Correlation of mining pit depth and flotation recovery of oxidized platinum group mineral ores

By

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DEDICATION

This thesis is dedicated to my loving Mother. I am grateful for her full support throughout my studies. I thank her for his continuous contribution and support as a mother, understanding and listening to me on my frustration days, my overenthusiastic days and everything in between.

DECLARATION

I, Mohale Lawrence Letseli hereby proclaim that this M-tech research thesis is entirely my very own work and has not been submitted anyplace else for scholarly credit either without anyone else's input or someone else. I comprehend what plagiarism suggests and proclaim that this exploration is my very own thoughts, words, state, contentions, illustrations, figures, results and association with the exception of where reference is unequivocally made to another authors work. I fully understand further that any untrustworthy scholarly conduct, which incorporates copyright infringement, is found in a genuine light by the University of Johannesburg and is deserving of disciplinary activity.

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ABSTRACT

South Africa (SA) has a total of three reefs from the Bushveld Igneous Complex (BIC), which are mined and processed for the recovery of Platinum Group Elements (PGEs). The three reefs are Merensky, the UG2 and the Platreef (Mungall & Naldrett, 2008). The Platinum Group Minerals (PGMs) are mostly sourced from the UG2 reef because they are easy to recover as compared to Merensky and Platreef (Hey, 1999) by froth flotation. The froth flotation concentration process has been at the forefront in concentrating the PGMs from their perspective ores. Different concentrating circuits have been developed and these are aimed at floating mainly the sulfide minerals. Despite success in floating the sulfide minerals, significant complications are being encountered when it comes to floating weathered ores. Poor recoveries and high chromite grades are achieved when using the current conventional processing circuits due to complex mineralogy of oxidized ores (Becker, et al., 2014; Bulatovic, 2010; Cramer, 2001).

The mining pit depth at Thaba Cronimet Mine had been observed as the factor that could be correlated to weathering of the PGM bearing ores. Maximum or economic recovery predictions could be attained in knowing the extent of weathering of mineral bearing ores. This research, therefore, focused on trying to improve flotation performance of oxidized ore from Thaba Cronimet Mine, Northwest Province, South Africa by correlating the degree of weathering to mine pit depth and flotation performance. Ore samples from different mine pit depth were characterized using X-ray Fluorescence (XRF) for chemical composition, Quantitative X-ray Diffraction (XRD) for the mineral composition of the samples, Scanning electron microscopy (SEM) for surface topography, Qemscan, and Mineral liberation analyzer (MLA) for the liberation of the minerals after milling. Mintek flotation predictor software was used to predict the potential recovery of PGMs from both oxidized and pristine. The samples were then floated using sodium isobutyl xanthate (SIBX) as a collector, Sendep 30D as depressant and Senfroth 200 as a frother.

The characterization of the weathered ore indicated a low concentration of sulfide minerals and the weathering of the ore decreased with the increase in mining pit depth. The targeted grind (80%-75µm) for pristine and weathered ore was achieved after milling the sample between 100-120min and 60-100min respectively. The highest achievable 4E PGE (Pt, Pd, Ru, and Au) grade and recovery for weathered ore is approximately 10 g/ t and 41% respectively at a depth

of 918m above sea level. Although the recoveries are low for the weathered ore, the major contributor to this was the low head grade. The highest achievable 4E PGE recovery for the pristine ore was approximately 76% at the concentrate grade of approximately 27 g/t. Recoveries of pristine and weathered ore were predicted using Mintek flotation predictor software. The Flotation Predictor results show that expected recovery for pristine ore is approximately74%, whilst that of the weathered ore is 69% 4E at the current grind size. The time vs. recovery curves show that the pristine ore has a higher fast-floating component (approximately 60% recovery in 5 min.) compared to the weathered ore (45% recovery in 5 min).

In both samples, reasons for losses are mainly PGMs that are still locked within gangue particles, and to a lesser degree, PGMs with too low a liberation index to be considered recoverable (i.e. very small PGMs attached to relatively large gangue particles). Longer residence times may lead to the recovery of the PGMs with low liberation index.

Therefore, with proper knowledge of the source (mine depth) of the ore and degree of oxidation, recoveries of PGEs and the associated base metals can be improved from what has been achieved by previous workers.

LIST OF ABBREVIATIONS

Abbreviations	Meaning
SG	Specific gravity
µm	Micron
BIC	Bushveld Igneous Complex
BMS	Base Metal Sulphides
MLA	Mineral Liberation Analyser
PGE	Platinum Group Element
PGM	Platinum Group Minerals
QEMSCAN	Quantitative Evaluation of Minerals by Scanning Electron Microscopy
Q-XRD	Quantitative Powder x-ray diffraction
SEM	Scanning Electron Microscopy
SIBX	Sodium Isobutyl Xanthate
EDS	Energy dispersive x-ray spectroscopy
XRD	X-ray diffractometer
XRF	X-ray fluorescence
ROM	Run-of-mine ore
Pt	Platinum
Pd	Palladium
Rh	Ruthenium
Au	Gold
Cr	Chromite
Gangue	Unwanted Material
MG	Middle Group

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PUBLICATIONS

Papers published and submitted on the contents of this dissertation are as follows:

Status	Title	Published in:
Published	Characterisation and Flotation of Weathered Platinum Group Metal ore.	Proceedings of the 4th World Congress on Mechanical, Chemical, and Material Engineering (MCM'18). Madrid, Spain – August 16 – 18, 2018
Published	Investigation on the correlation of degree of weathering to pit depth on platinum group metals recovery by flotation	Proceedings of the 19th International Multidisciplinary Scientific Geo-Conference SGEM 2019 at Albena Resort, Bulgaria – 27 June – 7 July 2019 *N.B (The paper received an award for being the best paper in its category)*

CHAPTER 1

INTRODUCTION

1.1   Research Background

South Africa (SA) has a total of three reefs that are mined and treated for the recovery of Platinum Group Elements (PGEs). The three reefs are Merensky, the UG2 and the Platreef. They are all found in the Bushveld Igneous Complex (BIC). Despite having three reefs, most of PGMs are sourced from the UG2 reef (Cawthorn, 1999). In these three reefs, the PGEs are predominantly found in Platinum Group Minerals (PGMs). The PGEs are associated with the base metal sulfides (BMS) (particularly chalcopyrite, pentlandite, and pyrrhotite), and insubstantially associated with silicate minerals. A similar relationship to silicates is also observed with chromite (McLaren & De Villiers, 1982).

All PGMs minerals found in South Africa's BIC are concentrated by froth flotation process. The basic concentration circuits used are milling and froth flotation (MF1) or milling-flotation – milling-flotation (MF2) (Goodall, 1995). These circuits are designed with the aim of floating sulfide minerals and reducing losses that occur due to fines, as well as entrainment of chromite into the final concentrates. Nonetheless, these circuits are not effective when treating oxidized ores. Due to the depletion of high-grade ores in the Merensky reef and expenses related to the processing of UG2 ores, a few organizations have resorted to the treatment of oxidized PGM minerals. These were initially disregarded because of low recoveries, yield, and grade of the final product (Mondal & Mathez, 2007).

The flotation of weathered ore for recovery of PGMs is complicated by its nature. PGMs within the ore body they are not only grouped with base metal sulfides (BMS), some they are associated with other gangue minerals such as silicates. Some of the valuable minerals are locked within gangue minerals which makes the entire flotation process more sophisticated and complex (Newell, et al., 2006).

The impacts of the oxidation process occurred in various parts of the ore body for sulfide-bearing PGM ores are fundamentally the same. The mineralogical nature of the ore and distribution of PGMs in the oxidized ore is highly concentrated with $(Ni, Fe)_9S_8$, and chalcopyrite ($CuFeS_2$). Thus, the final product after the flotation process is subsequently high

in Fe, Ni, and substituted with other base metals like Cu and Co (Eales & Cawthorn, 1996). The nature of PGMs during the formation is intently grouped with sulfide minerals in BIC ore body and most flotation set-up fully depends on these characteristics during the processing and recovery stage, the presence of base metal sulfide association of the PGE/PGMs (Evans, 2000).

In the BIC's mafic-ultramafic rocks and the Great Dyke of Zimbabwe, the effect of the weathering of sulphide ore is dominant down from 10-40m depth from the surface (Junge, et al., 2014). Sulphide minerals and PGMs are wiped out during weathering under oxidizing environment. The Pd is leached out of the PGE bearing thereby reducing the sulphur content within the ore (Oberthür, et al., 2013). This leaves the ore with more oxide mineral compared to sulphide minerals.

Therefore, the correlation of mining pit depth with weathered ore would be studied towards an improvement of the flotation performance of oxidized PGM bearing ore. The experimental work results would be compared to the flotation predictor simulator. The predictor is used to estimates the recoveries of PGMs from respective ores. The recovery predictions are based on the physicochemical properties of the ore, therefore prediction of the oxidized or would be performed. Since flotation of oxidized ores is complex by their nature and results in low recoveries, the floatability of the oxidized ore would then be improved to attain economically viable recoveries.

1.2 Research problem statement

Cronimet Thaba Mine operation now treats open pit material from the pilot plant and recovery of PGEs is low and not constant, especially with close surface material from the pit. The fire assay PGMs results range between 42-45% recovery and occasionally can go as high as 60% subject to the type of material being fed to the plant. The poor performance of PGM recovery is because of the weathering of the pristine ore (Ramonotsi, 2011). The production team at Thaba mine intends to recover PGMs that are only 0.5g/t or more. The ore with the PGMs less than 0.5g/t is pumped to tailings section despite having significant amount of PGMs. Currently, the tailings grade is between 1.2-1.5g/t against the rougher feed head evaluation of 1.92-2.5g/t.

PGM mineralogy within the ore body differs significantly with respect to the area of formation (Cawthorn, et al., 2002). The impact of such mineralogy characteristics assortments is seen on metallurgical material treatment and has been related to potholes, harsh pipes just as the

original nature of the footwall arrangement and the reef thickness (Ramonotsi, 2011). In addition, one of the most serious technical issues experienced during handling and recovery of PGMs from oxidized ore, apart from the low recovery, yields, and grades is various degrees of oxidation levels within the ore body. The mineralogy of oxidized PGM ores is complex and may represent any material from the mine, from almost non-oxidised ores to fully oxidized ores. It requires more complex circuits for processing and extraction of PGMs.

The other challenge is, the reef consists mainly of chromite layers between the critical zones of the BIC, which results in high chromite content (Penberthy & Merkle, 1999) in the concentrate after flotation. A high chromite content within the PGM concentrate imposes challenges during the smelting process. The maximum tolerance fed to the furnace during smelting of chromite in the PGM concentrate of 3%. In this work, the weathering of PGM bearing ore with the mining pit depth would be studied. Moreover, the recovery of PGMs by floatation would be performed, the results obtained would be compared to the flotation of pristine ore.

This research seeks to improve the PGM recovery from an oxidized ore by investigating the effect of mining pit depth and correlate to the degree of weathering and flotation performance of PGMs in the rougher flotation cell.

1.3   Research questions

i.   How does the MG4 mineralogy change with the increase in the mining pit depth?
ii. How can the loss of PGMs recovery be correlated to the mining pit depth?
iii. How can flotation and recovery of PGMs from weathered ore be improved?

1.4   Research Aim and Objectives

The aim of this research is focussed on increasing flotation recoveries of oxidised PGM ores by assessing the extent of weathering and correlate the degree of weathering to the mining pit depths. The main focus of this study is to determine the most efficient way of floating weathered ores and maximize the recovery and grade of PGMs during the flotation process. In order to achieve this, the research was carried out with the following specific objectives:

(i)   To investigate the chemical and mineralogical composition of MG4 ore at different mining pit levels and assess the degree of weathering.

(ii)   To use PGM flotation predictor software to estimate potential PGM recovery and correlate data with batch flotation results.

(iii)   To correlate the mine pit depth and flotation performance.

1.5   Research Scope

The project scope is divided into two stages, the 1st stage establishes the degree of weathering down the mining pit on pristine and oxidized MG4 chromite ore. The primary stage was done with the fundamental aim of finding how deep can mine pit go before pristine ore reached. The weathering profile of the ore body was established by considering the hardness and secondary minerals formed due to the weathering process. The profile of the host rock down the mining pit, as well as the froth flotation process performance of the ore, was intently observed during this campaign. The second phase of the study focused on improving the flotation recovery, yields, and grade of weathered ore and reducing the amount of chromite in the concentrates.

1.6   Research motivation

1.6.1   Global production and demand for PGMs

Worldwide PGM resources and demand keep on developing exponentially due to its extreme interest and its critical economic worth. South Africa contains 66% of reserves and assets and the availability of significant minerals followed by Russia (Mudd, et al., 2018). Social, ecological, and economic background factors are real imperatives on the PGMs market. PGMs production and the need on the market are probably going to stay basic minerals because of its significant value and their special properties (Mudd, et al., 2018).

Total worldwide PGMs supply and the yearly cost are given in Fig. 1.1: including recorded trends. The authentic information demonstrates that combined world supply of the mineral during 2015 was approximately 16,120 tonnes and 50% of PGMs supply was from SA mines, 33% sourced from Russia operations and the greater part of remaining PGMs from Canada, the USA, and Zimbabwe.

It is this supremacy by only two nations that is regularly the key function in the PGEs being surveyed as basic metal (a comparable circumstance to the precious metals, where China holds a close restraining infrastructure on worldwide supply (Weng, 2015). Although there are some

improvements in production, Worldwide PGM production presently can't seem to move back over its 2006 pinnacle.
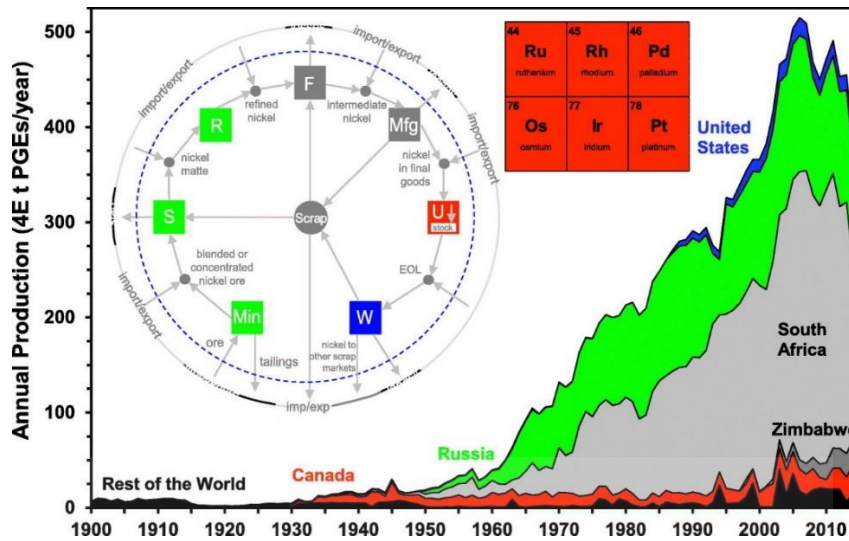
Figure 1.1: Supply of PGMs and the market cost (Mudd, et al., 2018)

The overall Pt and Pd demand by end-users are illustrated in Fig. 1.2. The vehicle auto catalyst continues to dominate in the uses of mined and recycled Pt and Pd. Recycling stream now supplies about one-third of the total demand of these elements in autocatalysts. The significant role played by Pt and Pd in the mining industry is massive and its demand is growing up significantly every year.
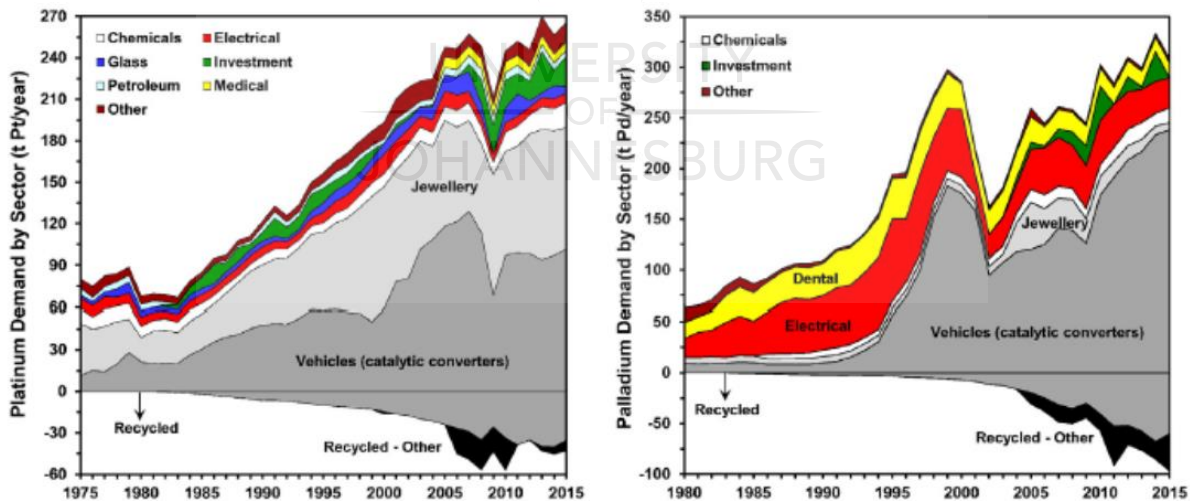
Figure 1.2: Pt and Pd demand by end-use sector worldwide
(Mudd, et al., 2018)

PGM ore body and assets by nation, including evaluated data production, figures 2015 information; arranged by size is shown in Table.1.1. Table 1.1 shows that the total production

of PGMs during 2015 was 16, 7754 t of PGMs including further additional 88,906.5 t PGEs in mineral resources, for example, all-out mineral assets of 105,681.9 t PGEs (4E premise). An examination data of PGMs supply by 2015 total PGM resources with respect to the 2010 production figures and PGM supply from (Mungall & Naldrett, 2008) demonstrates that the Eastern Bushveld figures have decreased by 3.4%, the Western Bushveld has expanded by 9.0% while the Northern Bushveld has almost multiplied production increment that exhibits the developing significance of Platreef ore mineralogy for the fate of the BIC. Great Dyke has expanded by 5.8%.

Table 1.1: Total PGMs ore reserves and production 2015 (Johnson , 2017-2018)

Country	Ore reserves			Additional mineral resources			Total	Cum. prod.
	Mt	4E g/t	t PGEs	Mt	4E g/t	t PGEs	t PGEs	t PGEs
South Africa	3275.0	3.29	10,790.3	16,667.0	3.68	61,411.2	72,201.4	~8230
Russia	946.1	4.16	3931.9	13,339.6	1.04	13,860.7	17,792.6	>5560
Zimbabwe	224.0	3.34	747.2	2439.5	3.46	8436.9	9184.1	~220
USA	293.0	2.48	727.3	4280.1	0.40	1710.4	2437.7	~413
Canada	1414.3	0.36	504.2	2842.7	0.55	1554.6	2058.7	~1101
Finland	156.2	0.48	74.5	662.0	1.23	816.8	891.3	--
Greenland	--	--	--	202.2	2.32	469.2	469.2	--
Australia	--	--	--	607.9	0.45	274.3	274.3	~28
Rest of World	--	--	--	977.1	0.36	351.8	351.8	--

Besides, worldwide PGE ore body production and supply, Ni-Cu-PGE ore deposits and other PGE-containing precious minerals have all went up, mostly identified with extra deposits currently incorporated into the database just as real increments in certain deposits or activities (for example particularly in Canada's Yukon area). The giant Platreef revelation in Northern Limpopo province by Ivanhoe mines likewise demonstrates evidently that even in developed areas there still great prospects to discover new deposits(for example Turfspruit is comparable in size and grade to Mogalakwena) (Mudd, et al., 2018).

1.6.2   Mining PGMs Production trends

SA mining patterns for PGMs production are illustrated in Fig. 1.3, including the total capacity that can be handled per each mine, with extra charts demonstrating the extent of ore sorts and PGMs ore resources. Fig. 1.3 demonstrates the drastic decline of Merensky reef value, while UG-2 reef value increases. The UG-2 reef shows 45% of precious metals and 58% of yearly PGE overall production (contrasted with just 37 and 34% of PGE production in 1995). It demonstrates further increase of new mineral sources, for example, the Great Dyke, Platreef, the Nkomati Ni-Cu-PGE venture and reprocessing of tailings material. Despite having Tailings

contribution of 11% on yearly production throughput in the previous five years, the Tailings still represent 1.4% of yearly PGMs production. This is due to much lower grade and recovery of PGEs from this section.
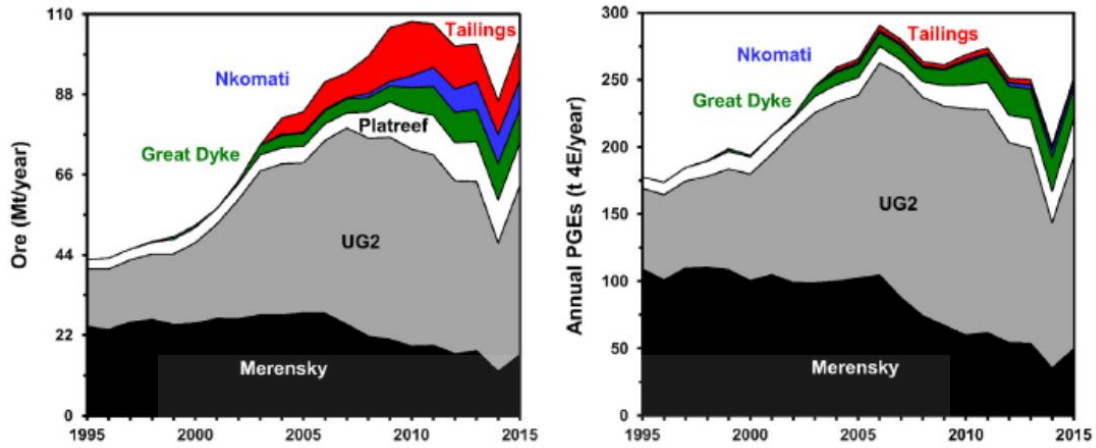
Figure 1.3: Annual PGE production trends over time in Southern Africa (Chinguno, 2013)

The overall grades by main PGM producer's organizations are illustrated in Fig. 1.4. Data presented many organizations including other activities a long term and continuous decrease overall of ore grade (for example Impala and Northam mine), despite the fact that this appears to slow as of late for some others (for example Elant Platinum, Lonmin, Noril'sk-Taimyr). This is most likely identified with discouraging economic situations and the need to keep up the grade to continue incomes as opposed to working expenses.
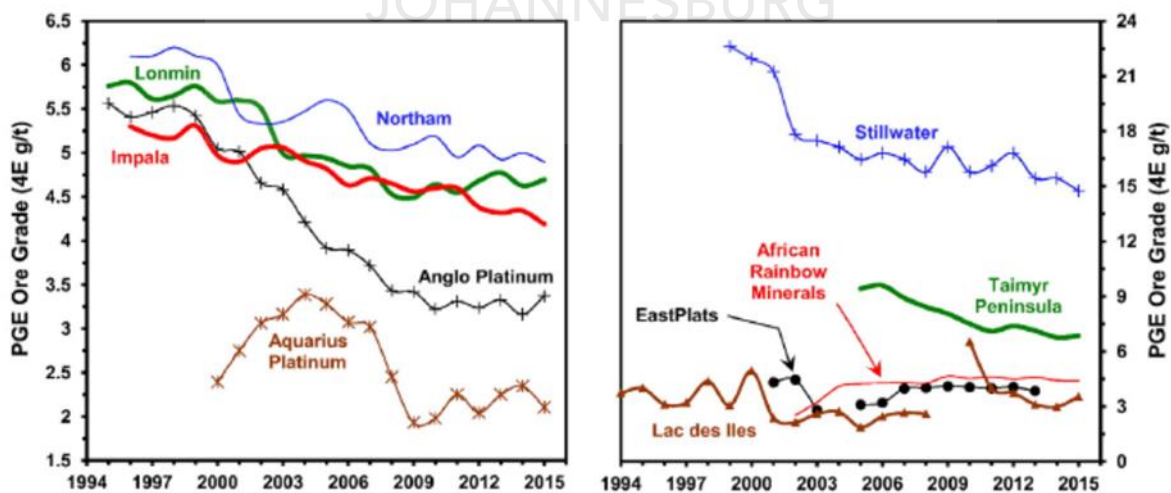
Figure 1.4: PGMs trends in selected projects (Chinguno, 2013).

