The Effect of Circuit Configuration on Regrind Circuit Performance

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A dissertation submitted to the Faculty of Engineering and the Built Environment, the University of Cape Town, in fulfilment of the requirements for the degree of Master of Science in Engineering.

September 2004

Acknowledgements

I would like to express my sincere appreciation to everyone who contributed directly and indirectly to this research project.

Thanks to my supervisors Dr. Malcolm Powell and Mr Craig Sweet for their valuable input, guidance and assistance during the course of the research.

My appreciation also goes to Mr. Aubrey Mainza for the guidance and assistance in the experimental work and the endless discussions.

Thanks to Mr. Jason Waters for helping with the batch flotation tests and Mr. Reuben Moalosi for assistance in most of the test work.

To everyone in the Comminution Group and the overall Mineral Processing Research Unit, my appreciation cannot be fully expressed in words for the invaluable assistance.

To Anglo Platinum, thanks for letting me pursue the MSc studies, for fully sponsoring this work and to the plants for allowing the work to be conducted in their sites.

Finally, but not least thanks to God, Almighty, Creator who is above all, to my family and friends for their endless support, love and care.

Abstract

It was identified that potential improvements in recoveries were to be found in the better use of the regrind stages at a number of Anglo-Platinum concentrators. Historically, these concentrators used to operate in an open circuit configuration, over a relatively short period, the circuits were changed to closed circuit configurations because of the reported benefits of operating in this configuration. The effect of closing the circuit had, however not been quantified in these concentrators.

This dissertation looks at quantifying the effects of each configuration with the purpose of drawing meaningful comparisons between them. The circuit configurations studied are open circuit, 1-stage and 2-stage closed circuit configurations. The primary difference between these configurations is the network of hydrocyclones. This study pays special attention to the effect of cyclone performance on the regrind circuit performance.

Comparisons were made between the open and the 2-stage closed circuits in three concentrators, and surprisingly the closed circuit did not show any improvement over the open circuit configuration from a comminution efficiency perspective. The cyclones performance was analysed, and it was observed that the cyclones were operating inefficiently, most likely because the cyclones employed were not appropriate for regrind circuits. A further comparison was made between the 1-stage and 2-stage circuit configurations and the 1-stage closed circuit configuration proved to be more efficient than the 2-stage closed circuit configuration.

Simulations were conducted to predict the performance of the circuit configurations by using more appropriate cyclones which are smaller in size. These cyclones are currently being used in a similar application in another concentrator. The simulations indicated that significant improvements in comminution performance could be realised with the utilisation of more appropriate cyclones, to achieve the closed circuit configuration.

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Chapter 1

INTRODUCTION

1.1 Introduction

A significant amount of work has been conducted to improve crushing and primary grinding at Anglo-Platinum. However, it was found that most plants suffered losses in flotation recovery due in the regrind stages either as a result of under-grinding or over-grinding of particles. It is well understood that particles outside of the required size region are not readily recovered by flotation.

The regrind circuits which were studied have historically been open circuit configurations, where the circuit feed was classified in a cyclone, the cyclone underflow was sent to the mill and the mill product would combine with the cyclone overflow to make the circuit product (float feed). The circuits were modified by adding a second stage of cyclones to classify the mill products combined with the 1st stage cyclone overflow. The 2nd stage cyclone overflow then became the circuit product and the underflow was re-circulated back to the mill for further grinding. (Flow diagrams are shown in Chapter 4, section 4.3).

The effect of closing the circuit had not been formally quantified at these concentrators, which treat a PGM rich seam known as Merensky ore. This thesis looks at quantifying the performances of three different circuit configurations with the purpose of comparing them. They are the historic open circuit, 1-stage closed circuit and 2-stage closed circuit.

The 2-stage closed circuit configuration in this thesis is not the typical 2-stages of classification where the cyclones are strictly in series. However, for argument sake this will be referred to as such in this thesis. In typical 2-stage closed circuits the underflow/overflow of the first cyclone feeds directly to the second cyclone making a series arrangement. However, the 2-stage closed circuit referred to here has the mill product stream combining with the first cyclone's overflow to feed the second cyclone.

The primary objective of a regrind circuit is to further liberate the locked valuable minerals from particles which have been subjected to flotation but were not recovered - the overall aim being to maximise the recovery from these streams.

The performance of a comminution circuit can be assessed by its final grind. The hypothesis of this investigation is described in the following section. The thesis objectives follow and a brief background to the section of mineral processing which relates to this thesis is discussed.

1.2 Hypotheses

The hypotheses were formulated in the following way:

- Operating a regrind circuit in closed circuit configuration with an appropriate network of cyclones should produce more particles in the optimal size range for flotation compared to an open circuit configuration. This is because in a closed circuit configuration there is classification of the mill products, and therefore the circuits' product stream is the result of a classification step.
- A 1-stage closed circuit should perform better than the 2-stage closed circuit, because a cyclone has inevitable inefficiencies that result in fines reporting to the underflow and then to the mill. Extending this argument, it can be held that by nominally doubling the cyclones, the amount of fines misclassified to the mill feed would also be doubled. This will lead to inefficient utilisation of energy, and result in over-ground particles.
- Implementing a grinding-classification system that releases particles in a well defined and controlled size range to the flotation stage would improve the mineral recovery because the flotation process is optimal for a specific size range. Very fine and coarse particles are not readily recovered by flotation.
- Different mill circuit configurations with the same nominal product size distribution may result in different flotation responses. This is owing to the process routes by which the particle size distributions are achieved potentially giving different surface properties and liberation characteristics.

1.3 Thesis Objectives

The objectives of the thesis are to investigate the effect of a circuit configuration on the performance of the regrind circuit by comparing different circuit configurations. The circuit configurations to be studied are the open circuit (as applied historically), and the 1-and 2 stage closed circuits. Performance of a circuit is in terms of the particle size reduction across the circuit.

This is taken a step further by exploring the possibility of assessing a comminution circuit by flotation response. In order to meet these objectives the following was required:

- Conducting sampling surveys around the regrind circuit for each circuit configuration studied where particle size, flowrates, percent solids and standard operation plant data were collected. The sampling surveys were to be conducted back to back to maintain the plant conditions between the surveys.
- Having collected all relevant data, a mass balance of each circuit configuration was conducted. The mass balance exercise was conducted to evaluate the integrity of the data collected and to calculate the missing streams data.

1.4 Background

Mineral beneficiation enterprises usually have multiple processing stages which include mining, concentrating, smelting and refining. The concentrator in a platinum ore treating plant consists of rock crushing of ore received from the mines, grinding to expose the valuable mineral and concentration, which in many cases is by froth flotation. The objective of this process is to separate valuable mineral from gangue.

The initial stage in the concentrator is the crushing of the so called "Run-Of-Mine" (RoM) ore that has been received from the mine. The crushing stage is followed by grinding using tumbling mills. In a platinum ore treating plant, the grinding is usually in more than one stage. The primary grinding stage usually employs semi-autogenous (SAG), autogenous (AG) or run of mine (RoM) ball mills. The secondary and tertiary (regrind) stages are usually carried out in the ball mills. The objective of a comminution circuit is to reduce particles' sizes to liberate the locked in valuable minerals. The valuable mineral is separated from the gangue mineral by means of froth flotation using mechanical flotation cells or flotation columns. The following section will briefly look at the comminution units that relate to this thesis and how they are influenced.

1.4.1 Factors Affecting Milling

Ball mills are cylinders that are rotated horizontally with steel balls as the grinding media. They have liners on the inside of the cylinder/shell that are used to protect the shell. In a regrind circuit, the mill feed is the slurry (mixture of ground particles with water) and it is transported through the mill as it rotates and grinds the particles with the steel balls. The mill product is discharged from the opposite end. Mill performance undoubtedly greatly influences the performance of a grinding circuit because the mill is the particle size reduction device. Some of the main factors that affect milling are discussed in this section. Figure 1-1 shows a picture of a ball mill.

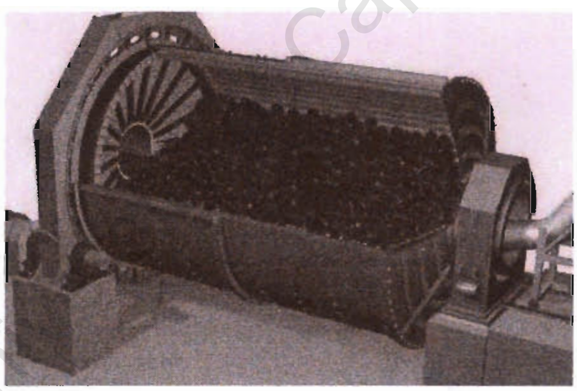


Figure 1-1 Cut-away of a ball mill (Napier-Munn *et al*, 1999)

MILL LINERS

The purpose of mill liners is to impart energy to the outer layer of media that's against the liner profile. Each profile of media then imparts energy to the layer on top of it. The liners are also intended to be sacrificed to prevent wearing of the mill shell. There are endless shapes of lifter and liners. In fine grinding smooth liners are used to enhance size reduction by abrasion.

POWER

The power input increases with the increase in volume and mass of the grinding media in tumbling mills. It reaches a maximum at a charge volume of about 50%. Ball mills work effectively over a wide range of sizes, from small power units to large units that use power up to 10-12 MW (Napier-Munn *et al*, 1999).

Figure 1-2 clearly indicates that a decrease in fill level decreases the power consumption at different mill rotation speeds. As the mill fill level decreases the power consumption continues to decrease. This is expected since the amount of charge that needs to be carried is being reduced.

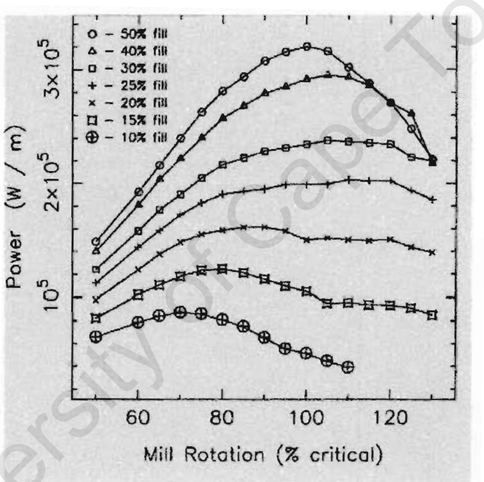


Figure 1-2 Effect of mill fill level on power draw as a function of mill speed, (Cleary, 2001)

Figure 1-3 shows the power per tonne as a function of mill fill level which is an indication of the rate of energy supply to each unit of ore fed to the mill. As the fill level decreases from 50% to 40% there is a strong increase of the power/tonne. This suggests that more effective grinding will occur at lower total energy consumption at 40% fill level rather than one at a capacity of 50%. The mill filling can be approximated by the Allis Chalmers filling formula (Equation 1-1).

Equation 1-1

$$V = 113 - \left(\frac{H}{D} \times 126\right)$$ (Napier-Munn *et al*, 1999);

where H is the vertical height from the charge surface to the roof of the mill in metres (between liners) and D is the internal diameter of the mill in metres(inside liners).

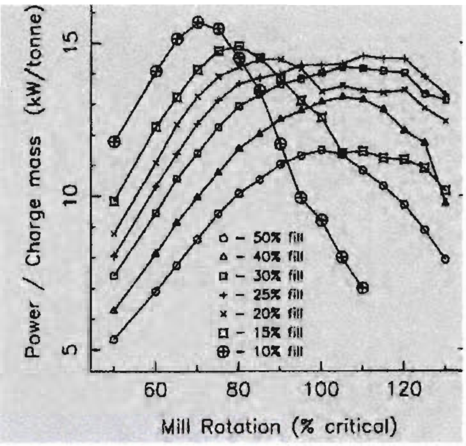


Figure 1-3 Effect of mill fill level on power draw as a function of power draw per ton (Cleary 2001)

BALL LOAD AND MILL SPEED

It is reported that ball mills normally carry a ball charge from 35 to 45% (or higher for grate discharge mills) of the mill volume with charge of up to 50% or slightly more (Cleary 2001, Napier-Munn *et al* 1999). Mill speed is expressed as a fraction (percentage) of the rotational speed at which centrifugal acceleration balances gravitational acceleration which is termed critical speed (Equation 1-2).

Equation 1-2

$C_s = \frac{42.3}{\sqrt{MillDiameter}}$ (RPM) (Napier-Munn *et al*, 1999 equation 9.1), where the mill diameter is in metres

EFFECT OF MILL ROTATION SPEED AND FILLING ON POWER DRAW AND TORQUE

Cleary (2001) predicted that for a mill filling of 50% the average torque increases slowly with speed and then it reaches its peak at a rotational speed of about 80% of critical speed (Cleary, 2001). Liddell and Moys (1988) reported that torque has a parabolic dependency on the mill filling for constant speeds and that the maximum torque occurs at a filling of about 45%. They reported that the breakage function is also at a maximum when the filling is approximately 45%.

MILL BALLS

Grinding balls are normally made of forged and cast iron or steel. Grinding media comes in different shapes. They can be in conical, cylindrical or even irregular shapes. For fine grinding, media that's got a large surface area per unit mass should be used. For larger rocks and coarser feeds large balls should be used because high energy impact is required to break the large particles.

It is desirable that the balls being used wear in a uniform pattern. This is of coarse dependent on the quality of the balls. Rowland and Kjos (1978) claim that an indication of good balls is when the worn balls discharging from the mill are about 16 mm in diameter or smaller, or have a polygon shape with 8 to 12 surfaces.

The size of balls has a large effect on the performance of a mill. Larger balls offer increased breakage for coarser particles and reduced breakage at finer particles. This can be explained by the argument that impact breakage predominantly occurs with coarse particles and that attrition breakage predominates at the fine particle sizes (Lynch and Morrell). A general rule is, the finer the feed the smaller the balls. The Allis Chalmers rule-of-thumb (Equation 1-3) is used to calculate the make up ball size (i.e. maximum ball size).

Equation 1-3

$$b = \left[\left(\sqrt{\frac{F80}{K}} \sqrt[3]{\frac{sg.WI}{\%C_s \sqrt{3.281D}}} \right) \right] \times 25.4$$ (Napier-Munn et al, 1999, equation 9.14)

Where,
b = make-up size (mm)
F80 = 80 % of material in the feed that passing, μm
K = mill type factor
C_s = critical speed, Equation 1-2
sg = specific gravity
D = Mill diameter (m)
WI =Feed ball mill work index

1.4.2 Classification

Classification is one of the most important processes in the mineral beneficiation industry. Napier-Munn *et al* (1999) identifies classifier operation as the most common cause for poor ball mill circuit performance. The degree of improvement from a closed circuit is dependent on the product size of interest. The classifier is a means of moderating the product size of a particular circuit. The product from an open circuit mill is dictated by the natural mill product size. This is the basic difference between open and closed circuits.

The product of an open circuit configuration is not moderated to classify what is ready for the next process, which is flotation in the regrind circuit's case. Efficient classification is a key to see the benefit of a closed circuit configuration. Napier-Munn *et al* (1999) notes that efficient classification offers 5-10% better performance and a poor classification offers 5-10 % below average performance.

The most common classifier found in the regrind circuits is the hydrocyclone (Figure 1-4). The hydrocyclone, or cyclone as it is commonly referred to, is a simple device which takes less floor space and is simple to operate compared to other classifiers.

The feed to the cyclone is introduced to the unit under pressure through the feed inlet. This creates a swirling motion of the pulp and a vortex is generated with a low pressure region at the vertical axis which is directly linked to the atmosphere. The classical theory is that particles in the swirling are exposed to both centrifugal outward force and the drag inward forces. Details of the individual forces are discussed by Heiskanen (1993).

The bigger and heavier particles are drawn to the wall of the cyclone where the velocity is low and go down to the apex, the small and lighter particles move to the low pressure zone, the core, where they are carried upward though the vortex finder to the overflow.

1.4.3 Cyclone Efficiency

Since the hydrocyclone performance greatly determines the effectiveness of a closed circuit grinding circuit, some background on the analysis of a hydrocyclone performance will be discussed briefly.

The most common way of representing the efficiency of a hydrocyclone is by means of a performance/partition curve. The performance curve relates the mass fraction of each particle size in the feed reporting to the overflow/underflow. The cut-point/separation size, normally referred to as the d_{50c}, is that point on the performance curve where 50% in the feed of that size goes to the underflow.

The sharpness of a cut/sharpness of separation is dependent on the slope of the central part of the performance/partition curve. The steeper the slope the higher is the classification efficiency.

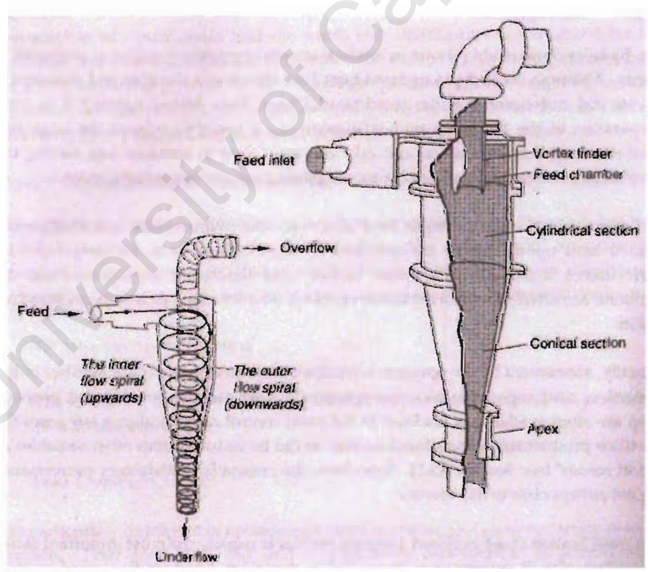


Figure 1-4 The hydrocyclone, showing main components and principal flows (Napier-Munn *et al*, 1999)

1.4.4 Factors Affecting Cyclone Performance

This section briefly discusses some of the factors that affect the performance of a hydrocyclone.

CYCLONE DIAMETER

The d_{50c} is dependent on the cyclone diameter. Cyclone diameter is therefore a principal design variable, this is derived from the resulting tangential velocities which establish the centrifugal forces which drive separation (Napier-Munn *et al*, 1999). Smaller cyclones should be used for finer separation.

APERTURES

The area of the inlet opening determines the velocity of the entering pulp. At a given pressure drop, an increase in the vortex finder diameter will result in a coarser cut point and an increase in capacity. The diameter of the apex/spigot opening determines the solids concentration in the underflow. Unclassified material leaves the underflow in proportion to the water that leaves the underflow; so the solids content must be as high as possible in the underflow (Wills, 1997).

PULP SOLIDS CONTENT AND PULP PRESSURE

The sharpness of separation is decreased when the pulp density is increased and the cut point rises because of the increased resistance to the swirling motion in the cyclone which then results in a reduction of the effective pressure drop. For finer feeds, separation occurs at low solids concentration (normally no greater than 30% solids by weight) and high pressures. However, for closed circuit grinding where coarser separations are required, high solids content of up to 60% by weight are combined with low-pressure drops, often less than 68.9 kPa (Wills 1997).

Increasing the pressure of the pulp feed can influence the sharpness of separation, although the effect is said not to be strong (Napier-Munn *et al*, 1999).

1.4.5 Mineral Concentration by Froth Flotation

The mineral beneficiation process used in the circuits studied is froth flotation. Froth flotation depends largely on the surface properties of the particles. For flotation to occur an air bubble must be able to attach itself to a particle and lift it to the surface of the agitated pulp, Agitation is by an impeller at the bottom of a flotation cell. Once the particle reaches the cell surface, it is recovered in a 3-phase froth layer which overflows the flotation cell, and is termed "concentrate".

For air bubbles to be able to attach themselves to the surface of the particles the air bubble must displace the water on the surface of the particle. This can happen if the particles are to some extent hydrophobic/aerophilic. Various reagents are used to render particles aerophilic. This process can only be applied to particles of relatively small size, if the particle is too large the weight of the particle will be larger than the adhesion forces between the particle and the air-bubble and the particle will drop. Alternatively the bubble-particle aggregate may sink.

If the particles are too fine, attachment becomes unlikely as the particles do not posses the inertia essential to penetrate the bubble shell and become attached. Recovery is therefore largely by entrainment, where particles (valuable and non-valuable) are recovered unselectively with the recovery of water. The objective of the comminution circuit is therefore to produce particles between the small non-floating and the large non-floating particles.

The reagents used in flotation include collectors which facilitate bubble attachment by rendering particles hydrophobic/aerophilic. Frothers help to maintain a stable froth so that the air bubbles do not burst and drop particles back into the pulp on reaching the cell surface. Depressants are used to enhance the selectivity of the flotation process by rendering the unwanted particles hydrophilic, thus preventing their flotation.

1.5 Plan of Development

Chapter 2 discusses the literature research where papers relevant to this study were each critically reviewed. The chapter is followed by the preliminary investigation chapter (chapter 3) where open circuit configuration was compared with the 2-stage closed circuit at Lebowa concentrator. This summarises the work conducted as a way of developing and assessing the experimental methodology.

An experimental method that was followed during this project is discussed in detail in chapter 4. Chapter 5 presents and discusses the results obtained from two concentrators (Amandelbult and Waterval concentrators) where open circuit was compared with the 2-stage closed circuit. In Waterval concentrator there is a pipeline that allows for a third circuit configuration, the 1-stage closed circuit. Chapter 6 looks at the performance between the 1- and 2-stage closed circuit configurations.

The circuits' experimental results were fitted to the mathematical models using the JKSimMet package. Simulations using these models were conducted to predict possible improvements in the 1-and 2-stage closed circuit of Waterval concentrator. This is presented in Chapter 7.

The conclusions and recommendations are discussed in the final chapter of the thesis.

Chapter 2

LITERATURE REVIEW

2.1. *Introduction*

There are two aspects covered in the literature review in light of the objectives outlined in Chapter 1. The literature survey was done by critically reviewing various papers that cover the following aspects:

- The effect of circuit configuration on the performance of a milling circuit. This review covers studies dealing with how hydrocyclones are used to close the circuit, and how these arrangements influence circuit performance in terms of particle size reduction.

- Studies dealing with the effect that the characteristics of the milling circuit product have on the recovery of valuable minerals by froth flotation.

2.2. *Preparation of Flotation Plant Feed*

By R.D. Carpenter (1962)

Carpenter conducted work to analyse the performance of the grinding circuit at Pine Creek. The plant treated low grade tungsten and molybdenum-copper ores. The investigation was conducted on a 6×5 ft (1.83×1.52m) grate discharge ball mill in closed circuit with a spiral classifier. The classifier overflow goes directly to the flotation section and the classifier oversize returns to the mill for regrinding.

The flotation studies conducted showed that the optimum particle size range for flotation was between 150μm and 38μm. The author claims that experience has shown that particles are reduced to that size range if the total ore is ground to minus 48mesh (300μm).

The general objective of the circuit under study was thus determined to grind the ore to minus 300μm. The experimental method included collecting samples of the new ball mill feed, ball mill discharge, classifier overflow and the classifier oversize. Sampling was conducted over a 24 hour period. The samples were sized and the resulting size distributions were analysed.

The analysis of the sizes showed that the new ball mill feed had 23% minus 300μm which is already suitable for flotation. There is some similarity between the composite ball mill feed (new mill feed plus classifier oversize) and the mill discharge. This is an indication of a short retention time of the solids inside the mill. This was confirmed by calculating the retention time which was found to be 2 minutes.

The classifier return was found to have 46% minus 300μm indicating that nearly half the material in the classifier underflow is ready for flotation. Further grinding of the classifier oversize is the reason for the occurrence of 38% of minus 38μm in the classifier overflow.

Since the desired separation size was 300μm the efficiency of the classifier was considered as the recovery of the minus 300μm in the classifier overflow. The efficiency of classification in this circuit was found to be 27% (% minus 300μm in the classifier overflow divided by % minus 300μm in the classifier feed).

Some particles, which were ready for flotation, were re-circulated back into the mill and a considerable amount of coarse material overflowed the classifier and fed the flotation section. It was then concluded that the objective of the circuit was not being fully attained. The main factors against achieving the objectives were identified as the high circulating load, low classifier efficiency, overloaded ball mill and differential grinding of minerals.

The efficiency of the classifier in a closed circuit is dependent to some degree upon the circulating load of that circuit. The circulation load is an empirical variable and peculiar to each circuit.

Discussion

The author mentioned that the optimum size range for flotation was found to be between 150-38μm; however he uses 300μm as a separation size for particles ready for flotation. His conclusions which were based on 300μm as a separation point were biased since there would be particles between 150 and 300μm considered ready for flotation although they are outside the optimum size fraction. This analysis would have been more accurate if the analysis was based on the material between 150 and 38μm which is the optimum size range for flotation.

In spite of the above, the conclusions reached are valid to a certain extent. The paper is informative on the problems associated with ball mills in closed circuit with classifiers. It seems that the circuit is meant to perform an outstanding task of only producing particles already reduced to liberation size and keeping coarse particles for further grinding. However, there are variables that need to be tuned for optimal performance of the circuit.

The author mentions that the calculated retention time in the mill was 2 minutes. He does not show the calculations for the retention time. From the calculation it would be possible to tell what influences the retention time. The retention time is size dependent; a small particle would not have the same mill retention time as a bigger particle.

2.3. About The Ways And Means to Improve the Performance of the Closed Grinding Circuit

By R.T. Hukki (1976)

Grinding is undoubtedly one of the highest consumers of energy in the mining industry. With the increasing tonnages of ore being processed and the grinding of lower grade ores, Hukki found it necessary to find ways to improve the closed grinding circuits.

Hukki conducted investigations in closed circuit grinding processes to get a clear understanding of these circuits in terms of the sharpness of classification, capacity, circulating load and power consumption. A summary of results from his investigations are shown in Figure 2-1. The curves relate increase in capacity and reduction in power consumption as functions of sharpness of classification and circulating load. The data used to formulate the curves was drawn from more than 300 studies.

The fundamental significance of the sharpness of classification on a closed grinding circuit can be illustrated by using the curves in Figure 2-1. Considering a typical plant operation case where the circulating load is 200% and the sharpness of classification is at 50%, it was shown that an increase in sharpness resulted in a drastic improvement of circuit performance. For example, a 20% increase in capacity or a 15% power reduction is achieved by improving the sharpness of classification by 30%.

Despite the encouraging conclusions drawn by the Hukki, there has never been any report of such effects before. The suitable high capacity apparatus to accomplish such improvements in practice has not been developed.

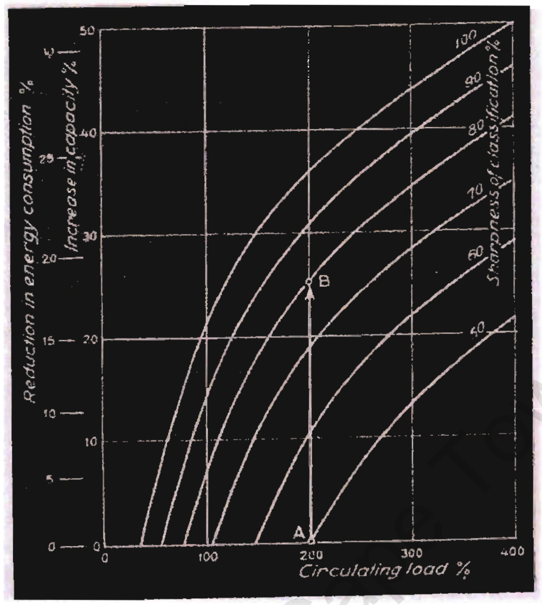


Figure 2-1 Experimental curves showing the fundamental relationships between circulating load, sharpness of classification, increase in capacity and the reduction in energy consumption in closed grinding circuits.

Hydrocyclones are the most common classification device used in wet closed grinding circuits. This is because they occupy small floor space, don't have moving parts and have low capital costs. The author claims that the hydrocyclone has no worthwhile classification improvements from other conventional classifiers. There have been attempts to improve the shortcomings of poor sharpness of classification by employing two stages of hydrocyclones; but little success was achieved.

Hukki reports that the findings from investigations that have been conducted in the past on the classification in closed grinding circuits show that for optimal size control, classification should not be based on one stage of a hydrocyclone. This is in agreement with what Carpenter (1962) concluded. A two-step classification consisting of a hydrocyclone and a wet fine screen is proposed for sharper classification. The hydrocyclone makes use of the simple classification and the fine screen corrects any shortcomings of the hydrocyclone. A new hydraulic screening apparatus has been developed for the second stage classification.

Discussion

The benefits of a sharper classification as discussed by Hukki in this paper are significant. For a high sharpness of classification; most particles that are already reduced to liberation size are released from the circuit, there is therefore a reduction of unnecessary grinding and more energy is used for necessary grinding.

Both the sharpness of classification and the circulating load appear to influence the performance of the closed grinding circuit. This was discussed by both Hukki in this paper and Carpenter (1962).

Hukki does not present the experimental details in this paper, so there is no way of reproducing the experimental procedure that he conducted. He claims that employing multiple stages of cyclones does not yield worthwhile improvements. However, he does not quantify the changes due to multiple cyclones.

It is interesting to note that the author does not mention anything about the mill as a way to improve the performance of a closed grinding circuit. This shows that in the author's opinion, a lot of the inefficiencies in the grinding circuits are attributed to poor classification.

For the purposes of the thesis, the analysis of the closed grinding circuit will be based on the variables discussed in the paper, sharpness of classification and the circulating load. Yianatos *et al.* (2002) stated that a high circulating load will provide a faster release of fine particles and decrease the over-grinding. While low circulating load will create space for new fresh feed.

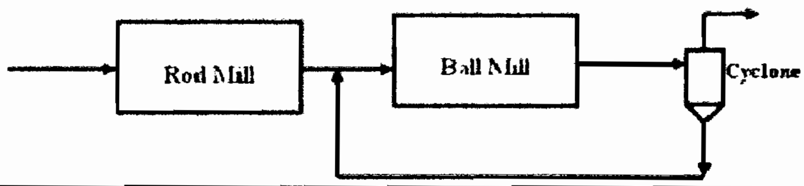
2.4. A New Way to Grind and Recover Minerals

By R.T. Hukki (1977)

Hukki quotes S.K. deKok as saying that the hydrocyclone must be regarded as a stumbling block to progress in the field of grinding. He said that without the hydrocyclone the industry would have been forced to find a better method for size separation since classification is observed to be the least efficient process in the industry.

Hukki affirms that a lot of attempts have been made to improve the performance of the hydrocyclone by fine adjustments. The results, if positive have been minimal. He declares that as long as the analysis of size separation in closed circuit grinding is out of focus, all attempts to cure the situation by fine adjustments appear barren.

Hukki declares that the mutual interdependence between fineness of classifier feed, fine product, and their effect on classifier sharpness of classification and the circulating load have not been recognised by the grinding experts. Recognition of such a relationship is the key to a closed grinding circuit that can produce the required fine product. Figure 2-1 shows such a relationship.



	Minus 65 mesh (212µm)	Minus 200 mesh (75µm)
Feed to rod mill	10-20 %	4-8%
Rod mill product	40-60%	20-35%
Ball mill product	60-95%	30-35%
Cyclone overflow	95%	55-70%
Cyclone underflow	40-70%	15-30%

Figure 2-2 Two-stage grinding and one stage classification circuit

The author presents some typical field results of a two-stage grinding and one stage classification circuit (Figure 2-2). Based on the results he came to the following conclusions.

- Rod mill product can be of ideal fineness for a closed circuit operation at a high sharpness of separation.
- In normal practice a ball mill product is greatly over-ground. At circulating loads of 150 to 250%, the hydrocyclone feed is characteristic of an over-ground mill product.
- The cyclone underflow includes a large portion of material that is fine enough to be included in the fine product.
- A comparison of field data and the information on Figure 2-1 indicates that there is a need for a drastic change both in fineness of the classification system feed and the oversize material returned to the mill.

Hukki proposes a one stage grinding and two stage classification circuit where grinding is by a rod mill, (a ball mill will be preferred for a regrind circuit), first stage classification is by a hydrocyclone and the second stage classification is by either the hydraulic cone classifier or hydraulic trommel screen. He presented figures from such a circuit at circulating loads of 50 to 150 %. The feed to the sizing system was coarser than that found in a conventional circuit. The hydrocyclone is observed to have an improved sharpness of separation and a finer overflow than in a conventional circuit.

Discussion

It appears that the problems associated with classification have been around for a very long time. This paper is a continuation from Hukki (1976) in an attempt to sell the idea of the proposed hydraulic gravitational cone classifier equipped with a newly developed sand cleaning system. It is suggested in the paper that the hydrocyclone has failed to achieve the required sharpness classification on its own and that fine tuning of the hydrocyclone has proved barren.

Hukki proposes a circuit configuration for improved performance where the first stage classification is by means of a hydrocyclone and then the second stage is either by the hydraulic cone classifier or by hydraulic trommel screen. It appears that the circuit configurations studied in the thesis cannot achieve the reported sharpness of separations.

2.5. A Practical Multiple Cyclone Arrangement for Improved Classification

By D.F. Kelsall, P.S.B. Stewart and C.J. Restarick (1974)

Kelsall *et al* identified that the inefficiencies in the cyclone may be limiting the recovery of valuable mineral in the flotation process. This was observed by conducting full scale plant tests on a closed circuit with a ball mill. They identified the main imperfections from the cyclone as:

a. The presence of fine particles down to zero size in the underflow.

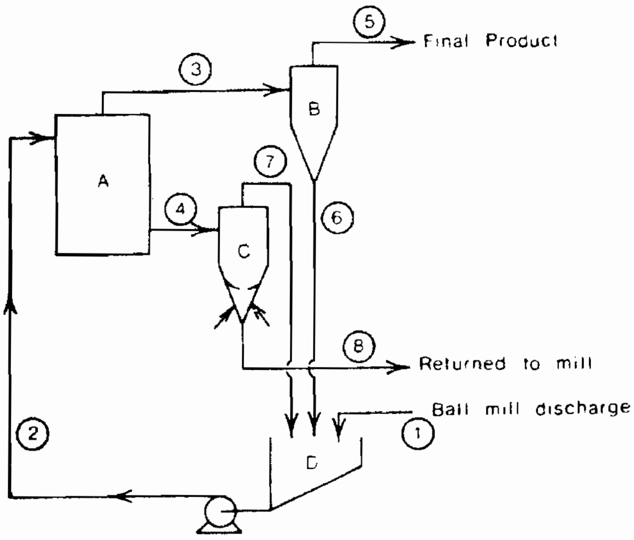
b. The deviation of the efficiency curve from a vertical straight line (i.e. the slope for the intermediate size).

c. The difficulty associated with the complete elimination of the coarsest particles from the overflow.

Earlier studies of size by size recovery studies of lead and zinc recovery in the flotation circuits at the Broken Hill South showed that recovery was poor in the fine particle region and dropped sharply in the coarser region, with superior recovery in the intermediate sizes. It was then realised that poor classification has an adverse effect on recovery.

In an endeavour to cure the classifying problems, Kelsall *et al* used mathematical modelling to predict that by combining appropriate cyclone units and using a single pump they might improve the classification efficiency and recovery. Laboratory scale and preliminary full scale tests were conducted to test the concept.

Conditions were established in a laboratory scale where the series arrangement would give sharper classification than a single unit. The composite cyclone used is shown in Figure 2-3.



Line diagram of composite cyclone arrangement.
A. Cylinder
B. Fine cyclone
C. Coarse cyclone
D. Pump sump

Figure 2-3 Line diagram of composite cyclone arrangement

The ball mill product makes the new feed (1). A single pump is used for the whole arrangement. A cylinder (A) is fed tangentially (2) so that the overflow (3) is retreated in the conventional cyclone (B) and the cylinder (A) tangential underflow feeds the second cyclone (C) fitted with the water injection. The underflow cyclone with the water injection is used to minimise the return of product size solids to the mill and hence the overflow (7).

