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# THE EFFECTS OF FEED SIZE DISTRIBUTION ON A RUN–OF–MINE BALL MILL

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A research report submitted to the Faculty of Engineering and the Built Environment, University of the Witwatersrand, in partial fulfilment of the requirements for the degree of Master of Science in Engineering.

Johannesburg, 2008

**DECLARATION**

I declare that this research report is my own, unaided work. It is being submitted in partial fulfilment for the Degree of Master of Science in Engineering in the University of the Witwatersrand, Johannesburg. It has not been submitted before for any degree or examination in any other University.



The eighteenth day of March, 2008

**ABSTRACT**

As there is little published data on the operation of Run-of-Mine Ball Mills a data inspection and modelling exercise was carried out using an Anglo Platinum concentrator plant mill as the subject. The purpose was to establish whether differences in the size distribution of the feed rock had any effects on the mill operation and performance, and to attempt to quantify these effects.

Analysis of data enabled correlations to be determined between the feed and product size ranges. It was found that there are good, positive correlations between the feed size and the product size. It was also found that the feed size influences the power drawn by the mill in that increasing quantities of the finer fractions decreases mill power and that increasing quantities of coarse material increases the mill power.

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# 1 INTRODUCTION

## 1.1 Run-of-Mine Ball Mills

Some Semi-Autogenous-Grind (SAG) mills are operated with high ball loads of up to thirty percent by volume. This is considered to improve stability with fluctuating feed ore types and sizes. It has been shown that reducing the ball load can result in increased throughput with little loss in grind and considerable cost savings (Powell et al., 2001, Hinde et al 2001)). Notwithstanding this, there are currently at least five mills in the South African platinum industry operated as "Run-of-Mine Ball mills" (ROM ball mills). These are situated at the Amandelbult, Mortimer, Waterval, Modikwa and Mototolo concentrators of Anglo Platinum.

Little work has been done on the modelling or optimization of these mills. The effects of feed size on SAG mills are well known, (Morell, 2003), but have not been extended to ROM ball mills. SAG mill models have been developed for substantially different mills, typically without direct measurement of the feed sizing, (Apelt, 2002). Feed size measurement has been proposed as a control input (Gillot, 2004; Bouajila et al, 2000; Simkus et al, 1998) but has not been implemented on ROM ball mills.

A literature search reveals only one paper specifically referring to "Run-of-Mine Ball Mills" (Mainza and Powell, 2006) and one, (Powell et al, 2001) which reported on the use of a SAG mill model. The 2001 work resulted in the mill in question, at Impala Platinum Mines, reverting to SAG operation at a drastically reduced ball load. Powell commented that this mill falls outside the then currently modelled parameters in terms of load, aspect ratio and speed. The mills operated by Anglo Platinum differ further from the Impala mill in terms of size and circuit configuration, (Table 1.1).

Table 1.1 Comparisons of Run-of-Mine Ball Mills

| **Mill** | | **Impala** | **Waterval** | **Mortimer** | **Modikwa** |
|---|---|---|---|---|---|
| **Diameter** | m | 4.770 | 7.320 | 6.096 | 6.096 |
| **Length** | m | 5.880 | 8.530 | 7.320 | 7.320 |
| **Installed Power** | kW | 3200 | 10400 | 4500 | 4500 |
| **Speed** | % Critical | 89.1 | 75 | 75 | 75 |
| **Liner type** | | Grid | Solid, with lifters | Solid, with lifters | Solid, with lifters |
| **Ball Load** | % | 37 | 32 | 30 - 35 | 30 |
| **Total Load** | % | 48 | 45 | 45 | 45 |
| **Classifier Type** | | Cyclones | Screens | Screens | Screens |
| **Design Feed Rate** | tph | 300 | 600 | 345 | 345 |

Mainza compared AG/SAG and ROM ball mills and concluded that the ROM ball mills exhibit higher breakage rates and are more suited to friable ores such as the UG2 seam. Feeds which were coarser than the comparative AG mill resulted in finer products, but no comparison of different feed sizes was done for the ROM mill. The effect of ball load was investigated and it was concluded that a higher ball load increased breakage at all feed sizes but proportionately more in the larger sizes.

## 1.2 SAG Mill Investigations

Other authors have investigated the behaviour of SAG mills and have concluded that finer feeds generally result in increases in feed rate. (Hart et al 2001, Strohmayr and Valery 2001, Sylvestre et al 2001, Bouajila, et al 2001, Dance 2001, Morell 2001, McGhee et al, 2001).

Sherman, (2001) reports the opposite effect, increasing fines resulting it lower throughput. He comments on the increased quantity of critical size material as the reason. The difficulties of distinguishing between the effects of feed size and feed type are also highlighted. An increased ball charge did not result in increased throughput.

Strohmayr and Valery (2001), confirm that the optimum grinding conditions are achieved with a bi-modal distribution where the quantity of intermediate sized material in the feed is at a minimum. The intermediate size range is defined as 20 mm to 50 mm. Dance (2001) reports similar findings.

Nelson et al, (1996), comment on increased throughput with increasing ball load at Kalgoorlie Consolidated Gold Mines Firmiston Plant. Also of interest is the fact that increasing the feed size by bypassing crushers resulted in an increased power draw and improved grind.

Sylvestre et al (2001), report on the pre-crushing of feed material to improve throughput but experienced an increase in the quantity of critical sized material reporting to the in-circuit crusher. The critical size range is defined as 12 mm to 50 mm.

McGhee et al, (2001), investigated the pre-crushing of ore and found that the throughput increased but at the expense of grind, resulting in a coarser product.

Some have reported improvements in grind resulting from finer feed material (Lam et al, 2001 Bouajila, et al 2001. Morell 2001), but at low steel loading of 11% to 13% by volume.

Lam et al, (2001), comment on unchanged breakage rates and the effects of recycling scats to the mill. A critical size range of 20 mm to 75 mm was identified for this ore type.

All the above authors refer specifically to typical North American-style SAG mills with a large diameter to length ratio and low ball filling. Where mills could be identified from the papers the mill details and findings have been summarised in table 1.2, below. It can be seen that the mills and operating conditions are dissimilar to those used in the South African platinum mining industry.

Table 1.2 Comparisons of SAG Mill Testwork.

| Reference | Plant | Mill type | Mill size | Mill Ball Load | Findings |
|---|---|---|---|---|---|
| | | | | | |
| Hart et al 2001 | Newcrest, Cadia Hill | SAG, closed circuit with pebble crushing | 12.2m Ø x 6.7 m 19 MW installed. | 14 % by volume | Finer feed = increased throughput |
| Strohmayr and Valery 2001 | Ernest Henry | SAG closed circuit with.screens | 10.4m Ø x 5.1m 11 MW installed. | 8 % by volume | Finer feed = increased throughput. Maximum efficiency at minimum quantity of 20 – 50 mm material. |
| Sylvestre et al 2001 | Inmet Troilus Mine | SAG, closed circuit with pebble crushing | 9.1 m Ø x 4.0 m 5.25 MW installed. | 10 % by volume | Finer feed = increased throughput. Pre-crushing of 25mm – 150mm fraction to -38mm increased the quantity of critical size material. |
| Dance 2001 | Highland Valley. | SAG closed circuit with.screens | 13.1 m Ø x 4.9 m 9.4 MW installed | Unknown | Finer feed = increased throughput. Maximum efficiency at minimum quantity of 20 – 50 mm material. |

| Reference | Plant | Mill type | Mill size | Mill Ball Load | Findings |
|---|---|---|---|---|---|
| Sherman, 2001 | Alumbrera | SAG | 11 m Ø x 5.8 m<br>13 MW installed | Unknown | No relationship between feed size and throughput rate.<br>No relationship between increased ball charge and throughput rate. |
| Nelson et al, 1996 | Kalgoorlie Consolidated Gold Mines, Firmiston plant | SAG, open circuit. | 11 m Ø x 4.9 m<br>12 MW installed | 13% | Coarser feed resulted in increased stability, increased power draw and a finer grind. |
| Lam et al, 2001 | Porgera Gold Mine | SAG, closed circuit with pebble crushing | 8.5 m Ø x 3.4 m<br>4.1 MW installed | 13% | Finer feed = increased throughput. |
| McGhee, et al 2001 | ASARCO, Ray Concentrator | SAG, closed circuit with pebble crushing | 10.4 m Ø x 5.8 m<br>10.4 MW installed | 12% | Finer feed = increased throughput but coarser grind. |

## 1.3 Design Issues

The problems facing design engineers and plant operators have been the subject of considerable debate but little investigation. These problems include:

- How should current mill design techniques developed for ball and SAG mills be modified for ROM ball mills?
- What is the optimum ball load for a given feed rate and ore type?
- Is there an optimum feed size?
- What is the optimum mill speed and liner profile?

Anecdotal accounts from plant personnel indicate that higher ball loads give better grinds at similar feed rates, as with SAG mills, but that feed particles above a certain critical size inhibit throughput. Thus any information on the applicability of SAG models to ROM ball mills is of particular interest

## 1.4 Test Programme

Three data capture campaigns were undertaken. The first, referred to as the "Long-term data" period represents data records captured at five minute intervals for a period of a month. Although useful, this data set was not considered appropriate for modelling due to the following concerns:

- The calibration of some equipment, specifically the milling circuit product particle size monitor, was doubtful.
- Data capture was not continuous, resulting in significant gaps.
- During a large portion of the time the mill operation was unsteady. This will also be discussed below.

The second period was restricted to a single day, ("Short-term data"), during which time the feed was increased in steps to give six relatively stable periods under different conditions. These periods form the basis for the modelling work and consequent conclusions.

The third period consisted of a number of weeks of data logging, but after major modifications to the secondary milling circuit of the plant. ("Medium-term data"). These modifications resulted in considerably more stable operation in terms of stoppages, feed rate and power draw, allowing the effect of feed size to be clearly seen.

Test periods are identified in the text as LT, MT or ST, depending on the data set.

# 2 DESCRIPTION OF THE TEST MILLING CIRCUIT

The mill targeted in this study is the primary mill at the Waterval UG2 plant of Anglo Platinum, situated near Rustenburg in the North-West Province. The mill was originally designed and commissioned in the year 2000 as a 7 MW fully autogenous unit. In 2003 it was modified to the current 10 MW ROM ball mill. This mill was chosen because it is the most fully instrumented of the five ROM ball mills. In addition to the standard instrumentation, (Table 2.1), there is an optical rock sizing instrument situated on the mill feed conveyor and an ultrasonic particle size monitor on the classified product stream. These instruments will be described below.

## 2.1 Process Description

### 2.1.1 Ore Type.

The ore fed to the Waterval UG2 concentrator consists mainly of chromite, pyroxenite and feldspar. (Clark-Mostert 2005). Pentlandite is the major Base Metal Sulphide (BMS) phase present in the feed. The BMS in the feed are largely liberated, with locked and middlings BMS mainly associated with pyroxene and feldspar. The average chromite grain size is of the order of 0.5mm

Table 2.1 Bulk Mineralogical Composition.

| **Mineral** | **Proportion %** |
|---|---|
| Pyroxene | 38.4 |
| Chromite | 36.8 |
| Feldspar | 17.5 |
| Olivine | 0.6 |
| Alteration Silicates | 3.9 |
| Other Silicates | 1.7 |
| Base Metal Sulphides | 0.3 |
| Other | 0.8 |
| Total | 100.0 |

Table 2.2 gives the relative abundance of the Base Metal Sulphides in the feed, which mainly consists of pyrite, pyrrhotite, pentlandite and chalcopyrite.

Table 2.2: Base Metal Sulphides: relative abundance.

| **Sulphide Group** | **Proportion %** |
|---|---|
| Fe-Sulphides | 26.9 |
| Ni-Sulphides | 49.3 |
| Cu-Sulphides | 23.8 |
| Total | 100.0 |
| Mass % | 0.26 |

### 2.1.2 Ore Receiving Section

The Waterval Concentrator receives approximately 250 000 tons per month of UG2 ore from the Waterval underground bulk mining operation and approximately 250 000 tons per month of UG2 ore from the narrow-width Rustenburg section mines. UG2 (Upper Group 2) is a chromite seam containing economically viable quantities of platinum group metals. Waterval ore is conveyed from the mine and fed to a primary grizzly and jaw crusher via a silo of 5000 ton capacity. The ore from the Rustenburg Section mines is conveyed from a rail receiving bin either to its primary grizzly and jaw crusher feed conveyor belt, or to a 50 000 ton stockpile. Stockpiled ore is recovered by front end loader and added to the conveyor from the rail receiving bin when required. The primary grizzly feed conveyor belts are equipped with electromechanical weightometers, metal detectors and belt magnets.

### 2.1.3 Primary Crushing Section

Waterval Section ore is fed at a rate of approximately 550 tons per hour over a 250mm aperture vibrating grizzly to a primary jaw crusher. The jaw crusher receiving the oversize is designed to receive a top size of 200x200x300mm and produce a maximum product size of 250mm. The crusher product combines with the -250mm undersize from the grizzly. The -250mm material

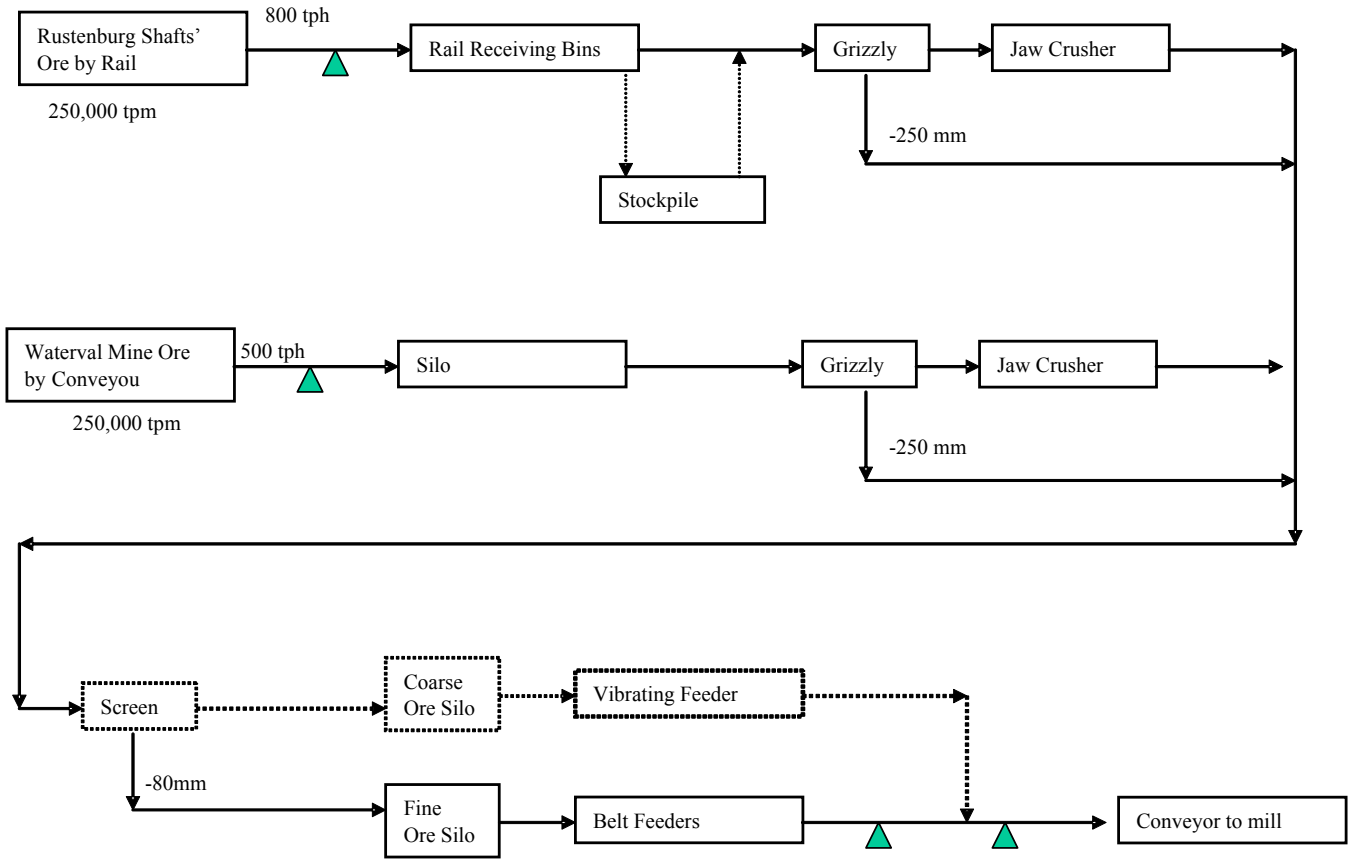
is combined with the -250mm ore from the Rustenburg Section and conveyed to a screen ahead of the mill feed silos.

Rustenburg Section ore from the rail receiving bin, or the stockpile, is fed at a rate of approximately 800 $t.h^{-1}$ over an identical grizzly and jaw crusher.

The combined stream is thus made up of Waterval ore and a balance of Rustenburg product to give 500 000 tons per month to the milling section. This is conveyed to a vibrating screen mounted on top of the fine ore mill feed silo. The screen has an 80mm cut point with undersize and oversize reporting to the fine and coarse ore silos respectively. This originally allowed a variable mix of fine and coarse ore, blended automatically as required, to be fed onto the mill feed conveyor from the two silos; the fine ore by means of one of two variable speed belt feeders equipped with electromechanical weightometers, the coarse ore by means of a variable speed vibrating feeder.

The mill feed rate is measured by an electromechanical weightometer installed on the mill feed conveyor and controlled to a setpoint by the variable speed feeders. The weightometer on the fine ore belt feeder is used in conjunction with the weightometer on the combined mill feed to control the coarse to fine ore ratio. Refer to figure 2.1

.




▲ = Weightometer

**Figure 2.1: Flowsheet: Waterval UG2 Concentrator, Ore Receiving Sections**

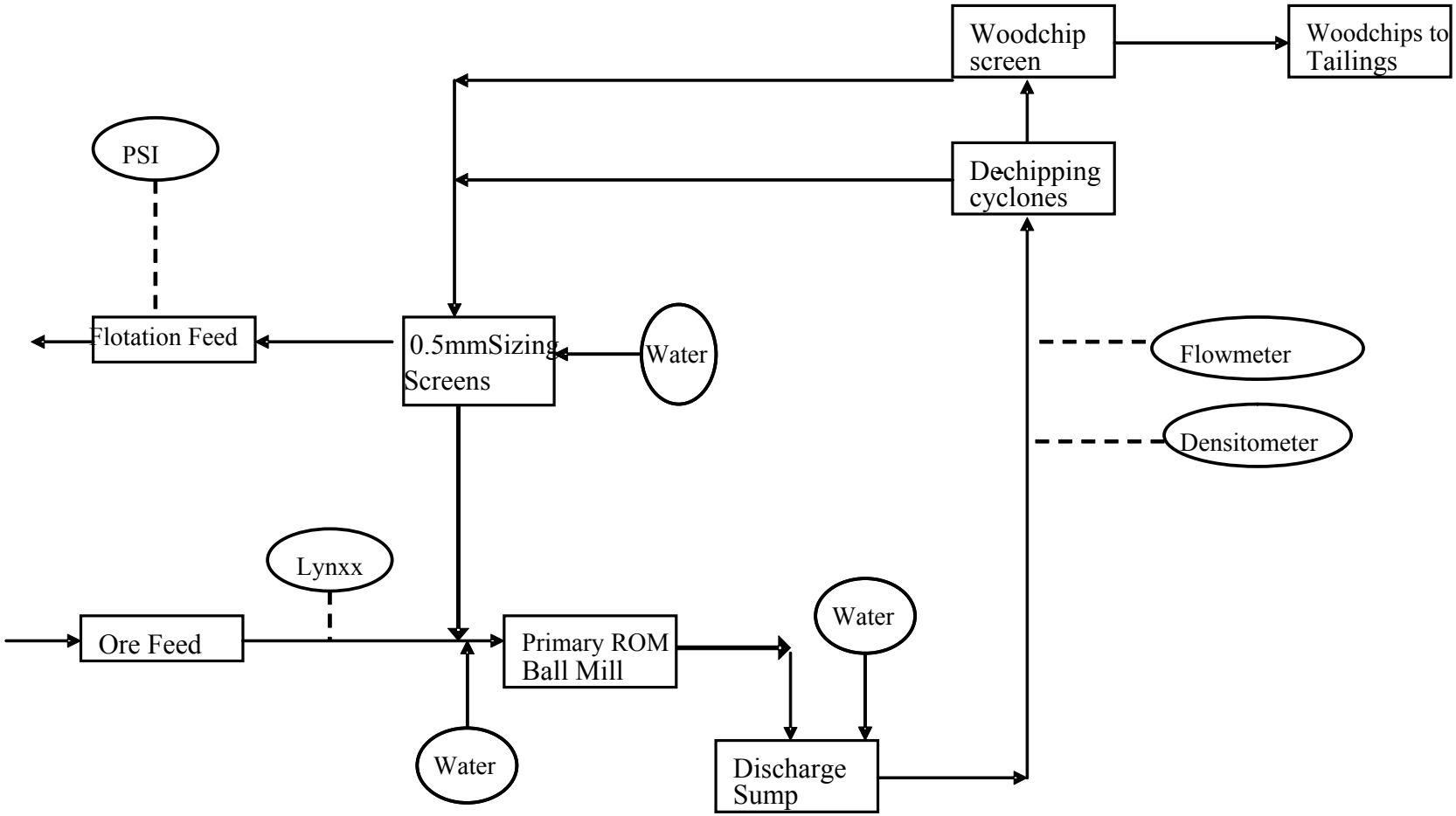



**Figure 2.2: Flowsheet: Waterval UG2 Concentrator: Primary Milling**

### 2.1.4 Primary Milling Section

The primary milling circuit consists of a SAG/ball mill of 7.32m diameter and 8.53m effective grind length. The mill is fitted with a grate discharge which conforms to the slope of the conical end. The liners are manganese-steel with integral 150mm high lifters. The mill product is pumped by an operating or standby set of two all-metal, variable-speed pumps to de-chipping cyclones. The de-chipping cyclone overflow gravitates to a woodchip removal screen. Wood chips from the screen are pumped to the tailings disposal sump.

The de-chipping cyclone underflows and woodchip screen underflow gravitate to three of four classification screens, (one standby). The classification screens are vibrating screens of nominal aperture 0.5mm, designed to produce a primary rougher flotation feed with $p_{80}$ equal to 300µm. Classification screen overflow reports via the mill feed conveyor to the mill feed inlet hopper. The classification screen underflow is sampled and pumped to a primary chromite removal cyclone. Cyclone overflow is the feed to the primary rougher flotation section. Cyclone underflow discharges into the secondary mill. The secondary mill is in closed circuit with a further stage of cycloning. The overflow from these cyclones feeds the secondary flotation cells.

Mill dilution water is measured and controlled either to a fixed setpoint or as a ratio to the mill feed. Process water is sprayed onto the mill trommel and added into the mill discharge sump to dilute the mill product from 80% solids to 55% solids for the de-chipping cyclone feed. The steel charge is made up on a daily basis with 80mm diameter, high-chrome balls, usually in proportion to the feed rate, adjusted monthly as required according to the mill load. Refer to figure 2.2.

## 2.2 Instrumentation

### 2.2.1 Mill Feed Size Measurement

The mill feed particle size is measured using an optical unit known as the "Lynxx". This was developed for Anglo Platinum by Stone Three Signal Processing Limited due to problems experienced with similar units which were commercially available.

The Lynxx software operates by using image processing techniques on the input video. Image frames are captured from the input video. Each image then forms the input to a segmentation procedure. The segmentation is used to separate visible rock fragments from the conveyor belt and fine material in the input image. Lynxx displays the actual segmentation for each input image. Given the boundary points of each rock fragment, the rock sieve-size and the

two-dimensional rock area as seen by the camera are determined. The rock volume for each of the fragments is then estimated by fitting an ellipsoid to each of the rock fragments. After the sieve size and volume for each rock fragment has been computed, a metric calibration factor is applied. This converts distance in the input image to actual distance. Typically, the distance between two pixels in the input image would correspond to approximately 1-2mm.




Figure 2.3: "Lynxx" Optical Sizer on a Mill Feed Conveyor

The rock-mass for each rock fragment is then estimated by multiplying the volume by a density factor for the specific ore type. This simply scales the

result and does not affect histograms. The rock mass of each fragment is then binned according to its sieve size in a set of discrete sieve size ranges. An un-normalised set of rock fragment mass bins versus sieve size now exists. The un-normalised rock mass bins are likely to vary rapidly, as each image processed could have a significantly different rock-size distribution from its predecessor. The un-normalised rock mass bins are therefore low-pass filtered over time. This filtered un-normalised rock mass versus sieve size bin set is normalised to form a time-filtered rock mass versus sieve size histogram. A cumulative fraction-passing indicator is also computed, given the above histogram.

The Lynxx system has been shown to perform well in terms of accuracy although this deteriorates with increased loading. An accuracy evaluation in which pre-sized feeds were presented to the instrument indicated that the error in each size fraction is of the order of 10-15% but that trends are accurately followed. (Plint et al, 2004).

Nine of these units are currently operational.

### 2.1.2 Mill Product Size Measurement

The mill product size is measured using an instrument known as the "UltraPS" developed and marketed by the CSIRO of Australia. The unit at the Waterval UG2 plant is the one of fourteen such installations in Anglo Platinum.



Figure 2.4: "Ultra PS" Particle Size Measurement System.

The UltraPS uses the measurement of ultrasonic velocity and the attenuation of pulsed multiple frequency ultrasonic waves passing through the slurry. The measuring unit includes a gamma transmission gauge which is used to correct for any variations in the density of the slurry. From the measured ultrasonic velocity and attenuation, particle size is obtained by applying mathematical inversion techniques to provide a full particle size distribution. This is correlated with particle size cut points determined by laboratory analysis to provide a calibration equation. The output consists of 31 points of particle size distribution and solids percentage and slurry temperature. The size points are then converted into cumulative passing size ranges typically understood by operational staff. These are, -212 µm, 150 µm, -106 µm, -75 µm, -53 µm, -38 µm, -25 µm and -10 µm. (Kumalo, 2002).

Table 2.3 Process Instrumentation at Waterval UG2 Primary Milling Circuit

| **Measurement** | **Instruments** |
|---|---|
| Waterval mine ore feed | Electro-mechanical weightometer |
| Rustenburg mine ore feed | Electro-mechanical weightometer |
| Coarse ore silo | Ultrasonic Level |
| Fine ore silo | Ultrasonic Level |
| Fine ore mill feed | Electro-mechanical weightometer |
| Total mill feed | Electro-mechanical weightometer<br>Rock Particle Sizer (Lynxx) |
| Mill load | Loadcells under mill inlet bearing. |
| Mill inlet water | Magnetic induction flowmeter |
| Combined trommel spray and mill discharge sump dilution water | Magnetic induction flowmeter |
| De-chipping cyclone feed, operational and standby lines. | Nuclear pulp density meter<br>Magnetic induction flowmeter |
| Screen undersize. | Nuclear pulp density meter<br>Magnetic induction flowmeter<br>Particle size range. (UltraPS) |

# 3 MODELLING TECHNIQUES

It was proposed that the Waterval UG2 mill be modeled using the techniques developed by the Julius Kruttschnitt Mineral Research Centre, (Napier-Munn et al., 1999). There are essentially three different models which are relevant:

- The Power Draw Prediction model.
- The Whiten ball mill model.
- The AG/SAG mill model.

The models are discussed in detail in the reference above and are discussed below in terms of their relevance to the mill in question. Data analysis uses the JKSimMet modelling package which is based on these models.

## 3.1 The Power Draw Prediction Model

The power drawn by a mill is one of the few operational parameters which can be accurately measured. The inputs to the model are mainly physical mill measurements which can also be accurately determined. These are:

- Diameter inside liners
- Belly length inside liners
- Centre length inside liners
- Trunion diameter inside liners
- Fraction of critical speed

- Ore relative density
- Ball relative density
- Discharge slurry: % solids by weight
- Total fractional mill filling
- Ball fractional mill filling

A good match between actual and measured power under steady state conditions would then indicate that the mill parameters have been correctly measured and give some confidence in the use of the JKMRC models in this application.

Although the model is incorporated into the JKSimMet programme it was decided that re-coding the model into a spreadsheet would be useful as it enables comparisons between test results to be made more easily. This was done and the model applied to individual data records as described below.

