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Investigating Ways of Improving Recovery of Slow Floating PGM Minerals at Zimplats

Munyaradzi Mugadza (689462)

A research report submitted to the Faculty of Engineering and the Built Environment, University of the Witwatersrand, in partial fulfilment of the requirements for the degree of Master of Science in Metallurgical Engineering

Supervisor: **Dr Vusumuzi Sibanda**

2 June 2015

Declaration

I declare that this research report is my own unaided work. It is being submitted in partial fulfilment of the requirements of the degree of Master of Science in Metallurgical Engineering at the University of Witwatersrand, Johannesburg.

It has not been submitted before for any degree or examination to any other university.

…….

(Signature of Candidate)

Dedication

Dedicated to my family for being a constant pillar of strength and support."

Acknowledgements

I would like to express my sincere gratitude to the following people:

1. My supervisor Dr Vusi Sibanda, for all the assistance towards the completion of this study sometimes going above and beyond the call of duty to make it a success.
2. My co-supervisor Mr Ngoni Mapurisa, for always lending a helpful hand and listening ear to the several challenges that came up during the course of this research.
3. Mr Louis Mabhiza and the Zimplats Technical Services team for providing the research opportunity and technical support.
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5. Dr Elias Matinde for all the research advice and guidance.

ABSTRACT

The work presented in this study involves investigation of ways of improving recovery of slow floating minerals in a single stage (MF1) Platinum Group Metal milling and flotation plant at the Zimplats Selous Metallurgical Complex. This study was conducted with an aim of improving flotation of PGMs by measuring the slow floating ratio (SFR) and analysing the effect on grade and recoveries as dosages of flotation reagents are altered. The fast floating fractions are usually recovered earlier on in the flotation circuit whilst the slower floating values are recovered at the back end of the circuit in the scavengers, cleaners, and high energy cells. The work presented here investigates the use of reagents in improving flotation kinetics of the slow floating fraction and improve its recovery in this part of the circuit to prevent the values being lost with the tailings.

Batch flotation rate tests were conducted on "in plant pulp" sampled from three sections of the flotation circuit namely, the scavenger, cleaner and high energy cells as the sections of the circuit where the slow floating fraction was most likely to be present. Different dosages of collector and depressant were added and flotation response measured. Chemical determination of the float test results was conducted by Nickel Sulphide fire assay with Inductively Coupled Plasma Optical Emission Spectroscopy (ICP-OES) finish. The kinetic parameters were investigated by fitting the experimentally obtained data into the Kelsall's unmodified rate equation using the KinCalc® flotation kinetics calculator.

Rate tests carried out on scavenger feed indicated that feed to the third bank of the scavenger circuit was most appropriate reagent testing station. Scavenger Bank 3 Feed was chosen as point of addition of reagents to isolate the banks with the majority of slow floating mineral. Addition of depressant to the feed to the third bank showed an improvement in SFR from 12.28 to 21.64 with the addition of 25g/t of depressant as the depressant acted on the floatable gangue. However, further depressant addition had a secondary effect of also reducing floatability of mineral values and SFR fell to 20.07 with 50g/t depressant and further to 11.69 with 100g/t depressant. These results showed the potential of staged addition of the depressant to the cells processing material with mostly slow floating values. Results on collector addition to third scavenger bank feed showed a minimal change in SFR with collector addition, however there were higher recoveries obtained due to the high mass pull observed. Better grades were obtained at a dosage of 30g/t collector than at 60g/t because the excess collector increased the pulp viscosity which led to massive entrainment of fine gangue material.

Rate tests done on cleaner feed showed a decrease in slow floating ratio (SFR) with depressant addition. Recovery of PGMs also fell with depressant addition as the depressant inhibited the flotation of both floatable gangue and mineral. Tests carried out with collector addition to cleaner feed indicated a small increase in SFR from 1.88 to 2.01. The recoveries achieved were lower with collector addition than without any reagent addition to cleaner feed.

Depressant addition to the high energy cell tailings showed that SFR increased in direct proportionality to dosage. Good recoveries of above 90% were obtained with all tests within the range of reagent dosage considered. The concentrate grades achieved were higher than as received flotation as floatability of gangue was reduced by the depressant. Collector addition also led to improvements in SFR however those improvements as well as recoveries obtained were generally lower than those observed with depressant addition as collector works less efficiently due to the lower number of liberated mineral faces to attach to in this part of a flotation circuit.

Sieve analysis was carried out on high energy cell tailings to ascertain if there was any need for regrinding showed that the particle size distribution of the tailings was 89.6% passing 75µm. This grind is adequate for the flotation of PGMs and hence regrinding was not done.

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Nomenclature

Symbols

Pt	Platinum
Pd	Palladium
Rh	Rhodium
Au	Gold
Pb	Lead
Cu	Copper
Ni	Nickel
Si	Silicon
Zn	Zinc
OH-	Hydroxyl Group
-COOH	Carboxyl Group
-NH2	Amine Group
I_{GF}	Fast Floating Fraction of Gangue
I_{PF}	Fast Floating Fraction of PGMs
φ	Kelsall model slow floating fraction of mineral
k_{GF}	Fast Floating Rate Constant for Gangue
k_{PF}	Fast Floating Rate Constant for PGMs
k_{PS}	Slow Floating Rate Constant for PGMs
k_{GS}	Slow Floating Rate Constant for Gangue
$\gamma_{M/W}$	Interfacial surface energy between mineral and water
$\gamma_{W/A}$	Interfacial surface energy between water and air
$\gamma_{m/A}$	Interfacial surface energy between mineral and air
$W_{M/A}$	The work required to separate the solid–air interface
λ	Kinetic constant for Gamma and Fully Mixed flotation models

Abbreviations

SMC	Selous Metallurgical Complex
MF1	Single Stage Mill-Float Plant
MSZ	Main Sulphide Zone
UG-2	Upper Group 2 PGM Reef
PGMs	Platinum Group Minerals
SFR	Slow Floating Ratio (equals kPS/kGS)
SIBX	Sodium Isobutyl Xanthate
CMC	Carboxymethyl Cellulose
DTTP	Dithiophosphate
MIBC	Methyl Isobutyl Carbinol
FFR	Fast Floating Ratio (equals kPF/kGF)
g/t	Grams per tonne
rpm	Revolutions per minute
µm	Micrometers

CHAPTER 1

INTRODUCTION

The Selous Metallurgical Complex (SMC) concentrator at Zimbabwe Platinum Mines (Zimplats) is a single stage mill float (MF1) plant that produces upgraded Platinum Group Metals (PGM) concentrate. This concentrate is smelted into matte in the SMC smelter and exported to South Africa for refining at Impala Refining Services.

The flotation circuit at SMC (Figure 1.1) uses a Rougher-Scavenger-Cleaner-Recleaner configuration. There are also two Delkor Column Cells and a Metso Reactor Cell System (high energy cells) as a way of improving recoveries by increasing the pulp residence time in the circuit so as to capture the slow floating value fraction. Typically in the configuration, the roughers recover the fast floating values. The rougher tailings are however also rich in PGMs and are sent to the rougher scavengers for further treatment to recover the remaining value and obtain the lowest possible tailings grade. The incremental recovery obtained in the treatment of rougher tailings is an important aspect of PGM flotation. The slower floating fractions of the ore require more residence time in the circuit in order to be floated. This fraction has a very important effect on the overall recovery of the flotation circuit.

Recovery of slow floating fractions is dependent on the rate differential between slow floating mineral and slow floating gangue. This rate differential is termed slow floating ratio (SFR). If SFR is less than a certain optimal value then the increment in recovery less than the fast floating mineral fraction falls away and becomes a circulating load in the cleaner tailings. The challenge presented in this study was to investigate the use of reagents in manipulating kinetics to increase the value of SFR at different points in the scavenger and cleaner circuits and hence improve incremental recovery of the slow floating fraction. The grade-recovery-time relationships obtained at each point were also studied.

Flotation is an engineering system and as such there are several different factors to take into account when studying performance. These factors can be summarised as mineralogy, kinetics, chemistry, operational, physical and hydrodynamic factors (Hay, 2010). In this study the effect of chemistry, reagents in particular, on the kinetics of flotation, grade and recovery

relationships obtained were considered. The work done thus centred on the effect of reagent dosage on the kinetics and grade recovery relationship of the flotation process.

The economic benefits that accrue with better recovery of slow floating minerals if optimized correctly are potentially immense since up to 29% of PGMs in the SMC feed can be considered slow floating (Zimplats Internal Report, 2012).

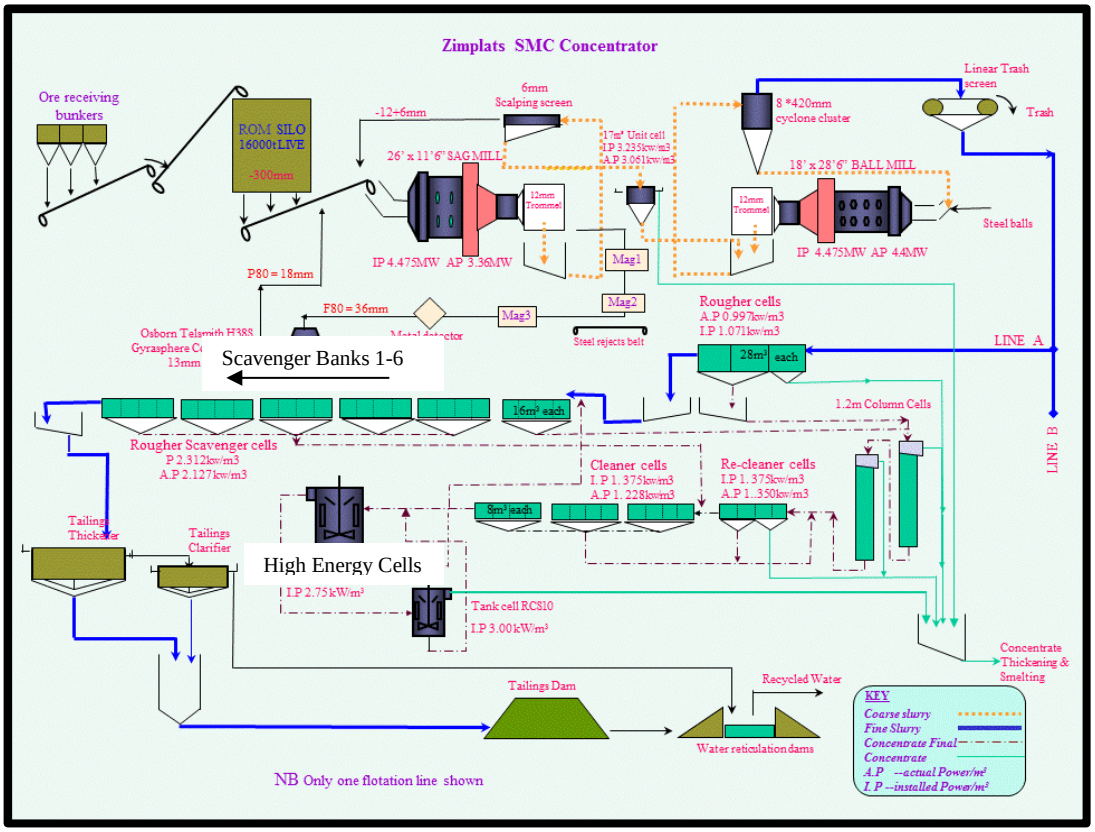


Figure 2.1: Zimplats SMC Flotation Flowsheet

1.1 Research Objectives

The objectives of the study were:

(i) To investigate the influence of flotation reagent dosage on the recovery of the slow floating fraction in the flotation circuit.

(ii) To determine the impact of recovery on grade at different reagent dosages.

(iii) To apply a suitable flotation rate model to determine flotation rate parameters at strategic points in the circuit using batch flotation rate tests.

CHAPTER 2

LITERATURE SURVEY

2.1 Flotation Theory

Flotation is a physico-chemical separation process that utilises the difference in surface properties of the valuable minerals and the unwanted gangue minerals (Wills, 2006). Mineral separations are achieved by the exploitation of differences in the surface properties of the minerals to be separated specifically by the ability of air bubbles to selectively adhere to specific mineral surfaces in mineral-water slurry. The particles with attached air bubbles are then carried to the surface and removed, while the particles that remain completely wetted stay in the liquid phase (Kawatra, 2011). Surface properties are very specific to a particular type of mineral because they are determined by its chemical composition and type of chemical bonding. These properties are unique to a mineral and thus offer very selective separation capability (Woollacot and Eric, 1994)

Flotation is a separation process that has found prominence because of the need to treat complex or low grade ores where the liberation particle size is too small for efficient gravity separation or where the gravity difference between minerals is too small (Gupta and Yan, 2006). Froth flotation can be adapted to a broad range of mineral separations, as it is possible to use chemical treatments to selectively alter mineral surfaces so that they have the necessary properties for the separation. This separation method derives its usefulness from its relatively high efficiency and selectivity, application to extraction of most minerals and high throughput capability. It finds its most common application in the extraction of sulphide ores but also used in separating coal from ash-forming minerals, removing silicate minerals from iron ores, separating phosphate minerals from silicates and even non-mineral applications such as de-inking recycled newsprint.

2.1.1 Flotation Principles

As mentioned earlier, the process of froth flotation is based on the differences in surface properties of the different minerals present in a slurry. Because the process is dependent on

the hydrophobicity imparted to one of the minerals, air bubbles have to be introduced to the slurry to collect the hydrophobic particles whilst the hydrophilic particles stay wetted and in the pulp. The air bubbles become coated with the hydrophobic particles as the bubbles rise through the slurry and levitate to the surface where they are removed as a mineral rich froth. The level of hydrophobicity that a mineral particle possesses dictates whether it gets attached to an air bubble and recovered to the froth or remains in the pulp. This is measured by the contact angle formed between the particle and air bubble. The forces acting on the solid-liquid-air interface are shown in Figure 2.1. The equilibrium tensile force balances are related by Young's equation (Kelly and Sportiswood, 1989);

$$\gamma_{M/A} = \gamma_{M/W} + \gamma_{W/A}cos\theta \qquad (2.1)$$

$\gamma_{M/A}$, $\gamma_{M/W}$ and $\gamma_{W/A}$ are the interfacial surface energies between mineral and air, mineral and water and water and air, respectively, and θ is the contact angle between the mineral surface and the bubble (Kelly and Sportiswood, 1989).

The force required to break the particle–bubble interface is called the work of adhesion, $W_{M/A}$, and is equal to the work required to separate the solid–air interface and produce separate air–water and solid–water interfaces (Wills, 2006), i.e.

$$W_{M/A} = \gamma_{W/A} + \gamma_{S/W} - \gamma_{\frac{S}{A}} \qquad (2.2)$$

Combining the two equations gives:

$$W_{M/A} = \gamma_{\frac{W}{A}}(1 - cos\theta) \qquad (2.3)$$

Equation 2.3 shows that the work of adhesion increases with contact angle thus the work required to disrupt he system can be said to increases with contact angle. Therefore, hydorophobicity increases with contact angle.

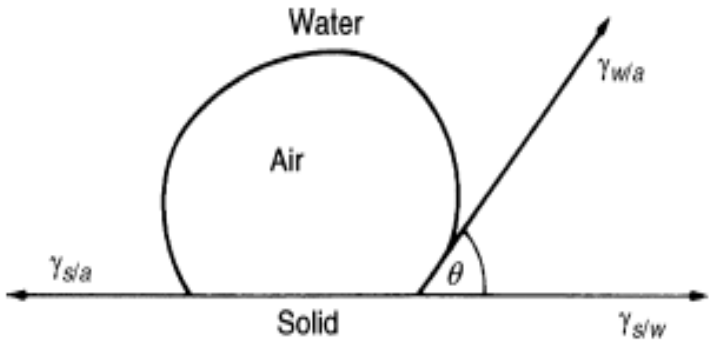


Figure 2.1: Idealised Illustration of the equilibrium contact angle between bubble and particle in an aqueous medium (Wills, 2006)

2.1.2 Flotation Reagents

Most minerals are not hydrophobic in their natural state with talc and graphite being two notable examples of the exception. As such, it is generally necessary to condition slurries with flotation reagents to impart the desired surface properties to the mineral of interest and give them the selective characteristics required for separation from the gangue. The flotation reagents used for this purpose are divided into three main categories as follows:

2.1.2.1 Collectors

Collectors are surfactants whose basic role in flotation is to selectively form a hydrophobic layer on a given mineral surface in the flotation pulp and thus provide conditions for attachment of the hydrophobic particles to air bubbles and recovery of such particles in the froth product. Collectors are generally bipolar organic compounds and may either be ionizing compounds that dissociate into ions in water or non-ionizing compounds that are practically insoluble and render the mineral water repellent by covering its surface with a thin film (Wills, 2006). The ionised collectors are either cationic, (amines) or anionic (fatty acids or sulphydril compounds such as xanthates or dithiophosphates (Gupta and Yan, 2006). Xanthates and dithiophosphates are the most widely used collectors in the flotation of sulphide ores. Collectors will normally have an ionic “head” that possesses an affinity for the mineral surface and an organic hydrophobic “tail” that then renders the otherwise hydrophilic mineral surface hydrophobic. Once the mineral is rendered hydrophobic, it tends to move

away from the water towards the air bubbles resulting in particle bubble attachment. The simplistic mechanism of the process is shown in Figure 2.2.

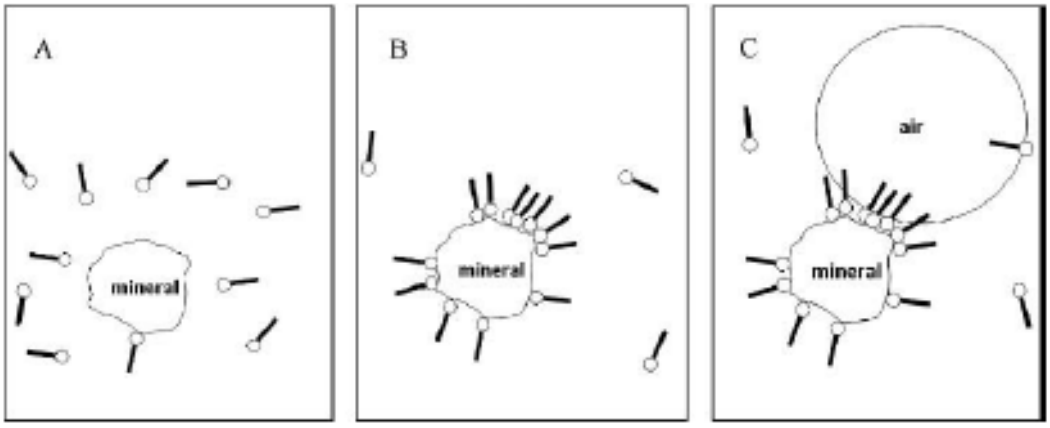


Figure 2.2: (a) Collector in aqueous phase (b): Adsorption onto mineral particle (c) Particle bubble attachment. (Gupta, 2006)

2.1.2.2 Modifiers

Modify the action of the collector on mineral surfaces and as a consequence govern the selectivity of the flotation process. In the presence of regulators, the collector only adsorbs on particles that are targeted for recovery. This group of flotation reagents consists of activators, depressants, dispersant and pH regulators.

Activators allow collector adsorption on minerals by changing the chemical character of the mineral surfaces to increase interaction with the collector molecule. An example of an activator is copper sulphate which is used in the flotation of PGMs. The mechanism suggested for this process is an ion exchange where the metal ion on the surface of the base metal sulphide particle is exchanged with the Cu^{2+} ions in solution thus precipitating a copper rich sulphide on the surface of the particle. The xanthate collector readily adsorbs on the copper rich surface and a xanthate collector salt is formed by chemisorption (Grobler *et al*, 2005).

2.1.2.3 Depressants

Depressants are used to increase the selectivity of flotation by rendering certain minerals hydrophilic (water avid), thus preventing their flotation. The process can be loosely defined as the opposite process to collection. Organic reagents such as starch, tannin and dextrin do

not ionize in solution but form colloidal particles in the pulp which can be deposited on mineral surfaces and thereby preventing flotation of talc, graphite and calcite. Examples of depressants used in industry include sodium cyanide for Pb-Cu-Zn ores, potassium dichromate to depress galena in copper–lead separations while polymeric depressants like guar and carboxymethyl cellulose (CMC) are commonly used in PGM flotation.

2.1.2.4 Frothers

These are organic chemicals which reduce the surface tension of the water to stabilize the bubbles into a froth layer at the top of the flotation cell to make concentrate removal easier. Frother also allows for finer bubbles to be formed and decreased bubble size implies the presence of more bubbles per unit volume and hence an increased flotation rate (Nashwa, 2007). An ideal frother should act entirely in the liquid phase and therefore should not influence the state of the mineral surface. It should have no collector properties. An ideal froth allows entrapped gangue particles to drain out whilst also ensuring that the values are retained long enough to be recovered into the concentrate and thereafter break down rapidly to prevent interference with subsequent processing operations. These properties are a function of the frother used. Frothers are generally heteropolar surface active organic reagents, capable of being adsorbed on the air–water interface. Frothing action is due to the ability of the frother to adsorb on the air–water interface because of its surface activity and to reduce the surface tension, thus stabilising the air bubble. The action of a frother is shown in Figure 2.3.

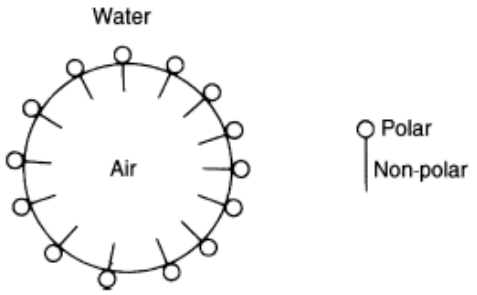


Figure 2.3: Action of a frother (Wills, 2006)

The most effective frothers include in their composition one of the following groups: hydroxyl (-OH), carboxyl (-COOH), carbonyl (-CO), amino ($–NH_2$) and sulpho ($–OSO_2OH_2SO_2OH$). The acids, amines and alcohols are the most soluble of the frothers. Alcohols have found wide applications since they have practically no collector properties and in this respect are preferable to other frothers such as the carboxyls, which are also powerful

collectors (Wills, 2006). Frothers commonly used include natural chemicals like pine oil, cresylie acid and synthetic reagents such as methyl isobutyl carbinol (MIBC) and polyglycol ethers.

2.2 Flotation Kinetics

Particles of the same mineral float at different rates due to different particle characteristics and cell conditions. The mechanism responsible for these different responses is flotation kinetics. A laboratory flotation batch rate test on an ore/slurry sample generates a recovery, grade, and concentrates mass profile with time. These profiles describe the flotation response and performance of the ore under the given conditions. It is important to note that the way mineral associations in the ore affect recovery by flotation is determined by the kinetics i.e. the fast and slow floating behaviour (Eurus Mineral Consultants, n.d.). The grade-recovery characteristics of the flotation of the ore in question also depend on the kinetics through the relative floatabilities of the gangue to value minerals. Figure 2.4 shows how kinetics fit into the overall picture of testing and characterising an ore.

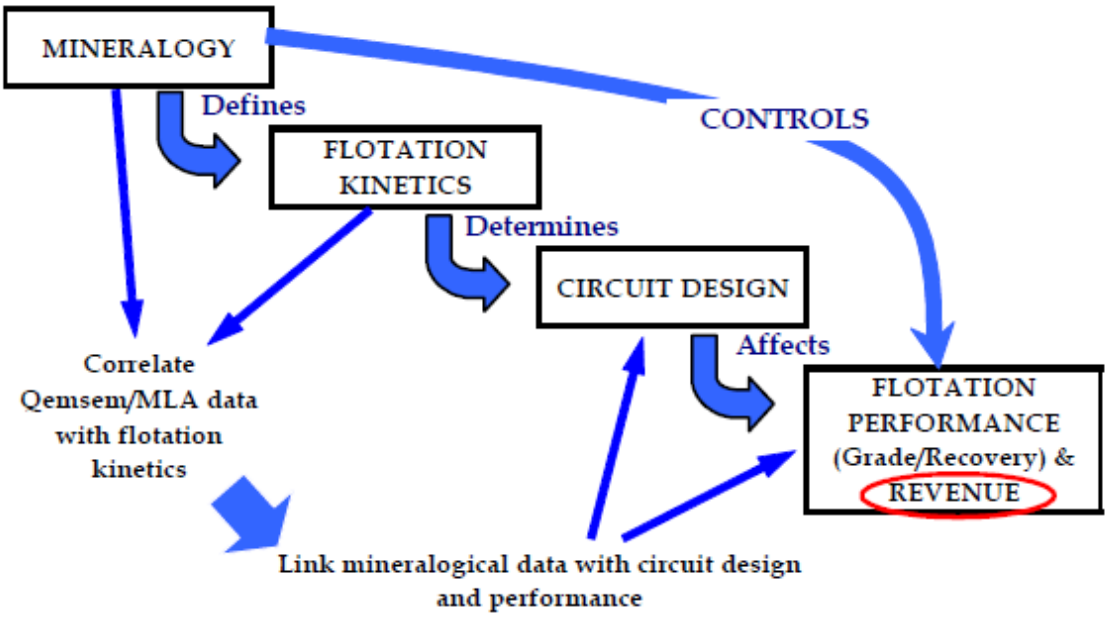


Figure 2.4 Flotation Performance Influence Diagram (Eurus Mineral Consultants, n.d.)

The majority of authors studying flotation kinetics have described flotation as a 1st order rate process (Wills, 2006). This assumption is the basis for the 1st order flotation rate equation on which most flotation models are based.

Considering a mineral *i* that consists of *j* subclasses;

$$C_i = \sum_j^n C_{ij} \qquad (2.1)$$

Where C_i represents the concentration (mol/l) of the floatable mineral *i* in the float cell and C_{ij} represents the fraction of that mineral in the *j*th class. k_{ij} is the flotation rate constant (in min^{-1}) for that fraction.

The Batch Process is considered to be a 1st order process hence;

$$\frac{dC_{ij}}{dt} = -k_{ij}C_{ij} \qquad (2.2)$$

Integrating both sides of the equation;

$$C_{ij}(t) = C_{ij}(0)\exp(-k_{ij}t)$$

$$C_i(t) = \sum_j C_{ij}\,(0)\exp(-k_{ij}t)$$

$$C_i(t) = C_i(0)\sum_j P_{ij}\,(0)\exp(-k_{ij}t)$$

Where P_{ij} (0) = fraction of mineral *i* which occurs as subclass *j* in the feed.

If R_i(t) = is the fraction of the mineral *i* recovered in the concentrate at time *t* then,

1- R_i (t) = is the fraction of the mineral *i* in the tailings at time *t* and, at constant volume is also $= \frac{C_i(t)}{C_i(0)}$ hence,

$$1 - R_i(t) = \frac{C_i\,(t)}{C_i(0)} = \sum_j P_{ij}(0)\exp\left(-k_{ij}(t)\right) \qquad (2.3)$$

2.2.1 Flotation Kinetics Modelling

There have been various mathematical models presented by several different authors that attempt to describe kinetics of flotation of different minerals. These models are present in simulation software such as MODSIM™, USIM™, PAC JKSimFloat, Kincal® (Brezani, 2010). This section discusses the flotation models that describe batch laboratory flotation testing.

