

Improved gold recovery by accelerated gravity separation.



JAN A. DU PLESSIS
10121323

Presented in partial fulfilment of the requirements for the degree of **M.Sc. Eng
Science**

SUPERVISOR: PROF. F.B. Waanders

CO- SUPERVISOR: PROF. Q.P. Campbell

FACULTY OF **ENGINEERING**

**SCHOOL OF CHEMICAL AND MINERALS
ENGINEERING**



NORTH-WEST UNIVERSITY
YUNIBESITHI YA BOKONE-BOPHIRIMA
NOORDWES-UNIVERSITEIT

2011

Declaration

I, Jan Antonie Du Plessis, hereby declare that this study entitled: “**Improved gold recovery by accelerated gravity separation**”, which was done for the completion of the degree of Magister of Engineering Science, is my own work and has never been submitted to any other university.

A handwritten signature in cursive script that reads "Jan Antonie Du Plessis".

Jan Antonie Du Plessis

Abstract

This project was specifically aimed at using increased acceleration separation, as a method to optimize the recovery of gold in an ore body mainly consisting of hematite.

The specific gravity of gold is much higher in comparison to the carrying material, making it possible to separate gold from other materials such as silica, base metals and their oxides, usually associated with gravitation-gold-recovery processes.

The ore body investigated in this project originated from a mined gold reef containing a large proportion of gold locked inside the gold pyrite complexes. In the mine's processing plant a gold pyrite concentrate was produced by means of a flotation process. The roasting process that followed, oxidized the pyrite to iron oxide (hematite) and sulphur dioxide. The gold particles which were locked up inside the pyrite gold complex were exposed or liberated, allowing the chemicals to penetrate the complex and dissolve the gold. After the cyanide gold extraction process, the material was pumped on to a mine reserve dump, referred to as tailings or tailings reserve dump. The tailings usually contain iron oxides, free gold, gold associated with iron oxides and gold associated with silica, and free silica, commonly referred to as calcine. The gold content on the calcine dump was significantly lower than the flotation concentrate before the extraction of the gold and it was no longer viable for the mine to process the tailings further.

As the volume of the mine reserve dump increased over the years, it became viable to recover the gold in a high volume low grade plant. Several attempts were made to recover the gold in this dump, but due to the high cost of processing and milling the material, it was not done. The norm in the mining industry is that it is impossible to concentrate the gold by means of gravity separation techniques where the average particle sizes are smaller than 50 μm in diameter and upgrading with inexpensive gravity separation techniques was ruled out by the mine, because the average particle sizes were too small.

The dump investigated in this project differed from other reserve dumps in that the main phase of material in this dump was hematite and not silica. A suspension of this material would have different fall-out properties than other mine reserve dumps, because of the hematite's high specific gravity compared to silica. This property of the material birthed the idea that the material will respond positively to high acceleration separation, although the particle sizes were too small for effective upgrading according to the norm in the mining industry.

Using acceleration concentration as a first stage in the gold recovery process the production cost per gram of gold produced could be reduced significantly. Firstly, the volume of concentrated material to be treated in the chemical extraction process was reduced ninety percent and secondly, the gold concentration was increased significantly. If the gold could be concentrated to more than twenty grams of gold per ton, it could be extracted economically with an aggressive chemical processes. This was not possible with low grade material contained in the dump.

The theoretical principle, on which this project was based, was to make use of high acceleration separation to establish separation between the particles associated with the gold, and the particles not associated with gold. Applying a high gravitational force would have an influence on the velocity by which the particles would fall-out in a suspension. As the acceleration force increased the fall-out velocity would also be increased and the particles with higher specific gravity would be affected more.

A factor that was equally important was the particle size and weight distribution. A large hematite particle would compete with a small gold particle due to the similarity in weight. This could cause loss in small gold particles or retention of hematite particles with no gold content.

Very little scientific information was available on the material investigated and in order to assemble a concentration plant setup, the head grade and particle size distribution for both the dump and bulk sample were determined accurately. Thereafter, chemical analyses and mineralogical examination were done on a representative sample of the bulk sample, determining the chemical composition of the material. The results obtained thereof were evaluated and used to configure

a pilot plant. A large bulk sample was processed in the pilot plant and from the analytical results the efficiency could be evaluated.

The results at optimum acceleration forces applied, resulted in a recovery of 5% of the mass, with a gold concentrate of 90 g/t Au, which represented 58% recovery of the gold. The hematite with high specific gravity as main phase positively influenced the high acceleration separation process. It proved that if the specific gravity of particles in a suspension were increased, high acceleration separation could be applied effectively to separate much smaller particle sizes.

Key words

Calcine, gold recovery, acceleration separation, small particle separation, increased specific gravity separation.

Sleutelwoorde

Goudherwinning, versnellingskeiding, kleindeeltjieskeiding, hoë soortlike massa skeiding.

Opsomming

Die ondersoek was spesifiek daarop gemik om hoë versnelling gravitasieskeiding te gebruik as 'n metode om die optimum goudherwinning uit 'n ertsliggaam, wat hoofsaaklik uit hematiet bestaan, te realiseer. Die soortlike massa van goud is baie hoër in vergelyking met die dra-materiaal. Hierdie eienskap van goud maak dit moontlik om die goud met gravitasie-herwinningsprosesse te skei van die ander minerale soos silika, basismetale en hulle oksides, wat gewoonlik met goudherwinning geassosieer word.

Die ertsliggaam wat ondersoek is in die huidige projek was afkomstig van 'n goudrif wat gemyn is met 'n groot gedeelte van die goud opgesluit binne-in goud-piriet komplekse. In die goudherwinningsaanleg was die goud-pirietkonsentraat geproduseer deur middel van die flotasiëproses. Die roosteringsproses wat gevolg het op die flotasië het die piriet geoksideer na hematiet en swaeldioksied. Die roosteringsproses het die gouddeeltjies in die piriet komplekse blootgestel en die goud is in die daaropvolgende sianiedproses opgelos. Na hierdie sianied-goud-ekstraksiëproses is die materiaal weggepomp na 'n myn-afvalbergingshoop. Die afvalstroom bevat normaalweg ysteroksied, vry goud en goud in assosiasie met oksied, silika en vry silika. Daar word algemeen na hierdie materiaal verwys as kalsien ("calcine"). Die goudkonsentrasie na die sianied-goud-ekstraksiëproses was beduidend laer as in die flotasiëkonsentraat en dit is weggepomp na die myn-bergingshoop. Goudherwinning uit die bergingshoop was nie verder lewensvatbaar vir die myn nie.

Soos wat die volume materiaal op die myn-bergingshoop vermeerder het, het dit weer ekonomies begin word om die goud te herwin in 'n hoë-volume-lae-goudkonsentrasie-aanleg. Daar is 'n paar pogings aangewend om die goud uit die hoop te herwin, maar as gevolg van die hoë maalkostes en herverwerking van die materiaal, is dit nie gedoen nie. Die norm in die mynbou-industrie is dat dit nie moontlik is om suksesvol met gravitasie-opgradering deeltjies waarvan die deeltjiegrootte kleiner as 50 μm is, te skei nie. Dit het die myn ontmoedig om die goedkoop gravitasie-opgraderingsmetode te gebruik.

Die hoop wat in die projek ondersoek is, verskil daarin van ander reserwehope, dat hierdie hoop hematiet het as hoofkomponent en nie silika nie. 'n Suspensie van die materiaal sou dus ander uitvaleienskappe hê as gevolg van die hoër soortlike massa van die hematiet. Hieruit het die gedagte ontstaan dat die materiaal positief sou reageer op gravitasieskeiding, alhoewel die deeltjiegrootte volgens die industrie te klein was vir effektiewe gravitasieskeiding.

Deur gebruik te maak van gravitasie skeiding as 'n eerste stap in die goud herwinningsproses kon die herwinningskoste van die proses aansienlik verminder word. Die volume van gekonsentreerde materiaal wat deur die chemiese ontrekkingsproses behandel moes word, is met negentig persent verminder en die goudkonsentrasie is aansienlik verhoog. Indien 'n goudkonsentrasie van meer as twintig gram per ton bereik kon word, kan die goud ekonomies herwin word met aggressiewe chemiese prosesse. Die proses is nie moontlik met die laegraadse materiaal nie.

Die teoretiese beginsel wat ondersoek is, is gebruikmaking van hoë-gravitasie-skeidingstegnieke om skeiding te bewerkstellig tussen deeltjies geassosieer met goud en deeltjies wat nie met goud geassosieer is nie. Die aanwending van hoë gravitasieversnelling op deeltjies in 'n suspensie, sou die snelheid waarteen die deeltjies uit die suspensie uitval, beïnvloed. Soos wat die gravitasiekrag verhoog word, is die uitvalsnelheid ook verhoog en die deeltjies met die hoogste soortlike massa is die meeste beïnvloed.

'n Eienskap wat net so belangrik was, was die deeltjiegrootte-verspreiding. 'n Groot hematietdeeltjie sou moes kompeteer met 'n relatiewe klein gouddeeltjie, bloot op grond van hulle vergelykende massa. Dit kon tot gevolg gehad het dat 'n klein gouddeeltjies verlore gaan en groot hematietdeeltjies met geen goudinhoud in die goudkonsentraat opgevang word.

Die goudinhoud, die deeltjie-grootteverspreiding van die hoop en die hoopmonster is akkuraat bepaal. Chemiese analise is gedoen op 'n verteenwoordigende monster uit die hoopmonster om te bepaal wat die chemiese samestelling van die materiaal is. Die hoopmonster is mineralogies ondersoek om die karakter daarvan te bestudeer. Hierdie bevindings is geëvalueer en daarna is 'n loodsaanleg saamgestel. Die hoopmonster is in die loodsaanleg geprosesseer en die resultate geëvalueer.

Die resultate wat verkry is tydens die optimale gravitasieversnelling, was 5% van die materiaal massa, met 'n goudinhoud van 90 g/t. Dit verteenwoordig 58% van die goud. Die hematiet met hoër soortlike massa as grootste gedeelte van die materiaal, het die gravitasie skeidingproses positief beïnvloed. Dit het bewys dat as die soortlike massa van die deeltjies in 'n suspensie verhoog, verhoog die gravitasie-skeidingeffek en kan gravitasieskeiding effektief toegepas word op materiaal met kleiner deeltjiegroottes.

Acknowledgments

The author of this dissertation would like to express his sincere thanks and appreciation to the following people and organizations, for help support and assistance throughout this study:

- I. We can do all things through Christ, according to the power that is working within us, but Jesus gets all the glory, for out of His hand we received it all.
- II. My supervisor Prof. F.B. Waanders and co-supervisor Prof. C.Q. Campbell, for, apart from their friendship, their outstanding guidance, assistance and inputs throughout the course of this study and beyond.
- III. My family and friends who supported me financially, emotionally and socially.
- IV. Special thanks to Prof John Davidtz for his inspiration and support.
- V. Lastly, I want to thank the North-West University for giving me this study opportunity.

Table of Contents

DECLARATION.....	2
ABSTRACT	3
OPSOMMING.....	6
ACKNOWLEDGMENTS.....	8
TABLE OF CONTENTS	9
LIST OF TABLES	13
LIST OF GRAPHS.....	13
LIST OF FIGURES	14
CHAPTER 1	16
INTRODUCTION	16
1.1. Brief discussion of the background to and the motivation for the procedure followed in this investigation.	16
1.2. The development of a method to increase gold recovery in the calcine dump material	18
1.2.1. Determining the head grade	18
1.2.2. The distribution of minerals and sizes	18
1.2.3. Basic mineralogy tests	18
1.2.4. Increased acceleration forces to enhance the separation of the minerals.....	19
1.3. Objectives of this project	22
1.4. Hypotheses investigated in this project.....	23
1.5. The four steps taken to prove the hypotheses of this project	23
a. Particle size distribution and head grade study:	23
b. Material characteristics study.....	23
c. Pilot plant study.....	23
d. Evaluating the objectives and hypotheses.....	23
CHAPTER 2.....	25
Background and literature survey	25
2.1. Background and literature survey	25

2.1.1. General aspects of gold mining.....	25
2.1.2. The formation of the Witwatersrand gold.....	26
2.1.2.1. Gold extraction.....	28
2.1.2.2. Refractory ore	29
2.1.3. How the material investigated in this present investigation was produced	30
2.1.4. Mineralogical investigation techniques used in this investigation.....	31
2.1.5. The theoretical principle of increased acceleration separation technology used to upgrade the dump	32
Chapter 3	34
EXPERIMENTAL PROCEDURES	34
3.1. Methods and procedures used to determine the material characteristics	34
3.1.1. The experimental procedures to sample the dump and the analytical testing to determine the dump's head grade.....	34
3.1.2. The experimental procedures to analyse the bulk sample.....	34
3.1.2.1. The experiment to measure the particle size distribution	35
3.1.2.2. The experiment to analyse the gold content of the bulk sample	35
3.1.2.3. The experiment to analyse the chemical composition of the bulk sample was done with Induction Coupled Plasma mass spectrometry (ICP-MS).....	35
3.1.3. Particle association was investigated with the Scanning Electron Microscope (SEM) and Energy Dispersive X-ray analysis (EDX).....	37
3.2. The mineralogical experimental procedures performed on the bulk sample	39
3.2.1. The flotation experimental procedures	39
3.2.2. The magnetic separation experimental procedures	40
3.2.3. The hydraulic separation experimental procedures	41
3.2.4. The dense media separation experimental procedures.....	43
3.2.5. The multi-gravity separation experimental procedures	45
3.2.6. The James table experimental procedures	46
CHAPTER 4.....	48
EXPERIMENTAL RESULTS	48
4.1. Results of experiments determining the material parameters	48

4.1.1. Results of the fire assay to determine the head grade of the dump.....	48
4.1.1.1 Discussion of the fire assay results to determine the head grade of the dump	50
4.1.2.1. Results on the particle size distribution	52
4.1.2.1.1. Discussion on the results of the particle size distribution measurements	52
4.1.2.1.2. Results of the gold content or head grade of the bulk sample	54
4.1.2.1.3. Discussion of the results of the head grade of the bulk sample	55
4.1.2.2. Results of the spread in chemical composition of the bulk sample	55
4.1.2.2.1. Discussion of the spread in chemical composition	57
4.1.2.2.2. The interpretation of the bulk sample parameters.....	59
4.2. Photographic images obtained with the SEM showing the different mineral particles and their association to each other	60
4.2.1. Discussion of the photographic images taken with the SEM showing the particles association with each other	61
4.3. Results obtained for the determination of the mineralogical aspects.....	62
4.3.1. Results of the flotation experiment	62
4.3.1.1. Discussion and interpretation on the results obtained for the flotation experiment	62
4.3.2. Results on the magnetic separation experiment.....	63
4.3.2.1. Discussion and interpretation of the results obtained for the magnetic separation experiment.....	63
4.3.3. Results on the hydraulic separation experiment	64
4.3.3.1. The results of the hydraulic classification experiment discussed and interpreted.....	65
4.3.4. Results on the dense medium separation experiment	66
4.3.4.1. The results discussed and interpreted obtained from the dense media separation experiment.....	66
4.3.5. Results on the multi-gravity separation experiment	67
4.3.5.1. Discussion and interpretation of the results obtained from the multi- gravity separation experiment.....	67
4.3.6. Results on the tabling experiment.....	68
4.3.6.1. The tabling experiment's results discussed and interpreted.....	69

4.4. Evaluating the results of the determination of the mineralogical aspects	70
4.4.1. The mass balance of the mineralogical experiments discussed and interpreted	70
CHAPTER 5	72
THE PILOT PLANT EXPERIMENT AND DISCUSSION	72
5.1 The pilot plant configuration.....	72
5.1.1. Experimental procedures used in the operation of the pilot plant.....	74
5.2. Results obtained for the pilot plant experiment	75
5.2.1. Discussions on the results of the pilot plant experiment.....	76
5.2.2. Interpretation of the results from the pilot plant.	77
CHAPTER 6.....	79
CONCLUSIONS	79
APPENDICES	86
Appendix 1.1	86
Ergo report on the size distribution and gold content of the calcine dump	86
Appendix 1.2.....	88
In the Table gold values analysed by Ergo was reported and the average gold content was calculated.....	88
Appendix 1.3.....	89
Calculations on the Itomak acceleration	89

List of tables

Table 4.1: Gold values analysed with fire assay of the dump samples	49
Table 4.2: The size distribution results of the bulk sample and the accumulative mass values passed through the different screen sizes.....	52
Table 4.3: The gold analysis of the head grade from the bulk sample done with the ICP analyser	54
Table 4.4: The chemical composition spread analysed over the particle sizes distribution	56
Table 4.5: The average analyzed values of the main minerals.....	56
Table 4.7: The results from the flotation experiment.....	62
Table 4.8: The results from the magnetic separation experiment	63
Table 4.9: The results from the hydraulic separation experiment.....	65
Table 4.10: The results from the dense media separation experiment	66
Table 4.11: The results from the multi-gravity separation experiment.....	67
Table 4.12: The results of the tabling experiment.....	68
Table 4.13: The table of mass balance done of the mineralogical experiments.....	70
Table 5.1: The results from the pilot plant experiment	76
Table of the results recorded by Ergo using a particle counter to measure the particle distribution in the sample suspension.	86
Table of Ergo gold sampling results documented previously.	88

List of Graphs

Graph 4.1: Graphical depiction of the accumulated mass% of the sample that passed through the different screen sizes	53
Graph 4.2: Graphical representation of the results obtained for the relative Au, Si and Fe contents analysed at different screen sizes	58
Graph of the screen sizes and the percentage material retained.....	87

List of Figures

Figure 1.1: Photographic representation of the Itomak high acceleration separator used in this present investigation	19
Figure 2.1. Geological map of the Witwatersrand super group, showing the locations where the major goldfields are (Frimmel, 2004).....	27
Figure 3.1. Schematic of an ICP-MS system (Skoog <i>et al.</i> , 1998).....	36
Figure 3.2.: Photographic representation of the sample chamber opened on the SEM, used in this present investigation.....	37
Figure 3.3. Schematic illustration of the SEM apparatus (Skoog <i>et al.</i> , 1998).....	38
Figure 3.4. Schematic illustration of the batch flotation cell.....	39
Figure 3.5: Photographic representation of the batch flotation cell used in this present investigation	40
Figure 3.6. Schematic illustration of the Davis magnetic separator.	41
Figure 3.7: Photographic representation of the Davis tube magnetic separator used for the magnetic separation experiment.	41
Figure 3.8: Schematic illustration of the separating principle of the MPC5 hydraulic separator.....	42
Figure 3.9: Photographic representation of the MPC5 concentrator used in this present investigation.	43
Figure 3.10: Schematic illustration of the functional unit in the DMS plant used in this present investigation.	44
Figure 3.12: Schematic representation of a cyclone.....	45
Figure 3.13: Photographic representation of a cyclone multi-gravity separator used in this present investigation.	46
Figure 3.14: Schematic representation made of a James Table used in this present investigation.....	47
Figure 3.15: Photographic representation of the calcine showing separation on the James table taken during experimentation.....	47
Figure 4.1. SEM photographic image of the material showing pyrite surrounded with hematite.....	60

Figure 4.2. SEM photographic image of the material showing gold particle surrounded with silica.	60
Figure 4.3. SEM photographic image of the material showing free gold particle with hematite next to it.	61
Figure 5.1: Diagrammatic representation of how the final pilot plant was configured.	73

Chapter 1

Introduction

In this chapter a general background of the material investigated in this study is given. A brief discussion is made concerning the characteristics of the specific material and how it differs from normal material in waste dumps, which motivated this investigation. An overview of the project and the objectives is given, with reference to the theoretical principles investigated in this project, leading to the formulated hypothesis. Also, a concept recovery process will be discussed and how the pilot plant was configured following the basic metallurgical test work.

