FREDA REBECCA SAGMILLING CIRCUIT OPTIMIZATION

MSc (50/50) RESEARCH REPORT

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ABSTRACT

Simulation work was carried out to analyze how feeding a finer size distribution to a single stage SAG mill in closed circuit with hydro cyclones at Freda Rebecca Gold Mine would affect through put and specific energy consumption using an Excel based milling process simulator developed by Hinde, a comminution specialist who has worked for Mintek for many years. Other initiatives to increase through put were evaluated which included; the installation of a regrind mill (1000kW) to treat the cyclone underflow and the installation of a 100μm aperture fine screen to reduce the re-circulating load. ModSim software was used to study the later. It was observed that when the feed size distribution became finer there was an increase in through put, a decrease in specific energy consumption and cyclone overflow fineness. A 13.1% increase in throughput was observed when $F_{80}$was reduced from 240 mm (split ratio=1) to 45 mm (split ratio=0), the specific energy dropped from 20.66 kWh/t to 18.5 kWh/t and the cyclone overflow product size distribution fineness decreased from 78.33% passing 75 microns to 74%. The optimum through put was obtained at a split ratio of 0.2 which corresponds to an F_{80} of 70 mm. The throughput at this point is 88.17t/h (10.2% increment), specific energy consumption of 18.94kWh/t and the cyclone overflow product size distribution is 75 % passing 75 microns (plant process requirement). The payback period of the project at optimum through put is 1.2 years.

Installing a 1000kW regrind ball mill increased plant capacity by 56% through treating 29% of the cyclone underflow and reduced the SAG mill specific energy from 20.66 to 16 kWh/t and the investment has a payback period of 1 year and a month, in 10 years the project would have earned the company more than 15 million dollars of profit.

It was also observed from the plant survey that there was a high re circulating load of fine material to the SAG mill amounting to 51.2 t/h affecting mill capacity as this took space and limited the capacity for new feed. Using ModSim simulator, incorporation of a 100 micron screen on the cyclone underflow stream was simulated. The main purpose of the screen is to reduce the re circulating load by removal of fines that are deemed gold liberated enough and directed to the leaching circuit. The re-circulating load was reduced from a base case of 238 % to 175% and a 50% saving on energy required.

Key words: Simulation, Hinde excel simulator, ModSim, Model, SAG mill, Feed size distribution, Circulating load, Comminution.

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1. INTRODUCTION

Freda Rebecca Gold Mine is a business unit of Mwana Africa group located in Bindura, Zimbabwe as part of the global mining industry it is affected by fluctuations in gold price. Projections of mining low grade ore have motivated the need to take advantage of economies of scale by increasing milled tonnage in the processing plant thereby pushing upward production at a low cost per ounce. It has been observed that the rate limiting process when it comes to increasing throughput is the single stage Semi autogenous milling (SAG milling), this is to a large extent due to major ore hardness variations and coarse feed size distribution.

This work is aimed at optimizing the SAG mills feed size distribution as this is an important parameter that affects milling performance after ore competency. A lot of operations have recognized the optimization opportunity that can be taken advantage of by manipulating the feed size distribution to improve SAG milling efficiency. Hence attention has been paid to partial or full crushing, changes in blasting practices or selective screening to obtain most appropriate feed size. Various researchers and investigators have documented studies in this respect (Morrel S, 2001). Increase in throughput has been realized by simply optimizing the feed size distribution to the SAG mill. This work will look at the effect of removing fine material in the cyclone underflow through fine screening to reduce the re-circulating load. Possible installation of a ball mill to treat the cyclone underflow on milled tonnage is also explored.

Freda Rebecca consists of an underground mine and a processing plant. The run of mine is transported using load haul and dump trucks to the primary crusher pad where size reduction starts at a Telsmith jaw crusher (50”x60”) with a closed setting of 170 mm. Particles as large as 300 mm have been observed in the crusher product due to the nature of the rock. The crusher product is stockpiled and six vibro feeders feed 2-low aspect SAG mills (16x24ft) which operate in closed circuit with 750 mm diameter cyclones. The primary cyclone overflow is pumped to the leaching tanks through the Delkor linear screen and the dewatering clustered cyclones, the underflow goes back to the mill for further comminution whilst the other portion (approximately 30%) feeds the Knelson concentrators (KC) via a 4.5 mm screen and the KC tail goes to the mill.

Figure 1 below illustrates the flow sheet of the comminution circuit with the dewatering cyclone overflow (hydro cyclone cluster) being used as dilution water at 14% solids (stream 19):

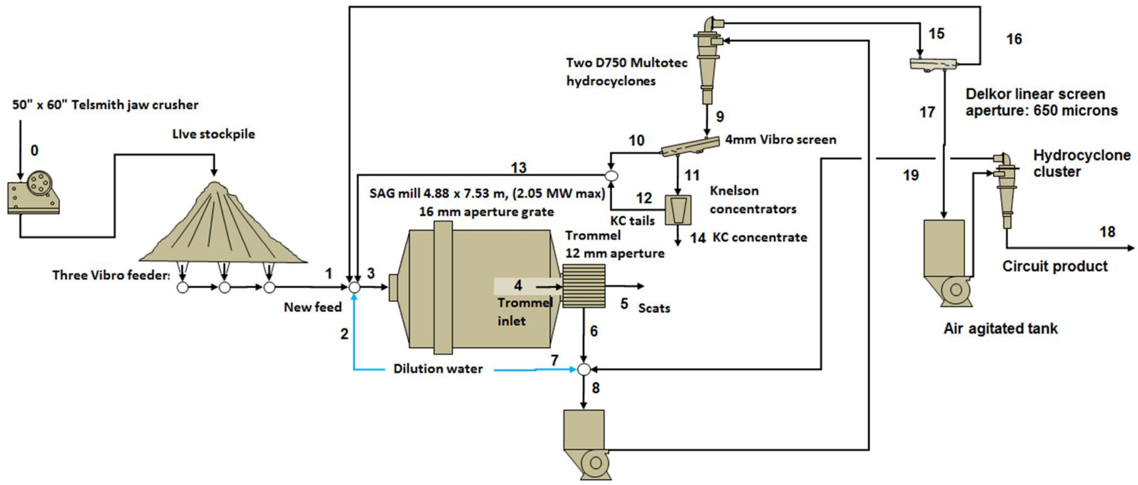

Figure. 1 Freda Rebecca Gold Mine flow sheet

The main research objective of this work is to investigate how SAG milling throughput can be improved. The main variables that are going to be considered are; feed size distribution and re circulating load control.

Software program developed by Adrian (Excel based) and the one by Minerals Technology International software (Modsim^{TM}) will be used for computer simulation in this study. Improvement options for the overall grinding performance will be considered, including possibility of installing a regrind mill to deal with mid size particles.

2. LITERATURE REVIEW

2.1. AUTOGENOUS (AG) AND SEMI AUTOGENOUS (SAG) MILL OPERATION

Semi autogenous grinding is one of the most used processes for comminution. The process involves feeding rock particles and water into the grinding mill containing a fraction of steel balls usually below 10% and running at a fraction of its critical speed. The slurry and ball charge adheres to the walls of the mill before cascading off at a given angle causing impact and fragmenting the ore (Nunez F, 2011).

There are different types of SAG mills; low aspect, medium aspect and the high aspect. The aspect ratio here refers to the ratio of the mill diameter to length. High aspect mills are typical in the Americas, where often diameter is twice the length, giving an aspect ratio of two. These are ideal for high throughputs and a coarse product that usually feeds a secondary grinding circuit for further reduction. Medium aspect mills are common in Australia with aspect ratio between 1.2 and 1.5 (Powel M, 2006). Low aspect mills are common in South Africa and Scandinavia. The length can be twice the diameter to give an aspect ratio of 0.5. The mill ensure a high residence time, resulting in a finer grind product. They are often operated as single stage mills that produce the final product as is the case with Freda Rebecca Gold Mine. To achieve this they are closed with classifiers (Powel M and Valery W, 2006).

Autogenous mills (AG mills) differ from SAG mills as they solely use larger rocks to grind the intermediate particles in the absence of secondary media, normally steel. In most cases AG mills are high aspect mills. The introduction and subsequent ascendancy of AG and SAG milling for comminution circuits has led to many economic advantages, from high processing capacity, energy efficiency, lower investment and maintenance cost (Nunez F and Silva D, 2011).

The major disadvantage is the sensitivity of these machines to process input variations. Foremost among these is ore competence and feed size distribution comes as close as second. This sensitivity is due to the reliance of AG/SAG mills on the feed ore to also act as grinding media. Thus there is a fine balance between having stable media that is available for comminution and grinding without too much difficulty. AG mills are the most sensitive in this respect with SAG mills being increasingly less sensitive as the ball charge is increased (Morrel S and Valery W, 2001).

2.2. EFFECT OF FEED SIZE DISTRIBUTION ON SAG MILLING

The qualitative effect of changes in particle size distributions (PSD) on the SAG mill performance can be addressed by using the grind curves. The operation of SAG mills is extremely sensitive to mill filling; from a qualitative point of view, for a given fill level and constant feed rate and speed, if the PSD becomes slightly coarser, then the fill level will increase. On the other hand, if the PSD becomes slightly finer then the fill level will decrease. Grindcurves provide information regarding the effect of changes in fill level on SAG mill power and throughput. Therefore, variables capable of explaining how changes in the PSD affect mill filling are required. They could be inferred based on the measurements delivered by visual sensors (Nunez F and Silva D, 2011).

AG and SAG mills respond differently to changes in feed size. It arises from the fact that in AG mills some large rocks are required to break intermediate sized ones. If these large rocks are not present in sufficient numbers then the intermediate sized ones are not broken at a sufficiently high rate, thus developing the so called critical size build up which results in limiting throughput (Morrel S and Valery W, 2011).

This is not to say that AG mill performance can be improved to infinity if feed size continues to rise. A balance needs to be struck between the number of coarse rocks and intermediate sized rocks ones. If too many coarse rocks are fed to the mill they will cause an imbalance and will themselves start to build up, resulting in throughput limitations. The same conditions apply to SAG milling. However, in this case the required balance between coarse and intermediate sized rocks is different. This is because the steel grinding balls in the SAG mill do the duty of the larger rocks. As a result, the more balls that are loaded into the SAG mill the fewer larger rocks are required. Hence, the general trend in SAG mills is that finer feeds tend to perform better than coarser feeds (Morrel S and Valery W, 2011).

Successful monitoring and manipulation of feed size starts with successful measurement. Taking a belt cut and sieving the sampled material is the most accurate method of determining the size distribution of the AG/SAG mill feed. However, fluctuations in feed size make such sizings valuable only if they can be done very frequently, which is impracticable in a production environment (Nunez F and Silva D, 2011).

Currently, the most widely accepted method for measuring ore size distribution is digital image analysis; including simple one or two dimensional methods, or sophisticated approaches such as multivariate and texture based image analysis. There are several commercial visual sensors currently operating in industrial mineral processing plants. Some of these are; SplitOnline, VisioRockand WipFrag. The incorporation of measurements delivered by visual sensors in control algorithms is still an open issue (Nunez F and Silva D, 2011).

WipFragis an image analysis system for sizing materials such as blasted or crushed rock. It is based on image processing techniques, and uses automatic algorithms to identify individual blocks. Then an outline net, using state of the art edge detection is created. Ore size distribution measurement is done in several stages; image acquisition, net generation, equivalent volume determination, and curve adjusting (Nunez F and Silva D, 2011).

Image acquisition is carried out using a camera installed over the conveyor belt in which the mineral is transported. The shutter speed is adjusted to obtain an image without slip (Nunez F and Silva D, 2011).

Net generation involves the identification of block edges; this is done in two stages. The first stage involves several conventional image processing techniques, including the use of thresholding and gradient operators. The operators detect the faint shadows between adjacent blocks, this works best on clean images with lightly textured rock surfaces. The second stage uses a number of reconstruction techniques to further delineate blocks that are only partly outlined during the first stage. These include both knowledge based and arbitrary reconstruction techniques, to complete the net (Nunez F and Silva D, 2011).

Finally, the identified blocks are approximated by an equivalent volume sphere, and then ore size distribution is estimated based on volume of the spheres. Curve adjusting is done to parameterize f the Rosin-Rammler distribution (Nunez F and Silva D, 2011).

In plant operation it is customary to monitor the passing percentage for a given size considered representative, regardless of the Rosin-Rammler distribution parameters, and to characterize the PSD based on this passing percentage.

A study done shows that there are several combinations which shows the Rosin-Rammler distribution with different parameters that would deliver the same passing percentage for a given size, yet with a completely different milling performance. Although the single size estimate is a useful indicator, it is dangerous to rely on it exclusively. This suggests that a better characterization of the feed size distribution should preferably use the Rosin- Rammler parameters rather than the passing percentage for a given size (Nunez F, 2011 and Morrel S, 2001).

2.3. PARTIAL CRUSHING: CASE STUDIES

The use of a partial secondary crushing circuit provides a flexible way to optimize existing SAG milling circuits limited by rock competency. It also enables the optimal processing of an ore through second hand equipment not initially selected for that ore (Putland B and Siddal B, 2004).

In Australia several examples of the retrofitting of secondary crushing circuits exist. The idea is to reduce feed size and hence increase SAG mill capacity. The most notable are Kidston Gold Mine, Mt Rawdon and St Ives Gold Mine. However circuits treating totally secondary crushed feed are often unstable and difficult to operate. They do not always provide an optimum circuit balance whereby the total installed power is utilized (Putland B and Siddal B, 2004).

By understanding the ore breakage characteristics and by utilizingcomminution circuit modelling, the benefits of SAG mill feed size manipulation can be readily assessed. At Mt Rawdon for example it was possible to show that the capacity could be raised from 2.2 Mtpa to 3.2 Mtpa by a combination of improved power efficiency from secondary crushing and optimized equipment performance. This necessitated increasing the power available to the SAG mill and reconfiguring the ball mill to deal with the grate-discharge more effectively. Manipulation of the proportion of secondary crushed feed allowed balancing of the duty between the SAG and ball mill. This was done by diverting primary crushed ore to a surge bin ahead of a secondary crusher, and allowing the bin to overflow into a bypass (Putland B and Siddal B, 2004).

The Mt Rawdon upgrade involved the installation of a secondary crushing circuit, new cyclones to cope with the increased flow, and modifications to the ball mill. The secondary crushing circuit was installed after the gyratory crusher and before the stockpile (Putland B and Siddal B, 2004).

For a competent ore such as Mt Rawdon's (18.3-20.7kwh/t), reducing the feed size by partial secondary crushing minimized the amount of coarse rock. This resulted in increased SAG mill capacity (Putland B and Siddal B, 2004).

By blending primary and secondary crushed ore it was possible to obtain the right amount of media to be presented to the mill, whilst lowering the energy demand. The circuit capacity was increased from ~270 tph to an excess of 395 tph, with 50 to 60% of the ore crushed. With an increased ball charge and load, an average 3.8 MW of the 4.2 MW available power was recorded on the SAG mill.

At St Ives Gold Mine JKSimMet was used as a simulation tool. In doing so, the throughput was increased from 2Mtpa to 3Mtpa. The adopted solution was the addition of a secondary crushing circuit and the removal of the scats crusher. Notwithstanding all the above operating the SAG mill at an F_{80} of 28 mm also resulted in a number of problems which include; high SAG liner maintenance, low SAG utilization due to frequent shutdowns for mill re-lines, high ball consumption rates and maintaining and operating a high cost secondary crushing circuit

Due to this fine feed size, the SAG mill is operated at a high ball charge levels to achieve target throughput rates. The optimum charge weight depends on ball charge, ore hardness and feed size. Any attempt to increase rock charge beyond 4-7% on top of the already existing ball charge level results in the SAG mill filling very quickly, dramatically reducing grinding rates. In this instance a partial or full mill grind-out is then required to reduce power draw and charge weight to acceptable levels; this represents a loss of throughput as well as loss of opportunity (Atasoy Y, 2001).

A study was proposed to optimize the liner life and feed size so as to reduce crushing and grinding costs. Original lifter bars had a 7 degree face angle and extra metal put around boltholes for extra strength. As a result, the original lifter bars had a “dog bone” shape, which did not help improve the throw of balls against the toe of charge. New lifters were designed with 25 degree face angle and smooth profile across the lifting face to help improve the throw of balls. Lifter heights on the feed and discharge head of the SAG mill were increased from 80 mm to 150mm. The seating arrangement between the shell plates and lifter bars was also re-designed with the objective to reduce bolt breakages and improve sealing for leakage prevention (Atasoy Y, 2001).

Preliminary simulations carried out have shown that the maximum throughput rates, which can be achieved with a coarse feed may be less than those achieved with a fine feed. And work is still underway to optimize the comminution circuit (Barrios G, 2001)

Cadia Hill SAG mill circuit was also commissioned following a research project and development exercise in July 1998. This was response to the fact that the SAG mill failed to meet expected throughput at the design operating conditions of ball and rock charge, mill speed and power draw. The process modifications were made with the aim of maintaining high availability (Hart S and Valery W, 2001).

SAG mill feed size distribution was recognized as being critical to the performance of the SAG mill circuit. The Split on-line image analysis system was installed in the Cadia Hill SAG mill circuit in June 1999 to confirm observations made since start up. A correlation between SAG mill feed size F_{80}, throughput and specific power consumption was developed from plant operating data. The CDI (conveyor dynamic incorporated) confirmed the magnitude of the relationship, filtering the data for the influence of all other factors, during an analysis of 12 months of process data using the MillStat program. A reduction in SAG mill feed size $F_{80}$from 100 to 70mm in Cadia Hill was found to increase throughput by 10 to 15%. Model simulations carried out by the Julius Kruttschnitt Mineral Research Centre (JKMRC) also confirmed the feed size relationship. Limited manipulation of feed size could be obtained by adjustment of primary crusher gap, as the closed side setting was operated at approximately 110mm.

2.4. MODELLING AND SIMULATION

Simulation is the process of designing computerized model of a system, for the purpose of understanding its behaviors and developing strategies to control the operation. Simulation is now an effective tool for mineral processing plants (Merks JW, 1991).

To maximize the efficiency and cost effectiveness of mines, mineral processing plants and metallurgical plants, the design team has to consider many operating parameters and design criteria These tools can evaluate performance for a variety of process concepts, design criteria such as ore grades, and throughput tonnages (Nikkhah K).

A tool such as IDEAS can simulate both steady and dynamic states, allowing the user to create a dynamic representation of the process with equipment such as pumps, valves, pipes and tanks. Scheduled and random stoppages, which influence availability and overall production performance, can be included to obtain a more realistic picture (Nikkhah K).

