UNIVERSITY OF CAPE TOWN

CENTRE FOR MINERALS RESEARCH

Effect of operating variables on IsaMill^{TM} performance using Platinum bearing Ores

by

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Declaration

I declare that this thesis, submitted for the degree of Master of Science in Engineering at the University of Cape Town, is my own unaided work. It has not been submitted before, for any degree or examination, at this or at any other university.

Brian Chaponda.

Abstract

Comminution involves crushing and grinding operations. The grinding operations use the traditional tumbling mills and stirred mills to reduce the ore to the required fineness. This thesis intends to investigate the influence of design and operating variables on the IsaMill^{TM} specific energy and product size, when grinding UG2 platinum-bearing ore. The main objectives of this work were to study the effects of operating variables on specific energy consumption and product fineness, and to investigate IsaMill^{TM} scale-up protocol. The experimental studies were conducted using the M4 IsaMill^{TM} on a laboratory scale and the M10 000 IsaMill^{TM} on an industrial scale.

The laboratory scale M4 IsaMill^{TM} was used to investigate the effects of some design and operating variables on the energy consumption and product fineness for UG2 PGM (Platinum Group Metals) bearing ore. There are many operating and design variables that have been shown to have a significant influence on the stirred mill operations (Clark *et al.*, 2004; Jankovic, 2003; Pease *et al.*, 2005; Zheng *et al.*, 1996). This study, however, has focused only on variables that influence specific energy consumption and the product fineness of grind. The variables investigated include; stirrer speed, media load, media size, feed size, solids concentration and flow rate.

Sampling campaigns were conducted on the M10 000 industrial scale IsaMill^{TM} to evaluate the performance of large scale units. The campaigns were conducted at Anglo Platinum’s Waterval UG2 Concentrator and Western Limb Tailings Re-treatment Plant (WLTRP), which are both located in the Rustenburg area in South Africa.

Waterval Concentrator treats UG2 platinum ore and has two IsaMill^{TM} operating in mainstream grinding. The WLTRP, on the other hand, treats reclaimed material from the old Klipfontein tailings dam. The reclaimed material contains a mixture of UG2 and Merensky ore. The WLTRP has one IsaMill^{TM} installed in fine-grinding applications to re-grind concentrates. While the results obtained from the test work have shown that when the IsaMill^{TM} mill is operated at different speeds the energy required to grind UG2 ore of feed F_{80} = 120µm to a given product size (P_{80}) varies. It was seen that lower speeds

(1500 rpm) required more energy for given product size, increasing the speed to 1800 rpm resulted in a decrease in energy and further increase led to higher energy utilization. This indicates an existence of optimum speed when grinding UG2 ore of F_{80} = 120µm at 1800 rpm. In terms of the effect of media load, it was found that different media loads are required to efficiently grind various feed sizes (F_{80}) of UG2 ore to desired product. Therefore, the IsaMill^{TM} media load should be optimized at different levels for efficient grinding process. This study has indicated that the optimum media load is dependant on feed particle size.

This test work also indicates that the performance of the IsaMill^{TM} is greatly affected by the media size and feed particle size distribution. For the same stirrer speed, media load and slurry percent solids, it was found that the best grinding efficiency when grinding UG2 ore of feed sizes F_{80} = 55µm and F_{80} = 120µm were achieved when the 2mm media was used. However, the 3.5mm media was the most efficient media to grind the UG2 ore of F_{80} = 250µm. Therefore, the ratio of media to feed size must be matched and optimized in order to maximize the grinding efficiencies.

Comparison of the data obtained from the sampling campaigns conducted on M10000 and the M4 IsaMill^{TM} test work exhibited a consistent behaviour of industrial IsaMill^{TM} that is closely matched by results achieved using the M4 mill. This suggests that the M4 laboratory scale IsaMill^{TM} can be utilized to accurately estimate the energy required to prepare desired product fineness in M10000 industrial scale IsaMill^{TM}. Therefore, the M4 can be used to generate data for design and operations optimization of industrial scale IsaMill^{TM}.

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Nomenclature

Mainstream grinding	– Grinding materials with feed sizes of more than 75µm
Fine grinding	– Grinding feed materials with feed sizes of less than 75µm
F_{80}	– Size at which 80% of feed material passes
P_{80}	– Size at which 80% of product material passes
RoM	– Run of Mine Ore
MIG	- Mainstream Inert Grinding
UFG	- Ultra Fine Grinding
PGMs	- Platinum Group Metals
Comminution	- A processing technique of reducing large particles to smaller sizes in order to liberate or free the minerals of interest
PEPT	- Positron Emission Particle Tracking

Dedication

To my mother -- for being such a great mum -- and all the sacrifices made to accord me the opportunity of getting such a good education. I can only hope that I have lived up to your expectations. To you, I will remain forever grateful, for facilitating the realization of my dreams.

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Last, but not least, special thanks to my parents and family for their encouragement, emotional and social support with which they provided me throughout this work.

1.0 Introduction

This thesis aims to investigate the influence of design and operating variables on the IsaMill^{TM} energy consumption and product size, when grinding UG2 platinum-bearing ore. The IsaMill^{TM} was originally designed for fine grinding application at Mount Isa Mines in Australia to liberate the lead/zinc minerals; it has in the recent past been configured for mainstream grinding application elsewhere. Studies to investigate the effects of design and operating variables on energy consumption and product fineness were performed on the M4 IsaMill^{TM}. A number of plant scale tests were also conducted on the M10 000 IsaMill^{TM} to evaluate the performance of large scale units. There are many operating and design variables that have a significant influence on the operation of the IsaMill^{TM} (Clark *et al.*, 2004; Jankovic, 2003; Pease *et al.*, 2006). This study focuses on the variables that influence specific energy consumption and the product fineness. The variables investigated were stirrer speed, grinding media size and load, feed size distribution, solids concentration and the feed flow rate.

Comminution is the breaking down of relatively coarse particles in order to liberate or unlock finely disseminated valuable minerals from the gangue minerals. The size to which the ore should be reduced is governed by the requirements of the subsequent process and the grain size of the valuable mineral (Wills & Nappier-Munn, 2006). In the mineral processing industry, the run of mine ore (RoM) is mined from open pit or underground mines and sent to the concentrator for the extraction of valuable minerals that are closely interlocked with gangue minerals. The purpose of the concentrator is to separate selectively a mineral-rich concentrate from the barren gangue contained in the bulk RoM ore. This is done by using physical methods in comminution to liberate or unlock the valuable minerals from gangue minerals before the actual separation process can be achieved by using either froth flotation, or any other suitable technique.

Comminution involves crushing and grinding operations. Grinding is an energy-intensive operation and is the highest consumer of power in mineral processing applications (Tromans, 2007). It is therefore, imperative to operate and control the grinding operations efficiently for the mine to be profitable. In most cases, water is added to the ground ore to form a pulp of the required solids concentration; which is then sent to the flotation circuit for the separation of valuable minerals from gangue material (Jain, 2001). The energy consumption in conventional comminution devices increases exponentially with the reduction of product size making them uneconomical for use in fine grinding. To meet the challenges of the low cost of production for ores that require finer grinding to liberate the valuable mineral, the mining industry has embarked on research to investigate technologies that can be applied to process high tonnages with relatively low power consumption. This has led to the development of specialized fine grinding mills that have better grinding efficiencies than conventional tumbling mills (Kwade, 1999). These specialized mills include stirred mills and other alternative mills, such as the Tower mill, Pin mill, Stirred Media Detritors and IsaMill^{TM} (Napier-Munn *et al.*, 1999).

The IsaMill^{TM} is increasingly being used in the minerals industry for grinding applications due to its high energy efficiency and the inert grinding environment (Jayasundara *et al.*, 2010). The IsaMill^{TM} which is indicated in Figure 3-1 is a horizontal stirred mill consisting of a motor, a gearbox and a grinding chamber. Discs mounted on a shaft inside the mill chamber rotate at high tip speeds resulting in high energy intensities. The mill is filled with a suitable grinding material, and the area between each disc essentially acts as a compartmentalized grinding chamber. As a result, the mill is effectively 6 - 8 grinding chambers operating in series, resulting in the low probability of feed short-circuiting to the discharge end without being involved in size reduction.

At the end of the mill is a product separator, consisting of a rotor and a displacement body. The distance between the last grinding disc and the rotor disc is smaller than the distance between two grinding discs to allow for the centrifuging of any coarse particles towards the outside of the mill and in essence acting as a classifier (Pease, 2007). The operating mechanism of the product separator precludes the use of screens or cyclones in the retention of media in the mill. IsaMill^{TM} can therefore, be operated in open circuit without cyclones; making the circuit simpler and providing a way of reducing capital, operating and maintenance costs without compromising performance.

Most of the expansions in the platinum industry in South Africa involve mining and processing the UG2 Platinum-bearing ore which generally has Platinum Group Metals (PGMs) finely disseminated in gangue minerals requiring fine grinding for effective liberation of the locked PGMs. The processing of UG2 ores has been reported (Rule *et al.,* 2008) to produce a significant proportion of PGM losses in the final tailings, occurring as locked or middling particles. This is because the PGMs in UG2 ore are smaller than 25μm and, therefore, require finer grinding to liberate them (Becker *et al.,* 2001; Rule et al., 2008). This has led to the introduction of IsaMill™ for finer grinding of UG2 ores to achieve better mineral liberation and facilitate improved recovery of minerals. Due to the relative successes in finer grinding, stirred mills are now being introduced for main stream grinding applications in the Platinum industry. However, there is need to evaluate machine and operating variables that can assist in achieving benefits obtained in fine grinding when these units are applied in main stream grinding.

In this work, studies to investigate effects of operating variables on specific energy consumption and product fineness were performed on the M4 IsaMill™. Two IsaMill™ scales were used in this work, namely M4 (4 litre laboratory scale) and M10 000 (10 000 litre industrial scale). The M4 was used to investigate the influence of key operating and design variables on the IsaMill™ performance. The M10 000 IsaMill™ industrial scale was used to evaluate the performance of this type of mill on a large scale, and to assess whether M4 test results can be used to provide design guidance for the M10 000 IsaMill™.

1.1 Hypothesis

The following hypothesis will be tested in this thesis:

- Breakage in stirred mills depends on the level of stress applied which is determined by stirrer speed, media size and media load. The dominant grinding mechanisms during the particle breakage processes in stirred mills are abrasion and attrition.

In stirred mills the ability to effectively produce fines is dependent on high stirrer speeds resulting in high shear stresses acting to create particle breakage. The use of small media also results in increased grinding surfaces per unit volume. An optimum ratio of media to particle size exists for efficient particle breakage. At this optimum ratio, the particles are large enough to have a high probability of contacting the media, but are small enough to be effectively caught and broken by the media.

1.2 Thesis objectives

The general objectives of this research are summarized as follows:

1. To investigate the effect of different feed sizes on the specific energy consumption and required media size for optimum operation;
2. To assess the effects of altering the operating variables, such as stirrer speed, mill media load, media size, solids concentration and flow rate on the IsaMill^{TM} specific energy consumption and product fineness;
3. To conduct sampling campaigns on industrial scale IsaMill^{TM} in order to evaluate the performance of large-scale units;
4. To generate correlations of operating variables with specific energy consumption, and product fineness;
5. To assess the IsaMill^{TM} scale up protocol.

1.3 Thesis lay-out

This thesis comprises six chapters. A review of the literature on comminution -- ball mills and stirred mills is given in Chapter 2. The experimental methodologies employed in this work are described in Chapter 3. Chapter 4 presents and discusses the results obtained from the tests performed on the M4 and M10 000 IsaMill^{TM} scales. A comparison of the results obtained using the M4 and those from the M10 000 is also given in Chapter 4. Chapter 5 presents a statistical analysis of the M4 IsaMill^{TM} results and correlations between the operating variables and specific energy, product fineness and capacity are provided. The conclusions drawn from this work are presented in Chapter 6.

2.0 Literature review

This chapter provides a review of the literature on tumbling and stirred mills. A background to comminution principles is also discussed in order to give a good understanding of the research topic. Stirred mills are discussed in greater depth by looking at operating and design variables that have a significant influence on energy consumption and product fineness.

2.1 Introduction

The comminution process aims at providing a product size with desired liberation characteristics, for the subsequent recovery process. Different mill characteristics and operating conditions affect the product due to variations between the fracture, attrition and abrasion components. This, in turn, affects the size distribution, liberation and surface characteristics of the product. The mill product characteristics are, therefore, closely related to the type of mill used (Lichter and Davey, 2002).

Expansions in the mineral processing industry have led to the processing of ores that have fine particle intergrowth, either with other minerals or with gangue. This has led to the need for finer grinding, to achieve good mineral liberation and improved recovery of these finely disseminated minerals (Becker *et al.,* 2001; Rule *et al.,* 2008). Generally, high amounts of energy are required to grind the ore finer. This imposes limitations on production. There is evidence that conventional ball mills are capable of grinding more finely than is traditionally accepted (Napier-Munn *et al.,* 1999), however, they are extremely wasteful in terms of energy utilization and could not, therefore, be economically used for fine grinding applications (Kwade, 1999). The actual proportion of the energy effectively used in size reduction for tumbling mills is thought to be very low -- perhaps in the range of 1–5% of the total energy input to the mill (Cleary, 2001). Significant financial and environmental benefits can be obtained by improving this efficiency even slightly.

Substantial benefits are also potentially available through higher downstream recoveries of the liberated mineral if the product particle size distributions (PSD) can be tailored for subsequent flotation or other separation processes. To meet the challenges of low production costs for ores that require grinding finer, the mining industry has embarked on research to investigate technologies that can be applied to process high tonnages at relatively low energy consumption. The stirred mills have been the enabling technology of many fine and ultra fine grinding applications. These include applications in the paint, chemical, pharmaceutical and mineral processing industries (He *et al.*, 2006; Stender *et al.,* 2004; Varinot *et al.*, 1999).

It has been reported in the literature that stirred mills are more energy efficient than conventional tumbling mills when grinding particles finer than ~100 μm (Pease *et al.,* 2005; Zheng *et al.,* 1996; Wang & Forssberg, 2007). This could be attributed to the stirred mills' capability of using very small grinding media of ~ 3mm, whilst the smallest possible grinding media in the tumbling mills is ~ 12mm (Burford & Clark, 2007; Wills, 1997). Therefore, stirred mills have a more suitable particle/media ratio for fine grinding than tumbling mills. In addition, the high energy intensity in stirred mills means that they are much smaller in size for the same production rate compared to tumbling mills. Because of the relative successes in finer grinding (F_{80} less than 75μm), stirred mills are now being introduced for mainstream (F_{80} of up to 150μm) grinding applications. However, there is a need to evaluate machine and operating variables that can be used to deliver the benefits obtained from fine grinding, when these units are applied in mainstream grinding.

2.2 Comminution

2.2.1 General comminution principles

The well-known laws of comminution have been the basis for assessing comminution energy, and indirectly, comminution efficiency. As reported by Wills and Napier-Munn

(2006), theoretical and empirical energy-size reduction equations proposed by Rittinger (1867), Kick (1885) and Bond (1952) -- commonly known as the three theories of comminution -- have been used to assess comminution efficiency. These laws are concerned with the relationship between energy consumption and product particle size achieved when grinding a given feed size.

Von Rittinger (1867) considered that the required energy for comminution is proportional to the new surface area produced. Since the specific surface area is inversely proportional to the particle size, Rittinger's hypothesis can be written in the following form:

$$E = K\left(\frac{1}{D_2} - \frac{1}{D_1}\right)$$ **Equation 2-1**

Where E is the energy input, D_1 is the initial particle size, D_2 is the final particle size, and K is a constant.

Von Rittinger's equation accounts only for the energy required to produce new surfaces. There are many other energy aspects not included in this equation, such as material resistance to crack propagation. This must be overcome for breakage to occur (Rodrigues *et al.*, 2005).

Kick (1885) considered that the energy for comminution is determined by the energy required to stress the particle to failure. Kick's equation is written as follows:

$$E = K_1\left[\frac{1}{x_p} - \frac{1}{x_f}\right]$$ **Equation 2-2**

Where E is the net specific energy; x_f and x_p are the feed and the product size indices, respectively; and K_1 is a constant.

The drawback with Kick's law is that it assumes that the energy required to achieve a given amount of size reduction will be the same for equivalent changes in particle size (Donovan, 2003). The effect of size on the amount of energy required for fracture is not considered by Kick's equation. However, as will be shown later, smaller particles would require more energy to break, which indicates increasing energy requirements with decreased particle size.

Bond (1952) developed an equation that is based on the theory that the work input is proportional to the new crack-tip length produced in particle breakage, and equals the work represented by the product minus that represented by the feed. This is expressed by Bond's equation (Morrel, 2007):

$$W = \frac{10W_i}{\sqrt{P}} - \frac{10W_i}{\sqrt{F}}$$ **Equation 2-3**

Where W_i is the work index, P is the particle size at which 80% product passes, and F is the particle size at which 80% of the feed passes.

Because of the widespread acceptance and use of Bond's equations over the years, comparisons that have been made of different circuit designs have invariably relied on his approach, particularly the use of so-called ‘‘operating work indices’’ (Morrell, 2007).

The operating work index associated with Bond's equation is written as:

$$W_i = W/10\left(\frac{1}{\sqrt{P}} - \frac{1}{\sqrt{F}}\right)$$ **Equation 2-4**

In theory, the operating work index can be used to compare different circuits that have different feed and product sizes. Provided the ore is the same, higher operating work indices imply less efficient circuits. Different circuits treating different ores can, in theory, also be compared by using the ratios of their operating work index (Morrell, 2007).

Various attempts have been made to show that the relationships of Rittinger, Kick and Bond are interpretations of single general equations (Fuestenau & Abouzeid, 2002; Morrell, 2007). Hukki (1975) suggested that the relationship between energy and particle size is the composite form of the three laws. The probability of breakage in comminution is high for large particles, but rapidly diminishes for fine sizes.

Hukki showed that Kick's law is reasonably accurate in the crushing range above about 1cm in diameter; Bond's theory applies reasonably in the range of conventional rod mill and ball mill grinding; and Rittinger's law applies fairly well in the fine grinding range of 10 – 1000μm. Hukki's revised energy-size equation has the following form:

$$dE = -C\frac{dx}{x^{f(x)}}$$ **Equation 2-5**

Where E is the net specific energy; x is the characteristic dimension of the product; $f(x)$ is the exponent indicating the order of the process, and is dependent on the characteristic dimension of the particle; C is the constant.

Schoenert (1972) took a different approach towards assessing comminution efficiency by considering that the upper limit of energy utilization in size reduction processes is that required for the mechanical breakage of single particles. This means that the maximum utilization of energy is only obtained when inter-particle friction and inter-particle energy transfer are absent.

2.2.2 Comminution models

Models have been developed to assess the comminution efficiency in both tumbling and stirred mills. The most popular model for tumbling mills is the "population balance model", which was introduced by Epstein (1947), and further developed by several researchers (Whiten, 1974; Herbst & Fuerstenau, 1968, 1973; Austin & Shah, 1983). For this model, a "first-order breakage rate" is assumed. First-order breakage rate is based on the hypothesis that for a given size range of particles, for example a $\sqrt{2}$ screen interval, the production of ground material per unit time within the mill depends only on the mass of that size fraction that is present in the mill content (Austin, 1990).

Breakage rate of size fraction $j = S_j w_j(t) W$ **Equation 2-6**

Where:

S_j is the specific breakage rate of size j,

W is the mill hold-up,

$w_j(t)$ is the weight fraction of size j material at grinding time t.

Therefore,

$$d w_j(t)/d_t = -S_j w_j(t) \quad \textbf{or} \quad w_j(t) = w_j(0) e^{-S_j(t)}$$ **Equation 2-7**

For a batch mill, the population balance equation is:

$$d w_i(t)/d_t = -S_i w_i(t) + \sum_{\substack{j=1 \\ i>1}}^{i-1} b_{ij} S_j w_j(t) \quad \textbf{Where } n > i > j > 1$$ **Equation 2-8**

Where, b_{ij} is the primary breakage distribution function.

The primary breakage distribution function describes the weight fraction of the products that occur in the size interval i if the material size interval j is broken. Equation 2-8 states that the net rate of production of size material equals the sum of the rates of appearance from the breakage of all sizes larger than size i, minus the rate of disappearance from breakage of size i by breakage. The population balance model has been used in scaling up of tumbling mills (Datta & Rajamani, 2001; Herbst & Fuerstenau, 1968, 1973; Austin & Shah, 1983). The selection and breakage functions are determined in a small laboratory mill. The laboratory experiments are performed with identical feed materials and operating conditions. These parameters are then scaled up for bigger industrial mills (Datta & Rajamani, 2001).

Many researchers have conducted studies to assess the use of population balance in stirred mills (Gao & Forssberg, 1994; Tuzun *et al.*, 1994; Kwade, 1999; Yue & Clein, 2005). However, there has been no significant effort to establish the region of transition for grinding from first- order to non-first-order to establish the range of size fractions where the population balance is applied in stirred mills.

Studies conducted by Kwade, (1999), Yue and Klein, (2004) show that first-order breakage in stirred mills can be assumed. The first-order breakage rate implies that a massive fracture-breakage mechanism was dominant for the operating conditions used in these studies. Therefore, the population balance can be used to characterize breakage in stirred mills. The first-order breakage in stirred mills, according to Yue and Klein (2004) can be fitted better using the Rosin-Rammler equation than the Gaudin-Shuhmann equation.

Results however, fitted poorly, as the particle size reduces to below 5µm. This indicates that there is a drift from first-order breakage in the very fine size range. This was confirmed using theoretical models by Hogg (1999) and Gao & Forssberg (1994). They suggested that first-order breakage in stirred mills during fine grinding does not exist; and therefore, the population balance cannot be used to characterize breakage.

Hogg’s theoretical model for breakage kinetics in stirred mills accounts for breakage by massive fracture and attrition. Hogg suggested that attrition becomes more important as particles become finer, particularly in the micron range. His model indicates that attrition grinding results in the acceleration of the particle degradation and non-first-order breakage occurs. In this case, the population balance equation could not be applied to stirred mill grinding -- where attrition is considered to be a major grinding mechanism.

Comparing the results obtained by Kwade (1999) with those obtained by Hogg indicates that the grinding mechanisms in stirred mills are highly influenced by the particle size of the feed, with attrition becoming more dominant in finer feed materials. However, no significant work has been done to better the understanding of grinding mechanisms in stirred mills.

Researchers have proposed models to analyse fine grinding processes, most of which though claimed to be flexible and applicable to various abrasion and attrition processes have complicated forms and a large number of parameters. This discourages their use in most engineering applications. Failing to produce an analytical model of the fine grinding process, empirical correlations between energy input and size reduction have been used. A commonly used correlation is that from Charles work (1957), which when the product is much finer than the feed takes the form shown in Equation 2-9.

$$E = ax_m^{-\propto} \qquad \textbf{Equation 2-9}$$

Where E is the specific grinding energy (kWh/t), a and $-\propto$ are constants, x_m is the characteristic size of the product.

However, because only one characteristic size of the product is being used, its application to predict the whole size distribution is limited. Equation 2-9 will, however, help to predict the size-reduction progress and the milling energy consumption, to compare the efficiency of different fine grinding devices, to optimize the system, as well as to yield an understanding of the process mechanism possible. The relationship between product size and the specific energy has been used in the scaling up of stirred grinding mills (Curry *et al.,* 2005; Weller *et al.,* 1999) and will be discussed further in section 2.4.3.