1.1. Brief discussion of the background to and the motivation for the procedure followed in this investigation.

It was established that the mine reserve dump investigated in this project, had formed from the effluent from a large flotation plant processing a pyrite gold concentrate. The material still contained some gold, as well as high iron oxide and low silica concentrations (Table 4.2), of which the gold recovery was the main challenge. Recovering gold from this material involved aggressive and expensive chemical recovery techniques not economically viable below twenty grams per ton. The approach was to upgrade the gold concentrate with gravity separation techniques to a gold concentrate of more than twenty grams per ton and then to recover the gold with chemical extraction processes. An investigation into the origin and character of the material was done before the material could be upgraded, because there was very little data available on the material.

The material in this investigation was gold pyrite containing ore from the Witwatersrand reefs that had been milled to average particle sizes smaller than 75 μm (Appendix 1.1), followed by an extraction process. In the flotation plant the gold and pyrite were collected as flotation concentrate. Selective floating chemicals were used to float the gold bearing particles and collect them as gold concentrate. The silica and alumina remained in suspension, separating them from the gold concentrates, and pumped away to the normal waste dump. The flotation

concentrate was roasted to oxidize the sulphur before the gold extraction in the cyanide leaching plant. The effluent from this process contained iron oxides, gold both free gold and gold associated with the iron oxides, and also gold associated with silica and free silica. This material was pumped to a different dump. Hematite represented a large portion (sixty per cent, Table 4.5) of the material and the specific gravity of hematite, being much higher than that of silica, caused this dump's specific gravity to be much higher than that other silica dumps. As specific gravity plays a vital role in gravity separation processes, it was logical to expect that this material would react differently in gravity separation processes.

All the parameters that influence gravity separation of the material had to be determining determined accurately in order to draw the correct evaluations and conclusions. The historical data available from Ergo (Ergo report, Appendix 1.1), on the head grade and particle size distribution, had to be confirmed in order to calculate an accurate mass balance and to evaluate the mineralogical experimental results. The basic mineralogical data of this dump, consisting mainly of calcine, was lacking, and tests had to be conducted to establish the character of this material.

Hematite, and not silica, was the dominant mineral phase present in the dump. This increased the relative specific gravity of the suspension medium. According to the Stokes Law of Separation the fall-out velocity of the particles would be different. This can be expected because the physical number of particles per unit volume would decrease as a result of the higher specific gravity of the hematite. Particles associated with the gold have a higher specific gravity than the particles not associated with gold, mainly as a result of the difference of specific gravity of the two compounds, gold and hematite (Perry *et al.*, 1984). The composition of this material, from pure gold particles (the heaviest) to silica (the lightest) caused a mass gradient and size distribution to be present in this dump. If the separation of this material could be based on gravity separation techniques, the acceleration could be manipulated to optimize the recovery.

Application of a high accelerating force to the suspension containing particles would result in an increased fall-out velocity of the particles. The fall-out velocity is directly proportional to the specific gravity of the particle and with the material under discussion, increasing the gravitational forces would make increased separation possible. The separation can be optimized by manipulating the acceleration of the particles during the pilot plant testing. Conclusions can be

made on the basis of the evaluation of the experimental results obtained in the pilot plant experiment about the correctness of the hypotheses, and also if the economically viable cut-off point of 20 g/t Au in the concentrate could be reached.

1.2. The development of a method to increase gold recovery in the calcine dump material.

1.2.1. Determining the head grade.

In all the experiments the head grade of 2.46 g/t Au (Table 4.5.) used, was obtained from a representative sample from the bulk sample. The experimental results were evaluated with reference to this value.

1.2.2. The distribution of minerals and sizes.

The bulk sample consisted of: 40% Fe, 20% Si, and 2.46 g/t Au (Table 4.5.). Fe and Si were reported in the elementary form, the hematite value calculated from the 40% iron was 61% Fe₂O₃ and the balance was made up mainly by silica – 39% as SiO₂. The distribution of minerals was even for the different size fractions (Graph 4.2). This even distribution was helpful in the evaluation of the separation results. The average particle size was 50% minus 38 µm. The small average size is a problem, as particles smaller than 50 µm tend to remain in suspension and add to the liquid density. It was expected that the particle size was too small for effective separation, but the perspective was changed by the following: In the suspension hematite represented a large part of the solid phase (61% as Fe₂O₃) with a specific gravity of 5.12 g/cm³ (Perry *et al.*, 1984). This is a significant factor when compared to silica's specific gravity of 2.4 g/cm³ (Perry *et al.*, 1984), which is less than half of that of hematite. Assuming that the fall-out velocity, as a consequence would also be doubled, this would cause an increase in the separation. Relating this factor back to the particle sizes, it suggests that separation would be possible for particles down to about 15 µm in diameter, which could explain the relatively good separation on the James Table.

1.2.3. Basic mineralogy tests.

The tabling experiments indicated most promising gravity separation results (Table 4.12). It indicated that the material was sensitive towards normal gravity

forces in the separation processes. A visual colour distinction could be made between the concentrate (dark red), the middling (hematite red) and the tailings (light red). The mass balance of the chemical composition calculated from the ICP measurements and the colour indications on the tabling experiment helped to configure the spiral column's collectors, namely 20% tailings (light fraction), 70% middling and 10% concentrate (heavy fraction). If the material responded so well to the normal gravity acceleration separation techniques, more efficient separation can be expected by using increased acceleration.

1.2.4. Increased acceleration forces to enhance the separation of the minerals.

By increasing the acceleration forces to the calcine suspension, from 9.8 m/s^2 (gravity acceleration) to 117.4 m/s^2 (Appendix 1.3), the separation between the particles could be increased and effective separation of the gold particles could be accomplished. The Itomak high acceleration concentrator shown in the photographic image in Figure 1.1 was used to apply the necessary acceleration in the separation process.

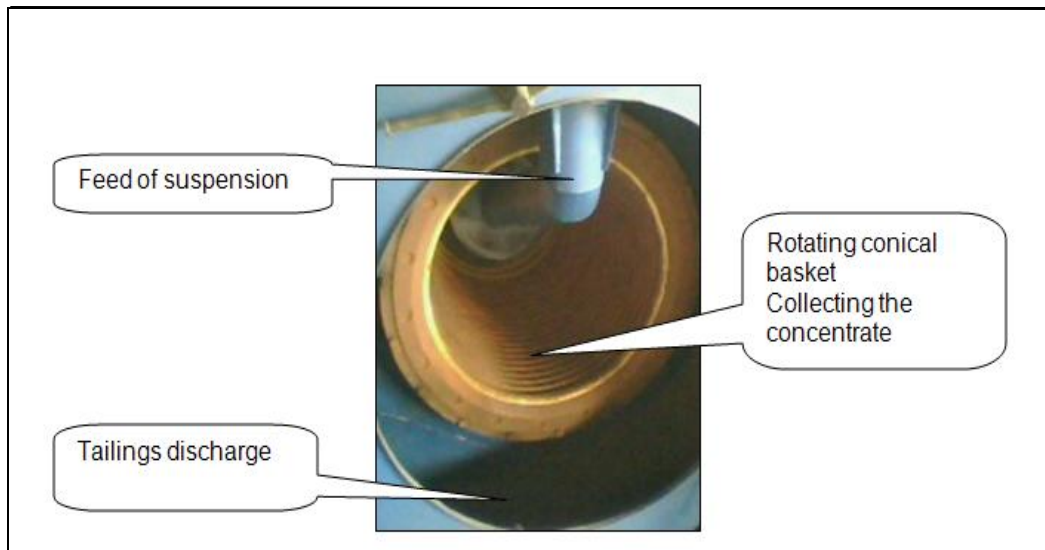


Figure 1.1: Photographic representation of the Itomak high acceleration separator used in this present investigation.

The Itomak, used in this project as high acceleration separator, consisted of a rotating conical basket that turned at high speed (8400 revolutions per minute), creating the required acceleration force on the particles. The basket had a 75 mm diameter ring at the inlet and 150 mm ring at the discharge end.

The suspension of calcine was pumped in at the smaller end of the basket and due to the centrifugal forces forced outward, with the heaviest fraction against the inner surface of the basket. High pressure water was pumped from micro holes in the basket to the inside and the material was washed towards the centre. The water current flowed towards the discharge end of the basket where the tailings were discharged.

The principle applied was the following: as the particles' specific gravity increased, the higher the fall-out velocity became. The Itomak was adjusted to the point where the fall-out velocity became so high that it was impossible for the water current to transport the heavier particles over the last concentric ring of the basket. The heavier particles remained between the rings of the basket and the lighter particles were washed out of the basket. The multiple rings with increasing diameter are necessary, because of the spread in particle size and relative weight of the particles. As the diameter of the basket increases, so does the velocity and also the acceleration forces exerted on the particles. The fall-out velocity is also increased ultimately. Separation of the coarser material takes place on the small diameter side, and progressively smaller sizes are separated towards the bigger diameter of the basket.

The G-force created by the circular movement, accelerate the particle away from the centre, proportional to the velocity of the particle, schematically illustrated in Figure 1.2.

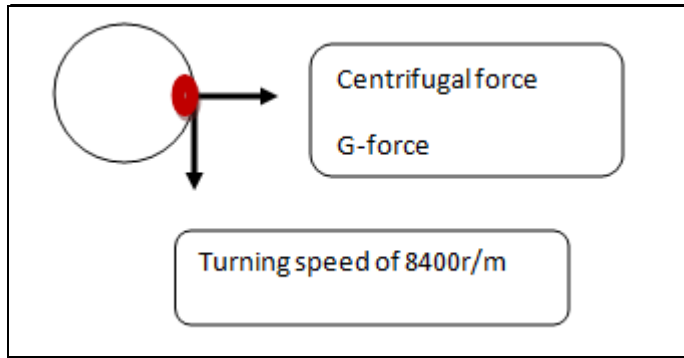


Figure 1.2: Schematic illustration of the centrifugal force (G- force) exerted on a particle subjected to a circular movement.

In order to calculate the acceleration, the velocity must first be calculated. The velocity is influenced by the distance the particle has to travel to complete one circle, or in this case the circumference of the basket.

At the big end of the Itomak basket the diameter is 150 mm and the calculated velocity **39.564 m/s**. The corresponding acceleration force was **117.4 m/s²** (Appendix 1.3). Likewise, at the small end of the basket the diameter is 75 mm and the calculated velocity was **19.824 m/s**, resulting in an acceleration force of **14.7 m/s²** (Appendix 1.3).

The fall-out velocity, also known as the settling, or terminal velocity (V_t) was calculated from equation 1.2 as follows:

The fall-out velocity (V_t) of an average size hematite particle at the 150 mm diameter side of the basket was calculated to be 36.2 $\mu\text{m}/\text{second}$ (Appendix 1.3) and for an average size gold particle was calculated to be 167 $\mu\text{m}/\text{second}$ (Appendix 1.3). The fall-out velocity (V_t) of an average size hematite particle at the end where the basket was 75 mm in diameter, was calculated to be 4.5 $\mu\text{m}/\text{second}$ (Appendix 1.3) and for an average size gold particle 20.9 $\mu\text{m}/\text{second}$ (Appendix 1.3).

When the maximum fall-out velocity (36.2 $\mu\text{m}/\text{s}$) of the hematite at maximum acceleration force (117.4 m/s²) and the fall-out velocity of the gold at minimum fall-out velocity (20.9 $\mu\text{m}/\text{s}$) are compared it shows that the maximum fall-out

velocity of the hematite and the minimum fall-out velocity of the gold were similar. In this calculation an optimum number of gold particles would be retained, and most of the hematite would leave with the tailings if the separation point could be adjusted for fall-out velocities of between 21 $\mu\text{m/s}$ and 36 $\mu\text{m/s}$.

The increase in suspension density reduced the mobility and separation, according to Stokes' Law of separation. The density of the medium (ρ_m) was calculated from equation 1.2.

There were other uncontrollable factors that influenced the separation negatively, such as a spread in particle sizes between 0 μm and 100 μm , and more than 50% of the particles being smaller than 38 μm , limiting the separation effect. Gold locked inside some of the hematite and silica particles, as seen in the SEM photographic images, created a spread in specific gravity of the particles. As a consequence of these two negative factors, it was expected that 100% separation would not be achieved, but that a recovery of more than 30% of the gold would prove the principle that the calcine with its higher specific gravity improved the separation at small particle sizes.

1.3. Objectives of this project.

The first objective of this project was to study the characteristics of the dump material and its response to high acceleration separation. There was not much accurate data available on calcine, and in order to fulfil the objectives of this investigation, the association of the minerals and the character of the materials in the dump had to be determined accurately.

The second objective of this project was to configure a pilot plant and to process a sample of the material by using a combination of acceleration separation processes, including the high acceleration separation techniques. The applied acceleration separation processes had to upgrade the material to a recoverable gold concentrate of higher than 20 g/t before it could become economically viable. Increased acceleration forces exerted on suspension of mixed particles with a difference in specific gravity, will increase the fall-out velocity of the particles. The particles with the higher specific gravity will fall-out faster, causing an increase in distance between the heavier and the lighter particles. The aim was to maximize this separation between the particles by increasing the force due to acceleration appreciably above the gravitational force of 9.8 m/s^2 .

The third objective was to test the hypotheses, namely that the higher specific gravity of the material making up the main phase of a suspension (hematite and not silica), would influence of the acceleration separation process positively and make efficient separation of much smaller particle sizes possible.

1.4. Hypotheses investigated in this project.

The concept, which states it is impossible to recover gold particles effectively with gravity separation techniques from of a suspension if the particle sizes are too small (smaller than 38 μm), was investigated.

The hypothesis that the higher specific gravity of the hematite, making up the main phase of a suspension, will influence the acceleration separation process positively to make separation effective on material with much smaller average particle sizes, was also investigated.

1.5. The four steps taken to prove the hypotheses of this project.

a. Particle size distribution and head grade study:

The particle size distribution and head grade of the material was measured by using standard analytical processes.

b. Material characteristics study.

Experimental testing of standard mineralogical processes was conducted to establishing the characteristics of the material.

c. Pilot plant study.

The recovery processes was optimized by applying a combination of acceleration concentration processes in the pilot plant experiment.

d. Evaluating the objectives and hypotheses.

The test results from the pilot plant experiment were evaluated and conclusions on the correctness of the hypotheses and the objectives of the investigation drawn.

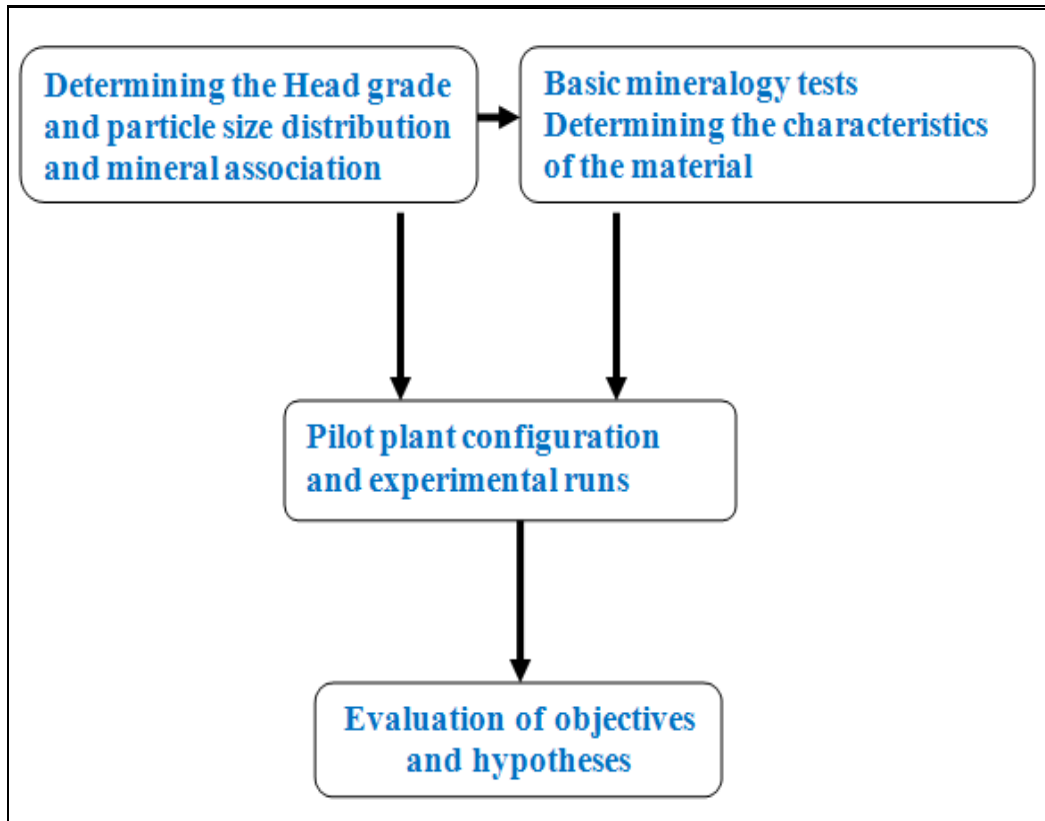


Diagram 1.1: Flow diagram of the experimental procedures employed

Chapter 2

Background and literature survey.