JKSimMet, MODSIM, HINDE's EXCEL BASED, METSIM, USIM PAC, are some of the simulation software used for designing and optimization of mineral processing circuits. With Hinde's excel based only used for steady state processes whilst the rest can be used for both steady and dynamic processes.

Different simulators are based on milling kinetics models that have been developed over the last half century. These are mostly centered on the selection function and the breakage factors. The selection function can be defined as the breakage rates of different particle size classes whereas the breakage function describe the distribution of fragments after each breakage event.

2.4.1. SELECTION AND BREAKAGE FUNCTION

The goal of a comminution circuit is to grind particles to their liberation size, so that the valuable minerals are completely broken free from the gangue minerals. The optimal design and control of comminution circuits require a mathematical model capable of depicting the size reduction behaviors of every size fraction (Fuerstenan, D. W, 2003).

A rigorous mathematical approach to comminution was published in 1954 but the catalyst that led to worldwide utilization of the population balance model (batch grinding model) to the analysis of comminution in tumbling mills was perhaps the paper presented by Gardner and Austin at the First European Comminution Symposium in 1962 (Fuerstenan, D. W, 2003).The selection function and breakage distribution functions can be determined directly in a laboratory scale or pilot scale mill operated under batch grinding conditions.

In population balance model grinding is treated as a rate process and breakage of the given size fraction usually follows the first order law. Thus following first order grinding hypothesis the breakage rate of material can be expressed as:

$\frac{dWp_1}{dt} = -S_1 Wp_1$……………….(1)

S_1---selection function/rate of breakage of particles in size class 1

P_1---Fraction of particle in class one

W---total weight of particles

Rate of grinding is proportional to the mass of particle in that size class as shown in the equation above. Solving equation 1 in conjunction with experimental data, the rate of breakage of particles of class 1 can be determined.

$B_{ij} = \sum_{k=n}^{i} b_{k,j}$……………………………………..(2)

The concept of selection function and breakage distribution function makes it possible to express the population balance of batch grinding and predict the product size distribution:

$\frac{dWp_i(t)}{dt} = \sum_{n}^{n-1} b_{ij} * S_j Wp_j - S_i Wp_i(t)$……….. (3) on condition that $n \geq i \geq j+1$

The specific rate of breakage function can be expressed as follows Austin et al.(1984):

$$S_i = \frac{a x_i^{\alpha}}{1+\left(\frac{x_i}{\mu}\right)^{\Lambda}} \quad \text{.................................(4)}$$

Where x_i is the upper limit of the particle size and a, α, μ and Λ are the model parameters that depend on the properties of the material and grinding conditions. This equation allows interpolation and extrapolation to obtain estimates of selection function values for all size intervals involved.

The weight fraction of the material broken from the size interval j which appears in the size interval i before re-breakage of the fragments occurs is defined as the primary breakage distribution function, $b_{i,j}$. It is convenient to represent this function in the cumulative form:

$B_{i,j}$ can be fitted to an empirical function proposed (Austin, 1984):

$$B_{ij}(x_i, x_j) = \emptyset(\frac{x_i}{x_i+1})^{\gamma} + (1-\emptyset_j)(\frac{x_i}{x_i+1})^{\beta} \quad \text{.............. (5)}$$

Where $\emptyset_j$, γ and β are the model parameters that depend on the properties of the material.

The breakage function is obtained by running grinding tests in the laboratory mill and can be scaled up to an industrial scale to describe the behavior of an industrial mill. A model for SAG and AG milling include abrasion in addition to the normal breakage processes which follows a different set of laws and self-breakage of big rocks that are capable of breaking on their own by the impact of their own fall, in a stream of tumbling rock and balls (Farzanegan A and Khodadadi, 2007). The breakage rates increase with increased lump size due to the increased impact force (Austin L.G and Percy F, 1987).

2.4.2.PLITT'S MODEL FOR HYDROCYCLONES

The design, selection and optimization of a cyclone can also be done by means of mathematical models. A model is an idealized representation of a physical reality in the form of a set of equations; it is used to predict the output characteristics in terms of input variables without doing an experiment. This saves a lot of time and resources associated with carrying experiments at pilots plants to predict the behaviour of a system (Nageswararao K, 2004).

They are different cyclone models but the most widely used is the Plitt model. In the original reference, Plitt (1976) offered two forms of the d_{50c} equation, one with and the other without feed size effects. The Plitt model in its current form as revised by Flintoff et al. (1987) has no dependence for feed size characteristics in any of the equations and is given below (Nageswararao K, 2004):

$$Ruc = 1 - \exp(-0.693(\frac{l}{l50c})^{m} \quad \ldots\ldots\ldots\ldots\ldots(6)$$

$$l_{50c} = \frac{50*5D_c^{0.46}*D_i^{0.6}*D_O^{1.12}\exp(0.063\emptyset)}{D_u^{0.71}*h^{0.38}*Q^{0.45}*(\rho_s-\rho)^{0.5}}\ldots\ldots\ldots\ldots\ldots(7)$$

$$S = \frac{1.9(\frac{Du}{Do})^{3.31}*h^{0.54}*(D_u^2-D_O^2)+\exp(0.054\emptyset)}{H^{0.24}-D_c^{1.11}}\ldots\ldots\ldots\ldots(8)$$

$$P = \frac{1.88*Q^{1.78}*\exp(0.005\emptyset)}{D_O^{0.37}*D_i^{0.94}*h^{0.28}*(D_u^2+D_O^2)^{0.87}}\ldots\ldots\ldots\ldots\ldots\ldots\ldots(9)$$

Q Cyclone throughput, l_{50} Cut size S Volumetric split, m Sharpness of classification. As design independent variables Plitt used: D_c, D_o, D_u, D_i,h; Diameter of the cyclone, vortex finder, spigot and inlet diameter, free vortex height.

With the ability to calculate the parameters mentioned above for a given set of conditions, it is possible to determine the complete mass balance together with the size distributions of the products of the cyclone.

Plitt (1976) took into account that the feed solids percentage affects the pulp viscosity, which in turn affects d_{50c} and also hinders settling and causes crowding. When proposing the equations for pressure drop, P (to design pumping system) and flow split, S (water balance across the cyclone), he used 297 sets of data, including the tests run with water only. As d_{50c} values were not available for all the data sets, only 179 of the sets were used for the d_{50c} equation. Only the 162 tests with sufficient data points above and below d_{50c} to form a complete classification curve were used for the equation for m (Nageswararao K, 2004).

2.4.3. POWER MODEL

The net power demand of a SAG mill may be well estimated by the Hogg and Fuerstenau model to represent the independent contribution of each component of the mill charge (balls, rocks and slurry) to the total net power draw of the mill (Hogg and Fuerstenau, 1972):

$$P_{net} = \eta P_{gross} = 0.238 * D^{3.5} * \left(L/_D\right) * N_{cri} * \rho_{ap} * (J - 1.065J^2) * sin\alpha \text{------------} (10)$$

Where:

P_{gross}= Gross power draw of the mill (kW) = P_{net} / η.

η = Electrical and power transmission efficiency.

D = Effective mill diameter, ft.

L = Effective mill length, ft.

N_c= Tumbling speed; expressed as a fraction of the critical centrifugation speed:

N_{cri}= $76.6/D^{0.5}$.

J= Apparent volumetric fractional mill filling, (Including the balls, the rocks and the interstitial voids in between such balls and rocks).

α= Charge lifting angle (defines the dynamic positioning of the center of gravity of the mill load (the 'kidney') with respect to the vertical direction. Typically in the range of 40° to 45°

ρ_{ap}= Apparent density of the charge (ton/m^3), which may be evaluated on the basis of the indicated charge components (balls, rocks and interstitial slurry):

$$\rho_{ap} = ((1 - f_v) * \rho_b * J_b + (1 - f_v) * \rho_m * (J - J_b) + \rho_b * J_b * f_v * J))/J \text{..........} (11)$$

f_v=Volume fraction of interstitial voids in between the balls (typically assumed to be 40% of the volume apparently occupied by the balls).

J_b= Apparent balls filling (including balls and the interstitial voids in between such balls).

J_p= Interstitial slurry filling, corresponding to the fraction of the available interstitial voids (in between the balls and rocks charge) actually occupied by the slurry of finer particles.

ρ_m= Mineral particle density, ton/m^3.

ρ_b= Slurry density (ton/m^3) directly related to the weight % solids of the slurry (f_s) by:

$1/(\frac{f_s}{\rho_m} + \ (1 - f_{vs}))$

In this formulation, the contribution to the net mill power by the balls in the charge becomes:

$P_b = ((1 - f_v) * \rho_b * J_b / \rho_b \ * J) * \eta P_{gross}$......... (12)

Similarly, the contribution to the net mill power by the rocks in the charge becomes:

$P_r = ((1 - f_v) * \rho_m * (J - J_b) / \rho_{ap} \ * J) * \eta P_{gross}$......... (13)

And finally, the contribution of the slurry in the charge becomes:

$P_s = ((f_v) * \rho_p * J_p * J / \rho_{ap} \ * J) * \eta P_{gross}$......... (14)

2.5. FINE SCREENING

Although hydrocyclones are widely used for sizing very fine particles, screens would be theoretically superior. But a considerably large screening surface area is required to separate fine particle sizes leading to high maintenance costs.

The concept of fine screening has been thought to lead to low capacity, high media consumption rates and blinding. This is no longer true, fine screening is now more practical than ever with high capacity Derrick stacksizer screening machine fitted with Derrick's unique, long life non blinding polyurethane screen surfaces (Ultrafine screen).

Since its inception in 2001, about 350 stacksizers have been produced by the mining industry worldwide and the majority, about 200 are used in iron ore applications (ultrafine). Derrick cooperation introduced an improved classification in grinding circuits which increase mill capacity and production rate by reducing the circulating load and reduce power consumption per ton .

Most Peruvian base metal mines have replaced hydrocyclones which are less efficient, i.e efficiency of 45%-65%. The end result of closing grinding circuits with this relatively low separation efficient device is that most Peruvian mines have operated with circulating loads in excess of 200%. On the other hand stacksizers operate with separation efficiencies in the range of 85% to 92% and separate according to size not density. The other problem with hydro cyclones is that heavier liberated minerals report to the underflow then back to the grinding mill, while lighter middlings particles report to overflow stream and on for concentration without the liberation of the desired mineral.

At the Sociedad Mineral El Brocal Lead/Zinc mine in Peru, after installing fine screens, the circulating load was reduced from 400% to 100%. This resulted in two regrind mills treating the primary rod mills discharge shut down due to lack of feed. This then created huge power savings along with maintenance cost elimination. Eventually throughput increased by 10% from the baseline and in a similar way production is increased by 30% (Barrios G, 2001).

In this work it is proposed to treat the cyclone underflow instead of replacing the hydrocyclones and processing the whole mill discharge stream. This is expected to cut down on the capital investment cost as only a smaller screen will be purchased. The feasibility of the proposed solution is also assessed as it has been observed that the Freda Rebecca ore is not grind sensitive to leaching. Here the objective of the screens is to reduce the recirculating load of fine material back to the mill rather than to reduce coarse material in the cyclone overflow.

3. PLANT SURVEY AND CIRCUIT SIMULATION WORK

3.1. PLANT SURVEY

A plant survey was carried out to determine the particle size distributions of important streams such as mill feed (1), mill discharge (6), cyclone feed (8), overflow (15) and under flow size distributions (9) (see Figure 1).

3.1.1. SAMPLING THE CYCLONE OVERFLOW AND UNDERFLOW

Cross sectional sampling was done for the cyclone underflow (stream 9) and overflow (stream 15). A sample cutter was passed across the entire stream at a constant speed using a sufficiently large sample cutter (more than three times the largest particle) with splashing avoided while cutting across. A composite sample was prepared from 2 hourly sample cuts over a period of 5 days and a particle size distribution was then determined by test sieving. The results were also used to determine the amount of fine material in the re-circulating load. The results obtained can be seen in Appendix D.

3.1.2. SAMPLING THE CYCLONE FEED (MILL DISCHARGE)

Sampling point for cyclone feed was not accessible, so it was considered to be the same size distribution as the mill discharge (stream 6). The sampling for the mill discharge was done by cutting across the trommel screen underflow with a sample cutter at a constant speed as explained for cyclone overflow and underflow. The mill discharge was assumed to approximate the cyclone feed although it is different due to the solids from the dewatering cyclone overflow used as dilution water.

3.1.3. SAMPLING OF THE MILL FEED

Mill feed (stream 1) was sampled by taking a 5 metre belt cut every 2 hours for a day.

These size distributions were used to establish a base case on the Hinde simulator and to ensure it was a physical representation of Freda Rebecca Plant.

3.2.THE EXCEL BASED MODEL SIMULATOR BY ADRIAN HINDE

Excel can be set up to run *VBA* code, do iterative calculations, and use *Solver* and other add-ins for model parameters. Consideration is then given to for instance the task of fitting equations to measured particle size distributions based on the Rosin-Rammler and Logistic probability distributions. Non-linear regression techniques are explained to show how the parameters of these equations can be estimated by using *Solver* to minimize a weighted sum of squares of the differences between the measured and calculated particle size distributions. It is also shown how the equations used to model the size distributions can be set up as *VBA* user-defined functions. Such functions are essential to the development of computer simulation models using Excel.

Consideration is then given to the mass balance smoothing of data generated from sampling surveys of grinding circuits operated at steady state. Mass balance smoothing is all about making minimal changes to measured data to ensure that all mass flows are self-consistent. This is done by making sure that for any given species of interest, what goes into each circuit unit must provide a perfect balance with what comes out of the unit.

Most grinding circuits use mills operating in closed circuit with screens or hydrocyclones. This can make the calculation of mass balances around the circuit a challenging task not amenable to simple analytical methods, especially when the model equations for the mills and classifying devices have nonlinear structures. In general, mass balances must be calculated using numerical methods involving circular or iterative calculations where a formula refers back to its own cell, either directly or indirectly. Fortunately, *Excel is very easy to set up to do iterative calculations*. The underlying models used in the Excel based simulator developed by Adrian are presented in the following sections.

3.2.1. BASIC MODEL FOR GRINDING MILLS

The simplest phenomenological model for a continuous grinding mill is one that assumes the content of the mill is fully mixed. This implies that the size distribution of the milled product formed P_i (mass fraction less than size x_i) is the same as that of the mill contentsAnother simplifying assumption is that the mass of the ore in the mill M [t] remains constant so that the total solids discharge rate [t/h] of the ore is always equal to the total inlet feed rate, F [t/h]. Similarly, the discharge flow rate of water is equal to the inlet water flow rate.

To quantify breakage behaviour it is necessary to invoke the concept of a cumulative specific breakage rate function K_i $[h]^{-1}$, which gives the fractional rate per unit mass that material greater than size x_i in the mill breaks to below this size. The accumulation or rate of change of the mass of material finer than x_i inside the mill is then given by the difference in the mass flow rates of material finer than x_i in the inlet and discharge streams plus the mass consumption rate of material coarser than x_i inside the mill than breaks to sizes less than x_i. This mass balance can be expressed in terms of a differential equation:

$$accumulation = flow\,in - flow\,out + consumption$$

$$\frac{d(MP_i)}{dt} = FF_i - FP_i + MK_i(1-P_i) \quad \text{.......(15)}$$

Where F_i is the inlet size distribution (mass fraction less than x_i in the inlet stream).

At steady state, the left hand side of equation 15 is zero.

$$P_i = \frac{F_i + K_i\,(M/F)}{1 + K_i\,(M/F)} \quad \text{.......(16)}$$

Since M/F is equal to the mean residence τ, Equation 16 simplifies to:

$$P_i = \frac{F_i + \tau K_i}{1 + \tau K_i} \quad \text{.......(17)}$$

It follows from equation 17 that for zero residence time, the product size distribution is the same as the feed size distribution, as expected. It is also evident that as the residence time increases, P_i approaches a value of unity for all size classes. So the structure of Equation 17 is physically sensible for extreme values of the residence time.

Although it is possible to express the product size distribution as a function of time, it is more convenient to express the product size distribution as a function of net specific energy input. This the leads to the definition of an energy-based breakage rate function: $K_i^E = K_i M / Pnet$ where $Pnet$ [kW] is the net power consumed by the mill. Equation 15 can then be rewritten as:

$$\frac{d(MP_i)}{dt} = FF_i - FP_i + MK_i^E\left(\frac{Pnet}{M}\right)(1-P_i) \quad \text{.......(18)}$$

And at steady state it simplifies to:

$$P_i = \frac{F_i + \xi K_i^E}{1 + \xi K_i^E} \quad \text{.......(19)}$$

where $\xi = P_{net}/F$ is the specific energy input.

The energy-based breakage rate function can be defined as the fractional amount of material coarser than size x_i in the mill that breaks to below this size per unit specific energy input. The units for the energy-based specific breakage rates can therefore be expressed as $[\mathrm{kWh/t}]^{-1}$. The attractive feature of using Equation 19 is that the energy based breakage rate function is invariant to scale-up. In other words, the specific energy required to achieve a given grind in a pilot mill closely approximates that required for a production scale mill.

The fully mixed model is best suited for pancake shaped mills with diameters greater than their lengths. However, a single reactor model can be forced to provide a reasonable fit to most tumbling mill (AG/SAG, rod and ball) data generated from plant surveys.

The function K_i^E can be calculated directly from Equations 15 and 16 in terms of the measured feed and product size distributions, provided values of P_i are less than unity. For fully mixed conditions, the cumulative specific rate of breakage is given by:

$$K_i^E = \frac{P_i - F_i}{\xi(1 - P_i)} \quad \text{(20)}$$

It should be evident that if $P_i = 1$, values of K_i^E are indeterminate. To help get around this problem, it is convenient to express K_i^E in terms of an equation. A commonly used expression is a third-order logarithmic polynomial function of particle size:

$$K_i^E = K(\exp(c1\ln(x_i) + c2(\ln(x_i))^2 + c3(\ln(x_i))^3)) \quad \text{(21)}$$

Where c_1, c_2, c_3 and K parameters that can be determined by regression. Once the parameters of the model have been identified from plant data, it is possible to incorporate the model into a simulator and guide the design and optimization of a production circuit treating any given tonnage. It is usually possible to get a good fit to test data using only a first order or second order polynomial from Equation 21. This is because, a mill model with only two or three parameters is usually adequate for many applications involving conventional tumbling ball mills and even other types of comminution equipment.

The code also allows for a mill simulated as three fully mixed reactors in series or as a mill with a plug flow residence time distribution. In the latter case all material passing through the mill is assumed to stay in the mill for the same time interval.

It is important to appreciate that the plug flow model can be used to identify parameters of the breakage rate function from simple laboratory batch tests where the size distributions can be measured for different specific energy inputs.