The results from the laboratory scale studies showed that the composite cyclone had a steeper efficiency curve than the single cyclone (Figure 2-4), which means a higher classification efficiency.

Table 2-1 **Composite cyclone results compared with single cyclone results (Kelsall et al, 1976)**

Component	Size Range µm	Percent in Size Range		
		Typical Plant Data with Single Cyclone	Possible Result Using Composite Cyclone	Preliminary Full Scale Test of Composite Cyclone
Pb	+295	0.3	-	0.3
	-295+147	2.0	1.0	1.0
	-147+2.4	87.8	90.1	90.0
	-2.4	9.9	8.9	8.7
Zn	+295	1.0	0.2	0.3
	-295+208	3.8	0.8	1.3
	-208+3.5	90.4	93.9	92.8
	-3.5	4.8	5.1	5.6

Although the models showed that composite cyclone would give improved classification, the preliminary full scale data obtained did not match calculated results. The separation obtained was however sharper than the single cyclone. Table 2-1 shows the preliminary results from the full scale tests. Kelsall *et al* identified the cylinder to have limited the performance of the composite cyclone in full scale and they said that by modifying the cylinder the composite cyclone could give worthwhile improvements.

Discussions

The paper shows that there is some potential in using multiple cycloning stages to improve classification, this was shown by the mathematical models and laboratory testing; however, the proposed composite cyclone in full plant scale has not been shown to give significant improvements compared with the simple single cyclone, if the capital required for the extra equipment is taken into consideration.

The ball mill model was earlier developed by the main author (cited in Kelsall *et al,* 1974), but there is no mention of the hydrocyclone model used. There is therefore no way of assessing the model when the model parameters are not presented.

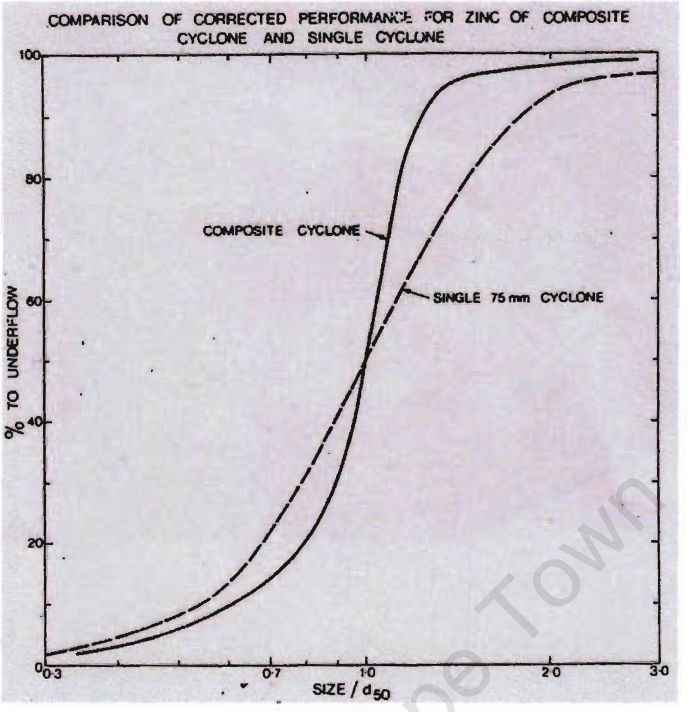


Figure 2-4 Corrected performance curves for zinc for the small scale composite cyclone and for the single 75mm cyclone (Kelsall et al, 1976)

2.6. Characteristics of Open- and Closed –Circuit Grinding Systems

By Kobayashi, A. Nagasaka, H., Iizuka, K. and Yoshida, H. (2003)

This paper attempts to examine the characteristic differences between the open –and closed-circuit grinding systems (Figure 2-5) via the use of a continuous vibration mill and a blade-type classifier. The parameters investigated were the size distributions of the product and the returning particles, and the energy consumption per unit mass production. The investigations were conducted using computer simulations and actual data from real processes. Simulations were based on equations obtained from the open circuit process data. The grinding device and the classifier were initially studied under various conditions to obtain empirical equations that describe each component.

The material used in this study was quartz type silica and the grinding system consisted of a vibratory mill and an O-sepa classifier. Three equations were used to estimate the performance of the mill because there were three particle size regions that behaved differently. A similar effect was observed with the classifier where separation efficiency increased by about 15% for particle size less than 3μm. This is due to the coagulation of small particles which are then collected at the coarse side. There was corroboration between the calculated and the experimental data for both the mill and the classifier.

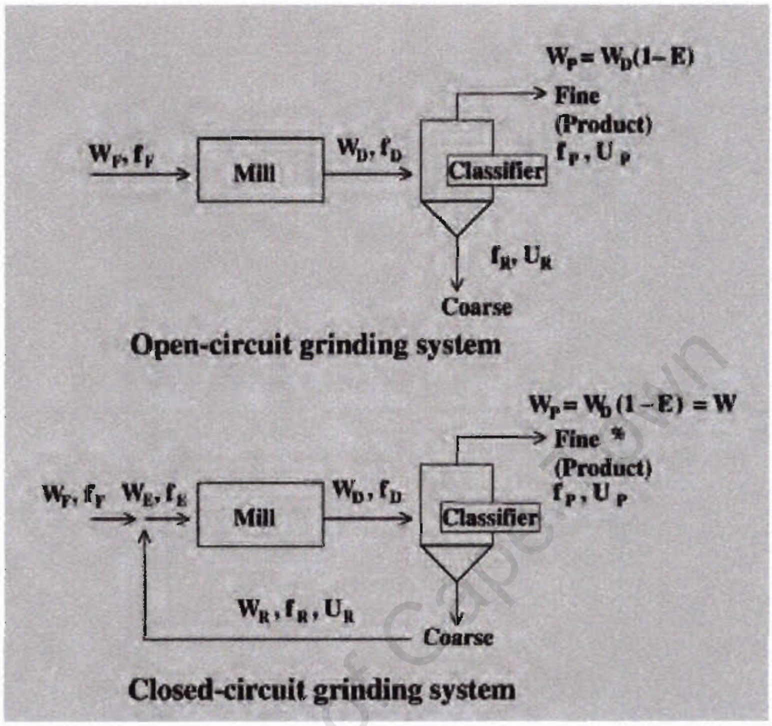


Figure 2-5 Schematic Diagram of Open-and Closed –Circuit Grinding Systems (Kobayashi *et al*, 2003)

The simulation method used involved calculating the feed rate and size distribution entering the mill using the circuit feed particle size distribution. Equation 2-1 and Equation 2-2 were used.

Equation 2-1

$$f_E \Delta D_P = \frac{W_F f_F \Delta D_P + W_R f_R \Delta D_P}{W_E},$$

Equation 2-2

$$W_E = W_F + W_R, \qquad \text{see Figure 2-5}$$

Where W and f are powder flowrate and size distributions respectively

The powder flowrate and size distributions after the mill were calculated using the equations developed for the mill. The fine and coarse size distributions of the classifier were calculated using the equations developed for the classifier. Iterations were performed until a steady state solution was reached. Experiments were conducted for a wide range of conditions. It was observed that the performance of the mill decreased with an increase in product flowrate. The median diameter of the product also increased as the median diameter of the classifier feed increased. It also increased with an increase in 50% cut size.

The experiment was arranged such that the product size was set, and then the new feed consisted of particles of a particular size distribution. Comparisons between the open and closed circuit grinding systems were based on the production rate of the product and the power requirement of the mill in both circuit systems for a particular product size.

The conclusions reached by the authors as far as open-and closed circuit systems are concerned included that for the same product median size the production rate is about 5% higher for the closed circuit system. The power consumption is 5% less for the closed circuit grinding system when the product median size produced is the same. The simulated results were compared with the experimental, and there was a good agreement.

Discussion

The authors observed that for the same median product, the closed circuit system yielded a higher product flowrate. This was because the classifier received a finer feed and hence produced more material going to the finer stream. The classifier received finer material from the mill for the closed circuit grinding system. The reason for a finer mill product on the closed circuit system is due to the re-circulating load which is already finer compared to the open circuit feed (new feed to the system of a particular constant size).

In one condition that was investigated, the throughput in the mill was 468 and 451 kg/h for the open and closed circuit systems respectively. This also contributed to the closed circuit system mill having a finer product than the open circuit mill. This is because as the flowrate increased the mill performance reduced. It is then evident that the mill discharge particle size distribution greatly influences the performance of any circuit system.

2.7. Potential Energy Savings in Comminution by Two-Stage Classification

By Donald A. Dahlstrom and Wai-Ping Kam (1988)

It is well known that comminution is the largest consumer of energy in the mineral processing industry. Studies have been conducted and there are indications of possible energy savings in various areas of comminution. One area of energy saving is in the design of the classification device. The energy saving is possible due to the elimination of fines already reduced to liberation size from the comminution circuit.

There are a couple of factors that affect the performance of the hydrocyclone closed circuit. For a finer ball mill product the hydrocyclone centrifugal force must be increased. This can be achieved by high pressure drops across the hydrocyclone, reducing feed and overflow nozzle dimensions, or by employing smaller hydrocyclone diameters.

In order to reduce the particle size, energy input to the comminution unit must be increased, either by increasing one or a combination of: mill speed, slurry hold-up, the ball charge, or the solids concentration in the unit. As the hydrocyclone underflow solid concentration is increased the amount of liberated fines that are recycled is reduced hence over-grinding is reduced.

The single stage hydrocyclone is a common method for closed circuit ball milling; however the authors have considered two-stage circuits for energy saving. The two two-stage hydrocyclone classification circuits that were considered are shown in Figure 2-6. In flowsheet 1 the feed from comminution is combined with the stage 2 hydrocyclone overflow. It is then pumped to the stage 1 hydrocyclone; the final product comes from the stage 1 hydrocyclone and the stage 2 hydrocyclone underflow is re-circulated back into the comminution unit (Ball mill).

For the second flowsheet the feed to the stage one and stage two of hydrocyclones is diluted with fresh water. The product comes from the overflow of the two hydrocyclones and the 2nd stage hydrocyclone underflow is re-circulated back to comminution.

The authors claim that the latter circuit will result in minimum re-circulation of fines and hence possible maximum energy savings. This is the circuit that they used in the study.

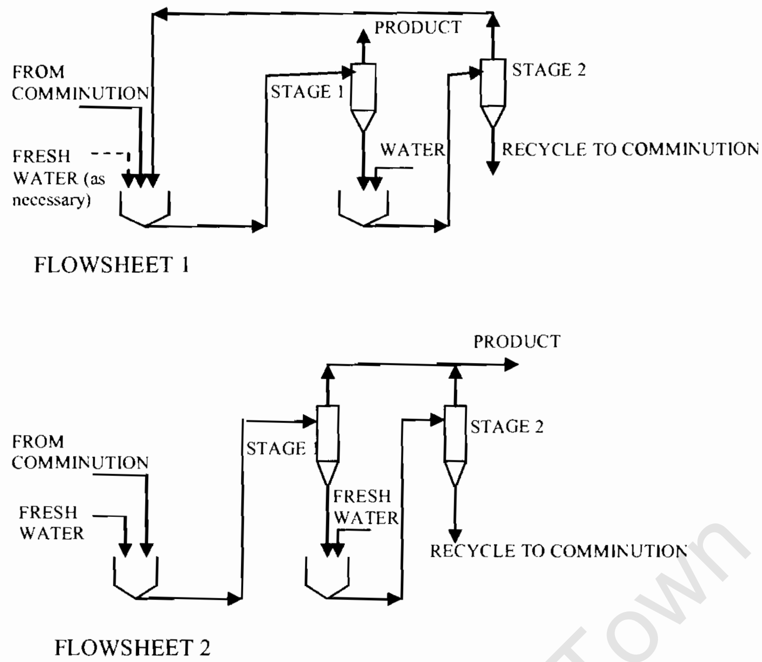


Figure 2-6 Possible two-stage hydrocyclone classification circuits for use in comminution energy savings

The authors conducted their studies using a computer program termed MODISM which was developed in the University of Utah. It is a general flowsheet simulator and one of the existing programs is the comminution circuit, including hydrocyclones and screens for closing the circuit. The program requires the following parameters as inputs: rod mill size distribution, ball mill power draw; selection and breakage function; hydrocyclone diameter and number of cyclones in parallel. The other inputs into the program include the tonnage and percent solids, hydrocyclone flowsheet, and dimensions.

The hydrocyclone model that they used was based on the work done by Plitt. They were able to determine the grind, recycle ratio, kWh/metric ton of product and kWh/ metric ton of product plus recycle. This was conducted for both the single-and two-stage classification in order to make the comparison.

The solid concentrations for the feed and the underflow for both the hydrocyclone stages were kept at about 54 and 75 %, respectively and the rod mill discharge size distribution was kept constant. By changing the number of hydrocyclones, the hydrocyclone diameter, the other dimensions of the hydrocyclone and the kW input to the ball mill the results were found on the basis of kWh/metric ton of product plus recycle, percent 100 mesh (150μm) or 200 mesh (75μm) in the product and percent solids in the final product along with the recycle ratio.

The conclusions reached were that for the two-stage hydrocyclone classification the energy saving will be a function of the grind and will increase as the grind becomes finer. The energy savings were reduced as the recycle ratio was increased. The predicted energy savings ranged from 6% to over 30%.

There has been a significant reduction in the -37μm solids generated by the two-stage classification at the same grind as the single stage classification due to reduced over-grinding. A disadvantage of the two-stage classification is the resulting low percent solids of the product due to the amount of fresh water required.

For the installations that are already in existence, energy savings cannot be as great as in new installations, however the following benefits were predicted:

- Percent increase in tonnage rate of solids at the same grind with the same energy consumption, and
- Potential finer grinds achievable at the same solids tonnage rate and energy consumption.

The results had not yet been confirmed experimentally at the time the paper was released. The authors were confident that the experimental work will yield similar if not better results.

DISCUSSION

The MODISM computer program enabled the authors to determine the results of the ball mill and hydrocyclones as they worked together. The authors say that the hydrocyclone model was based on the work by Plitt; however they did not mention what they used for the ball mill model which is also a critical part of the circuit.

It would be most informative if the authors had conducted experiments to confirm or compare their predicted results, because at the time of releasing the paper no experiments were done. It is therefore not known how accurate their predictions are. Contrary to the reports in this paper Hukki (1976) reported that a two stage hydrocyclone system did not yield any worthwhile improvements.

2.8. An Evaluation of the Use of Two Vs One Stage of Hydrocyclones in a Pilot Scale Ball Mill Circuit

By R.S.C. Rogers, A.M. Hukki, G.J. Steiner and R.A. Arterburn (1981)

The authors of the paper have examined the use of two versus one stage of hydrocyclone in a pilot scale ball mill circuit. Their study was based on the computer simulation with models that account for grinding and classification. The simulation results were validated by tests which are claimed to show a close agreement with the simulated results.

There are two apparent sources of inefficiencies in the hydrocyclones. The first inefficiency results because real hydrocyclones are non-ideal; the probability of the feed particle smaller than the cut-size to report to the coarse stream is not necessarily zero. The deviation of the real hydrocyclones from the ideal is characterized by the Sharpness Index which is defined as the ratio of the size of particles having a 25 % chance of reporting to the coarse stream to the size of particles having a 75% chance of reporting to the coarse stream.

The second inefficiency is as a result of the bypass particles, where some fine particles bypass the classification system and report to the coarse stream. In closed circuit grinding the bypass particles are re-circulated for further, and in most cases unnecessary grinding.

In this study an attempt is made to correct the limitations of a one stage classification by classifying the impure streams in an additional classifying stage; that is employing two stages of hydrocyclones.

The grinding system used in the study consists of a 0.91m by 1.54m (diameter by length) ball mill in closed circuit with one or two stages 76mm diameter Krebs hydrocyclones. The configuration chosen for the two stage classification was such that it should provide sharper classification and decreased bypass compared to the one stage classification.

The models for the grinding mill and hydrocyclones were developed and validated through experimental tests. A comparison of the one and two stages of hydrocyclones was based on:

- Classification Efficiency
- Circuit Capacity
- Circuit Product Size Distribution
- Material Balance

The configurations considered are the same as those considered by Dahlstrom and Kam (1988), Figure 2-6. The mill was operated at a critical speed of 73% and drawing a constant approximate power 7.5 kW with 35% filling with balls of 50.8mm top size and limestone ore with a top size of 4.75mm. In the single stage hydrocyclone test one 76mm Krebs hydrocyclone was used; for the two stage hydrocyclone test two of these hydrocyclones were used. Spigots and vortex finders of different sizes were used.

The mill model was initially derived by Rogers and Gardner (1980) which was based on a mechanistic segregated flow for continuous open-circuit ball mills. The actual model combines with the solution for batch ball milling and the residence time distribution models developed to give a model which accounts for size reduction and material transport inside a mill. The authors claim a good agreement between their simulation result and the test results; see Table 2-2 which shows the typical results from the study.

Table 2-2 Measured and Model Simulated Product Size Distributions Obtained for Limestone being ground in a 0.91m x 1.54m Open Circuit , Wet Ball Mill (Rogers *et al*, 1981)

	Percent Passing Stated Size			
Size (um)	Measured	Model	Measured	Model
1190	99.96	99.98	99.92	99.9
595	99.82	99.89	99.54	99.44
297	99.00	99.12	96.51	96.64
149	94.97	95.28	86.38	87.35
74	84.69	84.69	70.84	70.22

The authors say that the hydrocyclone model used was based on the size selectivity concept which assumes that the classification selection process for material of a given size is first order with respect to the quantity of the feed presented to the classifier.

A number of tests were performed on the experimental apparatus described above, and compared the results with those from the simulations. The authors found good agreement between all the results. Once satisfied that their simulators were capable of predicting the performance of a circuit, they performed simulations to compare the circuit configurations at product sizes of 60, 70, 80 and 90 percent by weight of material less than 75μm.

The authors came to conclude that at the product sizes considered, two-staging hydrocyclones improves overall classification, yields higher circuit product rates and gives equivalent or more closely sized circuit product size than the one staged hydrocyclone, the magnitudes however of the difference vary with the operating conditions. There was an increase of only about 6 percent in productivity of the study circuit.

Discussion

There is some degree of confidence in the predictions that the authors came up with. This is because they have run experimental tests to validate their models. The authors confirmed some of the predictions made by Dahlstrom and Wai-Ping Kam (1988), such as that the water requirements for flowsheet 2 are quite substantial. The paper also provides the approach for conducting simulations on a full scale circuit; however, validating the full scale predictions could be quite costly. The value of the paper is that it shows the potential improvements in running a ball mill circuit with two stages of hydrocyclones in certain configurations; but this contradicts the earlier findings by R.T. Hukki (1976) who reported that two stage hydrocyclones did not show any worthwhile improvements.

They reported that flowsheet 2 (Figure 2-6) yielded an overall Sharpness Index of 0.62 and that of Flowsheet 1 was 0.6. Both the two stage configurations reduced the apparent bypass by about 20% for all the tests except with the test with the product with material of 60% minus 75μm. In a regrind mill circuit which has a finer product, the prediction of reduced bypass should be expected.

Conducting simulations to predict outcomes from changing some aspects of the circuits is very useful. Making changes in the full scale is very costly, so simulations are used to indicate whether the proposed changes will improve the operation or not. Sometimes the time frame for the study is too short for major changes to be made on the circuit. The simulations are then used to predict outcomes due to the circuit changes. However, simulations are idealistic in a way and what they predict will differ in a full scale test, and although the pilot scale data corroborated the simulation results, in full scale operation, conditions are not as ideal and easy to control and the results may differ. This indicates the need for plant scale tests to assess the 'real' benefits of changing circuit configurations.

2.9. *The Effect of Grinding on Mill Performance at Division Salvador, Codelco-Chile*

By J.B. Yianatos, L.G. Bergh and J. Aguilera (2000)

The authors of this paper assessed the effect that the grinding level has upon the rougher flotation performance in a copper mine. The grinding level is presented as the percentage of material bigger than 212μm because the recovery of copper above this size decreases sharply. Their study was focused on the effect of changing the grinding level on the rougher flotation recovery and upon the ore throughput.

A grinding model was developed based on the operating variables; this was done by using data from one year of operation. The Bond model was used to correlate the power (kW), ore throughput (tpd) and the grinding level (% +212μm). The Bond model was validated using the plant data. The next step was the fitting and selection of the particle size distribution model in terms of the grinding level which gave a good agreement with the plant data at different plant conditions. They reported a correlation of the metal distribution as a function of the feed size distribution.

The flotation model was based on the operating variables. The rougher flotation kinetics was described by Klimpel's model (cited in Yianatos *et al,* 2000). The authors report that Klimpel's model has been used by other workers to describe other plant operations with a very good agreement with plant data. Klimpel's model is described by the following equation:

Equation 2-3

$$R_{iPLANT} = R_{i\infty}\left\{1 - \frac{1-(1+k_i\tau)^{1-N}}{k_i(N-1)\tau}\right\}$$

where $R_{i\,PLANT}$ = actual plant recovery

$R_{i\,\infty}$ = maximum plant recovery at infinite time

k_i = maximum rate constant

τ = effective residence time in one cell of the bank

N = number of cells in series

Two approaches were used in order to estimate the model parameters, namely the kinetic sampling of the rougher flotation banks and the second approach was to scale up the batch test data to Klimpel's parameters. The model outputs are the cumulative recovery and grade and the residence time along the rougher flotation cells. Yianatos *et al* integrated the grinding and the flotation models and made a prediction of the effect of grinding on recovery. The authors found that when the grinding level is improved from 30% to 20% + 212µm, the copper rougher recovery is increased by 4.5% while reducing the tpd of copper in rougher concentrate by 13.6% (Figure 2-7).

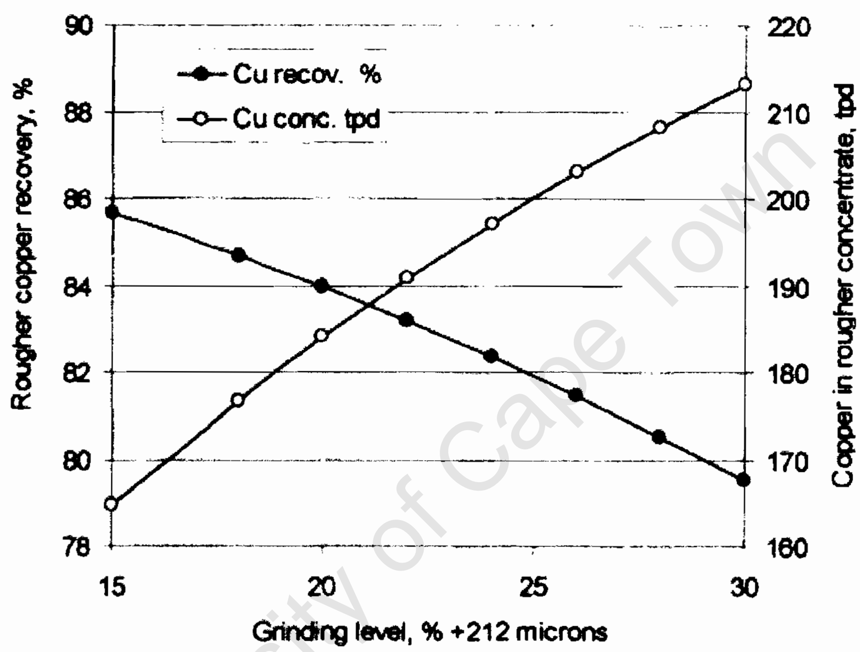


Figure 2-7 Copper recovery and copper production versus grinding level (Yianatos *et al*, 2000)

However operating the circuit with a rougher flotation feed of a grind level of less than 20% was not economically optimal due to reduced copper production despite the increased flotation recovery. This is due to the improved grind being traded off against the lower milling rate required to achieve the improved grind.

Discussion

The grinding model is purely empirical and it does not have any equipment variables; this limits the model to be used only at the Salvador concentrator.

The relevance of the paper to the thesis is how the grinding level affects the rougher flotation performance. As shown in the paper that at the Salvador, Codelco-Chile concentrator, the grinding level greatly affects recovery of copper; when reducing the percentage plus 212μm material, the copper recovery increases, albeit that the copper production decreases at the same time.

2.10. Effect of Particle Size on Flotation Performance of Complex Sulphide Ores

By D. Feng and C. Aldrich (1999)

This paper presents the effects that the particle size has on the flotation of two complex sulphide ores, Merensky and UG-2, as investigated by Feng and Aldrich. In particular they investigated the effects of particle size on grades and recoveries.

During their batch flotation experiments they controlled the pulp concentrations at 30% solids. The reagent dosages for the batch flotation tests were as follows:

- Activators → Copper Sulphate - 50g/ton
- Collectors → Sodium Isobutyl xanthate (SIBX)- 30 g/ton
  → Dithiophosphate – 30 g/ton
- Frother → Dow 200 – 60 g/ton
- Depressant → guar type – 100g/ton

The collectors and frother were added 5 minutes after the activator and the depressant was added after10 minutes. Aeration started after a minute and sampling commenced 20 seconds after aeration. The flotation cell used was 3 litres in volume and the aeration was kept at 3litres/min while the impeller was rotating at 1200 rpm. The froth height was kept at 15mm and sampling was at 10 second intervals with a scraper.

The authors conducted an investigation, where they studied the effect of particle size on flotation and the particles size distributions were characterized in terms of the percentage fines (-75μm).

Merensky ores exhibited lower recoveries at the fine size fraction and on the coarser fraction. Water recoveries (the carrying mechanism for the entrainment of particles) were higher at the finer particle size fraction, and this resulted in a greater degree of entrainment of particles at the finer fraction. As for the coarser fraction the authors reported unstable froths and lower froth volumes.

The medium size particle fraction was reported to have better flotation performance in terms of recovery and grade. This is probably due to the fact that fine particles show slow recovery rates, and are prone to entrainment. The coarser particles' recovery rates are affected by the disruption of bubble-particle aggregates in turbulent zones. The authors found the optimum flotation feed for Merensky ore to be approximately 60 % - 75µm, see Figure 2-8.

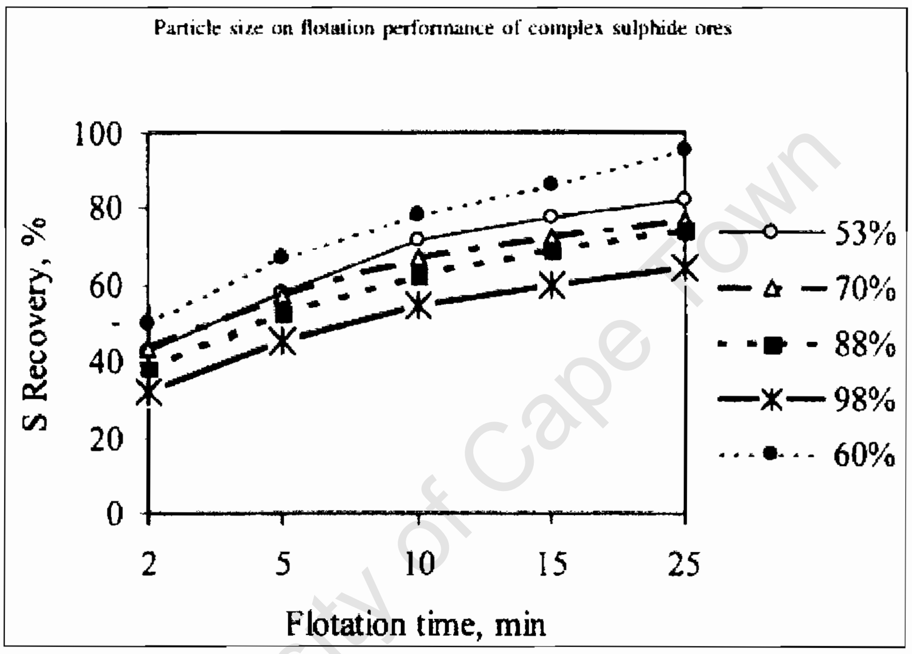


Figure 2-8 Variation in Sulphur recovery with concentration of fines (% minus 75µm) of Merensky ores at different flotation times (Feng and Aldrich, 1999)

Discussions

This paper studies the effect of particle size distribution on flotation recovery. This was accomplished by conducting batch flotation tests on samples with different percentages of minus 75µm. A one number description of a particle size distribution is, however very limiting. It would have been more informative to see the particle size distributions of the samples which were being compared, to assess the trade-off between decreased coarse fraction and increased fines fraction as the material subjected to flotation was incrementally finer.

2.11. A Rational Interpretation of the Role of Particle Size in Flotation

By W.J. Trahar (1981)

The influence of particle size on flotation is discussed. This is based on plant surveys of concentrator operations and related batch flotation test data. The batch data and the plant data apply to systems that have been optimised by the best methods available. The results of the tests have been presented as recovery-size curves in which the recovery from a particular size fraction has been plotted against the average size of that particular size fraction.

The recovery-size curves are arbitrarily divided into three regions; the fines region, usually 5-10μm which is difficult to float, the intermediate region, say from 10μm to about 70μm with particles that float the best and the coarse region which has sizes from about 70μm to an undefined upper limit as shown in Figure 2-9. This region might be easy or difficult to float, depending on the conditions of the system. The boundaries of the three regions are usually ill-defined, for that reason they will be referred to as fine, intermediate and coarse regions.

The flotation of fines is low compared to other size regions. This is mostly attributable to the decreased probability of successful collisions between particles and air bubbles as the particle size is reduced. A large proportion of the behaviour of fine particles is due to entrainment (carry over with water which enters the concentrate via the froth) of both floatable and non-floatable minerals. True flotation only attributes a small proportion of the total because the flotation rate is low for fine particles.

As for the intermediate region, the principal mechanism of recovery is true flotation. The contribution of entrainment is observed in the lower boundary of the size region. The particles in this region pose no difficulty to flotation. The range of the intermediate region varies with the mineral-collector system, where for sulphides the region extends to the coarser sizes.

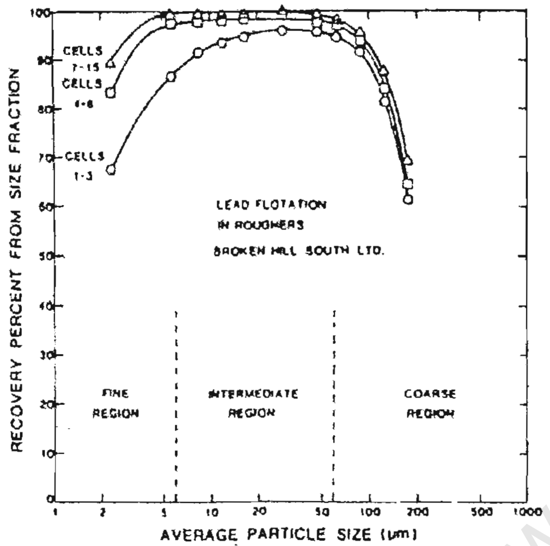


Figure 2-9 Cumulative lead recoveries in lead roughers (Trahar, 1981)

The recovery of the coarser region is mainly due to true flotation; the influence of entrainment is negligible. The flotation rate of the coarse region is usually lower than that of the intermediate region. As the collector dosage is increased the intermediate region widens and shifts to the coarser particles. However, increased collector additions slightly affect the fine region.

Other reagents that promote floatability have similar effects on the recovery-size curves. The differences are mainly observed at the coarse region. Influence of the activators has been observed to be similar to that of collectors on the floatability of malachite. It is shown that the effect of suppressants and depressants on floatability of pyrite in the presence of excess collector is the depression of the coarse particles. Flotation is suppressed in the coarser region if the environment becomes too reducing or oxidizing. It is reported that the coarse particles are the most susceptible to environmental changes.

A lot of the data presented in the paper was obtained from laboratory batch tests. This data was validated by plotting the plant data with the batch test data. The general pattern of behaviour was observed to be typical of the real plant data.

Discussion

This paper brings to light that grinding and classification systems that prepare the flotation feed, should be designed to yield high percentage of particles to be in the intermediate size region, which is associated with high flotation rates. The validation of the batch flotation data comes as a useful factor to the reliability of the data presented in the paper.

For the purposes of the thesis, the circuit configurations will be evaluated on the percentage of particles in the intermediate size region, which in the platinum industry is said to be between 10 and 100μm (Harris, 2000).

2.12. A Study of the Flotation Characteristics of Different Mineralogical Classes in Different Streams of an Industrial Circuit

By K.C. Runge, J.P. Franzidis and E.V. Manlapig (2003)

Runge *et al* opened their paper by saying: "Some researchers postulate that the floatability of a particle in a stream is a function of only its size and liberation properties. They do not discount the importance of reagent and particle contamination on particle floatability but assume that reagent coverage or surface contamination is uniform for particles of similar class and mineralogical properties when subjected to a particular chemical regime." However, through the use of a nodal analysis technique which was developed by Runge *et al* it was concluded that particles with the same size and mineralogical characteristics do not float at the same rate, but at a distribution of rates.

There is a possibility that the particles tested had different oxidation rates which resulted in varying particle flotation rates. It is normal to find a mineral exhibiting different recovery rates in a flotation circuit. Researchers have postulated that the turbulence in an aerated flotation environment could be resulting in attrition, de-conditioning and oxidation which could be resulting in the difference in the floatability of particles. Alternatively the difference being observed could be due to floatability distribution effects.

Runge *et al* conducted experiments within the lead cleaning circuit of Tech Cominco Red Dog Operation in April 1996. The experimental program consisted of collecting samples of major streams around the cleaning circuit. Batch flotation tests were carried out immediately after collecting a sample. This was done to minimise the ageing of samples between the sampling time and the batch flotation time. A sampling survey was conducted after all the samples were floated.

All samples from batch flotation tests and the flotation survey were assayed for lead, zinc and iron content. The samples were split into seven size fractions and each fraction was assayed for lead, zinc and iron content. This data was used to determine the mineral recovery in each stream.

NODAL ANALYSIS

The nodal analysis involves comparing the batch flotation tests recovery rates of streams, before a node and after a node. Where there are, say, two feed streams or product streams, they are mathematically combined based on their relative flow. The batch test recovery of a stream can be calculated using Equation 2-4.

Equation 2-4

$$R_{ParticleClass}^{CombinedStream,t,\min} = \frac{\sum_{s=1}^{n} F_{ParticleClass}^{Stream,s} \times R_{ParticleClass}^{Stream,s,t\min}}{\sum_{s=1}^{n} F_{ParticleClass}^{Stream,s}},$$

where R and F are the batch test recovery and solids flowrate of designated streams respectively.

The investigation endeavoured to find out whether a particular class group exhibits different recovery rates in different streams. If it does, nodal analysis is used to determine whether the observed variation is due to the modification of the particles. Using the nodal analysis Runge *et al* were able to tell that the variation of floatability was not due to the modification of particles.

Discussion

Nodal analysis is a useful technique to study the floatability of particles in a circuit. Runge *et al* used it to differentiate whether the variation of floatabilities that they observed around the cleaner circuit was due to modifications of the particles or due to the distribution of floatabilities within particles. Thus, it was concluded that the particle floatability is not based on size and mineralogical effects, alone.

There is therefore a possibility that even if particles are in the same size-mineralogy class, a distribution of flotabilities can be observed. The Nodal Analysis can be used for example, to directly compare recovery rates of two streams where one or both are made of one or more streams. Furthermore, it is a useful approach to show whether a change has been brought about by a particular process, be this as a result of a reagent addition or a further milling stage.