The recommended future approach for processing low grade and oxidized PGMs ores is probably a perplexing capacity to blend the ore (Merensky/UG-2/Platreef). The financial matters from the mining as the results of high operating cost due to the nature of the ore body also negatively impact PGE production. The ongoing deal by Anglo American Platinum of its more profound Bushveld ventures (for example Rustenberg), proposed that they are currently aimed at processing PGMs from shallower pit depths, progressively open pit, and more Ni-Cu rich ores.

Total production of Pt up to date and related metals recovered has positively impacted revenue income for the SA economy. During the processing stage, most significant loss of PGMs occurs in the concentrating stage whereby 10-15 % of PGMs are lost due to weathering. The flotation and recovery of PGMs from the material mined and processed near-surface area regularly met with restricted achievement. Due to the climatic conditions and the effect of weathering within the ore body, the material close to surface is significantly more altered and oxidized compared to the material found covered somewhere deep in the ground.

The level of oxidation within the pit ore tends to decrease with increasing mine pit depth, thus this research is focused on optimizing platinum flotation and improve recovery yields and grade of PGMs from oxidized ores.

1.7   *Significance of the research*

Processing of PGMs from oxidized ores is complicated during the flotation process due to the change in mineralogy and low concentration of Pd which remobilized during the weathering process. The low Pd content alters the Pt:Pd ratio, the only key used during the flotation process of PGMs. With the current processing circuits, the processing of oxidized ores normally results in very poor recovery. Due to high demand worldwide and diminishing supplies of Merensky Reef and the scarce locations of Platreef, most recent flotation research has been focused on improving the recovery of PGMs from weathered ore (Evans, 2000). During the processing period, significant losses of PGMs occurs in the concentrating stage because of weathering (10-15% in some cases even more). This research is focused on increasing recoveries during flotation of oxidized PGM ores.

1.8   Research Methodology

Representative samples of MG4 fresh ore were collected from the open-pit mine at Limpopo province, South Africa. The samples were collected at different mining pit depth. Each sample was then properly blended and 1.3kg of each ore was milled with 30% of solids-water in the rod mill approximately for two hours to produce a suitable grind of 80%-75µm. Prior to milling test work, the ore was characterized using XRF for chemical composition, XRD for mineralogical phases and SEM/EDS for surface morphology. The flotation predictor software was then used to estimate the potential PGM recovery. The ore was then concentrated using laboratory flotation cell to produce concentrates only at the rougher stage. Each fraction was filtered, dried and weighed (feed, concentrates and tails) and analyzed.

1.9   Structure of the dissertation

The dissertation is divided into five chapters. Chapter one details an introduction to the project, problem statement, research questions, aim and objectives, research scope of work, research motivation, the significance of the study and the methodology. Chapter two presents a comprehensive literature review on the study with the full description and the background of the Bushveld Igneous Complex (BIC). The flotation of weathered ore including the technical challenges involved during processing is also detailed. A brief description of the flotation process is also presented followed by the review of the mineralogy. Chapter three discusses the methods and analytical techniques that were used to analyze the samples and the software that was used to predict recoveries. In line with the problem statement, aim and objectives of this study, chapter four presents the results in the form of figures and tables with a discussion on observations. The conclusions and recommendations according to the research results are presented in Chapter 5. The detailed tables of results and the description of the analysis methods utilized are appended in the appendix.

CHAPTER 2

LITERATURE REVIEW

This section details a comprehensive study on PGM reefs found in South African BIC. The full evaluation and assessment of the ore body contemplations when treating oxidised ore and the impact of mining pit depth towards recovery, yields and grade during the flotation process is presented as well as the past work directed on the processing of oxidised ores using flotation and hydrometallurgical routes

2.1 Bushveld Igneous Complex

2.1.1 Geology and general overview

The BIC consist of an area approximately 65000 km^2 and thickness is stretching from 7-9 km. This is the main reserve of PGEs and an enormous reserve of chromite (Mungall & Naldrett, 2008). SA is the biggest producer of PGMs on the planet followed by Russia (Cawthorn, 1999). It is estimated that approximately about 75% of the world's platinum commodity is available in the BIC. There are three principle mineral bodies, viz. the Merensky reef, the UG2 reef and the Platreef. Among all the three ore bodies, Merensky contains relatively high concentration of PGMs with fewer complications during the processing and recovery.

 In these three reefs, PGEs are hosted in the PGMs that are related with base metal sulfides (particularly chalcopyrite, pentlandite and pyrrhotite), and less relationship with silicate minerals just as with chromite (McLaren & De Villiers, 1982). Geological formation map of BIC is illustrated in Figure 2.1, including the names of the mines currently mining and processing PGMs and chromite ore.

PGEs are mined from three territories in the complex: The Western, Eastern and Northern Limbs. Two reefs are mined from the Western and Eastern Limbs, known as the Merensky and UG2 reefs. The well-created foundation near the Western Limb has seen this territory experience significant exploration. Notwithstanding, as of late significant advancements have occurred along the Eastern Limb. The Northern Limb

contains Platreef, which has seen some advancement as of late and is probably going to be of developing significance of PGMs source in the future (Cawthorn, 1999).
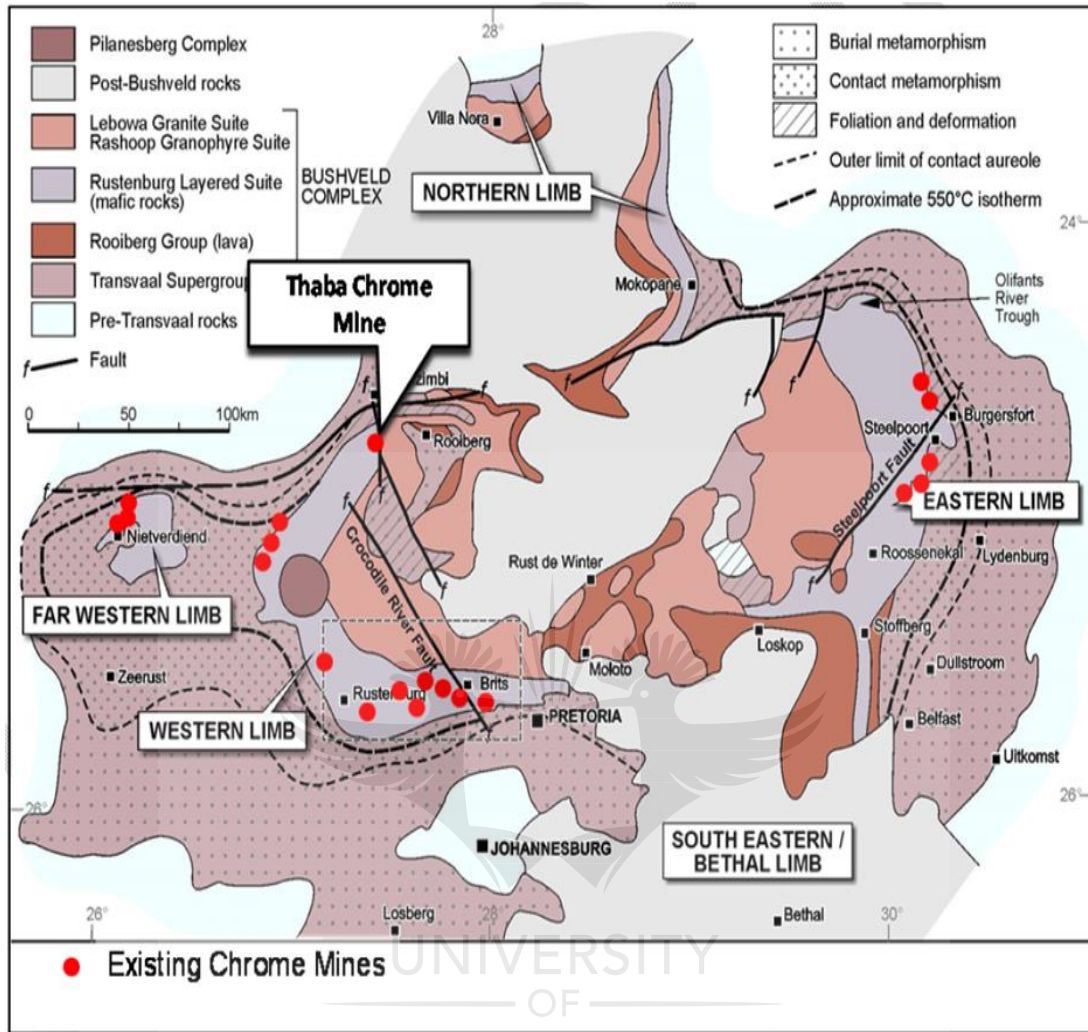
Figure 2.1: Geological map of the Bushveld Igneous Complex, obtained from a compilation by (Eales & Cawthorn, 1996).

The BIC Complex is plainly stratified into several layers, as exhibited in Figure 2.2. Over the previous years since exploration, mining and processing of PGMs from BIC, over 60% of the valuable minerals are found mainly inside the critical zone. As illustrated from Fig 2.2, the critical zone is divided by chromite layers namely, upper group, middle group and lower group respectively (Hochreiter, et al., 1985).
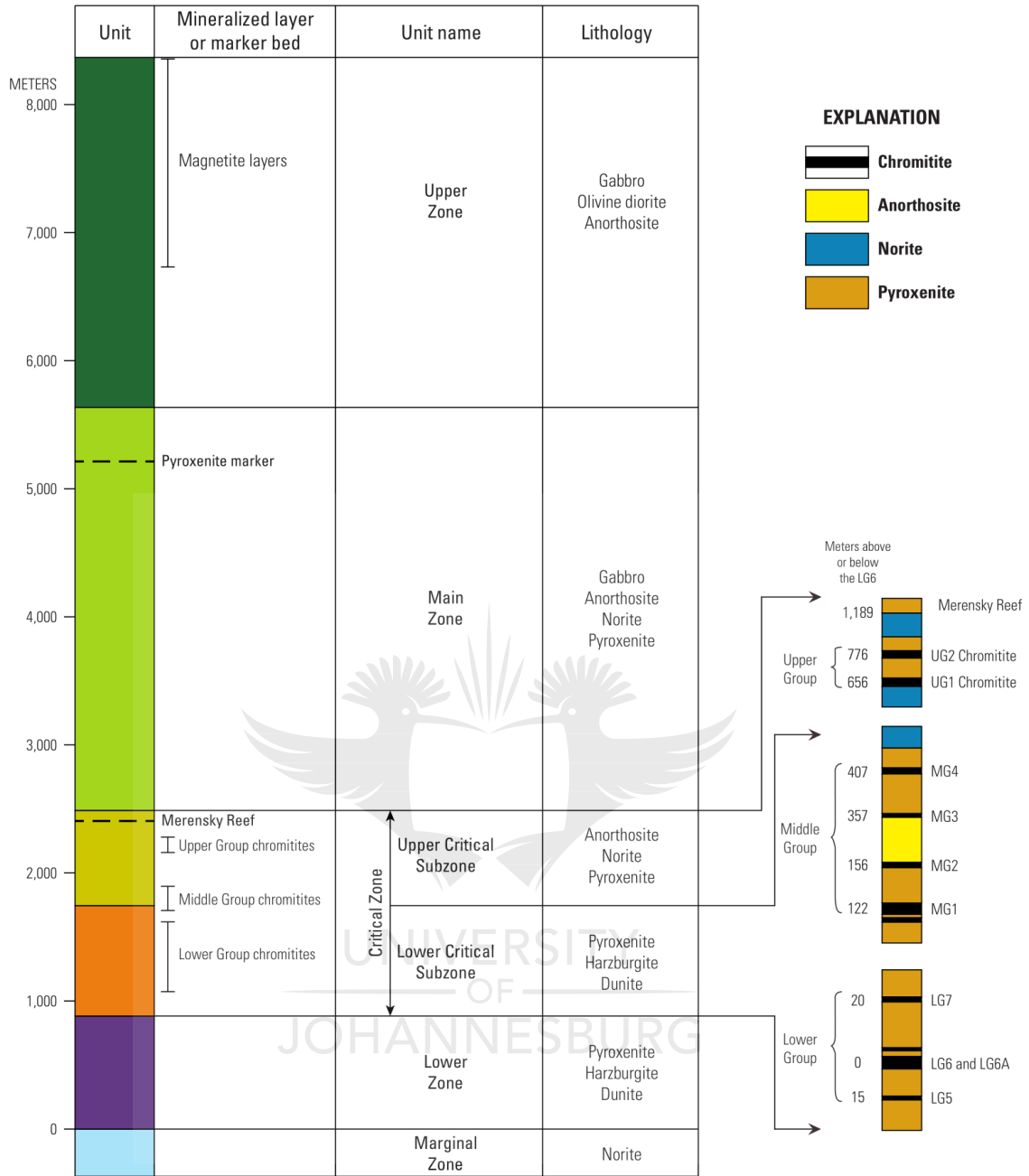
Figure 2.2: Stratigraphy of the Bushveld Igneous Complex (Hochreiter, et al., 1985)

The Critical Zone has the two fundamental reefs from which PGE's are mined and processed, the Merensky Reef and the UG2 reef. The Merensky Reef is located close to the highest point of the Critical Zone, where the chromitite layer is ineffectively created. As the name recommends, the UG2 reef is the second chromitite layer in the Upper Group. As indicated by (Liddell, et al., 1985) , the UG2 reef lies roughly 150

meters underneath the Merensky reef in the Western Bushveld complex and is somewhere in the range of 15 and 255 centimetres thick.

The content of chromite material from the UG2 reef extends in the range of 30% and 90%, with the additional small percentage of silicate minerals comprising mainly of orthopyroxene and plagioclase. The content of base metal sulfides (BMS) within the ore body normally integrated with chromite grain limits. Other sulphide minerals like Chalcopyrite, pyrrhotite and pentlandite are the principle BMS present with traceable amounts in the UG2 reef. PGE's in the UG2 reef is related to BMS and are present as single grains. These single grain platinum bunch minerals (PGM's) are present on the limit between chromitite grains, among chromite and silica gangue grains or secured different gangue minerals (Junge, et al., 2014; Cawthorn, 1999).

2.2   *Principles of Froth flotation*

Froth flotation is a process where valuable minerals are separated and recovered using differential physiochemical properties of both valuable and gangue minerals. Flotation has been in existence and implementation took place since 1860 with William Haynes' exhibition. William used oil devotion to isolate precious minerals from gangue. The flotation process is accomplished by transferring hydrophobicity to the ideal mineral particles and the supply of air bubbles rises through the slurry.

 The mineral is rendered hydrophobic by including a chemical reagent called collector, which is pump over the specific concentration, mixed and agitated together with the slurry. The total composite in a form of slurry is pumped at the specific rate to the flotation cell first stage/rougher cells to float and recover valuable minerals as illustrated in Fig 2.3, (Wang, et al., 2016; Becker, et al., 2014; Ekmekci, et al., 2003).
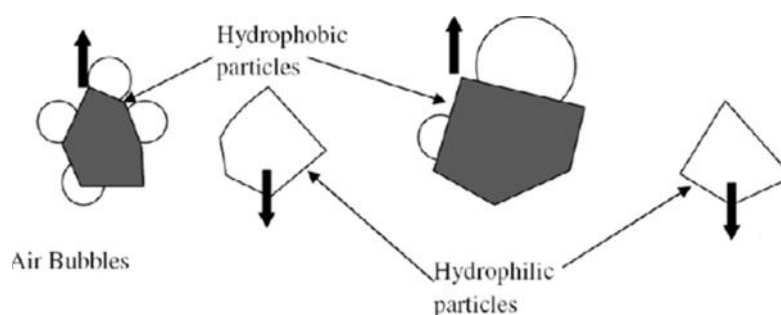
Figure 2.3: Selective attachment of air bubbles to hydrophobic particles. (Wang, et al., 2016)

Once chemical reagents dosing is accomplished, the flotation particles in the cell are implemented by the mechanical disturbance of the slurry utilizing the impeller. The air is then pumped into the slurry to promote the formation of air bubbles in order to float and recover the ideal mineral. Generated air bubbles will at that point connect themselves to the particles and thus suspend them to the water surface of the slurry, simultaneously while the gangue material is gravitating to the base of the flotation cell.
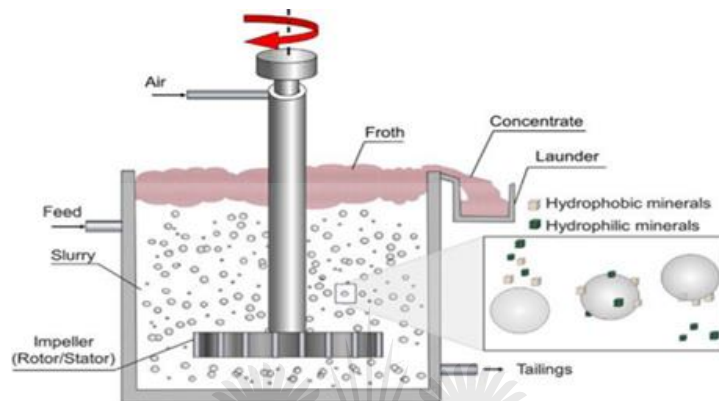
Figure 2.4: Simplified schematic of a conventional flotation cell (Jovanović & Miljanović, 2015).

Wills & Finch, (2016) stated that the molecule bubble connection can be portrayed by three autonomous sub-steps, i.e.: collision, connection and steadiness. Figure 2.5 demonstrates an old-style representation of the three-stage contact among water and air bubble and the mineral molecule surface.
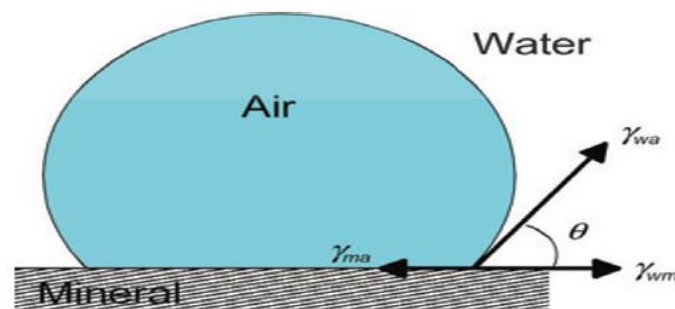
Figure 2.5: An idealised illustration of the three-phase equilibrium contact between air, water and mineral surface (Wark, 1993).

Figure 2.5 completely exhibits a classical diagram of the three-arrange contact among water and air bubble and the mineral atom surface. At balance, the three interfacial free energies are associated with the Young condition presented at equation 2.1:

$$(Y_{ma} - Y_{mw}) = Y_{wa} cos\theta_c \qquad (2.1)$$

Y_{ma}, Y_{mw} and Y_{wa} being the interfacial energies of the mineral-air, mineral-water and water-air respectively and θ_c is the contact angle.

Equation 2.2 represents the relationship between the bubble and the particle during the attachment period:

$$Y_{ma} - Y_{mw} < Y_{wa} \qquad (2.2)$$

The relationship between the changes in free energy $\Delta\gamma$ in correlation with the creation of the mineral air interface is presented in Equation 2.3,

$$\Delta y = Y_{ma} - (Y_{wa} + Y_{ma}) \qquad (2.3)$$

For the flotation process to occur the change in free energy $\Delta\gamma$ must always be negative.

2.2.1 Flotation bubble attachment

The likelihood of the valuable mineral particle to be connected to the flotation bubble during the flotation is controlled by the surface zone on the mineral that is accessible for collector adsorption, just as the surface zone on the flotation bubble that is readily accessible for attachment. The interaction between the bubble and the particle is a function of intermolecular and electrostatic forces. The reagents used in flotation modify the valuable particle surface to promote the attachment to the air bubble. The reagents used are collector, activator. The measure of bubbles generated, and their size impacts the total aggregate bubble surface area accessible for connection of the mineral onto the air bubble (Nguyen, et al., 2006; Ekmekci, et al., 2003).

2.2.2 Flotation bubble detachment

The detachment of valuable mineral after bubble loading, likelihood is impacted by capillary forces during the collection stage, molecule weight and the turbulent vortexes generated by the flotation agitator assist the bubbles to rise. Expanded vortex turbulent

in the flotation cell does not only assist in collision impact, yet shockingly assist on the increment particles to detach themselves from the flotation bubbles to a lesser extent degree. The contradicting forces of gravity and lightness on the joined minerals additionally influence the separation probability. On the off chance that the gravitational forces they are too high, the flotation bubbles will break, and particles will be released from the bubbles (Nguyen, et al., 2006; Jovanović & Miljanović, 2015).

2.2.3   True flotation

The relationship between the bubbles generated to the concentration of particles and the total rising from the pulp interphase is transferred through the pulp-froth interface. The relationship between the bubbles generated and % solids aggregate gradually and report to the top of the froth bed and collected from the flotation launder to the next stage for further processing. Therefore, true flotation in practice represents hydrophobicity of the particles and their selectivity properties of attaching themselves to the bubbles during the flotation process. True flotation is fully influenced by the behaviour of the pulp phase.(Bulatovic, 2010).

2.2.4   Entrainment

Flotation entrainment is a non-selective process, during the flotation process, liquid from the pulp stage conveying unselected particles enters the froth stage. Unattached particles get uplifted upwards through the interface because of an upwards water transition brought about by the upward development of the air pocket molecules. The amount and velocity of water at the froth phase is fully dependent on the rate of air supplied by the compressor. The measure of entrained material reporting to the final concentrate is fundamentally controlled by the qualities of the froth phase instead of the conditions in the pulp phase. The total mass of entrained particles has been found to diminish exponentially with increment in froth height as these unattached particles channel from suspended bubbles (Neethling, 2002; Cramer, 2001).

2.2.5   Drainage

Drainage is the gravity separation process where particles gravitate down due to their physical properties through the froth boundaries as illustrated in Fig 2.6. The watery air pocket shell is the layer around the air pocket that contains generally hydrophobic

particles, shaping the air pocket molecule and assist the bubble in aggregate. The following layer is the between air pocket lamella, which comprise of fluid and unattached particles. Unattached particles in the lamellae can drop out of the foam stage once again into the mash stage or they can reattach to the air bubble molecule beneath. The likelihood of unattached particles dropping out of the froth is represented by the lamella properties, descending gravitational forces just as upward water transition. Drainage is a significant process of flotation process as it upgrades the selectivity of the valuable particles attachment and recovered for the next processing stage (Neethling, 2008).
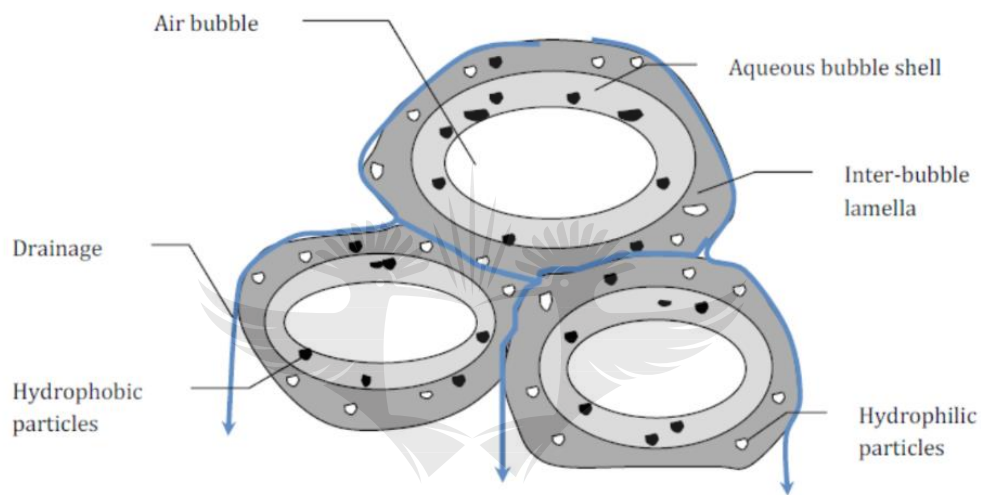
Figure 2.6: Illustration of the inner workings in the froth phase as suggested by (Cilliers, et al., 1998).