In this chapter certain general aspects of gold mining and the formation of the Witwatersrand gold are given. A short history and background of how the material investigated in the present investigation was produced and the theoretical principle used to separate the minerals in the material is further discussed. All the different mineralogical investigation techniques used in this investigation are discussed theoretically.

2.1. Background and literature survey.

2.1.1. General aspects of gold mining.

Gold is found in nature as an element, primarily as metal or in complex molecules, driven into the earth's surface by volcanic action. The high temperatures resulted in certain associations, for example with silicon oxide because of gold's similar melting point. The association with metals like iron is due to their mutual chemical affinity for sulphur. As the material migrated upwards, it froze at the surface and igneous gold bearing rocks were formed. The "Greenbelt" occurring from the South of the Democratic Republic of the Congo all the way down to Barberton and ending in Pongola district of Kwazulu Natal is a good example of such a gold deposit. When the associated minerals weathered away by oxidation and/or physical processes, the gold was distributed downstream (Bitrch, 2011). Mining of these gold deposits can be divided in two basic categories, namely alluvial and reef mining. Alluvial gold is normally mined where gold occurs in the metal form. The gold originates from reef formations that were oxidized by air in the atmosphere and eroded by water over time and then deposited in river beds. The reefs also oxidized in situ, causing free gold particles to be present in the weathered zone. Reef gold normally contains a portion of the gold in metal form, but gold can also occur as gold complexes or a combination of both.

2.1.2. The formation of the Witwatersrand gold.

The Witwatersrand gold deposit originated from the weathering of such deposits and was transported into the Witwatersrand basin. Sedimentation of the gold, minerals and rocks formed sequential layers in the basin, now mined as the Witwatersrand gold reefs (Norman & Whitfield, 2006). The Witwatersrand gold was discovered by George Harrison on the farm Langlaagte in March, 1886. The quartz-pebble conglomerates rapidly became the largest gold source on Earth.

The name “Witwatersrand” or “ridge of white waters” was given to the geological area in central South Africa by the Boers because of the many waterfalls which once cascaded off its stony ridges. As can be seen in Figure 2.1, the basin is asymmetric oval in shape and extends approximately 350 km along its long axis, which lies in a north-east to south-west direction (Norman & Whitfield, 2006).

The Witwatersrand was part of a much bigger basin complex. A gold-rich Archaean granite greenstone in the early continent acted as the source of the gold. Over a very long time span these gold-rich sediments migrated by hydrothermal activity into the basin. The atmospheric conditions were favourable in preserving pyrite with the placer minerals. The consecutive layers of reef and other sedimentations formed. A unique astral impact created structural features in the crust that favoured the preservation of the depository and outcropped some of it, that led to the discovery of the Witwatersrand gold (Johnson *et al.*, 2009).

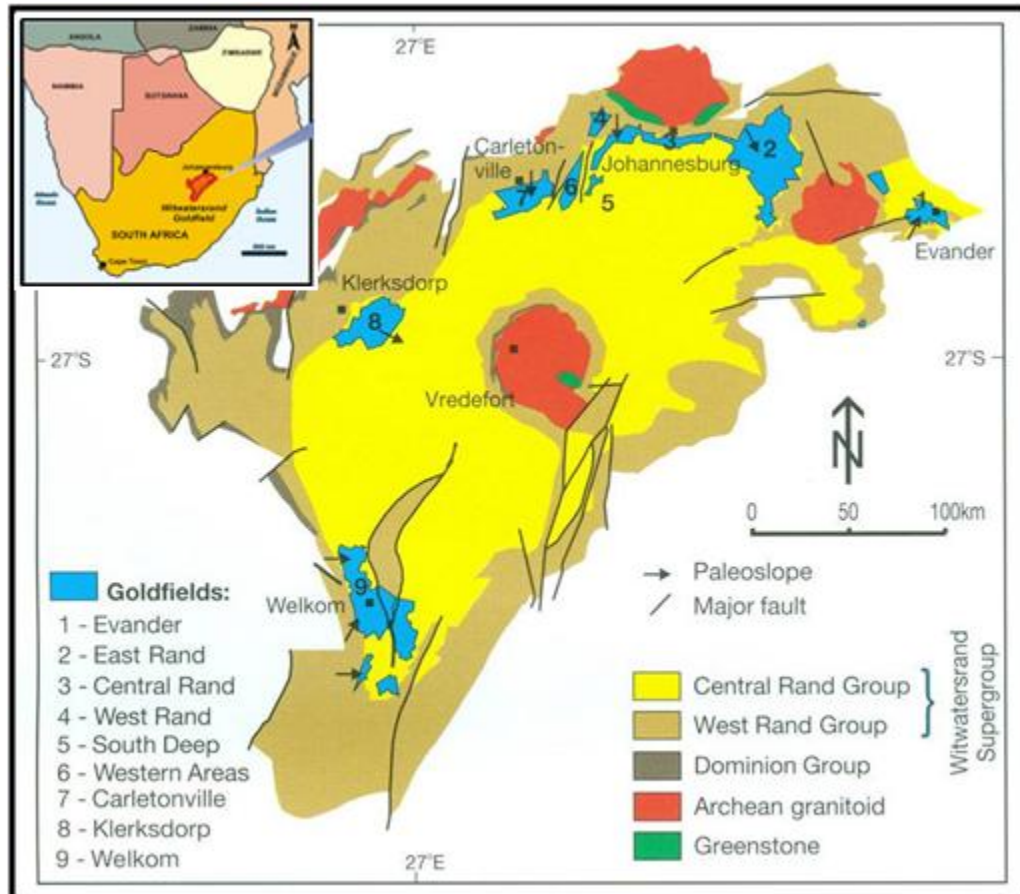


Figure 2.1. Geological map of the Witwatersrand super group, showing the locations where the major goldfields are (Frimmel, 2004).

The Witwatersrand Basin began forming about 3 500 million years ago when this area was covered by a large inland sea. Sediments were deposited along the bottom of this sea in an oxygen-deficient environment. Changing conditions in the earth's crust caused the sea to become shallower in time, and fill with coarse sand and thick layers of quartz pebbles. Surrounding volcanic activity covered the sediments with Basaltic lava. Alternating periods of sedimentation and volcanic eruptions caused layers of mixed lava and sediment to build up. Gold was deposited as fluvial sediments from river deltas at the shores of the inland sea. Seven major gold-bearing reefs occur within the Witwatersrand basin, with the gold being contained in 1 to 2 m thick tabular conglomerate layers (Norman and Whitfield, 2006; Tutu *et al.*, 2008).

The conglomerates consist of quartz pebbles in a matrix of quartz sand. Typical Witwatersrand ores, which are mined for gold and uranium, have the following composition (Tutu *et al.*, 2008):

- Quartz (70 to 90%)
- Phyllosilicates (10 to 30%)
- Accessory and minor minerals (1 to 5%)
- Base metal sulphides (most commonly pyrite (FeS₂) with an abundance of 3 to 5 %)
- Uranium bearing minerals (290 to 450 ppm)
- Gold bearing minerals (10 to 100 ppm)

Since the commencement of gold mining in South Africa over 50 000 tons of gold have been produced. In the late 1960s many of the mines closed, and those which remained were forced to mine deeper to reach the deeper lying gold reefs. Some mines have reached depths exceeding 3 000 m (Norman and Whitfield, 2006; Tutu *et al.*, 2008).

2.1.2.1. Gold extraction.

To extract gold, different chemical and gravitation methods were historically used. Primitive mining methods of alluvial deposits only focused on the extraction of free gold. Mining free gold usually involved gravity concentrating methods with mercury collecting before smelting. As the gold mining industry developed the emphases shifted to chemical processes for optimum recoveries. Gold bearing ore was oxidized by roasting, pressurized acid leaching, peroxide and bioleaching, (Deng and Liao, 2001) and in the recovery plants gold flotation processes were very prominent, dating far back (Wang and Poling, 1983). The first paper on true gold flotation was published by (Motherwell, 1914), which made it possible to concentrate low grade gold ore and recover gold complexes like pyrite. As newer technologies became available the recovery of gold improved. However, processed ore with relative high gold values were discarded on mine reserve dumps. A lack of technology and too short recovery plant retention times made it uneconomical to recover the gold (Petela, 1993). The discharge volumes were too small to process these tailings economically for the remainder of the gold.

2.1.2.2. Refractory ore.

Mining the unique Witwatersrand deposit had a few challenges; the gold particles were very small, making it not viable to recover with gravitational methods. A large portion of the ore mined was refractory ore and had to be oxidized and treated before it could be extracted with chemicals.

The direct cause of low recoveries is that gold is locked inside sulphide molecules. The solution to this problem was the implementation of an oxidative pre-treatment, which would completely break down the sulphide matrix and release the gold from solid solution as described by authors such as Dry and Coetzee, (1986) and Demopoulos and Papangelakis, (1989). It was suggested that a prior concentration step, such as flotation would be required to minimise the volume of material to be treated in the roasting plant, making the treatment cost effective (Swash, 1988).

In the flotation process gold and gold associated sulphides (Kelly, *et al*, 1982) are attached to an air bubble caused by the addition of surface active chemicals like xanthates. These chemicals have a high affinity for the gold and the sulphides, normally present in the ore. After the floating chemicals have attached to the gold, the hydrophobic side arranged on the surface of the air bubbles, produced by the flotation cell aeration system, the air bubbles are now transported to the surface as foam. The foam is collected as a flotation concentrate at the upper surface of the flotation cell where it accumulates. The remainder of the material drops to the bottom or remain in the pulp suspension.

In the milling process the ore is milled to very fine particles usually with an average particle size smaller than 70 μm (Appendix 1.1.), which would not only liberate the free gold particles, but also mills the gold into fine plates. Gold is very malleable and a large portion of free gold is milled to plate form. The chemical and physical condition of the gold is very important for addition to the floating chemicals. Flotation results are influenced negatively by physical conditions such as pulp viscosity and the roughness of the gold particle surface (Taggart, 1927). Imbedded particles on the gold flake surface are also contributing to bad flotation results. These imbedded particles influence the hydrophobic properties and reduced the floatability. A high pulp density is beneficial to free gold flotation

which suggested that a significant proportion of the gold may be recovered along with any entrained water and gangue material (Lange, 1935).

Another aspect that negatively influences the flotation process is the presence of a competition component for the attachment on the bubble, which could be gold surfaces, coated by large amounts of partly hydrophobic gangue fines such as pyrophyllite (Allison *et al.*, 1982.; Aksoy and Yarar, 1989). Ferric ions in the form of hydrated oxides act as physical barriers between the air bubbles and the gold surfaces during the flotation process.

In general, the recovery of refractory gold follows the same trend as the sulphide minerals, but normally the gold is locked inside the pyrite, pyrrhotite or arsenopyrite. Some gold bearing ores are refractory due to the smallness of the gold grains and the concentration of the sulphide minerals require flotation followed by roasting, bacterial leaching or pressure oxidation, liberating the gold prior to cyanide leaching (O'Connor and Dunne 1990).

Flotation gold concentrate is roasted in a roasting plant that liberates or exposes the locked in gold particles. In the cyanide leaching process that follows, the gold is dissolved and recovered chemically. The waste from the flotation plant still has a small amount of gold left after flotation, but extraction of gold in the floating process is considered sufficient with about 78% gold recovery (Lehmann *et al.*, 1999) for the recovery process.

2.1.3. How the material investigated in this present investigation was produced.

The specific dump investigated in this project was formed as a tailings dump of a large flotation and roasting plant. Pyrite-containing ore from the Witwatersrand reefs were milled to particle sizes smaller than 70 μm (Appendix 1.1). In the flotation plant the gold and pyrite were collected as flotation concentrates. The silica and alumina didn't bind to the floating chemicals, hence they were separated from the concentrate and pumped away to a waste dump.

In the roasting process that followed, the pyrite was converted into hematite and sulphur dioxide. The pyrite was oxidized in the roasting plant, converting the sulphur to sulphur dioxide which was further oxidized by vanadium pentoxide catalyst to sulphur trioxide and then collected in sulphuric acid scrubbers as sulphuric acid. The iron was oxidized to hematite (Fe_2O_3) and magnetite (Fe_3O_4). Magnetite formed in hypo-oxidation conditions, where the oxygen supply was low. The trapped gold particles were liberated enabling the chemical process to dissolve the gold (O'Connor and Dunne, 1994).

The cyanide leaching process that followed dissolved most of the gold. The waste tailings of the leaching process were pumped to a separate mine reserve dump. The effluent normally contained iron oxides, gold and silica, which are commonly referred to as calcine (De Wet *et al.*, 1995). The main phase of this calcine dump was found to be hematite and not silica (Table 4.5). In theory the gravity separation properties should differ from the norm in the industry, where silica represents the larger proportion of such a waste dump.

2.1.4. Mineralogical investigation techniques used in this investigation.

The uniqueness of the material studied in this present investigation made it necessary to conduct some chemical and mineralogy tests in order to assess all the factors that would influence separation of the minerals. To determine the material's character, proven mineralogical concepts were tested on the material. Some of the basic mineralogical tests were performed on the material because the waste dump investigated was different from the normal dump material. The tests conducted were flotation, magnetic separation, hydraulic separation, dense medium separation, multi-gravity concentration and tabling.

The parameter in the gravity separation process that could be changed was the acceleration. This was done by increasing the centrifugal force exerted on the particles during the separation process. In the high acceleration concentrator, an increase in the number of revolutions of the basket, or turning speed, resulted in an increased acceleration exerted on the particles (Appendix 1.3). In turn this would increase the fall-out velocity of the particles directly proportional to their weight and size, and ultimately be responsible for separation of the minerals inside the separator.

2.1.5. The theoretical principle of increased acceleration separation technology used to upgrade the dump.

The theoretical principle investigated in the present investigation was to make use of high acceleration separation technology. This method was used to concentrate the gold from the calcine mine reserve dump to a gold concentrate that could be recovered economically. The problem was, however, that the calcine particles were very fine, the gold occurred in very small particles and were even sometimes included in the pyrite (Anhaeusser *et al.*, 1987) and also still included in the hematite after roasting. This could comprise as much as 20% of the gold content (De Sousa, 1986). Particles measuring smaller than 38 μm accounted for more than 50%, which posed a problem to possible gravity separation (Dandios *et al.*, 1997). Even with high acceleration separation methods, only 32% gold could be recovered for particles under 40 μm (Forrest *et al.*, 2001).

Besides the size, the specific gravity of the particles also plays an important role in gravity separation techniques (Equation 1.2.). Because of the difference in the specific gravity of gold (19.3 g/cm^3), hematite (5.2 g/cm^3) and silica (2.4 g/cm^3) (Perry *et al.*, 1984), the gold in association with any of the other particles, such as hematite or silica, would cause an increase in the complex specific gravity. This increase in specific gravity would be relative to the size of gold particles present, to the rest of the complex influencing the acceleration force and the fall-out velocity. Particles will flow laminar with the suspension current if no unbalanced forces are present. However, the calcine particles are heavier than the water, resulting in an imbalance in the forces exercised on the particle. The magnitude of this force is dependent on the gravity or centrifugal acceleration given by the equation:

$$F = ma \qquad 1.1$$

where F is the force, m is the mass of the particle, and a is the acceleration (or gravity acceleration g).

The imbalanced forces will cause the particle to move relative to the direction of the force, resulting in a change in velocity. The force that counters the changes in velocity of the particles is called drag. The drag on the particle will tend to slow

down the changes in velocity of the particle and decrease the fall-out velocity. Drag on the particle is influenced by the symmetry of the particle, the viscosity of the fluid and the turbulence (Kelly *et al.*, 1982). Stokes assumed that, under laminar flow conditions, the drag force on a spherical particle was entirely due to viscous forces within the fluid, and he deducted the influence of the symmetry from the equation defining the fall-out velocity according to equation 1.2. According to the Stokes Law of Separation, there are certain factors influencing this separation process: terminal, fall or settling velocity, gravity acceleration, particle diameter, and medium density, specific gravity of the particle and the viscosity of the medium.

The density of a particle is defined the following equation by:

$$\rho_p = \frac{18\mu V_t}{gd^2} + \rho_m \quad 1.2$$

Where:

ρ_p = Density of the particle	measured in kilogram/cubic metre
μ = Viscosity of the medium	measured in kilogram/metre second
V_t = Fall-out velocity	measured in micrometre/second
g = Acceleration	measured in metre/second squared
d = Particle diameter	measured in millimetre
ρ_m = Density of the medium	measured in kilogram/cubic metre

It is evident from equation 1.2 that the particle density (ρ_p) is proportional equal to the fall-out velocity (V_t). In the present study, the parameter used was the fall-out velocity (V_t), or the velocity by which the suspended particles would settle out from the suspension. Parameters kept constant for the experiment were the particle diameter (d), the medium density (ρ_m), the density of particle (ρ_p) and the viscosity of the medium (μ).

Chapter 3

Experimental procedures.

In this chapter the experimental procedures are discussed. In addition more detailed data of the experimental results can be found in Appendix 1.1 & 1.2.

3.1. Methods and procedures used to determine the material characteristics.

3.1.1. The experimental procedures to sample the dump and the analytical testing to determine the dump's head grade.

Seven boreholes were drilled with an auger drill on a 50 m drill pattern apart and a total of 62 samples of the dump material were taken. The depth the holes varied from 1 m to 13 m as the dump varies in depth from 9 m to 13 m. A gold fire assay was done on each sample after drying at 120 °C for three hours, before measuring the gold content. The results were recorded and the average gold content was calculated and reported.

3.1.2. The experimental procedures to analyse the bulk sample.

A representative sample was taken by the *coning and quartering method* of the bulk sample and samples were analysed to determine the:

- Particle size distribution.
- Gold content or head grade of the bulk sample.
- Chemical composition.

The “coning and quartering” method is performed as follows:

The dry material is mixed and poured in a conical heap, then divided into four equal quarters. The process is repeated with the material of two opposite quarters reducing the material volume every time by half, till the required sample size is obtained.

