

**A Comparison between Column and Mechanical Cell
Performance in Platinum Flotation**

By

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PREFACE

The work in this dissertation was performed at the University of KwaZulu-Natal, School of Chemical Engineering. The project was supervised by Professor Emeritus B.K. Loveday and Mr. L Maharaj.

As the candidate's main supervisor I, Professor Emeritus B.K. Loveday agree/do not agree to the submission of this dissertation.

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Professor Emeritus B.K. Loveday

As the candidate's co-supervisor I, Mr. L. Maharaj agree/do not agree to the submission of this dissertation.

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Reshmi Hurdeyal

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To my spiritual mentor, Kadamba Kanana Swami, the best engineer I know. Thank you for never accepting any excuses.

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ABSTRACT

Froth flotation has been used to concentrate minerals for over a century (Finch and Dobby, 1990) and it is essential for the economical recovery of metals which are present at low concentrations. The traditional flotation cell consists of a stirred tank incorporating air injection. In recent decades columns have appeared to offer advantages for platinum flotation. This project is a comparative study of the use of column and mechanical cells as cleaners for the flotation of an UG-2 (Upper Group 2) platinum ore.

A laboratory column cell, 1.8 m high and 60 mm in diameter, was constructed to have the same volume as a conventional 5L Denver mechanical cell. Batch cleaning tests were used to compare the two types of cells, while varying the depressant dosage and conditioning technique. The performance of the column cell when the pulp was recycled was also compared. Finally the effect of doubling the column height, whilst maintaining the same mass percent of solids as in the original column, was investigated. Samples were subjected to acid digestion to determine Copper (Cu) and Nickel (Ni) recovery as an indicator of platinum recovery. Platinum Group Metals and Gold (PGM+Au) assays were performed on samples from a final series of tests that used a different but similar ore sample.

The investigations showed that the mechanical cell performed better than the column. The PGM grade in the mechanical cell was 6.5ppm higher for a common recovery of 80% PGM in both cells. The performance of the column cell was only marginally improved when the pulp was recycled. Mass recovery data indicated that the column cell required less depressant for reducing the mass recovery. The corresponding grade recovery curves showed that for a 10 g/t depressant addition, the grade of the concentrate produced in the column cell doubled whilst that of the mechanical cell revealed no significant change. It was suggested in literature (Whitney and Yan, 1996) that the depressant was more effective in a column cell, due to the absence of agitation by an impeller. Platinum recovery data from the final series did not confirm the interesting trend observed with the preliminary tests (Cu/Ni) at a low depressant dosage (10g/t), but marginal improvements were noted at higher dosages. The effect of agitation during conditioning with depressant was also investigated. Gentle conditioning of the depressant, prior to column flotation, resulted in a higher grade and recovery of PGM as compared to conditioning by means of the recirculation pump. The PGM grade when gentle conditioning was employed was 9 ppm higher for a common PGM recovery of 80% for both conditioning techniques. Finally it was found that increasing the column height improved the grade-recovery relationship. The PGM grade obtained by the longer column was 8 ppm higher for a common

PGM recovery of 75% in both columns. The performance of the longer column was similar to that of the mechanical cell.

Tests were also conducted at a platinum concentrator, to compare the performance of a 378L pilot-plant column (5.5m high and 0.3m diameter), to that of a pilot-plant consisting of six mechanical cells. The pilot test rig of mechanical cells (FCTR) achieved a significantly better recovery than a pilot column. However, by decreasing the column feed flowrate and thereby increasing the residence time, a substantial improvement in the PGM grade and recovery was obtained by the column cell.

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NOMENCLATURE

SYMBOL	DESCRIPTION	UNITS
A_{CS}	Cross sectional area	cm^2
a	mass fraction	
C_f	Pressure correction factor	
d	Diameter of steel balls	m
D_C	Diameter of column cell	cm
D_M	Diameter of mill	m
F	Force	N
g	Gravitational acceleration	m^2/s
k	Distribution rate constant	1/min
m	Mass of object	Kg
M_{Solids}	Mass of ore	Kg
M_{water}	Mass of Water	Kg
$M_{\text{Steel balls}}$	Mass of milling media	Kg
P	Power	W
q_{Air}	Flowrate of air	m^3/min
r	Radius of mill	m

R	Rotameter Reading	
R_m	Metal Recovery	%
V	Voltage	V
V_{Charge}	Volume of slurry and media combined	m^3
V_{Mill}	Volume of the cylindrical mill	m^3
V_{Solids}	Volume of ore	m^3
$V_{\text{Steel balls}}$	Volume of the milling media used	m^3
V_{Water}	Volume of water	m^3
ρ_{steel}	Density of steel	Kg/m^3
ρ_{solids}	Density of ore	Kg/m^3
τ	Torque	N.m
v_M	Superficial velocity in the mechanical cell	m/min
v_C	Superficial velocity in the column cell	m/min
ω_c	Critical speed for mill operation	rpm

CHAPTER 1: INTRODUCTION

1.1. BACKGROUND

Platinum is among the most valuable of metals in the world with a wide variety of uses ranging from a variety of jewellery and different electronic devices to everyday dental care (<http://minerals.usgs.gov/minerals/pubs/commodity/platinum/myb1-2006-plati.pdf>). It is estimated that about 20% of the products purchased by modern consumers either contain platinum or use it in production. Platinum's market price fluctuates around \$1700 (<http://www.platinum.matthey.com/pgm-prices/price-charts/>). South Africa has 52% of the world's platinum reserves. As the largest producers of PGMs (Platinum Group Minerals), methods of improving platinum recovery or reducing operating costs are of interest in South Africa.

Flotation is essential for the concentration of platinum group minerals, as the concentration in the ore is very low (4-5g/t). Industry has predominantly used the standard mechanical cell, consisting of a tank with a centralized impeller, for more than a century. The limitations of mechanical cells became prominent when dealing with fine particles that had a low probability of collision with air bubbles and were more prone to entrainment (Honaker, 1993).

Over the last few decades the concept of column cells has slowly taken hold (Finch and Dobby, 1990). The column cell employs the use of a sparger to generate bubbles instead of a mechanical impeller. Hence, the degree of agitation is reduced. The structure of a column cell allows for the use of deep froths. Most columns also use froth washing to improve the grade of the concentrate. In the flotation circuit, the rougher cells are used to recover as much of the valuable minerals as possible and cleaner cells are used to improve the concentration (grade) of valuable minerals. Column cells seem to be most suitable as cleaners. Industrial producers, such as Northam Platinum has made definitive changes to their circuit design to include the use of column cells in the capacity of cleaners and recleaners. These changes have improved the overall PGM recovery by 7% (Minnaar *et al.*, 2005)

The batch performance of a column flotation cell was compared to that of a mechanical cell. The cells were operated as cleaners in platinum flotation. A laboratory column cell, 1.8m high and 60mm in diameter, was constructed to have the same volume as a conventional 5L Denver mechanical cell. The height of the column cell was more than 7 times that of the mechanical

cell. The bubble path length in a cell is proportional to the cell height therefore the bubble path length in the column cell was significantly longer. The probability of a bubble collecting mineral particles is proportional to the bubble path length. The volumetric flow of air in the column was significantly smaller than the flow in the mechanical cell, as the same superficial velocity was used. The path length was increased proportionately. One would logically expect that the overall recovery as a function of time would be equivalent, for the same size of bubbles, unless the bubbles became overloaded with particles (Loveday, 1983). This relationship is hypothetical, however, and needs to be explored.

1.2. OBJECTIVES OF THE PROJECT

This investigation focused on the following parameters of flotation:

- Preliminary (base case) tests on both mechanical and column cells with no reagent addition during cleaner flotation to obtain a general comparison between the two cells.
- Column performance with the pulp in continual recycle using an external pump.
- The effect of varying the depressant dosages in the mechanical and column cells.
- The effect of both gentle and vigorous depressant conditioning techniques in the column cell to determine the effect of agitation in depressant behavior.
- The effect of increasing the height of the column whilst maintaining the mass percent solids.

The objective of these laboratory investigations was to explore the possibility of using column cells instead of the traditional mechanical cells for cleaning.

Pilot plant tests were conducted on a 5.5 m column cell at Lonmin Platinum and compared with the performance of a pilot-plant test rig of 6 mechanical cells in series. Both tests utilized feed to the final cleaners in the Lonmin Platinum circuit. The flowrate in the column cell was varied at different froth depths so that the performance of the column cell could be evaluated over a range of pulp residence times.

The objective of the pilot plant tests was to compare the performance of column and mechanical cells using a continuous feed and at a significantly larger scale than the experiments done at UKZN.

CHAPTER 2: LITERATURE REVIEW

2.1 OVERVIEW

This section provides a review of the fundamentals of platinum production and the flotation process. It includes an appraisal of the studies undertaken by researchers to evaluate the flotation parameters of column cells and improve their performance in comparison to mechanical cells. An evaluation of the relevant literature was conducted with the aim of determining how best mechanical and column cells can be systematically compared, what the best position for column cells in the flotation circuit would be, and what parameters could effectively be tested on a laboratory scale.

The survey commences with a briefing on UG-2 ore, its location, mineralogy, and associated platinum group metals (PGM).

2.2 UPPER GROUP 2 (UG-2) ORE AND PLATINUM GROUP METALS (PGM)

2.2.1 Location and Mineralogy of UG-2 ore

UG-2 reef in the Bushveld Igneous Complex (BIC) is probably Earth's greatest source of PGM (Lee, 1996). The BIC is found in the northern portion of South Africa (Fig. 2.1(a)).

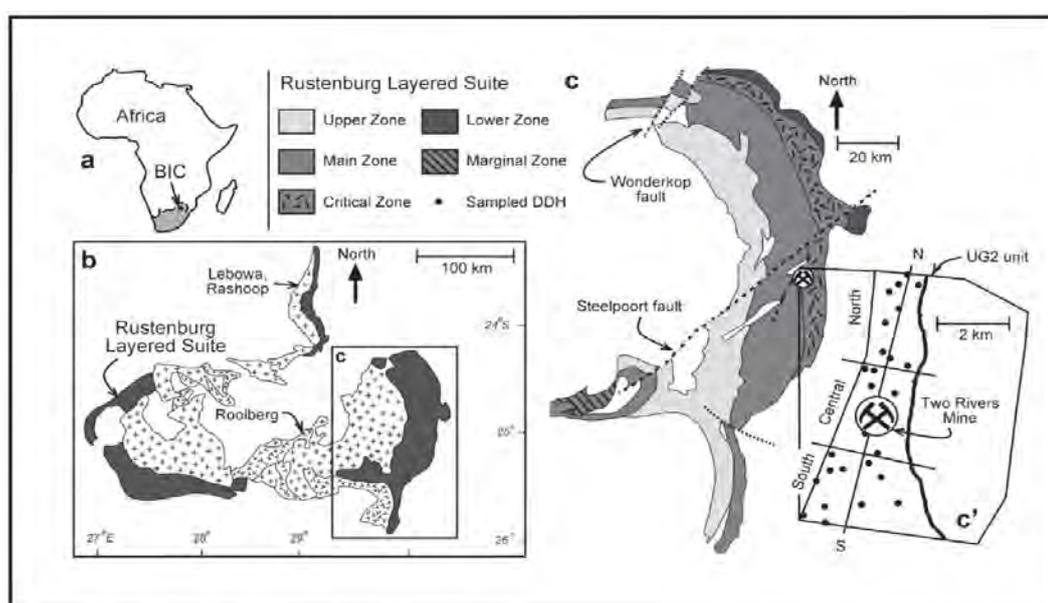


Figure 2.1 Schematic map of the Bushveld Igneous Complex
<http://sajg.geoscienceworld.org/cgi/content-nw/full/112/1/47/F1>

Traditionally, platinum mines have focused on the Merensky Reef. In the last 20 years the Merensky reserves have decreased significantly and producers have developed methods of utilizing the UG-2 Reef (Fig. 2.1(c), upper zone) for expanding the production of platinum (Cramer *et al.*, 2004). The UG-2 Reef is now the major source of South Africa's platinum group metals (Hay and Roy, 2010).

Chromite ($\text{FeO} \cdot \text{Cr}_2\text{O}_3$) forms up to 75% of the reef by mass and is by far its major constituent (Hay and Roy, 2010). Cramer *et al.*, (2004) indicated that the chromite content of the ore itself is between 10% - 25% whereas Nel *et al.*, (2005) placed the percentage chromite between 20% - 30%. The rest of the ore mainly comprises of aluminum silicates (Nel *et al.*, 2005).

The UG-2 reef has a PGM head grade of approximately 4-5.5 g/t (Valenta, 2007). The platinum group metals consist of seven different elements (Hay and Roy, 2010). The four listed below are the primary metals that define the head grade.

Table 2.1 Percentage composition of different PGMs in UG-2 ore (Nel *et al.*, 2005)

<i>Platinum group metal</i>	<i>Percentage (%)</i>
Platinum (Pt)	45
Palladium (Pd)	25
Ruthenium (Ru)	15
Rhodium (Rh)	10

The average diameter of a PGM particle was reported as 9 μm by Lee (1996). Hay and Roy (2010) placed the average grain size to be about 12 μm with rare occurrences of 30 μm particles. However, PGM grains of 50 μm and larger have been reported within pockets of silicates (Hay and Roy, 2010). The PGM grain size in the Merensky reef is much larger by comparison, an average of 150 μm (Lee, 1996).

The sulphide content of UG-2 ore (0.2%) is a lower than that of Merensky ore. The majority of PGM particles are attached to base metal sulphides (Lee, 1996). They have been termed "sulphide-associated" by Lee (1996). Figure 2.2 shows a general view of UG-2 ore.

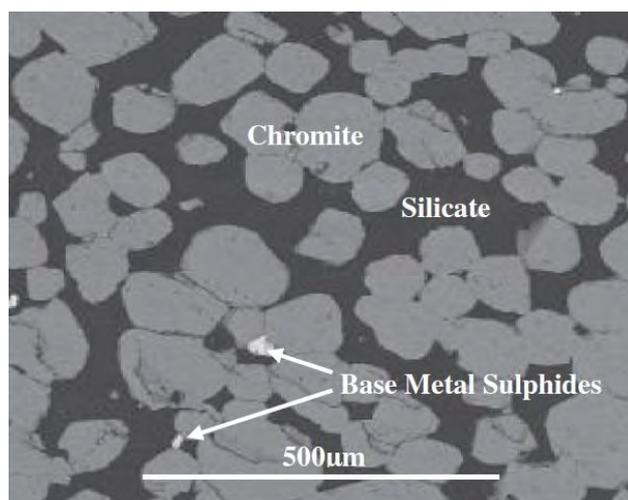


Figure 2.2 A general view of UG-2 ore (Hay and Roy, 2010, pg 856)

The base metal sulphides, although occurring in trace amounts, are important because they are closely associated with the platinum-bearing minerals (Valenta, 2007). Table 2.2 shows the main types and percentages of major sulphides.

Table 2.2 Different Types and Percentages of sulphides in UG-2 ore (Nel *et al.*, 2005)

<i>Main types of sulphides</i>	<i>Percentage (%)</i>
Pyrrhotite (FeS)	50
Pentlandite (Fe,Ni) ₉ S ₈	35
Chalcopyrite (CuFeS ₂)	10

The copper (0.005%) found in UG-2 ore occurs mainly as chalcopyrite and nickel (0.025%) occurs as pyrrhotite or pentlandite (Valenta, 2007).

2.2.2 Platinum recovery from UG-2 ore

An overview of the platinum recovery process is found in Figure 2.3. In the first process of platinum production the UG-2 ore is concentrated by two stages of milling and flotation to produce a feed to the smelter. This is followed by two stages of refining. In the first stage base metals, such as copper and nickel, are extracted and then precious metals such as PGM and gold. (Valenta, 2007).

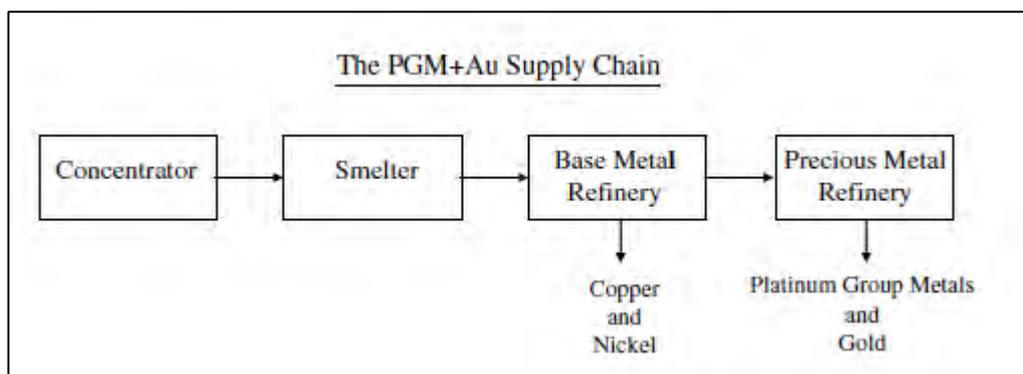


Figure 2.3 Recovery of platinum (Valenta, 2007, pg 980)

A typical concentrator circuit commonly known as an MF2 (mill-float-mill-float) circuit is shown Figure 2.4

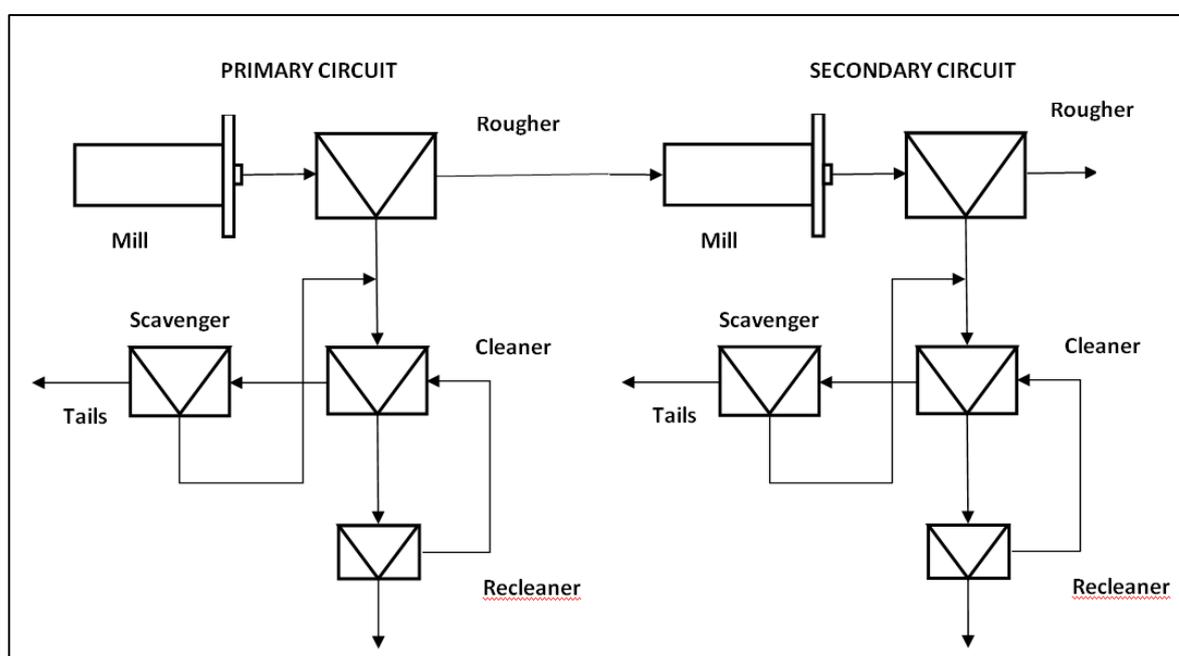


Figure 2.4: Typical MF2 Circuit (adapted, Valenta, 2007, Pg 981)

Primary milling reduces the ore size to about 35% passing 75 μm (Valenta, 2007). The milled slurry is fed to a bank of rougher cells. The rougher concentrate is then fed to the cleaner cells and recleaner cells, where the ore is further upgraded. The tailings from the cleaner cell are sent to the scavenger to retrieve any lost PGMs. The above process is referred to as the primary flotation circuit. The primary rougher tails enter the secondary circuit. Secondary milling further reduces the particle size to approximately 75% passing 75 μm (Valenta, 2007) and the subsequent flotation is identical to the primary circuit. The recleaner concentrate contains the

high grade material of approximately 120g/t to 300g/t (Valenta, 2007). The circuit produces a high grade product and a low grade product.

Several variations to this circuit are possible, including the use of flash flotation and modifications to the cleaner circuit configuration, as seen in circuit used at Northam Platinum in figure 2.5.

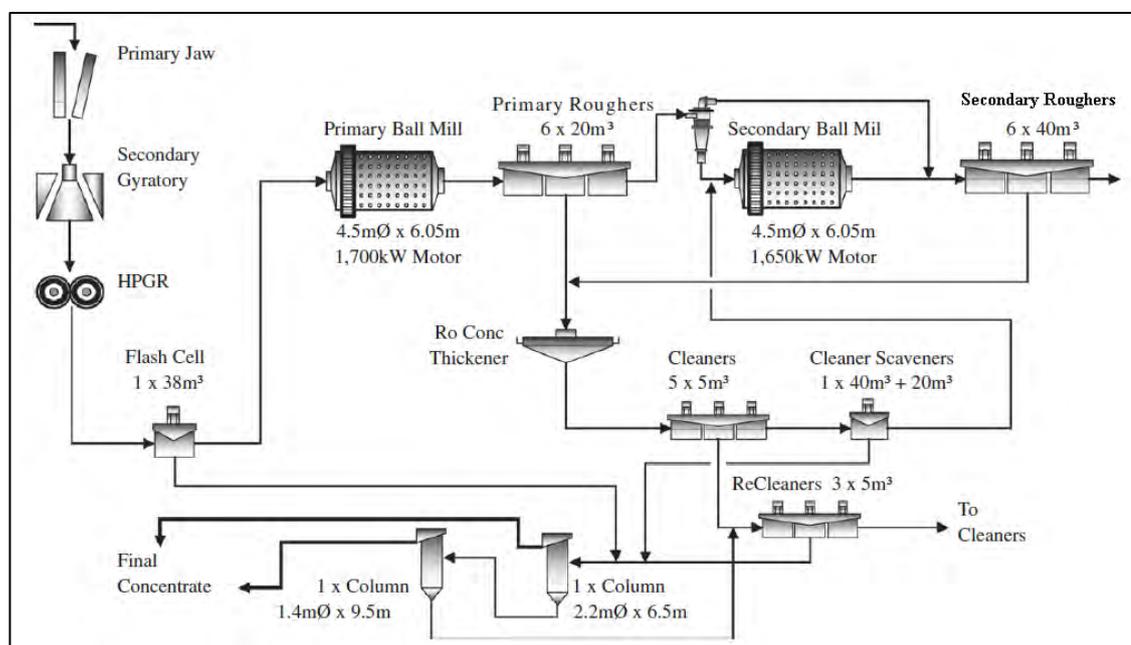


Figure 2.5 Northam Platinum process flowsheet (Hay, 2010, pg 869)

Above is the schematic view of the UG-2 circuit currently used by Northam Platinum. The primary jaw crusher reduces the ore size to 80% passing 6 mm. The ore is further reduced to 80% passing 1.5 mm and 10% passing 75 μm by the high pressure grinding rolls (HPGR). The flash cell produces a concentrate of 300g/t PGM with a corresponding recovery of 40%, as well as 5% Cr_2O_3 . The flash product is sent directly to the column cells (Hay, 2010).

The flash tails are further reduced in size to 30% passing 75 μm during primary milling. The milled slurry is fed to the primary rougher flotation cells, which operate at 40% solids by mass. The rougher tailings are diluted and classified in a cyclone and the classifier underflow is fed to the secondary mill to achieve a grind of 65% passing 75 μm before proceeding to the secondary rougher flotation. The combined concentrates from the roughers are thickened and fed to a set of cleaners and then recleaners. A series of two column cells produce the final concentrate (Hay 2010).

2.2.3 Processing problems on a UG-2 concentrator

One of the problems faced when processing UG-2 ore is its high chromite content. Ultra-fine particles of chromite may be entrained into the concentrate. A concentration of chromite that is above 3% will result in complications in the downstream smelting process (Valenta, 2007). The overall PGM recovery may vary between 75% and 90%. This is as a result of multiple factors such as operation procedures, plant capacity and changes in mineralogy. (Valenta, 2007)

2.3 GENERAL FLOTATION THEORY

2.3.1 Froth flotation

Froth flotation is a method where particles are physically separated depending on their hydrophobicity (Finch and Dobby, 1990). Figure 2.6 illustrates the principles of the process. The valuable minerals present in the ore become hydrophobic due to the addition of a collector. The hydrophobic particles become attached to air bubbles that rise to the surface and collect in a froth layer.

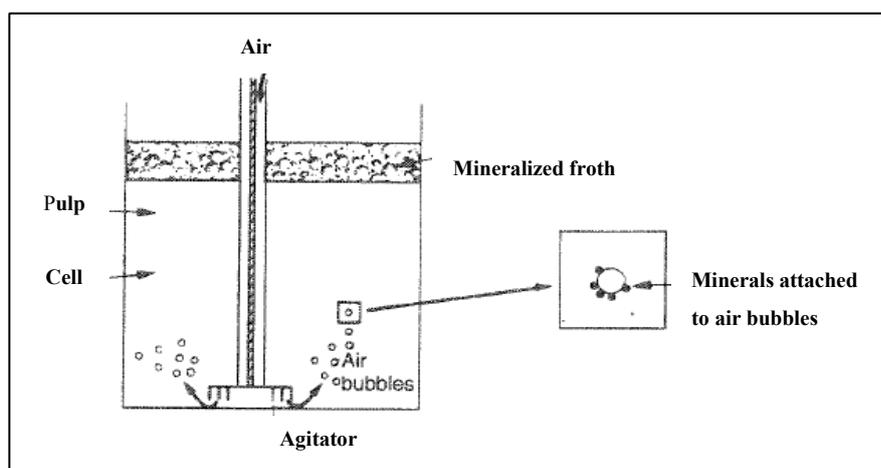


Figure 2.6 Principle of Flotation (Wills, 1997, pg 259)

The froth overflow is collected over time as a concentrate. The hydrophilic gangue (unwanted) material remains in the pulp. Some gangue mineral is recovered in the concentrate by entrainment (Wills, 1997).

Froth flotation has been used for a variety of applications including the separation of coal from ash forming minerals, the separation of phosphates from silicates and the separation of sulphide minerals from silica gangue. (Finch and Dobby, 1990) The later is of key importance in this investigation.

2.3.2 The use of reagents

Froth flotation has been used for the separation of a wide range of minerals. This is possible because reagents are used to alter the surface properties of the minerals in terms of their hydrophobicity. Table 2.3 describes the function of different flotation reagents and their functions.

Table 2.3 Flotation Reagents and their functions (Wills, 1997, pg 260)

<i>Reagent</i>	<i>Purpose</i>
Collector	Reacts with mineral surfaces and makes them water-repellent and attachable to air bubbles
Frother	A surface-active reagent that aids in the formation of an air induced froth layer. It stabilizes the froth layer and facilitates transport of floated minerals
Depressant	Retards adsorption of collector onto gangue preventing undesired product from floating
Activator	Enhances the action of the collector by altering the mineral surfaces so that the collector can act.

The reagents used for the flotation investigations were an anionic, sulphide mineral collector, SIBX (Sodium Isobutyl Xanthate), a Senmin frother (XP200) and Carboxymethyl Cellulose (CMC) was used as a depressant. The use of an reagent was not considered.

2.3.3 Grade versus recovery curves

There is no general method of expressing the separation efficiency of flotation; however, grade versus recovery curves are used widely. The grade or assay refers to the mass of the desired mineral per ton of ore. The recovery refers to the proportion of metal present in the original feed what was recovered in the concentrate (Kawatra, 1984). These curves are most useful for comparing separations where both the grade and the recovery are changing.

In figure 2.7 a set of grade versus recovery curves are illustrated. A 100% recovery corresponds to the lowest grade, i.e. the feed grade. Similarly, the highest grade achievable by physical separation will be the grade of the pure mineral. As the illustrated curve moves in the indicated direction, the grade increases for a given recovery. In figure 2.7 it is therefore shown that curves that are higher and towards the right show a better grade and recovery relationship than curves

that are lower and towards the left.. One cannot opt for a high recovery at the expense of the grade due to the cost incurred during smelting. Ultimately a balance between the grade and recovery is required.

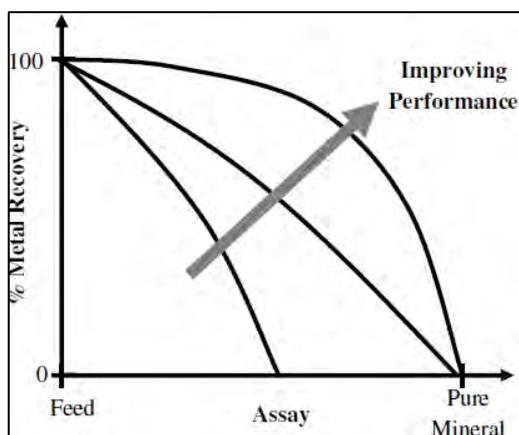


Figure 2.7 Illustration of grade recovery curves (Kawatra, 1984, pg 482)

2.4 THE MECHANICAL FLOTATION CELL

2.4.1 Basic principle of operation

The conventional mechanical cell has been used as the standard flotation equipment for over a century. It consists of a square or cylindrical cell with a mechanically driven impeller.

The impeller mixes the slurry within the cell as shown in figure 2.8. Its shearing action disperses the incoming air into fine bubbles while maintaining the particles in suspension. The surface of the desired mineral is rendered hydrophobic by addition of reagents and particles with exposed surfaces of this mineral can attach to bubbles. Generally a number of such cells are combined consecutively, creating a bank of cells (Wills, 1997). Figure 2.8 illustrates the operation of a standard Denver flotation machine.

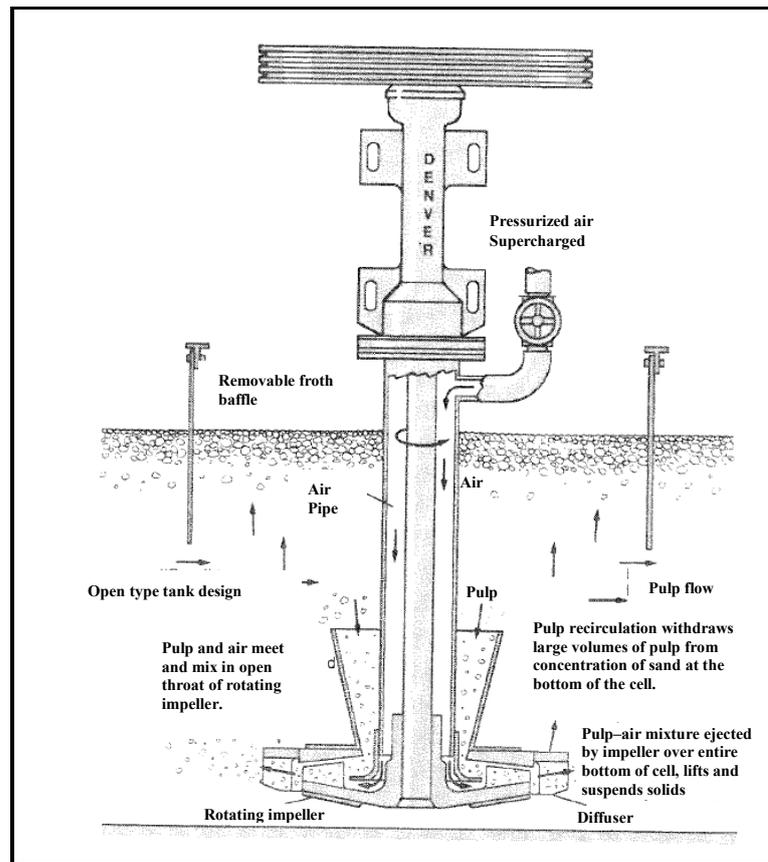


Figure 2.8: Typical Denver D-R flotation machines (Wills, 1997, pg 300)

2.4.2 Industrial Trends

The operational principle of mechanical cells has not changed significantly since 1912 (Fuerstenau *et al.*, 2007) although certain aspects have been improved. One of the most noticeable trends in mechanical cell flotation, in recent years, was an increase in the cell size. In the 1960's cells were typically 5.7 m³ and by the 1980's they increased to 20m³ (Wills, 1997). Flotation cell volume has increased tenfold in the last 20 years and one hundred fold since the 1940's (Miller and Parekh, 1996). Currently, cells of 250m³ capacity are in operation in certain plants (Yianatos *et al.*, 2006). One of the reasons for this shift was that concentrators have opted for using fewer cells with a large capacity as opposed to using many small cells. This has resulted in reduced floor space, simplified control systems and it has reduced required capital and power usage (Miller and Parekh, 1996).

2.4.3 Limitations of the mechanical cell

The limitations of mechanical cells have become apparent. Studies have indicated that the mechanical cell was ineffective when dealing with fine particles or siliceous gangue (Honaker, 1994). Fine particles have a low probability of collision with air bubbles and are more prone to entrainment

Column flotation cells have been used as cleaners in a number of applications because froth washing can be used to reduce the entrainment of unwanted minerals.

2.5 COLUMN FLOTATION

2.5.1 Basic principle of operation

Column flotation is debatably a breakthrough in the field of mineral processing (Finch and Dobby, 1990). Fuerstenau *et al.*, (2007) placed columns in the category of pneumatic cells. Pneumatic cells differ from mechanical cells mainly because they introduce air through a diffuser or sparger instead of using an impeller. Figure 2.9 illustrates a typical column cell.

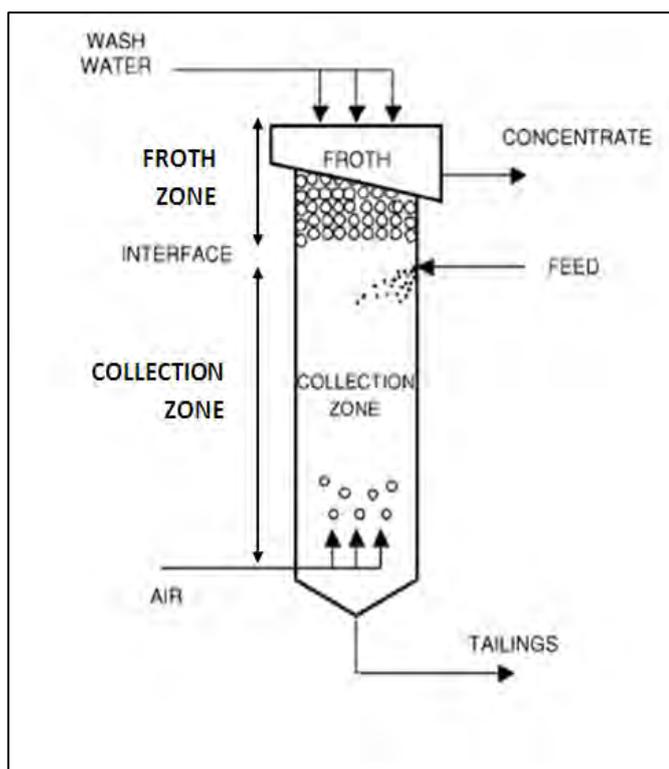


Figure 2.9: Typical column cell (Bergh and Yianatos, 2003, pg 68)

Finch and Dobby (1990) suggested that the feed port of the column cell should be located one third the distance from the top of the cell. Columns were initially designed to operate in a counter current fashion (Miller and Parekh, 1996). This implies that the feed contacts rising bubbles generated at the bottom of the column as it moves down the column. Hydrophobic particles that attach to the bubbles form “bubble-particle aggregates” which rise to the surface (Fuerstenau et al., 2007).

The column cell may be divided into two zones, namely the collection zone and the froth zone, as indicated in Figure 2.9. There are interacting bubble-particle dispersions in the collection zone. The bubbles carry the particles to the surface, forming a layer in the froth zone. Non-floatable material exits from the bottom of the column as tailings (Wills, 1997).

The bubble generation system is one of the unique features of column flotation. Froth washing is another key feature whereby water is injected into the froth layer to increase the grade of the concentrate by removing gangue particles which accumulate in the froth by mechanical entrainment (Finch and Dobby, 1990). The use of these unique features has often raised the question as to whether the column cell would be a plausible alternative to the use of mechanical cells.

2.5.2 The evolution of the column cell

The conventional column cell, known as the “Canadian Column”, was patented in the early 1960’s by Boutin and Tremblay (Finch and Dobby, 1990). It used an internal sparger system consisting of perforated pipes through which a frother and water mixture was passed together with air. Two flotation columns were subsequently installed as cleaners in the molybdenum circuit at Mines Gaspé in Canada (Wills, 1997).

Over the next half a century the use of column flotation expanded to include variety of ore types. Column cells have been used in the capacity of roughers, cleaners and scavengers (Wills, 1997). Many different designs have emerged such as the packed column (Honacker, 1994) with the hope of benefiting from decreased mixing within the cell. Columns have been fitted with baffles that increase the bubble path length and the probability of bubble-particle collision. (Kawatra and Eisele, 1993).

Some designs were highly unconventional, such as the Jameson cell (Fig. 2.10(b)). The column was divided into two sections. The first segment consisted of a downcomer section where slurry and air were intimately mixed (co-current) in a venturi, to shear the air into fine bubbles. The

second segment allowed for the separation of the froth and the pulp. Froth washing was also used (Jameson, 1988).

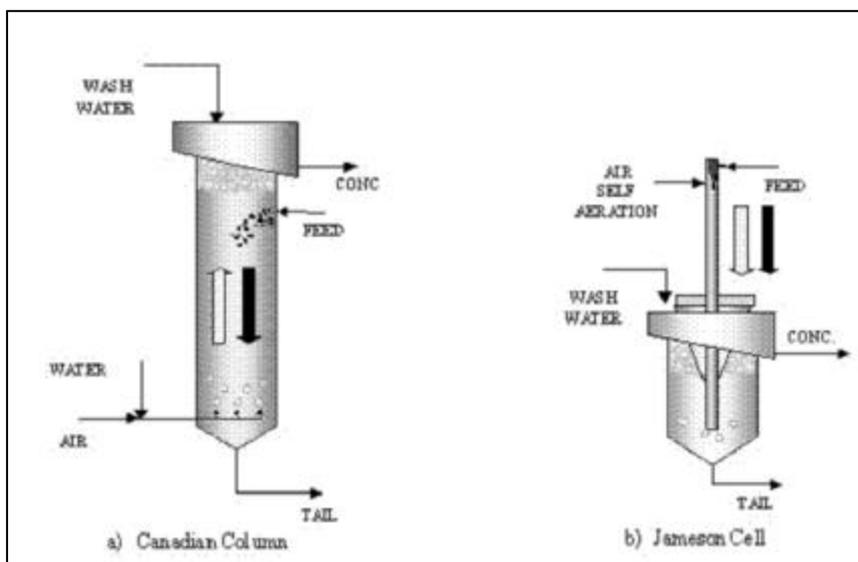


Figure 2.10: Schematic view of Canadian column (a) and Jameson cell (b)
(Bergh and Yianatos, 2003, pg 71)

The Canadian column (Fig. 2.7(a)) is still used widely today, although bubble dispersion mechanisms in different columns do vary considerably (Fuerstenau *et al.*, 2007). Packed columns did not show a consistent performance advantage on a laboratory scale (Finch, 1994). It was decided that the batch test would be performed in a device similar to the conventional Canadian cell.

2.5.3 The choice between column and mechanical cells

Fuerstenau *et al.*, (2007) considered the column cell an advantage for a number of reasons. The authors mentioned that the column cell would be able to achieve a better product without sacrificing recovery or, in other words, it would give an improved separation. The column cell requires less capital and the operational costs are also lower than a mechanical cell. The column cell is also more adaptable to automatic control. Jena *et al.*, (2008) added that the column cell can reduce the number of stages in the operation and that the column is able to handle a finer feed. The authors also suggested that the amount of collector needed in a column cell is lower than the mechanical cell. The construction and design of a column cell is also simpler. The emphasis in column flotation was on height rather than diameter and therefore floor space can be efficiently utilized.

Finch (1994) identified certain challenges that limit the use of column cells, such as axial mixing within the column and the blockage of spargers. Columns of substantial height may be difficult to install.

Miller and Parekh (1996) noted the problem of inconsistent results with column cells. Jena *et al.*, (2008) mentioned that columns require more water, and therefore more frother, than a mechanical cell. However, this seems to be entirely based on the size of the column cell and the operating conditions.

2.5.4 The use of column flotation for various ore types.

Column flotation has been studied as an alternative to the mechanical cell for a variety of different ores. Hacifazlioglu and Sutcu (2007) showed that the ash content of coal was reduced from 47.5% to 19.5% by using conventional flotation. The authors added that the ash content was reduced to 15.6% when column flotation was used. However, combustible recovery in the column cell (81.85%) was slightly less than the mechanical cell (82.32%).

The coal industry has not been alone in testing column cells. Mount Isa mines were among the first to implement column flotation for the recovery of zinc in 1987. Three columns of 2.5m diameter and 13m height were installed to upgrade the lead/zinc rougher concentrate. The performance of the new plant circuit with columns was better than the pilot results (Espinosa-Gomez *et al.*, 1989).

A pilot scale column flotation was also investigated at two different copper concentrators in Turkey. The Kure copper plant required very fine grinding for an adequate level of liberation. This resulted in entrainment of fine gangue and the maximum grade of the final concentrate was 16% Cu. Column flotation tests were conducted at the plant site using two different columns. The experiments proved that the grade of the concentrate could be increased from an average of about 14% Cu to 25-26% Cu (Sirkeci, 2003). This was clear evidence that column cells have proven quite successful for various minerals and there is definitely scope for use of column cells in the flotation process. It is still unclear, however, as to where exactly in the flotation circuit a column cell would be most effective.

2.5.5 The use of column flotation in cleaner cells

Yahyaei and Banisi (2006) investigated the effect of using column cells as cleaners and re-cleaners at a pilot plant on a copper mine in Iran. The separation efficiency increased by 7%

when the conventional cleaner cells were replaced with column cells. An improvement of 10% was noted when the column cell was used as both a cleaner and re-cleaner. In addition a reduction in energy costs by 76% and 83% was reported for the cleaner and re-cleaner respectively.

The flotation of fines has been listed as one of the major problems facing the mineral processing industry (Rule and Anyimadu, 2007). Since column flotation has been noted to deal well with fine material it may prove very useful in the capacity of cleaner or re-cleaner cells.

Cleaner flotation is aimed at improving the concentrate grade and the froth at this stage in the flotation process is highly mineralised. The length to diameter ratio in a column cell allows for deeper froths than a mechanical cell and this may suit the requirements of cleaner flotation.

The performance of column cells as cleaners has yielded sufficient interest to warrant further investigation. The subsequent section elaborates on the use of cleaner columns specifically for platinum processing on an industrial level.

2.5.6 The use of column cells as cleaners in platinum flotation: an industrial trend.

In June 2004 Northam Platinum upgraded their re-cleaner column with a newly installed external sparger system. A PGM recovery of 81% and an upgrade ratio of 105 was obtained with the new sparger system. The previous column cell with an internal sparger had obtained 79.7% recovery and an upgrade ratio of 83. The chromite content was reduced from 5% to 2.8% in the improved column. The mechanical cell cleaners obtained a PGM recovery of only 74% but at an upgrade ratio of 125. The authors suggested that an upgrade ratio of 125 and recovery of 80% PGM was achievable in the improved column (Minnaar *et al.*, 2005).

Northam Platinum's investigations highlight the potential advantages of using column cells for the flotation of platinum ore, particularly in a cleaner capacity. Their findings form a credible basis for this investigation.

2.6 FACTORS EFFECTING THE OPTIMIZATION OF THE COLUMN CELL

2.6.1 Depressant dosage

The function of a depressant is to it is to inhibit the flotation of a given mineral. Depressants prevent collectors from adsorbing onto particular mineral surfaces. Depressants increase

selectivity by allowing certain minerals to float unimpeded whilst preventing others from doing so. (Kawatra, 1984).

2.6.1.1 Mechanism of depression

Laskowsky and Pugh (1992) described two ways by which a depressant was able to perform its function:

- (i) By preventing the collector from adsorbing on to existing hydrophilic minerals e.g. simple inorganic depressants such as Sodium Sulphide (Na_2S).
- (ii) By adsorbing onto a mildly hydrophobic mineral and rendering it hydrophilic e.g. polymeric depressants.

Polymeric depressants are commonly used in the platinum industry. Bradshaw *et al.*, (2005) described polysaccharides as long-chain macromolecules that adsorb onto the gangue mineral surfaces, making them hydrophilic. The amount of gangue recovered is thereby decreased. Rhodes (1981) hypothesized long before that hydrogen bonding is the determining mechanism in the adsorption of polymeric depressants on readily floating silicates.

2.6.1.2 The possibility of depressants negatively effecting sulphide recovery.

Platinum group minerals are associated with base metal sulphide minerals, as explained in Section 2.1.1. If the depressant dosage is above a certain optimum level, the inhibition of valuables may occur as an undesired consequence. Wiese *et al.*, (2007) explained the possible reasons for this occurrence:

- If the sulphide mineral is not fully liberated, it may also be depressed as the polymers are inclined to absorb onto any hydrophilic particle.
- If the adsorption of the relevant collector on the sulphide particle is weak or incomplete then the particle may still be depressed by the polymer.
- Polysaccharide molecules are substantially heavier than those of the collector (by a factor between 500 and 1500). It can therefore interfere with the attachment of the particle onto a bubble.

Whitney and Yan (1996) also stated that sulphide minerals may be blocked or caught in the formation of large flocculated particles. It is evident that the use of depressants may also result in secondary effects that can have a negative outcome on the flotation of sulphide minerals.

Bradshaw *et al.*, (2005) warned that the increased depressant dosage reduces the gangue content of the froth thereby decreasing froth stability.

A delicate balance must be reached in order to use depressant effectively. Hence, the choice of depressant is of crucial importance and is discussed further in the subsequent section.

2.6.1.3. Choice of Depressant

Carboxymethyl cellulose (CMC) and guar gum are two of the most commonly used polymeric depressants in the platinum industry. Comparative studies favoured the use of CMC (Whitney & Yan, 1996; Wiese *et al.*, 2007). One of the main reasons behind this choice was that an excess of CMC did not result in such a pronounced decrease in sulphide recovery as Guar gum (Whitney & Jan, 1996). CMC also ionises in solution and becomes negatively charged, thus causing the gangue surfaces to become negatively charged. At high dosages the gangue particles start to repel each other creating strongly dispersed pulps (Bradshaw *et al.*, 2005; Wiese *et al.*, 2007)

2.6.1.4 The use of CMC as a depressant in column flotation

CMC adsorbs onto naturally hydrophobic silicates by hydrogen bonding. Polymer bridging occurs between the long hydrocarbons resulting in flocculation. Higher CMC dosages allow for the formation of bigger and longer flocculated particles. Impeller agitation in a mechanical cell tends to scrub the absorbed CMC off the depressed mineral, which may then be refloated. However, column cells require less vigorous agitation so the above phenomenon is not as likely to occur.

A variety of depressant types are available, with complex mechanisms, but in most instances they are not well understood. This makes depressant behaviour more difficult to control than other reagents (Wills, 1997). The platinum industry has conducted extensive research into the effect of depressants on PGM recovery, but this information may be specific to the ore tested and very little is publically available. It would therefore be of interest to investigate the effect of depressant dosage in the column cleaner flotation as compared to a mechanical cell. This is especially significant in light of the studies with the nickel/magnesium ore (Whitney & Yan, 1996).

2.6.2 Cell Height

Original column cells, like the Canadian units, were typically between 12m to 14m high (Wheeler, 1985), as was standard for cells of that era. More consideration was placed on the number of columns and the column diameter than height (Finch, 1994). However, the need for column height became a debateable issue. Investigations into the effect of the length of the collection zone as well as the height to diameter ratio have yielded many interesting possible repercussions.

2.6.2.1 Longer columns increase mineral recovery

One reason why longer columns were favoured was the effect of height on mixing within the column cell. It was suggested (Finch and Dobby, 1990) that an increase in height results in less mixing within the column. This would imply an improvement in recovery and selectivity.

The problem of low particle collection rates is eliminated in long columns. An increase in column height implies an increase in the bubble path length and a greater chance of bubble particle interaction. Yalchin (1994) was in agreement with this theory and modified the design of the column cell so as to increase flotation rates. Kawatra and Eisele (1993) opted for baffling as an alternative solution to increasing the particle collection rate instead of increased cell height.

Garibay *et al.*, (2002) proposed that tall columns would improve the metal recovery as long as the solids content of the slurry was less than 20%.

2.6.2.2 Longer columns decrease mineral recovery

The risk of bubble overloading is much greater when the column height is increased (Garibay *et al.*, 2002). The residence time of a particle is increased in longer columns and Ityokumbul (1996, cited in Finch, 1994) suggested that this did not necessarily imply better recovery.

Garibay *et al.*, (2002) further elaborated that the height of the column, directly related to the residence time, improves the mineral collection up to a certain point. The mineral collection is limited by the carrying capacity (surface bubble saturation) of the bubble. When mineralized bubbles coalesce, they lose a fraction of the particles that they transport due to a reduction in overall surface area. The recovery and the processing capacity of the column decreases in response. This phenomenon is effectively known as bubble overloading. Bubble overloading is

more pronounced when dense slurries of floatable material are used (Garibay *et al.*, 2002). Figure 2.11 demonstrates that when the solids content of the feed is relatively high (30%), increasing the height of the collection zone decreases the percentage recovery (Maksimov 1991, cited in Garibay *et al.*, 2002). To substantiate this hypothesis mineralised bubbles were found in the tailings (Garibay *et al.*, 2002).