3.2.2. BASIC MODEL FOR SCREENS AND HYDROCYCLONES

Screens and hydrocyclones are size classifying devices used to separate a feed stream into coarse and fine components. The performance of screens and hydro cyclones is determined by the partition function R_i. It gives the fractional recoveries to the coarse stream of feed particles in a given size class i. Because size distributions are usually measured using sieves with mesh sizes conforming to a geometric progression, the representative size $\hat{x}_i$ of particles in a given size class is usually taken as a geometric mean of the upper and lower limits of the size class ($\hat{x}_i = \sqrt{x_i x_{i+1}}$; $with\ i > n$). An arithmetic mean is usually used as a representative size of particles in the sink size class. In this case ($\hat{x}_i = x_n / 2$; $with\ i = n$).

Partition functions for both screens and hydrocyclones usually take on the form:

$$R_i(\hat{x}_i) = r_f(\hat{x}_i) + (1 - r_f(\hat{x}_i))R_i^c(\hat{x}_i) \quad (22)$$

where $r_f(\hat{x}_i)$ allows for the fact that a portion of the feed can bypass the normal classification process as a result of water entrainment or due to the adherence of fine particles to coarser particles. This bypass fraction is either constant or decreases monotonically with increase in particle size. In most cases, the bypass fractions can be assumed to be a constant for all size classes. When the bypass is not constant it can usually be mathematically by:

$$r_f(\hat{x}_i) = \frac{r_0}{1 + (\hat{x}_i / \delta)^\eta} \quad (23)$$

Where r_0, δ, and η are constant parameters. $R_i^c(\hat{x}_i)$ is the partition function for normal classification, after correcting for the effects of bypass.

It turns out that the partition functions for screens and hydrocyclones can be represented by equations very similar to those used for particle size distributions. For a screen, one would expect the partition function to have values of unity for particles coarser than the effective mesh size of the screen. It is then appropriate to use a truncated form of the Rosin-Rammler distribution:

$$R_i = r_f + (1 - r_f)\frac{1 - \exp(-a(\hat{x}_i / d_{mesh})^m)}{1 - \exp(-a)} \quad ; for\ \hat{x}_i \le d_{mesh} \qquad (24)$$

$$R_i = 1 \,;\, for\ \hat{x}_i > d_{mesh}$$

Where d_{mesh} is the effective mesh size of the screen while a and m are model parameters.

Commonly used equations for hydrocyclones are based on a Rosin-Rammler distribution (Equation 25) or a logistic distribution (Equation 26):

$$R_i = r_f + (1 - r_f)(1 - \exp\left(1 - \exp(-0.6931(\hat{x}_i / d_{50c})^m\right) \qquad (25)$$

$$R_i = r_f + \frac{1 - r_f}{1 + \left(\dfrac{\hat{x}_i}{d_{50c}}\right)^{-\lambda}} \qquad (26)$$

The parameter d_{50c} is the cut-size for the hydrocyclone, after correcting for the effects of bypass. The parameters m and λ are measures of the sharpness of cut. The partition function for a screen is usually much sharper than that achievable with a hydrocyclone for the same cut size, especially at the coarser sizes.

It is also pertinent to point out that for the same cut-size and slope at the cut-size, the plot of the Logistic function is much flatter than that of the Rosin-Rammler function at sizes above the cut-size as indicated in figure 2:

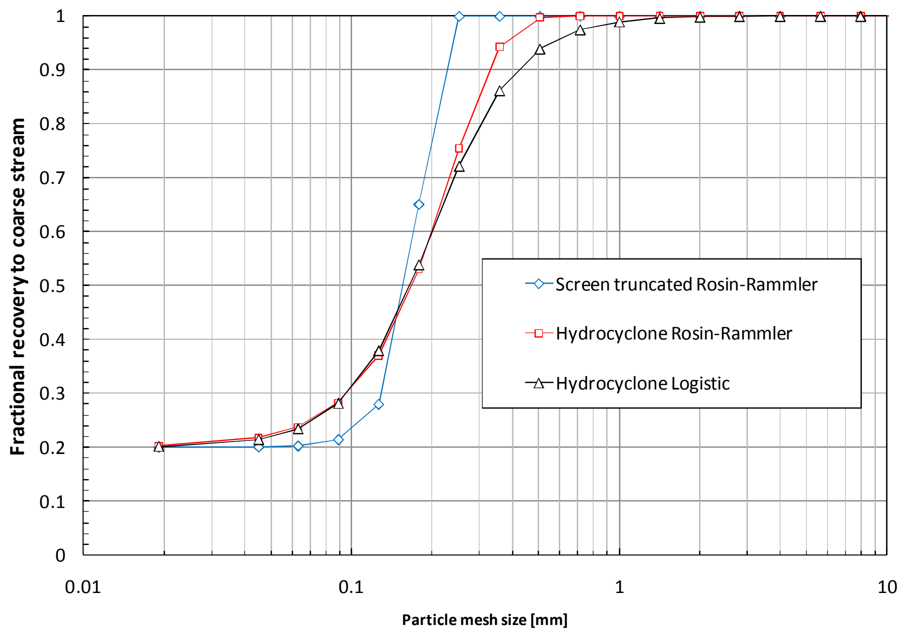

Figure.2 Partition functions for screen and hydrocyclones

Although the performance of a hydrocyclone can be quantified in terms of a partition function involving only three parameters, these parameters can vary with its geometry and operating conditions. It follows that the ideal hydrocyclone model should involve equations relating the parameters to the dimensions of the hydrocyclone, the feed volumetric flow rate, the feed pulp density, and the feed size distribution. These relationships can be quite complex.

In this work only simple model is used to calculate the hydrocyclone parameters. The model is based on the "crowding" theory originally developed by Fahlstrom in 1963 who provided evidence that except for operations with low feed pulp densities, the cut-size is primarily a function of the underflow orifice or spigot diameter and the size analysis of the feed. Moreover, the underflow pulp density remains essentially constant over a typical range of operating conditions (Hinde A, 2011).

According to Arterburn in 1982 an underflow of 50% to 53% solids by volume is typical for primary grinding circuits, whereas an underflow density of 40% to 45% solids by volume is normally for regrind circuits. Arterburn also asserts that the underflow volumetric flow rate asymptotes to a constant value determined mainly by the diameter of the spigot (Hinde A, 2011).

These empirical observations, coupled with the known structure of the partition function, can serve as the basis for a useful phenomenological model for hydro cyclones. Unfortunately, the model does not allow for the onset of unstable behaviour when the capacity limit of the spigot is reached and roping occurs. Under these conditions, the vortex air core disappears and the pulp in the vicinity of the spigot becomes overcrowded with solids, resulting in a drastically reduced and erratic underflow flow rate. However, it is generally acknowledged that hydrocyclones operate best close to the roping state that is at low discharge flare angles when the discharge volumetric flow rate is close to its maximum value.

3.2.3. SUMMARY OF THE EXCEL BASED SIMULATOR

- The simulator considers cumulative breakage rate function, which is proportional to the specific power.
- It works with a concept of a specific discharge rate function which is proportional the cross sectional area of the mill multiplied by the fractional open area of the grate.
- Functions can be estimated directly from plant data and fitted to equations with parameters obtained by minimising the sum of squares of the difference between the measured function values and the fitted equations.
- Allowance made for changes in the breakage rate function with changes in ball load.
- The result is a universal SAG model that can be fitted to production plant data with a single calibration parameter for the breakage rate function as well as for the discharge rate function. The model can be fine-tuned if the circuit can be controlled to run over a range of different operating conditions by adjusting some of the other model parameters to reconcile any differences in the plant performance. Although the model may not provide complete accuracy of predictions, it is certainly helpful in guiding the optimisation process on a rational basis.
- The simulator uses a set of simple model equations applicable to where crushing particle size distribution with top size is controlled by gap setting.
- Mathematical algorithms is used in the Excel workbook for the base case scenario iterations

- Within iterations where the cycle around the circuit unit by unit and at the same time adjust the new feed rate to maintain the mill charge filling to a given set-point (basically a steady state model with a dynamic component).
- *Solver* is used to satisfy pulp density constraints in the mill discharge, classifier oversize streams and final product.

The VBA codes are not be included in the report for intellectual property reasons. Only the information is extracted from Dr Hinde dummies guide is presented (Hinde A, 2011).

3.3. MODSIM SIMULATOR

The Modular Simulator for Mineral Processing Plants (Modsim^{TM}) is a basic and easy to use simulator. It calculates a detailed mass balance for any ore dressing plant. The mass balance include; total flow rates of water and solids, the particle size distribution of the solid phase, the distribution of particle composition and average assay of the solid phase. The ore dressing unit operations include the size reduction (crushing and grinding), size and solid-liquid separation.

Modsim^{TM} is a steady state simulator and is therefore not designed to simulate dynamic conditions. It is also not suitable for the design and simulation of process control systems. Modsim is unique in that it can simulate the liberation of minerals during comminution operations and calculate detailed mass balance for any ore dressing plant (King R.P, 2001).

The SAG mill is modelled in Modsim^{TM} using a population balance framework that includes attrition and wear as developed by Austin and Hoyer in 1985. Three distinct breakage processes are modelled: surface attrition, impact breakage and self breakage. The rate of self breakage is modelled using the variation of particle fracture energy and the consequence breakage probability with size. The mills are assumed to be perfectly mixed with post classification at the grate. The load in the mill is calculated from the mill dimensions and the average residence time calculated as the ratio of the load to the throughput. The power drawn by the mill is determined using formulas of Austin (1984) and Morrell (1996). This model permits the use of a pseudo stream from the mill to carry the size distribution of the mill load. Water can be added directly to the mill feed at a prescribed rate or the simulator will calculate the water addition rate that is required to achieve a specified solid in the mill discharge. The model parameters were estimated to closely match actual plant performance (King R.P, 2001).

4. EXCEL-BASED SIMULATION RESULTS

4.1. EFFECT OF PRE CRUSHING ON SINGLE STAGE SAG MILL THROUGHPUT

The results discussed below show the effect of splitting ratio to the cone crusher on the mill feed size distribution and subsequent SAG mill tonnage and specific energy consumption.

4.1.1. PRIMARY AND SECONDARY CRUSHING CIRCUIT

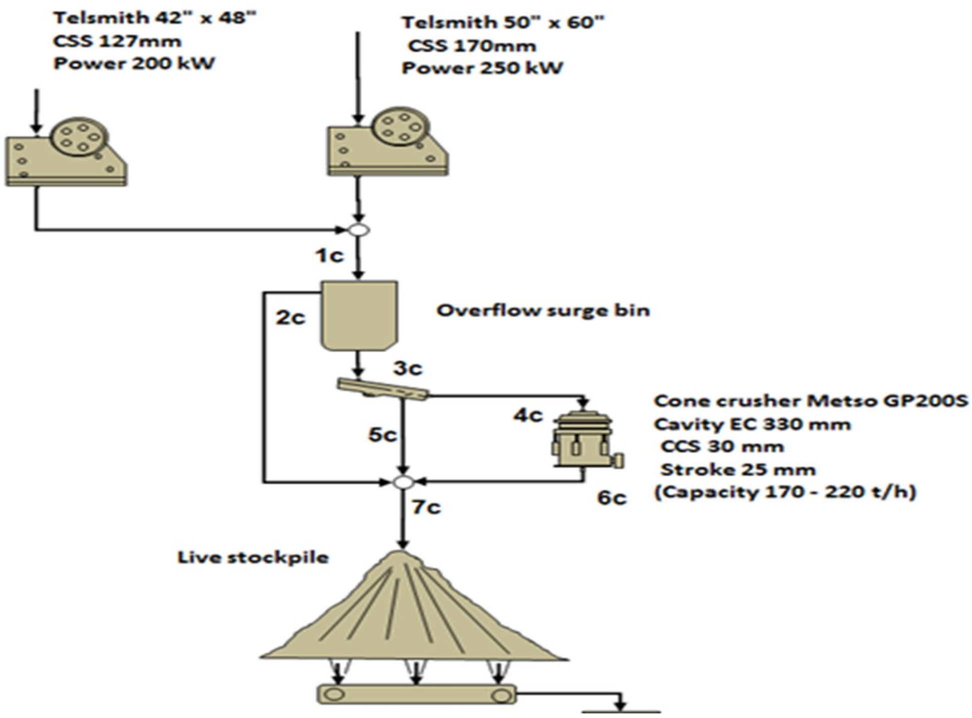

Figure. 3 Proposed pre crush circuit

Crushers

Tables for respective crushers and with the corresponding cusher gap (as shown in figure 3) use to obtain the product size distribution with a feed size top size of 900 cm.

Screen specifications

Aperture size...60mm

Screen efficiency……97%

Figure. 4 show the proposed pre crushing circuit, the ROM is crushed by the two existing jaw crushers which can operate simultaneously or individually. The portion of their products at different split ratio is passed through a 60mmaperturesize screen whereby the screen oversize is conveyed to the cone crusher and its product combines with screen undersize to the stockpile. This allows the SAG mill to be fed with different size distributions depending on the split ratio. The particle size distribution of the mill feed (stockpile) is shown in the diagram below at different split ratios:

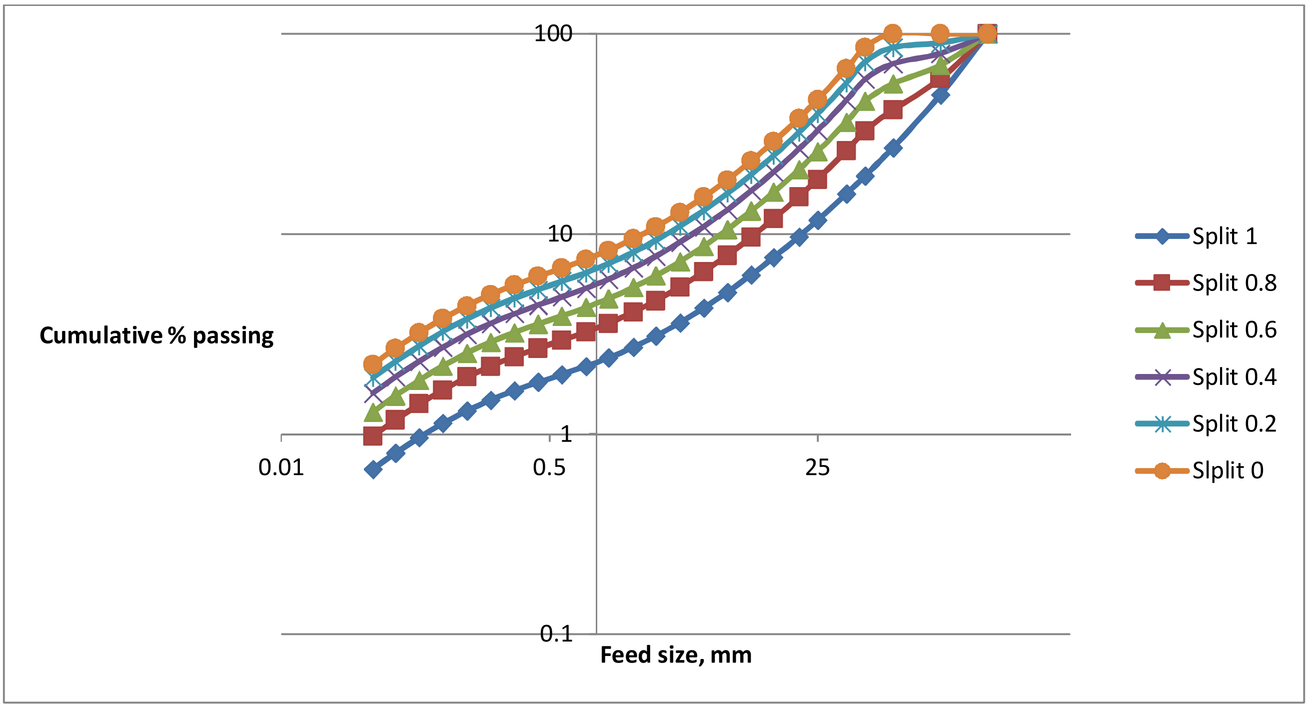

Figure. 4 SAG mill feed size distribution at different split ratios.

At a split ratio of 1, the stockpile consists only of a jaw crusher product and the mill feed size distribution is coarser as it can be observed in Figure 4. As more material is passed through the screen and the oversize subsequently to the cone crusher the mill feed size distribution becomes finer and the split ratio decreases reaching an F_{80} of 45 mm at split ratio of 0.

4.1.2. EFFECT OF PRECRUSHING ON MILLING RATE AND ENERGY CONSUMPTION

Figure 5 show shows how splitting a portion of the primary crusher product to go through the secondary crusher circuit affects SAG mill feed size distribution. The results are based on the Hinde simulator discussed in section 3.3. The same simulator is used to study the effect of feed size distribution on the SAG milling circuit performance.

The SAG milling simulation results at different split ratios are shown in Table 1:

Table. 1 Simulation results summary

Split ratio	**F_{80} (mm)**	**Milled tonnes (t)**	**Specific energy consumption (kWh/t)**	**Primary cyclone o/f Product size (% passing 75 microns)**
0	45	90.5	18.50	74.11
0.2	70	88.17	18.94	74.99
0.4	150	85.9	19.37	75.84
0.6	180	83.8	19.81	76.69
0.8	200	81.8	20.25	77.51
1	240	80	20.68	78.33

Table 1 shows an increase in milled tonnage from 80 to 90.5 t/h and a decrease in specific energy consumption from 20.68 to 18.50 kWh/t for a change in split ratio from 1 to 0.