3.1.2.1. The experiment to measure the particle size distribution.

The determination of the size distribution was very crucial for the experiment and physical screen sizes were used to classify the material. The material sample was dried at 120°C for three hours and weighed accurately. The sample was then put through a stack of vibrating screening sieves of sizes 104 µm, 75 µm, 53 µm, 38 µm and 0 µm respectively. The screen sizes were not evenly distributed but chosen according to the sizes normally used in the mining industry as size indicators. The weight retained on each screen was accurately recorded and the samples were further subjected to ICP analytical analyses.

3.1.2.2. The experiment to analyse the gold content of the bulk sample.

Standard Induction Couple Plasma (ICP) protocol and fire assays were used to analyse the gold content of the bulk sample. In the standard ICP protocol all the solid material samples were fused by heating in the presence of sodium peroxide, then dissolved in concentrated hydrofluoric acid. When noble elements were analysed, Aqua Regia (70% hydrochloric and 30% nitric acid) was added to dissolve all the remaining solids. The mixture was diluted accurately to ppm levels and subsequently analysed.

3.1.2.3. The experiment to analyse the chemical composition of the bulk sample was done with Induction Coupled Plasma mass spectrometry (ICP-MS).

The standard (ICP) protocol was used to analyse the chemical composition of the bulk sample. Samples from every screen size were analysed separate to determine

its chemical composition. This would have given an indication if there were a difference in chemical composition for the different particle sizes

Induction Coupled Plasma mass spectrometry (ICP-MS) is used for the chemical analysis of the different elements in the material. Since the early 1980s ICP-MS has become one of the most important techniques for elemental analysis (Skoog *et al.*, 1998).

Advantages of ICP-MS include:

- High degree of selectivity
- Low detection limits for most elements (as low as parts per trillion)
- Good precision and accuracy

The process of analysing by ICP-MS can be broken down into four phases, schematically shown in Figure 3.1: sample introduction and aerosol generation (nebulisation), ionisation by an argon plasma source, mass discrimination and finally, being registered by the detection source. For solutions, sample introduction is accomplished by either an ultrasonic or a conventional nebulizer (Skoog *et al.*, 1998). Samples are decomposed to neutral elements in the high temperature argon plasma and analysed, based on their mass to charge ratios (Worley and Kvech, 2000).

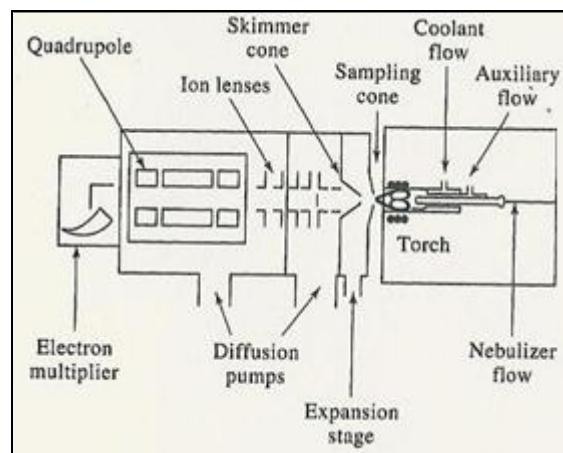


Figure 3.1. Schematic of an ICP-MS system (Skoog *et al.*, 1998).

3.1.3. Particle association was investigated with the Scanning Electron Microscope (SEM) and Energy Dispersive X-ray analysis (EDX).

Small samples were taken from the bulk sample, mixed with poly-resin and fixated. Electron microscope slides were cut and prepared by polishing the surfaces. The slides were put in the Scanning Electron Microscope (SEM) and studied under high magnification. Photographic slides were taken of the different minerals with the SEM (Figure 3.2. photographic representation of the SEM). The relationships of the minerals to each other were determined with the SEM's EDX (Energy Dispersive X-ray analysis) function that made it possible to distinguish between the minerals on the image



Figure 3.2.: Photographic representation of the sample chamber opened on the SEM, used in this present investigation.

Scanning Electron Microscopy (SEM) is used to determine the physical structure and association of the minerals. Scanning electron microscopy is used when detailed knowledge of the physical and chemical nature of the surfaces of solids are required. The surface to be studied is swept in a raster pattern with a fine focussed beam of electrons. The electron beam is first swept across the surface in a straight line (x -direction), and then returned to its starting position, from where it is shifted downwards (y -direction) by a standard increment. This is repeated until the entire required area is scanned (Skoog *et al.*, 1998; Tiedt and Pretorius, 2006). A schematic illustration of the SEM apparatus is shown in Figure 3.3.

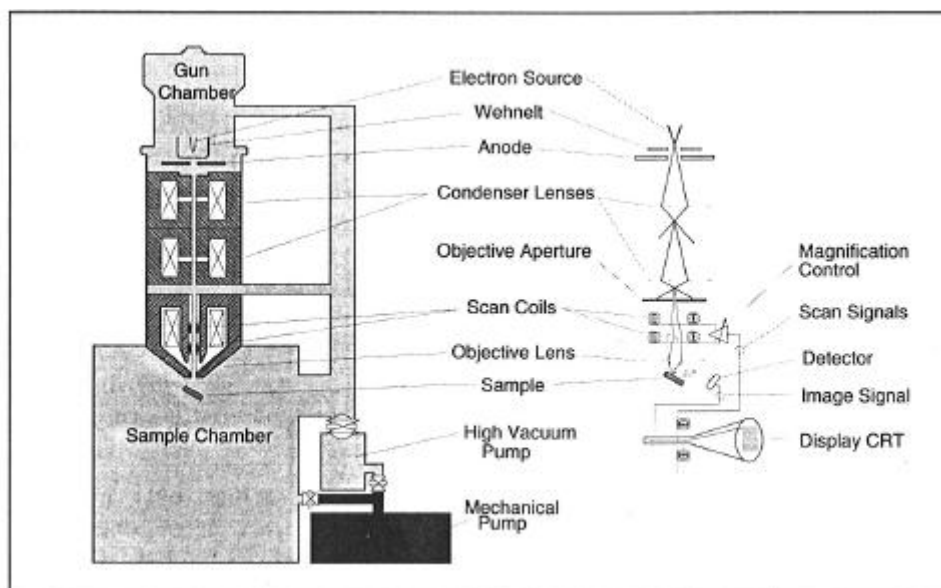


Figure 3.3. Schematic illustration of the SEM apparatus (Skoog et al., 1998).

Several types of signals are produced when the surface is bombarded with electrons. Most important are the backscattered and secondary electrons, which serve as the basis of scanning electron microscopy (Skoog *et al.*, 1998).

Images are obtained, based on the backscatter of electrons. Backscattered electrons are the result of incident electrons being re-emitted due to collisions within the sample. The atomic number determines the intensity of the backscattered signal. A lighter image corresponds to a high atomic number, and darker images are associated with lighter elements. The backscattered signal is received above the surface of the sample and stored in a computer system where it is converted into an image. X-ray emission can also be used to identify certain elements (Skoog et al., 1998).

X-ray diffraction provides a means for the qualitative identification of crystalline compounds. The arrangement and spacing of atoms in crystalline structures can be determined by diffraction studies. This method is based upon the fact that an X-ray diffraction pattern is unique for each crystalline substance (Skoog *et al.*, 1998).

Quantitative analysis of crystalline mixtures is made possible by measuring the intensities of diffraction lines and comparing them to standards. Furthermore, if an exact match can be found between the pattern of an unknown and an authentic sample, chemical identity can be assumed (Skoog *et al.*, 1998).

3.2. The mineralogical experimental procedures performed on the bulk sample.

3.2.1. The flotation experimental procedures.

The flotation separation process is based on chemical binding of the flotation chemicals to sulphides or metal molecules in the ore. The molecules of the flotation chemicals consist of a sulphide- or metallophillic and a hydrophobic part. The one side of the chemical molecule attaches to the mineral and the opposite hydrophobic side to the surface of the micro-bubbles. The air bubbles are formed as a result of the flotation cell's mechanical aeration action (schematically represented in Figure 3.4.). The bubbles migrate to the surface where it is collected as concentrate and the rest of the ore sinks to be discharged as tailings (Kelly, *et al* 1982).

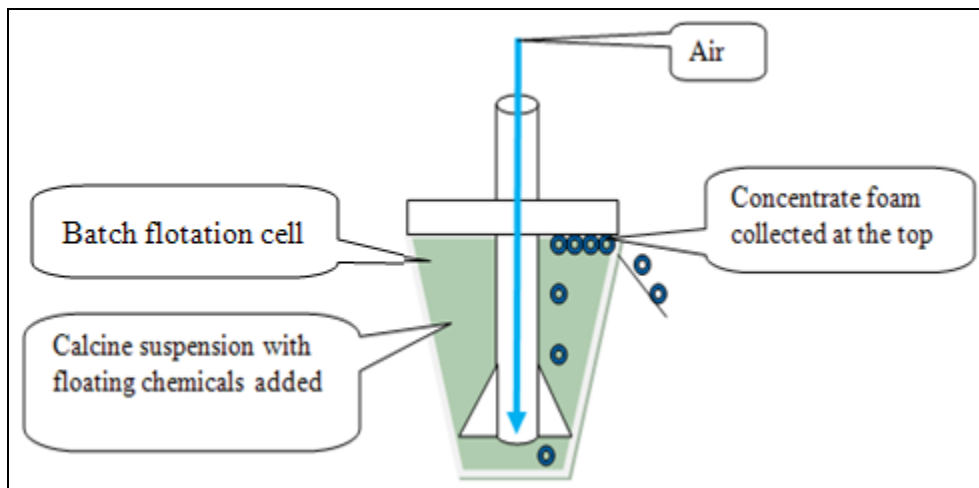


Figure 3.4. Schematic illustration of the batch flotation cell.

A 30% mass per volume water suspension was made with the calcine obtained from the bulk sample and floated in a batch flotation cell. Xanthate was used as flotation agent. The floated concentrate was collected and analysed.

The equipment used for the floating experiment was a batch flotation cell (Figure 3.5.), consisting of a hollow drive agitator that pumped air into the pulp at the bottom. The agitator suspended the pulp and delivered the air for the bubble formation, with the aid of the floating chemicals. The foam formed, migrated to the top of the cell and frothed over the top edge of the cell. Samples of the overflow foam and pulp from the cell were collected and analysed. A schematic representation of the batch flotation cell is shown in Figure 1.4.

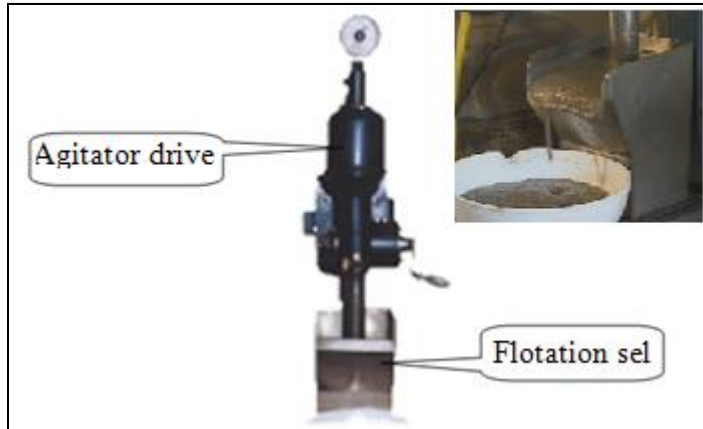


Figure 3.5: Photographic representation of the batch flotation cell used in this present investigation.

3.2.2. The magnetic separation experimental procedures.

Magnetic separation is used to separate strong magnetic and non-magnetic materials in the ore. In the gold process, the magnetic component is often magnetite, and strong permanent magnets or electromagnets are configured in a separation stream where the ore is separated by a thin layer of non-magnetic material. The magnetic field is used to direct the magnetic particles into a collector chute opposite the flow direction. The magnetic ore is separated by routing the thin non-magnetic material away from the magnet source, which releases the magnetic ore in the collecting chute (Kelly, *et al* 1982). The Davis magnetic separator is schematically illustrated in Figure 3.6.

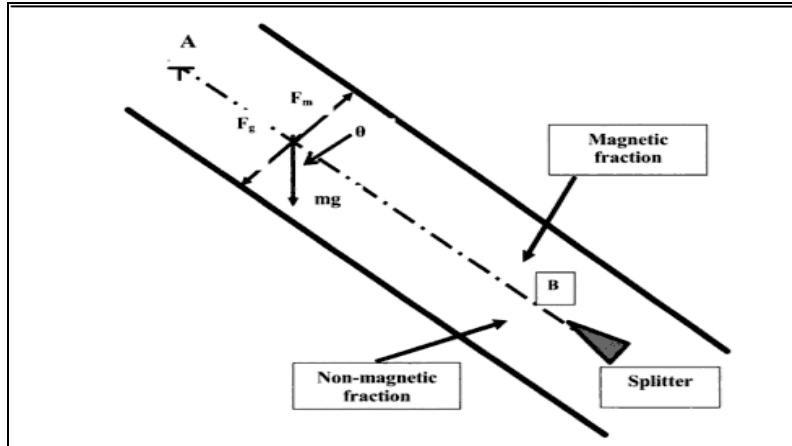


Figure 3.6. Schematic illustration of the Davis magnetic separator.

A water suspension was made of 30% mass per volume, of the calcine from the bulk sample and put through a Davis tube magnetic separator. The magnetic and nonmagnetic fractions were collected separately and analysed. A photographic representation of the Davis tube magnetic separator is shown in Figure 3.7.

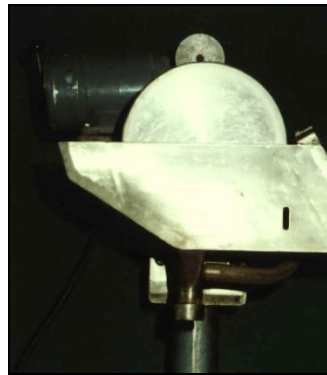


Figure 3.7: Photographic representation of the Davis tube magnetic separator used for the magnetic separation experiment.

3.2.3. The hydraulic separation experimental procedures.

In hydraulic separation the settling characteristics of the heavy particles in turbulent water flow is used to separate the heavy particles from the lighter

particles. By continually repeating this process separation is made possible. The Multi-Purpose Concentrator (MPC) has a rotating drum with a concurrent collecting spiral that reverses the current, collecting the heavy particles. Figure 1.6 shows a schematic illustration of the spirals, while Figure 3.8. shows the inverse spiral rib configuration moving through the fluidized slurry bed.

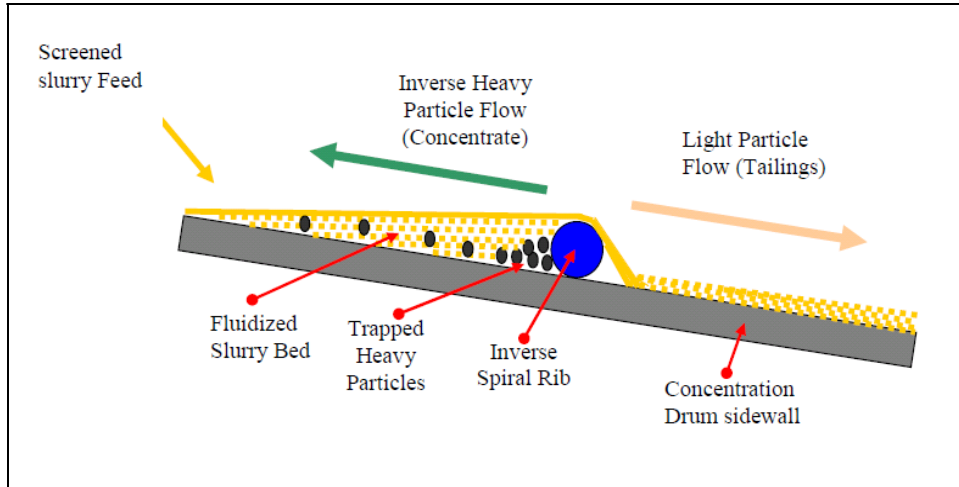


Figure 3.8: Schematic illustration of the separating principle of the MPC5 hydraulic separator.

A one ton sample of calcine was processed with a hydraulic separator, a five ton multipurpose concentrator (MPC5 model), shown in the photographic image in Figure 3.9. The suspension processed through the MPC5 had a ratio of 1.5 m³ water to 1 ton of calcine, at a feed rate of five tons of calcine per hour. The rotating speed of the separation drum was 18 rpm and the settling angle of the MPC5 was 13 degrees to the horizontal. Samples were collected from the tailings and concentrate.

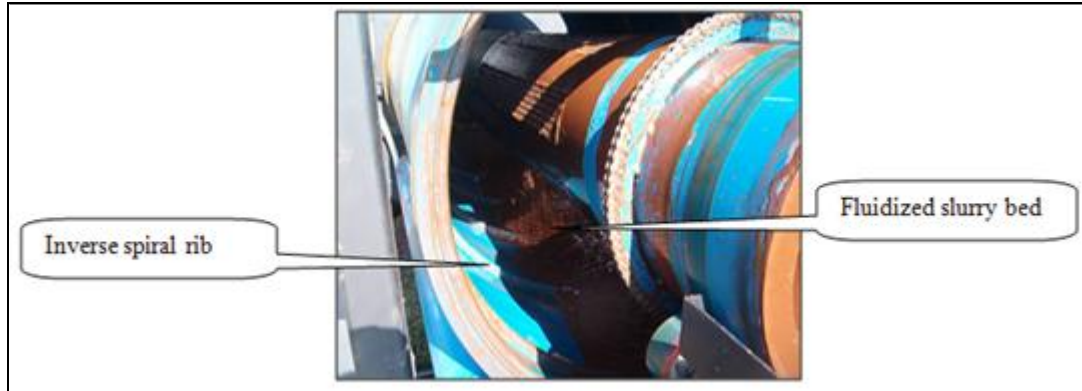


Figure 3.9: Photographic representation of the MPC5 concentrator used in this present investigation.

3.2.4. The dense media separation experimental procedures.

Gravity dense medium separation (DMS) techniques are applied to the material, testing the gravity separation properties through heavy dense medium material. The DMS plant consists of a bed of ferrous-silicon with a specific gravity corresponding to the separation cut-off point to be measured in the ore. A pulse action is applied to the stream flowing over the separation bed. Heavier particles than the separation bed will fall through to be collected as heavy concentrate and the lighter material will flow over the bed as tailings. Normally a magnetic material is used and collected with a magnetic separator in the overflow stream. A schematic illustration of the DMS plant is shown in Figure 3.10.