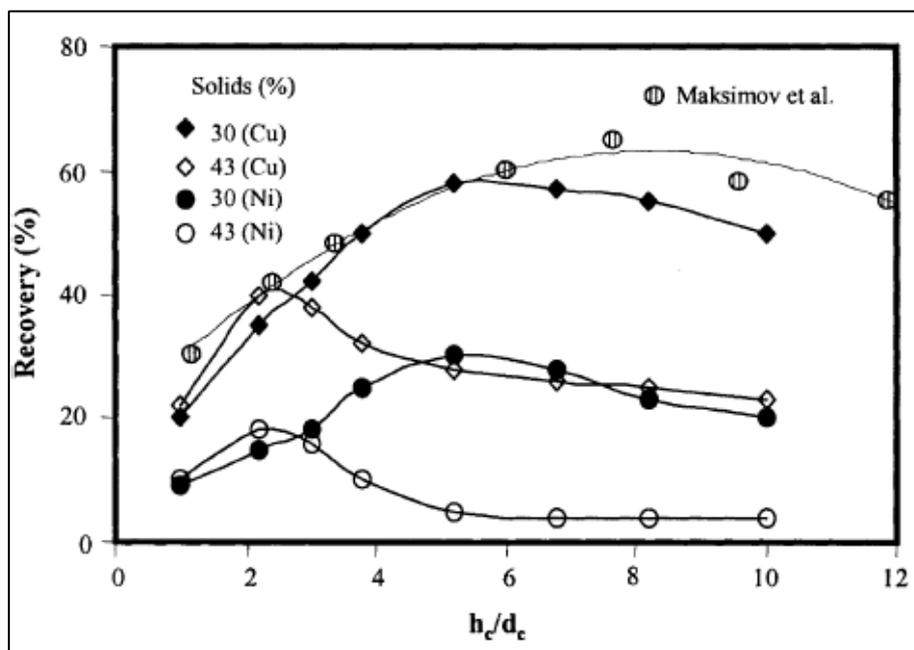


Figure 2.11 The effect of height to diameter ratio on recovery for different percent solids (Marzikov 1991, cited in Garibay *et al.*, 2002, pg 326)

Garibay *et al.*, (2002) was able to increase the recovery of silica from 55% to 78% when the column height was decreased from 4m to 1.5m for a slurry with 30% solids by mass.

2.6.2.3 The optimum height to diameter (h/d) ratio for improved recovery

The degree of mixing within the column cell at high h/d (height to diameter) ratios is decreased, thereby increasing the recovery. The gas flowrate per unit volume, on the other hand, is lowered in order to maintain the same air flux at the surface. Hence, the column approaches overloading. Yianatos *et al.*, (1987) identified the maximum h/d ratio to be dependant on the gas carrying capacity. Under typical column operating parameters, Yianatos *et al.*, (1988) found that a h/d ratio of 10 is a reasonable compromise. Maksimov *et al.*, (1991 cited in Garibay *et al.*, 2002) later suggested that with an h/d ratio greater than 5 would result in decreasing the mineral recovery.

The size of solid particles in the slurry is another contributing factor to obtaining the optimum h/d ratio. Bubble-particle collision probability models have been used to investigate how the size of particles would affect the optimum cell height. The minimum column height was defined as that height resulting in at least one collision between the particle and a bubble. The models show that height is a strong function of particle size with heights increasing from less than 1 m for particles greater 50 μ m to heights that exceed 10m for particles smaller than 10 μ m (Oteyaka and Soto, 1994).

Investigations into the effect of cell height have been contradictory. Research has sometimes shown that short columns suffice and at other times established that they did not (Nicol *et al.*, 1988). Across the globe, column size has varied quite drastically. A plant in Norway was designed with short (less than 5 m) and tall columns in series (Finch and Dobby, 1990). The effect of column height is a plausible area for further research in column flotation.

2.6.3 Froth Depth

2.6.3.1 The General Trend

It is well known that an increase in the froth depth improves the product grade at the expense of the recovery. This has been confirmed in mechanical flotation cells with different ores such as pyrite (Englebrecht and Woodburn, 1975) and galena (Feteris *et al.*, 1987).

Bisshop and White (1976) offered a possible reason for this trend. The authors suggest that the drainage of hydrophilic particles from the froth, back to the pulp, was dependent on the residence time of the particles in the froth. The residence time was directly proportional to the froth height. A deeper froth provides a longer time for particles to drain from the bubble surface and drop back to the collection zone. Hence, an increase in froth depth increases the grade but decreases the recovery of the concentrate.

2.6.3.2 Effect of varying froth depth in column cells

The separation efficiency of column flotation is highly dependent on the froth depth (Yianatos *et al.*, 1988). Engelbrecht and Woodburn (1975) and Feteris *et al.*, (1987) have demonstrated that froth stability is also dependent on the height of the froth zone.

Tao *et al.*, (2000) investigated the effect of froth depth in column cells and were in agreement with the grade recovery trend found when using mechanical cells. This was later confirmed by Hacifazlioglu and Sutcu (2007). The latter conducted experiments with coal in a 150 cm high

column with a diameter of 7cm. The authors investigated the effect of varying the froth depth between 15cm, 20cm, 25cm and 30cm and concluded that the optimum froth depth was 25cm. However, for a similar column of 5cm diameter and a height of 170cm, Tao *et al.*, (2000) chose a froth height of 50cm. Column cells may be operated at very deep froths. This may prove advantageous in cleaner flotation where a high grade is desired, as opposed to roughers where recovery is of a higher priority.

However, Engelbrecht and Woodburn (1975) and Feteris *et al.*, (1987) warned that by increasing the froth depth the froth would eventually collapse due to bubble coalescence; The liquid drains away, resulting in thinning of the films between the bubbles. Amelunxen *et al.*, (1988) and Kosick *et al.*, (1988) have also discussed that bubbles larger than 1mm in diameter showed only minimal improvement in selectivity with an increase in froth depth. This provides some insight as to why thin froths are sometimes opted for.

Obtaining a suitable froth depth is important in this comparative study, in light of the above research

2.6.4 Recirculating the pulp in a column cell

Hay (2010) commented on the need for a greater input of energy in column cells:

“It is well known that to maximise recovery, flotation cells treating UG-2 ore require a higher energy input than is normal for any other ore. The reduction in PGM recovery in the column cell illustrates the consequence of a low power input.”

Hay (2010) further stated that the recirculation pump employed at Northam Platinum was sized to pump 7 to 8 times the volume of the feed. This encouraged contact between the bubbles and the particles and increased the energy input to the column cell.

It was suggested by Loveday (personal communication, April 2009) that the recirculation pump in a column may imitate the shearing action of the impeller in a mechanical cell. The recirculation pump could increase the turbulence in the cell and it would also cost less than a mechanical impeller. The recirculation pump would help prevent the coarser ore particles from settling at the bottom of the column.

2.6.5 Froth Washing

Unwanted particles are often physically entrained in the interstices between the bubbles of the froth. Froth washing reduces the amount of gangue particles recovered in the froth by inserting a gentle stream of water within or above the froth (Finch and Dobby, 1990). However, an actual understanding of how wash water behaves in the froth is limited.

It must be mentioned that existing wash water systems are largely based on trial and error, as confirmed by Ireland *et al.*, (2007). Most of the design characteristics are debatable.

2.6.5.1 Types of wash water distributors

An early design of a wash water distributor utilized a single perforated pipe immersed in the froth (Finch, 1994). Ireland *et al.*, (2007) mentioned the use of a vertical array of pipes situated above the froth.

A comparison was made between a “shower head” distributor and a single jet impinging into the froth. Ireland *et al.*, (2007) claimed that a single jet yielded better results. However, Banisi *et al.*, (2003) opted for the shower head system due to its easy maintenance and because it helped maintain plug flow in the column. The latter obtained a 1.5% increase in the final copper grade.

However, a sprinkler system was not able to penetrate stiff froths unless some force was used. The use of jets deserved closer inspection. Although horizontal jets were prone to forming vortex pairs (a pair of counter-rotating vortices that form at the ends of the T-shaped tube), they may be more efficient at “spreading” the wash water than vertical jets. Horizontal jets yielded better results when both jets were submerged in the froth (Ireland *et al.*, (2007).

2.6.5.2 Wash water entry point

The placement of the wash water distributor was an important consideration. Neethling and Cilliers (2001) reported that when the wash water was added toward the back of the froth instead of at the front, near the weir, the concentrate grade was higher.

A froth washing system that is submerged in the froth runs the risk of the apertures becoming blocked with solids. Structures situated above the froth also risk the possibility of the wash water not penetrate the froth (as with the sprinkler system) and short circuiting to the overflow (Ireland *et al.*, (2007).

If a submerged system is preferred then the froth depth at which the system is located would determine the amount of wash water that actually enters the pulp, as compared to that which ends up in the concentrate. This is generally referred to as the bias rate (Kosick *et al.*, 1988).

2.6.5.3 The bias rate

The bias rate refers to the ratio of wash water to the water leaving in the (washed) concentrate. The use of a large bias may result in a higher grade and it also could help to minimize coalescence (Whitney & Yan, 1996). However, an increased bias rate may also lead to an unstable froth (less gangue material to stabilize the froth) which would hamper the column's operation (Ireland *et al.*, 2007).

The optimum bias rate has been a subject of contention. It was initially thought that a bias rate greater than 0.1 cm/s would lead to a stable froth (Finch *et al.*, 1989). Tests on industrial columns yielded a recommended bias rate of 0.3 to 0.4 cm/s (Yianatos *et al.*, 1987). Later (Finch and Dobby, 1990) claimed that the bias rate greater than zero was plausible.

2.6.5.4 The effect of froth washing on grade and recovery

Studies show that as the wash water rate is increases, the metal grade increases and the recovery decreases. Tao *et al.*, (2000) stated that despite a rise in the grade, the recovery does not decrease much. Overall, it was obvious that an increased rate of wash water is beneficial up to a certain limit (Hacifazlioglu & Sutcu, 2007).

Froth washing has the potential to significantly improve column flotation results, with appropriate investigation into design and performance, taking the above factors into consideration.

2.7 LITERATURE SURVEY SUMMARY AND RESEARCH PROPOSAL

- The rate of mining and processing of the UG-2 reef of the BIC has increased dramatically, in recent years, as an alternative to the diminishing Merensky reserves. The UG-2 reef consists mainly of chromite (75%) but it has a head grade of 4-5.5 g/t of PGM. (section 2.2)
- The process of platinum recovery requires that the ore is first pulverised and upgraded in the concentrator plant by two stages of milling and flotation.

- Traditional flotation cells comprised of a rectangular or circular cell and a motorized impeller with air injection.
- The limitations of mechanical cells were noted for fine particles that had a low probability of collision with air bubbles and were more prone to entrainment.
- Column cells were introduced as a possible alternative to the conventional mechanical cell. Various forms of air dispersion in column cells have been developed to reduce costs and improve contact between particles and bubbles.
- Column cells seem to be most suitably used as cleaners. Rougher flotation cells are used to obtain maximum recovery, whereas in the cleaner cells obtaining a high grade is priority. The bubbles in column cells have a large loading of particles and the froth is more stable, when compared to mechanical cells. Hence column cells can be operated with a deep froth, which make it possible to produce a high grade concentrate. Froth washing is also possible.
- Platinum processors, like Northam Platinum, have made convincing modifications to include the use of columns as cleaner cells in their circuit. The PGM recovery improved by 7% with the use of cleaner columns.

A review of the relevant literature has confirmed that column cells may perform better than mechanical cells for cleaner flotation of UG-2 ore.

It is proposed that a column cell and a mechanical cell of equal volume but significantly different heights be used for comparison. The bubble path length in a cell is proportional to the cell height therefore the bubble path length in the column cell would be significantly longer. Hence, the column cell should theoretically obtain a better recovery. The superficial velocity (flowrate of air per cross sectional area) in each cell should be equivalent to aid comparison.

The review of the literature brought into focus various operational parameters that are worth investigation:

- The use of a recirculation pump in conjunction with the column cell in order to improve metal recovery (section 2.5.4) as suggested by Hay (2010).
- Whitney and Jan (1996) hypothesized that the action of the impeller in the mechanical cell scrubs off the absorbed depressant (section 2.6.1). The column cell does not utilize

mechanical agitation. Varying the depressant dosage in both the mechanical and column cells may result in an interesting comparison.

- The action of depressant in the column cell may be related to many factors and not solely the absence of agitation. An investigation into the effect of different depressant conditioning techniques, both vigorous and gentle, could verify whether the action of the depressant in the column cell was based on the degree of agitation.
- Garibay *et al.*, (2002) explained that the performance of the column cell, in response to increasing the column height, is based on the percentage of solids by mass (section 2.6.2). However, different authors expressed conflicting views. This warrants an investigation into the effect of increasing the column height whilst maintaining the percentage of solids by mass and superficial air velocity.

During the commissioning of the column cell the effect of froth depth (as suggested by Tao *et al.*, (2000) and Hacifazlioglu and Sutcu (2007) in section 2.5.3) is taken into account. The possibility of employing froth washing and its various parameters of design (section 2.5.5) are also considered.

CHAPTER 3: EQUIPMENT AND EXPERIMENTAL PROCEDURES

This chapter provides a description of the equipment used in the laboratory and pilot plant tests as well as an overview of the experimental procedures followed.

3.1. LABORATORY EQUIPMENT

The mechanical cell and column cell used in the laboratory investigations on cleaner flotation are described below. Associated equipment, such as the mechanical rougher and ball mill, are also included.

3.1.1 Mechanical rougher cell

A 43L Perspex flotation cell was used for the rougher flotation tests. The cell is depicted below in Figure 3.1.

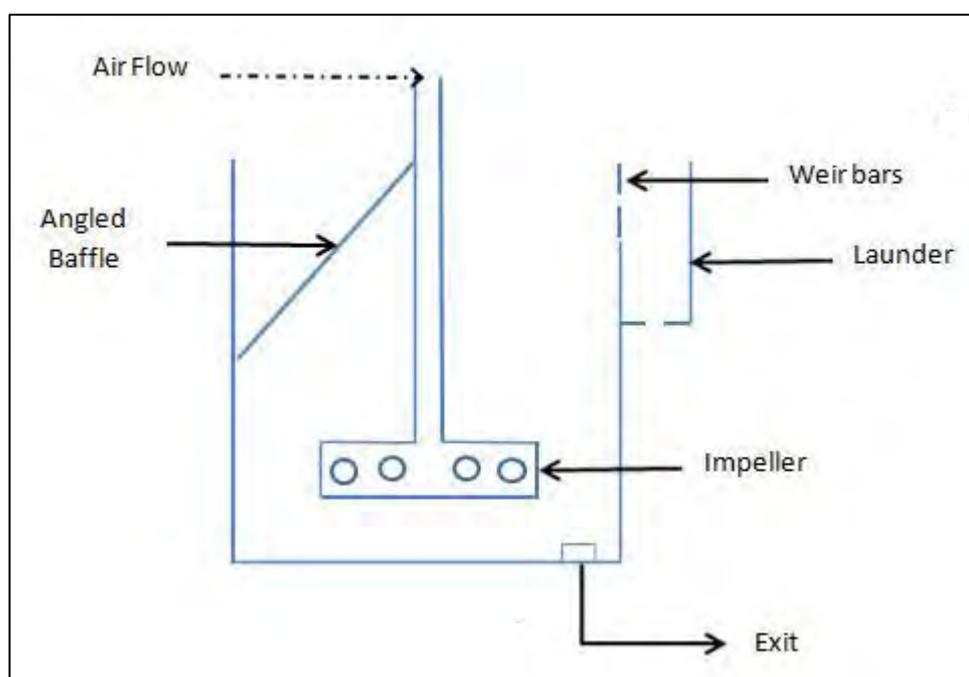


Figure 3.1 Schematic representation of the rougher Cell

The rougher cell was fitted with an Outokumpu impeller. An angled baffle was utilised to assist the flow of froth. The baffle also reduced the possibility of error by eliminating the need to “hand-scrape” the froth. The weir bars on the rougher cell made it possible to perform tests at

different depths, while maintaining the same pulp volume. Air flow into the rougher cell was controlled by a gas rotameter. Figure 3.2 shows the side and top views of the 43 L laboratory rougher cell.

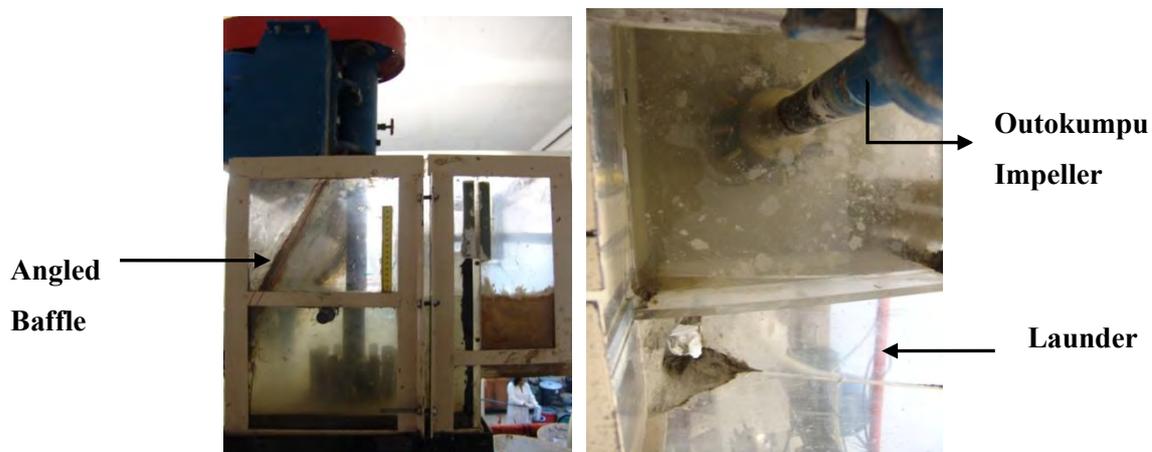


Figure 3.2 Side view (left) and top view (right) of the mechanical rougher cell

3.1.2 Mechanical cleaner cell

A 5 L mechanical cell with a Denver flotation mechanism was used as a mechanical cleaner. Figure 3.3 shows the structure of the stainless steel cell.

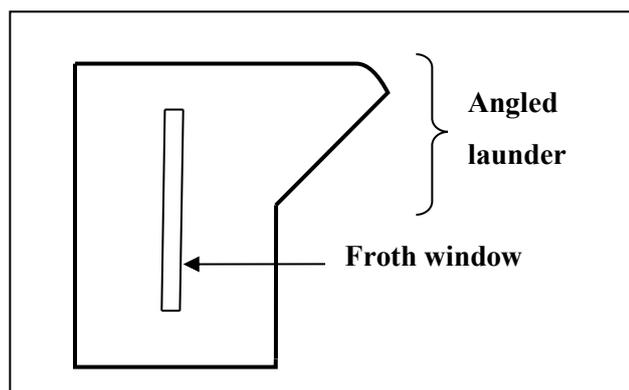


Figure 3.3 Structure of the mechanical cleaner cell

The mechanical cell was a conventional Denver cell. The clear perspex window allowed for the froth-pulp interface to be monitored. Figure 3.4 shows the configuration of the rougher and mechanical cleaner cell. The Denver flotation mechanism included a variable speed motor. The air flow into the impeller was controlled from a compressed air source via a rotameter.

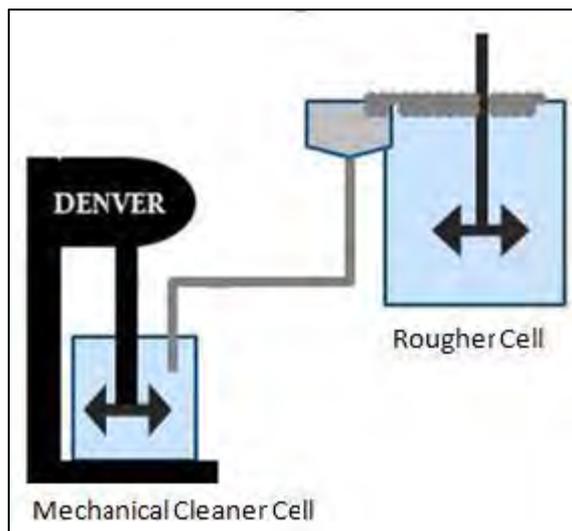


Figure 3.4 Rougher and mechanical cleaner configuration

3.1.3 Column cleaner cell

The column cell was designed to have the same volume as the mechanical cell (5 L). This was an important design consideration because although the two cells were of equal volume, the column height was 7.8 times longer than the mechanical cell (Fig 3.5).

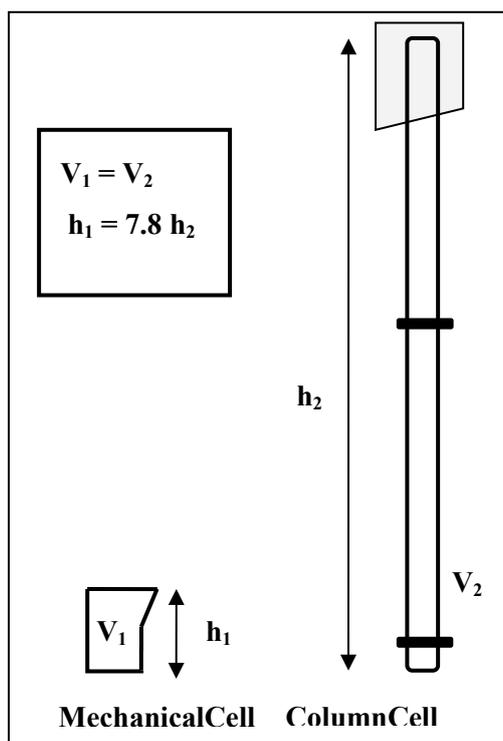


Figure 3.5 Comparative dimensions of the mechanical and column cell

The bubble path length was proportional to the cell height therefore the bubble path length in the column cell was 7.8 times longer. This should logically result in more floatable material being given the opportunity to attach to the bubble. This concept is advantageous as long as the floatable material does not overload the bubbles (Loveday, 1983). The superficial gas velocities were kept constant for a fair comparison between the 5L column cell and the 5L mechanical cell.

A 1.86 m perspex tube with a 6cm diameter was used to construct the column cell to achieve the above specifications. The tube design was modified to accommodate for an internal air distribution mechanism, a recirculation pump, and an angled launder, as seen in figure 3.5

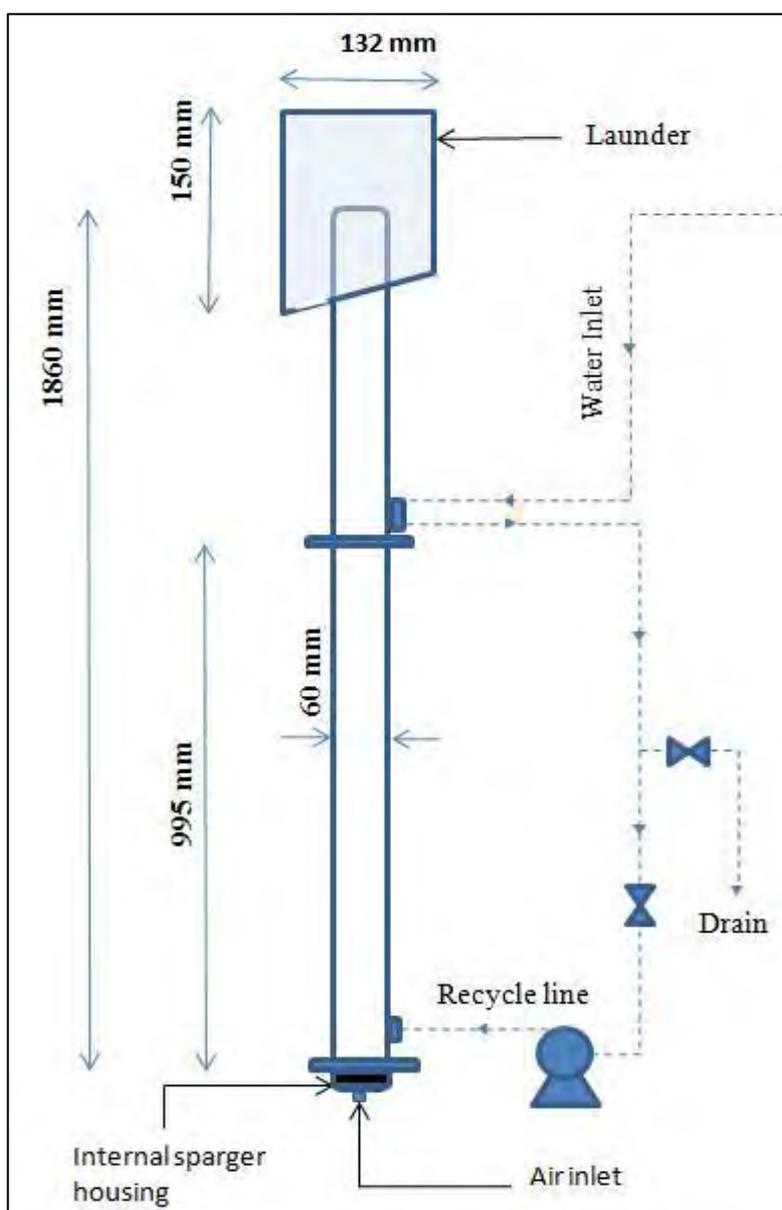


Figure 3.6 Schematic view of the laboratory column cell

3.1.3.1 Air distribution mechanism

The shearing action of the impeller in a mechanical cell produces fine bubbles that are adequate for flotation. Hence, one of the main considerations of the column cell design was its air distribution mechanism. A variety of possible air distributors, such as perforated perspex, rubber tubing and porous foam piping, were tested. In each case a visual comparison of the bubble size was made and the bubbles in the column were found to be larger than the mechanical cell.

Finally, two sintered silica discs of different porosities (E1 and E2) were tested. Compressed air entered at the bottom of the column and was passed through the sintered disc creating fine bubbles. As the bubbles passed through the column a certain degree of coalescence was visible. The E2 disc was found to provide an adequate bubble size. Later on supplier shortages resulted in the use of an E3 disc of smaller pore size (Table 3.1). The E3 disc resulted in the most stable froth and the bubbles produced were visually more similar to the mechanical cell. Coalescence was also less apparent. The three different porosities are defined in Table 3.1.

Table 3.1 Pore sizes for different sintered discs

<i>Sintered disc</i>	<i>Pore size (μm)</i>
E1	100 - 160
E2	40 – 100
E3	16 - 40

Attempts to measure the actual bubble sizes were difficult as the bubble size analyser available was not fully operational. A visual comparison was used to select the best of the sintered discs.

3.1.3.2 The recirculation pump

The height of the column cell was similar to a large, modern mechanical cell therefore it was decided that a recycle of slurry (and air bubbles) would be useful. To investigate the effect of using a recycling the slurry a centrifugal pump was employed. The pulp was withdrawn from a point mid-way up the column. The pulp re-entered the column at the bottom, just above the

sparger, as shown in figure 3.6. The flowrate of the pulp in the recycle was adjusted by means of a control valve. The reasons for using the recycle were manifold. It was initially anticipated that the coarse particles may settle at the base of the column. In that case a recirculation pump would be very convenient. The recycle could also help reduce the bubble size due to the shearing action of the pump.

3.1.3.3 Level Control

It was initially problematic to control the pulp-froth interface. The column diameter was very small (6 cm) therefore any minor adjustment to the airflow resulted in a significant change in the pulp level. Changes in the pulp level were rapid, especially when operating with a deeper froth. Automatic level control was an expensive solution and it was not required for batch experimentation. On the other hand, water could not be added from above as with the laboratory mechanical cells (rougher and cleaner) as this created significant disturbance in the froth. A direct water line was added midway up the column that allowed for quick and effective manual level control without disturbing the froth.

3.1.3.4 Angled launder

The column cell was fitted with a circular launder around the top end. Initial flotation tests showed that fast flowing froth tended to flood the launder and a steep floor angle (45°) was used to help alleviate this problem.

3.1.3.5 Adjustable column height

The column cell consisted of two pieces of perspex tubing separated by a flange. This arrangement allowed for adjustment of the column height. The bottom end of the first piece housed the sparger and the angled launder was attached to the top end of the second piece. The column height was doubled by separating the two pieces and accommodating another 1.86 m tube in between them.

The column remained, in essence, prototype. One difference between the laboratory column and a prototype column was the absence of froth washing, a common feature of column cells. Froth washing was attempted with a porous vertical tube in the centre of column but the froth became unstable it was decided that this line of investigation was terminated. The data was not recorded. Hence, all tests were performed without froth washing.

3.1.4 Conventional Ball Mill

A conventional ball mill was utilized to grind the ore before flotation. The mill (fig 3.7) had an inner diameter of 0.3m and an additional mill shell was added to make it possible to grind 10kg of ore for the rougher cell. The total internal length was 0.51m. The mill was equipped with a variable speed controller and a voltmeter.



Figure 3.7 Conventional ball mill

3.2 LABORATORY EXPERIMENTAL PROCEDURES

This section provides an overview of the experimental procedures for the processing of UG-2 ore from receipt through to the milling, rougher flotation and cleaner flotation. It includes a summary of the methods employed when different parameters were tested on the mechanical cell and the column cell.

3.2.1 Overview of the experimental programme

Two samples of UG-2 ore were received in the course of the investigations. The first sample of UG-2 ore was crushed and passed through a 3mm screen. The ore was then sub-sampled and a standard milling curve was obtained. Initial tests performed with the rougher cell were aimed at obtaining a suitable reagent suite for maximum mass recovery, without excessive froth stability in the cleaner. Once repeatability was achieved in the rougher flotation, the conditions were kept standard for all subsequent cleaner tests.

Commissioning tests were performed in order to set suitable air flowrates and froth depths for the cleaner cells. The cleaner tests were conducted in three series. The first series of tests (Series A) were performed on the mechanical cell and the column cell, using the E2 sparger (E2

column). The cleaner cells were tested under base conditions and at different depressant dosages. The effect of recycling the pulp in the column cell was also investigated. Midway through the investigations the E2 sintered disc fractured. At that point no E2 discs were available and a disc of lower porosity (E3) was used. This proved to be a blessing in disguise because the bubble sizes (in the froth) appeared to be similar to those obtained in the mechanical cell.

Series B refers to the tests conducted in the column cell with the E3 sparger (E3 column). The parameters originally investigated with the E2 column in series A were repeated with the E3 column in series B. In addition, the effect of gently conditioning the depressant, as opposed to conditioning by means of the recirculation pump, was investigated. The tests performed in series A and B were analysed for Cu/Ni content.

Series C refers to tests performed on the mechanical cell and the E3 column but these tests were performed with a new ore sample and analysed for PGM + Au. The investigations conducted in series B were repeated in series C. In the final phase of the laboratory investigations the column height was doubled and standard tests were performed on the taller column.

A summary of all the experimental procedures is covered in the subsequent section.

3.2.2 Ore receipt and preparation

A total of 1100 kg of UG-2 ore was received from Anglo-platinum. The first sample consisted of 800 kg of ore (used in series A and B) that was crushed and sub-sampled in the laboratory. The second sample of 300 kg (used in series C) was crushed on site before being received and sub-sampled in the laboratory.

3.2.2.1 Preparation and sampling of the first UG-2 ore sample (800 kg)

- The ore was passed through a 3mm screen.
- The coarse and fine fractions were weighed. The finer fraction consisted of 70% of the total ore.
- The coarse fraction was crushed using a laboratory jaw crusher.
- The crushed ore was passed through a 3mm screen.

- The fine and crushed coarse fraction were individually blended and separated into 50 kg portions by coning and quartering.
- The 50 kg portions were riffled into 3.5 kg samples (fine) and 1.5 kg samples (coarse).
- The riffled coarse and fine fractions were then combined to form 5 kg portions of ore in the same ratio that was found in the original ore i.e. 70% of fine material.

3.2.2.2 Preparation and sampling of the second UG-2 ore sample (300 kg)

- The second UG-2 ore sample was crushed to below 3 mm on site.
- The crushed ore was blended and separated into 50 kg portions by coning and quartering in the laboratory.
- The 50 kg portions were riffled into 5 kg portions which were then bagged.

3.2.3 Mill Calibration and general milling procedure

A mill calibration was performed to determine the milling time for all subsequent tests.

3.2.3.1 Mill Calibration

The ball mill was loaded with 80 kg of steel balls, 10 kg of ore and 6 L of water. The grinding media consisted of three different sizes of steel balls, namely 25 mm, 35 mm and 40 mm. The mill was operated at these conditions for different lengths of time (20 min, 39 min and 78 min). The mill calibration curve was then obtained (Fig. 3.8).

Generally in industry, a grind of 40% passing 75 μm is typically used in the primary circuit and 80% passing 75 μm for the secondary circuit. It was decided that an average grind of 65% passing 75 μm was targeted for the laboratory work, as per discussion with staff at Anglo American Research on laboratory test procedures. The milling curves for both ore samples (Fig. 3.8) revealed that the required grind was achievable in 45 minutes at a speed of 60 rpm.

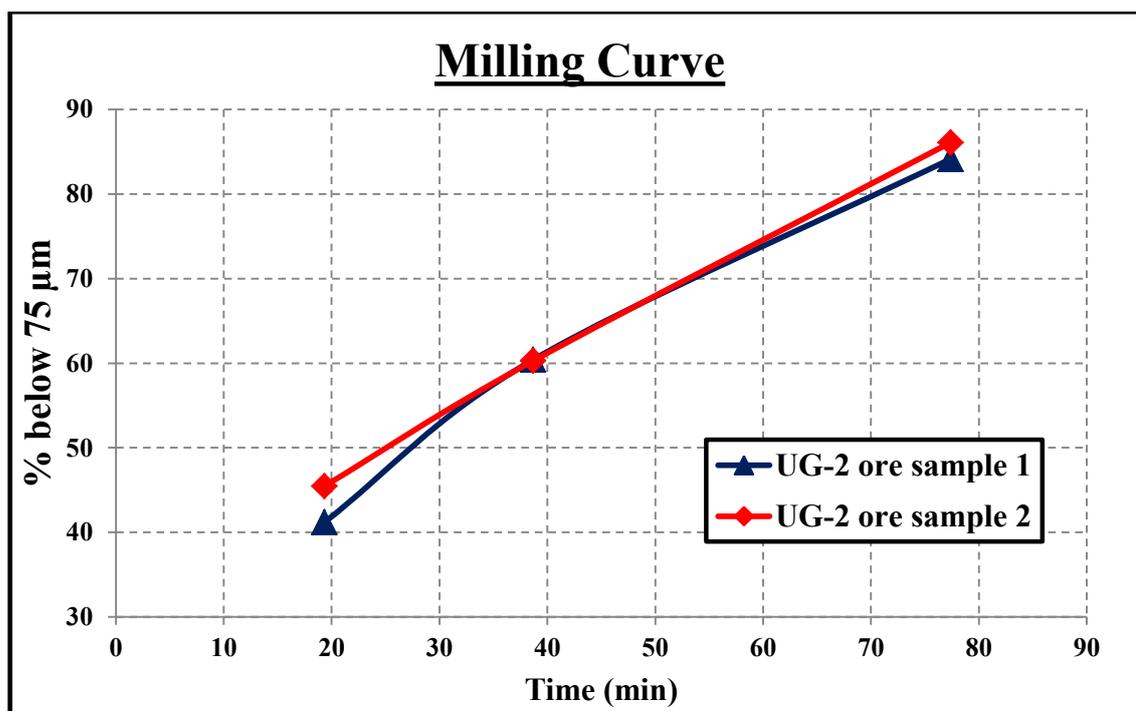


Figure 3.8 Milling curves for ore samples 1 and 2

3.2.3.2 General milling procedure

The general milling procedure that was followed for all subsequent tests is outlined below.

- The mill was loaded with 80 kg of steel balls of three different sizes.
- 10 kg of the sub-sampled UG-2 ore was added to the mill.
- The mill was then closed and bolted.
- 6L of water was through the inlet on the lid of the mill by means of a funnel and the inlet was sealed with a rubber stopper.
- The mill motor was switched on and the speed was set at 60 rpm.
- The ore was milled for 45 minutes.
- The speed was reduced to zero and the motor switched off.
- The mill was unbolted and the mixture of slurry and steel balls were transferred onto a steel screen where the balls were washed and the slurry emptied in 25 L buckets.
- The slurry was then transferred to the rougher cell for the first stage of flotation.

3.2.4 Standardising the rougher flotation

The rougher cell operated with 20% solids by mass. A percentage mass closer to 40% was preferable; however, it was not attainable because the maximum amount of ore that could be loaded into the mill was 10 kg. Initial tests performed on the rougher cell were aimed at attaining the highest mass pull. Hence, no depressant was utilized in the rougher tests. A recommended collector dosage (SIBX) of 150 g/t was used. The rougher cell was operated with a 6cm froth depth. The air flowrate was adjusted to maintain the froth depth.

3.2.4.1 Optimising the frother dosage

In order to standardize the rougher flotation the frother dosage (Senmin XP200) was varied (20 g/t, 30 g/t and 50 g/t) and the rougher concentrate was weighted to obtain the best operating conditions. The following graph (Fig 3.9) was obtained.

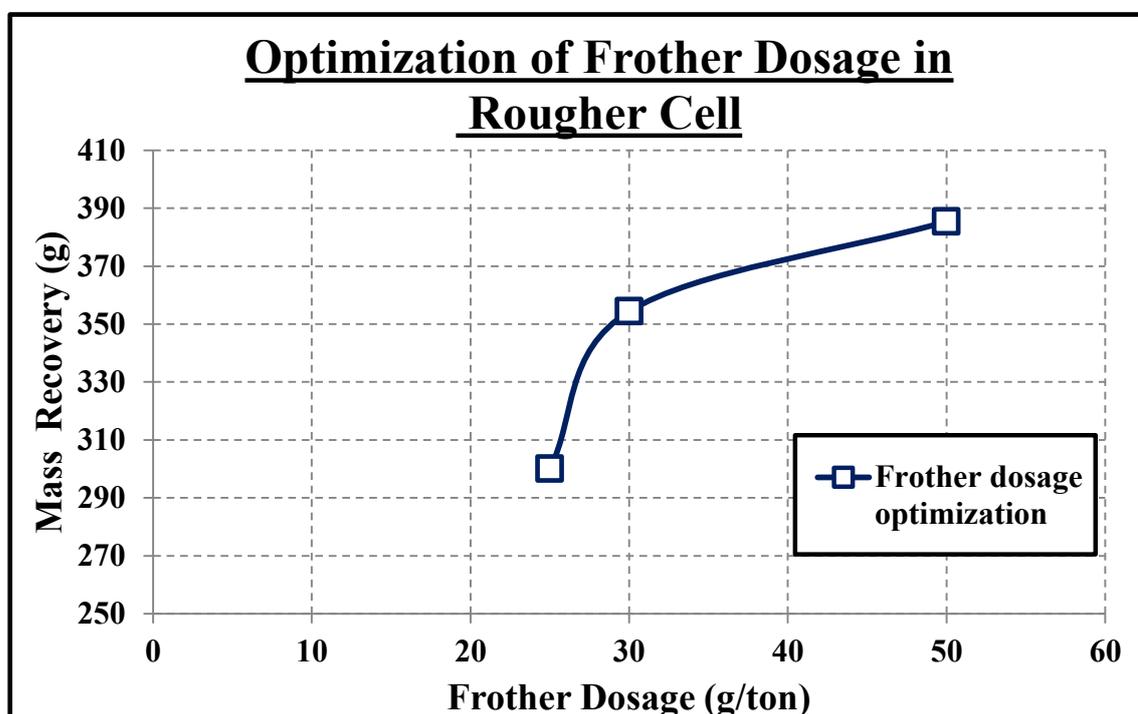


Figure 3.9 Optimization of frother dosage in rougher float

A frother dosage of 30g/t was decided upon because higher dosages resulted in the froth in the cleaner column being too dense.

3.2.4.1 The general procedure for rougher flotation

The rougher test represents the first stage of flotation. The aim of the rougher test was to produce the feed to the mechanical cleaners. Once the rougher reagent dosages were optimized the rougher tests were standard for all experiments.

- After milling, the slurry was transferred to the 45 L rougher cell with Outokumpu impeller.
- Impeller motor was switched on and the pulp was allowed to condition.
- 150g/ton of collector (SIBX) was added to the cell and allowed to condition for 5 minutes.
- 30g/ton of frother (Senmin XP200) was added to the cell and allowed to condition for 2 minutes. (Dosage varied for reagent optimization tests).
- The air flow valve was opened and the air flowrate was periodically increased to maintain the 6cm froth depth.
- Test was terminated when 4.2L of concentrate was obtained in an average of 14 minutes.
- The rougher tailings were properly discarded.

The rougher concentrate was transferred to either the column cell or the mechanical cell for batch experimentation.

3.2.5 Base tests on the mechanical cell

Once the rougher concentrate was standardized, base tests were performed on the mechanical cleaner cell. Base tests imply that no further reagents were added to the feed (rougher concentrate) during the cleaner float. The general procedure for the mechanical cell flotation was as follows:

- The 4.2 L rougher concentrate sample was transferred to the mechanical cleaner cell.
- The impeller of the Denver flotation mechanism was switched on at a speed of 850 rpm.
- The rotameter valve was opened, allowing an air flowrate of 20.6 L/min.

- A froth depth of 6cm was maintained.
- Cleaner concentrate was collected at timed intervals of 1 minute, 2minutes, 4 minutes and 6 minutes. The froth was allowed to flow naturally and was not scraped.
- Water was added manually with wash bottles to maintain the pulp-froth interface.
- The sides of the cell and the circumference of the impeller shaft were regularly washed during the test.
- The test was terminated after 6 minutes.

The concentrates and tailings were pressure filtered and dried in an oven. Base tests were performed on the mechanical cell in series A and C. They were analyzed for Cu/Ni and PGM content respectively.

3.2.6. Base case tests on the column cell

Attempts were made to determine a suitable froth depth in the column cell. However, tests using the E2 sparger in series A showed that level control was problematic with deep froths because the froth layer would occasionally collapse. The column froth depth was therefore set at 6cm by observation. When the E3 sparger was operational in series B it was noted that the froth layer was much more stable therefore the froth depth was varied in series C. The three different froth depths (20 cm, 15cm and 6cm) were investigated.

The mechanical and column cells were operated at the same superficial velocity. The superficial velocity in the mechanical cell determined the required air flowrate in the column cell. Hence, the flowrate in the column was calculated to be 2.27 L/min. The column experienced a pressure drop of 30 kPa therefore the corrected rotameter reading was 2 L/min. The effect of varying the air flow rate was still tested. Base tests were repeated at air flowrates of 1.14 L/min, 2.27 L/min and 3.41 L/min.

The column was operated with 6% solids by mass.

The general procedure of the standard column test was as follows:

- 4.2 L of rougher concentrate was transferred to the cleaner column
- The rotameter valve was opened to an air flowrate of 2.27 L/min. (Flowrate was varied for optimization tests.)

- A froth depth of 6cm was used.
- The concentrate was collected at timed intervals of 1minute, 2minutes, 4 minutes, 6 minutes and 9 minutes.
- The water flowrate was adjusted to maintain the pulp-froth interface.
- The test was generally terminated after 9 minutes.

The sides of the launder was washed using a wash bottle. The cleaner concentrates and tailings were pressure filtered and dried in an oven.

Base tests were performed on the column cell with the E2 sparger in series A and the E3 sparger in series B. These tests were analyzed for Cu/Ni content. Base tests were repeated on the E3 sparger in series C and analyzed for PGM content.

3.2.7 The use of the recirculation pump in column flotation

The reasoning behind using a recirculation pump was the anticipated settling of the ore on the base of the sparger. During the initial tests, however, settling did not occur. This was possibly due to the dilute operating conditions. There was, therefore, no need to use the recycle during the base tests. However, the shearing action of the pump could improve column flotation therefore the effect of the recycle was tested.

The recycle was operated at a flowrate of 7.5 L/min because suitable level control was possible at this flowrate. At higher flowrates the recirculation of air through the pump resulted in a build-up of air in the system. The recycle operated at a velocity of 4.42 cm/s down the column.

The general procedure for a column test using the recycle was as follows:

- 4.2 L of rougher concentrate was transferred to the cleaner column
- The air flowrate was set to 2.27 L/min.
- A froth depth of 6cm was maintained.
- The concentrate was collected at timed intervals of 1minute, 2minutes, 4 minutes, 6 minutes and 9 minutes.
- The water flowrate was adjusted to maintain the pulp level.

- The test was terminated after 9 minutes.

The sides of the launder was washed using a wash bottle. The cleaner concentrates and tailings were pressure filtered and dried in an oven.

The effect of using the recycle was tested on the column cell with the E2 sparger in series A and the E3 sparger in series B. These tests were analyzed for Cu/Ni content. The recycle tests were repeated on the E3 sparger in series C and analyzed for PGM content.

3.2.8 The effect of depressant dosage on mechanical cell

Cleaners are required to produce a high grade product therefore the use of depressant in the cleaner operation was the essential next step after the base tests. The depressant used was carboxymethyl cellulose (CMC).

The general procedure for depressant tests were similar to the base tests in section 3.2.6 except:

- Depressant was added before the impeller was switched on.
- Impeller was then switched on and pulp was allowed to condition for 5 minutes.
- The air flowrate was set to 20.6 L/min.

In series A, the mechanical cell was tested at depressant dosages of 10 g/t and 30 g/t. These tests were analysed for Cu/Ni content. In series C these tests were repeated and an additional test at 20 g/t depressant dosage was performed. These tests were analysed for PGM content.

3.2.9 The effect of depressant dosage on column cell

The effect of depressant dosage was also investigated using the column cell. The procedure was similar to the column base test in section 3.2.7 except:

- The rougher concentrate was transferred to the column and the depressant (5g/t, 10g/t or 30 g/t) was added to the cell.
- The recycle valve was opened to 7.5 L/min and the pump was switched on.
- The pulp was conditioned for 5 minutes.

- The pump was switched off and the recycle valve was closed.
- The airflow was set to 2.27 L/min.

The procedure continued as per the base test. At high depressant dosages the duration of the column test was reduced as most of the floatable material was depressed.

In series A, tests were performed on the E2 column for 10 g/t and 30 g/t depressant dosages. The depressant tests were repeated on the E3 column in series B. These tests were analysed for Cu/Ni content. In series C, tests were conducted on the E3 column at 10 g/t, 20 g/t and 30 g/t depressant dosages and analysed for PGM content.

3.2.10 The effect of gentle conditioning of the depressant.

Whitney and Jan (1996) stated that the absence of agitation in the column cell would enhance the action of the depressant because the agitation tends to scrub the depressant off the particles. The method of conditioning the depressant could also affect the action of the depressant for the same reason. The general procedure for tests with gentle conditioning of depressant was conducted as follows:

- Depressant was added to the rougher concentrate.
- Pulp was gently stirred with a stirring rod for 5 minutes.
- The concentrate was transferred to the column cell.

The procedure continued as per the base test in section 3.2.7.

The effect of manual conditioning was tested with the E3 column cell in series B and C. The tests were analysed for Cu/Ni and PGM content respectively.

3.2.11 The effect of cell height

The height of the column cell was doubled by including an extra length of perspex tubing. Two rougher concentrates were combined to form the feed to the column (2H) with the intention of maintaining the same percentage of solids as the previous tests on the original column. The rotameter correction factor was re-estimated at the new pressure drop of 50 kPa (7.5 PSIG) in order to maintain the superficial velocity of the original column. The resulting rotameter reading was 1.87 L/min. There was no further reagent addition in the cleaner tests.

The general procedure for tests with the column (2H) was as follows:

- 8.4 L of rougher concentrate was transferred to the cleaner column.
- The rotameter valve was opened to a reading of 1.87 L/min (as compared to R = 2 l/min in the shorter column).
- The concentrate was collected at timed intervals of 1minute, 2minutes, 4 minutes, 6 minutes, 9 minutes and 12 minutes.
- The water flow was adjusted to maintain 6cm froth depth.
- The tests were terminated after 12 minutes.

The concentrate and tailings were pressure filtered and dried in an oven. The dried concentrates and tailings were then prepared for analysis. Tests on the column (2H) utilized the E3 sparger in series C and were analysed for PGM content.

3.2.12 Analyses

It is well known the PGM found in UG-2 ore is closely associated with sulphide minerals (section. 2.2). The grade of Cu and Ni was analysed in series A and B as a cost-effective alternative to direct platinum analysis. Cu and Ni grades would provide an indication of the trends one may expect from the actual PGM grades. Samples were analyzed for Cu and Ni at local UKZN laboratories.

3.2.12.1 Cu/Ni analysis

After the concentrates were pressure filtered and dried in ovens, the ore was pulverised and sub-sampled. The samples were digested with hydrochloric and nitric acid. The digestion method for Cu/Ni samples were as follows:

- The dried concentrate and tailings were finely pulverised using a pestle and mortar.
- 10g sub-samples were obtained using the coning and quartering sampling technique.
- 20 ml of nitric acid and 20ml of hydrochloric acid was added to the 10g ore sample.

- The mixture was heated under medium heat and mechanically stirred for 8 minutes (till the temperature of the mixture was above 50°C)
- The heated mixture was then filtered under suction.
- The filtrate was diluted to a 1 L solution.
- 12 cm³ of the solution was transferred into a vial and sent for ICP analysis.