2.2.6   Bubble size

Top froth section and bubble size is a result of both the pulp and froth stage conditions and exhibitions. The conditions of the pulp phase and the bubble particle size influence the froth interface to select and attach valuable minerals to the inlet and bubble scattering. The conditions of the froth phase also influence the particles residence time inside the flotation bubbles and improve the properties of the drainage system determined by lamellae properties (Moolman, et al., 1996).

2.2.7 Bubble shape

Froth layer depth on the upper zone of the flotation cell, determines the shape of the bubble. The relationship between the bubble-particle on the froth zone and lamellae that are thick and watery their shape during the flotation process tends to be rounder because of liquid filling the spaces between them. As layers become thinner due to water seepage, totals push against each other and compete for space. As the aggregates swarm around each other, their shapes take on a polyhedral structure Similar to bubble size the key froth phase contributors to the upper froth layer bubble shape are froth residence time and the rate of draining due to the lamellae properties (Moolman, 1996)

2.2.8 Recovery

Equation 2.4 represents the metal recovery of valuable minerals. This equation illustrates the total mass of the targeted valuable minerals which reports to the final product over the total mass of targeted minerals from the feed.

$$\%R_0 = \frac{M_P}{M_F} * 100 \qquad (2.4)$$

The overall total metal recovery is fully depended on the condition of the pulp and the froth. Total residence time during the flotation process and the froth should be enough to allow recovered valuable particles to be released back into the pulp so that the quality of the final product is not severely compromised.

2.2.9 Grade

Equation 2.5 represents the percentage grade of the final. This is the upgraded final product quality and is the total mass of the targeted valuable material that reports to the concentrate as a percentage of the total mass of solids reporting to the concentrate.

$$\%G = \frac{m_{p,PGM}}{m_p} * 100 \qquad (2.5)$$

External operational factors influencing flotation selectivity, such as liberation, density, froth height and reagents dossing they play a significant role during the flotation process and they aid on producing concentrate of a desirable grade.

2.2.10 Mass pull

Mass pull during the flotation process represents the total mass of valuable minerals recovered on the final product. It can also be classed under operational requirement/ plant target, a certain amount of valuable minerals must be recovered and reports to the final product. Pulp phase conditions, recovery rate and froth residence time they are the key performance influencing the mass pull. High mass pull can be achieved by adjusting the air flowrate and pulp height (Muller, 2010).

2.3 *Factors Influencing Froth Flotation*

Critical physical and chemical parameters involved in froth flotation are classified into three groups as shown in Figure 2.7.
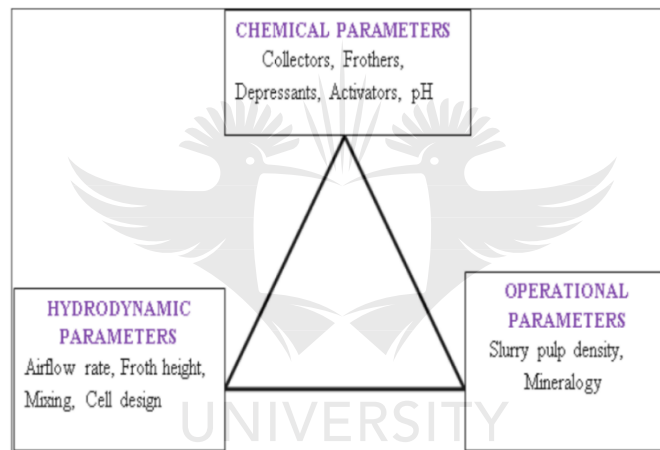
Figure 2.7: Flotation components (Neethling & Brito-Parada, 2018)

The properties of crude mineral blends suspended in plain water are seldom reasonable for froth flotation. Specific flotation reagents are required both to control the overall hydrophobicities of the particles and to keep up the correct flotation attributes. It is along these lines a wide range of chemical reagents engaged with the flotation process, with the determination of reagents relying upon the particular mineral being processed.

2.3.1   Collectors

Collectors are chemical flotation reagents which are utilized to specifically adsorb onto the surfaces of particles being floated. They structure a monolayer on the molecule surface that basically makes a slight film of non-polar hydrophobic hydrocarbons. They enormously increase the contact angle with the goal that bubbles will cling to the

surface. The choice of the right collector is critical for a viable selectivity of the valuable minerals within the ore. Collectors can be by and large classified relying upon their ionic charge: they can be nonionic, anionic, or cationic, as shown in Fig 2.8.
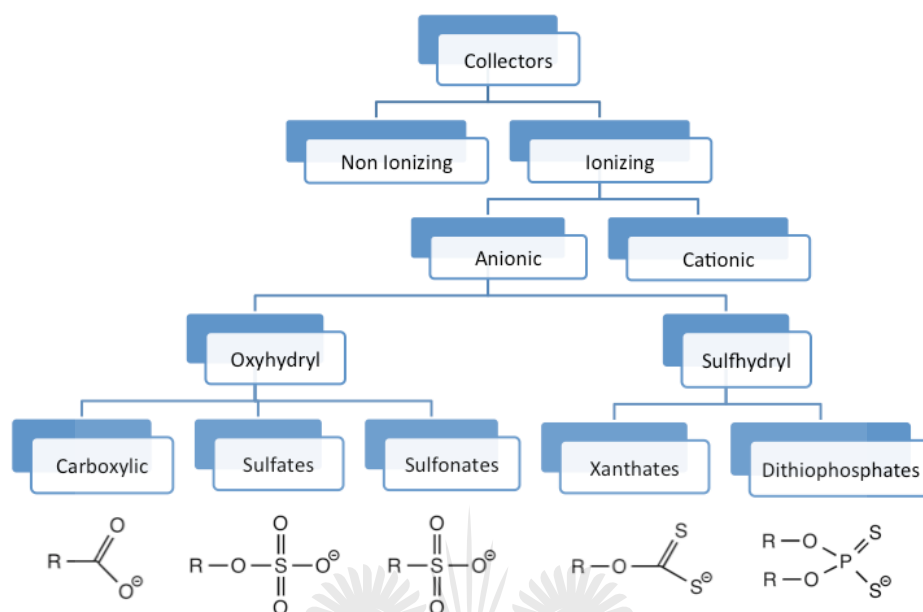
Figure 2.8: Different types of collectors, or surfactants, used in froth flotation.
(Lotter & Bradshaw, 2010)

The non-ionic collector is straightforward hydrocarbon oils, while the anionic and cationic collectors comprise of a polar part that specifically connects to the mineral surfaces, and a non-polar part that tasks out into the arrangement and makes the surface hydrophobic. Collectors can either artificially attach to the mineral surface (chemisorption) or be attached superficially by physical adsorption (Lotter & Bradshaw, 2010).

When the collector is joined onto the molecule surface (Fig 2.9.) with its hydrophilic useful gathering, the natural end arranges opposite to the surface. Thus, a gathering of these collector coats the molecule's surface and consequently give hydrophobicity to the mineral molecule surface, which is generally hydrophilic. When the molecule surface is hydrophobic, it will in general move out of the water to the air, bringing about the bubble molecule connection. When the molecule is connected to the bubble, it is then lifted to the slurry surface, leaving the hydrophilic gangue material to tumble to the base of the flotation cell. These mineral particles overflow with the froth from the

slurry surface into the cell wash or gathering skillet of flotation cell (Lotter & Bradshaw, 2010).
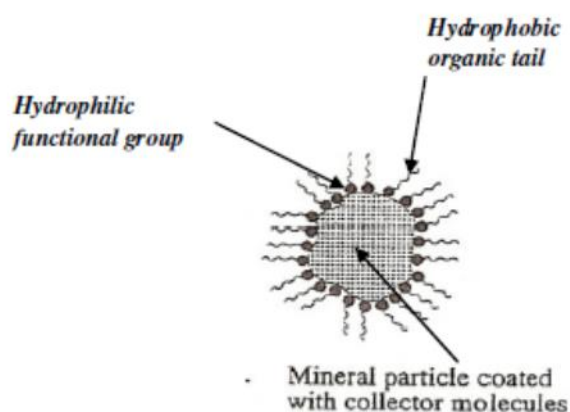
Figure 2.9: Collector molecule on the mineral particle surface (Lotter & Bradshaw, 2010).

Collectors can be arranged by the sort of ion that is shaped upon their interaction in solution. They can be ordered based on synthesis and whether they exist as anionic, cationic or covalent species in solution

2.3.2   Depressant

The main function of depressant during the flotation process is to suppress the influence of hydrophilic gangue mineral furthermore improve the grade of the final product (Wiese, et al., 2008). All flotation depressants they must have the following characteristics:

(i) Properties which suppress the influence of gangue material.

(ii) Strongly hydrophilic properties by virtue of either the same or other functional groups in the molecular structure.

(iii) They must present functional groups that compete effectively with the collector for the surface of the minerals which are to be floated.

The polymeric depressants regularly utilized in PGM flotation applications are either adjusted guar gum or carboxymethyl cellulose (CMC). The UG2 mineral contains normally floatable talc and guar gum and CMC are great depressants for suppressing talc (Wiese, 2009) . The real contrast between them is that CMC ionizes in arrangement

and it is adversely charged while guar is just somewhat charged. CMC particles subsequently, once adsorbed onto gangue minerals, cause these minerals to turn out to be adversely charged and, especially at high doses, they thus will in general repulse one another. Guar gum is a fanned polysaccharide with galactomannan shaping the essential unit. The hydroxyl gatherings are orchestrated in a cis-design on the C-2 and C-3 iotas. In this manner, the guar gum has been more grounded depressant of normally skimming gangue than CMC at low doses (Wiese, 2009). High depressant measurement expands the evaluation and diminishes the recuperation of the concentrates (Wiese, et al., 2010) .

2.3.3   Frothers

Frothers generally classified under compounds that demonstration to balance out air bubbles with the goal that they will stay well-scattered in the slurry, and they will typically frame a steady froth layer that can be expelled before the air bubble burst (Figure 2.10). The most ordinarily utilized frothers are alcohols, especially Methyl Isobutyl Carbinol (MIBC), or 4-methyl-2-pentanol, an expanded chain aliphatic liquor) or any of various water-dissolvable polymers dependent on propylene oxide (PO), for example, polypropylene glycols. The polypropylene glycols specifically are exceptionally flexible and can be custom fitted to give a wide scope of froth properties (McFadzean, et al., 2016).

Numerous different frothers are accessible, for example, cresols and pine oils, yet the majority of these are viewed as outdated and are not as broadly utilized as they used to be. If there should be an occurrence of mineral flotation it is usually accepted that, if the frother dose increments, there is an expansion in water recovery. Water transport is affected by the air bubble size or more definitely the bubble surface region transition. The high concentration of froth reagent prompts a steady froth. The conduct of the froths is constrained by the properties of the frother molecules at the air/water interface (Wiese & Harris, 2012).
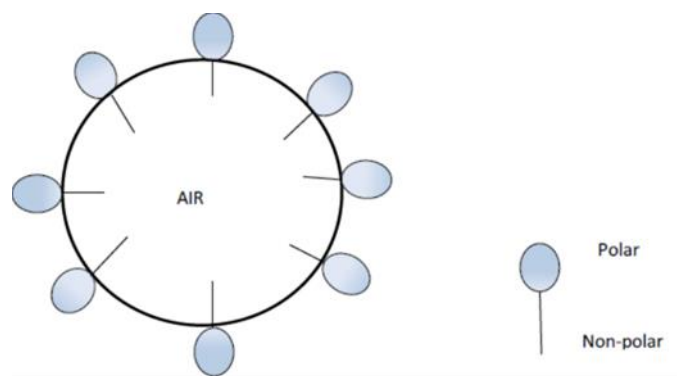
Figure 2.10: Working principle of a frother molecule (McFadzean, et al., 2016)

This interface moves toward visco-versatile, progressively fixed and progressively stable when expanded measures of frother particles are adsorbed at the air/water interface. A definitive impact is that fluid waste from the lamellae is hindered and subsequently, bubble mixture is repressed. The general impact is that froth stability is expanded with increasing the frother concentration. This is due to of the Gibbs-Marangoni theory, the impact which proposes that if the lamellae are diminished, at that point the convergence of frother particles at the interface is upset from its steady state. Accordingly, if abundance frother molecules are available in the pulp stage, these will relocate to the interface and therefore re-establish the balance concentration bringing more water being dismantled in to expand the film thickness. On the off chance that water recovery increments, there is increasingly strong recovery because of the steady froth which lessens selectivity and prompts a non-entrainment (Wiese, et al., 2011).

2.3.4   Activators

Activators are normally added to adjust the mineral surface to upgrade or expand the collaboration of the collector with the particle surface, which is generally not efficient alone. One case of a generally utilized activator in the sulfide flotation is copper sulfate (CuSO4). Different activators, for example, Fe^{2+}, Pb^{2+} can likewise be utilized, contingent upon the mineral sort. CuSO4 is broadly utilized in Merensky PGM flotation tasks (Cramer, 2001).

To support the flotation of PGMs for instance, $CuSO_4$ is typically added to the pulp. An examination directed by (Bradshaw, et al., 2005) demonstrated that dossing of CuSO4 significantly expanded PGMs flotation execution. These outcomes affirmed

results gotten by (Bradshaw, et al., 2005) express that the improvement in PGMs flotation execution shows that copper activation takes place at alkaline pHs, however, the idea of the initiation component was not electrochemical and was probably going to be synthetic. The Cu^{2+} particles adjust the outside of the pyrrhotite mineral and make it workable for the collector to effectively connect itself to the actuated surface.

2.3.5   PH Control

The pH is used as a modifier during the flotation of PGMs. During the flotation of PGMs is adjusted and regulated between 9-11 pH. Its main application during the flotation process is used to affect the surface chemistry of the floated PGMs minerals. In most cases, PGMs targeted minerals have a positive surface charge under acidic conditions and a negative charge under alkaline conditions (Wiese, et al., 2011).Since each mineral changes from negatively-charged to positively-charged at some particular pH, it is possible to manipulate the attraction of collectors to their surfaces by pH adjustment (Wiese, et al., 2011).

Moreover, there are other complex effects due to pH that change the way that collectors adsorb on mineral surfaces. Sulfhydryl collectors such as xanthate ions compete with OH- ions to adsorb on mineral surfaces, and so adsorption is a function of pH. This makes it possible for sulfhydryl collectors to be used to progressively separate specific minerals. The pH where the xanthate ion wins the competition with OH- ions depends both on the concentration of xanthate in solution, and on the specific sulfide mineral present (Wills & Finch, 2016).

2.3.6   Effect of Froth Height

Recovery of talc and other gangue minerals can be reduced significantly through the column flotation circuit since froth depth levels can be adjusted directly during the flotation process. Flotation model developed by Cho & Laskowski, (2002), states that the mechanical entrainment is a complex mechanism and is caused by changes in froth residence time, froth structure and froth properties. In order to improve the floated particle residence time, the froth height is maintained in the froth zone. The flotation dynamics philosophy which controls the fundamental actions which have an impact on flotation is the attachment of hydrophobic particles onto the bubbles in the collecting

zone. Flotation research by Ekmekci, et al. (2003) states clearly that the increase in froth height during the flotation process has the potential of reducing mechanical entrainments of chromite concentrates.

2.3.7   Impeller speed

The speed of the flotation impeller must be controlled and maintained at the steady-state to achieve pulp homogeneity during the flotation process (Gupta & Yan, 2006). The impellor rotation is used to keep solids in suspension and to blend flotation reagents. In the normal operation the drive of the impeller is connected to the Variable Speed Drive (VSD) control unit and the speed can be adjusted accordingly per respective flotation cell.

2.3.8   Airflow rate

Air rate supplied by the flotation compressors is used to generate flotation bubbles in order to keep the solids in suspension within the flotation cell. The air supply is used to influence the froth conditions and stability also improve flotation performance. The amount of air supply plays significant role in the recovery, yield and the grade of the final product. (Hadler & Cilliers, 2009).

2.3.9   Particle size distribution

Recommended particle size distribution for flotation of PGMs is 80%-75µm. This is chosen to ensure proper liberation between the gangue minerals and targeted valuable minerals before the flotation process can take place. The performance of the flotation process is also fully depended on efficient liberation between the mineral and the waste particles (Gupta & Yan, 2006).

2.3.10 Temperature

The level of temperature is not controlled during the flotation process. However, it plays a significant role during the flotation process. It influences the rate of PGMs recovery depending on the type of ore been processed. Generally high temperatures are expected from the milled ore and the material cool down naturally as it enters the flotation cells (Moolman, 1996).

2.4   *Mineralogy of oxidised ores of the Bushveld Igneous Complex*

PGMs from oxidized ore during the characterisation period are identified by a loss of Pd content to Pt relationship, Pd is more mobile and normally leach out during the weathering process in a supergene environment through surface and groundwater. The presence of PGE within the ore body occur as (i) primary PGM, (ii) in solid solution with relict BMS, (iii) as secondary PGE alloys, (iv) as PGM oxides/hydroxides and (v) as PGE associated with secondary oxides/hydroxides (e.g. Fe, Mn) or silicates (e.g. smectite, chlorite) (Evans, 2000).

The processing and recovery of PGEs from these oxidised ores is complicated due to the remobilisation of Pd alters the Pt:Pd ratio which is the only parameter matrix currently been used as an indicating factor during the processing period. During the flotation process using the current flotation circuit low recoveries are achieved, therefore various hydrometallurgical, pyrometallurgical, and future technologies may provide alternate solutions (Evans, et al., 1994).

The extraction of BMS associated with PGM ores in oxidized mineralization has been investigated by Mpinga & Eksteen, (2018) and Becker, et al., (2014). The mineralogy of oxidized PGMs ore from BIC ore is presented in Table 2.1. According to Oberthür, et al. (2003), the feed ore is highly concentrated with orthopyroxene (24.6 wt.%), clinopyroxene (10.3 wt.%), the alteration minerals (10.6 wt.% serpentine, 18.0 wt.% talc, 5.1 wt.% chlorite, 6.8 wt.% amphibole) and the Feoxides/hydroxides (11.0 wt.%).

Table 2.1: Mineralogy of oxidised ore (Becker, et al., 2014)

	Feed	Tail: AM 28, X, CMC	Tail: AM 28, X, NaHS, CMC	Tail: X, CMC	Tail: X, NaHS, CMC
Base metal sulfides	0.2	0.2	0.2	0.3	0.2
Olivine	3.8	2.1	2.3	2.4	2.3
Orthopyroxene	24.6	30.5	31.6	31.9	33.2
Clinopyroxene	10.3	11.2	10.6	10.7	10.8
Serpentine	10.6	9.2	9.4	9.4	8.8
Talc	18.0	16.6	15.3	14.8	14.5
Chlorite	5.1	5.7	5.3	5.1	5.3
Amphibole	6.8	5.5	6.1	6.1	5.9
Plagioclase	4.4	1.8	2.3	2.4	2.0
Epidote	1.2	2.0	1.9	1.9	2.2
K-feldspar	0.1	0.1	0.1	0.1	0.2
Mica	0.9	1.0	1.0	1.0	0.9
Calcite	1.0	1.0	0.8	0.8	0.9
Quartz	0.3	0.4	0.4	0.5	0.4
Chromite	0.8	0.9	0.8	0.8	0.8
Fe oxides/hydroxides	11.0	10.8	10.9	11.0	10.6
Other	0.9	1.0	0.9	1.0	0.9

Other valuable minerals beside BMS are low in a concentration such as plagioclase content representing 4.4% on the feed as compared to other Merensky ores (Evans, 2000). This low concentration may be due to the mining cut and the processing of ''silicate reef'' compared to pure Merensky reef rather than the effect of oxidation.

2.5 Flotation of Oxidised Ores

Vast mining and processing of Merensky ore have let to depletion of BMS and PGE bearing minerals. More than 60% of PGE production from BIC originates from UG2 ore due to diminishing, socio-political environments in most primary PGM producing countries have triggered interest in exploring the recovery of these BMS from oxidized ores. The processing of this oxidized ore sometimes contains relatively high marketable PGM values which have proven to be more difficult to process by conventional metallurgical practice which involves grinding, milling, froth flotation into a sulphide concentrate, smelting and matte production and chemical refining (Sefako, et al., 2017).

The main technical challenge when processing oxidized ores by conventional flotation techniques is that the performance of the flotation circuit is very poor due to characteristics of the ore. The flotation of such ores results in very poor recovery, yield and grade of the final product (Becker, et al., 2014). A lot of research is still ongoing on the flotation of weathered ore. At the present, several operations which are mining and extracting PGMs from such ore have proved uneconomic due to low recoveries

($\ll 50\%$) achieved by conventional metallurgical methods (Oberthür, et al., 2013) . The other technical challenge with oxidised ores is low grade. This leads to the treatment of large volumes of relatively low-grade material containing PGMs. The procedure on its own is not environmentally friendly and not economically viable in a wide variety of mineral forms that cause hydrometallurgical processes to be inappropriate (Oberthür, et al., 2003).

Previously these ores were left in left in-situ, stockpiled or discarded as waste because of weathering (Oberthür, et al., 2013). Some reach has been conducted with the primary aim of improving flotation recoveries of oxidized PGM ores by employing techniques such as (1) controlling the content of sulfidation before the commencement of flotation (Kalichini, et al., 2017). (2) Employing acid pre-introduction process on the ores before flotation (this process is not economically viable) and (3) Application of different flotation reagents, to improve mineral selectivity present within the ore.(Becker, et al., 2014).