2.3 Tumbling mills

Tumbling mills are cylindrical or cylindro-conical in shape and rotate about a horizontal axis. These include ball mills, autogenous and semi-autogenous mills and rod mills. A load of crushing bodies, either rods, balls, or pebbles -- called the grinding media -- forms part of the mill load (Wills & Napier-Munn, 2006; Jain, 2001). Tumbling mills are conventionally used in grinding applications. The grinding media in the mill is agitated by the rotating vessel, and in the process comminute the ore particles. The mechanism of grinding in tumbling mills is mainly that of impact and abrasion (Wills, 1997). A schematic indication of the different zones in a tumbling mill is shown in Figure 2-1. The tumbling mill used in continuous grinding is nearly half-filled with the charge media and the ore is fed in the mill at one end, and discharges continuously at the other end. At the feed end, water is added (wet grinding) to assist in transporting the material through the mill (Jain, 2001).

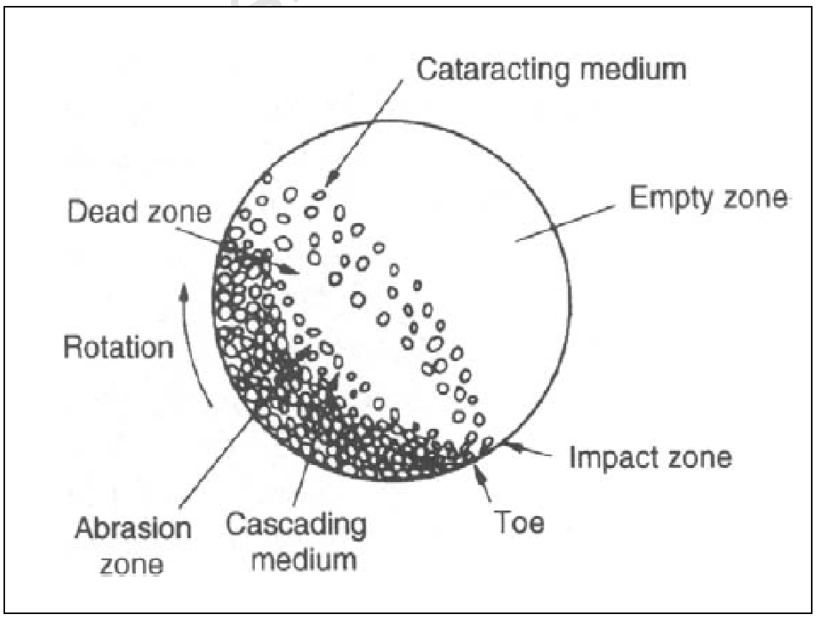

Figure 2-1: Motion of charge in a tumbling mill (After Wills, 1997).

Control of the grinding section is very important, as grinding significantly affects mineral processing as a whole. Under-grinding of the ore will result in a product which is too coarse, with liberation too low for economic separation. This leads to poor recovery and the enrichment ratio achieved in the flotation process. Needless over-grinding reduces the particle size of the material below the size required for most efficient separation. Much energy is then wasted in the process.

Although tumbling mills have been developed to a high degree of mechanical efficiency, they are extremely wasteful in terms of the energy expended. The power draw is dictated by the motion and magnitude of the charge (Fuerstenau & Abouzeid, 2002). As a result of the rotation of the mill shell, the grinding medium is lifted until a position of dynamic equilibrium is reached, when the bodies cascade and cataract down to the toe of the mill charge. The ore is mostly broken, as a result of repeated, random impacts and the abrasion caused by the cataracting and cascading charge; which breaks the liberated, as well as the unliberated particles (Wills & Napier-Munn, 2006). Size reduction can therefore be viewed as a by-product of moving the charge around in the mill.

The efficiency of tumbling mills is influenced by several factors, such as charge volumetric filling, slurry density, grinding media size and mill speed. The pulp density must be as high as possible, but low enough so that it does not affect the easy of flow of slurry through the mill. Usually, the pulp density is between 65% and 80% solids by weight, depending on the ore and the type of mill (Wills, 1997). The efficiency of grinding also depends on the surface area of the grinding media. Therefore, balls should be as small as possible and the charge should be graded such that the largest balls are just heavy enough to grind the largest and hardest particles in the feed (Partyka & Yan, 2007).

Ball mills are the most commonly used type of tumbling mill. They dominate the minerals industry over a wide feed and product-size range, from a few millimetres to a few tens of microns. Ball mills are used as primary grinding mills with feed up to 20mm, as well as secondary/tertiary and regrinding operations with fine feed and products. At product sizes finer than 80% passing 75µm, the efficiency of ball mill grinding rapidly decreases. The practical limit to ball mill product fineness is considered as 40 - 45μm (Shi *et al*, 2009; Wills & Nappier-Munn, 2006; Morrell, 2007; Weller & Gao, 1999). However, these product sizes are usually not achieved, as it becomes uneconomic for ball mills to grind to such fineness.

2.4 Stirred mills

2.4.1 General features of stirred mills

Stirred media mills have stationary mill chambers, and the motion of charge is imparted by the movement of an internal stirrer. Stirred media mills are different from tumbling mills, such as ball mills, where the motion is imparted to the charge via the rotation of the mill shell. Stirred mills are operated with a set of stirrers inside the mill which rotate at high tip speeds. The mill is usually loaded with small media that are agitated at high intensity by the rotating stirrers. The high intensity stirring action in stirred mills, according to Gao *et al.* (2007) efficiently energises small grinding media and produces numerous compressed and rapidly rotating media layers that create compressional and torsional forces. These forces are far more efficient for breaking micron particles than impact or abrasive stresses generated inside ball mills (Gao *et al.*, 2007).

Stirred mills differ from one another in terms of the design of the stirrer they incorporate. Typically, stirred mills comprise a stationary cylindrical chamber that can be mounted either vertically or horizontally. The differences between the stirred mills available in the mineral processing industry are evident in terms of stirrer design, mill orientation, power intensity and the methods of separating the slurry product from media and coarse particles (Weller & Gao, 1999). The stirrer designs that are used in stirred mills are shown in Figure 2-2.

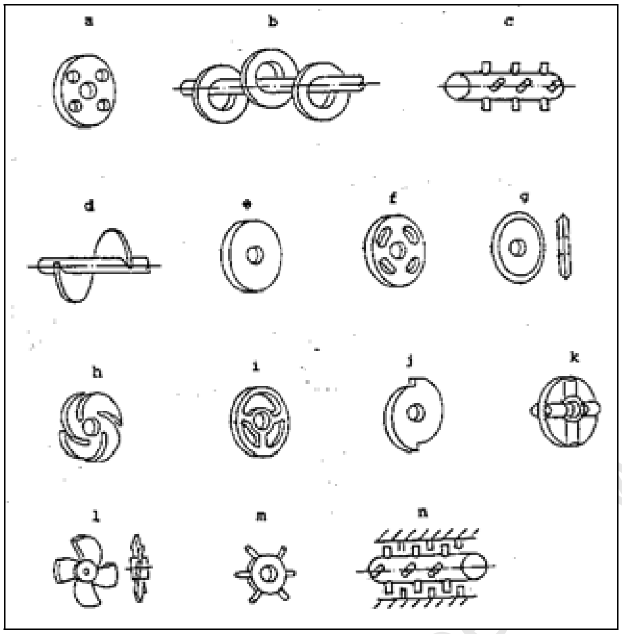

Figure 2-2: Stirrer designs for various stirred mills (After Weller & Gao, 1999)

Weller and Gao (1999) reported that only two designs of stirrers used in vertical stirred mills are applied in mineral processing, while horizontal stirred mills are available with most of the stirrer designs (shown in Figure 2-2). The restriction on stirrer types used in vertical stirred mills could be due to the tendency of the mill charge overflowing at the top of the mill chamber. Hence, stirrers with lower overflow tendencies are selected for vertical mills.

The horizontal stirred mills use a closed milling chamber, in which the milled product is separated from the media mechanically using a screen or a ring gap at the discharge end (Kwade, 1999). A certain pressure needs to be maintained in the milling chamber to keep the mill charge suspended and to retain the necessary residence time for achieving the required product. Since the media and product are separated mechanically, the stirrer tip speed can be as high as the mechanical design of the mill, and can allow media size as small as 0.5mm. The high stirrer speed and small media size have improved enormously the efficiency of grinding to very fine sizes (He *et al.*, 2006).

One of the fundamental differences between vertical and horizontal stirred mills is in the way grinding media and the ground product is separated (Weller & Gao 1999). In general, vertical stirred mills rely on a settlement zone at the top to separate the media from the slurry product. To prevent the media from overflowing, the highest stirrer tip speed is limited to about 3m/s and the smallest media size is about 3mm (Wills & Napier-Munn, 2006).

These two limitations of the vertical stirred mills appear to confine the finest economical top product size to about 10µm. The restricted mill speed for the vertical stirred mill also means lower design-energy intensity than those used in horizontal stirred mills. Therefore, horizontal stirred mills have a higher ability to grind effectively much finer product sizes (Kwade, 1999).

Three types of stirred mills, such as Tower mill, IsaMill^{TM} and Detritor mill, are widely accepted and used in the mineral-processing industry (Gao *et al.*, 2007). The Tower mill and Detritor mills are vertical stirred mills that use spirals and pins as stirrers, respectively. The IsaMill^{TM}, on the other hand, is a horizontal stirred mill with discs as stirrers.

2.4.2 Operating principles for stirred mills

Stirred mills are usually operated wet because of the lower power consumption and higher throughput achieved in wet grinding, compared with those achieved in dry grinding (Hassibi *et al.*, 1999). Material transportation and handling is also easier using pumps and sumps. The product particles are reduced in size between the loose grinding media. The media is usually composed of locally available sands, smelter slag, glass, steel or ceramic material. The latest trend has been the use of high quality ceramic media for improved grinding efficiency (Curry & Clermont, 2006, Rule *et al.*, 2008). Using inert media in stirred mills apparently reduces particle surface degradation leading to improved performance of downstream flotation processes.

According to Yue (1983), two conditions are necessary in order to break a sufficient quantity of feed particles in a certain time. Firstly, the number of grinding media collisions per unit time referred to as stress events must occur at high rates to increase the frequency of breakage. Secondly, the intensity at which these stress events occur must be high enough to break the coarsest feed particles. The most important stress mechanism is determined by calculating how often each of the possible stress events appears in the mill, and by determining the stress intensity (Kwade, 1999).

The grinding mechanism in stirred mills depends on the material properties of the product, such as particle strength and size, and on the orientation and intensity of the forces exerted on the particles. The forces result from the movement of the grinding media in the grinding chamber (Hennart *et al.*, 2009). In the IsaMill^{TM}, the mill chamber is filled with the desired grinding media, and the area between each disc is essentially an individual grinding chamber, making the mill effectively 8 grinding chambers in series thereby, preventing the short-circuiting of mill feed to the discharge end.

There are three main grinding mechanisms that have been identified in grinding devices. These mechanisms are attrition, abrasion and impact (Gao & Forssberg, 1994). Attrition of particles occurs when intense stresses are slowly applied on a particle (compression). This produces fragments of sizes 50–80 volume percent smaller than the initial particles. Abrasion occurs when the stress is applied on particles along their tangential axis (shear). Particle breakage here gives a bimodal particle size distribution of fine particles that are released from the surface of the initial particle with a size close to that of the initial particles. Attrition and abrasion of particles are the results of compression forces and shear in the centrifuged packed bed of grinding media and are the main grinding mechanisms in fine grinding. Impact occurs by rapidly applying intense stresses. The particle size distribution here will range between 20 and 70 volume percent of the size of the initial particles (Redner, 1990). According to Hennart *et al.,* (2009), the dominating grinding mechanism in grinding mills under given conditions will depend on the size of the feed particles. These researchers conducted tests on three feed sizes: coarse (15μm), intermediate (0.8μm) and fine (0.15μm) to investigate the nature of the grinding mechanisms in a Dynomill (horizontal stirred mill). The following grinding mechanisms were found to dominate for the three feed particle sizes: impact and attrition for coarse particles, attrition and abrasion for intermediate particles and attrition for fine particles. Hennart *et al.,* (2009) indicated that impact by massive fracture becomes an important grinding mechanism for breaking down coarser particles. However, the particle sizes (15μm) referred to in the work by Hennart *et al.,* (2009), according to conventional mineral processing applications can not be classified as coarse particles (Wills & Napier-Munn, 2006; He *et al.,* 2006). At these particle sizes, grinding mechanisms can not easily be discerned and viscosity effects become dominant and should have been considered.

High speed stirred media mills operate with high stirrer speeds of up to 21 m/s, leading to high energy intensity in the mill. The grinding media used are also much smaller than those used in tumbling mills (Burford, & Clark. 2007; Kwade, 1999). The grinding surface area is, therefore, higher in stirred mills than it is in conventional ball mills given the same total media volume. Usually, 60% to 80% of the free grinding chamber volume is filled with the bulk of the grinding media.

The grinding media in stirred mills are agitated by the rotating stirrer, so that centrifugal accelerations of more than 50 times the acceleration due to gravity can be attained (Kwade, 1999). According to Kwade (1999), the specific energy consumption of stirred media mills for producing very fine particles is lower than that for tumbling mills. This could probably be because of the high number of stress events per unit time and unit volume, and the appropriate stress intensity that is thus achieved in stirred mills.

2.4.3 Energy consumption for comminution processes

The power used in stirred mill operations is influenced by several operational and design variables. The major variables affecting energy input are stirrer tip speed, media and slurry properties (Gao *et al.,* 1996; Jankovic, 2003; Weller & Gao 1999). A control of these variables will determine the mill's grinding and energy efficiency. The design and orientation of the chamber and stirrer will also affect the energy input to the mill (Gao *et al.,* 1996).

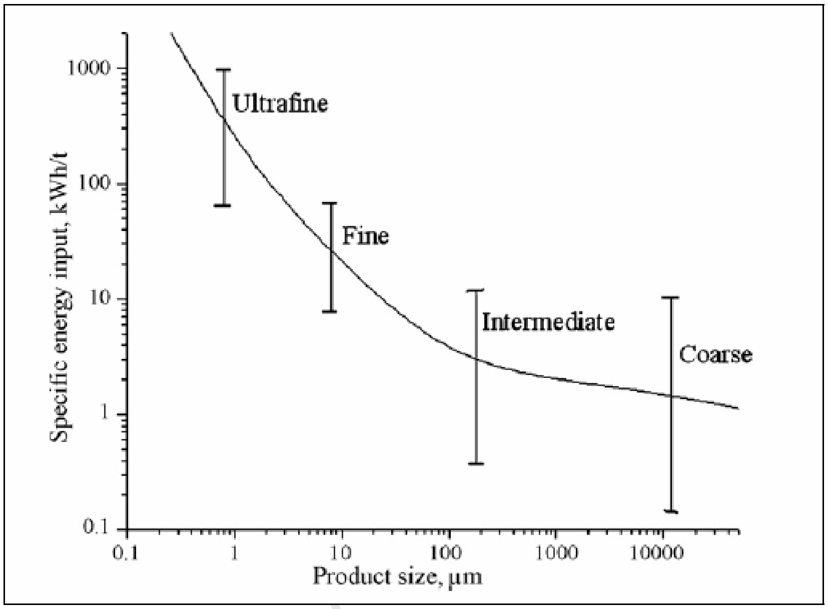

Figure 2-3: Required energy for size reduction in comminution (Wang & Forssberg, 2007)

Figure 2-3 shows the observed specific energies for various product size requirements (Wang & Forssberg, 2007). It can be observed that the relationship between specific energy and product fineness is not linear; the trend indicates that a reduction in required product size leads to an exponential increase in the specific energy consumption. The increase in energy consumption may be attributed to the fact that larger particles have a greater chance of being captured and broken than the smaller ones.

Therefore, smaller particles require more grinding events where breakage can occur. Larger particles are also alleged to have many more flaws that are broken down during breakage into smaller fragments. A reduced number of flaws in smaller particles lead to less chances of particle breakage (Wang & Forssberg, 2007; Curry *et al.,* 2005; Donovan, 2003; Weller *et al.,* 1999). Thus, an increase in energy input is necessary to raise the number of events whereby the smaller particles may be broken. This is achieved by increasing the stirrer speed and/or using smaller media in order to increase the grinding surfaces. Similar trends have been observed by other researchers (Jankovic, 2003; Pease *et al.*, 2006). The profile of specific energy and product-size relationship, according to Wang and Forssberg (2007) is dependent on the design of the mill and the macroscopic conditions that operate relevant to the individual particle properties. The relationship between specific energy and product size is very vital in the design and evaluation of stirred mills. This plot shown in Figure 2-4 is referred to as a signature plot, is very important in assessing the performance of the mill under different grinding conditions (Lichter & Davey, 2002; Weller & Gao, 1999). The specific energy and product size relationship remains constant during the process of mill scale-up and is unique to pulp conditions and the media selected (Curry *et al.*, 2005; Weller *et al.*, 1999). In this project, the signature plot will be used in assess the performances of the IsaMill™ at different operating conditions.

Although it is a common practice in the minerals industry to define the product fineness by the particle size at which 80 percent of the particle mass is smaller (P_{80}), it does not give a true representation of the mill product size distribution. Therefore, in some other grinding applications such as the paint and pharmaceuticals industry, the product fineness is defined with tight specifications such as maintaining a constant ratio of P_{98}/P_{80} (Curry *et al.*, 2005). Such stringent restrictions, however, are not usually necessary in mineral processing applications and often not achievable when used as a target measure of performance for wet grinding. Therefore, many operators in the mineral-processing industry have continued to make use of the P_{80} to characterize particle size in performance descriptions. Even though full particle-size distributions for each test are presented for the work conducted in this study, F_{80} and P_{80} will be used to characterize feed and product size, respectively.

Several studies comparing the energy consumption required to grind material to the desired product size in ball mills and stirred mills have shown that, while the energy required to grind finer increases in both cases as indicated in Figure 2-4, stirred mills appear to be more efficient in fine grinding than ball mills (Kwade, 1999; Jankovic, 2003; Pease *et al.*, 2006; Wang & Forssberg, 2007; Shi *et al.*, 2009). It has been reported by Pease *et al.*, (2006) and Jankovic (2003) that grinding to product sizes of less than 75µm using a ball mill becomes uneconomical, as the energy required for grinding to such fineness rises exponentially. This is because the ball mill uses large media (usually not less than 12mm) and has a restricted mill speed, as centrifugation occurs at speeds above critical levels (Wills, 1997). Stirred mills, on the other hand, have the ability to effectively use small media of up to 1mm size and can operate at very high speeds of up to 21m/s (Burford & Clark, 2007). The small media size significantly increases the grinding surface area and the number of grinding events required to efficiently grind fine particles. The motion in stirred mills is imparted to the charge by the movement of an internal stirrer, while the shell remains stationary. This differs from the tumbling mills, where the motion is transmitted to the charge via the rotation of the mill shell. The use of stirrers leads to higher power intensity and better energy utilization for the IsaMill^{TM}.

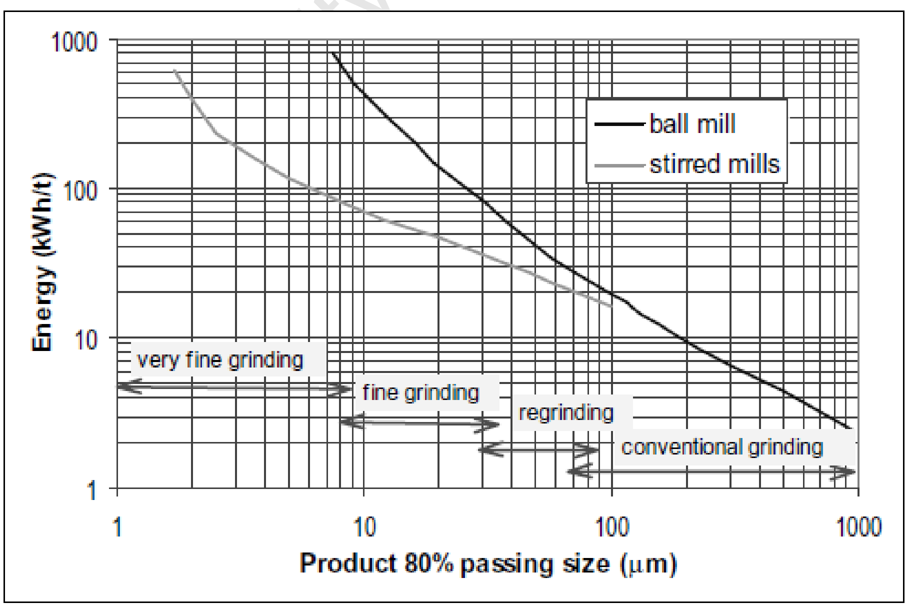

Figure 2-4: Schematic of the energy consumption at different grinding stages for balls mill and stirred mills (Jankovic, 2003)

Ball mills, Tower mills and IsaMill™ are currently the most prominent grinding devices used for grinding in the minerals industry (Wills & Napier-Munn, 2006; Gao *et al.,* 2007). A summary of typical Ball mill, Tower mill and IsaMill™ energy intensities and their grinding surface area, reported by Pease *et al.,* (2006) is given in Table 2-1. It is indicated that the IsaMill™ energy intensity and grinding surface area are much higher than the other alternatives, thereby making the IsaMill™ more efficient than both the tower mill and ball mill (Pease *et al.,* 2006). However, this comparison has a bias in that the authors are comparing power intensities and number of grinding surfaces of the different equipments when using different media sizes. No mention is made on what would happen if, for example, similar media sizes were used for Tower Mill and IsaMill™. The media size used for the IsaMill™ (1mm) in this comparison also appears to be much smaller than what is conventionally used for the IsaMill™ operations (3.5mm) thereby inflating the grinding surface area. However, studies conducted by Lichter and Davey (2002), indicated that the Tower mill can operate just as efficiently as high energy intensity stirred mills like the IsaMill™. The key to efficient mill operations for stirred mills should be the correct media selection for a given particle size (Lichter & Davey, 2002).

Table 2-1: Mill comparison of power intensity and grinding surface area (After Pease et al., 2005)

	Power Intensity (kW/m^3)	Media Size (mm)	No. Balls/m^3	Media Surface area (m^3)
Ball Mill	20	20	95, 500	120
Tower Mill	40	12	440, 000	200
IsaMill™	280	1	1, 150, 000, 000	3600

Since the conventional tumbling mills appear to be uneconomical and inefficient when grinding to product sizes finer than 80 percent passing 75µm, stirred mills have been introduced in circuits where finer products are desired (Pease, 2007; Curry *et al.,* 2005). The minerals industry has since capitalized on the relative successes gained in fine grinding, and introduced stirred mills in mainstream grinding applications to transfer some of the benefits achieved in fine grinding (Burford & Clark, 2007, Shi *et al.,* 2009). However, very little work has been conducted to compare the performance of the Tower mill, IsaMill^{TM} and ball mill operating in mainstream grinding applications. The work in this study is aimed at investigating the possibilities of using the IsaMill^{TM} to treat primary ball mill products (P_{80} = 250 μm) an application which is typically designed for the secondary ball mill in a conventional grinding circuit.

2.5 Factors affecting comminution processes in stirred mills

Comminution processes are affected by a number of operating variables; and the control of these variables will determine the grinding and energy efficiency of the mill (Burford & Clark, 2007; Gao *et al.,* 2007; Yue & Klein, 2004; Jankovic, 2003; Wang & Forssberg, 2000; Varinot *et al.,* 1999; Lofthouse & Johns, 1999). This section discusses operating variables that have been reported to have a significant influence on specific energy consumption and product fineness.

2.5.1 Mill design (Kwade 1999)

Various stirred-mill designs of different chamber and stirrer geometry, as well as different separation devices, exist. Concerning the chamber and stirrer geometry, three types of stirred media mills can be distinguished.