The Cu/Ni analysis was then performed using Inductively Coupled Plasma mass spectrometry (ICP). The analyses were performed by an analytical technician in the School of Pure and Applied Chemistry, UKZN.

3.2.12.2 PGM + Au analysis

PGM analysis required a much larger sample mass than the ICP analysis. The previous samples retained from tests performed in Series A and B could not be re-assayed due to insufficient masses remaining after repeated acid digestion and ICP analyses. The tests performed in series B were therefore repeated using the second ore sample and analysed for PGM content. These tests were referred to as series C. The dry concentrates and tailings were bagged and sent to Mintek. The PGM analysis and all sample preparation were performed by Mintek.

3.3 PILOT PLANT EQUIPMENT

A weeklong plant visit to Lonmin Platinum in Rustenburg was conducted in order to perform comparative tests between a newly acquired pilot column and a pilot rig of mechanical flotation cells known as the Floatability Characterization Test Rig (FCTR). This section provides a description of the pilot plant equipment consisting of the pilot test rig with 6 mechanical cells and the pilot column cell.

3.3.1 The Floatability Characterisation Test Rig (FCTR)

The FCTR is pilot scale test rig that runs alongside the actual plant. The pilot plant allowed metallurgists to treat specific streams of plant material and the material used was returned to the main stream. The FCTR is fully instrumented and it incorporates Mintek's Plantstar flotation

control system. The Lonmin FCTR was based on an original rig developed at the University of Queensland's Mineral Research Centre and is shown in Figure 3.10.



Figure 3.10 Original FCTR developed at the University of Queensland
(<http://www.p9project.com/Documents/News%20Archive/FCTR%20strengthens.pdf>)

The Lonmin FCTR test rig consisted of 6 mechanical flotation cells. Each cell was of the same dimension as shown in Table 3.2.

Table 3.2 Dimensions of each cell in FCTR test rig

<i>FCTR Dimensions per cell</i>		
Height	0.7	m
Diameter	0.52	m
Volume	150	L

Each cell was equipped with its own feed port and a connection to the previous cell making it versatile for the production of high, medium or low grade concentrates. The airflow was controlled by control valves that were attached to rotameters and adjusted by PI (Proportional - Integral) controllers. The FCTR employed automatic level control.

The mechanical cells were operated in series. The configuration of the cells is shown below in Figure 3.11. The FCTR accommodated for sampling of the feed, final tailings and all concentrates. All sampling could be automatically or manually performed.

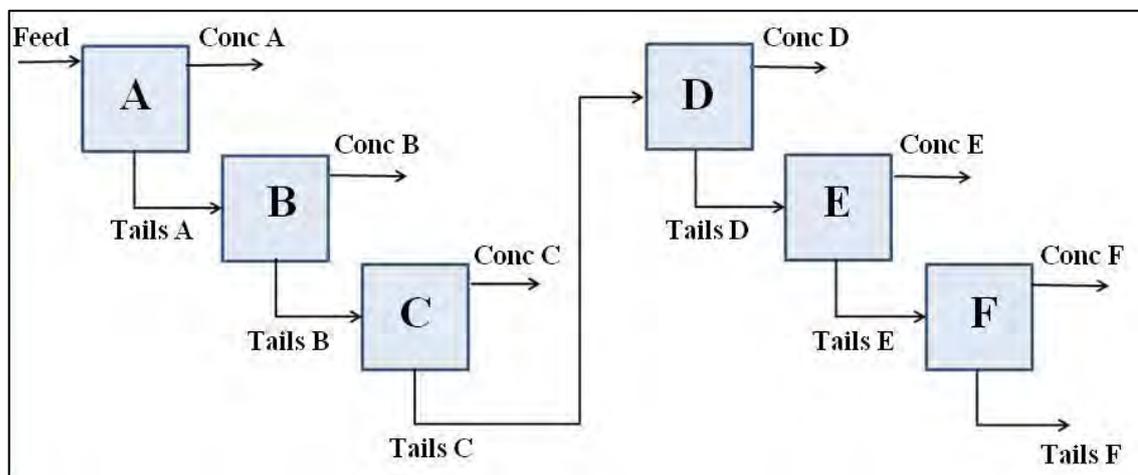


Figure 3.11 Configuration of the FCTR

3.3.2 The pilot column

A 378 L column cell was recently acquired by Lonmin Platinum. The column diameter was 0.3 m and the height was 5.5m. At the time of the pilot plant tests, the column cell had just been installed and was in the process of being optimised.

The column consisted of two feed ports situated midway and two thirds up the column. Automatic level control had not been implemented therefore the pulp level was maintained by monitoring the pressure within the column

The air distribution mechanism comprised of both an external sparger and an internal sparger. The internal sparger consisted of perforated metal tubes in the sparger ports situated on the column (fig. 3.12). The external sparger operated via the recycle line. The external sparger was employed for pilot plant tests.

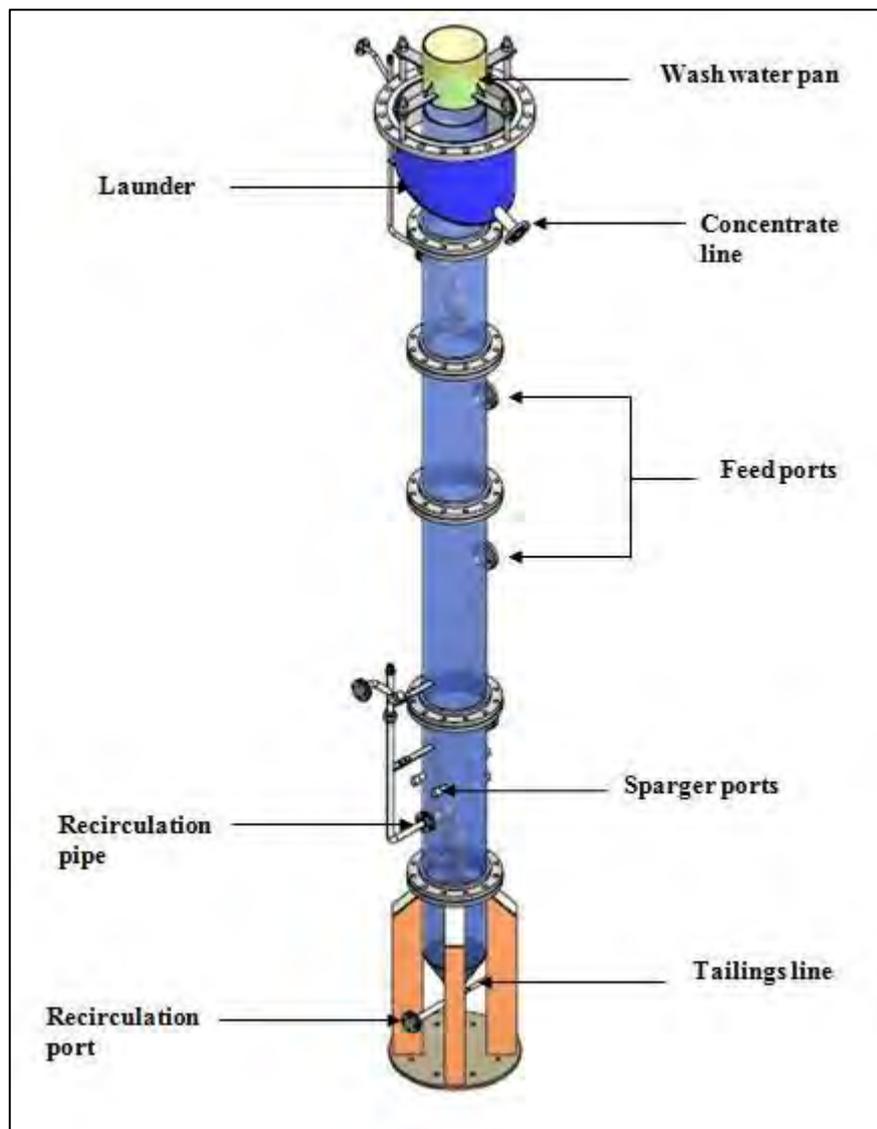


Figure 3.12 Isometric view of the pilot column (Lonmin Platinum Research Laboratories)

The pilot column was designed to operate with a wash water pan at the top to allow for froth washing. However, froth washing was not employed during the pilot plant tests.

3.4 PILOT PLANT EXPERIMENTAL PROCEDURES

3.4.1 Experimental procedure in the FCTR

The feed to the final cleaner in the secondary circuit was diverted to the FCTR. The impellers were switched on and the air flow was set for each cell. The level was automatically controlled to maintain a certain froth depth in each cell. The feed to the final cleaner cell has a large proportion of floatable minerals therefore a deep froth was required in the FCTR cells to prevent excessive flow of froth. The froth depth could be reduced down the bank of cells as shown in

Table 3.3. The system was given time to reach steady state. The pulp volume in each cell was calculated, allowing for the froth depth. The average residence time per stage was calculated on the basis of the tailings flow leaving that stage. This is consistent with the assumption of perfect mixing in the pulp phase.

Table 3.3 Calculation of the residence time in each cell of the FCTR

	<i>Froth Depth</i> (<i>cm</i>)	<i>Air flow</i> (<i>L/min</i>)	<i>Pulp volume</i> (<i>L</i>)	<i>Residence Time</i> (<i>min</i>)
cell A	37	90	63	2.84
cell B	30	80	78	3.65
cell C	28	80	83	3.95
cell D	26	75	87	4.37
cell E	24	70	91	5.17

A sampling team was at hand to obtain samples of the tailings, the concentrates and the feed in that order. The feed was sampled last because by sampling the feed disruptions may occur downstream. Three samples were taken at each sampling point. This meant that three buckets were alternated at a particular sampling point until each bucket contained three cuts. One cut was taken every 10 minutes at each sampling point. The samples were weighted, filtered and then dried. The dry samples were analysed for PGM content by Mintek.

3.4.2 Experimental procedure in the pilot column

The pilot column was still in the process of being optimized and tests were performed on the column at three different froth depths and three different flowrates. A total of nine tests were performed. The froth depth was determined by measuring the pressure drop. The column did not possess a very sensitive pressure gauge therefore the three froth depths investigated were the lowest and highest froth depths that appeared to be stable, and a froth depth somewhere in the middle. The flowrates were chosen so as to cover a wide range of residence times within the column. Laboratory tests in series A showed that the column cell recovered mass at a slower rate and therefore required a longer residence time. The different froth depths and flowrates appear in table 3.4.

Table 3.4 Parameters tested in the Pilot column

<i>Flow (L/min)</i>	<i>Froth Depth (cm)</i>
9	55.5
35	37
59	18.5

The feed to the final cleaner in the secondary circuit was diverted to the pilot column. The air flowrate was set at 50 L/min for all tests. The pressure drop across the column was adjusted to maintain the froth depth. Samples of the feed, concentrate and tailing was taken for each test. These samples were weighted, filtered and the dried. The dry samples were analysed for PGM content at Mintek.

CHAPTER 4: LABORATORY RESULTS AND DISCUSSION

4.1 OVERVIEW OF ANALYSIS AND REPEATABILITY

The laboratory experiments were divided into three series as stated in section 3.2.1. Series A refers to tests performed on the column cell with the E2 sparger (E2 column) and the mechanical cell. Series B refers to tests performed on the column cell with the E3 sparger (E3 column). Both series A and B utilized the same ore sample and were analysed for Cu/Ni content (detailed procedures are available in section 3.2).

4.1.1 Analysis of the rougher tailings

The data from initial tests showed a significant variation in the recalculated head grades of Cu and Ni in the feed. The sampling of the rougher tailing was identified as a significant contributor to this variation. This was due to difficulties in obtaining a representative sample for analysis from approximately 9.5 kg of rougher tails remaining in the flotation cell (Some of the water was removed after settling and the pulp was mixed using the impeller. Samples of pulp were removed, dried and split). It was decided that a more reliable method was to use a standard rougher flotation procedure and compare the cleaner efficiency. The degree of fluctuations in the recalculated head grade of the rougher concentrate was not as significant as the rougher feed.

4.1.2 Problems encountered with repeatability and its possible causes

The experiments performed in series A and B were analysed in batches. It was concluded from the analysis that the recalculated head grades in different batches still showed significant variations. This sometimes occurred within the same batch as well as for particular concentrates within individual tests.

An obvious consideration for such variation would be the sampling of the ore itself. It has been explained in the experimental procedure that the ore was screened (3mm). The coarse fraction was crushed in a jaw crusher and re-screened. The crushed ore and fine ore was individually blended, sub-sampled and riffled and then combined to form 5kg portions (detailed procedures in Section 3.2.2). It was therefore concluded that all reasonable measures to obtain a representative feed were applied.

The subsequent considerations were either that there were errors in the experimental method or in the analysis itself. General precautions were put in place to ensure that the experimental method was consistent. Significant shifts in the grade occurred between different batches sent for analysis. This led to the conclusion that the analysis techniques had to be investigated.

The analysis itself can be divided into the acid digestion of the ore sample and the ICP analysis. These techniques are discussed in section 3.2.12. The acid digestion was performed by the author and the ICP analysis was performed by a local laboratory. A study into the possibility of errors in both techniques was pursued. The investigation entailed the following:

- Multiple samples of a particular concentrate were individually digested with acid to investigate the extent of errors in the digestion technique.
- Multiple sub-samples of a single concentrate were submitted for analysis to investigate the possibility of errors in the ICP analysis within a particular batch of tests.
- Liquid samples from previously digested concentrates were resubmitted to investigate errors in ICP analysis between different batches of tests.

Substantial variations in Cu/Ni grade were discovered for tests that were resubmitted from previous batches. In a particular case a concentrate that was reported to have a Ni grade of 3156 ppm two months prior, was re-analysed to have a grade of 5960 ppm. It was also evident that there were discrepancies within the same batch of tests as well. A repeat sample for the above concentrate in the same batch reported a Ni grade of 5120 ppm. However, these discrepancies were infrequent and were less drastic.

The investigation into experimental errors was inconclusive to a certain degree because having ascertained that errors did exist in the ICP analysis it was not possible to determine whether the differences in the individually digested tests were from the digestion itself or the ICP. The differences in the grades of Ni and Cu for the individually digested concentrates were of a much smaller degree (200-400 ppm) than that of the ICP analysis for repeated batches (more than 800 ppm as in the example above).

Attempts were made to determine exactly what was causing these inconsistencies in the ICP analysis. Repeat analyses were conducted at adjusted calibrations and fresh standard solutions were utilized but neither option yielded much success.

In total, 80 flotation tests were performed in Series A, B and C. Series A and B consisted of 50 tests which had been analysed for Cu/Ni content locally. The plausible but expensive alternative was to analyse the pgm content of these tests. This technique also required significantly larger masses of concentrate for analysis. After multiple attempts at Cu/Ni analyses, there was not

enough sample left from the tests performed in Series A and B to be re-analysed for pgm content.

A new ore sample was required and series C contained a repeat of the base tests. It may be questioned as to why the Cu/Ni analyses were not discontinued sooner as they currently form the bulk of the experimental work. It was initially hoped that the cost of direct pgm analysis could be avoided but it was eventually decided that PGM analysis was required to draw final conclusions. Nevertheless the initial tests performed with Cu/Ni analysis provided essential experience for developing experimental procedure. Physical difficulties, such as the crack in the E2 sparger and mechanical problems whilst milling, hampered and prolonged ICP investigations.

The tests analysed for Cu/Ni content could not be discarded altogether because tests that were analysed in the same batch could provide suitable data. These tests were selected based on criteria that will be discussed below.

4.1.3 Section Criteria for tests in Series A and B

It was decided that only tests with a similar calculated head grade of rougher concentrate would be used for comparison. The average rougher grades of the selected tests were 0.37% Cu and 0.54% Ni. The head grade for each test was required to be within 12% of the average head grade stated above. About 46% of the tests did not satisfy this criterion (for the reasons discussed in Section 4.1.2) and they were discarded. In order to make the selection process more transparent to the reader, graphs depicting the head grade and total recovery for each test are shown below. Each graph compares the mechanical and column data for selected changes in operating conditions. Tests that were discarded can be identified by hollow markers.

Figures 4.1 and 4.2 show all base tests performed in series A and B. Certain tests may appear within range, such as the Ni grade (65.1%, 0.53%) of a base test with the E2 column. Figure 4.2, however, shows that the corresponding Cu grade for that test was too low (0.29%). Similarly, the Ni grade for the mechanical cell test (78.9%, 0.55%) seems to be acceptable but the corresponding Cu recovery was uncharacteristically low (44.1%). It is not possible to visually judge from the graphs which Cu and Ni head grades correspond to the same test, therefore the both grades appear together for each test in Table 4.1. The values in bold signify those tests that were regarded as acceptable.

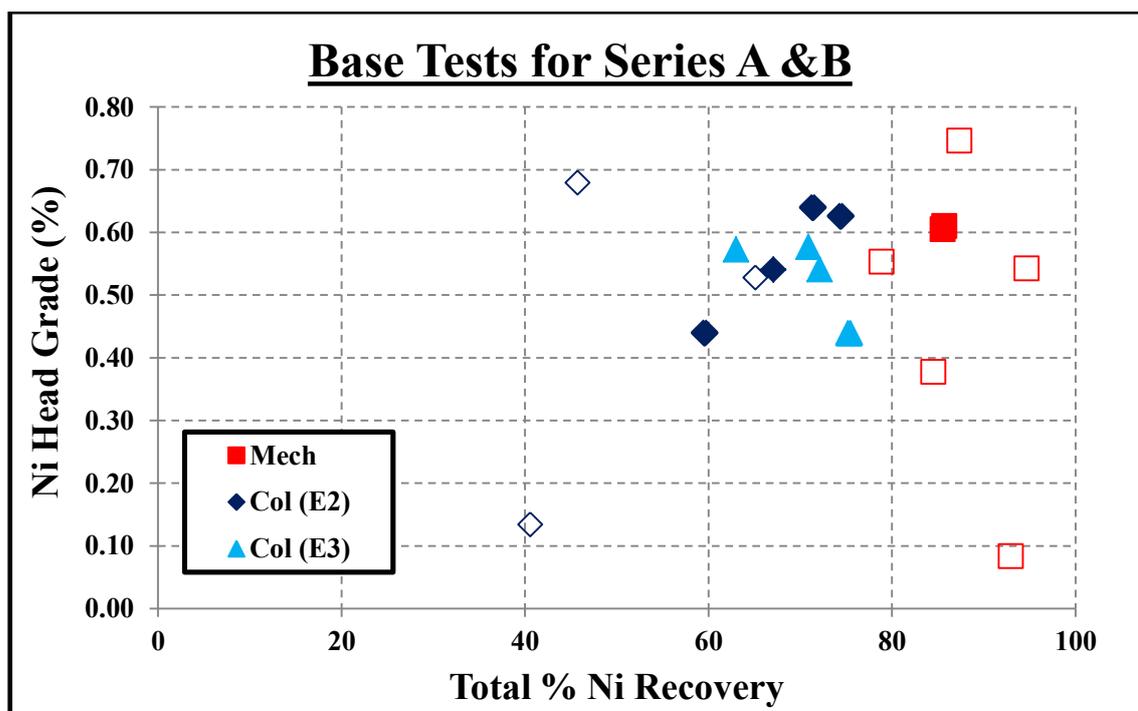


Figure 4.1 Head grade vs total recovery of Ni for base tests in series A and B

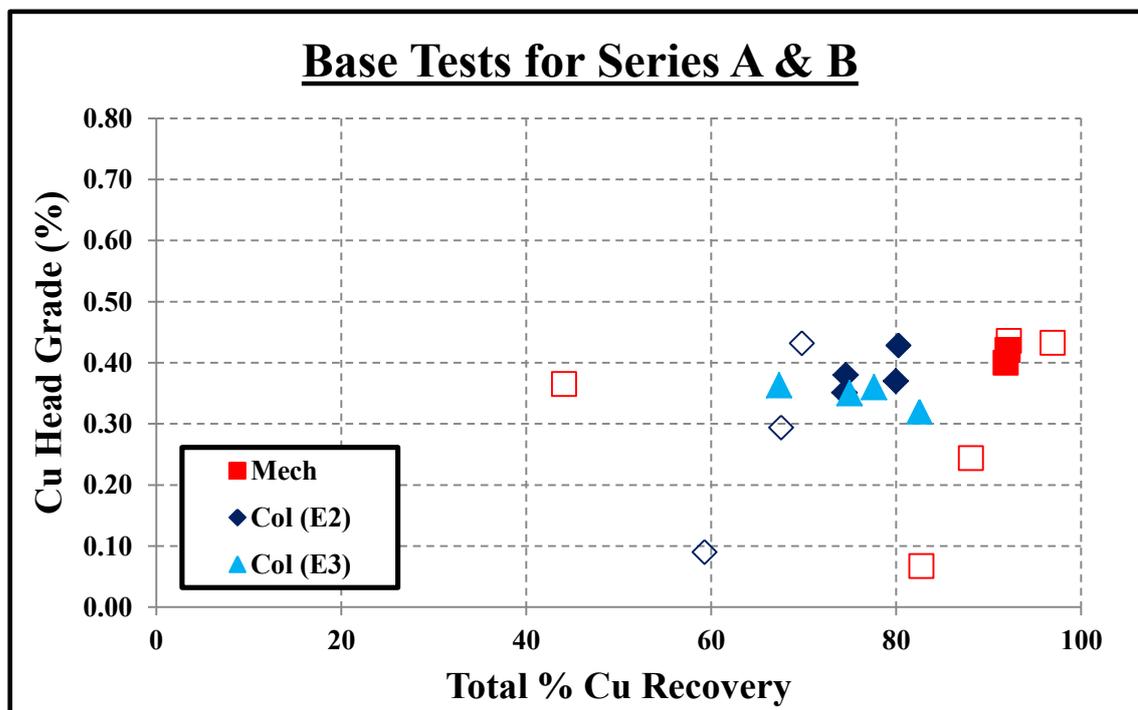


Figure 4.2 Head grade vs total recovery of Cu for base tests in series A and B

Table 4.1 Base tests for mechanical and column cells

<i>Mechanical Cell</i>				
Head Grade(%)			Recovery(%)	
No	Ni	Cu	Ni	Cu
1	0.08	0.07	92.9	82.7
2	0.55	0.37	78.9	44.1
3	0.54	0.43	94.7	96.9
4	0.75	0.44	87.3	92.2
5	0.60	0.42	85.6	92.1
6	0.38	0.24	84.5	88.1
7	0.61	0.40	85.7	91.9

<i>Column Cell (E2)</i>				
Head Grade(%)			Recovery(%)	
No	Ni	Cu	Ni	Cu
1	0.13	0.09	40.6	59.3
2	0.68	0.43	45.7	69.8
3	0.44	0.38	59.6	74.6
4	0.64	0.37	71.4	80.0
5	0.54	0.35	67.1	74.4
6	0.63	0.43	74.4	80.3
7	0.53	0.29	65.1	67.6

<i>Column Cell (E3)</i>				
Head Grade(%)			Recovery(%)	
No	Ni	Cu	Ni	Cu
1	0.54	0.36	72.1	77.6
2	0.57	0.36	63.0	67.3
3	0.44	0.32	75.3	82.5
4	0.58	0.35	70.9	75.0

Figures 4.3 and 4.4 depict all tests performed with a depressant dosage of 10 g/t.

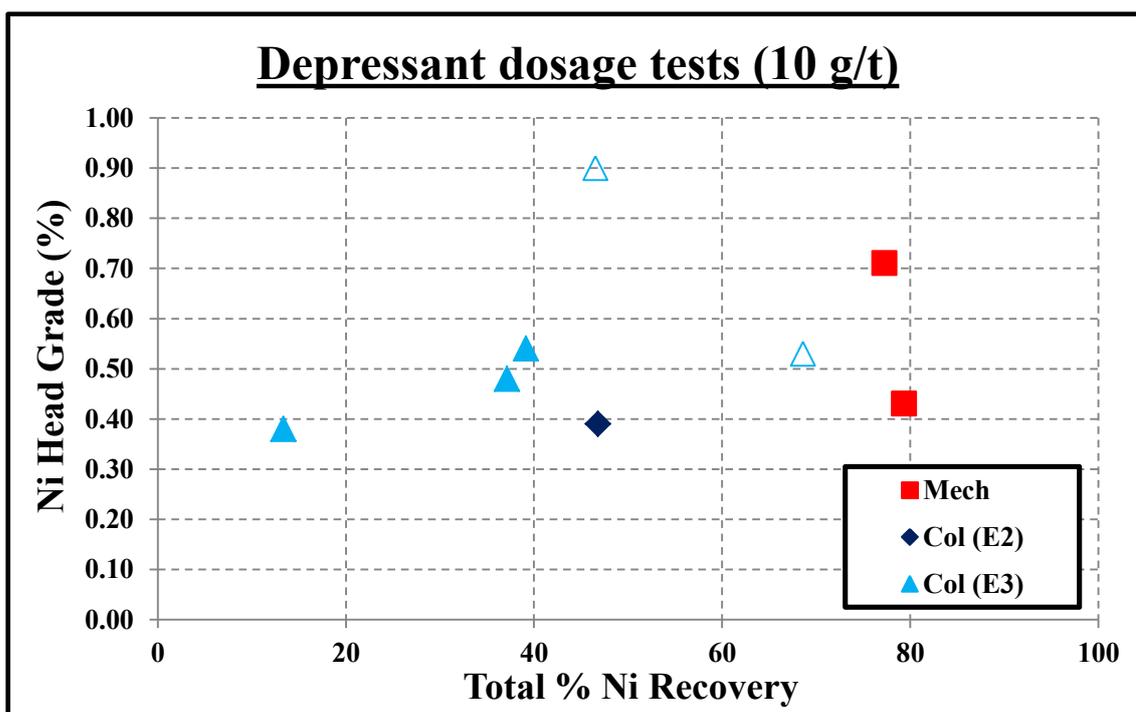


Figure 4.3 Head grade vs total recovery of Ni for a depressant dosage of 10 g/t in series A and B

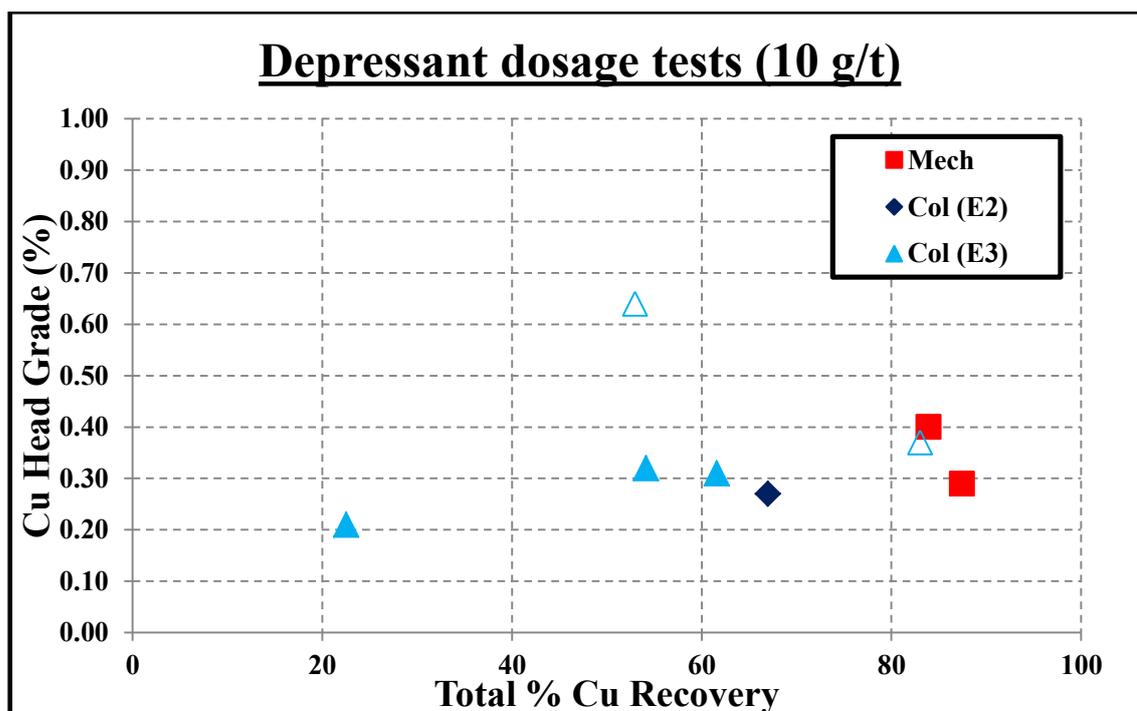


Figure 4.4 Head grade vs total recovery of Cu for a depressant dosage of 10 g/t in series A and B

Table 4.2 Depressant dosage tests (10g/t) for mechanical and column cells

<i>Mechanical Cell</i>				
Head Grade(%)			Recovery(%)	
No	Ni	Cu	Ni	Cu
1	0.71	0.40	77.3	83.9
2	0.43	0.29	79.4	87.4

<i>Column Cell (E2)</i>				
Head Grade(%)			Recovery(%)	
No	Ni	Cu	Ni	Cu
1	0.39	0.27	46.8	67.0

<i>Column (E3)</i>				
Head Grade(%)			Recovery(%)	
No	Ni	Cu	Ni	Cu
1	0.48	0.31	37.1	61.6
2	0.38	0.21	13.3	22.5
3	0.90	0.64	46.5	53.0
4	0.54	0.32	39.2	54.1
5	0.53	0.37	68.6	83.0

In figure 4.4, a test performed on the E3 column (83.0%, 0.37%) was rejected because the Cu recovery was uncharacteristically high as compared with the other tests. The same was found with the Ni recovery (68.6%, 0.53%) of another test in figure 4.3. However, in the table 4.2, it can be seen that test 2, performed on the E2 column, was accepted even though the Cu and Ni recoveries seem unusually low. The reason behind this decision was that this test was the first

test performed with a dosage of 10 g/t depressant and manual conditioning. It was expected that a difference in recovery would occur when the conditioning technique changed. It was the only test performed at this condition and was therefore not discarded despite the Cu grade being low (0.21%). Tests performed at a depressant dosage of 30g/t are depicted in figures 4.5 and 4.6.

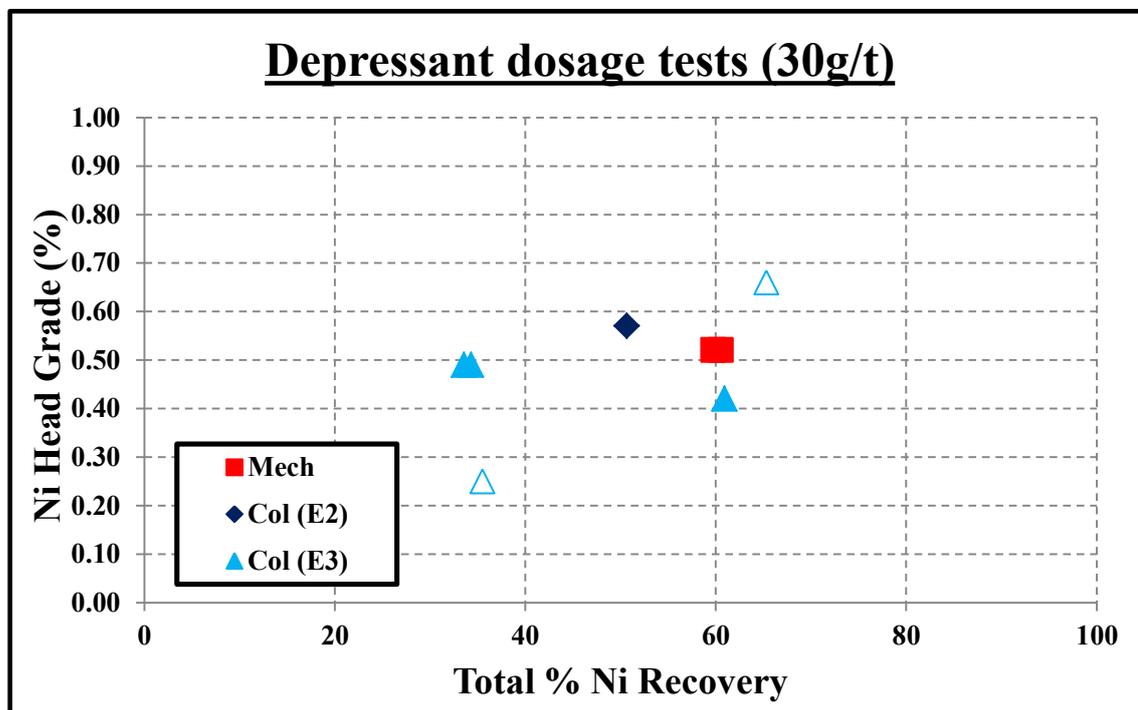


Figure 4.5 Head grade vs total recovery of Ni for a depressant dosage of 30g/t in series A and B

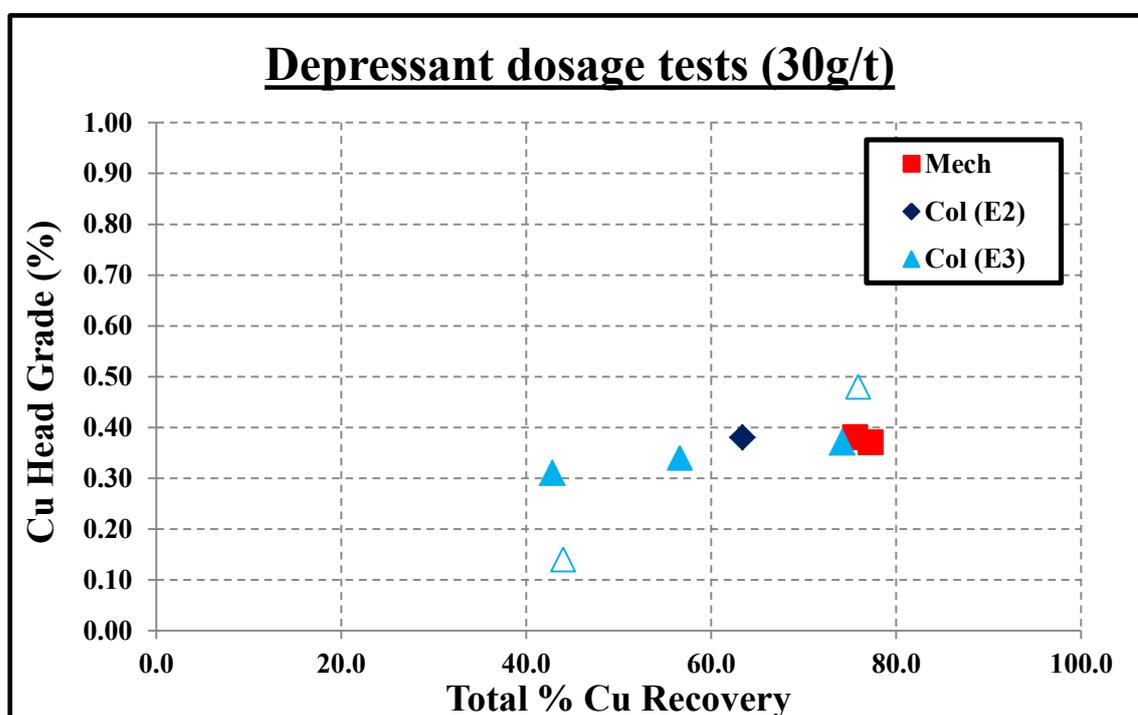


Figure 4.6 Head grade vs total recovery of Cu for a depressant dosage of 30 g/t in series A and B

Table 4.3 Depressant dosage tests (20g/t) for mechanical and column cells

<i>Mechanical Cell</i>				
Head Grade(%)			Recovery(%)	
No	Ni	Cu	Ni	Cu
1	0.71	0.40	77.3	83.9
2	0.43	0.29	79.4	87.4

<i>Column Cell (E2)</i>				
Head Grade(%)			Recovery(%)	
No	Ni	Cu	Ni	Cu
1	0.39	0.27	46.8	67.0

<i>Column (E3)</i>				
Head Grade(%)			Recovery(%)	
No	Ni	Cu	Ni	Cu
1	0.48	0.31	37.1	61.6
2	0.38	0.21	13.3	22.5
3	0.90	0.64	46.5	53.0
4	0.54	0.32	39.2	54.1
5	0.53	0.37	68.6	83.0

The discarded tests on the E3 column are clearly out of range in figures 4.5 and 4.6.

The mechanical cell was tested at a depressant dosage of 20g/t. The Cu and Ni head grade and total recoveries simultaneously appear in figure 4.7. All tests were out of range and hence they were not utilised in analysis.

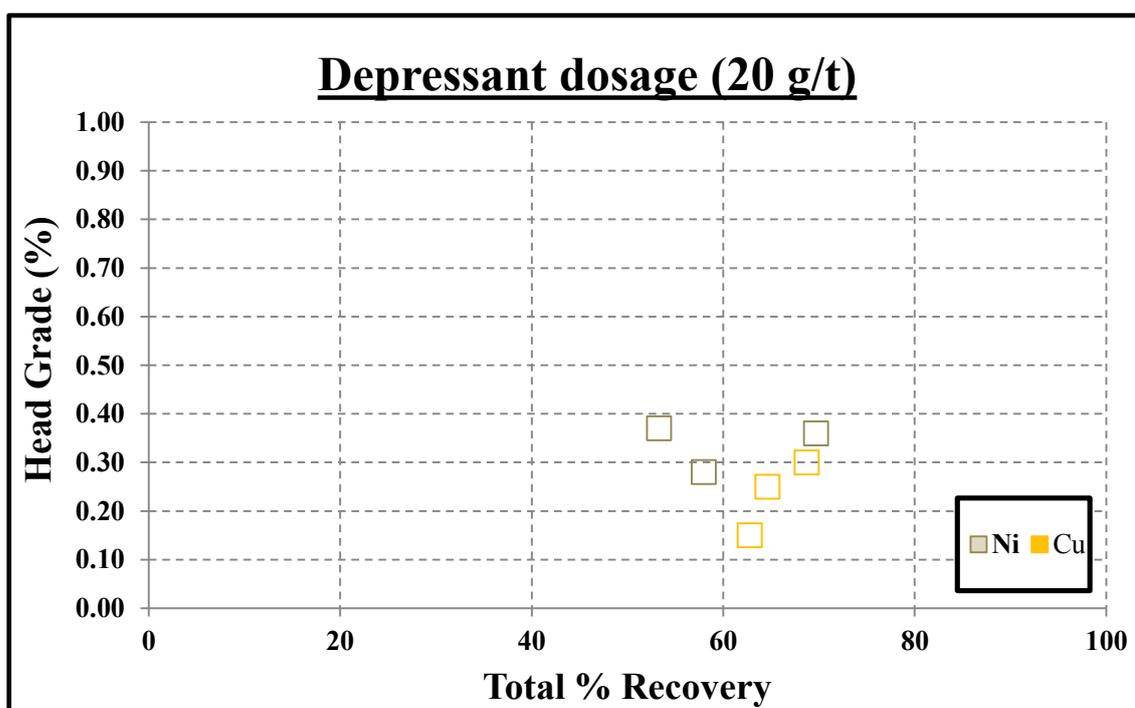


Figure 4.7 Head grade vs total recovery for a depressant dosage of 20 g/t in mechanical cell

Table 4.4 Depressant dosage tests (20g/t) in mechanical cell

<i>Mechanical Cell</i>				
Head Grade(%)		Recovery(%)		
No	Ni	Cu	Ni	Cu
1	0.37	0.25	53.3	64.6
2	0.28	0.15	58.0	62.8
3	0.36	0.30	69.7	68.7

The column cell was tested at a depressant dosage of 5g/t as shown in figure 4.8. The Cu and Ni head grades and total recoveries appear simultaneously. Both tests were acceptable.

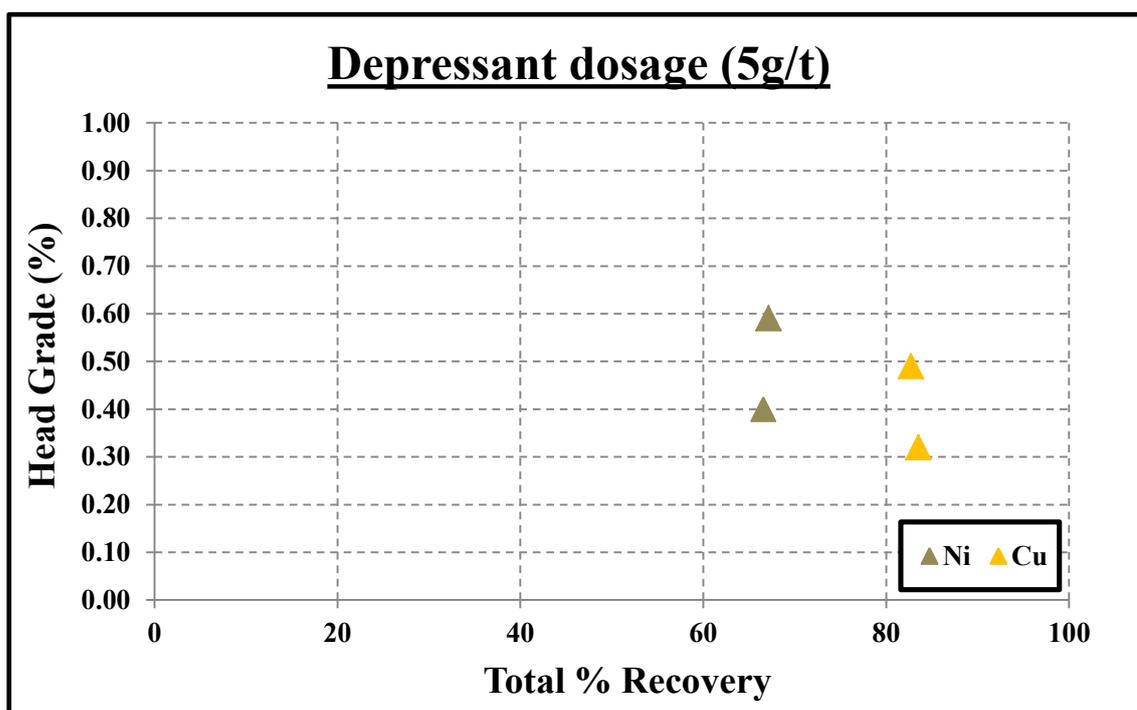


Figure 4.8 Head grade vs total recovery for a depressant dosage of 5 g/t in mechanical cell

Table 4.5 Depressant dosage tests (5g/t) in Column Cell

<i>Column (E3)</i>				
Head Grade(%)		Recovery(%)		
No	Ni	Cu	Ni	Cu
1	0.59	0.49	67.1	82.7
2	0.40	0.32	66.6	83.5

4.1.4 Repeatability of tests analysed for pgm (Series C)

Series C refers to tests performed on the column cell with E3 sparger and the mechanical cell. These tests were analysed for PGM content. The first ore sample was depleted and therefore series C utilized a second, smaller ore sample. PGM analysis showed that in the second ore sample there was some variation in the mineral content and as a result two distinct average rougher concentrate grades were identified. The recalculated PGM grades of rougher concentrates were within 10% of an average of 41.6 ppm for the base tests, depressant dosage tests and column height tests. However, the tests comparing manual conditioning with conditioning by means of the recycle were within 10% of 33.3 ppm i.e. they were comparable, but at a different feed grade.

It was interesting to note that tests that were analysed for pgm showed much less variation in the head grade than those in series A and B. This can be visually perceived in Figure 4.9 and 4.10 that show tests with an average of 41.6 ppm and 33.3 ppm head grade respectively.

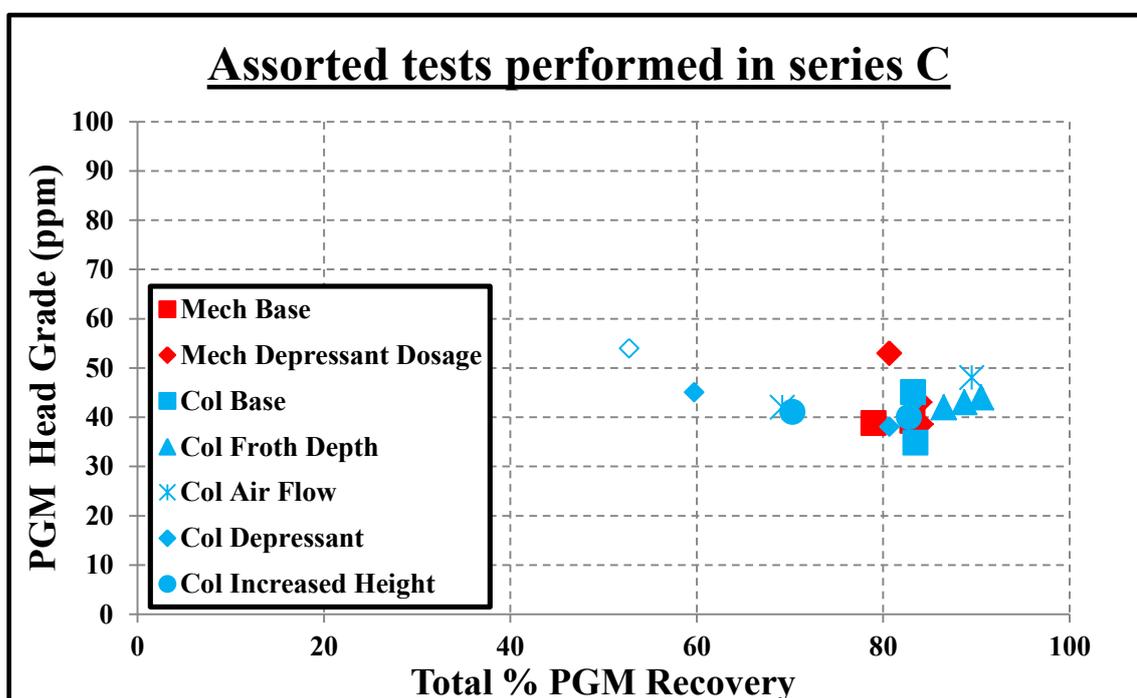


Figure 4.9 Head grade vs total recovery for assorted tests in series C (Average = 41.6 ppm)

Figure 4.9 shows that only a single test (52.7%, 54 ppm) was discarded from the entire batch. The 30 g/t depressant test in the mechanical cell reported a slightly higher grade (80.6%, 53 ppm). It was not discarded because it was the only test performed at this dosage and was required for comparison. Figure 4.10 shows that all tests using the recycle were also within a reasonable range.

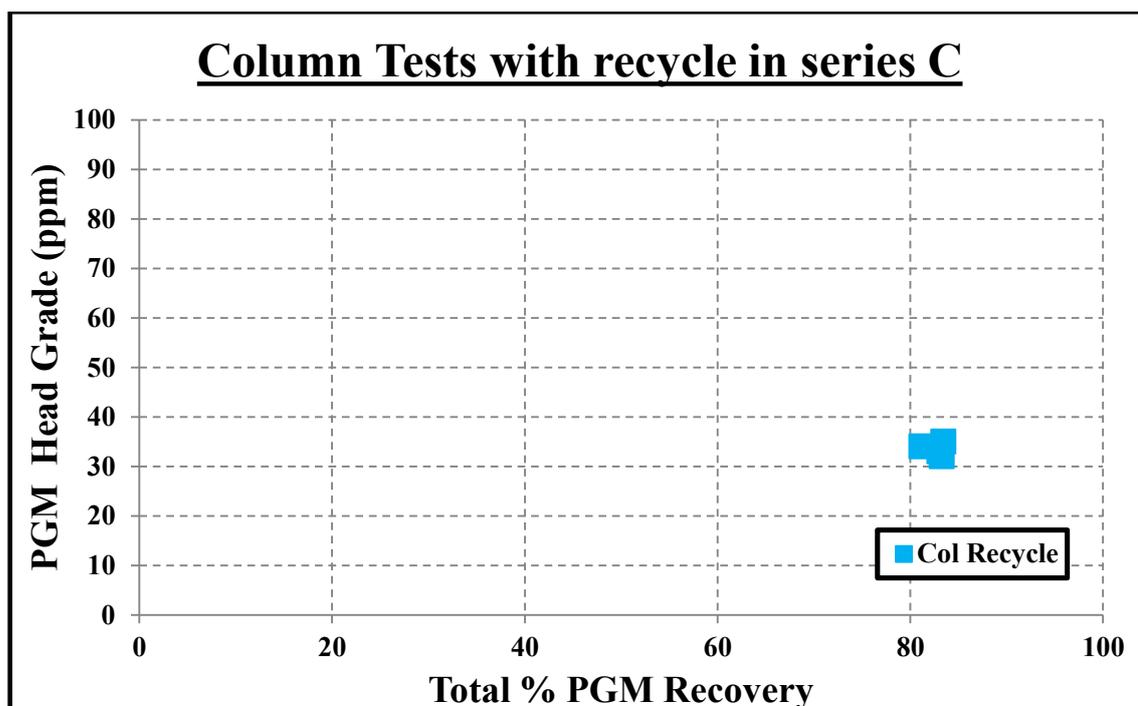


Figure 4.10 Head grade vs total recovery for column test with recycle (Average = 33.3 ppm)

Only a few tests in series C were replicated. The reason for this was that each test required 10 kg of ore and tests on the longer column (2H) required 20 kg of ore per test. The second ore sample consisted of only 300 kg and hence only a limited number of tests were possible. The expense of the PGM analyses was also a critical factor. The total cost of the PGM analysis was in excess of R60 000 therefore mainly single tests were compared per parameter.

4.1.5 Important points on repeatability and error analysis

Error bars were employed in all graphs for all three series to give an indication of the repeatability of the data. In cases where the error is very small the bars may not be visible.