2.13. The flotation of pentlandite from pyrrhotite with particular reference to the effects of particle size

By G.D. Senior, L.K. Shannon and W.J. Trahar (1994)

A majority of the world's nickel is extracted from the mineral pentlandite which frequently occurs in ores containing pyrrhotite. The usual method of extracting nickel is concentration of pentlandite by flotation. This paper looks at a method of floating pentlandite selectively from pyrrhotite. This is because of the imposed stringent limits on smelter production of SO_2. So, to maximise production, the producers have to smelt concentrates high in pentlandite and low in pyrrhotite. Smelting pyrrhotite produces large quantities of SO_2 and small quantities of nickel.

Because of the above, the authors set out to investigate the floatabilities of pentlandite and pyrrhotite and to explore their behaviours that can be exploited to improve their separation. Batch flotation tests were conducted on pentlandite-quartz, pyrrhotite-quartz and pentlandite-pyrrhotite-quartz mixtures. Pentlandite was observed to be moderately more floatable than pyrrhotite in nickel ore under self induced conditions.

Another observation made by the authors is that it is more difficult for pentlandite to be separated from pyrrhotite in the absence of a collector, which means that pentlandite and pyrrhotite show similar responses to flotation in the absence of a collector. This observation is supported by Heiskanen *et al* (1991) who conducted experiments in collector-less conditions and the only variable was pH values. It was observed that at acidic pH values, the flotation recoveries of all sulphides (including chalcopyrite) were found to be good. Pentlandite did not float well at pH values above neutral and significant pyrrhotite flotation was observed only at acidic pH values.

Senior *et al* compared the recovery-size dependence of pentlandite to that of chalcopyrite. The results showed that chalcopyrite is more flotable than pentlandite in coarse and fine size fractions. Apparently the poor recovery of fine pentlandite is widespread in practice. Henley, Eltham and Tilyard (cited in Senior *et al*, 1994) analysed plant tailings and consistently reported large amounts of liberated fine pentlandite.

The authors found that pentlandite can be floated selectively by controlling the pH to about 9 and collector addition is maintained at moderate levels.

Discussion

It is interesting to note that the sulphides pentlandite, chalcopyrite and pyrrhotite which are intimately associated with the PGM's show different behaviours under specific conditions. The observations made in the paper emphasise the importance of a grinding and classification system that does not over-grind/under-grind particles, because, for instance the large proportion of pentlandite losses are due to poor flotability of the fine and coarse fractions.

Understanding of the behaviours of pentlandite, chalcopyrite and pyrrhotite are crucial in discussing their flotation recoveries. This is because these are the tested minerals for this thesis in batch flotation tests products.

2.14. Summary

The flotation process, being the concentrating stage in the mineral beneficiation plant, requires its feed to be prepared in a certain way. It is reported (Feng and Aldrich, 1999 and Trahar, 1981) that generally particles in the fine and coarse fractions do not float well and the Merensky ore is no exception. The medium size fraction normally yields higher recoveries. Senior *et al* (1994) reported that particles of the same size but different chemical composition respond differently to flotation. Chalcopyrite was found to be more flotable than pentlandite in coarse and fine fractions.

The fine fractions do not float well because of the decreased probability of particle-bubble collision caused by the reduced particle size. The poor flotation on the coarse fraction is due to unstable froths and lower froth volumes, flotable mineral not liberated and particles too big for bubbles to lift them.

In light of that, a grinding circuit should be designed to produce particle sizes in the optimal size range. Carpenter (1962) conducted an investigation to evaluate the performance of a closed circuit grinding system. His findings showed that some shortcomings to achieving the optimal results are associated with high circulating load, low classifier efficiency and overloaded ball mill. Due to the above, the flotation feed was found to have 38% of the particles (by weight) in the fine fraction where poor flotation has been observed.

The significance of high classifier efficiency in a closed grinding system is emphasised by Hukki (1976, 1977). He shows that an improvement in classifier sharpness of separation from 50% to 80% could improve the circuit capacity by 20% and the reduce power consumption by 15%. Attempts have been made to improve the classification efficiency by using two stages of hydrocyclones, however Hukki (1976) claims that there was not much success achieved with that.

Contrary to an above report, Dahlstroom and Kam (1988) reported that simulations demonstrated that two-stages of hydrocyclones (Figure 2-1, Flowsheet B) improved classification and reduced the amount of over-ground particles. They also reported that a two stage classification will improve the energy reduction since fines will be removed form the circuit. Dahlstroom and Kam had not conducted experimental work to support these findings at the time the paper was written.

Rogers *et al* (1981) support the case that two-stages of hydrocyclones improve classification efficiency and yield a higher product flowrate than the one stage classification at the same product size. Kelsall (1974) also investigated using multiple cyclone arrangement using mathematical models, lab scale experiments and full scale tests. The lab scale experiments supported the calculations, however the full scale tests did not show as good results as predicted. However, he is adamant that by making a few adjustments, this process can greatly improve classification by multiple cyclones.

Kobayashi *et al* (2003) reported that mill performance decreased with an increase in product flowrate. It was also reported that an increase in product (classifier overflow) median resulted as the classifier feed median was increased. Energy savings of about 5% were reported for a closed circuit grinding system as compared to an open circuit grinding system with a product of the same particle size median.

Runge *et al* (2003) used a Nodal Analysis Technique to study the floatability of particles in a flotation circuit. In particular, they used the method to differentiate whether the variation of flotabilities that they observed around the cleaner circuit was due to modifications of the particles or due to the distribution of flotabilities within particles. Thus, it was concluded that the particle floatability is not based on size and mineralogical effects, alone.

There is not much literature reported on the comparison between open and closed circuit configurations in industrial circuits and as a result there is no documentation on how to make this comparison in an industrial operation. The thesis will make this comparison in a regrind circuit of an industrial operation. Chapter 3 explores the possibility of conducting such a comparison in an industrial circuit.

Chapter 3

EXPLORATORY INVESTIGATION INTO COMPARING THE PERFORMANCE BETWEEN OPEN AND CLOSED CIRCUIT CONFIGURATIONS

3.1 Introduction

Chapters 1 and 2 describe the focus of this work and the level of knowledge that exists in the public domain pertaining to the impact that circuit configuration has on regrind milling performance and effectiveness. Little was however found in the literature suggesting or describing how best to assess the relationship between circuit configuration and performance for industrial circuits.

A preliminary investigation was therefore conducted as a way of developing an experimental methodology to make such a comparison. This survey was conducted at the Lebowa Platinum Mine (LPM) concentrator, and is described in the chapter that follows. There is a brief description of the concentrator process flow, which is followed by a description of the experimental procedure employed. The findings are discussed and an improved experimental methodology is developed. The detailed experimental procedure, as followed during the remainder of the investigation, is detailed in Chapter 4.

3.2 Background

The LPM concentrator receives and treats both Merensky and UG-2 ore in separate streams. The Merensky plant treats an average of 130 to 155 tph of ore. The LPM concentrator has three stages of grinding, namely primary, secondary and tertiary grinding. The ore is received in the concentrator after primary crushing, which is done in the mining section. The primary mill grinds the ore and the product is classified using the vibratory screens. The screen oversize is re-circulated to the primary mill and the undersize goes to the primary rougher flotation cells for the first stage of flotation.

The primary rougher concentrate is sent to the cleaner flotation cell for cleaning to obtain the final concentrate. The primary rougher tails are sent to the secondary mill for further grinding. A cyclone is placed ahead of the mill to produce a high density underflow which is fed to the mill and overflow with fine material which does not need further grinding.

The secondary milling circuit is given in Figure 3-1 and is operated in closed circuit with a cyclone. This is the section on the plant that was used to conduct this work, as the installed equipment and piping arrangement allows for operation in both open- and closed-circuit configurations.

For the closed circuit mill configuration, the mill product is sent to a classifying cyclone. The cyclone overflow is sent to the secondary rougher flotation cells and the underflow is sent back to the secondary mill for further grinding. For the open circuit mill configuration the mill product goes to the secondary rougher flotation cells shown with dotted lines in Figure 3-1.

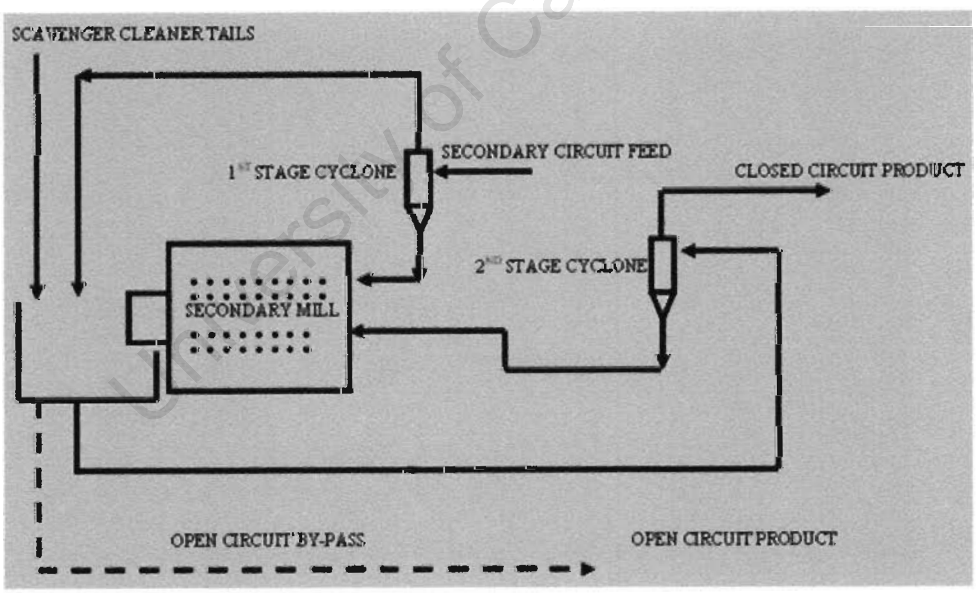


Figure 3-1 Lebowa Secondary Grinding Circuit

3.3 Experimental Procedure

The first task was to identify the critical streams where samples and flowrate measurements would be taken in order to conduct a complete mass balance around the open and closed circuit configurations. A couple of the identified sampling points required modification to enable the collection of representative samples. Figure 3-2 shows an example of a modified pipe for representative sampling. Pictures of examples of before and after modifications are given in Appendix G. After carrying out the modifications at all the relevant sampling points, the plant was ready to be surveyed.

The plant was required to be operated as close to steady state conditions as possible for at least 2 hours prior to sampling and these conditions maintained during the entire sampling period. This involved operating the plant with the same ore type at a consistent feed rate, mill inlet water addition, mill speed, mill load and mill power draw. Although the tests were being conducted on the secondary mill, the stability of the primary mill was critical in the stabilization of the circuit of interest. Therefore it was attempted to keep the primary mill in the most stable operating conditions possible during the sampling campaign. The only data available on the SCADA for the secondary mill was the power draw; the rest of the data logged during the test was for the primary grinding circuit.

Figure 3-2 Modified stream for representative sampling

The open circuit test started at about 18h30 on the 17th of February of 2003 and all sampling was finished at 20h00. For the closed circuit configuration the test started at around 5h30, the next morning and finished at 7h00. It was important to run the tests close to each other because the conditions (such as the ore type) had to be the same between the two tests.

The control room personnel were requested to record the operational data of the plant at intervals of five minutes and they were plotted on graphs for the two tests in Figure 3-3 and Figure 3-4.

Samples were cut at intervals of approximately 15 minutes around the circuit. Two sample buckets were collected at every sampling point, namely the primary sample and the back-up sample. An extra sample bucket was collected at the circuit product streams for batch flotation tests.

Batch flotation tests were conducted on the samples from the circuit product streams in open and closed circuit configurations. The procedure followed is laid out in Chapter 4. Batch flotation products were assayed for elements sulphur, copper and nickel. These are intimately associated with the Platinum Group Minerals (PGM's), and as such their behaviour approximate the behaviour of the PGM's.

3.4 Results and Discussions

3.4.1 Circuit Conditions

Figure 3-3 and Figure 3-4 show the operational information on the days of the experiments. The duration of the experiments are highlighted by a dashed box. The secondary mill inlet water (MIW), secondary mill power, secondary mill product flow rate and the tertiary mill power were fairly constant. They all have low standard deviations as can be seen in Table 3-1. There is no observed trending of any of the logged data. In reality, it is very difficult to achieve true steady state conditions in an operation such as the one under investigation. Operation personnel indicated that the operation of the circuit during the time of the survey was indeed typical of normal operation. For this reason the surveys were accepted as being representative.

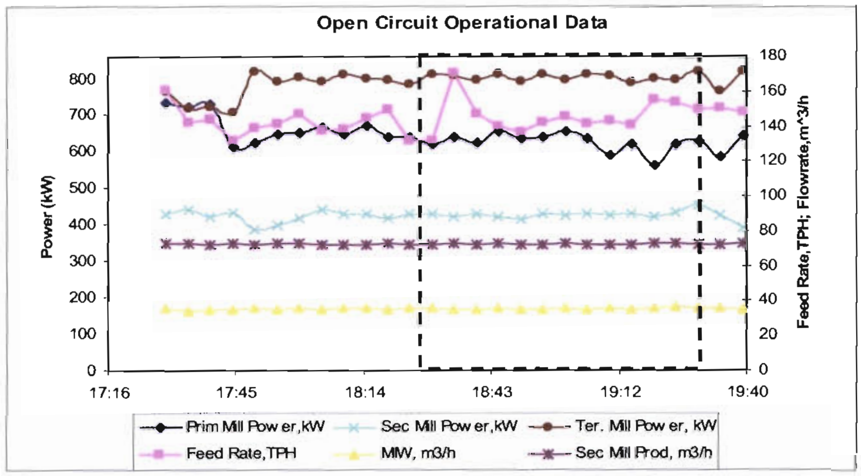


Figure 3-3 Lebowa Open circuit configuration operational plant data

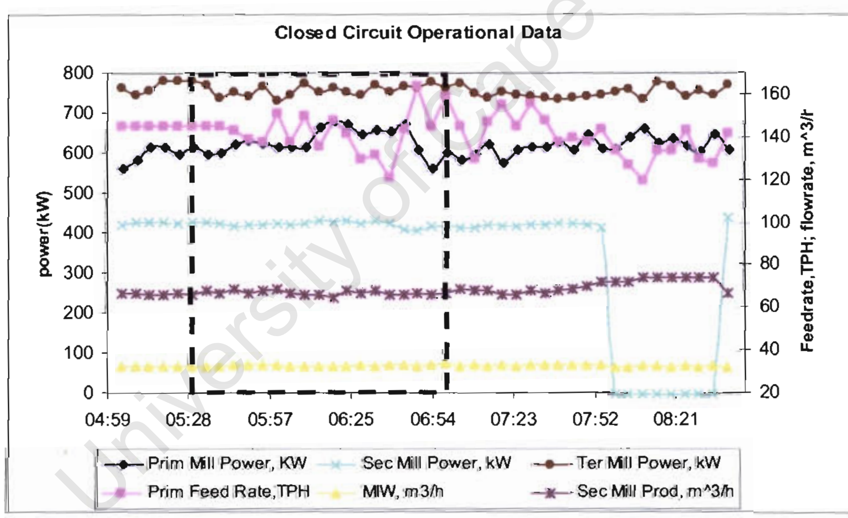


Figure 3-4 Lebowa closed circuit configuration operational plant data

Table 3-1 Summary of the averaged operational logged data of the plant

	Open Circuit		Closed Circuit	
	Averaged Logged Data	Standard Deviation	Averaged Logged Data	Standard Deviation
Prim Mill Power, KW	626	25.3	629	31.7
Prim Feed Rate,TPH	147	9.8	143	9.8
Primary MIW, m3/h	36	0.5	33	0.3
Sec Mill Power,kW	430	8.9	423	6.9
Sec Mill Prod, m3/h	73	0.3	67	1.1
Ter. Mill Power, kW	805	9.1	759	13.7

Table 3-2 shows the mass flowrates of various streams on the circuit and some conditions of the circuit for both tests. The circuit was receiving roughly the same amount of feed during the two circuit tests.

Table 3-2 Operational conditions for the open and closed circuit tests

		OPEN CIRCUIT TEST		CLOSED CIRCUIT TEST	
		Exp	Bal	Exp	Bal
Mass Flowrate (tph)	Secondary Circuit Feed	147	147	143	143
	1st stage cyclone U/F		92.2	68.0	71.7
	1st stage cyclone O/F		53.1		68.0
	Sec Ball Mill Prod		92.2		91.0
	sec mill sump		153		177
	2nd stage cyclone U/F			19.4	19.3
	2nd stage cyclone O/F				158
	Scavenger Cleaner Tails	7.6	7.6	18.3	18.2
	Slurry Level (mm)	59		319	
	Secondary Ball Mill Filling (%)	31.6		31.3	
	Secondary Mill Speed (RPM)	16.0		16.0	
	Mill Fraction C_s (Critical Speed)	0.71		0.71	
	Secondary Mill Inlet Water (TPH)	5.9	6.3	1.6	2.0

3.4.2 Stream Particle Size Distribution

Figure 3-5 and Figure 3-6 show the particle size results in a graphical form. The particle size distribution results shown are the experimental (markers) and mass balanced (solid lines) results. The agreement between the experimental and the mass balanced data is an indication of the good quality of the data.

There are however some streams that indicate a disagreement, the primary circuit product (secondary circuit feed) for both open and closed circuit configurations and the open circuit (OC) cyclone underflow in Figure 3-5. This could be as a result of sampling and analytical errors. The errors of the OC cyclone underflow are attributed to analytical errors, associated with sample handling and processing.

The deviation of the circuit feed can be attributed to sampling errors because it is observed in both open and closed circuit cases, and is a large and difficult stream to sample effectively. There is a rougher floatation stage between the primary circuit and the secondary circuit. The primary circuit product was the measured stream, it was used to approximate the secondary circuit feed by assuming a 5% mass pull in the flotation cells and assuming that the flotation concentrate consisted of the same size distribution as the rougher flotation tails which feeds the secondary circuit. It could be that one of these assumptions was indeed not accurate.

It should be noted that CC and OC in Figure 3-5 and Figure 3-6 are used to denote 'closed circuit' and 'open circuit' configurations respectively. This convention is maintained throughout the thesis.

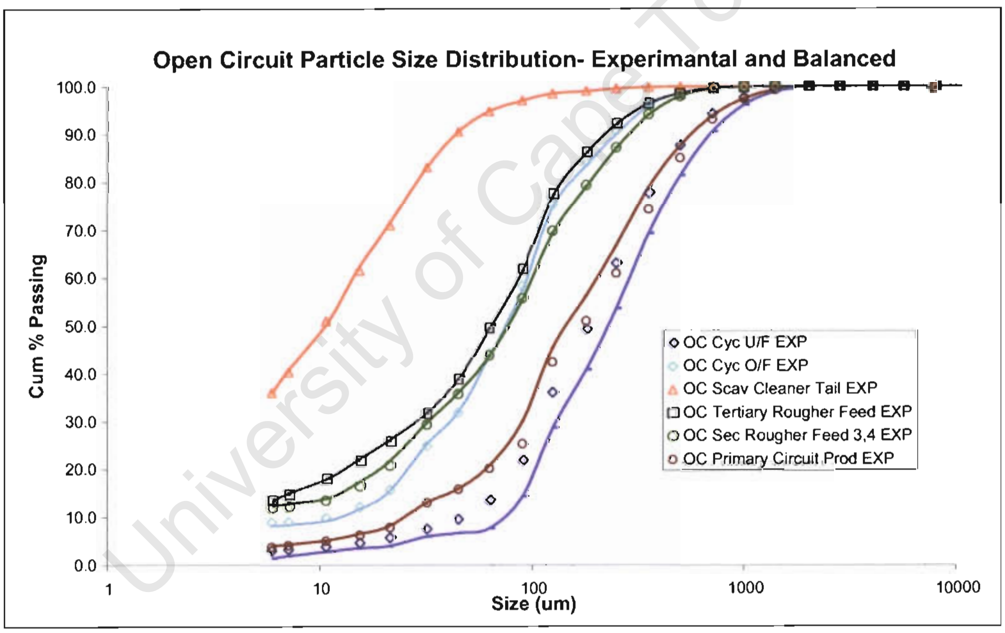


Figure 3-5 Open circuit streams particle size distributions

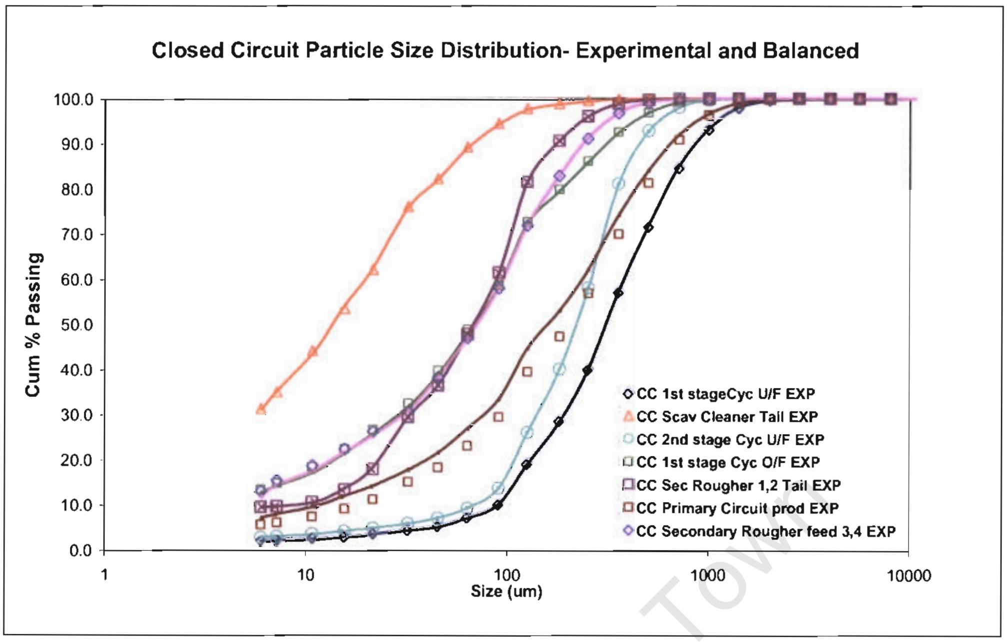


Figure 3-6 Closed circuit streams particle size distributions

3.4.3 Streams' Percent Solids

The percent solids of the various streams were calculated from the wet and dry masses of the samples and the results are presented together with the mass balanced results in Table 3-3.

The cyclones had very dilute underflows. They operated in the range of 50 to 58% by weight; the percent solids of the underflows should at least be 75% for optimal operation. A low percentage of solids concentration is extremely detrimental to the mill performance. The first stage cyclone overflow products are over-dilute, at 20% solids. Given that the feed density is in the typical region for this application (38% Solids), and the fact that the underflow is also dilute, suggests that the cyclone used is too big for this application.

Table 3-3 Lebowa Concentrator Secondary Circuit-Streams Percent Solids

	Open Circuit		Closed Circuit	
Stream	**% Solids**	**% Solids**	**% Solids**	**% Solids**
	Exp	**Bal**	**Exp**	**Bal**
primary circuit product	38.1	37.1	39.0	39.0
1st stage Cyclone U/F	50.0	50.3	52.5	51.8
1st stage Cyclone O/F	19.1	25.4	31.7	31.0
Sec Ball Mill Prod	-	48.6	-	52.8
Mill Sump Prod	-	31.5	-	34.7
2nd stage cyclone U/F	-	-	59.4	59.4
2nd stage cyclone O/F	-	-	-	33.0
Scavenger Cleaner Tails	9.0	8.7	13.2	15.4

It was noticed that 6 tph and 2 tph mill inlet water was added during the open and closed circuit tests, respectively; this made the mill contents even more dilute. The first stage cyclone underflow is already too dilute for good mill operation and the addition of water to the mill would have a further detrimental effect on the milling performance in both cases.

3.4.4 Product grind comparison between open and closed circuit configurations

The comparison of the particle size distribution between the open and closed circuit configuration is presented graphically in Figure 3-7. The circuit feed and product in both configurations have close particle size distributions. This shows that contrary to the hypotheses, the closed circuit configuration did not yield a better performance than the open circuit configuration. To quantitatively make a comparison between the two circuit configurations, the Bond calculation was used.

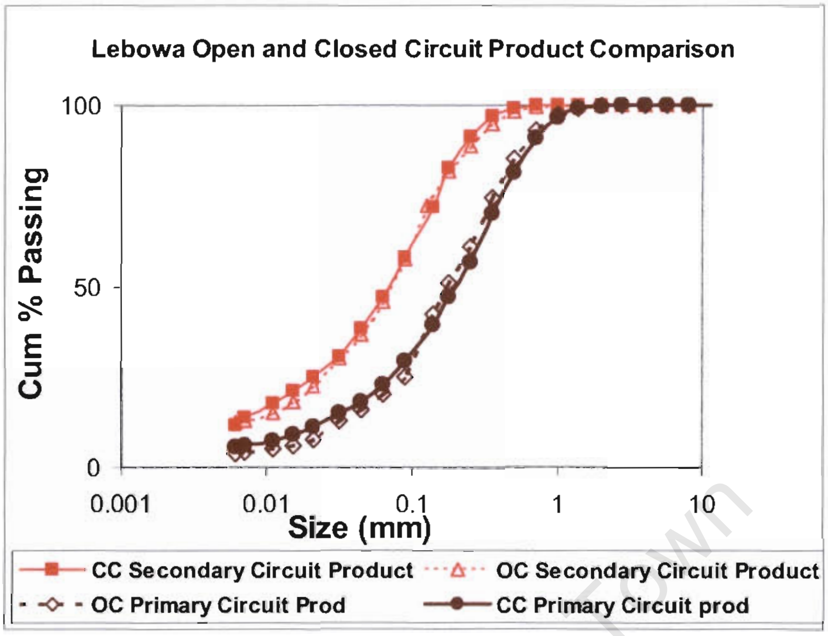


Figure 3-7 Comparison of the feed and product size distribution between open circuit and closed circuit

The Grinding efficiency for the circuits was worked out using the Bond Work equation (Equation 3-1). The results of the calculations are shown in Table 3-4. The closed circuit configuration had a slightly higher grinding efficiency as indicated by the low operating work index (*WIo)*. The mills during the two circuit configuration performed differently. The mill product during the open circuit mode had a grind level of 47.5% -75μm compared to 35.5% for the closed circuit mode (Table 3-5). Similar observations were made in the tests conducted at Waterval and Amandelbult, which will be expanded on later in Chapter 5.

Equation 3-1

$$WI = \frac{E}{10}\left(\frac{\left(\sqrt{P80}\right)\left(\sqrt{F80}\right)}{\sqrt{F80}-\sqrt{P80}}\right);$$

Where

E = Specific energy, kWh/t
F80= 80 % of material in the feed that passing, μm
P80 = 80 % of material in the product that passing, μm
WI = Work Index

Table 3-4 Operating Work Index- Lebowa Secondary Circuit

	Closed Circuit Configuration	Open Circuit Configuration
Power Draw, kW	423	430
FeedRate, tph	143	147
Specific Energy, E,kWh/t	3.0	2.9
P80 ,µm	162	175
F80 ,µm	431	368
Wio (Operating Work Index)	9.7	12.5

Table 3-5 Open and closed circuit modes summary of stream properties

	Open Circuit			
Stream	**P 80 (mm)**		**% - 75µm**	
	Exp	**Bal**	**Exp**	**Bal**
Primary Circuit Product/ Secondary Circuit Feed	0.421	0.368	22.7	25.2
1st stage Cyclone U/F	0.381	0.479	17.4	10.7
1st stage Cyclone O/F	0.15	0.154	51.5	50.5
Sec ball mill feed	-	0.479	-	10.68
Sec Ball Mill Prod	-	0.196	-	47.5
Scavenger Cleaner Tails	0.0284	0.0285	96.1	96.1
Mill Sump Prod/ Secondary Circuit Product	-	0.175	-	51.0
	Closed Circuit			
Stream	**P 80 (mm)**		**% - 75µm**	
	Exp	**Bal**	**Exp**	**Bal**
Primary Circuit Product/ SecondaryCircuit Feed	0.481	0.431	26.1	30.1
1st stage cyclone O/F	0.619	0.611	8.5	8.5
1st stage cyclone U/F	0.182	0.185	53.7	53.2
Sec ball mill feed		0.552		9.06
Sec ball mill prod	-	0.21	-	35.5
Sec mill sump prod	-	0.19	-	48.1
Scavenger Cleaner Tails	0.0397	0.04	92.2	92.0
2nd stage cyclone U/F	0.349	0.349	11.2	11.2
2nd stage cyclone O/F / Secondary Circuit Product	-	0.162	-	52.7

Although about 6 tph of water was added into the mill during the open circuit mode and only 2 tph for the closed circuit mode, the slurry level inside the mill above the balls was measure to be 59 mm for the open circuit test and 319 mm for the closed circuit test. These were measured after crash stopping a mill immediately when sampling was completed. That could explain the lower power consumption (as seen in Table 3-1) for the secondary mill during the closed circuit mode.

This is explained by the argument that power draw is proportional to the torque. Mills with a higher slurry level will have more slurry that lags behind as the charge rotates with the mill. The slurry pool has a torque opposing the charge and the resultant torque is reduced, and hence the power drawn decreases.

3.5 Conclusions

The circuit configurations can be assessed successfully by conducting sampling surveys on each circuit configurations. The mass balance, although fairly good showed where additional care should be practiced when collecting, preparing and analyzing samples.

There was no significant difference between the circuit products' particle size distributions for the two circuit configurations. The mill was grinding finer during the open circuit configuration than in closed circuit configuration. The effect of closing the circuit with the cyclones was shown by bringing the circuit's grind level to the same as that of the open circuit configuration although the closed circuit mill was grinding coarser.

Chapter 4

EXPERIMENTAL DETAILS

4.1. Introduction

This chapter presents the revised experimental procedure as partly discussed in the preliminary investigations. The chapter starts by discussing some of the requirements of conducting a sampling survey where streams' data is obtained and the principles of the technique used during the experiment.

There is also a brief description of possible errors associated with conducting a sampling survey. The list of equipment and their characteristics are outlined later in the chapter and then the test procedure as followed during each of the plant campaigns is described.

There are three circuit configurations that were studied, that is, the open circuit, 1-stage and 2-stage closed circuits. The flowsheets of these circuit configurations are presented in this chapter.

4.2. Description of Procedures and Principles

The performance of the circuit configurations was assessed by conducting a sampling survey around each circuit. A sampling survey consists of collection of samples which are representative of the streams in a circuit and collecting standard plant data to conduct a complete mass balance of a particular circuit, where the solids mass flowrate, water and particle sizes in each streams are determined.

Prior to conducting a survey which aims to estimate the behaviour of a highly dynamic system, such as the typical minerals processing operation, an effort needs to be made to ensure that the processes are at "steady state". True steady state is highly improbable given the large number of interacting disturbances at play in a typical ore treatment plant, however every effort needs to be made to ensure that the process is as close to steady state as reasonably possible. This is attained by keeping the feedrate as constant as possible and keeping the cyclone feed pump speed constant. This condition should be maintained for the duration of the sampling survey.

The collected samples are processed to determine size distributions and percent solids of the various streams. The percent solids are obtained from the wet and dry masses of the samples. The dry bulk samples are sub-sampled to produce aliquots of manageable mass. The particle size distribution is then determined. Two methods were used to size the samples, that is, the screening method and the laser diffraction method. The principles of these are discussed in Appendix C. Screen analysis was conducted down to 32μm and the lower sizes were obtained using the laser diffraction system. The splitting and screening methods are presented later in this chapter.

Batch flotation tests were conducted on selected streams of each circuit configuration. In Amandelbult five streams were floated for open and 2-stage closed circuit configurations. The streams floated are the circuit feed, 1st stage cyclone overflow, both mill products and the circuit product. At the Waterval concentrator the streams floated are the circuit feed and product for the 1-and 2-stage closed circuit configurations. Circuit configurations are shown from Figure 4-1 to Figure 4-3.

The aim of the batch flotation program was to assess the effect that each circuit configuration has on the flotation response. Flotation testing is usually conducted on plants for improvement of procedures, development of new reagents, testing of reagents and in order to develop a circuit for new ore.

'Hot' batch flotation tests as referred to in this thesis, are batch flotation tests conducted on samples taken directly from a stream in an industrial circuit. The duration between taking the sample and conducting the batch flotation test is kept to a minimum to eliminate potential ageing effects. The sample would still be warm, hence the term *hot batch flotation.* The streams were selected around a node, so that what is coming into a node was measured and the effect of the node seen by measuring what is coming out of the node.

4.3. Description of the Studied Circuit Configurations

There are three circuit configurations that were studied. These are the open, 1- and 2-stage closed circuit configurations. In all the three concentrators, the regrind circuit is operated in the 2-stage configuration. They all have a bypass to allow an open circuit configuration. In Waterval concentrator, the circuit design is such that the 1-stage closed circuit configuration can be operated.

Figure 4-1 shows the open circuit configuration of a tertiary milling circuit. The feed to the circuit is classified by the 1st stage cyclones and the cyclone underflow is sent to the mill. The cyclone overflow combines with the mills' product to make the circuit product. The primary node being the circuit with the circuit feed and product as the streams entering and leaving the node.

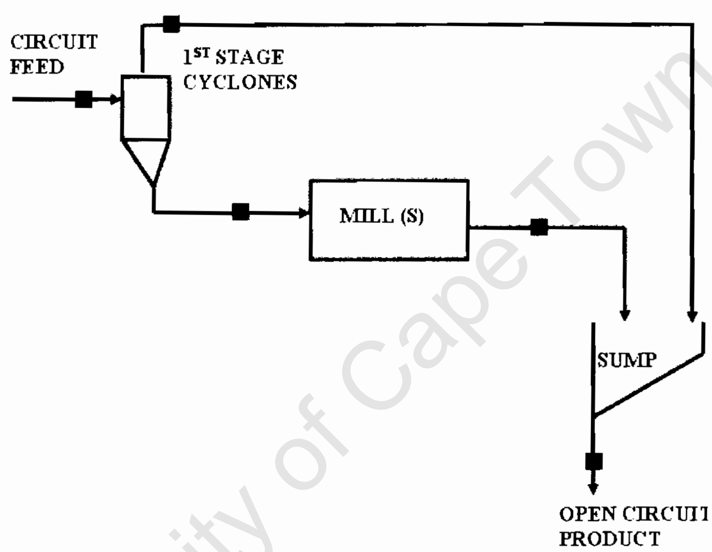


Figure 4-1 Open circuit configuration

As mentioned above, during each survey, samples were collected for size analysis and batch flotation tests. In Figure 4-1 to Figure 4-3, streams from which samples were collected for size analysis are marked with a square.

Figure 4-2 shows the same circuit in a 2-stage closed circuit configuration. Unlike the open circuit configuration, the mills' product together with the 1st stage cyclone overflow is sent to the 2nd stage cyclones. The 2nd stage cyclone underflow is re-circulated back into the mill and the overflow is the circuit product. This is not the most common configuration for closing a circuit. The circuits that were studied were converted (prior to the commencement of this work) from open to closed circuit by adding the second stage of cycloning.

