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MM 590 FINAL YEAR PROJECT REPORT

FULL RELEASE ANALYSIS OF ORE FROM ALL NKANA MOPANI COMPANY MINE SHAFTS.

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ABSTRACT

Due to low copper and cobalt recoveries that were being experienced, and thus a considerable rise in mineral losses to the tailings, this study was proposed. The specific objective was to establish the optimum mesh of grind for all Nkana copper and cobalt sulphide ores. This was done through laboratory batch flotation test work. Therefore the current work is an extension of previous work done by Mutentwa (2001) who investigated Chibuluma, South Ore Body and Mindola ores.

This study was done through laboratory flotation tests conducted under different experimental conditions for all Mopani Copper Mine sulphide ores. Mineral samples were collected, prepared and subjected to grinding and flotation tests. The average head grades were 1-2.5% copper and about 0.2% cobalt.

From the results obtained, the optimum mesh of grind for central shaft ore was 70% passing 75µm with recoveries of 97.6% and 88.2% for copper and cobalt respectively. That for South Ore Body ore was also 70% passing 75µm giving recoveries of 95% and 80% for copper and cobalt respectively. Mindola Subvertical ore had 60% passing 75µm as the optimum mesh of grind with recoveries of 96.4% and 87.6% for copper and cobalt respectively. The optimum mesh of grind for Mindola North ore was also 60% passing 75µm with recoveries of 94.6% and 93.8% for copper and cobalt respectively. These results are different from the optimum mesh of grind being practiced which was 65% passing 75µm for all the shaft ores.

It was therefore concluded that the low copper and cobalt recoveries, as well as the considerable rise in mineral losses to the tailings were due to the change in the optimum meshes of grind for the different shaft ores.

Therefore, it is recommended that the plant personnel may implement the above optimum meshes of grind as established in this study.

DEDICATION

This report is dedicated to my mother, Loveness Nkhoma, for her unwavering support.
And also brothers and sisters who have often offered me support in different ways during my stay at the University Of Zambia.

TABLE OF CONTENTS **Page**

List of tables

List of figures

CHAPTER ONE

1.0 INTRODUCTION

1.1 Background

This project was carried out at Mopani Copper Mines PLC, Nkana Concentrator in Kitwe. The concentrator treats about 320600 tonnes per month of sulphide ores containing both copper and cobalt. The ores are received from different shafts namely, South Ore Body, Central, Mindola North and Subvertical shafts. The mineralogical composition of the ores is mainly Chalcopyrite, Bornite, Chalcocite, Pyrite and Carrolite. The methods of processing of these ores include crushing, grinding, flotation, segregation, filtration followed by disposal of gangue material.

1.2 Process Description (Nkana Concentrator)

1.2.1 Ore Reception

The concentrator receives ores from four different shafts namely South Ore Body (SOB), Central, Mindola Subvertical and North shafts. SOB, Mindola Subvertical and North shafts are outside the plant area and their ore is delivered by rail wagons with capacity of 40 tonnes. Central shaft is within the plant and delivers its ores using skips of 10 tonnes capacity into the 600 tonnes capacity central bin. The ores from SOB, Mindola Subvertical and North shafts, having passed through primary crushing from their respective shafts are 15 to 24 cm in size. This ore is discharged into the Foreign Ore Bin (FOB) situated under the rail line. The bin consists of 10 compartments of 180 tonnes capacity giving a total bin capacity of 1800 tonnes. The ore is discharged from the bin to a conveyor by means of electric vibratory feeders placed under each compartment. It is then taken for secondary crushing. This is shown in flow sheet Figure 1.1

1.2.2 Secondary Crushing

The 15cm primary crushed ore from the four shafts is conveyed into a 600 tonnes capacity standard bin before being fed into the standard cone crushers at 350 tonnes per hour per crusher through a 1m by 2.8m double deck mogul screen with 7.6cm by 7.6cm oversize screens and 38mm by 38mm undersize screen. The minus 38mm undersize ore from the mogul screen proceeds to the 5500 tonnes capacity mill bin while the crusher product goes to the 2.1 by 4.3m Hewitt Robbins screen. The Hewitt Robbins oversize reports to the short head crushers (tertiary crushers) via the 200 tonnes capacity tertiary bin for further reduction and circulate back to the Hewitt Robbins screen thus forming a closed circuit. To effect these reductions at secondary crushing, the standard crusher gaps are set at 21mm to 25mm and the short head crusher gaps are set at 16mm to 18mm. The minus 38mm undersize ore from the Hewitt Robbins screen joins the mogul screen undersize and are taken to the mill bin through the 6 A weightometer. The 20000 tonnes capacity fines bin provides ore storage space before the mill bin. The Hewitt Robbins oversize ore can be alternatively taken to the 200 tonnes capacity oxide bin and crushed by the 1.53m oxide crusher to the1600 tonnes capacity silo.

1.2.3 Grinding

The minus 38mm ore from the mill bin is weighed by the inflow weightometer before being fed to the 2.75m by 3.66m allis chalmers rod mills with a charge of 3.5m diameter mill rods at 80 to 90 tonnes per hour (TPH). The rod mill discharge is pumped to the 76.2cm hydrocyclones where the underflow is fed to the 2.75m by 3.66m allis chalmers ball mills charged with 80mm mill balls. The ball mill discharge is circulated back to the hydrocyclones together with fresh feed thus forming a closed circuit. The 38% solids and 65% passing 75μm cyclone overflow is first diluted with water to 34% solids before being taken as feed to flotation. There are 8 units of allis chalmers mills where a unit comprises one rod mill and one ball mill. The ore from the silo on the oxide route is fed to the 3.35m by 4.27m vecor rod mill (charged with 4.12m) mill rods at an ore throughput of 120 tonnes per hour. The rod mill discharge together with the discharge from the ball mill (charged with 80mm mill balls) is pumped to 30 inch (76.2cm) hydrocyclones. The hydrocyclones underflow circulates back via the ball mill, while the overflow is taken as feed to flotation together with the cyclone overflow from the allis chalmers mill units.

1.2.4 Flotation

1.2.4.1 Bulk Flotation

The cyclone overflow from grinding section is fed into the two 80 cubic meters tank cells at a head grade of 1.60% copper and 0.09% cobalt. An 8% solution of the collector Sodium Ethyl Xanthate (SEX) at 60g/t ore milled and a 100% solution of Aerofroth 68 is added to the head of flotation (tank cell feed). The tank cell tailings 0.80% copper and 0.05% cobalt and proceed as feed to the fagergren banks while the concentrates (rougher concentrates) at about 15% copper and 0.80% cobalt proceed to the column cells for cleaning. The tank cells recovery is around 60% copper and 45% cobalt. The tank cell tailings proceed as feed to the 4 by 12 cells of fagergren banks having a capacity of $8.5m^3$ each. The bank is divided into three blocks of four cells each called primary, secondary and scavenger. The rougher concentrates from the first five cells (the primary cells and secondary cells) proceed to the cleaning section while the rest of the concentrates from the last seven cells are recycled back to the primary grinding section as middlings. The Fagergren Banks tailings are the final concentrator tailings pumped to the effluent disposal plant at 0.08% copper and 0.023% cobalt. The rougher concentrates are in the range of 8.0% copper and 0.4% cobalt. The Fagergren banks recovery to the rougher concentrates is in the range of 70% copper and 30% cobalt.

In the past, Fagergren tailings were cycloned at the pachucas and the cyclone overflow was taken as the final tailings while the underflow was taken to the coarse flotation plant which no longer exists. The combined rougher concentrates from the tank cells and bulk flotation Fagergren cells, at about 10% copper and 0.8% cobalt are fed into two of the three $54m^3$ cominco column cells, column 1 and 2 at slurry volume ranging between 1400 to 2000 litres/min and a density of around 1280g/l. Columns 1 and 2 tailings are fed to the third column while concentrates, at around 24% copper and 1% cobalt, proceed to copper/cobalt segregation banks. The tailings from column three are recycled back to primary grinding section as the fresh feed while the concentrates, at around 18% copper and 1.2% cobalt, join those from columns one and two to segregation. The average recovery from the columns is 60% copper and 50% cobalt. The cleaning section occasionally includes the 1.87 cubic meters by 8 cells cobalt re-flotation bank. Part of the rougher concentrates is fed to cobalt re-flotation cells where the concentrates proceed to segregation and the tailings are recycled back to the head of flotation circuit.

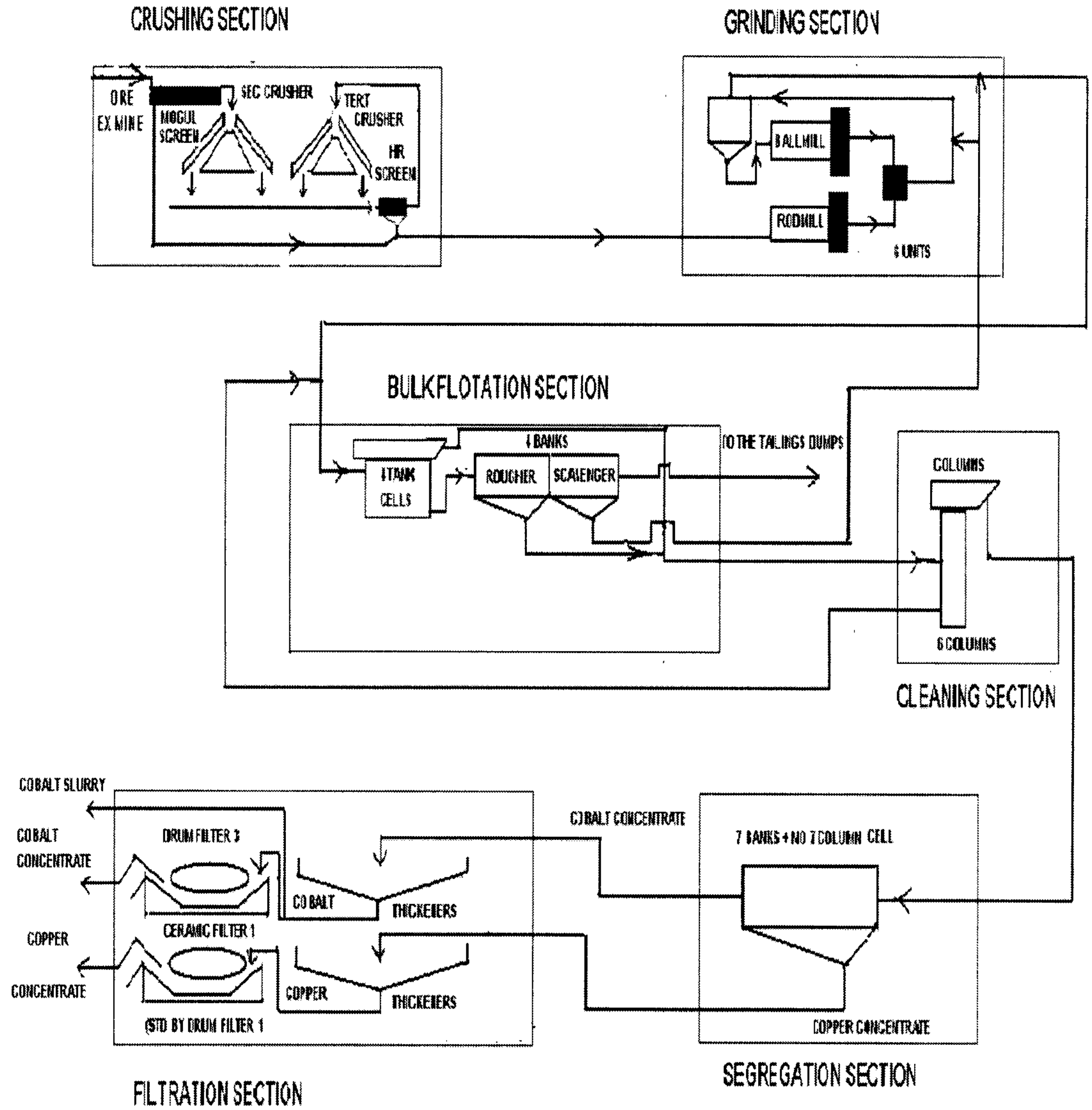

Figure 1.1; Concentrator process flow sheet

1.2.4.2 Segregation

In this section, copper/cobalt separation is effected by raising the pH to 12 and the use of sodium cyanide which depresses the cobalt mineral and only copper minerals float. The column concentrates are fed into the two 1.87m^3 x 7 cells second stage banks. The concentrates from the second stage report to copper re-cleaner cells for further cleaning while the tailings are fed to the third stage cells.