This model assumes, for ball mills, that the fractional mill filling and the fractional ball filling are the same. Any rock or pulp is therefore assumed to fill the inter-ball spaces. Visual inspection of the interior of the Waterval UG2 mill during maintenance indicates that pebbles comprise a significant part of the load. In order to estimate the fractional fillings it was necessary to

measure the actual filling after a crash stop (fractional mill filling) and after an extended grind-out (fractional ball filling).

A second assumption is that slurry discharge from a grate-discharge mill is perfect and that no pooling occurs. Pooling has, however, been observed in other Anglo Platinum grate discharge mills, possibly due to insufficient open area in the grates or to excessive return flow through grates which conform to the conical mill-end shape. Pooling decreases the power draw of the mill by providing a balancing force on the downward-turning side of the mill. The extent of any pooling can be estimated by measuring the thickness of the water and slurry layer above the charge after a crash stop.

## 3.2 The Whiten Ball Mill Model

The model considers a ball mill as a perfectly stirred tank. For any feed particle size, the mill output is a product size distribution which is defined by a breakage function. The model then requires data on feed and product size distributions at steady state, which in this case are available from the plant instrumentation. The breakage function may be determined by a standardised drop-weight test. This has been done for Waterval UG2 ore.

The major problem with this model as applied to a ROM Ball Mill is that it was developed using a maximum feed size of 20 mm, although any feed size range can be used as the input.

The model may be used for comparative purposes by adjusting factors such as mill size, load, speed, and ball size. In this study it was proposed to input the known parameters to obtain a standard model. The simulation function would then be used to calculate new breakage functions at different feed size distributions in order to determine the effects of feed top size.

## 3.3 The AG/SAG Mill Model

This model has three components; a breakage rate, the resulting size distribution function and a discharge rate. The breakage rate algorithm is not detailed in the literature but is explained as a function of feed size, feed rate, mill filling and load, and is predicted using a perfect mixing mass balance model. This is the primary function which is changed in order to fit the model to the experimental data.

The appearance function is obtained from two laboratory tests. The first determines a high energy function. A standardised drop-weight test is used to measure breakage at different energy inputs and the percentage of material passing one tenth of the original size is determined. This is referred to as the

t10. The t10 is related to the energy input, (specific comminution energy, Ecs) by the equation:

$$t10 = A(1-e^{-b.Ecs})$$

Fitting this curve gives the A and b values which are used as the model input. Testwork has shown that the relationships between t10 and other breakage sizes may be represented as a standard series of curves. Thus the appearance function for a range of sizes can be determined from the t10, or A and b values.

Similarly, a standard milling test is used to give a low energy function and the two results are combined to give a total appearance function. The discharge rate is calculated using Bernoulli's equation, taking into account the mill volume filling, mill diameter, mill speed and the attributes of the grates. As with the ball mill model, study it was proposed to input the known parameters to obtain a standard model. The simulation function would then be used to calculate new breakage functions at different feed size distributions in order to determine the effects of feed top size.

# 4 LONG-TERM DATA ANALYSIS

## 4.1 Data Capture

The initial data set comprised relevant information from the plant historian at five minute intervals. The period is from the 27th of October, 2005 to the 30th November, 2005. During this period the mill feed size data supplied from the Lynxx was not captured on the historian due to the failure of the optical fibre link. This data was subsequently acquired separately from the Lynxx control computer. Problems were also experienced with the product particle sizer due to choking of the detector by wood chips. This eventually resulted in plant staff switching off the instrument. Similar problems at other installations have been successfully overcome with the installation of a sieve bend on the sample pipeline and a similar unit has now been fitted at Waterval.

This leaves a data set with approximately 9600 records at five minute intervals.

## 4.2 Data Processing

The data set was inspected and the following operations carried out:

- All “null” values were deleted.

- When the mill was standing, as indicated by the power reading, the values of the other variables were deleted. This affected 503 records in sixteen occurrences ranging in length from 5 minutes to 23.5 hours and amounting to 5.2% 0f the total. Five of the occurrences were of ten minutes or less and it is the opinion of the plant instrumentation staff that these are missing records due to data transfer conflicts rather than mill stops.

- The UltraPS results are the least complete records as discussed above. There are 3971 complete records, only 41% of the total. Where the gaps in the data consisted of one or two missing records, these were then interpolated by filling in an average. The number of such substitutions was 401, increasing the number of complete records to 45%. The resulting data set was again inspected and all isolated blocks of UltraPS data consisting of less than six records, half an hour, were deleted as being

of no significance. This affected 99 records leaving 4273 or 44% of the original total. The effects of these manipulations may be seen in Appendix A, where the percentage of + 150 micron material is used as an example.

Additional information was then calculated from the base data. This included:

- Mill power was recalculated from megawatts to kilowatts.
- Mill volume filling was calculated from mill load.
- Silo level readings were averaged from two instruments.
- Single operational cyclone feed flows and densities were calculated from readings from operating and standby equipment.
- Mill discharge slurry density was calculated from the cyclone feed flows and densities with mill discharge sump dilution water subtracted.
- Circulating load, defined as the difference between the solids flowrate in the cyclone feed and the mill feed rate, expressed as a percentage of the mill feed.

Lynxx data, contained in daily comma-separated-variable files was appended to the data from the plant historian. This data was time-stamped but not recorded at discrete intervals. A significant amount of manual data manipulation was

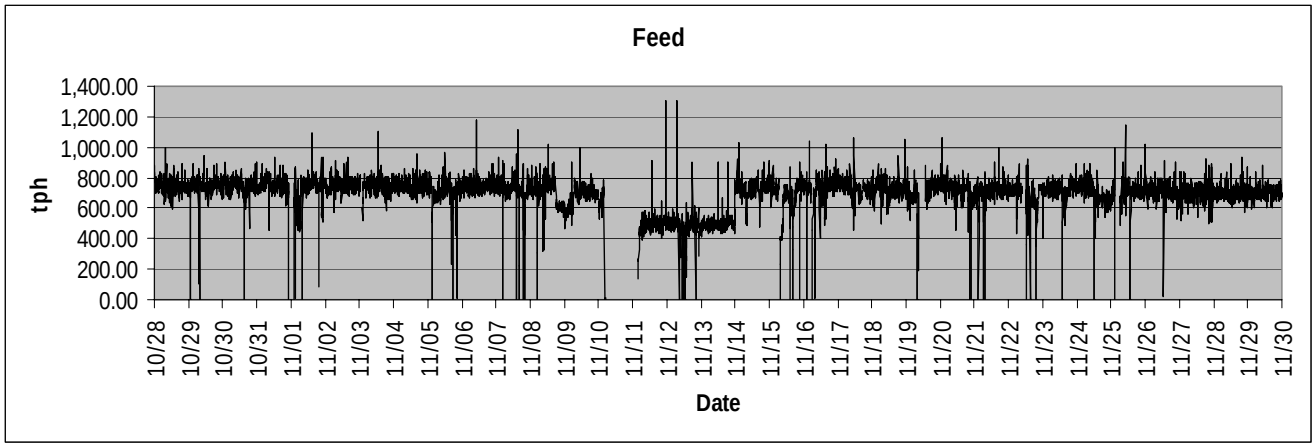
required in order to match it with plant data. It is also unfortunate that the instrument was not restarted immediately after the plant shutdown, resulting in the loss of five and a half days' data.

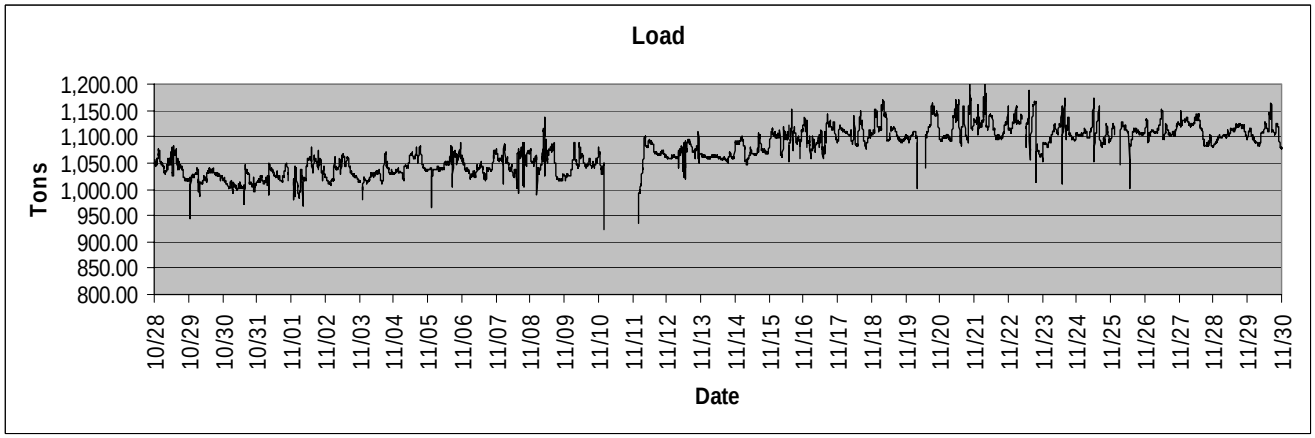
## 4.3 Data Inspection

The mill feed data suggests three distinct periods of operation:

- An initial period from 27/10/2005 to 10/11/2005, ending at data point 3905, during which the average feed rate was relatively constant at an average of 725 tph. The setpoint varied between 700 tph and 750 tph with an average of 737 tph.
- After a maintenance shutdown there follows a period of lower throughput. Between 11/11/2005 and 13/11/2005 the setpoint was 500 tph, with an average of 490 tph being achieved.
- Sixteen days of relatively constant feed follow, the setpoint mainly constant at 700 tph to give an average of 701 tph.

These periods will be referred to as Period LT1, Period LT2 and Period LT3 respectively.



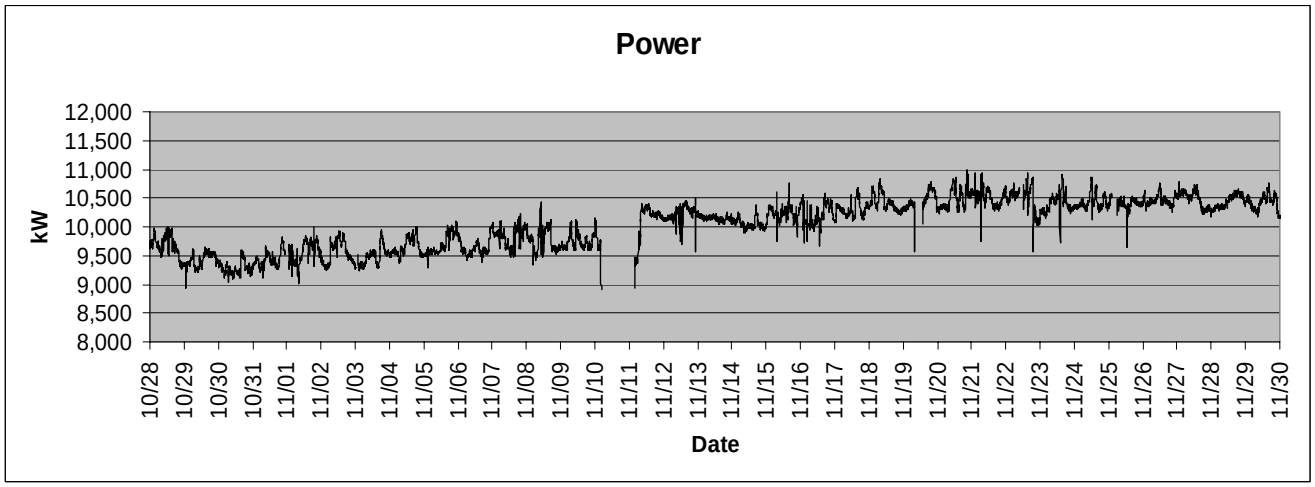
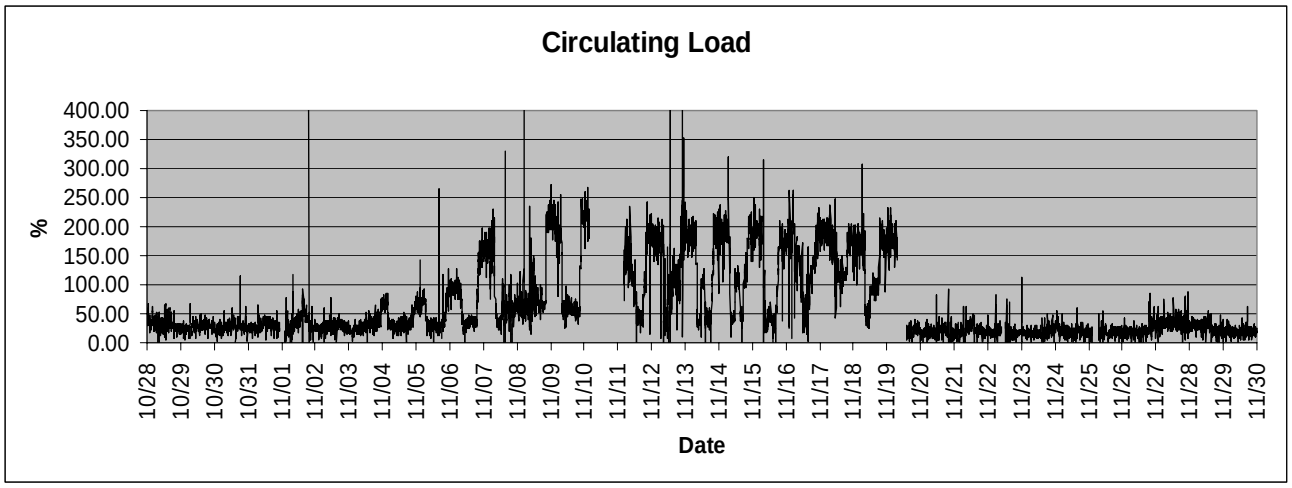
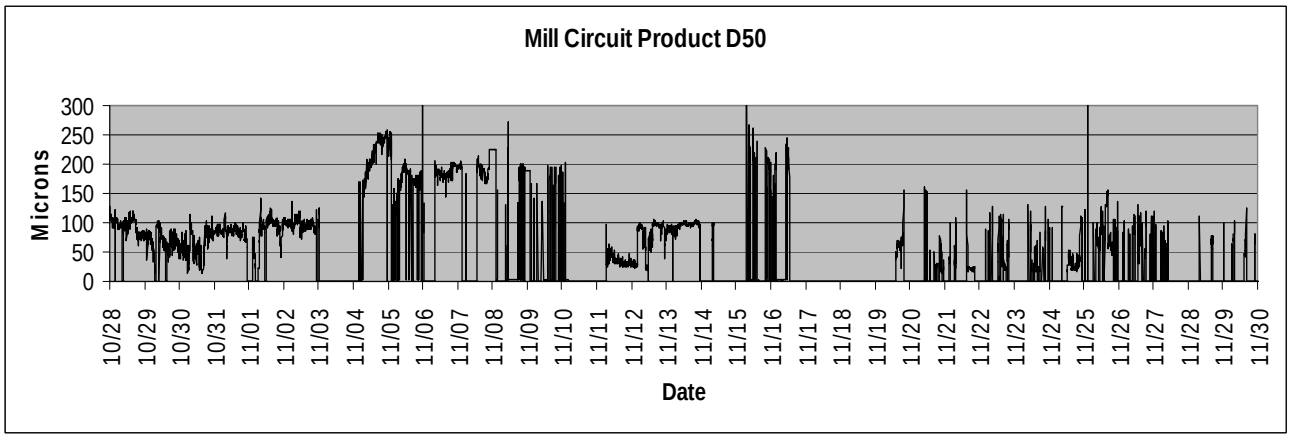




Figure 4.1 Time Series Graphs of Significant Readings



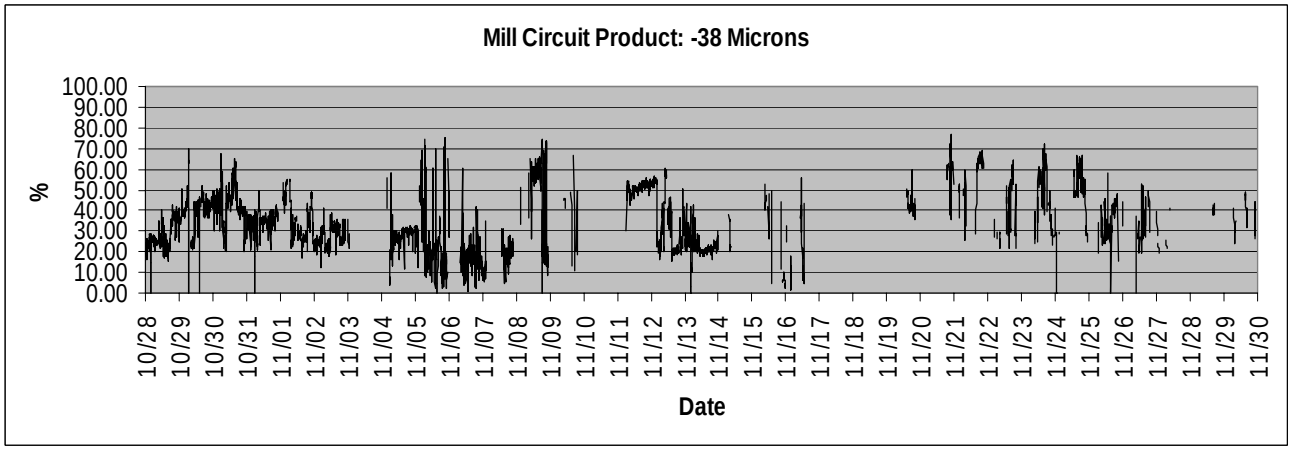
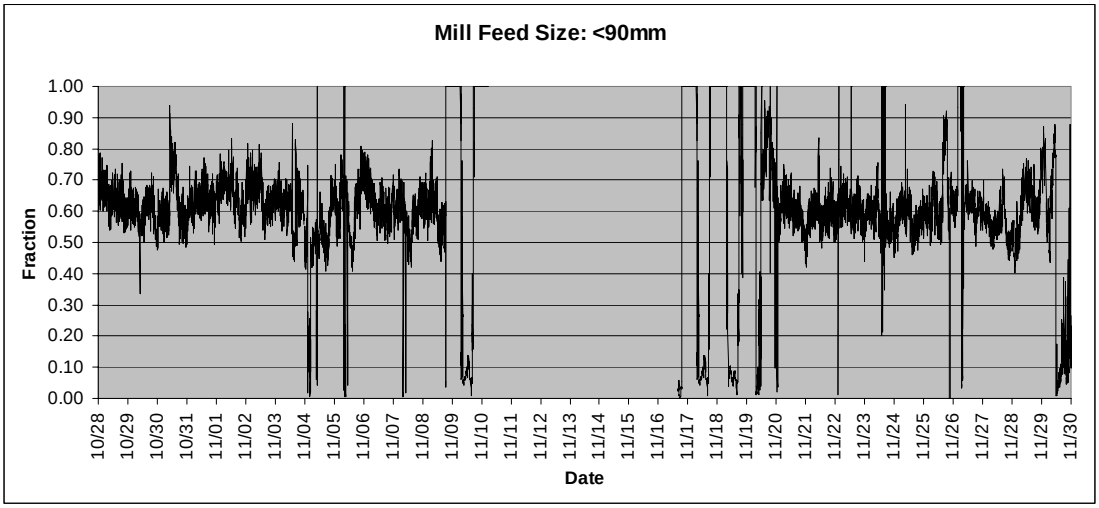




Figure 4.1 (Continued) Time Series Graphs of Significant Readings



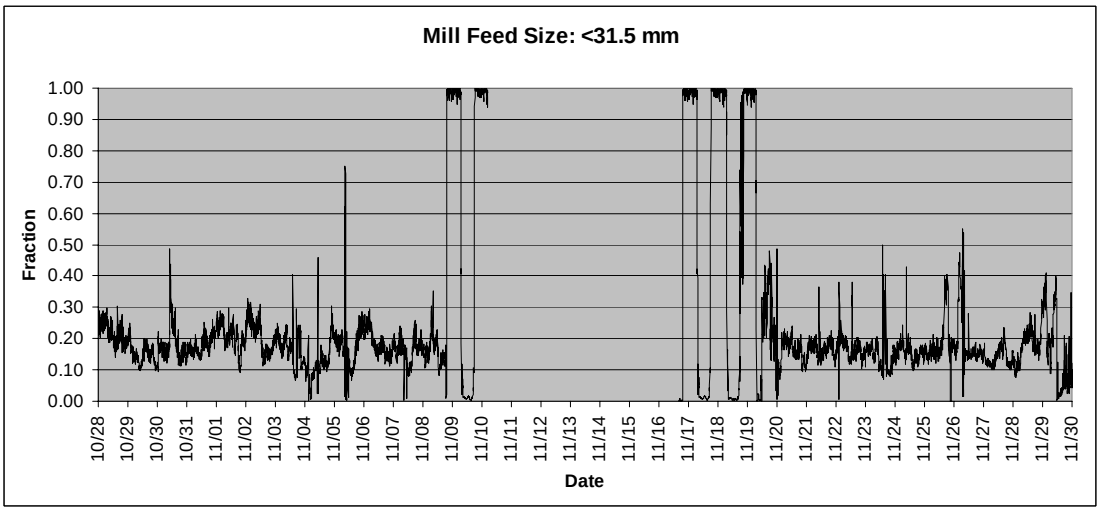




Figure 4.1 (Continued) Time Series Graphs of Significant Readings

The most obvious factor from the graphs is the strong correlation between the mill load and the power drawn. The correlation factor in this case is 0.92 shown in figure 4.2. This is typical ball mill behaviour. The strong correlation also indicates that the mill does not run at its peak power draw i.e.: increasing the load should increase the power consumption but this is not possible due to the limiting size of the mill motors. The corollary to this is that the mill is some way from optimum energy efficiency as also indicated by the degree of filling, 30 %.

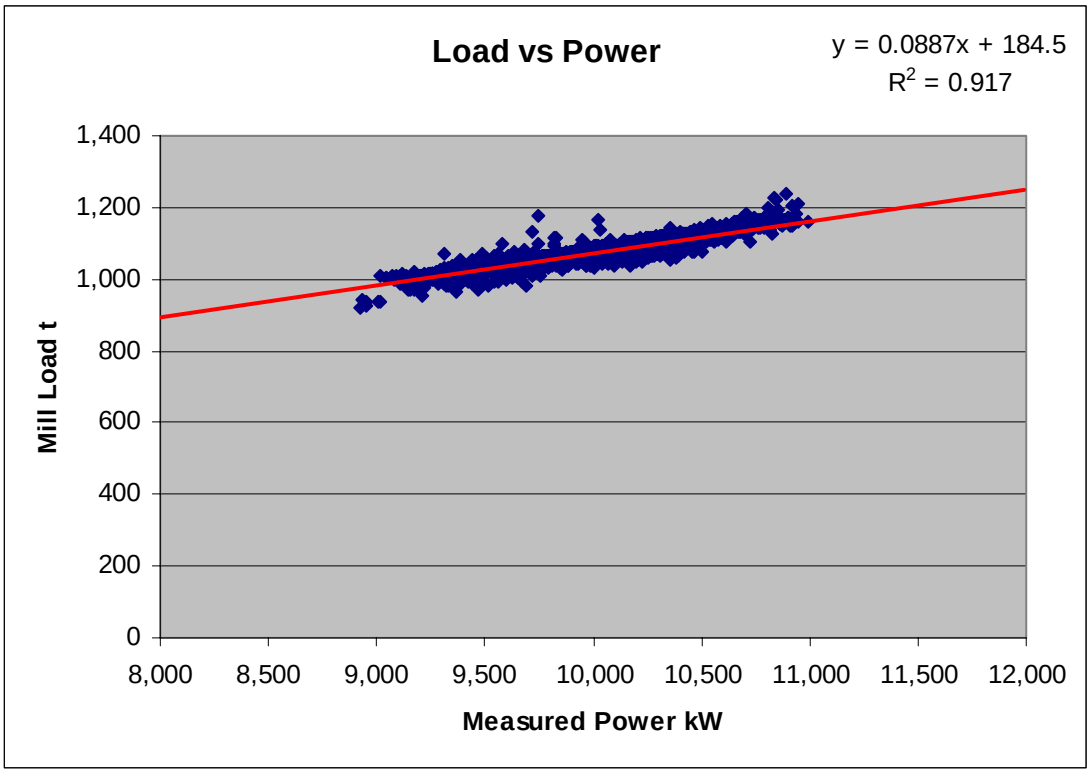



Figure 4.2 Mill Load vs Power

This result may have immediate practical application. Load measurement has proved to be a problem with large mills as failure of one of the load-cells or drift of the measurement can cause severe overloading and consequent inefficiencies. Use of a smoothed JK power estimation, which is largely load based, could provide a warning signal if run on-line.

During Period LT1 the overall effect of the constant feed rate is to cause a gradual increase in the mill load from approximately 1000 tons to 1050 tons with a corresponding increase in power. In Period LT2, after the shut-down, the feed rate is decreased and there is a slight increase in the load and power. Increasing the feed at the beginning of period 3 causes the load to increase and steady at approximately 1100 tons.

It can therefore be concluded that the power drawn by this mill is almost entirely dependent on the mill load. The mill load is influenced to a degree by the feed rate, although the reaction is slow, taking from two to ten days to reach a steady state. Plant staff indicated that the steel ball addition during this period was constant at 1.8 tons per day, equivalent to 1 kg of steel per ton of new feed, although it is suspected that additional steel balls may have been added. A gradual increase in ore hardness is also a possibility.

Table 4.1 Comparison Between Periods: Average Values

| **Variable** | **Period LT1** | **Period LT2** | **Period LT3** |
|---|---|---|---|
| Mill Feed (tph) | 737 | 490 | 701 |
| Mill Power (kW) | 9599 | 10166 | 10387 |
| Mill Load (tons) | 1038 | 1065 | 1108 |
| Circulating Load (%) | 54 | 123 | 61 |
| kWh/t | 13.3 | 20.9 | 14.8 |
| Feed percent -90mm | 61 | n/a | 59 |
| Feed percent -31.5mm | 18 | n/a | 9 |
| Mill Circuit Product $P_{50}$ mm | 99 | 70 | 55 |

Also obvious is the extreme variability in the circulating load. This is of interest as it does not correlate with feed rate or cyclone feed density. (See figure 4.3) There does appear to be some interaction on a short time-scale but this may be due to disturbances in the circulating load propagating back through the mill. The silo levels also appear to have some influence on the circulating load as changes in the direction of silo level change seem to coincide with changes in the circulating load. (See figure 4.4).

