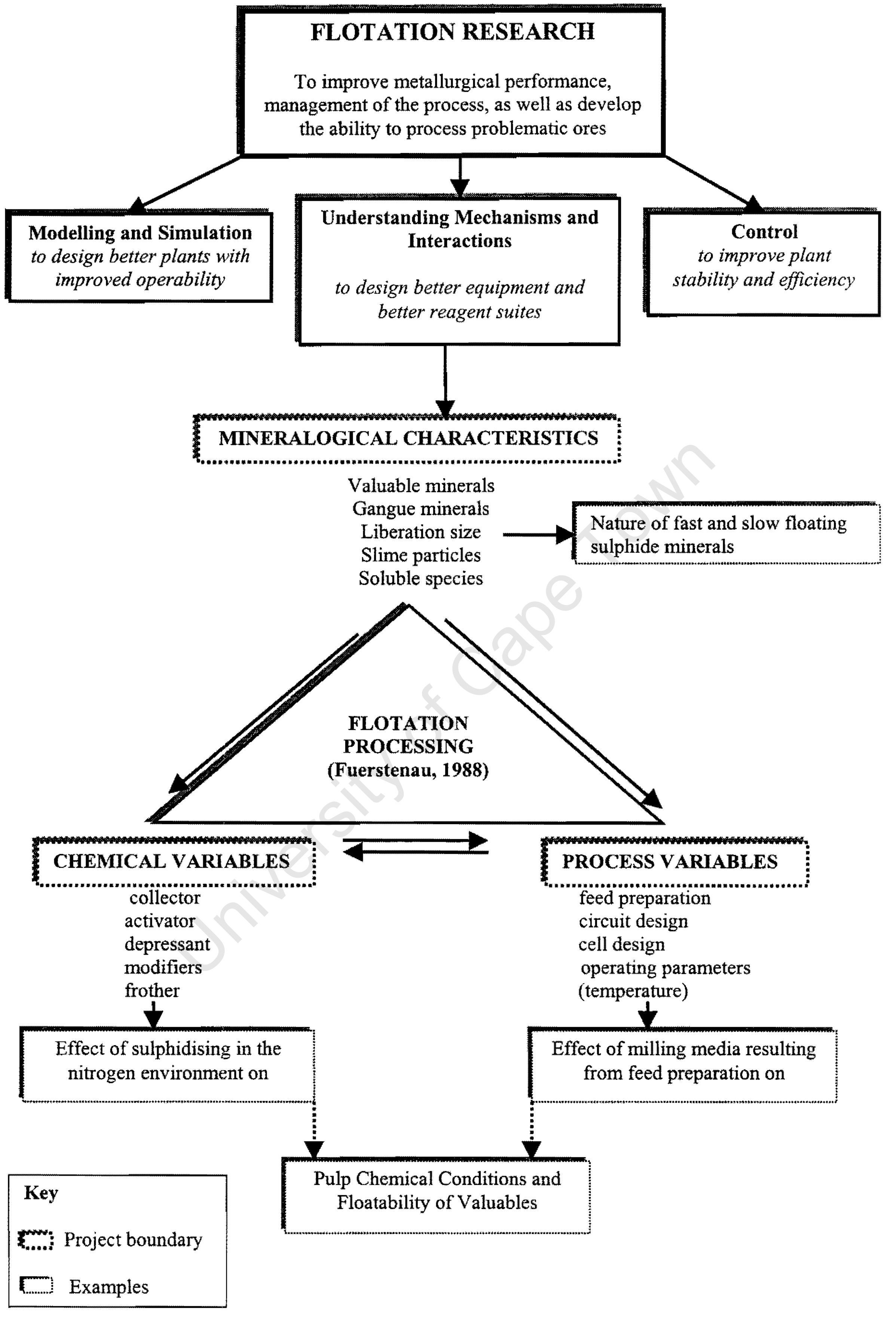


Figure 1.2: The Context of this Research in Flotation Research

CHAPTER 2

Literature Review

2.0 LITERATURE REVIEW

2.1 Flotation Fundamentals

2.1.1 Froth Flotation

Froth flotation is a process used to separate minerals, suspended in liquids, by attaching them to gas bubbles to provide selective levitation of the solid particles. It is the most extensively used process for the separation of minerals, to concentrate ore for economic smelting.

Floatable minerals can be classified into non-polar and polar types according to Wills (1988). The segregation into these two types of minerals is based on their surface bonding. The surfaces of non-polar minerals have relatively weak molecular bonds, difficult to hydrate, and in consequence such minerals are hydrophobic. Non-polar minerals include graphite, sulphur, molybdenite, diamond, coal, and talc, all naturally floatable in the pure state. The ores containing these minerals for beneficiation usually require the addition of non-specific collectors to their pulp to aid the natural hydrophobicity of the floatable fraction. Polar minerals (e.g. oxide minerals) have a strong covalent or ionic surface bonding, and exhibit high free energies at their surfaces. Therefore, surface hydration is rapid due to the strong reaction with water molecules, which form multilayers on the mineral. Thus these species are hydrophilic and need surface modifiers to render them amenable to flotation.

Bulk flotation is a somewhat imprecise term that covers nearly all the normal rougher or scavenger flotation, where a single mineral or a group of related minerals is separated from gangue and other low value minerals in a single flotation step. An example would be the recovery of a mixed copper sulphide concentrate from an ore containing pyrite and gangue.

Differential flotation is the term normally used to describe the separation of complex ores, and is generally restricted to describing the separation of similar minerals from each other (e.g. copper, lead, zinc, cobalt and silver) where the successful and economic recovery of each component involves the sophisticated use of collectors, depressants and flotation activators.

Flotation Reagents

For a flotation process to be successful, reagents must be added to the pulp. Flotation reagents ensure that the flotation process is highly selective and efficient. Elaborate control of the wetting or non-wetting of mineral surfaces facilitates the separation, in the flotation system. The reagents employed are generally interfacial surface tension modifiers, surface chemistry modifiers, and/or flocculants. Usually they are classified under five headings: collectors (sometimes known as flotation promoters), frothers, modifiers, activators and depressants. Each type of reagent has a specific purpose:

Collectors

Collectors are organic substances, which are used in flotation to render selected minerals water-repellent by the adsorption of molecules or ions of the collector on the mineral surface. Under these conditions, the stability of the hydrated layer, which separates the mineral particle and the air-bubble adjacent to it, is reduced to the level necessary for the formation of a three phase contact perimeter when the particles come in contact with a bubble, and for attachment of the bubble after a short interval of time.

Frothers

Frothers are surface-active reagents that aid in the formation and stabilisation of air-induced flotation froths. They concentrate at the air-water interface, helping to keep the air bubbles dispersed and preventing their coalescence. Frothing agents increase the stability of the flotation froth by stabilising the mineralised bubble as it rises to the pulp surface.

Modifiers, Activators and Depressants

Flotation modifiers, activators and depressants are sometimes known as regulating agents. The main purpose of these types of reagents is to control the effect of collectors on mineral particles, so that the selectivity of the flotation process is increased. In the presence of a regulating agent, the collector activates basically those minerals, which are required to pass into the froth.

In some cases, the regulating agent acts directly on the surface of a definite material, facilitating the subsequent interaction of this mineral and the collector and thus improving the results of flotation (activation). In other cases the regulating agent has the reverse effect, producing an adverse condition for mineral activation by the collector and thus leading to relatively poor results in flotation (depression). For pH control an environmental modifier such as lime may be used.

2.2 Flotation Processing

Processing of ores involves the understanding of the interaction between the three components of flotation processing: the mineralogical characteristics of the ore, the chemistry by which the system is controlled, and the process variables as illustrated in Figure 2.2 (Fuerstenau, 1988).

The favourable economics of a milling operation depend on optimal and effective manipulation of variables of the flotation system in order to achieve the desired flotation efficiency. Synthesis of an optimum circuit requires detailed consideration of numerous interacting factors (Fuerstenau, 1988; Sutherland, 1981; Dey, Kapur and Mehrotra, 1989). As seen by the arrows of Figure 2.2, mineralogical characteristics of the ore determine both the chemical and process variables required for efficient flotation. Therefore, optimum flotation conditions largely depend on manipulating the chemical and process variables for given feed characteristics of the ore. However, the ultimate objective or profit function in flotation processing may range from simply maximising recovery to highly complex combinations of feed and grade, equipment cost, operating expenses etc., subject to appropriate constraints on process and structural variables.

2.2.1 Mineralogical Characteristics of the Ore

Since the specific process is determined by the nature of the ore, the arrows from the mineralogical corner of the triangle are unidirectional. The nature of the ore determines liberation characteristics, that is, the particle size to which the ore must be ground for liberation, or whether the circuit can involve bulk flotation together with regrinding before selective flotation. Soluble species in solution and the amount and nature of slime particles is a consequence of ore mineralogy.

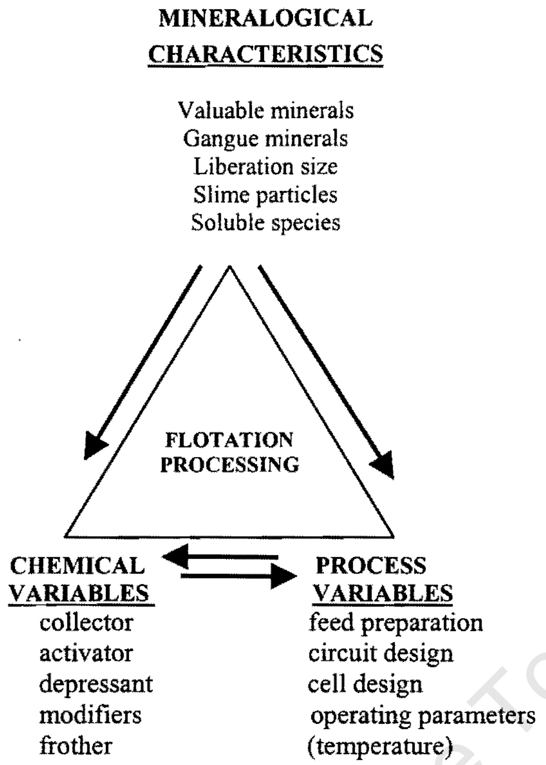


Figure 2.2: Components of Flotation Processing (Fuerstenau, 1988).

2.2.2 Chemical Variables

The effective flotation processing of an ore of given mineralogy is regulated by control of the bulk chemistry of the pulp (chemical conditions in the pulp), which in turn controls the surface chemistry of the particles and bubbles in the system. Judicious use of collectors, frothers, depressants, activators and modifiers achieve the physico-chemical effects induced on mineral particles in order to influence their floatability.

2.2.3 Process Variables

Process variables influence the potential floatability of the valuable minerals by sustaining optimal hydrodynamic conditions necessary for efficient flotation. The following sub- or micro-processes must be realised in a flotation machine. The suspension of the particles in the pulp, feeding and dispersion of air into bubbles. Others are mixing of the aerated pulp for reagent distribution and conditioning, increased collision efficiency between air bubbles and mineral particles as the basic requirement for bubble-particle attachment and rising of the loaded bubbles and removal of the froth (Schubert, Bischofberger and Koch, 1982). Process variables are in part determined by the ore and by the design/operating engineers. This corner of the triangle interacts with the chemical control but is dictated by the nature of the ore body. In the broadest sense, overall process variables include feed preparation (comminution, desliming), circuit design (rougher flotation, cleaner flotation, regrind), cell design, and operating parameters (such as froth depth, aeration, throughput) that are in the hands of the plant operator.

2.3 Liberation

There are two fundamental operations in mineral processing namely the release, or liberation, of the valuable minerals from their waste gangue minerals, and the separation of these values from the gangue, this latter process being known as concentration.

Liberation of the valuable minerals from the gangue is achieved by comminution, which involves crushing and grinding to such a particle size that the product is a mixture of relatively clean particles of mineral and gangue. Grinding is often the greatest energy consumer, accounting for up to 50% of a concentrator's energy consumption. As it is this process which achieves liberation of values from gangue, it is also the process which is essential to efficient separation of the minerals, and it is often said to be the key to good mineral processing. In order to produce clean concentrates with little contamination with gangue minerals, it is necessary to grind the ore finely enough to liberate the associated minerals. Fine grinding, however, increases energy cost, and can lead to the production of very fine untreatable "slime" particles which may be lost into the tailings. Grinding therefore becomes a compromise between clean (high-grade concentrates), operating costs and losses of fine minerals. If the ore is low-grade, and the minerals have very small grain size and are disseminated through the rock, then grinding energy costs and fines losses can be high.

An intimate knowledge of the mineralogical assembly of the ore is essential if efficient processing is to be carried out. Not only is knowledge of the nature of the valuable and gangue minerals required, but also of the ore texture. The texture refers to the aggregation (size), dissemination (distribution) and shape of the minerals within the ore.

One of the major objectives of comminution is the liberation, or release, of the valuable minerals from the associated gangue minerals at the coarsest possible particle size. If such an aim is achieved, then not only is energy saved by the reduction of the amount of fines produced, but any subsequent separation stages become easier and cheaper to operate.

In practice, complete liberation is seldom achieved, even if the ore is ground down to the grain size of the desired mineral particles. The particles of "locked" mineral and gangue are known as middling, and further liberation from this fraction can only be achieved by further comminution.

The "degree of liberation" refers to the percentage of the mineral occurring as free particles in the ore in relation to the total content. This can be high if there are weak boundaries between mineral and gangue particles, which is often the case with ores composed mainly of rock-forming minerals, particularly sedimentary minerals. Usually, however, the adhesion between mineral and gangue is strong and, during comminution, the various constituents are cleft across. This produces much middlings and a low degree of liberation. New approaches to increasing the degree of liberation involve directing the breaking stresses at the mineral crystal boundaries, so that the rock can be broken without breaking the mineral grains (Wills and Atkinson, 1993). Poor liberation leads to slow floating of mineral particles and the subsequent loss to tailings.

Many researchers have tried to quantify degree of liberation with a view to predicting the behaviour of particles in a separation process (Barbery, 1991). Gaudin (1939) made the first attempt at the development of a model for the calculation of liberation. King (1982) developed an exact expression for the fraction of particles of a certain size that contained less than a prescribed fraction of any particular mineral. These models, however, suffer from many unrealistic assumptions that must be made with respect to the grain structure of the minerals in the ore, and as a result have not found much practical application.

Attempts at quantifying liberation by means of automated optical image analysis have also been relatively unsuccessful due to the inherent inadequacies of the instrument in working with ore assemblies. Recent developments in scanning electron microscopy have made automated mineralogical assessment a real possibility for the near future (Sutherland and Gottlieb, 1991; Sutherland et al, 1991; Sutherland et al, 1993).

In practice, ores are ground to an optimum mesh of grind, determined by laboratory and pilot scale testwork, to produce an economic degree of liberation. The concentration process is then designed to produce a concentrate consisting predominantly of valuable mineral, with an accepted degree of locking with gangue minerals, and a middlings fraction, which may require further grinding to promote optimum release of minerals. The tailings should be mainly composed of gangue minerals, but often includes poorly liberated valuables.

2.4 Floatability and the Forces at the Mineral / Oil / Water Interface

Forces present on its surface will determine the wettability of a mineral by a liquid. That is, it will be a function of the relative number of hydrophilic and hydrophobic sites.

The thermodynamic condition for froth flotation is that the work of adhesion of water to mineral ($\mathbf{W_{w/m}}$) should be less than the work of cohesion of water ($\mathbf{W_{w/w}}$). In a two-liquid flotation system where the mineral surface is in contact with water and non-polar oil, the liquid with the higher work of adhesion to the solid should preferentially wet the solid and form a contact angle at the mineral/oil/water interface. That is, the liquid of polarity closer to that of the solid should preferentially wet the solid (Voyutsky, 1978).

In a two-liquid flotation system, in the presence of long chain reagents or any other electrolyte which adsorb at the oil/water interface, the assumption that dispersion forces alone are present will hold for the oil/water and mineral/oil interfaces (Fowkes, 1964). However, for mineral / water interfaces, it applies only to the minerals, when broken, whose fracture and cleavage surfaces form without rupture of interatomic bonds other than residual bonds.

With most minerals the work of adhesion of water to mineral ($\mathbf{W_{w/m}}$) involves other forces, in addition to dispersion forces which exist at all interfaces (Laskowski and Kitchener, 1969; Rao, 1974).

$$\mathbf{W_{w/m} = W_d + W_h + W_i}$$

Where $\mathbf{W_d}$ is the contribution from London-van der Waals (dispersion) forces, $\mathbf{W_h}$ is the contribution from hydration of non-ionic polar sites, as with hydrogen bonding of water to surface groups, and $\mathbf{W_i}$ is the contribution from ionic sites. The hydrophilicity of a mineral/water interface is caused by the $\mathbf{W_h}$ and $\mathbf{W_i}$ energies, which lead to the hydration of surfaces and the formation of electrical double layers. For oxygen containing minerals, $\mathbf{W_h} + \mathbf{W_i}$ is large compared to **Ww/m** and $\mathbf{W_d}$ (Laskowski and Kitchener, 1969; McCafferty and Zettlemoyer, 1971; Finkelstein et al., 1975). As a result oxide minerals have a strong surface hydration layer and are very hydrophilic.

The electrical double layer forces ($\mathbf{W_i}$) arising from the presence of a surface charge, affect the equilibrium thickness of the hydration layer, therefore the attachment of mineral particles to gas bubbles or oil drops (Read and Kitchener, 1969; Rao, 1974). Read and Kitchener suggest that for hydration films thicker than 30nm on solid surfaces, the electrical double layer forces should be predominant in determining the stability of hydration layers.

Sulphide minerals differ from most of the other groups of minerals such as oxides and silicates in various aspects.

(a) They are predominantly covalent bonded (Aplan and Fuersteneau, 1962).

(b) Under inert atmosphere, i.e. in the absence of oxygen, their solubilities are extremely low, reflected in the solubility product values: 10^{-28}, 10^{-45}, and 10^{-23} for PbS, CuS, and ZnS, respectively (Shergold, 1984).

(c) While oxygen of oxygen-containing minerals form hydrogen bonding with water molecules, sulphur of sulphide minerals does not form hydrogen bonding (Finkelstein et al., 1975; Fuerstenau and Sabacky, 1981).

The lack of hydrogen bonding by sulphur, because of its relatively large size and smaller electronegativity (Evans, 1979), reduces the adsorption of water hence the hydration of sulphide surfaces. The adhesion between a liquid and a solid undoubtedly is due to the chemical affinity, or the tendency of the solid to dissolve in or combine with the liquid (Hildebrand, 1936). This means that the more soluble a solid in a liquid is, the more the liquid wets it.

Therefore the low solubility of sulphides in water suggests that they should be relatively inert in aqueous solutions. Thus it can be said that the $\mathbf{W_h}$ component of the equation will be very small for sulphide minerals which have not undergone extensive oxidation. It has been reported that the hydrophilicity of minerals decreases as the covalency of the bonding within the mineral increases. Because W_i in the equation decreases, so does the work of adhesion of the water to the mineral surface.

2.5 Froth Stability, Particle Entrainment and Drainage in Flotation

The froth and its stability, the entrainment and the drainage of particles in flotation were long before recognised as important factors, which affect recovery and grade. The flotation process is known to be governed by a multitude of interacting variables and it becomes necessary to have a knowledge of those that contribute to the final yield. In a three-phase system like froth flotation involving solid-liquid-gas, the efficiency of separating the values from the gangue is considered to be a function of the adherence of an air bubble to the required mineral particle. The role of the gas phase in the form of dispersed bubbles in a pulp is to carry the hydrophobic mineral particles, obtained as concentrate, along with the froth from the flotation cell.

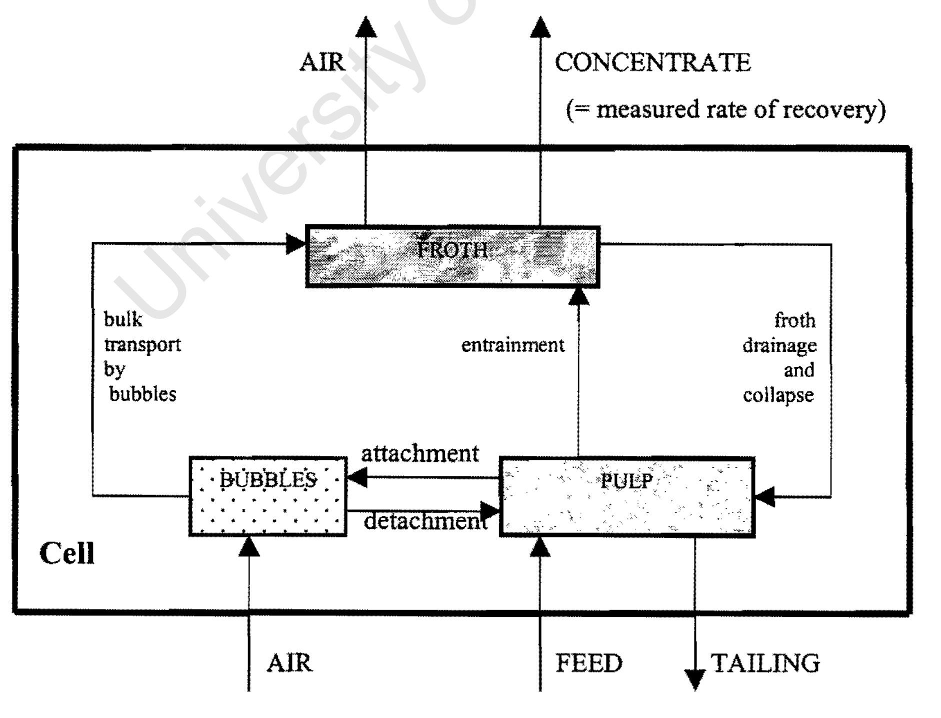


Figure 2.5: Transport Paths of Material in Flotation (Flint, 1973).

The froth phase is important since it can affect the flotation recovery and grade. It is considered that a small-bubbled closely-knit froth is favourable for high recoveries and a loosely knit froth of large bubbles for good grades. The entrainment and drainage of particles in flotation are concerned with the pulp/froth phases. All these phenomena are interrelated and influence the recovery and grade. Figure 2.5 illustrates the material transport paths in flotation.

2.5.1 Froth Stability

Froth stability refers to the persistency of the froth. Harris (1982) referred to two types of froths - unstable and metastable. Unstable froths are those which continuously break, due to draining of liquid from between the bubbles, and metastable froths can persist for longer times in the absence of disturbances.

A bubble produced in water is unstable. One of the prerequisites for a successful flotation operation is the stability of the bubble-particle aggregate. Using a frother, the function of which is to decrease the surface tension of the air-liquid interface, produces a stable bubble. The frother molecules impart stability to a bubble by migrating from a region of low surface tension to a higher value due to surface tension gradient.

A frother has a number of functions in flotation; first, it reduces the surface tension of the air-liquid interface in order that a stable bubble is produced in the system; secondly, it influences the kinetics of bubble-particle adhesion; thirdly, it thins the liquid layer by interacting with collector molecules and finally, it stabilises the bubble-particle aggregate (Schulman and Leja, 1954; Leja, 1956-57). In addition to the frother properties like surface activity, surface viscosity of the medium, etc., the nature of solids, i.e. the particle size and the hydrophobicity, play a dominant role in imparting stability to the froth. Lekki and Laskowski (1975) classified the frothers into surface-active and surface-inactive based on their mechanism of action, i.e. the ability to adsorb or co-adsorb on the mineral surface with a collector.

2.5.2 Particle Entrainment

The two important mechanisms by which particle collection takes place in a flotation operation are the flotation by adhesion and entrainment. Not much information exists on other possible mechanisms such as entrapment (Gaudin, 1957) and carrier flotation (Greene and Duke, 1962).

Several workers investigated the effect of particle and bubble sizes on flotation rate recovery; experimental studies and theoretical analysis have led to varying results. These aspects were dealt with in earlier works (Trahar, 1976, 1981; Jameson et al., 1977; Szatkowski and Freyberger, 1985). The influence of particle size on the rate of recovery of minerals has been investigated before (Gaudin et al., 1931; Anthony et al., 1975; Trahar, 1976). In industrial concentrators the recovery for copper, lead and zinc minerals was shown to be maximum in the size range of 10-100 μm (Gaudin et al., 1931). While the slow recovery rate of fine particles was due to decreased particle-bubble collisions, that of coarse particles was attributed to the disruption of bubble-particle aggregates in turbulent zones (Morris, 1950; Schultze, 1977). One of the reasons for the low flotation rate of coarse particles was that, with increasing particle size, the density of the bubble-particle aggregate approaches that of the pulp density and thereby the aggregate becomes less buoyant (Jameson et al., 1977). The argument put forth by Jowett (1980) for the poor recovery is in agreement with the observation made by Dippenaar (1982). With increasing particle size the induction time increases and hence the poor floatability of valuables.

For bubble-particle adhesion it is necessary that the bubble should collide with the particle and consequently film rupture will follow by the establishment of a three- phase contact. Whereas the entrainment is a characteristic feature of fine particles and is non-selective, there being no distinction between hydrophilic or hydrophobic particles. An arbitrary classification of particles into fine (5-10 μm), intermediate (10-70 μm) and coarse (> 70 μm) on the basis of their floatabilities was discussed by Trahar (1981). It is, of course, difficult to generalise since the floatabilities vary from mineral to mineral. However, it is known that the adhesion of particles > 10 μm in size occurs due to collision with air bubbles. For particles < 10 μm the collision efficiencies are low and the mechanism of collection takes place by entrainment.

In general the high recoveries obtained in flotation either at plant level or in batch tests are not separately accounted for in terms of recovery by entrainment and by true flotation. The contribution to the final yield by entrainment is significant, especially with fine particles present in the system, and it is important to estimate this factor for a better evaluation of the process performance.

The entrainment of particles in flotation is closely related to the recovery of water. Several workers investigated the effects of operating variables like froth depth, froth concentration and air addition which influence the water recovery. Mitrofanov et al. (1985) pointed out that the factors responsible for entrainment are the ascending and descending streams as a result of cell hydrodynamics, bubble population and size, amount of slimes and the concentration of reagents.

Wrobel (1953), in an investigation on flotation frothers, observed a direct relationship between water content in the froth and the concentrate grade. The varying behaviour of a mineral (flotation and entrainment) with different size fractions present in the pulp is evident from the work of Engelbrecht and Woodburn (1975) and Subrahmanyam and Forssberg (1986). Engelbrecht and Woodburn (1975) observed a linear relationship with water recovery in the case of hydrophobic pyrite $< 7.7 \mu m$ and silica $< 12 \mu m$. The recovery of coarse pyrite 20-27 μm was found to be dependent on hydrophobicity. For hydrophobic particles the recovery by entrainment was measured by differences in the relative contribution of true flotation in the presence of collector and by entrainment in the absence of the collector, i.e. with only frother present at the same water recoveries (Trahar, 1981).

2.5.3 Drainage

The particles are carried into the froth by bubbles rising through the pulp phase. These bubbles with their load of particles accumulate at the pulp/froth interface. A variable amount of the material carried into the froth returns to the pulp by drainage. The re-entry of particles into the pulp occurs because of the continuous drainage of liquid and bubble coalescence. The drainage of gangue particles is desirable from the viewpoint of enriching the grade. It was suggested that water spraying on the froth layer might improve both grade and recovery (Klassen and Mokrousov, 1963). Obviously this accelerates the liquid drainage and hence the drainage of gangue material.

2.6 Flotation of Sulphide Minerals

2.6.1 The Electrochemical Theory of Flotation

While flotation has been used in the processing of sulphide bearing ores since the early 20th century (Louis, 1909), it has proven difficult to develop a complete understanding of the underlying mechanism(s). This is due to the complexity of the process and the number of physical and chemical factors, which affect its success. Numerous mechanisms were proposed in the earlier years (Gaudin, 1957; Sutherland and Wark, 1955; Taggart, 1945). However, the electrochemical approach, first proposed by Salamy and Nixon (1952), has now gained wide support (Woods, 1976; Chander, 1988).

The floatability of a mineral is dependent on its hydrophobic nature. Sulphide minerals exhibit different degrees of natural hydrophobicity, though few have sufficient hydrophobicity to ensure flotation (natural collectorless flotation). Generally organic reagents (collectors) are required to prepare a hydrophobic surface for flotation, although the hydrophobicity of some minerals can be changed through chemical reaction in the absence of conventional collectors (induced collectorless flotation).

To date, collectorless flotation has been largely of academic interest (although the separation of molybdenite from copper sulphides uses kerosene or fuel oil instead of a conventional collector) (Sutulov, 1970). Significant to this discussion are observations that under sufficiently oxidising conditions (which can be resultant from autogenous grinding or grinding with high chrome or stainless media), collectorless flotation of some chalcopyrite, sphalerite and pyrites can occur (Heyes and Trahar, 1977, 1984; Lepetic, 1974; Plaksin, 1959; Rao and Finch, 1987-I, 1987-II; Trahar, 1983; Walker et al., 1986). The causes for collectorless flotation are, however, a subject open to many debates. The literature hosts a number of contributions indicating precisely opposing views (Luttrel and Yoon, 1984a,b; Yoon, 1981).

Normally, selective flotation of minerals is achieved using collectors, of which thiol collectors are the most important (Aplan and Chander, 1987). The reaction between collector and mineral is believed to be electrochemical in nature, involving the transfer of electrons between the solid (sulphide mineral) and liquid (water) phases. The electrochemical theory of flotation involves the anodic (oxidation) reaction between mineral and collector coupled with the cathodic reduction of dissolved oxygen in the water:

Anodic reaction:

$$2X^- \longrightarrow X_2 + 2e^- \qquad (1)$$

(Formation of dixanthogen)

or:

$$MeS + 2X^- \longrightarrow MeX_2 + S + 2e^- \qquad (2)$$

(Formation of metal xanthate)

Cathodic reaction:

$$1/2O_2 + H_2O + 2e^- \longrightarrow 2OH^- \qquad (3)$$

(Reduction of oxygen)

Where:
MeS: sulphide mineral
X^-: Xanthate

The above equations indicate that two criteria for reaction between collector and sulphide mineral are:

1) The mineral must be conducting or semi-conducting to permit the flow of electrons from the site of reactions 1 and 2 to that of 3.
2) Dissolved oxygen must be present in solution to act as the electron accepting element (reaction 3).

However, the presence of oxygen also causes oxidation of a mineral surface, which can reduce its ability to interact with the collector (Forssberg et al., 1988).

2.6.1.1 Native Floatability of Sulphide Minerals in the absence of Oxygen

Some sulphides, such as molybdenite, are naturally floatable under most conditions whereas others are considered to have intrinsic hydrophobic character in the absence of oxygen (Ravitz and Porter, 1933; Fuersteneau and Sabacky, 1981). The basis for native floatability rests on the assumption that sulphide lattice ions are weakly hydrated and do not interact strongly with water molecules. The critics of this hypothesis argue that sulphide minerals are thermodynamically unstable and sufficient oxygen remains in the system to cause oxidation

(Miller, 1988), presumably leading to the formation of elemental sulphur (a hypothesis originally proposed by Wark, 1938, and later suggested by several others). More recently, Woods (1987) has postulated a metal-deficient sulphur layer. The minimum quantity of the hydrophobic entity (elemental sulphur or metal-deficient layer) needed for complete flotation is yet to be established.

Other reasons for native floatability of sulphides have been postulated in the past. Gaudin (1932) considered that surfaces formed by rupture of van de Waals bonds are naturally hydrophobic. Chander and Fuersteneau (1972) postulated that molybdenite retains its hydrophilic character because the product of oxidation of the lattice metal ion is a soluble anion, which does not have the same hydration characteristics as other metal cations. In other words, many sulphides may acquire surface hydrophilic character through dissolution and readsorption of hydroxylated cations. These arguments suggest that the reasons for the floatability of sulphide minerals observed in the absence of collectors may vary from mineral to mineral. In most cases, the flotation behaviour depends upon the nature of the surface, which might be readily altered by electrochemical reactions.

2.6.1.2 Collectorless Flotation of Sulphide Minerals

Several investigators have suggested that sulphide minerals can be floated under mild to oxidising conditions although the reasons vary considerably. As early as 1949, Plaksin (1949) proposed that adsorbed oxygen decreases surface hydration, thereby making a mineral hydrophobic. In contrast, Heyes and Trahar (1977) observed that modest oxidation is required for collectorless flotation of chalcopyrite. Although several investigators have suggested that elemental sulphur is responsible for flotation, quantitative correlation has been difficult to establish.

In a recent study of oxidation of sulphide ions at a gold electrode, Chander and Briceno (1988a) distinguished three different forms in which sulphur could be present in its zero oxidation state. These are atomic S^{o}, sulphur in polysulphides (which in fact is a mixture of sulphur in two oxidation states), and elemental sulphur in S_8 form. It was observed that both polysulphides and elemental sulphur make the surface hydrophobic. In another study, Buckley, et al. (1985), has proposed a metal-deficient sulphide layer on the basis of XPS studies.

2.6.1.3 Collector Induced Floatability of Sulphide Minerals

Most of the sulphide minerals can be floated with the aid of small quantities of thiol collector. Some of the commercially used thiol collectors are xanthates, dithiophosphates, thioncarbonates and xanthogen formates (Aplan and Chander, 1987). Several other collectors such as thiocarbamates, xanthic esters, thioureas etc., are known to have collecting properties and some are used commercially on a small scale. Aplan and Chander give a more detailed discussion of collector type and its properties. The species responsible for inducing flotation are the metal salts of thiol reagents and / or dithiols of the collector. A necessary condition for floatability is that such compounds be present on the surface of the mineral to be floated.

2.6.2 Collector Adsorption on Sulphide Minerals

Thiol collectors like xanthates adsorb on sulphide minerals via a mixed potential mechanism, involving anodic oxidation of collectors and cathodic reduction of oxygen. The electrochemical process involved in collector adsorption is thought to be very similar to that involved in the corrosion of steel: electron deposition of the dissolving metal followed by reduction of dissolved oxygen to maintain electroneutrality. In the flotation process the collector molecule approaches the mineral surface and is absorbed only after it donates electrons to the mineral particle. These electrons migrate to an oxygen rich point on the surface where they react with oxygen to form OH^- ions, or H_2O or H_2O_2. The anodic oxidation of xanthate involves xanthate chemisorption, metal xanthates formation or catalytic oxidation to dixanthogen.

Depending on the mineral involved, the electrochemical mechanisms can be subdivided into four classes (Yoon and Basilio, 1993).

2.6.2.1 Chemisorption

This involves the formation of a monolayer of the thiol oxidation product at potentials below the thermodynamic potential for the metal thiol compound formation, that is, underpotential deposition. The chemisorption bond between the collector and the mineral surface is not a metal-sulphur bond but a sulphur-sulphur bond forming compounds of the type:

$$ROCS_2\text{-}S\text{-}S_2COR$$

It is generally concluded that, for the majority of the sulphide minerals, the negative charge on the metal forces the collector-mineral electron transfer to be from the collector sulphur atom to the mineral sulphur atom (Crozier, 1991).

2.6.2.2 Catalytic Oxidation

In this mechanism, the mineral provides a passage for the transfer of electrons from the site where the collector is oxidised to the site where oxygen is reduced, but does not participate in the reaction itself. This mechanism applies to the adsorption of xanthates on noble minerals such as pyrite, chalcopyrite, pyrrhotite and gold to form dixanthogen (Majima and Takeda, 1968; Woods, 1971; Gardner and Woods, 1977). However these minerals do not liberate metal ions to form metal-thiol compounds. Xanthate adsorption occurs via the catalytic oxidation mechanism forming dixanthogen, which can be represented by the following reactions:

$$ROCS^-_2 = (ROCS_2)\ ads + e-$$

$$Or\ (ROCS^-_2)\ ads = ROCS_2 + e-$$

$$Or\ 2(ROCS^-_2)\ ads = ROCS^-_2 + ROCS_2 + e-$$

These reactions are balanced electro-chemically by

$$\tfrac{1}{2} O_2 + 2H^+ + 2e- = H_2O$$

$$\tfrac{1}{2} O_2 + H_2O + 2e- = 2OH^-$$

$$O_2 + 2H^+ + 2e- = H_2O_2$$

The exact points to which collectors will attach themselves on mineral particle will depend on the micro semi-conductor topology of the surface. As may be recalled, the n-type semi-conductor conducts through electrons in the conductance band, and thus is an electron donor, while the p-type conducts through the movement of vacancies in the valency band and thus is an electron acceptor. Ideally, for rapid adsorption of collector molecules, the mineral particles should have n and p regions distributed on the surface, the p-type to provide a point of adherence of the collector, and n-type for the transfer of electrons to oxygen dissolved in the pulp, thus maintaining electroneutrality for each mineral particle (Crozier, 1991). In summary, a mineral surface must be a catalyst for the oxidation reaction as is the case for mineral surfaces with semi-conducting properties

2.6.2.3 EC-Mechanism

This mechanism represents the formation of bulk-like metal-thiol compounds at potentials greater than or equal to the thermodynamic potentials. In this mechanism, the mineral participates in the adsorption process to form a metal thiol compound on the surface. The mechanism of metal-thiol formation may be viewed as involving the electrochemical reaction (E):

$$MS = M^{2+} + S^{o} + 2e-$$

(Both of which represent oxidation of the sulphide mineral (MS) surface)

and a chemical reaction (C):

$$M^{2+} + 2X^{-} = MX_2$$

leading to the formation of a thiol compound MX_2 with metal component of the mineral, where X and M represent a thiol collector and metal cation. The overall adsorption mechanism can then be represented by an electrochemical reaction:

$$MS + 2X^{-} = MX_2 + S^{o} + 2e^{-}$$

The sulphur component of the mineral surface can also be oxidised to sulphur-oxy anions, such as sulphate, or thiosulphate in reaction leading to the formation of metal-thiol compounds for example:

$$2MS + 4X^{-} + 3H_2O = 2MX_2 + S_2O_3^{2-} + 6H^{+} + 8e^{-}$$

In the EC- mechanism, the mineral oxidation is controlled by the electrochemical potential (E_h), while the chemical step is controlled by the stability constant (pK) of the metal-thiol compound. In effect, E_h determines whether the metal-thiol compounds can be formed at given E_h. Thus, metal-thiol compounds can be formed when the mineral is easily oxidised, and / or when the metal-collector compound has a very large pK value. The former is the case for the flotation of chalcocite, galena, and silver with short chain xanthates, while the latter applies to the adsorption of thionocarbamates and other modified thiol collectors.

2.6.2.4 Metathetical Substitution

This mechanism involves Metathetical substitution of the oxidation product on a mineral surface by a thiol collector, which constitutes the 'chemical theory' proposed by Taggart et al, (1934). Although this mechanism does not directly involve a charge transfer process, it may be included in this classification since an electrochemical process forms the oxidation products. This mechanism occurs only on heavily oxidised sulphide minerals. When the collector is added, Metathetical reaction that could take place is:

$$M(OH)_2 + 2X^- = MX_2 + 2OH^-$$

The latter process is often considered as a chemical or ion exchange mechanism. However, since the initial oxidation is a corrosion-type process, it is readily accommodated within the electrochemical approach.

Richardson and Walker (1985) show that at potentials below the thermodynamic potential for copper xanthate formation (Eh = -70 mV), xanthate adsorbs to form chemisorbed xanthate. At potentials between –70 mV and 200 mV, copper ethyl xanthate (CuX)-like species are formed on the surface through an EC-type mechanism. In this potential range, the mineral is oxidised, releasing the copper ions with xanthate to form CuX on the surface. Thus, both the electrochemical potential of the system and pK of CuX control xanthate adsorption on chalcocite. At higher potentials where a considerable amount of copper ions is released into solution by oxidation of chalcocite, multilayers of bulk copper ethyl xanthate are formed.

On chalcopyrite, xanthate forms dixanthogen at potentials close to the reversible potential for the xanthate / dixanthogen couple. However, at higher potentials where chalcopyrite oxidises to liberate sufficient amounts of copper ions, CuX is formed and co-exists with dixanthogen. Dixanthogen is the only xanthate species identified on pyrite. There are no indications of underpotential deposition of xanthate on chalcopyrite or pyrite. Xanthate adsorption on these two minerals may therefore be ascribed to the catalytic oxidation mechanism.

2.6.3 Different Floatabilities of Sulphide Minerals

The structure and characteristics of sulphide mineral samples from different locations vary (Wards, 1996) and this results in differences in the flotation behaviour of samples (Bothelho de Sousa, 1984). The reactions of these minerals, such as the oxidation characteristics and interactions with water or other reagents used in flotation are known to strongly influence the conditions necessary for successful flotation.

The surfaces of sulphide minerals oxidise in the presence of water and oxygen, with the reduction of oxygen being facilitated by the semiconducting properties of the mineral. This oxidation hinders flotation performance, however in some circumstances reduced performance can be overcome by increased collector addition (Leja, 1982). Different sulphide minerals oxidise at different rates and the extent of oxidation and the product of oxidation depend on the mineral under investigation, pH, time of contact (Subrahmanyam and Forssberg, 1993). The general sulphide oxidisation proceeds as follows:

$$M_zS \longrightarrow zM^{n+} + S^0 + zne^- \qquad \text{(acid solutions)}$$

$$M_zS + znH_2O \longrightarrow zM(OH)_n + S^0 + znH^+ + zne^- \qquad \text{(neutral / alkaline)}$$

Further oxidation may form the oxy – sulphur species

$$xM_zS + (zxn + y)H_2O \longrightarrow zxM(OH)_n + S_xO^{2-} + (2y + zxn)H^+ + (2y + zxn - 2)e^-$$

In general the minerals which oxidise to form sulphur exhibit the highest flotation rates (Woods, 1984) and while the sulphur rich surface can promote floatability, the presence of oxy-sulphur species depresses flotation. Hydroxide species also suppress flotation by forming on the metal deficient sulphur surfaces.

It has been shown that the ease of collectorless flotation of sulphide minerals follows approximately the opposite ranking as the ease of oxidation (Guy and Trahar, 1984). The order of descending collectorless flotation is as follows: chalcopyrite > galena > pyrrhotite > pentlandite > covellite > bornite > chalcocite > sphalerite > pyrite > arsenopyrite.

Chalcopyrite is the anomalous mineral in the sulphide mineral series, as it has the highest natural floatability but not the lowest susceptibility to oxidation in the series. Eh has also been known to affect flotation performance and different minerals have potentials above which flotation is successful (Richardson and Walker, 1985).

