

https://dspace.org/

Optimisation of reagent addition during flotation of a nickel sulphide ore at the Nkomati Mine concentrator

Kahn, Riyard

2017

http://hdl.handle.net/10539/22965

Kahn, Riyard (2016) Optimisation of reagent addition during flotation of a nickel sulphide ore at the Nkomati Mine concentrator, University of the Witwatersrand, Johannesburg, <http://hdl.handle.net/10539/22965>

Downloaded from a DSpace repository. https://dspace.org/



Optimisation of Reagent Addition during Flotation of a Nickel Sulphide ore at the Nkomati Mine Concentrator

Prepared by:

Riyard Kahn (783709)

Supervised by:

Prof Vusumuzi Sibanda

28 January 2017

A dissertation submitted to the faculty of Engineering and the Built Environment, University of Witwatersrand in fulfilment of the requirements for the degree of Master of Science in Engineering

Declaration

I declare that this thesis, submitted for the degree of Master of Science in Engineering at the University of the Witwatersrand, is my own work and has not been submitted before for any degree at this university or any other institution. I understand the meaning of plagiarism and declare that all the work in this document, except for that which is referenced, is my own.

Riyard Kahn

Synopsis

Batch scale laboratory testwork was conducted to evaluate collector and depressant addition on flotation performance of a nickel sulphide ore. The objectives of the study were to:

1. develop an understanding of the effects of collector and depressant dosage, and its interactive effects, on flotation performance and
2. determine the effect of stage dosing collector and depressant on flotation performance.

Testwork was conducted on the Nkomati Main Mineralized zone orebody, a nickel sulphide orebody in the Mpumulanga Province of South Africa consisting of pentlandite, chalcopyrite, pyrrhotite, pyrite and magnesium bearing silicates.

Characterisation testwork was conducted, including mineralogy on the major plant streams (by QEMSCAN) and a process survey. The results indicated that there was potential to increase the recovery of coarse pentlandite and that major nickel losses were observed in ultrafine pentlandite. Milling optimisation requires the minimisation of ultrafine generation while ensuring adequate liberation of the course nickel. Stage dosing of collector at nodal points (where more than one stream meets) is currently practiced on the plant, however, its effect had not yet been quantified on the plant or in the laboratory. Stage dosing of depressant is currently practiced on the cleaner flotation stage, however, this too has not been compared to upfront dosage on its own. Significant gangue depression was noted specifically for the cell at which stage dosing was done. The current study would provide an understanding of the current practices with the possibility of offering improvements.

The addition of collector progressively improved the hydrophobicity of the sulphide minerals and gangue (with particular emphasis on magnesium bearing gangue), improving recovery significantly. As a result of additional gangue recovery at the higher collector dosages, increased depressant dosages were required to maximise nickel recovery. The collector improved valuable mineral recovery, however, gangue recovery was increased simultaneously, albeit at a reduced rate or in reduced quantities. Furthermore, increased gangue entrainment was evident at higher collector dosages from the increase in water recovery. Excessive depressant addition destabilised the froth phase by the rejection of

froth stabilising gangue, which resulted in reduced recovery of the valuable minerals. Therefore, a careful balance must be maintained in order to maximise nickel recovery. Iron recovery was markedly increased at higher reagent dosages, indicative of increased pyrrhotite recovery. Pyrrhotite, although containing nickel, reduces the concentrate grade and may need to be depressed in the latter stages of flotation to ensure the final concentrate specification is achieved. This is an important observation as any improvement in nickel recovery in the roughing stages must be evaluated against the subsequent effect on the cleaning stages.

Stage dosing both collector and depressant, individually and collectively, proved to be beneficial by improving the nickel recovery. Stage dosing of both collector and depressant produced higher recoveries than stage dosing of the reagents individually. The time at which the reagent is dosed also proved to have an effect on the performance with an increased dosage in the latter stages providing the highest recovery. The typical recovery by size performance for flotation is characterised by low recovery of fines and coarse with an optimum recovery of an intermediate size fraction. Stage dosing ensures that fine particles are recovered with minimal reagent addition upfront, thereby, coarser particles can be effectively recovered once the high reagent consuming fines are removed. The results have indicated that stage dosing improved the recovery of both coarse and fine particles, whilst reducing the recovery of the intermediate size fraction.

Stage dosing can be implemented for two reasons:

1. maximising recovery
2. minimising reagent consumption to achieve the same recovery as upfront dosing

A financial evaluation should be conducted to quantify the optimum operating solution. Minimising reagent consumption could be beneficial under conditions of very low commodity prices and excessive reagent costs.

Nomenclature

Nodal point – a point where one or more process streams are combined

MMZ – Main Mineralised Zone

PCMZ – Peridotite Chromititic Mineralised Zone

PGMs – Platinum Group Metals

MgO – Magnesium Oxide; used as an indicator for magnesium bearing species

FAG – Fully Autogenous

QEMSCAN – Quantitative Evaluation of Minerals by Scanning Electron Microscopy

CMC – Carboxymethyl Cellulose

m^3 – cubic metres

XRF – X-Ray Flourescence

ICP – Inductively Coupled Plasma

AA – Atomic Absorption

μm – microns

Table of Contents

Declaration ..... **i**

Synopsis ..... **ii**

Nomenclature ..... **iv**

1. Chapter 1: Introduction **1**

1.1 Problem Statement and Research Objectives 2
1.2 Aim and objectives 3
1.3 Research questions 4

2. Chapter 2: Literature Review **5**

2.1 Principles of Flotation 5
2.2 Nickel Sulphide Production 9
2.2.1 General Flotation Characteristics 10
2.2.2 Reagents used in Nickel Sulphide Concentrators 10
2.3 Flotation Reagents 11
2.3.1 Collectors 11
2.3.2 Depressants 12
2.4 Interaction between Collector and Depressant 13
2.5 Entrainment 14
2.6 Recovery by Size and Distribution of Reagents down a Flotation Bank 15
2.7 Nkomati Mine 18
2.7.1 Orebody 18
2.7.2 Process 21

3. Chapter 3: Characterisation Testwork **24**

3.1 Introduction 24
3.2 Methodology 24
3.2.1 Mineralogical Testwork 24
3.2.2 Process Survey 26
3.3 Mineralogical Evaluation 27
3.3.1 Particle Size Distribution 27
3.3.2 Elemental Composition 28
3.3.3 Mineral Composition 31
3.4 Process Survey 36
3.4.1 Summary of Characterisation Testwork 42

4. Chapter 4: Interaction between Collector and Depressant **44**

4.1 Introduction 44
4.2 Experimental Procedure 44
4.2.1 Equipment and Procedure 44

4.2.2 Experimental Programme 49
4.3 Feed Analysis 50
4.4 Base case float 50
4.5 Effect of Collector – No Depressant 53
4.6 Effect of Depressant – No Collector 58
4.7 Interactive Effects of Collector and Depressant 61
4.8 Chapter 4 Conclusions 65
4.9 Chapter 4 Recommendations 66
5. Chapter 5: Stage Dosing 67
5.1 Introduction 67
5.2 Experimental Procedure 67
5.2.1 Equipment and Procedure 67
5.2.2 Experimental Programme 67
5.3 Assay by Size 69
5.4 Base Case Float – Upfront Dosing of Reagents 69
5.5 Stage Dosing Reagents 71
5.6 Chapter 5 Conclusions 80
5.7 Chapter 5 Recommendations 81
6. Study Conclusions 83
7. References 84
Appendix A – Detailed Mass Balance 86
Appendix B – Experimental Procedures 92
Appendix C – Chapter 4 Appendices 96
Appendix D – Chapter 5 Appendices 121

List of Figures

Figure 1-1: MMZ Flotation Process Block Flow 2
Figure 2-1: Schematic of Flotation Processes 7
Figure 2-2: Electrical Double Layer and Zeta Potential (Bose) 8
Figure 2-3: Recovery of Galena in a Batch Flotation Cell (King, 2001) 16
Figure 2-4: Recovery by Size using Split Conditioning (Singh, 1995) 17
Figure 2-5: Location of the Nkomati Mine 19
Figure 2-6: Idealized Geological Cross-Section through the Uitkomst Complex 20
Figure 2-7: Block Flow Diagram of the MMZ Plant 23
Figure 3-1: QEMSCAN Component Configuration (SGS Mineralogical Report) 25
Figure 3-2: Sample Size Distributions 27
Figure 3-3: Discrete Elemental Recoveries to Concentrate – Sulphides 30
Figure 3-4: Discrete Elemental Recoveries to Concentrate – Non Sulphides 30
Figure 3-5: Nickel Deportment by Mineral in Feed and Products 32
Figure 3-6: Feed and Product Deportment of Pentlandite Nickel by Size Fraction 33
Figure 3-7: MMZ Plant Basic Flotation Block Flow Diagram 37
Figure 3-8: MMZ Plant Rougher and Scavenger Flotation Nickel Grade-Recovery Curve 40
Figure 4-1: Laboratory Mill Rig 45
Figure 4-2: Laboratory Denver Flotation Cell 45
Figure 4-3: Feed Size and Discrete Nickel Distribution 50
Figure 4-4: Base Case Mass Recovery 51
Figure 4-5: Base Case Elemental Recovery vs Mass Pull Curve (Cumulative Data) 51
Figure 4-6: Base Case Grade-Recovery Curve 52
Figure 4-7: Effect of Collector Dosage on Mass Pull 54
Figure 4-8: Effect of Collector Dosage on Ni Grade-Recovery Curve 54
Figure 4-9: Effect of Collector Dosage on Cu Grade-Recovery Curve 55
Figure 4-10: Effect of Collector Dosage on Fe Grade-Recovery Curve 55
Figure 4-11: Effect of Collector Dosage on MgO Grade-Recovery Curve 56
Figure 4-12: Effect of Depressant Dosage on Ni Grade-Recovery Curve 58
Figure 4-13: Effect of Depressant Dosage on Cu Grade-Recovery Curve 59
Figure 4-14: Effect of Depressant Dosage on Fe Grade-Recovery Curve 59
Figure 4-15: Effect of Depressant Dosage on MgO Grade-Recovery Curve 60

Figure 4-16: Effect of Collector and Depressant Dosage on Ni Recovery 62
Figure 4-17: Effect of Collector and Depressant Dosage on Fe Recovery 62
Figure 4-18: Effect of Collector and Depressant Dosage on MgO Recovery 63
Figure 5-1: Feed Size Distribution 69
Figure 5-2: Upfront Dosing Grade-Recovery Curves 70
Figure 5-3: Nickel Recovery by Size 70
Figure 5-4: Nickel Recovery by Size after 3 Minutes of Flotation 73
Figure 5-5: Nickel Recovery by Size after 30 Minutes of Flotation 74
Figure 5-6: Mass Distribution for Upfront Dosing 75
Figure 5-7: Mass Distribution for Test 5 – Collector and Depressant Stage Dosing 76
Figure 5-8: MgO Recovery by Size after 30 Minutes of Flotation 76
Figure 5-9: Iron Recovery by Size after 30 Minutes of Flotation 77
Figure 5-10: Nickel Recovery by Size – comparison of low stage dosing with low upfront and high upfront dosing and 78

List of Tables

Table 2-1: Sulphide Compositions of Sulphide Mineralization 20
Table 3-1: Feed Composition.....28
Table 3-2: Concentrate Composition 28
Table 3-3: Tailings Composition 29
Table 3-4: Overall Recovery 30
Table 3-5: Mineral Compositions and Recoveries 31
Table 3-6: Pentlandite Concentrate Liberation Data 33
Table 3-7: Pentlandite Tailings Liberation Data 34
Table 3-8: Pyrrhotite Concentrate Liberation Data.....34
Table 3-9: Non-Sulphide Gangue Concentrate Liberation Data.....35
Table 3-10: MMZ Plant Overall Mass Balance.....38
Table 3-11: MMZ Plant Reagent Dosages.....39
Table 3-12: MMZ Plant Rougher and Scavenger MgO Grade-Recovery Data 40
Table 3-13: MMZ Plant Cleaner Grade-Recovery Data 41
Table 3-14: MMZ Plant Re-Cleaner Grade-Recovery Data 42
Table 4-1: Screen Aperture Sizes for Mill Feed and Product Sizing 47
Table 4-2: Laboratory Scale Testwork – Interaction between Reagents.....49
Table 4-3: Summary of Final Concentrate Results for Testwork Without Depressant 56
Table 4-4: Selectivity Factor for Collector Addition 57
Table 4-5: Summary of Final Concentrate Results for Testwork Without Collector 60
Table 5-1: Laboratory Scale Testwork – Stage Dosing.....68
Table 5-2: Summary of Reagent Dosing Testwork at 100 g/t 71
Table 5-3: Summary of Reagent Dosing Testwork at 30 g/t 78
Table 5-4: Financial Analysis of Stage Dosing.....79

1. Chapter 1: Introduction

The Nkomati Mine, located in the Mpumalanga Province of South Africa, mines a sulphide orebody and is the only primary nickel producer in the country. The mine has two flotation concentrator plants, one processing ore from the Main Mineralized Zone (MMZ) and the other processing ore from the Peridotite Chromititic Mineralized Zone (PCMZ). The metals recovered from the concentrators are nickel, copper, cobalt and platinum group metals (PGMs) and these are recovered in the form of a combined concentrate. The major metal bearing minerals are pentlandite, pyrrhotite, chalcopyrite and pyrite, with MgO bearing silicates as the major gangue mineral.

The MMZ process involves primary crushing, fully autogenous (FAG) milling, ball milling and flotation unit operations. The gyratory crusher feed is sourced from a combination of open-pit and underground mining operations. The ore is crushed down to generate a feed sufficient for FAG milling. The FAG mill operates in closed circuit with pebble crushers to provide a sufficiently reduced particle size for secondary milling. The ball mill grinds the particles further to ensure that the desired particle size for flotation is obtained. The valuable minerals are separated from the bulk of the gangue in the flotation process and the flotation products are processed through thickeners and filters to prepare the concentrate for shipment to the smelter and tailings to the tailings storage facility. Reagents are added to the flotation process to maximise the recovery of the valuable minerals and to achieve a suitable product purity for downstream processing and product handling. A frother is added to ensure the froth is stable, collectors improve the hydrophobic nature of the valuable minerals to promote collection to the concentrate and depressants reduce the flotability of gangue to the concentrate.

In the current plant practice, reagents are dosed either at the head of each flotation bank or at process stream nodes (where two streams meet). A block flow diagram of the MMZ flotation circuit at Nkomati mine is presented.

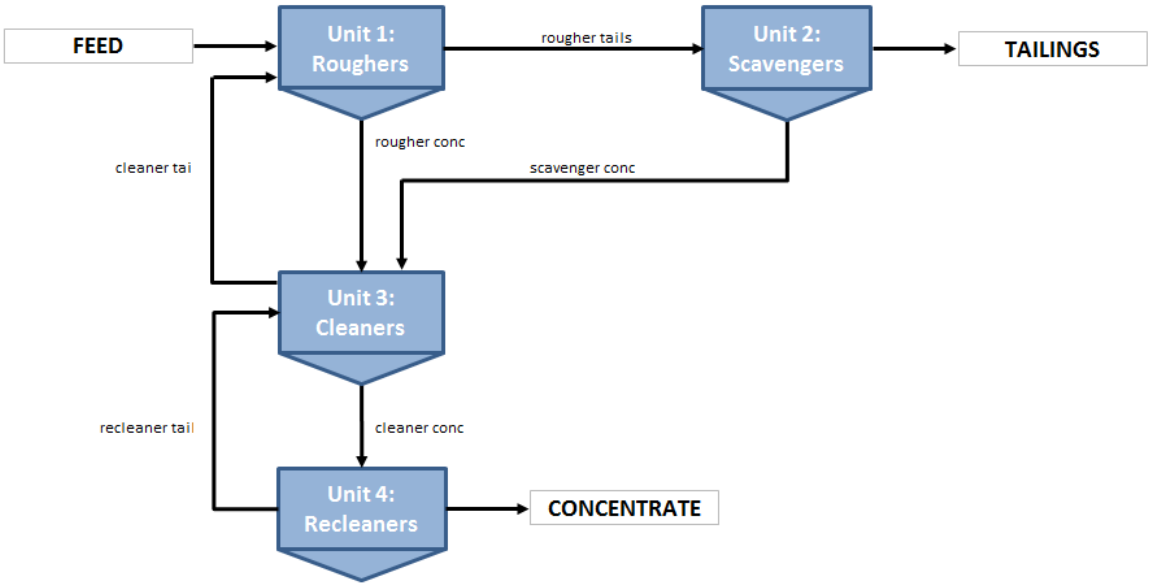


Figure 1-1: MMZ Flotation Process Block Flow

The MMZ and PCMZ plants have surpassed design throughput capacities and one of the current focuses is optimization of process inputs and parameters. The study proposed in this research involves mineralogical testwork, metallurgical testwork and process surveys on the MMZ plant to identify areas of improvement with the main focus being the reagent addition to improve metal recovery and product quality.

1.1 Problem Statement and Research Objectives

The study is motivated by the need to improve the concentrate grade and recovery from the MMZ plant to maximize the financial return to the Mine. This will be done by first evaluating the effect of flotation reagents, used at the plant, on the recovery of valuable minerals and gangue flotation, particularly nickel and talc (or more appropriately, magnesium bearing gangue). The reagents to be considered in the study will be the collector and depressant used at the plants, which are sodium isobutyl xanthate (SIBX) and carboxymethyl cellulose (CMC).

Secondly, the dosing regimen at the plant will be evaluated. The current operating philosophy involves using a set collector dosage, while varying the depressant dosage to control the gangue content. Often the depressant dosage on the roughers is adjusted, however, its effect on the rougher concentrate is not clear. Testwork on the interaction between collector and depressant will provide a better understanding of the process and improve the addition of reagents in the circuit. In the current plant practice, reagents are also dosed either at the head of each flotation bank or at process stream nodal points. In this research an alternative stage dosing method will be investigated with the view of improving reagent consumption, metal recovery and operating costs.

1.2 Aim and objectives

The aims of the study are to:

1. Evaluate the effect of the collector and depressant on the flotation of valuable and gangue minerals and this will include measuring the:
    a. Effect of SIBX collector dosage on flotation
    b. Effect of CMC depressant dosage on flotation
    c. The interaction between collector and depressant – which one is more selective to which mineral
2. To apply stage dosing of flotation reagents in specific areas in the plant and measuring the mineral recovery and product grade. The effect of reagent addition on particle size recovery will also be evaluated.

All work will be based on the MMZ orebody consisting of a mixture of underground and open pit material.

1.3 Research questions

The research questions are summarised as follows:

i. What is the relationship between collector dosage and valuable mineral and gangue flotation?
ii. What is the relationship between depressant dosage and valuable mineral and gangue flotation?
iii. What is the combined effect of collector and depressant dosage on valuable mineral and gangue flotation?
iv. What are the current areas of improvement for the process?
v. How can stage dosage of reagents be used to improve process performance?

2. Chapter 2: Literature Review

2.1 Principles of Flotation

Flotation is a separation process that exploits the difference in surface properties between two or more particles. Flotation can be defined as a process for the separation of particles of type A (e.g. a valuable mineral) from a mixture of particles of A and B (e.g. gangue or a second valuable mineral) that involves

- the addition of chemicals to the slurry which makes particle A hydrophobic and ensures particle B remains hydrophilic,
- addition of air bubbles for the collection of particle A, and
- the rising of the bubble with particle A attached to the bubbles, followed by the recovery of the froth as concentrate.

The process occurs in a three phase system of water, solids and froth. The solids phase contains a variety of minerals, valuable and unwanted gangue. Furthermore, the targeted metal may be present in different minerals and as such possess different properties. Other complications of the solid phase include mineral liberation and the distribution of minerals across a relatively wide size range and variations in crystal structure, shapes and textures of milled flotation feed. The liquid phase is the medium in which the mineral separation occurs. The liquid phase is affected by the hydration of ions, solubility of minerals in water, impurities in water and dissolved gases. The gas phase is the air bubbles that are injected into the flotation cell for attachment of the mineral and recovery to the froth (Bulatovic, 2007).

The flotation process involves the collision of a hydrophobic particle with an air bubble, attachment of the particle to the air bubble and, sometimes, the detachment of the particle from the air bubble. The probability of successful attachment is, therefore, related to these properties as shown in equation (1).

$$P = P_C \times P_A \times (1 - P_D) \qquad (1)$$

Where P_C is the probability of collision, P_A is the probability of attachment and P_D is the probability of detachment. The process of attachment and recovery of a hydrophobic particle from the froth phase is termed True Flotation. Other sub-processes that occur in the flotation cell are:

1. Adsorption of chemicals onto mineral surfaces to induce hydrophobicity and enable attachment onto an air bubble
2. Collision of bubbles with particles
3. Entrainment of the gangue particles from the pulp into the froth phase, which can collect in the concentrate, ultimately reducing the grade. Entrained particles are those that are drawn into the froth phase by water and are typically ultrafine material
4. Precipitation which involves dissolved oxygen precipitating on a hydrophobic site and forming a bubble
5. Froth drainage and bubble rupture. Froth stability is an important property to allow selective drainage

Processes 1 and 2 above are non-selective, causing both gangue and valuable minerals to be collected. An illustration of the abovementioned processes is provided in Figure 2-1.

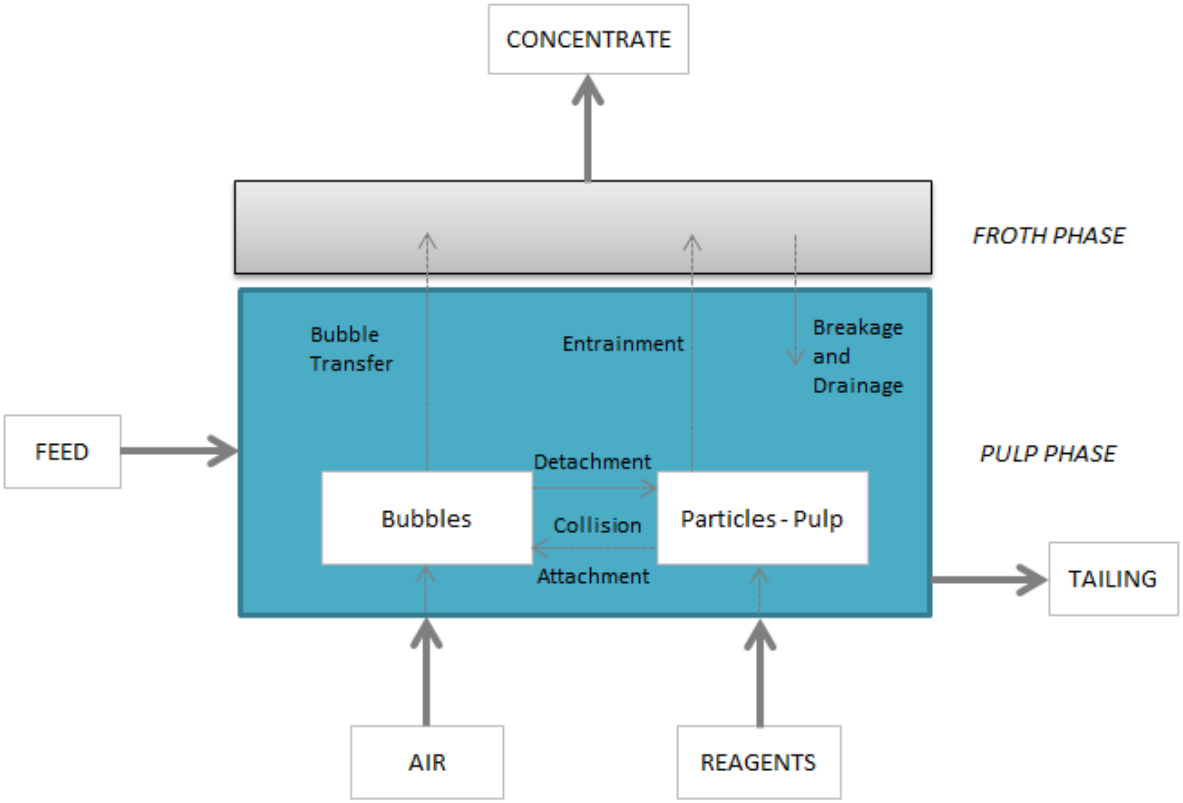


Figure 2-1: Schematic of Flotation Processes

One of the key properties of the flotation system is the contact angle. The contact angle provides an indication of the tendency of a particle to attach to a bubble and, thus, its hydrophobicity. Particles with larger contact angles have a higher probability of collection while particles with a zero contact angle are highly hydrophilic and would not adhere to the bubble.

Reagents are added to the flotation cell to assist with the flotation processes. Collectors are organic chemicals added to the pulp to render the valuable mineral hydrophobic, enhancing the probability of mineral attachment to the air bubble and subsequent recovery in the froth phase. Depressants are modifying chemicals that aim to render gangue minerals hydrophilic such that the valuable mineral is selectively floated. Activators prepare the mineral surface for collector adsorption. Frothers reduce the surface tension of the bubbles to stabilize bubbles during particle collision and attachment.

The electrical double layer describes the interaction between the solid and liquid phases. The phenomenon is presented in Figure 2-2. If a negatively charged particle/ion is present in solution, other co-ions would repel while counter (positive)-ions would be attracted to it. The counter ions are relatively strongly bound and are considered immovable. This layer is called the Stern layer. The Diffuse layer exists on the outside of the Stern layer and consists of ions that are more weakly adsorbed. The electrical double layer refers to both these layers.

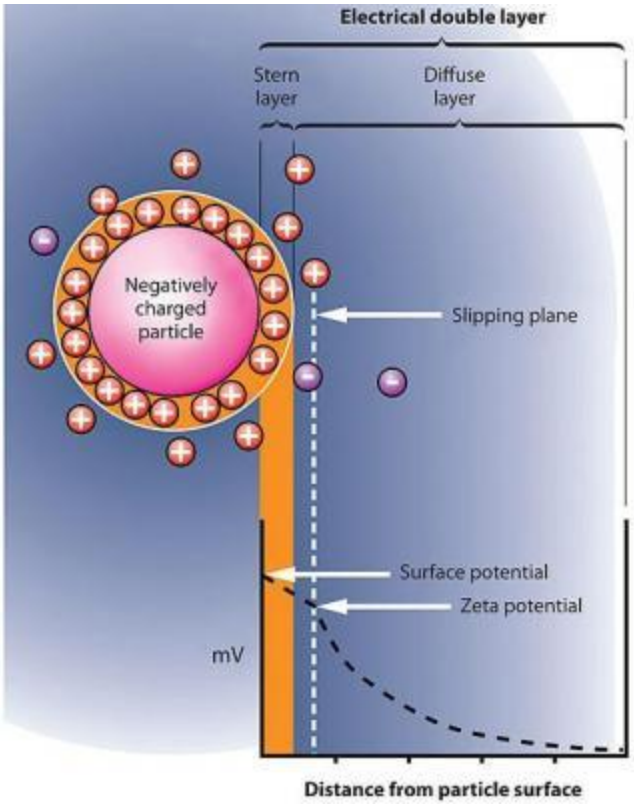


Figure 2-2: Electrical Double Layer and Zeta Potential (Bose)

Figure 2-2 also shows the electric potential as a function of the distance from the particle/ion. The slipping plane of the particle exists on the boundary of the stern layer and is the point at which shear occurs. The potential at the slipping plane is known as the zeta potential. Therefore, the zeta potential provides an indication of the resultant

charge of the particle. Most flotation phenomena can be ascribed to the electrical double layer (Salopek et al., 1992) which include:

- the physical adsorption of flotation reagents
- the dispersion of particle in the slurry
- the coating of slimes on larger particles, with the potential charge and strength affecting slimes coating and its removal
- flotation kinetics relate directly to the effect of double layers on the kinetics of film thinning (Salopek et al., 1992)
- gangue activation

2.2 Nickel Sulphide Production

Laterite and sulphide ores are the two primary sources of nickel. Laterite ores were typically formed near the earth's surface and have subsequently weathered. Historically, laterite ores have been relatively difficult to process due to their mineralogical composition. Laterites are oxide ores commonly found in two ore-types, near surface limonite ore and deeper saprolite ore. The substitution of nickel in the crystal structure of the host mineral (iron hydroxide for limonite and magnesium silicate for saprolite) complicates the recovery process and restricts physical beneficiation potential (Crundwell, 2011). Most of the world's nickel has been produced from sulphide ores. However, pyrometallurgical and hydrometallurgical techniques have successfully been developed to process laterite ores, depending on the limonite and saprolite compositions.

Limonitic laterite ores are processed by crushing and grinding the ore to liberate the valuable mineral(s). Once the mineral is liberated, leaching under fairly harsh conditions is required to solubilize the nickel. High pressure sulphuric acid leach and high temperature ammonia leach are processes that have been used. Once in solution, the solids are washed in a counter current decantation circuit to separate the solid and

solution whilst minimizing the nickel lost to the solution in the tailings stream. The nickel is selectively extracted from solution by solvent extraction. The nickel product can then be recovered from solution as a final product (e.g. by electrowinning) or as an intermediate product (e.g. by precipitation or crystallisation). Most saprolitic laterite ores are dried, followed by processing through a rotating calcination/reduction kiln. The reduced calcine is then treated by electric reduction to produce a crude molten ferronickel product.

Sulphide ores are also processed by crushing and grinding to liberate the nickel minerals. The ground ore is concentrated by flotation. The concentrate is then filtered. Smelting of the concentrate produces a nickel matte that is subsequently refined to produce nickel metal. A description of the general flotation practices for nickel sulphide ores is provided below.

2.2.1 General Flotation Characteristics

Flotation of nickel ores tends to vary due to the differences in the ore mineralogy. The gangue is also important to the process, with massive sulphide ore generally easier to process than magnesium containing ores. Dithiophosphates and Mercaptan are better collectors than xanthate when naturally floatable gangue is present (Bultovic, 2007). Nickel is depressed at high pH (above 9.5) and, therefore, pyrite can be selectively floated under alkaline conditions, i.e. reverse floatation where pyrite reports to concentrate and pentlandite to tailings. Pentlandite can be reactivated with copper sulphate after depression.

2.2.2 Reagents used in Nickel Sulphide Concentrators

The nickel ores from Western Australia (Mt. Keith, Mt Windarra, Yakabindie) do not contain copper and a bulk concentrate is usually produced. These ores have high talc content, but pyrrhotite and pyrite are not problematic. Xanthate collector is used and combinations of xanthate and mercaptan have shown improved nickel recoveries and selectivity. The major gangue mineral is talc, however, magnesite dolomite is also present in certain ores. Carboxymethyl cellulose (CMC) works well as a depressant for

talc while dextrin and guar are good for magnesite containing ores. Work conducted at Mt Windara showed that higher molecular weight and low sodium glycol content of the CMC improved talc depression. High molecular weight also had a positive effect on pyrrhotite.

Copper-nickel ores are often associated with pyrrhotite. The process flowsheet is dependent on the nature of pyrrhotite present. Pyrrhotite is removed by magnetic separation either before or after flotation. Xanthates are also typically used as collectors for these ores. Secondary collectors in the form of mercaptans or thionocarbomates have improved recoveries of base metals and PGMs. An example is the Strathcona plant in Canada where xanthate in the rougher circuit was replaced by a combination of mercaptan and thionocarbomate. A variety of depressants, from soda ash, CMC to polyacrylic acid and water-soluble polyamines, have been tested and used for pyrrhotite depression. At the Sudbury plant, polyacrylic acid was found to depress pyrrhotite, however, pentlandite depression was also observed.

2.3 Flotation Reagents

2.3.1 Collectors

A collector can be ionising or non-ionising in nature where ionisation involves dissociation of the molecule into ions. Non-ionising collectors merely form a coating on the mineral surface. Ionising collectors have a polar and non-polar group. The polar group reacts with water and adsorps onto the mineral surface while the non-polar group is positioned to the water phase. Xanthates are typical ionising collectors used for sulphide flotation. They are anionic in nature and are sulphur based, with the ionic form $ROCS^{2-}$ (where R is the hydrocarbon group, O is oxygen, C is carbon and S is sulphur). Sodium isobutyl xanthate has the chemical formula $C_5H_9OS_2Na$; the functional group being the carbon atom bonded to the sulphur atom. The adsorption process is complex and there remains unresolved aspects of the mechanisms involved (Bulatovic, 2007). Mendiratta (2000) summarized the adsorption mechanisms as a mixed potential mechanism involving a cathodic reduction of oxygen coupled with one of the following anodic reactions:

a. Chemisorption of xanthate ion: $X^- \rightarrow X_{ads} + e^-$
b. Metal xanthate formation: $MS + nX^- \rightarrow MX_n + S^0 + ne^-$
c. Oxidation of xanthate to dixanthogen at the mineral surface: $2X^- \rightarrow X_2 + 2e^-$

Where X is the xanthate, M is the metal, e^- denotes electrons, ads denotes adsorbed, MS is metal sulphide and S is sulphur.

The detailed reaction mechanisms of reagents are complex and considered outside the scope of this work.

2.3.2 Depressants

Inorganic modifiers can be used as depressants, however, organic modifiers, particularly organic polymers, are depressants for sulphide and non-sulphide minerals. As with collectors, these reagents are grouped according to ionic charge: non-ionic, cationic, anionic and amphoteric. Typical anionic polymers used in sulphide flotation are carboxymethyl cellulose (CMC). CMC's target magnesium-bearing minerals like talc.

Depressants also destabilise the froth due to its nature as well as the absence of froth stabilising talc (Bradshaw et al., 2005). Wiese (2009) reports that mechanisms of depressant adsorption have been studied:

- Steenberg and Harris suggested that hydrophobic bonding (bonding of depressant and talc to minimize the entropy of the system) occurs on talc planes/faces while hydrogen bonding takes place on talc edges (where charged ions of Mg^{2+} and OH^- exist)
- Liu and Lakowski suggested that acid-base interaction occurs where the depressant behaves as an acid and adsorps onto metallic sites.

Steenberg and Harris (1984) studied the adsorption of CMC, guar gum and starch on a variety of minerals including talc, pyrite and galena. Their work indicated that the polysaccharides adsorbed onto both hydrophobic and hydrophilic surfaces. In addition, adsorption was found to occur on both the hydrophobic, non-polar basal plane as well

as the hydrophilic, highly polar edges of talc. Despite the presence of counter-ions in the polymer layer, it did not prevent the adsorption of the polysaccharide. It was also inferred that adsorption on select minerals did not occur and that selectivity was rather a function of the reactivity with the collector (displacing weakly bonded depressant).

Khraisheh (2005) investigated the effect of molecular weight and concentration of CMC on the adsorption onto talc. It was postulated that the depressant was first adsorbed onto talc planes and subsequently onto the edges with similarities to the results of Steenberg and Harris with respect to adsorption mechanisms. The adsorption of CMC was found to be proportional to the molecular weight of the CMC and that ionic strength improved adsorption.

2.4 Interaction between Collector and Depressant

Bradshaw et al (2004) studied the interaction of depressant and collectors. Two different depressants (high and low charge, same molecular weight) and collectors (different chain lengths to create different hydrophobicity) were used in the study. Collector dosage remained constant while depressant dosage was increased from 100 g/t to 500 g/t. The results indicated:

1. Higher collector chain length did not improve metal recovery due to negative effect of increased hydrophobicity on froth stability
2. The same effect was noticed irrespective of depressant dosage
3. High dosages of depressant improved gangue depression but also reduced sulphide recovery
4. Nickel recoveries were more severely affected by increased depressant dosage than chalcopyrite due to its lower floatability

In other work by Bradshaw et al (2005), the effect of different collectors and depressants on the flotation of various ores was tested. Flotation performance was measured by solid and water mass recovery. Engelbrecht and Woodburn (1975)

showed entrainment and water recovery is directly related. Microflotation tests indicated that longer chain xanthate gave higher recoveries on a galena (lead) ore. In the absence of depressant, the bubble sizes were significantly larger than when depressant was added, resulting in higher water recoveries (bubble sizes were measured using SmartFrothTM, a machine vision system providing descriptors of the froth surface). It was postulated that stronger collectors improved hydrophobicity, however, at the same time reduced the froth stability which resulted in reduced solids and water recovery. In the absence of froth stabilizing gangue (caused by depressant addition), this effect was more severe. Destabilization of the froth was also inferred through the recovery of highly hydrophobic chalcopyrite, the recovery of which reduced with the use of stronger collector. Particularly at higher depressant dosages, the sulphide recovery was lower. It has been hypothesized that although depressant does not specifically target sulphide minerals, at high dosages, the recovery can be reduced. Over-depressing of naturally floatable gangue can reduce froth stability and reduce sulphide recovery. Thus, collector and depressant dosages have a significant effect on the froth stability and metal recovery.

2.5 Entrainment

Wang (2014) reviewed the entrainment process. A summary is provided below.

Entrainment is a process in which both hydrophilic and hydrophobic minerals report to the concentrate, thus reducing the grade of the product. Entrainment involves transfer of minerals that are in the path of rising bubbles and are carried upward into the froth without bubble-particle attachment followed by collection from the froth into the concentrate launder. Drainage of the entrained particles back into the pulp occurs through the plateau borders or on collapse of froth. Factors that affect entrainment include:

1. Water recovery: there seems to be a direct relationship between the recovery of water and entrainment for both fine and coarse particles. Furthermore, an increase in the water recovery results in reduction in the concentrate grade due to an increase gangue reporting to the concentrate.

2. Pulp solids concentration: the content of solids in the region just below the froth-pulp interface is important for entrainment. An increase in pulp solids concentration results in an increase in entrainment.
3. Particle size: entrainment increases significantly for fines. Larger particles drain more effectively. This is particularly noticeable with hydrophilic particles, since hydrophobic particles tend to have an optimum intermediate flotation size with recovery decreasing for both finer and coarser particles.
4. Impeller speed: higher impeller speeds results in higher water recovery which increases entrainment. This occurs as a result of reduced selectivity under turbulent conditions (Cilek, 2008).
5. Particle density: lower density particles tend to be more mobile in the water phase and, therefore, recovered more by entrainment than denser particles.
6. Gas rate: mixed reviews are found on the effect of air rate on entrainment. Higher air flowrates would carry over more solids to the froth, but the resultant froth structure would affect entrainment (or drainage).
7. Froth depth and retention time: deeper froth heights promote drainage and reduce entrainment. A longer froth retention time allow more time for drainage and reduces entrainment.
8. Froth structure: improved structure (measured by stability) affects the drainage of gangue and, therefore affects entrainment.

2.6 Recovery by Size and Distribution of Reagents down a Flotation Bank

A typical recovery by size is presented in Figure 2-3.

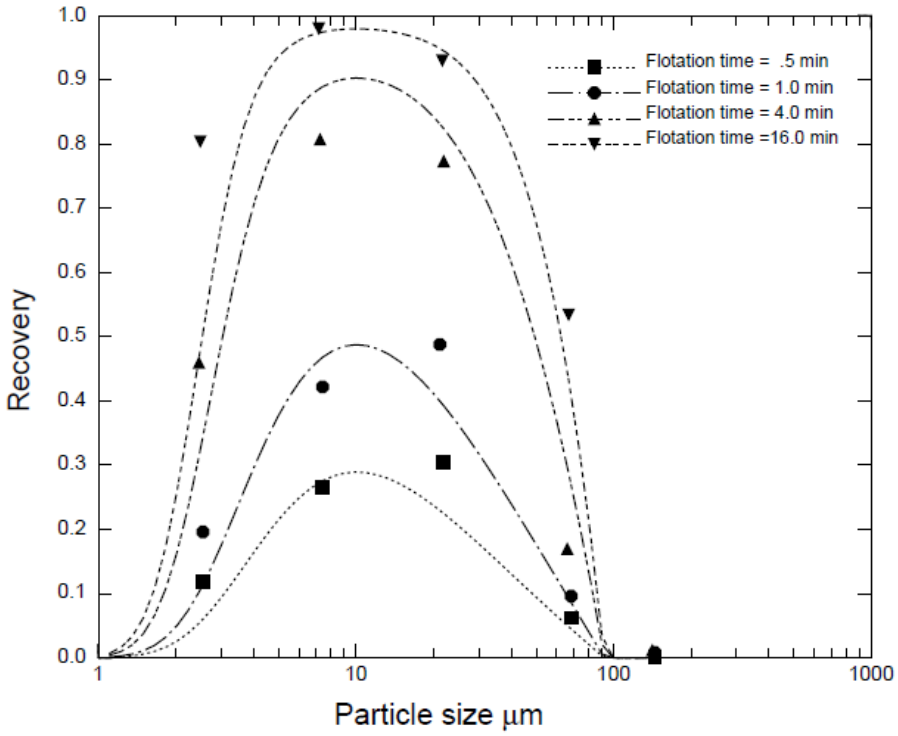


Figure 2-3: Recovery of Galena in a Batch Flotation Cell (King, 2001)

Recovery of the intermediate size range is higher than the recovery of coarse and fine particles. Losses in fines are due to the reduced probability of collision with bubbles. Large particles recovery is reduced due to rupture of the particle/air bubble aggregates due to insufficient hydrophobic coverage of the particles (Bazin, 2001) or by excessive forces of weight and turbulence in the cell.

Small particles require less collector than coarse particles in terms of effectiveness, however, due to increased surface area, consume more collector. Split conditioning is a process whereby an ore is separated into coarse and fine fractions and reagents are conditioned separately. The conditioned slurry is recombined before flotation. Trahar (1976) tested split conditioning on Broken Hill lead-zinc ore in which the split size was 38 μm. The results indicated that split conditioning improved lead recovery and a higher selectivity was achieved. Split conditioning and combined flotation of a phosphate ore yielded improved recovery in all size fractions (Singh) as shown in
Figure 2-4. Split conditioning uses classification to condition different particle sizes separately, but requires additional capital expenditure.