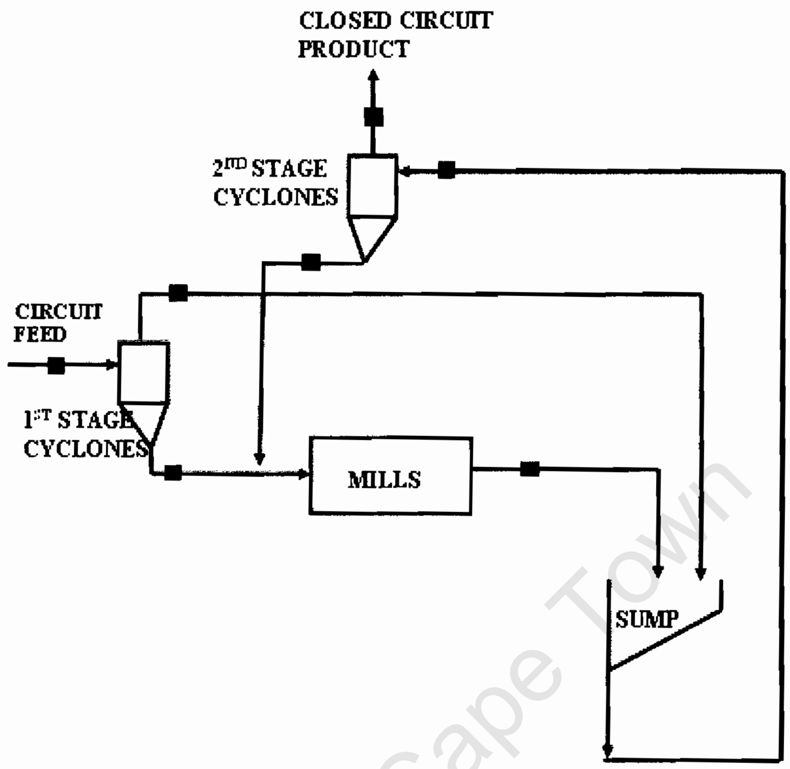


Figure 4-2 2-stage closed circuit configuration

Figure 4-3 shows the more common configuration for closing the milling circuit with circuit's feed being introduced into the mill discharge sump. The cyclone then classifies a combined stream consisting of the mill discharge and circuit feed

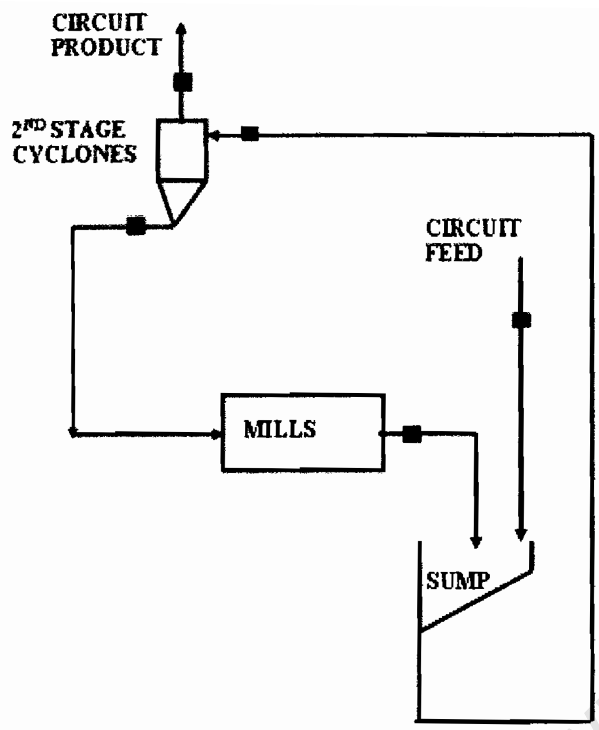


Figure 4-3 1-stage closed circuit configuration

The reagents typically added to the circuit were:

- Collector (such as sodium iso-butyl xanthate, or SIBX) at the mill product sump; and,
- Activator (copper sulphate, $CuSO_4$) dosed to the mill feed.

To ensure a similar chemical environment existed during the batch flotation tests, samples downstream of the mill feed did not have the activator added during the batch flotation tests and the samples downstream of the mill product sump did not have the collector added. However, batch flotation tests on samples taken ahead of these dosage points would have these reagents added in the required amounts.

4.4. Details of a Sampling Survey

A sampling survey involves collecting samples from the streams around a circuit. It was crucial that the tests, comparing two circuit configurations, were conducted as close together as possible. This was done so the operating conditions of the circuit between the tests would be as similar as possible. These conditions include ore type, flowrate through the circuit, etc. The same sampling procedure was followed during the test campaigns at the Lebowa, Amandelbult and Waterval Merensky concentrators.

4.4.1. Sampling Procedure

The procedure for each sampling survey is outlined below. This is divided in three sub-sections according to the order of events.

a) Pre-Test Inspection

The procedure prior to the sampling survey is outlined below:

- Two labelled buckets were placed at each sampling point. Three buckets were placed where a sample for batch flotation was required. All buckets and lids had tarred masses marked on them.

- The circuit was monitored at least two hours prior to the sampling survey for its stability. Since the circuits under study are regrind circuits, pressure was monitored where a pressure gauge was installed, otherwise circuit feed flowrate was monitored by measuring the flowrate at the beginning and at the end of the test.

b) During the Survey

Once it was determined that the circuit is stable, sampling was conducted for an hour. Samples were collected at intervals of 15 minutes at all the sampling points. The general procedure for cutting a sample is outlined in point form below:

- The opening of the sample cutter was held perpendicular to the flow of the stream
- The sample cutter was passed across the flow at an even speed, and then back again ensuring full coverage of the stream flow
- The sample was carefully emptied into a primary bucket smoothly and quickly
- The sample cutter was then tapped upside down to make sure there was nothing left in the sample cutter.

- A second sample was cut and emptied into a backup bucket
- A third sample was collected for the sampling points which needed a batch flotation test sample.
- Whenever the sample was not cut successfully, that is, if it was spilled or something else went wrong, the sample was dumped and another one was taken.
- The stability of the circuit was being monitored during the sampling survey.

c) Post Sampling Survey

Flowrate measurements were made at some of the sampling points, to give enough information to be able to conduct a circuit mass balance. The methods used for flowrate measurements are discussed in section 4.4.2.

After the sampling survey, all samples were weighed wet; they were then filtered and then dried in the ovens. When dry, they were removed from the ovens, and then left at room temperature to cool. Once they had cooled down, the samples were weighed to determine the dry mass. The percent solids of the samples were calculated from the wet and dry masses of the samples.

4.4.2. Flowrate Measurement Techniques

A couple of techniques were used for the measurement of streams' flowrates. The most basic used was the bucket and stop watch method, where a bucket of known volume was put under a stream and the time it takes for the slurry to fill the bucket was measured. This gives the slurry flowrate in m^3/h. The solids flowrate (F) in tph is calculated by:

Equation 4-1

$$SolidsFlowrate(tph) = SlurryFlowrate(tph) - WaterFlowrate(tph)$$

$$F = SG_{solids}\left[SlurryFlowrate(\mathrm{m}^3/\mathrm{h}) - \left(\frac{100 - \%SOL}{\%SOL}\right)F\right](tph)$$

Therefore

Equation 4-2

$$F = \left(\frac{SG_{solids} \times SlurryFlowrate}{1 + SG_{solids} \times \left(\frac{100 - \%SOL}{\%SOL} \right)} \right) (tph)$$

Where SG is the specific gravity and % SOL is the percentage of solids in the slurry.

Alternatively the bucket of slurry would be weighed to get the wet mass. This would give the slurry flowrate in tph, and the solids flowrate in tph was calculated from:

Equation 4-3

$$F = \%SOL \times SlurryFlowrate(tph)$$

In a number of situations a launder, which was either already in place or was installed specifically for the campaign, was used to estimate the flowrate of the stream flowing across it. A sample cutter would go across the stream at a constant speed. The time that the sample cutter spends in the stream will be taken. A couple of these cuts would be taken, accumulating the sample. The solids flowrate would be calculated from:

Equation 4-4

$$F = \left(\frac{AccumulatedSample \times \%SOL \times WidthOfLaunder}{SampleCollectionTotalTime \times SampleCutterWidth} \right) (tph)$$

4.5. Sample Processing Procedures

Once the samples were dry, they were then split into representative smaller fractions, to facilitate easy handling. The section presents the splitting technique used, and later, the screening procedure that was followed.

4.5.1. Sample Dividing/Splitting

The splitting procedure for the dry samples is outlined below in point form

- The splitter and all the cups were cleaned before they were used.

- The dry sample was added slowly into the cone of the rotary splitter (Figure 4-7)
- The motor was started and then feeder vibration which was set to a minimum was then started.
- At the end of the first split, cups which were directly opposite to each other were combined. Sets of cups at opposite sides were put together to make half of the sample and it was put back into the cone.
- This was continued until each cup had about half of the required mass. Four sub-samples were collected: a screening sample, a laser sizing sample, an assay sample and a back-up sample. Each of these weighed at least 300 g.

4.5.2. Sizing Methods

Screen sizing was conducted down to 32μm. The sub-32μm sizing was conducted using laser diffraction method. Principles of both the screening technique and the laser diffraction method are discussed in Appendix C. Screening was conducted in two stages, namely, wet and dry screening. This section presents the method of conducting both wet and dry screening.

a) Wet Screening

The screening sub-sample was then taken for wet screening. A brief description of the wet screening method is given below:

- A 32μm screen was placed into the ring of the screen shaker (Figure 4-9)
- An empty clean bucket was placed underneath the screen.
- A pre-weighed sub-sample of at least 300g was added onto the 32μm screen
- Using a hosepipe, water was introduced on the screen to wet the whole sample
- The shaker was started up and controlled amounts of water was added on to the screen

- Screening was continued until only clear water was being discharged into the bucket
- Water was turned off and the 32μm screen was removed from the shaker
- A 125μm screen was placed onto the ring of a screen shaker and a clean empty bucket was placed underneath
- All the plus 32μm material was placed on the screen
- The shaker was started and controlled amounts of water was added continually onto the screen
- Screening was continued until only clear water was being discharged into the bucket
- The plus 125μm material was transferred into a laboratory dish and the screen was cleaned thoroughly to ensure that all particles are removed from the screen into the dish
- Excess water was removed from the dish and the sample was put in the oven to dry
- The procedure was repeated for 90, 63, 45, and 32μm screens
- The minus 32μm material from the first and second screening were combined, then filtered and dried.
- Masses from all the dry samples were recorded
- All the plus 125μm particles were kept for dry screening

b) Dry Screening

The dry plus 125μm is dry screened as follows:

- The following sieve series was used for dry screening: 710, 500, 355, 250, 180 and 125μm
- A stack of screens, starting with the one with the largest apertures on top was made and one with the smallest apertures was placed at the bottom
- The stack was placed on the screen shaker with the pan at the bottom of the sieve stack
- The timer of the shaker to was set to 25 minutes and the vibration was set at an amplitude of 0.8 mm

- The plus 125μm particles from the wet screening were placed on the top screen and the lid was placed on the shaker and the clamps were tightened
- The shaker was then started up
- When the shaker stopped at the end of the 25 minutes, the screens were off loaded and each size fraction was weighed separately; each time cleaning the screen with a brush.

4.6. List of Equipment

This section describes the list of equipment during the test work and sample preparation.

4.7.1. Sample Cutters

The sample cutters used were designed such that the cutting edges are straight and parallel. The depth of the sample cutter was made to be large enough so that the sample does not splash out or overflow. Different sample cutters were used for different streams. The sample cutter in Figure 4-4 to Figure 4-6 were used for the cyclone underflows, cyclone overflow and mill discharges (or inclined cyclone underflows) respectively.

The cutters where designed specifically for the streams as described above. The cutter widths were decided based on the size of particles in that particular stream. For instance, a 10mm cutter width was used for a cyclone overflow because this stream carries relatively fine particles and for the cyclone underflow 60mm was used for the cutter width.

A piece of a conveyor belt was cut to cover the top of the sample cutter to stop any material from splashing out of the sample cutters. Whenever necessary handles for sample cutters were designed to enable easy access to difficult to reach streams. Handles were bolted on to the sample cutters, for easy attachment and removal, depending on the accessibility of the stream to be cut.

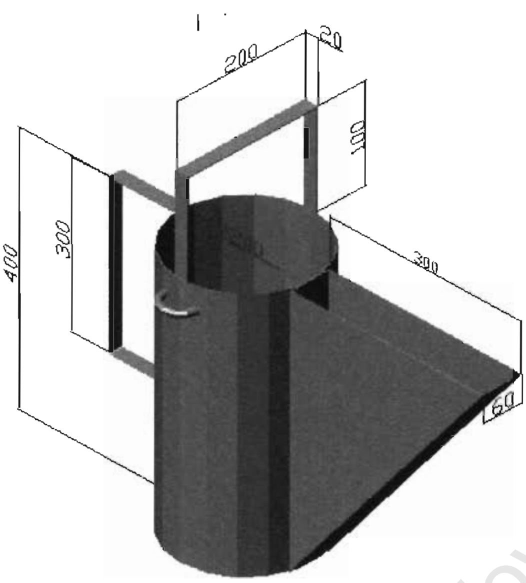


Figure 4-4 Hydrocyclone Underflow Sample Cutter

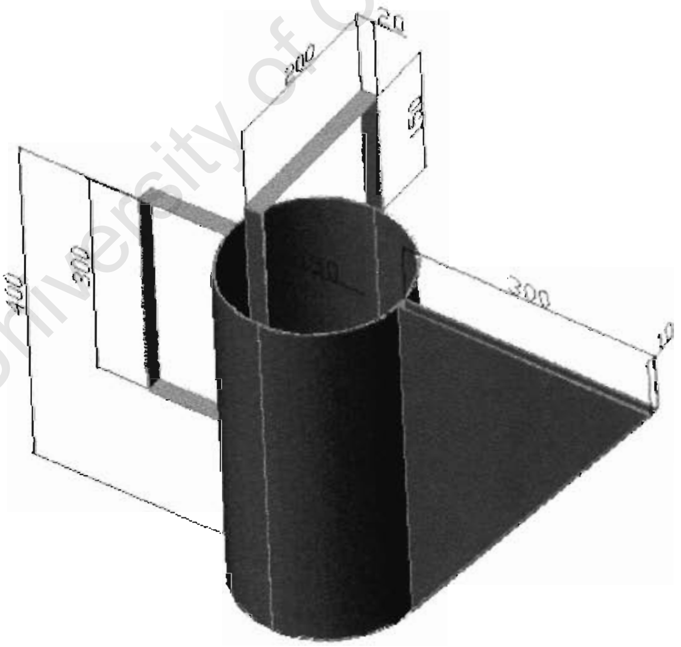


Figure 4-5 Hydrocyclone Overflow Sample Cutter

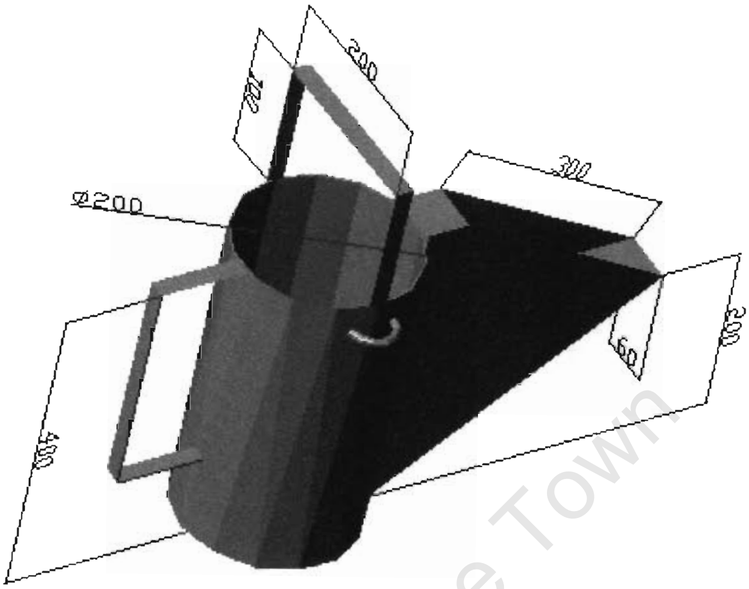


Figure 4-6 Mill Discharge Sample Cutter

4.7.2. Sample Processing Equipment

The equipment described in this section was used for the preparation of samples, which includes splitting and sizing. Dry samples were split into equally representative manageable quantities. The samples were sized in two stages, the wet screening and then dry screening.

Figure 4-7 Rotary splitter

The rotary splitter shown in Figure 4-7 was used to split the dry sample into smaller masses of at least 300g, but equally representative. The screening shakers in Figure 4-8 and Figure 4-9 were used for dry and wet screening as described earlier in the chapter.

Figure 4-8 Dry screening shaker

Figure 4-9 Wet screening shaker

4.7. Mass Balancing

The obtained size distributions, flowrates and percent solids were entered into JKSimMet. The results were mass balanced to check the integrity of the results. JKMBal which is included in JKSimMet was used for mass balancing. Steady state conditions are assumed i.e. input tph = output tph. One stream in the circuit is assumed accurate. This stream is usually from a trusted reading; in this case it was a measured flowrate. JKMBal is based on the minimization of the sum of weighted squared errors, given by Equation 4-5 (Napier-Munn *et al*, 1999):

Equation 4-5

$$SSQ = \sum_{j=1}^{N}\sum_{i=1}^{L}\left(\frac{X_{ij} - x_{ij}}{\sigma_{ij}}\right)^2 + \sum_{i=1}^{L}\left(\frac{A_i - a_i}{\sigma_i}\right)^2 \text{ with respect to } x_{ij} \text{ and } a_{ij}$$

N = number of measurements

L = number of streams

X = measurement (solids, assay, size)

x = adjuxted measurement

A = measured flow

a = adjusted flow

σ_{ij} = weight or standard deviation for measurements

σ_i = weight or standard deviation for the flow

JKMBal can display values of the estimated SSQ which can be used as a guide to interprete if a balance is good or bad. Low values of the SSQ indicate a good balance whereas high values of SSQ show a poor balance.

In conjuction with looking at the SSQ value, one can judge the success of the balance by comparing the order of magnitude of the SD value to that of the associated flow value. If the SD is small compared to the flow value, this implies a well defined flowrate. A plot of the experimental and the adjusted size values gives an indication of the the success or the failure of the balance.

There is no absolute measure for a "correct" mass balance. The usual assesment is that the balanced and the experimental size values lie on top of each other and the percent solids and the solids flowrate require minimal adjustments; less than 1% and less than 5% for percent solids and solids flowrate respectively.

4.8. *Possible Errors*

The main objective of a sampling survey is to obtain samples that are representative of the operation of a circuit at that period. The samples are then processed through a number of steps, which include drying, splitting, sizing and assaying to obtain the properties of the circuit including particle size distribution, streams' percent solids, streams' flowrate and sulphur and base metal contents. However there are disturbances and errors associated with this process and are discussed below (Napier-Munn *et al*, 1999).

a) A process is rarely in **steady state**. Plant dynamics is a problem. A plant is sampled over a period of 1-2 hours to smooth out the short term disturbances in the process.

b) **Sample Cutter designs** should be such that the edges are horizontal and parallel. The cutter width should at least be three times the particle size diameter for streams with the top size particles of greater than 3mm. As for streams with particles with sizes less than 3mm the cutter width should at least be 10mm.

c) A consistent **Sampling Technique** should be adopted, where the sample cutter is passed through the stream at the same slow and even rate for each increment taken from that stream.

d) **Sub-sampling a primary sample**. There are a few standard methods available to sub-sample a primary sample. A method of obtaining a representative sub-sample is a requirement for analysis of stream size data and for assays. The bulk sample is already a representation of the stream, a sub-sample, which represents the bulk sample, also represents the stream.

   There is an old Cornish method called Coning and quartering (Wills, 1997). It consists of gathering the material into a heap and relying on its radial symmetry to give four identical samples when the heap is flattened and divided into four sections. Two opposite corners are taken as a sample, the other two are discarded. The chosen portion may be coned and quartered again until a required size is obtained. This method is very dependant on the operator and should not be used for accurate sampling.

A second method is table sampling, where the material is poured at the top of an inclined table which has got a series of holes. Prisms are placed in the path of the stream to break it into fractions. Some material falls into the holes and is discarded while material remaining in the plate goes to the following row of prisms and holes, more material is recovered. This goes on for a couple of rows. The material reaching the end of the table is sample. This method is often used to divide samples of 5kg and above.

The rotary splitter method is the modern and most common sub-sampling technique used. A picture of a rotary splitter is shown in Figure 4-7. The material is poured into a funnel and a vibrating feeder feeds the material into the rotating rack of cups. The amplitude of vibration can be varied depending on the fineness of the material. This method is well accepted as the most accurate splitter technique and was used in this investigation.

e) **Analytical errors**, which include weighing, using screens with worn or incorrect apertures, inadequate screening time, using incorrectly calibrated or un-calibrated equipment, malfunctioning rotary splitters. Other factors giving rise to errors include loss of sample, especially of the fines.

f) **Propagation of errors in calculated quantities.** Some quantities are calculated when they are difficult or impossible to measure. **The fundamental error**-This is a statistical uncertainty involved in choosing a small sample to represent a large population. How big should a sample be? An equation (Equation 4-6) has been developed based on Gy's theory (Barberry (cited in Napier-Munn *et al* 1999)), which accommodates the fundamental error.

Equation 4-6

$$M = \frac{f \rho d_m^3}{\theta^2 P}$$

where	M	=	mass of sample required (g)
	f	=	shape factor for material (0<f<1)

ρ	=	density of material (g/cm^3)
d_m	=	mean size in size range of interest (cm)
P	=	expected proportion of material in size range of interest
θ	=	standard deviation of the number of particles in that size range.

In reality the mass requirements for assay, size distribution measurement and backup meant that samples taken in the surveys far exceeded the theoretical minimum mass predicted by the above equation.

Chapter 5

OUTCOMES FROM THE STUDY OF OPEN AND TWO-STAGE CLOSED CIRCUIT CONFIGURATIONS

5.1 Introduction

Three sets of investigation programs were conducted where open circuit was compared with the 2-stage closed circuit configuration. These were conducted on three different plants, namely, Lebowa, Amandelbult and Waterval concentrators. The Lebowa findings were presented in the preliminary investigation section (Chapter 3). This chapter focuses on the results from the open and 2-stage closed circuit comparative tests conducted at the Waterval and Amandelbult concentrators.

A further set of tests was conducted at Waterval Concentrator; to compare the performance of the 2-stage closed circuit configuration with a more traditional 1-stage closed circuit configuration (as shown in Figure 4-3). This is presented in Chapter 6.

Prior to each sampling survey, the plant was setup and confirmed to operate in the chosen configuration. The plant was then allowed to stabilise for 2 hours prior to commencing with sampling.

This chapter starts by looking at the plant conditions under which the tests were conducted. It then presents the results from the comparison between open and 2-stage closed circuit, based on the degree of particle size reduction across the circuit. Particle size distribution results are presented in the form of graphs showing cumulative percent passing particular size fractions. Partition curves are used as a graphical representation of the hydrocyclone performance.

5.2 Test Conditions

This section discusses the plant conditions for the days of the tests. This is done in two sections, starting with the conditions from Amandelbult and then from Waterval concentrators.

AMANDELBULT

The tests at the Amandelbult concentrator were conducted on the 8th of May 2003. The tests were conducted back to back in order to maintain the same conditions, and allow the best comparison between the two configurations.

Figure 5-1 to Figure 5-4 show the logged operational data of the milling plant at the Amandelbult concentrator during the tests. The primary and the secondary stages were fairly stable. This is judged by the stable power consumption, feedrate and the mill loads, although there is some short-term fluctuation in power there is no long term trend. The tertiary circuit of Amandelbult concentrator was monitored for its stability during the test by measuring the flowrate of the circuit feed stream, this was done both at the beginning and at the end of the test. This method was used because there were no pressure gauges installed on the circuit.

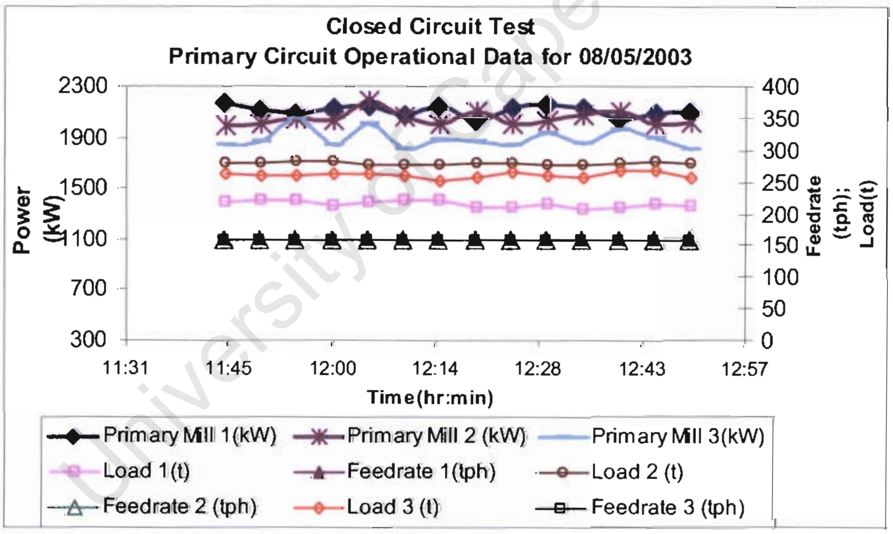


Figure 5-1 Amandelbult primary circuit operational data – closed circuit test

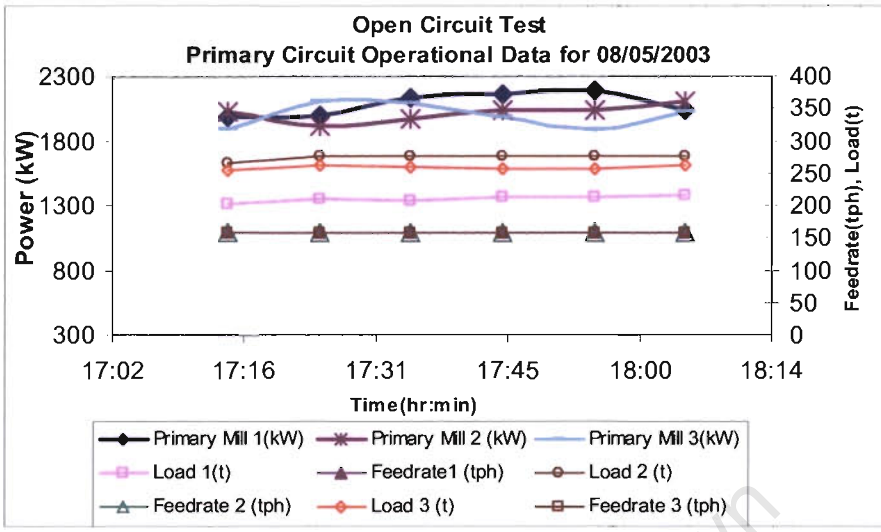


Figure 5-2 Amandelbult primary circuit operational data – open circuit test

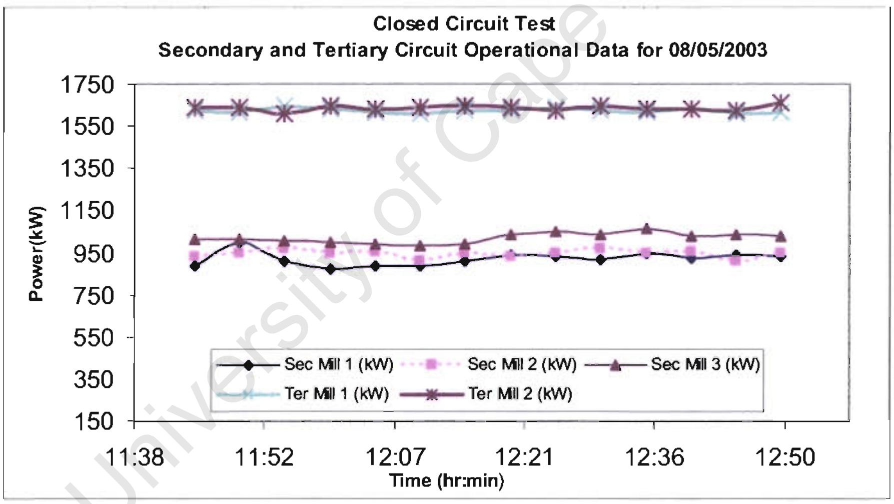


Figure 5-3 Secondary and tertiary circuits operational data – closed circuit test

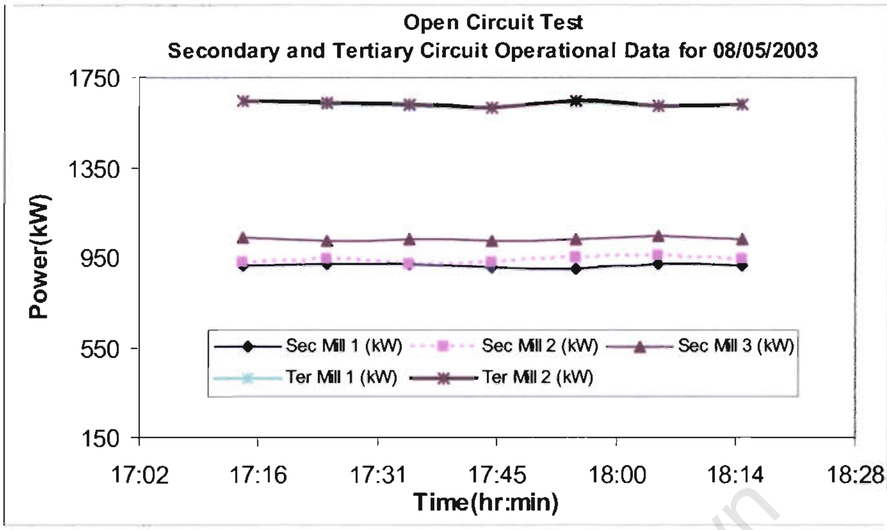


Figure 5-4 Secondary and tertiary operational data – open circuit test

Table 5-1 shows the table of the summary of operational data during the open and the closed circuit configuration tests. This table summarises the data from Figure 5-1 to Figure 5-4. It also shows the standard deviation of these logged data which are relatively low indicating that the deviation of the data from the mean is not significant with the highest deviation being 5%.

Table 5-1 Summary of operational data - Amandelbult

		Closed Circuit Test		Open Circuit Test	
Primary Circuit		Average	Standard deviation	Average	Standard deviation
	Primary Mill 1 (kW)	2113	42.20	2088	81.07
	Primary Mill2 (kW)	2050	54.85	2020	61.13
	Primary Mill 3 (kW)	1888	70.97	2003	88.03
	Load 1 (t)	215	4.92	211	4.38
	Load 2(t)	280	2.14	276	4.41
	Load 3 (t)	261	4.91	260	2.99
	Feedrate 1 (tph)	160	0.00	160	0.00
	Feedrate2 (tph)	160	0.00	160	0.00
	Feedrate 3 (tph)	160	0.00	160	0.00
Secondary Circuit	Sec Mill 1 (kW)	921	32.29	912	8.82
	Sec Mill 2 (kW)	943	16.88	939	11.27
	Sec Mill 3 (kW)	1019	24.37	1032	8.64
Tertiary Circuit	Tertiary Mill 1 (kW)	1624	11.17	1632	10.21
	Tertiary Mill 2 (kW)	1636	13.22	1634	9.78

From the above figures it can be seen that stable operation was maintained prior to and during each test at Amandelbult Concentrator.

Waterval

The stability of the Waterval Concentrator was monitored by the pressure at the 1st stage of hydrocyclones. There is a pressure gauge in the central box that distributes the feed to the individual cyclones. Figure 5-5 and Figure 5-6 show the pressure readings in graphical forms over the time of the closed and open circuit tests, respectively. The circuit was fairly stable during the two tests with no trending of pressure over the periods of the tests.

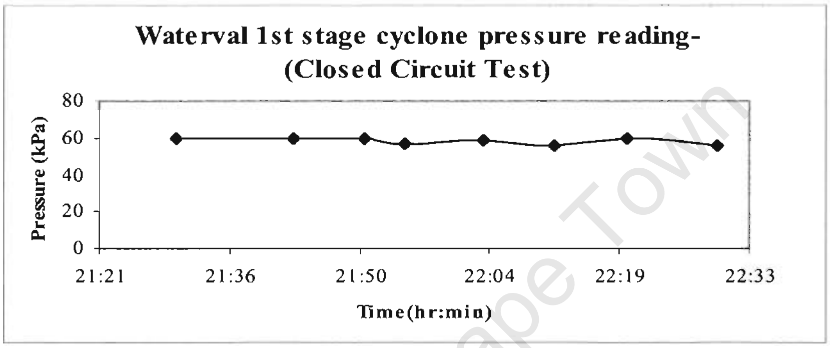


Figure 5-5 Pressure readings of the 1st stage hydrocyclone during the open circuit test

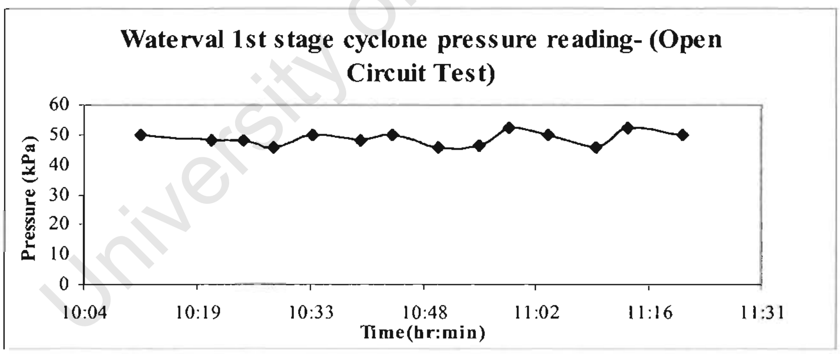


Figure 5-6 Pressure readings of the 1st stage hydrocyclone during the open circuit test

From the limited information available on Waterval concentrator, it can be seen that plant operation was stable at the time of the campaign. There was a higher pressure during the closed circuit test at roughly 60kPa than during the open circuit test at roughly 50kPa – the feedrates agree with the pressure because the flowarate was higher during the closed circuit test as shown in Table 5-3.

5.3 Experimental and Mass Balanced Results

Figure 5-7 to Figure 5-10 show the particle size distributions of both the open and closed circuit configuration tests from Amandelbult and Waterval concentrators. The solid lines indicate the mass balanced particle size distributions and the markers show the experimental data. A table of experimental and mass balanced results is presented in the Appendix F. There is a good agreement between the experimental and balanced results for both campaigns, which is an indication of quality data.

A mass balance is said to be of good quality if the difference between the experimental and the balanced data is within one percent for percent solids, five percent for mass flowrates and the particle size distributions curves match closely (Powell, 2004).

The particle size distributions for the Amandelbult were sized down to below 32μm using the laser sizing system, however, due to time constraints and the fact that the laser sizing procedures were conducted in Johannesburg, the Waterval results were only sized down to 32μm.