2.6.4 The Role of Milling Media in the Flotation of Sulphide Minerals

Wet grinding in steel mills followed by flotation is the common practice adopted in concentrating mineral sulphide ores. It is known that the type of grinding media used in the previous grinding operation influences the flotation of sulphide minerals. The effect of different types of grinding (such as autogenous and ball mill) on the flotation of chalcopyrite, sphalerite, galena and pyrrhotite has been reported (Rey and Formanek, 1960; Thornton, 1973; Cline et al., 1974). The effect of electrochemical interaction between grinding media and a sulphide mineral and its effect on flotation has been studied in the past with respect to pyrrhotite (Pavlica and Iwasaki, 1982; Adam and Iwasaki, 1984) and galena (Learmont and Iwasaki, 1984). Beside the grinding media-mineral interaction, mineral-mineral interactions in a grinding mill may also affect the floatability of the sulphide minerals (Nakazava and Iwasaki, 1985, 1986). In many sulphide grinding systems, the steel ball materials act as anodes relative to the sulphide mineral which serves as a cathode. Galvanic interactions between the ball material and the ground mineral could therefore be expected in ball mill grinding which could affect not only the surface property of the ground mineral, but could also increase the corrosive wear of the grinding media. The sulphide fines produced, due to a large cathodic surface area, accelerate the anodic oxidation of the media. The dissolved iron ions produced as a result of the anodic oxidation of the grinding media react with the hydroxyl ions generated by the cathodic reduction of oxygen, thus forming hydroxyl complexes of iron, which are hydrophilic, and may coat the mineral surface. Such coatings may impair the floatability of the sulphide mineral by interfering with xanthate adsorption (Rao and Natarajan, 1990). Besides such a surface reaction, caused by migration of anodically dissolved ions, increased surface passivation and tarnishing will occur on the cathodic mineral due to oxygen reduction at its surface.

Galvanic interactions (electron transfer) between minerals and media are one of the more recently proposed models of media and mineral effects in flotation (Adam and Iwasaki, 1984; Kocabag and Smith, 1985). The principle is illustrated by Figure 2.6.4, and is described below.

1) Mild steel media is anodic relative to sulphide minerals, leading on contact to a flow of electrons from the steel to the mineral:

$$Fe \longrightarrow Fe^{2+} + 2e^-$$

2) As a result, the media will corrode at an accelerated rate and the extra electrons will lead to the reduction of dissolved oxygen at the mineral-water interface, and reaction with water to form hydroxyl ions. The dissolved ferrous species would further react with the hydroxyl ions generated at the cathodic sulphide mineral sites forming oxy-hydroxide species:

$$O_2 + 2H_2O + 4e^- \longrightarrow 4OH^-$$

$$\begin{array}{ccc} Fe & \longrightarrow & Fe^{3+} \\ \downarrow & & \downarrow \\ Fe(OH)_2 & & FeOOH, Fe(OH)_3 \end{array}$$

3) The presence of hydroxyl ions close to the mineral surface accelerates the dissociation of the mineral surface and leads to the formation of a stable coating of metal hydroxides, which inhibits the collector reaction with the mineral surface.

In multimineral combination, an electrode with the most positive potential will always be cathodic and that with the most negative potential will be anodic (Tomashov, 1968). Thus in the multimineral combinations in the presence of steel ball material, the sulphide minerals will be cathodic (most positive), and the grinding media anodic (most negative). Such galvanic interactions occur between minerals and stainless steel media, but are stronger between mild steel media and sulphide minerals by several orders of magnitude (Cline et al., 1974). Corrosion of mild steel, consumes oxygen and lowers the pulp potential (Graham and Heathcote, 1982; Johnson et al., 1982).

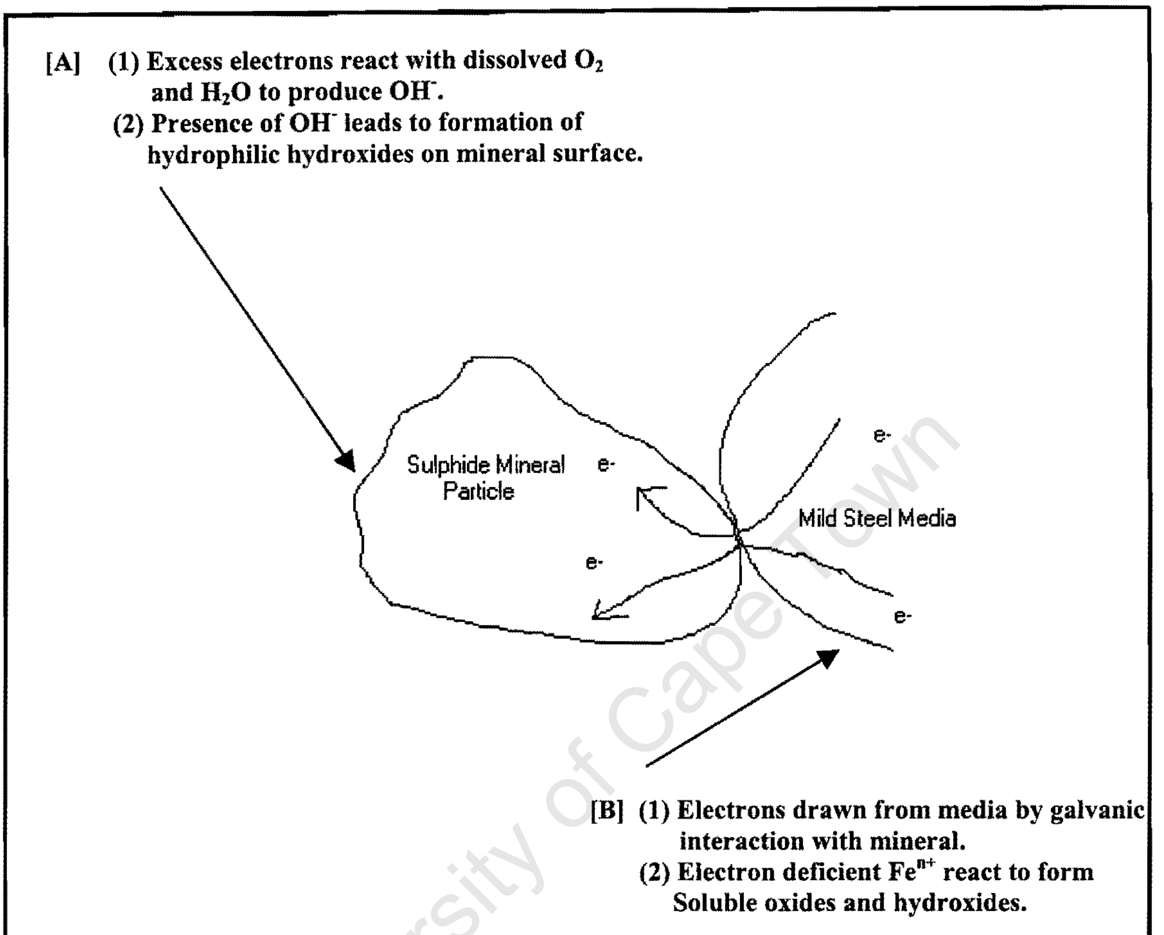


Figure 2.6.4: Mechanism for galvanic interaction between mineral and media leading to (a) hydrophilic sulphide mineral surface and (b) accelerated media corrosion (Kocabag and Smith, 1985).

Therefore, mild steel grinding media has the following effect on flotation:

- Lowers dissolved oxygen concentration
- Lowers pulp potential
- Increases opportunity for galvanic interaction between mineral and media.
- Increases iron hydroxyl species in solution, which coat the sulphide mineral surfaces and impair collector adsorption. These species may also stabilise the froth zone and cause increased recovery and reduced grade.

2.7 Flotation of Oxide Minerals

The recovery of oxidised copper minerals from predominant sulphide ore bodies is an important technological problem facing the mineral processing industry. Considerable work has been carried out in the past to increase the recovery of oxidised copper minerals by flotation (Soto and Laskowski, 1973; Castro et al., 1974a,b; Queirolo and Castro, 1976; Decuyper, 1977; Aplan and Fuersteneau, 1984; Fuersteneau and Pradip, 1984). Based on these and other studies, the various methods available for flotation of oxidised copper ores are:

(1) sulphidisation followed by flotation with a xanthate collector;
(2) carboxylic acid process;
(3) leach-precipitation-flotation process;
(4) direct flotation by using a variety of ionic, long chain sulphydryl, and chelating reagents.

The first three of these methods are practised in industry. However, the carboxylic acid method is not applicable for the ores containing carbonate gangue minerals due to the simultaneous flotation of copper oxide and gangue minerals (Gaudin, 1957). When significant quantities of sulphide minerals are present in the ore, the third method is not very effective due to loss of sulphide components in the leaching stage. Therefore, for the mixed sulphide /oxide copper ores the sulphidisation-flotation process is the most widely used and effective method for copper recovery.

2.7.1 Oxide Copper Recovery by Sulphidisation

Oxidised copper minerals without prior sulphidisation generally do not float with low molecular weight thiol collectors at the concentrations that are normally employed in the flotation of sulphide minerals. In such a system, the reaction products between the collector and the metal ions in the mineral do not adhere well to the surface (Wright and Prosser, 1965; Bustamante and Castro, 1975; Castro et al., 1976). The role of sulphidisation, according to these researchers, is therefore to provide a copper sulphide layer on the mineral surface, which stabilises the collector coating. The following equation was proposed for malachite sulphidisation with hydrosulphide (Bustamante and Castro, 1975):

$$CuCO_3 \cdot Cu(OH)_2 + HS^- + OH^- \longrightarrow Cu(OH)_2 \cdot CuS + CO^{2-}_3 + H_2O$$

This reaction is not confined to the surface, but continues within the bulk of the particles to form copper sulphide coatings. The extent of reaction in excess of monolayer was also observed in sulphidisation of tenorite (Castro et al., 1974a).

Even though sodium sulphide is one of the most widely used alkali metal sulphides in the sulphidisation-flotation process, its effectiveness is not entirely satisfactory. Sulphidisation treatment with such a reagent can enhance the hydrophobicity of the oxide minerals, but the sulphidized particles require quiescent conditions for flotation (Fuersteneau and Raghavan, 1986), possibly due to the fact that the sulphidized layer may detach readily from the surface. When an alkali sulphide is used as the reagent, a careful control of the sulphidisation stage is critical to maximising the flotation recovery (Malgan, 1986). An insufficient amount of sulphide gives poor recoveries because of inadequate sulphidisation, while an excess of sulphide causes poor flotation due to the depressant action of sulphide ions. A major difficulty in sulphidisation is to establish and maintain an optimum sulphide concentration.

The microstructure of the oxidised mineral surface seems to play an important role in the sulphidisation-flotation process. The effect of sulphidisation on the porous nature and the surface area of brochantite and chrysocolla have been investigated (Raghavan et al., 1984). The surface area of brochantite was substantially reduced upon sulphidisation and the flotation was improved. In contrast, it was observed that sulphidisation neither affected the surface area nor the flotation response of chrysocolla, which had an extraordinarily large surface area.

The effect of thermal treatment on the flotation behaviour of chrysocolla was studied by several investigators (Parks and Kovacs, 1966; Castro et al., 1974b; Queirolo and Castro, 1976; Gonzalez and Soto, 1978). Thermally activated chrysocolla was found amenable to the sulphidisation-flotation process. The pore structure of chrysocolla was studied and found to contain a major fraction of micropores (Raghavan and Fuersteneau, 1977). The use of gas adsorption technique revealed that heat treatment of chrysocolla at 555°C sintered pores whose radii were below 1.5 nm, and thus reduced the specific surface area. The thermal activation of chrysocolla flotation was ascribed to the reduction in the number of pores where precipitates could form.

2.7.2 Controlled Potential Sulphidisation

It has been indicated that in the flotation of presulphidised oxidised ores, the sulphidisation stage is critical in that the addition of either too much or too little sodium sulphide results in poor metallurgy (Rey 1953). Due to a delicate balance between the sulphidising and the depressing effects of the hydrosulphide ions, the industrial application of sulphidisation has proved difficult. Several plant applications practise slug additions where control of hydrosulphide and sulphide ions is absent and usually results in excess additions of the sulphidiser. Jones and Woodcock (1979) developed the controlled Potential Sulphidisation (CPS) technique to control and regulate sulphidiser additions by using an ion-selective electrode. The potential for CPS has been demonstrated in effecting substantial improvements in metallurgical performance over that obtained using conventional sulphidisation. Apart from the faster flotation rates, the simultaneous application of CPS on both sulphide and oxide copper minerals also increased overall recovery and produced superior concentrate grades (Jones, Wong and Woodcock, 1986).

2.7.3 Recovery of oxidised copper sulphide minerals by Nitrogen gas and Sulphidisation Conditioning (Maxifloat^{TM})

Success of the use of nitrogen conditioning prior to sulphidisation to recover oxidised sulphide minerals has been reported (Clark et al., 2000). BOC Gases is pursuing commercially the application (Clark et al, 1995) of this in a process called Maxifloat^{TM}. The nature of the improvements with copper sulphide bearing ores using Maxifloat^{TM} has been described (Clark et al, 2000). Clark et al (2000) found that by using nitrogen gas to exclude oxygen and by careful control of the sulphidisation process it was possible to increase the flotation recovery of chalcocite, chalcopyrite and bornite.

The technique of applying the Maxifloat^{TM} conditioning step incorporated the following features that were varied according to ore characteristics (Clark et al, 2000):

1. Application was prior to either rougher or scavenger flotation.
2. Nitrogen gas was added to control the dissolved oxygen concentration before, during and after sulphidisation conditioning. Typically, dissolved oxygen values of less than 2 ppm are appropriate to initiate sulphidisation in the presence of nitrogen.

3. During sulphidisation, sufficient sulphidising agent was added to maintain target E_s values. Typically this was between –300 mV and –600 mV. Sulphidising agent (NaHS) was added in liquid form directly to the ore pulp stirred in the flotation cell. Cell agitation was adjusted to minimise the induction of air during sulphidisation conditioning.
4. The time taken in the conditioning step was found to be of some importance. Generally this time was between 1 and 3 minutes.
5. Collector addition could be made either prior to sulphidisation or afterwards.
6. Some aeration after sulphidisation was advantageous to promote collector reactions for flotation.
7. Air was used as a flotation gas.

From their Bench scale studies, the improvements from Maxifloat™ were between 0.6% to 9.5% additional copper recovery for various mineral assemblages examined at the same concentrate grade. In the pilot plant studies, the Maxifloat™ process improved copper recovery by 2.2%, significant at the 95% level of confidence (Clark et al, 2000).

During the bench scale studies, a significant amount of learning was gained in the investigations and the most pertinent of which has been summarised as follows (Clark et al, 2000):

(a) Certain ores respond well to Maxifloat™. Ores containing tarnished or oxidised primary copper minerals in tailings, secondary copper minerals and native copper show the greatest improvements.

(b) By using nitrogen gas to exclude oxygen and by careful control of the sulphidisation process it is possible to increase the flotation recovery of chalcocite, chalcopyrite and bornite. The additional recovery is believed to be from particles with oxidised or tarnished surfaces where surface oxidation has been reversed.

(c) The use of nitrogen to exclude oxygen increases the efficacy of new sulphide surface formation and improves process efficiencies by maximising the availability of sulphide ions for surface sulphidisation to occur at the lowest sulphide consumption levels.

(d) The use of nitrogen conditioning, even in the absence of sulphidisation, has been found to have positive metallurgical effects on certain ore types. It appears that, in the absence of dissolved oxygen, some ores can generate sulphide ions, effectively allowing autogenous sulphidisation to occur, with attendant improvement in metal recoveries.

(e) NaHS is the preferred sulphidiser causing less change to slurry pH while providing more active sulphidising species. Small addition rates of sulphidiser are often sufficient.

(f) For ores containing chalcocite, chalcopyrite and bornite, the preferred location for applying the Maxifloat^{TM} is the tail of the Rougher flotation circuit.

(g) Collector addition could be made either prior to sulphidisation or afterwards. Sulphidising reagents are capable of reconverting oxidised collector species into active forms. This is also believed to contribute to the additional copper flotation recovery (Garip et al, 1998).

The presence of dissolved oxygen during sulphidisation conditioning is expected to adversely affect the process efficiency in the following three ways (Clark et al, 200):

1. Sulphidising reagent can react with oxygen to form sulphoxy species such as thiosulphate ions ($S_2O_3^{2-}$), sulphite ions (SO_3^{2-}) and sulphate ions (SO_4^{2-}) (Hecker et al, 1985). As a result, sulphidising ions (HS^-) are consumed by dissolved oxygen in significant quantities, necessitating additional quantities of sulphidiser to be used in order to achieve satisfactory surface sulphidisation.

2. Dissolved oxygen is known to affect the performance of both platinum and silver/silver sulphide electrodes (Hecker et al, 1985) thereby impairing reliable potential measurements. The interference of dissolved oxygen with the potential measurement is more than due to its contribution to the Eh value (>50 mV). Electrode manufacturers recommend that special precautions be taken to exclude dissolved oxygen from the system being measured. This includes using techniques such as nitrogen purging (Kokholm).

3. Dissolved oxygen would react with the freshly formed sulphide layer. Sulphidisation of secondary copper minerals proceeds through a two-stage process (Zhou et al, 1993). The first stage involves the formation of a primary sulphidised layer and the second involves precipitation of copper ions that have diffused through the primary layer. The presence of dissolved oxygen is believed to disrupt these reactions by reacting with freshly formed sulphide surface and thereby resulting in a spongy, less coherent sulphidised layer. This is evident from the leach test performed on sulphidised copper minerals in the presence of oxygen and nitrogen atmospheres (Orwe et al, 1997). A coherent, compact sulphidised layer can only be achieved by eliminating dissolved oxygen with nitrogen purging as is done in the Maxifloat™ process. As an example, the application of Maxifloat™ on malachite bearing ores resulted in economic recoveries being achieved in fewer stages than compared to traditional controlled potential sulphidisation (CPS) methods (Wong, 1994).

2.7.4 Oxide Copper Recovery with Alkyl hydroxamate

Oxide copper minerals generally do not respond well to traditional methods of concentration using known sulphide copper collectors. Their recovery in a froth flotation circuit requires special treatment. The traditional method involves sulphidisation (at –500 to –600 mV vs. combination Sulphide Ion Electrode) using sodium sulphide (Na_2S), sodium hydrosulphide (NaHS), or ammonium sulphide ($(NH_4)_2S$) followed by flotation using xanthate or other sulphide collectors (Jones et al., 1986; Nagaraj and Gorken, 1989). In principle, this method is quite attractive, but in practice it suffers from two disadvantages. The first is that it is difficult to control the dosage of the sulphidising agent; excess causes depression of both sulphide and oxide minerals, and an insufficient amount produces poor recoveries. Secondly, the different oxide minerals respond differently to sulphidisation (Nagaraj and Gorken, 1989; Soto and Laskowski, 1973; Castro et al., 1974; Deng and Chen, 1991), and frequently sulphidisation simply fails to provide acceptable oxide copper recovery.

A variety of collectors has been proposed for oxide copper flotation without sulphidisation in the past six decades. These include a large number of complexing agents, fatty acids, fatty amines, and petroleum sulphonates (Nagaraj, 1979; Nagaraj, 1987; Deng and Chen, 1991). Except for a very limited use of fatty acids (which are quite non-selective), none of the proposed reagents has been used in an operating plant, though a large number of collectors have shown considerable promise in laboratory tests. The high cost, high consumption, and inadequate performance are three important drawbacks for the majority of the proposed collectors. Alkyl hydroxamates, however, are among the very few collectors that have shown the most promise. They are available commercially, and they offer many advantages over alternative technologies for oxide copper recovery.

The use of alkyl hydroxamates as a collector for non-sulphide minerals is well documented. Popperle (1940) first introduced the use of hydroxamic acids or their salts as collectors in ore flotation. Fuerstenau and Peterson (1965, 1969) used alkyl hydroxamates for the flotation of chrysocolla and suggested that flotation was due to the formation of an insoluble complex between surface metal ions and hydroxamate. Evrad and De Cuyper (1975) found the use of alkyl hydroxamate for floating copper-cobalt oxide ores advantageous.

Danilova et al. (1975) noted the benefits of using alkyl hydroxamates to recover chrysocolla without prior sulphidisation. Lenormand et al. (1979) discussed the mechanism of adsorption of potassium octyl hydroxamates on malachite. Fuerstenau and Pradip (1984) reviewed the abundant literature available on the use of hydroxamates in flotation and the mechanism of adsorption on oxide minerals. Dekun et al. (1984) and Rule (1982) described the use of hydroxamic acids for natural ore flotation of oxide copper ores.

It is evident that much of the published literature deals with mechanistic aspects of hydroxamate interaction with oxide minerals and laboratory flotation of single minerals. Although, there is little useful information in the literature on the practical aspects of the use of alkyl hydroxamates in a flotation plant, there is certainly overwhelming evidence to support the view that alkyl hydroxamates should indeed be excellent collectors for oxide copper flotation, and that they offer many advantages.

In spite of the many positive attributes of hydroxamates, their large-scale usage is not fully exploited. Several reasons have been proposed to explain this (Lee et al. 1998):

1. The very low oxide content in the ores does not justify added cost of recovery by flotation.
2. The use of alkyl hydroxamate has been limited to academic interest.
3. It is generally assumed that sulphidisation-flotation is the preferred method.

4. Practical guidelines for the use of alkyl hydroxamate in a plant are not available. Most of the available literature has done little to provide such guidelines.
5. On some ores the desired results were not obtained with hydroxamate for reasons unrelated to the chemistry of the collector.
6. Oxide copper in the ore is perhaps not recoverable by flotation.
7. Insufficient efforts have been made to demonstrate the efficacy and cost benefits of using alkyl hydroxamate in plant.

It must be noted that the discussion above pertains to oxide copper that is typically associated with economic amounts of sulphide copper and, consequently, not treated by leaching-SX (the latter is widely practised on low-grade ores from the "oxide zone" that are not treated by flotation). In specific cases where the oxide content in the mixed sulphide-oxide ore is relatively high, the tailings produced after maximum sulphide recovery, and partial oxide recovery, may be leached to recover oxide minerals that were not recovered in flotation. Sulphidisation-flotation has been practised in the industry, often with some degree of success, depending on the mineralogy of the oxide minerals in the ore (Deng and Chen, 1991). Such is not the case with alkyl hydroxamate.

2.7.5 Sulphidisation Activators of Oxide Copper Minerals

Zhang (1993) tested several alternative sulphidising activators on Chinese copper oxidised ores at the Dongchuan Copper Mining Bureau (DCMB). The activators were calcium sulphide, ammonium sulphate and ethylene diamine phosphate. These are included for the reader's interest for future work but are not part of this thesis.

Calcium Sulphide

In the early 1970s because of the shortage of sodium sulphide, DCMB had to substitute calcium sulphide for sodium sulphide. Dongchuan ore is rich in high-quality gypsum and as such reduction roasting gypsum powder with coal powder as reducing agent produced calcium sulphide. Calcium sulphide was then added as powder into the primary and secondary mills to grind with the ore. Sulphidisation of oxidised copper minerals took place during the grinding time and the subsequent conditioning time. However, the improvement in flotation recovery with CaS was marginal for both oxide and sulphide copper minerals.

Ammonium Sulphate [$(NH_4)_2SO_4$]

Zhang et al. (1973) have shown in both laboratory and plant tests that copper recoveries in flotation of oxidised copper ores and the selectivity of the process can be markedly improved by use of ammonium sulphate as a kind of activator.

For Yinmin oxidised copper ores (China), for example, the addition of ammonium sulphate improved oxide copper recoveries by the order of seven per cent to 11 per cent. A further study by Zhang and Poling (1989) has shown that prior or simultaneous addition of ammonium sulphate can not only enhance the activating effect of hydrosulphide ions on xanthate flotation of malachite but it can also eliminate possible detrimental or depressing effects of excess residual hydrosulphide. Use of relatively inexpensive ammonium sulphate appears to reduce the consumption of the much more expensive sulphidising agent for optimal flotation recovery.

Detailed studies of surface chemistry on the role of ammonium sulphate in a malachite system by Zhang and Poling (1991) has revealed that ammonium sulphate has three promoting effects on sulphidisation of malachite, namely, the 'catalytic effect' on sulphidising reactions, the stabilising effect on copper sulphide coating formed on malachite, and the hydrophobic effect on the sulphidised surface of malachite. Therefore, the two researchers have suggested ammonium sulphate to be termed ' sulphidisation-promoter'.

Ethylene Diamine Phosphate

Ethylene diamine phosphate (EDP) is a new type of flotation activator for oxidised copper minerals. The research centre of DCMB (Dongchuan Copper Mining Bureau, China) originally created it in 1974. It is prepared by reacting ethylenediamine and phosphoric acid:

$$NH_2CH_2CH_2NH_2 + H_3PO_4 \longrightarrow NH_2CH_2CH_2NH_2 \cdot H_3PO_4$$

EDP is a crystalline powder and readily soluble in water.

As Hu (1981) reported, based on the preliminary tests in laboratory, plant tests of using EDP as flotation activator were conducted in three concentrators from December 1974 to September 1975. The results showed that when EDP was added, the recoveries of oxide copper increased by 5.59 per cent to 11.24 per cent and the recoveries of sulphide copper increased by 0.12 per cent to 1.75 per cent with the grades of concentrates maintaining at the same levels. In addition, the consumption of sodium sulphide reduced by 34 per cent and butyl xanthate up to 43 per cent.

It has been shown by Zhang (1980) that the reaction mechanism of EDP is enhancing the adsorption rate and density of sodium sulphide and xanthate on copper minerals and hence the floatability of the minerals.

2.7.6 Summary of the Methods of Oxide Copper Recovery

Sulphidisation-flotation process, from all the methods of oxide copper recovery discussed above, is the most widely used in industry (Nchanga Concentrator in Zambia and Palabora Copper Mine Concentrator in South Africa).

In spite of the many positive attributes of hydroxamates, their large-scale usage is not fully exploited for reasons explained in section 2.7.3. The successful use of sulphidisation promoters has been reported in China.

The recovery of oxidised copper sulphide mineral by nitrogen gas and sulphidisation by a process called Maxifloat^{TM}, has been successful on some ores as discussed in section 2.7.2.

In this research two methods were applied: (1) sulphidisation-floatation process, and (2) sulphidisation in the nitrogen environment.

CHAPTER 3

Nchanga Concentrator Operations

3.0 NCHANGA CONCENTRATOR OPERATIONS

3.1 Historical Background

The ore body at Nchanga was discovered in 1923, near the Nchanga stream. Development of the mine commenced in 1927 and centred around the Dambo and River Lodes situated to the north and east of the present Nchanga Open Pit. Ore reserves estimated at 146 million tonnes with an average copper content of 4.66 were disclosed in 1931.

Preliminary work on the mine started in November 1936 and concentrated around the Nchanga West Ore body. Later, in 1939, a pilot concentrator with a capacity of 508 tonnes of ore per day was commissioned. Subsequently, a full-scale concentrator with a capacity of 4 064 tonnes of ore per day was commissioned in February 1946.

The decision to mine the lower grade Nchanga ore body by open pit methods was undertaken in 1954. In 1957, ore production from the Nchanga open pit began and extensions to the concentrator were completed in the same year to cope with the increased production.

In 1967, the East Mill was commissioned and the plant is rated at 500 000 tonnes of Open Pit ore per month. The ground product is pumped to the West Mill Flotation section. Cobalt ore treatment circuit was commissioned and incorporated in the concentration process in January 1986.

3.2 Nchanga Concentrator

The Nchanga Concentrator is made up of the West Mill and the East Mill sections. The East Mill handles copper ore from the Open Pits and after crushing and grinding, the pulp is pumped to West Mill for flotation. Underground copper and open pit cobalt ores are ground and floated in the West Mill flotation section.

3.2.1 Underground Copper Ore Circuit

The Underground copper ore circuit (West Mill) treats about 8 000 tonnes per day of ore at an average grade of 3.00 % TCu. The flowsheet is given in Figure 3.2.1. The ore is primary crushed to 6 inches in three 30-inch Allis Chalmers, Superior-McCully gyratory crushers. Secondary and tertiary crushing is done using 5 ½ ft short head symons crushers.

The fine crushed ore (3/4 of an inch) is ground in nine 8ft x 9ft Headwrightson overflow type ball mills, each in closed circuit with a single unit 20-inch Krebs hydrocyclone, to produce a feed to flotation (cyclone overflow) with a size distribution of 55-60% minus 74 microns and 15-20% plus 150 microns.

The ground ore is floated in one bank consisting of six 1500 cubic ft Wemco cells as sulphide roughers to produce a concentrate grade at 5-10%TCu. A high-grade concentrate at 45%TCu is produced from the two 72-inch column cells through single stage cleaning of sulphide rougher concentrates. The column tailings are scavenged in the sulphide cleaner and re-cleaner mechanical banks to produce another medium grade concentrate for smelters.

The sulphide rougher tailings at 1.5%TCu are rougher floated in the oxide rougher circuit and the concentrate cleaned to produce a medium grade concentrate at 30-35%TCu which is sent to the smelters for further processing. The final tailings after oxide roughing are pumped to the Tailings Leach Plant for hydrometallurgical processing.

3.2.2 Open Pit Copper Ore Circuit

The Open pit copper circuit (East Mill) is rated at 15 000 tonnes of feed to flotation per day. The flowsheet is given in appendix A. The ore is primary crushed in a 54-inch by 74-inch Allis Chalmers Superior Gyratory crusher to less than 6 inches and then washed and secondary crushed to ¾ inch in two 7 ft Nordberg Symons Short Head Cone crushers. The crushed ore is ground in three 11.5ft by 14.5ft Vecor rod mills. The rod mill discharge is further reground in five 11.5ft by 14.5ft Vecor overflow discharge ball mills, each in closed circuit with a double unit 20-inch Krebs hydrocyclone to produce a cyclone overflow feed to flotation of 55-60% minus 74 microns and 15-20% plus 150 microns.

The ground product is floated in two banks of six 1500 cubic ft Wemco cells each as sulphide roughing stage to produce a concentrate grade of 5-10%TCu. This concentrate joins the scavenger re-cleaner tails as feed to the two 15ft cleaner columns to yield a high grade concentrate of 45%TCu after re-cleaning in the 8ft column cell. The sulphide rougher

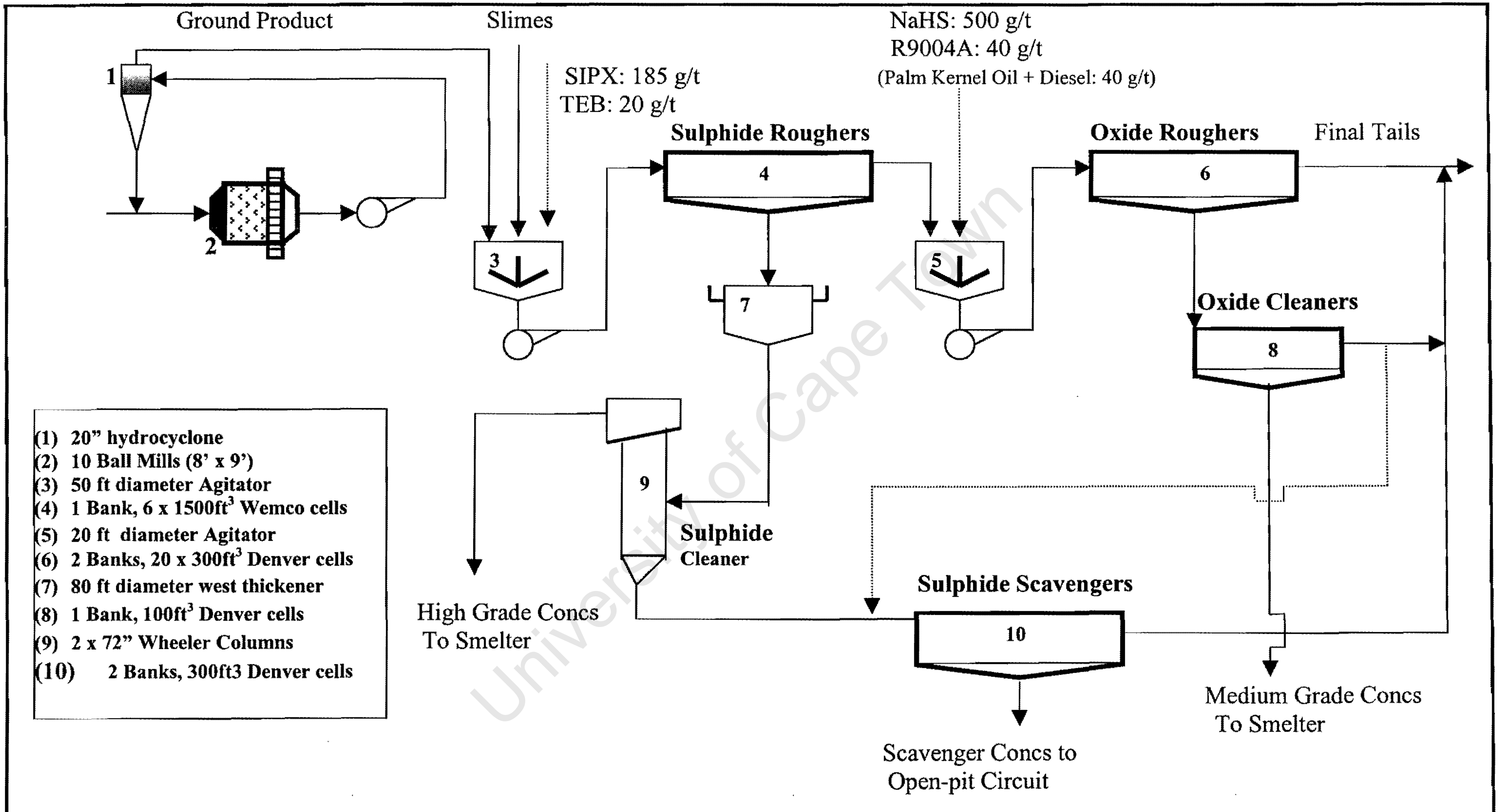


Figure 3.2.1: Nchanga Underground Concentrator Copper Circuit

tailings at 1.00-1.30%TCu are pumped to the Tailings Leach Plant for hydrometallurgical processing. The flowsheet is given in appendix A.

3.2.3 Open Pit Cobalt Ore Circuit

The cobalt circuit at West Mill treats 2 300 tonnes per day of cobalt ore from the Nchanga Open Pit at a grade of 1.00-1.50%TCu and 0.30-0.80%TCo. The ore is delivered by trucks and crushed to less than 6 inches in a 54-inch by 74-inch Allis Chalmers Superior Gyratory primary crusher. It is fine crushed to 1.5-inch in two 5.5ft Symons short head cone crushers in closed circuit with 1.5-inch x 1.5-inch aperture screens. The minus 1.5-inch product is discharged to the cobalt mill feed bins and subsequently ground in one 9ft by 12ft Headwrightson overflow discharge rod mill. The rod mill product is reground in three 9ft by 8ft Headwrightson overflow discharge ball mills in closed circuit with 20-inch Krebs hydrocyclones. The flotation flowsheet is given in appendix A.

The cyclone overflow is floated in three rougher banks of twenty-45 cubic ft cells and two banks for scavenging. The rougher concentrates are cleaned in a 72-inch column. The final cobalt concentrate assays 7-14%TCu and 3.0-8.0%TCo and 8-15% S.

3.3 Ore Mineralogy

3.3.1 Copper Ore

Nchanga copper ore is a complex mixture of sulphide and oxide minerals. The sulphides are the primary copper minerals. The main sulphide minerals in order of decreasing abundance are given in Table 3.3.1 (a). Associated with the copper sulphides are carrollite and pyrite.

Table 3.3.1 (a): Sulphide Minerals Present in Nchanga Copper Ore.

Mineral	Formula	RA%	Wt%	%TCu	%ASCu
Chalcocite	Cu_2S (79.8% Cu)	62.50	2.73	2.18	0.03
Chalcopyrite	$CuFeS_2$ (34.65% Cu)	26.31	1.15	0.40	-
Bornite	Cu_5FeS_4 (63.3% Cu)	3.35	0.15	0.09	<<0.01
Pyrite	FeS_2	7.17	0.31	-	-
Carrollite	Co_2CuS_4 (15.4% Cu)	0.38	0.02	<<0.01	-
Native Copper	Cu (100% Cu)	0.29	0.01	0.01	-
Total		100.00		2.68	0.03

RA = Relative Abundance

The copper 'oxide' minerals include the copper silicates, sulphates and carbonates. In general, this includes all copper ores where oxidation and weathering have proceeded past the stage of chalcocite and native copper to yield the carbonates, malachite, azurite, chrysocolla and related minerals. The major copper oxides present in the ore with their respective approximate percent relative abundance are shown in Table 3.3.1 (b). Cupriferous mica is also present in minor amounts. Detailed mineralogical analysis is given in appendix E.

Table 3.3.1 (b): Oxide Minerals Present in Nchanga Copper Ore.

Mineral	Formula	RA%	Wt%	%TCu	%ASCu
Malachite	$Cu_2(OH)_2CO_3$ (57.6% Cu)	90.0	1.97	1.13	1.13
Pseudomalachite	$Cu_5(PO_4)_2(OH)_4$ (55.2% Cu)	4.0	0.09	0.05	0.05
Chrysocolla	$CuOSiO_2.nH_2O$ (36.2% Cu)	4.0	0.09	0.03	0.03
Azurite	$Cu_3(OH)_2(CO_3)_2$(55.53% Cu)	1.0	0.02	0.01	0.01
Cuprite	Cu2O (88.8% Cu)	1.0	0.02	,0.02	0.02
Cu/mica	Cu-Vermiculite	-	0.2-0.3	0.01	<<0.01
Total		100.0		1.25	1.24

3.3.2 Gangue Minerals

The gangue minerals present in the ore include: quartz, feldspars, mica, talc, iron 'oxides', carbonates and carbonaceous shale. The approximate percent relative abundance is given in Table 3.3.2.

Table 3.3.2: Approximate Relative Percentage Distribution of Gangue in Nchanga Copper Ore.

Gangue	%RA
Quartz/Feldspars	80-85
Carbonaceous shale	11-13
Carbonates	2-3
Mica	2-3
Talc	<<1
Argillite	<<1
Iron 'Oxides'	<1
Accessories	-
TOTAL	100

3.4 Oxide and Sulphide Copper and its Significance on the Copperbelt

The term oxide copper is loosely applied on the Copperbelt to all copper minerals that are not chalcopyrite, bornite, chalcocite or native copper. These so called oxide copper minerals include a large number of copper phosphates, carbonates and silicates whose only common properties, apart from being formed as a result of weathering, are that they are often green and to some extent dissolve in dilute sulphuric acid solution. A good deal of confusion arises from time to time on the various interpretations of the term oxide and sulphide copper. To the mineral dressing engineer it is convenient to divide the copper content or copper minerals in an ore sample into sulphide and oxide copper. This supplies basic information which determines not only the concentration process that can be applied to the ore but also the method of copper extraction which will be most suited to the copper concentrates produced. For example, if the deposit consists entirely of sulphide copper minerals, then gravity concentration, or flotation concentration using straight xanthate type collectors, may be applicable for its treatment. But where the ore deposit contains only oxide copper minerals consideration can be given to bulk acid or alkaline leaching, reduction roasting or, if it is not economic to treat the whole of the ore, to the application of gravity concentration or separate sulphide-oxide flotation (O'Meara, 1961).

Many non-sulphide copper minerals are somewhat soluble in dilute sulphuric acid solution. One of the quickest ways of supplying the mineral dressing engineer with basic information is to assay the ore sample for total copper and for the copper that is soluble in dilute sulphuric acid, the difference thus obtained being taken as sulphide copper. This analytical method would be generally satisfactory if the ore sample contained say, only a mixture of chalcopyrite or only malachite, for the former is insoluble in dilute sulphuric acid, sulphur dioxide solution (up to a period of two hours) and the latter is completely soluble. A true indication of the sulphide and oxide copper content of the ore would thus be obtained. On the Copperbelt, however, the mineral composition of mixed oxide/sulphide copper deposits, such as those which occur at Nchanga, are not as simple as the example outlined above. A dilute acid leach of an ore sample or concentrate does not always give a true picture of the sulphide and oxide copper content of the sample. Such a leach will, however, usually give a rough working estimate of the oxide copper content of an ore sample and for this reason an empirical acid leach assay method has been in use at some mines for many years (Chevel and Pierrot, 1954).

To the analytical chemist, therefore, the term oxide copper has come to be synonymous with acid soluble copper. The following empirical assay method has been in use for several decades to determine acid soluble copper in ores and metallurgical products. Acid soluble or oxide copper is that which is taken into solution when a sample is treated at room temperature with a 90 g/l sulphuric acid solution saturated with sulphur dioxide and with continuous mechanical agitation for one hours. The difference between total copper and the acid leached copper was, until recently, termed sulphide copper. The sulphide copper thus determined is now referred to as acid insoluble copper, since the copper minerals that are left in the leach residue are not always true sulphide minerals. Many anomalies arise in the use of the terms oxide and sulphide copper when such copper has been determined by a sulphuric acid/sulphur dioxide leach. Some of the implications involved in connection with plant operation are (O'Meara, 1961):

(a) A small percentage of true sulphide copper present as chalcocite and bornite reports as acid soluble or oxide copper.
(b) Native copper reports by assay as sulphide copper.
(c) Hydrated copper silicate chrysocolla reports as oxide copper.
(d) From a mineralogical viewpoint cuprite is the only true oxide copper mineral. But interestingly, it does not entirely dissolve when leached by the sulphuric acid /sulphur dioxide leach method, and consequently some of this true oxide copper reports as sulphide (acid insoluble) copper.
(e) The copper in cupriferous vermiculite reports partly as oxide copper and partly as sulphide or non-acid soluble copper when acid leached.
(f) Any copper present as the hydrated copper phosphate mineral, pseudomalachite reports as oxide copper.

It can be appreciated from the foregoing description of the copper minerals and some of their properties that it is preferable if copper or copper minerals are referred to as acid soluble and acid insoluble. The terms, acid soluble and acid insoluble, have been adopted in preference to sulphide and oxide copper (O'Meara, 1961).