- The stirred media mill with disc stirrer. The discs can be provided with holes, slots and slits. These are fixed eccentrically, so that by displacement forces, additional energy can be transferred from the stirrer to the grinding media/product mixture.
- Stirred media mill with pin counter and pin stirrer. The pin stirrers move the grinding media/product mixture mainly by displacement forces. Counter pins can be used to increase the power density.

- Stirred media mill with annular gap geometry. The width of the annular gap is usually small, up to 4 to 10 times the grinding media diameter (Kwade, 1999). Some mills consist of smooth rotor and chamber walls, while others are equipped with pins to increase the energy density.

The highest power density, according to Kwade (1999), can be obtained in the grinding chamber of an annular gap mill.

The separator device in stirred media mills can be a screen or a rotating gap. Contact of the media with the screen can mostly be avoided by using the centrifugal forces inside the mill. The separator device allows free discharge of the product, but prevents the grinding media from leaving the mill.

2.5.2 Stirrer speed

The stirrer speed has been shown to have a significant effect on the operations of the stirred mill. Higher stirrer speeds tend to increase both the energy input and fineness of grind, but the energy efficiency of the stirred mill declines (Weller & Gao, 1999; Jankovic, 2003; Zheng *et al.*, 1996).

The reduced energy efficiency could be attributed to the increased energy consumption with increasing stirrer speed. For both vertical and horizontal stirred mills, a higher stirrer speed appears to offer the best processing conditions (Weller & Gao, 1999). Studies conducted by Van der Westhuizen *et al.*, (2010) using Positron Emission Particle Tracking (PEPT) to track tracer particles inside the IsaMill™ also indicated an increase in media velocity with higher stirrer speed. This is in agreement with the results obtained by Zheng *et al.*, (1996) who using a vertical stirred mill with pin stirrers to grind limestone, indicated that an increase in stirrer speed (speed ranging from 400rpm to 1600rpm) will lead to increased product fineness and the energy consumption will also increase, but the energy efficiency decreases, as illustrated in Figure 2-5. These results are in contradiction with those obtained by Becker *et al.*, (2001) which indicated a continual deterioration of fineness of grind as the stirrer tip speed was increased from 6.2m/s to 12.8m/s during the comminution of limestone.

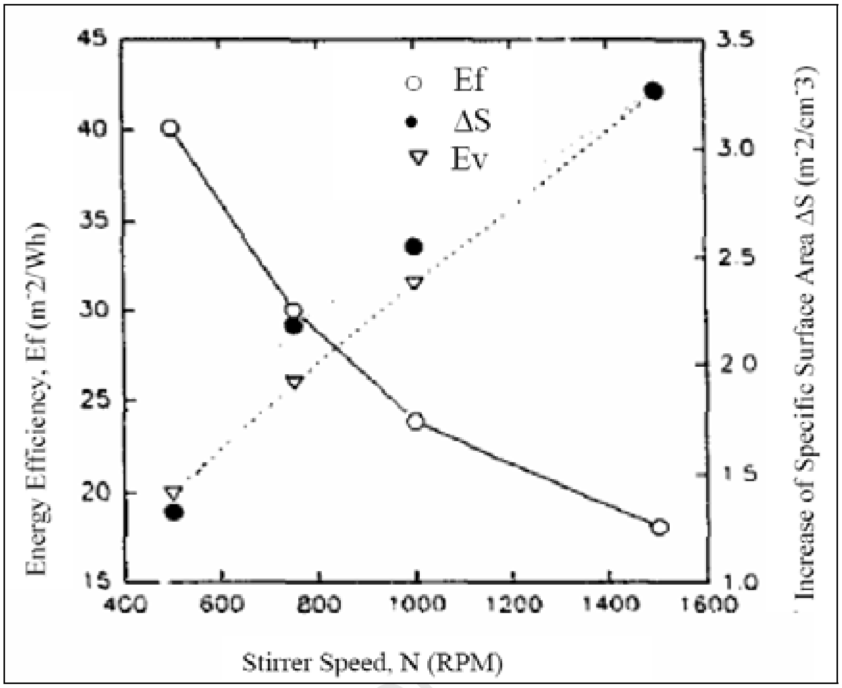

Figure 2-5: Effect of stirrer speed on energy efficiency and product surface area (After Zheng *et al.*, 1995)

The centrifugal forces of a rotating object are proportional to its rotation speed, as expressed in Equation 2-10.

$$F = Rm\omega^2$$ **Equation 2-10**

Where:

R is the rotating radius

m is the object mass

ω is its rotating speed

Equation 2-10 indicates that in a stirred mill, for any given mill dimensions and media size, the intensity of the milling forces is proportional to the square of the stirrer rotation speed. Hence, at higher mill speeds, the grinding time for producing a required product can be reduced significantly thereby increasing the grinding efficiency.

Comparison of the results obtained by Zheng *et al.,* (1996) and those of Becker *et al.,* (2001) suggests the existence of an optimum speed required to produce a desired product. At speeds lower than optimum, the grinding media is not fully energised to break the particles while centrifugation of mill charge may occur at higher speeds. This is in line with results obtained by Yang *et al.,* (2006) using Discrete Element Method (DEM) simulations in an IsaMill^{TM} which suggested the existence of an optimum speed beyond which the energy efficiencies deteriorates. This optimum velocity according to Yang *et al.,* (2006) may vary with flow and operational conditions. Therefore, an optimum stirrer speed must be established for specific stirred mills applications in order to optimize mill energy utilization. For vertical mills with open-top discharge arrangements, media may overflow from the top of the stirrer if the speed is too high. The wear rate and the packing characteristics of the media will restrict the stirrer speed in horizontal stirred mills (Gao *et al.,* 2007; Weller & Gao 1999).

2.5.3 Media Type

Stirred mills can use a range of media types; low cost locally available media, such as sand, smelter slag and ceramic have been used providing good performance at acceptable energy efficiency. Pease *et al.,* (2006) and Burford and Clark, (2007) reported the use of locally available media such as sand and smelter slag in the IsaMill^{TM} with good mill performance and energy efficiency. While the IsaMill^{TM} has been run with low-cost media the low quality of this media, however, limits the energy efficiency of the mill and also the size of the feed that can be milled.

The need for improved energy efficiency for many stirred mill installations has led to the introduction of high-quality ceramic media to replace the local sand and slag media. Key characteristics of an ideal media include hardness, high sphericity, high roundness, mechanical integrity, specific gravity, definite initial charge particle-size distribution and top-up size and chemical composition. Keramax MT1 developed by Magotteaux appear to have most of the characteristics as described above and Curry *et al.*, (2005), reported a consistent Keramax MT1 hardness. The properties of Keramax MT1 are presented in Table 2-2.

Table 2-2: Keramax MT 1 properties (After Burford & Clark, 2007)

Keramax MT1	Properties
Composition	79% Al_2O_3 6.5% SiO_2 14.0% ZrO_2
Hardness	1300-1400 HV
Fracture Toughness	5-6
Specific Gravity	3.7
Bulk Density	2.3-2.4

2.5.4 Media size

Media selection is dependent on the desired product fineness and the feed size. For a given feed size, mineral type and specific energy consumption, the media size plays a very important role in the comminution process (Kwade, 1999). Generally, a finer product requires smaller media, while a coarse feed size would need a larger media size than the fine feed size. Reduced media size would produce greater product fineness, and the tendency continues until the media size becomes too small to cause particle breakage effectively (Jankovic, 2003; Zheng *et al.*, 1996).

Results obtained by Weller and Gao (1999), Jankovic (2003) and Zheng *et al.*, (1996) indicate that the rates at which specific energy consumption changed with product fineness varies with media size, in such a way that there is a product-size range for which each of the media sizes becomes the most efficient. Therefore, determining the best media size for the required product fineness is an important step in optimizing stirred-mill performance.

The energy required for grinding is transferred through the media to the particles. Jankovic (2003) showed that two conditions need to be satisfied for breaking particles in a grinding mill; the media must exert sufficient stress intensity to the particles and there must be direct contact between the media and the particles. He concluded that at a fixed stirrer speed, the transfer of momentum from the stirrer to the bulk of the media charge reduces as the media size decreases.

Consequently, if the media size is too small, the mill produces a wide product size distribution containing a portion of unbroken feed material (Gao *et al.*, 2007). Therefore, selection of the optimum media size for a particular stirrer speed is critical for stirred milling efficiency. Correct selection of media size for a particular stirred mill application would greatly reduce the energy consumption and also provide a higher mill throughput for the installed power (Wang & Forssberg, 2007; Gao *et al.*, 2007). According to Peukert (2004) stirred mills behave like fluidized beds, creating voids between the media particles. The feed particles tend to fill these voids in a highly agitated grinding environment. Different feed-particle sizes will fill the voids between the media differently; hence, it is necessary to match media size with the feed size distribution for optimum mill performance.

2.5.5 Media load

Media load in stirred mills is one of the most important variables in terms of energy efficiency (He *et al.*, 2006). Increasing the media load in horizontal stirred mills increases the flow velocity and compressive forces on particles (Yang *et al.*, 2006). According to Yang *et al.*, (2006) and Weller & Gao (1999) higher media loads would, therefore, tend to give better energy efficiencies and finer product sizes at the same specific energy consumption. Studies conducted by Weller & Gao (1999), and illustrated in Figure 2-6, indicate that increasing the media load in horizontal stirred mills not only increases the mill energy efficiency but also the capacity of the mill. This suggests that a horizontal mill needs to be operated with the maximum allowable media load for optimum energy efficiencies. Operating the horizontal stirred mill with less media reduces the interaction of media with stirrer thereby decreasing the efficiency of the grinding process. (Weller & Gao, 1999; Yang *et al.,* 2006). The maximum media load according to Gao *et al.,* (2007) can be determined by the onset of media compression that can cause severe destruction of mill liners and stirrers.

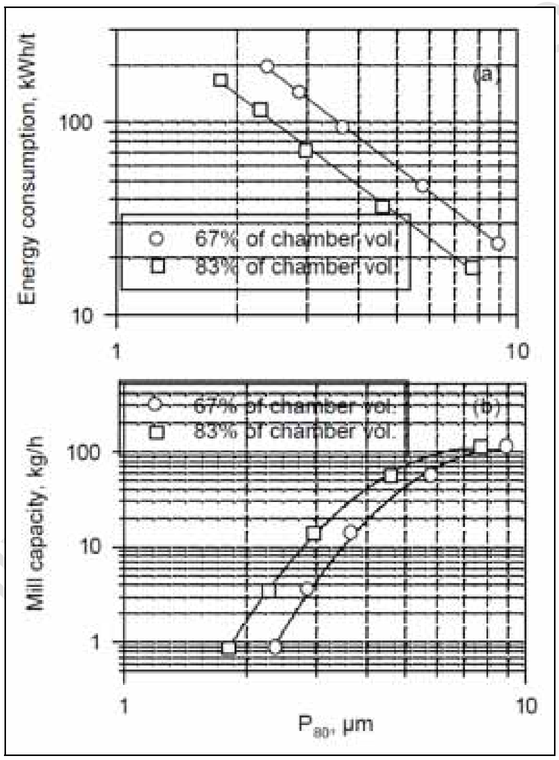

Figure 2-6: Effect of media load on energy consumption and mill capacity of a horizontal stirred mill (After Weller & Gao, 1999)

He *et al.,* (2006) also in their study of effect of media load on mill efficiency concluded that higher media loads will continue to improve the efficiency of grinding. However, the observed improvements in efficiencies with increased media load in horizontal stirred mills have a limit as observed by other researchers (Gao *et al.,* 2007; Van der westhuizen *et al.,* 2010). Van der westhuizen *et al.,* (2010) suggested the existence of an optimum media load for different stirred mill applications starting from media loads of 60%. The optimum media load is highly influenced by the media properties such as size and density of media (Van der Westhuizen *et al.,* 2010; Gao *et al.,* 2007).

For vertical stirred mills, Weller & Gao (1999) suggested that provided a minimum number of stirrers are covered by media adding more media increases the effective grinding volume of the mill, but has no effect on the efficiency of grinding within the volume. The media load limitation for a vertical mill is fixed largely by the minimum height of the media separation zone at the top of the mill, and/or the position of the top-most stirrer relative to the product outlet (Weller & Gao, 1999). Researchers have also observed that the mill load significantly influences the mill power draw (He *et al.,* 2006; Gao *et al.,* 2007; Weller & Gao, 1999). The linear relationship observed between mill power draw and media load is often used at industrial scale to estimate the media load volume and to maintain the power set point in the mill (Gao *et al.,* 2007; Wills & Napier-Munn, 2006).

The IsaMill^{TM} can be operated at low power by reducing the media load in the grinding chamber. Decreasing the power by reducing media load is necessary especially in single mill-grinding lines. This is done to avoid over-grinding and over-heating during periods of low-feed tonnage. According to Curry *et al.* (2005), this method of power control is common to all IsaMill^{TM} and is important to operations where the IsaMill^{TM} feed fluctuates during normal operations. An example of such an operation is the Western Limb Tailings Re-treatment Plant (WLTRP) where the rougher flotation mass pull fluctuates with head grade. Figure 2-7 shows the M10, 000 at WLTRP operating at similar energy inputs, but greatly different power draws. Despite the extreme operating conditions, the IsaMill^{TM} produces the same P_{80} in both cases (Curry *et al.,* 2005).

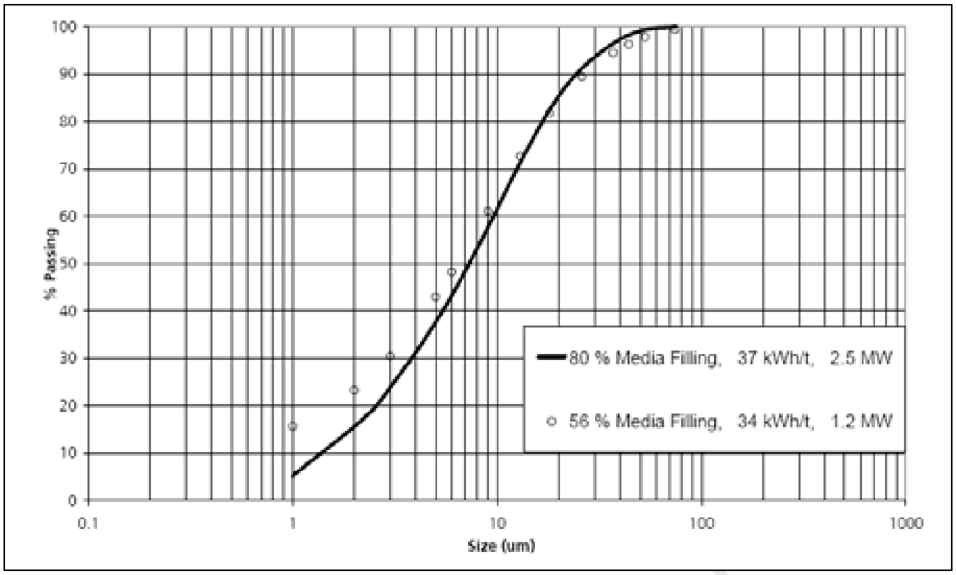

Figure 2-7: M10, 000 turn-down capability (After Curry *et al.* 2005)

2.5.6 Solids concentration

The solids concentration is an important factor in wet-grinding operations -- due to its direct influence on the product fineness and the specific energy consumption (Gao *et al.*, 2007; He *et al.*, 2006; Jankovic, 2003; Zheng *et al.*, 1996). Increases in solids concentration tend to increase the specific energy consumption required for a comminution process. The product fineness also appears to increase with increased solids concentration, but the trend is only true up to a certain limiting high solids concentration -- beyond which point, the product fineness decreases (He *et al.*, 2006; Jankovic, 2003; Zheng *et al.*, 1996).

Solids concentration is also reported to have influence on the slurry viscosity which in turn influences the movement of the grinding media and therefore the stress intensity (Kwade, 1999; Napier-Munn *et al.*, 1999). Viscosity has a very non linear (exponential) dependence on the solids concentration of the slurry as indicated in Figure 2-8.

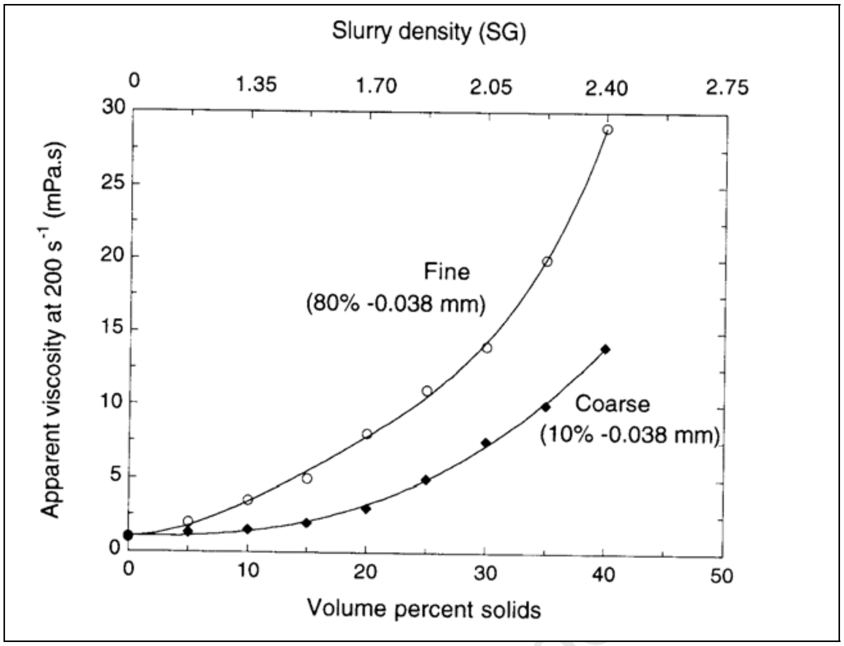

Figure 2-8: Typical relationship between viscosity and solids concentration of mineral ore after grinding (After Napier-Munn, 1999)

Slurry viscosity can be defined as the resistance of particles to flow and according to Napier-Munn *et al.*, (1999) viscosity controls the mill hold-up and affects mill power draw. For low solids concentration, (usually below 40% solids) there is large inter-particle distance making it difficult for the grinding media to capture effectively the particles (Gao *et al.*, 2007; Napier-Munn *et al.*, 1999). This increases the possibility of direct collision between the media, resulting in higher energy loss. A higher solids concentration, on the other hand, gives a smaller average inter-particle distance leading to a larger than average number of stress events of each particle (He *et al.*, 2006; Jankovic, 2003). However, if the solids concentration is too high, the changes in mill rheology and larger viscosity dampens the motion of the grinding media in the mill and significantly reduces the velocity and kinetic energy of the media (He *et al.*, 2006; Tangsathitkulchai, 2003). This brings about lower stress intensities of collisions on media/particle/chamber wall caused by the coating of slurry, media and mill chamber. Therefore, the captured particles cannot be ground effectively and brings about an ineffective grinding operation.

It can be concluded that an increase in the solids concentration will improve mill performance up to a certain point after which a decrease in mill efficiency should be expected, together with slurry flow-problems due to increased viscosity (He *et al.,* 2006; Jankovic, 2003; Zheng *et al.,* 1996). Therefore, controlling the solids concentration at the optimum is very important in order to improve grinding efficiency of the milling process.

2.5.7 Flowrate

The flowrate of slurry in horizontal stirred mills have been reported to affect mills performance (Lane, 1999; Weller *et al.*, 1999). According to Weller et al., (1999) the energy consumption for horizontal stirred mills tend to reduce with increased flowrate. The reduced energy consumption at high flowrates can be attributed to the increased mill pressure which leads to decrease in slurry residence time distribution. However, if the flowrate is too high, the back-mixing of the media becomes ineffectual and hydraulic packing develops, where the media migrates to the discharge end, giving rise to accelerated wear of media and stirrer (Lane, 1999; Weller et al., 1999).

Weller and Gao (1999) conducted tests to investigate the influence of flowrate on the operations of Netzsch mill. It was observed that the energy consumption reduced, while product fineness improved with increased feed flowrate. The pressure in the mill chamber was also seen to rise with increasing flowrate; and this change in pressure according to Weller and Gao (1999), could have led to an increased rate of particle breakage. The findings of Weller and Gao (1999) suggested that the changes in flowrate in horizontal stirred mills affects the mill operation. The flowrate in horizontal stirred mills should be optimized because, once the flowrate exceeds a certain maximum point, the flow characteristics within the mill would begin to change, and the media tend to pack towards the discharge end. Such high flowrates can cause severe localized wear of the media and stirrer, and should, therefore, be avoided where possible (Lane, 1999; Weller et al., 1999).

2.6 Summary of literature review

A review of the literature has indicated that stirred mills are being increasingly applied to the grinding of minerals. However, since stirred mills have only recently been introduced to the mineral-processing industry, the understanding of their application in mineral processing has not been developed to the same degree as it has for tumbling mills. A lot of research, therefore, needs to be done in characterizing stirred milling in the mineral-processing industry.

The abundance of modeling work conducted in stirred mills to characterize breakage is not conclusive enough to apply in general models. The population balance developed for tumbling mills has been employed for stirred mills, with varied results being obtained by different researchers. It is assumed that population balance works better for coarser feed sizes and is not ideal when applied in fine grinding. The use of the population-balance model has shed light on the study of breakage characteristics in stirred mills, and there is no doubt that numerous benefits can be derived by using this model. However, it is not clear as to what particle size range the population model can be applied.

The energy consumption in conventional comminution devices increases exponentially with the reduction of product size, making them uneconomical to use in fine grinding. This has led to the development of specialized fine grinding mills which have better grinding efficiencies than conventional tumbling mills in the low size range (Kwade, 1999). These specialized mills include stirred mills and other alternative mills, such as the Tower mill, the Pin mill, Detritors, the IsaMill^{TM} and High Pressure Grinding Rolls (HPGR) (Napier-Munn *et al.*, 1999).

Stirred mills have been found to be more efficient in fine grinding when compared with ball mills; and they also appear to perform well in mainstream grinding. But it is not clear whether coarse-grinding stirred mills are preferable to ball mills. It has been established that many operating variables affect stirred mill performance, though their level of influence is highly dependent on mill type and ore properties. In this project, the influence of a number of operation variables on the IsaMill™ performance has been investigated.

Operation and design variable interactions have been said to influence mill operations. However, not so much work has gone into establishing the associations of different variables on mill performance.

The nature and intensity of the applied stresses on the particles affect the particle-size reduction process and are influenced by the stirrer speed and media size (Pease *et al.,* 2006). Therefore, media size and stirrer speed are among the most important variables in a stirred mill and these must be fully optimized for efficient mill operation.

3.0 Methodology

This chapter presents a description of the experimental methodology employed to study the performance of the laboratory (M4) and the industrial scale (M10000) IsaMills under different operating conditions. The effect of several key variables was evaluated. The surveys for the industrial scale IsaMill (M10000) were performed at Waterval Concentrator and Western Limb Tailings Re-treatment Plant. The experimental methods and procedures used to collect the data for this study is described in this chapter.

3.1 Experimental apparatus and methodology

The IsaMill^{TM} Technology was introduced in 1990 by Mount Isa Mines limited and Netzch-Feinmahltechnik GmbH to treat the lead/zinc deposit at Mount Isa and McArthur River which are both in Australia (*Pease et al.,* 2006*; Gao et al.,* 2002). The McArthur River deposit was discovered in 1955, but could not be treated because there was no economically viable method for the treatment of the fine-grained deposit to produce saleable lead/zinc concentrate.

On the other hand, the plant performance at Mount Isa started to deteriorate in the mid-1980s due to increased amounts of refractory pyrite in the ore and the decreasing liberation size. Studies conducted at Mount Isa and reported by Burford & Clark, (2007) have shown that conventional ball-and-tower mill technology became uneconomical to grind the ore finer in order to liberate the valuable minerals. This was due to high power consumption required to achieve the desired fine liberation sizes; and also because of the high consumption rate of steel media and contamination of the mineral surfaces with iron resulting in poor flotation responses.