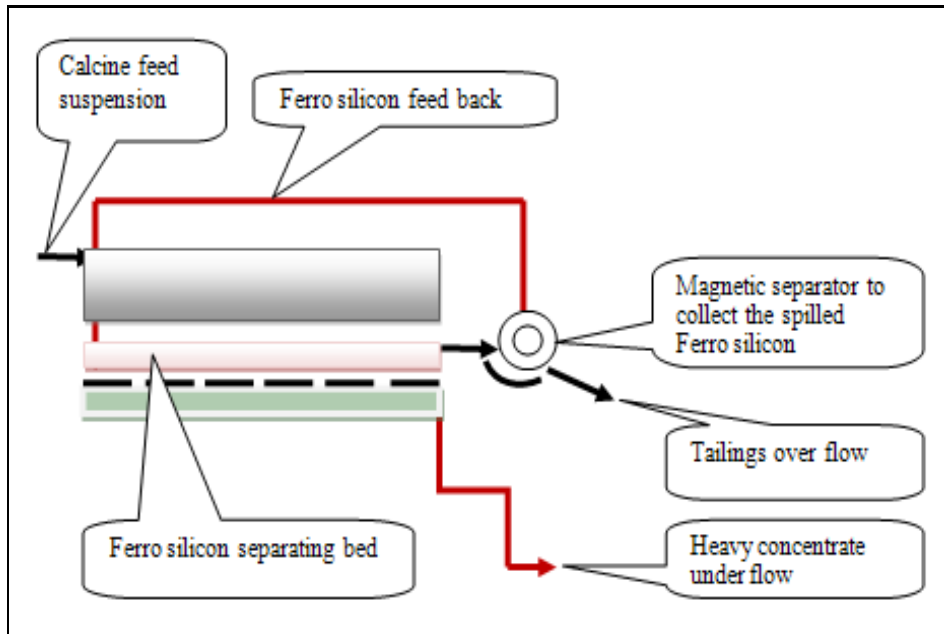


Figure 3.10: Schematic illustration of the functional unit in the DMS plant used in this present investigation.

In the dense media separation (DMS) experiment a 30% mass per volume water suspension was made from the bulk sample and run through the DMS plant. The dense media bed was made of ferro-silicon with a specific gravity of 3.2 g/cm^3 . The overflow was collected as tailings and the underflow was collected as concentrate. Samples of each stream were dried and analysed. A photographic image of the DMS plant used in the present investigation is shown in Figure 3.11.

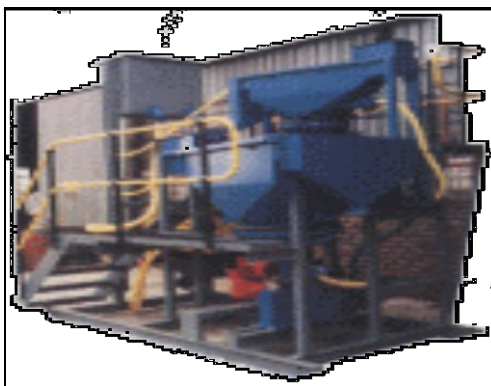


Figure 3.11: Photographic representation of the DMS plant used in the present investigation.

3.2.5. The multi-gravity separation experimental procedures.

Multi-gravity separation is done with a series of hydro cyclones. The material suspension is pumped into the cyclone at very specific flow rates, causing a centrifugal movement in the cyclone forcing the heavier particles to the outside and the lighter particles to the inside. The overflow lighter fraction is collected from the middle and the heavier fraction drops down in the cone, as the velocity decreases against the wall of the cyclone and the heavy particles are eventually discharged from the bottom as heavy concentrate (Kelly, *et al* 1982). The flow in the cyclone is schematically illustrated in Figure 3.12. The separation can be altered in the same cyclone by altering the flow rates and height of the discharge, or by changing the cyclone's physical dimensions.

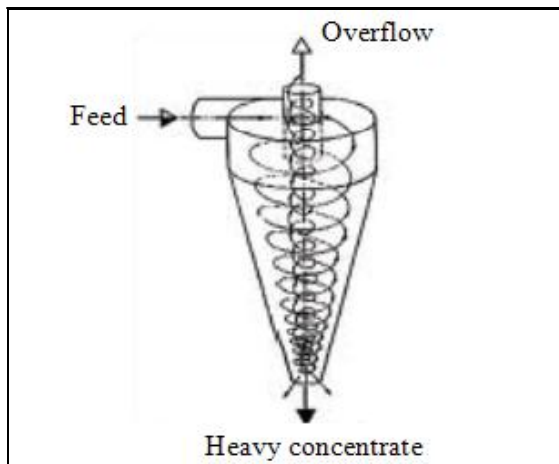


Figure 3.12: Schematic representation of a cyclone.

The multi-gravity separation experiment was done with a multi cyclone separator. A 30% mass per volume water suspension was made from the bulk sample and pumped through the cyclone separator. The cyclone underflow represented the concentrate or heavy fraction and the overflow represented the light tailings fraction. Samples were taken from the concentrate and tailing and analysed. Figure 3.13 is a photographic representation of the cyclone multi-gravity separator used in this investigation.

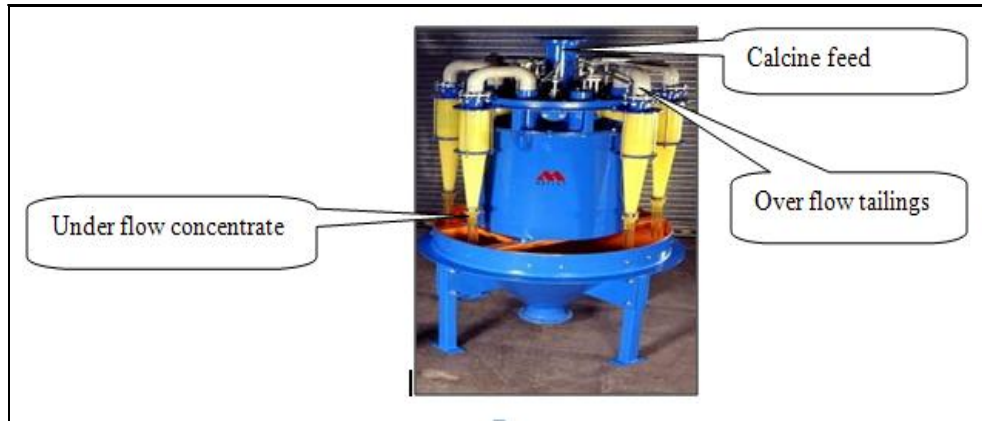


Figure 3.13: Photographic representation of a cyclone multi-gravity separator used in this present investigation.

3.2.6. The James table experimental procedures.

The small James Table, used to separate the material is based on the momentum changes of the particles on the table's surface. The table is also called a shaking table and it accelerates the particles by shaking the table's surface horizontally. The heavier the particles, the further the momentum will displace them. The recoil action causes turbulence and the friction force between the table surface and the particles is limited, causing also limited momentum to be generated in the reverse direction. Simultaneously the water current is running at 90° over the table's ribs tilted at a slight angle, washing down the lighter material that settles slower over the ribs as schematically illustrated in Figure 3.14.

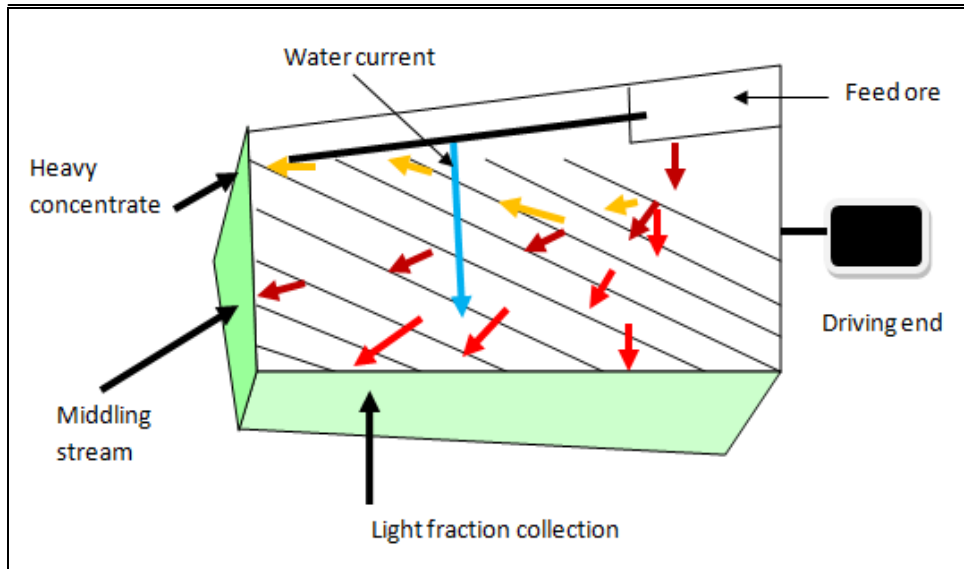


Figure 3.14: Schematic representation made of a James Table used in this present investigation.

A 30% mass per volume water suspension was made from the bulk sample and sent over a 1 metre long James table. Samples were collected from the concentrate, middling and tailing and analysed. A photographic representation the James table used during the tabling experiment is shown in Figure 3.15.

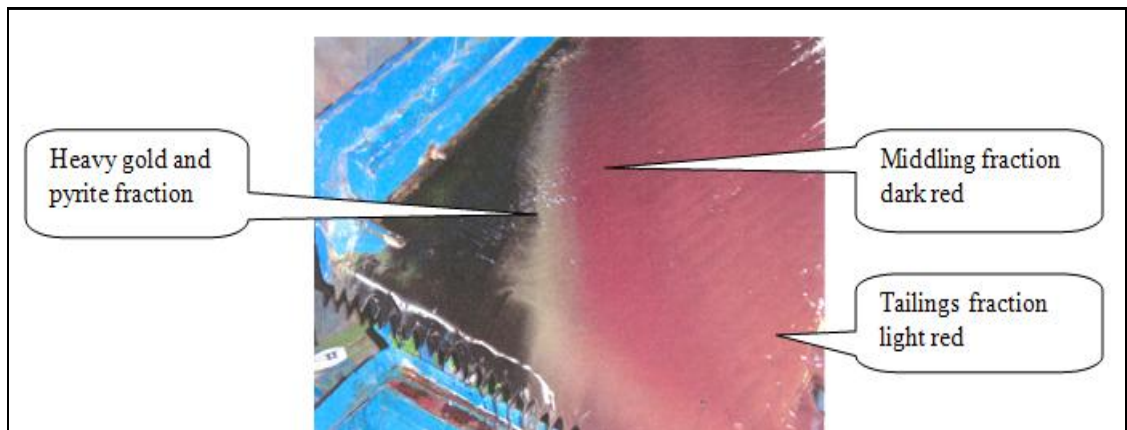


Figure 3.15: Photographic representation of the calcine showing separation on the James table taken during experimentation.

Chapter 4.

Experimental results.

In this chapter the actual experimental results are shown for the experiments needed to develop the input parameters for the pilot scale gold recovery process.

4.1. Results of experiments determining the material parameters.

The parameters of the dump would give an indication whether the gold was linked to other materials or occurred as free gold. The material originated from the same plant and if the gold content was higher at the bottom, it would indicate that the gold is present as free gold and have migrated downwards in the dump.

4.1.1. Results of the fire assay to determine the head grade of the dump.

The gold content of the dump was determined by analyzing the samples with a fire assay analyzing technique done on all 62 samples, and the average gold contents were calculated and are shown in Table 4.1.

Table 4.1: Gold values analysed with fire assay of the dump samples.

Ref. No.	Sample No	Au g/t	Ref. No.	Sample No	Au g/t
41273	1	1.44	41539	33	1.54
41274	2	1.35	41540	34	1.54
41275	3	1.4	41541	35	1.42
41276	4	1.38	41542	36	1.34
41277	5	1.2	41543	37	1.36
41278	6	1.4	41544	38	1.4
41279	7	1.38	41545	39	1.2
41280	8	1.2	41546	40	1.43
41281	9	1.4	41547	41	1.7
41282	10	1.52	41548	42	1.68
41283	11	1.74	41549	43	1.36
41284	12	1.64	41550	44	1.74
41519	13	1.84	41551	45	1.42
41520	14	0.14	41552	46	1.44
41521	15	1.56	41553	47	1.44
41522	16	1.38	41554	48	1.3
41523	17	1.36	41555	49	1.34
41524	18	1.32	41556	50	1.32
41525	19	1.34	41557	51	1.34
41526	20	1.52	41558	52	1.5

41527	21	1.54	41559	53	1.64
41528	22	1.54	41560	54	1.72
41529	23	1.7	41561	55	1.78
41530	24	1.34	41562	56	1.32
41531	25	1.3	41563	57	1.52
41532	26	1.38	41564	58	1.46
41533	27	1.42	41565	59	1.38
41534	28	1.16	41566	60	1.44
41535	29	1.32	41567	61	1.22
41536	30	1.58	41568	62	1.47
41537	31	1.76			88.81
41538	32	1.5	Average	g/t Au	1.432

4.1.1.1 Discussion of the fire assay results to determine the head grade of the dump.

The 62 samples were taken every metre vertically, from the 7 holes on a grit pattern 50 m apart. This sampling was done to verify if the gold migrated downwards in the dump or if the gold concentration was uniform throughout the dump. The average gold value obtained also represents the gold head grade for the whole dump. The gold fire assay from dump reported an average gold grade of 1.432 g/t. This gold value corresponds with the head grade reported by Ergo of

1.446 g/t. (Appendix 1.2). Ergo's samples were obtained from random grab samples on the dump assayed by fire assay techniques.

The Standard Deviation (SDEV) of the gold values were calculated with the "n-1" method for large samples and calculated as follows:

$$STDEV = \sqrt{\frac{\sum(X-\bar{X})^2}{n-1}} \quad 4.1.$$

Where: n is the number of samples, X is the gold value of the sample and \bar{X} is the average gold value.

The SDEV for the gold values calculated from the dump was: 1.432 ± 0.229 g/t Au.

The gold value for the dump was similar to the original values determined and was evenly spread throughout the dump. The ± 0.229 g/t Au deviation from the average of 1.432 g/t Au head grade indicated that the gold grade in the dump was relatively constant, both horizontally and vertically, which indicated that although the dump was deposited more than thirty years ago, limited vertical migration of the gold had taken place. The normal tendency for free gold particles is to migrate downwards with time. The constant gold value for the vertical sampling indicated that a large portion of the gold was associated with hematite or silica and the free gold particles were the exception rather than the rule.

Although free gold was visible on the SEM photographic images, it is more likely that the gold was attached to the hematite or to the silica, making it immobile.

4.1.2. Results reported on the bulk sample parameters.

The bulk sample was analysed to determine the particle size and mass distribution. Chemical analysis was done to determine the gold head grade and spread in chemical composition.

4.1.2.1. Results on the particle size distribution.

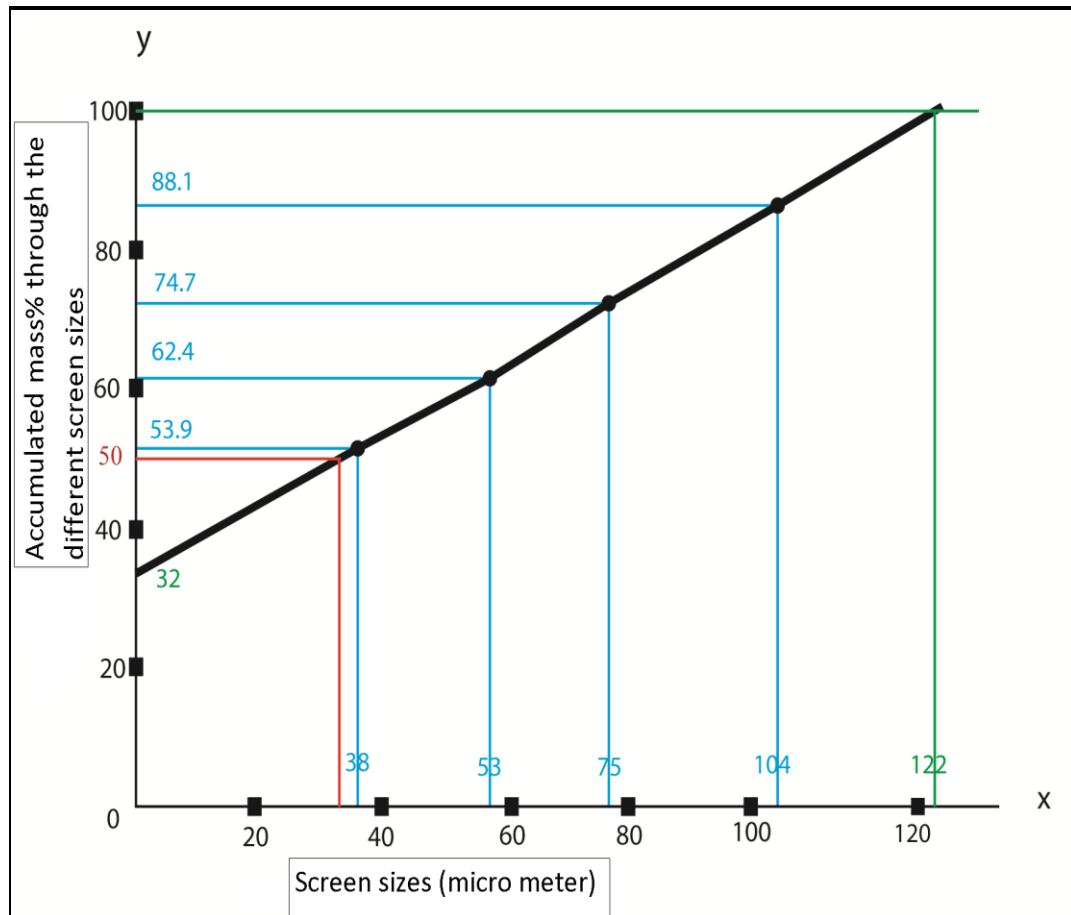
A representative sample was dried, screened and every fraction was weighed to determine the percentage retained by the specific sieve sizes, and reported in Table 3.2. A large portion of the material, more than 50%, still passed through the -38 μ m sieve as shown in Table 4.2.