3.5 Differential Sulphide-Oxide Flotation at Nchanga

In the Nchanga mixed sulphide/oxide orebodies the chief sulphide minerals are chalcocite, associated with small amounts of chalcopyrite, bornite, pyrite and carrollite. Chalcocite is also associated with some native copper and cuprite, and all of the copper in the former mineral and half of the copper in the later reporting, as already discussed, as acid insoluble or sulphide copper. The differential sulphide/oxide flotation operation aims at floating all of the above minerals into a sulphide concentrate and the remaining non-sulphide copper minerals into the oxide concentrate. The remaining minerals are chiefly malachite, with smaller amounts of pseudomalachite, azurite, chrysocolla and cupriferous vermiculite. All of these 'oxide' minerals, with the exception of cupriferous mica, are completely soluble in dilute sulphuric acid solution.

Separate sulphide and oxide concentrates are made at the Nchanga concentrator from washed ore, washing plant slimes and open pit flotation feed respectively. The sulphide concentrates are made by flotation at a pH of about 9.3 using Isopropyl xanthate collector and TEB frother whilst the oxide concentrates are made from the respective sulphide rougher tailings by the addition of sodium hydrosulphide sulphidiser, palm kernel/fuel oil mixture, and additional frother as required.

The problem of oxide and sulphide terminology enters into this aspect of concentrator operation for the aim is to produce acid insoluble and acid soluble concentrates. However, it is fortunate at Nchanga that most of the acid insoluble copper minerals that report into the sulphide concentrates are in fact true sulphide minerals. The only exception is native copper and cuprite. As discussed in the previous section, cuprite is normally the chief contributor to acid soluble copper in Nchanga sulphide concentrate, the only other acid soluble mineral of any significance being malachite. However, as very little of the acid soluble copper in the sulphide concentrates is present as malachite, the flotation of so-called oxide copper minerals into the sulphide concentrates generally gives little cause for concern. The chief problem in differential sulphide/oxide flotation, therefore, is not the presence of acid soluble copper in the sulphide concentrates but of acid insoluble copper in the oxide or leach grade concentrates. If the oxide concentrates are destined for smelting, as they are now at Nchanga, then the presence of acid insoluble copper in the oxide concentrates is not significant. When the various Nchanga ores are ground to the optimum fineness for flotation it is interesting to note that, of the locked copper values remaining, very little acid insoluble copper is locked to acid soluble copper. Most locks are with gangue fragments. Thus, there are comparatively few locks of chalcocite to malachite for example and far more locks of chalcocite to gangue and malachite to gangue respectively. Had chalcocite and malachite, for example, been very finely intergrown with each other it may not have been economically possible to produce separate sulphide and oxide concentrates. The fact that cuprite is always intergrown with chalcocite and native copper is also fortunate, for any particles of cuprite locked to either chalcocite or native copper will tend to float into the sulphide concentrates.

It is natural to suppose that the sulphide or acid insoluble minerals that are most likely to escape from the sulphide flotation circuit and float into the oxide concentrates are firstly, the most abundant sulphide minerals and secondly, the slower floating sulphide minerals. Chalcocite, the most abundant sulphide mineral in the ore, is generally the biggest contributor to acid insoluble copper in the oxide concentrates and bornite, generally considered to be a slow-floating copper sulphide mineral, is another major contributor to acid insoluble copper in oxide concentrates.

CHAPTER 4

Experimental

4.0 EXPERIMENTAL

4.1 Sample Collection and Preparation

Primary crusher product of underground copper ore was obtained as belt cut from Nchanga Mine in January 1998. Eight buckets of ore were collected each day for 7 days. The total amount of wet sample obtained was approximately 1.2 tons. The material was dried in an oven before being crushed stage-wise.

Although sample preparation is a dull and tedious job, careful sampling and preparation of the head sample ensures uniform, dependable and reproducible results in the subsequent unit process. Laboratory tests are meant to reflect the response of the ore treatment to technique, reagent, or process variable. This being the case, the sample to be treated in the laboratory should represent the bulk sample in all aspects, physical, chemical, and mineralogical.

All the samples were reduced in size stage wise to 100% passing 2000 microns using a laboratory Jaw Crusher and Roll Crusher prior to homogenisation. Large rocks were initially hand-hammered to a size suitable for feed to jaw crusher. The crusher discharge was screened on a 10-mesh screen. The screen undersize was fed to a Roll Crusher and subsequent discharge was screened on a 2000-micron sieve. The screen oversize was recycled back to the roll crusher for further size reduction. This was repeated until material was 100% passing 2000 microns. The ore samples were homogenised using the cone and quartering technique and chute riffler. The ore was split into one-kilogram lots for laboratory work.

50 kg of the ore sample was transported to each of the University of Zambia (UNZA) and the University of Cape Town (UCT) for laboratory test work. The work was done in three phases. Phase one involved the work at UCT. Phases two and three involved the work at Nchanga concentrator and UNZA respectively.

4.2 Phase I (UCT)

The work done at UCT involved the following:

(a) Evaluation of the Effect of Milling Media on Flotation Response [Stainless Steel (SS) and Mild Steel Media (MS)].

(b) Effect of Regrind of Sulphide Rougher Tailings on Flotation Recovery of Oxide Minerals (SS & MS).

4.2.1 Flotation Procedure

Prior to flotation, the ore was ground in a Sala laboratory stainless steel rod mill, length 30-cm by diameter 30 cm, with either a charge of 21.5-kg stainless steel rods of diameter 2.1 cm or 22.5 kg mild steel rods of diameter 2.3 cm as specified. The ore was ground for various times and screen analysed. The grind time at which 55% of the material passed 75 microns was determined (5 minutes) and used for the subsequent flotation tests (Figure 4.2.1 (a)). The same was done for the regrind tests. The determined regrind time (7 minutes) at which 80% of the material passed 75 microns was used for the subsequent flotation tests (see Figure 4.2.1 (b)). The milling curves are given in appendix G.

The flotation experiments were performed in a modified laboratory Leeds cell with a pulp volume of about 3 litres. The speed of the impeller was set at 1200 r.p.m. The air flowrate was set at 6 l/min. The Nchanga laboratory flotation flowsheet for copper ore shown in Figure 4.2.1 (a) was followed. The reagents used were:

- Collector: Sodium Isopropyl Xanthate (SIPX) at 185g/t, conditioned for 3 minutes
- Sulphidiser: Sodium Hydrogen Sulphide (NaHS) at 500g/t, conditioned for 5 minutes
- Frothers: Tri– ethoxy-butane (TEB) for sulphide flotation (30g/t) and R9004A for oxide flotation (30g/t).

Senmin supplied the collector and frothers.

The flotation process involved the sulphide rougher and oxide rougher flotation steps. Prior to flotation, the collector SIPX (185g/t) was added to the pulp in the cell and conditioned for 3 minutes. A few drops of the frother TEB at a dose rate of 30g/t of ore were added, after conditioning the collector for a minute, and conditioned for 2 minutes before aeration, thus bringing the total conditioning time for collector to 3 minutes. The sulphide rougher concentrates were collected into four separate pans for 6 minutes at a stroke interval of 10 seconds (see Figure 4.2.1 (a)).

In order to float the oxide minerals, NaHS (500g/t) was added to the remaining pulp in the cell (i.e. after the sulphide rougher flotation step). NaHS was conditioned for 3 minutes before a few drops of frother (R9004A) at a dose rate of 30g/t of ore were added and conditioned for 2 minutes. This brings the total conditioning time of NaHS to 5 minutes. The oxide rougher concentrates were collected into four separate pans for 8 minutes.

Note that in the regrind test collector was added after NaHS had been added, and conditioned for 2 min before the oxide flotation step. The collector was conditioned for one minute before the frother was added and conditioned for 2 minutes bringing the total conditioning time to 5 minutes for NaHS and 3 minutes for collector (see Figure 4.2.1 (b)). Without a regrind step only NaHS and frother and no additional collector were added before the oxide flotation stage.

The pulp chemistry parameters were measured directly in the pulp by a TPS-meter (see section 4.5).

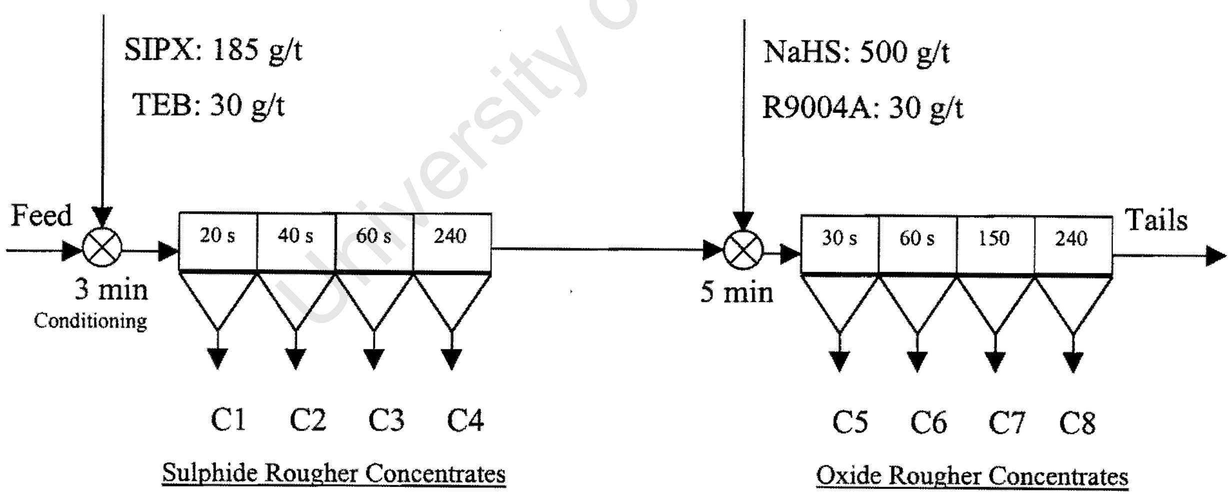


Figure 4.2.1 (a): Nchanga Laboratory Flotation Circuit for Copper Ore

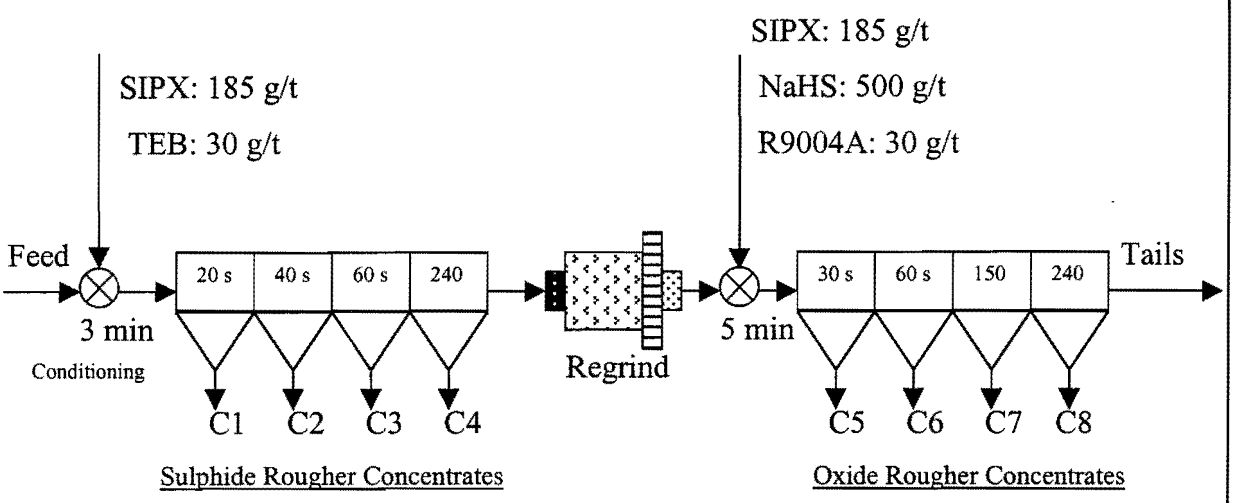


Figure 4.2.1 (b): Laboratory Flotation Circuit for Sulphide-Regrind-Oxide Flotation

4.3 Phase II (Nchanga Concentrator)

The objective of the work at Nchanga Concentrator was:
To investigate whether the distribution of dissolved oxygen concentration (DO), pH and pulp potential (Eh) from milling stage to flotation stage in the Plant is the same as that obtained in the laboratory with mild steel or stainless steel milling media so that the benefits could be predicted.

4.3.1 In-Plant Studies

In-plant studies involved the measurement of pulp chemistry parameters; pH, dissolved oxygen concentration (DO), and oxidation-reduction potential (ORP) by a TPS meter. The sampling points were:

(a) Ball mill discharge pulp

There are 10 ball mills each in closed circuit with a cyclone in the underground milling circuit at Nchanga. The cyclone overflow is pumped to the 50-ft diameter West Agitator that handles the underground ore pulp (see flotation circuit in Figure 3.2.1 in chapter 3.0). The pulp chemistry parameters were measured directly in the pulp at a point where the pulp from all the mills converges before entering the agitator. The TPS-meter was used (see section 4.5).

(b) Flotation banks

The pulp chemistry parameters were measured directly in the flotation banks as follows: (i) Sulphide Rougher Banks (SRB), which consists of 1 bank, 6 x 1500ft^3 Wemco cells, (ii) Oxide Rougher Banks (ORB) consisting of 2 banks, 20 x 300ft^3 Denver cells, and (ii) Oxide Cleaner Banks (OCB), which is made up of 1 bank, 100ft^3 Denver cells. The TPS-meter probes were immersed in the pulp and not in the froth zones. Two sets of measurements were taken from the first cell of each bank since there wasn't much difference in the readings of the subsequent cells.

The readings were taken in the night, because sometimes during the day, the Plant was shut down for repairs. These pulp chemistry measurements were intended for the Underground copper ore circuit only. As such the experiment was brought to a stand still for about 3 months when they were mixing the underground and open pit ores due to increased production from the open pit mine at that time.

One bank was analysed on each day of the experiment. Before being used in another bank, the TPS-probes had to be cleaned and recalibrated. Two sets of readings were obtained from each bank due to inadequate time.

4.4 Phase III (UNZA)

The objectives of Phase III of the work at UNZA were:

(a) To investigate the effect of using nitrogen as a conditioning gas during sulphidisation, on the flotation response of oxide minerals.
(b) Determination of the nature of fast and slow floating sulphide minerals based on flotation kinetics and mineralogical analysis.

4.4.1 Flotation Tests

In order to determine the grind time at which 55% of the ore passed 75 microns, the ore was ground for various times and screen analysed. The determined grind time (26 min) at which

55% of the material passed 75 microns was used for the subsequent flotation tests. The Nchanga laboratory flotation flowsheet shown in Figure 4.4.1 was used. The milling curve is given in appendix G.

The reagents used were Collector: SIPX at 185 g/t; Frother: Tri-ethoxy-butane (TEB) for sulphide flotation and palm kernel oil & diesel (PKD) for oxide flotation all at 30 g/t; Sulphidiser: Sodium Hydrogen Sulphide (NaHS) at 500 g/t. Nchanga concentrator supplied the reagents. The flotation experiments were performed in a modified laboratory Leeds cell with a pulp volume of 3 litres. The speed of the impeller was set at 1200 r.p.m. The air flowrate was set at 6 l/min.

The flotation process involved the sulphide rougher and oxide rougher flotation steps. Prior to flotation, the collector SIPX (185 g/t) was added to the pulp in the cell and conditioned for a minute before a few drops of the frother TEB at a dose rate of 30 g/t of ore were added and conditioned for 2 minutes before aeration. This brings the total conditioning time for the collector to 3 minutes. The sulphide rougher concentrates were collected into four separate pans for 6 minutes at a stroke interval of 10 seconds (see Figure 4.4.1). In the oxide flotation step, NaHS (500 g/t) was added to the remaining pulp in the cell and conditioned for 3 minutes before a few drops of frother palm kernel oil and diesel (30 g/t) were added and conditioned for 2 minute, thus bringing the total conditioning time for NaHS to 5 minutes. The oxide rougher concentrates were collected into four pans for 8 minutes.

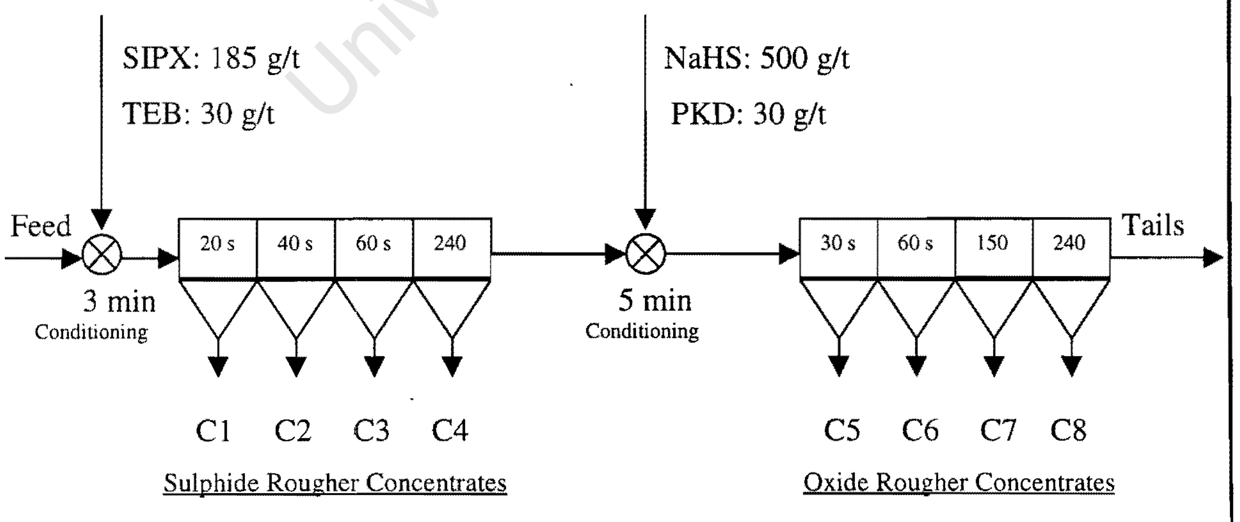


Figure 4.4.1: Nchanga Laboratory Flotation Circuit for Copper Ore

4.4.1.1 Nature of Fast- and Slow-floating Sulphide minerals

The sulphide rougher flotation stage shown in Figure 4.4.1 was analysed. The concentrates were obtained in four fractions. Recovery-time curves were plotted. The mineralogical analysis of concentrate fractions was performed in order to identify the nature of fast and slow floating sulphide minerals.

The concentrate fractions were mixed with epoxy powder and mounted. The mounted samples were polished and ready for microscopic examination to quantify the sulphide minerals present in the concentrate fractions. The sulphide minerals are opaque and are studied under reflected light microscopy. Optical grain counting method was followed.

4.4.1.2 Effect of Nitrogen Conditioning on Recovery of Oxide Minerals

Nitrogen gas at 4 l/min was bubbled in the pulp before and upon addition of NaHS to the pulp. Nitrogen was added to drop the dissolved oxygen (DO) level to < 2ppm and NaHS added and conditioned for 3 minutes. Then Nitrogen was turned off and the DO raised to > 2ppm by air, and the collector added. The collector and frother were conditioned for 2 minutes before aeration. During sulphidisation, sufficient sulphidising agent (NaHS) was added to maintain target E_{pt} values (Platinum potential). Typically this was between –300 mV and -500 mV. Sulphidising agent was added in liquid form directly to the ore pulp stirred in the flotation cell. Cell agitation was adjusted to minimise the introduction of air during sulphidisation conditioning. Oxide flotation followed (Figure 4.4.1.2).

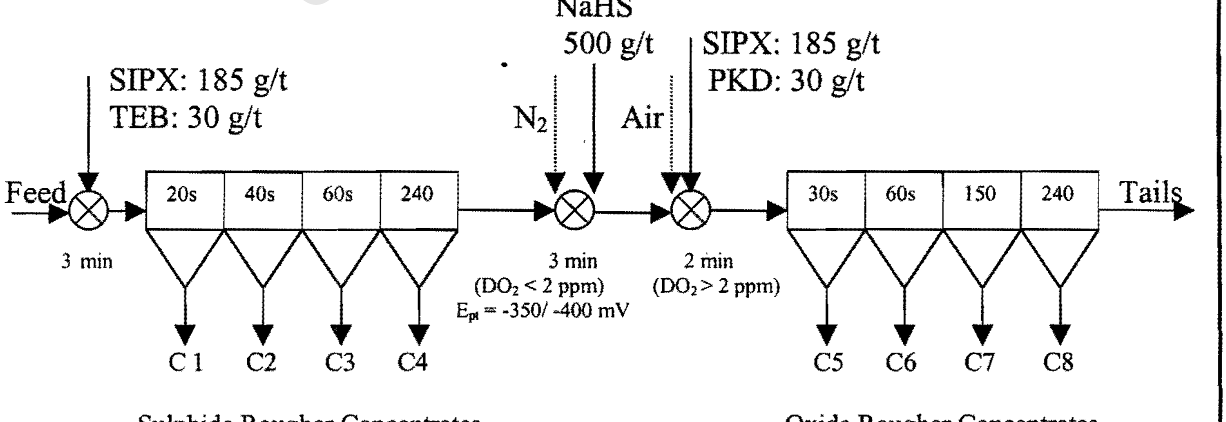


Figure 4.4.1.2: Laboratory Flotation Circuit for Nitrogen Conditioning Test

4.5 Pulp Chemistry Measurements

The parameters measured to determine the chemical conditions in the pulp were pH, temperature, oxidation-reduction potential (ORP) and dissolved oxygen concentration (DO). These parameters were measured with a TPS meter (Australian-made electrochemical instrumentation, model 90 FLMV) directly in the pulp. The ORP was measured with a Pt-Ag/AgCl electrode and the DO measured with the YSI 5739 Field Probe. All potentials reported in this work have been converted to the standard hydrogen electrode (S.H.E) scale using the following equation (Bates, 1964):

$$Eh = E_{Pt} + 207 \text{ mV}$$

4.6 Chemical analysis

The concentrates obtained during the flotation, together with the feed and tailings were analysed for total copper (TCu) and acid soluble copper (ASCu). The acid soluble copper is mainly derived from the oxide minerals. The acid insoluble copper, which is mostly derived from the sulphide minerals, is obtained as the difference between the total copper and the acid soluble copper. Digestion methods are given in appendix B.

CHAPTER 5

Results

5.0 RESULTS

5.1 Phase I (UCT)

The reproducibility results of the sulphide flotation step are given in Tables 5.1 (a) and (b) for mild steel and stainless steel grinding respectively. The sulphide flotation stage is the same with or without a regrind step. The regrind step is done after sulphide rougher flotation.

Table 5.1 (a): Reproducibility of Sulphide Flotation Step after Mild Steel Grinding

Test	Concentrate Wt%	Grade (%)		Recovery (%)	
		ASCu	AICu	ASCu	AICu
1 (a) MS	5.19	2.98	32.04	11.70	81.10
1 (b) MS	5.33	2.97	30.07	11.80	81.80
1 (c) MSR	4.79	2.70	31.89	10.00	79.30
1 (d) MSR	4.47	2.46	31.18	8.80	73.50
Mean	4.95	2.78	31.30	10.58	78.93
Standard Deviation	0.39	0.25	0.90	1.44	3.77

Table 5.1 (b): Reproducibility of Sulphide Flotation Step after Stainless Steel Grinding

Test	Concentrate Wt%	Grade (%)		Recovery (%)	
		ASCu	AICu	ASCu	AICu
1 (a) SS	4.89	4.58	30.61	15.80	79.20
1 (b) SS	5.73	4.29	27.59	17.90	83.10
1 (c) SSR	4.79	4.12	31.16	15.80	69.70
1 (d) SSR	5.03	4.18	31.51	16.10	75.20
Mean	5.11	4.29	30.22	16.4	76.80
Standard Deviation	0.42	0.20	1.79	1.01	5.73

As seen from the Tables, the standard deviation of the grade and recovery respectively, of acid soluble copper to the sulphide concentrates are lower in Stainless steel media than in mild steel media. The standard deviation of the grade and recovery of acid insoluble copper are slightly lower in mild steel media than in stainless steel media.

In order to determine whether the sulphide flotation step gives the same results for mild steel and stainless steel grinding media, the means of the recoveries and grades given in Tables 5.1 (a) and (b) were compared statistically. The t-test for two means was used and the detailed calculations are given in appendix F.

Statistically, it was found that the difference in the grade of acid soluble copper between mild steel and stainless steel grinding media was real. The difference in the grade of acid insoluble copper was not significant.

Similarly, the difference in the recovery of acid soluble copper to the sulphide concentrate was significant. Stainless steel recovered more acid soluble copper than mild steel media. The difference in the recovery of acid insoluble copper between the two media was not significant.

5.1.1 Effect of Grinding Media on Flotation Performance

Summary of metallurgical results obtained during flotation after mild steel grinding (MS) and regrinding (MSR), and stainless steel grinding (SS) and regrinding (SSR) is presented in Table 5.1.1. The flotation performance was assessed in terms of grade and recovery of acid soluble copper (ASCu) and acid insoluble copper (AICu). Detailed metallurgical results are given in appendix C1.

As seen from Table 5.1.1, the recovery of acid soluble copper to the sulphide concentrate is higher in pulp ground with stainless steel than with mild steel media. If all sulphides were acid insoluble, and all oxides acid soluble, then all acid insoluble copper would be recovered during the sulphide flotation stage and all acid soluble copper recovered in the oxide flotation stage. The recovery of acid insoluble copper to the oxide concentrates and acid soluble copper to the sulphide concentrates shows the limitation of this approximation of sulphide and oxide minerals as acid insoluble copper and acid soluble copper, respectively. It is known that about 1.5% of chalcocite and 2% of bornite is acid soluble copper, and native copper and part of cuprite respond as acid insoluble copper.

Table 5.1.1: Summary of Metallurgical Results of the Effect of Grinding Media

Milling Media	Test Number	Flotation Stage	Mass Pull (%)	Grade (%)		Recovery (%)	
				ASCu	AICu	ASCu	AICu
Mild Steel (MS)	1 (a)	Sulphide	5.19	2.98	32.04	11.70	81.10
		Oxide	3.56	1.57	1.44	4.30	2.50
		Total	**8.75**	**2.41**	**19.59**	**16.00**	**83.60**
	1 (b)	Sulphide	5.33	2.97	30.07	11.80	81.80
		Oxide	4.16	1.53	1.39	4.80	3.00
		Total	**9.49**	**2.34**	**17.50**	**16.60**	**84.80**
Mean			**9.12**	**2.38**	**18.55**	**16.30**	**84.20**
Standard Deviation			**0.52**	**0.05**	**1.48**	**0.42**	**0.85**
Mild Steel Regrind (MSR)	2 (a)	Sulphide	4.79	2.70	31.89	10.00	79.30
		Oxide	3.62	4.39	7.75	12.5	14.70
		Total	**8.41**	**3.43**	**21.50**	**22.50**	**94.00**
	2 (b)	Sulphide	4.47	2.46	31.18	8.80	73.50
		Oxide	4.24	3.95	8.78	13.40	19.80
		Total	**8.71**	**3.19**	**20.28**	**22.20**	**93.30**
Mean			**8.56**	**3.31**	**20.89**	**22.35**	**93.65**
Standard Deviation			**0.21**	**0.17**	**0.86**	**0.21**	**0.49**
Stainless Steel (SS)	3 (a)	Sulphide	4.89	4.58	30.61	15.80	79.20
		Oxide	4.71	2.18	6.72	7.30	16.90
		Total	**9.60**	**3.40**	**18.89**	**23.10**	**96.10**
	3 (b)	Sulphide	5.73	4.29	27.59	17.90	83.10
		Oxide	3.66	1.92	4.48	5.10	8.70
		Total	**9.39**	**3.37**	**18.58**	**23.00**	**91.80**
Mean			**9.50**	**3.39**	**18.74**	**23.05**	**94.00**
Standard Deviation			**0.15**	**0.02**	**0.22**	**0.07**	**3.04**
Stainless Steel Regrind (SSR)	4 (a)	Sulphide	4.79	4.12	31.16	15.80	69.70
		Oxide	2.77	3.63	13.36	8.10	17.20
		Total	**7.56**	**3.94**	**24.64**	**23.90**	**86.90**
	4 (b)	Sulphide	5.03	4.18	31.51	16.10	75.20
		Oxide	3.19	3.12	10.69	7.80	16.20
		Total	**8.22**	**3.77**	**23.43**	**23.90**	**91.40**
Mean			**7.89**	**3.86**	**24.04**	**23.90**	**89.15**
Standard Deviation			**0.47**	**0.12**	**086**	**0.00**	**3.18**

5.1.1.1 Flotation Results

It can be seen from Table 5.1.1 that the overall flotation recovery of acid insoluble copper after grinding with stainless steel media was higher than that of mild steel media (94.0% Vs 84.2%). The difference in the recovery of acid soluble copper and acid insoluble copper between mild steel and stainless steel grinding was compared statistically and found to be significant. The calculations are given in appendix F. Regrinding the sulphide rougher tails with stainless steel media did not increase the flotation performance (89.15% AICu). The reduction in the recovery of the acid insoluble copper (AICu) to the sulphide concentrate in the regrind test with stainless steel media was not expected to occur since the regrind step is performed after the sulphide rougher flotation stage. In practice, the recovery of acid insoluble copper to the sulphide concentrate during the sulphide flotation stage was expected to be equal for both conditions of stainless steel grinding (SS) and regrinding (SSR). This discrepancy observed demonstrates the limitation of the reproducibility tests and can be attributed to the complexity and sensitivity of the flotation of this ore.

Mild steel milling media favoured the initial flotation recovery of acid insoluble copper to the sulphide concentrate over the recovery of acid soluble copper to the oxide concentrate. After regrinding the sulphide rougher tails with mild steel media, the recovery of acid soluble copper to the oxide concentrate increased (from 4.55% to 12.95%). The overall recovery of acid soluble copper and acid insoluble copper was better after regrinding the sulphide rougher tails with mild steel media. The difference in the recovery of acid soluble copper between mild steel grinding (MS) and mild steel with regrind (MSR) was found to be significant statistically. The detailed calculations using t-test are given in appendix F. However, the recovery observed after regrinding with mild steel media (22.35% ASCu) was almost the same as that obtained without a regrind step with stainless steel media (22.05% ASCu). The low recovery of acid insoluble copper to the sulphide concentrate in the regrind test with mild steel media was offset after regrinding the sulphide rougher tails.

Figure 5.1.1.1 (a) shows the recovery of copper as a function of time for the condition of stainless steel grinding and regrinding. It can be seen from the figure that similar recoveries of acid soluble copper were obtained after stainless steel grinding (SS) and regrinding (SSR).

Figure 5.1.1.1 (b) gives the flotation recovery of copper as a function of time for the condition of mild steel grinding (MS) and regrinding (MSR). It can be observed from the figure that there was an increase in the recovery of both acid soluble and acid insoluble copper after regrinding the sulphide rougher tails and adding the collector.

Figure 5.1.1.1 (c) shows the grade-recovery profiles of acid soluble copper obtained during oxide flotation stage both after grinding and regrinding in either mild steel (MS) or stainless steel (SS) media as specified. It can be seen from the figure that regrinding the sulphide rougher tails in mild steel media resulted in higher grade and recovery of acid soluble copper. The mass pulls of the initial concentrate after regrinding with mild steel media were lower than without regrind although the grade was higher. The overall mass pulls of the acid soluble copper with or without a regrind step with mild steel media were similar.

Regrinding the sulphide rougher tails with stainless steel media improved the grade of the acid soluble copper obtained and not the recovery. The mass pull of the acid soluble copper after regrinding with stainless steel media was lower than that obtained without regrind but was accompanied by a corresponding increase in grade. This shows that the change in grade–recovery profile was not due to changes in mass pull and froth characteristics, but rather to increased selectivity.

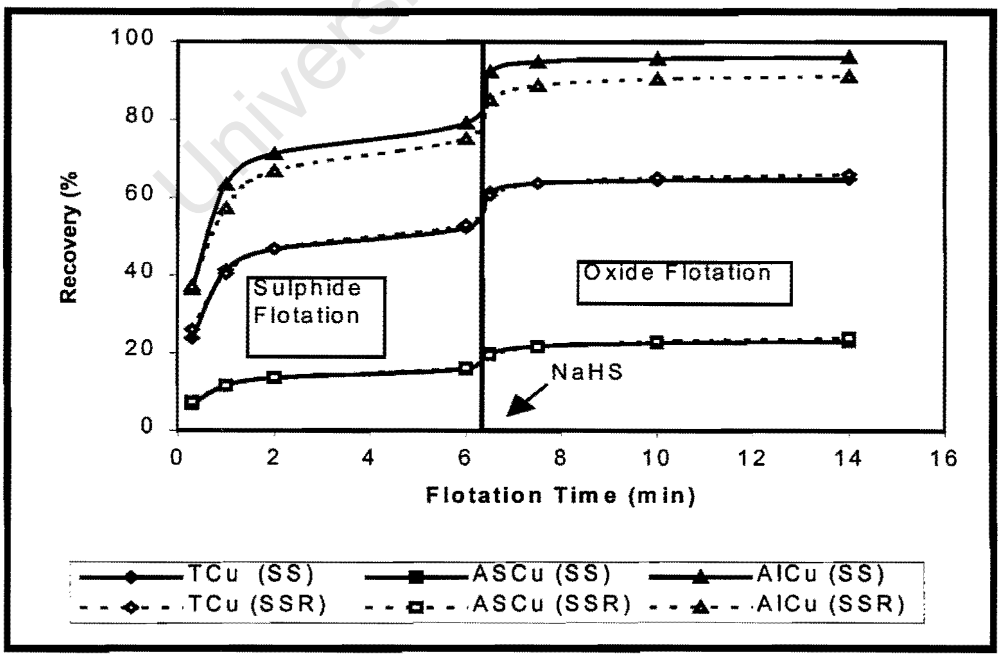


Figure 5.1.1.1 (a): Flotation Recovery of Acid Soluble Copper (ASCu), Total Copper (TCu) and Acid Insoluble Copper (AICu) as a Function of Time after Stainless Steel Grinding (SS) and Regrinding (SSR).

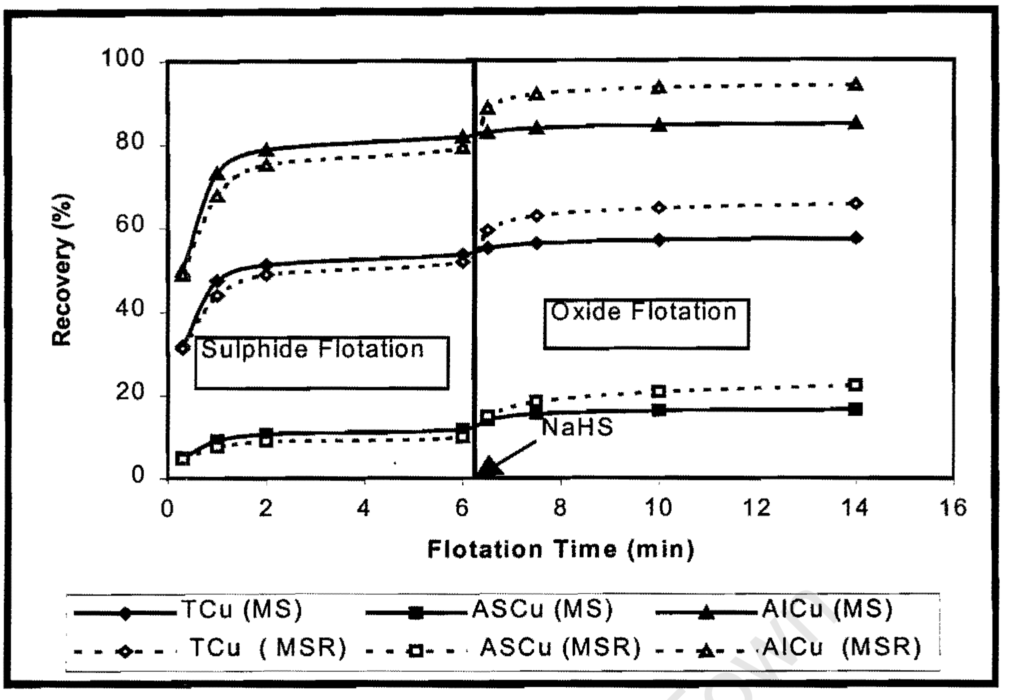


Figure 5.1.1.1 (b): Flotation Recovery of Acid Soluble Copper (ASCu), Total Copper (TCu) and Acid Insoluble Copper (AICu) as a Function of Time after Mild Steel Grinding (MS) and Re-grinding (MSR).

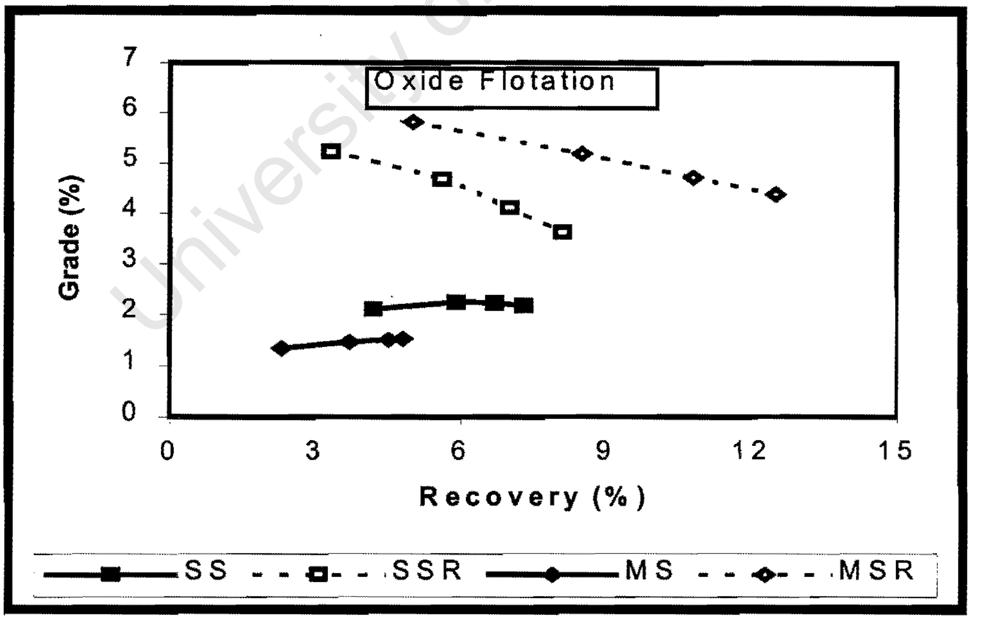


Figure 5.1.1.1 (c): Grade-Recovery Profiles for Acid Soluble Copper (ASCu) Obtained During Oxide Flotation Stage after Stainless Steel Grinding (SS) and Re-grinding (SSR), and after Mild Steel Grinding (MS) and Re-grinding (MSR).

5.1.1.2 Pulp Chemistry Results

Summary of pulp chemistry measurements obtained during flotation after mild steel grinding (MS) and regrinding (MSR), and after stainless steel grinding (SS) and regrinding (SSR) are given in Table 5.1.1.2. Detailed pulp chemistry results are given in appendix D1.

Table 5.1.1.2: Summary of Pulp Chemistry Measurements of the Effect of Grinding Media

Milling Media	pH		Pulp potential (Eh) vs SHE (mV)					Dissolved Oxygen concentration (DO) (ppm)				
			After				End Flot.	After				End Flot.
	Initial	Final	Milling	SIPX Addition	Start Flot.	NaHS Addition		Milling	SIPX Addition	Start Flot.	NaHS Addition	
MS	7.45	8.04	196	182	193	183	200	1.59	3.02	7.24	6.56	6.80
MSR	7.13	8.47	186	176	186	157	164	1.59	2.95	7.30	1.70	5.74
SS	7.71	8.09	210	202	201	199	200	7.96	8.15	8.31	7.84	7.16
SSR	7.72	8.32	213	206	208	201	205	5.93	6.12	6.07	7.15	7.33

It can be seen from Table 5.1.1.2 that the dissolved oxygen (DO) level and pulp potential (Eh) were lower in the pulp ground with mild steel than stainless steel media. The pH in both milling media was in the same range.

Figure 5.1.1.2 (a) shows the dissolved oxygen levels (DO) obtained during flotation after grinding and regrinding with either mild steel (MS) or stainless steel (SS) media as specified. It can be observed from the figure that the dissolved oxygen level after grinding with mild steel media was less than that obtained with stainless steel media, which is consistent with the literature (Kocabag and Smith, 1982). During the sulphide and oxide flotation stages when air was turned on, the DO came to similar levels for both milling media. However, the DO levels from the pulp ground with mild steel media decreased substantially during sulphidisation, compared to stainless steel media which were unaffected. The dissolved oxygen levels in pulp ground by stainless steel are supposed to be the same during the sulphide rougher flotation stage, the difference observed between the two tests can be attributed to the sensitivity of the dissolved oxygen probe.

The DO levels after regrinding the sulphide rougher tails with mild steel media decreased further, possibly due to the scavenging of oxygen by ferrous-ferric ions in the pulp as a result of media corrosion. The DO levels in pulp ground with stainless steel media did not show any significant change during both the sulphide and oxide flotation stages, demonstrating that the xanthate-mineral reaction would not be limited by dissolved oxygen concentration.

Figure 5.1.1.2 (b) gives the pulp potential (Eh) profiles measured during flotation after grinding and regrinding the ore with either mild steel media (MS) or stainless steel (SS) media as specified. As seen from the figure the pulp potential in pulp ground with stainless steel media were slightly higher than those obtained after milling with mild steel media. There was a slight drop in potential both after collector and NaHS additions for the pulp milled with mild steel media. The pulp potential after regrinding with mild steel media decreased, probably due to the introduction of more iron ions in the pulp as a result of media corrosion. This demonstrated the stronger response of pulp to sulphidisation after regrind and resulted in better flotation performance.

There was no appreciable change in pulp potential after regrinding with stainless steel media and upon addition of either collector or NaHS, as was the case with mild steel milling media, and there was no corresponding benefit in flotation performance. The temperature ranged between 18 °C and 29 °C, and the pH ranged between 7 and 9 in both milling media, from the sulphide flotation stage to the oxide flotation stage.