Observations made by Jankovic (2003) and Harbort *et al.,* (1999) also indicate that conventional Ball mills and Tower mills are not economical in fine grinding. A series of test work to investigate efficient grinding to higher product fineness led to the development of the IsaMill^{TM} Technology. The first industrial scale installation was an

M300 IsaMill™ operating with a 1.12MW motor at the Mount Isa mines lead/zinc concentrator in 1994. The IsaMill™ was later commercialised in 1999. The IsaMill™ has since been installed in various mineral-processing operations around the world and it is now being applied in main-stream grinding applications (Pease, 2007).

3.1.1 General features

The IsaMill™ has a set of discs horizontally mounted on a cantilevered shaft; a schematic is shown in Figure 3-1.

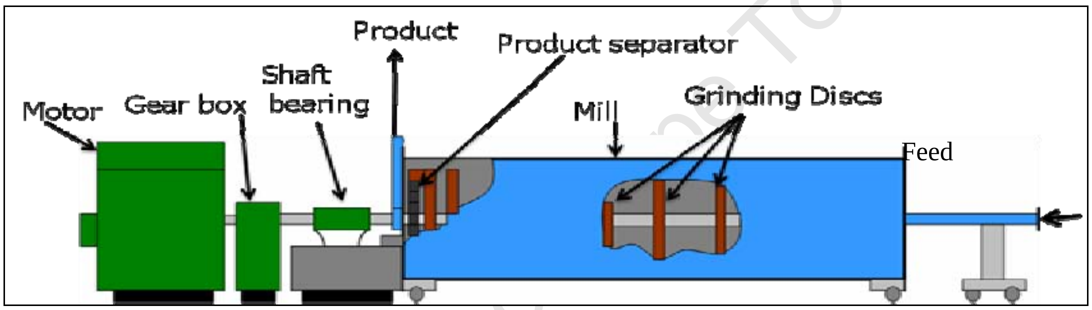

Figure 3-1: IsaMill™ schematic diagram

The discs rotate at high tip speeds of up to 21 – 23m/s resulting in high-energy intensities of up to 300kW/m^3. The high rotational speed is critical for the effective use of small grinding media. According to Burford and Clark (2007), the energy intensity of the IsaMill™ is significantly higher than that of any other commercially available grinding equipment. The combination of high energy intensity and the high-grinding efficiency leads to a compact mill that can be fitted into existing plants where floor space is limited (Burford & Clark 2007).

It is stated that IsaMill™ high energy intensity is critical for efficient mill operations (Burford & Clark, 2007; Pease *et al.*, 2007; Shi *et al.*, 2009). However, studies conducted by Lichter and Davey (2002), indicated that the stirred mill's energy intensity does not have a strong influence on the relative performance of the mill. They observed that low-energy intensity stirred mills, such as the Tower mill can operate just as efficiently as high energy intensity stirred mills like the IsaMill™. The key to efficient mill operations for the IsaMill™ should be the correct media selection for a given particle size (Lichter & Davey, 2002).

The grinding mechanisms in IsaMill™ can be described by the stress energy distribution in the mill chamber as illustrated in Figure 3-2 (Burford & Clark, 2007; Pease *et al.,* 2006; Stender *et al.,* 2004). According to Stender *et al.* (2004) the most important comminution mechanisms occur because of media–media contacts with different tangential velocities. They showed that different stress energy regions occur in the mill chamber. This is in agreement with findings of Van der Westhuizen *et al.,* (2010) who using PEPT experiments suggested that the highest velocities occur to regions close to the disc stirrers while significantly lower velocities and, therefore, smaller stress energies occur in a much larger part of the grinding chamber volume. The rotating discs set the media in motion and accelerate it towards the shell. Between the discs where the media is not prone to the high outward acceleration of the disc face, the media is forced back in towards the shaft, creating a circulation of the media between each set of discs (Burford & Clark, 2007; Pease *et al.,* 2006).

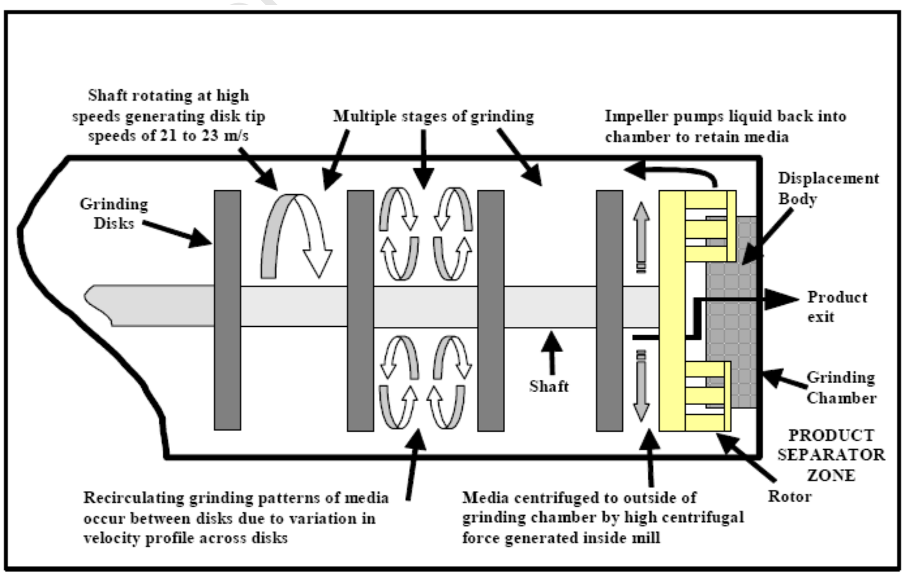

Figure 3-2: IsaMill™ grinding mechanism

3.2 M4 IsaMill^{TM} testwork

3.2.1 Ore type used

The UG2 Ore from Anglo Platinum mines was used for all the tests performed in this project. The UG2 ore is a Platinum Group Metal (PGM) bearing ore, mined in the Bushveld igneous complex in the Northern West and Northern provinces of South Africa. The PGMs are typically associated with the base-metal sulphides, such as chalcopyrite ($CuFeS_2$), pyrrhotite ($Fe_{1-x}S$), pentlandite ($(FeNi)_9S_8$) and pyrite (FeS_2) (Lee, 1996). Chromite is usually a predominant gangue mineral in UG2 ores.

Due to the fineness of the PGMs (< 25μm), extensive grinding is required to liberate the PGMs (Becker *et al.,* 2008; Rule *et al.,* 2008). The UG2 ore was selected because most of the industrial scale IsaMill^{TM} applications in the platinum industry are operations treating this ore. The IsaMill^{TM} is important in the treatment of the UG2 ore because this ore needs to be ground to a fineness of less than 25μm to liberate the PGMs which are typically between 2 to 15μm. For such fineness, it becomes uneconomical to use the conventional tumbling mills (Jankovic, 2003; Wills & Napier-Munn, 2006; Shi *et al.,* 2009).

3.2.2 Feed preparation

The feed samples used for the M4 testwork were prepared at the Anglo Platinum's pilot plant situated at the Frank Concentrator. The circuit configuration of the pilot plant comminution flow-sheet, which was used to prepare the feed, consisted of the primary ball mill, the secondary ball mill and the M100 IsaMill^{TM} as illustrated in Figure 3-3. The primary and secondary ball mills were operated in closed circuit, with hydrocyclones, while the IsaMill^{TM} was run in open circuit.

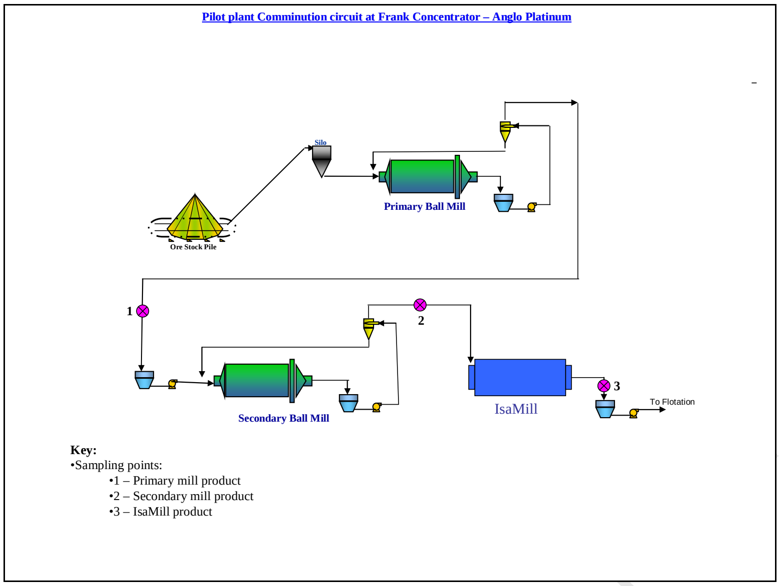

Figure 3-3: Pilot plant comminution circuit at Frank – Anglo Platinum

The particle size distributions for the range of feed sizes obtained from the pilot plant and used for the M4 tests are presented in Figure3-4.These bulk samples, termed as coarse, intermediate, and fine feed sizes, were collected from the pilot plant and used to perform the M4 experiments. The coarse-size material was collected from the product of the primary ball mill; the intermediate size was the secondary mill product; and the fine size was from the IsaMill^{TM} product. The pilot plant was stabilized before taking the three mill discharges. This was done by controlling and making sure that the flow-rate, the mill power draw and the density were running as close to a steady state as possible. Operating the pilot plant close to steady state was done to ensure uniformity in the size fractions of the bulk samples collected for use in subsequent experiments.

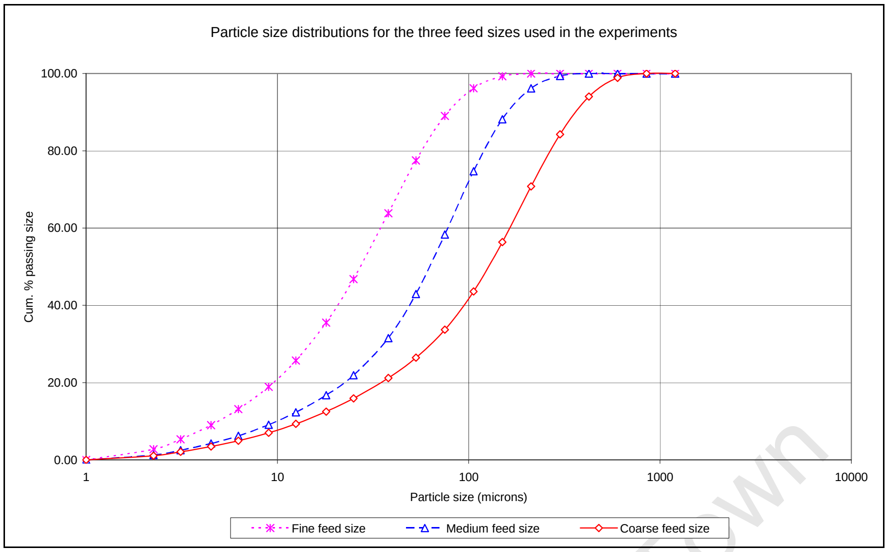

Figure3-4: Particle size distributions for the three feed size ranges used in the tests. Fine feed size F_{80} = 55µm, Medium feed size F_{80} = 120µm, Coarse feed size F_{80} = 250µm.

After obtaining stable operations, the bulk samples were collected at 30 minute intervals by directing the discharge hose of the pumps for the primary mill, the secondary mill and the IsaMill™ product into 100 litre drums. The drums full of slurry were left overnight to allow for the solids to settle. The water was then decanted from the drums the following morning. The solids were allowed to settle overnight to ensure no solids were removed with the water when decanting.

The samples were decanted to allow for ease of handling the drums during transportation from the pilot plant to Anglo Platinum's Divisional Metallurgical Laboratory (DML), where the M4 tests were performed.

3.2.3 Media type

The Magotteaux' Keramax MT1 ceramic media of sizes 1.8 – 2.2mm and 3 – 4mm, and AZ 2000 alumina media of size 4.5 – 5.5mm were used for testing the media size in this study. The AZ2000 alumina media type was used for the tests performed at the media size of 4.5 – 5.5mm because the Magotteaux Keramax MT1 ceramic media type was not available in this size. Keramax MT1 is a typical media type that was used in IsaMill™

applications at Anglo platinum's Waterval and Western Limb Tailings re-treatment plant. The chemical properties of the two media types are similar, as shown in Table 3-1.

Table 3-1: Media chemical composition

Media type	**% Al_2O_3**	**% Zr_2O_2**	**% ZrO_2**	**% SiO_2**	**% $Al_2O_3SiO_2$**
Keramax MT1	79	14	-	6.5	-
AZ 2000	60 – 75	-	5 – 15	-	10 – 20

3.2.4 Media preparation

The media were preconditioned in water before testing on the slurry. This was to simulate a conditioned charge achievable by a full scale IsaMill^{TM}. The required volume of media was weighed and charged into the mill, where it was preconditioned in water for 1 hour. The media were then dried and weighed again to determine the preconditioning mass loss. Fresh media amounting to the mass loss was then added to the conditioned media to maintain the required media load and also to introduce the media top size. All operating variables were kept at their normal operating levels during the media preconditioning. The conditioned media were then reloaded into the mill in readiness for the test.

3.2.5 M4 Laboratory scale IsaMill^{TM} apparatus

The experimental apparatus for the laboratory tests performed in this study was the M4 IsaMill^{TM} shown in Figure 3-5 and Figure 3-6. It consists of a horizontal grinding chamber containing 6 shaft-mounted grinding discs arranged for intensive grinding. The shaft is connected to a 5.5kW motor. The discharge end of the mill is fitted with an integrated product separator, which helps retain the grinding media and the coarse material in the mill, while allowing the product out of the mill. The M4 IsaMill^{TM} has an effective volume of 3.1 litres, with the rest of the volume being taken up by the mill inserts, such as the grinding discs, the product separator and the shaft. The mill is equipped with a variable speed drive and can be operated between 1400 to 2500 rpm.

Figure 3-5: Picture of the M4 Laboratory scale IsaMill^{TM}

The M4 IsaMill^{TM} used in the testwork for this study is mounted on a stand, and is equipped with digital panel in which the operating conditions and output variables, such as stirrer speed, pump speed, mill pressure, temperature and mill power can be monitored and recorded during the tests. Stirrer speed and pump speed are independent variables, which are set to their desired values at the beginning of the experiments. Power draw, pressure and temperature are dependent variables, which are used to monitor the operations of the mill; exceeding their set limits will automatically trip the mill. Figure 3-6(a) indicates the M4 IsaMill^{TM} mounted on a stand and the control panel, while Figure 3-6(b) shows the internals of the mill pointing out the grinding discs and the product separator.

Figure 3-6: Pictures showing the M4 IsaMill™ utilized in this test work

Figure 3-7 shows the experimental set-up for the M4 IsaMill™ grinding tests. This set up allowed a continuous stage-wise operation. At the beginning of each test, the conditioned media were loaded to the required volumetric filling level. The test material was then re-pulped to the required slurry density. Slurry volumes of between 100 – 120 litres were prepared for each M4 IsaMill™ test. Two samples of the feed were then collected in properly labelled bottles. Thereafter, the mill was started up and material introduced into the mill for four passes. A variable drive stirrer was used to stir the slurry continuously in the drum, in order to maintain the feed slurry in a homogeneous state for the duration of the test.

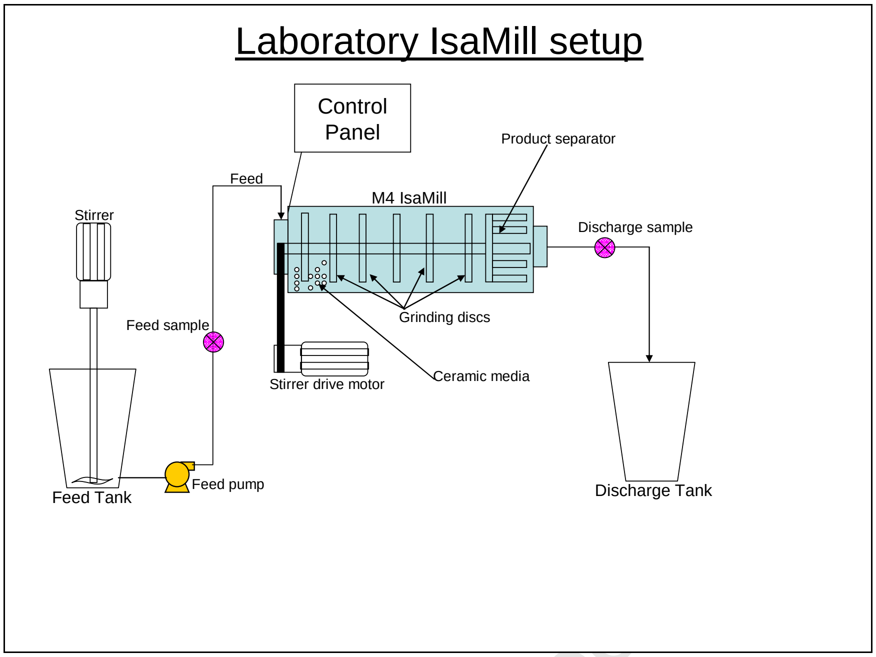

Figure 3-7: Laboratory IsaMill experimental set-up

The slurry densities were measured, using a Marcy scale. An on-site calculation was performed with the slurry densities obtained in order to obtain the approximate concentrations of the solids, based on the measured slurry density, using Equation 3-1 (Mainza, 2006). The slurry solids concentration was subsequently determined accurately by using the wet and dry mass method.

$$\%solids = \frac{\rho_{ore}\left(\left(\frac{\rho_{slurry}}{V}\right)-1\right)}{\left(\frac{\rho_{slurry}}{V}\right)(\rho_s - 1)}*100$$ **Equation 3-1**

Where:

ρ_{ore} - Ore specific gravity

ρ_{slurry} - Measured slurry density

V - Volume of the Marcy scale cup

As discussed in section 2.5, various design and operational variables of the IsaMill^{TM} influence the specific energy consumption and product fineness. The ranges of the key variables used in this study are shown in Table 3-2.

Table 3-2: Test matrix indicating the highest, intermediate and lowest level for the variables considered in the M4 IsaMill experimental work

Variable	High	Intermediate	Low
Feed size	P_{80} = 250µm	P_{80} = 150µm	P_{80} = 53µm
% Solids	60%	50%	40%
Flow-rate	3.0 l/min	2.5 l/min	2.0 l/min
Stirrer speed	2100rpm	1800rpm	1500rpm
Media size	1.8 – 2.2mm	3 – 4mm	4.5 – 5.5mm
Media load	80%	65%	50%

The standard procedure developed by Xstrata Technology for the M4 tests was adopted for this work. This procedure is outlined in the section that follows.

After preparing the feed slurry and loading the media into the mill, the pump was then started up and adjusted to the desired flow-rate. The pump speeds corresponding to different flow rates were noted during the trial runs, conducted prior to the actual tests to avoid wasting test slurry on parameter setting. Immediately after starting the pump, the mill was started up at the required speed, and the timer started to record the duration of each pass. Each test was performed for two hours, with every pass running for 30 minutes. Samples for the feed and product-particle size analysis were taken at five minute intervals in order to collect a representative sample over the period of the test. Readings of the power draw, the mill pressure, the stirrer speed, the flow rate and the temperature were also recorded every five minutes to monitor the stability of the mill operation throughout the testing period. The flow rates were measured by pumping the slurry into a measuring cylinder and recording the time taken to fill up a certain volume, while noting the pump speed.

As soon as the feed tank was empty, the mill was stopped, together with the timer. The hose pipes rearranged so that the product from the complete pass could be pumped back through the mill as feed for the next pass. The above procedure was repeated up to four passes for every set of test conditions, in order to obtain enough data points required for the signature plot and other analyses.

At the end of the fourth pass, a media wash-out was carried out. This involved adding water to the feed tank and pumping it through the mill for 10 minutes, while the mill was still running. The wash-out was done in order to wash off all the slurry material from the mill, and to avoid the mill and feed pipe line from choking. The time taken and the power consumed for the wash-out were recorded separately. At the end of the wash-out, the media were then removed from the mill, dried and screened on a 1mm screen to remove any fines and then weighed to determine the media loss during the test.

3.2.6 Operating variables

3.2.6.1 Power draw

The laboratory IsaMill power draw was constantly monitored and recorded, every five minutes during the test periods – in order to capture the changes in power draw during the test. These power readings were used to calculate the specific energy consumption for each pass required for signature plots. Table 3-3 shows typical power draw trends for the four passes constituting a test. Standard deviations were used to check the variability of the power draw for each pass during the test. Lower standard deviations for the power drawn during each pass meant better stability in mill operation.

It was observed that the standard deviations were low, and the results could be used to assess the performance of the IsaMill™. The consistency in the power draw is also an indication that the test was being performed under conditions close to steady state.

Table 3-3: Power draw trends and standard deviations for a typical M4 IsaMill test (Test 1 results)

Time (minutes)	0	5	10	15	20	25	30	Average Power	Standard Dev
Pass 1	2.9	3.2	3.2	3.2	3.1	3.1	3.1	3.11	0.11
Pass 2	3.1	2.9	2.8	2.8	2.8	2.8	2.8	2.86	0.11
Pass 3	2.8	2.7	2.7	2.7	2.7	2.7	2.7	2.71	0.04
Pass 4	2.7	2.7	2.7	2.7	2.7	2.7	2.7	2.70	0.00

For each test, the no-load power was subtracted from the gross power readings to obtain the net power draw. The net power was subsequently used in the calculations of the specific energy. To obtain the no-load power, the mill was run empty (with no media). The no load tests were performed at different stirrer speeds as it is alleged that stirrer speed affects the no load power achieved in the IsaMill™. The no-load tests that were conducted achieved the following results; 1.2kW, 1.0kW and 0.9kW for stirrer speeds 1500rpm, 1800rpm and 2100rpm respectively.

3.2.6.2 Stirrer speed

IsaMill™ stirrer speed is regarded as a very important parameter. The effect of stirrer speed was evaluated by running the IsaMill at three different speeds ranging from 1500rpm to 2100 rpm, by adjusting the knob on the control panel. Electronically, the M4 IsaMill™ was pre-set to operate at a minimum stirrer speed of 1500 rpm; thus the lowest speed for the experimental programme was set to be 1500rpm. There is, however, on the M4 IsaMill™ according to the Xstrata, an upper critical speed at which point the separator retains more solids and shows a more biased fine grind. This phenomenon begins to occur at stirrer speeds exceeding 2000 rpm for the M4 IsaMill™. Therefore, for this project 2100 rpm was chosen as the upper limiting stirrer speed.

3.2.6.3 Media load

IsaMill™ in industrial applications are typically operated with media loads of approximately 75%. The media volume tested in this project ranged between 50 and 80 percent. The volume of media required to fill up the mill to the desired level for each test was calculated from the net mill chamber volume. The net mill chamber volume was taken to be the total chamber volume less the volume of insets like disks, product separator and shaft.

3.2.6.4 Flowrate

The volumetric flow rate tested ranged from $0.138m^3/hr$ to $0.342m^3/hr$. This was achieved by operating the feed pump at speeds that would give the desired flowrate. The actual flowrate was measured by using a stop-watch and a 100 millilitre measuring cylinder. Three independent measurements were taken; at the beginning, in the middle and at the end of each test. The flowrate measurements were obtained by directing the mill discharge pipe into the measuring cylinder and filling it up to 100 milliltre. The time taken to fill up the 100 millilitre measuring cylinder was obtained from the stop watch and recorded.