The two 1.87m^3 x 5 copper re-cleaner banks of which one is double discharge receive the second stage concentrates. The tailings from copper re-cleaner banks are recycled back to second stage and the concentrates are the final copper smelter concentrates with a grade ranging between 27% and 32% copper and 0.30% cobalt.

The two 1.87m^3 x 5 cells third stage banks receive second stage tailings. The concentrate report back to second stage while the tailings are the final cobalt concentrates with 7 to 10% copper and 1.5 to 2% cobalt.

1.2.5 Dewatering (Filter Plant)

The copper concentrate is filtered separately after being thickened to 70% solids in two 15m diameter thickeners. The underflow pumped to agitator tank from where the material is drawn and fed to a ceramic or drum filters no 2 and 3 as shown in Figure 1.2. The filtered concentrate at 15% moisture is sent for drying in a rotary kiln to reduce the moisture to 11% before being conveyed to smelter shed. The cobalt concentrate is thickened separately at 55% solids and is filtered to 17% moisture using drum filters four and five as shown in Figure 1.2. The concentrate is conveyed to the cobalt shed from where it is dispatched to the cobalt plants.

1.2.6 Effluent Disposal

The effluent disposal plant receives the concentrator final tailings, cobalt and acid plants effluents. Hydrocyclones at the pachucas split the concentrator tailings effluents into the underflow which reports directly to the receiving tanks and the overflow which together with cobalt and acid plant effluents reports to the pachucas where they are mixed before being discharged into 76.25m tailings thickener. The feed to the thickener is around 20% solids at about pH 9 and the thickener underflow is around 38 to 45% solids. The thickener underflow is pumped to the receiving tanks where together with the pachucas cyclone underflow is pumped to the tailings dams.

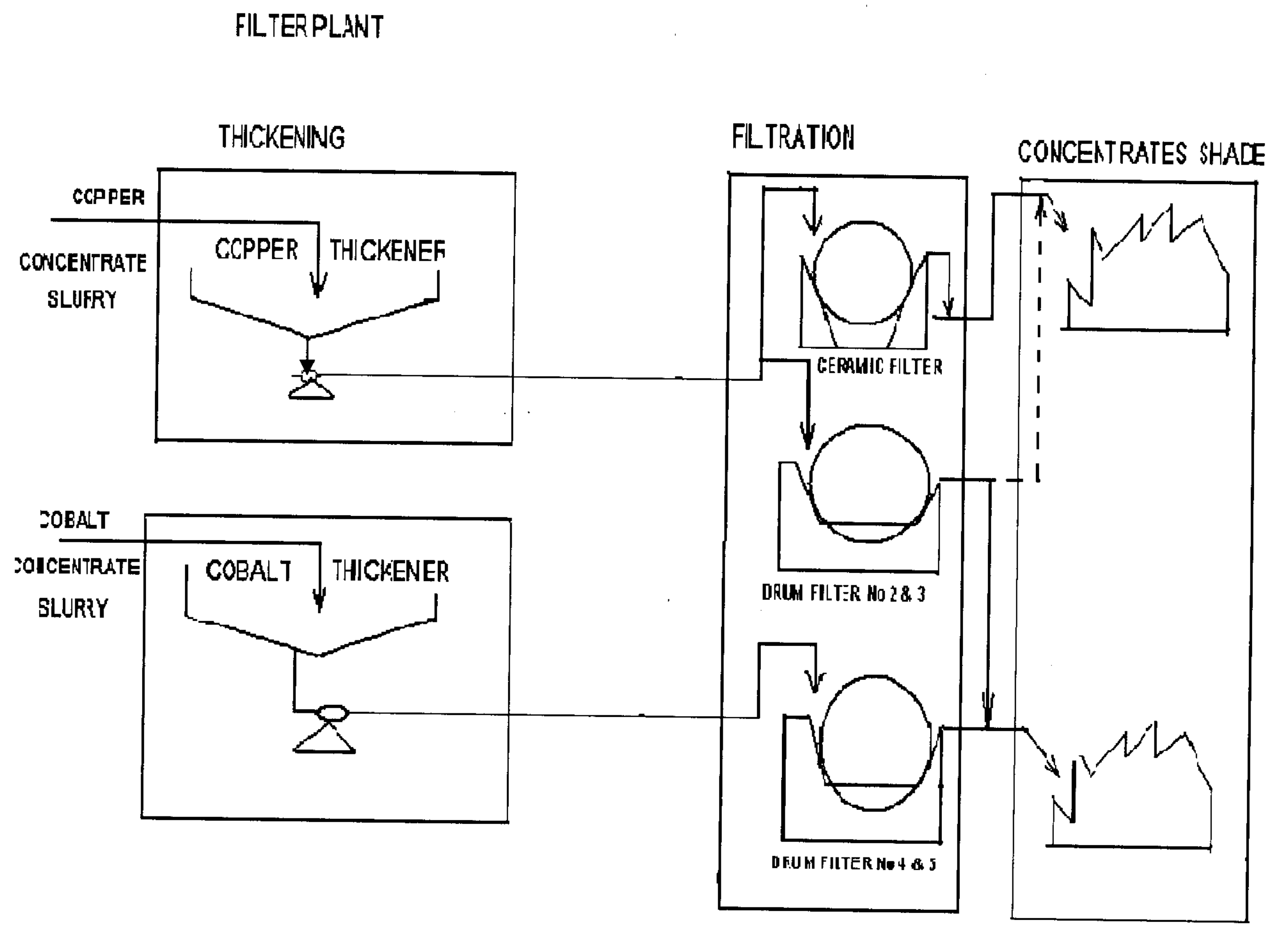

Figure 1.2; Schematic flowsheet of filter plant showing filter types used.

1.3 Previous work

Similar work was done by Mutentwa (2001), the difference being that in place of central shaft ores, work was done on Chibuluma shaft ores. The results obtained after this work showed the optimum meshes of grind shown below:

1 Chibuluma: 70 % passing 75µm and a grinding time of 30 minutes.
2 South Ore Body: 60 % passing 75µm and grinding time of 7 minutes.
3 Mindola: 60 % passing 75µm and grinding time of 6 minutes

The current investigation was done as an extension of the work by Mutentwa (2001) and was aimed at studying additional ores from different sources.

1.4 Problem statement

Due to low copper and cobalt recoveries that were being experienced, and thus a considerable rise in mineral losses to the tailings, this study was proposed to establish the optimum mesh of grind for all Nkana ores. The effects of some operating parameters on metal recoveries were also investigated.

1.5 Objectives.

The main objective of this project was to investigate the cause for the drop in recovery of copper and cobalt for Mopani Copper Mine Nkana Ores.

1. The specific objective was to determine the optimum mesh of grind for all the Mopani Copper Mine Nkana shaft ores through release analysis.

CHAPTER TWO

2.0 LITERATURE REVIEW

2.1 Comminution

Comminution is size reduction of run-of-mine ores to liberate the enclosed minerals. The first stage in comminution is crushing which is done by impact and abrasion in Jaw, Gyratory, Cone and many more types of crushing machines. Crushing comprises primary, secondary and tertiary stages. Primary crushing reduces run-of-mine ore from large rocks to a maximum of 15cm in diameter. Secondary crushing reduces ores from 15cm diameter to a range from 2.5 to 5cm while tertiary crushing reduces ores from maximum of 5cm to a range of 1 to 0.5cm diameter size. This particle size goes for grinding where it is further reduced to optimum size of mineral liberation (Wills, 1987).

2.1.1 Grinding Principles.

It is the last stage in the process of comminution. This is done in cylindrical rotating mills called tumbling mills. Among the factors that affect wet grinding are; speed of mill, pulp level, amount of charge, hardness of ores, time taken during grinding, size of grinding media and the circulating load. All ores have an economic optimum mesh of grind which is obtained if ore is ground at optimum conditions. Undergrinding leads to coarse product with a low degree of liberation for economic separation. Overgrinding reduces the particle size of the gangue and may reduce the mineral value to size below that required for efficient separation. Minute particles easily float, oxidize, flocculate and agglomerate causing low mineral liberation efficiency Ø. (Jain et al., 1987)

2.2 Causes Of Mineral Losses To The Tailings.

Under grinding produces coarse particles that are too heavy to be raised by air bubbles. Hence, a lot of mineral values remain in the pulp after concentration by flotation. Another cause of mineral losses to tailings is slimes production from over grinding. Slimes have a large surface area to mass ratio causing them to easily oxidise, flocculate and agglomerate. Agglomerates and flocculated particles are too heavy to be floated while oxides cannot be floated using sulphide flotation methods. Another problem with slimes is that they do not easily settle to the bottom but remain in suspension in fluid medium (process water). Thus some fine tailings report to the float phase, increasing tailings content in the concentrate. This reduces flotation efficiency (Jain, 1987; Wills, 1987)

2.3 Flotation Principles

Release analysis is based on the principles of flotation. Flotation is a concentration process by which desired minerals are separated from the pulp by attaching themselves to air bubbles that lift them to the surface of the pulp where they are separated from the gangue. This process uses the differences in physical-chemical surface properties of particles of various minerals. This process requires attachment of air bubbles to mineral particles to float them. Hydrophilic minerals are made hydrophobic and aerophilic by collector addition. Collectors added to the pulp adsorb on mineral surfaces rendering them hydrophobic and facilitating bubble attachment. The mineral is usually transferred to the froth, or float fraction leaving the gangue in the pulp or tailing. Frothers are used to stabilize the froth while regulators control the flotation process. Regulators either activate or depress mineral attachment to air bubbles and are also used to control the pH of the system. (Jain, 1987; Wills, 1987)

2.3.1 Collectors

The property responsible for mineral attachment to air bubbles is either natural or acquired character of mineral surface. Molybdenite and graphite are examples of substances which naturally attach to air bubbles, meaning that they are aerophylic and hydrophobic. Collectors are reagents which when added to the pulp causes the hydrophilic minerals to be hydrophobic and aerophylic for easy attachment to air bubbles during flotation process. These are made up of polar groups and one or more non polar hydrocarbon groups. Sulphide polar groups are predominantly R-O-C (-S)-SM where M stands for ions like sodium, potassium and ammonium ions. Non polar groups include alkyl groups (i.e. C_2H_5, C_5H_{12} etc) and cresyl group ($CH_3C_6H_4$). Upon attachment to air bubbles, the mineral particles are raised to the surface of the pulp thereby being separated from the gangue materials. Thus the desired flotation mineral particles must be light enough in order to be raised by the air bubbles. If they are too heavy, they detach themselves from the air bubble and sink into the pulp. This lowers flotation efficiency.

The main cause for this is coarse grinding. On the other hand, over ground particles cause flotation of too much gangue material due to their lightness as they do not easily settle in the pulp. This also reduces flotation efficiency. Therefore, optimum grinding is required for efficient flotation of the required minerals. Examples of collectors include Xanthates (i.e. potassium ethyl xanthate, sodium ethyl xanthate, and sodium isopropyl xanthate e.t.c.), monothiophosphates, dithiophosphates, and also nitrogen based collectors like dialkyl thionocarbamates. Others are amines, fatty acids, tall oil and petroleum sulphonates (Day, 2002; Weiss, 1985).

2.3.2 Frothers

Frothers are reagents which stabilise the floating minerals (froth) so that they do not sink into the pulp. They achieve this by making bubble-air attachments to be stronger. Addition of these reagents causes bubble size reduction in the pulp and also formation of froth phase. Bubble diameters measured within pulps range from 0.1mm to 1mm while those in the froth increase to as much as 15cm. The maximum value of bubble diameter depends upon frother type, its concentration, mineral loading and froth height among other factors. The ideal frother or frother combination chosen should produce frothing conditions suitable for mineral transport to the froth phase and subsequent cell overflow while also allowing drainage of entrained gangue particles. Factors affecting frothing action of frothers include; type of flotation cell, ore granulometry, minerals present, retention time and presence of slimes. Examples of frothers include; dow froth, pine oil, cresylic acid and fuel oil e.t.c. (Day, 2002; Weiss, 1985).