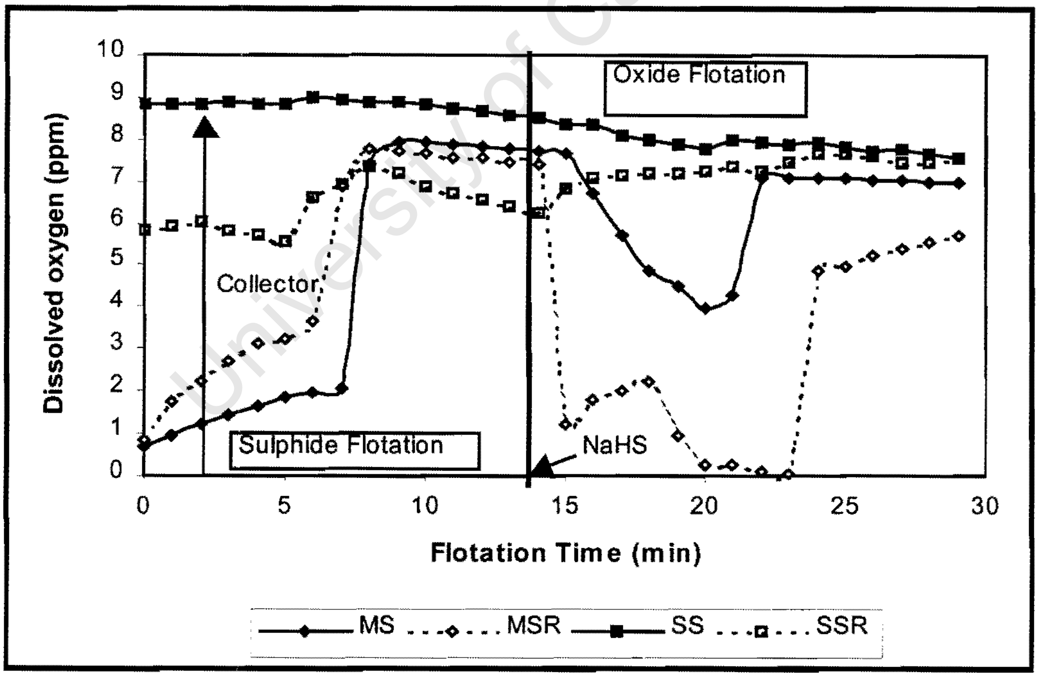


Figure 5.1.1.2 (a): Dissolved Oxygen Profiles Measured During Flotation after Mild Steel Grinding (MS) and Re-grinding (MSR), and after Stainless Steel Grinding (SS) and Re-grinding (SSR).

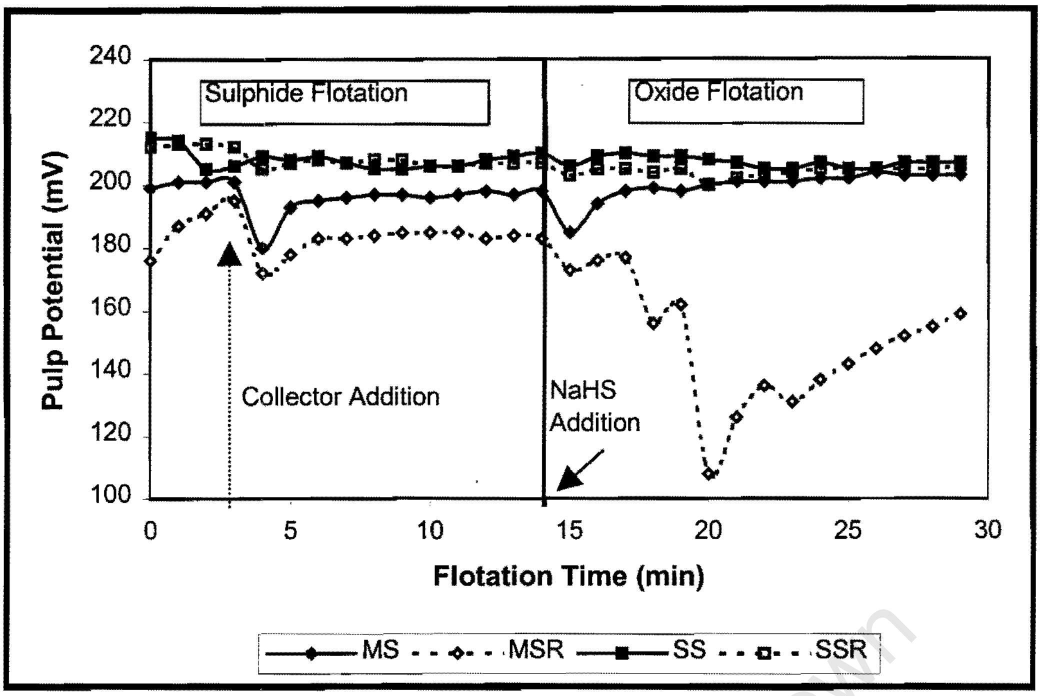


Figure 5.1.1.2 (b): Pulp Potential Profiles Measured During Flotation after Mild Steel Grinding (MS) and Re-grinding (MSR), and after Stainless Steel Grinding (SS) and Re-grinding (SSR).

5.2 Phase II (Nchanga Concentrator)

5.2.1 Distribution of Pulp Chemistry Parameters from Grinding to Flotation Banks

The detailed graphical representation of DO, pH, temperature and Eh profiles are given in appendix D2. Table 5.2.1. shows the summary of pulp chemistry measurements obtained from the cyclone overflow up to the flotation banks.

Table 5.2.1: Summary of Pulp Chemistry Measurements Obtained from Cyclone Overflow to Flotation Banks.

Parameter	Cyclone Overflow	Sulphide Rougher Banks	Oxide Rougher Banks	Oxide Cleaner Banks
DO (ppm)	3.27 – 3.61	7.07 – 7.08	7.74 – 7.32	7.60 – 7.43
pH.	7.53 – 7.46	8.45 – 9.11	8.44 – 7.73	7.92 – 8.00
Eh (mV)	221 - 214	210 - 206	203 - 211	214 – 216
Temp. (°C)	28.5 – 28.1	25.9 – 25.9	26.1 – 25.8	25.8 – 26.1

The profile follows those obtained in the laboratory with mild steel media where the DO level during conditioning, but before sulphide or oxide flotation, is below 4 ppm and rises to about 7 ppm during sulphide and oxide flotation stages respectively. The pH also follows the same profile obtained during laboratory flotation. However, the Eh is slightly higher than that obtained in the laboratory. The effect of NaHS on pulp potential was observed by the reduction in the pulp potential obtained in the oxide rougher banks (ORB). Figure 5.2.1 shows the distribution of dissolved oxygen (DO) and pulp potential (Eh) values obtained from the cyclone overflow (COF), sulphide rougher banks (SRB), oxide rougher banks (ORB), and the oxide cleaner banks (OCB) at Nchanga concentrator.

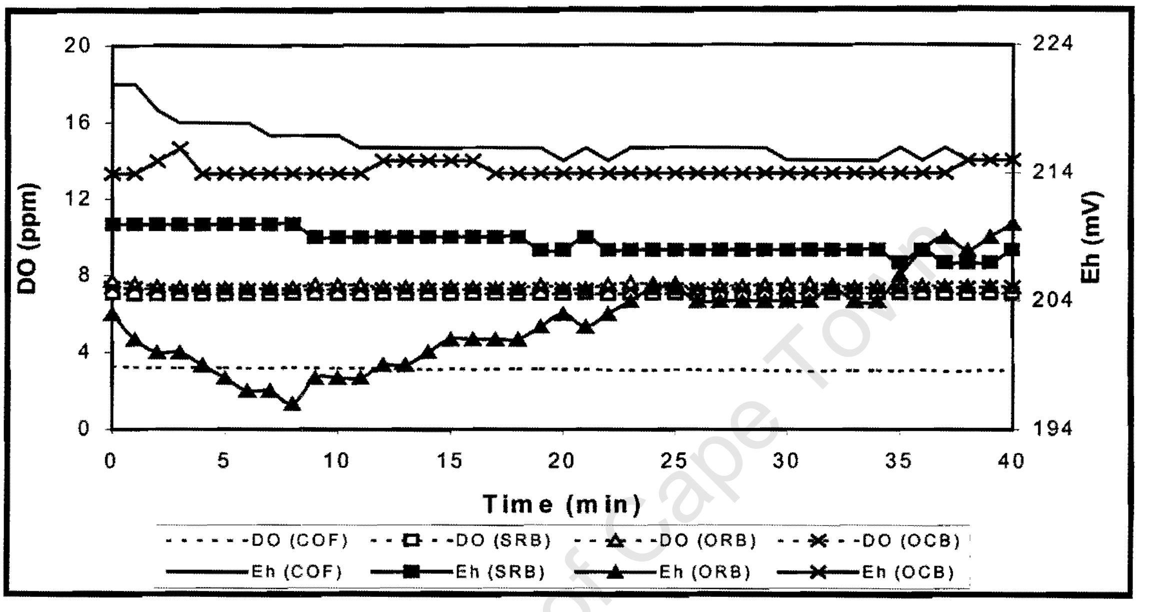


Figure 5.2.1: Distribution of Dissolved Oxygen (DO) and Pulp Potential (Eh) Profiles obtained in the Cyclone Overflow (COF), Sulphide Rougher Banks (SRB), Oxide Rougher Banks (ORB) and Oxide Cleaner Banks (OCB) at Nchanga Concentrator.

The DO level of the cyclone overflow is almost the same as that obtained in the laboratory with ore milled with mild steel media. The pulp potential in the cyclone overflow is higher than that obtained in the flotation banks because of the presence of collector and NaHS in the flotation banks. The collector and NaHS are known to reduce the pulp potential. As seen from the figure, the pulp potentials obtained from cyclone overflow, sulphide rougher banks and the oxide cleaner banks were steady compared to that obtained from the oxide rougher banks. This simply shows that there was more uniform distribution of reagents in the former flotation banks than in the latter. The sulphide rougher tails report to an oxide agitator where reagents (collector and NaHS) are added before the oxide rougher flotation stage. Unfortunately, the impeller to the agitator does not function and this caused the instability observed from the pulp potential measured in the oxide rougher banks. Reagent conditioning ensures that the necessary chemical reactions between chemicals and minerals such as xanthate-mineral or sulphidiser-mineral occur, which is a prerequisite for flotation recovery of valuables.

5.3 Phase III (UNZA)

The Phase III (UNZA) flotation experiments were done in a modified laboratory Leeds cell after grinding in a laboratory mild-steel ball mill. Nchanga concentrator supplied the reagents, collector and NaHS. Phase I (UCT) flotation experiments were also done in a modified laboratory Leeds cell. In Phase I, mild steel or stainless steel rods were used to grind the ore in a Sala laboratory stainless steel rod mill as specified. The ore in both Phases was ground to about 55% < 75 μm. The results are given in Table 5.3, which include the Phase I mild steel grinding results.

The difference in the recovery of acid soluble copper after grinding with mild steel media between Phase I (16.3% ASCu) and Phase III [Standard test (63.2% ASCu)] has been attributed to insufficient sulphidisation at UCT because of the aged reagent used (NaHS). The comparison between Phase I and Phase III, for acid soluble copper showed that the NaHS used in Phase I was less effective. On average the recovery of acid insoluble copper in both Phases was almost similar, 84.2% AICu (UCT) and 80.9% AICu (UNZA).

In order to determine whether there was a significant difference in the standard flotation performance between Phase I and Phase III, the means of the mass pull, acid soluble copper and acid insoluble copper were tested statistically (Table 5.3). The t-test was used and detailed calculation is given in appendix F.

It was found that there was no significant difference in the mass pull between Phase I and Phase III. The difference in the recovery of acid insoluble copper was found to be not significant but that of the acid soluble copper was found to be significant. This confirms the ineffectiveness of the NaHS used in Phase I.

It was not possible to compare the effect of nitrogen in Phase III with regrind in Phase I due to the ineffectiveness of the NaHS.

Table 5.3: Comparison of the Standard Flotation Performance between Phase I and Phase III

Phase of Work	Milling Media	Test No.	Flotation Stage	Mass Pull (%)	Grade (%)		Recovery (%)	
					ASCu	AICu	ASCu	AICu
Phase I (UCT)	Mild Steel (MS)	1 (a)	Sulphide	5.19	2.98	32.04	11.70	81.10
			Oxide	3.56	1.57	1.44	4.30	2.50
			Total	**8.75**	**2.41**	**19.59**	**16.00**	**83.60**
		1 (b)	Sulphide	5.33	2.97	30.07	11.80	81.80
			Oxide	4.16	1.53	1.39	4.80	3.00
			Total	**9.49**	**2.34**	**17.50**	**16.60**	**84.80**
Mean				**9.12**	**2.38**	**18.55**	**16.30**	**84.20**
Standard Deviation				**0.52**	**0.05**	**1.48**	**0.42**	**0.85**
Phase III (UNZA)	Mild Steel (MS)	2 (a)	Sulphide	6.85	3.28	26.55	17.50	78.40
			Oxide	3.12	20.69	2.50	50.30	3.40
			Total	**9.97**	**8.73**	**19.02**	**67.80**	**81.80**
		2 (b)	Sulphide	7.03	3.12	25.56	16.80	82.20
			Oxide	3.49	15.78	1.94	42.20	3.10
			Total	**10.52**	**7.32**	**17.72**	**59.00**	**85.30**
		2 (c)	Sulphide	6.00	4.14	24.76	19.00	72.80
			Oxide	3.70	17.20	2.83	48.60	5.10
			Total	**9.70**	**9.12**	**16.40**	**67.60**	**77.90**
		2 (d)	Sulphide	5.52	3.88	28.54	14.90	75.90
			Oxide	3.41	18.35	1.62	43.40	2.70
			Total	**8.93**	**9.41**	**18.26**	**58.30**	**78.60**
Mean				**9.78**	**8.65**	**17.87**	**63.18**	**80.90**
Standard Deviation				**0.66**	**0.93**	**1.10**	**5.23**	**3.39**

5.3.1 Effect of Nitrogen Conditioning on the Recovery of Oxide Minerals

Table 5.3.1 presents the summary of metallurgical results obtained during flotation with and without sulphidisation in the nitrogen environment. Four flotation experiments were done each with air and nitrogen conditions respectively, to show reproducibility of the results. The values given in the subsequent graphs are the average. Flotation performance is measured in terms of recovery and grade. Table 5.3.1 shows that sulphidising in the nitrogen resulted in increased mass recovery of the acid soluble copper (ASCu) to oxide concentrates.

Table 5.3.1: Summary of Metallurgical Results of the Effect of Nitrogen Conditioning

Test No.	NaHS Conditioning	Flotation Stage	Mass Pull (%)	Grade (%)		Recovery (%)	
				ASCu	AICu	ASCu	AICu
1 (a)	Nitrogen	Sulphide	7.42	3.30	18.99	17.30	84.20
		Oxide	4.80	13.90	2.40	47.20	6.90
		Total	**12.22**	**7.46**	**12.47**	**64.50**	**91.10**
1 (b)	Nitrogen	Sulphide	7.71	2.94	19.44	16.90	77.70
		Oxide	5.48	13.27	1.03	54.10	2.90
		Total	**13.19**	**7.23**	**11.79**	**71.00**	**80.60**
1 (c)	Nitrogen	Sulphide	8.18	2.84	18.98	16.80	76.00
		Oxide	4.66	15.56	1.18	52.40	2.70
		Total	**12.84**	**7.46**	**12.52**	**69.20**	**78.70**
1 (d)	Nitrogen	Sulphide	8.12	3.43	17.39	21.10	64.40
		Oxide	4.45	16.70	3.43	56.30	6.90
		Total	**12.57**	**8.13**	**12.45**	**77.40**	**71.30**
Mean			**12.71**	**7.57**	**12.31**	**70.52**	**80.43**
Standard Deviation			**0.41**	**0.39**	**0.35**	**5.34**	**8.17**
2 (a)	Air (Standard Flotation)	Sulphide	6.85	3.28	26.55	17.50	78.40
		Oxide	3.12	20.69	2.50	50.30	3.40
		Total	**9.97**	**8.73**	**19.02**	**67.80**	**81.80**
2 (b)	Air (Standard Flotation)	Sulphide	7.03	3.12	25.56	16.80	82.20
		Oxide	3.49	15.78	1.94	42.20	3.10
		Total	**10.52**	**7.32**	**17.72**	**59.00**	**85.30**
2 (c)	Air (Standard Flotation)	Sulphide	6.00	4.14	24.76	19.00	72.80
		Oxide	3.70	17.20	2.83	48.60	5.10
		Total	**9.70**	**9.12**	**16.40**	**67.60**	**77.90**
2 (d)	Air (Standard Flotation)	Sulphide	5.52	3.88	28.54	14.90	75.90
		Oxide	3.41	18.35	1.62	43.40	2.70
		Total	**8.93**	**9.41**	**18.26**	**58.30**	**78.60**
Mean			**9.78**	**8.65**	**17.87**	**63.18**	**80.90**
Standard Deviation			**0.66**	**0.93**	**1.10**	**5.23**	**3.39**

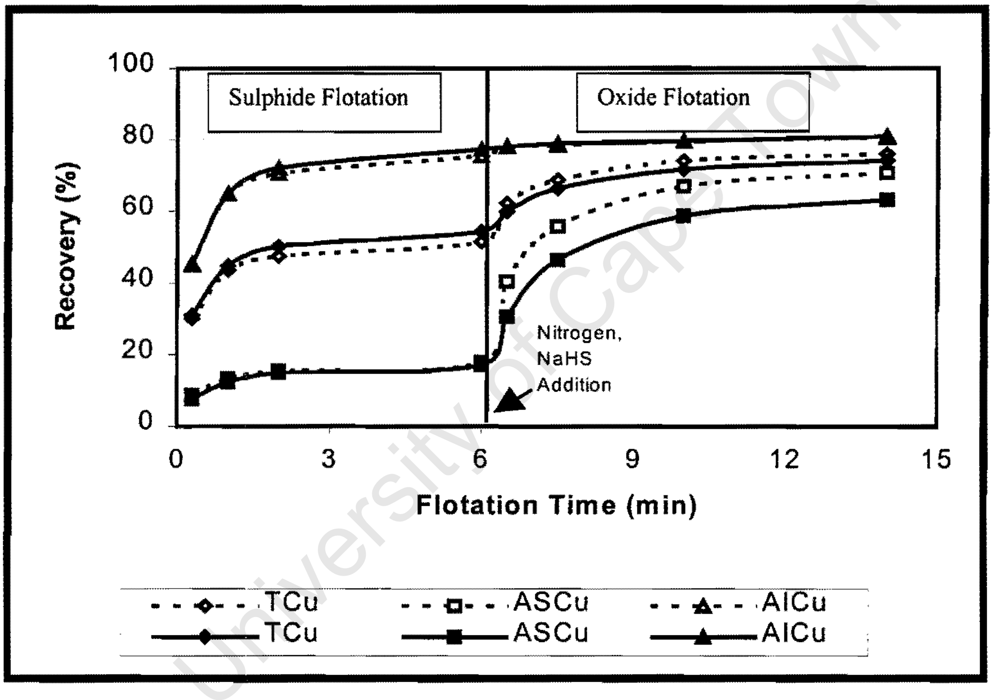
The increased mass yield observed with nitrogen was accompanied by a reduction in the grades of acid soluble copper. In the case of air, the decreased mass recovery of acid soluble copper to the oxide concentrates was accompanied by a slight increase in grade of the acid soluble copper. The detailed metallurgical balance sheets are given in appendix C2.

The results of the effect of sulphidisation in the nitrogen environment on the flotation recovery of oxide minerals shown in Table 5.3.1 above, has been tested statistically using the analysis of variance (ANOVA).

From the ANOVA, we conclude that the difference of 7.34% ASCu in the recovery for the flotation process with and without nitrogen conditioning is real. Detailed calculation, explanation of symbols and mathematical expressions is given in appendix F.

5.3.1.1 Copper Recovery and Grade Results

Figure 5.3.1.1 (a) shows the recovery of acid soluble copper (ASCu), total copper (TCu) and acid insoluble copper (AICu) as a function of flotation time for the condition of nitrogen and air sulphidisation. It can be seen that there is an increase in the recovery of acid soluble copper to the oxide concentrates after sulphidising in the nitrogen environment. The recoveries on average were about 63.18% ASCu and 70.52% ASCu without and with nitrogen conditioning respectively. It can also be observed from the figure that sulphidising in the nitrogen environment did not have any influence on the recovery of the acid insoluble copper to the oxide concentrate.



NB: Continuous Lines for Standard Test (Air) and Dotted lines for Nitrogen Conditioning Test

Figure 5.3.1.1 (a): Flotation Recovery of Acid Soluble Copper (ASCu), Total Copper (TCu) and Acid Insoluble Copper (AICu) as a Function of Time with and without Nitrogen Conditioning.

Figure 5.3.1.1 (b) presents the grade-recovery profiles of acid soluble copper. It can be seen that a drop in grade accompanied the slight increase in the recovery of acid soluble copper to the oxide concentrates.

5.3.1.2 Mass Pull Results

Figure 5.3.1.2 shows grade as a function of mass pull (Wt%) for acid soluble copper. As seen from the Figure the mass pull increased after nitrogen conditioning but was accompanied by a slight drop in grade of the concentrates. On average the mass pulls ranged between 1.45% - 4.85% and 0.8% - 3.36% with and without nitrogen conditioning respectively.

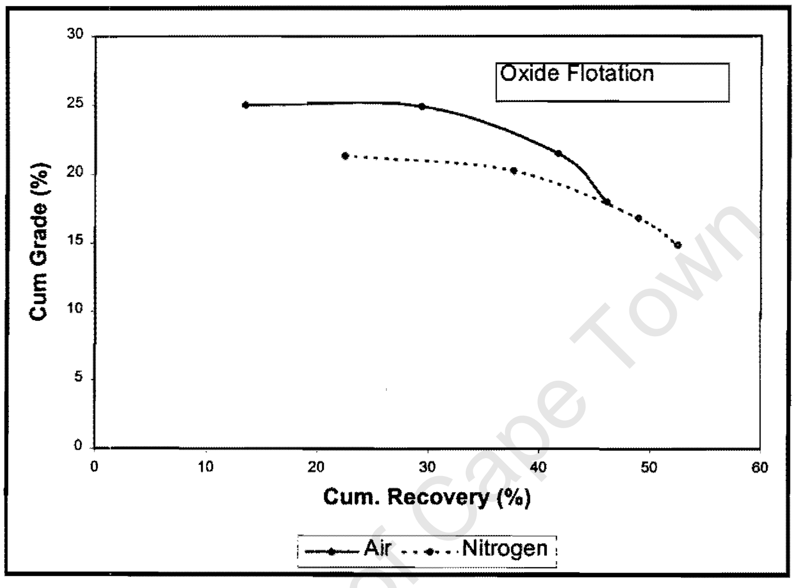


Figure 5.3.1.1 (b): Grade-Recovery Profiles for Acid Soluble Copper (ASCu) Obtained during the Oxide Flotation Stage with and without Nitrogen Conditioning.

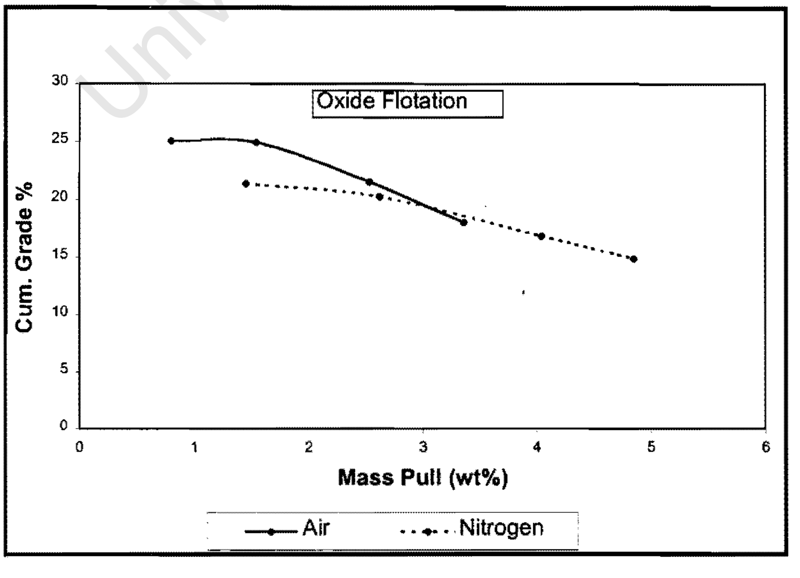
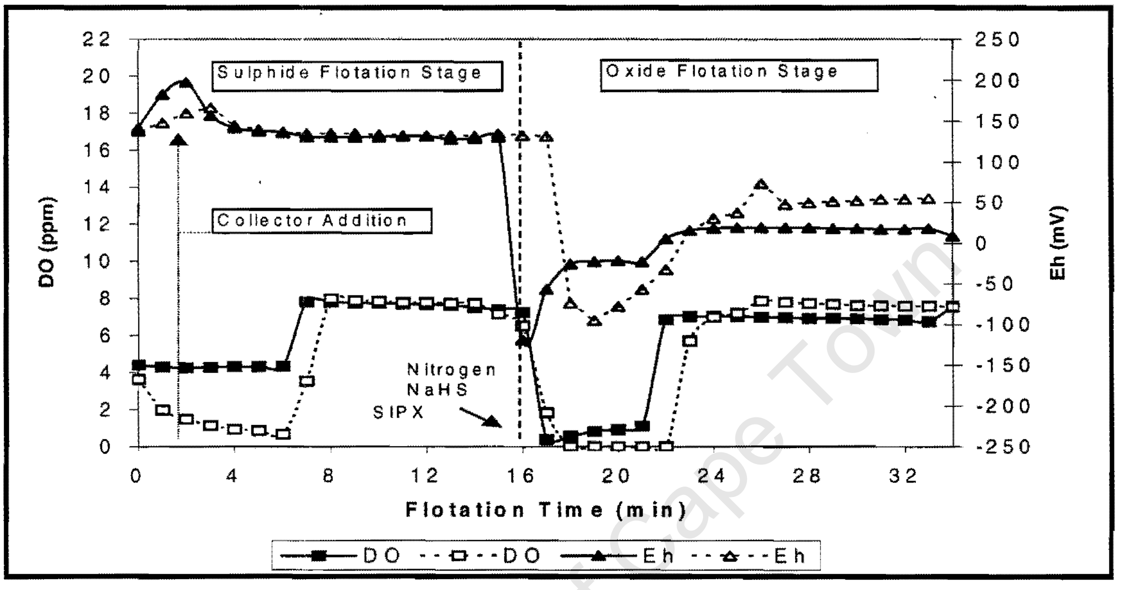


Figure 5.3.1.2: Grade-Mass Pull Profiles for Acid Soluble Copper (ASCu) Obtained during the Oxide Flotation Stage with and without Nitrogen Conditioning.

5.3.1.3 Pulp Chemistry Results

The pulp potential, pH, temperature and dissolved oxygen content were monitored. Figure 5.3.1.3 shows the pulp potential (Eh) and dissolved oxygen (DO) content as a function of time from the start of the flotation test. Detailed pulp chemistry measurements are given in appendix D3.



NB: Continuous lines for Standard Test (Air) and dotted lines for Nitrogen conditioning

Figure 5.3.1.3: Dissolved (DO) and Pulp Potential (Eh) Profiles obtained during Flotation with and without Nitrogen Conditioning.

As seen from the figure, the oxygen content of the pulp after grinding was less than 5 ppm. This increased rapidly through aeration to between 6-8 ppm during the sulphide flotation stage and decreased rapidly during nitrogen conditioning to almost zero before the oxide flotation stage. The dissolved oxygen and pulp potential were almost similar with or without nitrogen conditioning because mild steel media also consumes oxygen and lowers the potential as observed in the results of phase I.

The effectiveness of sulphidisation in the nitrogen environment can be explained in terms of interrupting galvanic interaction that results in the production of hydroxyl species of iron that interfere with NaHS and collector adsorption. The use of nitrogen to exclude oxygen increases the efficacy of new sulphide surface formation and improves process efficiencies by maximising the availability of sulphide ions for surface sulphidisation.

5.3.2 Nature of Fast and Slow Floating Sulphide Minerals

5.3.2.1 Chemical Analysis

The results of the chemical analysis are presented in Table 5.3.2.1 below.

Table 5.3.2.1: Results of Chemical Analysis

Fraction	Flot. Time (sec.)	Wt. (g)	%TCu	%ASCu	%AICu
Concentrate 1	20	16.39	42.65	4.50	38.15
Concentrate 2	40	14.44	31.03	4.25	26.78
Concentrate 3	60	14.87	22.35	4.45	17.9
Concentrate 4	240	11.24	14.79	3.06	11.73

5.3.2.2 Mineralogical Results

The mineralogical results are shown in Tables 5.3.2.2 (a) to (d). Microscopic examination of the concentrates showed that the fast-floating sulphide minerals consisted mainly of coarser and liberated mineral grains with lesser to minor amounts consisting of smaller mineral grains. The slow floating sulphide minerals composed of fines and partially locked coarser and smaller mineral grains.

Table 5.3.2.2 (a): Sulphide minerals present in concentrate 1.

Mineral	T (grains)	T*S.G	RA%	Wt%	%TCu	%ASCu
Chalcopyrite	112	470.40	17.56	10.03	3.46	-
Bornite	18	93.60	3.50	2.00	1.27	0.03
Chalcocite	338	1892.80	70.66	40.3	32.16	0.48
Pyrite	28	140.60	5.25	3.00	-	-
Carrollite	6	28.20	1.05	0.6	0.09	-
Native- copper	6	53.1	1.98	1.13	1.13	-
Total		2678.70	100.00		38.11	0.51

RA% = Relative Abundance

Table 5.3.2.2 (b): Sulphide Minerals Present in Concentrate 2

Mineral	T (grains)	T*S.G	RA%	Wt%	%TCu	%ASCu
Chalcopyrite	83	348.60	19.19	7.49	2.58	-
Bornite	13	67.60	3.72	1.45	0.92	0.02
Chalcocite	238	1332.8	73.38	28.65	22.86	0.34
Pyrite	8	40.16	2.21	0.86	-	-
Carrollite	2	9.40	0.52	0.20	0.03	-
Native-copper	2	17.70	0.98	0.38	0.38	-
Total		1816.261	100.00		26.77	0.36

Table 5.3.2.2 (c): Sulphide Minerals Present in Concentrate 3

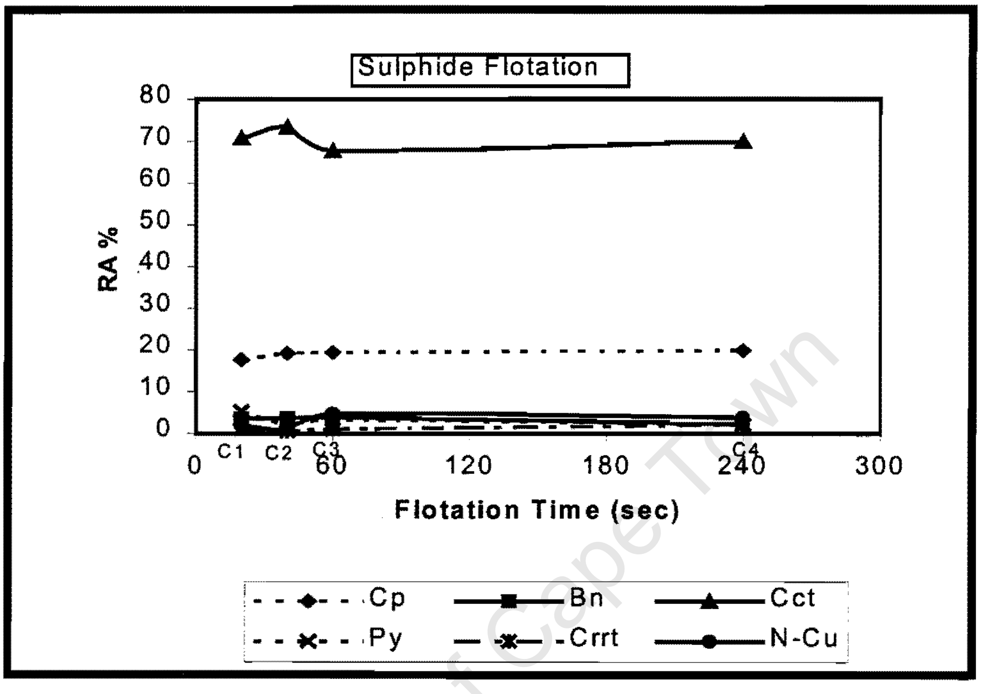
Mineral	T (grains)	T*S.G	RA%	Wt%	%TCu	%ASCu
Chalcopyrite	43	180.60	19.34	5.08	1.75	-
Bornite	7	36.40	3.90	1.02	0.65	0.01
Chalcocite	113	632.80	67.78	17.81	14.21	0.21
Pyrite	6	30.12	3.23	0.85	-	-
Carrollite	2	9.40	1.01	0.27	0.04	-
Native-copper	5	44.25	4.74	1.25	1.25	-
Total		933.57	100.00		17.9	0.22

Table 5.3.2.2 (d): Sulphide Minerals Present in Concentrate 4

Mineral	T (grains)	T*S.G	RA%	Wt%	%TCu	%ASCu
Chalcopyrite	11	46.20	19.88	3.42	1.18	-
Bornite	1	5.20	2.24	0.39	0.25	0.01
Chalcocite	29	162.40	69.89	12.03	9.60	0.14
Pyrite	1	5.02	2.16	0.37	-	-
Carrollite	1	4.70	2.02	0.35	0.05	-
Native-copper	1	8.85	3.81	0.66	0.66	-
Total		232.37	100.00		11.74	0.15

The mineralogical Tables show the distribution of sulphide minerals in the concentrate fractions obtained during the sulphide flotation stage. As seen from the results of the chemical and mineralogical analyses, approximately 90% of the total copper in concentrate 1 (38.15% of AICu in 45.65% TCu), 86% in concentrate 2, 80% in concentrate 3 and 79% in concentrate 4 occur in acid insoluble form derived from mainly chalcocite and lesser to minor amounts being contributed by chalcopyrite, bornite, native copper and carrollite.

Figure 5.3.2.3 shows the percent relative abundance of the sulphide minerals: chalcopyrite (Cp), bornite (Bn), chalcocite (Cct), pyrite (Py), carrollite (Crrt) and native copper (N-Cu) as a function of flotation time. It can be observed that in all concentrate fractions chalcocite was the predominant and fast floating sulphide mineral.



NB: C1 = Concentrate 1, C2 = Concentrate 2, etc.

Figure 5.3.2.3: Percent Relative Abundance as a Function of Flotation time for Sulphide Minerals Obtained in the Sulphide Flotation Stage.

The trend of the relative abundance of sulphide minerals in the concentrates follows that of the feed as shown in Table 5.3.2.4 below.

Table 5.3.2.4: Summary of Relative Abundance of Feed and Concentrate

Sulphide Minerals	Relative Abundance (RA%)				
	Feed	Conc. 1	Conc. 2	Conc. 3	Conc. 4
Chalcopyrite	26.31	17.56	19.19	19.34	19.88
Bornite	3.35	3.50	3.72	3.90	2.24
Chalcocite	62.50	70.66	73.38	67.78	69.89
Pyrite	7.17	5.25	2.21	3.23	2.16
Carrollite	0.38	1.05	0.52	1.01	2.02
Native-Copper	0.29	1.98	0.98	4.74	3.81
Total	100.00	100.00	100.00	100.0	100.00

It can be seen from Table 5.3.2.4 that the relative abundance of chalcopyrite goes down from feed to concentrates, that of bornite stays the same and that of chalcocite goes up. Also the relative abundance of pyrite decreased from feed to concentrates while that of carrollite and native copper increased.

CHAPTER 6

Discussion

6.0 DISCUSSION

As discussed in the introduction, Nchanga copper ore is made up of sulphide and oxide minerals and processing consists of sulphide flotation followed by sulphidisation and an oxide flotation step.

The difference in the recovery of acid soluble copper in the standard flotation tests after milling in mild steel media, between Tepa (1998) and those obtained in Phase I, could be attributed to the ineffective sulphidiser (NaHS) used in Phase I of this work. In Tepa's (1998) work the recoveries obtained were 59.6% ASCu, 97.3% AICu and those I obtained in this work were 16.3 % ASCu and 84.2 % AICu. After regrinding the sulphide rougher tails, the recovery of acid soluble copper obtained by Tepa and those of Phase I were 81.8% and 22.4% respectively. In both cases, regrind caused an increase in the recovery of acid soluble copper. The ore in both cases was ground at the same mesh of grind (~ 55% < 75 microns for the standard grind and 80% < 75 microns for regrind). Tepa (1998) did his flotation tests in a Denver laboratory cell (at the Mineral Investigation Department, ZCCM) and the tests of Phase I were done in a modified Leeds laboratory cell (UCT).

The standard flotation tests were compared between Phase I (UCT) and Phase III (UNZA). It was found that the difference in the mean recovery of acid insoluble copper between the two Phases was not significant (84.2% versus 80.9%). The difference in the recovery of acid soluble copper was found to be significant (16.3% versus 63.2%) and has been attributed to poor quality NaHS used in Phase I.

6.1 Effect of Milling Media on Flotation Response

The combined flotation recovery of both acid soluble copper and acid insoluble copper was better in pulp ground in stainless steel media. The overall flotation recovery obtained after regrinding in mild steel media was similar to that obtained without regrind in stainless steel media. This showed that the increase observed with mild steel media after regrind was not linked to liberation but to some other factors influenced by the pulp chemical condition such as dissolved oxygen level and pulp potential.

The slightly lower flotation recoveries of sulphide minerals obtained with mild steel grinding media, compared to that obtained with stainless steel grinding media, may be linked to the coatings of hydroxyl complexes of iron on mineral surfaces. These coatings are hydrophilic and impair the floatability of sulphide minerals by interfering with xanthate adsorption (Rao and Natarajan, 1960) as discussed in Chapter 2.

The grade of the concentrate obtained after regrinding with mild steel media was better than that obtained without a regrind step. In other words, there was better selectivity after regrinding. Stainless steel regrind did not have any significant effect on the recovery but improved the grade of the acid soluble copper. The mass pulls of the acid soluble copper after regrind in both milling media was lower than without regrind, but was accompanied by an increase in grade, showing that it was not a mass pull or froth stabilisation effect as observed by Bradshaw et al. (1999), but rather improved valuable mineral recovery.

More acid soluble copper, in pulp ground with stainless steel, was recovered in the sulphide flotation step than with mild steel media. With mild steel media, the increase in the recovery of acid soluble copper was only seen after regrinding the sulphide rougher tails, and the effectiveness of sulphidisation has been ascribed to the lower dissolved oxygen level in pulp. As discussed in the literature, the presence of dissolved oxygen during sulphidisation affects the efficiency of the process. Sulphidising agents can react with oxygen to form sulphoxy species such as thiosulphate ions, sulphite ions and sulphate ions (Hecker et al, 1985), and as a result, the sulphidising ions are consumed by dissolved oxygen. The overall recovery of acid soluble copper obtained with ore milled with stainless steel, without a regrind step, was similar to that obtained with mild steel regrind and collector added. It is interesting to note that, with ore milled with stainless steel media, most of the acid soluble copper (oxides) was recovered during the sulphide rougher flotation stage. The low oxide recoveries in both media have been masked by the poor quality NaHS used and should be further investigated with better quality NaHS.

The oxygen level in pulp ground in stainless steel media did not show any significant change as compared to pulp ground in mild steel media in which the oxygen level dropped during conditioning. Additionally, the pulp potentials in pulp ground in stainless steel media were slightly higher than that ground in mild steel media. This indicates that corrosion of mild steel takes place during grinding and that it consumes oxygen and lowers the potential as observed by Graham et al. (1982) and Johnson et al (1982). The pH range in both pulps was similar.

The influence of grinding on the floatability of some sulphide minerals was studied in terms of galvanic interactions (Rao et al., 1976; Berglund and Forssberg, 1987; Forssberg et al., 1988). Such galvanic interactions occur between minerals and stainless steel, but are stronger between mild steel media and sulphide minerals. The type of grinding media influenced the surface chemistry of the sulphide mineral and consequently its flotation behaviour as discussed before. In wet grinding of sulphides in a conventional mill the mineral-grinding media interaction plays a dominant role since the particles are always in contact with the grinding media (iron) which has a lower rest potential than the sulphide minerals. The fact that grinding media type brings about different flotation conditions is seen from the pulp chemistry measurements obtained during flotation.

Therefore, as discussed in the literature, mild steel grinding in this investigation had the following effects on flotation (Graham and Heathcote, 1982; Johnson et al, 1982):

(1) Lowered dissolved oxygen concentration in the pulp
(2) Lowered pulp potential
(3) Increased opportunity for galvanic interaction between mineral and media
(4) Increased iron hydroxyl species in solution, which coat the sulphide mineral surfaces and impaired collector adsorption.

The low flotation recovery of acid soluble copper at UCT (16.3% ASCu) compared to that obtained at UNZA (63.2% ASCu) has been attributed to the NaHS used at UCT which was ineffective. The ineffective NaHS could have been oxidised to form sulphoxy species such as thiosulphate ions, sulphite ions and sulphate ions, thereby reducing the sulphidising ions. The Sulphidiser (NaHS) used at UNZA was freshly supplied by Nchanga concentrator. Both sets of experiments were done in a modified laboratory Leeds flotation cell.

6.2 Pulp Chemistry Measurements in Laboratory (Phases I and III) and at Nchanga Concentrator (Phase II)

Generally the pulp chemistry parameters obtained in the plant (Phase II) were similar to those obtained in the laboratory with mild steel media (Phase I and Phase III). The dissolved oxygen (DO) level in the cyclone overflow (flotation feed) was lower than that of the flotation banks due to aeration in the latter. The dissolved oxygen levels determined in the cyclone overflow and flotation banks, relate to the DO levels obtained in the laboratory with mild steel milling media during conditioning and flotation respectively. There was a slight reduction in pulp potential in the oxide flotation banks equivalent to that obtained in the laboratory before the flotation of oxide minerals when NaHS was added. The pulp potential in the oxide rougher banks (ORB) was lower compared to that obtained in the cyclone overflow (COF), sulphide rougher banks (SRB) and oxide cleaner banks (OCB) because of the effect of NaHS. The flowsheet for Nchanga underground concentrator copper circuit is given in Figure 3.2.1, in chapter 3.

