

# **AN INVESTIGATION OF FROTH EFFECTS IN SCAVENGING FLOTATION OF PLATINUM FROM UG-2 ORE**

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## ABSTRACT

South Africa is the largest supplier of platinum group metals (PGMs), which are mined from three reefs in the Bushveld Igneous Complex. About 60% of the world's mined PGMs come from a single reef, the UG-2 reef (Mudd, 2010). Flotation is the primary method used to concentrate the PGMs. There are currently two major problems which are experienced during the flotation of UG-2 ore. Firstly, mineralogical studies have shown that the platinum losses in flotation plants are currently in excess of 10% and secondly the high chromite content in the flotation concentrate leads to downstream smelting problems. This project was aimed at improving the recovery of platinum and reducing the amount of chromite in the feed to the smelter.

Platinum concentrators in South Africa normally consist of two stages of grinding and flotation and this investigation was focussed on the second stage, where platinum-containing particles tend to float slowly and the fine grinding leads to entrainment of chromite. Tests were performed on a low-grade UG-2 ore sample, obtained from a plant (the feed to the secondary grinding mill). Sub-samples of the ore were ground to a size at which 80% passed 75  $\mu\text{m}$ . Subsequent flotation tests were done in two stages, the rougher and scavenger stages.

Focus of this project will be on optimising the scavenger stage as it has the potential to recover most of the 'lost' platinum. The objective of the research was to improve PGM recovery and reduce the chromite recovery in the scavenger. This was to be achieved by varying different parameters, which included froth washing, froth depth, the use of a baffle (an innovative technique, in which two baffle lengths were used), and the replacement of the standard frother dosage with: a mixture of diesel and frother; a mixture of paraffin and frother; a reduced frother dosage and no frother dosage. Flotation concentrates were obtained from the experiments and sent for PGM and chromite analysis to an external laboratory where the fire assay analysis was done to determine the PGM content and an inductively coupled plasma- mass spectroscopy analysis was used to determine the % chromite.

Base case experiments showed that the overall recovery of platinum in the secondary rougher/scavenger, using standard hand scraping, was 71%, with a cumulative chromite content of 6.33%. The wet mass of concentrate was controlled, by weighing the concentrate. Promising results were obtained for tests with a nearly horizontal longer baffle and the diesel and frother replacement for the standard frother. This combination had an overall PGM recovery of 82 % and the chromite content was reduced to 4.18%. The reduced frother dosage (10 g/ton as compared to the standard 20 g/ton dosage) showed promising results with a 77.5% PGM recovery and 4.10% chromite content. Tracer tests showed that froth washing had potential, but the desired reduction of chromite did not take place, due to dilution of the froth.

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## **NOMENCLATURE**

<i><b>Symbol</b></i>	<i><b>Description</b></i>	<i><b>Units</b></i>
C(t)	Concentration of liquid in the flotation cell	$\text{g.L}^{-1}$
C <sub>o</sub>	Concentration of feed salt water solution	$\text{g.L}^{-1}$
Ø	Outer diameter	cm
S(t)	Concentration of liquid in the flotation concentrate	$\text{g.L}^{-1}$
η	Efficiency	Dimensionless
σ	Conductivity	mS
φ	Air flow rate per surface area	$\text{m.min}^{-1}$
Au	Gold	-

<i><b>Abbreviation</b></i>	<i><b>Description</b></i>
BIC	Bushveld Igneous Complex
CMC	Carboxymethyl cellulose
Cr <sub>2</sub> O <sub>3</sub>	Chromite (gangue material in PGM flotation)
D and F	Diesel and frother
HG	Head grade
ICP MS	Inductively coupled plasma mass spectroscopy
MF1	Single stage mill float configuration
MF2	Staged mill-float-mill-float circuit
P and F	Paraffin and frother
PAX	Potassium Amyl Xanthate
PGE's	Platinum group elements
PGMs	Platinum group metals
RF	Reduced frother
SIBX	Sodium Isobutyl Xanthate
SIPX	Sodium Isopropyl Xanthate
SNPX	Sodium Normal-propyl Xanthate
UG-2	Upper group-2

# CHAPTER 1: INTRODUCTION

## 1.1. Background to study

The Bushveld Igneous Complex (BIC) in South Africa is renowned for its vast platinum group metal (PGM) reserves and most of the world's mined PGMs come from this source. Platinum, as well as the other five group elements i.e. palladium, rhodium, iridium, osmium and ruthenium, constitute a major source of revenue for South Africa. Mining and processing of the UG-2 reef of the Bushveld Igneous Complex has increased dramatically in recent years, due to depletion of the Merensky reef. Existing PGM concentration circuits that were initially designed for Merensky ores are now being used for the concentration of PGMs from UG-2 ore. The UG-2 is a chromitite reef and hence the mined material (ore) has much higher chromite content than the Merensky ore. Chromite is not floatable, but due to the large proportion of chromite in the ore, some of the chromite is recovered in the concentrate. The high chromite content of the concentrate reduces the efficiency of the subsequent smelting, which puts a strain on energy requirements. Consequently penalties have been imposed on the upper limit of chromite content in the smelter. Hence methods of reducing the chromite content in the flotation concentrate (on route to the smelter) are needed.

Froth flotation is the main separation process in the concentration of the PGMs from UG-2 ore. In flotation, particles may be recovered by two main mechanisms, namely (i) direct attachment to air bubbles and (ii) hydraulic entrainment of the particles in the water of the froth (McKeon, 2001). Direct attachment is a selective process whereby particles attach to air bubbles depending on the surface hydrophobicity of the particle. This is also known as 'true flotation'. Hydraulic entrainment is a non-selective process whereby hydrophilic gangue particles enter the flotation froth by being pulled upwards in the liquid around the air bubbles due to the upward force created around the bubbles as they rise. Recovery of particles by hydraulic entrainment is usually unfavourable as it mainly transports hydrophilic material, which is gangue material, to the froth phase. Due to the large quantity of chromite present in the UG-2 reef, increased amounts report unfavourably via entrainment to the flotation concentrate.

It has been known that washing of the froth layer of the flotation cells causes a reduction in gangue transport to the concentrate. Froth washing has been investigated in the past on coal and sulphide ores (Kaya, 1989 and McKeon, 2001) in mechanical and column flotation cells respectively, some reduction in the entrainment of gangue has been achieved. Hence an investigation of froth washing on the scavenger stage of flotation of UG-2 platinum containing ores proves to be a viable area of investigation. Although the primary role of froth washing is a reduction of gangue, it may also have a potential to improve PGM recovery as well.

It is commonly found that research goes into optimising the rougher and cleaner stages of flotation. The scavenger stage of flotation, however, is a good place to test methods of reducing entrainment of chromite, but maintain, or improve the recovery of the PGMs, which are floating slowly at this stage. Froth washing has been applied in industry particularly in cleaner flotation stages. It was found here to significantly affect the residence time of the pulp (Kawatra, 2012). It was because of this reason that focus has shifted to the application of froth washing on the scavenger stages of flotation where the effect on residence time would be small and the entrainment of chromite is significant.

Not much research in the past has been done on froth washing of UG-2 ore, particularly in the scavenger stage of flotation. Chemical and mineralogical analysis of the secondary scavenger tailings, by Deeplaul and Bryson (2004), indicated that a fairly significant portion of PGMs were not recovered by flotation and methods of improving this recovery may prove to be valuable. Optimisation of the reagent added in the scavenger stage of flotation has the potential to improve the recovery of PGMs since the scavenger stage is the place where the final trade-off between gangue recovery and PGM recovery is made. Hence this research was focussed on improving the performance of the scavenger stage of flotation.

The use of froth-directing baffles in the flotation industry is a relatively new concept. Baffles are known to direct the flow of the froth to the overflow lip and into the concentrate launders. It also functions to inhibit the upward flow of entrained liquid that may enter the froth phase with the bubbles (Kawatra and Eisele, 2001) and can be used to reduce gangue entrainment.

## **1.2. Dissertation objectives**

The aim of this project was to attempt to alleviate the problems, as explained above, which are currently being faced in the PGM flotation industry. This was done by investigating the effect of froth washing on the reduction in chromite recovery and also investigating the effect of substituting the standard frother dosage for: (i) different frother dosages and (ii) various other chemical combinations, on the recovery of PGMs. Both these investigations aimed to optimise the scavenger stage of flotation.



## CHAPTER 2: LITERATURE REVIEW

### 2.1. Overview

The Bushveld Igneous Complex, located in South Africa, is renowned for its vast platinum group element reserves. The platinum bearing ores are found in layers, or reefs, as they are more commonly known. A number of processes are required to concentrate and finally separate the PGMs. Flotation is the main process for separating the valuable minerals from unwanted material, commonly known as gangue material, and it has been used successfully in the past to extract the PGMs from the platinum bearing ore bodies.

The Merensky reef was exploited initially for PGMs but since the 1980's the UG-2 reef has been exploited for valuable PGMs (Impala Platinum Limited, 2011). UG-2 ore differs from Merensky ore by having significantly higher chromite contents. This eventually led to larger amounts of chromite intruding into the flotation concentrates (via entrainment, as discussed in section 2.5.1) which causes problems in the subsequent smelter operations. Since exploitation of the UG-2 reef, large amounts of chromite have been known to coat and solidify on the walls of smelters thereby causing a reduction in efficiency and wastage of energy in the smelters. As a result, smelters have set limits on the amount of chromite allowed in the concentrate (Valenta and Mapheto, 2011). The UG-2 reef is a chromitite (rock type) and most of it chromite ( $\text{Cr}_2\text{O}_3$ ). Consequently, concentrators have been forced to re-examine their current operations and seek out opportunities for the reduction of chromite in the concentrate.

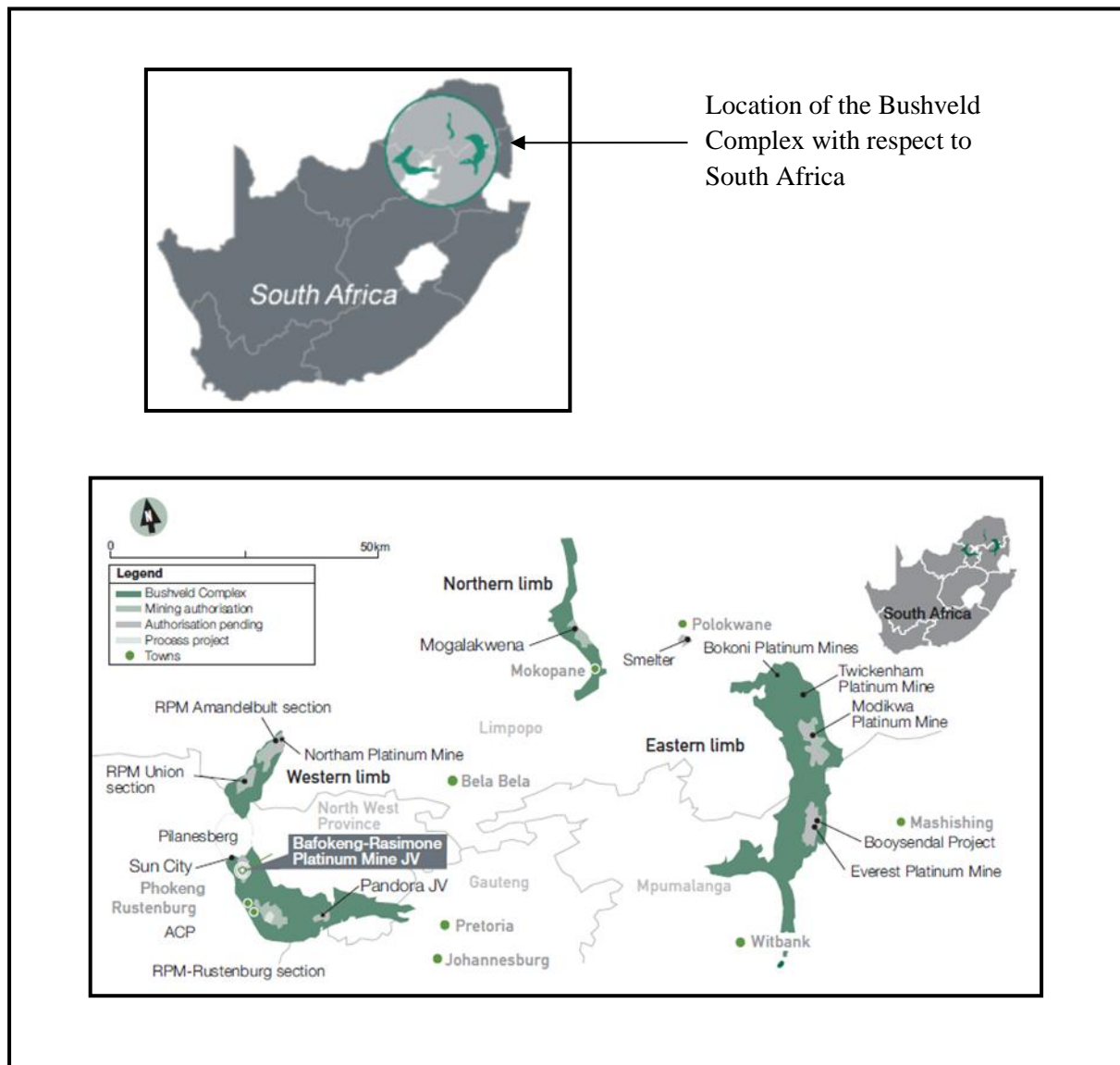
Froth washing in the flotation stage of the PGM recovery process provides a potential avenue for reducing contamination by chromite and its impact on the concentrate grade and recovery. Froth washing is known to remove the gangue material from the froth and it has been scarcely studied in the past (Kaya, 1989; McKeon, 2001). Mineralogical studies (Nel et al, 2004; Bryson, 2004) on the tailings of the existing flotation circuits have indicated that a significant amount of PGMs are not being recovered by flotation and means of improving this recovery needs to be sought. Methods of improving the recovery in the scavenger stage include the optimisation of the reagents, as well as considering the use of diesel and paraffin mixed with the frother. Baffles also provide a relatively new avenue on which to investigate the PGM recovery.

This investigation aims to explore all aspects relating to: gangue rejection via froth washing, the use of the baffle and its effect on PGM and  $\text{Cr}_2\text{O}_3$  recovery, and it also will provide insight into the effect of optimising the reagent suite on flotation cell performance. The literature review uses the following sequence:

- Mineralogy of the UG-2 ore in the Bushveld Igneous Complex: This section is relevant to facilitate the understanding of the variability and complexity of the ore in terms of its composition, which in turn complicates the processes that concentrate platinum group metals.
- Froth flotation in general: The basic but essential understanding of flotation is presented in this section. This section provides the fundamental knowledge on which flotation is based.
- Flotation with respect to UG-2 ore: Flotation can be used to recover a variety of minerals depending on their origin. Due to the complexity of especially UG-2 ore, (its variability) it's processing is complex and requires the mill-float / mill-float (MF2) circuit for its processing. The major problem of UG-2 ore (as compared to its predecessor, Merensky ore) is the abundance of chromite. How the chromite intrudes into the flotation concentrate and the further problems it creates for downstream processes will be discussed.
- Froth effects: This section will detail past documented work on experiments carried out in the froth phase of flotation and variables investigated to improve the recovery and grade of the valuable minerals.

## 2.2. The Bushveld Igneous Complex

The Bushveld Igneous Complex (BIC), also known as the Bushveld Complex, in South Africa was formed approximately 2 billion years ago and is well known for its large reserves of platinum bearing ores. It is the largest layered mafic intrusion in the world (McCandless and Ruiz, 1991). The saucer shaped geological intrusion of platinum bearing ore spans the area along the Northern parts of South Africa in the North-West, Limpopo and Mpumalanga Provinces and it covers an irregular oval area of approximately 66 000 square kilometres (Impala Platinum Limited, 2011). The location of the BIC with respect to the rest of South Africa can be seen in the figure below.

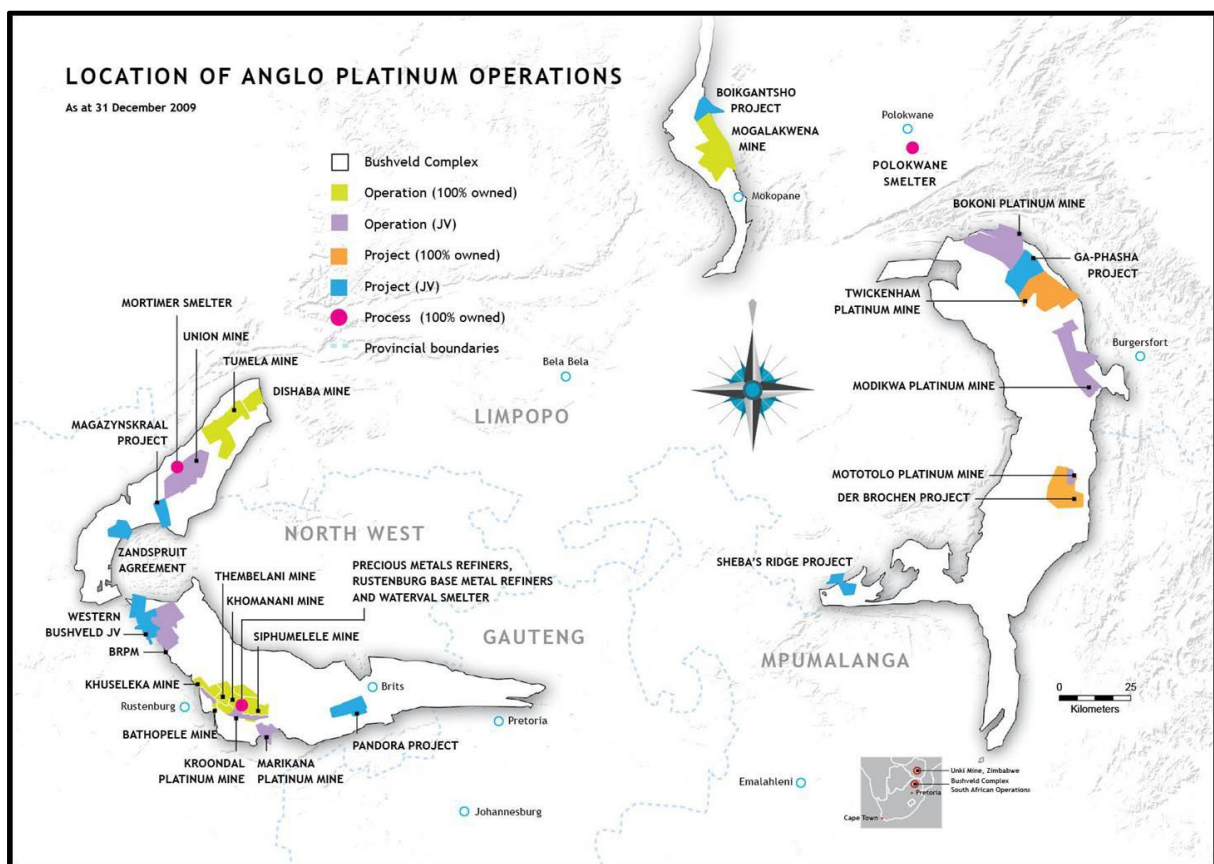


***Figure 2-1: Location of the Bushveld Igneous complex with respect to South Africa (top) and a more descriptive inset of the complex (bottom) (taken from Royal Bafokeng Platinum, 2010).***

### 2.2.1. Geology of the ore body

The Bushveld Igneous Complex is the largest source of the platinum group metals, being platinum, palladium, ruthenium, rhodium, iridium and osmium. The six PGMs as well as gold and silver are classified as precious metals. The complex also contains vast reserves of tin, chromium, titanium and vanadium and the base metals cobalt, copper and nickel (Dube, 2010). Volcanic eruptions in the past have caused these metals to emerge from within the earth's mantle and be brought to the surface of the earth's crust. Cooling and solidifying of the different minerals at different temperatures resulted in the formation of the layer-like structure of the complex with three main PGM bearing reefs. The reefs are: the Merensky reef, Platreef and the Upper group-2 reef (UG-2).

The mining of platinum first began in 1925 and throughout the years platinum reserves in South Africa have been exploited by many PGM producers <sup>[1]</sup> for extraction of valuable PGMs (Djadji Platinum, 2007). Anglo American is the largest platinum producer in South Africa and the location of their mines can be seen in Figure 2-2.



**Figure 2-2: Location of Anglo American (largest PGM producer in South Africa) mines and smelters in the Bushveld Igneous Complex in South Africa. (taken from Hundermark et al, 2011).**

[1] – Anglo American, Implats, Lonmin, Northam Platinum Limited.

Platinum occurs in the Bushveld Igneous Complex at a relative abundance of approximately 4-7 grams per ton of ore mined (Chamber of mines of South Africa, 2008). The variance in the abundance is due to geological complexity. Recovery of a PGM concentrate through the flotation of UG-2 ore has certain specific challenges due to the intrinsic variations in the mineral composition of the ore of the BIC. Variability in the ore, together with upstream process dynamics, creates a challenging environment in which to achieve optimal recovery of a flotation concentrate with accurate grade control (Dawson, 2010). Variations in the ore imply the variation of the physical, metallurgical and surface properties of the ore and these variations result in significant changes in the flotation circuit. For example, a simple variation in the hardness of the ore results in a variation of particle sizes in the grinding operations which will cause further variations in the flotation performance even under constant operating conditions (Meech, 2012). Nevertheless, through all the complexities, PGMs have been collected by flotation and concentrate recoveries of up to 95 % have been achieved (Tatarnikov et al, 2004).

The Bushveld Igneous Complex has the world's largest platinum ore bodies concentrated in its reefs. Statistics South Africa (2011) reported that in 2008 alone, 276 tons (imperial) of platinum group minerals were produced (extracted from ore) and at the current extraction rate, the complex still has another 254 years before the PGM reserves are depleted.

As previously mentioned, the main PGM bearing reefs are the Merensky <sup>[2]</sup> and the UG-2 reef's that are present in the eastern and western areas (or 'limbs') of the Bushveld Igneous Complex. The Merensky and the UG-2 reefs are confined in the Rustenburg layered suite <sup>[3]</sup>, which is composed of a well-layered ultramafic to mafic igneous succession (Cousins, 1959). The UG-2 reef lies below the Merensky reef and hence the same shaft can be used for mining both reefs. The vertical parting between the Merensky and UG-2 reefs changes from about 125 meters in the South and to about 45 meters in the North (Impala Platinum Limited, 2011). The UG-2 and Merensky ores respond differently to grinding and flotation and hence a new process flowsheet was required for UG-2 ore. The UG-2 ore has a considerably lower concentration of base metal sulphides (nickel and copper), but it contains much more chromite than the ore of the Merensky reef. The Platreef is similar to Merensky ore in terms of its metallurgy, but it is enriched in palladium (Jones, 1999). The Platreef is located in the northern portion of the Bushveld Complex and recovery of this reef is now being done by open pit mining.

[2] – Named after Hans Merensky, the first person to discover Platinum deposit in the Bushveld Igneous Complex.

[3] – The Rustenburg layered suite of the BIC can be divided vertically into five zones: the Marginal zone, the Lower zone, the Critical zone, the Main zone and the Upper zone (Dube, 2010).

Components of UG-2 ore include chromite, plagioclase, orthopyroxene, minor aggregates of phlogopite and talc chlorite, along with lesser quantities of base metal sulphides and platinum group metals. The base metal sulphides are largely chalcopyrite, pentlandite, pyrite, pyrrhotite, and minor amounts of millerite. Ore from the Merensky reef contains more base metal sulphides than that from UG-2 reef (Jones, 1999). The UG-2 ore comprises of a chromite matrix and its high chromite content poses significant problems when processing the ore. The PGM content of UG-2 ore (4.4 – 10.6 g/ton) is higher than that of the ore from the Merensky reef (4 – 10 g/ton). An interesting aspect of UG-2 ore is that it has an abundance of rhodium which is the most expensive precious metal and it is also a vital constituent in motor vehicle exhaust system. Palladium, a metal which is high in demand, also creates a market for the processing of UG-2 ore.

Despite the relatively small amounts of base metal (copper, cobalt and nickel) sulphides, they are of importance since they occur in association with the PGMs. The head grade of copper can be anything from 0.005% to 0.02%, and the head grade of nickel varies between 0.025% and 0.05%. PGMs are found as various minerals, for example metal alloys, oxides, sulphides, tellurides, and many others. The grains of the platinum group metals in these alloys are tiny and seldom exceed 30 µm with the average grain size being as fine as 6 µm. These fine sizes and the complex mineralogy pose mineral liberation and extraction challenges (Valenta, 2007).

The grain size of PGM minerals in the Merensky reef is approximately 45 µm and the grain size of PGMs in the UG-2 reef is around 15 µm (Jones, 2005). Therefore, for liberation of the fine PGMs in UG-2 ore, the ore requires milling to a finer particle size. Some of the fine PGMs occur locked in gangue material and this is known as a composite particle. PGMs locked inside base metal sulphides can be recovered by flotation, but if they are locked in gangue minerals, they may be lost. The effect of composite particles on flotation will be discussed in section 2.5.11. Due to the finer grain sizes of the PGMs, the UG-2 ore requires milling to a finer size in order to liberate these fines and to become more susceptible to the flotation process. While the fine grains of PGMs are liberated by grinding, the chromite that is present in the ore is also ground to a fine size. This creates a major problem in the flotation process, as will be described later.

Although the UG-2 ore has a higher chromite content, it has some benefits over Merensky ore: For a given mass of feed into the flotation cells, the mass of the flotation concentrate obtained using UG-2 ore is lower than the mass obtained using Merensky. Consequently, the UG-2 concentrate has a higher grade and a smaller throughput into the smelter. UG-2 ore has lower crushing costs due to it being more friable (Jones, 2005). Smelting costs are also theoretically lower due to a smaller amount of concentrate being smelted for a fixed amount of platinum produced. Hence theoretically, the total treatment cost for UG-2 ore should be lower than the cost to treat ore of the Merensky reef (Corrans et al, 1982). Unfortunately, the reality indicates that the smelting costs have been increasing due to the large amount of chromite in the flotation concentrates (discussed in section 2.10).

The PGM output of South Africa is derived almost entirely from the Bushveld Igneous Complex (Jones, 1999). PGMs are South Africa's largest mining export earner, accounting for approximately a third of all sales in 2004. Revenue from PGMs even exceeded that of gold as the platinum industry began gaining more importance. Recently, however, the price of gold has risen above that of platinum, due to uncertainty in the world economy (Jones, 2005).

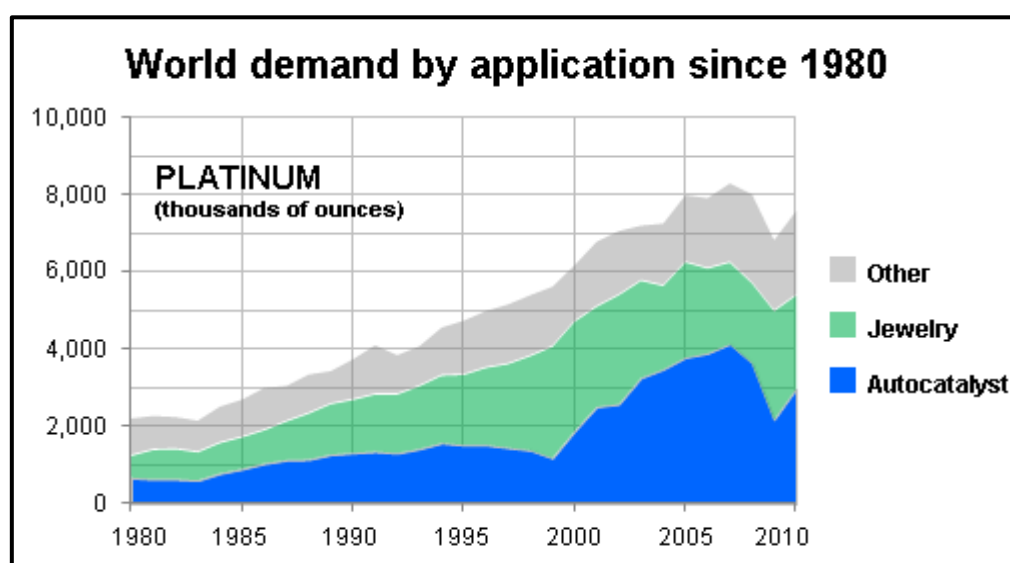
UG-2 reef is a rock type called chromitite, because its major constituent is chromite ( $\text{Cr}_2\text{O}_3$ )<sup>[4]</sup>. The two main phases in UG-2 ore are a high density chromite rich phase and a lower density silicate rich phase. Most of the PGM minerals are found within the chromite rich parts of the UG-2 reef. The PGMs associated with the high density chromite phase occur alongside the grain boundaries and they are easily liberated when crushed due to the friable characteristics of the chromite grains. PGMs that occur with the lower density silicate phase are intergrown in the silicate grains and are frequently locked in this phase. The silicate rich fractions in the reef contain a lower quantity of PGMs and consequently have a low grade. The importance of fine grinding to liberate the locked and intergranular PGMs is depicted by this finding (Nel et al, 2004).

[4]- Chromite is a mineral with an approximate formula  $\text{Cr}_2\text{O}_3$ . Chromitite is a rock type and in this case it was formed by slow crystallisation and sedimentation of chromite crystals in molten rock, to form a layer rich in chromite. The liquid between the crystals solidified later.

The Merensky and UG-2 reefs are situated in close proximity to each other. Platinum mining usually occurs at an average depth of 800 meters with the reefs varying in thickness and depth. The width of the Merensky reef is usually 1 meter thick whilst the width of the UG-2 reef is in the region of 0.9 meters (Impala Platinum Limited, 2011). A typical characteristic of the Merensky reef is its high grade of platinum group elements (PGE's) and high ratios of platinum to PGE's. The UG-2 reef differs from Merensky reef by being rich in its chromite content (making it a low grade layer), although it's platinum group metal reserve may be nearly twice as large as that of the Merensky reef (Chamber of mines of South Africa, 2008). Merensky ore has a chromite content of approximately 0.1% as opposed to UG-2 ore with a chromite content of around 30% (Jones, 2005).

### 2.2.2. Importance of Platinum

Platinum is one of the rarest metals found on the earth's crust (Mineral Information Institute, 2008). Approximately 80% of the world supply of platinum arises from South Africa (Maksimainen et al, 2010). Platinum is extremely dense, highly malleable and ductile. Other favourable properties of platinum are that it is extremely resistant to oxidation and corrosion at high temperatures and it is a very good conductor of electricity. Its main industrial use is as a catalyst, notably for oxidising unburnt car emissions (Implats, 2005). Platinum has significant economic importance for the country and the demand for platinum has been steadily increasing due to platinum finding increased use in various applications as can be seen in Figure 2-3. A few uses of platinum include its role in catalytic converters, laboratory equipment, wireless and electrical contacts and electrodes, thermocouple devices, medical equipment and jewellery (Wikipedia, 2012).



**Figure 2-3: Annual growth in platinum demand from the year 1980 through to 2010 (adapted from Matthey, 2012).**



The rise in the demand for platinum in the automotive industry has exceeded the fall in its demand for jewellery. The demand for platinum catalysts in the automotive industry is expected to increase by 25% due to the influence of three main factors (Robinson, 2002):

- ✱ The growth in the diesel market which uses catalysts which are made up of entirely platinum (instead of gasoline automobiles that utilise Pt and/or Pd in combination with Rh).
- ✱ Higher loadings on catalysts in diesel automobiles.
- ✱ A change in catalysts for gasoline vehicles from palladium-based catalysts to platinum based catalysts.

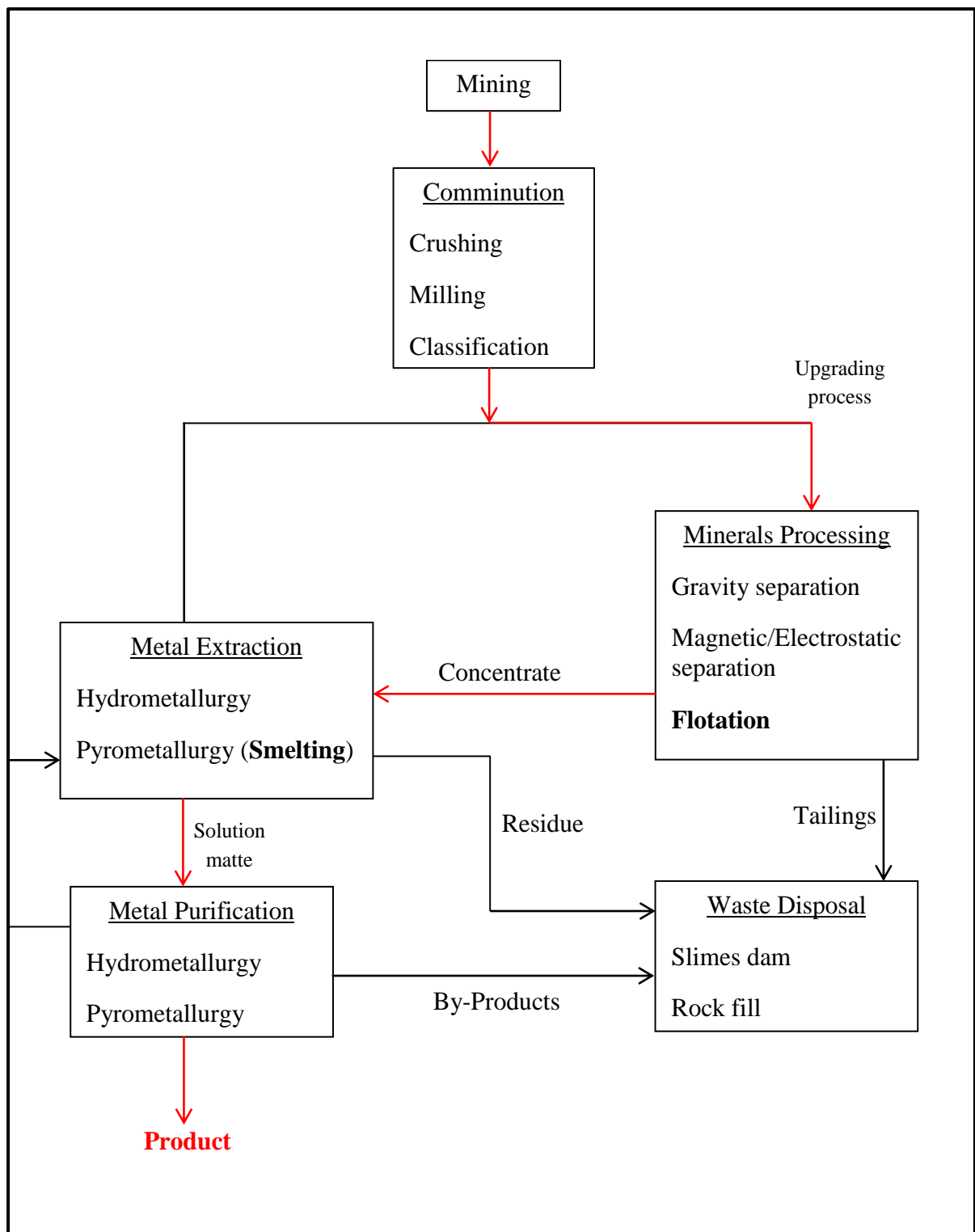
This demand has sparked the increase in production of platinum by the platinum producing companies. This has put pressure on the companies to improve their existing process operations.

### **2.2.3. Extraction of PGE's from the ore**

The PGE's from the relatively narrow reefs are extracted using various drilling and blasting techniques. The ore is ground to a size that will expose the valuable minerals or host minerals such as nickel sulphide, so that they can be concentrated. Many processes exist that separate and concentrate the valuable material from the waste material, including gravity concentration, flotation, magnetic and electrostatic separation (U.S Environmental Protection Agency, 1994). This investigation focuses mainly on the process of flotation for the beneficiation of ore.

UG-2 ore, due to its complex mineralogy, usually requires two stages of crushing and flotation. The first stage recovers the (coarser) valuable minerals that have been liberated by primary grinding. The second stage of grinding helps to liberate the (finer) unrecovered valuable minerals trapped within composite particles so that they can be amenable to flotation.

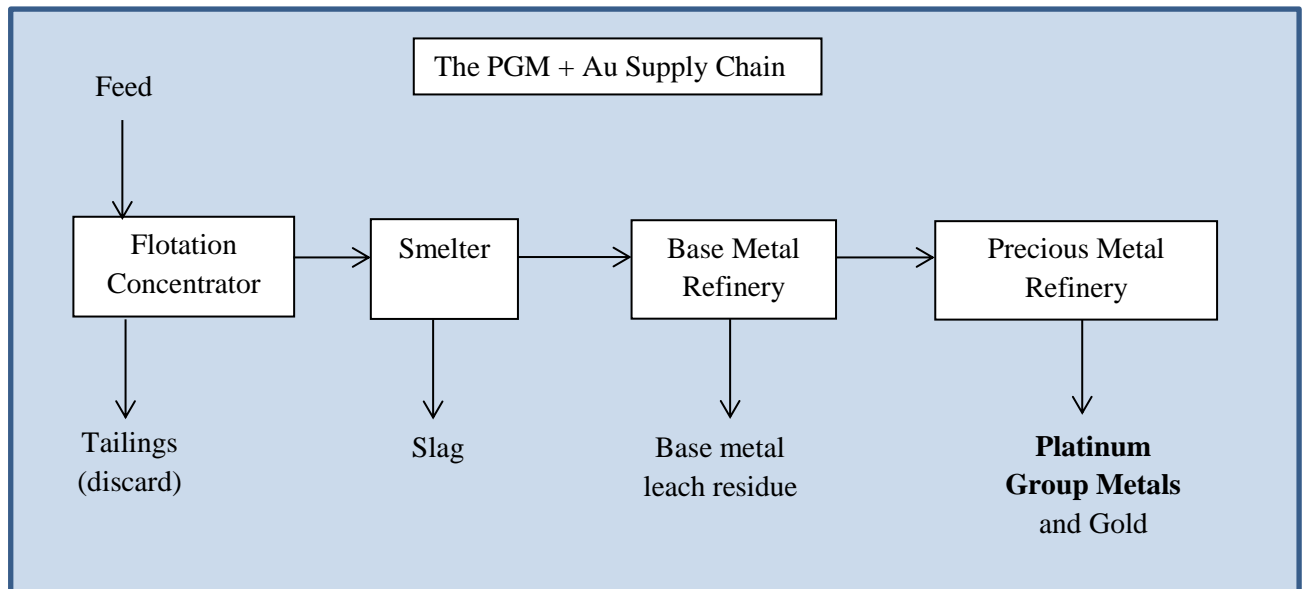
Recovery of the valuable PGE's from the ore takes place mainly through extraction, concentration and refining. Extraction is the labour-intensive process that involves drilling and blasting of the ore with explosives. The ore is then removed and transported to the surface for concentration of the valuables. Froth flotation is the main process for concentrating PGMs. The ore from the extraction step undergoes a series of crushing and milling (comminution) operations to obtain particle sizes that are suitable for froth flotation. The crushing and grinding also help to liberate valuable PGE's that may be trapped within larger sized particles. A simplified diagram of the operations involved in extraction of the PGE's can be seen in Figure 2-4.



*Figure 2-4: Flow diagram of the processes involved in extracting precious metals from the ore (adapted from Loveday, 2006).*

### 2.3. Process of precious metal extraction

The ore has to undergo many processes before it becomes fully refined to obtain the final products. A basic flow diagram of the PGM + Au supply chain can be seen in the PGM (+ Au) process flow sheet.

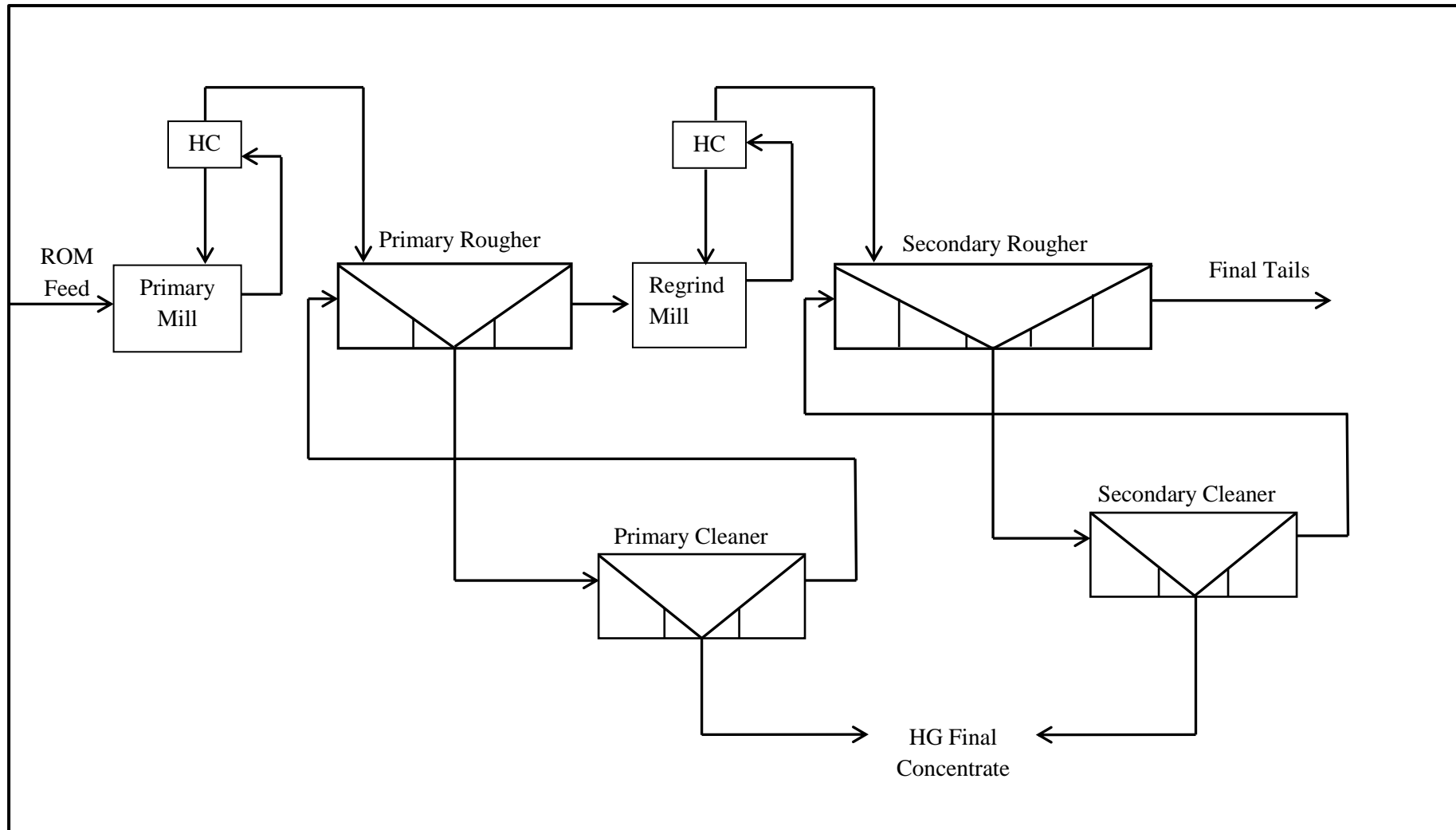


**Figure 2-5: Simplified process flow sheet for PGMs and gold (adapted from Valenta and Mapheto, 2011).**

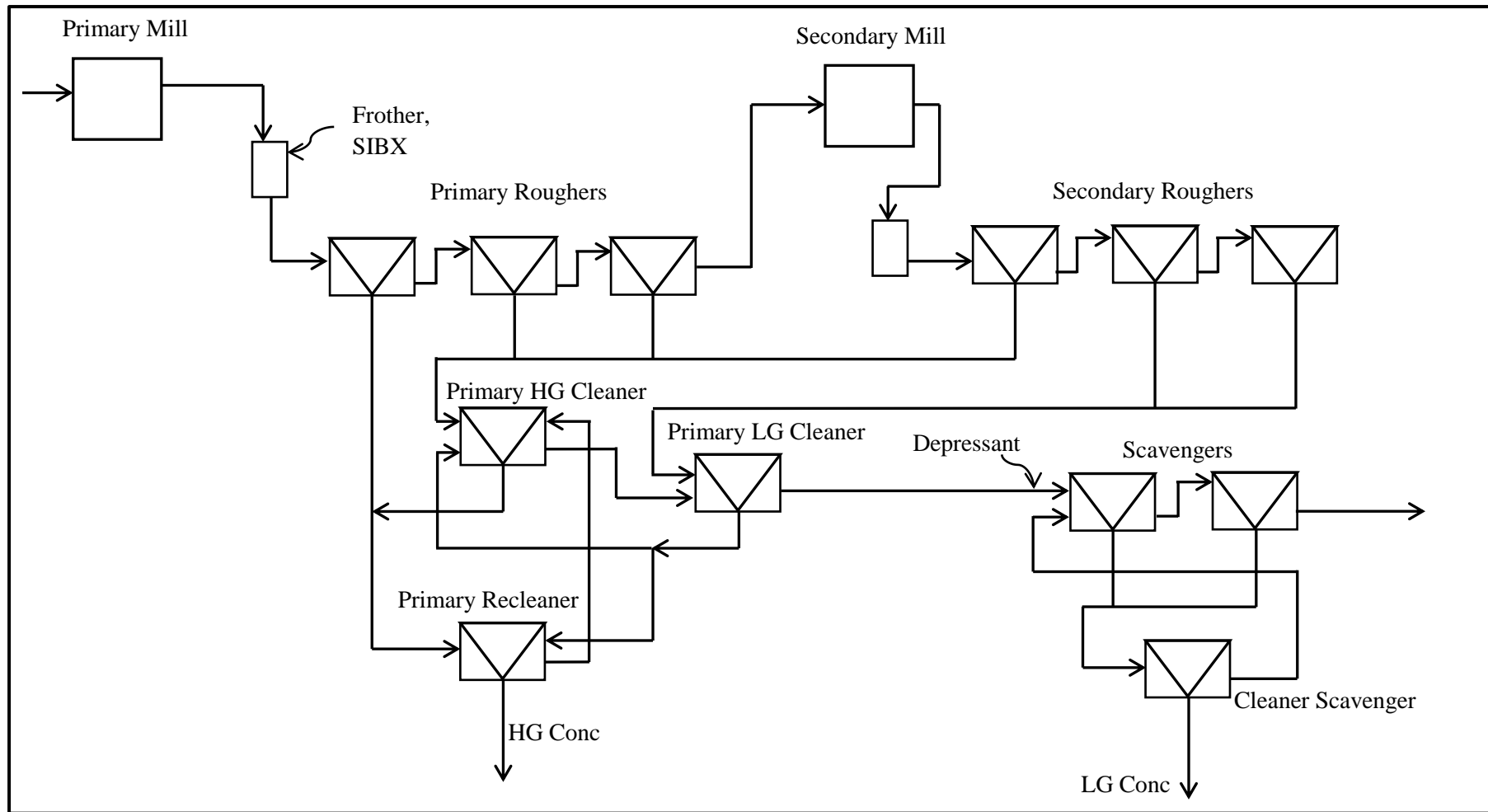
As can be seen, the final products are the base metals (copper, cobalt and nickel) and the precious metals (PGMs and gold). Of the many concentration stages, flotation is the main process for separating valuable minerals from the ore.

## **2.4. Background of flotation of UG-2 ore**

The flotation circuit that forms the basis of most UG-2 concentrators is illustrated in Figure 2-6. This is commonly referred to as the MF2 (mill-float-mill-float) circuit and it is commonly used in the processing of UG-2 ore. Previously, the MF1 circuit was used, but due to the large amount of PGMs and base metals found in the tailings end of the circuit, it was decided that a second (and sometimes a third) stage of grinding and flotation is needed. The problem of MF1 was that the primary milling rendered coarse particles that reduced chrome entrainment but also limited the PGM recovery into the final concentrate. The MF1 was unable to recover the finer sized PGMs that were associated with silicates (Nel et al, 2004). In around 1994 the recoveries of PGMs were improved by setting up a regrind mill and a secondary rougher/cleaner flotation configuration. The resulting system was then known as MF2 (Figures 2-6 and 2-7). The second stage of grinding aims to liberate smaller PGM grains that are possibly attached to gangue material. The improved fineness of the particle size provided further liberation of the locked PGMs but also caused a problem of fine chromite entrainment into the flotation concentrate. The increased recovery of chromite in the flotation concentrates causes downstream furnace problems as previously mentioned (Nel et al, 2004). Variations to the MF2 circuits depend on the individual flotation plant requirements. Another type of MF2 flotation circuit can be seen in Figure 2-7.



*Figure 2-6: MF2 circuit at Impala's UG-2 plant (Nel et al, 2004).*



***Figure 2-7: Mintek's MF2 process (Deeplaul and Bryson, 2004).***

The entire flotation process (the selective attachment of valuable particles to air bubbles takes place in the presence of a much larger amount of gangue). The base metals and platinum group metals are present at very low concentration in the ore. However, recoveries of 90% and above have been achieved, which encourages the flotation of these minerals and research into improving their recovery.

Each processing stage in the process of precious metal extraction is designed to increase the concentration of PGMs. Flotation increases the concentration of base metals and PGMs at a ratio of 30:1 and the smelter increases concentration by a further 10:1. In the converter, an improvement in concentration of 3:1 is possible and the base metals refinery increases the concentration two hundred times (200:1). The costs for each processing stage are distributed as follows: 72% for mining; 10% for concentrating; 9% for smelting and 9% for refining (Jones, 1999). The following table shows how the process of flotation increases the concentration of PGMs from 4-7 g/ton (Chamber of mines of South Africa, 2008) in the natural ore to the values as shown in Table 2-1.

**Table 2-1: Concentrate analyses (adapted from Jones, 1999).**

<b>Plant</b>	<b>PGM (g/ton)</b>
Amplats: Waterval	143
Amplats: Union	142
Impala	138
Lonmin: Merensky	130
Lonmin: UG-2 blend	340
Northam	132

PGM grades are over 100 g/ton for Merensky concentrates and above 300g/ton for UG-2 concentrates. Blending of the ore may take place to alter grade of the concentrate (Jones, 1999).

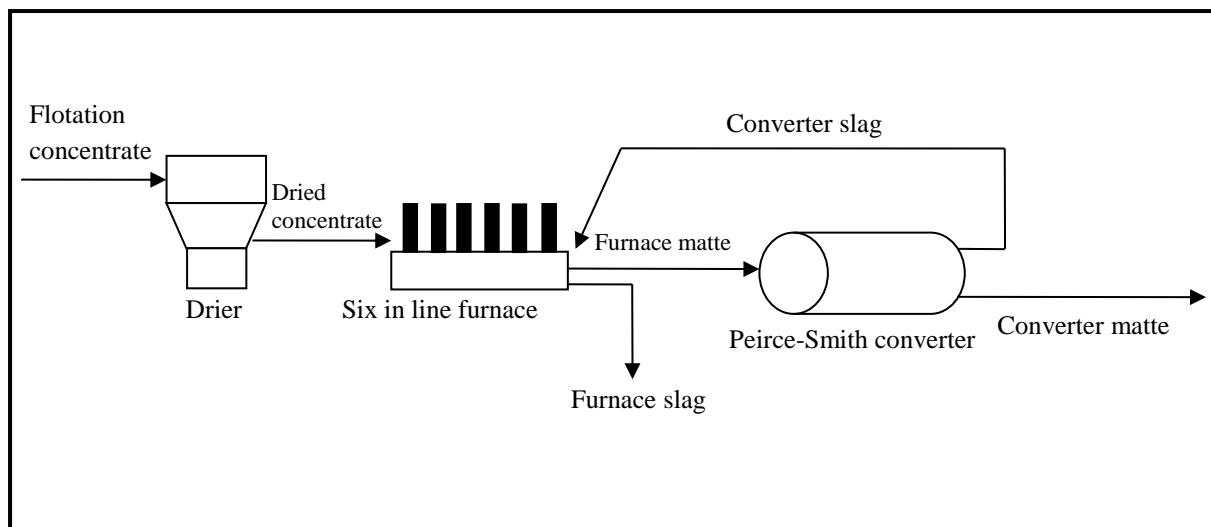
The flotation concentrates undergo conversion and smelting in a furnace, to produce a PGM containing nickel-copper matte. Smelting further increases the concentration of PGMs as can be seen in Table 2-2.

**Table 2-2: Furnace matte analyses in various smelters (Jones, 1999).**

Plant	PGM (g/ton)
Amplats: Waterval	640
Amplats: Union	830
Impala	1050
Lonmin: Merensky	1000
Lonmin: UG-2 blend	2500
Northam	724

The matte is treated via hydrometallurgical means to separate the base metals (Cu, Ni, Co) from the precious metals (PGMs). Finally the PGM concentrate is refined to separate the individual precious metals into their pure forms. PGM recovery is typically about 85% in the concentration stage, 95% to 98% in smelting, and 99% in refining. The largest loss of precious metals occurs during the crushing, milling and flotation operations (Jones, 2005).

In the concentration circuit, after flotation, the concentrate is sent to a spray (or flash) drier (as can be seen in Figure 2-8). The purpose of the intermediate drier is to reduce the energy requirement of the smelter and also to reduce the incidence of explosions in the smelter. The output contents of the drier are transported pneumatically to the smelter (Jones, 1999).



**Figure 2-8: A typical platinum smelting process (after Jones, 1999).**





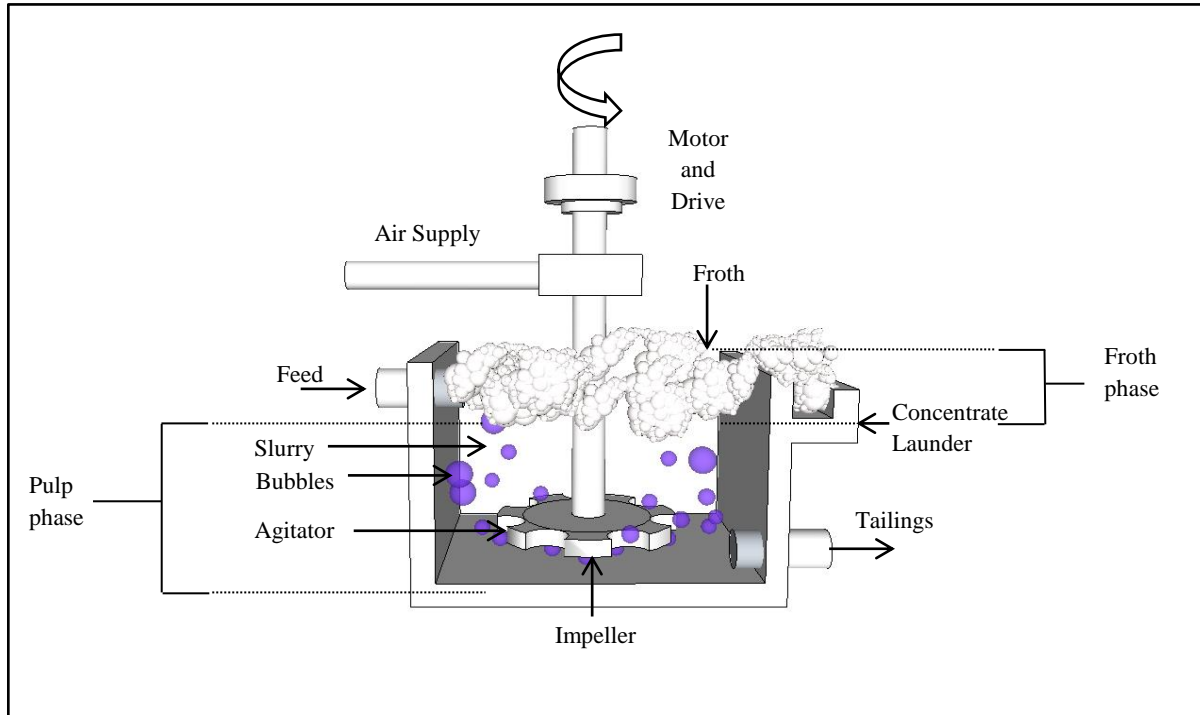
**Figure 2-9: Exit of smelter containing the PGM + Au and Ni–Cu matte. The matte then goes for further refining to separate precious metals from base metals. The base metals are removed by standard electrolyte techniques (Anglo American, 2008).**

The aim of smelting is to separate the oxide and silicate gangue minerals from the precious minerals. The output of the smelter consists of two phases – the matte and the slag. The matte is a dense phase, rich in base metal sulphides. The PGMs occur in association with these sulphides. The matte is separated from the slag and sent for further treatment. The slag is the lighter of the two phases and is silicate and iron enriched. The slag phase contains mostly gangue and is discarded (Jones, 1999).

Smelting of UG-2 concentrates occur most commonly in submerged arc electric furnaces at temperatures of around 1600 °C or higher. Due to the low concentration of valuable minerals in the flotation concentrate, the smelters operate at high slag to matte ratios. The slag and matte phases are tapped separately from the smelter. The matte, containing nickel, copper, cobalt iron, sulphur and PGMs, are tapped into ladles and transported to a converter (Jones, 1999). Air is injected into the converter to oxidise most of the iron and sulphur. The resulting sulphur dioxide is converted to sulphuric acid, and the iron oxide slag is added back to the smelter, where the iron finally leaves with the discarded slag. The high temperature in the smelter leads to energy loss by radiation. It is common practice to continuously add a layer of concentrate to the top of the submerged molten phases in smelters, to reduce the energy loss by radiation to the roof and walls (Jones, 1999).

## 2.5. Flotation in general

Froth flotation was introduced early in the 20<sup>th</sup> century and it is considered to be the most widely used method for the beneficiation of ore (Ata, 2011). Beneficiation is the separation of valuable material from the waste material. This waste material in flotation is termed as gangue material. A froth flotation cell is shown in Figure 2-10.



**Figure 2-10: A mechanically stirred flotation cell showing feed entry and concentrate and tailings exit points (Google SketchUp 8).**

Two main types of froth flotation cells are used to achieve flotation of ore i.e. column cells and mechanical cells. Emphasis in this investigation will be placed on the mechanical flotation cell due to its use in plants for rougher and scavenger flotation (to be explained later) and the laboratory device used in this investigation. Mechanical flotation cells are the most widely used type of flotation cells used in flotation plants and several single cells are mounted together to form a bank of flotation cells (Minerals Processing Techniques, 2004). The flotation process is carried out in a flotation cell fitted with an air supply and an impeller. The feed to the flotation cell is finely ground ore. Wet grinding of the ore is necessary to produce a suspension, called ‘pulp’. This suspension is then subjected to vigorous mixing via the impeller. Once in operation, air is pumped into the eye of the impeller and eddies behind the blades of the impeller break the air up into small bubbles, which are dispersed into the pulp (Minerals Processing Techniques, 2004).

The flotation cells are designed to keep the ore particles in suspension whilst the air bubbles are introduced. The hydrophobic (water- repelling) particles selectively attach to the air bubbles which then move up to the surface of the pulp layer where they form a froth layer. The froth is collected as it flows over the top of the flotation cell. This overflowing froth is known as the concentrate. In industrial flotation cells, the froth flows over the cell lip into launders which transport the froth to subsequent flotation stages, depending on the type of ore being floated. Frothing agents are added to control the bubble size by reducing the amount of bubble coalescence in the froth layer. The coalescence of bubbles become more probable as the liquid films that separate the bubbles become thinner (Ata, 2011). The froth flotation process may be used to recover both sulphide and non-sulphide minerals. PGE's are usually associated with the sulphide minerals and are hence recovered by the flotation process (Tatarnikov et al, 2004).

The recovery of valuable minerals via flotation is based almost entirely on probability, including the probability of particle-bubble contact; the probability of movement of the laden bubbles from the pulp phase to the froth phase and the probability of collecting the froth into the concentrate launders (Atkinson et al, 1994).

Separation of valuable minerals from gangue material through flotation is based on the differences in particle hydrophobicity. During flotation, the finely ground particles are suspended in the pulp and are made hydrophobic by the addition of surfactants (more specifically a collector chemical which will be discussed in section 2.5.5.1). Air is pumped into the pulp suspension at the bottom of the flotation cell, so that there is a sufficient path length for bubble / particle collisions to take place. The froth, which is enriched in valuable minerals overflows into the concentrate launders. Contemporary flotation tanks have a capacity of 100 to 300 m<sup>3</sup> depending on the requirements of the plant. Plants generally also have many flotation stages (about 3 or 4 depending on plant capacity) and each stage has about 6 to 10 flotation tanks. An increase in the number of stages and tanks improve the flotation separation efficiency and the purity of the final product, (up till a certain limit), but it depends largely on plant economics (Cilliers, 2006).

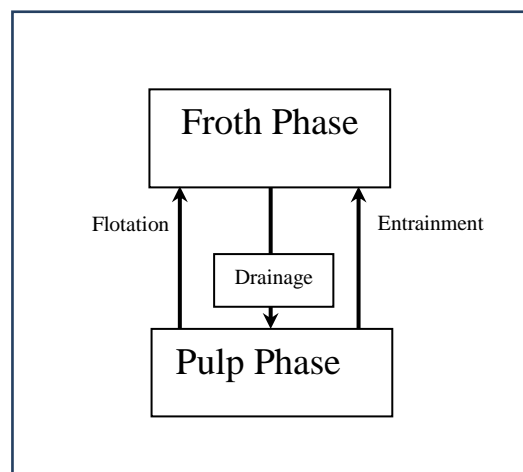
The froth phase occupies approximately 10% by volume of the flotation tank but it essentially governs the recoveries of the valuable minerals from the pulp. Much research has been done to enhance the characteristics of the froth phase in order to improve the recovery of the valuable minerals and rejection of gangue minerals. These enhancements include methods that make the froth phase more favourable for improved separation of valuable minerals from gangue material. Some of these enhancements include modification of the froth flow and the addition of water to the froth phase. Past work (Runge et al, 2010) also states that there is a potential to improve flotation performance through better understanding of the froth phase.