2.2.1.1 Classical Model

Among flotation models, the most acceptable model is the kinetic model which uses the first order reaction equation (Equation 2.1) as the starting point (Zhang, 1989). It is based on rules of mass transport from one phase to another. The derivation is shown in section 2.2.

$$R = R_\infty(1 - e^{-kt)} \qquad (2.4)$$

C = the concentration of mass at time t (mol/l)

R = Fractional recovery of mineral (%)

R_∞= Fractional recovery at infinite time (%)

k = flotation rate constant (min^{-1})

t = time (min)

This first order model assumes monodisperse feed with particles of constant floatability. A plot of In (1-R) against time should give a straight line whose gradient represents -k, the rate constant. If the plot of is not linear then either the rate is not first order or the floatability of particles is not constant. If the rate is first order but the floating particles in the pulp do not have identical properties, i.e., there is a continuous range of rate constants for the mineral being recovered, the integrated form of the rate equation for a semi-batch process becomes

$$C = C_0 \int_0^\infty e^{-kt} f(k,0) dk \qquad (2.5)$$

where f*(k,o)* represents a continuous distribution of rate constants.

The distribution of rate constants may arise from intergrowths of minerals or a distribution of particle sizes (Gupta, 2006). The problem then is related to the accurate estimation of the distribution of rate constants. Different interpretations of this distribution of rate constants give rise to a number of first order rate models as discussed in the flowing section.

2.2.1.2 Klimpel Model

The Klimpel model uses two parameters to describe flotation. This model differs from the classical model in the representation of the rate constant distribution as uniform or rectangular. A rectangular distribution is one in which the quantity is constant over a fixed interval. For example, the rate constant has a fixed or constant value over a limited property range. For low and high values of the property, the rate constant is zero. Compare this with the classical first order model where the rate constant is assumed to have a constant value for all property values (Gupta, 2006).

$$R = R_\infty \left(1 - \frac{1}{kt}(1 - e^{kt})\right) \qquad (2.6)$$

R = Recovery of mineral (%), R_∞ = Recovery at infinite time (%), k = modified first order rate constant (min^{-1}), t = flotation time (min)

2.2.1.3 Kelsall (Unmodified) Model

This is a two fraction kinetic model with only two rate constants. The behaviour of each fraction is described with a corresponding rate constant. The two fractions are named as fast floating and slow floating fractions. A third parameter used in this kinetic model, φ, is used to represent the fraction in the feed that floats with the slow constant. The use of two rate constants was considered to give a better approximation to the distribution of particle floatabilities than could be obtained with a single rate constant. The model is shown below.

$$R = (1-\varphi)\left(1-e^{-k_f t}\right)+\varphi\left(1-e^{-k_s t}\right) \quad (2.7)$$

k_f = fast floating rate constant (min^{-1}), k_s = slow floating rate constant (min^{-1}), φ = Slow floating fraction and t = time (min)

2.2.1.4 Modified Kelsall Model

The modified Kelsall's model by Jowett (1974) includes the effect of infinite recovery R_∞, and increases the number of independent variables to four.

$$R = R_\infty\,(1-\varphi)\left[1-\exp\left(-k_f t\right)\right]+\varphi[1-\exp(-k_s t)] \quad (2.8)$$

k_f = fast floating rate constant (min^{-1}) , k_s = slow floating rate constant (min^{-1}) , φ = Slow floating fraction and t = time (min)

2.2.1.5 Gamma Model

The Gamma model was proposed by Loveday 1966 and Imaizumi and Inoue 1968 (Gupta, 2006). It has three independent parameters and can be simplistically described as being made up of P exponential distributions.

$$R = R_\infty\left[1-\left[\frac{\lambda}{\lambda+t}\right]^P\right] \quad (2.9)$$

R_∞ = Recovery of mineral, R = infinite recovery of mineral, λ = kinetic constant (min), P = exponent, t = flotation time (min)

2.2.1.6 Fully Mixed Model

Flotation results may be represented by an expression analogous to an equation describing the time concentration for a series of fully mixed reactors. In recovery terms this may be expressed in a mathematical form, similar to the Gamma model

$$R = R_{\infty}\left[1 - \left[\frac{\lambda}{\lambda + \frac{t}{k}}\right]\right] \tag{2.10}$$

R_{∞} = Recovery of mineral, R = recovery of mineral at infinite time, k = kinetic constant (min), t = flotation time (min)

2.2.2 Kinetic Parameter Estimation

The study of flotation is made convenient by the fact that most of the useful information can obtained from a standard batch flotation rate test. The flotation rate parameters in any model describe the flotation response and the performance of the ore under question and these can be calculated to give a numerical representation of the flotation performance. The basic method used to calculate the flotation parameter is the graphical method;

Considering the example of *Equation 2.3* and a mineral with 2 subclasses i.e. fast floating (P_f) and slow floating (P_s);

$$1 - R_i(t) = P_f e^{-k_f t} + P_s e^{-k_s t}$$

Where P_{ij} = fraction of mineral i which occurs as subclass j in the feed hence, $P_f + P_s = 1$

To estimate the parameters it is assumed that $k_f >> k_s$ and at large values of t, $P_f e^{-k_f t}$ is negligible and

$$1 - R_i(t) = P_s e^{-k_s t}$$

And taking natural logarithms of both sides;

$$\ln\{1 - R_i(t)\} = \ln P_s - k_s t$$

Plotting $\ln\{1 - R_i(t)\}$ against time gives a straight line where ln P_s is the intercept and k_s is the slope.

To estimate P_f and k_s using the values obtained above;

$$\ln\{1 - R_i(t) - P_s e^{-k_s t}\} = \ln P_f - k_f t$$

And similarly, plotting $\ln\{1 - R_i(t) - P_s e^{-k_s t}\}$ against time gives a straight line with P_f as the intercept and gradient equal to k_f.

2.2.3 Kinetic Parameter Estimation using KinCalc®

The results of the flotation batch rate tests carried out in this study were used to estimate all the required kinetic parameters relating to Kelsall's unmodified equation. This model was an obvious choice for any study involving slow floating minerals because it describes a slow floating rate and fraction. Kelsall's unmodified equation was preferred over the classic first-order Klimpel and Jowett/modified Kelsall models because it is the only model that does not use recovery at infinite time. Instead, this model assumes a recovery of 100% at infinite time (Hay and Rule, 2003). The software used in this study was KinCalc® which uses Excel Solver to fit the experimental data into the Kelsall's unmodified model using the least squares regression method and generates the parameters relating to the model. Kinetic values estimated by KinCalc® from the flotation rate test can be used to account for ore's behaviour and measure the floatability at any point in a circuit. KinCalc® uses preloaded testwork information for an ore to estimate its kinetic parameters at any given point in the flotation circuit. The parameters that are generated from the software and used for kinetics investigations are defined by Hay, 2005 as:

I_{PF}= fast floating fraction of PGMs

k_{PF} = fast floating rate constant of PGMs

k_{PS} = slow floating constant rate of PGMs

I_{GF} = fast floating fraction of gangue

k_{GF} = fast floating rate of gangue

k_{GS} = slow floating rate of gangue

FFR = (k_{PF}/k_{GF}) fast floating ratio i.e. the fast floating flotation rate of PGMs relative to gangue

SFR = (k_{PS}/k_{GS}) slow floating ratio i.e. the slow floating flotation rate of PGMs relative to gangue

2.3 Flotation Optimization

In studying flotation optimization, it is clear that due to the high number of variables that affect flotation performance a good understanding of the underlying principles is required.

This understanding is critical in assessing the factors that affect flotation and how they can be manipulated to improve the system as a whole. The flotation performance of a system is dependent on six main components which are defined as mineralogy, kinetics, chemistry, operational, physical and hydrodynamic factors (Hay, 2005). Each of these factors has different components that are interrelated and thus any change in one of these will cause or demand changes in other parts of the system (Kawatra, 2011). It is therefore important in analyzing flotation performance that this is taken into account as it increases the degree of difficulty in performing the required analysis. Another factor to take into consideration is the large number of variables that affect flotation performance. Should all these factors be taken into consideration individually, it would require collection of an enormous amount of data from an equally large number of different experimental procedures (Figure 2.4). The benefit associated with taking this approach is however outstripped by the cost and time required to carry this out. As a result an alternative approach can be adopted where the factors that have the largest effect on the flotation performance are investigated and analyzed. One of the factors with a significant influence on the flotation performance of a system is the kinetics of the system. This is true because most of the other factors that affect flotation performance do so by affecting the kinetics in some way or the other. It has been suggested that 90% of flotation performance is dependent on the kinetics in the system (Hay, 2010). As a result, it is possible to analyze the flotation system comprehensively on the basis of a minimum number of variables which affect the flotation kinetics. The advantage with this approach is that the investigation can be done on the basis of the flotation rate test using relatively simple methods and apparatus (Hay, 2005). This kinetic approach is based on the simplification of flotation as a first order kinetic phenomenon. Figure 2.5 is a schematic of factors that affect flotation.



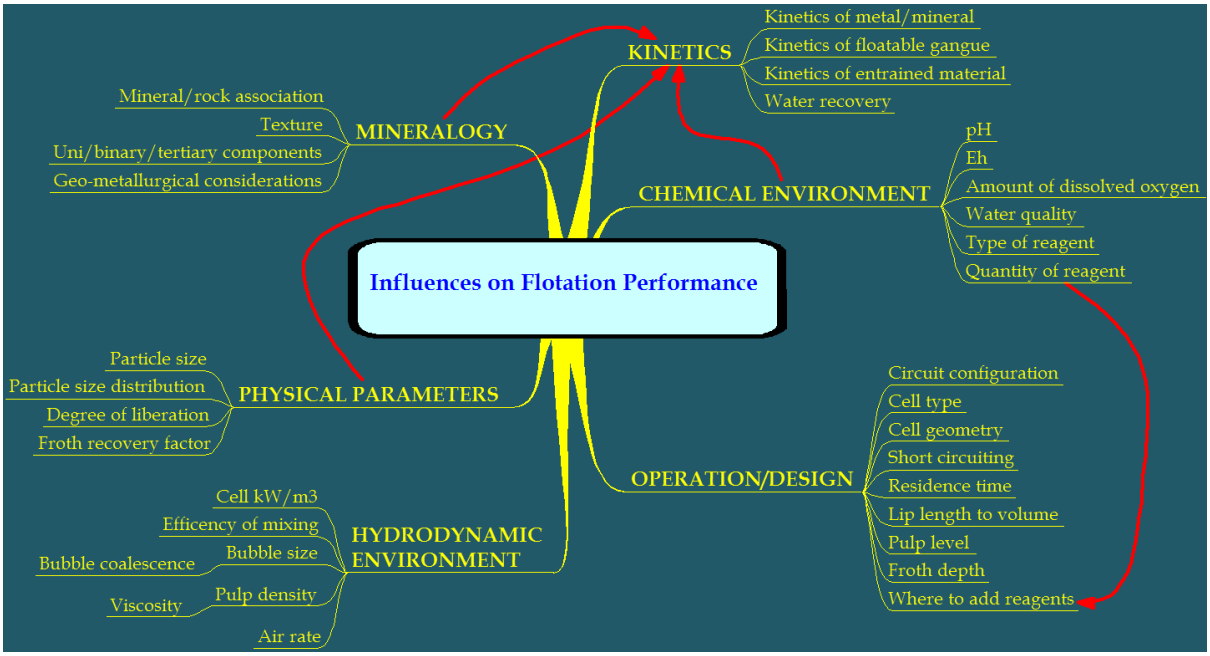


Figure 2.5: The Factors Affecting Flotation Performance (Hay, 2010)

2.4 Factors Affecting Floatability

2.4.1 Kinetics and Effect of Slow Floating

The Source of Recovery: Recovery originates from three sources in the ore's mineralogical structure;

1. Its liberated, easily floatable fast floating metal/mineral fraction,
2. Its fast to medium floating metal/mineral fraction that may or may not be overwhelmed by fast floating gangue and
3. Its slow floating metal/mineral fraction that competes with slow floating gangue.

The debilitating effect that floatable gangue has upon plant performance may not be fully appreciated. Taking an average of all streams in a base metal sulphide circuit, floatable gangue constitutes probably 92% (Bryson, n.d.). The easily recoverable, fast floating portion of mineral is recovered in about the first quarter to one third of a circuit. Thereafter, ore performance, circuit design and the capital expenditure that goes with it is predominantly about how not to float the gangue. On average the slow floating rate of mineral is 0.03 and that of gangue is 0.0023 (Bryson, n.d.). Once the stream masses are taken into consideration it is not surprising that plants struggle to convert rougher recovery in the last half of the

rougher bank into final concentrate recovery most, if not all ends up as cleaner tailings. A kinetic balance of the slow floating fractions is as follows;

Values: *8% mass x 0.0300* *= 0.0024*

Gangue: *92% mass x 0.0023* *= 0.0021*

The kinetic differential of slow value to gangue, the slow floating ratio (SFR), is 13.04 but when the masses in the float plant in which they are active are brought into the equation the differential drops to only 1.14. (Hay, 2005)

The overall performance of a flotation system depends on the floatability of the material being floated. Floatability is determined by how well the metal and the gangue float. The two main components that drive floatability are the fast floating rate (FFR) and the slow floating rate (SFR) and these in turn are the main driving forces of recovery in flotation plant. The fast floating fraction should always be recovered and the incremental recovery sourced over and above this is what is termed the slow floating fraction. The recovery of this fraction is dependent on the flotation rate differential between the slow floating mineral and the slow floating gangue which has to be large enough to allow the mineral to be upgraded to the concentrate. This rate differential is termed the slow floating ratio (SFR). If SFR is less than a certain value then the increment in recovery less than the fast floating PGM faction (I_{PF}) falls away and becomes a circulating load in the cleaner tailings. The challenge presented in this study is to investigate ways of manipulating the kinetics to increase the value of SFR at different points in the scavenger and cleaner circuits and hence improve the incremental recovery of the slow floating fraction.

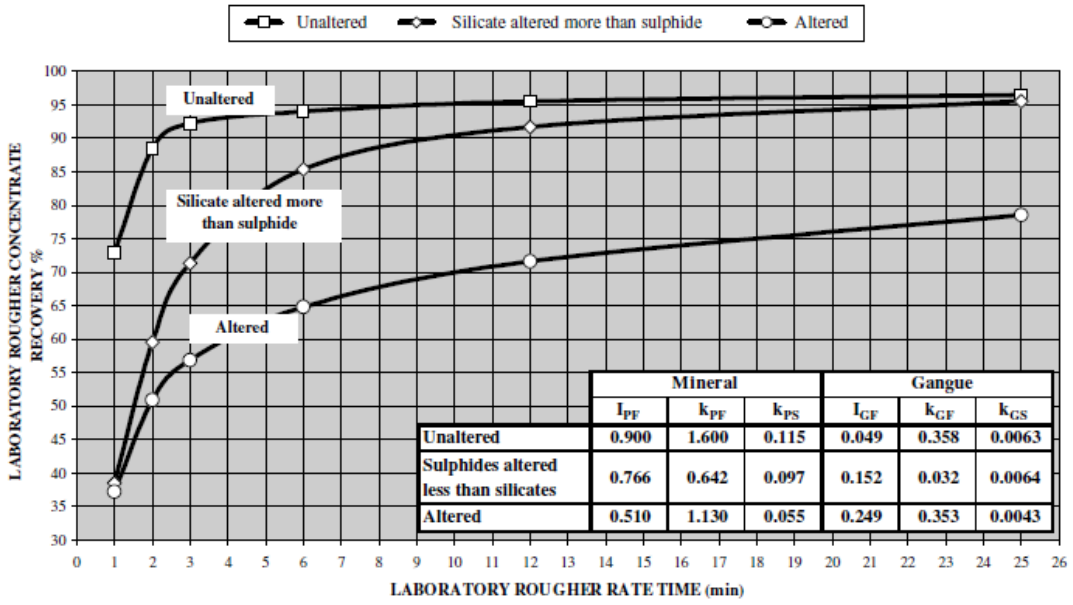
2.4.2 The Effect of Mineralogy

The link between mineralogy and flotation performance can be illustrated by considering how ores with different textural association of minerals respond to flotation.



Figure 2.6: Clean, intermediately and considerably altered ores (Hay, 2005)

Figure 2.6 shows ores with similar mineral content but different textural association between the minerals. On milling, the clean ore will show good liberation and recovery characteristics. The ore where intermediate alteration of sulphide minerals has taken place with silicates altered more than the sulphides will show good recovery characteristics but its gangue component is very floatable. The ore that has been subjected to considerable alteration processes with overgrowth of sulphides by secondary silicate minerals will exhibit low to medium degree of liberation and recovery. The response to flotation of the three ores is summarised in Figure 2.7. From the summary it can be seen that the unaltered ore is associated with higher and faster fast floating fractions. It can be concluded that the flotation of the three ores must necessarily differ depending on mineralogical associations in order to get the optimum performance from each.



	Mineral			Gangue		
	I_{PF}	k_{PF}	k_{PS}	I_{GF}	k_{GF}	k_{GS}
Unaltered	0.900	1.600	0.115	0.049	0.358	0.0063
Sulphides altered less than silicates	0.766	0.642	0.097	0.152	0.032	0.0064
Altered	0.510	1.130	0.055	0.249	0.353	0.0043

Figure 2.7: Recovery Time Profiles for Unaltered, Intermediate and Altered Ores (Hay, 2005)

The mineralization and mode of occurrence of the PGMs in the Great Dyke of Zimbabwe is of utmost importance considering the low concentrations of these elements in the ores in nature (1-15g/t) (Vermark, 2005). The most commonly occuring PGMs are sulphides, arsenides and tellurides. The Great Dyke PGMs generally have a diameter of less than 10 micron. This textural association will generally influence recovery during flotation. Mineralogy studies to develop the relationship between pyrrhotite mineralogy and the flotation performance of selected PGM deposits in terms of their crystallography, mineral association, mineral chemistry and reactivity have been carried out by several researchers. By characterizing the different pyrrhotite samples using ore petrography, X-Ray Diffraction (XRD) and mineral chemistry analysis and investigating flotation performance using microflotation, Becker (2009) found that there were differences in flotation performance in the different samples linked to their reactivity towards oxidation. The main sulphide zone MSZ of the Great Dyke contains 0.5-10% volume of the sulphides pyrrhotite, pentlandite, chalcopyrite and subordinate pyrite and the PGMs are usually included mainly in pyrrhotite and chalcopyrite. The PGM proportions in the different chambers of the Great Dyke vary, and this coupled with the varying average grain size dictates that mineralogy of the ores should be taken into account when investigating the flotation performance (Oberthur *et al.*, 2002). As is well-known the chalcopyrite is fast floating and pentlandite, although not as fast floating as the former, tends to float readily. Pyrrhotite on the other hand is well-known to be "notoriously slow floating" (Allison and O'Connor, 2011). Studies have thus been done to ascertain how the flotation performance of individual minerals affects the overall flotation performance of an ore containing these minerals. Ekmekci et al., (2005) did a plant performance evaluation using quantitative mineralogical and chemical analysis on a Cu-Zn flotation plant using a detailed plant sampling survey. The sampling survey involved carrying out a performance evaluation of all streams of the plant. The quantitative mineralogical data used included mineral quantities, size distribution of free and locked mineral grains. From this investigation the losses of value mineral were tracked on the basis of size fraction and recommendations made based on the analysis of those results. Martin and Mckay, (2003) carried out a mineralogy study at Lac Des Ill Mill with the twin objectives of:

(i) Understanding feed PGM mineralogy and establishing relationships between PGM floatability and mineralogy, and

(ii) Bulk sample mineralogy, including sulphide assemblage, and liberation, and establishment of the mineralogy of the culprit silicate minerals floating to the final concentrate.

From a study of the mineralogy of fast floating (rougher concentrate) and slow floating (scavenger concentrate) fractions, they concluded the difference in floatability is driven by grain size and liberation. PGM speciation was found to be not very important to flotation performance however what should be regarded as important from the mineralogical perspective is the mineral rock associations and geometallurgy.

2.4.3 The Effect of Physical Factors

2.4.3.1 Particle Size

The significance of particle size on flotation performance is easily observable even from first principles. Generally speaking, flotation performance deteriorates rapidly when operating in the very fine or very coarse particle size ranges. This dictates that there is a size range in which flotation will take place optimally. Recovery-size-curves where the recovery for each particle size was plotted against the average particle size are the diagnostic tool generally used in these types of studies (Trahar, 1981). The typical recovery-size-liberation relationship is illustrated in Figure 2.8.

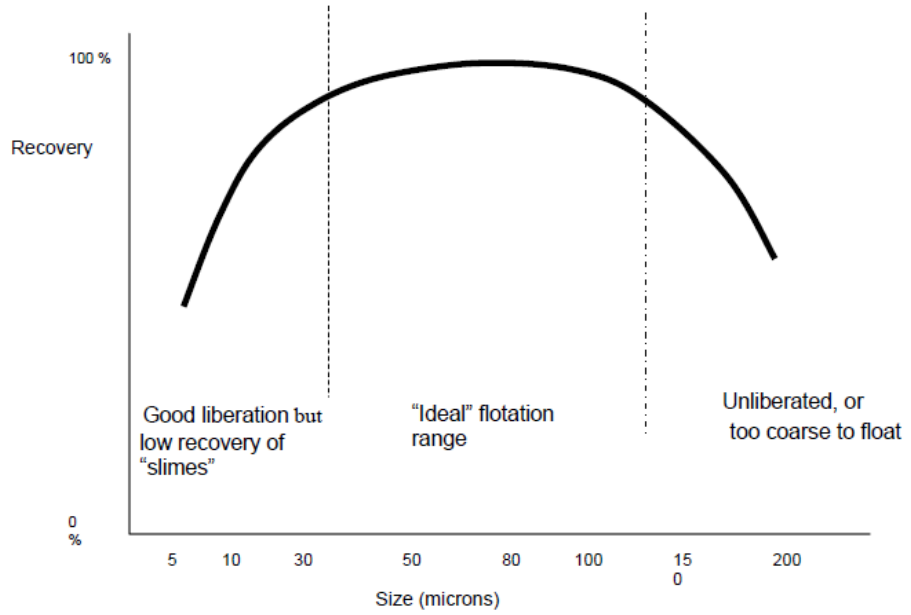


Figure 2.8: Typical Recovery Size Curve (Pease et al., 2004)

2.4.4.2 Liberation

Generally, in nature PGMs are finely disseminated, the average grain size being less than 10 micron, so that grain size, liberation and association tend to dictate mineral floatability. Platinum group mineral grain size and association can be split into four categories in descending order of ease of flotation:

• liberated PGM,
• PGM associated with base metal and iron sulphides,
• PGM occurring on host mineral grain boundaries (mainly silicates),
• PGM locked in silicates.

The role of liberation in flotation is closely influenced by the particle size and generally one cannot be considered in isolation of the other. As mentioned before the PGM speciation is not a very important metallurgical consideration when analysing flotation performance. The reason for this is that it does not dictate fast or slow floating in the ore. From metallurgical perspective, the species of PGM does not significantly affect flotation kinetics. QemSCAN studies of the Impala UG-2 plant feed have shown that the composition of fast-floating PGM is similar to that of the slow floating PGMs, and the unfloated PGMs, indicating no clear ‘‘hierarchy’’ in mineral floatability (Nel *et al*., 2005) . They suggested that the nature of occurrence of the PGMs was what drove floatability. Martin and McKay, 2003 did similar

studies on the Lac des Iles (LDI) circuit in Canada and found that rather than being of any particular species, the faster floating PGM are coarser and better liberated or middlings whilst the finer grains float much slower. They also concluded that if the grain size distribution of the slow floating PGM overlapped that of the non-floating PGM, then the flotation had not reached its natural endpoint.

The study at LDI recommended the following ways of improving recoveries of slow floats

- Regrinding and floating ultra-fine liberated PGM
- Floating more of the low grade PGM middlings

Both of the above measures require long residence times and highly optimized cell hydrodynamics. Lab and pilot studies at LDI showed that a fine primary grind of 80% passing 40 micron gives improved recoveries, however recovery of these ultra-fines in industrial flotation cells was less satisfactory. This approach however presents problems due to the lack of comparability of the cell dynamics at lab scale and those at industrial scale.

2.4.4 The Effect of Reagents

The typical reagent suites employed by concentrators in the treatment of PGMs focus largely on recovering the sulphides using typically sodium iso-butyl xanthate (SIBX) as the primary collector often, in the case of PGM concentrators, in conjunction with a secondary collector such as a dithiophosphate (DTTP). Many, but not all, of the flotation plants add copper sulphate as an activator. Generally these concentrators use a polysaccharide depressant, either carboxymethyl cellulose (CMC) or a modified guar gum, to reduce the recovery of naturally floating gangue minerals such as talc, pyroxene, plagioclase, etc. The flotation is carried out at the natural pH of the milled ore which may vary between 7.5 and 9 but is typically closer to the higher value (Allison and O'Connor, 2011).

Bradshaw *et al.,* (2004) did a study on the flotation behaviour of collectors and depressants for a copper nickel ore. By considering the copper and nickel recoveries and grades, mass of floatable and entrained gangue recovered and the water recovery, they found that effect of changing reagents can be masked and hence primary and secondary effects should be taken into account. They concluded that the effect of reagents should be assessed holistically in both the froth and pulp phase paying particular attention to the effects of entrainment for each different reagent suite.

2.4.4.1 Depressant

Although addition of depressant is one of the more commonly used ways of increasing SFR the optimization of depressant dosages has not been fully investigated. Some studies have been carried out on mixed depressants which have shown that there is no performance benefit accrued from blending depressants (Corin and Harris, 2010). There is need for further investigation of the role of depressants in improving recoveries and specifically the effect of depressant addition on SFR at different points in the scavenger and cleaner circuits. Gangue has a tendency to rapidly reactivate; hence the need to stage add depressant and other reagents in order to continuously change the state (and reactivity) of the PGM and gangue components. Slow floating mineral competes with slow floating gangue, depressants assume greater importance as they hold the key in increasing the selectivity of the flotation. Addition of depressant progressively decreases SFR but also reduces the fast and slow rates of PGMs. If SFR does not change with addition of depressant, it is an indication that slow mineral and gangue are in the form of binary particles and further grinding is needed to increase liberation and thus recovery.