Table 4.2: The size distribution results of the bulk sample and the accumulative mass values passed through the different screen sizes.

Fraction screened in microns	Mass% on screen	Accumulated mass% on screen
+104	11.9	100
-104+75	13.4	88.1
-75+53	12.3	74.7
-53+38	8.5	62.4
-38	53.9	53.9

4.1.2.1.1. Discussion on the results of the particle size distribution measurements.

In Table 4.2 the size distribution results of the bulk sample was reported, showing that more than 50% of the material still passed through the 38 μ m screen. The cumulative percentage of material passing through each screen was calculated and is shown in Table 4.2 and also plotted graphically in Graph 4.1. The 50% retention value is clearly indicated on Graph 4.1. This was done to illustrate the spread of particles smaller and bigger than the 38 μ m parameter. A trend line was also drawn to show the relative spread in screen sizes; this was done because the screens were not chosen with frequent size intervals, but according to the sizes normally used in the mining industry as size indicators.



Graph 4.1: Graphical depiction of the accumulated mass% of the sample that passed through the different screen sizes

.

An extrapolation of the size distribution line on Graph 4.1 clearly indicates the average particle sizes were smaller than 38 μ m. On the + 0 μ m screen all the

material was collected that passed through the 38 μm screen, accounting for almost 54% of the material.

4.1.2.1.2. Results of the gold content or head grade of the bulk sample.

The average gold content for the different particle sizes were determined by analysing the samples by means of an ICP technique. The gold samples' value-average was about 2 g/t Au except for the +104 μm fraction that was exceptionally low, as shown in Table 4.3. The lower value of the rest in the +38-53 fraction could not be explained and the lab discarded the remainder of the sample before a retesting could be done. It was assumed that the value was similar to the rest of the gold values.

Table 4.3: The gold analysis of the head grade from the bulk sample done with the ICP analyser.

The gold values of the size distribution	
Fractions screened in micrometres	Gold concentration (g/t Au)
+104	0.32
-104+75	2.44
-75+53	2.14
-53+38	1.04
-38	2.44
Reconstructed head	2.036
Actual head	2.46

4.1.2.1.3. Discussion of the results of the head grade of the bulk sample.

The average gold content was obtained by fire assay analysis of a representative sample out of the bulk sample. This value was reported as the “actual” head grade of the bulk sample (2.46 g/t). The average of the gold samples analysed by the ICP was about 2.0 g/t and differed from the fire assay. The standard ICP protocol used to analyse the samples did not compensate for the high iron content of the samples. The interference of the high iron value could explain the difference in the Au values obtained in the two analytical methods used.

For the purpose of this project it was decided to use the gold value from the fire assay results and to use the ICP analysis only for the spread in chemical composition.

4.1.2.2. Results of the spread in chemical composition of the bulk sample.

The three elements of interest, Fe, Si and Au, were analysed simultaneously, verifying the chemical composition and their constancy in the different particle sizes. The values of the chemical compositions were tabled in Table 4.4. Notable was the inconsistency of the +104 μm fraction values compared to the rest of the sample values. The average element values for the bulk sample are shown in Table 4.5.

Table 4.4: The chemical composition spread analysed over the particle sizes distribution.

Mineral value distribution of the different sizes				
Fraction screened in micrometres	Mass% of material	Au (g/t)	Si (%)	Fe (%)
+104	11.9	0.32	28.25	38.24
-104+75	13.4	2.44	22.2	39.72
-75+53	12.3	2.14	20.66	40.1
-53+38	8.5	1.04	20.04	40.18
-38	53.9	2.44	18.16	40.96
Reconstructed Head	100	2.036	20.37	40.3
Actual Head		2.46	20.2	40.25

Table 4.5: The average analyzed values of the main minerals.

Average concentrations of the minerals		
Au (gold)	2.46	g/t
Fe (Iron)	40.25	%m/m
Si (Silica)	20.2	%m/m

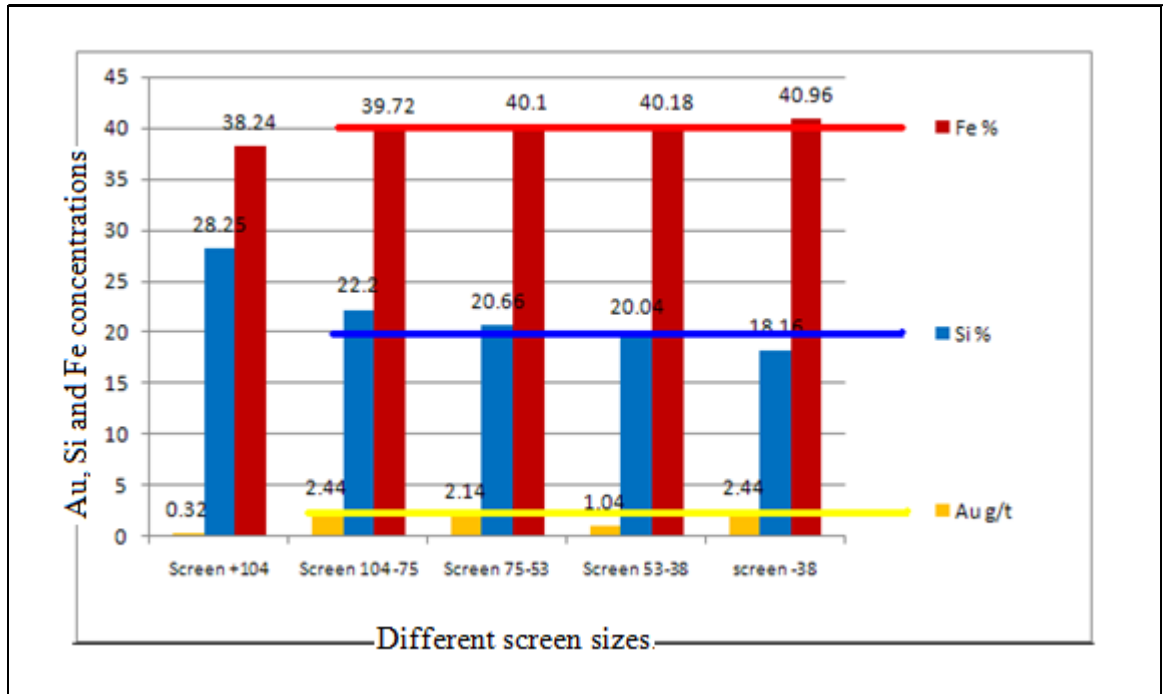
Note that the elements were reported as elements, the balance was made up by oxygen and sulphur. (40% Fe represented 60% Fe₂O₃ and 20% Si represented 40% SiO₂).

4.1.2.2.1. Discussion of the spread in chemical composition

The values 2.46 g/t for Au, about 20% for Si and about 40% for Fe, were considered as constant over the range of partial sizes. The sizes above the 104 μm fraction differed from these values. This contamination of the bulk sample material with soil under the dump was as a result of wrong excavation of the sample. The indication from the size distribution experiment results were that the particle sizes would rather be smaller than 100 μm sizes. The high silica value 28.25% and corresponding lower gold 0.32 g/t concentration and iron value of 38.24% also supported the presence of footprint material in the +104 μm fraction.

In the design of the pilot plant, the +100 μm fraction was discarded because of the deviation in chemical composition compared to the rest of the material. The volume of material not processed, above 100 μm , represented less than 12% of the sample volume.

A graphical presentation is shown in Graph 4.2 with the element values relative to each other, plotted against the relative screened sizes. Graph 4.2 also clearly shows the inconsistency in the +104 μm fraction values in concentration of minerals to the values of the values of the other screen sizes.



Graph 4.2: Graphical representation of the results obtained for the relative Au, Si and Fe contents analysed at different screen sizes.

In Graph 4.2, the chemical composition results are shown relative to each other and the trend lines highlight the content difference in the values for each element.

- 1) Iron was constant with little deviation from 40% in all the fractions.
- 2) When the +100 μm fraction was discarded, the silica values were more or less constant at about 20% Si for all the other screen sizes.
- 3) The same trend was experienced with the gold values, with the +100 μm fraction left out, the calculated gold value would be 2.26 g/t on average as shown in Table 4.6, which corresponds better with the actual head grade of 2.46 g/t rather than with the 2.036 g/t Au reported in Table 4.4.

Table 4.6: The calculated gold values excluding the +100 µm fraction.

The gold values of the size distribution without the +104µm fraction			
Fractions screened in micrometres	Gold concentration (g/t Au)	Mass retained on screen	Calculated gold mass
+104	0.32	-	-
-104+75	2.44	13.4	32.7
-75+53	2.14	12.3	26.32
-53+38	1.04	8.5	8.84
-38	2.44	53.9	131.52
Reconstructed head	2.036	88.1	2.26
Actual head	2.46		

4.1.2.2.2. The interpretation of the bulk sample parameters.

The bulk sample consisted of 40% Fe and Si at 20% as the main phases of the material in addition to the 2.5 g/t Au. The distribution of these elements was evenly throughout the size fractions as shown in Graph 4.2. It was also noted that 60% to 65% of the material was in the minus 50 µm fraction of the material. The theoretical recovery according to the industry was expected to be below 30% of the gold content due to the small average particle size. The particle sizes were found to be slightly larger than the reported values of Ergo where 50% was smaller than 19 µm (Graph 6.1., Appendix 1). This difference emphasizes the need to have determined the size distribution accurately. The average size was measured at 53.9% smaller than 38 µm, still well below the size regarded for good separation under normal conditions.

4.2. Photographic images obtained with the SEM showing the different mineral particles and their association to each other.

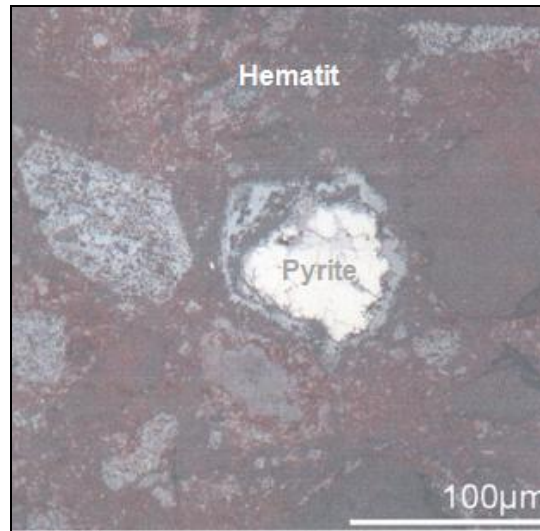


Figure 4.1. SEM photographic image of the material showing pyrite surrounded with hematite.

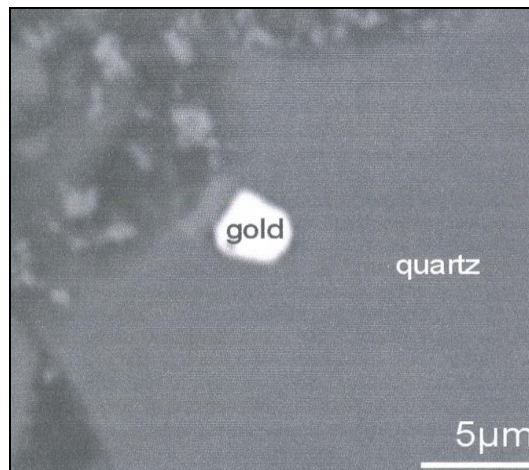


Figure 4.2. SEM photographic image of the material showing gold particle surrounded with silica.

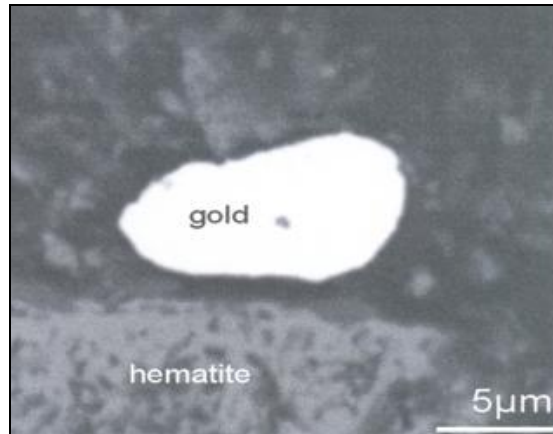


Figure 4.3. SEM photographic image of the material showing free gold particle with hematite next to it.

4.2.1. Discussion of the photographic images taken with the SEM showing the particles association with each other.

In Figure 4.1 a particle of pyrite can clearly be seen with hematite particles around it. The particle sizes were not well defined, but compared to the microscope bar scale, the particles measured smaller than 100 μm . Some of the pyrite was not converted to hematite due to an incomplete roasting process and it could be expected that magnetite (Fe_3O_4) would also be present. Magnetite would be formed at low oxygen pressure during the oxidation in the roasting process; with ample oxygen all the pyrite would have been converted to hematite.

In the Figure 4.2, a 4 μm gold particle can clearly be seen inside a much larger silica particle. The gold in this association would increase the particle's specific gravity, making it possible for the particle to be selected with the gold concentrate.

In Figure 4.3, a 10 μm free gold particle with rather large diameter can clearly be seen. These large free gold particles are common in cyanide processing, especially where there is a high concentration of other metals present, like the Fe concentration in the ore investigated in the present investigation. In the “high

volume gold plants”, negative factors like short plant retention times, high base metal content and relative large gold particles, the cyanide process has a tendency to form an oxide body around the gold. It is impossible for the cyanide to dissolve the entire gold particle under these conditions in a short leaching process. This explains the gold particles that were pumped away to the calcine waste dump as seen in this image.

4.3. Results obtained for the determination of the mineralogical aspects.

4.3.1. Results of the flotation experiment.

The batch flotation cell produced a small amount of material in the foam collected as a flotation concentrate, being only 0.6% of the material floated. From Table 4.7 it can be seen that the flotation concentrate had the highest Au concentration in comparison to the actual head grade. The majority of material did not bind to the flotation chemicals and could not be floated, and the Fe concentration was relatively constant in all the fractions.

Table 4.7: The results from the flotation experiment.

Fraction	Mass%	Au (g/t)	Fe (%)
Floating concentrate	0.6	3.92	39.16
Sinking	99.4	2.44	40.12
Reconstructed Head	100	2.44	39.92
Actual Head		2.46	40.25

4.3.1.1. Discussion and interpretation on the results obtained for the flotation experiment.

In the flotation experiment, the volume flotation concentrate collected was only 0.6% of the material and the gold content of the concentrate was only 3.92 g/t.

This poor recovery was not considered significant enough to investigate flotation as a separation technique in this investigation.

The poor flotation values could be as a result of the roasting process that converted most of the pyrite to hematite, destroying the needed sulphide binding sites, preventing the floating chemicals to attach to the sulphides and making the particles on longer floatable. The gold association shown in Figure 4.2, inside the non-floatable materials, also contribute to the poor flotation results.

4.3.2. Results on the magnetic separation experiment.

The magnetic separator separated the magnetite better than the hematite due to the higher magnetic properties of the magnetite. In Table 4.8 this can clearly be seen in the higher Fe values of the magnetic fraction compared to the non-magnetic fraction. The magnetic fractions have a higher gold concentration, directly related to the gold associated with the magnetite.

Table 4.8: The results from the magnetic separation experiment.

Fraction	Mass%	Au (g/t)	Fe (%)
Magnetic	6.8	10.2	64.66
Non-magnetic	93.2	1.92	38.48
Reconstructed Head		2.48	40.26
Actual Head		2.46	40.25

4.3.2.1. Discussion and interpretation of the results obtained for the magnetic separation experiment.

In the magnetic separation experiment, the magnetic component, (Fe) was recovered directly proportional to the magnetite concentration. This could clearly be seen in the black colour of the recovered material, as supposed to the red of the

hematite, and furthermore, magnetite is more magnetic than hematite. Also, the higher Fe value of the concentrate supported the presence of magnetite. It would be expected that the hematite and magnetite will react similar to the high acceleration separation because their specific gravities are similar.

The result of the magnetic separation experiment was not related to acceleration separation techniques, ruling out the interest of this project to pursue magnetic separation further, but for the purpose of maximizing the gold recovery it would be important to recover the magnetic component separately. When subjecting the material to magnetic forces, as during a magnetic separation, the magnetite keeps a magnetic memory, causing the remaining magnetic particles to flocculate, thus influencing the particle sizing drastically. The incorporation of a magnetic separator in the pilot plant design would have an effect on particle sizes and influence the separation. The extraction of the magnetic fraction would also change the composition of the material and the results of the gravity separation would no longer be related to the feed material.

4.3.3. Results on the hydraulic separation experiment.

From table 4.9 it can be seen that there is a 100% increase in the concentration of the Au and Fe from the tailings to the concentrate, but the upgrading of Au shows significant upgrading has not taken place, if it is compared to the reconstructed head grade.

Table 4.9: The results from the hydraulic separation experiment.

Fraction	Mass %	A (g/t)	Fe (%)	Au %
Overflow	30.6	3.94	62.16	49.39
Underflow	69.4	1.78	30.12	50.60
Reconstructed Head	100	2.44	39.92	100.00
Actual Head		2.46	40.25	

4.3.3.1. The results of the hydraulic classification experiment discussed and interpreted

With the experiment on the hydraulic classification, a new technology was applied, capable of recovering free gold very successfully. The gold recovered was 49% of the gold in 30.6% of the material, with a concentration of 3.94 g/t Au.

In this experiment the concentration of the gold was not good enough for a pre-concentrator and the volume of concentrate recovered was too large for the Itomak's capacity. The poor upgrading was mainly as a result of the high settling rate of the hematite from the suspension. The MPC5 machine makes use of normal gravity force for separation. The water flow was increased to counter the high settling rate, causing more turbulence, and the turbulence kept the small gold particles in suspension and reduced the particles fall-out properties.

4.3.4. Results on the dense medium separation experiment.

In Table 4.10 it is seen that the heavy fraction, containing more gold, sank through the dense medium, which shows the highest concentration of gold compared to the fraction that floated over the bed.

Table 4.10: The results from the dense media separation experiment.