Theoretically, the process of froth flotation can be considered to be a system of two operations namely reaction and separation. The reaction process takes place in the pulp phase. The product of the reaction is a (hydrophobic) particle-bubble union. This combination is of lower density than the medium (the pulp) hence it moves up towards the froth phase. The hydrophilic particles remain in the pulp phase and report to the tailings. The separation occurs at the interface between the pulp and the froth and through transport into concentrate launders. As mentioned before, there are two ways in which particles may enter the froth phase: through particle-bubble combinations or through entrainment (Yianatos et al, 2008).

### 2.5.1. True flotation and Entrainment

Floated material that reports to the concentrate generally does so via two mechanisms: True flotation which depends on particle surface properties and entrainment which depends mainly on particle size and density (Steyn, 2011).

True flotation deals with the hydrophobic particles being attached to air bubbles and rising to the surface (with the aid of collectors). Entrainment deals with generally undesired materials intruding into the concentrate. Undesired material may be transported into the concentrate due to the sub processes occurring in the flotation cell (Figure 2-11):



**Figure 2-11: The various sub-processes of mineral transport in the flotation cell (Valenta, 1994).**

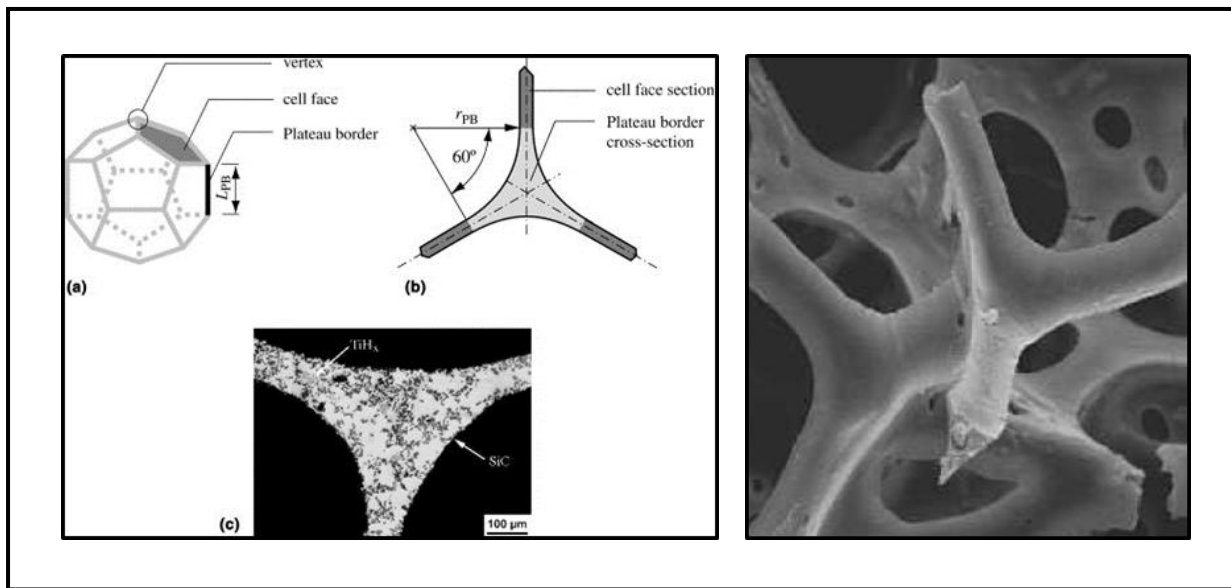
The flotation process as a whole can be broken down into sub processes. These sub process can be referred to as true flotation, entrainment and drainage. True flotation, as described previously, is the selective means of attachment of the hydrophobic particles to air bubbles which rise up into the froth phase to be recovered in the concentrate. Entrainment on the other hand is a non-selective process that causes the transfer of solids and water into the froth (Valenta and Mapheto, 2011). Figure 2-11 shows a two stage model consisting of the pulp phase (which is related to the collection of the valuable minerals) and the froth separation zone. Mineral-laden bubbles coalesce and pop in the froth and the surrounding water (and solids) and can drain back into the pulp. Drainage is also known as drop-back (Yianatos et al, 2008). As mentioned previously, the collection and separation are based on probability of particle-bubble contact.

### **2.5.2. Froth Phase**

The froth phase is the place where most of the valuable minerals are concentrated. The froth phase is formed once the air bubbles enter the space above the pulp (Breward, 1999). The froth phase appears on the surface of the flotation cell because the density of air bubbles being less than the density of the pulp. This froth that is formed is unstable and requires the addition of the frother for stabilisation. Once stable, the froth can be easily collected as the flotation concentrate in the launders. The stability of the froth phase is the focal feature of froth flotation since it determines the grade and recovery of the final product and the overall efficiency (Ata, 2011). Processes that refine the flotation concentrates, such as frother addition; froth washing or the addition of other chemicals, generally take place in the froth phase. The type of froth and frothing agent is significantly important in determining the success of the flotation process. A weak, brittle froth reduces the efficiency of the flotation process, whereas a robust, bulky froth, formed with the use of a strong frother, produces higher recoveries (Twidle et al, 1984). However, the froth must not be too stable, as entrainment of gangue occurs. A certain amount of ‘brittleness’ is required, to allow the gangue to drain (Kawatra, 2012).

### 2.5.2.1. Froth Structure

The structural components of the froth are the lamellae, the plateau borders and nodes (Figure 2-12). These structures only form in the higher reaches of the froth where the bubbles have lost most of their water, are polyhedral in shape and are closely packed. The bubble lamella is simply the film of the bubble, but because the polyhedral bubbles are closely packed, they resemble polyhedrons and the lamella corresponds to a ‘face’ of a side of the polyhedron bubble. Plateau borders are the formed when three lamella meet which is almost always at an angle of  $120^\circ$  since this is the most stable configuration as confirmed by Joseph Plateau<sup>[5]</sup>.

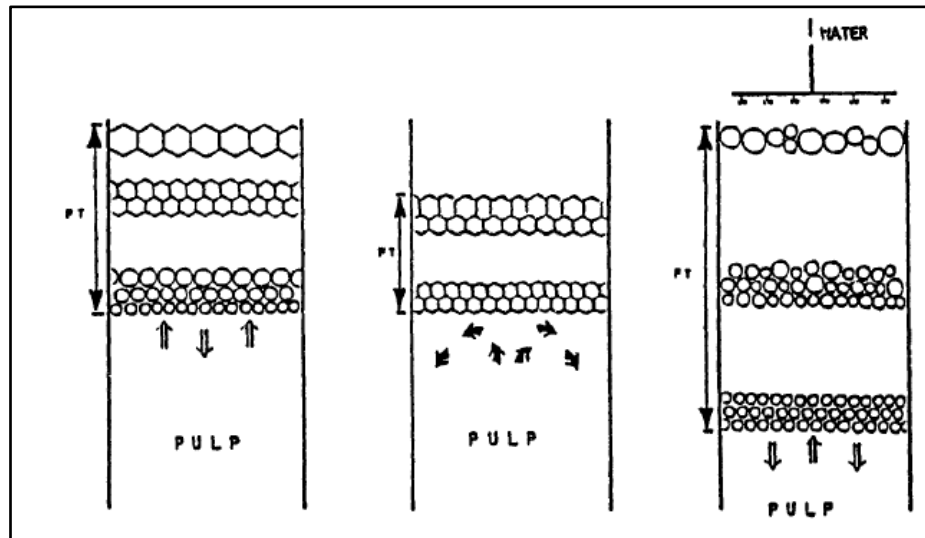


***Figure 2-12: Plateau borders in a foam (taken from Cilliers, 2006).***

Four plateau borders meet at a vertex at an angle of  $109.47^\circ$  (the tetrahedral angle). The plateau borders and vertices form a network of channels in the froth. Almost all the liquid in the froth is contained in the channels (Cilliers, 2006).

[5] – A Belgian physicist who formulated ‘Plateau’s laws’ that describe the structure of soap films.

Froth may be classified as two different types. These are wet froths (high liquid content with spherical bubbles) and dry froths (low liquid content with polyhedral bubbles). Froth structures of three different types (as can be seen in Figure 2-13) are encountered in flotation practice namely: conventional draining froth, polyhedral froth and washed froth (Kaya, 1989).

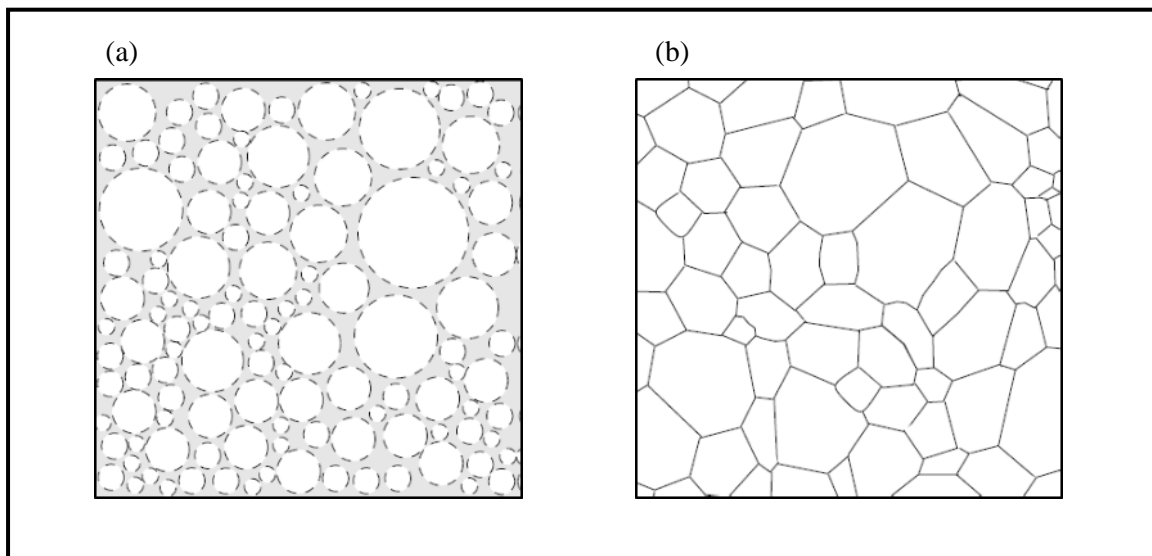


**Figure 2-13: The three froth structures encountered in flotation according to Kaya (1989): conventional draining froth, polyhedral froth and washed froth.**

**Table 2-3: Characteristics of the three different froth structures as described by Kaya (1989)**

<b>Draining Froth</b>	<b>Polyhedral Froth</b>	<b>Washed Froth</b>
More entrainment	Less entrainment	Less entrainment
More coalescence	More coalescence	Less coalescence
More drainage	Less drainage	More drainage
Not very stable	Not stable	Very stable
Difficult to remove	Difficult to remove	Move easily
Easy to produce	Difficult to produce	Produced easily
Shallow froth	Shallow froth	Deep froth

In wetter froths, the bubbles are spherical and have more space between the bubbles as compared to the less spaces available between the larger polyhedral bubbles of drier froths (see Figure 2-14). The larger spaces between the spherical bubbles are taken up by liquid. Thus spherical, wetter froths increases the entrainment of gangue in the inter-bubble water (Kaya, 1989).

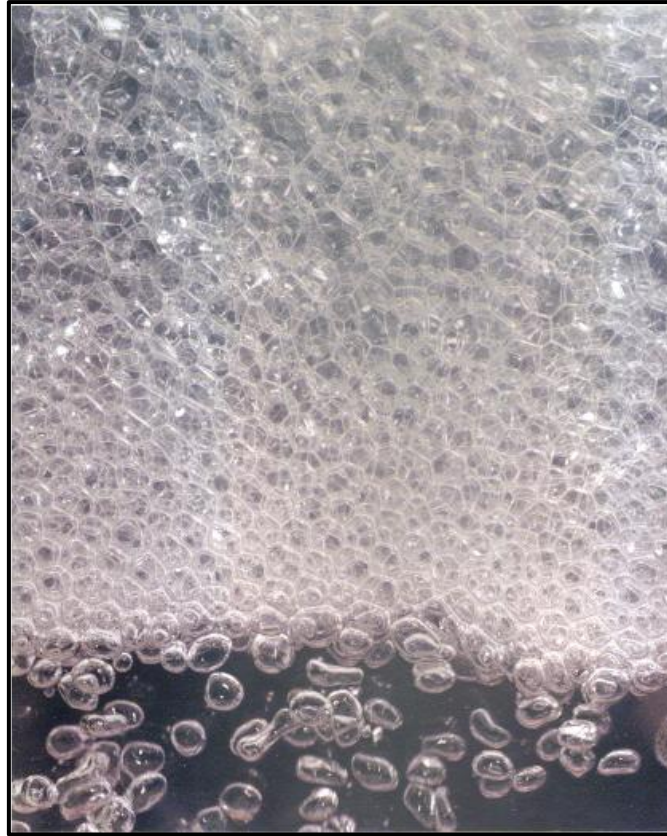


**Figure 2-14: Schematic of a wet froth (a) and a dry froth (b) (taken from Breward, 1999).**

Polyhedral froths fill the available space more efficiently than small spherical bubbles therefore it limits the entrainment into the froth phase. The disadvantage of polyhedral froths is that they are difficult to generate in flotation operations. Polyhedral bubbles can be generated at the top of washed froths if wash water is added a few centimetres below the top of the froth surface but this position, according to Kaya (1989), may not be advantageous. No polyhedral bubbles may be seen if the wash water is added above the froth surface. In mechanical cells with wash water addition, an expanded and washed froth can be created, that tends towards column froths (Kaya, 1989).

The air bubbles are rounder in shape in the lowest part of the froth and are essentially a collection of closely packed spheres. In general, the foam starts off wet near the pulp-froth interface and becomes dryer as it moves up and the liquid drains out. In the upper and drier parts of the froth, the bubbles have a polyhedral shape (Figure 2-15) and collapse when the lamellae separating the bubbles fail (Cilliers, 2006). From an entrainment point of view, polyhedral froths seem to have the best geometrical characteristics for the minimisation of gangue carry-over into the concentrate due to the reduced liquid content surrounding the bubbles.





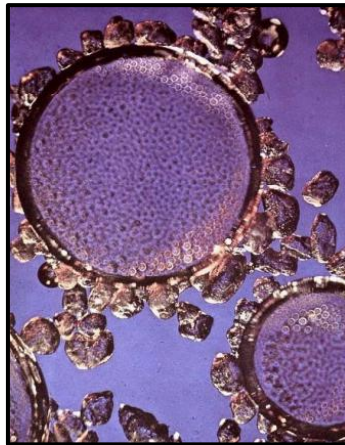
**Figure 2-15: Vertical cross section through a typical flowing foam showing rapid decrease in liquid content and bubble growth (adapted from Cilliers, 2006).**

#### **2.5.2.2. Particle motion in the froth**

Particles in the froth (hydrophobic or hydrophilic), in computational fluid dynamics, may occur in the froth as attached to the bubble lamellae or they may occur as unattached and moving freely with the liquid in the network of plateau borders. Attached particles may become detached from the bubble lamellae due to coalescence of bubbles or surface bursting (Cilliers, 2006).

### 2.5.2.3. Froth movement

The mineral laden air bubbles (as seen in Figure 2-16) rise through the pulp phase and enter the froth phase. Thereafter, the bubbles will be merged into the froth phase. In cylindrical flotation cells, the froth near the edge of the cell migrates easily to the concentrate launder. The froth near the middle of the cell, however, does not have such an easy journey. Due to the distance, the froth near the middle of the cell is subject to bubble collapsing and loss of valuable mineral. Further, on the way to the concentrate launder, the valuable minerals may have the opportunity to be re-captured by rising bubbles from below and hence the process of detachment and re-attachment may occur several times as the froth moves from the middle to the concentrate launder. Various arrangements of internal launders are used to reduce the loss of mineral as the froth traverses the surface.



**Figure 2-16: Picture of a mineralised bubble on route to the froth phase and thereafter to be collected in the concentrate (taken from Minerals Processing Techniques, 2004).**

As mentioned previously, flotation is based on the probability of particle-bubble attachment and also on the probability that the mineral laden bubble flowing into the froth phase and into the concentrate launder. It is not guaranteed that all the bubbles forming the froth are recovered in the concentrate.

Froth recovery is low in scavenging cells. Previous work that measured froth recoveries in scavenger cells (Savassi et al, 1997) found these recoveries to be as little as 10%. Froth recoveries in the rougher were found to be around 50-80% (Runge et al, 2010). This proves that there is great potential to improve the flotation performance in the scavenger stage of flotation through better comprehension of the froth phase. A method of increasing the recovery of the froth is through the use of froth directing baffles. This functions to direct the froth out of the ‘dead zones’, where the froth is stagnant. In the flotation cell, the role of the froth is to selectively transport the valuable particles to the concentrate, while at the same time discarding the undesirable gangue minerals that have entered the froth phase by entrainment.

#### 2.5.2.4. Importance of bubble size

Smaller bubble sizes increase the rate of flotation (Atkinson, 1994). A reduction in bubble size (with a fixed air flow rate) results in an increased bubble surface area available for particle attachment (see Table 2-4) due to the larger number of smaller bubbles produced. An increased bubble surface area allows for more mineral attachment.

***Table 2-4: Comparison of bubble sizes, depicting the relationship between bubble size and surface area (adapted from Rodrigues and Rubio, 2006).***

Bubble size ( $\mu\text{m}$ )	20	50	100
Number of bubbles in $1\text{ cm}^3$	1 250 000	100 000	14 000
Surface area ( $\text{cm}^2$ ) in $1\text{ cm}^3$	23	12	6.6

Various authors have noticed the trend of bubble-particle detachment in a fine particle, coarse bubble combination (Crawford and Ralston, 1988 and Dai et al, 1998). It was concluded that a wide distribution of bubble sizes is required for the recovery of fine minerals. This distribution should include micro or middle-sized bubbles (approximately 100-600  $\mu\text{m}$ ). The generation of these bubble sizes is an immense task (Rodrigues and Rubio, 2006). Mechanical flotation cells generate bubble sizes in the range of 600-2500  $\mu\text{m}$  and are not as good as column cells for the separation of fine particles (Yianatos, 1989; Rubio et al, 2002). Column flotation cells are able to develop finer bubbles with the aid of spargers or micro bubble generators.

Rodrigues and Rubio, (2006) found that micro bubbles were able to capture small mineral particles and also that a linear relationship exists between water and particle recoveries through the use of micro bubbles in flotation. This implies that fine gangue material will also be recovered (via entrainment) with the valuable particles. Thus, the micro bubbles increase the likelihood of entraining gangue particles, due to the water recovery. It was concluded that practical applications of micro bubbles in flotation were dubious unless means of reducing fine gangue entrainment was possible. There has been not much research into the use micro bubble generators in mechanical flotation cells. Hence the production of fine bubbles in roughing and scavenging applications, where mostly mechanical cells are used, is limited and methods of improving fines recovery other than with the use of fine bubble generation should be investigated.

### **2.5.3. Solids Drainage**

The bubbles rise up in the froth and the liquid flows down due to the difference in density. The liquid that drains holds mostly gangue minerals (since the gangue material is hydrophilic, it is most likely to be present in the water). Bubbles that are packed with highly hydrophobic particles are known to be some cause for gangue entrainment (Ata et al, 2004). The process of solids drainage is considered to be a vital process in flotation affecting both recovery and grade of the final product. The rate of solids drainage from the air bubbles varies throughout the froth depending on its location. In the upper layers where the froth is usually dry, the solids drainage is somewhat slow (Ata, 2011).

### **2.5.4. Froth effects - Means of reducing gangue**

#### **2.5.4.1. Froth washing**

Spraying water on top of the froth is a way of accelerating the drainage of gangue. Froth washing was investigated in this study and it will be discussed in detail in sections 2.5.4.1, 2.6.4 and 2.9. As wash water is added, either one of two things may happen depending on the position of the wash water distribution device. Some of the wash water may just travel into the concentrate with the froth bubbles or it may flow down through the froth which has the effect of flushing out the entrained gangue particles, surrounding the bubbles, transporting them into the liquid zone below (Ata, 2011). The flushing out action is the preferred route for the wash water as it removes the gangue particles in the lamella.

#### **2.5.4.2. Baffles in flotation**

Tiitinen et al (2005) described the use of an angled baffle to ‘crowd’ the froth in the direction of concentrate launder. This takes the form of a cone in large circular tank cells, typically when there is a low flow of froth. Kawatra and Eisele (2001) have also stated the beneficial use of horizontal baffles in suppressing the amount of axial mixing that occurs in column flotation cells. Axial mixing is mixing along the vertical axis of the column which can lead to detachment of valuable articles and a probability of re-attachment with uprising bubbles. Through the upward momentum of the bubble, the liquid flows with the bubble around its streamlines and is transported along with the bubble to the froth phase. The liquid then flows back down again (due to gravity, since it is heavier than the froth phase and hence will not remain there) thereby setting up a strong mixing action. The turbulence created by the mixing is unfavourable and causes detachment of valuable particles. Kawatra and Eisele (2001) have stated that the introduction of a baffle suppresses the axial mixing. The baffles function in interrupting the flow of the uprising liquid thereby preventing it from being carried to the surface. The entrainment problem can also be alleviated by this.

Research has been based on both trial and error as well as computational methods (computational fluid dynamics). However, the computational fluid dynamics approach has to take into account the unique (but essential) properties of the froth which may bring in complications in the modelling of the froth phase (Cilliers, 2006).

### **2.5.5. Surface chemistry**

The flotation process depends on the surface properties of the particles to be floated. The principal surface property to be manipulated by chemicals is the surface hydrophobicity of the valuable mineral. Hydrophobicity is a characteristic of being water repellent, or non-polar. Interfacial forces are established between air, water and the mineral surface, which can hold the particle to the surface of the bubble. The surface of the valuable minerals may or may not be naturally hydrophobic. Hence chemicals are added in the flotation mixture in order to increase the hydrophobicity of the valuable minerals. In some cases, the mineral exists as a composite particle, with only a small proportion of the surface area of the particle being hydrophobic. The probability of this part of the particle striking a bubble is low and hence the rate of flotation is low.

#### **2.5.5.1. Collectors**

The main chemical used in the flotation process is the collector. The collector is responsible for making the surface of the valuable mineral hydrophobic. It does this by reacting selectively with the valuable mineral (Loveday, 2006). Although certain minerals may already possess a naturally hydrophobic surface, (and so do not need a collector), collectors can be added to improve the recovery (Chevron Phillips, 2012). There are different types of collectors depending on the type of mineral being floated and the surface charge of the mineral (collectors can be non-ionic, cationic or anionic) as well as various other factors. Typical collectors used in industry are xanthates (X), dithiophosphates, dithiocarbamates, amines and many others. Xanthate collectors (or dithiocarbonates) have been used in bulk PGM flotation for many years. The use of trithiocarbonates (TTCs) as collectors has been investigated recently (Vos et al, 2007). Trithiocarbonates are currently used in the flotation of gold ores in South Africa. An improvement in recovery was obtained by using the short chain TTC collectors (as compared to xanthates), but they had a disadvantage of an unpleasant odour that is associated with the decomposed products.

### 2.5.5.2. Frother

The second most important chemical in flotation (as it forms the most important phase in flotation, the froth phase) is the frothing agent, which is more commonly known as the frother.

A frother is a chemical that assists in the formation and stabilisation of the froth phase. It also increases flotation kinetics and hence the rate of flotation (Pretorius, 2010). The frother reduces the surface tension between air and water. Air bubbles are formed in the turbulent wake behind the impeller blades. By reducing the surface tension, the work that is needed to create the froth is reduced and hence the froth forms with ease (Chevron Phillips, 2012). This reduced surface tension enables a relatively small, stable bubble to be produced (Cilek et al, 2003). The frother increases the stability of the froth by inhibiting the coalescence of bubbles (Ireland et al 2006) and therefore the froth is able to reach the concentrate launder without collapsing. In other words, the stability that the frother creates in the froth phase (by preventing the bubbles from collapsing) should enable the froth to be just stable enough to transfer the minerals that have been floated from the flotation cell to the concentrate launder. The froth phase is the most important phase in the flotation cell as it provides the physical separation that is needed between the minerals that are floated (the valuable minerals) and the pulp containing the gangue material. Typical examples of frothers are: Pine oil, alcohols, and low molecular weight polypropylene glycols (Chevron Phillips, 2012). The frother selected for a particular application must be such that it promotes the formation of good quality froth.

According to Wright (1999), a visual description of a good froth may be explained as follows:

- Consists of small mineralised bubbles with some of them collapsing on the surface of the froth without forming larger bubbles that have a tough film covering.
- The froth should be mobile and not viscous.
- Bubble sizes should be around 1-3 centimetres when it reaches the froth surface (although this depends on the size of mineral particles being floated as well). The bubbles in the froth appear in the froth with a film of mineral particles.
- The degree of froth stability should be somewhere in the middle because a very stable froth consists of bubbles that do not collapse once over the overflow lip, whereas an unstable froth collapses right away as soon as it reaches the froth phase in the flotation cell. Froths that are too 'dry' are known to become too stable.

The froth must be strong enough to support the weight of the mineral floated but not be too rigid and non-flowing. The froth should ideally break down once removed from the flotation cell. The dosage of frother into the flotation cells is an important factor to consider and it is known that an increase in frother concentration causes a decrease in power consumption (Cilek et al, 2003).

The overall characteristics of the froth are made up of many individual physical characteristics such as: froth depth, frequency of froth removal, method of froth removal, bubble size, dispersion of particles in the froth and the intensity of pulp movement in the zone under the pulp-froth interface (Wright, 1999). In the event of the gangue mineral already being in the froth phase (usually in the case of naturally floatable gangue material), even the best froth cannot assist in its removal. Hence means must be taken to prevent the gangue from moving into the froth phase in the first place. This is done through the use of a gangue depressant.

#### **2.5.5.3. Depressants**

A depressant functions to inhibit the flotation of certain minerals. It does this by preventing the adsorption of the collector chemical on the surface of the mineral, hence assisting in the selectivity of the collector chemical. The talc present in UG-2 ore has natural floatability and it tends to float rapidly in the rougher stage. High molecular weight polymers, which are hydrophilic, are added to depress the talc in the cleaning stage. The polymer is adsorbed on the talc surface by molecular bonding, but it does not adsorb on the coated minerals, because of its negative charge. Depressants are also used to selectively depress one or more of the base metal minerals (Cu, Pb and Zn). Common depressants used in the industry are sodium cyanide, dextrin, guar gum, or carboxymethyl cellulose (CMC) and starch (Chevron Phillips, 2012).

#### **2.5.5.4. Activators**

The activator improves the action of the collector. Copper sulphate ( $\text{CuSO}_4$ ) is generally used as an activator in the flotation of base metal sulphides that occur in association with PGMs in UG-2 ore (Grobler et al, 2005). The metal ion on the base metal sulphide mineral is exchanged with a Cu ion thereby precipitating a Cu-rich sulphide on the surface of the mineral. It is hypothesised that the collector readily adsorbs on the Cu-rich surface and thereafter carries out its function to make the surface hydrophobic and susceptible to flotation (Grobler et al, 2005). Other chemicals can be used to assist in the flotation of valuable minerals. These chemicals are selected based on trial and error in experimentation and the effects they have on the recovery and grade are the criteria on which they are judged on for their use.

#### **2.5.5.5. Secondary reagents: Diesel and paraffin**

Secondary reagents, such as co-collectors or promoters may be added in addition to the prescribed collector to enhance the function of the collector. Various co-collectors are widely used in the flotation industry (Choi et al, 1993; Vos et al, 2007; Grobler et al, 2005; Hangone et al, 2005) and these co-collectors have made it possible to reduce the amount of collector and hence the cost, without affecting metallurgical performance.

Diesel and paraffin may be used in flotation to act as a collector or to optimise the function of the collector chemical. Diesel and paraffin are both hydrocarbons and are hence non-polar. The use of diesel and paraffin in the flotation industry as an aid to the collector is not uncommon. Diesel is a non-polar hydrocarbon oil and is used in flotation for a number of reasons. A few of these reasons are reducing the overall reagent cost by replacing part of the collector addition and improving the recovery of coarse particles without reducing the selectivity (Seitz and Kawatra, 1986). Paraffin is strongly hydrophobic and has been used in the past to assist the collector chemical in the flotation of graphite (Kaya and Canbazoglu, 2012). Diesel and paraffin are commonly used in coal flotation (Pretorius, 2010).

#### **2.5.5.6. pH**

The pH of the system is also adjusted through the use of chemicals. Some minerals are found to float better in an acidic medium or in a more alkaline medium. Calcium hydroxide and sulphuric acid are usually used to adjust the pH. The optimum pH for flotation of PGMs is between 8 and 9 (Cheminova, 2012). The UG2 ore reacts with water to produce a pH in the optimum range and adjustment of pH was not investigated in this study.

#### **2.5.6. Requirements for an ideal flotation in a mechanical flotation cell**

It is important that the valuable particles are in suspension (in order to attach onto rising bubbles and to enter the froth phase) and not at the bottom of the flotation cell. Therefore the contents of the flotation cell should be thoroughly mixed. The valuable particles must also be sufficiently coated with the collector chemical (sufficiently hydrophobic) for attachment to air bubbles. There should be an adequate flow of air bubbles to collect the mineral particles and carry them to the launder.



### **2.5.7. Mechanism for particles entering the froth**

After the addition of the required chemicals with the suitable conditioning time during the flotation process, the particle's hydrophobic surface makes contact and adheres to surface of the air bubble. It is then transported to the upper section of the flotation cell (the froth zone). The froth laden with valuable minerals accumulates at the surface, creating the froth layer, which migrates slowly towards the edge of the cell and flows over the lip to be collected into a launder. The overflowing froth is the flotation concentrate. The bubbles coalesce as they rise through the froth and about 60% 'pop' at the surface (Cilliers, 2006). The particles adhering to these bubbles are shed and some re-attach to bubbles rising from below. This process of shedding and re-attachment contributes to concentration of the valuable minerals, but leads to the loss of composite particles.

### **2.5.8. Mechanism for gangue particles entering the froth through entrainment**

Ideally, the unwanted gangue (hydrophilic) particles which do not adhere to the air bubbles remain in the pulp phase and exit in the tailings. In practice, the rising air bubbles drag a certain amount of liquid behind them (in their wakes). Since gangue particles are hydrophilic (water-loving), they are present in the entrained liquid which follows the air bubbles to the top of the cell. This phenomenon is known as entrainment of gangue. It is by this process of that gangue particles enter the froth and exit with the concentrate, thereby reducing the grade of the product and increasing the amount of impurities in the flotation concentrate. Various authors in the field of flotation have acknowledged the strong relationship between the recovery of entrained solids and the recovery of water into the concentrate (Ross, 1987; Smith et al, 1989; Savassi et al, 1997 and Ireland et al, 2006). Recent publications on froth washing by Neethling and Cilliers (2001) showed that washing increased the grade, but at the expense of the recovery of the valuable mineral.

### **2.5.9. Gangue transport/Entrainment**

There are mainly two mechanisms by which gangue is recovered. Firstly, the transfer of gangue units in the liquid which are locked between bubbles as they rise from the pulp phase and secondly through the coating of slimes on the surface of the particles (Yianatos et al, 2008). The gangue slimes covers the originally hydrophobic sulphide particles through a heterocoagulation mechanism, rendering the once hydrophobic particles to be hydrophilic (Hanumantha Rao and Chernyshova, 2011).

According to Yianatos et al (2008), the five main variables that influence gangue entrainment are:

1. Feed solids percentage
2. Froth characteristics (residence time, froth stability, froth transport)
3. Density of the solid particles
4. Particle size
5. Liquid recovery

The main variable liable for the entrainment of gangue, during the collection and separation processes in the flotation cell, is water recovery (Yianatos et al, 2008). Many authors in the field of flotation, as previously mentioned, have also consistently found strong correlations between gangue recovery and water recovery. This relationship exhibits a linear behaviour for the fine particles (less than 40 µm). It was also found that the gangue entrainment was not significant for coarse particles.

Neethling and Cilliers (2009) investigated entrainment of gangue and found a proportionality between the fractional gangue and fractional water recoveries. This relationship was referred to as the degree of entrainment or the entrainment factor and is given the symbol *Ent*. The entrainment factor was calculated as follows:

$$Ent = \frac{M_c Q_F}{M_F Q_c} \quad (1)$$

Where M is the mass flow of entrained solids, Q the volumetric flow of water and subscripts F and C signify the feed and concentrate correspondingly. The entrainment factor is expressed as the recovery of the entrained species divided by the recovery of water. Many others have developed correlations to determine the entrainment of gangue material into the flotation concentrate. The correlation by Neethling and Cilliers (2009) is just one of the many relationships developed. Various semi-empirical and empirical equations have been derived to quantify the amount of entrainment based on the amount of water recovered and the physical properties of the gangue material.

Numerical models of entrainment have been developed from a more fundamental point of view and the simulated results show a good correlation with the results obtained experimentally on a laboratory scale (Neethling and Cilliers, 2002; Stevenson et al, 2007). However the models did not apply in industrial operations (Yianatos et al, 2008), because the assumptions made during laboratory experimentation were not applicable.

### 2.5.10. Flotation in industry

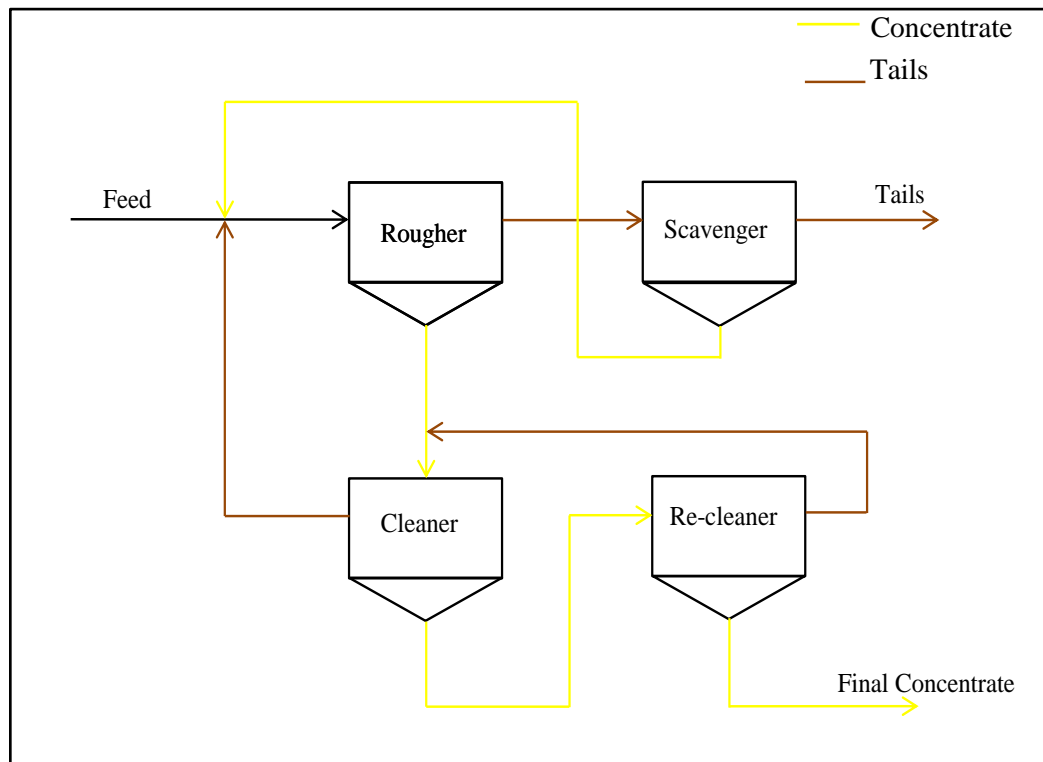
Flotation, in industry, takes place in either a column flotation cell or a mechanical flotation cell depending on the requirements of the process and on the nature of the material being floated. Generally column cells form a deep froth and are therefore used when deep froths are required such as in coal flotation. Mechanical flotation cells form a relatively shallow froth phase compared to the column flotation cells and they are commonly used to float sulphide minerals and PGMs. In industry, flotation usually occurs in banks of cells (see Figure 2-17) and in multi-stage circuits to provide improved separation between valuable and gangue minerals.



**Figure 2-17: Bank of WEMCO® SmartCell™ flotation cells in a flotation circuit (taken from FLSmidth, 2010).**

There are many types of flotation cells that are used in industry. Some examples include *WEMCO®*, *Dorr Oliver* and *Denver* flotation machines. The choice of which cell to use depends on many factors such as the volume of pulp to be floated at a given time and the froth height required.

In industry, it is common for flotation to take place in banks of flotation cells. Flotation cell are considered to be ‘well mixed’, and hence for a given residence time for recovery of minerals, it is necessary to use a bank of cells, rather than a single large cell, to avoid by-pass effects. These banks of cells can be classified as the rougher, cleaner, scavenger, and occasionally flotation circuits may have a re-cleaner or a cleaner-scavenger. A general example of a multi-stage flotation circuit can be seen in Figure 2-18.



**Figure 2-18: A basic configuration of a multi-stage flotation circuit showing four stages of flotation (rougher, scavenger, cleaner and re-cleaner). Between the stages, there is intermediate re-grinding of the pulp or the concentrate in re-grind mills (not shown) (Loveday, 2006).**

The different stages of flotation perform different functions in a flotation circuit. The rougher stage of flotation is responsible for the primary separation of valuable minerals from gangue. Following the rougher stage is the cleaner and the scavenger stages of flotation. The cleaner flotation cells receive the concentrate (mostly valuable content) of the rougher and the aim is to further remove gangue material from this concentrate, thereby creating a cleaner concentrate, hence the name ‘cleaner’. The concentrate of the rougher flotation stage may contain gangue material which must be removed or minimised. The cleaner and re-cleaner stages of flotation serve this purpose of improving the grade of the rougher concentrate hence the final product will be of a much higher purity.

The scavenger stage of flotation receives the tailings (mostly gangue content) of the rougher cells and therefore functions to recover the remaining valuable minerals that were not recovered in the preliminary rougher cells. Hence the scavenger functions to minimise losses to the tailing and to improve overall recovery. Different industries have different circuit configurations to suit the minerals being floated. Flotation circuits are designed so as to obtain the maximum recovery and grade whilst keeping reagent usage and power consumption to a minimum.

In the flotation cell, the pulp suspension is understood to contain fast floating particles (with a high rate of flotation), slow floating particles (with a low rate of flotation) and non-floating particles with the resultant rate of flotation being a function of each of these fractions (Lynch et al, 1974). Hence the flotation circuit needs to be designed to accommodate the different floating particles. The fast floating particles are recovered in the rougher stage of the flotation whereas the slow floating particles are generally recovered in the scavenger stage. Knowledge of these rates can greatly assist the design of the flotation circuit such that adequate flotation residence time is allocated for the main sections of the plant (Bryson, 2003). Flotation kinetics, however, do not affect the phenomenon of particle detachment and reattachment as this is determined almost entirely by chance.

#### **2.5.11. Detachment and re-attachment of hydrophobic particles**

Some of the mineral-laden bubbles collapse at the surface of the froth and shed their particle load. These detached particles may or may not re-attach to bubbles arriving from below. The detachment and recapturing procedure may create an internal flow (or turbulence) between the froth phase and pulp phase which may possibly reduce the flotation cell throughput significantly (Ata, 2011). On the other hand, this process also contributes to enrichment of the valuable minerals whilst the froth migrates to the edge of the cell. When the bubbles shed their load of particles, the hydrophobic particles are most likely to re-attach to the air bubbles rising up from below. This has the effect of a loss of gangue material as the process of detachment and re-attachment takes place. Consequently, the bubbles reaching the top and flowing into the concentrate have a smaller gangue loading as compared to a bubble not having undergone detachment and re-attachment.

Composite particles are also subject to detachment and re-attachment. Part of the valuable mineral could be exposed to the air bubble and hence attaches to it. The down side to this process is when the process of detachment takes place whereby a potential loss of valuable mineral is expected since the composite particles may not easily re-attach to the rising bubbles due to the hydrophilic nature of gangue in the composite particle.

Falutsu and Dobby (1989) reported indications of the detachment of floatable particles within the froth body and also found that there are indications of detachment occurring at the pulp-froth interface. The above authors state a possibility of up to 60% of hydrophobic particles that enter the froth could be detached at the pulp-froth interface. It is theoretically known that the detachment of hydrophobic particles occurs in the froth phase. The re-attachment process is also quite theoretical and the debate as to whether detached particles actually are recaptured within the froth before they come out of the froth zone has not yet been resolved (Ata, 2011).

Laboratory tests in a Denver flotation cell that monitored the trajectory of a single hydrophobic and hydrophilic particle were conducted by Waters et al (2009). These tests were done by using Positron Emission Particle Tracking (PEPT). The results clearly showed that detachment and re-attachment took place for the hydrophobic particle and also found the hydrophilic particle rarely entered the froth phase and mainly followed the flow of water. Further evidence of reattachment in the froth is provided in the froth upgrading phenomenon whereby one mineral type is concentrated at the cost of others.

## 2.6. Process optimisation of flotation circuits

Process optimisations in flotation circuits include grade and recovery improvement and control, cost and energy minimisation, and in the case of UG-2 ore, chromite reduction. Various parameters are to be considered when optimizing a flotation circuit. Some of these parameters are listed in the following table taken from Valenta and Mapheto (2011). The parameters were listed in three categories namely inherent, mechanical and operational which each contain sub-parameters. Other parameters not originally in the table but will be intensively studied in this project are the effect of the addition of wash water (froth washing) and the use of a baffle. The use of reagents to enhance the function of the collector was also investigated and these fall under the sub-parameter secondary reagents.

**Table 2-5: Parameters to consider when optimising a flotation circuit, taken from Valenta and Mapheto (2011).**

Inherent parameters	Mechanical parameters	Operational parameters
PGM mineralogy Feed grade Gangue dilution	Circuit design Type of equipment Classification circuit Flotation cell geometry	Mill power Pulp density Reagent type Reagent dosage Reagent dosage point Froth depth Aeration rates *Froth washing *Baffle *Secondary reagents

\* Parameters investigated in this investigation.

Inherent factors such as ore variability stem from parameters such as ore mineralogy which cannot be changed since it is characteristic of the ore. Parameters that can more likely be optimised are the mechanical and the operational.

### 2.6.1. Aeration rates

Valenta and Mapheto (2011) found that an increase in the aeration rate in plant flotation cells did not improve the recovery. It caused an additional problem, as the bubbles became larger, causing the froth to ‘geyser’ and to become disturbed. This is in contrast to Bulatovic (2007) who stated that an increase in air flow rate increases the floatability of the particles thereby improving the recovery. Clearly it depends upon the range of air flow that the mechanism can tolerate. However, in general, the bubbles will get larger as the air rate is increased, unless the impeller speed is increased simultaneously.

### 2.6.2. Reagent type and dosage

It is vital to ensure the correct type and balance in the reagents used since the quantities of the reagents added into the flotation cell affect recovery and entrainment processes. The collector type and dosage determines which floatable mineral species are recovered into the concentrate and also affects the amount of entrained solids reporting to the concentrate. Frothing agents have been shown to be closely linked to the entrainment of particles into the concentrate hence the dosage should be selected carefully. The depressant dosage must be balanced with frother and collector addition as it was found that high dosages of depressant resulted in the froth of the cleaner flotation cells collapsing (Valenta and Harris, 2007). Other findings (Vos et al, 2007) state that an increase in depressant dosage results in a reduction in the amount of floatable gangue that reports to the concentrate which is what ideally is supposed to happen. This illustrates the difficulty in determining the optimum quantity of chemicals to add. There are various reagent suites used by different PGM producers in the flotation of UG-2 ore. Some of these may be seen in the following table:

**Table 2-6: Common reagents for flotation of UG-2 ore.**

Activator	Collector	Depressant	Frother
CuSO <sub>4</sub>	SNPX	KU5 (or CMC)	DOW 200
	SIPX	MN49	Polyglycol ether
	SIBX	IMP4 (guar gum)	
	PAX		
	Trithiocarbonates		



The use of trithiocarbonates is a relatively new concept and an improvement in gangue rejection has been demonstrated when small quantities were added with the traditional dithiocarbonate (xanthate) collectors (Vos et al, 2007). The use of DOW 200 as a frother is well established for flotation of UG-2 ore and it is almost always the frother used by the various PGM producers. Copper sulphate ( $\text{CuSO}_4$ ) is generally used as an activator in the flotation of PGE's but its use is questionable and it is known to cause a wide range of responses in flotation (Mailula et al, 2003).  $\text{CuSO}_4$ , as an activator, is added to enhance action of the collector. Interactive and competing effects with other reagents in the flotation cell may prevent  $\text{CuSO}_4$  from carrying out its proper function although it has been proved that the use of  $\text{CuSO}_4$  in the flotation of sulphide minerals has been effective (Wiese et al, 2006). It should be noted that the copper precipitates as  $\text{Cu}(\text{OH})_2$  at the high pH, which may inhibit its effect. The addition of  $\text{CuSO}_4$  before the addition of the collector may cause chromite to be hydrophobic and hence susceptible to true flotation (Wesseldijk et al. 1999). Wiese et al (2007) has also found that  $\text{CuSO}_4$  has been known to cause some gangue activation.

The occurrence of chromite in the concentrate can result in ineffective smelting processes since it is a member of the spinel group which are able to form compounds that are stable to temperatures of up to  $2000^\circ\text{C}$  (Jones, 2005). The reagents used in flotation should be designed to maximize the recovery of the sulphide minerals that contain the PGMs and reduce the gangue content in the concentrate.

Valenta and Mapheto (2011) made numerous changes to the reagent suite for the flotation of UG-2 ore. The changes that were made include: an increase in the overall collector dosage; the use of a different collector (dithiophosphate). The results of changing the reagent suite proved to be futile and the mass recovery to the concentrate remained low. The results of this experiment prove that altering the reagent suite is not favoured, and potential investigators in the field of flotation should consider other factors when attempting to optimise PGM and base metal recovery.

### **2.6.3. Froth depth**

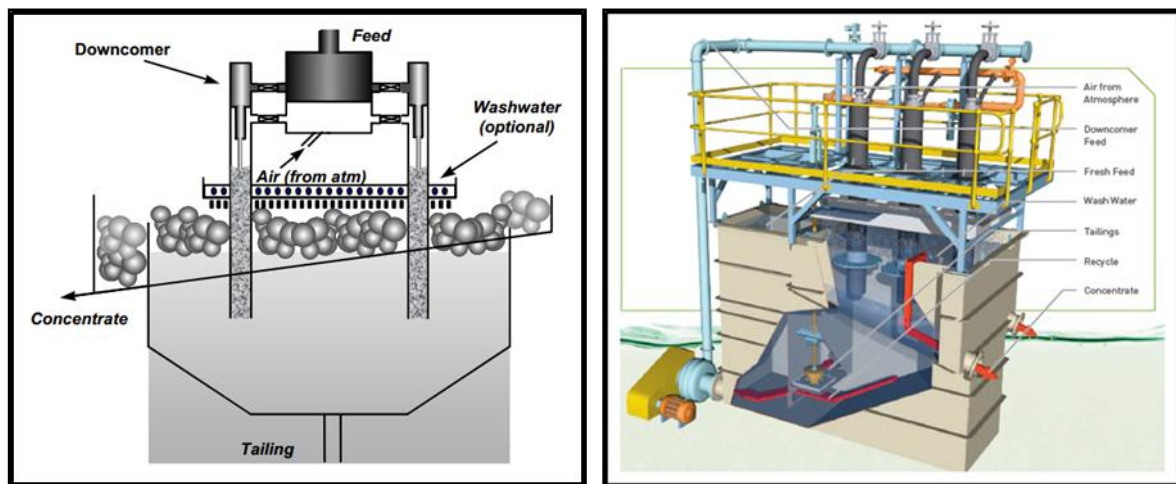
An important operating variable in flotation is the froth depth. The general belief is that deeper froths have a lower flow rate of overflow water because additional drainage of the water has occurred. Cilliers (2006) stated that although this behaviour is generally observed, the reason for this occurrence is incorrect. The decrease in the flow rate of overflowing water in deep froths may be ascribed to an increase in coalescence and bubble bursting which are both a result of an increased residence time. It was found (by Ross, 1987; McKeon, 2001 and Runge et al, 2010) that the rate of flotation decreases with an increase in froth depth. It was also found that deep froths play a role in promoting the selectivity of flotation due to the increased coalescence that is caused (Yianatos, 1989). Coalescence (or collapsing) of the froth causes the particles to detach and again re-attach onto air bubbles coming from below. McKeon (2001) found that the froth depth had a significant effect on the effectiveness of the system.

### **2.6.4. Froth Washing**

Froth washing is the addition of wash water into the froth phase of the flotation cell in the hope of washing away the entrained gangue particles (Cilliers, 2006). Frothing agents may be added to the wash water (at the same concentration as in the feed) to assist in the formation of fine bubbles and also to ensure that a stable froth is formed as the bubbles rise and separate from the liquid (Jameson, 2010). Many types of wash water distributors may be used as a means of directing the wash water into the froth. As stated previously, froth washing aims to reduce the amount of entrained gangue and thereby improve the grade of the final product. More particularly to the flotation of UG-2 ore, froth washing aims to reduce the entrained chromite and hence reduce the problems caused by chromite in the smelters. The addition of wash water into flotation cells is a relatively new research field in flotation to improve gangue rejection. Few authors have experimented on froth washing techniques in the past (Kaya, 1989; McKeon, 2001). There have also been few authors that simulated the effect of froth washing in flotation cells (Neethling and Cilliers, 2001).

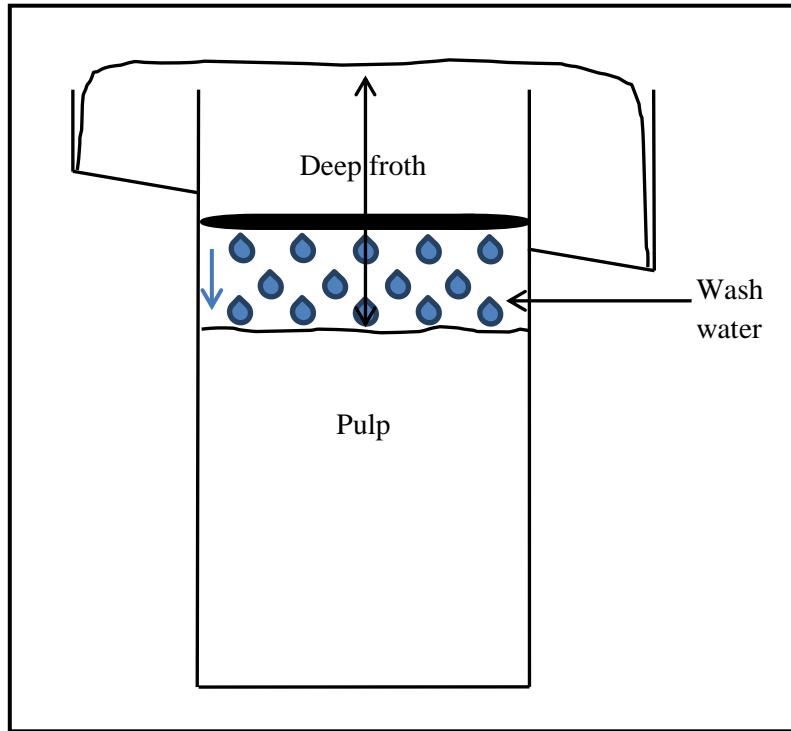
Kaya (1989) investigated washing of the entire froth surface in both laboratory and part of the froth surface on a pilot plant scale. It was found that the recovery of the valuable minerals generally increased due to a stabilized froth. On the plant scale, two perforated cylindrical tubes (wash water distributors) close to the concentrate weir yielded the highest rejection of gangue. On the plant scale, the stabilisation effect by the wash water was not as great as that of the laboratory tests since washing of the froth only took place in one section and not the whole froth. Consequently the recoveries were lower in the plant scale as compared to laboratory scale. On the plant scale the froth washing was limited to the area adjacent to the concentrate weir in order to minimise water requirements.

Froth washing has been applied successfully in column flotation cells (Yianatos, 1989) and in the Jameson cell. Column cells are used mainly in the cleaning stages of flotation. The role of the cleaner flotation cells is mainly to improve the grade of the concentrate hence froth washing in this stage is appropriate. McKeon (2001) applied froth washing on coal concentrates (in cleaner flotation cells) and found that washing improved the product grade. Hacifazlioglu et al (2007) also investigated froth washing on coal and found that an increase in wash water addition rate caused an increase in product purity although at a lower recovery. The Jameson cells have been designed to incorporate froth washing in the cell and can be seen in the following illustration:



**Figure 2-19: The Jameson cell incorporating the froth washing mechanism (after Young et al, 2006).**

The Jameson flotation cell was designed as an improvement over the mechanical and the column flotation cells (see Figure 2-20), incorporating features of both. Like the column cell, there are no moving parts, the froth depth is much deeper than the mechanical cell and flotation is carried out with the use of air spargers. The Jameson cell is able to provide better selectivity and product grade (Xstrata Technology, 2012). It is superior to the mechanical flotation cell due to its ability to produce concentrates of a high grade. As previously mentioned, column cells are mainly used as cleaners and their use as scavengers is rare, especially in the flotation of sulphide minerals (in the case of UG-2 ore). Existing UG-2 flotation circuits already possess mechanical flotation cells and cannot merely change their circuits to incorporate Jameson or column flotation cells. Hence, investigation of froth washing in a mechanical flotation cell was selected for this investigation. A major aspect of this investigation involved finding the appropriate wash water distributor device.



*Figure 2-20: Column flotation cell demonstrating froth washing and its characteristic deeper froth (adapted from McKeon, 2001).*

#### **2.6.4.1. Types of wash water distributor devices**

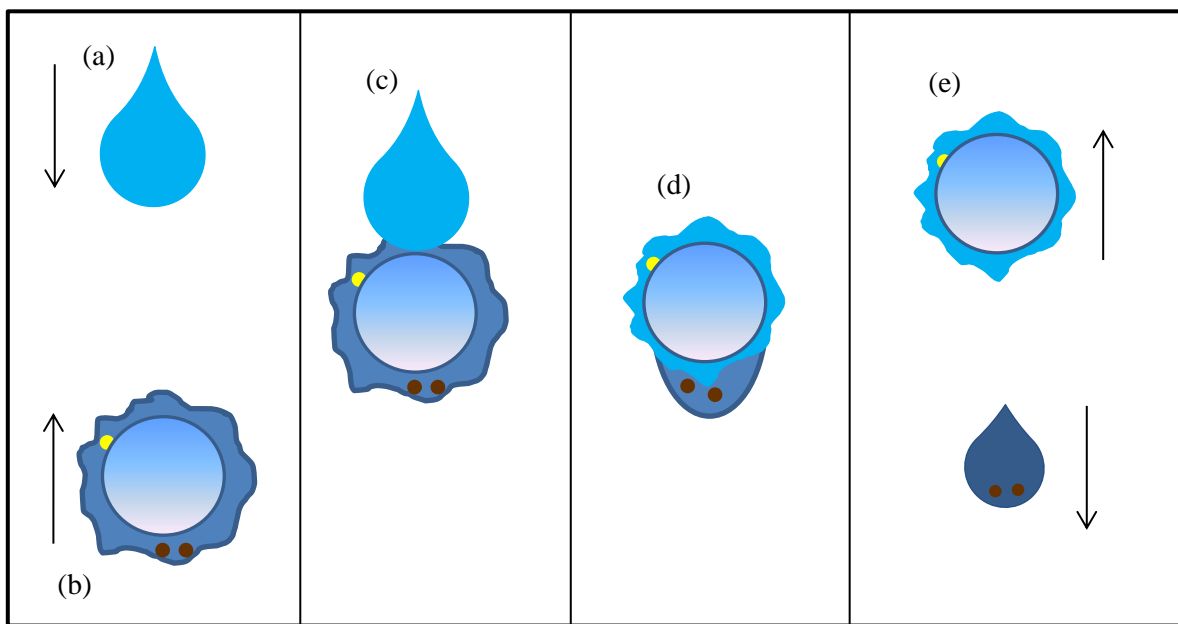
A wash water distribution device (wash water distributor) is an object that transfers water into the froth of the flotation cell. It may be any shape and size depending on the shape and size of the flotation cell onto which it is mounted. The wash water distributor plays an important role in the effectiveness of the wash water addition since it signifies how the wash water enters the froth (through injection via a small aperture or a much calmer entrance into the froth for example a light spray). Froth washing devices currently in use are relatively simple distributors, consisting of horizontal pipes or tubes with small holes made at equal intervals from which jets of water flow. The wash water distributor may be placed anywhere from above the froth to within it. Types of distributors may be described as follows:

- The wash water distributor may be in the form of a wash box (as in McKeon, 2001) whereby water is fed into a box with perforated holes at the bottom, and the box is suspended (positioned on a frame) over the froth phase of the flotation cell.
- Other common distributors may be in the form of nozzles (Jameson, 2010) whereby water enters the froth in an injection-like fashion through the small openings of the nozzles. Many small nozzles (a network of nozzles) may be distributed evenly over the surface (or even within) of the froth.
- The last common distributor that is used to enable froth washing is a wash bar which simply is a hollow rod with holes made on the underside. The rod may be made of tubing (to allow easy construction of holes on the underside) and bound to a rigid support and placed either over the froth or within the froth in the flotation cell. The tubes may be assembled in a radial manner for circular flotation cells.

A fuller description of the advantages and disadvantages of various froth washing distributor devices is given in Appendix E.

## 2.7. Mechanism of froth washing

The main function of wash water addition is to wash out the entrained gangue particles from the froth. Water addition wets and cleans the froth while at the same time still maintaining the froth's characteristic structure (Ata, 2011). Diagrams (a) to (e) in Figure 2-21 below depict the ideal processes that occur when water enters the froth. In (a) the wash water flows down the froth while (b) the air bubbles in the froth move upward. The valuable minerals are attached to the air bubble (as depicted by the yellow particle) and the gangue particles (brown) are dragged upwards with the bubble in its wake. At some point in the descent of the wash water and rise of the air bubble, contact is made (c) and the wash water 'washes' the froth (d) by coating the bubble and replacing the entrained liquid that carries the entrained gangue particles. In (e) the entrained liquid is removed from the bubble and moves downward into the pulp and the 'washed' bubble moves upward in the froth, toward the cell lip and overflows into the concentrate launder.



**Figure 2-21: The ideal mechanism of froth washing.**

### **2.7.1. Effect of froth washing on entrainment**

The wash water displaces the entrained liquid between the bubbles that contain gangue particles which improves the grade of the final product. The wash water addition lowers the viscosity and allows the entrained water to drain away more easily (Neethling and Cilliers, 2001). The gangue particles are hydrophilic and hence will follow the path of the water as it flows downward through the froth. Ideally the valuable minerals, through their hydrophobicity, remain attached to the air bubbles in the froth but there may be incidents where the valuable minerals also are washed away from the froth whilst froth washing occurs. This results in a froth that is reduced in the amount of gangue but still enriched with the valuable minerals.

### **2.7.2. Effect of froth washing on grade and recovery**

McKeon (2001) performed froth washing tests on coal concentrates and found that the gangue content for the tests with wash water was lower than the tests without wash water, hence the grade of the concentrate increased. The recovery of valuable minerals was however found to decrease with the addition of wash water. A possible reason for the decrease in recovery was that the wash water causes detachment of some of the valuable mineral from the air bubbles. The addition of water has the effect of a reduction in residence time (as previously stated). This causes a decrease in the time for a single valuable particle to be recovered. From this experience, it was concluded that froth washing is very limited in its effectiveness. McKeon (2001) found that froth washing coupled with a deeper froth resulted in the significant reduction of gangue and improvement of grade.

## **2.8. Position of wash water addition point**

Wash water may be added over (or within) the entire froth or it may also be added to just a portion of the froth. In the latter case, the position of the wash water addition is an aspect of investigation. The position of the wash water entry into the froth should ideally be optimised such that the wash water is used to its maximum potential in washing away the gangue particles from the froth phase. McKeon (2001), via experiments performed on flotation of coal, found the optimum position of wash water addition to be near the overflow lip. This optimum position was determined based on the ability of wash water to remove gangue particles and hence was determined by the grade of the concentrate. Neethling and Cilliers (2001) carried out simulations which incorporated the effect of moving the wash water addition point on the grade of the concentrate. It was found, as opposed to McKeon (2001), that the highest grade was achieved when the wash water is added further back in the cell which was away from the overflow lip. Neethling and Cilliers (2001) found that when the wash water is added near the overflow lip, much of it would flow over the cell lip without removing gangue particles from the froth.

There are mainly two alternatives for the vertical position of the wash water entry point; the wash water may be added on top of the froth or it may be added within the froth (below the froth surface). Washing below the surface of the froth was found to be favourable since it produced a less wetter concentrate and a smaller reduction in the recovery of the valuable mineral as compared to washing on the surface of the froth. Although this general trend is observed, there is no fixed rules for which wash water addition method is suitable or applies to a certain situation (Cilliers, 2006).

### **2.8.1. Speed and frequency of wash water addition**

The wash water addition into the froth may be continuous or it may be periodic. Wash water has an effect of stabilising the froth (Kaya, 1989; McKeon, 2001), but very high water additions may also have an unfavourable effect of destabilising the froth. An excessively high wash water flow may strip the bubble surfaces from its coat of chemicals which results in the destabilisation effect. Also, the high wash water rates may impact negatively on the grade of the product by increasing the amount of axial mixing which may lead to valuable particle detachment (Kennedy, 2008). On the other hand, a very low wash water flow rate is not favoured as well since it will not provide a stabilising effect.