Other external technical factors contributing towards poor overall flotation performance of weathered ore can be attributed by two major factors: poorly targeted mineral liberation and the effect of material alteration. An example is on Merensky PGM ore which consists of PGMs that show a strong association to the BMS (Schouwstra, et al., 2000) and hence flotation concentrators target the recovery of composite PGM and BMS particles. Under these circumstances, poor PGM liberation (defined by area% PGM in a particle) does not necessarily mean poor PGE recovery, if the PGM are locked in or associated with the coarser-grained BMS. Effect of Mining Pit Depths towards the recovery of PGMs

For ore mined from the top surface of the pit area and processed through froth flotation for recovery of PGMs, good performance is met with limited success. The direct influence of weather conditions within the ore body, the quality of material from the top surface normally found to be much more altered and oxidised as compared to the material found buried deep in the ground. The effect of weathering level within the ore body tends to decrease with increasing quality of the ore.
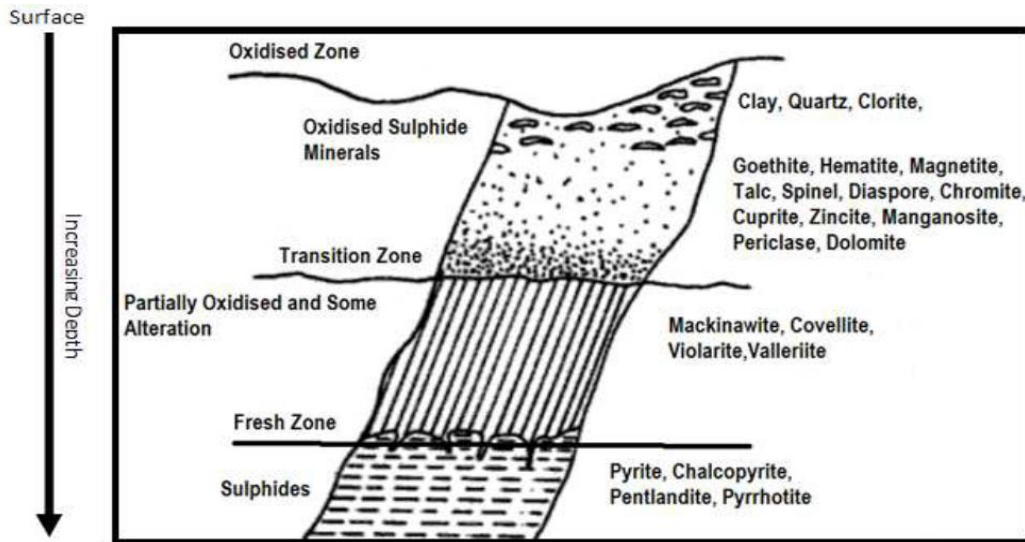
Figure 2.11: Idealised model of ore deposit showing oxidation near-surface and transitioning to secondary sulfide and fresh sulfides at the deep end, adapted from (Oberthür, et al., 2013)

The relationship between the mining pit depths and the effect of weathering is illustrated in Figure 2.11. Oberthür, et al. (2013) developed the cross-sectional area representing a different mining pit depth and modelled the ore deposit showing oxidation near top surface and transitioning to secondary sulfide and fresh sulfides at the deep level. The content of primary sulfide minerals is more dominant with the increasing mining pit depth. Slightly above this level approximately 20m is a transition zone with a secondary enrichment zone and the oxidized zone. The material near the top surface is highly enriched with silicates gangue minerals and clay weathered material (Mungall & Naldrett, 2008).

During the weathering process transformation of minerals take place within the ore body, silicate minerals are transformed into various hydrated phyllosilicates such as talc, serpentine and chlorites. Becker, et al. (2014), identified the naturally floating pyroxene within some of BIC weathered ores. Table 2.2 below shows that altered silicate minerals generally have lower density compared to parent silicate minerals.

Table 2.2: Altered Silicate Minerals (Oberthür, et al., 2013)

Silicate Minerals	Composition	Specific Gravity
Enstatite	$(Mg, Fe)SiO_3$	3.2 – 3.5
Diopside	$CaMgSi_2O_6$	3.2
Plagioclase	$(Na, Ca)AlSi_2O_8$	2.6 – 2.8
Forsterite	$MgSiO_4$	3.2
Hornblende	$(Ca, Na)_{2-3}(Mg,Fe,Al)_5Si_6(Si,Al)_2O_{22}(OH)_2$	3 – 3.4
Lizardite	$Mg_3Si_2O_5(OH)_4$	2.5 – 2.6
Talc	$Mg_3Si_4O_{10}(OH)_2$	2.7 – 2.8
Chlorite	$(Mg,Fe)_3(Si,Al)_4O_{10}(OH)_2$	2.6 – 3.3
Quartz	SiO_2	2.65

The trend distributions of Pt and Pd from pristine to oxidized PGE ores are presented in Fig 2.12. The relationship between the two valuable metals during the weathering process, Pd normally leach out. The metal is much more mobile as compared to Pt during the weathering process, which makes the recovery process more complicated. The ratio between the two metals is used during the flotation process (Cawthorn, 1999; Oberthür, et al., 2003). Fe-sulfide minerals oxidize much more quickly and mainly give the relatively thin layer of Fe(III) oxyhydroxide, perhaps in the form of FeOOH.

During the weathering process identified by (Newell, et al., 2006), the pyrrhotite surface is highly exposed and oxidizes more rapidly as compared to other sulfide minerals present within the ore such as pentlandite.

Another interesting characteristics observed, even though pentlandite is less responsive to oxidation, the mineral is highly enriched with nickel hydroxide ($Ni(OH)_2$) and nickel sulfate (NiSO4) in the Fe(III) oxyhydroxide (FeOOH) - dominated over the layer. Data represents the relationship between oxidation and alteration on the mineral sulfide ore. The rate of oxidation on the mineral surface is entirely consistent with newly developed thermodynamic calculations made on the relative stabilities of pentlandite, violarite, nickel hydroxide, nickel sulfate, and iron oxyhydroxide (Bulatovic, 2010).

Nickel sulfate during the weathering process normally relocates from the mineral surface area and leaves iron (III) oxyhydroxide behind. (Becker, et al., 2014). Moreover, sulfide minerals undergo anodic oxidation in time when they are exposed to atmospheric conditions or highly enriched dissolved oxygen. Other BMS minerals such as pyrrhotite argued the susceptibility of pyrrhotite to oxidation compared to other sulfide minerals such as pentlandite, pyrite, and chalcopyrite by comparing relative conductivity of these minerals. Pyrrhotite is a highly metallic conductor and amenable to oxidation than other sulfide minerals which were more like semiconductors" (Bulatovic, 2010).

(Mungall & Naldrett, 2008) , found out that most of the sulfide mineral surfaces differ significantly in hydroxide content. Different layers were identified within the ore body ranging between (5–80 nm) and oxidized fine particles (0.1–5 μm) were also attached to larger sulfide particles or colloidal such spheroidal iron hydroxide that precipitated from solution during the weathering process. According to Cawthorn, et al., (2002) observed that in order for effective flotation process to take place the flotation of sulfide minerals is fully dependent on the exposure of the clean surface of the mineral that can complex with collectors and attach to the rising bubble. The surface oxidation of sulfide minerals as observed by Newell, et al., (2006) can have a significant impact on flotation.

2.6 *Other alternative existing processing technologies for processing and recovery of PGMs from weathered ores*

Other alternative techniques such as direct leaching have also been tested, where sperrylite has been found to be the main residual platinum bearing mineral remaining undissolved (Mpinga & Eksteen, 2018). The valuable targeted minerals within the ore can be recovered in the Intec Copper process through ion exchange, or reduction (i.e. cementation) using recycled metallic copper dendrites Li, et al. (2013) conducted a full investigation on direct leaching process on copper and/or nickel hydrometallurgy or the combination pyro-hydrometer.

However, the exact chalcopyrite leaching mechanism was not proven. The treatment of weathered ore material using the leaching process is still not determined. To date, chalcopyrite leaching has not been widely adopted by industry due to its extremely slow leach kinetics (Li, et al., 2013; Cilliers, et al., 1998).

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2.7   *Role of Modelling in Flotation*

Flotation is a complex physiochemical process involving solid, liquid, and gas phases; therefore, the number of parameters affecting the process is large. These parameters can be subdivided broadly into two groups, *i.e.*, hydrodynamic and surface chemistry parameters. The former include particle size, bubble size, energy dissipation rate, *etc.*, while the latter includes contact angle (θ), potential, Hamaker constants, and surface tension (γ) (Soni, 2013).

Many investigators developed flotation models in the past, most of which are based on the hydrodynamic parameters. Having a first principle model has many advantages, the most important aspects including predictive and diagnostic capabilities. There are some parameters that are difficult to be tested in the experiment without affecting other parameters. For example, changing the pH to study the effect of potentials of particles also affect the potentials of air bubbles as well as the collector adsorption and hence the particle contact angles. A first principle model can easily study the effects of isolated process variables one at a time and, thereby, optimize a flotation circuit. In addition, the model-based simulator can be used to design flotation plants with minimal input from the experiment (Hay & Rule, 2003).

Flotation is a three-phase separation process. Therefore, modelling flotation is difficult simply because of the large number of parameters involved. Furthermore, a bubble-particle interaction involves several different sub-processes, which need to be modelled separately. Many factors and interactions between them need to be considered in the flotation model since all of these can affect the flotation results in different ways (Kirjavainen, 1992). The flotation models or simulators require basic input data from flotation tests, which causes the limitation of applying these models to predict flotation performance.

2.7.1   The value of flotation kinetics

The feasibility study on how ore will respond on the full plant scale, at milling and flotation circuit, data is derived at laboratory-scale batch tests. Under chosen conditions, a flotation rate test is performed which generates a recovery, grade and concentrate mass profile over time. The shape of these profiles and the relationship

between recovery, grade, and mass with time is described mathematically by Kelsall's equation where R equals recovery and t equals time (Sherrell, 2004).

$$\langle 100 - \emptyset|)[1 - \exp(-kf * t] + \emptyset(|1 - \exp)(-ks * t)]\rangle \qquad (2.6)$$

Where, I = fraction k = rate P = Platinum Group Metals (PGMs) G = Gangue

The equation describes fast and slow floating components that relate to material that is easily and quickly recovered at the beginning of the process and material that is not (ie the slow and difficult to recover component). The equation has four unknowns (kinetics), a fast fraction, a fast rate, a slow fraction and a slow rate.

2.8 Summary

South Africa (SA) has a total of three reefs that are mined and treated for the recovery of Platinum Group Elements (PGEs). The three reefs are Merensky, the UG2 and the Platreef. They are all found in the Bushveld Igneous Complex (BIC). Despite having three reefs, most of PGMs are sourced from the UG2 reef (Cawthorn, 1999). Most of the material from this reef is highly weathered and consist of altered material which results in complications during processing and recovery period.

The froth flotation concentration process has been at the forefront in concentrating the PGMs from their perspective ores. Different concentrating circuits have been developed and these are aimed at floating mainly the sulfide minerals. Despite success in floating the sulfide minerals, significant complications are being encountered when it comes to floating weathered ores. Poor recoveries and high chromite grades are achieved when using the current conventional processing circuits. During the weathering process transformation of minerals take place within the ore body, silicate minerals are transformed into various hydrated phyllosilicates such as talc, serpentine and chlorites. Becker, et al. (2014).Other alternative techniques to deal with weathered material such as direct leaching have also been tested, where sperrylite has been found to be the main residual platinum bearing mineral remaining undissolved (Mpinga & Eksteen, 2018). The valuable targeted minerals within the ore can be recovered in the Intec Copper process through ion exchange, or reduction (i.e. cementation) using recycled metallic copper dendrites Li, et al. (2013) conducted full investigation on

direct leaching process on copper and/or nickel hydrometallurgy or the combination pyro-hydrometer.

Since the flotation is a complex physiochemical process involving solid, liquid, and gas phases; therefore, the number of parameters affecting the process is large. Many investigators developed flotation models in the past, most of which are based on the hydrodynamic parameters. Having a first principle model has many advantages, the most important aspects including predictive and diagnostic capabilities. There are some parameters that are difficult to be tested in an experiment without affecting other parameters. For example, changing the pH to study the effect of potentials of particles also affect the potentials of air bubbles as well as the collector adsorption and hence the particle contact angles. A first principle model can easily study the effects of isolated process variables one at a time and, thereby, optimize a flotation circuit. In addition, the model-based simulator can be used to design flotation plants with minimal input from experiment (Hay & Rule, 2003).
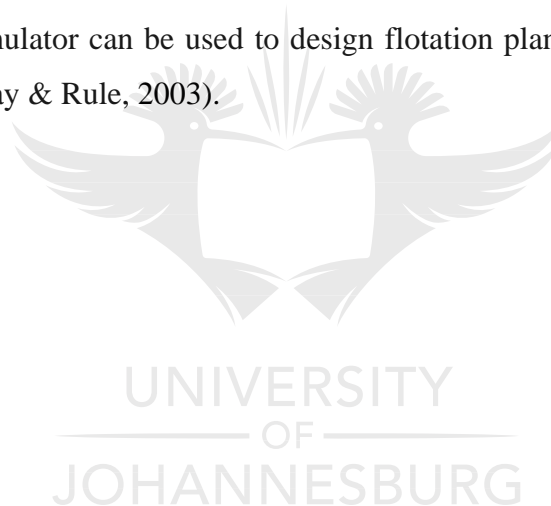
CHAPTER 3

EXPERIMENTAL METHODOLOGY

The following chapter details the steps followed in the characterisation of MG4 PGM ore and test work is done to improve PGMs recovery by flotation. These include sampling, ore characterization, milling test work, quality control, froth flotation experiments, and application of flotation predictor software.

3.1   Sampling:

3.1.1   In situ metallurgical profiling

Representative samples from Cronimet Thaba Mine's open cast mine were drawn initially from the reverse flow drill samples over the mine ore body. The instrument used to collect samples is called a reverse circulation drill (RCD). It consists of a dual walled drill pipe rod on the sides drill bit and produces the whole formation while cutting to the surface of the ore body. The instrument is designed to prevent the upcoming material collected from being contaminated with the material stripped off from the side holes of the instrument. Since it is important to collect representative samples the drill whole position is sealed at the collar, thus the material collected is compelled to go through the drill stem and into the authority at the highest point of the pole.

The operating technique of the instrument is used by applying a high force until the highest point of the drill touches the ground level, up to 1.5m in height below the ground surface. This procedure is done to clear every material of the cuttings from the drill stem before proceeding with the following development. There is an outward classifier through which the air debilitates, which catches shake chippings into the sampler. Total material collected per each RCD opening is then physically riffled to get sizable representative sample amounting to approximately 3 kg per each drill.

The RCD composite representative samples during the collection were drilled in front of mining at a framework of 5 by 5 m intervals. Samples collected from this RC battle were accepted to intently represent the geography of the mining pit area and could give the possible view with respect to how the ore would behave during the processing and

recovery stage. During this campaign 6 from each mining bench (recoil) were composited into one representative sample. During the collection process each sample from the mining zone, RCD test openings were drilled before mining at a matrix of 4 by 4 m division.

Approximately 5tonnes of chromium MG4 ore was collected from Cronimet Thaba Mine both weathered and pristine ore for batch flotation experiments. This material came from the location of MG4 seam as presented on stratigraphy indexed Fig 3.1. Representative samples were then collected at different pit levels, initially starting from the upper, centre and bottom of the pit area for both pristine and weathered ore (940, 935 and 918m above sea level respectively) on the MG4 pit area. Size reduction of each collected material was conducted using mineral processing laboratory stainless steel jaw crusher followed by cone crusher for further size reduction down to -3mm prior to milling test.

Furthermore, the crushed samples were also properly blended to ensure sample homogeneity. After blending the whole composite, the material was then divided equally using a rotary splitter and combined to 1.3 kg by weight for milling test work. A total of 24 samples (each 1.3kg) were prepared of which 6 were used for the establishment of milling test work corresponding to each pit level and the remaining 18 were then used for flotation experiments and for application of the flotation software.

Unit	Thickness
MG4a	0.80
Interburden	1.62
MG4	1.08
Interburden	12.71
MG3	0.82
Interburden	7.31
MG2	0.57
Interburden	12.72
MG1	0.64
Interburden	48.11
LG6a	0.24
Interburden	13.29
LG6	0.78
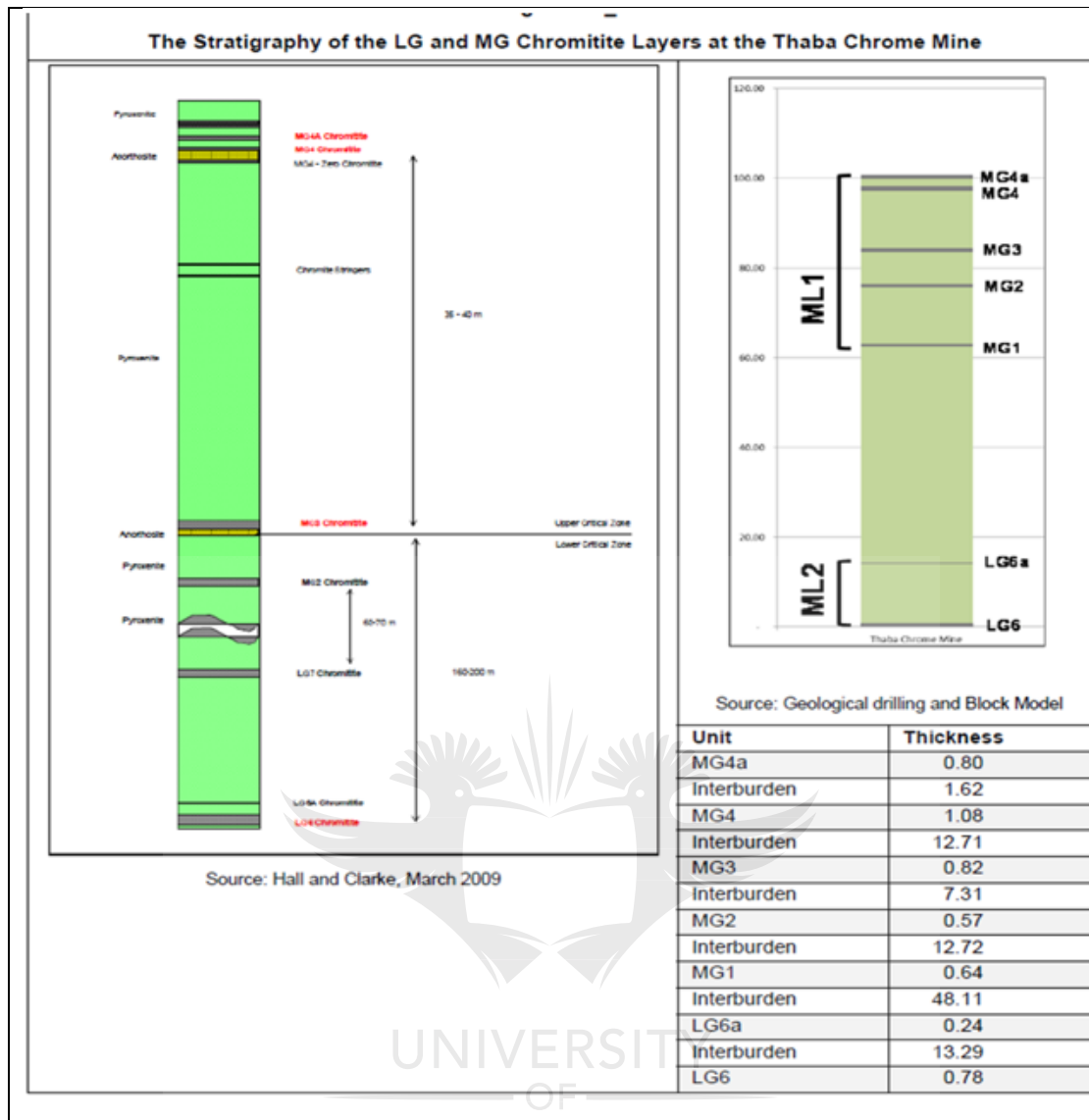
Figure 3.1: Stratigraphy index of Taba Mine

3.2 Ore Characterisation

3.2.1 Chemical composition of the samples

The XRF Rigaku ZSX Primus II spectroscopy was used to obtain the elemental composition of the oxidized and pristine ores. The representative samples were obtained using a rotary micro spinning riffler. A 10.0 g of the sample was pulverized to -75 mm and mixed with 2.0 g of Sasol wax. Pestle and mortar were used to get a homogenous mixture. The mixed sample was transferred to the aluminium cup and pressed with hydraulic presser up to 30.0 bar. The sample was taken to XRF for elemental analysis. This procedure was done for all the samples analyzed with XRF.

3.2.2 Mineral composition of the samples

For different mineral phase determination, Rigaku XRD spectroscopy was used. The representative samples were obtained similarly as for the XRF analysis. The XRD multi-stage sample holder was used for analysis. 3.0 g of the sample was placed and flattened with a glass slide on the aluminium sample holder. The analysis was executed with 20.0 KV with the scan rate of 0.1Hz, at the initial θ angle of 5º to maximum angle of 90º. Surface topography

As the way of confirmation of different phases present in the ores and the understanding of mineral liberation after milling tests, SEM/EDX was used to analyze the samples. Prior to analysis, the samples were mounted on the resin, polished, and coated with pure graphite rods to increase their conductivity as shown in Fig 3.2. The high vacuum mode was used for analysis with the backscatter electron detector. The morphology, mineral liberation, and elemental composition of different phases were determined.
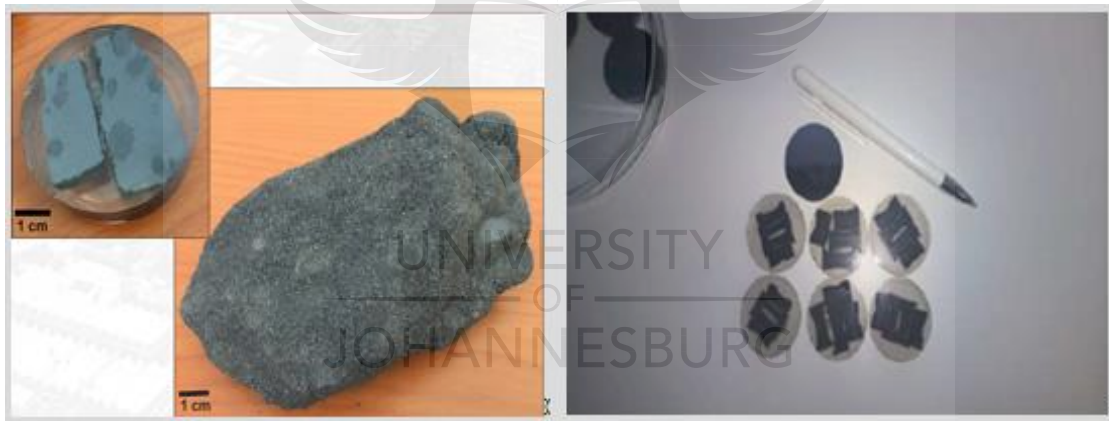
Figure 3.2: SEM Samples, Polished section

3.3 Milling Experiments

A fixed speed rod mill, operating at 85 rpm, was used to produce a suitable grind of 80%-75µm for both weathered and pristine ore. Dimensions of the cylindrical mill were the total internal volume of approximately 9 litres and measured diameter of 300mm. A total of 15 rods made of stainless steel were charged to the mill to produce suitable grind, the milling rods had different diameters between 15-25mm. These were all 290 mm in length. A total of 1.3kg x 6 samples were prepared and milled at 30, 45, 60 and 120 minutes respectively for samples collected. This was done in order to investigate

the hardness of two ores and how long each ore type needed to be milled to achieve the required suitable grind for flotation experiments. The determination of milling curves corresponding to each level of the mining pit area was established for each ore.

3.4   Quality control

Quality control measures were taken into consideration during the performance of batch flotation test work. Batch flotation samples were taken for 60 consecutive days, blended and split into batch samples which were a representative of the mining pit. Metallurgical laboratory standards for sample preparation were always adhered to and avoid contaminations and other possible errors. Fresh reagents samples were taken every morning prior to flotation. All equipment was thoroughly washed before and after use to avoid possible contaminations. Scrubbing consistency was carefully followed by scrubbing after every 15 seconds. Quality control measures have been taken, those include but not limited to:

- Mass Accountability
- Flotation scrapping consistency
- Metal Accountability

3.5   Flotation Predictor

Two crushed feed samples were mineralogy and liberation were studied with flotation predictor. The tests were done on both weathered and pristine ore. Prior to the application of the software, the samples were milled to a PGM flotation grind size of 80% -75 µm, split using a rotary sample splitter to produce twenty unsized polished sections of each sample. The samples were then analyzed by the auto-SEM system to locate and characterize PGM-bearing particles.

The PGM search analyses provided PGM species identification, grain size distribution, liberation characteristics and gangue associations of PGM-bearing particles detected in each sample. PGM mode of occurrence was classified into one of six pre-defined modes of occurrence classes, as described in Table 3.1 and illustrated in Figure 3.2. The PGM search examinations gave PGM species recognizable identity, total grain size dissemination of the particles, liberation attributes and the amount of gangue present within the PGM-bearing particles identified in each representative sample. The PGM

method of the event was arranged into one of six pre-characterized methods of event classes, as depicted in Table 3.1 and showed in Figure 3.2.