In other instances where a single test was performed the error bars obviously could not be used. All subsequent curves that used error bars (whether visible or not) are demarcated by solid markers (representing an average of two tests) and single tests are demarcated by hollow markers to distinguish these two occurrences from each other.

The discussion of the laboratory results commences with the standardization of the rougher flotation for the production of the feed to the mechanical cell and column cell.

4.2 STANDARDIZING THE ROUGHER FLOTATION FOR CLEANER FEED PRODUCTION

The 43L rougher cell with an Outokumpu impeller was used to produce the feed for all batch cleaner tests performed on the mechanical and column cells. The rougher flotation was standardized with the aim of recovering sufficient mass for the cleaning tests, in a volume that did not exceed the cleaner volume (4.2L). Hence, no depressant was used during the rougher flotation. The recommended collector dosage of 150 g/t was kept constant for all tests. The frother dosage was varied between 20 g/t, 30 g/t and 50 g/t.

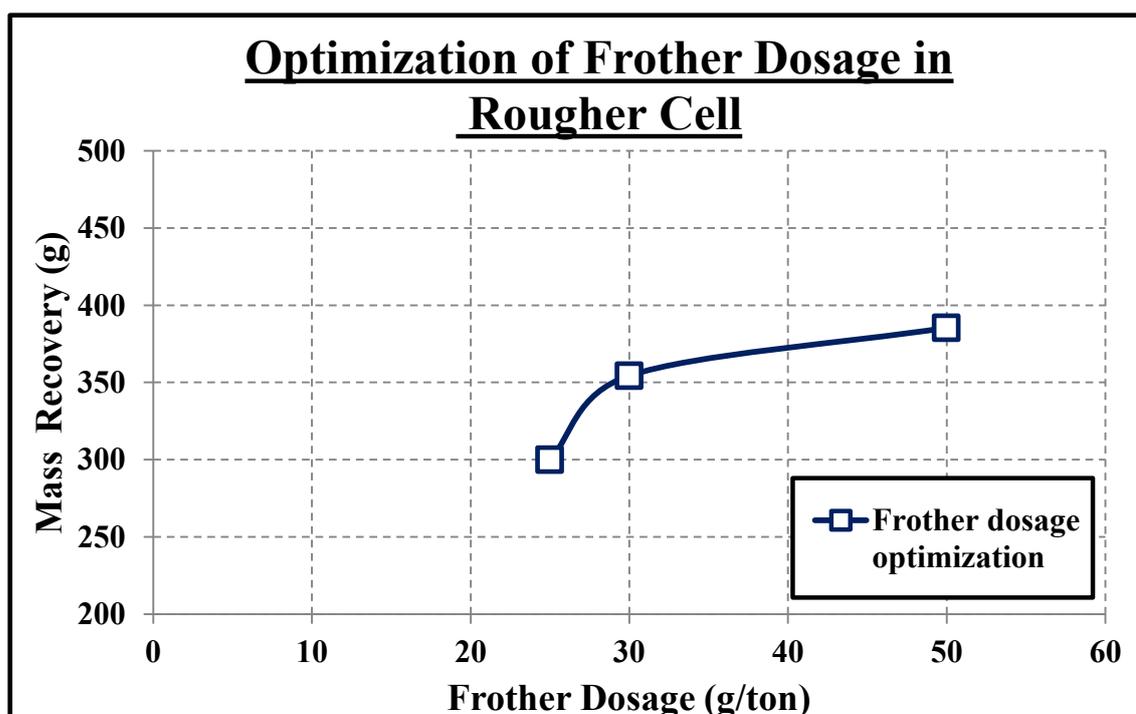


Figure 4.11 Optimization of frother dosage in rougher cell

Figure 4.11 shows the different mass recoveries obtained during the rougher flotation when the frother dosage was varied. A 50 g/t frother dosage recovered the most mass but subsequently the froth in the cleaner column was too voluminous. The standard frother dosage was set at 30 g/t because at that dosage a substantial mass recovery as achieved without producing a too voluminous froth in the cleaner. A rougher concentrate of 354 g was obtained using a reagent suite of 30 g/t of frother, 150 g/t of collector and no depressant.

4.3 COMMISSIONING OF THE COLUMN CELL

Commissioning tests were performed on the column cell to obtain an appropriate froth depth and air flowrate.

4.3.1 Froth depth

Initial attempts were made to investigate the effect of varying the froth depth in the E2 column in series A. However, it was found that froth layer in the column cell collapsed periodically when a deep froth was used. A suitable froth depth of 6cm was obtained by observation.

When the E3 sparger was operational in series B it was noted that the froth layer was much more stable therefore the froth depth was varied in series C. Three different froth depths were investigated i.e. 20 cm, 15cm and 6cm.

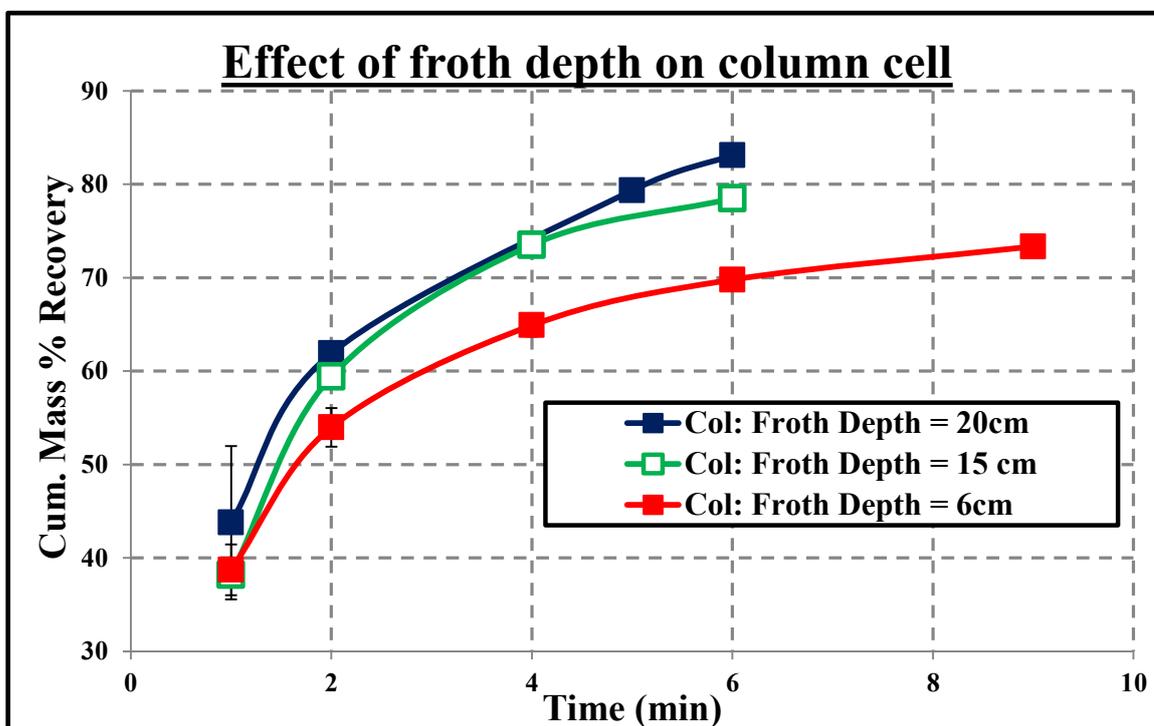


Figure 4.12 Effect of froth depth on the mass recovery of the E3 column

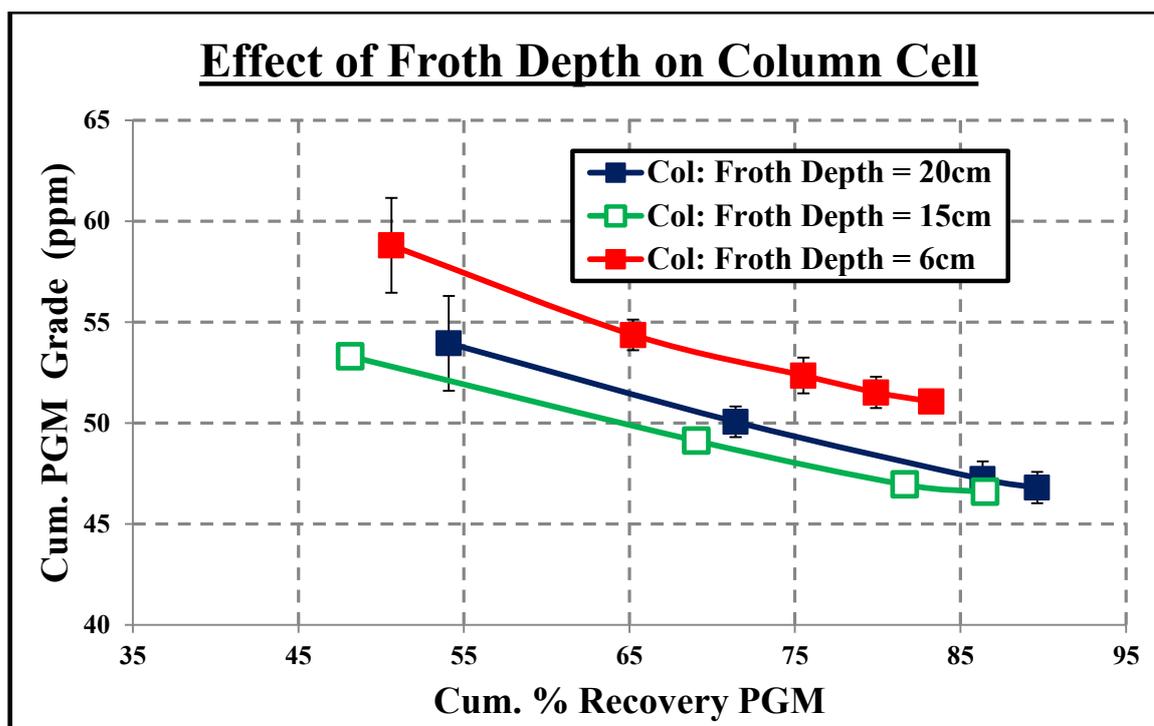


Figure 4.13 Effect of froth depth on the PGM grade and recovery of the E3 column

Figure 4.12 shows an unexpected trend in the mass recovery for varying froth depths. The 6cm froth depth achieved the lowest mass recovery and the 20cm froth depth achieved the highest mass recovery. Both the 6cm and 20cm froth depth tests were repeated but similar results were produced. There was no reasonable explanation for this trend. It possibly may be related to the build-up of froth within the column. It was decided that the froth depth for the remaining tests in series C would continue to be 6cm because it resulted in the highest grade (fig. 4.13). It would aid consistency as all prior tests were performed at that froth depth.

4.3.2 Air flowrate

The superficial velocities of the column cell and the mechanical cell were maintained at an equivalent value in order to aid appropriate comparison. The mechanical cell operated at a superficial velocity (v) of 0.80 m/min based on the standard air flowrate of 20.6 L/min for all tests (Appendix C2.1). The required air flowrate in the column cell (q_{air}) was subsequently calculated to be 2.27 L/min (Table 4.1). The gauge pressure at the rotameter was 30 kPa when the 1.86 m column was used, due to the hydrostatic head and the pressure drop across the sintered disk. When the column height was doubled the new pressure drop was 50 kPa. The pressure drop was used to determine the correction factor needed in order to obtain the air flowrate that the rotameter should register (R), so as to maintain the required air velocity at the

top of the column (i.e. in the froth). The air flowrate of 2.27 L/min was used for all column tests.

Table 4.6 Summary of superficial velocities and air flowrates

<i>Unit</i>	<i>v</i> (m/min)	<i>q_{air}</i> (L/min)	<i>R</i> (L/min)
Mechanical cell	0.80	20.6	20.6
Column cell (H)	0.80	2.27	2.00
Column cell (2H)	0.80	2.27	1.86

In series C the effect of varying the air flowrate was investigated over a range of air flowrates (q_{air}) such as 1.14 L/min, 2.27 L/min and 3.41 L/min. Figures 4.14 and 4.15 show the overall mass recovery and the PGM grade recovery curves respectively.

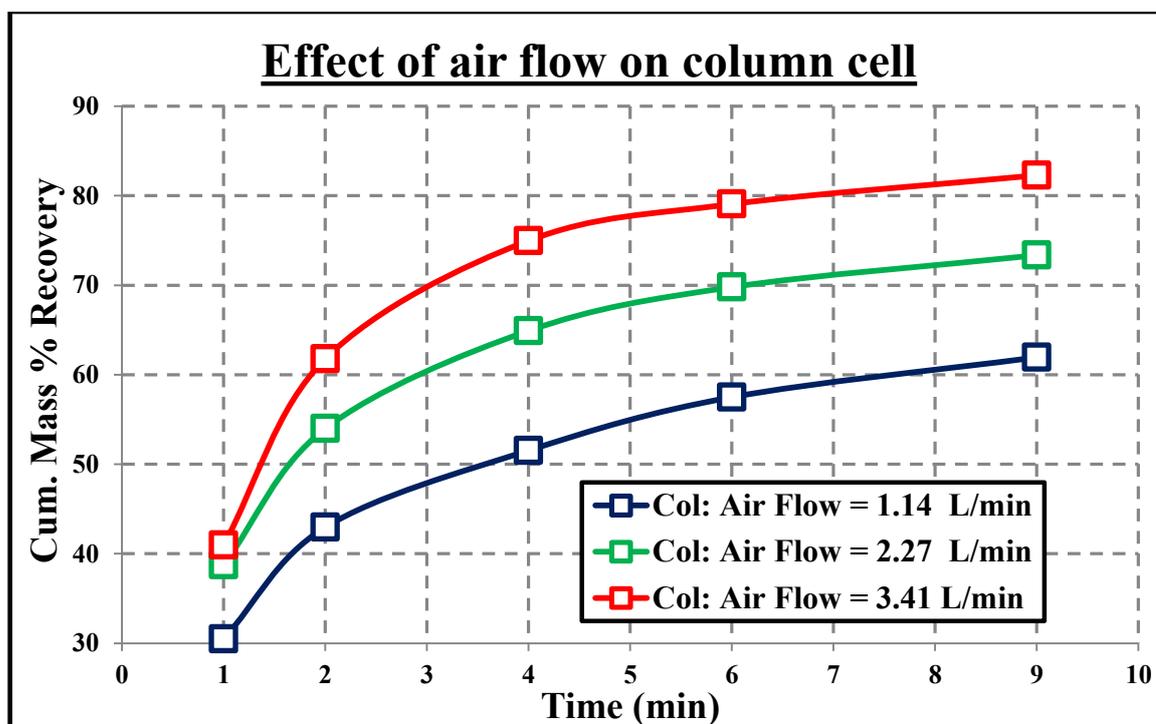


Figure 4.14 Effect of air flowrate on the mass recovery of the E3 column cell

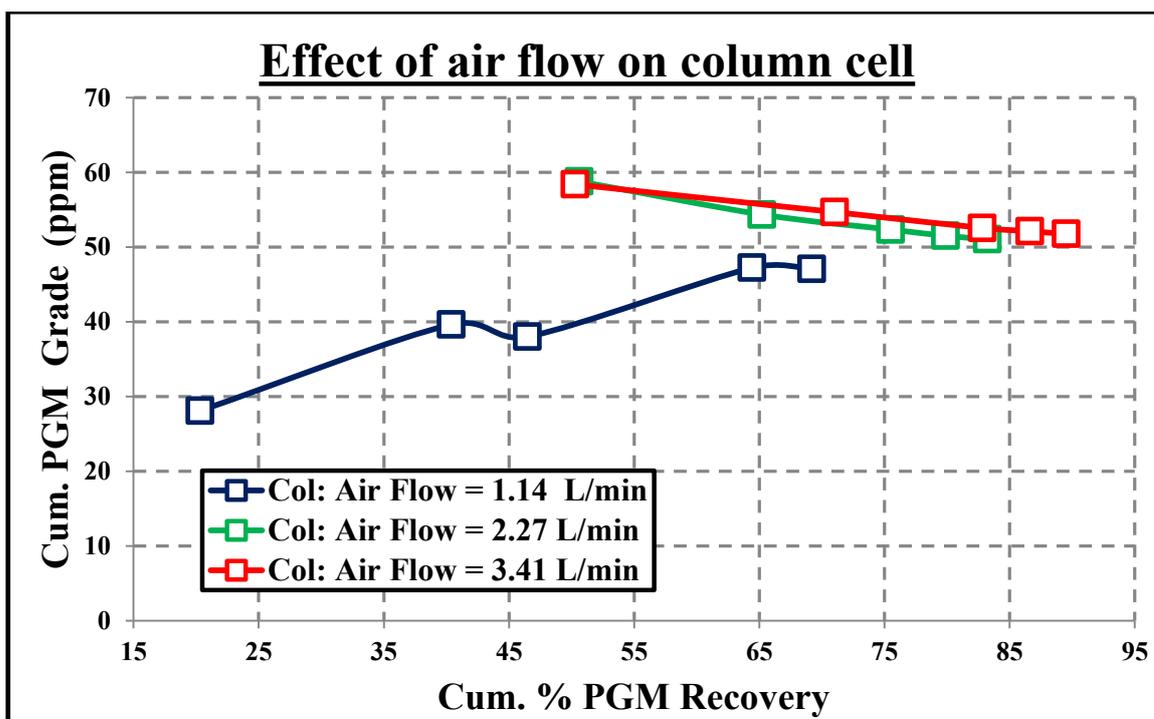


Figure 4.15 Effect of air flowrate on the PGM grade-recovery of the E3 column

The PGM recovery increased when the air flowrate was increased. The grade-recovery relationship for the 2.27 L/min and 3.41 L/min flowrates was much better than the 1 L/min flowrate. A flowrate of 3.41 L/min resulted in 6 % increase in the PGM recovery as compared with the 2.27 L/min test at a common PGM grade of 52 ppm. However, all column tests continued to operate at 2.27 L/min in order to main the same superficial velocity as the mechanical cell for adequate comparison.

4.4 BASE TESTS FOR BOTH MECHANICAL CELL AND COLUMN CELL

Standard cleaner flotation tests with no depressant (base tests) were performed on the column cell and mechanical cell. The base tests were performed in series A, B and C and are discussed below. Comparisons made between the Cu/Ni data and PGM data were based on the trends observed rather than actual percentages. All tests were done in duplicate. Error bars are shown, but are not visible in some cases, when the repeatability was good.

In certain cases the Cu and Ni trends differed from each other. In an attempt to gain clarity on this issue, a batch model with two rate constants (Kelsall, 1961), as in equation 4.1 was fitted to the recovery versus time data in series A and B.

$$R_m = a_1(1 - e^{-k_1 t}) + a_2(1 - e^{-k_2 t})$$

4.1

R_m : Cumulative mass or metal recovery (%)

a_1, a_2 : mass fraction of floatable material (fast floating and slow floating)

k_1, k_2 : distribution rate constants (1/min)

4.4.1 Base tests performed on the mechanical cell and E2 column cell (Series A)

Standard flotation tests were performed on the mechanical cell and the E2 column cell. These tests were analysed for Cu/Ni content. The mass recovered over time is shown in Figure 4.16.

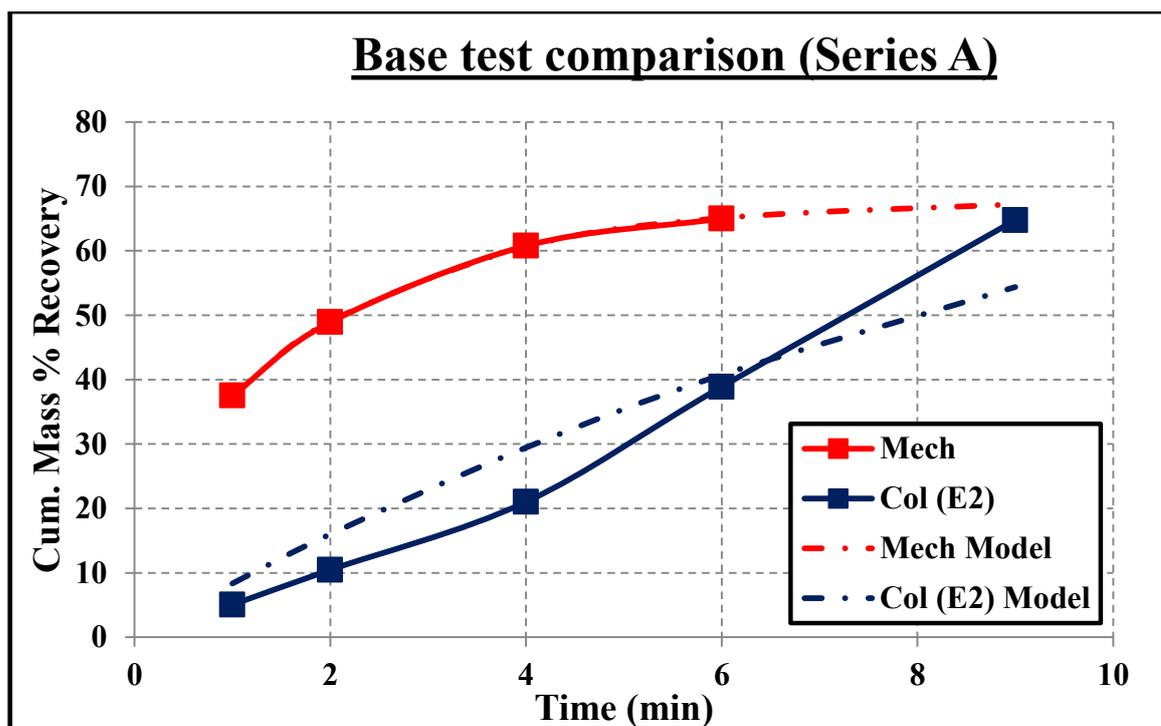


Figure 4.16 Base test mass recoveries for mechanical cell and E2 column cell

The column and mechanical cells achieved the same recovery, of about 65%, but the column was only able to do so in 9 minutes, as compared to the 6 minute duration for the mechanical cell. The rate of mass recovery in the column cell was slower and more linear, not having the usual exponential reduction in rate. This appeared to be due to bubble overloading, resulting in a constant rate of recovery at the maximum capacity of the bubbles.

The Cu/Ni recovery versus time curves were plotted (Fig. 4.17 & 4.18) in the hope on getting a better insight into the matter. The linear trend is evident for both Cu and Ni recovery in the column cell, increasing the suspicion of bubble overloading. A linear model may be applicable until the overloading condition was over.

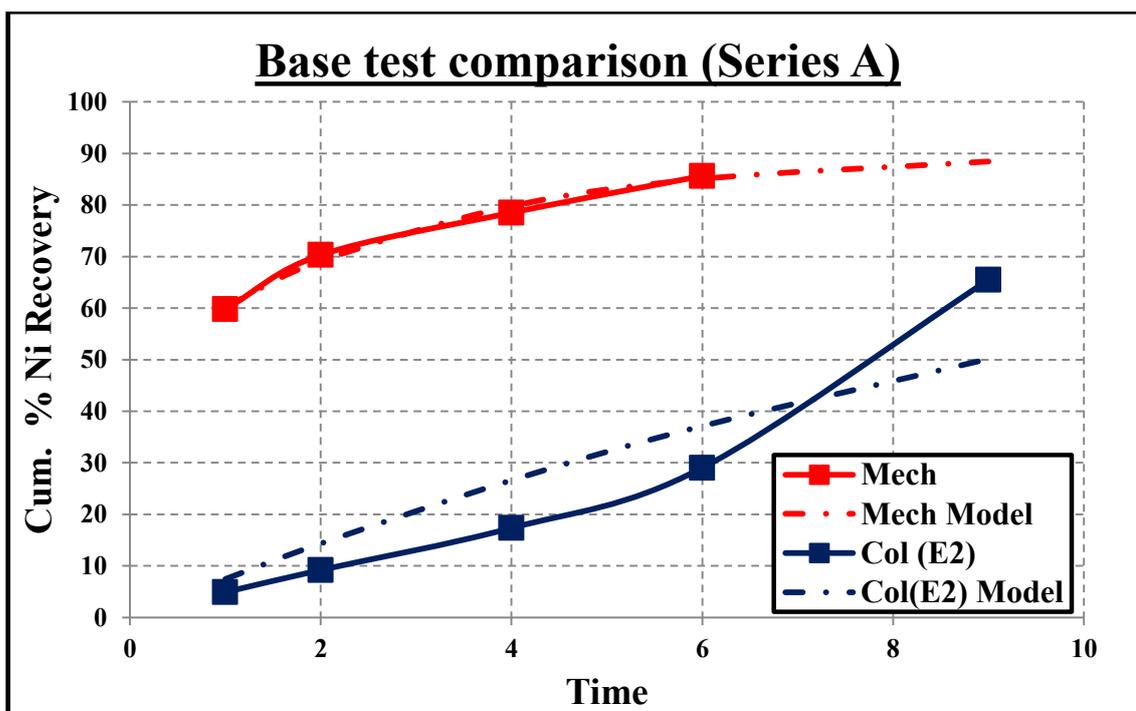


Figure 4.17 Base test Ni recovery for the mechanical cell and E2 column cell

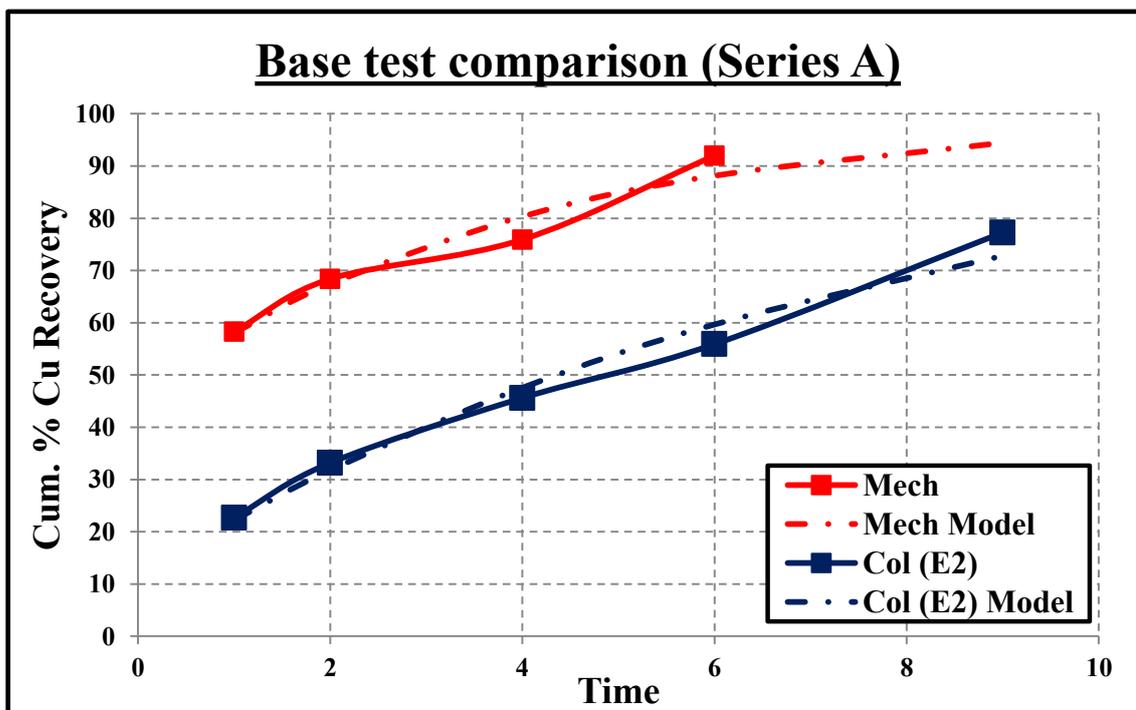


Figure 4.18 Base test Cu recovery for the mechanical cell and E2 column cell

The Cu/Ni grade recovery curves for the base tests are shown in Figures 4.19 and 4.20. The dotted lines were calculated from the regressed batch models for metal and mass recovery

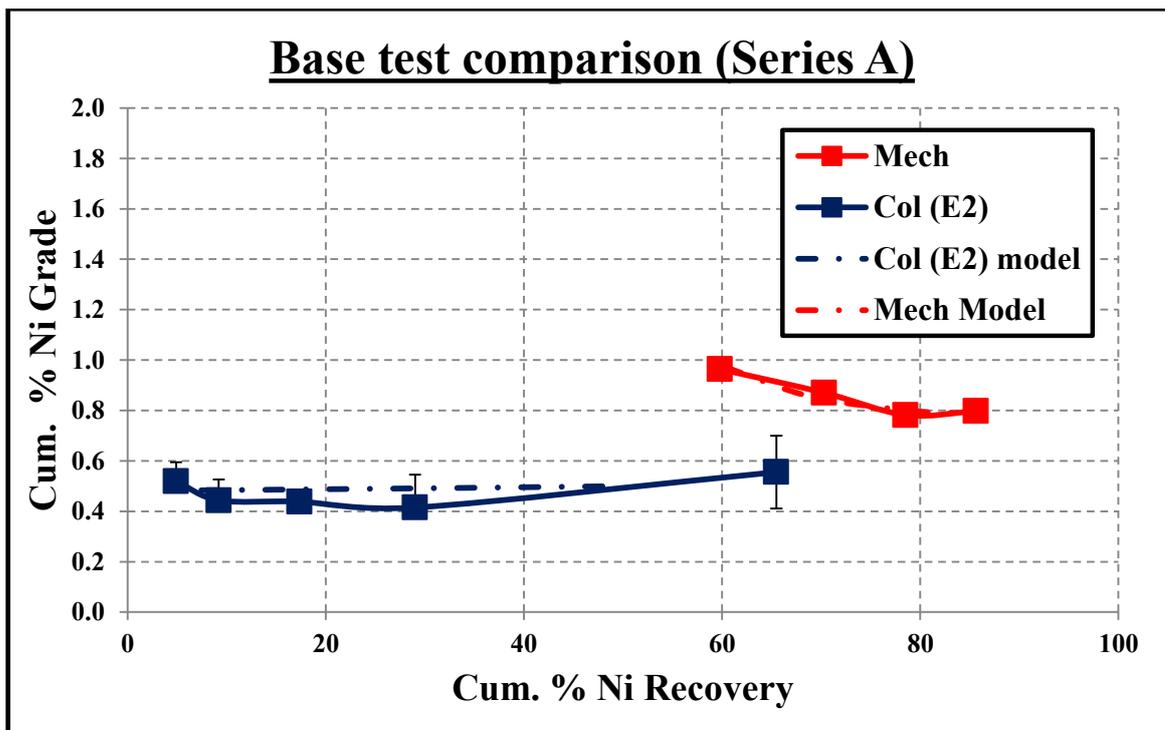


Figure 4.19 Ni grade-recovery for base tests with the mechanical cell and E2 column cell

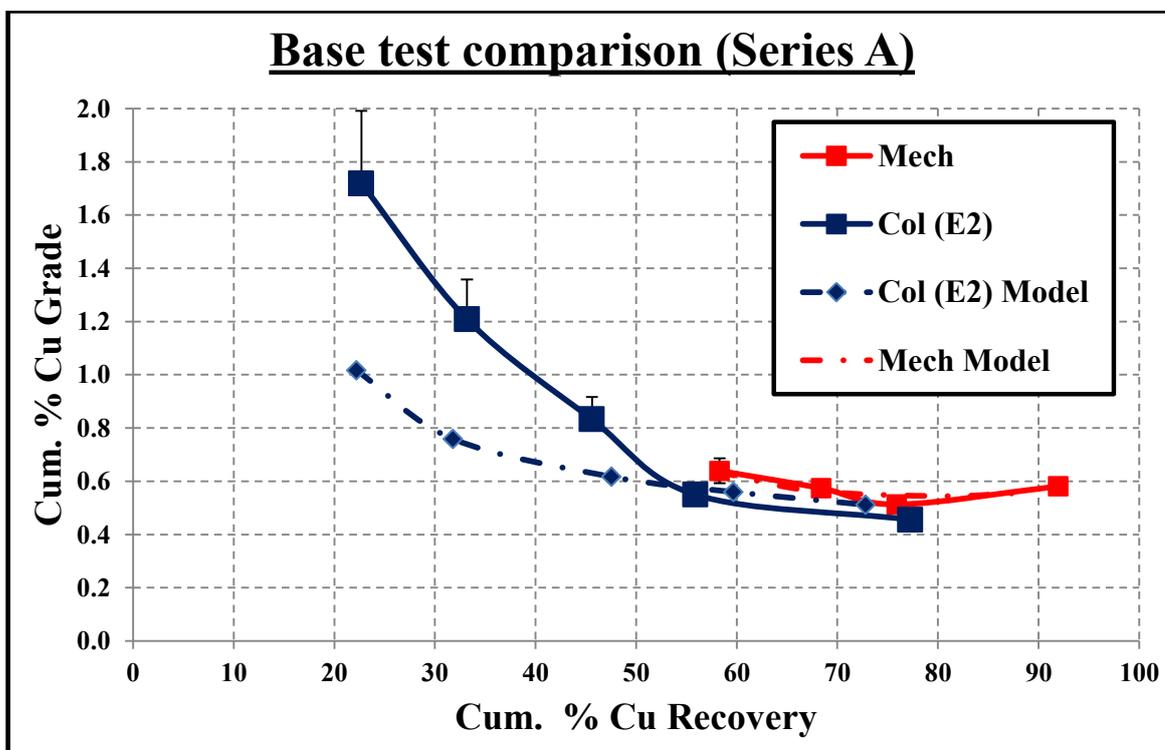


Figure 4.20 Cu grade-recovery for base tests with the mechanical cell and E2 column cell

The mechanical cell achieved a significantly higher Ni grade when compared to E2 column cell. The Cu grade and recovery (Fig. 4.3) in the mechanical cell were also higher, but only marginally so. The Cu and Ni grade-recovery curves in the column are very different, as

expected, in view of the lower concentration of copper in the ore. However, it was hoped that the recovery versus time curves would be similar and could be used as an indicator of platinum recovery. The recovery curves of copper and nickel in the mechanical cell were similar, (Figures 4.17 and 4.18), but significant differences were noted when comparing Cu and Ni recovery in the column cell.

The linear mass recovery in the column cell warranted further base tests of a longer duration. However, due to a fracture in the E2 sparger, repeat tests could not be performed.

4.4.2 Base tests performed on E3 column cell (Series B)

Standard flotation tests were performed on the column cell using the E3 sparger. These tests were also analysed for Cu/Ni content. The base tests in the column cell were performed for duration of 12 minutes, at which point no concentrate was overflowing. The graphs of mass recovery versus time and metal recoveries versus time are shown in Figures 4.21, 4.22 and 4.23.

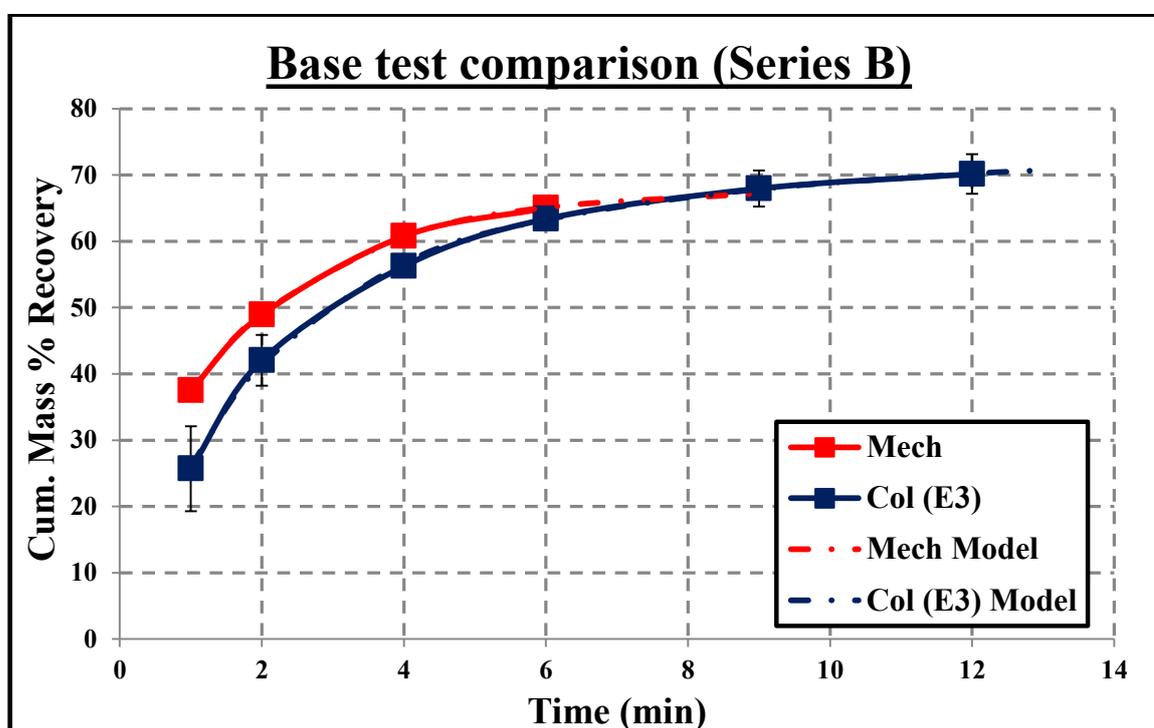


Figure 4.21 Base test mass recoveries mechanical cell and E3 column cell

The E3 column was able to achieve a similar mass recovery as the mechanical cell and in the same duration of time. The linear trend obtained in Series A was not observed in series B. The two-rate constant model fitted well. It was concluded that bubble overloading was no longer taking place and this was attributed to the finer bubbles produced by the E3 sparger, which had a smaller pore size (16-40 μm as compared to 40-100 μm). This increased the surface area available for mineral attachment.

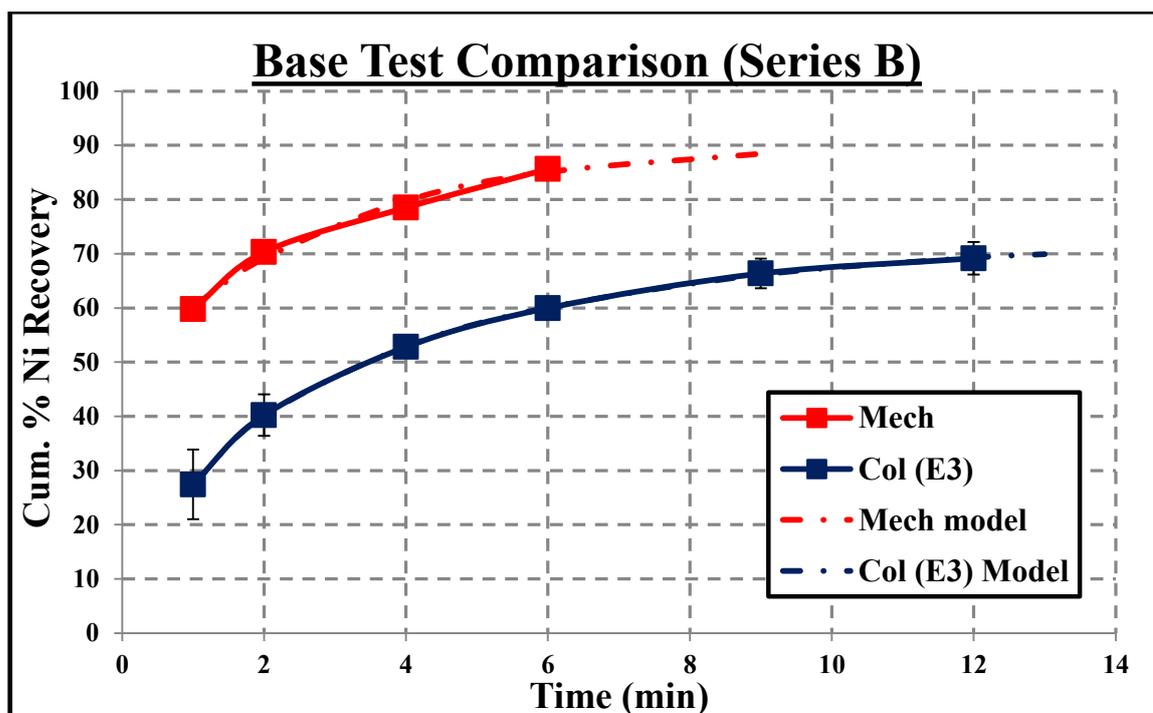


Figure 4.22 Base test Ni recovery for the mechanical cell and E2 column cell

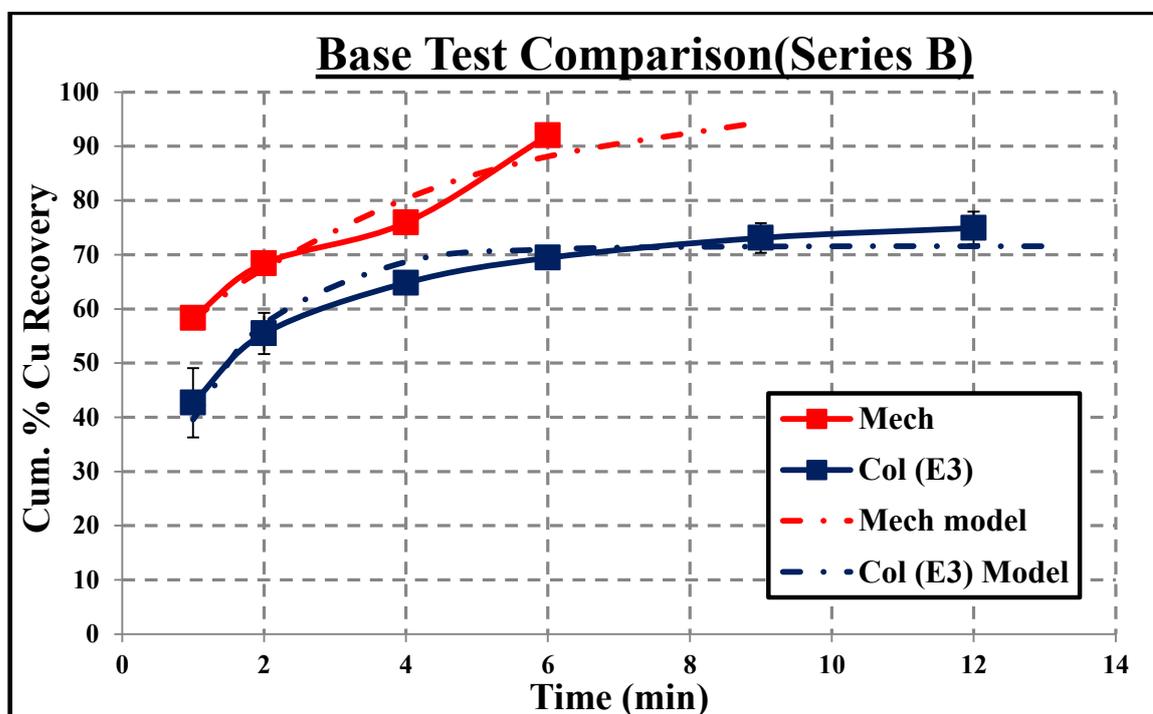


Figure 4.23 Base test Cu recovery for the mechanical cell and E2 column cell

The metal grade versus recovery relationships, for Series B, are shown in Figures 4.24 and 4.25.

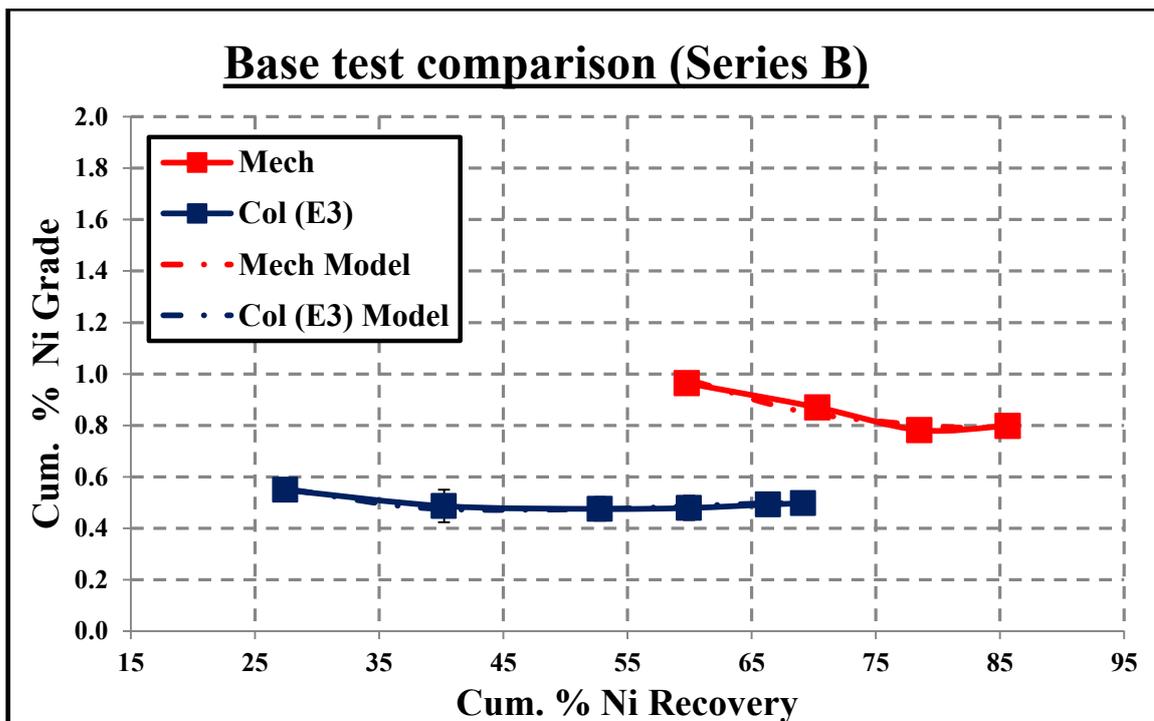


Figure 4.24 Ni grade-recovery for base tests with the mechanical cell and E3 column cell

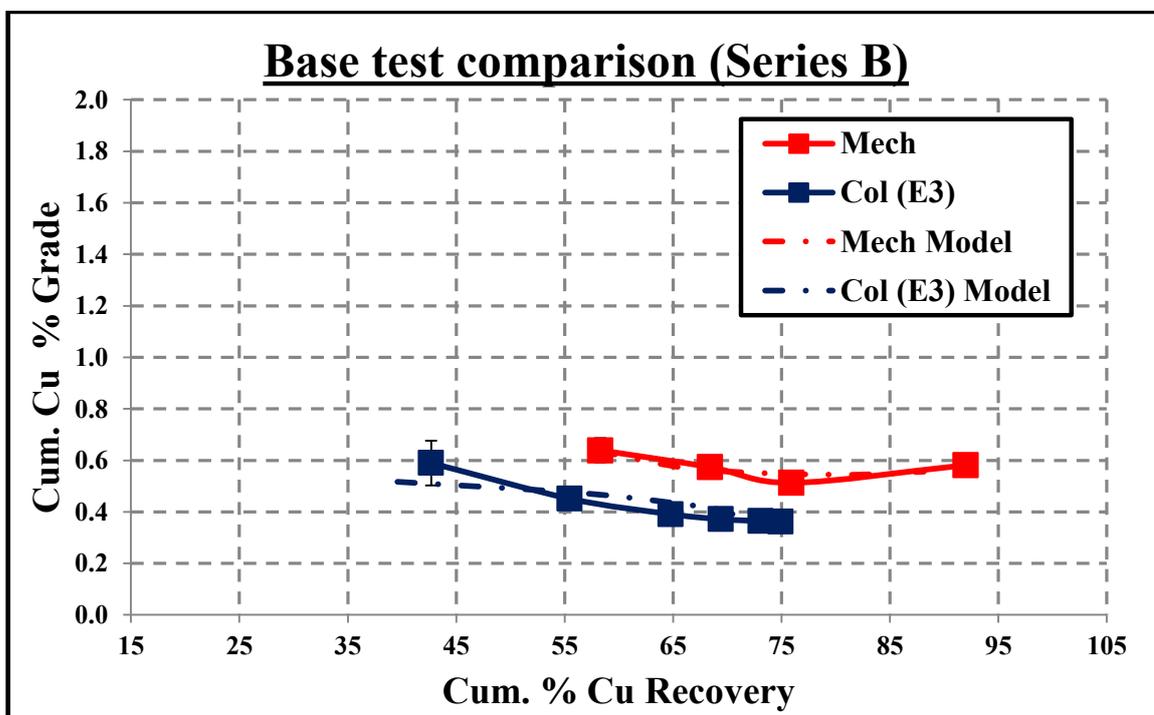


Figure 4.25 Cu grade-recovery for base tests with the mechanical cell and E3 column cell

Figures 4.24 and 4.25 show that the mechanical cell achieved a significantly higher metal grade and recovery than the E3 column thus confirming the findings of series A (fig. 4.18). No significant improvement was noted in the grade-recovery relationship as a result of the change

from the E2 sparger to E3 sparger. However, the Cu and Ni grade recovery curves in the column in series B appear much more consistent with each other than in series A.