## **2.9. Froth washing of UG-2 ore**

The main reason for investigating froth washing on UG-2 ore is due to its possibility in alleviating the chromite entrainment problem that is currently being faced in the flotation plants of practically all PGM producers. Froth washing aims to reduce chromite, the main gangue material in UG-2 ore, from the froth by the mechanism stated in section 2.7 whereby the wash water replaces the entrained liquid in the froth and removes the chromite, leaving behind the PGMs attached to the bubble.

McKeon (2001) and Ata (2011) report the use of froth washing to be more common for column flotation cells rather than mechanically stirred cells. In the PGM industry, however, column cells are scarcely used since the recoveries achieved by these cells are said to be poor (Bryson, 2004). Murdock (1991) however states that the column flotation cells have been successively in the flotation of sulphide ores but mainly for the cleaning stages of flotation.

## **2.10. The effects of chromite on smelting**

The entrainment of chromite in the flotation concentrates causes a reduction in efficiency of the smelters. An excess of chromite in the flotation concentrate causes an accumulation of layers of highly refractory chromite spinel on the smelter walls that are stable at temperatures of up to 2000 °C (Jones, 2005). Chromite spinels are high density, high melting point substances that solidify on furnace walls and they can accumulate at the interface between the slag and the matte. This layer can inhibit the flow of molten matte droplets from the slag into the matte. Hence, they may lock up PGMs inside which is a further cause for a loss in recovery of valuable minerals (Vos et al, 2007).

Due to the complications caused by an excess of chromite in flotation concentrates, penalties have been enforced on the upper limit of chromite in the feed to the smelters (Valenta et al, 2011). Recent publications (Nell, 2004; Jones, 2005; Valenta and Mapheto, 2011) state an upper limit value of chromite content to the smelters to be 2.5 to 3 %. Penalties may be incurred when the chromite content exceeds 1.5 %.

## **CHAPTER 3: EXPERIMENTAL WORK**

### **3.1. Preliminary information**

The project was entirely laboratory based and small scale laboratory tests were done in a 4L Denver flotation cell at University of KwaZulu-Natal.

A 250 kg UG-2 ore sample was received (at the University of KwaZulu-Natal) in sealed buckets approximately 40 kg (wet mass) each. The sample was obtained from a platinum flotation plant, being the feed to the secondary mill (i.e. the underflow from the secondary densifier). Hence, it had passed through flotation cells for primary recovery of PGMs and the maximum particle size was of about 400 microns. The solids settled in the buckets and the excess water was decanted when preparing the ore for sub-sampling.

#### **3.1.1. Sub-sampling procedure**

The contents of the bucket were emptied onto a large sheet of tough plastic where thorough mixing of the ore took place. The cross cut method of mixing was applied for thorough mixing such that representative samples may be obtained for the flotation tests. Sub-samples of approximately 2 kg were required for each test.

### **3.2. Determination of moisture content**

The dry mass of solids in each test was required to be 2 kg. The ore that was received from Lonmin was not dry due to it being taken from a secondary circuit that has already been through primary flotation. Hence a moisture content test needed to be done to obtain the amount of moist ore that is required that to result in 2 kg of dry solids. This was done by weighing many samples of moisture-laden ore and placing it the oven overnight to dry. The dry masses were then weighed and a simple moisture content calculation was carried out based on the amount of water contained in the ore. The calculation for moisture content can be found in Appendix B1.

### 3.3. Reagent preparation

The reagents used in the entire experiment comprised of  $\text{CuSO}_4$ , SNPX, MN49, DOW 200, diesel and paraffin. The supplier of the ore also supplied the SNPX, MN49 and the DOW 200. The remainder of the reagents were available in the laboratory.

The  $\text{CuSO}_4$ , SNPX, MN49 and the DOW 200 chemicals were all prepared as 1 % (by mass) solutions. All the chemical solutions were made in 500 mL volumetric flasks. The  $\text{CuSO}_4$ , SNPX and MN49 were in solid form and the DOW 200 was in a form of a viscous liquid. The solutions were freshly prepared every week.

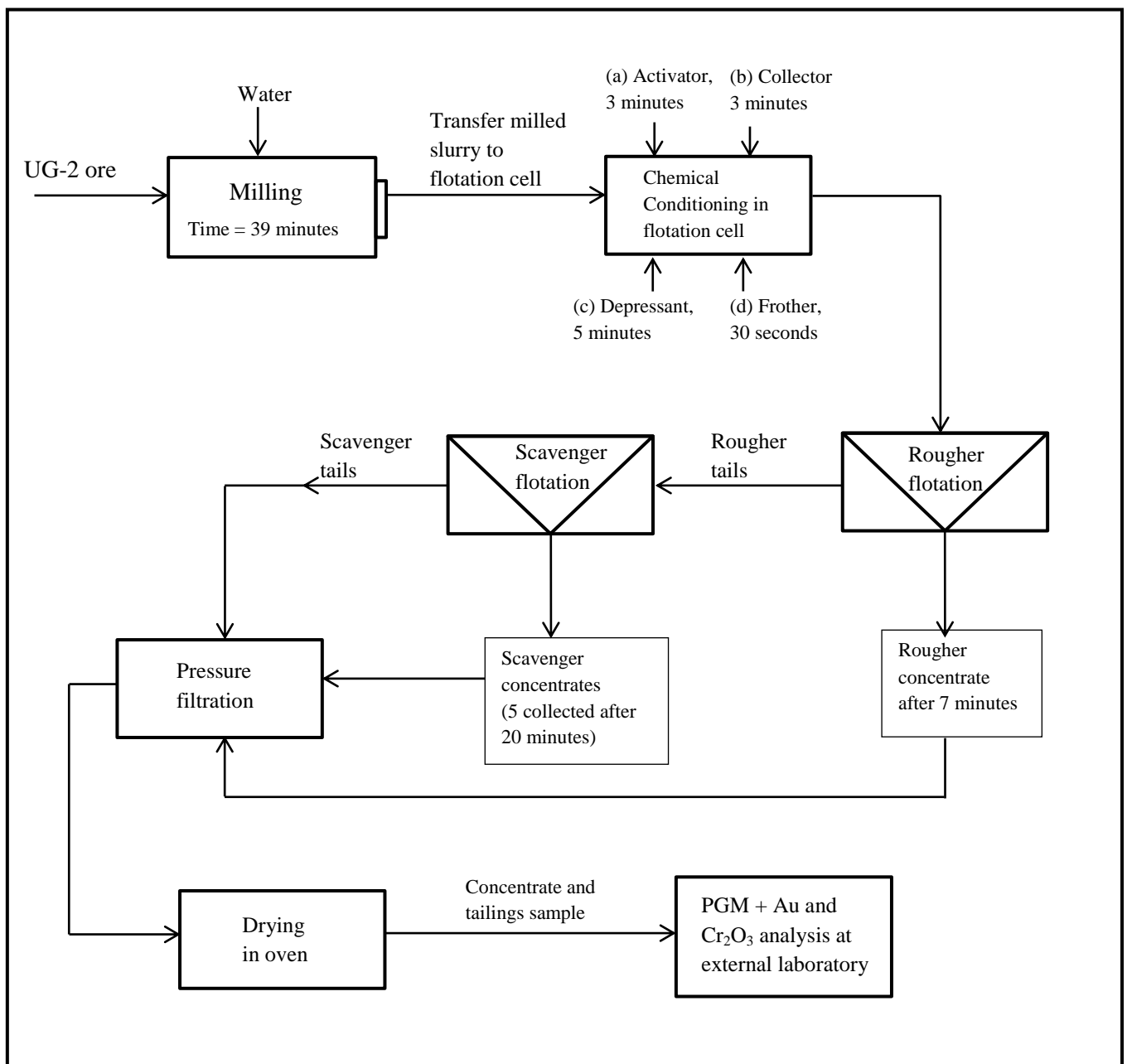
The diesel and the paraffin were used as a mixture with the frother. The combinations that were prepared were:

Diesel/frother mixture: 3 parts diesel to one part frother

Paraffin/frother mixture: 3 parts paraffin to one part frother

These mixtures were used directly and not diluted.

### 3.4. Process flow diagram of the experimental set up



**Figure 3-1: Process flow diagram.**

### **3.4.1. Process description**

Thoroughly mixed UG-2 ore (2kg dry mass) was charged into the mill, along with 2L of water. This resultant slurry was milled for a time of 39 minutes as obtained through the milling curve (to be discussed later). The milled slurry was then transferred to the 4L perspex flotation cell. The contents of the flotation cell were left to stand for approximately 15 minutes such that the excess water may be decanted easily. The water was decanted until a specific level was achieved (two pulp levels were investigated). Once the appropriate level was attained, the contents of the flotation cell were thoroughly mixed to achieve a uniform phase. Next, the chemical reagents were added and conditioned for the respective times. Thereafter, flotation of the rougher stage took place. This lasted for 7 minutes (as determined by the mass recovery for the rougher, explained later in section 3.6.3.1) and always took place via manually scraping of the froth at 15 second intervals. Only one concentrate was taken from the rougher. The tails of the rougher remained in the flotation cell. The rougher tails then became the scavenger and flotation of the rougher tails is known as the scavenger stage of flotation.

Flotation of the scavenger took place for 20 minutes within which five concentrates were collected. Timed samples of 0-3 minutes; 3-6 minutes; 6-10 minutes; 10-15 minutes and 15-20 minutes were collected in separate containers. The scavenger concentrates and tails, along with the rougher concentrate were then weighed, filtered and dried overnight in an oven. Thereafter, the dried concentrates were packaged in plastic bags to be sent to the external laboratory to be analysed for PGM and chromite. The scavenger concentrates were first combined while packaging. This was due to the minimum mass regulations of the external laboratory (i.e. in order to for a sample to be analysed, it must be above a certain mass). The first and second concentrates were combined as one concentrate. The third, fourth and fifth concentrates were also combined. A total of 3 concentrates were then sent for analysis (taking into account the rougher concentrate). The scavenger tails, which was just less than 2 kg, was split twice in a sample splitter such that a reasonable mass was obtained to send for PGM and chromite analysis.

### **3.5. Laboratory scale tests: Milling**

#### **3.5.1. Mill geometry and particle size**

A mild steel rod mill with a diameter of 240 mm was used to mill 2 kg samples of UG-2 ore to reduce the particle size. The mill was charged with 48 stainless steel rods of three diameter sizes: 16 x 20 mm, 16 x 15 mm and 16 x 10 mm. The ore samples were milled at 50 % solids in water to achieve a grind of 80% passing 75  $\mu\text{m}$ <sup>[6]</sup>.

#### **3.5.2. Milling curve**

Milling time for the ore varies depending on the type of ore being milled. Due to this variation, a milling curve was formulated to determine the time it takes to grind the UG-2 ore into a size fraction of 80% passing 75  $\mu\text{m}$ .

The milling curve was formulated using tests with sufficient wet solids for a 2 kg dry mass of sample. Water was added achieve 50% solids in the mill. The following milling times were tested: 5, 10, 20, 30, 40 and 50 minutes. The resulting slurry was wet screened at 75  $\mu\text{m}$ , using a vibrating device. The solids coarser than 75  $\mu\text{m}$  were dried and the screened to determine the % passing 75  $\mu\text{m}$ . A graph was plotted showing the relationship between the milling time and the % passing 75  $\mu\text{m}$  (Figure 4-1 in section 4.2). The tests carried out to compile the milling curve were performed in duplicate and triplicate to ensure reliability of results. The time for 80% of the solids to pass 75  $\mu\text{m}$  was interpolated to be 39 minutes. The milling procedure was standard for all flotation tests (50% water, for 39 minutes).

The data used to plot the milling curve can be found in Appendix D2.

[6] – It is common practise to mill the ore to 75-80% passing 75  $\mu\text{m}$  in secondary circuit prior to secondary flotation (Cramer et al, 2004)

### 3.6. Laboratory scale tests: Flotation

Flotation tests were carried out in a 4-litre Denver flotation cell. The ground ore from the mill (in the form of a pulp suspension) was transferred to the 4L cell. The contents were left to stand for approximately 15 minutes to decant some water to the appropriate level. Two pulp levels were investigated: 12 cm and 16 cm.

#### 3.6.1. Materials of construction of the flotation cell

The flotation cell was constructed out of perspex. The cell was designed to have a capacity to contain 4L of the flotation mixture. The perspex flotation cell, being clear, was beneficial since the pulp level could be easily seen and controlled. The behaviour of the froth could also be observed.

#### 3.6.2. Reagent conditioning

Once the appropriate pulp level was achieved, the reagents were added into the flotation cell in the following doses, as used by the plant where the ore sample was obtained.

Water was added (from a squeeze bottle) to control the pulp level. Two types of froth heights were investigated: deep froth (8 cm) and shallow froth (4 cm).

The standard (plant) reagent suite, consisting of  $\text{CuSO}_4$ , SNPX, MN49 and DOW 200, was used. The reagent dosage can be seen in the following table:

***Table 3-1: Reagents and dosages of and conditioning times of chemicals used in the flotation process.***

Reagent	Role	Dosage (g/t)	Dosage (mL)	Conditioning time (min)
$\text{CuSO}_4$	Activator	120	24	3
SNPX	Collector	150	30	3
MN49	Depressant	72	14.4	5
DOW 200	Frother	20	4	0.5

$\text{CuSO}_4$  was added first and the pulp was conditioned, or mixed in the pulp, for 3 minutes before the introduction of the rest of the chemicals. The order of the addition of chemicals and the respective conditioning times can be seen in Table 3-1. The conditioning time was sufficient enough to ensure that the reagents were properly mixed with the pulp and distributed in the cell.

Flotation was carried out in 2 stages: the rougher stage and the scavenger stage. The reagents in Table 3-1 were added prior to the rougher stage. The aim of this work is to investigate the various effects in the scavenger stage and hence reagent changes were done prior to scavenger flotation. These changes can be summarised as follows:

**Table 3-2: Reagents used in the scavenger stage of flotation (all with a conditioning time of 30 seconds).**

	<b>Dosage</b>
DOW 200 frother (used as the ‘standard’ dosage)	4 mL
Reduced frother dosage	2 mL
Diesel and frother mixture	3 drops
Paraffin and frother mixture	3 drops
No frother	-

In the scavenger part of the flotation, the ‘normal’ or standard frother dosage was replaced by either the addition of a reduced frother dosage, no frother dosage, a mixture of diesel and frother or a mixture of paraffin and frother.

### **3.6.3. Preliminary flotation tests**

Preliminary tests were done on flotation for familiarisation of aspects such as the air flow rates and mass of concentrates. This part of the experimentation was vital, to establish repeatable operating procedures. All preliminary tests were conducted in duplicate.



### **3.6.3.1. Mass recovery: Rougher and scavenger**

The mass recovery tests were needed to determine the time for the flotation experiment for both the rougher and scavenger. The mass recovery for the rougher was carried out after milling and after reagent conditioning. The method of scraping of the froth phase in 15 second intervals was used to determine the mass recovery since this was used as a basis for all flotation operations. Flotation was carried out for a total time of 30 minutes. In the 30 minutes, at varying time intervals, numerous concentrates were collected. The concentrates were filtered and dried overnight in an oven. Thereafter, the dry masses were weighed and plotted against time to achieve the mass recovery curve which is a plot of cumulative % solids recovered versus time (see Figures 4-2 and 4-3 in section 4.3). Plant data was examined and a target of 4% for solids recovery in the secondary rougher was selected. This corresponded to a time of 7 minutes according to the mass recovery curve in Figure 4-2. For all subsequent tests, the rougher flotation time was fixed at 7 minutes.

The same procedure was carried out to determine the mass recovery curve for the scavenger but the total time for the scavenger was 60 minutes because the rate of flow of concentrate in the scavenger was much slower. It is a well-known fact that the scavenger flotation demonstrates slower kinetics than the rougher and hence the longer time it takes for recovery of mass in the scavenger flotation. This is since most of the easily floating particles are floated off first in the rougher stage of flotation. Various concentrates were collected in varying time intervals and the dry masses were plotted to yield Figures 4-2 and 4-3 in section 4.3. Tests were done in triplicate to test reproducibility.

### **3.6.4. Flotation of rougher**

The rougher conditions were kept constant for all experiments. The rougher concentrate was then dried and sent for analysis.

### 3.6.5. Changing scavenger flotation performance

Flotation of the scavenger was carried out in a time of 20 minutes as obtained from the mass recovery curve. The concentrates were collected via hand scraping of the froth as the base case. For the other tests, the froth was made to overflow naturally off the lip of the cell by adjusting the air flow rate. Make-up water (tap water) was added in the tests without froth washing to maintain the pulp level. The make-up water was added in the corner of the flotation cell so as to not disturb the froth but just to maintain the pulp level.

Tests were also done with the use of a baffle that was placed at an angle in the upper section of the flotation cell. Two baffles were investigated: a short baffle that was placed at an angle of approximately 45° into the flotation cell, to simulate a ‘froth crowder’. A longer baffle was also tested, that was placed at a nearly horizontal angle. For the scavenger, the influence of many variables on flotation performance were investigated and compared to the performance achieved in the base case (hand scraping). A summary of the different variables and combinations applied to the scavenger stage of flotation can be summarised in Table 3-3.

**Table 3-3: The different combinations of flotation reagents and flotation methods that were investigated.**

Flotation Method Investigated \ Flotation Reagents Investigated	Standard frother dosage*	Diesel and frother	Reduced frother	No frother	Paraffin and frother
<b>Scraping (Base case)</b>	✓	✓	✓	✓	✓
<b>Baffle</b>	✓	✓	✓	✓	✓
<b>Baffle + Washing (Up)</b>	✓	✓	✓	✓	✓
<b>Baffle + Washing (Down)</b>	✓				
<b>No scraping, no baffle, no washing</b>	✓				
<b>Longer baffle</b>	✓	✓	✓	✓	✓
<b>Longer baffle + washing (Down only)</b>	✓	✓	✓	✓	✓

\* The standard frother dosage (as per Lonmin) described in section Table 3-1

All tests were carried out in duplicate, and some in triplicate for reliability of results. The tests carried out with scraping of the froth were done similar to the rougher stage of flotation whereby the froth was scraped in intervals of 15 seconds. It must be noted that the impeller speed was kept constant at 1000 rpm for the conditioning, rougher flotation and the scavenger flotation. The mass pulls (dry mass, expressed as %) for the batch flotation tests were controlled by changing the air flow. Each of the other types of flotation tests was required to have the same mass of solids collected in the concentrates as the standard (base case) in order to compare the differences in PGM and chromite contents that were obtained through the various methods.

### **3.6.6. Process of achieving similar masses of concentrates for each test**

Flotation tests with scraping of the froth for the scavenger were carried out in duplicate. The masses of the 5 scavenger concentrates were recorded. In the other non-scraping tests, the aim was to achieve the same mass for each of the 5 concentrates as achieved in the scraping test. This was done by using a scale that weighed the mass of the concentrate as flotation occurred. As soon as the specified wet mass was reached for a concentrate, the concentrate collection container was changed. This process was carried out for all 5 concentrates. The following figure shows concentrate being collected with the scale being used as a guide for the mass.



**Figure 3-2: Collection of concentrate without the use of scraping the froth into the collection container. In this particular case, the longer nearly horizontal baffle was used.**

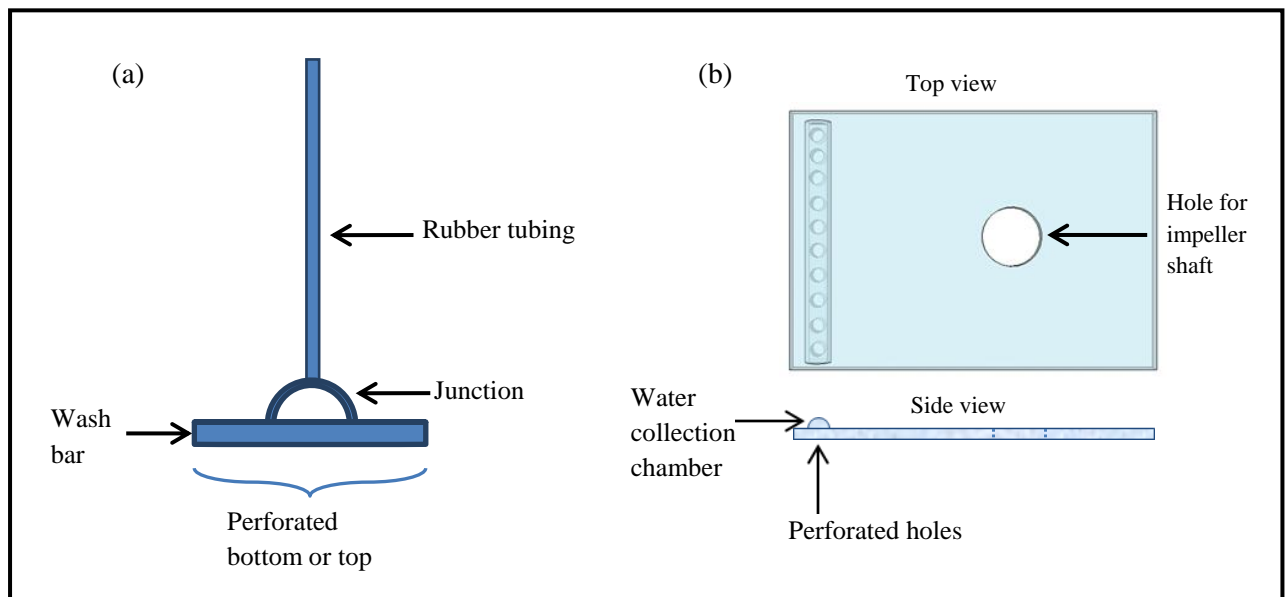
Other tests without scraping were performed similarly with the scale used to give an indication of the mass at which to change the concentrate collection container. The red arrow in Figure 3-2 shows the movement of the froth from the cell to the concentrate collection container. The natural overflow of froth was maintained by changing the air flow rate.

The effect of each of the variables (reagents and flotation procedure) will be compared to the base case experiment to determine the effect of that variable on the recovery of PGMs and the grade of  $\text{Cr}_2\text{O}_3$ .

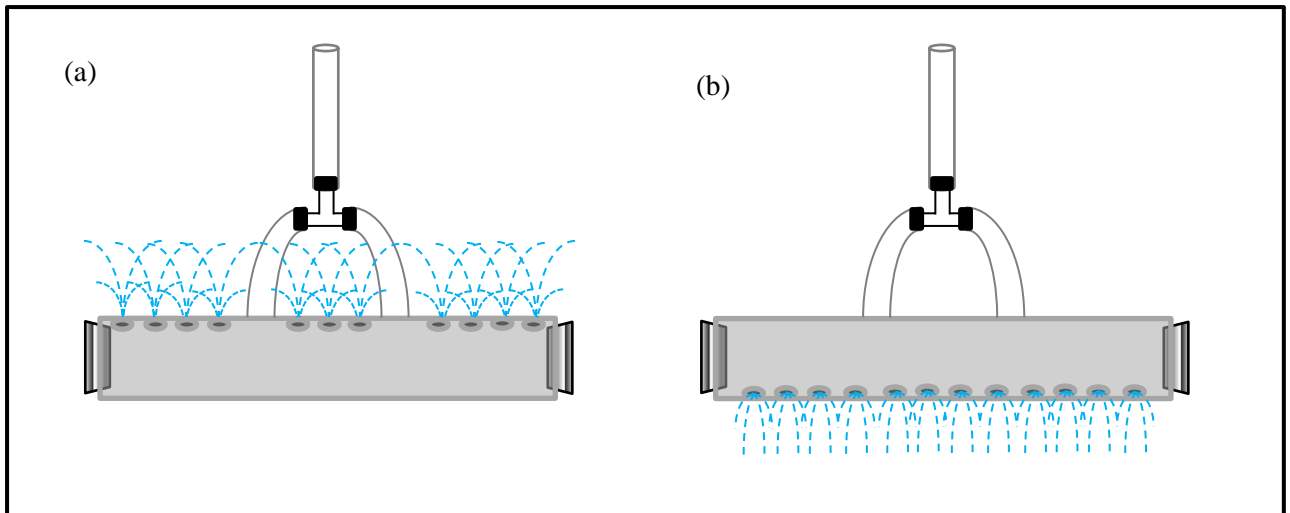
### 3.6.7. Wash water distribution devices

Three wash water distribution devices were used to carry out the experiments:

- Wash bar with holes on the underside of the bar for downward washing.
- Wash bar with holes made on the upper side of the bar for upward washing.
- Baffle with a row of holes made near the front to allow for froth washing.



***Figure 3-3: The wash water distribution devices that were used. (a) The wash bar and (b) the (longer) baffle incorporated wash water device.***



**Figure 3-4: The upward (a) and downward (b) wash water distributors. A detailed version of diagram 'a' in Figure 3-3.**

### **3.6.7.1. Construction of the water distribution device**

The wash bars in Figure 3-4 were made out of 15 cm of tubing with plugs at each end to prevent the exit of water from the ends. The tip of a sharp nail was used to puncture evenly spaced holes on the base (for the wash water device in downward washing) and the upper side (for the wash water distribution device in upward washing) of the wash bar. Plastic tubing was used to form a junction that allowed the transport of water from the source through rubber tubing and into the wash water distributor and also to keep the wash bar stable. The junction was fitted into holes that were made on the wash bar and was glued to the wash bar. A picture of one of the wash bars (for the downward washing) can be seen in Figure 3-5.

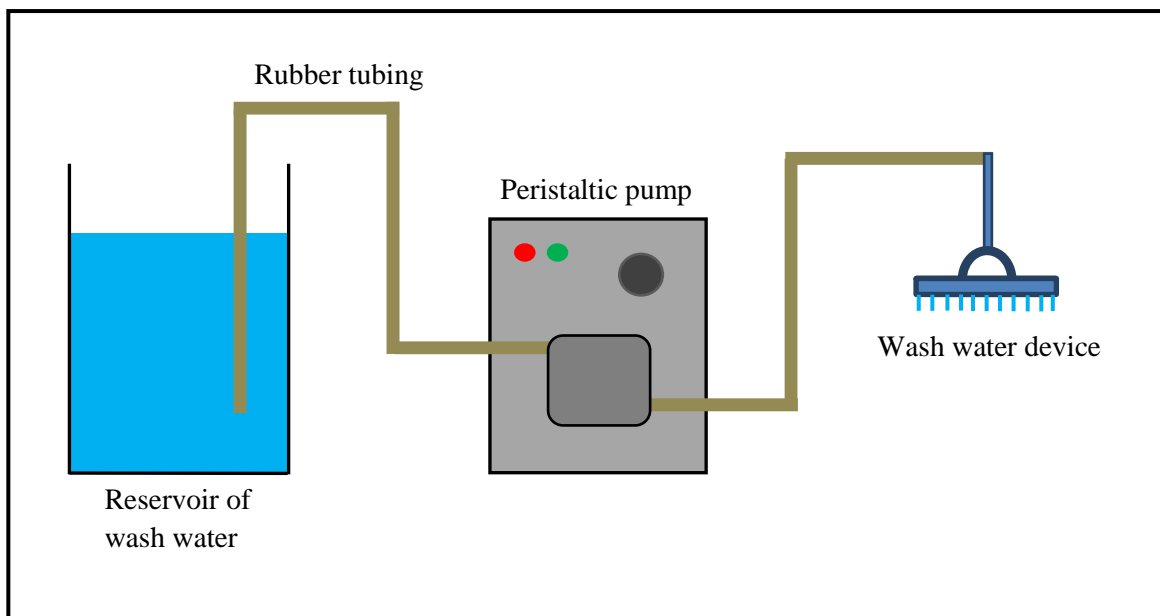


**Figure 3-5: Photograph of the wash bar for the downward washing in the flotation cell.**

The wash water distributor device/baffle in Figure 3-3 (b) was constructed out of a sheet of polypropylene to fit the cell. A plastic tube was connected to the wash water chamber. The semi-circular chamber enclosed the row of perforations through the baffle.

### 3.6.7.2. Operation of the wash water devices

Wash water was pumped from the reservoir using intermittent operation of the peristaltic pump, set at its lowest flow, to maintain a constant pulp level (i.e. the wash water replaced water lost in the concentrate). The flow of wash water from the reservoir to the wash water device can be seen in Figure 3-6:



**Figure 3-6: System of water flow from the reservoir through the peristaltic pump into the wash water device which was inserted into the flotation cell.**

### 3.6.7.3. Position of wash water addition point

The wash bar (a) in Figure 3-3 was mounted and secured onto the flotation cell after flotation of the rougher. The wash bar was placed at various positions to determine the most optimum wash water addition position. The addition point of wash water may be above the froth or submerged within the froth.

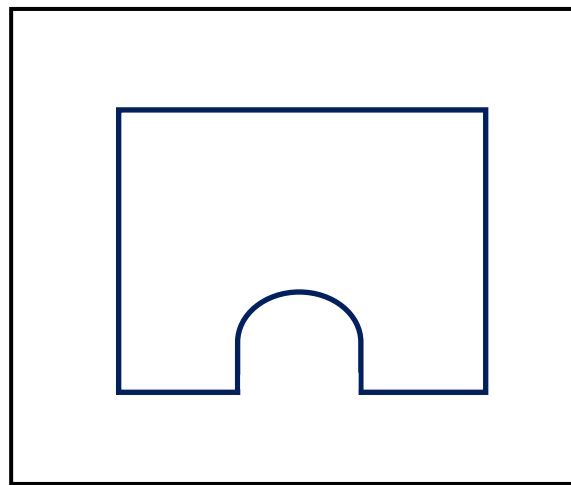
### **3.6.8. Salt tracer efficiency test on froth washing**

Tracer tests were only performed on the froth washing tests in the scavenger stage of flotation and were carried out via two methods. The first was a more visual method incorporating the use of blue dye. The second method was more quantitative since it employed a salt water solution that had a measurable conductivity. For the blue dye testing, a concentrated mixture of blue dye was prepared in a container. The colour had to be strong enough to be seen even in the dark brown colour of the pulp. The froth washing tests were carried out and the path of the wash water was observed.

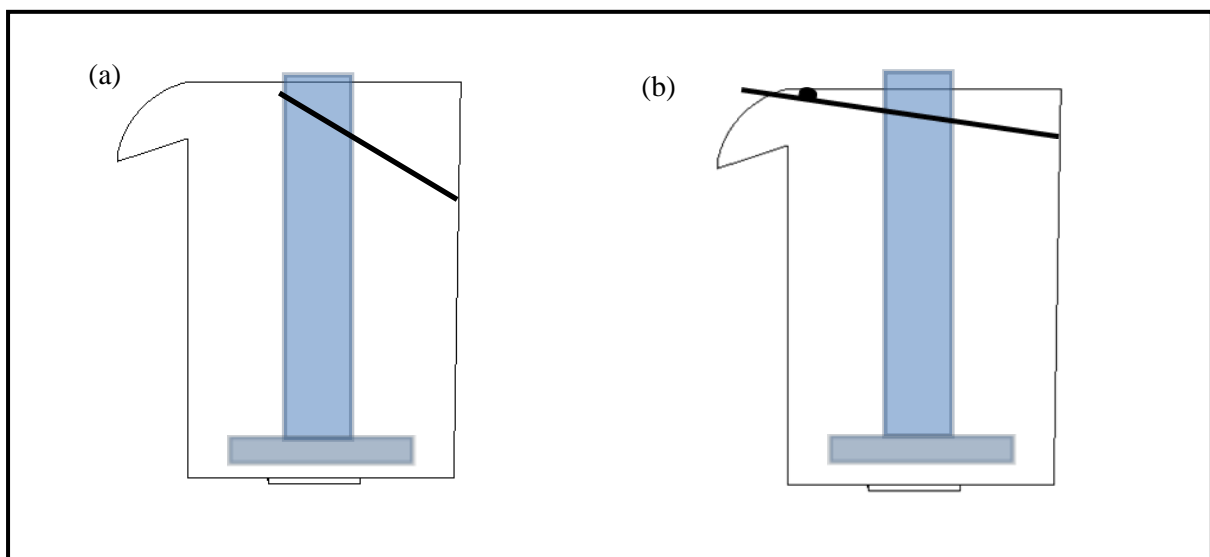
The salt water solution tracer tests were used to measure the efficiency of froth washing. This was done by incorporating the use of a salt water solution as the wash water in the reservoir. After flotation, the concentrates were filtered and their filtrates were collected in beakers and analysed for conductivity using the EC215 conductivity meter. The conductivity readings were measured by inserting the conductivity probe into the various beakers of filtrates collected. The efficiency of froth washing was calculated by comparing the salt concentration in froth samples with the salt concentration in the cell. Sample calculations of the efficiency of froth washing can be seen in Appendix B2. The tracer tests were done in duplicate to confirm consistency. The conductivity meter was calibrated daily using the standard conductivity solution (12.88 mS, H17030L) to ensure accuracy of results.

### 3.6.9. Baffles

Two types of baffles were used (longer and shorter). The longer baffle that incorporated the wash water device was previously described (see Figure 3-3, b). The shorter baffle was constructed of a sheet of 15 cm by 13 cm aluminium. Double sided foam tape was attached to the edges of the baffle to prevent air leaks and to protect the flotation cell from being scraped by the aluminium. The shorter baffle was placed at an angle of approximately  $45^\circ$  to the horizontal, behind the impeller shaft. A diagrammatic representation of the shorter baffle can be seen in Figure 3-7. The positions that both the baffles were placed in the flotation cell may be seen in Figure 3-8.



**Figure 3-7: The shorter baffle that was used.**



**Figure 3-8: Positions of the shorter and longer baffles that were used. The space in front of the shorter baffle was used to place the wash water distributor device (a) in Figure 3-3.**



### **3.6.10. Concentrate collection procedure**

Five concentrates were collected at intervals of 3 (0-3 min), 3 (3-6 min), 4 (6-10 min), 5 (10-15 min) and 5 minutes (15-20 min). The mass of the water and solids was measured for all concentrates. Concentrates and tails were filtered, dried overnight in an oven, weighed again for the dry mass and then packaged in plastic bags to be sent for analysis. PGM + Au and chromite ( $\text{Cr}_2\text{O}_3$ ) analysis was done at an external laboratory.

### **3.6.11. PGM + Au and $\text{Cr}_2\text{O}_3$ analysis**

Five scavenger concentrate samples were combined into two concentrates – concentrates 1 and 2 and concentrates 3, 4 and 5 were combined. These, along with the rougher concentrate were placed into plastic bags and weighed. The tailings samples (approximately 2 kg) were split twice in a sample splitter to obtain a mass of approximately 500 g. The samples then were sent to an external laboratory for 3 PGE+Au and  $\text{Cr}_2\text{O}_3$  analysis. The procedure that was carried out in the external laboratory for the analyses can be seen in Appendix F.

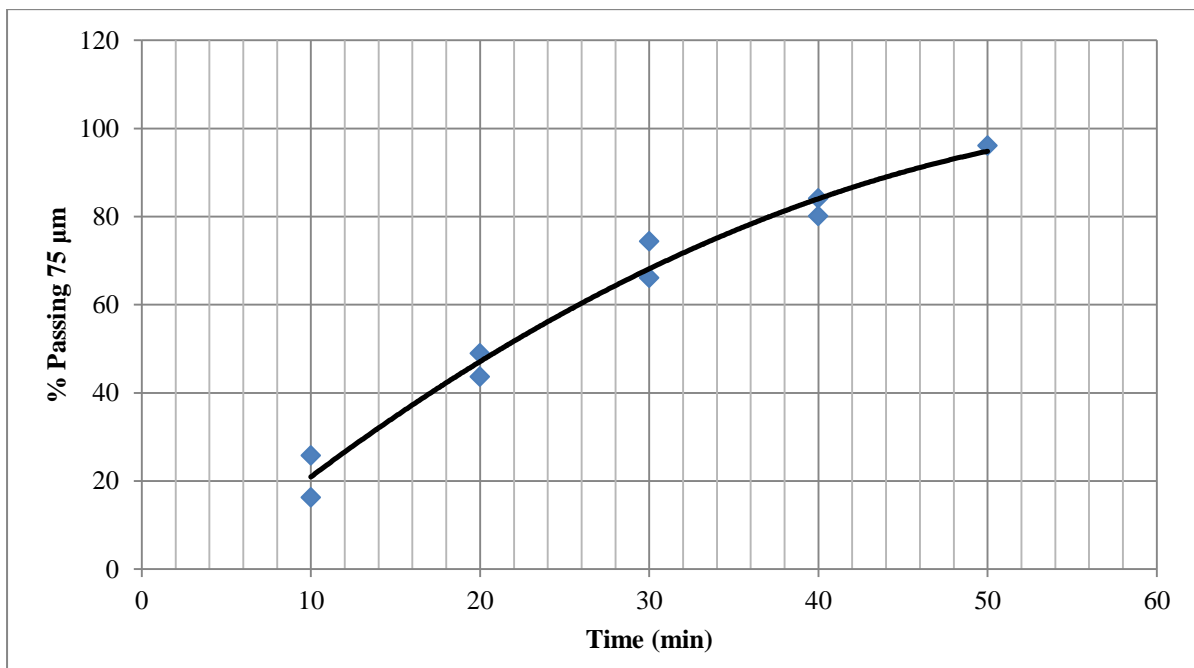
## CHAPTER 4: RESULTS AND DISCUSSION

### 4.1. Moisture content test

The ore for experimentation was received in buckets with a slight pool of water on the surface indicating that the ore was wet. A moisture calculation was done to determine the % moisture of the ore. A dry solids mass of 2 kg was required to be milled hence the relevance of the moisture content. The moisture content of ore was found to be 10.44 %. Therefore for 2 kg of dry solids, 2.23 kg of moist ore was required to be milled. Water was added to make up a total of 2kg.

### 4.2. Milling curve

Prior to flotation, the ore had to undergo size reduction via milling for liberation of the values. The targeted size range was 80 % passing 75  $\mu\text{m}$  which is common in secondary circuit flotation. The milling curve (Figure 4-1 below) was plotted in order to determine the milling time for 80 % of the particles to pass the 75  $\mu\text{m}$  sieve tray.

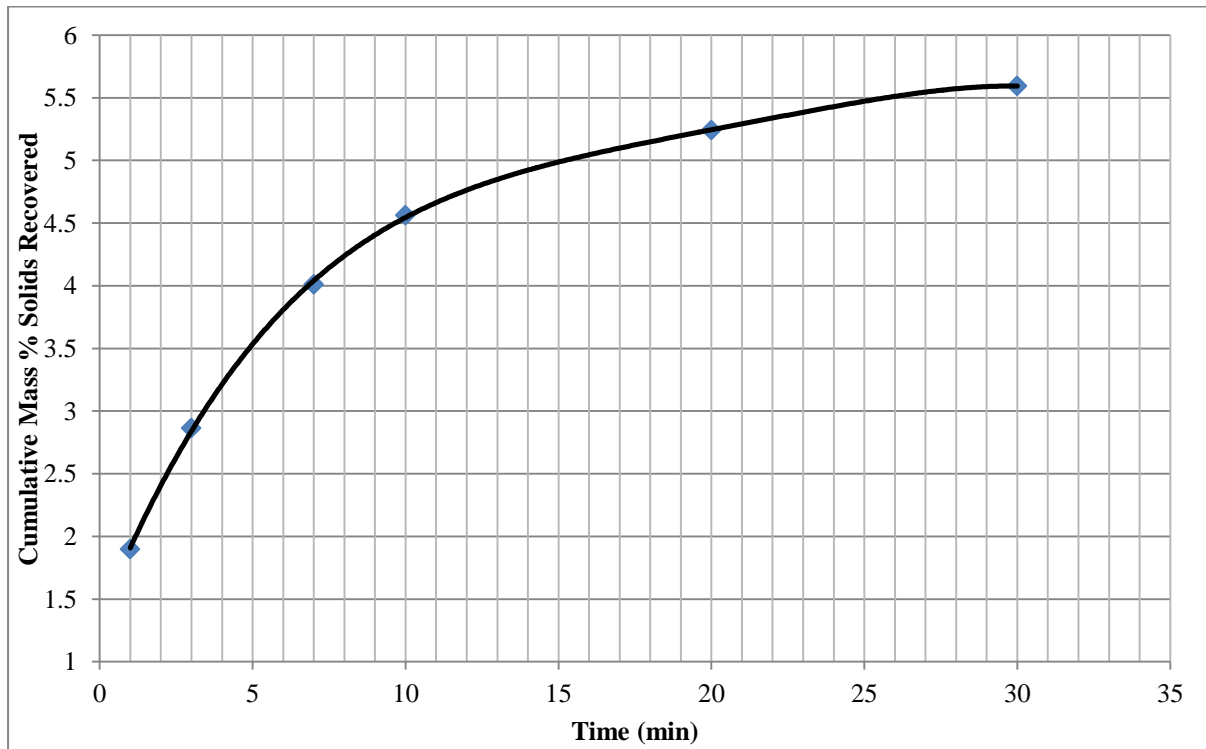


**Figure 4-1: Milling curve for the UG-2 ore.**

A polynomial was fitted to the data points for interpolation, since the data showed non-linearity. According to the graph, for 80 % of the particles to pass the 75  $\mu\text{m}$  screen, a time of 39 minutes must be allowed for milling. Subsequent tests for flotation therefore underwent 39 minutes of milling prior to flotation.

### 4.3. Rougher and scavenger mass recoveries

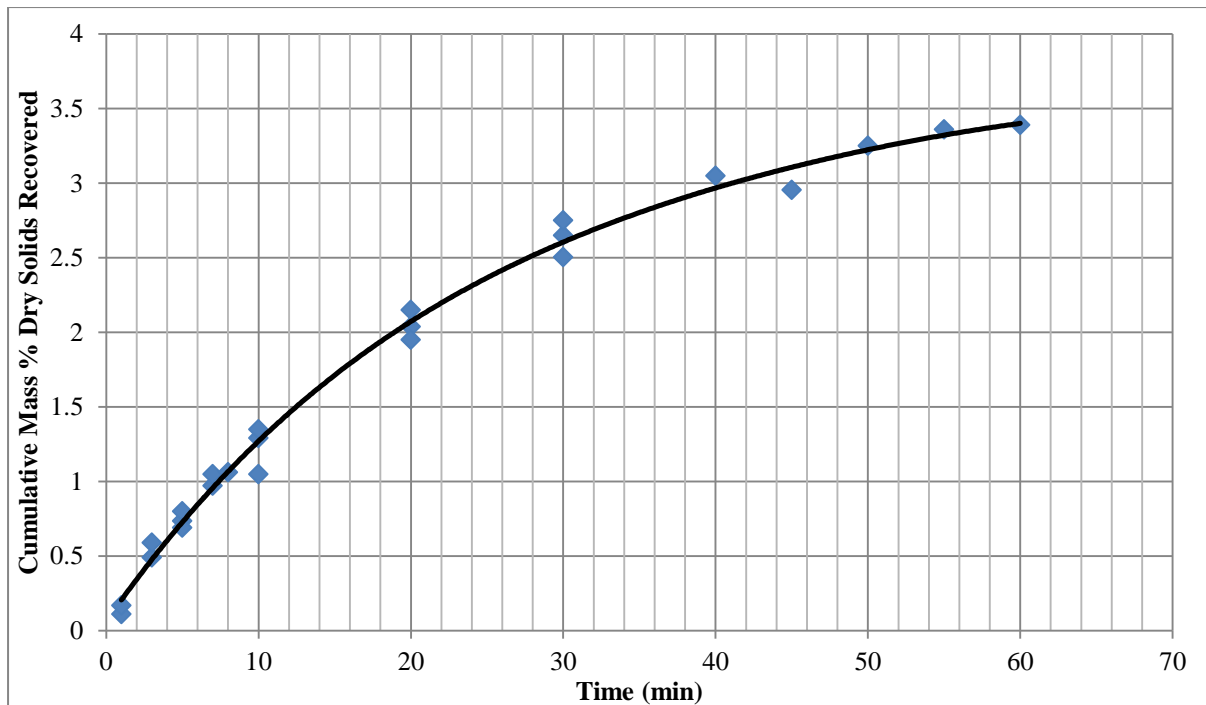
Mass recoveries were obtained to determine the time for flotation of the rougher and scavenger flotation stages. The mass recovery is a plot of cumulative mass % of dry solids recovered in the concentrate versus time.



**Figure 4-2: Mass recovery for rougher flotation.**

The mass recovery in the rougher shows an initial fast increase in the amount of solids recovered thereafter followed by a decline in the solids recovery as represented by the levelling of the graph. The rapid rise in gradient is due to recovery of the readily floating particles initially. The reason for the levelling off is due to the dissipating of the froth layer (the froth dies out) hence less mass overflowed and was recovered. As indicated by the graph, the highest % of solids that could be recovered was approximately 6%. A value of 4 % of solids recovered was chosen as an appropriate recovery for the rougher which, according to the Figure 4-2, corresponds to a time of 7 minutes.

Scavenger flotation, in any flotation plant, functions to recover the remaining valuable minerals that were lost in the tailings of the rougher. Flotation in the rougher recovers the bulk of the valuable minerals. The scavenger stage inherently tries to improve the recovery of valuable minerals from the ore but the flotation is not very selective with significant recovery of gangue minerals by entrainment.



**Figure 4-3: Mass recovery for scavenger flotation.**

From Figure 4-2, the maximum to be recovered was approximately 6%, 4% of which was recovered in the rougher stage thereby leaving behind 2% to be recovered in the scavenger. According to Figure 4-3 a solids recovery of 2% corresponds to a flotation time of approximately 20 minutes. Thus flotation of the scavenger was carried out for 20 minutes for subsequent testing. Increasing the mass pull will not necessarily result in the recovery of the lost PGMs of the rougher tails. Therefore the mass recovered was not aimed at a markedly high mass.

#### 4.4. Flotation tests

The various configurations and chemical reagents investigated in this work may be summarised in Table 3-3 in section 3.6.5 with the standard tests being done with scraping of the froth every 15 seconds. The other tests were compared to the standard test to see the effect of each of the variables on the grade and recovery of PGMs and  $\text{Cr}_2\text{O}_3$  content. All tests were done in duplicate to confirm validity of the results.

##### 4.4.1. Manual scraping of the froth – Base case

Using the flotation times that were obtained from the mass recoveries (rougher – 7 minutes; scavenger – 20 minutes); flotation with the aid of scraping paddles was carried out with adjustment of the air flow rate as required. The froth was removed manually by using two scraping paddles (as can be seen in the figure below) that covered the entire froth surface area at intervals of 15 seconds. Flotation by means of scraping of the froth is commonly used in laboratory tests as a standard method. Therefore it was used as the base (or reference) case in this investigation. The other tests were then carried out based on the mass achieved with the scraping test. The aim was to achieve similar mass pulls for each concentrate (for the respective tests) so that the PGM content was based on a similar mass of concentrate.



**Figure 4-4: Flotation test with the use of scraping paddles. Two paddles were used to scrape off the surface of the froth in constant intervals of 15 seconds according to laboratory flotation standards.**

#### **4.4.1.1. Reasons for replacing the scraping method**

Industrial flotation cells do not have periodic scraping of the froth; instead it comprises of naturally overflowing froth that flows from the flotation cell into the launders that are on the side of the flotation cell. In order to better represent the industrial flotation cell, scraping of the froth had to be replaced by naturally overflowing froth in the laboratory.

#### **4.4.1.2. Implication for the scraping of the froth to be used as a reference basis**

The flotation test with scraping of the froth was used as a standard. This implies that the mass recovered in the concentrates was used as a standard. The PGM and  $\text{Cr}_2\text{O}_3$  contents in the scraping tests were also used as a standard (or reference). All other non-scraping tests were compared to this standard. The real comparison was in the cumulative PGM recovery and cumulative  $\text{Cr}_2\text{O}_3$  contents at similar mass recovery.

It must be noted that the rougher was always scraped for every test that was carried out and that similar masses were achieved for each rougher test because the rougher was floated under the same conditions all the time. No modifications made in the rougher stage, since the scavenger stage is the focus of this investigation.

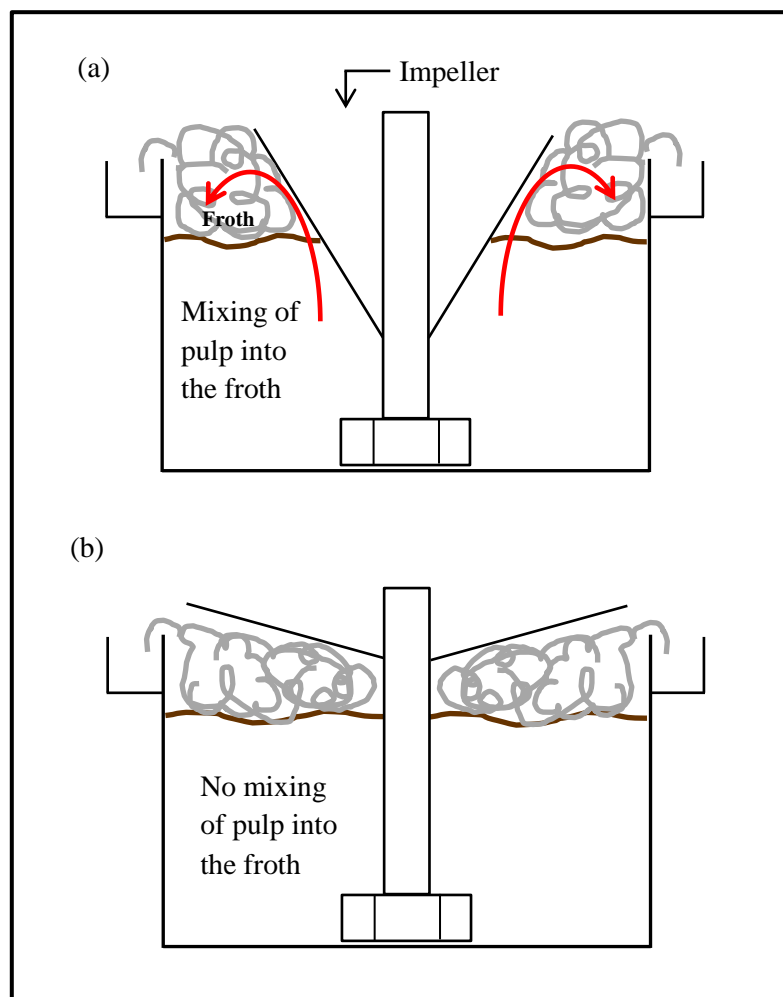
Another very important reason to eliminate the periodic scraping of the froth was that the washing was applied as continuously as possible. Hence the short (or long) baffles were used for tests of froth washing, to provide a continuous flow of froth.

#### **4.4.2. Tests with no scraping**

Tests with no action/enhancement performed on the froth phase (i.e. no scraping, no baffle and also no froth washing) were carried out to compare with the base case. The tests were carried out by manipulation of the air flow rate in order to enable a natural flow of froth out of the flotation cell, over the cell lip, and into the collection container. The aim was to get the same mass of concentrates as obtained in the base case. In these tests, it was noticed that an area behind the impeller was relatively stagnant and did not move forward towards the lip to be collected in the container. This stagnant area probably contributed to loss of valuable minerals. The angled baffle was then constructed and used to move the froth forward from the back of the flotation cell.

#### 4.4.3. Use of baffle: Types investigated

The shorter baffle was designed to move the froth from the back of the cell to the front to be collected. In preliminary tests, this baffle demonstrated good performance. Hence, another longer baffle was designed, constructed and investigated. The longer baffle was built to include a means of washing the froth hence perforations were made on the baffle near the front to enable this. The gentler angle was investigated in order to reduce the possibility of mixing of the pulp with the froth. In industry, current baffles run from deep in the pulp at the base of the impeller to the surface of the froth at an angle of about 45°. The common term is a 'froth crowder'. The rising bubbles accumulate on the baffle, creating an upward flow of pulp, into the froth phase, thereby leading to a local reduction in grade. The pictorial representation of this phenomenon can be seen in the following figure.



**Figure 4-5: The effect of a baffle placed at a larger angle (a) and with a smaller angle (b) in an industrial circular flotation cell. (a) is currently in place in industry while (b) is a new idea and is investigated on a laboratory scale in this project.**

The mixing of the pulp, which is associated with the gangue material, results in a reduction of the grade of the concentrate obtained because unwanted material now enters the froth. The steep angle promotes mixing. The near horizontal position was investigated due to the reduced amount of mixing caused by the slighter angle.

#### **4.4.4. Froth washing tests**

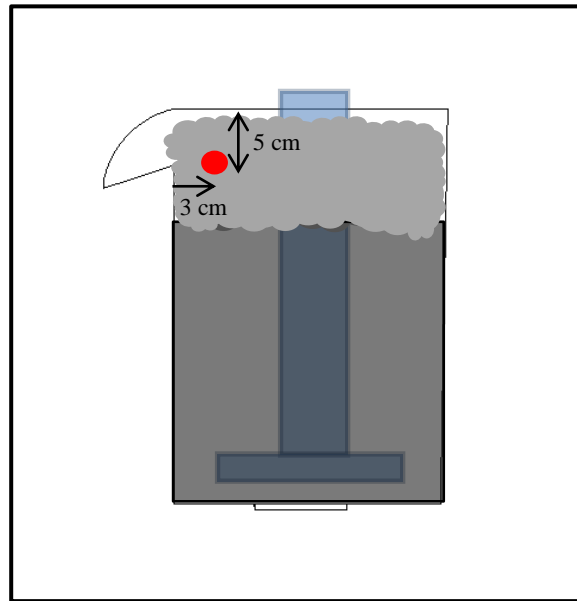
Froth washing tests were carried out with the aim of reducing the  $\text{Cr}_2\text{O}_3$  content of the flotation concentrates. Upward and downward washing tests were investigated. Upward washing was investigated to determine whether it provides a better washing effect than downward washing. The improved effect was assumed due to the motion of the water in upward washing. The upward washing differed from the downward washing by having the perforations on the upper side of the wash bar which was always placed inside the froth phase (for downward washing, the wash bar may be placed above the froth phase or within). This had the effect of the wash water moving upward into the froth, washing the froth (coating the bubbles), and then moving down again due to gravity. This upward spurting motion was assumed to cause a favourable washing effect. Salt water was used, so that the entrained liquid displacement could be measured at the same time as the actual effects on PGM recovery and chromite content. The results of these tests were compared to the base case. The addition of wash water caused the froth to be visibly cleaner (lighter in colour). Also, the wash water had an effect of stabilising the froth which was evident by smaller bubbles being formed in the froth.

##### **4.4.4.1. Position of froth washing device**

The froth washing device specifically referred to at this point was the wash bars in Figure 3-4 in section 3.6.7 of the experimental procedure. The best position for the wash bar was determined through weighing the dry solids and measuring the water displacement by salt solution. PGM and  $\text{Cr}_2\text{O}_3$  analysis was not carried out to determine the best position for the wash water device since numerous tests were carried out which would have been uneconomical if all the tests were to be sent for PGM and  $\text{Cr}_2\text{O}_3$  analysis.



The positioning of the wash water injection point was optimised and the optimal position was found to be near the overflow lip and within the froth (as represented in the figure below). This was for both (upward washing and downward washing) wash bars investigated. Specifically, for this investigation with the 4L flotation cell, it corresponded to 3 cm away from the lip and 5 cm below the top of the flotation cell.



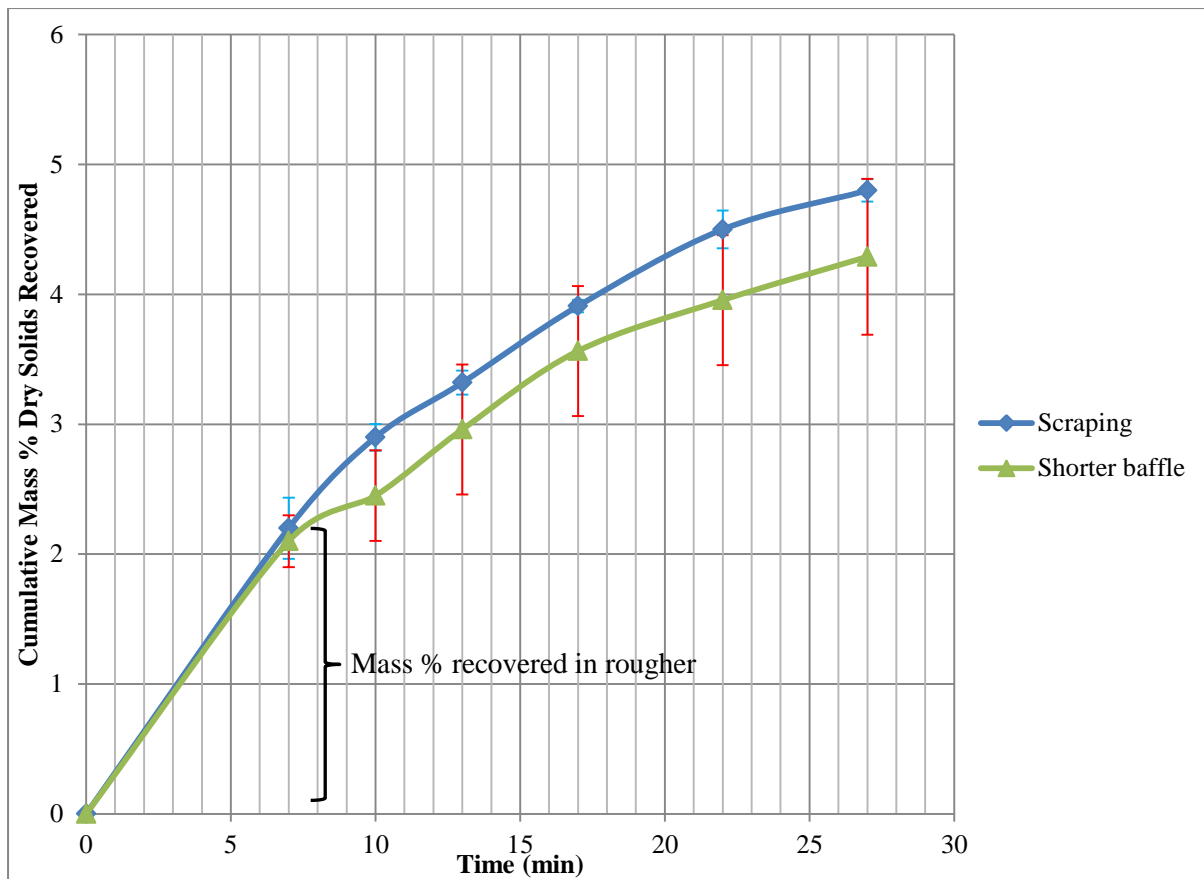
**Figure 4-6: Optimised position for the wash bar (as in Figure 3-4) represented by the red dot.**

Numerous tests were carried out in determining the optimum position for the wash bar. Tests were conducted with the wash bars above, on and within the froth phase. Based on visual observations, the washing was found to be most effective when the wash bar was placed within the froth.

#### **4.4.5. Change of scavenger frother dosage**

The standard frother dosage of 20 g/ton (or 4 mL) was replaced by four other variations namely: a diesel and frother dosage, a reduced frother dosage, no frother dosage and paraffin and frother dosage. The mixture with diesel and frother was investigated since it is known that diesel enhances the action of the collector. The same may be said for the paraffin. The idea for a reduced frother dosage stemmed from the fact that with a lower amount of frother going into the cell, a higher air flow rate will have to be applied to cause the froth to flow over the lip naturally. This higher flow rate of air would then cause a higher selectivity inside the flotation cell and hence increase the flotation of the valuable particles. The tests with no frother dosage were carried out to amplify this notion.

#### 4.5. Mass recovered for the flotation tests



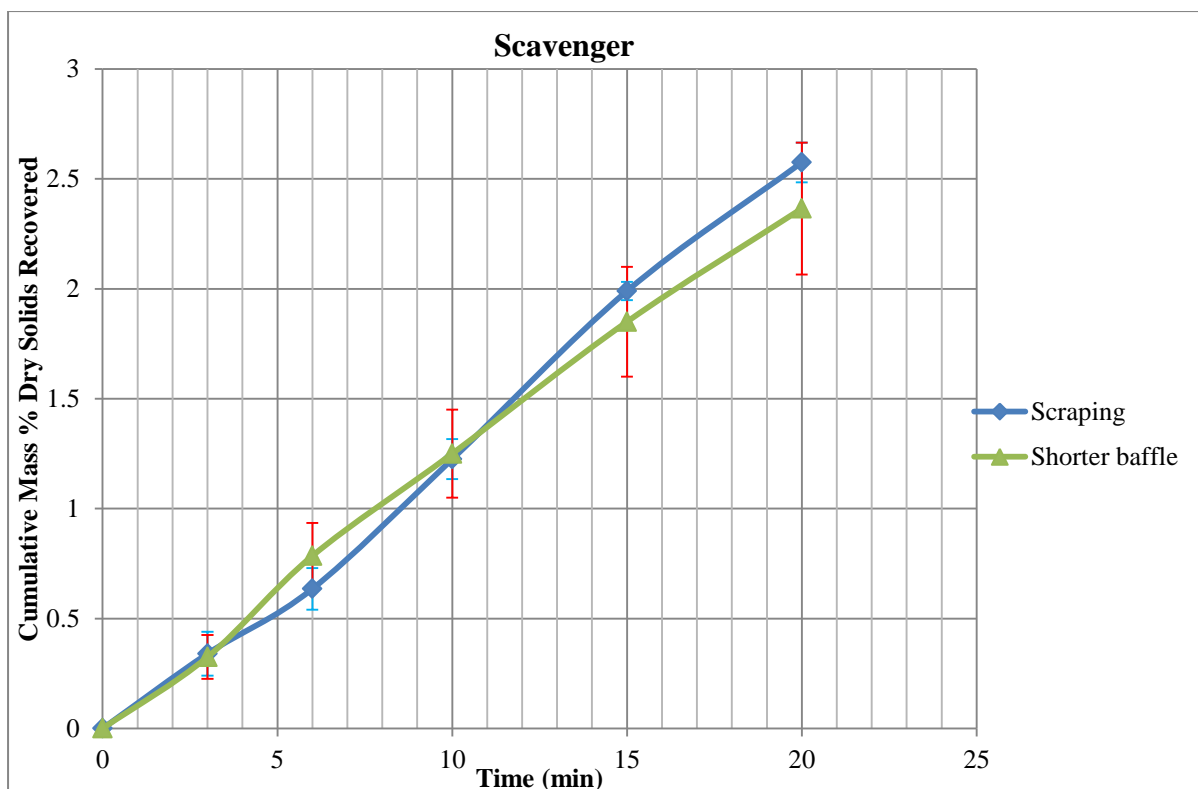
***Figure 4-7: Cumulative mass % dry solids recovered versus time for selected flotation tests (standard frother concentration).***

The test with scraping of the froth phase in the scavenger was used as a base case. The masses of the remaining types of tests were based on this mass. The aim was to attain the same mass recovered in the concentrates for all the tests such that results of the PGM + Au content will be based on similar units of mass. To attain the similar masses recovered in the concentrate, the air flow rate (via the rotameter) was adjusted.