Table 3.1: PGM-bearing particle characteristics index

PGM mode of occurrence	Description
L	PGM Liberated from gangue
SL	PGM grouped with BMS
AG	PGM particles attached silica mineral
SAG	PGM particles grouped together with BMS and attached to
SG	PGM grouped with BMS and locked within silicate or
G	PGM entirely locked within silicate or oxide gangue
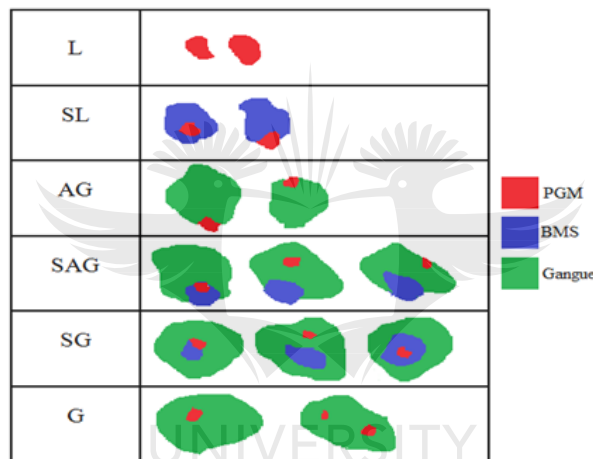
Figure 3.3: PGMs mode of occurrence

PGM grain territories were utilized to ascertain a liberation list for each PGM-bearing molecule. This proportion of PGM grain freedom is determined by separating the zone of the conceivably floatable segment (PGM + BMS) by the all-out zone of the molecule (PGM + BMS + Gangue). The resultant figure runs somewhere in the range of 0 and 1, the last demonstrating either a freed PGM grain or a parallel molecule containing PGM and BMS as it were. Conversely, the freedom file of a PGM grain completely encased inside a BMS-fruitless silicate molecule (low likelihood of flotation) likewise moved toward zero.

The individual PGM-bearing molecule qualities portrayed above were utilized as a contribution to Mintek's PGM Flotation Predictor programming, to give assessments of expected PGM recuperation from each example.

3.6   Batch Flotation experiments (Laboratory test work)

Froth flotation test work was conducted using a 2.5L recently modified flotation cell as presented in (Fig. 3.3). After milling the ore, the slurry was then filtered and transferred to the flotation cell.
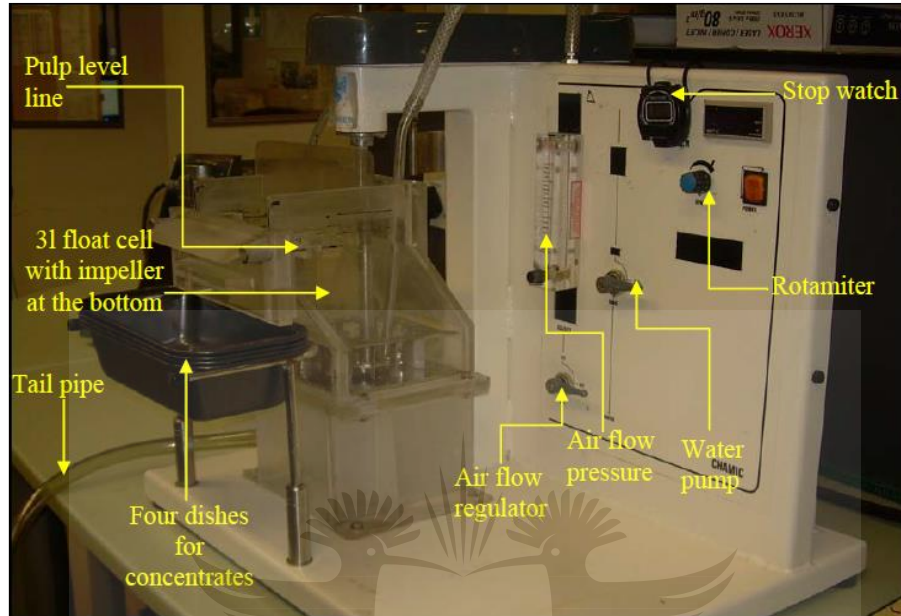
Figure 3.4: Leed Backer Flotation Cell

Direct flotation during the experimental test work using the rougher stage only was used to concentrate the valuable mineral (PGMs). The performance of the flotation test was measured by measuring the PGMs grade in the concentrate, calculating metal recovery and metal loss as follows:

$$Metal \ Recovery = \frac{Cc}{Ff} * 100 \qquad\qquad (3.1)$$

$$Metal \ Loss \ (M) = 100 - R \qquad\qquad (3.2)$$

Where C is the concentrates mass in grams, c is the PGM assay in the concentrates (mass %), F is the feed mass in grams and f is PGM assay in the feed (mass %). Metal loss refers to the metal in the tails and miscellaneous losses (e.g. losses due to spillages, washings, materials left in filter papers, etc). Related measures to grade and recovery include the ratio of concentration, the weight recovery/yield, and the enrichment ratio and separation efficiency (Wills & Finch, 2016).

The reagents used in the flotation of the ore were obtained from SENMIN, South Africa. These are sodium isobutyl xanthate (SIBX) as a collector, Sendep 30D (Sodium Carboxy Methyl Cellulose) - depressant, (and Senfroth 200 (triethylene glycol monobutyl ether; butoxytriethylene glycol) - frother, as presented on Table 3.2.

Table 3.2: Flotation Reagents and dosage

Reagents Name	Type	Reagent dosage
Collector	SIBX	200 g/t
Activator	Copper Sulfate	20g/t
Depressant	Sendep 30D	500 g/t
Frother	Senfroth 200	10 g/t

The concentration of all reagents was 1 % by mass (or 1 gram per 100 ml solution). The Mintek laboratory standard procedure was used to perform flotation tests (Deeplaul & Bryson, 2004).

The rotation speed of the flotation agitator was set and kept constant at 1600rpm, and the samples were agitated until all reagents were properly mixed. Lime was added to regulate and maintain the pH steady at 10.5. Copper sulfate was added as an activator and conditioning was done for 2 minutes followed by 0.2g/kg of SIBX and 0.5g/kg of Sendep 30D depressant and conditioned for 2 minutes. One drop Senfroth 200 was added and conditioned for 30 seconds. The air valve from the compressor was slowly opened until the froth completely covered the cell surface, then the froth was scraped off from the lip of the cell for 10 seconds into the collection container.

This was done several times over a period of 30 minutes to collect representative amounts of concentrates. Each fraction (feed, concentrates and tails) was filtered, dried, weighed and prepared for analysis for fire Assay. During the batch flotation process, representative samples were collected for feed, concentrates, and tailings. All samples were properly filtered, dried and prepared for further processing.

These samples were properly labelled and transferred to stainless steel plates for drying. The samples were all placed into the oven and they were dried for approximately 12 hours at 100°C. After drying, the steel plates were removed from the oven and allowed

to cool down to room temperature. The samples were then pulverized using vertical spindle pulverizer since some material still had small agglomerated particles in a form of small granules.

Nickel sulfide method was used for analyses of 4Es (Pt, Pd, Ru, and Au). This analytical method was selected over other methods because it is considered to be the best due to its capability to analyze the total of 4 precious metals and gold and all this can be routinely done. However, the method has its own limitations, the whole analytical procedure is extremely expensive and is conducted to a critical metal accounting sample. The method uses nickel carbonate which decomposes when subjected to heat to form nickel oxide, which during the smelting process, in turn, reacts with sulfur in the flux to produce the nickel matte. All PGE are dissolved in this matte.

The matte was crushed and pulverized and prepared for wet chemical analysis. The final product (matte) in size was less 100 mesh (100%-120mm) wire mesh screen in order to be efficiently leached, if not valuable minerals would have remained during the filtration process. The pulverized matte was leached using highly concentrated hydrochloric acid. All PGEs were freely liberated at this stage and dissolved in a solution. During this process hydrogen sulfide was generated and it precipitated all PGEs present as sulfides. Furthermore, the un-precipitated PGEs sulfide residues were then filtered and recovered. Finally, the PGE content was measured using a sophisticated Nikel sulfide instrumental technology.

The ICP-OES techniques was used for the analysis of 4E, platinum, palladium, rhodium and gold, as well as base metals such as copper, zinc and nickel, being accurate to 1 ppm. The technique is destructive but it does accurately determine both major and trace elements. In the present investigation, the Inductively Coupled Plasma (ICP-OES) "Quantima sequential by GBC" was used as multi-elements measurement equipment. The generated prill from the fire assay was digested using appropriated medium made of HNO3 and HCl. then subjected to temperatures (plasma) high enough to cause excitation of the sample atoms. Once the atoms or ions are in their excited states, they decay to lower stable states by releasing energy. Light is emitted at a specific wavelength by each element present and these emission lines are measured and used to determine the concentrations of the elements of interest.

3.7 Summary of the Experimental Procedure
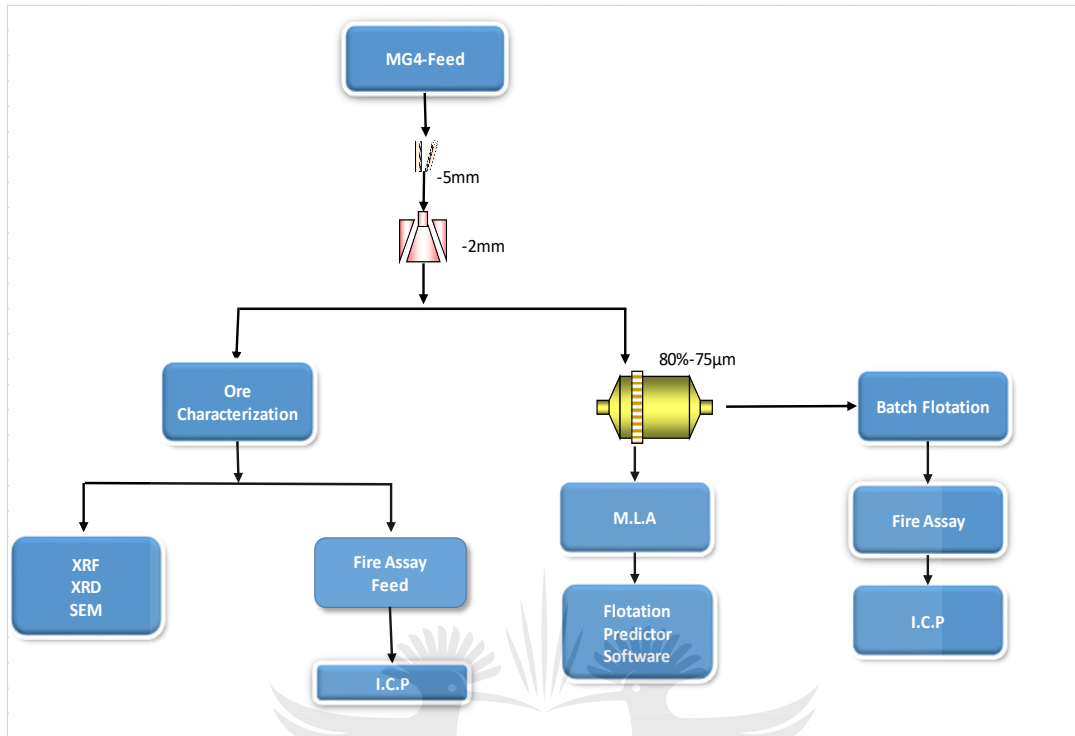
Figure 3.5: Summary of Experimental procedure

CHAPTER 4

RESULTS AND DISCUSSION

This section begins with the presentation of characterization results for both pristine and weathered PGMS ore followed by PSD analysis of the feed from different levels of the ore body. Results from batch flotation experiments that were conducted at different levels of MG4 orebody, flotation predictor software and optimization flotation experiments on weathered ore are also presented.

4.1   Characterization of MG4 ore bodies

4.1.1   Chemical composition of the feed (MG4 Ore)

The chemical composition of the feed was obtained using XRF and the results are shown in Table 4.1. Since PGMs are extremely low in concentrations, they were not detected by this equipment. It is clear from Table 4.1 that the slurry still contains a high % of chromite and is increasing as the mining pit depth increases. The chromite content increased from 26.24-38.75% with an increase in depth from 940 to 918m above sea level. This possesses a serious constrains during the processing period. The high amounts of chromite within the ore which is highly hydrophilic in nature, report to the final product during the flotation process by means of mechanical entrainment (Hey, 1999). This is a serious technical problem for the downstream smelting process, which generally has a constraint of 3% chromite. BMS is also present in small quantities especially in the upper zone of the ore body. This is a typical result of weathering processes were most BMS minerals are leached out (Junge, et al., 2014).

During the characterization, it was identified that the ore body is also dominated and highly enriched with iron content. The impact of oxidation could not only be confined to Fe content since is not stable and the highest observed during the characterization is 31.85%. Research conducted by Oberthür, et al. (2003) confirms that the sulfide minerals oxidize rapidly to give mostly a relatively thin layer of Fe(III) oxyhydroxide, perhaps in the form of FeOOH. During data analysis, the presence of violarite was also identified ($FeNi_2S_4$), $Ni(OH)_2$, and $NiSO_4$. Traces of Fe(III)-S and polysulphide species were also found in the over layers. Apart from that, the ore contains low base metal

content. There is a need to establish the degree of weathering of these base metal sulfides since they are associated with PGMs.

Table 4.1: Chemical Composition of the slurry feed to the flotation circuit

Element	Na	Mg	Al	Si	P	S	Cl	K	Ca	Ti
	Pit Depth 940m *(above sea level)*									
%wt	0.17	9.90	6.85	10.49	0.00	0.08	0.02	0.11	6.58	0.61
Element	V	Cr	Mn	Fe	Co	Ni	Cu	Zn	Sr	Zr
%wt	0.21	26.24	0.30	31.85	0.04	0.25	0.06	0.11	0.03	0.00
	Pit Depth 935 m *(above sea level)*									
	Na	Mg	Al	Si	P	S	Cl	K	Ca	Ti
%wt	0.67	13.84	10.67	20.41	0.67	0.67	0.67	0.01	2.72	0.64
	V	Cr	Mn	Fe	Co	Ni	Cu	Zn	Sr	Zr
%wt	0.35	30.29	0.20	16.31	0.03	0.15	0.02	0.09	0.00	0.00
	Pit Depth 918m *(above sea level)*									
	Na	Mg	Al	Si	P	S	Cl	K	Ca	Ti
%wt	0.20	14.44	11.25	20.60	0.00	0.11	0.00	0.03	2.25	0.89
	V	Cr	Mn	Fe	Co	Ni	Cu	Zn	Sr	Zr
%wt	0.51	38.75	0.20	18.37	0.03	0.15	0.02	0.09	0.00	0.00

4.1.2   Mineralogical composition of the feed

The mineralogical composition of the feed was determined using the XRD machine and the results are presented in Fig. 4.1 and Table 4.2. The analysis was conducted only at the pit elevation of 940m. In the normal stratification, the pristine ore mainly consists of heterogeneous pegmatitic feldspathic orthopyroxene and pyroxenites bounded by narrow stringers of chromite boundaries (Smith & Basson, 2003). The base metal present within pristine and weathered ore differ significantly due to the effect of weathering and may comprise very coarse-grained feldspathic harzburgite or even medium-grained melanorite (Viljoen, 1998).

Pristine ore primarily mainly consists of orthopyroxene and orthopyroxene while weathered ore the formation of secondary minerals occurred because of weathering. The base metal sulfide minerals, such as chalcopyrite, pentlandite, and pyrrhotite which host PGE ores were not identified at all during the characterization period.
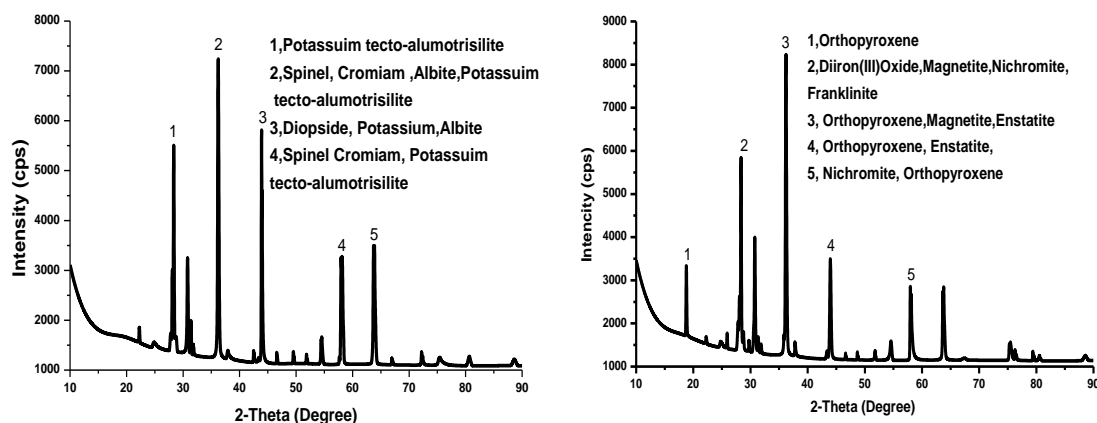
Figure 4.1: Mineralogy composition of the samples, weathered and pristine ore at 940m

The chemical and XRD data reconciliation correlates with the chemical composition data from XRF. Minerals identified in the samples as well as their abundances are presented in Table 4.2. Both elevations are mostly composed of chromite i.e. 24.6 and 30.6 mass % respectively.

The results also show that most of the seams contain a high content of Fe at 19.6%. Low content of base metal sulfides with traceable amount was identified confirming that most of the seams are oxidized at this level (<30m). The major silicate gangue minerals present are orthopyroxene and plagioclase (10-20%), while minor amounts of talc and chlorite are also present.

Table 4.2: Assay reconciliation of Pristine and weathered ore

Elements	Pristine MG4 Ore		Weathered MG4 Ore	
	Chemical Assay Measured	XRD Assay Calculated	Chemical Assay Measured	XRD Assay Calculated
Cr	30.6	29.3	24.6	23.2
Fe	18.4	18.4	19.6	19.4
Ca	0.9	0.3	0.2	0
Mg	6.3	6.4	3.2	3.1
Ti	0.4	0.3	0.8	0.6
Al	8.4	5.9	2.1	2
Si	4.3	5.6	6.2	5.9

4.1.3   Surface topography of the MG4 reef

SEM instrument was used to determine the surface topography of minerals present in the MG4 ore body on different pit levels and the results are presented in Fig. 4.2. and Figure 4.3. The results obtained using the analytical instrument also correlates with data obtained using XRD. Major gangue minerals are silicate oxide minerals, talc, and Chromite. Fine-grained chromite's particles with intercumulus silicates forming ovoid structures. The texture surrounding the voids are pure coarse-grained chromite cumulates. BMS minerals within the ore body appear to be more associated with the coarser-grained chromite particles than with ovoids associated with calcite and silicates in pristine ore. Material again consists of more intercumulus silicates, closely packed which require more energy and expenses to crush and mill during the processing period. All seams displayed mixed chromite textures, thus removal and recovery of chromite
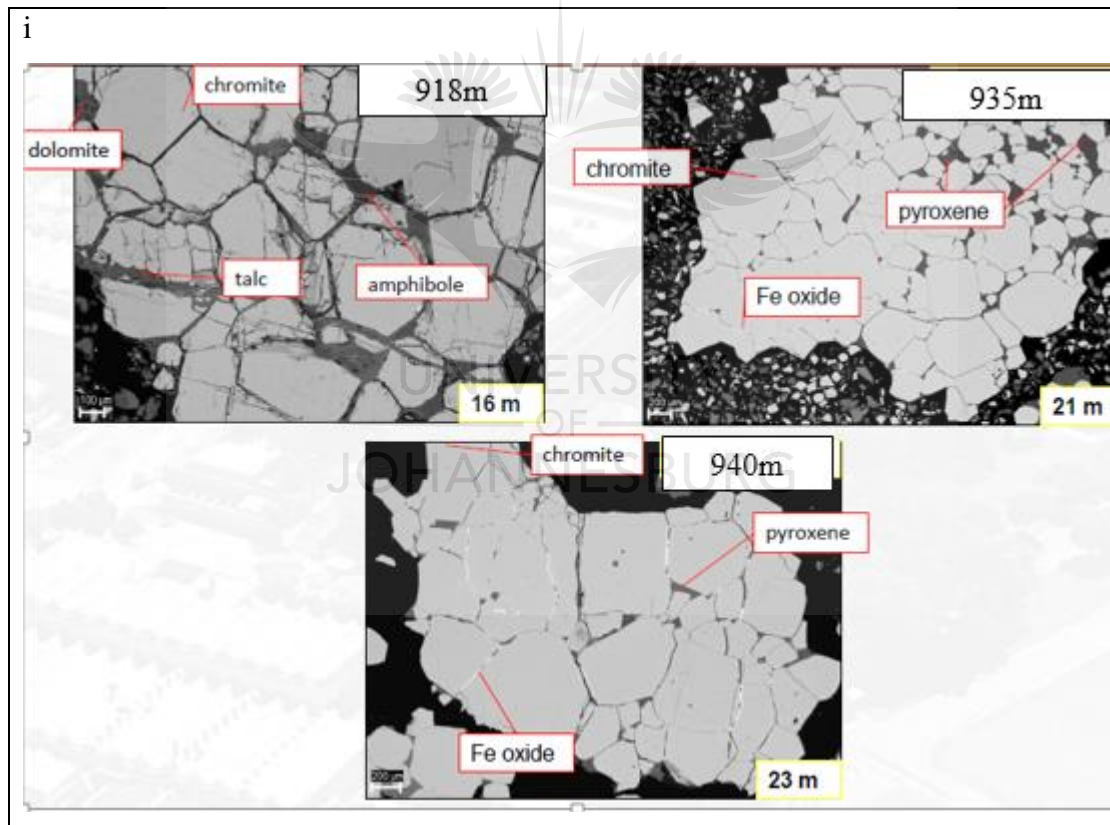
Figure 4.2: Surface topography of Pristine Ore at different depth (m above sea level)

Weathered ore slightly differs in mineralogy based on depth, most BMS minerals were leached out from the top level of the mining reef. Most seams are highly characterized by depletion in BMS and enrichment of alteration minerals such as talc, chlorite,

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serpentine and amphibole (also Fe oxide/oxyhydroxide). The ore is more oxidized and closely packed chromite grain textures, meaning the ore will break easily during the milling process. Processing of such ores with similar characteristics using conventional techniques normally results in low recoveries (Oberthür, et al., 2003).
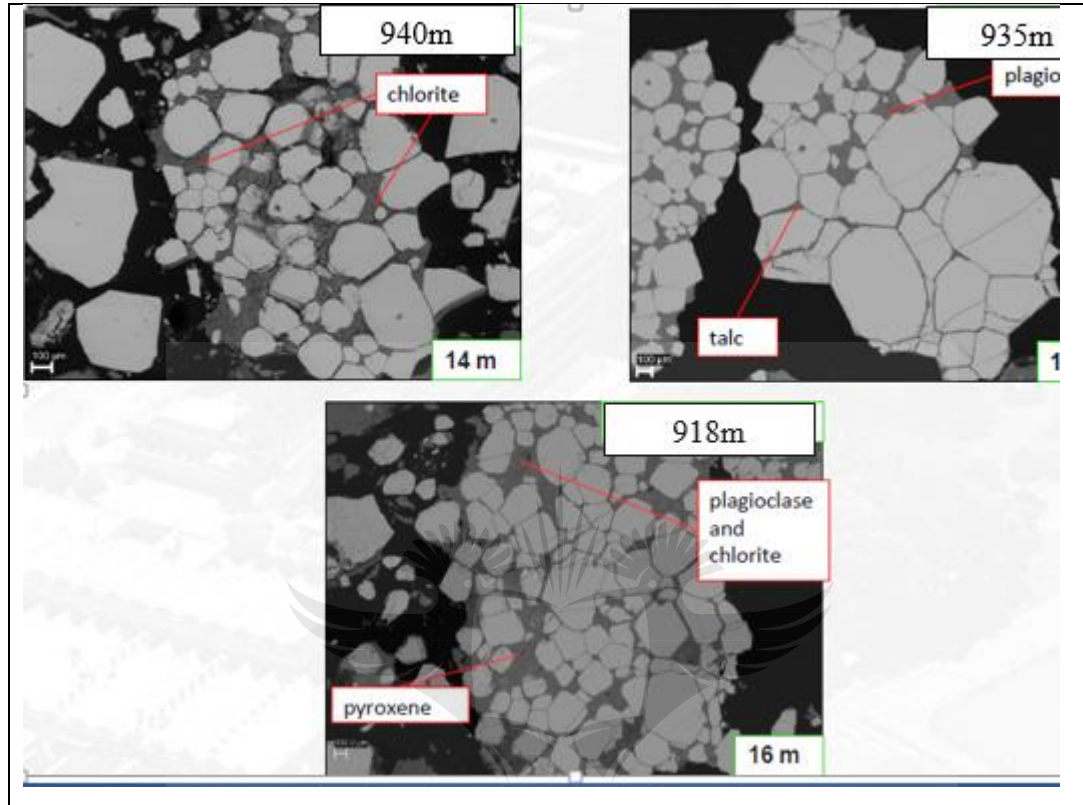
Figure 4.3: Surface topography of weathered ore at different depth (m above sea level).