2.3.3 Modifiers

In addition to providing collector adsorption to produce floatability, surface treatment must also control selectivity through preferential adsorption of collector by the mineral to be floated and prevention of its adsorption by other minerals. Modifiers or regulators are reagents which either activate or depress mineral attachment to air bubbles. They are added mainly to achieve selectivity during flotation process thereby regulating or controlling the flotation process. Modifiers are either activators or depressants. Activators activate attachment of particular minerals to air bubbles causing them to float. Depressant prevents particular mineral attachment to air bubbles preventing them from floating. Therefore these are used in separation of two or more desired minerals which are in the same froth. A typical example of an activator is sodium aerofloat which promotes flotation of chalcocite, various copper sulphides and sphalerite at pH values of 11.8, 11 and 9.0 respectively. An example of depressant is sodium cyanide solution which depresses flotation of chalcopyrite at a pH of 11.4 while galena is floated using anyl xanthate collector. More examples of regulators include sodium silicate, sodium ferrocyanide, hydrofluoric acid and sodium sulphide (Day, 2002; Weiss, 1985).

2.4 Release Analysis

The release analysis procedure can be split into two stages. In the first, cleaning stage, the ground ore sample is re-floated several times to separate the floatable from the non-floatable material. In the second stage, the floatable particles are separated into a number of fractions of progressively decreasing grade. This is based on the particles flotation rates, high-grade particles float faster than low-grade particles. Any laboratory cell may be used for a release analysis. However, the procedure used depends upon the degree of control available on impeller speed and airflow rate. The initial sample weight should be large enough to give fractions easily assayed and 1 kg is generally sufficient.

Wet grinding is preferred as this prevents surface oxidation and allows conditioning to be done in the mill. An excess of collector is maintained throughout the test in order to maximize recovery. Frother dosages, however, should be kept as low as possible so as not to produce a too stable froth and to minimise gangue carry over during cleaning. This coupled with slow froth removal can cut down the number of cleaning stages required and shorten the number of the time taken for the test. The main practical difficulty to be encountered is that water should be decanted from the collecting basins before the concentrate is re-introduced to the cell for cleaning. Care should be exercised here so that no concentrate is lost. An alternative to this problem is using progressively larger cells.

CHAPTER THREE

3.0 MATERIALS AND METHODS

This chapter describes the materials and the different procedures used in carrying out the experiments.

3.1 Materials

3.1.1 Apparatus

The laboratory equipment used in the test work were; Flotation Machine Agitator, LA 500 with impellor speed of 1200 rpm, Laboratory Jaw Crusher, Laboratory Ball Mill, Laboratory Scale, Oven and an Oakton pH meter.

3.1.2 Reagents

The reagents used included; Sodium Ethyl xanthate, a collector at the dosage of 60g/t, A Frother, Aerofroth 68, Concentrated Sulphuric Acid and Lime Solution.

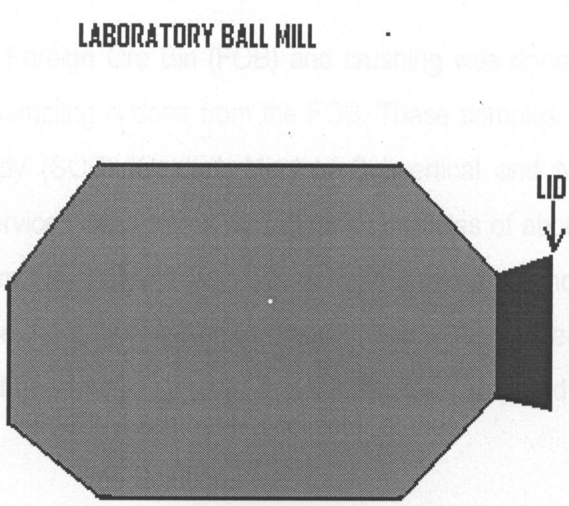

Fig 3.1: Laboratory Ball Mill

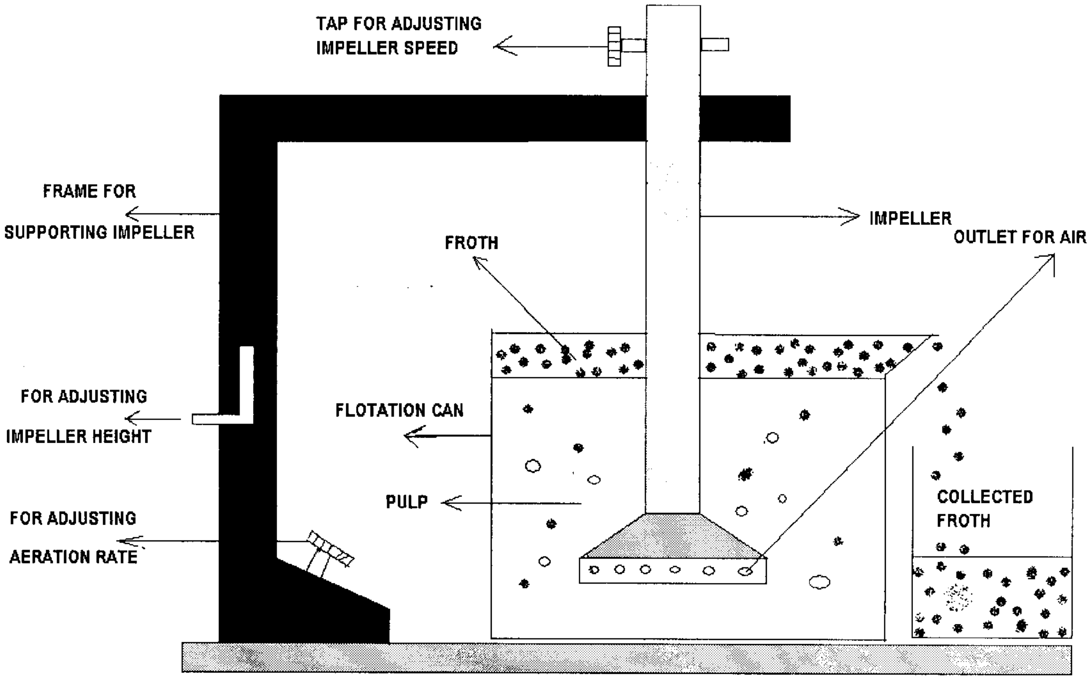

Fig 3.2: Laboratory Flotation Machine

3.2 Methods

3.2.1 Sample collection and preparation

The sampling of ores from the Foreign Ore Bin (FOB) and crushing was done by the Analytical Services department. Plant operational sampling is done from the FOB. These samples included ores from the four shafts namely; South Ore Body (SOB), Central, Mindola Subvertical and North shafts. The ore size collected from the Analytical Services department was -8 mesh particles of about 50 kg for each shaft ore. They were then transported from the analytical services department to the concentrator laboratory using a wheelbarrow. Further, they were thoroughly mixed to ensure homogeneity of each ore. Before grinding, 1 kg samples were also cone and quartered to ensure that the samples prepared were representative of the bulk of the ores.

3.2.2 Elemental analysis

Table 1 Ore Head Grades

Ore type	South Ore Body		Central Shaft		Mindola Subvertical		Mindola North	
Metals Present	Cu	Co	Cu	Co	Cu	Co	Cu	Co
Head Grades	1.89	0.078	2.05	0.140	2.15	0.10	3.60	0.24

Table 2: Mineralogical composition of ores (Phiri Maxwell, 2007)

Minerals	Ore sample				
	Weight %	% Cu	% ASCu	% Co	% TS
Chalcopyrite	2.27	0.78	-	-	0.79
Bornite	2.21	1.40	0.03	-	0.57
Chalcocite	0.47	0.38	0.01	-	0.09
Pyrite	0.64	-	-	0.02	0.34
Carrolite	0.23	0.05	-	0.09	0.09
Malachite	0.23	0.13	0.13	-	-
Pseudo malachite	0.01	0.01	0.01	-	-
Chrysocolla	0.04	0.01	0.01	-	-
Gangue	93.90	-	-	-	-
Total	100.00	2.76	0.19	0.11	1.88

3.3 Experimental Procedures

During the experiments that were done, the variables that were kept constant were; speed of grinding mill, pH values, flotation machine impeller speed, reagent dosages. The only changing variables were grinding time and grind size. The pH value for flotation of Mindola North and Mindola Subvertical ores was 9 while for South Ore Body and Central shaft ores was 8.5. This difference in pH values was according to plant requirements. The flotation impeller speed was fixed at 1200 revolutions per minute. Sodium ethyl xanthate was fed at 60 grams per tone of ore as a collector. A total of 4 drops of 100 percent (weight to volume) of aerofroth 68 was fed as a frother. During grinding the percentage of solids in the pulp was 66.67 on the basis of weight. The percentage of solids in the pulp during flotation was 33.33 percent on weight basis.

3.3.1 Grind Time Determination

The grind time determination experiments done would be used to find the grindability of the different shaft ores. By definition, grindability is the ease with which an ore is ground. Ores with higher grindability are easier to grind than those with lower grindability, meaning that they are softer. These softer ores reach finer sizes quickly, meaning time taken to grind to very fine sizes is short. This means that their work indexes are lower compared to those with lower grindability. The experiment below is mainly based on these facts.

Using laboratory Ball Mill, 1 kg of minus 8 mesh samples were ground at time intervals of 4, 8, 12, 20 and 30 minutes and dried. Then 300g samples were wet screened to remove very fine particles (minus 45µm mesh). This was done using water at high pressure, which was passed through the sample placed on a 45µm sieve until the liquid coming from below was clear. The plus 45µm material remaining on the sieve was again dried and dry screening using sieves of sizes 212, 150, 106, 75 and 45µm was done. A curve relating cumulative percentage passing 75µm and grinding time was constructed from screen analysis data.

3.3.2 Grind Optimisation

Five 1 kg samples of each ore were ground at 40, 50, 60, 70, and 80 percent passing 75µm. The times taken to grind to these percent passing 75µm were read from the curve-relating percentage passing 75µm and grinding time. These ground samples were then fed for flotation. The pulp was transferred from the ball mill to the laboratory flotation cell having a capacity of 2litres, which is suitable for a 1 kg sample. The pH of the pulp was kept constant at 8.5 for Central and SOB shaft ores and 9.0 for Mindola Subvertical and north shaft ores. These pH values are recommended at plant levels. The flotation was carried out according to the flow sheet (Fig 3.3) and test conditions for each experiment.

During flotation the froth was removed by skimming at regular time intervals (every after 10 seconds) using plastic paddles. Total flotation time was 8 minutes (i.e. 3 minutes for rougher flotation and 5 minutes for scavenger flotation). Conditioning time for rougher and scavenger flotation were 2 and 1 minute respectively. The concentrates and tailings obtained after flotation were filtered and dried in the oven or drying pans. After drying, the fractions were weighed and packed in sample bags and sent to analytical services department for chemical analysis of percentages of copper and cobalt.

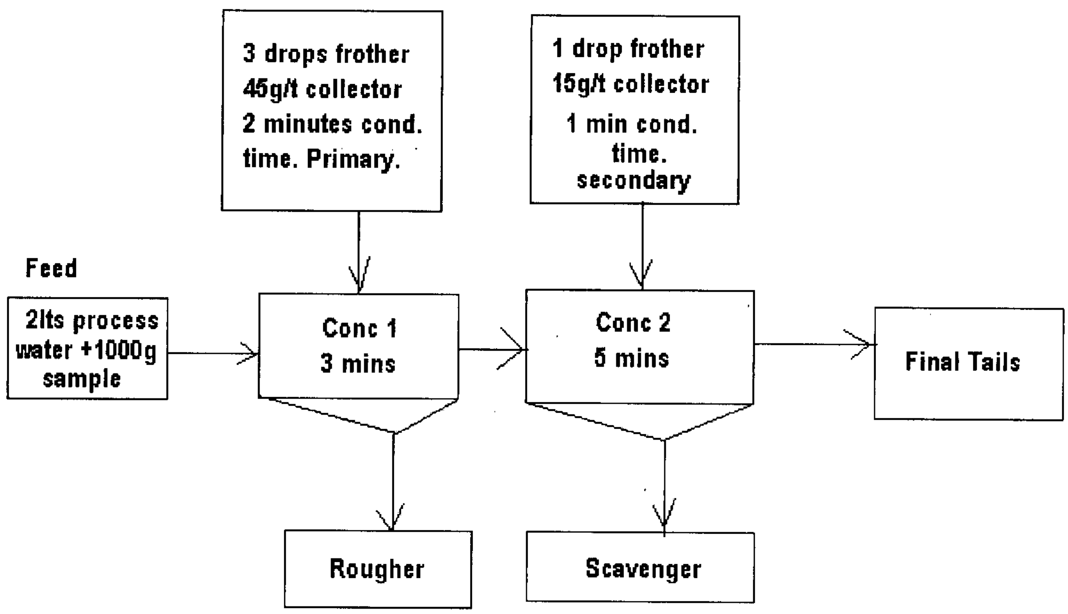

Fig 3.3: flotation flow sheet

3.4 Limitations

- Sampling and crushing was done by the analytical services department. This was due to defective roll crusher and lack of transport by the concentrator laboratory.
- PH and Reagent optimisation was not done due to lack of adequate time. If this was the cause for the fall in recovery, then it would not be found.
- Equipment were not always available as they were also used for plant work and therefore, there was need to wait until available for use.