The concentrates for the remaining tests (with baffle, with washing, without washing and baffle) were then compared to the base case. This was done by on-line measurement of the wet mass of the concentrate with the use of a scale. The concentrates were filtered and dried with the dry masses used for the data analysis. A graph of cumulative mass % of dry solids obtained with time was plotted to show that the masses obtained via the various methods were similar. This was for the purpose of the second part of the analysis which was carried out via the external laboratory (PGM + Au and  $\text{Cr}_2\text{O}_3$  analysis). Figure 4-7 shows similarities in the trends of the mass % recovered in the tests. It also shows that the 'rougher point' is fairly similar for both tests (indicated by coinciding first points). This is the first of the mass recovery curves that was obtained from experimentation and was necessary to show that the masses obtained in the various flotation tests were similar to the base case. Similar results in mass recoveries were achieved in the other types of tests (see Appendix C1).

The data points plotted were the average of duplicate and, in some instances, triplicate tests for a particular flotation situation. Error bars were used to quantify the tests that were used to create the average. The upward and downward error bars were obtained by using the Microsoft Excel built-in function *CONFIDENCE.NORM* with the level of significance of 0.05 (by convention) and standard deviation as the arguments. This function was used to obtain the 95 % confidence interval error bars for each of the types of tests in Figures 4-7 and 4-8. The shorter error bars of the scraping tests show that they are repeatable. The longer error bars obtained for the shorter baffle tests show that the results were varied. The same procedure was carried out in the other tests (upward washing, downward washing, and no baffle + no washing) but the results are not shown here due to the overlapping of error bars crowding the diagram. The plots of all the types of tests, however, may be seen in Figure C-1 of Appendix C1. The more important results will be in the form of PGM recovered versus mass recovered. Figure 4-7 was used as a means to demonstrate that the mass recoveries were similar to the base case (scraping).

The different methods of flotation caused minor amounts of variance with the largest error bar being achieved for the final data points. The length of the error bar corresponded approximately to 0.6 mass % which is a reasonable amount to state that the mass percentages are similar. As can be seen in Figure C-1 in Appendix C1, the mass recovery in some tests were reduced significantly, despite efforts to keep the froth flowing by increasing the flow of air. This may be good for gangue rejection.



***Figure 4-8: Cumulative mass % of dry solids recovered with time for selected flotation tests for the scavenger only (standard frother dosage).***

The remainder of the scavenger plots can be found in Appendix C2. The results for the platinum group metal and  $\text{Cr}_2\text{O}_3$  analysis can be seen in the figures that follow.

#### 4.6. PGM and chromite analysis

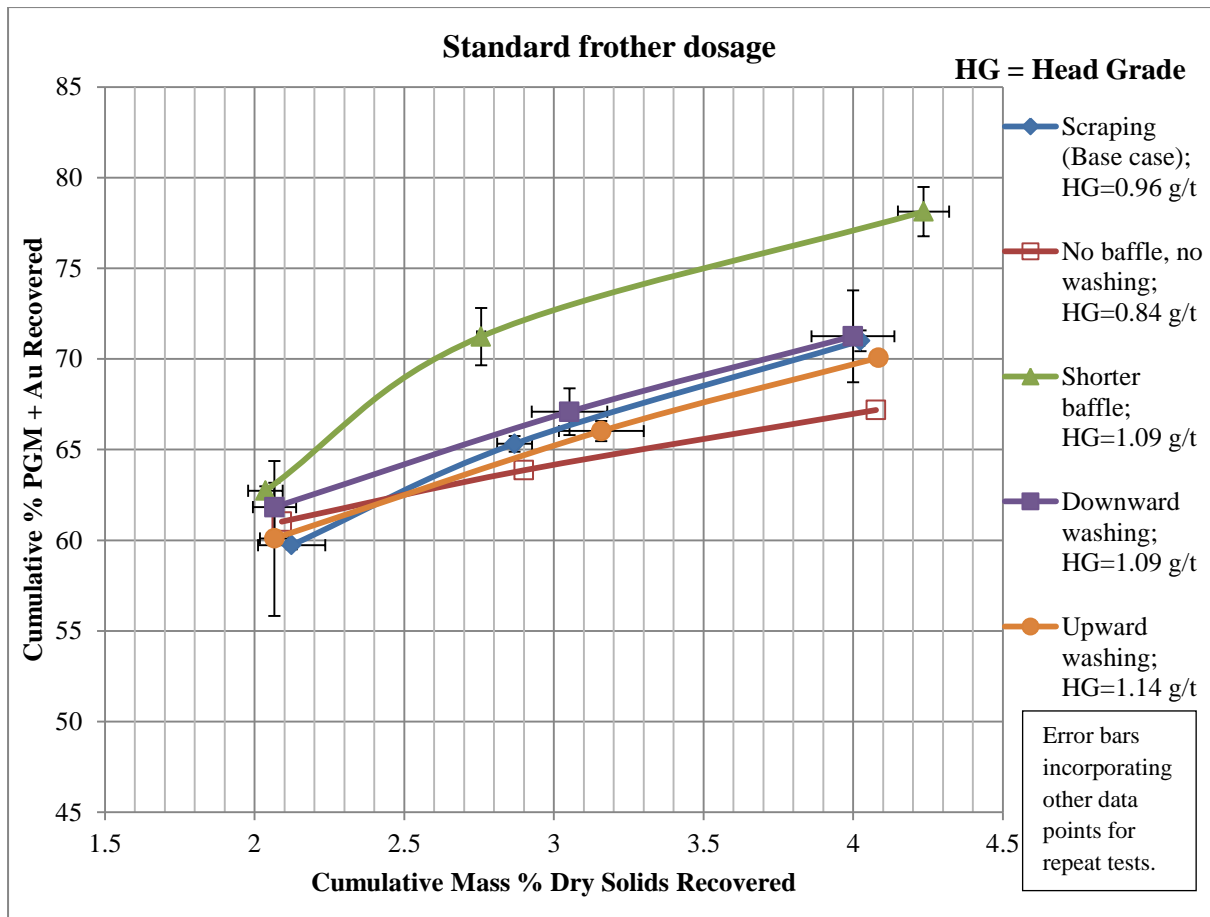
PGM and chromite analysis was necessary to determine the effect of the various methods and reagents on the flotation in the scavenger. The PGM and chromite analysis were not done at the university due to insufficient facilities, hence the samples were sent to an external laboratory to be analysed. The dry masses of the flotation concentrates were packaged into plastic bags and sent to the external laboratory. The external laboratory analysed precious metals in the form of PGM + Au, and chromite in the form of  $\text{Cr}_2\text{O}_3$ , hence the results were presented accordingly. PGM + Au analysis was carried out through fire assay (procedure explained in Appendix F1).  $\text{Cr}_2\text{O}_3$  was analysed through ICP analysis (Appendix F2). The PGM + Au and  $\text{Cr}_2\text{O}_3$  analysis was relatively expensive which placed a restriction on the amount of samples that could be analysed for PGMs and  $\text{Cr}_2\text{O}_3$ . This consequently put a restriction on the amount of repeat samples that could be analysed. Four batches of samples were sent and the total cost of the analysis was R 75 853. Most of the tests were done in duplicate, although some tests were not, due to the economics. The graphical representation for duplicate tests was a filled data point, and for a non-repeated tests, an open figure. This depiction was used for all the figures in the results and discussion section.

The flotation concentrates were, by nature, small in mass, and consequently a combination of the concentrates were required to obtain a suitable mass to conduct the analysis as per requirement of the external laboratory. One concentrate of adequate mass was obtained from the rougher flotation; hence no combinations were done for the rougher. In the scavenger flotation five concentrates were collected overall. Concentrates 1 and 2 were combined as the first concentrate, and concentrates 3, 4 and 5 were combined to give the second concentrate. This can be represented by the three data points obtained for each test in the subsequent figures to follow (one point from the rougher and two points from the scavenger). The closed-figure data points represent the tests performed in duplicate and the open-figure data points represent the non-duplicate tests. This convention was used throughout for purposes of conformity.

Table 3-3 in section 3.6.5 contains the various configurations investigated in this project. In the first column: scraping; baffle; baffle + washing (upward and downward); no baffle + no washing; longer baffle and, longer baffle + washing (downward) were all known as flotation ‘methods’. In the first row: standard frother dosage; diesel and frother; reduced frother; no frother, and, paraffin and frother were the flotation reagents investigated in the scavenger stage. The base case situation was the scraping test with the standard frother dosage. All the other flotation methods and chemical reagents were compared to the base case to determine any improvement in PGM recovery and decrease in  $\text{Cr}_2\text{O}_3$  content.

Cumulative % PGM + Au recovered against cumulative mass % dry solids recovered in the flotation tests was used as a measure of PGM recovery. Cumulative %  $\text{Cr}_2\text{O}_3$  content against cumulative mass % dry solids recovered was used as a measure of  $\text{Cr}_2\text{O}_3$  grade. The recovery of PGMs and the grade of  $\text{Cr}_2\text{O}_3$  are represented as overall recoveries and grades because recoveries for just the scavenger have no definite meaning. The range of recoveries experienced in the various tests will be due to the changes (method of flotation, chemical reagents) in the scavenger only. The rougher flotation was uniformly carried out for all the tests such that negligible variations in recovery of PGM or  $\text{Cr}_2\text{O}_3$  content are due to this.

The plot of PGM + Au for the standard frother dosage can be seen in the following figure. The figures will be plotted in pairs, namely, a plot of PGM + Au recovery and thereafter a plot of  $\text{Cr}_2\text{O}_3$  grade for the various combinations that were investigated. The terms PGM + Au and PGMs will be used synonymously in the following part since the external laboratory carried out the PGM analysis in the form of PGM + Au. The same will be applied for chromite and  $\text{Cr}_2\text{O}_3$ .

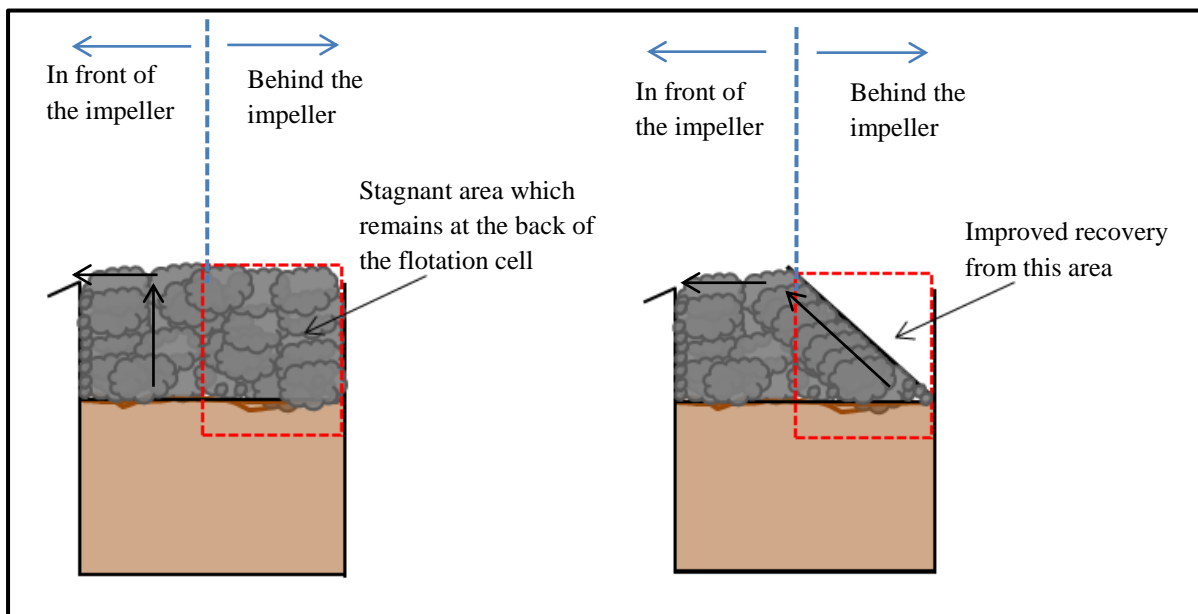


***Figure 4-9: Effects of washing and the short baffle, at a standard frother dosage.***

Figure 4-9 shows that washing did not make a significant improvement to PGM recovery, compared to scraping, at equivalent mass recoveries. However, the results with the baffle were significantly better. Figure 4-9 also shows the average PGM head grade (HG) for each data set (sample calculations for head grade may be seen in Appendix B3). There is a natural variability in the PGM content, with recoveries tending to increase as the head grade increases. This has to be taken into account, when comparing performance. Considering an equivalent mass recovery of about 4%, an improvement in recovery of PGM + Au by 6 % from the base case, as displayed in the test with the shorter baffle, is extremely significant in the minerals processing industry as it may correspond to millions of rands worth of income. The marked improvement in recovery through the use of the baffle was unexpected and hence an investigation into the use of a longer baffle was undertaken. The error bars indicate that the difference in the curves was significant.

The recovery for the rougher is inherently high (60 – 63 %) since the rougher is known to recover nearly all the fast floating particles leaving behind the lower grade tailings to be floated in the scavenger.

The wash water, in the froth washing tests, may have washed away valuable PGMs as well as gangue particles. This contributed to a reduction in recovery and hence an overall insignificant improvement to the recovery of PGMs as compared to the base case. The washing tests in Figure 4-9 were all performed with the use of the shorter baffle as well and hence the lower recovery achieved in the washing tests (as compared to the test with the baffle only), may be attributed to the washing action removing the valuable particles. These results therefore showed that the net effect of froth washing was negative. The results of tests with no froth action (no baffle + no washing) were significantly worse than the standard scraping method. The baffle, on the other hand, produced significantly better results, as discussed above. This is due to the baffle recovering the froth, containing the valuable minerals, that was otherwise stagnant in the tests with no action performed on the froth phase, and allowing natural drainage to occur.



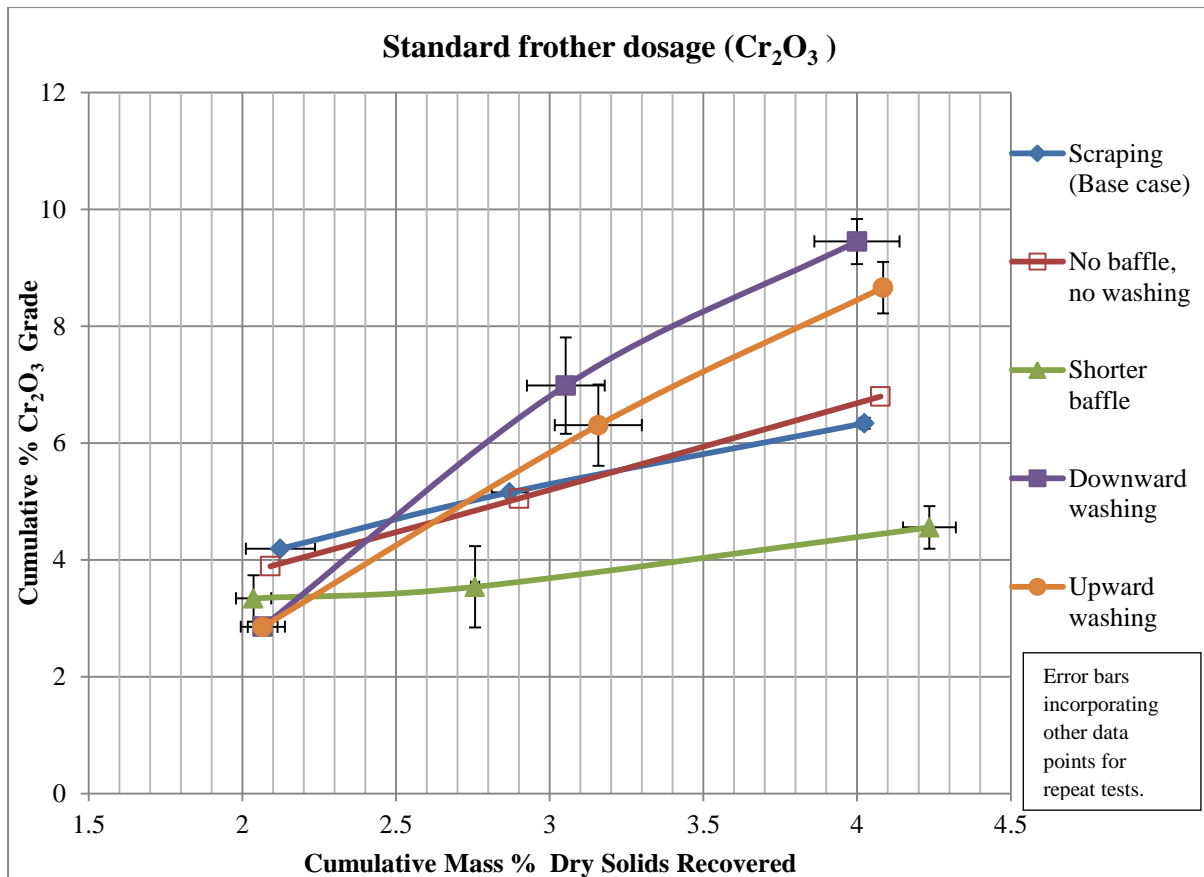
**Figure 4-10: Depicting how the angled baffle improves recovery. The black arrows show the direction of the moving froth. The baffle allows the froth at the back of the flotation cell to move forward. The baffle was constructed to include the impeller shaft.**

Overall, the baffle shows a significant improvement of PGM + Au recovery as compared to the base case and is a viable option for improving recovery of PGMs. The washing tests seem to be futile in improving recovery, but the effect on  $\text{Cr}_2\text{O}_3$  reduction is the more important focus for the washing tests.



The addition of wash water significantly reduced the amount of solids and increased the water content in the concentrates. This resulted in an increase in the water to solids ratio in the froth washing tests as compared to the non-washing tests, and hence the dry masses were much smaller than expected, when based on controlled wet mass of concentrate. To compensate for the increase in the water to solids ratio trend that was experienced in the washing tests, a larger mass of concentrate (solids + water) was collected such that the dry masses of the concentrates were similar to the base case.

The froth, with the standard frother dosage, was initially quite stable with smaller bubbles but thereafter became unstable (and with larger bubbles) half-way through the scavenger float. The instability may have taken place due to depletion of the frother chemical. A slightly unstable froth is a desirable phenomenon in flotation. Unstable froth is known to cause a 'distillation' effect because the unstable bubbles burst and shed their load consisting of gangue and valuable minerals. The valuable minerals have a higher probability of being re-attached to uprising bubbles due to their hydrophobicity but the gangue material have a lower probability of being re-attached. Hence, this enrichment process caused by unstable froth improves the separation of valuable minerals from the gangue material.



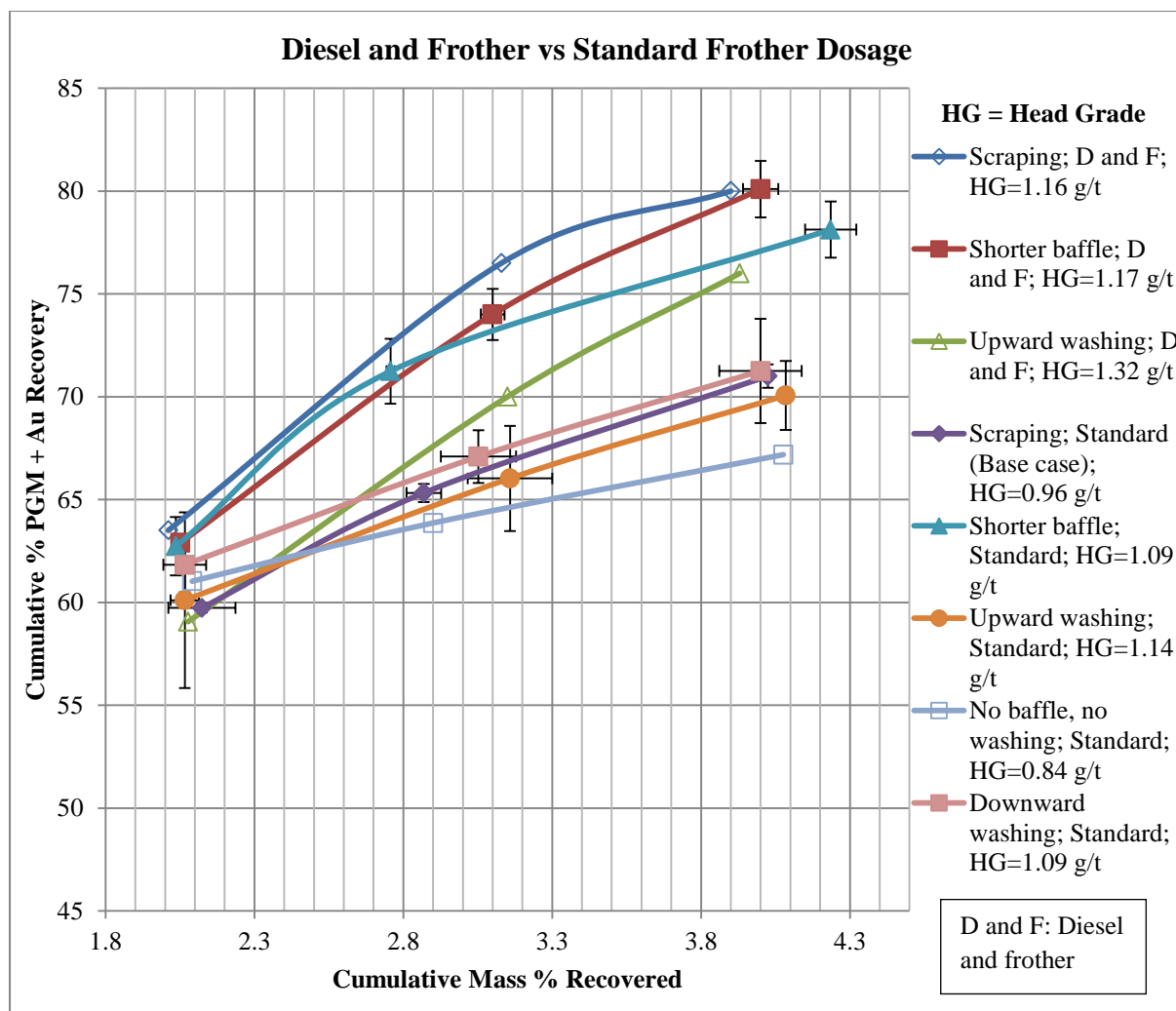
**Figure 4-11: Effect of washing and a baffle on cumulative  $\text{Cr}_2\text{O}_3$  grade, using standard frother addition.**

Figure 4-11 shows that the chromite content of the froth increased in most cases, but that the baffle provided the best result, with an almost constant  $\text{Cr}_2\text{O}_3$  grade of concentrate. The amount of water recovered in the washing tests was considerably higher than that in the non-washing tests, and this appears to have contributed to increased chromite recovery, despite displacement of the water in the froth by wash water. The relationship between water and gangue recovery states that an increase in water recovered causes an increase in the amount of entrained gangue. Significantly higher amounts of water recovered were achieved for the washing tests (see plots of  $\text{Cr}_2\text{O}_3$  recovery versus water recovery in Appendix C5) which may have led to the increased amount of  $\text{Cr}_2\text{O}_3$  recovered.

From Figure 4-11, it can be seen that the baffle shows a lower  $\text{Cr}_2\text{O}_3$  grade than the base case by approximately 2 %, at an equivalent mass recovery of about 4 %. Both washing tests show a higher  $\text{Cr}_2\text{O}_3$  content. It was anticipated that the washing tests would show a lower chromite content due to the washing action of the wash water. The results depict the opposite from what was anticipated. This was due to the entrained water in the froth phase undergoing inadequate drainage with the entrained gangue material into the pulp layer as theoretically supposed. Instead, the wash water and the entrained liquid exited with valuable minerals in the concentrate. The wash water may also have moved straight down through the froth without coating and removing the gangue material from the froth and therefore inadequately performing its function.

The  $\text{Cr}_2\text{O}_3$  content for the tailings was fairly constant at a value around 24.2 to 25.3 % (see Appendix D7 for raw data). As before, the tests analysed in duplicate are represented by the closed figure data points, and the non-repeated tests are represented as open data points. All tests were done in duplicate except for the upward washing test which was done in triplicate. The error bars incorporated the data points in cases of repeat tests and showed variation in the specific tests that were repeated. Small error bars were calculated which show that the graph plotted for the various tests had a limited range to be in. Overlapping of the error bars indicate that the difference between the tests containing the overlapping may not be significant. Error bars for the upward washing overlapped the error bars for the downward washing showing that the two methods are not statistically different to each other. Since no overlapping of error bars was noted in Figure 4-11 with the base case, it can be said that both washing tests and the baffle tests are statistically different from the base case.

The results above show that froth washing of the froth phase in both directions was inefficient in reducing the amount of entrained chromite. Examining Figure 4-11, it can be deduced that an increase in entrainment occurred in the washing tests. This is indicated by the higher chromite content as compared to the base case. The plot shows that the washing tests do not result in a reduction of chromite as theoretically predicted.

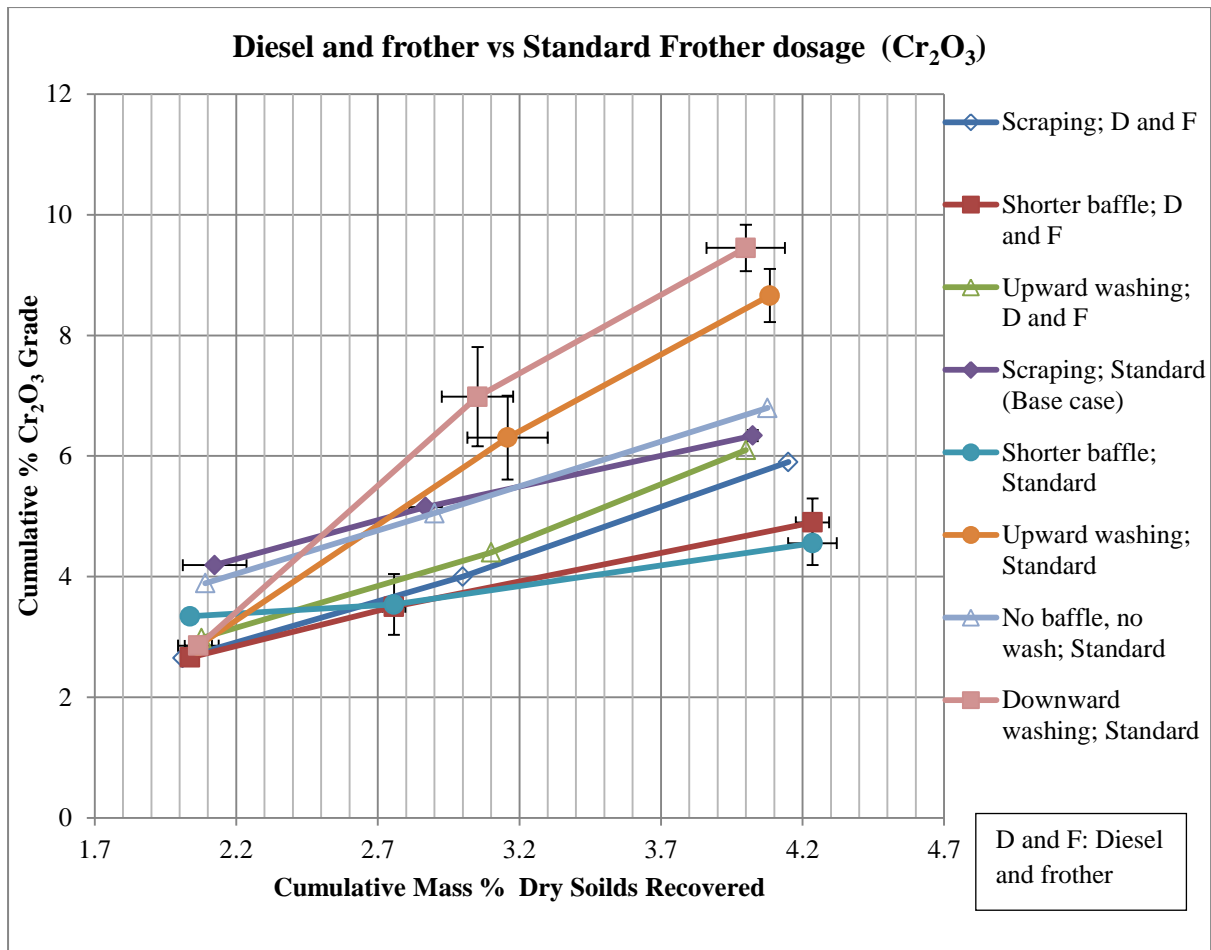


**Figure 4-12: Effect of diesel and frother on the PGM recovery, compared to standard frother dosage.**

The mixture of diesel and frother was added to the scavenger stage of flotation only. The diesel that was used was the normal diesel fuel used in motor vehicles. The tests carried out with the use of diesel and frother as a reagent also shows positive linear trends as the standard reagent dosage. The scraping and baffle tests with the use of diesel and frother show a significant improvement as compared to the base case. At an equivalent mass recovery of about 4%, recovery improvements of up to 9 % are noted for the tests with the baffle + diesel and frother dosage as compared to the base case. The washing tests with diesel and frother show a smaller improvement in recovery as compared to the baffle and scraping tests with diesel and frother. The froth washing tests with diesel and frother show an approximate improvement in recovery of 5 % as compared to the base case. The diesel and frother improves the hydrophobicity of the mineral which leads to an increase in recovery. Overall, the addition of diesel and frother to the scavenger stage of flotation assists significantly in the improvement of PGM recovery.

Error bars plotted on the data in Figure 4-12 represent the 95 % confidence interval on the repeated tests (closed figure data points) as explained previously and are there to prove that the tests with diesel and frother are significantly different than the base case which is evident by distinct error bars. Comparing the performance of the baffle against the two chemical reagents, it can be seen that the test with the diesel and frother shows a slightly higher recovery than the standard frother dosage (approximately 3 % higher) at an equivalent mass recovery of around 4%.

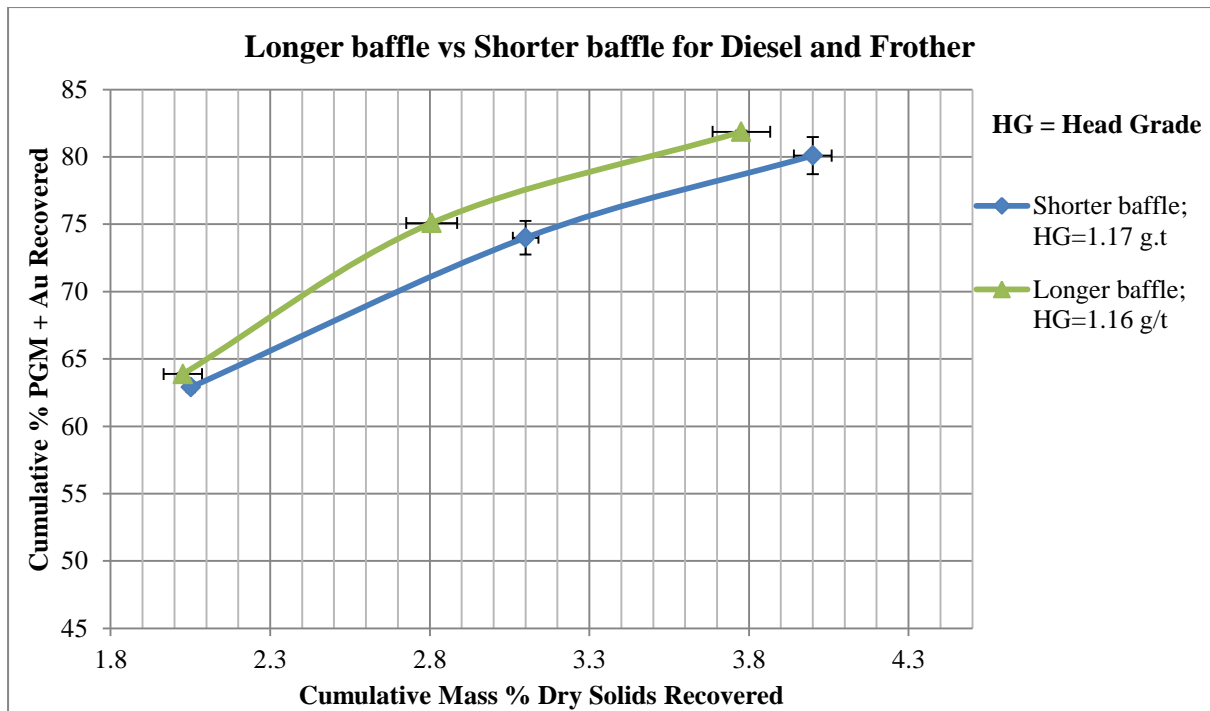
The froth, with the addition of diesel and frother, was darker in colour than the standard reagent dosage and was much more unstable as well. The additional instability may have assisted in the significant improvement in recovery as compared to the standard reagent dosage. The froth was also visually observed to be more sticky and glossy and was probably caused by interaction with the diesel.



**Figure 4-13: Effect of diesel and frother on the  $\text{Cr}_2\text{O}_3$  grade compared to standard frother dosage.**

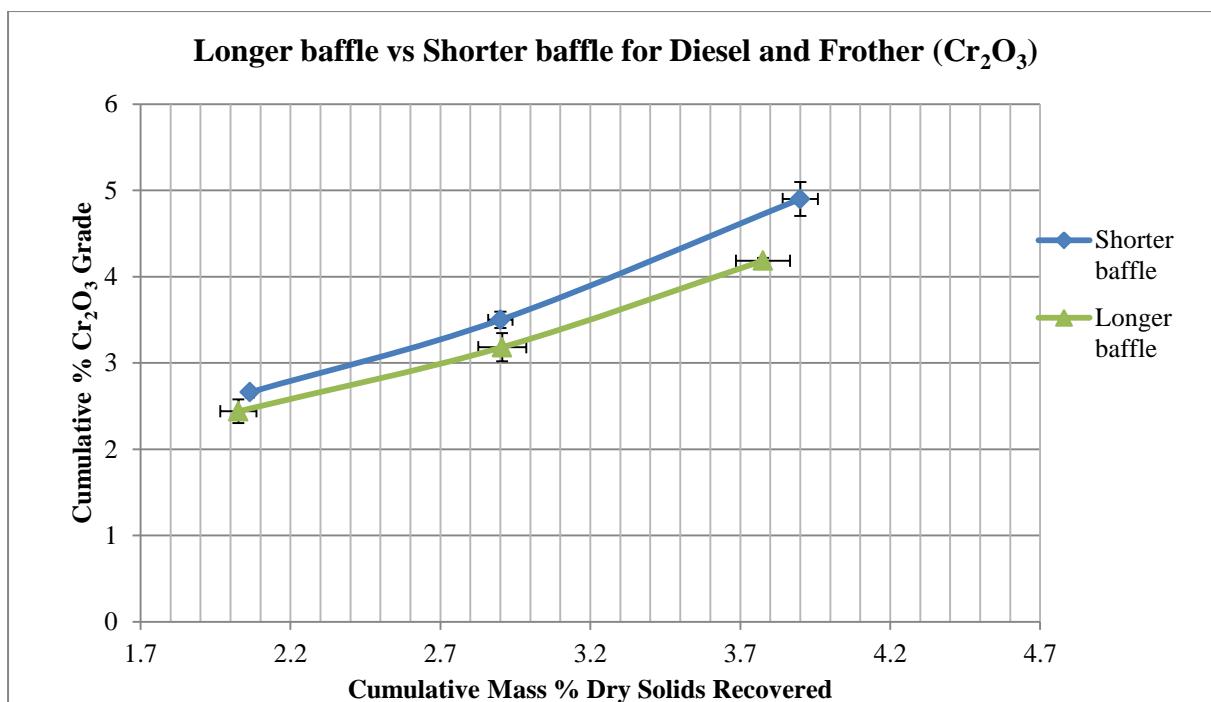
The highest  $\text{Cr}_2\text{O}_3$  grade can be seen in the tests with the standard frother dosage. The tests with diesel and frother show lower  $\text{Cr}_2\text{O}_3$  grades. The diesel and frother test with the shorter baffle shows an approximate 2 % lower  $\text{Cr}_2\text{O}_3$  content as compared to the base case, compared at an equivalent mass recovery of about 4%. The addition of diesel and frother to the scavenger stage of flotation decreases the amount of entrained  $\text{Cr}_2\text{O}_3$  in the flotation concentrate and, from Figure 4-12; flotation with the addition of diesel and frother improves the recovery of PGMs. The graph representing the series of repeat tests for the baffle (diesel and frother) is a good representation of the tests as depicted by the short error bars.

Due to the surprisingly favourable results achieved with the baffle tests, it was decided to construct a longer baffle on which to conduct experiments. As seen previously the tests with diesel and frother significantly improved the recovery of PGMs and reduced the grade of  $\text{Cr}_2\text{O}_3$ . Hence tests with the diesel and frother were also carried out with the longer baffle in order to compare with the shorter baffle.



***Figure 4-14: Comparison of longer baffle and shorter baffle performance for PGM + Au recovery using the diesel and frother dosage.***

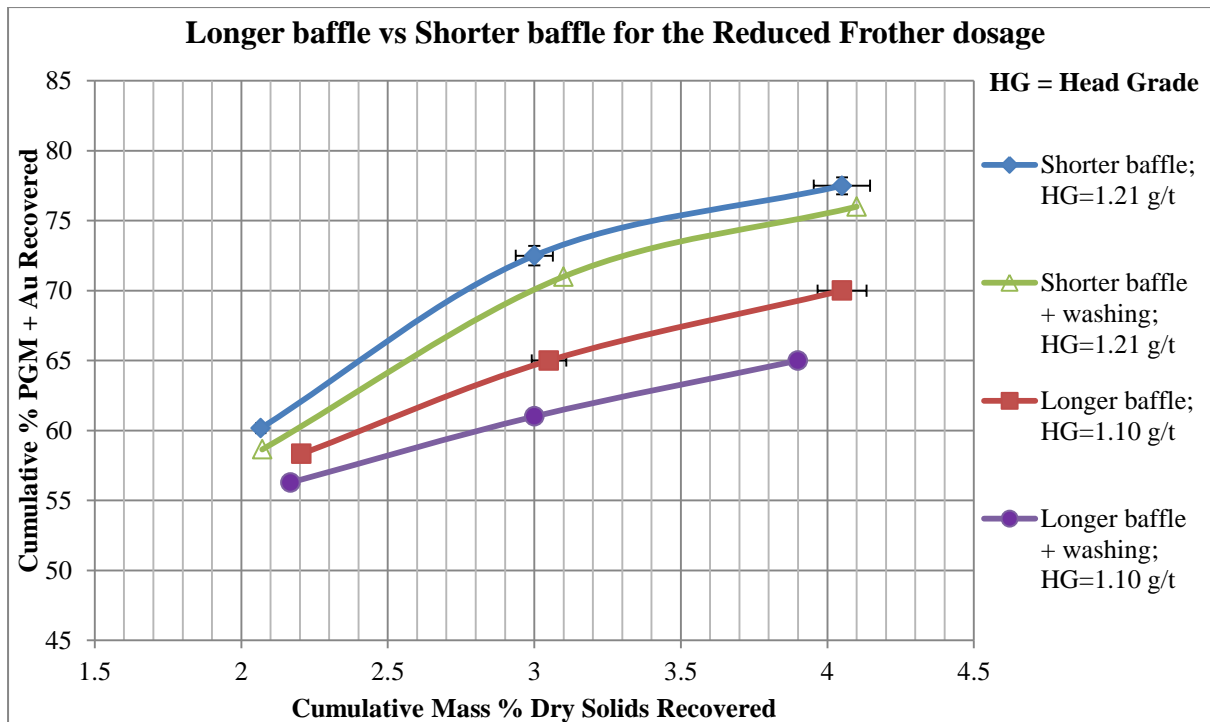
Figure 4-14 clearly shows a slightly higher recovery of PGMs for the longer baffle as compared to the shorter baffle. This may be attributed to the longer baffle covering the entire froth surface area as compared to the reduced portion covered by the shorter baffle (see Figure 3-8 in section 3.6.9). The longer baffle allows for continued forward movement of the froth which contains the valuable minerals. Error bars plotted show that the two tests are significantly different.



**Figure 4-15: Comparison of longer baffle and shorter baffle performance for Cr<sub>2</sub>O<sub>3</sub> grade using the diesel and frother dosage.**

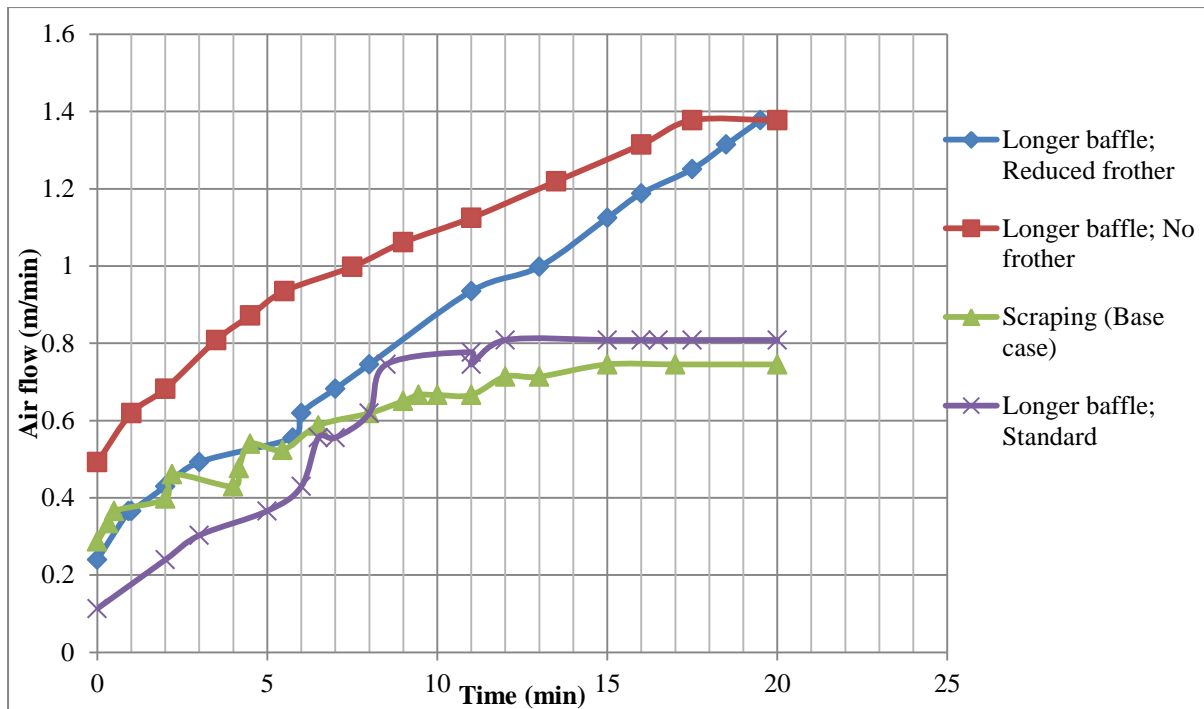
Figure 4-15 shows that the longer baffle provides a reduced Cr<sub>2</sub>O<sub>3</sub> recovery as compared to the shorter baffle. The longer baffle with the extended surface area is able to reduce the amount of Cr<sub>2</sub>O<sub>3</sub> that reports to the concentrate by the mechanism as explained in section 2.5.4.2. The baffle functions to suppress the uprising liquid. This causes the entrained gangue in the uprising liquid to fall back down to the pulp zone. It is this action that causes a reduction in Cr<sub>2</sub>O<sub>3</sub> that is recovered in the flotation concentrates. Since the longer baffle with the diesel and frother dosage in the scavenger demonstrates the highest PGM recovery and Cr<sub>2</sub>O<sub>3</sub> reduction thus far, further tests will be compared against this and the final conditions will represent the best situation in the optimisation of the scavenger flotation. Tests with a reduced frother dosage were also carried out both with the longer baffle and the shorter baffle. The comparison as to which baffle outperforms the other will be seen as follows.





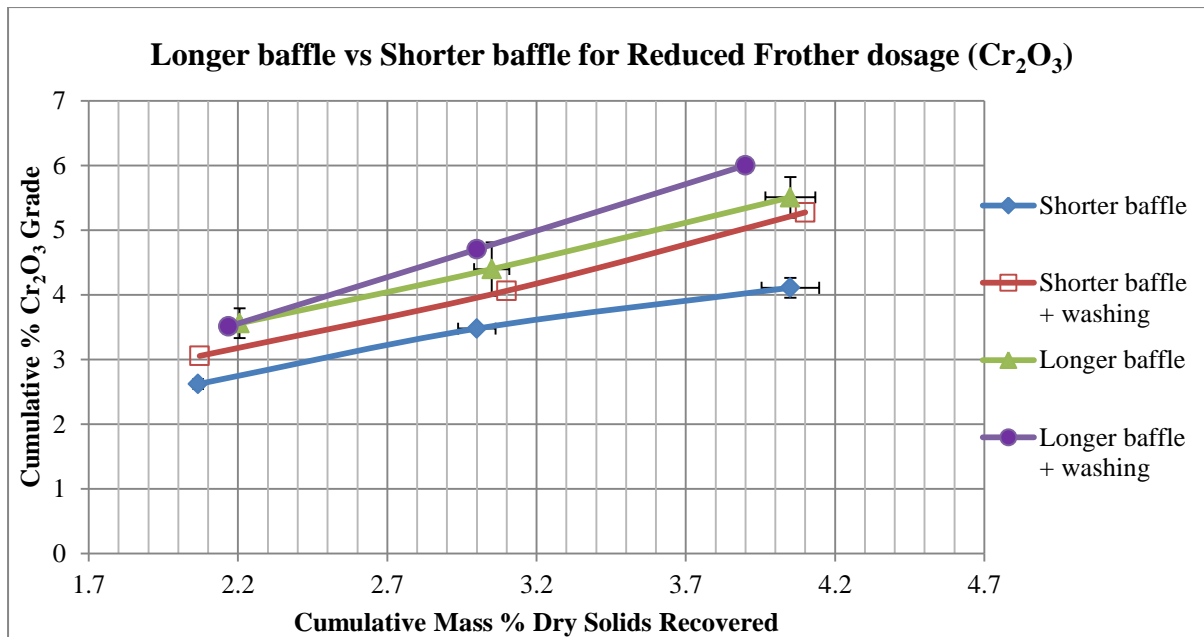
**Figure 4-16: Comparison of longer baffle and shorter baffle performance for PGM + Au recovery using the reduced frother dosage.**

From Figure 4-16, it can be seen that the shorter baffle, on its own and also with froth washing, demonstrates higher PGM + Au recoveries than the longer baffle tests for a reduced frother dosage. The highest PGM recovery can be seen in the test with the shorter baffle only and the lowest recovery in the test with the longer baffle with froth washing. The previous comparisons with diesel and frother show that the longer baffle performs better. The results with the reduced frother prove the opposite. To keep the froth level constant in the tests with the reduced frother dosage, the air flow rate had to be increased more than usual. Consequently, higher air flows were experienced with the tests with the reduced frother (and no frother, to follow) dosage as can be seen in the following figure: (see Appendix C3 for additional air flow plots and Appendix B4 for sample calculations).



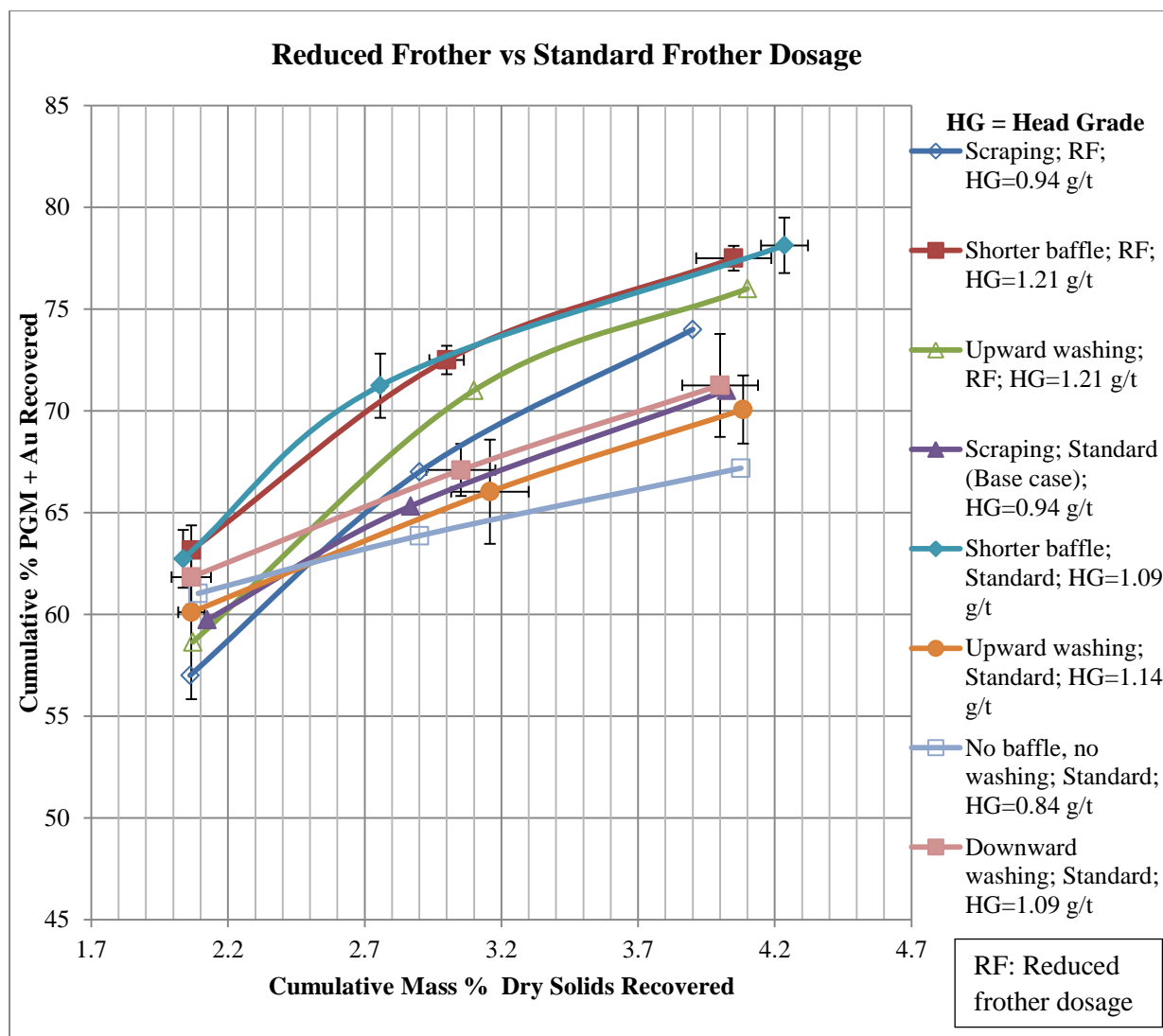
**Figure 4-17: Air flow with time showing impact of frother dosage on air flow rate.**

These higher air flow rates caused more bubbles to be formed and hence a higher push-up of froth into the froth phase and towards the concentrate launder directed by the shorter baffle. An increase in the amount of bubbles produced as a result of an increase in air flow rate results in an increase in selectivity of the flotation to capture the valuable minerals and not the gangue material. The longer baffle, in these tests may have caused the faster uprising bubble motion to decrease thereby reducing the overflow into the concentrate launder which resulted in a loss of recovery. The longer baffle with froth washing experienced a further lower recovery due to the washing action of the wash water removing valuable minerals as well as gangue material. In the case of the shorter baffle, the froth washing test experienced a lower recovery than the test with only the baffle for the same reason.



**Figure 4-18: Comparison of longer baffle and shorter baffle performance on Cr<sub>2</sub>O<sub>3</sub> grade using the reduced frother dosage.**

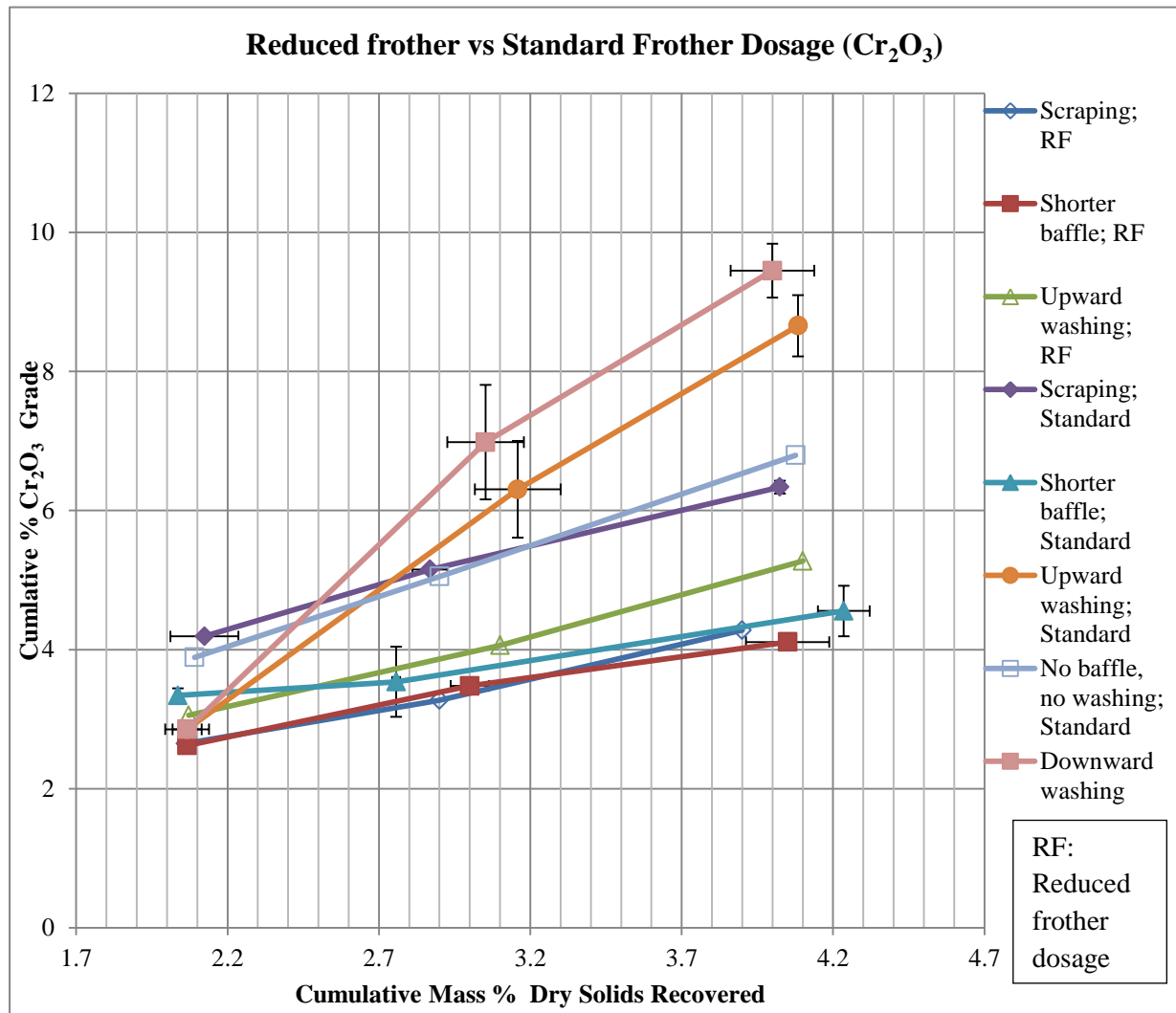
Figure 4-18 shows that lower Cr<sub>2</sub>O<sub>3</sub> grades are achieved for the shorter baffle tests than the longer baffle tests. For the froth washing tests, the shorter baffle demonstrates approximately a 1 % reduction in Cr<sub>2</sub>O<sub>3</sub> as compared to the longer baffle + washing. For the tests with the baffle only, the shorter baffle has a lower Cr<sub>2</sub>O<sub>3</sub> content by up to 1.5 %, as can be seen at an equivalent mass recovery of around 4%. From Figures 4-16 and 4-18, it can be concluded that the shorter baffle with the reduced frother dosage is the better option when considering the baffles. Hence the shorter baffle will be used in the future comparisons when dealing with the reduced frother dosage.



**Figure 4-19: Effect of reduced frother dosage on PGM recovered compared to standard frother dosage.**

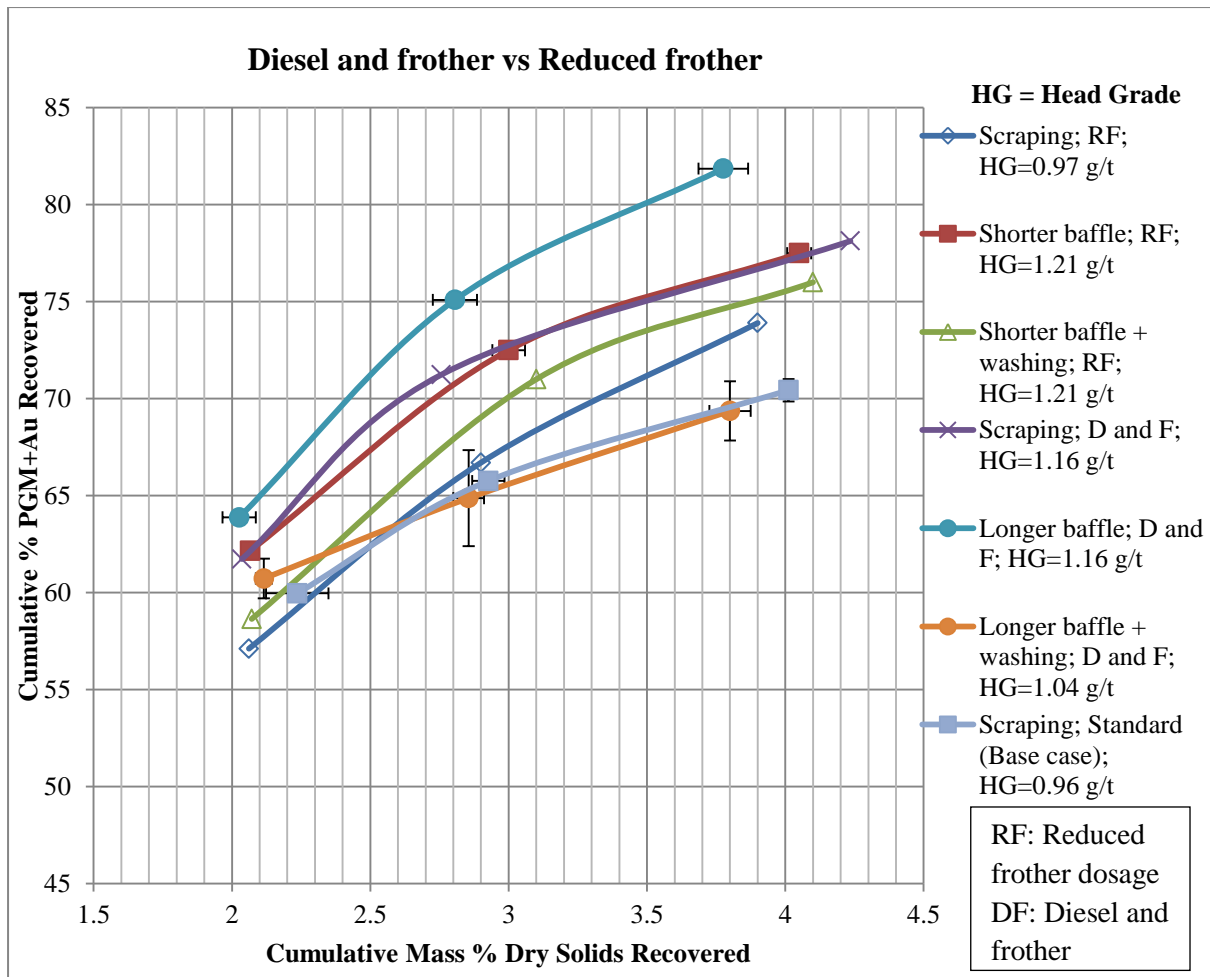
Figure 4-19 shows that the reduced frother dosage improves the precious metal recovery. At an equivalent mass recovery of around 4%, the shorter baffle + reduced frother dosage provides a high recovery of approximately 77.5 % which is close to the shorter baffle + standard frother dosage (78 %). This implies that, for the shorter baffle, a similar recovery can be reached with half the frother dosage. Compared to the base case (at 71 % PGM + Au recovered), the shorter baffle + reduced frother dosage shows an improvement in recovery by 6.5 %. The reduced frother dosage with froth washing showed reasonably good results since it achieved higher recoveries than the base case (approximately 4.5 % higher). For the froth washing tests, the reduced frother dosage was only investigated with upward washing due to economics.