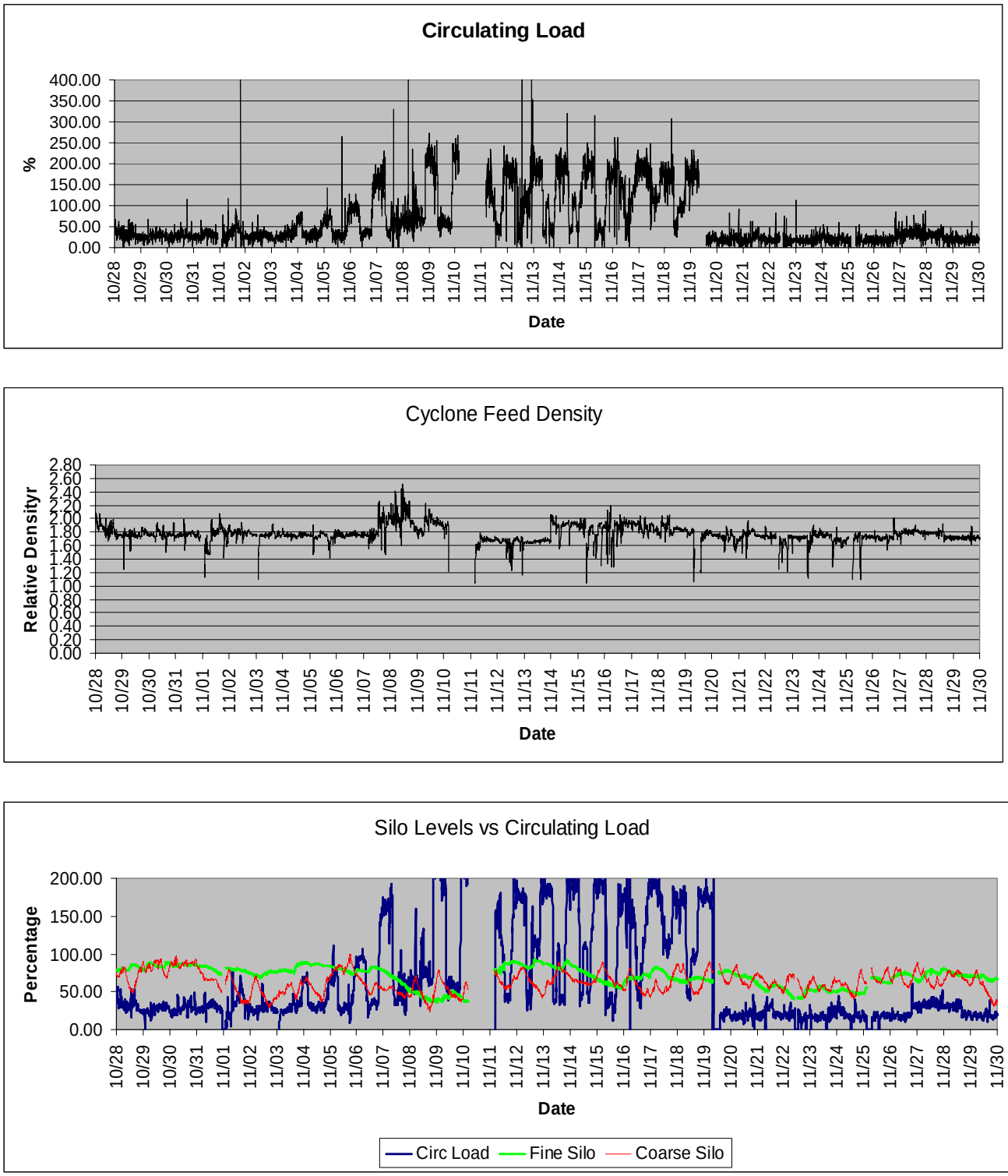



Figure 4.3 Time Series Graphs of Circulating Load and Associated Variables

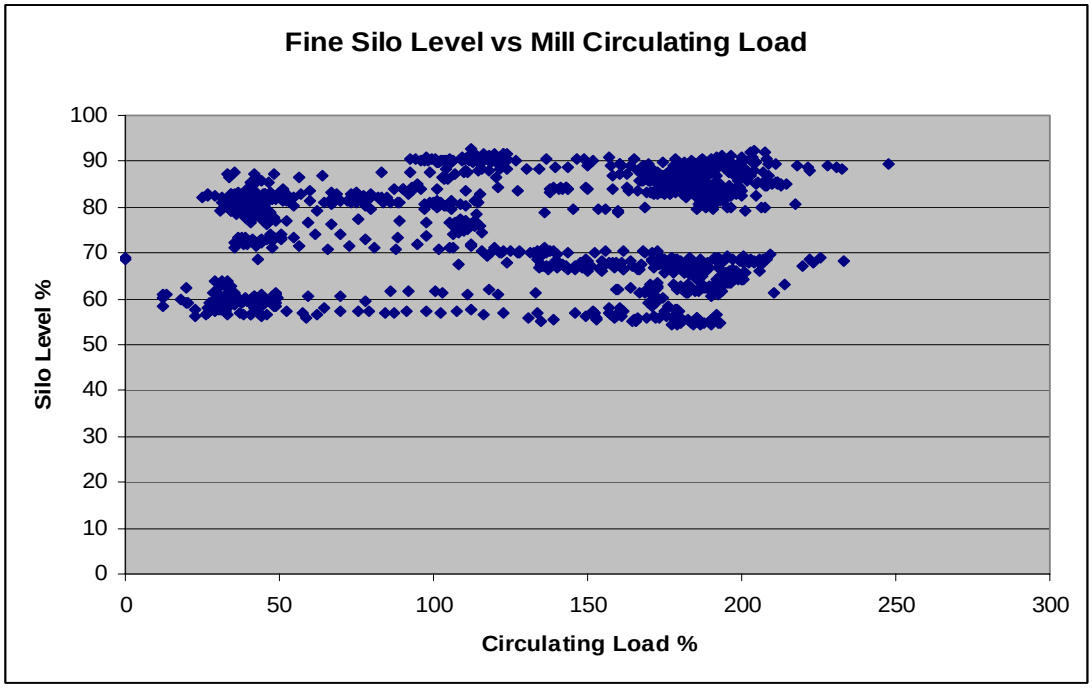



Figure 4.4 Graph of Fine Silo Level and Circulating Load.

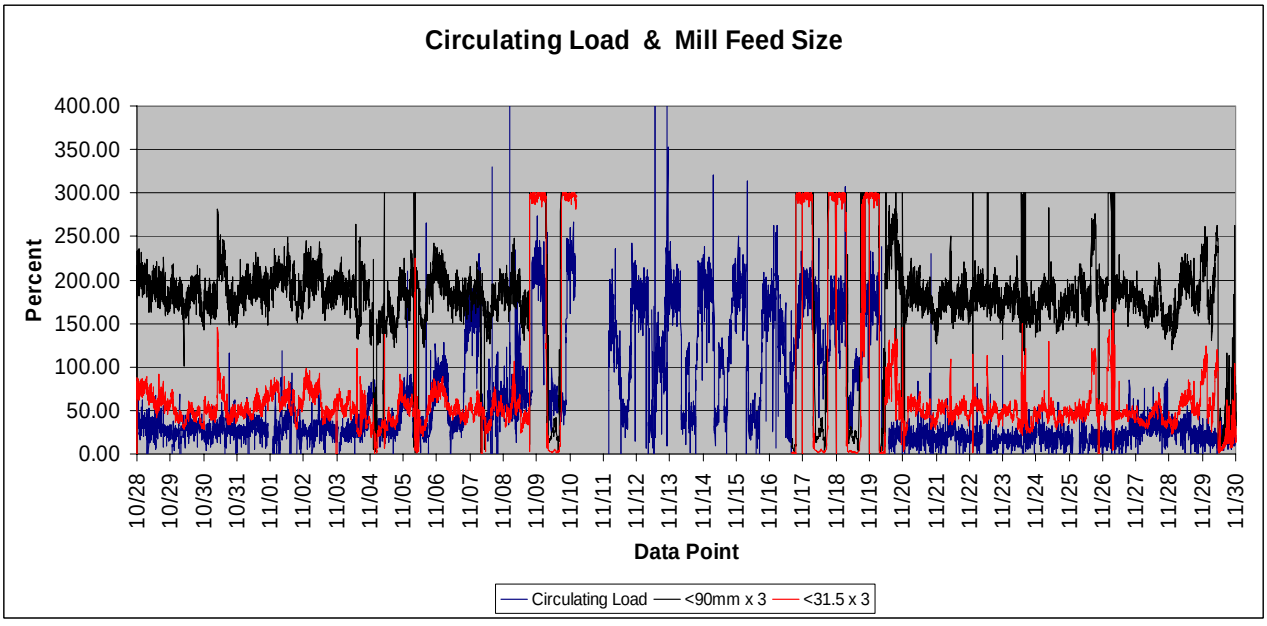



Figure 4.5 Time Series Graph of Circulating Load and Mill Feed Size.

( Nb: Mill feed size percent passing value multiplied by 3 for comparison)

The reason for the variability in the circulating load is not obvious from the data. It is actually due to overloading of the screens. This does not appear to be related to the feed rate but may well be a result of a change in the sizing of the ore feed. This is indicated by figure 4.5. When the ore feed was fine, all less than 31.5mm, (indicated in figure 4.5 by the red line peaking at 300), the circulating load increased from the normal 30% to 200%. This is probably caused by a lack of coarse media in the mill resulting in a decrease in the surface area available for grinding and so resulting in a coarser discharge. When the ore feed was coarse, only about 10% less than 90mm, (indicated in figure 4.5 by the red line bottoming at 30), the circulating load decreased to 40% - 50%, still higher than the normal 30%.

In the areas where the circulating load is relatively stable, inspection shows that increasing the proportion of fines in the feed material generally increases the circulating load. Comparison with the mill product sizing is approached with caution due to the known problems with this reading. Period LT3 is therefore not taken into account. The remaining data can also be resolved into three periods, as seen in figure 4.6:

Period a: Dates 28 October to 3 November

Here the product size appears to be inversely related to the mill feed sizing. An increase in the quantity of coarse material results

in the coarsening of the product, but the relationship is not very clear.

Period b: Dates 4 November to 8 November

Increased instability in the circulating load appears to result in a coarsening of product size. As discussed above, the instability is triggered by screen overloading.

Period c: Dates 11 November to 13 November

No particular trends can be discerned during this period.

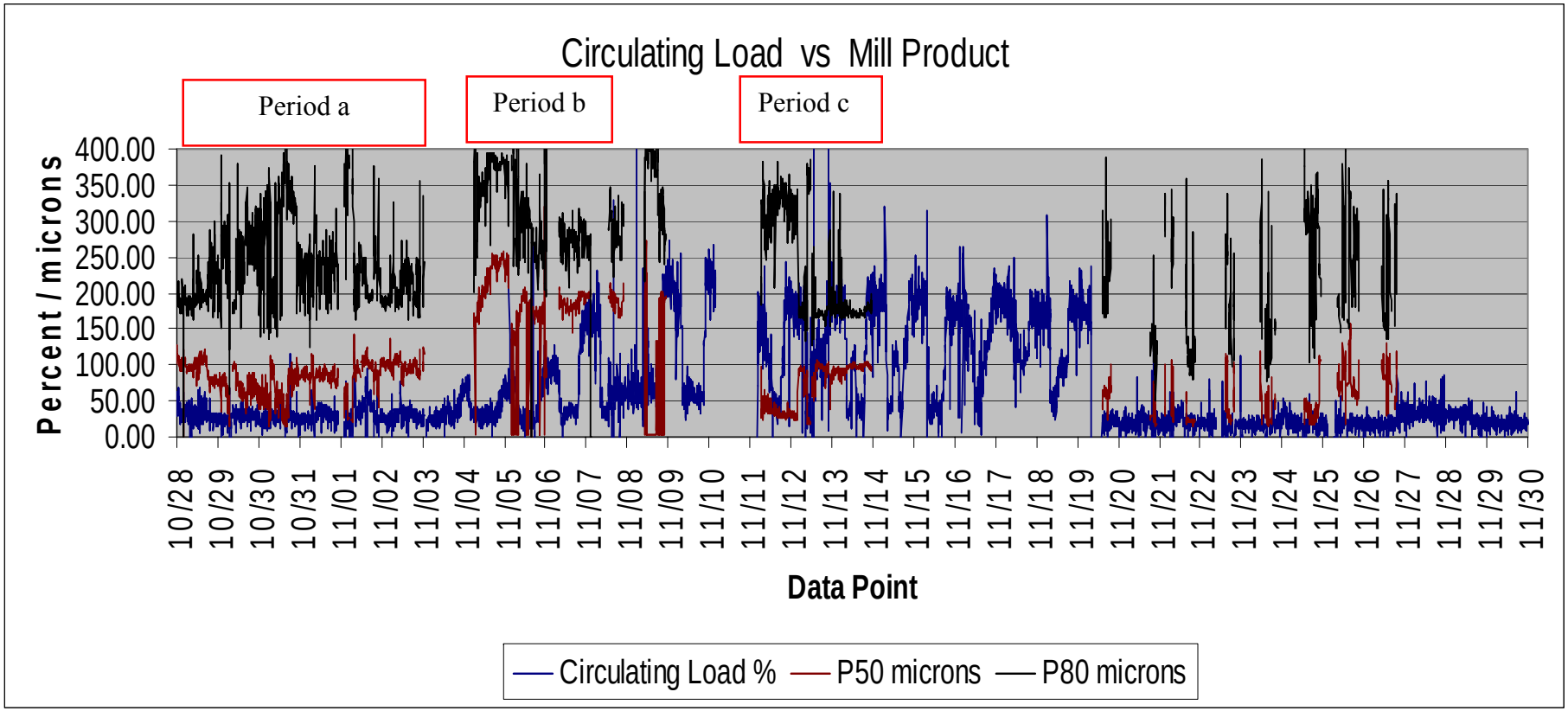



Figure 4.6 Time Series Graphs of Circulating Load and Mill Product Size.

## 4.4 Modelling

### 4.4.1 Power Model

As discussed in 3.1, above, the JK power model was programmed as an Excel spreadsheet. The resulting power estimation equation was then applied to the plant data for each five-minute period. As the model was developed as a design tool it was not expected that the fit would be good. However, the resulting array compares favourably with the measured values, the correlation being 0.93. The complete trend is shown in figure 4.7, and the comparison between model and measurement in figure 4.8.

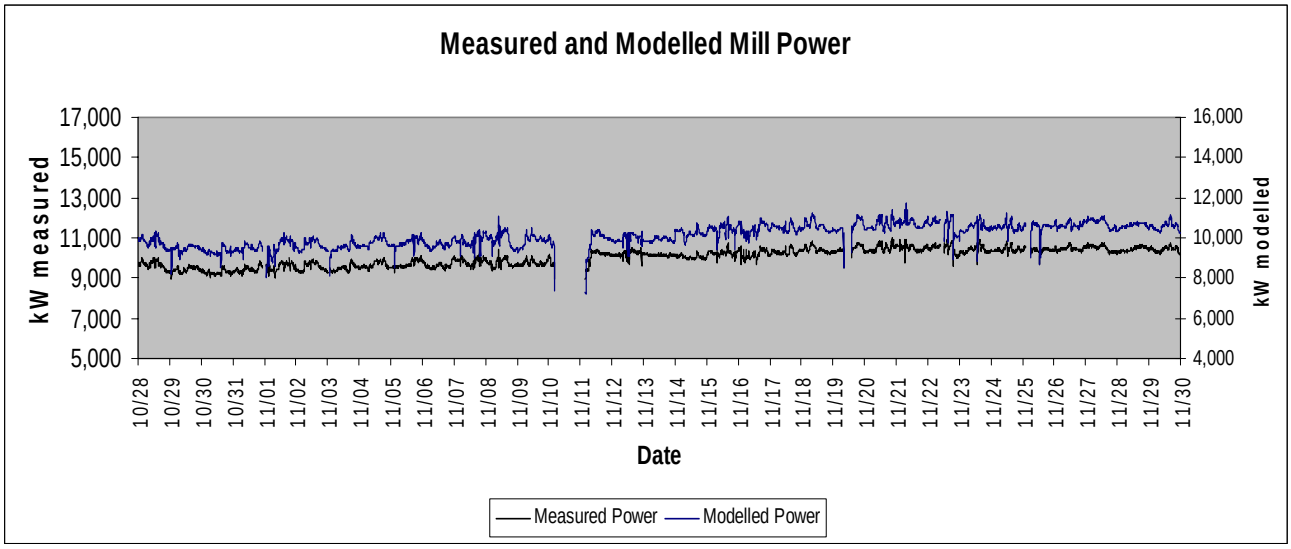



Figure 4.7 Time Series Graph of Measured and Modelled Mill Power.

(NB: Different scales have been used to separate the lines.)

Downward spikes in the modelled curve represent mill stops; the low densities measured on starting result in the prediction of a low in-mill density and hence low power.

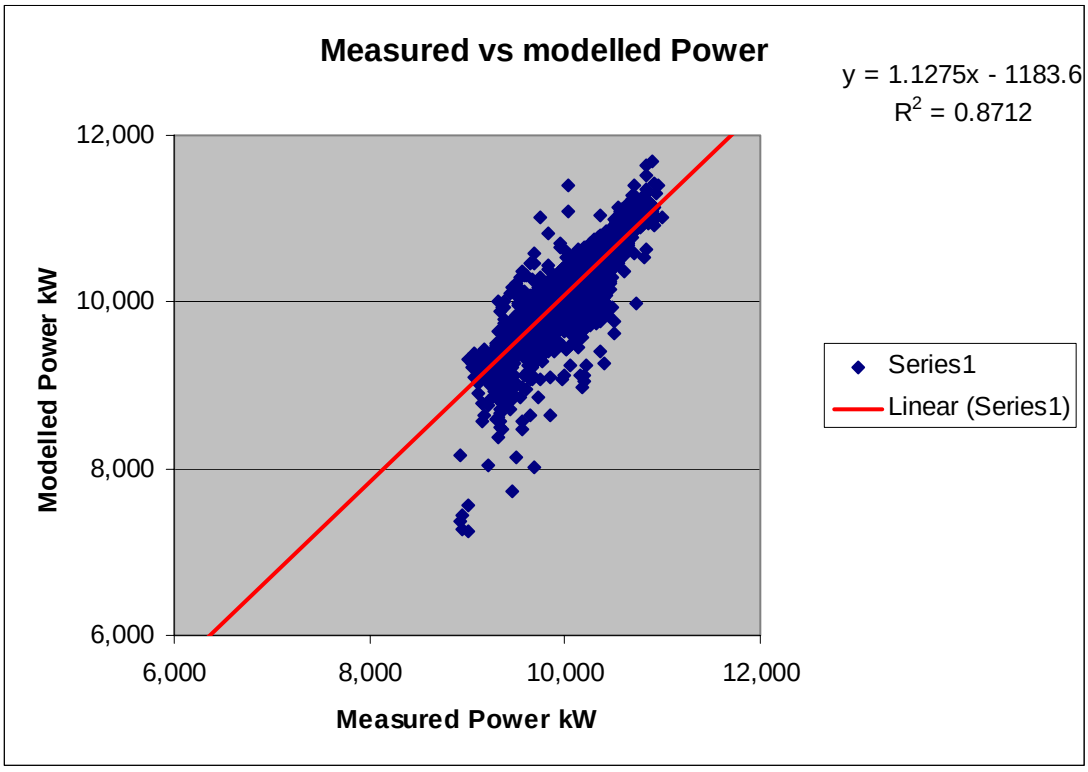



Figure 4.8 Graph of Measured vs Modelled Mill Power.

### 4.4.2 JK Ball Mill Model

The good fit of the power model appeared to indicate that the basic modelled variables were correct and the averaged data for the first 2500 points were chosen as representing relatively stable mill operation. The circuit was modelled in its simplified form as illustrated in figure 4.9.

The cyclones and other woodchip removal steps were ignored as having little effect on the overall circuit.

Fitting the average values to the model resulted in the circulating load being adjusted, the model suggesting a value of 209 tph rather than the 258 tph calculated from the cyclone feed flow and density readings. The final product size gives a good fit at the coarser sizes, less good at finer sizes., although the correlation coefficient is 0.97, over a somewhat limited range. This can be seen in figure 4.10.

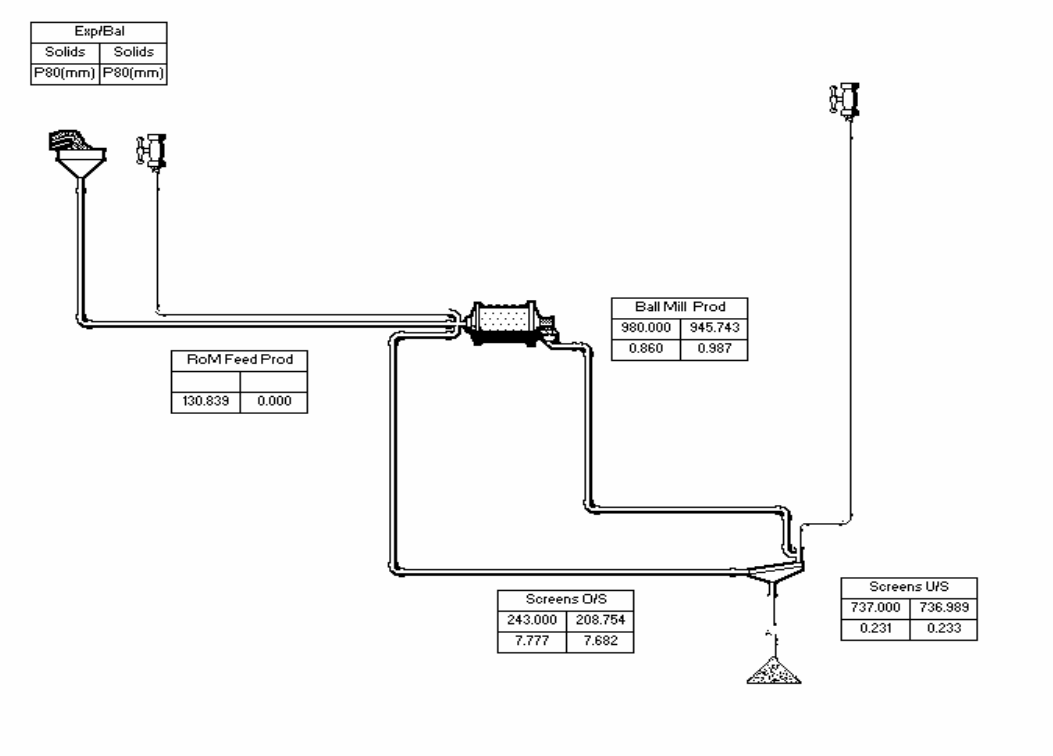



Figure 4.9 Modelled Ball Mill Flowsheet..

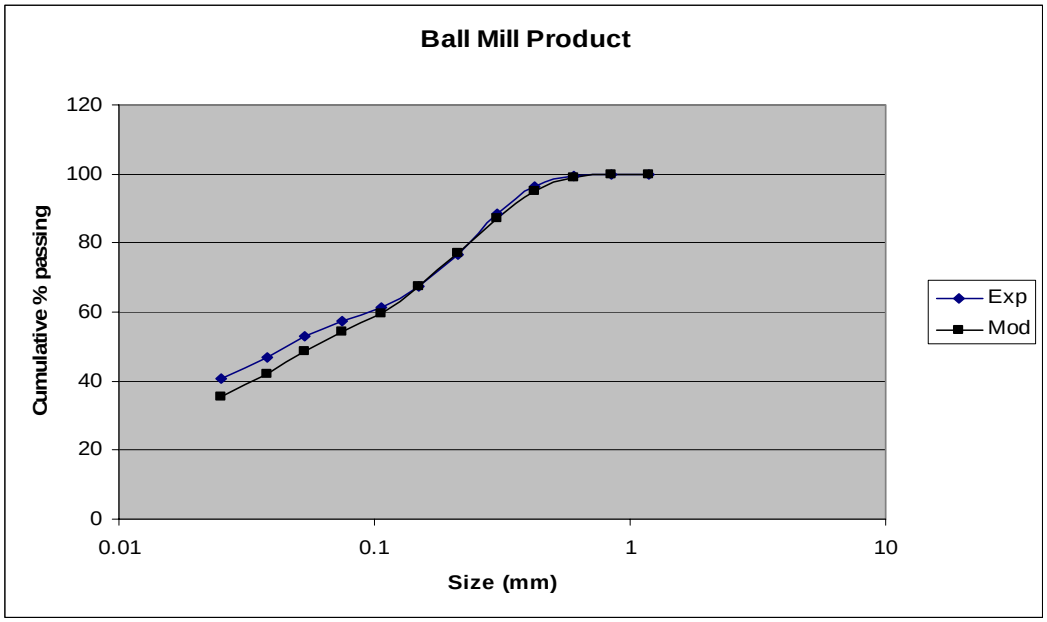



Figure 4.10 Modelled and Measured Mill Circuit Product Sizing: Period 1

Given a reasonable fit of the model, the next step was to change the feed size to represent one of the disturbed periods where the mill feed was finer. The circulating load during this period was 181 % of new feed. The model, however predicts 67 %. The resulting product sizing also does not fit the results, as shown in figure 4.11. The final p80 is unchanged, but the model predicts a coarser final product. This is possibly due to the screen overloading discussed above.

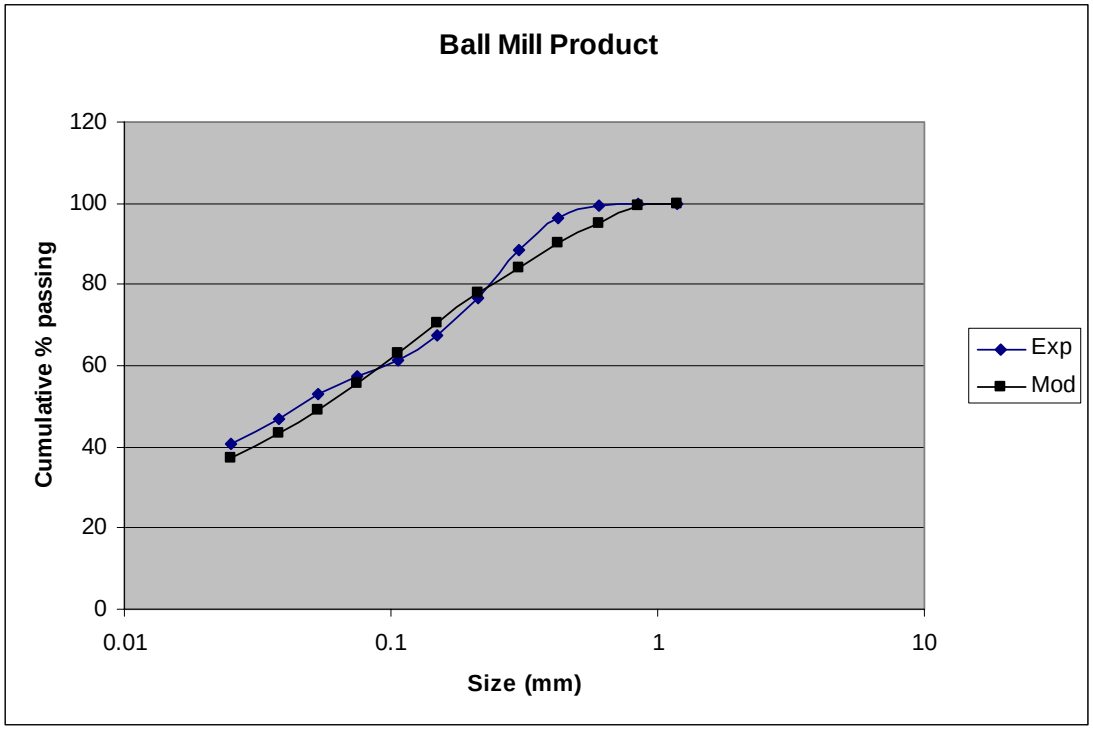



Figure 4.11 Modelled and Measured Mill Circuit Product Sizing: Fine Feed

At this stage it was advised that the ball mill model was not suitable for ROM Ball Mill modelling due to the upper limit of the feed size range for the model being only 20 mm. In addition, the model does not cater for the inefficiencies resulting from screen overloading, which is the cause of the increase in circulating load.

Concerns as to the calibration of the particle size monitor also resulted in the decision not to attempt further modelling on this data set.

## 4.5 Conclusions: Long-Term Data

Several conclusions can be drawn from the analysis of the Long-term Data set:

- There is a strong correlation between the mill load and the power drawn.
- The silo levels appear to influence the circulating load, as changes in the direction of silo level change coincide with changes in the circulating load. It has been seen that during filling of the silo coarse rocks tend to flow to the circumference of the silo and that the off-take contains more fine material. Similarly, during emptying of the silo the larger material is drawn to the middle and the off-take coarsens.

- Instability in the circulating load results in the coarsening of product size. This instability is triggered by screen overloading.
- There is some evidence that a coarser feed results in a coarser product but no definite conclusions can be drawn on the effect of feed size on the mill circuit product due to the overwhelming effect of the screen overloading on the circulating load.
- The mill can be modelled as a ball mill using JKSimMet, and the model does react to changing feed sizes but is not very accurate, probably due to the model limitation of feed top size to 20mm.

# 5 SHORT-TERM DATA ANALYSIS

Following a plant visit on 3rd May 2006, and recommendations that detailed analysis of a typical day would be of benefit, a series of step-tests was undertaken. Monday, 15th May was chosen as being the least disruptive time for the plant and the mill feed was increased in discrete steps.

The initial feed rate was planned to begin at 550 tph, increasing in 50 tph steps every four hours to 700 tph. Some changes proved necessary, resulting in the final programme as follows:

| | | |
|---|---|---|
| Period ST1 | 06h00 to 10h00 | 550 tph |
| Period ST2 | 10h00 to 14h00 | 600 tph |
| Period ST3 | 14h00 to 16h30 | 650 tph |
| Period ST4 | 16h30 to 23h00 | 700 tph |
| Period ST5 | 23H25 to 02h30 | 700 tph |
| Period ST6 | 04h15 to 06h00 | 600 tph |

Period ST3 was shortened as plant management wished to increase the throughput at the end of the day shift. At 20h30 the operation was left to the plant personnel. The feed rate was then kept constant at 700 tph until approximately 02h30 the following morning. A feed interruption at 23h00 provides a convenient break. Period ST5 was then taken to occur from the

restart at 23h25 to 02h30 when feeder problems caused a series of disturbances. Finally, Period ST6 between 04h15 and 06h00 completes the day with a reduced feed of 600 tph.