2.4.4.1.1 The Mintek Two Concentrate Process

The Mintek two concentrate process developed for the South African platinum industry is an example of the use of depressant to manipulate SFR. If mineral-gangue liberation is good then depressant can be used to selectively change gangue kinetics relative to mineral kinetics. The fast floating fraction of mineral is then less encumbered by floatable gangue and can lead to an improvement in both recovery and grade. Thus without altering the value of fast floating fraction (usually achieved by finer milling) performance improves as a result of the environment being tailored to its particular needs. Figure 2.9 is a schematic of a circuit where the Mintek two concentrate process was used to produce two concentrates and the addition of depressant was both increased and proportioned between the two cleaner banks. The change to the two concentrate process improved recovery by 4.2% and grade increased from 405 to 660 g/t (Hay, 2005).

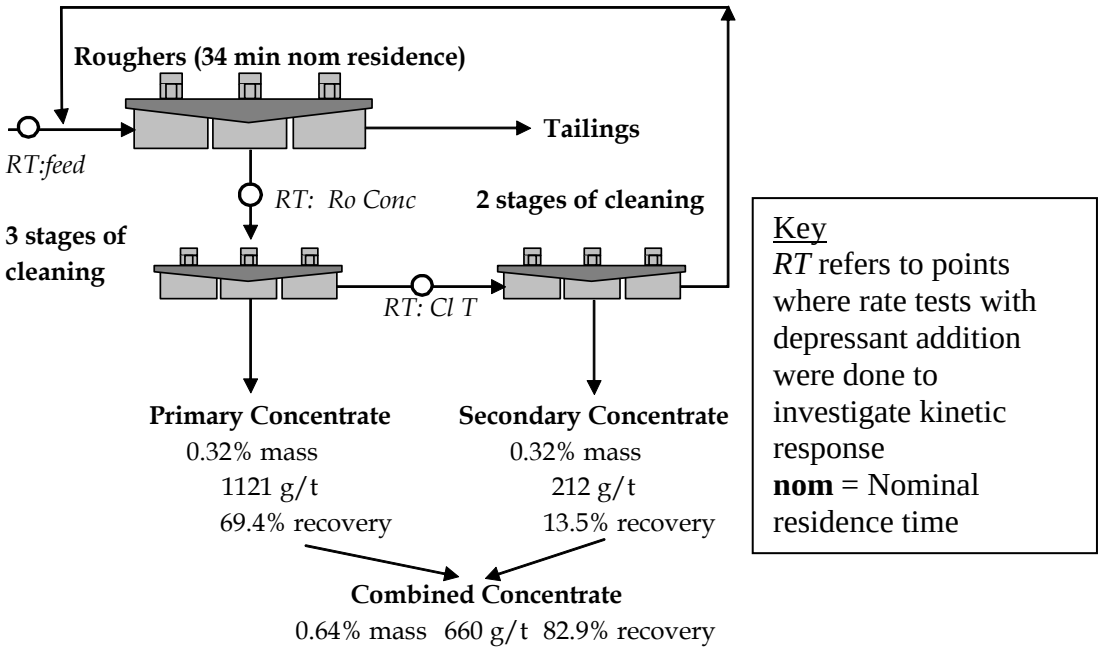


Figure 2.9: Mintek Two Concentrate Process (Hay, 2005)

2.4.4.2 Collectors

Mixed collectors have been used in optimization of sulphide flotation for many years with the synergistic effect improving recoveries significantly in some cases. In platinum flotation the use of sodium di-isobutyl dithiophosphate as a co-collector with SIBX has been widely used. Initially used for the frothing properties of dithiophosphates (DTTP), it was also found to improve recoveries (Lotter and Bradshaw, 2010) however there is limited evidence of this having been applied specifically for improving recoveries of slow floating minerals.

Different PGM processing operations each have a different reagent suite that is optimum for maximum recoveries. Wiese *et al.,* (2005), Wiese *et al.,* (2006) studied the effects of reagent suite on flotation of Merensky reef ores. They found that there appeared to be competitive adsorption and interactions taking place between collectors and depressants. As such, there is need for further investigation into the exact effect of these interactions in flotation reagent studies. The investigation of effect, if any, of collector addition at different points in the scavenger and cleaner circuits on the kinetics, in particular SFR, would be useful in this study.

2.4.4.3 Activators

Mineralogical analysis of concentrate from batch flotation tests carried out by Wiese *et al*, (2006) also indicated that the different sulphide components of the ore under study responded differently to addition of copper sulphate as an activator. Of particular interest was that pyrrhotite which is known to be slow floating had enhanced recoveries due to copper sulphate addition. In some cases and notably in the Ngezi ores processed at Zimplats, there seems to be little benefit in recovery of slow floating minerals accrued from the use of copper sulphate activator (Zimplats Internal Reports, 2005).

2.4.4.4 Staged Addition of Reagents

An industrial flotation cell contains material of a wide range of sizes and reactivity to the reagents added. As a result, the optimization of recovery of this range of particles is virtually impossible (Schubert and Bischofberger, 1979). As an example, in a cell with bimodal particle distribution the finer particles consume less collector due to the higher surface area presented whilst coarser particles require significantly more coverage (Trahar, 1989). In order to resolve this problem, split conditioning can be carried out so that the different particles can be conditioned independently. This is done by distributing reagent down the flotation bank. By distributing reagents down a bank, the slow floating particles in the tails of one cell can be transformed in the fast floating particles of the next cell (Lynch et al., 1981). This staged addition of reagents improves selectivity due to less entrainment of hydrophilic particles in the first flotation cells as they would be pulled smoother. A study on optimization of recovery of coarse particles by Bazin and Proulx, (2000) showed that split conditioning by staged addition of collector gave higher recovery with equal or less consumption of reagent. This study regarded the coarse particles as analogous to the slow floating mineral in any industrial flotation cell. The effect of staged addition of collector is shown in Figure 2.10. The flotation of fines (0-37µm) is readily promoted by the initial xanthate addition. On the other hand, recovery of medium and coarse size particles becomes significant only after the second addition point. If the entire collector were added in one location at the top of the bank, the fine particles would have probably consumed most of the collector available and floated out of the bank in the first two or three cells, leaving coarse particles (slow floating fractions) insufficiently covered by the collector in the tailings of the flotation bank (Mckee, 1979).

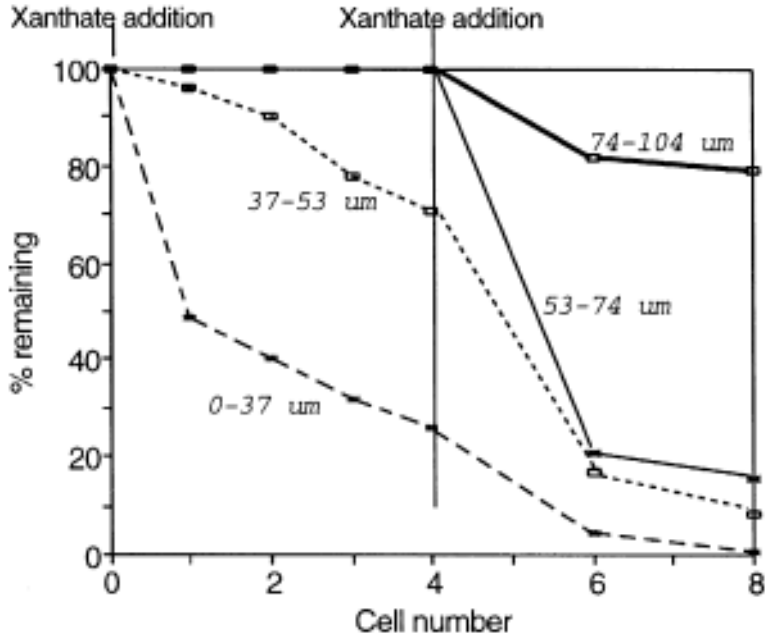


Figure 2.10 Effect of collector distribution on particles recovery (McKee et al., 1976)

2.4.5 Effect of Residence Time

Increased residence time is one of the ways to improve recovery in a flotation plant when tailings test work shows that the flotation process is incomplete at the end of the circuit. A careful look is however required because with slow floating minerals the residence time must be considered in conjunction SFR. Increasing residence time alone might not give the desired effect of improving recovery of slow floating minerals because they build up in the cleaner circuit tailings with increased residence time. What is important is to combine increased residence time with other ways of increasing SFR. This implies that residence time has to be determined according to the kinetics of a given ore.

CHAPTER 3

EXPERIMENTAL METHODS

3.1 Overview

The points selected for in-line sampling of the plant pulp were the scavenger, cleaner and high energy circuits because at these locations, according to the Zimplats flotation circuit, the bulk of the fast floating fraction would have been recovered making these points suitable to attempt the upgrading and recovery of the slower floating PGMs to the concentrate. The samples obtained from these selected streams were subjected to batch flotation rate tests. The concentrates and tailings samples obtained from the flotation rate tests were then assayed for PGM content using NiS fire assay procedure with ICP-OES finish. The results of the tests were used to calculate the flotation kinetic parameters of metal or mineral and gangue under chosen test conditions by making use of the Kinclac® Flotation Kinetics Calculator. KinCalc® allows calculation of kinetic parameters from float tests performed and enables comparison of one test, or set of test conditions against another (Eurus Mineral Consultants, n.d.)

3.2 Sampling Point Selection

3.2.1 Scavengers

After the fast floating mineral is recovered in the roughers, the little remaining fast floating mineral and the slow floating mineral gravitates to the scavenger circuit. The scavenger feed consists of rougher tails together with high energy tails and thus most of the mineral values here are slow floating. The remaining fast floating mineral is recovered in the first scavenger bank whose concentrate gravitates to the column cells. The slow floating mineral is then treated in the remainder of the scavenger circuit. Feed to each bank of the scavenger was sampled to track the movement of the slow floating minerals and any changes in their behaviour.

3.2.2 Cleaner Feed

Cleaner feed consists of the combined scavenger concentrates and together with recleaner tails. This is the other point of entry for slow floating material that has failed to be upgraded to the final concentrate in the recleaners and is recirculated back to the cleaners for another chance at upgrading.

3.2.3 High Energy Tails

The high energy circuit presents the last chance for slow floating to be upgraded to the concentrates and failure to be upgraded here means that any values remaining will be

recycled to the scavengers to start the process again. It must be noted that mineral values with low SFR will become a circulating load in the high energy tailings hence optimizing this stream offers a chance at increasing SFR and upgrading these values.

3.3 Sampling Procedures

Sampling was carried out using the sample cutters provided in the SMC flotation plant. All samples cut from a point at each interval were combined to form a composite sample that is representative of the stream in question. A sample of 1000cm^3 was collected from each stream at intervals of 30 minutes and added to the sample bucket. In total 16 samples were collected as a composite per stream every 8 hour shift. Standard operating procedure at SMC requires a composite of at least 6 samples to be taken over a 3 hour period to be considered as representative. Care was taken to obtain an adequate size of composite sample in order to enable the standard triplicate tests to be done and still maintain a contingency sample. This was done by cutting two separate samples at each sampling interval giving two composite samples for each sampling operation.

3.3.1 Sample Preparation

The composite sample obtained from the flotation plant was transferred to the flotation lab in the sample holding buckets. Due to the fine nature of solids, sedimentation occurred in transferring the samples from the plant to the flotation lab. The samples were immediately agitated using a Denver D12 impeller to obtain consistent slurry from which the smaller samples to use for the flotation rate tests were obtained.

3.4 Batch Flotation Rate Tests

The standard flotation rate test procedure was carried out to simulate the plant conditions and the results analysed to obtain the recovery/grade/mass/time relationships.

3.4.1 Equipment Utilized

- Denver D12 flotation machine
- 2.5l stainless steel flotation cells
- Stop Watch
- Sample scrappers
- Impellers and dispersers
- Wash bottles for topping up level during flotation
- Electronic balance

- Stainless steel concentrate bowls
- Electric stirrer
- Glassware for mixing and dosing reagents
- Laboratory Oven
- Paper sample bags

3.4.2 Reagents Utilized

The reagent dosages used in all the rate tests was measured in grams (of reagent) per tonne (solids). This is based on the average percent solids in the SMC streams to be sampled and translated to the mass of solids in the test cell for each test.

1. Collector used was SIBX (Sodium Isobutyl Xanthate) at 1% solution strength. 10g of SIBX powder were dissolved in 1000ml of water. Precautions: SIBX solution is unsuitable for use 48 hours after preparation as xanthate is unstable in water and oxidizes easily.

2. Depressant: CMC (Carboxyl Methyl Cellulose) at 0.1% solution strength was used. 1g CMC powder was mixed in 1000ml of water for an hour using an electric stirrer.

3. Frother: XP 200 (polypropylene glycol) at 1% solution strength. 1ml XP 200 was dissolved in 100ml of water.

3.4.3 Technical Specifications

The float tests were standardized by using the same operating parameters,

Airflow rates: As indicated on the Denver machine for a 2.5l cell.

Impeller speeds: 1200rpm

Pulling rate: Uniform Throughout as described in the flotation test procedure.

3.4.4 Flotation Test Procedure

1. Slurry was transferred to the 2.5l flotation cell and agitated using the Denver flotation machine set at 1200rpm with air inlet closed (Figure 3.1).
2. The pulp level was initially set at 25mm below the cell overflow lip. With agitation and airflow, pulp level should be about 10-15mm below the froth overflow lip as shown in Figure 3.2. For ease of movement and flow of equipment and materials was set up as shown in Fig 3.3.

3. For tests involving collector dosage, the pulp was conditioned with SIBX for 10 minutes with air inlet still closed. Depressant and frother conditioning time was 1 minute.
4. Air inlet was opened to initiate flotation.
5. Froth was removed every 15 seconds for 30 minutes using a scrapper than spans the whole float cell and maintaining a constant froth depth. There were four collections of concentrate every minute i.e. after 15, 30, 45 and 60 seconds.
6. After each second sweep of the froth, any material adhering to the paddles was washed off with water into the concentrate collection bowl. Material adhering to the sides of the cell and impeller was also occasionally washed down,
7. From a cost and sample processing point of view the minimum number of concentrates to adequately describe the recovery-time curve and the ore's flotation characteristic is four timed at 2, 6, 14 and 30 minutes. Concentrate bowls were removed at each of these intervals and replaced with fresh ones after floating for these set times to give 4 concentrates.
8. After 30 minutes the air was closed off and the machine switched off. Each test generated 5 samples for chemical analysis i.e. the 4 concentrates obtained at the set time intervals and the tails remaining in the float cell.
9. The concentrate samples, still in their concentrate bowls were marked and placed in a laboratory oven for drying and weighed periodically until there was no change in weight.
10. Concentrate bowls were removed from the oven, the concentrates (or tails) carefully scrapped off and placed in pre-weighed paper sample bags.
11. Net dry sample weight was obtained by subtracting the weight of the sample bag.
12. The samples were then sent for assay analysis.



Figure 3.1 Batch Flotation Test Setup (Denver D12 Flotation machine)

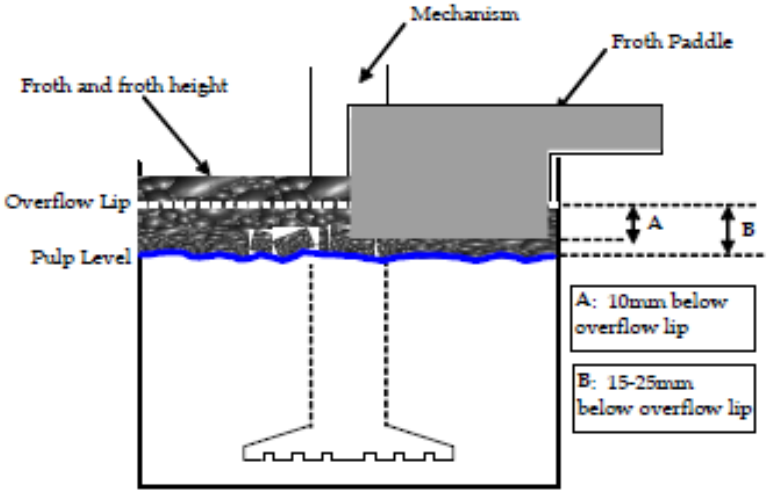


Figure 3.2: Correct Pulp and Concentrate Scrapping Level for Batch Test (Eurus Mineral Consultants, n.d.)

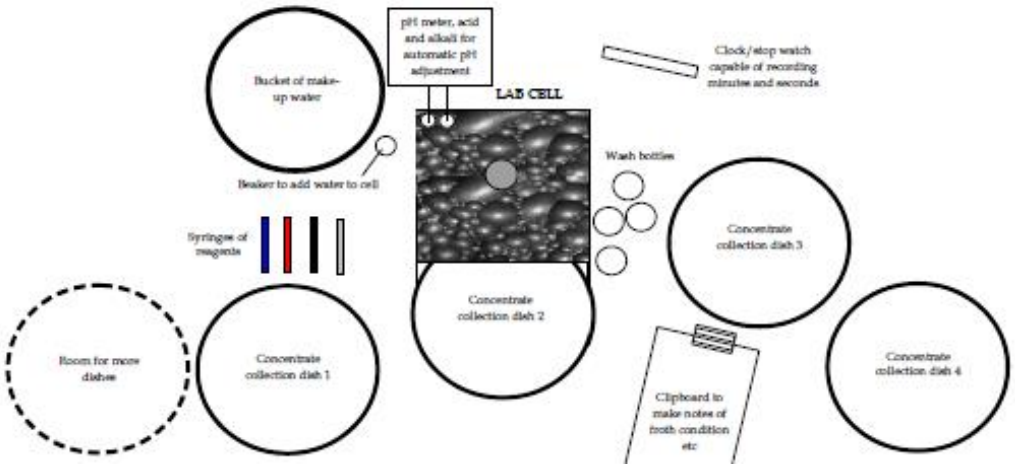


Figure 3.3: Suggested Layout for a Rate Test (Eurus Mineral Consultants, n.d.)

3.5 Determination of PGMs

The determination of the PGM content of the flotation result samples was carried out using NiS fire assay collection followed by acid leach of the NiS button and Inductively Coupled Plasma Optical Emission Spectroscopy (ICP-OES) finish. The fire assay procedure put forward by Frimpong *et al*., (1995) was used.

3.5.1 Fire Assay Procedure

1. A 10g amount of sample, 6.7g of Na_2CO_3, 13.3g of $Na_2B_4O_7$, 5g of silica and different collector masses of Ni and S in the ratio 1 part Ni to 0.66 parts S were weighed into clay crucibles and mixed thoroughly with a spatula.
2. The mixture was fused in a preheated oven at 1050°C for 75 minutes before removing the crucible and allowing it to cool.
3. The crucible was broken open and the NiS button retrieved, weighed and crushed into small chips and transferred in to a 1000ml pyrex beaker. The NiS transferred was weighed to correct for loss on crushing.
4. A 400ml portion of hydrochloric acid (HCl) of $12mol/l^{-1}$ was added. The beaker covered with a watch glass and transferred to hot plate at 150°C to and the bead dissolved for at least 3 hours (up to 24 hours).
5. The solution was allowed to cool to between 35°C and 45°C. A 2.5ml sample of Tellurium (Te) solution (2000ppm) was added. The solution was diluted with 400ml of water and 10ml of $SnCl_2$ solution was added to precipitate the Te. The solution was brought to the boil slowly for 30 minutes to coagulate the black Te precipitate formed.
6. The solution was cooled to between 35°C and 45°C and then filtered and washed with de-ionised water.
7. The filter paper was placed in a test tube and 5ml of $16mol//l^{-1}$ nitric acid (HNO_3) was added, a reflux condenser attached and the filter paper allowed to dissolve. 5ml of $12mol/l^{-1}$ HCl were added through the top of the condenser and the solution warmed to just below 100°C.
8. The solution was cooled, the inside of the condenser washed and with de-ionised water. The solution was quantitatively transferred to a propylene bottle and diluted to 100ml with de-ionised water.

3.5.2 Chemical Analysis

The solution obtained from the digestion described in the fire assay procedure was chemically determined using inductively coupled plasma-optical emission spectrometry (ICP-OES). The analysis by ICP-OES offers the unique advantage of sub-ppm detection limits, limited interference effects and simultaneous analysis. The method gives the concentrations in parts per million (ppm) of the target metals namely Pt, Pd, Rh and Au.

3.6 Use of KinCalc® to Calculate Kinetic Parameters

The results of a flotation rate test and the subsequent chemical analysis are used to estimate all the required kinetic parameters relating to Kelsall's unmodified equation using the Kincalc® Flotation Kinetics Calculator. Figure 3.4 shows the Kincalc® data entry sheet for an example of cleaner feed with 30g/t dosage of SIBX. The experimental values that are input are shown in the figure and headed as follows:

Sample: The concentrate under consideration, i.e. at either 2,6,14 or 30 minutes or the tailings for test.

Time - The time of flotation referring to the concentrate sample under consideration as above.

Mass - The dry mass in grams of the concentrate under consideration (obtained from weighing of concentrates/tailings of the rate tests).

Cu, Ni - Assays of Cu and Ni in the concentrate/tails sample under consideration (not considered in this study)

Pt, Pd, Rh, Au - Assays of each the 4 metals (in g/t) in the concentrate/tails sample under consideration (obtained from ICP-OES analysis). N.B. g/t are equivalent to ppm.

PGMs: The sum of the assays of 4 PGMs above in the concentrate/tails sample under consideration (obtained from ICP-OES analysis).

Tails (Remaining in Conc) - Mass and assays entered for the tailings sample of each test

Combined Conc - Total calculated mass and assay for the combined concentrate for the entire test.

Once all the data is input, Kincalc® calculates the kinetic parameters relating to Kelsall's unmodified equation by data fitting using linear regression. The parameters that are generated (output) from the software are used for kinetics investigations. These parameters are:

I_{PF} = fast floating fraction of PGMs

k_{PF} = fast floating rate of PGMs

k_{PS} = slow floating rate of PGMs

I_{GF} = fast floating fraction of gangue

k_{GF} = fast floating rate of gangue

k_{GS} = slow floating rate of gangue

The values obtained above were used to calculate:

SFR = (k_{PS}/ k_{GS}) slow floating ratio i.e. the slow floating flotation rate of PGMs relative to gangue

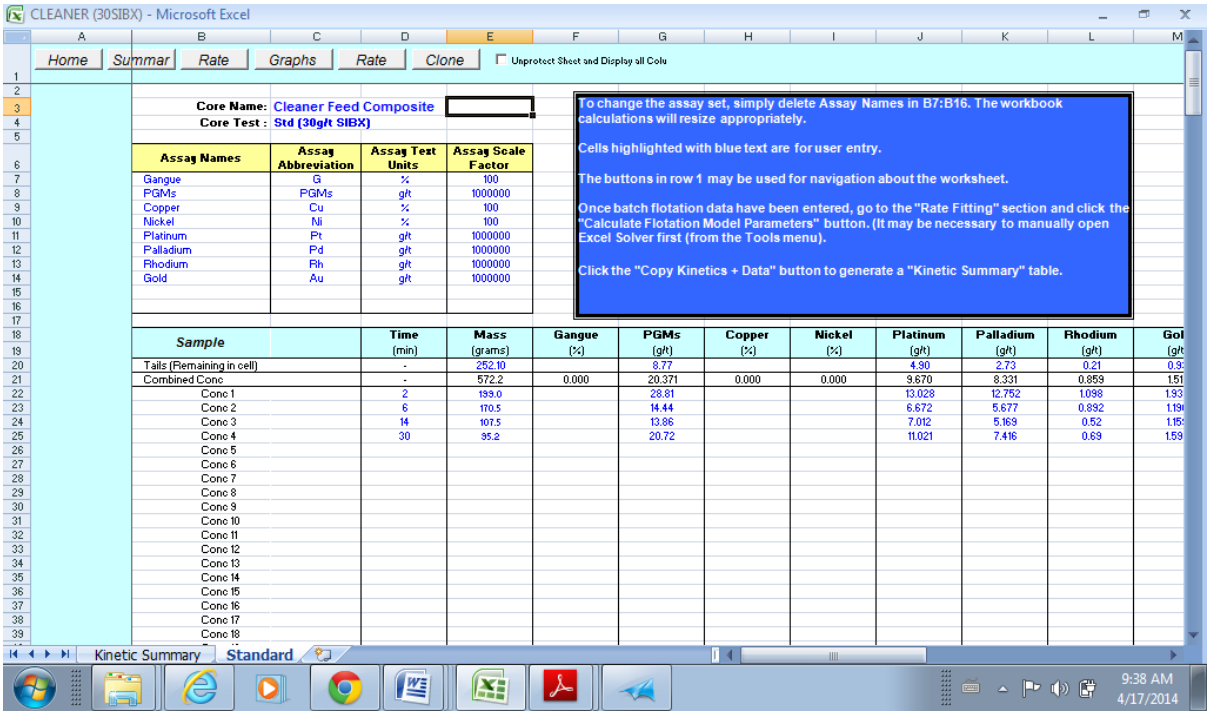


Figure 3.4: Kincalc® Data Entry Page

3.7 Sieve Analysis (High Energy Cell Tailings)

3.7.1 Equipment Utilized

- Wet sieving set used with mesh sizes (in µm) 106,75, 53, 38
- Wet sieving shaker (Figure 3.5)
- Stainless steel concentrate bowls
- Laboratory oven
- Electronic balance

3.7.2 Technical Specifications

Amplitude: 15mm

3.7.3 Procedure

1. Slurry was introduced to the uppermost sieve.
2. The set was clamped, amplitude set and the shaker started with the water inlet opened (Figure 3.5).
3. The sieving was continued until the water outlet produced clear water.
4. The solids were washed into the sample collecting bowls and placed in a drying oven at 65 ºC and weighed periodically until there was no change in weight.
5. Concentrate bowls were removed from the oven, the solids carefully scrapped off and placed in pre-weighed paper sample bags.
6. Net dry sample weight was obtained by subtracting the weight of the sample bag.
7. The samples were then sent for assay analysis.

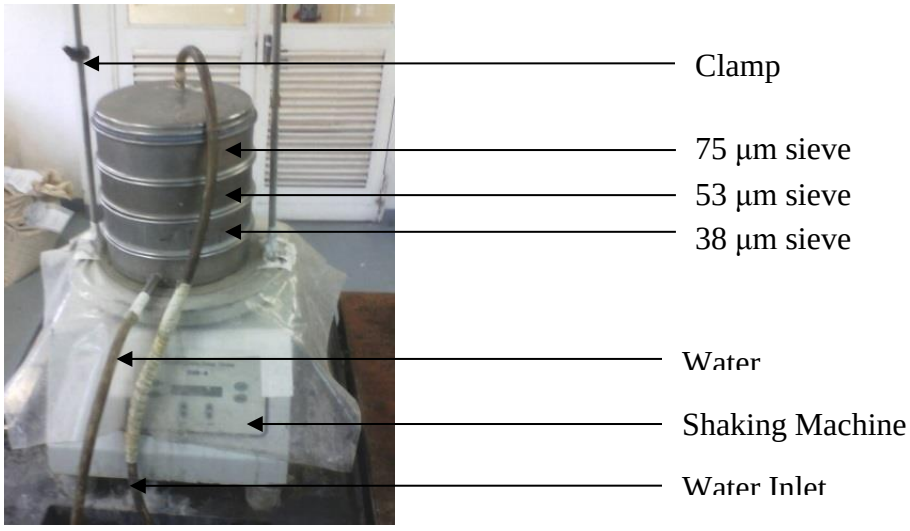


Figure 3.5: Wet Sieve Shaking Set

CHAPTER 4

RESULTS

4.1 As Received Scavenger Feed Tests

Results of calculated SFR from the floatability tests across the scavenger banks are shown in Table 4.1 and figure 4.1. Laboratory rate tests were carried out on "as received" feed to each of the 6 scavenger banks (Scav 1-6). The scavengers are arranged in series from 1 to 6 as shown in Figure 1.1. "As received" refers to the sampled plant slurry without any reagent addition. The kinetic parameters relating to Kelsall's unmodified equation were generated using Kincalc®. The main parameter targeted for analysis was the slow floating ratio (SFR). SFR is the ratio of the slow floating rate constant for PGMs to the slow floating rate constant for gangue, (k_{PS}/k_{GS}). The fractions of fast floating gangue (I_{GF}) and PGMs (I_{PF}) were also generated together with their rate constants (k_{GF} and k_{PF}). For the purposes of this study the parameters analysed were those related to slow floating fractions of mineral and gangue.