CHAPTER FOUR

4.0 RESULTS AND DISCUSSION

This chapter presents a discussion of the results obtained during the various experiments conducted. They are discussed according to type of experiments done. These are grind time determination and batch flotation tests.

4.1 Grind time determination

This section discusses grind time determination curves for all the four shafts and their importance in relation to this study.

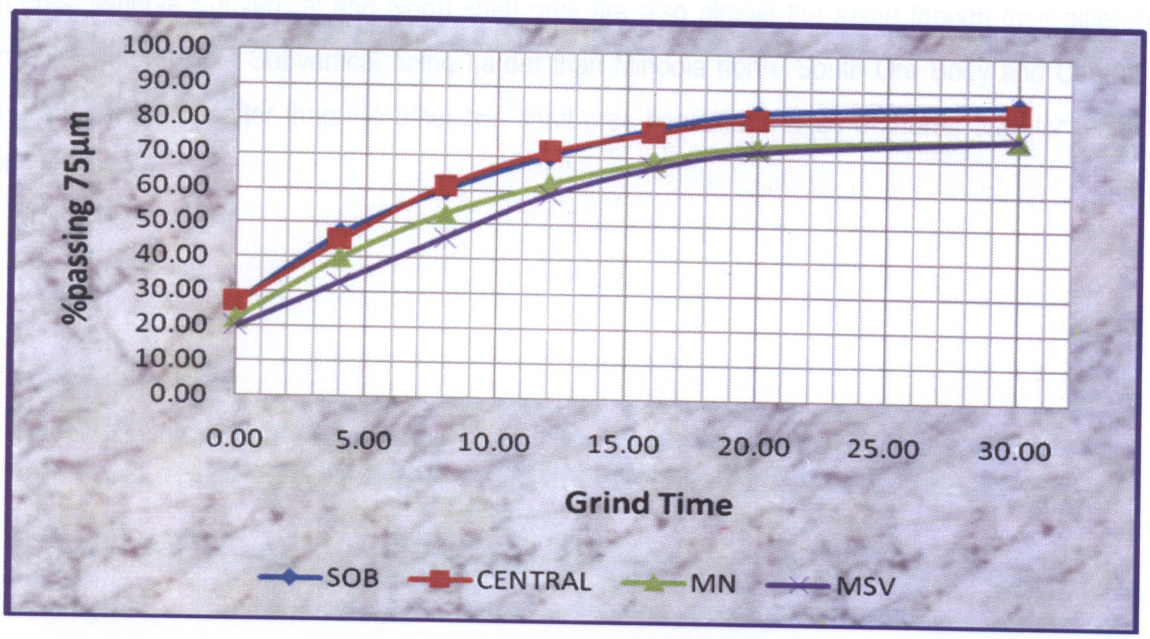

Fig 4.1.1: % passing 75µm against grind time

Figure 4.1.1 shows curves with a continuously decreasing slope. As the percentage of particles passing through 75µm increases (size becoming finer), the curves approach horizontal straight lines. If grind time is increased indefinitely, a stage can be reached where the curves become horizontal straight lines meaning that there is no further reduction in the particle size despite continued grinding.

These types of curves are used to read the time taken to grind to a particular size distribution (percent passing 75μm). For example, time taken to grind central shaft ore to 80 % passing 75μm is 20 minutes. Similarly, time taken to grind South Ore Body (SOB) ore to 80 % passing 75 μm is 17 minutes. For Mindola Subvertical (MSV) and Mindola North (MN), after grinding for up to 30 minutes, the size of particles could not be reduced to 80 percent passing 75μm. The smallest size for these two ores in this investigation was 76 % passing 75 μm which took about 30 minutes. Smaller sizes could have been reached if grinding time was increased.

These types of curves can also be used to compare the grindabilities of the different shaft ores. It can be read from the graphs (Fig 4.1.1) that Central and SOB shafts had almost the same hardness as they are almost the same. These two ores have a higher grindability than Mindola North and Mindola Subvertical shaft ores. Mindola Subvertical and North shaft ores are also almost the same though their difference is noticeable, with Mindola Subvertical being harder than Mindola north. South Ore Body and Central shaft ores are reported as softer than the other two shaft ores because it takes a shorter time for them to be ground to finer sizes.

4.2 Grind Optimisation (with regard to copper recoveries) for all ores

This section discusses the copper recoveries obtained for all the shaft ores, how they compare with each other and how the optimum mesh of grind was obtained for each ore.

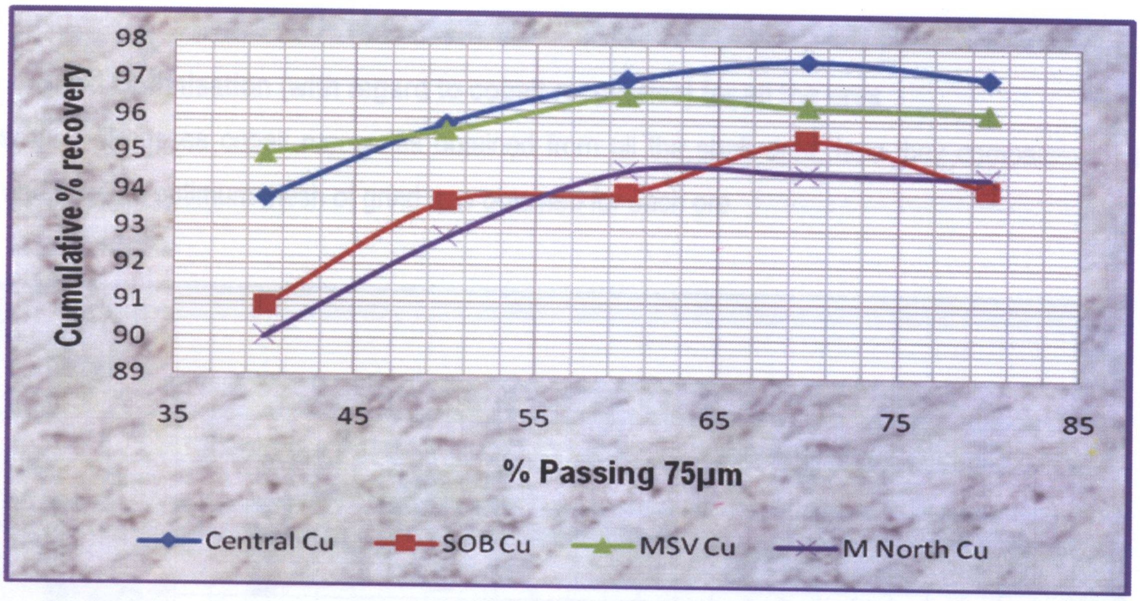

Fig 4.2.1: cumulative % recovery against % passing 75µm

Fig 4.2.1 shows that as grind size continued decreasing, cumulative percent recoveries increased. At a certain particle size, the recoveries began to drop with decreasing particle size. At coarser particle sizes, some mineral particles are too heavy to be lifted by the air bubbles while at finer particle sizes a lot of gangue materials end up being floated due to their lightness. This explains the trends shown by the plotted curves.

It can also be deduced that the copper recoveries for central and Mindola Subvertical ores were higher than for SOB and Mindola North. The highest recoveries were obtained from central shaft ore.

The maximum recovery obtained for Central shaft ore was 97.6%. This was at a grind size of 70 percent passing 75µm which is taken as the optimum mesh of grind (MOG). From Fig 4.1.1, the time taken to grind to this optimum mesh of grind was read as 11 minutes 30 seconds. For SOB ore, the maximum recovery was 95.4 % at an optimum mesh of grind of 70 % passing 75µm. Time taken to grind to this size was 12 minutes (Fig 4.1.1). Moreover, the maximum recovery for Mindola Subvertical ore was 96.6 % which was at the optimum mesh of grind of 60 % passing 75μm. The time taken to grind to this size was 12 minutes 30 seconds. Furthermore, the maximum recovery for Mindola North ore was 94.6 % and was at the optimum mesh of grind of 60 % passing 75μm. The time taken to grind to this size was 11 minutes.

4.3 Grind Optimisation (with regard to cobalt recoveries) for all the ores

This section discusses cobalt recoveries obtained from all the shaft ores, how they compare with each other and how the optimum mesh of grind was obtained for each ore.

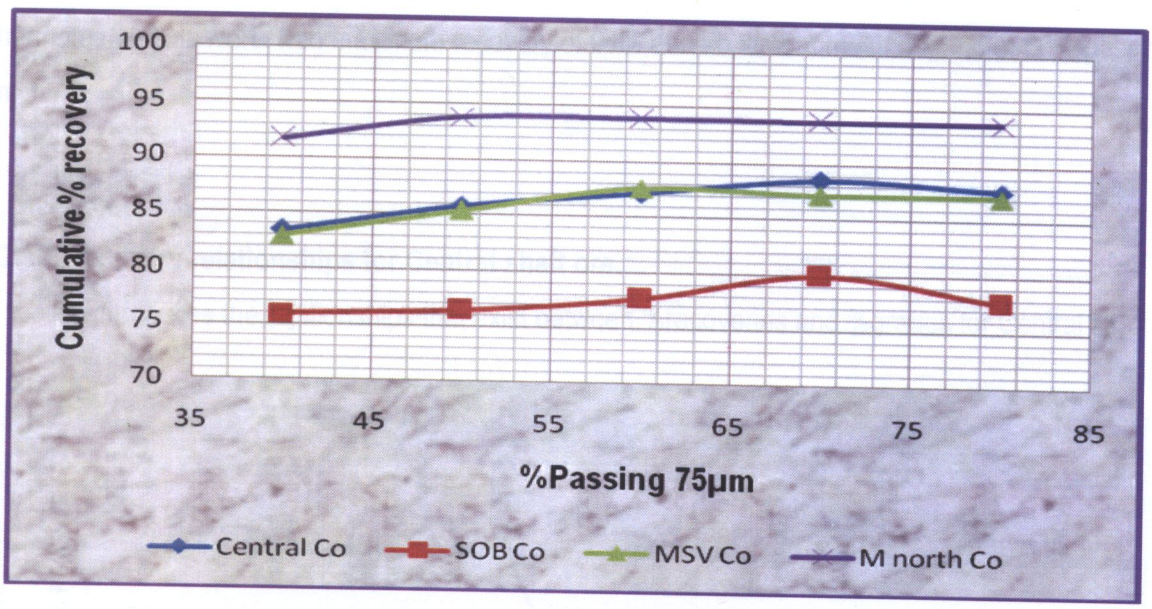

Fig 4.3.1: Cumulative % recovery against % passing 75μm

Fig 4.3.1 shows that as grind size decreased continuously percent recoveries kept increasing until a maximum recovery is reached. After this point, recoveries started decreasing. As explained earlier (Section 4.2), the reason for this was that at coarser particle sizes the air bubbles are not able to lift some mineral particles due to their heavy weight. Finer particles do not easily settle causing fine gangue material to be floated with the concentrate. Hence, flotation efficiency is poor at these particle sizes leading to the trend shown.

It was also observed that Mindola North had the highest cobalt recoveries compared to all other shaft ores ranging from 90% to 94%. On the other hand, SOB shaft ores gave the lowest recoveries ranging from 75% to 80%. In addition, Mindola Subvertical and Central shaft had intermediate cobalt recoveries ranging from 83% to 87%.