Fraction	Mass %	A (g/t)	Fe (%)	Au %
Float	97.2	2.15	38.9	86.51
Sinks	2.8	11.64	63.88	13.49
Reconstructed Head	100	2.42	39.6	100.00
Actual Head		2.46	40.25	

4.3.4.1. The results discussed and interpreted obtained from the dense media separation experiment.

The dense media separation experiment worked well and the material was upgraded to 11.64 g/t Au as reported in the concentrate, but the total volume recovered was only 2.8% of the material, which represents 13% of the gold, which in turn, indicates that the material reacted positively to separation by using gravity forces. The free gold particles, seen on the SEM slides, could also contribute to this result.

4.3.5. Results on the multi-gravity separation experiment.

In Table 4.11 the results show that both the Fe and Au are present in much higher concentrations in the concentrate than in the tailings. The increase in cogeneration is almost 100% for the Au and 50% for the increase in Fe, compared with the reconstructed head grade of the minerals.

Table 4.11: The results from the multi-gravity separation experiment.

Fraction	Mass %	A (g/t)	Fe (%)	Au %
Concentrate	20.8	5.12	57.46	43.00
Middling	56.6	2.04	39.14	40.00
Tailing	22.6	1.18	27.15	17.00
Reconstructed Head	100	2.49	40.24	100.00

4.3.5.1. Discussion and interpretation of the results obtained from the multi-gravity separation experiment.

The multi-gravity separation technique upgraded the material to about 20.8% concentrate at a gold grade of 5.11 g/t Au. This experiment was also a good indicator that the material was susceptible to gravity separation.

This technique is very sensitive to the particle size variation and variation in flow rates. The sensitivity towards variation in the flow rates made it impractical to use in line with other equipment, like the Itomak, in a batch process configuration.

4.3.6. Results on the tabling experiment.

In table 4.12, the Au concentration of the concentrate is much higher compared to the middling and the tailings fraction. The Fe concentration is almost constant, both in the concentrates and middling fractions, but much reduced in the tailing fraction.

Table 4.12: The results of the tabling experiment.

Fraction	Mass %	Au (g/t)	Fe (g/t)	Relative Au value	Relative Au (%)
Concentrate heavy 1	0.8	11.16	65.88	0.089	4.00
Concentrate heavy 2	1.6	8.2	63.55	0.131	5.00
Total concentrate	2.4	9.4		0.220	9.00
Middlings1	12.5	4.58	62.42	0.572	23.00
Middlings2	20.1	3.02	62.04	0.607	25.00
Total	35.00	4.0		1.40	57.00
Tailing light	65.0	1.62	28.65	1.053	43.00
Reconstructed Head	100	2.45	40.44	2.453	100.00
Actual Head		2.46	40.25	8.928	

4.3.6.1. The tabling experiment's results discussed and interpreted.

The experiments done on the James table showed an upgrading of the material to be significant, but a very small recovery of only 2.4% of the material was collected as concentrate with a value of 9.40 g/t Au. Although this indicated that the material reacted to gravity separation and could be useful in this investigation, the overall recovery of the gold was still not very promising with 8.99% material recovered as concentrate.

The combined middling and concentrate fraction accounted for 57% of the gold. It was possible to increase the gold recovery by adding the volume of middling to the concentrate, resulting in a viable pre-concentration technique.

The tabling experiment made a clear colour distinction between the concentrate, middling and tailings. This colour difference was helpful in designing the pilot plant and adjusting its collectors on the spirals.

The concentrate had high gold value, the middling fraction was high in iron and the tailings were high in silica. This corresponds with the specific gravity of these elements.

4.4. Evaluating the results of the determination of the mineralogical aspects.

The mineralogical experiments were evaluated by comparing the relative gold mass of each experiment and calculating the relative per cent of gold recovered in the corresponding experiment. These values were reported in Table 4.13.

Table 4.13: The table of mass balance done of the mineralogical experiments.

	Mass%	Au concentration g/t	Relative gold mass	Relative gold % recovered
Flotation	0.6	3.92	2	1
Magnetic separation	6.6	10.2	67	27
Hydraulic classification	30.6	3.94	121	49
Dense medium separation	2.8	11.64	33	13
Multi-gravity separation	20.8	5.12	106	43
Tabling	35	4.0	140	57

4.4.1. The mass balance of the mineralogical experiments discussed and interpreted.

In Table 4.14 a mass balance was calculated for every mineralogical test done, to determine the most effective upgrading technique to be used as a pre-concentration step in the pilot plant. The tabling showed the best recovery of 57% Au but, hydraulic classification and multi-gravity separation also proved to be viable, judging from the mass balance calculations. There were other factors that also played a role in the choice of the pre-concentration step.

The water flow of the multi-gravity separators were too large for the pilot plant setup and by reducing the water flow the separation was compromised. The hydraulic classifier was physically too large for the pilot plant area available. The only option left was the James table, but the volume of material that could be processed was only 60 kg/h. The spiral columns used to perform this function produced similar results using the same separation principle. The spiral column's results were shown with the pilot plant results, all other the features of the spirals were compatible with the pilot plant configuration.

Chapter 5

The pilot plant experiment and discussion.

In this chapter, the relevant mineralogical experiments and the results were incorporated in the designing of the pilot plant experiment. The pilot plant configuration is discussed in this chapter and the result of the pilot plant experiment is reported. Finally the results of the pilot plant are discussed and an interpretation of the results is given.

5.1 The pilot plant configuration.

The chemical composition analysed in the size distribution experiment showed that the particles above 100 μm were not uniform compared to the rest of the material. Contamination with the soil under the dump was expected, and this fraction was eliminated by screening the material, as it left the mixing tank. Only the minus 100 μm fraction was used to pump to the Rougher spiral column, for the first stage separation.

The second part of the pilot plant configuration was mainly based on the values obtained from the James table experiment. The spiral collectors were adjusted according to the chemical distribution of the minerals (Table 4.2) and the visual separation on the James table; 10% concentrate, 70% middling and 20% tailing.

The pilot plant configuration was done with a double spiral (gravity separation processes) reducing the volume of material going through to the final stage of high acceleration separation done in the Itomak. To increase the effectiveness of the Rougher spiral, a recycled feed from the Scavenger spiral concentrate was incorporated. Recycling was done by sending the tailings from the Rougher spiral, through the Scavenger spiral and then the concentrates of the Scavenger were sent back to the Rougher spiral feed.

A final stage high acceleration separation was done with a Itomak high acceleration separator. There was also a second recycling done by sending the Itomak tailing back to the Rougher spiral feed.

The middling stream came from the Rougher spiral, and the tailings stream was collected from the Scavenger spiral and the concentrate stream from the Itomak concentrate stream.

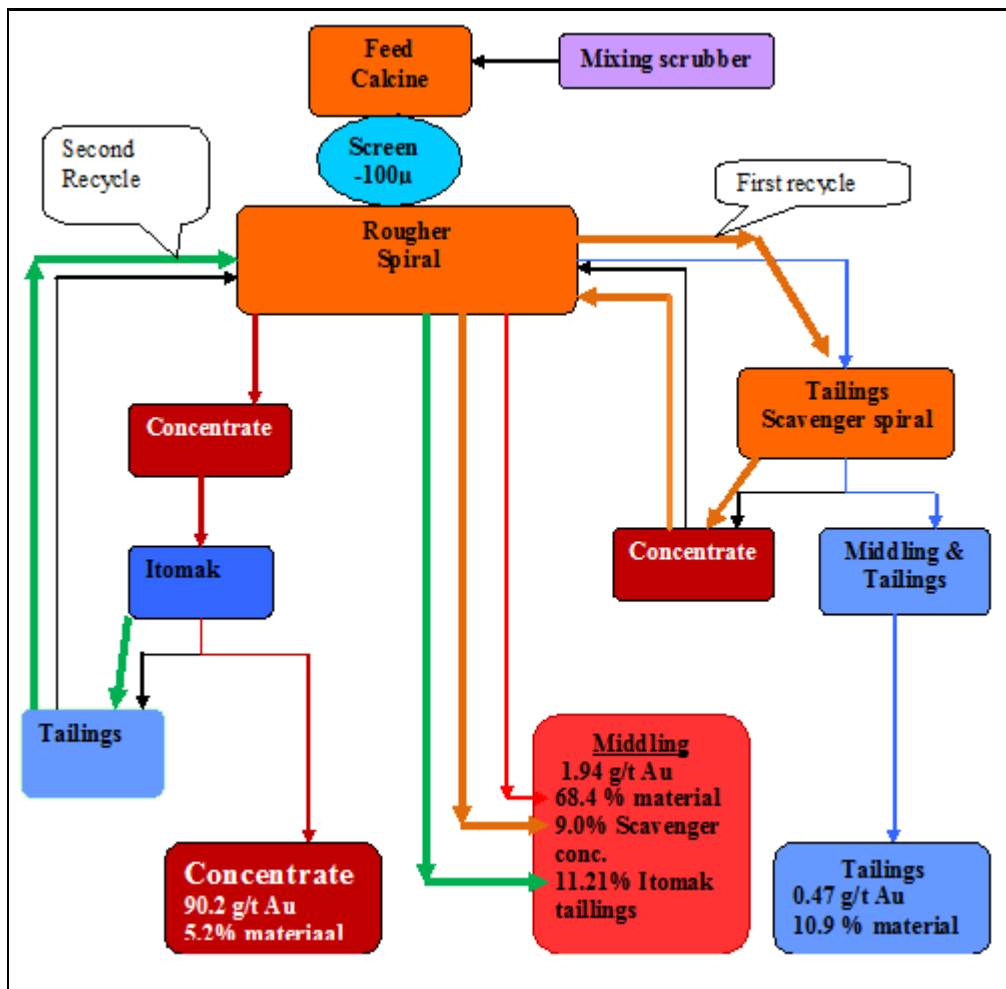


Figure 5.1: Diagrammatic representation of how the final pilot plant was configured.

5.1.1. Experimental procedures used in the operation of the pilot plant.

Calcine from the bulk sample was conveyed into a 2000ℓ mixing tank with a feed belt, at a rate of 400 kg per hour. Water was added to the suspension to maintain a 30% m/m suspension feed to the Rougher spiral column. The suspension settled fast and continuous mixing was maintained by recycle pumping and agitation throughout the experiment.

The material was pumped to the Rougher spiral after particle classification (100 µm screen) at a rate of 10% of the recycle pumped stream.

The collectors on the column split the effluent in three streams:

- 20% tailings (light fraction)
- 70% middling
- 10% concentrate (heavy fraction)

The first recycling was done by pumping the Rougher tailings through a Scavenger spiral column. The heavier concentrate fraction of the Scavenger spiral was sent back to the Rougher spiral column, and the light and middling fraction was collected as tailings. The quantity and composition of each sample was analysed from the samples collected.

The concentrate from the Rougher spiral column was sent to the Itomak and the tailings from the Itomak were sent back to the Rougher spiral column, for the second recycling circuit.

The middling fraction was collected from the Rougher spiral column's middling stream and the tailings were collected from the Scavenger spiral middling and tailing streams combined. The 5% high gold concentrate was collected from the Itomak concentrate and analysed for gold content.

The plant was run continuously until twenty tons of the material were processed. A mass balance was calculated out of the representative samples collected from the different streams.

The samples were analysed and the results were reported as follows:

- Rougher spiral samples; concentrate **middling** and tailings.
- Scavenger spiral samples; concentrate and **middling & tailings**
- Itomak samples; **concentrate** and tailings

5.2. Results obtained for the pilot plant experiment.

In table 5.1, the first spiral Rougher produced the middling fraction of about 70% of the material. The Au value in the tailings was considerable lower compared to the concentrate's Au value. The Fe was high in the concentrate and very low in the tailing, indicating that separation took place.

In the second spiral, the Rougher spiral's tailings were recycled and it contained less than 0.5 g/t Au when discharged as tailings.

The concentrate fraction of the Rougher was upgraded in the Itomak from 8 g/t Au to over 90 g/t Au. This section represented the high acceleration separation phase of the pilot plant. Increased upgrading took place compared to the spiral's low gravity separation results. The Fe was already upgraded to the maximum and only a slight increase in Fe concentration could be seen here.

Table 5.1: The results from the pilot plant experiment.

Fraction	Mass%	Au(g/t) concentration	Fe (% m/m) concentration	Au %	Fe %
Rougher concentrate	11.8	8	66.85	38.27	19.44
Rougher tailings	19.8	0.99	22.23	7.95	10.85
Rougher middling	68.4	1.94	41.36	53.78	69.71
Scavenger concentrate	45.2	3.82	65.64	87.02	28.06
Scavenger middling & tailings	34.8	0.47	21.12	12.98	71.84
Itomak concentrate	5.2	90.2	66.12	58.43	5.15
Itomak tailings	94.8	3.52	66.84	41.57	94.85
Reconstructed head	100			100	100
Actual head		2.46	40.25		

5.2.1. Discussions on the results of the pilot plant experiment.

The pilot plant ran continuously and processed 20 tons of feed material from the bulk sample. Sampling was done continuously with regular intervals to compile a representative sample from every stream. A mass balance was calculated and reported as a percentage of the masses recovered.

In the first separation stage on the Rougher spirals a concentration increase of gold from 2.46 g/t Au head grade to 8 g/t Au was encountered. It was calculated that almost 40% of the gold was concentrated into 12% of the material.

The Rougher spiral collectors were adjusted to split the effluent stream in three separate streams, namely a high gold concentrate fraction, iron oxides fraction and high silica fraction. The tailings from the Rougher spirals were recycled through the Scavenger spirals. The middling and tailings of the Scavenger spirals were collected as pilot plant tailings. The tailings collected at the Scavenger spirals had only 0.7% of the gold and represented 11% of the original material with a gold grade of less than 0.5 g/t.

In the high acceleration separation, the Itomak further concentrated the Au, coming from the Rougher spiral, from 8 g/t Au to 90 g/t Au. The Itomak produced 5% concentrate material with a gold content of 90 g/t Au. The gold recovered by the high acceleration separation in the experiment, was almost 60%.

The overall upgrading of gold was from the 2.46 g/t Au head grade to 8 g/t Au with the spiral column (low acceleration, gravity), which represents a one to three upgrading of the gold concentration.

In the Itomak high acceleration the gold was upgraded from 8 g/t Au to over 90 g/t Au and represents an eleven fold upgrading.

The overall pilot plant recovery was 22% of the gold, recovered in 5% of the material, with a gold concentration of 90 g/t Au. The gold recovered in the high acceleration separation stage of the pilot plant was almost 60%.

5.2.2. Interpretation of the results from the pilot plant.

In the first stage of the plant that operated under normal gravity acceleration force of 9.8 m/s^2 , the recovery of gold was 40%. This recovery was slightly higher than expected, but better than the recoveries reported from normal gravity separation processes, where silica was the main phase of the material. The recovery would be in the mid 20% or even lower for the corresponding particle size distribution,

had the main phase been silica. The recovery showed almost double this recovery at 40% of the gold in only 12% of the material, collected at the Rougher concentrate. This phase of the pilot plant already indicated a much higher recovery because of the increased specific gravity of the elements in suspension. The stage was set for maximizing this effect by increasing the acceleration forces, the only variable parameter available.

The second phase, where high acceleration forces were applied to separate the gold, started with a head grade of 8.0 g/t Au. The silica was now almost 100% eliminated from the suspension.

There were two prominent factors influencing the recovery in this stage:

1. The spiral columns reduced the volume of feed material, but also selected the heavier material already responding positively to normal gravity force influences.
2. The second factor influencing the separation was that the tailings stream of the Scavenger spiral removed most of the silica in circulation, contributing to the increased hematite concentration of the suspension.

The specific gravity of the average particles in the suspension was now running much more uniform and close to 5 g/cm³, positively contributing to the separation process. The physical volume of gold collected made it possible to refine the gold, proving the process to the end.

Chapter 6

Conclusions

In this chapter, conclusions are made on the chemical composition, the associations of the minerals with each other and the reaction to mineralogical separation processes in conjunction with the success of the pilot plant. An evaluation of the hypothesis was also made as to the effectiveness of applying high acceleration forces on material with high specific gravity and whether the small particle sizes still resulted in poor separation results. Certain recommendations are made based on the results of this investigation, which could maximize the recovery of gold in this material.

The key to unlock the minerals with the separation process in the present investigation was to know exactly what the mineral composition and association to each other were. The physical properties and reaction to physical elements were also investigated in this project. The best gravity separation technique was employed to increase the sample volume to be processed. The Itomak a high acceleration separation machine was used for the final separation stage. The final step was to evaluate the results and draw a conclusion regarding the influence of particles with a high specific gravity on the separation process for smaller particle sizes.

The dump was accurately sampled and analysed with a head grade of 1.432 g/t, corresponding to the historical value obtained from the Ergo data, namely 1.446 g/t (Appendix 1.2). It was concluded from the results obtained for the gold values that the gold did not migrate vertically over time through the dump, but indicated that the gold was mostly associated with the other minerals present in the dump. A further prove of this associations was found when the SEM images of the material were studied. The gold particles could clearly be seen inside some of the other mineral particles. The size distribution experiment was done simultaneously with the spread in chemical composition and mass balance experiment, which proved that the minerals were evenly spread throughout the material and the average particle size, was below the normal size for effective separation, at 53% smaller than 38 μm . The mineralogical experiment that proved to give the best

gravity separation results was the tabling experiment, but the low capacity made the James table impractical to be used as a pre-concentration stage for the high acceleration separation to follow. The next alternative was to use spiral columns as a pre-concentrating phase, using the same separation principle as the James table, but having the capability of handling much larger volumes.

The bulk sample was processed through the pilot plant, screening off the +100 μm fraction and thus eliminating from the rest of the fractions the chemical inconsistency of this fraction. The next phase, in the spiral columns, reduced the volume of material to a volume that the Itomak high acceleration concentrator could process. The high acceleration separation of the material, with a high hematite concentration resulted in effective upgrading of the gold concentration. The experiment proved that the high specific gravity of the particles in suspension increased separation where high acceleration forces were applied, although the particle sizes were considered too small for effective gravity separation. The influence of the hematite's high specific gravity and the increase in the acceleration force increased the fall-out velocity which made the separation more effective. The increased effect of the acceleration forces could clearly be seen in the different results of the spiral column's upgrading of 1:3 under normal gravity forces and the Itomak upgrading of 1:11 at accelerated acceleration forces.

The hypothesis was that the higher the specific gravity of the particles, making up the main phase of a suspension, the more it would influence the high acceleration separation process positively and make separation effective for a material with much smaller particle size.