Table 4.1: Summary of Kelsall Parameters for "As Received" Scavenger Bank Tests

	I_{GF}	k_{GF}	k_{GS}	I_{PF}	k_{PF}	k_{PS}	SFR
Scav 1	0.1190	0.0700	0.0019	0.6248	0.2958	0.0176	9.32
Scav 2	0.1078	0.0756	0.0014	0.2570	0.2324	0.0175	12.95
Scav 3	0.1224	0.0247	0.0033	0.2571	0.0471	0.0404	12.28
Scav 4	0.1224	0.0246	0.0033	0.1100	0.3851	0.0404	12.19
Scav 5	0.2014	0.1815	0.0056	0.5449	0.5014	0.0348	6.21
Scav 6	0.0311	5.3884	0.0197	0.3078	0.7088	0.0618	3.13

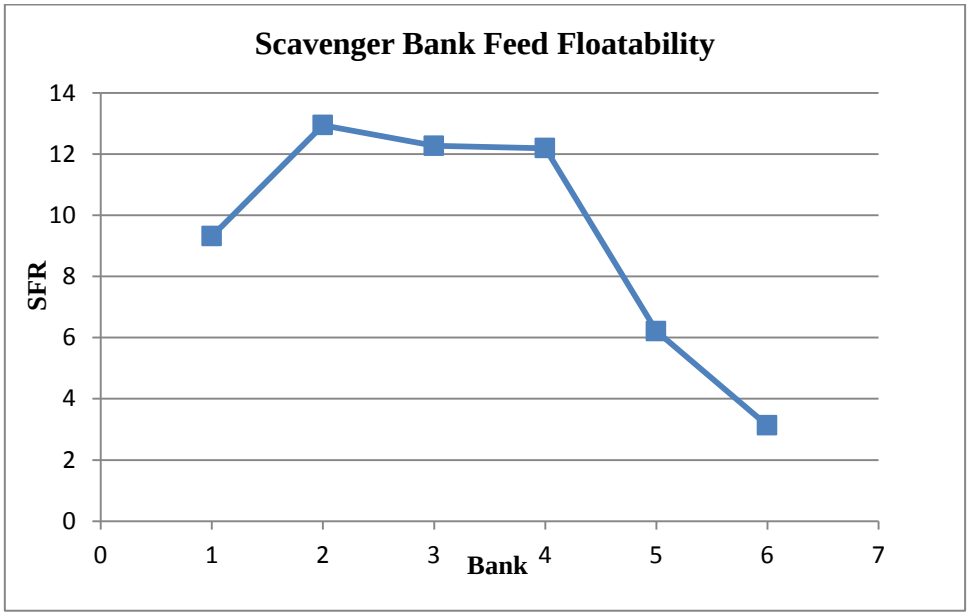


Figure 4.1: Scavenger Floatability

4.1.1 Flotation Behaviour of Scavenger Feed with no Reagent Addition

The scavenger banks are arranged in series from 1-6 (Refer to Figure 1.1) and the general trend is that the floatability of the PGMs in the slurry diminishes across the scavengers, the tailings from the final scavenger bank being final tails of the flotation circuit. Figure 4.1 shows that SFR is low in the feed to the first bank, increases to the maximum in the second and starts to fall in the third and fourth banks after which it falls rapidly. An explanation of this trend can be offered by the nature of the scavenger circuit feed. Since scavenger feed is composed of the tailings of the rougher and high energy cells, it contains the fastest floating mineral which remains unfloated from the roughers and the slowest floating material recirculated from the high energy cell tailings. The first bank of the scavenger thus contains an abundance of faster floating mineral competing to attach to the available air bubbles with the slow floating mineral. In the first scavenger bank the slow floating mineral thus stays in the slurry as the faster floating fractions are recovered and the slow floating ratio is low. As the fast PGM fraction diminishes in the second and third banks, SFR increases, however it falls in the last three banks (banks 4-6) as the reagents get used up and only the slowest floating mineral remains unrecovered. Due to the observed maximum of SFR in the feed to the second bank, for this study, feed to the third bank of the scavenger was selected as a reagent addition station as this is the point where SFR starts to fall and a boost is required to improve recovery of mineral to the concentrate (froth) by further reagent addition.

4.2 Scavenger Bank 3 Test Results

The kinetic results obtained for tests carried out on feed to the third scavenger bank are shown in Table 4.2. The tests were carried out with addition of depressant and collector and compared with the results of the tests with "as received" feed. The reagents used for the tests were carboxymethyl cellulose (CMC) brand named Finnfix 300 as a depressant and sodium isobutyl xanthate (SIBX) as a collector. All tests were conducted in triplicate.

Table 4.2 : Summary of Kelsall Parameters for Scavenger Bank 3 Feed

	IGF	k_{GF}	k_{GS}	I_{PF}	k_{PF}	k_{PS}	SFR
As Received	0.1224	0.0247	0.0033	0.2571	0.0471	0.0404	12.27
25 g/t Finnfix	0.1205	0.0507	0.0019	0.2158	0.1839	0.0422	21.64
50 g/t Finnfix	0.1192	0.0661	0.0023	0.3233	0.1617	0.0452	20.07
100 g/t Finnfix	0.1225	0.0282	0.0023	0.4802	0.0744	0.0268	11.69
30 g/t SIBX	0.1929	0.3807	0.0070	0.1231	5.7570	0.0690	9.91
60 g/t SIBX	0.2120	0.3909	0.0063	0.2265	5.7513	0.0806	12.78

4.2.1 Flotation of As Received Scavenger Bank 3 Feed

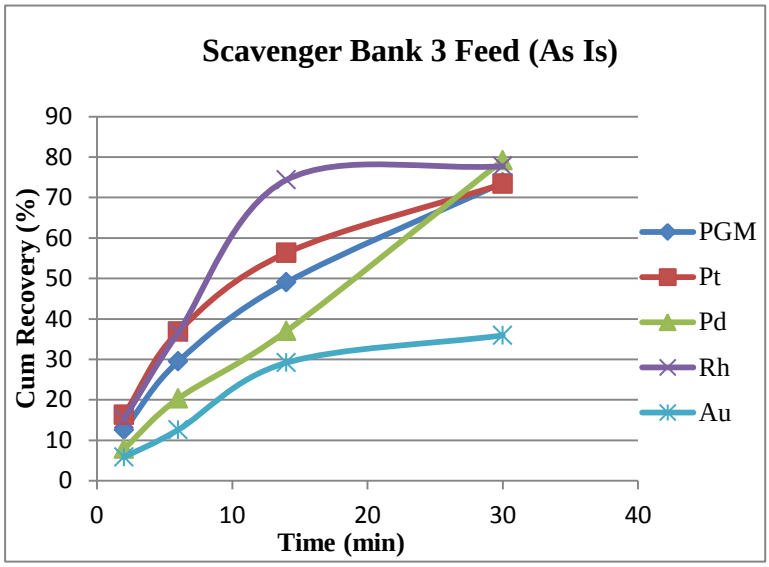


Figure 4.2: Time for Flotation Scavenger

[1]Recovery-Relationship of As Received Bank 3 Feed

[1] Note that that all Grade – Time and Grade – Recovery graphs in this section and the rest of the chapter are based on aggregate PGM content. For Recovery – Time Graphs, PGMs also refers to the aggregate content.

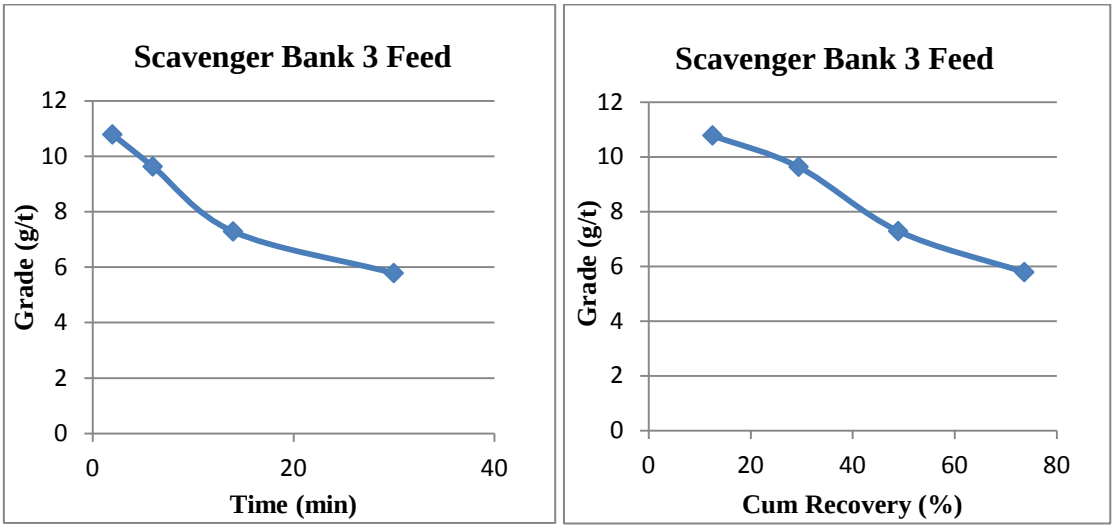


Figure 4.3: (a) Grade Recovery and (b) Grade-Time Relationships for Flotation of As Received Scavenger Bank 3 Feed (Based on aggregate PGMs)

The recovery time curves (Figure 4.2) indicate a fairly uniform rate of recovery throughout the duration of the test. This uniformity is broken for Rh which is distinctly faster floating and is recovered almost entirely within the first 10 minutes of the test. This means that most of the Rh was concentrated in the initial stages of the test (up to 14 minutes). The explanation would be that the fractions associated with Rh exhibit a higher level of liberation and hence float much faster than the rest of the material in the cell (Hay, 2005). The maximum recovery for PGMs obtained in this test was 73.7% which leaves potential for further recovery down the scavenger circuit with increased residence time. The grade time curve (Figure 4.3(a)) also indicates uniform recovery to concentrate with time until about 15 minutes in to the test where the rate of uptake of gangue decreases as overall flotation rate decreases resulting in grade falling less sharply. The grade recovery curve (Figure 4.3(b)) shows an expected decrease in grade with increasing recovery. The total grade differential between the first concentrate and the final concentrate at the end of the batch test is 5g/t. This is a result of small differences in the PGM assay obtained across the whole test because at this stage in the circuit most of the values float at the same slow rate.

4.2.2 Flotation of Scavenger Bank 3 Feed with Depressant Addition

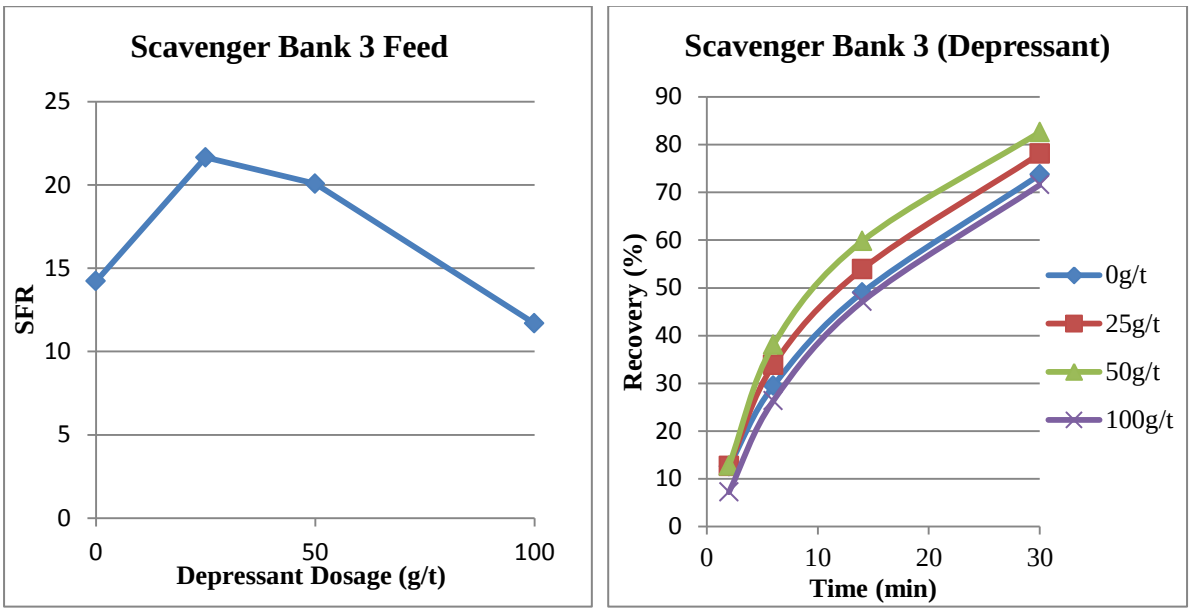


Figure 4.4: Variation of (a) SFR and (b) Recovery-Time Relationship with Depressant Dosage for Scavenger Bank 3 Feed

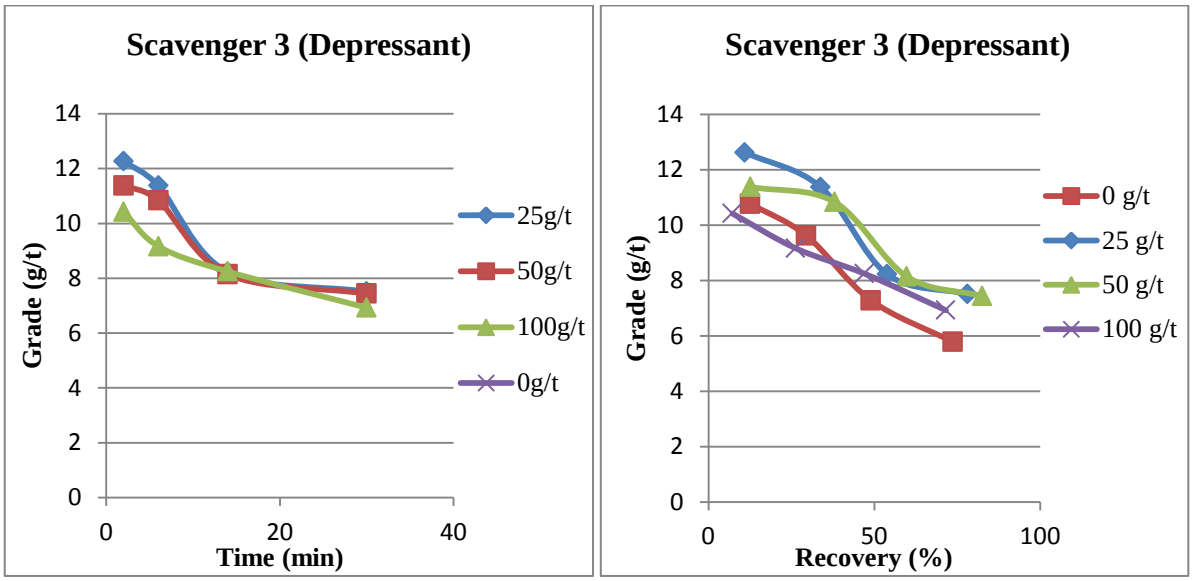


Figure 4.5: Variation of (a) Grade-Time and (b) Grade-Recovery Relationship with Depressant Dosage for Scavenger Bank 3 Feed

4.2.2.1 Effect of Depressant Addition

Figure 4.4(a) shows the variation of SFR with increasing depressant addition to the scavenger bank 3 feed. As can be seen the slow floating ratio increases initially with depressant addition to a high value of 21.64 at 25g/t depressant dosage. Further increase in depressant dosage decreases the SFR slightly to 20.07 at 50g/t and even further depressant addition reduces SFR to values below those obtained without depressant addition. This means

that the slow floating rate constant for mineral (k_{PS}) increases as depressant is added and falls significantly at higher depressant dosages. This is visible from inspection of Table 4.2. The observed similarity between the SFR and k_{PS} trends is broken at 50 g/t dosage of depressant. Whilst SFR decreases slightly with addition of 50g/t depressant, k_{PS} actually increases. The decrease in SFR at this dosage is attributable to an increase in the slow floating rate constant for gangue (k_{GS}). At this dosage, the depressant is more abundant in the slurry and allows the value slow floating mineral to float at a faster rate by inhibiting gangue flotation. At a dosage of 100g/t depressant, k_{PS} falls significantly whilst k_{GS} remains constant. At this point there is excess depressant in the slurry and it inhibits the flotation of both values and gangue. It can be seen that depressant addition in small quantities improves the selectivity between slow floating mineral and slow floating gangue. However with increased addition, depressant exhibits a secondary effect of inhibiting recovery of the value sulphide mineral as well (Bradshaw et al., 2005). As a result SFR is diminished. Depressant acts to inhibit the flotation of gangue by rendering the gangue minerals hydrophilic and thus unnameable to attachment to air bubbles. However, with excessive addition, a highly inactive atmosphere is created in the slurry due to a higher proportion of the mineral surfaces in the slurry that are coated with depressant. As a result, the value mineral surfaces also become unavailable for attachment to the air bubbles. Additionally, the residual collector at this point cannot act on the value mineral surfaces due to the excess depressant acting on them. The result is very little flotation takes place and only a small amount of material floats to the concentrate.

The grade-recovery-time comparison for the depressant addition tests are shown in Figures 4.4(b) and 4.5. The results obtained for tests done with 25g/t and 50g/t depressant show much better grades at any given recovery than the tests done with 0g/t and 100g/t. The final recovery is highest for addition of 50g/t depressant and lowest for addition of 100g/t depressant. This is due to the effect of the high depressant dose that allows only a small amount of material to float at 100g/t of depressant. The 50g/t depressant dosage is the most ideal in the tested range as it allows the highest recovery of the value minerals compared to the other dosages (Figure 4.4 (b)). Figure 4.5 (a) shows higher grades at any given recovery for the 50g/t over the 25g/t dose at all but the initial stages of the tests at low recovery (up to 6 minutes). This shows that the effect of depressant is limited at 25g/t as not enough gangue is inhibited from floating.

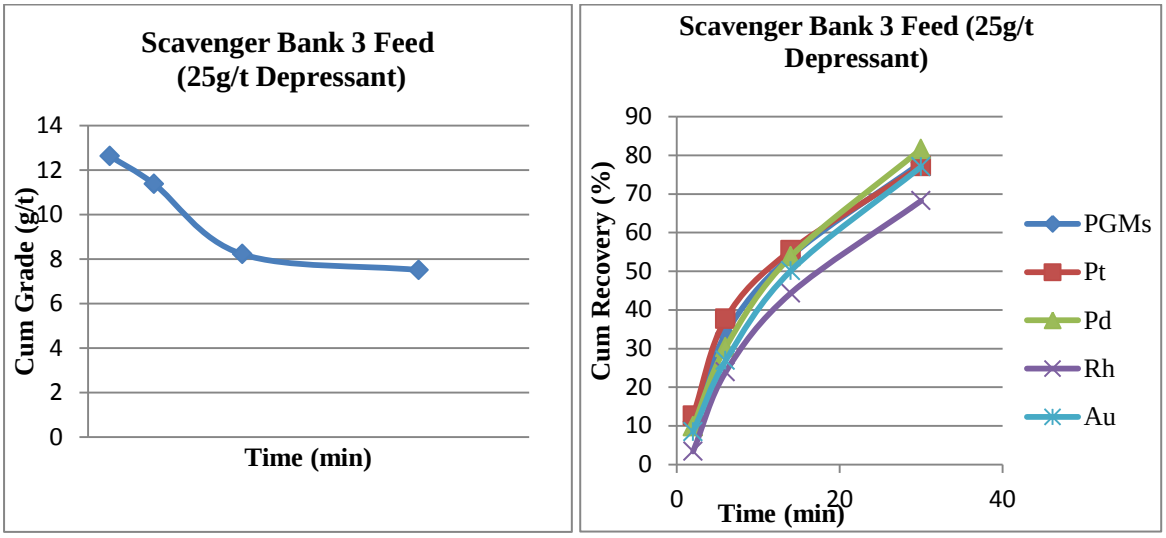


Figure 4.6: (a) Grade-Time and (b) Recovery-Time Relationships for Flotation of Scavenger Bank 3 Feed with 25g/t Depressant

The grade-time curve for the tests caried out with depressant dosage of 25g/t (Figure 4.6 (a)) shows a distinct drop in the grade with recovery. The drop in grade with time is initially sharp up to about 10 minutes into the test and levels out in the later stages. This suggests that this depressant dosage causes the mineral values to be preferentially floated in the early stages of the test at 2 and 6 minutes as depressant acts on the gangue. Between 6 and 15 minutes the grade fell sharply due to depletion of mineral values in the float cell that resulted in a proportionally higher uptake of gangue into the concentrate. From 15 minutes onwards, the grade fell less sharply as there was proportional uptake of values and gangue into the concentrate resulting from the higher poprtion of less floatable material in the cell due to depletion of floatable material by flotation in the earlier stages of the test. This results in less proprtionately less concentrate being recovered per unit time and thus smaller effect on the overall grade.The recovery time profile (Figure 4.6 (b)) shows the same trend with the rate of recovery decreasing for all the metals at around 6 minutes into the test. After this point the recovery rate for the value is slower implying a higher uptake of gangue into the concentrate..

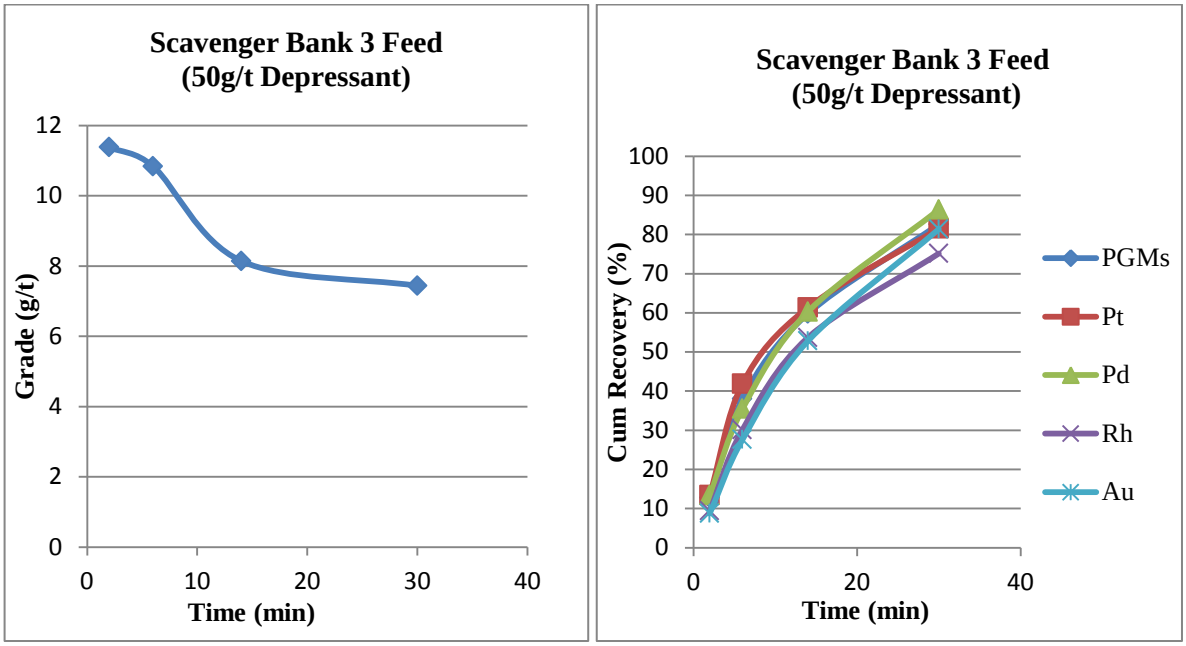


Figure 4.7: (a) Grade-Time and (b) Recovery-Time Relationship for Flotation of Scavenger Bank 3 Feed with 50g/t Depressant

The grade-recovery-time behaviour for the tests done with addition of 50 g/t of depressant (Fig 4.7) is very similar to that for 25g/t depressant addition. Both the grade recovery and recovery time curve show similar trends. There are very small differences in the final recovery and final grade and hence the behaviour at the two different dosages is comparable.

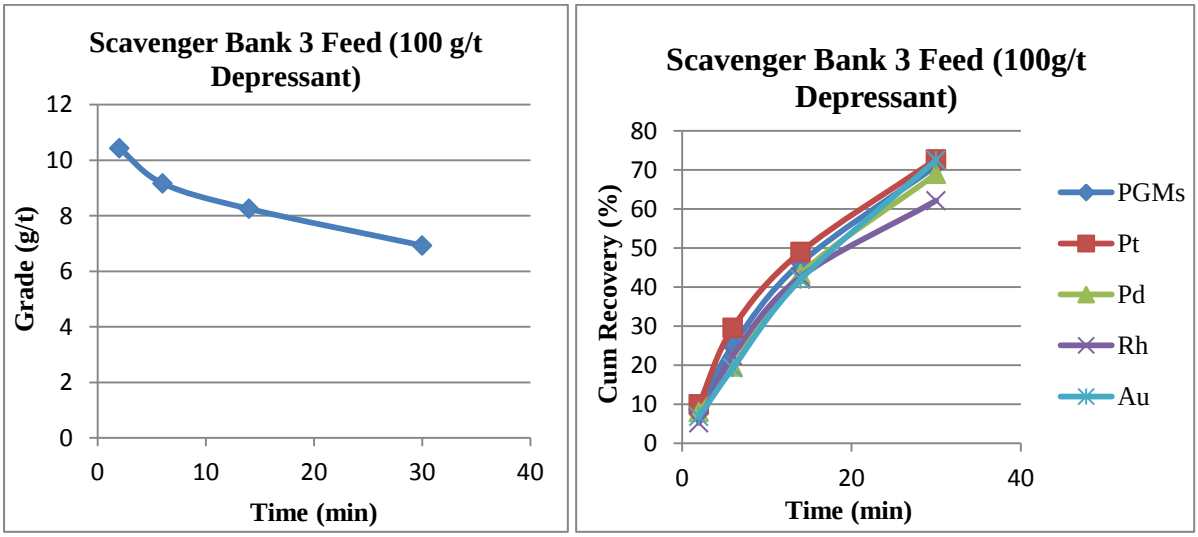


Figure 4.8: (a) Grade-Time and (b) Recovery-Time Relationship for Flotation of Scavenger Bank 3 Feed with 100g/t Depressant.