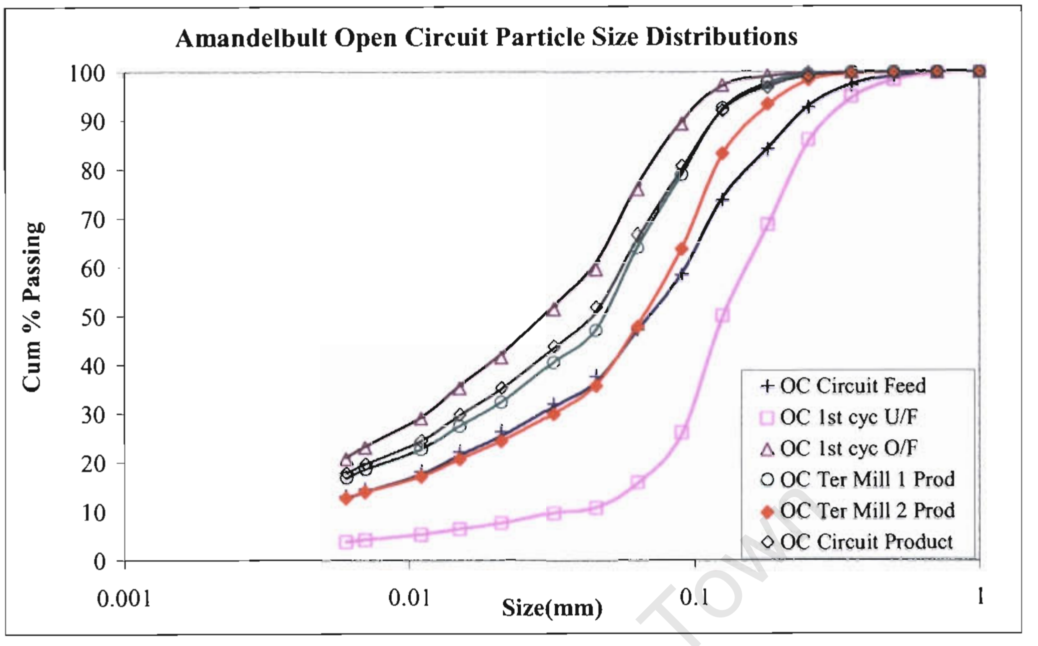


Figure 5-7 Amandelbult open circuit particle size distributions

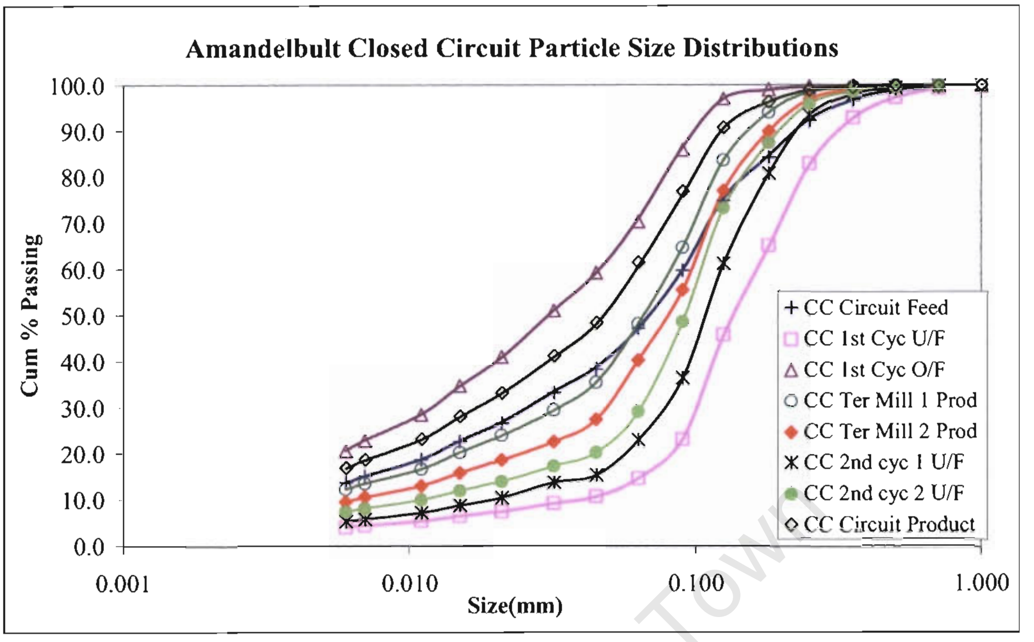


Figure 5-8 Amandelbult closed circuit particle size distributions

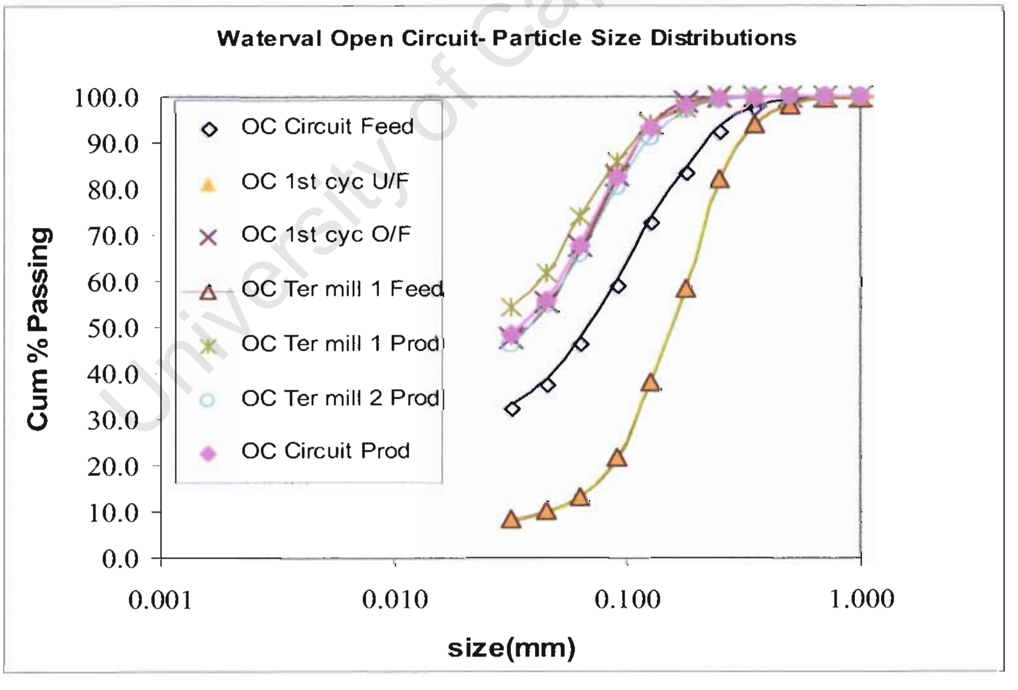


Figure 5-9 Waterval open circuit particle size distributions

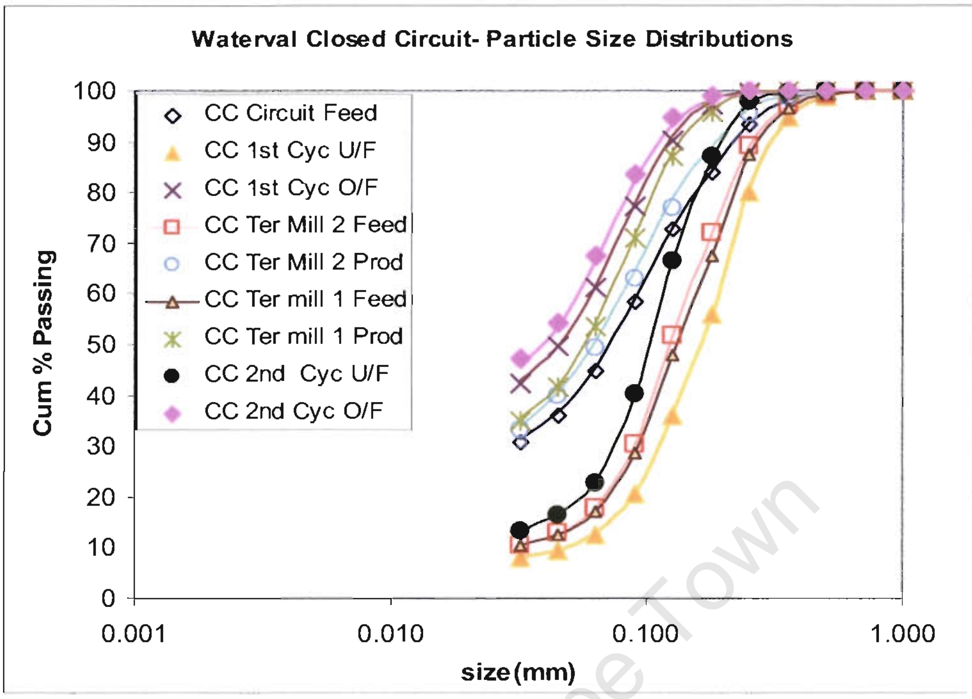


Figure 5-10 **Waterval closed circuit particle size distributions**

Table 5-2 and Table 5-3 show the stream properties around the Amandelbult and the Waterval tertiary circuits, respectively. The tables compare the experimental and balanced solids flowrate (tph), P80 (mm), % solids and the % -75µm between the open and closed circuit configurations. There is a close correlation between the balanced and the experimental results. This is another indication of the success of the mass balance.

Table 5-2 Amandelbult experimental and balanced stream properties

	TPH Solids				% Solids			
	Closed Circuit		Open Circuit		Closed Circuit		Open Circuit	
Stream	Experimental	Balanced	Experimental	Balanced	Experimental	Balanced	Experimental	Balanced
Circuit Feed	291.6	291.6	353	353	36.6	35.8	35.9	34.4
1st Cyc U/F	-	122.7	-	171	63.4	62.4	63.8	62.1
1st Cyc O/F	-	168.9	-	181	26.2	27.4	26.2	24.2
Ter Mill 1 Prod	61.0	61.9	65	65	62.6	63.5	61.2	61.8
Ter Mill 2 Prod	74.3	76.1	107	107	59.7	60.8	61.4	62.3
2nd Cyc1 U/F	-	6.9	-	-	61.7	61.6	-	-
2nd Cyc 1 O/F	-	133.7	-	-	-	32.9	-	-
2nd Cyc2 U/F	-	8.3	-	-	57.2	57.1	-	-
2nd Cyc 2 O/F	-	157.9	-	-	-	38.8	-	-
Circuit Product	291.6	291.6	353	353	36.0	35.8	33.3	33.3
	P80 (mm)				% - 75 microns			
	Closed Circuit		Open Circuit		Closed Circuit		Open Circuit	
Stream	Experimental	Balanced	Experimental	Balanced	Experimental	Balanced	Experimental	Balanced
Circuit Feed	0.151	0.148	0.155	0.153	53.7	53.8	52.9	53.1
1st Cyc U/F	0.235	0.236	0.22	0.222	18.8	18.7	20.8	20.6
1st Cyc O/F	0.077	0.076	0.069	0.068	79.4	80.0	83.8	84.5
Ter Mill 1 Prod	0.116	0.116	0.092	0.092	56.9	56.9	72.0	72.3
Ter Mill 2 Prod	0.134	0.134	0.118	0.118	48.2	48.2	56.1	56.4
2nd Cyc1 U/F	0.177	0.177	-	-	29.5	29.4	-	-
2nd Cyc 1 O/F	-	0.093	-	-	-	71.1	-	-
2nd Cyc2 U/F	0.146	0.146	-	-	38.9	38.9	-	-
2nd Cyc 2 O/F	-	0.100	-	-	-	67.3	-	-
Circuit Product	0.096	0.097	0.089	0.090	69.8	69.0	74.3	73.6

Table 5-3 Waterval experimental and balanced stream properties

	TPH Solids				% Solids			
	Open Circuit		Closed Circuit		Open Circuit		Closed Circuit	
Stream	Exp	Bal	Exp	Bal	Exp	Bal	Exp	Bal
Circuit Feed	-	300	-	368	33.0	33.4	36.8	36.8
1st stage cyc UF	105	105	120	121	67.5	66.4	69.1	68.3
1st stage cyc OF	-	195	-	247	26.5	26.4	30.1	30.0
Ter mill 1	36	36	121	121	65.6	67.7	65.1	66.7
Ter Mill 2	69	69	95	97	64.6	65.7	68.5	68.5
2nd stage cyc UF	-	-	-	96	-	-	66.5	66.4
Circuit Product	-	300	367	368	33.2	33.4	36.8	36.8
	P80 (mm)				% -75 microns			
	Open Circuit		Closed Circuit		Open Circuit		Closed Circuit	
Stream	Exp	Bal	Exp	Bal	Exp	Bal	Exp	Bal
Circuit Feed	0.160	0.152	0.158	0.156	52.8	54.5	51.7	52.4
1st stage cyc UF	0.241	0.242	0.249	0.247	17.2	17.1	16.3	16.4
1st stage cyc OF	0.084	0.085	0.096	0.094	76.1	75.2	69.8	70.5
Ter mill 1	0.089	0.089	0.107	0.105	73.9	73.8	62.8	63.6
Ter Mill 2	0.075	0.075	0.138	0.129	80.7	80.7	56.4	57.9
2nd stage cyc UF	-	-	0.104	0.103	-	-	31.1	31.4
Circuit Product	0.084	0.084	0.156	0.158	75.9	76.3	76.5	75.3

5.4 Comparison between the circuit configurations – The Mills' performance

This section compares the performance of the mills between open and closed circuit configurations. Both at Amandelbult and Waterval concentrators there have two mills in parallel. Figure 5-11 and Figure 5-12 show the particle size distributions for the mills' feed (1st stage cyclone underflow in open circuit configuration), and mill 1 and 2 product streams for Amandelbult and Waterval concentrators respectively.

It is observed that the product of mill 1 is consistently finer than that of mill 2 in open and closed circuit configurations from both Amandelbult and Waterval concentrators (Figure 5-11 and Figure 5-12). It is questionable whether the split is perfect (i.e. what goes into mill 1 is the same as what is going into mill 2) because the measured throughput for mill 1 and 2 is not the same (Table 5-2 and Table 5-3). It could be that mill 2 was consistently receiving a coarser feed than mill 1. This could not be tested during the campaigns as the individual mill feed streams could not be accessed.

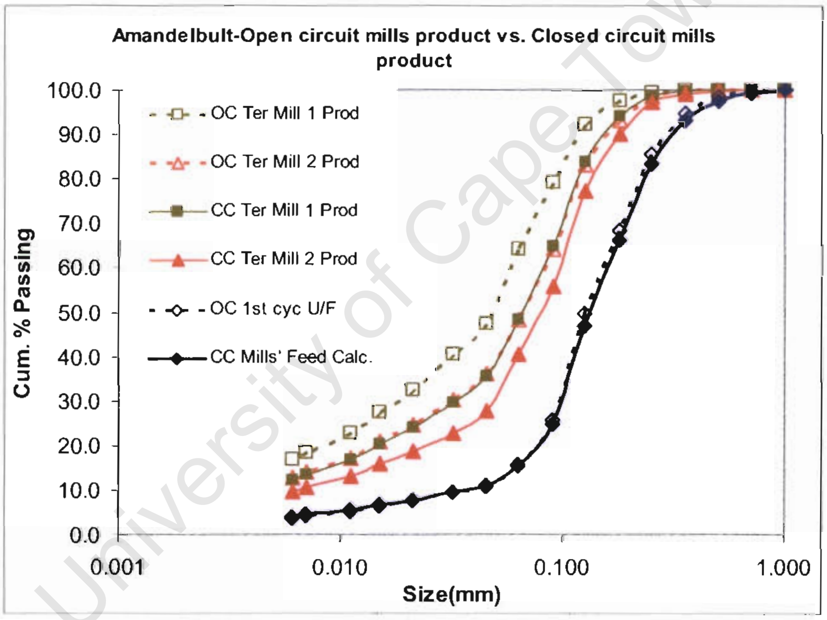


Figure 5-11 Comparison between the mill feed and the mill products for open and closed circuit configuration at Amandelbult

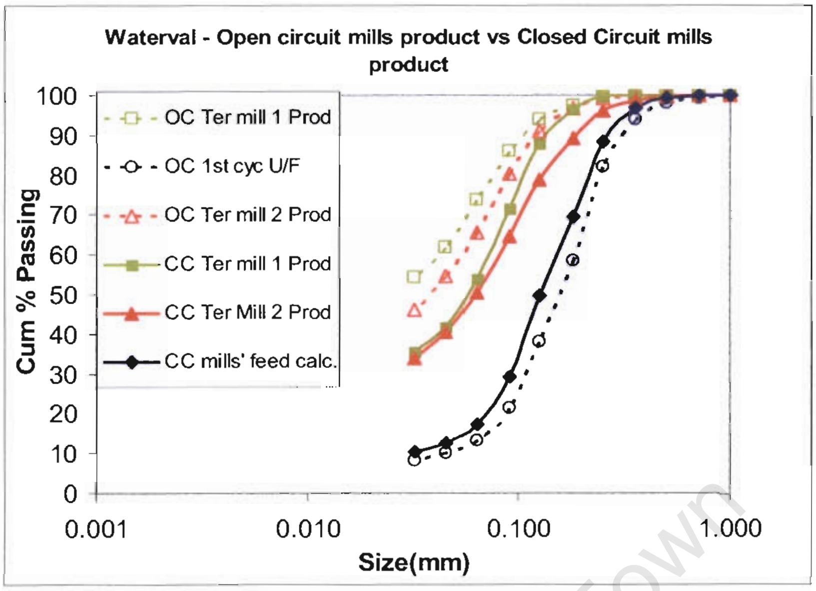


Figure 5-12 Comparison between the mill feed and the mill products for open and closed circuit configuration at Waterval

From Amandelbult concentrator (Figure 5-11) the mills' feed was the same in both open and closed circuit configuration. However, the combined mill product is finer for the open circuit configuration than in the closed circuit configuration.

At Waterval concentrator the mills' feed was coarser during the open circuit mode. However, the open circuit mode had a finer product than the closed circuit mode (Figure 5-12) suggesting that the mills were grinding more efficiently in the open circuit than in the closed circuit configuration. A similar observation was made at Lebowa concentrator (see Chapter 3), where the mill products had 48% and 36% -75μm for open and closed circuit configurations respectively.

This behaviour could be attributed to the high amount of fines in the circulating load due to inefficiencies in the cyclones (this will be discussed in more detail in section 5.6). The mills in closed circuit configuration receive fine particles which could have dropped the efficiency of grinding. Shi and Napier-Munn (2002) reported that decreasing the amount of fines in the mill feed increased the Grinding Index which they used to describe a mill performance. The Grinding Index represents the overall breakage of particles passing through the mill. It varies from 0%, in which there is no particle breakage, to 100% in which all the material in the feed above a specified size is reduced to below this size.

Although the throughput could cause the observed effect, this is discounted as similar results were observed at Lebowa where the mills' throughputs were identical (Chapter 3). At Amandelbult the mills' throughput was higher during the open circuit test (Table 5-2) and at Waterval the mills' throughput was higher for the closed circuit test (Table 5-3).

In all the mills tested, the make-up ball size was 30mm. This was found to be bigger than the recommended sizes for this application, using the Allis Chalmers rule of thumb (Equation 1-3). From the both the Waterval and Amandelbult concentrators the recommended make-up sizes were calculated to be less than 20mm. Using bigger balls has an adverse effect in terms of the production of fines. This indicates that the mill grind can also be improved by using smaller balls. However, it may not have been practical to use smaller balls due to the high cost associated.

5.5 Comparison between the circuit configurations – Circuit performance

A comparison of the performances between the circuit configurations was made by comparing the particle size reduction between the circuit configurations. Comparison of the feed and the product streams between open and closed circuit configuration for Amandelbult and Waterval concentrators are given in Figure 5-13 and Figure 5-14 respectively.

During the Amandelbult programme, the circuit feed was very similar for the two configurations (54% and 53% -75μm for open and closed circuit configurations respectively). Surprisingly, the open circuit product had 74% -75μm which was slightly finer than the closed circuit product which had 70% -75μm (refer to Table 5-2 and Figure 5-13). This clearly indicates that the open circuit configuration had a higher degree of size reduction than the closed circuit configuration.

Because these investigations were conducted in an industrial regrind circuit, it was not possible to control the throughput in the circuits. It was later learned that the circuit throughput was not the same between the tests. The circuit throughput was lower during the open circuit configuration test, at 353 tph, whereas it was 292 tph for closed circuit test (Table 5-2). Even though that was the case, the closed circuit configuration had no benefit over the open circuit configuration in terms of particle size reduction in contrast to the hypothesis (refer to section 1.2. for the hypotheses).

At Waterval concentrator, the feed size distribution was close with 53% and 52% -75μm for the open and closed circuit configurations respectively. The product was also close for the open and closed circuit modes. The circuit throughput was higher at 368 tph during the closed circuit test than during the open circuit test with 300 tph. Kobayashi *et al* (2004) studied the characteristics of open and closed circuit grinding system and observed that for the same product size the closed circuit system had a higher throughput. From Waterval the throughput was higher in closed circuit, and the particle size distribution of the product was close for both open and closed circuit configurations. This shows that the closed circuit configuration was grinding more efficiently than the open circuit configuration.

In both the Amandelbult and the Waterval campaigns, it appears that the 2-stage closed circuit configurations were not meeting the objectives of the regrind circuit due to the inappropriate cyclones. Data supporting this statement is given in Table 5-4 and Table 5-5 which show the characteristics of the streams around the cyclones. At Amandelbult the cyclone underflows solid concentrations were lower than in Waterval concentrator. For the closed circuit configurations, the solids concentrations of the 1st stage cyclone underflow were 62% and 66% for Amandelbult and Waterval respectively and for the 2nd stage cyclone underflow, they were 57% and 66%. For optimal mill operation this should be 75% by weight. A low solids concentration in the underflow is an indication that a significant amount of water is reporting to the underflow and carries with it a high amount of bypassing fine particles.

Slurries with high fines content and medium solids concentration, tertiary or regrind mill product, exhibit dilatant characteristics. For this type of slurry Shi and Napier-Munn (2002) observed that the Grinding Index - which is a measure of particle breakage inside a mill - was increased by raising the slurry density. This can be achieved by reducing the water addition, but since no water is added to the circuits, increasing the percent solids in the cyclone underflow could be used to increase the percent solids in the mill feed.

Additionally the presence of fines in the mill is reported to have a detrimental effect to mill performance (Shi and Napier-Munn, 2002). This could explain the poor performance of the closed circuit configuration in Amandelbult and a better performance in Waterval concentrator. The 2nd stage cyclone was re-circulating underflow streams with 39% and 32% -75μm for closed circuit at Amandelbult and Waterval, respectively. This indicates that a large portion of particles in those streams that should be in the float feed are being misclassified to the cyclone underflow.

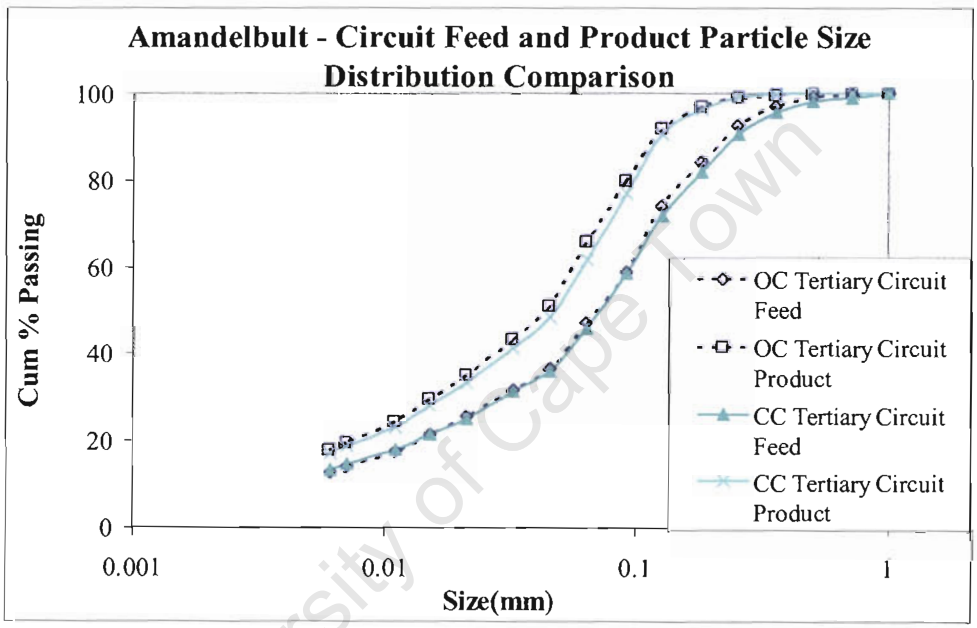


Figure 5-13 Amandelbult – circuit feed and product comparison of the particle size distribution

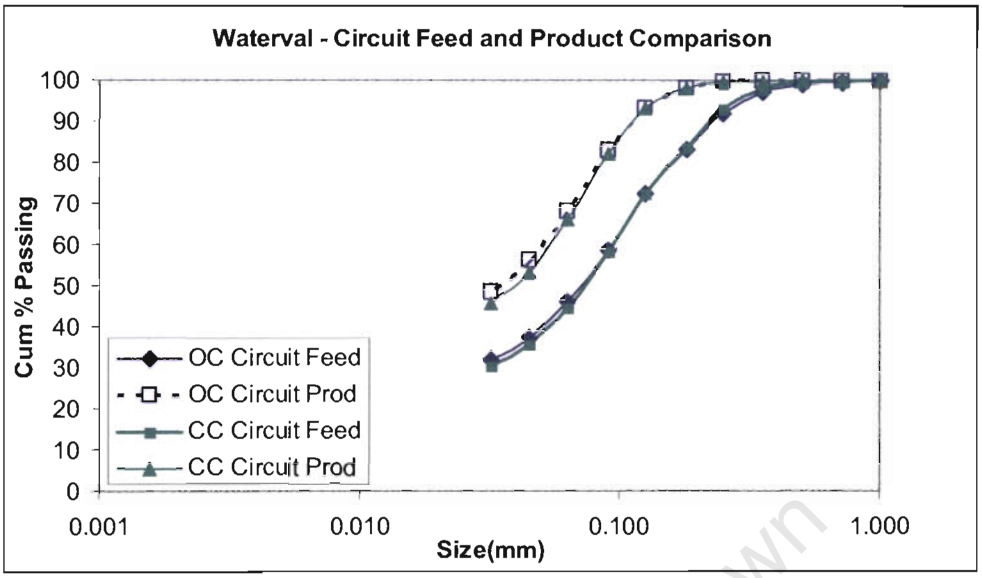


Figure 5-14 Waterval – circuit feed and product comparison of the particle size distribution

5.6 *Performance of the hydrocyclones*

The hydrocyclones were found to be performing poorly across all three concentrators investigated. Table 5-4 shows the properties of streams around the 1st stage cyclones across the concentrators tested. The reason it is claimed that the hydrocyclones were performing poorly is because the 1st stage cyclones underflows had excessively high values of % -100μm of 19, 34 and 27% at Lebowa (secondary circuit), Amandelbult (tertiary circuit) and Waterval (tertiary circuit) concentrators respectively. These streams return to the mills and result in over-grinding.

Similar results were observed for the 2nd stage hydrocyclones (Table 5-5) where a high percentage %-100μm material reported to the cyclone underflow streams. The values of about 30% -10μm in the 2nd stage cyclone overflow are characteristic of over-ground particles. The slurry densities are too low for optimal mill operation as discussed

The diagnosis for the poor performance of the closed circuit configuration has been identified as the poor design of the classification circuits. The hydrocyclones employed are too large for the operation. Additionally, smaller cyclones operating at a higher pressure are favourable for a fine separation. The plants were not able to replace the hydrocyclones in the short term and as a result optimal operation could not be tested. Simulations were conducted to predict the outcome when smaller cyclones replaced the current ones, and this is presented in Chapter 8.

Table 5-4 1st stage hydrocyclones stream properties

		Lebowa		Amandelbult		Waterval	
		CC 1st stage cyclone	OC 1st stage cyclone	CC 1st stage cyclone	OC 1st stage cyclone	CC 1st stage cyclone	OC 1st stage cyclone
Feed	% passing 100um	37.2	35.0	66.0	64.3	59.4	65.6
	% passing 75um	30.2	25.2	54.7	53.1	48.1	54.5
	% passing 10um	9.2	5.1	18.2	16.9	17.9	21.5
	% solids	39.0	37.0	35.7	34.4	38.7	33.4
Overflow	% passing 100um	63.2	63.7	92.2	93.3	83.3	87.1
	% passing 75um	53.2	50.5	81.2	84.5	70.4	75.2
	% passing 10um	16.5	9.1	27.5	28.0	29.2	32.8
	% solids	30.9	25.4	27.5	24.2	29.7	26.4
Underflow	% passing 100um	12.6	18.5	29.6	33.5	25.6	26.6
	% passing 75um	8.5	10.7	18.3	20.6	16.6	17.1
	% passing 10um	2.3	25.4	5.0	5.1	4.4	4.4
	% solids	52.5	50.3	62.2	62.1	68.4	66.4

Table 5-5 2nd stage hydrocyclones stream properties

		Lebowa	Amandelbult	Waterval
		2nd Stage Cyclone	2nd Stage Cyclone	2nd Stage Cyclone
Feed	% passing 100um	58.0	80.6	78.5
	% passing 75um	48.1	67.5	65.2
	% passing 10um	15.3	21.3	25.5
	% solids	34.7	39.4	42.9
Overflow	% passing 100um	63.0	81.7	87.6
	% passing 75um	52.7	68.8	75.9
	% passing 10um	16.8	21.9	32.1
	% solids	33.0	38.8	38.7
Underflow	% passing 100um	12.6	57.7	49.6
	% passing 75um	8.5	38.8	31.5
	% passing 10um	2.3	9.5	7.6
	% solids	52.0	57.1	66.2

Figure 5-15 shows the corrected partition curves for the hydrocyclones at the Waterval Concentrator. The steeper the curve the higher the classification efficiency, an ideal classifier would have a vertical partition curve. The corrected partition curve is used to represent the true classification, as the fraction of fine particles that bypass classification and report to the underflow along with the water has been corrected for. The curve therefore spans 0 to 100% in terms of material reporting to the overflow (in this case).

It can be seen that the partition curves indicate that the cyclones were operating inefficiently. As an example, a 120μm particle has a 50 % chance of reporting to the overflow for the open circuit. For the closed circuit configuration 165μm particles had a 50 % chance of reporting to the overflow.

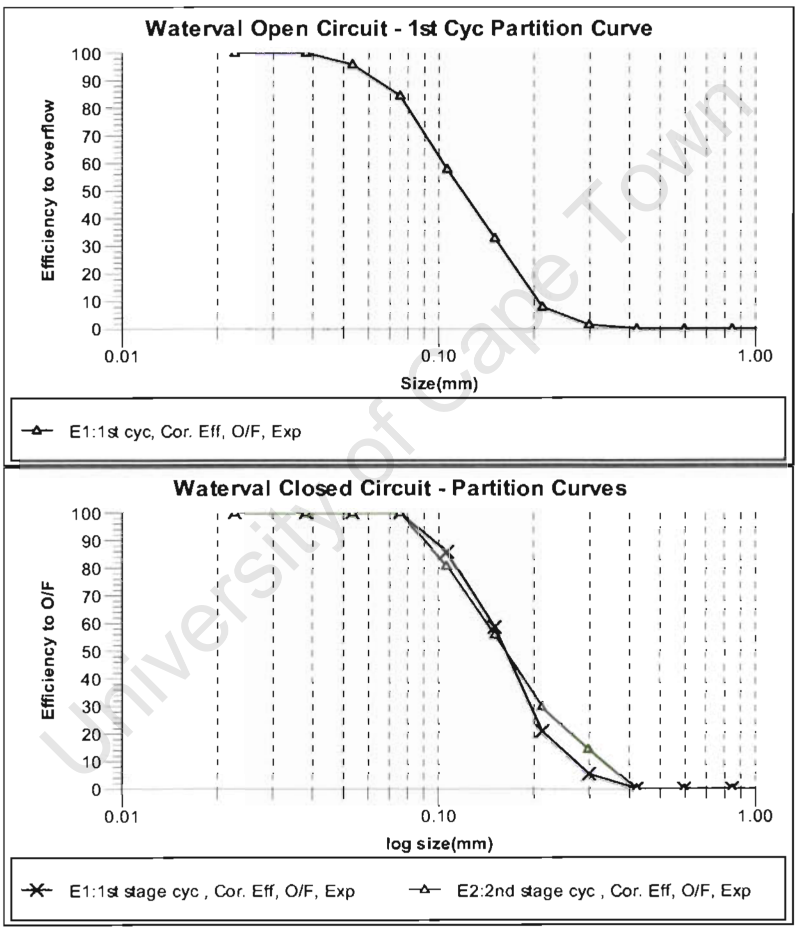


Figure 5-15 **Waterval hydrocyclones' partition curves – open and closed circuit**

Bearing in mind that the required size range for flotation is 10 to 100μm (Harris, 2000), it can be seen that the hydrocyclones were not meeting the classification objective with particles as big as 165μm having a 50 % chance of reporting to the overflow. The 2^{nd} stage hydrocyclone partition curve has a flatter gradient, indicating lower separation efficiency. This is the hydrocyclone that determines what leaves the circuit and what is retained in the circuit for further grinding.

The classification efficiency can be improved by limiting the proportion of water reporting to the underflow. Napier-Munn *et al* (1999) noted that there is some evidence that increased pressure can improve the classification efficiency; however, the effect is not strong.

It is not unusual to find that the initial cyclone selected for a particular operation turns out to be inappropriate, sometimes due to conditions changing. Napier-Munn *et al*, 1999 observed that it is a common error to operate wrong diameter cyclones especially in fine grinding.

It would have been ideal if this argument could be tested by optimising 2nd stage hydrocyclones at the Waterval concentrator, this was to be achieved by using a smaller spigot (smaller than the current 110mm). The hydrocyclones were too big to fit the spigot with a diameter of less than 110mm. The manufacturer does not make spigots with an internal diameter of less than 110mm for the same class of spigots employed for this particular hydrocyclones. An extra conical section has to be added to the cyclone to reduce the current outlet to the required diameter. A possible solution is to raise the whole nest of cyclones which is expensive. Alternatively, the concentrator could install smaller hydrocyclones. Unfortunately, it was not been feasible to implement either of the solutions within the time frame of this study.

5.7 Conclusions

Comparative test work was conducted at both Amandelbult and Waterval concentrators looking at open and 2-stage closed circuit configurations. From Amandelbult concentrator, the closed circuit configuration did not show a benefit over the open circuit configuration in terms of size reduction across the circuit. However, at Waterval concentrators the closed circuit configuration showed a higher grinding efficiency over the open circuit configuration. This is because the throughput was higher during the 2-stage closed circuit configuration test. This is attributed to the better (higher) underflow densities being achieved in this concentrator.

However, it is believed that these closed circuit configurations can be improved by improving the sharpness of classification; this will lead to less over-grinding and the production of floatable size fractions will increase. This would be primarily because particles that are ready for flotation will be released from the circuit without being over-ground, but also because less oversize particles will be carried over in the cyclone overflows.

The closed circuit configuration mills were observed to be less efficient than the open circuit mills. This is attributed to the circulation of fine particles due to the inefficiencies in the 2nd stage cyclones.

The partition curves indicated that the hydrocyclones were not suited for the operation. The cut sizes are too high for a regrind application. This suggests that the employed cyclones are too big for the operation since the cut-size is determined by the cyclone diameter.

Chapter 6

OUTCOMES FROM THE STUDY OF SINGLE- AND TWO-STAGE CLOSED CIRCUIT CONFIGURATIONS

6.1 Introduction

This chapter focuses on the comparison in performance between 1-stage and 2-stage closed circuit configurations. The flowsheets for the circuit configurations are shown in chapter 4. This investigation was conducted at the Waterval concentrator, where the design of the tertiary circuit allows for the operation of open, 1-stage closed and 2-stage closed circuit configurations.

The apparent advantage of the 1-stage closed circuit over the 2-stage closed circuit configuration is that the total fines that bypass classification and report to the underflow is reduced, because this inefficiency will be restricted to one stage of hydrocyclones.

As discussed in Chapter 5, the results were mass balanced to assess the data integrity. The comparison is based on the degree of particle size reduction across the circuits. Because the effectiveness of a closed circuit depends largely on the performance of the hydrocyclones, the performance of the hydrocyclones was assessed by means of the partition curves.

6.2 Streams Properties

Figure 6-1 and Figure 6-2 show the particle size distributions for the 1- and 2 –stage closed circuits, respectively. There is a strong agreement between the experimental (markers) and the mass balanced (lines) data for all streams for both the 1- and 2-stage closed circuit configurations.

The 1st stage hydrocyclone underflow has solids concentration as low as 53% (Table 6-1) for the 2-stage closed circuit configuration. This is an indication of poor classification; there is a significant amount of water carrying fine particles that bypass true classification. In addition to this, low percent solids are detrimental for mill operation, where a number in the order of 75%, is thought to be ideal by mineral processing norms (Powell, 2004).