3.2.7 Sample collection and processing

Samples were collected at 5 minutes intervals during the test period. Separate samples were collected for each pass during the test. A 100 millilitre bottle was used for sampling the IsaMill™ feed and discharge streams. The samples were collected by directing the slurry pipe into a sampling bottle before being transferred into 2 litre or 200 millilitre containers. For each pass, samples were collected into three containers; “A”, “B” and “C”. Sample “C” was collected in 200 millilitre bottles, and was used to obtain the particle-size distributions achieved at each stage of the testing. The Malvern laser sizer at Divisional Metallurgical Laboratory (DML) was used for this.

The "A" and "B" samples collected in 2 litre containers were weighed wet, filtered and dried. The dried samples were then weighed again. The wet and dry masses were used in the calculations of the solids' concentration. After dry weighing, the samples were de-lumped on 1mm screen; and sample "A" was stored at the University of Cape Town, while the "B" samples were left at Divisional Metallurgical Laboratory only to be used when an error was found with the results.

3.2.8 Particle size analysis

Laser diffraction techniques using the Malvern laser analyser were used to obtain the particle-size distribution. The principle of the Malvern Master Sizer system is based on the measurement of light diffraction from the particles. A low power laser transmitter produces a parallel monochromatic beam of light, which is arranged to illuminate the particles to give a stationary diffraction pattern, regardless of any particle movement.

Integration, over a suitable period, with a continuous flux of particles through the sample cell, gives the measured diffraction pattern of a representative bulk sample. A computer uses the method of non-linear least squares analysis to find the size distribution, giving the closest fitting diffraction pattern (Master Sizer manual).

Each slurry sample collected for the Malvern laser sizer particle-size distribution test was properly mixed, using a stirrer, following which a smaller sample was siphoned out using a 5 millilitre syringe. Only a couple of grams were needed to make the measurements in the Malvern laser sizer. Hence, just about 2 millilitres of slurry from the syringe were put into the Malvern machine for size analysis. To ensure that there was no bias in the particle size distributions results obtained, the Malvern laser sizer particle-size distribution tests were performed in triplicates.

3.3 M10 000 IsaMill™ testwork

This section describes the methodology followed for the sampling campaigns conducted on the industrial scale IsaMill™ at Waterval UG2 Concentrator and Western Limb Tailings re-treatment Plant (WLTRP). Both located in the Rustenburg area in South Africa. The sampling campaigns were conducted to evaluate the performance of the large-scale units. The sample processing and procedures for industrial scale IsaMill™ testwork will also be presented in this section.

3.3.1 Waterval UG2 Concentrator

The Waterval Concentrator has two IsaMill™ operating in main-stream grinding and treat the UG2 platinum ore. The Waterval UG2 Concentrator flow-sheet showing the main comminution and flotation sections is given in Figure 3-8.

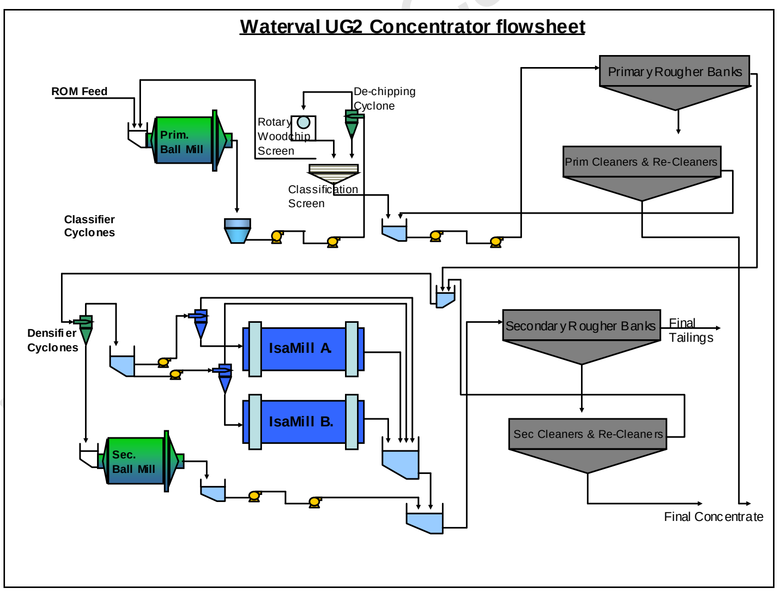

Figure 3-8: Schematic of the Waterval UG2 Concentrator

The Waterval UG2 Concentrator flow-sheet shown in Figure 3-7 consists of the RoM ball mill operating in closed circuit with screens. The de-chipping cyclone together with the wood chip screen is used to screen out all the wood chips from the circuit. The undersize material from the screens is fed to the flotation banks for platinum concentrate recovery, while the oversize is recycled back to the RoM ball mill for further grinding. The screen undersize is pumped to the primary rougher bank, from which the concentrates are floated again in the primary cleaners and re-cleaner banks to improve the grade of the platinum concentrates.

The primary rougher tailings are pumped to the secondary milling section for further grinding to liberate locked PGMs, and then sent to the secondary flotation section for further PGM recovery. Prior to sending the primary rougher tailings to the secondary milling section the material is pumped to the densifier cyclone. The densifier cyclones separate the slurry into the underflow and the overflow streams which feed the secondary ball mill, and the IsaMill^{TM}, respectively. The densifier cyclones are employed to increase the density of the slurry getting into the secondary mill from about 1350 to 2450 Kg/m^3. During the densification process, most of the chrome ends up reporting to the densifier underflow because of its higher specific gravity, while most of the silica component being lighter reports to the overflow.

Consequently, there is a significant reduction in the amount of chrome reporting to the IsaMill circuit thereby reducing the wear rates of IsaMill internals, such as mill liners, grinding discs and product separator. The products of the secondary ball mill and the IsaMills are then combined to feed the secondary rougher flotation banks. The secondary rougher concentrates are cleaned in secondary cleaners and re-cleaners, while the tailings report to the final tailings for disposal.

There are two M10 000 IsaMills operating in parallel; each mill runs with a cluster of pre-grinding hydrocyclones, which removes the fines (minus 75µm) from the IsaMill feed. The fines bypass the IsaMill^{TM} and are combined with the IsaMill^{TM} product at the mill discharge sump. The underflow from the cluster of hydrocyclones is then pumped to the IsaMill^{TM} for grinding. The IsaMill^{TM} product then combines with the overflow from the same cluster of hydrocyclones prior to being pumped to the secondary flotation section.

The secondary flotation section consists of the rougher, cleaner and re-cleaner banks. The rougher bank receives the slurry from the secondary milling section, from which the tailings are final tailings, while the concentrate reports to the secondary cleaner and re-cleaner for concentrate upgrade. The re-cleaner concentrate is the final concentrate, which is sent to the smelter, while the secondary cleaner tailings are recycled back to secondary rougher banks.

The surveys for this thesis were conducted on the streams around the IsaMill^{TM}. No variable was changed during the sampling campaigns. During the survey periods, the plant was operated as close as possible to the desired daily operating conditions such as power draw, feed flow rate, mill media load and slurry density. The desired conditions for these tags are given in Table 3-4.

Process data from the control system were obtained for the duration of the test. The data were used to assess the stability of the IsaMill^{TM} – both before and during the test. The operating conditions for the IsaMill^{TM} are shown in Figure 3-7. Notice that the operating variables were operated under sufficiently stable conditions during the whole test period.

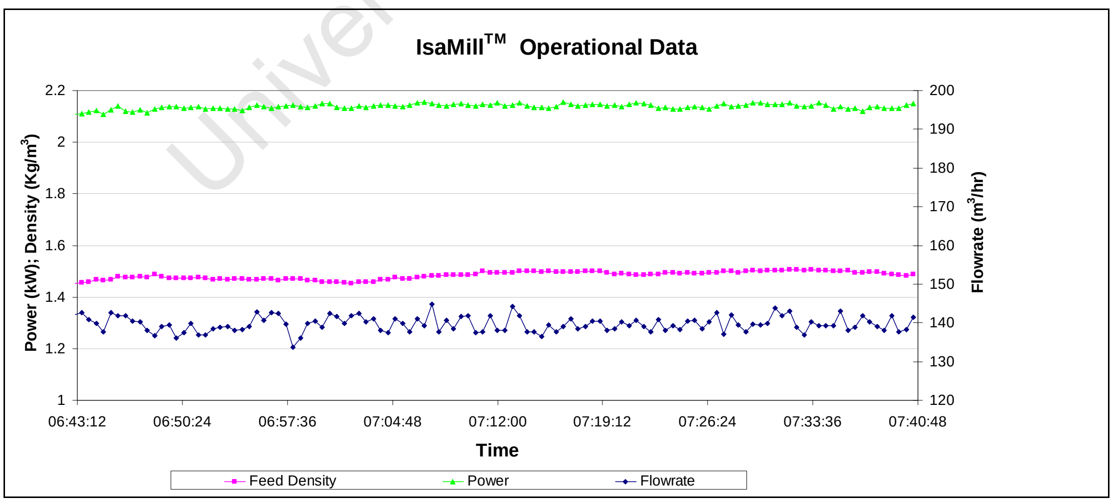

Figure 3-9: IsaMill operating data for the test period

Table 3-4: Actual operating variable levels of the M10 000 IsaMill^{TM} utilized during the survey

Variable	Level
Feed size	80 percent passing particle size ~ 110µm
Slurry density	1.48 kg/l
Volumetric flow-rate	140 m^3/hr
Power draw	2.123 MW
Media type & size	Keramax MT1 media, top size 3.5mm

3.3.1.1 Experimental procedures

Conducting the surveys on the IsaMill™ circuit required careful planning on how the sample at each sampling point would be taken, so as to ensure that it would be representative of the stream from which it was taken. Composite samples from the IsaMill™ feed and discharge were collected over a period of one hour. The feed samples were taken from the spigot discharge of each hydrocyclone on the cluster by using the sample cutter shown in Figure 3-10.

While the discharge samples were collected from the pipe feeding the mill discharge sump using the sample cutter indicated in Figure 3-11. The custom made sample cutters were used in order to achieve an optimum cut of the stream flow and a statistically adequate sample size. The selected sample cutters were such that they were able to access the whole range of the stream without difficulties. The speed of the cutter across the stream was, as far as possible, kept constant in order to take an even volume of sample per cut.

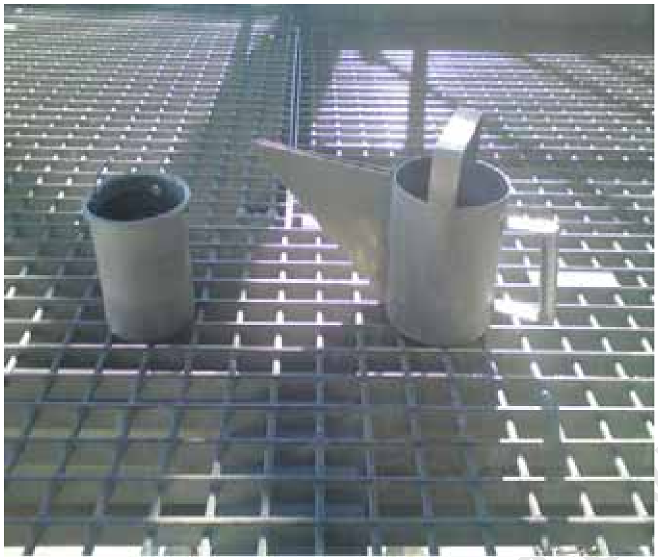

Figure 3-10: Photo indicating the sample cutter used for taking the IsaMill™ feed sample

Figure 3-11: Researcher with sample cutter used for taking the IsaMill™ discharge sample

During the sampling campaign, the plant operating data were constantly checked from the control room monitors, to assess whether the circuit was operating under sufficiently steady state conditions. If the operating conditions revealed any significant variation from the desired settings, the samples were discarded immediately; and the test was restarted at a later more suitable period. This was done because any fluctuations in mill operations would reflect in the components of each stream and introduce errors into the samples (Mainza, 2006). The IsaMill^{TM} power draw and flowrates in the streams were carefully monitored during the entire test.

For each survey, a series of five samples from each stream were taken at 15 minute intervals at all sampling points on the IsaMill^{TM} circuit, for a test period of 60 minutes, during which time, the mill was held as close as possible to steady the operation. All the samples collected from a sampling point were poured into the same sample bucket and any suspect sample was immediately discarded and a fresh one taken.

Two samples were taken in separate buckets for each sampling interval: the “A” sample and the “B” sample. The “A” sample is the primary sample used for analysis. The “B” sample was taken as a back-up only to be used if there was an error in the processing of samples or any uncertainty in the results (Mainza, 2006).

3.3.2 Western Limb Tailings Re-treatment Plant

The Anglo Platinum’s Western Limb Tailings Re-treatment Plant (WLTRP) treats reclaimed material from the old Klipfontein tailings dam. The reclaimed material contains a mixture of UG2 and Merensky ore. The process flow-sheet for the WLTRP is shown in Figure 3-12.

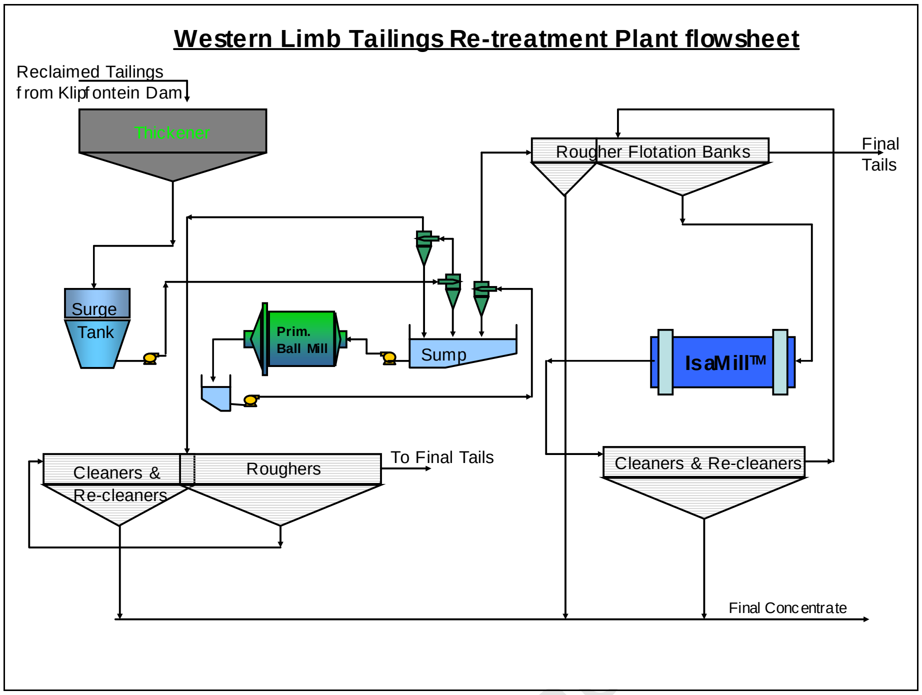

Figure 3-12: Schematic of the Western Limb Tailings Re-treatment Plant flow-sheet

The tailings dam material is re-pulped by the use of high-pressure water guns. The slurry gravitates to the launder and into a pump box, and is then pumped to a reclamation transfer pump, where it is screened to remove any unwanted oversize grit and vegetation, and then to the booster pump station and finally pumped to the reclaim thickener. The reclaimed material density varies between 1300 – 1500 Kg/m^3.

The density of the slurry is increased by the sedimentation method in the thickener were the solids settle to the bottom, while water overflows from the top of the thickener. The overflow from the thickener is fed into water reservoirs for service water and reclamation purposes. The thickener underflow is pumped to a series of hydrocyclones for solids classification. The hydrocyclones overflow is fed to the rougher flotation bank for PGM recovery.

The underflow reports to the ball mill for grinding. The ball mill is operated in closed circuit with a hydrocyclone. The hydrocyclone overflow is fed to a second set of rougher flotation cells, while the underflow is recycled back to the ball mill. The tailings from the rougher banks are disposed off to the tailings dam as final tailings. The concentrates obtained from the rougher bank stream are fed to the IsaMill^{TM} for re-grinding. Second-stage grinding, using the IsaMill^{TM} is done to liberate any more platinum minerals from the middling material thereby, increasing the grade of the PGM concentrate. The IsaMill^{TM} discharge then goes to cleaners and the re-cleaner flotation banks for further concentration of the platinum mineral.

Three sampling campaigns were conducted on the IsaMill^{TM} at WLTRP. These surveys were conducted to investigate the effect of flowrate on mill performance. The plant was operated as close to steady state as possible, before taking the samples. The plant stability during the sampling campaigns was assessed by carefully monitoring the IsaMill^{TM} operating variables, such as power draw, flowrates and feed density on the control room monitor. The trend obtained during one of the surveys is shown in Figure 3-13.

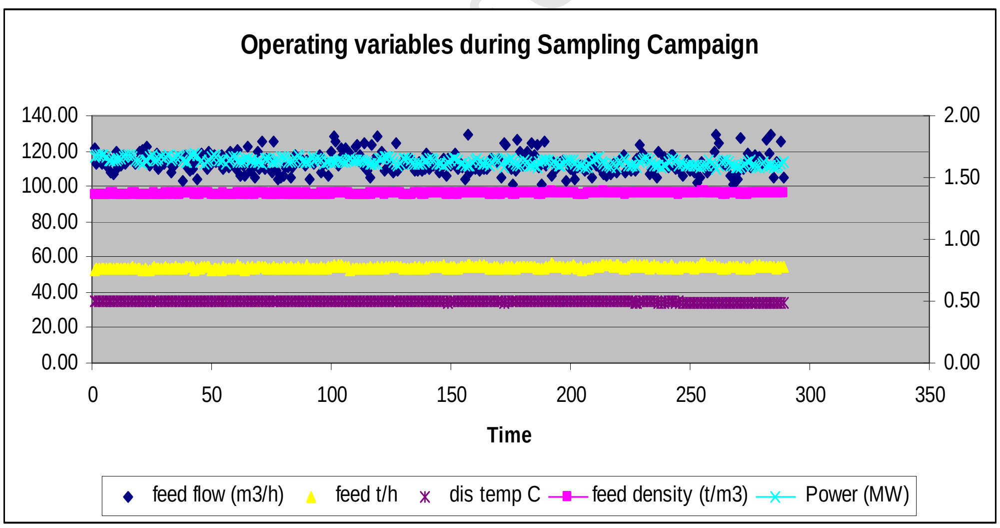

Figure 3-13: Monitoring of operating variables during the sampling campaign

The survey was stopped immediately if significant differences in operating variables, such as mill power draw, feed flow rate, slurry density and mill pressure were observed. The plant was operated under similar conditions in terms of mill power draw, slurry density and mill pressure, while the flowrate was varied, as indicated in Table 3-5.

Table 3-5: Operating variables during one of the sampling campaign at WLTRP

Variable	**Level**
Feed sizes	P_{80} ~ 33 and 40µm
Slurry density	1.35 t/ m^3
Volumetric flow-rates	100, 80 and 76 m^3/hr
Power draw	1.95 MW
Media	Silica sand media, top size 3.5mm

3.3.2.1 Experimental procedures

Samples of the IsaMill™ feed and discharge were collected using custom made sample cutters, similar to those used for surveys at Waterval Concentrator. The samples were then transferred into properly labeled clean buckets. Six to seven cuts were necessary to collect the sample required for processing.

In order to achieve a representative sample for the survey period, sampling for each test was conducted over a period of 60 minutes, and at 15-minute intervals. For each survey, samples of the IsaMill™ feed and discharge were taken into separate buckets. These were prepared and processed using the methodology described in Section 3.3.3. The samples collected were weighed immediately after the survey to obtain wet masses. The samples were then filtered, using a press filter and then placed in the oven for drying at the temperature of 105ºC. The weight of the dry sample was then taken. The wet and dry masses were used to calculate the slurry solids concentration for each sample.

3.3.3 Sample preparation and processing

Samples collected from the testwork were handled with care to avoid contamination or loss during preparation and processing. The standard sample preparation and processing used by the Centre for Minerals Research at the University of Cape Town was employed in this project (MPTech, 2008).

Figure 3-14: Rotary splitter

After getting all the samples' dry masses, the "A" samples were de-lumped on a 1mm screen and split into smaller samples of at least 300g, using a rotary splitter, as shown in Figure 3-14. One sub-sample from each sample was used for screening, to obtain the particle-size distributions. The other sub-samples were kept as back-up samples in case of problems, such as spillages that would require the immediate replacement of a sample.

3.3.4 Particle size distribution analysis

Two methods of analyzing for particle-size distributions were used for the samples collected during the M10 000 sampling campaigns. These were screen-sizing and laser-sizing methods. The laser sizing was done using the Malvern laser analyzer; and the details of the procedure are described in section 3.2.8. A description of the screen-sizing method is presented in this section (Powell & Mainza, 2002). Each sub-sample was weighed to establish its exact mass at the beginning of the screening process. This weight was used to check for the material loss during the screening process. Wet screening was applied for finer sizes, and dry screening for coarser sizes. The sub-sample was first wet screened on the 32μm sieve size. The 32μm screen was firmly clamped to the vibratory sieve shaker. The sample was first mixed with water to form slurry, before it was poured on to the screen. The shaker was turned on, and the water introduced onto the screen continuously throughout the wet screening process, and the undersize was collected in a properly labeled, clean bucket. The screening process was carried out, until only clear water was discharged from the undersize.

Table 3-6 indicates the size fractions which where applied in the wet and dry screening procedures.

Table 3-6: Size fractions applied in the dry and wet screening processes

Dry screening	Wet screening
+710µm	-125µm+90µm
-710µm +500µm	-90µm+63µm
-500µm +355µm	-63µm+45µm
-355µm +250µm	-45µm+32µm
-250µm +180µm	-32µm
-180µm +125µm	

The oversize particles were then placed on a 90µm screen, and wet-screening was conducted, using the same method, as described for the 32µm screen. The undersize was collected in a bucket. When only clear water was discharged at the bottom of the screen, the 90µm screen oversize was placed on a pan and dried in the oven. The dried material was then weighed, packed and stored for dry screening.

The undersize was then wet-screened on the 63µm screen, and the procedure used on the 90µm screening was repeated for the screen sizes of 63µm, 45µm and 32µm. The minus 32µm from the second pass was then combined with the original, minus 32µm fraction; and finally, it was filtered and dried. The dried oversize from 90mm was weighed and dry- screened on a stack consisting of screen sizes 125µm, 180µm, 250µm, 355µm, 500µm and 710µm. The screen stack was vibrated by a vibratory sieve shaker, as shown in Figure 3-15.

Figure 3-15: Vibratory sieve shaker used for dry screening

The resulting fractions from both wet and dry screening were weighed and the masses recorded. The integrity of the screening and weighing process was verified by immediately adding up all the screen fraction masses and the total sub 32μm fraction, and checking whether it was close to the total sub-sample mass, recorded at the beginning of the screening.

4.0 Results and discussions

Results showing the influence of some operating variables on the performance of the laboratory scale IsaMill^{TM} are presented in this chapter. The variables tested were stirrer speed, media load, media size, feed size, solids concentration and flow-rate. The results obtained from the M4 IsaMill^{TM} are then compared with the results obtained from sampling campaigns performed on M10 000 industrial scale IsaMill^{TM} at Waterval UG2 Concentrator and Western Limb Tailings Re-treatment Plant.

4.1 Effect of various operational and design variables of the IsaMill^{TM}

The results presented in this section indicate the effects of design and operational variables on the specific energy consumption, particle size distribution and reduction ratio for the M4 laboratory scale M4 IsaMill^{TM}.