The tests carried out with 100g/t depressant (Figure 4.8) also show more or less the same behaviour as the preceding depressant tests. The major difference is that the final recovery at 30 minutes falls to 71.6 % compared to 82% for the 50g/t dosage whilst the grade time

relationship falls steeply in the first 6 minutes of the test after which it falls less steeply. The fall in recovery is a result of the excessive depressant inhibiting flotation of values in the slurry. The grade time behaviour is a result of the recovery of the fastest floating material in the first 6 minutes after which flotation slows down as the high depressant dosage inhibits flotation of the bulk of the material in the cell. This results in less material being floated to the concentrate per unit time. As a result, the effect on the grade of the concentrate being recovered is lower and grade falls less steeply. The recovery time profiles show a slow down in recovery rate after 6 minutes. At this point, the rate of recovery decreases with time due to the effect of the high depressant dosage on the less floatable slurry remaining in the cell.

4.2.3 Flotation of Scavenger Bank 3 Feed with Collector Addition

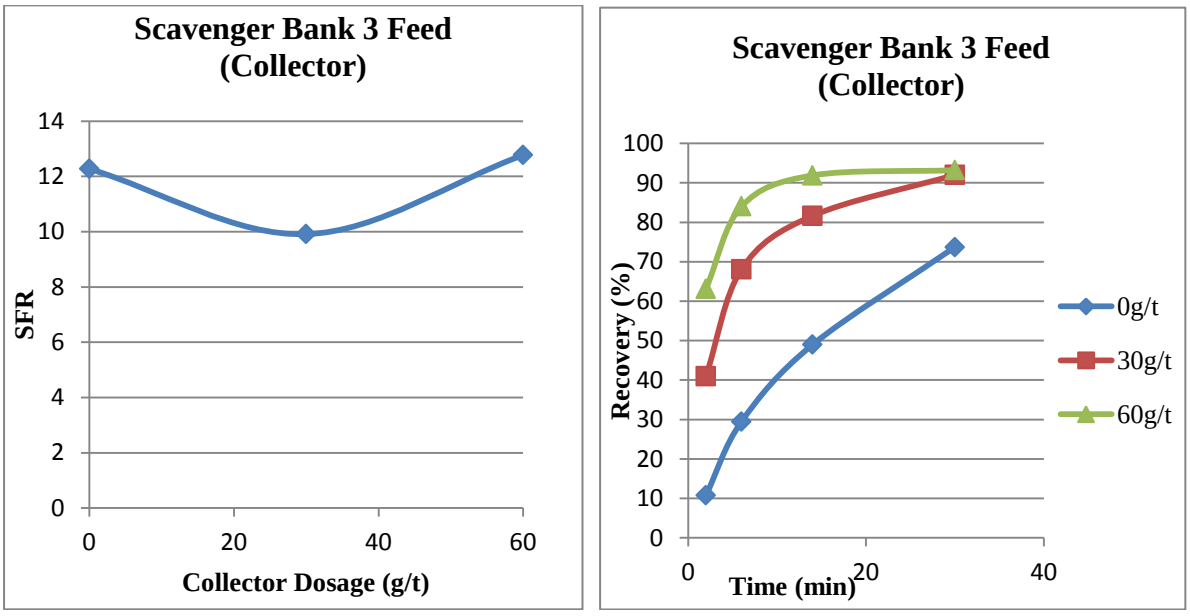


Figure 4.9 Variation of (a) SFR and (b) Recovery-Time relationship with collector dosage for Scavenger Bank 3 Feed

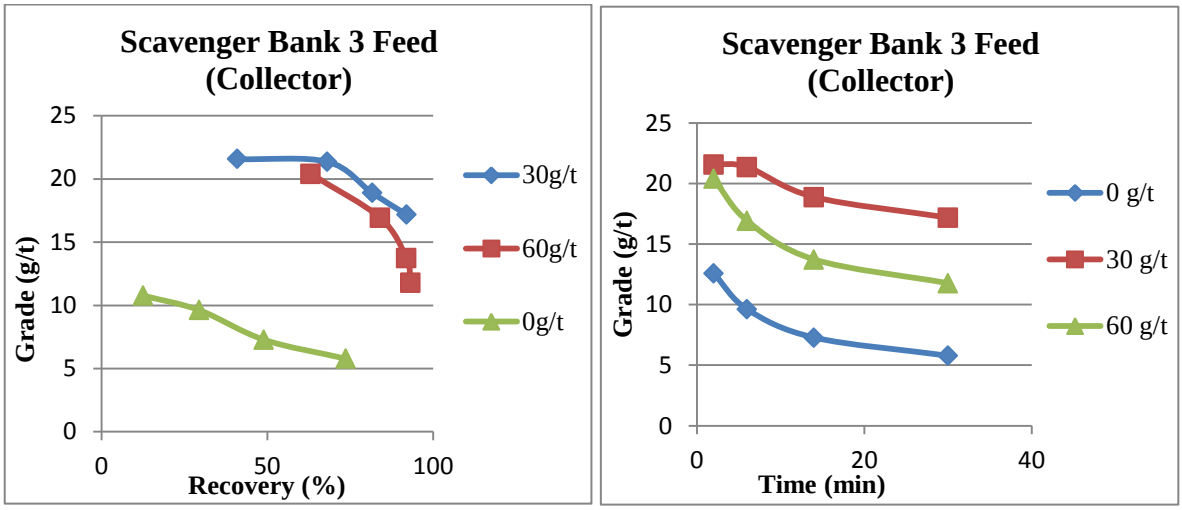


Figure 4.10: (a) Grade-Recovery and (b) Grade-Time Relationships with Collector Dosage for Scavenger Bank 3 Feed

The rate tests done with collector addition show a decrease in SFR at 30g/t collector addition as compared to the values obtained with "as received" scavenger bank 3 feed (Figure 4.9 (a)). At a dosage of 30g/t collector, the increase in the rate of flotation of gangue is higher than that of PGMs causing a drop in SFR. On addition of 60g/t collector, the SFR increases to slightly above that obtained from flotation without addition of reagent. Table 4.2 shows the calculated values of k_{PS}, k_{GS} and SFR for all the tests conducted on Scavenger Bank 3 Feed. As expected k_{PS} increases with additional collector, however k_{GS} also increases. The increase in the rate of slow floating gangue (k_{GS}) results in the low SFR values. Also, k_{PS} and SFR increase with collector dosage however due to the corresponding increase in k_{GS} the values of SFR obtained do not represent an improvement. The increase in k_{GS} is attributable to rendering of hydrophobicity to a larger proportion of the particles within the float cell. As a result, there is a higher amount of gangue that competes with the slow floating mineral for recovery to the concentrate. Although k_{GS} falls with addition of 60g/t collector the much higher value of k_{PS} gives a higher SFR. The jump in k_{PS} and k_{GS} values observed with collector addition infer faster kinetics for the mineral values and a corresponding increase in slow floating gangue kinetics. The difference in SFR between the two collector dosages is due to the higher recovery observed with addition of 60g/t SIBX. The higher collector dosage results in a higher proportion of mineral achieving hydrophobicity and being recovered, however there is also a higher amount of gangue recovered to the concentrate.

The recovery-time graphs (Figure 4.9(b)) show that collector addition increased total recovery to a certain level above which the increment in recovery starts to slow down. Whilst the dosage of 60g/t of collector initially gives a faster flotation response, the final recovery obtained at the end of the test is comparable to that obtained with 30g/t. This is because excessive collector has an adverse effect on the recovery of the valuable minerals due to the development of multi-layers on the mineral particles, which reduce the proportion of hydrocarbon radicals oriented into the slurry and thus available for attachment to the air bubbles in order to be floated out as concentrate (Wills, 2006). The grade recovery curves (Figure 4.10(a)) show a typical grade recovery relationship. The curves show high recoveries in the early stages of the tests and this is explained by the high amount of the value containing mass pulled to the concentrate even after the first concentrate was removed. As the tests wore on, there was more gangue pulled to the concentrate due to the high mass pull and since the most of the values were pulled in the initial stages of the test the grade fell more steeply in the latter stages. Notably, the collector addition gives better grade recovery relationships than the as received test.

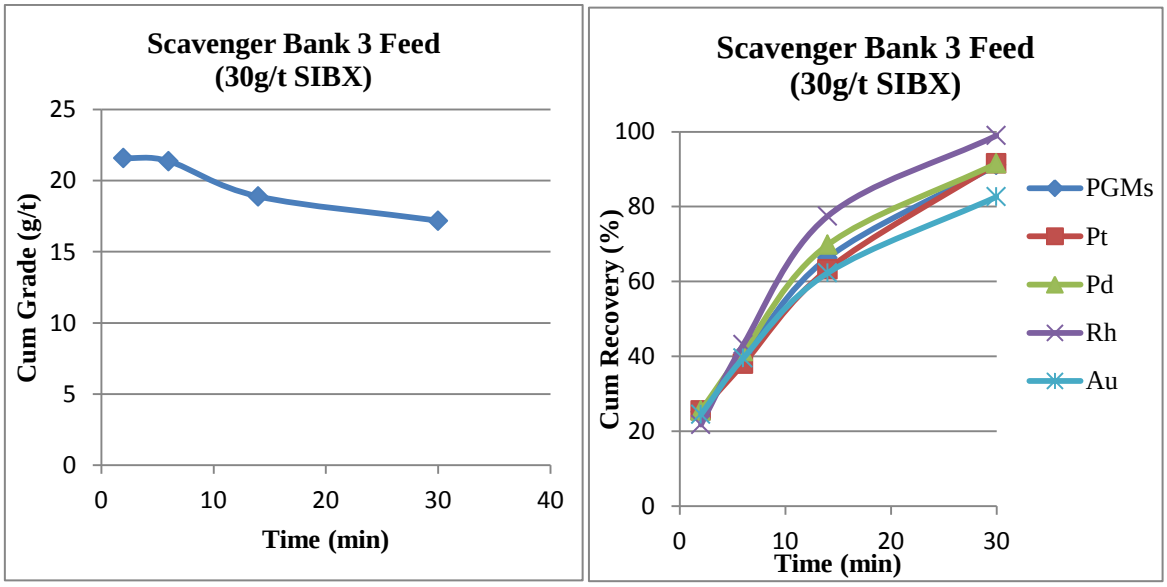


Figure 4.11: (a) Grade-Time and (b) Recovery-Time Relationships for Flotation of Scavenger Bank 3 Feed with 30g/t Collector

The flotation of Scavenger Bank 3 feed with 30g/t collector addition gave the grade-time relationship shown in Figure 4.11(a). In the first stages of the test (up to around 6 minutes) grade fell marginally with time. This suggests that a high proportion of the value mineral was preferentially recovered in the initial stages of the test. As the test wore on and the bulk of the floatable values had been recovered, the grade started to fall more sharply with time as more

gangue material was recovered to the concentrate. The time recovery profile for the test is shown in figure 4.11(b). The profiles show a fast rate of recovery with the first concentrate (after 2 minutes) having grades of over 20% for all the metals. The increased rate of recovery observed is due the hydrophobicity imparted to a higher proportion of the value mineral particles in the slurry by the added collector (Mpongo & Siame, 2006). This allows the particles to be more readily attached to the air bubbles and floated out as concentrate early in the test.

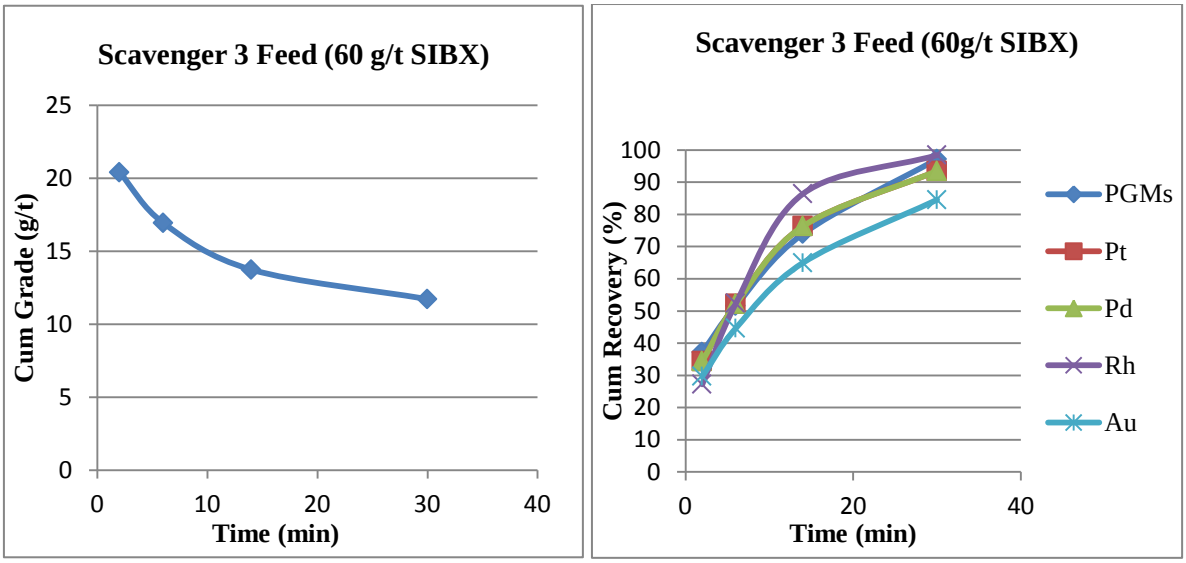


Figure 4.12: (a) Grade-Time and (b) Recovery-Time Curve for Flotation of Scavenger Bank 3 Feed with 60g/t Collector

The significant finding from the grade versus time curve for Scavenger Bank 3 feed with addition of 60g/t collector addition is the grade differential of 8.69g/t (Figure 4.12(a)). This grade differential is significantly higher than that realised with addition of 30g/t collector (4.4g/t). This suggests that the addition of the extra collector causes a marked drop in the grade due to the higher collector dosage rendering extra hydrophobicity to a larger proportion of gangue particles in the float cell.This results in a higher amount of gangue material being pulled to the concentrate thereby lowering grade. The recovery time curves (Figure 4.12(b) all exhibit a similar fairly similar profile. The first concentrates (at 2 minutes) represent a significantly high recovery (greater than 20%) showing a degree of fast floating imparted to the slurry by collector dosage.This is due to the high mass pull that results in a high amount of material being quickly recovered to concentrate in the early stages of the test. As such a proportionally higher amount of values is recovered to the concentrate. The high mass pull is explained in section 4.2.3.3. The rate of recovery slows down around 10 minutes into the test

as the amount of values in the slurry diminishes due to recovery. Collector concentration in the slurry also dimimishes as flotation proceeds. This is due to collector being floated out with the concentrate and this also contributes to the slow down in recovery.The PGMs recovery time profile for 30g/t collector shows a the slow down in recovery rate earlier in the test than with 60g/t collector. This shows that the lower dosage of collector diminishes due to flotation earlier.

4.2.3.1 Mass Pull

Table 4.3: Mass Recovery for Scavenger 3 Tests

	As Is	25g/t Depressant	50g/t Depressant	100g/t Depressant	30g/t Collector	60g/t Collector
Time (min)	Mass Rec (%)	Mass Rec (%)	Mass Rec (%)	Mass Rec (%)	Mass Rec (%)	Mass Rec (%)
2	1.4	1.3	1.7	1.0	13.1	14.4
6	3.7	4.4	5.4	3.4	21.9	23.1
14	8.2	9.7	11.4	7.6	29.7	31.2
30	15.8	15.4	17.2	13.8	36.8	37.1
Conc (%)	15.8	15.4	17.2	13.8	36.8	37.1
Tails (%)	84.2	84.6	82.8	86.2	63.2	62.9

Table 4.3 shows the mass recovery data for the rate tests done on the scavenger bank 3 feed. It is observed that the collector test runs generally have a much higher mass pull than the tests done with both as received and addition of depressant for all concentrates collected. The total mass recovered to the concentrate is 36.8% and 37.1% for the 30g/t and 60g/t collector dosage respectively whilst for the depressant and as received test the mass recovered to the concentrate ranges between 13.8% and 17.2%. The collector tests more than doubled the mass yield as compared to “as received” and depressant tests. This is a result of the excess collector causing increased froth viscosity and thereby increasing the amount of fines recovered by entrainment (Drzymala et al., 2005). At 30g/t SIBX the concentrate grade does not fall significantly even though mass pull is high indicating that the flotation process pulls a high amount of the values to the concentrate. On the other hand the mass pulled with the higher collector dose of 60g/t gives a concentrate with a diminished grade. This means the higher collector dose is excessive and results in higher entrainment of fine gangue to the concentrate due to the increase in the froth viscosity. It is of interest to note that depressant

addition also initially increases mass pull by increasing selectivity between values and gangue however with further depressant addition the floatability of both gangue and values is suppressed and the mass pull falls rapidly.

4.3 Cleaner Feed Test Results

Cleaner feed at SMC is a mixture of scavenger banks 2 to 6 concentrates mixed with tailings from the recleaners. Because the cleaner tailings are further processed in the high energy circuit, rate tests were carried out on the cleaner feed to investigate the floatability drop between the cleaners and high energy cells. The following tests were carried out on the cleaner feed

1. Standard rate tests were carried out on "as received" cleaner feed.
2. Standard rate tests with depressant addition were also carried out to investigate if any benefit accrues from depressant addition to the cleaners.
3. Tests were also done with collector addition.

The kinetic results of the tests carried on cleaner feed are summarized in Table 4.4. The results show that generally cleaner feed has a very low slow floating ratio. This is expected because the slowest floating material is treated in this part of the circuit.

Table 4.4: Summary of Kelsall Parameters for Cleaner Feed Flotation

	I_{GF}	k_{GF}	k_{GS}	I_{PF}	k_{PF}	k_{PS}	SFR
As Is	0.3274	0.1358	0.0184	0.4312	0.3148	0.0347	1.88
30g/t SIBX	0.4161	0.3378	0.0190	0.4903	0.7287	0.0383	2.01
50g/t Finnfix	0.0099	0.0950	0.0309	0.2057	0.1935	0.0375	1.21
100g/t Finnfix	0.0049	0.1326	0.0185	0.1206	0.3878	0.0290	1.57

4.3.1 Effect of Depressant Addition

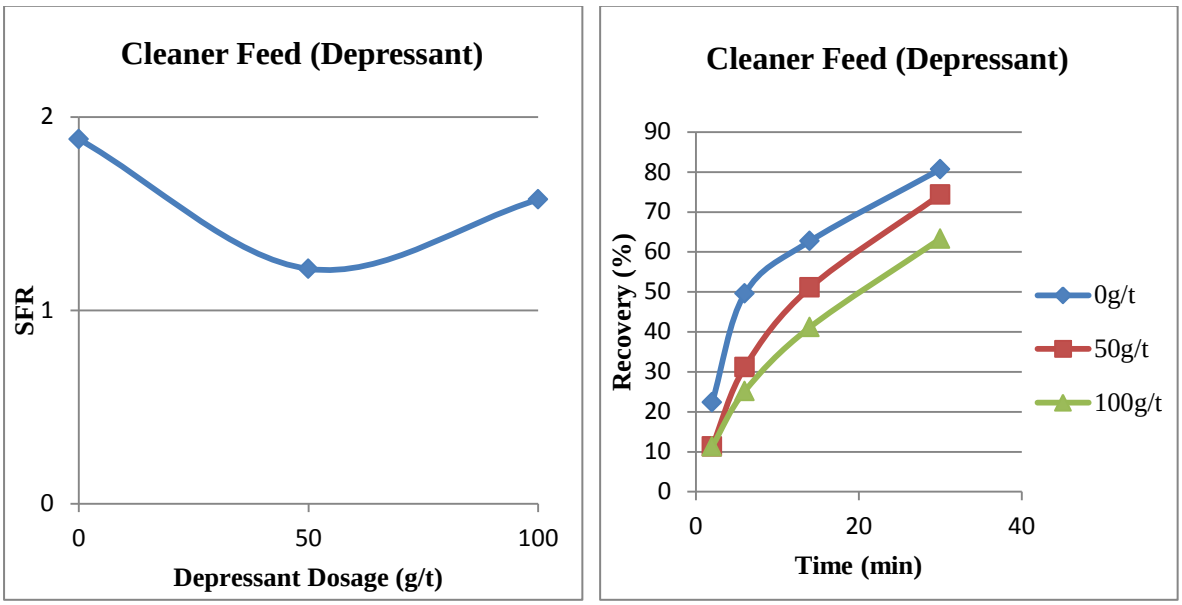


Figure 4.13: Variation of (a) SFR and (b) Recovery-Time relationship with depressant dosage for Cleaner Feed Flotation

Addition of depressant to the cleaner feed does not appear to have a significant effect on SFR as shown in Figure 4.13(a). The values of SFR vary only slightly for all the tests. The overall effect of depressant does not seem to be positive as in both cases SFR is lower than the flotation of as received cleaner feed. This is due to the fact that the dosage of depressant inhibits the flotation of both the gangue and values. Table 4.4 shows that at 100g/t dosage k_{GS} (0.185) falls significantly from the value at 50g/t (0.309) however k_{PS} does not fall proportionally and hence there is an increase in SFR. This shows that the high depressant dosage inhibits flotation of both values and gangue although more significantly on the gangue.

4.3.1.1 Analysis of Grade-Recovery-Time Relationships

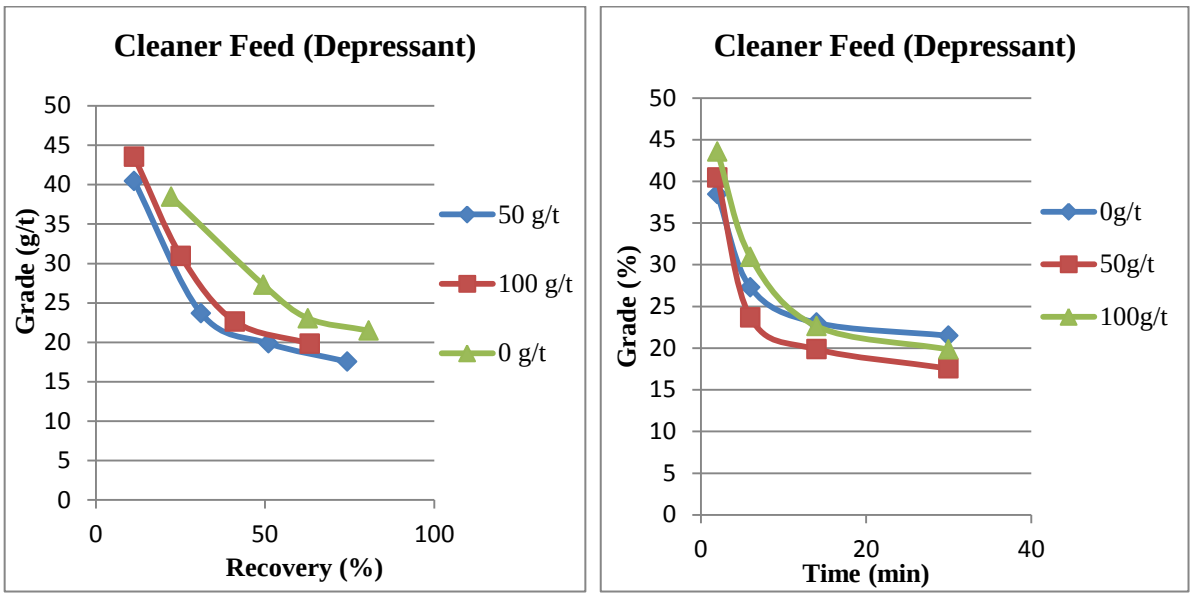


Figure 4.14: Variation of (a) Grade-Recovery and (b) Grade-Time with Depressant Dosage for Cleaner Feed Flotation

The recovery time graphs (Figure 4.13(b)) indicate that the highest recovery is obtained with no depressant dosage. This confirms that the addition of depressant to the fine cleaner feed inhibits flotation of both values and gangue as observed from the analysis of SFR. The recovery-time profiles show that for the cleaner tests with dosage of 50g/t and 100g/t depressant the total PGMs recovery is 72% and 65% respectively. This suggests that the cleaner fee is not very floatable in the presence of high doses of depressant and requires extra residence time for all the values to be recovered (Martin & McKay, 2003). This problem is already addressed in the SMC circuit with the cleaner tails fed to the high energy cells. Depressant addition to the cleaner feed thus seems to have an adverse effect on the recovery of slow floating mineral in the cleaners.

The grade versus recovery relationship for tests done on cleaner feed with addition of depressant is shown in Figure 4.14(a). The grade-recovery relationship of the tests on as received cleaner feed was found to be much better than that with depressant addition. For both depressant tests the fall in grade across the test is quite significant. The difference in grade between the first concentrate and the final combined concentrate is 23.45g/t for 50g/t depressant and 23.70g/t for 100g/t dosage. The trend for both the tests shows an initial sharp drop in grade with recovery with the grade drop slowing down with increasing recovery.