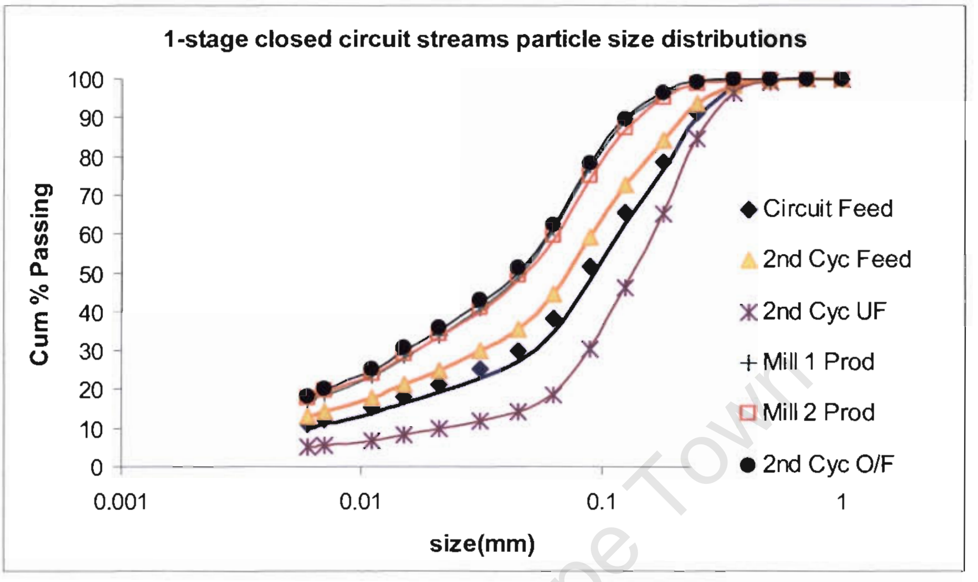


Figure 6-1 Waterval 1 stage closed circuit particles size distributions

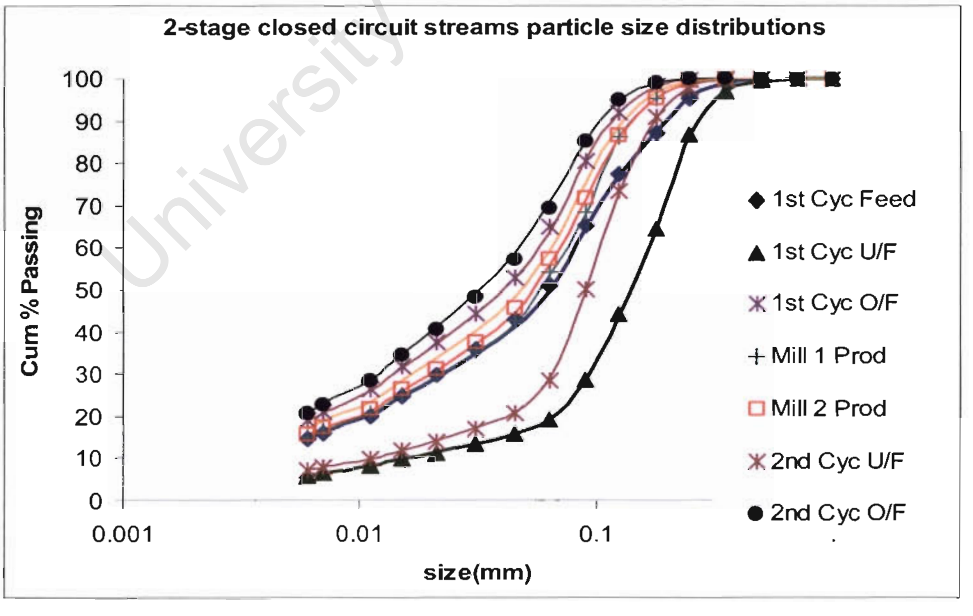


Figure 6-2 Waterval 2 stage closed circuit particles size distributions

Table 6-1 Waterval Streams' properties – 1- and 2-stage closed circuit

	TPH Solids				% Solids			
	2-stage closed circuit		1-stage closed circuit		2-stage closed circuit		1-stage closed circuit	
	Exp	Bal	Exp	Bal	Exp	Bal	Exp	Bal
Circuit Feed	342	342	353	353	35.7	36.1	39.8	39.7
1st Cyc U/F	-	110	-	-	54.8	52.1	-	-
1st Cyc O/F	-	232	-	-	31.6	31.5	-	-
Mill 1 Feed	-	82	-	91	-	55.9	-	68.1
Mill 1 Prod	82	82	91	91	55.9	55.9	68.6	68.1
Mill 2 Feed		128	-	157	-	56.2	-	67.9
Mill 2 Prod	128	128	157	157	56.2	56.2	68.0	67.9
2nd Cyc Feed	-	442	-	601	39.8	39.8	48.1	47.9
2nd Cyc U/F	-	101	-	248	61.8	61.2	67.7	68.0
Circuit Product/2nd Cyc O/F	-	342	-	353	36.3	36.1	39.5	39.7
	P80 (mm)				% - 75μm			
	2-stage closed circuit		1-stage closed circuit		2-stage closed circuit		1-stage closed circuit	
	Exp	Bal	Exp	Bal	Exp	Bal	Exp	Bal
Circuit Feed	0.138	0.139	0.197	0.199	57.3	57.1	39.6	39.2
1st Cyc U/F	0.223	0.223	-	-	23.7	23.7	-	-
1st Cyc O/F	0.089	0.088	-	-	73.4	73.4	-	
Mill 1 Feed	-	0.188	-	0.229	-	30.1	-	24.2
Mill 1 Prod	0.110	0.110	0.095	0.095	61.5	61.5	70.4	70.3
Mill 2 Feed	-	0.198	-	0.228	-	31.1	-	24.3
Mill 2 Prod	0.107	0.107	0.101	0.102	64.7	64.7	68.0	68.0
2nd Cyc Feed	0.101	0.098	0.159	0.159	66.8	68.6	50.9	51.4
2nd Cyc U/F	0.141	0.143	0.229	0.229	39.1	38.5	24.2	24.3
Circuit Product/2nd Cyc O/F	0.079	0.081	0.094	0.094	78.2	77.5	71.0	70.7

6.3 Comparison of Mills' Performances between 1- and 2-stage closed circuit configurations

The mills' performances between 1- and 2-stage closed circuit configurations are considered in isolation in this section. Figure 6-3 shows the size distributions for the 1- and 2-stage closed circuit mills feed and product. It should be noted that the 1-stage closed circuit uses the cluster of cyclones that is used as the 2nd stage cyclones in the 2-stage closed circuit configuration. For discussion purposes this will be referred to as the 2nd stage cyclones even in the 1-stage closed circuit. By this convention, the 1-stage closed circuit has the "2nd stage cyclone" underflow as a feed to the mills. This configuration bypasses the 1st stage cyclones. The 2-stage closed circuit mills are fed by the 1st stage and 2nd stage cyclone underflows.

The two parallel mills were receiving a coarser feed in 1-stage closed circuit (Figure 6-3) than in 2-stage closed circuit. At the same time, the mill product was finer in 1-stage closed circuit than in 2-stage closed circuit. This indicates that the mills were grinding more efficient in 1-stage closed circuit than in 2-stage closed circuit. Actual mill feed rate is discounted as a possible cause of this observation, as the feedrate during the 2- stage closed circuit was 210tph, as opposed to 248tph for 1-stage closed circuit. Similarly the circulating loads were eliminated as a possible because these were 70 and 67% for the 2-stage and 1-stage closed circuit tests respectively (Appendix E).

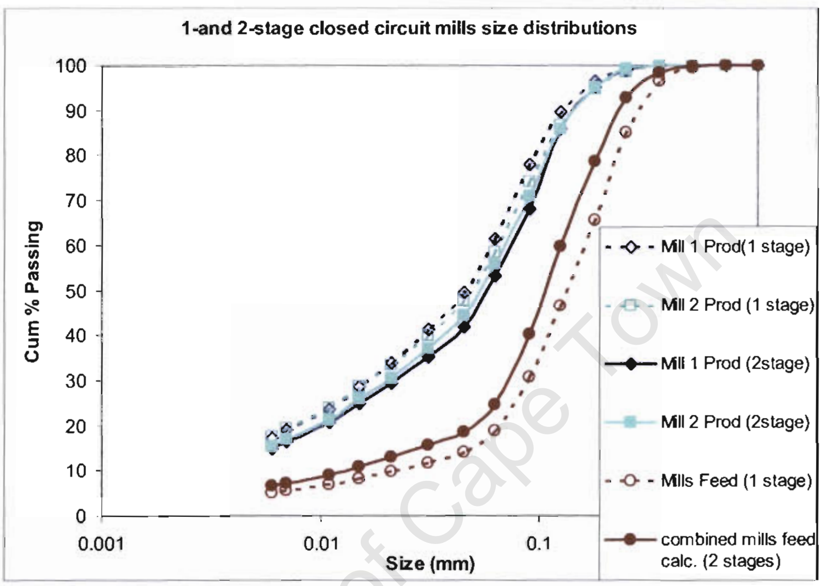


Figure 6-3 Comparison of 1-and 2-stage closed circuit mills' product size distributions

As mentioned above, in the 1-stage closed circuit the inefficiency due to fines being misplaced into cyclone underflows is limited to one stage of cyclones. This is supported by the %-75µm in the mills' feed in the two configurations. The %-75µm were about 30% in 2-stage and 24% in 1-stage closed circuit configuration (refer to Table 6-1). The presence of the fine particles in the mills could be hindering the mill performance as observed by Shi and Napier-Munn (2002).

The most likely explanation for the drop in mill performance in the 2-stage closed circuit may be the low percent solids of the mill feed, which was about 56% compared with 68% for the 1-stage closed circuit. Shi and Napier-Munn (2002) also reported that the Grinding Index for this kind of application can be increased by raising the slurry density. Since the mills' feed in the 1-stage closed circuit configuration had a higher slurry density it can be concluded that is a strong contributory factor in the observed better mills performance.

6.4 The comparison in performance between the 1– and 2–stage closed circuit configurations

A comparison between the 1-stage and 2-stage closed circuit is illustrated in Figure 6-4. The feed was dramatically coarser for the 1-stage test, with F80 of 199μm compared with 139μm in 2-stage closed circuit. It was unfortunate that the feed was not the same for these two consecutive tests. It is possible that an external stream (the Frank circuit tailings stream) was switched on between the tests without the knowledge of the surveying team. Precaution had been taken to have this stream switched off for the test period, but a delay in the test had resulted in the 1-stage closed circuit test being conducted 2 hours later than planned.

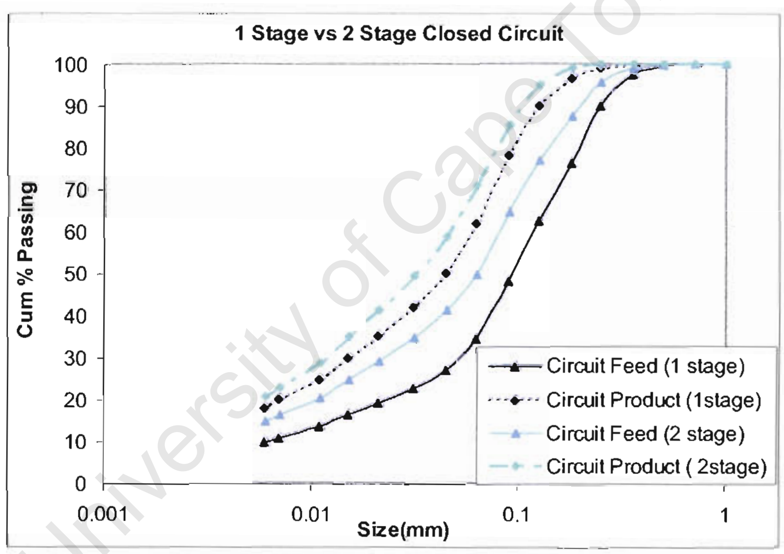


Figure 6-4 Comparison between 1-stage and 2-stage closed circuit size distributions

Due to the differences in the feed size distribution between the two circuit configurations it was realised that graphical comparison may not be the most appropriate method of quantifying the differences in milling performance. The Bond equation (Equation 6-1) was used as a method to clarify the relative milling performance with the two different feed characteristics. The Bond equation is centred on two assumptions, the first being that energy in grinding is dominated by the production of new surface area and the feed type. Secondly, it is assumed that the feed and product can be represented by a single size, provided the distributions being compared are approximately parallel when plotted as cumulative passing vs. log size. By calculating the operating work index (WIo), the grinding efficiencies of the circuits were compared.

Equation 6-1

$$WIo = \frac{E}{10}\left(\frac{\left(\sqrt{P80}\right)\left(\sqrt{F80}\right)}{\sqrt{F80}-\sqrt{P80}}\right)\text{, where}$$

E = Specific energy, *kWh/t*

F80= 80 % of material in the feed passing size, *μm*

P80 = 80 % of material in the product passing size, *μm*

WIo = Operating Work Index, *kWh/t*

Table 6-2 Operating work index-Waterval

	1 stage Closed Circuit Configuration	2 stage Closed Circuit Configuration
Power Draw, kW	3320	3346
FeedRate, tph	353	303
Specific Energy, E,kWh/t	9.4	11.0
P80 ,μm	94	78
F80 ,μm	193	138
WIo	30.2	39.3

The low calculated work index of 30.2 kWh/t shown in Table 6-2 for the 1-stage closed circuit indicates that this configuration has a higher grinding efficiency than the 2-stage closed circuit configuration (which had a work index of 39.3 kWh/t). It can also be observed from Figure 6-4 that the change from the circuit feed to the circuit product is greater in 1-stage closed circuit configuration.

It has been reported by Morrell (2004) that the basic assumption of the Bond relationship is flawed. Based on testing a wide range of data, he has concluded that the *WIo* always increases as the product becomes finer, even for the same degree of reduction. This may partially explain the increase in *WIo* in 2-stage closed circuit, but not to the extent that is calculated.

The higher throughput during the 1-stage closed circuit test has also contributed to the lower *WIo* and it is also possible that the new stream (the Frank tailings stream) had a lower *WI*, but as this constitutes less than 15% of the feed, it can also not account for the considerable drop in *WIo*. Based on the above analysis, it can be concluded that the 1-stage closed circuit is more efficient than the 2-stage closed circuit in terms of energy utilisation for size reduction.

6.5 Hydrocyclone Performance

The hydrocyclones performance was assessed by the partition curves. Figure 6-5 shows the experimental and balanced corrected partition curves for the 1-stage closed circuit and 2-stage closed circuit configurations.

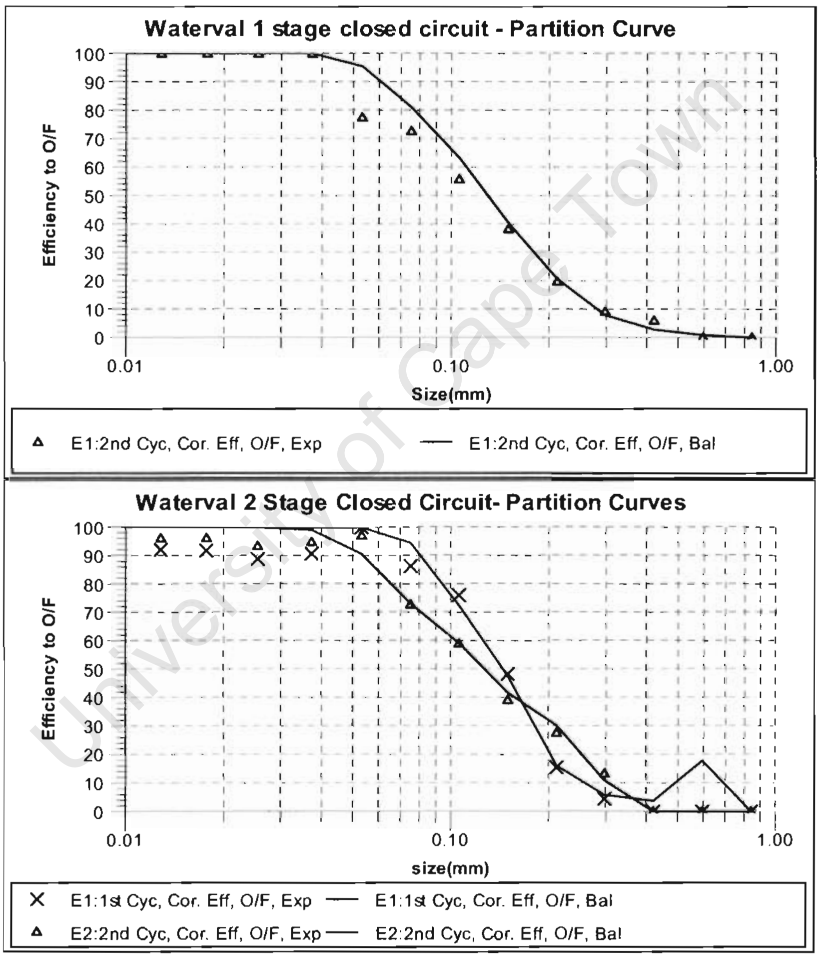


Figure 6-5 Waterval hydrocyclones' partition curves -1-and 2-stage closed circuit

The cut size was the same between the two configurations for the 2nd stage cyclone at 130μm (Table 6-3) which approximated from the graphs in Figure 6-5. The slope of the 2nd stage hydrocyclone partition curve has a lower gradient which indicates poor classification efficiency. As in chapter 5, the 1st stage hydrocyclone had better classification efficiency than the 2nd stage hydrocyclone.

A fish hook is observed in the 1st stage cyclone efficiency curve. The fish hook has been reported by a number of authors (Heiskanen, 1993 and Napier-Munn *et al*, 1999) and an explanation for the fish hook and other unusual efficiency curve shapes is given as that it could be as a result of sampling errors and size analysis. However, improved size analysis techniques at very fine sizes (laser sizing) have demonstrated that this is a real effect. It could also be as a result of the entrainment of particles in water, the entrainment increasing with the decrease in particle size.

Table 6-3 Estimated cut-sizes – Waterval 1-stage and 2-stage closed circuit

Waterval	1-stage closed circuit	2-stage Closed Circuit	
	2nd stage cyclone	1st stage cyclone	2nd stage cyclone
d_{50c}	130	143	130

6.6 Conclusions

Operating work indices were calculated for the 1-stage and 2-stage closed circuit configurations and the result showed that the 1-stage closed circuit had a higher grinding efficiency than the 2-stage closed circuit configuration. The 2nd stage cyclone was observed to have a low separation efficiency, which means that both circuits could be improved by improving the separation efficiency. Hukki (1976 and 1977) showed that by improving the sharpness of classification the circuit improves drastically (refer to sections 2.3 and 2.4).

Chapter 7

SIMULATIONS OF POSSIBLE IMPROVEMENTS TO THE REGRIND CIRCUITS

7.1 Introduction

The experimental results were mass balanced, and model fitted, where mathematical models, available in JKSimMet, were fitted to the plant data. From the fitted results, simulations were conducted in which the existing cyclones were replaced with smaller cyclones, as a way of estimating the positive influence that more appropriately sized cyclones would have on the circuit's performance. This chapter briefly discusses the models that JKSimMet uses. Simulations were conducted on the Waterval 1-stage and 2-stage closed circuit configuration using the experimental data reported in Chapter 6.

JKSimMet is a steady state simulation program based on many years of industrial research by the Julius Kruttschnitt Mineral Research Centre (JKMRC) (Napier-Munn *et al*, 1999). JKSimMet contains robust models of comminution and classification processes found in mineral processing plants. Once the models are fitted to the plant data, the package can be used to predict the results of changing design paramters, operating parameters and in some cases circuit configuration.

There are various mathematical models available in the package for the various comminution units. For this thesis, only the ball mill and the hydrocyclone models were used. These two models will be briefly discussed in Appendix C; a detailed study of the models is outside the scope of the thesis. The results from the simulations are discussed in this chapter.

The ball mill and cyclone models used were developed from real plant data collected over a long period. However, like all empirical models they work well within the conditions under which they were established. The Nageswararao model (cyclone model) for instance was developed by predominantly using limestone and Krebs geometry cyclones with diameters of 102-381mm. It is reported that the model is quite robust for cyclones with diameters up to 760mm (Napier-Munn *et al*, 1999). Therefore care was taken in this thesis to simulate conditions that are within the defined range of applicability for this model.

7.2 Model Fitting

Table 7-1 and Table 7-2 compare the fitted with the experimental results for the 1-stage and 2-stage closed circuit configurations from Waterval concentrator. The closeness of the fitted values to the experimental values shows that the mathematical models, described in Appendix C, have successfully fitted the experimental data. The particle size distributions are compared in Appendix F and the results show a good fit.

As with mass balancing, the test for a good fit is when the difference between the experimental and fitted results is within one percent for percent solids, five percent for flowrates and the particle size distributions are on top of each other (Powell, 2004). Difficulty was experienced in accurately fitting the 2-stage circuit, partially arising from the enforced assumption that the feed to the two mills was identical. Although the absolute values did not match very closely-the final product is within 1.2% which is reasonable for such a circuit, the model fit will still give realistic outcomes of reasonable changes to the circuit. The quality of the 1-stage circuit fit is good. Now that the mathematical models have been successfully fitted to the experimental data, simulations of circuit changes can proceed with confidence.

Table 7-1 Waterval 2-stage closed circuit – Exp. and Fitted Stream Values

	TPH Solids			% Solids			P80 (mm)			%-75mm		
Stream	Exp	Bal	Fit	Exp	Bal	Fit	Exp	Bal	Fit	Exp	Bal	Fit
Circuit Feed	-	342	342	35.7	36.1	35.7	0.138	0.139	0.138	57.3	57.1	57.3
1st Cyc U/F	110	110	109	54.8	52.1	56.4	0.223	0.223	0.226	23.7	23.7	27.3
1st Cyc O/F	-	232	233	31.6	31.5	30.4	0.089	0.088	0.091	73.4	73.4	71.9
Mill 1 Feed	-	82	128	-	55.9	59.4	-	0.194	0.192	0.0	30.1	32.8
Mill 2 Feed	82	82	82	55.9	55.9	58.4	0.110	0.110	0.192	61.5	61.5	32.8
Mill 1 Prod	-	128	82	-	56.2	58.4	-	0.188	0.111	0.0	31.1	62.4
Mill 2 Prod	128	128	128	56.2	56.2	59.4	0.107	0.107	0.107	64.7	64.7	64.7
2nd Cyc Feed	-	442	443	39.8	39.8	39.5	0.101	0.098	0.099	66.8	68.6	68.0
2nd Cyc U/F	-	101	101	61.8	61.2	62.0	0.141	0.143	0.147	39.1	38.5	38.8
2nd Cyc O/F	-	342	342	36.3	36.1	35.7	0.079	0.081	0.082	78.2	77.5	76.8

Table 7-2 Waterval 1-stage closed circuit – Experimental, Balanced and Fitted Stream Values

Port	TPH Solids			% Solids			P80 (mm)			%-75mm		
Stream	Exp	Bal	Fit	Exp	Bal	Fit	Exp	Bal	Fit	Exp	Bal	Fit
Circuit Feed	353	353	353	39.8	39.7	39.8	0.197	0.199	0.197	39.6	39.2	39.6
2nd Cyc Feed	-	601	612	48.1	47.9	48.6	0.159	0.159	0.157	50.9	51.4	51.9
2nd Cyc U/F	-	248	259	67.7	68.0	69.6	0.229	0.229	0.226	24.2	24.3	27.2
2nd Cyc O/F	353	353	353	39.5	39.7	39.8	0.094	0.094	0.095	71.0	70.7	70.4
Mill 1 Feed	-	91	95	-	68.1	69.6	-	0.229	0.226	-	24.2	27.2
Mill 1 Prod	91	91	95	68.6	68.1	69.6	0.095	0.095	0.095	70.4	70.3	70.4
Mill 2 Feed	-	157	164	-	67.9	69.6	-	0.228	0.226	-	24.3	27.2
Mill 2 Prod	157	157	164	68.0	67.9	69.6	0.101	0.102	0.105	68.0	68.0	68.1

It was observed in Chapter 6 that the mills were grinding finer in 1-stage closed circuit than in 2-stage closed circuit. The breakage rates of the mills are plotted in Figure 7-1. Mill 2 in the 1-stage configuration appears to have higher breakage rates than all the other mills' regardless of the configuration.

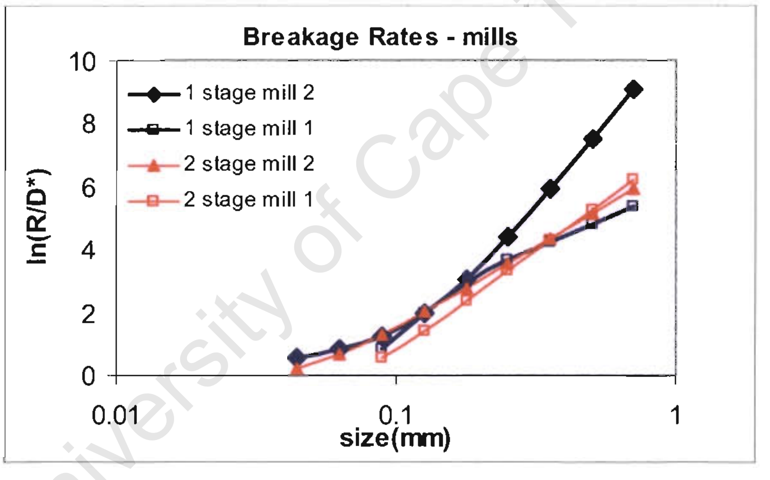


Figure 7-1 Breakage rates of mills in 1-and 2-stage closed circuits

It was discussed in section 6.3 that the mills were grinding more efficiently in 1-stage closed circuit than in 2-stage closed circuit. Figure 7-1 shows that mill 2 (1-stage circuit) had higher breakage rates than mill 1(1-stage circuit) although in Figure 6-3 mill 1 and 2 product size distributions were similar. This is attributed to the fact that mill 2 had a significantly high throughput than mill 1.

However, it appears that the combined breakage rates were higher for the 1-stage closed circuit than the combined 2-stage closed circuit configurations. This is thought to be attributable to the higher percent solids in the mills' feed for the 1-stage closed circuit configuration as discussed in section 6.3. The mills were not heavily loaded since the breakage rates did not indicate any curvature at coarser sizes as in Figure C-1.

7.3 Simulation Outcomes

This section discusses the outcomes from simulating the 1-and 2-stage closed circuit configurations, as a way of exploring potential improvements to the operation of the circuits. It was observed that the cyclones being used in the concentrators that were studied in this thesis were inappropriate for the regrind section. As a result simulations were conducted to predict the circuit performance by using more appropriate smaller cyclones. The simulator being used has been used with reliable predictions (Napier-Munn et al, 1999).

7.1.1 1-Stage Closed Circuit Configuration

As discussed in Chapter 2, the cyclone efficiency strongly drives the performance of a closed circuit configuration. Smaller cyclones from a different concentrator in a similar application (i.e. also classifying Merensky ore in a regrind application) were used in the simulations to replace the exiting cyclones. The dimensions of the existing and the simulated cyclones are shown in Table 7-3. During the simulation, the pressure of the replacement cyclones were kept as close to the measured pressure as possible, by increasing or decreasing the number of units "on-line".

Fernandes and Peres (1999) reported an increase of 16.5% in apatite flotation recovery at Arafertil by refurbishing the regrind circuit. Among the changes made to the circuit was the replacement of the 10 inch cyclone with 2 inch ones because the regrind product was finer.

7.1.1.1 Circuit Improvement

Figure 7-2 shows the size distribution of the circuit product as is, and of the circuit product as predicted with the simulated cyclones. Table 7-4 shows the stream data of the circuit by replacing the existing two 750mm diameter cyclones by nine 400mm diameter cyclones. Simulations show a marked improvement in circuit performance by using the 400mm cyclones in the 1-stage closed circuit. The circuit product has improved from having 71% to 78% -75μm. The more effective classification system can now remove more fine particles from the circuit.

In the concentrator where the 400mm cyclones are currently used, three of them are used in parallel at a measured pressure of 70kPa. To maintain this pressure in the proposed application, nine of these smaller cyclones will have to be used to ensure that the predictions are within operating conditions of the cyclones (conditions should be close to the conditions of the current application). Table 7-5 shows the current performance data of the 2nd stage cyclone at Waterval concentrator, performance data of the proposed cyclone in the current application, and the predicted performance of the simulated cyclones in the Waterval concentrator.

Table 7-3 Existing and simulated cyclones model parameters and cyclone parameters

Parameter	**Waterval - Existing 1st Stage Cyclones**	**Waterval - Existing 2nd Stage Cyclones**	**New Cyclones in Original Application**
Cyclone Diameter - Dc (m)	0.640	0.750	0.400
Inlet Diameter - Di (m)	0.290	0.300	0.110
Vortex Finder Diameter - Do (m)	0.240	0.256	0.160
Spigot Diameter - Du (m)	0.110	0.110	0.060
Cylinder Length - Lc (m)	0.345	0.600	0.450
Cone Angle - Theta (deg)	20	15	9.8
Model Parameters			
Parameter			
D50 Constant - KD0	2.41E-04	1.02E-04	8.94E-05
Capacity Constant - KQ0	282.1	512.4	317.7
Volume Split Constant - KV1	5.874	8.161	8.451
Water Split Constant - KW1	16.64	17.67	24.13
Sharpness of Efficiency Curve - Alpha	4.861	2.941	7.363
Initial Dip in Efficiency Curve - Beta	0	0	0
Calculated Value - Beta*	1	1	1

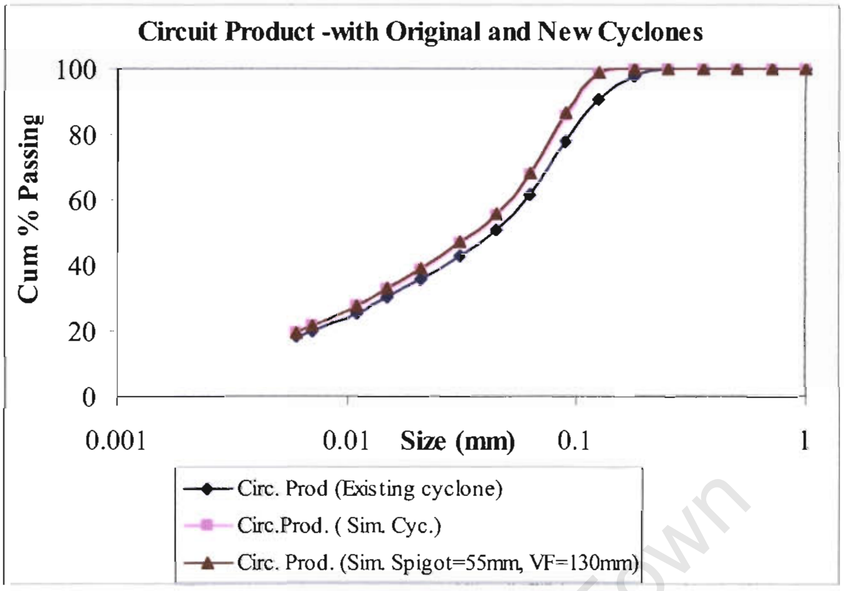


Figure 7-2 Simulated cyclones and Existing size distribution for the circuit product

Table 7-4 Waterval 1-stage closed circuit - Exp. and Simulated Stream Values (Simulated Cyclones)

	TPH Solids		% Solids		P80 (mm)		%-75μm	
Stream	Fit	Sim	Fit	Sim	Fit	Sim	Fit	Sim
Circuit Feed	353	353	39.8	39.8	0.197	0.197	39.6	39.6
2nd Cyc Feed	612	650	48.6	50.1	0.157	0.156	51.9	51.2
2nd Cyc U/F	259	297	69.6	72.5	0.226	0.220	27.2	20.7
2nd Cyc O/F	353	353	39.8	39.8	0.095	0.080	70.4	77.9
Mill 1 Feed	95	109	69.6	72.5	0.226	0.220	27.2	20.7
Mill 1 Prod	95	109	69.6	72.5	0.095	0.104	70.4	67.1
Mill 2 Feed	164	188	69.6	72.5	0.226	0.220	27.2	20.7
Mill 2 Prod	164	188	69.6	72.5	0.105	0.118	68.1	63.7

The cyclone underflow percent solids improve from 68% to 73% (Table 7-4). This will improve milling and reduce the amount of fines bypassing classification carried by the water to the underflow. Although the %-75μm in the circuit product has increased by about 7%, the %-75μm of the mills product has reduced by about 3%. This is attributable to the more efficient cyclone delivering a coarser feed to the mills. It should be noted that there is still room for more improvement in the 1-stage closed circuit because the cyclone underflows percent solids are still low compared to the norm values of at least 75%.

Table 7-5 Performance data for the existing and predicted cyclones

Performance Data	**Waterval - Existing 2nd Stage Cyclones**	**New Cyclones in Original Application**	**New 2nd stage Cyclones in Waterval Conc.**	**New 2nd stage Cyclones in Waterval Conc. (SP=55mm; V.F.=135mm)**
Number of parallel cyclones	2	3	9	10
Water Split To O/F (%)	82.49	85.01	82.61	82.99
Corrected D50, mm (Total)	0.137	0.0465	0.114	0.11
Operating Pressure, kPa	59.96	70.06	76.6	78.48
SP=spigot diameter, V.F.= vortex finder diameter				

7.1.1.2 Classification Improvement

Classification improvement is shown by means of partition curves in Figure 7-3. The slope of the curve has become steeper approaching the ideal vertical limit. The cut-size has been reduced as expected since smaller cyclones are employed. The curve shows that coarser particles have a reduced chance to report in the overflow. In the tested situation a 200μm particle had more than a 20% chance of reporting to the overflow; however, with the simulated cyclones this chance has reduced to close to almost zero.

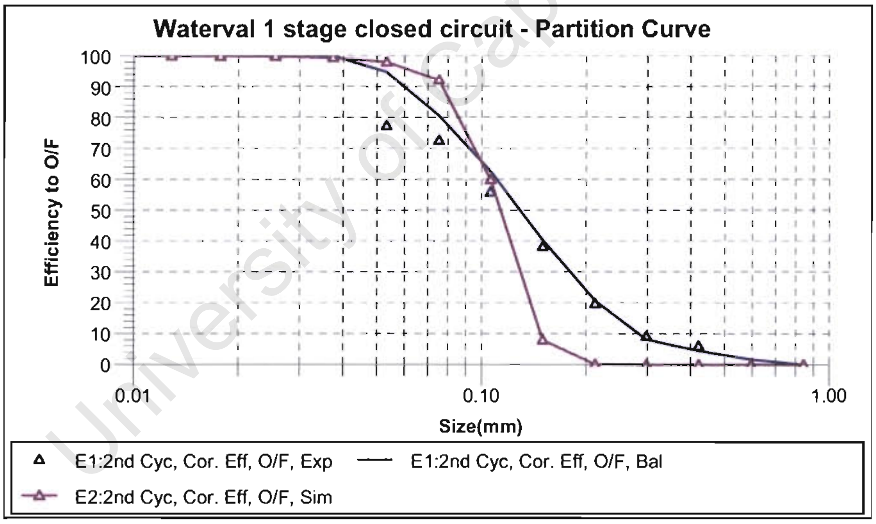


Figure 7-3 Waterval 1-stage closed circuit partition curve with simulated cyclones

7.1.1.3 Further Optimisation of the 1-Stage Closed Circuit

A conservative approach was followed in optimising the cyclones. A rule of thumb calculation for the spigot and vortex finder diameters was used, that the spigot diameter should be about a seventh of the cyclone diameter and the vortex finder diameter should be about a third of the cyclone diameter (Mainza, 2004). The improvements are shown in Table 7-6. The circuit product is predicted to be finer, with

79% -75µm (Table 7-6) compared to 77.9% (Table 7-4) for the simulated cyclones without the tuned apertures.