4.1.1 Stirrer speed

Stirrer speed is one of the most important variables in stirred milling. In this study, tests were performed at three stirrer speeds: 1500rpm (low), 1800rpm (intermediate) and 2100rpm (high).

4.1.1.1 Effect of stirrer speed on specific energy

Figure 4-1 shows the relationship between stirrer speed and the cumulative specific energy consumed, after four passes. A linear relationship between the total specific energy and the stirrer speed was obtained within the range of stirrer speeds (1500rpm – 2100rpm) tested. The error bars shown are within a 95% confidence interval; and therefore, the cumulative specific energy consumed at each pass when the IsaMill^{TM} is operated at different stirrer speeds can be said to be significantly different from one another. Tests performed at higher stirrer speeds appeared to consume more energy than tests performed at lower stirrer speeds. The trends observed show typical behavior of stirred mills as observed by Van der Westhuizen *et al.*, (2010); Weller & Gao (1999) and Zheng et al., (1996). The increase in specific grinding energy consumed at higher stirrer speeds could be postulated to be due the increased energy of motion required to drive the stirrer and charge at higher speed.

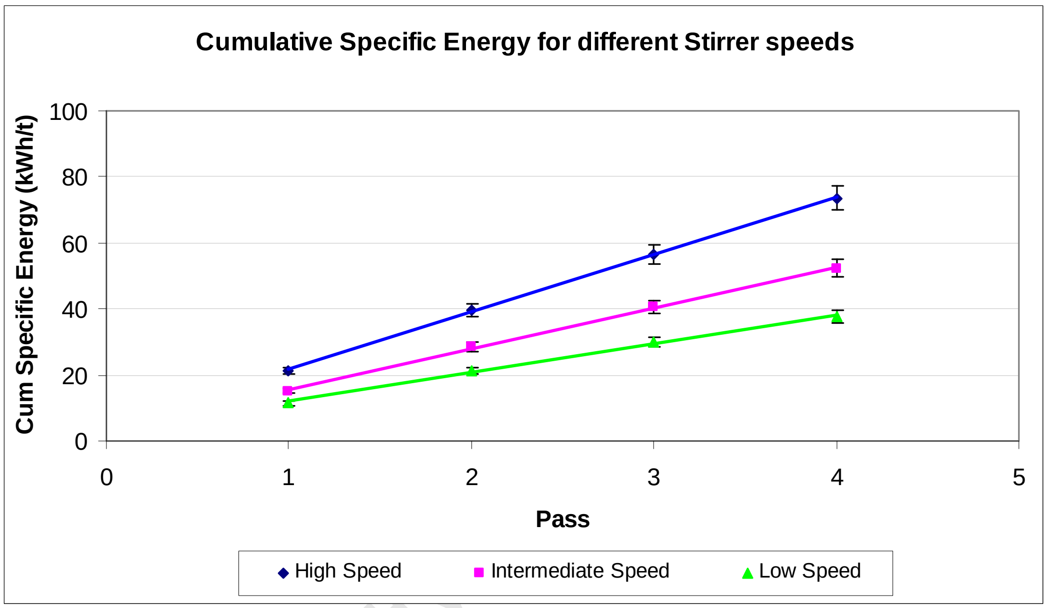

Figure 4-1: Effect of stirrer speed on total specific energy consumption

4.1.1.2 Effect of stirrer speed on particle size distributions.

The data was analysed further to determine the influence of stirrer speed on particle size distributions and fineness of grind that can be achieved when the M4 IsaMill™ is used to treat UG2 ore.

Figure 4-2 to Figure 4-5 shows the comparison of product particle size distributions obtained from the different speeds at similar passes. Similar feed particle size distributions (P_{80} = 120µm) were used in pass 1 for all the stirrer speed tests as indicated in Figure 4-2. The subsequent product obtained from the preceding pass was then used as feed material for the next pass. This imposes a limitation of discussing particle size distributions obtained for passes 2, 3 and 4 due to the differences in feed particle sizes used.

Figure 4-2 shows that the increases in stirrer speed from 1500rpm to 2100rpm produced a finer product. This is in line with the findings of Weller & Gao (1999) and Zheng *et al*., (1996) who suggested that higher stirrer speeds in stirred mills offer the better processing conditions resulting in increased product fineness. Figure 4-2 also indicates that the product particle size distributions obtained at stirrer speed 2100rpm and 1800rpm (P_{80} = 45µm and P_{80} = 50µm respectively) are much finer than that achieved at 1500rpm (P_{80} = 85µm). This could be because at lower speed (1500rpm), the grinding media is not fully energised to break the particles compared to speeds of 1800rpm and 2100rpm (Becker *et al*., 2001). Similar particle size distributions were obtained for 1800rpm and 2100rpm indicating existence of optimum speed around 1800rpm as there was seemingly no significant benefit in increasing the speed to 2100rpm (Yang *et al*., 2006).

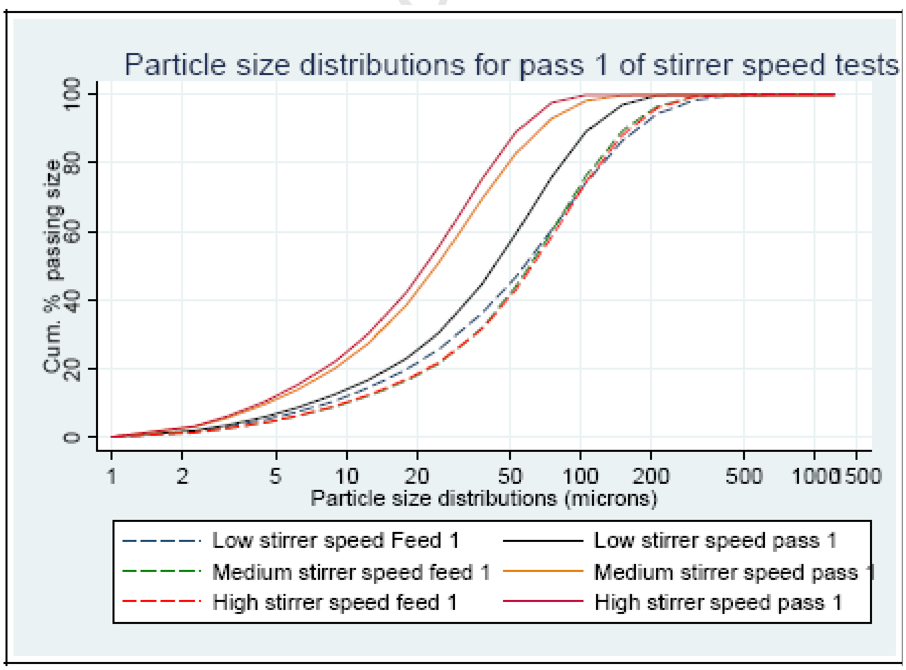

Figure 4-2: Pass 1 product particle size distributions obtained for stirrer speed tests. Conditions: high speed = 2100rpm, medium speed = 1800rpm, low speed = 1500rpm.

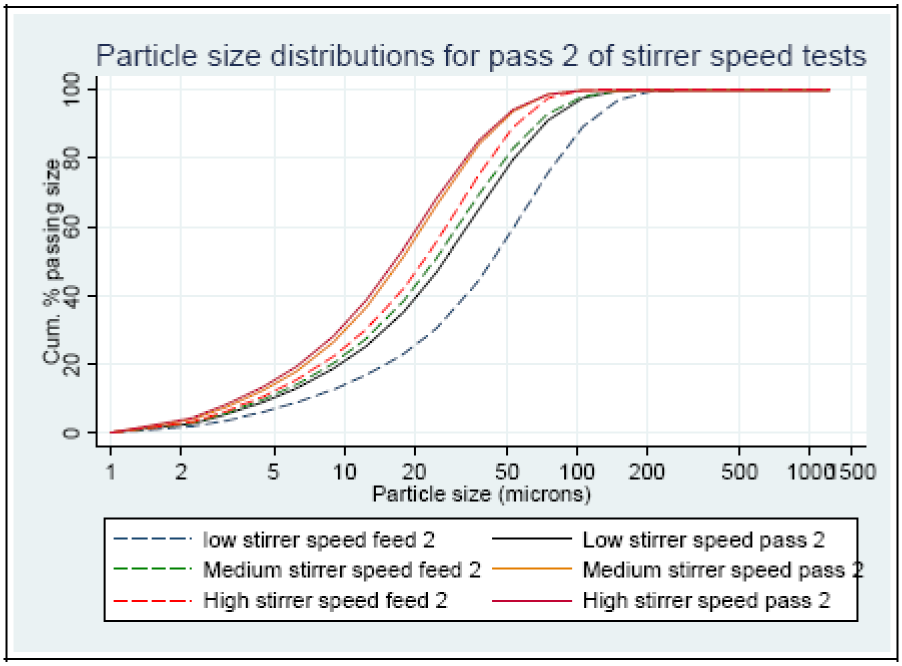

Figure 4-3: Pass 2 product particle size distributions obtained for stirrer speed tests. Conditions: high speed = 2100rpm, medium speed = 1800rpm, low speed = 1500rpm.

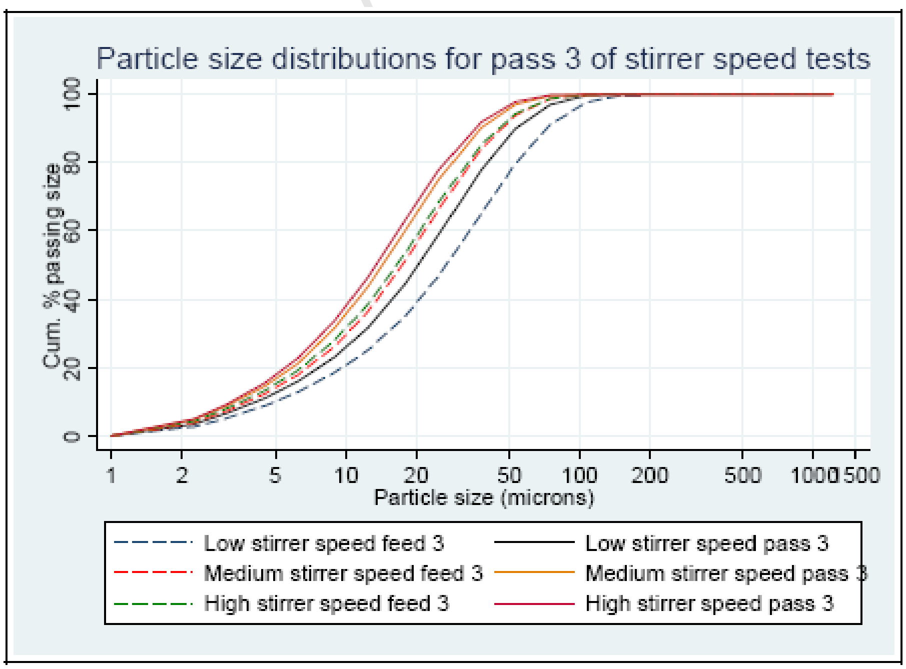

Figure 4-4: Pass 3 product particle size distributions obtained for stirrer speed tests. Conditions: high speed = 2100rpm, medium speed = 1800rpm, low speed = 1500rpm.

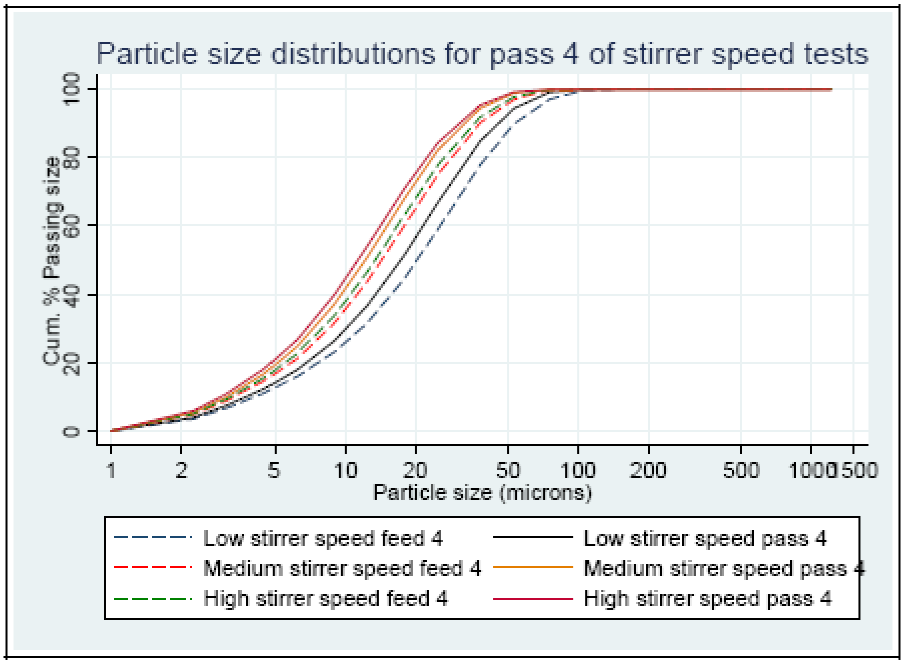

Figure 4-5: Pass 4 product particle size distributions obtained for stirrer speed tests. Conditions: high speed = 2100rpm, medium speed = 1800rpm, low speed = 1500rpm.

Figure 4-6 shows the effect of stirrer speed on the energy required to grind the feed material (P_{80}=120µm) to a product size of P_{80}=38µm. The trend shows a decrease in specific energy required to producing P_{80}=38µm from the speed of 1500rpm to 1800rpm, but increasing energy consumption was noted as the speed was increased from 1800rpm to 2100rpm. This indicates that for the three speeds used in this study, 1800rpm appeared to be the optimum speed to achieve product of P_{80}=38µm. This result differs from those observed by Zheng *et al.*, (1996) who suggested that the stirred mills efficiency increases with increasing stirrer speed but is in agreement with the findings by Yang *et al.*, (2006) indicating the existence of an optimum stirrer speed for stirred mills applications. This implies that the media in the mill gets energised more with increased stirrer speed leading to better energy efficiencies, but beyond a certain point centrifugation of media and media compression starts to impact negatively on mill efficiency (Yang *et al.*, 2006).

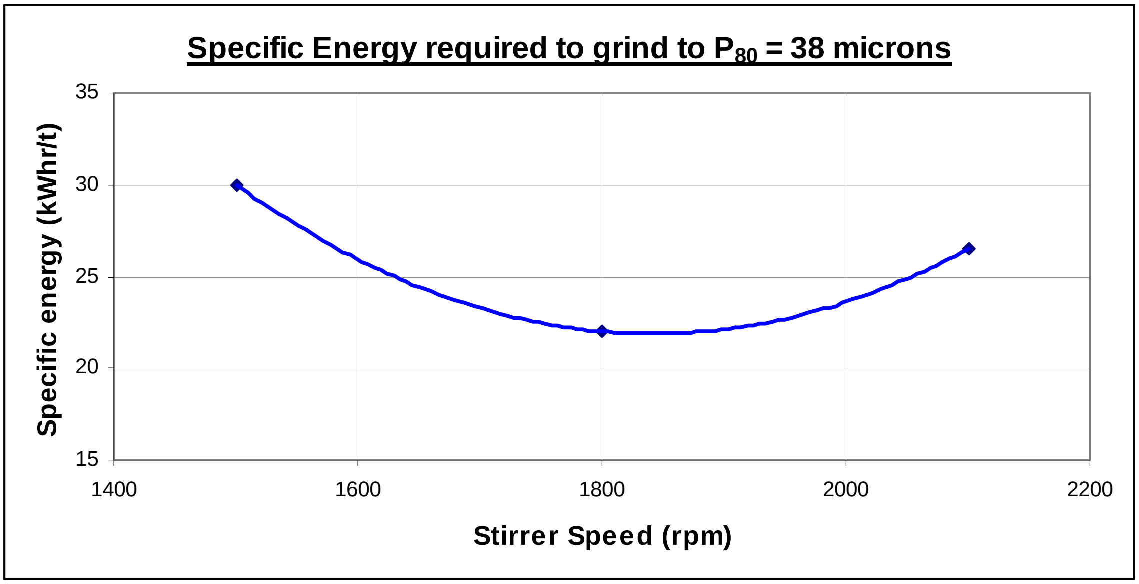

Figure 4-6: Effect of stirrer speed on energy required to grind to P_{80} = 38µm. Conditions: high speed = 2100rpm, medium speed = 1800rpm, low speed = 1500rpm.

4.1.1.3 Effect of stirrer speed on specific energy-particle size distribution relationship

Conventionally, the performance of the IsaMill™ is described in terms of specific energy (kWh/t) against product fineness. The product fineness is characterized by the particle size at which 80 percent of the material passes. The graph indicating the relationship between specific energy and product size is usually referred to as the signature plot (Weller & Gao, 1999; Shi *et al.*, 2009). The relationship between specific energy and reduction ratio has been analysed to determine the efficiency of the size reduction that can be achieved when the mill is operated at different levels of operational variables.

Figure 4-5 shows signature plots obtained for high (2100rpm), medium (1800rpm) and low (1500rpm) stirrer speed tests. Other operating variables such as media size, media load, slurry percent solids and flowrate were kept constant while varying the stirrer speed. The media size used in these tests was 3.5mm, while media load was maintained at 65%. The slurry percent solids was maintained at 40% and feed slurry flowrates at

2.5l/min. It can be seen that specific energy consumption dropped when the stirrer speed was increased from 1500rpm to 1800rpm. However, the specific energy consumption increased with increasing stirrer speeds when the speed was increased from 1800rpm to 2100rpm. The lowest specific energy consumption was achieved when the IsaMill™ was operated at stirrer speed of 1800rpm compared with those obtained for 1500rpm and 2100rpm. Therefore, the energy efficiencies of the IsaMill™ appear to be low when the Mill was operated at 1500rpm and 2100rpm and the optimum is around 1800rpm. This is in line with the results obtained by Yang *et al.*, (2006) using DEM simulations which indicated the existence of an optimum stirrer speed for stirred mills applications.

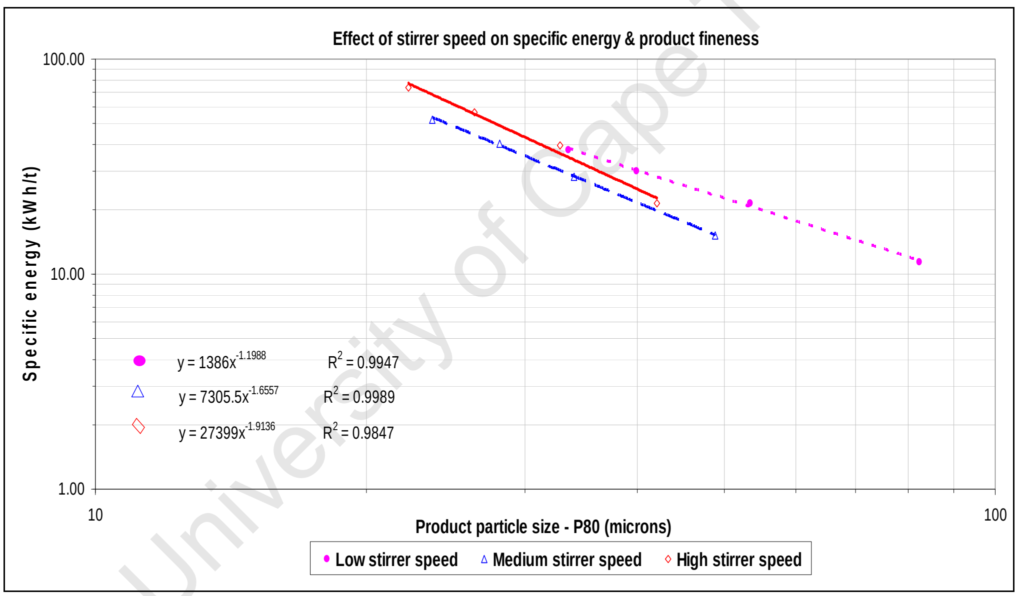

Figure 4-7: Effect of stirrer speed on specific energy and product fineness. Conditions: high speed = 2100rpm, medium speed = 1800rpm, low speed = 1500rpm

It appears that a critical stirrer speed necessary for a certain media size and slurry solids concentration required to break a given feed particle size exists. Once this has been obtained, there is no benefit in increasing the stirrer speed on a specific energy/grind basis. These results are similar to the findings of Yang *et al.* (2006), who suggested that an optimum stirrer speed in IsaMill™ exists. This optimum may vary with flow and other operational conditions. According to Yang *et al.* (2006), an increase in stirrer speed drives the particles outwards and forms a faster flow with an increased velocity gradient along the radial direction. The compressive forces on particles also increase with increasing stirrer speeds, but the increase of the flow velocity and stirrer speed is not linear (Yang *et al.*, 2006). As the energy dissipation within the mill increases with the flow velocity, the flow velocity reaches a point where it does not increase any more. Further increases in stirrer speed would then have no impact on the flow velocity leading to deterioration in the energy efficiency. At lower stirrer speeds (1500rpm), the stress intensity and flow velocity within the mill are low, giving limited media-particle interaction, and thereby causing poor energy efficiency. According to Wang and Forssberg (2000), sufficient mechanical stress intensity of the grinding media within the stirred mill is required for effective comminution to occur. Therefore, the stirrer speed must be sufficiently high for the use of the smaller media used in stirred mills. Grinding becomes inefficient at low stirrer speeds, when the stress intensity of the media is rather low, and leads to coarser product fineness as indicated by the results obtained by Becker *et al.,* (2001).

4.1.2 Media load

The media load in stirred media mills is one of the most important operating variables in terms of energy efficiency. In horizontal stirred mills, much better energy utilization can be achieved at higher media loads; however, an optimum media load exists and overloading the grinding media in a stirred mill should be avoided (Van der Westhuizen *et al.,* 2010; Gao *et al.,* 2007). In this project, media loads of 50% (low), 65% (intermediate) and 80% (high) loads were tested. The range of media load (50% to 80%) for this project was selected based on literature which indicates optimum media load within the chosen span (Van der Westhuizen *et al.,* 2010; Gao *et al.,* 2007).

4.1.2.1 Effect of media load on specific energy

Figure 4-8 shows the cumulative specific energy consumed at every pass for different media load tests. It can be seen that the cumulative specific energy for media load tests linearly increased from pass 1 through to pass 4. The specific energy consumed also appeared to increase for higher media loads. The specific energy consumption increased for higher media loads because more energy is required to set in motion the extra media in the mill.

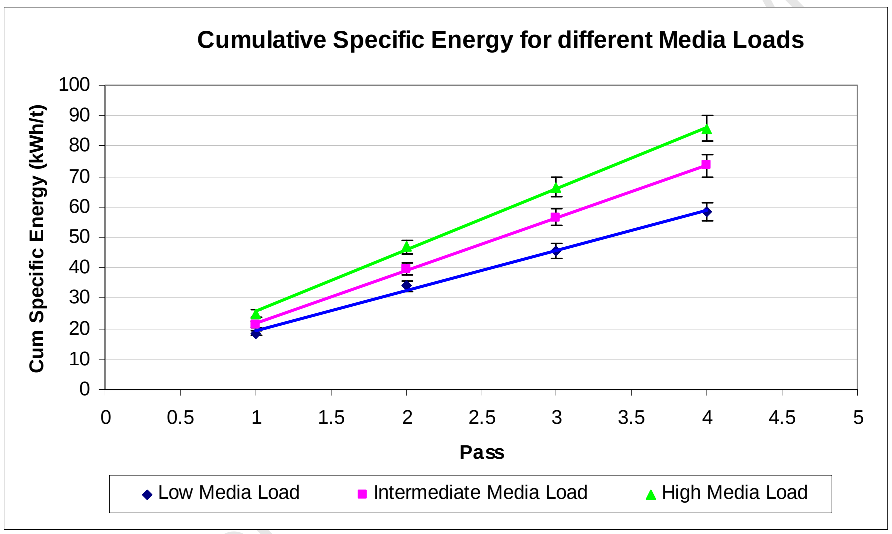

Figure 4-8: Effect of stirrer speed on total specific energy consumed

4.1.2.2 Effect on particle size distributions and reduction ratios

Figure 4-9 to Figure 4-12 shows the particle size distributions obtained when grinding UG2 platinum ore at different media loads (50%, 65% and 80%). The particle size distributions obtained for high and intermediate media loads appear to be similar and finer than those achieved at low media load. This trend, however, drifts for passes 2, 3 and 4 where the particle size distributions of high media load are much finer than for both intermediate and low media loads. No significant differences were observed in the particle size distributions achieved for intermediate and low media loads during passes 2, 3 and 4. These results indicate that higher media loads would tend to produce finer product. This could be attributed to the increased grinding surfaces and compressive forces on particles achieved at higher media loads (Yang *et al.*, 2006). A more uniformly distribution of particles is also obtained at higher media load increasing the probability of particle breakage (Jayasundra *et al.*, 2010).