The discrete periods chosen for analysis can be seen in the graph of the feed rate shown below (figure 5.1)

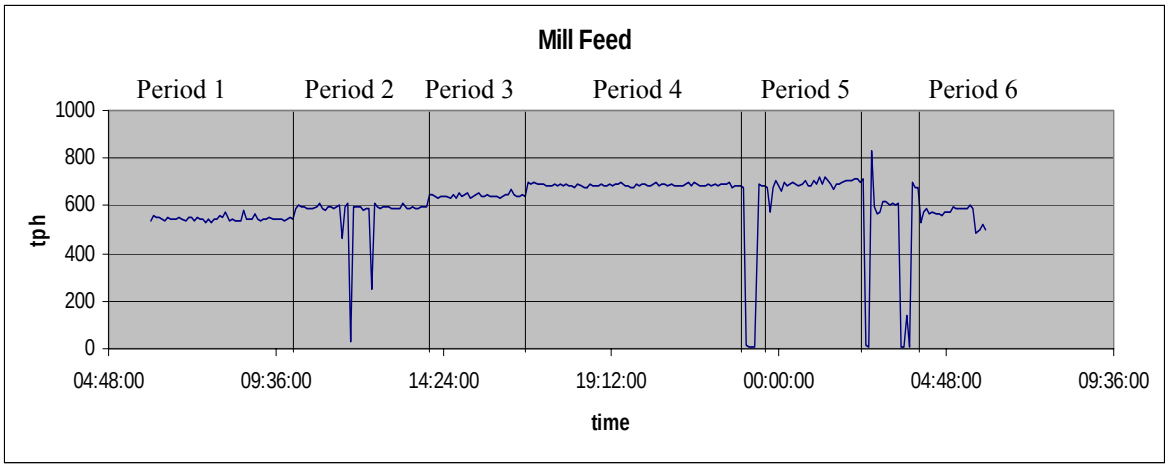



Figure 5.1 Analysis periods according to feed.

## 5..1 Data Processing

The data set was inspected as detailed in the previous chapter and no interpolations or deletions from the data set were considered necessary.

Additional information was calculated from the base data, as before:

- Mill power was recalculated from megawatts to kilowatts.
- Mill volume filling was calculated from mill load.
- Silo level readings were averaged from two instruments.
- Mill discharge slurry density was calculated from the cyclone feed flows and densities, less the mill discharge sump dilution water .
- Circulating load, defined as the difference between the solids in the cyclone feed and the mill feed, expressed as a percentage of the mill feed.
- Plant Work Index, PWI, was calculated as:

$$PWI = \frac{\text{Mill Power per Feed ton}}{\frac{10}{\sqrt{P_{80}}} - \frac{10}{\sqrt{F_{80}}}}$$

Where $P_{80}$ is the size at which 80% of the product is finer than that size and where $F_{80}$ is the size at which 80% of the feed is finer than that size.

Lynxx feed particle size data, contained in a daily comma-separated-variable file, was appended to the data from the plant historian. A significant amount of manual data manipulation was required in order to match it with plant data.

## 5.2 Data Inspection

Table 5.1, below, gives the averages and standard deviations of selected variables for the six periods and the averages for the day.

Table 5.1 Averages and standard deviations during six time periods.

| | | ST 1 | ST 2 | ST 3 | ST 4 | ST 5 | ST 6 | Ave |
|---|---|---|---|---|---|---|---|---|
| | | | | | | | | |
| **Mill Mass** | t | 993 | 996 | 1005 | 1007 | 995 | 992 | 998 |
| | SD | 1.42 | 3.75 | 2.94 | 5.61 | 1.64 | 0.91 | 9.30 |
| | | | | | | | | |
| **Mill Volume Filling** | % | 26.6 | 26.7 | 27.3 | 27.4 | 26.7 | 26.5 | 26.9 |
| | SD | 0.08 | 0.22 | 0.17 | 0.33 | 0.09 | 0.05 | 0.56 |
| | | | | | | | | |
| **Power** | kW | 9642 | 9629 | 9776 | 9820 | 9541 | 9708 | 9695 |
| | SD | 44.99 | 35.04 | 57.85 | 112.83 | 48.77 | 35.12 | 122.03 |
| | | | | | | | | |
| **Power Modelled** | kW | 9598 | 9585 | 9792 | 9788 | 9557 | 9634 | 9663 |
| | SD | 74.75 | 69.81 | 117.99 | 105.54 | 74.61 | 61.92 | 157.74 |
| | | | | | | | | |
| **Input Power** | kWh/t | 17.6 | 16.8 | 15.2 | 14.3 | 13.8 | 17.2 | 16.0 |
| | | | | | | | | |
| **Plant work index** | kWh/t | 22.6 | 21.6 | 20.13 | 18.64 | 18.18 | 20.26 | 20.29 |
| | SD | 0.70 | 1.53 | 0.43 | 0.51 | 1.16 | 1.58 | 3.07 |
| | | | | | | | | |
| **Feed** | tph | 547 | 572 | 642 | 687 | 690 | 564 | 604 |
| | SD | 11.40 | 97.96 | 7.68 | 4.93 | 24.68 | 33.79 | 133.05 |
| | | | | | | | | |
| **Feed, <31.5mm** | % | 35.9 | 38.2 | 38.6 | 35.4 | 34.7 | 51.5 | 38.6 |
| | SD | 3.62 | 3.23 | 3.71 | 3.71 | 6.61 | 5.29 | 7.07 |
| | | | | | | | | |
| **Product, -75um** | **%** | 38.4 | 35.8 | 33.6 | 33.8 | 39.3 | 59.7 | 39.0 |
| | SD | 1.98 | 7.84 | 3.89 | 3.97 | 7.79 | 2.22 | 9.51 |
| | | | | | | | | |
| **Circulating load** | % of feed | 55.8 | 41.6 | 71.2 | 53 | 37 | 80 | 54.6 |
| | SD | 7.34 | 19.32 | 9.35 | 11.19 | 10.65 | 15.61 | 19.83 |
| | | | | | | | | |
| **Silo Level** | % Full | 61.8 | 55.2 | 56.9 | 69.6 | 77.2 | 81.3 | 67.2 |

It should be noted that the power and mass readings were relatively steady with the maximum variability due to spikes in the data only covering a range of 6 and 7 % respectively.

Period ST1:

This period can be taken as the base case for comparison as all the variables are relatively constant. The feed rate of c550 tph is low compared to both the design of 620 tph and the previous analysis at 700 to 730 tph. The circulating load of 55% is close to the 50% assumed for design purposes. The product size of 38% less than 75 microns is coarser than the design figure of 50%.

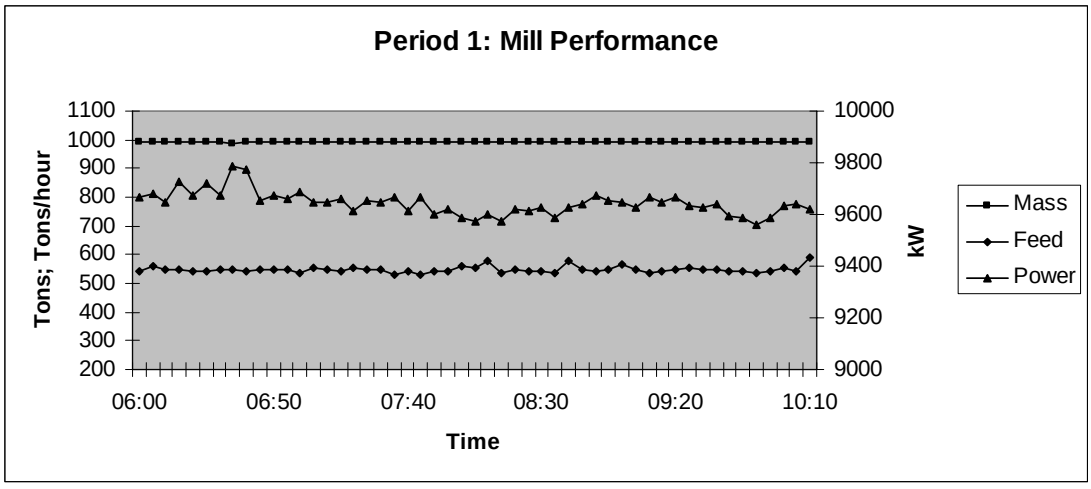



Figure 5.2 Period 1 Mill Performance.

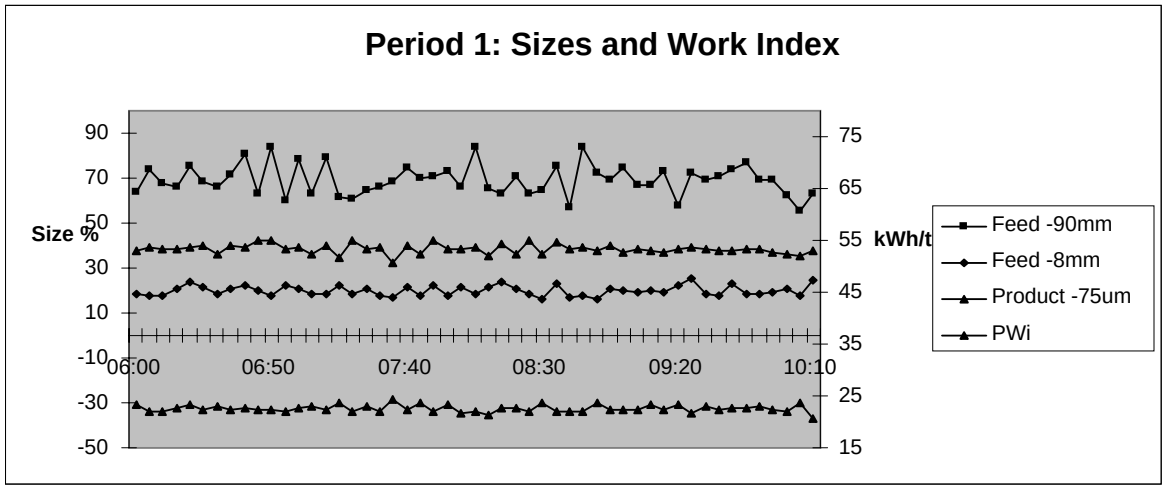



Figure 5.3 Period 1 Sizes and Plant Work Index.

Period ST2

All variables continued to be steady during this period. The feed setpoint was increased to 600 tph and the measured feed was controlled at this level apart from three partial interruptions due to feeder chokes. The feed size was coarser, the amount of -8mm and -90mm decreasing significantly. Not surprisingly, the product size also coarsened to 36% -75 microns, but the circulating load decreased. This is possibly due to the fact that the cyclone feed volume setpoint was not adjusted when the feed was increased.

The circulating load spiked at each feed interruption, due to the method of calculation. The reason for the spike of fines in the product at approximately 13h15 is unknown.

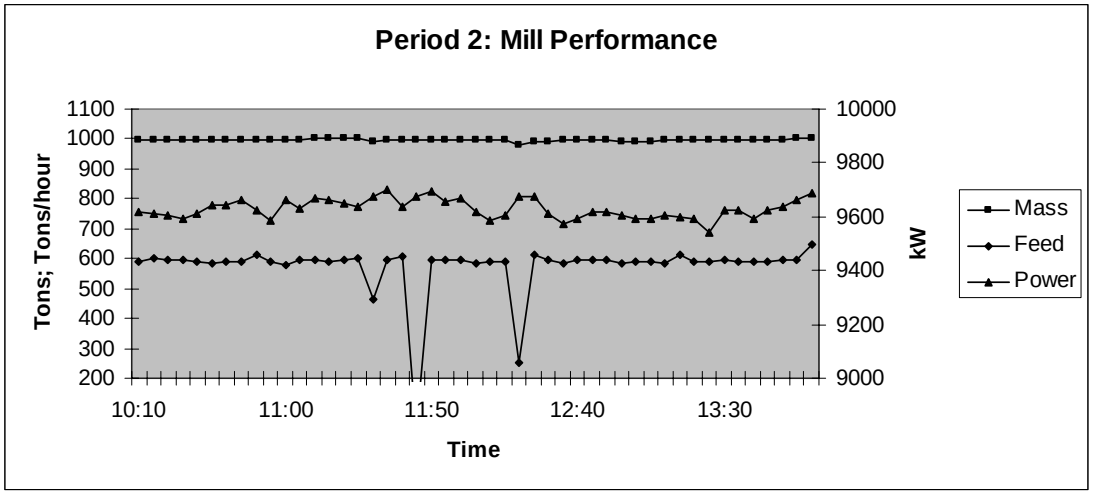



Figure 5.4 Period 2 Mill Performance.

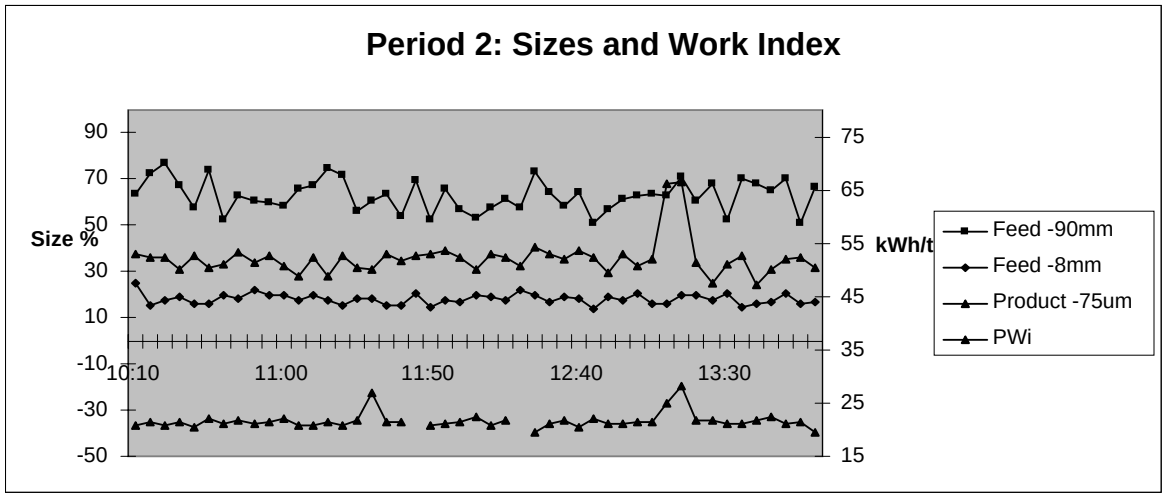



Figure 5.5 Period 2 Sizes and Plant Work Index

Period ST3

The feed was increased to 650 tph and the cyclone feed rate increased from 600 $m^3$/h to 800 $m^3$/h. This resulted in small changes in power and mass but a substantial increase in circulating load, which moved from 40% to above 70%. The average feed size changed slightly as can be seen in figure 5.8, with period 3 being slightly coarser below 16mm, but with less material in the range of 125mm to 250mm.

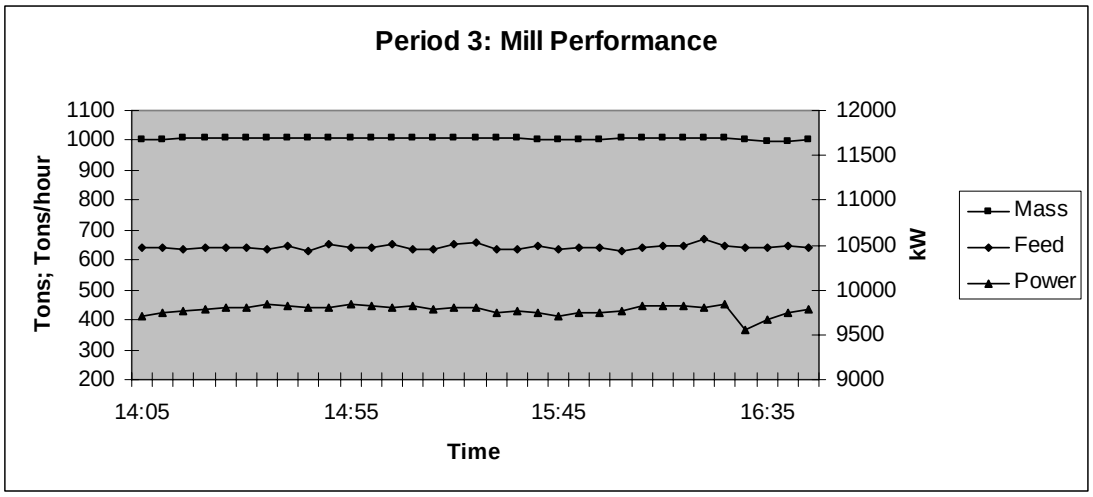



Figure 5.6 Period 3 Mill Performance.

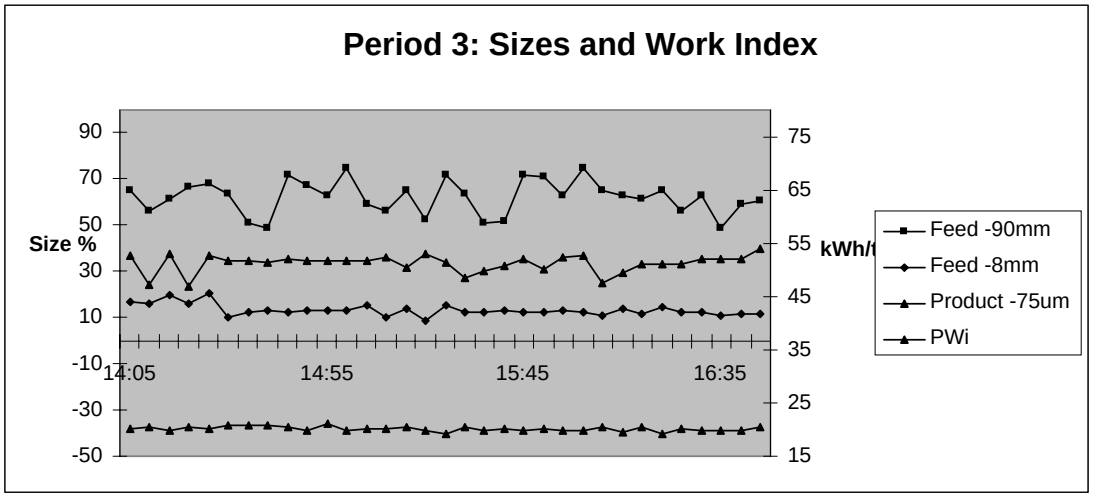



Figure 5.7 Period 3 Sizes and Plant Work Index

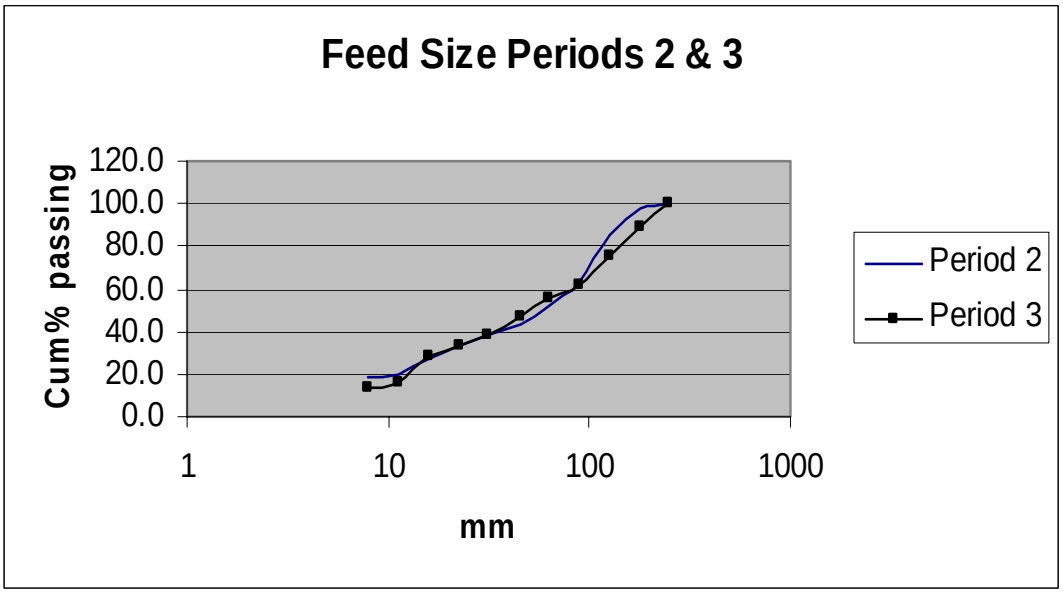



Figure 5.8: Comparison of feed size: Periods ST2 and ST3.

The increase in circulating load seems to have offset any other changes, as the product sizing for the two periods is almost identical. (Figure 5.9).

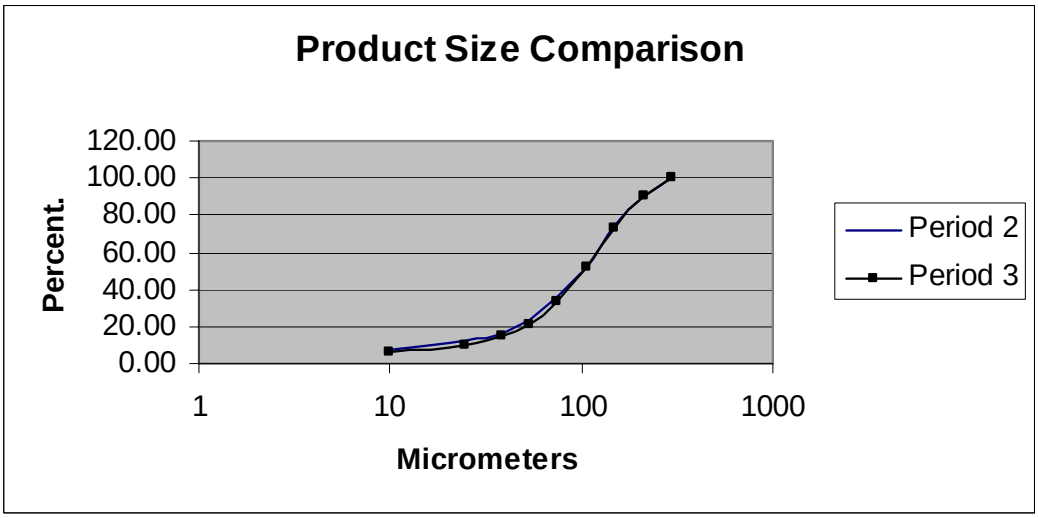



Figure 5.9: Comparison of product size: Periods ST2 and ST3.

Towards the end of this period, at approximately 16h20, it was noticed that Screen 3 was partially blinded and that significant quantities of pulp were being returned to the mill. This manifested as a drop in mill load and power as the mill was being washed out. Screens 1 and 2 were coping with their feed so no deterioration in grind was seen. Screen 4 was started and screen 3 stopped.

The current drawn by the screens is interesting. Screen 2 showed no change from a steady 41A. Screen 3 increased abruptly from 41A to 65A. Screen 1 jumped to 69A for five minutes before returning to its normal 44A, confirming an increase in screen loading. Screen 4, after starting at 16h30,

runs constantly at 39A. Similar problems have been reported with screens in previous ROM ball mill applications. (Hinde et al, 2001).

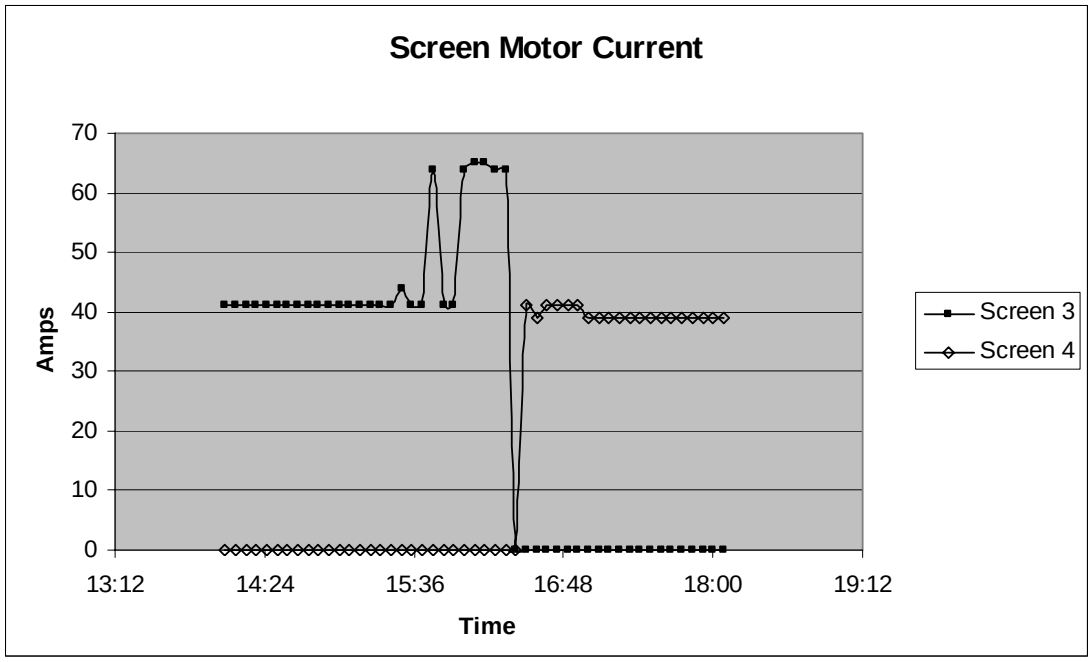



Figure 5.10 Overloading of Screen 3

Period ST4

Again, operation was at a steady feed, now at a setpoint of 700 tph, with a cyclone feed setpoint of 850 $m^3$/h. Mill mass and power decrease slightly, but noticeably, over the period by 1% and 3% respectively.

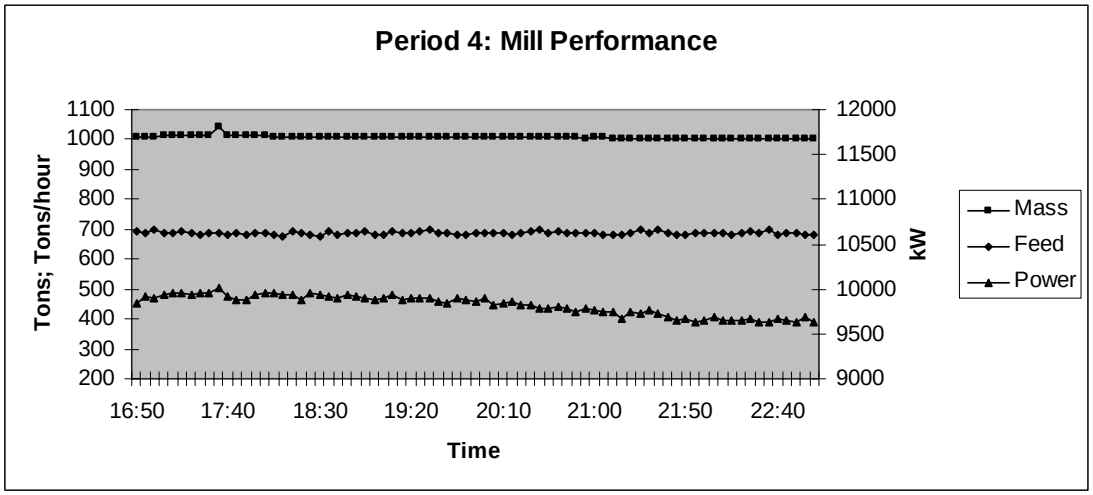



Figure 5.11 Period 4 Mill Performance

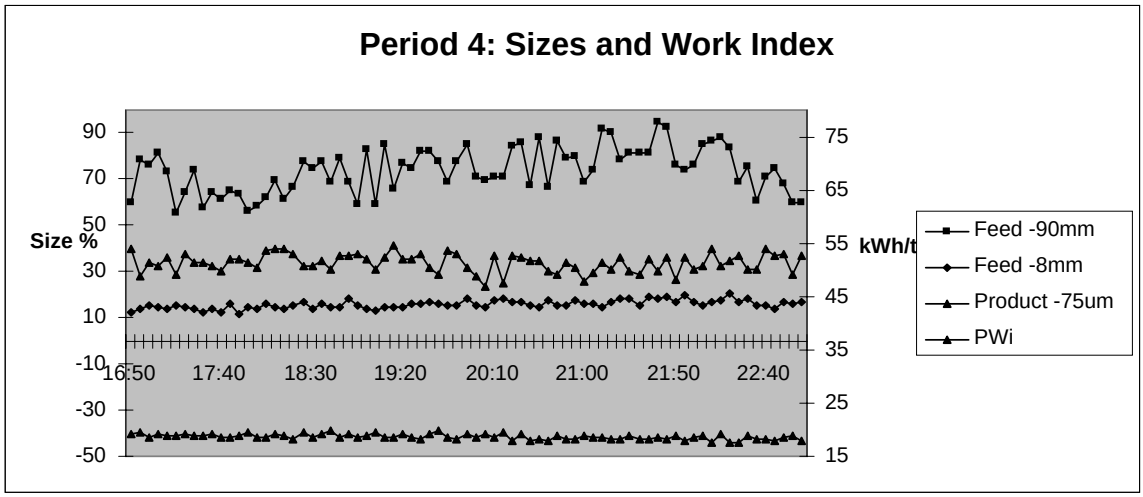



Figure 5.12 Period 4 Sizes and Plant Work Index

The only other factor apparently influencing the reduction in mill load is a general reduction in the size of the feed until approximately 21h50. This may be an indication of a softer feed as the Plant Work Index dropped from 20.1 kWh/t in ST 3 to 18.6 kWh/t in ST 4. This also indicates the danger of using

a single size as an indicator, as the fraction of material between 45 mm and 63 mm is only slightly changed but the distributions above and below this size are different.