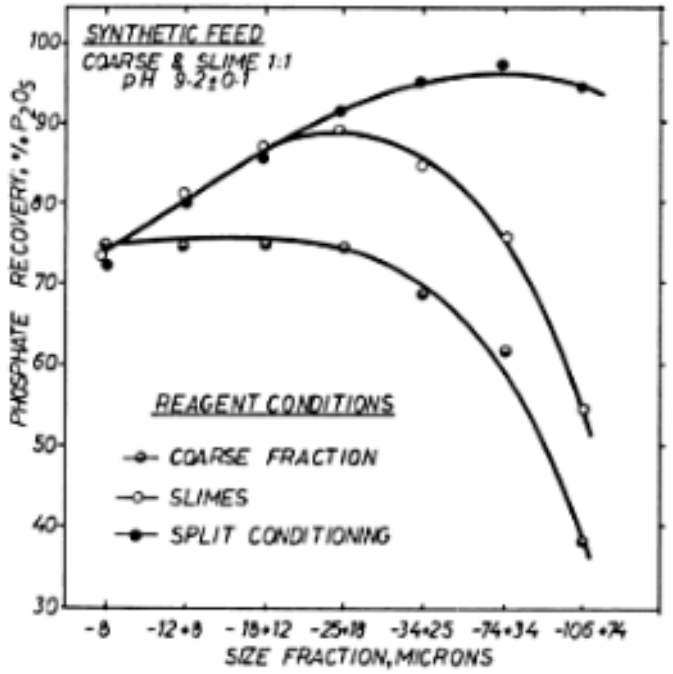


Figure 2-4: Recovery by Size using Split Conditioning (Singh, 1995)

Down the bank reagent addition or stage dosing has the ability to mimic this phenomenon exhibited by split conditioning.

Bazin (2001) explained the effect of particle size on the efficiency and selectivity of reagent consumption:

> "Since fine particles exhibit a larger specific surface (cm^2/g) than coarse particles, the consumption of collector required to produce a given degree of particle coverage is much higher per unit of mass of fine particles than it is for coarse ones. If a mixture of fine and coarse particles is contacted with a given dosage of collector, most of the collector will be consumed by the fine particles, which in fact need little coverage to be efficiently floated. On the other hand, there would be no sufficient collector available to produce the hydrophobic coverage required to float the coarse particles."

Bazin (2001) studied the effect of minimal addition of reagent at the top of a flotation bank to collect fine particles with reduced entrainment. Batch testwork on a copper-lead ore showed an increased lead recovery of 2%. The plant scale trial was conducted on the rougher-scavenger-scavenger bank with original reagent additions of 85/15/5 compared to 50/45/5 which resulted in an improved lead recovery from 73% to 79%. Another trial was conducted which indicated that a 3% recovery improvement was obtained at 23% reduction in collector consumption. Stage dosing is typically practiced; however, the majority of the reagent is dosed upfront. This paper suggested that improved recovery at lower dosages, particularly for larger particles, can be achieved by reducing upfront dosage.

2.7 Nkomati Mine

2.7.1 Orebody

The location of the Nkomati Mine is indicated in Figure 2-5. It is situated within the Uitkomst Complex between Barberton and Machadodorp in the Mpumalanga Province of South Africa. The ore is predominantly mined from an open pit operation with a portion of material mined underground.

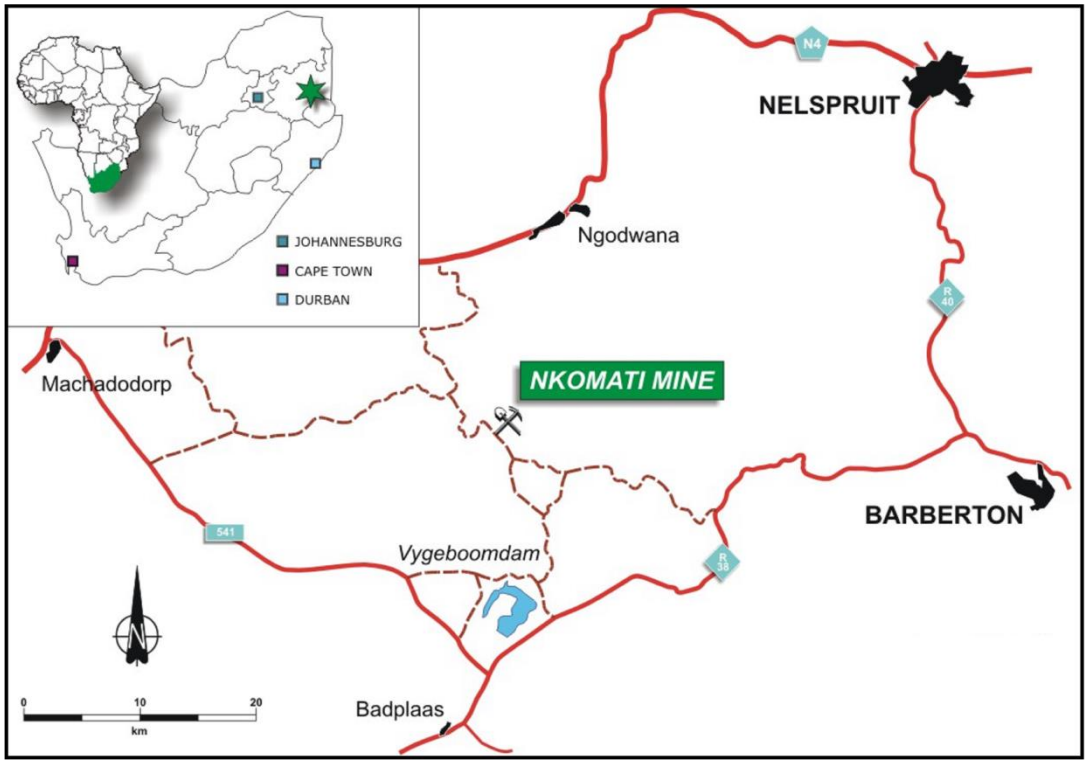


Figure 2-5: Location of the Nkomati Mine

The Uitkompst complex consists of the following lithologies, with the Basal Gabbro Unit situated at the bottom and the Norite Unit at the top of the complex:

1. Basal Gabbro Unit
2. Lower Pyroxenite Unit
3. Chromititic Peridotite Unit
4. Massive Chromitite Unit
5. Peridotite Unit
6. Upper Pyroxenite Unit
7. Norite Unit

An illustration presenting a cross-section through the complex is presented in Figure 2-6.

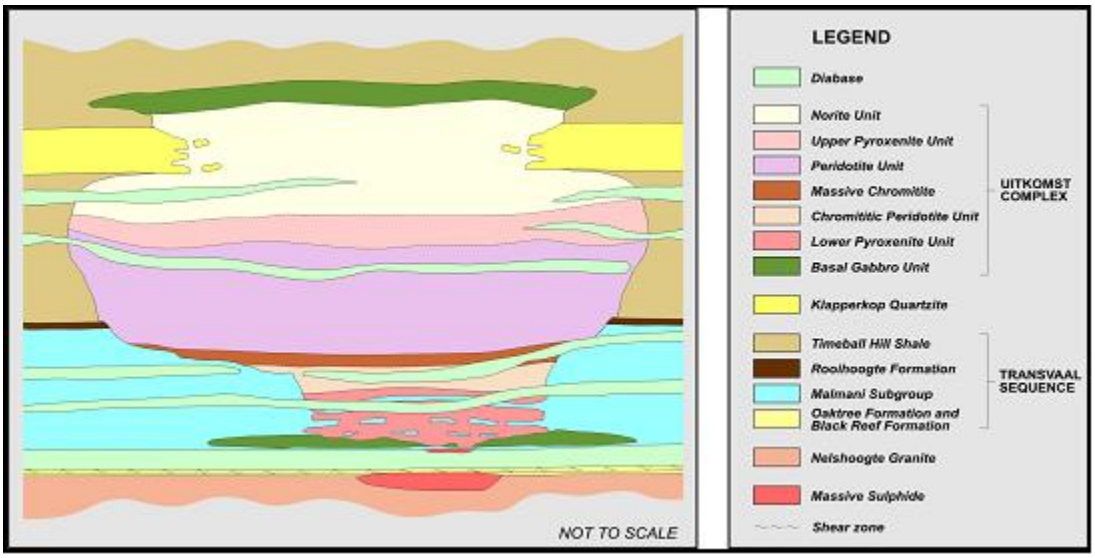


Figure 2-6: Idealized Geological Cross-Section through the Uitkomst Complex

The sulphide mineralization is categorized by four regions; these are the massive sulphide body (MSB), main mineralized zone (MMZ), peridotitic chromititic mineralized zone (PCMZ) and the basal mineralized zone (BMZ). The main sulphide minerals in the MSB, MMZ and PCMZ zones are pentlandite, pyrrhotite, chalcopyrite and pyrite. The approximate sulphide compositions for the BMZ and MMZ are provided in Table 2-1.

Table 2-1: Sulphide Compositions of Sulphide Mineralization

Sulphide Mineral	Molecular Formula	BMZ (%)	MMZ (%)
Chalcopyrite	$CuFeS_2$	23-37	5
Pyrrhotite	$Fe_{1-x}S$	21-32	85
Pentlandite	$(Fe,Ni)_9S_8$	17-25	10
Pyrite	FeS_2	2-15	varied

The BMZ forms part of the GAB unit and is adjacent to the MMZ and the two zones are mined together. The sulphide mineralization is generally fine-grained and varies from being disseminated to net textured. The average Ni:Cu ratio in the BMZ ore is 0.9:1. This is due to the significantly higher chalcopyrite content. Chromite is absent from the BMZ.

The MMZ is contained in the lower pyroxenite unit. The following rock types are present in this unit:

- Pyroxenite, with clinopyroxene as the dominant primary silicate, and amphibolite, abundant in phlogopite
- Wehrlite: contains mostly olivine (50-55%) and clinopyroxene (20-25%)
- Pegmatoidal: contains pyroxenes and amphibole
- Xenoliths: calc-silicate inclusions and quartzitic inclusions

The MMZ can be categorized as either net textured, massive/semi-massive, blebby or disseminated. MMZ typically has a Ni:Cu ratio of 2.7:1. Palladium-bismuth-tellurides are the main PGM-bearing phases, forming small grains (5-25 µm) included in sulphide and silicate minerals.

MSB is found in the footwall lithologies. The main base metal minerals are pentlandite and chalcopyrite. The dominant platinum group mineral is merenskyite, with minor michenerite, testibiopalladinite, and sperrylite.

The PCMZ ore is hosted by the chromititic peridotite unit and is characterized by chromitite layers, pods, lenses and wisps. The major silicate minerals are chlorite, amphibole and talc. The talc content in the PCMZ is considerably higher than MMZ, containing between 15% and 25%.

2.7.2 Process

A block flow diagram of the MMZ plant at the Nkomati Mine is presented in Figure 2-7.

Ore is received at the primary gyratory crusher from the open pit mine or from stockpiles. The top size fed to the crusher cavity is 1 m. The target crusher feed size distribution is 80% passing 450 mm. The crusher discharge has a size distribution of 80% passing 120-130 mm. The primary gyratory crusher is campaign fed with MMZ and PCMZ ore. Therefore, crushed ore is transferred via overland conveyors to stockpiles before feeding the plants. The MMZ stockpile receives ore from the open pit and

underground in a ratio of approximately 11:1 (open pit:underground). Underground ore is mined and jaw crushed before being stockpiled.

The stockpiled ore is transferred to the milling circuit. The MMZ milling circuit consists of a FAG mill operated in closed circuit with pebble crushers to ensure that critical size is effectively removed from the circuit. FAG milling is followed by secondary ball milling in closed circuit with cyclones. The product from the milling circuit (i.e. the cyclone overflow) serves as the feed to the flotation circuit. The target particle size distribution of the float feed is 70% passing 75 μm at a pulp specific gravity of approximately 1.3.

The flotation circuit is a rougher-cleaner-recleaner circuit. The circuit consists of a 100 m^3 conditioning tank, two banks of 5 X 130 m^3 rougher cells each and 2 banks of 3 X 100 m^3 scavenger cells. The original design included the scavengers to recover pyrrhotite for separate tailings disposal. Due to the slow nature of the pyrrhotite flotation, the plant currently uses these cells for additional roughing. The rougher concentrate is processed in 5 X 50 m^3 cleaner cells. The cleaner concentrate is treated in 3 X 30 m^3 cells to produce the final concentrate. The tails are thickened and treated with lime in a tailings disposal tank before being pumped to the tailings dam. The concentrate is thickened and filtered.

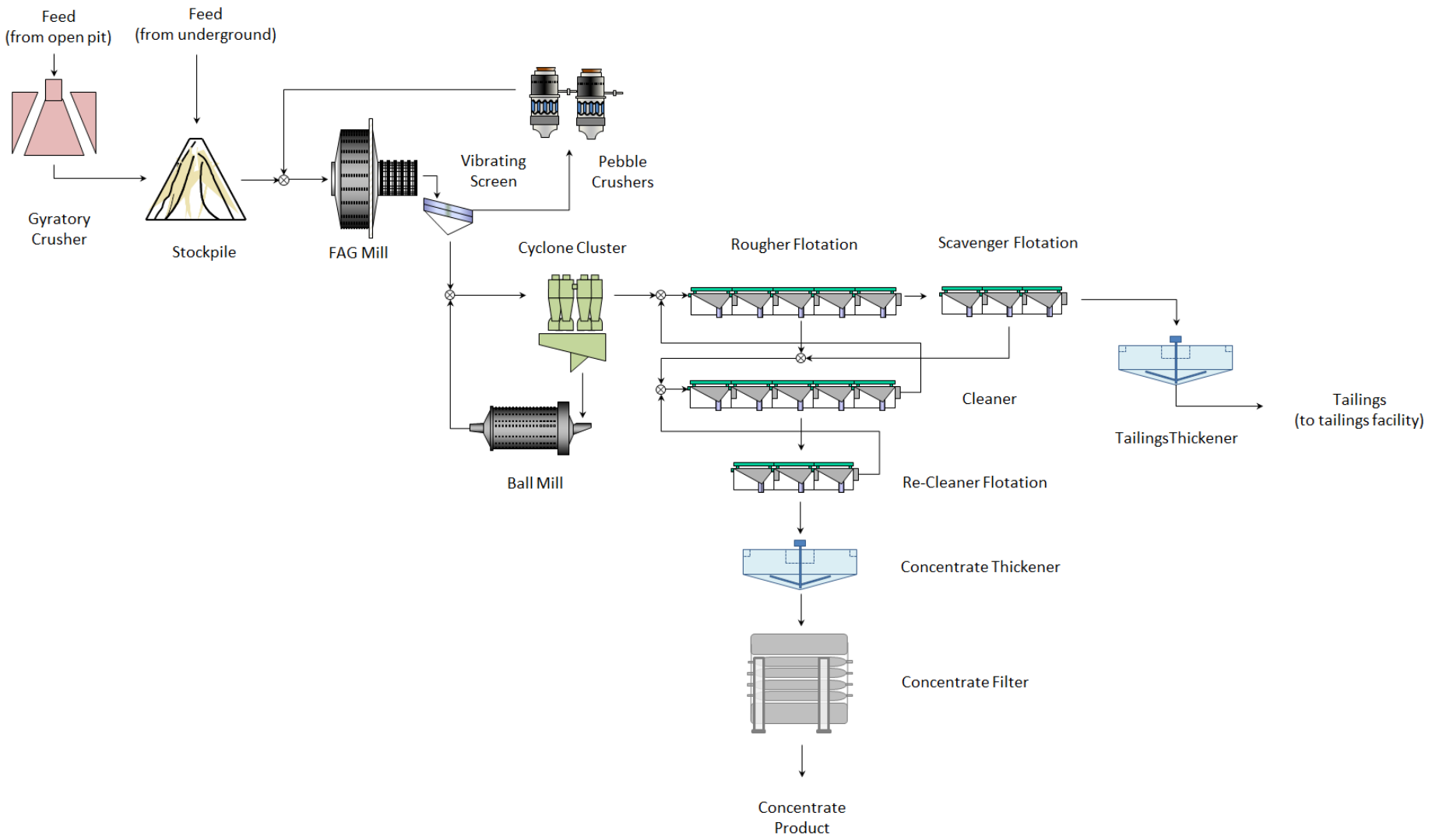


Figure 2-7: Block Flow Diagram of the MMZ Plant

3. Chapter 3: Characterisation Testwork

3.1 Introduction

Characterisation testwork was conducted to determine the mineralogical composition of the ore, the mineral distribution of the target and gangue elements and the losses of nickel as well as misplacement of gangue to the concentrate.

The areas covered in this chapter are:

- Procedure used to conduct mineralogical evaluation and plant survey
- Particle size distribution of the flotation feed
- Elemental and mineral composition of the various process streams
- Deportment of selected minerals
- Evaluation of the process survey
- Summary of characterisation work

3.2 Methodology

3.2.1 Mineralogical Testwork

All mineralogical testwork was conducted by an external laboratory. The plant samples submitted were:

1. Flotation Feed
2. Final Concentrate
3. Final Tailings
4. Cleaner Tailings

These samples represent the input and output streams of the process, and the cleaner tailings representing the rejects of the cleaning (cleaner and re-cleaner) circuit.

The samples were cyclosized to separate the various products into distinct size fractions to ensure that cross-sections do not affect particle size characterization. Each fraction was submitted for:

1. chemical analysis by X-Ray Fluorescence (XRF) technique. The method involves exciting atoms by an external energy source. This emits x-ray photons with a

characteristic energy/wavelength, thus identifying the specific element. The amount of photons emitted are counted to quantify the elements present.

2. QEMSCAN (Quantitative Evaluation of Minerals by Scanning Electron Microscopy) analysis using the particle mineral analysis (PMA) mode for bulk mineralogy and specialised mineral search (SMS) mode for detailed mineralogy on the sulphide minerals. QEMSCAN collects chemical and atomic number information at every point on a grid analysis to measure particle compositions. The minerals are identified by comparing the data to a mineral compositional database. The PMA mode is a two-dimensional scanning method which analyses the mineral composition of the exposed surface area of the particle, identified using a back scatter electron scan – in excess of 3 000 particles are analysed in a typical PMA analysis. The SMS mode is based on the PMA mode, however, the search is only focussed on specific minerals of interest. This is particularly useful for low grade compositions. Particles with a specific back scatter electron intensity/brightness range are targeted and only these particles are fully analysed. A flow diagram of the QEMSCAN process is presented in Figure 3-1.

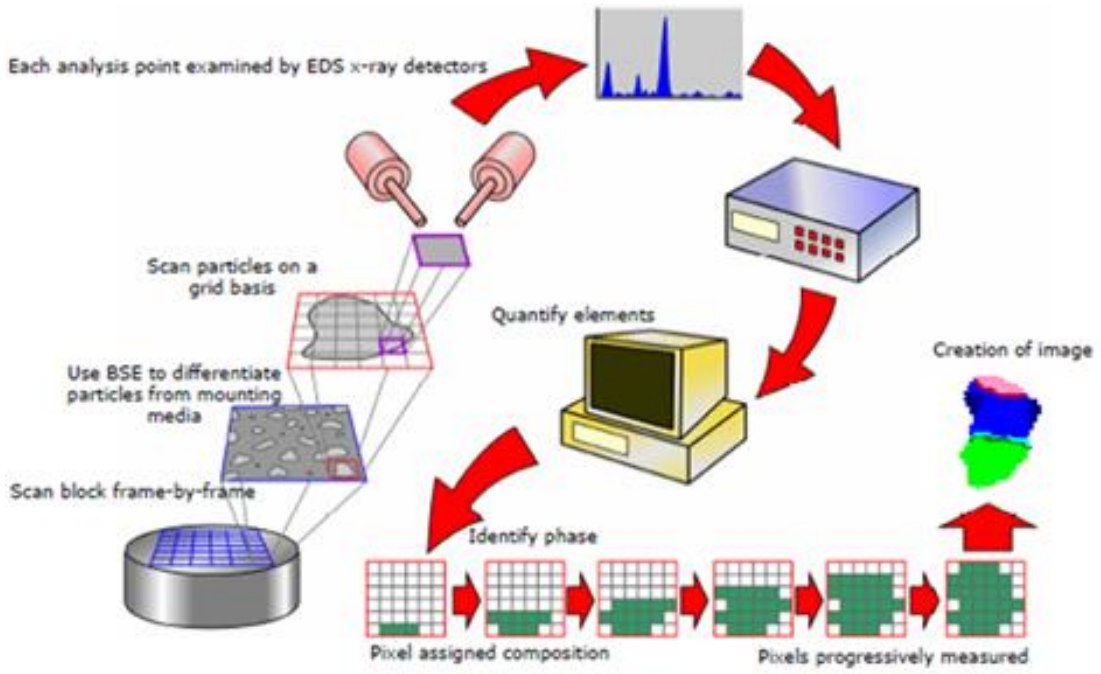


Figure 3-1: QEMSCAN Component Configuration (SGS Mineralogical Report)

3.2.2 Process Survey

A process survey was conducted over the flotation plant. A basic block-flow diagram of the process is presented in Figure 1-1 and repeated below for ease of reference.

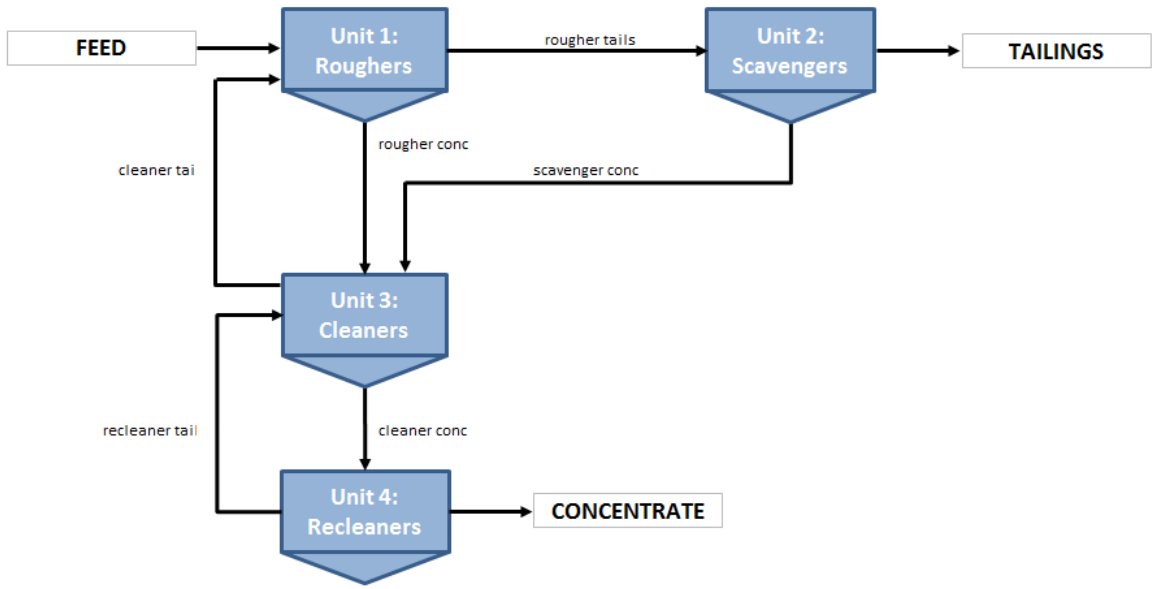


Figure 1-1: Process Block Flow

Samples of the following streams were collected for analysis:

- Rougher flotation feed: cross-cut sampler
- Final tailings: cross-cut sampler
- Final concentrate: poppet sampler
- Concentrate from each flotation cell (rougher, scavenger, cleaner and re-cleaner cells): manual sampling
- Tailings from each flotation cell: manual sampling

The survey was conducted when the plant had been operating at steady state for a sufficiently long period (2 hours). Various process parameters such as plant feed, mill power and load, cyclone feed flow and density, cyclone pressure, flotation stream volumetric flowrates, air flowrates and flotation conditions were monitored to confirm steady state operation.

The concentrate from each cell was collected by taking manual stream cuts from the concentrate pipes. These pipes are fed by gravity into a combined launder and were collected just before the streams are combined.

The tailings from each cell was collected by pumping slurry out of the cell from the region around the discharge port of the cell. This was done by placing a 1-inch steel pipe into the cell and pumping the slurry out. Manual stream cuts were taken from the pumped stream.

Samples were collected every 15 minutes over a period of approximately 2 hours.

3.3 Mineralogical Evaluation

3.3.1 Particle Size Distribution

Particle size distributions for the flotation feed, final concentrate and final tailings is presented in Figure 3-2.

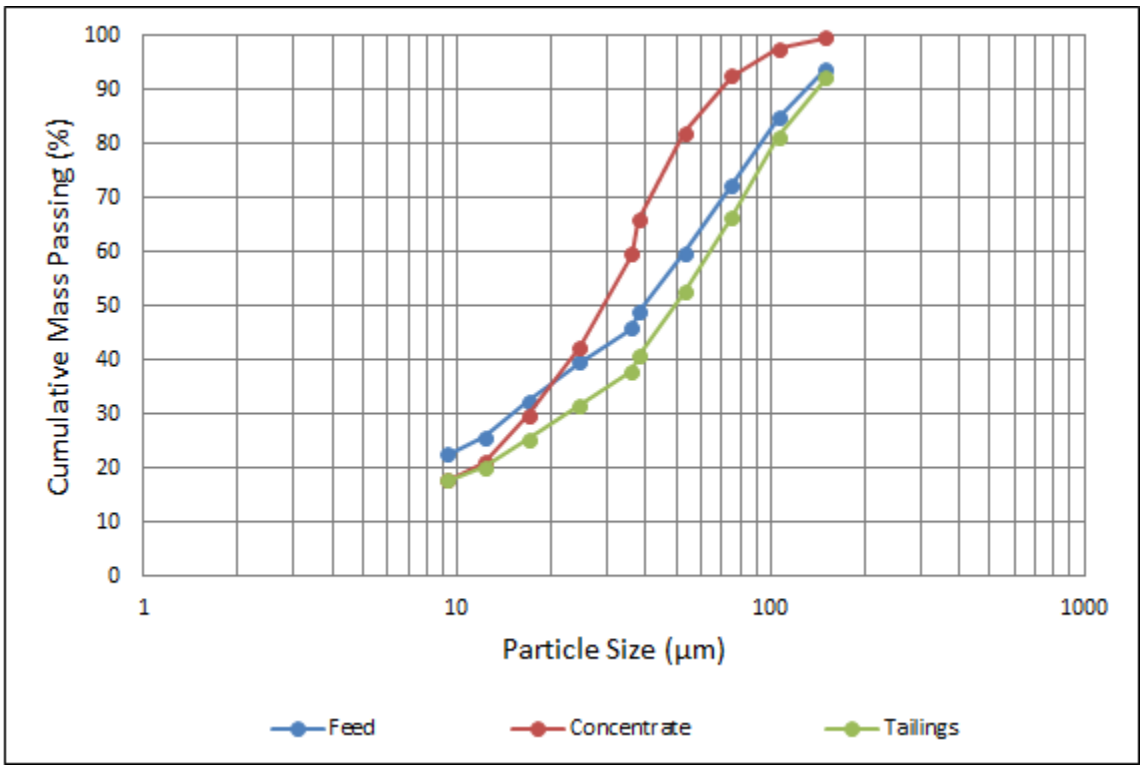


Figure 3-2: Sample Size Distributions

The plant target feed size distribution is 70% passing 75 µm, which corresponds to the sample analysed at 72% passing 75 µm. A comparison of the 80% passing size (P_{80}) of each of the streams is shown below:

- Feed: 94 µm

- Concentrate: 49 µm
- Tailings: 104 µm

The general observation from Figure 3-2 is that the concentrates are much finer than the feed and that the larger size fractions of the feed are not sufficiently floated and end up in the tailings stream. The composition of the tailings is detailed in section 3.3.2.

3.3.2 Elemental Composition

Assay by size information for the feed, concentrate and tailings is detailed in Table 3-1, Table 3-2 and Table 3-3 respectively.

Table 3-1: Feed Composition

Size Range	Mass	Feed Grade (%)					
µm	%	Ni	Cu	Fe	Mg	S	Si
+75	34.14	0.14	0.07	7.99	13.41	1.48	20.40
-75+38	25.22	0.58	0.17	14.07	10.68	5.22	17.66
-38+24.7	9.77	1.57	0.37	23.11	8.16	10.54	13.83
-24.7+9.3	13.35	0.86	0.30	15.37	10.06	5.95	16.81
-9.3	17.52	0.46	0.24	12.42	12.05	3.67	17.75
Total	100.00	0.54	0.19	12.76	11.52	4.29	18.12

Table 3-2: Concentrate Composition

Size Range	Mass	Concentrate Grade (%)					
µm	%	Ni	Cu	Fe	Mg	S	Si
+75	0.30	5.77	4.83	29.66	5.08	25.7	7.08
-75+38	1.09	10.28	3.59	37.38	1.93	33.4	2.50
-38+24.7	1.07	12.75	3.18	41.07	1.05	35.6	1.14
-24.7+9.3	0.94	10.68	3.98	34.63	3.12	29.9	4.11
-9.3	0.72	6.58	4.16	23.94	7.38	18.6	10.60
Total	4.11	10.04	3.76	34.79	3.16	30.1	4.27

Table 3-3: Tailings Composition

Size Range	Mass	Tailings Grade (%)					
µm	%	Ni	Cu	Fe	Mg	S	Si
+75	33.8	0.09	0.03	7.79	13.5	1.26	20.5
-75+38	24.1	0.14	0.02	13.03	11.1	3.95	18.3
-38+24.7	8.70	0.20	0.02	20.91	9.03	7.46	15.4
-24.7+9.3	12.4	0.12	0.02	13.91	10.6	4.14	17.8
-9.3	16.8	0.20	0.07	11.93	12.2	3.03	18.1
Total	95.9	0.13	0.03	11.82	11.9	3.18	18.7

The feed nickel grade was 0.54%, relatively high grade material, but not uncommon for MMZ ore. The iron and magnesium grades are typical at 11.8% and 11.9% respectively. The concentrate contained 10.0% nickel, 34.8% iron and 3.2% magnesium. The Fe/MgO ratio is important for the smelting process and should be in the range of 5-7. The concentrate had an Fe/MgO ratio of 6.6 and was, therefore, on specification.

Magnesium in the feed is higher in the coarser and ultra-fine material with the same trend identified in the concentrate. Nickel grades in the tailings are higher in the fines size fraction, however, a large proportion of the tailings material consists of coarse grained particles. Furthermore, very low grade nickel tends to be present in the coarser particle size. The overall recovery (relative to the flotation feed) of each element is presented in Table 3-4 and recovery by size shown in Figure 3-3 and Figure 3-4 for sulphides and non-sulphide gangue respectively. The majority of the nickel is recovered from the -75+9 µm fraction, representative of their distribution in the feed as shown in Table 3-1. Overall magnesium recovery is low, however, it is evident from the concentrate composition that ultrafine magnesium preferentially reports to the concentrate. This is indicative of entrainment.

Table 3-4: Overall Recovery

Element	Recovery
	%
Ni	76.22
Cu	83.32
Fe	11.22
S	28.84
Mg	1.13
Si	0.97

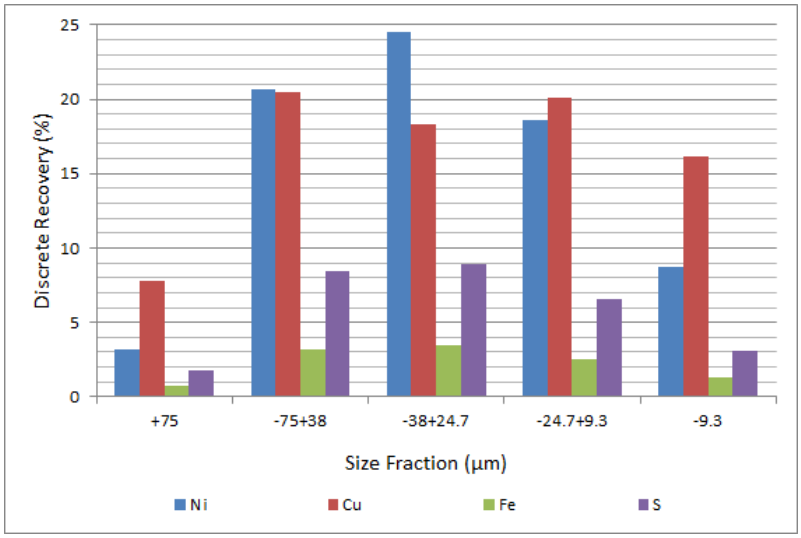


Figure 3-3: Discrete Elemental Recoveries to Concentrate – Sulphides

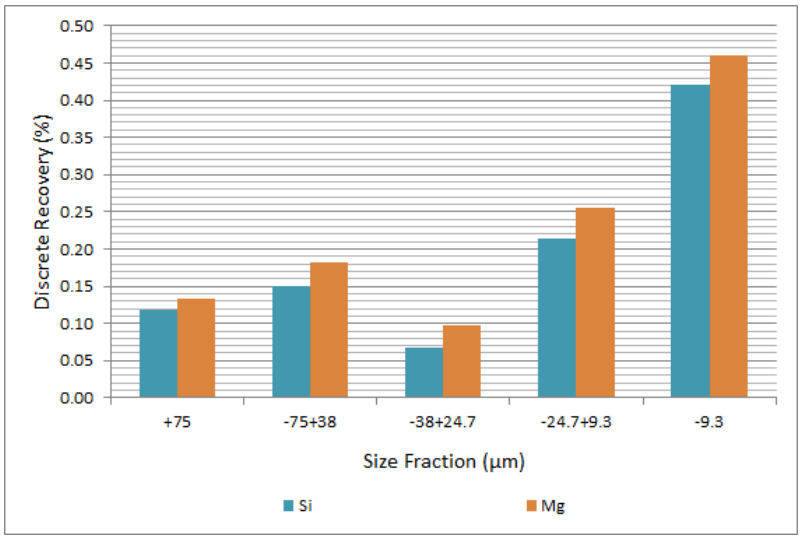


Figure 3-4: Discrete Elemental Recoveries to Concentrate – Non Sulphides

3.3.3 Mineral Composition

The mineral composition of each stream and mineral recoveries are presented in Table 3-5. These recoveries are based on a calculated concentrate mass pull to ensure that the recovery of all elements sum to 100%. The applied factors are presented in Appendix A.

Table 3-5: Mineral Compositions and Recoveries

Mineral	Mineral Formula	Feed	Conc	Tails	Recovery
		%			
Pentlandite	$(Fe,Ni)_9S_8$	1.33	30.6	0.21	86.1
Chalcopyrite	$(CuFeS_2)$	0.59	12.6	0.08	86.9
Pyrrhotite	$Fe_{1-x}S$	7.72	22.8	6.97	12.0
Pyrite	FeS_2	0.57	14.3	0.04	94.3
Olivine	$(Mg,Fe)_2SiO_4$	4.33	1.14	4.59	1.02
Orthopyroxene	enstatite $((Mg,Fe)SiO_3)$	4.31	2.12	4.97	1.75
Clinopyroxene	$(Ca(Mg,Fe)Si_2O_6)$; $((Ca,Na)(Mg,Fe,Al)(Si,Al)_2O_6))$.	18.3	1.41	19.1	0.31
Amphibole	$(Ca_2Mg_5Si_8O_{22}(OH)_2)$; $((Mg,Fe)_7Si_8O_{22}(OH)_2)$	24.7	3.73	24.5	0.63
Chlorite	$(Mg,Fe)_5Al(Si_3Al)O_{10}(OH)_8$	8.03	0.96	8.08	0.49
Serpentine	$(Mg,Fe)_3Si_2O_5(OH)_4$	11.9	3.42	10.5	1.34
Talc	$Mg_3Si_4O_{10}(OH)_2$	1.35	3.66	1.67	8.39
Micas	$(KMg_3(Si_3Al)O_{10}(F,OH)_2)$	3.02	0.25	3.36	0.31
Feldspars	$(NaAlSi_3O_8)$	3.27	0.39	4.06	0.40
Epidote	$Ca_2Al_2O(Al,Fe)OH(Si_2O_7)(SiO_4)$	1.88	0.18	1.97	0.38
Quartz	SiO_2	1.22	0.09	1.86	0.21
Carbonates	$(CaCO_3)$	2.17	0.25	1.97	0.53
Fe oxides/oxyhydroxides	Various	2.84	1.31	3.10	1.74
Total		100	100	100	-

The major sulphide minerals include pentlandite, chalcopyrite, pyrrhotite and pyrite. Recoveries of pentlandite and chalcopyrite were both around 86-87% and have a combined composition of 43%. The sulphide gangue minerals, pyrrhotite and pyrite, form 38% of the concentrate with nearly all the pyrite recovered to the concentrate. Despite low overall magnesium recovery (1.1%), talc tends to be selectively recovered. Talc comprises

3.7% of the concentrate. The other substantial magnesium-bearing mineral in the concentrate is serpentine at 3.4% however, its recovery too was only 1.3%.

Ni Deportment

The distribution of the primary nickel minerals, pentlandite and pyrrhotite, are presented in Figure 3-5 and Figure 3-6.

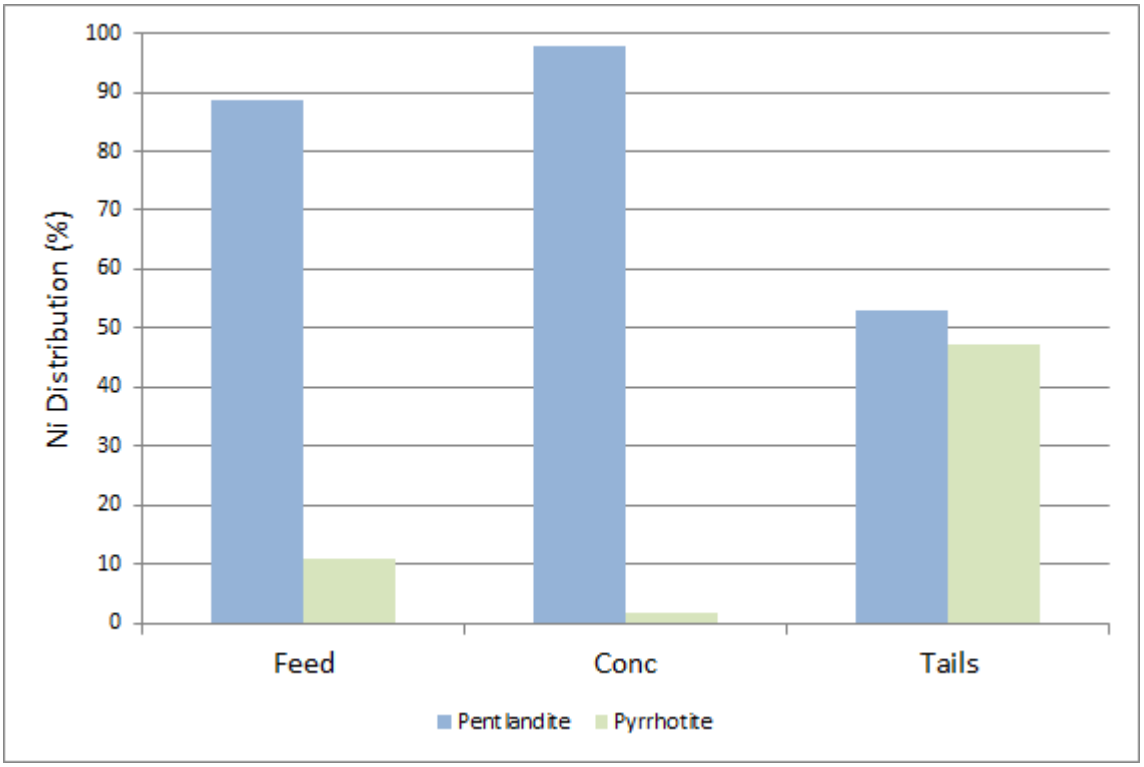


Figure 3-5: Nickel Deportment by Mineral in Feed and Products

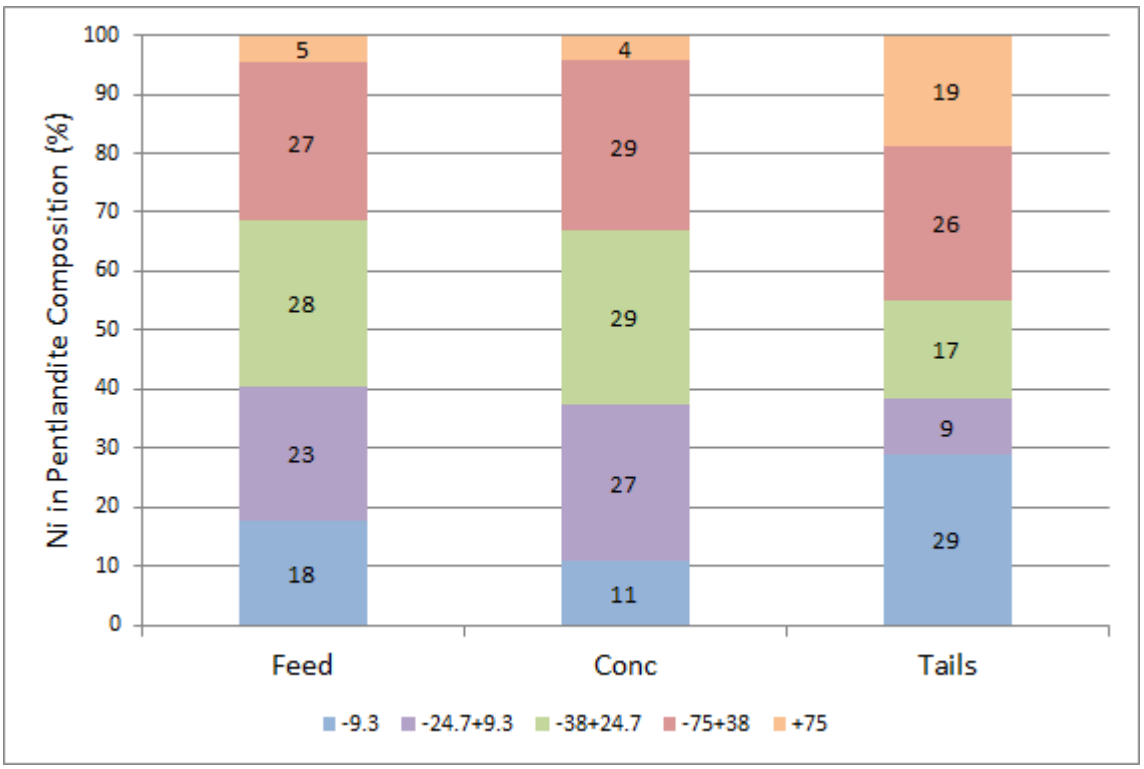


Figure 3-6: Feed and Product Deportment of Pentlandite Nickel by Size Fraction

The nickel in the concentrate is composed of approximately 98% pentlandite and 1.6% pyrrhotite. Nickel content in the pyrrhotite is approximately 0.75%, making it an uneconomical mineral to recover because it reduces the concentrate quality and, as a result, increases transportation costs. The pentlandite composition by size and liberation characteristics is presented in Table 3-6. Liberation is based on the mineral's fractional area per particle.