The model parameters for series A and B are listed in Tables 4.2 and 4.3

Table 4.7 Modelling Parameters for Ni recovery

<i>Cell</i>	α_1	α_2	$\alpha_1+\alpha_2$	k_1 (1/min)	k_2 (1/min)
Mech	0.48	0.43	0.91	20.49	0.35
Col(E2)	0.00	1.00	1.00	20.46	0.08
Col (E3)	0.14	0.56	0.70	12.95	0.30

Table 4.8 Modelling Parameters for Cu recovery

<i>Cell</i>	α_1	α_2	$\alpha_1+\alpha_2$	k_1 (1/min)	k_2 (1/min)
Mech	0.46	0.54	1.00	19.33	0.25
Col(E2)	0.11	0.89	1.00	13.76	0.13
Col (E3)	0.28	0.71	1.00	15.00	0.80

The slower overall rate of flotation in the column, when using the E2 sparger, has been noted. However, regression of a model to the data has not helped to identify clear differences in the ultimate recovery between the column cell and the mechanical cell. cases (which is not achievable in practice), while un-floatable material was indicated in others. There was almost an even split between mass fractions of fast-floating and slow-floating material in the mechanical cell. However, majority of the mineral recovered in the column cell was slow-floating.

4.4.3 Base tests performed on the mechanical cell and E3 column cell (Series C)

Standard flotation tests were performed on the E3 column cell and the mechanical cell. These tests were analysed for PGM content. The cumulative mass recovered over time is shown in Figure 4.26.

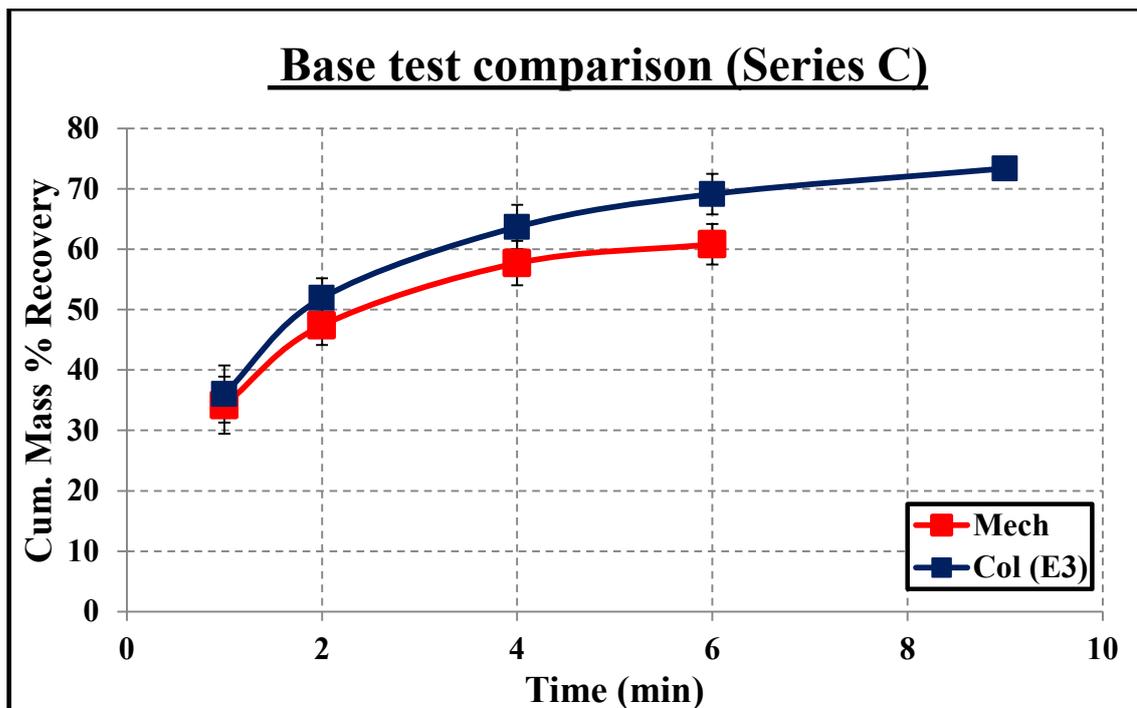


Figure 4.26 Base test mass recoveries mechanical cell and E3 column cell

The cumulative mass recovered by the E3 column was 10 % higher than the mechanical cell after 6 minutes of flotation. This was inconsistent with series A and B. This change in the mass recovery can be attributed to different ore sample utilized in series C.

The corresponding grade recovery curve is shown below in Figure 4.27.

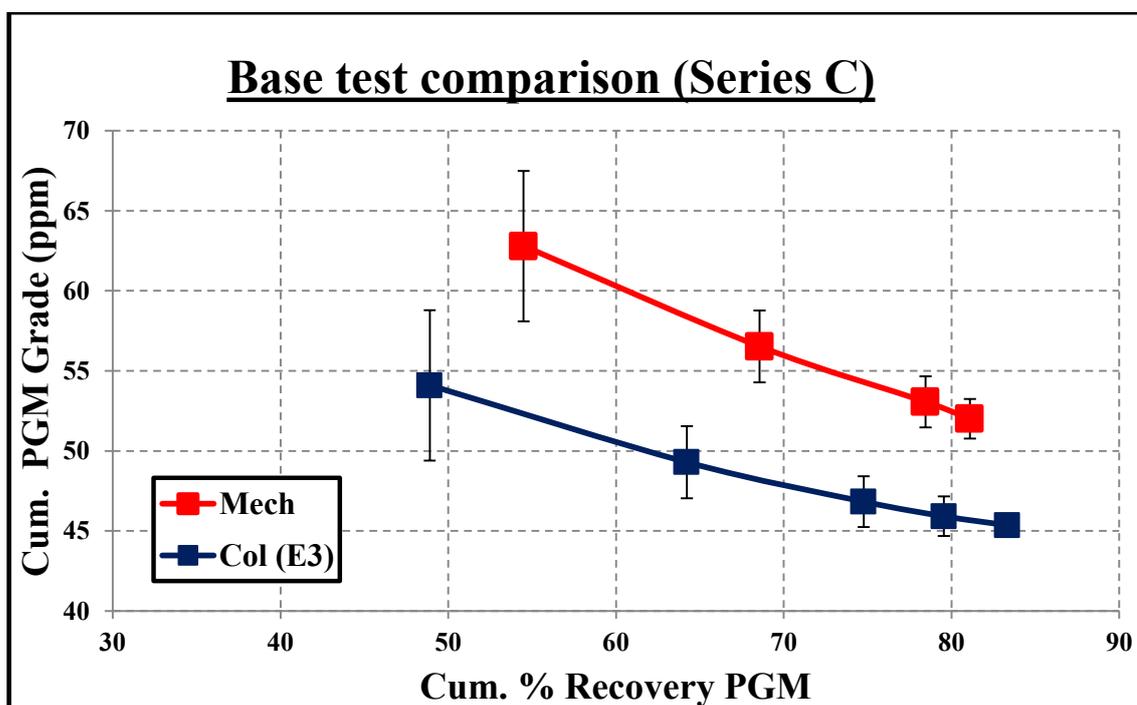


Figure 4.27 PGM grade-recovery for base tests with the mechanical cell and E3 column cell

The mechanical cell produced a higher PGM grade, with the differential being 6.5ppm at 80% recovery of PGM. The grade-recovery relationship in the mechanical cell was again significantly better.

4.4.4 Overview of the base tests for Series A, B and C

All the data from Series A, B and C demonstrated that the mechanical cell achieved a better grade-recovery relationship than the column cell. The mechanical cell produced a higher PGM grade, with the differential being 6.5ppm at 80% recovery of PGM in the mechanical and column cell in series C.

The E2 column recovered the same amount of mass as the mechanical cell but at a slower rate in series A. The mass recovery curve was more linear than exponential indicating that the bubbles may have been overloaded. The E3 column in series B used a sintered disc with a smaller pore size. The smaller bubbles produced implied that a larger surface area would be available for hydrophobic particles to attach to therefore the mass recovery increased. Simultaneously, the probability of bubble overloading was decreased. However, there was no significant improvement in the grade-recovery relationship with the E3 column as compared to the E2 column.

It is important bear in mind that the column used in this study was a prototype column and parameters such as the bubble size were not optimized. The column operated under dilute conditions of 6% solids rather than the recommended 23% (Anglo Platinum Research Laboratories) due to limitations in the milling capacity and because the volume of rougher concentrate was also limited by the cleaner capacity.

4.5 THE EFFECT OF USING A RECYCLE DURING COLUMN FLOTATION

The column used in this study allowed for the optional use of a recirculation pump in accordance with Hay's (2010) findings in section 2.6.4. Tests were performed on the column cell using the recycle in series A, B and C. These tests were compared to the corresponding base tests.

4.5.1 The effect using the recycle with the E2 column cell (Series A)

Standard column tests were performed on the E2 column employing the use of a recycle. These tests were analysed for Cu/Ni content.

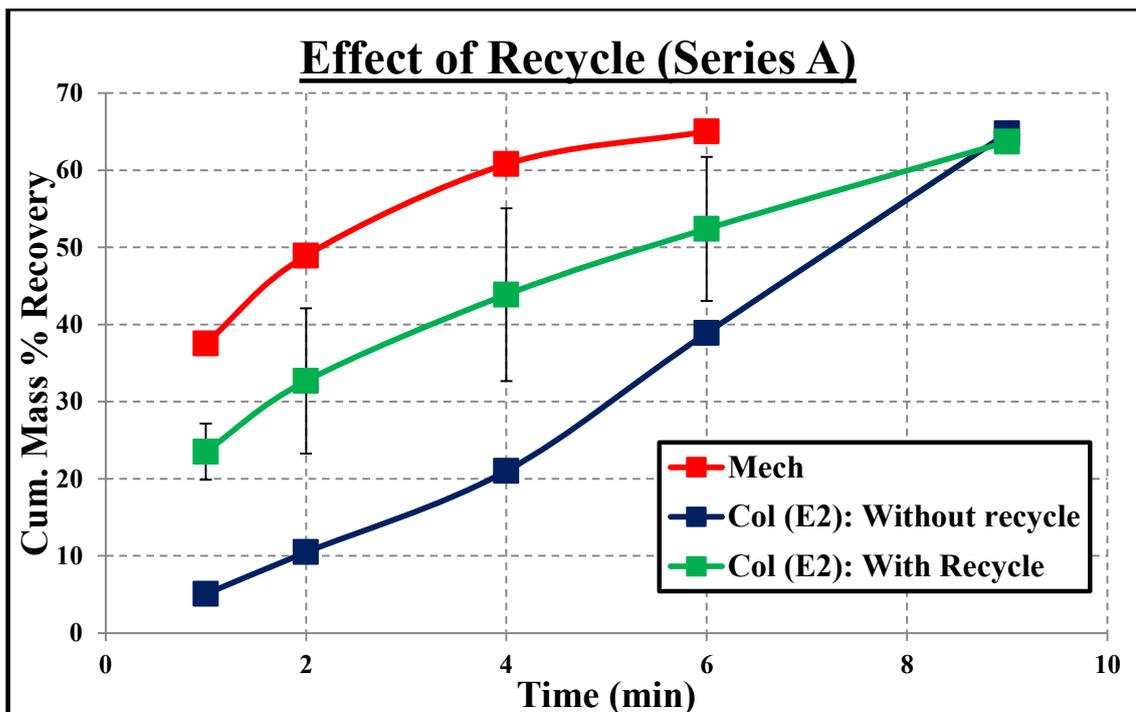


Figure 4.28 Effect of using a recycle on the mass recovery of the E2 column

Figure 4.28 showed that using the recycle increased mass recovery initially, but the overall recovery fairly similar to the base column test.

The Cu/Ni grade-recovery curves are shown below in Figures 4.29 and 4.30.

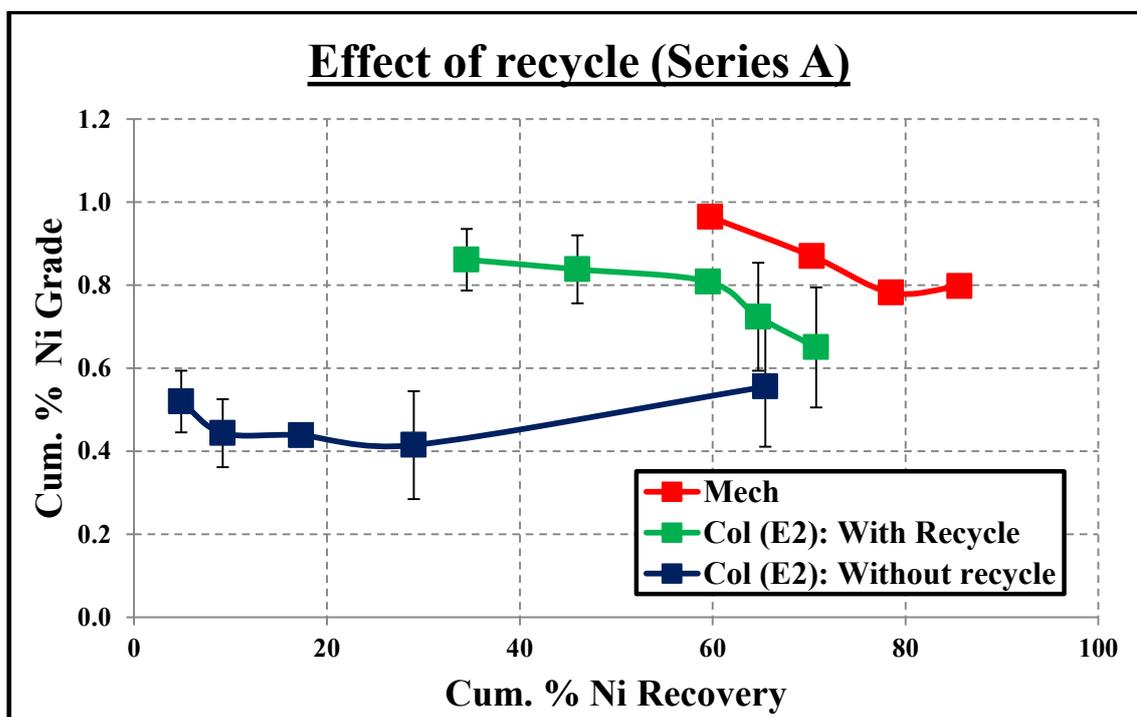


Figure 4.29 Effect of using a recycle on the Ni grade-recovery of the E2 column

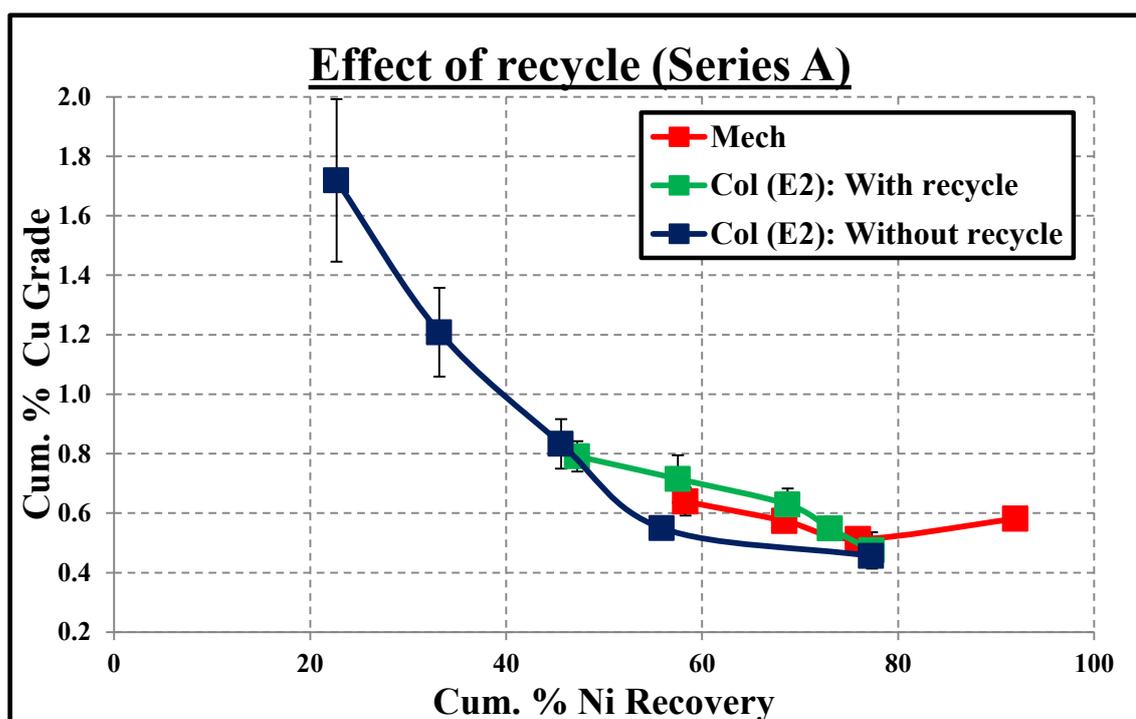


Figure 4.30 Effect of using a recycle on the Cu grade-recovery of the E2 column

The Ni grade-recovery relationship improved with the use of the recycle. The Cu grade and recovery, on the other hand, remained fairly similar to the base test in the E2 column. The grade-recovery results were therefore inconclusive in series A.

4.5.2 The effect of using the recycle with E3 column cell (Series B)

Standard column tests were performed on the E3 column employing the use of the recycle. These tests were also analysed for Cu/Ni content.

The cumulative mass recovered over time is shown in Figure 4.31. The overall mass recovered in the column cell when the recycle was used was marginally lower than base tests in series B.

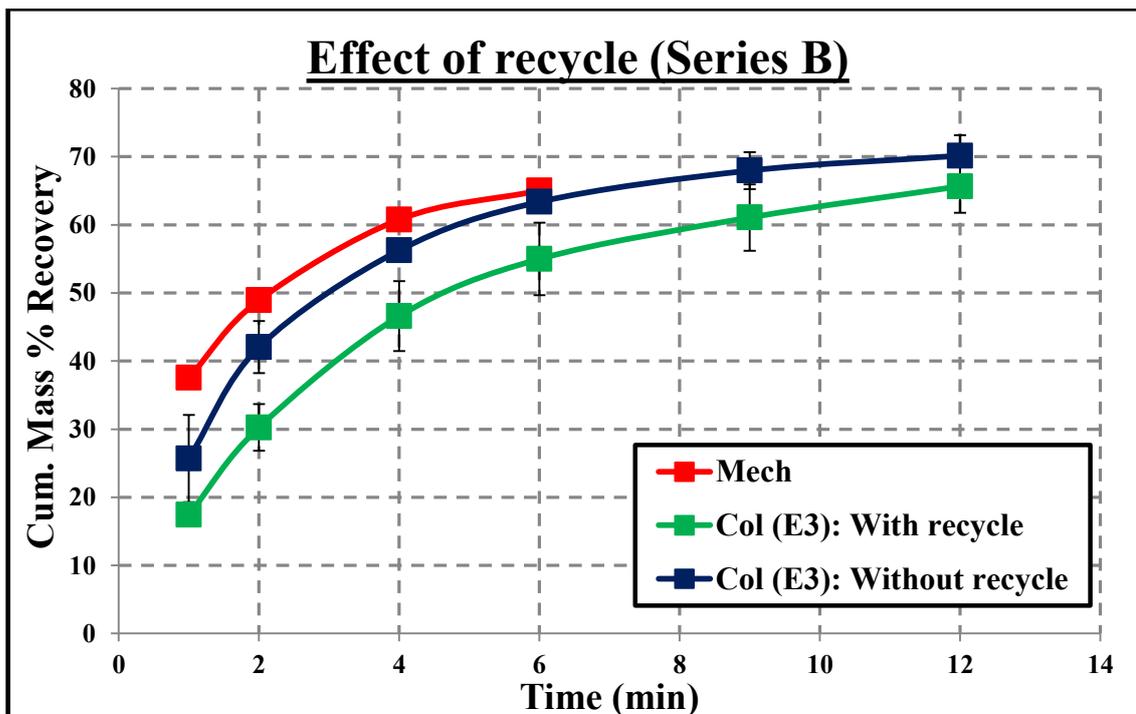


Figure 4.31 Effect of using a recycle on the mass recovery of the E3 column

The corresponding grade-recovery curves are shown below in Figures 4.32 and 4.33 for Ni and Cu respectively.

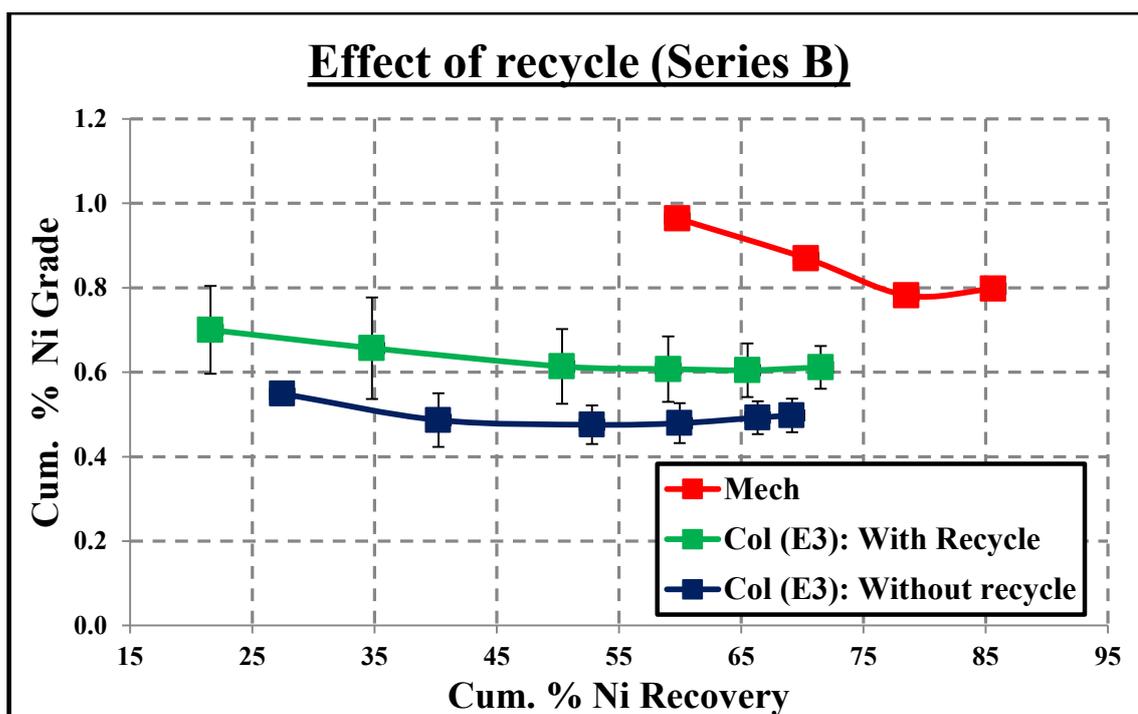


Figure 4.32 Effect of using a recycle on the Ni grade-recovery of the E3 column

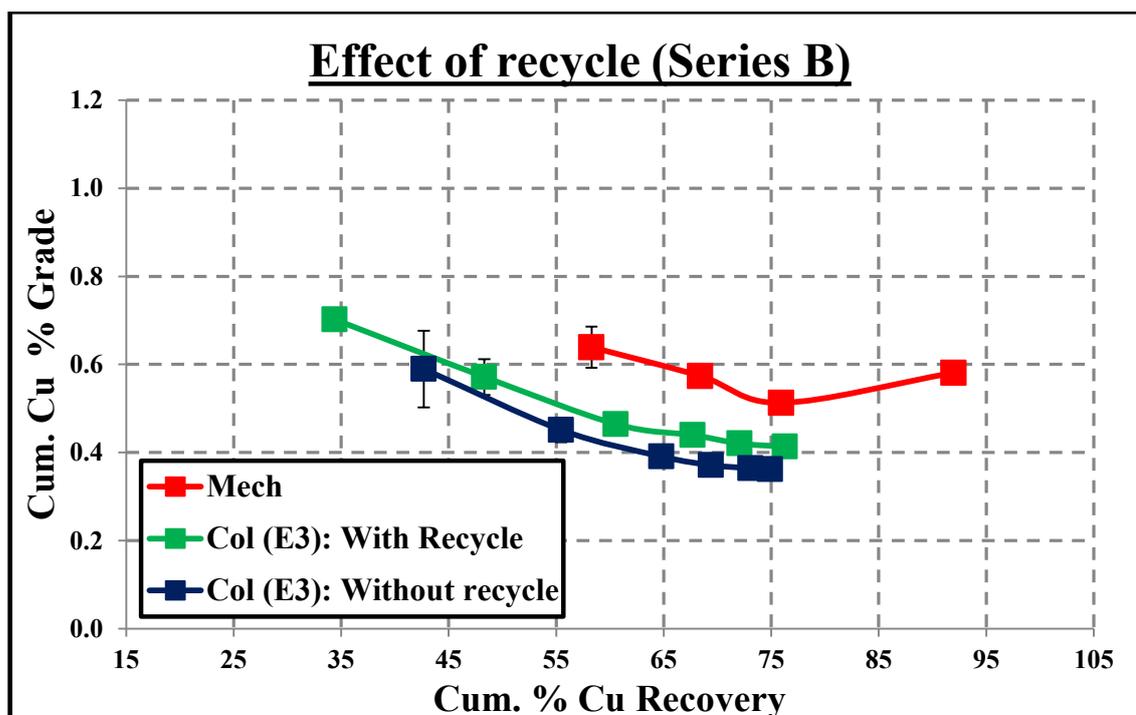


Figure 4.33 Effect of using a recycle on the Cu grade-recovery of the E3 column

Figures 4.32 and 4.33 indicated that by using the recycle the grade and recovery of both the Cu and Ni minerals improved. This confirmed the trend found with Ni grade-recovery curve for the E2 column in Series A (Fig. 4.29).

4.5.3 The effect using the recycle with E3 column cell (Series C)

Standard column tests were performed with the E3 column employing the use of a recycle. These tests were analysed for PGM content. The overall mass recovered is shown in Figure 4.34. The mass recovery was marginally lower than the base tests. This re-affirmed the trend in the mass recovery in series B.

The corresponding PGM grade-curve (Fig. 4.35) indicated that the improvement in the grade-recovery curve seen in series B was at best only marginal when the recycle was used in series C. For a common head grade of 40 ppm, the difference between the base test and the test with the recycle was only 0.1 ppm.

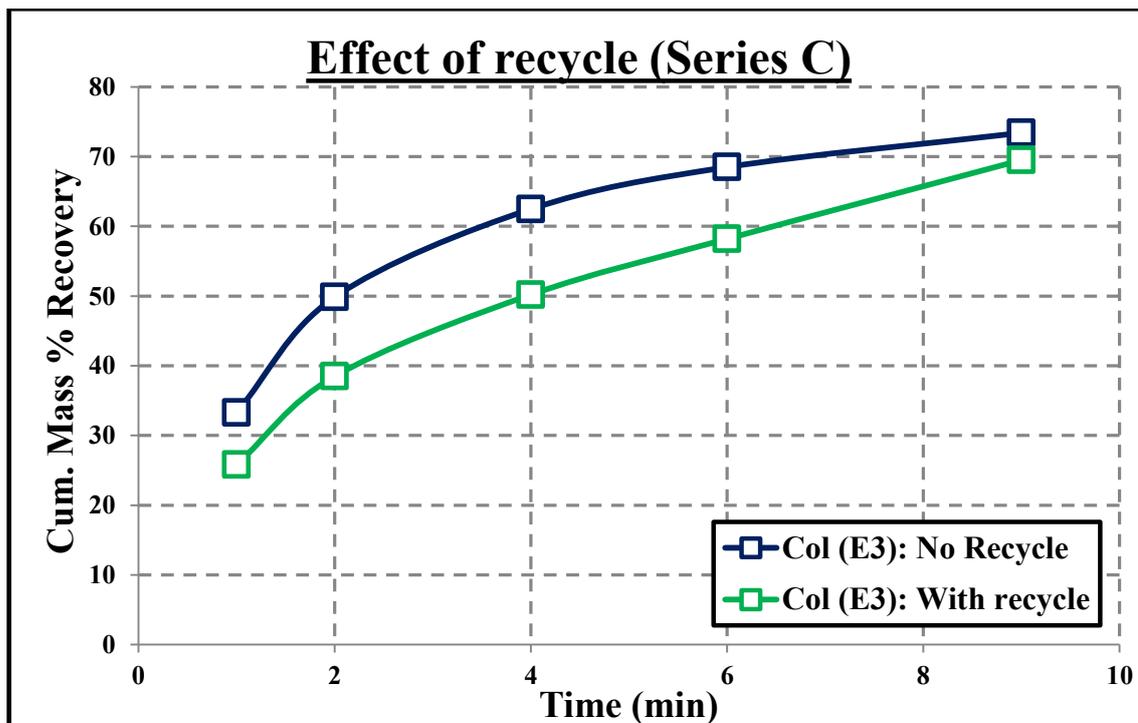


Figure 4.34 Effect of using a recycle on the mass recovery of the E3 column

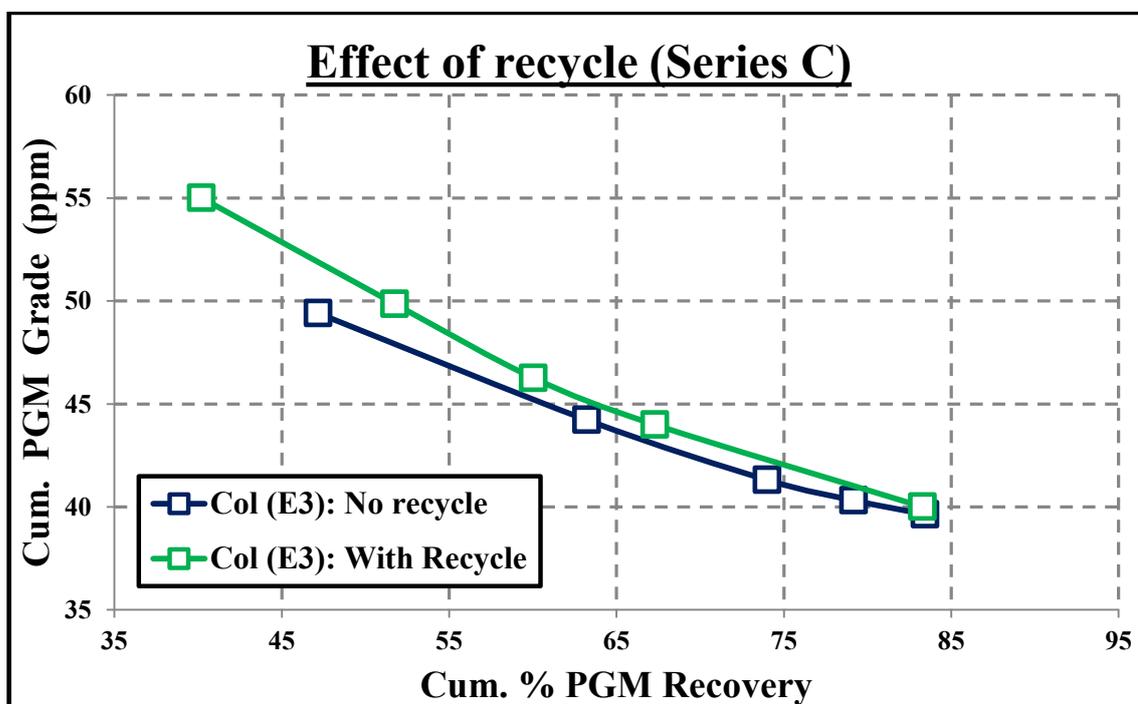


Figure 4.35 Effect of using a recycle on the PGM grade-recovery of the E3column

In this series the results for the mechanical cell do not appear in Figures 4.34 and 4.35 because these tests were in the region of the lower recalculated head grade of 33.3 ppm for which a mechanical cell grade recovery curve was not available.

4.5.4 Overview of the effect of the recycle on column flotation for Series A, B and C

In series A the Ni grade-recovery relationship showed that the column cell performed better when the recycle was used. The Cu results showed that the using the recycle did not effect the grade-recovery relationship. Series A was therefore inconclusive. Series B showed an improvement in the grade-recovery relationship was noted for both Cu and Ni when the recycle was used. However, ultimately the PGM analysis in series C showed that the improvement noted was only marginal (0.1 ppm).

4.6 THE EFFECT OF DEPRESSANT DOSAGE

Various dosages of depressant were added to the mechanical and column cell in this section. The range of depressant dosage was 10g/t, 20g/t and 30 g/t. The effect of varying the depressant dosage was tested in Series A, B and C. The results are reported in the following two sub-sections.

4.6.1 The effect of depressant dosage on the mass recovery of the mechanical & column cells

The cumulative mass recovered over time was reported for all three series of depressant dosage tests.

4.6.1.1 The effect of depressant dosage on the mechanical cell and E2 column cell (Series A)

The mechanical cell and E2 column cell were tested at 10g/t and 30g/t depressant dosages. The overall mass recovered is shown in Figure 4.36.

The overall mass recovered by the mechanical cell was more than double that of the column for a 10g/t depressant addition. A similar difference occurred for a 30 g/t depressant addition. This is surprising considering that the overall mass recovered with mechanical cell in the base tests was almost identical to that of the E2 column cell.

This behaviour of the depressant was in line with the theory proposed by Whitney and Jan (1996) who explained that vigorous impeller action in the mechanical cell tends to scrub the absorbed CMC from the depressed mineral which is then refloated. The column does not require such vigorous action and therefore the mineral remains depressed (section 2.4.1.4) resulting in a lower mass recovery, as seen in Figure 4.36.

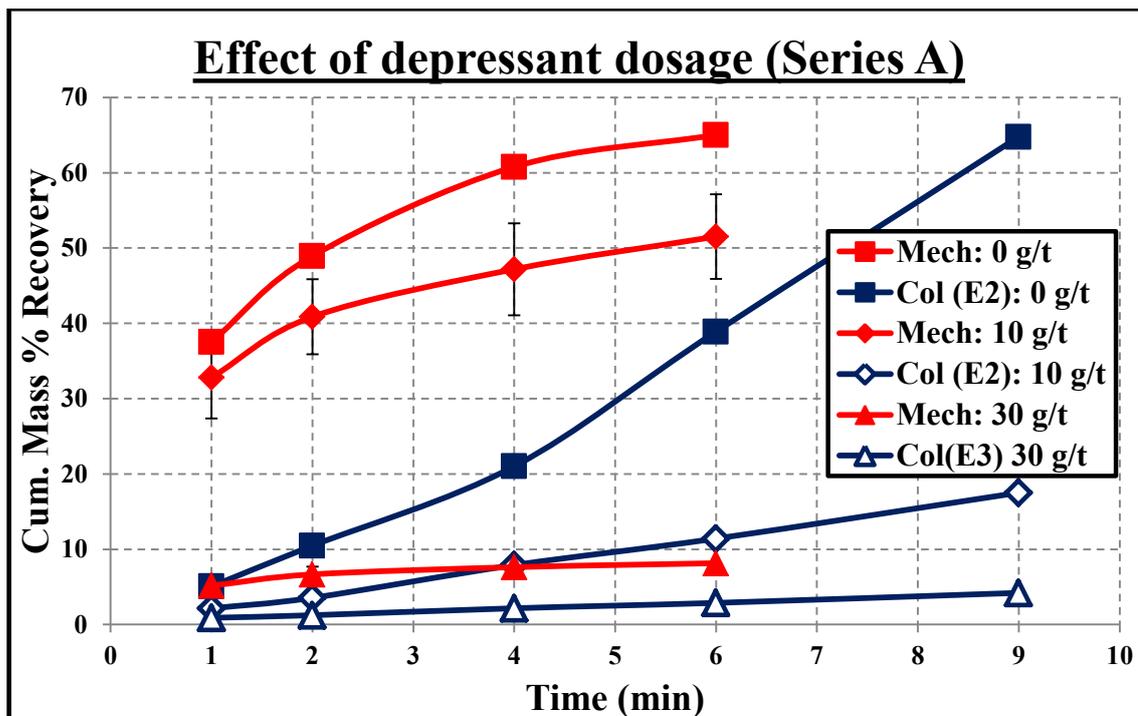


Figure 4.36 Effect of depressant dosage on mass recovery of mechanical cell and E2 column cell

4.6.1.2 The effect of depressant dosage on the mechanical cell and E3 column cell (Series B)

The mechanical cell and E3 column cell were tested at 10g/t and 30g/t depressant dosages. The overall mass recovered is shown in Figure 4.37.

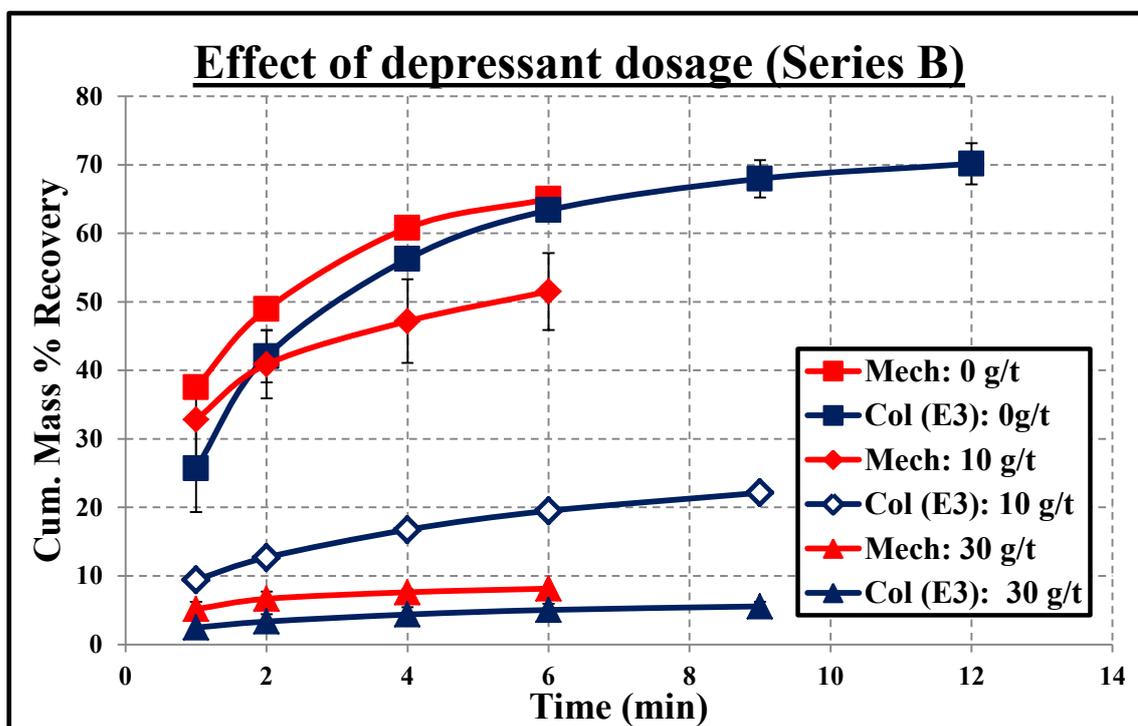


Figure 4.37 Effect of depressant dosage on mass recovery of mechanical cell and E2 column cell

The trend observed in the mass recovered with the E3 column in series B was almost identical to that of the E2 column for different depressant dosages. Twice as much mass was recovered in the mechanical cell as compared to the column cell for a 10g/t depressant addition. The column cell recovered one third of the mass recovered by the mechanical for an addition of 30 g/t of depressant. The mass recovery data indicated that the column cell was more sensitive to the depressant dosage. The cost of the depressant is significant and hence this was an interesting finding, pointing to a saving in reagent costs when using column cells for cleaning.

4.6.1.3 The effect of depressant dosage on the mechanical cell and E3 column cell (Series C)

Tests were performed on the mechanical and E3 column cell for 10 g/t, 20 g/t and 30 g/t depressant dosages. The overall mass recoveries are shown in Figure 4.38.

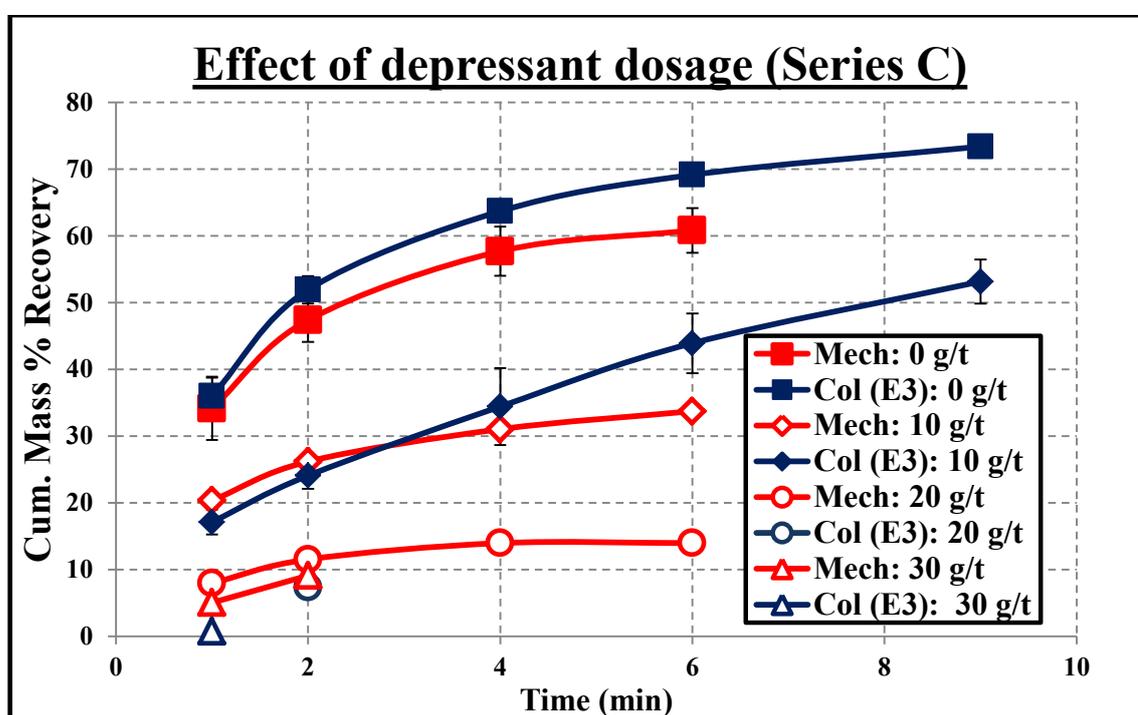


Figure 4.38 Effect of depressant dosage on mass recovery of mechanical cell and E3 column cell.

At first glance, the mass recovery in series C seemed to contradict that of series A and B. The mass recovery in the E3 column cell exceeded that of the mechanical cell for a 10 g/t depressant dosage. However, Figure 4.19 was rescaled in Figure 4.20. The base tests and 10g/t depressant dosage tests were omitted for easier visibility.

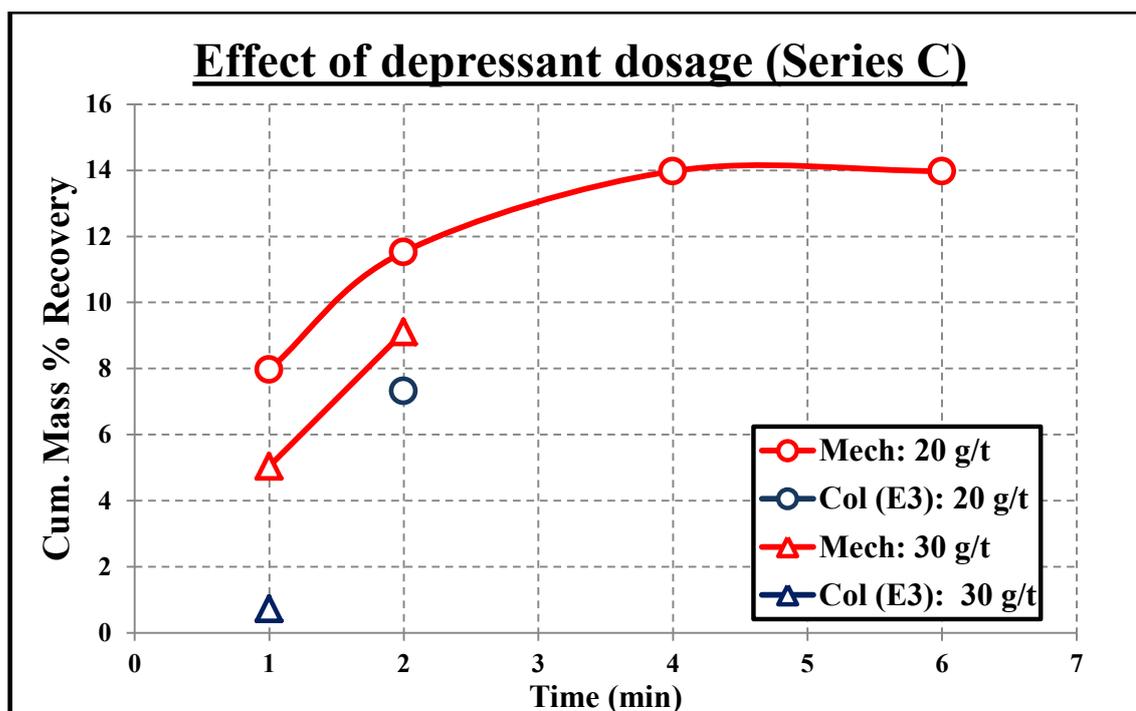


Figure 4.39 Effect of depressant dosage on mass recovery (magnified)

The column cell recovered half the mass that was recovered by the mechanical cell for a depressant dosage of 20 g/t. The mechanical test recovered 9 times more mass than the column cell for a 30 g/t addition of depressant.

The change in the mass recovery for the 10 g/t test in series C can be attributed to the fact that a new ore sample was used for the tests in this series. The ore sample was unlikely to be of an identical mineralogical content as the first sample. It can be assumed that the effect of mechanical agitation on the depressant behaviour became more visible at the higher depressant dosages (20 g/t and 30 g/t) with the new ore sample. The mass recoveries at different depressant dosages for series A, B and C indicated that the column cell appeared to be more sensitive to depressant dosage than the mechanical cell. However, these results would have to be confirmed in the subsequent sections.

4.6.2 The effect of depressant dosages on the grade-recovery curves of mechanical & column cell

This section contains the grade-recovery results for all three series of depressant dosage tests.

4.6.2.1 The effect of depressant dosage of the mechanical cell and E2 column cell (Series A)

The mechanical cell and E2 column cell were tested at 10g/t and 30g/t depressant dosages. The Cu/Ni grade-recovery curves are shown in Figures 4.40 and 4.41.

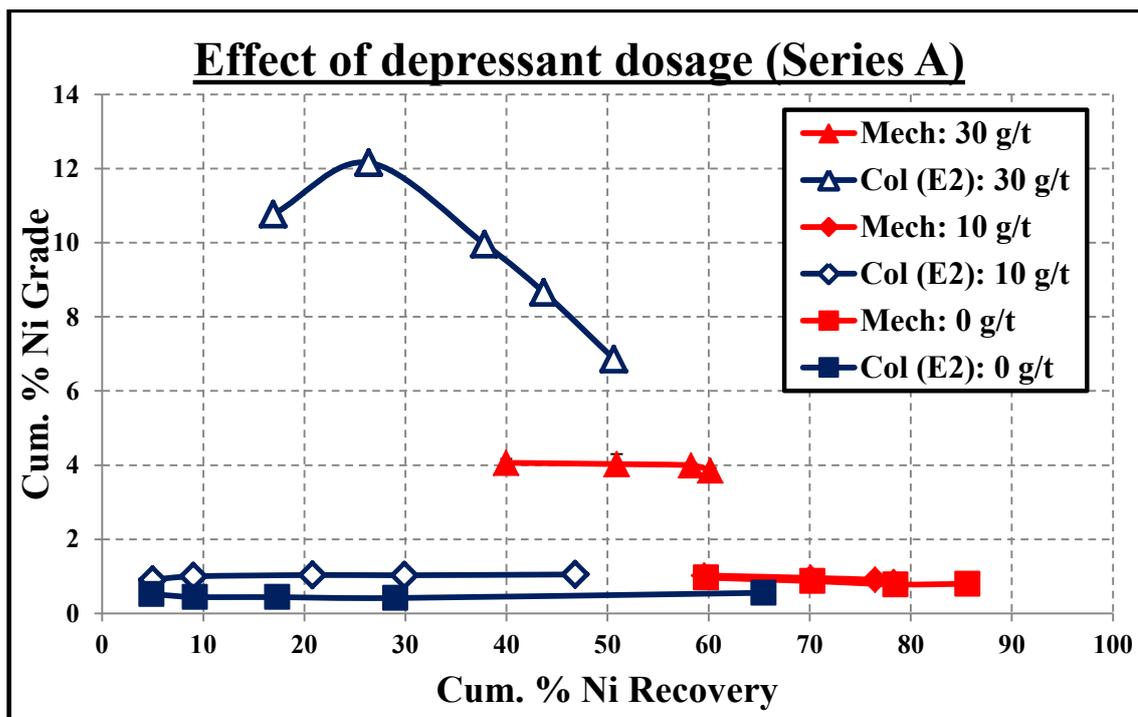


Figure 4.40 Effect of depressant dosage on the Ni grade-recovery on the mechanical cell and E2 column cell

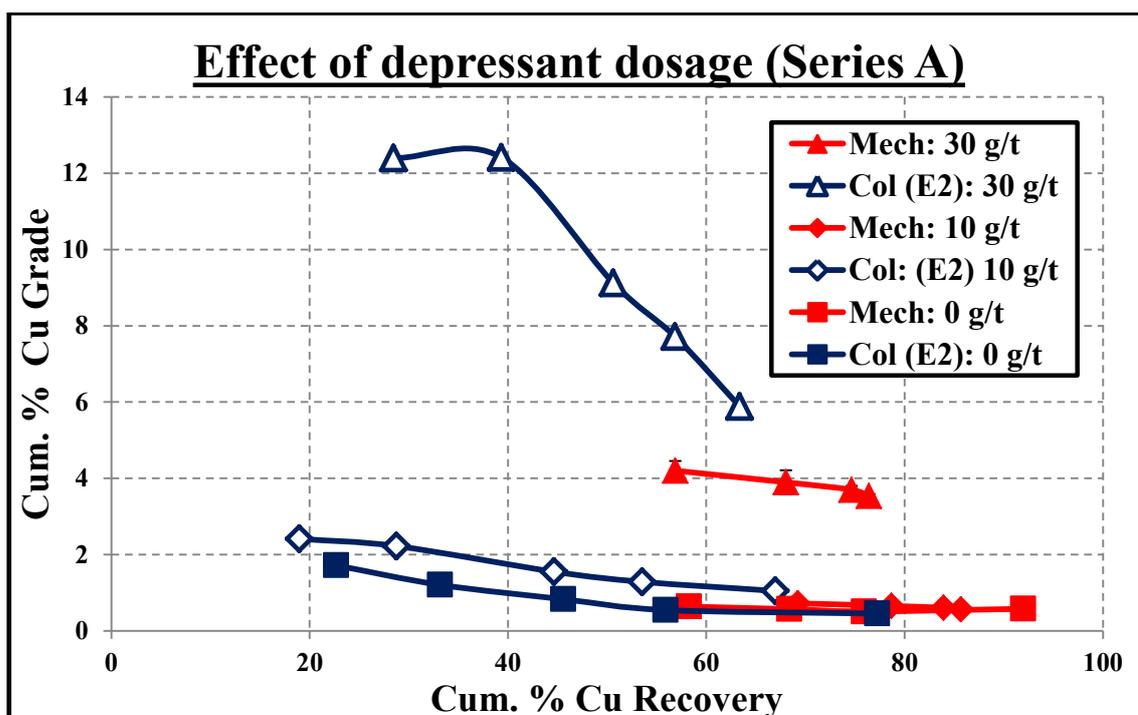


Figure 4.41 Effect of depressant dosage on the Cu grade-recovery on the mechanical cell and E2 column cell

Figures 4.40 and 4.41 showed that the mechanical cell achieved a significantly higher metal recovery. However, the E3 column achieved a substantially higher grade for the 30 g/t depressant dosage test. Figures 4.40 and 4.41 are rescaled in Figures 4.42 and 4.43 respectively. The 30 g/t depressant tests were omitted for easier visibility.