The measurement of dissolved oxygen concentration and pulp potential in the sulphide rougher banks in the plant was done to indicate the efficiency of the xanthate-mineral reaction. Thiol collectors like xanthates adsorb on sulphide minerals via a mixed potential mechanism, involving anodic oxidation of collectors and cathodic reduction of oxygen. If the dissolved oxygen level in the pulp was too low, the xanthate-mineral reaction would be limited. In flotation the collector molecule approaches the mineral surface and is adsorbed only after it donates electrons to the mineral particle. These electrons migrate to an oxygen rich point on the surface where they react with oxygen to form OH^- ions or H_2O or H_2O_2. This is explained in detail in Chapter 2, section 2.6.2.

In the oxide rougher banks, the dissolved oxygen can react with sulphidising reagents to form sulphoxy species such as thiosulphate ions ($S_2O_3^{2-}$), sulphite ions (SO_3^{2-}) and sulphate ions (SO_4^{2-}) (Hecker et al, 1985). As a result, sulphidising ions (HS-) are consumed by dissolved oxygen in significant quantities as explained in Chapter 2, section 2.7.2 on the Maxifloat™ technique.

The importance of this section is that it shows that we can translate behaviour of flotation pulps milled with mild steel media in laboratory to Plant behaviour, and can use laboratory tests to simulate Plant performance e.g. the effects of sulphidising in the nitrogen environment and regrind on the flotation recovery of problematic oxide minerals.

DO was less than 4 ppm and rose to about 7 ppm during flotation both in the laboratory (Phase I and Phase III) and in the Plant (Phase II).

The pulp potential profiles obtained in the plant also follow that of the mild steel laboratory-milled ore of Phase I. The pulp potentials obtained in Phase III were relatively lower due to the greater effect of NaHS. The NaHS used in Phase I was found to be less effective than that of Phase III.

6.3 Effect of Nitrogen Conditioning on the recovery of Oxide Minerals

There was an improvement in the recovery of oxide minerals after sulphidising in the nitrogen environment. From a metallurgical viewpoint nitrogen has two effects of interest. Firstly, it lowers the activity of oxygen in the pulp and, secondly, it reduces the pulp potential.

Although the Maxifloat™ (Clark et al, 200) procedure was not fully utilised, the influence observed after sulphidising in the nitrogen environment in the recovery of acid soluble copper in this research was significant. Clark et al (2000) also reports the improvement in the recovery of valuables after applying the Maxifloat™ technique as discussed in the literature (Chapter 2).

The reduced activity of oxygen may also influence galvanic interactions between sulphide minerals. Nakazawa and Iwasaki (1986, 1986), Kocabag and Smith (1985) and Rao and Finch (1988) have studied galvanic interactions. It has been shown that the more cathodic mineral draws electrons from the less cathodic one upon contact. The presence of oxygen appears to be essential in this galvanic coupling: oxygen acting as an electron acceptor probably reacts with the transferred electron to form hydroxyl ions (OH^-) (Woods, 1976). The use of nitrogen would interrupt this galvanic coupling.

The effect of nitrogen is probably due to purging of oxygen from the system, rather than to any reaction of the minerals with nitrogen. Martin and co-workers (1989) who used another inert gas, argon, which gave results similar to those using nitrogen, confirmed this. The removal of oxygen by nitrogen has three consequences, which combine to promote flotation: (1) it weakens the galvanic coupling; (2) it prevents formation of OH^- ions; and (3) it removes OH^- ions which compete with xanthate or hydrosulphide ions (HS^-) for adsorption sites (Rao and Finch, 1988).

Therefore, carrying out sulphidisation in the nitrogen environment is expected to enhance the adsorption of HS^- ions on the oxide mineral surface since there will be less OH^- ions to compete with for adsorption sites. Under these conditions collector adsorption on the mineral surface will be facilitated to promote flotation, as there are less hydration layers on the mineral surface. Hydration layers on the mineral surface hinder collector adsorption and as a consequence retard the flotation of the desired minerals.

The mass pulls of oxide concentrates after nitrogen conditioning increased. The high mass recoveries and lower grades of acid soluble copper obtained after nitrogen conditioning indicate that the froth zone was stabilised, increasing the recovery by entrainment in addition to improved recovery of acid soluble copper.

However, it must be mentioned that after sulphidising in the nitrogen environment, the dissolved oxygen level was raised to above 2 ppm by aeration before collector was added. This is because oxygen is generally held to be essential for xanthate film formation (Leja, 1982).

The use of nitrogen to exclude oxygen increases the efficacy of new sulphide surface formation and improves process efficiencies by maximising the availability of sulphide ions for surface sulphidisation to occur at the sulphide consumption levels.

As discussed in the literature, the presence of dissolved oxygen during sulphidisation conditioning is expected to adversely affect process efficiency in the following ways (Clark et al, 2000):

1. Sulphidising reagents can react with oxygen to form sulphoxy species such as thiosulphate ions ($S_2O_3^{2-}$), sulphite ions (SO_3^{2-}) and sulphate ions (SO_4^{2-}) (Hecker et al., 1985). As a result, sulphidising ions (HS^-) are consumed by dissolved oxygen in significant quantities, necessitating additional quantities of sulphidiser to be used in order to achieve satisfactory surface sulphidisation.

2. Dissolved oxygen would react with the freshly formed sulphide layer. Sulphidisation of secondary copper minerals proceeds through a two-stage process (Zhou et al, 1993). The first stage involves the formation of a primary sulphidised layer and the second involves

precipitation of copper ions that have diffused through the primary layer. The presence of dissolved oxygen is believed to disrupt these reactions by reacting with freshly formed sulphide surface and thereby resulting in a spongy, less coherent sulphidised layer. This is evident from leach tests performed on sulphidised copper minerals in the presence of oxygen and nitrogen atmospheres (Orwe et al, 1997). A coherent, compact sulphidised layer can only be achieved by eliminating dissolved oxygen with nitrogen purging.

6.4 Fast- and Slow-Floating Sulphide Minerals

The fast floating sulphide minerals consisted mainly of fairly liberated coarser mineral grains with minor to lesser amounts comprising smaller mineral grains. The slow-floating fraction comprised disseminated and partially locked coarser and finer mineral grains. Chalcocite, the most abundant mineral, was found to be the fast floating sulphide mineral. The slow floating minerals were chalcopyrite, bornite and carrollite. Liberation and the nature of these minerals affected their floatability.

In the froth flotation process, the upper size limit of the recoverable particles is determined by what an air bubble can lift; i.e. it is dependent on the size, surface characteristics, composition, and relative density of the mineral. The flotation efficiency decreases towards the upper and lower grain size limits as follows: slimes (smaller than 10 microns) do not easily attach to bubbles, and the larger grains are too heavy to adhere to the bubbles and thus are not readily transported. Locked mineral grains tend to float very slowly because of the incomplete coverage of collector on the mineral surface.

While the slow recovery rate of fine particles has been attributed to decreased particle-bubble collisions, that of coarse particles has been attributed to the disruption of bubble-particle aggregate in turbulent zones (Morris, 1950; Schultze, 1977). One of the reasons for the low flotation rate of coarse particles is that, with increasing particle size, the density of the bubble-particle aggregate approaches that of the pulp density and thereby the aggregate becomes less buoyant (Jameson et al., 1977).

In general, the recovery of slow-floating partially locked mineral particles can be improved by further grinding if the proportion of the slow-floating fine particles is very low. If the proportion of fine particles is very high, the material should be deslimed before further grinding.

As discussed in the literature, different sulphide minerals have different floatabilities. The structure and characteristics of sulphide mineral samples from different locations vary (Wards, 1996) and this results in differences in the flotation behaviour of samples (Bothelho de Sousa, 1984). The reactions of these minerals, such as the oxidation characteristics and interactions with water or other reagents used in flotation, are known to strongly influence the conditions necessary for successful flotation.

The surfaces of sulphide minerals oxidise in the presence of water and oxygen, with the reduction of oxygen being facilitated by the semiconducting properties of the mineral. This oxidation hinders flotation performance, however in some circumstances reduced performance can be overcome by increased collector addition (Leja, 1982).

Different sulphide minerals oxidise at different rates and the extent of oxidation and the product of oxidation depend on the mineral under investigation, pH, and time of contact (Subrahmanyam and Forssberg, 1993). In general the minerals which oxidise to form sulphur exhibit the highest flotation rates (Woods, 1984) and, while the sulphur rich surface can promote floatability, the presence of oxy-sulphur species depresses flotation.

CHAPTER 7

Conclusions and Recommendations

7.0 CONCLUSIONS AND RECOMMENDATIONS

7.1 Conclusions

(a) The combined flotation recovery of both copper sulphide and oxide minerals were higher in pulp ground in stainless steel media than that ground in mild steel media. This showed the benefit of higher dissolved oxygen in sulphide flotation stage. Regrind with stainless steel media after the sulphide rougher flotation stage did not benefit the recovery but improved the grade of oxide minerals obtained in the oxide flotation stage, which corresponded to no observed change in pulp chemistry measurements (DO and Eh).

(b) The recovery of acid soluble copper in the sulphide flotation stage after stainless steel milling was higher than after mild steel milling. This corresponded to the presence of high dissolved oxygen levels in the pulp and showed that collector adsorption was better after stainless steel milling.

(c) With mild steel, the sulphide rougher stage was not as efficient as stainless steel. However, mild steel regrind benefited sulphidisation in the oxide flotation stage and this corresponded to pulp chemistry measurements showing greater drop in dissolved oxygen (DO) and pulp potential (Eh).

(d) The benefit observed in the recovery of acid soluble copper with mild steel regrind corresponded to Tepa's finding (1998), although to a lesser extent, due to the poor quality NaHS. The fact that the results from mild steel regrind (MSR) and stainless steel with no regrind (SS) were similar showed that the improvement observed in the recovery of acid soluble copper after regrind with mild steel media was not a liberation problem, but rather that the effectiveness of flotation was governed by pulp chemistry.

(e) The quality of the NaHS used was found to be poor only after the experiments of Phase I were done. This accounts for the low recoveries of acid soluble copper obtained in Phase I compared to Phase III and Tepa's results.

(f) Since the same recovery of copper could be obtained with stainless steel without a regrind step and additional collector, or with mild steel regrinding and additional collector, an economic evaluation of the best condition to be used is needed, after confirmatory tests with better NaHS.

(g) In Phase II the pulp chemistry parameters obtained in the plant at Nchanga concentrator were similar to those obtained in the laboratory flotation tests with mild steel milling media in Phase I. This indicates that the pulp chemical conditions were similar and changes to flotation performance of copper observed in the laboratory could be extrapolated to Nchanga concentrator, such as the effects of regrinding the sulphide rougher tails and sulphidising in the nitrogen environment. It is recommended that mild steel milling media be used to evaluate the effects of the parameter under investigation such as regrind or nitrogen conditioning. Lowering of potential with mild steel in the laboratory, improved the efficiency of sulphidisation, and indications are that by lowering potential in the Plant (by regrind or nitrogen conditioning), sulphidisation and the resulting acid soluble copper recovery could be improved.

(h) Flotation recovery of oxide minerals was improved by carrying out sulphidisation in the nitrogen environment in Phase III. The use of nitrogen to exclude oxygen increases the efficacy of new sulphide surface formation and improves process efficiencies by maximising the availability of sulphide ions for surface sulphidisation to occur.

(i) The effect of nitrogen and regrind on the flotation recovery of acid soluble copper could not be directly compared due to the differences in the quality of NaHS between Phase I and Phase III.

(j) The fast-floating sulphide minerals occurred in the coarser size range and were liberated. Fines, dissemination, and partially locked mineral grains characterised the slow floating sulphide minerals. Chalcocite was the predominant mineral in the feed, as well as the fast floating sulphide mineral and as such the usual one stage sulphide flotation should be continued.

(k) The slow-floating sulphide minerals consisted of chalcopyrite, bornite and carrollite. Slow floating particles were partially locked mineral grains and recovery could be improved by further grinding to improve liberation, or with better collector.

7.2 Recommendations

(a) The economic comparison of using stainless steel milling media and mild steel regrinding should be done.

(b) Since there is an increase in the flotation recovery of the acid soluble copper after sulphidising in the nitrogen environment, further work is recommended to find out whether the NaHS dose rate could be reduced, and the economics investigated.

(c) The effective of sulphidising in the nitrogen environment and regrind after mild steel milling should be directly compared.

(d) The effect of mild steel and stainless steel regrind with good quality NaHS should be investigated.

(e) Chalcocite is the most abundant and fast floating sulphide mineral and as such the usual one stage sulphide rougher flotation followed by one stage oxide rougher flotation should be continued.

(f) Other collectors for sulphides must be tested in an attempt to improve the recovery of the slow-floating minerals.

CHAPTER 8

References

8.0 REFERENCES

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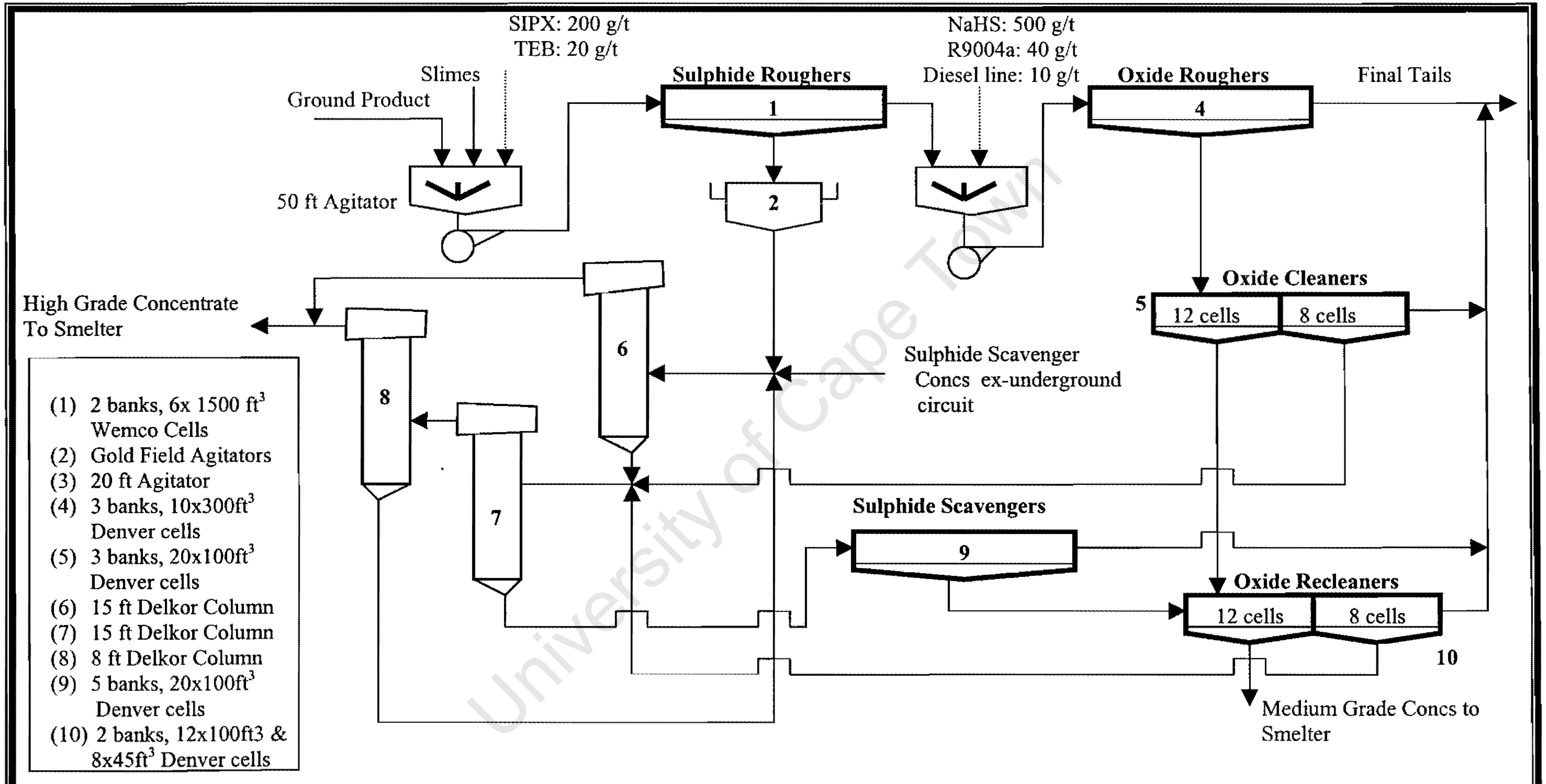
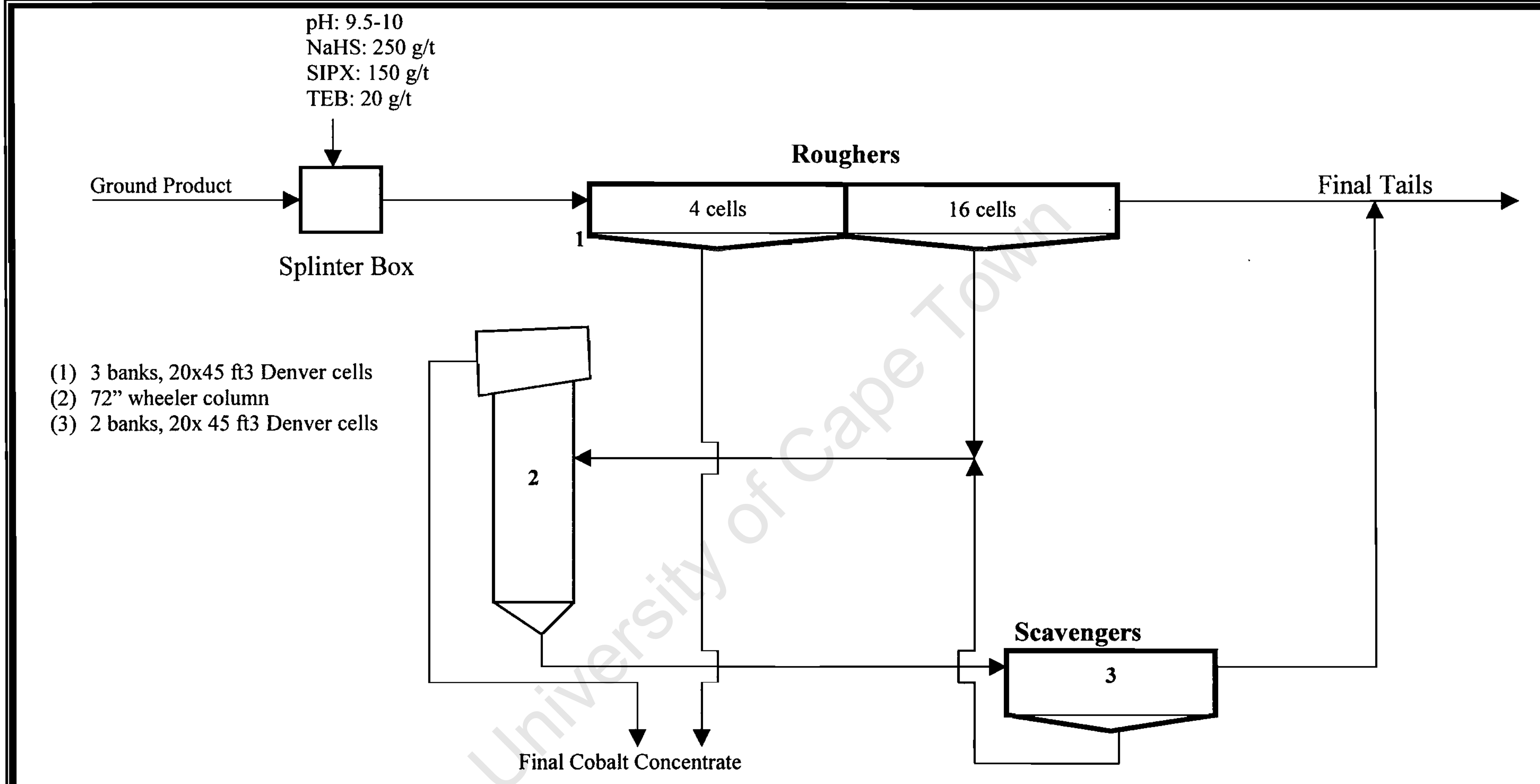
APPENDIX A

Plant Flow sheets



Nchanga Open-pit Concentrator Copper Circuit



Nchanga Concentrator Cobalt Circuit

APPENDIX B

Chemical Analysis Methods

DIGESTION METHODS FOR TCu AND ASCu

(A) Analysis of Total Copper (TCu) [ex Karbochem R & D Lab]

Principle

The concentrate or finely milled ore sample is dissolved in aqua regia with the addition of hydrofluoric acid and perchloric acid. The sample is made up to a known volume, filtered and read on the AA spectrophotometer.

Reagents

Hydrofluoric acid	CP	approx. 40% concentration
Nitric acid	CP	approx. 60% concentration
Perchloric acid	CP	concentrated
Hydrochloric acid	CP	approx. 30% concentration

For use: Mix 4 parts HCl to 1 part HF per volume.

CP = Chemically Pure

Method:

1. Sample is weighed out according to the type, into a 250 ml wide mouthed Erlenmeyer flask as follows:

Concentrates........................0.1g

Tails and feed (head)..............0.5g

2. Digest:

Add 10 ml of the HCl/HF mixture to sample and heat to boiling.

Add 10 ml HNO_3 to flask and boil until sample volume is approx. 2 ml.

Add 5 ml $HClO_4$ to flask and boil until sample volume is approx. 2 ml (a white cloud will form which will rise once reaction has taken place).

3. Make up to volume and filter:

Transfer the sample quantitatively to a 100-ml volumetric flask and make up to 100 ml with distilled H_2O. Filter through Whatman No1 into sample bottle. Filtrate to be read on AA. Store in fridge if delay in reading.

4. Read on AA.
5. Calculation

E.g. $\%TCu \text{ in sample} = \dfrac{R \text{ (in ppm)}}{100 \times a \text{ (sample wt)}}$

(B) Analysis of Acid Soluble Copper (ASCu) [Nchanga method]

Preparation of Leaching Acid

Add 400 ml concentrated sulphuric acid slowly and with stirring to one litre of distilled water in a two-litre boiling flask. Allow cooling. Add 36g of sodium sulphite (Na_2SO_3) or 72g of sodium sulphite hepta-hydrated ($Na_2SO_3.7H_2O$) and, when dissolved, dilute solution to 8 litres with distilled water.

Procedure

1. Weigh 1.0g sample into a plastic bottle.
2. Add 25 ml of leaching acid and swirl the contents of the bottle and allow to stand for one minute for the gases to escape.
3. Stopper the bottle and shake sample on a mechanical shaker for one hour.
4. Filter through Whatman No 541 filter paper into a 100-ml volumetric flask and wash the bottle thoroughly into the filter paper with distilled or deionised water.
5. Wash the residue and the filter paper several times with water and finally dilute to the mark with distilled water.
6. Aspirate the standard and sample solutions into the frame of the atomic absorption spectrophotometer (AAS).

Calculation.

Calculation of %ASCu same as that for TCu.

APPENDIX C1

Detailed Metallurgical Results of the Effect of Milling Media

Table 5.1.1A: Metallurgical Balance Sheet [Mild Steel Grinding (MS)]

Stage	Time (min)	Fraction	Weight				Grade (%)			Recovery (%)		
			Wt (g)	Wt%	S.C. Wt%	O.C Wt%	TCu	ASCu	AlCu	TCu	ASCu	AlCu
Sulphide Flotation	0.3	Conc. 1	24.65	2.32	2.32	2.32	43.79	2.83	40.96	30.2	4.9	46.4
	0.6	Conc. 2	15.75	1.48	3.80	3.80	35.78	3.44	32.34	15.8	3.9	23.4
	1	Conc. 3	7.45	0.70	4.51	4.51	23.88	3.07	20.81	5.0	1.6	7.1
	4	Conc. 4	7.25	0.68	5.19	5.19	15.02	2.41	12.61	3.0	1.3	4.2
Combined Sulphide Concentrates			**55.10**	**5.19**			35.02	2.98	32.04	**54.0**	**11.7**	**81.1**
Oxide Flotation	0.5	Conc. 5	18.20	1.71	1.71	6.90	2.37	1.39	0.98	1.2	1.8	0.8
	1	Conc. 6	9.59	0.90	2.62	7.80	3.42	1.66	1.76	0.9	1.1	0.8
	2.5	Conc. 7	6.38	0.60	3.22	8.41	3.89	1.85	2.04	0.7	0.8	0.6
	4	Conc. 8	3.59	0.34	3.56	8.74	3.59	1.77	1.82	0.4	0.5	0.3
Combined Oxide Concentrates			**37.76**	**3.56**			3.01	1.57	1.44	**3.2**	**4.3**	**2.5**
Oxide Rougher Tails			969.19	91.26	91.26	100.00	1.58	1.21	0.37	42.8	84.1	16.5
Head (Calculated)			1062.05	100.00			3.37	1.31	2.05	100.0	100.0	100.0
Head (Assay)							3.54	1.33	2.21			
Fractions	**Cum. Time (min)**	**Water Recovery**		**Cumulative Grade (%)**			**Stage cumulative Recovery (%)**			**Overall Cumulative Recovery (%)**		
		Wt (g)	**Wt%**	**TCu**	**ASCu**	**AlCu**	**TCu**	**ASCu**	**AlCu**	**TCu**	**ASCu**	**AlCu**
Conc. 1	0.3	22.96	5.46	43.79	2.83	40.96	30.2	4.9	46.4	30.2	4.9	46.4
Conc. 2	1	58.73	13.96	40.67	3.07	37.60	46.0	8.8	69.8	46.0	8.8	69.8
Conc. 3	2	32.48	7.72	38.05	3.07	34.99	50.9	10.4	76.9	50.9	10.4	76.9
Conc. 4	6	17.30	4.11	35.02	2.98	32.04	54.0	11.7	81.1	54.0	11.7	81.1
Conc. 5	6.5	133.72	31.78	2.37	1.39	0.98	1.2	1.8	0.8	55.2	13.5	81.9
Conc. 6	7.5	90.31	21.47	2.73	1.48	1.25	2.1	3.0	1.6	56.1	14.6	82.7
Conc. 7	10	41.92	9.96	2.95	1.55	1.40	2.8	3.8	2.2	56.8	15.5	83.3
Conc. 8	14	23.30	5.54	3.01	1.57	1.44	3.2	4.3	2.5	**57.2**	**15.9**	**83.6**
Oxide Rougher Tails							42.8	84.1	16.5	100.0	100.0	100.0
Heads (Calculated)				3.37	1.31	2.05	100.0	100.0	100.0			

Table 5.1.1B: Metallurgical Balance Sheet [Mild Steel Grinding (MS)]

Stage	Time (min)	Fraction	Weight				Grade (%)			Recovery (%)		
			Wt (g)	Wt%	S.C. Wt%	O.C Wt%	TCu	ASCu	AICu	TCu	ASCu	AICu
Sulphide Flotation	0.3	Conc. 1	27.69	2.58	2.58	2.58	40.55	2.67	37.88	31.9	5.1	50.0
	0.6	Conc. 2	16.60	1.55	4.13	4.13	32.80	3.51	29.29	15.5	4.1	23.2
	1	Conc. 3	6.57	0.61	4.75	4.75	21.29	3.28	18.01	4.0	1.5	5.6
	4	Conc. 4	6.24	0.58	5.33	5.33	12.72	2.54	10.18	2.3	1.1	3.0
Combined Sulphide Concentrates			**57.10**	**5.33**			33.04	2.97	30.07	**53.6**	**11.8**	**81.8**
Oxide Flotation	0.5	Conc. 5	24.88	2.32	2.32	7.65	2.28	1.34	0.94	1.6	2.3	1.1
	1	Conc. 6	10.90	1.02	3.34	8.67	3.54	1.72	1.82	1.1	1.3	0.9
	2.5	Conc. 7	6.43	0.60	3.94	9.27	3.98	1.84	2.14	0.7	0.8	0.7
	4	Conc. 8	2.40	0.22	4.16	9.49	3.90	1.88	2.02	0.3	0.3	0.2
Combined Oxide Concentrates			**44.61**	**4.16**			2.92	1.53	1.39	**3.7**	**4.8**	**3.0**
Oxide Rougher Tails			970.12	90.51	90.51	100.0	1.55	1.22	0.33	42.7	83.3	15.3
Head (Calculated)			1071.83	100.00			3.28	1.33	1.96	100.0	99.9	100.0
Head (Assay)							2.98	1.27	1.71			

Fractions	Cum. Time (min)	Water Recovery		Cumulative Grade (%)			Stage cumulative Recovery (%)			Overall Cumulative Recovery (%)		
		Wt (g)	Wt%	TCu	ASCu	AICu	TCu	ASCu	AICu	TCu	ASCu	AICu
Conc. 1	0.3	52.33	10.85	40.55	2.67	37.88	31.9	5.1	50.0	31.9	5.1	50.0
Conc. 2	1	41.89	8.69	37.65	2.98	34.66	47.4	9.2	73.2	47.4	9.2	73.2
Conc. 3	2	19.96	4.14	35.53	3.02	32.51	51.3	10.7	78.8	51.3	10.7	78.8
Conc. 4	6	16.11	3.34	33.04	2.97	30.07	53.6	11.8	81.8	53.6	11.8	81.8
Conc. 5	6.5	171.89	35.65	2.28	1.34	0.94	1.6	2.3	1.1	55.2	14.1	82.9
Conc. 6	7.5	112.59	23.35	2.66	1.46	1.21	2.7	3.7	2.1	56.3	15.5	83.9
Conc. 7	10	54.05	11.21	2.86	1.51	1.35	3.4	4.5	2.7	57.0	16.3	84.5
Conc. 8	14	13.33	2.76	2.92	1.53	1.39	3.7	4.8	3.0	**57.3**	**16.6**	**84.8**
Oxide Rougher Tails							42.7	83.3	15.3	100.0	99.9	100.0
Heads (Calculated)				3.28	1.33	1.96	100.0	99.9	100.0			

Table 5.1.1C: Metallurgical Balance Sheet [Mild Steel Re-Grinding (MSR)]

Stage	Time (min)	Fraction	Weight				Grade (%)			Recovery (%)		
			Wt (g)	Wt%	S.C. Wt%	O.C Wt%	TCu	ASCu	AICu	TCu	ASCu	AICu
Sulphide Flotation	0.3	Conc. 1	24.19	2.30	2.30	2.30	43.51	2.71	40.80	31.3	4.8	49.0
	0.6	Conc. 2	12.46	1.19	3.49	3.49	33.98	3.04	30.94	12.6	2.8	18.8
	1	Conc. 3	6.56	0.62	4.11	4.11	25.27	2.62	22.65	4.9	1.3	7.4
	4	Conc. 4	7.13	0.68	4.79	4.79	13.97	2.13	11.84	3.0	1.1	4.2
Combined Sulphide Concentrates			**50.34**	**4.79**			34.59	2.70	31.89	**51.9**	**10.0**	**79.3**
Oxide Flotation	0.5	Conc. 5	11.52	1.10	1.10	5.89	22.05	5.82	16.23	7.6	5.0	9.3
	1	Conc. 6	10.36	0.99	2.08	6.87	11.07	4.51	6.56	3.4	3.5	3.4
	2.5	Conc. 7	8.86	0.84	2.93	7.72	6.78	3.53	3.25	1.8	2.3	1.4
	4	Conc. 8	7.30	0.69	3.62	8.41	4.56	3.03	1.53	1.0	1.6	0.6
Combined Oxide Concentrates			**38.04**	**3.62**			12.15	4.39	7.75	**13.8**	**12.5**	**14.7**
Oxide Rougher Tails			962.26	91.59	91.59	100.00	1.20	1.08	0.12	34.4	76.9	5.7
Head (Calculated)			10.50.64	100.00			3.20	1.28	1.92	100.0	99.4	99.7
Head (Assay)							3.30	1.25	2.05			
Fractions	**Cum. Time (min)**	**Water Recovery**		**Cumulative Grade (%)**			**Stage cumulative Recovery (%)**			**Overall Cumulative Recovery (%)**		
		Wt (g)	**Wt%**	**TCu**	**ASCu**	**AICu**	**TCu**	**ASCu**	**AICu**	**TCu**	**ASCu**	**AICu**
Conc. 1	0.3	36.53	14.28	43.51	2.71	40.80	31.3	4.8	49.0	31.3	4.8	49.0
Conc. 2	1	27.12	10.60	40.27	2.82	37.45	44.0	7.6	67.8	44.0	7.6	67.8
Conc. 3	2	11.08	4.33	37.99	2.79	35.20	48.9	8.9	75.2	48.9	8.9	75.2
Conc. 4	6	20.55	8.03	34.59	2.70	31.89	51.9	10.0	79.3	51.9	10.0	79.3
Conc. 5	6.5	40.54	15.85	22.05	5.82	16.23	7.6	5.0	9.3	59.4	15.0	88.7
Conc. 6	7.5	49.33	19.29	16.85	8.20	11.65	11.0	8.5	12.7	62.8	18.5	92.0
Conc. 7	10	26.75	10.46	13.95	4.72	9.23	12.8	10.8	14.1	64.6	20.8	93.5
Conc. 8	14	43.87	17.15	12.15	4.39	7.75	13.8	12.5	14.7	**65.6**	**22.4**	**94.0**
Oxide Rougher Tails							34.4	76.9	5.7	100.0	99.4	99.7
Heads (Calculated)				3.20	1.28	1.92	100.0	99.4	99.7			

Table 5.1.1D: Metallurgical Balance Sheet [Mild Steel Re-Grinding (MSR)]

Stage	Time (min)	Fraction	Weight				Grade (%)			Recovery (%)		
			Wt (g)	Wt%	S.C. Wt%	O.C Wt%	TCu	ASCu	AlCu	TCu	ASCu	AlCu
Sulphide Flotation	0.3	Conc. 1	13.30	1.30	1.30	1.30	41.48	2.47	39.01	17.2	2.6	26.8
	0.6	Conc. 2	15.65	1.52	2.82	2.82	36.77	2.62	34.15	17.9	3.2	27.2
	1	Conc. 3	9.61	0.94	3.76	3.76	29.86	2.49	27.37	8.9	1.9	13.6
	4	Conc. 4	7.30	0.71	4.47	4.47	17.62	2.05	15.57	4.0	1.2	5.9
Combined Sulphide Concentrates			**45.86**	**4.47**			33.64	2.46	31.18	**48.0**	**8.8**	**73.5**
Oxide Flotation	0.5	Conc. 5	14.51	1.41	1.41	5.88	21.08	4.70	16.38	9.5	5.3	12.4
	1	Conc. 6	15.46	1.51	2.92	7.39	10.51	3.81	6.70	5.1	4.6	5.4
	2.5	Conc. 7	9.16	0.89	3.81	8.28	6.90	3.34	3.56	2.0	2.4	1.7
	4	Conc. 8	4.36	0.42	4.24	8.70	5.05	3.19	1.86	0.7	1.1	0.4
Combined Oxide Concentrates			**43.49**	**4.24**			12.73	3.95	8.78	**17.2**	**13.4**	**19.8**
Oxide Rougher Tails			937.46	91.30	91.30	100.00	1.19	1.06	0.13	34.7	77.5	6.3
Head (Calculated)			10.26.81	100.00			3.13	1.24	1.88	100.0	99.8	99.7
Head (Assay)							3.07	1.23	1.84			

Fractions	Cum. Time (min)	Water Recovery		Cumulative Grade (%)			Stage cumulative Recovery (%)			Overall Cumulative Recovery (%)		
		Wt (g)	Wt%	TCu	ASCu	AlCu	TCu	ASCu	AlCu	TCu	ASCu	AlCu
Conc. 1	0.3	14.30	5.22	41.48	2.47	39.01	17.2	2.6	26.8	17.2	2.6	26.8
Conc. 2	1	26.60	9.70	38.93	2.55	36.38	35.1	5.8	54.1	35.1	5.8	54.1
Conc. 3	2	19.49	7.11	36.67	2.54	34.14	44.0	7.7	67.7	44.0	7.7	67.7
Conc. 4	6	17.97	6.56	33.64	2.46	31.18	48.0	8.8	73.5	48.0	8.8	73.5
Conc. 5	6.5	57.50	20.98	21.08	4.70	16.38	9.5	5.3	12.4	57.6	14.2	85.9
Conc. 6	7.5	82.40	30.06	15.63	4.24	11.39	14.6	9.9	17.7	62.6	18.8	91.3
Conc. 7	10	40.17	14.65	13.58	4.03	9.55	16.5	12.3	19.4	64.6	21.2	93.0
Conc. 8	14	15.68	5.72	12.73	3.95	8.78	17.2	13.4	19.8	**65.3**	**22.3**	**93.4**
Oxide Rougher Tails							34.7	77.5	6.3	100.0	99.8	99.7
Heads (Calculated)				3.13	1.24	1.88	100.0	99.8	99.7			

Table 5.1.1E: Metallurgical Balance Sheet [Stainless Steel Grinding (SS)]

Stage	Time (min)	Fraction	Weight				Grade (%)			Recovery (%)		
			Wt (g)	Wt%	S.C. Wt%	O.C Wt%	TCu	ASCu	AICu	TCu	ASCu	AICu
Sulphide Flotation	0.3	Conc. 1	20.32	1.89	1.89	1.89	41.58	5.16	36.42	23.8	6.8	36.4
	0.6	Conc. 2	14.83	1.38	3.27	3.27	42.01	4.88	37.13	17.5	4.8	26.9
	1	Conc. 3	6.68	0.62	3.89	3.89	28.30	4.10	24.20	5.3	1.8	8.0
	4	Conc. 4	10.84	1.01	4.89	4.89	18.14	3.38	14.76	5.5	2.4	7.9
Combined Sulphide Concentrates			**52.67**	**4.89**			35.19	4.58	30.61	**52.2**	**15.8**	**79.2**
Oxide Flotation	0.5	Conc. 5	30.02	2.79	2.79	7.68	10.96	2.11	8.85	9.3	4.2	13.2
	1	Conc. 6	10.04	0.93	3.72	8.62	7.85	2.68	5.17	2.2	1.8	2.6
	2.5	Conc. 7	5.66	0.53	4.25	9.14	4.80	2.05	2.75	0.8	0.8	0.8
	4	Conc. 8	4.98	0.46	4.71	9.61	3.31	1.75	1.56	0.5	0.6	0.4
Combined Oxide Concentrates			**50.7**	**4.71**			8.91	2.18	6.72	**12.7**	**7.3**	**16.9**
Oxide Rougher Tails			972.75	90.39	90.39	100.00	1.28	1.20	0.08	35.1	76.7	3.8
Head (Calculated)			1076.12	100.00			3.30	1.41	1.89	100.0	99.7	99.9
Head (Assay)							3.48	1.37	2.11			
Fractions	**Cum. Time (min)**	**Water Recovery**		**Cumulative Grade (%)**			**Stage cumulative Recovery (%)**			**Overall Cumulative Recovery (%)**		
		Wt (g)	**Wt%**	**TCu**	**ASCu**	**AICu**	**TCu**	**ASCu**	**AICu**	**TCu**	**ASCu**	**AICu**
Conc. 1	0.3	52.07	8.27	41.58	5.16	36.42	23.8	6.8	36.4	23.8	6.8	36.4
Conc. 2	1	48.09	7.64	41.76	5.04	36.72	41.3	11.5	63.4	41.3	11.5	63.4
Conc. 3	2	26.64	4.23	39.61	4.89	34.72	46.7	13.3	71.3	46.7	13.3	71.3
Conc. 4	6	116.45	18.49	35.19	4.58	30.61	52.2	15.8	79.2	52.2	15.8	79.2
Conc. 5	6.5	194.55	30.89	10.96	2.11	8.85	9.3	4.2	13.2	61.5	19.9	92.4
Conc. 6	7.5	83.99	13.33	10.18	2.25	7.93	11.5	5.9	15.7	63.7	21.7	94.9
Conc. 7	10	53.99	8.57	9.51	2.23	7.29	12.3	6.7	16.5	64.5	22.5	95.7
Conc. 8	14	54.07	8.57	8.91	2.18	6.72	12.7	7.3	16.9	**64.9**	**23.0**	**96.1**
Oxide Rougher Tails							35.1	76.7	3.8	100.0	99.7	99.9
Heads (Calculated)				3.30	1.41	1.89	100.0	99.7	99.9			

Table 5.1.1F: Metallurgical Balance Sheet [Stainless Steel Grinding (SS)]

Stage	Time (min)	Fraction	Weight				Grade (%)			Recovery (%)		
			Wt (g)	Wt%	S.C. Wt%	O.C Wt%	TCu	ASCu	AlCu	TCu	ASCu	AlCu
Sulphide Flotation	0.3	Conc. 1	18.43	1.84	1.84	1.84	41.74	4.59	37.15	23.6	6.1	36.1
	0.6	Conc. 2	15.95	1.60	3.44	3.44	37.11	4.80	32.31	18.1	5.6	27.0
	1	Conc. 3	10.23	1.02	4.46	4.46	27.91	4.28	23.63	8.7	3.2	12.7
	4	Conc. 4	12.70	1.27	5.73	5.73	14.17	3.20	10.97	5.5	3.0	7.3
Combined Sulphide Concentrates			**57.31**	**5.73**			31.87	4.29	27.59	**55.9**	**17.9**	**83.1**
Oxide Flotation	0.5	Conc. 5	16.09	1.61	1.61	7.34	8.19	1.90	6.29	4.0	2.2	5.4
	1	Conc. 6	10.85	1.09	2.69	8.43	5.82	2.08	3.74	1.9	1.7	2.1
	2.5	Conc. 7	6.74	0.67	3.37	9.10	4.27	1.82	2.45	0.9	0.9	0.9
	4	Conc. 8	2.90	0.29	3.66	9.39	3.58	1.70	1.88	0.3	0.4	0.3
Combined Oxide Concentrates			**36.58**	**3.66**			6.40	1.92	4.48	**7.2**	**5.1**	**8.7**
Oxide Rougher Tails			906.11	90.61	90.61	100.00	1.33	1.16	0.17	36.9	76.8	8.1
Head (Calculated)			1000.00	100.00			3.27	1.37	1.90	100.0	99.8	99.9
Head (Assay)							3.14	1.27	1.87			