Table 3-6: Pentlandite Concentrate Liberation Data

Liberation Class (mineral area %)		**+75**	**-75+38**	**-38+24.7**	**-24.7+9.3**	**-9.3**	**Total**
Description	**Area**	**Pentlandite Mass in Concentrate (%)**					
Liberated	>90%	2.81	23.0	23.8	23.7	10.6	84.0
High grade middlings	60-90%	0.46	1.61	1.08	1.85	3.56	8.55
Low grade middlings	30-60%	0.28	0.98	0.53	0.85	1.77	4.42
Locked	<30%	0.35	0.59	0.43	0.62	1.04	3.03
Total		3.91	26.2	25.8	27.1	17.0	100.0

Nickel in the tailings consists of almost equal proportions of pentlandite and pyrrhotite. As per typical flotation recovery by size profiles (Figure 2-3), the mineral recovery of the fines

and coarse particles is poorer than that in the mid-sized particles. The pentlandite liberation by size, shown in Table 3-7, provides some insights as to the potential areas for improvement.

Table 3-7: Pentlandite Tailings Liberation Data

Liberation Class (mineral area %)		+75	-75+38	-38+24.7	-24.7+9.3	-9.3	Total
Description	**Area**	**Pentlandite Mass in Tailings (%)**					
Liberated	>90%	1.2	1.1	1.0	4.3	25.6	33.2
High grade middlings	60-90%	1.3	2.1	0.7	1.5	7.9	13.6
Low grade middlings	30-60%	1.6	4.3	2.1	2.9	4.0	14.9
Locked	<30%	9.5	13.7	5.6	4.8	4.9	38.4
Total		13.5	21.1	9.38	13.4	42.5	100.0

Approximately 25% of the pentlandite in the tailings is present as finely liberated grains (-9.3 μm). 53% of the unrecovered pentlandite was present in the locked and low grade middlings fractions, mostly associated with pyrrhotite. The unliberated pentlandite is closely associated with pyrrhotite, mainly as fine-grained particles. Pentlandite of the aforementioned types would most likely be difficult to recover. Additional recovery could be achieved by targeting the liberated and middlings coarse grained pentlandite. Improvement in the ultrafine recovery would be highly beneficial. Therefore, optimisation of milling could reduce ultrafine pentlandite generation and improve pentlandite recovery and it is recommended that a sampling and balance around milling circuit be conducted. Further testwork would be to identify the effect of grinding, with particular focus on ultrafine generation, on flotation performance.

The pyrrhotite liberation data for the concentrate is presented in Table 3-8.

Table 3-8: Pyrrhotite Concentrate Liberation Data

Liberation Class (mineral area %)		+75	-75+38	-38+24.7	-24.7+9.3	-9.3	Total
Description	**Area**	**Pyrrhotite Mass in Concentrate (%)**					
Liberated	>90%	2.38	22.5	27.1	21.1	6.96	80.0
High grade middlings	60-90%	0.96	2.91	2.28	2.42	3.70	12.3
Low grade middlings	30-60%	0.53	1.00	0.78	0.90	1.52	4.73

Locked	<30%	0.39	0.82	0.41	0.54	0.87	3.03
Total		4.26	27.2	30.6	24.9	13.1	100.0

Pyrrhotite reporting to the concentrate was predominantly liberated with the bulk being recovered in the -75+9.3 µm size fraction. Despite a large proportion of the feed consisting of -9.3 µm liberated pyrrhotite (23%), only a small fraction of this is recovered in the concentrate. The pyrrhotite content of the tailings is similar to that of the feed (7.0 % compared to 7.7 %, respectively), reflecting its slow-floating nature.

Non-Sulphide Gangue

Non-sulphide gangue deportment in the concentrate is presented in Table 3-9.

Table 3-9: Non-Sulphide Gangue Concentrate Liberation Data

Liberation Class (mineral area %)		**+75**	**-75+38**	**-38+24.7**	**-24.7+9.3**	**-9.3**	**Total**
Description	**Area**	**Non-Sulphide Gangue Mass in Concentrate (%)**					
Liberated	>90%	6.88	10.5	5.09	24.1	28.5	75.0
High grade middlings	60-90%	3.43	3.60	1.31	2.22	5.60	16.2
Low grade middlings	30-60%	0.87	1.17	0.56	0.56	2.39	5.55
Locked	<30%	0.42	0.94	0.46	0.32	1.09	3.25
Total		11.6	16.3	7.42	27.2	37.6	100.0

This gangue accounts for almost 20% of the concentrate. 75% of the non-sulphide gangue in the concentrate is liberated. Magnesium bearing minerals are the major non-sulphide gangue contaminating the concentrate. Approximately 8% of the talc in the feed is recovered to concentrate, preferentially in the fines. The size distribution of the talc in the samples are:

- Feed: P_{80} 86 µm
- Concentrate: P_{80} 35 µm
- Tailings: P_{80} 101 µm

The recovery of fine talc to the concentrate is significant, even in the liberated fraction. Approximately 85% of the talc in the concentrate is collected in the -24.7 µm fraction. This is not as significant as for the non-sulphide gangue overall. A major opportunity exists to depress gangue more effectively to improve concentrate quality. Importantly, as can be

seen from Table 3-6, depression of gangue may result in reduced nickel recovery of the locked and middlings fractions.

For all the above mentioned major minerals, about 5-10% of the minerals in the concentrate consists of locked and low grade middlings.

3.4 Process Survey

The overall circuit block flow diagram and mass balance is presented in Figure 3-7 and Table 3-10 respectively. A more detailed mass balance showing the flows, grades and recoveries for each stream in the various flotation stages is provided in Appendix A – Detailed Mass Balance.

The nickel grade of the rougher feed was 0.33%. The nickel recovery for the rougher bank was 80.3% at a mass pull of 12.8%. Nickel recovery for the scavenger bank was 3.2% at a mass pull of 1.8%. Therefore, the combined rougher and scavenger recovery was 83.5% at a mass pull of 14.5%. The MgO grade in the rougher concentrate was slightly higher than in the feed, which is typical of normal operation at the MMZ plant and is evidence of the naturally hydrophobic talc. Additional depressant could assist in reducing the recovery of MgO to the rougher concentrate, reducing the load to the cleaner circuit. In comparison to laboratory testwork, the rougher mass pull on the plant is significantly higher than that typically achieved in the laboratory (typically around 7-8%), however, recovery is comparable. The overall plant recovery for nickel was 72.5%, therefore, an 87% recovery over the cleaning stages was achieved. MgO is reduced from 20% in the feed to 6% in the concentrate with cleaning only achieved in the cleaner and re-cleaner stages.

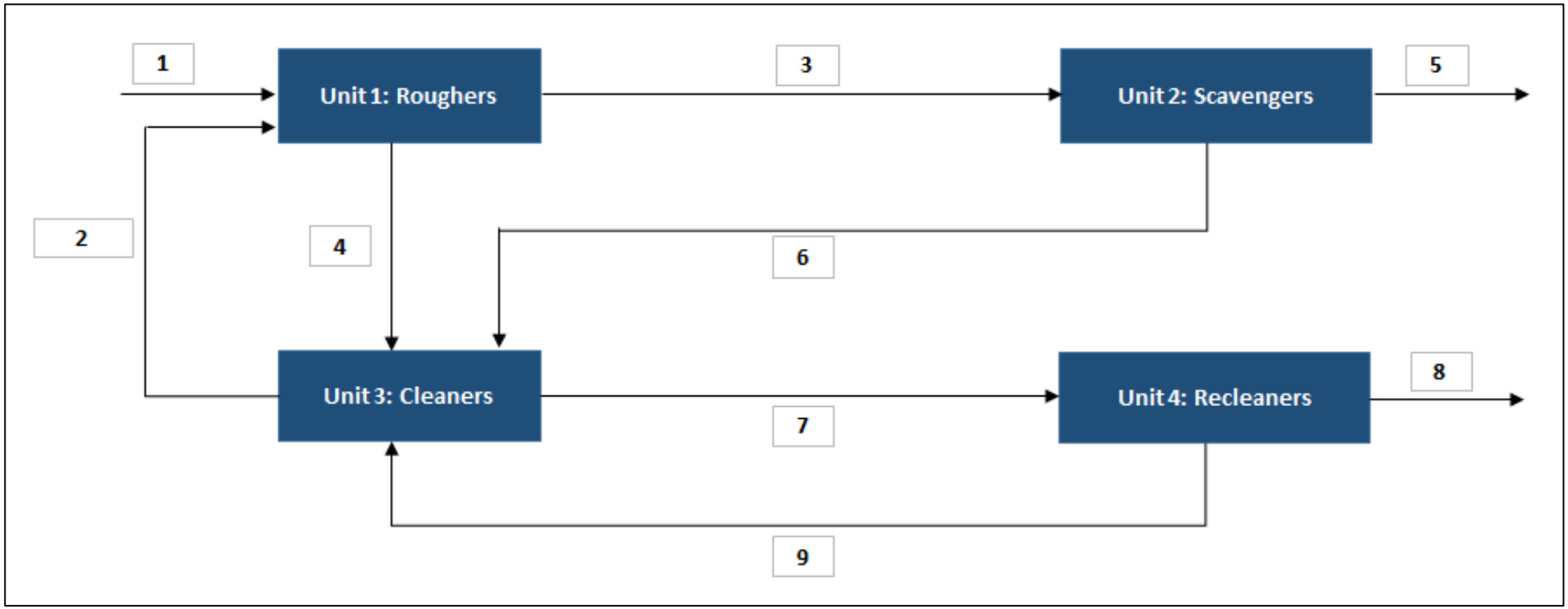


Figure 3-7: MMZ Plant Basic Flotation Block Flow Diagram

Table 3-10: MMZ Plant Overall Mass Balance

Stream		Mass (%)	Grade (%)				Recovery (%)				Solids Conc (%)
			Ni	Cu	MgO	Fe	Ni	Cu	MgO	Fe	
1	Flotation Feed	100.00	0.33	0.13	20.40	12.12	100.0	100.0	100.0	100.0	24
2	Cleaner Tails	12.21	0.30	0.05	24.60	9.72	11.0	4.9	14.7	9.8	31
3	Rougher Tails	99.47	0.10	0.01	20.81	11.63	30.8	10.3	101.5	95.4	24
4	Rougher Conc	12.74	2.09	0.99	21.19	13.67	80.3	94.7	13.2	14.4	31
5	Scavenger Tails	97.68	0.09	0.01	20.75	11.67	27.5	8.7	99.3	94.0	24
6	Scavenger Conc	1.79	0.60	0.11	24.33	9.48	3.2	1.5	2.1	1.4	21
7	Cleaner Conc	6.10	5.87	2.20	10.99	27.18	107.7	101.1	3.3	13.7	16
8	ReCleaner Conc	2.32	10.36	5.21	5.71	31.19	72.5	91.3	0.7	6.0	24
9	ReCleaner Tails	3.78	3.10	0.35	14.24	24.71	35.2	9.8	2.6	7.7	11

Collector is added to the rougher feed, rougher cells 3 and 4 (due to re-circulation of cleaner tailings to the roughers) and scavenger feed. Depressant is added to the rougher feed, scavenger feed, cleaner feed, cleaner cell 3 (i.e. stage dosing point) and re-cleaner feed. Table 3-11 indicates the reagent dosages and distribution within each bank. It is evident that stage dosing is practiced, however, this is mainly at nodal points. The only point where stage dosing occurs on a single stream is depressant dosage in cleaner cell 3 in the cleaner bank. The majority of reagents are typically added upfront.

Table 3-11: MMZ Plant Reagent Dosages

Bank	Units	Upfront	Stage 1	Stage 2	Total
Collector					
Roughers and Scavengers	g/t	140	40	40	220
	%	64	18	18	100
Depressant					
Roughers and Scavengers	g/t	150	60	0	210
	%	71	29	0	100
Cleaners	g/t	80	20	0	100
	%	80	20	0	100
Re-Cleaners	g/t	160	0	0	160
	%	100	0	0	100
Total	g/t	-	-	-	470

The effect of stage dosing of collector at the rougher 3 and 4 and scavengers on nickel is not clearly identified from the general grade-recovery curve. A graph of the nickel grade-recovery over both the roughers and scavengers is shown in Figure 3-8.

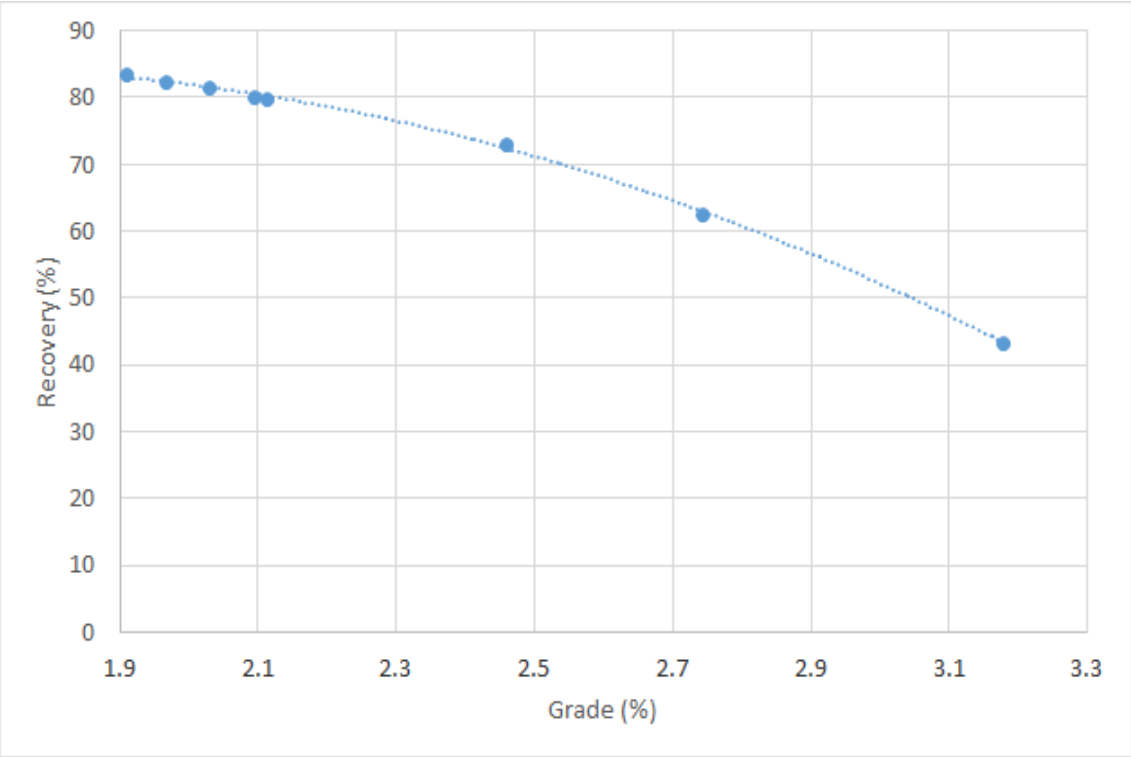


Figure 3-8: MMZ Plant Rougher and Scavenger Flotation Nickel Grade-Recovery Curve

The three data points, on the right hand side of the graph, represent the scavenger concentrates. There is no change in the nickel grade-recovery curve from roughers to scavengers, therefore, it is unclear whether the process is enhanced by the stage dosing of collector from the process survey. More detailed testwork and sampling would be required.

The MgO grade-recovery data is provided in Table 3-12.

Table 3-12: MMZ Plant Rougher and Scavenger MgO Grade-Recovery Data

Stream	Mass Pull (%)	MgO			
		Grade (%)	Recovery (%)	Cum Grade (%)	Cum Recovery (%)
Ro Conc 1 & 2	4.55	23.21	5.18	23.21	5.18
Ro Conc 3 & 4	3.05	19.50	2.92	21.72	8.09
Ro Conc 5 & 6	2.29	21.36	2.40	21.64	10.49
Ro Conc 7 & 8	2.68	19.46	2.55	21.17	13.05

Ro Conc 9 & 10	0.17	22.51	0.19	21.19	13.24
Sc Conc 1 & 2	0.60	22.98	0.68	21.27	13.91
Sc Conc 3 & 4	0.60	24.83	0.73	21.42	14.64
Sc Conc 5 & 6	0.60	25.18	0.74	21.58	15.37
Rougher Feed	100.00	20.40	100.00	20.55	-

Considering that the MgO grade increases from the roughers to scavengers indicates that that the rate of MgO collection to the concentrate is retarded by depressant addition, but collection is still significant due to naturally floating and froth stabilising characteristics. Stage dosing testwork, covered in Chapter 5, will provide some details on the effects of stage dosing of both the collector and depressant.

The cleaner bank nickel recovery was 90.7% with concentrate grades of 5.9% nickel and 11.0% MgO. 80 g/t Depressant was added to the cleaner feed and the MgO grade dropped from 24.3% to 12.3%, the grade remained similar in the second concentrate. A further 20 g/t depressant was added to the cleaner cell 3 feed and the MgO grade reduced to 6.4%, with the grade increasing thereafter in cleaner cells 4 and 5 concentrates. This shows the potential benefit of stage dosing and could be beneficial to the re-cleaners as well. Table 3-13 presents the performance of the cleaner circuit with:

1. *Mass pull* data relative to the rougher feed
2. *Recovery* data relative to the cleaner feed (this includes the re-cleaner tailings stream).

Table 3-13: MMZ Plant Cleaner Grade-Recovery Data

Stream	Mass Pull (%)	Ni		MgO	
		Grade (%)	Recovery (%)	Grade (%)	Recovery (%)
Cl Conc 1	2.61	5.61	37.06	12.27	7.19
Cl Conc 2	1.51	5.92	22.61	12.27	4.15
Cl Conc 3	0.90	9.29	21.25	6.43	1.30
Cl Conc 4	0.18	3.41	1.54	16.82	0.67
Cl Conc 5	0.90	3.60	8.25	12.84	2.61
Cl Feed	18.31	2.16	90.72	24.31	15.93

The overall recovery during this campaign was 72.5%. The recovery in the roughers and scavengers are good, but any improvement in recovery could be attained through the cleaner stage. This is identified by the high cleaner tails grade (0.3% on this day). As an example, if the grade in the cleaner tails were instead 0.25%, the cleaner recovery would have improved by approximately 2% (gangue collection and floatability/depression would have to be considered as well).

The re-cleaner bank recovery was 67%, however, the MgO grade was 5.7%. The grade-recovery profile is peculiar. The first concentrate had a significantly lower nickel grade than concentrates 2 and 3 and vice versa for the MgO. This could be an indication of :

1. over-depressing froth stabilising gangue in the first cell and perhaps stage dosing could be a better approach. However, this assumption would have to be verified through more sampling and testwork. Stage dosing can be tested more effectively using a laboratory experiment.
2. too high a mass pull from the first cell, resulting in entrainment.

Table 3-14: MMZ Plant Re-Cleaner Grade-Recovery Data

Stream	Mass Pull (%)	Ni		MgO	
		Grade (%)	Recovery (%)	Grade (%)	Recovery (%)
Re-Cl Conc 1	1.62	9.16	41.35	6.00	14.47
Re-Cl Conc 2	0.31	13.15	11.54	4.87	2.28
Re-Cl Conc 3	0.40	13.06	14.41	5.20	3.07
Re-Cl Feed	6.10	5.87	67.31	10.99	19.82

3.4.1 Summary of Characterisation Testwork

Approximately 86% of pentlandite was recovered from the flotation process. The majority of nickel losses are in pyrrhotite and fine liberated pentlandite. A small proportion of the nickel can be recovered as coarse pentlandite. There is potential to increase recovery of the liberated as well as middlings pentlandite. Milling optimisation to reduce ultrafine generation should improve pentlandite recovery.

Approximately 20% of the concentrate consists of non-sulphide gangue, the majority being liberated and present as fines. Furthermore, 14% of the concentrate consists of pyrite and due to its fast floating nature, would need special treatment to reduce its concentration in the product. Improvement in concentrate grade could be obtained by depressing pyrite and pyrrhotite reporting to concentrate, at the cost of some nickel recovery. Optimisation of talc and other magnesium bearing minerals depression will be beneficial as well.

Reasonable nickel recovery was achieved on the rougher-scavenger flotation circuit, but at the expense of a high talc recovery. Additional depressant could improve MgO grades in the rougher concentrate and improve the performance in the cleaner circuit. Additional depressant or less air could be possible solutions to this. Stage dosing of collector on the rougher flotation circuit (at nodal points) does not show a visible impact on the process, however, stage dosing of depressant in the cleaner flotation cells clearly showed suppression of MgO to the concentrate. It is recommended that stage dosing of depressant be tested in all the flotation stages. The nickel recovery on the cleaner circuit was low – a mass balance analysis on the control assays should be setup for plant personnel to easily identify major areas of focus to optimise recovery. There were indications that froth stabilising gangue could be over-depressed in the re-cleaner flotation stage and a stage dosing approach at a reduced reagent dosage may be beneficial.

4. Chapter 4: Interaction between Collector and Depressant

4.1 Introduction

Laboratory scale testwork was conducted to understand the effects of the collector and depressant individually as well as the interaction when added together. It is hypothesised that an optimum addition of collector and depressant dosage exists to recover valuable mineral and reject gangue. The areas covered in this chapter are:

- Procedure used for testwork
- Flotation without collector and depressant
- Effect of collector addition
- Effect of depressant addition
- Effect of combined collector and depressant addition

4.2 Experimental Procedure

4.2.1 Equipment and Procedure

All sample splitting was conducted using a rotary splitter.

Sizing was conducted using a Pascal shaker for dry screening and a vibrating shaker for wet screening. Dry screening was conducted on all screen sizes above 38 µm and wet screening was conducted on the 38 µm screen.

Crushing was conducted using laboratory jaw crushers. Two crushers were used to achieve the feed size to the mill. The closed side setting for the two crushers were:

Crusher 1: CSS = 10 mm

Crusher 2: CSS = 2 mm

Milling was conducted in a laboratory mill with mill diameter of 270 mm and mill length of 310 mm. The mill was filled with 31 kg of stainless steel rods to achieve a target charge of approximately 30% (v/v). A picture of the lab mill is provided in Figure 4-1.



Figure 4-1: Laboratory Mill Rig

Flotation testwork was conducted in a 5 litre Denver flotation cell. A picture of the flotation cell in operation is presented in Figure 4-2.

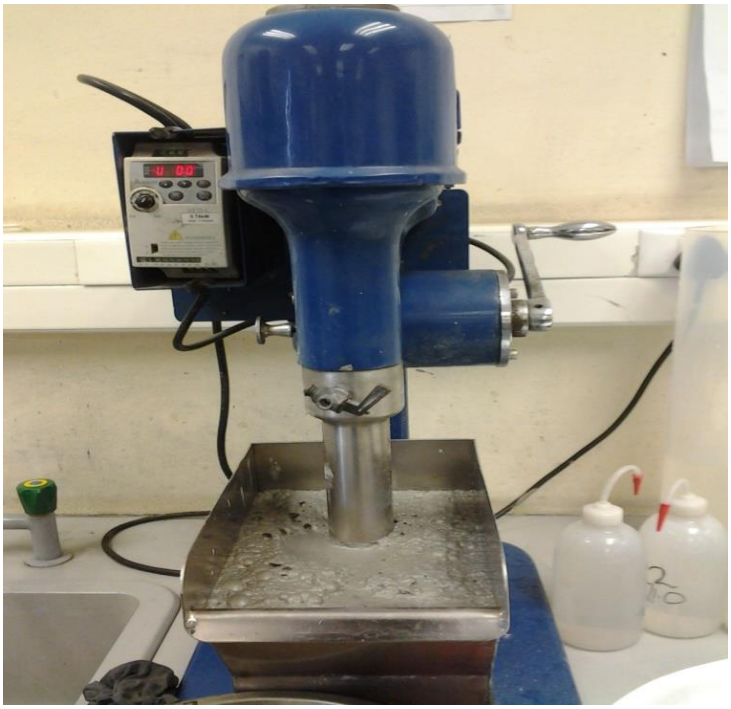


Figure 4-2: Laboratory Denver Flotation Cell

4.2.1.1 Sample Preparation

A plant feed sample was collected as a belt cut during typical operation. The feed sample was air dried and crushed in a laboratory jaw crusher to 100% passing 10 mm. The -10 mm material was then processed in another laboratory jaw crusher to achieve a mill feed of 100% passing 2 mm. The crushed material was rotary split into 2 kg batches. A 100 g sub-sample was removed for head chemical analysis. One 2 kg sample was used for mill feed assay by size analysis (see section 4.2.1.3).

4.2.1.2 Milling

A milling curve was generated to determine the milling time required to achieve the target grind of 67-70% passing 75 µm. Milling was conducted at approximately 62.5% solids concentration. Four samples were used for milling curve construction. One samples was not milled, and the other three samples were milled for 15, 25 and 30 minutes The four samples were then wet screened at 75 µm. The products were dried and weighed. The time determined to mill to 67% passing 75 µm was 21 minutes. The milling curve is presented in Appendix B.

A separate sample was milled for 21 minutes to conduct assay by size analysis on the mill product/flotation feed. All other flotation test samples were milled for 21 minutes; collector (SIBX) and promoter (thionocarbomate) were added to the mill before milling.

4.2.1.3 Assay by Size Analysis

An assay by size analysis was conducted on the mill feed and product. The samples were initially wet screened at 38 µm. The +38 µm fraction was dried, lumps removed by rolling and screened over the entire screen range (inclusive of the 38 µm sieve). The screen sizes used for each sample is presented in Table 4-1.

Table 4-1: Screen Aperture Sizes for Mill Feed and Product Sizing

Screen Size (µm)	
Mill Feed	**Mill Product**
1700	
1180	
850	
600	
425	
300	
212	212
150	150
106	106
75	75
53	53
38	38

4.2.1.4 Flotation

All flotation testwork was conducted at the natural pH of the slurry. This was measured at 9.5. The standard laboratory and plant reagents were used for testwork and current plant dosages were used as a guideline for the testwork. The following reagents were used for testwork:

- Promoter: Senkol 700 (isopropyl ethyl thionocarbomate), added to the mill feed
- Collector: Sodium Isobutyl Xanthate (SIBX), added to the mill feed
- Frother: Dowfroth 200 (tripopylene glycol methyl ether), added during conditioning
- Depressant: Carboxy Methyl Cellulose (CMC), added during conditioning

Testwork was conducted using Nkomati process water. Collector and promoter were added to the sample just before milling. The sample was milled for 21 minutes. After milling, the sample was placed in the flotation cell and process water added. Water was added to a predetermined level that ensured that the froth height was approximately 10-15

mm thick, once the agitator was switched on and the air introduced to the cell. After the water had been added, the agitator was switched on, the frother was added and conditioned for 1 minute. This was followed by the addition of depressant and conditioning for an additional 3 minutes. The air was then introduced at a rate of 7 l/min and flotation commenced. Concentrates were collected after 1 minute, 3 minutes, 10 minutes and 30 minutes of flotation.

Concentrate and tailings samples were dried in a laboratory oven at 80 ºC. Samples were weighed wet and dry. A sub-sample of each of the flotation products were submitted for chemical analysis. All testwork was conducted in duplicate.

4.2.1.5 Chemical Analysis

All samples submitted for chemical analysis were analysed by the following techniques:

1. Feed and Tailings samples were digested and analysed by the Inductively Coupled Plasma (ICP) method. The following elements and compounds were determined via this technique:
   Ni, Cu, Co, Fe, MgO, SiO_2, Cr_2O_3, Al_2O_3
2. Concentrate samples were digested and analysed by both ICP and atomic absorption (AA). The following elements were determined by AA:
   Ni, Cu, Co
   The other elements were determined using the ICP method.

The above techniques were specifically developed for the Nkomati ore at the Nkomati laboratory. All samples were analysed at the Nkomati site laboratory.

Detailed methodology for analytical method is provided in Appendix B.

4.2.2 Experimental Programme

Table 4-2 outlines the testwork programme.

Table 4-2: Laboratory Scale Testwork – Interaction between Reagents

Test Number	Collector	Depressant
#	g/t	g/t
1	0	0
2	50	0
3	50	50
4	50	150
5	50	300
6	100	0
7	100	50
8	100	150
9	100	300
10	200	0
11	200	50
12	200	150
13	100	500
14	0	100

4.3 Feed Analysis

Cumulative particle size distribution and discrete nickel distribution for the flotation feed is presented in Figure 4-3.

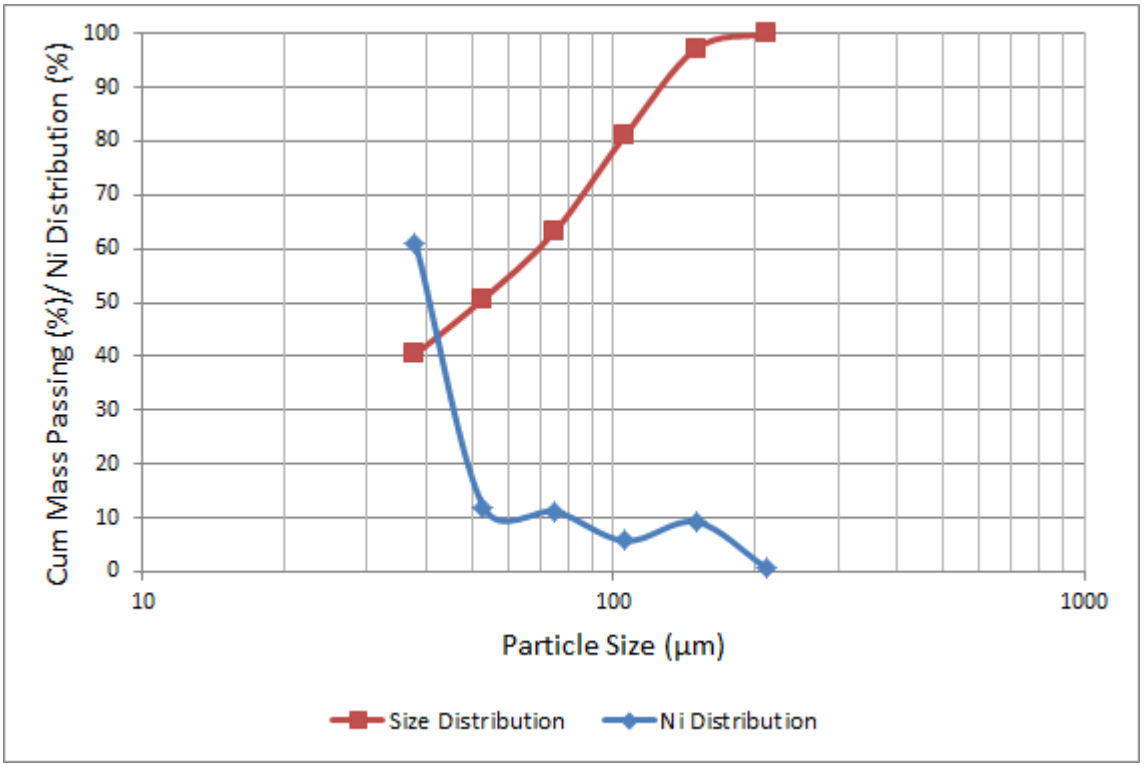


Figure 4-3: Feed Size and Discrete Nickel Distribution

The flotation feed particle size distribution was 63% passing 75 µm, slightly coarser than the target grind of 67% passing 75 µm. 60% of the nickel reports to the -38 µm fraction, while 15% of the nickel is present in the +75 µm fraction.

4.4 Base case float

Two mass pull figures are presented below, the solids and water mass recovery in Figure 4-4 and the mass pull in comparison with recovery in Figure 4-5. Note that solids and water mass recovery are relative to the total concentrate recovered over the flotation period. The grade-recovery curve of the base case float is presented in Figure 4-6.

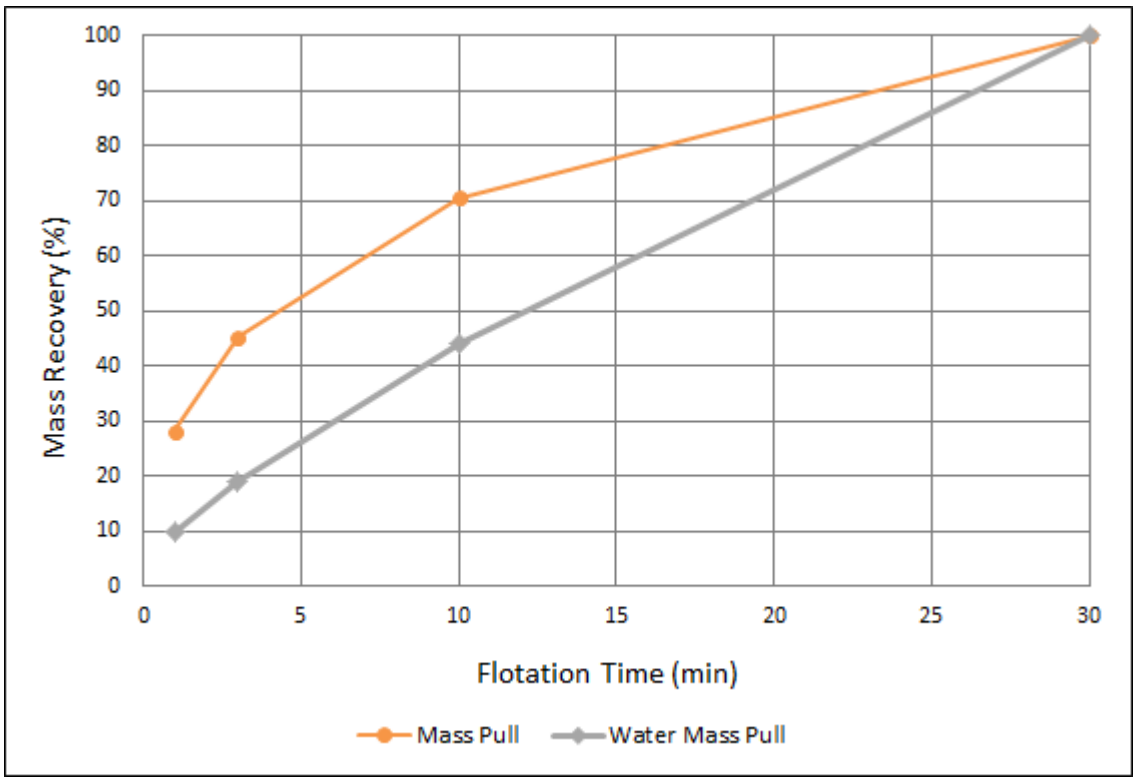


Figure 4-4: Base Case Mass Recovery

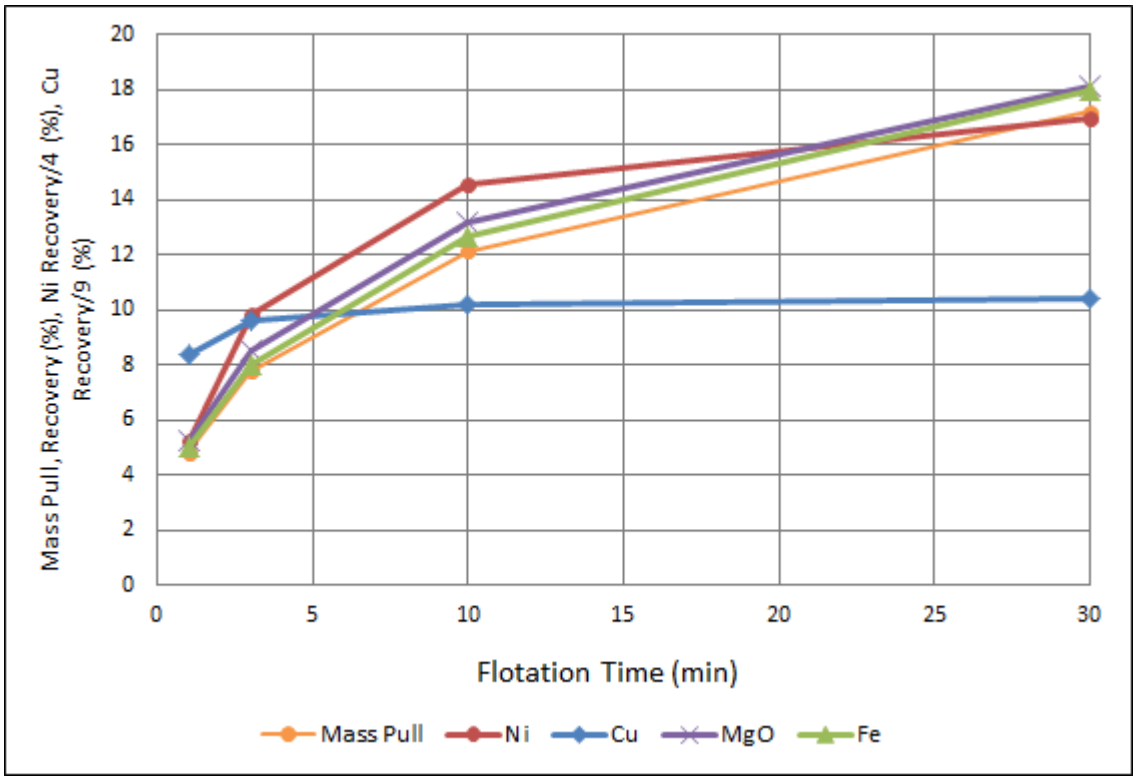


Figure 4-5: Base Case Elemental Recovery vs Mass Pull Curve (Cumulative Data)

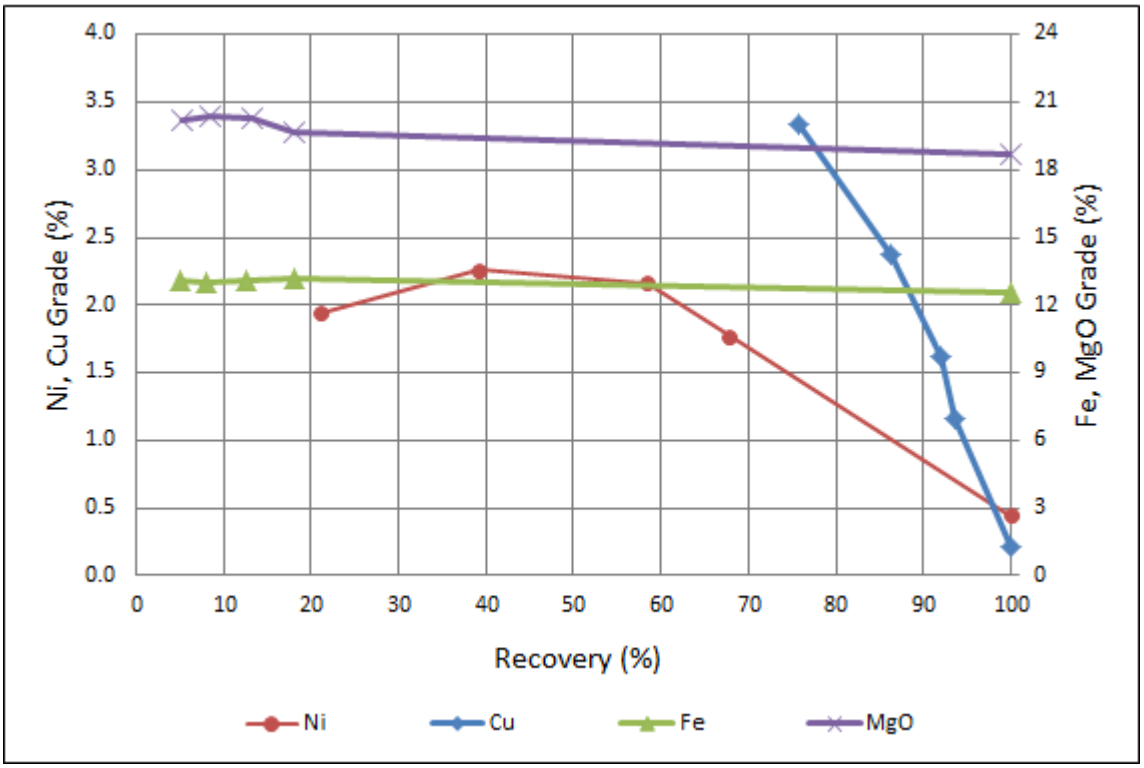


Figure 4-6: Base Case Grade-Recovery Curve

The rate of solids mass collected tends to decrease with time, however, the rate of water recovered to the concentrate has a strong linear trend. The solids mass pull for the test was 17.2%. The nickel and copper sulphides are faster floating than the iron and MgO. Copper (chalcopyrite) floats the fastest, with the highest initial recovery and total recovery. The concentrate nickel grade increases from the first concentrate (collected after 1 minute) to the second concentrate (collected after 3 minutes) due to reduced competition from the copper. The copper crowds the air bubble in the initial stages, resulting in the characteristic nickel grade recovery curve. The iron and MgO grades are similar in each of the concentrate (with very slight upgrade in the concentrate) and recovery is directly proportional to the mass of solids. The upgrade ratios for nickel, copper, iron and MgO were 3.94, 5.44, 1.05 and 1.05 respectively. It is clear that preferential flotation is not significant for iron and MgO under these flotation conditions. The liberation data for pyrrhotite in the feed, final concentrate and tailings is very similar:

- Locked: 80-85%
- High Grade Middlings: 10-12%
- Low Grade Middlings: 3-5%

- Locked: 2-3%

Initially, the concentrates should be dominated by chalcopyrite, pentlandite and pyrite while in the later stages the float is dominated by pyrrhotite. However, due to the consistent grade of iron in the concentrate, it is deduced that a fair amount of pyrrhotite is collected throughout the float. Therefore, although pyrrhotite is well known to be slow floating, its recovery to the concentrate is only slightly inhibited in the first few minutes of flotation but offset by the recovery of other iron sulphides (chalcopyrite, pyrite and pentlandite). This is possibly due to the high concentration of pyrrhotite in the feed and association with other sulphides. Similar behaviour has been reported on the flotation of pyrrhotite in other ores (Becker, 2009). The pyrite liberation shows that the tailings contains less liberated pyrite and more locked pyrite. Based on the mineralogical evaluation, it is estimated that the iron in the feed consists of the following:

- Pentlandite, chalcopyrite and pyrite: 10%
- Pyrrhotite: 50%
- Non-sulphide gangue: 40%

According to the mineralogical evaluation in Chapter 2 (Table 3-5), talc only forms 3.6% of the concentrate while all magnesium bearing minerals consist of approximately 17%. The liberation data indicates that liberated and high grade middlings talc floats preferentially. Considering the major contribution of other primary and secondary silicate minerals to the concentrate, it is understandable that the upgrade ratio of magnesium in the base case float is not high. As with iron, the recovery of silicates is recovered fairly consistently over the flotation period.