The maximum recovery for Central shaft ore was 89%. This was at a mesh of grind of 70% passing 75µm and the time taken to grind to this size was 11 minutes 30 seconds. For SOB ore, the maximum recovery was 80% at an optimum mesh of grind of 70% passing 75µm. Time taken to grind to this size was read as 12 minutes. Moreover, Mindola Subvertical had maximum recovery of 87% at an optimum mesh of grind of 60% passing 75µm. The time taken to grind this ore to this mesh is was 12 minutes 30 seconds. Furthermore, Mindola North had maximum recovery of 94% at an optimum mesh of grind of 60% passing 75µm

4.4 Grade/recovery relationships for Central shaft ore.

This section introduces the relationships that exist between %recoveries and %grades for Central shaft ore

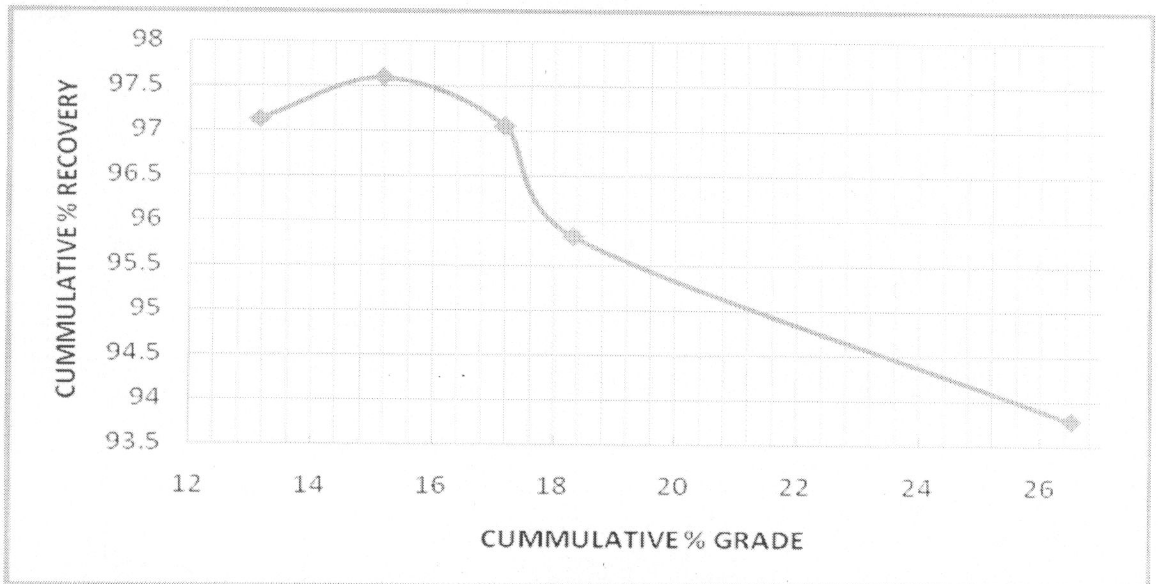

Figure 4.4.1: Cumulative % recovery against Cumulative % grade of copper

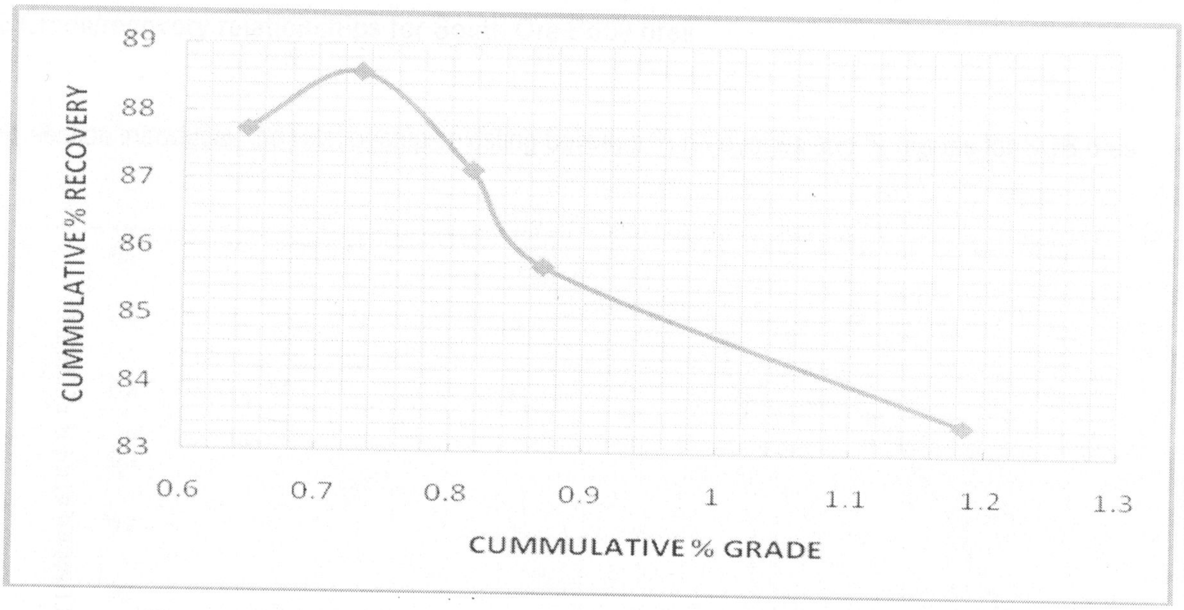

Figure 4.4.2: Cumulative % Recovery against Cumulative % grade of cobalt

Figures 4.4.1 and 4.4.2 show cumulative percent recovery against cumulative percent grade, the pattern displayed is that as the recovery is increased, the grade becomes lower. This means that the recovery and grade are inversely related. So the optimum recovery is a compromise between good grades and good recoveries.

4.5 Grade/recovery relationships for South Ore Body ores

This section introduces the relationships existing between % recoveries and % grades for SOB ores

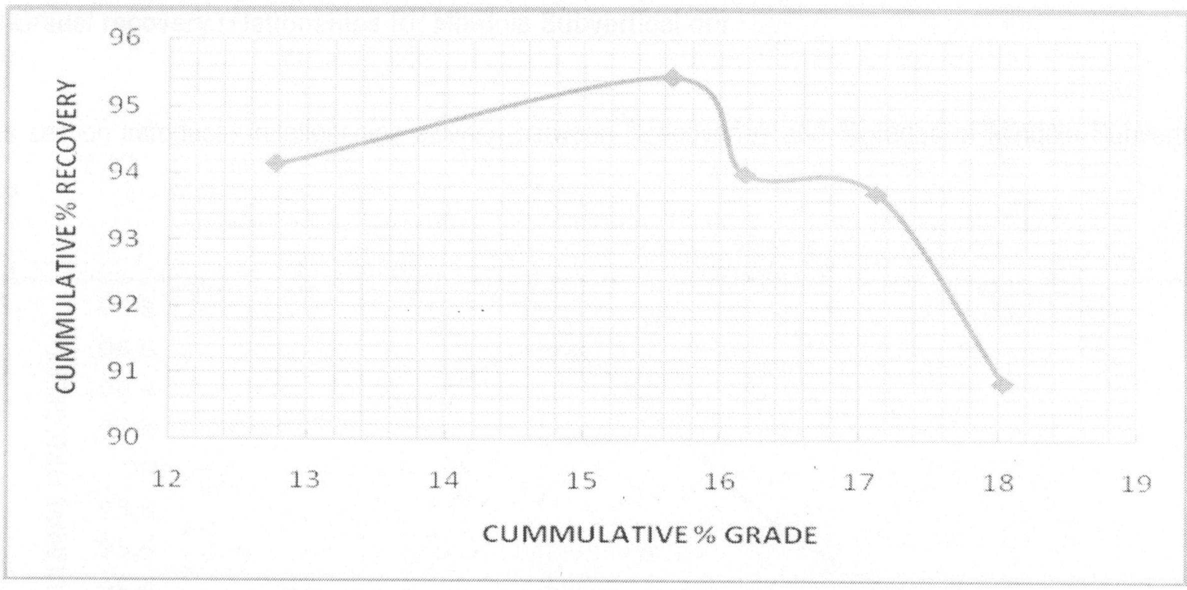

Figure 4.5.1: Cumulative percent recovery against cumulative percent grade for copper.

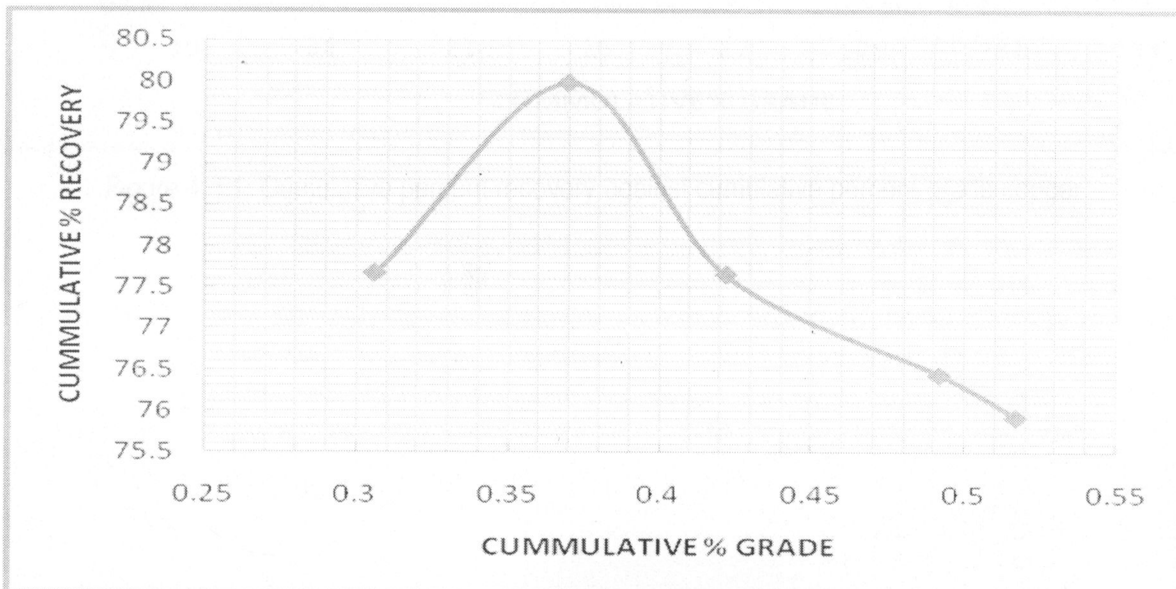

Figure 4.5.2: Cumulative Percent Recovery against Cumulative Percent Grade for cobalt

Figures 4.5.1 and 4.5.2 show cumulative percent recovery against cumulative percent grade. They display the same curves as for central shaft ore. As the recoveries increase, the grades decrease. Both curves in this case are not smooth due to some errors which could be systematic or random.

4.6 Grade/ recovery relationships for Mindola Subvertical ore

This section introduces relationships existing between %recoveries and %grades of Mindola Subvertical ores.