4.1.4   Bulk Modal analysis

Bulk modal analysis for pristine and weathered ore at 940m pit elevation was determined using the Qemscan instrument and the results are illustrated in Fig. 4.3. The instrument during the analysis uses its total energy dispenser X-ray spectrum and back distribution electron image technique to present minerals present within the ore.

Qemscan results also correlate with SEM results. The overall mineralogy present within the feed after analysis presents major gangue minerals as silicate oxide minerals, talc, and chromite. The reasonable concentration of BMS on pristine ore followed by low concentration of BMS on weathered was identified with approximately 2.08% and 0.64% on pristine and weathered ore respectively.
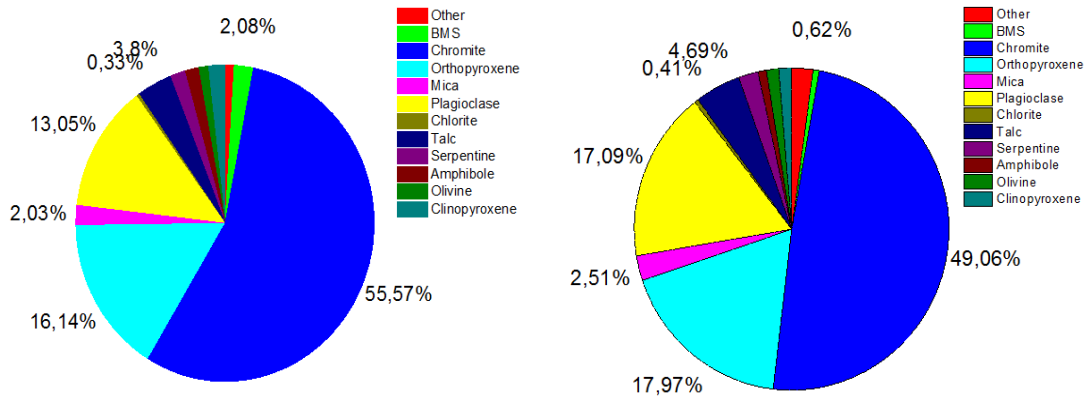
Figure 4.4: Bulk Modal analysis for Pristine and weathered ore at 940m

In addition, other waste minerals present within the ore like amphibole, mica, clinopyroxene, are present in significant quantities. Problematic gangue mineral during the flotation and recovery of PGMs, talc is also present. Its content from pristine and weathered ore is 3.8 and 4.7% respectively. Due to the high content of Chromite and talc in both ores, processing of such ores using conventional techniques normally results in low recoveries (Oberthür, et al., 2013).

4.2   Milling experimental Results
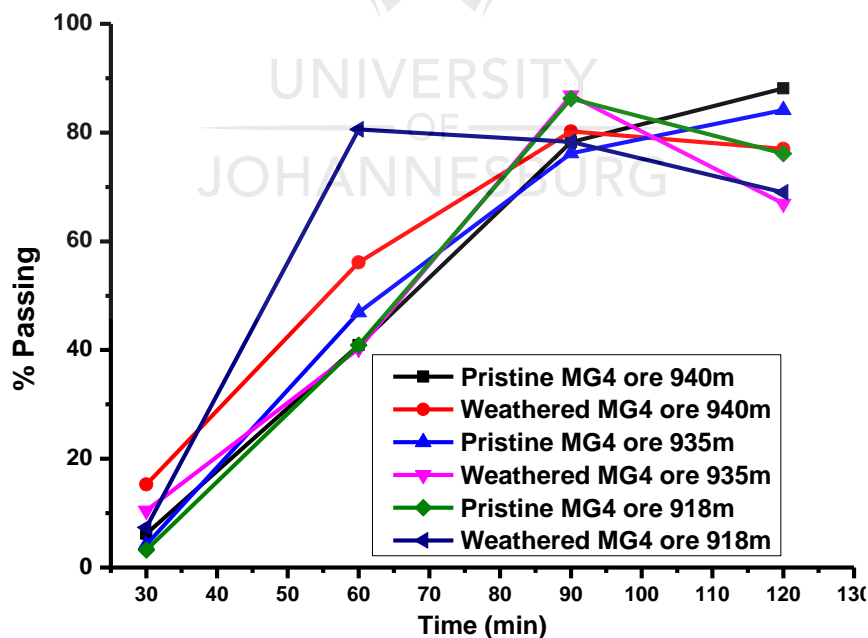
Figure 4.5: Milling test work on MG4 ore at different mining pit depth

The influence of milling time on the particle size distribution of the pristine ore and weathered ore was investigated using a laboratory rod mill and the results are shown in Fig. 4.5.

It is evident that weathered ore is softer than the pristine ore. The results obtained, indicated that the average grind of pristine ore, 80%-75µm, was achieved after milling the sample between 100-120min while for weathered ore was achieved after milling the ore between 60-100min as the mining pit depth increases.

4.3   Quality control

During flotation, it is very important to monitor the quality of the process in terms of mass, metal accountability and scrapping consistency. Fig. 4.6 represents the results on mass accountability which refer to the comparison between the initial sample mass prior to the test work and built-up sample mass after the test work. The mass accountability for both tests (weathered and pristine ore) is within the limits of 95%-100%.
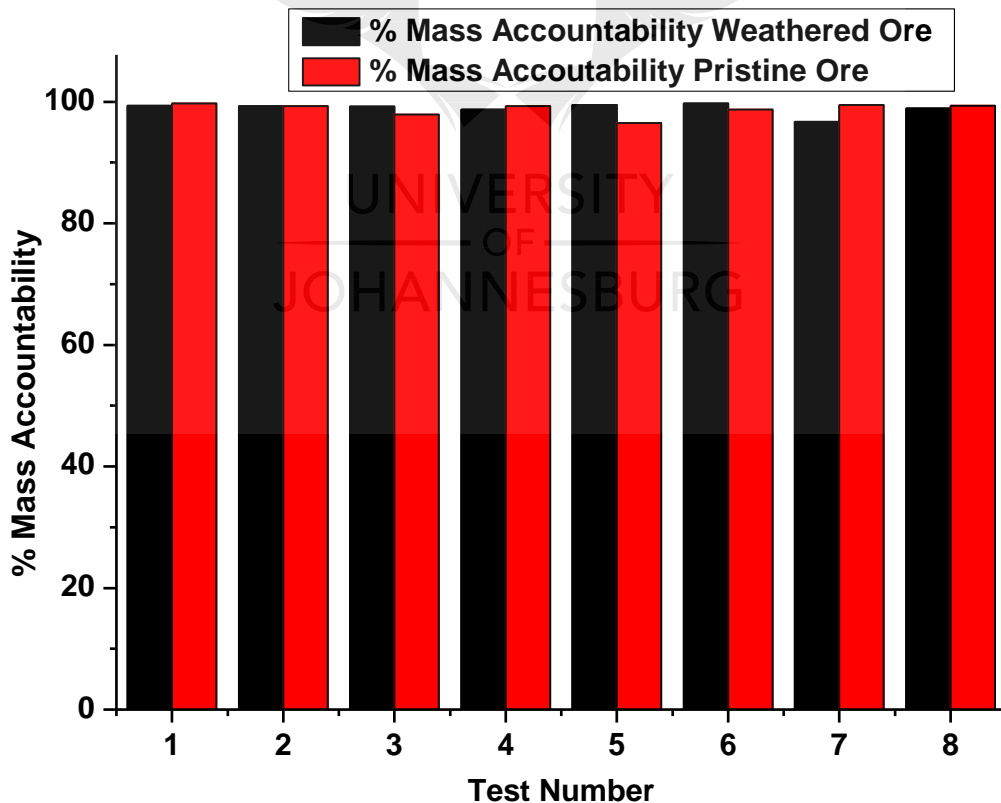
Figure 4.6: Mass Accountability of different flotation experiments

Fig. 4.7, represents the Metal accountability which refers to the comparison between the sample 4E PGE head grade prior to the test work and built-up 4E PGE head grade after the test work. The metal accountability was calculated using equation 4.1 and the results are shown in Fig. 4.5.

$$Mass\ Accountability = \frac{Built\ up\ Mass}{Initial\ Mass} * 100 \qquad (4.1)$$

The mass accountability for both tests weathered and pristine ore is within the limits of 90- 100%, except for test number 5 for weathered ore. The metal accountability was slightly above the limit with a total of 101.45%. This might be due to the generation of metal entrainment during the milling and the flotation process.
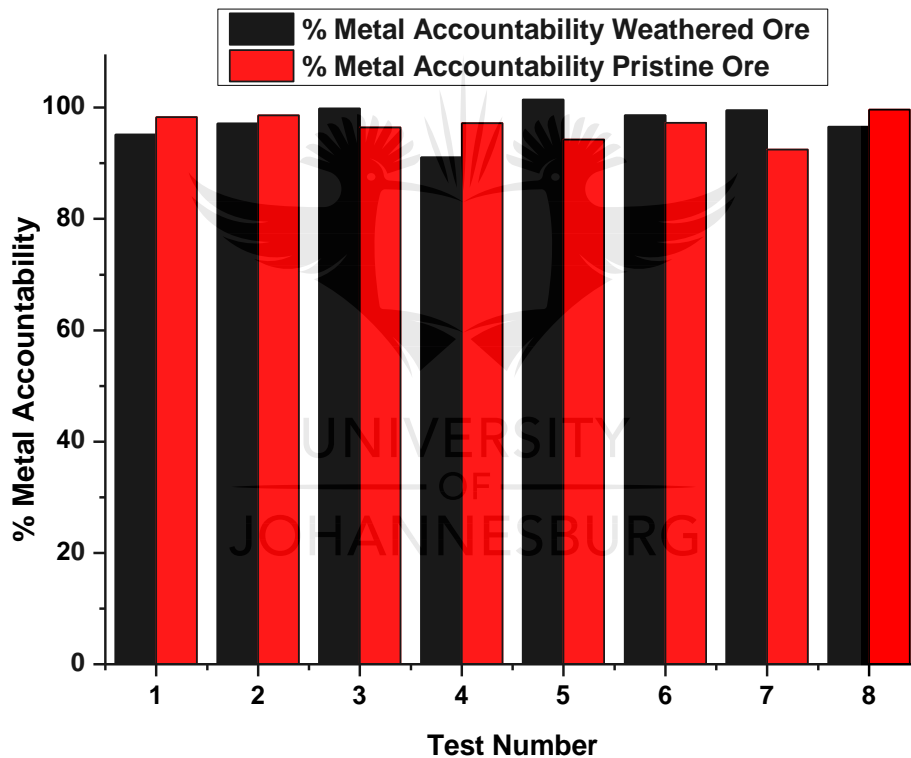
Figure 4.7: Metal accountability of different flotation experiments

Scrapping consistency shows the consistency of producing concentrates between the 3 runs in each test. The % Relative Standard Deviation (RSD)for all concentrates throughout the tests should not exceed 20%. Fig. 4.8 shows that %RSD's for both tests were within the acceptable limits.
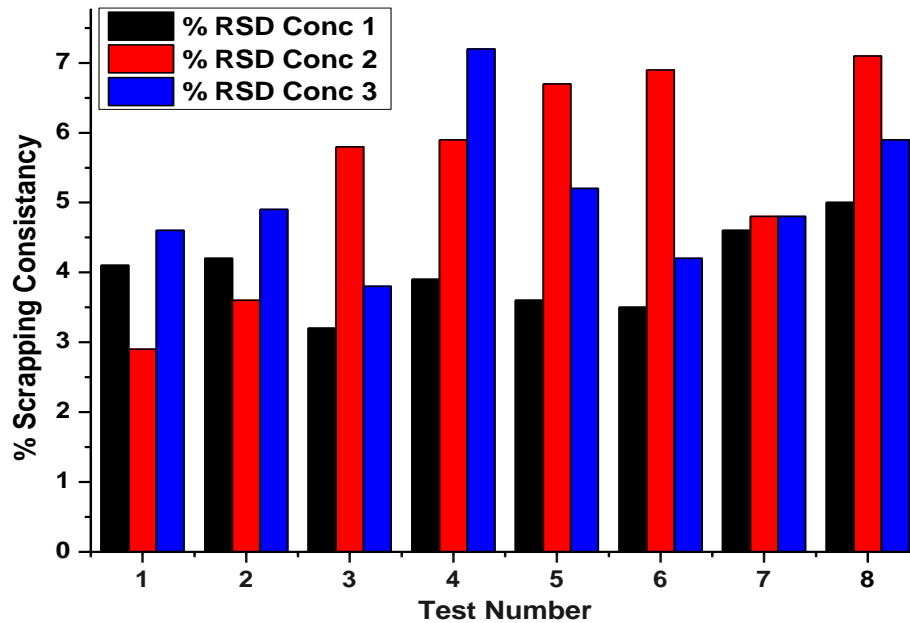
Figure 4.8: Scrapping consistency during different flotation experiments

4.4   Flotation Results

4.4.1   The effect of mining pit depth on PGMs flotation performance

The effect of weathering of PGM ore on flotation performance was investigated by floating the MG4 ore from different pit levels at 940, 935 and 918m above sea level under the same conditions and the results are shown in Fig. 4.9. It was found that the correlation between the MG4 ore mining pit depths and the grade of PGMs concentration grade was trending upwards as mining progresses to deeper depth with the average head grade of PGMs ranging between 1.03-2.49 on weathered ore and 2.03-3.62 on pristine ore. The performance of the flotation process was also improving with recovery and grade as floating material on deeper mining pit depths with the with the average recovery of 11.76%-22.76% and 41.99%-47.07% 4E on weathered and pristine respectively The content of 4E reported to tailings stream during the flotation process was almost the same with the average grade of 0.64-0.78 and 0.67-1.01 for weathered and pristine ore.

The flotation of PGMs from the material mined close surface is regularly met with restricted achievement. Because of weathering around the ore body, the material close surface will, in general, be discovered considerably more modified and oxidized as compared with the material discovered covered somewhere down in the ground. The

degree of oxidation on the ore body will, in general, reduce with increasing mining pit depth (Mungall & Naldrett, 2008).

This can be explained by the fact that flotation of PGMs from the ore mined near-surface is normally met with limited success. Due to weathering, the material near the surface tends to be much more altered and oxidized compared to the material found buried deep in the ground. The extent of oxidation on the ore body tends to diminish with increasing depth (Mungall & Naldrett, 2008).

Weathered ore is mainly characterized by depletion in BMS and enrichment of alteration minerals such as talc, chlorite, serpentine and amphibole (also Fe oxide/oxyhydroxide) (Becker, et al., 2014). Low flotation recoveries on weathered ore are due to the fact that some of the valuable minerals are still locked within the gangue minerals, the presence of high chromite content and availability of talc (naturally floating gangue) (Letseli, et al., 2018).
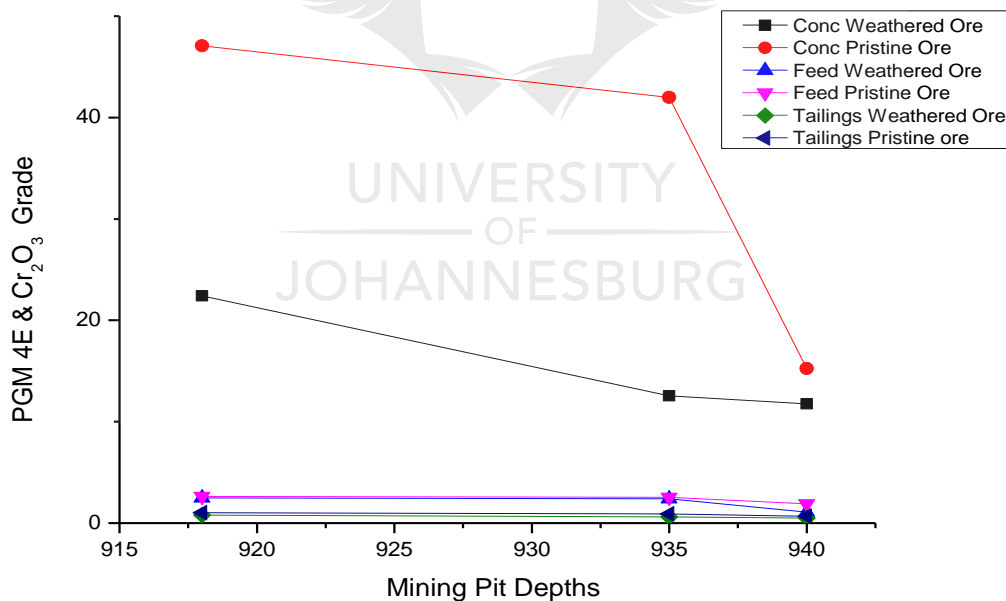
Figure 4.9: Mining Pit Depths m (Above Sea Level)

4.4.2   The impact of weathering on the head grade

The impact on the relationship between the content of Pt and Pd during the weathering process was investigated and the results are illustrated in Fig. 4.10. The material near-

surface tends to be found much more altered and weathered compared to the material found buried deep in the ground from the pit area. It can clearly be seen that Pd is much more mobile than platinum during surface weathering and thus, many weathered PGE ores are depleted in Pd relative to Pt in comparison to their pristine precursors (Cawthorn, 1999). Early attempts to recover Pt and Pd from oxidized ore by flotation resulted in recovery yields of considerably less than 50% (Bulatovic, 2010). The extent of weathering on the ore body between two ores tends to diminish with increasing content of Pd as the mine depth increases.
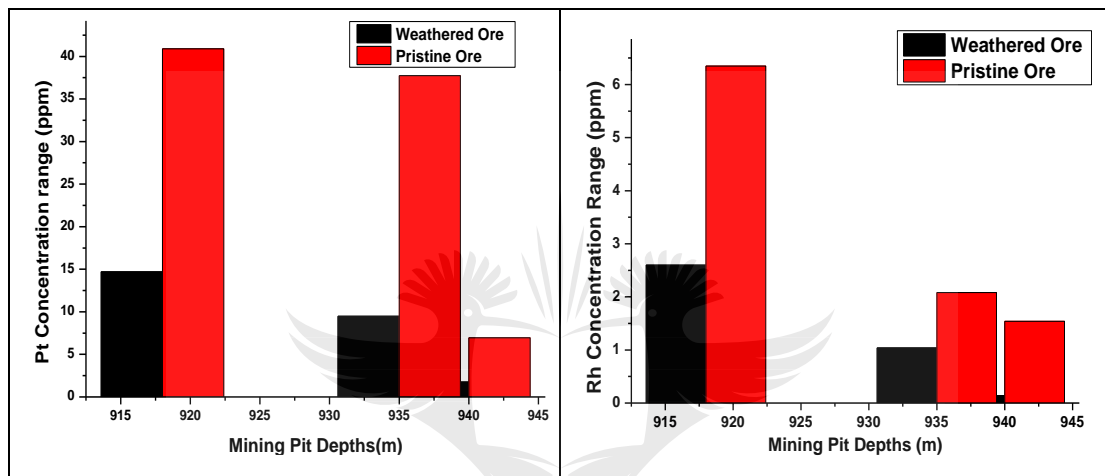
Figure 4.10: Pt and Pd content on different mining pit depths after the flotation process

During the flotation of weathered and pristine ore, the difference in bubble size and colour was identified as illustrated in Fig. 4.11. Both ores were floated under the same conditions. The collector selectively attaches to the surface of the PGMs and hence inhibits the natural hydrophobicity of these particles. Bubble and particles loading on weathered ore, the amount of PGMs mineral present in the froth was low and hence bubble coalescence was big and lighter in colour (Wiese, et al., 2011) . It was therefore decided that the proportion of clusters of small bubbles and their colour during the flotation test would be a useful indicator to decide whether the ore is weathered or not.

Figure 4.11: Bubble sizes and colour during flotation pristine and weathered ore

4.5   Flotation Predictor Software results

4.5.1   PGM search

The PGM bearing species were investigated using Mintek's Flotation predictor software and the results are represented in Table 4.3. Several PGM-bearing particles were detected in the two samples submitted for the PGM search. Mineral identification, grain size, liberation and mode of occurrence data were gathered from each PGM-bearing particle detected. Relative abundances of PGM species are reported in volume percent, as several PGM species occur as solid solution series (eg. PtS-PtPdS-PdS), and as a result, it is not possible to apply a fixed density to these species to calculate accurate mass percent values.

Table 4.3: PGM species detected

PGM Species	Pristine (Vol. %)	Weathered (Vol. %)
PtRhCuS	29.3	0.2
PtS	27.8	33.5
Ru(Os, Ir)S	14.4	21.6
PtFe	14.2	26.2
PtPdS	12.4	2.4
PdHg	0.8	2.4
PdS	0.6	n.d.
PtSb	0.5	2.5
PtPdSb	n.d.	1.7
PtAsSb	n.d.	3.8
PdAsSb	n.d.	2.5
PtPdAsS	n.d.	1.5
PtRhAsS	n.d.	1.3
PdBiTe	n.d.	0.3
Total	**100**	**100**

4.5.2 PGM species

The PGM species detected in each sample are presented in Table 4.4, to simplify the PGM species detected, and to make it easier to compare the species detected in each sample, PGM species were grouped into PGE-sulphides, PGE-alloys, PGE-arsenides, and PGE-tellurides. Laurite (RuS_2), although it is a PGE-sulphide, has been kept separate from the groups in Table 4.4, since it is a low-value PGM that does not contribute to 4E PGE assays.

Table 4.5, PGE-sulphides are the most abundant species in the non-weathered ore, comprising 70 volume % of the total PGMs in the sample. PGE alloys and laurite comprise ~ 15 volume % each, to make up the remainder of the PGMs in the sample. PGE arsenides and PGE-tellurides were not detected in this sample.

PGE-sulfides are less abundant in the weathered ore, totalling 39 volume %. PGE-alloys are more abundant in this sample, at a level of ~ 33 volume %. Laurite levels are ~ 22 volume % in the weathered sample, whilst PGE arsenides comprise ~6 volume %, and PGE-tellurides < 1 volume % of the sample.