The hypothesis was proven correct; calcine with high specific gravity, in this case a gold-rich hematite ore body, was effectively concentrated to 90 g/t Au with a high acceleration separation process, although the average particle sizes were smaller than 38 μm . High acceleration forces were applied effectively to concentrate 60% of the gold to a concentrate of above 20 g/t, making it an extractable process that is economical viable.

The investigation's original aim was to produce a high grade hematite Fe_2O_3 for pigment production and in addition to this a gold concentrate that would be viable to process and recover the gold. This approach led to some features in the pilot plant that could be altered to recover more gold. The first recommendation is the

addition to the pilot plant would be to incorporate a second Itomak to upgrade the middling stream of the Rougher spiral column. The second addition to the plant is certainly to add a magnetic separator to recover the magnetite. By incorporating a magnetic separation process an additional volume of the gold could be recovered, as a result of the magnetite's high gold content.

References

Aksoy, B.S. and Yarar, B., 1989. *Natural hydrophobicity of native gold flakes and their flotation under different conditions*. Processing of Complex Ores. Ed. G.S. Dobby and S.R. Rao. Pergamon Press, N.Y., pp. 19-27,

Allison, S.A., Dunne, R.C. and De Waal, S.A., 1982. *The flotation of gold and pyrite from South African gold-mine residues*. XIV Int. Min. Proc. Congress, Toronto, pp. 17-23, 11-9.1-II-9.18.

Anhaeusser, C.R., Feather, C.E., Liebenberg, W.R., Smits, G. & Snegg, J.A., 1987. *The geology and mineralogy of the principal goldfields in South Africa*. The Extractive Metallurgy of Gold in South Africa. It. Stanley G.G. (Ed.). S. Afr. Inst. Min. Metall, pp. 1-67.

Birch, B., February 2011. *SBS; Gold; Science and Technology; Geology of gold*.

Coetzee, C.B., 1976. *Delfstowwe van die Republiek van Siud Afrika*. Vyfde uitgawe handbook 7, pp. 39-73.

Dandois, P. Ckeres, G. Joly, P. And Frenay, J., 1997. *Technical note development of a pilot plant using clean technology for recovery of gold from small scale mines*, Miner & Engineering, pp. 453-455.

De Sousa, Botelho A.M.R., O'Connor, C.T. & Dunne, R.C., 1986. *The influence of various, chemical, physical and mineralogical factors on the flotation of gold-bearing pyrites*. GOLD 100. Proceedings of the International Conference on Gold. Vol. 2: Extractive metallurgy of gold. Johannesburg. S. Afr. Inst. Min. Metall, pp. 493-503.

De Wet, J. R. Hodkinson, G. Pistorius, P. C. Prinsloo, L. C. and Sandenbergh, R. F. June 1995. *The influence of cyanide on the flotation of pyrite from Witwatersrand gold leach residues*, Minerals Engineering, pp. 1333.

Demopoulos, G.P. and Papangelakis, V.G., 1989. *Recent Advances in Refractory Gold Processing*, CIM Bulletin, 82(931), pp. 85-91.

Deng, T. And Liao, M., 1988. *Gold recovery enhancement from a refractory flotation concentrate beneficiation, with Special Reference to Arsenical Ores*, Journal of the South African Institute of Mining and Metallurgy, 88 (5), pp. 173-180.

Dry, M.J. and Coetzee, C.F.B., 1986. *'The Recovery of Gold from Refractory Ores'*, In *Proc. Of international Conference on Gold, Johannesburg, S. Africa*, pp. 259-274.

Kelly, E.G., Spottiswood, D.J., 1982, *Introduction to Mineral Processing*, pp. 68-70, 199-322.

Forrest, K., Yant, D. and Dune, R. 2001. *Optimisation of gold recovery by selective gold flotation for copper-gold-ores*, Minerals Engineering, Vol. 14, No. 2, pp. 227-241.

Frimmel, H.E., 2004. *Archaean atmospheric evolution: evidence from the Witwatersrand gold fields, South Africa*. Earth-Science Reviews, 70, pp. 1 – 46.

Johnson, M.R., Anhaeusser, C.R., Thomas, R.J., 2009. *The Geology of South Africa*., pp. 155-182.

Lange, L.H., 1935. *More facts on the flotation of free gold*. Eng. and Min., 136(3), pp. 116-118.

Lehmann, M.N., O'Leary, S., and Dunn, J.G., 1999, *An evaluation of pre-treatment to increase gold recovery from a refractory ore containing arsenopyrite and pyrrhotite*. Mineral Engineering, Vol. 13, No. 1, pp. 1-18.

Motherwell, W., 1914. *Flotation test at Mount Morgan*. Mining and Scientific Press, 53140.

Norman, N. And Whitfield, G., 2006. *De Beers Geological Journeys*, pp. 42-45, 320.

O'Connor, C.T., Bradshaw, D.J. and Upton, A.E., 1990. *The use of dithiophosphates and dithiocarbamates for the flotation of arsenopyrite*. Minerals Eng., 3(5), pp. 447-459.

O'Connor, C.T. and Dunne, R.C., 1994. *The flotation of gold bearing ores-A review*. Minerals Eng., 7(7), pp. 839-849.

Perry , R., *et al.* 1984. *Perry's Chemical engineers' handbook*; sixth Edition, 3-12/13/22.

Petela, R., 1993. *A design method for scale-up of selective agglomeration plant for upgrading coal with respect to sulphur and mineral matter content*. Science and Technology, Fuel 72 (4), pp. 511–516.

Skoog, D.A., Holler, F.J. & Niemand, T.A., 1998. *Principles of instrumental analysis*. 5th Ed. United States of America: Brooks/Cole Thomson Learning, pp. 262, 294, 550, 551, 552.

Swash, P.M., 1988. *A Mineralogical Investigation of Refractory Gold Ores and Their by sequential bioleaching and thiourea leach*, pp. 173-180.

Taggart, A.F., 1945. *Handbook of Mineral Dressing*. John Wiley and Sons, Inc., N.Y.

Tiedt, L.R. & Pretorius, W.E., 2006. *An introduction to electron microscopy and x-ray microanalysis*. Information pamphlet. North West University.

Tutu, H., McCarthy, T.S. & Cukrowska, E. 2008. *The chemical characteristics of acid mine drainage with particular reference to sources, distribution and remediation: The Witwatersrand Basin, South Africa as a case study*. Applied Geochemistry, 23(2008): 3 666 – 3 684.

Wang W. and Poling G.W., 1983. *Methods for recovering fine placer gold*. Canadian Mining and Metallurgical Bulletin, 76(860), pp. 47-56.

Worley, J. & Kvech, S. 2000. ICP-MS. *Environmental and Water Resource Engineering*. Virginia Tech: Department of Civil and Environmental Engineering, Virginia Tech, Blacks bury, VA. Available online at:
www.cee.vt.edu/ewr/environmental/teach/smprimer/icpms/icpms.htm, accessed 28 January 2010

Appendices

Appendix 1.1

Ergo report on the size distribution and gold content of the calcine dump.

Table of the results recorded by Ergo using a particle counter to measure the particle distribution in the sample suspension.

Result: Analysis Table

ID: 1030900654	Run No: 4	Measured: 1/10/2003 15:12PM
File: 30900654	Rec. No: 3	Analysed: 1/10/2003 15:12PM
Path: C:\SIZERS\DATA\		Source: Analysed

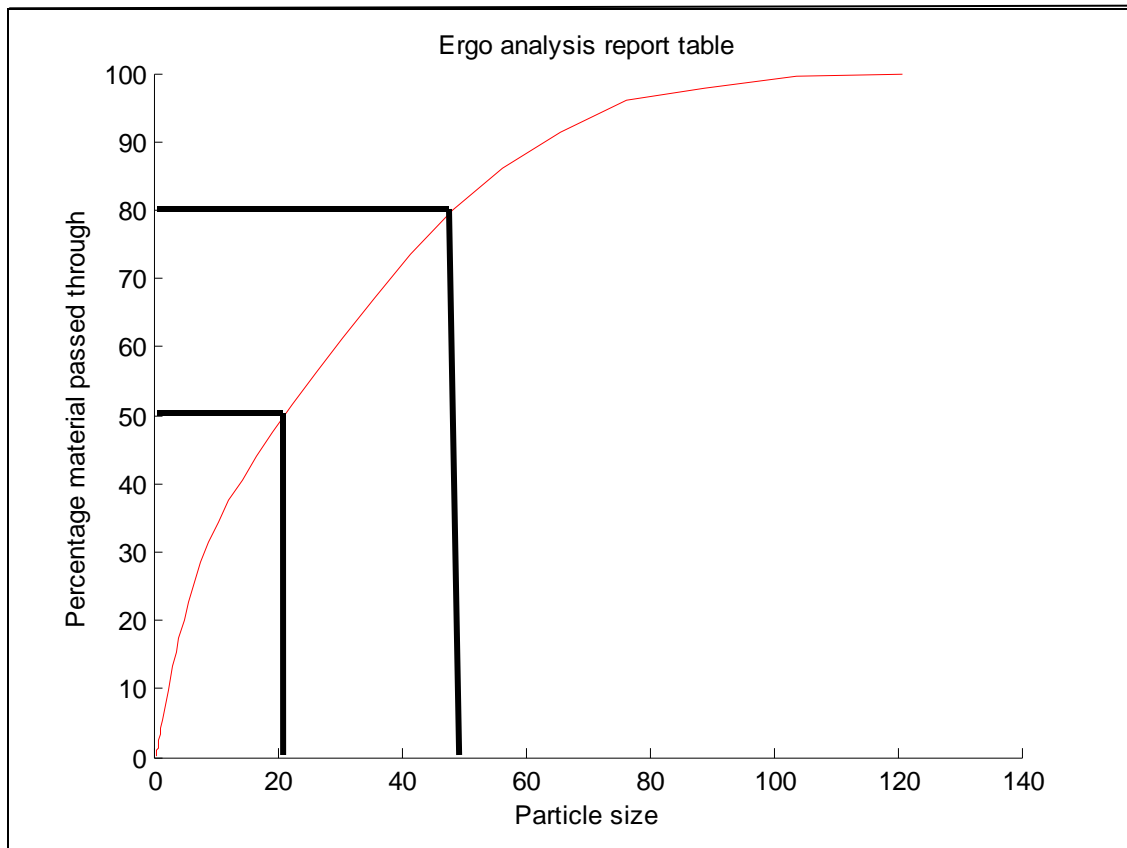
Range: 300RF mm	Beam: 2.40 mm	Sampler: MS17	Obs': 20.1 %
Presentation: 3\$\$D		Analysis: Polydisperse	Residual: 2.660 %
Modifications: None			

Conc. = 0.0185 %Vol	Density = 1.000 g/cm ³	S.S.A. = 1.0112 m ² /g
Distribution: Volume	D[4, 3] = 27.56 um	D[3, 2] = 5.93 um
D(v, 0.1) = 2.33 um	D(v, 0.5) = 21.25 um	D(v, 0.9) = 62.84 um
Span = 2.847E+00	Uniformity = 9.228E-01	

Size (um)	Volume Under%	Size (um)	Volume Under%	Size (um)	Volume Under%	Size (um)	Volume Under%
0.05	0.00	0.67	1.28	9.00	31.43	120.67	100.00
0.06	0.00	0.78	1.81	10.48	34.47	140.58	100.00
0.07	0.00	0.91	2.48	12.21	37.52	163.77	100.00
0.08	0.00	1.06	3.31	14.22	40.64	190.80	100.00
0.09	0.00	1.24	4.30	16.57	43.93	222.28	100.00
0.11	0.00	1.44	5.45	19.31	47.52	258.95	100.00
0.13	0.01	1.68	6.75	22.49	51.56	301.68	100.00
0.15	0.01	1.95	8.19	26.20	56.16	351.46	100.00
0.17	0.02	2.28	9.76	30.53	61.39	409.45	100.00
0.20	0.03	2.65	11.46	35.56	67.19	477.01	100.00
0.23	0.06	3.09	13.31	41.43	73.47	555.71	100.00
0.27	0.10	3.60	15.33	48.27	80.06	647.41	100.00
0.31	0.16	4.19	17.54	56.23	86.16	754.23	100.00
0.36	0.25	4.88	19.98	65.51	91.29	878.67	100.00
0.42	0.40	5.69	22.63	76.32	95.19		
0.49	0.60	6.63	25.46	88.91	97.88		
0.58	0.89	7.72	28.42	103.58	99.44		

Particle Size distribution of the Calcine.

Work done previously, shown in Graph 7.1 was statistically examined to determine the parameters of the dump.



Graph of the particle sizes measured and the percentage material passed through.

Rosin Rammler Parameters

1. Distribution Parameter = 0.908
2. Size Parameter = 28.850
3. 50% Passing Size = 19.268
4. 80% Passing Size = 48.724

Appendix 1.2

In the Table gold values analysed by Ergo was reported and the average gold content was calculated.

Table of Ergo gold sampling results documented previously.

no	Au g/t	no	Au g/t
1	1.49	16	1.43
2	1.53	17	1.38
3	1.72	18	1.4
4	1.51	19	1.32
5	1.44	20	1.388
6	1.43	21	1.458
7	1.34	22	1.415
8	1.37	23	1.458
9	1.5	24	1.333
10	1.41	25	1.476
11	1.35	26	1.538
12	1.38	27	1.506
13	1.19	28	1.446
14	1.61	sum	40.486
15	1.67	average	1.446

The average gold (Au) value determined in the past was 1.446 g/t Au.

Appendix 1.3

Calculations on the Itomak acceleration.

At the big end of the basket the diameter was 150 mm

The circumference at 150 mm diameter was:

$$C = 2\pi r \quad 1.3$$

With

$$C = 150 \text{ mm} \times \pi$$
$$= 471 \text{ mm}$$

and where: C is the circumference, and r is the radius of the basket measured in mm.

The velocity calculated at 150 mm diameter was:

$$\begin{aligned} \text{Velocity}_{150\text{mm}} &= \text{Circumference} \times \text{revolutions} \\ &= 471 \text{ mm/r} \times 8400 \text{ r/min} \\ &= 3956400 \text{ mm/min} \\ &= 39.564 \text{ m/s} \end{aligned}$$

At the small end of the basket the diameter was 75 mm.

The circumference at 75 mm diameter was:

$$\begin{aligned} C &= 2\pi r \\ &= 75 \text{ mm} \times \pi \\ &= 236 \text{ mm} \end{aligned}$$

Where: C is the circumference, and r is the radius of the basket measured in mm.

The velocity calculated at 75 mm diameter was:

$$\begin{aligned}\text{Velocity}_{75\text{mm}} &= \text{Circumference} \times \text{revolutions} \\ &= 236 \text{ mm/r} \times 8400 \text{ r/min} \\ &= 1982400 \text{ mm/min} \\ &= 19.824 \text{ m/s}\end{aligned}$$

According to Newton's second law of acceleration:

$$a = v^2/r \quad 1.4$$

Where:

a - acceleration

v - velocity

r - radius

$$\begin{aligned}\text{The acceleration at 150 mm diameter was: } a_{150} &= (39.564 \text{ m/s})^2/75 \text{ mm} \\ &= 117.4 \text{ m/s}^2\end{aligned}$$

$$\begin{aligned}\text{And the acceleration at 75 mm diameter was: } a_{75} &= (19.824 \text{ m/s})^2/37.5 \text{ mm} \\ &= 14.7 \text{ m/s}^2\end{aligned}$$

The fall-out velocity or settling or terminal velocity (V_t) from equation 1.1 calculated for the different rings

The fall-out velocity (V_t) on the 150mm diameter side of the basket for hematite was calculated as:

$$= 36.2 \text{ } \mu\text{m/second.}$$

Where:

$$g = \text{Acceleration} = 117.4 \text{ metre/ (second)}^2$$

$$d = \text{Particle diameter} = 0.038 \text{ millimetre}$$

$$\rho_p = \text{Density of particle} = 5120 \text{ kilogram/ (metre)}^3$$

$$\rho_m = \text{Density of medium} = 1200 \text{ kilogram/ (metre)}^3$$

$$\mu = \text{Viscosity of medium} = 1.02 \text{ kilogram/ (metre) (second)}$$

The fall-out velocity (V_t) on the 150 mm diameter side of the basket for gold was calculated as:

$$=167 \mu\text{m/second.}$$

Where:

$$\begin{aligned} g &= \text{Acceleration} &&= 117.4 \text{ metre/ (second)}^2 \\ d &= \text{Particle diameter} &&= 0.038 \text{ millimetre} \\ \rho_p &= \text{Density of particle} &&= 5120 \text{ kilogram/ (metre)}^3 \\ \rho_m &= \text{Density of medium} &&= 1200 \text{ kilogram/ (metre)}^3 \\ \mu &= \text{Viscosity of medium} &&= 1.02 \text{ kilogram/ (metre) (second)} \end{aligned}$$

The fall-out velocity (V_t) on the 75 mm diameter side of the basket for hematite was calculated as:

$$= 4.5 \mu\text{m/second.}$$

Where:

$$\begin{aligned} g &= \text{Acceleration} &&= 117.4 \text{ metre/ (second)}^2 \\ d &= \text{Particle diameter} &&= 0.038 \text{ millimetre} \\ \rho_p &= \text{Density of particle} &&= 5120 \text{ kilogram/ (metre)}^3 \\ \rho_m &= \text{Density of medium} &&= 1200 \text{ kilogram/ (metre)}^3 \\ \mu &= \text{Viscosity of medium} &&= 1.02 \text{ kilogram/ (metre) (second)} \end{aligned}$$

The fall-out velocity (V_t) on the 75 mm diameter side of the basket for gold was calculated as:

$$=20.9 \mu\text{m/second.}$$

Where:

$$\begin{aligned} g &= \text{Acceleration} &&= 117.4 \text{ metre/ (second)}^2 \\ d &= \text{Particle diameter} &&= 0.038 \text{ millimetre} \\ \rho_p &= \text{Density of particle} &&= 5120 \text{ kilogram/ (metre)}^3 \\ \rho_m &= \text{Density of medium} &&= 1200 \text{ kilogram/ (metre)}^3 \\ \mu &= \text{Viscosity of medium} &&= 1.02 \text{ kilogram/ (metre) (second)} \end{aligned}$$