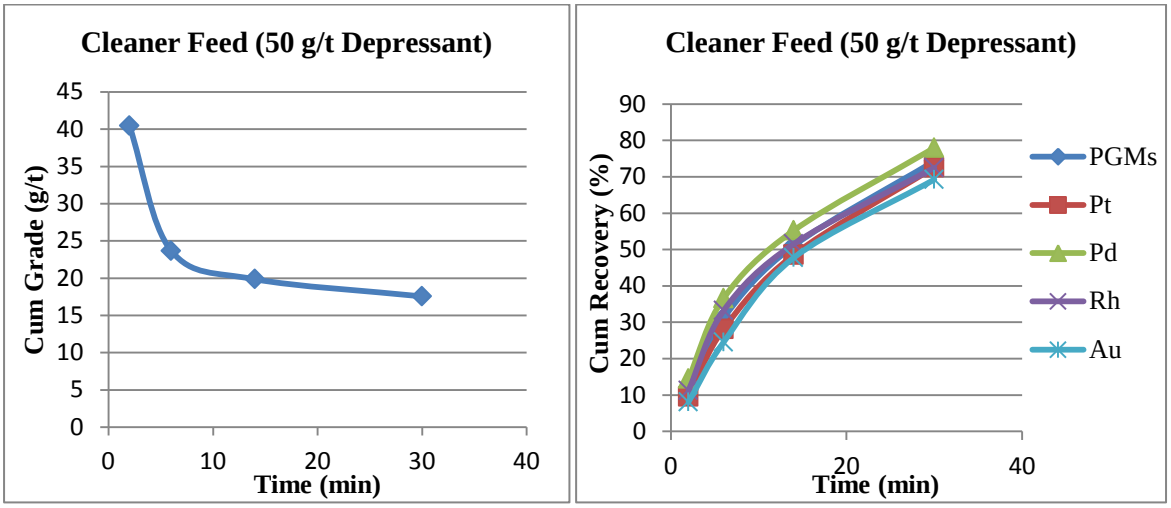


Figure 4.15: (a) Grade-Time and (b) Recovery-Time Curves for Cleaner Feed with 50g/t Depressant

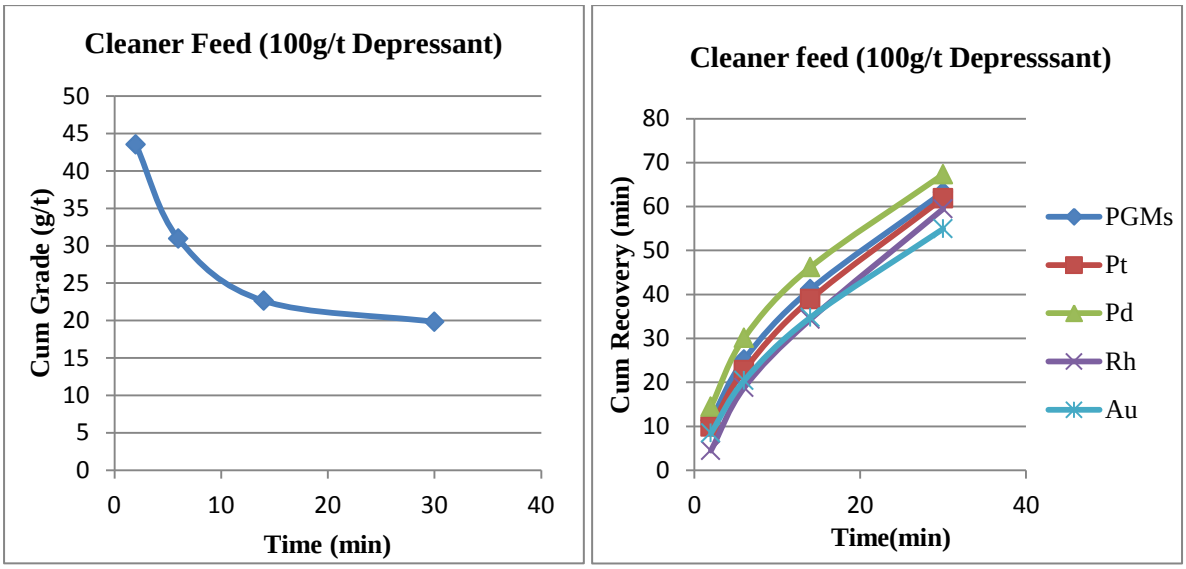


Figure 4.16: (a) Grade-Time and (b) Recovery-Time Curves for Cleaner Feed with 100g/t Depressant

Figures 4.15(a) and 4.15(b) show the grade-recovery and grade-time relationship for the tests carried out with 50g/t depressant dosage to the cleaner feed. The graph shows that mineral is preferentially recovered in the first 2 minutes of the test. As the process proceeds and the values are depleted whilst more gangue material floats, the grade falls sharply to a point where the fall in grade with time diminishes. At this point flotation slows down due to the depressant acting on the less floatable material remaining in the cell and only a small amount of concentrate is recovered per unit time. As a result, the effect of the material added to the concentrate is less notable and grade falls less sharply. The grade-recovery/grade-time behaviour for the tests carried out with 100g/t depressant (Figures 4.16(a) and (b)) showed a trend similar to that observed with 50g/t depressant. The difference was lower grades and

recoveries which were a result of the high depressant dosage inhibiting flotation of the values to a greater extent (Bradshaw et al., 2006).

4.3.2 Effect of Collector Addition

The fine nature of SMC cleaner feed presents problems when performing flotation tests with collector dosage. Tests done with high doses of collector gave inconclusive results as all the material in the float cell floated out within the first few minutes of the test. This is due to the high amount of fines reporting to the concentrate by entrainment (Dryzmala et al,, 2005). As a result, tests were conducted at a single dosage that allowed the test to go to completion to give an indicative result of the effect of collector. The kinetic results obtained from the tests done with addition of 30g/t SIBX collector showed an insignificant variation from those obtained for the tests done with no reagent addition (Table 4). Both k_{PS} and k_{GS} increase slightly giving a net improvement in SFR from 1.88 to 2.01. This improvement is however very minimal and suggests that there is no benefit to the kinetics from collector addition. The grade-recovery-time relationships for the test with collector addition are shown in Figures 4.17.

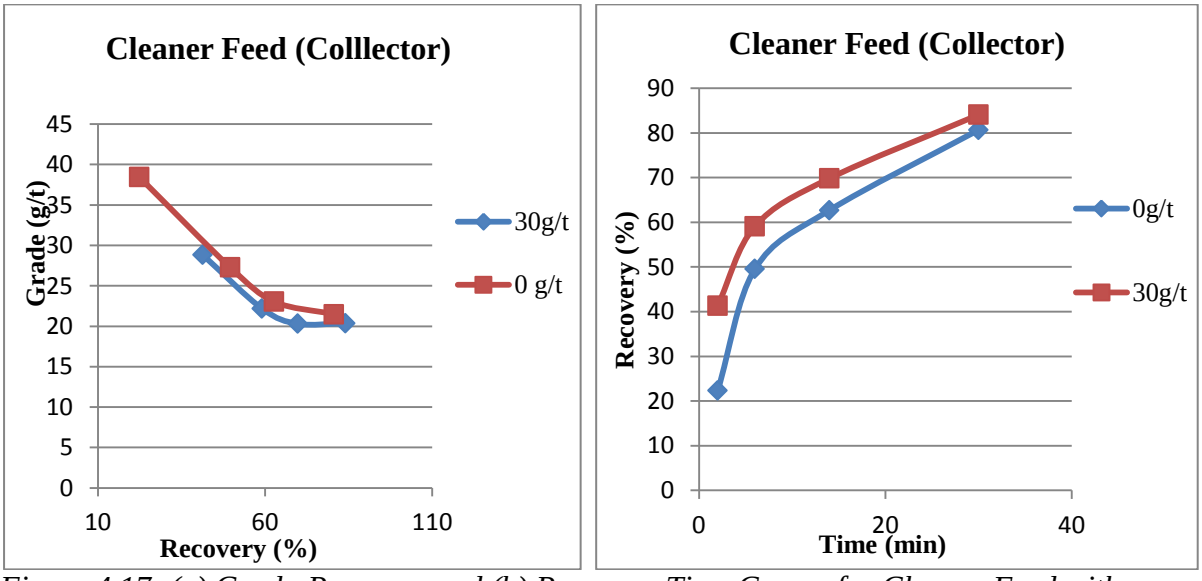


Figure 4.17: (a) Grade-Recovery and (b) Recovery-Time Curves for Cleaner Feed with Collector Addition (30g/t)

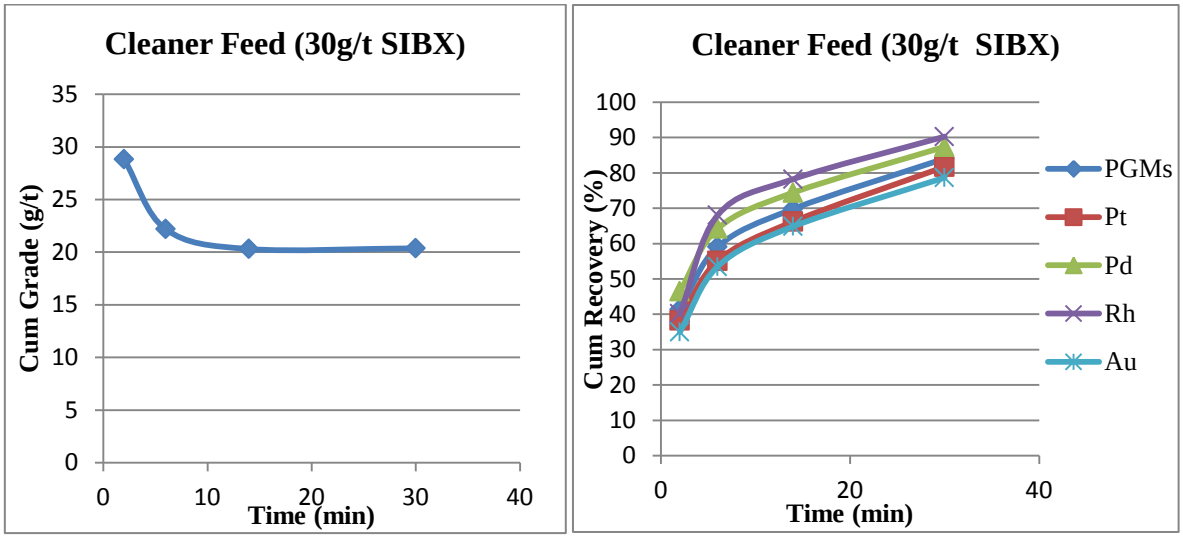


Figure 4.18: (a) Grade-Time and (b) Recovery-Time Relationships for Cleaner Feed with 30g/t Collector

The grade-recovery-time profiles show that collector addition has minimal effect on the recovery of the slow floating fraction. The grade-recovery curves show that collector addition only slightly increases the final recovery (at 30 minutes) whilst grade is lower than that obtained without collector dosage. Collector addition causes the drop in grade due to the collection of gangue material which is then recovered to the concentrate. The high mass pull observed early in the collector test results in the recovery of most of the values in the early stages as shown in the recovery time curve. The kinetic results also show a slight increase in SFR with the addition of collector. The addition of this amount of collector (30g/t) has negligible effect on the flotation kinetics in this case. This is because the slow floating gangue and mineral float at almost the same rate and it is highly unlikely that there would be clean mineral surfaces at this stage in the circuit for the collector to coat and impart hydrophobicity (Hay 2005). The selectivity between mineral and gangue is not enhanced in any way by addition of collector and as a result, the effect of collector on SFR is minimal. The individual graphs for the test with 30g/t collector dosage are shown in Figure 4.18.

4.4 High Energy Circuit Tailings Test Results

The tests carried out to attempt to improve the SFR and recoveries in high energy cell tailings were as follows:

1. Rate tests on the “as received” tails of this circuit as a control experiment.

2. Tailings rate tests were done with increasing depressant dosage to ascertain if there is value to higher depressant dosage.
3. Tailings rate tests with collector addition. The problem of massive entrainment as observed with the tests carried out on cleaner feed is not present with the high energy tailings because they contain residual depressant from the high energy cells unlike the cleaner feed.
4. Tailings particle size distribution to investigate if incomplete liberation is an issue and if regrind of tailings is an option.

The kinetic results of the tests done on high energy cell tailings are summarized in table 4.5

Table 4.5: Summary of Kelsall Parameters for High Energy Cell Tailings

	I_{GF}	k_{GF}	k_{GS}	I_{PF}	k_{PF}	k_{PS}	SFR
As Is	0.0581	0.1976	0.0095	0.1010	0.5805	0.0284	2.99
20g/t SIBX	0.0985	0.1524	0.0098	0.0674	1.3598	0.0360	3.67
40g/t SIBX	0.0985	0.1524	0.0098	0.2080	0.2299	0.0311	3.17
50g/t Finnfix	0.3102	0.4194	0.0214	0.5655	0.8663	0.0931	4.36
100g/t FinnFix	0.2316	0.3979	0.0198	0.3146	6.3805	0.1149	5.80

4.4.1 Effect of Depressant Addition

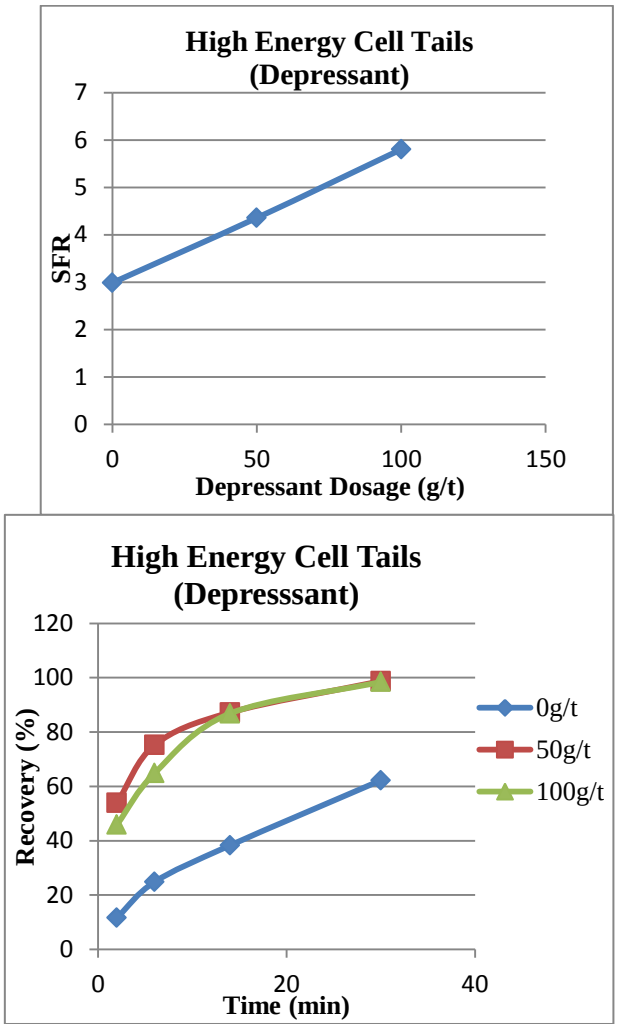


Figure 4.19: Variation of (a) SFR and (b) Recovery-Time relationship with Depressant Dosage for High Energy Cell Tails Flotation

The results of the depressant addition tests are shown in Figures 4.19(a).The figure shows that an increase in the depressant dosage increases SFR proportionally. The increase in SFR with depressant is significant as it more than doubles (2.8 for as received and 5.9 with 100g/t depressant) within the range investigated. Table 4.5 shows that k_{PS} increases as k_{GS} decreases with depressant addition thereby increasing SFR. This indicates that that the depressant effectively suppresses flotation of gangue and improves the selectivity within the range under consideration for these tests

4.4.1.1 Analysis of Grade-Recovery-Time Relationships

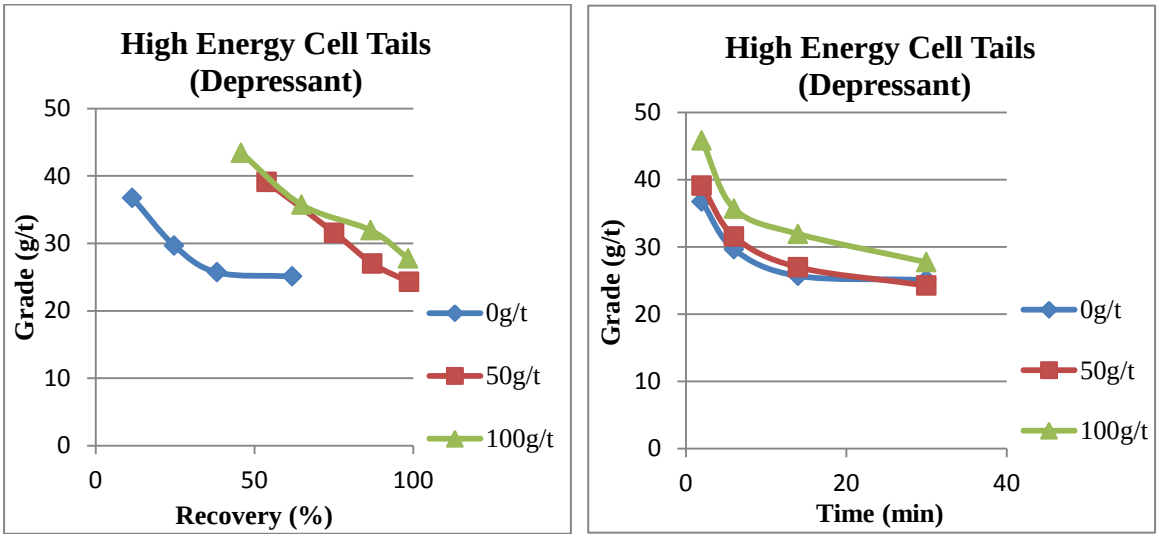


Figure 4.20: (a) Grade-Recovery and (b) Grade-Time relationships with Depressant Dosage for High Energy Cell Tails Flotation.

The recovery-time profiles indicate fast floating behaviour in the initial stages of the tests as seen with the comparatively high recoveries at 2 minutes for both depressant tests (Figure 4.19(b)). This is due to the selective inhibition of gangue flotation due to dosage of depressant. The improvement in slow floating kinetics observed from the kinetics investigation results in the slow floating mineral floating faster gangue is depressed.

The grade recovery curves for tests carried out with depressant addition are shown in Figure 4.20(a). The grade recovery curves for the depressant tests show comparable trends. Good overall recoveries were achieved in the 30 minutes flotation run. The higher recoveries obtained are as a result of the extra residence time afforded to cleaner tailings the slurry in the high energy cells. Although the recoveries were very similar, a slightly higher grade was obtained with 100g/t depressant dosage. This indicates that higher depressant addition led to more effective depression of the gangue. Generally the grades obtained were high as they were both above 20g/t. The grade time curve for 50g/t depressant addition (Figure 4.20(b)) indicates a steep fall in grade with time up to 10 minutes after which grade falls less steeply up to the end of the test. For 100g/t depressant addition, the point at which grade starts to fall less steeply is earlier in the test at around 6 minutes. The levelling out of the fall in grade is due to the less amount of material recovered to the concentrate per unit time as a result of the diminishing of the floatable material in the cell. This results in a lower rate of fall in grade. The higher depressant dosage naturally acts to suppress flotation of a higher proportion of particles and hence flotation slows down earlier in the test. The individual graphs for the tests are shown in Figures 4.21 and 4.22.

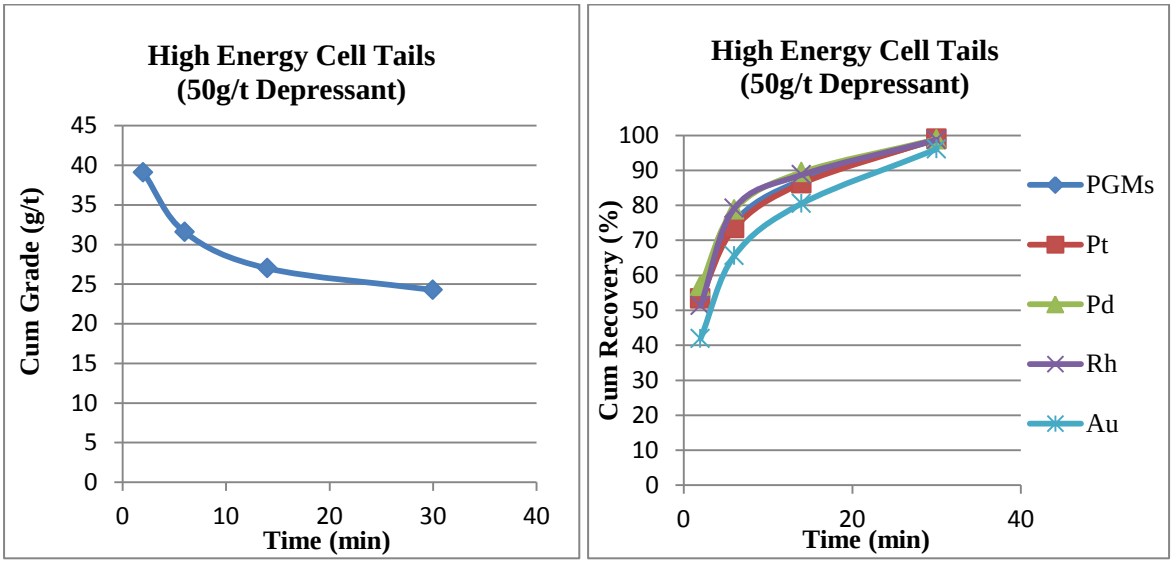


Figure 4.21: (a) Grade-Time and (b) Recovery-Time Relationships for High Energy Cell Tails Flotation with 50g/t Depressant

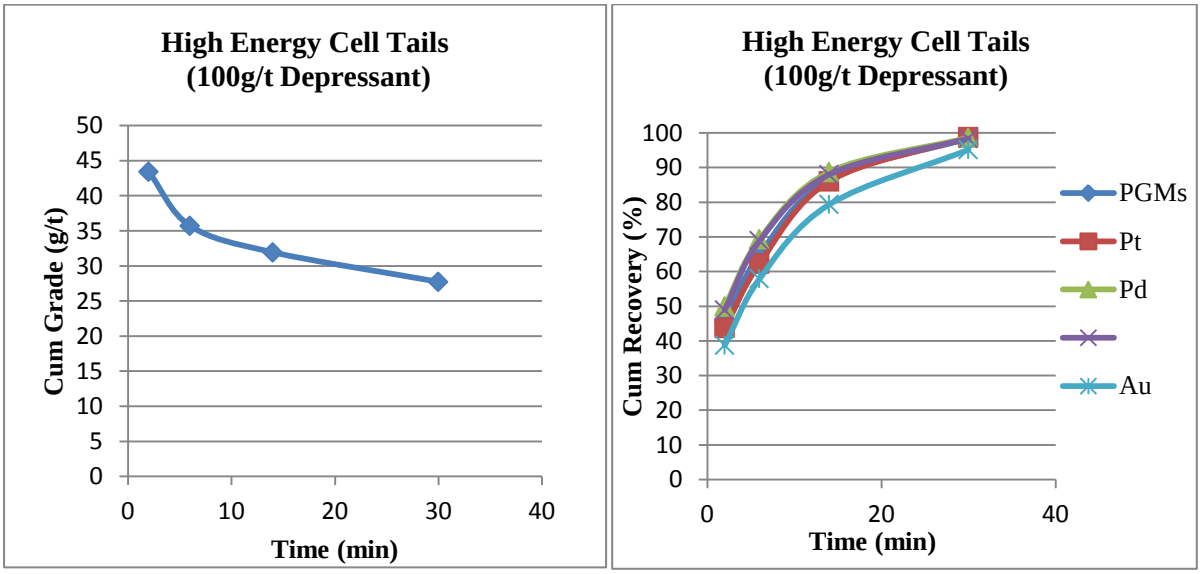


Figure 4.22: (a) Grade-Time and (b) Recovery-Time Relationships for High Energy Cell Tails Flotation with 100g/t Depressant

4.4.2 Effect of Collector Addition

The results for the collector addition tests to high energy tailings are shown in Figures 4.23-4.26. The results show a general increase in SFR with collector addition (Figure 4.23(a)). Maximum SFR was achieved at a dosage of 20g/t SIBX. On further addition of collector to 40g/t, SFR falls as the value of k_{PS} falls whilst k_{GS} remains the same. This is a result of the multi layers formed on the mineral surfaces which reduce the proportion of hydrophobic "tails" which orient into the slurry with a higher collector dosage (Wills, 2006). This leads to

a reduction in the flotation rate of the mineral and the flotation differential between mineral and gangue falls leading to a fall in SFR. As a result of this, the grade achieved for this collector dosage is lower than that achieved with 20g/t collector.