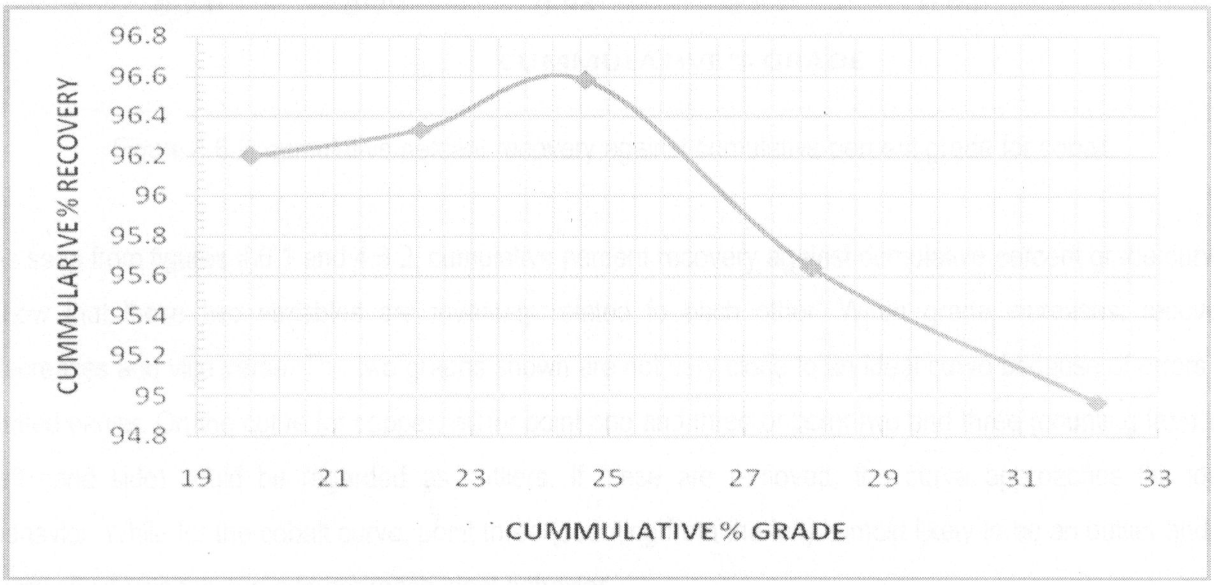

Figure 4.6.1: Cumulative percent recovery against cumulative percent grade copper

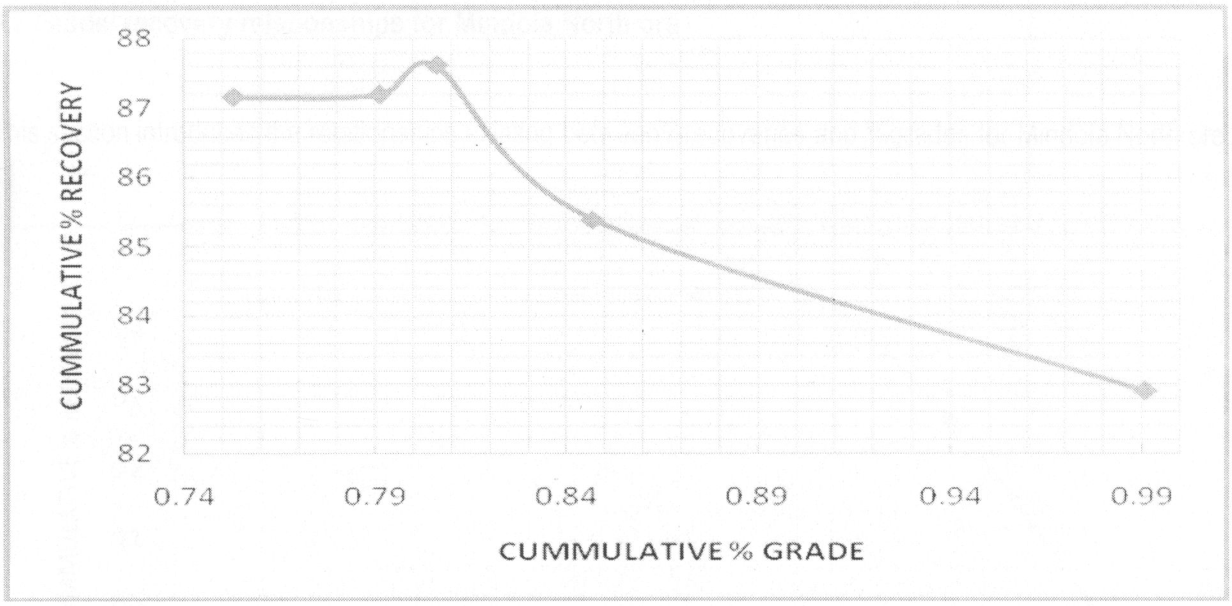

Figure 4.6.2: cumulative percent recovery against cumulative percent grade for cobalt.

As seen from figures 4.6.1 and 4.6.2, cumulative percent recovery against cumulative percent grade curves show that these two variables are inversely related to each other. When grade increases, recovery decreases and vice versa. The two graphs shown are not very close to an ideal curve because of errors as stated earlier. On the curve for copper, either point one and three or point two and three (counting from the left hand side) could be regarded as outliers. If these are removed, the curve approaches the ideal behavior. While for the cobalt curve, point three (counting from the left) is most likely to be an outlier and its removal causes the curve to approach ideal behavior.

4.7 Grade/ recovery relationships for Mindola North ore

This section introduces the relationships existing between%recoveries and %grades for Mindola North ore.

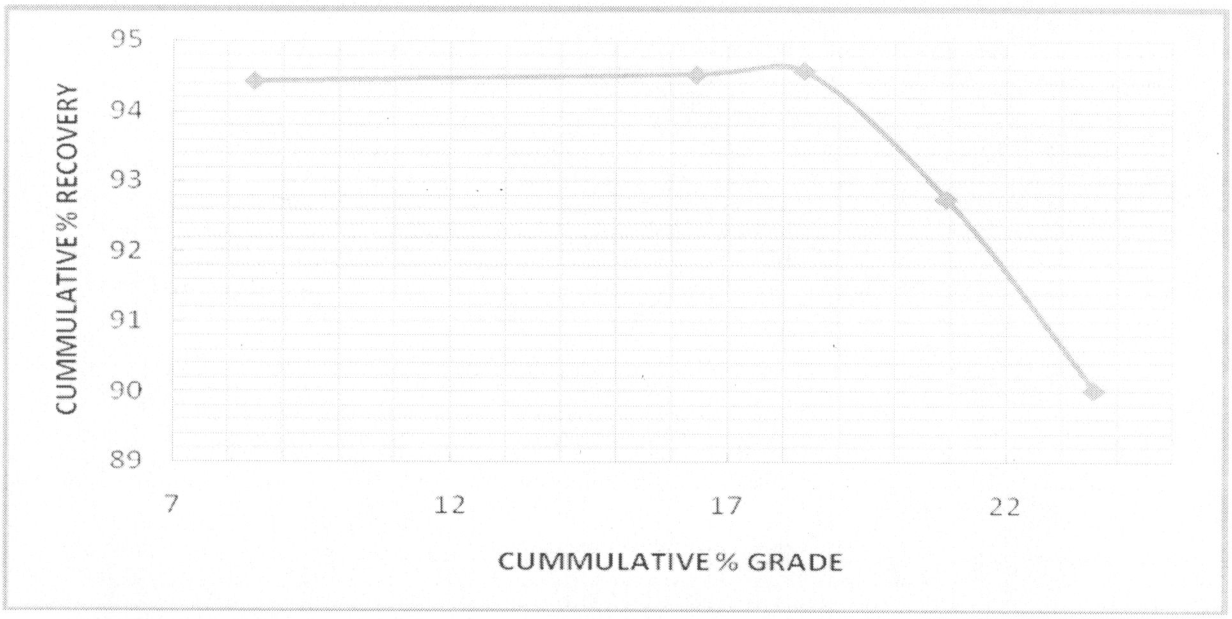

Figure 4.7.1: Cumulative percent recovery against cumulative percent grade graph for copper.

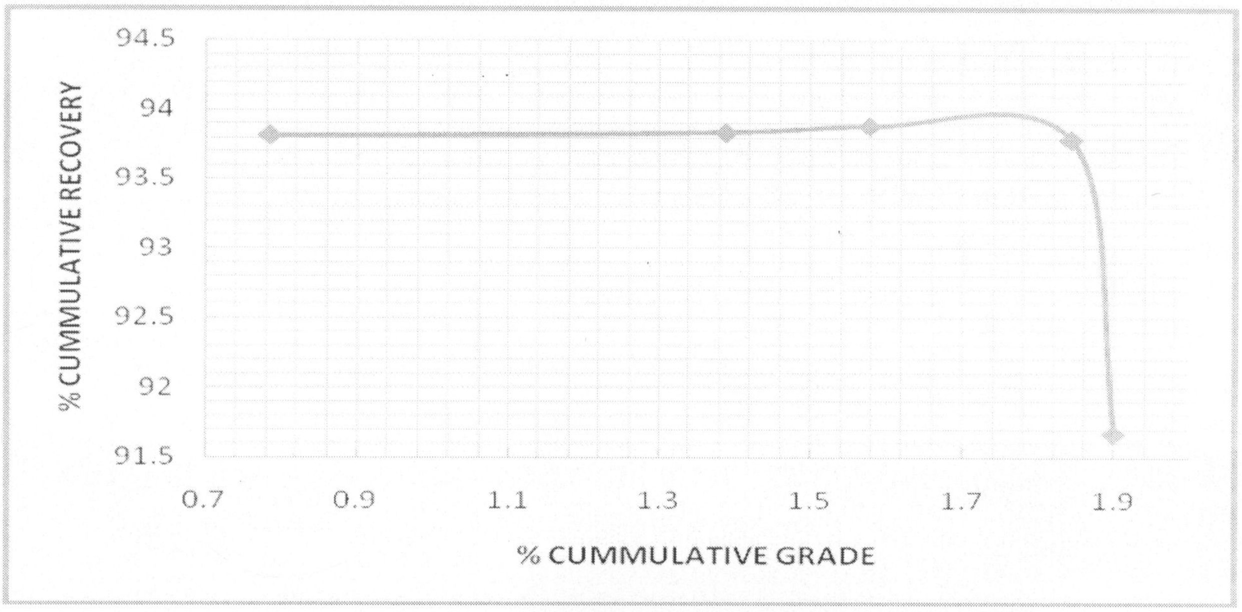

Figure 4.7.2: Cumulative percent recovery against cumulative percent grade

As seen from Fig 4.7.1 and 4.7.2, the same behavior as seen from previous cumulative percent recovery against cumulative percent grade curves are shown. Increasing cumulative recoveries causes a decrease in the cumulative grades of the ores. These two curves are smoother than the others and show more ideal behavior than the previous ones.

CHAPTER FIVE

5.0 CONCLUSIONS AND RECOMMENDATIONS

From the release analysis results obtained, it can be concluded that the optimum meshes of grind and optimum recoveries for the different ores were as follows:

1 For Central shaft ore: Optimum cumulative recoveries were 97.6 and 88.2 percent for copper and cobalt respectively. The optimum mesh of grind was 70 percent passing 75µm for both copper and cobalt. The time taken to grind to this size was approximately 11 minutes.

2 For South Ore Body ore: Optimum cumulative recoveries were 95 and 80 percent for copper and cobalt respectively. Optimum mesh of grind was 70 percent passing 75µm for both metals. The time taken to grind to this size was 12 minutes

3 For Mindola Subvertical ore: Optimum cumulative recoveries were 96.4 and 87.6 percent copper and cobalt respectively. The optimum mesh of grind was 60 percent passing 75µm for both metals. The time taken to grind to this size was 12 minutes 30 seconds.

4 Mindola North ore: Optimum cumulative recoveries were 94.6 and 93.8 for copper and cobalt respectively. Optimum mesh of grind was 60 percent passing 75µm for both metals. The time taken to grind to this size was 11 minutes.

It was also observed that the ore hardness for all the ores had changed. Previously, South Ore Body ores took about 7 minutes to grind to 60% passing 75µm (Mwansa, 2007), while Mindola North mixed with Mindola Subvertical took about 4.6 minutes to grind to the same size distribution (Mwansa, 2007). From this investigation; South Ore Body took 8 minutes, Mindola Subvertical took 12 minutes and Mindola North took 11 minutes to grind to 60 percent passing 75µm.

Therefore the low recoveries experienced were due to the changes in the optimum meshes of grind and ore hardnesses for all the ores as established in this investigation.

The following are the recommendations:

- Mindola North and Subvertical ores should be ground separately as they have different grind times to grind to their optimum sizes.
- Central shaft ore should be ground to 70 percent passing 75µm which is done at 11 minutes grinding time.
- South Ore Body ore should be ground to 70 percent passing 75µm at a grinding time of 12 minutes.
- Mindola Subvertical ore should be ground to 60 percent passing 75µm at a grinding time of 12 minutes 30 seconds.
- Mindola North ore should be ground to 60 percent passing 75µm at a grind time of 11 minutes.