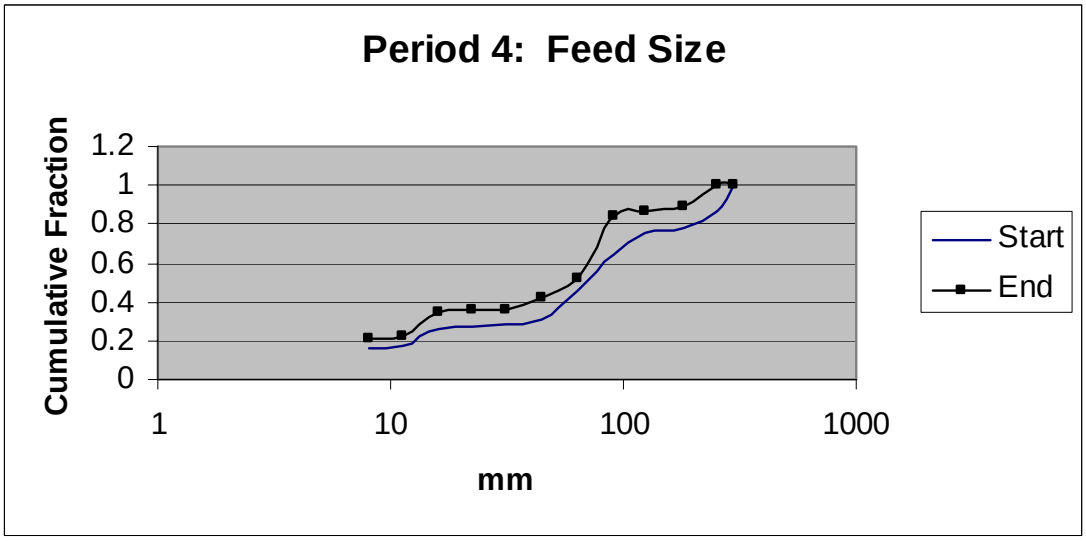



Figure 5.13 Comparison of feed size: Beginning and end of Period ST4

The unusually shaped feed distribution was originally thought to be the result of different mining methods. Half of the feed originates from deep-level stope and pillar mining, the remainder from shallow-level trackless mining. A data set from Amandelbult UG2 has a similar distribution, leading to the conclusion that the different breakage characteristics of the friable chromite seam and the associated pyroxenite and feldspar, which are more competent, may be the reason. Similar results have been reported, (Mainza and Powell, 2006).

The average product size is unchanged from Period ST3 as can be seen in figure 5.14.

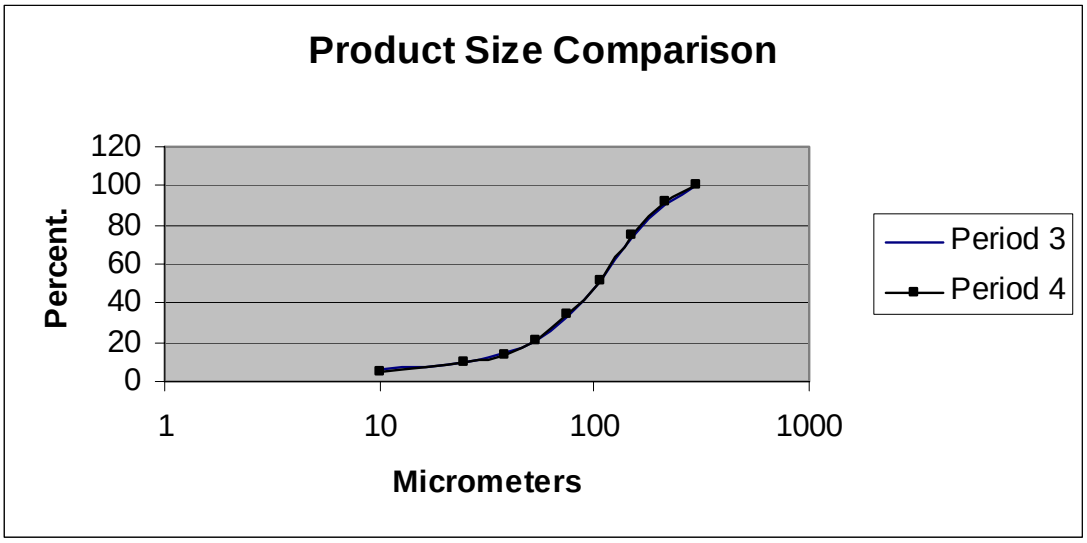



Figure 5.14 Comparison of product size: Periods ST3 and ST4

Period ST5

This period follows a disturbance in the feed flow which coincides with abrupt changes in level in the feed silo. Compared to the previous period, this is characterised by a further small increase in the average coarseness of the feed, a decrease in the circulating load and an increase in the product fineness. The circulating load and fineness of the product appear to be a continuation of the trends observed in period 4 and may be attributed to the change in feed size.

The Plant Work Index was similar to ST 4, 18.6 kWh/t in ST 4, 18.2 kWh/t in ST 5.

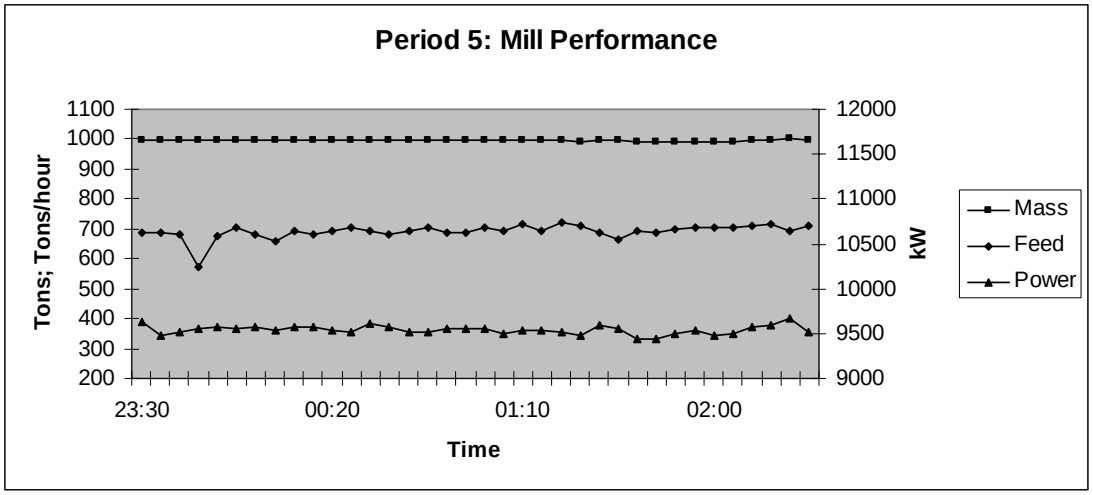



Figure 5.15 Period 5 Mill Performance

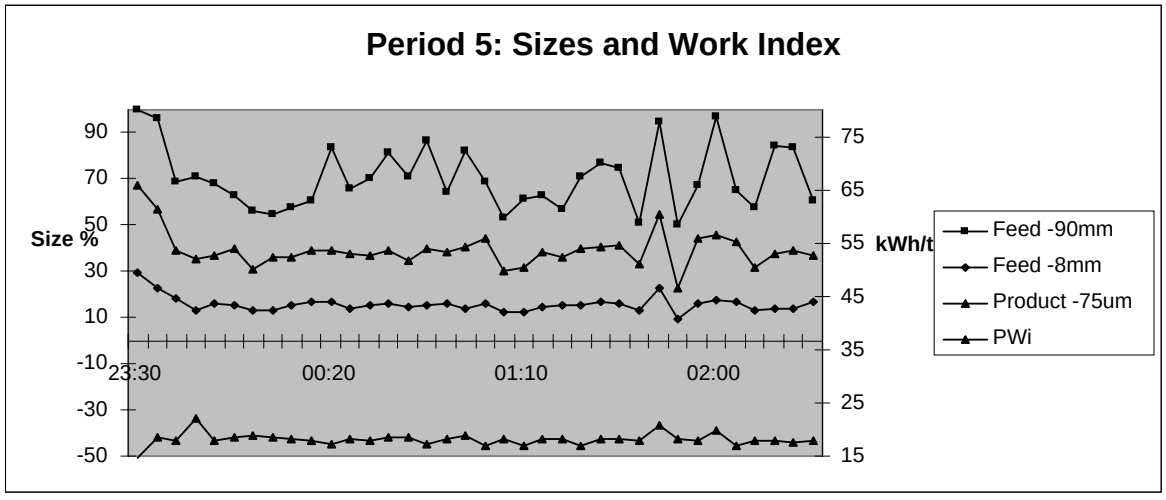



Figure 5.16 Period 5 Sizes and Plant Work Index

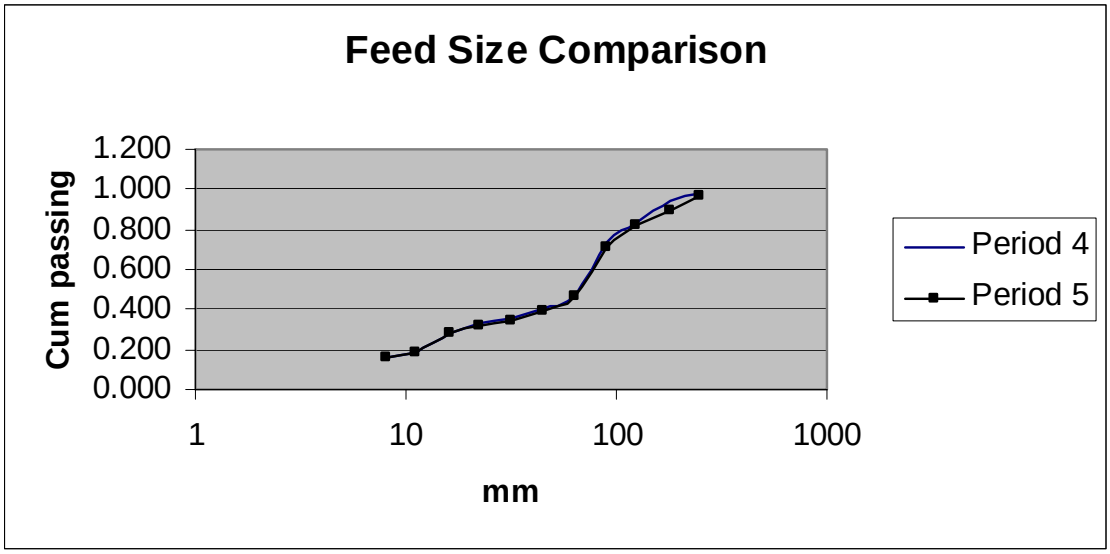



Figure 5.17 Comparison of feed size: Periods ST4 and ST5

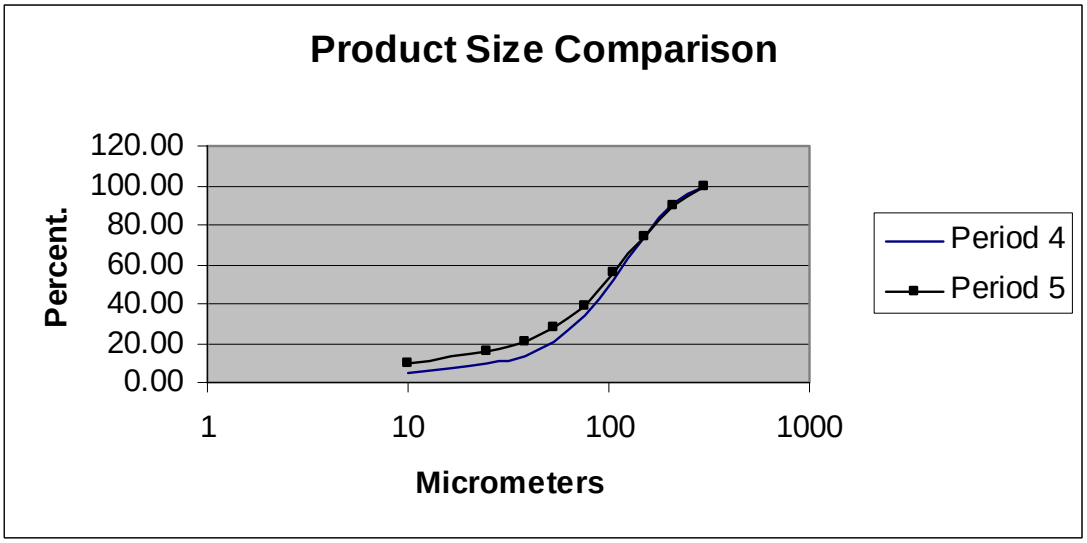



Figure 5.18: Comparison of product size: Periods ST4 and ST5.

Period ST6

The interruptions in feed prior to Period ST6 appeared to be the result of damp, fine material bridging above the feed slot in the silo. The feed rate was therefore dropped to 600 tph to maintain a flow from the silos. If Periods ST5 and ST6 are compared, the resulting improvement in the fineness of grind could be attributed both to the decreased feed rate and to the finer feed.

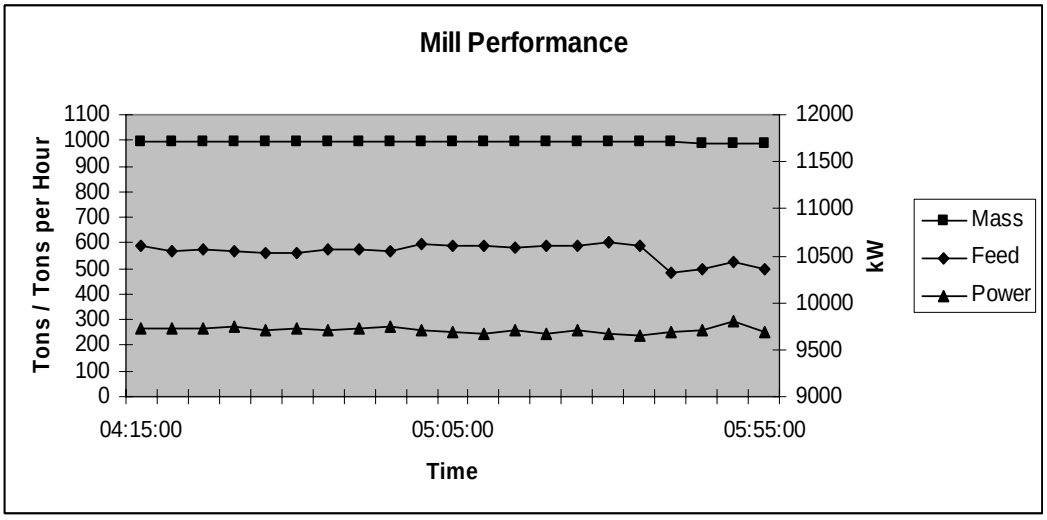



Figure 5.19 Period 6 Mill Performance

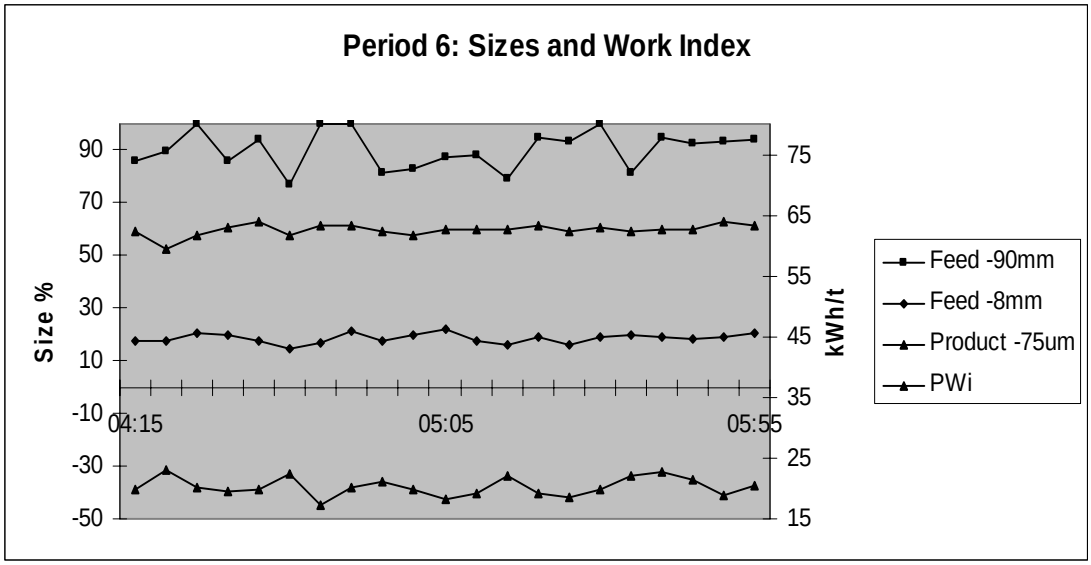



Figure 5.20 Period 6 Sizes and Plant Work Index

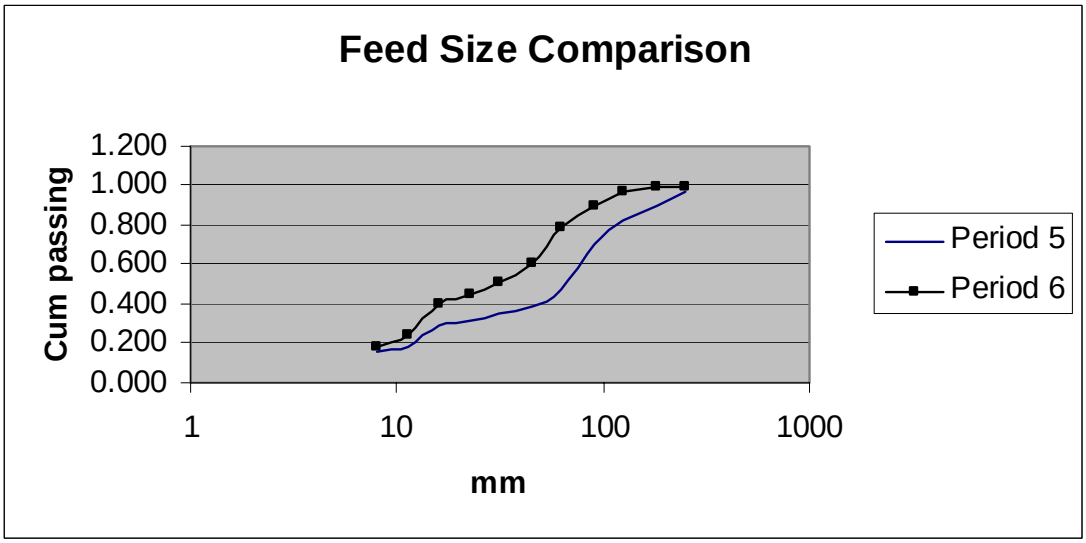



Figure 5.21 Comparison of feed size: Periods ST5 and ST6

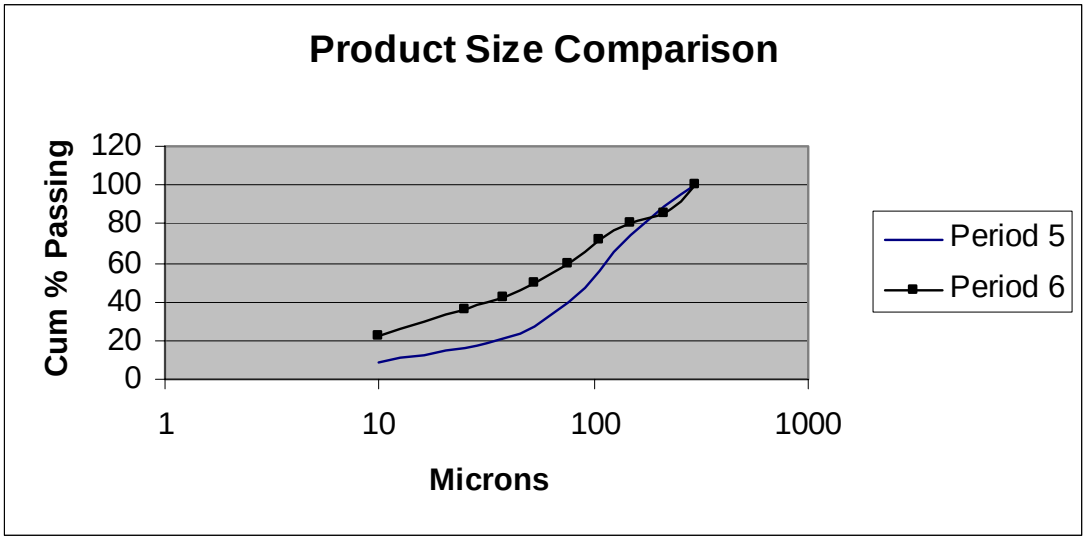



Figure 5.22 Comparison of product size: Periods ST5 and ST6

If Period ST6 is compared to Period ST1, where the feed rates, mill mass and power, are the same, a different conclusion is reached. The change in feed to a significantly finer range appears to be the main factor which causes a finer product size. This can also be demonstrated by comparing figures 5.25 and 5.26 where the difference in size, both for feed and product, between periods ST1 to ST5 is small. Period ST6 is the exception.

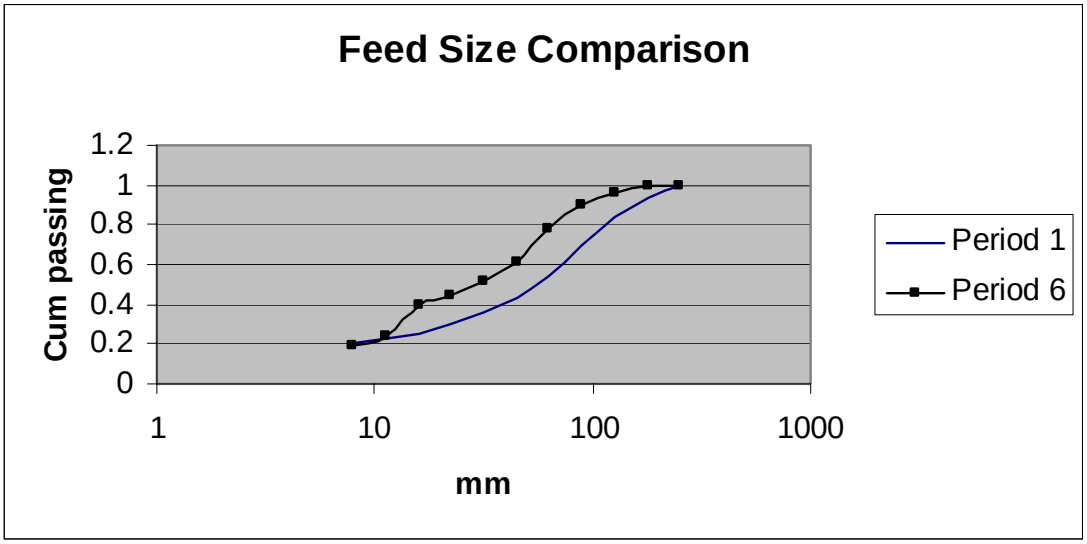



Figure 5.23 Comparison of feed size: Periods ST1 and ST6

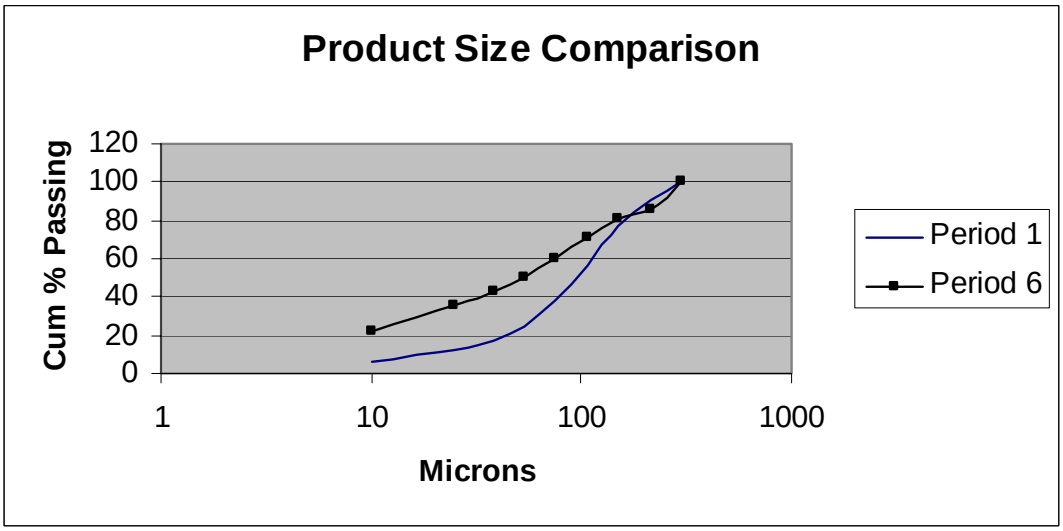



Figure 5.24 Comparison of product size: Periods ST1 and ST6

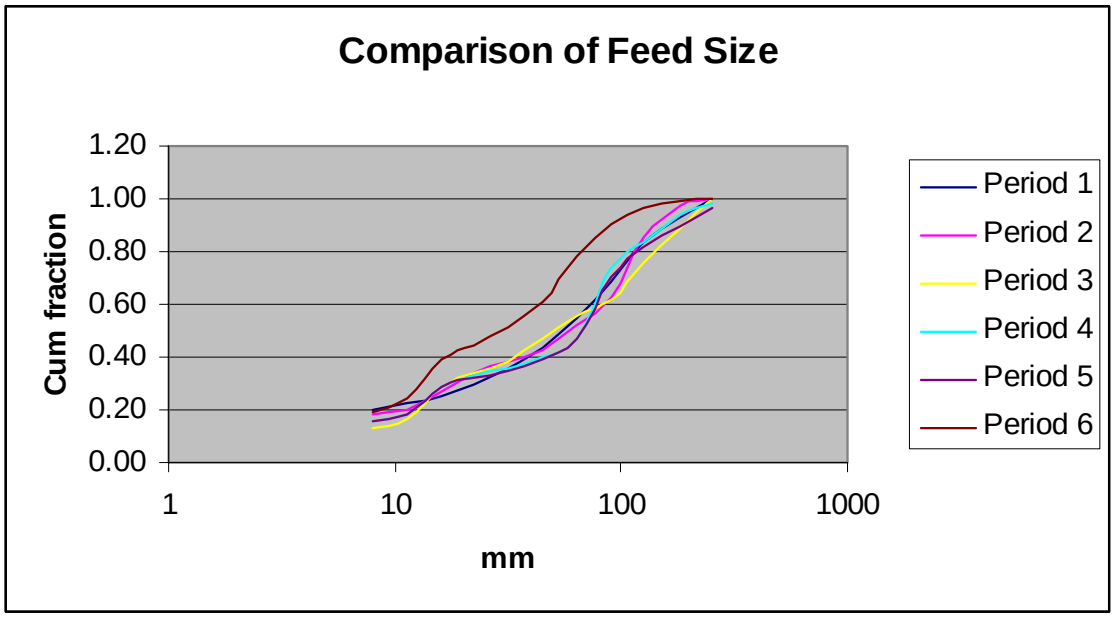



Figure 5.25 Comparison of feed size: All periods

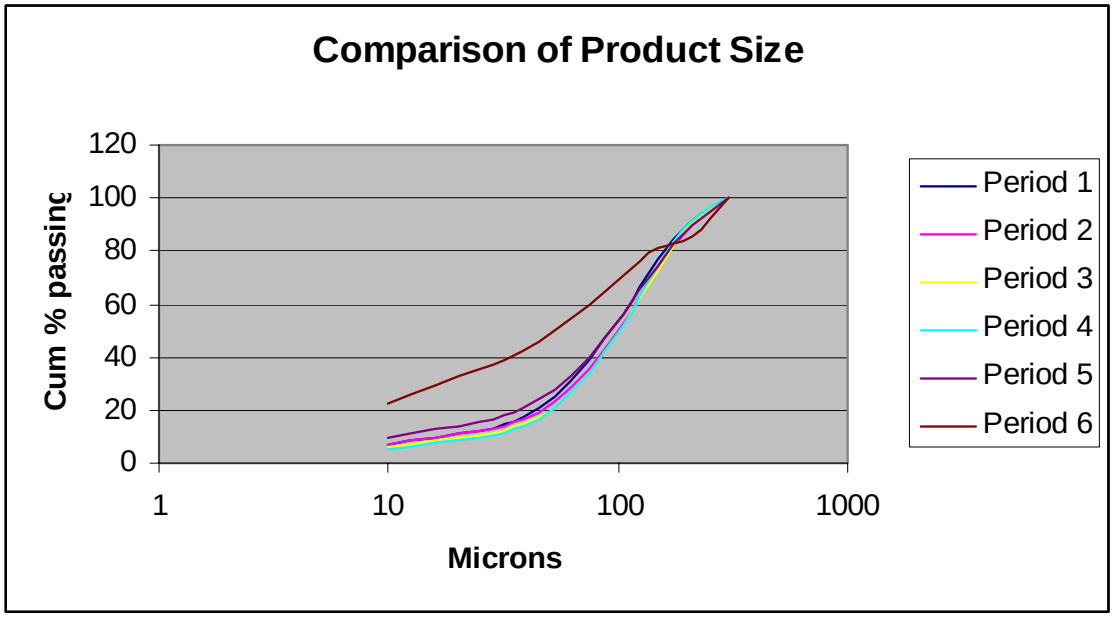



Figure 5.26 Comparison of product size: All periods

The change in the shape of the feed curve from Period ST1 to Period ST6 also indicates a change in the type of material. If the assumption that the binomial type of distribution seen in Period ST6 is due to a mixture of chromite, pyroxenite and feldspar, typical of UG2 ore, then the feed in Period ST1 could have a greater proportion of waste rock. It is known that more waste is sent to the plant over weekends so this is entirely feasible. Feed analyses for the shifts covered by this period show a small increase in chromite content with time which also supports this hypothesis.