Table 7-6 1-stage closed circuit stream data – simulated cyclone with optimized cyclones

	TPH Solids		% Solids		P80 (mm)		%-75mm	
Stream	Fit	Sim	Fit	Sim	Fit	Sim	Fit	Sim
Circuit Feed	353	353	39.8	39.8	0.197	0.197	39.6	39.6
2nd Cyc Feed	612	658	48.6	50.5	0.157	0.155	51.9	51.1
2nd Cyc U/F	259	305	69.6	73.6	0.226	0.218	27.2	20.1
2nd Cyc O/F	353	353	39.8	39.8	0.095	0.078	70.4	79.0
Mill 1 Feed	95	112	69.6	73.6	0.226	0.218	27.2	20.1
Mill 1 Prod	95	112	69.6	73.6	0.095	0.105	70.4	66.5
Mill 2 Feed	164	193	69.6	73.6	0.226	0.218	27.2	20.1
Mill 2 Prod	164	193	69.6	73.6	0.105	0.117	68.1	63.2
Vortex Finder =135mm, Spigot = 55mm, Number of Cyc.=10, Pressure = 78kPa								

The cyclone underflow percent solids went from 72.5% (simulated cyclones) to 73.6% (simulated and optimised) by bringing down the spigot and vortex finder diameters to 55mm and 135mm. This exercise increased the pressure required, so by adding another cyclone to total ten, the pressure was at 78kPa which is much higher than the required pressure. By changing the apertures' diameters there was some gain, the %-75µm of the circuit product and the percent solids in the cyclone underflow have improved by about 1 %.

To bring down the pressure, eleven cyclones were used, and bringing down the vortex finder diameter to 130mm which is still close to a ninth of a cyclone diameter. The spigot diameter was kept at 55 mm. The outcomes of the simulations are show in Table 7-7. The pressure was brought down to 71kPa and the %-75µm stayed at 79%.

Table 7-7 1-stage closed circuit stream data – simulated cyclone with optimized apertures

	TPH Solids		% Solids		P80 (mm)		%-75µm	
Stream	Fit	Sim	Fit	Sim	Fit	Sim	Fit	Sim
Circuit Feed	353	353	39.8	39.8	0.197	0.197	39.6	39.6
2nd Cyc Feed	612	673	48.6	50.5	0.157	0.155	51.9	50.9
2nd Cyc U/F	259	320	69.6	72.1	0.226	0.215	27.2	21.5
2nd Cyc O/F	353	353	39.8	39.8	0.095	0.078	70.4	78.7
Mill 1 Feed	95	118	69.6	72.1	0.226	0.215	27.2	21.5
Mill 1 Prod	95	118	69.6	72.1	0.095	0.106	70.4	65.6
Mill 2 Feed	164	202	69.6	72.1	0.226	0.215	27.2	21.5
Mill 2 Prod	164	202	69.6	72.1	0.105	0.119	68.1	62.2
Vortex Finder =130mm, Spigot = 55mm, Number of Cyc.=11, Pressure = 71.38kPa								

7.1.2 Simulation of the 2-Stage Closed Circuit Configuration

The 2-stage closed circuit configuration was simulated in two stages. The first step was to predict the circuit performance by using the 400mm cyclones in the 2nd stage cluster and retain the existing 1st stage cyclones. The second step was to use the 400mm cyclones in the 1st stage cluster as well and predict the circuit performance. This is discussed in the following sub-sections.

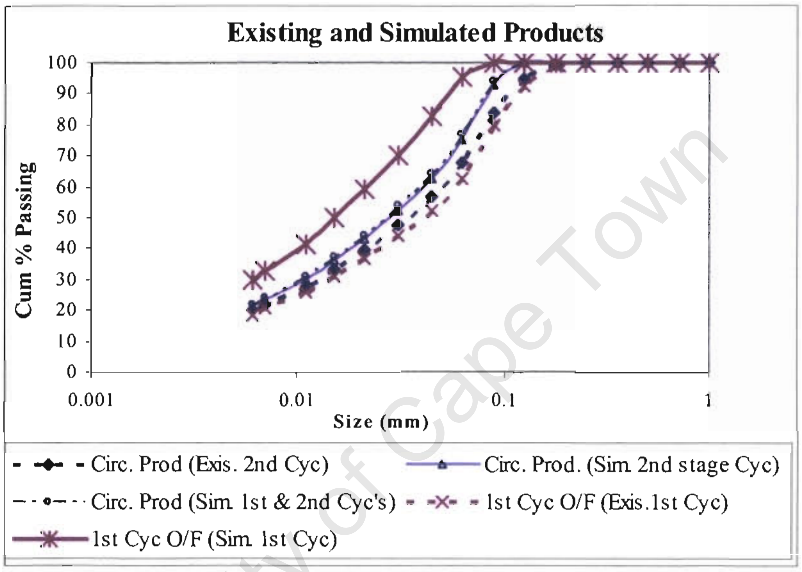


Figure 7-4 Existing and simulated 1st and 2nd stage cyclone products

7.1.2.1 Simulated 2nd stage cyclones

Table 7-8 shows the stream data of the simulation using the 400mm cyclones for the 2nd stage. The effect of improving the 2nd stage cyclone is observed by the improved 2nd cyclone overflow (Figure 7-4). The simulation predicts an improvement of the circuit product from 76.8% to 87.2% -75μm by simply changing the 2nd stage cyclones. This is in line with the findings by Hukki (1976) (refer to chapter 2) that by improving the sharpness of classification, more particles ready for flotation will be released from the circuit. In this case the sharpness of classification (alpha) was improved from 2.9 to 7.4, where a value of 10 would indicate an ideal cyclone.

Less over-grinding is observed, since the mills are receiving less fines at 26% -75μm compared with the original 33% -75μm. This resulted in mill products with 43.7% and

51.4% -75μm for mill 1 and 2 respectively, compared with 62.4 and 64.7% -75μm measured on the real circuit. Even though the predicted mills products are coarser than existing, the circuit product is much finer than the existing product. Kobayashi *et al* (2003) observed similar findings, that the median of the classifier overflow was increased as the classifier feed was increased. This is definitely attributed to a more effective fine particles removal system. This will reduce unnecessary grinding in the mills, hence save energy. It should be noted that there is still more room for improvement since the cyclone underflows did not reach the optimal percent solids of at least 75%.

Table 7-8 2-stage closed circuit stream data – simulated 2nd stage cyclone (spigot = 55mm, vortex finder =135mm) and existing 1st stage cyclones

	TPH Solids		% Solids		P80 (mm)		%-75μm	
Stream	Fit	Sim	Fit	Sim	Fit	Sim	Fit	Sim
Circuit Feed	342	342	35.7	35.7	0.138	0.138	57.3	57.3
1st Cyc U/F	109	147	56.4	55.8	0.226	0.208	27.3	28.3
1st Cyc O/F	233	195	30.4	28.0	0.0906	0.0762	71.9	80.2
Mill 1 Feed	128	223	59.4	64.9	0.192	0.160	32.8	26.0
Mill 2 Feed	82	223	58.4	64.9	0.192	0.160	32.8	26.0
Mill 1 Prod	82	223	58.4	64.9	0.111	0.125	62.4	43.7
Mill 2 Prod	128	223	59.4	64.9	0.107	0.117	64.7	51.4
2nd Cyc Feed	443	641	39.5	46.3	0.099	0.112	68.0	57.2
2nd Cyc U/F	101	299	62.0	70.5	0.147	0.140	38.8	24.8
2nd Cyc O/F	342	342	35.7	35.7	0.0819	0.0675	76.8	87.2

7.1.2.2 Simulated 1st and 2nd stage cyclone

Table 7-9 shows the stream data of the simulation with the 1st and 2nd stage cyclones replaced with the simulated cyclones. By also changing the 1st stage cyclones a further improvement is observed although, the improvement is not very striking. Kelsall (1974) did not observe striking improvement by re-classifying the cyclone overflow, but he was adamant that the improvement can be increased. Hukki (1976) noted that multiple cyclones do not give worthwhile improvements. A similar observation is made here, improving the sharpness of classification from 4.9 to 7.4 of the 1st stage cyclone only results in the 2nd cyclone overflow moving from 87.2% to 88.8% -75μm (Table 7-8 and Table 7-9)

It appears that the efficiency of 1st stage cyclone does not have a huge effect on the overall performance of the circuit. However, the amount of fines in the 1st stage cyclone overflow has increased significantly from 80.2% to 99.7% -75μm. This indicates the amount of fines that could have gone to the mills have been redirected to the overflow due to more efficient classification.

Table 7-9 2-stage closed circuit stream data – simulated 2nd stage cyclone and simulated 1st stage cyclones (spigot = 55mm, vortex finder =135mm)

	TPH Solids		% Solids		P80 (mm)		%-75μm	
Stream	Fit	Sim	Fit	Sim	Fit	Sim	Fit	Sim
Circuit Feed	342	342	35.7	35.7	0.138	0.138	57.3	57.3
1st Cyc U/F	109	206	56.4	69.3	0.226	0.183	27.3	28.7
1st Cyc O/F	233	136	30.4	20.9	0.0906	0.037	71.9	99.7
Mill 1 Feed	128	245	59.4	70.4	0.192	0.154	32.8	26.9
Mill 2 Feed	82	245	58.4	70.4	0.192	0.154	32.8	26.9
Mill 1 Prod	82	245	58.4	70.4	0.111	0.121	62.4	44.4
Mill 2 Prod	128	245	59.4	70.4	0.107	0.154	64.7	26.9
2nd Cyc Feed	443	625	39.5	46.5	0.099	0.109	68.0	59.4
2nd Cyc U/F	101	283	62.0	71.3	0.147	0.138	38.8	25.5
2nd Cyc O/F	342	342	35.7	36.1	0.0819	0.065	76.8	89.0

Table 7-10 Performance data of the simulated 1st and 2nd cyclones (with vortex finder diameter = 135mm; spigot diameter = 55mm)

	Original 1st stage cyc	New 1st stage cyclone	New 2nd stage cyclone	New 2nd stage cyclone*
Number of cyclones in parallel	4	8	11	11
Water Split To O/F (%)	81.16	85.17	83.01	83.14
Corrected D50, mm (Total)	0.114	0.0561	0.0883	0.0896
Operating Pressure, kPa	55.1	75.71	75.53	76.03
* in operation with the original 1st stage cyclone				

7.1.2.3 Simulated Classification Improvement

The improvement in classification is presented in Figure 7-5 and Figure 7-6 by means of partition curves. The simulated cyclone shows higher classification efficiency than the existing cyclone. This is seen by the steepness of the simulated 2nd stage cyclone partition curve. Both the 1st and 2nd stage cyclone partition curves have shifted to the left, giving a smaller cut size. There is an observed improvement in separation efficiency in the 2nd stage cyclone as can be seen by the increased steepness of the slope (Figure 7-5). For fine separation it is recommended that smaller cyclones should be used. By using smaller diameter cyclones, the cyclones were now cutting at a fine point (Table 7-10, Figure 7-5, and Figure 7-6).

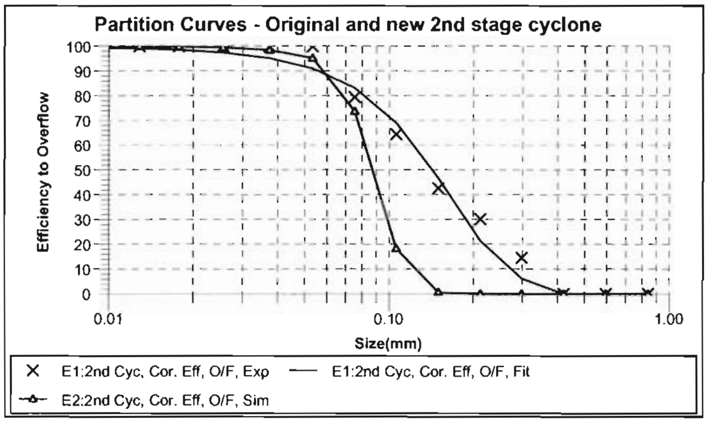


Figure 7-5 Simulated and existing 2nd stage cyclone partition curve – 2 stage closed circuit

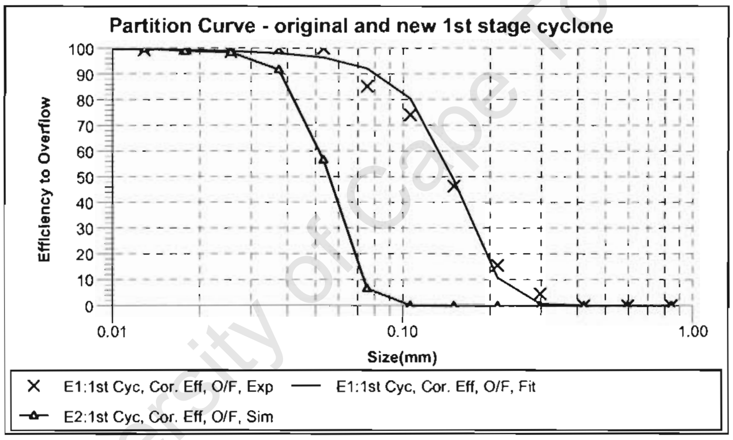


Figure 7-6 Simulated and existing 1st stage cyclone partition curve – 2 stage closed circuit

7.4 Comparison of Performance between Open, 1-Stage and 2-Stage Closed Circuit

In order to make a meaningful comparison between the open, 1-stage and 2-stage closed circuit configuration, the simulations had to be compared under similar conditions. The fitted data from the 2-stage closed circuit test were used as a base point from which the simulations were based. By rearranging the components in the simulator, the 2-stage closed circuit was modified into the open and 1-stage closed circuit configurations. The breakage rates from the 2-stage circuit were used for all the simulations because the breakage rates between the 1-stage and 2-stage closed circuit were very different as shown in Figure 7-1. By using the same fitted circuit, simulations were conducted and the results are shown in Table 7-11 which shows the 2-stage closed circuit fitted data and the circuits' performance predictions. The 2-stage closed circuit data was used as a base case because this is the current configuration employed in the studied circuits. Again the 2-stage circuit was simulated in two steps, the first by using the 400mm cyclones in the 2nd stage only of which the results are given in the B-column of Table 7-11 and for the second step both stages were using the 400mm cyclones–B columns.

The simulations conducted above may not be realistic due to the high circulating loads, because of that in the simulations discussed in this section the circulating loads were brought down by using bigger vortex (160mm) finders and smaller spigots (50mm) in order to coarsen the cut sizes. Although these may not be optimal operations, they are realistic and thus the predictions can be trusted with high confidence.

Table 7-11 Comparison between open, 1-stage and 2-stage closed circuit

	tph solids					% solids					%-75microns				
Stream	Fit	Open	1-stage	2-stage		Fit	Open	1-stage	2-stage		Fit	Open	1-stage	2-stage	
				A	B				A	B				A	B
Circuit Feed	342					35.7					57.3				
1st Cyc U/F	109	206		147	188	56.4	69.3		55.8	73.1	27.3	99.7		28.3	23.6
1st Cyc O/F	233	136		195	154	30.4	20.9		28.0	22.3	71.9	28.7		80.2	98.8
Mills Feed	211	206	192	326	349	59.0	69.3	75.0	65.2	71.7	32.8	28.7	18.5	23.8	24.7
Mills Prod	211	206	192	326	349	39.5	69.3	75.0	65.2	71.7	68.0	63.8	63.9	51.7	53.1
2nd Cyc U/F	101		192	179	160	62.0		75.0	75.5	70.0	38.8		18.5	20.0	25.8
Circuit Prod	342					35.7					76.8	77.9	83.9	85.5	87.1

From the same circuit feed the simulation predicts a %-75μm of 78% for the open circuit configuration, which is lower than the closed circuit configurations at least 87%. The 2-stage closed circuit with both classification stages using 400mm cyclones predicts the highest %-75μm of 89%. This is owing to a finer feed being received in the 2nd stage cyclones partly due to the increased classification efficiency of the 1st cyclone- which led to a better mill performance. This is seen by the finest mill product for the 2-stage-B circuit of 59%-75μm.

The particle size distributions from the four conditions tested are shown in Figure 7-7. The open circuit product size distribution shows a flatter gradient and the presence of coarse material whereas the closed circuits product size distribution curves show steeper gradient at the coarse end, showing an efficient elimination of coarse particles from the circuit product. From this it is apparent that for a given circuit feed and the appropriate cyclones the closed circuit configurations will exhibit a finer circuit product. It appears that for a more optimal set of cyclones the 2-stage closed circuit slightly outperforms the 1-stage closed circuit. It is therefore felt that due to the minimum difference between the performances of the 1-stage and the 2-stage circuit, it might not be worth the trouble and cost to operate these circuits in two stages.

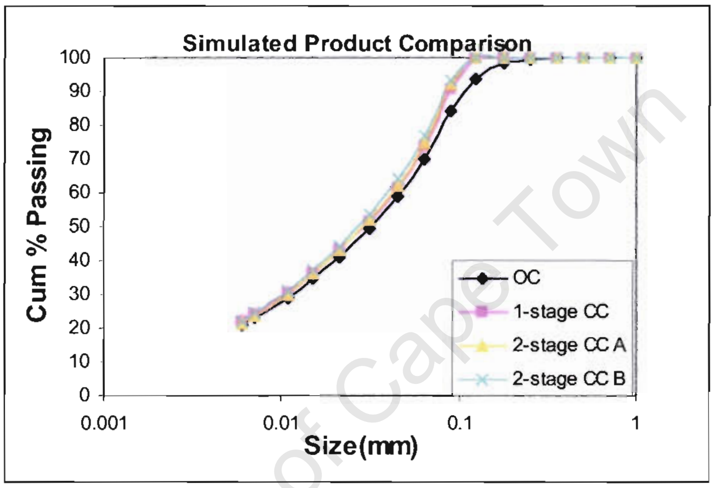


Figure 7-7 Comparison of the predicted product size distribution for the open circuit(OC), 1-stage closed circuit (1 stage CC), 2-stage closed circuit-A, and 2-stage closed circuit-B

7.5 Conclusions

Mathematical models found in JKSimMet can be fitted to the experimental data in order to predict the outcomes of circuit changes. Smaller cyclones from a regrind circuit of another Merensky ore concentrator were used to replace the existing cyclones in the simulation in order to assess the circuit performance after the proposed changes. The outcomes of the simulations showed that improving the efficiency of the 2^{nd} stage cyclones can result in a marked improvement in the circuit performance, which shows that the 2^{nd} stage cyclone performance is critical to the performance of a closed circuit configuration. Simulations show that the 2-stage closed circuit with appropriate cyclones can outperform both the open and 1-stage closed circuit configurations with the open circuit being the least efficient.

A similar improvement on the 1st stage cyclone does not show as much improvement to the circuit. The 1-stage closed circuit would be the more favourable, because in 2-stage closed circuit the 1st stage cyclone does not seem to add worthwhile improvements to the circuit.

Chapter 8

CONCLUSIONS AND RECOMMENDATIONS

8.1 Conclusions

Based on the findings of the investigations, the following conclusions were reached:

8.1.1 Effect of Circuit Configuration on Size Reduction

From the studies conducted at three concentrators, Lebowa, Amandelbult and Waterval, to compare the performance between the open and 2-stage closed circuit configurations it was concluded that there is no significant difference in operation of the two circuit configurations. The 2-stage closed circuit did not show any improvement over the open circuit in terms of the particle size reduction across the circuit in three concentrators. This indicates that installing hydrocyclones to close a circuit will not necessarily improve the performance of that circuit. Napier-Munn et al (1999) noted that efficient classification in a closed circuit will offer 5-10% improvement in performance and poor classification will give 5-10% below average performance.

Due to the differences in the particle size distribution and the flowrate of the circuit feed between the 1- and 2-stage closed circuit tests in Waterval, the Bond work equation was used to calculate the grinding efficiency in order to make a performance comparison between the two circuit configurations. The results indicated that the 1-stage closed circuit showed a higher grinding efficiency than the 2-stage closed circuit configuration. This is attributed to the doubled inefficiencies because of the two stages of cyclones in the 2-stage closed circuit.

Simulations were conducted to compare the performance between open, 1-stage and 2-stage closed circuit configurations. It was found that the open circuit configuration had the least efficiency. For a more efficient classification system, the 2-stage closed circuit outperforms the 1-stage closed circuit, probably because the amount of fines bypassing classification due to inefficiencies has been greatly reduced.

8.1.2 Effect of Classification Efficiency on Size Reduction

The hydrocyclones in the circuits were observed to be operating inefficiently. If the classification efficiency can be increased in the closed circuit configuration, then these circuits will show a significant improvement over the open circuit configuration. Hukki (1976) reported drastic improvement in closed circuit operation by increasing the classification efficiency.

Simulations with smaller cyclones which are suited for the regrind applications were conducted and the following conclusions were drawn from the simulated circuit:

- Considerable improvements were predicted where %-75μm was increased by 7% in 1-stage closed circuit and by 8% in 2-stage closed circuit configuration.

- For 2-stage closed circuit configuration the results showed that the overall circuit performance does not improve significantly by improving classification efficiency of the 1st stage cyclones.

8.2 Recommendations

In light of the findings and conclusions, the following recommendations were made to improve future work, and the performance of the circuits tested in this investigation.

1. This work has shown that cyclone performance dictates the success of a closed circuit configuration. It is therefore recommended that smaller cyclones be installed in the regrind circuits rather than trying to fine-tune the apertures. Predictions of using smaller cyclones showed significant increase in % -75μm in the circuit product of the closed circuit configurations.

2. The observed outcomes of the simulations in the 2-stage closed circuit configuration showed that the 1st stage cyclones appeared not to add any benefit to the circuit performance. It will therefore be a cost saving to use the 1-stage closed circuit configuration. Additionally, the inevitable cyclone inefficiencies will be limited to one stage of cyclones.

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Appendix A.

Brief Background to the Concentrators

A.1. Amandelbult Merensky Concentrator

The Merensky plant at the Amandelbult concentrator has three grinding stages, namely three primary SAG mills, three secondary and two tertiary ball mills. There are three belts feeding the three primary parallel SAG mills with fresh feed. Each mill product is sent to a vibratory screen whereby the screen undersize is sent to the primary rougher flotation cells and the oversize is re-circulated to the primary SAG mills. There are two vibratory screens per mill, one serves as a stand-by.

The secondary circuit has three parallel ball mills; the secondary circuit is fed by a combination of rougher tails and the primary cleaner tails. The secondary circuit feed is classified by three cyclones, one cyclone per mill. The cyclone underflow goes into the mill and the cyclone overflow goes into the mill discharge sump where it combines with the respective mill discharge. Number 1 and number 2 secondary mill discharge sumps feed into the number three mill discharge sump; the combined product is sent to scavenger 1 floatation cells.

The scavenger 1 tails are sent to a nest of cyclones (1^{st} stage cyclone) in the tertiary circuit, the underflow is split at a T-piece and goes to the two parallel tertiary ball mills and the overflow is sent to the common tertiary mills' discharge sump. The mills' product goes into the discharge sump and then for closed circuit configuration the mill sump product is sent to two sets of 2^{nd} stage nests of cyclones for classification. The cyclones' overflows are combined in a sump from where the slurry is sent to the scavenger 2 flotation cells, and the underflows are re-circulated to the tertiary mills for further grinding.

For the open circuit configuration the mill discharge sump product is sent directly to the second sump and then to the scavenger 2 flotation cells; which means that it bypasses the 2^{nd} stage cyclones.

A.2. Waterval Merensky Concentrator

The Waterval Merensky concentrator is not very different from the Amandelbult concentrator. The concentrator has a crushing plant where ore from the mine is crushed. In Waterval there are two identical limbs, each limb with five primary and secondary ball mills and two tertiary ball mills. Unlike at Amandelbult, at Waterval concentrator primary milling is by means of ball mills.

The tertiary circuit is fed by the secondary rougher tails, an external tailing stream from the Frank concentrator and the crusher classifier overflow tails. These streams combine in a sump, the combined stream is pumped to the 1st stage cyclone when operating in 2-stage closed circuit. The 1st stage cyclone overflow bypasses the mills and combines with the mills product in a mill discharge sump. The mills are fed by both the 1st and 2nd stage cyclone underflows. The sump product is pumped to the 2nd stage cyclone. The 2nd stage cyclones overflow is the circuit product which feeds the scavenger flotation cells.

In 1-stage closed circuit, the feed to the circuit goes into the mill product sump and combines with the mills' product. The sump product feed the '2nd stage cyclones and the cyclone overflow is the circuit product and the cyclone underflow is the mill feed. This circuit bypasses the 1st stage cyclone underflow. This arrangement is not available in the other concentrators.

The open circuit arrangement bypasses does not use the 2nd stage cyclones. The circuit product is composed of the mill products and the 1st stage cyclone overflow.

Appendix B.
Efficiency Curve Calculations

Partition /efficiency curves are used to evaluate the performance of hydrocyclones. The partition curve relates the weight fraction of particles in the feed which reports to the underflow or overflow. A cut-point/ cut size normally referred to as d_{50} is the point on the partition curve for which 50% of particles of that size have a chance of reporting to the overflow. In an ideal classifier, the particles that are bigger than the d_{50} will report to the coarse stream and those smaller will report to the fine stream. This cannot be achieved in real classifiers. The closer the real partition curve is to the ideal one the better is its classification efficiency.

In earlier work at the JKMRC, it was argued that the classifier overflow is generally the product in many grinding circuits, because of that it was decided to use the partition curve to the overflow. In much of the literature the partition curve to the underflow is used; the two just complement each other. The actual efficiency to the overflow of size *i* can be calculated from Equation B-1.

Equation B-1 Actual recovery, E_{oi}, to overflow (Napier-Munn *et al*, 1999, Eq. 12.1)

$$E_{oi} = 100\left(\frac{W_{oi}M_o}{W_{fi}M_f}\right)\%$$

W_{oi} and W_{fi} are proportions by weight of material of size I in overflow and feed solids respectively,

M_o and M_f are total mass flowrates of overflow and feed streams respectively

W_{oi} and W_{fi} are determined from size analysis of samples of overflow and feed streams. These are not always available; in that case, this would be calculated provided information about the other streams is available. M_o and M_f are measured by different methods, some of which are discussed in Chapter 4 under experimental details.

Most classifier curves do not reach 100% at zero size, this is attributed to the idea that fine particles do not respond to classification forces and they simply follow the water flow, hence bypass classification.

There are different correction procedures and these are discussed by Napier-Munn *et al* (1999). Kelsall (1953) developed an approach which is based on the theory that particles of all sizes report to the underflow in proportion to the water recovery in the underflow. This, however, has been questioned by other researchers (Finch, 1983) who suggested that classification bypass decreases with increasing particle size.

JKSimMet which was employed in this thesis uses Equation B-2 for the corrected efficiency curve.

Equation B-2 Corrected efficiency curve used in JKSimMet (Napier-Munn *et al*, 1999, Eq. 12.11)

$$E_{oa} = C\left[\frac{(1+\beta\beta^{*}x)(\exp(\alpha)-1)}{\exp(\alpha\beta^{*}x)+\exp(\alpha)-2}\right]$$

Where

C = proportion of water to overflow

α = efficiency parameter

β^* and β are fish-hook parameters, when no fish hook is present, β^* is set to 1 and β to 0.

$x = d/d_{50C}$

Appendix C. Simulation Models

C.1. *Ball Mill Model*

The Whiten perfect mixing ball mill model is well known in mineral processing circles. The model is based on a basic material balance, at steady state the balance in the mill for a particular size class is:

Feed in + Breakage in = product out + Breakage out,

and can be represented mathematically by Equation C-1.

Equation C-1

$$f_i + \sum_{j=1}^{i} a_{ij} r_j s_j = p_i + r_i s_i \ ,$$

Where , a_{ij} = appearance function

r_i = rate of breakage

s_i = mill contents

For a perfectly mixed mill the contents of the mill are related to the mill product with a discharge rate, d_i, such that:

Equation C-2

$$p_i = d_i s_i \quad \text{or} \quad s_i = p_i / d_i$$

Now, writing Equation C-1 in terms the measurable parameters, feed size and the product size,

Equation C-3

$$f_i + \sum_{j=1}^{i} a_{ij} \frac{r_j p_j}{d_j} = p_i + \frac{r_i p_i}{d_i}$$

The function a_{ij} is measured or assumed from previous results .The only needed parameters are the feed and product size distribution which are also measured, then r_i/d_i can be calculated and then the mill is defined. Due to variations in residence time, d_i is scaled in terms of the mill volume and the volumetric flowrate, Q, to the term d_i^*.

Equation C-4

$$d_i^* = \left(\frac{D^2 L}{4Q}\right) d_i ,$$

L = mill length

D = mill diameter

The values of r/d^* are poorly determined for each size fraction, for that reason the r/d^*function is represented by a cubic spline function. This spline function provides a smooth curve which is defined by 3 or 4 values (x_1-x_4) selected by the user as shown in Figure C-1.

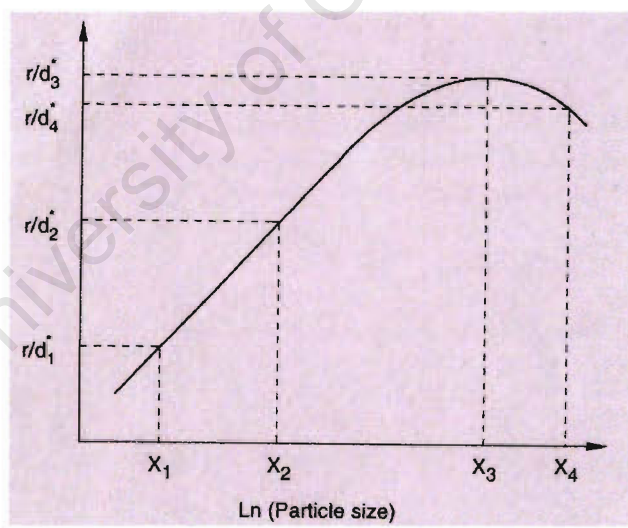

Figure C-1 Grinding rates variation with particle size (Napier-Munn *et al*, 1999, figure 2.6)

C.2. Hydrocyclone Model

There are a couple of models out there as discussed by Napier-Munn *et al* (1999). The principal hydrocyclone used by JKSimMet is the Nageswararao Model (cited in

Napier-Munn *et al*, 1999). The model comprises empirical equations. The predictive equations are:

Equation C-5 Separation size

$$\frac{d_{50c}}{D_c} = K_{D1}\left(\frac{D_o}{D_c}\right)^{0.52}\left(\frac{D_u}{D_c}\right)^{-0.47}\lambda^{0.93}\left(\frac{P}{\rho_p g D_c}\right)^{-0.22}\left(\frac{D_i}{D_c}\right)^{-0.5}\left(\frac{L_c}{D_c}\right)^{0.2}\theta^{0.15}$$

And $K_{D1} = K_{D0} D_c^{-0.65}$, K_{D0} depends on feed solids characteristics only.

Equation C-6 Volumetric flowrate

$$Q_f = K_{Q1} D_c^2 \left(\frac{P}{\rho_p}\right)^{0.5}\left(\frac{D_o}{D_c}\right)^{0.68}\left(\frac{D_i}{D_c}\right)^{0.45}\theta^{-0.1}\left(\frac{L_c}{D_c}\right)^{0.2}$$

For cyclones of Krebs geometry treating identical feed

$K_{Q1} = K_{Q0} D_c^{-0.1}$ where K_{Q0} depends on feed solids characteristics only

Equation C-7 Water split to underflow

$$R_f = K_{w1}\left(\frac{D_o}{D_c}\right)^{-1.19}\left(\frac{D_u}{D_c}\right)^{2.40}\left(\frac{P}{\rho_p g D_c}\right)^{-0.53}\lambda^{0.27}\left(\frac{D_i}{D_c}\right)^{-0.50}\theta^{-0.24}\left(\frac{L_c}{D_c}\right)^{0.22}$$

Equation C-8 Volumetric recovery of slurry to underflow

$$R_v = K_{v1}\left(\frac{D_o}{D_c}\right)^{-0.94}\left(\frac{D_u}{D_c}\right)^{1.83}\left(\frac{P}{\rho_p g D_c}\right)^{-0.31}\left(\frac{D_i}{D_c}\right)^{-0.25}\theta^{-0.24}\left(\frac{L_c}{D_c}\right)^{0.22}$$

Where D_i = inlet diameter (m)

D_o = vortex finder diameter (m)

D_u = apex (spigot) diameter (m)

D_c = cyclone cylinder diameter (m)

L_c = length of cylindrical section (m)

θ = cone full angle (degrees)

P = feed pressure at inlet (kPa)

ρ_p = feed slurry density (t/m^3)

g = acceleration due to gravity (9.81 m/s^2)

Q_f = flowrate (m^3/h)

R_f = recovery of water to underflow (%)

R_v = volumetric recovery of feed slurry to underflow (%)

K = constant to be estimated from data

λ = hindered settling correction term = $10^{1.82C_v} / (8.05[1-C_v]^2)$

C_v = volumetric fraction of solids in feed slurry

The K-constant is dependent on feed solids characteristics; it can be estimated from surveys of existing plants in the application where JKSimMet is used to optimise an existing cyclone.

Appendix D. Principles of Sizing Techniques

D.1. *Particle Sizing – Sieve Analysis*

The sieve analysis is used to determine the particle size distribution of the streams around the circuit. The sieving/screening analysis is accomplished by passing a known weight of sample through finer sieves/screens and weighing the amount retained in each sieve. This is done to determine the percentage fraction in each size. The screening process happens such that the particles smaller than the aperture size pass rapidly and secondly the so called 'near-size' particles gradually go through the apertures and rarely reach completion. Screening is conducted up to size 32µm. The sub 32µm analysis was conducted by using laser diffraction method.

The success of the sieve analysis technique largely depends on the size of the sample, which affects the chance of each particle being presented to an aperture, and the type of movement of the screen/sieve shaker. Typical screen shakers are shown in Figure 4-8 and Figure 4-9. If the screen is overloaded with the sample, then not all particles are going to have a chance to meet an aperture. On the other hand the sample must be large enough to be representative (Wills, 1997). Errors can occur due to blinding, particle breakage and mesh stretching caused by overloading. Blinding is most common on the smaller sieves.

D.2. *Sub-sieve Particle Sizing – Laser Diffraction Method*

The aim of conducting the sub-sieve analysis of particle size is primarily to determine the fractions of sizes below the 32µm screen size. The size range of interest in this study goes down to sub 10µm. The method of sizing the sub-sieve particles which uses the scattering of light by suspensions began to be implemented in the late 1970's. The original manufacturers were Cilas Granulometer, the Leeds and Northrup Microtrac, and the Malvern Particle Sizer (Syvitski, 1991).

The method is based on the measurements of the forward diffracted light from a disperse suspension. The angle of diffraction is inversely proportional to particle size and the intensity of the diffracted beam at any angle is a measure of the mean projected areas of particles of a specific size (Allen, 1981).

The principle is illustrated in Figure D-1; laser light is passed through a dilute suspension of particles which circulate through the optical cell. The light is scattered by the particles in the suspension, and it is captured by the light detector which measures light intensities over a wide range of angles. A theory of light scattering is used to calculate the particle size distribution from the light distribution pattern, finer material inducing more scatter than coarse (Napier-Munn *et al*, 1999). The technique is based on the assumption that all particles are spheres (Xu and Di Guida, 2003).