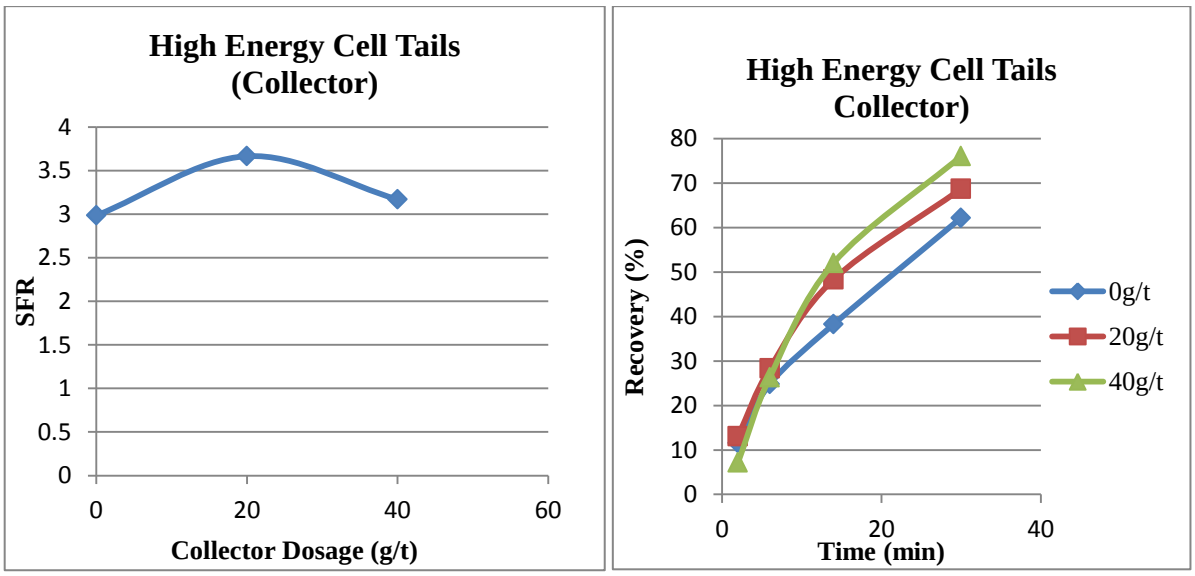


Figure 4.23: Variation of (a) SFR and (b) Recovery-Time with Collector Dosage for High Energy Cell Tails Flotation

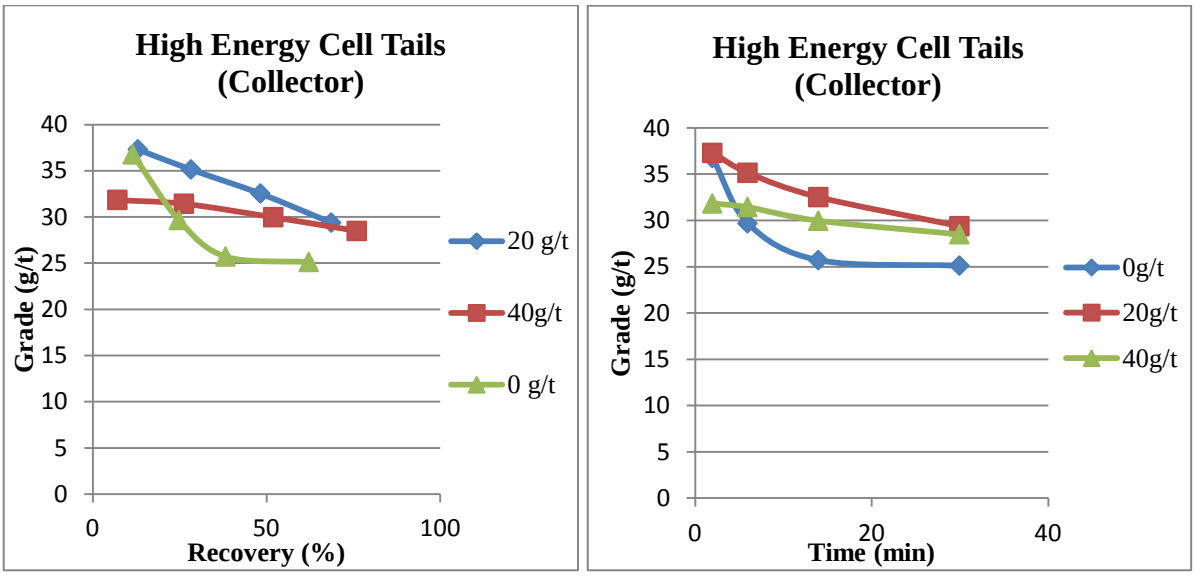


Figure 4.24: (a) Variation of Grade-Recovery and (b) Grade-Time with Collector Dosage for High Energy Cell Tails Flotation

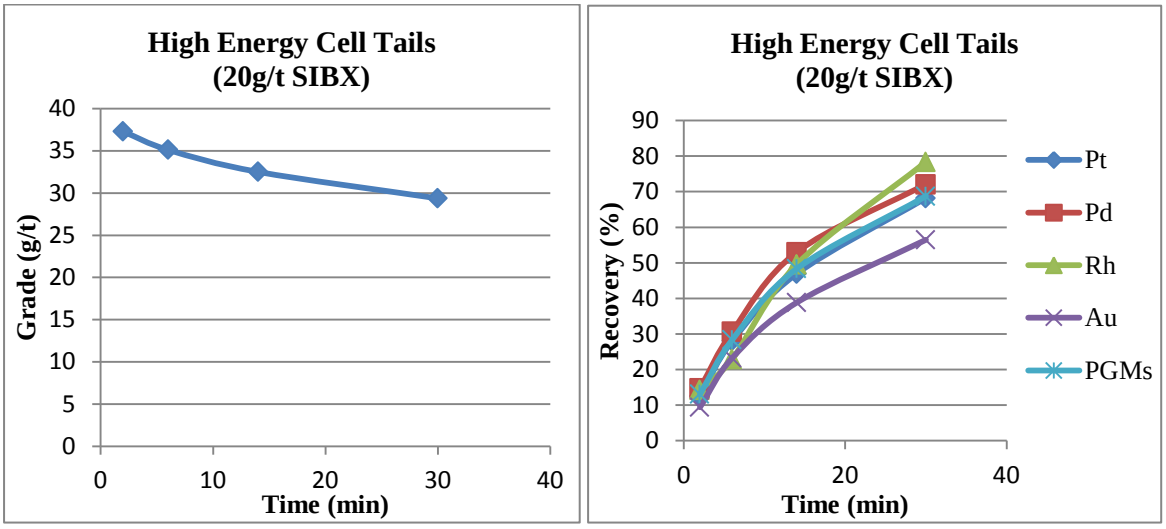


Figure 4.25: (a) Grade-Time and (b) Recovery-Time Relationships for High Energy Cell Tails with Collector (20g/t)

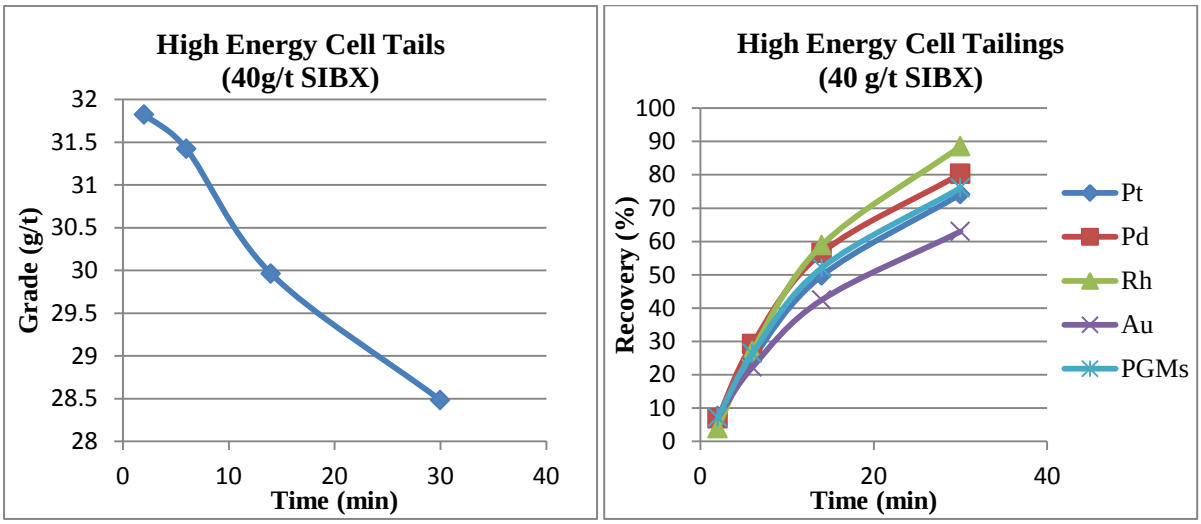


Figure 4.26: (a) Grade-Time and (b) Recovery-Time Relationships for High Energy Cell Tails with Collector (40g/t)

4.4.2.1 Analysis of Grade-Recovery-Time Relationships

Recoveries of 68.7% and 76.2% are obtained for the 20g/t and 40g/t dosages respectively as shown in Figures 4.23(b). The grade recovery curves shown in Figure 4.24(a) indicate that better grades and recoveries are obtained with collector dosage than with "as received" slurry. This indicates that with the "as received" slurry, the mineral surfaces do not exhibit sufficient hydrophobicity in order attach themselves to the air bubbles and be recovered. The 40g/t dosage achieves a greater recovery at lower grade than the 20g/t dosage. This means that at the 20g/t dosage there is insufficient collector to coat all the available mineral surfaces whilst the 40g/t dosage results in higher recovery of both values and gangue to the concentrate as

more material is recovered. The grade obtained is not much lower than that achieved with 20g/t (29.3g/t vs 28.5g/t) however the recovery achieved with collector addition is generally lower than that achieved with addition of depressant. This is likely because collector usage depends on the presence of exposed mineral surfaces unlikely to be present in large quantities at this late point in the flotation circuit The grade-time graphs (Figure 4.24(b)) show that in both tests the grade initially fell steeply between the first two concentrates obtained (2 and 6 minutes for the 20g/t test and 2 and 14 minutes for the 40g/t test) after which it fell less steeply. This is due to the action of the collector that results in the values being selectively recovered in the earlier stages of the test. In the later stages of the test flotation slows down and there is less concentrate recovered with time as values are depleted. The results for the individual collector tests are shown in Figures 4.25 and 4.26.

4.4.3 Sieve Analysis of High energy Cell Tailings

The results of the sieve analysis of high energy tailings are shown in Table 4.6.

Table 4.5: Wet Sieve Analysis Results for High energy Cell Tailings

Fraction	Mass (g)	Mass %	Cum Passing (%)	Cum Retained (%)	PGM (ppm)
106	11.4	0.74	99.26	0.74	2.14
75	142.2	9.29	89.96	10.04	4.29
53	105.7	6.914	83.06	16.94	1.08
38	107	6.99	76.07	23.93	1.59
-38	1164.2	76.07	0	100	2.19

The results show that the high energy tailings have 89.96% passing 75µm. This is more than 80% passing 75µm and therefore there is no obvious need for regrinding of the high energy tailings. There is however significant value locked in the material retained on the 75µm sieve. A study by Pease et al, (2004) showed that an ultrafine grind of 80% passing 40µm may help in improving recoveries.

CHAPTER 5

CONCLUSIONS

Recovery of slow floating PGMs is plays an important role in increasing overall recoveries in a flotation plant. Optimization of reagent dosage is essential in achieving the maximum possible recovery. Degree of liberation is also important and the optimum grind has to be achieved for the optimum degree of liberation of PGM values to be achieved so that they are floated efficiently. By applying Kelsall's unmodified rate model, flotation rate parameters were generated to determine the response to different reagent dosages applied at different selected points in the flotation circuit. The slow floating ratio (SFR) was calculated from the Kelsall parameters generated. This ratio indicated the relative floatability of slow floating mineral to slow floating gangue and can be used to compare the relative effects of different reagent dosages. The grade-recovery-time relationships generated indicate the practical applicability of the different reagent dosages.

Addition of depressant to the feed to the third bank of the scavengers increased slow floating ratio (SFR) from 12.27 to a maximum of 21.64 at a dosage of 25g/t. Recovery and grade improved 73% and 5% to 83 and 7% at that dosage. Excessive depressant addition in the scavengers diminished SFR to 11.69 at a dosage of 100g/t. Collector addition to the scavengers has a small positive effect with SFR increasing to 12.78 at dosage of 60g/t. The results showed high grade and recovery obtained at lower collector dosage of 30g/t. Increasing collector dosage diminished the grade obtained as a high proportion of fine gangue was pulled to the concentrate by entrainment due to excess collector increasing the pulp viscosity .

Addition of reagents to the cleaner feed did not significantly improve SFR. The highest improvement in SFR was achieved with the addition of 30g/t of collector that improved SFR from 1.89 to 2.01. Due to the fine particles contained in cleaner feed, any further collector addition led to excessive mass pull due to entrainment. Recoveries achieved for flotation of cleaner feed were low even with reagent addition however the grades were high indicating the need for longer residence times to achieve higher recovery.

Addition of depressant to high energy cells led to an improvement in SFR. With a dosage of 100g/t of depressant, SFR increased from 2.99 to 5.80. The slurry in this section of the circuit needed a high depressant dosage to suppress flotation of the abundant slow floating gangue at this late stage of the process. Collector addition only gave a very small improvement at lower

collector dose. A 20g/t dosage increased SFR from 2.99 to 3.67 as the collector acted on the limited amount of clean mineral faces at this stage in the circuit. From the particle size analysis carried out on high energy cell tailings, it was concluded that no regrinding was necessary. This is because the grind of more 80% passing 75µm was considered adequate for good liberation and recovery of the ore processed.

For the SMC concentrator, reagent manipulation is a useful way of improving the recovery of slow floating mineral. Depending on the section of the circuit, reagent type and dosage can be varied in order to improve the relative floatability of mineral to gangue. Depressant offered more scope for improvement of SFR and recoveries as depressants suppress flotation of gangue thereby improving selectivity of the flotation process. Collectors have a less profound effect since they work better with exposed mineral surfaces that are unlikely to present in the back end of the flotation circuit.

CHAPTER 6

RECOMMENDATIONS FOR FUTURE WORK

1. Optimization of collector and depressant addition to the scavengers should be done in the scavengers to ascertain how staged addition affects recovery. This should also investigate where in the scavengers the best reagent addition points are.
2. Optimization of depressant addition to the high energy cells must be done to optimize recovery.
3. In order to avoid slow floating minerals becoming a circulating load that is recovered in the scavengers and falls away in the high energy cell tailings, an investigation must be done for a different treatment route for the scavenger concentrates to recover some mineral to the final concentrate before they are sent to the cleaners.
4. The scavengers treat rougher tailings some of which are still fast floating. An investigation into increasing rougher residence time so that the scavenger cells process only slow floating mineral would be useful. This would be useful in ascertaining if any benefit accrues from the extra residence time in the scavengers for the slow floating mineral.

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APPENDIX A: Scavenger Bank 3 Composite Raw Data

As Received	Time (min)	Mass (g)	**Pt** *(ppm)*	**Pd** *(ppm)*	**Rh** *(ppm)*	**Au** *(ppm)*	**Total PGM** *(ppm)*
Conc 1	2	11.6	7.163	2.457	0.878	0.276	10.774
Conc 2	6	18.8	5.578	2.38	0.76	0.197	8.915
Conc 3	14	36.4	2.729	1.654	0.693	0.25	5.326
Conc 4	30	59.8	1.455	2.548	0.038	0.062	4.103
Tails		676.1	0.201	0.111	0.052	0.022	0.386

25g/t Finnfix	Time (min)	Mass (g)	**Pt** *(ppm)*	**Pd** *(ppm)*	**Rh** *(ppm)*	**Au** *(ppm)*	**Total** *(ppm)*
Conc 1	2	10.5	8.132	3.189	1.159	0.146	12.626
Conc 2	6	25.3	6.704	2.743	1.041	0.368	10.856
Conc 3	14	43.1	2.769	1.85	0.771	0.214	5.604
Conc 4	30	46.2	3.189	2.041	0.833	0.234	6.297
Tails		684.7	0.225	0.091	0.048	0.021	0.385

50g/t Finnfix	Time (min)	Mass (g)	**Pt** *(ppm)*	**Pd** *(ppm)*	**Rh** *(ppm)*	**Au** *(ppm)*	**Total PGM** *(ppm)*
Conc 1	2	12.3	6.86	3.288	0.925	0.309	11.382
Conc 2	6	26.3	6.802	2.533	0.929	0.317	10.581
Conc 3	14	42.1	2.9	14.777	0.774	0.225	18.676
Conc 4	30	41.4	3.062	1.898	0.894	0.209	6.063
Tails		588.4	0.197	0.07	0.041	0.017	0.325

100g/t Finnfix	Time (min)	Mass (g)	**Pt** *(ppm)*	**Pd** *(ppm)*	**Rh** *(ppm)*	**Au** *(ppm)*	**Total PGM** *(ppm)*
Conc 1	2	7.4	5.904	2.013	0.807	0.439	9.163
Conc 2	6	16.4	6.643	3.067	0.987	0.296	10.993
Conc 3	14	30	3.194	2.233	0.806	0.282	6.515
Conc 4	30	43.5	2.687	1.672	0.749	0.188	5.296
Tails		609.9	0.221	0.144	0.048	0.026	0.439

30g/t SIBX	Time (min)	Mass (g)	**Pt** *(ppm)*	**Pd** *(ppm)*	**Rh** *(ppm)*	**Au** *(ppm)*	**Total PGM** *(ppm)*
Conc 1	2	96.9	5.798	4.534	1.077	0.525	11.934
Conc 2	6	65.8	4.182	4.127	0.976	0.756	10.041
Conc 3	14	58.1	9.572	8.43	1.67	1.38	21.052
Conc 4	30	52.6	11.86	7.113	1.649	0.955	21.577
Tails		469	0.397	0.31	0.158	0.005	0.87

60g/t SIBX	Time (min)	Mass (g)	**Pt** *(ppm)*	**Pd** *(ppm)*	**Rh** *(ppm)*	**Au** *(ppm)*	**Total PGM** *(ppm)*
Conc 1	2	103.9	7.493	5.319	1.133	0.6	14.545
Conc 2	6	62.9	4.776	4.542	0.944	0.916	11.178
Conc 3	14	58	8.086	6.702	1.388	1.345	17.521
Conc 4	30	42.5	11.423	6.5	1.828	0.647	20.398
Tails		453.3	0.13	0.23	0.135	0.008	0.503

APPENDIX B: Cleaner Feed Compiled Experimental Data

As Is	Time (min)	Mass (g)	**Pt** *(ppm)*	**Pd** *(ppm)*	**Au** *(ppm)*	**Rh** *(ppm)*	**Total PGM** *(ppm)*
Conc 1	2	91.4	16.013	18.635	2.381	1.41	38.439
Conc 2	6	194.2	10.485	9.025	1.593	0.911	22.014
Conc 3	14	142.3	7.414	5.416	1.251	0.462	14.543
Conc 4	30	162.3	9.338	6.224	1.376	0.451	17.389
Tails		338.5	5.012	2.817	0.925	0.231	8.985

30g/t SIBX	Time (min)	Mass (g)	**Pt** *(ppm)*	**Pd** *(ppm)*	**Au** *(ppm)*	**Rh** *(ppm)*	**Total PGM** *(ppm)*
Conc 1	2	199	13.028	12.752	1.932	1.098	28.81
Conc 2	6	170.5	6.672	5.677	1.196	0.892	14.437
Conc 3	14	107.5	7.012	5.169	1.159	0.516	13.856
Conc 4	30	95.2	11.021	7.416	1.593	0.687	20.717
Tails		252.1	4.901	2.729	0.93	0.21	8.77

50g/t Finnfix	Time (min)	Mass (g)	**Pt** *(ppm)*	**Pd** *(ppm)*	**Au** *(ppm)*	**Rh** *(ppm)*	**Total PGM** *(ppm)*
Conc 1	2	33.3	17.66	19.219	2.417	1.16	40.456
Conc 2	6	123.3	9.268	7.936	1.326	0.613	19.143
Conc 3	14	198	7.473	5.068	1.079	0.364	13.984
Conc 4	30	149.5	8.377	5.515	1.546	0.422	15.86
Tails		301.3	5.581	3.235	1.015	0.314	10.145

100g/t Finnfix	Time (min)	Mass (g)	**Pt** *(ppm)*	**Pd** *(ppm)*	**Au** *(ppm)*	**Rh** *(ppm)*	**Total PGM** *(ppm)*
Conc 1	2	27.8	19.865	20.552	2.671	0.455	43.543
Conc 2	6	59.1	12.081	10.444	1.781	0.687	24.993
Conc 3	14	107.3	8.387	5.936	1.178	0.411	15.912
Conc 4	30	146.6	8.734	5.686	1.211	0.486	16.117
Tails		422.3	5.021	3.056	0.939	0.272	9.288

APPENDIX C: High Energy Cell Tailings Composite Raw Data

As Received	Time (min)	Mass (g)	**Pt** *ppm*	**Pd** *ppm*	**Au** *ppm*	**Rh** *ppm*	**Total PGM** *ppm*
Conc 1	2	22.6	19.067	13.865	2.724	1.086	36.742
Conc 2	6	37.2	12.543	9.757	2.002	1.046	25.348
Conc 3	14	46.6	10.094	8.172	1.649	0.722	20.637
Conc 4	30	70.7	12.324	8.936	1.98	0.967	24.207
Tails		392.7	3.666	2.272	0.773	0.163	6.874

20g/t SIBX	Time (min)	Mass (g)	**Pt** *ppm*	**Pd** *ppm*	**Au** *ppm*	**Rh** *ppm*	**Total PGM** *ppm*
Conc 1	2	29.8	19.239	14.42	2.504	1.13	37.293
Conc 2	6	38.8	14.872	11.307	2.139	1.088	29.406
Conc 3	14	57.7	13.333	7.689	1.931	0.938	23.891
Conc 4	30	72.4	18.072	12.021	2.834	0.527	33.454
Tails		372.4	3.877	2.189	0.927	0.137	7.13

40g/t SIBX	Time (min)	Mass (g)	**Pt** *ppm*	**Pd** *ppm*	**Au** *ppm*	**Rh** *ppm*	**Total PGM** *ppm*
Conc 1	2	15.5	14.26	13.679	1.977	1.358	31.274
Conc 2	6	42.3	14.074	11.451	1.804	1.271	28.6
Conc 3	14	61.9	13.274	9.547	1.766	1.141	25.728
Conc 4	30	64.3	17.09	11.543	2.567	0.625	31.825
Tails		373.8	2.417	1.368	0.548	0.076	4.409

50g/t Finnfix	Time (min)	Mass (g)	**Pt** *ppm*	**Pd** *ppm*	**Au** *ppm*	**Rh** *ppm*	**Total PGM** *ppm*
Conc 1	2	159.8	20.112	15.617	1.933	1.448	39.11
Conc 2	6	116.4	10.418	8.229	1.496	1.076	21.219
Conc 3	14	98.4	7.827	4.761	1.118	0.441	14.147
Conc 4	30	97.3	7.881	4.245	1.176	0.464	13.766
Tails		244.7	0.247	0.203	0.118	0.023	0.591

100g/t Finnfix	Time (min)	Mass (g)	**Pt** *ppm*	**Pd** *ppm*	**Au** *ppm*	**Rh** *ppm*	**Total PGM** *ppm*
Conc 1	2	118.8	21.538	17.537	2.523	1.836	43.434
Conc 2	6	85.6	12.742	9.493	1.737	1.039	25.011
Conc 3	14	100.8	13.762	8.022	1.645	0.845	24.274
Conc 4	30	94.1	7.92	4.396	1.3	0.491	14.107
Tails		270.5	0.271	0.215	0.139	0.027	0.652

Particle Size Distribution Composite Raw Data

Sieve (µm)	Mass Retained (g)	**Pt (ppm)**	**Pd (ppm)**	**Au (ppm)**	**Rh (ppm)**	**Total PGM (ppm)**
106	11.4	21.461	15.193	2.786	1.698	41.138
75	42.2	46.552	34.056	5.116	2.705	88.429
53	105.7	37.669	24.773	3.784	1.882	68.108
38	107	26.347	14.574	2.068	2.67	45.659
-38	1164.2	6.168	4.361	1.043	0.621	12.193

APPENDIX D: Raw Experimental Data

Note: In all cases μ (in bold print) represents the mean of the 3 experimentally obtained values. These are the figures compiled in the tables in Appendices A to C and these were input to Kincalc® for each concentrate or tails sample. σ represents the standard deviation of the experimental values in %.

Sample Id	Matrix	Mass	Pt	Pd	Au	Rh	Total PGM
Scavenger 3	As Is	g	ppm	ppm	ppm	ppm	ppm
1	Conc 1	12.4	7.46	2.98	1.003	0.333	11.776
2	Conc 1	11.6	7.026	2.012	0.953	0.29	10.281
3	Conc 1	10.8	7.003	2.379	0.678	0.205	10.265
μ	**Conc 1**	**11.6**	**7.163**	**2.457**	**0.878**	**0.276**	**10.774**
σ		6.9	3.6	19.9	19.9	23.6	8.1
	Conc 2	19.8	5.575	2.44	0.799	0.188	9.002
	Conc 2	14.7	4.93	1.706	0.649	0.181	7.466
	Conc 2	21.9	6.229	2.994	0.832	0.222	10.277
μ	**Conc 2**	**18.8**	**5.578**	**2.38**	**0.76**	**0.197**	**8.915**
σ		19.7	11.6	27.1	12.8	11.1	15.8
	Conc 3	46.1	3.127	1.749	0.678	0.319	5.873
	Conc 3	32.5	2.387	1.431	0.808	0.203	4.829
	Conc 3	30.6	2.673	1.782	0.593	0.228	5.276
μ	**Conc 3**	**36.4**	**2.729**	**1.654**	**0.693**	**0.25**	**5.326**
σ		23.2	3.6	19.9	19.9	23.6	8.1
	Conc 4	65.9	1.49	2.782	0.045	0.087	4.404
	Conc 4	55.8	1.499	2.517	0.036	0.049	4.101
	Conc 4	57.7	1.376	2.345	0.033	0.05	3.804
μ	**Conc 4**	**59.8**	**1.455**	**2.548**	**0.038**	**0.062**	**4.103**
σ		9.0	4.7	8.6	16.4	34.9	7.3
	Tails	589	0.198	0.099	0.044	0.019	0.36
	Tails	741.3	0.215	0.12	0.052	0.018	0.405
	Tails	698	0.19	0.114	0.06	0.029	0.393
μ	**Tails**	**676.1**	**0.201**	**0.111**	**0.052**	**0.022**	**0.386**
σ		11.6	6.4	9.7	15.4	27.6	6.0

Sample Id	Matrix	Mass	Pt	Pd	Au	Rh	Total PGM
Scavenger 3	25g/t Finnfix	g	ppm	ppm	ppm	ppm	ppm

1	Conc 1	11.4	8.308	2.093	1.441	0.138	11.98
2	Conc 1	9.8	9.602	3.113	0.729	0.094	13.538
3	Conc 1	10.3	6.486	4.361	1.307	0.206	12.36
μ	**Conc 1**	**10.5**	**8.132**	**3.189**	**1.159**	**0.146**	**12.626**
σ		7.8	19.3	35.6	32.6	38.6	6.4
	Conc 2	17.4	7.394	2.393	0.663	0.402	10.852
	Conc 2	19.9	5.394	3.595	0.855	0.303	10.147
	Conc 2	19.1	7.324	2.241	1.605	0.399	11.569
μ	**Conc 2**	**25.3**	**6.704**	**2.743**	**1.041**	**0.368**	**10.856**
σ		5.0	16.9	27.0	47.8	15.3	6.5
	Conc 3	36.3	3.404	2.11	0.937	0.267	6.718
	Conc 3	44.3	2.348	1.99	0.82	0.204	5.362
	Conc 3	48.7	2.555	1.45	0.556	0.171	4.732
μ	**Conc 3**	**43.1**	**2.769**	**1.85**	**0.771**	**0.214**	**5.604**
σ		14.6	20.2	19.0	25.3	22.8	18.1
	Conc 4	57.3	2.663	1.538	0.927	0.253	5.381
	Conc 4	44.7	3.749	2.393	0.748	0.298	7.188
	Conc 4	36.6	3.155	2.192	0.824	0.151	6.322
μ	**Conc 4**	**46.2**	**3.189**	**2.041**	**0.833**	**0.234**	**6.297**
σ		22.6	17.1	21.9	10.8	32.2	14.4
	Tails	704.2	0.202	0.101	0.038	0.019	0.36
	Tails	599.4	0.183	0.083	0.045	0.022	0.333
	Tails	750.5	0.29	0.089	0.061	0.022	0.462
μ	**Tails**	**684.7**	**0.225**	**0.091**	**0.048**	**0.021**	**0.385**
σ		11.3	25.4	10.1	24.6	8.2	17.7

Sample Id	Matrix	Mass	Pt	Pd	Au	Rh	Total PGM
Scavenger 3	50g/t Finnfix	g	ppm	ppm	ppm	ppm	ppm
1	Conc 1	11.3	7.22	3.838	1.101	0.311	12.47
2	Conc 1	10.9	5.99	2.93	0.937	0.36	10.217
3	Conc 1	14.7	7.37	3.096	0.737	0.256	11.459
μ	**Conc 1**	**12.3**	**6.86**	**3.288**	**0.925**	**0.309**	**11.382**
σ		17.0	11.0	14.7	19.7	16.8	9.9

	Conc 2	29.3	6.912	3.008	0.892	0.362	11.174
	Conc 2	26.7	6.019	2.229	0.999	0.297	9.544
	Conc 2	22.9	7.475	2.362	0.896	0.292	11.025
μ	**Conc 2**	**26.3**	**6.802**	**2.533**	**0.929**	**0.317**	**10.581**
σ		12.2	10.8	16.5	6.5	12.3	8.5
	Conc 3	47.2	3.078	1.551	0.876	0.288	5.793
	Conc 3	44.4	2.989	1.777	0.757	0.189	5.712
	Conc 3	34.7	2.633	1.403	0.689	0.198	4.923
μ	**Conc 3**	**42.1**	**2.9**	**1.577**	**0.774**	**0.225**	**5.476**
σ		15.6	8.1	11.9	12.2	24.3	8.8
	Conc 4	44.2	4.01	2.481	0.997	0.201	7.689
	Conc 4	41.3	3.019	1.869	0.898	0.232	6.018
	Conc 4	38.7	2.157	1.344	0.787	0.194	4.482
μ	**Conc 4**	**41.4**	**3.062**	**1.898**	**0.894**	**0.209**	**6.063**
σ		6.6	30.3	30.0	11.8	9.7	26.5
	Tails	604.5	0.193	0.008	0.041	0.011	0.253
	Tails	529.3	0.122	0.007	0.038	0.016	0.183
	Tails	631.4	0.276	0.006	0.044	0.024	0.35
μ	**Tails**	**588.4**	**0.197**	**0.007**	**0.041**	**0.017**	**0.262**
σ		9.0	39.1	14.3	7.3	38.6	32.0