Table 5.2: Flotation feed analyses for the 15th May, 2006.

| **Shift** | **Periods** | **$Cr_2O_3$ (%)** | **Grind (%-75um)** | **Grind (%+150um)** |
|---|---|---|---|---|
| | | | | |
| Morning | 1 & 2 | 14.82 | 44.8 | 28.8 |
| Afternoon | 3 & 4 | 16.09 | 37.6 | 32.8 |
| Night | 5 & 6 | 16.28 | 50.4 | 25.6 |
| | | | | |

## 5.3 Statistical Analysis

The data set was analysed using the multiple regression function from a Statistica software package. The best correlation with product grind, using seventeen variables including all the feed sizing data, gave an $R^2$ of 0.59. The predicted and observed values for the final grind (% < 75 microns) are illustrated in figure 5.27

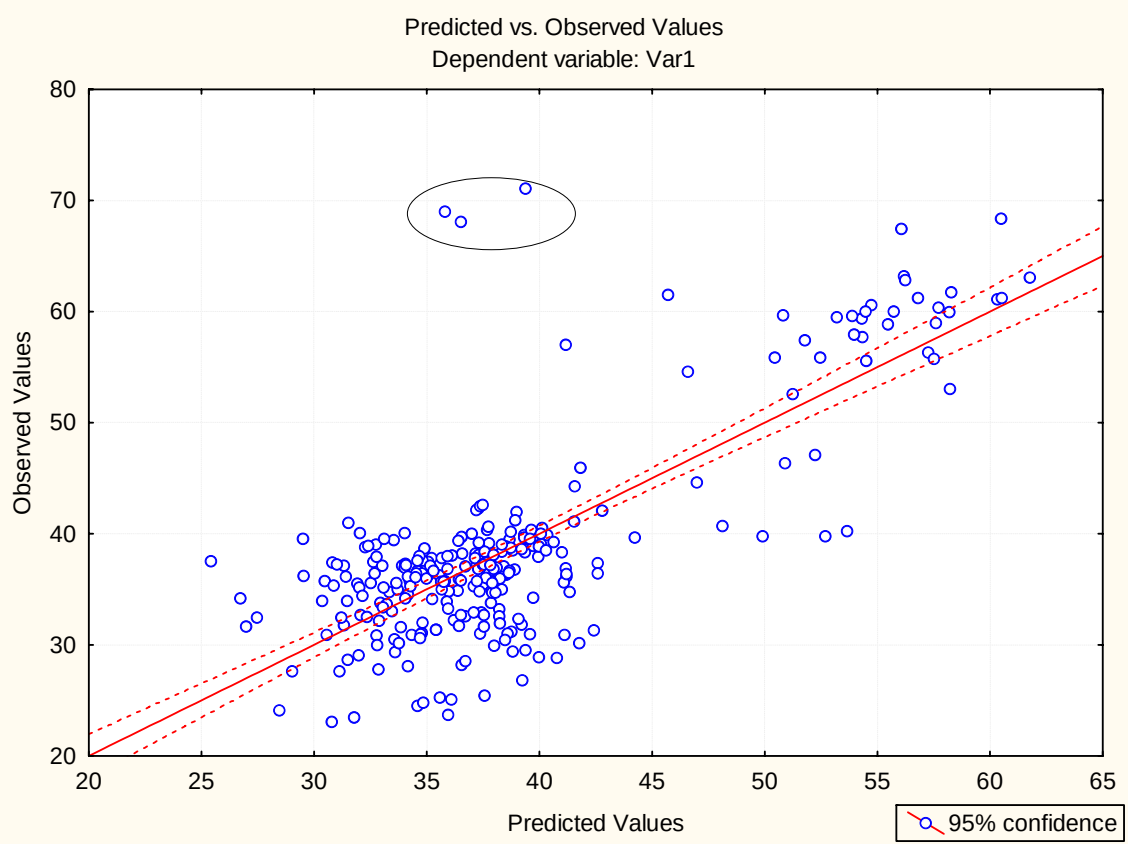



Figure 5.27 Multivariable analysis: predicted and observed final grind

The results above 45 microns for both predicted and observed variables are, with the exception of two outliers, all from period 6. Without period 6, the $R^2$ is only 0.20. The three observed values around 70 % <75 microns that were predicted to be between 35 and 40 coincide with feed interruptions during period 2. Deleting these outliers, seen circled in figure 2.27, improves the $R^2$ to 0.63.

Multivariable analysis for each individual period gives the following $R^2$ values:

| | |
|---|---|
| Period ST1 | 0.47 |
| Period ST2 | 0.44 |
| Period ST3 | 0.54 |
| Period ST4 | 0.31 |
| Period ST5 | 0.95 |
| Period ST6 | 0.89 |

A combined regression for Periods ST5 and ST6 gives an $R^2$ value of 0.98. The resulting predicted and observed values are depicted in figure 5.28. This confirms that period ST6 is definitely different from the other periods.

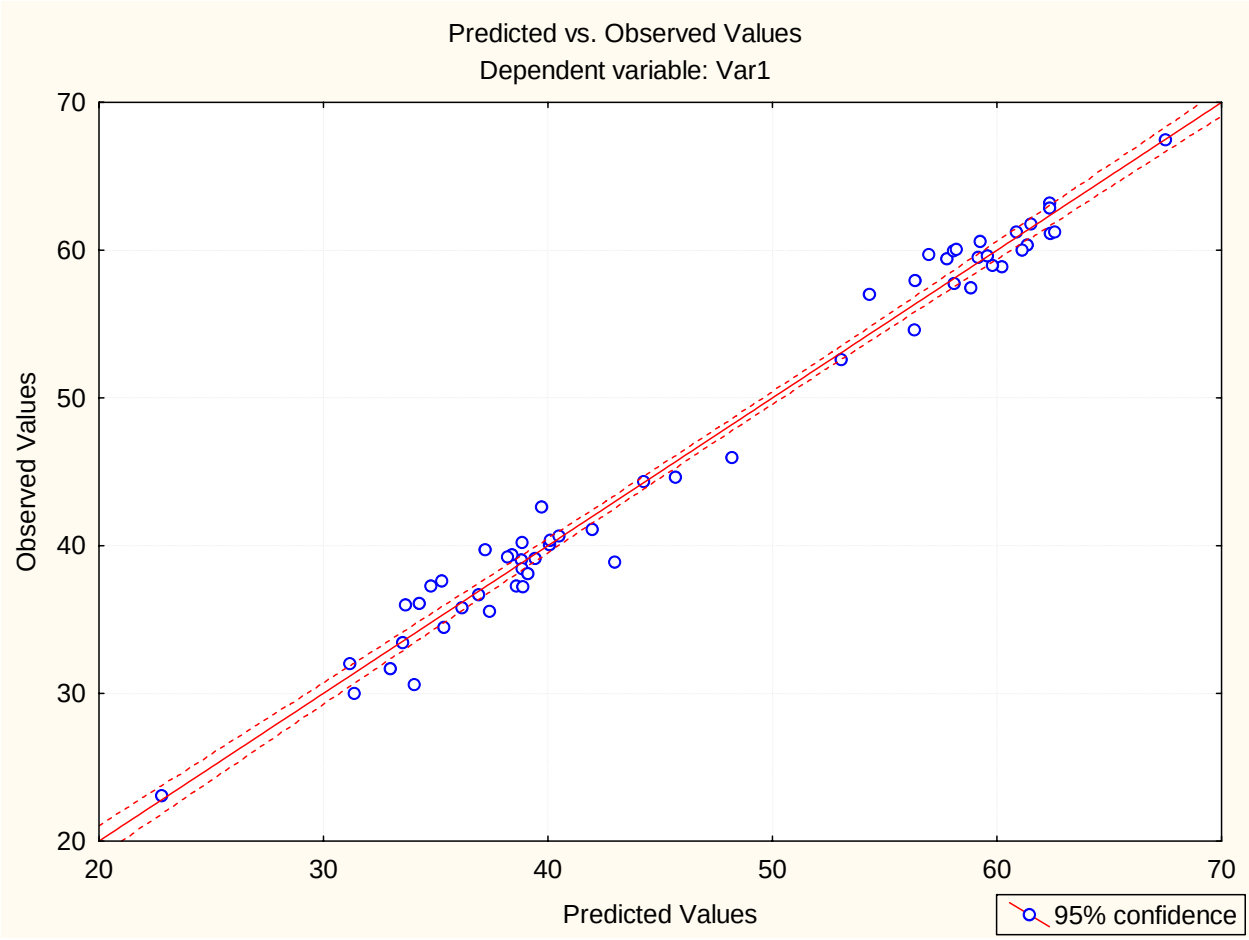



Figure 5.28: Multivariable analysis: predicted and observed final grind. Periods ST5 and ST6

It is therefore not possible to attribute the change in product sizing solely to the change in feed size, as the finer feed may be indicative of a change to a softer or more friable ore.

## 5.4 Modelling

### 5.4.1 Mill Power Model

The mill power model was fitted to the total data set and a correlation of 0.82 achieved for the day. Correlations for the individual periods ranged from 0.29 to 0.89. The poor correlations occur where there is little change in the readings over the period. The highest correlation was for period ST4 where the change in power draw was 4%. The modelled and measured values for the day are shown below in figure 5.29.

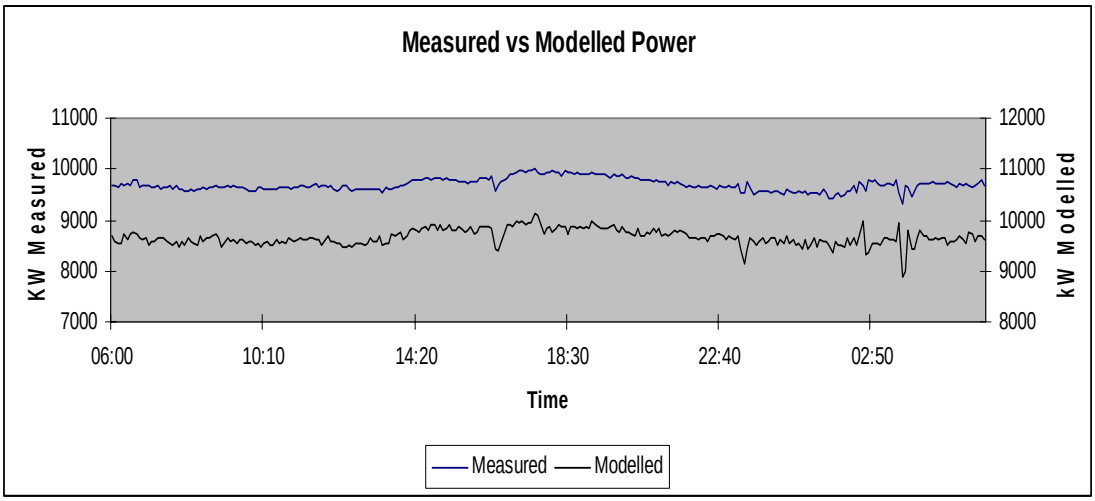



Figure 5.29 Measured and modelled mill power.

The model relies on two main parameters; mill filling and in-mill density. The filling is relatively constant but the calculation of the density uses the cyclone feed flows and densities which are less stable, hence the short-term variability in the modelled results.

The mismatch between 02h30 and 03h30 was caused by rapid changes of the cyclone feed flow set point by the operator. The reason for this was interruptions in the mill feed caused by choked feeders. Abrupt changes in the silo levels at the same time indicate possible build-up and sloughing within the silo. Similar behaviour was observed earlier around 12h00.

In general, the modelled power gives a good fit over a day, although not as good as the 0.93 correlation previously shown over a month because the range is wider.

### 5.4.2 JKSimMet Model

The total data set was then used to develop a model using the JKSimMet SAG model programme. The balanced circuit diagram is shown in figure 5.30 below. When this model was applied to the individual periods the modelled grind remained substantially unchanged despite the changes in feed size feed rate power draw etc.. A typical result is shown in figure 5.31.

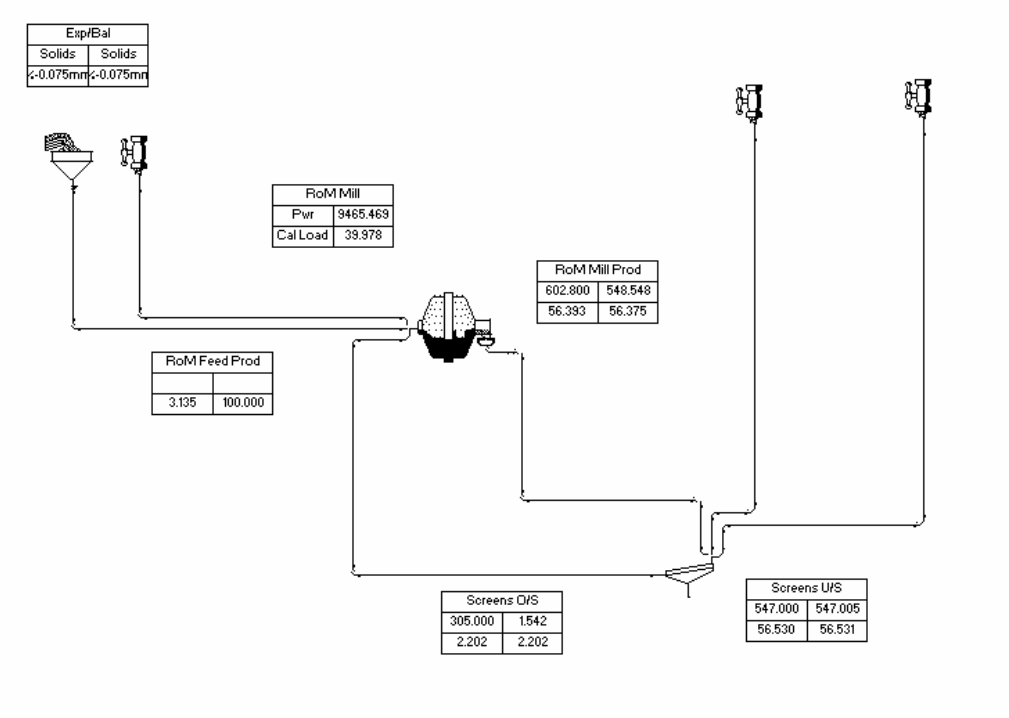



Figure 5.30 Modelled SAG Mill Circuit.

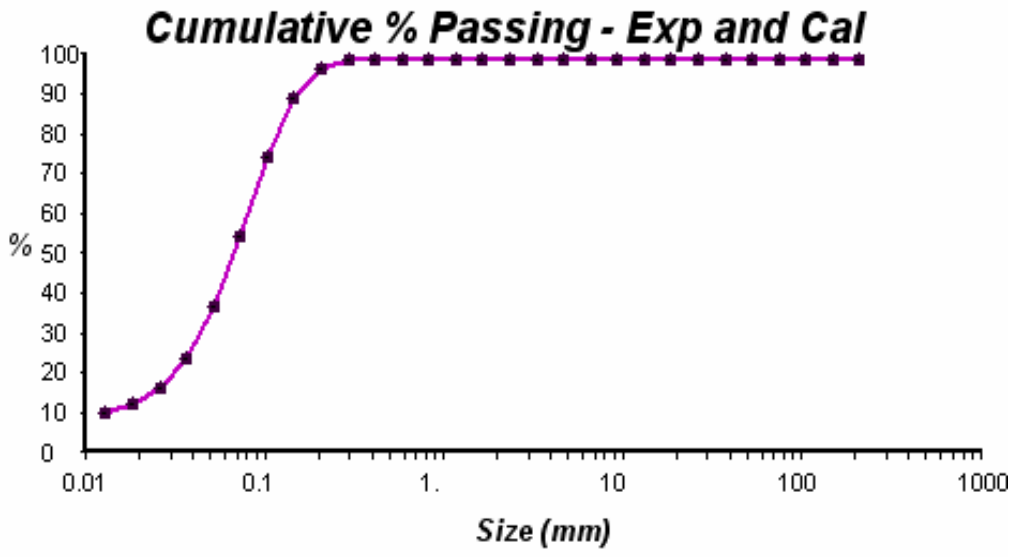



Figure 5.31 Typical example of measured and modelled grind

Each individual period was then mass balanced and modelled to obtain a breakage function. In no case can the models be called a good fit. The standard deviations of the errors ranges from 4.9 to 7.9. The usually accepted error standard deviation for a good fit is 3 or less. The reason for the poor fit is the circulating loads which are modelled to be considerably less than the measured values.

The model for period ST1 was then used as a base case for the simulation of the effects of changing feed rate and size. This period gives the best model fit. The model predicts a coarser product size and greater variation than the measured values but follows the trend, except for period 6.

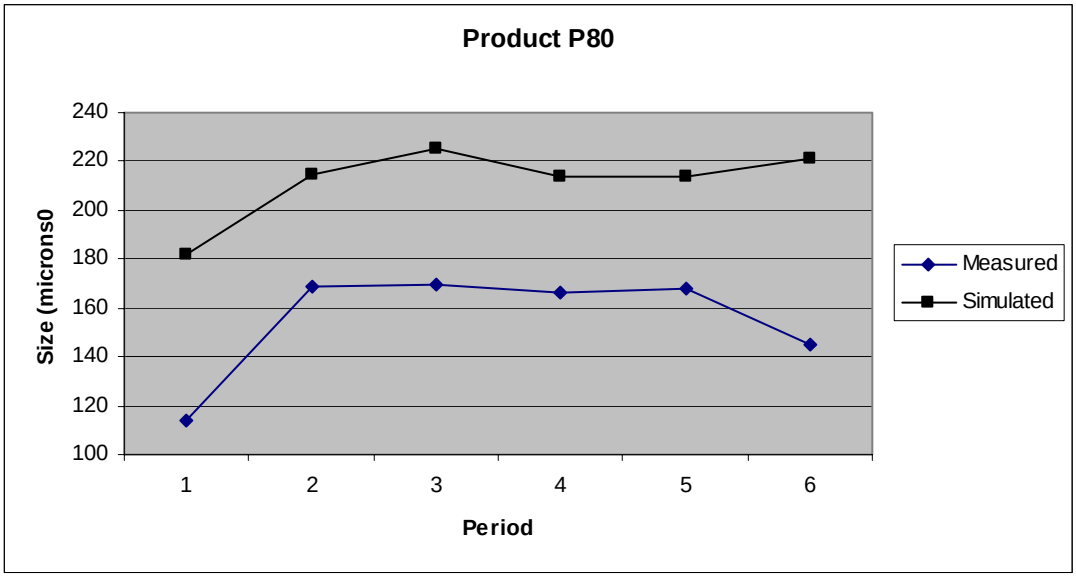



Figure 5.32 Product P80 comparison by period

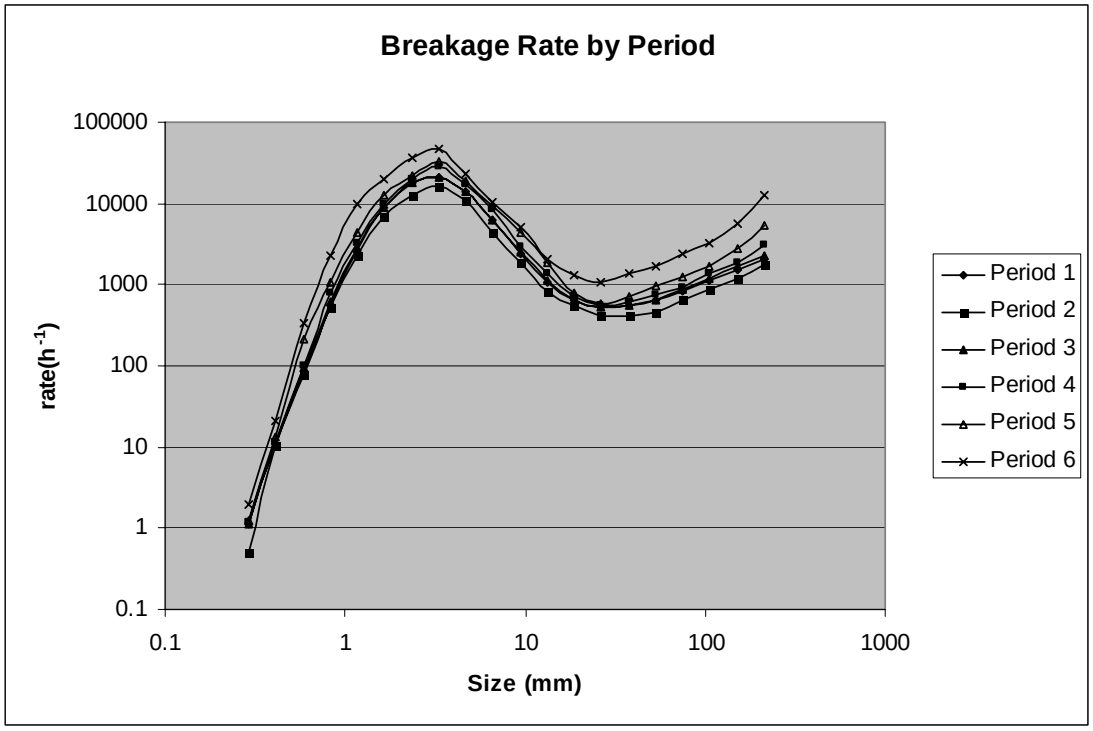



Figure 5.33 Breakage rate comparison by period

As can be seen from the breakage rate curves, figure 5.33 above, there is a significantly higher rate for period ST6 than for the other periods. This confirms a change in the feed material between periods 5 and 6

Use of the model to simulate the effects of changes in feed size at constant throughput results in the following product sizes for the feed size distributions from each period:

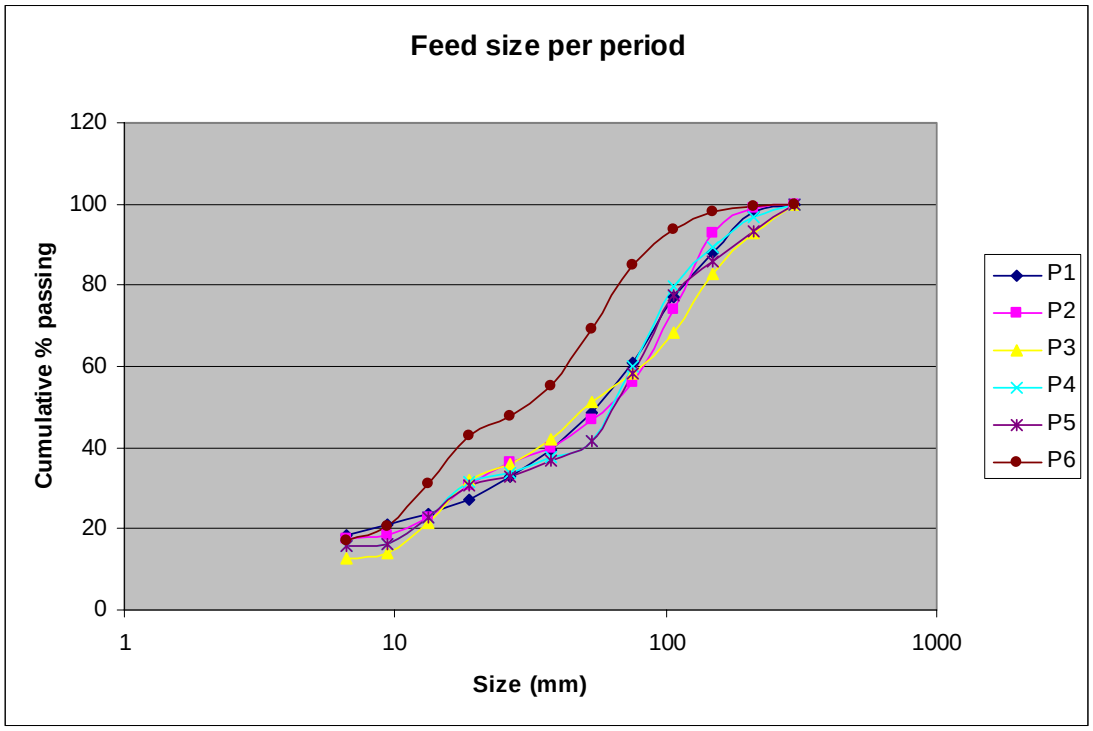



Figure 5.34 Measured feed size distribution, all periods

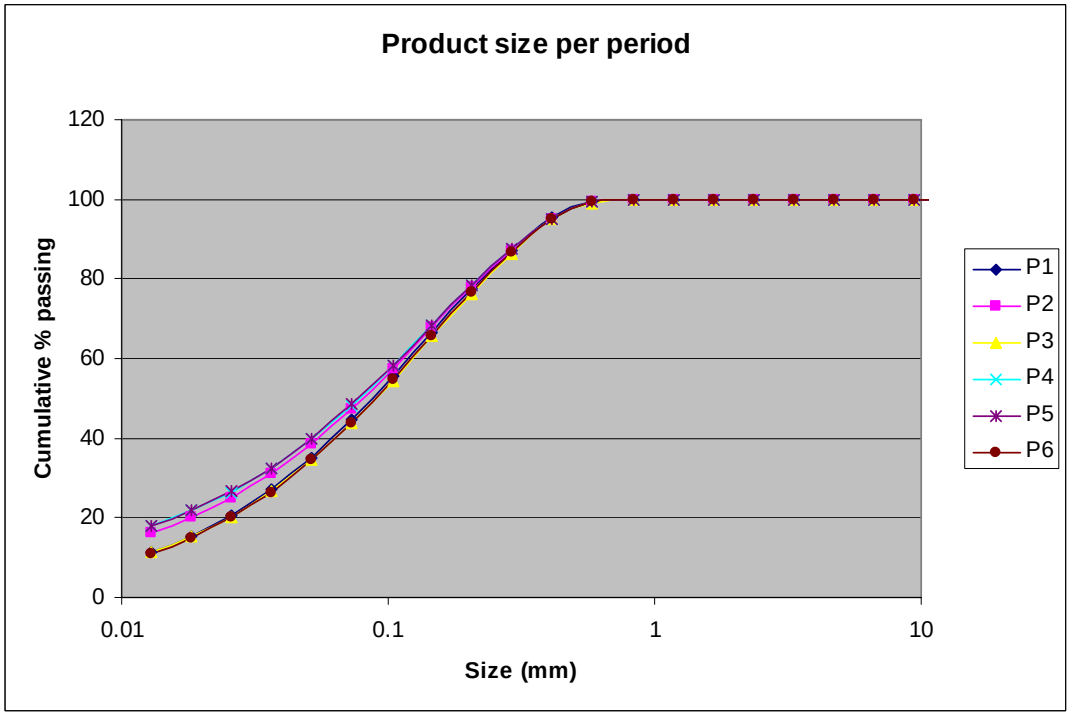



Figure 5.35 Modelled product size distribution, all periods

The model suggests that the differences in the feed curve make little difference to the final product. This would imply that top-size crushing, for example by reducing the jaw crushers' gap, would not have a major effect on the mill grind. However, findings reported from Geita Gold Mine, (Mwehonge, 2006) and an unnamed operation, (Major, 2006) that pre-crushing of the total feed stream increased the material less than one millimetre and thereby improved the overall grind, do not support this conclusion.