The graph below show the SAG mill (8% ball load) performance at different mill feed size distributions (split ratios):

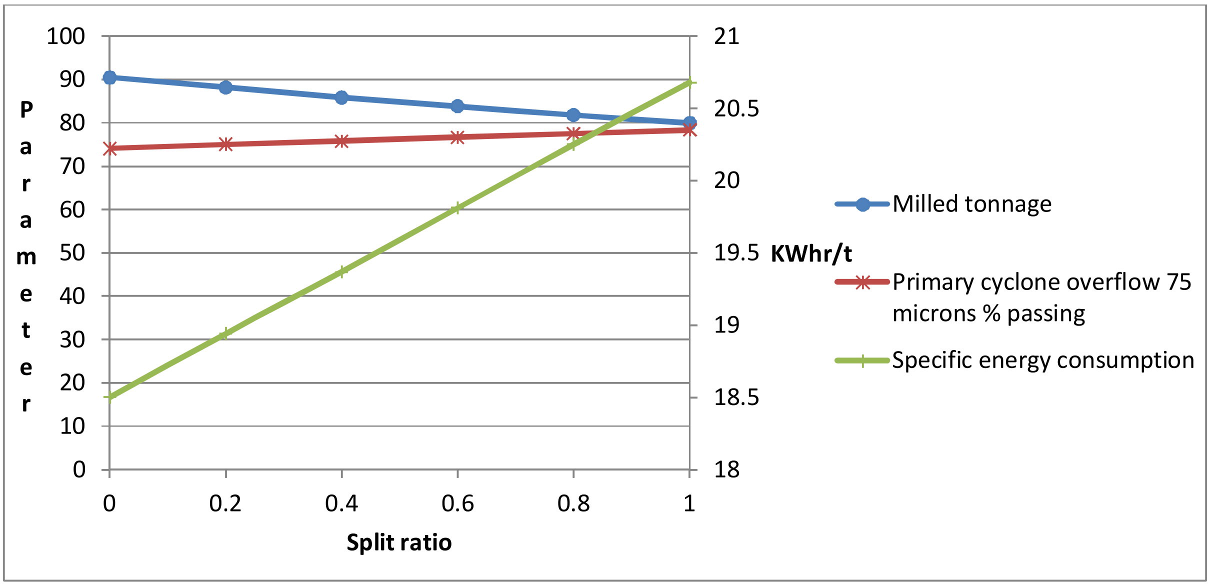

Figure. 5 Effect of pre crush on SAG mill performance

As the feed size distribution to the mill becomes finer (the split ratio decreases) the milled tonnage (t/hr) increases, the specific energy consumption (kWh/t) decreases and the cyclone overflow fineness decreases. Split ratio 0.2 (F_{80} of 70 mm) is considered ideal as it has the highest tonnage (88.17t/hr) whilst fulfilling the plant requirements of the primary cyclone overflow product size (75% passing 75 microns).

4.1.3. ECONOMIC JUSTIFICATION FOR INSTALLING A SECONDARY CRUSHING CIRCUIT.

Capital cost= cost of the crusher and installation= $ 1 000 000 (refurbished sold by Shamva mine Zimbabwe)

Operating cost=Mining and processing costs=$ 1000/ounce (Mine operating cost, Freda financial stamen, 2013-2014)

Revenue due to an increase in tonnage of 8.17t/hr at a grade of 1.6g/t, 80% recovery, 22h/day operation and gold price of $1326/ounce.

Operating cost=$1000/onz*2700.145onz= $2 700145

Revenue=$1326/onz*2700.1450nz=$3580393

Discounted cash flow considers value of money with time. The internal rate of return (IRR) used is 20% which is the rate at which most banks in the country give loans.

Table. 2 Economic evaluation of a secondary crusher installation

YEAR	CF	PVF	DCF	CDC
0	-1000000	1.00000	-1000000	-1000000.00000
1	880247.46	0.83333	733539.55327	-266460.44673
2	880247.46	0.69444	611282.96106	344822.51433
3	880247.46	0.57870	509402.46755	854224.98187
4	880247.46	0.48225	424502.05629	1278727.03816
5	880247.46	0.40188	353751.71357	1632478.75174
6	880247.46	0.33490	294793.09465	1927271.84639
7	880247.46	0.27908	245660.91220	2172932.75859
8	880247.46	0.23257	204717.42684	2377650.18543
9	880247.46	0.19381	170597.85570	2548248.04113
10	880247.46	0.16151	142164.87975	2690412.92087

KEY

CF: Cash flow=Revenue-Operating cost

PVF: $\frac{1}{(1+0.2)^{Year}}$

DCF=Discounted cash flow=CF*PVF (Take into consideration value of money with time)

CDC=Cumulative discounted cash flow= Cumulative sum of DFC

Payback period of the project is 1 year and a month and the project would have generated a profit of over 2.5 million dollars of profit after 10 years.

4.2. OPTION OF PUTTING A REGRIND BALL MILL TO TREAT A PORTION OF THE PRIMARY CYCLONE UNDERFLOW.

Figure 6 shows the proposed circuit showing the regrind ball mill (1000kW) installation:

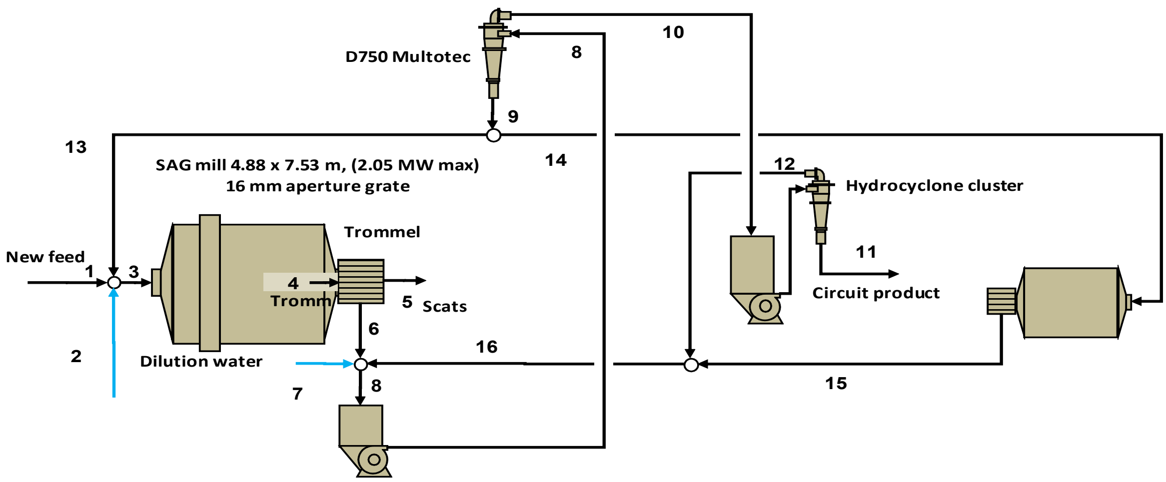

Figure. 6 Proposed SAG-Ball mill circuit (SAB)

The results of the simulation show an increase in tonnage from 80 to 125t/hr representing a 56% increment (appendix B). The SAG mill specific energy consumption decreases from 20.66 to 16kWh/t. The ball mill treats 70t/hr of the cyclone underflow which represents 29% of the total stream; its specific energy consumption is 10.7 kWh/t (750 kW is the available power). The hydrocyclone cluster is used for dewatering and its overflow used as dilution water in the mill sump.

The economic justification for installing the regrind ball mill is shown below:

Capital cost = ball mill cost + mill discharge pumps+ installation and piping cost

=$5 218 000 USD

Revenue due to an increase in tonnage of 45t/hr at a grade of 1.6g/t, 80% recovery and gold price of $1326/ounce (Freda Rebecca financial statement 2013-2014).

Operating cost=$1000/ounce (mining and processing cost)

Total operating cost=$1000/ounce*14870.609 ounces (for 45t/hr increment in tonnage) =$14870609

Annual revenue= (45t/hr*22hrs*365d/yr*1.6g/t*0.8*$1326/onz)/31.1035 g/onz=$19718428.09/yr

The payback period using the net present value (NPV) and internal rate of return (IRR) can be obtained from Table 3.

Table 3 below shows a summary of the NPV analysis:

Table. 3 Economic evaluation of a re grind mill installation

YEAR	CF	PVF	DCF	CDC
0	-6018000	1.00000	-6018000	-6018000.00000
1	4847818.7	0.83333	4039848.89160	-1978151.10840
2	4847818.7	0.69444	3366540.74300	1388389.63461
3	4847818.7	0.57870	2805450.61917	4193840.25378
4	4847818.7	0.48225	2337875.51597	6531715.76975
5	4847818.7	0.40188	1948229.59665	8479945.36640
6	4847818.7	0.33490	1623524.66387	10103470.03027
7	4847818.7	0.27908	1352937.21989	11456407.25016
8	4847818.7	0.23257	1127447.68324	12583854.93340
9	4847818.7	0.19381	939539.73604	13523394.66944
10	4847818.7	0.16151	782949.78003	14306344.44947

The payback period of the project is 1.2 years. After 10 years the project would have generated a profit of over 15 million dollars.

5. MODSIM SIMULATION RESULTS

Effect of installing a fine screen in the cyclone underflow to remove fine material circulating load (modsim simulator). The use of Modsim™ instead of excel based was preferred as in it's initially development the circuit modification shown in figure 7 was not considered.

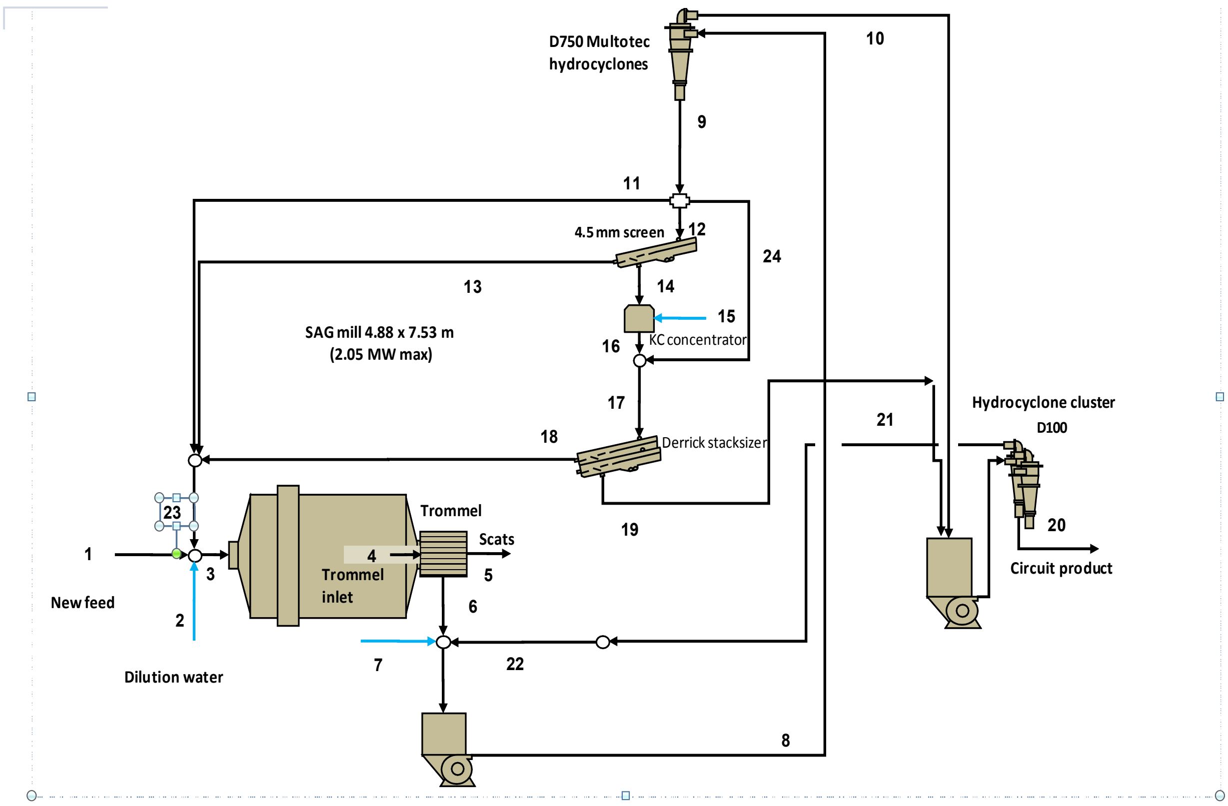

Figure. 7 Derrick screen installation

After doing a plant survey it was realized that around 32% of the cyclone underflow (51.2t/hr in each mill at 200% circulating load, see Appendix C) constitutes the fine material (-100 microns) which can be sent directly to the leaching circuit. An introduction of a100 micron scalping screen to treat the cyclone underflow was proposed.

This would have many advantages which include; reduction of the circulating load which could be linked to improved grinding kinetics as fine material that cause slurry pooling and subsequent cushioning will be minimized and more space will be available for fresh feed hence translating into increased plant capacity. The modified flow sheet in figure 7 is a representation of the simulation setup in the Modsim^{TM} software. This simulation model is used to investigate the effect of installing a Derrick stacksizer fine screen on the circulating load of the fine material. Figure 8 shows the simulation base case whereby no material is passed through the screen.

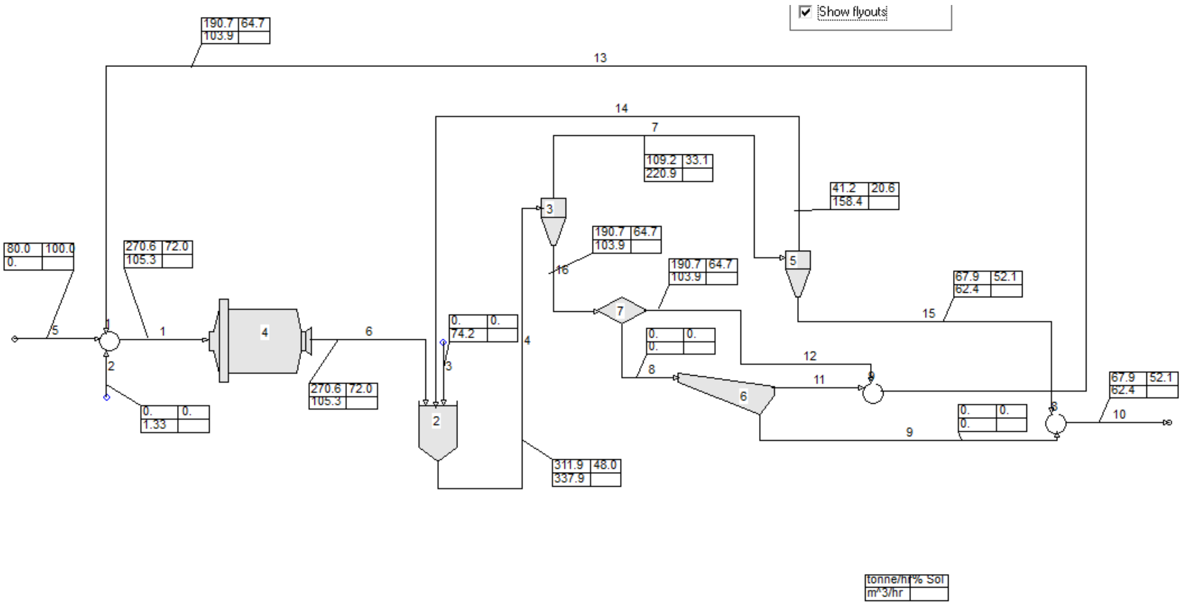

Figure 8 Current Freda Rebecca milling circuit flow sheet

The simulation results show a circulating load of 190.7t/hr (238%) compared to the actual plant circulating load of 200%. And 22% of the material is below 100 microns in the cyclone underflow compared to the actual plant performance of 32%.The primary cyclone overflow product size was very fine 80% passing 56 microns compared to 75% passing 75 microns for the actual plant operation.

The installation of the derrick stacksizer (100 micron aperture size) which treats the whole cyclone underflow to remove fine material hence reducing the circulating load.

The circulating load was found to decrease from 238% to 175%. The density of the final product sent to leach (stream 10 in Appendix C1) dropped slightly from 52.1 to 45.4% solids while remaining in the acceptable range (52%-45%) for leaching and adsorption operations.

The ModSim^{TM} simulation results (particle size distributions) differed significantly from the plant data except for the cyclone underflow where the fine screen was placed. This was done so as to closely monitor the reduction of the circulating load which can be comparable to plant operation. Simulation results also showed a coarser cyclone underflow particle size distribution shown (see figure 9). That is why the reduction in circulating load through fine screening for actual plant performance was higher than that of the simulation:

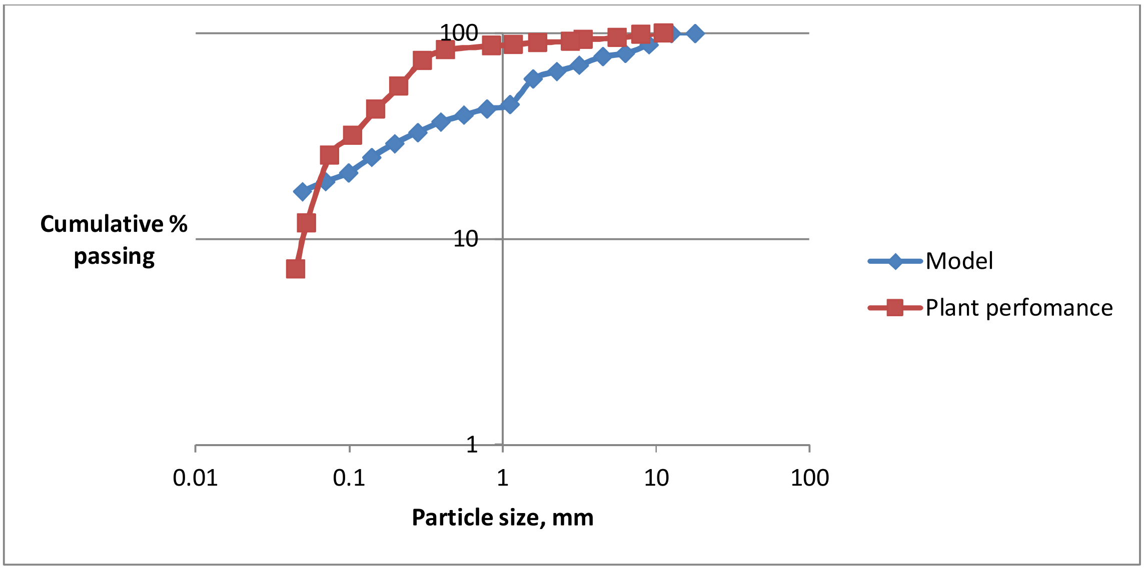

Figure. 9 Cyclone underflow particle size distribution: Model versus Plant performance

The difference in plant and simulation results is mainly due to the fact that most of the parameters were not known and the academic simulator version that was used doesn't have the parameter search option. Although this was a major problem, the simulator does highlight the advantages associated with removing fine material circulating load which is expected to enhance the performance of the milling circuit at Freda Rebecca if implemented.

6. DISCUSSION

When the split ratio in the crushing circuit decreases (as more material is passed through the cone crusher) the particle size distribution of the stockpile (mill feed) becomes finer reaching an F_{80} of 45 mm at a split ratio of 0, from an F_{80} of 240mm when the entire primary crusher product is conveyed straight to the stockpile (split ratio = 1). There is an increase in throughput associated with feeding the SAG mill with a finer feed size distribution. This increase is accompanied by a decrease in the cyclone overflow fineness and a decrease in the specific energy consumption.