Xu and Di Guida (2003) conducted experiments where they compared different sizing techniques on differently shaped particles. They found that the techniques agree to the size distribution for particles that are of a spherical shape. As soon as the shape departs from that of the spheres, the laser diffraction method gives oversized size distribution and exaggerated distribution broadness due to the orientation of the particles.

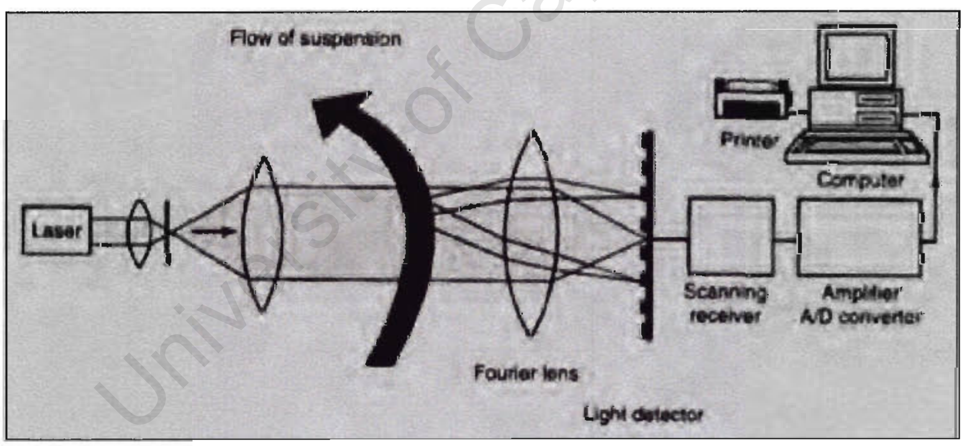

Figure D-1 Laser diffraction instrument principle, (Napier-Munn, 1999, figure A3.4)

The method is fast, easy to use and it gives very reproducible results, however, the results are not compatible with other methods, such as screening, there is a discontinuity when moving from screen sizes to the laser-sizer sizes. It was observed that the off-set was constant for the mineral being studied. A method of inter-conversion was developed by regression for material of consistent characteristics.

Figure D-2 shows a plot of particles size distribution from the laser diffraction method. It shows the results from two streams, the tertiary feed and the cyclone overflow. The graphs show good reproducibility of the system for both streams.

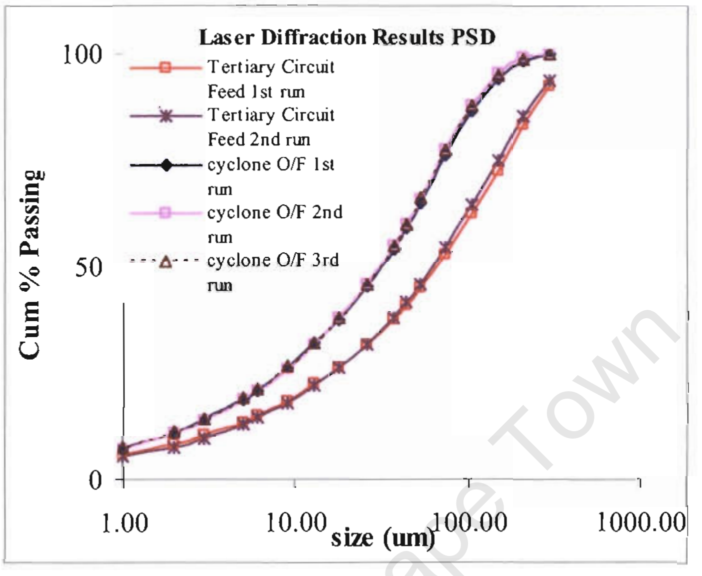

Figure D-2 Laser diffraction results showing reproducibility

Appendix E.

Machine Details

Table E-1 Circuit Configurations details

Site	Configuration tested (cyc. stages)	No of Parrallel Mills	No. of stages of hydroclones	Stage in operation	No. of stage 1 hydrocyclones in operation	No. of stage 2 hydrocyclones in operation
Lebowa	Open	1	1	first	1	-
	Closed (2)	1	1	first	1	1
Amandelbult	Open	2	1	first	4/5	-
	Closed (2)	2	2	first and second	4/5	2 x 2/3
Waterval	Open	2	1	first	3/5	-
	Closed (1)	2	1	second	-	2/4
	Closed (2)	2	2	first and second	3/4	2/4
	Closed (2)	2	2	first and second	4/5	2/4

Table E-2 Mills' Details

	Lebowa	Amandelbult		Waterval	
		Mill 1	Mill 2	Mill 1	Mill 2
Internal Diameter (mm)	3527	4163	4122	4107	4167
Internal Length (mm)	5502	6550	6580	6760	6695
Percent Filling %	31.6	43.3	39.8	35.7	32.5
Fraction of Critical Speed	0.71	0.75	0.75	0.7	0.7

Table E-3 Hydrocyclones' Dimensions

	Lebowa		Amandelbult		Waterval	
	1st stage cyc	2nd stage cyc	1st stage cyc	2nd stage cyc	1st stage cyc	2nd stage cyc
cyclone diamater (mm)	320	405	700	500	640	750
inlet dimension (mm)	223	281	270	200	293	300
spigot diameter (mm)	104	104	100	100	110	110
vortex finder (mm)	167	183	300	224	240	256
cylindrical length (mm)	560	540	618	450	344	600
cone angle (degrees)	30	30	20	15	20	15

Table E-4 Circulating Loads* across the Concentrators

	Lebowa	Amandelbult	Waterval - 2 stage		Waterval -1 stage
Circulating Load	65	46	72	70	67

$$*CirculatingLoad = \frac{MillFeed}{NewCircuitFeed} \times 100$$ (Napier-Munn et al, 1999)

Appendix F. Streams Data

Table F-1 Amandelbult –Open Circuit Data-Exp (calc.) and mass balanced

	OC Circuit Feed		OC 1st cyc U/F		OC 1st cyc O/F		OC Ter Mill 1 Prod		OC Ter Mill 2 Prod		OC Mill sump Prod		OC Circuit Product	
	exp	bal	exp	bal	exp		exp	bal	exp	bal	exp	bal	exp	bal
TPH Solids	353	353	172	171	181	181	65	65	107	107	353	353	353	353
Solids SG (t/m³)	3.2	3.2	3.2	3.2	3.2	3.2	3.2	3.2	3.2	3.2	3.2	3.2	3.2	3.2
TPH Water	630	672	97	105	510	568	41	40	67	64	706	672	706	705
% Solids	35.9	34.4	63.8	62.1	26.2	24.2	61.2	61.8	61.4	62.3	33.3	34.4	33.3	33.3
Pulp SG (t/m³)	1.3	1.3	1.7	1.6	1.2	1.2	1.6	1.6	1.6	1.6	1.3	1.3	1.3	1.3
Vol. Flow (m³/h)	760	803	161	168	577	635	65	64	107	104	837	803	837	836
% -75μm	52.9	53.1	20.8	20.6	83.8	84.5	72.0	72.3	56.1	56.4	74.3	73.6	74.3	73.6
P80 (mm)	0.155	0.153	0.220	0.222	0.069	0.068	0.092	0.092	0.118	0.118	0.089	0.090	0.089	0.090
Size (mm)	OC Circuit Feed	OC Scavenger 1 Tails Bal	OC 1st cyc U/F	OC 1st cyc U/F Bal	OC 1st cyc O/F	OC 1st cyc O/F Bal	OC Ter Mill 1 Prod	OC Ter Mill 1Prod Bal	OC Ter Mill 2 Prod	OC Ter Mill 2 Prod Bal	OC Mill sump Prod	OC Mill sump Prod Bal	OC Circuit Product	OC Scav 2 Feed Bal
1.400	100.0	100.0	100.0	100.0	100.0	100.0	100.0	100.0	100.0	100.0	100.0	100.0	100.0	100.0
1.000	100.0	100.0	99.9	99.9	100.0	100.0	100.0	100.0	100.0	100.0	100.0	100.0	100.0	100.0
0.710	99.8	99.8	99.7	99.7	100.0	100.0	100.0	100.0	100.0	100.0	100.0	100.0	100.0	100.0
0.500	99.0	99.1	98.4	98.3	100.0	99.9	100.0	100.0	99.9	99.9	99.9	99.9	99.9	99.9
0.355	97.1	97.4	94.9	94.7	100.0	99.9	99.9	99.9	99.6	99.6	99.8	99.8	99.8	99.8
0.250	92.6	92.9	86.1	85.8	99.8	99.7	99.5	99.5	98.4	98.3	99.1	99.2	99.1	99.2
0.180	84.1	84.2	68.8	68.6	99.2	99.1	97.7	97.7	93.4	93.1	96.8	97.0	96.8	97.0
0.125	73.7	74.2	50.2	49.8	97.3	97.2	92.3	92.3	83.3	83.1	91.9	92.0	91.9	92.0
0.090	58.5	58.8	26.2	25.9	89.4	89.9	78.8	79.0	63.8	64.0	80.6	80.1	80.6	80.1
0.063	47.3	47.3	16.0	15.9	76.0	76.9	64.0	64.3	47.9	48.2	66.7	65.9	66.7	65.9
0.045	37.7	36.7	10.8	10.9	59.6	61.0	47.1	47.4	35.8	36.1	51.7	51.0	51.7	51.0
0.032	32.1	31.5	9.7	9.8	51.5	52.1	40.5	40.6	30.0	30.2	43.7	43.4	43.7	43.4
0.021	26.3	25.4	7.6	7.7	41.5	42.2	32.3	32.4	24.4	24.5	35.2	35.1	35.2	35.1
0.015	22.2	21.5	6.4	6.5	35.2	35.7	27.4	27.5	20.7	20.8	29.8	29.7	29.8	29.7
0.011	18.2	17.6	5.3	5.4	29.0	29.3	22.7	22.7	17.2	17.2	24.4	24.4	24.4	24.4
0.007	14.6	14.1	4.2	4.3	23.1	23.3	18.5	18.5	14.0	14.0	19.6	19.6	19.6	19.6
0.006	13.2	12.7	3.8	3.9	20.8	21.0	16.8	16.8	12.8	12.8	17.8	17.8	17.8	17.8

Table C-2 Amandelbult –Closed Circuit Data-Exp (calc.) and mass balanced

	CC Circuit Feed		1st Cyc U/F		1st Cyc O/F		Ter Mill 1 Prod		Ter Mill 2 Prod		2nd cyc 1 U/F		2nd cyc 1 O/F		2nd cyc 2		2nd cyc 2		CC Circuit Product	
	EXP	Bal	EXP	Bal	EXP	Bal	EXP	Bal	EXP	Bal	EXP	Bal	EXP	Bal	EXP	Bal	EXP	Bal	EXP	Bal
TPH Solids	292	292	122	123	170	169	61	62	74	76	7	7	134	134	8	8	158	158	292	292
Solids SG (t/m^3)	3.2	3.2	3.2	3.2	3.2	3.2	3.2	3.2	3.2	3.2	3.2	3.2	3.2	3.2	3.2	3.2	3.2	3.2	3.2	3.2
TPH Water	505	523	70	74	479	449	36	36	50	49	4	4	284	273	6	6	252	249	518	523
% Solids	36.6	35.8	63.4	62.4	26.2	27.4	62.6	63.5	59.7	60.8	61.7	61.6	32.0	32.9	57.2	57.1	38.5	38.8	36.0	35.8
Pulp SG (t/m^3)	1.3	1.3	1.7	1.6	1.2	1.2	1.7	1.7	1.6	1.6	1.6	1.6	1.3	1.3	1.6	1.6	1.3	1.3	1.3	1.3
Vol. Flow (m^3/h)	613.1	630.6	115.1	119.5	541.8	511.2	59.0	58.6	77.6	77.1	7.0	6.9	333.8	322.8	9.1	9.3	310.7	307.9	626.4	630.6
% -75μm	18.0	18.0	5.2	5.2	27.3	27.4	16.0	16.0	12.6	12.6	6.9	6.9	22.7	22.8	9.6	9.6	21.6	21.5	22.2	22.1
P80 (mm)	0.151	0.148	0.235	0.236	0.077	0.076	0.116	0.116	0.134	0.134	0.177	0.177	0.094	0.093	0.146	0.146	0.100	0.100	0.096	0.097
Size (mm)	CC Circuit Feed	CC Circuit Feed	CC 1st Cyc U/F	CC 1st Cyc U/F Bal	CC 1st Cyc O/F	CC 1st Cyc O/F Bal	CC Ter Mill 1 Prod	CC Ter Ball Mill 1 Prod Bal	CC Ter Mill 2 Prod	CC Ter Mill 2 Prod Bal	CC 2nd cyc 1 U/F	CC 2ndd cyc 1 U/F Bal	CC 2nd cyc 1 O/F	CC 2nd cyc 1 O/F Bal	CC 2nd cyc 2 U/F	CC 2nd cyc 2 U/F Bal	CC 2nd cyc 2 O/F	CC 2nd cyc 2 O/FBal	CC Circuit Product	CC Circuit Product
1.400	100	100	100	100	100	100	100	100	100	100	100	100	100	100	100	100	100	100	100	100
1.000	100	100	100	100	100	100	100	100	100	100	100	100	100	100	100	100	100	100	100	100
0.710	99.6	99.7	99.4	99.4	100	100	100	100	99.9	99.9	99.9	99.9	100	100	99.9	99.9	100	100	99.9	100
0.500	98.8	98.9	97.5	97.5	100	100	99.9	99.9	99.7	99.7	99.4	99.4	99.9	99.9	99.6	99.6	99.9	99.9	99.9	99.9
0.355	96.8	97.0	93.1	93.0	99.9	99.9	99.6	99.6	99.2	99.2	98.1	98.1	99.8	99.8	98.8	98.8	99.7	99.6	99.7	99.7
0.250	92.3	92.6	83.1	82.8	99.7	99.7	98.6	98.6	97.1	97.1	93.6	93.6	99.2	99.3	96.1	96.1	98.9	98.8	99.0	99.0
0.180	84.3	84.7	65.3	65.0	99.1	99.1	94.2	94.2	90.1	90.1	80.8	80.8	97.0	97.2	87.6	87.6	96.0	95.9	96.5	96.5
0.125	75.0	75.5	46.0	45.7	97.2	97.2	83.8	83.8	77.2	77.2	61.3	61.3	91.8	92.0	73.4	73.4	89.7	89.5	90.8	90.7
0.090	59.9	60.0	23.3	23.3	86.1	86.7	64.9	64.9	55.7	55.7	36.5	36.5	78.0	78.2	48.8	48.8	74.8	74.6	76.9	76.2
0.063	47.4	47.4	14.8	14.8	70.5	71.1	48.3	48.3	40.5	40.5	23.0	23.0	62.5	62.6	29.3	29.3	59.1	58.9	61.5	60.6
0.045	38.5	38.9	10.9	10.8	59.4	59.3	35.6	35.6	27.6	27.6	15.4	15.4	49.6	49.8	20.4	20.4	47.0	46.8	48.4	48.2
0.032	33.3	33.6	9.4	9.4	51.2	51.3	29.6	29.6	22.8	22.8	13.8	13.8	42.2	42.4	17.5	17.5	40.1	40.0	41.3	41.1
0.021	26.7	27.0	7.6	7.6	41.2	41.1	24.1	24.1	18.8	18.8	10.5	10.5	34.1	34.2	14.1	14.1	32.4	32.3	33.2	33.2
0.015	22.7	22.9	6.5	6.5	34.8	34.8	20.3	20.3	15.9	15.9	8.8	8.8	28.8	29.0	12.0	12.0	27.4	27.3	28.2	28.0
0.011	18.7	18.8	5.4	5.4	28.5	28.6	16.7	16.7	13.1	13.1	7.2	7.2	23.7	23.8	10.0	10.0	22.5	22.4	23.2	23.1
0.007	15.1	15.2	4.4	4.4	22.8	23.0	13.5	13.5	10.6	10.6	5.8	5.8	19.1	19.2	8.1	8.1	18.1	18.1	18.7	18.6
0.006	13.7	13.7	4.0	4.0	20.5	20.7	12.3	12.3	9.6	9.6	5.3	5.3	17.3	17.3	7.4	7.4	16.4	16.3	16.9	16.8

Table F-2 Waterval – Open Circuit Data-Exp (calc.) and mass balanced

	OC Circuit Feed		OC 1st cyc U/F		OC 1st cyc O/F		OC Ter mi	OC Ter mill 1 Prod		OC Ter mi	OC Ter mill 2 Prod		OC Circuit Prod	
	Exp	Bal	Exp	Bal	Exp	Bal	Bal	Exp	Bal	Bal	Exp	Bal	Exp	Bal
TPH Solids	300.0	299.9	105.0	105.0	194.0	194.9	69.4	69.4	69.4	35.6	35.6	35.6	300.0	299.9
Solids SG (t/m³)	3.2	3.2	3.2	3.2	3.2	3.2	3.2	3.2	3.2	3.2	3.2	3.2	3.2	3.2
TPH Water	609.1	597.4	50.5	53.2	538.1	544.2	36.3	38.0	36.3	17.0	18.7	17.0	603.2	597.4
% Solids	33.0	33.4	67.5	66.4	26.5	26.4	65.7	64.6	65.7	67.7	65.6	67.7	33.2	33.4
Pulp SG (t/m³)	1.3	1.3	1.7	1.7	1.2	1.2	1.7	1.7	1.7	1.7	1.7	1.7	1.3	1.3
Vol. Flow (m³/h)	720.2	708.5	89.4	92.1	609.9	616.3	62.0	63.7	62.0	30.2	31.8	30.2	714.3	708.5
% -75μm	52.8	54.5	17.2	17.1	76.1	75.2	17.2	80.7	80.7	16.9	73.9	73.8	75.9	76.3
P80 (mm)	0.160	0.152	0.241	0.242	0.084	0.085	0.241	0.075	0.075	0.243	0.089	0.089	0.084	0.084
Size (mm)	OC Circuit Feed	OC Circuit Feed	OC 1st cyc U/F	OC 1st cyc U/F	OC 1st cyc O/F	OC 1st cyc O/F	OC Ter mill 1 Feed	OC Ter mill 1 Prod	OC Ter mill 1 Prod	OC Ter mill 2 Feed	OC Ter mill 2 Prod	OC Ter mill 2 Prod	OC Circuit Prod	OC Circuit Prod
2.000	100.0	100.0	100.0	100.0	100.0	100.0	100.0	100.0	100.0	100.0	100.0	100.0	100.0	100.0
1.400	100.0	100.0	99.9	99.9	100.0	100.0	99.9	100.0	100.0	99.9	100.0	100.0	100.0	100.0
1.000	99.9	99.9	99.8	99.8	100.0	100.0	99.8	100.0	100.0	99.8	100.0	100.0	100.0	100.0
0.710	99.8	99.8	99.5	99.5	100.0	100.0	99.5	100.0	100.0	99.5	100.0	100.0	100.0	100.0
0.500	99.2	99.3	98.2	98.2	100.0	99.9	98.2	100.0	100.0	98.2	100.0	100.0	99.9	100.0
0.355	97.2	97.8	93.9	93.9	100.0	99.9	93.9	100.0	100.0	93.9	99.9	99.9	99.8	99.9
0.250	92.2	93.6	82.3	82.1	99.9	99.8	82.3	99.7	99.7	81.8	99.3	99.3	99.6	99.7
0.180	83.2	84.7	58.6	58.5	98.9	98.8	58.6	97.5	97.5	58.2	96.9	96.9	98.3	98.3
0.125	72.6	74.2	38.0	38.0	94.0	93.7	38.0	93.9	93.9	37.9	91.2	91.2	93.4	93.5
0.090	59.1	60.9	21.7	21.5	82.9	82.1	21.7	86.1	86.1	21.2	80.5	80.5	82.5	82.8
0.063	46.3	48.0	13.4	13.3	67.6	66.6	13.4	73.9	73.9	13.2	65.9	65.9	67.8	68.2
0.045	37.4	38.9	10.2	10.1	55.5	54.4	10.2	62.0	62.0	9.8	54.7	54.7	55.8	56.2
0.032	32.2	33.5	8.4	8.2	47.9	47.1	8.4	54.3	54.3	7.7	46.2	46.2	48.4	48.7

Table F-3 Waterval 2 stage Closed Circuit test 1 Data-Exp (calc.) and mass balanced

	CC Circuit Feed		CC 1st Cyc U/F		CC 1st Cyc O/F		CC Ter Mi	CC Ter Mill 2 Prod		CC Ter mil	CC Ter mill 1 Prod		CC 2nd	Cyc U/F	CC 2nd Cyc O/F	
	Exp	Bal	Exp	Bal	Exp	Bal	Bal	Exp	Bal	Bal	Exp	Bal	Exp	Bal	Exp	Bal
TPH Solids	367	368	120	121	246	247	97	95	97	121	121	121	95	96	367	368
3.2	3.2	3.2	3.2	3.2	3.2	3.2	3.2	3.2	3.2	3.2	3.2	3.2	3.2	3.2	3.2	3.2
TPH Water	630	631	54	56	572	575	45	44	45	60	65	60	48	49	631	631
% Solids	36.8	36.8	69.1	68.3	30.1	30.0	68.5	68.5	68.5	66.7	65.1	66.7	66.5	66.4	36.8	36.8
Pulp SG (t/m^3)	1.3	1.3	1.8	1.8	1.2	1.2	1.8	1.8	1.8	1.7	1.7	1.7	1.7	1.7	1.3	1.3
Vol. Flow (m^3/h)	766	767	98	101	663	666	81	79	81	105	109	105	83	84	767	767
% -75µm	51.7	52.4	16.3	16.4	69.8	70.5	23.6	56.4	57.9	22.5	62.8	63.6	31.1	31.4	76.5	75.3
P80 (mm)	0.2	0.2	0.2	0.2	0.1	0.1	0.2	0.1	0.1	0.2	0.1	0.1	0.2	0.2	0.1	0.1
Size (mm)	CC Circuit Feed	CC Circuit Feed	CC 1st Cyc U/F	CC 1st Cyc U/F	CC 1st Cyc O/F	CC 1st Cyc O/F	CC Ter Mill 2 Feed	CC Ter Mill 2 Prod	CC Ter Mill 2 Prod	CC Ter mill 1 Feed	CC Ter mill 1 Prod	CC Ter mill 1 Prod	CC 2nd Cyc U/F	CC 2nd Cyc U/F	CC 2nd Cyc O/F	CC Circuit Prod
2.000	100	100	100	100	100	100	100	100	100	100	100	100	100	100	100	100
1.400	100	100	100	100	100	100	100	100	100	100	100	100	100	100	100	100
1.000	100	100	100	100	100	100	100	100	100	100	100	100	100	100	100	100
0.710	99.9	100	99.9	99.9	100	100	100	100	100.0	99.9	100	100	100	100	100	100
0.500	99.6	99.6	98.9	98.9	100	100	99.4	99.7	99.7	99.3	100	100	99.9	99.9	100	100
0.355	98.2	98.2	94.6	94.6	100	100	97.3	98.7	98.9	96.5	99.9	100.0	99.7	99.7	100	99.8
0.250	93.4	93.6	80.2	80.7	99.7	99.9	89.3	95.3	96.4	87.4	99.3	99.5	97.8	97.7	99.9	99.5
0.180	83.8	84.1	55.8	55.9	97.3	98.0	72.0	87.1	89.3	67.5	95.8	96.4	87.2	86.6	98.9	98.2
0.125	72.6	73.1	35.9	36.2	90.1	91.2	51.6	76.9	79.1	48.1	87.1	88.1	66.6	66.5	94.6	93.5
0.090	58.4	59.1	20.5	20.7	77.2	78.0	30.4	63.0	64.7	28.6	71.0	71.8	40.1	40.3	83.6	82.4
0.063	44.9	45.6	12.7	12.8	61.1	61.6	17.7	49.4	50.7	17.2	53.6	54.2	22.9	23.2	67.5	66.4
0.045	35.9	36.6	9.5	9.5	49.7	49.8	12.8	39.9	40.7	12.5	41.5	41.9	16.4	16.6	54.2	53.5
0.032	30.9	31.4	8.2	8.0	42.3	42.9	10.6	33.3	34.0	10.3	35.0	35.5	13.2	13.5	47.3	45.9

Table F-4 Waterval 1 stage closed circuit data-Exp (calc.) and mass balanced

	Circuit Feed		2nd Cyc Feed		2nd Cyc UF		Mill 1 Prod		Mill 2 Prod		2nd Cyc O/F	
	Exp	Bal	Exp	Bal	Exp	Bal	Exp	Bal	Exp	Bal	Exp	Bal
TPH Solids	353	353	601	601	248	248	91	91	157	157	353	353
Solids SG (t/m³)	3.2	3.2	3.2	3.2	3.2	3.2	3.2	3.2	3.2	3.2	3.2	3.2
TPH Water	534	537	649	654	118	117	42	43	74	74	541	537
% Solids	39.8	39.7	48.1	47.9	67.7	68.0	68.6	68.1	68.0	67.9	39.5	39.7
Pulp SG (t/m³)	1.334	1.333	1.434	1.432	1.743	1.748	1.76	1.751	1.749	1.746	1.331	1.333
Vol. Flow (m³/h)	665	668	871	876	210	209	76	76	132	132	672	668
% -75μm	39.6	39.2	50.9	51.4	24.2	24.3	70.4	70.3	68.0	68.0	71.0	70.7
P80 (mm)	0.197	0.199	0.159	0.159	0.229	0.229	0.0948	0.0951	0.101	0.102	0.0939	0.0942
Size (mm)												
1	100	100	100	100	100	100	100	100	100	100	100	100
0.71	100	100	100	100	100	100	100	100	100	100	100	100
0.5	100	100	100	100	99.4	99.4	100	100	100	100	100	100
0.355	98.0	97.7	98.5	98.5	96.3	96.4	100	100	100	100	100	100
0.25	91.2	90.2	93.6	93.7	84.6	85.1	99.2	99.0	98.8	98.7	99.3	99.4
0.18	78.8	76.7	84.2	84.2	65.2	65.8	96.5	96.0	95.3	95.1	96.5	96.7
0.125	65.6	62.8	72.8	72.8	46.3	46.8	89.6	88.9	87.4	86.8	89.9	90.3
0.09	51.7	48.4	59.2	59.2	30.4	30.7	77.9	77.1	75.2	74.2	78.3	78.4
0.063	38.2	34.7	44.6	44.6	18.7	18.9	61.6	60.8	59.8	58.6	62.4	62.1
0.045	30.1	27.1	35.7	35.7	14.1	14.2	49.7	49.0	49.3	48.1	51.5	50.2
0.031	25.4	22.9	29.9	29.9	11.7	11.8	41.3	40.7	41.1	40.1	43.1	42.2
0.021	21.4	19.3	24.9	25.0	9.7	9.8	34.0	33.5	34.2	33.4	36.0	35.2
0.015	18.3	16.5	21.3	21.3	8.2	8.3	28.7	28.3	29.2	28.6	30.7	30.1
0.011	15.3	13.7	17.7	17.7	6.8	6.9	23.6	23.2	24.3	23.8	25.4	25.0
0.007	12.3	11.0	14.3	14.3	5.5	5.6	19.0	18.7	19.7	19.3	20.3	20.2
0.006	11.2	10.0	12.9	12.9	5.0	5.1	17.3	17.0	17.9	17.6	18.3	18.3

	1st Cyc Feed		1st Cyc U/F		1st Cyc O/F		Mill 1 Prod		Mill 2 Prod		Mill Sump Prod		2nd Cyc U/F		2nd Cyc O/F	
	Exp	Bal	Exp	Bal	Exp	Bal	Exp	Bal	Exp	Bal	Exp	Bal	Exp	Bal	Exp	Bal
TPH Solids	342	341.8	110	109.8	232	232	82.33	82.33	128.3	128	442.3	442.3	101	100.5	342	341.8
Solids SG (t/m³)	3.2	3.2	3.2	3.2	3.2	3.2	3.2	3.2	3.2	3.2	3.2	3.2	3.2	3.2	3.2	3.2
TPH Water	617.1	605.2	90.73	101.1	502.2	504.1	65	64.99	100	99.71	668.7	668.8	62.43	63.63	600.1	605.2
% Solids	35.66	36.09	54.8	52.08	31.6	31.51	55.88	55.88	56.2	56.22	39.81	39.81	61.8	61.24	36.3	36.09
Pulp SG (t/m³)	1.29	1.294	1.527	1.488	1.248	1.248	1.543	1.543	1.548	1.548	1.335	1.334	1.637	1.628	1.296	1.294
Vol. Flow (m³/h)	743.7	731.8	131.5	141.8	588.1	590	95.5	95.48	147.5	147.1	832.5	832.7	99.84	100.9	726.8	731.8
% -75μm	57.34	57.11	23.7	23.72	73.37	73.43	61.46	61.46	64.7	64.71	66.8	68.62	39.05	38.5	78.19	77.5
P80 (mm)	0.138	0.139	0.223	0.223	0.0889	0.0877	0.11	0.11	0.107	0.107	0.101	0.098	0.141	0.143	0.0794	0.081
Size (mm)																
1.000	100	100	100	100	100	100	100	100	100	100	100	100	100	100	100	100
0.710	100	99.97	99.9	99.91	100	100	100	100	100	100	100	100	100	100	100	100
0.500	99.8	99.82	99.5	99.49	100	99.97	100	100	100	99.99	100	99.98	99.9	99.91	100	100
0.355	98.9	98.99	97	96.91	100	99.93	99.9	99.89	99.9	99.87	99.9	99.9	99.6	99.62	100	100
0.250	95.4	95.61	86.9	86.56	99.8	99.68	99	98.95	99.2	99.12	99.3	99.36	97.8	97.88	99.9	99.93
0.180	87.3	87.52	64.6	64.1	98.2	98.07	95.2	95.14	95.3	95.19	96.4	96.61	90.8	90.61	98.9	98.92
0.125	77.2	76.87	44.1	43.71	92	91.81	86.3	86.07	86.4	86.01	88.2	88.91	73.3	73.07	94.9	95.01
0.090	65.3	64.54	28.7	28.52	80.5	80.76	68.2	68.05	71.5	71.22	74.9	75.35	49.9	49.67	85.1	85.24
0.063	50.9	49.77	19.2	19.24	64.7	63.52	54.2	53.4	57.2	55.89	57.4	59.21	28.5	28.89	69.2	70.88
0.045	43	41.31	15.8	15.89	52.7	52.75	42.4	41.91	45.5	44.66	47.5	48.16	20.5	20.78	57.2	58.7
0.031	35.8	34.78	13.5	13.56	44.4	44.33	35.5	35.16	37.6	37.03	39.9	40.31	16.9	17.12	48.2	49.23
0.021	29.6	29.14	11.3	11.32	37.4	37.16	29.6	29.36	31	30.61	33.3	33.64	14	14.17	40.5	41.12
0.015	24.8	24.72	9.7	9.696	31.8	31.47	25.1	24.93	26.2	25.93	28.3	28.51	11.8	11.94	34.5	34.87
0.011	20.1	20.32	8	7.974	26.3	25.86	20.7	20.58	21.5	21.31	23.3	23.45	9.7	9.815	28.5	28.68
0.007	16	16.23	6.4	6.372	21	20.66	16.6	16.51	17.2	17.06	18.7	18.76	7.8	7.901	22.8	22.92
0.006	14.4	14.61	5.8	5.774	18.9	18.58	15	14.91	15.6	15.46	16.9	16.93	7.1	7.201	20.5	20.65

Table F-5 Waterval 2 stage closed test 2 circuit-Exp (calc.) and mass balanced

Appendix G. Sampling Points Modifications

This section shows some of the modifications made on the pipes where samples were collected. Pipes were modified to enable collection of representative samples. Figure G-2 shows the original arrangement of the stream that was feeding a mill before installing a modifying "box". The stream was modified to enable easy sampling as can be seen in Figure G-3.

A round pipe at an angle would result in a biased sample, because there would be segregation of the pulp across the pipe (Powell, 2004). The dense particles would segregate to the bottom of the pipe, and as a result particles would not have the same opportunities of making it in the sample cutter. If the sample cutter is moving across the pipe at constant speed, then there would be more slurry collected at the centre of the pipe than on right or left from the centre; see Figure G-1 for the illustration. Figure G-4 shows such a pipe and Figure G-5 shows the same pipe with a box designed to spread out the flow so that there would be no segregation.

There are other streams which were modified because the pipes were worn and were leaking on the sides. Figure G-8 and Figure G-9 shows an example of such a stream and the modified stream after the modification box was installed.

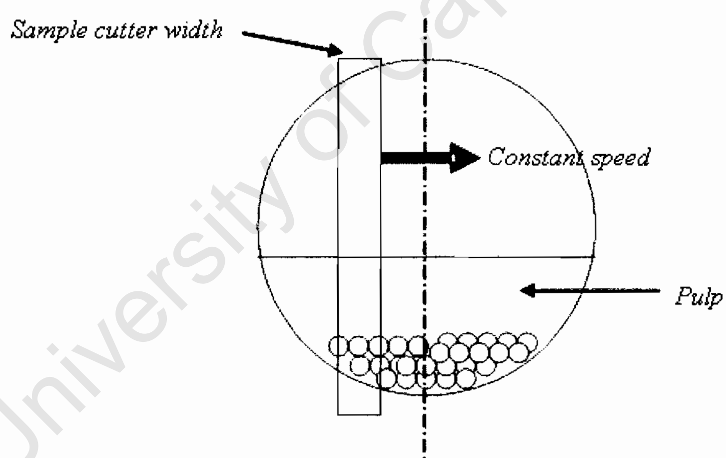

Figure G-1 Schematic illustration of segregation across a pipe (not vertical)

Figure G-2 Mill feed stream before modification

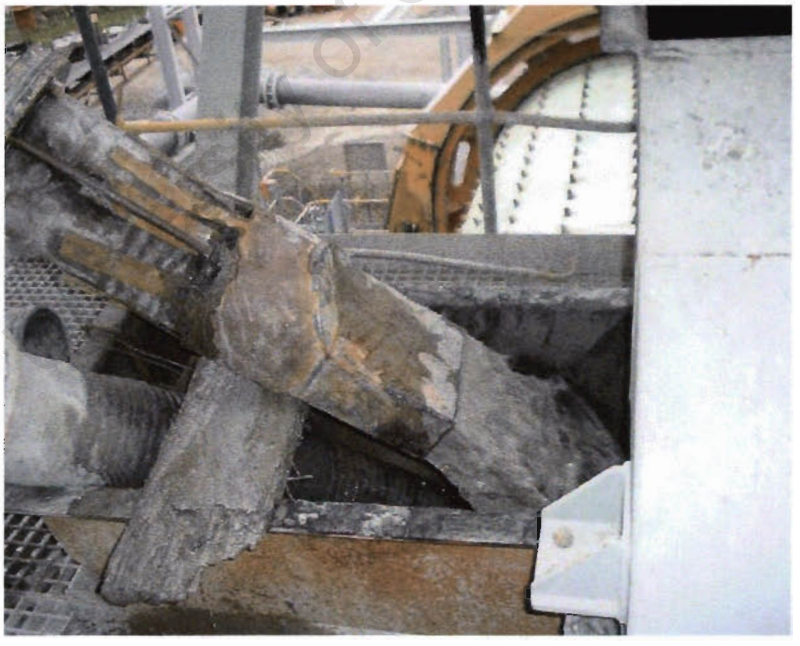

Figure G-3 Mill Feed Stream sampling point after modification

Figure G-4 A stream with possible segregation across a pipe

Figure G-5 The same stream as Figure G-4 with a modified discharge box

Figure G-6 A circuit product stream before modification - Lebowa

Figure G-7 A circuit product stream after modification - Lebowa

Figure G-8 Original setting of the first stage cyclone overflow - Lebowa

Figure G-9 Modified setting of the first stage cyclone overflow - Lebowa