Mwehonge reports that crushing of an intermediate fraction, between 30mm and 120mm, has improved throughput by 14% but with little change in the final grind. Major also reports increased throughput of 20% as a result of increasing the proportion of fines less than 25mm in the mill feed from 11% to 94% but does not comment on final grind.

As the size distribution of the feed below 8mm is an artefact of the modelling programme, which extrapolates the curve when calculating a standard size distribution from the measured distribution, it is not possible to treat this result as anything more than an indication of possible areas for improvement.

## 5.5 Conclusions: Short-Term Data

- Screen current draw is a good indicator of screen overloading.
- A finer feed is observed to result in a finer grind, but it is not possible to determine whether this is due to a change in ore hardness affecting both feed and product.
- The Morell power model gives a good fit over a day, although the correllation of 0.82 is not as good as the 0.93 correlation previously shown over a month.
- The JKSimMet SAG model is a poor fit to the available data. There are two probable reasons for this. Firstly, the data may well be inaccurate, particularly in the measurement of slurry flows and densities and the consequent calculation of circulating loads. Secondly, the model itself has been developed on substantially different mills.
- The model indicates that changes to the proportion of material less than ten millimetre in the feed has the biggest effect on product size. This cannot be confirmed from the data due to the influence of other factors, notably feed rate. This indication also needs to be treated with caution due to the interpolation of the -8mm fractions by the model.

# 6 MEDIUM TERM DATA ANALYSIS

Modifications to the secondary grinding circuit of the plant in mid 2006 resulted in much improved operations. Data from October, in particular, showed consistent primary mill operation with few stops. In addition, routine analyses indicate little variation in the ore type. The October data set consists of five minute readings from the 2nd to the 19th day of the month.

The data set was inspected as detailed in section 4 above and no interpolations or deletions from the data set were considered necessary, except to remove incomplete records from the UltraPS readings. As discussed above, it appears that these are due to a mismatch in timing between the UltraPS readings and the plant control system..

## 6.1 Data Inspection

Figure 6.1, below, indicates that both power and feed remained reasonably constant for this period. The power draw averaged 9.62 MW with a standard deviation of only 4.0%. The feed rate averaged 747 tph with a standard deviation of 6.4%. Equally importantly, the cyclone feed flow rate averaged 813 $m^3/h$ with a standard deviation of only 4.2%, indicating little operator intervention.

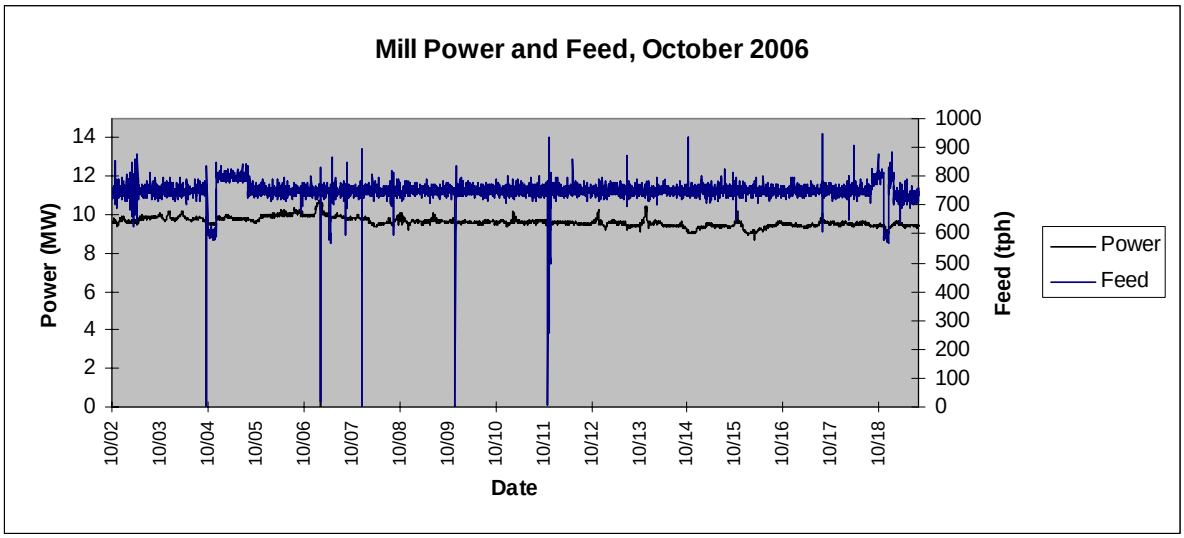



Figure 6.1: Mill Power and Feed Rate, October 2006.

The chromite content in the feed indicates that the feed was relatively constant in terms of ore type. This is confirmed by the PGE analyses.

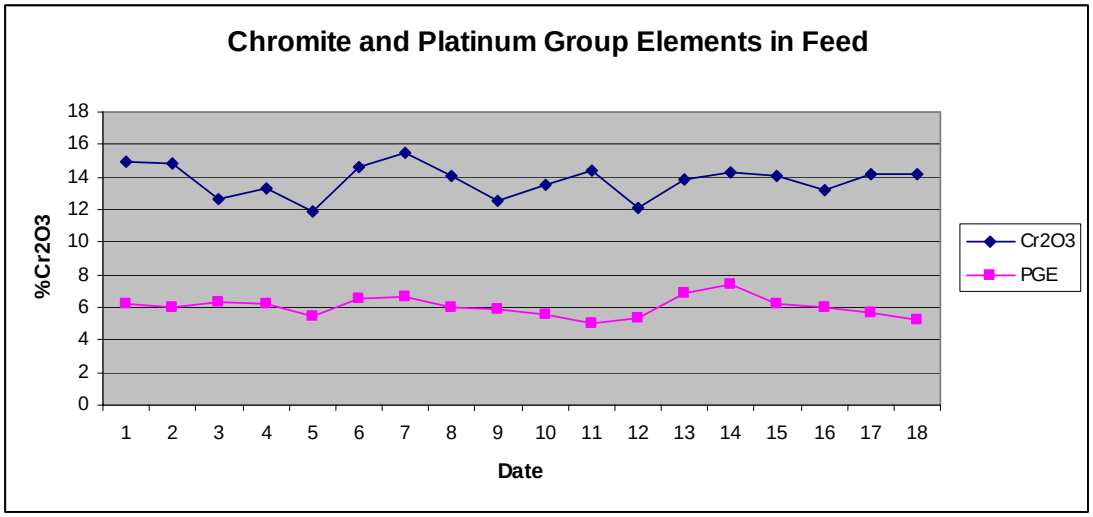



Figure 6.2: Feed Chromite and Platinum Group Element Content, October 2006. (Nb: PGE grades not shown due to confidentiality requirements)

Each of the feed and product size classes was then correlated in order to determine if there were any obvious links. Table 6.1, below, shows the correlation factors between each feed size and each product size. This was done to determine the best comparison between feed and product sizes. Table 6.1 shows good correlations between feed sizing and product sizing, particularly in the middle ranges of the size classes. In the case of the Lynxx readings this is not surprising as the -8mm is below its normal calibration range and there is little material in the higher classes during this period. The UltraPS is calibrated to have the highest accuracy in the -75µm class as this is the sizing most used by plant operating staff.

Table 6.2 below, shows improved correlations if the UltraPS readings are correlated with the Lynxx readings from the previous record, ie imposing a five minute delay on the Lynxx data. The best correlation, the cumulative percentage of -31.5 mm material in the feed with the cumulative percentage of -75 µm material in the product, is shown in figure 6.3 and 6.4.

Table 6.1: Correlation Table of Feed and Product Sizing, October 2006.

| Product µm | Feed mm -8 | -11.2 | -16 | -22.4 | -31.5 | -45 | -63 | -90 | -125 | -250 | -300 |
|---|---|---|---|---|---|---|---|---|---|---|---|
| **-10 µm** | 0.1260 | 0.1461 | 0.1387 | 0.1565 | 0.1630 | 0.1572 | 0.1355 | 0.0798 | 0.0347 | 0.0212 | -0.0009 |
| **-25 µm** | 0.2139 | 0.2451 | 0.2390 | 0.2653 | 0.2718 | 0.2605 | 0.2249 | 0.1390 | 0.0527 | 0.0111 | -0.0068 |
| **-38 µm** | 0.2904 | 0.3290 | 0.3280 | 0.3636 | 0.3708 | 0.3558 | 0.3092 | 0.1924 | 0.0705 | 0.0087 | -0.0072 |
| **-53 µm** | 0.3994 | 0.4449 | 0.4591 | 0.5085 | 0.5193 | 0.5026 | 0.4389 | 0.2723 | 0.0926 | -0.0014 | -0.0132 |
| **-75 µm** | 0.4796 | 0.5204 | 0.5638 | 0.6252 | 0.6413 | 0.6305 | 0.5547 | 0.3450 | 0.1099 | -0.0141 | -0.0230 |
| **-106 µm** | 0.3297 | 0.3490 | 0.3938 | 0.4356 | 0.4474 | 0.4444 | 0.3946 | 0.2460 | 0.0819 | -0.0132 | -0.0288 |
| **-150 µm** | 0.2309 | 0.2367 | 0.2695 | 0.2921 | 0.2965 | 0.3008 | 0.2738 | 0.1735 | 0.0681 | -0.0037 | -0.0084 |
| **-212 µm** | 0.1823 | 0.1802 | 0.2165 | 0.2313 | 0.2348 | 0.2439 | 0.2171 | 0.1270 | 0.0462 | -0.0059 | -0.0020 |

Table 6.2: Correlation Table of Feed and Product Sizing With Five Minute Offset.

| Product µm | Feed mm -8 | -11.2 | -16 | -22.4 | -31.5 | -45 | -63 | -90 | -125 | -250 | -300 |
|---|---|---|---|---|---|---|---|---|---|---|---|
| **-10 µm** | 0.1422 | 0.1608 | 0.1540 | 0.1725 | 0.1813 | 0.1763 | 0.1561 | 0.1035 | 0.0491 | 0.0197 | 0.0096 |
| **-25 µm** | 0.2476 | 0.2767 | 0.2720 | 0.3020 | 0.3124 | 0.3030 | 0.2704 | 0.1855 | 0.0837 | 0.0260 | 0.0093 |
| **-38 µm** | 0.3350 | 0.3721 | 0.3737 | 0.4143 | 0.4270 | 0.4159 | 0.3729 | 0.2576 | 0.1174 | 0.0317 | 0.0104 |
| **-53 µm** | 0.4557 | 0.5013 | 0.5193 | 0.5756 | 0.5926 | 0.5813 | 0.5207 | 0.3557 | 0.1551 | 0.0317 | 0.0109 |
| **-75 µm** | 0.5403 | 0.5836 | 0.6319 | 0.7014 | 0.7228 | 0.7186 | 0.6438 | 0.4340 | 0.1831 | 0.0268 | 0.0056 |
| **-106 µm** | 0.3653 | 0.3870 | 0.4350 | 0.4812 | 0.4948 | 0.4977 | 0.4460 | 0.2974 | 0.1209 | 0.0097 | -0.0040 |
| **-150 µm** | 0.2521 | 0.2588 | 0.2939 | 0.3190 | 0.3251 | 0.3329 | 0.3050 | 0.2054 | 0.0932 | 0.0139 | 0.0038 |
| **-212 µm** | 0.1999 | 0.1981 | 0.2359 | 0.2532 | 0.2566 | 0.2657 | 0.2401 | 0.1514 | 0.0660 | 0.0040 | -0.0022 |

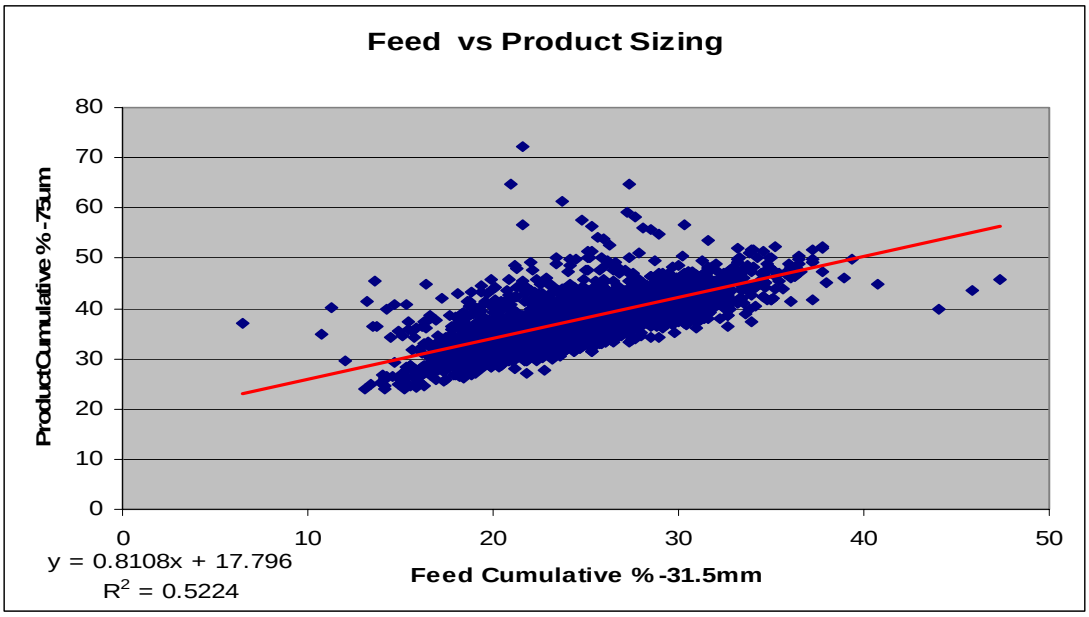



Figure 6.3 Feed vs Product Comparison, October 2006.

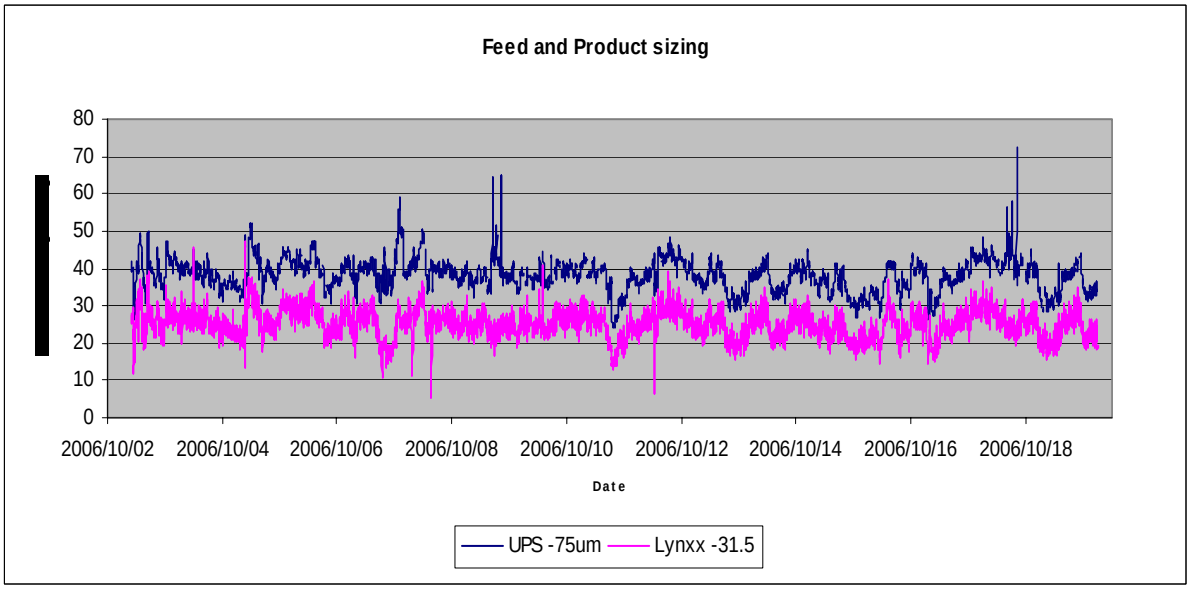



Figure 6.4 Feed and Product Comparison, October 2006.

The regression line indicates that the relationship between feed and product sizing is given by the equation:

Product %-75 µm = 0.8108 x ( Feed % -31.5mm ) + 17.796

It is also noted that there are no significant correlations with the mill operating variables such as power, feed, cyclone flowrate etc. This is not unexpected as these variables were extremely constant during this period.

The next step was to determine whether this finding was unique to the October 2006 data set. Although this project has been complicated by the difficulty in obtaining data sets where the sizing instruments and the data logging systems were operational simultaneously, there was a period of complete data from 6$^{th}$ to 11$^{th}$ September which appeared to exhibit a similar degree of stability to the October period. The averages and standard deviations are shown in table 6.3.

Table 6.3 Selected averages and standard deviations for variables during September and October, 2006.

| | | **September** | | **October** | |
|---|---|---|---|---|---|
| | | Average | SD | Average | SD |
| | | | | | |
| **Mill Mass** | t | 976 | 12.0 | 953 | 16.0 |
| | | | | | |
| **Power** | kW | 9745 | 198.4 | 9618 | 38.9 |
| | | | | | |
| **Specific Power** | kWh/t | 13.34 | | 12.88 | |
| | kWh/t -75um | 37.16 | | 34.16 | |
| | | | | | |
| **Feed** | tph | 728 | 34.7 | 747 | 47.6 |
| | | | | | |
| **Feed, <31.5mm** | Cum.**%** | 24.2 | 5.0 | 25.1 | 3.9 |
| | | | | | |
| **Feed,<90mm** | Cum.% | 73.9 | 6.1 | 74.4 | 5.9 |
| | | | | | |
| **Product, -75um** | Cum % | 41.1 | 5.2 | 37.7 | 2.7 |
| | | | | | |

The increased variability in the power in September was noticeable and there appeared to be some correlation between power and feed size which had not been seen in the October data. The relevant graphs are shown in figures 6.5 and 6.6.

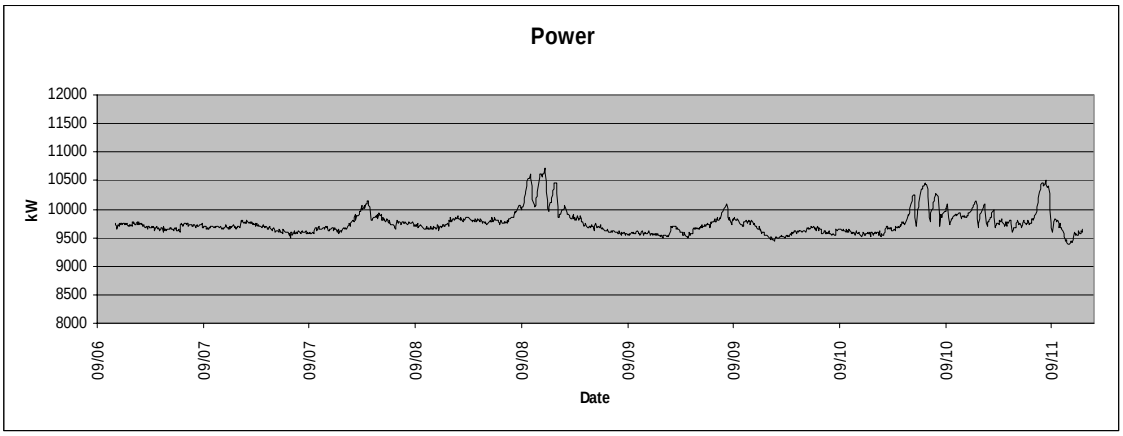



Figure 6.5 Mill Power September 2006.

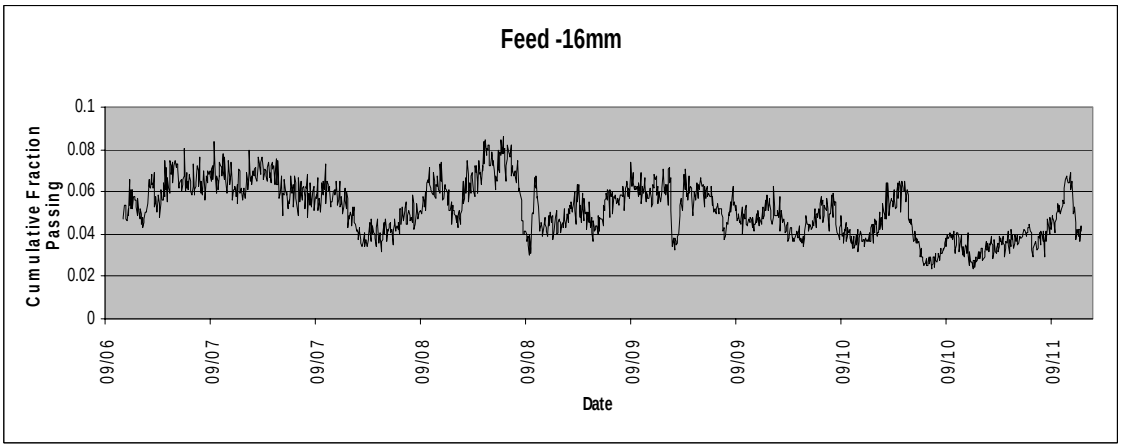



Figure 6.6 Mill Feed % <16mm cumulative fraction, September 2006.

Correlating each of the feed sizes with power gives an increasing negative correlation at finer sizes as shown in figure 6.7. It can be concluded that increasing the proportion of fines fed to the mill decreases the power drawn.

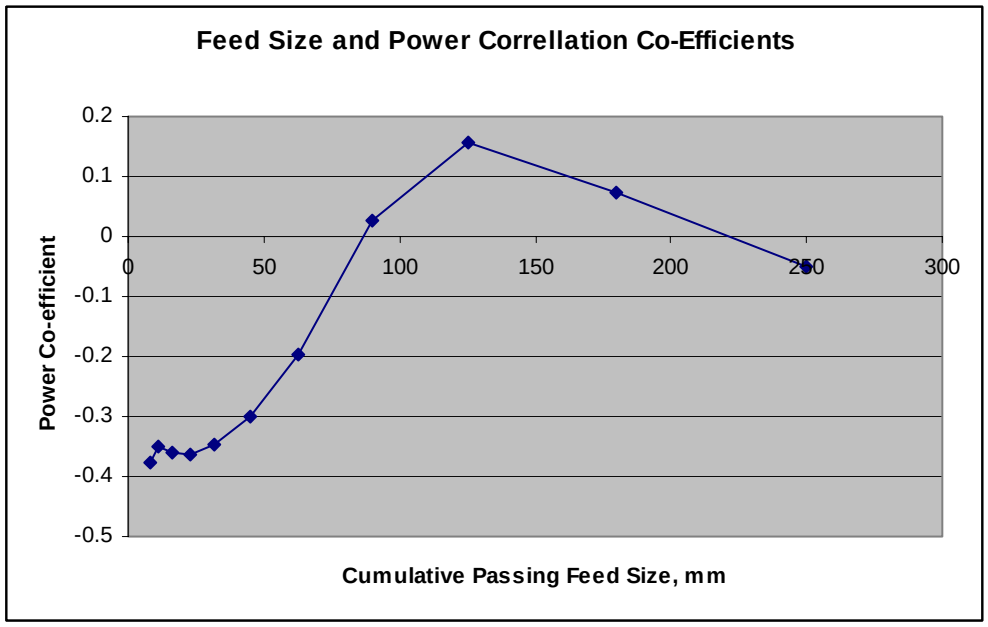



Figure 6.7 Cumulative Mill Feed Size and Correlation to Power, September 2006.

If the data is represented as the fraction of feed occurring in each size range, the picture becomes clearer and figure 6.8 illustrates that increasing quantities of the finer fractions decreases mill power and that increasing quantities of coarse material increases the mill power. The critical size which has least effect on power is the 31.5 mm to 45 mm fraction, despite having the largest quantity of material, as can be seen from figure 6.8. The small quantity of fines appears to have an effect out of proportion to the mass in the feed. This may indicate poor transport of fines through the mill. The larger sizes, above 125 mm, also appear to have little effect on power, probably because this only amounts to approximately 10% of the feed material.

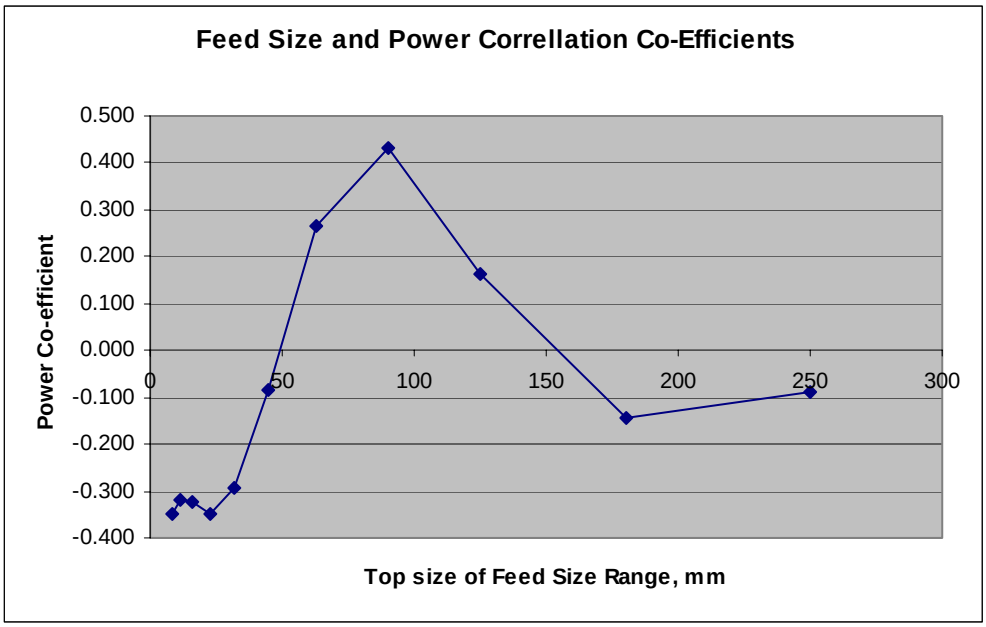



Figure 6.8 Fractional Mill Feed Size and Correlation to Power, September 2006.

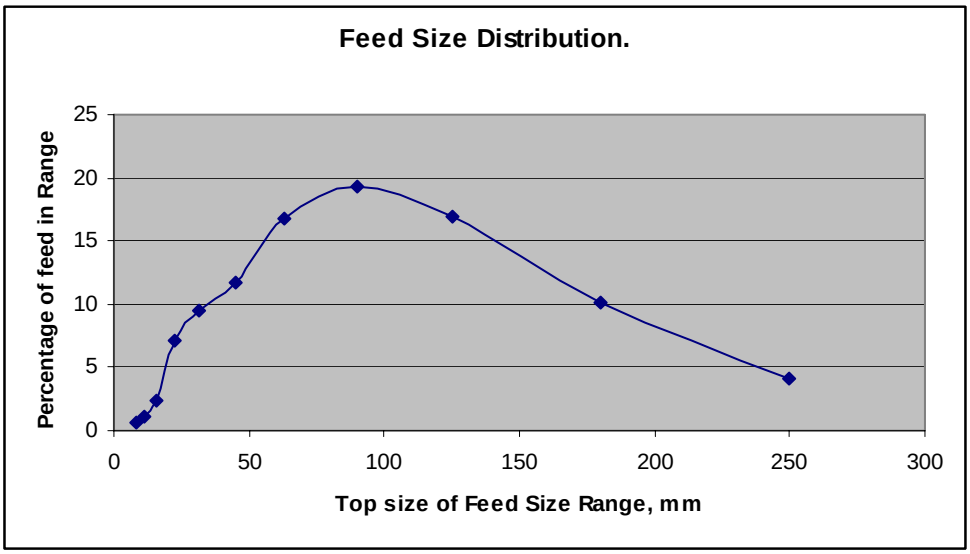



Figure 6. Mill Feed Size Distribution, September 2006.