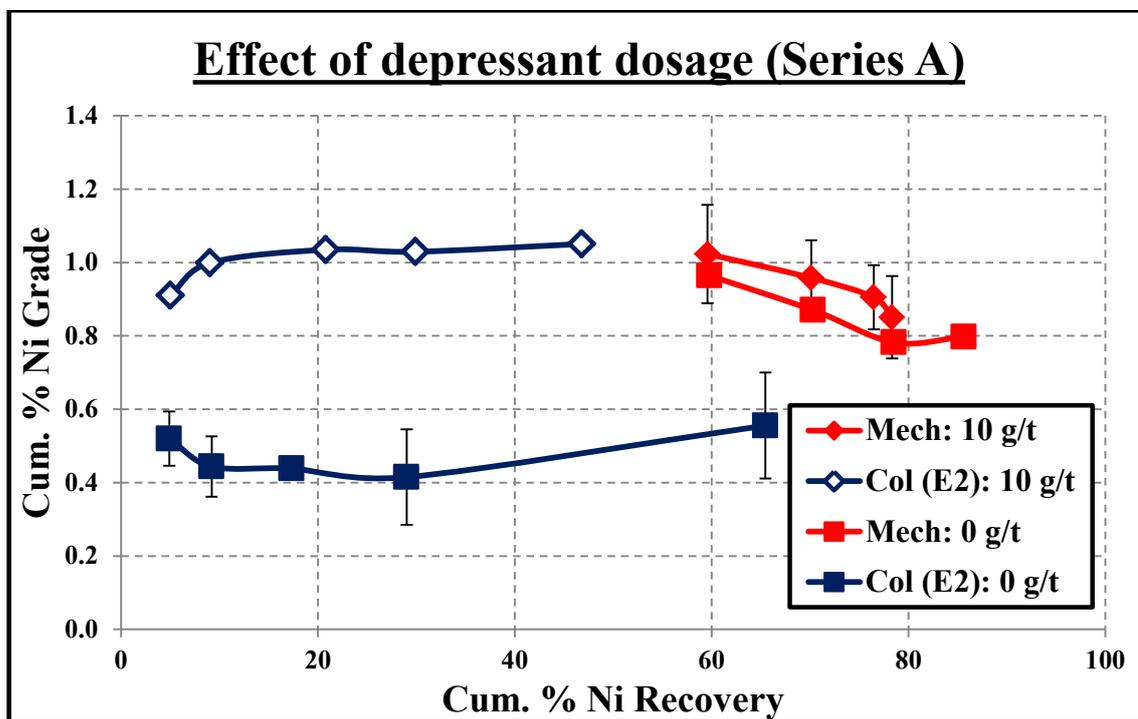


Figure 4.42 Effect of depressant dosage on the Ni grade-recovery (magnified)

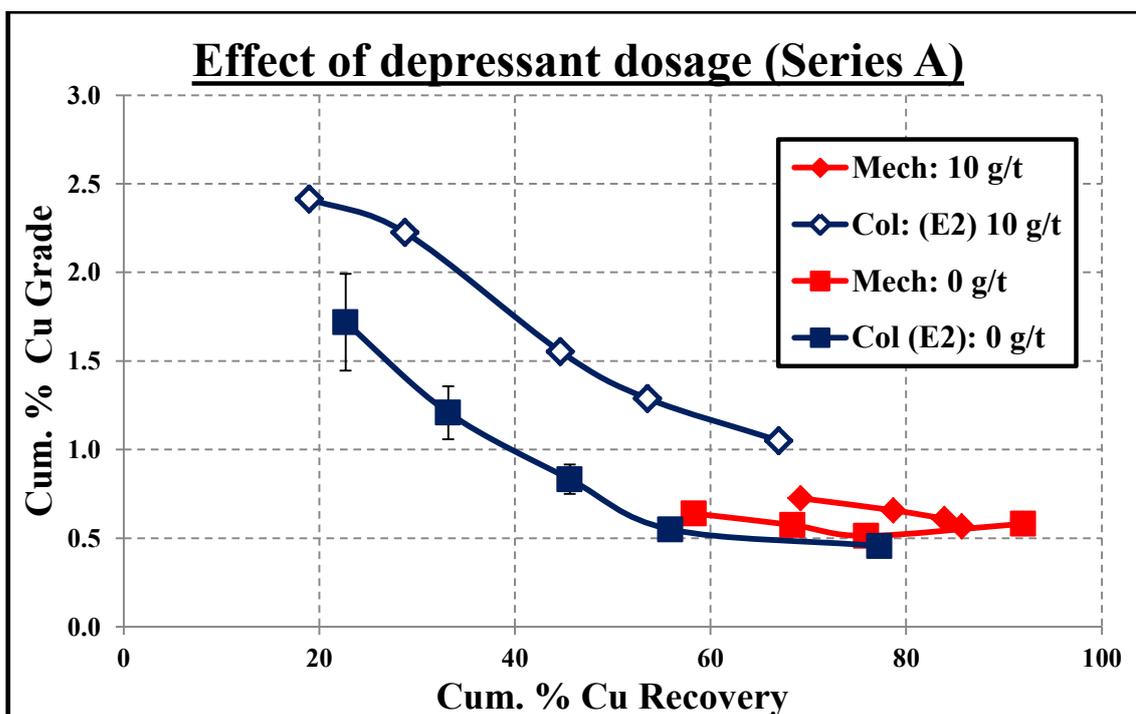


Figure 4.43 Effect of depressant dosage on the Cu grade-recovery (magnified)

It was interesting to note that the addition of 10 g/t of depressant did not result in any significant change in the metal grade or recovery in the mechanical cell as compared to its base test. The metal grade in the column cell, on the other hand, doubled with an expected decrease in recovery for a 10 g/t depressant addition. This further substantiates Whitney and Jan's (1996) theory (section 2.6.1.4).

The depressant dosage tests with the E2 column were not repeated because the E2 sintered disc fractured. The recalculated head grade of the 30 g/t depressant test were within the stipulated Ni and Cu grade range (section 4.1). However, the recalculated head grade of the 10 g/t depressant test was 20% lower than the Ni and Cu grade range. Since the previous sparger was not available for further tests, no replicate tests were performed. The experiments were repeated with the new sparger (E3) instead.

4.6.2.2 The effect of depressant dosage on the mechanical cell and E3 column cell (Series B)

The mechanical cell and E3 column cell were tested at 10g/t and 30g/t depressant dosages. The Cu/Ni grade-recovery curves are shown in Figures 4.44 and 4.45. The grade relationship obtained by the mechanical cell was superior at all depressant dosages.

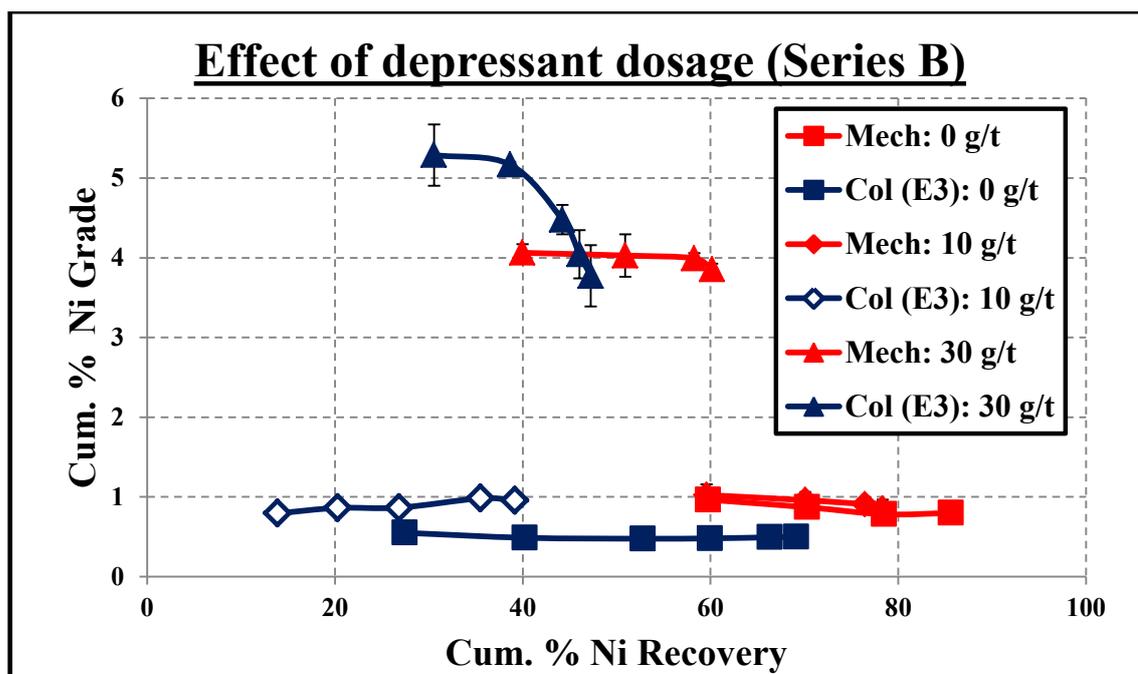


Figure 4.44 Effect of depressant dosage on the Ni grade-recovery of the mechanical cell and E3 column cell

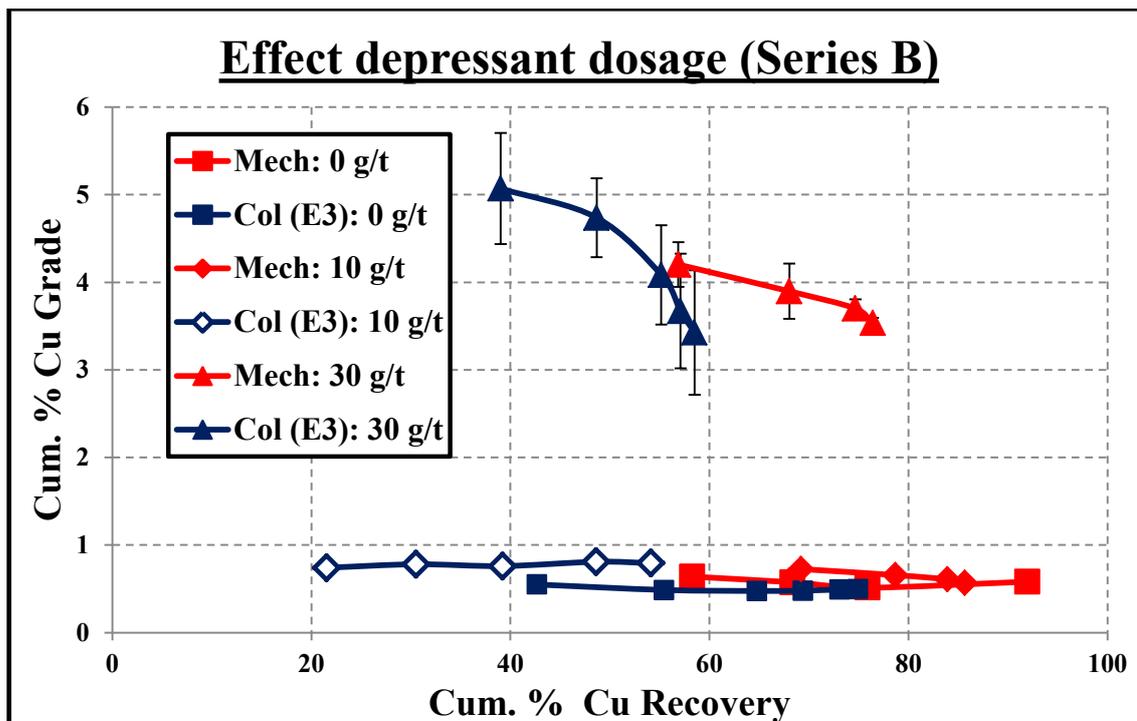


Figure 4.45 Effect of depressant dosage on the Cu grade-recovery of the mechanical cell and E3 column cell

The metal grade obtained by the column cell was similar to that of the mechanical cell for a dosage of 30 g/t but the column recovery was significantly lower. The above graphs were rescaled in figures 4.46 and 4.47. The 30g/t depressant test was omitted for easier visibility.

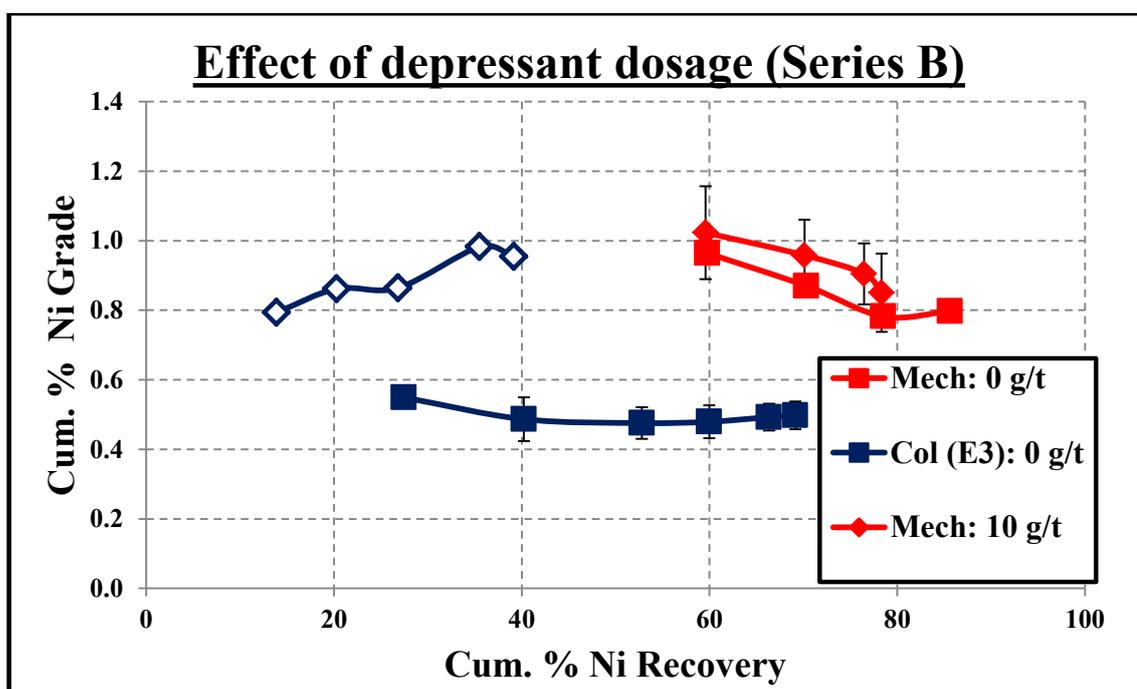


Figure 4.46 Effect of depressant dosage on the Ni grade-recovery (magnified)

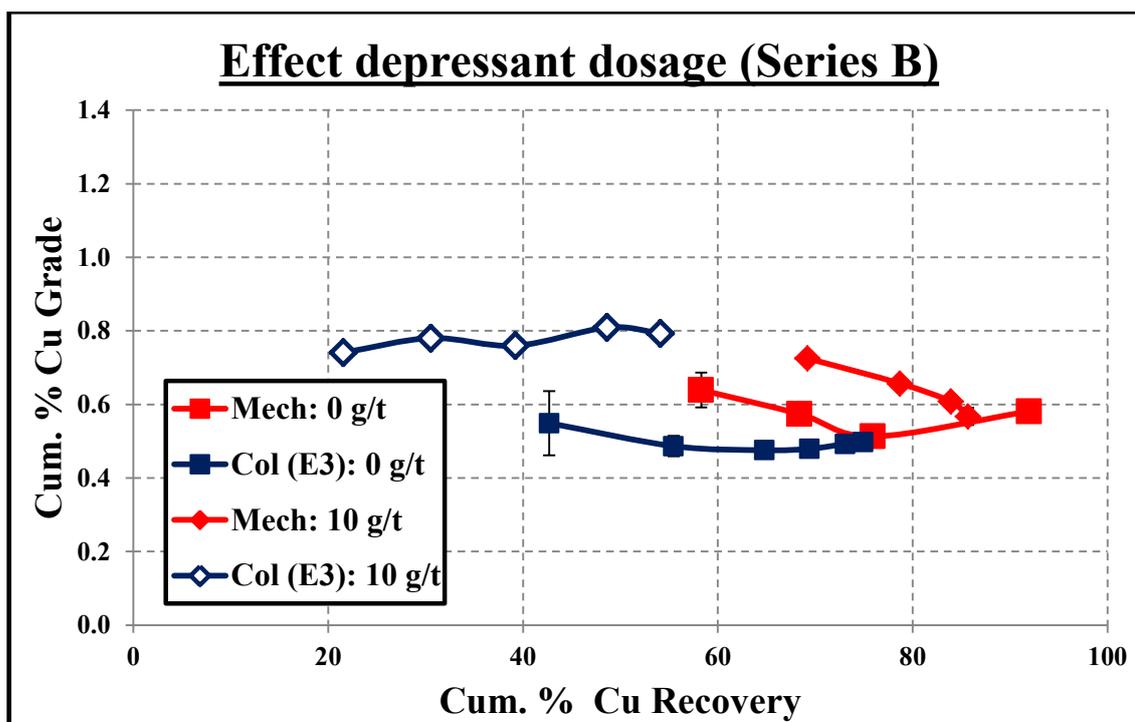


Figure 4.47 Effect of depressant dosage on the Cu grade-recovery (magnified)

The metal grade obtained in the E3 column for a 10g/t depressant dosage was almost double that of the base test. The mechanical cell showed no significant change in the grade and recovery in response to the 10g/t addition of depressant. These results reaffirmed the experiments performed in Series A. It further validated the observation that the action of the depressant was more pronounced in the column cell.

4.6.2.3 The effect of depressant dosage on the mechanical cell and E3 column cell (Series C)

The mechanical cell and E3 column cell were tested at 10g/t, 20g/t and 30g/t depressant dosages. The tests were analysed for PGM content. The PGM grade-recovery curves are shown in Figure 4.48 which is rescaled in Figure 4.49. The 30 g/t and 20 g/t depressant dosage tests are omitted for easier visibility.

The mechanical cell consistently obtained higher metal recoveries at all depressant dosages. The 10g/t depressant addition tests showed that PGM grade obtained by the mechanical cell was 40 ppm higher than the column cell for a common PGM recovery of 80%. This was a reversal of the findings in series A and B.

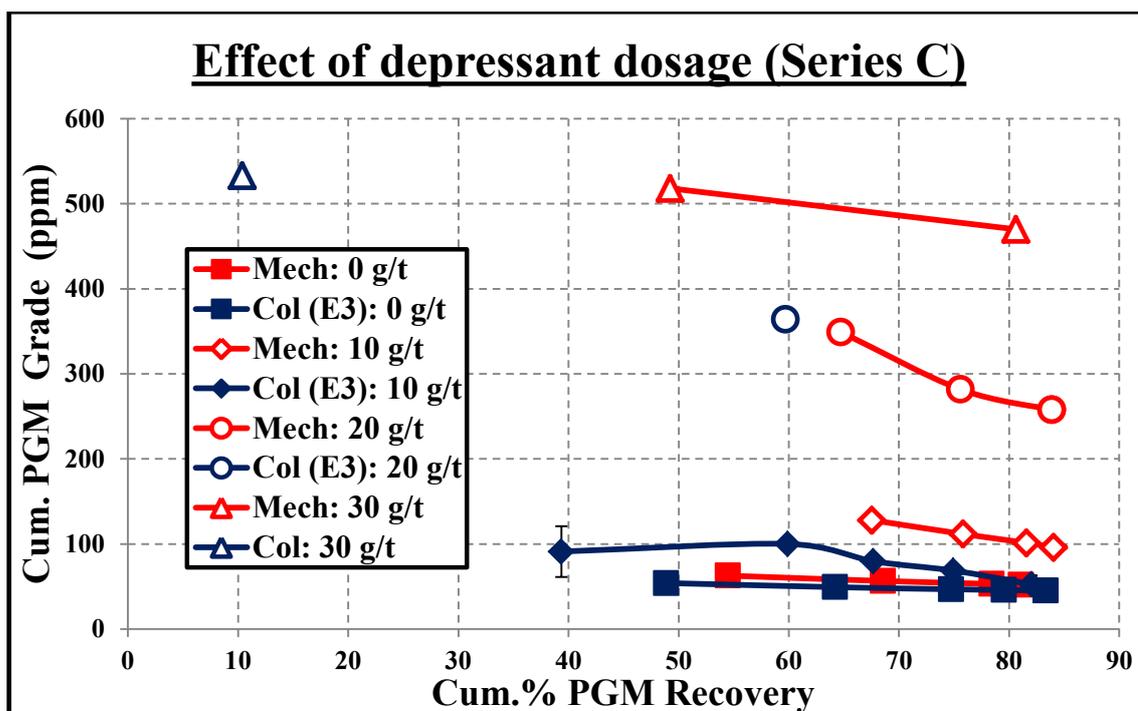


Figure 4.48 Effect of depressant dosage on the PGM grade-recovery of the mechanical cell and E3 column cell

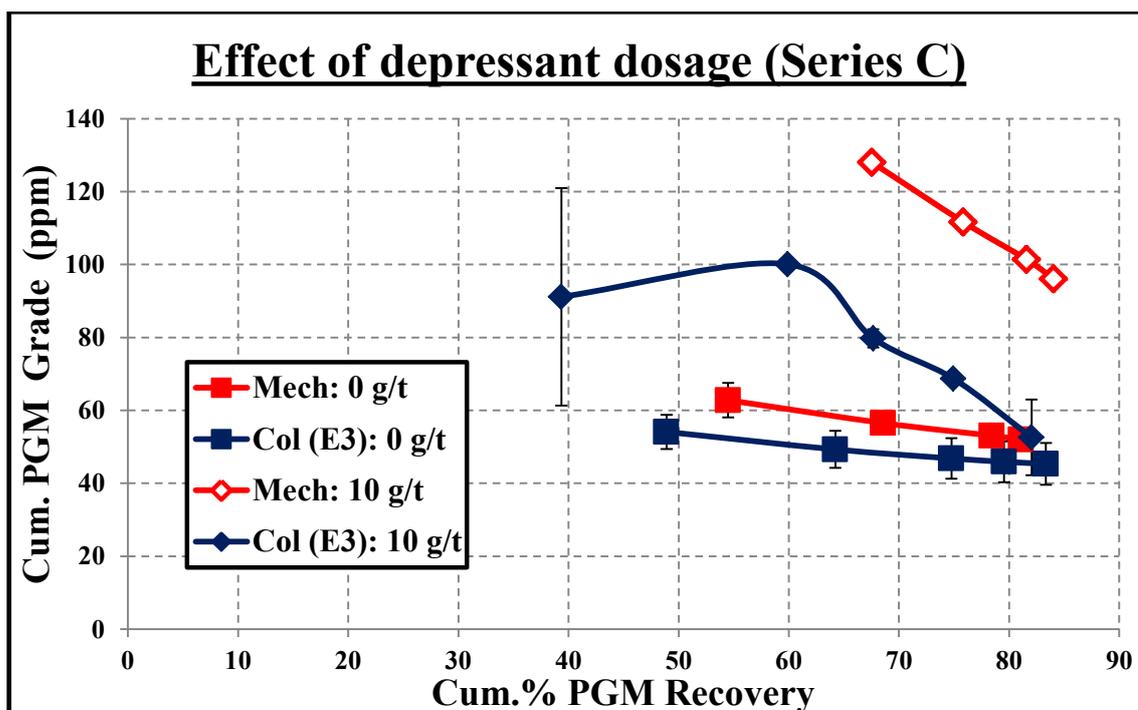


Figure 4.49 Effect of depressant dosage on the PGM grade-recovery (magnified)

However, the cumulative PGM grade achieved by the mechanical cell was 63 ppm lower than the column cell for a 30 g/t depressant addition. The 20 g/t depressant addition tests indicated

that the column obtained a higher PGM grade by 106 ppm. Although the mechanical cell obtained a lower grade, the corresponding metal recovery was much greater. However, the higher metal grade obtained the column cell at the 20 g/t and 30 g/t dosages indicated that the column cell was more sensitive to the depressant at higher dosages.

4.6.3 Overview of depressant dosage tests in series A, B and C

The mechanical cell obtained higher metal recoveries than the column cell in all three series of depressant tests. The column cell recovered half the mass that the mechanical cell recovered for a 10 g/t depressant dosage in series A and B. A similar result was found with the 30 g/t depressant dosage tests. The corresponding grade-recovery curves showed that the metal grade of the column cell for a 10 g/t addition of depressant was double that of the base test. The mechanical cell grade-recovery at the same depressant dosage showed no significant change from the base tests. The Cu/Ni results indicated that the column cell was more sensitive to depressant dosage possibly due to the absence of impeller agitation as explained by Whitney and Jan (1996).

The mass recovered by the column cell in series C was higher than that of the mechanical cell for the base tests and 10g/t depressant addition tests. At higher depressant dosages (20g/t and 30g/t) the column cell recovered significantly less mass than the mechanical cell. The corresponding grade-recovery curve showed that mechanical cell obtained a PGM grade that was 44 ppm higher than the column cell. This was contrary to the trend observed in series A and B. However, at higher depressant dosages of 20 g/t and 30 g/t the PGM grade achieved by the column cell was higher than that of the mechanical cell by 106 ppm and 63 ppm respectively. It was assumed that due to the change in ore sample the effect of impeller agitation (or lack of it) was more visible at higher depressant dosages.

Overall the mechanical cell grade recovery was significantly better than the column cell; however, the column cell was more sensitive to the depressant dosage.

4.7 THE EFFECT OF GENTLE CONDITIONING WITH DEPRESSANT .

The depressant dosage tests indicated that impellor agitation could result in the depressant performance being less effective. Previously the recirculation pump was used to condition the depressant prior to the test. The vigorous action of the pump may have, to some degree, simulated the action of the impeller in the mechanical cell. If the pulp was conditioned more gently then it could result in a more effective use of depressant. The effect of conditioning with depressant by means of manual stirring was investigated in series B and C.

4.7.1 The effect of gentle conditioning of the depressant in the E3 column (Series B).

The cumulative mass recovered over time for a 30 g/t depressant is shown in Figure 4.50.

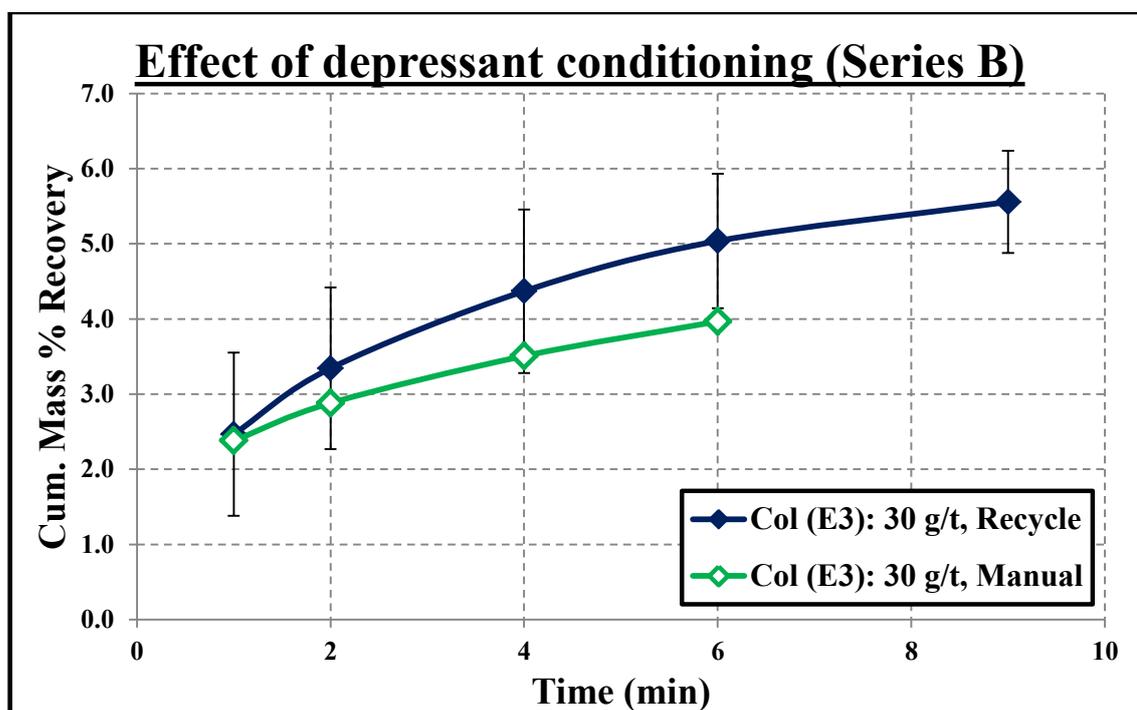


Figure 4.50 Effect of depressant conditioning on the mass recovery of the E3 column

The cumulative mass recovered was lower when the depressant was conditioned manually. The corresponding grade-recovery curves are shown in Figure 4.51 and 4.52.

Figure 4.51 showed that conditioning the depressant by means of the recycle improved the Ni grade-recovery curve. The Cu grade-recovery curve in Figure 4.52, on the other hand, indicated the opposite trend. The Cu/Ni results were conflicting in this case and therefore inconclusive. This was one of the contributing factors that encouraged direct PGM analysis in series C.

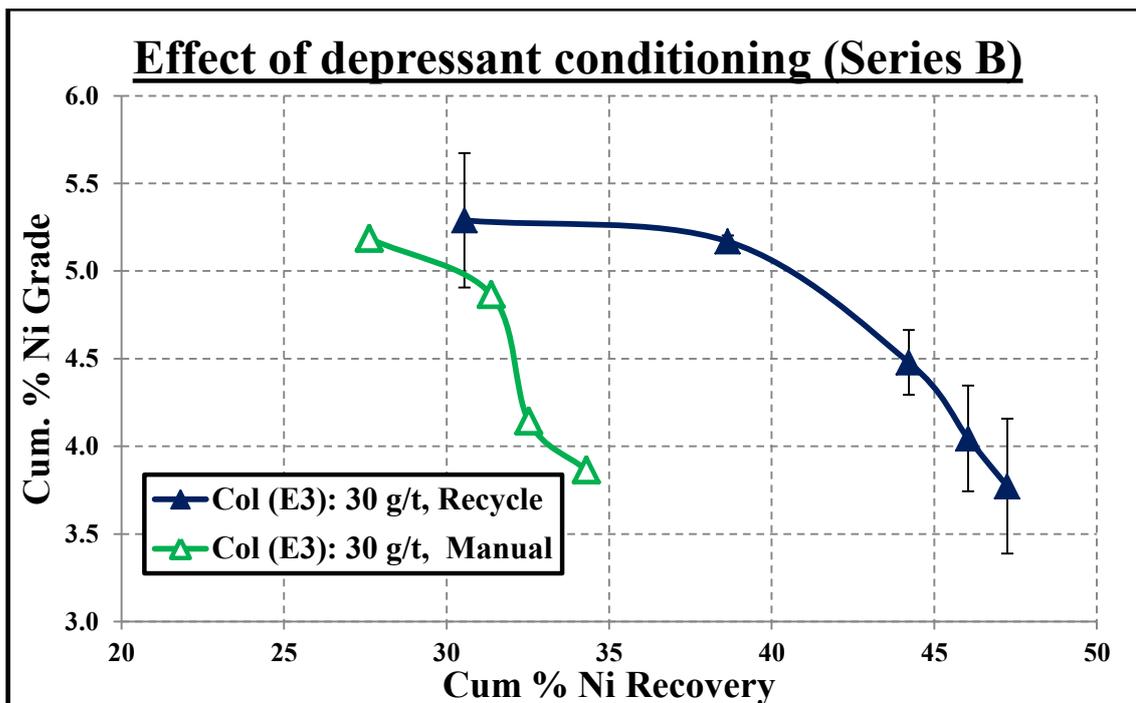


Figure 4.51 Effect of depressant conditioning on the Ni grade-recovery of the E3 column cell

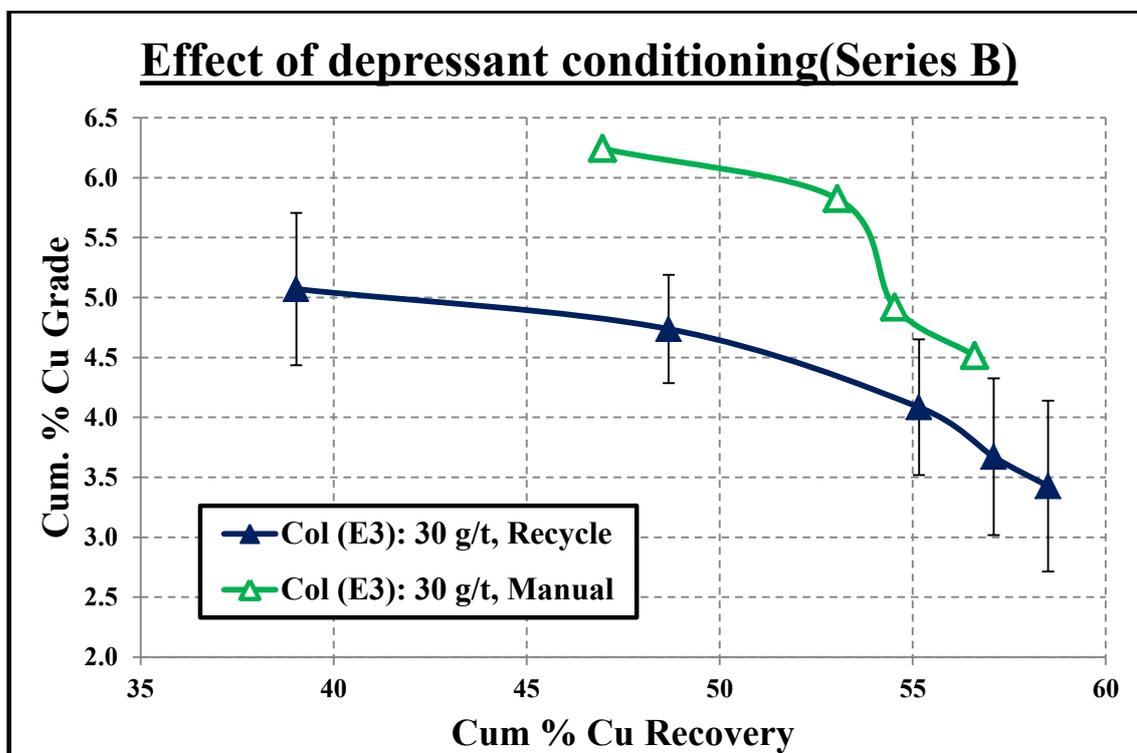


Figure 4.52 Effect of depressant conditioning on the Cu grade-recovery of the E3 column cell

4.7.2 The effect of gentle conditioning of the depressant in the E3 column (Series C).

The mass recovery obtained for both conditioning techniques are shown in Figure 4.53. The tests were performed at a depressant dosage of 10 g/t.

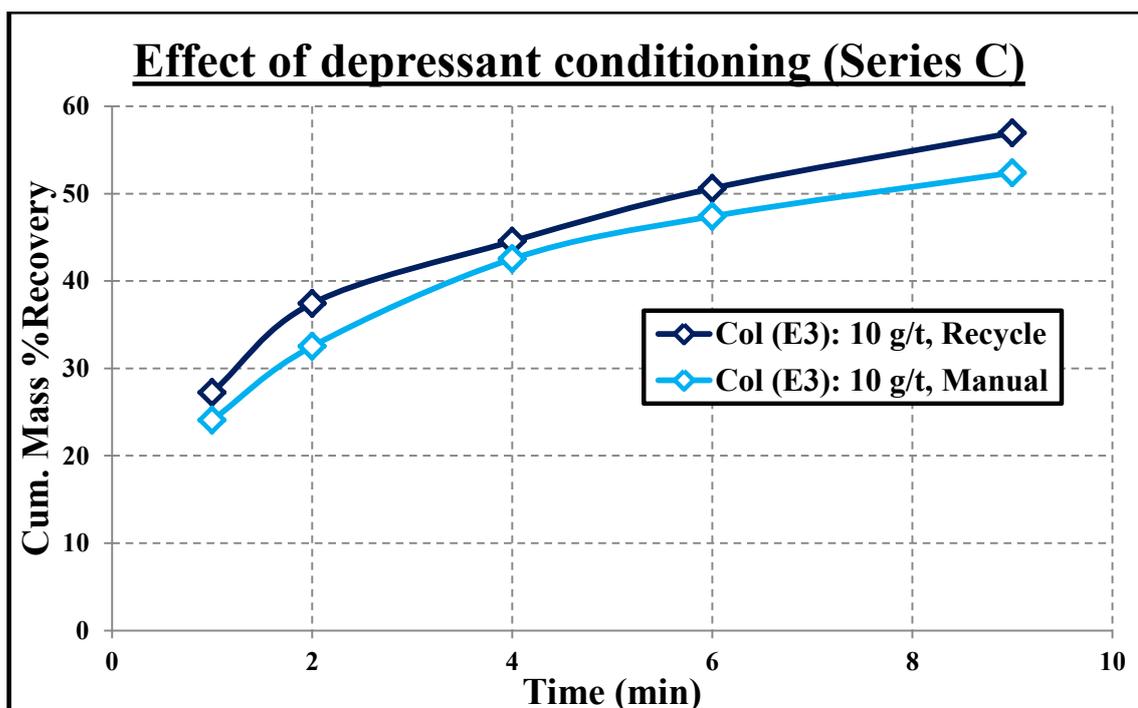


Figure 4.53 Effect of depressant conditioning on the mass recovery of the E3 column

The corresponding PGM grade-recovery curves are shown below in Figure 4.54.

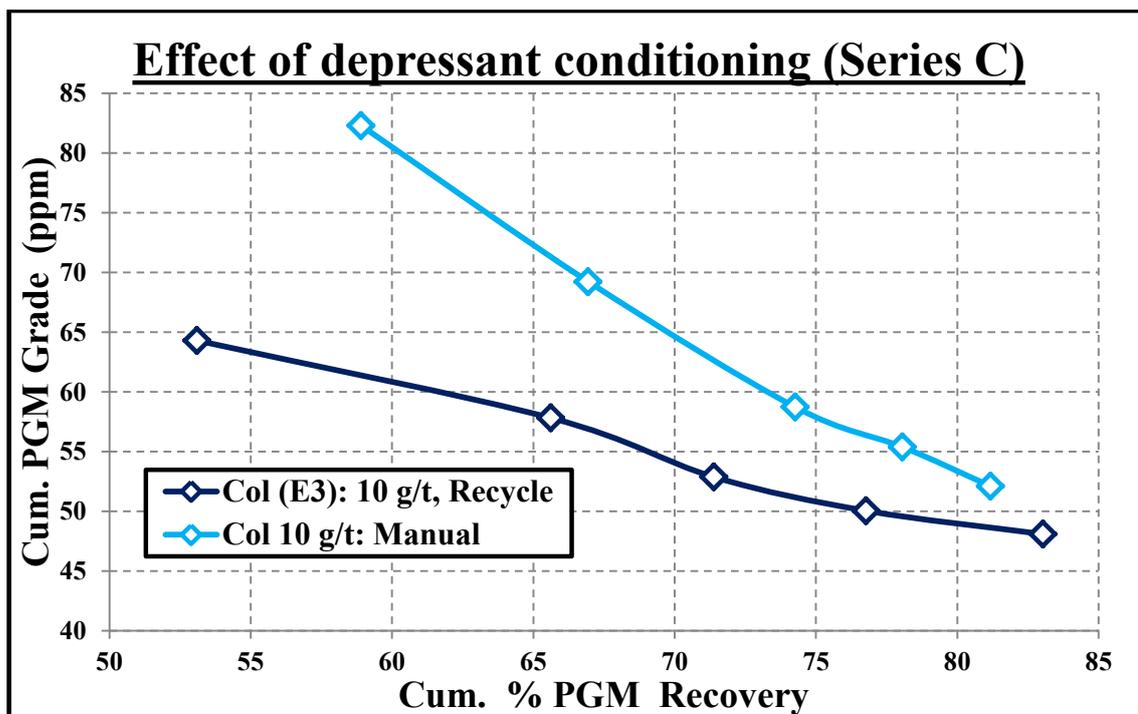


Figure 4.54 Effect of depressant conditioning on the mass recovery of the E3 column

The cumulative mass recovered was lower when manual conditioning was employed. This is in agreement with series B.

The PGM grade-recovery relationship was improved when manual conditioning was employed. The PGM grade obtained when the depressant was conditioned with the recycle was 9ppm lower for a common PGM recovery of 80%. These results concurred with the Cu grade-recovery curve in series B.

4.7.3 Overview of depressant dosage tests in series A, B and C

Series B was inconclusive as to whether the manual conditioning or conditioning by means of the recycle was a better technique. This was due to the grade recovery trends of the Cu and Ni minerals being contradictory. In series C the PGM grade recovery curve confirmed that manual conditioning improved the PGM grade by 9ppm for a common recovery of 80% PGM.

These observations supported the conclusions of the depressant dosage tests. Manual conditioning showed that a gentler means on conditioning can further enhance the action of the depressant in the column cell.

One of the major operational costs in flotation is the amount of depressant required. Column cells have the potential to reduce the amount of depressant needed. It is concluded that the use of low-intensity conditioning of depressant should be investigated further, in order to reduce depressant consumption.

4.8 THE EFFECT OF INCREASING THE COLUMN HEIGHT

The height of the column (with an E3 sparger) was doubled, by inserting a 1,8m central section. The mass of solids was also doubled to maintain the same concentration of solids. The cumulative percentage mass recovered over time (fig 4.54) and PGM grade recovery curves (fig 4.55) appear below.

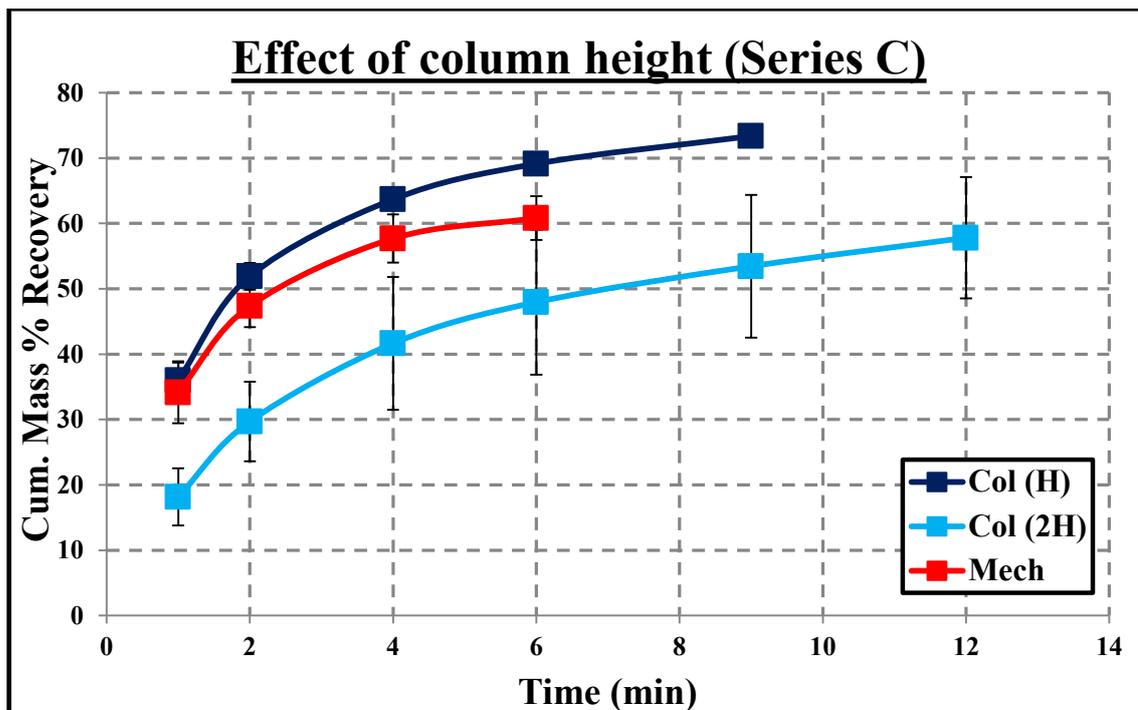


Figure 4.54 Mass recoveries of E3 columns of different heights

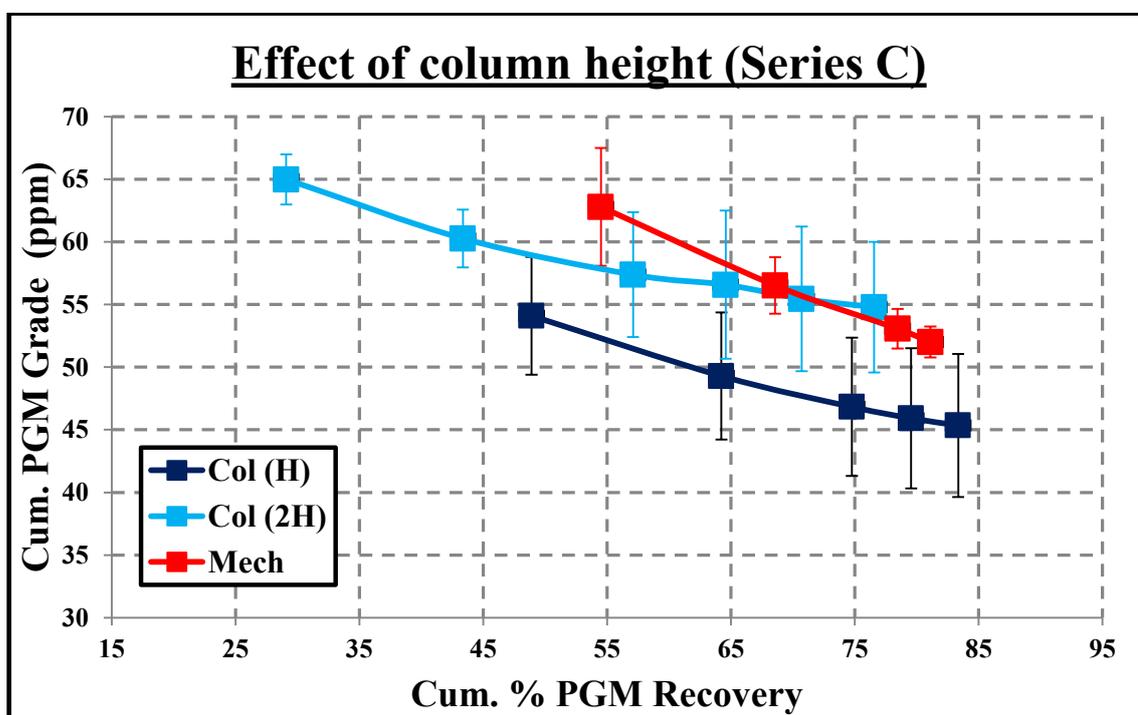


Figure 4.55 Grade- recovery curves of E3 columns of different heights

Figure 4.54 showed that the percentage of mass recovered in the longer column (2H) was less than both the original column and the mechanical cell. The grade-recovery relationship of the column cell (fig.4.55) improved when the column height was increased. The column (2H) grade was higher than the original column (H), by a differential of 8ppm, for a common recovery of

75% PGM. The grade recovery relationship obtained with the longer column (2H) was similar to that of the mechanical cell.