4.5 Effect of Collector – No Depressant

The effect of collector addition was explored to identify the exclusive effect that a collector has on the flotation of the major elements. The mass pull and grade-recovery curves for each of the elements are presented in Figure 4-7 to Figure 4-11 and a summary of the results is provided in Table 4-3.

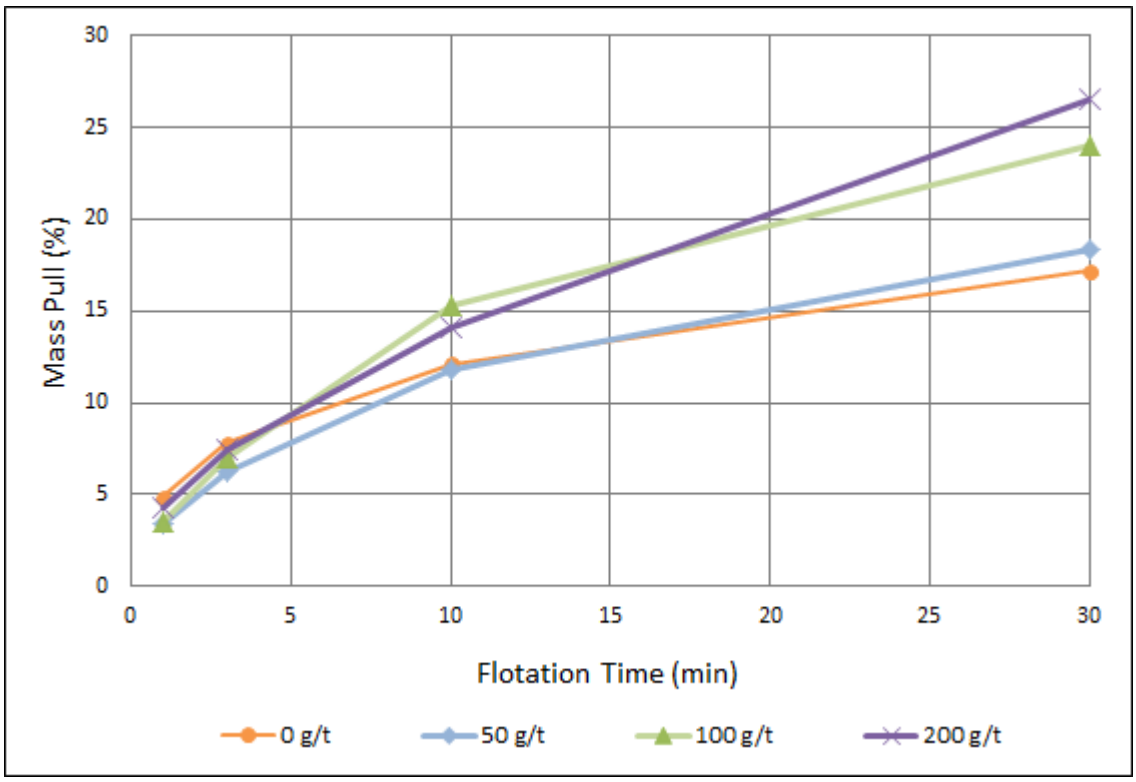


Figure 4-7: Effect of Collector Dosage on Mass Pull

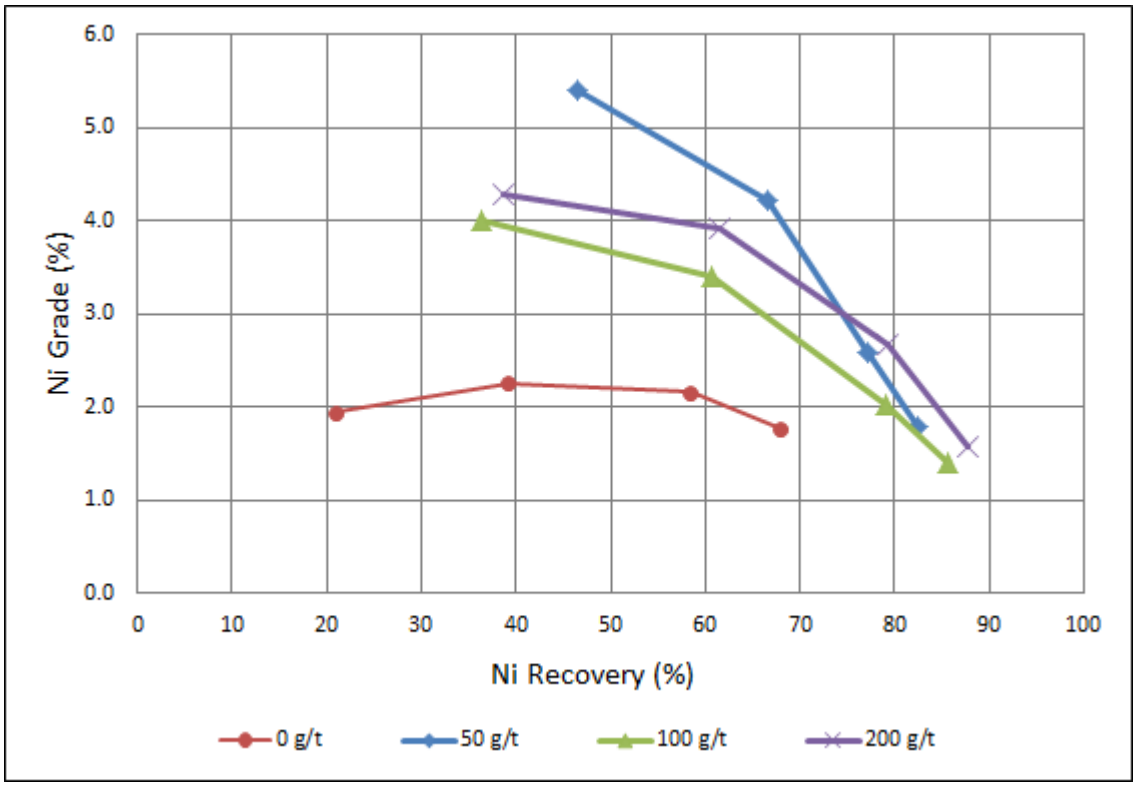


Figure 4-8: Effect of Collector Dosage on Ni Grade-Recovery Curve

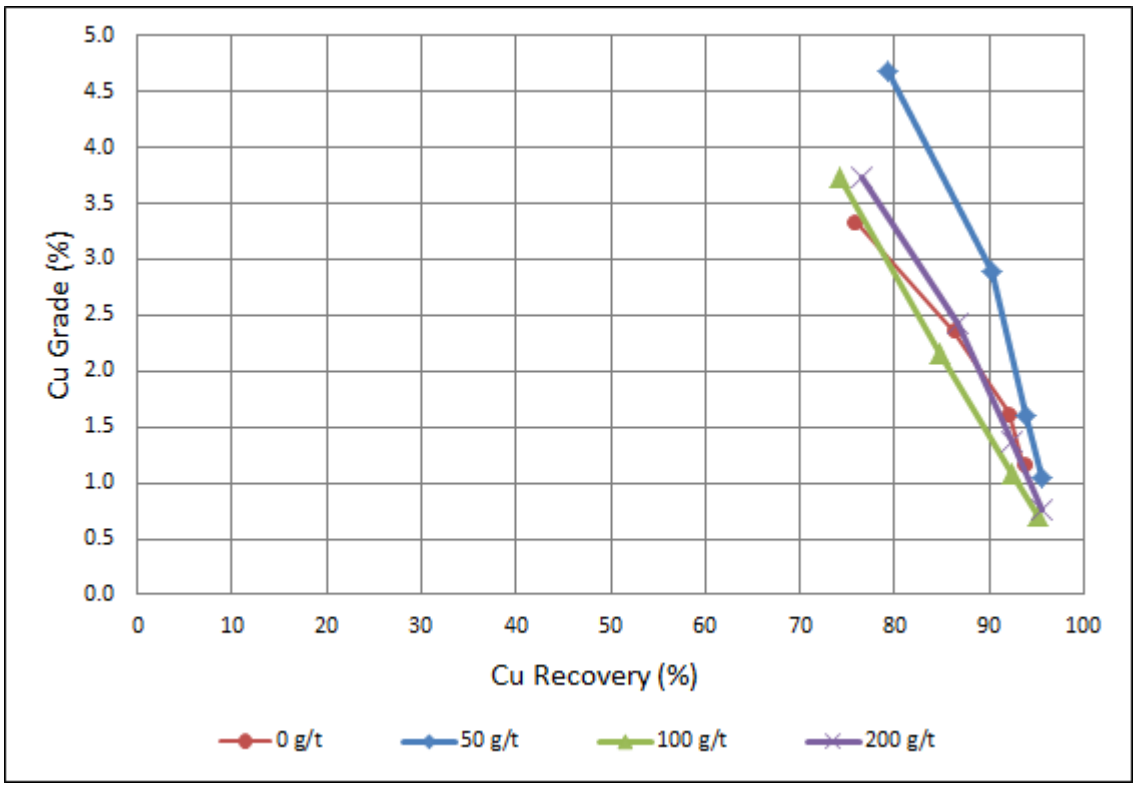


Figure 4-9: Effect of Collector Dosage on Cu Grade-Recovery Curve

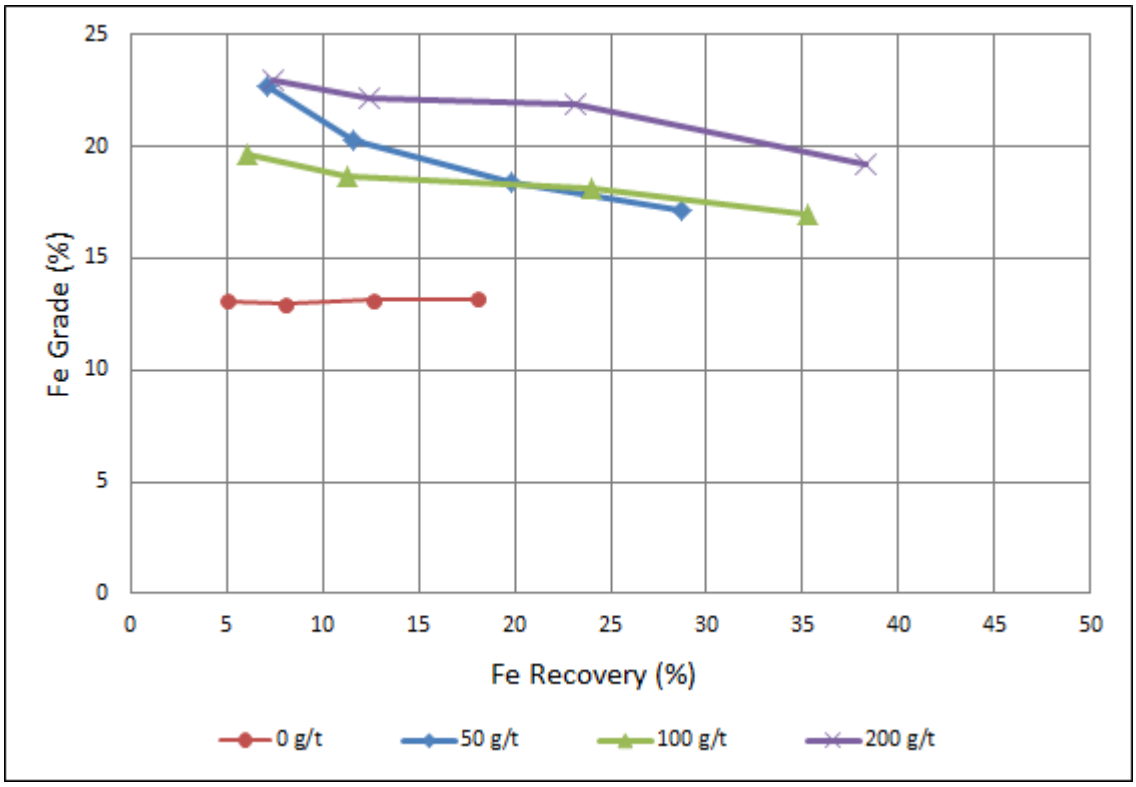


Figure 4-10: Effect of Collector Dosage on Fe Grade-Recovery Curve

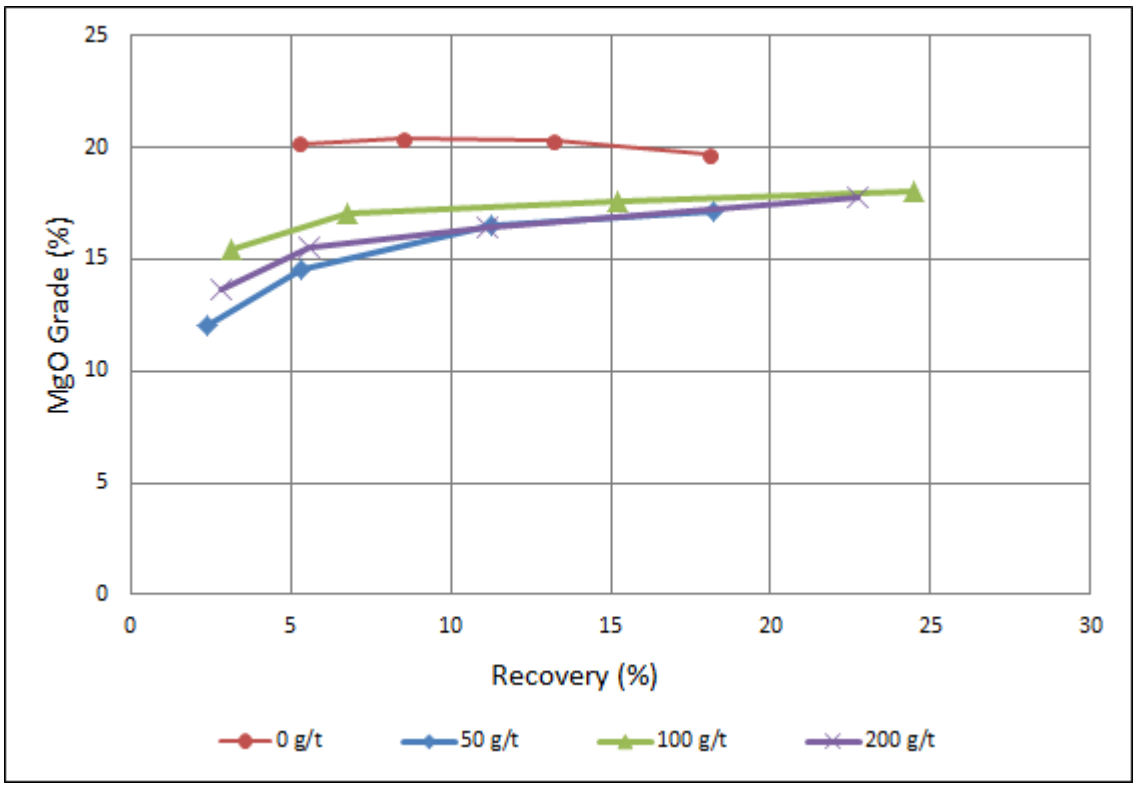


Figure 4-11: Effect of Collector Dosage on MgO Grade-Recovery Curve

Table 4-3: Summary of Final Concentrate Results for Testwork Without Depressant

SIBX (g/t)	Mass Pull (%)	Final Concentrate Grade (%)				Final Recovery (%)			
		Ni	Cu	Fe	MgO	Ni	Cu	Fe	MgO
0	17.20	1.77	1.16	13.16	19.66	67.8	93.6	18.0	18.1
50	18.33	1.79	1.05	17.16	17.14	82.4	95.5	28.7	18.2
100	24.06	1.40	0.71	16.98	18.01	85.7	95.1	35.3	24.5
200	26.58	1.57	0.75	19.24	17.75	87.8	95.5	38.3	22.8
Feed	-	0.39	0.19	11.79	18.77	-	-	-	-

The results indicate that the addition of collector increases the recovery of all the major elements. Due to the strong natural hydrophobicity of chalcopyrite, the copper recovery only increases by approximately 4% between dosages of 50 g/t and 200 g/t collector. Nickel recovery is increased by approximately 20% to 30% (relative to the base case), with a significant increase observed at 50 g/t and only marginal increases in 100 g/t and 200 g/t dosages. The extremely fast floating nature of chalcopyrite is noted by the significant increase in copper recovery at 50 g/t, after which, additional collector does not provide any

benefit. However, nickel recovery is not maximised at this dosage. The addition of collector is particularly effective for recovery of iron, doubling its recovery at higher dosages. It is postulated that the addition of collector, thus, has a significant impact on the pyrrhotite recovery due to the low composition of iron in other sulphide minerals. This is also seen in the grade recovery curves where the concentrate grade reduces at higher reagent dosages. A selectivity ratio was calculated to determine whether collector preferentially targeted any of the minerals. The ratio was defined as the relative improvement in recovery divided by the relative improvement in the mass pull (with the base case used as the reference).

$$Selectivity\ Factor = \left(\frac{Recovery\ @\ x\,{}^{g}/_{t}}{Recovery\ @\ 0\,{}^{g}/_{t}}\right) \div \left(\frac{Mass\ Pull\ @\ x\,{}^{g}/_{t}}{Mass\ Pull\ @\ 0\,{}^{g}/_{t}}\right) \qquad \text{Equation (2)}$$

Where: Recovery x g/t refers to the recovery of the element for the specific dosage in g/t
Recovery 0 g/t refers to the recovery for the collectorless test
Mass pull x g/t refers to the mass pull of the concentrate for the specific dosage in g/t
Mass pull 0 g/t refers to the mass pull for the collectorless test

Table 4-4: Selectivity Factor for Collector Addition

SIBX (g/t)	Selectivity Factor			
	Ni	Cu	Fe	MgO
50	1.14	0.96	1.50	0.94
100	0.90	0.73	1.40	0.97
200	0.84	0.66	1.38	0.81

The preferential increase in iron and nickel recovery, especially in the case of 50 g/t addition, infers that the sulphides are more selectively, but not exclusively, targeted by the collector. Due to the natural hydrophobicity of chalcopyrite, the selectivity factor is not that high. The rate of nickel and copper recovery tends to decrease at higher dosages as indicated by low selectivity factors at higher dosages. The same trend is noted with MgO and it is hypothesised that its flotation is as a result of entrainment and natural hydrophobicity, rather than induced hydrophobicity through surface alteration. Entrainment is most likely a major factor due to:

1. Significantly higher water recovery at higher dosages, which relates well with results by various researchers and summarised by Wang (2014) – see section 2.5. The water mass recovered in the 0 and 50 g/t tests were around 3 500 to 4 000 g compared to 6 000 to 6 500 g for the 100 and 200 g/t tests.
2. Recovery of fine talc confirmed by mineralogy – see section 3.3.3

These results confirm that iron, in the form of pyrrhotite and MgO are the major gangue minerals of concern.

The significant increase in iron recovery and subsequent dilution of the concentrate may be detrimental to the operation. The impact of increased rougher flotation recoveries on the reagent addition and final product concentrate quality must be economically evaluated before maximising recovery through collector addition.

4.6 Effect of Depressant – No Collector

The effect of depressant addition was explored to identify the exclusive effect that a depressant has on the flotation of the major elements. The grade-recovery curves for each of the elements are presented in Figure 4-12 to Figure 4-15 and a summary of the results is provided in Table 4-5.

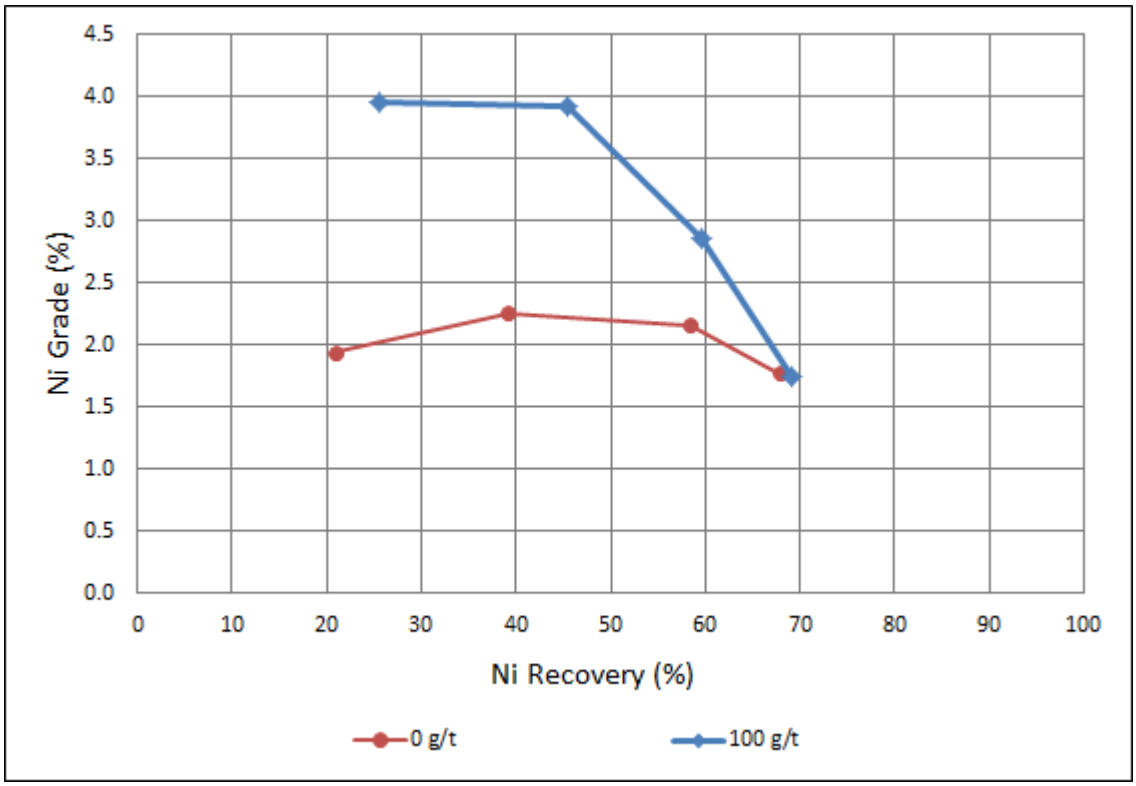


Figure 4-12: Effect of Depressant Dosage on Ni Grade-Recovery Curve

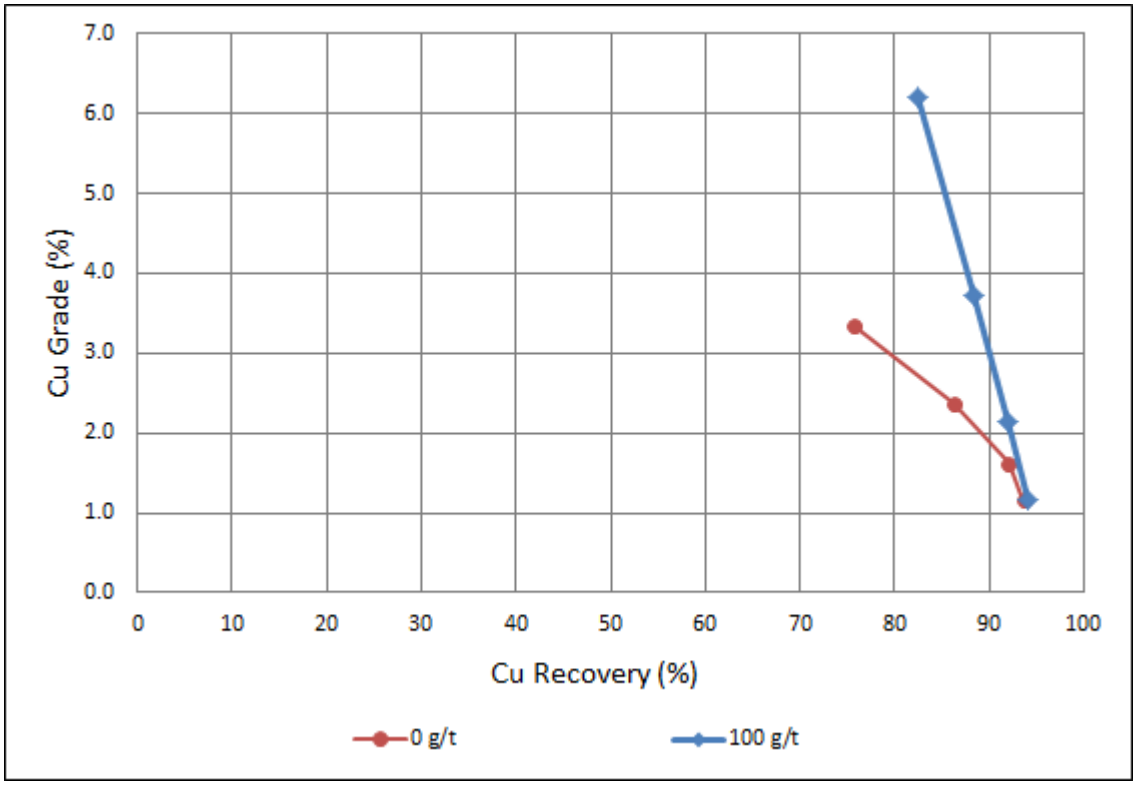


Figure 4-13: Effect of Depressant Dosage on Cu Grade-Recovery Curve

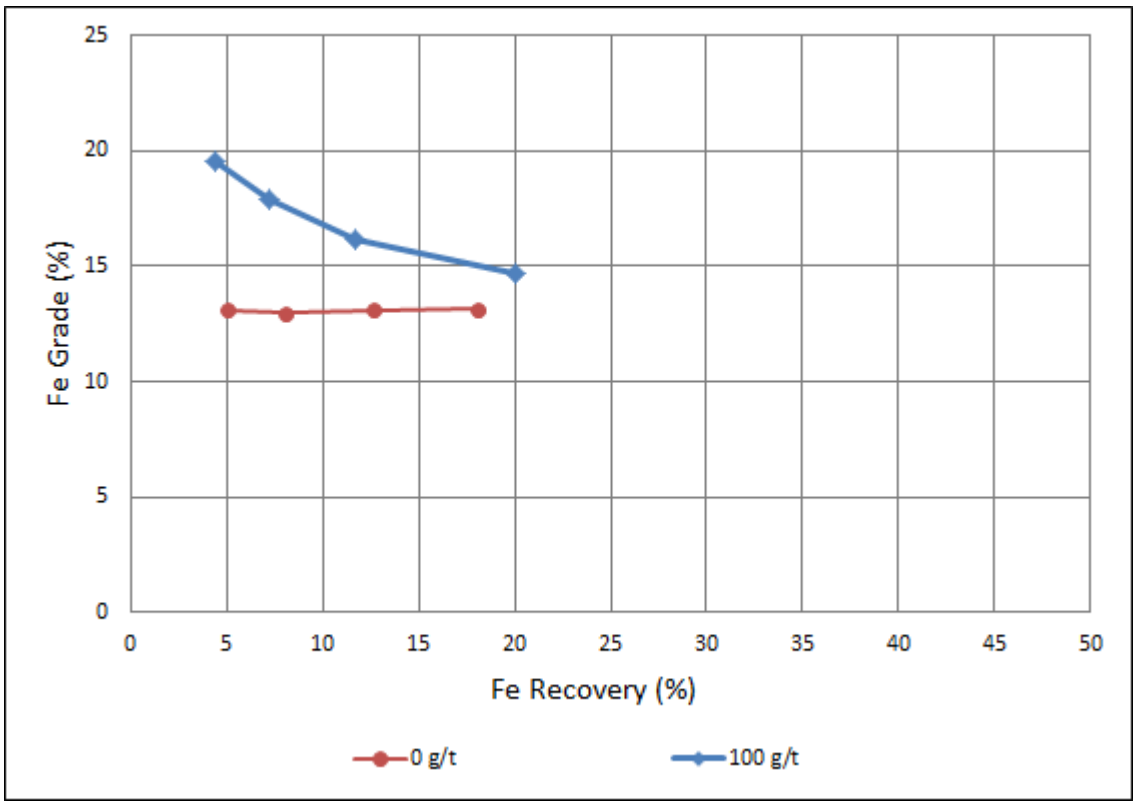


Figure 4-14: Effect of Depressant Dosage on Fe Grade-Recovery Curve

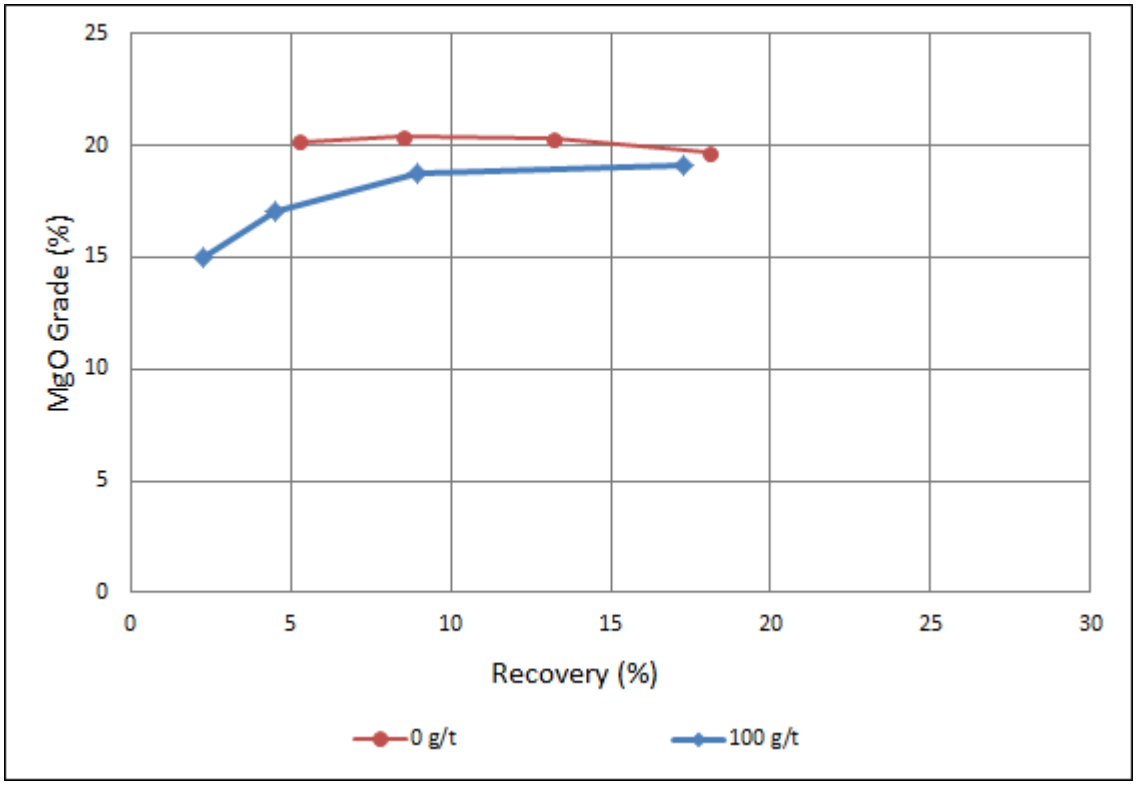


Figure 4-15: Effect of Depressant Dosage on MgO Grade-Recovery Curve

Table 4-5: Summary of Final Concentrate Results for Testwork Without Collector

CMC (g/t)	Mass Pull (%)	Final Concentrate Grade (%)				Final Recovery (%)			
		Ni	Cu	Fe	MgO	Ni	Cu	Fe	MgO
0	17.20	1.77	1.16	13.16	19.66	67.8	93.6	18.0	18.1
100	16.01	1.75	1.17	14.72	19.15	69.0	94.0	20.1	17.3

The addition of depressant reduces the mass pull, increases sulphide recovery (inclusive of iron) and reduces MgO recovery. However, the effect is marginal when compared to the enhancement achieved using collector, with 0.8% reduction in MgO recovery and an increase of 1.3% in nickel recovery. From Figure 4-12 to Figure 4-15, the MgO is strongly depressed in the first concentrate. The increase in sulphide recoveries confirms that magnesium silicates are partially collected by true flotation, i.e. bubble area originally occupied by MgO is now available for sulphide recovery. The decrease in MgO in the initial minutes of flotation was offset by additional iron recovery, despite significant increases in nickel and copper. Therefore, the effect of depressant dosage is marginal and a selective

gangue iron sulphide depressant or oxidant would be required to have a significant impact on gangue recoveries in the rougher stage. Possible methods could be solution chemistry adjustments (E_h and pH) and addition of amine depressants such as TETA (Lawson, 2015). Adjustments in solution chemistry aim to selectively oxidise gangue iron to minimise its flotation and simultaneously striving to minimise the oxidation of valuable mineral sulphides.

At the time of this testwork, typical depressant dosages on the rougher flotation cells were approximately 0 to 50 g/t (for headgrades of about 0.4% nickel), with very rare occasions where depressant dosages were as high as 100 g/t. Lower headgrades typically require more depressant.

It is recommended that further testwork be conducted at higher depressant dosages and no collector to further understand the effects of depressant dosage on elemental recoveries (i.e. comparison between improved gangue rejection and nickel recovery). Furthermore, specific iron sulphide and primary silicate (pyroxene) depressants or depressing mechanism (such as oxidation) could be beneficial to the performance of the float.

4.7 Interactive Effects of Collector and Depressant

The effect of collector and depressant on nickel, iron and MgO recoveries is presented in Figure 4-16 to Figure 4-18.

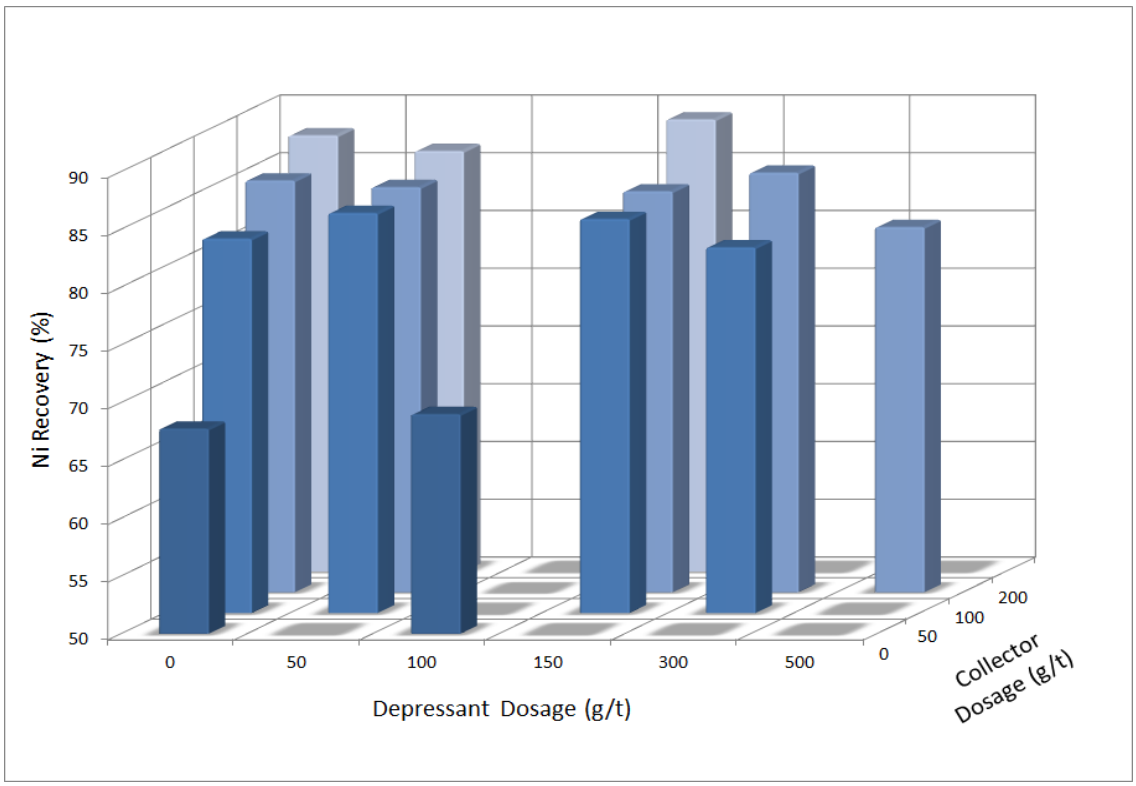


Figure 4-16: Effect of Collector and Depressant Dosage on Ni Recovery

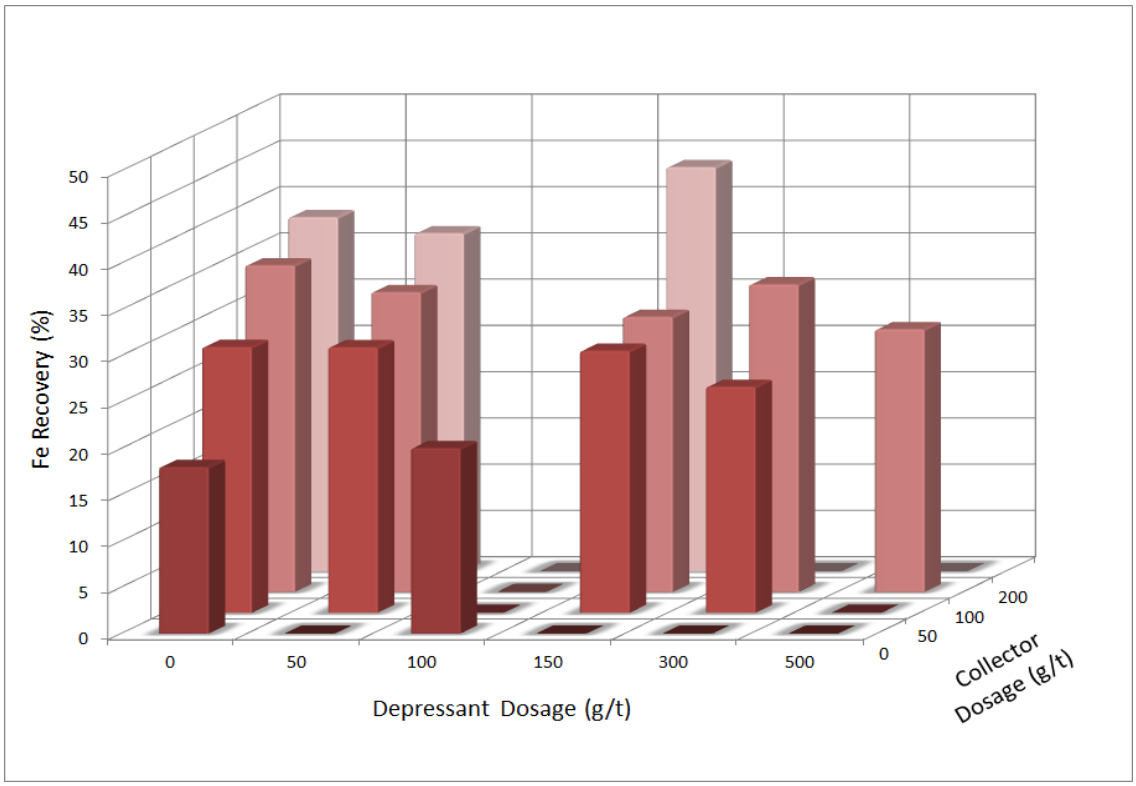


Figure 4-17: Effect of Collector and Depressant Dosage on Fe Recovery

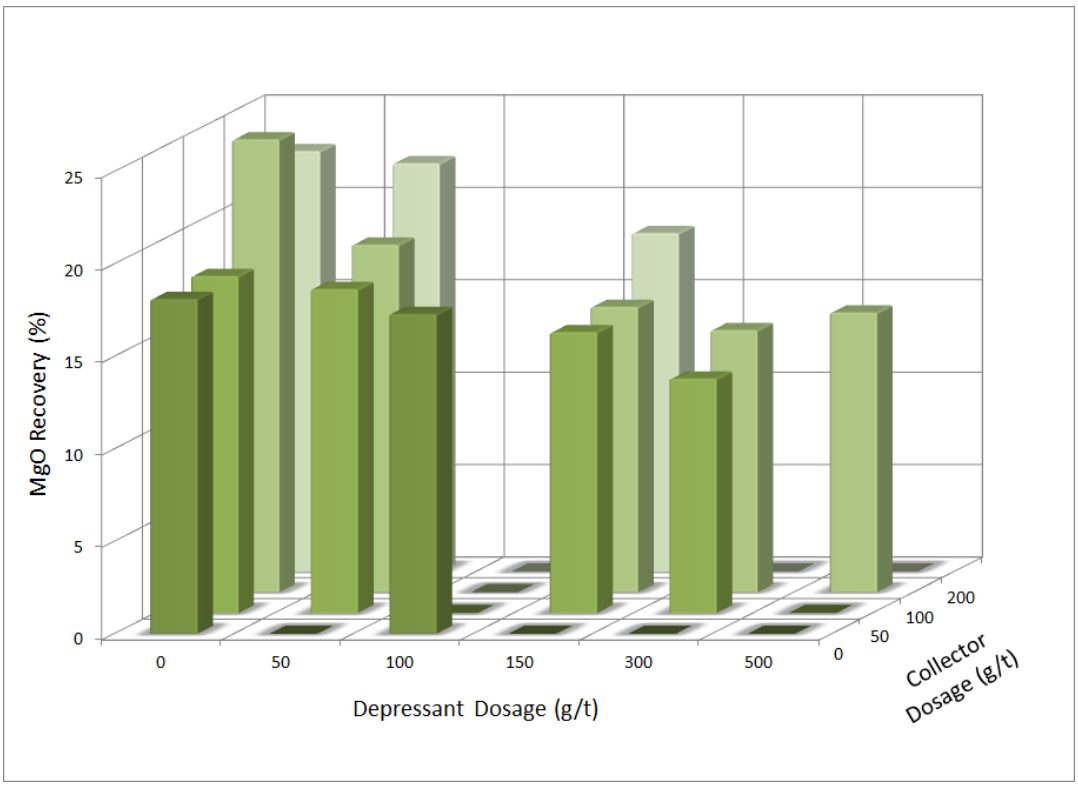


Figure 4-18: Effect of Collector and Depressant Dosage on MgO Recovery

Increasing collector dosage improved the hydrophobicity of the minerals in the ore, thereby increasing the recovery of all the elements and at all depressant dosages. The impact of a higher nickel recovery must be evaluated against the effect of higher gangue on the operating costs (particularly depressant costs) and final concentrate grade (affecting off mine costs). Excessive talc (and other silicates) recovery requires higher depressant consumption while higher pyrrhotite recovery significantly reduces the concentrate nickel grade (or valuables concentration). A significant increase in mass pull was observed at a collector dosage of 200 g/t. This resulted in increased recovery of all elements, including MgO. The nickel grade in the concentrate reduced from about 2% to 1.5%. The reduced nickel grade was also reported at 100 g/t collector dosage and lower depressant dosages. This is most likely due to a higher recovery of pyrrhotite. The evidence is not strong from the iron concentrate grades, however, a slight increase in the iron content is observed. This testwork should be repeated and the concentrate samples submitted for mineralogical analysis to confirm this hypothesis.