Table 6.4: Correlation Table of Feed and Product Sizing, September 2006.

| Product | Feed mm 8 | 11.2 | 16 | 22.4 | 31.5 | 45 | 63 | 90 | 125 | 250 | 300 |
|---|---|---|---|---|---|---|---|---|---|---|---|
| **-10 µm** | 0.5645 | 0.6029 | 0.5845 | 0.6512 | 0.6771 | 0.6702 | 0.6046 | 0.4128 | 0.1853 | 0.0040 | -0.0398 |
| **-25 µm** | 0.5815 | 0.6214 | 0.6043 | 0.6679 | 0.6910 | 0.6788 | 0.6051 | 0.4054 | 0.1744 | -0.0022 | -0.0316 |
| **-38 µm** | 0.6029 | 0.6469 | 0.6307 | 0.6952 | 0.7198 | 0.7100 | 0.6355 | 0.4297 | 0.1882 | 0.0029 | -0.0267 |
| **-53 µm** | 0.6254 | 0.6716 | 0.6561 | 0.7200 | 0.7449 | 0.7369 | 0.6618 | 0.4491 | 0.1985 | 0.0070 | -0.0230 |
| **-75 µm** | 0.6372 | 0.6847 | 0.6717 | 0.7348 | 0.7601 | 0.7540 | 0.6778 | 0.4588 | 0.2012 | 0.0078 | -0.0192 |
| **-106 µm** | 0.6016 | 0.6458 | 0.6389 | 0.6951 | 0.7191 | 0.7147 | 0.6431 | 0.4337 | 0.1893 | 0.0071 | -0.0110 |
| **-150 µm** | 0.4594 | 0.4963 | 0.4956 | 0.5347 | 0.5533 | 0.5528 | 0.5030 | 0.3497 | 0.1645 | 0.0198 | -0.0018 |
| **-212 µm** | 0.2286 | 0.2613 | 0.2617 | 0.2762 | 0.2875 | 0.2951 | 0.2832 | 0.2340 | 0.1395 | 0.0382 | 0.0037 |

The feed and product sizing correlations shown in table 6.4 follow a similar pattern to the October data and are strongly correlated. The feed sizing has a five minute delay imposed, as for the October data. The comparison between feed and product sizing can be seen in figure 6.9 and is given by the equation:

Product %-75 µm = 0.8461 x ( Feed % -31.5mm ) + 19.254

This is similar to the previously determined equation for October:

Product %-75 µm = 0.8038 x ( Feed % -31.5mm ) + 17.485

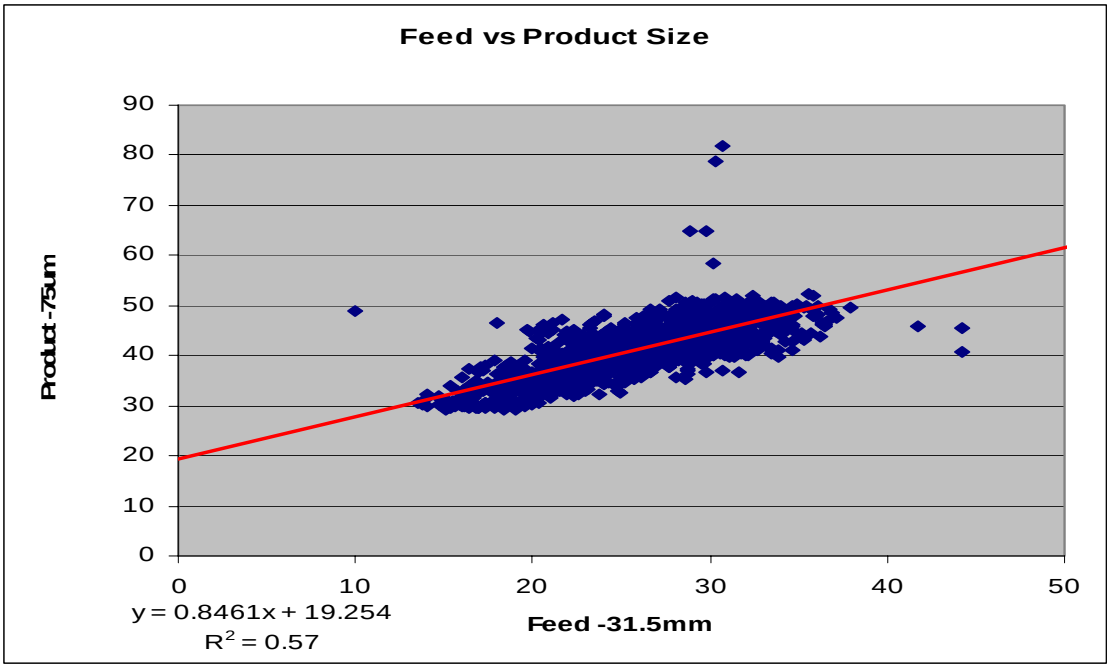



Figure 6.9 Feed vs Product Comparison, September 2006.

## 6.2 Modelling

The October data was mass balanced and modelled as before. A reasonable fit with the measured variables was obtained, as can be seen from the experimental and modelled final product in figure 6.10, below. The error SD was 4.77, a figure of 5 is usually considered as being the upper limit for SAG mill models.

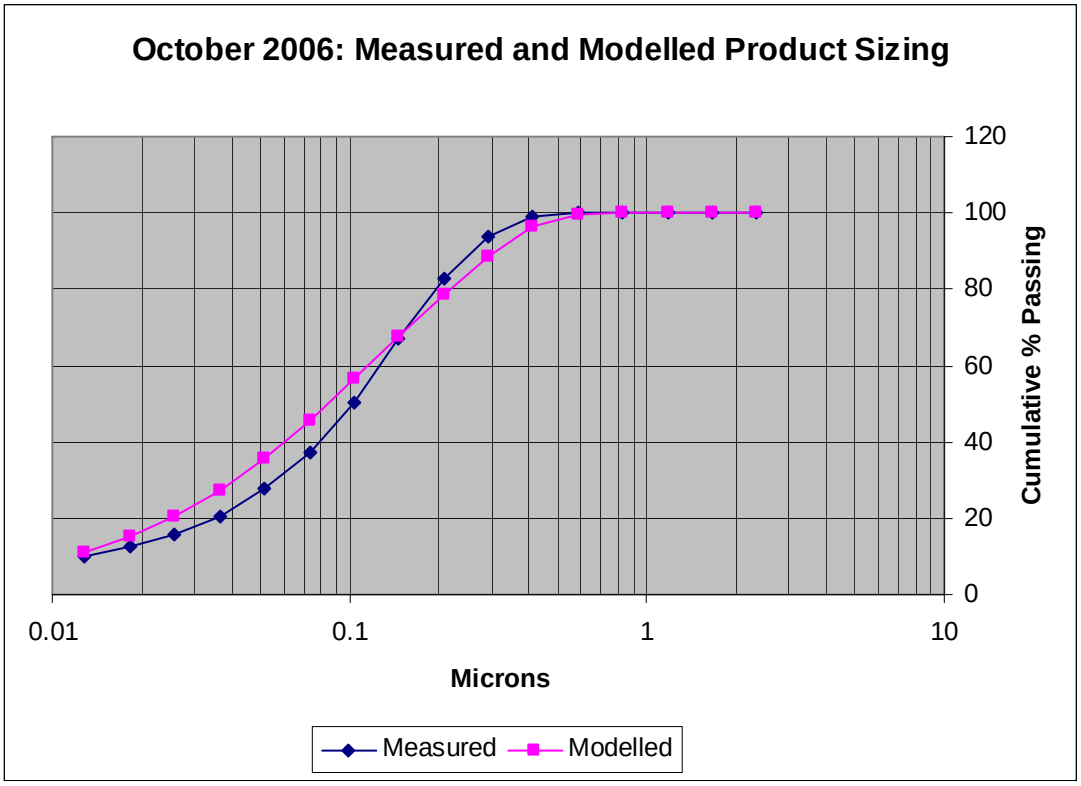



Figure 6.10 Measured and modelled mill circuit product, October 2006.

The September data was then entered into this model. The result can be seen in figure 6.11.

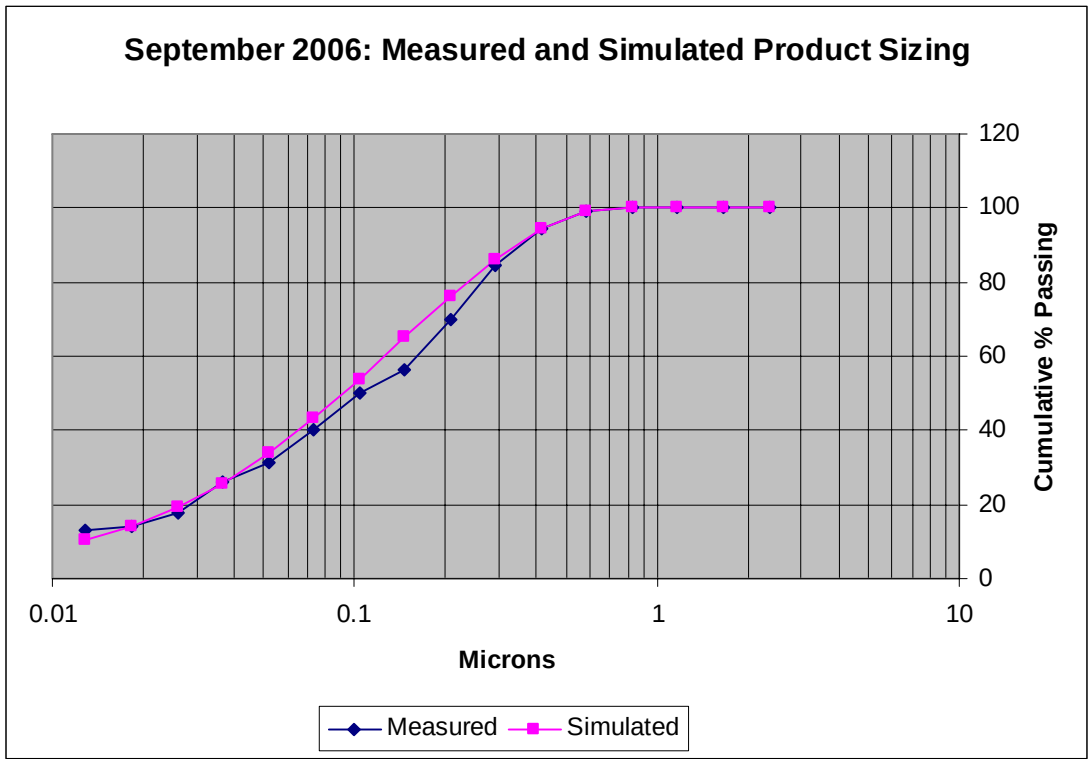



Figure 6.11 Measured and simulated mill circuit product, September 2006.

The model gives a better indication of the September mill product although the overall size is predicted to be slightly finer than the actual. The measured and modelled results agree that the product was finer in September than October, despite the feed sizes being, on average, almost identical. These can be seen in figures 6.12 and 6.13. This is probably due to the 2.6% lower feed rate in September, along with a 23 ton greater ball load which is indicated by the increased load and power draw for September. The model therefore appears to have responded to the changes in these variables

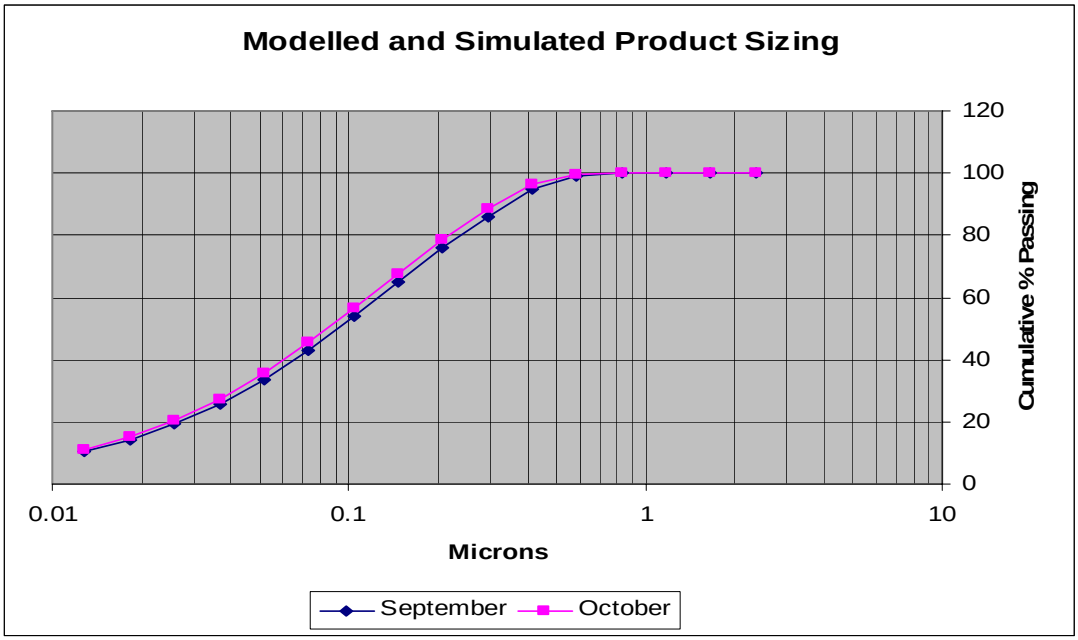



Figure 6.12 Modelled October and Simulated September Product.

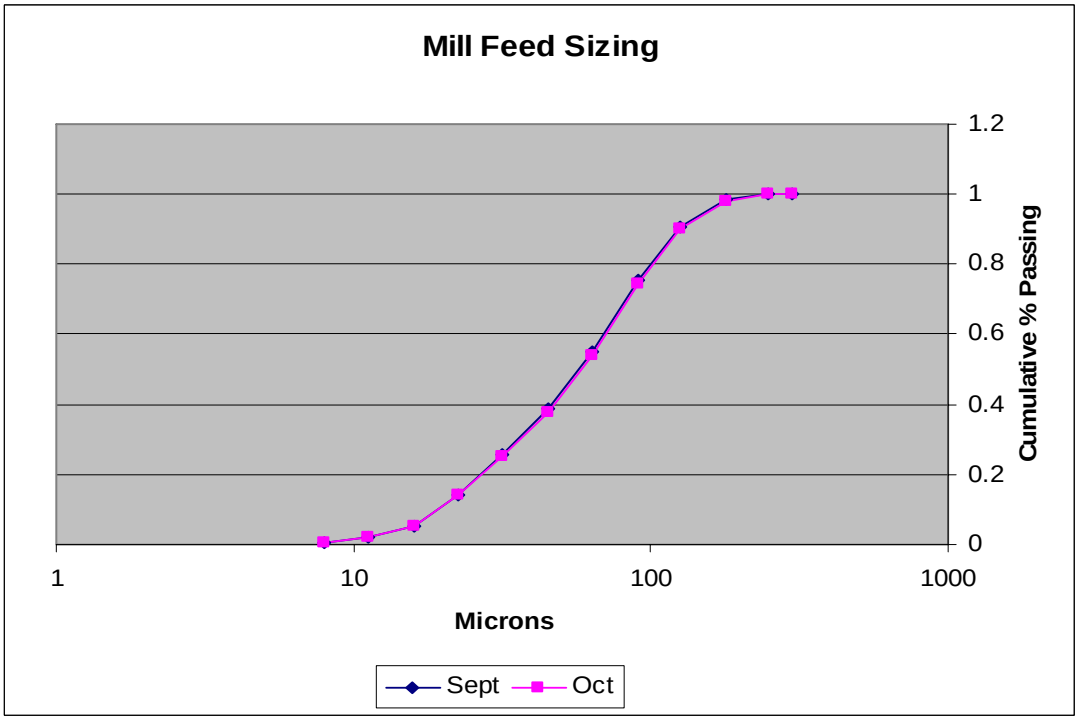



Figure 6.13 Mill feed sizing, September and October 2006.

## 6.3 Conclusions: Medium Term Data

With the milling circuit running in steady state conditions the effects of feed size on the milling circuit become more visible. There is a good, positive correlation between the feed size and the product size which is most apparent in the middle ranges of each measurement. This correlation may be expressed as an equation of the form:

Product %-75 µm = 0.8 x ( Feed % -31.5mm ) + 17

Feed sizing also appears to influence the power drawn by the mill in that increasing quantities of the finer fractions decreases mill power and that increasing quantities of coarse material increases the mill power. The critical size which has least effect on power is the 31.5 mm to 45 mm fraction. This finding seems to be independent of the mass proportions within the size ranges.

Modelling gives an indication of the response of the mill, but the outputs of the model in terms of product sizing are not an accurate indication of the measured results. The reasons have been discussed in paragraph 5.5, above.

# 7 CONCLUSIONS

Detailed inspection of the individual periods and comparisons between them strongly indicates that a change in feed to a significantly finer range correlates with an improvement in the product size. There is a good, positive correlation between the feed size and the product size when other variables are relatively constant. This is most apparent in the middle ranges of each measurement. It is postulated that this may be due to the method of calibration of the product particle size monitor where the calibration of the percentage of -75 µm material is the most critical factor. The best such correlation may be expressed as an equation of the form:

Product %-75 µm = 0.8 x ( Feed % -31.5mm ) + 17

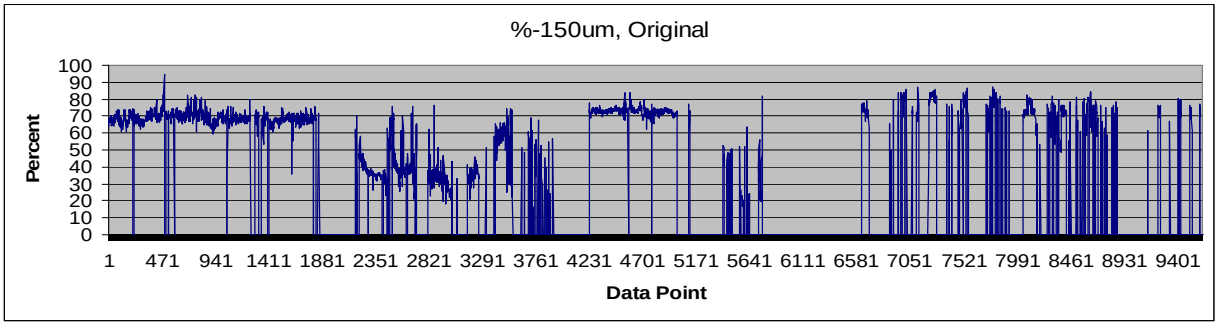
Feed sizing also appears to influence the power drawn by the mill in that increasing quantities of the finer fractions decreases mill power and that increasing quantities of coarse material increases the mill power. The critical size which has least effect on power is the 31.5 mm to 45 mm fraction. This finding seems to be independent of the mass proportions within the size ranges, although it has only been seen in one data set.

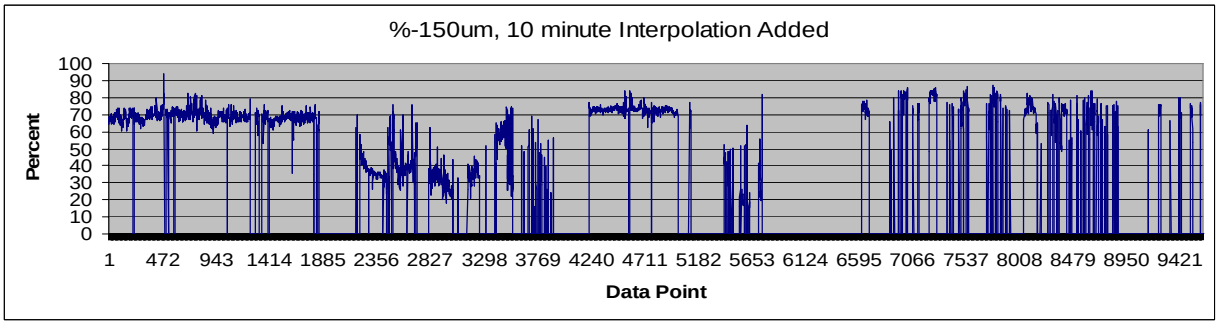
Modelling gives an indication of the response of the mill, ie the product becoming finer or coarser, but the outputs of the model in terms of product sizing are not an accurate indication of the measured results

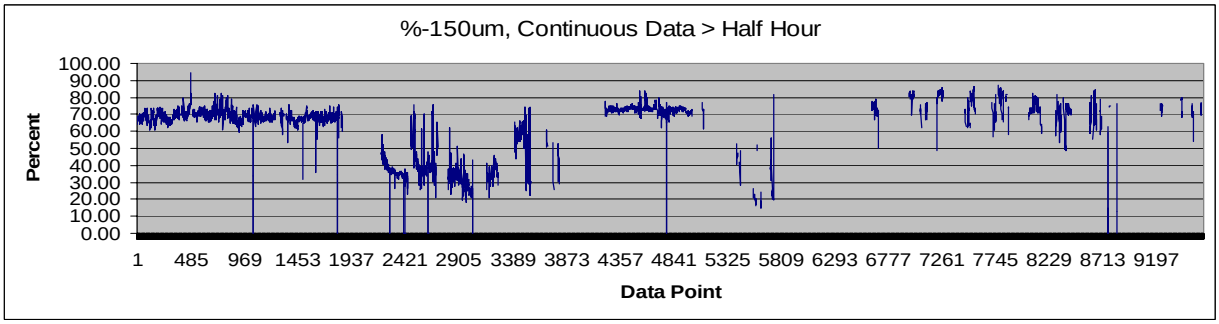
In general, the JKSimMet power model gives a good fit over a day, with the trends being followed. The correlation between observations and the model using five minute averages is 0.82 over periods of some hours or 0.93 over a month. Short-term changes over periods of 30 minutes in the cyclone/screen feed flow rate appear to be the main factor affecting the fit. This is due to the mill load pulp density being back-calculated from the cyclone feed flow and density data.

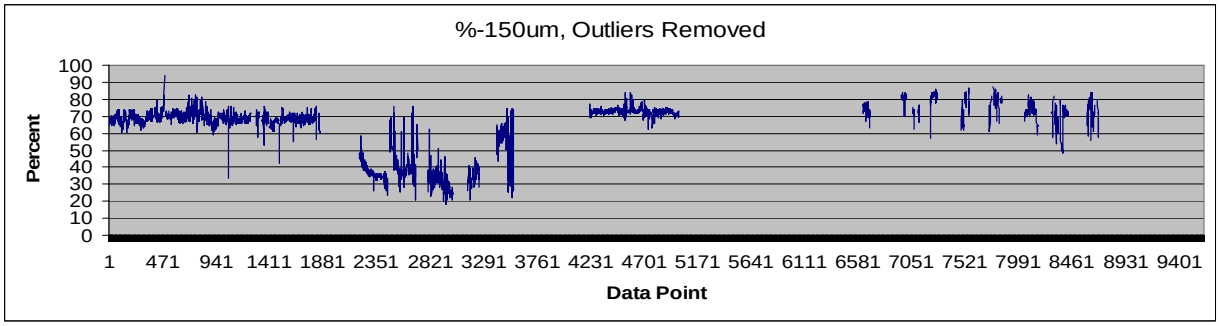
The JKSimMet ball and SAG models produce poor model fits. It was concluded the parameters used to develop these models are well outside the range of the data obtained from this exercise.

# LONG - TERM DATA SET: EXAMPLE OF DATA MANIPULATION



**REFERENCES**


Apelt, T.A. (2002) Inferential Measurement Models for Semi-autogenous Grinding Mills, University of Sydney.

Bouajila, A., Bartolacci, G., Kock, N., Cayoucette, J., and Cote, C. (2000) Toward the Improvement of Primary Grinding Productivity and Energy Consumption Efficiency, Part 1: Investigation of the Feed Ore Size Effects. *IFAC 2000*.

Bouajila, A., Bartolacci, G. and Cote, C, (2001) The impact of feed size analysis on the autogenous grinding mill, *Proceedings, SAG 2001,* Vancouver, Canada.

Clarke-Mostert, V (2005) Benchmark Mineralogical Investigation of Waterval UG-2 Concentrator Samples – June 2005, Anglo Platinum internal report.

Dance, A, (2001) The Importance of Primary Crushing in Mill Feed Size Control, *Proceedings, SAG 2001,* Vancouver, Canada.

Gillot, P.L. ( c2004) Pit to Plant Optimisation at Morila Gold Mine, INTERNET.http://www.splitonline.co.us. Cited 29 July 2005.

Hart, S., Valery, W., Clements, B., Reed, M., Song, M. and Dunne, R, (2001) Optimisation of the Cadia Hill SAG Circuit, *Proceedings, SAG 2001,* Vancouver, Canada.

Hinde, A. and Pearson, C. (2001) Conversion of Primary Run-Of-Mine Ball Mills at Impala Platinum Limited to Fully Autogenous Mills, *Proceedings, SAG 2001,* Vancouver, Canada..

Kumalo, T. (2002) Comparative Study between CSIRO Ultrasonic Particle Size Analyzer and Sympatec In-line Particle Size Analysis, Anglo Platinum internal report.

Lam, M., Jankovic, A., Valery, W. and Kanchibotla, S. (2001) Maximising SAG Mill Throughput at Porgera Gold Mine By Optimising Blast Fragmentation, *Proceedings: SAG 2001*, Vancouver, Canada.

McGhee, S, Mosher, J, Richardson, M, David, D and Morrison, R, (2001), SAG Feed Pre-crushing at ASARCO's Ray Concentrator: Development Implementation and Evaluation, *Proceedings: SAG 2001*, Vancouver, Canada.

Mainza, A.N, and Powell, M.S. (2006) ROM Ball Mills - a comparison with AG/SAG Milling, *Proceedings: SAG 2006*, Vancouver, Canada.

Major, K. (2006) Precrushing SAG Mill Feed, *Proceedings: SAG 2006*, Vancouver, Canada.

Morrell, S. (2003) The Influence of Feed Size on Autogenous and Semi-autogenous Grinding and the Role of Blasting in its Manipulation, *Proceedings: XXII International Mineral Processing Conference*, Cape Town, South Africa.

Morrell, S. and Valery, W, (2001) Influence of Feed Size on AG/SAG Mill Performance, *Proceedings: SAG 2001*, Vancouver, Canada.

Mwehonge, G.L.K. (2006) Crushing Practise Impact on SAG Milling: Addition of Secondary Crushing Circuit at Geita Gold Mine, *Proceedings: SAG 2006*, Vancouver, Canada.

Nelson, M., Valery, W. and Morrell, S, (1996) Performance Characteristics and Optimisation of the Firmiston (KCMG) SAG Mill Circuit, *Proceedings: SAG 1996*, Vancouver, Canada.

Napier-Munn, T.J., Morrell, S., Morrison, R.D. and Kojovic, T. (1999) *Mineral Comminution Circuits, Their Operation and Optimisation*, JKMRC, pp206-229.

Plint, N., Bye, A., Dippenaar, J. Fanner, R. (2005) The Evaluation of On -line Optical Rock Sizing, Anglo Platinum internal report.

Powell, M., Morrell, S. and Latchireddi, S. (2001) Developments in the understanding of South African style SAG mills, *Minerals Engineering*. Vol. 14 No. 10, pp. 1143-1153.

Sherman, M, (2001), Optimisation of the Alumbrera SAG mills, *Proceedings: SAG 2001*, Vancouver, Canada.

Silvestre, Y., Abols, J. and Baratt, D, (2001) The Benefits of Pre-Crushing at the Inmet Troilus Mine, *Proceedings: SAG 2001*, Vancouver, Canada.

Simkus, R., Dance, A. (1998) Tracking Hardness and Size: Measuring and Monitoring ROM Ore Properties at Highland Valley Copper, *Proceedings: Mine-to-Mill 98*, Brisbane, Australia.

Strohmayr, S., and Valery, W, (2001) SAG Mill Circuit optimisation at Ernest Henry Mining, *Proceedings: SAG 2001*, Vancouver, Canada.

Wills. B.A., *Minerals Processing Technology*, (1981) Pergamon international library, pp 151-187.