Table 4.4: Grouped PGM species detected

Grouped PGM Species	Non-Weathered (Vol. %)	Weathered (Vol. %)
PGE-sulfides	70.2	38.9
PGE-Alloys	15.4	32.8
PGE-Arsenides	n.d.	6.4
PGE-Tellurides	n.d.	0.3
Laurite	14.4	21.6
Total	**100.0**	**100.0**

4.5.3   PGM grain size distribution

"The grain size distribution of PGM grains detected is presented in Table 4.5 and Fig. 4.12. Grain sizes are expressed in equivalent circle diameter (ECD), which is defined as the diameter of a circle with the same area as the measured PGM grain. Note that the grain size data presented have been obtained from measurements on sectioned grains (i.e. in two dimensions only), and although stereological corrections have been applied to the data, grain sizes reported are not comparable to true 3D grain sizes. The 2D grain sizes can, however, be used for comparative purposes between samples".

Table 4.5: PGM grain size distribution in each sample

Grain Size Class (µm ECD)	Non-Weathered		Weathered	
	No. of PGMs	PGM Vol. %	No. of PGMs	PGM Vol. %
0 - 3	31	8.3	35	13.7
3 - 6	65	53.0	62	69.4
6 - 9	17	38.7	6	17.0
Total	**113**	**100.0**	**103**	**100.0**

All the PGM grains detected in all samples fall into the 0 – 9 µm size classes. The 3-6 µm class contains the most PGMs in both samples (53 volume % in the non-weathered ore, and 69 volume % in the weathered ore). The PGM grain size data suggest that PGMs in the weathered ore are slightly finer than those in the non-weathered ore, but since only ~100 PGMs were detected in each sample, these data should be interpreted with caution, due to the potential relatively large statistical error that is possible on

datasets with low populations. The PGMs smaller than 3 µm are likely to be slower floating than those larger than 3 µm.
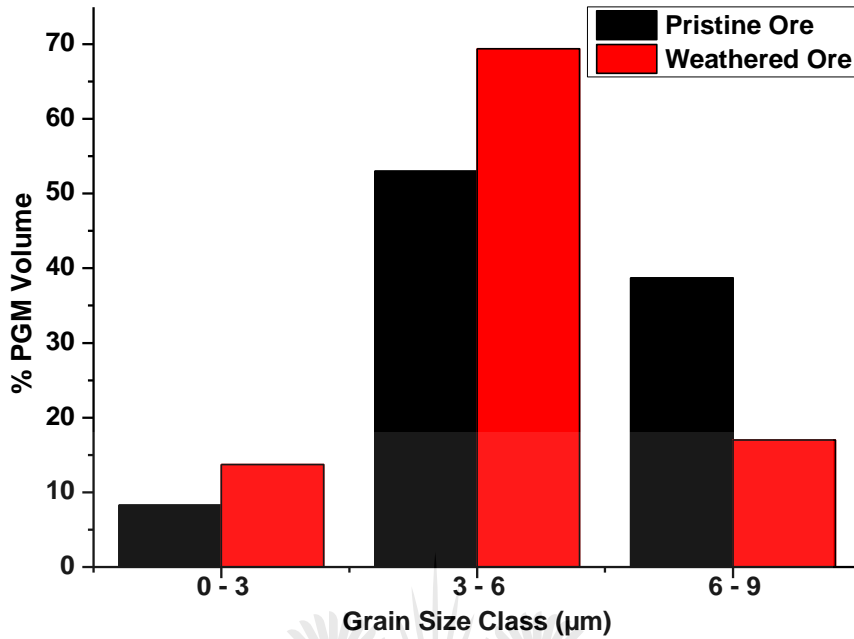
Figure 4.12: PGM grain size distribution in each sample

4.5.4 PGM mode of occurrence

Figure 4.13 represents the number of PGM grains in each mode of occurrence class that is smaller than 3 µm ECD, as these PGMs are likely to be slower-floating than those larger than 3 µm.
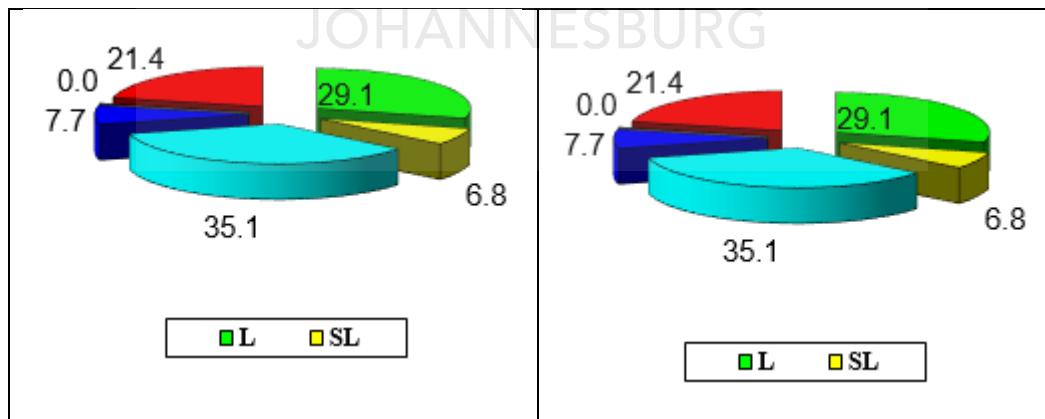
Figure 4.13: PGM mode of occurrence for Pristine and weathered ore respectively

PGMs are better liberated in the non-weathered ore compared to the weathered ore. This is most likely due to intergrowths of altered gangue minerals in the weathered ore, which can affect the liberation of PGM grains during milling.

The number of PGMs associated with base metal sulfides (BMS) is also higher in the non-weathered ore. The likely reason for this is that the BMS in weathered ore has been altered, and as a result is lower in abundance than in the non-weathered ore.

4.5.5   Flotation Predictor results

Mintek's PGM Flotation predictor software was used to estimate potential PGM recovery in two samples. The Flotation Predictor uses the PGM characteristics described in the methodology to estimate potential PGM recovery in each sample. Note that the Flotation Predictor does not consider the PGM species as a parameter when calculating potential recovery and considers all PGM species equally floatable. The recovery values produced by the Flotation Predictor are comparable to rougher flotation recovery rates at the current grind size of the samples (in this case, 80 % passing 75 µm).
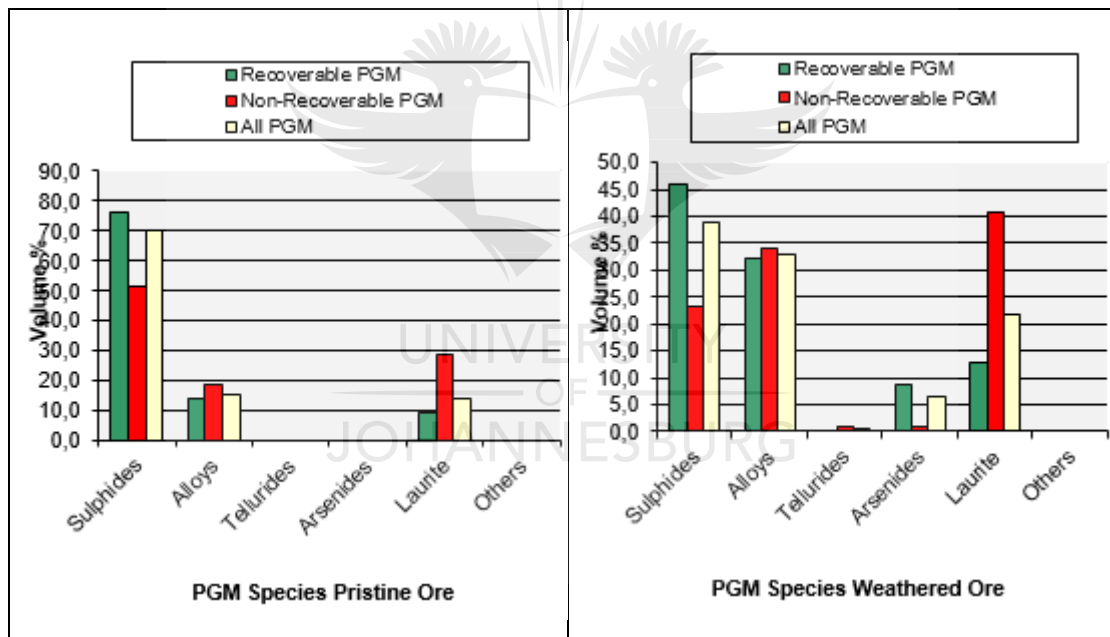
Figure 4.14: Grouped PGMs species for Pristine and weathered ore

Grouped PGM species for non-weathered and weathered ore are presented in Fig. 4.15. As the data set of PGM bearing particles obtained by the auto-SEM is small due to the low PGE grade of typical samples, grouping can help to achieve a mineral balance across the recovery circuit. The metallurgical significance of mineralogical results from automated mineralogy systems can often be difficult to interpret predictor software is run.

This was achieved by performing mineralogical investigations on two representative samples collected on MG4s, weathered and non-weathered ore during the flotation rate tests on each one. The physical characteristics of PGM-bearing particles detected in each of the concentrate samples were noted, and these characteristics were used to establish the cut-off values for PGM size, mode of occurrence and liberation indices for particles that would be expected to report to individual concentrates within the five abovementioned time limits.

"The physical characteristics of PGM-bearing particles detected in each of the concentrate samples were noted, and these characteristics were used to establish the cut-off values for PGM size, mode of occurrence and liberation indices for particles that would be expected to report to individual concentrates within the five abovementioned time limits".

"The link between ore mineralogy and plant performance is not always clearly revealed by traditional liberation, particle size and mineral association result from these systems. The aim of the flotation predictor is to interpret the mineralogical results in a repeatable manner to produce meaningful metallurgical data in an easy to understand format".
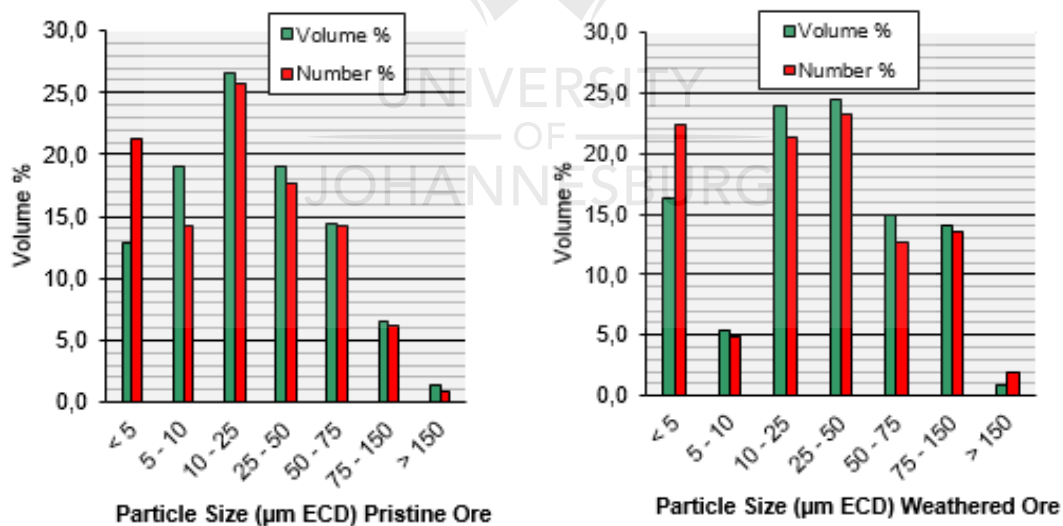
Figure 4.15: PGM distribution by particle size (Pristine and Weathered)

Predicted PGM time vs. recovery curve on non-weathered and weathered ore is presented in Fig.4.16. The Flotation Predictor results show that expected recovery of the non-weathered ore is ~74%, whilst that of the weathered ore is ~69% at the current grind size. The time vs. recovery curves show that the non-weathered ore has a higher

fast-floating component (~60% recovery in 5 min.) compared to the weathered ore (~45% recovery in 5 min).

In both samples, reasons for losses are mainly PGMs that are still locked within gangue particles, and to a lesser degree, PGMs with too low a liberation index to be considered recoverable (i.e. very small PGMs attached to relatively large gangue particles). Longer residence times may lead to the recovery of the PGMs with low liberation index.
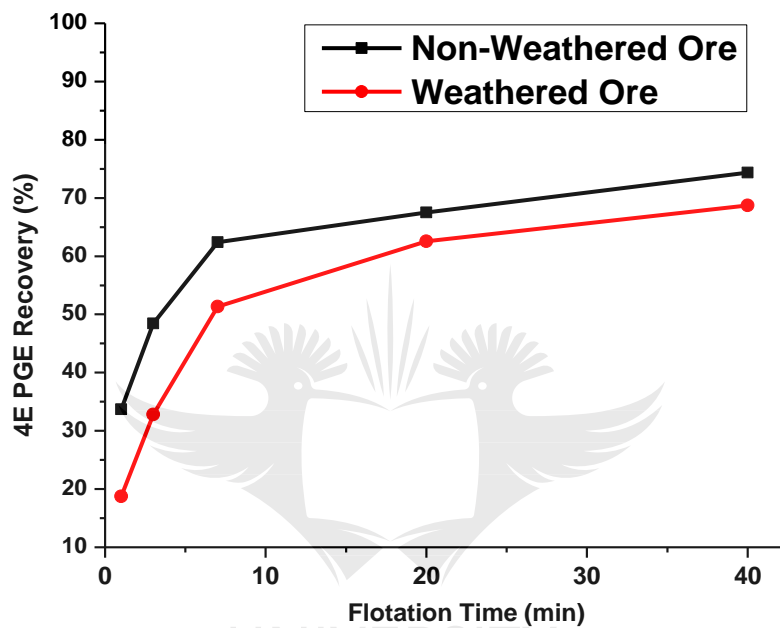
Figure 4.16: Predicted PGM time vs. recovery curve: Weathered ore and non-weathered ore

4.6   Froth Flotation (Laboratory Test Work)

Fig. 4.17, represent the relationship between the grade and recovery of the ore during the flotation process. This test work covers the floatability of two ores: weathered and pristine ore at 918m pit elevation. Two tests were completed at four runs each.

The highest 4E PGE grade and recovery relationships achieved were approximately 41% at 10 g/t and 76% at the concentrate grade of approximately 27 g/t for weathered and pristine ores respectively. Although the recoveries are low for the weathered ore, the major contributor to this was low head grade. This observation further supports the argument that the ore near-surface is somewhat passivated compared to the material deep into the earth (Mungall & Naldrett, 2008).
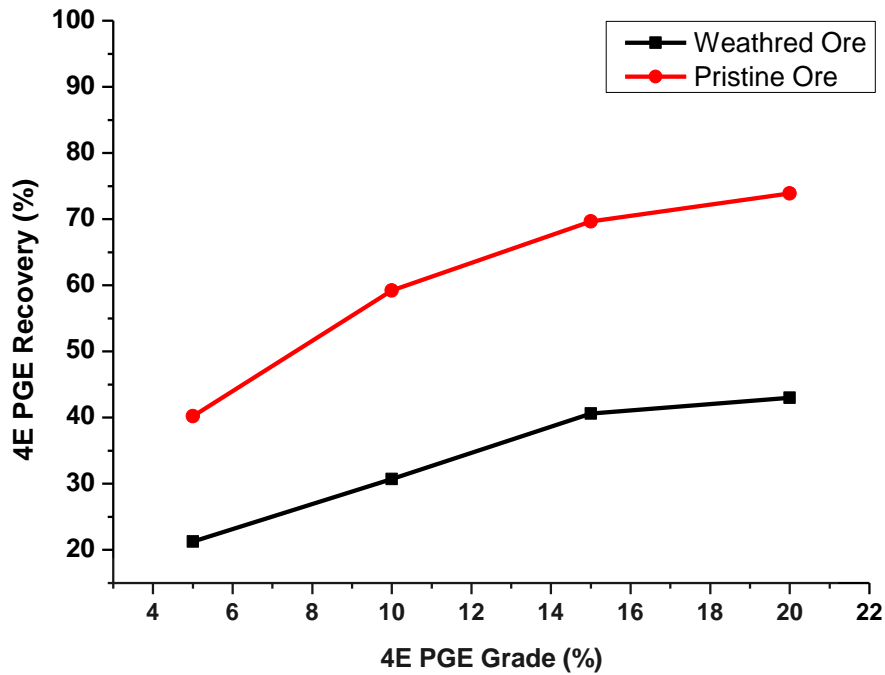
Figure 4.17: Showing 4E PGE % Grade and Recovery Relationships

The mass pull curves depicted in Fig. 4.18, shows the correlation between the mining pit depth and the grade of the ore. The highest mass pull was achieved at the pristine ore due to the nature of the ore. The material had less altered material as compared to weathered ore. It shows clearly that on pristine ore the highest (~76 %) 4E PGE recovery was achieved at relatively high mass pull (4.91 %) and a relatively low (41 %) 4E PGE recovery was achieved at relatively low (4.82 %) mass pull on the weathered ore.

This confirms the observations that were done during the flotation test. The higher mass pull curves are generally shifted higher and to the right as the head grade of the ore is higher. The flotation of PGMs from the ore mined near-surface is normally met with limited success. Due to atmospheric and weather realities around the ore body, the material near-surface tends to be found much more altered and oxidized compared to the material found buried deep in the ground. The extent of oxidation on the ore body tends to diminish with increasing depth" (Sefako, et al., 2017).
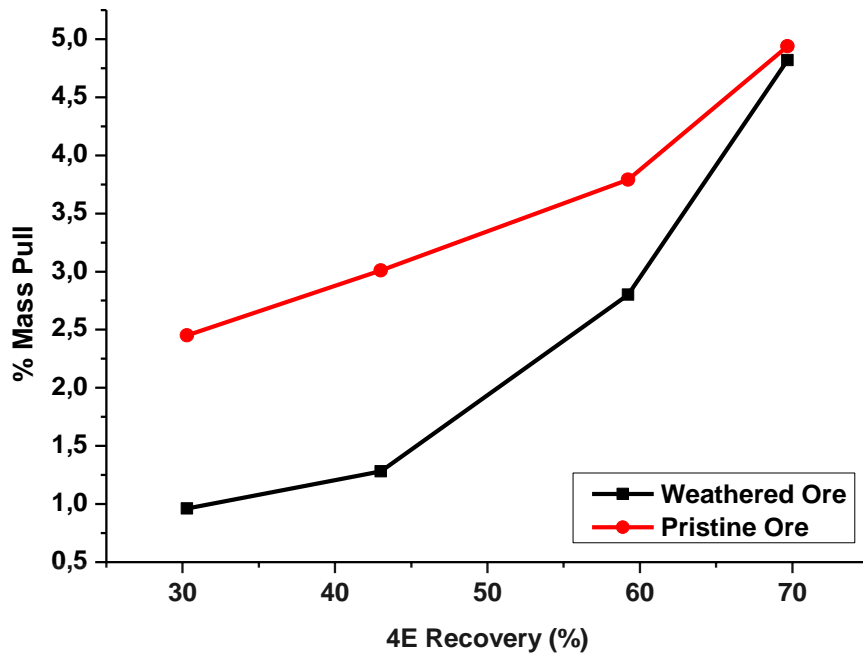
Figure 4.18: 4E PGE Recovery and Mass Pull Relationships

Fig.4.19 illustrates the relationship between the recovery of PGMs and the flotation time between pristine and weathered ore. The flotation was performed in the rougher cell with 918m material and MG4 pit area. The PGMs associated with high-grade material is fast floating minerals that are rapidly recovered in the concentrate. These are generally recovered in the initial period of the test. PGMs associated with low-grade material is slow floating minerals which are difficult to recover.

The curve which is found above others from the start till the end of the flotation time is considered to have relatively high flotation kinetics. It is then evident that pristine ore has valuable mineral sulfides which respond well to flotation within a relatively low time period as opposed to weathered ore. It is also evident that after 13 minutes, the weathered ore was responding very slowly to flotation.

Hence, they are, if at all, recovered later in the flotation test. The high-grade material is largely associated with talc. This is because talc is also a fast-floating gangue mineral. Hydrophobic particles cause bubble coalescence in the. froth phase. This results in the formation of larger bubbles and produces a slower moving, stickier froth (Becker, et al., 2014).
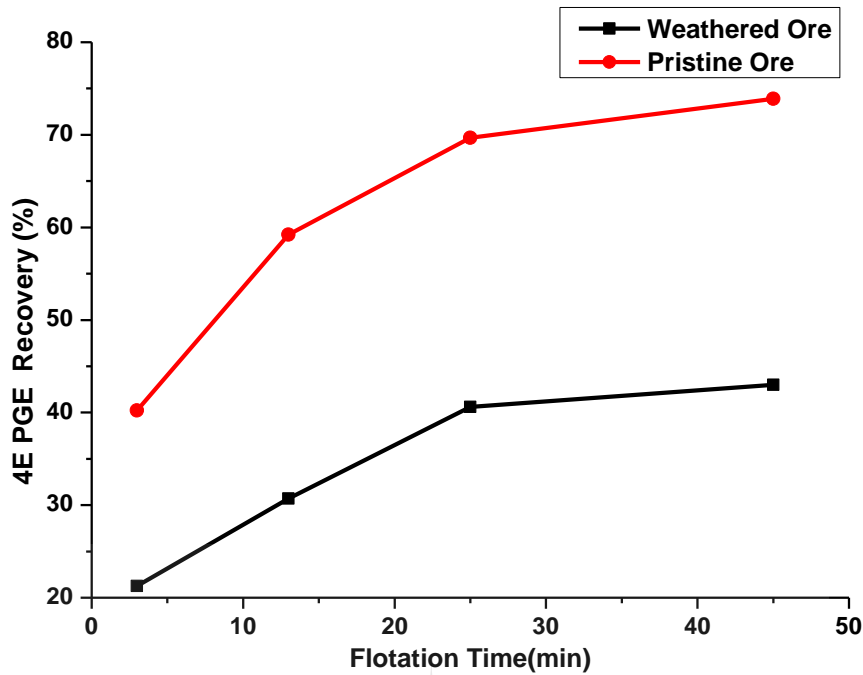
Figure 4.19: Showing the relationship between 4E PGE Recovery and Time (min)

Fig 4.20 represents the relationship between the recovery of PGEs and chromite content on the final product. As stated, the MG4 ore body consists of high chromitite content in the reef ranges between 30% and 90%, and BMS occur in trace quantities, typically on chromitite grain boundaries. PGE's in the UG2 reef are associated with these BMS or occur as single grains. These single grain platinum group minerals (PGM's) occur on the boundary between chromitite grains, between chromite and silica gangue grains or locked in various gangue minerals (Junge, et al., 2014). The content of chromite on the rougher stage was reduced from 32.4% and 35.6% for weathered and pristine ore down to 17.2%-22.21% and 25.6%-25.36% for weathered and pristine ore respectively.