At all depressant dosages, the highest relative increase in recovery is observed for iron, as was noted in the base case. The higher nickel recovery is caused by the effective

depression of magnesium silicates. MgO is progressively reduced in the concentrate as depressant dosage is increased. A collaborative effect is observed between the collector and depressant, with higher nickel recoveries achieved when dosages of both reagents are increased. As expected, however, nickel recovery is reduced at higher depressant dosages. This is in agreement with testwork done by other researchers (Bradshaw et al, 2005). This phenomena has been attributed to the depression of froth stabilising gangue (talc) reducing the froth stability, thereby reducing entrainment of valuable minerals. The optimum depressant dosage depends on the specific dosage of collector. This is evident at 150 g/t depressant for a 50 g/t collector dosage and 300 g/t depressant for a 100 g/t collector dosage. Nickel recovery loss also depends on the collector addition, with less depressant required to over-depress froth stabilising gangue at lower collector dosages. This is of great importance to the concentrator for defining the reagent regime because increasing the depressant dosage would require an adjustment to the collector, and therefore, an appropriate recipe or algorithm would be required for reagent changes. Reduced copper recovery was also observed at high depressant dosage, however, the effect was lesser than on nickel due to its high floatability. This finding is in agreement with work done by Bradshaw (2004). The depressant addition in the Nkomati flotation circuit should be re-evaluated as froth destabilisation could be occurring due to high depressant dosages. It is recommended that depressant dosage be reduced in the cleaning stages whilst adjusting the air flowrate to achieve the product specification. This has been shown to be beneficial on a UG2 plant, where an upward shift in the grade-recovery curve was observed and the depressant dosage was reduced significantly (Harris et al, 2013).

Collector dosages are fairly high at the plant and the effects of excessive dosage should be highlighted. Stronger collectors are known to reduce froth stability, resulting in reduced recoveries of the valuable metal (Bradshaw et al, 2005). In addition, high depressant dosages could remove froth stabilising gangue, further reducing recovery. It is recommended that weaker collectors be considered in future testwork to explore this phenomenon.

4.8 Chapter 4 Conclusions

Based on the findings presented in Chapter 4, the following conclusions can be made:

1. Collectorless flotation resulted in a nickel recovery of 67%.

2. The addition of collector enhances the flotation of both valuable and gangue sulphide minerals. Collector has a significant impact on the recovery of iron bearing sulphide minerals.

3. The SIBX collector does not increase the hydrophobicity of the MgO and its collection is driven more by natural hydrophobicity and entrainment.

4. Depressant addition successfully reduces MgO recovery to the concentrate. However, sulphide recovery, including the iron sulphide gangue, is increased. This reduces the final concentrate grade. A depressant selective to the iron sulphides, pyrrhotite and pyrite, would assist in improving both concentrate nickel grade and recovery.

5. Nickel is depressed at very high dosages of depressant.

6. An understanding of the interaction between collector and depressant is vital to the optimisation of process recoveries. There exists an optimum dosage of depressant at a specific collector addition and vice versa.

7. The relative recovery of iron is considerably greater than that for nickel when collector is introduced. This increases the need to depress iron in the cleaning stages. Losses of nickel are caused at higher depressant dosages, therefore, the effect of increased recovery in the rougher stage should be evaluated against any potential losses in the cleaning stages (as a result of additional depressant requirements).

8. A range of collectors (weaker and stronger) should be considered for further optimisation.

4.9 Chapter 4 Recommendations

1. A process recipe should be developed to optimise the addition of collector and depressant.

2. Testwork at 200 g/t collector dosages and perhaps higher should be conducted to confirm the results achieved in this report. Furthermore, mineralogical analysis should be conducted to determine whether pyrrhotite recovery increases and its contamination of the final concentrate.

3. Further optimisation should include the effect of all other reagents such as frother and co-collectors.

4. A pyrrhotite/pyrite specific depressant should be evaluated to improve grade and potentially recovery.

5. The depressant dosages in the Nkomati cleaner circuit should be re-evaluated. Reduced depressant dosages should be tested, while adjusting the air flowrate simultaneously to achieve the product specification.

6. The impact of increased nickel recovery, and subsequently iron recovery, in the rougher stages on the overall recovery should be evaluated by conducting rougher-cleaner-recleaner testwork.

5. Chapter 5: Stage Dosing

5.1 Introduction

Laboratory scale testwork was conducted to understand the effects of stage dosing on the recovery of valuable and gangue minerals. It is hypothesised that stage dosing collector improves valuable mineral recovery and stage dosing depressant improves rejection of unwanted gangue from the concentrate. The areas covered in this chapter are:

- Procedure used for testwork
- Stage dosing of collector
- Stage dosing of depressant
- Stage dosing of collector and depressant

5.2 Experimental Procedure

5.2.1 Equipment and Procedure

Testwork procedures have been detailed in Chapter 4. The following testwork was conducted:

1. Assay by size analysis (section 4.2.1.3). Assay by size on flotation products was only conducted on one of the duplicate flotation tests.
2. Flotation (section 4.2.1.4)
3. Chemical Analysis of products (section 4.2.1.5)

5.2.2 Experimental Programme

The testwork programme was designed as follows:

1. Test 1: Base case float – upfront reagent dosage only
2. Test 2: Stage dosing of collector
3. Test 3: Stage dosing of depressant
4. Test 4: Stage dosing of collector and depressant – higher dosages in the earlier stages of flotation (see Table 5-1 for details)
5. Test 5: Stage dosing of collector and depressant – higher dosages in the later stages of flotation (see Table 5-1 for differences between Test 4 and Test 5)
6. Test 6: Base case float at lower collector dosage

7. Test 7: Stage dosing of collector at lower collector dosage

Table 5-1: Laboratory Scale Testwork – Stage Dosing

Test	Reagent	T = 0 min	T = 1 min	T = 3 min	T = 10 min
		g/t			
1	Collector	100	0	0	0
	Depressant	50	0	0	0
2	Collector	50	25	15	10
	Depressant	50	0	0	0
3	Collector	100	0	0	0
	Depressant	15	15	10	10
4	Collector	50	25	15	10
	Depressant	20	15	10	5
5	Collector	30	30	20	20
	Depressant	10	10	15	15
6	Collector	30	0	0	0
	Depressant	50	0	0	0
7	Collector	15	8	4	3
	Depressant	50	0	0	0

5.3 Assay by Size

Cumulative particle size distribution and discrete metal distribution for the flotation feed is presented in Figure 5-1.

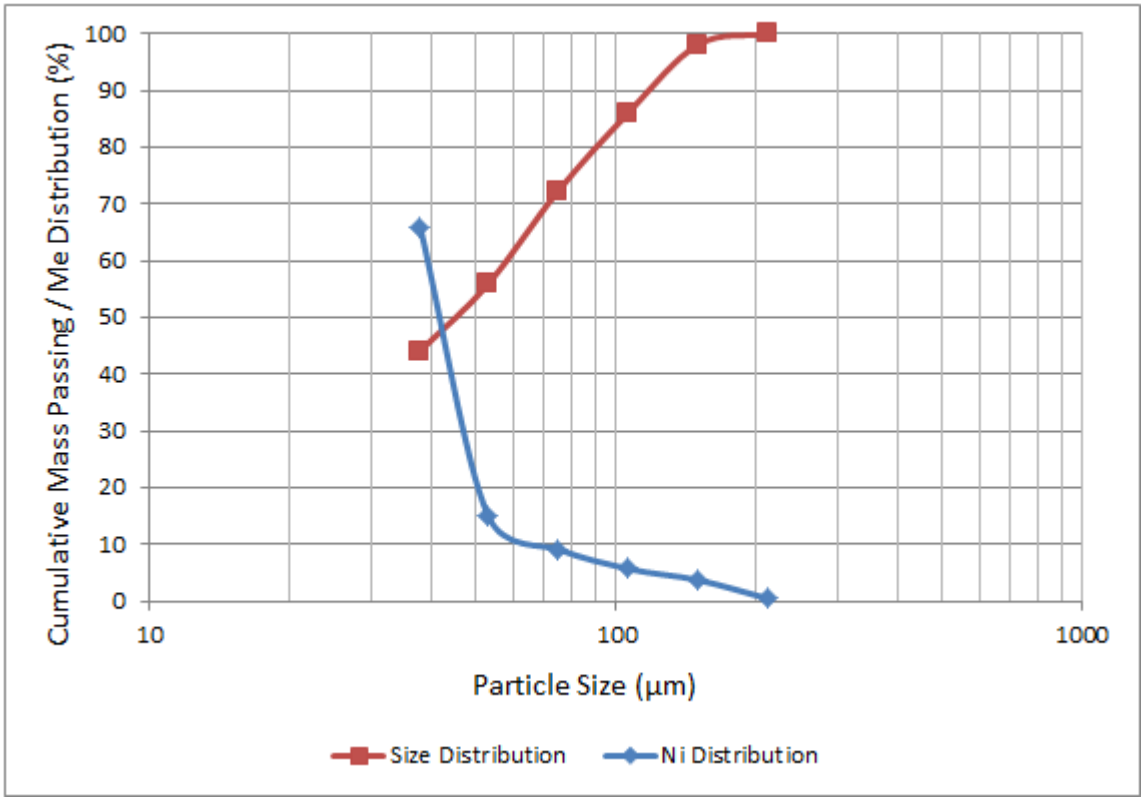


Figure 5-1: Feed Size Distribution

The flotation feed particle size distribution was 72% passing 75 µm, the same as the target grind according to the process target. 65% of the nickel reports to the -38 µm fraction, while 10% of the nickel is present in the +75 µm fraction. Since stage dosing optimises the coarse recovery, it is clear that there is a significant amount of nickel in this fraction of the feed. It will, therefore, be important to identify the proportions recovered using upfront dosing as compared to stage dosing.

5.4 Base Case Float – Upfront Dosing of Reagents

The grade-recovery curves for the upfront dosing testwork is presented in Figure 5-2 and the nickel recovery by size is presented in Figure 5-3.

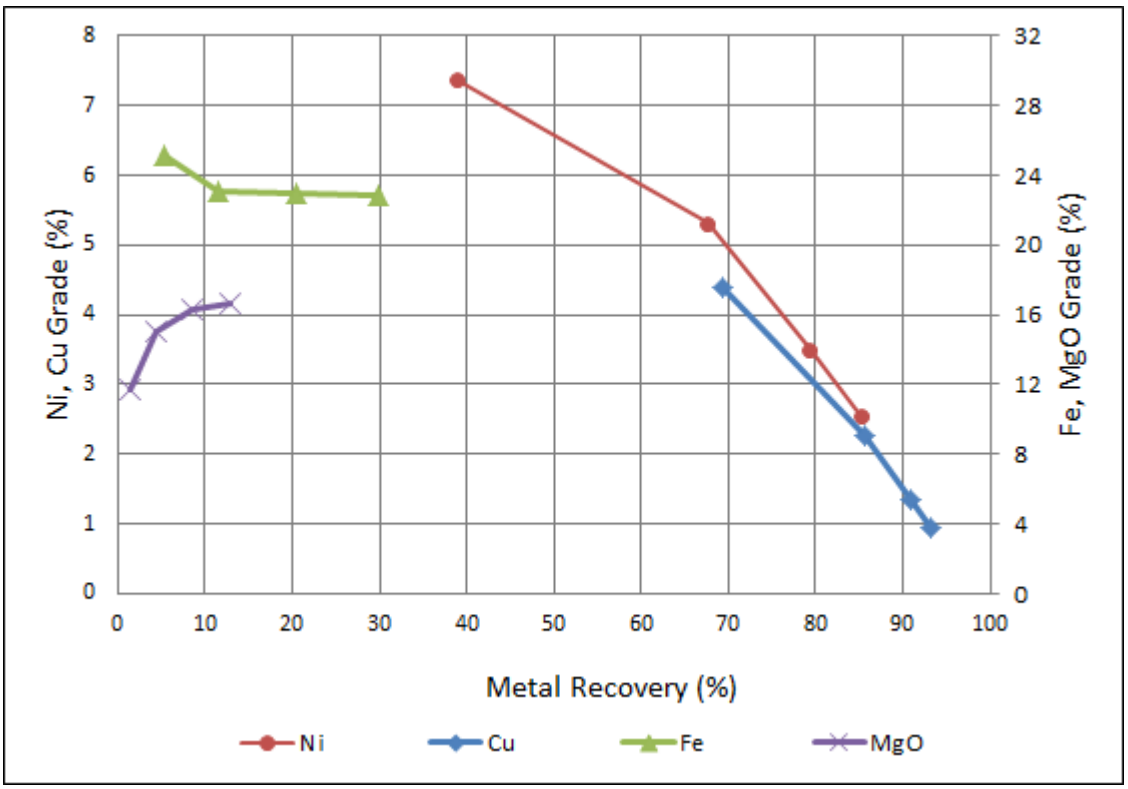


Figure 5-2: Upfront Dosing Grade-Recovery Curves

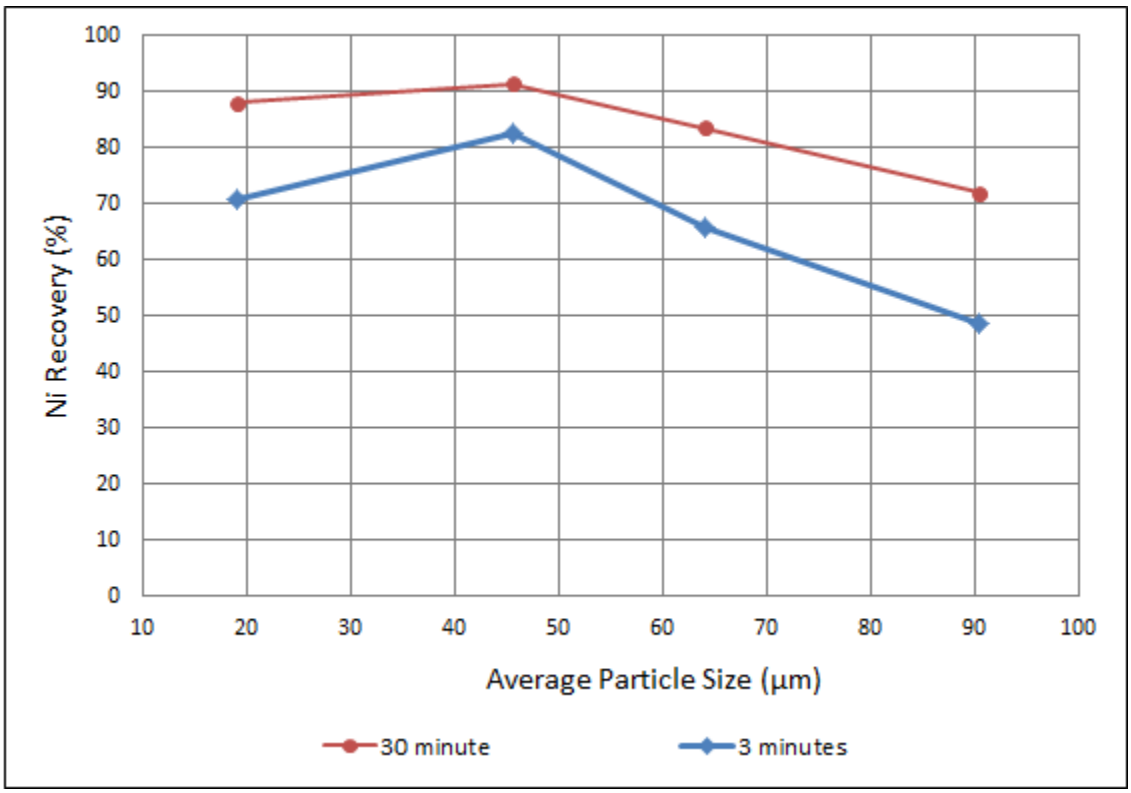


Figure 5-3: Nickel Recovery by Size

The nickel recovery was 85.1% at a nickel grade of 2.6%. The nickel recovery curves are consistent with the general recovery by size curves as presented in the literature (Figure 2-3) with a lower recovery in finer and coarser size fractions than the intermediate size range. Additional flotation time improves the recovery in all fractions, particularly fines and coarse. Fines recovery increased from 70% to 90%, while coarse recovery was increased from 50% to 70%, illustrating the benefit to be gained from improved coarse grained nickel recovery. Iron and MgO recoveries were approximately 30% and 13% respectively.

5.5 Stage Dosing Reagents

A summary of the test results is provided in Table 5-2.

Table 5-2: Summary of Reagent Dosing Testwork at 100 g/t

Test	SIBX (g/t)	CMC (g/t)	Final Grade (%)				Final Recovery (%)			
			Ni	Cu	Fe	MgO	Ni	Cu	Fe	MgO
1	100	50	2.56	0.94	22.86	16.70	85.12	93.23	29.86	12.91
2	100	50	2.33	0.97	27.56	14.02	86.84	93.87	40.13	12.74
3	100	50	2.26	0.90	21.20	15.78	86.87	93.65	34.90	14.85
4	100	50	2.37	0.92	26.27	14.23	87.90	94.48	41.46	13.55
5	100	50	2.67	1.02	29.37	13.63	88.94	93.24	39.76	10.68

Test 1 and Test 2 provide a comparison between upfront dosing and stage dosing collector. Stage dosing collector increased the nickel recovery by 1.7%. Increased nickel recovery is accompanied by an increase in iron recovery. A noticeable MgO recovery change was not observed with stage dosing collector.

Test 1 and Test 3 provide a comparison between upfront dosing and stage dosing depressant. A similar increase in nickel recovery of 1.7% was observed when depressant was stage dosed. Although the nickel recovery was similar to stage dosing collector, the effect on iron and MgO is different, with less iron reporting to the concentrate. This could potentially be better than Test 2 if the iron is in the form of pyrrhotite. This is a reasonable assumption because pyrite is fast floating and the majority is most likely recovered in the

early stages of flotation. Improved depression is not clear from the elemental distribution, however, the increased nickel recovery suggests this.

Tests 4 and 5 provide a comparison between stage dosing of both collector and depressant simultaneously. Test 4 was conducted with a collector distribution of 50/25/15/10 and a depressant distribution of 40/30/20/10 with a resultant increase in nickel recovery of 2.8%. Test 5 was conducted with higher dosages at 3 minutes and 10 minutes flotation times with collector distribution of 30/30/20/20 and depressant distribution of 20/20/30/30 and resulted in an increased nickel recovery of 3.8%. The lower gangue recovery for test 5 indicates the desired outcome of improved gangue rejection by stage dosing of depressant.

These results confirm the findings of stage dosing by Bazin (2001) and split conditioning by Trahar (1976) in that stage dosing reagents improves metal recovery and that lower dosages in the earlier stages of flotation are preferable. The specific findings presented above indicate that:

1. Stage dosing collector improves nickel recovery
2. Stage dosing depressant improves nickel recovery
3. Stage dosing collector and depressant had a similar effect on nickel recovery improvement
4. Stage dosing both collector and depressant had an additive effect on the nickel recovery
5. Lower dosages of collector and depressant in the earlier stages provided the biggest improvement in nickel recovery.

Furthermore, the previous researchers also identified that the recovery of metal in coarse particles improved on stage dosing with minimal impact on the recovery of metal in fine particles. The results of the recovery by size on the Nkomati ore is presented in Figure 5-4 (after 3 minutes of flotation) and Figure 5-5 (at the end of the testwork – 30 minutes of flotation).

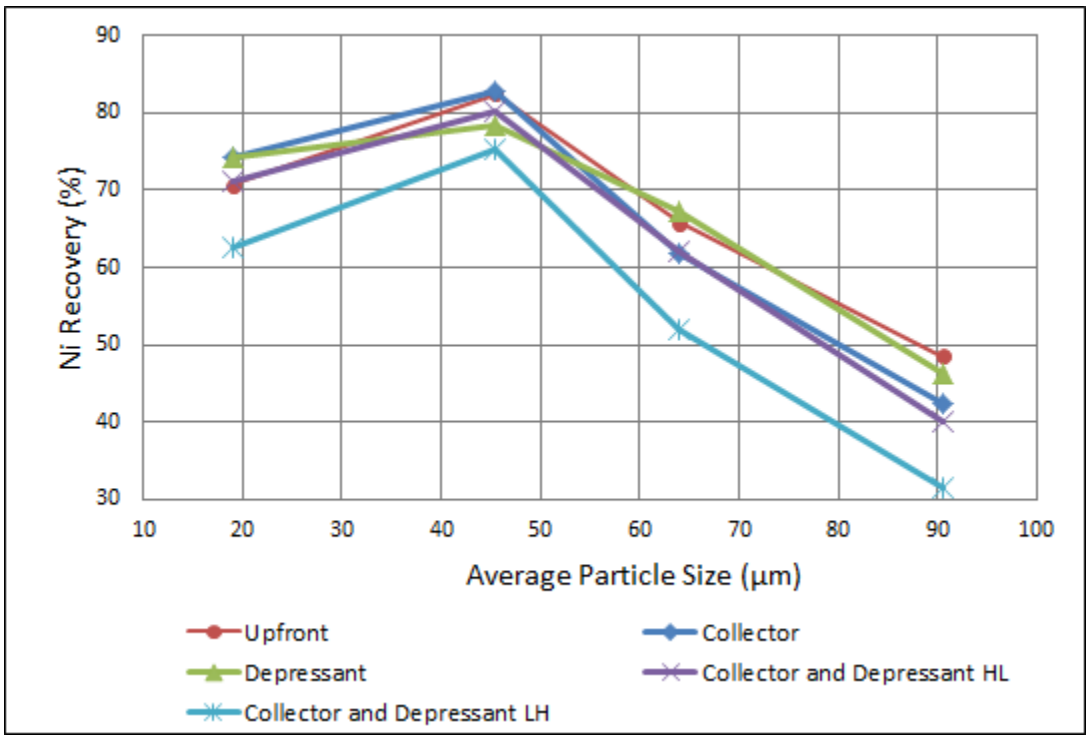


Figure 5-4: Nickel Recovery by Size after 3 Minutes of Flotation

After 3 minutes of flotation, upfront dosing produced the best recovery in all size fractions except for the fine fraction, which is not unexpected, considering that the reagent dosage is higher for this scenario. Furthermore, stage dosing of depressant had a similar performance to upfront dosing. In addition, stage dosing of collector and depressant had a similar performance to stage dosing of collector only. Thus, in the early stages of flotation, depressant addition does not seem to have a significant effect on the nickel recovery and presents a strong case for minimising the dosage of depressant upfront. The test with the lowest recovery performance was the low-high addition and is most likely due to insufficient coverage in the early stages of flotation (due to low dosage of reagents, particularly depressant). An anomalous result was the finest fraction where stage dosing of collector and depressant were slightly higher which is probably due to selective adsorption of reagent onto the finer particles.

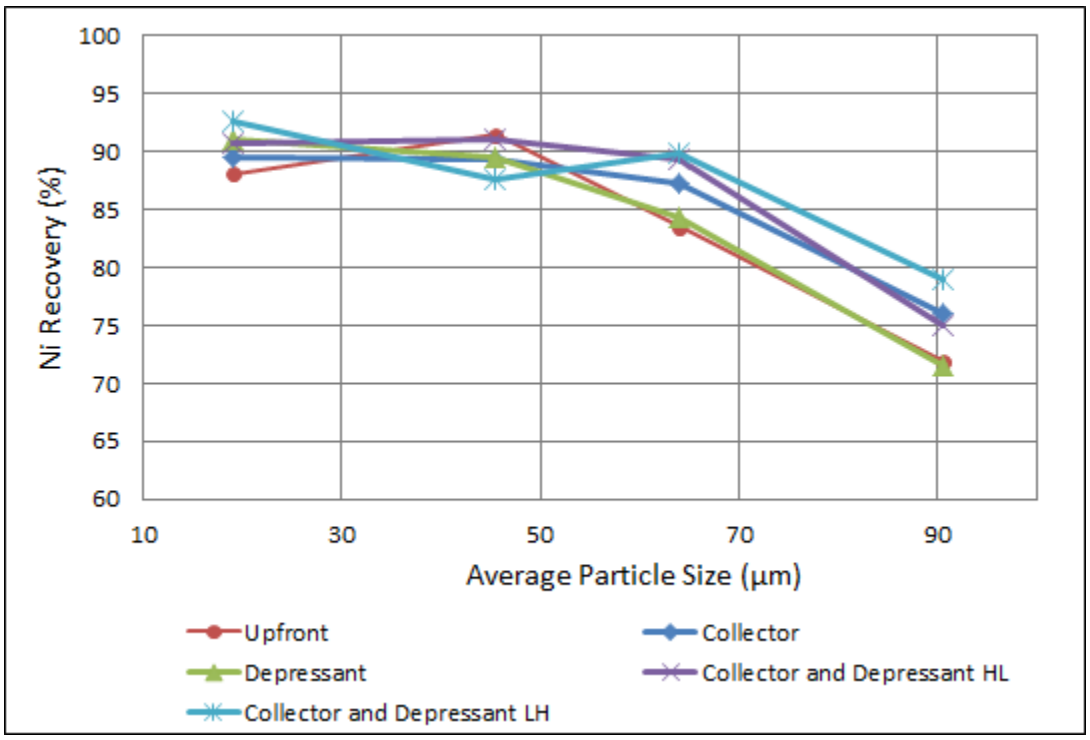


Figure 5-5: Nickel Recovery by Size after 30 Minutes of Flotation

The results after 30 minutes of flotation are very different, with stage dosing outperforming upfront dosing, except for the -53+38 µm size range. Recovery of coarse particles increased significantly from approximately 72% to as high as 79%. The nickel recovery in the fines fractions improved for all forms of stage dosing relative to upfront dosing. Recovery of the fine nickel increased from between 88% and 93%. According to the assay by size presented in section Figure 5-1, 65% of the nickel reports to the fine size fraction, therefore, improved recovery in the fines size fraction drives overall recovery. The high grade middlings and liberated pentlandite in the fine tailings was also confirmed by the mineralogical evaluation (Chapter 3), and it's recovery is favourable. The initial understanding for stage dosing was to improve recovery of coarse nickel. Although this is true, the impact of fine nickel recovery by stage dosing may be more beneficial, especially in the case of the Nkomati ore. It would be more appropriate to state that stage dosing improves nickel recovery of both coarse and finely grained nickel. Bazin (2001) found that although laboratory scale testwork did not show an improvement in recovery, the opposite was observed on plant scale trials and, therefore, required further testwork to confirm. The results on the Nkomati ore confirm this finding. An irregular result is the significantly lower recovery of the -53+38 µm nickel recovery for the collector and depressant stage dosing (LH). Repeat testwork to confirm this result is recommended.

As stated previously, stage dosing of depressant in the first 3 minutes of flotation was not conclusive in proving performance benefits, however, the performance after 30 minutes shows improved nickel recovery. Therefore, the additional depressant in the later stages improves the depressing action.

It is suggested that stage dosing not only improves the recovery of coarse particles, but also improves the surface coverage of reagents onto non-selectively coated finer particles. Dosing reagents later in the process, improves coverage due to the absence of the faster floating species that may have unnecessarily consumed reagent, particularly when fine particles are excessive. Figure 5-6 and Figure 5-7 shows the mass distribution of the flotation products for upfront and stage (test 5) dosing. It is evident that the first few concentrates for both tests contain around 60-70% fines. Although less fine particles (by mass) are recovered in test 5, the recovery of fine nickel is higher, as presented in Figure 5-5. Thus, stage dosing in the later stages of flotation results in more effective particle coverage of non-selectively coated particles.

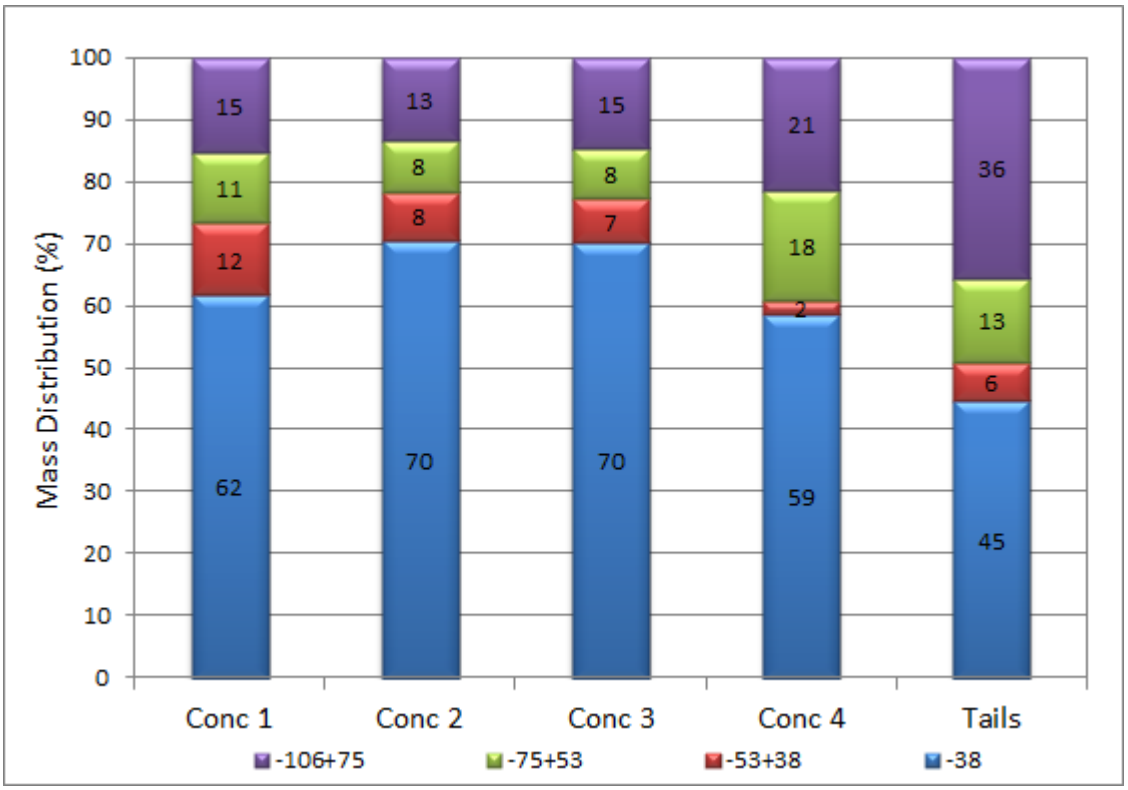


Figure 5-6: Mass Distribution for Upfront Dosing

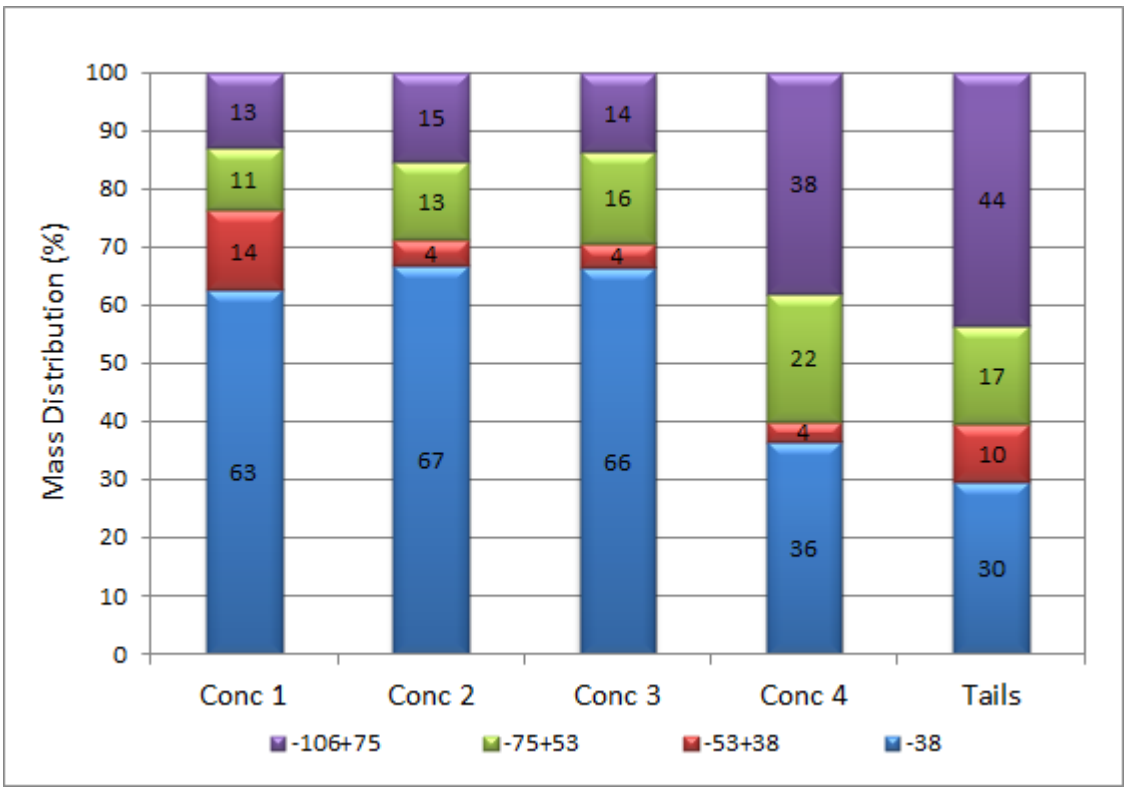


Figure 5-7: Mass Distribution for Test 5 – Collector and Depressant Stage Dosing

The effect of stage dosing on gangue is presented in Figure 5-8 and Figure 5-9.

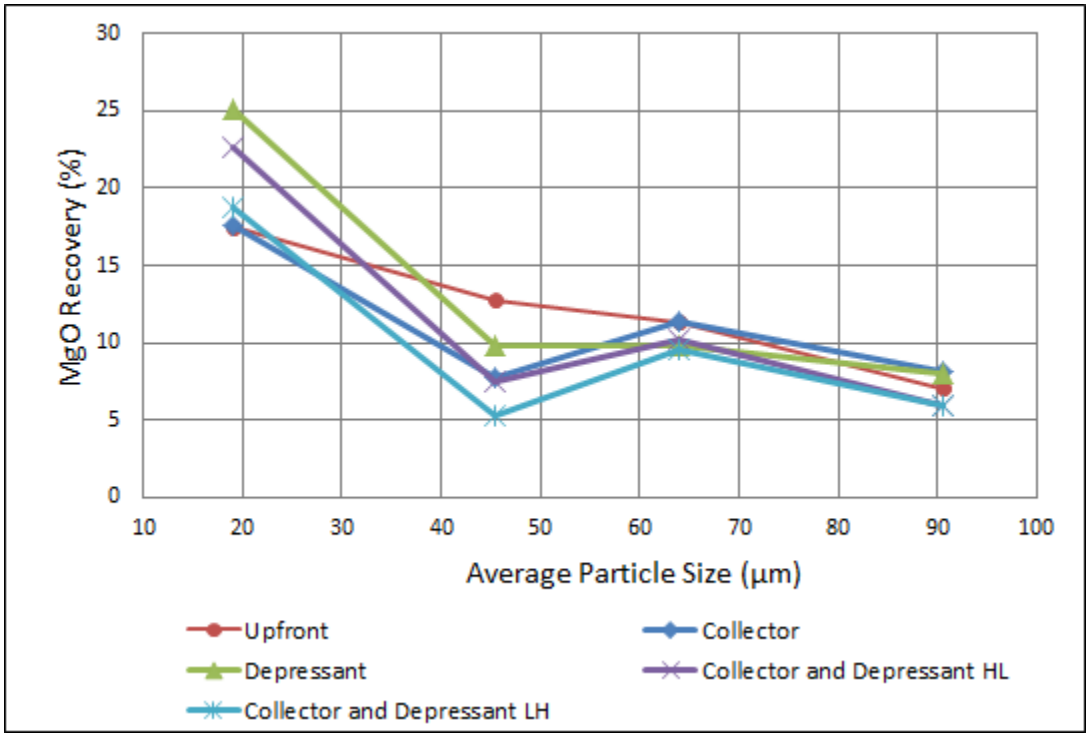


Figure 5-8: MgO Recovery by Size after 30 Minutes of Flotation

The effect of stage dosing on MgO recovery is not clear. As presented in Table 5-2, MgO generally showed an increase in recovery with stage dosing. Even when depressant was stage dosed in isolation the recovery of MgO increased. It is evident that fine MgO recovery is significantly increased along with the overall mass pull. This does not compare well with the testwork that was conducted in Chapter 4. More effective addition of depressant should improve MgO depression. Considering that nickel recovery was increased when depressant was stage dosed, an improvement in gangue rejection could be inferred. It is recommended that this testwork be repeated to confirm the results. Similarly, iron recovery is generally higher across all particle sizes. The improved iron recovery is in agreement with the Chapter 4 with recovery increasing due to gangue depression.

In Chapter 3, the process survey revealed low nickel recovery in the re-cleaner float, most likely as a result of excessive talc depression. Stage dosing is suggested to improve the nickel recovery in the re-cleaner float. The significant increase in iron recovery may result in additional depressant consumption in the cleaner stages to achieve concentrate specification. This does not annul the case for stage dosing, it rather demonstrates that a lower overall dosage would be required with stage dosing.

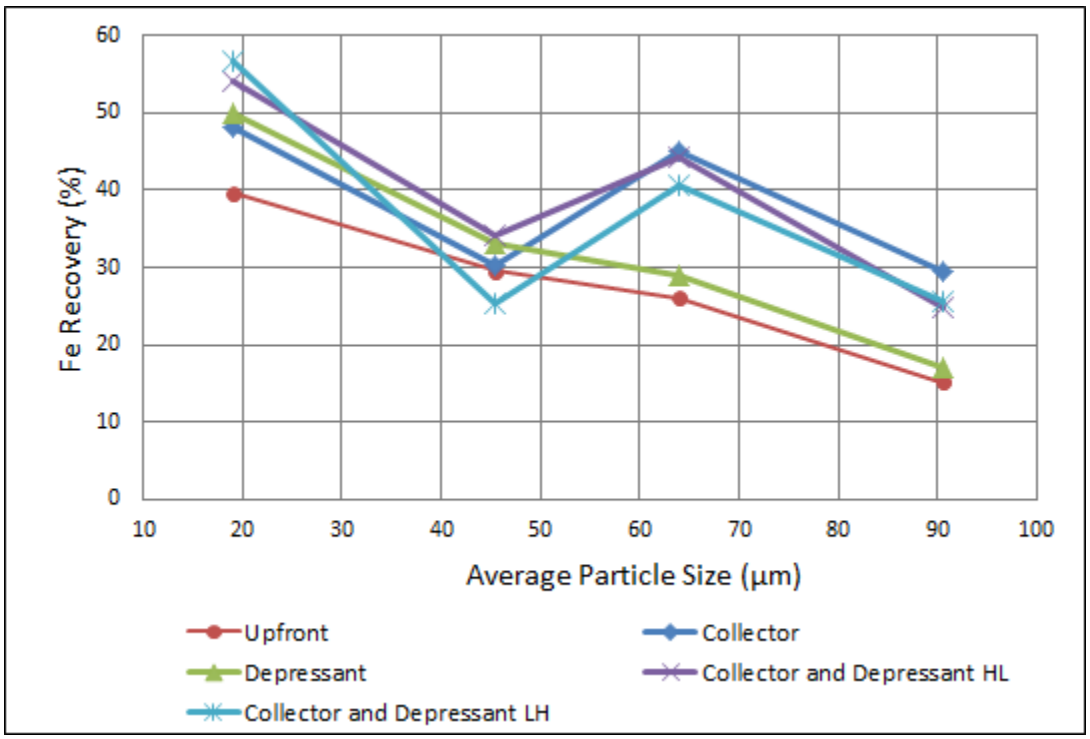


Figure 5-9: Iron Recovery by Size after 30 Minutes of Flotation

Tests 6 and 7 were conducted to identify the potential for reagent consumption optimisation by stage dosing. The results are presented in Table 5-3.

Table 5-3: Summary of Reagent Dosing Testwork at 30 g/t

Test	SIBX (g/t)	CMC (g/t)	Final Grade (%)				Final Recovery (%)			
			Ni	Cu	Fe	MgO	Ni	Cu	Fe	MgO
1	100	50	2.56	0.94	22.86	16.70	85.12	93.23	29.86	12.91
6	30	50	2.96	1.27	20.69	17.38	83.15	92.89	22.18	11.77
7	30	50	2.98	1.27	22.71	16.72	84.78	93.45	26.66	12.05

Stage dosing at 30 g/t recovered 1.6% more nickel than upfront dosing at 30 g/t and slightly lower recovery (0.3%) than upfront dosing at 140 g/t. The recovery by size is presented in Figure 5-10. Size recovery of fine and coarse particles at 30 g/t are similar to recoveries at 140 g/t. However, recovery of the intermediate size (-53+38 µm) is reduced.