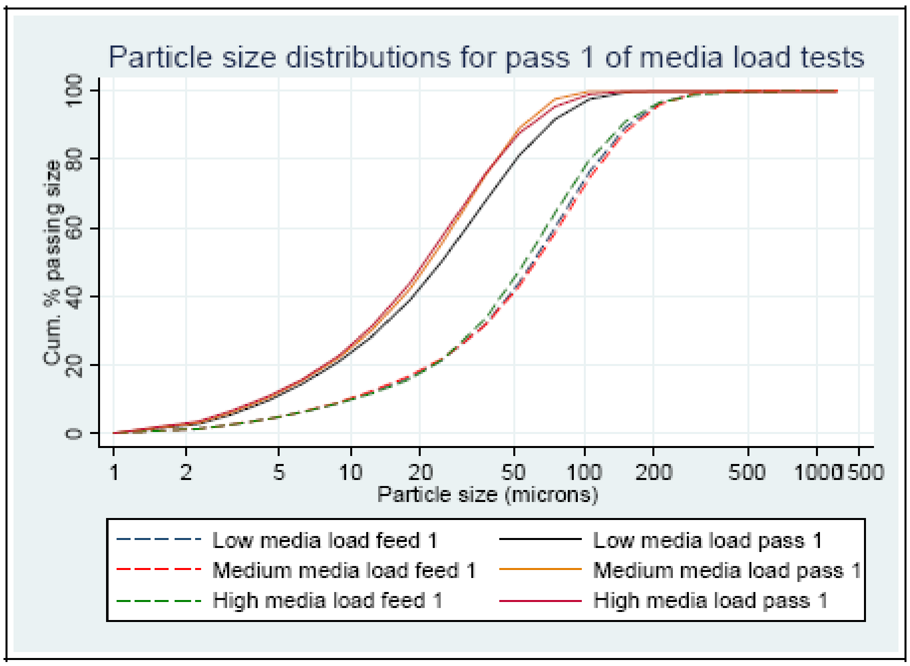

Figure 4-9: Particle-size distributions obtained in the first pass of media load test; low media load = 50%, medium media load = 65%, high media load = 80%

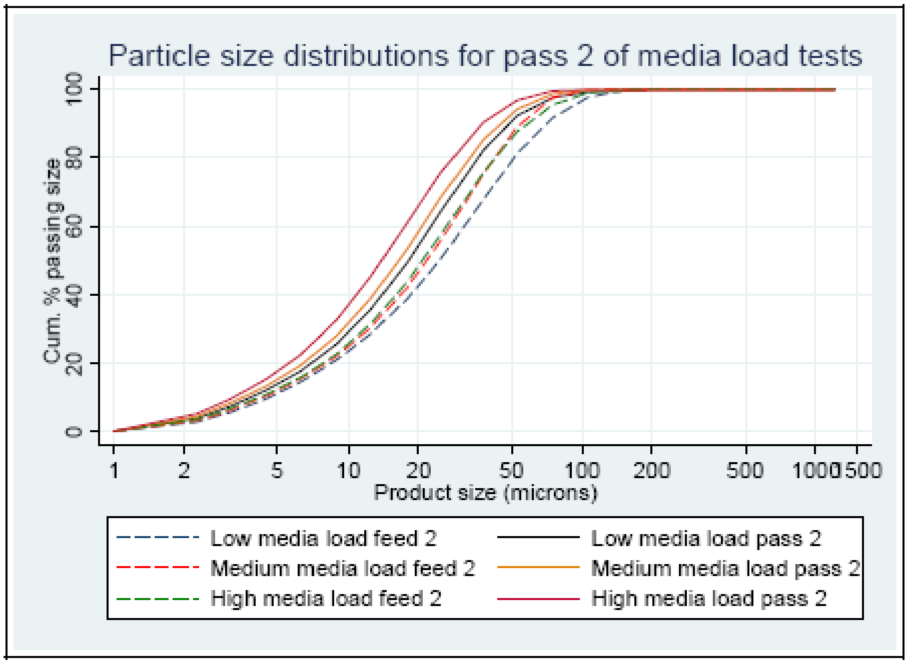

Figure 4-10: Particle-size distributions obtained during the second pass for media load tests; low media load = 50%, medium media load = 65%, high media load = 80%

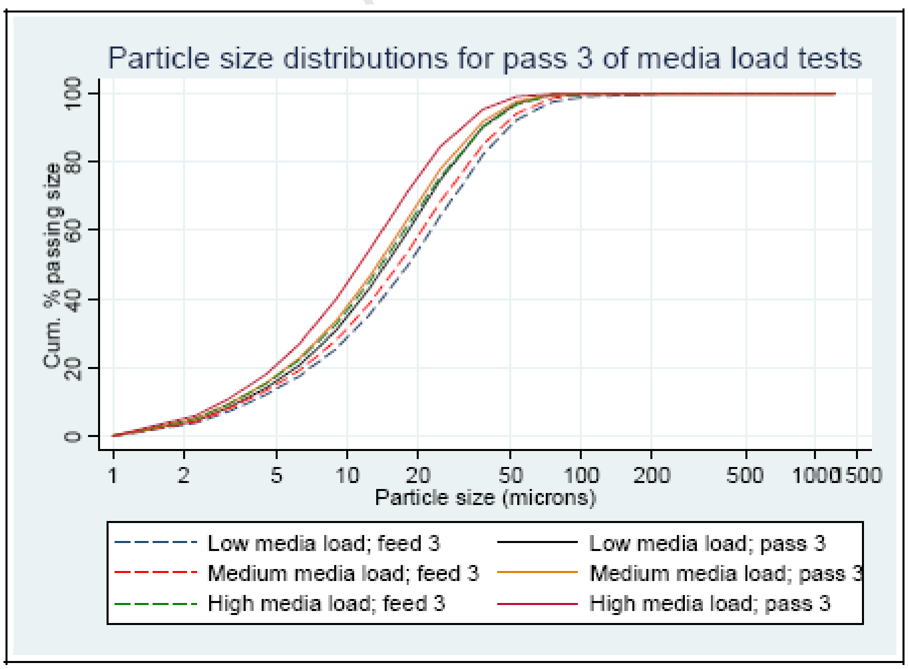

Figure 4-11: Particle-size distributions obtained during the third pass for media load tests; low media load = 50%, medium media load = 65%, high media load = 80% tests

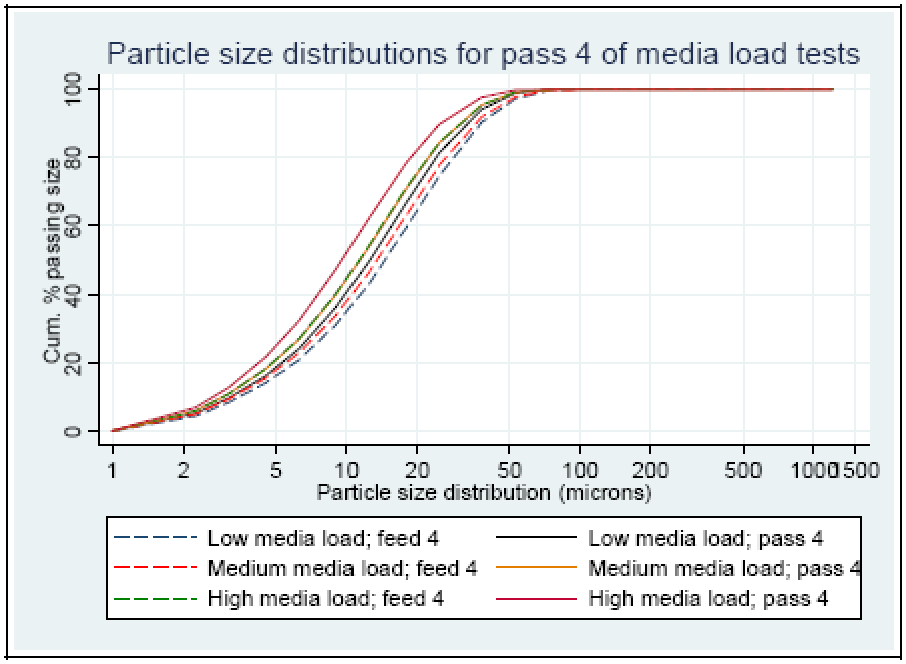

Figure 4-12: Particle-size distributions obtained during the fourth pass for media load tests; low media load = 50%, medium media load = 65%, high media load = 80% tests

To further investigate the grind achieved for the media load tests, the reduction ratios obtained at different media loads were plotted and presented in Figure 4-13. The reduction ratios appear to be higher in the first pass for all the media loads evaluated. This could be attributed to the larger particles in pass 1 having a greater chance of being captured and broken than the smaller ones in subsequent passes (Wang & Forssberg, 2007). The highest reduction ratio during pass 1 was achieved when grinding the ore at intermediate media load, while the lowest was obtained for the low media load test. No significant differences in reduction ratios achieved were observed for passes 2 to 4 with variation in media loads.

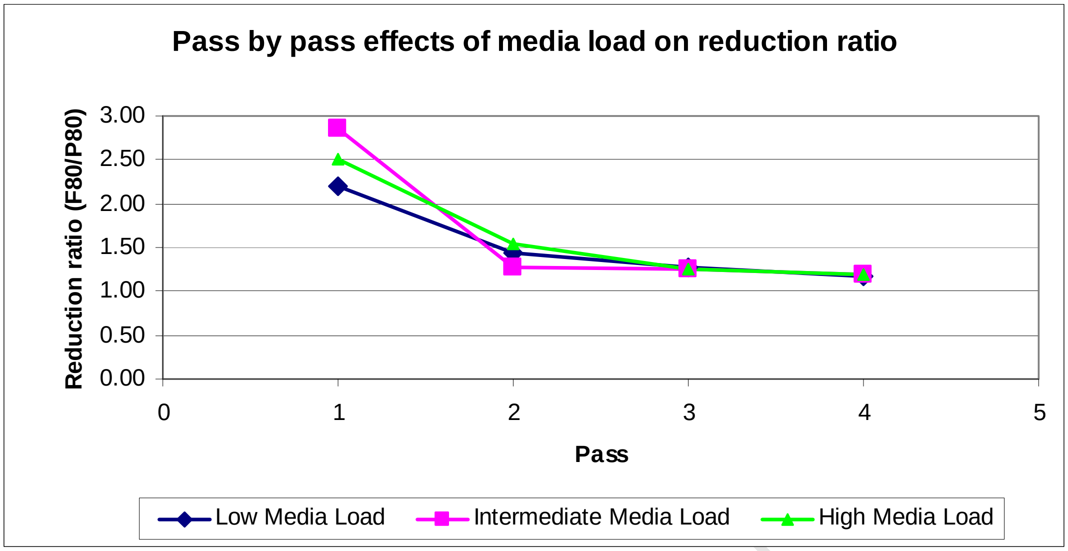

Figure 4-13: Effect of media load on reduction ratio

The reduction ratios obtained during pass 1 for media load tests were analysed further and presented in Figure 4-14. The results indicate an increase in reduction ratio with increased media load from 50% to 65% and a decrease as media load continues to increase from 65% to 80%. This suggests that there is an optimum media load which is close to intermediate media load in this case. This is in line with the observations made by Van der Westhuizen *et al.*, (2010) and Gao *et al.*, (2007) who indicated the existence of an optimum media load in stirred mills applications.

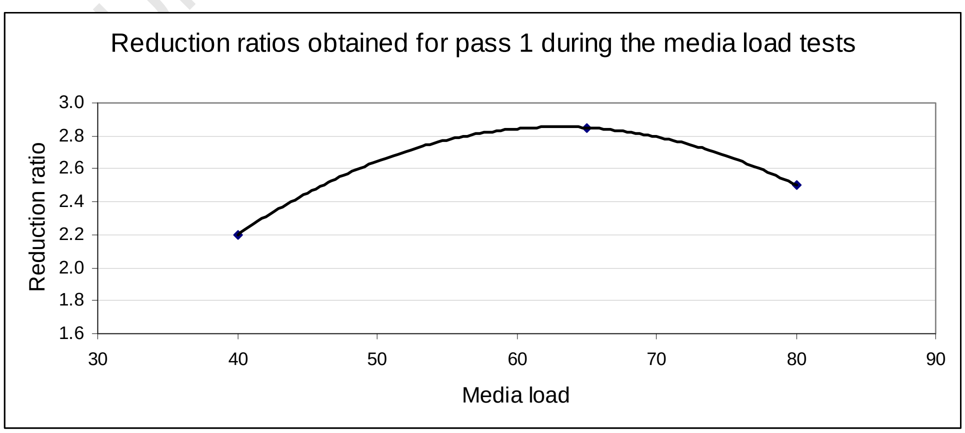

Figure 4-14: Reduction ratios achieved during the first pass of media load tests.

4.1.2.3 Effect of media load on specific energy particle size distribution relationship

Figure 4-15 shows signature plots obtained for high (80%), intermediate (65%) and low (50%) media load tests. Other operating variables such as stirrer speed, media size, slurry percent solids and flowrate were kept constant while varying the media load. The stirrer speed was set at 2100rpm, media size used in these tests was 3.5mm, while the slurry percent solids was maintained at 50% and feed slurry flowrates at 2.5l/min. The signature plots obtained when grinding UG2 ore of F_{80}=120µm indicate no significant differences in the specific energy required to achieve product size ranging from P_{80}=20µm to P_{80}=40µm. This shows that the efficiencies of grinding UG2 platinum ore of F_{80}=120µm are similar for all the media loads tested.

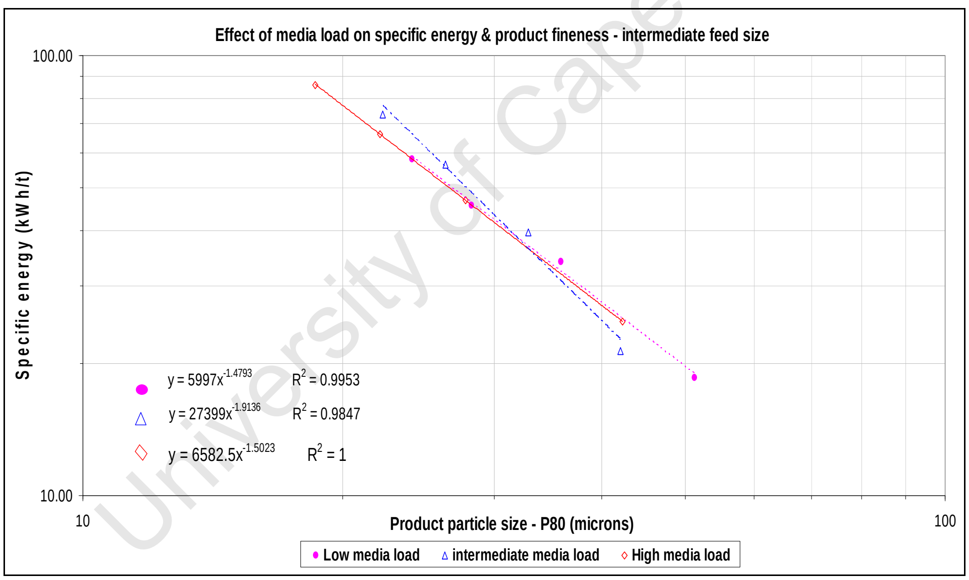

Figure 4-15: Effect of media load on specific energy and product fineness. Conditions: Feed F_{80}=120µm, Low media load = 50%, Intermediate media load = 65%, High media load = 80%

To further investigate the effect of media load on the relationship between specific energy consumption and product fineness more tests were conducted using finer (F_{80}=55µm) and coarser (F_{80}=250µm) feed sizes and results are presented in Figure 4-16 and Figure 4-17 respectively. The signature plots obtained for feeds of F_{80}=55µm and F_{80}=250µm indicate higher specific energy consumption for high media load (80%)

compared with those utilized for intermediate (65%) and low (50%) media loads. The trends obtained in Figure 4-16 also indicate similar specific energy consumption at 65% and 50% media load required for producing the desired product size for intermediate and low media load tests. This indicates that grinding process with fine ore feed was more efficient at 65% and 50% media loads compared with 80% media loads.

The signature plots obtained when grinding coarse (F_{80}=250µm) feed shown in Figure 4-17 appear to be significantly different for the three media loads tested. To achieve a product size of P_{80} between 20µm and 90µm, the specific energy consumption was seen to reduce with increased media load from 50% to 65% but increased as media load was increased from 65% to 80%. This suggests that there is an optimum media load when grinding coarse feed (F_{80}=250µm) which is close to 65% load confirming the observations made using reduction ratios in section 4.1.2.2.

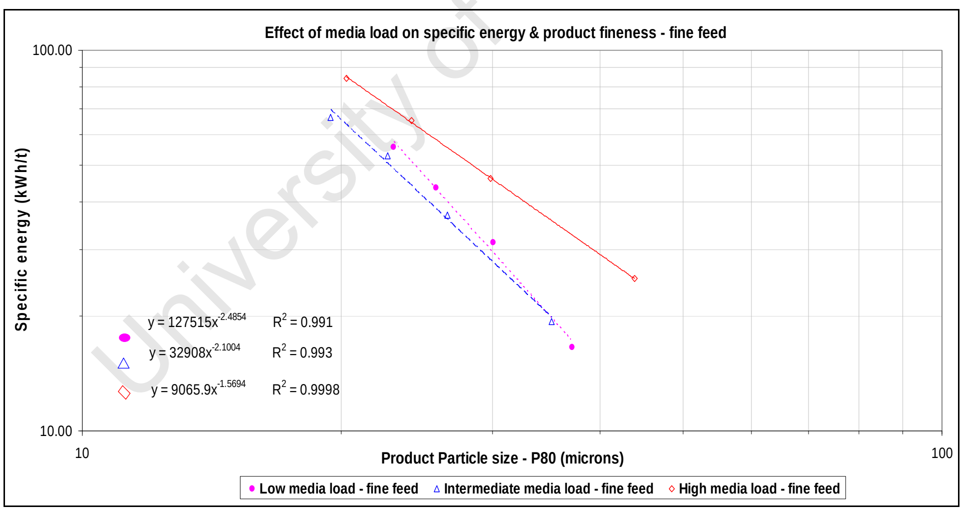

Figure 4-16: Effect of media load on specific energy and product fineness. Conditions: Fine feed size (F_{80}=55µm), Low media load = 50%, Intermediate media load = 65%, High media load = 80%

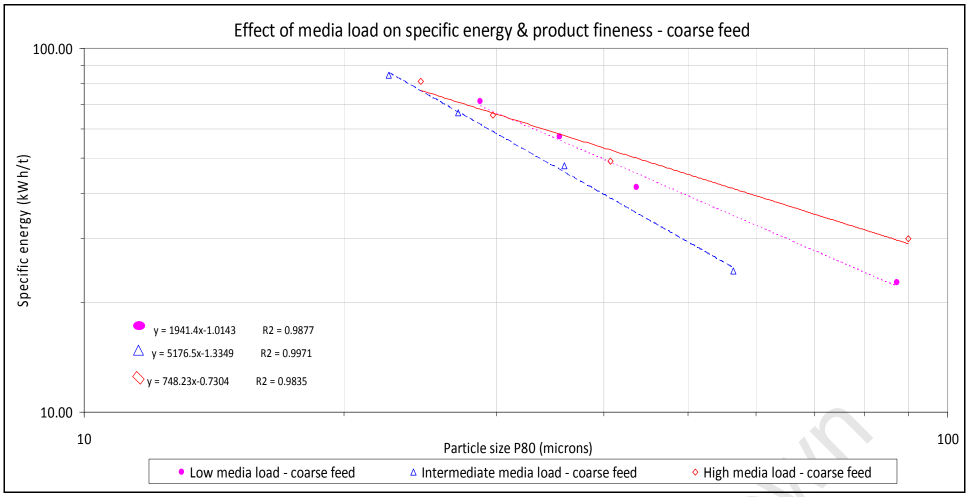

Figure 4-17: Effect of media load on specific energy and product fineness. Conditions: Coarse feed size (F80=250µm), Low media load = 50%, Intermediate media load = 65%, High media load = 80%

Figure 4-18 shows the summary plots of specific energy consumed to grind different feed sizes of UG2 ore to P_{80}=38µm at the three media loads; 50%, 65% and 80%. It is observed that higher specific energies were consumed when grinding coarse feed (F_{80}=250µm) at all media load tests followed by those consumed for intermediate feed (F_{80}=120µm). Lowest specific energy was consumed when grinding fine feed material (F_{80}=55µm).

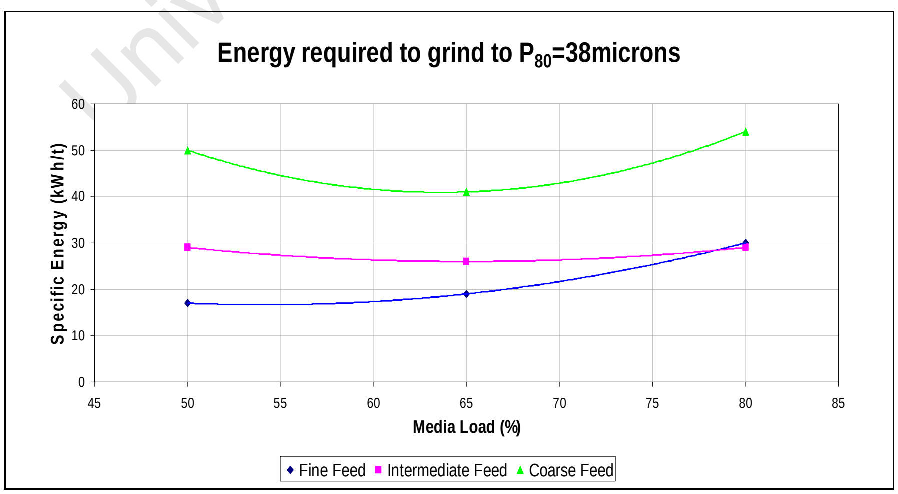

Figure 4-18: Summary plot of specific energy consumption at media loads of 50%, 65% and 80% feed sizes of F_{80}=55µm, F_{80}=120µm and F_{80}=250µm.