REFERENCES

1. Arbiter, N. (1985), Flotation, In Weiss, N. L.(Ed), SME Mineral Processing Handbook, American Institute of Mining, Metallurgical and Petroleum Engineers Inc. New York, USA.
2. Day, A. (2002), Mining Chemicals Handbook, Revised Edition, Cytec Mining Inc. USA.
3. Jain s. K (1987), Mineral Processing, 2nd Edition, National Book Trust, India.
4. Mulenshi, M. J. (2003), Optimisation of Grinding And Flotation Conditions On Nchanga Open Pit Cobalt Ore, B. Min. Sc. Thesis, University of Zambia, Lusaka, Zambia.
5. Mutentwa, V. (2001), Release Analysis of Nkana Ores to Determine their Optimum Mesh of Grind, B. Min. Sc. Thesis, University Of Zambia, Lusaka, Zambia.
6. Mwansa, D. M.(2007), Optimisation of Bulk Flotation Parameters on Mindola, South Ore Body (SOB) Ores and when the two are Blended in ratio 2:1, B. Min. Sc. Thesis, University of Zambia, Lusaka, Zambia.
7. Perry (1884) Chemical Engineer's handbook, 5th Edition.
8. Prasher C. L. (1987), Crushing and Grinding Process Handbook, John Wiley and Sons Limited, Britain.
9. Svenska, G. and Jernkontoret (1958), Progress in Mineral Dressing, Transactions of Mineral Dressing Congress, Stockholm.
10. Wills B. A. (1987), Mineral Processing Technology, 4th Edition, Pe

APPENDICES

Appendix 1

Miscellaneous Formulae and Sample Calculations:

$$\text{Recovery} = \frac{\text{concentrate weight} \times \text{percent concentrate grade}}{\text{head weight} \times \text{percent head grade}}$$

$$\text{Head grade} = \frac{(\textit{Concentrate wt\%} \times \textit{concentrate grade\%}) + (\textit{tailings wt\%} \times \textit{tailings grade\%})}{100}$$

Calculation of first recovery for Central shaft ore, 40 percent passing 75µm particle size distribution

$$\text{Percent recovery} = \frac{60 \times 23.07 \times 100}{(60 \times 23.07) + (108 \times 3.26) + (640 \times 0.18)} = 74.56$$

Appendix 2: Screen Analysis Results for All Mopani Copper Mine Shaft Ores

Central Shaft Ores

0 mins

SIEVE SIZE (Microns)	WT(g)		%WT	CUMULATIVE % RETAINED	CUMULATIVE % PASSING
212	171		57.00	57.00	43.00
150	15		5.00	62.00	38.00
106	17		5.67	67.67	32.33
75	15		5.00	72.67	**27.33**
-75	82		27.33	100.00	0.00
TOTAL	**300**		**100.00**		

4 min

SIEVE SIZE (Microns)	WT(g)		%WT	CUMULATIVE % RETAINED	CUMULATIVE % PASSING
212	55		18.33	18.33	81.67
150	36		12.00	30.33	69.67
106	38		12.67	43.00	57.00
75	36		12.00	55.00	**45.00**
-75	135		45.00	100.00	0.00
TOTAL	**300**		**100.00**		

8 min

SIEVE SIZE (Microns)	WT(g)		%WT	CUMULATIVE % RETAINED	CUMULATIVE % PASSING
212	29		9.67	9.67	90.33
150	22		7.33	17.00	83.00
106	44		14.67	31.67	68.33
75	22		7.33	39.00	**61.00**
-75	183		61.00	100.00	0.00
TOTAL	**300**		**100.00**		

12 min

SIEVE SIZE (Microns)	WT(g)		%WT	CUMULATIVE % RETAINED	CUMULATIVE % PASSING
212	10		3.33	3.33	96.67
150	18		6.00	9.33	90.67
106	26		8.67	18.00	82.00
75	32		10.67	28.67	**71.33**
-75	214		71.33	100.00	0.00
TOTAL	**300**		**100.00**		

16 min

SIEVE SIZE (Microns)	WT(g)		%WT	CUMULATIVE % RETAINED	CUMULATIVE % PASSING
212	6		2.00	2.00	98.00
150	8		2.67	4.67	95.33
106	24		8.00	12.67	87.33
75	31		10.33	23.00	**77.00**
-75	231		77.00	100.00	0.00
TOTAL	**300**		**100.00**		

20 min

SIEVE SIZE (Microns)	WT(g)		%WT	CUMULATIVE % RETAINED	CUMULATIVE % PASSING
212	4		1.33	1.33	98.67
	9		3.00	4.33	95.67
106	11		3.67	8.00	92.00
75	33		11.00	19.00	**81.00**
-75	243		81.00	100.00	0.00
TOTAL	**300**		**100.00**		

30 mins

SIEVE SIZE (Microns)	WT(g)		%WT	CUMULATIVE % RETAINED	CUMULATIVE % PASSING
212	1		0.33	0.33	99.67
150	5		1.67	2.00	98.00
106	15		5.00	7.00	93.00
75	28		9.33	16.33	**83.67**
-75	251		83.67	100.00	0.00
TOTAL	**300**		**100.00**		

South Ore Body Shaft Ores

0 mins

SIEVE SIZE (Microns)	WT(g)			CUMULATIVE % RETAINED	CUMULATIVE % PASSING
212	172		57.33	57.33	42.67
150	16		5.33	62.67	37.33
106	19		6.33	69.00	31.00
75	12		4.00	73.00	**27.00**
-75	81		27.00	100.00	0.00
TOTAL	**300**		**100.00**		

4 mins

SIEVE SIZE (Microns)	WT(g)		%WT	CUMULATIVE % RETAINED	CUMULATIVE % PASSING
212	73		24.33	24.33	75.67
150	30		10.00	34.33	65.67
106	30		10.00	44.33	55.67
75	26		8.67	53.00	**47.00**
-75	141		47.00	100.00	0.00
TOTAL	**300**		**100.00**		

8 mins

SIEVE SIZE (Microns)	WT(g)		%WT	CUMULATIVE % RETAINED	CUMULATIVE % PASSING
212	34		11.33	11.33	88.67
150	24		8.00	19.33	80.67
106	28		9.33	28.67	71.33
75	34		11.33	40.00	**60.00**
-75	180		60.00	100.00	0.00
TOTAL	**300**		**100.00**		

12 mins

SIEVE SIZE (Microns)	WT(g)		%WT	CUMULATIVE % RETAINED	CUMULATIVE % PASSING
212	12		4.00	4.00	96.00
150	22		7.33	11.33	88.67
106	32		10.67	22.00	78.00
75	24		8.00	30.00	**70.00**
-75	210		70.00	100.00	0.00
TOTAL	**3OO**		**100**		

16 mins

SIEVE SIZE (Microns)	WT(g)		%WT	CUMULATIVE % RETAINED	CUMULATIVE % PASSING
212	8		2.67	2.67	97.33
150	16		5.33	8.00	92.00
106	26		8.67	16.67	83.33
75	16		5.33	22.00	**78.00**
-75	234		78.00	100.00	0.00
TOTAL	**300**		**100.00**		

20 mins

SIEVE SIZE (Microns)	WT(g)		%WT	CUMULATIVE % RETAINED	CUMULATIVE % PASSING
212	4		1.33	1.33	98.67
	8		2.67	4.00	96.00
106	18		6.00	10.00	90.00
75	21		7.00	17.00	**83.00**
-75	249		83.00	100.00	0.00
TOTAL	**300**		**100.00**		

30 mins

SIEVE SIZE (Microns)	WT(g)		%WT	CUMULATIVE % RETAINED	CUMULATIVE % PASSING
212	0		0.00	0.00	100.00
150	6		2.00	2.00	98.00
106	13		4.33	6.33	93.67
75	23		7.67	14.00	**86.00**
-75	258		86.00	100.00	0.00
TOTAL	**300**		**100.00**		

Mindola Subvertical Ores

0 mins

SIEVE SIZE (Microns)	WT(g)	%WT	CUMULATIVE % RETAINED	CUMULATIVE % PASSING
212	184	61.33	61.33	38.67
150	14	4.67	66.00	34.00
106	15	5.00	71.00	29.00
75	27	9.00	80.00	**20.00**
-75	60	20.00	100.00	0.00
TOTAL	**300**	**100.00**		

4 mins

SIEVE SIZE (Microns)	WT(g)	%WT	CUMULATIVE % RETAINED	CUMULATIVE % PASSING
212	62	20.67	20.67	79.33
150	62	20.67	41.33	58.67
106	50	16.67	58.00	42.00
75	27	9.00	67.00	**33.00**
-75	99	33.00	100.00	0.00
TOTAL	**300**	**100.00**		

8 mins

SIEVE SIZE (Microns)	WT(g)	%WT	CUMULATIVE % RETAINED	CUMULATIVE % PASSING
212	66	22.00	22.00	78.00
150	24	8.00	30.00	70.00
106	40	13.33	43.33	56.67
75	32	10.67	54.00	**46.00**
-75	138	46.00	100.00	0.00
TOTAL	**300**	**100.00**		

12 mins

SIEVE SIZE (Microns)	WT(g)	%WT	CUMULATIVE % RETAINED	CUMULATIVE % PASSING
212	14	4.67	4.67	95.33
150	24	8.00	12.67	87.33
106	40	13.33	26.00	74.00
75	46	15.33	41.33	**58.67**
-75	176	58.67	100.00	0.00
TOTAL	**300**	**100.00**		

16 min

SIEVE SIZE (Microns)	WT(g)	%WT	CUMULATIVE % RETAINED	CUMULATIVE % PASSING
212	2	0.67	0.67	99.33
150	13	4.33	5.00	95.00
106	43	14.33	19.33	80.67
75	41	13.67	33.00	**67.00**
-75	201	67.00	100.00	0.00
TOTAL	**300**	**100.00**		

20 min

SIEVE SIZE (Microns)	WT(g)	%WT	CUMULATIVE % RETAINED	CUMULATIVE % PASSING
212	4	1.33	1.33	98.67
150	12	4.00	5.33	94.67
106	29	9.67	15.00	85.00
75	39	13.00	28.00	**72.00**
-75	216	72.00	100.00	0.00
TOTAL	**300**	**100.00**		

30 mins

SIEVE SIZE (Microns)	WT(g)	%WT	CUMULATIVE % RETAINED	CUMULATIVE % PASSING
212	1	0.33	0.33	99.67
150	4	1.33	1.67	98.33
106	34	11.33	13.00	87.00
75	33	11.00	24.00	**76.00**
-75	228	76.00	100.00	0.00
TOTAL	**300**	**100.00**		

Mindola North Ores

0 mins

SIEVE SIZE (Microns)	WT(g)	%WT	CUMULATIVE % RETAINED	CUMULATIVE % PASSING
212	198	66.00	66.00	34.00
150	12	4.00	70.00	30.00
106	11	3.67	73.67	26.33
75	12	4.00	77.67	**22.33**
-75	67	22.33	100.00	0.00
TOTAL	**300**	**100.00**		

4mins

SIEVE SIZE (Microns)	WT(g)	%WT	CUMULATIVE % RETAINED	CUMULATIVE % PASSING
212	90	30.00	30.00	70.00
150	30	10.00	40.00	60.00
106	30	10.00	50.00	50.00
75	30	10.00	60.00	**40.00**
-75	120	40.00	100.00	0.00
TOTAL	**300**	**100.00**		

8 min

SIEVE SIZE (Microns)	WT(g)	%WT	CUMULATIVE % RETAINED	CUMULATIVE % PASSING
212	23	7.67	7.67	92.33
150	26	8.67	16.33	83.67
106	36	12.00	28.33	71.67
75	56	18.67	47.00	**53.00**
-75	159	53.00	100.00	0.00
TOTAL	**300**	**100.00**		

12 mins

SIEVE SIZE (Microns)	WT(g)	%WT	CUMULATIVE % RETAINED	CUMULATIVE % PASSING
212	20	6.67	6.67	93.33
150	22	7.33	14.00	86.00
106	32	10.67	24.67	75.33
75	40	13.33	38.00	**62.00**
-75	186	62.00	100.00	0.00
TOTAL	**300**	**100.00**		

16 mins

SIEVE SIZE (Microns)	WT(g)	%WT	CUMULATIVE % RETAINED	CUMULATIVE % PASSING
212	7	2.33	2.33	97.67
150	17	5.67	8.00	92.00
106	27	9.00	17.00	83.00
75	42	14.00	31.00	**69.00**
-75	207	69.00	100.00	0.00
TOTAL	**300**	**100.00**		

20 mins

SIEVE SIZE (Microns)	WT(g)	%WT	CUMULATIVE % RETAINED	CUMULATIVE % PASSING
212	6	2.00	2.00	98.00
150	14	4.67	6.67	93.33
106	26	8.67	15.33	84.67
75	34	11.33	26.67	**73.33**
-75	220	73.33	100.00	0.00
TOTAL	**300**	**100.00**		

30 mins

SIEVE SIZE (Microns)	WT(g)	%WT	CUMULATIVE % RETAINED	CUMULATIVE % PASSING
212	1	0.33	0.33	99.67
150	6	2.00	2.33	97.67
106	22	7.33	9.67	90.33
75	43	14.33	24.00	**76.00**
-75	228	76.00	100.00	0.00
TOTAL	**300**	**100.00**		