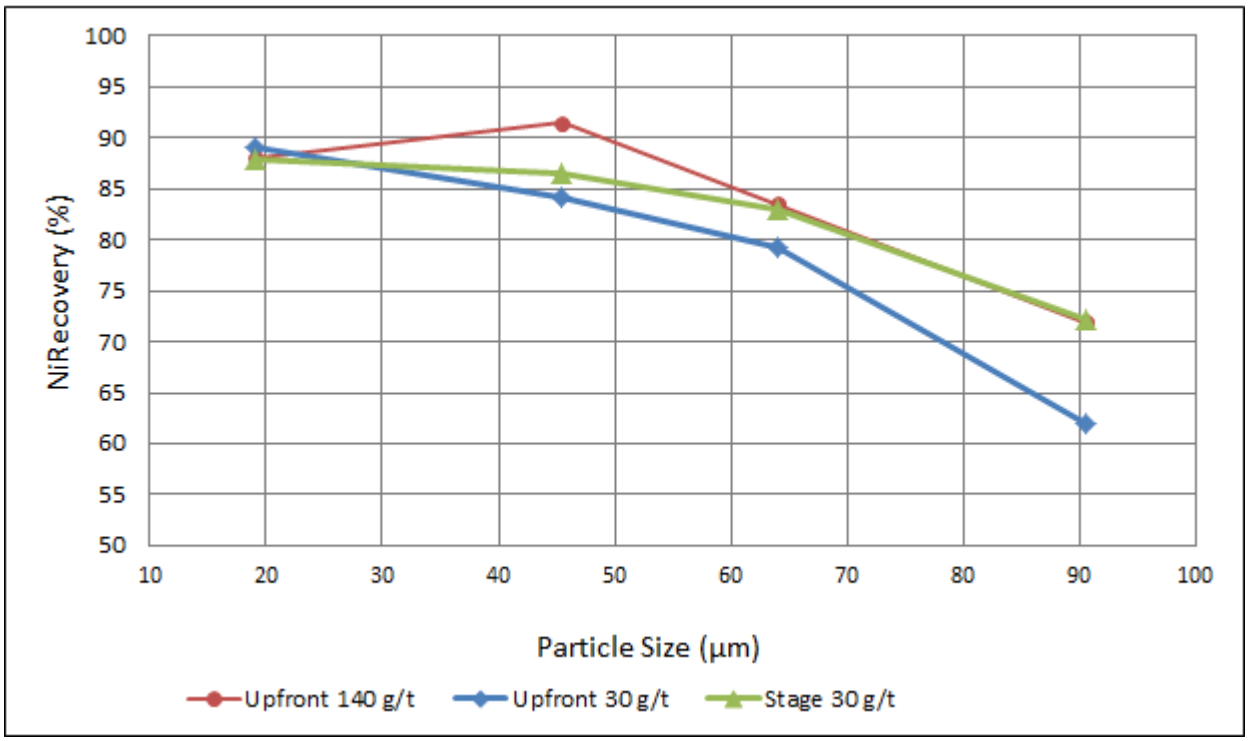


Figure 5-10: Nickel Recovery by Size – comparison of low stage dosing with low upfront and high upfront dosing and

The relevance of this analysis is best represented by a financial analysis. Stage dosing provides optimisation of either recovery or reagent consumption. A cash forecast has been generated under the following conditions:

1. Stage dosing collector at the current plant dosage with at a relative recovery increase as identified by the testwork (i.e. relative recovery increase of 2.0% used). Since the financial benefit of additional recovery is driven by metal prices, a sensitivity analysis was conducted on price between 8 000 and 20 000 USD/t of nickel; other metal prices have been adjusted by the same percentage.
2. Stage dosing collector at reduced collector dosages of 25% and 33% of current dosage based on an estimated improvement factor from the comparison between 30 g/t and 140 g/t testwork. This analysis is only based on collector consumption and, therefore, is not affected by the nickel price.
3. It has been assumed that nickel and cobalt recoveries are improved simultaneously and that copper recoveries are not improved (due to its fast floating nature – testwork shows that copper recovery is not significantly affected).

The results of the financial analysis is presented in Table 5-4.

Table 5-4: Financial Analysis of Stage Dosing

Simulation	Nickel Price	Reduction in Collector	Relative NPV
#	USD/t	% of Current	-
1	8 000	-	1.0
2	9 000	-	1.4
3	10 000	-	1.9
4	11 000	-	2.3
5	12 000	-	2.8
6	15 000	-	3.8
7	20 000	-	5.3
8	-	25%	2.0
9	-	35%	1.8

The optimum philosophy is dependent on the nickel price. At the current nickel prices around 8 000 to 9 000 USD/t, reducing the operating costs (i.e. collector dosage) is more profitable for the operation. Maximising recovery becomes more beneficial around a nickel price of 10 000 USD/t. This basic analysis shows the importance of considering both these options under the prevailing circumstances. The true operational benefits would have to be determined on the plant before deciding on the approach.

The financial analysis only accounts for stage dosing of collector. The testwork results have shown that depressant could provide additional recovery benefits. Further testwork on the reduction of depressant consumption (in the laboratory and on the plant) would be required to complete the financial evaluation for the combined effect of stage dosing collector and depressant.

5.6 Chapter 5 Conclusions

Based on the findings presented in Chapter 5, the following conclusions can be made:

1. Recovery of coarse grained nickel in the base case float (upfront reagent dosing) was 70%. This size fraction showed the highest potential for improvement, with recoveries in the other size fractions in excess of 80%.

2. Stage dosing of collector in isolation showed improved nickel recoveries.

3. Stage dosing of depressant did not improve selective gangue rejection as initially presumed. Depressant dosage seemed to improve the overall mass pull to concentrate, thereby increasing the recovery of all the components, including nickel. Stage dosing resulted in lower iron content in the concentrate, an indication of reduced pyrrhotite collection.

4. Stage dosing of both collector and depressant showed the biggest improvement in nickel recovery, thus, indicating an additive improvement effect. Lower dosages of reagent upfront improved overall recovery.

5. Bazin (2001) indicated that stage dosing would be beneficial for coarse particle recovery, however, the work conducted in this thesis indicates and confirms areas of uncertainty in previous research that stage dosing increased the recovery of valuable minerals in both coarse and fine particles.

6. It has been hypothesised that stage dosing improves the surface coverage of non-selectively adsorbing particles by eliminating high reagent consumers in the initial stages of flotation.

7. Optimisation of reagent addition can be done to maximise recovery or minimise reagent dosage. A financial evaluation should be conducted to determine the optimum route.

5.7 Chapter 5 Recommendations

1. Stage dosing is recommended for the Nkomati ore. Various distributions should be tested. An evaluation of the recovery and reagent consumption reduction should be conducted.

2. Additional testwork for stage dosing of depressant is required to confirm the findings in this thesis. Mineralogical analysis would be important to obtain a clear understanding of the process.

3. Collector and depressant stage dosing (starting with low dosage at the top of the bank and increasing further down) should be repeated to confirm the finding that the nickel recovery was significantly lower for the 45 µm fraction compared to all other testwork conducted.

4. Collector stage dosing at reduced dosage was only conducted in a high-low configuration. The low-high configuration has shown to be beneficial and should either be evaluated in the laboratory or plant scale.

5. Over-depression of froth stabilising gangue was highlighted in the re-cleaner flotation stage on the plant in the process survey conducted (Chapter 2). Stage dosing depressant is recommended and could reduce depressant addition, thereby improving nickel recovery.

6. Study Conclusions

This report has investigated reagent addition as a means of improving the flotation process. The effect of individual reagent addition, particularly collector and depressant, and their interaction and the effect of stage dosing has been considered. The major findings of the study are highlighted in this section.

Collector addition had a significant impact on the hydrophobicity of the sulphide minerals, with a proportional effect on recovery. The gangue recovery, MgO and silicate minerals, was largely driven by their natural hydrophobicity and entrainment. Higher depressant dosages are valuable for MgO depression, however, the CMC depressant was not selective to the iron sulphide gangue which reduces the valuable base metal sulphides in the final concentrate. Future testwork should focus on identifying reagents or processes that may assist in selectively depressing both silicates and sulphide gangue. Excessive depressant addition removes the froth stabilising gangue, which, in effect reduces the recovery of nickel to the concentrate. Therefore, optimising the combined addition of reagents is essential to maximising recovery. The effect of other reagents such as co-collectors and frothers should also be included as these too would have an effect on the process.

Stage dosing has been known to provide benefits in the recovery of coarse-grained particles. The results presented in this report have shown benefits in both coarse- and fine-grained particles, thereby improving the surface coverage of reagents. Lower dosages upfront ensure that reagent wastage on finer particles is minimized. Progressive addition down the bank would be beneficial for all particle sizes.

7. References

1. Bazin C, Proulx M, 2001; *Distribution of reagents down a flotation bank to improve the recovery of coarse particles*; International Journal of Minerals Processing 61 (2001) 1-12; Elsevier Scientific Publishing Company
2. Becker M, 2009; *The Mineralogy and Crystallography of Pyrrhotite from Selected Nickel and PGE Ore Deposits and its Effect on Flotation Performance*; University of Pretoria
3. Bose, A; Emerging Trend of Nanotechnology in Pharmacy; http://www.pharmainfo.net/book/emerging-trends-nanotechnology-pharmacy-physicochemical-characterization-nanoparticles/zeta (picture); accessed on 13 September 2015.
4. Bradshaw D.J., Oostendorp B., Harris P.J., 2004; *Development of methodologies to improve the assessment of reagent behavior in flotation with particular reference to collectors and depressants*; Minerals Engineering 18 (2005) 239-246; Elsevier Scientific Publishing Company.
5. Bradshaw D. J., Harris P.J., O'Connor C.T, 2005; *The effect of collectors and their interactions with depressants on the behaviour of the froth phase in flotation*; Centenary of Flotation Symposium, Brisbane; Australia 6–9 June 2005.
6. Bulatovic, S.M.,2007; *Handbook of Flotation Reagents*; Chemistry, Theory and Practise: Flotation of Sulfide Ores, Volume 1; Elsevier Science and Technology Books.
7. Cilek E.C., 2009; *The effect of hydrodynamic conditions on true flotation and entrainment in flotation of a complex sulphide ore*; International Journal of Mineral Processing 90 (2009) 35-44; Elsevier Scientific Publishing Company
8. Crundwell F., Moats S.M., Ramachandran V., Robinson G.T., Williamson G.W., 2011, *Extractive Metallurgy of Nickel, Cobalt and Platinum-Group Metals*, Elsevier Scientific Publishing Company.
9. Engelbrecht J.A. and Woodburn E.T., 1975; *The effects of froth height, aeration rate, and gas precipitation of flotation*; Journal of the South African Institute of Mining and Metallurgy; October:125-132.
10. Harris A., Venkatesan L., Greyling M., 2013; A practical approach to plant-scale flotation optimisation. *The Journal of The Southern African Institute of Mining and Metallurgy;* Vol. 113. pp.263-272.

11. King R.P.; *Modeling and Simulation of Mineral Processing Systems*; Butterworth-Heinemann; 2001.
12. Klimpel R.R.,1999; *A Review of Sulphide Meneral Collector Practise; Advances in Flotation Technology*; Parekh B.K. & Miller, J.D., Society for Mining, Metallurgy and Exploration, Inc.; pp 115-127.
13. Khraisheh M, Holland C, Creany C, Harris P, Parolis L, 2005; *Effect of molecular weight and concentration on the adsorption of CMC onto talc at different ionic strength*; International Journal of Mineral Processing 75 (2005) 197-206; Elsevier Scientific Publishing Company.
14. Lawson V, Hill G, Kormos L, Marrs G, 2014; The Separation of Pentlandite from Chalcopyrite, Pyrrhotite and Gangue in Nickel Projects Throughout the World; AusIMM Mill Operators Conference.
15. Mendiratta, 2000; *Kinetic Studies of Sulfide Mineral Oxidation and Xanthate Adsorption* (Doctoral Dissertation); Retreived from Virginia Tech website: http://vtechworks.lib.vt.edu/handle/10919/27545 on 13 September 2015.
16. Salopek B, Krasi D, Filipovi S, 1992; *Measurement and Application of Zeta-Potential*; Rudarsko-geoloiko-naftni zbornik, Volume 4 pp. 147-151; University of Zagreb.
17. Singh, R; *On the Development of an Integrated Flotation Process*; Mineral Processing: Recent Advances and Future Trends; Allied Publishers Limited; 1995; pp 390-398.
18. Steenberg H, Harris P.J., 1984; *Adsorption of carboxymethylcellulose, guar gum, and starch onto talc, sulphides, oxides, and salt-type minerals*; S. Afr. J. Chem., 1984 37(3).
19. Trahar, W.J. (1976); *The selective flotation of galena from sphalerite with special reference to the effects of particle size*; International Journal of Mineral Processing, 3 (1976) 151-166; Elsevier Scientific Publishing Company.
20. Wang L, Peng Y, Runge K, Bradshaw D, 2014; *A review of entrainment: Mechanisms, contributing factors and modelling in flotation*; Minerals Engineering 70 (2015) 77-91.
21. Wiese, J.G., 2009; Investigating Depressant Behaviour in the Flotation of Selected Merensky Ores (Masters Dissertation); University of Cape Town.

Appendix A – Detailed Mass Balance

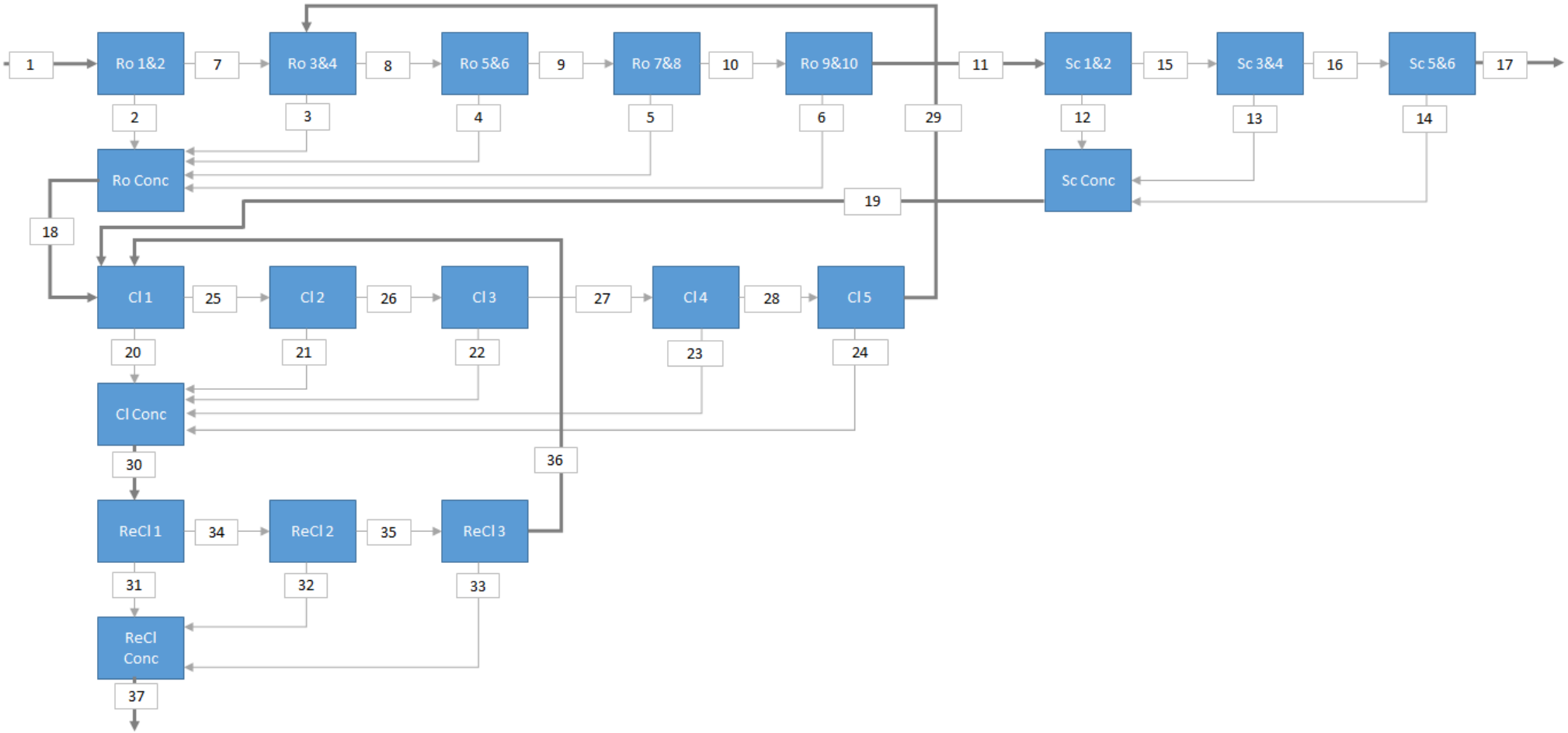


Figure A1: Block Flow Diagram of the MMZ Flotation Circuit

TableA1: MMZ Flotation Circuit Mass Balance

Stream No.	Units	1	2	3	4	5	6	7	8	9	10	11	12	13	14	15	16
Stream Name		Feed	Ro 1&2 Conc	Ro 3&4 Conc	Ro 5&6 Conc	Ro 7&8 Conc	Ro 9&10 Conc	Ro 1&2 Tails	Ro 3&4 Tails	Ro 5&6 Tails	Ro 7&8 Tails	Ro 9&10 Tails	Sc 1&2 Conc	Sc 3&4 Conc	Sc 5&6 Conc	Sc 1&2 Tails	Sc 3&4 Tails
Solids	t/h	655.8	29.8	20.0	15.0	17.6	1.1	625.9	686.0	671.0	653.4	652.3	3.9	3.9	3.9	648.3	644.4
Water	t/h	2060.2	74.2	47.9	29.0	34.6	2.9	1989.8	2107.4	2085.3	2058.5	2055.7	16.8	15.4	12.9	2039.8	2024.9
Total	t/h	2716.0	104.1	67.9	44.0	52.2	4.0	2615.8	2793.4	2756.2	2711.9	2708.0	20.8	19.3	16.8	2688.2	2669.4
Mass Pull	%	100.0	4.5	3.1	2.3	2.7	0.2	95.5	104.6	102.3	99.6	99.5	0.6	0.6	0.6	98.9	98.3
Solids Conc	%	24.1	28.7	29.5	34.1	33.6	27.9	23.9	24.6	24.3	24.1	24.1	19.0	20.2	23.3	24.1	24.1
Density	t/m^3	1.20	1.25	1.25	1.31	1.30	1.24	1.20	1.20	1.20	1.20	1.20	1.15	1.16	1.19	1.20	1.20
Volume	m^3/h	2265.1	83.6	54.1	33.7	40.1	3.2	2185.4	2321.8	2295.0	2262.7	2259.6	18.1	16.6	14.1	2242.4	2226.3
SIBX	g/t	140	-	-	-	-	-	40	-	-	-	40	-	-	-	-	-
Depressant	g/t	150	-	-	-	-	-	-	-	-	-	60	-	-	-	-	-
Grade																	
Ni	%	0.33	3.18	2.10	1.52	0.83	0.78	0.20	0.15	0.12	0.10	0.10	0.66	0.52	0.61	0.10	0.10
Cu	%	0.13	2.25	0.45	0.27	0.11	0.15	0.03	0.02	0.02	0.01	0.01	0.12	0.09	0.14	0.01	0.01
Fe	%	12.12	17.90	13.51	10.75	9.52	8.23	11.85	11.55	11.57	11.62	11.63	12.03	8.00	8.39	11.63	11.65
MgO	%	20.40	23.21	19.50	21.36	19.46	22.51	20.26	20.79	20.78	20.81	20.81	22.98	24.83	25.18	20.80	20.77
Distribution																	
Ni	%	100.0	43.5	19.2	10.5	6.7	0.4	56.5	48.3	37.9	31.2	30.8	1.2	0.9	1.1	29.6	28.6
Cu	%	100.0	77.2	10.5	4.6	2.2	0.2	22.8	17.3	12.7	10.5	10.3	0.5	0.4	0.6	9.7	9.3
Fe	%	100.0	6.7	3.4	2.0	2.1	0.1	93.3	99.7	97.6	95.5	95.4	0.6	0.4	0.4	94.8	94.4
MgO	%	100.0	5.2	2.9	2.4	2.6	0.2	94.8	106.6	104.2	101.7	101.5	0.7	0.7	0.7	100.8	100.1

Stream No.		17	18	19	-	-	20	21	22	23	24	25	26	27	28	29	30
Stream Name	**Units**	Sc 5&6 Tails	Comb Ro Conc	Comb Sc Conc	Sc & Ro Conc	Cl Feed	Cl 1 Conc	Cl 2 Conc	Cl 3 Conc	Cl 4 Conc	Cl 5 Conc	Cl 1 Tails	Cl 2 Tails	Cl 3 Tails	Cl 4 Tails	Cl 5 Tails	Comb Cl Conc
Solids	t/h	95.3	120.1	17.1	9.9	5.9	1.2	5.9	103.0	93.1	87.2	86.0	80.1	40.0	10.6	2.1	2.6
Water	t/h	231.7	423.4	57.0	79.6	31.8	10.7	78.6	287.8	239.6	215.4	209.9	180.7	206.2	28.9	7.5	15.0
Total	t/h	327.0	543.5	74.1	89.5	37.7	11.9	84.5	390.7	332.7	302.6	295.8	260.8	246.2	39.5	9.5	17.6
Mass Pull	%	14.5	18.3	2.6	1.5	0.9	0.2	0.9	15.7	14.2	13.3	13.1	12.2	6.1	1.6	0.3	0.4
Solids Conc	%	29.1	22.1	23.1	11.0	15.7	9.9	7.0	26.4	28.0	28.8	29.1	30.7	16.2	26.8	21.7	14.7
Density	t/m^3	1.25	1.18	1.19	1.08	1.12	1.07	1.05	1.22	1.24	1.25	1.25	1.27	1.13	1.23	1.17	1.11
Volume	m^3/h	261.5	460.9	62.3	82.7	33.6	11.1	80.5	319.9	268.7	242.6	236.7	205.7	218.7	32.2	8.1	15.8
SIBX	g/t	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
Depressant	g/t	-	80	-	-	-	-	-	-	20	-	-	-	160	-	-	-
Grade																	
Ni	%	1.91	2.16	5.61	5.92	9.29	3.41	3.60	1.58	1.12	0.57	0.53	0.30	5.87	9.16	13.15	13.06
Cu	%	0.88	0.32	1.04	1.04	3.74	0.55	0.76	0.41	0.34	0.11	0.10	0.05	2.20	5.24	5.65	4.73
Fe	%	13.15	5.98	28.13	28.13	34.63	22.72	22.72	13.82	12.30	10.78	10.62	9.72	27.18	30.57	33.61	31.77
MgO	%	21.58	24.31	12.27	12.27	6.43	16.82	12.84	21.60	22.59	23.69	23.79	24.60	10.99	6.00	4.87	5.20
Distribution																	
Ni	%	83.5	118.7	44.0	26.8	25.2	1.8	9.8	74.7	47.9	22.6	20.8	11.0	107.7	44.5	12.4	15.5
Cu	%	96.2	44.4	20.4	11.8	25.5	0.7	5.2	48.1	36.3	10.8	10.1	4.9	101.1	63.8	13.4	14.1
Fe	%	15.8	9.0	6.1	3.5	2.6	0.3	1.7	17.9	14.4	11.8	11.5	9.8	13.7	4.1	0.9	1.0
MgO	%	15.4	21.8	1.6	0.9	0.3	0.1	0.6	16.6	15.7	15.4	15.3	14.7	3.3	0.5	0.1	0.1

Stream No.	Units	31	32	33	34	35	36	37	31	32	33	34	35	36	37
Stream Name		ReCl 1 Conc	ReCl 2 Conc	ReCl 3 Conc	ReCl 1 Tails	ReCl 2 Tails	ReCl 3 Tails	Comb ReCl Conc	ReCl 1 Conc	ReCl 2 Conc	ReCl 3 Conc	ReCl 1 Tails	ReCl 2 Tails	ReCl 3 Tails	Comb ReCl Conc
Solids	t/h	10.6	2.1	2.6	29.4	27.3	24.8	15.25	10.6	2.1	2.6	29.4	27.3	24.8	15.25
Water	t/h	28.9	7.5	15.0	206.9	205.4	191.7	48.13	28.9	7.5	15.0	206.9	205.4	191.7	48.13
Total	t/h	39.5	9.5	17.6	236.3	232.7	216.5	63.37	39.5	9.5	17.6	236.3	232.7	216.5	63.37
Mass Pull	%	1.6	0.3	0.4	4.5	4.2	3.8	2.32	1.6	0.3	0.4	4.5	4.2	3.8	2.32
Solids Conc	%	26.8	21.7	14.7	12.4	11.8	11.4	24.06	26.8	21.7	14.7	12.4	11.8	11.4	24.06
Density	t/m^3	1.23	1.17	1.11	1.09	1.09	1.09	1.20	1.23	1.17	1.11	1.09	1.09	1.09	1.20
Volume	m^3/h	32.2	8.1	15.8	216.1	213.9	199.4	52.89	32.2	8.1	15.8	216.1	213.9	199.4	52.89
SIBX	g/t	-	-	-	-	-	-	-	-	-	-	-	-	-	-
Depressant	g/t	-	-	-	-	-	-	-	-	-	-	-	-	-	-
Grade															
Ni	%	9.16	13.15	13.06	4.68	4.04	3.10	10.36	9.16	13.15	13.06	4.68	4.04	3.10	10.36
Cu	%	5.24	5.65	4.73	1.10	0.76	0.35	5.21	5.24	5.65	4.73	1.10	0.76	0.35	5.21
Fe	%	30.57	33.61	31.77	25.96	25.38	24.71	31.19	30.57	33.61	31.77	25.96	25.38	24.71	31.19
MgO	%	6.00	4.87	5.20	12.78	13.38	14.24	5.71	6.00	4.87	5.20	12.78	13.38	14.24	5.71
Distribution															
Ni	%	44.5	12.4	15.5	63.1	50.7	35.2	72.5	44.5	12.4	15.5	63.1	50.7	35.2	72.5
Cu	%	63.8	13.4	14.1	37.3	23.9	9.8	91.3	63.8	13.4	14.1	37.3	23.9	9.8	91.3
Fe	%	4.1	0.9	1.0	9.6	8.7	7.7	6.0	4.1	0.9	1.0	9.6	8.7	7.7	6.0
MgO	%	0.5	0.1	0.1	2.8	2.7	2.6	0.7	0.5	0.1	0.1	2.8	2.7	2.6	0.7

TableA2: Mineralogical Balance Factors

Estimated Mass Pull	4.008						
Balance Factors:							
Concentrate							
+75	1.002	0.995	1.001	0.999	1.000	0.998	1.000
-75+38	1.018	0.970	0.947	0.994	1.000	1.001	1.000
-38+24.7	1.135	1.004	1.003	0.994	1.000	1.001	1.000
-24.7+9.3	0.954	0.971	1.004	0.995	1.000	1.001	1.000
-9.3	1.006	0.987	1.003	0.998	1.000	1.001	1.000
Tailings							
+75	1.054	0.992	1.001	0.962	1.096	1.002	0.982
-75+38	0.978	0.991	1.000	0.965	1.016	1.003	0.997
-38+24.7	0.985	0.995	1.000	0.973	1.005	1.003	0.999
-24.7+9.3	0.934	0.996	1.000	0.973	1.008	1.002	0.998
-9.3	0.984	0.981	1.001	0.970	1.012	1.002	0.998

Appendix B – Experimental Procedures

1. Milling Curve

The cumulative mass passing 75 µm was plotted against time. A straight line was fitted to the data and the time to achieve 67% passing 75 µm was interpolated. The raw data and solved time is presented in Table B1 and the milling curve is presented in Figure B1.

Table B1: Milling Curve Data

Time	**Passing 75 µm**
min	**%**
0	15.3
15	55.1
25	73.5
30	89.4
Solved Time	**Passing 75 µm**
21	67

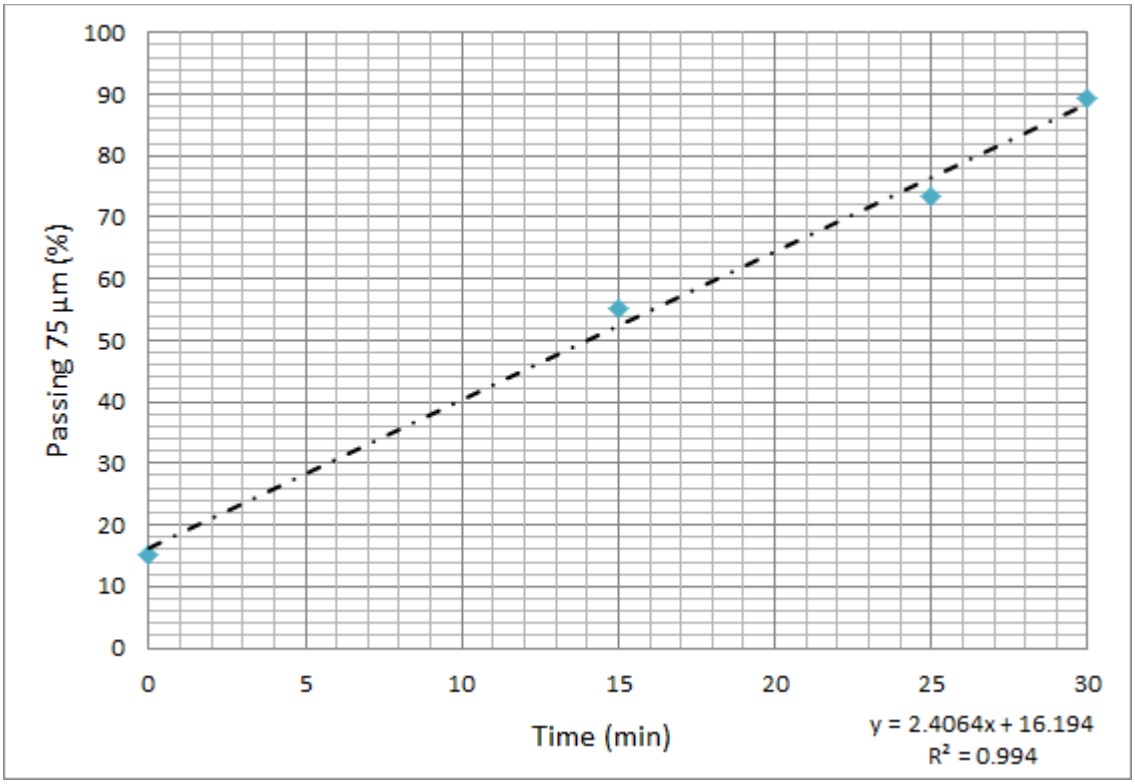


Figure B1: Milling Curve Plot

2. Laboratory Procedure for Chemical Analysis:

2.1 ICP

a. Preparation of 50% (v/v) Hydrochloric acid solution

1. Using a 500 ml measuring cylinder, measure 500 ml of de-ionized water and transfer the water into a 2.5 L empty HCL container.
2. Using a 500 ml measuring cylinder, measure 500 ml of 32% A.R hydrochloric acid into the 2.5 L container containing the 500ml of de-ionized water.
3. Gently mix the solution before use.

b. Method

1. Tare a clean zirconium crucible on a four decimal place analytical balance.
2. Accurately weigh 0.2 g ±0.0005 g of samples and record the sample weight on a worksheet.
3. Using plastic weighing boat, weigh 2.0 g ±0.3 g of sodium peroxide, on a top pan balance, into the same crucible. Using the crucible tray, swirl the crucible contents for a uniform mix.
4. Using a long tray tong, place the tray containing crucibles into a muffle furnace preheated to 720 °C. Keep the tray inside the furnace for 15 minutes.
5. Using a long tray tong remove the crucible tray from the furnace and place the tray on a steel plate. Observe if the colour of the sample is brick-red. If the colour of the samples is not brick-red, then return the tray with the crucibles containing samples which were not totally fused into the furnace.
6. If the colour of the samples is brick-red, allow the crucibles to cool to room temperature.
7. Prepare clean and labelled 400 ml tall form glass beakers according to the worksheet.
8. Place the cooled crucible on its side into a 400 ml tall form glass beaker.
9. Carefully and swiftly, add 100 ml to 150 ml of de-ionized water and immediately cover the beaker with a watch glass.

10. When the reaction is no longer vigorous, add 40ml of 50% (v/v) hydrochloric acid.
11. Swirl the beaker gently to mix the solution.
12. Warm the solution on a hot plate (200-260 °C) for a few minutes; the solution must not boil.
13. If the solution is not clear or has black substance, then repeat step 6.2.1
14. Remove from the hot plate and allow cooling.
15. Using a fine-tip squeeze bottle, quantitatively rinse the watch glass with de-ionized water.
16. Using plastic tweezers and fine tips bottle with de-ionized water, remove the crucible from the beaker, rinsing the crucible into the beaker at least three times. Ensure that the crucible is rinsed/washed thoroughly.
17. Using a dispenser, add 5 ml of Scandium solution into an empty clean 500ml glass volumetric flask.
18. Transfer the solution from the beaker into a 500 ml glass volumetric flask. Rinse the beaker with de-ionized water each time transferring the washings into the flask.
19. Make the solution up to the mark with de-ionized water and close the flasks with stopper.
20. Carefully swirl the flasks to mix contents.
21. Prepare a 60-place ICP-OES plastic rack with the the samples to be analysed.
22. Transfer solutions into clean ICP-OES test tubes.
23. Take the test tube rack to ICP-OES room for analysis.
24. Analyse samples on the ICP instrument

2.2 AA

a. Digestion of Samples

1. Tare clean weighing boat on a four decimal point analytical balance with a readability of 0.0001 g.
2. Weigh accurately the actual sample weight as tabulated below to ± 0.0005 g of sample.
3. Carefully transfer the weighed sample from the weighing boat into a glass beaker.
4. Using the calibrated dispensette, carefully dispense 5 ml of nitric acid and 15 ml of hydrochloric acid into the 100 ml beaker containing the sample.

5. Put the beaker covered with clean watch glass on the hotplate (200-260 °C) and observe the reaction of the acid and samples until the solution just start to boil.
6. Once the samples start boiling, remove immediately from the hotplate and leave on to cool for approximately 5 minutes.
7. Return the beakers to the hotplate and leave to digest until the entire nitrous fumes have evolved and subsided.
8. Remove the beakers from the hotplate and allow cooling to room temperature.
9. Once cooled, rinse all watch glasses transfer solution into beakers, which they were covering, and then transfer the entire solution from the beaker into 100 ml volumetric flasks.
10. Carefully rinse the beaker with electro deionised (EDI) water and transfer the solution into the volumetric flask.
11. Make up to the mark with electro deionised (EDI) water.
12. Shake the final solution.
13. Analyse samples on the AA instrument

Appendix C – Chapter 4 Appendices

1. Collector and Depressant Interaction Testwork Conditions and Average Results

Testwork was conducted in duplicate. Average results are provided in the first section of Appendix C. Standard deviations of the testwork are presented in the second section of Appendix C.

Table C1: Test 1 Average Results

REAGENT DOSAGE		
Reagent	**Dosage (g/t)**	**Conditioning Time (min)**
Depressant	0	2
Collector	0	Added to Mill

TESTWORK RESULTS																		
Product	**Time**	**Solids Conc**	**Solids Mass**		**Grade**							**Recovery**						
					Ni	**Cu**	**Fe**	**MgO**	**SiO_2**	**Cr_2O_3**	**Al_2O_3**	**Ni**	**Cu**	**Fe**	**MgO**	**SiO_2**	**Cr_2O_3**	**Al_2O_3**
	min	**%**	**g**	**Rel %**	**%**							**%**						
Conc 1	1	22	96.83	4.85	1.94	3.34	13.10	20.15	38.29	0.12	2.69	20.9	75.7	5.0	5.2	4.5	2.1	2.5
Conc 2	2	16	58.94	2.95	2.79	0.76	12.75	20.77	40.15	0.18	3.13	18.3	10.5	3.0	3.3	2.9	2.0	1.7
Conc 3	7	9	86.56	4.33	1.99	0.29	13.40	20.13	39.67	0.22	4.07	19.2	5.8	4.6	4.7	4.1	3.5	3.3
Conc 4	20	5	101.39	5.07	0.83	0.07	13.26	18.14	36.85	0.28	4.74	9.4	1.6	5.4	4.9	4.5	5.2	4.6
Conc Total		**9**	**343.71**	**17.20**	**1.77**	**1.16**	**13.16**	**19.66**	**38.53**	**0.20**	**3.71**	**67.8**	**93.6**	18.0	18.1	16.0	12.8	12.1
Tails			1654.74	82.80	0.18	0.02	12.45	18.45	42.04	0.28	5.61	32.2	6.4	82.0	81.9	84.0	87.2	87.9
Head (calc.)			1998.45	100.00	0.450	0.21	12.57	18.66	41.44	0.27	5.28	100.0	100.0	100.0	100.0	100.0	100.0	100.0
Head (meas.)					0.39	0.19	11.79	18.77	42.68	0.39	5.50							
Combined Concentrates																		
Conc 1	1		101.08	5.05	1.89	3.24	12.76	19.90	37.90	0.12	2.81	21.5	77.1	4.8	5.3	4.6	2.2	2.7
Conc 1+2	3		162.84	8.14	2.24	2.29	12.90	20.19	38.83	0.14	3.00	40.9	87.8	7.9	8.7	7.6	4.1	4.6
Conc 1+2+3	10		253.12	12.66	2.06	1.54	12.91	20.09	39.16	0.17	3.41	58.5	91.8	12.2	13.4	12.0	7.9	8.2

Conc 1+2+3+4	30		350.03	17.50	1.71	1.13	13.12	19.82	38.97	0.22	3.86	67.2	93.4	17.2	18.3	16.5	14.3	12.8

Table C2: Test 2 Average Results

REAGENT DOSAGE																		
Reagent					**Dosage (g/t)**							**Conditioning Time (min)**						
Depressant					0							2						
Collector					50							Added to Mill						
TESTWORK RESULTS																		
Product	**Time**	**Solids Conc**	**Solids Mass**		**Grade**							**Recovery**						
					Ni	**Cu**	**Fe**	**MgO**	**SiO_2**	**Cr_2O_3**	**Al_2O_3**	**Ni**	**Cu**	**Fe**	**MgO**	**SiO_2**	**Cr_2O_3**	**Al_2O_3**
	min	**%**	**g**	**Rel %**	**%**							**%**						
Conc 1	1	26	68.19	3.41	5.41	4.68	22.69	12.02	23.83	0.08	1.72	46.5	79.3	7.1	2.4	2.0	1.4	1.1
Conc 2	2	17	57.14	2.86	2.80	0.76	17.40	17.61	35.32	0.14	2.72	20.1	10.9	4.5	2.9	2.5	2.0	1.5
Conc 3	7	11	110.44	5.53	0.76	0.13	16.27	18.63	37.76	0.17	3.70	10.6	3.7	8.2	6.0	5.2	4.8	3.9
Conc 4	20	6	130.53	6.53	0.32	0.05	14.92	18.33	38.32	0.18	4.76	5.3	1.6	8.9	7.0	6.2	6.2	5.9
Conc Total		9	**366.29**	**18.33**	**1.79**	**1.05**	**17.16**	**17.14**	**34.98**	**0.15**	**3.56**	**82.4**	**95.5**	**28.7**	18.2	**15.9**	**14.5**	**12.4**
Tails			1554.36	77.78	0.09	0.01	10.03	18.10	43.53	0.21	5.90	17.6	4.5	71.3	81.8	84.1	85.5	87.6
Head (calc.)			1920.65	96.11	0.41	0.21	11.39	17.92	41.90	0.20	5.45	100.0	100.0	100.0	100.0	100.0	100.0	100.0
Head (meas.)					0.39	0.19	11.79	18.77	42.68	0.39	5.50							
Combined Concentrates																		
Conc 1	1		3.41	5.41	4.68	22.69	12.02	23.83	0.08	1.72	46.5	79.3	7.1	2.4	2.0	1.4	1.1	3.41

Conc 1+2	3		6.27	4.22	2.89	20.28	14.57	29.06	0.10	2.18	66.6	90.2	11.6	5.3	4.5	3.4	2.6	6.27
Conc 1+2+3	10		11.80	2.60	1.60	18.40	16.47	33.14	0.13	2.89	77.1	93.9	19.8	11.3	9.7	8.2	6.5	11.80
Conc 1+2+3+4	30		18.33	1.79	1.05	17.16	17.14	34.98	0.15	3.56	82.4	95.5	28.7	18.2	15.9	14.5	12.4	18.33

Table C3: Test 3 Average Results

REAGENT DOSAGE		
Reagent	**Dosage (g/t)**	**Conditioning Time (min)**
Depressant	50	2
Collector	50	Added to Mill

TESTWORK RESULTS

Product	**Time**	**Solids Conc**	**Solids Mass**		**Grade**							**Recovery**						
					Ni	**Cu**	**Fe**	**MgO**	**SiO_2**	**Cr_2O_3**	**Al_2O_3**	**Ni**	**Cu**	**Fe**	**MgO**	**SiO_2**	**Cr_2O_3**	**Al_2O_3**
	min	**%**	**g**	**Rel %**	**%**							**%**						
Conc 1	1	21	67.46	3.38	4.18	4.59	21.17	15.81	29.56	0.10	1.56	33.9	77.0	5.7	2.8	2.3	1.0	1.0
Conc 2	2	20	48.71	2.44	3.73	0.91	17.04	20.51	38.19	0.17	2.18	21.9	11.0	3.3	2.6	2.2	1.2	1.0
Conc 3	7	13	112.31	5.62	1.59	0.18	20.29	18.81	36.66	0.27	3.18	21.4	5.2	9.1	5.6	4.8	4.5	3.4
Conc 4	20	7	138.01	6.91	0.45	0.07	19.45	18.00	36.57	0.29	4.23	7.5	2.5	10.7	6.5	5.9	5.9	5.5
Conc Total		11	**366.48**	**18.34**	**1.92**	**1.05**	**19.71**	**18.18**	**35.52**	**0.23**	**3.14**	**84.6**	**95.7**	**28.7**	17.5	**15.2**	**12.6**	**10.9**
Tails			1610.92	80.61	0.08	0.01	11.14	19.46	44.91	0.36	5.85	15.4	4.3	71.3	82.5	84.8	87.4	89.1
Head (calc.)			1977.39	98.95	0.42	0.20	12.73	19.22	43.17	0.34	5.35	100.0	100.0	100.0	100.0	100.0	100.0	100.0
Head (meas.)					0.39	0.19	11.79	18.77	42.68	0.39	5.50							