A maximum of13.1 % increase in throughput was realized through feeding material with an F_{80} of 45 mm (split ratio=0) compared to feeding one with an $F_{80}$of 240 mm (split ratio=1). The cyclone overflow product size obtained at these conditions was coarser than the required product size which is 75% passing 75 microns. The optimum throughput was obtained at a split ratio of 0.2 corresponding to 88.17t/hr (10.2% increment), specific energy consumption of 18.94kWh/t and the cyclone fineness of 75% passing 75 microns. The payback period of installing a secondary crusher and operating it at an optimum through put is 1.2 years and the project is expected to generate over 2.5 million dollars after 10 years.

The SAG-Ball mill circuit (SAB) where the ball mill treats the cyclone underflow was also evaluated as a potential option to increase through put. This option allows the ball mill to treat the cyclone underflow and the SAG mill to be the primary millwhich is fed with coarser material. In this case the work is distributed between the two mills. The ball mill works at high ball load and discharges a finer product whilst the SAG mill discharges a relatively coarser product. In this instance the SAG mill is optimally utilized with improved overall circuit efficiency and throughput.

The simulation involving the SAB circuit showed an increase in tonnage from 80 to 125t/hr representing a 56% increment. The SAG mill specific energy consumption decreased from 20.66 to 16 kWh/t. The ball mill treated 70 t/hr of the cyclone underflow which represented 29% of the total stream; its specific energy consumption was found to be 10.7 kWh/t (750 kW available power). This suggests that installing a regrind ball mill to treat a fraction of the cyclone underflow is a viable route, with a payback period of 1.2 years and massive return in an acceptable period of time.

Installing a cyclone underflow 100 micron aperture scalping screen reduces the circulating load from 238% to 174% and then power required by 50%. This is expected to translate into increased milled tonnage as more space is made available for fresh feed in the mill due to the reduction of the circulating load as well as improved grinding kinetics.

7. CONCLUSION

The use of simulations has shown that three changes can be implemented on the existing SAG mill circuit at Freda Rebecca Mine. If 80% of the current SAG mill feed goes through the secondary crusher circuit throughput can be increased by 10.2% while still managing the desired cyclone overflow product of 75% passing 75µm. Further improvement of the throughput by about 50% can be achieved by installing a 1000kW regrind ball to treat 29% of the cyclone underflow. Finally, it has also been shown that installing a 100µm screen in the cyclone underflow can also lead to a substantial reduction of re circulating load from 238% to 174%. The installation cost of the regrind mill and the secondary crushing circuit has been estimated to have a payback period of just over a year and the regrind mill having the highest profit of 15 million US dollars in 10 years compared to over 2.5 million dollars for the secondary crushing circuit in the same period. Therefore the regrind mill is considered to be the best option.

8. RECOMMENDATIONS

It has been shown in this work that the milling capacity can be improved by; crushing a fraction of the feed in secondary crusher, installing a regrind mill and introducing the screens to remove -100µm from the circulating load. The findings are all based on simulations, so further experimental work is required to verify these simulation results. The following cause of action is recommended:

- To run a simulation with a high throughput regrind mill at different SAG mill feed size distributions to study its effect on milled tonnage for a SAB circuit.
- To rent a 100 micron aperture size Derrick stacksizer screen from Mintek in the Republic of South Africa and carry out test work. This will enable to monitor the effect of removing fine material from the cyclone underflow on re-circulating load and milled tonnage. To do an economic analysis of the project before considering implementation and permanent installation.

- To run the same simulations with other professional simulation software such as JKSimmet to ascertain the observed results.
- To study the effect of crushing the critical size portion of the SAG mill feed instead of crushing the whole feed.
- To study the effect of crushing the pebble and re circulating them into the SAG mill

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APPENDIX A: PARAMETERS USED FOR SIMULATION

Mill internal diameter [m]	4.640
Mill effective grind length	7.530
Total charge volume (fraction of internal mill volume)	0.250
Ball charge volume (fraction internal mill volume)	0.083
Pinion power draw [kW]	1656.595
Ore SG	2.840
Ball SG	7.850
Mill speed (fraction of critical)	0.730
Closing screen mesh [mm]	16.000

Cumulative breakage rate model parameters (see equation 21)

Parameter	**Value**
k	3.78
a	3.331
b	0.850
c	0.099

Crusher circuit screen parameters

Parameters	**Value**
Bypass fraction of feed to screen oversize **(rf)**	**0.03**
dMesh is the effective screen mesh size [mm]	**60**
a determines the sharpness of cut at coarse sizes	**0**
m determines the sharpness of cut at fine sizes	**5**

APPENDIX B: EXCEL BASED PRE CRUSH SIMULATION RESULTS

Split ratio=0

Size	Stream 1 New feed	Stream 2 Mill inlet water	Stream 3 Mill inlet	Mill holdup Copy holdup	Mill holdup Calc. Hold up	Stream 4 Mill discharge	Stream 5 Trommel o/s	Stream 6 Trommel u/s	Stream 7 Sump water	Stream 8 Cyclone feed	Stream 9 Cyclone u/f	Stream 10 Cyclone o/f	Stream 11 Dewater u/f	Stream 12 Dewater o/f
300.000	0.000		0.000	0.000	0.000	0.000	0.000	0.000		0.000	0.000	0.000	0.000	0.000
150.000	0.000		0.000	0.000	0.000	0.000	0.000	0.000		0.000	0.000	0.000	0.000	0.000
75.000	12.937		12.937	1.127	1.127	0.000	0.000	0.000		0.000	0.000	0.000	0.000	0.000
50.000	17.025		17.025	2.266	2.266	0.000	0.000	0.000		0.000	0.000	0.000	0.000	0.000
38.000	18.089		18.089	3.573	3.573	0.000	0.000	0.000		0.000	0.000	0.000	0.000	0.000
25.000	8.228		8.228	2.016	2.016	0.000	0.000	0.000		0.000	0.000	0.000	0.000	0.000
19.000	8.009		8.009	2.191	2.191	0.000	0.000	0.000		0.000	0.000	0.000	0.000	0.000
13.200	5.238		6.348	1.565	1.565	1.333	0.224	1.109		1.109	1.109	0.000	0.000	0.000
9.500	4.121		7.324	1.432	1.432	3.305	0.102	3.203		3.203	3.203	0.000	0.000	0.000
6.700	3.020		7.913	1.276	1.276	4.919	0.027	4.892		4.892	4.892	0.000	0.000	0.000
4.750	2.302		8.922	1.232	1.232	6.626	0.006	6.620		6.620	6.620	0.000	0.000	0.000
3.350	1.741		10.300	1.236	1.236	8.560	0.001	8.559		8.559	8.559	0.000	0.000	0.000
2.360	1.246		11.535	1.214	1.214	10.289	0.000	10.289		10.289	10.289	0.000	0.000	0.000
1.700	1.062		15.831	1.488	1.488	14.769	0.000	14.769		14.769	14.769	0.000	0.000	0.000
1.180	0.741		18.510	1.539	1.539	17.769	0.000	17.769		17.769	17.769	0.000	0.000	0.000
0.850	0.640		24.685	1.959	1.959	24.046	0.000	24.046		24.046	24.046	0.000	0.000	0.000
0.600	0.551		30.116	2.412	2.412	29.613	0.000	29.613		29.613	29.565	0.048	0.048	0.000
0.425	0.528		34.375	2.836	2.836	34.810	0.000	34.810		34.810	33.847	0.962	0.962	0.000
0.300	0.529		28.117	2.573	2.573	31.590	0.000	31.590		31.590	27.588	4.002	4.002	0.000
0.212	0.537		18.589	2.056	2.056	25.243	0.000	25.243		25.243	18.052	7.191	7.191	0.000
0.150	0.539		12.335	1.714	1.714	21.039	0.000	21.039		21.039	11.796	9.243	9.243	0.000
0.106	0.517		8.684	1.491	1.491	18.301	0.000	18.301		18.303	8.167	10.137	10.134	0.002
0.075	0.482		6.629	1.333	1.333	16.370	0.000	16.370		16.473	6.147	10.326	10.223	0.103
0.053	0.415		5.314	1.149	1.149	14.103	0.000	14.103		14.808	4.900	9.908	9.204	0.705
0.038	2.013		30.546	5.513	5.513	67.676	0.000	67.676		98.729	28.533	70.195	39.142	31.053
Water	0.000	25.000	177.466	14.456	14.456	177.466	0.000	177.466	85.183	549.770	152.466	397.304	110.183	287.121
Tot solids	90.511		350.361	45.192	45.192	350.361	0.361	350.000		381.863	259.850	122.012	90.150	31.863
% solids	100.000	0.000	66.378	75.765	75.765	66.378	100.000	66.355	0.000	40.989	63.022	23.495	45.000	9.989

Size [mm]	Stream 1 New feed	Stream 2 Mill inlet water	Stream 3 Mill inlet	Mill holdup Copy holdup	Mill holdup Calc. Hold up	Stream 4 Mill discharge	Stream 5 Trommel o/s	Stream 6 Trommel u/s	Stream 7 Sump water	Stream 8 Cyclone feed	Stream 9 Cyclone u/f	Stream 10 Cyclone o/f	Stream 11 Dewater u/f	Stream 12 Dewater o/f
300.000	100.000		100.000	100.000	100.000	100.000	100.000	100.000		100.000	100.000	100.000	100.000	100.000
150.000	100.000		100.000	100.000	100.000	100.000	100.000	100.000		100.000	100.000	100.000	100.000	100.000
75.000	100.000		100.000	100.000	100.000	100.000	100.000	100.000		100.000	100.000	100.000	100.000	100.000
50.000	85.707		96.308	97.507	97.507	100.000	100.000	100.000		100.000	100.000	100.000	100.000	100.000
38.000	66.897		91.448	92.492	92.492	100.000	100.000	100.000		100.000	100.000	100.000	100.000	100.000
25.000	46.912		86.285	84.586	84.586	100.000	100.000	100.000		100.000	100.000	100.000	100.000	100.000
19.000	37.821		83.937	80.124	80.124	100.000	100.000	100.000		100.000	100.000	100.000	100.000	100.000
13.200	28.972		81.651	75.275	75.275	100.000	100.000	100.000		100.000	100.000	100.000	100.000	100.000
9.500	23.184		79.839	71.812	71.812	99.619	37.946	99.683		99.709	99.573	100.000	100.000	100.000
6.700	18.631		77.749	68.644	68.644	98.676	9.712	98.768		98.871	98.340	100.000	100.000	100.000
4.750	15.294		75.490	65.820	65.820	97.272	2.283	97.370		97.590	96.458	100.000	100.000	100.000
3.350	12.750		72.944	63.093	63.093	95.381	0.514	95.479		95.856	93.910	100.000	100.000	100.000
2.360	10.827		70.004	60.357	60.357	92.938	0.116	93.033		93.615	90.616	100.000	100.000	100.000
1.700	9.450		66.712	57.670	57.670	90.001	0.028	90.094		90.920	86.657	100.000	100.000	100.000
1.180	8.276		62.193	54.378	54.378	85.786	0.006	85.874		87.053	80.973	100.000	100.000	100.000
0.850	7.458		56.910	50.972	50.972	80.714	0.002	80.797		82.400	74.135	100.000	100.000	100.000
0.600	6.751		49.864	46.638	46.638	73.851	0.000	73.927		76.103	64.882	100.000	100.000	100.000
0.425	6.142		41.269	41.300	41.300	65.399	0.000	65.466		68.348	53.504	99.961	99.947	100.000
0.300	5.559		31.457	35.026	35.026	55.463	0.000	55.521		59.232	40.478	99.172	98.879	100.000
0.212	4.974		23.432	29.332	29.332	46.447	0.000	46.495		50.959	29.861	95.892	94.439	100.000
0.150	4.380		18.126	24.782	24.782	39.242	0.000	39.283		44.349	22.914	89.998	86.463	100.000
0.106	3.785		14.606	20.990	20.990	33.237	0.000	33.271		38.839	18.375	82.423	76.210	100.000
0.075	3.214		12.127	17.691	17.691	28.014	0.000	28.043		34.046	15.232	74.115	64.969	99.992
0.053	2.682		10.235	14.740	14.740	23.341	0.000	23.365		29.732	12.866	65.652	53.629	99.670
0.038	2.224		8.718	12.198	12.198	19.316	0.000	19.336		25.854	10.981	57.531	43.419	97.459
Water [t/h]	0.000	25.000	177.466	14.456	14.456	177.466	0.000	177.466	85.183	549.770	152.466	397.304	110.183	287.121
Solids [t/h]	90.511		350.361	45.192	45.192	350.361	0.361	350.000		381.863	259.850	122.012	90.150	31.863
P_{80}	45.602		9.777	18.817	18.817	0.818	10.484	0.815		0.741	1.124	0.095	0.119	< 38 µm
P_{95}	53.410		45.695	42.539	42.539	3.150	11.025	3.102		2.895	3.832	0.198	0.218	< 38 µm
P_{98}	55.633		51.653	50.588	50.588	5.549	11.266	5.419		5.201	6.195	0.252	0.269	< 38 µm
mass % solids	100.000		66.378	75.765	75.765	66.378	100.000	66.355		40.989	63.022	23.495	45.000	9.989

Net Mill [kW]	J_T	J_B
1674.63	0.250	0.083

Split ratio=0.2

Size	Stream 1 New feed	Stream 2 Mill inlet water	Stream 3 Mill inlet	Mill holdup Copy holdup	Mill holdup Calc. Hold up	Stream 4 Mill discharge	Stream 5 Trommel o/s	Stream 6 Trommel u/s	Stream 7 Sump water	Stream 8 Cyclone feed	Stream 9 Cyclone u/f	Stream 10 Cyclone o/f	Stream 11 Dewater u/f	Stream 12 Dewater o/f
300.000	8.906		8.906	0.111	0.111	0.000	0.000	0.000		0.000	0.000	0.000	0.000	0.000
150.000	3.968		3.968	0.526	0.526	0.000	0.000	0.000		0.000	0.000	0.000	0.000	0.000
75.000	11.398		11.398	1.483	1.483	0.000	0.000	0.000		0.000	0.000	0.000	0.000	0.000
50.000	13.916		13.916	2.216	2.216	0.000	0.000	0.000		0.000	0.000	0.000	0.000	0.000
38.000	14.827		14.827	3.371	3.371	0.000	0.000	0.000		0.000	0.000	0.000	0.000	0.000
25.000	6.767		6.767	1.861	1.861	0.000	0.000	0.000		0.000	0.000	0.000	0.000	0.000
19.000	6.604		6.604	2.003	2.003	0.000	0.000	0.000		0.000	0.000	0.000	0.000	0.000
13.200	4.329		5.341	1.427	1.427	1.215	0.204	1.011		1.011	1.011	0.000	0.000	0.000
9.500	3.412		6.346	1.312	1.312	3.028	0.093	2.934		2.934	2.934	0.000	0.000	0.000
6.700	2.504		7.029	1.180	1.180	4.549	0.025	4.524		4.524	4.524	0.000	0.000	0.000
4.750	1.911		8.103	1.153	1.153	6.198	0.006	6.192		6.192	6.192	0.000	0.000	0.000
3.350	1.447		9.545	1.170	1.170	8.099	0.001	8.098		8.098	8.098	0.000	0.000	0.000
2.360	1.036		10.868	1.160	1.160	9.833	0.000	9.832		9.832	9.832	0.000	0.000	0.000
1.700	0.883		15.117	1.434	1.434	14.234	0.000	14.234		14.234	14.234	0.000	0.000	0.000
1.180	0.617		17.852	1.493	1.493	17.235	0.000	17.235		17.235	17.235	0.000	0.000	0.000
0.850	0.533		23.956	1.908	1.908	23.423	0.000	23.423		23.423	23.423	0.000	0.000	0.000
0.600	0.459		29.448	2.363	2.363	29.011	0.000	29.011		29.011	28.989	0.022	0.022	0.000
0.425	0.441		35.047	2.872	2.872	35.260	0.000	35.260		35.260	34.606	0.654	0.654	0.000
0.300	0.442		30.436	2.722	2.722	33.416	0.000	33.416		33.416	29.993	3.423	3.423	0.000
0.212	0.449		20.323	2.171	2.171	26.649	0.000	26.649		26.649	19.874	6.776	6.776	0.000
0.150	0.451		13.223	1.774	1.774	21.773	0.000	21.773		21.773	12.772	9.001	9.001	0.000
0.106	0.433		9.074	1.517	1.517	18.625	0.000	18.625		18.627	8.641	9.986	9.984	0.002
0.075	0.403		6.771	1.343	1.343	16.482	0.000	16.482		16.584	6.368	10.216	10.115	0.102
0.053	0.347		5.341	1.150	1.150	14.114	0.000	14.114		14.812	4.994	9.818	9.120	0.698
0.038	1.686		30.125	5.473	5.473	67.184	0.000	67.184		97.922	28.439	69.483	38.745	30.738
Water	0.000	25.000	173.560	14.138	14.138	173.560	0.000	173.560	82.360	535.686	148.560	387.126	107.360	279.765
Tot solids	88.170		350.330	45.192	45.192	350.330	0.330	350.000		381.540	262.160	119.381	87.840	31.540
% solids	100.000	0.000	66.871	76.171	76.171	66.871	100.000	66.850	0.000	41.597	63.829	23.569	45.000	10.132