This investigation highlighted the effect of the scale up problems that one may encounter when investigating column cells in a laboratory and trying to implement them in industry. Longer columns are generally used in industry and therefore the results from the 2H column may prove to be more realistic.

CHAPTER 5: RESULTS AND DISCUSSION OF PLANT DATA

The results obtained from the pilot plant tests conducted at Lonmin Platinum are discussed in this section. The performance of pilot column and the pilot test rig (FCTR) were compared.

5.1 FCTR GRADE-RECOVERY RESULTS

The FCTR samples were taken in triplicate over a period of ninety minutes. Each cut contained three samples. The PGM grade and mass recovery relationship for each cut as well as the average are shown in figure 5.1

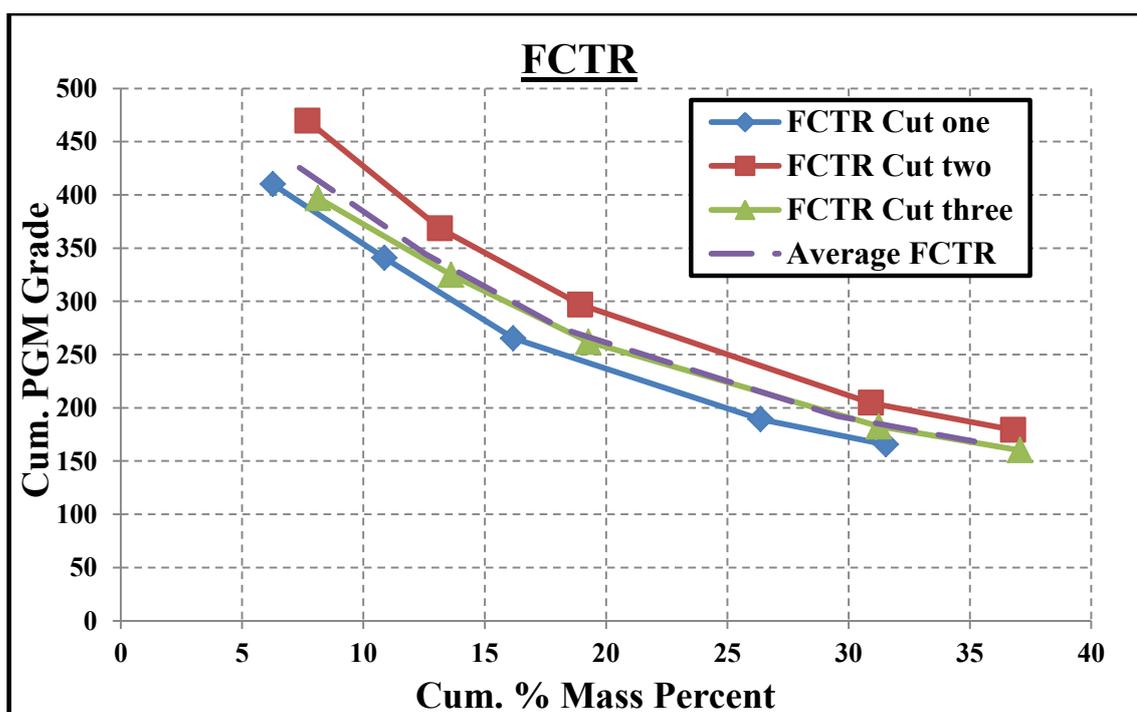


Figure 5.1 Cumulative percent PGM grade versus mass percent for various FCTR cuts

Figure 5.1 showed that the relationship between cumulative PGM grade and cumulative mass was similar for the three sample sets. An average grade of 170ppm was obtained overall mass recovery of 35%.

Figure 5.2 showed the corresponding cumulative PGM recovery versus mass percent. The FCTR obtained a PGM recovery of 73% for the corresponding 35% overall mass recovery.

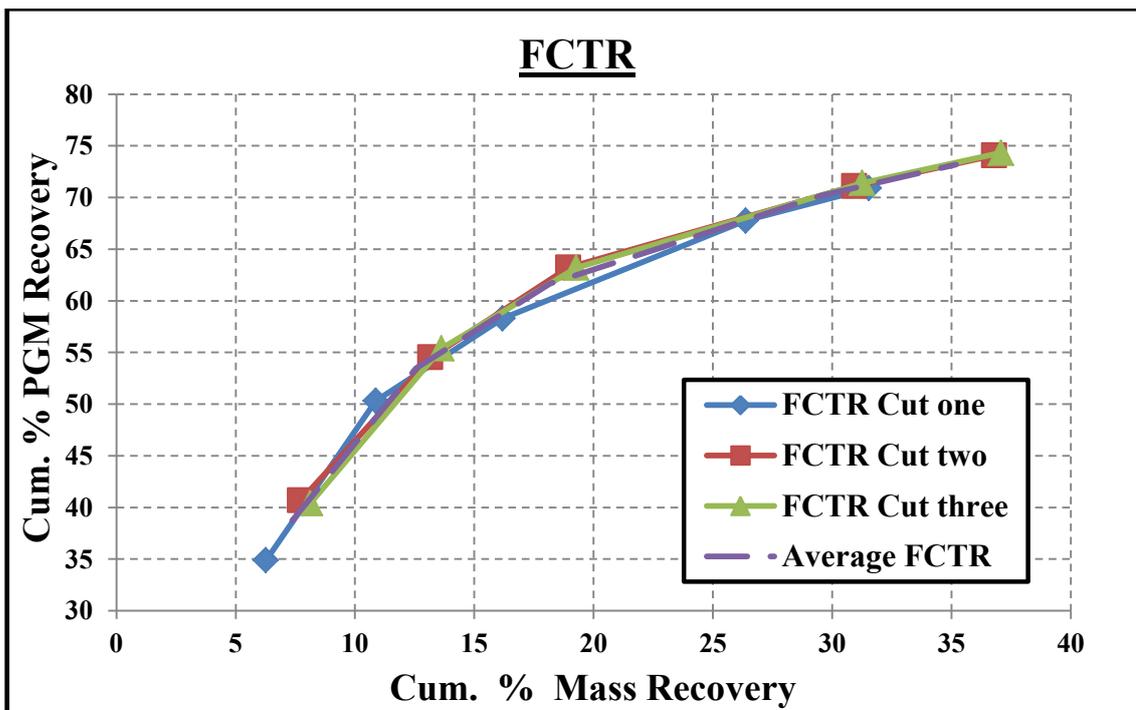


Figure 5.2 Cumulative percent PGM recovery versus mass percent for various FCTR cuts

Figures 5.3 illustrated the cumulative PGM grade versus residence time in the FCTR. The cumulative PGM recovery versus residence time is shown in Figure 5.4.

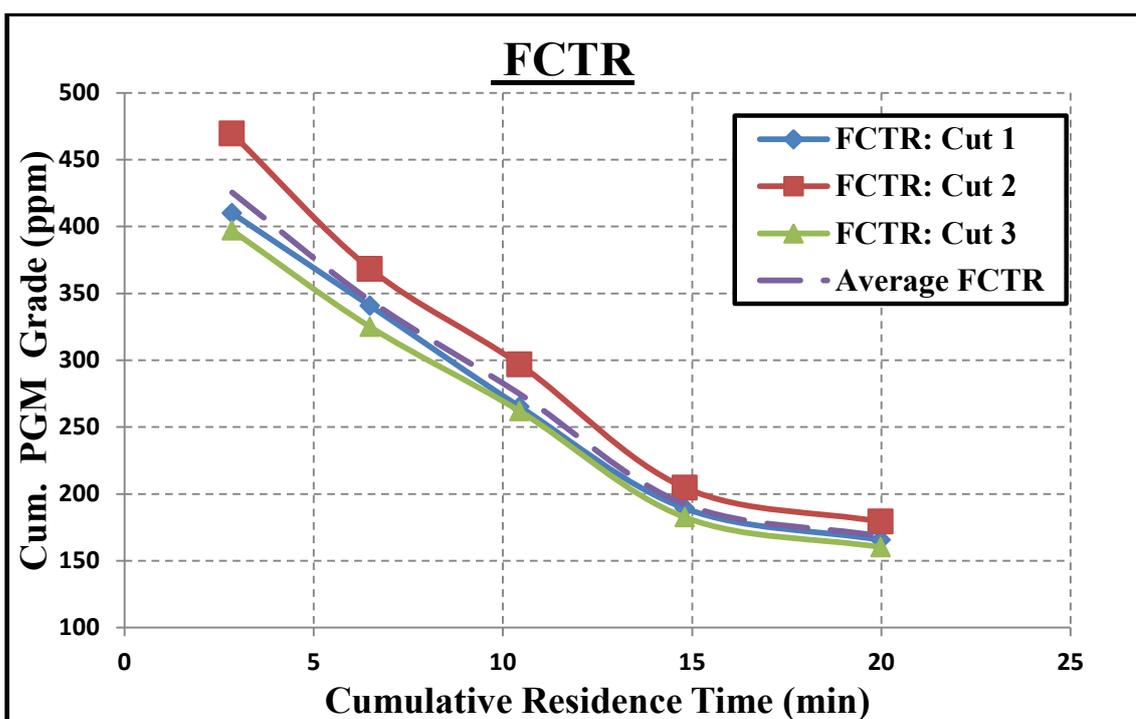


Figure 5.3 Cumulative PGM grade versus residence time for various FCTR cuts

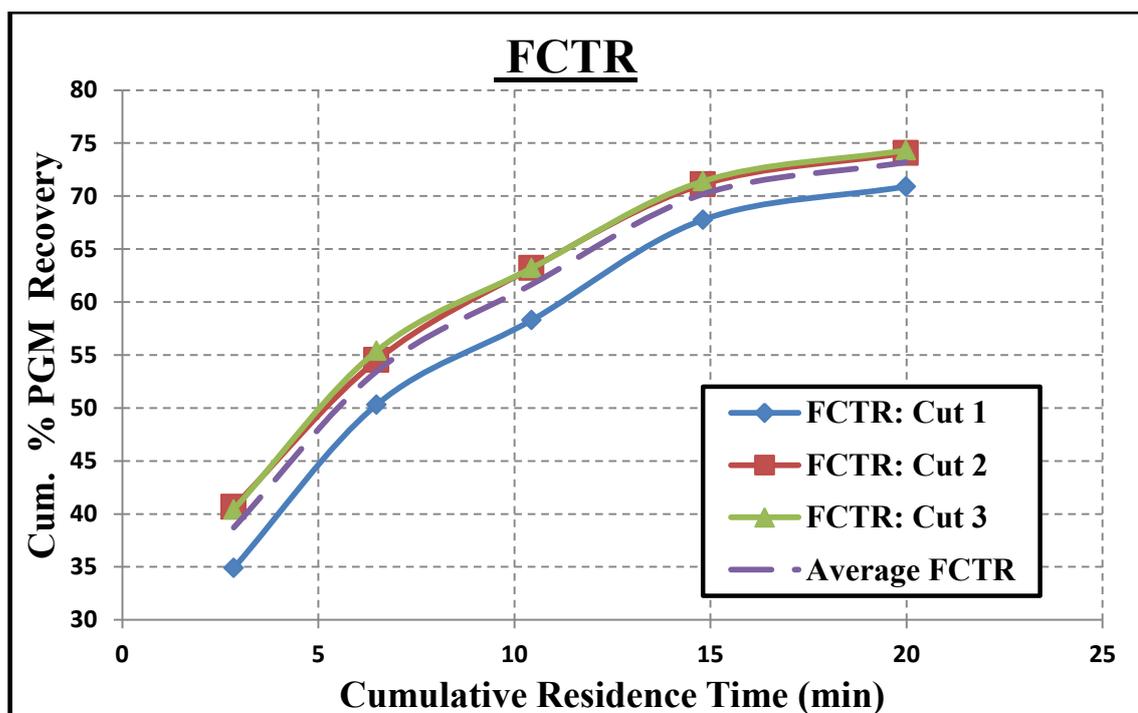


Figure 5.4 Cumulative % PGM recovery versus residence time for various FCTR cuts

As the residence time increased the cumulative % PGM recovery increased and the cumulative grade decreased. The final PGM grade of 170ppm for a corresponding PGM recovery of 73 % was obtained in 20 minutes.

5.2 THE PILOT CLEANER COLUMN

During the laboratory investigations it was ensured that the column cell and mechanical cell were of equal volumes and operated at the same superficial velocities for appropriate comparison. These considerations could not be met on the plant site. The FCTR consisted of 6 mechanical cells of 150L volume and the pilot column was a single 378L unit. Comparisons were made between the two units on the basis of the cumulative residence time in the FCTR and a variety of tests at different residence times in the pilot column.

Pilot column tests operated at different feed flowrates and froth depths. The three different feed flowrates were 9L/min, 25L/min and 59L/min. The different froth depths tested at each flowrate were 18.5cm, 37.0cm and 55.5cm.

Figure 5.5 showed the effect of varying the feed flowrate and froth depth on the PGM grade as a function of the percentage mass recovered. The percentage mass increased and PGM grade decreased as the froth depth decreased for a particular flowrate.

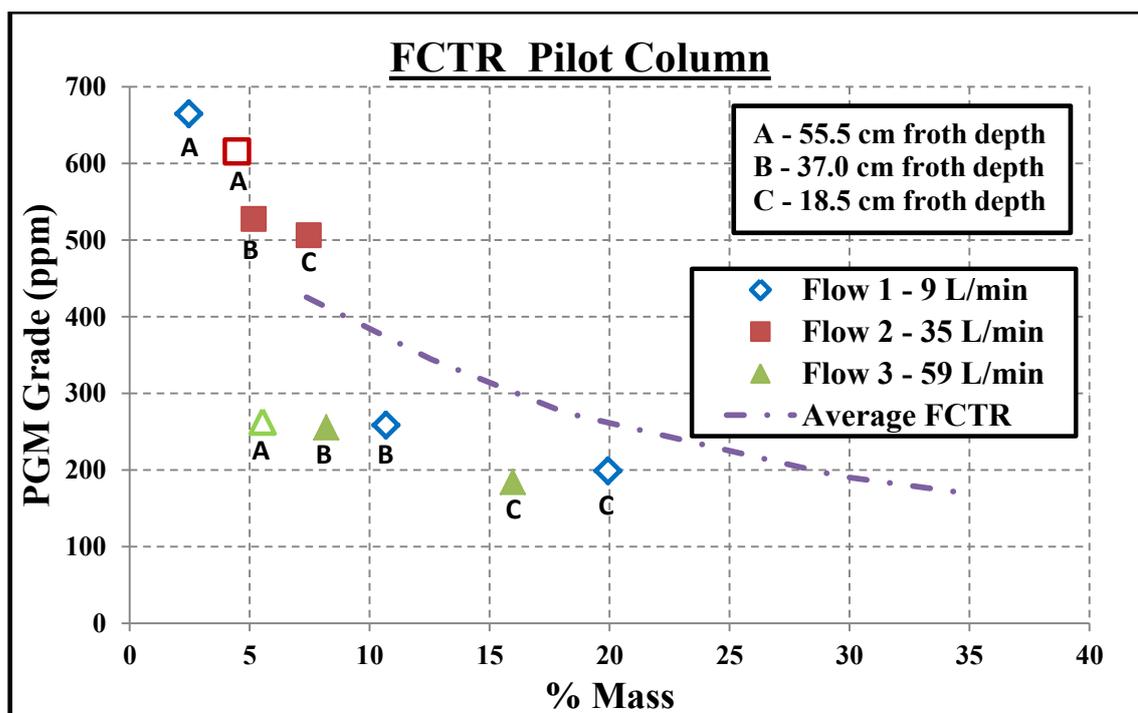


Figure 5.5 % PGM grade versus Mass recovered in FCTR (cumulative) and pilot column

The effect of varying the flowrate and froth depth on the percentage of PGM recovered as a function of the percentage mass was demonstrated in Figure 5.6.

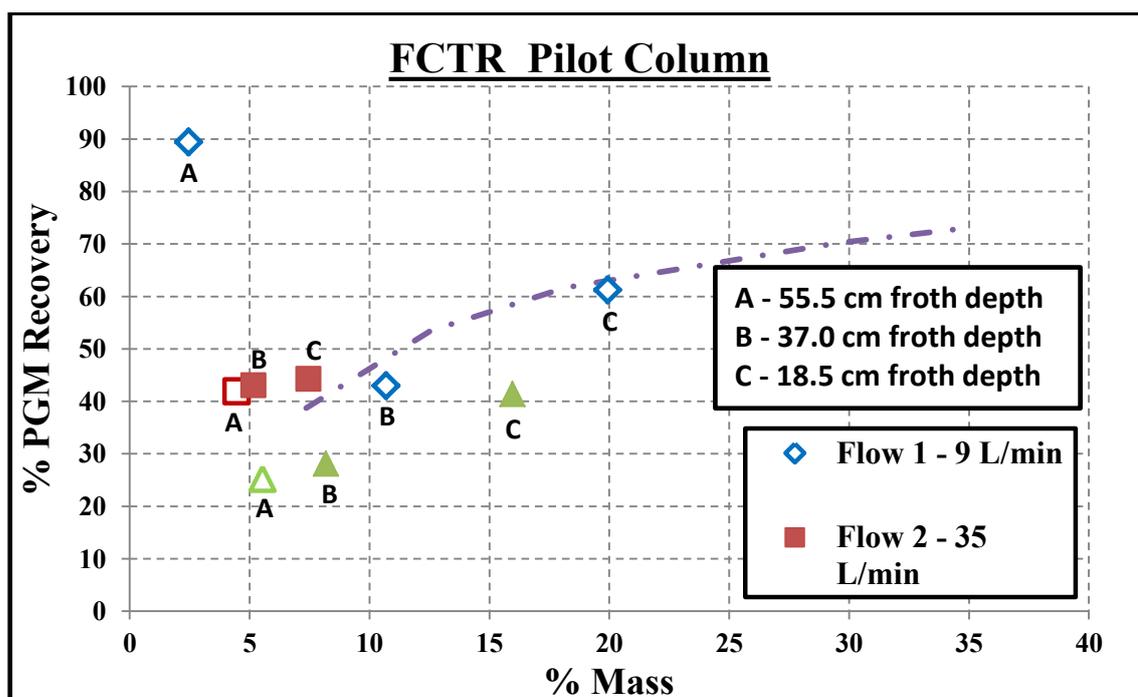


Figure 5.6 % PGM recovered versus mass percent in FCTR (cumulative) and pilot column

Figure 5.6 showed that as the froth depth decreased, the percentage mass recovered increased and so too did the percentage PGMs recovered for each flowrate. All flow rates except for Flow 1 (9L/min) followed this trend. A closer look at the data for Flow 1 revealed why this occurred.

Certain column tests showed inconsistencies between the feed grade (sample) and the recalculated head grade. The ratio of feed grade to calculated head is shown in Table 5.1. Ideally, these ratios should be 1.

Table 5.1 Grade ratio: Variances between and calculated head grade and feed grade

<i>Flow (L/min)</i>	Froth Depth (cm)		
	55.5	37	18.5
9	0.39	1.89	1.41
35	0.68	1.07	1.23
59	0.76	0.82	1.13

The three tests that operated with a 55.5cm froth depth and the three tests that utilised a flowrate of 9L/min, showed significant variations between the recalculated and feed head grades.

These deviations could have occurred for a number of different reasons. One of them could be that the system may not have been at steady state when the samples were taken due to the limited time available for the pilot plant tests. A visible change in the appearance of the froth was noted when the 9L/min flowrate was investigated. This was possibly due to alterations in the upstream plant processes. The data captured at the 55.5cm froth depth and the 9L/min flowrate were therefore shown with hollow markers on the graphs.

The elimination of tests at feed flowrate of 9L/min resulted in one disadvantage: the range of residence times tested in the column cell was narrower. Figure 5.7 and 5.8 showed the effect of varying the flowrate on the grade and recovery as a function of residence time. The different froth depths are included for easy identification.

The lowest feed flowrate in the column corresponded to the longest residence time. Laboratory tests on the E2 column showed that the column cell may recover mass at a slower rate than the mechanical cell therefore the wide range of flowrates tested in the pilot column (9L/min, 35 L/min and 59L/min) would result in a wide range of residence times (7 minutes, 16 minutes and 41 minutes). It was expected the lowest flowrate would yield the best grade and recovery in the

column and therefore may compare favourably with the FCTR. Since the data of the lowest flow was to be ignored this comparison was not possible.

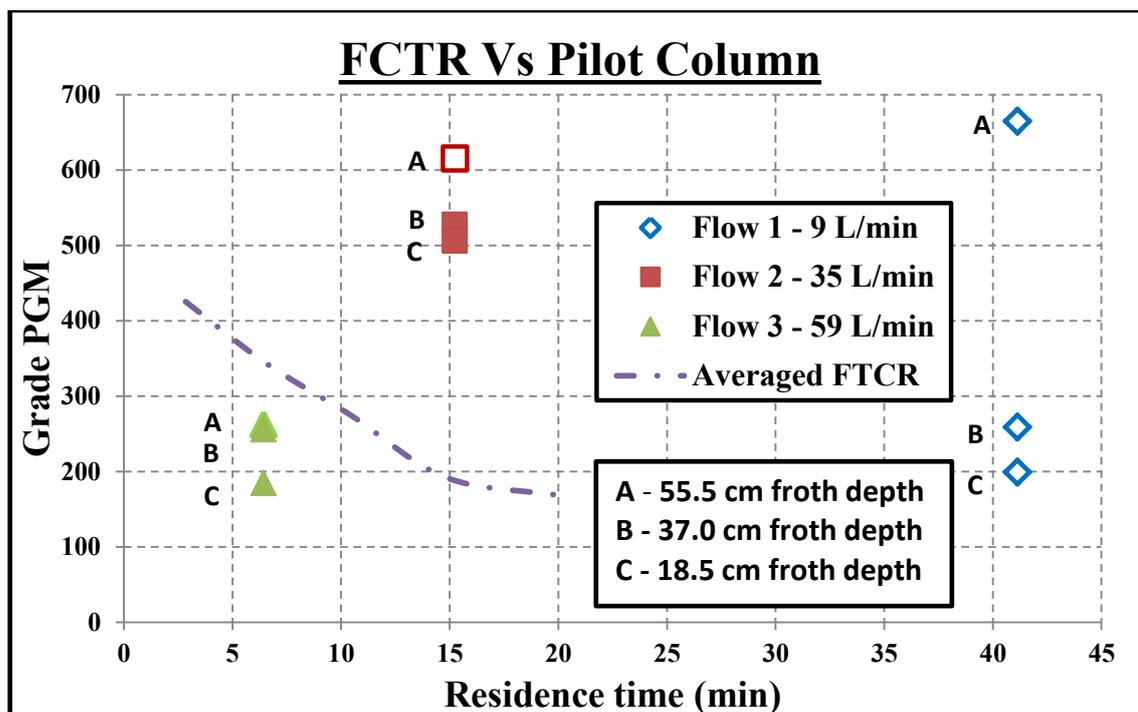


Figure 5.7 PGM Grade versus residence time in FCTR (cumulative) and pilot column

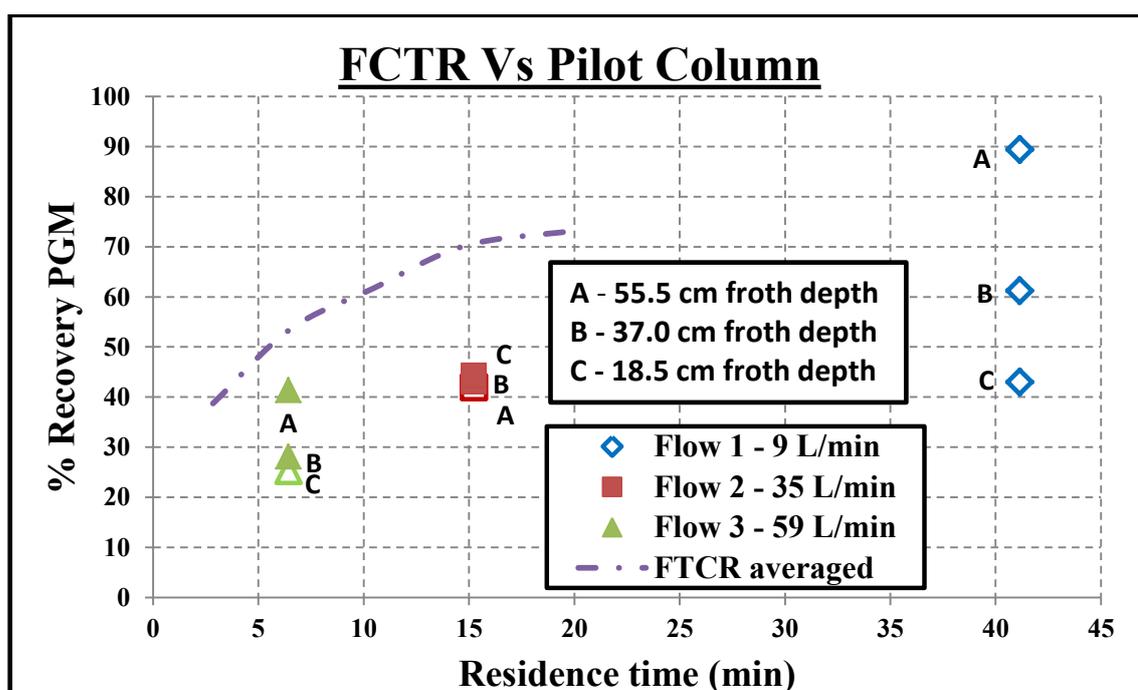


Figure 5.8 % PGM recovered versus residence time in FCTR (cumulative) and pilot column

5.3 COMAPRISON BETWEEN THE FCTR AND THE PILOT COLUMN

Figures 5.9 and 5.10 compared the performance of the FCTR and the pilot column cell. Only column tests with a reasonable grade ratio appeared on these plots. The grade and recovery for the FCTR was expressed cumulatively.

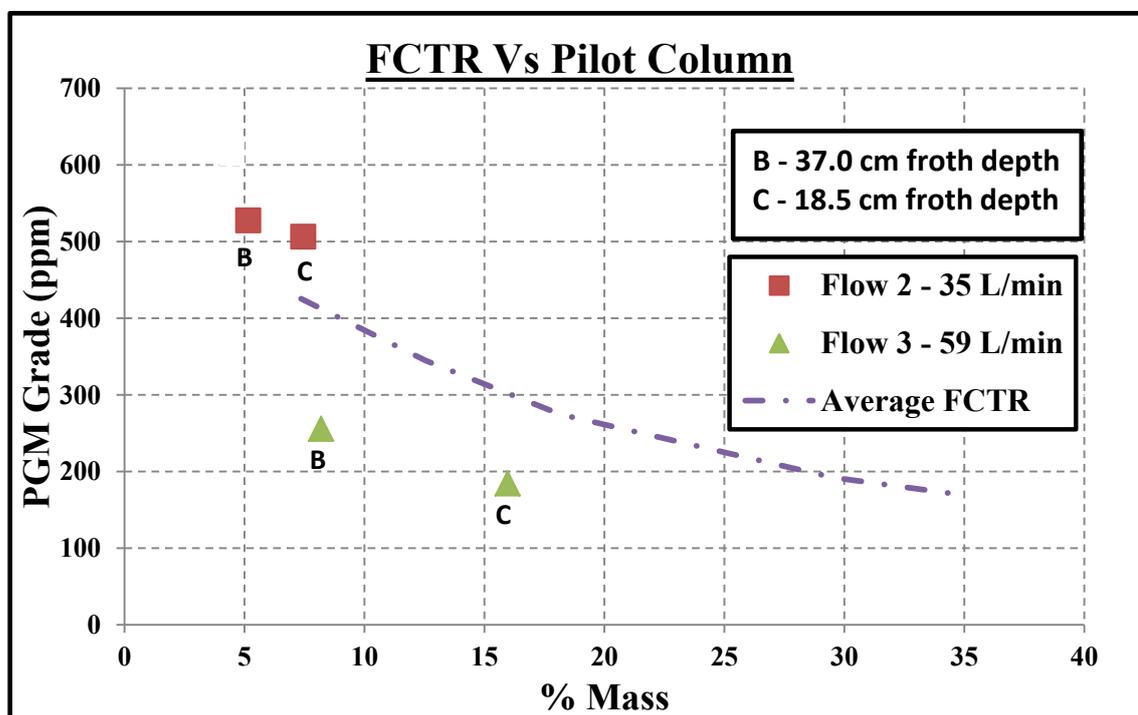


Figure 5.9 PGM grade versus mass percent in FCTR (cumulative) and pilot column

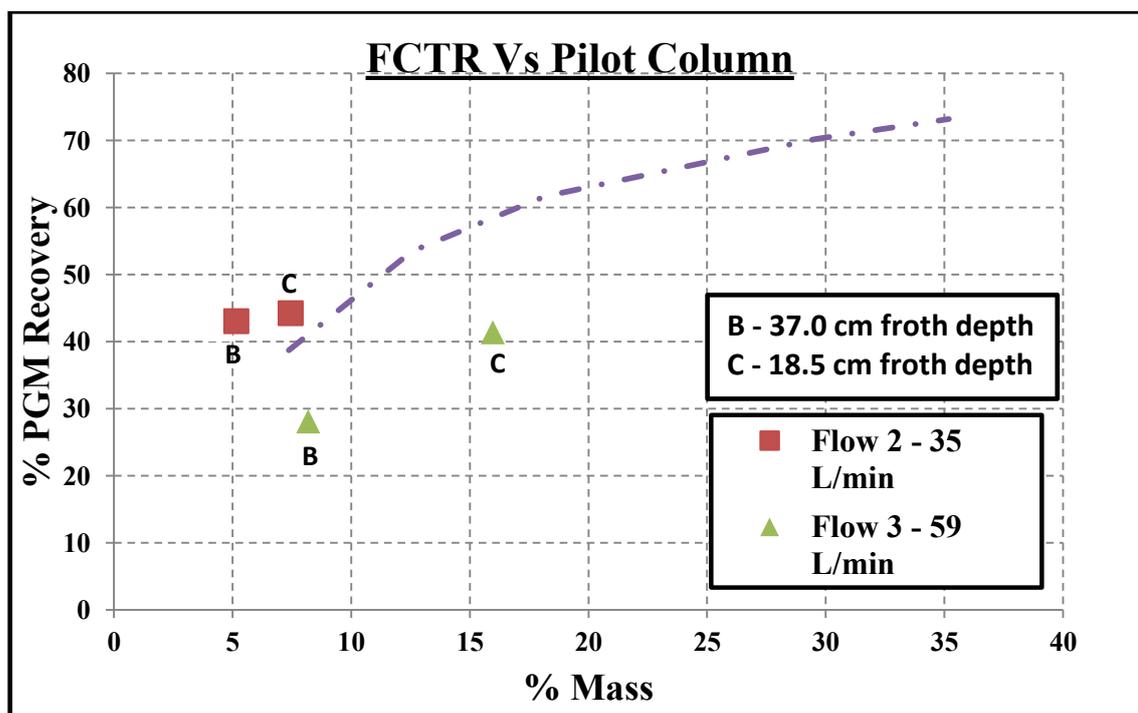


Figure 5.10 % PGM recovered versus mass percent in FCTR (cumulative) and pilot column

It should be noted that at Flow 2 (35L/min), the average residence time in the column (16 minutes) was similar to the total residence time in the FCTR (20 minutes at the end point, where recovery is maximum and the grade is low). Figure 5.9 and 5.10 showed that FCTR achieved a final cumulative PGM grade and recovery of 73% and 180ppm respectively.

When the pilot column was operated at a flowrate of 59L/min, it obtained grade of 255ppm and 184ppm with a 37.0cm and 18.5cm froth depth respectively. The PGM grade and recovery were both lower than achieved in the FCTR.

When the pilot column was operated at a flowrate of 35 L/min it obtained grade of 527ppm and 505ppm with a 37.0cm and 18.5cm froth depth respectively. The PGM grade was higher than that of the FCTR. The corresponding percentage PGM recovery in the pilot column was 43% and 44%. The PGM recovery was again significantly lower than the FCTR

If the 59L/min and 35L/min tests are compared to each other a significant improvement in the PGM grade and recovery was noted as the flowrate decreased. However, this was at the expense of a significant loss in recovery. For a froth depth of 37.0cm the PGM grade improved by 320 ppm and the recovery by 15% when the residence time was increased from 7 minutes to 15 minutes. An additional increase in the residence time could further improve the column performance. This would require further investigation.

It was also important to bear in mind that these tests were among the first tests that were conducted on the pilot column unit as it was in the process of being commissioned. Hence, the column was not optimised at that stage. The FCTR was operated at a high froth depth (average of 30cm) relative to its height (70cm). The feed to final cleaner, which is generally highly mineralized, was diverted to the FCTR, therefore a deep froth was required for stable operation. Lastly, the level control on the pilot column was based on the change in the column pressure and the pressure gauge used was of low sensitivity.

The results showed that the FCTR obtained a much better PGM recovery and therefore performed better than the pilot column. However, the pilot column should be further tested at lower flowrates.

Although the laboratory test campaign focussed on other parameters such as depressant dosage and conditioning technique, testing these parameters were not permitted on site as depressant addition occurred further upstream and could affect the other operating processes. Hence, the parameters investigated were restricted to froth depth and flowrate.

CHAPTER 6: CONCLUSIONS

- The overall objective of the investigations was to compare a column and mechanical cell of equal volume and equal superficial air velocities, but significantly different heights. It was proposed that the increased bubble path length in the column cell would result in a better grade recovery relationship. This proposition was found to be untrue for the 1,8m column cell.
- A better grade recovery relationship was obtained in the mechanical cell for all parameters tested, with the exception of column height.
- Standard flotation tests were performed on the mechanical and column cell without any reagent addition in the cleaner flotation. The investigations revealed that the PGM grade in the mechanical cell was higher than the column cell, by a differential of 6.5ppm, for a common recovery of 80% PGM.
- The use of recirculation pump improved the grade recovery relationship in the column cell marginally.
- The mechanical cell obtained a higher PGM recovery for all depressant dosages (10g/t, 20g/t and 30g/t).
- Investigations into the effect of depressant dosage revealed that the action of the depressant was enhanced in the column cell due to the absence of impeller agitation. This trend was observed in the Cu/Ni analysis and confirmed in the PGM analysis at higher depressant dosages.
- Two different depressant conditioning techniques were investigated, namely gentle conditioning prior, and conditioning with the aid of a recirculation pump. When gentle conditioning of the depressant was employed, the PGM grade was higher, by a differential of 9ppm, for a common PGM recovery of 80%. These tests confirmed that agitating the pulp curtailed the effect of the depressant.
- The effect of doubling the column height was investigated. The tests showed that the PGM grade of the longer column was higher, by a differential of 8ppm, for a common PGM recovery of 75% in both columns. The grade recovery relationship in the longer column was the similar to the mechanical cell.

- Investigations at a flotation plant at Lonmin Platinum showed that a pilot test rig of mechanical cells (FCTR) achieved a significantly better recovery than a pilot column. However, by decreasing the column feed flowrate and thereby increasing the residence time a substantial improvement in the PGM grade and recovery obtained by the column was noted.

CHAPTER 7: RECOMMENDATIONS

- The grade recovery relationship obtained using the longer column (3,6m) was similar to that obtained with the mechanical cell. It would therefore be interesting to investigate the effect of depressant dosage in the column (2H) as compared to the mechanical cell.
- Methods of gentle conditioning of depressant should be investigated on a larger scale, to reduce the consumption of depressant. This would be of particular interest when column cells are used for cleaner flotation.
- The inclusion of froth washing in the column cell tests may improve the columns performance in comparison the mechanical cell. This aspect could not be investigated in the equipment used for this investigation.
- The surface superficial air velocities were equivalent in this investigation. A more accurate analysis of the effect of the bubble path length can be made if the bubble surface area flux in the mechanical and column cells were equal. The comparison is complicated by the expansion of bubbles as they rise and the possibility of coalescence.
- The investigations conducted at the Lonmin pilot plant showed that further tests on the pilot column at reduced feed flowrates may improve the column performance and therefore should be investigated further.

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APPENDICES

Appendix A1: Tables of laboratory results for series A

A1.1: MECHANICAL CELL

The following tables provide the laboratory data for tests performed on the mechanical cell in series A. It is divided into 2 sub-sections namely the base tests and depressant dosage tests.

A1.1.1: Base tests

Table A1.1 Laboratory data for base test 1 in mechanical cell

<i>TEST 1</i>	Cum. % mass	Cum. % Rec		Cum. % Grade	
		Ni	Cu	Cu%	Ni %
Conc 1	36.84	59.0	60.1	0.69	0.97
Conc2	49.08	70.1	67.9	0.58	0.86
Conc4	60.40	78.2	74.9	0.52	0.78
Conc6	64.68	85.6	92.1	0.60	0.80
Tails	100.00	100.0	100.0	0.42	0.60

Table A1.2 Laboratory data for base test 2 in mechanical cell

<i>TEST 2</i>	Cum. % mass	Cum. % Rec		Cum. % Grade	
		Ni	Cu	Cu%	Ni %
Conc 1	38.24	60.6	56.5	0.59	0.96
Conc2	48.78	70.5	68.9	0.57	0.88
Conc4	61.15	78.9	76.9	0.50	0.78
Conc6	65.35	85.7	91.9	0.56	0.80
Tails	100.00	100.0	100.0	0.40	0.61

Table A1.3 Averaged data for base test in mechanical cell

<i>Average</i>	Cum. % mass	Cum. % Rec		Cum. % Grade	
		Ni	Cu	Cu%	Ni %
Conc 1	37.54	59.8	58.29	0.64	0.96
Conc2	48.93	70.31	68.39	0.57	0.87
Conc4	60.78	78.51	75.92	0.51	0.78
Conc6	65.02	85.64	91.98	0.58	0.80
Tails	100.00	100.00	100.00	0.41	0.61
Variance(%)				9.30	10.4

A1.1.2: Depressant Tests

In series A the depressant tests were performed for 10 g/t and 30 g/t dosages as shown below.

Table A1.4 Laboratory data for depressant test 1 (10 g/t) in mechanical cell

<i>TEST 1</i>	<i>Cum. % mass</i>	<i>Cum. % Rec</i>		<i>Cum. % Grade</i>	
		<i>Ni</i>	<i>Cu</i>	<i>Cu%</i>	<i>Ni %</i>
Conc 1	38.23	62.1	69.5	0.73	1.16
Conc2	45.84	68.3	75.8	0.67	1.06
Conc4	53.29	74.3	81.0	0.61	0.99
Conc6	57.12	77.3	83.9	0.59	0.96
Tails	100.00	100.0	100.0	0.40	0.71

Table A1.5 Laboratory data for depressant test 2 (10 g/t) in mechanical cell

<i>TEST 2</i>	<i>Cum. % mass</i>	<i>Cum. % Rec</i>		<i>Cum. % Grade</i>	
		<i>Ni</i>	<i>Cu</i>	<i>Cu%</i>	<i>Ni %</i>
Conc 1	27.38	57.0	69.0	0.72	0.89
Conc2	35.89	71.9	81.5	0.65	0.86
Conc4	41.06	78.6	86.8	0.60	0.82
Conc6	45.91	79.4	87.4	0.54	0.74
Tails	100.00	100.0	100.0	0.30	0.43

Table A1.6 Averaged data for 10g/t depressant dosage test in mechanical cell

<i>Average</i>	<i>Cum. % mass</i>	<i>Cum. % Rec</i>		<i>Cum. % Grade</i>	
		<i>Ni</i>	<i>Cu</i>	<i>Cu%</i>	<i>Ni %</i>
Conc 1	32.81	59.58	69.23	0.73	1.02
Conc2	40.87	70.12	78.69	0.66	0.96
Conc4	47.18	76.46	83.93	0.61	0.91
Conc6	51.51	78.33	85.69	0.57	0.85
Tails	100.00	100.00	100.00	0.35	0.57
Variance(%)				6.1	4.7

Table A1.7 Laboratory data for depressant test 3 (30 g/t) in mechanical cell

<i>TEST 3</i>	Cum. % mass	Cum. % Rec		Cum. % Grade	
		Ni	Cu	Cu%	Ni %
Conc 1	6.22	47.1	65.5	3.95	3.95
Conc2	7.71	55.5	73.7	3.58	3.76
Conc4	8.05	60.6	77.3	3.60	3.93
Conc6	8.05	60.6	77.3	3.60	3.93
Tails	100.00	100.0	100.0	0.37	0.52

Table A1.8 Laboratory data for depressant test 4 (30 g/t) in mechanical cell

<i>TEST 4</i>	Cum. % mass	Cum. % Rec		Cum. % Grade	
		Ni	Cu	Cu%	Ni %
Conc 1	4.10	32.9	48.2	4.46	4.17
Conc2	5.62	46.3	62.4	4.21	4.29
Conc4	7.18	56.0	72.1	3.81	4.06
Conc6	8.24	59.8	75.6	3.48	3.78
Tails	100.00	100.0	100.0	0.38	0.52

Table A1.9 Averaged data for 30g/t depressant dosage test in mechanical cell

<i>Average</i>	Cum. % mass	Cum. % Rec		Cum. % Grade	
		Ni	Cu	Cu%	Ni %
Conc 1	5.16	39.97	56.88	4.20	4.06
Conc2	6.66	50.91	68.03	3.90	4.03
Conc4	7.62	58.27	74.67	3.70	3.99
Conc6	8.14	60.18	76.42	3.54	3.85
Tails	100.00	100.00	100.00	0.38	0.52
Variance(%)				1.2	4.0

A1.2 E2 COLUMN CELL

The following tables provide the laboratory data for tests performed on the E2 Column cell in series A. It is divided into 3 sub-sections namely the base tests, the recycle tests and the depressant dosage tests.

A1.2.1 Base case tests

Table A1.10 Laboratory data for base test 1 in E2 column cell

<i>TEST 1</i>	Cum. % mass	Cum. % Rec		Cum. % Grade	
		Ni	Cu	Cu%	Ni %
Conc 1	5.08	5.1	19.5	1.45	0.45
Conc2	10.49	12.5	37.9	1.36	0.53
Conc4	19.98	19.1	48.6	0.92	0.42
Conc6	38.94	25.2	53.9	0.52	0.28
Conc 9	63.76	59.6	74.6	0.44	0.41
Tails	100.00	100.0	100.0	0.38	0.44

Table A1.11 Laboratory data for base test 2 in E2 column cell

<i>TEST 2</i>	Cum. % mass	Cum. % Rec		Cum. % Grade	
		Ni	Cu	Cu%	Ni %
Conc 1	5.04	4.6	25.9	1.99	0.59
Conc2	10.45	5.9	28.5	1.06	0.36
Conc4	22.05	15.6	42.6	0.75	0.46
Conc6	38.85	32.8	57.9	0.58	0.54
Conc 9	65.77	71.4	80.0	0.47	0.70
Tails	100.00	100.0	100.0	0.39	0.64

Table A1.12 Averaged data for base test in E2 column cell

<i>Average</i>	Cum. % mass	Cum. % Rec		Cum. % Grade	
		Ni	Cu	Cu%	Ni %
Conc 1	5.06	4.9	22.7	1.72	0.52
Conc2	10.47	9.2	33.2	1.21	0.44
Conc4	21.01	17.3	45.6	0.83	0.44
Conc6	38.89	29.0	55.9	0.55	0.41
Conc 9	64.77	65.5	77.3	0.46	0.56
Tails	100.00	100.0	100.0	0.38	0.54
Variance(%)	2.03	69.59	14.70	2.5	0.01

A1.2.2 Recycle tests

Table A1.13 Laboratory data for recycle test 1 in E2 column cell

<i>TEST 1</i>	Cum. % mass	Cum. % Rec		Cum. % Grade	
		Ni	Cu	Cu%	Ni %
Conc 1	19.88	31.7	47.7	0.84	0.86
Conc2	23.26	36.8	52.6	0.79	0.86
Conc4	32.66	50.7	63.5	0.68	0.84
Conc6	43.05	55.6	66.9	0.55	0.70
Conc 9	63.25	67.1	74.4	0.41	0.57
Tails	100.00	100.0	100.0	0.35	0.54

Table A1.14 Laboratory data for recycle test 2 in E2 column cell

<i>TEST 2</i>	Cum. % mass	Cum. % Rec		Cum. % Grade	
		Ni	Cu	Cu%	Ni %
Conc 1	27.15	37.3	46.9	0.74	0.86
Conc2	42.09	55.2	62.4	0.64	0.82
Conc4	55.08	68.3	73.9	0.58	0.78
Conc6	61.74	73.9	79.2	0.55	0.75
Conc 9	64.10	74.4	80.3	0.54	0.73
Tails	100.00	100.0	100.0	0.43	0.63

Table A1.15 Averaged data for recycle test in E2 column cell

<i>Average</i>	Cum. % mass	Cum. % Rec		Cum. % Grade	
		Ni	Cu	Cu%	Ni %
Conc 1	23.51	34.5	47.3	0.79	0.86
Conc2	32.67	46.0	57.5	0.72	0.84
Conc4	43.87	59.5	68.7	0.63	0.81
Conc6	52.40	64.7	73.1	0.55	0.72
Conc 9	63.68	70.7	77.3	0.47	0.65
Tails	100.00	100.0	100.0	0.39	0.58
Variance(%)	0.36	26.95	17.11	0.01	0.01

A1.2.3: Depressant Test

The depressant tests were performed for 30 g/t and 10 g/t depressant dosages in series A

Table A1.16 Laboratory data for depressant test 1 (30 g/t) in E2 column cell

<i>TEST 1</i>	Cum. % mass	Cum. % Rec		Cum. % Grade	
		Ni	Cu	Cu%	Ni %
Conc 1	0.90	16.9	28.5	12.39	10.77
Conc2	1.24	26.4	39.3	12.41	12.16
Conc4	2.16	37.8	50.6	9.12	9.96
Conc6	2.87	43.7	56.9	7.71	8.66
Conc 9	4.20	50.6	63.4	5.88	6.86
Tails	100.00	100.0	100.0	0.39	0.57

Table A1.17 Laboratory data for depressant test 2 (10 g/t) in E2 column cell

<i>TEST 2</i>	Cum. % mass	Cum. % Rec		Cum. % Grade	
		Ni	Cu	Cu%	Ni %
Conc 1	2.16	5.0	19.0	2.41	0.91
Conc2	3.55	9.0	28.8	2.22	1.00
Conc4	7.89	20.8	44.7	1.55	1.03
Conc6	11.41	29.9	53.6	1.29	1.03
Conc 9	17.50	46.8	67.0	1.05	1.05
Tails	100.00	100.0	100.0	0.27	0.39

Appendix A2: Tables Of Laboratory results for series B

A2.1: E3 COLUMN CELL

The following tables provide the laboratory data for tests performed on the E3 Column cell in series B. It is divided into 4 sub-sections namely the base tests, the recycle tests , the depressant dosage tests and the depressant conditioning tests.

A2.3.1 Base case test

Table A2.1 Laboratory data for base test 1 in E3 column cell

<i>TEST 1</i>	Cum. % mass	Cum. % Rec		Cum. % Grade	
		Ni	Cu	Cu%	Ni %
Conc 1	32.10	32.0	44.4	0.50	0.57
Conc2	45.87	44.0	53.5	0.42	0.55
Conc4	56.70	51.6	60.4	0.39	0.52
Conc6	61.94	56.9	63.6	0.37	0.53
Conc 9	65.24	60.5	65.8	0.37	0.53
Conc 14	67.15	63.0	67.3	0.36	0.54
Tails	100.00	100.0	100.0	0.36	0.57

Table A2.2 Laboratory data for base test 2 in E3 column cell

<i>TEST 2</i>	Cum. % mass	Cum. % Rec		Cum. % Grade	
		Ni	Cu	Cu%	Ni %
Conc 1	19.30	22.9	41.0	0.68	0.53
Conc2	38.23	36.4	57.5	0.48	0.42
Conc4	55.79	54.0	69.2	0.40	0.43
Conc6	64.80	63.0	75.2	0.37	0.43
Conc 9	70.69	72.2	80.3	0.36	0.45
Conc 12	73.14	75.3	82.5	0.36	0.46
Tails	100.00	100.0	100.0	0.32	0.44

Table A2.3 Averaged data for base test in E3 column cell

<i>Average</i>	Cum. % mass	Cum. % Rec		Cum. % Grade	
		Ni	Cu	Cu%	Ni %
Conc 1	25.70	27.4	42.7	0.59	0.55
Conc2	42.05	40.2	55.5	0.45	0.49
Conc4	56.25	52.8	64.8	0.39	0.48
Conc6	63.37	60.0	69.4	0.37	0.48
Conc 9	67.96	66.4	73.1	0.36	0.49
Conc 12	70.15	69.2	74.9	0.36	0.50
Tails	100.00	100.0	100.0	0.34	0.51
Variance(%)				9.25	6.68

B2.3.2 Recycle tests

Table A2.4 Laboratory data for recycle test 1 in E3 column cell

<i>TEST 1</i>	Cum. % mass	Cum. % Rec		Cum. % Grade	
		Ni	Cu	Cu%	Ni %
Conc 1	18.75	20.7	35.9	0.69	0.60
Conc2	33.70	33.4	49.7	0.53	0.54
Conc4	51.74	50.2	63.4	0.44	0.53
Conc6	60.29	59.1	69.5	0.41	0.53
Conc 9	65.96	66.0	73.6	0.40	0.54
Conc 14	69.56	72.1	77.6	0.40	0.56
Tails	100.00	100.0	100.0	0.36	0.54

Table A2.5 Laboratory data for recycle test 2 in E3 column cell

<i>TEST 2</i>	Cum. % mass	Cum. % Rec		Cum. % Grade	
		Ni	Cu	Cu%	Ni %
Conc 1	16.14	22.5	32.9	0.71	0.80
Conc2	26.82	36.1	46.9	0.61	0.78
Conc4	41.44	50.5	57.7	0.49	0.70
Conc6	49.67	59.0	65.9	0.46	0.69
Conc 9	56.17	65.1	70.5	0.44	0.67
Conc 14	61.76	70.9	75.0	0.43	0.66
Tails	100.00	100.0	100.0	0.35	0.58

Table A2.6 Averaged data for recycle test in E3 column cell

<i>Average</i>	Cum. % mass	Cum. % Rec		Cum. % Grade	
		Ni	Cu	Cu%	Ni %
Conc 1	17.44	21.6	34.4	0.70	0.70
Conc2	30.26	34.8	48.3	0.57	0.66
Conc4	46.59	50.3	60.6	0.46	0.61
Conc6	54.98	59.0	67.7	0.44	0.61
Conc 9	61.07	65.5	72.1	0.42	0.60
Conc 12	65.66	71.5	76.3	0.41	0.61
Tails	100.00	100.0	100.0	0.36	0.56
Variance(%)				4.94	3.12

A2.3.3 Depressant dosage tests

Depressant tests were performed for 10 g/t and 30 g/t depressant addition as shown below.