The froth behaviour with a reduced frother dosage was more unstable than the standard frother dosage as well. The froth was noted to become unstable in the first quarter of the scavenger flotation due to a faster depletion of the frother as compared to the standard frother dosage. It should be noted that the air flow was increased to achieve a comparable froth flow, see Figure 4-17, and the ability to do this on the plant would have to be checked.



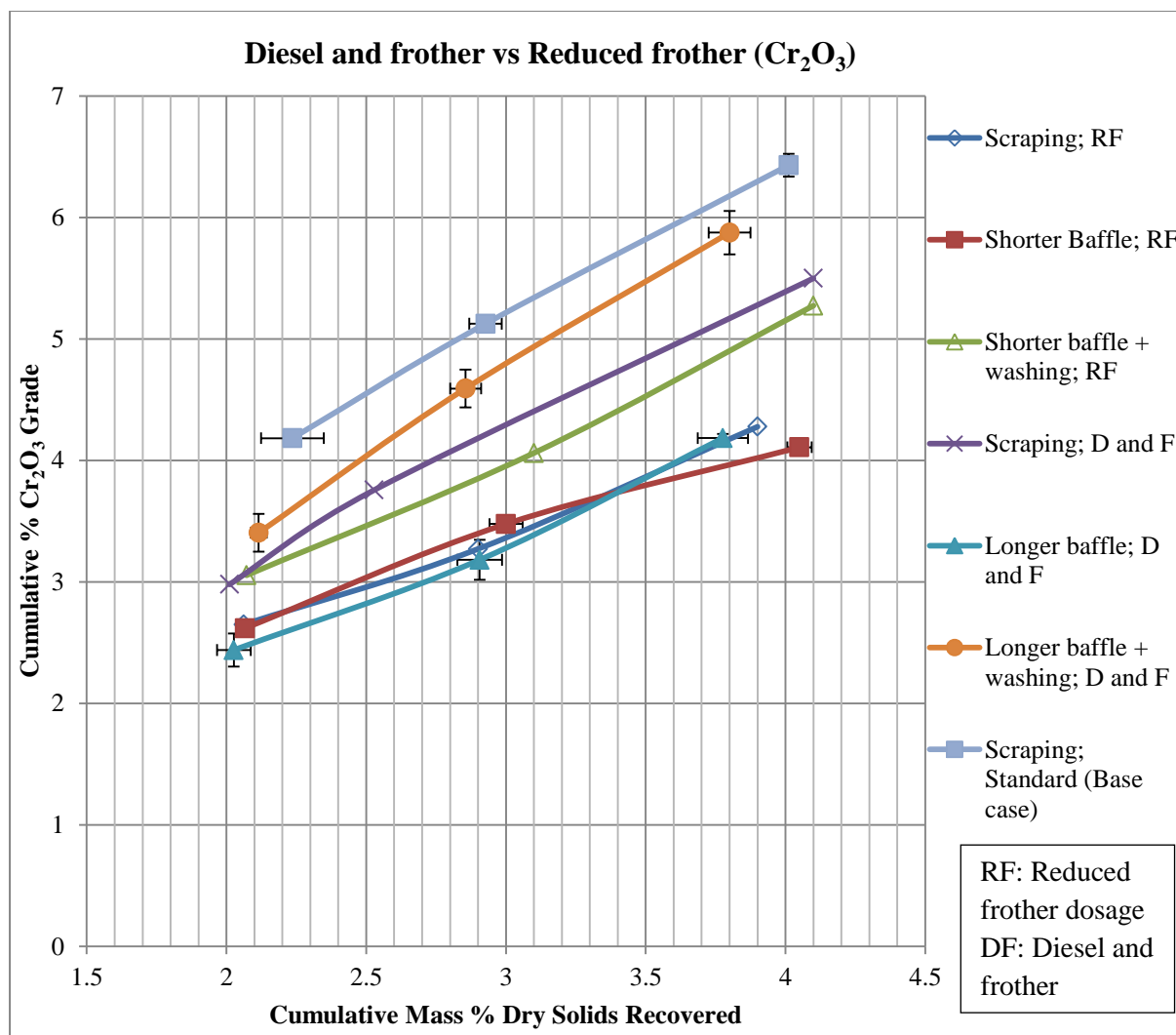
**Figure 4-20: Effect of reduced frother dosage on  $\text{Cr}_2\text{O}_3$  grade compared to standard frother dosage.**

There was no significant difference in the cumulative chromite grade for the best results which were the shorter baffle with the standard frother dosage and the shorter baffle with the reduced frother dosage.



**Figure 4-21: Comparison of diesel and frother vs reduced frother dosage on PGM recovery.**

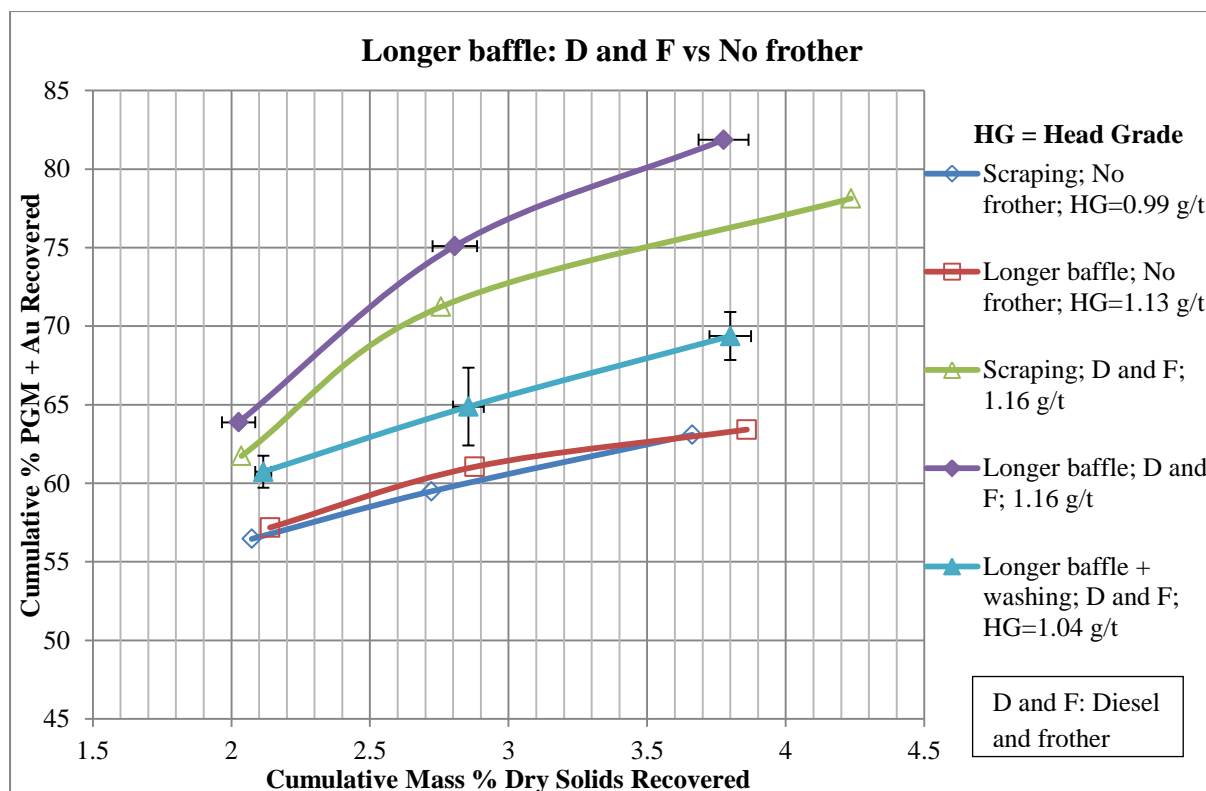
Figure 4-21 shows a comparison of the best results shown thus far. At an equivalent mass recovery of around 3.8%, the longer baffle + diesel and frother show a PGM recovery of approximately 82 % which is 5 % more than the PGM recovery of the shorter baffle + reduced frother dosage. In the froth washing tests, the diesel and frother provides the lowest recovery which may be attributed to unfavourable interaction between wash water and the mixture of diesel and frother. Comparing the scraping tests for the diesel and frother and the reduced frother, it can be seen that the diesel and frother improves the hydrophobicity of the PGMs (hence improved the recovery) much more than the ability of the reduced frother dosage.



**Figure 4-22: Comparison of diesel and frother vs reduced frother dosage on the  $\text{Cr}_2\text{O}_3$  grade.**

The lowest  $\text{Cr}_2\text{O}_3$  grade was achieved for three tests: longer baffle + diesel and frother, shorter baffle + reduced frother and the scraping + reduced frother. The froth washing test with the longer baffle + diesel and frother dosage had approximately 2 % higher  $\text{Cr}_2\text{O}_3$  content than the abovementioned tests. The PGM recovery for the longer baffle + diesel and frother dosage is significantly higher than the shorter baffle + reduced frother dosage but the  $\text{Cr}_2\text{O}_3$  grades for both baffles are approximately similar. Hence the longer baffle + diesel and frother dosage is overall the better option and would be used in the subsequent comparisons.

The performance of the reduced frother was found to be fairly good in the PGM recovery and the  $\text{Cr}_2\text{O}_3$  reduction; hence it was thought that completely eliminating the frother dosage to the scavenger stage would be an improvement to the reduced frother dosage.

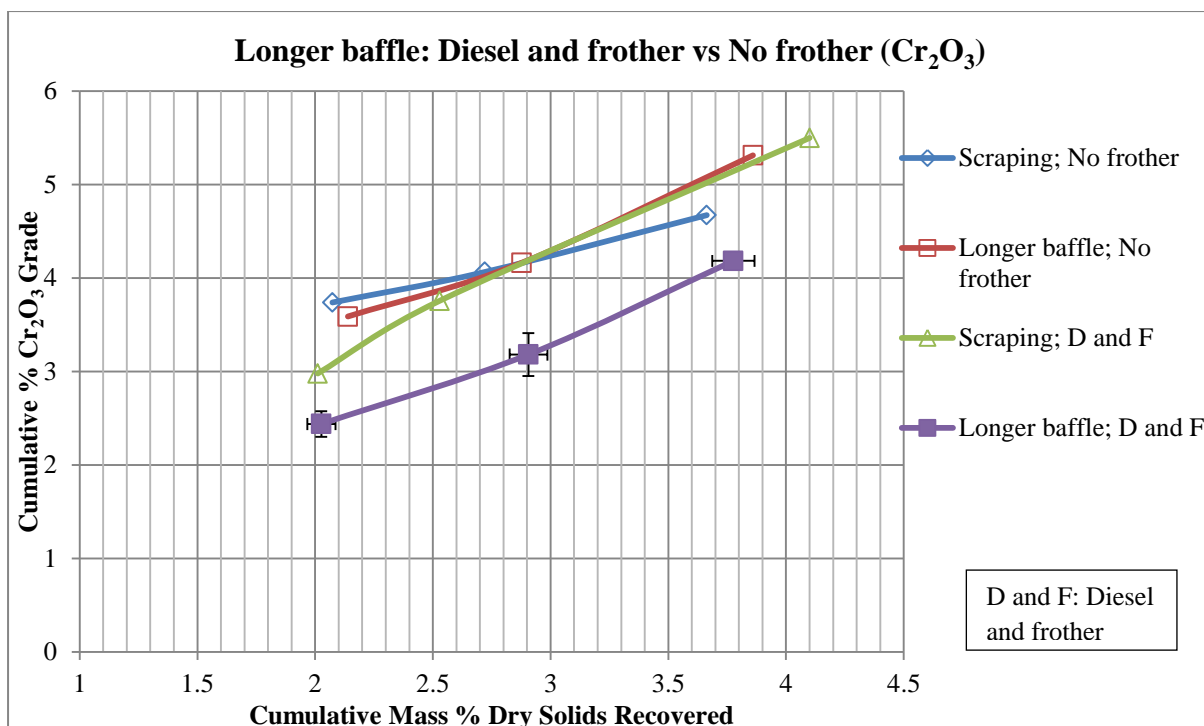


***Figure 4-23: Comparison of diesel and frother dosage to no frother for PGM recovered.***

It can be clearly seen that flotation without frother added to the scavenger stage does not result in an improvement in recovery. The bubbles were larger which thereafter died off quickly during the course of the run, despite the increase in air flow. This suggests that a finite amount of frother is required to exhibit reasonable recoveries of PGMs.

When diesel and frother is used in the reagent suite in scavenger flotation, it produces a stable, slimy (oily) froth layer for the first half of the scavenger flotation. When reduced frother dosage is used, an unstable voluminous froth with a great amount of bubble coalescence is formed for the first half of the scavenger flotation. The froth behaviour of no frother addition is similar to the reduced frother addition with the froth dying off quickly during the course of the scavenger flotation.

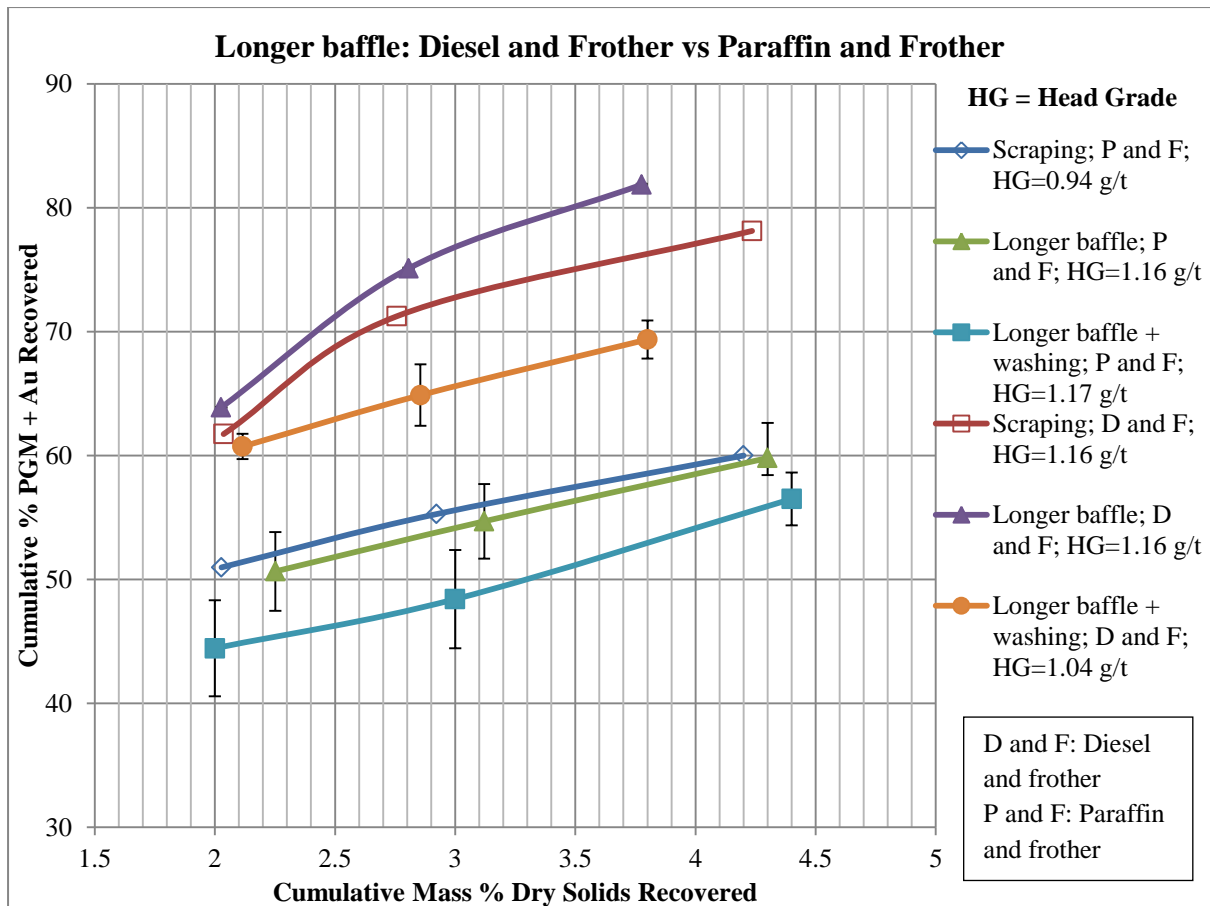




**Figure 4-24: Comparison of diesel and frother dosage to no frother for  $\text{Cr}_2\text{O}_3$  grade.**

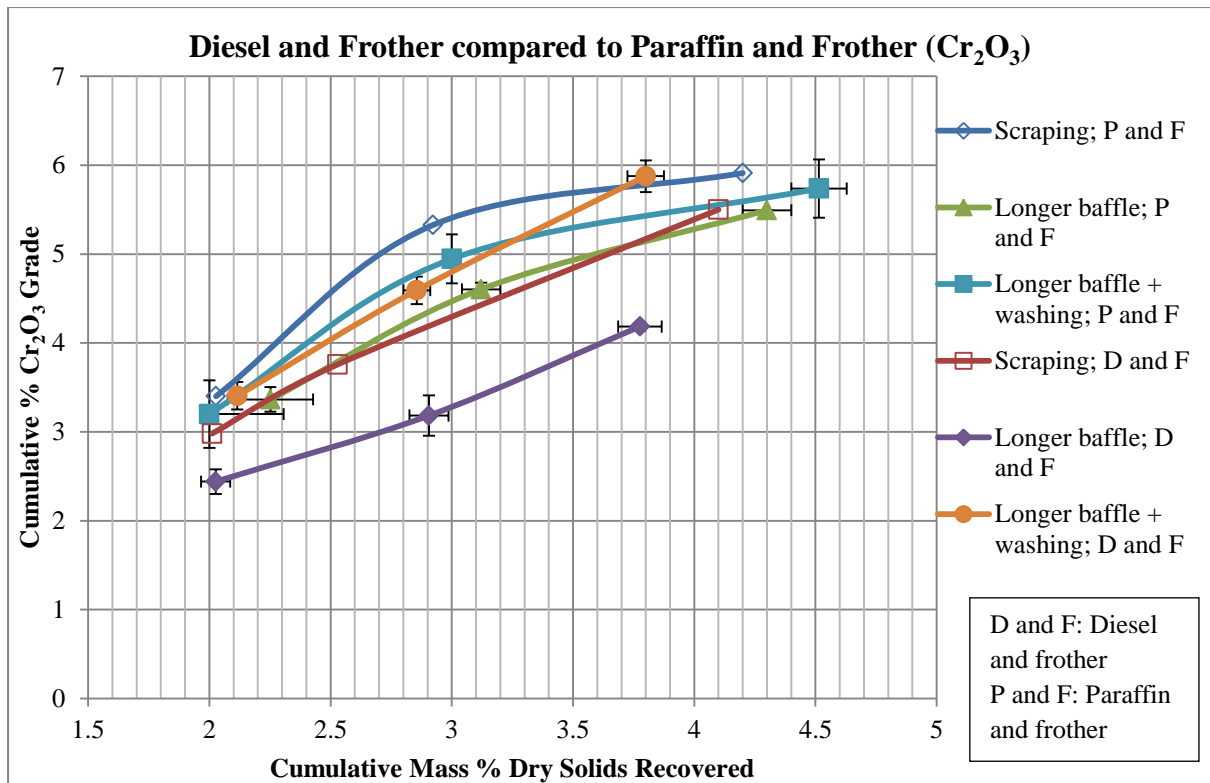
Figure 4-24 shows that the elimination of frother addition resulted in more entrainment of chromite, probably due to unstable froth conditions.

Another chemical reagent, paraffin, was investigated in the optimisation of the scavenger stage of flotation. The paraffin was mixed with the frother (proportions of paraffin and frother may be found in section 3.3 in the experimental procedure. Results of the paraffin and frother compared to the best combination of baffle and reagents (longer baffle + diesel and frother) are shown in Figure 4-25. The diesel and frother clearly outperforms the paraffin and frother. Figure 4-26 confirmed that the chromite grade was also better with the diesel and frother mixture.



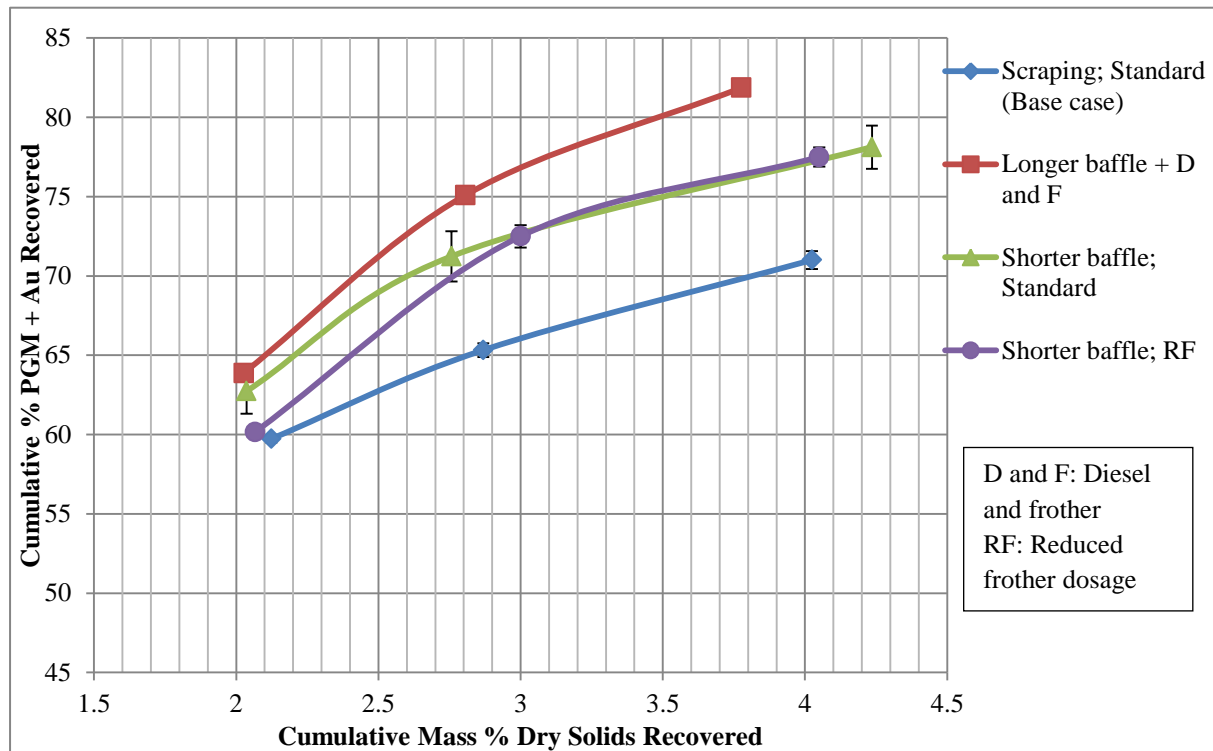
**Figure 4-25: Comparison of diesel and frother with paraffin and frother for PGM recovered.**

The froth behaviour with the addition of paraffin and frother was similar to the standard frother dosage with a slight instability towards the end of the flotation. Visually, the paraffin was not noticed to enhance the froth phase significantly.



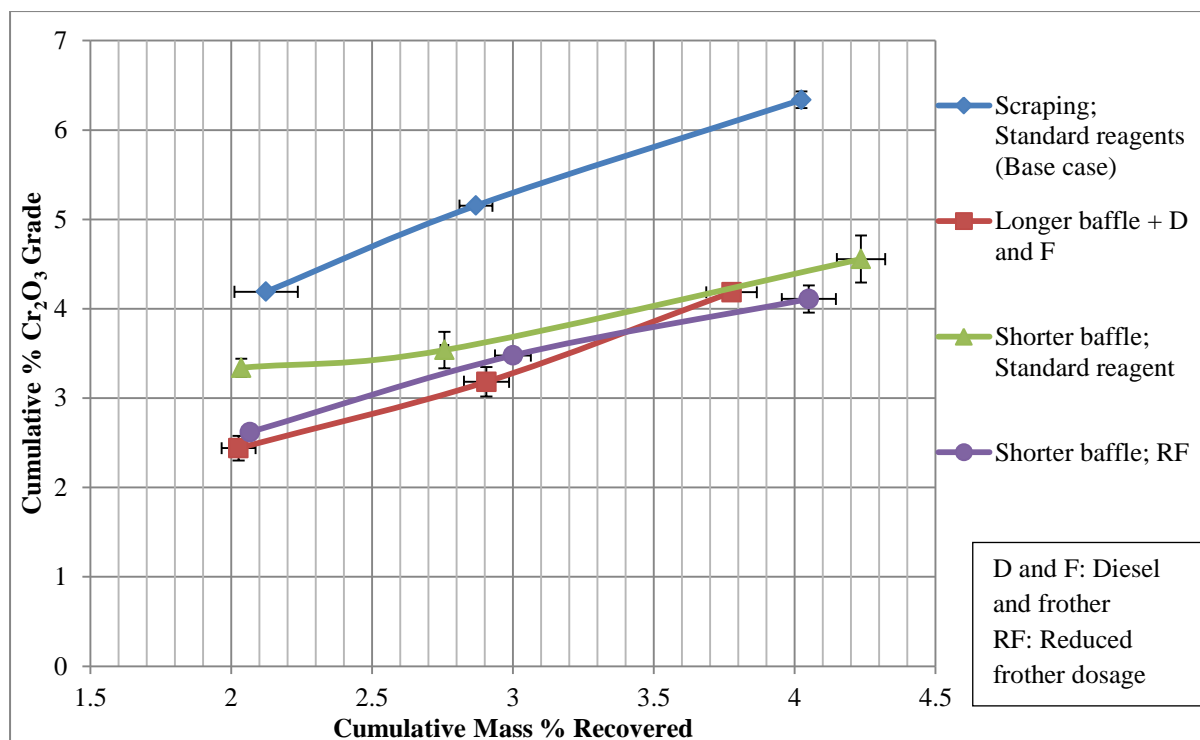
**Figure 4-26: Comparison of diesel and frother with paraffin and frother for  $\text{Cr}_2\text{O}_3$  grade.**

#### 4.7. Final optimised results



***Figure 4-27: Final results with the highest PGM + Au recovery.***

Figure 4-27 shows that the longer baffle + diesel and frother dosage improved the recovery of PGM + Au by 11 % when comparing at an equivalent mass recovery of about 3.8 %. If the standard frother dosage is used, the situation that results in the highest recovery is the shorter baffle which shows a 6% improvement in recovery. If the shorter baffle + reduced frother dosage is used, the maximum improvement in recovery that can be achieved is 6.5 %. The reduced frother option is also a favourable option since there is a possibility of savings in reagent costs since lesser amounts of frother are required.



**Figure 4-28: Final results with the lowest  $\text{Cr}_2\text{O}_3$  recovery.**

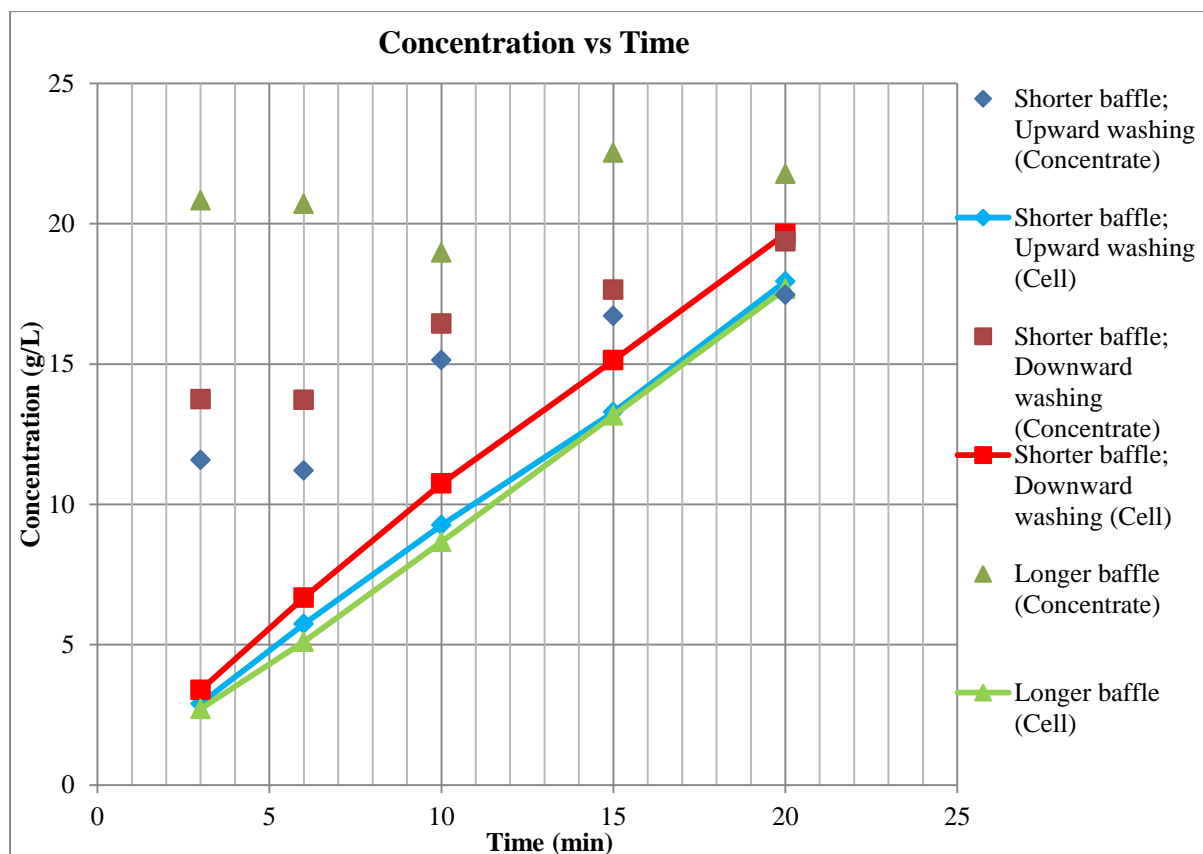
It can be seen that all the above tests effectively reduce the amount of entrained  $\text{Cr}_2\text{O}_3$  as compared to the base case. Compared at an equivalent mass recovery of around 4%, the lowest overall  $\text{Cr}_2\text{O}_3$  grade of 4.1 %, was obtained for the shorter baffle + reduced frother test with a 2.2 % reduction in  $\text{Cr}_2\text{O}_3$  content from the base case. The longer baffle + diesel and frother test achieved similar results.

Figures 4-27 and 4-28 summarise the performance obtained for the longer baffle + diesel and frother, the shorter baffle + reduced frother and, the shorter baffle + standard frother dosage. The test with the longer baffle + diesel and frother outperformed the other types of tests and significantly improved the PGM + Au recovered compared to the base case. Although the shorter baffle + reduced frother dosage did not result in the highest PGM + Au recovery, the increase in PGM + Au recovery of 6.5 % is significant considering that only half the standard frother dosage was used.

#### **4.8. Tracer tests –Efficiency of froth washing**

Wash water tracer studies were done visibly with blue dye to try and observe the path of the wash water. Tracer studies were also done quantitatively with the use of a salt solution where the amount of salt water that displaced the entrained liquid was used as an indication of froth washing efficiency. The criterion for an effective wash water distribution configuration was the one that provides an even distribution of water to the froth. The blue dye tracer tests showed that the wash water initially moved downwards into the froth and coated the bubbles. The froth was also naturally mobile and carried some of the water out over the cell lip and into the collection containers. Some blue dye was noticed in the pulp suspension which indicated that a fraction of the wash water also moved down into the pulp. From the poor results obtained for the froth washing tests it may be deduced that the wash water did wash the froth (as observed) but that it did not have enough time to drain from the froth and hence travelled to the concentrate, not removing the gangue as planned.

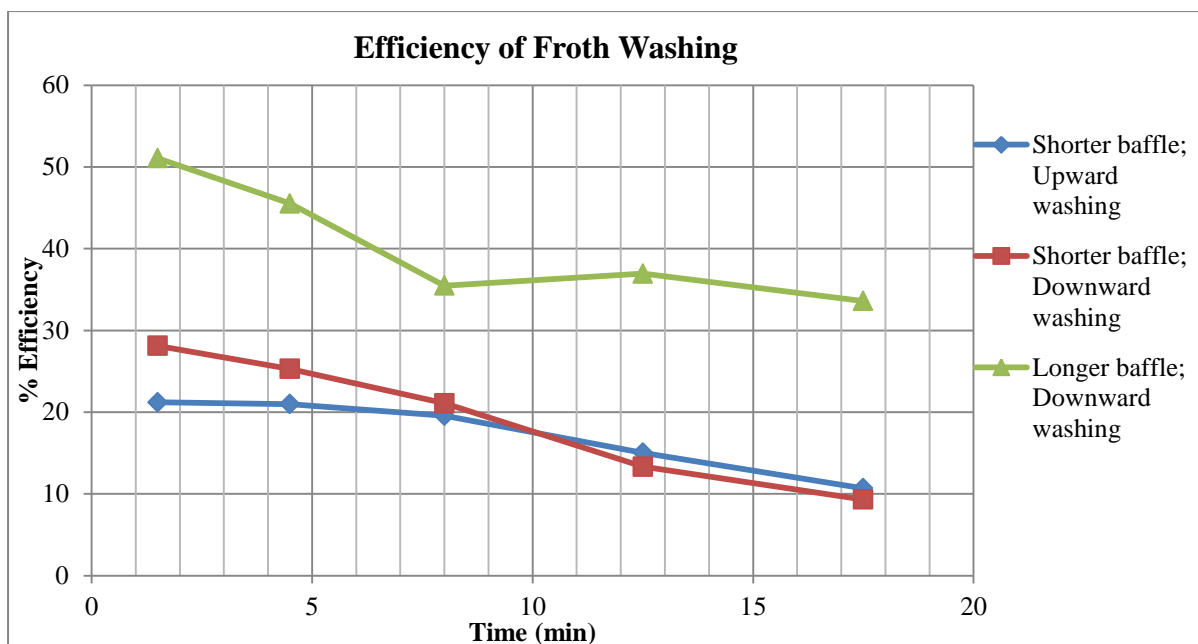
The efficiency of displacement of water from the froth by washing was measured by using a salt solution. The concentration of salt in the samples of concentrate was measured and gave a direct indication of the washing efficiency (see Figures 4-29 and 4-30 below, and, for more information, see Tables D-4 to D-21 in Appendix D5). Figure 4-29 that follows depicts the concentration pattern in the cell and in the concentrate.



**Figure 4-29: Salt concentration in the cell and in the concentrate with time during flotation. The points represent the concentration in the concentrate and the lines represent concentration in the cell.**

The trend for the salt concentration in the cell is fairly linear with an increase in salt concentration with time. The relationship between the salt concentration in the concentrate that is collected with time is not clear since no steady trend is observed. The result of an increase in salt concentration in the cell with time was expected as a portion of the salt wash water went into the cell and another portion exited with the concentrate. Individual concentration versus time plots for each of the washing tests can be seen in Figures C-28 to C-22 of Appendix C4.

Figure 4-29 shows that the washing tests utilising the longer baffle obtained concentrates of a much higher conductivity as evident by the higher concentration values. This means that the longer baffle offers a better potential displacement of entrained liquid with wash water. Efficiencies of the froth washing were then calculated based on the conductivities obtained. Results for the efficiency tests can be seen in Figure 4-30:



**Figure 4-30: Efficiency of the froth washing tests with the different baffles used.**

The trend for % efficiency with time is approximately linear with a negative slope indicating a decrease in efficiency with time. The trend of the efficiency decreasing as the flotation takes place could have been due to the froth becoming brittle towards the latter part of the flotation. The brittle froth was unstable which caused the bubbles to break easily which led to the collapse of the bubbles as the wash water was added. This caused most of the wash water to go down into the pulp and not be recovered in the concentrate hence the reduction in efficiency. Sample calculations for the efficiency may be found in Appendix B2.

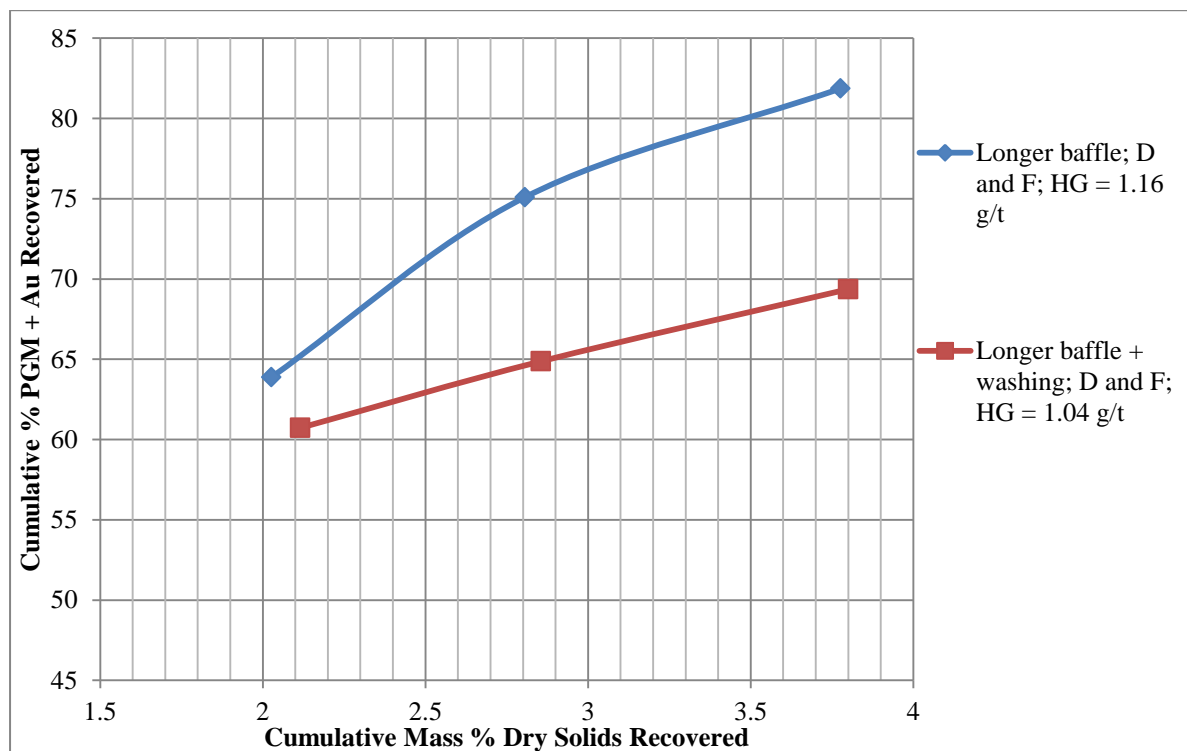
The washing was more efficient when the longer baffle, with an integrated wash bar, was used. The longer baffle had higher average efficiencies that ranged from 30 to 50 % compared to 10 to 30 % for the shorter baffle. The upward and downward washing tests show similar efficiencies.

The efficiencies obtained in Figure 4-30 imply that the froth washing is significantly efficient in replacing the entrained liquid with wash water thereby removing the  $\text{Cr}_2\text{O}_3$ . From the results of  $\text{Cr}_2\text{O}_3$  recovered for the washing tests with the standard reagent dosage (Figure 4-11), it may be deduced that the froth washing does not effectively remove  $\text{Cr}_2\text{O}_3$  from the froth phase. It actually increases the amount of entrained  $\text{Cr}_2\text{O}_3$ . This implies that the efficiencies obtained are theoretical and do not match reality. For the tests with reduced frother dosage, no frother dosage and paraffin and frother dosage; similar and slightly lower  $\text{Cr}_2\text{O}_3$  recoveries were obtained which imply slight correspondence with the efficiencies calculated.

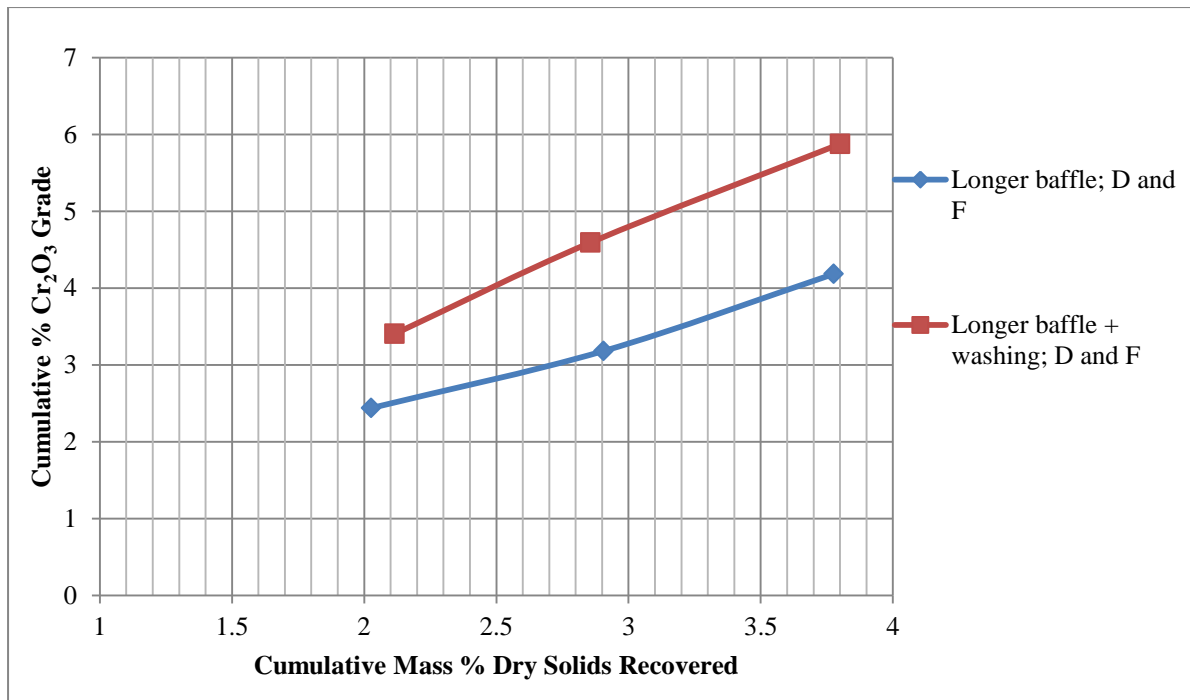


Simulations done by Neethling and Cilliers (2009) resulted in a linear relationship between water and gangue recovery for fine particle sizes. Very similar behaviour has been observed experimentally as can be seen in Appendix C5 ( $\text{Cr}_2\text{O}_3$  recovered versus water recovered) where all the tests rendered approximately linear relationships between gangue and water recovery.

Although an even distribution of wash water was supplied, the froth washing tests rendered poor results. The bubbles were noted to be stabilised by the wash water – i.e. bubble coalescence was inhibited. Hence no logical explanation can be given (i.e. a wash efficiency of 50 % implies that the gangue would be reduced to half the previous value). The difference in recoveries can be seen in Figures 4-31 and 4-32, when comparing the washing tests with the non-washing tests for the longer baffle as an example. Although the longer baffle + washing tests show a maximum of 50 % efficiency in the froth washing, the recoveries obtained for PGMs are significantly lower.



**Figure 4-31: Washing versus non-washing for the longer baffle tests for the diesel and frother dosage on the PGM + Au recovery.**



**Figure 4-32: Washing versus non-washing for the longer baffle tests for the diesel and frother dosage on the  $\text{Cr}_2\text{O}_3$  grade.**

It can also be seen that the efficiencies are misleading as, in Figure 4-32; the washing tests provide higher chromite grades than the non-washing tests which is contradictory to the froth washing being efficient in removing the gangue from the froth.

## CHAPTER 5: CONCLUSIONS

### 5.1. Recovery of PGMs (+ gold)

The following improvements were observed at an equivalent mass recovery for rougher and scavenger of about 4%:

- The diesel and frother mixture, in combination with the longer baffle, provided the largest improvement in PGM recovery of approximately 11 %.
- A significant improvement of 6.5 % in PGM recovery was achieved by reducing the frother dosage in the scavenger from 20g/t to 10 g/t, in combination with the shorter baffle.
- Washing tests did not perform as expected. The test that displayed positive results for PGM recovery, relative to the base case, was the washing test with the shorter baffle and a reduced frother dosage. Nevertheless, the washing caused a decrease in recovery, relative to the second condition above. Hence it was concluded that washing had an adverse effect on PGM recovery, at equivalent mass recovery.
- Diesel is known for its ability to enhance the collector activity. The data confirmed the suspicion that it would be beneficial in the scavenger, where composite particles are likely to occur. The limited exposure of floatable mineral needed to be made more hydrophobic to retain composite particles.

## 5.2. Cr<sub>2</sub>O<sub>3</sub> grade of the concentrate

The grades of Cr<sub>2</sub>O<sub>3</sub> in the combined rougher and scavenger concentrates were compared at an equivalent mass recovery of about 4 %.

- The average Cr<sub>2</sub>O<sub>3</sub> grade achieved for the base case was 6.3 %. The test with the shorter baffle and reduced (10g/t) frother demonstrated an average grade of 4.1 %, a significant reduction.
- The tests with the longer baffle and the diesel/frother mixture also demonstrated significant reductions in the Cr<sub>2</sub>O<sub>3</sub> content (2%).
- The nearly horizontal baffle effectively promotes the movement of the froth to the front of the cell (thereby improving PGM recoveries) while at the same time not causing significant mixing with the pulp, as was observed when using the short baffle. Hence the Cr<sub>2</sub>O<sub>3</sub> content was reduced by 2%
- The tests with no frother displayed some Cr<sub>2</sub>O<sub>3</sub> reduction (approximately by 1 %). The tests with paraffin and frother showed only a slight reduction in Cr<sub>2</sub>O<sub>3</sub> content.
- The washing tests showed the highest Cr<sub>2</sub>O<sub>3</sub> grades as compared to the other types of tests without washing (scraping, shorter baffle, longer baffle and no baffle) Cumulative grades were up to 9.5 %, which was 3.2% higher than the base case.
- The diesel and frother mixture, in combination with the long baffle, resulted in the best PGM recovery. However, the variables to be applied for the maximum Cr<sub>2</sub>O<sub>3</sub> rejection would be the shorter baffle and reduced frother dosage.

## 5.3. Washing tests

Despite efforts to optimise the addition of wash water to the froth, which was constrained by the need to maintain the level of the pulp, the following conclusions were reached:

- Froth washing does not reduce the amount of chromite in the scavenger stage of flotation of UG-2 ore.
- Froth washing works well in column flotation cells, with a deep froth, as stated in literature, but it does not operate well in the scavenger stage of flotation, where relatively large mechanical flotation cells are used and the froth is shallow and unstable. Angled baffles are used to 'crowd' the froth in some large mechanical cells.

#### 5.4. Tracer tests

- Tracer tests with the blue dye was effective in demonstrating visually the path of the wash water but the salt tracer tests had to be carried out to determine a quantitative split of the wash water.
- Theoretical calculations showed that the froth washing is a maximum of 50 % efficient with the longer baffle and 30 % efficient with the shorter baffle. The relative reductions in  $\text{Cr}_2\text{O}_3$  do not correspond to the calculated efficiencies. The  $\text{Cr}_2\text{O}_3$  content was consistently worse than the equivalent test with no washing.

## **CHAPTER 6: RECOMMENDATIONS**

It is recommended that additional laboratory and pilot-plant tests be done on the following:

- Nearly horizontal baffle
- Reduced frother dosage to the scavenger flotation cells
- Diesel and frother dosage to the scavenger flotation cells

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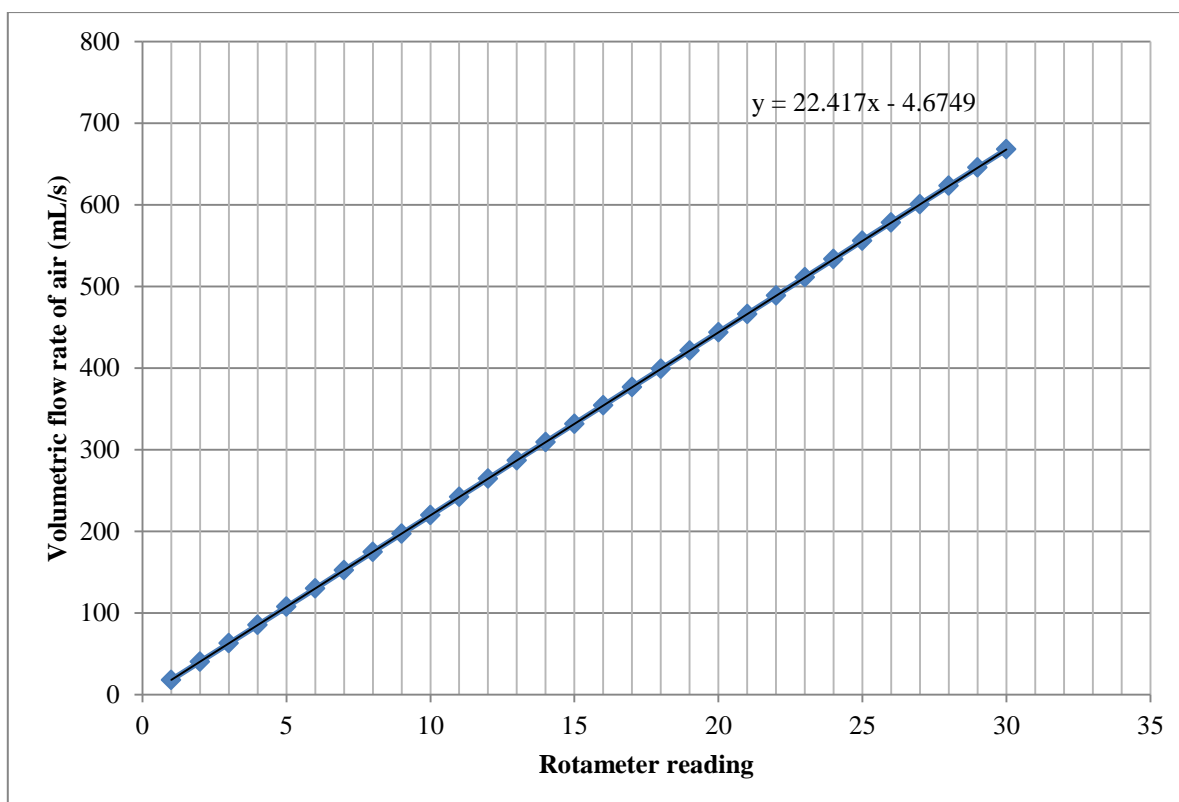
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## APPENDIX A: EQUIPMENT CALIBRATION GRAPHS

### Appendix A1: Rotameter calibration

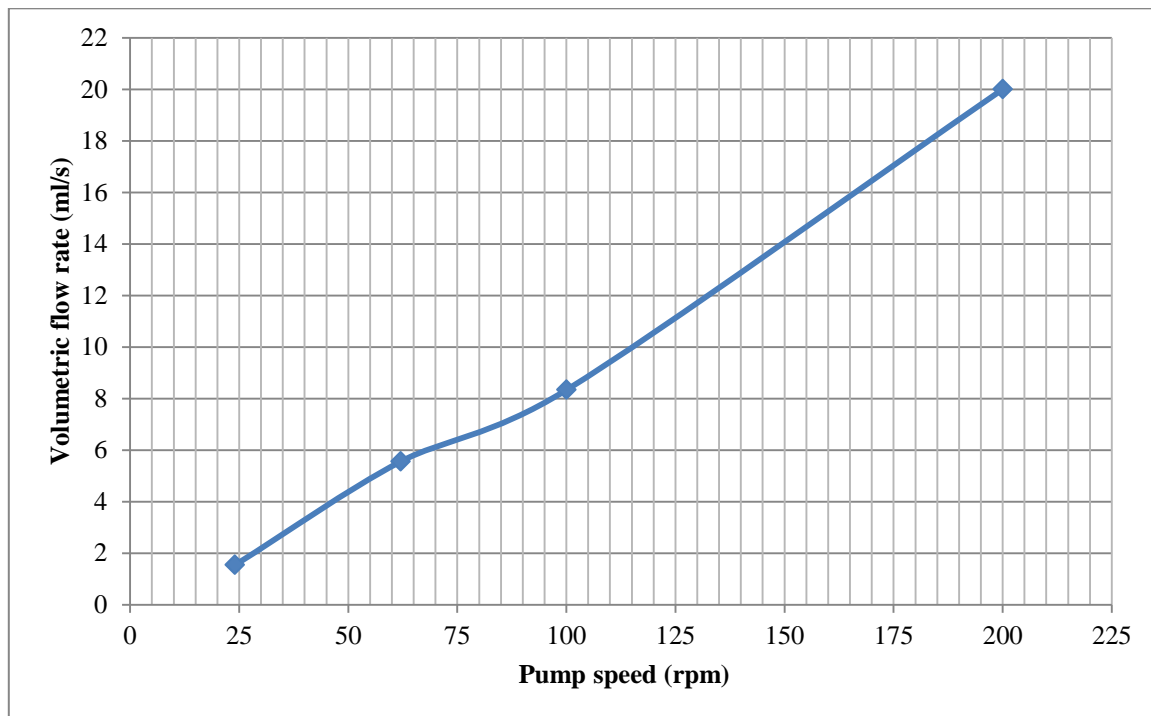
The rotameter was used to control the air supply into the flotation cell. The calibrations on the rotameter were not in terms of a volumetric flow rate; hence a calibration was necessary to convert the rotameter readings into volumetric flow rates. The plot below shows the relationship between the rotameter readings and the volumetric flow rate of air. It is noted to exhibit a linear trend. Experiments were done in triplicate to obtain the rotameter calibration curve.



**Figure A-1: Rotameter calibration for the (air) rotameter that was used in the flotation process.**

## Appendix A2: Peristaltic pump calibration

The peristaltic pump was used as means of transferring wash water from a source to the wash water distribution device. The settings on the pump were in revolutions per minute (rpm) therefore a calibration had to be done to determine the volumetric flow rate of water pumped into the device. The graph below shows the relationship between the speed of the pump and the volumetric flow rate of wash water transferred by the pump to the wash water distribution device. This relationship, according the figure below, is approximately linear. Experiments were done in triplicate to determine the pump calibration.

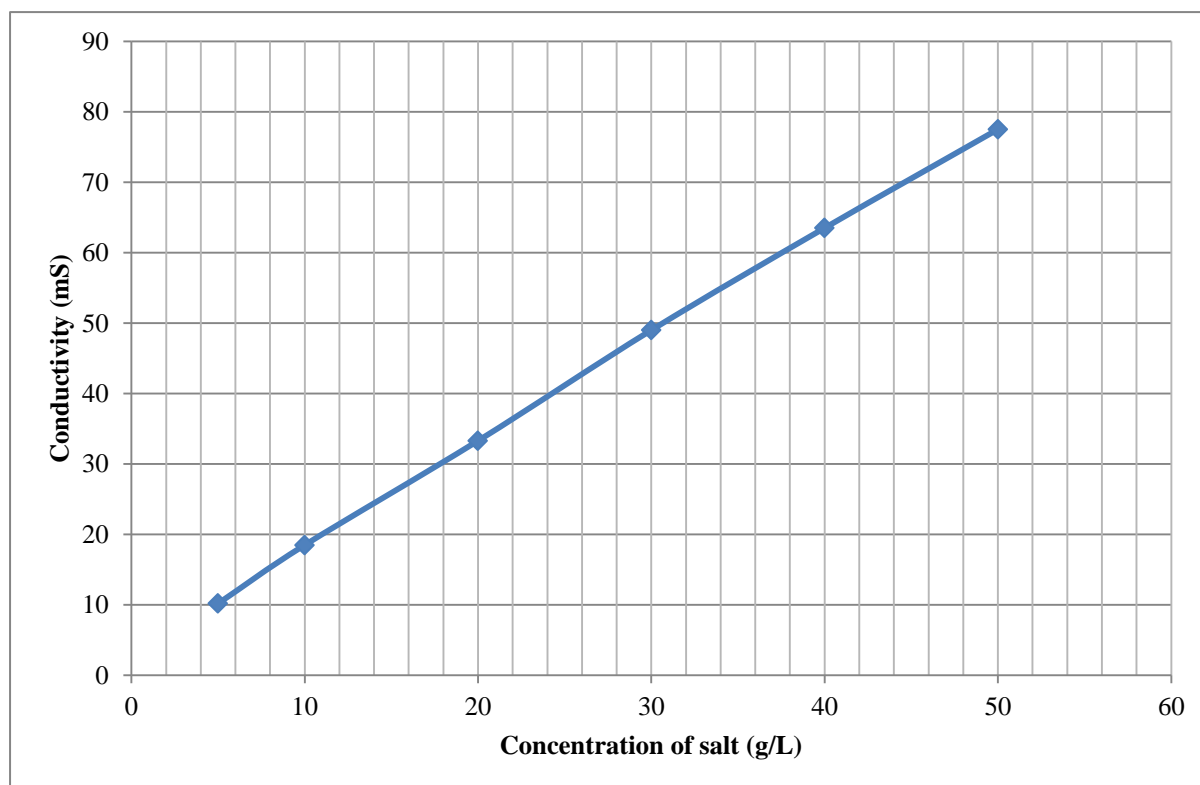


**Figure A-2: Calibration for the peristaltic pump into mL/s shows almost a linear trend.**



### Appendix A3: Conductivity meter calibration

A solution of salt water was used to determine the efficiency of the froth washing. The conductivity meter was used to determine the conductivity of the salt solutions. The liquid part of the concentrate was collected and tested for its conductivity. The calibration was necessary to translate into conductivity values into concentration values in g/L. The calibration shows a linear trend. Experiments were done in triplicate to determine the conductivity meter calibration.



**Figure A-3: Calibration for the conductivity meter.**

## APPENDIX B: SAMPLE CALCULATIONS

### Appendix B1: Moisture content

The UG-2 ore obtained for the experiments was obtained from the underflow of a hydro cyclone in the secondary circuit. Therefore, water was present in the solids. After the remaining water was drained off the crushed ore, it was thoroughly mixed. Numerous sub samples of the ore was then weighed and dried in the oven overnight. Thereafter the dry masses were recorded and the moisture content calculated. The raw data used for the calculation can be found in Table D-1 in Appendix D1. The sample calculations were based on sample 1.

Mass of moist ore + filter paper + tray = 136.7 g

Mass of dry ore + filter paper + tray = 124.1 g

Mass of water = Moist mass – Dry mass

$$= 136.7 \text{ g} - 124.1 \text{ g}$$

$$= \mathbf{12.6 \text{ g}}$$

Mass of filter paper = 3.0 g

Mass of tray = 12.7 g

Mass of moist ore = (Mass of moist ore + filter paper + tray) - Mass of filter paper - Mass of tray

$$= 136.7 \text{ g} - 3.0 \text{ g} - 12.7 \text{ g}$$

$$= \mathbf{121.0 \text{ g}}$$

$$\% \text{ Moisture} = \frac{\text{Mass of water}}{\text{Moist mass of solids}} \times 100$$

$$= \frac{12.6 \text{ g}}{121.0 \text{ g}} \times 100$$

$$= \mathbf{10.41 \%}$$

Similarly, the moisture content for the other samples were calculated and the average was computed to be **10.44%**.

## Appendix B2: % Efficiency of the froth washing

A solution of salt water as the wash water was used to test the efficiency of the froth washing. The efficiency was measured by the degree of displacement of entrained water by wash water and was calculated using the following formula:

$$\eta = \frac{S(t) - C(t)}{C_o - C(t)}$$

Where:  $\eta$  – Efficiency (dimensionless)

$S(t)$  – Concentration of liquid in the flotation concentrate ( $\text{g.L}^{-1}$ )

$C(t)$  – Concentration of liquid in the flotation cell ( $\text{g.L}^{-1}$ )

$C_o$  – Concentration of feed salt water solution ( $\text{g.L}^{-1}$ )

After flotation of the scavenger, the (5) concentrates were filtered and their filtrates collected and analysed for conductivity. The conductivity measurements were translated into concentration values (in  $\text{g.L}^{-1}$ ) using the calibration for the conductivity meter which can be seen in Appendix A3.

The conductivity readings ( $\sigma$ ) for the froth washing tests can be seen in Appendix D4 (raw data). The following sample calculation was based on Test 45.

Concentrate 1 of Test 45 had a conductivity reading 20.0 mS. Inputting this value into the conductivity meter calibration, the concentration in  $\text{g.L}^{-1}$  of the liquid can be obtained:

$$y = 0.6676x - 2.2062$$

$$x = 20.0 \text{ mS}$$

$$y = 0.6676 \times 20.0 - 2.2062$$

$$= \mathbf{11.14 \text{ g.L}^{-1}}$$

$S(t) = 11.14 \text{ g.L}^{-1}$  for concentrate 1 (Table D-14, 'Concentration of salt water' column, Appendix D5)

Similarly, all the other concentrations in were calculated for each of the flotation concentrates and can be seen in the tables of Appendix D5. Plots of  $S(t)$  with time may be found in Figure C-20, Test 45, Appendix C4.

Using the volume of liquid collected from each concentrate, the amount of salt (in grams) in the concentrate was determined.

From, Appendix D3 (for Test 45), the volume of liquid for concentrate 1 is 332.9 mL

$$\begin{aligned}\text{Amount of salt in concentrate 1} &= 11.14 \frac{\text{g}}{\text{L}} \times 0.3329 \text{ L} \\ &= \mathbf{3.71 \text{ g}} \text{ of salt (second column)}\end{aligned}$$

The same procedure was carried out for all five concentrates to obtain the mass of salt that went into the concentrate. The cumulative amount of salt collected by the concentrates was then calculated and can be seen in Table D-14, 'Cumulative salt' column, Appendix D5.

The total amount of salt (in grams) that went into the cell was calculated by multiplying the flow rate of the salt solution wash water (in g/min) by the time (minutes) of each respective concentrate to yield a mass of the total salt that was input into the whole cell in the prescribed time interval.