Size [mm]	Stream 1 New feed	Stream 2 Mill inlet water	Stream 3 Mill inlet	Mill holdup Copy holdup	Mill holdup Calc. Hold up	Stream 4 Mill discharge	Stream 5 Trommel o/s	Stream 6 Trommel u/s	Stream 7 Sump water	Stream 8 Cyclone feed	Stream 9 Cyclone u/f	Stream 10 Cyclone o/f	Stream 11 Dewater u/f	Stream 12 Dewater o/f
300.000	100.000		100.000	100.000	100.000	100.000	100.000	100.000		100.000	100.000	100.000	100.000	100.000
150.000	89.900		97.458	99.754	99.754	100.000	100.000	100.000		100.000	100.000	100.000	100.000	100.000
75.000	85.399		96.325	98.590	98.590	100.000	100.000	100.000		100.000	100.000	100.000	100.000	100.000
50.000	72.472		93.072	95.308	95.308	100.000	100.000	100.000		100.000	100.000	100.000	100.000	100.000
38.000	56.689		89.100	90.404	90.404	100.000	100.000	100.000		100.000	100.000	100.000	100.000	100.000
25.000	39.873		84.867	82.944	82.944	100.000	100.000	100.000		100.000	100.000	100.000	100.000	100.000
19.000	32.197		82.936	78.826	78.826	100.000	100.000	100.000		100.000	100.000	100.000	100.000	100.000
13.200	24.707		81.050	74.394	74.394	100.000	100.000	100.000		100.000	100.000	100.000	100.000	100.000
9.500	19.797		79.526	71.237	71.237	99.653	38.139	99.711		99.735	99.614	100.000	100.000	100.000
6.700	15.927		77.715	68.335	68.335	98.789	9.855	98.873		98.966	98.495	100.000	100.000	100.000
4.750	13.087		75.708	65.723	65.723	97.490	2.343	97.580		97.780	96.769	100.000	100.000	100.000
3.350	10.920		73.395	63.172	63.172	95.721	0.533	95.811		96.157	94.407	100.000	100.000	100.000
2.360	9.279		70.671	60.584	60.584	93.409	0.122	93.497		94.035	91.318	100.000	100.000	100.000
1.700	8.104		67.568	58.016	58.016	90.603	0.030	90.688		91.458	87.568	100.000	100.000	100.000
1.180	7.102		63.253	54.843	54.843	86.539	0.007	86.621		87.727	82.138	100.000	100.000	100.000
0.850	6.402		58.157	51.539	51.539	81.620	0.002	81.697		83.210	75.564	100.000	100.000	100.000
0.600	5.798		51.319	47.318	47.318	74.934	0.000	75.004		77.071	66.629	100.000	100.000	100.000
0.425	5.277		42.914	42.088	42.088	66.653	0.000	66.715		69.467	55.572	99.981	99.975	100.000
0.300	4.777		32.910	35.733	35.733	56.588	0.000	56.641		60.225	42.371	99.433	99.230	100.000
0.212	4.275		24.222	29.710	29.710	47.049	0.000	47.094		51.467	30.930	96.566	95.333	100.000
0.150	3.766		18.421	24.906	24.906	39.442	0.000	39.480		44.483	23.350	90.890	87.620	100.000
0.106	3.255		14.646	20.982	20.982	33.227	0.000	33.259		38.776	18.478	83.351	77.372	100.000
0.075	2.763		12.056	17.625	17.625	27.911	0.000	27.937		33.894	15.182	74.985	66.006	99.992
0.053	2.306		10.124	14.654	14.654	23.206	0.000	23.228		29.547	12.753	66.428	54.492	99.670
0.038	1.912		8.599	12.110	12.110	19.177	0.000	19.195		25.665	10.848	58.203	44.109	97.457
Water [t/h]	0.000	25.000	173.560	14.138	14.138	173.560	0.000	173.560	82.360	535.686	148.560	387.126	107.360	279.765
Solids [t/h]	88.170		350.330	45.192	45.192	350.330	0.330	350.000		381.540	262.160	119.381	87.840	31.540
P_{80}	62.577		10.512	20.499	20.499	0.778	10.480	0.775		0.705	1.056	0.092	0.115	< 38 µm
P_{95}	161.600		62.260	48.930	48.930	2.970	11.022	2.928		2.733	3.616	0.188	0.208	< 38 µm
P_{98}	174.066		153.295	67.540	67.540	5.323	11.263	5.208		4.998	5.942	0.239	0.255	< 38 µm
mass % solids	100.000		66.871	76.171	76.171	66.871	100.000	66.850		41.597	63.829	23.569	45.000	10.132

Net Mill [kW]	J_T	J_B
1669.77	0.250	0.083

Split ratio=0.4

Size	Stream 1 New feed	Stream 2 Mill inlet water	Stream 3 Mill inlet	Mill holdup Copy holdup	Mill holdup Calc. Hold up	Stream 4 Mill discharge	Stream 5 Trommel o/s	Stream 6 Trommel u/s	Stream 7 Sump water	Stream 8 Cyclone feed	Stream 9 Cyclone u/f	Stream 10 Cyclone o/f	Stream 11 Dewater u/f	Stream 12 Dewater o/f
300.000	17.362		17.362	0.217	0.217	0.000	0.000	0.000		0.000	0.000	0.000	0.000	0.000
150.000	7.736		7.736	1.028	1.028	0.000	0.000	0.000		0.000	0.000	0.000	0.000	0.000
75.000	9.937		9.937	1.824	1.824	0.000	0.000	0.000		0.000	0.000	0.000	0.000	0.000
50.000	10.963		10.963	2.169	2.169	0.000	0.000	0.000		0.000	0.000	0.000	0.000	0.000
38.000	11.730		11.730	3.178	3.178	0.000	0.000	0.000		0.000	0.000	0.000	0.000	0.000
25.000	5.380		5.380	1.713	1.713	0.000	0.000	0.000		0.000	0.000	0.000	0.000	0.000
19.000	5.270		5.270	1.822	1.822	0.000	0.000	0.000		0.000	0.000	0.000	0.000	0.000
13.200	3.466		4.384	1.295	1.295	1.103	0.185	0.918		0.918	0.918	0.000	0.000	0.000
9.500	2.739		5.416	1.197	1.197	2.763	0.085	2.678		2.678	2.678	0.000	0.000	0.000
6.700	2.014		6.187	1.089	1.089	4.196	0.023	4.173		4.173	4.173	0.000	0.000	0.000
4.750	1.540		7.323	1.077	1.077	5.789	0.006	5.784		5.784	5.784	0.000	0.000	0.000
3.350	1.167		8.825	1.106	1.106	7.659	0.001	7.658		7.658	7.658	0.000	0.000	0.000
2.360	0.837		10.233	1.109	1.109	9.397	0.000	9.397		9.397	9.397	0.000	0.000	0.000
1.700	0.714		14.437	1.382	1.382	13.723	0.000	13.723		13.723	13.723	0.000	0.000	0.000
1.180	0.499		17.225	1.449	1.449	16.726	0.000	16.726		16.726	16.726	0.000	0.000	0.000
0.850	0.431		23.259	1.859	1.859	22.828	0.000	22.828		22.828	22.828	0.000	0.000	0.000
0.600	0.372		28.753	2.313	2.313	28.390	0.000	28.390		28.390	28.380	0.010	0.010	0.000
0.425	0.358		35.321	2.883	2.883	35.393	0.000	35.393		35.393	34.963	0.430	0.430	0.000
0.300	0.360		32.618	2.861	2.861	35.128	0.000	35.128		35.128	32.259	2.869	2.869	0.000
0.212	0.366		22.177	2.293	2.293	28.154	0.000	28.154		28.154	21.811	6.342	6.342	0.000
0.150	0.367		14.197	1.839	1.839	22.580	0.000	22.580		22.580	13.830	8.750	8.750	0.000
0.106	0.353		9.508	1.547	1.547	18.987	0.000	18.987		18.989	9.155	9.834	9.832	0.002
0.075	0.329		6.936	1.353	1.353	16.615	0.000	16.615		16.715	6.607	10.108	10.008	0.101
0.053	0.283		5.380	1.151	1.151	14.136	0.000	14.136		14.828	5.097	9.731	9.039	0.692
0.038	1.376		29.743	5.436	5.436	66.734	0.000	66.734		97.172	28.367	68.805	38.367	30.438
Water	0.000	25.000	169.851	13.836	13.836	169.851	0.000	169.851	79.680	522.312	144.851	377.461	104.680	272.781
Tot solids	85.948		350.300	45.192	45.192	350.300	0.300	350.000		381.233	264.353	116.880	85.647	31.233
% solids	100.000	0.000	67.346	76.561	76.561	67.346	100.000	67.327	0.000	42.193	64.602	23.644	45.000	10.274

Size [mm]	Stream 1 New feed	Stream 2 Mill inlet water	Stream 3 Mill inlet	Mill holdup Copy holdup	Mill holdup Calc. Hold up	Stream 4 Mill discharge	Stream 5 Trommel o/s	Stream 6 Trommel u/s	Stream 7 Sump water	Stream 8 Cyclone feed	Stream 9 Cyclone u/f	Stream 10 Cyclone o/f	Stream 11 Dewater u/f
300.000	100.000		100.000	100.000	100.000	100.000	100.000	100.000		100.000	100.000	100.000	100.000
150.000	79.799		95.044	99.520	99.520	100.000	100.000	100.000		100.000	100.000	100.000	100.000
75.000	70.799		92.835	97.244	97.244	100.000	100.000	100.000		100.000	100.000	100.000	100.000
50.000	59.237		89.999	93.209	93.209	100.000	100.000	100.000		100.000	100.000	100.000	100.000
38.000	46.482		86.869	88.409	88.409	100.000	100.000	100.000		100.000	100.000	100.000	100.000
25.000	32.834		83.521	81.376	81.376	100.000	100.000	100.000		100.000	100.000	100.000	100.000
19.000	26.574		81.985	77.585	77.585	100.000	100.000	100.000		100.000	100.000	100.000	100.000
13.200	20.443		80.480	73.553	73.553	100.000	100.000	100.000		100.000	100.000	100.000	100.000
9.500	16.410		79.229	70.688	70.688	99.685	38.361	99.738		99.759	99.653	100.000	100.000
6.700	13.224		77.683	68.039	68.039	98.896	10.020	98.973		99.057	98.640	100.000	100.000
4.750	10.880		75.916	65.630	65.630	97.699	2.411	97.780		97.962	97.061	100.000	100.000
3.350	9.089		73.826	63.248	63.248	96.046	0.555	96.128		96.445	94.874	100.000	100.000
2.360	7.731		71.307	60.800	60.800	93.860	0.128	93.940		94.436	91.977	100.000	100.000
1.700	6.758		68.385	58.346	58.346	91.177	0.032	91.255		91.972	88.422	100.000	100.000
1.180	5.927		64.264	55.287	55.287	87.259	0.007	87.334		88.372	83.231	100.000	100.000
0.850	5.347		59.347	52.081	52.081	82.485	0.002	82.555		83.985	76.904	100.000	100.000
0.600	4.845		52.707	47.967	47.967	75.968	0.000	76.033		77.997	68.268	100.000	100.000
0.425	4.412		44.499	42.849	42.849	67.864	0.000	67.922		70.550	57.533	99.992	99.989
0.300	3.996		34.416	36.470	36.470	57.760	0.000	57.810		61.266	44.307	99.624	99.487
0.212	3.577		25.105	30.138	30.138	47.732	0.000	47.773		52.052	32.104	97.169	96.137
0.150	3.151		18.774	25.064	25.064	39.695	0.000	39.729		44.667	23.853	91.743	88.731
0.106	2.724		14.721	20.994	20.994	33.249	0.000	33.278		38.744	18.621	84.256	78.515
0.075	2.313		12.006	17.571	17.571	27.829	0.000	27.853		33.763	15.158	75.842	67.035
0.053	1.931		10.027	14.577	14.577	23.086	0.000	23.106		29.378	12.659	67.194	55.350
0.038	1.601		8.491	12.029	12.029	19.050	0.000	19.067		25.489	10.731	58.868	44.797
Water [t/h]	0.000	25.000	169.851	13.836	13.836	169.851	0.000	169.851	79.680	522.312	144.851	377.461	104.680
Solids [t/h]	85.948		350.300	45.192	45.192	350.300	0.300	350.000		381.233	264.353	116.880	85.647
P_{80}	150.228		11.624	22.579	22.579	0.741	10.475	0.738		0.671	0.993	0.088	0.111
P_{95}	174.739		147.698	58.159	58.159	2.797	11.018	2.761		2.578	3.408	0.179	0.197
P_{98}	186.460		162.892	87.091	87.091	5.097	11.260	4.995		4.793	5.688	0.228	0.242
mass % solids	100.000		67.346	76.561	76.561	67.346	100.000	67.327		42.193	64.602	23.644	45.000

Net Mill [kW]	J_T	J_B
1665.16	0.250	0.083

Split ratio=0.6

Size	Stream 1 New feed	Stream 2 Mill inlet water	Stream 3 Mill inlet	Mill holdup Copy holdup	Mill holdup Calc. Hold up	Stream 4 Mill discharge	Stream 5 Trommel o/s	Stream 6 Trommel u/s	Stream 7 Sump water	Stream 8 Cyclone feed	Stream 9 Cyclone u/f	Stream 10 Cyclone o/f	Stream 11 Dewater u/f	Stream 12 Dewater o/f
300.000	25.403		25.403	0.318	0.318	0.000	0.000	0.000		0.000	0.000	0.000	0.000	0.000
150.000	11.318		11.318	1.508	1.508	0.000	0.000	0.000		0.000	0.000	0.000	0.000	0.000
75.000	8.548		8.548	2.149	2.149	0.000	0.000	0.000		0.000	0.000	0.000	0.000	0.000
50.000	8.156		8.156	2.124	2.124	0.000	0.000	0.000		0.000	0.000	0.000	0.000	0.000
38.000	8.785		8.785	2.994	2.994	0.000	0.000	0.000		0.000	0.000	0.000	0.000	0.000
25.000	4.061		4.061	1.572	1.572	0.000	0.000	0.000		0.000	0.000	0.000	0.000	0.000
19.000	4.001		4.001	1.650	1.650	0.000	0.000	0.000		0.000	0.000	0.000	0.000	0.000
13.200	2.645		3.474	1.169	1.169	0.995	0.167	0.828		0.828	0.828	0.000	0.000	0.000
9.500	2.098		4.531	1.087	1.087	2.510	0.077	2.432		2.432	2.432	0.000	0.000	0.000
6.700	1.548		5.385	1.001	1.001	3.858	0.021	3.837		3.837	3.837	0.000	0.000	0.000
4.750	1.186		6.579	1.004	1.004	5.398	0.005	5.393		5.393	5.393	0.000	0.000	0.000
3.350	0.901		8.138	1.046	1.046	7.239	0.001	7.237		7.237	7.237	0.000	0.000	0.000
2.360	0.647		9.627	1.060	1.060	8.981	0.000	8.980		8.980	8.980	0.000	0.000	0.000
1.700	0.553		13.787	1.333	1.333	13.235	0.000	13.235		13.235	13.235	0.000	0.000	0.000
1.180	0.386		16.625	1.407	1.407	16.239	0.000	16.239		16.239	16.239	0.000	0.000	0.000
0.850	0.335		22.593	1.813	1.813	22.258	0.000	22.258		22.258	22.258	0.000	0.000	0.000
0.600	0.290		28.058	2.262	2.262	27.773	0.000	27.773		27.773	27.768	0.004	0.004	0.000
0.425	0.279		35.277	2.873	2.873	35.271	0.000	35.271		35.271	34.998	0.273	0.273	0.000
0.300	0.281		34.590	2.987	2.987	36.665	0.000	36.665		36.665	34.309	2.356	2.356	0.000
0.212	0.287		24.135	2.423	2.423	29.742	0.000	29.742		29.742	23.849	5.893	5.893	0.000
0.150	0.288		15.260	1.911	1.911	23.461	0.000	23.461		23.461	14.973	8.489	8.489	0.000
0.106	0.277		9.987	1.579	1.579	19.388	0.000	19.388		19.390	9.710	9.680	9.678	0.002
0.075	0.258		7.123	1.366	1.366	16.767	0.000	16.767		16.867	6.865	10.002	9.903	0.100
0.053	0.222		5.432	1.154	1.154	14.170	0.000	14.170		14.856	5.209	9.646	8.960	0.686
0.038	1.081		29.397	5.402	5.402	66.322	0.000	66.322		96.473	28.316	68.157	38.006	30.151
Water	0.000	25.000	166.325	13.548	13.548	166.325	0.000	166.325	77.131	509.596	141.325	368.271	102.131	266.139
Tot solids	83.834		350.272	45.192	45.192	350.272	0.272	350.000		380.939	266.438	114.501	83.562	30.939
% solids	100.000	0.000	67.804	76.935	76.935	67.804	100.000	67.787	0.000	42.776	65.341	23.717	45.000	10.414

Size [mm]	Stream 1 New feed	Stream 2 Mill inlet water	Stream 3 Mill inlet	Mill holdup Copy holdup	Mill holdup Calc. Hold up	Stream 4 Mill discharge	Stream 5 Trommel o/s	Stream 6 Trommel u/s	Stream 7 Sump water	Stream 8 Cyclone feed	Stream 9 Cyclone u/f	Stream 10 Cyclone o/f	Stream 11 Dewater u/f	Stream 12 Dewater o/f
300.000	100.000		100.000	100.000	100.000	100.000	100.000	100.000		100.000	100.000	100.000	100.000	100.000
150.000	69.699		92.748	99.296	99.296	100.000	100.000	100.000		100.000	100.000	100.000	100.000	100.000
75.000	56.198		89.516	95.958	95.958	100.000	100.000	100.000		100.000	100.000	100.000	100.000	100.000
50.000	46.002		87.076	91.202	91.202	100.000	100.000	100.000		100.000	100.000	100.000	100.000	100.000
38.000	36.274		84.748	86.502	86.502	100.000	100.000	100.000		100.000	100.000	100.000	100.000	100.000
25.000	25.795		82.240	79.877	79.877	100.000	100.000	100.000		100.000	100.000	100.000	100.000	100.000
19.000	20.951		81.080	76.400	76.400	100.000	100.000	100.000		100.000	100.000	100.000	100.000	100.000
13.200	16.179		79.938	72.749	72.749	100.000	100.000	100.000		100.000	100.000	100.000	100.000	100.000
9.500	13.023		78.946	70.163	70.163	99.716	38.617	99.763		99.783	99.689	100.000	100.000	100.000
6.700	10.520		77.653	67.757	67.757	98.999	10.211	99.068		99.144	98.776	100.000	100.000	100.000
4.750	8.673		76.115	65.542	65.542	97.898	2.490	97.972		98.137	97.336	100.000	100.000	100.000
3.350	7.258		74.237	63.321	63.321	96.357	0.581	96.431		96.721	95.312	100.000	100.000	100.000
2.360	6.183		71.914	61.007	61.007	94.290	0.135	94.363		94.821	92.596	100.000	100.000	100.000
1.700	5.412		69.165	58.662	58.662	91.726	0.034	91.798		92.464	89.225	100.000	100.000	100.000
1.180	4.753		65.229	55.712	55.712	87.948	0.008	88.016		88.989	84.258	100.000	100.000	100.000
0.850	4.292		60.482	52.599	52.599	83.312	0.002	83.377		84.727	78.163	100.000	100.000	100.000
0.600	3.892		54.032	48.587	48.587	76.957	0.000	77.017		78.884	69.809	100.000	100.000	100.000
0.425	3.547		46.022	43.581	43.581	69.028	0.000	69.082		71.593	59.387	99.996	99.995	100.000
0.300	3.214		35.951	37.224	37.224	58.959	0.000	59.004		62.334	46.251	99.758	99.668	100.000
0.212	2.879		26.075	30.615	30.615	48.491	0.000	48.529		52.709	33.374	97.700	96.849	100.000
0.150	2.537		19.185	25.254	25.254	40.000	0.000	40.031		44.902	24.423	92.553	89.796	100.000
0.106	2.193		14.828	21.025	21.025	33.302	0.000	33.328		38.743	18.804	85.140	79.637	100.000
0.075	1.863		11.977	17.531	17.531	27.767	0.000	27.788		33.653	15.159	76.685	68.056	99.993
0.053	1.555		9.943	14.508	14.508	22.980	0.000	22.998		29.225	12.583	67.950	56.205	99.671
0.038	1.290		8.393	11.954	11.954	18.934	0.000	18.949		25.325	10.628	59.525	45.482	97.454
Water [t/h]	0.000	25.000	166.325	13.548	13.548	166.325	0.000	166.325	77.131	509.596	141.325	368.271	102.131	266.139
Solids [t/h]	83.834		350.272	45.192	45.192	350.272	0.272	350.000		380.939	266.438	114.501	83.562	30.939
P_{80}	160.215		13.460	25.180	25.180	0.705	10.470	0.703		0.639	0.934	0.085	0.107	< 38 µm
P_{95}	183.753		155.969	67.853	67.853	2.632	11.014	2.600		2.429	3.199	0.171	0.188	< 38 µm
P_{98}	194.897		168.718	102.891	102.891	4.871	11.256	4.782		4.559	5.435	0.218	0.231	< 38 µm
mass % solids	100.000		67.804	76.935	76.935	67.804	100.000	67.787		42.776	65.341	23.717	45.000	10.414