Flotation Results Tables For All Shaft Ores Analyzed

Central Shaft Ores

40% passing 75µm

FRACTION	WT(g)	%Cu	%Co	METAL CONT		% RECOVERY		% CUM REC	
				Cu(g)	Co(g)	Cu	Co	Cu	Co
CONC 1	60	23.07	0.931	13.84	0.56	74.76	57.83	74.76	57.83
CONC 2	108	3.26	0.229	3.52	0.25	19.02	25.60	93.78	83.44
TAILS	640	0.18	0.025	1.15	0.16	6.22	16.56	100.00	100.00
Head Calc.	808	2.29	0.120	18.51	0.97	100.00	100.00		
Head Assay		2.05	0.140						

50% passing 75µm

FRACTION	WT(g)	%Cu	%Co	METAL CONT		% RECOVERY		% CUM REC	
				Cu(g)	Co(g)	Cu	Co	Cu	Co
CONC 1	106	16.80	0.730	17.81	0.77	90.11	76.35	90.11	76.35
CONC 2	80	1.41	0.119	1.13	0.10	5.71	9.39	95.82	85.74
TAILS	688	0.12	0.021	0.83	0.14	4.18	14.26	100.00	100.00
Head Calc.	874	2.26	0.116	19.76	1.01	100.00	100.00		
Head Assay		2.05	0.140						

60% passing 75µm

FRACTION	WT(g)	%Cu	%Co	METAL CONT		% RECOVERY		% CUM REC	
				Cu(g)	Co(g)	Cu	Co	Cu	Co
Cleaner Cons	114	16.39	0.710	18.68	0.81	94.09	80.01	94.09	80.01
Cleaner Tails	84	0.70	0.086	0.59	0.07	2.96	7.14	97.05	87.15
Final Tails	650	0.09	0.020	0.59	0.13	2.95	12.85	100.00	100.00
Head Calc.	848	2.34	0.119	19.86	1.01	100.00	100.00		
Head Assay		2.05	0.140						

70% passing 75µm

FRACTION	WT(g)	%Cu	%Co	METAL CONT		% RECOVERY		% CUM REC	
				Cu(g)	Co(g)	Cu	Co	Cu	Co
CONC 1	122	15.03	0.666	18.34	0.81	97.20	82.87	97.20	82.87
CONC 2	112	0.07	0.050	0.07	0.06	0.39	5.71	97.59	88.58
TAILS	700	0.07	0.016	0.46	0.11	2.41	11.42	100.00	100.00
Head Calc.	934	2.02	0.105	18.86	0.98	100.00	100.00		
Head Assay		2.15	0.100						

80% passing 75µm

FRACTION	WT(g)	%Cu	%Co	METAL CONT		% RECOVERY		% CUM REC	
				Cu(g)	Co(g)	Cu	Co	Cu	Co
CONC 1	148	12.00	0.552	17.76	0.82	91.09	79.50	91.09	79.50
CONC 2	110	1.07	0.077	1.18	0.08	6.04	8.24	97.13	87.74
TAILS	700	0.08	0.018	0.56	0.13	2.87	12.26	100.00	100.00
Head Calc.	958	2.04	0.107	19.50	1.03	100.00	100.00		
Head Assay		2.15	0.100						

South Ore Body Shaft Ore

40% passing75µm

FRACTION	WT(g)	%Cu	%Co	METAL CONT		% RECOVERY		% CUM REC	
				Cu(g)	Co(g)	Cu	Co	Cu	Co
CONC 1	104	13.73	0.477	14.28	0.50	74.40	73.95	74.40	73.95
CONC 2	78	4.05	0.017	3.16	0.01	16.46	1.98	90.86	75.93
TAILS	702	0.25	0.023	1.76	0.16	9.14	24.07	100.00	100.00
Head Calc.	884	2.17	0.076	19.19	0.67	100.00	100.00		
Head Assay		1.89	0.078						

50% passing 75µm

FRACTION	WT(g)	%Cu	%Co	METAL CONT		% RECOVERY		% CUM REC	
				Cu(g)	Co(g)	Cu	Co	Cu	Co
CONC 1	90	12.52	0.289	11.27	0.26	77.68	63.66	77.68	63.66
CONC 2	52	4.47	0.110	2.32	0.06	16.02	14.00	93.71	77.66
TAILS	702	0.13	0.013	0.91	0.09	6.29	22.34	100.00	100.00
Head Calc.	844	1.72	0.048	14.51	0.41	100.00	100.00		
Head Assay		1.89	0.078						

60% passing 75µm

FRACTION	WT(g)	%Cu	%Co	METAL CONT		% RECOVERY		% CUM REC	
				Cu(g)	Co(g)	Cu	Co	Cu	Co
CONC 1	84	13.22	0.359	11.10	0.30	73.47	54.46	73.47	54.46
CONC 2	110	2.82	0.115	3.10	0.13	20.52	22.85	94.00	77.31
TAILS	698	0.13	0.018	0.91	0.13	6.00	22.69	100.00	100.00
Head Calc.	892	1.69	0.062	15.11	0.55	100.00	100.00		
Head Assay		3.60	0.240						

70% passing 75µm

FRACTION	WT(g)	%Cu	%Co	METAL CONT		% RECOVERY		% CUM REC	
				Cu(g)	Co(g)	Cu	Co	Cu	Co
CONC 1	98	11.92	0.290	11.68	0.28	66.56	60.13	66.56	60.13
CONC 2	140	3.62	0.067	5.07	0.09	28.88	19.85	95.44	79.98
TAILS	728	0.11	0.013	0.80	0.09	4.56	20.02	100.00	100.00
Head Calc.	966	1.82	0.049	17.55	0.47	100.00	100.00		
Head Assay		3.60	0.240						

80% passing 75µm

FRACTION	WT(g)	%Cu	%Co	METAL CONT		% RECOVERY		% CUM REC	
				Cu(g)	Co(g)	Cu	Co	Cu	Co
CONC 1	76	11.18	0.230	8.50	0.17	82.79	59.97	82.79	59.97
CONC 2	80	1.51	0.065	1.21	0.05	11.77	17.70	94.56	77.67
TAILS	620	0.09	0.011	0.56	0.07	5.44	22.33	100.00	100.00
Head Calc.	776	1.32	0.038	10.26	0.29	100.00	100.00		
Head Assay		2.05	0.140						

Mindola Subvertical Ores

40% passing 75µm

FRACTION	WT(g)	%Cu	%Co	METAL CONT		% RECOVERY		% CUM REC	
				Cu(g)	Co(g)	Cu	Co	Cu	Co
CONC 1	67	29.50	0.890	19.77	0.60	86.02	74.32	86.02	74.32
CONC 2	84	2.45	0.082	2.06	0.07	8.96	8.58	94.97	82.90
TAILS	722	0.16	0.019	1.16	0.14	5.03	17.10	100.00	100.00
Head Calc.	873	2.63	0.092	22.98	0.80	100.00	100.00		
Head Assay		2.15	0.100						

50% passing 75µm

FRACTION	WT(g)	%Cu	%Co	METAL CONT		% RECOVERY		% CUM REC	
				Cu(g)	Co(g)	Cu	Co	Cu	Co
CONC 1	101	24.99	0.750	25.24	0.76	87.08	78.29	87.08	78.29
CONC 2	88	2.82	0.078	2.48	0.07	8.56	7.09	95.64	85.39
TAILS	744	0.17	0.019	1.26	0.14	4.36	14.61	100.00	100.00
Head Calc.	933	3.11	0.104	28.99	0.97	100.00	100.00		
Head Assay		2.15	0.100						

60% passing 75µm

FRACTION	WT(g)	%Cu	%Co	METAL CONT		% RECOVERY		% CUM REC	
				Cu(g)	Co(g)	Cu	Co	Cu	Co
CONC 1	113	22.77	0.745	25.73	0.84	91.58	84.18	91.58	84.18
CONC 2	78	1.78	0.044	1.39	0.03	4.94	3.43	96.52	87.61
TAILS	751	0.13	0.017	0.98	0.12	3.48	12.39	100.00	100.00
Head Calc.	942	2.98	0.106	28.09	1.00	100.00	100.00		
Head Assay		1.89	0.078						

70% passing 75µm

FRACTION	WT(g)	%Cu	%Co	METAL CONT		% RECOVERY		% CUM REC	
				Cu(g)	Co(g)	Cu	Co	Cu	Co
CONC 1	97	20.48	0.702	19.87	0.68	89.96	79.95	89.96	79.95
CONC 2	85	1.66	0.073	1.41	0.06	6.37	7.24	96.33	87.19
TAILS	704	0.12	0.016	0.81	0.11	3.67	12.81	100.00	100.00
Head Calc.	886	2.49	0.096	22.08	0.85	100.00	100.00		
Head Assay		1.89	0.078						

76% passing 75µm

FRACTION	WT(g)	%Cu	%Co	METAL CONT		% RECOVERY		% CUM REC	
				Cu(g)	Co(g)	Cu	Co	Cu	Co
CONC 1	104	17.22	0.660	17.91	0.69	84.50	78.16	84.50	78.16
CONC 2	100	2.48	0.079	2.48	0.08	11.70	9.00	96.20	87.15
TAILS	806	0.10	0.014	0.81	0.11	3.80	12.85	100.00	100.00
Head Calc.	1010	2.10	0.087	21.19	0.88	100.00	100.00		
Head Assay		3.60	0.240						

Mindola North Ores

40% passing 75µm

FRACTION	WT(g)	%Cu	%Co	METAL CONT		% RECOVERY		% CUM REC	
				Cu(g)	Co(g)	Cu	Co	Cu	Co
CONC 1	80	17.59	1.670	14.07	1.34	68.75	81.80	68.75	81.80
CONC 2	76	5.73	0.212	4.35	0.16	21.28	9.87	90.03	91.67
TAILS	756	0.27	0.018	2.04	0.14	9.97	8.33	100.00	100.00
Head Calc.	912	2.24	0.179	20.47	1.63	100.00	100.00		
Head Assay		3.60	0.240						

50% passing 75µm

FRACTION	WT(g)	%Cu	%Co	METAL CONT		% RECOVERY		% CUM REC	
				Cu(g)	Co(g)	Cu	Co	Cu	Co
CONC 1	88	16.92	1.400	14.89	1.23	80.02	77.10	80.02	77.10
CONC 2	62	3.82	0.430	2.37	0.27	12.73	16.68	92.75	93.78
TAILS	710	0.19	0.014	1.35	0.10	7.25	6.22	100.00	100.00
Head Calc.	860	2.16	0.186	18.61	1.60	100.00	100.00		
Head Assay		2.05	0.140						

60% passing 75µm

FRACTION	WT(g)	%Cu	%Co	METAL CONT		% RECOVERY		% CUM REC	
				Cu(g)	Co(g)	Cu	Co	Cu	Co
CONC 1	118	15.92	1.440	18.79	1.70	87.88	89.87	87.88	89.87
CONC 2	62	2.31	0.122	1.43	0.08	6.70	4.00	94.58	93.88
TAILS	772	0.15	0.015	1.16	0.12	5.42	6.12	100.00	100.00
Head Calc.	952	2.25	0.199	21.38	1.89	100.00	100.00		
Head Assay		2.05	0.140						

70% passing 75μm

FRACTION	WT(g)	%Cu	%Co	METAL CONT		% RECOVERY		%CUM REC	
				Cu(g)	Co(g)	Cu	Co	Cu	Co
CONC 1	122	14.08	1.144	17.18	1.40	87.54	85.14	87.54	85.14
CONC 2	62	2.21	0.230	1.37	0.14	6.98	8.70	94.52	93.83
TAILS	722	0.15	0.014	1.08	0.10	5.48	6.17	100.00	100.00
Head Calc.	906	2.17	0.181	19.62	1.64	100.00	100.00		
Head Assay		2.05	0.140						

76% passing 75μm

FRACTION	WT(g)	%Cu	%Co	METAL CONT		% RECOVERY		%CUM REC	
				Cu(g)	Co(g)	Cu	Co	Cu	Co
CONC 1	162	7.69	0.604	12.46	0.98	88.62	77.40	88.62	77.40
CONC 2	122	0.67	0.170	0.82	0.21	5.81	16.41	94.43	93.81
TAILS	652	0.12	0.012	0.78	0.08	5.57	6.19	100.00	100.00
Head Calc.	936	1.50	0.135	14.06	1.26	100.00	100.00		
Head Assay		2.05	0.140						