Combined Concentrates																		
Conc 1	1		67	3.38	4.18	4.59	21.17	15.81	29.56	0.10	1.56	33.9	77.0	5.7	2.8	2.3	1.0	1.0
Conc 1+2	3		116	5.81	3.99	3.05	19.44	17.78	33.18	0.13	1.82	55.8	88.0	9.0	5.4	4.5	2.3	2.0
Conc 1+2+3	10		228	11.43	2.81	1.64	19.86	18.29	34.89	0.20	2.49	77.2	93.1	18.0	11.0	9.3	6.8	5.4
Conc 1+2+3+4	30		366	18.34	1.92	1.05	19.71	18.18	35.52	0.23	3.14	84.6	95.7	28.7	17.5	15.2	12.6	10.9

Table C4: Test 4 Average Results

REAGENT DOSAGE		
Reagent	**Dosage (g/t)**	**Conditioning Time (min)**
Depressant	150	2
Collector	50	Added to Mill

TESTWORK RESULTS

Product	**Time**	**Solids Conc**	**Solids Mass**		**Grade**							**Recovery**						
					Ni	**Cu**	**Fe**	**MgO**	**SiO_2**	**Cr_2O_3**	**Al_2O_3**	**Ni**	**Cu**	**Fe**	**MgO**	**SiO_2**	**Cr_2O_3**	**Al_2O_3**
	min	**%**	**g**	**Rel %**	**%**							**%**						
Conc 1	1	9	26.74	1.34	5.27	5.06	26.62	10.56	21.26	0.13	1.98	18.2	28.6	3.1	0.8	0.7	0.5	0.5
Conc 2	2	11	53.46	2.67	6.41	5.43	27.99	8.85	17.72	0.10	1.66	44.4	61.4	6.5	1.3	1.2	0.8	0.9
Conc 3	7	9	116.92	5.85	1.08	0.21	18.71	18.43	36.69	0.24	3.35	16.4	5.3	9.5	6.0	5.3	4.1	3.9
Conc 4	20	6	131.91	6.60	0.30	0.06	16.13	19.20	39.03	0.28	4.20	5.2	1.7	9.2	7.1	6.3	5.4	5.5
Conc Total		7	**329.01**	**16.46**	**1.98**	**1.39**	**19.83**	**16.54**	**33.29**	**0.23**	**3.30**	**84.1**	**97.1**	**28.3**	**15.2**	**13.5**	**10.8**	**10.8**
Tails			1577.27	78.92	0.08	0.01	10.47	19.25	44.53	0.39	5.67	15.9	2.9	71.7	84.8	86.5	89.2	89.2

Head (calc.)			1906.28	95.39	0.41	0.25	12.08	18.79	42.59	0.36	5.26	100.0	100.0	100.0	100.0	100.0	100.0	100.0
Head (meas.)					0.39	0.19	11.79	18.77	42.68	0.39	5.50							
Combined Concentrates																		
Conc 1	1		26.74	1.34	5.27	5.06	26.62	10.56	21.26	0.13	1.98	18.2	28.6	3.1	0.8	0.7	0.5	0.5
Conc 1+2	3		80.19	4.01	6.03	5.30	27.53	9.42	18.90	0.11	1.76	62.6	90.1	9.6	2.1	1.9	1.3	1.4
Conc 1+2+3	10		197.11	9.86	3.10	2.28	22.30	14.76	29.45	0.19	2.70	79.0	95.3	19.1	8.1	7.1	5.4	5.3
Conc 1+2+3+4	30		329.01	16.46	1.98	1.39	19.83	16.54	33.29	0.23	3.30	84.1	97.1	28.3	15.2	13.5	10.8	10.8

Table C5: Test 5 Average Results

REAGENT DOSAGE		
Reagent	**Dosage (g/t)**	**Conditioning Time (min)**
Depressant	300	2
Collector	50	Added to Mill

TESTWORK RESULTS

Product	**Time**	**Solids Conc**	**Solids Mass**		**Grade**							**Recovery**						
					Ni	**Cu**	**Fe**	**MgO**	**SiO_2**	**Cr_2O_3**	**Al_2O_3**	**Ni**	**Cu**	**Fe**	**MgO**	**SiO_2**	**Cr_2O_3**	**Al_2O_3**
	min	**%**	**g**	**Rel %**	**%**							**%**						
Conc 1	1	29	22.81	1.14	6.80	10.47	32.45	4.22	8.22	0.06	0.96	20.6	64.6	3.2	0.3	0.2	0.2	0.2
Conc 2	2	15	36.90	1.85	7.52	1.95	27.29	9.94	20.03	0.19	2.22	36.8	19.5	4.4	1.0	0.9	1.1	0.8
Conc 3	7	8	93.47	4.68	1.48	0.31	19.35	16.83	34.19	0.28	3.67	18.4	7.9	7.9	4.3	3.9	3.9	3.4
Conc 4	20	6	131.40	6.57	0.34	0.07	15.62	19.74	39.03	0.28	4.20	6.0	2.5	8.9	7.1	6.2	5.5	5.4

Conc Total		8	**284.58**	**14.24**	**2.17**	**1.23**	**19.71**	**16.27**	**32.51**	**0.25**	**3.51**	**81.7**	**94.6**	**24.4**	**12.7**	**11.3**	**10.7**	**9.8**
Tails			1674.46	83.79	0.08	0.01	10.36	19.03	43.58	0.36	5.51	18.3	5.4	75.6	87.3	88.7	89.3	90.2
Head (calc.)			1959.04	98.03	0.39	0.19	11.72	18.63	41.97	0.34	5.22	100.0	100.0	100.0	100.0	100.0	100.0	100.0
Head (meas.)					0.39	0.19	11.79	18.77	42.68	0.39	5.50							
Combined Concentrates																		
Conc 1	1		22.81	1.14	6.80	10.47	32.45	4.22	8.22	0.06	0.96	20.6	64.6	3.2	0.3	0.2	0.2	0.2
Conc 1+2	3		59.71	2.99	7.24	5.20	29.26	7.76	15.52	0.14	1.74	57.3	84.1	7.6	1.3	1.1	1.3	1.0
Conc 1+2+3	10		153.18	7.66	3.73	2.22	23.22	13.29	26.91	0.23	2.91	75.7	92.0	15.5	5.6	5.0	5.2	4.4
Conc 1+2+3+4	30		284.58	14.24	2.17	1.23	19.71	16.27	32.51	0.25	3.51	81.7	94.6	24.4	12.7	11.3	10.7	9.8

Table C6: Test 6 Average Results

REAGENT DOSAGE		
Reagent	**Dosage (g/t)**	**Conditioning Time (min)**
Depressant	0	2
Collector	100	Added to Mill

TESTWORK RESULTS

Product	**Time**	**Solids Conc**	**Solids Mass**		**Grade**							**Recovery**						
					Ni	**Cu**	**Fe**	**MgO**	**SiO_2**	**Cr_2O_3**	**Al_2O_3**	**Ni**	**Cu**	**Fe**	**MgO**	**SiO_2**	**Cr_2O_3**	**Al_2O_3**
	min	**%**	**g**	**Rel %**	**%**							**%**						
Conc 1	1	28	71.17	3.56	4.01	3.73	19.68	15.43	29.54	0.12	1.38	36.3	74.2	6.1	3.1	2.6	1.3	1.0
Conc 2	2	16	69.36	3.47	2.77	0.54	17.54	18.68	36.68	0.22	2.53	24.5	10.5	5.3	3.7	3.2	2.4	1.8

Conc 3	7	9	165.10	8.26	0.87	0.17	17.73	18.13	37.28	0.29	3.88	18.2	7.7	12.7	8.5	7.7	7.5	6.5
Conc 4	20	5	175.30	8.77	0.30	0.05	14.96	18.67	38.99	0.29	4.92	6.7	2.7	11.3	9.3	8.5	8.2	8.8
Conc Total		7	**480.92**	**24.06**	**1.40**	**0.71**	**16.98**	**18.01**	**36.67**	**0.25**	**3.70**	**85.7**	**95.1**	**35.3**	**24.5**	**22.0**	**19.4**	**18.1**
Tails			1432.82	71.70	0.08	0.01	10.44	18.62	43.67	0.35	5.61	14.3	4.9	64.7	75.5	78.0	80.6	81.9
Head (calc.)			1913.74	95.76	0.41	0.19	12.08	18.47	41.91	0.33	5.13	100.0	100.0	100.0	100.0	100.0	100.0	100.0
Head (meas.)					0.39	0.19	11.79	18.77	42.68	0.39	5.50							
Combined Concentrates																		
Conc 1	1		71.17	3.56	4.01	3.73	19.68	15.43	29.54	0.12	1.38	36.3	74.2	6.1	3.1	2.6	1.3	1.0
Conc 1+2	3		140.53	7.03	3.40	2.16	18.62	17.04	33.06	0.17	1.95	60.8	84.7	11.3	6.8	5.8	3.7	2.8
Conc 1+2+3	10		305.62	15.29	2.03	1.08	18.14	17.63	35.34	0.23	2.99	79.0	92.4	24.0	15.2	13.5	11.3	9.3
Conc 1+2+3+4	30		480.92	24.06	1.40	0.71	16.98	18.01	36.67	0.25	3.70	85.7	95.1	35.3	24.5	22.0	19.4	18.1

Table C7: Test 7 Average Results

REAGENT DOSAGE																		
Reagent				**Dosage (g/t)**							**Conditioning Time (min)**							
Depressant				50							2							
Collector				100							Added to Mill							
TESTWORK RESULTS																		
Product	**Time**	**Solids Conc**	**Solids Mass**		**Grade**							**Recovery**						
					Ni	**Cu**	**Fe**	**MgO**	**SiO_2**	**Cr_2O_3**	**Al_2O_3**	**Ni**	**Cu**	**Fe**	**MgO**	**SiO_2**	**Cr_2O_3**	**Al_2O_3**
	min	**%**	**g**	**Rel %**	**%**							**%**						

Conc 1	1	15	55.45	2.77	5.65	4.90	25.72	10.75	20.94	0.11	1.26	40.5	74.9	6.1	1.6	1.4	1.0	0.7
Conc 2	2	18	68.43	3.42	3.01	0.71	20.58	16.88	33.34	0.18	2.34	26.6	13.3	6.0	3.2	2.8	2.0	1.6
Conc 3	7	11	124.33	6.22	0.76	0.15	19.08	18.08	36.50	0.25	3.41	12.1	5.1	10.1	6.2	5.5	5.0	4.1
Conc 4	20	5	154.83	7.75	0.29	0.05	15.50	18.52	38.64	0.28	4.62	5.8	2.3	10.2	7.9	7.2	6.9	7.0
Conc Total		8	**403.04**	**20.17**	**1.63**	**0.86**	**18.87**	**17.04**	**34.64**	**0.23**	**3.40**	**85.1**	**95.7**	**32.4**	**18.8**	**16.9**	**14.9**	**13.4**
Tails			1559.10	78.02	0.07	0.01	10.17	19.01	44.03	0.34	5.68	14.9	4.3	67.6	81.2	83.1	85.1	86.6
Head (calc.)			1962.14	98.18	0.39	0.18	11.96	18.60	42.10	0.32	5.21	100.0	100.0	100.0	100.0	100.0	100.0	100.0
Head (meas.)					0.39	0.19	11.79	18.77	42.68	0.39	5.50							
Combined Concentrates																		
Conc 1	1		55.45	2.77	5.65	4.90	25.72	10.75	20.94	0.11	1.26	40.5	74.9	6.1	1.6	1.4	1.0	0.7
Conc 1+2	3		123.88	6.20	4.20	2.58	22.88	14.14	27.79	0.15	1.86	67.1	88.3	12.1	4.8	4.2	3.0	2.3
Conc 1+2+3	10		248.21	12.42	2.47	1.36	20.98	16.11	32.15	0.20	2.64	79.3	93.3	22.2	11.0	9.7	8.0	6.4
Conc 1+2+3+4	30		403.04	20.17	1.63	0.86	18.87	17.04	34.64	0.23	3.40	85.1	95.7	32.4	18.8	16.9	14.9	13.4

Table C8: Test 8 Average Results

REAGENT DOSAGE							
Reagent				**Dosage (g/t)**		**Conditioning Time (min)**	
Depressant				150		2	
Collector				100		Added to Mill	
TESTWORK RESULTS							
Product	**Time**	**Solids**	**Solids Mass**	**Grade**		**Recovery**	

		Conc			Ni	Cu	Fe	MgO	SiO_2	Cr_2O_3	Al_2O_3	Ni	Cu	Fe	MgO	SiO_2	Cr_2O_3	Al_2O_3
	min	%	g	Rel %	%							%						
Conc 1	1	35	38.66	1.93	7.11	8.26	32.76	5.98	11.77	0.15	1.05	30.2	72.0	4.9	0.6	0.5	0.8	0.4
Conc 2	2	17	62.03	3.10	5.19	1.25	26.15	13.26	26.15	0.26	2.31	35.4	17.5	6.3	2.1	1.8	2.3	1.3
Conc 3	7	11	121.56	6.08	1.10	0.16	20.13	17.81	35.61	0.22	3.59	14.7	4.4	9.5	5.6	4.9	3.7	4.0
Conc 4	20	6	149.36	7.47	0.32	0.04	15.52	18.54	39.30	0.25	4.73	5.2	1.4	9.0	7.1	6.7	5.2	6.4
Conc Total		9	**371.61**	**18.59**	**2.09**	**1.14**	**20.60**	**16.11**	**33.04**	**0.23**	**3.57**	**85.4**	**95.3**	**29.8**	**15.4**	**14.0**	**12.0**	**12.1**
Tails			1713.26	85.73	0.08	0.01	10.55	19.13	44.12	0.37	5.63	14.6	4.7	70.2	84.6	86.0	88.0	87.9
Head (calc.)			2084.87	104.32	0.44	0.21	12.34	18.59	42.14	0.35	5.26	100.0	100.0	100.0	100.0	100.0	100.0	100.0
Head (meas.)					0.39	0.19	11.79	18.77	42.68	0.39	5.50							
Combined Concentrates																		
Conc 1	1		38.66	1.93	7.11	8.26	32.76	5.98	11.77	0.15	1.05	30.2	72.0	4.9	0.6	0.5	0.8	0.4
Conc 1+2	3		100.69	5.04	5.93	3.94	28.69	10.47	20.63	0.22	1.82	65.6	89.5	11.2	2.7	2.4	3.1	1.7
Conc 1+2+3	10		222.25	11.12	3.29	1.87	24.01	14.48	28.82	0.22	2.79	80.3	94.0	20.7	8.3	7.3	6.7	5.6
Conc 1+2+3+4	30		371.61	18.59	2.09	1.14	20.60	16.11	33.04	0.23	3.57	85.4	95.3	29.8	15.4	14.0	12.0	12.1

Table C9: Test 9 Average Results

REAGENT DOSAGE		
Reagent	**Dosage (g/t)**	**Conditioning Time (min)**
Depressant	300	2
Collector	100	Added to Mill

TESTWORK RESULTS																		
Product	**Time**	**Solids Conc**	**Solids Mass**		**Grade**							**Recovery**						
					Ni	**Cu**	**Fe**	**MgO**	**SiO_2**	**Cr_2O_3**	**Al_2O_3**	**Ni**	**Cu**	**Fe**	**MgO**	**SiO_2**	**Cr_2O_3**	**Al_2O_3**
	min	**%**	**g**	**Rel %**	**%**							**%**						
Conc 1	1	28.59	1.43	6.52	9.57	35.13	4.04	7.96	0.07	1.03	19.9	62.0	3.7	0.3	0.2	0.3	0.3	28.59
Conc 2	2	68.02	3.40	5.88	1.45	33.09	9.12	18.08	0.13	2.22	42.6	22.3	8.4	1.5	1.3	1.3	1.3	68.02
Conc 3	7	113.94	5.70	1.45	0.28	26.08	14.42	29.58	0.20	3.46	17.6	7.3	11.1	4.1	3.6	3.4	3.4	113.94
Conc 4	20	171.17	8.57	0.34	0.07	15.82	19.68	39.37	0.21	4.51	6.2	2.6	10.1	8.3	7.3	5.3	6.6	171.17
Conc Total		**381.72**	**19.10**	**2.12**	**1.09**	**23.40**	**15.06**	**30.30**	**0.18**	**3.53**	**86.3**	**94.2**	**33.3**	**14.2**	**12.5**	**10.3**	**11.6**	**381.72**
Tails		1789.16	89.53	0.07	0.01	10.02	19.42	45.30	0.34	5.74	13.7	5.8	66.7	85.8	87.5	89.7	88.4	1789.16
Head (calc.)		2170.88	108.63	0.43	0.20	12.37	18.65	42.66	0.31	5.35	100.0	100.0	100.0	100.0	100.0	100.0	100.0	2170.88
Head (meas.)					0.39	0.19	11.79	18.77	42.68	0.39	5.50							
Combined Concentrates																		
Conc 1	1		28.59	1.43	6.52	9.57	35.13	4.04	7.96	0.07	1.03	19.9	62.0	3.7	0.3	0.2	0.3	0.3
Conc 1+2	3		96.61	4.83	6.07	3.85	33.69	7.62	15.09	0.11	1.87	62.5	84.3	12.1	1.8	1.6	1.6	1.6
Conc 1+2+3	10		210.55	10.54	3.57	1.92	29.57	11.30	22.93	0.16	2.73	80.1	91.6	23.2	5.9	5.2	5.0	4.9
Conc 1+2+3+4	30		381.72	19.10	2.12	1.09	23.40	15.06	30.30	0.18	3.53	86.3	94.2	33.3	14.2	12.5	10.3	11.6

Table C10: Test 10 Average Results

REAGENT DOSAGE		
Reagent	**Dosage (g/t)**	**Conditioning Time (min)**

Depressant					0							2						
Collector					200							Added to Mill						
TESTWORK RESULTS																		
Product	**Time**	**Solids Conc**	**Solids Mass**		**Grade**							**Recovery**						
					Ni	**Cu**	**Fe**	**MgO**	**SiO_2**	**Cr_2O_3**	**Al_2O_3**	**Ni**	**Cu**	**Fe**	**MgO**	**SiO_2**	**Cr_2O_3**	**Al_2O_3**
	min	**%**	**g**	**Rel %**	**%**							**%**						
Conc 1	1	29	85.91	4.30	4.29	3.73	22.99	13.62	25.37	0.09	1.71	38.7	76.5	7.4	2.8	2.4	1.1	1.3
Conc 2	2	18	63.75	3.19	3.41	0.67	20.99	18.04	32.46	0.16	2.25	22.8	10.2	5.0	2.8	2.3	1.4	1.3
Conc 3	7	11	132.10	6.61	1.28	0.18	21.68	17.47	33.04	0.22	3.27	17.8	5.6	10.7	5.6	4.9	4.4	3.9
Conc 4	20	6	249.33	12.48	0.32	0.05	16.20	19.26	38.30	0.26	4.87	8.5	3.2	15.1	11.6	10.6	9.5	10.9
Conc Total		9	**531.09**	**26.58**	**1.57**	**0.75**	**19.24**	**17.75**	**34.20**	**0.21**	**3.65**	**87.8**	**95.5**	**38.3**	**22.8**	**20.2**	**16.4**	**17.4**
Tails			1608.47	80.49	0.07	0.01	10.25	19.90	44.49	0.35	5.71	12.2	4.5	61.7	77.2	79.8	83.6	82.6
Head (calc.)			2139.56	107.06	0.45	0.20	12.48	19.37	41.94	0.32	5.20	100.0	100.0	100.0	100.0	100.0	100.0	100.0
Head (meas.)					0.39	0.19	11.79	18.77	42.68	0.39	5.50							
Combined Concentrates																		
Conc 1	1		85.91	4.30	4.29	3.73	22.99	13.62	25.37	0.09	1.71	38.7	76.5	7.4	2.8	2.4	1.1	1.3
Conc 1+2	3		149.66	7.49	3.91	2.43	22.14	15.50	28.39	0.12	1.94	61.5	86.7	12.4	5.6	4.7	2.5	2.6
Conc 1+2+3	10		281.76	14.10	2.68	1.37	21.93	16.42	30.57	0.17	2.56	79.3	92.4	23.1	11.2	9.6	6.9	6.5
Conc 1+2+3+4	30		531.09	26.58	1.57	0.75	19.24	17.75	34.20	0.21	3.65	87.8	95.5	38.3	22.8	20.2	16.4	17.4

Table C11: Test 11 Average Results

REAGENT DOSAGE		
Reagent	**Dosage (g/t)**	**Conditioning Time (min)**
Depressant	50	2
Collector	200	Added to Mill

TESTWORK RESULTS																		
Product	**Time**	**Solids Conc**	**Solids Mass**		**Grade**							**Recovery**						
					Ni	**Cu**	**Fe**	**MgO**	**SiO_2**	**Cr_2O_3**	**Al_2O_3**	**Ni**	**Cu**	**Fe**	**MgO**	**SiO_2**	**Cr_2O_3**	**Al_2O_3**
	min	**%**	**g**	**Rel %**	**%**							**%**						
Conc 1	1	15	69.23	3.46	4.79	4.53	26.65	11.85	22.13	0.11	1.74	36.1	75.4	7.0	2.0	1.7	1.1	1.1
Conc 2	2	15	103.33	5.17	2.82	0.53	22.43	17.28	31.95	0.19	2.76	31.7	13.1	8.8	4.4	3.7	2.8	2.6
Conc 3	7	10	152.83	7.65	0.78	0.12	18.83	18.89	36.29	0.24	3.76	12.9	4.6	10.9	7.1	6.3	5.2	5.2
Conc 4	20	6	175.44	8.78	0.30	0.05	15.01	19.71	38.66	0.26	4.81	5.7	2.1	9.9	8.5	7.7	6.3	7.6
Conc Total		9	**500.83**	**25.06**	**1.59**	**0.79**	**19.32**	**17.87**	**34.27**	**0.22**	**3.64**	**86.4**	**95.2**	**36.5**	**22.1**	**19.4**	**15.4**	**16.4**
Tails			1617.58	80.94	0.08	0.01	10.39	19.51	44.13	0.37	5.73	13.6	4.8	63.5	77.9	80.6	84.6	83.6
Head (calc.)			2118.41	106.00	0.43	0.20	12.50	19.12	41.80	0.34	5.24	100.0	100.0	100.0	100.0	100.0	100.0	100.0
Head (meas.)					0.39	0.19	11.79	18.77	42.68	0.39	5.50							
Combined Concentrates																		
Conc 1	1		69.23	3.46	4.79	4.53	26.65	11.85	22.13	0.11	1.74	36.1	75.4	7.0	2.0	1.7	1.1	1.1
Conc 1+2	3		172.56	8.63	3.61	2.13	24.12	15.10	28.01	0.16	2.35	67.7	88.5	15.7	6.4	5.5	3.9	3.7
Conc 1+2+3	10		325.39	16.28	2.28	1.19	21.64	16.88	31.90	0.20	3.01	80.7	93.1	26.6	13.6	11.7	9.1	8.8

Conc 1+2+3+4	30		500.83	25.06	1.59	0.79	19.32	17.87	34.27	0.22	3.64	86.4	95.2	36.5	22.1	19.4	15.4	16.4

Table C12: Test 12 Average Results

REAGENT DOSAGE		
Reagent	**Dosage (g/t)**	**Conditioning Time (min)**
Depressant	150	2
Collector	200	Added to Mill

TESTWORK RESULTS

Product	**Time**	**Solids Conc**	**Solids Mass**		**Grade**							**Recovery**						
					Ni	**Cu**	**Fe**	**MgO**	**SiO_2**	**Cr_2O_3**	**Al_2O_3**	**Ni**	**Cu**	**Fe**	**MgO**	**SiO_2**	**Cr_2O_3**	**Al_2O_3**
	min	**%**	**g**	**Rel %**	**%**							**%**						
Conc 1	1	18	28.83	1.44	6.17	8.40	35.43	4.48	8.81	0.06	1.05	21.9	63.1	4.3	0.3	0.3	0.3	0.3
Conc 2	2	26	104.02	5.21	3.60	0.92	32.16	10.04	20.63	0.18	2.42	46.1	24.9	14.0	2.8	2.6	3.1	2.5
Conc 3	7	18	157.58	7.89	0.83	0.14	24.75	15.66	30.76	0.23	3.17	16.1	5.9	16.3	6.6	5.9	6.0	5.0
Conc 4	20	6	157.53	7.88	0.26	0.05	13.82	20.15	39.62	0.21	4.56	5.1	1.9	9.1	8.5	7.6	5.6	7.1
Conc Total		12	**447.96**	**22.42**	**1.62**	**0.82**	**23.32**	**15.21**	**30.11**	**0.20**	**3.35**	**89.1**	**95.8**	**43.7**	**18.3**	**16.5**	**15.0**	**14.9**
Tails			1500.44	75.08	0.06	0.01	8.98	20.26	45.60	0.34	5.71	10.9	4.2	56.3	81.7	83.5	85.0	85.1
Head (calc.)			1948.40	97.50	0.42	0.20	12.27	19.10	42.04	0.31	5.16	100.0	100.0	100.0	100.0	100.0	100.0	100.0
Head (meas.)					0.39	0.19	11.79	18.77	42.68	0.39	5.50							
Combined Concentrates																		
Conc 1	1		28.83	1.44	6.17	8.40	35.43	4.48	8.81	0.06	1.05	21.9	63.1	4.3	0.3	0.3	0.3	0.3

Conc 1+2	3		132.85	6.65	4.16	2.54	32.87	8.83	18.06	0.15	2.12	68.0	88.0	18.3	3.2	2.9	3.4	2.8
Conc 1+2+3	10		290.43	14.53	2.35	1.24	28.46	12.54	24.95	0.19	2.69	84.0	93.9	34.6	9.8	8.8	9.4	7.8
Conc 1+2+3+4	30		447.96	22.42	1.62	0.82	23.32	15.21	30.11	0.20	3.35	89.1	95.8	43.7	18.3	16.5	15.0	14.9

Table C13: Test 13 Average Results

REAGENT DOSAGE		
Reagent	**Dosage (g/t)**	**Conditioning Time (min)**
Depressant	500	2
Collector	100	Added to Mill

TESTWORK RESULTS

Product	**Time**	**Solids Conc**	**Solids Mass**		**Grade**							**Recovery**						
					Ni	**Cu**	**Fe**	**MgO**	**SiO_2**	**Cr_2O_3**	**Al_2O_3**	**Ni**	**Cu**	**Fe**	**MgO**	**SiO_2**	**Cr_2O_3**	**Al_2O_3**
	min	**%**	**g**	**Rel %**	**%**							**%**						
Conc 1	1	22	68.96	3.45	6.70	4.48	26.49	10.93	21.55	0.15	2.14	53.5	79.0	7.1	2.0	1.7	1.5	1.3
Conc 2	2	15	88.61	4.43	1.66	0.41	19.91	17.92	35.55	0.30	3.64	17.0	9.3	6.8	4.1	3.6	3.9	2.8
Conc 3	7	8	67.78	3.39	0.79	0.17	21.41	17.71	34.80	0.24	3.48	6.2	2.9	5.6	3.1	2.7	2.4	2.0
Conc 4	20	6	124.10	6.21	0.34	0.06	18.46	18.16	37.41	0.27	4.64	4.9	1.9	8.9	5.9	5.3	5.0	5.0
Conc Total		9	**349.45**	**17.49**	**2.02**	**1.04**	**20.99**	**16.59**	**33.30**	**0.25**	**3.67**	**81.6**	**93.1**	**28.4**	**15.1**	**13.2**	**12.8**	**11.1**
Tails			1712.01	85.67	0.09	0.02	10.80	18.98	44.72	0.35	5.98	18.4	6.9	71.6	84.9	86.8	87.2	88.9
Head (calc.)			2061.46	103.15	0.42	0.19	12.52	18.57	42.78	0.33	5.59	100.0	100.0	100.0	100.0	100.0	100.0	100.0
Head (meas.)					0.39	0.19	11.79	18.77	42.68	0.39	5.50							

Combined Concentrates																		
Conc 1	1		68.96	3.45	6.70	4.48	26.49	10.93	21.55	0.15	2.14	53.5	79.0	7.1	2.0	1.7	1.5	1.3
Conc 1+2	3		157.57	7.88	3.87	2.19	22.79	14.86	29.42	0.23	2.99	70.5	88.3	13.9	6.1	5.3	5.4	4.1
Conc 1+2+3	10		225.35	11.28	2.94	1.58	22.38	15.72	31.04	0.24	3.14	76.7	91.2	19.5	9.3	7.9	7.8	6.1
Conc 1+2+3+4	30		349.45	17.49	2.02	1.04	20.99	16.59	33.30	0.25	3.67	81.6	93.1	28.4	15.1	13.2	12.8	11.1

Table C14: Test 14 Average Results

REAGENT DOSAGE																		
Reagent					**Dosage (g/t)**							**Conditioning Time (min)**						
Depressant					100							2						
Collector					0							Added to Mill						
TESTWORK RESULTS																		
Product	**Time**	**Solids Conc**	**Solids Mass**		**Grade**							**Recovery**						
					Ni	**Cu**	**Fe**	**MgO**	**SiO_2**	**Cr_2O_3**	**Al_2O_3**	**Ni**	**Cu**	**Fe**	**MgO**	**SiO_2**	**Cr_2O_3**	**Al_2O_3**
	min	**%**	**g**	**Rel %**	**%**							**%**						
Conc 1	1	19	52.69	2.64	3.95	6.20	19.52	15.03	30.18	0.22	2.82	25.7	82.4	4.4	2.2	1.9	2.0	1.3
Conc 2	2	11	41.40	2.07	3.90	0.58	15.90	19.67	38.57	0.15	3.08	19.9	6.0	2.8	2.3	1.9	1.1	1.2
Conc 3	7	8	75.17	3.76	1.52	0.18	13.99	20.76	41.99	0.20	4.13	14.1	3.4	4.5	4.4	3.8	2.6	2.8
Conc 4	20	6	150.62	7.54	0.51	0.06	13.08	19.64	42.02	0.25	5.46	9.4	2.2	8.4	8.4	7.6	6.5	7.4
Conc Total		8	**319.88**	**16.01**	**1.75**	**1.17**	**14.72**	**19.15**	**39.62**	**0.22**	**4.40**	**69.0**	**94.0**	**20.1**	**17.3**	**15.2**	**12.3**	**12.7**
Tails			1596.91	79.91	0.16	0.01	11.75	18.32	44.43	0.32	6.05	31.0	6.0	79.9	82.7	84.8	87.7	87.3

Head (calc.)			1916.79	95.91	0.424	0.21	12.25	18.46	43.63	0.30	5.77	100.0	100.0	100.0	100.0	100.0	100.0	100.0
Head (meas.)					0.386	0.19	11.79	18.77	42.68	0.39	5.50							
Combined Concentrates																		
Conc 1	1		52.69	2.64	3.95	6.20	19.52	15.03	30.18	0.22	2.82	25.7	82.4	4.4	2.2	1.9	2.0	1.3
Conc 1+2	3		94.09	4.71	3.93	3.73	17.92	17.07	33.87	0.19	2.94	45.5	88.4	7.2	4.5	3.8	3.1	2.5
Conc 1+2+3	10		169.26	8.47	2.86	2.15	16.18	18.71	37.48	0.20	3.47	59.6	91.8	11.7	8.9	7.6	5.7	5.3
Conc 1+2+3+4	30		319.88	16.01	1.75	1.17	14.72	19.15	39.62	0.22	4.40	69.0	94.0	20.1	17.3	15.2	12.3	12.7

2. Collector and Depressant Interaction Testwork Standard Deviations of Results

Table C15: Test 1 Standard Deviation

STANDARD DEVIATIONS								
Product	Solids Mass	Grade						
		Ni	Cu	Fe	MgO	SiO_2	Cr_2O_3	Al_2O_3
	Rel %	%						
Conc 1	0.30	0.07	0.15	0.48	0.36	0.55	0.00	0.17
Conc 2	0.20	0.02	0.05	0.54	0.16	0.28	0.01	0.25
Conc 3	0.26	0.37	0.14	0.66	0.32	0.13	0.02	0.13
Conc 4	0.32	0.05	0.01	0.60	1.35	2.30	0.12	0.43
Tails	0.43	0.00	0.00	1.40	0.46	0.10	0.01	0.04

Table C16: Test 2 Standard Deviation

STANDARD DEVIATIONS								
Product	Solids Mass	Grade						
		Ni	Cu	Fe	MgO	SiO_2	Cr_2O_3	Al_2O_3
	Rel %	%						
Conc 1	1.47	0.22	1.54	2.35	1.36	3.53	0.00	0.37
Conc 2	0.42	1.64	0.36	1.35	1.25	3.08	0.00	0.75
Conc 3	0.10	0.38	0.04	0.37	0.12	0.07	0.01	0.56

Conc 4	0.44	0.04	0.00	0.06	0.37	0.15	0.01	0.20
Tails	1.55	0.00	0.00	0.27	0.08	0.16	0.01	0.02

Table C17: Test 3 Standard Deviation

STANDARD DEVIATIONS								
Product	**Solids Mass**	**Grade**						
		Ni	**Cu**	**Fe**	**MgO**	**SiO_2**	**Cr_2O_3**	**Al_2O_3**
	Rel %	**%**						
Conc 1	0.19	0.76	0.29	1.40	0.59	1.74	0.00	0.06
Conc 2	0.30	0.49	0.09	1.42	1.05	2.09	0.01	0.03
Conc 3	0.18	0.04	0.01	1.44	0.31	1.57	0.01	0.04
Conc 4	0.02	0.01	0.01	0.02	0.28	0.45	0.01	0.04
Tails	0.33	0.00	0.00	0.24	0.21	1.00	0.04	0.18

Table C18: Test 4 Standard Deviation

STANDARD DEVIATIONS								
Product	**Solids Mass**	**Grade**						
		Ni	**Cu**	**Fe**	**MgO**	**SiO_2**	**Cr_2O_3**	**Al_2O_3**
	Rel %	**%**						
Conc 1	0.03	0.95	5.07	2.33	3.61	6.61	0.02	0.14

Conc 2	0.32	1.30	5.76	5.45	4.67	9.74	0.06	1.01
Conc 3	0.40	0.04	0.02	0.20	0.38	0.28	0.01	0.06
Conc 4	0.39	0.02	0.00	0.39	0.37	0.44	0.00	0.06
Tails	5.63	0.00	0.00	0.13	0.31	0.23	0.09	0.07

Table C19: Test 5 Standard Deviation

STANDARD DEVIATIONS								
Product	**Solids Mass**	**Grade**						
		Ni	**Cu**	**Fe**	**MgO**	**SiO_2**	**Cr_2O_3**	**Al_2O_3**
	Rel %	**%**						
Conc 1	0.10	0.55	1.14	1.03	0.74	1.69	0.01	0.16
Conc 2	0.29	0.32	0.37	0.56	0.98	2.11	0.09	0.31
Conc 3	0.75	0.00	0.01	0.02	0.41	0.13	0.11	0.02
Conc 4	2.00	0.08	0.01	1.53	0.79	1.20	0.06	0.38
Tails	4.36	0.004	0.004	0.040	0.175	0.478	0.049	0.076

Table C20: Test 6 Standard Deviation

STANDARD DEVIATIONS								
Product	Solids Mass	Grade						
		Ni	Cu	Fe	MgO	SiO_2	Cr_2O_3	Al_2O_3
	Rel %	%						
Conc 1	0.21	0.03	0.01	0.20	0.62	0.96	0.01	0.06
Conc 2	0.16	0.14	0.04	0.98	0.54	1.30	0.02	0.06
Conc 3	1.77	0.18	0.04	1.74	0.17	1.68	0.00	0.37
Conc 4	1.11	0.04	0.01	0.73	0.70	0.47	0.02	0.08
Tails	0.94	0.00	0.00	0.12	0.37	0.12	0.00	0.01

Table C21: Test 7 Standard Deviation

STANDARD DEVIATIONS								
Product	Solids Mass	Grade						
		Ni	Cu	Fe	MgO	SiO_2	Cr_2O_3	Al_2O_3
	Rel %	%						
Conc 1	0.43	0.45	1.27	1.18	1.19	2.44	0.01	0.15
Conc 2	0.27	0.71	1.07	1.72	1.50	3.26	0.03	0.37
Conc 3	0.52	0.15	0.04	0.69	0.27	0.70	0.03	0.18
Conc 4	0.71	0.04	0.01	0.61	0.64	0.78	0.04	0.19
Tails	2.34	0.00	0.00	0.40	0.24	0.33	0.04	0.06

Table C22: Test 8 Standard Deviation

STANDARD DEVIATIONS								
Product	Solids Mass	Grade						
		Ni	Cu	Fe	MgO	SiO_2	Cr_2O_3	Al_2O_3
	Rel %	%						
Conc 1	0.49	0.53	1.43	1.65	1.29	2.64	0.01	0.17
Conc 2	0.34	0.78	0.85	1.75	1.24	3.07	0.03	0.39
Conc 3	0.64	0.21	0.03	0.87	0.28	0.77	0.02	0.23
Conc 4	0.63	0.04	0.01	0.62	0.74	0.95	0.03	0.25
Tails	2.87	0.00	0.00	0.34	0.22	0.44	0.02	0.05

Table C23: Test 9 Standard Deviation

STANDARD DEVIATIONS								
Product	Solids Mass	Grade						
		Ni	Cu	Fe	MgO	SiO_2	Cr_2O_3	Al_2O_3
	Rel %	%						
Conc 1	0.42	0.51	1.24	1.16	1.21	2.36	0.01	0.16
Conc 2	0.20	0.57	1.43	1.94	1.46	3.47	0.04	0.49
Conc 3	0.45	0.19	0.05	0.65	0.31	0.71	0.03	0.23
Conc 4	0.67	0.05	0.01	0.53	0.48	0.60	0.05	0.19
Tails	2.27	0.00	0.00	0.34	0.26	0.37	0.03	0.07

Table C24: Test 10 Standard Deviation

STANDARD DEVIATIONS								
Product	Solids Mass	Grade						
		Ni	Cu	Fe	MgO	SiO_2	Cr_2O_3	Al_2O_3
	Rel %	%						
Conc 1	0.43	0.53	1.58	1.47	1.48	3.25	0.01	0.17
Conc 2	0.31	0.70	1.23	1.75	1.79	2.48	0.02	0.36
Conc 3	0.67	0.18	0.05	0.81	0.36	0.81	0.03	0.15
Conc 4	0.74	0.05	0.01	0.50	0.78	1.06	0.03	0.14
Tails	2.58	0.00	0.00	0.28	0.35	0.43	0.03	0.09

Table C25: Test 11 Standard Deviation

STANDARD DEVIATIONS								
Product	Solids Mass	Grade						
		Ni	Cu	Fe	MgO	SiO_2	Cr_2O_3	Al_2O_3
	Rel %	%						
Conc 1	0.38	0.35	1.12	1.38	1.44	2.84	0.01	0.16
Conc 2	0.22	0.70	0.90	1.91	1.79	3.57	0.03	0.48
Conc 3	0.51	0.13	0.04	0.90	0.36	0.80	0.02	0.16

Conc 4	0.73	0.05	0.01	0.48	0.68	0.84	0.04	0.21
Tails	2.85	0.00	0.00	0.28	0.21	0.39	0.03	0.08

Table C26: Test 12 Standard Deviation

STANDARD DEVIATIONS								
Product	**Solids Mass**	**Grade**						
		Ni	**Cu**	**Fe**	**MgO**	**SiO_2**	**Cr_2O_3**	**Al_2O_3**
	Rel %	**%**						
Conc 1	0.47	0.52	1.52	1.60	1.02	2.97	0.01	0.14
Conc 2	0.29	0.60	0.79	1.85	1.63	3.01	0.03	0.39
Conc 3	0.47	0.15	0.03	0.78	0.23	0.83	0.02	0.25
Conc 4	0.63	0.05	0.01	0.62	0.80	1.06	0.04	0.23
Tails	2.65	0.00	0.00	0.44	0.30	0.41	0.04	0.05

Table C27: Test 13 Standard Deviation

STANDARD DEVIATIONS								
Product	**Solids Mass**	**Grade**						
		Ni	**Cu**	**Fe**	**MgO**	**SiO_2**	**Cr_2O_3**	**Al_2O_3**
	Rel %	**%**						
Conc 1	0.36	0.41	1.13	1.53	0.92	2.69	0.01	0.13

Conc 2	0.31	0.66	0.89	1.70	1.57	2.79	0.03	0.30
Conc 3	0.57	0.19	0.03	0.81	0.24	0.65	0.03	0.25
Conc 4	0.77	0.03	0.01	0.54	0.67	0.96	0.04	0.24
Tails	1.61	0.00	0.00	0.44	0.27	0.43	0.03	0.06

Table C28: Test 14 Standard Deviation

STANDARD DEVIATIONS								
Product	**Solids Mass**	**Grade**						
		Ni	**Cu**	**Fe**	**MgO**	**SiO_2**	**Cr_2O_3**	**Al_2O_3**
	Rel %	**%**						
Conc 1	0.32	0.49	1.35	1.36	1.53	2.84	0.01	0.15
Conc 2	0.31	0.52	1.32	1.87	1.77	3.94	0.03	0.48
Conc 3	0.70	0.12	0.05	0.80	0.24	0.45	0.02	0.14
Conc 4	0.81	0.05	0.01	0.51	0.47	1.00	0.03	0.23
Tails	1.88	0.00	0.00	0.43	0.27	0.45	0.04	0.06

Appendix D – Chapter 5 Appendices

Testwork was conducted in duplicate. Average results are provided in the first section of Appendix D. Standard deviations of the testwork are presented in the second section of Appendix D. The assay by size data is presented in the third section of Appendix D.