Fractions	Cum. Time (min)	Water Recovery		Cumulative Grade (%)			Stage cumulative Recovery (%)			Overall Cumulative Recovery (%)		
		Wt (g)	Wt%	TCu	ASCu	AlCu	TCu	ASCu	AlCu	TCu	ASCu	AlCu
Conc. 1	0.3	55.33	9.37	41.74	4.59	37.15	23.6	6.1	36.1	23.6	6.1	36.1
Conc. 2	1	65.45	11.08	39.59	4.69	34.90	41.7	11.7	63.1	41.7	11.7	63.1
Conc. 3	2	50.97	8.63	36.91	4.59	32.32	50.4	14.9	75.8	50.4	14.9	75.8
Conc. 4	6	86.95	14.72	31.87	4.29	27.59	55.9	17.9	83.1	55.9	17.9	83.1
Conc. 5	6.5	122.11	20.68	8.19	1.90	6.29	4.0	2.2	5.4	60.0	20.1	88.5
Conc. 6	7.5	114.65	19.41	7.24	1.97	5.26	6.0	3.9	7.5	61.9	21.8	90.6
Conc. 7	10	75.39	12.77	6.64	1.94	4.70	6.8	4.8	8.4	62.8	22.7	91.5
Conc. 8	14	19.72	3.34	6.40	1.92	4.48	7.2	5.1	8.7	**63.1**	**23.1**	**91.8**
Oxide Rougher Tails							36.9	76.8	8.1	100.0	99.8	99.9
Heads (Calculated)				3.27	1.37	1.90	100.0	99.8	99.9			

Table 5.1.1G: Metallurgical Balance Sheet [Stainless Steel Re-Grinding (SSR)]

Stage	Time (min)	Fraction	Weight				Grade (%)			Recovery (%)		
			Wt (g)	Wt%	S.C. Wt%	O.C Wt%	TCu	ASCu	AICu	TCu	ASCu	AICu
Sulphide Flotation	0.3	Conc. 1	20.02	1.85	1.85	1.85	41.78	4.52	37.26	22.9	6.6	32.3
	0.6	Conc. 2	13.36	1.23	3.08	3.08	39.01	4.51	34.50	14.3	4.5	19.6
	1	Conc. 3	7.72	0.71	3.80	3.80	32.62	3.75	28.87	6.9	2.2	9.7
	4	Conc. 4	10.76	0.99	4.79	4.79	20.48	3.17	17.31	6.0	2.5	8.1
Combined Sulphide Concentrates			**51.86**	**4.79**			35.28	4.12	31.16	**50.1**	**15.8**	**69.7**
Oxide Flotation	0.5	Conc. 5	8.43	0.78	0.78	5.57	32.34	5.24	27.10	7.5	3.3	9.8
	1	Conc. 6	7.53	0.70	1.47	6.27	17.52	4.06	13.46	3.6	2.3	4.4
	2.5	Conc. 7	6.98	0.64	2.12	6.91	9.35	2.81	6.54	1.8	1.5	2.0
	4	Conc. 8	7.04	0.65	2.77	7.56	5.60	2.04	3.56	1.1	1.1	1.1
Combined Oxide Concentrates			**29.98**	**2.77**			16.99	3.63	13.36	**14.0**	**8.1**	**17.2**
Oxide Rougher Tails			1000.47	92.44	92.44	100.00	1.31	1.02	0.29	35.9	75.5	12.6
Head (Calculated)			1082.31	100.00			3.37	1.24	2.13	100.0	99.4	99.5
Head (Assay)							3.33	1.27	2.06			

Fractions	Cum. Time (min)	Water Recovery		Cumulative Grade (%)			Stage cumulative Recovery (%)			Overall Cumulative Recovery (%)		
		Wt (g)	Wt%	TCu	ASCu	AICu	TCu	ASCu	AICu	TCu	ASCu	AICu
Conc. 1	0.3	57.35	15.85	41.78	4.52	37.26	22.9	6.6	32.3	22.9	6.6	32.3
Conc. 2	1	50.99	14.09	40.67	4.52	36.16	37.2	11.1	51.9	37.2	11.1	51.9
Conc. 3	2	32.06	8.86	39.16	4.37	34.79	44.1	13.2	61.6	44.1	13.2	61.6
Conc. 4	6	68.51	18.93	35.28	4.12	31.16	50.1	15.8	69.7	50.1	15.8	69.7
Conc. 5	6.5	37.74	10.43	32.34	5.24	27.10	7.5	3.3	9.8	57.6	19.1	79.4
Conc. 6	7.5	39.76	10.99	25.35	4.68	20.66	11.1	5.6	14.2	61.2	21.3	83.8
Conc. 7	10	42.25	11.68	20.48	4.11	16.37	12.9	7.0	16.1	63.0	22.8	85.8
Conc. 8	14	33.17	9.17	16.99	3.63	13.36	14.0	8.1	17.2	**64.1**	**23.9**	**86.9**
Oxide Rougher Tails							35.9	75.5	12.6	100.0	99.4	99.5
Heads (Calculated)				3.37	1.24	2.13	100.0	99.4	99.5			

Table 5.1.1H: Metallurgical Balance Sheet [Stainless Steel Re-Grinding (SSR)]

Stage	Time (min)	Fraction	Weight				Grade (%)			Recovery (%)		
			Wt (g)	Wt%	S.C. Wt%	O.C Wt%	TCu	ASCu	AICu	TCu	ASCu	AICu
Sulphide Flotation	0.3	Conc. 1	22.94	2.12	2.12	2.12	41.50	4.59	36.91	26.0	7.4	37.3
	0.6	Conc. 2	12.84	1.19	3.31	3.31	40.50	4.46	36.04	14.2	4.1	20.1
	1	Conc. 3	7.39	0.68	3.99	3.99	33.05	3.95	29.10	6.7	2.1	9.5
	4	Conc. 4	11.17	1.03	5.03	5.03	19.96	3.15	16.81	6.1	2.5	8.3
Combined Sulphide Concentrates			**54.34**	**5.03**			35.69	4.18	31.51	**53.0**	**16.1**	**75.2**
Oxide Flotation	0.5	Conc. 5	10.69	0.99	0.99	6.01	25.94	4.42	21.52	7.6	3.4	10.1
	1	Conc. 6	8.83	0.82	1.81	6.83	12.53	3.31	9.22	3.0	2.1	3.6
	2.5	Conc. 7	8.04	0.74	2.55	7.58	6.91	2.31	4.60	1.5	1.3	1.6
	4	Conc. 8	6.90	0.64	3.19	8.21	4.72	1.83	2.89	0.9	0.9	0.9
Combined Oxide Concentrates			**34.46**	**3.19**			13.81	3.12	10.69	**13.0**	**7.8**	**16.2**
Oxide Rougher Tails			992.34	91.79	91.79	100.00	1.25	1.06	0.19	33.9	75.4	8.3
Head (Calculated)			1081.14	100.00			3.38	1.28	2.10	100.0	99.3	99.6
Head (Assay)							3.45	1.32	2.13			
Fractions	**Cum. Time (min)**	**Water Recovery**		**Cumulative Grade (%)**			**Stage cumulative Recovery (%)**			**Overall Cumulative Recovery (%)**		
		Wt (g)	**Wt%**	**TCu**	**ASCu**	**AICu**	**TCu**	**ASCu**	**AICu**	**TCu**	**ASCu**	**AICu**
Conc. 1	0.3	61.62	14.44	41.50	4.59	36.91	26.0	7.4	37.3	26.0	7.4	37.3
Conc. 2	1	47.68	11.18	41.14	4.54	36.60	40.3	11.5	57.4	40.3	11.5	57.4
Conc. 3	2	80.48	18.87	39.76	4.44	35.31	46.9	13.6	66.9	46.9	13.6	66.9
Conc. 4	6	49.68	11.65	35.69	4.18	31.51	53.0	16.1	75.2	53.0	16.1	75.2
Conc. 5	6.5	50.31	11.79	25.94	4.42	21.52	7.6	3.4	10.1	60.6	19.5	85.2
Conc. 6	7.5	51.02	11.96	19.87	3.92	15.96	10.6	5.5	13.7	63.7	21.6	88.8
Conc. 7	10	47.33	11.09	16.09	3.45	12.64	12.1	6.9	15.3	65.2	23.0	90.5
Conc. 8	14	38.48	9.02	13.81	3.12	10.69	13.0	7.8	16.2	**66.1**	**23.9**	**91.3**
Oxide Rougher Tails							33.9	75.4	8.3	100.0	99.3	99.6
Heads (Calculated)				3.38	1.28	2.10	100.0	99.3	99.6			

APPENDIX C2

Detailed Metallurgical Results of the Effect of Nitrogen Conditioning

Table 5.3.1-1A: Metallurgical Balance Sheet (NaHS - Nitrogen Conditioning)

Stage	Time (min)	Fraction	Weight		Grade (%)			Recovery (%)		
			Wt (g)	Wt%	TCu	ASCu	AICu	TCu	ASCu	AICu
Sulphide Rougher Flotation	0.3	Conc. 1	31.50	3.31	27.44	3.69	23.75	29.4	8.7	47.0
	0.6	Conc. 2	20.25	2.13	24.42	3.16	21.26	16.8	4.8	27.0
	1	Conc. 3	7.48	0.79	10.53	2.66	7.87	2.7	1.5	3.7
	4	Conc. 4	11.38	1.20	11.98	2.88	9.10	4.6	2.4	6.5
Combined Sulphide Concentrates			**70.61**	**7.42**	22.29	3.30	18.99	**53.6**	**17.3**	**84.2**
Oxide Rougher Flotation	0.5	Conc. 5	14.70	1.55	24.43	17.83	6.60	12.2	19.5	6.1
	1	Conc. 6	10.35	1.09	17.51	17.51	0.00	6.2	13.5	0.0
	2.5	Conc. 7	14.40	1.51	10.74	9.99	0.75	5.3	10.7	0.7
	4	Conc. 8	6.17	0.65	7.85	7.6	0.25	1.7	3.5	0.1
Combined Oxide Concentrates			**45.62**	**4.80**	16.30	13.90	2.40	**25.3**	**47.2**	**6.9**
Oxide Rougher Tails			835.00	87.78	0.74	0.57	0.17	21.1	35.4	8.9
Head (calculated)			951.23		3.09	1.41	1.67	100.0	100.0	100.0
Head (assay)					3.24	1.24	2.00			
Concentrate Fractions	**Cum. Flot. Time (min)**	**Cumulative Grade (%)**			**Stage Cumulative Recovery (%)**			**Overall Cumulative Recovery**		
		TCu	**ASCu**	**AICu**	**TCu**	**ASCu**	**AICu**	**TCu**	**ASCu**	**AICu**
Conc. 1	0.3	27.44	3.69	23.75	29.4	8.7	47.0	29.4	8.7	47.0
Conc. 2	1	26.26	3.48	22.78	46.3	13.4	74.0	46.3	13.4	74.0
Conc. 3	2	24.27	3.38	20.89	49.0	14.9	77.7	49.0	14.9	77.7
Conc. 4	6	22.29	3.30	18.99	53.6	17.3	84.2	53.6	17.3	84.2
Conc. 5	6.5	24.43	17.83	6.60	12.2	19.5	6.1	65.9	36.9	90.3
Conc. 6	7.5	21.57	17.70	3.87	18.4	33.0	6.1	72.0	50.4	90.3
Conc. 7	10	17.62	14.88	2.73	23.7	43.7	6.8	77.3	61.1	91.0
Conc. 8	14	16.30	13.90	2.40	25.3	47.2	6.9	**78.9**	**64.6**	**91.1**
Oxide Rougher Tails					21.1	35.4	8.9	100.0	100.0	100.0
Heads (Calculated)		3.09	1.41	1.67	100.0	100.0	100.0			

Table 5.3.1-1B: Metallurgical Balance Sheet (NaHS - Nitrogen Conditioning)

Stage	Time (min)	Fraction	Weight		Grade (%)			Recovery (%)		
			Wt (g)	Wt%	TCu	ASCu	AlCu	TCu	ASCu	AlCu
Sulphide Rougher Flotation	0.3	Conc. 1	32.53	3.40	28.32	2.71	25.61	29.4	6.8	45.1
	0.6	Conc. 2	19.33	2.02	22.88	3.20	19.68	14.1	4.8	20.6
	1	Conc. 3	9.63	1.01	17.94	3.54	14.40	5.5	2.6	7.5
	4	Conc. 4	12.36	1.29	9.45	2.70	6.75	3.7	2.6	4.5
Combined Sulphide Concentrates			**73.85**	**7.71**	22.38	2.94	19.44	**52.7**	**16.9**	**77.7**
Oxide Rougher Flotation	0.5	Conc. 5	14.73	1.54	18.59	18.43	0.16	8.7	21.1	0.1
	1	Conc. 6	17.43	1.82	14.99	13.96	1.03	8.3	18.9	1.0
	2.5	Conc. 7	15.55	1.62	11.65	9.84	1.81	5.8	11.9	1.5
	4	Conc. 8	4.79	0.50	7.11	5.98	1.13	1.1	2.2	0.3
Combined Oxide Concentrates			**52.50**	**5.48**	14.29	13.27	1.03	**23.9**	**54.1**	**2.9**
Oxide Rougher Tails			831.00	86.80	0.88	0.45	0.43	23.3	29.0	19.3
Head (calculated)			957.35		3.27	1.35	1.93	100.0	100.0	100.0
Head (assay)					3.24	1.35	1.89			
Concentrate Fractions	Cum. Flot. Time (min)	Cumulative Grade (%)			Stage Cumulative Recovery (%)			Overall Cumulative Recovery		
		TCu	ASCu	AlCu	TCu	ASCu	AlCu	TCu	ASCu	AlCu
Conc. 1	0.3	28.32	2.71	25.61	29.4	6.8	45.1	29.4	6.8	45.1
Conc. 2	1	26.29	2.89	23.40	43.5	11.6	65.7	43.5	11.6	65.7
Conc. 3	2	24.98	2.99	21.99	49.0	14.3	73.2	49.0	14.3	73.2
Conc. 4	6	22.38	2.94	19.44	52.7	16.9	77.7	52.7	16.9	77.7
Conc. 5	6.5	18.59	18.43	0.16	8.7	21.1	0.1	61.5	38.0	77.9
Conc. 6	7.5	16.64	16.01	0.63	17.1	40.0	1.1	69.8	56.9	78.8
Conc. 7	10	15.01	14.00	1.02	22.8	51.9	2.6	75.6	68.7	80.4
Conc. 8	14	14.29	13.27	1.03	23.9	54.1	2.9	**76.7**	**71.0**	**80.7**
Oxide Rougher Tails					23.3	29.0	19.3	100.0	100.0	100.0
Heads (Calculated)		3.27	1.35	1.93	100.0	100.0	100.0			

Table 5.3.1-1C: Metallurgical Balance Sheet (NaHS - Nitrogen Conditioning)

Stage	Time (min)	Fraction	Weight		Grade (%)			Recovery (%)		
			Wt (g)	Wt%	TCu	ASCu	AICu	TCu	ASCu	AICu
Sulphide Rougher Flotation	0.3	Conc. 1	35.11	3.72	30.6	3.20	27.40	33.2	8.6	49.9
	0.6	Conc. 2	21.12	2.24	17.44	2.68	14.76	11.4	4.3	16.2
	1	Conc. 3	9.72	1.03	12.74	2.70	10.04	3.8	2.0	5.1
	4	Conc. 4	11.19	1.19	10.45	2.13	8.32	3.6	1.8	4.8
Combined Sulphide Concentrates			**77.14**	**8.18**	21.82	2.84	18.98	**52.1**	**16.8**	**76.0**
Oxide Rougher Flotation	0.5	Conc. 5	12.90	1.37	22.71	22.65	0.06	9.1	22.4	0.0
	1	Conc. 6	9.46	1.00	19.86	19.71	0.15	5.8	14.3	0.1
	2.5	Conc. 7	11.33	1.20	15.24	12.83	2.41	5.3	11.1	1.4
	4	Conc. 8	10.30	1.09	8.04	5.88	2.16	2.6	4.6	1.2
Combined Oxide Concentrates			**43.99**	**4.66**	16.74	15.56	1.18	**22.8**	**52.4**	**2.7**
Oxide Rougher Tails			822.30	87.16	0.99	0.49	0.50	25.2	30.8	21.3
Head (calculated)			943.43		3.43	1.38	2.04	100.0	100.0	100.0
Head (assay)					3.41	1.35	2.06			

Concentrate Fractions	Cum. Flot. Time (min)	Cumulative Grade (%)			Stage Cumulative Recovery (%)			Overall Cumulative Recovery		
		TCu	ASCu	AICu	TCu	ASCu	AICu	TCu	ASCu	AICu
Conc. 1	0.3	30.6	3.20	27.40	33.2	8.6	49.9	33.2	8.6	49.9
Conc. 2	1	25.66	3.00	22.65	44.2	12.9	66.1	44.2	12.9	66.1
Conc. 3	2	23.75	2.96	20.79	48.4	14.9	71.2	48.4	14.9	71.2
Conc. 4	6	21.82	2.84	18.98	52.1	16.8	76.0	52.1	16.8	76.0
Conc. 5	6.5	22.71	22.65	0.06	9.1	22.4	0.0	61.1	39.1	76.0
Conc. 6	7.5	21.50	21.41	0.10	14.9	36.6	0.1	66.9	53.4	76.1
Conc. 7	10	19.40	18.52	0.88	20.2	47.8	1.5	72.3	64.5	77.5
Conc. 8	14	16.74	15.56	1.18	22.8	52.4	2.7	**74.8**	**69.2**	**78.7**
Oxide Rougher Tails					25.2	30.8	21.3	100.0	100.0	100.0
Heads (Calculated)		3.43	1.38	2.04	100.0	100.0	100.0			

Table 5.3.1-1D: Metallurgical Balance Sheet (NaHS - Nitrogen Conditioning)

Stage	Time (min)	Fraction	Weight		Grade (%)			Recovery (%)		
			Wt (g)	Wt%	TCu	ASCu	AICu	TCu	ASCu	AICu
Sulphide Rougher Flotation	0.3	Conc. 1	37.12	3.87	25.28	3.59	21.69	27.9	10.5	38.3
	0.6	Conc. 2	18.78	1.96	20.77	3.43	17.34	11.6	5.1	15.5
	1	Conc. 3	8.55	0.89	18.40	3.43	14.97	4.7	2.3	6.1
	4	Conc. 4	13.41	1.40	10.06	2.98	7.08	4.0	3.2	4.5
Combined Sulphide Concentrates			**77.86**	**8.12**	20.82	3.43	17.39	**48.1**	**21.1**	**64.4**
Oxide Rougher Flotation	0.5	Conc. 5	12.87	1.34	32.43	26.44	5.99	12.4	26.9	3.7
	1	Conc. 6	7.21	0.75	27.25	25.01	2.24	5.8	14.3	0.8
	2.5	Conc. 7	13.09	1.37	12.51	10.84	1.67	4.9	11.2	1.0
	4	Conc. 8	9.48	0.99	8.51	5.26	3.25	2.4	3.9	1.5
Combined Oxide Concentrates			**42.65**	**4.45**	20.12	16.70	3.42	**25.5**	**56.3**	**6.9**
Oxide Rougher Tails			838.30	87.43	1.06	0.34	0.72	26.4	22.5	28.7
Head (calculated)			958.81		3.51	1.32	2.19	100.0	100.0	100.0
Head (assay)					3.41	1.46	1.95			
Concentrate Fractions	**Cum. Flot. Time (min)**	**Cumulative Grade (%)**			**Stage Cumulative Recovery (%)**			**Overall Cumulative Recovery**		
		TCu	**ASCu**	**AICu**	**TCu**	**ASCu**	**AICu**	**TCu**	**ASCu**	**AICu**
Conc. 1	0.3	25.28	3.59	21.69	27.9	10.5	38.3	27.9	10.5	38.3
Conc. 2	1	23.76	3.54	20.23	39.4	15.6	53.8	39.4	15.6	53.8
Conc. 3	2	23.05	3.52	19.53	44.1	18.0	59.9	44.1	18.0	59.9
Conc. 4	6	20.82	3.43	17.39	48.1	21.1	64.4	48.1	21.1	64.4
Conc. 5	6.5	32.43	26.44	5.99	12.4	26.9	3.7	60.5	48.0	68.0
Conc. 6	7.5	30.57	25.93	4.64	18.2	41.2	4.4	66.4	62.3	68.8
Conc. 7	10	23.44	19.97	3.47	23.1	52.4	5.5	71.2	73.5	69.8
Conc. 8	14	20.12	16.70	3.42	25.5	56.3	6.9	**73.6**	**77.5**	**71.3**
Oxide Rougher Tails					26.4	22.5	28.7	100.0	100.0	100.0
Heads (Calculated)		3.51	1.32	2.19	100.0	100.0	100.0			

Table 5.3.1-2A: Metallurgical Balance Sheet (NaHS - Air Conditioning)

Stage	Time (min)	Fraction	Weight		Grade (%)			Recovery (%)		
			Wt (g)	Wt%	TCu	ASCu	AICu	TCu	ASCu	AICu
Sulphide Rougher Flotation	0.3	Conc. 1	29.93	3.23	41.89	3.51	38.38	37.6	8.8	53.5
	0.6	Conc. 2	16.62	1.80	25.00	3.49	21.51	12.4	4.9	16.6
	1	Conc. 3	7.24	0.78	15.92	2.92	13.00	3.5	1.8	4.4
	4	Conc. 4	9.66	1.04	11.23	2.48	8.75	3.2	2.0	3.9
Combined Sulphide Concentrates			**63.45**	**6.85**	29.83	3.28	26.55	**56.7**	**17.5**	**78.4**
Oxide Rougher Flotation	0.5	Conc. 5	5.5	0.59	31.53	31.08	0.45	5.2	14.4	0.1
	1	Conc. 6	8.54	0.92	30.37	28.15	2.22	7.8	20.2	0.9
	2.5	Conc. 7	8.13	0.88	20.67	16.51	4.16	5.0	11.3	1.6
	4	Conc. 8	6.72	0.73	10.28	7.75	2.53	2.1	4.4	0.8
Combined Oxide Concentrates			**28.89**	**3.12**	23.19	20.69	2.50	**20.1**	**50.3**	**3.4**
Oxide Rougher Tails			833.50	90.03	0.93	0.46	0.47	23.2	32.2	18.2
Head (calculated)			925.84		3.61	1.28	2.32	100.0	100.0	100.0
Head (assay)					3.58	1.16	2.42			
Concentrate Fractions	Cum. Flot. Time (min)	Cumulative Grade (%)			Stage Cumulative Recovery (%)			Overall Cumulative Recovery (%)		
		TCu	ASCu	AICu	TCu	ASCu	AICu	TCu	ASCu	AICu
Conc. 1	0.3	41.89	3.51	38.38	37.6	8.8	53.2	37.6	8.8	53.2
Conc. 2	1	35.86	3.50	32.36	50.0	13.7	70.1	50.0	13.7	70.1
Conc. 3	2	33.18	3.42	29.75	53.5	15.5	74.5	53.5	15.5	74.5
Conc. 4	6	29.83	3.28	26.55	56.7	17.5	78.4	56.7	17.5	78.4
Conc. 5	6.5	31.53	31.08	0.45	5.2	14.4	0.1	61.9	31.9	78.5
Conc. 6	7.5	30.82	29.30	1.53	13.0	34.6	1.0	69.7	52.1	79.4
Conc. 7	10	27.10	24.61	2.49	18.0	45.9	2.6	74.7	63.4	81.0
Conc. 8	14	23.19	20.69	2.50	20.1	50.3	3.4	**76.8**	**67.8**	**81.8**
Oxide Rougher Tails					23.2	32.2	18.2	100.0	100.0	100.0
Heads (Calculated)		3.61	1.28	2.32	100.0	100.0	100.0			

Table 5.3.1-2B: Metallurgical Balance Sheet (NaHS - Air Conditioning)

Stage	Time (min)	Fraction	Weight		Grade (%)			Recovery (%)		
			Wt (g)	Wt%	TCu	ASCu	AICu	TCu	ASCu	AICu
Sulphide Rougher Flotation	0.3	Conc. 1	31.61	3.36	38.37	334	35.03	36.9	8.6	53.8
	0.6	Conc. 2	16.85	1.79	25.22	3.26	21.96	12.9	4.5	18.0
	1	Conc. 3	6.36	0.68	19.92	3.12	16.8	3.9	1.6	5.2
	4	Conc. 4	11.30	1.20	11.67	2.32	9.35	4.0	2.1	5.1
Combined Sulphide Concentrates			**66.12**	**7.03**	28.68	3.12	25.56	**57.7**	**16.8**	**82.2**
Oxide Rougher Flotation	0.5	Conc. 5	12.41	1.32	13.99	13.16	0.83	5.3	13.3	0.5
	1	Conc. 6	5.43	0.58	24.43	24.39	0.04	4.0	10.8	0.0
	2.5	Conc. 7	7.62	0.81	23.28	20.30	2.98	5.4	12.6	1.1
	4	Conc. 8	7.38	0.78	13.31	9.19	4.12	3.0	5.5	1.5
Combined Oxide Concentrates			**32.84**	**3.49**	17.72	15.78	1.94	**17.7**	**42.2**	**3.1**
Oxide Rougher Tails			841.20	89.47	0.96	0.60	0.36	24.6	41.0	14.7
Head (calculated)			940.18		3.49	1.31	2.19	100.0	100.0	100.0
Head (assay)					3.93	1.26	2.67			
Concentrate Fractions	Cum. Flot. Time (min)	Cumulative Grade (%)			Stage Cumulative Recovery (%)			Overall Cumulative Recovery		
		TCu	ASCu	AICu	TCu	ASCu	AICu	TCu	ASCu	AICu
Conc. 1	0.3	38.37	3.34	35.03	36.9	8.6	53.8	36.9	8.6	53.8
Conc. 2	1		3.31	30.49	49.8	13.1	71.8	49.8	13.1	71.8
Conc. 3	2		3.29	28.90	53.7	14.7	77.0	53.7	14.7	77.0
Conc. 4	6		3.12	25.56	57.7	16.8	82.2	57.7	16.8	82.2
Conc. 5	6.5		13.16	0.83	5.3	13.3	0.5	63.0	30.1	82.7
Conc. 6	7.5		16.58	0.59	9.3	24.1	0.5	67.0	40.9	82.7
Conc. 7	10		17.69	1.30	14.7	36.6	1.6	72.4	53.4	83.8
Conc. 8	14	17.72	15.78	1.94	17.7	42.2	3.1	**75.4**	**59.0**	**85.3**
Oxide Rougher Tails					24.6	41.0	14.7	100.0	100.0	100.0
Heads (Calculated)		3.49	1.31	2.19	100.0	100.0	100.0			

Table 5.3.1-2C: Metallurgical Balance Sheet (NaHS - Air Conditioning)

Stage	Time (min)	Fraction	Weight		Grade (%)			Recovery (%)		
			Wt (g)	Wt%	TCu	ASCu	AICu	TCu	ASCu	AICu
Sulphide Rougher Flotation	0.3	Conc. 1	16.39	1.73	42.65	4.50	38.15	22.0	5.9	32.3
	0.6	Conc. 2	14.44	1.52	31.03	4.25	26.78	14.1	4.9	20.0
	1	Conc. 3	14.87	1.57	22.35	4.45	17.90	10.4	5.3	13.7
	4	Conc. 4	11.24	1.18	14.79	3.06	11.73	5.2	2.8	6.8
Combined Sulphide Concentrates			**56.94**	**6.00**	28.90	4.14	24.76	**51.7**	**19.0**	**72.8**
Oxide Rougher Flotation	0.5	Conc. 5	7.57	0.80	33.63	27.77	5.86	8.0	16.9	2.3
	1	Conc. 6	9.93	1.05	24.74	22.96	1.78	7.7	18.3	0.9
	2.5	Conc. 7	10.62	1.12	13.80	13.22	0.58	4.6	11.3	0.3
	4	Conc. 8	7.01	0.74	8.09	3.65	4.44	1.8	2.1	1.6
Combined Oxide Concentrates			**35.13**	**3.70**	20.03	17.20	2.83	**22.1**	**48.6**	**5.1**
Oxide Rougher Tails			857.50	90.30	0.97	0.47	0.50	26.1	32.4	22.1
Head (calculated)			949.57		3.35	1.31	2.04	100.0	100.0	100.0
Head (assay)					3.52	1.42	2.10			
Concentrate Fractions	**Cum. Flot. Time (min)**	**Cumulative Grade (%)**			**Stage Cumulative Recovery (%)**			**Overall Cumulative Recovery (%)**		
		TCu	**ASCu**	**AICu**	**TCu**	**ASCu**	**AICu**	**TCu**	**ASCu**	**AICu**
Conc. 1	0.3	42.65	4.50	38.15	22.0	5.9	32.3	22.0	5.9	32.3
Conc. 2	1	37.21	4.38	32.82	36.1	10.9	52.2	36.1	10.9	52.2
Conc. 3	2	32.37	4.40	27.97	46.5	16.2	65.9	46.5	16.2	65.9
Conc. 4	6	28.90	4.14	24.76	51.7	19.0	72.8	51.7	19.0	72.8
Conc. 5	6.5	33.63	27.77	5.86	8.0	16.9	2.3	59.7	35.9	75.0
Conc. 6	7.5	28.59	25.04	3.54	15.7	35.3	3.2	67.5	54.2	76.0
Conc. 7	10	23.00	20.58	2.43	20.3	46.6	3.5	72.1	65.5	76.3
Conc. 8	14	20.03	17.20	2.83	22.1	48.6	5.1	**73.9**	**67.6**	**77.9**
Oxide Rougher Tails					26.1	32.4	22.1	100.0	100.0	100.0
Heads (Calculated)		3.35	1.31	2.04	100.0	100.0	100.0			

Table 5.3.1-2D: Metallurgical Balance Sheet (NaHS - Air Conditioning)

Stage	Time (min)	Fraction	Weight		Grade (%)			Recovery (%)		
			Wt (g)	Wt%	TCu	ASCu	AICu	TCu	ASCu	AICu
Sulphide Rougher Flotation	0.3	Conc. 1	22.60	2.40	40.38	4.18	36.20	27.6	7.0	41.8
	0.6	Conc. 2	15.24	1.62	34.79	4.09	30.70	16.0	4.6	23.9
	1	Conc. 3	4.94	0.52	23.42	3.59	19.83	3.5	1.3	5.0
	4	Conc. 4	9.18	0.98	13.75	2.95	10.80	3.8	2.0	5.1
Combined Sulphide Concentrates			**51.96**	**5.52**	32.42	3.88	28.54	**50.9**	**14.9**	**75.9**
Oxide Rougher Flotation	0.5	Conc. 5	4.58	0.49	28.43	28.11	0.32	3.9	9.5	0.1
	1	Conc. 6	6.51	0.69	33.46	29.17	4.29	6.6	14.0	1.4
	2.5	Conc. 7	10.97	1.17	17.71	17.60	0.11	5.9	14.2	0.1
	4	Conc. 8	10.02	1.06	9.80	7.68	2.12	3.0	5.7	1.1
Combined Oxide Concentrates			**32.08**	**3.41**	19.97	18.35	1.62	**19.3**	**43.4**	**2.7**
Oxide Rougher Tails			857.10	91.07	1.15	0.66	0.49	29.8	41.7	21.5
Head (calculated)			941.14		3.52	1.44	2.08	100.0	100.0	100.0
Head (assay)					3.41	1.35	2.06			
Concentrate Fractions	**Cum. Flot. Time (min)**	**Cumulative Grade (%)**			**Stage Cumulative Recovery (%)**			**Overall Cumulative Recovery (%)**		
		TCu	**ASCu**	**AICu**	**TCu**	**ASCu**	**AICu**	**TCu**	**ASCu**	**AICu**
Conc. 1	0.3	40.38	4.18	36.20	27.6	7.0	41.8	27.6	7.0	41.8
Conc. 2	1	38.13	4.14	33.98	43.6	11.6	65.8	43.6	11.6	65.8
Conc. 3	2	36.43	4.08	32.35	47.1	12.9	70.8	47.1	12.9	70.8
Conc. 4	6	32.42	3.88	28.54	50.9	14.9	75.9	50.9	14.9	75.9
Conc. 5	6.5	28.43	28.11	0.32	3.9	9.5	0.1	54.8	24.4	75.9
Conc. 6	7.5	31.38	28.73	2.65	10.5	23.5	1.5	61.4	38.4	77.4
Conc. 7	10	24.58	23.20	1.39	16.4	37.7	1.6	67.3	52.6	77.4
Conc. 8	14	19.97	18.35	1.62	19.3	43.4	2.7	**70.2**	**58.3**	**78.5**
Oxide Rougher Tails					29.8	41.7	21.5	100.0	100.0	100.0
Heads (Calculated)		3.52	1.44	2.08	100.0	100.0	100.0			

APPENDIX D1

Detailed Pulp Chemistry Measurements of the Effect of Milling Media

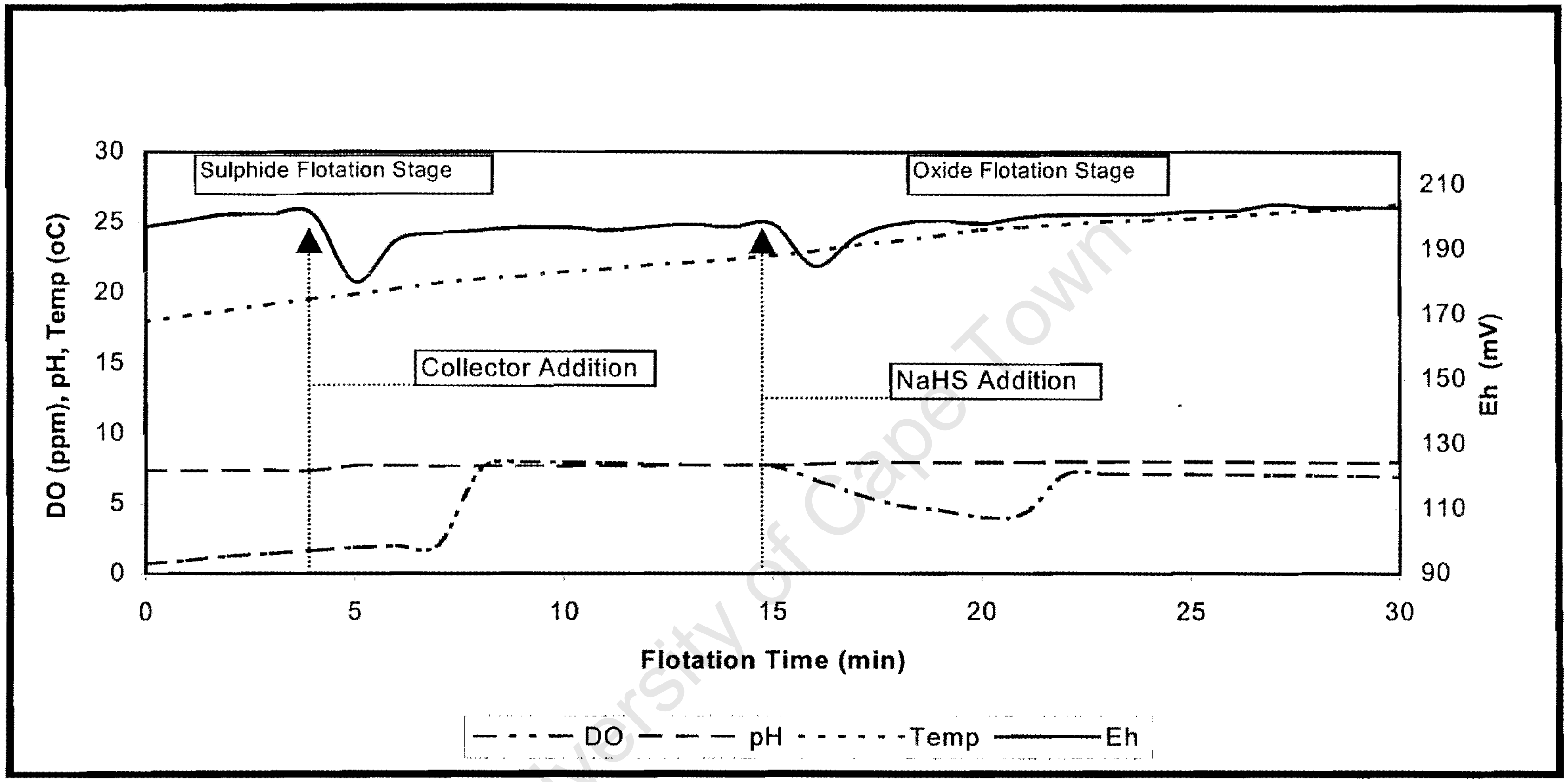


Figure 5.1.2-1 (a): Dissolved Oxygen (DO), pH, Temperature (Temp.) and Pulp Potential (Eh) Profiles obtained during Flotation after Mild Steel Grinding (MS)

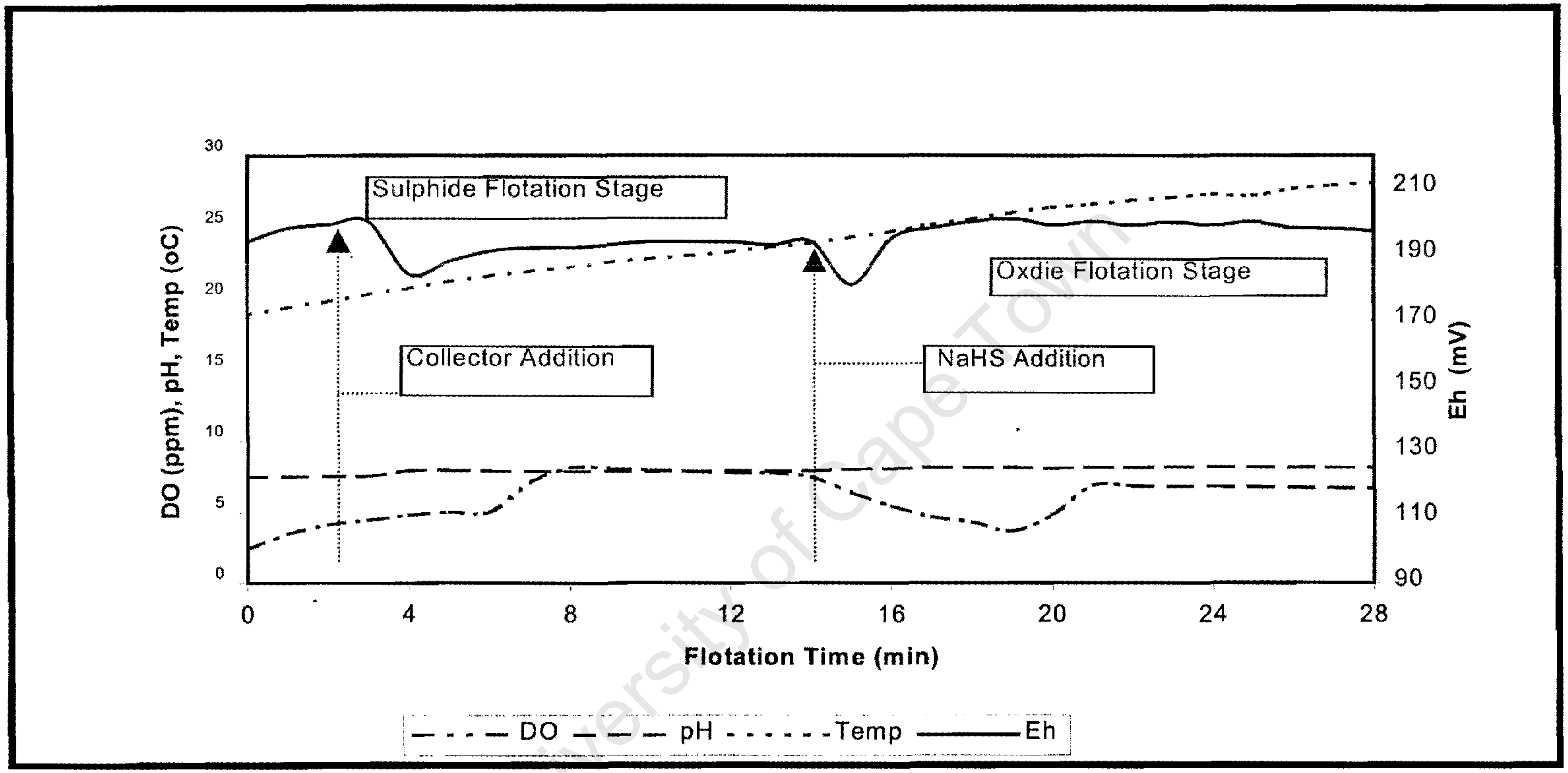


Figure 5.1.2-1 (b): Dissolved Oxygen levels (DO), pH, Temperature (Temp.) and Pulp Potential (Eh) Profiles obtained during Flotation after Mild Steel Grinding (MS)