The relationship between 4E PGE recovery and Chromic Oxides for both weathered and pristine ore is almost similar. The pristine ore has slightly lower chromic oxides content, this can be lowered further during the cleaning stages as the test only simulates the rougher stage of flotation.
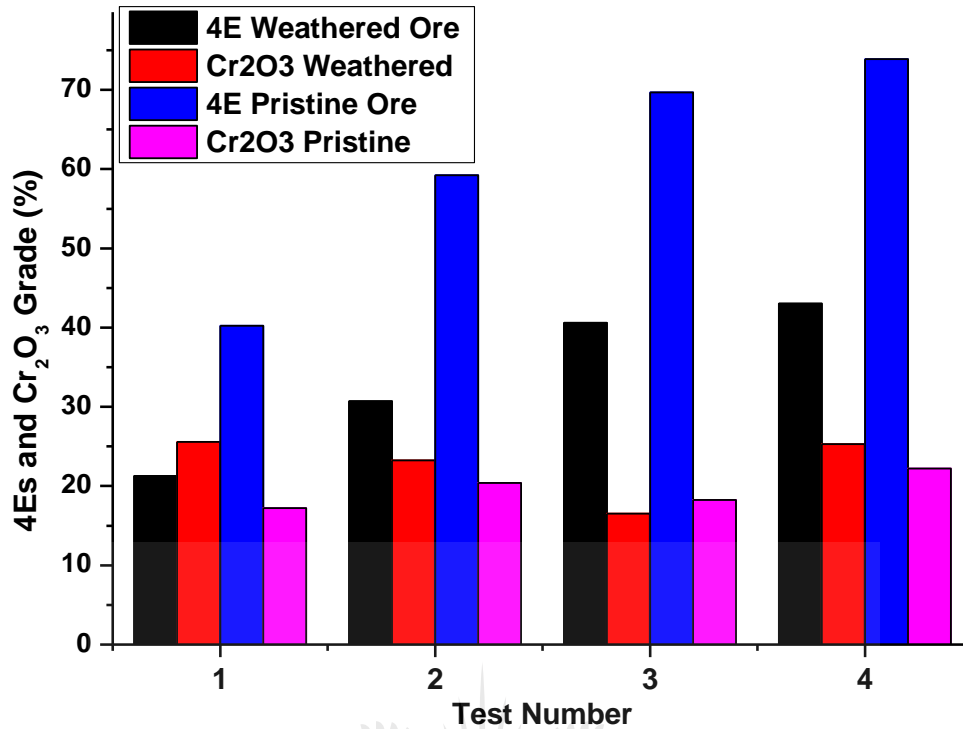
Figure 4.20: 4E PGE recovery and Chromic Oxides grade

4.7   Significance of Results

This chapter has fully elaborated some of the technical challenges currently being encountered when processing weathered material by froth flotation and how these problems can be addressed.

The main objective of the study was to improve the flotation performance of oxidized ore from Thaba Cronimet Mine. The sample was collected at different mining pit depth to correlate the degree of weathering to mine pit depth and monitor the flotation performance. Ore samples from different mine pit depth were characterized using XRF for chemical composition, XRD for the mineral composition of the samples, SEM for Surface topography, Qemscan, and MLA. To obtain the liberation of the PGM containing minerals, milling tests were performed. The hardness, milling curves, and mineral liberation were obtained as the target was 80%-75µm. Mintek flotation predictor software was used to predict the potential recovery of PGMs from both oxidized and pristine. The samples were floated using sodium isobutyl xanthate (SIBX) as a collector, Sendep 30D as depressant and Senfroth 200 as a frother.

The presence of naturally floating gangue material and the presence of chromite content in PGM concentrate is a major concern during the flotation process. Besides reducing the overall flotation concentrate grades, it also affects downstream processing during the smelting process.

According to Jovanović & Miljanović, (2015), some of these materials contain content of PGMs which need modifications on the current conventional circuits to accommodate its modified characteristics due to the weathering process. Based on a solid's recovery of 1%, typical UG-2 final concentrate grades and PGM recoveries are 76% and 47% for pristine and weathered ore respectively. These recoveries were achieved only at the rougher stage, this ensures a reasonable profit can be obtained with the full operating flotation circuit. The Cr_2O_3 content limit for typical concentrates is 2.9%. This constraint can be increased to between 4-9% provided the grade of the product is improved to about 1000 g/t (Xiao & Laplante, 2004). In light of this, it is apparent that the concentrates obtained in this investigation raise certain concerns since the Cr_2O_3 contents were significantly greater than 2.9% on the rougher cells.

The results obtained in this dissertation can be used to solve an industry-wide problem. Industrial concentrator plants consist of many different flotation circuits that treat various ore bodies' i.e. pristine and weathered ore. The results between the flotation predictor software and flotation experiments they also correlate the results obtained from the pristine and weathered ore investigation showed a potential increase in PGMs recovery whilst reducing the chromite content on the final product when re-cleaner stage can be implemented. The purpose of lab test work is to obtain preliminary experimental conditions which are then used as the basis for larger-scale pilot plants. However this thesis highlighted the shortcomings of regular laboratory procedures, particularly in the cleaner cells. Due to insufficient volume of concentrate obtained in the rougher concentrate,

CHAPTER 5

CONCLUSION AND RECOMMENDATIONS

4.8   Conclusions.

The aim of this study was to improve the recovery of PGMs from oxidized ores by investigating the flotation behaviour of MG4 ore from different mining pit depth and floating under the same conditions.

It was concluded that the ore under study was a low-grade PGM ore. The ore contained a reasonable concentration of BMS on pristine ore (2.08%) and a very low concentration on weathered ore (0.64%). The major gangue minerals are silicate oxide minerals, talc, and chromite. More than 70% of the ore body, both pristine and weathered ore is dominated and highly enriched with chromite and iron content. The weathered ore is softer than the pristine ore and it required 60-100min (depending on mine pit depth) to mill weathered ore and achieve the targeted grind of 80%-75μm whilst it took 100-120min for pristine ore from the same pit depth.

The correlation between the MG4 ore mining pit depths and the head grade show that the grade of PGMs was trending upwards (1.08, 2.40 and 2.49 4E for weathered ore and 1.9, 2.55 and 2.62 4E for pristine ore) with increase mining pit depth from 940, 935 and 918m above sea level respectively. The Flotation Predictor results show that the expected recovery of the non-weathered ore is ~74%, whilst that of the weathered ore is ~69% at the current PSD. The time vs. recovery curves show that the non-weathered ore has a higher fast-floating component (~60% recovery in 5 min.) compared to the weathered ore (~45% recovery in 5 min).

The highest 4E PGE grade and recovery that was achieved floating ore from 918m above sea level was approximately 10 g/t at 41% and 27 g/t at 76% for weathered and pristine ores respectively. Although the recoveries were low but increase with the increase in mine pit depth of 940 to 918m respectively, the final tailings grade improved from 0.78g/t to 0.49g/t for weathered ore. It can be concluded that the reagent suite currently used at the plant is not suitable for rougher flotation of weathered ore taking into consideration that the Mintek flotation software predicted recovery of 69%. Apart

from that, current suite is producing concentrates with high chromite grades of 17.2%-22.21% and 25.6%-25.36% respectively for both pristine and oxidized ores.

4.9   Recommendations

- A new reagent suite for flotation of the ore understudy must be investigated with the aim of improving both grade and recoveries of oxidized ores.
- There should be attempts to clean the rougher concentrate and reduce the chromite concentrate in the final product. Due to the high content of chromite within the ore, investigations on the removal of chromite before flotation must be carried out.
- The flotation conditions used in this study should be further investigated and optimize the whole flotation process.

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APPENDIX

MILLING EXPERIMENTS

Table A.1: PSD test work at 940m

Time (min)	Normal MG4 ore 940	Weathered MG4 ore 940
	(% Passing)	(% Passing)
30	6,2	15,26
60	40,92	56,13
90	78,25	80,24
120	88,12	76,98

Table A.2: PSD test work at 935m

Time (min)	Normal MG4 ore 935	Weathered MG4 ore 935
	(% Passing)	(% Passing)
30	4,2	10,5
60	46,92	40,23
90	76,21	86,84
120	84,13	66,93

Table A.3: PSD test work at 918

Time (min)	Normal MG4 ore 918	Weathered MG4 ore 918
	(% Passing)	(% Passing)
30	3,2	7,4
60	40,92	80,6
90	86,25	78,3
120	76,12	68,98

FLOTATION OF MG4 ORE ON DIFFERENT MINING PIT DEPTHS (PRISTINE AND WEATHERED ORE)

Table A.4: Flotation of MG4 at 940m

940m	Grade	Weathered Ore(Ppm)			Pristine Ore (Ppm)		
		Feed	Conc	Tails	Feed	Conc	Tails
Run 1	**Au**	<0.02	<0.02	<0.02	<0.02	<0.02	<0.02
	Pd	0,11	0,1	0,03	0,09	1,28	0,1
	Pt	0,89	1,76	0,57	1,78	6,95	0,31
	Rh	0,03	0,09	0,05	0,1	1,11	0,02
	4E PGE	1,03	1,95	0,65	1,97	9,34	0,43
Run 2	**Au**	<0.02	<0.02	<0.02	0,03	0,08	0,08
	Pd	0,14	0,64	0,11	1,54	1,3	0,06
	Pt	0,95	5,69	0,32	2,4	26,2	0,71
	Rh	0,09	0,94	0,03	0,54	1,63	0,09
	4E PGE	1,18	7,27	0,46	4,48	29,13	0,86
Run 3	**Au**	<0.02	<0.02	<0.02	<0.02	<0.02	<0.02
	Pd	0,11	0,09	0,05	0,11	0,64	0,54
	Pt	0,89	1,78	0,38	0,89	5,69	0,89
	Rh	0,03	0,1	<0.02	0,03	0,94	0,03
	4E PGE	1,03	1,97	0,43	1,03	7,27	1,46

Table A.5: Flotation of MG4 at 935m

935m	Grade	Weathered Ore(Ppm)			Pristine Ore (Ppm)		
		Feed	Conc	Tails	Feed	Conc	Tails
Run 1	Au	0,05	<0.02	<0.02	<0.02	0,13	0,23
	Pd	0,93	1,04	0,03	0,11	1,86	0,14
	Pt	1,99	9,47	0,33	2,05	37,3	0,27
	Rh	0,32	1,59	<0.02	0,23	2,78	0,11
	4E PGE	3,24	12,1	0,36	2,39	41,94	0,52
Run 2	Au	<0.02	0,21	<0.02	0,01	0,48	<0.02
	Pd	0,11	1,22	0,07	0,27	2,08	0,05
	Pt	2,05	7,63	0,29	2,02	36,9	0,63
	Rh	0,23	1,26	0,03	0,12	3,14	0,05
	4E PGE	2,39	10,11	0,39	2,41	42,12	0,73
Run 3	Au	0,05	<0.02	<0.02	<0.02	0,12	<0.02
	Pd	0,13	1,11	0,05	0,11	1,65	0,05
	Pt	1,91	10,3	0,63	2,05	37,3	0,66
	Rh	0,19	1,66	0,05	0,23	2,95	0,05
	4E PGE	2,23	13,07	0,73	2,39	41,9	0,76

Table A.6: Flotation of MG4 at 918m

	Grade	Weathered Ore(Ppm)			Pristine Ore (Ppm)		
		Feed	Conc	Tails	Feed	Conc	Tails
Run 1	Au	0,01	<0.02	<0.02	<0.02	0,48	<0.02
	Pd	0,6	2,6	0,03	0,11	2,08	0,2
	Pt	2,64	14,7	0,57	2,05	36,9	0,59
	Rh	0,59	0,49	0,05	0,23	3,14	0,19
	4E PGE	3,83	17,79	0,65	2,39	42,12	0,98
Run 2	Au	0,01	0,92	<0.02	0,01	<0.02	<0.02
	Pd	0,6	1,6	0,12	0,27	6,35	0,11
	Pt	2,64	23,4	0,36	1,02	40,4	0,89
	Rh	0,59	2,01	0,08	0,12	5,18	0,03
	4E PGE	3,83	27,01	0,56	1,41	51,93	1,03

FLOTATION PREDICTOR SOFTWARE

Table A.7: Liberation Index

L	PGM Liberated from gangue
SL	PGM grouped with BMS
AG	PGM particles attached silica mineral
SAG	PGM particles grouped together with BMS and attached to silicate or oxide
SG	PGM grouped with BMS and locked within silicate or oxide gangue
G	PGM entirely locked within silicate or oxide gangue
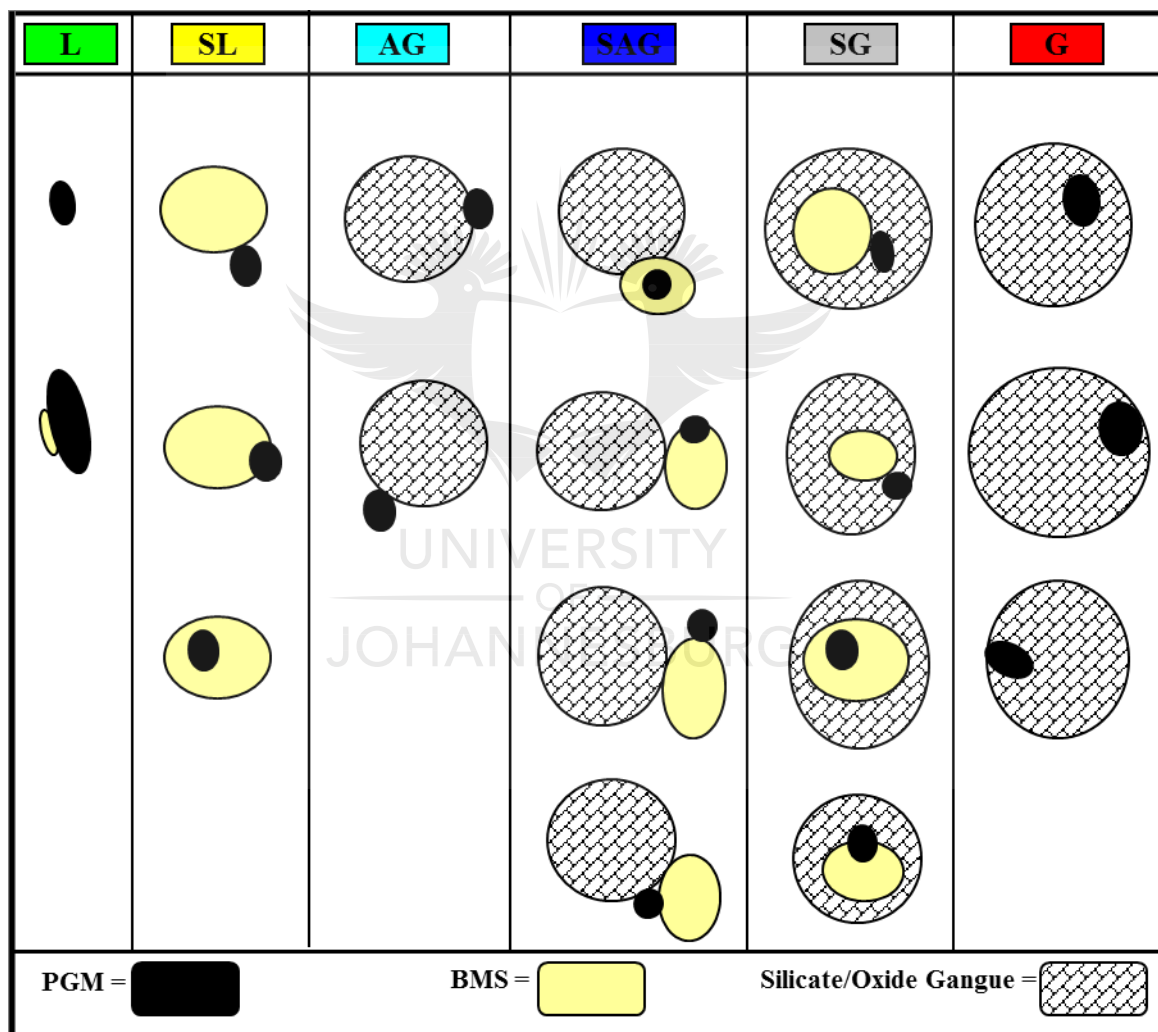
Figure A.1: PGM grain mode of occurrence Pristine Ore

Table A.8: PGM grain mode of occurrence pristine ore

	PGM Vol %	No of PGM grains	No of PGM grains smaller than 3 µm ECD
L	29,1	33	8
SL	6,8	8	3
AG	35,1	40	13
SAG	7,7	6	1
SG	0,0	0	0
G	21,4	26	6
	100,0	**113**	**31**

Table A.9: PGM grain mode of occurrence Weathered ore

	PGM Vol %	No of PGM grains	No of PGM grains smaller than 3 µm ECD
L	19,9	25	7
SL	1,2	2	1
AG	54,9	50	15
SAG	0,3	1	1
SG	0,0	0	0
G	23,7	25	11
	100,0	**103**	**35**

Table A.10: PGM grain Floatability Index Pristine Ore

	Particle Characteristics	PGM Vol %	No of PGMs
Fast Floating	Liberated PGM's >3µm ECD	27,0	25
	Liberated BMS >10µm ECD	5,8	6
Slow Floating 1	Liberated PGM's <3µm ECD	2,1	8
	Liberated BMS <10µm ECD	1,0	2
	PGM's >3µm ECD attached to gangue	31,7	27
	BMS >10µm ECD attached to gangue	6,1	4
Slow Floating 2	PGM's <3µm ECD attached to gangue	3,4	13
	BMS <10µm ECD attached to gangue	1,6	2
No Floating	PGM's and/or BMS locked in gangue	21,4	26
		100,0	**113**

Table A.11: PGM grain Floatability Index weathered Ore

	Particle Characteristics	PGM Vol %	No of PGMs
Fast Floating	Liberated PGM's >3µm ECD	17,5	18
	Liberated BMS >10µm ECD	1,2	2
Slow Floating 1	Liberated PGM's <3µm ECD	2,4	7
	Liberated BMS <10µm ECD	0	0
	PGM's >3µm ECD attached to gangue	48,5	35
	BMS >10µm ECD attached to gangue	0	0
Slow Floating 2	PGM's <3µm ECD attached to gangue	6,4	15
	BMS <10µm ECD attached to gangue	0,3	1
No Floating	PGM's and/or BMS locked in gangue	23,7	25
		100,0	**103**

Table A.12: Distribution of PGM types Pristine Ore

PGM Type	PGM Vol %	No of PGMs	Average ECD (µm)
PtRhCuS	29,3	33	4,2
PtS	27,8	34	3,9
Ru(Os,Ir)S	14,4	17	4,1
PtFe	14,2	15	4,2
PtPdS	12,4	9	5,2
PdHg	0,8	1	4,1
PdS	0,6	3	2,0
PtSb	0,5	1	3,2
	100,0	**113**	

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Table A.13: Distribution of PGM types weathered Ore

PGM Type	PGM Vol %	No of PGMs	Average ECD (µm)
PtS	33,5	32	3,9
PtFe	26,2	29	3,6
Ru(Os,Ir)S	21,6	22	3,8
PtAsSb	3,8	4	3,7
PtSb	2,5	2	4,3
PdAsSb	2,5	2	4,1
PdHg	2,4	2	4,3
PtPdS	2,4	4	3,1
PtPdSb	1,7	1	5,2
PtPdAsS	1,5	1	5,0
PtRhAsS	1,3	2	3,2
PdBiTe	0,3	1	2,3
PtRhCuS	0,2	1	1,8
	100,0	**103**	

Table A.14: Potential PGM Recovery pristine ore

	Volume %	Number %	No. of Particles
Recoverable PGM	74,3	71,7	81
Non-Recoverable PGM	25,7	28,3	32
Total	**100,0**	**100,0**	**113**

Table A.15: Potential PGM recovery weathered ore

	Volume %	Number %	No. of Particles
Recoverable PGM	68,7	63,1	65
Non-Recoverable PGM	31,3	36,9	38
Total	**100,0**	**100,0**	**103**

Table A.16: PGM Distribution by Particle Size pristine ore

Particle Size (µm ECD)	Volume %	Number %	No. of Particles
< 5	12,9	21,2	24
5 – 10	19,1	14,2	16
10 – 25	26,6	25,7	29
25 – 50	19,1	17,7	20
50 – 75	14,5	14,2	16
75 – 150	6,5	6,2	7
> 150	1,5	0,9	1
Total	**100,0**	**100,0**	**113**

Table A.17: PGM Distribution by Particle Size weathered ore

Particle Size (µm ECD)	Volume %	Number %	No. of Particles
< 5	16,3	22,3	23
5 - 10	5,4	4,9	5
10 - 25	24,0	21,4	22
25 - 50	24,5	23,3	24
50 - 75	14,9	12,6	13
75 - 150	14,1	13,6	14
> 150	0,8	1,9	2
Total	**100,0**	**100,0**	**103**

Table A.18: Grouped PGM Species - Total PGM Population Pristine ore

	Volume %	Number %	No. of Particles
Sulphides	70,1	69,9	79
Alloys	15,4	15,0	17
Tellurides	0,0	0,0	0
Arsenides	0,0	0,0	0
Laurite	14,4	15,0	17
Others	0,0	0,0	0
Total	**100,0**	**100,0**	**113**

Table A.19Grouped PGM Species - Total PGM Population weathered ore

	Volume %	Number %	No. of Particles
Sulphides	38,9	38,8	40
Alloys	32,8	33,0	34
Tellurides	0,3	1,0	1
Arsenides	6,3	5,8	6
Laurite	21,6	21,4	22
Others	0,0	0,0	0
Total	**100,0**	**100,0**	**103**

FLOTATION KINETICS

SAMPLE ID	Pt (g/t)	Pd (g/t)	Rh (g/t)	Au (g/t)	Cr_2O_3	4T PGE (g/t)
MOD – Weathered 1 - C1	13,200	2,520	1,730	0,020	25,30	17,470
MOD – Weathered 1 - C2	2,580	0,860	0,620	0,020	23,20	4,080
MOD – Weathered 1 - C3	1,850	0,660	0,580	0,020	23,30	3,110
MOD – Pristine - C1	33,400	5,840	5,110	0,020	17,20	44,370
MOD – Pristine - C2	23,200	3,900	3,390	0,020	20,50	30,510
MOD – Pristine - C3	5,290	1,550	1,400	0,020	20,40	8,260
MOD - Weathered TAIL	0,250	0,080	0,130	0,020	37,50	0,480
MOD - Pristine TAIL	0,340	0,140	0,120	0,020	38,10	0,620
MOD - HEAD NW1	1,970	0,450	0,490	0,020	38,7	2,93
MOD - HEAD NW2	1,560	0,320	0,390	0,020	37,5	2,29
MOD - HEAD NW3	1,810	0,400	0,460	0,020	37,6	2,69
MOD - HEAD W1	0,450	0,120	0,130	0,020	36,8	0,72
MOD - HEAD W2	0,410	0,120	0,100	0,020	37,6	0,65
MOD - HEAD W3	0,390	0,100	0,010	0,020	37,7	0,52

Table A.20: PGE Head Grade

Weathered ore			Pristine ore		
4T PGE (g/t)	4T PGE	Mass	4T PGE (g/t)	4T PGE	Mass
Head 1	**0,72**	498	Head 1	**2,93**	500
Head 2	**0,65**	501	Head 2	**2,29**	492
Head 3	**0,52**		Head 3	**2,69**	
STD	**0,10**		STD	**0,32**	
Average	**0,69**		**Average**	**2,61**	
RSD	**14,82%**		**RSD**	**12,39%**	
weighted Aver	**0,68**		**weighted Aver**	**2,61**	

Table A.21: Flotation Kinetics Pristine ore

Test 1 weathered ore							
Sample ID	Run 1	Run2	Run3	Sum	Average	Stdev	%RSD
Con1	**11**	**11**	**18**	40,00	13,33	4,04	**30,31%**
Con2	**29**	**25**	**22**	76	25,33	3,51	**13,86%**
Con3	**35**	**45**	**50**	130	43,33	7,64	**17,63%**
Tail	**1308**	**1294**	**1299**	3901			
Total	**1383**	**1375**	**1389**	**4147**			
Feed	**1386**	**1379**	**1393**	4158			
Mass Accountability	**99,78%**	**99,71%**	**99,71%**	**99,74%**			

Table A.22: Flotation Kinetics Weathered Ore

Test 2 Non-weathered ore							
Sample ID	Run 1	Run2	Run3	Sum	Average	Stdev	%RSD
Con1	**82**	**76**	**72**	230,00	76,67	5,03	**6,57%**
Con2	**19**	**19**	**21**	59	19,67	1,15	**5,87%**
Con3	**28**	**32**	**28**	88	29,33	2,31	**7,87%**
Tail	**2388,59**	**2458,35**	**2407,02**	7253,96			
Total	**2517,59**	**2585,35**	**2528,02**	**7630,96**			
Feed	**2511**	**2580**	**2525**	7616			
Mass Accountability	**100,26%**	**100,21%**	**100,12%**	**100,20%**			