Volume of salt solution wash water added = 1500 mL (Table D-22 for Test 45, Appendix D6)

Duration of scavenger flotation = 20 minutes

$$\text{Flow rate of salt solution wash water} = \frac{1.5 \text{ L}}{20 \text{ min}} = 0.075 \text{ L} \cdot \text{min}^{-1}$$

The flow rate of the salt solution wash water was calculated by dividing the amount of salt solution wash water by the time of the scavenger flotation (20 minutes).

To obtain the salt input in  $\text{g} \cdot \text{min}^{-1}$ , the feed concentration was used.

$$\text{Feed concentration (4 \% by mass for all solutions)} = 40 \text{ g} \cdot \text{L}^{-1}$$

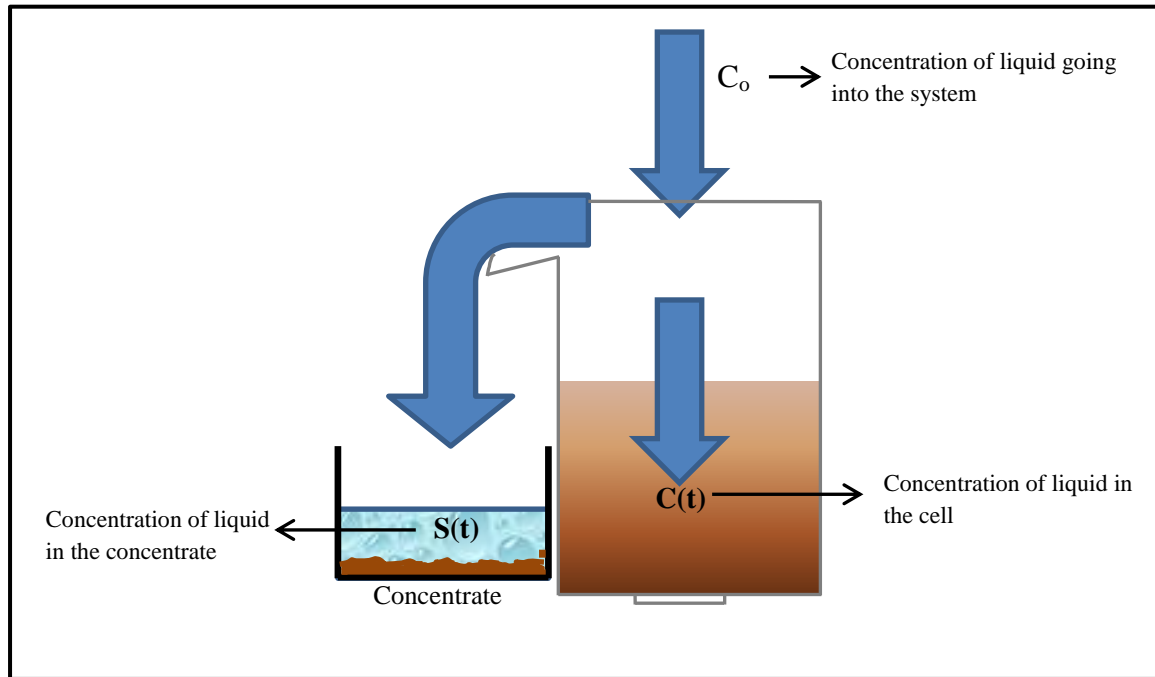
$$\text{Salt input} = 0.075 \text{ L} \cdot \text{min}^{-1} \times 40 \text{ g} \cdot \text{L}^{-1} = 3 \text{ g} \cdot \text{min}^{-1}$$

Time to collect concentrate 1 = 3 minutes

$$\begin{aligned}\text{Total amount of salt that went into the flotation system in 3 minutes} &= 3 \frac{\text{g}}{\text{min}} \times 3 \text{ min} \\ &= \mathbf{9.0 \text{ g}}\end{aligned}$$

Similarly, the amount of salt that was delivered into the cell for the remainder of the time for the other concentrates was calculated (Table D-14, 'Salt that went into the cell' column, Appendix D5).

Knowing the total amount of salt going into the flotation system and the amount of salt that came out in the concentrate, the amount of salt remaining in the cell can be calculated by mass balance.



**Figure B-1: The split of the wash water (salt solution). A fraction of the salt solution went into the concentrate and another fraction went into the cell.**

Mass of salt remaining in the cell = Mass of salt fed into the system – Mass of salt in the concentrate

$$= 9.0 \text{ g} - 3.71 \text{ g}$$

$$= \mathbf{5.29 \text{ g}}$$

The same was done for the other concentrates ('Salt that remained in the cell' column of Table D-14, Appendix D5). The concentration of liquid in the cell was calculated by dividing the mass of salt remaining in the cell by the volume of water in the cell. The volume of water in the cell was fixed by maintaining the pulp level at a constant height of 12 cm.

$$\text{Volume of pulp in cell} = 15.5 \text{ cm} \times 15 \text{ cm} \times 12 \text{ cm} = 2790 \text{ cm}^3$$

$$\text{Volume of solids in cell} = \frac{\text{Mass of solids}}{\text{Density of solids}} = \frac{2000 \text{ kg}}{3.57 \text{ kg.cm}^{-3}} = 560.2 \text{ cm}^3$$

$$\text{Volume of water in cell} = \text{Volume of pulp} - \text{Volume of solids}$$

$$= 2790 - 560.2$$

$$= 2229.8 \text{ cm}^3 \text{ (2.3 L)}$$

$$\begin{aligned}
\text{Concentration of liquid in the cell } C(t) &= \frac{\text{Cumulative mass of salt retained in the cell}}{\text{Volume of water in cell}} \\
&= \frac{5.29 \text{ g}}{2.23 \text{ L}} \\
&= \mathbf{2.37 \text{ g.L}^{-1}}
\end{aligned}$$

This procedure was carried out for each concentrate. Cumulative masses of salt retained in the cell were calculated to obtain the concentration of the liquid in the cell with time to plot the cumulative salt concentration in the cell  $C(t)$  during the flotation period (see Figure, C-20, Test 45, Appendix C4). This was plotted on the same axes as salt concentration in concentrate  $S(t)$ . Trend lines were fitted to the respective curves to obtain  $C(t)$  and  $S(t)$  at average times (See Table D-14, 'Average times between concentrates', Appendix D5).

Trend line equation for:  $C(t)$ :  $y = 0.7388x + 0.314$

$$S(t): y = 0.4696x + 8.6777$$

The average times for the intervals: 0-3; 3-6; 6-10 and 10-20 minutes are 1.5; 4.5; 8; 12.5 and 17.5 minutes respectively.

$$\text{Hence } C(1.5 \text{ minutes}) = 0.7388(1.5) + 0.314 = 1.42 \text{ g.L}^{-1}$$

$$S(1.5 \text{ minutes}) = 0.4696(1.5) + 8.6777 = 9.3821 \text{ g.L}^{-1}$$

The efficiencies were calculated at the average times.

Using the formula for calculating efficiency ( $\eta$ ) as stated above (for the average time interval of 0 to 1.5 minutes):

$$\begin{aligned}
\eta &= \frac{S(t) - C(t)}{C_o - C(t)} \\
&= \frac{9.38 \text{ g.L}^{-1} - 1.42 \text{ g.L}^{-1}}{40.0 \text{ g.L}^{-1} - 1.42 \text{ g.L}^{-1}} \\
&= \frac{8.83 \text{ g.L}^{-1}}{37.13 \text{ g.L}^{-1}} \\
&= 0.2063
\end{aligned}$$

$$\% \text{ Efficiency} = \eta \times 100 \% = 0.2063 \times 100\% = 20.63 \%$$

Similarly, efficiencies for the other concentrates were calculated and may be seen in Table C-1 of Appendix C4.

## Appendix B3: Head grade

The results from the external laboratory on PGM + Au analysis were used to calculate the head grade. The table below shows how the head grade is calculated and a description will explain how the table is obtained. The sample calculation is based on Test 24 (raw data can be found in Appendix D7).

**Table B-1: Data used to calculate the head grade for Test 24.**

	Mass (g)	Mass %	PGM + Au (g/t)	Units
Rougher	41.90	2.12	27.00	57.34
Scav. Conc. 1	14.70	0.74	7.20	5.36
Scav. Conc. 2	22.80	1.15	4.72	5.45
Tails	1893.60	95.97	0.29	27.83
	1973.0	100.00		95.99
Calculated Head				<b>0.959</b>

Note: The scavenger concentrate masses are the combined masses of concentrates 1 and 2; and concentrates 3, 4 and 5 respectively in the raw data section of Appendix D3.

$$\begin{aligned}
 \text{Mass \%} &= \frac{\text{Mass of concentrate}}{\text{Total mass}} \times 100 \\
 &= \frac{41.90}{1973.0} \times 100 \\
 &= \mathbf{2.12 \%}
 \end{aligned}$$

Similarly, all other mass percentages were calculated.

The results for the PGM + Au content were obtained from an external laboratory.

$$\text{Units} = \text{Mass \%} \times \text{PGM + Au (g/t)}$$

$$\begin{aligned}
 &= 2.12 \% \times 27.00 \text{ g/t} \\
 &= \mathbf{57.34 \% \text{ g/t}}
 \end{aligned}$$

The Units for all the samples were calculated similarly and added.

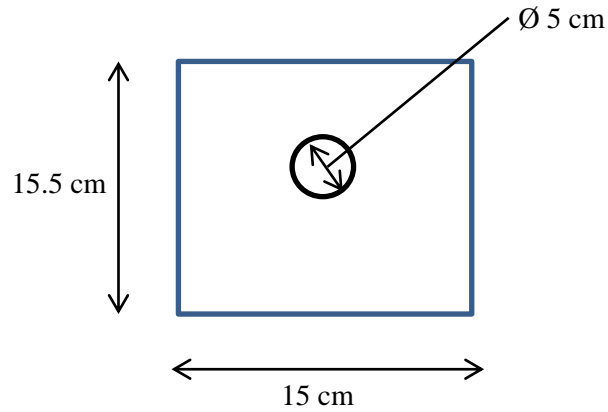
$$\text{Sum units} = 57.34 + 5.36 + 5.45 + 27.83 = 95.99 \% \text{ g/t}$$

$$\text{Head grade} = \frac{\text{Sum units}}{\text{Sum mass \%}}$$

$$= \frac{95.99 \% \text{ g/t}}{100.00 \%}$$

$$= \mathbf{0.96 \text{ g/t}}, \quad \text{Similarly, the head grades for all other experiments was calculated.}$$

#### Appendix B4: Air flow rate per unit area ( $\phi$ )



**Figure B-2: Top view of flotation cell showing the surface area used to calculate the air flow rate through the surface area of the cell (cell lip not shown).**

Surface area of rectangle =  $15.5 \text{ cm} \times 15 \text{ cm}$

$$= 232.5 \text{ cm}^2 \text{ or } \mathbf{0.0232 \text{ m}^2}$$

Area taken up by impeller on surface =  $\frac{\pi d^2}{4}$

$$= \frac{\pi(0.05)^2}{4}$$

$$= \mathbf{0.00196 \text{ m}^2}$$

Area by which air leaves the surface = Surface area of rectangle - Area taken up by impeller on surface

$$= 0.0232 \text{ m}^2 - 0.00196 \text{ m}^2$$

$$= \mathbf{0.0212 \text{ m}^2}$$

Calculating air flow rate for a rotameter reading of 6:

Using the rotameter calibration in Appendix A1:

Air flow rate =  $22.417 \times 6 - 4.6749$

$$= 129.83 \text{ ml.s}^{-1} \times \frac{60 \text{ s}}{10^6 \text{ m}^3}$$

$$= \mathbf{0.00778 \text{ m}^3.\text{min}^{-1}}$$



Air flow rate per unit surface area  $\phi$ :

$$\begin{aligned}\phi &= \frac{\text{Air flow rate}}{\text{Surface area}} \\ &= \frac{0.00778 \text{ m}^3 \text{ min}^{-1}}{0.0212 \text{ m}^2} \\ &= \mathbf{0.366 \text{ m}^3 \cdot \text{min}^{-1} \cdot \text{m}^{-2} \quad \text{or} \quad \text{m} \cdot \text{min}^{-1}}\end{aligned}$$

Similarly, the air flows for all the other rotameter readings were calculated and results depicted in Figures C-15 to C-17 in Appendix C3.

## Appendix B5: Significance of an improvement in recovery of platinum by 1 %

- Typical quantity of ore mined a day: 6000 tons per day
- Typical abundance of platinum in the ore: 4 g/t
- Typical recoveries in platinum concentrator plants: 80 %
- Exchange rate for rand to dollar: \$ 1 = R 8.50

For an improvement in recovery of platinum by 1 %, from 80 % to 81 %:

$$\begin{aligned}\text{Quantity of platinum mined} &= 6 \frac{g}{t} \times 6000 \frac{t}{day} \\ &= 24\,000 \frac{g}{day}\end{aligned}$$

Amount of platinum achieved by an improvement in recovery by 1 %:

$$(0.81 - 0.8) \times 24\,000 \frac{g}{day} = 240 \frac{g}{day}$$

Current price of platinum = \$ 1635 per troy ounce

1 troy ounce = 31.10 g

$$\text{Therefore } 240 \frac{g}{day} \times \frac{1 \text{ t oz}}{31.01 \text{ g}} = 7.72 \text{ t } \frac{t \text{ oz}}{day}$$

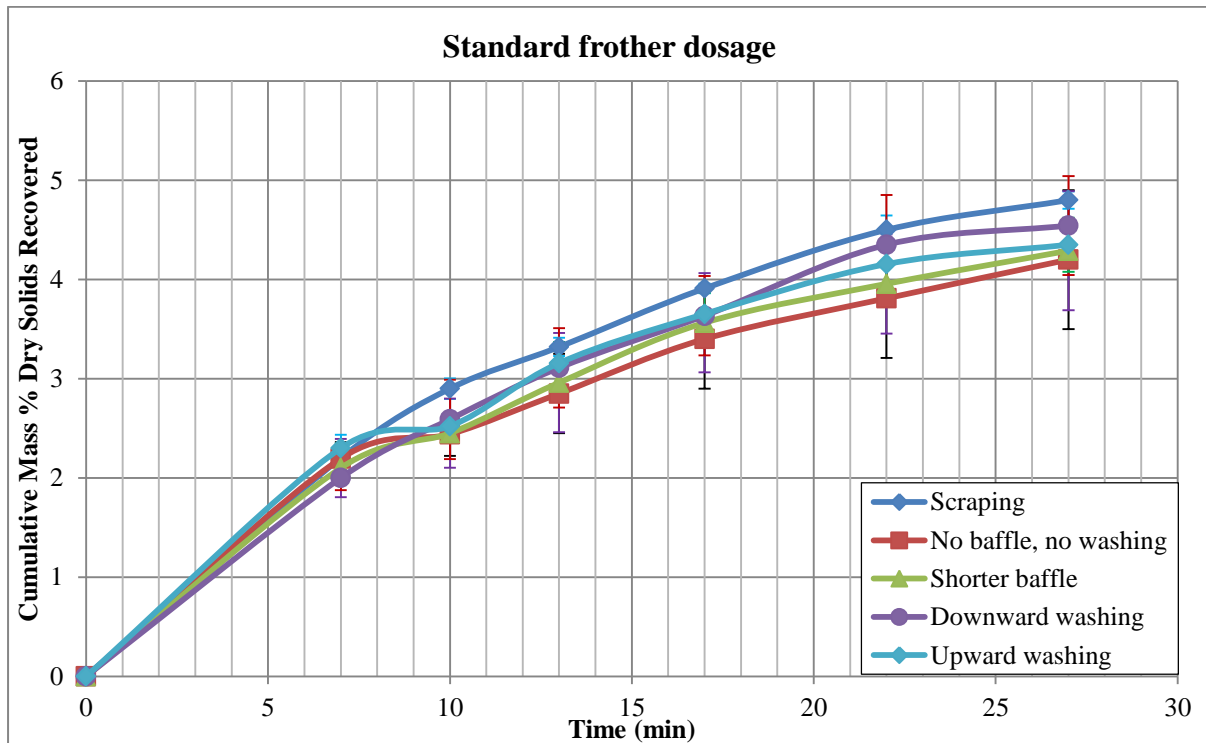
$$7.72 \text{ t } \frac{t \text{ oz}}{day} \times 1635 \frac{\$}{t \text{ oz}} = 12\,616 \frac{\$}{day}$$

$$12\,616 \frac{\$}{day} \times 8.5 \frac{R}{\$} = R\,107\,235 \frac{R}{day}$$

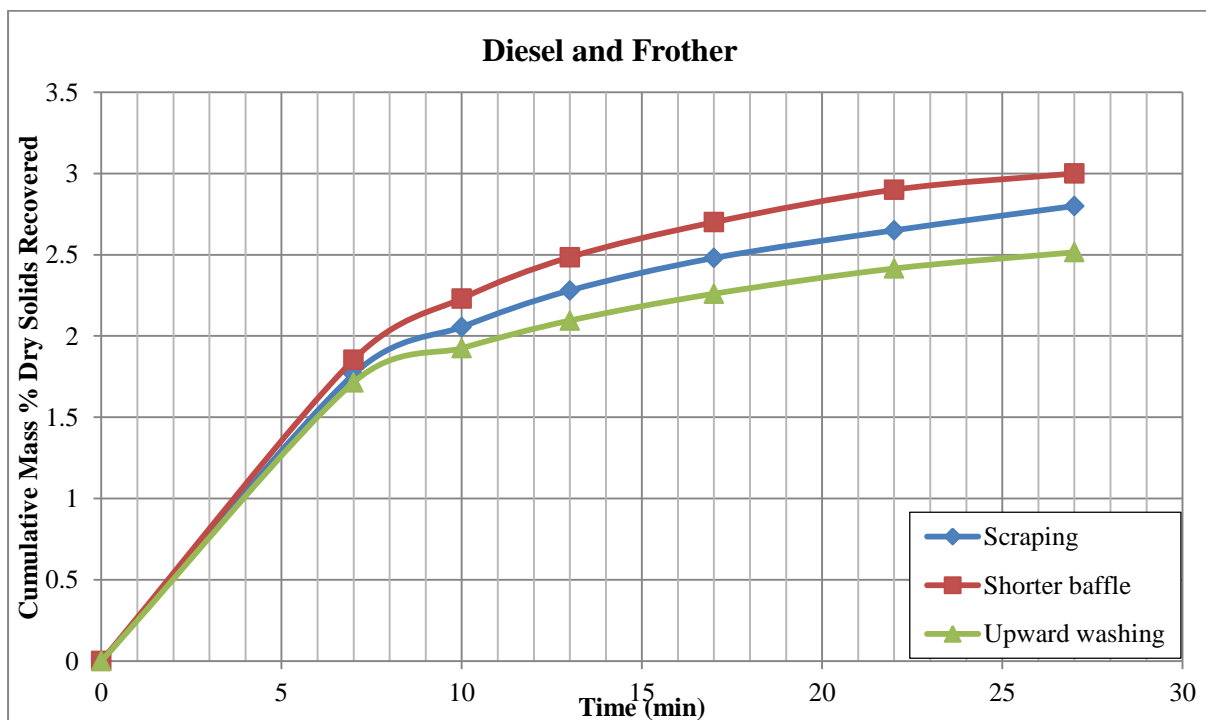
Hence an increase in income through an increase of platinum recovery by 1 % approximates to R100 000 a day.

## APPENDIX C: ADDITIONAL RESULTS

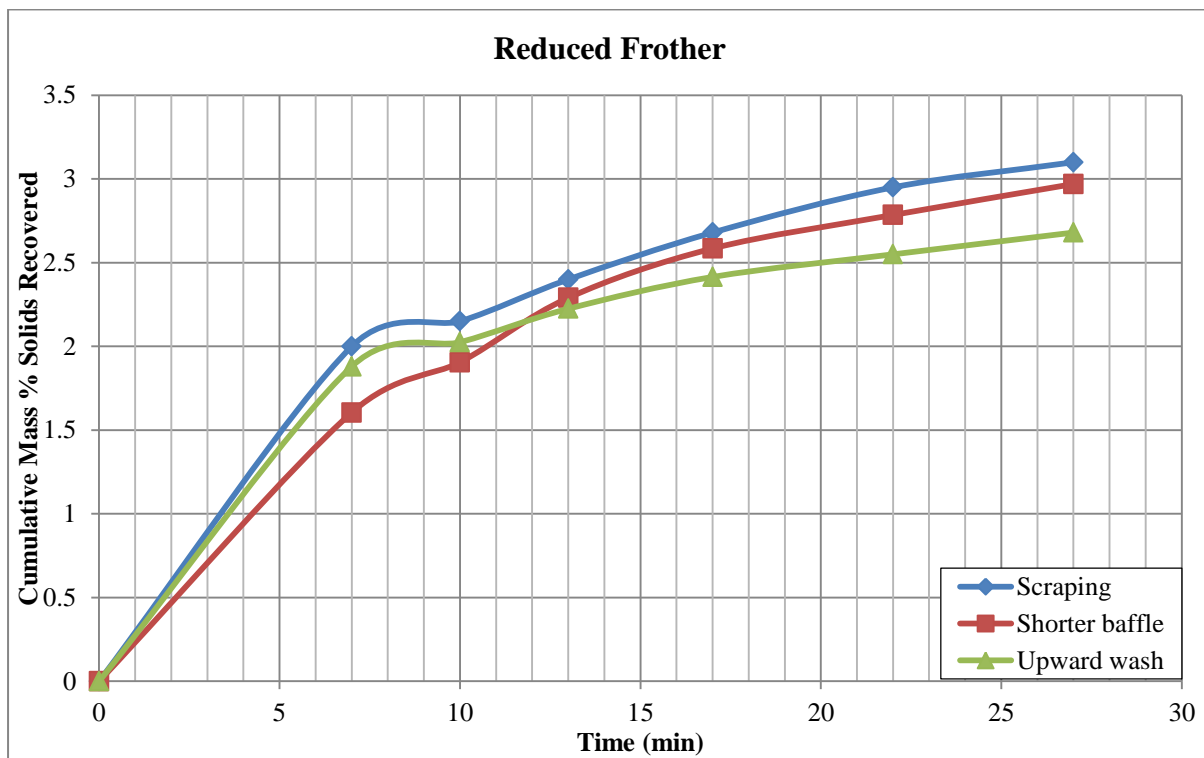
**Appendix C1: Mass recovery curves. Plot of cumulative mass recovered versus time (including the rougher).**



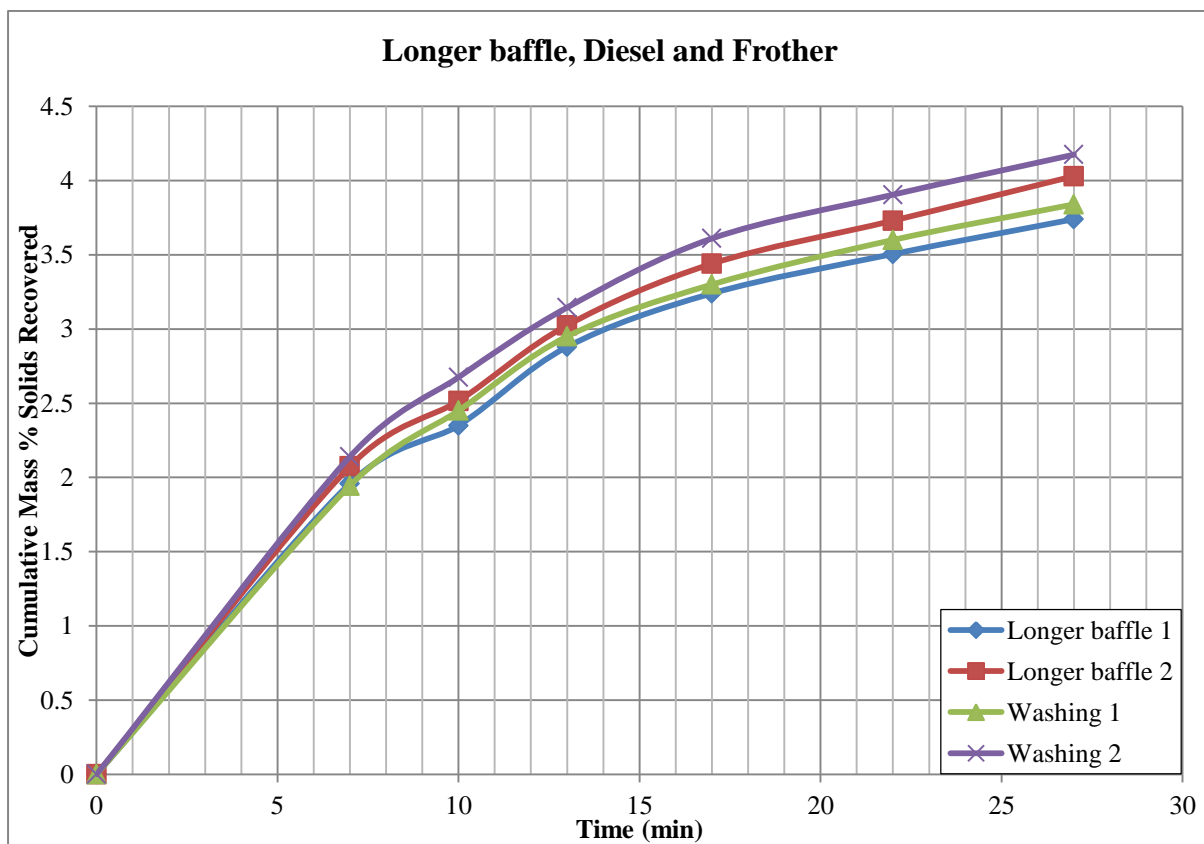
***Figure C-1: Cumulative mass % dry solids recovered versus time (standard frother concentration).***



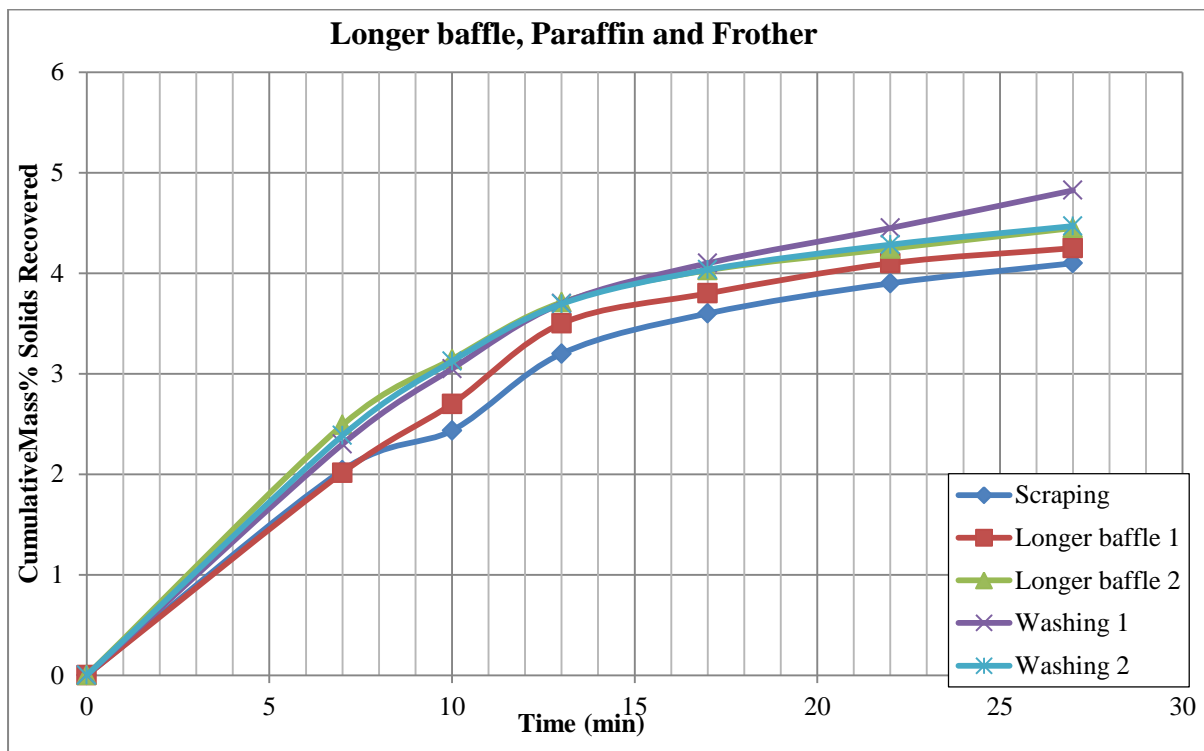
***Figure C-2: Cumulative mass % dry solids recovered versus time for diesel and frother.***



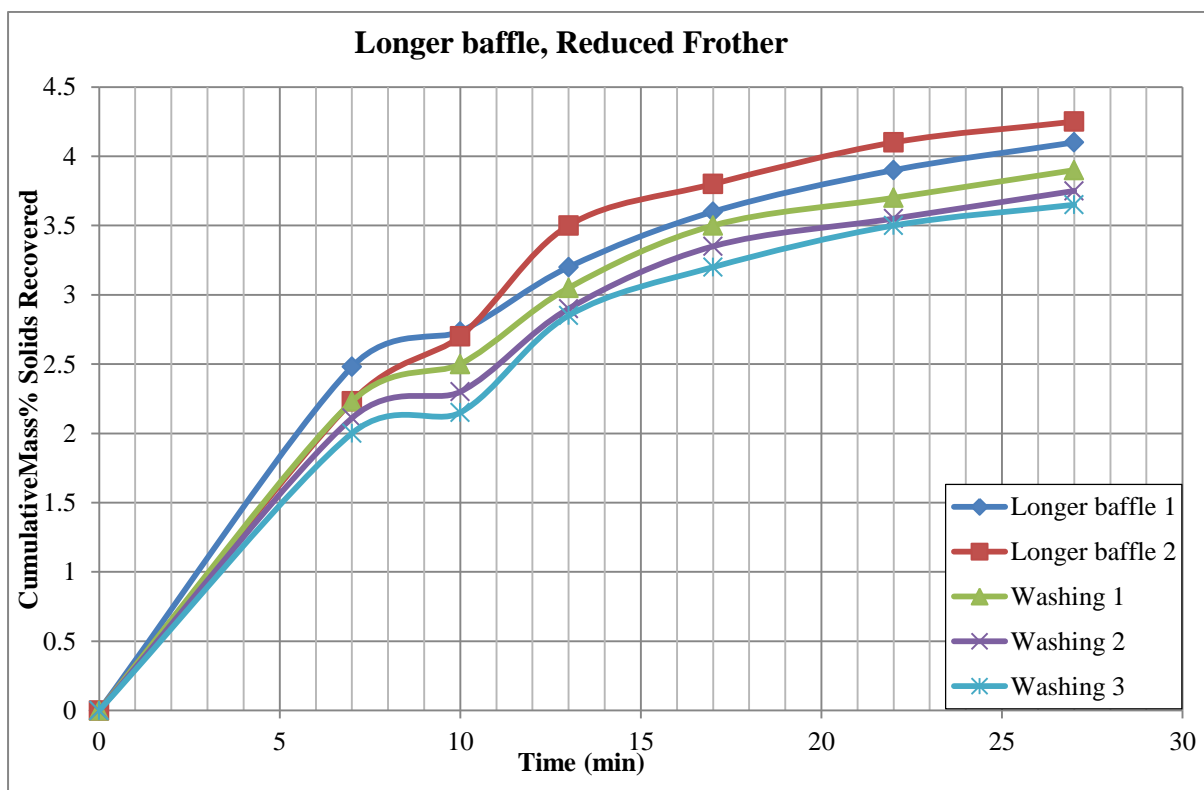
***Figure C-3: Cumulative mass % dry solids recovered versus time for reduced frother.***



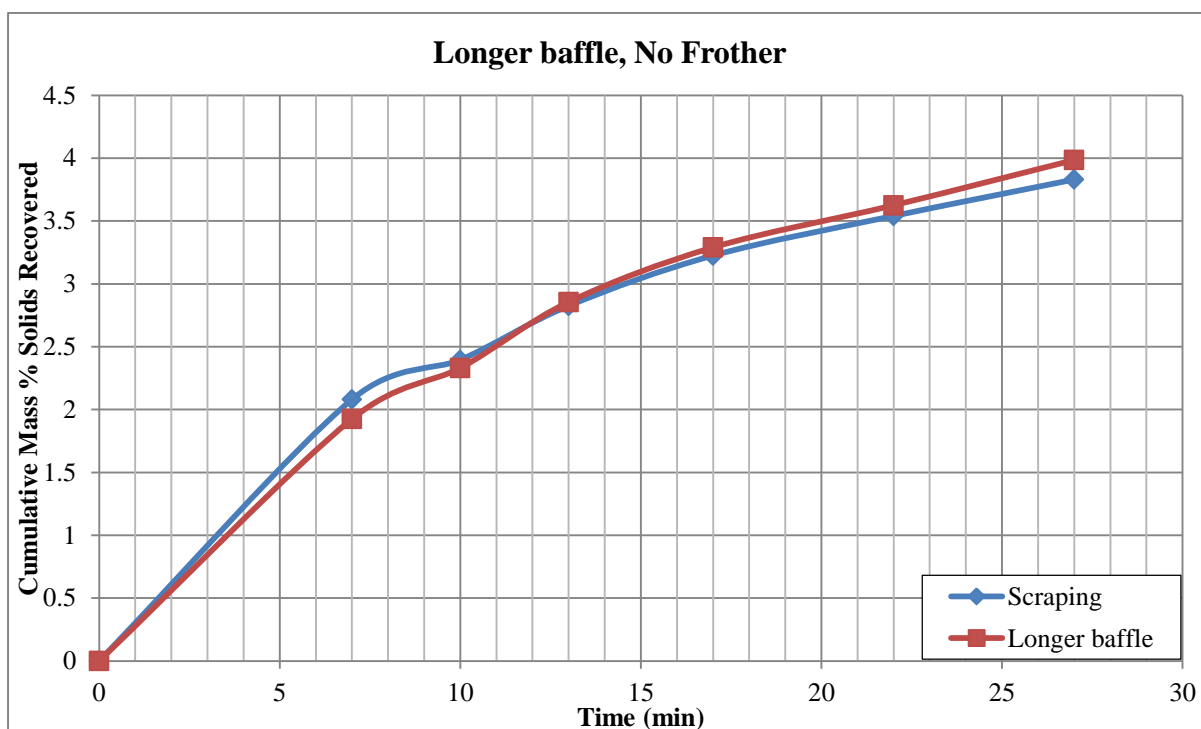
***Figure C-4: Cumulative mass % dry solids recovered versus time for diesel and frother.***



***Figure C-5: Cumulative mass % dry solids recovered versus time for paraffin and frother.***

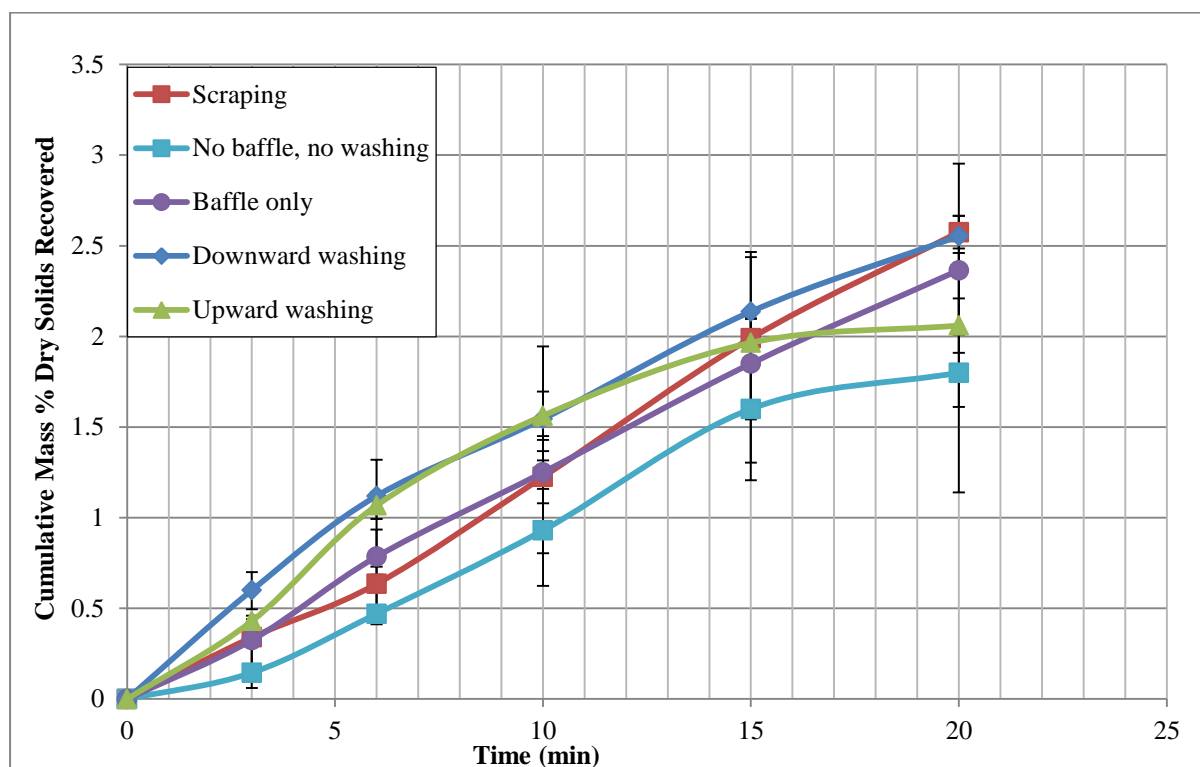


***Figure C-6: Cumulative mass % dry solids recovered versus time for reduced frother dosage.***

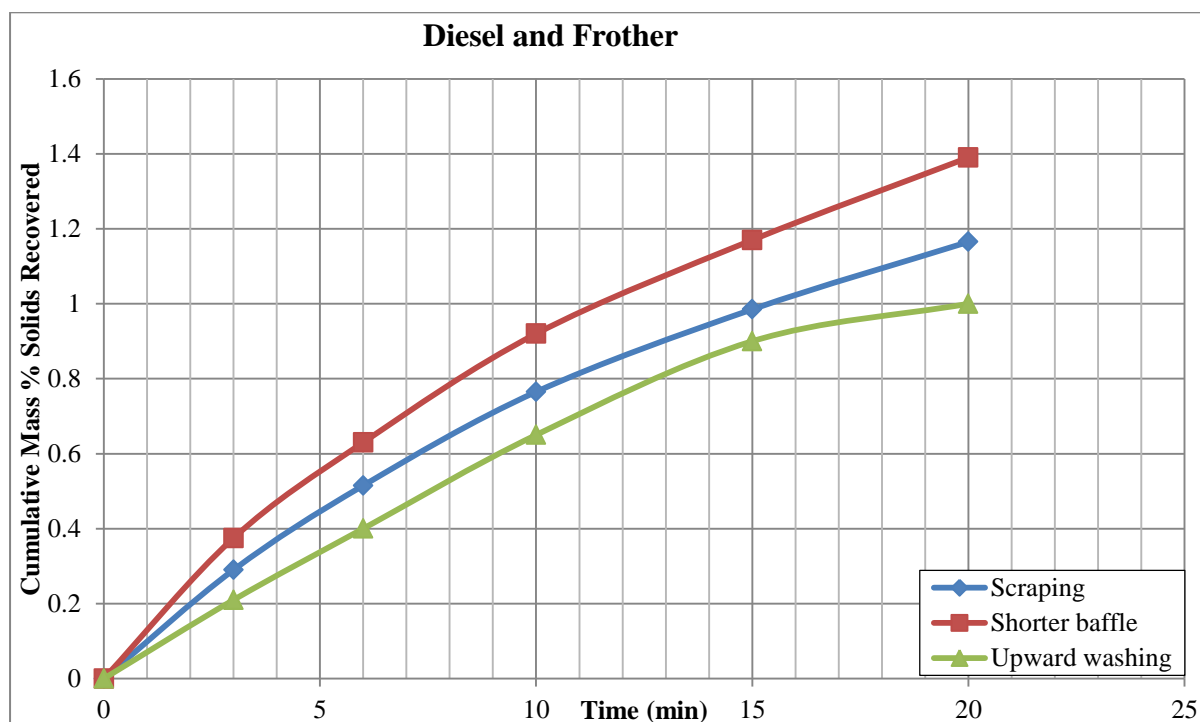


**Figure C-7: Cumulative mass % dry solids recovered versus time for no frother dosage.**

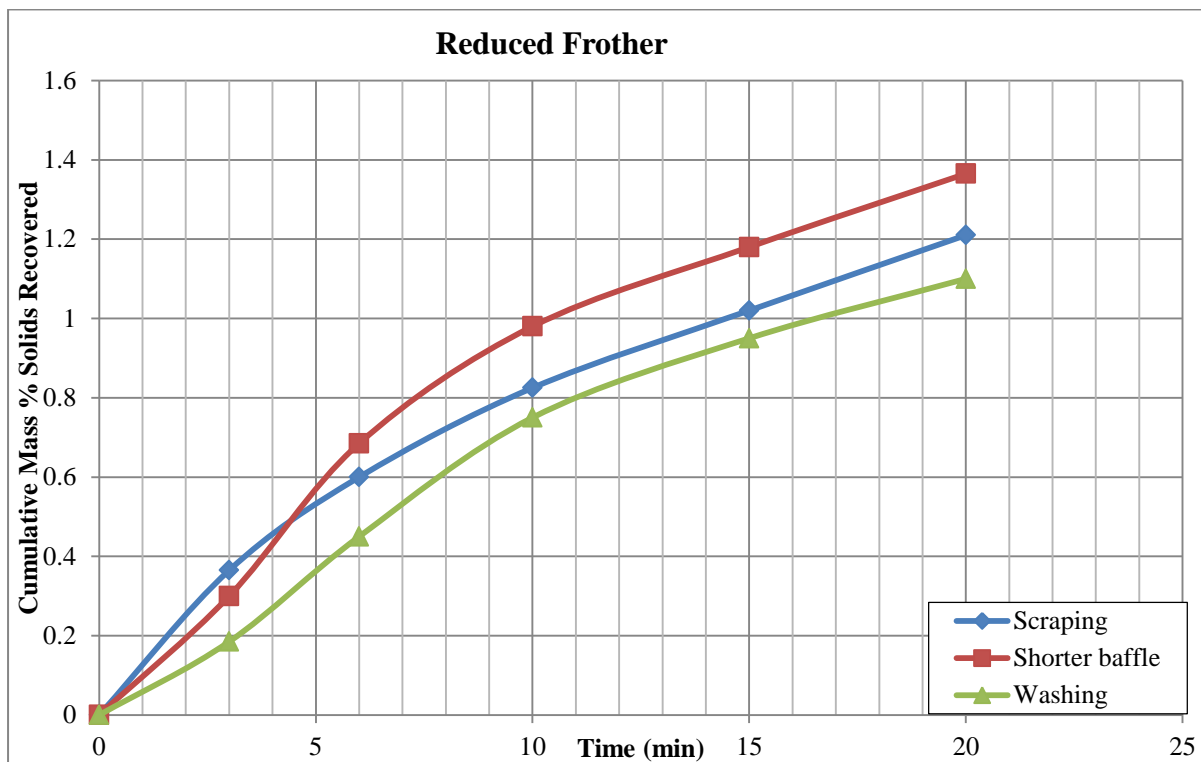
**Appendix C2: Mass recovery curves. Plot of cumulative mass recovered versus time for the scavenger only.**



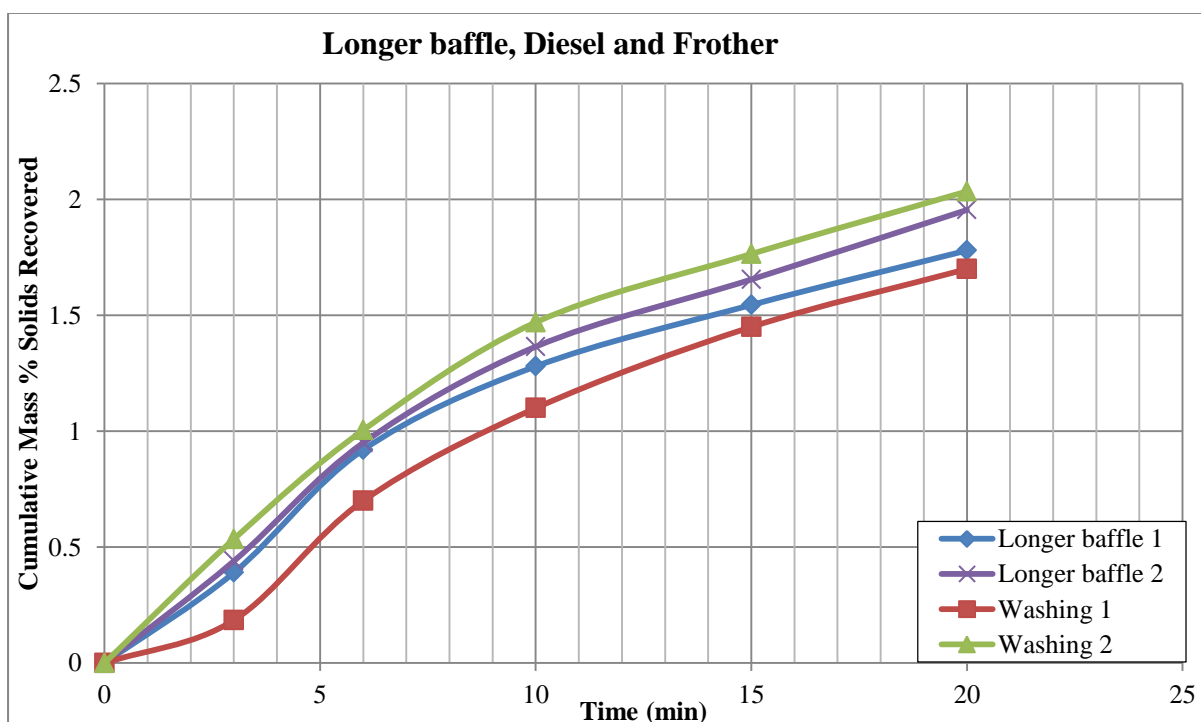
***Figure C-8: Cumulative mass % dry solids recovered versus time for standard frother dosage.***



***Figure C-9: Cumulative mass % dry solids recovered versus time for diesel and frother dosage.***

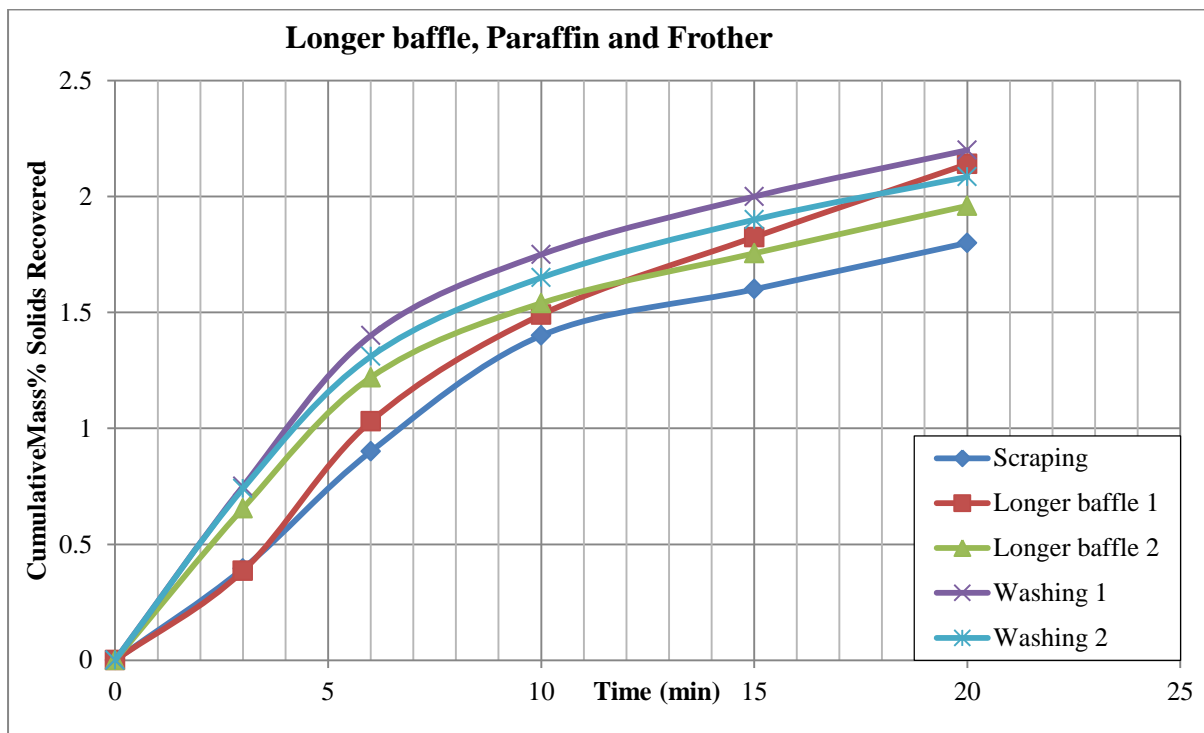


***Figure C-10: Cumulative mass % dry solids recovered versus time for reduced frother dosage.***

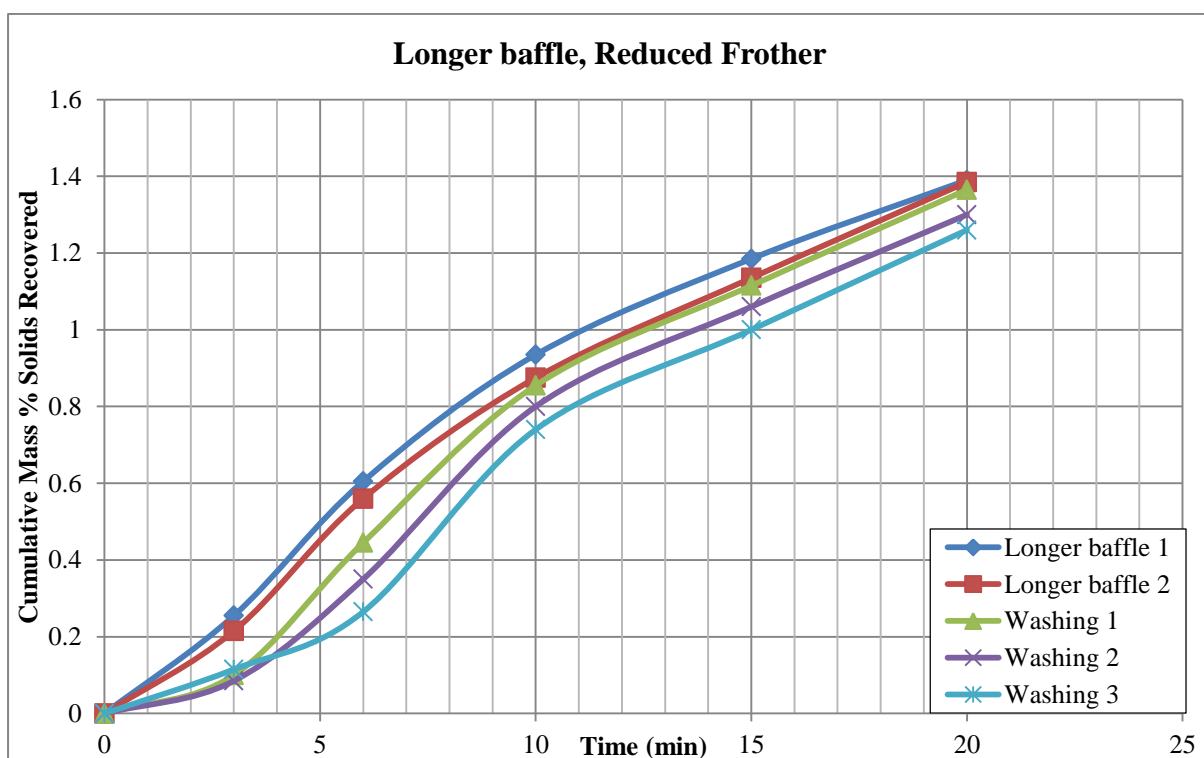


***Figure C-11: Cumulative mass % dry solids recovered versus time for diesel and frother dosage.***

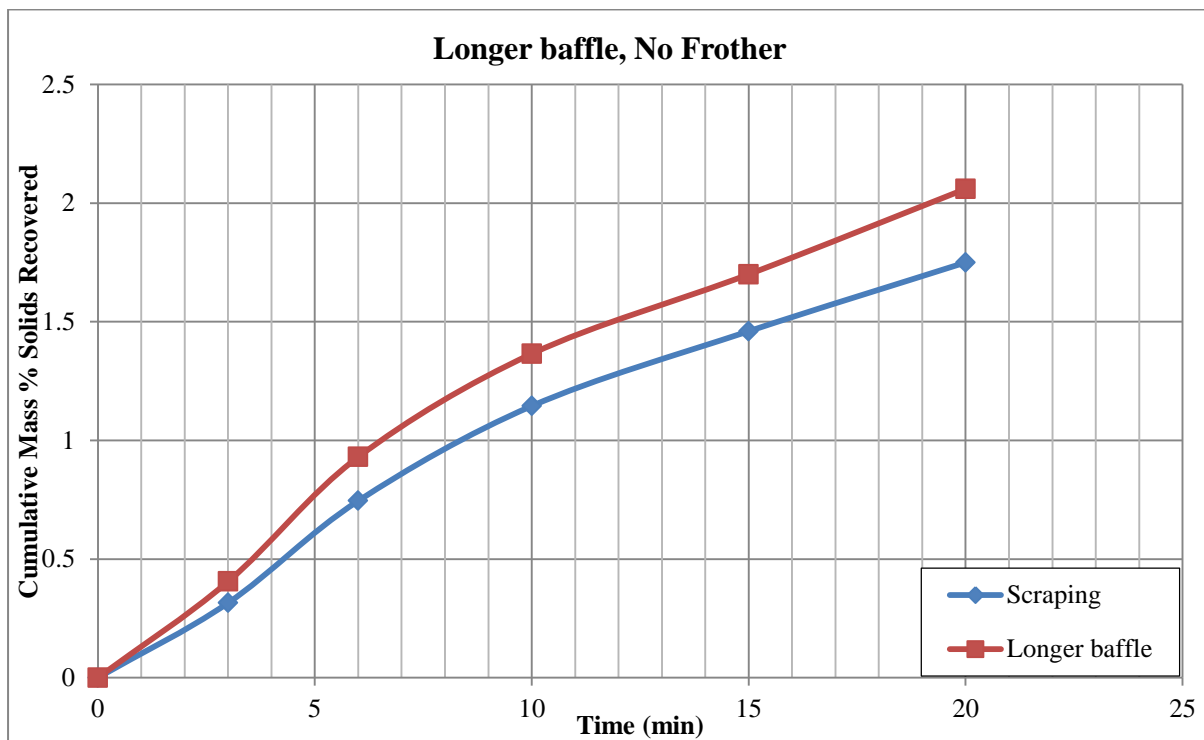




***Figure C-12: Cumulative mass % dry solids recovered versus time for paraffin and frother dosage.***

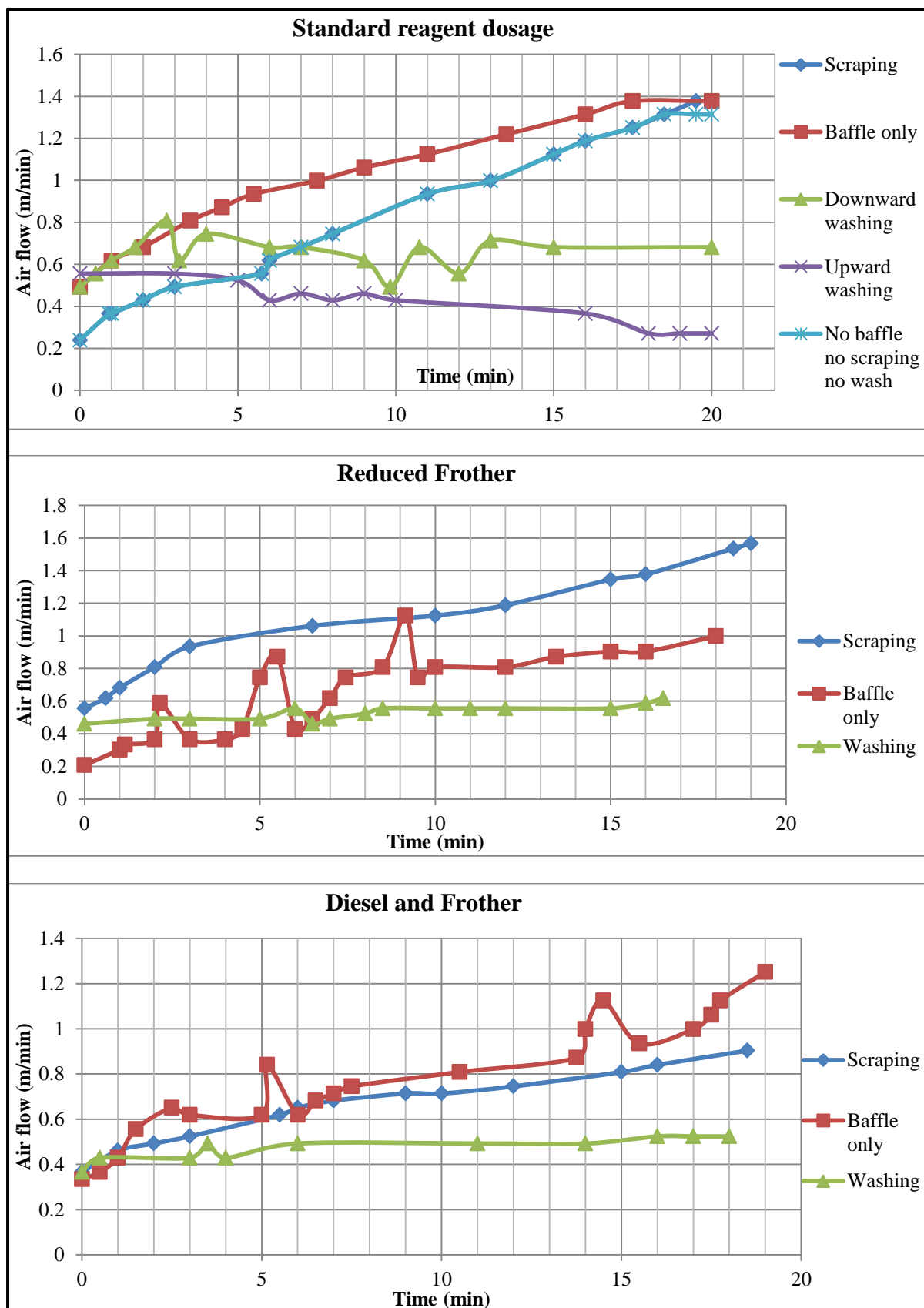


***Figure C-13: Cumulative mass % dry solids recovered versus time for reduced frother dosage.***

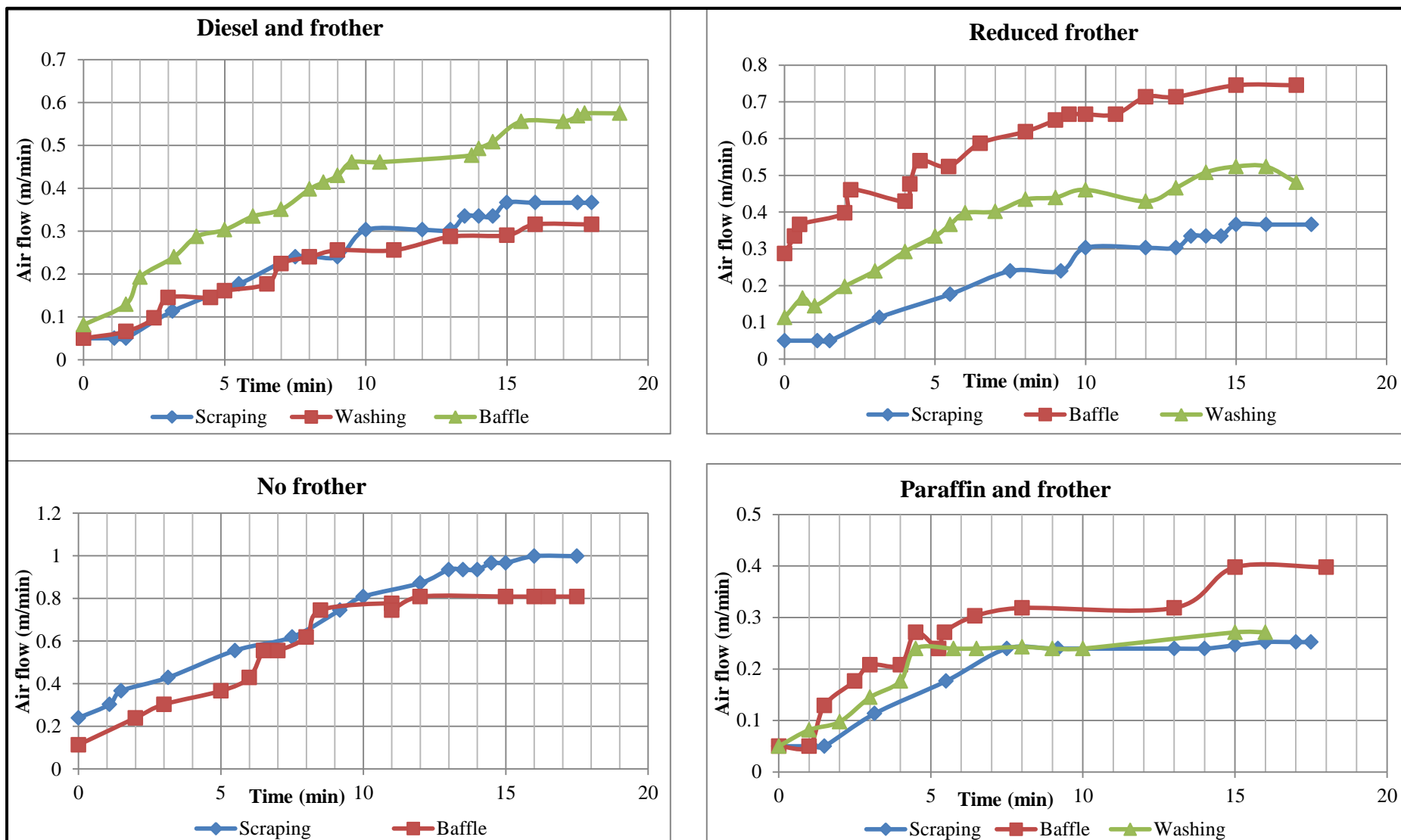


***Figure C-14: Cumulative mass % dry solids recovered versus time for no frother dosage.***

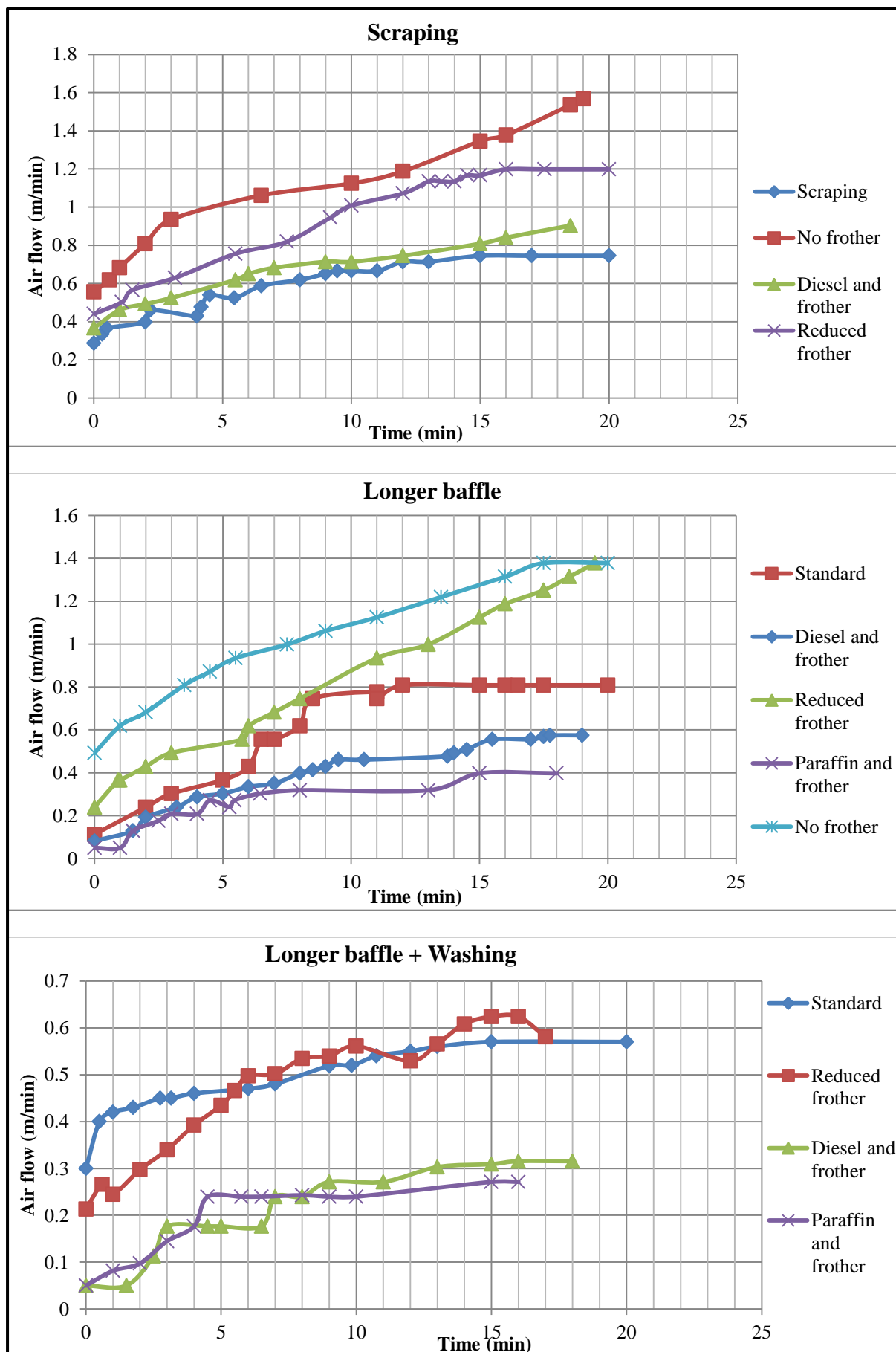
### Appendix C3: Air flow plots



**Figure C-15: Plot of air flow versus time for the standard frother, reduced frother and diesel and frother dosage respectively (The shorter baffle is implied where necessary).**