Net Mill [kW]	J_T	J_B
1660.77	0.250	0.083

Split ratio=0.8

Size	Stream 1 New feed	Stream 2 Mill inlet water	Stream 3 Mill inlet	Mill holdup Copy holdup	Mill holdup Calc. Hold up	Stream 4 Mill discharge	Stream 5 Trommel o/s	Stream 6 Trommel u/s	Stream 7 Sump water	Stream 8 Cyclone feed	Stream 9 Cyclone u/f	Stream 10 Cyclone o/f	Stream 11 Dewater u/f	Stream 12 Dewater o/f
300.000	33.058		33.058	0.415	0.415	0.000	0.000	0.000		0.000	0.000	0.000	0.000	0.000
150.000	14.729		14.729	1.968	1.968	0.000	0.000	0.000		0.000	0.000	0.000	0.000	0.000
75.000	7.225		7.225	2.461	2.461	0.000	0.000	0.000		0.000	0.000	0.000	0.000	0.000
50.000	5.483		5.483	2.081	2.081	0.000	0.000	0.000		0.000	0.000	0.000	0.000	0.000
38.000	5.981		5.981	2.817	2.817	0.000	0.000	0.000		0.000	0.000	0.000	0.000	0.000
25.000	2.805		2.805	1.436	1.436	0.000	0.000	0.000		0.000	0.000	0.000	0.000	0.000
19.000	2.793		2.793	1.485	1.485	0.000	0.000	0.000		0.000	0.000	0.000	0.000	0.000
13.200	1.864		2.607	1.048	1.048	0.892	0.150	0.743		0.743	0.743	0.000	0.000	0.000
9.500	1.489		3.686	0.982	0.982	2.267	0.070	2.197		2.197	2.197	0.000	0.000	0.000
6.700	1.105		4.621	0.917	0.917	3.535	0.019	3.516		3.516	3.516	0.000	0.000	0.000
4.750	0.850		5.870	0.934	0.934	5.024	0.005	5.019		5.019	5.019	0.000	0.000	0.000
3.350	0.648		7.483	0.987	0.987	6.836	0.001	6.835		6.835	6.835	0.000	0.000	0.000
2.360	0.466		9.048	1.013	1.013	8.582	0.000	8.582		8.582	8.582	0.000	0.000	0.000
1.700	0.399		13.167	1.286	1.286	12.768	0.000	12.768		12.768	12.768	0.000	0.000	0.000
1.180	0.280		16.052	1.366	1.366	15.773	0.000	15.773		15.773	15.773	0.000	0.000	0.000
0.850	0.243		21.957	1.769	1.769	21.714	0.000	21.714		21.714	21.714	0.000	0.000	0.000
0.600	0.211		27.379	2.213	2.213	27.170	0.000	27.170		27.170	27.169	0.002	0.002	0.000
0.425	0.204		34.997	2.848	2.848	34.962	0.000	34.962		34.962	34.793	0.168	0.168	0.000
0.300	0.206		36.291	3.094	3.094	37.978	0.000	37.978		37.978	36.084	1.894	1.894	0.000
0.212	0.211		26.175	2.557	2.557	31.396	0.000	31.396		31.396	25.964	5.432	5.432	0.000
0.150	0.212		16.413	1.989	1.989	24.418	0.000	24.418		24.418	16.201	8.217	8.217	0.000
0.106	0.205		10.511	1.615	1.615	19.828	0.000	19.828		19.830	10.307	9.523	9.521	0.002
0.075	0.191		7.333	1.380	1.380	16.940	0.000	16.940		17.039	7.142	9.897	9.798	0.098
0.053	0.165		5.496	1.158	1.158	14.215	0.000	14.215		14.895	5.331	9.564	8.884	0.680
0.038	0.801		29.087	5.372	5.372	65.947	0.000	65.947		95.824	28.286	67.538	37.661	29.878
Water	0.000	25.000	162.967	13.275	13.275	162.967	0.000	162.967	74.705	497.490	137.967	359.522	99.705	259.817
Tot solids	81.822		350.245	45.192	45.192	350.245	0.245	350.000		380.658	268.423	112.235	81.577	30.658
% solids	100.000	0.000	68.246	77.295	77.295	68.246	100.000	68.230	0.000	43.348	66.051	23.791	45.000	10.555

Size [mm]	Stream 1 New feed	Stream 2 Mill inlet water	Stream 3 Mill inlet	Mill holdup Copy holdup	Mill holdup Calc. Hold up	Stream 4 Mill discharge	Stream 5 Trommel o/s	Stream 6 Trommel u/s	Stream 7 Sump water	Stream 8 Cyclone feed	Stream 9 Cyclone u/f	Stream 10 Cyclone o/f	Stream 11 Dewater u/f	Stream 12 Dewater o/f
300.000	100.000		100.000	100.000	100.000	100.000	100.000	100.000		100.000	100.000	100.000	100.000	100.000
150.000	59.598		90.562	99.081	99.081	100.000	100.000	100.000		100.000	100.000	100.000	100.000	100.000
75.000	41.597		86.356	94.726	94.726	100.000	100.000	100.000		100.000	100.000	100.000	100.000	100.000
50.000	32.767		84.293	89.281	89.281	100.000	100.000	100.000		100.000	100.000	100.000	100.000	100.000
38.000	26.066		82.728	84.677	84.677	100.000	100.000	100.000		100.000	100.000	100.000	100.000	100.000
25.000	18.756		81.020	78.443	78.443	100.000	100.000	100.000		100.000	100.000	100.000	100.000	100.000
19.000	15.328		80.219	75.266	75.266	100.000	100.000	100.000		100.000	100.000	100.000	100.000	100.000
13.200	11.914		79.422	71.979	71.979	100.000	100.000	100.000		100.000	100.000	100.000	100.000	100.000
9.500	9.636		78.678	69.661	69.661	99.745	38.917	99.788		99.805	99.723	100.000	100.000	100.000
6.700	7.816		77.625	67.487	67.487	99.098	10.434	99.160		99.228	98.905	100.000	100.000	100.000
4.750	6.466		76.306	65.458	65.458	98.089	2.583	98.155		98.304	97.595	100.000	100.000	100.000
3.350	5.427		74.630	63.390	63.390	96.654	0.611	96.721		96.985	95.725	100.000	100.000	100.000
2.360	4.636		72.494	61.205	61.205	94.702	0.144	94.769		95.190	93.179	100.000	100.000	100.000
1.700	4.066		69.910	58.964	58.964	92.252	0.036	92.317		92.935	89.981	100.000	100.000	100.000
1.180	3.578		66.151	56.118	56.118	88.607	0.008	88.669		89.581	85.225	100.000	100.000	100.000
0.850	3.236		61.568	53.095	53.095	84.103	0.002	84.162		85.438	79.349	100.000	100.000	100.000
0.600	2.939		55.299	49.181	49.181	77.904	0.000	77.958		79.733	71.259	100.000	100.000	100.000
0.425	2.682		47.482	44.284	44.284	70.146	0.000	70.195		72.596	61.138	99.999	99.998	100.000
0.300	2.432		37.489	37.982	37.982	60.164	0.000	60.206		63.411	48.176	99.849	99.792	100.000
0.212	2.180		27.128	31.137	31.137	49.321	0.000	49.355		53.434	34.733	98.161	97.470	100.000
0.150	1.922		19.655	25.477	25.477	40.357	0.000	40.385		45.186	25.060	93.321	90.811	100.000
0.106	1.663		14.968	21.076	21.076	33.385	0.000	33.409		38.772	19.024	86.000	80.738	100.000
0.075	1.413		11.967	17.502	17.502	27.724	0.000	27.743		33.562	15.185	77.515	69.067	99.993
0.053	1.179		9.874	14.449	14.449	22.887	0.000	22.903		29.086	12.524	68.697	57.056	99.671
0.038	0.978		8.305	11.887	11.887	18.829	0.000	18.842		25.173	10.538	60.176	46.166	97.453
Water [t/h]	0.000	25.000	162.967	13.275	13.275	162.967	0.000	162.967	74.705	497.490	137.967	359.522	99.705	259.817
Solids [t/h]	81.822		350.245	45.192	45.192	350.245	0.245	350.000		380.658	268.423	112.235	81.577	30.658
P_{80}	168.531		17.180	27.613	27.613	0.672	10.464	0.670		0.609	0.880	0.082	0.104	< 38 μm
P_{95}	191.165		160.420	77.030	77.030	2.474	11.009	2.447		2.288	2.997	0.164	0.180	< 38 μm
P_{98}	201.798		172.947	114.554	114.554	4.627	11.252	4.537		4.320	5.181	0.208	0.221	< 38 μm
mass % solids	100.000		68.246	77.295	77.295	68.246	100.000	68.230		43.348	66.051	23.791	45.000	10.555

Net Mill [kW]	J_T	J_B
1656.60	0.250	0.083

Split ratio=1

Size	Stream 1 New feed	Stream 2 Mill inlet water	Stream 3 Mill inlet	Mill holdup Copy holdup	Mill holdup Calc. Hold up	Stream 4 Mill discharge	Stream 5 Trommel o/s	Stream 6 Trommel u/s	Stream 7 Sump water	Stream 8 Cyclone feed	Stream 9 Cyclone u/f	Stream 10 Cyclone o/f	Stream 11 Dewater u/f	Stream 12 Dewater o/f
300.000	40.354		40.354	0.508	0.508	0.000	0.000	0.000		0.000	0.000	0.000	0.000	0.000
150.000	17.980		17.980	2.408	2.408	0.000	0.000	0.000		0.000	0.000	0.000	0.000	0.000
75.000	5.964		5.964	2.759	2.759	0.000	0.000	0.000		0.000	0.000	0.000	0.000	0.000
50.000	2.936		2.936	2.039	2.039	0.000	0.000	0.000		0.000	0.000	0.000	0.000	0.000
38.000	3.308		3.308	2.648	2.648	0.000	0.000	0.000		0.000	0.000	0.000	0.000	0.000
25.000	1.608		1.608	1.306	1.306	0.000	0.000	0.000		0.000	0.000	0.000	0.000	0.000
19.000	1.642		1.642	1.327	1.327	0.000	0.000	0.000		0.000	0.000	0.000	0.000	0.000
13.200	1.119		1.780	0.932	0.932	0.794	0.133	0.661		0.661	0.661	0.000	0.000	0.000
9.500	0.908		2.881	0.882	0.882	2.035	0.063	1.973		1.973	1.973	0.000	0.000	0.000
6.700	0.682		3.890	0.837	0.837	3.226	0.018	3.208		3.208	3.208	0.000	0.000	0.000
4.750	0.530		5.191	0.868	0.868	4.666	0.004	4.661		4.661	4.661	0.000	0.000	0.000
3.350	0.406		6.856	0.932	0.932	6.451	0.001	6.449		6.449	6.449	0.000	0.000	0.000
2.360	0.294		8.494	0.968	0.968	8.201	0.000	8.200		8.200	8.200	0.000	0.000	0.000
1.700	0.253		12.573	1.241	1.241	12.320	0.000	12.320		12.320	12.320	0.000	0.000	0.000
1.180	0.178		15.504	1.328	1.328	15.326	0.000	15.326		15.326	15.326	0.000	0.000	0.000
0.850	0.155		21.347	1.726	1.726	21.192	0.000	21.192		21.192	21.192	0.000	0.000	0.000
0.600	0.136		26.723	2.166	2.166	26.588	0.000	26.588		26.588	26.587	0.001	0.001	0.000
0.425	0.132		34.556	2.812	2.812	34.524	0.000	34.524		34.524	34.424	0.101	0.101	0.000
0.300	0.135		37.680	3.180	3.180	39.035	0.000	39.035		39.035	37.545	1.490	1.490	0.000
0.212	0.139		28.269	2.696	2.696	33.093	0.000	33.093		33.093	28.130	4.964	4.964	0.000
0.150	0.140		17.656	2.073	2.073	25.450	0.000	25.450		25.450	17.515	7.934	7.934	0.000
0.106	0.136		11.082	1.654	1.654	20.308	0.000	20.308		20.311	10.947	9.364	9.362	0.002
0.075	0.127		7.566	1.396	1.396	17.133	0.000	17.133		17.231	7.439	9.792	9.695	0.097
0.053	0.109		5.571	1.163	1.163	14.271	0.000	14.271		14.946	5.462	9.484	8.810	0.674
0.038	0.533		28.809	5.344	5.344	65.606	0.000	65.606		95.221	28.276	66.946	37.330	29.615
Water	0.000	25.000	159.767	13.014	13.014	159.767	0.000	159.767	72.393	485.951	134.767	351.184	97.393	253.791
Tot solids	79.905		350.220	45.192	45.192	350.220	0.220	350.000		380.389	270.315	110.074	79.685	30.389
% solids	100.000	0.000	68.672	77.641	77.641	68.672	100.000	68.659	0.000	43.908	66.731	23.864	45.000	10.694

Size [mm]	Stream 1 New feed	Stream 2 Mill inlet water	Stream 3 Mill inlet	Mill holdup Copy holdup	Mill holdup Calc. Hold up	Stream 4 Mill discharge	Stream 5 Trommel o/s	Stream 6 Trommel u/s	Stream 7 Sump water	Stream 8 Cyclone feed	Stream 9 Cyclone u/f	Stream 10 Cyclone o/f	Stream 11 Dewater u/f	Stream 12 Dewater o/f
300.000	100.000		100.000	100.000	100.000	100.000	100.000	100.000		100.000	100.000	100.000	100.000	100.000
150.000	49.498		88.478	98.875	98.875	100.000	100.000	100.000		100.000	100.000	100.000	100.000	100.000
75.000	26.996		83.344	93.547	93.547	100.000	100.000	100.000		100.000	100.000	100.000	100.000	100.000
50.000	19.532		81.641	87.442	87.442	100.000	100.000	100.000		100.000	100.000	100.000	100.000	100.000
38.000	15.858		80.803	82.929	82.929	100.000	100.000	100.000		100.000	100.000	100.000	100.000	100.000
25.000	11.717		79.858	77.070	77.070	100.000	100.000	100.000		100.000	100.000	100.000	100.000	100.000
19.000	9.704		79.399	74.179	74.179	100.000	100.000	100.000		100.000	100.000	100.000	100.000	100.000
13.200	7.650		78.930	71.242	71.242	100.000	100.000	100.000		100.000	100.000	100.000	100.000	100.000
9.500	6.249		78.422	69.180	69.180	99.773	39.274	99.811		99.826	99.756	100.000	100.000	100.000
6.700	5.113		77.599	67.229	67.229	99.192	10.699	99.248		99.308	99.026	100.000	100.000	100.000
4.750	4.259		76.488	65.377	65.377	98.271	2.694	98.331		98.464	97.839	100.000	100.000	100.000
3.350	3.596		75.006	63.457	63.457	96.939	0.647	96.999		97.239	96.115	100.000	100.000	100.000
2.360	3.088		73.049	61.395	61.395	95.097	0.154	95.157		95.544	93.729	100.000	100.000	100.000
1.700	2.720		70.623	59.254	59.254	92.755	0.039	92.814		93.388	90.695	100.000	100.000	100.000
1.180	2.403		67.033	56.508	56.508	89.238	0.009	89.294		90.149	86.138	100.000	100.000	100.000
0.850	2.181		62.606	53.570	53.570	84.861	0.002	84.915		86.120	80.468	100.000	100.000	100.000
0.600	1.986		56.511	49.750	49.750	78.810	0.000	78.860		80.549	72.628	100.000	100.000	100.000
0.425	1.816		48.880	44.957	44.957	71.219	0.000	71.263		73.559	62.792	99.999	99.999	100.000
0.300	1.651		39.013	38.735	38.735	61.361	0.000	61.399		64.483	50.058	99.908	99.873	100.000
0.212	1.482		28.254	31.699	31.699	50.215	0.000	50.246		54.221	36.168	98.554	98.003	100.000
0.150	1.308		20.183	25.734	25.734	40.766	0.000	40.791		45.521	25.762	94.045	91.774	100.000
0.106	1.132		15.141	21.146	21.146	33.499	0.000	33.520		38.831	19.282	86.837	81.817	100.000
0.075	0.963		11.977	17.486	17.486	27.700	0.000	27.717		33.491	15.233	78.330	70.069	99.993
0.053	0.804		9.817	14.398	14.398	22.808	0.000	22.822		28.962	12.481	69.434	57.903	99.672
0.038	0.667		8.226	11.825	11.825	18.733	0.000	18.745		25.033	10.460	60.819	46.847	97.453
Water [t/h]	0.000	25.000	159.767	13.014	13.014	159.767	0.000	159.767	72.393	485.951	134.767	351.184	97.393	253.791
Solids [t/h]	79.905		350.220	45.192	45.192	350.220	0.220	350.000		380.389	270.315	110.074	79.685	30.389
P_{80}	176.121		26.613	30.644	30.644	0.640	10.457	0.638		0.583	0.832	0.080	0.100	< 38 µm
P_{95}	197.861		163.938	84.996	84.996	2.324	11.003	2.302		2.154	2.805	0.158	0.173	< 38 µm
P_{98}	208.004		176.280	123.678	123.678	4.367	11.246	4.289		4.085	4.926	0.198	0.212	< 38 µm
mass % solids	100.000		68.672	77.641	77.641	68.672	100.000	68.659		43.908	66.731	23.864	45.000	10.694