1. Stage Dosing Flotation Testwork Conditions and Average Results

Table D1: Test 1 Average Results

REAGENT DOSAGE		
Reagent	**Dosage (g/t) 1min/3min/10min/30min**	**Type of Dosage**
Depressant	50/0/0/0	Upfront
Collector	140/0/0/0	Upfront

TESTWORK RESULTS

Product	Time		Solids Mass		Grade							Recovery						
					Ni	Cu	Fe	MgO	SiO_2	Cr_2O_3	Al_2O_3	Ni	Cu	Fe	MgO	SiO_2	Cr_2O_3	Al_2O_3
	min		g	Rel %	%							%						
Conc 1	1		49.65	2.51	7.37	4.40	25.11	11.74	21.65	0.12	1.36	38.9	69.3	5.2	1.4	1.3	1.0	0.7
Conc 2	2		69.85	3.53	3.86	0.74	21.61	17.36	32.06	0.18	2.03	28.7	16.4	6.3	3.0	2.8	2.2	1.5
Conc 3	7		93.28	4.71	1.18	0.18	22.76	17.94	33.11	0.26	2.70	11.7	5.2	8.9	4.1	3.8	4.2	2.6
Conc 4	20		100.30	5.06	0.55	0.07	22.72	17.53	32.47	0.29	3.47	5.9	2.3	9.5	4.3	4.0	5.1	3.6
Conc Total			**313.065**	**15.81**	**2.56**	**0.94**	**22.86**	**16.70**	**30.86**	**0.23**	**2.59**	**85.1**	**93.2**	**29.9**	**12.9**	**11.9**	**12.5**	**8.3**
Tails			1667.30	84.19	0.08	0.01	10.08	21.16	43.02	0.30	5.39	14.9	6.8	70.1	87.1	88.1	87.5	91.7
Head (calc.)			49.65	100.00	0.47	0.16	12.10	20.45	41.10	0.29	4.94	100.0	100.0	100.0	100.0	100.0	100.0	100.0
Head (meas.)					0.46	0.17	12.47	20.90	40.92	0.28	4.87							
Combined Concentrates																		
Conc 1	1		49.645	2.51	7.37	4.40	25.11	11.74	21.65	0.12	1.36	38.9	69.3	5.2	1.4	1.3	1.0	0.7
Conc 1+2	3		119.49	6.03	5.32	2.26	23.06	15.03	27.73	0.16	1.75	67.6	85.7	11.5	4.4	4.1	3.2	2.1
Conc 1+2+3	10		212.765	10.74	3.50	1.35	22.93	16.30	30.09	0.20	2.17	79.2	90.9	20.4	8.6	7.9	7.4	4.7

Conc 1+2+3+4	30		313.065	15.81	2.56	0.94	22.86	16.70	30.86	0.23	2.59	85.1	93.2	29.9	12.9	11.9	12.5	8.3

Table D2: Test 2 Average Results

REAGENT DOSAGE																		
Reagent				**Dosage (g/t) 1min/3min/10min/30min**							**Type of Dosage**							
Depressant				50/0/0/0							Upfront							
Collector				70/35/20/15							Stage							
TESTWORK RESULTS																		
Product	**Time**		**Solids Mass**		**Grade**							**Recovery**						
					Ni	**Cu**	**Fe**	**MgO**	**SiO_2**	**Cr_2O_3**	**Al_2O_3**	**Ni**	**Cu**	**Fe**	**MgO**	**SiO_2**	**Cr_2O_3**	**Al_2O_3**
	min		**g**	**Rel %**	**%**							**%**						
Conc 1	1		58.55	2.95	7.12	4.67	21.95	13.40	24.79	0.14	1.56	43.1	73.4	5.2	2.0	1.8	1.2	0.9
Conc 2	2		78.07	3.93	3.09	0.69	23.91	16.07	29.79	0.24	2.17	24.9	14.5	7.6	3.2	2.9	2.7	1.8
Conc 3	7		109.09	5.50	1.01	0.13	31.10	13.37	24.65	0.35	2.07	11.4	3.7	13.7	3.7	3.3	5.4	2.3
Conc 4	20		114.06	5.75	0.63	0.07	29.54	13.57	24.94	0.36	2.68	7.4	2.2	13.6	3.9	3.5	5.7	3.2
Conc Total			**359.76**	**18.12**	**2.33**	**0.97**	**27.56**	**14.02**	**25.88**	**0.30**	**2.20**	**86.8**	**93.9**	**40.1**	**12.7**	**11.6**	**14.9**	**8.2**
Tails			1625.19	81.88	0.08	0.01	9.10	21.26	43.72	0.37	5.43	13.2	6.1	59.9	87.3	88.4	85.1	91.8
Head (calc.)			1984.95	100.00	0.49	0.19	12.45	19.95	40.49	0.36	4.85	100.0	100.0	100.0	100.0	100.0	100.0	100.0
Head (meas.)					0.48	0.20	12.82	20.40	41.22	0.29	4.88							
Combined Concentrates																		
Conc 1	1		58.545	2.95	7.12	4.67	21.95	13.40	24.79	0.14	1.56	43.1	73.4	5.2	2.0	1.8	1.2	0.9

Conc 1+2	3		136.61	6.88	4.82	2.40	23.07	14.92	27.65	0.20	1.91	68.0	88.0	12.8	5.1	4.7	3.8	2.7
Conc 1+2+3	10		245.7	12.38	3.13	1.39	26.64	14.23	26.31	0.27	1.98	79.5	91.6	26.5	8.8	8.0	9.2	5.1
Conc 1+2+3+4	30		359.76	18.12	2.33	0.97	27.56	14.02	25.88	0.30	2.20	86.8	93.9	40.1	12.7	11.6	14.9	8.2

Table D3: Test 3 Average Results

REAGENT DOSAGE		
Reagent	**Dosage (g/t)**	**Conditioning Time (min)**
Depressant	15/15/10/10	Stage
Collector	140/0/0/0	Upfront

TESTWORK RESULTS

Product	**Time**	**Solids Conc**	**Solids Mass**		**Grade**							**Recovery**						
					Ni	**Cu**	**Fe**	**MgO**	**SiO_2**	**Cr_2O_3**	**Al_2O_3**	**Ni**	**Cu**	**Fe**	**MgO**	**SiO_2**	**Cr_2O_3**	**Al_2O_3**
	min	**%**	**g**	**Rel %**	**%**							**%**						
Conc 1	1		69.51	3.48	6.54	3.81	21.61	12.76	24.49	0.11	1.37	48.0	75.7	6.8	2.3	2.2	1.3	1.0
Conc 2	2		59.81	3.00	3.39	0.59	20.86	16.04	30.08	0.21	2.01	21.4	10.1	5.6	2.5	2.3	2.1	1.3
Conc 3	7		89.32	4.48	1.09	0.19	22.89	16.29	30.19	0.28	2.51	10.3	4.8	9.3	3.8	3.5	4.2	2.5
Conc 4	20		145.05	7.27	0.47	0.07	20.11	16.79	30.95	0.28	3.36	7.1	3.1	13.2	6.3	5.8	6.7	5.3
Conc Total			**363.68**	**18.22**	**2.26**	**0.90**	**21.20**	**15.78**	**29.38**	**0.24**	**2.55**	**86.9**	**93.7**	**34.9**	**14.9**	**13.8**	**14.3**	**10.2**
Tails			1631.86	81.78	0.08	0.01	8.81	20.16	40.82	0.32	5.02	13.1	6.3	65.1	85.1	86.2	85.7	89.8
Head (calc.)			1995.535	100.00	0.47	0.18	11.07	19.36	38.73	0.30	4.57	100.0	100.0	100.0	100.0	100.0	100.0	100.0
Head (meas.)					0.48	0.20	12.64	20.76	42.13	0.31	4.98							

Combined Concentrates																		
Conc 1	1		69.51	3.48	6.54	3.81	21.61	12.76	24.49	0.11	1.37	48.0	75.7	6.8	2.3	2.2	1.3	1.0
Conc 1+2	3		129.315	6.48	5.09	2.32	21.26	14.28	27.07	0.16	1.66	69.4	85.8	12.4	4.8	4.5	3.4	2.4
Conc 1+2+3	10		218.63	10.96	3.45	1.45	21.93	15.10	28.34	0.21	2.01	79.7	90.6	21.7	8.5	8.0	7.6	4.8
Conc 1+2+3+4	30		363.68	18.22	2.26	0.90	21.20	15.78	29.38	0.24	2.55	86.9	93.7	34.9	14.9	13.8	14.3	10.2

Table D4: Test 4 Average Results

REAGENT DOSAGE		
Reagent	**Dosage (g/t)**	**Conditioning Time (min)**
Depressant	15/15/10/10	Stage (High-Low)
Collector	70/35/20/15	Stage (High-Low)

TESTWORK RESULTS

Product	**Time**	**Solids Conc**	**Solids Mass**		**Grade**							**Recovery**						
					Ni	**Cu**	**Fe**	**MgO**	**SiO_2**	**Cr_2O_3**	**Al_2O_3**	**Ni**	**Cu**	**Fe**	**MgO**	**SiO_2**	**Cr_2O_3**	**Al_2O_3**
	min	**%**	**g**	**Rel %**	**%**							**%**						
Conc 1	1		64.73	3.24	6.66	4.33	21.83	12.71	24.12	0.09	1.26	41.8	75.0	5.8	2.0	1.9	0.9	0.8
Conc 2	2		70.59	3.53	3.50	0.64	22.78	15.33	28.89	0.19	1.90	24.0	12.1	6.6	2.7	2.5	2.1	1.4
Conc 3	7		123.24	6.17	1.27	0.16	30.50	13.53	24.74	0.29	2.12	15.2	5.1	15.5	4.1	3.7	5.7	2.7
Conc 4	20		124.87	6.25	0.58	0.07	26.35	15.10	28.13	0.27	3.05	7.0	2.2	13.5	4.7	4.3	5.3	3.9
Conc Total			**383.42**	**19.19**	**2.37**	**0.92**	**26.27**	**14.23**	**26.51**	**0.23**	**2.24**	**87.9**	**94.5**	**41.5**	**13.6**	**12.5**	**14.1**	**8.8**
Tails			1614.21	80.81	0.08	0.01	8.81	21.57	44.17	0.34	5.49	12.1	5.5	58.5	86.4	87.5	85.9	91.2

Head (calc.)			1997.63	100.00	0.52	0.19	12.16	20.16	40.78	0.32	4.87	100.0	100.0	100.0	100.0	100.0	100.0	100.0
Head (meas.)					0.49	0.20	12.89	20.86	40.99	0.25	4.79							
Combined Concentrates																		
Conc 1	1		64.725	3.24	6.66	4.33	21.83	12.71	24.12	0.09	1.26	41.8	75.0	5.8	2.0	1.9	0.9	0.8
Conc 1+2	3		135.315	6.77	5.01	2.40	22.33	14.08	26.61	0.14	1.60	65.7	87.1	12.4	4.7	4.4	3.1	2.2
Conc 1+2+3	10		258.555	12.94	3.23	1.33	26.22	13.82	25.72	0.22	1.85	80.9	92.3	27.9	8.9	8.2	8.8	4.9
Conc 1+2+3+4	30		383.42	19.19	2.37	0.92	26.27	14.23	26.51	0.23	2.24	87.9	94.5	41.5	13.6	12.5	14.1	8.8

Table D5: Test 5 Average Results

REAGENT DOSAGE		
Reagent	**Dosage (g/t)**	**Conditioning Time (min)**
Depressant	10/10/15/15	Stage (Low-High)
Collector	40/40/30/30	Stage (Low-High)

TESTWORK RESULTS																		
Product	**Time**	**Solids Conc**	**Solids Mass**		**Grade**							**Recovery**						
					Ni	**Cu**	**Fe**	**MgO**	**SiO_2**	**Cr_2O_3**	**Al_2O_3**	**Ni**	**Cu**	**Fe**	**MgO**	**SiO_2**	**Cr_2O_3**	**Al_2O_3**
	min	**%**	**g**	**Rel %**	**%**							**%**						
Conc 1	1		45.31	2.30	6.67	5.38	22.90	13.22	23.20	0.06	1.07	31.1	68.9	4.3	1.4	1.3	0.4	0.5
Conc 2	2		52.82	2.68	4.57	1.08	22.27	17.06	30.70	0.16	1.91	24.9	16.1	4.9	2.2	2.0	1.4	1.1
Conc 3	7		105.64	5.35	2.08	0.19	31.16	13.03	22.72	0.26	1.76	22.6	5.8	13.8	3.3	3.0	4.5	1.9
Conc 4	20		120.13	6.09	0.83	0.07	33.37	12.82	22.62	0.29	2.32	10.3	2.4	16.8	3.7	3.3	5.6	2.9

Conc Total			**323.895**	**16.42**	**2.67**	**1.02**	**29.37**	**13.63**	**24.05**	**0.23**	**1.89**	**88.9**	**93.2**	**39.8**	**10.7**	**9.6**	**11.9**	**6.4**
Tails			1648.78	83.58	0.07	0.01	8.74	22.39	44.48	0.33	5.43	11.1	6.8	60.2	89.3	90.4	88.1	93.6
Head (calc.)			1972.67	100.00	0.49	0.18	12.13	20.95	41.13	0.31	4.85	100.0	100.0	100.0	100.0	100.0	100.0	100.0
Head (meas.)					0.48	0.20	12.82	20.40	41.22	0.29	4.88							
Combined Concentrates																		
Conc 1	1		45.31	2.30	6.67	5.38	22.90	13.22	23.20	0.06	1.07	31.1	68.9	4.3	1.4	1.3	0.4	0.5
Conc 1+2	3		98.13	4.97	5.54	3.07	22.56	15.28	27.24	0.11	1.52	56.0	85.0	9.3	3.6	3.3	1.8	1.6
Conc 1+2+3	10		203.765	10.33	3.75	1.58	27.02	14.11	24.89	0.19	1.64	78.6	90.8	23.0	7.0	6.3	6.3	3.5
Conc 1+2+3+4	30		323.895	16.42	2.67	1.02	29.37	13.63	24.05	0.23	1.89	88.9	93.2	39.8	10.7	9.6	11.9	6.4

Table D6: Test 6 Average Results

REAGENT DOSAGE		
Reagent	**Dosage (g/t)**	**Conditioning Time (min)**
Depressant	50/0/0/0	Upfront
Collector	30/0/0/0	Upfront (Low Dosage)

TESTWORK RESULTS																		
Product	**Time**	**Solids Conc**	**Solids Mass**		**Grade**							**Recovery**						
					Ni	**Cu**	**Fe**	**MgO**	**SiO_2**	**Cr_2O_3**	**Al_2O_3**	**Ni**	**Cu**	**Fe**	**MgO**	**SiO_2**	**Cr_2O_3**	**Al_2O_3**
	min	**%**	**g**	**Rel %**	**%**							**%**						
Conc 1	1		41.08	2.10	7.70	6.16	25.79	10.74	19.42	0.10	1.18	33.4	69.7	4.3	1.1	1.0	0.7	0.5
Conc 2	2		46.78	2.39	5.64	1.22	19.88	16.98	31.38	0.18	2.09	27.8	15.8	3.7	2.0	1.8	1.3	1.0

Conc 3	7		82.61	4.22	1.82	0.23	19.43	19.34	35.52	0.26	2.84	15.9	5.2	6.5	4.1	3.7	3.4	2.5
Conc 4	20		95.77	4.89	0.60	0.08	19.99	18.74	35.24	0.33	3.63	6.0	2.2	7.7	4.6	4.2	4.9	3.7
Conc Total			**266.235**	**13.58**	**2.96**	**1.27**	**20.69**	**17.38**	**32.21**	**0.25**	**2.74**	**83.1**	**92.9**	**22.2**	**11.8**	**10.7**	**10.2**	**7.8**
Tails			1693.61	86.42	0.09	0.02	11.41	20.48	42.10	0.34	5.08	16.9	7.1	77.8	88.2	89.3	89.8	92.2
Head (calc.)			1959.84	100.00	0.48	0.19	12.67	20.06	40.75	0.33	4.76	100.0	100.0	100.0	100.0	100.0	100.0	100.0
Head (meas.)					0.49	0.20	12.89	20.86	40.99	0.25	4.79							
Combined Concentrates																		
Conc 1	1		41.08	2.10	7.70	6.16	25.79	10.74	19.42	0.10	1.18	33.4	69.7	4.3	1.1	1.0	0.7	0.5
Conc 1+2	3		87.855	4.48	6.61	3.53	22.64	14.07	25.79	0.14	1.66	61.2	85.5	8.0	3.1	2.8	2.0	1.6
Conc 1+2+3	10		170.465	8.70	4.29	1.93	21.09	16.62	30.51	0.20	2.23	77.1	90.7	14.5	7.2	6.5	5.4	4.1
Conc 1+2+3+4	30		266.235	13.58	2.96	1.27	20.69	17.38	32.21	0.25	2.74	83.1	92.9	22.2	11.8	10.7	10.2	7.8

Table D7: Test 7 Average Results

REAGENT DOSAGE		
Reagent	**Dosage (g/t)**	**Conditioning Time (min)**
Depressant	50/0/0/0	Upfront
Collector	15/8/4/3	Upfront (Low Dosage)

TESTWORK RESULTS																		
Product	**Time**	**Solids Conc**	**Solids Mass**		**Grade**							**Recovery**						
					Ni	**Cu**	**Fe**	**MgO**	**SiO_2**	**Cr_2O_3**	**Al_2O_3**	**Ni**	**Cu**	**Fe**	**MgO**	**SiO_2**	**Cr_2O_3**	**Al_2O_3**
	min	**%**	**g**	**Rel %**	**%**							**%**						

Conc 1	1		47.49	2.39	7.15	6.51	20.84	14.46	27.16	0.08	1.51	32.6	77.0	3.9	1.7	1.5	0.6	0.7
Conc 2	2		49.61	2.50	6.45	0.77	22.97	15.51	28.97	0.16	1.90	30.8	9.5	4.5	1.9	1.7	1.2	0.9
Conc 3	7		98.04	4.94	1.64	0.18	22.77	17.95	33.50	0.25	2.97	15.4	4.4	8.8	4.3	3.9	3.8	2.9
Conc 4	20		101.04	5.09	0.61	0.10	23.41	17.18	32.47	0.32	3.44	6.0	2.5	9.4	4.2	3.9	4.9	3.5
Conc Total			**296.17**	**14.93**	**2.98**	**1.27**	**22.71**	**16.72**	**31.38**	**0.23**	**2.72**	84.8	93.5	26.7	12.1	11.0	10.5	8.1
Tails			1687.12	85.07	0.09	0.02	10.97	21.42	44.40	0.34	5.42	15.2	6.5	73.3	87.9	89.0	89.5	91.9
Head (calc.)			1983.29	100.00	0.52	0.20	12.72	20.72	42.46	0.33	5.02	100.0	100.0	100.0	100.0	100.0	100.0	100.0
Head (meas.)					0.49	0.20	12.89	20.86	40.99	0.25	4.79							
Combined Concentrates																		
Conc 1	1		47.485	2.39	7.15	6.51	20.84	14.46	27.16	0.08	1.51	32.6	77.0	3.9	1.7	1.5	0.6	0.7
Conc 1+2	3		97.095	4.90	6.80	3.57	21.93	15.00	28.09	0.12	1.71	63.4	86.5	8.4	3.5	3.2	1.8	1.7
Conc 1+2+3	10		195.13	9.84	4.20	1.87	22.35	16.48	30.81	0.19	2.34	78.8	90.9	17.3	7.8	7.1	5.6	4.6
Conc 1+2+3+4	30		296.17	14.93	2.98	1.27	22.71	16.72	31.38	0.23	2.72	84.8	93.5	26.7	12.1	11.0	10.5	8.1

2. Stage Dosing Testwork Standard Deviations of Results

Table D8: Test 1 Standard Deviation

STANDARD DEVIATIONS								
Product	Solids Mass	Grade						
		Ni	Cu	Fe	MgO	SiO_2	Cr_2O_3	Al_2O_3
	Rel %	%						
Conc 1	0.02	0.97	0.08	1.38	1.12	2.12	0.00	0.09
Conc 2	0.41	0.80	0.14	1.04	1.04	2.13	0.05	0.29
Conc 3	0.29	0.29	0.00	0.47	1.42	2.10	0.02	0.20
Conc 4	0.44	0.01	0.01	0.35	0.91	1.15	0.01	0.14
Tails	0.02	0.01	0.00	0.10	0.07	0.14	0.04	0.00

Table D9: Test 2 Standard Deviation

STANDARD DEVIATIONS								
Product	Solids Mass	Grade						
		Ni	Cu	Fe	MgO	SiO_2	Cr_2O_3	Al_2O_3
	Rel %	%						
Conc 1	0.11	0.03	1.04	0.01	0.71	0.83	1.68	0.06
Conc 2	0.25	0.55	0.04	0.03	1.52	0.34	0.73	0.04
Conc 3	0.13	0.07	0.01	0.00	0.77	0.14	0.21	0.06

Conc 4	0.09	0.04	0.01	0.00	1.00	0.05	0.11	0.05
Tails	1.23	0.01	0.00	0.00	0.80	0.45	0.94	0.03

Table D10: Test 3 Standard Deviation

STANDARD DEVIATIONS								
Product	**Solids Mass**	**Grade**						
		Ni	**Cu**	**Fe**	**MgO**	**SiO_2**	**Cr_2O_3**	**Al_2O_3**
	Rel %	**%**						
Conc 1	0.05	0.40	0.61	2.23	1.16	2.51	0.02	0.34
Conc 2	0.09	0.42	0.03	3.28	1.01	2.65	0.02	0.19
Conc 3	0.06	0.11	0.01	3.09	1.03	1.70	0.01	0.29
Conc 4	0.15	0.08	0.02	2.44	1.52	3.06	0.00	0.42
Tails	2.00	0.00	0.00	0.54	0.57	1.48	0.01	0.30

Table D11: Test 4 Standard Deviation

STANDARD DEVIATIONS								
Product	**Solids Mass**	**Grade**						
		Ni	**Cu**	**Fe**	**MgO**	**SiO_2**	**Cr_2O_3**	**Al_2O_3**
	Rel %	**%**						
Conc 1	0.34	0.30	0.48	1.19	0.04	0.30	0.00	0.00

Conc 2	0.19	0.11	0.02	1.05	0.46	0.57	0.00	0.08
Conc 3	0.68	0.00	0.00	1.62	0.55	0.96	0.00	0.12
Conc 4	0.71	0.03	0.01	1.51	1.05	1.75	0.01	0.23
Tails	0.77	0.01	0.00	0.40	0.24	0.19	0.02	0.10

Table D12: Test 5 Standard Deviation

STANDARD DEVIATIONS								
Product	**Solids Mass**	**Grade**						
		Ni	**Cu**	**Fe**	**MgO**	**SiO_2**	**Cr_2O_3**	**Al_2O_3**
	Rel %	**%**						
Conc 1	0.11	0.62	0.09	1.33	0.40	0.75	0.00	0.09
Conc 2	0.07	0.35	0.17	1.01	0.79	1.39	0.02	0.26
Conc 3	0.11	0.28	0.00	0.37	0.16	0.07	0.00	0.14
Conc 4	0.13	0.03	0.00	1.03	0.26	0.44	0.00	0.11
Tails	1.06	0.00	0.00	0.31	0.07	0.16	0.02	0.02

Table D13: Test 6 Standard Deviation

STANDARD DEVIATIONS								
Product	Solids Mass	Grade						
		Ni	Cu	Fe	MgO	SiO_2	Cr_2O_3	Al_2O_3
	Rel %	%						
Conc 1	0.06	0.22	0.25	0.16	0.19	0.19	0.01	0.02
Conc 2	0.27	0.24	0.08	0.03	0.52	0.01	0.02	0.10
Conc 3	0.12	0.27	0.02	0.26	0.72	0.42	0.00	0.29
Conc 4	0.32	0.03	0.02	0.72	0.58	0.58	0.03	0.03
Tails	0.46	0.00	0.00	0.01	0.81	0.18	0.00	0.03

Table D14: Test 7 Standard Deviation

STANDARD DEVIATIONS								
Product	Solids Mass	Grade						
		Ni	Cu	Fe	MgO	SiO_2	Cr_2O_3	Al_2O_3
	Rel %	%						
Conc 1	0.02	0.97	0.08	1.38	1.12	2.12	0.00	0.09
Conc 2	0.41	0.80	0.14	1.04	1.04	2.13	0.05	0.29
Conc 3	0.29	0.29	0.00	0.47	1.42	2.10	0.02	0.20
Conc 4	0.44	0.01	0.01	0.35	0.91	1.15	0.01	0.14
Tails	0.02	0.01	0.00	0.10	0.07	0.14	0.04	0.00

3. Assay by Size Data

Table D15: Test 1 Assay by Size Data

Screen Size (µm)	Mass (g)	Mass (%)	Grade					Distribution				
			Ni	Fe	Cu	SiO_2	MgO	Ni	Fe	Cu	SiO_2	MgO
Concentrate 1												
-106+75	5.88	15.39	5.85	29.94	4.95	16.98	9.47	4.49	0.95	11.77	0.16	0.18
-75+53	4.3	11.26	9.44	29.57	5.27	14.85	8.17	5.29	0.69	9.15	0.10	0.11
-53+38	4.44	11.62	10.34	27.68	5.12	17.01	9.30	5.98	0.66	9.18	0.12	0.13
-38	23.58	61.73	7.52	22.83	4.12	25.34	13.47	23.13	2.90	39.20	0.94	1.02
Total	38.2	100	7.81	25.25	4.49	21.91	11.77	38.89	5.20	69.31	1.32	1.44
Concentrate 2												
-106+75	5.22	13.41	3.86	20.20	0.99	32.70	18.22	3.71	0.83	2.56	0.36	0.41
-75+53	3.18	8.17	5.51	20.95	0.93	31.45	17.29	3.23	0.53	1.48	0.21	0.23
-53+38	3.12	8.02	5.38	20.98	0.84	32.25	17.73	3.09	0.52	1.31	0.21	0.24
-38	27.4	70.40	3.70	20.42	0.81	33.45	18.11	18.66	4.42	11.04	1.96	2.12
Total	38.92	100	4.00	20.48	0.85	33.09	18.03	28.68	6.30	16.38	2.75	2.99
Concentrate 3												
-106+75	7.17	14.71	1.57	16.61	0.29	37.54	21.57	2.31	0.98	1.25	0.62	0.72
-75+53	3.91	8.02	1.46	19.65	0.20	35.21	19.76	1.18	0.63	0.47	0.32	0.36
-53+38	3.47	7.12	1.21	23.67	0.16	33.90	18.29	0.86	0.68	0.34	0.27	0.29
-38	34.19	70.15	1.04	23.34	0.15	33.12	17.36	7.31	6.57	3.18	2.59	2.76

Total	48.74	100	1.16	22.08	0.18	33.99	18.24	11.66	8.86	5.23	3.80	4.13
Concentrate 4												
-106+75	10.28	21.28	0.75	19.11	0.14	36.41	20.06	1.63	1.61	0.88	0.97	1.09
-75+53	8.63	17.87	0.61	27.46	0.07	29.14	15.54	1.12	1.95	0.37	0.65	0.71
-53+38	1.08	2.24	0.54	28.43	0.06	27.98	14.80	0.12	0.25	0.04	0.08	0.08
-38	28.31	58.61	0.50	24.50	0.06	31.52	16.41	3.01	5.70	1.01	2.31	2.46
Total	48.3	100	0.57	23.97	0.08	32.06	17.00	5.89	9.51	2.30	4.00	4.34
Tailings												
-106+75	28.42	35.70	0.07	10.25	0.01	44.79	21.42	4.74	24.49	2.37	32.28	31.42
-75+53	10.67	13.40	0.09	12.00	0.01	43.48	20.13	2.13	10.77	0.78	11.76	11.09
-53+38	4.93	6.19	0.08	12.16	0.01	42.78	20.10	0.94	5.04	0.34	5.35	5.11
-38	35.58	44.70	0.09	9.97	0.01	42.95	21.50	7.06	29.83	3.28	38.74	39.47
Total	79.6	100	0.08	10.48	0.01	43.67	21.20	14.88	70.14	6.77	88.13	87.09

Table D16: Test 2 Assay by Size Data

Screen Size (µm)	Mass (g)	Mass (%)	Grade					Distribution				
			Ni	Fe	Cu	SiO_2	MgO	Ni	Fe	Cu	SiO_2	MgO
Concentrate 1												
-106+75	7.86	13.17	5.78	25.27	4.53	20.36	11.34	4.52	0.80	10.84	0.19	0.22
-75+53	6.17	10.34	9.11	25.79	4.74	17.82	9.84	5.60	0.64	8.90	0.13	0.15
-53+38	6.46	10.82	10.30	26.32	4.65	20.09	11.27	6.62	0.68	9.13	0.16	0.18

-38	39.21	65.68	6.76	19.51	3.74	27.81	14.90	26.36	3.08	44.56	1.32	1.43
Total	59.7	100	7.25	21.65	4.04	24.96	13.51	43.10	5.20	73.42	1.81	1.98
Concentrate 2												
-106+75	6.77	11.32	2.65	20.91	0.86	30.60	17.01	2.82	0.79	2.16	0.34	0.38
-75+53	4.79	8.01	3.57	23.81	0.82	27.62	15.28	2.68	0.63	1.45	0.22	0.24
-53+38	5.62	9.40	3.15	25.55	0.68	26.14	14.37	2.78	0.80	1.41	0.24	0.27
-38	42.63	71.28	2.49	22.55	0.60	29.75	15.94	16.67	5.34	9.52	2.09	2.27
Total	59.81	100	2.66	22.74	0.66	29.33	15.86	24.95	7.55	14.54	2.89	3.17
Concentrate 3												
-106+75	20.42	23.27	1.16	27.76	0.15	26.35	14.68	3.13	2.81	1.09	0.83	0.96
-75+53	14.04	16.00	0.91	35.55	0.13	19.36	10.79	1.70	2.48	0.63	0.42	0.48
-53+38	2.98	3.40	0.93	37.28	0.08	20.40	10.97	0.37	0.55	0.09	0.09	0.10
-38	50.3	57.33	0.93	31.61	0.10	25.82	13.36	6.21	7.89	1.85	2.00	2.14
Total	87.74	100	0.98	31.54	0.12	24.73	13.18	11.41	13.73	3.66	3.35	3.68
Concentrate 4												
-106+75	29.62	30.90	0.73	32.16	0.07	23.01	12.52	2.72	4.61	0.82	0.94	1.07
-75+53	20.75	21.65	0.65	37.91	0.04	19.58	10.38	1.69	3.81	0.35	0.56	0.62
-53+38	5.26	5.49	0.56	31.87	0.08	25.83	13.62	0.37	0.81	0.17	0.19	0.21
-38	40.23	41.97	0.51	22.64	0.05	33.43	17.19	2.60	4.41	0.90	1.85	2.00
Total	95.86	100	0.61	29.39	0.06	26.80	14.08	7.38	13.64	2.24	3.54	3.91
Tailings												
-106+75	34.28	34.87	0.07	8.58	0.03	44.99	21.24	4.17	21.67	2.32	30.38	29.77

-75+53	13.42	13.65	0.08	9.29	0.02	46.71	21.18	1.70	9.19	0.59	12.35	11.63
-53+38	10.32	10.50	0.07	8.67	0.01	46.67	21.30	1.22	6.60	0.39	9.49	8.99
-38	40.3	40.99	0.09	7.55	0.03	45.58	22.37	6.07	22.42	2.84	36.19	36.87
Total	98.32	100	0.08	8.26	0.02	45.64	21.70	13.16	59.87	6.13	88.42	87.26

Table D17: Test 3 Assay by Size Data

Screen Size (µm)	Mass (g)	Mass (%)	Grade					Distribution				
			Ni	Fe	Cu	SiO_2	MgO	Ni	Fe	Cu	SiO_2	MgO
Concentrate 1												
-106+75	6.09	12.79	5.06	27.34	4.61	15.58	8.70	4.40	1.00	10.48	0.21	0.22
-75+53	5.7	11.97	8.85	28.57	5.18	13.70	7.54	7.20	0.98	11.02	0.17	0.18
-53+38	5.58	11.72	9.82	26.55	4.97	15.39	8.52	7.82	0.89	10.36	0.19	0.20
-38	30.23	63.51	6.62	21.50	3.88	24.75	13.20	28.59	3.92	43.85	1.64	1.69
Total	47.6	100	7.06	23.69	4.26	21.16	11.40	48.01	6.80	75.72	2.20	2.30
Concentrate 2												
-106+75	4.9	10.13	4.45	20.17	0.78	30.10	16.95	2.61	0.50	1.34	0.25	0.28
-75+53	4.01	8.29	5.63	22.53	0.69	27.91	15.21	2.71	0.46	0.96	0.19	0.20
-53+38	4.26	8.81	5.06	23.90	0.57	27.09	14.74	2.58	0.51	0.85	0.20	0.21
-38	35.18	72.76	3.21	23.49	0.57	28.36	15.15	13.52	4.18	6.93	1.69	1.79
Total	48.35	100	3.70	23.11	0.60	28.39	15.30	21.42	5.65	10.08	2.33	2.48
Concentrate 3												

-106+75	9.97	14.87	1.46	17.71	0.30	35.08	18.92	1.94	0.97	1.14	0.62	0.67
-75+53	5.61	8.37	1.41	23.97	0.24	30.25	16.52	1.05	0.74	0.51	0.30	0.33
-53+38	5.49	8.19	1.10	27.31	0.16	25.68	14.06	0.81	0.82	0.33	0.25	0.28
-38	45.97	68.57	1.06	26.68	0.16	28.12	15.16	6.50	6.73	2.79	2.31	2.49
Total	67.04	100	1.15	25.17	0.19	29.13	15.74	10.29	9.25	4.77	3.49	3.77
Concentrate 4												
-106+75	17.62	22.95	0.51	16.17	0.10	36.80	20.55	1.91	2.41	1.25	1.52	1.68
-75+53	12.8	16.67	0.54	25.74	0.06	26.84	14.89	1.46	2.78	0.54	0.81	0.88
-53+38	6.14	8.00	0.52	30.79	0.04	24.37	13.19	0.68	1.60	0.18	0.35	0.38
-38	40.23	52.39	0.36	18.87	0.04	33.04	18.08	3.10	6.42	1.10	3.12	3.37
Total	76.79	100	0.44	20.35	0.06	32.18	17.72	7.15	13.20	3.08	5.81	6.31
Tailings												
-106+75	37.64	37.98	0.06	9.42	0.01	43.02	20.80	4.31	23.89	2.63	33.24	32.83
-75+53	17.27	17.43	0.07	10.41	0.01	41.93	20.31	2.32	12.12	1.00	14.87	14.71
-53+38	11.3	11.40	0.07	10.12	0.01	41.95	20.46	1.39	7.71	0.60	9.73	9.70
-38	32.9	33.20	0.09	9.64	0.01	41.96	20.23	5.10	21.38	2.12	28.34	27.91
Total	99.11	100	0.07	9.74	0.01	42.36	20.49	13.13	65.10	6.35	86.17	85.15

Table D18: Test 4 Assay by Size Data

Screen Size (µm)	Mass (g)	Mass (%)	Grade					Distribution				
			Ni	Fe	Cu	SiO_2	MgO	Ni	Fe	Cu	SiO_2	MgO
Concentrate 1												
-106+75	5.48	12.93	5.27	29.19	5.61	16.41	9.21	4.08	0.96	11.65	0.17	0.19
-75+53	5.39	12.72	8.81	29.50	5.98	13.83	7.54	6.71	0.95	12.21	0.14	0.16
-53+38	5.18	12.22	9.93	27.42	5.69	16.20	8.60	7.27	0.85	11.18	0.16	0.17
-38	26.34	62.14	6.37	19.48	4.01	28.32	14.94	23.70	3.06	39.98	1.44	1.52
Total	42.39	100	6.97	22.98	4.67	23.46	12.49	41.76	5.82	75.02	1.92	2.04
Concentrate 2												
-106+75	4.77	10.19	3.78	20.61	0.91	30.41	16.65	2.53	0.60	1.79	0.27	0.29
-75+53	4.22	9.01	5.62	23.55	0.76	27.04	14.71	3.33	0.61	1.33	0.21	0.23
-53+38	4.04	8.63	5.16	25.16	0.60	25.75	14.15	2.93	0.62	1.00	0.19	0.21
-38	33.8	72.18	3.20	23.26	0.57	29.21	15.68	15.18	4.79	8.01	1.83	1.95
Total	46.83	100	3.65	23.18	0.63	13.50	15.56	23.96	6.62	12.12	2.50	2.69
Concentrate 3												
-106+75	13.37	14.71	1.84	22.63	0.26	31.75	17.27	3.34	1.79	1.39	0.72	0.81
-75+53	14.75	16.22	1.41	34.16	0.13	20.31	11.11	2.82	2.99	0.79	0.51	0.57
-53+38	6.46	7.11	1.33	37.78	0.11	18.62	10.28	1.16	1.45	0.28	0.20	0.23
-38	56.33	61.96	1.03	27.71	0.12	24.23	12.82	7.85	9.25	2.67	2.31	2.53
Total	90.91	100	1.23	28.72	0.14	24.30	13.01	15.18	15.48	5.13	3.74	4.14
Concentrate 4												

-106+75	21.24	21.92	0.87	34.21	0.10	20.03	11.28	2.42	4.05	0.67	0.66	0.77
-75+53	17.91	18.48	0.66	38.15	0.05	17.01	9.28	1.55	3.81	0.31	0.48	0.54
-53+38	3.05	3.15	0.54	31.32	0.05	22.48	12.38	0.22	0.53	0.05	0.11	0.12
-38	54.7	56.45	0.39	16.89	0.06	35.91	18.41	2.81	5.15	1.17	3.07	3.25
Total	96.9	100	0.55	25.07	0.07	28.51	14.97	7.00	13.55	2.21	4.31	4.68
Tailings												
-106+75	37.64	37.98	0.06	9.42	0.01	43.02	20.80	4.31	23.89	2.63	33.24	32.83
-75+53	17.27	17.43	0.07	10.41	0.01	41.93	20.31	2.32	12.12	1.00	14.87	14.71
-53+38	11.3	11.40	0.07	10.12	0.01	41.95	20.46	1.39	7.71	0.60	9.73	9.70
-38	32.9	33.20	0.09	9.64	0.01	41.96	20.23	5.10	21.38	2.12	28.34	27.91
Total	99.11	100	0.07	9.74	0.01	42.36	20.49	13.13	65.10	6.35	86.17	85.15

Table D19: Test 5 Assay by Size Data

Screen Size (µm)	Mass (g)	Mass (%)	Grade					Distribution				
			Ni	Fe	Cu	SiO_2	MgO	Ni	Fe	Cu	SiO_2	MgO
Concentrate 1												
-106+75	5.00	12.88	5.30	32.50	5.95	12.75	7.49	3.02	0.76	9.65	0.10	0.12
-75+53	4.20	10.82	8.92	31.89	6.48	10.92	6.48	4.26	0.63	8.83	0.07	0.08
-53+38	5.30	13.66	10.34	29.41	6.49	12.93	7.17	6.23	0.73	11.16	0.11	0.12
-38	24.31	62.64	6.37	19.46	4.98	27.11	14.91	17.61	2.22	39.29	1.02	1.13
Total	38.81	100	7.05	23.85	5.47	21.57	11.99	31.13	4.34	68.93	1.30	1.45

Concentrate 2												
-106+75	5.90	15.38	4.15	19.66	1.68	31.20	17.22	3.26	0.64	3.51	0.33	0.36
-75+53	5.10	13.29	6.20	21.44	1.39	29.68	16.20	4.21	0.60	2.52	0.27	0.29
-53+38	1.70	4.43	7.40	23.97	1.44	26.05	14.71	1.67	0.22	0.87	0.08	0.09
-38	25.67	66.90	4.61	24.32	1.01	28.84	15.83	15.74	3.45	9.20	1.32	1.44
Total	38.37	100	4.87	23.20	1.18	11.20	16.04	24.88	4.92	16.10	2.00	2.18
Concentrate 3												
-106+75	10.30	13.56	2.97	21.66	0.39	32.55	18.19	4.73	1.25	1.52	0.58	0.66
-75+53	12.00	15.80	2.23	30.33	0.20	21.77	12.12	4.15	2.04	0.92	0.45	0.51
-53+38	3.20	4.21	2.02	33.99	0.19	19.83	10.67	1.00	0.61	0.23	0.11	0.12
-38	50.46	66.43	1.63	34.84	0.16	21.00	11.45	12.75	9.85	3.10	1.82	2.04
Total	75.96	100	1.93	32.31	0.20	22.64	12.44	22.62	13.76	5.78	2.96	3.33
Concentrate 4												
-106+75	28.60	38.07	0.94	35.75	0.09	20.04	11.21	4.66	6.65	1.14	1.16	1.33
-75+53	16.50	21.96	0.71	39.02	0.06	18.14	9.85	2.04	4.19	0.38	0.61	0.67
-53+38	2.70	3.59	0.65	29.88	0.07	25.72	13.80	0.30	0.52	0.08	0.14	0.15
-38	27.33	36.38	0.70	30.32	0.07	26.08	13.90	3.31	5.39	0.84	1.44	1.57
Total	75.13	100	0.79	34.28	0.08	22.02	11.98	10.32	16.75	2.44	3.35	3.73
Tailings												
-106+75	42.40	43.54	0.06	9.43	0.01	45.30	22.07	4.19	27.23	2.82	39.59	39.23
-75+53	16.50	16.94	0.06	9.69	0.01	45.26	21.25	1.67	10.88	0.99	15.39	14.70
-53+38	9.70	9.96	0.08	9.31	0.02	44.93	21.19	1.30	6.15	0.73	8.98	8.62

-38	28.78	29.55	0.08	8.15	0.02	44.55	22.18	3.90	15.98	2.21	26.43	26.76
Total	97.38	100.00	0.06	9.08	0.01	45.03	21.87	11.06	60.24	6.76	90.40	89.32