Sample Id	Matrix	Mass	Pt	Pd	Au	Rh	Total PGM
Scavenger 3	100g/t Finnfix	g	ppm	ppm	ppm	ppm	ppm
1	Conc 1	8.0	5.734	1.484	0.737	0.457	8.412
2	Conc 1	7.5	6.383	2.439	0.793	0.526	10.141
3	Conc 1	6.7	5.595	2.116	0.891	0.334	8.936
μ	**Conc 1**	**7.4**	**5.904**	**2.013**	**0.807**	**0.439**	**9.163**
σ		8.9	7.1	24.1	9.7	22.2	9.7
	Conc 2	16.9	6.364	3.193	0.748	0.274	10.579
	Conc 2	17.4	7.329	3.475	0.954	0.304	12.062
	Conc 2	14.9	6.236	2.533	1.259	0.31	10.338

μ	**Conc 2**	**25.3**	**6.643**	**3.067**	**0.987**	**0.296**	**10.993**
σ		5.2	9.0	15.8	26.0	6.5	8.5
	Conc 3	26.4	3.304	2.19	0.805	0.199	6.498
	Conc 3	33.6	2.747	2.484	0.907	0.29	6.428
	Conc 3	30	3.531	2.025	0.706	0.357	6.619
μ	**Conc 3**	**30**	**3.194**	**2.233**	**0.806**	**0.282**	**6.515**
σ		12.0	12.6	10.4	12.5	28.1	1.5
	Conc 4	35.7	3.474	1.96	0.839	0.201	6.474
	Conc 4	44.5	2.646	2.028	0.799	0.175	5.648
	Conc 4	50.3	1.941	1.028	0.609	0.188	3.766
μ	**Conc 4**	**43.5**	**2.687**	**1.672**	**0.749**	**0.188**	**5.296**
σ		16.9	28.6	33.4	16.4	6.9	26.2
	Tails	673.3	0.199	0.179	0.054	0.033	0.465
	Tails	590	0.254	0.138	0.039	0.027	0.458
	Tails	566.4	0.21	0.115	0.051	0.018	0.394
μ	**Tails**	**609.9**	**0.221**	**0.144**	**0.048**	**0.026**	**0.439**
σ		9.2	13.2	22.5	16.5	29.0	8.9

Sample Id	Matrix	Mass	Pt	Pd	Au	Rh	Total PGM
Scavenger 3	30g/t SIBX	g	ppm	ppm	ppm	ppm	ppm
1	Conc 1	100.5	5.944	5.123	1.199	0.588	12.854
2	Conc 1	90.4	5.574	4.015	0.9	0.414	10.903
3	Conc 1	99.8	5.876	4.464	1.132	0.573	12.045
μ	**Conc 1**	**96.9**	**5.798**	**4.534**	**1.077**	**0.525**	**11.934**
σ		5.8	3.4	12.3	14.6	18.4	8.2
	Conc 2	74.8	4.392	3.596	0.933	0.86	9.781
	Conc 2	66.2	3.942	4.918	0.975	0.563	10.398
	Conc 2	56.4	4.212	3.867	1.02	0.845	9.944
μ	**Conc 2**	**65.8**	**4.182**	**4.127**	**0.976**	**0.756**	**10.041**
σ		14.0	5.4	16.9	4.5	22.1	3.2
	Conc 3	49.7	8.98	7.78	1.05	0.98	18.79
	Conc 3	59.6	10.516	8.86	2.06	1.61	23.046

	Conc 3	65	9.22	8.65	1.9	1.55	21.32
μ	**Conc 3**	**58.1**	**9.572**	**8.43**	**1.67**	**1.38**	**21.052**
σ		13.4	8.6	6.8	32.5	25.2	10.2
	Conc 4	56	11.09	7.99	1.946	0.87	21.896
	Conc 4	43.8	12.47	5.899	1.671	0.965	21.005
	Conc 4	58	12.02	7.45	1.33	1.03	21.83
μ	**Conc 4**	**52.6**	**11.86**	**7.113**	**1.649**	**0.955**	**21.577**
σ		14.6	5.9	15.3	18.7	8.4	2.3
	Tails	437.4	0.42	0.358	0.201	0.004	0.983
	Tails	476	0.404	0.233	0.086	0.006	0.729
	Tails	493.6	0.367	0.339	0.187	0.005	0.898
μ	**Tails**	**469**	**0.397**	**0.31**	**0.158**	**0.005**	**0.87**
σ		6.1	6.8	21.7	39.7	20.0	14.9

Sample Id	Matrix	Mass	Pt	Pd	Au	Rh	Total PGM
Scavenger 3	60g/t SIBX	g	ppm	ppm	ppm	ppm	ppm
1	Conc 1	98	7.465	5.293	1.102	0.604	14.464
2	Conc 1	126.3	8.321	5.913	1.353	0.793	16.38
3	Conc 1	87.4	6.693	4.751	0.944	0.403	12.791
μ	**Conc 1**	**103.9**	**7.493**	**5.319**	**1.133**	**0.6**	**14.545**
σ		19.4	10.9	10.9	18.2	32.5	12.3
	Conc 2	53.4	4.028	4.103	0.826	0.839	9.796
	Conc 2	76	5.123	4.837	1.032	0.92	11.912
	Conc 2	59.3	5.177	4.686	0.974	0.989	11.826
μ	**Conc 2**	**62.9**	**4.776**	**4.542**	**0.944**	**0.916**	**11.178**
σ		18.6	13.6	8.5	11.3	8.2	10.7
	Conc 3	67.3	8.202	6.938	1.457	1.563	18.16
	Conc 3	48.4	7.937	5.954	1.283	1.128	16.302
	Conc 3	58.3	8.119	7.214	1.424	1.344	18.101
μ	**Conc 3**	**58**	**8.086**	**6.702**	**1.388**	**1.345**	**17.521**
σ		16.3	1.7	9.9	6.7	16.2	6.0
	Conc 4	50.9	11.863	7.262	1.937	0.711	21.773

	Conc 4	39.6	11.363	6.465	1.738	0.638	20.204
	Conc 4	37	11.043	5.773	1.809	0.592	19.217
μ	**Conc 4**	**42.5**	**11.423**	**6.5**	**1.828**	**0.647**	**20.398**
σ		17.4	3.6	11.5	5.5	9.3	6.3
	Tails	482.9	0.137	0.253	0.163	0.01	0.563
	Tails	399.4	0.103	0.193	0.118	0.006	0.42
	Tails	477.6	0.15	0.244	0.124	0.008	0.526
μ	**Tails**	**453.3**	**0.13**	**0.23**	**0.135**	**0.008**	**0.503**
σ		10.3	18.7	14.1	18.1	25.0	14.8

Sample Id	Matrix	Mass	Pt	Pd	Au	Rh	Total PGM
Cleaner Feed	As Is	g	ppm	ppm	ppm	ppm	ppm
1	Conc 1	102.3	17.283	16.826	1.993	1.826	37.928
2	Conc 1	89.7	16.028	20.273	2.735	1.028	40.064
3	Conc 1	82.2	14.728	18.806	2.415	1.376	37.325
μ	**Conc 1**	**91.4**	**16.013**	**18.635**	**2.381**	**1.41**	**38.439**
σ		11.1	8.0	9.3	15.6	28.4	3.7
	Conc 2	173.4	9.927	8.823	1.552	0.869	21.171
	Conc 2	199.4	10.639	9.192	1.728	0.905	22.464
	Conc 2	209.8	10.889	9.06	1.499	0.959	22.407
μ	**Conc 2**	**194.2**	**10.485**	**9.025**	**1.593**	**0.911**	**22.014**
σ		9.7	4.8	2.1	7.5	5.0	3.3
	Conc 3	150.3	7.293	5.293	1.278	0.464	14.328
	Conc 3	120.5	6.593	4.837	0.826	0.393	12.649
	Conc 3	156.1	8.356	6.118	1.649	0.529	16.652
μ	**Conc 3**	**142.3**	**7.414**	**5.416**	**1.251**	**0.462**	**14.543**
σ		13.4	12.0	12.0	32.9	14.7	13.8
	Conc 4	170.4	9.402	6.072	1.287	0.426	17.187
	Conc 4	202.8	10.327	6.826	1.732	0.502	19.387
	Conc 4	113.7	8.285	5.774	1.109	0.425	15.593
μ	**Conc 4**	**162.3**	**9.338**	**6.224**	**1.376**	**0.451**	**17.389**
σ		27.8	10.9	8.7	23.3	9.8	11.0

	Tails	298.5	4.937	1.927	0.749	0.192	7.805
	Tails	337.1	5.028	2.828	0.829	0.232	8.917
	Tails	379.9	5.071	3.696	1.197	0.269	10.233
μ	**Tails**	**338.5**	**5.012**	**2.817**	**0.925**	**0.231**	**8.985**
σ		12.0	1.4	31.4	25.8	16.7	13.5

Sample Id	Matrix	Mass	Pt	Pd	Au	Rh	Total PGM
Cleaner Feed	30g/t SIBX	g	ppm	ppm	ppm	ppm	ppm
1	Conc 1	209.3	13.099	12.846	1.998	1.092	29.035
2	Conc 1	168.4	12.293	11.825	1.793	0.983	26.894
3	Conc 1	219.3	13.692	13.585	2.005	1.219	30.501
μ	**Conc 1**	**199**	**13.028**	**12.752**	**1.932**	**1.098**	**28.81**
σ		13.6	5.4	6.9	6.2	10.8	6.3
	Conc 2	189.2	6.892	5.827	1.392	0.968	15.079
	Conc 2	166.8	6.529	5.327	0.905	0.792	13.553
	Conc 2	155.5	6.595	5.877	1.291	0.916	14.679
μ	**Conc 2**	**170.5**	**6.672**	**5.677**	**1.196**	**0.892**	**14.437**
σ		10.1	2.9	5.4	21.5	10.1	5.5
	Conc 3	99.3	6.742	4.937	0.927	0.466	13.072
	Conc 3	100.3	7.102	5.193	1.273	0.485	14.053
	Conc 3	122.9	7.192	5.377	1.277	0.597	14.443
μ	**Conc 3**	**107.5**	**7.012**	**5.169**	**1.159**	**0.516**	**13.856**
σ		12.4	3.4	4.3	17.3	13.7	5.1
	Conc 4	89.7	10.637	7.472	1.736	0.723	20.568
	Conc 4	100.5	11.293	7.273	1.504	0.865	20.935
	Conc 4	95.4	11.133	7.503	1.539	0.473	20.648
μ	**Conc 4**	**95.2**	**11.021**	**7.416**	**1.593**	**0.687**	**20.717**
σ		5.7	3.1	1.7	7.9	28.9	0.9
	Tails	197.4	4.493	2.732	0.883	0.155	8.263
	Tails	262.9	4.939	3.103	1.031	0.221	9.294
	Tails	296	5.271	2.352	0.876	0.254	8.753
μ	**Tails**	**252.1**	**4.901**	**2.729**	**0.93**	**0.21**	**8.77**
σ		19.9	8.0	13.8	9.4	24.0	5.9

Sample Id	Matrix	Mass	Pt	Pd	Au	Rh	Total PGM
Cleaner Feed	50g/t Finnfiix	g	ppm	ppm	ppm	ppm	ppm
1	Conc 1	25.6	16.349	17.283	1.893	0.883	36.408
2	Conc 1	33.8	18.231	20.122	2.394	1.179	41.926
3	Conc 1	40.5	18.4	20.252	2.964	1.418	43.034
μ	**Conc 1**	**33.3**	**17.66**	**19.219**	**2.417**	**1.16**	**40.456**
σ		22.4	6.4	8.7	22.2	23.1	8.8
	Conc 2	123.8	9.263	7.928	1.353	0.677	19.221
	Conc 2	100.2	8.384	7.309	1.183	0.594	17.47
	Conc 2	145.9	10.157	8.571	1.442	0.568	20.738
μ	**Conc 2**	**123.3**	**9.268**	**7.936**	**1.326**	**0.613**	**19.143**
σ		18.5	9.6	8.0	9.9	9.3	8.5
	Conc 3	168.4	7.293	4.389	1.021	0.332	13.035
	Conc 3	204.3	7.826	5.284	0.977	0.364	14.451
	Conc 3	221.3	7.3	5.531	1.239	0.396	14.466
μ	**Conc 3**	**198**	**7.473**	**5.068**	**1.079**	**0.364**	**13.984**
σ		13.6	4.1	11.9	13.0	8.8	5.9
	Conc 4	128.3	7.283	4.953	1.102	0.387	13.725
	Conc 4	203.8	7.937	6.229	1.839	0.441	16.446
	Conc 4	116.4	9.911	5.363	1.697	0.438	17.409
μ	**Conc 4**	**149.5**	**8.377**	**5.515**	**1.546**	**0.422**	**15.86**
σ		31.7	16.3	11.8	25.3	7.2	12.0
	Tails	283.9	4.833	3.137	0.929	0.309	9.208
	Tails	364.3	5.328	3.622	1.228	0.398	10.576
	Tails	255.7	6.582	2.946	0.888	0.235	10.651
μ	**Tails**	**301.3**	**5.581**	**3.235**	**1.015**	**0.314**	**10.145**
σ		18.7	16.2	10.8	18.3	26.0	8.0

Sample Id	Matrix	Mass	Pt	Pd	Au	Rh	Total PGM
Cleaner Feed	100g/t Finnfix	g	ppm	ppm	ppm	ppm	ppm
1	Conc 1	27.2	16.374	15.373	2.664	0.494	34.905
2	Conc 1	24.4	22.436	22.686	2.485	0.302	47.909

3	Conc 1	31.8	20.785	23.597	2.864	0.569	47.815
μ	**Conc 1**	**27.8**	**19.865**	**20.552**	**2.671**	**0.455**	**43.543**
σ		13.4	15.8	21.9	7.1	30.3	17.2
	Conc 2	66.2	12.937	10.836	1.938	0.745	26.456
	Conc 2	59.4	12.033	10.553	1.773	0.647	25.006
	Conc 2	51.7	11.273	9.943	1.632	0.669	23.517
μ	**Conc 2**	**59.1**	**12.081**	**10.444**	**1.781**	**0.687**	**24.993**
σ		12.3	6.9	4.4	8.6	7.5	5.9
	Conc 3	133.5	8.434	5.364	1.274	0.373	15.445
	Conc 3	89.4	8.374	5.946	0.843	0.435	15.598
	Conc 3	99	8.353	6.498	1.417	0.425	16.693
μ	**Conc 3**	**107.3**	**8.387**	**5.936**	**1.178**	**0.411**	**15.912**
σ		21.6	0.5	9.6	25.4	8.1	4.3
	Conc 4	144.6	8.384	5.734	1.363	0.527	16.008
	Conc 4	150.4	8.937	6.011	1.118	0.423	16.489
	Conc 4	144.8	8.881	5.313	1.152	0.508	15.854
μ	**Conc 4**	**146.6**	**8.734**	**5.686**	**1.211**	**0.486**	**16.117**
σ		2.2	3.5	6.2	11.0	11.4	2.1
	Tails	444.3	5.229	2.948	1.028	0.322	9.527
	Tails	399.4	4.947	3.635	0.884	0.299	9.765
	Tails	423.2	4.887	2.585	0.905	0.195	8.572
μ	**Tails**	**422.3**	**5.021**	**3.056**	**0.939**	**0.272**	**9.288**
σ		5.3	3.6	17.4	8.3	24.9	6.8

Sample Id	Matrix	Mass	Pt	Pd	Au	Rh	Total PGM
[2]H-E Tails	As Is	g	ppm	ppm	ppm	ppm	ppm
1	Conc 1	27.4	19.373	12.937	2.846	1.038	36.194
2	Conc 1	22.3	18.373	13.757	2.018	1.227	35.375
3	Conc 1	18.1	19.455	14.901	3.308	0.993	38.657
μ	**Conc 1**	**22.6**	**19.067**	**13.865**	**2.724**	**1.086**	**36.742**
σ		20.6	3.2	7.1	24.0	11.4	4.6

[2] H-E Tails: High Energy Cell Tailings

	Conc 2	40.5	12.918	10.021	1.726	0.927	25.592
	Conc 2	38.5	11.826	9.282	2.226	1.199	24.533
	Conc 2	32.6	12.885	9.968	2.054	1.012	25.919
μ	**Conc 2**	**37.2**	**12.543**	**9.757**	**2.002**	**1.046**	**25.348**
σ		11.0	5.0	4.2	12.7	13.3	2.9
	Conc 3	46.2	10.302	8.728	1.272	0.729	21.031
	Conc 3	55.7	9.028	8.292	1.933	0.801	20.054
	Conc 3	37.9	10.952	7.496	1.742	0.636	20.826
μ	**Conc 3**	**46.6**	**10.094**	**8.172**	**1.649**	**0.722**	**20.637**
σ		19.1	9.7	7.6	20.6	11.5	2.5
	Conc 4	61.9	13.992	9.239	2.003	1.191	26.425
	Conc 4	69.3	11.113	9.028	1.727	0.893	22.761
	Conc 4	80.9	11.867	8.541	2.21	0.817	23.435
μ	**Conc 4**	**70.7**	**12.324**	**8.936**	**1.98**	**0.967**	**24.207**
σ		13.5	12.1	4.0	12.2	20.4	8.1
	Tails	333.2	3.029	2.736	0.928	0.188	6.881
	Tails	401.3	3.293	2.202	0.677	0.126	6.298
	Tails	443.6	4.676	1.878	0.714	0.175	7.443
μ	**Tails**	**392.7**	**3.666**	**2.272**	**0.773**	**0.163**	**6.874**
σ		14.2	24.1	19.1	17.5	20.1	8.3

Sample Id	Matrix	Mass	Pt	Pd	Au	Rh	Total PGM
H-E Tails	20g/t SIBX	g	ppm	ppm	ppm	ppm	ppm
1	Conc 1	25	19.192	15.332	2.102	1.348	37.974
2	Conc 1	29.9	18.832	14.112	3.012	0.922	36.878
3	Conc 1	34.5	19.693	13.816	2.398	1.12	37.027
μ	**Conc 1**	**29.8**	**19.239**	**14.42**	**2.504**	**1.13**	**37.293**
σ		15.9	2.2	5.6	18.5	18.9	1.6
	Conc 2	44.2	15.018	11.928	2.419	1.128	30.493
	Conc 2	35.2	14.655	11.002	1.992	0.978	28.627
	Conc 2	37	14.943	10.991	2.006	1.158	29.098
μ	Conc 2	**38.8**	**14.872**	**11.307**	**2.139**	**1.088**	**29.406**
σ		12.3	1.3	4.8	11.3	8.9	3.3

	Conc 3	56.3	12.849	7.103	1.792	0.917	22.661
	Conc 3	47.9	13.738	8.028	1.992	0.882	24.64
	Conc 3	68.9	13.412	7.936	2.009	1.015	24.372
μ	**Conc 3**	**57.7**	**13.333**	**7.689**	**1.931**	**0.938**	**23.891**
σ		18.3	3.4	6.6	6.2	7.3	4.5
	Conc 4	68.2	17.927	11.927	2.628	0.483	32.965
	Conc 4	77.1	17.586	11.723	2.112	0.553	31.974
	Conc 4	71.9	18.703	12.413	3.762	0.545	35.423
μ	**Conc 4**	**72.4**	**18.072**	**12.021**	**2.834**	**0.527**	**33.454**
σ		6.2	3.2	2.9	29.8	7.3	5.3
	Tails	388.2	3.528	1.839	0.928	0.12	6.415
	Tails	327.1	4.002	2.229	0.993	0.129	7.353
	Tails	401.9	4.101	2.499	0.86	0.162	7.622
μ	Tails	**372.4**	**3.877**	**2.189**	**0.927**	**0.137**	**7.13**
σ		10.7	7.9	15.2	7.2	16.1	8.9

Sample Id	Matrix	Mass	Pt	Pd	Au	Rh	Total PGM
H-E Tails	40g/t SIBX	g	ppm	ppm	ppm	ppm	ppm
1	Conc 1	16.5	14.552	13.425	1.829	1.193	30.999
2	Conc 1	14.8	13.993	13.772	2.101	1.732	31.598
3	Conc 1	15.2	14.235	13.84	2.001	1.149	31.225
μ	**Conc 1**	**15.5**	**14.26**	**13.679**	**1.977**	**1.358**	**31.274**
σ		5.7	2.0	1.6	7.0	23.9	1.0
	Conc 2	50.2	14.949	11.862	2.028	1.712	30.551
	Conc 2	40.2	13.623	11.872	2.028	0.972	28.495
	Conc 2	36.5	13.65	10.619	1.356	1.129	26.754
μ	**Conc 2**	**42.3**	**14.074**	**11.451**	**1.804**	**1.271**	**28.6**
σ		16.8	5.4	6.3	21.5	30.7	6.6
	Conc 3	70.3	13.453	9.238	2.183	0.879	25.753
	Conc 3	59.7	12.738	10.202	1.495	1.009	25.444
	Conc 3	55.7	13.631	9.201	1.62	1.535	25.987
μ	**Conc 3**	**61.9**	**13.274**	**9.547**	**1.766**	**1.141**	**25.728**
σ		12.2	3.6	5.9	20.8	30.4	1.1

	Conc 4	60.4	16.544	10.928	2.293	0.638	30.403
	Conc 4	82.2	16.937	11.002	2.536	0.599	31.074
	Conc 4	50.3	17.789	12.699	2.872	0.638	33.998
μ	**Conc 4**	**64.3**	**17.09**	**11.543**	**2.567**	**0.625**	**31.825**
σ		25.4	3.7	8.7	11.3	3.6	6.0
	Tails	332.7	1.937	1.392	0.638	0.108	4.075
	Tails	408.3	2.239	1.774	0.489	0.069	4.571
	Tails	380.4	3.075	0.938	0.517	0.051	4.581
μ	**Tails**	**373.8**	**2.417**	**1.368**	**0.548**	**0.076**	**4.409**
σ		10.2	24.4	30.6	14.5	38.3	6.6

Sample Id	Matrix	Mass	Pt	Pd	Au	Rh	Total PGM
H-E Tails	50g/t Finnfix	g	ppm	ppm	ppm	ppm	ppm
1	Conc 1	147.4	19.193	15.102	1.928	1.203	37.426
2	Conc 1	189.3	19.838	16.002	2.124	1.896	39.86
3	Conc 1	142.7	21.305	15.747	1.747	1.245	40.044
μ	**Conc 1**	**159.8**	**20.112**	**15.617**	**1.933**	**1.448**	**39.11**
σ		16.1	5.4	3.0	9.8	26.8	3.7
	Conc 2	109.3	9.83	7.768	1.337	1.233	20.168
	Conc 2	89.5	10.677	8.754	1.554	0.948	21.933
	Conc 2	150.4	10.747	8.165	1.597	1.047	21.556
μ	**Conc 2**	**116.4**	**10.418**	**8.229**	**1.496**	**1.076**	**21.219**
σ		26.7	4.9	6.0	9.3	13.4	4.4
	Conc 3	120.7	7.293	5.102	1.293	0.513	14.201
	Conc 3	92.4	8.211	4.293	0.938	0.449	13.891
	Conc 3	82.1	7.977	4.888	1.123	0.361	14.349

μ	**Conc 3**	**98.4**	**7.827**	**4.761**	**1.118**	**0.441**	**14.147**
σ		20.3	6.1	8.8	15.9	17.3	1.7
	Conc 4	102.4	7.283	4.193	1.039	0.398	12.913
	Conc 4	90.6	8.293	4.637	0.894	0.457	14.281
	Conc 4	98.9	8.067	3.905	1.595	0.537	14.104
μ	**Conc 4**	**97.3**	**7.881**	**4.245**	**1.176**	**0.464**	**13.766**
σ		6.2	6.7	8.7	31.5	15.0	5.4
	Tails	285.3	0.238	0.193	0.093	0.036	0.56
	Tails	232.4	0.263	0.219	0.193	0.019	0.694
	Tails	216.4	0.24	0.197	0.068	0.014	0.519
μ	**Tails**	**244.7**	**0.247**	**0.203**	**0.118**	**0.023**	**0.591**
σ		14.7	5.6	6.9	56.1	50.1	15.5

Sample Id	Matrix	Mass	Pt	Pd	Au	Rh	Total PGM
H-E Tails	100g/t Finnfix	g	ppm	ppm	ppm	ppm	ppm
1	Conc 1	100.4	20.382	17.373	2.837	2.192	42.784
2	Conc 1	133.5	21.383	17.837	2.384	1.738	43.342
3	Conc 1	122.5	22.849	17.401	2.348	1.578	44.176
μ	**Conc 1**	**118.8**	**21.538**	**17.537**	**2.523**	**1.836**	**43.434**
σ		14.2	5.8	1.5	10.8	17.3	1.6
	Conc 2	99.8	12.384	9.293	1.958	0.948	24.583
	Conc 2	84.2	13.032	9.759	1.593	1.232	25.616
	Conc 2	72.8	12.81	9.427	1.66	0.937	24.834
μ	**Conc 2**	**85.6**	**12.742**	**9.493**	**1.737**	**1.039**	**25.011**
σ		15.8	2.6	2.5	11.2	16.1	2.2
	Conc 3	119.3	13.938	8.398	1.948	0.586	24.87
	Conc 3	89.5	13.294	7.859	1.595	0.883	23.631

	Conc 3	93.6	14.054	7.809	1.392	1.066	24.321
μ	**Conc 3**	**100.8**	**13.762**	**8.022**	**1.645**	**0.845**	**24.274**
σ		16.0	3.0	4.1	17.1	28.7	2.6
	Conc 4	102.9	7.39	3.857	1.294	0.544	13.085
	Conc 4	88.56	7.94	4.485	1.738	0.499	14.662
	Conc 4	90.84	8.43	4.846	0.868	0.43	14.574
μ	**Conc 4**	**94.1**	**7.92**	**4.396**	**1.3**	**0.491**	**14.107**
σ		8.2	6.6	11.4	33.5	11.7	6.3
	Tails	247.4	0.33	0.193	0.163	0.028	0.714
	Tails	301.3	0.283	0.244	0.138	0.038	0.703
	Tails	262.8	0.2	0.208	0.116	0.015	0.539
μ	**Tails**	**270.5**	**0.271**	**0.215**	**0.139**	**0.027**	**0.652**
σ		10.3	24.3	12.2	16.9	42.7	15.0