Net Mill [kW]	J_T	J_B
1652.61	0.250	0.083

APPENDIX C

REGRIND MILL SIMULATION

Size	Stream 1 New feed	Stream 2 Mill inlet water	Stream 3 Mill inlet	Mill holdup Copy holdup	Mill holdup Calc. Hold up	Stream 4 Mill discharge	Stream 5 Trommel o/s	Stream 6 Trommel u/s	Stream 7 Sump water	Stream 8 Cyclone feed	Stream 9 Cyclone u/f	Stream 10 Cyclone o/f	Stream 11 Dewater u/f	Stream 12 Dewater o/f	Stream 13 Cyc. u/f to mill	Stream 14 Ball mill feed	Stream 15 Ball mill disch.	Stream 16 Stream 12+15
300.000	37.883		37.883	0.520	0.520	0.000	0.000	0.000		0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000
150.000	16.879		16.879	2.419	2.419	0.000	0.000	0.000		0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000
75.000	12.747		12.747	3.339	3.339	0.000	0.000	0.000		0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000
50.000	12.162		12.162	3.212	3.212	0.000	0.000	0.000		0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000
38.000	13.100		13.100	4.361	4.361	0.000	0.000	0.000		0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000
25.000	6.056		6.056	2.203	2.203	0.000	0.000	0.000		0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000
19.000	5.966		5.966	2.237	2.237	0.000	0.000	0.000		0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000
13.200	3.945		4.756	1.485	1.485	1.265	0.212	1.052		1.135	1.135	0.000	0.000	0.000	0.811	0.324	0.083	0.083
9.500	3.129		5.296	1.253	1.253	2.893	0.089	2.804		3.032	3.032	0.000	0.000	0.000	2.166	0.865	0.228	0.228
6.700	2.309		5.419	1.046	1.046	4.031	0.022	4.009		4.352	4.352	0.000	0.000	0.000	3.110	1.242	0.343	0.343
4.750	1.769		5.792	0.960	0.960	5.161	0.005	5.156		5.630	5.630	0.000	0.000	0.000	4.023	1.607	0.475	0.475
3.350	1.344		6.363	0.922	0.922	6.382	0.001	6.381		7.025	7.025	0.000	0.000	0.000	5.019	2.005	0.644	0.644
2.360	0.965		6.816	0.870	0.870	7.369	0.000	7.368		8.189	8.189	0.000	0.000	0.000	5.851	2.338	0.820	0.820
1.700	0.824		9.024	1.028	1.028	10.209	0.000	10.209		11.476	11.476	0.000	0.000	0.000	8.200	3.276	1.267	1.267
1.180	0.576		10.151	1.020	1.020	11.773	0.000	11.773		13.400	13.400	0.000	0.000	0.000	9.575	3.825	1.627	1.627
0.850	0.499		13.393	1.273	1.273	15.625	0.000	15.625		18.045	18.045	0.000	0.000	0.000	12.894	5.151	2.420	2.420
0.600	0.432		16.487	1.563	1.563	19.186	0.000	19.186		22.531	22.470	0.062	0.062	0.000	16.055	6.414	3.345	3.345
0.425	0.416		19.312	1.875	1.875	23.014	0.000	23.014		27.451	26.445	1.006	1.006	0.000	18.896	7.549	4.437	4.437
0.300	0.419		17.615	1.905	1.905	23.387	0.000	23.387		28.222	24.066	4.155	4.155	0.000	17.196	6.870	4.835	4.835
0.212	0.427		13.361	1.754	1.754	21.532	0.000	21.532		26.184	18.101	8.083	8.083	0.000	12.934	5.167	4.652	4.652
0.150	0.429		9.852	1.622	1.622	19.911	0.000	19.911		24.358	13.187	11.171	11.171	0.000	9.423	3.764	4.447	4.447
0.106	0.413		7.479	1.509	1.509	18.528	0.000	18.528		22.822	9.889	12.933	12.930	0.003	7.066	2.823	4.291	4.294
0.075	0.385		6.041	1.410	1.410	17.314	0.000	17.314		21.662	7.916	13.746	13.609	0.137	5.656	2.260	4.211	4.348
0.053	0.332		5.061	1.249	1.249	15.327	0.000	15.327		20.279	6.619	13.660	12.688	0.971	4.730	1.890	3.981	4.952
0.038	1.612		33.223	6.299	6.299	77.331	0.000	77.331		153.606	44.240	109.367	60.985	48.382	31.611	12.629	27.894	76.275
Water	0.000	25.000	175.684	14.311	14.311	175.684	0.000	175.684	127.399	760.409	210.882	549.527	152.399	397.128	150.684	60.199	60.199	457.327
Tot solids	125.020		300.237	47.332	47.332	300.237	0.330	299.907		419.400	245.218	174.183	124.690	49.493	175.218	70.000	70.000	119.493
% solids	100.000	0.000	63.086	76.784	76.784	63.086	100.000	63.060	0.000	35.548	53.764	24.068	45.000	11.082	53.764	53.764	53.764	20.716

Size	Stream 1 New feed	Stream 2 Mill inlet water	Stream 3 Mill inlet	Mill holdup Copy holdup	Mill holdup Calc. Hold up	Stream 4 Mill discharge	Stream 5 Trommel o/s	Stream 6 Trommel u/s	Stream 7 Sump water	Stream 8 Cyclone feed	Stream 9 Cyclone u/f	Stream 10 Cyclone o/f	Stream 11 Dewater u/f	Stream 12 Dewater o/f	Stream 13 Cyc. u/f to mill	Stream 14 Ball mill feed	Stream 15 Ball mill disch.	Stream 16 Stream 12+15
300.000	100.000		100.000	100.000	100.000	100.000	100.000	100.000		100.000	100.000	100.000	100.000	100.000	100.000	100.000	100.000	100.000
150.000	69.699		87.382	98.902	98.902	100.000	100.000	100.000		100.000	100.000	100.000	100.000	100.000	100.000	100.000	100.000	100.000
75.000	56.198		81.761	93.792	93.792	100.000	100.000	100.000		100.000	100.000	100.000	100.000	100.000	100.000	100.000	100.000	100.000
50.000	46.002		77.515	86.738	86.738	100.000	100.000	100.000		100.000	100.000	100.000	100.000	100.000	100.000	100.000	100.000	100.000
38.000	36.274		73.464	79.952	79.952	100.000	100.000	100.000		100.000	100.000	100.000	100.000	100.000	100.000	100.000	100.000	100.000
25.000	25.795		69.101	70.739	70.739	100.000	100.000	100.000		100.000	100.000	100.000	100.000	100.000	100.000	100.000	100.000	100.000
19.000	20.951		67.084	66.084	66.084	100.000	100.000	100.000		100.000	100.000	100.000	100.000	100.000	100.000	100.000	100.000	100.000
13.200	16.179		65.096	61.358	61.358	100.000	100.000	100.000		100.000	100.000	100.000	100.000	100.000	100.000	100.000	100.000	100.000
9.500	13.023		63.512	58.221	58.221	99.579	35.616	99.649		99.729	99.537	100.000	100.000	100.000	99.537	99.537	99.881	99.930
6.700	10.520		61.749	55.573	55.573	98.615	8.581	98.714		99.006	98.301	100.000	100.000	100.000	98.301	98.301	99.555	99.740
4.750	8.673		59.944	53.363	53.363	97.273	1.921	97.377		97.969	96.526	100.000	100.000	100.000	96.526	96.526	99.066	99.453
3.350	7.258		58.014	51.336	51.336	95.554	0.414	95.658		96.626	94.230	100.000	100.000	100.000	94.230	94.230	98.387	99.055
2.360	6.183		55.895	49.388	49.388	93.428	0.090	93.531		94.951	91.365	100.000	100.000	100.000	91.365	91.365	97.468	98.517
1.700	5.412		53.625	47.551	47.551	90.974	0.021	91.074		92.999	88.026	100.000	100.000	100.000	88.026	88.026	96.296	97.830
1.180	4.753		50.619	45.378	45.378	87.573	0.004	87.670		90.262	83.346	100.000	100.000	100.000	83.346	83.346	94.486	96.770
0.850	4.292		47.238	43.224	43.224	83.652	0.001	83.744		87.067	77.881	100.000	100.000	100.000	77.881	77.881	92.161	95.408
0.600	3.892		42.777	40.535	40.535	78.448	0.000	78.534		82.765	70.522	100.000	100.000	100.000	70.522	70.522	88.703	93.382
0.425	3.547		37.286	37.233	37.233	72.058	0.000	72.137		77.393	61.359	99.964	99.950	100.000	61.359	61.359	83.924	90.583
0.300	3.214		30.854	33.272	33.272	64.392	0.000	64.463		70.847	50.575	99.387	99.144	100.000	50.575	50.575	77.586	86.870
0.212	2.879		24.986	29.247	29.247	56.603	0.000	56.665		64.118	40.761	97.001	95.811	100.000	40.761	40.761	70.679	82.824
0.150	2.537		20.536	25.542	25.542	49.432	0.000	49.486		57.875	33.379	92.361	89.328	100.000	33.379	33.379	64.033	78.930
0.106	2.193		17.255	22.115	22.115	42.800	0.000	42.847		52.067	28.001	85.947	80.370	100.000	28.001	28.001	57.680	75.208
0.075	1.863		14.764	18.926	18.926	36.629	0.000	36.669		46.626	23.969	78.523	70.000	99.994	23.969	23.969	51.550	71.615
0.053	1.555		12.752	15.946	15.946	30.862	0.000	30.896		41.461	20.740	70.631	59.085	99.717	20.740	20.740	45.535	67.977
0.038	1.290		11.066	13.309	13.309	25.757	0.000	25.785		36.625	18.041	62.788	48.909	97.755	18.041	18.041	39.848	63.832
Water [t/h]	0.000	25.000	175.684	14.311	14.311	175.684	0.000	175.684	127.399	760.409	210.882	549.527	152.399	397.128				
Solids [t/h]	125.020		300.237	47.332	47.332	300.237	0.330	299.907		419.400	245.218	174.183	124.690	49.493				
P_{80}	160.215		63.146	38.068	38.068	0.663	10.531	0.659		0.500	0.961	0.080	0.105	< 38 µm				
P_{95}	183.753		165.587	83.494	83.494	3.030	11.064	2.975		2.381	3.719	0.178	0.200	< 38 µm				
P_{98}	194.897		177.839	122.318	122.318	5.601	11.301	5.454		4.789	6.230	0.235	0.254	< 38 µm				
mass % solids	100.000		63.086	76.784	76.784	63.086	100.000	63.060		35.548	53.764	24.068	45.000	11.082				

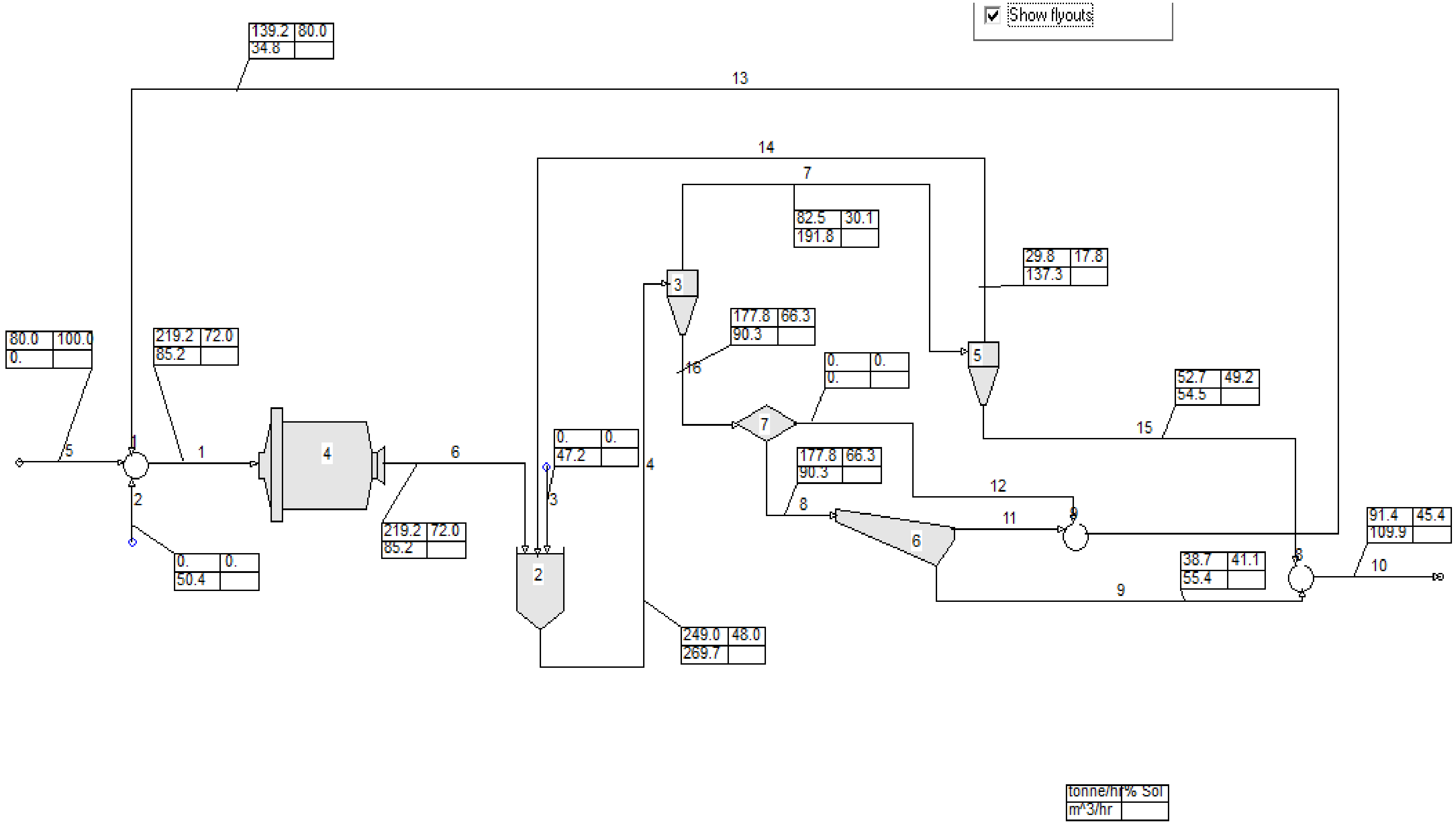
APPENDIX D

MODSIM SIMULATION OF BASE CASE

CYCLONE UNDERFLOW PARTICLE SIZE DISTRIBUTION:

	Model	
Cumulative% Passing		size, mm
100		17.9
100		12.6
88		8.94
80		6.31
77		4.47
70		3.16
65		2.24
60		1.58
45		1.12
43		0.789
40		0.559
37		0.395
33		0.279
29		0.197
25		0.14
21		0.987
19		0.698
17		0.494

Appendix E: Scalping of fine material from the cyclone underflow from the base case circuit.



APPENDIX F

Mill feed

MILL FEED				
SEIVE SIZE mm	MASS RETAINED (g)	% RETAINED	CUM % RETAINED	CUM % PASSING
300	0	0	0	100
250	16.28	19.07	19.07	80.93
125	8.88	10.40	29.47	70.53
75	15.78	18.48	47.95	52.05
53	7.02	8.22	56.17	43.83
45	2.64	3.09	59.26	40.74
31.5	6.1	7.14	66.41	33.59
25	3.18	3.72	70.13	29.87
13.2	8.3	9.72	79.85	20.15
11.2	2.96	3.47	83.32	16.68
8	1.98	2.32	85.64	14.36
5.6	1.44	1.69	87.33	12.67
2.8	1.84	2.16	89.48	10.52
2	0.96	1.12	90.61	9.39
1.7	0.25	0.29	90.90	9.10
1.18	0.72	0.84	91.74	8.26
0.5	2.03	2.38	94.12	5.88
0.425	0.83	0.97	95.09	4.91
0.3	0.94	1.10	96.19	3.81
0.212	0.89	1.04	97.24	2.76
0.15	0.67	0.78	98.02	1.98
0.106	0.64	0.75	98.77	1.23
0.075	0.41	0.48	99.25	0.75
0.053	0.28	0.33	99.58	0.42
0.045	0.19	0.22	99.80	0.20
-0.045	0.17	0.20	100	
TOTAL	**85.38**	**100**		

Mill discharge

+11.20	0	0	0	100
+8.00	2.0	0.1	0.1	99.9
+5.60	9.0	0.6	0.7	99.3
+3.35	11.0	0.7	1.5	98.5
+2.80	6.0	0.4	1.8	98.2
+1.70	17.0	1.1	3.0	97.0
+1.18	4.0	0.3	3.2	96.8
+0.85	35.0	2.3	5.5	94.5
+0.425	136.0	9.0	14.5	85.5
+0.300	225.0	14.9	29.4	70.6
+0.212	213.0	14.1	43.5	56.5
+0.150	153.0	10.1	53.6	46.4
+0.106	278.0	18.4	71.9	28.1
+0.075	109.0	7.2	79.1	20.9
+0.053	43.0	2.8	82.0	18.0
+0.045	54.0	3.6	85.5	14.5
-0.045	219.0	14.5	100.0	
Total	**1514**	**100.0**		

Mill Primary cyclone overflow

Sieve Size mm	Mass Retained g	Mass Retained %	Cumulative % Retained	Cumulative % Passing
+0.850	0	0	0	100
+0.425	12.0	11.0	11.0	614.7
+0.300	40.0	36.7	47.7	578.0
+0.212	31.0	28.4	76.1	549.5
+0.150	54.0	49.5	125.7	500.0
+0.106	109.0	100.0	225.7	400.0
+0.075	109.0	100.0	325.7	300.0
+0.053	109.0	100.0	425.7	200.0
+0.045	109.0	100.0	525.7	100.0
-0.045	109.0	100.0	625.7	
Total	109.0	**625.7**		

Mill Primary cyclone underflow

Sieve Size mm	Mass Retained g	Mass Retained %	Cumulative % Retained	Cumulative % Passing
+11.20	0	0	0	100
+8.00	22.0	2.0	2.0	98.0
+5.60	37.0	3.3	5.3	94.7
+3.35	16.0	1.4	6.8	93.2
+2.80	22.0	2.0	8.8	91.2
+1.70	13.0	1.2	9.9	90.1
+1.18	23.0	2.1	12.0	88.0
+0.85	16.0	1.4	13.5	86.5
+0.425	43.0	3.9	17.4	82.6
+0.300	103.0	9.3	26.7	73.3
+0.212	200.0	18.1	44.8	55.2
+0.150	140.0	12.7	57.4	42.6
+0.106	120.0	10.9	68.3	31.7
+0.075	69.0	6.2	74.5	25.5
+0.053	150.0	13.6	88.1	11.9
+0.045	53.9	4.9	92.9	7.1
-0.045	78.0	7.1	100.0	
Total	**1105.9**	**100.0**		