**Figure C-16: Plot of air flow versus time for the diesel and frother, reduced frother, no frother and paraffin and frother dosage respectively (The longer baffle is implied where necessary).**



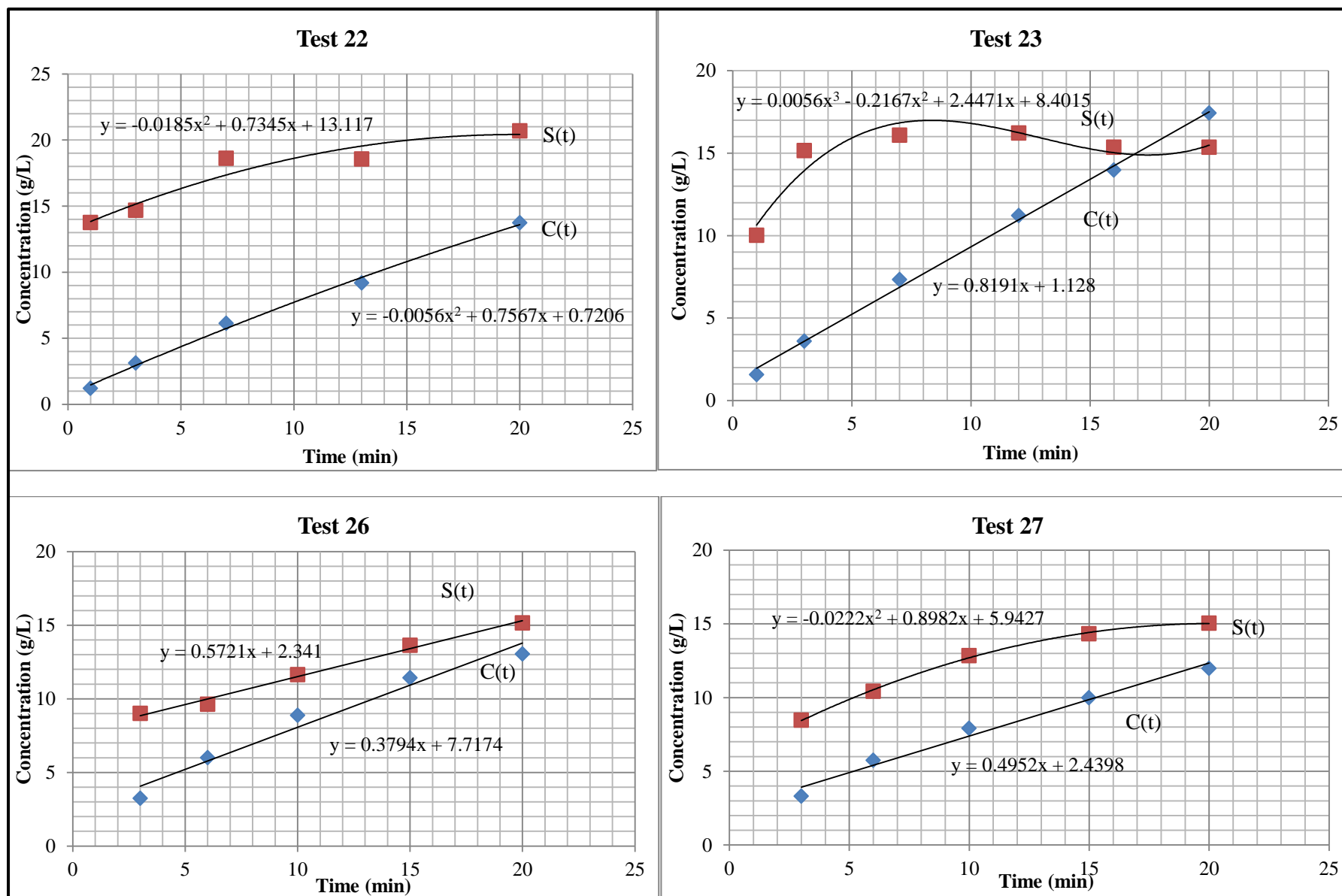
**Figure C-17: Plot comparing the various flotation methods for different reagents used in the scavenger flotation.**

## Appendix C4: Salt tracer tests additional results

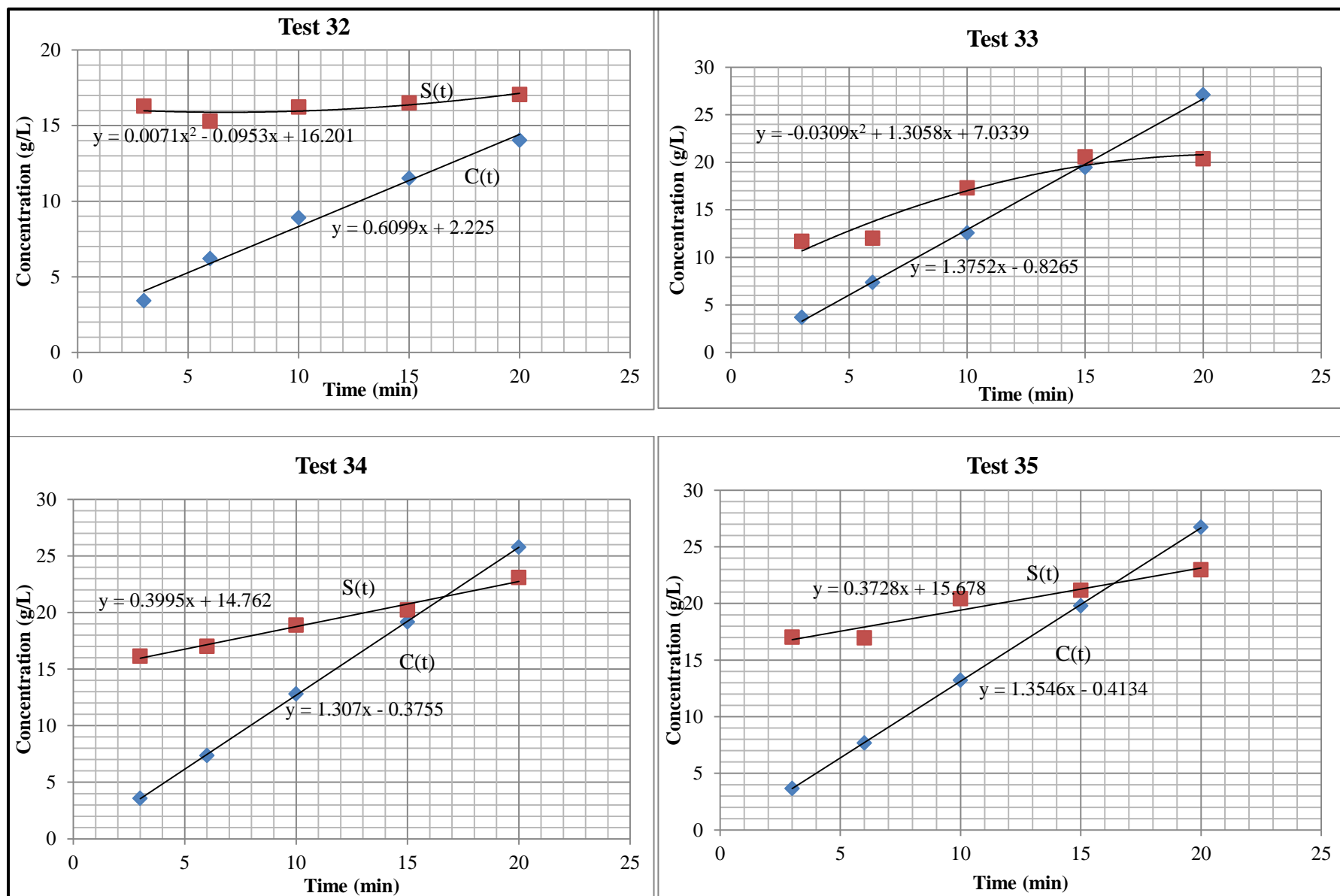
**Table C-1: Efficiencies ( $\eta$ ) for the froth washing tests.**

% Efficiency ( $\eta$ )																		
Test	22	23	26	27	32	33	34	35	36	39	45	46	49	50	53	54	62	63
S. C1	32.03	20.42	14.12	11.19	35.52	20.11	35.71	38.46	16.77	32.52	20.63	21.00	87.29	86.87	36.54	43.32	44.17	44.88
S. C2	32.73	23.99	13.14	14.00	31.65	20.31	30.57	34.39	16.68	29.65	19.97	20.26	80.95	41.95	40.76	47.39	49.51	50.41
S. C3	33.72	29.99	11.87	16.07	27.08	18.19	22.47	28.21	15.77	25.29	18.69	19.28	51.68	11.23	43.19	46.93	50.42	51.23
S. C4	33.88	24.97	9.98	16.47	21.11	9.37	6.12	16.53	12.87	17.57	16.72	17.78	56.29	3.21	41.04	41.13	47.00	47.51
S. C5	30.47	10.46	7.46	13.27	14.29	17.38	31.13	6.67	6.19	4.87	13.94	15.69	74.31	18.18	28.43	35.10	45.06	45.64
S. C6		3.89																

S.C – Scavenger concentrate

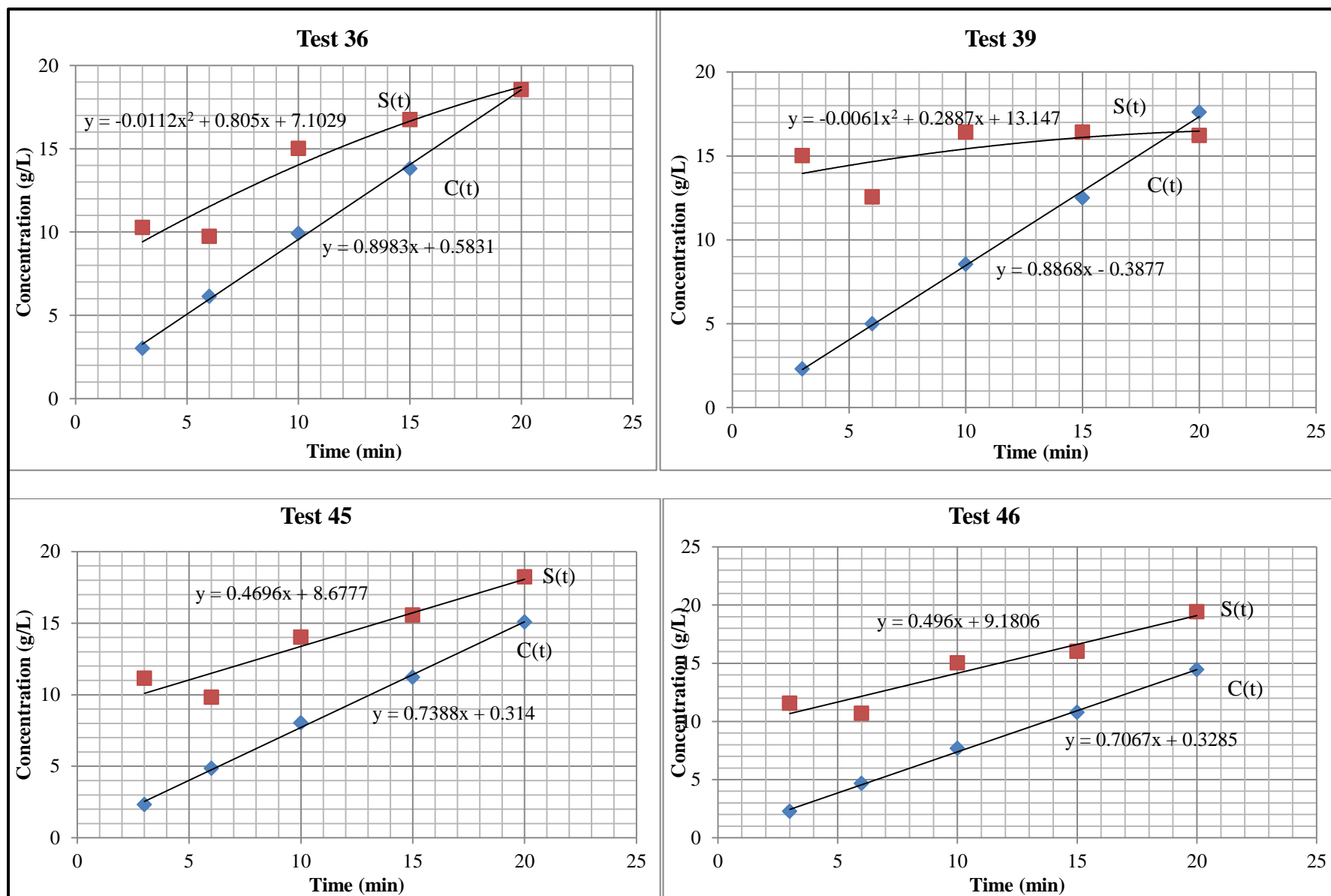


***Figure C-18: Concentration versus time plots of the froth washing tests among tests 22-27.***

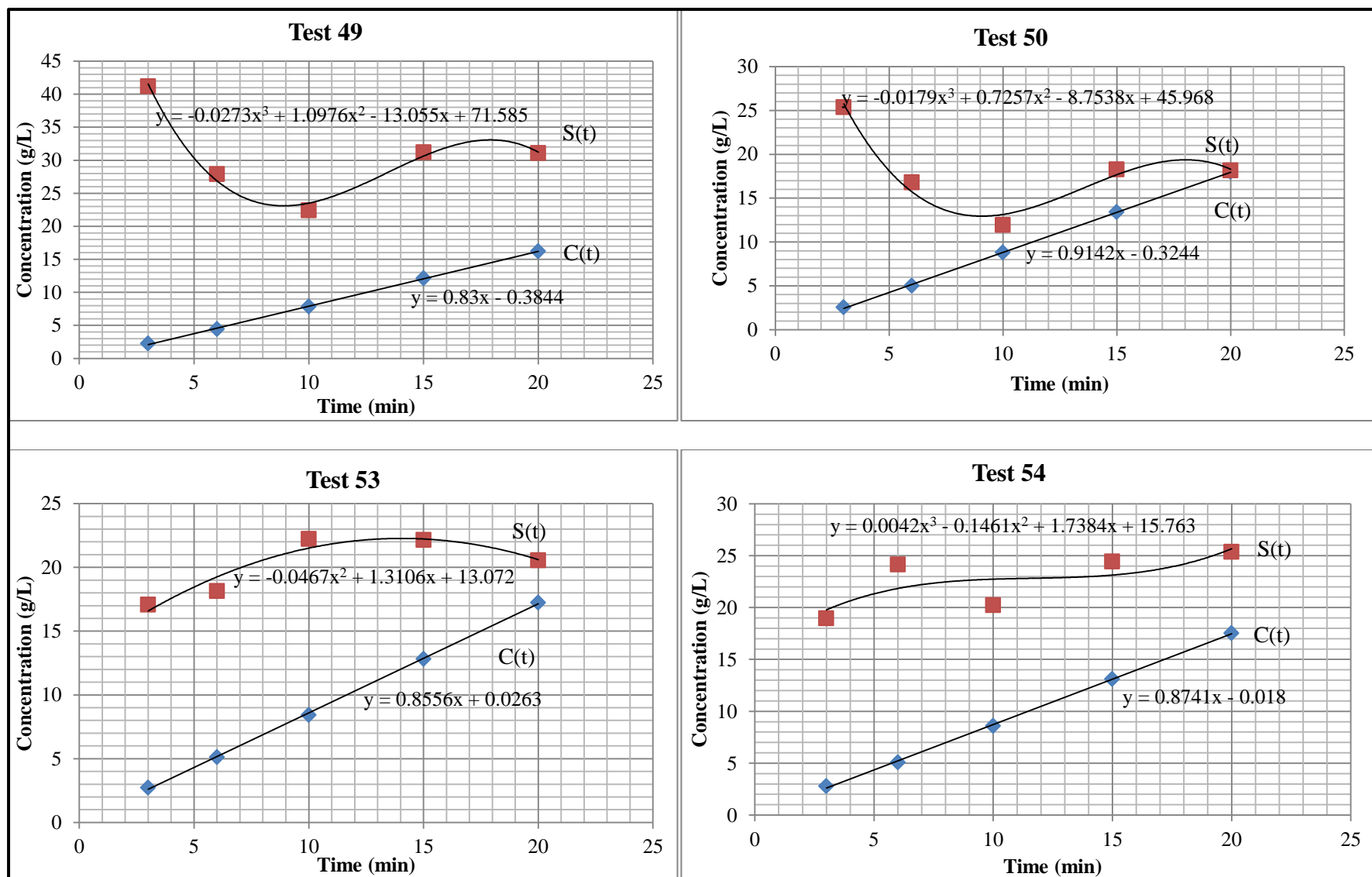


**Figure C-19: Concentration versus time plots or the froth washing tests among tests 32-35.**

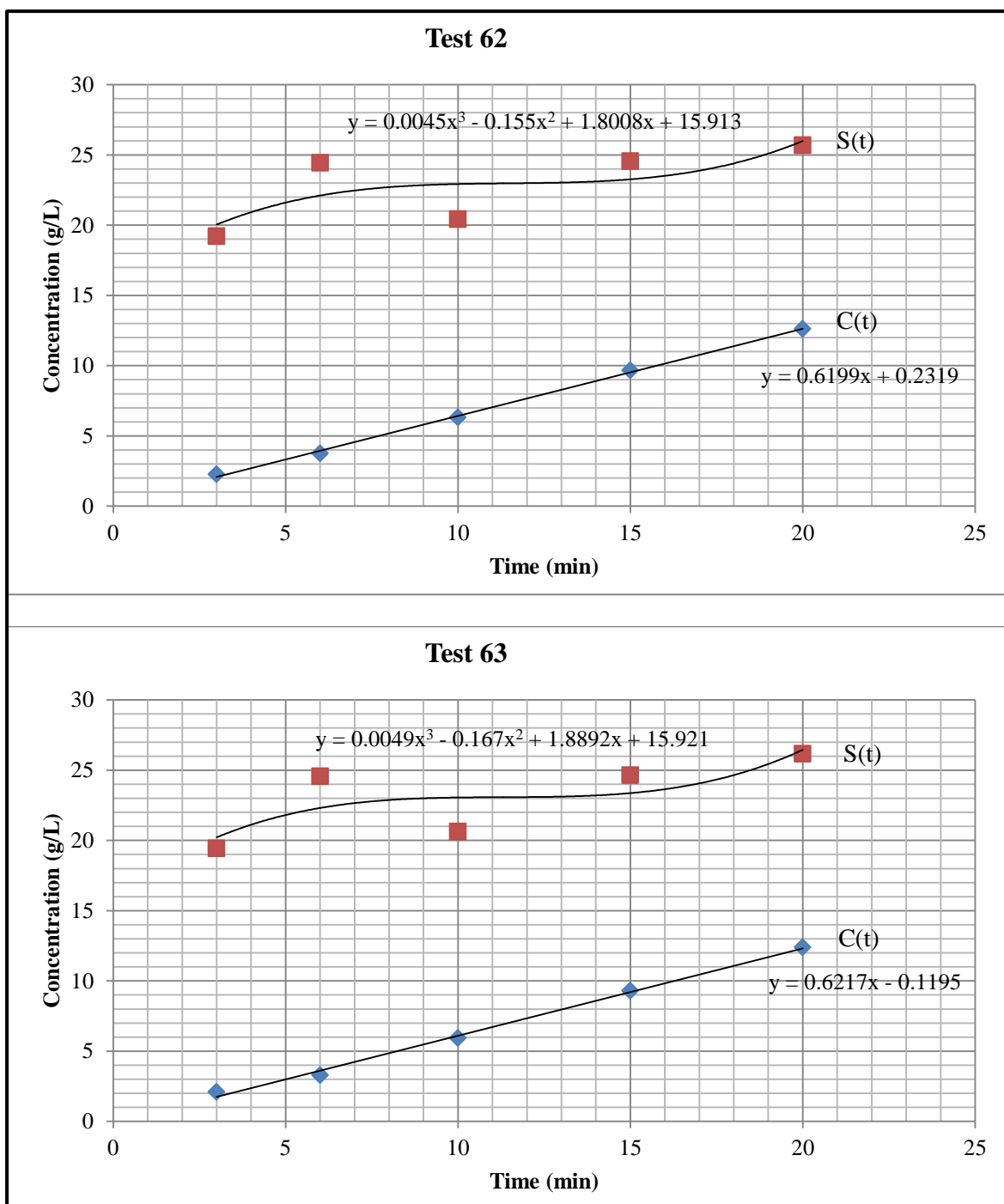




**Figure C-20: Concentration versus time plots of the froth washing tests among tests 36-46.**

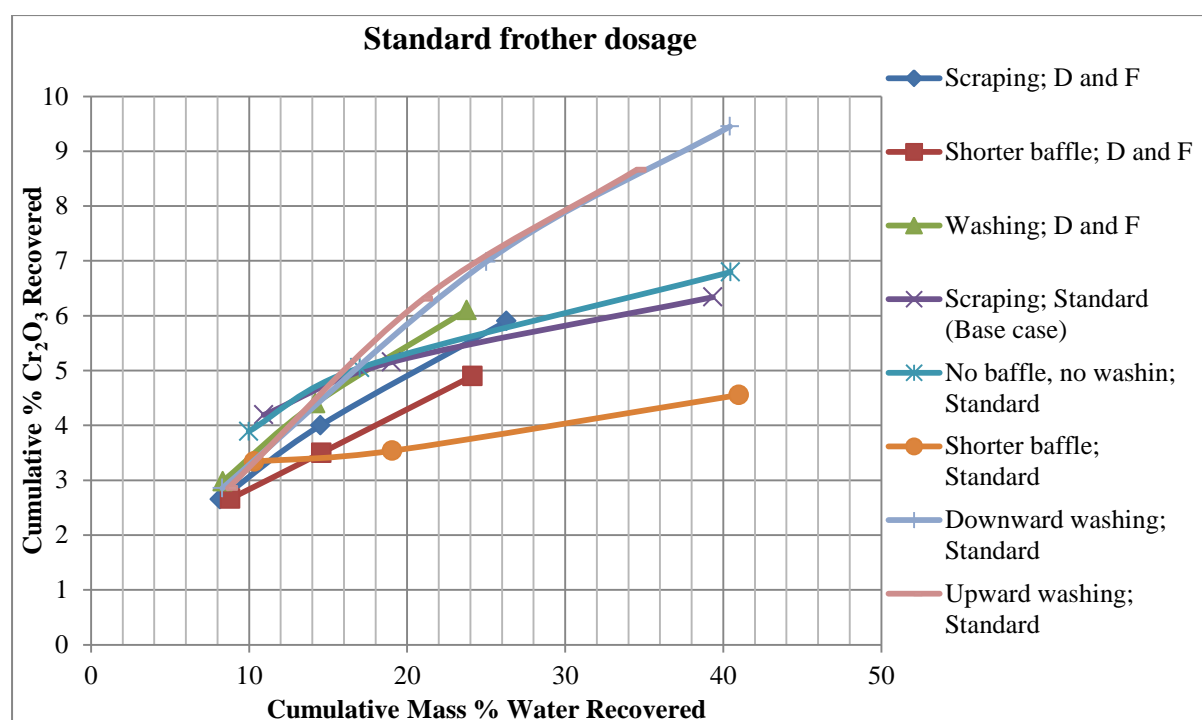


**Figure C-21: Concentration versus time plots for the froth washing tests among tests 49-54.**

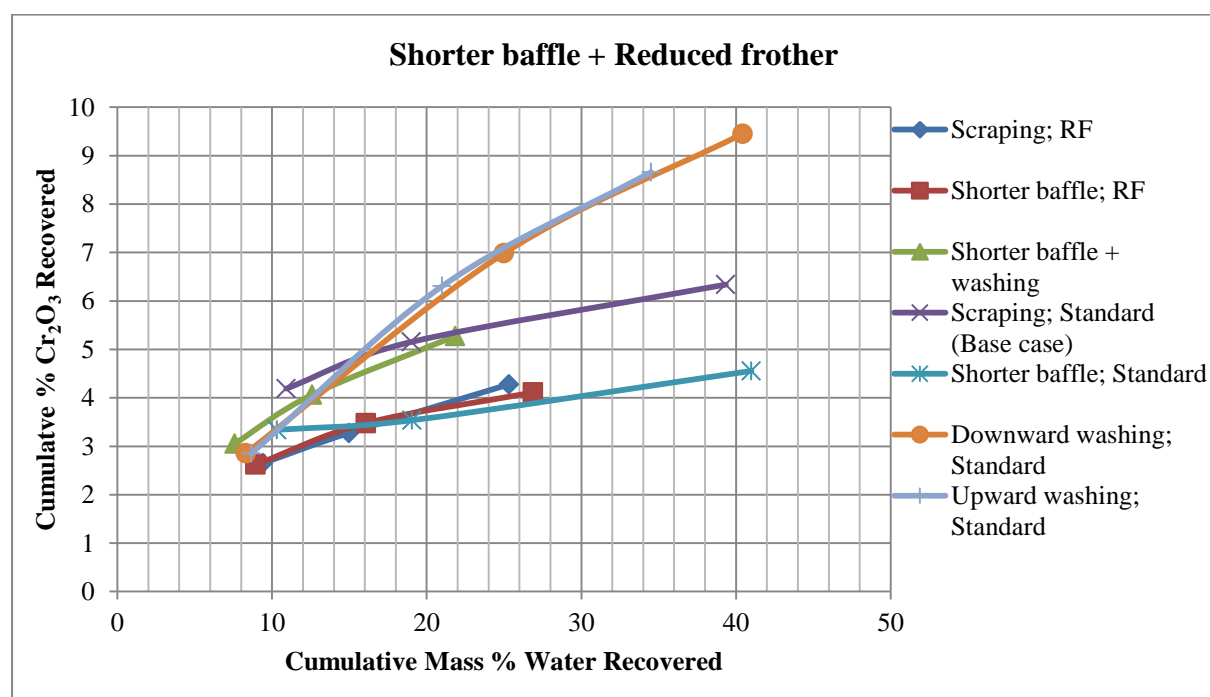


**Figure C-22: Concentration versus time plots of the froth washing tests 62-63.**

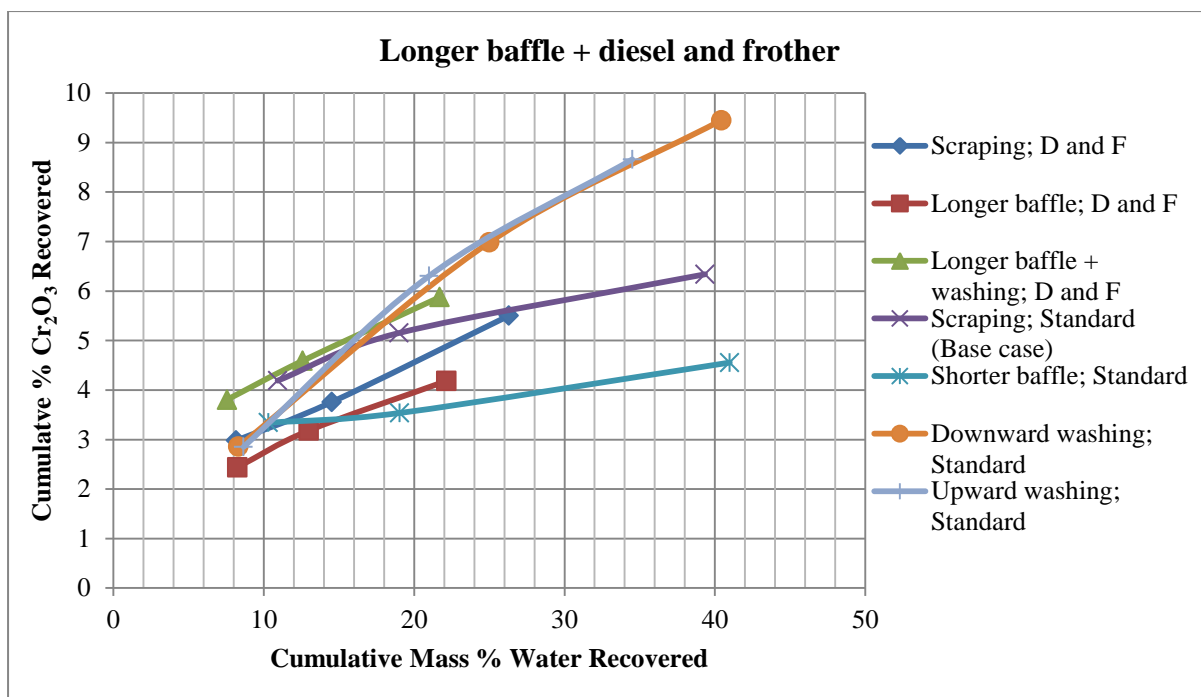
## Appendix C5: Water recovery plots



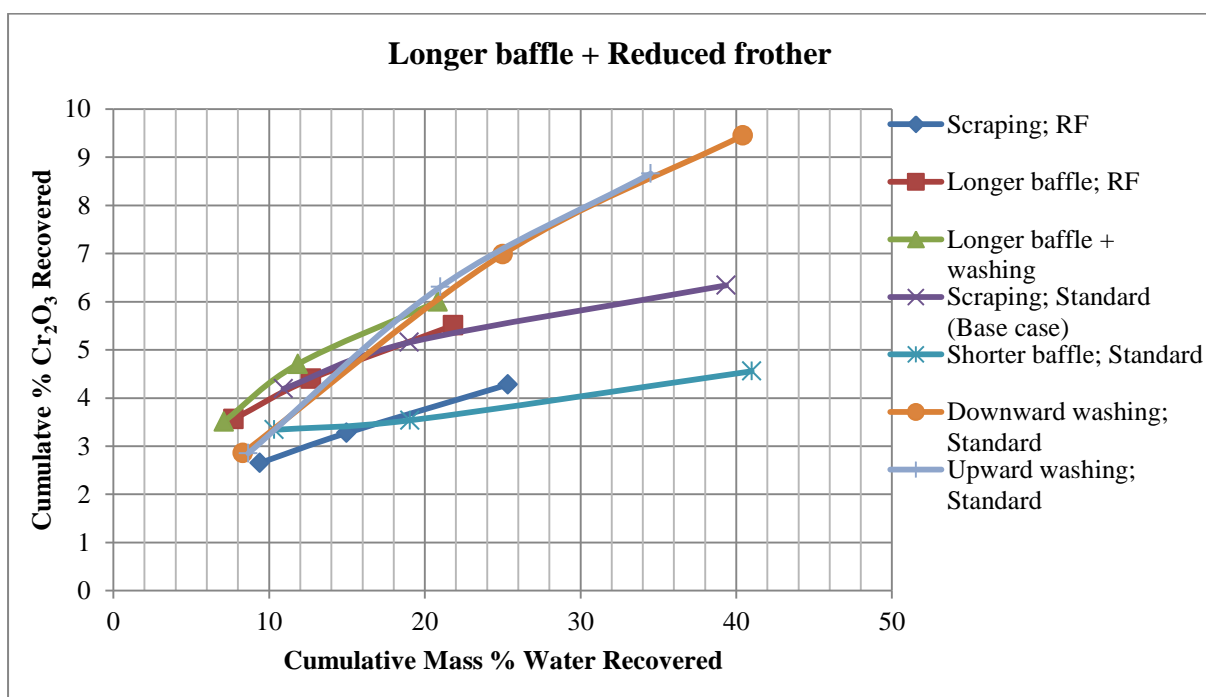
**Figure C-23: Water recovery plot for the standard frother dosage (shorter baffle implied where necessary).**



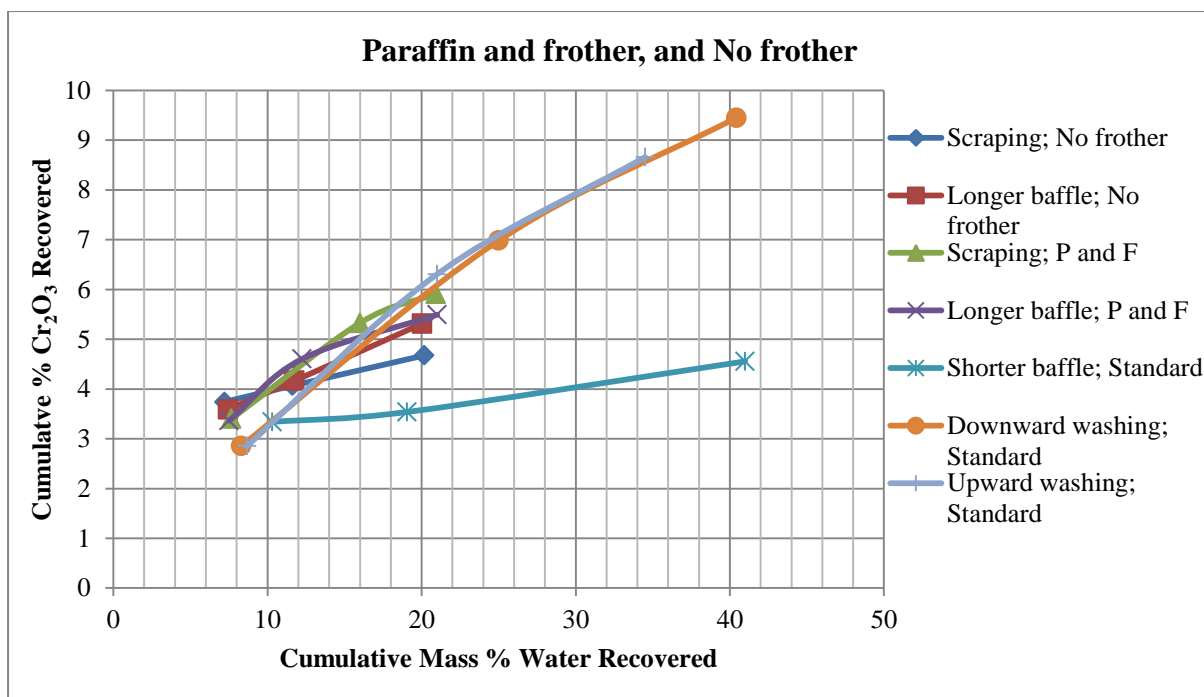
**Figure C-24: Water recovery plot for the reduced frother dosage (shorter baffle implied where necessary).**



**Figure C-25: Water recovery plot for the diesel and frother dosage for the longer baffle.**



**Figure C-26: Water recovery plot for the reduced frother dosage for the longer baffle.**



**Figure C-27: Water recovery plot for the paraffin and frother dosage for the longer baffle.**

## APPENDIX D: RAW DATA

### Appendix D1: Raw data from moisture content tests

***Table D-1: Moisture content data.***

Sample	Mass of tray (g)	Mass of filter paper (g)	Mass of solids + filter paper + tray before drying (g)	Mass of solids + filter paper + tray after drying (g)
1	12.7	3.0	136.7	124.1
2	11.0	3.1	118.6	107.8
3	12.2	3.2	111.6	101.3
4	12.3	3.2	106.2	97.0
5	12.3	3.1	111.0	101.2
6	12.5	3.1	116.5	105.6

## Appendix D2: Raw data from milling curve tests

The milling curve was determined by milling 2 kg of Tests for various times which was then passed through the 75  $\mu\text{m}$  sieve tray.

**Table D-2: Mass data from milling curve.**

	Test 1	Test 2	Test 3	Test 4	Test 5	Test 6	Test 7	Test 8	Test 9	Test 10
Milling time (min)	10	10	20	20	30	30	40	40	50	50
Passed 75 $\mu\text{m}$ (g)	324.5	515.0	978	871.1	1484.4	1319.7	1599.4	1679.7	1918.7	1920.5



### Appendix D3: Mass data from flotation tests

Test		Configuration	Reagent dosage	Mass dry solids recovered (g)	Time (min)	Volume of water recovered (mL)
1	Rougher	Scraping	Standard	132.1	7	166.2
	S. Concentrate 1			4.4	8	11.7
	S. Concentrate 2			6.5	10	14.9
	S. Concentrate 3			12.3	14	22.6
	S. Concentrate 4			22.3	27	35.8
	Tailings			1836.2	27	2142.9
2	Rougher	Scraping	Standard	48.5	7	68.6
	S. Concentrate 1			3.2	8	10.5
	S. Concentrate 2			8.4	10	17.7
	S. Concentrate 3			12.6	14	22.9
	S. Concentrate 4			20.4	27	33.5
	Tailings			1938.3	27	2306.8
3	Rougher	Scraping	Standard	40.2	7	59.2
	S. Concentrate 1			5.3	8	13
	S. Concentrate 2			11	10	20.7
	S. Concentrate 3			14.3	14	25.1
	S. Concentrate 4			24.6	27	37.3
	Tailings			1922.2	27	2261.2
4	Rougher	No scraping, no baffle, no washing	Standard	48.8	7	69.4
	S. Concentrate 1			16.9	8	27.8
	S. Concentrate 2			20.5	10	32.8
	S. Concentrate 3			28.9	14	44.2
	S. Concentrate 4			18.3	27	48.6
	Tailings			1866.0	27	2131.7
5	Rougher	Shorter baffle only	Standard	46.2	7	67.6
	S. Concentrate 1			2.6	8	9.5
	S. Concentrate 2			10	10	19.2
	S. Concentrate 3			18.3	14	29.5
	S. Concentrate 4			29.1	27	43.7
	Tailings			1908.6	27	2253.7
6	Rougher	Shorter baffle only	Standard	44.1	7	296.2
	S. Concentrate 1			4.4	8	74.1
	S. Concentrate 2			6.5	10	139.2
	S. Concentrate 3			12.3	14	206
	S. Concentrate 4			22.3	27	609.7
	Tailings			1902.2	27	2099.7
7	Rougher	Shorter baffle only	Standard	37.5	7	402.3
	S. Concentrate 1			3.2	8	83
	S. Concentrate 2			7.7	10	183
	S. Concentrate 3			8.3	14	216.2
	S. Concentrate 4			22.3	27	667.3
	Tailings			1910.6	27	2095.6
8	Rougher	Shorter baffle only	Standard	40.7	7	368.8
	S. Concentrate 1			4.4	8	88.6
	S. Concentrate 2			9.5	10	170.3
	S. Concentrate 3			21.9	14	237.5
	S. Concentrate 4			26.7	27	705.4
	Tailings			1891.6	27	2098.4

Test		Configuration	Reagent dosage	Mass dry solids recovered (g)	Time (min)	Volume of water recovered (mL)
9	Rougher	Shorter baffle + downward washing	Standard	44.3	7	441.1
	S. Concentrate 1			2.8	8	106.2
	S. Concentrate 2			11.6	10	369.6
	S. Concentrate 3			15.1	14	630.9
	S. Concentrate 4			46.9	27	1820.6
	Tailings			1859.2	27	2899.8
10	Rougher	Shorter baffle only	Standard	21.7	7	420.2
	S. Concentrate 1			2.2	8	62.1
	S. Concentrate 2			5.8	10	150.4
	S. Concentrate 3			7.7	14	219.7
	S. Concentrate 4			23.6	27	737.8
	Tailings			1927.2	27	1988.3
11	Rougher	Shorter baffle + downward washing	Standard	49.3	7	540.1
	S. Concentrate 1			2.7	8	117.3
	S. Concentrate 2			13.8	10	413
	S. Concentrate 3			19.6	14	1011.3
	S. Concentrate 4			79.4	27	2599.3
	Tailings			1808.2	27	3100.3
12	Rougher	Shorter baffle + downward washing	Standard	50.5	7	602.1
	S. Concentrate 1			3	8	81.5
	S. Concentrate 2			15.8	10	322.7
	S. Concentrate 3			22.8	14	549.8
	S. Concentrate 4			60.3	27	1639.8
	Tailings			1851.3	27	3096.2
13	Rougher	Shorter baffle + downward washing	Standard	52.7	7	668.5
	S. Concentrate 1			1.8	8	85.1
	S. Concentrate 2			11.9	10	279.2
	S. Concentrate 3			21.4	14	533.2
	S. Concentrate 4			54.1	27	1555.9
	Tailings			1863.4	27	3161.4
14	Rougher	Shorter baffle + downward washing	Standard	52.3	7	577.7
	S. Concentrate 1			3.8	8	90.6
	S. Concentrate 2			17.5	10	337.1
	S. Concentrate 3			21.9	14	494.1
	S. Concentrate 4			60.3	27	1356
	Tailings			1836.0	27	3269.5
15	Rougher	Shorter baffle + downward washing	Standard	51.7	7	546.3
	S. Concentrate 1			3.2	8	88.2
	S. Concentrate 2			13.4	10	279.3
	S. Concentrate 3			21.6	14	479.5
	S. Concentrate 4			64.4	27	1504.8
	Tailings			1834.4	27	3272.0
16	Rougher	Shorter baffle + downward washing	Standard	53.1	7	549.8
	S. Concentrate 1			3	8	88.1
	S. Concentrate 2			13.9	10	285.4
	S. Concentrate 3			22.1	14	502.1
	S. Concentrate 4			57.4	27	1355.1
	Tailings			1834.5	27	3251.7

Test		Configuration	Reagent dosage	Mass dry solids recovered (g)	Time (min)	Volume of water recovered (mL)
17	Rougher	Shorter baffle + downward washing	Standard	46.7	7	553.5
	S. Concentrate 1			3.3	8	103.6
	S. Concentrate 2			14.6	10	334.1
	S. Concentrate 3			19.9	14	547.9
	S. Concentrate 4			87.4	27	2479.8
	Tailings			1812.8	27	2559.5
18	Rougher	Shorter baffle + upward washing	Standard	43.0	7	504.8
	S. Concentrate 1			2.7	8	95.8
	S. Concentrate 2			12.5	10	332.6
	S. Concentrate 3			17.5	14	529.2
	S. Concentrate 4			53.9	27	1719.7
	Tailings			1851.6	27	2801.1
19	Rougher	No baffle, no washing	Standard	49.0	7	542.6
	S. Concentrate 1			0.5	8	46.2
	S. Concentrate 2			4.8	10	125.5
	S. Concentrate 3			18.6	14	314.1
	S. Concentrate 4			79.5	27	1372
	Tailings			1849.8	27	2950.4
20	Rougher	Shorter baffle + downward washing	Standard	48.3	7	430.6
	S. Concentrate 1			5.3	8	111.6
	S. Concentrate 2			18.9	10	313.9
	S. Concentrate 3			25	14	475.4
	S. Concentrate 4			57.3	27	1547.8
	Tailings			1847.7	27	3076.5
21	Rougher	Shorter baffle + downward washing	Standard	48.2	7	476.5
	S. Concentrate 1			11.6	8	277
	S. Concentrate 2			12.4	10	410.2
	S. Concentrate 3			15.1	14	675.2
	S. Concentrate 4			76.5	27	2547.3
	Tailings			1837.9	27	2692.2
22	Rougher	Shorter baffle + upward washing	Standard	47.8	7	504.6
	S. Concentrate 1			4.5	10	137
	S. Concentrate 2			13.1	13	358.7
	S. Concentrate 3			19.5	17	702.1
	S. Concentrate 4			42.5	22	1294.1
	S. Concentrate 5			38.8	27	1208.4
	Tailings			1833.3	27	2492.5
23	Rougher	Shorter baffle + upward washing	Standard	52.6	7	625.1
	S. Concentrate 1			2.1	8	75
	S. Concentrate 2			11.2	10	320.1
	S. Concentrate 3			21.1	14	667.8
	S. Concentrate 4			32.5	19	973.5
	S. Concentrate 5			35.9	23	889.7
	S. Concentrate 6			27	27	753.6
	Tailings			1801.9	27	2760.6

Test		Configuration	Reagent dosage	Mass dry solids recovered (g)	Time (min)	Volume of water recovered (mL)
24	Rougher	Scraping	Standard	41.9	7	489.0
	S. Concentrate 1			6.8	10	159.8
	S. Concentrate 2			7.9	13	135.8
	S. Concentrate 3			7.6	17	289.4
	S. Concentrate 4			7.5	22	379.7
	S. Concentrate 5			7.7	27	308.1
	Tailings			1893.6	27	2716.3
25	Rougher	Scraping	Standard	42.9	7	446.8
	S. Concentrate 1			6.5	10	160.2
	S. Concentrate 2			6.9	13	137.3
	S. Concentrate 3			7.8	17	278.3
	S. Concentrate 4			7.4	22	381.2
	S. Concentrate 5			7.5	27	310.4
	Tailings			1897.2	27	2718.7
26	Rougher	Shorter baffle + downward washing	Standard	53.8	7	522.5
	S. Concentrate 1			14	10	374.7
	S. Concentrate 2			18.2	13	499.7
	S. Concentrate 3			27.9	17	766.9
	S. Concentrate 4			39.2	22	1047.5
	S. Concentrate 5			44.7	27	1129.1
	Tailings			1782.8	27	2703.6
27	Rougher	Shorter baffle + upward washing	Standard	48.0	7	557.9
	S. Concentrate 1			20.2	10	418.7
	S. Concentrate 2			28.1	13	591.2
	S. Concentrate 3			32.5	17	896.3
	S. Concentrate 4			42.7	22	1143.2
	S. Concentrate 5			41.5	27	1102
	Tailings			1786.8	27	2757.0
28	Rougher	No baffle, no washing	Standard	47.9	7	483.0
	S. Concentrate 1			5.3	10	76.2
	S. Concentrate 2			6.6	13	148.1
	S. Concentrate 3			14.7	17	310.9
	S. Concentrate 4			27.5	22	469.1
	S. Concentrate 5			21.2	27	472.4
	Tailings			1884.1	27	2884.1
29	Rougher	Shorter baffle	Standard	47.4	7	335.8
	S. Concentrate 1			5.6	10	111.8
	S. Concentrate 2			8.4	13	188.5
	S. Concentrate 3			18.4	17	397.7
	S. Concentrate 4			18	22	442.4
	S. Concentrate 5			8.5	27	306.5
	Tailings			1897.0	27	2167.3
30	Rougher	Shorter baffle	Standard	36.5	7	353.7
	S. Concentrate 1			6.5	10	115.9
	S. Concentrate 2			9.2	13	183.4
	S. Concentrate 3			9.3	17	222.1
	S. Concentrate 4			12	22	321.4
	S. Concentrate 5			10.3	27	208.8
	Tailings			1917.4	27	2023.1

Test		Configuration	Reagent dosage	Mass dry solids recovered (g)	Time (min)	Volume of water recovered (mL)
31	Rougher	Shorter baffle	Standard	38.5	7	313.8
	S. Concentrate 1			4.6	10	163.7
	S. Concentrate 2			11.6	13	356.2
	S. Concentrate 3			9.2	17	499.8
	S. Concentrate 4			14	22	397
	S. Concentrate 5			10.4	27	63.5
	Tailings			1915.4	27	2189.3
32	Rougher	Shorter baffle, downward washing	Standard	42.0	7	470.1
	S. Concentrate 1			13.3	10	370.2
	S. Concentrate 2			17.2	13	486.6
	S. Concentrate 3			30.9	17	750.4
	S. Concentrate 4			44.2	22	1026.8
	S. Concentrate 5			41.9	27	1003.3
	Tailings			1805.8	27	2403.5
33	Rougher	Shorter baffle, upward washing	Standard	40.1	7	523.7
	S. Concentrate 1			20.7	10	387.5
	S. Concentrate 2			18.7	13	375.7
	S. Concentrate 3			9.8	17	310.7
	S. Concentrate 4			5.6	22	284.6
	S. Concentrate 5			2.6	27	203.6
	Tailings			1889.5	27	3097.2
34	Rougher	Shorter baffle, downward washing	Standard	39.6	7	416.2
	S. Concentrate 1			11.3	10	269.7
	S. Concentrate 2			7.2	13	227.8
	S. Concentrate 3			7.3	17	228
	S. Concentrate 4			7.4	22	315.1
	S. Concentrate 5			1.7	27	251.9
	Tailings			1911.2	27	3141.8
35	Rougher	Shorter baffle, downward washing	Standard	37.7	7	384.0
	S. Concentrate 1			7.5	10	218.1
	S. Concentrate 2			7.7	13	263.6
	S. Concentrate 3			5.9	17	309.6
	S. Concentrate 4			10	22	476.5
	S. Concentrate 5			11.2	27	532.3
	Tailings			1898.1	27	2603.2
36	Rougher	Shorter baffle, downward washing	Diesel and frother	42.1	10	443.9
	S. Concentrate 1			17.2	13	332.9
	S. Concentrate 2			16.3	17	329.8
	S. Concentrate 3			12.3	22	340.3
	S. Concentrate 4			18.1	27	492.7
	S. Concentrate 5			12.1	27	339.9
	Tailings			1850.7	7	2134.4

Test		Configuration	Reagent dosage	Mass dry solids recovered (g)	Time (min)	Volume of water recovered (mL)
37	Rougher	No baffle, no washing	Standard	39.7	7	404.2
	S. Concentrate 1			3.9	10	65.5
	S. Concentrate 2			9.2	13	149.9
	S. Concentrate 3			12.6	17	218.2
	S. Concentrate 4			10.5	22	220.3
	S. Concentrate 5			6.4	27	172.4
	Tailings			1888.2	27	2190.8
38	Rougher	No baffle, no washing	Standard	43.9	7	441.7
	S. Concentrate 1			2.9	10	58
	S. Concentrate 2			6.5	13	110.6
	S. Concentrate 3			9.2	17	170.7
	S. Concentrate 4			7.7	22	169.3
	S. Concentrate 5			3.9	27	107.8
	Tailings			1892.1	27	2119.5
39	Rougher	Shorter baffle, upward washing	Standard	41.2	7	333.5
	S. Concentrate 1			7.7	10	189.5
	S. Concentrate 2			16.9	13	326
	S. Concentrate 3			9.9	17	274.1
	S. Concentrate 4			8.4	22	243.5
	S. Concentrate 5			2.3	27	125.9
	Tailings			1918.0	27	2729.3
40	Rougher	Shorter baffle, upward washing	Standard	42.4	7	378
	S. Concentrate 1			9.5	10	178.1
	S. Concentrate 2			8.6	13	194.5
	S. Concentrate 3			9.9	17	265.6
	S. Concentrate 4			7.7	22	240
	S. Concentrate 5			1.5	27	95
	Tailings			1923.9	27	2662.8
41	Rougher	Scraping	Reduced frother	50.4	7	281.1
	S. Concentrate 1			7.3	10	58.8
	S. Concentrate 2			4.7	13	107.6
	S. Concentrate 3			4.5	17	109.6
	S. Concentrate 4			3.9	22	91
	S. Concentrate 5			3.8	27	111.4
	Tailings			1923.8	27	2233.5
42	Rougher	Scraping	Diesel and frother	35.3	7	223.2
	S. Concentrate 1			5.8	10	91
	S. Concentrate 2			4.5	13	83
	S. Concentrate 3			5	17	106
	S. Concentrate 4			4.4	22	102.7
	S. Concentrate 5			3.6	27	113.4
	Tailings			1933.8	7	2017.1
43	Rougher	Shorter baffle	Diesel and frother	37.1	10	252.6
	S. Concentrate 1			7.5	13	57.7
	S. Concentrate 2			5.1	17	108.5
	S. Concentrate 3			5.8	22	98.7
	S. Concentrate 4			5	27	89.1
	S. Concentrate 5			4.4	27	86.5
	Tailings			1922.9	7	2179.7

Test		Configuration	Reagent dosage	Mass dry solids recovered (g)	Time (min)	Volume of water recovered (mL)
44	Rougher	Shorter baffle	Reduced frother	32.1	7	243.5
	S. Concentrate 1			6	10	90
	S. Concentrate 2			7.7	13	77.3
	S. Concentrate 3			5.9	17	100
	S. Concentrate 4			4	22	102.1
	S. Concentrate 5			3.7	27	119.4
	Tailings			1920.6	27	1922.3
45	Rougher	Shorter baffle, upward washing	Diesel and frother	34.3	7	276.7
	S. Concentrate 1			4.2	10	332.9
	S. Concentrate 2			3.4	13	329.8
	S. Concentrate 3			3.3	17	340.3
	S. Concentrate 4			3.1	22	492.7
	S. Concentrate 5			2	27	339.9
	Tailings			1921.1	27	2533.4
46	Rougher	Shorter baffle, upward washing	Reduced frother	37.6	7	236.4
	S. Concentrate 1			2.9	10	58.4
	S. Concentrate 2			4	13	97.5
	S. Concentrate 3			3.8	17	98.4
	S. Concentrate 4			2.7	22	98.5
	S. Concentrate 5			2.6	27	91.7
	Tailings			1922.2	27	2435.7
47	Rougher	Longer baffle	Reduced frother	49.6	7	260.5
	S. Concentrate 1			5.1	10	74.5
	S. Concentrate 2			7	13	109.6
	S. Concentrate 3			6.6	17	104.7
	S. Concentrate 4			5	22	89.7
	S. Concentrate 5			4.1	27	72.5
	Tailings			1923.7	27	2689.9
48	Rougher	Longer baffle	Reduced frother	44.6	7	274.0
	S. Concentrate 1			4.3	10	67.4
	S. Concentrate 2			6.9	13	109.2
	S. Concentrate 3			6.3	17	111.4
	S. Concentrate 4			5.2	22	94.7
	S. Concentrate 5			5	27	118.1
	Tailings			1923.9	27	2775.2
49	Rougher	Longer baffle, downward washing	Reduced frother	44.6	7	294.5
	S. Concentrate 1			2	10	71.7
	S. Concentrate 2			6.9	13	113.7
	S. Concentrate 3			8.2	17	113.6
	S. Concentrate 4			5.2	22	104.9
	S. Concentrate 5			5	27	125.3
	Tailings			1924.0	27	3096.7

Test		Configuration	Reagent dosage	Mass dry solids recovered (g)	Time (min)	Volume of water recovered (mL)
50	Rougher	Longer baffle, downward washing	Reduced frother	42.2	7	288.8
	S. Concentrate 1			1.7	10	77.8
	S. Concentrate 2			5.3	13	136.4
	S. Concentrate 3			5	17	118.3
	S. Concentrate 4			3.1	22	119.6
	S. Concentrate 5			2.6	27	124.1
	Tailings			1930.2	27	3219.1
51	Rougher	Shorter baffle	Reduced frother	40.9	7	324.4
	S. Concentrate 1			7.4	10	190.2
	S. Concentrate 2			15.8	13	323.6
	S. Concentrate 3			10.1	17	269.8
	S. Concentrate 4			8.8	22	245.2
	S. Concentrate 5			2.8	27	119.6
	Tailings			1920.1	27	2730.7
52	Rougher	Longer baffle	Diesel and frother	39.2	7	284.4
	S. Concentrate 1			7.8	10	60
	S. Concentrate 2			10.6	13	104.1
	S. Concentrate 3			7.2	17	104.4
	S. Concentrate 4			5.3	22	97.9
	S. Concentrate 5			4.7	27	113.8
	Tailings			1900.9	27	2686.5
53	Rougher	Longer baffle, downward washing	Diesel and frother	38.9	7	292.9
	S. Concentrate 1			3.7	10	72.1
	S. Concentrate 2			5.7	13	123.3
	S. Concentrate 3			3.3	17	121.6
	S. Concentrate 4			2.7	22	114
	S. Concentrate 5			2.4	27	117.6
	Tailings			1921.3	27	3036.1
54	Rougher	Longer baffle, downward washing	Diesel and frother	42.8	7	280.4
	S. Concentrate 1			10.7	10	183.5
	S. Concentrate 2			9.4	13	277.9
	S. Concentrate 3			9.3	17	270.8
	S. Concentrate 4			5.9	22	256
	S. Concentrate 5			5.4	27	282.4
	Tailings			1878.4	27	2802.9
55	Rougher	Longer baffle	Diesel and frother	41.5	7	285.1
	S. Concentrate 1			8.8	10	61
	S. Concentrate 2			10.2	13	107
	S. Concentrate 3			8.3	17	113.1
	S. Concentrate 4			5.8	22	98.4
	S. Concentrate 5			6	27	112.4
	Tailings			1908.5	27	2809.8
56	Rougher	Scraping	No frother	41.6	7	273.2
	S. Concentrate 1			6.3	10	65.7
	S. Concentrate 2			8.6	13	101.6
	S. Concentrate 3			8	17	109.7
	S. Concentrate 4			6.3	22	99.6
	S. Concentrate 5			5.8	27	114.7
	Tailings			1932.9	27	3025.5