Table A2.7 Laboratory data for depressant test 1 (30 g/t) in E3 column cell

<i>TEST 1</i>	Cum. % mass	Cum. % Rec		Cum. % Grade	
		Ni	Cu	Cu%	Ni %
Conc 1	3.55	47.3	58.2	5.71	5.67
Conc2	4.42	53.3	65.9	5.19	5.13
Conc4	5.46	59.8	73.0	4.65	4.66
Conc6	5.93	60.6	73.8	4.33	4.35
Conc 9	6.24	60.9	74.2	4.14	4.16
Tails	100.00	100.0	100.0	0.37	0.42

Table A2.8 Laboratory data for depressant test 2 (30 g/t) in E3 column cell

<i>TEST 2</i>	Cum. % mass	Cum. % Rec		Cum. % Grade	
		Ni	Cu	Cu%	Ni %
Conc 1	1.38	13.8	19.8	4.44	4.91
Conc2	2.27	24.0	31.5	4.29	5.20
Conc4	3.28	28.6	37.4	3.52	4.29
Conc6	4.14	31.5	40.4	3.02	3.74
Conc 9	4.88	33.6	42.8	2.71	3.39
Tails	100.00	100.0	100.0	0.31	0.56

Table A2.9 Averaged data for 30g/t depressant dosage test in E3 column cell

<i>Average</i>	Cum. % mass	Cum. % Rec		Cum. % Grade	
		Ni	Cu	Cu%	Ni %
Conc 1	2.47	30.5	39.0	5.07	5.29
Conc2	3.34	38.6	48.7	4.74	5.17
Conc4	4.37	44.2	55.2	4.08	4.48
Conc6	5.04	46.0	57.1	3.67	4.04
Conc 9	5.56	47.2	58.5	3.43	3.77
Tails	100.00	100.0	100.0	0.34	0.49
Variance(%)				9.73	10.72

Table A2.10 Laboratory data for depressant test 3 (10 g/t) in E3 column cell

<i>TEST 3</i>	Cum. % mass	Cum. % Rec		Cum. % Grade	
		Ni	Cu	Cu%	Ni %
Conc 1	6.56	15.1	33.6	1.57	1.11
Conc2	9.77	25.7	50.4	1.57	1.27
Conc4	12.84	31.6	56.8	1.35	1.19
Conc6	15.06	36.3	60.8	1.23	1.16
Conc 9	17.86	37.1	61.6	1.05	1.00
Tails	100.00	100.0	100.0	0.31	0.48

Table A2.11 Laboratory data for depressant test 4 (10 g/t) in E3 column cell

<i>TEST 4</i>	Cum. % mass	Cum. % Rec		Cum. % Grade	
		Ni	Cu	Cu%	Ni %
Conc 1	9.44	13.9	21.6	0.74	0.79
Conc2	12.69	20.3	30.5	0.78	0.86
Conc4	16.74	26.8	39.2	0.76	0.86
Conc6	19.50	35.5	48.7	0.81	0.98
Conc 9	22.13	39.2	54.1	0.79	0.95
Tails	100.00	100.0	100.0	0.32	0.54

Table A2.12 Averaged data for 10g/t depressant dosage test in E3 column cell

<i>Average</i>	Cum. % mass	Cum. % Rec		Cum. % Grade	
		Ni	Cu	Cu%	Ni %
Conc 1	8.00	14.5	27.6	1.15	0.95
Conc2	11.23	23.0	40.4	1.18	1.07
Conc4	14.79	29.2	48.0	1.05	1.03
Conc6	17.28	35.9	54.7	1.02	1.07
Conc 9	20.00	38.1	57.9	0.92	0.98
Tails	100.00	100.0	100.0	0.31	0.51
Variance(%)				17.40	6.12

A2.3.5 Manual Conditioning of depressant

Table A2.13 Laboratory data for depressant test 5 (30 g/t) in E3 column cell

<i>TEST 5</i>	Cum. % mass	Cum. % Rec		Cum. % Grade	
		Ni	Cu	Cu%	Ni %
Conc 1	2.38	27.6	47.0	6.24	5.19
Conc2	2.88	31.4	53.0	5.82	4.87
Conc4	3.51	32.5	54.5	4.92	4.14
Conc6	3.97	34.3	56.6	4.51	3.86
Tails	100.00	100.0	100.0	0.34	0.49

Appendix A3: Tables of laboratory results for series C

A3.1: MECH. CELL

The following tables provide the laboratory data for tests performed on the mechanical cell in series C. It is divided into 2 sub-sections namely the base tests and depressant dosage tests.

A3.4.1 Base case tests

Table A3.1 Laboratory data for base test 1 in mechanical cell

<i>TEST 1</i>	Cum. % mass	Cum. % Rec	Cum. % Grade
		pgm	pgm
Conc 1	29.43	51.3	68
Conc2	44.13	67.0	59
Conc4	54.01	76.2	55
Conc6	57.46	79.0	53
Tails	100.00	100.0	39

Table A3.2 Laboratory data for base test 2 in mechanical cell

<i>TEST E</i>	Cum. % mass	Cum. % Rec	Cum. % Grade
		pgm	pgm
Conc 1	38.90	57.7	58
Conc2	50.64	70.2	54
Conc4	61.39	80.7	51
Conc6	64.18	83.2	51
Tails	100.00	100.0	39

Table A3.3 Averaged data for base test in mechanical cell

<i>Average</i>	Cum. % mass	Cum. % Rec	Cum. % Grade
		pgm	Pgm
Conc 1	34.16	54.50	62.80
Conc2	47.38	68.56	56.53
Conc4	57.70	78.45	53.07
Conc6	60.82	81.10	52.01
Tails	100.00	100.00	38.95
Variance(%)			10.14

A3.4.2 Depressant dosage tests

Depressant dosage tests were performed for 10 g/t, 20 g/t, and 30 g/t depressant dosages.

Table A3.4 Laboratory data for depressant test 1 (10 g/t) in mechanical cell

<i>TEST 1</i>	Cum. % mass	Cum. % Rec	Cum. % Grade
		pgm	pgm
Conc 1	20.35	67.6	128
Conc2	26.19	75.8	112
Conc4	31.04	81.6	101
Conc6	33.73	84.1	96
Tails	100.00	100.0	39

Table A3.5 Laboratory data for depressant test 2 (20 g/t) in mechanical cell

<i>TEST 2</i>	Cum. % mass	Cum. % Rec	Cum. % Grade
		pgm	pgm
Conc 1	7.97	64.7	349
Conc2	11.53	75.6	282
Conc4	13.98	83.9	258
Conc6	13.98	83.9	258
Tails	100.00	100.0	43

Table B A3.6 Laboratory data for depressant test 3 (30 g/t) in mechanical cell

<i>TEST 3</i>	Cum. % mass	Cum. % Rec	Cum. % Grade
		pgm	pgm
Conc 1	5.04	49.2	518
Conc 2	9.11	80.6	470
Conc 4	9.11	80.6	470
Conc 6	9.11	80.6	470
Tails	100.00	100.0	53

A3.2: E3 COLUMN CELL

The following tables provide the laboratory data for tests performed on the mechanical cell in series C. It is divided into 6 sub-sections namely the froth depth test, air flowrate tests, base tests, recycle tests, depressant dosage tests, depressant conditioning tests.

A3.2.1 Froth depth tests

Tests were performed on the E3 column cell for 3 different froth depths (20 cm, 15 cm and 6 cm)

Table A3.7 Laboratory data for froth depth test 1 (20 cm) in E3 column cell

<i>TEST 1</i>	Cum. % mass	Cum. % Rec	Cum. % Grade
		pgm	Pgm
Conc 1	51.98	62.5	52
Conc 2	62.29	71.6	49
Conc 4	79.81	86.2	46
Conc 6	82.71	88.8	46
Tails	100.00	100.0	43

Table A3.8 Laboratory data for froth depth test 2 (20 cm) in E3 column cell

<i>TEST 2</i>	Cum. % mass	Cum. % Rec	Cum. % Grade
		pgm	Pgm
Conc 1	35.58	45.6	56
Conc 2	61.52	71.2	51
Conc 4	78.89	86.5	48
Conc 6	83.50	90.5	48
Tails	100.00	100.0	44

Table A3.9 Averaged data for froth depth test in 20cm in E3 column cell

<i>Average</i>	Cum. % mass	Cum. % Rec	Cum. % Grade
		pgm	pgm
Conc 1	43.78	54.09	53.95
Conc2	61.90	71.42	50.06
Conc4	79.35	86.35	47.22
Conc6	83.10	89.64	46.80
Tails	100.00	100.00	43.39
Variance(%)	0.32	1.58	1.20

Table A3.10 Laboratory data for froth depth test 3 (15 cm) in E3 column cell

<i>TEST 3</i>	Cum. % mass	Cum. % Rec	Cum. % Grade
		pgm	pgm
Conc 1	38.17	48.1	53
Conc 2	59.42	69.0	49
Conc 4	73.51	81.7	47
Conc 6	78.49	86.5	47
Tails	100.00	100.0	42

Table A3.11 Laboratory data for froth depth test 4 (6 cm) in E3 column cell

<i>TEST4</i>	Cum. % mass	Cum. % Rec	Cum. % Grade
		pgm	pgm
Conc 1	38.74	50.6	59
Conc 2	53.98	65.2	54
Conc 4	64.91	75.5	52
Conc 6	69.79	79.9	52
Conc 9	73.35	83.2	51
Tails	100.00	100.0	45

A3.2.2 Air flowrate tests

Tests were performed on the E3 column cell for 3 different air flow rates (1ℓ/min, 2ℓ/min and 3ℓ/min)

Table A3.12 Laboratory data for air flow test 3 (1ℓ/min) in E3 column cell

TEST 3	Cum. % mass	Cum. % Rec	Cum. % Grade
		pgm	pgm
Conc 1	30.47	20.3	28
Conc 2	42.96	40.4	40
Conc 4	51.54	46.5	38
Conc 6	57.49	64.4	47
Conc 9	61.94	69.2	47
Tails	100.00	100.0	42

Table A3.13 Laboratory data for air flow test 1 (2ℓ/min) in E3 column cell

TEST 1	Cum. % mass	Cum. % Rec	Cum. % Grade
		Pgm	pgm
Conc 1	38.74	50.6	59
Conc 2	53.98	65.2	54
Conc 4	64.91	75.5	52
Conc 6	69.79	79.9	52
Conc 9	73.35	83.2	51
Tails	100.00	100.0	45

Table A3.14 Laboratory data for air flow test 2 (3ℓ/min) in E3 column cell

TEST 2	Cum. % mass	Cum. % Rec	Cum. % Grade
		Pgm	pgm
Conc 1	40.97	50.3	58
Conc 2	61.78	71.0	55
Conc 4	74.98	82.8	53
Conc 6	79.06	86.6	52
Conc 9	82.28	89.5	52
Tails	100.00	100.0	48

A3.2.3 Base tests

Table A3.16 Laboratory data for base test 1 (6 cm) in E3 column cell

<i>TEST 1</i>	Cum. % mass	Cum. % Rec	Cum. % Grade
		Pgm	pgm
Conc 1	33.30	47.2	49
Conc 2	49.86	63.2	44
Conc 4	62.47	74.0	41
Conc 6	68.49	79.2	40
Conc 9	73.41	83.4	40
Tails	100.00	100.0	35

Table A3.17 Laboratory data for base test 2 (6 cm) in E3 column cell

<i>TEST 2</i>	Cum. % mass	Cum. % Rec	Cum. % Grade
		pgm	pgm
Conc 1	38.74	50.6	59
Conc 2	53.98	65.2	54
Conc 4	64.91	75.5	52
Conc 6	69.79	79.9	52
Conc 9	73.35	83.2	51
Tails	100.00	100.0	45

Table A3.18 Averaged data for base test in 6cm in E3 column cell

<i>Average</i>	Cum. % mass	Cum. % Rec	Cum. % Grade
		pgm	pgm
Conc 1	36.02	48.90	54.10
Conc2	51.92	64.23	49.29
Conc4	63.69	74.76	46.83
Conc6	69.14	79.55	45.91
Conc 9	73.38	83.34	45.35
Tails	100.00	100.00	39.93
Variance(%)	0.00	0.02	4.13

A3.2.4 Recycle tests

Tests were performed on the E3 column cell with and without the use of a recycle.

Table A3.19 Laboratory data for recycle test 1 (without recycle) in E3 column cell

<i>TEST 1</i>	Cum. % mass	Cum. % Rec	Cum. % Grade
		pgm	Pgm
Conc 1	33.30	47.2	49
Conc 2	49.86	63.2	44
Conc 4	62.47	74.0	41
Conc 6	68.49	79.2	40
Conc 9	73.41	83.4	40
Tails	100.00	100.0	35

Table A3.20 Laboratory data for recycle test 2 (with recycle) in E3 column cell

<i>TEST2</i>	Cum. % mass	Cum. % Rec	Cum. % Grade
		pgm	pgm
Conc 1	25.80	40.2	55
Conc 2	38.50	51.8	50
Conc 4	50.20	60.0	46
Conc 6	58.20	67.3	44
Conc 9	69.50	83.3	40
Tails	100.00	100.0	32

A3.2.5 Depressant Dosage test

Depressant dosage tests were performed for 10 g/t, 20 g/t, and 30 g/t depressant dosages

Table A3.21 Laboratory data for depressant test 1 (10g/t) in E3 column cell

<i>TEST 1</i>	Cum. % mass	Cum. % Rec	Cum. % Grade
		pgm	Pgm
Conc 1	19.00	53.9	121
Conc 2	26.07	61.5	101
Conc 4	40.18	72.8	77
Conc 6	48.39	78.4	69
Conc 9	56.47	83.4	63
Tails	100.00	100.0	43

Table A3.22 Laboratory data for depressant test 2 (10g/t) in E3 column cell

<i>TEST 2</i>	Cum. % mass	Cum. % Rec	Cum. % Grade
		pgm	Pgm
Conc 1	15.26	24.8	61
Conc 2	22.09	58.4	100
Conc 4	28.68	62.6	82
Conc 6	39.45	71.5	68
Conc 9	49.85	80.7	42
Tails	100.00	100.0	38

Table A3.23 Averaged data for 10g/t depressant dosage test in E3 column cell

<i>Average</i>	Cum. % mass	Cum. % Rec	Cum. % Grade
		pgm	pgm
Conc 1	17.13	39.35	91.15
Conc2	24.08	59.91	100.08
Conc4	34.43	67.67	79.76
Conc6	43.92	74.94	68.71
Conc 9	53.16	82.04	52.62
Tails	100.00	100.00	40.18
Variance(%)	21.92	3.56	3.49

Table A3.24 Laboratory data for depressant test 3 (20g/t) in E3 column cell

<i>TEST 3</i>	Cum. % mass	Cum. % Rec	Cum. % Grade
		pgm	Pgm
Conc 2	7.33	59.7	364
Tails	100.00	100.0	45

Table A3.25 Laboratory data for depressant test 4 (30g/t) in E3 column cell

<i>TEST 4</i>	Cum. % mass	Cum. % Rec	Cum. % Grade
		pgm	pgm
Conc 2	0.73	10.4	533
Tails	100.00	100.0	38

A3.2.6 Depressant conditioning test

Tests were performed on the E3 column cell for 2 different depressant conditioning tests (manual depressant test and recycle depressant test.)

Table A3.26 Laboratory data for depressant conditioning test 1(manual) in E3 column cell

<i>TEST 1</i>	Cum. % mass	Cum. % Rec	Cum. % Grade
		pgm	Pgm
Conc 1	24.07	58.9	82
Conc 2	32.53	66.9	69
Conc 4	42.54	74.3	59
Conc 6	47.41	78.1	55
Conc 9	52.40	81.2	52
Tails	100.00	100.0	34

Table A3.27 Laboratory data for depressant conditioning test 2 (recycle) in E3 column cell

<i>TEST2</i>	Cum. % mass	Cum. % Rec	Cum. % Grade
		pgm	pgm
Conc 1	27.24	53.1	64
Conc 2	37.44	65.6	58
Conc 4	44.55	71.4	53
Conc 6	50.61	76.8	50
Conc 9	56.96	83.0	48
Tails	100.00	100.0	33

A3.3 COLUMN CELL (2H)

The height of the column cell was doubled and standard flotation tests were performed

Table A3.28 Laboratory data for height test 2 in column cell (2H)

<i>TEST 2</i>	Cum. % mass	Cum. % Rec	Cum. % Grade
		pgm	pgm
Conc 1	13.79	20.6	63
Conc 2	23.62	35.1	63
Conc 4	31.48	46.6	62
Conc 6	36.86	54.7	62
Conc 9	42.54	61.8	61
Conc 12	48.52	70.3	60
Tails	100.00	100.0	41

Table A3.29 Laboratory data for height 3 in column cell (2H)

<i>TEST3</i>	Cum. % mass	Cum. % Rec	Cum. % Grade
		pgm	pgm
Conc 1	22.53	37.6	67
Conc 2	35.78	51.6	58
Conc 4	51.82	67.6	52
Conc 6	59.06	74.5	51
Conc 9	64.37	79.6	50
Conc 12	67.10	82.8	50
Tails	100.00	100.0	40

Table A3.30 Averaged data for height test in column cell (2H)

<i>Average</i>	Cum. % mass	Cum. % Rec	Cum. % Grade
		pgm	pgm
Conc 1	18.16	29.09	65.00
Conc2	29.70	43.35	60.27
Conc4	41.65	57.09	57.38
Conc6	47.96	64.57	56.58
Conc 9	53.46	70.71	55.46
Conc12	57.81	76.56	54.79
Tails	100.00	100.00	40.59
Variance(%)			2.45

Appendix A4: Tables of plant audit results for FCTR

The following tables are the compiled results for all investigations conducted on the FCTR .

Table A4.1 Raw data for FCTR tests

	Wet mass (g)	Dry mass (g)	Time (s)	Density (g.cm⁻³)	Grade (PPM)	Ave Grade (PPM)
Feed						
cut 1	8063.1	3357	14.6		64.86	
cut 2	6402.2	2691	15.34		94.653	
cut 3	7949.1	3335	16.06	1.317	80.347	79.95
Conc A						
cut 1	3056.5	1250	86.72		409.964	
cut 2	3069	1282.2	86.6		469.415	
cut 3	3884.1	1628.8	102.26	2.745	397.037	425.47
Conc B						
cut 1	3132.7	1275.1	120.34		246.429	
cut 2	3024.8	1249.6	119.14		225.511	
cut 3	2491.4	1252.9	116.56	2.537	217.941	229.96
Conc C						
cut 1	3321.5	1349	110.5		110.558	
cut 2	3059.1	1245.3	112.15		133.953	
cut 3	3102.6	1262	113.85	1.694	110.334	118.28
Conc D						
cut 1	3783.3	1549.4	66.09		68.427	
cut 2	3802.2	1559.3	68.13		58.577	
cut 3	4569.3	1870.9	79.73	1.500	54.618	60.54
Conc E						
cut 1	1960.2	814.5	68.57		44.672	
cut 2	2180	908.6	80.69		45.038	
cut 3	2531.5	1059.9	93.01	1.400	40.154	43.29
Tails						
cut 1	8208.2	3401	25.28		31.3	
cut 2	8546.7	3634.1	29.95		36.51	
cut 3	9295.5	3920.8	31.82	1.863	32.6	33.47

Table A4.2 Calculation of the residence time in each cell of the FCTR

	<i>Froth Depth</i> (cm)	<i>Air flow</i> (L/min)	<i>Pulp volume</i> (L)	<i>Residence Time</i> (min)
cell A	37	90	63	2.84
cell B	30	80	78	3.65
cell C	28	80	83	3.95
cell D	26	75	87	4.37
cell E	24	70	91	5.17

Table A4.3 Wet and dry mass balances for cut 1 of FCTR

Cut 1: Wet Mass Balance			Cut 1: Dry Mass Balance				
	Feed (g/s)	Conc (g/s)	Tails (g/s)		Feed (g/s)	Conc (g/s)	Tails (g/s)
cell A	552.27	35.25	517.02	cell A	229.95	14.41	215.53
cell B	517.02	26.03	490.99	cell B	215.53	10.60	204.94
cell C	490.99	30.06	460.93	cell C	204.94	12.21	192.73
cell D	460.93	57.24	403.69	cell D	192.73	23.44	169.28
cell E	403.69	28.59	375.10	cell E	169.28	11.88	157.40
		Measured	324.69			Measured	134.53
		Variance	13.44			Variance	14.53

Table A4.4 Wet and dry mass balances for cut 2 of FCTR

Cut 2: Wet Mass Balance			Cut 2: Dry Mass Balance				
	Feed (g/s)	Conc (g/s)	Tails (g/s)		Feed (g/s)	Conc (g/s)	Tails (g/s)
cell A	417.35	35.44	381.91	cell A	175.39	14.81	160.59
cell B	381.91	25.39	356.53	cell B	160.59	10.49	150.10
cell C	356.53	27.28	329.25	cell C	150.10	11.10	138.99
cell D	329.25	55.81	273.44	cell D	138.99	22.89	116.11
cell E	273.44	27.02	246.42	cell E	116.11	11.26	104.85
		Measured	285.37			Measured	121.34
		Variance	15.80			Variance	15.73

Table A4.5 Wet and dry mass balances for cut 3 of FCTR

Cut 3: Wet Mass Balance				Cut 3: Dry Mass Balance			
	Feed (g/s)	Conc (g/s)	Tails (g/s)		Feed (g/s)	Conc (g/s)	Tails (g/s)
cell A	494.96	37.98	456.98	cell A	207.68	15.93	191.76
cell B	456.98	21.37	435.61	cell B	191.76	10.75	181.01
cell C	435.61	27.25	408.35	cell C	181.01	11.08	169.92
cell D	408.35	57.31	351.04	cell D	169.92	23.47	146.46
cell E	351.04	27.22	323.83	cell E	146.46	11.40	135.06
		Measured	292.13			Measured	123.22
		Variance	9.79			Variance	8.77

Table A4.6 Overall all wet and dry mass balances of FCTR

Overall Wet Mass Balance				Overall Dry Mass Balance			
	Feed (g/s)	Conc (g/s)	Tails (g/s)		Feed (g/s)	Conc (g/s)	Tails (g/s)
cell A	488.19	36.22	451.97	cell A	204.34	15.05	189.29
cell B	451.97	24.27	427.71	cell B	189.29	10.61	178.68
cell C	427.71	28.20	399.51	cell C	178.68	11.47	167.21
cell D	399.51	56.79	342.72	cell D	167.21	23.27	143.95
cell E	342.72	27.61	315.12	cell E	143.95	11.51	132.44
		Measured	300.73			Measured	126.00
		Variance	4.57			Variance	4.86

Table A4.7 Grade and recovery calculations for cut 1

Cut One	Cum Mass (%)	Grade (PPM)	Cum Grade (PPM)	Cum Recovery (%)
cell A	6.27	410	410	34.90
cell B	10.88	246	341	50.32
cell C	16.19	111	265	58.29
cell D	26.38	68	189	67.77
cell E	31.55	45	165	70.90
Tails	100.00	31	74	100.00
Recalc Head Grade		73.63		
Feed Grade		64.86		
Variance		11.92		

Table A4.8 Grade and recovery calculations for cut 2

Cut Two	Cum Mass (%)	Grade (PPM)	Cum Grade (PPM)	Cum Recovery (%)
cell A	7.71	469	469	40.67
cell B	13.17	226	368	54.52
cell C	18.96	134	297	63.22
cell D	30.94	59	205	71.11
cell E	36.80	45	179	74.07
Tails	100.00	37	89	100.00
Recalc Head Grade		89.00		
Feed Grade		94.653		
Variance		6.35		

Table A4.9 Grade and recovery calculations for cut 3

Cut Three	Cum Mass (%)	Grade (PPM)	Cum Grade (PPM)	Cum Recovery (%)
cell A	8.13	397	397	40.42
cell B	13.62	218	325	55.39
cell C	19.28	110	262	63.21
cell D	31.26	55	182	71.40
cell E	37.08	40	160	74.33
Tails	100.00	33	80	100.00
Recalc Head Grade		79.89		
Feed Grade		80.35		
Variance		0.57		

Table A4.10 Overall grade and recovery calculations

Overall	Cum Mass (%)	Grade (PPM)	Cum Grade (PPM)	Cum Recovery (%)
cell A	7.36	425	425	38.72
cell B	12.56	230	345	53.47
cell C	18.17	118	275	61.67
cell D	29.55	61	192	70.19
cell E	35.19	43	168	73.20
Tails	100.00	33	81	100.00
Recalc Head Grade		80.94		
Feed Grade		79.95		
Variance (%)		1.22		

Appendix A5: Tables of plant audit results for pilot column

The following tables are the compiled results for all investigations conducted on the pilot column at Lonmin Platinum.

Table A5.11 Dimensions of pilot column

Dimensional Data for Column		
Diameter	0.3	m
Height	5.345	m
Volume	0.38	m ³

Table A5.12 Relationship between pressure and froth depth in pilot column

Pressure (kPa)	Froth Depth (cm)
48	55.50
50	37.00
52	18.48

Table A5.13 Density and residence time calculations for pilot column for three different flows

	Slurry mass (g)	Volume (L)	Time (s)	Volumetric Flow (L/min)	Residence time(min)	Density (g.cm ³)
Flow 1	20967	15	98.00	9.2	41.14	1.398
Flow 2	19804	15	36.31	24.8	15.24	1.320
Flow 3	19770	15	15.32	58.7	6.43	1.318

Table A5.14 Grade and recovery calculations for different froth depths for flow 1

P(kPa)	Flow 1	Mass (g/s)	Mass Recovered (%)	Grade (PPM)	Recovery (%)
48	Feed	99.29	2.46	46.40	89.39
48	Conc	2.33			
48	Tails	92.40		1.993	
	Recalculated	94.73		18.33	
	Variance	4.59		60.50	
50	Feed	96.57	10.69	34.00	42.96
50	Conc	9.18			
50	Tails	76.72		41.10	
	Recalculated	85.90		64.35	
	Variance	11.04		89.28	
52	Feed	98.03	19.94	46.10	61.23
52	Conc	13.65			
52	Tails	54.81		31.37	
	Recalculated	68.46		64.79	
	Variance	30.17		40.53	

Table A5.15 Grade and recovery calculations for different froth depths for flow 2

P(kPa)	Flow 2	Mass (g/s)	Mass Recovered (%)	Grade (PPM)	Recovery (%)
48	Feed	130.02	4.93	96.88	44.56
48	Conc	6.83		615.67	
48	Tails	131.50		39.77	
	Recalculated	138.33		68.19	
	Variance	6.39		29.61	
50	Feed	173.80	7.46	69.68	44.24
50	Conc	10.59		505.92	
50	Tails	131.28		51.44	
	Recalculated	141.87		85.36	
	Variance	18.37		22.50	
52	Feed	194.19	4.27	59.26	38.23
52	Conc	9.13		527.28	
52	Tails	204.46		38.04	
	Recalculated	213.58		58.95	
	Variance	9.99		0.53	

Table A5.16 Grade and recovery calculations for different froth depths for flow 3

P(kPa)	Flow 3	Mass (g/s)	Mass Recovered (%)	Grade (PPM)	Recovery (%)
48	Feed	499.83	8.19	90.49	28.11
48	Conc	37.31		255.54	
48	Tails	418.18		58.32	
	Recalculated	455.49		74.48	
	Variance	8.87		17.70	
50	Feed	519.07	5.54	75.75	25.24
50	Conc	26.63		263.02	
50	Tails	454.52		45.66	
	Recalculated	481.16		57.69	
	Variance	7.30		23.84	
52	Feed	533.85	15.97	69.36	41.39
52	Conc	76.71		184.34	
52	Tails	403.58		49.61	
	Recalculated	480.29		71.12	
	Variance	10.03		2.54	

APPENDIX B: SAMPLE CALCULATIONS

This section includes the milling calculations and the superficial velocity calculations in the mechanical and column cells as well a sample of the grade and recovery calculations performed on each of the cleaner tests.

APPENDIX B1: MILLING CALCULATIONS

B1.1 Determining the milling parameters.

The amount of ore required as feed to the mill was determined by the percentage solids needed in the rougher. 10 kg of ore would result in 20% solids in the rougher. This was not optimum (40% was recommended) but it was the best conditions achievable taking into account the dimension of the available rougher cell. Given 10 kg of feed, the resulting volume of slurry was determined.

The volume of the ore fed to the mill:

$$\begin{aligned} V_{solids} &= \frac{M_{Solids}}{\rho_{Solids}} && (\text{m}^3) && \text{(a)} \\ &= (10/3571) \\ &= 0.0028 \text{ m}^3 \end{aligned}$$

It was recommended that the mill operate at 63% solids. The resulting mass of water was determined.

$$\begin{aligned} M_{water} &= \frac{M_{Solids} \times 0.35}{0.65} && (\text{kg}) && \text{(b)} \\ &= (10/0.65) \times 0.35 \\ &= 5.8 \text{ kg} \end{aligned}$$

The corresponding volume of water required was determined, given that the density of water is 1000 kg/m³.

$$\begin{aligned} V_{Water} &= \frac{M_{water}}{D_{water}} && (\text{m}^3) && \text{(c)} \\ &= (5.8/1000) \\ &= 0.0058 \text{ m}^3 \text{ (5.8 L)} \end{aligned}$$

The resulting volume of slurry was then calculated.

$$\begin{aligned} V_{Slurry} &= V_{Water} + V_{Solids} \text{ (m}^3\text{)} \\ &= 0.0058 + 0.0028 \\ &= \mathbf{0.0086 \text{ m}^3} \end{aligned} \quad \text{(d)}$$

It was important to determine, at this point, the volume of slurry that the available milling apparatus could accommodate.

The volume of the cylindrical mill was:

$$\begin{aligned} V_{Mill} &= \frac{\pi D_m^2 L}{4} \quad \text{(m}^3\text{)} \\ &= (\pi \times (0.30)^2 \times 0.51) / 4 \\ &= 0.036 \text{ m}^3 \end{aligned} \quad \text{(e)}$$

A volumetric charge of 40% is normally used. Wills (1997) recommend a volumetric charge between 40% and 50%. Initially the best case scenario (40%) was considered.

$$\begin{aligned} V_{Charge} &= 0.4 \times V_{Mill} \quad \text{(m}^3\text{)} \\ &= 0.014 \text{ m}^3 \end{aligned} \quad \text{(f)}$$

The volumetric charge comprises of media and slurry. The recommended proportion is 60% media.

$$\begin{aligned} V_{Media} &= 0.6 \times V_{Charge} \quad \text{(m}^3\text{)} \\ &= 0.0086 \text{ m}^3 \end{aligned} \quad \text{(g)}$$

$$\begin{aligned} M_{SteelBalls} &= V_{SteelBalls} \times \rho_{Steel} \quad \text{(kg)} \\ &= 0.0086 \times 7850 \\ &= 68 \text{ kg} \end{aligned} \quad \text{(h)}$$

The volume of slurry that the mill could accommodate at optimum conditions was calculated.

$$\begin{aligned} V_{Slurry} &= V_{charge} - V_{Steelballs} \\ &= \mathbf{0.0058 \text{ m}^3} \end{aligned} \quad \text{(i)}$$

Evidently, the mill would not be able to accommodate the required volume of slurry at optimum conditions. If a volumetric charge of 50% (upper end of Wills' range) was accepted then the resulting volume of slurry that the mill would be able to accommodate would be 0.0072 m^3 . This was still short of the required 0.0086 m^3 .

The following operating conditions, which are below optimum, were chosen as a compromise, to accommodate the 10kg of solids:

- A charge volume of 52%
- A ratio of 54% media and 46% slurry by volume was employed.

The volume of charge was then recalculated.

$$\begin{aligned} V_{Charge} &= 0.52 \times V_{Mill} && (\text{m}^3) && \text{(j)} \\ &= 0.019 \text{ m}^3 \end{aligned}$$

The volume and mass of media required was determined.

$$\begin{aligned} V_{Media} &= 0.54 \times V_{Charge} && (\text{m}^3) && \text{(k)} \\ &= 0.010 \text{ m}^3 \end{aligned}$$

$$\begin{aligned} M_{SteelBalls} &= V_{SteelBalls} \times \rho_{Steel} && (\text{kg}) && \text{(h)} \\ &= 0.010 \times 7850 \\ &= 80 \text{ kg} \end{aligned}$$

Thus the required volume of slurry could be accommodated.

$$\begin{aligned} V_{Slurry} &= V_{charge} - V_{Steelballs} && \text{(i)} \\ &= \mathbf{0.0086 \text{ m}^3} \end{aligned}$$

Hence 80kg of steel balls and 5.8L of water was used to mill 10kg of UG-2 ore.

B1.2 Determining the mill speed

The mill speed was based on the critical speed which is calculated below:

$$\begin{aligned}\omega_{critical} &= \frac{42.29}{\sqrt{d}} \quad (\text{rpm}) \\ &= 42.29/(0.30)^{1/2} \\ &= 77 \text{ rpm}\end{aligned}\tag{1}$$

The operating of the speed of the mill must be in the range of 75 % to 85% of the critical speed. The operating speed was set at 60 rpm, 78% of the critical speed.

B1.3 Determining the power usage

In order to determine the power used by the mill a power calibration was performed. The mill power calibration related the measured voltage to the actual power used by the mill. To obtain the calibration curve different masses were attached to the empty mill as seen below. The corresponding voltages were recorded using the voltmeter.

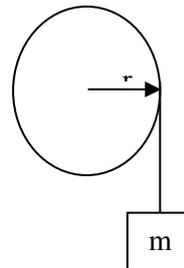


Figure B1.1 Schematic diagram indicated method for determining torque.

Using the known mass, it is possible to determine the force exerted by that mass and the resulting torque that was exerted on the mill. The power can then be easily determined. The first mass of used was 0.678 kg. The force exerted by the mass (m) was calculated.

$$\begin{aligned}F &= m \times g \\ &= 0.678 \times 9.81 \\ &= 6.65 \text{ N}\end{aligned}\tag{m}$$

The resulting torque, τ , exerted on the mill was determined.

$$\begin{aligned}\tau &= F \times r \quad (\text{N.m}) \\ &= 6.65 \times (0.34/2) \\ &= 1.13 \text{ N.m}\end{aligned}\tag{n}$$

The power (P) was determined calculated by the following equation:

$$\begin{aligned}
 P &= \tau \times \omega \times \left(\frac{2\pi}{60} \right) \quad (\text{W}) \\
 &= 1.13 \times 60 \times (2\pi/60) \\
 &= 7.10 \text{ W}
 \end{aligned}
 \tag{1}$$

Where: ω = rotational speed of the shell in rpm

By repeating the procedure for different masses the following data was obtained.

Table B1.1 Power Calibration Data

No	mass (kg)	Volts (V)	Force (N)	Torque (N.m)	Energy (N.m)
1	0.678	1.01	6.65	1.13	7.10
2	10.718	1.64	105.14	17.87	112.31
3	21.008	2.31	206.09	35.04	220.13
4	31.708	3.01	311.06	52.88	332.25
5	42.308	3.68	415.04	70.56	443.32

The power was then determined for a range of masses used and then plotted against the measured voltage. Figure B1.2 is the subsequent calibration curve.

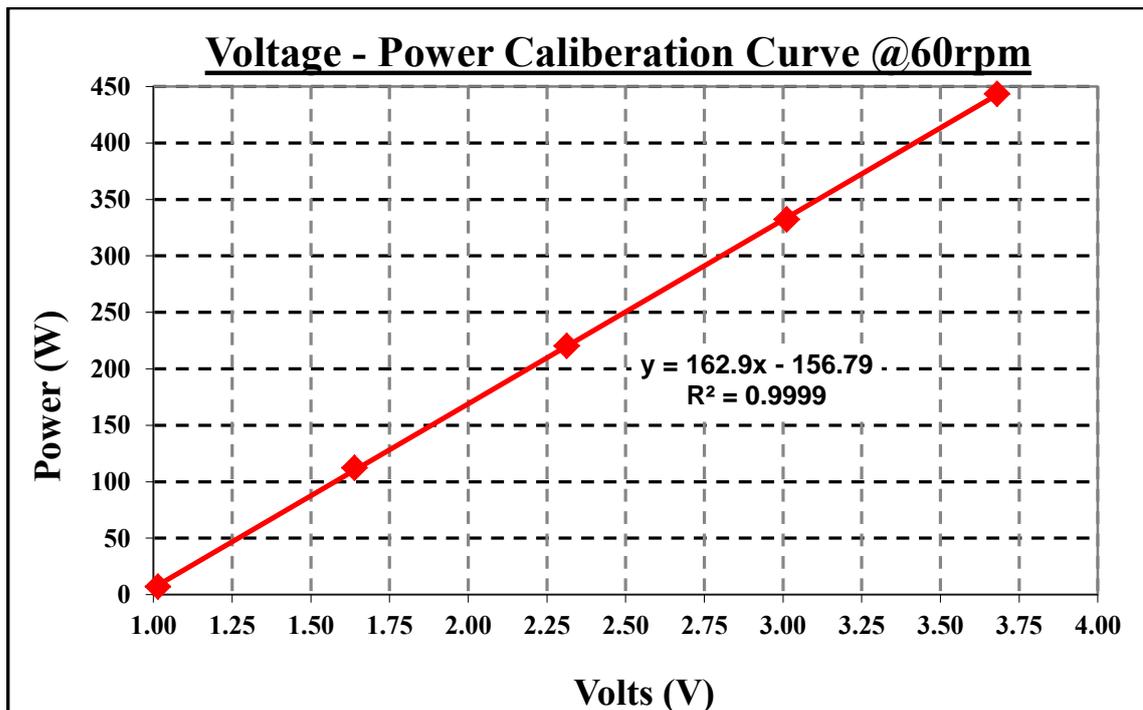


Figure B1.2 Voltage–Power calibration for ball mill

During the actual experiments the voltage during milling was monitored. The average voltage measured was 5.20 volts. From the equation of the calibration curve the actual power used can be determined.

$$\begin{aligned}
 P &= 162.5V - 156.7 & (o) \\
 &= (162.5 \times 5.20) - 156.7 \\
 &= 690.3 \text{ W}
 \end{aligned}$$

APPENDIX B2: CALCULATING THE SUPERFICIAL VELOCITY

One of the criteria for a plausible comparison between the mechanical and column cell was that the superficial velocity of the mechanical and column cells are the same.

B2.1 Calculating the superficial velocity in the mechanical cell

An air flowrate of 20.6 L/min was used in the Denver flotation cell for all mechanical cell tests. In order to calculate the superficial velocity the cross sectional area of the mechanical was required. For the purpose of calculating the cross sectional area the mechanical cell was taken to be square i.e. the angling of the cell toward the mouth was ignored.

$$\begin{aligned}
 A_{CS} &= s^2 & (p) \\
 &= 16.0 \times 16.0 \\
 &= 256 \text{ cm}^2
 \end{aligned}$$

Hence the superficial velocity of the mechanical cell was calculated as follows:

$$\begin{aligned}
 v_m &= \frac{q_{air}}{A_{CS}} & (q) \\
 &= 0.0206 / 0.0256 \\
 &= 0.805 \text{ m/min}
 \end{aligned}$$

B2.2 Calculating the required air flowrate in column cell (H)

The air flowrate in the column cell was determined by setting the superficial velocity of the column equivalent to that of the mechanical cell.

$$\begin{aligned}
 v_c &= v_m & (r) \\
 &= 0.805 \text{ m/min}
 \end{aligned}$$

The cross sectional area of the column cell was determined.

$$\begin{aligned}
 A_{CS} &= \frac{\pi D_c^2}{4} & (s) \\
 &= (\pi \times (6.0)^2)/4 \\
 &= 28.3 \text{ cm}^2
 \end{aligned}$$

The required flowrate in the column cell would therefore be:

$$\begin{aligned}
 q_{air} &= \frac{V_c}{A_{CS}} & (t) \\
 &= (0.805/0.00283) \\
 &= 0.00227 \text{ m}^3/\text{min} = 2.27 \text{ L/min}
 \end{aligned}$$

The pressure drop in the column was measured to be 30 kPa (gauge pressure)

For a rotameter calibrated at 0 kPa (gauge pressure) and operating at 30 kPa (4.35 PSIG) , the correction factor for the rotameter reading is 1.137 as seen below in Figure C2.1. The resulting rotameter reading would be:

$$\begin{aligned}
 R &= q_{air} / C_f & (u) \\
 &= 2.27 / 1.137 \\
 &= 2.00 \text{ L/min}
 \end{aligned}$$

B2.3 Calculating the required air flowrate in the column cell (2H)

Doubling the height of the column increased the pressure drop in the column to 50 kPa (7.5 PSIG). According to Figure C2.1 the applicable correction factor is 1.219. the superficial velocity and air flowrate will remain the same as the original column however, the rotameter reading will change.

$$\begin{aligned}
 R &= q_{air} / C_f & (u) \\
 &= 2.27 / 1.219 \\
 &= 1.87 \text{ L/min}
 \end{aligned}$$



ROTAMETER PRESSURE CORRECTION FACTORS

PRESSURE (PSIG) FOR WHICH ROTAMETER IS CALIBRATED	PRESSURE (PSIG) FOR WHICH CORRECTION IS TO BE MADE																									
	0	2	4	6	8	10	15	20	25	30	35	40	50	60	70	80	90	100	120	140	160	180	200	220	240	260
0 OR 14.7 PSIA	1.000	1.066	1.128	1.187	1.243	1.296	1.421	1.536	1.643	1.744	1.838	1.929	2.098	2.254	2.400	2.538	2.669	2.793	3.027	3.244	3.447	3.639	3.822	3.996	4.163	4.323
2	0.938	1.000	1.059	1.114	1.166	1.217	1.334	1.442	1.542	1.638	1.725	1.810	1.967	2.114	2.251	2.381	2.504	2.621	2.839	3.043	3.234	3.413	3.583	3.749	3.905	4.056
4	0.867	0.945	1.000	1.052	1.102	1.149	1.261	1.362	1.457	1.546	1.631	1.710	1.860	1.999	2.128	2.251	2.368	2.476	2.663	2.876	3.056	3.226	3.387	3.543	3.691	3.833
6	0.843	0.898	0.951	1.000	1.047	1.093	1.198	1.295	1.385	1.469	1.549	1.625	1.767	1.900	2.022	2.138	2.248	2.354	2.550	2.733	2.905	3.066	3.219	3.368	3.508	3.643
8	0.805	0.858	0.908	0.955	1.000	1.043	1.144	1.235	1.323	1.403	1.480	1.552	1.688	1.814	1.932	2.043	2.149	2.247	2.435	2.610	2.774	2.928	3.073	3.215	3.350	3.479
10	0.771	0.822	0.870	0.915	0.958	1.000	1.098	1.185	1.268	1.345	1.418	1.488	1.618	1.739	1.852	1.959	2.059	2.155	2.335	2.502	2.658	2.806	2.946	3.083	3.211	3.335
15	0.704	0.750	0.793	0.835	0.874	0.912	1.000	1.081	1.156	1.227	1.293	1.357	1.476	1.586	1.690	1.786	1.877	1.965	2.129	2.282	2.424	2.559	2.687	2.811	2.928	3.041
20	0.651	0.693	0.734	0.772	0.808	0.844	0.925	1.000	1.070	1.135	1.197	1.255	1.365	1.467	1.562	1.652	1.737	1.818	1.970	2.111	2.243	2.368	2.486	2.601	2.709	2.814
25	0.609	0.649	0.688	0.722	0.756	0.789	0.865	0.935	1.000	1.061	1.119	1.174	1.277	1.371	1.460	1.544	1.623	1.700	1.842	1.974	2.098	2.215	2.323	2.431	2.533	2.630
30	0.573	0.611	0.647	0.681	0.713	0.743	0.815	0.881	0.943	1.000	1.055	1.106	1.204	1.292	1.376	1.455	1.530	1.603	1.736	1.860	1.977	2.087	2.190	2.291	2.387	2.474
35	0.544	0.579	0.613	0.646	0.676	0.705	0.773	0.835	0.894	0.948	1.000	1.049	1.142	1.226	1.305	1.380	1.451	1.520	1.646	1.764	1.874	1.979	2.077	2.173	2.264	2.351
40	0.518	0.552	0.585	0.615	0.644	0.672	0.738	0.797	0.852	0.904	0.953	1.000	1.085	1.168	1.243	1.315	1.383	1.448	1.569	1.682	1.787	1.887	1.980	2.071	2.158	2.241
50	0.477	0.508	0.538	0.566	0.592	0.618	0.678	0.733	0.784	0.831	0.876	0.919	1.000	1.075	1.144	1.210	1.272	1.331	1.443	1.548	1.643	1.735	1.822	1.905	1.984	2.061
60	0.444	0.473	0.500	0.526	0.551	0.575	0.631	0.682	0.729	0.773	0.816	0.856	0.931	1.000	1.065	1.126	1.184	1.239	1.343	1.439	1.529	1.615	1.695	1.773	1.847	1.916
70	0.417	0.444	0.470	0.494	0.518	0.540	0.592	0.640	0.685	0.728	0.766	0.804	0.874	0.939	1.000	1.057	1.112	1.164	1.261	1.351	1.436	1.516	1.591	1.665	1.734	1.801
80	0.394	0.420	0.444	0.468	0.489	0.511	0.560	0.605	0.648	0.687	0.724	0.760	0.828	0.888	0.948	1.000	1.052	1.100	1.192	1.278	1.358	1.434	1.505	1.574	1.640	1.703
90	0.375	0.399	0.423	0.445	0.466	0.486	0.533	0.576	0.616	0.654	0.689	0.723	0.788	0.845	0.899	0.951	1.000	1.046	1.134	1.216	1.292	1.364	1.432	1.497	1.560	1.620
100	0.359	0.382	0.404	0.425	0.445	0.464	0.509	0.550	0.588	0.624	0.658	0.691	0.751	0.807	0.859	0.909	0.956	1.000	1.094	1.161	1.234	1.303	1.368	1.430	1.490	1.548
120	0.330	0.352	0.373	0.392	0.411	0.428	0.470	0.508	0.543	0.576	0.608	0.637	0.693	0.745	0.793	0.839	0.882	0.925	1.000	1.072	1.138	1.202	1.262	1.320	1.375	1.428
140	0.306	0.326	0.348	0.365	0.383	0.400	0.438	0.474	0.507	0.538	0.567	0.594	0.647	0.695	0.740	0.782	0.822	0.861	0.933	1.000	1.063	1.122	1.177	1.232	1.283	1.333
160	0.290	0.309	0.327	0.344	0.360	0.376	0.413	0.448	0.477	0.508	0.534	0.560	0.609	0.654	0.696	0.736	0.774	0.810	0.879	0.941	1.000	1.055	1.108	1.159	1.207	1.254
180	0.275	0.293	0.310	0.326	0.342	0.356	0.391	0.422	0.452	0.479	0.505	0.530	0.576	0.619	0.660	0.697	0.733	0.767	0.832	0.891	0.948	1.000	1.049	1.098	1.144	1.188
200	0.262	0.279	0.295	0.311	0.325	0.339	0.373	0.402	0.430	0.456	0.481	0.505	0.549	0.590	0.629	0.664	0.698	0.731	0.792	0.850	0.903	0.953	1.000	1.045	1.089	1.131
220	0.250	0.267	0.282	0.297	0.311	0.324	0.356	0.385	0.411	0.436	0.460	0.483	0.525	0.564	0.601	0.635	0.668	0.699	0.758	0.812	0.863	0.911	0.957	1.000	1.042	1.083
240	0.240	0.256	0.271	0.285	0.299	0.311	0.342	0.369	0.395	0.419	0.442	0.463	0.504	0.542	0.577	0.610	0.641	0.671	0.727	0.779	0.828	0.874	0.916	0.960	1.000	1.039
260	0.231	0.247	0.261	0.275	0.288	0.300	0.329	0.355	0.380	0.403	0.425	0.446	0.485	0.522	0.555	0.587	0.617	0.646	0.700	0.750	0.798	0.842	0.884	0.924	0.963	1.000

FOR ROTAMETER CALIBRATED IN FREE UNITS: ROTAMETER SCALE READING X FACTOR = CORRECTED FLOW FOR NEW CONDITION

Figure B2.1 Rotameter correction factors

(<https://www.mathesongas.com/pdfs/flowchart/Rotameter-Pressure-Correction-Factors.pdf>)