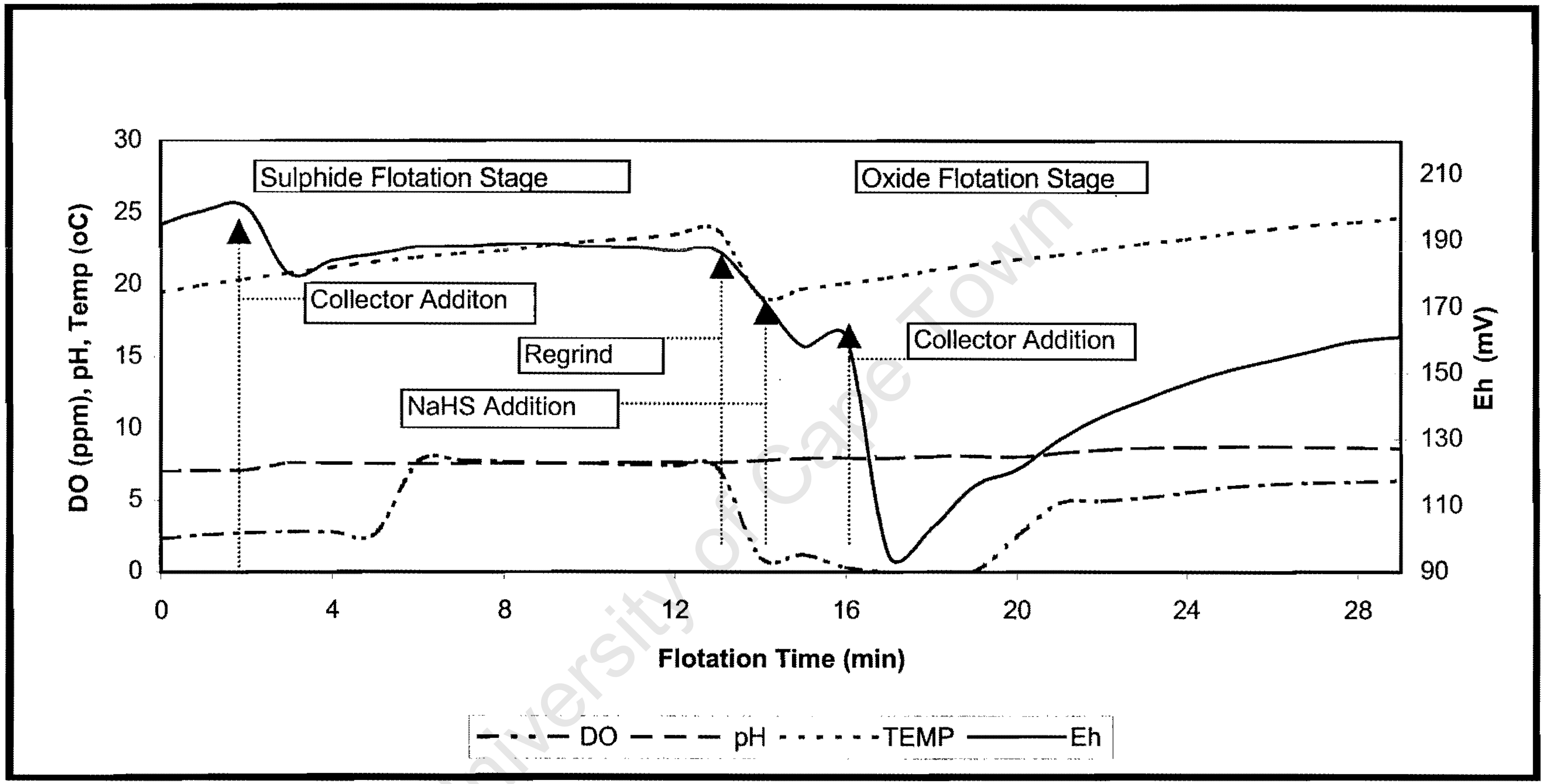


Figure 5.1.2-2 (a): Dissolved Oxygen levels (DO), pH, Temperature (Temp.) and Pulp Potential (Eh) Profiles obtained during Flotation after Mild Steel Grinding (MS) and Re-grinding (MSR)

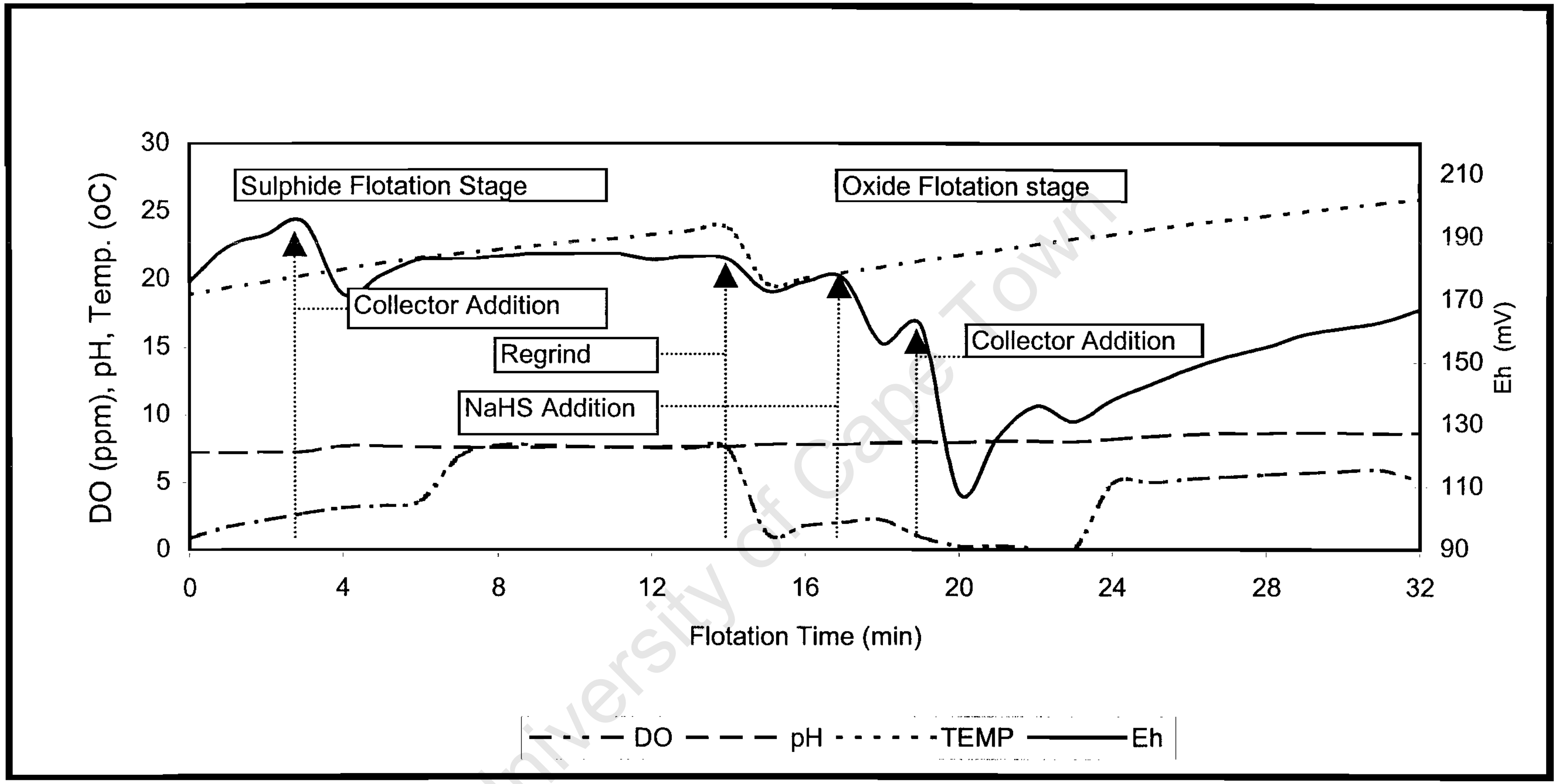


Figure 5.1.2-2 (b): Dissolved Oxygen levels (DO), pH, Temperature (Temp.) and Pulp Potential (Eh) Profiles obtained during Flotation after Mild Steel Grinding (MS) and Re-grinding (MSR)

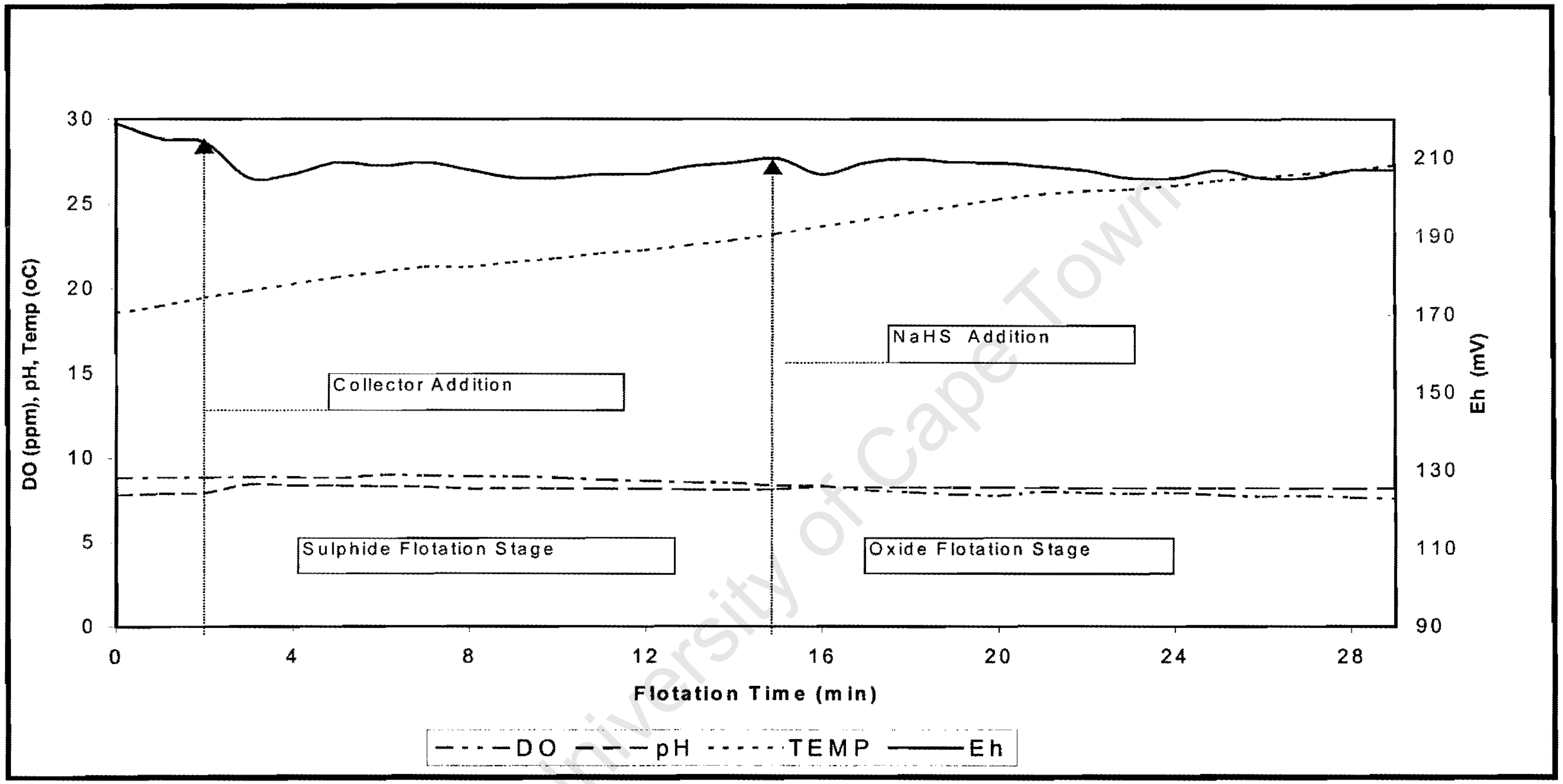


Figure 5.1.2-3 (a): Dissolved Oxygen levels (DO), pH, Temperature (Temp.) and Pulp Potential (Eh) Profiles obtained during Flotation after Stainless Steel Grinding (SS)

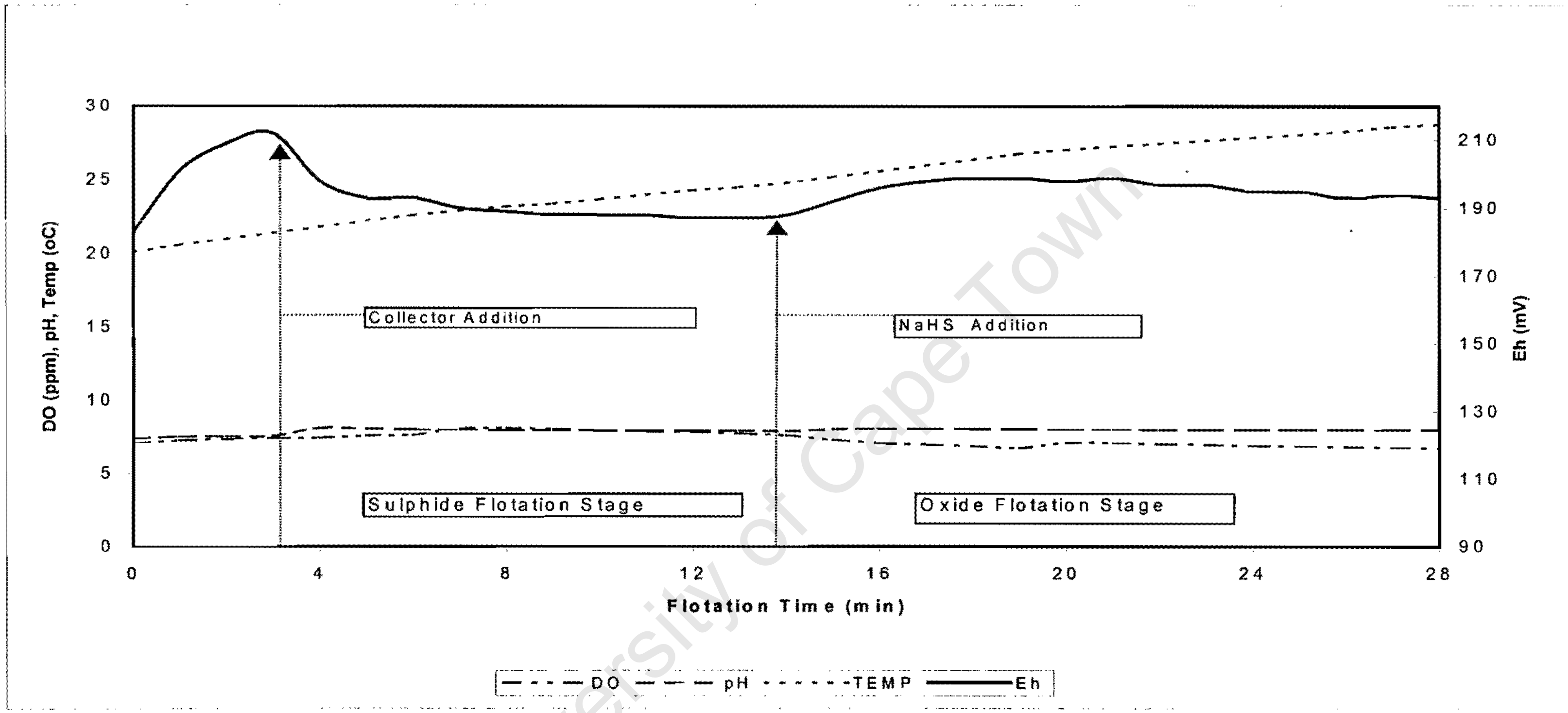


Figure 5.1.2-3 (b): Dissolved Oxygen levels (DO), pH, Temperature (Temp.) and Pulp Potential (Eh) Profiles obtained during Flotation after Stainless Steel Grinding (SS)

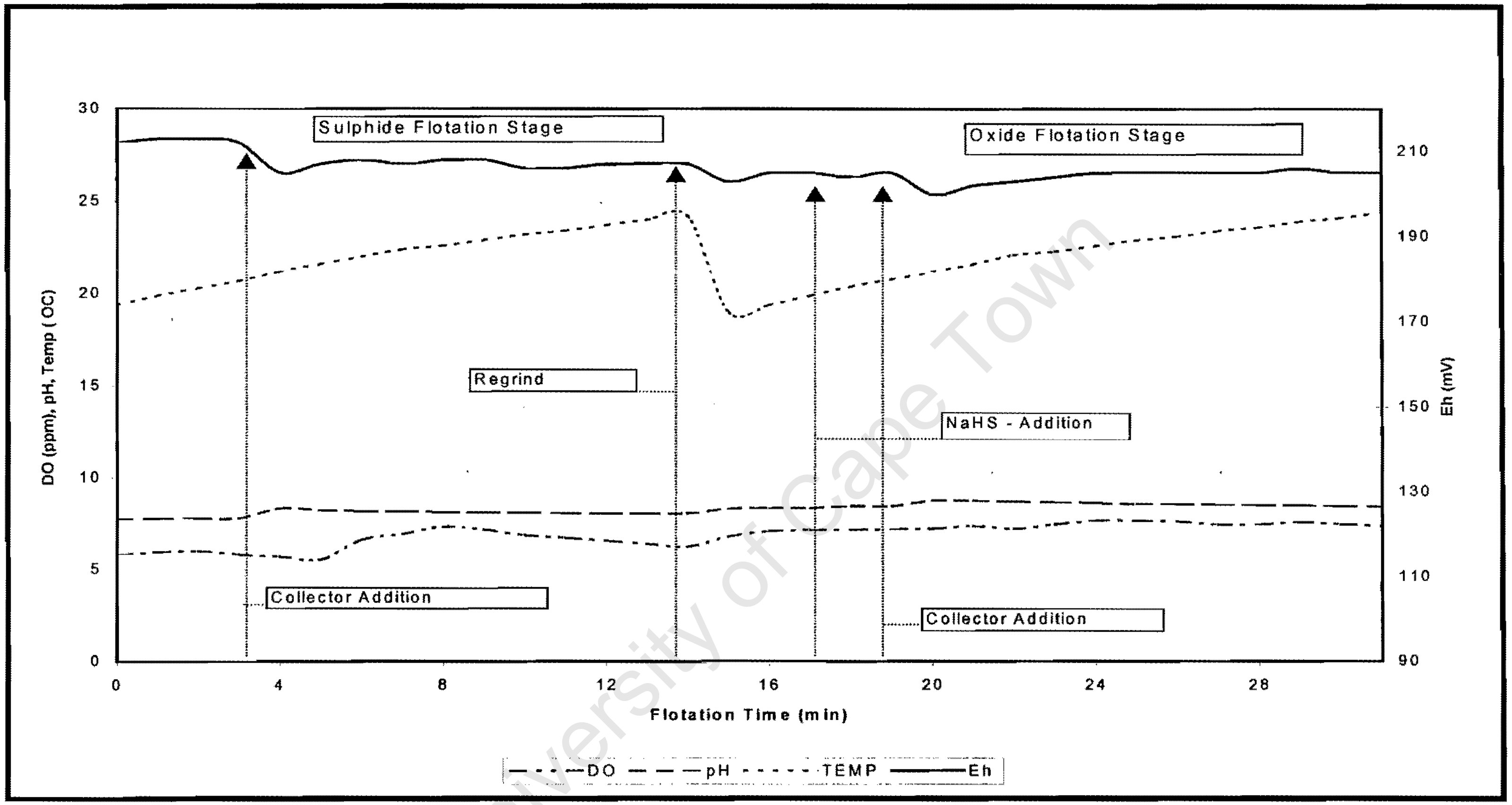


Figure 5.1.2-4 (a): Dissolved Oxygen levels (DO), pH, Temperature (Temp.) and Pulp Potential (Eh) Profiles obtained during Flotation after Stainless Steel Grinding (SS) and Re-grinding (SSR)

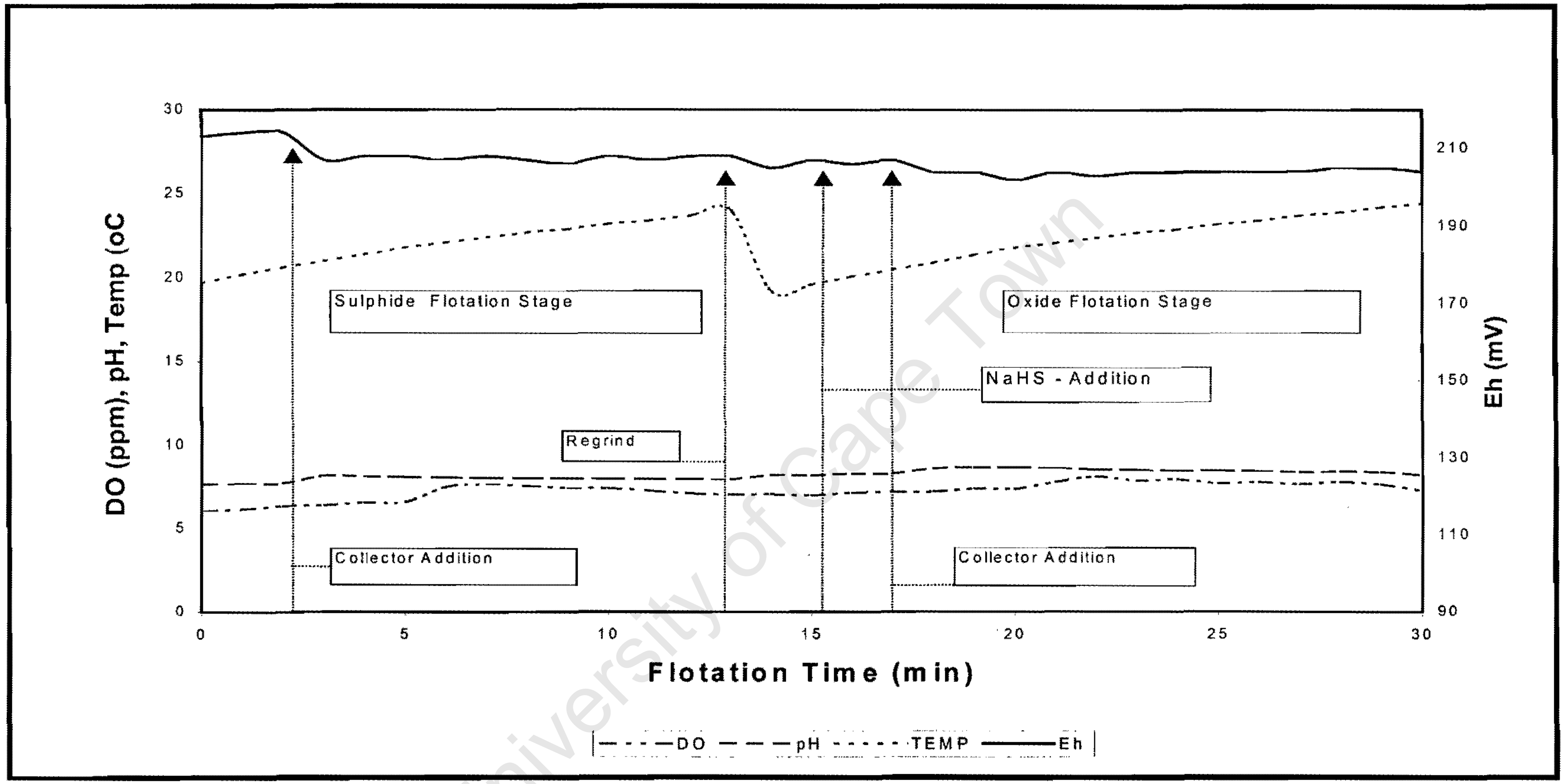


Figure 5.1.2-4 (b): Dissolved Oxygen levels (DO), pH, Temperature (Temp.) and Pulp Potential (Eh) Profiles obtained during Flotation after Stainless Steel Grinding (SS) and Regrinding (SSR)

APPENDIX D2

Detailed Pulp Chemistry Measurements At Nchanga Concentrator Plant

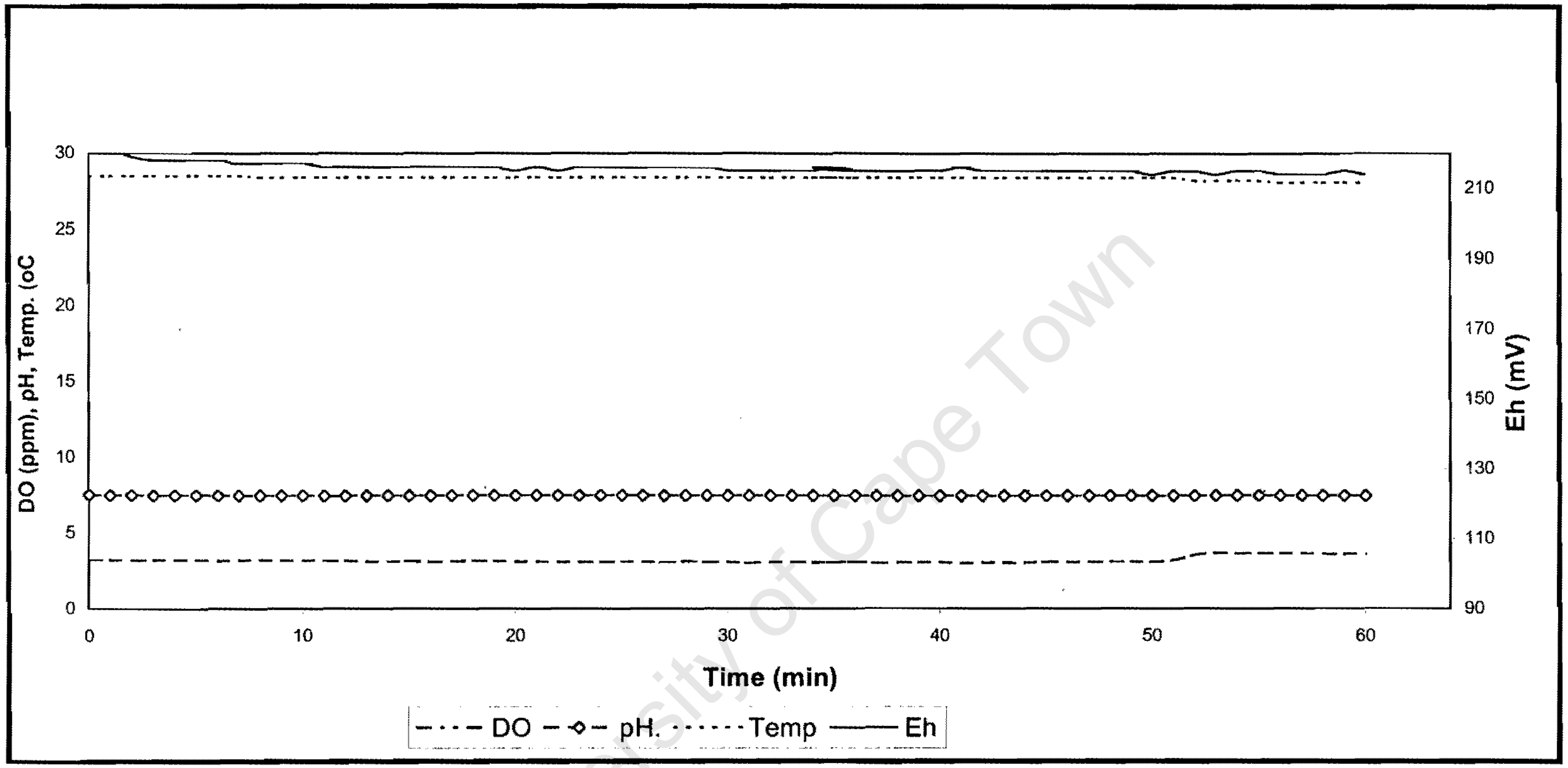


Figure 5.2.2.1: Dissolved Oxygen levels (DO), pH, Temperature (Temp.) and Pulp Potential (Eh) Profiles obtained in the Cyclone Overflow (COF)

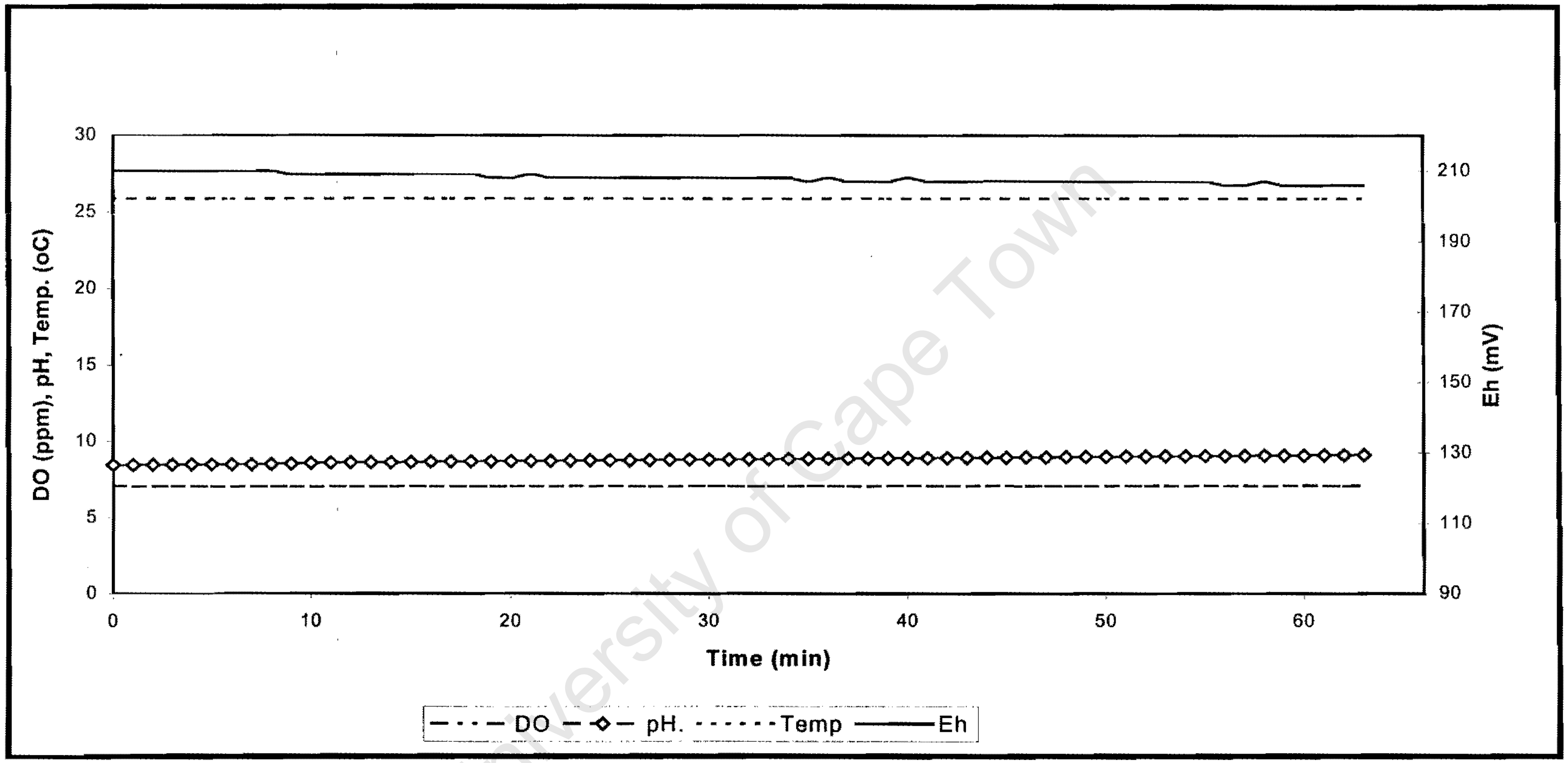


Figure 5.2.2.2: Dissolved Oxygen levels (DO), pH, Temperature (Temp.) and Pulp Potential (Eh) Profiles obtained in the Sulphide Rougher Banks (SRB)

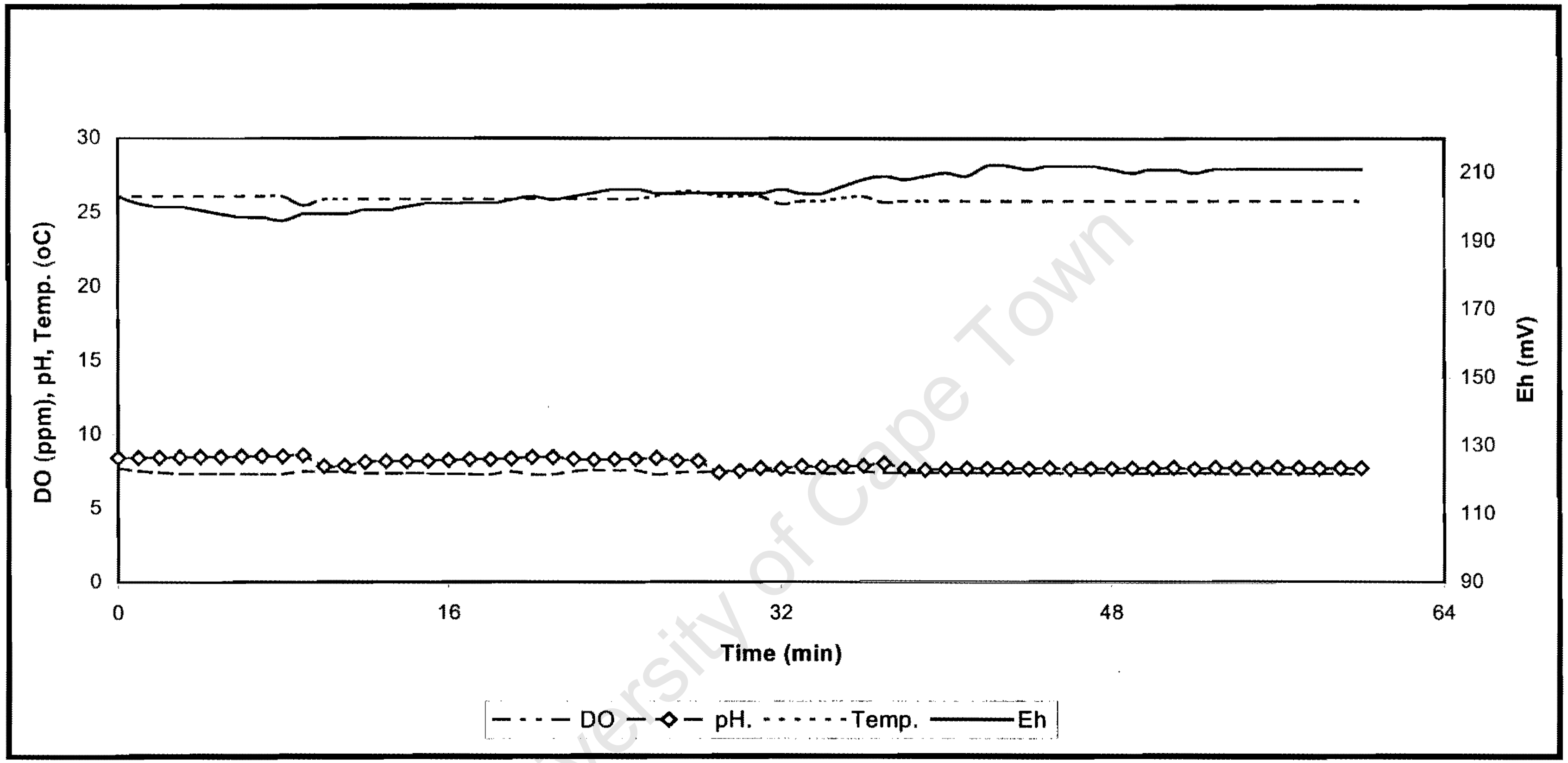


Figure 5.2.2.3: Dissolved Oxygen levels (DO), pH, Temperature (Temp.) and Pulp Potential (Eh) Profiles obtained in the Oxide Rougher Banks (ORB)

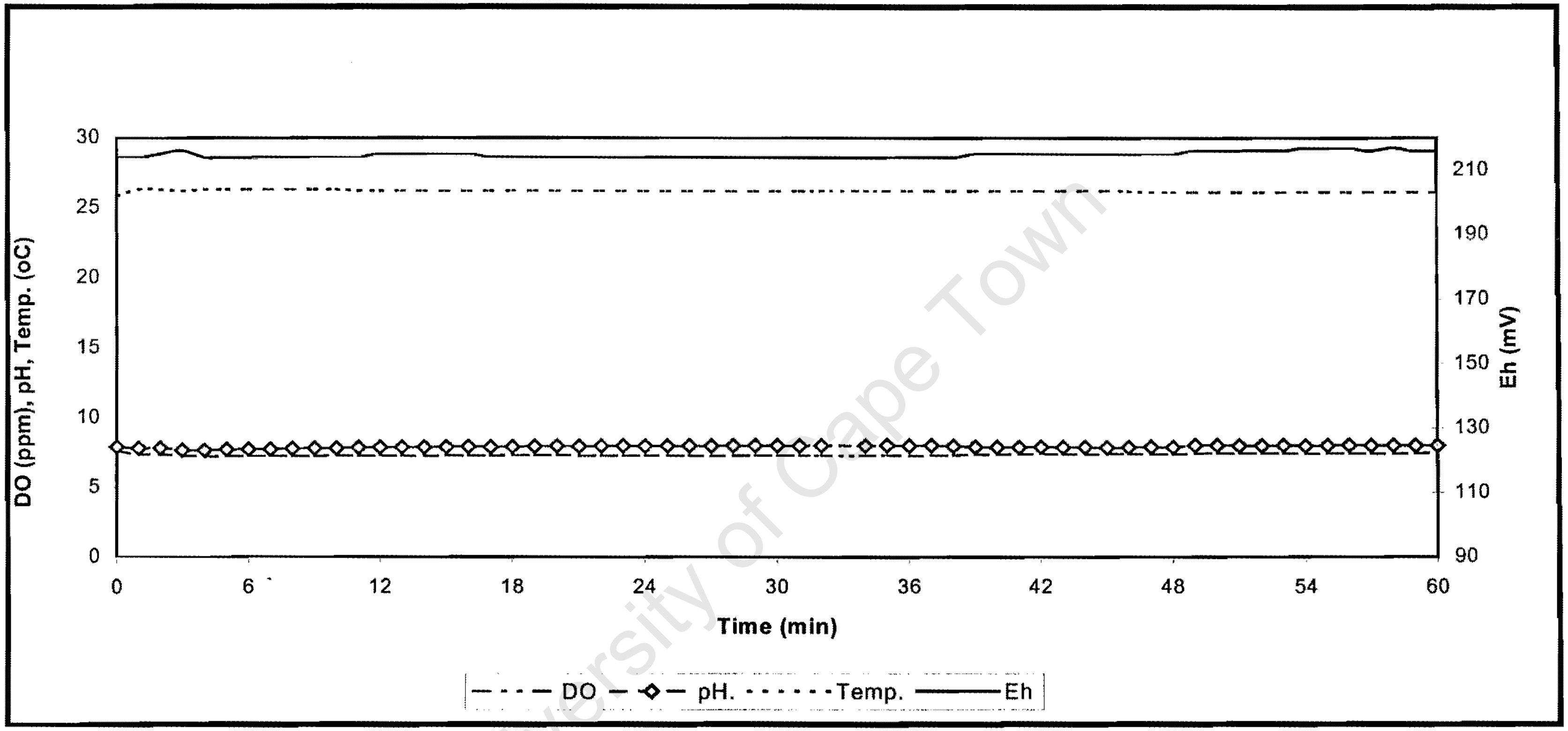


Figure 5.2.2.4: Dissolved Oxygen levels (DO), pH, Temperature (Temp.) and Pulp Potential (Eh) Profiles obtained in the Oxide Cleaner Banks (OCB)

APPENDIX D3

Detailed Pulp Chemistry Measurements of the Effect of Nitrogen Conditioning

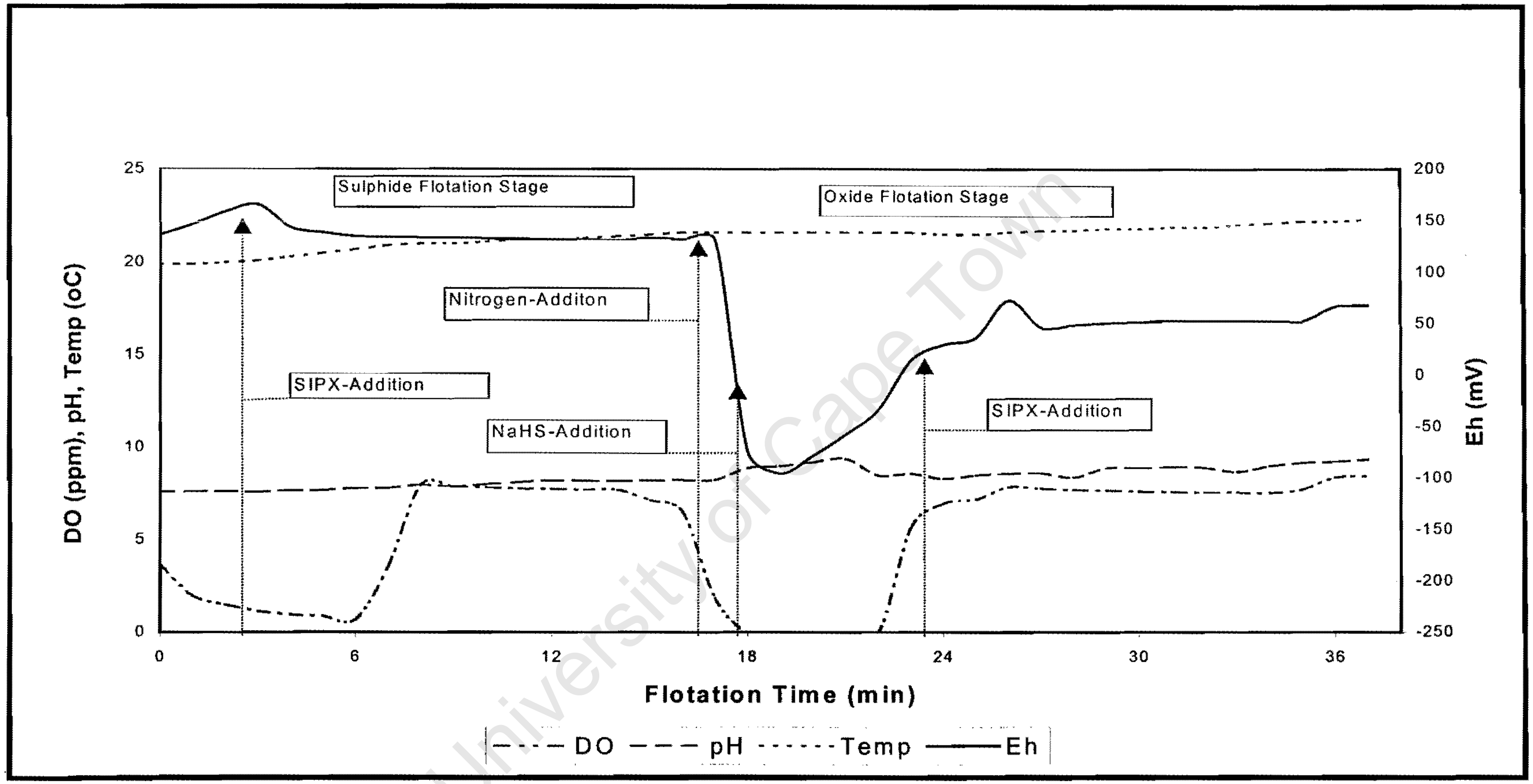


Figure 5.3.4 (a): Dissolved Oxygen levels (DO), pH, Temperature (Temp.) and Pulp Potential (Eh) Profiles obtained during Flotation before and after Nitrogen Conditioning