Test		Configuration	Reagent dosage	Mass dry solids recovered (g)	Time (min)	Volume of water recovered (mL)
57	Rougher	Longer baffle	No frother	38.5	7	283.0
	S. Concentrate 1			8.1	10	60.5
	S. Concentrate 2			10.5	13	103.1
	S. Concentrate 3			8.7	17	106.2
	S. Concentrate 4			6.7	22	95.6
	S. Concentrate 5			7.2	27	113.7
	Tailings			1928.0	27	3041.5
58	Rougher	Scraping	Paraffin and frother	40.8	7	280.1
	S. Concentrate 1			7.9	10	66.9
	S. Concentrate 2			10.1	13	133.7
	S. Concentrate 3			5.3	17	114.9
	S. Concentrate 4			3.4	22	106.1
	S. Concentrate 5			4	27	118.9
	Tailings			1940.8	27	2980.6
59	Rougher	Longer baffle	Paraffin and frother	40.3	7	279.4
	S. Concentrate 1			7.7	10	64.3
	S. Concentrate 2			12.9	13	102.8
	S. Concentrate 3			9.2	17	105.4
	S. Concentrate 4			6.7	22	99.5
	S. Concentrate 5			6.3	27	115.8
	Tailings			1919.2	27	3001.0
61	Rougher	Longer baffle	Paraffin and frother	49.8	7	266.5
	S. Concentrate 1			13.1	10	59.6
	S. Concentrate 2			11.3	13	102.7
	S. Concentrate 3			6.4	17	106.4
	S. Concentrate 4			4.3	22	96.3
	S. Concentrate 5			4.1	27	114.9
	Tailings			1910.2	27	2873.7
62	Rougher	Longer baffle, downward washing	Paraffin and frother	46.0	7	270.2
	S. Concentrate 1			41.6	10	155.8
	S. Concentrate 2			32.2	13	224.6
	S. Concentrate 3			13.7	17	267.2
	S. Concentrate 4			3.7	22	262.3
	S. Concentrate 5			3.3	27	296.3
	Tailings			1854.0	27	3032.0
63	Rougher	Longer baffle, downward washing	Paraffin and frother	47.7	7	269.9
	S. Concentrate 1			14.8	10	185.5
	S. Concentrate 2			11.4	13	260.5
	S. Concentrate 3			6.8	17	255.8
	S. Concentrate 4			5	22	256.6
	S. Concentrate 5			3.7	27	277.8
	Tailings			1890.9	27	2952.6
64	Rougher	Longer baffle	No frother	39.2	7	284.9
	S. Concentrate 1			8.6	10	59.9
	S. Concentrate 2			11.1	13	110.1
	S. Concentrate 3			9.1	17	102.5
	S. Concentrate 4			7.1	22	98.4
	S. Concentrate 5			7.6	27	103.4
	Tailings			1923.9	27	3030.4

## Appendix D4: Conductivity readings for the froth washing tests

**Table D-3: Conductivity readings ( $\sigma$ ) for the froth washing tests.**

σ (mS)																		
Test	22	23	26	27	32	33	34	35	36	39	45	46	49	50	53	54	62	63
S. C1	23.9	18.3	16.79	15.99	27.7	20.8	27.5	28.8	18.7	25.8	20.0	20.6	32.4	41.3	28.9	31.7	32.1	32.4
S. C2	25.3	26.0	17.74	18.92	26.2	21.3	28.8	28.7	17.9	22.1	18.0	19.3	40.1	28.5	30.5	39.5	39.9	40.1
S. C3	31.2	27.4	20.75	22.55	27.6	29.2	31.6	33.9	25.8	27.9	24.3	25.8	34.2	21.2	36.6	33.6	33.9	34.2
S. C4	31.1	27.6	23.73	24.76	28.0	34.1	33.6	35	28.4	27.9	26.6	27.3	40.2	30.7	36.5	39.9	40.1	40.2
S. C5	34.3	26.3	26.0	25.85	28.8	33.8	37.9	37.7	31.1	27.6	30.6	32.4	42.5	30.5	34.1	41.3	41.8	42.5
Tails	21.3	16.5	23.46	18.9	20.3	30.75	31.6	29.36	30.7	28.9	27.3	27.33	10.1	10.4	5.59	5.36	6.2	6.43

## Appendix D5: Data for efficiency calculation

**Table D-4: Information used to calculate efficiency for Test 22.**

Test 22	Conc. of salt water (g/L)	Salt rec. in conc (g)	Cumulative salt (g)	Time (min)	Time for collection of concentrate (min)	Salt that went into cell (g)	Salt that remained in the cell (g)	Salt recovered in conc (g/min)	S(t) at average times (g/L)	C(t) (g/L)	Average times between concentrates (min)	C(t) at average times (g/L)
S. C1	13.75	1.88	1.88	1	1	5.6	3.7	1.88	13.48	1.21	0.5	0.45
S. C2	14.68	5.26	7.157	2	1	11.1	5.8	2.63	14.51	3.12	2	1.56
S. C3	18.62	13.07	20.22	7	4	22.2	9.1	3.27	16.33	6.12	5	3.72
S. C4	18.56	24.01	44.24	13	6	33.3	9.3	4.004	18.61	9.17	10	7.08
S. C5	20.69	25.00	69.24	20	7	38.9	13.9	3.57	20.19	13.72	16.5	11.03

**Table D-5: Information used to calculate efficiency for Test 23.**

Test 23	Conc. of salt water (g/L)	Salt recovered in conc (g)	Cumulative salt (g)	Time (min)	Time for collection of concentrate (min)	Salt that went into cell (g)	Salt that remained in the cell (g)	Salt recovered in conc (g/min)	S(t) at average times (g/L)	C(t) (g/L)	Average times between concentrates (min)	C(t) at average times (g/L)
S. C1	10.01	0.75	0.75	1	1	5.5	4.8	0.75	9.57	1.57	0.5	1.54
S. C2	15.15	4.85	5.60	3	2	11.0	6.2	2.42	11.60	3.60	1.5	2.35
S. C3	16.08	10.74	16.34	7	4	22.1	11.3	2.68	15.92	7.33	5	5.22
S. C4	16.22	15.79	32.13	12	5	27.6	11.8	3.15	16.89	11.21	9.5	8.91
S. C5	15.35	13.65	45.79	16	4	22.1	8.4	3.41	15.55	13.97	14	12.59
S. C6	15.35	11.56	57.36	20	4	22.1	10.5	2.89	14.89	17.43	18	15.87

**Table D-6: Information used to calculate efficiency for Test 26.**

Test 26	Conc. of salt water (g/L)	Salt recovered in conc (g)	Cumulative salt (g)	Time (min)	Time for collection of concentrate (min)	Salt that went into cell (g)	Salt that remained in the cell (g)	Salt recovered in conc (g/min)	S(t) at average times (g/L)	C(t) (g/L)	Average times between concentrates (min)	C(t) at average times (g/L)
S. C1	9.0	3.37	3.37	3	3	13.2	9.9	1.24	8.28	3.24	1.5	3.19
S. C2	9.63	4.82	8.19	6	3	13.2	8.4	1.61	9.42	6.00	4.5	4.92
S. C3	11.64	8.93	17.12	10	4	17.6	8.7	2.23	10.75	8.87	8	6.91
S. C4	13.64	14.29	31.41	15	5	22.1	7.8	2.85	12.45	11.42	12.5	9.49
S. C5	15.15	17.11	48.51	20	5	22.1	5.0	3.42	14.35	13.05	17.5	12.35

**Table D-7: Information used to calculate efficiency for Test 27.**

Test 27	Conc. of salt water (g/L)	Salt rec. in conc (g)	Cumulative salt (g)	Time (min)	Time for collection of concentrate (min)	Salt that went into cell (g)	Salt that remained in the cell (g)	Salt recovered in conc (g/min)	S(t) at average times (g/L)	C(t) (g/L)	Average times between concentrates (min)	C(t) at average times (g/L)
S. C1	8.47	3.54	3.54	3	3	13.6	10.1	1.18	7.24	3.30	1.5	3.18
S. C2	10.42	6.16	9.71	6	3	13.6	7.4	2.05	9.54	5.74	4.5	4.66
S. C3	12.85	11.52	21.22	10	4	18.1	6.6	2.88	11.71	7.92	8	6.40
S. C4	14.33	16.37	37.60	15	5	22.7	6.3	3.28	13.71	9.98	12.5	8.62
S. C5	15.05	16.58	54.19	20	5	22.7	6.1	3.31	14.86	11.98	17.5	11.10

**Table D-8: Information used to calculate efficiency for Test 32.**

Test 32	Conc. of salt of water (g/L)	Salt rec. in conc (g)	Cumulative salt (g)	Time (min)	Time for collection of concentrate (min)	Salt that went into cell (g)	Salt that remained in the cell (g)	Salt recovered in conc (g/min)	S(t) at average times (g/L)	C(t) (g/L)	Average times between concentrates (min)	C(t) at average times (g/L)
S. C1	16.29	6.03	6.03	3	3	13.6	7.6	2.00	16.07	3.42	1.5	3.13
S. C2	15.28	7.43	13.47	6	3	13.6	6.2	2.48	15.92	6.19	4.5	4.96
S. C3	16.22	12.17	25.64	10	4	18.2	6.0	3.04	15.89	8.90	8	7.10
S. C4	16.48	16.93	42.57	15	5	22.7	5.8	3.38	16.12	11.51	12.5	9.85
S. C5	17.05	17.11	59.68	20	5	22.7	5.6	3.42	16.71	14.03	17.5	12.99

**Table D-9: Information used to calculate efficiency for Test 33.**

Test 33	Conc. of salt water (g/L)	Salt recovered in conc (g)	Cumulative salt (g)	Time (min)	Time for collection of concentrate (min)	Salt that went into cell (g)	Salt that remained in the cell (g)	Salt recovered in conc (g/min)	S(t) at average times (g/L)	C(t) (g/L)	Average times between concentrates (min)	C(t) at average times (g/L)
S. C1	11.67	4.52	4.52	3	3	12.7	8.2	1.51	8.92	3.67	1.5	1.23
S. C2	12.01	4.51	9.04	6	3	12.7	8.2	1.50	12.28	7.35	4.5	5.36
S. C3	17.28	5.37	14.41	10	4	17.0	11.6	1.34	15.51	12.55	8	10.17
S. C4	20.55	5.85	20.26	15	5	21.2	15.4	1.17	18.53	19.44	12.5	16.36
S. C5	20.35	4.14	24.41	20	5	21.2	17.1	0.83	20.42	27.09	17.5	23.24



**Table D-10: Information used to calculate efficiency for Test 34.**

Test 34	Conc. of salt water (g/L)	Salt recovered in conc (g)	Cumulative salt (g)	Time (min)	Time for collection of concentrate (min)	Salt that went into cell (g)	Salt that remained in the cell (g)	Salt recovered in conc (g/min)	S(t) at average times (g/L)	C(t) (g/L)	Average times between concentrates (min)	C(t) at average times (g/L)
S. C1	16.15	4.36	4.36	3	3	12.3	8.0	1.45	15.36	3.57	1.5	1.88
S. C2	17.02	3.88	8.23	6	3	12.3	8.5	1.29	16.56	7.36	4.5	6.41
S. C3	18.89	4.31	12.54	10	4	16.4	12.1	1.08	17.95	12.81	8	11.68
S. C4	20.23	6.37	18.91	15	5	20.6	14.2	1.27	19.75	19.17	12.5	18.46
S. C5	23.09	5.82	24.73	20	5	20.6	14.7	1.16	21.75	25.77	17.5	25.99

**Table D-11: Information used to calculate efficiency for Test 35.**

Test 35	Conc. of salt water (g/L)	Salt recovered in conc (g)	Cumulative salt (g)	Time (min)	Time for collection of concentrate (min)	Salt that went into cell (g)	Salt that remained in the cell (g)	Salt recovered in conc (g/min)	S(t) at average times (g/L)	C(t) (g/L)	Average times between concentrates (min)	C(t) at average times (g/L)
S. C1	17.02	4.59	4.59	3	3	12.8	8.2	1.53	16.23	3.67	1.5	1.62
S. C2	16.95	3.86	8.45	6	3	12.8	8.9	1.29	17.35	7.66	4.5	5.68
S. C3	20.42	4.65	13.11	10	4	17.0	12.4	1.16	18.66	13.22	8	10.42
S. C4	21.15	6.66	19.77	15	5	21.3	14.6	1.33	20.33	19.78	12.5	16.52
S. C5	22.96	5.78	25.56	20	5	21.3	15.5	1.16	22.20	26.74	17.5	23.29

**Table D-12: Information used to calculate efficiency for Test 36.**

Test 36	Conc. of salt water (g/L)	Salt recovered in conc (g)	Cumulative salt (g)	Time (min)	Time for collection of concentrate (min)	Salt that went into cell (g)	Salt that remained in the cell (g)	Salt recovered in conc (g/min)	S(t) at average times (g/L)	C(t) (g/L)	Average times between concentrates (min)	C(t) at average times (g/L)
S. C1	10.28	3.42	3.42	3	3	10.2	6.7	1.14	8.28	3.02	1.5	1.93
S. C2	9.74	3.21	6.63	6	3	10.2	6.9	1.07	10.49	6.13	4.5	4.63
S. C3	15.01	5.11	11.74	10	4	13.5	8.4	1.27	12.82	9.91	8	7.77
S. C4	16.75	8.25	20.00	15	5	16.9	8.7	1.65	15.42	13.79	12.5	11.81
S. C5	18.55	6.31	26.30	20	5	16.9	10.6	1.26	17.76	18.56	17.5	16.30

**Table D-13: Information used to calculate efficiency for Test 39.**

Test 39	Conc. of salt water (g/L)	Salt rec. in conc (g)	Cumulative salt (g)	Time (min)	Time for collection of concentrate (min)	Salt that went into cell (g)	Salt that remained in the cell (g)	Salt recovered in conc (g/min)	S(t) at average times (g/L)	C(t) (g/L)	Average times between concentrations (min)	C(t) at average times (g/L)
S. C1	15.02	4.99	4.99	3	3	10.1	5.1	1.67	13.56	2.30	1.5	0.94
S. C2	12.55	4.14	9.14	6	3	10.1	6.0	1.38	14.32	4.99	4.5	3.60
S. C3	16.42	5.59	14.73	10	4	13.5	7.9	1.39	15.06	8.55	8	6.71
S. C4	16.42	8.09	22.82	15	5	16.9	8.8	1.62	15.80	12.49	12.5	10.69
S. C5	16.22	5.51	28.33	20	5	16.9	11.4	1.10	16.33	17.60	17.5	15.13

**Table D-14: Information used to calculate efficiency for Test 45.**

Test 45	Conc. of salt water (g/L)	Salt rec. in conc (g)	Cumulative salt (g)	Time (min)	Time for collection of concentrate (min)	Salt that went into cell (g)	Salt that remained in the cell (g)	Salt recovered in conc (g/min)	S(t) at average times (g/L)	C(t) (g/L)	Average times between concentrates (min)	C(t) at average times (g/L)
S. C1	11.14	3.71	3.71	3	3	9.0	5.3	1.23	9.38	2.37	1.5	1.42
S. C2	9.81	3.23	6.95	6	3	8.9	5.6	1.07	10.79	4.84	4.5	3.63
S. C3	14.01	4.77	11.72	10	4	11.8	7.1	1.19	12.43	8.02	8	6.22
S. C4	15.55	7.66	19.38	15	5	14.8	7.1	1.53	14.54	11.21	12.5	9.55
S. C5	18.22	6.19	25.57	20	5	14.8	8.6	1.24	16.89	15.07	17.5	13.24

**Table D-15: Information used to calculate efficiency for Test 46.**

Test 46	Conc. of salt of water (g/L)	Salt recovered in conc (g)	Cumulative salt (g)	Time (min)	Time for collection of concentrate (min)	Salt that went into cell (g)	Salt that remained in the cell (g)	Salt recovered in conc (g/min)	S(t) at average times (g/L)	C(t) (g/L)	Average times between concentrates (min)	C(t) at average times (g/L)
S. C1	11.54	3.84	3.84	3	3	8.88	5.0	1.28	9.38	2.57	1.5	1.42
S. C2	10.67	3.52	7.37	6	3	8.9	5.4	1.17	10.79	4.66	4.5	3.64
S. C3	15.01	5.11	12.47	10	4	11.8	6.7	1.27	12.43	7.67	8	6.22
S. C4	16.02	7.89	20.37	15	5	14.8	6.9	1.58	14.54	10.77	12.5	9.55
S. C5	19.42	6.60	26.97	20	5	14.8	8.2	1.32	16.89	14.44	17.5	13.24

**Table D-16: Information used to calculate efficiency for Test 49.**

Test 49	Conc. of salt of water (g/L)	Salt rec. in conc (g)	Cumulative salt (g)	Time (min)	Time for collection of concentrate (min)	Salt that went into cell (g)	Salt that remained in the cell (g)	Salt recovered in conc (g/min)	S(t) at average times (g/L)	C(t) (g/L)	Average times between concentrates (min)	C(t) at average times (g/L)
S. C1	41.2	2.95	2.95	3	3	10.1	7.1	0.98	54.37	2.24	1.5	0.86
S. C2	27.9	3.17	6.13	6	3	10.1	6.9	1.05	32.57	4.42	4.5	3.35
S. C3	22.4	2.54	8.67	10	4	13.4	10.9	0.63	23.41	7.86	8	6.25
S. C4	31.2	3.27	11.94	15	5	16.8	13.5	0.65	26.57	12.14	12.5	9.99
S. C5	31.1	3.89	15.84	20	5	16.8	12.9	0.77	32.95	16.21	17.5	14.14

**Table D-17: Information used to calculate efficiency for Test 50.**

Test 50	Conc. of salt water (g/L)	Salt recovered in conc (g)	Cumulative salt (g)	Time (min)	Time for collection of concentrate (min)	Salt that went into cell (g)	Salt that remained in the cell (g)	Salt recovered in conc (g/min)	S(t) at average times (g/L)	C(t) (g/L)	Average times between concentrates (min)	C(t) at average times (g/L)
S. C1	25.37	1.97	1.97	3	3	10.0	8.1	0.65	34.41	2.55	1.5	1.48
S. C2	16.82	2.29	4.26	6	3	10.0	7.7	0.76	19.64	5.00	4.5	5.36
S. C3	11.95	1.41	5.68	10	4	13.4	12.0	0.35	13.21	8.79	8	9.90
S. C4	18.29	2.18	7.86	15	5	16.7	14.6	0.43	14.97	13.40	12.5	15.73
S. C5	18.15	2.25	10.12	20	5	16.7	14.5	0.45	19.09	17.98	17.5	22.21



**Table D-18: Information used to calculate efficiency for Test 53.**

Test 53	Conc. of salt of water (g/L)	Salt recovered in conc (g)	Cumulative salt (g)	Time (min)	Time for collection of concentrate (min)	Salt that went into cell (g)	Salt that remained in the cell (g)	Salt recovered in conc (g/min)	S(t) at average times (g/L)	C(t) (g/L)	Average times between concentrates (min)	C(t) at average times (g/L)
S. C1	17.08	1.23	1.23	3	3	9.8	8.6	0.41	14.93	2.72	1.5	1.31
S. C2	18.15	2.23	3.47	6	3	9.8	7.6	0.75	18.02	5.13	4.5	3.88
S. C3	22.22	2.70	6.17	10	4	13.1	10.4	0.68	20.56	8.45	8	6.87
S. C4	22.16	2.52	8.70	15	5	16.4	13.9	0.51	22.15	12.82	12.5	10.72
S. C5	20.56	2.42	11.12	20	5	16.4	14.0	0.48	21.70	17.24	17.5	14.99

**Table D-19: Information used to calculate efficiency for Test 54.**

Test 54	Conc. of salt water (g/L)	Salt recovered in conc (g)	Cumulative salt (g)	Time (min)	Time for collection of concentrate (min)	Salt that went into cell (g)	Salt that remained in the cell (g)	Salt recovered in conc (g/min)	S(t) at average times (g/L)	C(t) (g/L)	Average times between concentrates (min)	C(t) at average times (g/L)
S. C1	18.96	1.37	1.37	3	3	10.2	8.8	0.45	18.05	2.79	1.5	1.29
S. C2	24.16	2.97	4.35	6	3	10.2	7.2	0.99	21.01	5.08	4.5	3.91
S. C3	20.23	2.46	6.81	10	4	13.6	11.1	0.61	22.47	8.60	8	6.97
S. C4	24.43	2.79	9.59	15	5	17.0	14.2	0.56	22.86	13.09	12.5	10.91
S. C5	25.37	2.98	12.57	20	5	17.0	14.0	0.59	23.95	17.53	17.5	15.27

**Table D-20: Information used to calculate efficiency for Test 62.**

Test 62	Conc. of salt water (g/L)	Salt recovered in conc (g)	Cumulative salt (g)	Time (min)	Time for collection of concentrate (min)	Salt that went into cell (g)	Salt that remained in the cell (g)	Salt recovered in conc (g/min)	S(t) at average times (g/L)	C(t) (g/L)	Average times between concentrates (min)	C(t) at average times (g/L)
S. C1	19.22	2.99	2.99	3	3	10.2	7.2	0.99	18.28	2.27	1.5	1.16
S. C2	24.43	5.49	8.48	6	3	10.2	4.7	1.83	21.28	3.76	4.5	3.02
S. C3	20.43	5.46	13.94	10	4	13.6	8.1	1.36	22.70	6.33	8	5.19
S. C4	24.56	6.44	20.38	15	5	17.0	10.5	1.29	22.99	9.66	12.5	7.98
S. C5	25.69	7.61	27.99	20	5	17.0	9.4	1.52	24.08	12.62	17.5	11.08

**Table D-21: Information used to calculate efficiency for Test 63.**

Test 63	Conc. of salt water (g/L)	Salt recovered in conc (g)	Cumulative salt (g)	Time (min)	Time for collection of concentrate (min)	Salt that went into cell (g)	Salt that remained in the cell (g)	Salt recovered in conc (g/min)	S(t) at average times (g/L)	C(t) (g/L)	Average times between concentrates (min)	C(t) at average times (g/L)
S. C1	19.42	3.60	3.60	3	3	10.2	6.6	1.20	18.39	2.09	1.5	0.81
S. C2	24.56	6.40	10.00	6	3	10.2	3.8	2.13	21.48	3.29	4.5	2.68
S. C3	20.63	5.28	15.28	10	4	13.6	8.3	1.32	22.86	5.92	8	4.85
S. C4	24.63	6.32	21.60	15	5	17.0	10.7	1.26	23.01	9.30	12.5	7.65
S. C5	26.17	7.27	28.87	20	5	17.0	9.7	1.45	24.09	12.38	17.5	10.76

## Appendix D6: Amount of wash water added to the flotation cell

***Table D-22: Amount of salt solution wash water added to the tests.***

	Salt solution wash water added (mL)
Test 22	2700
Test 23	2700
Test 26	2250
Test 27	2300
Test 32	2300
Test 33	2150
Test 34	2080
Test 35	2150
Test 36	1700
Test 39	1700
Test 45	1500
Test 46	1500
Test 49	1700
Test 50	1700
Test 53	1700
Test 54	1700
Test 62	1700
Test 63	1700

### Appendix D7: PGM + Au and Cr<sub>2</sub>O<sub>3</sub> raw data

	Test 24		Test 25		Test 28		Test 30	
	PGM + Au (g/t)	Cr <sub>2</sub> O <sub>3</sub> (%)	PGM + Au (g/t)	Cr <sub>2</sub> O <sub>3</sub> (%)	PGM + Au (g/t)	Cr <sub>2</sub> O <sub>3</sub> (%)	PGM + Au (g/t)	Cr <sub>2</sub> O <sub>3</sub> (%)
Rougher	27	4.19	26.8	4.21	22	3.89	37.6	2.34
S.C 1	7.2	7.9	7.4	8.0	3.57	8.05	6.8	7.56
S.C 2	4.72	9.28	4.80	9.26	2.13	11.1	5.1	9.18
Tails	0.29	24.9	0.29	25.0	0.25	24.5	0.25	24.2

	Test 31		Test 33		Test 34		Test 35	
	PGM + Au (g/t)	Cr <sub>2</sub> O <sub>3</sub> (%)	PGM + Au (g/t)	Cr <sub>2</sub> O <sub>3</sub> (%)	PGM + Au (g/t)	Cr <sub>2</sub> O <sub>3</sub> (%)	PGM + Au (g/t)	Cr <sub>2</sub> O <sub>3</sub> (%)
Rougher	35.8	2.545	29.45	2.925	30.8	2.87	37.75	2.58
S.C 1	6.55	10.6	11.9	11.9	5.8	13.2	5.92	12.7
S.C 2	4.75	12.3	4.69	15.7	3.58	15.1	3.08	14.9
Tails	0.28	24.735	0.35	24.8	0.33	24.8	0.38	24.6

	Test 39		Test 40		Test 41		Test 42	
	PGM + Au (g/t)	Cr <sub>2</sub> O <sub>3</sub> (%)	PGM + Au (g/t)	Cr <sub>2</sub> O <sub>3</sub> (%)	PGM + Au (g/t)	Cr <sub>2</sub> O <sub>3</sub> (%)	PGM + Au (g/t)	Cr <sub>2</sub> O <sub>3</sub> (%)
Rougher	24.3	3.01	36.95	2.62	24.63	2.6	36.7	2.65
S.C 1	6	14.4	13.3	6.05	15.1	4.26	15.3	5.69
S.C 2	3.05	15.9	9.36	6.655	11.2	7.28	11.3	8.17
Tails	0.34	24.75	0.3	24.735	0.28	24.735	0.28	24.735

	Test 43		Test 44		Test 45		Test 46	
	PGM + Au (g/t)	Cr <sub>2</sub> O <sub>3</sub> (%)	PGM + Au (g/t)	Cr <sub>2</sub> O <sub>3</sub> (%)	PGM + Au (g/t)	Cr <sub>2</sub> O <sub>3</sub> (%)	PGM + Au (g/t)	Cr <sub>2</sub> O <sub>3</sub> (%)
Rougher	35.4	2.66	36.95	2.62	37.6	2.98	34.2	3.055
S.C 1	15	5.44	13.3	6.05	14	6.39	15.6	7.49
S.C 2	9.42	7.21	9.36	6.655	13	9.65	11.3	10.1
Tails	0.28	24.735	0.3	24.735	0.38	24.735	0.34	24.735

S.C – Scavenger concentrates (combined for PGM + Au and Cr<sub>2</sub>O<sub>3</sub> analysis)

	Test 47		Test 48		Test 49		Test 50	
	PGM + Au (g/t)	Cr <sub>2</sub> O <sub>3</sub> (%)	PGM + Au (g/t)	Cr <sub>2</sub> O <sub>3</sub> (%)	PGM + Au (g/t)	Cr <sub>2</sub> O <sub>3</sub> (%)	PGM + Au (g/t)	Cr <sub>2</sub> O <sub>3</sub> (%)
Rougher	28.65	3.79	29.2	3.33	26.3	3.46	27.6	3.56
S.C 1	9.9	8.98	8.44	6.1	10.3	5.45	8.25	11.9
S.C 2	6.98	8.53	5.77	7.51	7.5	7.63	5	13.1
Tails	0.38	24.9	0.35	25.2	0.4	24.8	0.43	25.0

	Test 51		Test 52		Test 53		Test 54	
	PGM + Au (g/t)	Cr <sub>2</sub> O <sub>3</sub> (%)	PGM + Au (g/t)	Cr <sub>2</sub> O <sub>3</sub> (%)	PGM + Au (g/t)	Cr <sub>2</sub> O <sub>3</sub> (%)	PGM + Au (g/t)	Cr <sub>2</sub> O <sub>3</sub> (%)
Rougher	30.1	3.44	30.8	3.36	29.3	3.25	24.9	4.36
S.C 1	15	5.44	7.9	7.36	6.57	6.68	6.09	7.7
S.C 2	7.8	7.71	6.375	8.915	5.17	8.63	5	11.1
Tails	0.48	24.735	0.39	25.0	0.4	24.6	0.38	25.0

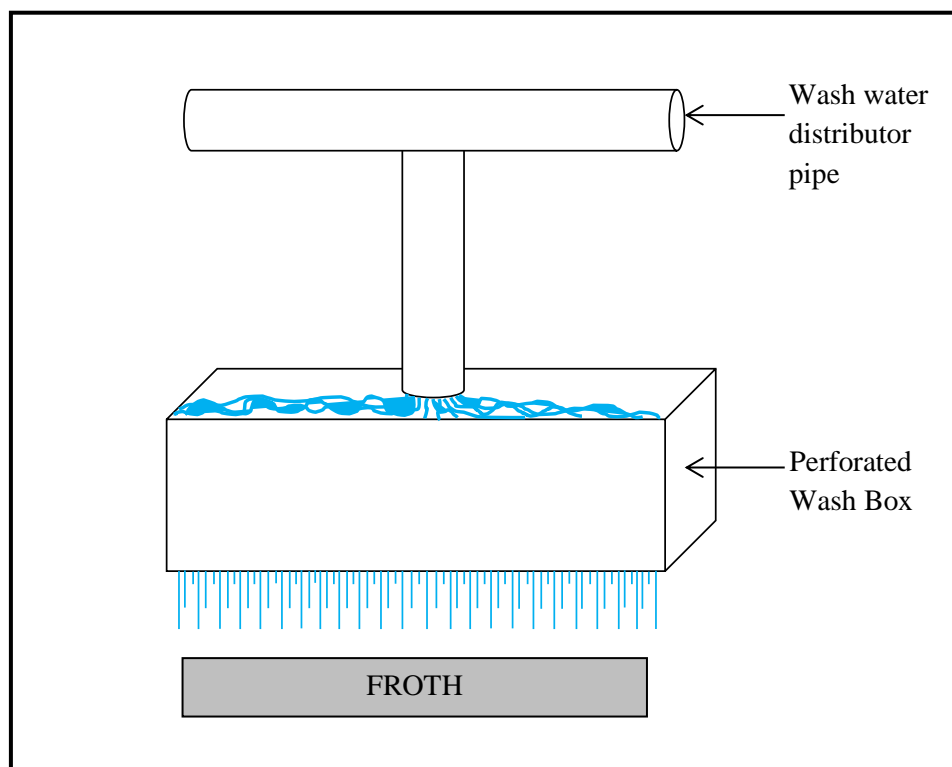
	Test 55		Test 56		Test 57		Test 58	
	PGM + Au (g/t)	Cr <sub>2</sub> O <sub>3</sub> (%)	PGM + Au (g/t)	Cr <sub>2</sub> O <sub>3</sub> (%)	PGM + Au (g/t)	Cr <sub>2</sub> O <sub>3</sub> (%)	PGM + Au (g/t)	Cr <sub>2</sub> O <sub>3</sub> (%)
Rougher	29.9	4.03	27	3.74	30.2	3.59	29.4	4.61
S.C 1	5.2	5.75	4.64	5.11	5.92	5.83	5	6.95
S.C 2	5.29	8.07	3.79	6.43	2.73	8.68	5.29	8.62
Tails	0.45	24.735	0.38	23.8	0.43	25.3	0.28	24.735

	Test 59		Test 61		Test 62		Test 63	
	PGM + Au (g/t)	Cr <sub>2</sub> O <sub>3</sub> (%)	PGM + Au (g/t)	Cr <sub>2</sub> O <sub>3</sub> (%)	PGM + Au (g/t)	Cr <sub>2</sub> O <sub>3</sub> (%)	PGM + Au (g/t)	Cr <sub>2</sub> O <sub>3</sub> (%)
Rougher	31.3	4.5	21.9	4.4	22	3.96	21.8	4.24
S.C 1	4.4	4.59	7.2	5.24	7	16.5	7.6	6.23
S.C 2	3.14	8.14	5.14	9.58	6.36	12.6	6.43	9.51
Tails	0.48	24.735	0.5	24.735	0.45	24.735	0.43	25.0

## APPENDIX E: WASH WATER DISTRIBUTOR DEVICES

### Appendix E1: Wash water distributor configuration: Wash box

McKeon (2001) investigated the effect of wash water addition through means of a rectangular wash box as can be seen in Figure E-1.



***Figure E-1: Wash box configuration that was tested by McKeon (2001) to add wash water into the froth of the flotation cell.***

**Operation:** The wash water is fed through a distribution piping system to the wash box. The wash water fills into the box and exits through the perforated holes below and the water, in the form of small streams, washes the froth. The arrangement of the perforated holes can be seen in Figure E-2.

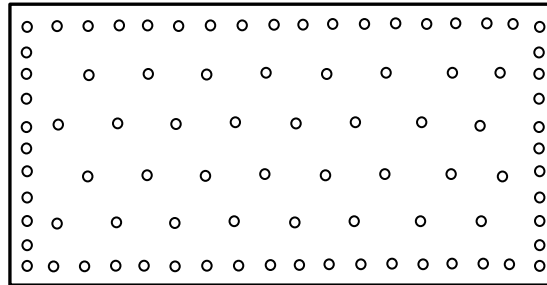
**Advantages:**

- Simple to construct and material of construction (perspex) is easily available.
- The holes on the bottom of the box may be easily drilled through without compromising the strength of the box.



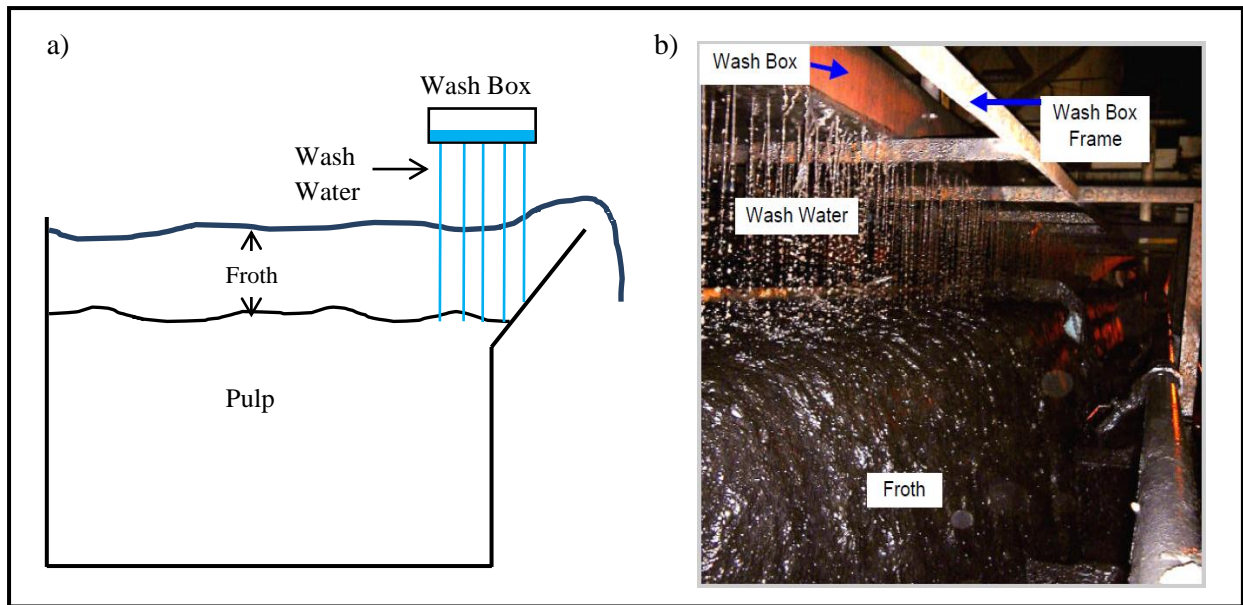
**Disadvantages:**

- Many wash boxes in a row may have to be used due to the large sizes of the industrial flotation cells.
- Only washing on the surface can take place and not inside the froth.
- The wash boxes will not cover the entire froth surface, hence the entire froth will not benefit from the effect of froth washing.



**Figure E-2: Wash box hole pattern (adapted from McKeon, 2001).**

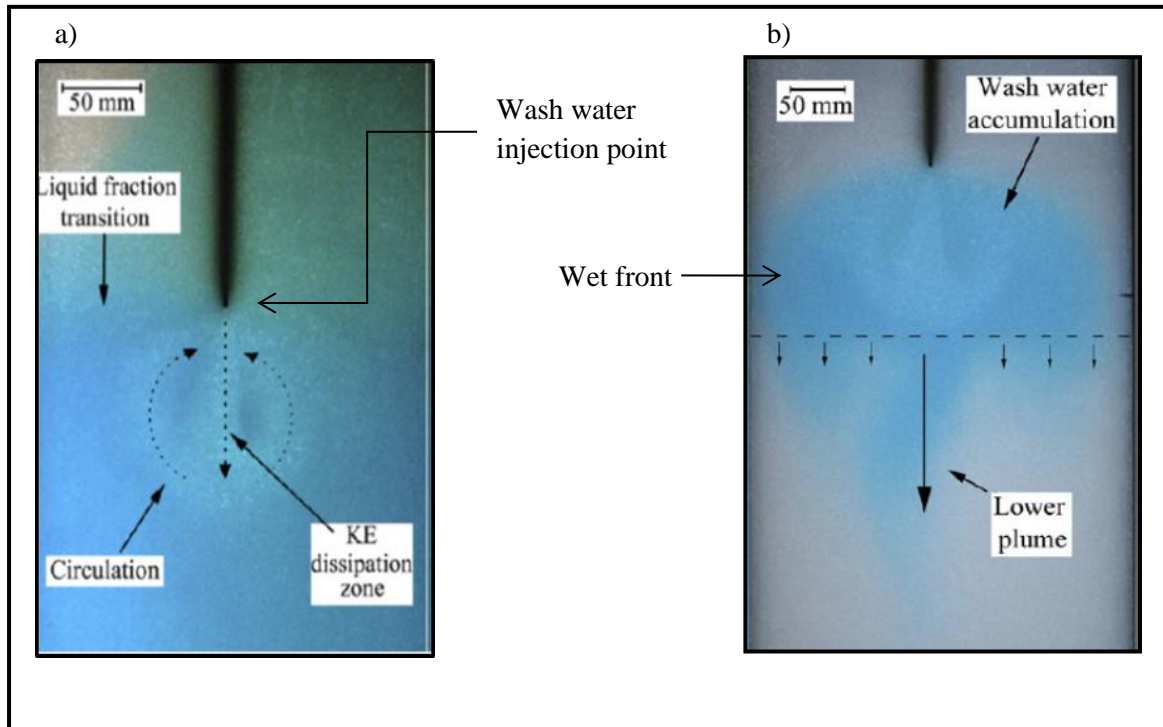
McKeon (2001) found that the hole pattern, as evident by Figure E-2, resulted in a uniform distribution of wash water. Various positions of the wash box were also tested. It was found that the optimum wash box position was near the lip of the cell and an array of wash boxes were used to cover the length of the cell lip. The wash water exited the wash box as a steady shower of water to the froth. This position allowed the froth to be washed just before it flowed into the concentrate launders (see Figure E-3). If a larger flow rate of wash water was required the number of holes was increased with a much smaller hole spacing to double the wash water added to the froth.



***Figure E-3: Wash box placement over the flotation cell (a) a schematic and (b) a photograph adapted from McKeon (2001).***

## Appendix E2: Wash water distributor configuration: Nozzle type

Any wash water distributor of the nozzle type is characterised by having a smaller opening for the exit of the wash water thereby causing the water to exit at a higher velocity. Ireland et al (2006) examined wash water configurations that include vertical and horizontal water jet injections to the flotation froth. An example of the vertical water jet injector can be seen in the following figure:



***Figure E-4: (a) Photograph of vertical injection of wash water into the froth of a column flotation cell with blue dye mixed in the wash water stream and (b) A zoom out version of (a) showing that as the wash water is injected, a portion of the wash water moves down quickly (lower plume) but the majority of the wash water moving down slowly and creating the effect of a wet front (Ireland et al, 2006)***

**Operation:** The wash water is fed into the nozzle and exits at a higher velocity at the open end in the form of liquid jets plunging into the froth.

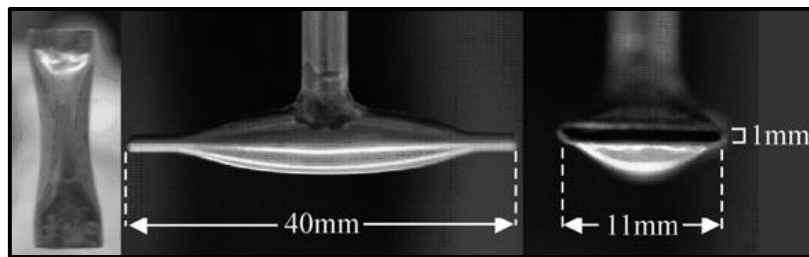
### **Advantages:**

- Simplest nozzle design.

**Disadvantages:**

- A relatively small area of froth is capable of being influenced by this type of injector hence many injectors will have to be used (as with all types of nozzle distributors).
- Axial mixing.

According to Ireland et al (2006) a superior dispersal performance was detected, the vertical injection tubes, when the tip of the injection tube was submerged in the froth, as compared to when the tip was on the surface of the froth. It was also found that a high flow rate of wash water was found to lead to a substantial reduction in the amount of gangue into the froth above the level of injection, leading to a product of a higher grade. In a washed froth, the bubbles are separated from each other and the drainage of entrained gangue is stimulated by the downward flow rate of wash water. The second type of nozzle as investigated by Ireland et al (2006) was the horizontal nozzle which can be seen in the following figure:



**Figure E-5: Photograph of the 'T' injector used to create a horizontal jet of wash water into the froth (Ireland et al, 2006).**

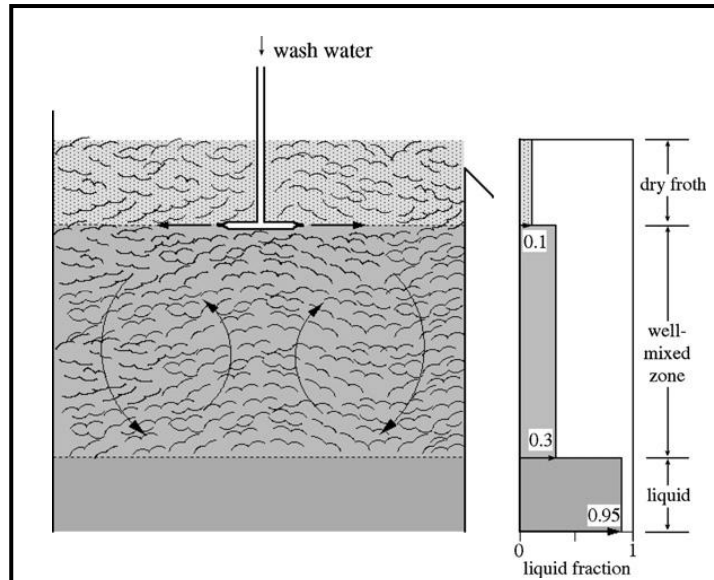
**Operation:** The wash water exits the 'T' injector through the nozzle. The horizontal nozzle causes the wash water to exit initially horizontally into the froth, thereafter downwards forming vortex pairs.

**Advantages:**

- Good ability to spread wash water in the horizontal direction (as compared to the vertical injection).

**Disadvantages:**

- Expensive to construct and intricate installation.
- Axial mixing.



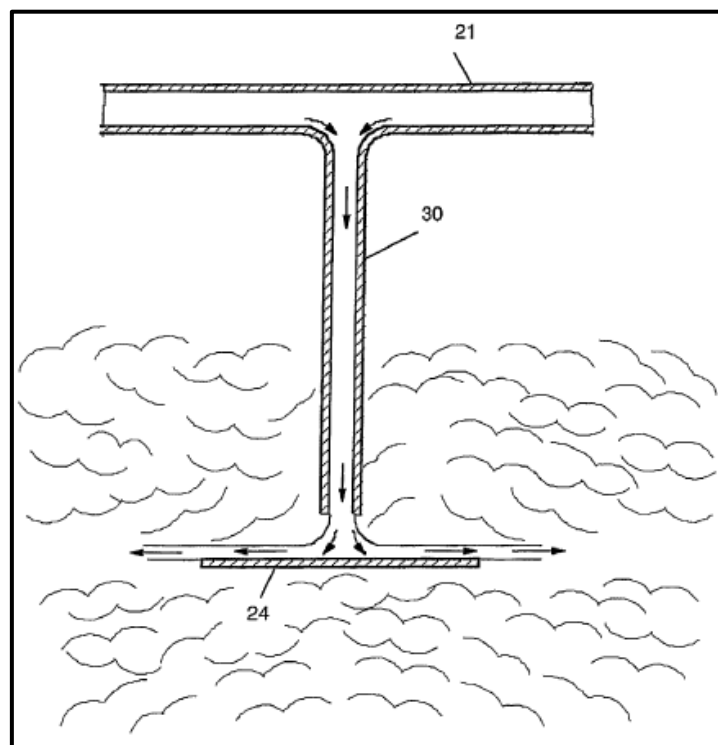
**Figure E-6: Schematic of the direction of movement of the water when injected into the froth. The scale on the right shows the liquid fraction that is experienced in the respective ‘zones’ of the froth (taken from Ireland et al, 2006).**

In both the systems (vertical and horizontal injection), it was found that substantial back mixing occurred which led to the conclusion that it is not possible to entirely remove the gangue particles from the froth phase. The back mixing has the effect of washing the gangue out (as it should) but then bringing it back up again.

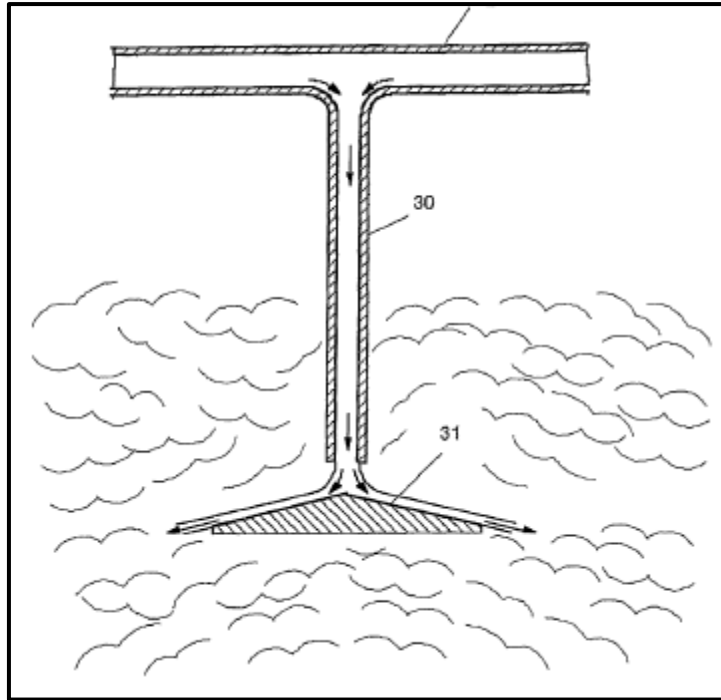
The wash water distributors by Jameson (2010) are all nozzle based and each design is aimed at a certain issue. The following configurations (Figures E-7 to E-14 by Jameson, 2010) were initially designed for column flotation cells, but can be used for a mechanically stirred flotation cell with some adjustment made to accommodate the impeller. A manifold system is mounted above the column that provides water to each of the wash water distribution ‘heads’ or nozzles (as can be seen in the following figure). Since the following wash water distributors are all nozzles, their advantages and disadvantages may be combined and presented at the end.



**Operation:** Wash water is delivered through a manifold pipe (21). The water then flows down through a nozzle (22) in the form of a jet of liquid. This stream strikes a circular plate (24). This plate could be immersed in the froth layer depending on required position. As the water strikes the plate, it changes direction from vertical to horizontal and travels radially outwards, leaving the apparatus at (26). The result of this configuration is that it delivers to the froth a horizontal sheet of wash water. The height H can be varied to control the velocity of the stream leaving the apparatus whilst keeping wash water flow rate constant. Following nozzle configurations have similar methods of operation and hence will not be repeated.

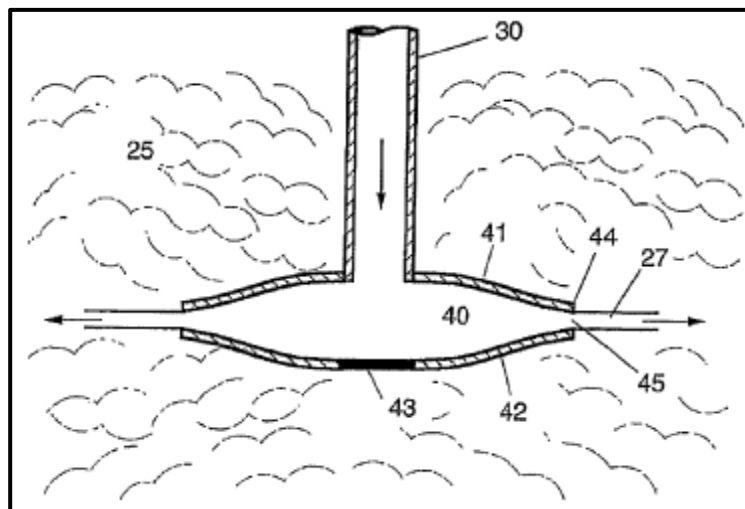


**Figure E-9: Cross section of an alternative wash water distributor nozzle showing the horizontal movement of wash water into the froth (reproduced from Jameson, 2010).**



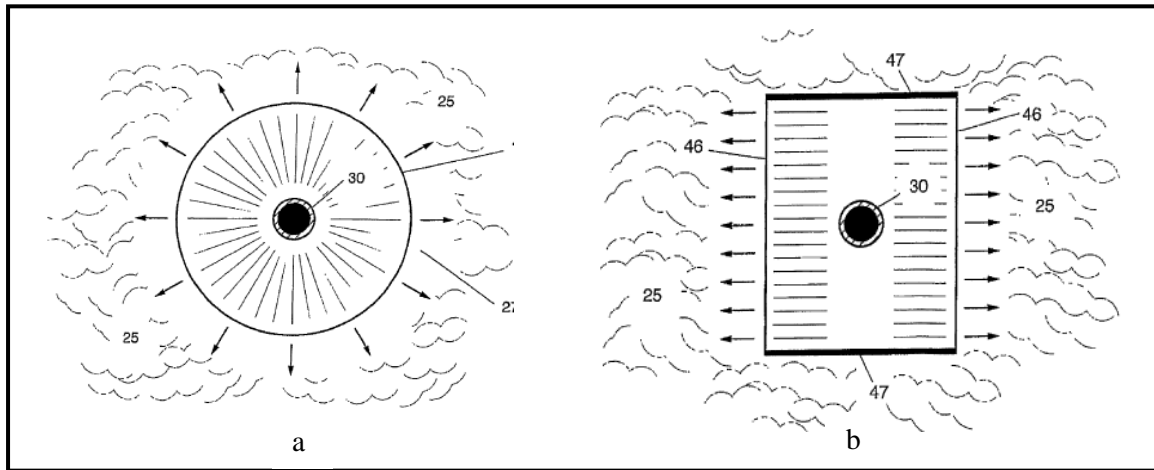
***Figure E-10: Distributor showing the bottom angled to minimise the force of the falling water on the bottom plate (reproduced from Jameson, 2010).***

The upper surface of the impacting plate has a small slope forming a conical upper surface. This has the advantage of moving possible sediments away from the centre by being washed away with the wash water thereby preventing blockage.



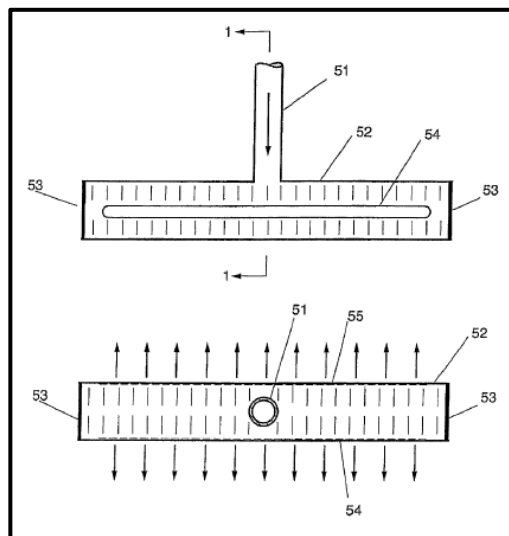
***Figure E-11: Wash water flows through a chamber and passes through the nozzle at a higher velocity as it enters the froth (reproduced from Jameson, 2010).***





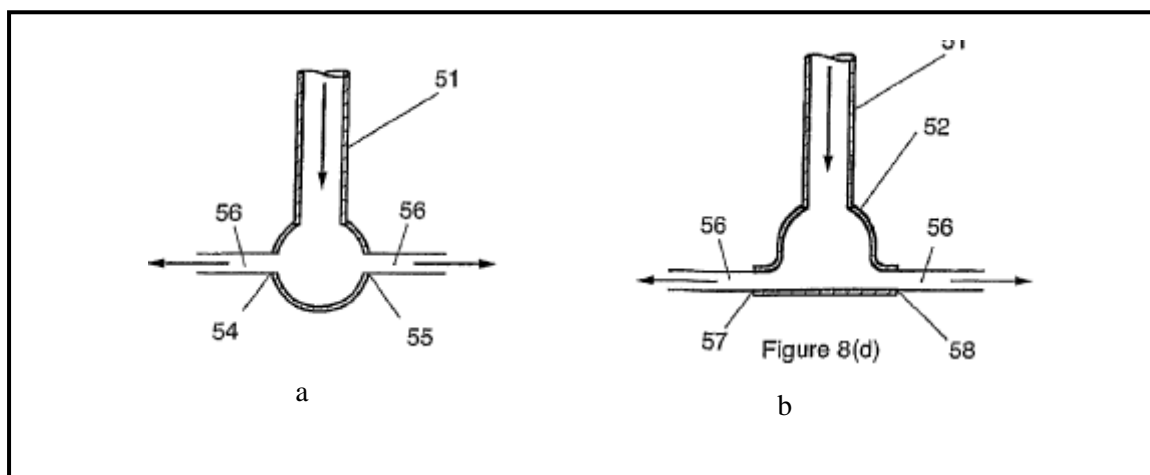
**Figure E-12: Two possibilities of plan views for Figure E-11 (reproduced from Jameson, 2010).**

The wash water is passed into a chamber (40) between two concentric discs whose surfaces are contoured as shown in Figure E-11. The water flows radially outwards in (a). The upper disc is fixed to the entry pipe (30) whilst the lower disc is fixed to a stationary central plate supported by means not shown. In (b), the wash water issues through rectilinear slits (46) bounded by walls (47)



**Figure E-13: Another distributor configuration (reproduced from Jameson, 2010)**

In the above wash water configuration, the wash water enters the distribution chamber (52) which can be constructed in the form of a cylindrical tube sealed at the ends with a plug (53). The slits (54 and 55) are made on the side of the tube at opposite ends, horizontally.



***Figure E-14: Showing the possible plan views for Figure E-13 (reproduced from Jameson, 2010)***

The wash water springs from the slits as liquid sheets (56) that move into the froth in a horizontal direction.

**Advantages:**

- Horizontal streams of wash water are favoured over vertical jets. Horizontal streams have both a vertical and horizontal reach on its spread in the froth whereas vertical jets have just a vertical reach.

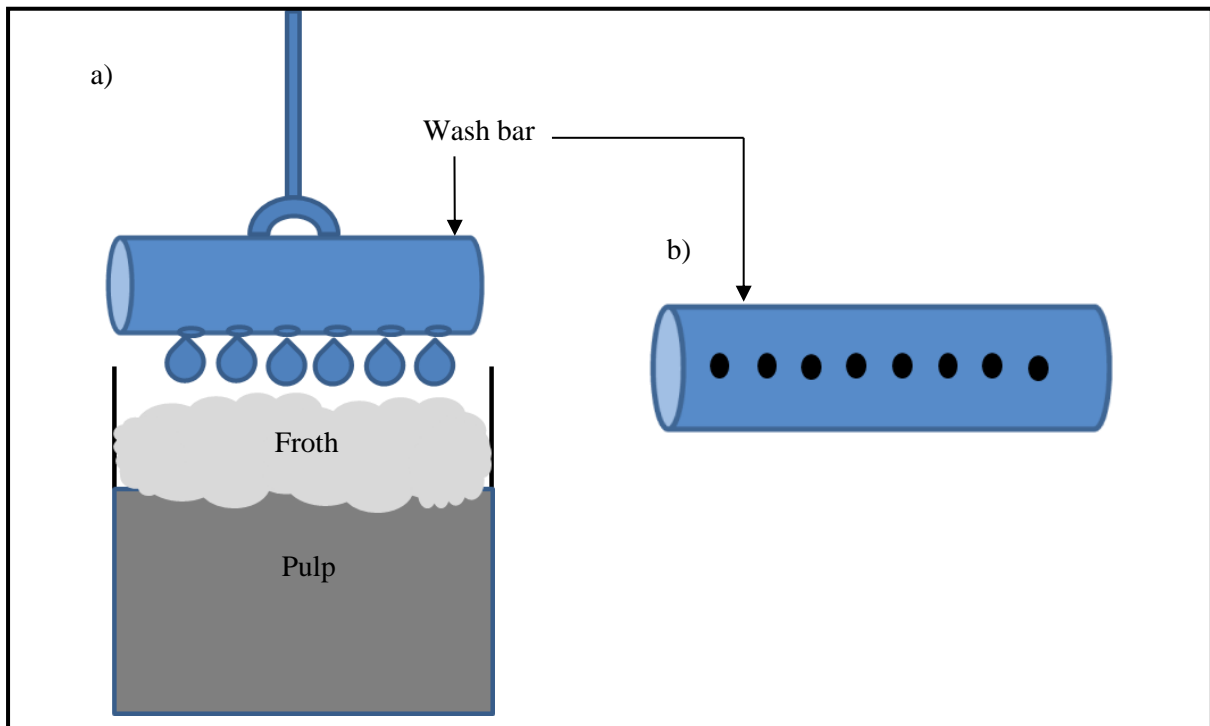
**Disadvantages:**

- Sediments from the froth may block the hole from which the water issues. Holes on the pipe may then be constructed at the bottom of the pipe but then the horizontal-derivation of the wash water stream will no longer be present.
- The nozzle designs are intricate to construct.
- Many nozzles will have to be used for maximum froth coverage.
- Especially in the case of mechanical flotation cells, the design of the manifold system (onto which the nozzles attach) will have to be re-designed to accommodate the impeller.

A factor to be taken into account in the nozzle type distributors is the effectiveness of single streams of water, in the form of essentially cylindrical jets, in washing the froth. Jameson (2010) investigated the effect of single streams of water being injected into the froth. It was found that the vertical distance required for a constant wash water distribution (the desired effect) to be reached is approximately 0.5 to 0.8 m which is relatively large. Therefore, injection of wash water into shallow froths will not benefit from the wash water addition since much of the wash water passes through the froth layer into the liquid layer below.

### Appendix E3: Wash water distributor configuration: Wash bar

The wash bar as a wash water distribution device is a relatively new and simple concept and research on it has not been intensively published in the past. The wash bar is a hollow rod with evenly distributed perforations on the underside for the exit of the wash water. The wash bar is quite similar to the wash box, although instead of the box, a cylindrical tube is used. The wash bar may be of a solid type or it may be constructed of rubber tubing that is bound to a support structure.



***Figure E-15: (a) Schematic of wash bar distributor and (b) the underside of the distributor depicting the perforations.***

**Operation:** As stated previously, the operation is similar to the wash box whereby wash water enters the bar via piping. Thereafter, the wash water exits the device through the perforated holes and into the froth. The perforations should be evenly spaced to ensure a uniform distribution.

**Advantages:**

- The rubber tubing is easily available and holes on the underside may be punctured easily.
- May be placed over the froth as well as inside the froth (as compared to the wash box which may only be placed over the froth surface)

**Disadvantages:**

- The rubber tubing may be subject to clogging since flotation plants often use recycled water which may contain suspended solids and other minerals that may have precipitated.
- The washing operation is limited to a certain section (not the whole froth) hence the optimum position for froth washing is required.

In order to deliver the required flow rate of wash water, a large number of holes are required irrespective of whether the wash water distribution device is a wash box or a cylindrical pipe. Many holes are required due to the need of providing an even distribution of wash water. The optimal distribution would involve an unlimited number of injection points but would be impractical to actually construct.

## APPENDIX F: PROCEDURE FOR PGM + Au AND Cr<sub>2</sub>O<sub>3</sub> ANALYSIS

### Appendix F1: Fire assay (3 PGE+Au)

- Each of the bagged samples was mixed thoroughly (until uniform in colour) with a lead collection flux. This mixture was then placed into a fusion furnace at a temperature of 1150°C for 50 minutes. The reducing conditions in the furnace convert the lead oxide (PbO) to metallic lead (Pb) that forms a complex with 3 of the PGMs (Pt, Pd and Rh) and Au.
- The Pb-3PGE-Au complex is then placed into a cupellation furnace at a temperature of 1000°C for one hour. During the hour, the Pb will be absorbed by the cupel. After cupellation, the PGMs, Au and Ag (added as AgNO<sub>3</sub>), present as a prill, will remain on the hollow surface of the cupel.
- The prill is transferred to a borosilicate glass tube. A solution of aqua regia is then added to the tube. The tube is then adequately sealed and placed in a metal enclosing which thereafter is placed in a dissolution oven where the solution will be left to dissolve at a temperature of 220°C for 8 hours. After the 8 hours have elapsed, the solution of PGMs and Au is transferred into a flask where a standard solution is added up to make up the volume in the volumetric flask. This solution is then assayed on the inductively coupled plasma optical emission spectroscopy (ICP-OES) or the inductively coupled plasma mass spectroscopy (ICP-MS) depending on the quantity of trace metals present (Mintek, 2011).

## Appendix F2: ICP15 (Cr<sub>2</sub>O<sub>3</sub>)

- A pre-determined mass of the sample are mixed with a solution of 5 % HNO<sub>3</sub> in a test tube, the total volume being around 4 mL per sample. The liquid form is then required to be analysed by the spectrometer.
- Lighting of the plasma then occurs since it requires to be lit for 20 minutes prior to the start of each measurement. Argon is fed into the instrument to form the plasma. The plasma is formed by passing the Argon through the tubes that are heated by a coil. Energy is supplied to the coil to provide enable the plasma to have a temperature of 6000°C.
- The sample to be analysed is inserted into the spectrometer in the form of fine droplets. This may be done through means of a nebuliser or a peristaltic pump.
- Inside the spectrometer, the liquid sample is vaporised. Thereafter, the components of the sample are ionised by exposure to electron beams. This results in the formation of ions.
- The mass spectrometer then separates the ions according to their mass to charge ratio.
- The ions are counted by the detector and then the signal that is obtained is processed.
- ICP software translates the data obtained into more useful information – i.e. into % Cr<sub>2</sub>O<sub>3</sub> (Mintek, 2011)