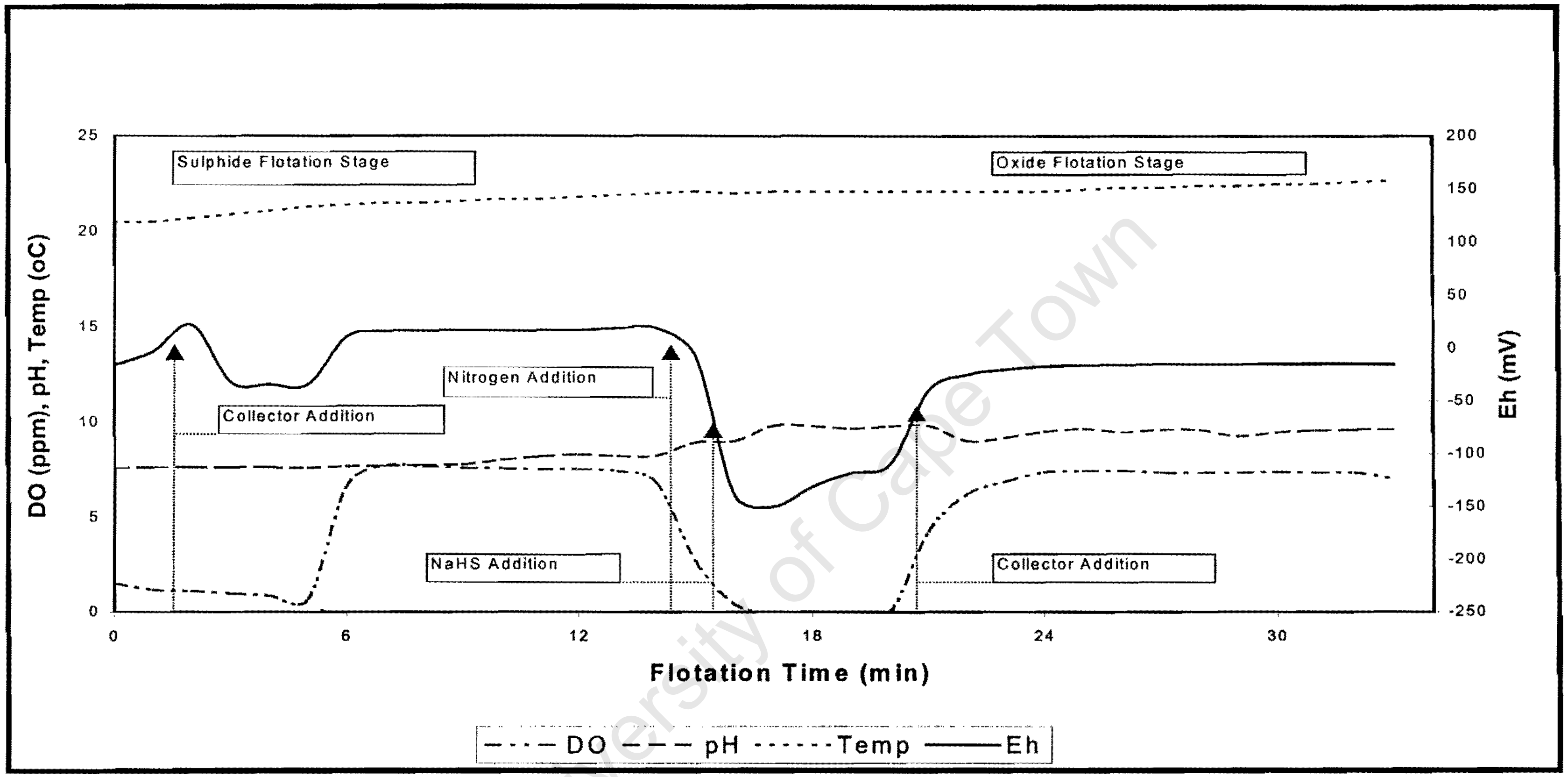


Figure 5.2.4 (b): Dissolved Oxygen levels (DO), pH, Temperature (Temp.) and Pulp Potential (Eh) Profiles obtained during Flotation before and after nitrogen Conditioning

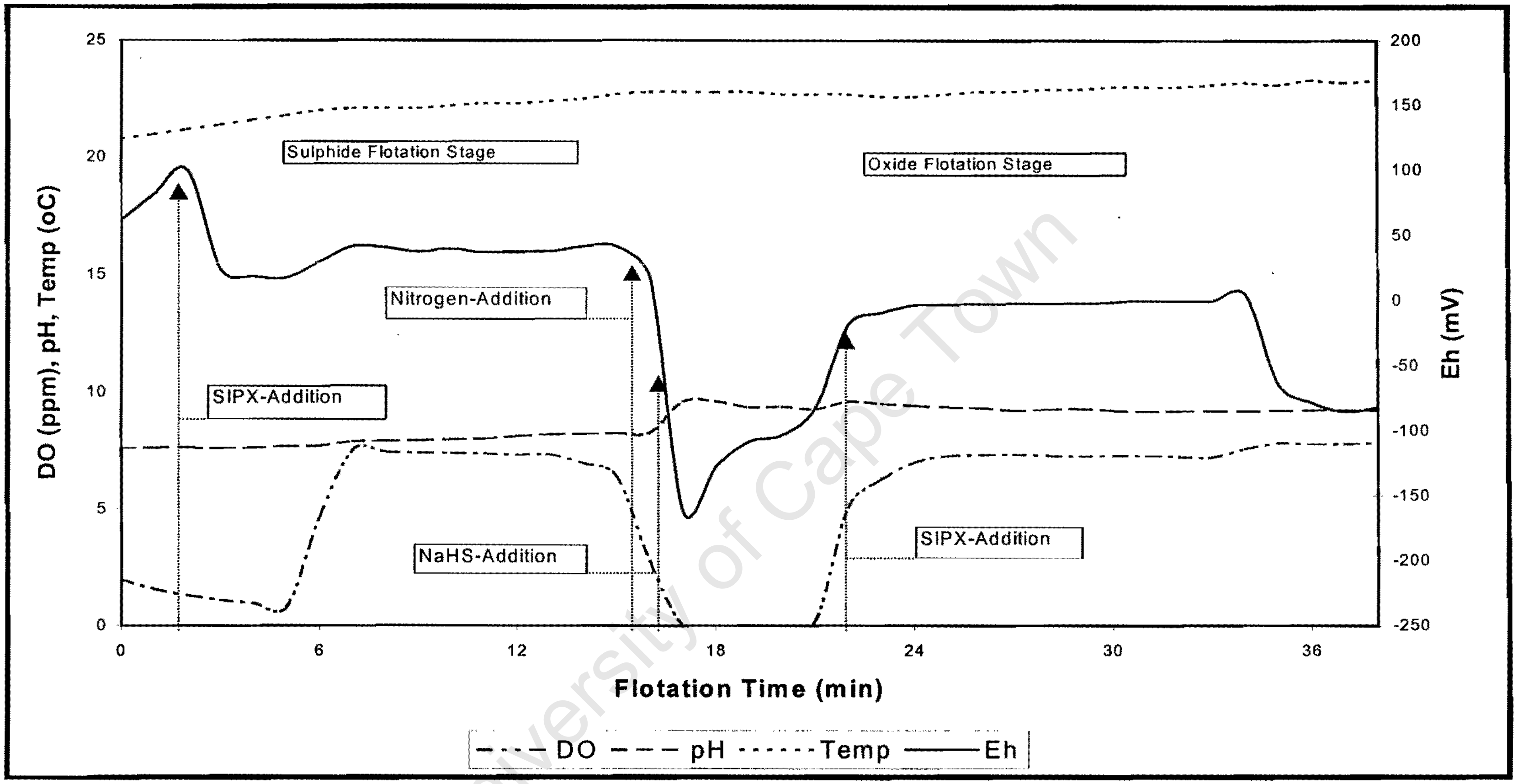


Figure 5.2.4 (c): Dissolved Oxygen levels (DO), pH, Temperature (Temp.) and Pulp Potential (Eh) Profiles obtained during Flotation before and after nitrogen Conditioning

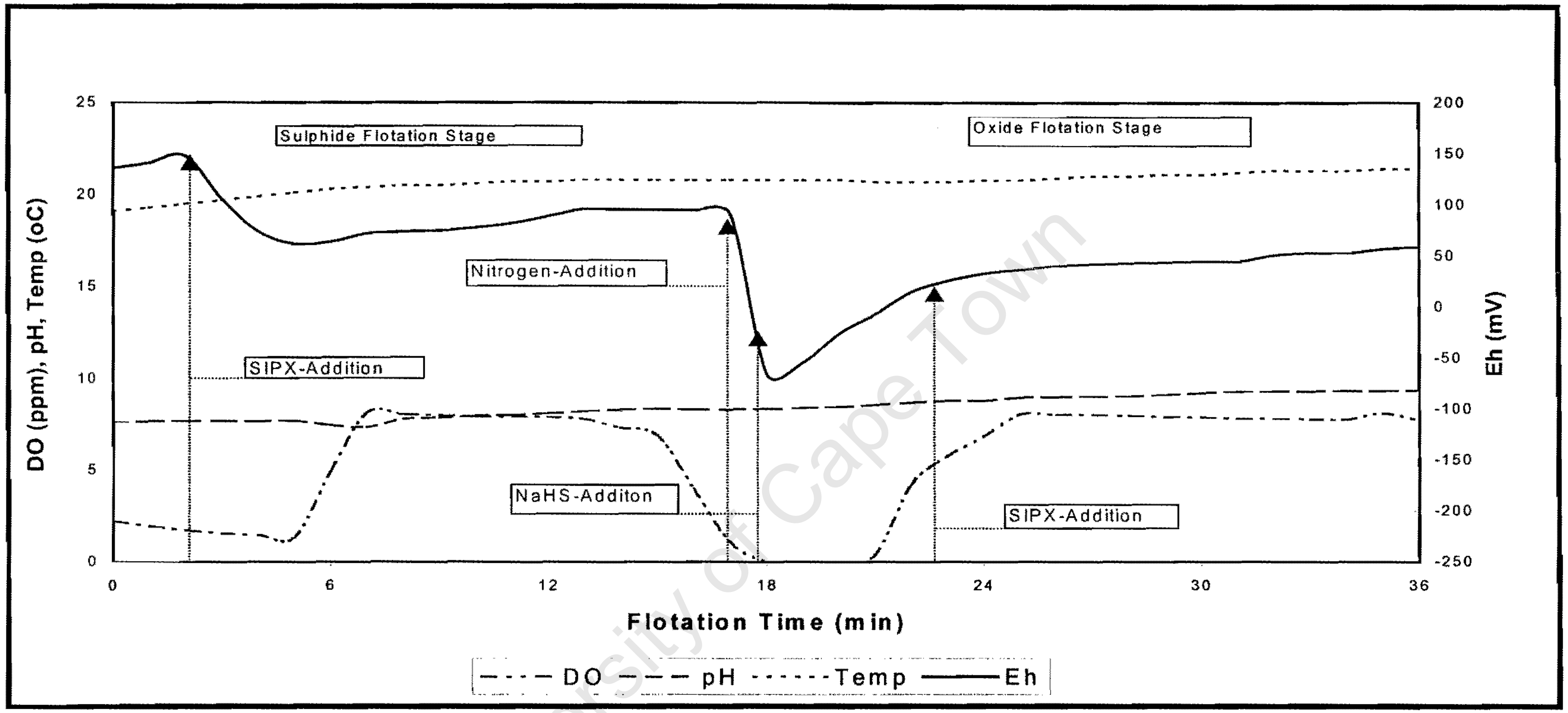


Figure 5.2.4 (d): Dissolved Oxygen levels (DO), pH, Temperature (Temp.) and Pulp Potential (Eh) Profiles obtained during Flotation before and after nitrogen Conditioning

APPENDIX E

Mineralogy of Nchanga Underground Copper Ore

ABSTRACT

Nchanga concentrator treats underground and open pits copper ores. The ores are a complex mixture of sulphide and oxide minerals and processing consists of sulphide flotation followed by sulphidisation and an oxide flotation step.

The oxide minerals have proved difficult to recover. The reasons for poor recovery have been attributed to complex mineralogical characteristics of the ore.

In this work a mineralogical examination was carried out on Nchanga underground copper ore feed. The investigation involved the examination of ores ground to -1700μm under both reflected and transmitted light microscopy to quantify the sulphides, 'oxides', and gangue minerals in the samples.

It was found that about 68% of the total copper (TCu) in the sample [2.66% of acid insoluble copper (AICu) in 3.93% total copper] occurs in acid insoluble form derived from mainly chalcocite and lesser to minor amounts being contributed by chalcopyrite, bornite, native copper and cuprite. The acid soluble copper is subordinate and was drawn mostly from malachite with lesser to minor amounts being derived from pseudomalachite, chrysocolla, azurite and cuprite. The gangue comprised predominantly quartz/feldspars with lesser carbonaceous shale while the carbonates and micas were minor. The calculation and tabulation of results are discussed. This exercise was later used as a basis to come up with suitable mineral processing approach for Nchanga Underground ore.

Keywords
Sulphide minerals; oxide minerals; gangue minerals

INTRODUCTION

Nchanga concentrator treats various ores mined from around Chingola, namely: Nchanga open pit, Luano open pit, Chingola open pit and Underground. The concentrator treats about 24000 tonnes of ore per day.

The ores are a complex mixture of sulphide and oxide minerals. The term 'oxide' is not limited to the actual copper oxides but includes the copper silicates, sulphates, phosphates and carbonates. In general, the term covers all copper ores where oxidation and weathering have proceeded past the stage of Chalcocite and native copper to yield the carbonates, malachite, azurite, chrysocolla and related minerals.

The copper oxides have proved difficult to recover and the reasons for poor recovery have been attributed to the complex mineralogical characteristics of the ore.

Originally the underground and open pits flotation pulps were mixed prior to flotation, but because of the problems the concentrator experienced with regard to the recovery of valuables, the underground and open pit flotation circuits were separated [1].

In this work a mineralogical examination was carried out on the underground copper ore feed. The investigation involved the examination of the ore ground to -1700μm under both reflected and transmitted light microscopy to quantify the sulphides, 'oxides' and gangue minerals in the sample.

EXPERIMENTAL METHOD

The primary crusher product of underground copper ore was obtained as a belt cut from Nchanga Mine (Zambia Consolidated Copper Mines, Division). The ore was crushed to -1700μm in a laboratory roll crusher.

The sample was mixed with epoxy powder and mounted. The mounted materials were polished and were ready for microscopic examination to identify the sulphide minerals. The sulphide minerals are opaque and are studied under reflected light microscopy.

The oxide minerals are examined under binocular microscopy. The sample was mixed thoroughly and a small amount put on a glass slide for examination.

All gangue minerals are transparent and are studied under transmitted light microscopy. The sample must be of standard size and as such it was further ground in a pestle and mortal, sieved on 106μm and 53μm sieve apertures. The –53μm was discarded and the –106μm or +53μm fraction retained for examination. After mixing the material thoroughly, a small amount was put on a glass slide and examined.

The properties studied were colour, shape, cleavage and isotropism and the method used to quantify the respective minerals was microscopic grain counting.

RESULTS AND DISCUSSION

Identification of Minerals

The identification of minerals was based on the following properties of the various groups of minerals.

(a) Sulphide Minerals

- Chalcocite: This is blueish-gray in colour and isotropic.
- Bornite: Pink-brown in colour and isotropic.
- Chalcopyrite: This is brass-yellow in colour and isotropic.
- Carrollite: Cream-white in colour and isotropic.
- Pyrite: Light yellow in colour and isotropic.

(b) Oxide Minerals

- Malachite: It is light to dark green in colour and semi-transparent. When a drop of dilute HCl is added to the mineral, it dissolves completely with effervescence.
- Pseudomalachite: This is grassy-green in colour, transparent and greasy in appearance. The mineral dissolves completely in dilute HCl without effervescence.
- Chrysocolla: Light blue in colour, dissolves in dilute HCl without effervescence and leave a skeleton of silica.
- Azurite: Dark-blue in colour and dissolves in dilute HCl with effervescence.
- Cuprite: Red in Colour and semi-transparent. A grain of Cuprite dissolves in concentrated HNO_3 with effervescence. When a drop of Potassium Mercuric Tricyanide is added a yellow precipitate indicates copper.

(c) Gangue Minerals

- Quartz (SiO_2): Under polarised light it is colourless. It has no cleavage but with low relief. Between crossed nicols it appears grey (interference colour). Turning the stage of the microscope it shows extinction.

- Feldspars: Microcline ($KAlSi_3O_8$) and Albite (NaAlSi3O8). Under ordinary light it is colourless and cloudy. Between crossed nicols it shows crosshatches (twinning).
- Carbonaceous shale: Appears dark under polarised light. Between crossed nicols it appears black.
- Carbonates: Colourless under polarised light. Between crossed nicols it shows higher order interference colours. Rotating the stage the relief changes.
- Mica: Brown in colour under polarised light and gives brown interference colour between crossed nicols.
- Talc: Colourless and flaky under polarised light. Between crossed nicols it appears dark.
- Argillite: This is an intergrowth between mica and quartz.
- Iron Oxides: Appears red under polarised light.
- Accessories: These are gangue minerals, which are not in larger amounts.

Chemical Analysis

The results of the chemical analysis are presented in the following Table I below.

Table I: Results of Chemical Analysis

Sample	%TCu	%ASCu	AICu
Copper Ore	3.93	1.27	2.66

Mineralogical Results

Table II gives the minerals found in the copper ore together with their respective formulae, composition and specific gravity used in the calculation [2]. The results of the mineralogical examination together with the calculations are given in Tables III, IV and V.

Table II: Minerals Found in Nchanga Copper Ore

MINERAL	FORMULA	S.G
Chalcopyrite	$CuFeS_2$ - Cu : 34.5, Fe : 30.5, S : 35	4.1-4.3 (4.2)
Bornite	Cu_5FeS_4 Cu:63.3 (2%A.S), Fe:11.1 S: 25.6	4.9-5.4 (5.2)
Chalcocite	Cu_2S - Cu : 79.8 (1.5% A.S)*, S: 20.2	5.5-5.8 (5.6)
Native Copper	Cu Cu : 100	8.8-8.9 (8.85)
Covellite	CuS - Cu :66.4, S : 33.6	4.6
Carrollite	Co_2CuS_4 - Cu:15.4, Co: 43.3, S: 41.3	4.5-4.9 (4.7)
Pyrite	FeS_2 - Fe :46.6, S : 53.4, (Co : 3)	4.95-5.10
Cuprite	Cu_2O - Cu : 88.8 (50-70% A.S)*	5.85-6.15
Malachite	$Cu_2(OH)_2CO_3$ - Cu:57.6 (100% A.S)*	3.9-4.0
Pseudomalachite	$Cu_5(PO_4)_2(OH)_4$ - Cu:55.2 (100% A.S)*	4.36
Chrysocolla	$CuOSiO_2.nH_2O$ - Cu: 36.2 (100% A.S)*	2.0-2.2
Azurite	$Cu_3(OH)_2(CO_3)_2$ - Cu:55.53 (100% A.S)*	3.84
Cupriferous Mica	Cu-VERMICULITE - TCu: 4, ASCu: 1	

* A.S = Acid Soluble

Calculation Steps

1st Step

Calculate the relative abundance (**RA%**) of the individual sulphides in the sample. Multiply the specific gravity by the **total grains** of individual sulphides. Then add the products of **SG** and total **grains** of all the sulphides. The percentage of each product of **SG** and **total grains** on the total sum gives the **RA%** of the respective mineral. Determine the theoretical copper present in the copper sulphides. This is done by multiplying the **RA%** of each mineral by the theoretical copper contained in it e.g. Bornite contains 63.3% Cu. Then determine the multiplication factor (**MF**) i.e. **AICu** divided by the theoretical copper. The **MF** multiplied by the **RA%** of individual mineral gives the weight percentage (**Wt%**). Weight percentage multiplied by the theoretical copper contained in each mineral give the total copper **(%TCu)** contributed by individual sulphides. Also determine the quantity of **ASCu** contributed by **bornite (bn)** and **chalcocite (cct).**

2nd Step

Determine the copper oxides present in the sample under binocular microscope. Determine the **RA%** of copper oxides. Adding the total grains of each mineral and then calculating percentage of each total grain over the sum do this. Determine the percentage of cupriferous mica present. Heating the sample on a Bunsen burner does this. After cooling a small amount is put on the glass slid under binocular microscope. Cupriferous mica looks like worms. Calculate the **%TCu** and **%ASCu** contributed by cupriferous mica.

3rd Step

Determine the theoretical copper present in the copper oxides. Deduct **ASCu** if any contributed by the sulphides and **ASCu** contributed by **Cupriferous mica** from the **ASCu** of the chemical analysis. Determine the **MF** by dividing the remaining ASCu by the theoretical copper of the oxides. Then determine the **Wt%, %TCu, %ASCu** of the individual copper oxides.

4th Step

Deduct the **TCu** of the copper oxides from the **TCu** of the chemical analysis to get **AICu**. Using this **AICu** calculate the **MF, Wt%, %TCu, %ASCu** of individual sulphides. The sum of the **TCu** of the sulphides and oxides must be equal to the **TCu** of the chemical analysis. Also the sum of the **ASCu** of the sulphides and oxides must be equal to the **ASCu** of the chemical analysis.

Example

Table III: Sulphide Minerals

MINERAL	$T_{(grains)}$	T*S.G	RA%	Wt%	%TCu	%ASCu
Chalcopyrite	389	1633.8	26.31	1.15	0.40	-
Bornite	40	208.0	3.35	0.15	0.09	<<0.01
Chalcocite	693	3880.8	62.50	2.73	2.18	0.03
Pyrite	89	445.0	7.17	0.31	-	-
Carrollite	5	23.5	0.38	0.02	<<0.01	-
Native - Cu	2	17.7	0.29	0.01	0.01	-
Total		6208.8	100.00		2.68	0.03

RA% = Relative Abundance

Theoretical copper in the copper sulphides = 26.31*0.345+3.35*0.0633 +62.5*0.798 + 0.38*0.154 + 0.29*1 = 61.42102

Chemical analysis: **TCu** = 3.93
-ASCu = 1.27
AICu **2.66**

$$MF = \frac{2.66}{61.42102} = 0.043307649$$

We use this first **MF** to calculate the **ASCu** contributed by **bn** and **cct**. In this case **ASCu** contributed by **bn** is negligible (<<0.01) where as that of **cct** is **0.03%ASCu**. (Then we calculate for oxides).

TCu = 3.93
-TCu in oxides = 1.25
2.68

$$\therefore MF = \frac{2.68}{61.42102} = 0.04363327$$

We use this second **MF** to calculate the **Wt%, %TCu**, and **%ASCu** for all minerals above except **pyrite** (which does not contain copper).

Table IV: Oxide Minerals

MINERAL	RA%	Wt%	%TCu	ASCu
Malachite	90	1.97	1.13	1.13
Pseudomalachite	4	0.09	0.05	0.05
Chrysocolla	4	0.09	0.03	0.03
Azurite	1	0.02	0.01	0.01
Cuprite	1	0.02	0.02	0.02
Cupriferous Mica	-	0.2 - 0.3	0.01	<<0.01
Total	100		1.25	1.24

Theoretical copper in **Cu** oxides = 0.9*57.6 + 0.04*55.2 + 0.04*36.2 + 0.01*55.53 + 0.01*88.8*0.7 = 56.6729

Chemical analysis: **ASCu** = 1.27
-ASCu in sulphides = 0.03
1.24

$$MF = \frac{1.24}{56.6729} = 0.021879946$$

Note: If the **ASCu** contributed by **cupriferous mica** is of substantial amount, it has to be subtracted from the **ASCu** of the chemical analysis.

When we add the **TCu** and the **ASCu** for sulphides and oxides they must be equal to that of the chemical analysis.

Chemical analysis: **TCu** = 3.93
ASCu = 1.27

Mineralogical analysis: **TCu** = 3.93
ASCu = 1.26

Table V: Gangue Minerals

MINERAL	T (grains)	RA%	RANGE	Wt%
Quartz / Feldspars	567	84	80-85	77
Carbonaceous Shale	73	11	11-13	11
Carbonates	13	2	2-3	2
Mica	14	2	2-3	2
Talc	2	<<1	<<1	<<1
Argillite	1	<<1	<<1	<<1
Iron 'Oxides'	5	1	<1	<<1
Accessories	-	-	-	-
Total		100		

The weight percentage for gangue is determined by adding the Wt% of individual sulphide and oxide minerals and the total subtracted from 100%. For copper ore, Wt% of gangue = 93.19. To get the Wt% of individual gangue minerals e.g. Quartz / feldspars (RA% = 82.5), Wt% = 0.825*93.19 = 77.

CONCLUSION

Approximately 68% of the total copper (TCu) sample [(2.66% of acid insoluble copper (AICu) in 3.93% TCu] occurs in acid insoluble form derived from mainly Chalcocite and lesser to minor amounts being contributed by chalcopyrite, Bornite, native copper and Cuprite. The acid soluble copper is subordinate and is drawn mostly from malachite with lesser to minor amounts being derived from Pseudomalachite, chrysocolla, azurite and Cuprite. The Cupriferous mica content is low (0.2-0.3% by weight).

The gangue in the ore comprises predominantly quartz/feldspars with lesser carbonaceous shale while the carbonates and micas are minor. Talc, Argillite and iron 'oxides' are negligible.

After the above mineralogical characterisation of the ore, suitable mineral processing method for this particular ore was arrived at as reported elsewhere.

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1. Mpashi, P. and Kasanda, J., Evaluation and commissioning of the separate treatment of open pit and underground ores at Nchanga concentrator (ZCCM), Concentrator Department, Report No 431 (1995).

2. Minerals table for Nchanga copper ore, Zambia Consolidated Copper Mines Limited (ZCCM), Technical Services, Mineralogy Department, Kalulushi, (1998).

APPENDIX F

Statistical Analysis of Results

1.0 The t-test for Two Means (Napier-Munn, 1994-2000)

To test the significance of the observed difference $\bar{x}_1 - \bar{x}_2$ between the means of two samples of size n_1 and n_2 with standard deviations (s.d.s) s_1 and s_2, calculate

$$t = \frac{\bar{x}_1 - \bar{x}_2}{s\sqrt{1/n_1 + 1/n_2}}$$

where $$s = \sqrt{\frac{(n_1 - 1)s_1^2 + (n_2 - 1)s_2^2}{n_1 + n_2 - 2}}$$

(s^2 is called the pooled variance). Compare this value of t with tables, using ($n_1 + n_2 - 2$) degrees of freedom (d.f), at the desired probability level (level of significance).

When the sample sizes are the same ($n_1 = n_2 = n$), then the formulae simplify considerably to:

$$s = \sqrt{\frac{s_1^2 + s_2^2}{2}} \quad \text{and} \quad t = \frac{\bar{x}_1 - \bar{x}_2}{s\sqrt{2/n}}$$

The confidence limits on the difference $(\bar{x}_1 - \bar{x}_2)$ are given by $\pm t\, s(1/n_1 + 1/n_2)^{0.5}$ where t is obtained from t-tables with an appropriate confidence level and ($n_1 + n_2 - 2$) d.f.

The calculations for the analysis shown in the results are given below.

Examples

1.1 Reproducibility of the Sulphide Flotation Step after Mild Steel and Stainless Steel grinding.

In order to determine whether the sulphide flotation step, after mild steel and stainless steel grinding gives the same results, the means of the recoveries and grades were compared.

The null hypothesis, H_0, is $\bar{x}_{\text{mild steel}} = \bar{x}_{\text{stainless steel}}$, where $\bar{x}$ is equal to the mean value of recovery or grade as specified.

(a) Recovery of Acid Soluble Copper (ASCu)

	Mild Steel	Stainless Steel
$\bar{x}$ (%)	10.58	16.40
s (%)	1.44	1.01
n	4	4

Null hypothesis, H_0: $\bar{x}_{mild} = \bar{x}_{staileless}$ (d.f. 4 + 4 – 2 = 6)
since $n_1 = n_2$,

$$s = \sqrt{\frac{1.44^2 + 1.01^2}{2}} = 1.24, \qquad t = \frac{10.58 - 16.40}{1.24\sqrt{2/4}} = -6.64$$

Comparing with tables ($t_{90;\,6}$ = 1.94), the value of t is found to be significant, and thus we conclude that the two milling methods give different recoveries of acid soluble copper to sulphide concentrates.

(b) Recovery of Acid Insoluble Copper (AICu)

	Mild Steel	Stainless Steel
$\bar{x}$ (%)	78.93	76.80
s (%)	3.77	5.73
n	4	4

Null hypothesis, H_0: $\bar{x}_{mild} = \bar{x}_{staileless}$ (d.f. 4 + 4 – 2 = 6)
since $n_1 = n_2$,

$$s = \sqrt{\frac{3.77^2 + 5.73^2}{2}} = 4.85, \qquad t = \frac{78.98 - 76.80}{4.85\sqrt{2/4}} = 0.62$$

Comparing this with tables ($t_{90;\,6}$ = 1.94), the value of t is found to be not significant, and thus we conclude that the two milling methods give the same recovery of acid insoluble copper to sulphide concentrate.

(c) Grade of ASCu

	Mild Steel	Stainless Steel
$\bar{x}$ (%)	2.78	4.29
s (%)	0.25	0.20
n	4	4

Null hypothesis, H_0: $\bar{x}_{mild} = \bar{x}_{stailess}$ (d.f. 4 + 4 – 2 = 6)
since $n_1 = n_2$,

$$s = \sqrt{\frac{0.25^2 + 0.20^2}{2}} = 0.226, \qquad t = \frac{2.78 - 4.29}{0.226\sqrt{2/4}} = -9.45$$

Comparing this with tables ($t_{90;\,6}$ = 1.94), the value of t is found to be significant, and thus the difference in grade of acid soluble copper between mild steel and stainless steel is real.

(d) Grade of AICu

	Mild Steel	Stainless Steel
$\bar{x}$ (%)	31.30	30.22
s (%)	0.90	1.79
n	4	4

Null hypothesis, H_0: $\bar{x}_{mild} = \bar{x}_{stailess}$ (d.f. 4 + 4 – 2 = 6)
since $n_1 = n_2$,

$$s = \sqrt{\frac{0.90^2 + 1.79^2}{2}} = 1.42, \qquad t = \frac{31.30 - 30.22}{1.42\sqrt{2/4}} = 1.08$$

Comparing this with tables ($t_{90;\,6}$ = 1.94), the value of t is found to be not significant, and thus we conclude that the two milling methods give the same grade of acid insoluble copper to sulphide concentrate.

1.2 Evaluation of the Effect of Milling Media and Regrind

In order to evaluate the effect of grinding media and of regrind, the means of the recovery and grade were compared.

Examples

1.2.1 Mild Steel Grinding (MS) Versus Mild Steel Regrind (MSR)

1.2.1.1 Recovery of Acid Soluble Copper (ASCu)

	MS (ASCu)	MSR (ASCu)
$\bar{x}$ (%)	16.30	22.35
s (%)	0.42	0.21
n	2	2

Null hypothesis, H_0: $\bar{x}_{MS} = \bar{x}_{MSR}$ (d.f. 2 + 2 – 2 = 2)
since $n_1 = n_2$,

$$s = \sqrt{\frac{0.42^2 + 0.21^2}{2}} = 0.332, \qquad t = \frac{16.30 - 22.35}{0.332\sqrt{2/2}} = -18.22$$

Comparing this with tables ($t_{90;\,2} = 2.92$), the value of t is found to be significant, and thus we conclude that the mild steel regrind gives high recovery of acid soluble copper compared to mild steel with no regrind.

1.2.2 Mild Steel Grinding (MS) Versus Stainless Steel Grinding (SS)

1.2.2.1 Recovery of Acid Soluble Copper (ASCu)

	MS (ASCu)	SS (ASCu)
$\bar{x}$ (%)	16.30	23.05
s (%)	0.42	0.07
n	2	2

Null hypothesis, H_0: $\bar{x}_{MS} = \bar{x}_{SS}$ (d.f. $2 + 2 - 2 = 2$)
since $n_1 = n_2$,

$$s = \sqrt{\frac{0.42^2 + 0.07^2}{2}} = 0.301, \qquad t = \frac{16.30 - 23.05}{0.301\sqrt{2/2}} = -22.43$$

Comparing this with tables ($t_{90;\,2} = 2.92$), the value of t is found to be significant, and thus we conclude that the two milling methods give different recovery of acid soluble copper.

1.2.2.2 Recovery of Acid Insoluble Copper (AICu)

	MS (AICu)	SS (AICu)
$\bar{x}$ (%)	84.2	94.0
s (%)	0.85	3.04
n	2	2

Null hypothesis, H_0: $\bar{x}_{MS} = \bar{x}_{SS}$ (d.f. $2 + 2 - 2 = 2$)
since $n_1 = n_2$,

$$s = \sqrt{\frac{0.85^2 + 3.04^2}{2}} = 2.232, \qquad t = \frac{84.2 - 94.0}{2.232\sqrt{2/2}} = -4.39$$

Comparing this with tables ($t_{90;\,2} = 2.92$), the value of t is found to be significant, and thus we conclude that the two milling methods give different recovery of acid insoluble copper.

1.3 Comparison between the Standard Flotation Performance of Phase I and Phase III

1.3.1 Recovery of Acid Soluble Copper (ASCu)

	Phase I (ASCu)	Phase III (ASCu)
$\bar{x}$ (%)	16.30	63.18
s (%)	0.42	5.23
n	2	4

Null hypothesis, H_0: $\bar{x}_{\text{Phase I}} = \bar{x}_{\text{Phase III}}$ (d.f. 2 + 4 – 2 = 4),

$$s = \sqrt{\frac{(2-1)\,0.42^2 + (4-1)\,5.23^2}{2+4-2}} = 4.534, \qquad t = \frac{16.30 - 63.18}{4.534\sqrt{1/2 + 1/4}} = -11.94$$

Comparing this with tables ($t_{90;\,4} = 2.13$), the value of t is found to be significant, and thus we conclude that the difference in the recovery of acid soluble copper between Phase I and Phase III is real.

1.3.2 Recovery of Acid Insoluble Copper (AICu)

	Phase I (AICu)	Phase III (AICu)
$\bar{x}$ (%)	84.20	80.90
s (%)	0.85	3.39
n	2	4

Null hypothesis, H_0: $\bar{x}_{\text{Phase I}} = \bar{x}_{\text{Phase III}}$ (d.f. 2 + 4 – 2 = 4),

$$s = \sqrt{\frac{(2-1)\,0.85^2 + (4-1)\,3.39^2}{2+4-2}} = 2.966, \qquad t = \frac{84.20 - 80.90}{2.966\sqrt{1/2 + 1/4}} = 1.285$$

Comparing this with tables ($t_{90;\,4} = 2.13$), the value of t is found to be not significant, and thus we conclude that the recovery of acid insoluble copper in Phase I and Phase III is the same.

2.0 Testing the Significance of Nitrogen Conditioning Statistically by using the Analysis of Variance (ANOVA)

The Analysis of Variance (ANOVA) [Napier-Munn, 1994-2000]

The ANOVA is a powerful procedure, which can be used to determine whether several means are significantly different to one another, as a group, by using the F-test to assess the significance of the variance due to the different means.

The ANOVA essentially partitions the total data variance into its components, and makes comparisons between them. For example in a simple replicated experiment to compare two 'treatments', there are two sources of variation: treatment, and error. Each has a sum of squares associated with it, which is a measure of variation. The sum of squares due to treatment, S_1, is:

$$S_1 = \sum_{i=1}^{k} n_i \left(\bar{x}_i - \bar{x}\right)^2$$

where k = number of treatments (2 in this case)

$\bar{x}$ = overall data mean

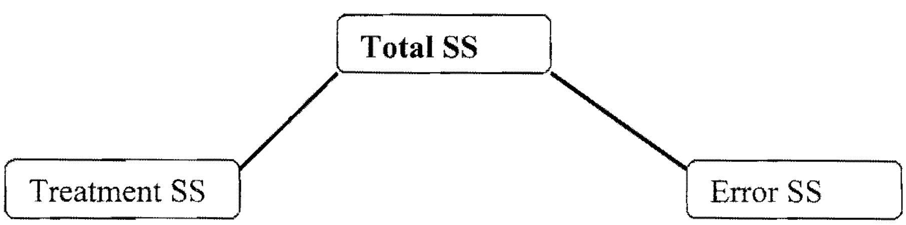
$\bar{x}_i$ = mean of i^{th} treatment

n_i = number of replicates of i^{th} treatment

The sum of squares due to error (the residual sum of squares after the treatment differences have been accounted for), S_0, is:

$$S_0 = \sum_{j=1}^{n_1} \left(x_{1j} - \bar{x}_1\right)^2 + \ldots + \sum_{j=1}^{n_k} \left(x_{kj} - \bar{x}_k\right)^2$$

where x_{ij} is the j^{th} observation for the i^{th} treatment. The sums of squares (SS) are additive:



Example

In order to find the effect of sulphidisation in the nitrogen environment on the flotation recovery of oxide minerals, four flotation tests were conducted on the copper ore. One experiment was done with sulphidisation in air and the other with nitrogen. The following were the results of the recovery of acid soluble copper (ASCu) during the oxide flotation stage:

Air (ASCu)	Nitrogen (ASCu)
67.8	64.5
59.0	71.0
67.6	69.2
58.3	77.4

$\bar{x} = 66.85$

$\bar{x}_{air} = 63.18$

$\bar{x}_{N_2} = 70.52$

$S_1 = 4(63.18 - 66.85)^2 + 4(70.52 - 66.85)^2$
$= 107.76$

$S_0 = (67.80 - 62.18)^2 + ...(58.3 - 63.18)^2 + (64.5 - 70.52)^2 + ...(77.40 - 70.52)^2$
$= 167.70$

(The total SS is given by $\sum_{i=1}^{k} \sum_{j=1}^{n_k} \left(x_{ij} - \bar{x}\right)^2$, but this is more easily obtained in this example as $S_0 + S_1$).

The F-test is used to compare the SS_s, but before this can be done they must be converted to units of variance by dividing by the corresponding degrees of freedom, to give the mean square (MS).

$Df_{treatment}$ = number of treatment – 1 = 1

Df_{total} = total number of results - 1 = 7

Df_{error} = Df_{total} - $Df_{treatment}$ = 6

Where:

$Df_{treatment}$ = degrees of freedom for the treatment in this case, two treatments, air and nitrogen conditioning. Therefore, $Df_{treatment} = 2 - 1 = 1$.

Df_{total} = degrees of freedom of the total number of results = 8 – 1= 7.

Df_{error} is the difference between Df_{total} and $Df_{treatment = 7 - 1 = 6}$.

The results of the analysis are presented in the ANOVA table, as follows:

Source of variation	SS	Df	MS	F
Treatment (on ore)	107.76	1	107.76	3.85
Error (residual)	167.70	6	27.95	-
Total	275.46	7	-	-

$$MS = \frac{SS}{Df}$$

The F-test assesses whether the MS due to treatments (the real difference between the test conditions) is significantly greater than that due to error.

$$F = \frac{MS_{treatment}}{MS_{error}} = 3.85 \text{ with } 1{,}6 \text{ df}$$

From tables, this value is significant at the 90% level, and thus we can conclude that the difference of 7.34%ASCu in the recovery between the flotation process with and without nitrogen conditioning is real.

Reference

Napier, T.J., (1994-2000). An Introduction to Comparative Statistics and Experimental Design for Minerals Engineers, 2nd Edition, Version 4.0. *Julius Kruttschnit Mineral Research Centre*, University of Queensland, Australia.

APPENDIX G

Milling Curves

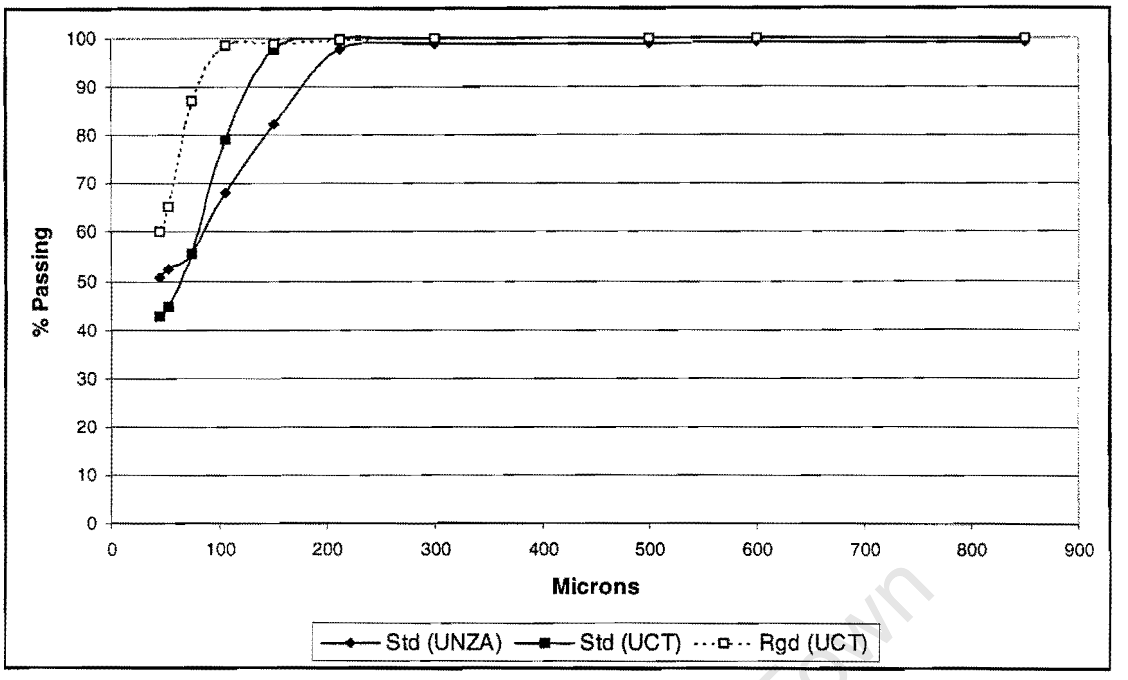


Figure 1: Screen Analysis of Standard Grinding (Std.) and Regrind (Rgd) at UCT and UNZA.