The trends observed in Figure 4-15 to Figure 4-18 suggest that the feed size has a significant influence on the relationship between the specific energy and product fineness at any given media load. In Figure 4-18, the IsaMill™ energy efficiency was seen to improve when the media load is increased from low to intermediate; and appears to decrease with further increases up to high media loads When grinding coarse material (F_{80}=250µm). This implies that the best energy efficiency and product fineness for grinding coarse feed size were obtained at a media load of 65% (intermediate load). No significant differences in specific energy consumption were observed at the three media loads tested for intermediate feed (F_{80}=120µm). The specific energy consumption when grinding fine feed (F_{80}=55µm) increased with increase in media load from 50% to 80% load. The influence of feed size on the relationship between the specific energy and product fineness at any given media load is in line with the principles of ideal packing conditions. According to ideal packing conditions for two component mixtures, the critical ratio of filling for media and particles is given by:

$$R = \frac{1-\varepsilon_m}{\varepsilon_m(1-\varepsilon_p)}$$ **Equation 4-1**

Where: Media porosity is ε_m and Particle packing porosity is ε_p.

Consequently, the best grinding conditions occur when the voids in the grinding media packing are just filled with particles. These ideal packing conditions occurs at different media loads when grinding various sizes of feed materials and confirms the existence of optimum media load in stirred milling as also observed by Van der Westhuizen *et al.*, (2010) and Gao *et al.*, (2007). For this project, the optimum media load was seen to lie close to the intermediate media load (65% media load), where the lowest specific energy was consumed when grinding the UG2 platinum ore to product sizes between 20µm and 40µm.

4.1.3 Media sizes and feed sizes

The most appropriate media size is a function of both the feed size and the desired product fineness. Media size ensures that the coarsest particles can be broken and do not build up in the IsaMill^{TM}. The media, therefore, need to be sufficiently coarse to break the largest particles in the feed. When the media size is too small for the feed, the mill tends to produce a wider product size distribution still containing a portion of the unbroken feed which would have a negative effect on flotation processes (Gao *et al.,* 2007). Therefore, correct selection of media size for a particular stirred mill application would greatly reduce the energy consumption and also provide a higher mill throughput for the installed mill power (Wang & Forssberg, 2007; Gao *et al.*, 2007).

The data obtained in this project provide a basic understanding of the relationships of media size and feed particle size distribution with specific energy consumption. The effect of media size on energy efficiency was evaluated by operating the IsaMill^{TM} with 2mm, 3.5mm and 5mm from here, henceforth referred to as small, intermediate and large media sizes, respectively. The three media sizes (2mm, 3.5mm and 5mm) were chosen based on the fact that 2mm and 3.5mm grinding media is used for ultra fine and main stream grinding applications when processing UG2 platinum ores respectively. The 5mm media size was also included to investigate the use of larger media sizes in main stream grinding applications. To demonstrate the effect of feed particle size on the IsaMill^{TM} performance, three different feed sizes were tested in this work. These are feed sizes of F_{80} ~ 250µm, F_{80} ~ 120µm and F_{80} ~ 55µm referred to as coarse, medium and fine feed sizes, respectively. Other operating parameters such as stirrer speed, media load, solids concentration and feed flowrate were kept constant during these tests. Stirrer speed was set at 1200rpm, media load was 65%, solids concentration at 40% and flowrate was maintained at 2.5l/min.

4.1.3.1 Effects of media size on specific energy

Figure 4-19 shows pass-by-pass cumulative specific energy consumed when the IsaMill™ was operated using different media sizes (2mm, 3.5mm and 5mm). A linear relationship between the passes and specific energy consumed at each pass was obtained within the range of media sizes (2mm – 5mm) tested. The error bars shown are within a 95% confidence interval; and therefore, results obtained indicate that for the range tested there is no significant difference between the cumulative specific energy consumed at each pass when different media sizes are used in the IsaMill™. This indicates that the pass-by-pass specific energy required to grind UG2 platinum ore of F_{80} = 120µm is similar when using media sizes of 2mm, 3.5mm and 5mm. However, as will be shown later, the product particle size distributions achieved by the three media sizes are different. Using the 2mm media, the IsaMill™ produced a much finer product compared to both intermediate and coarse media sizes. This means that the selection of media size for different application should include the desired product size. If finer product size is required then smaller media must be used.

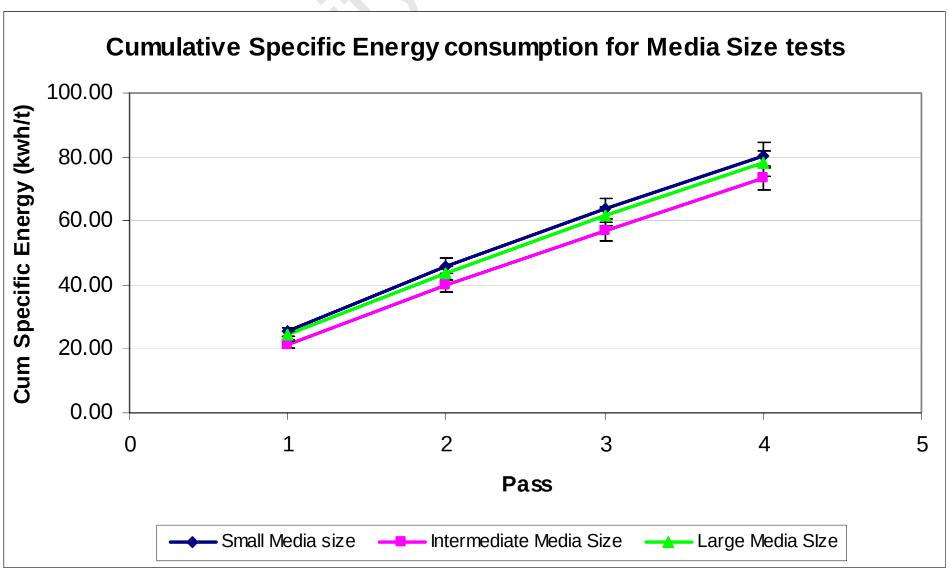

Figure 4-19: Effect of media size on total specific energy consumed.

4.1.3.2 Effect of media size on particle size distributions

Figure 4-20 and Figure 4-21 show respective particle-size distributions obtained during the first and fourth passes for the media size tests. It is observed in Figure 4-20 that similar product particle size distributions (P_{80} =45µm) were obtained during the first passes of the media size tests. However, differences in particle size distributions obtained increased from pass to pass and as indicated in Figure 4-21 the particle size distributions achieve in pass four for the small media is much finer (P_{80} = 15µm) compared to both intermediate and large media sizes (P_{80} = 25µm). Intermediate and large media sizes produced similar particle size distributions (P_{80} = 25µm) during their fourth passes. This implies that though similar energy efficiencies were obtained during the first pass for the media sizes tested, small media appear to be more competent in breaking the UG2 platinum ore as the particle size reduces in the consequent passes compared to both intermediate and large media. This is in agreement with findings of Wang and Forssberg (2000) that smaller media is more efficient in grinding finer materials. However, media size should be matched with the feed particle size for optimum energy utilization. Very small media would tend to give a coarser and broader product size distribution. This is because the media velocities in the tangential direction become low resulting in reduced abrasive energies (Wang & Forssberg, 2000).

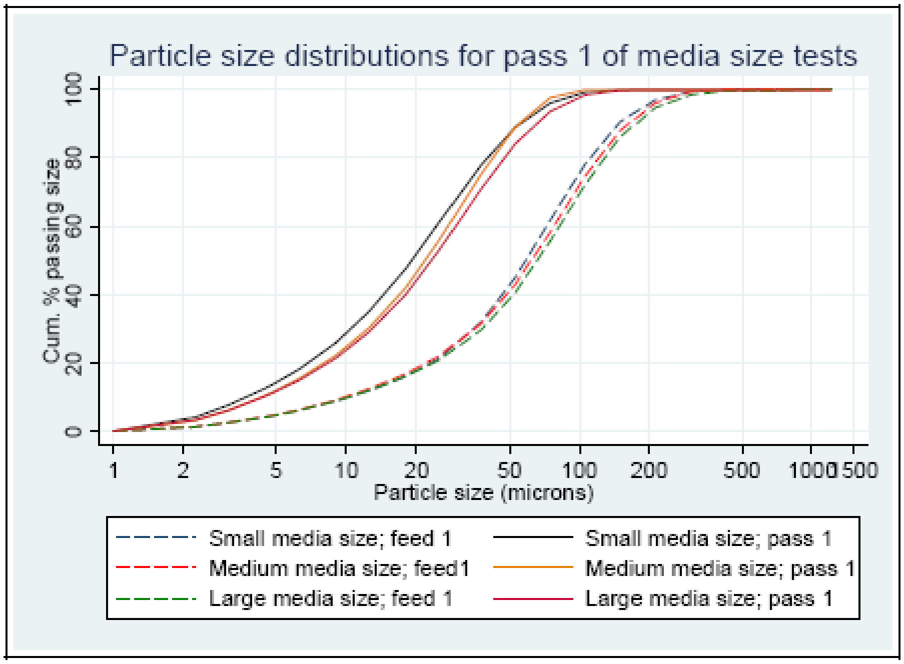

Figure 4-20: Pass 1 particle size distributions obtained for media size tests. Conditions: Large media = 5mm, Intermediate media = 3.5mm, Small media = 2mm.

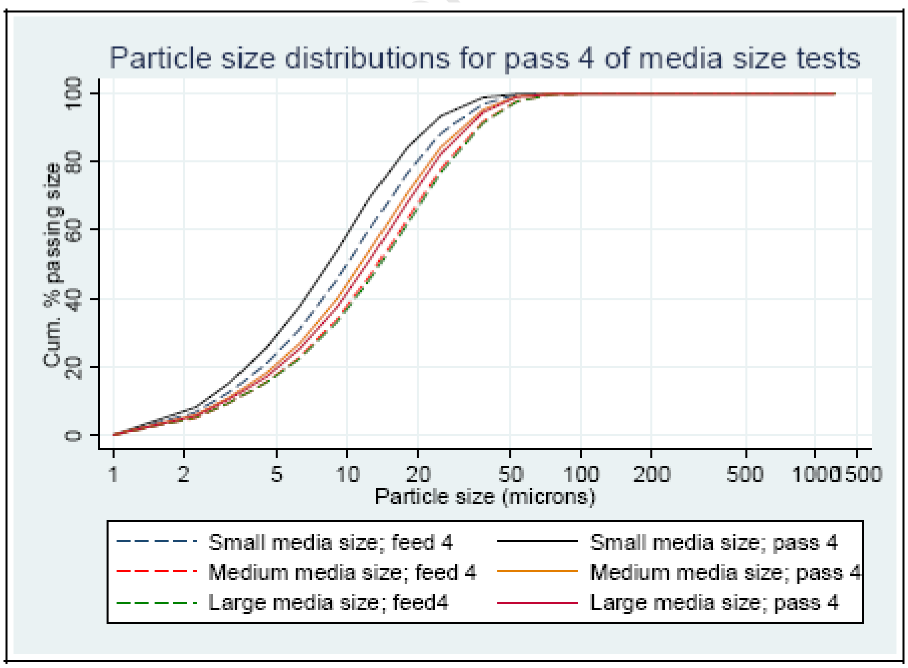

Figure 4-21: Pass 4 particle size distributions obtained for media size tests. Conditions: Large media = 5mm, Intermediate media = 3.5mm, Small media = 2mm.

4.1.3.3 Effect of feed size on relationship between specific energy and reduction ratio

Figure 4-22 shows the relationship between specific energy and reduction ratio obtained when grinding fine (F_{80}=55µm), intermediate (F_{80}=120µm) and coarse (F_{80}=250µm) feed sizes. It was observed that at the same specific energy consumption, the reduction ratios increased with increasing feed sizes. At specific energy consumption of 30kWh/t, the reduction ratios of 1.5, 2.1 and 3.5 were achieved during the comminution of fine, medium and coarse feed respectively. Higher reduction ratios were obtained for coarser feed materials because particle breakage is dependent on the presence of inherent flaws in particles. These flaws are more in larger particles and decreases with decreasing particle size making it more difficult to break the fine particles (Wang & Forssberg, 2007; Donovan, 2003). This means more energy would be required to achieve reduction ratios in finer feeds similar to those obtained with coarser feed. Particle breakage also occurs when energy is supplied by an external force or by the release of stored energy of the new crack surface (Wang & Forsberg, 2007; Donovan, 2003; Kwade, 1999). Since the capacity to store energy decreases with reduced particle size, smaller particles will require more energy by an external force in order for particle breakage to occur.

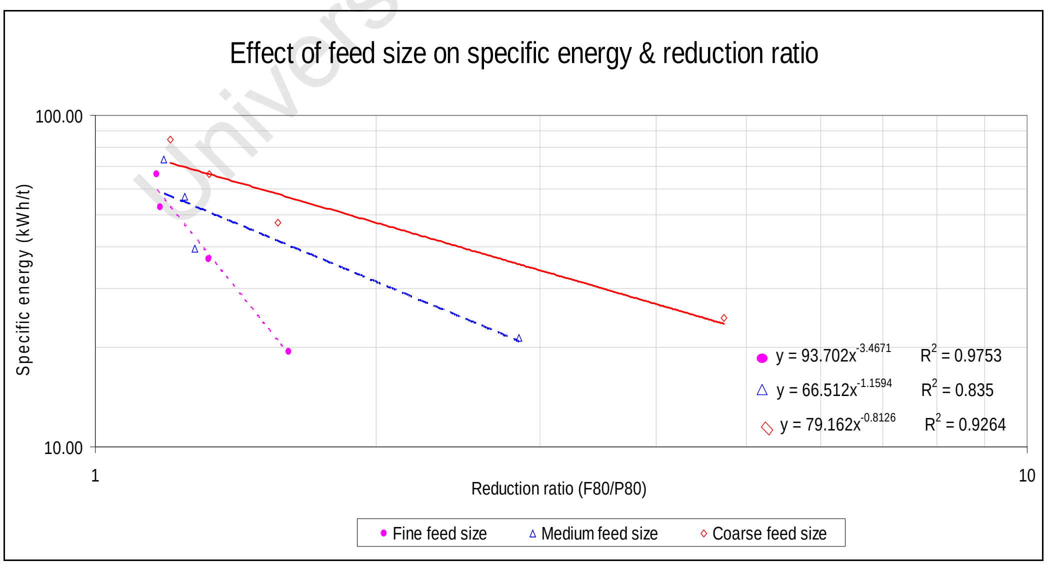

Figure 4-22: Effect of feed size on specific energy and product fineness. Conditions: Fine feed size F_{80} = 55µm, Medium feed size P_{80} = 120µm, Coarse feed size F_{80} = 250µm , media size = 3.5mm

4.1.3.4 Effect of media size on specific energy-particle size distribution relationship

Figure 4-23 shows signature plots obtained from the experiments conducted to assess the influence of media size on IsaMill^{TM} performances when grinding UG2 platinum ore of F_{80} = 120µm. The other operational variables such as stirrer speed, media load, solids concentration and flowrate were kept constant during the media size tests. Stirrer speed was 2100rpm, media load at 65%, solids concentration at 40% and flowrate at 2.5 l/min. It is observed that the specific energy consumption required to grind UG2 ore of F_{80} = 120µm to products with P_{80} between 15µm and 50µm increased with increasing media size. The small media produced the best grinding efficiency followed by the intermediate media whilst the large media appeared to be the least efficient media size to achieve the desired fineness of grind. For coarser products (P_{80} = + 38µm) the intermediate media (3.5mm) appeared to utilize energies similar to those obtained by the small media (2mm). However, the efficiencies of intermediate media drifts from those of small media towards those of the large (5mm) media as the desired product size reduced for P_{80} = 15µm and below. This indicates that though the 3.5mm media can be used interchangeably with the small media when desired product size is coarser (P_{80} = +38µm), it is not efficient for finer product sizes. Similar results were obtained by Becker *et al.,* (2001) and Weller & Gao (1999) indicating higher efficiencies for smaller media in fine grinding. The reduced grinding efficiency for larger media sizes could be attributed to the decreased number of grinding media in the grinding chamber which simultaneously decreases the number of stress events.

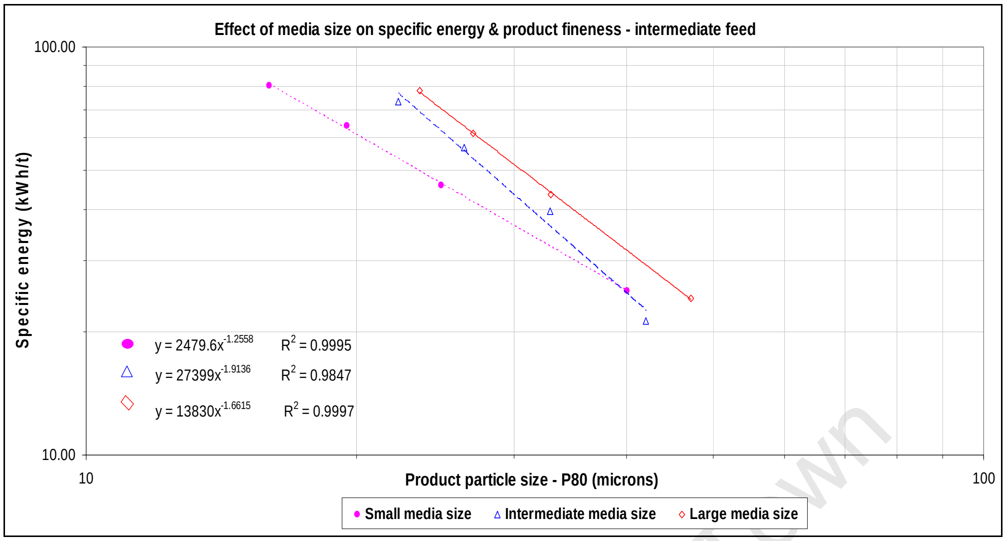

Figure 4-23: Effect of media size on specific energy and product fineness. Conditions: Small media size =2mm, Intermediate media size = 3.5mm, Large media size = 5mm, Feed size P_{80} ~ 120µm

According to Zheng et al. (1996), the media size must be matched with both the feed size and desired product in order to achieved optimum grinding efficiency of stirred mills. To investigate this relationship, media size tests were performed using two extra feed sizes of F_{80} = 250µm and F_{80} = 55µm. Figure 4-24 and Figure 4-25 indicates the signature plots obtained with F_{80} = 250µm and F_{80} = 55µm respectively. It is observed in Figure 4-24 that specific energy consumed to grind UG2 ore with F_{80} = 55µm increased with increasing media size. Best grinding efficiency was achieved with small media while the large media size was the least efficient media when grinding the fine (F_{80} = 55µm) UG2 ore. These results are similar to those obtained for grinding UG2 platinum ore of F_{80} = 120µm presented in Figure 4-23. However, different results were obtained when the feed size was increased to F_{80} = 250µm as shown in Figure 4-25. In this instance, small media (2mm) appeared to be the least efficient while the intermediate media (3.5mm) gave the best energy efficiency when grinding UG2 ore having a F_{80}= 250µm.

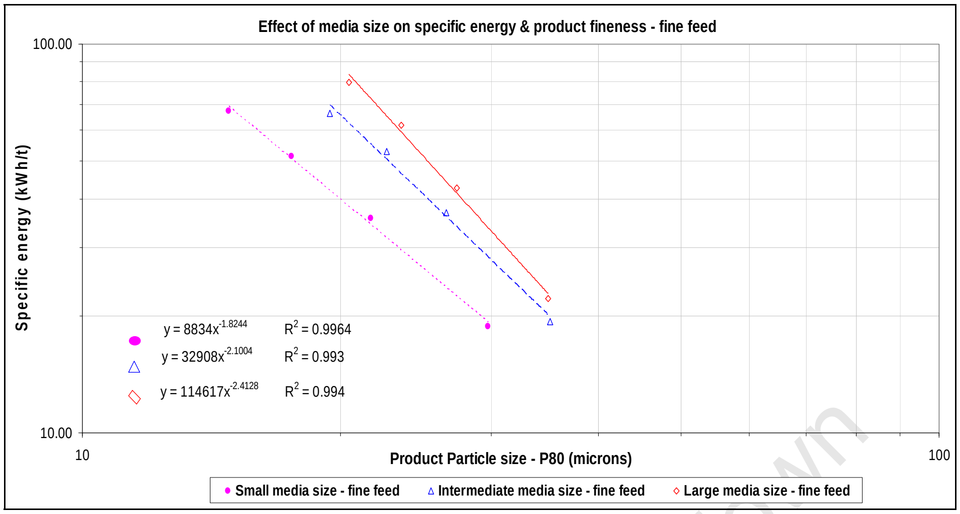

Figure 4-24: Effect of media size on specific energy and product fineness. Conditions: Small media size =2mm, Intermediate media size = 3.5mm, Large media size = 5mm, Feed size F_{80} ~ 55µm

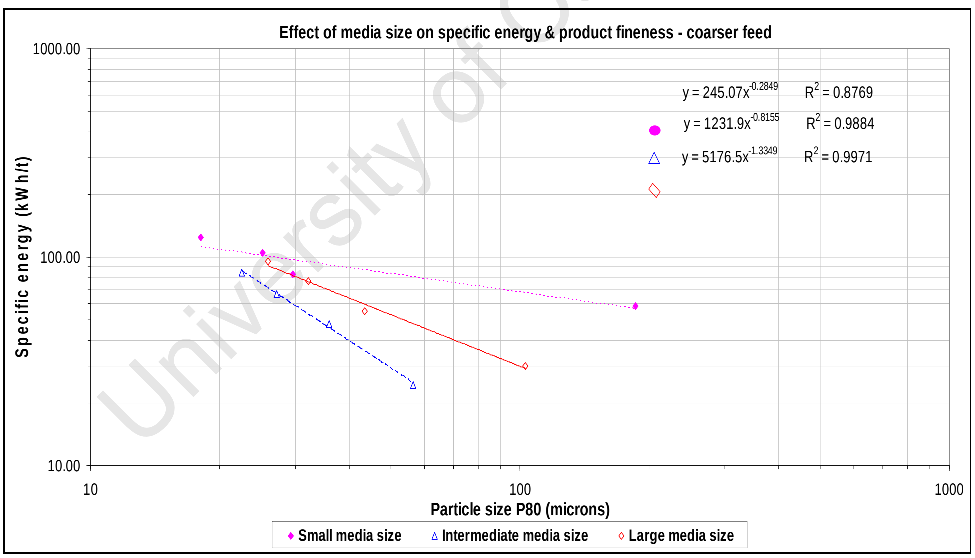

Figure 4-25: Effect of media size on specific energy and product fineness. Conditions: Small media size =2mm, Intermediate media size = 3.5mm, Large media size = 5mm, Feed size F_{80} ~ 250µm

Figure 4-20 to Figure 4-25 indicate that the best grinding efficiency when grinding UG2 ore of feed sizes F_{80} = 55µm and F_{80} = 120µm were achieved when the 2mm media was used. However, the 3.5mm media was the most efficient media to grind the UG2 ore of

F_{80} = 250µm. These results also indicate that the 5mm media was an over-size media to grind UG2 ore of feed F_{80} ranging from 55µm to 250µm as it achieved the least grinding efficiencies. The 2mm media, though quite efficient when grinding fine ore (F_{80} = 55µm) and medium ore (F_{80} = 120µm), it was found to be unsuitable for grinding coarse feed (F_{80} = 250µm). The poor results obtained when grinding coarse feed using small media could be because the applied pressure during the subsequent stress events of such small and light media were insufficient to cause effective particle breakage (Peukert, 2004; Zheng *et al.,* 1996). The velocities of small media in the tangential direction are also low when pressed tightly, which results in a reduction of their abrasive energies. The grinding of coarse ore (F_{80}=250µm) was seen to be more efficient when using the 3.5mm media size. The large media (5mm), though having larger energy for breakage, compared with both intermediate (3.5mm) and small media (2mm) sizes, were not as efficient as the intermediate media size, when grinding coarse feed material. According to Wang and Forssberg (2000), changing the media size may result in changes of the relative proportion of massive impact and attrition events between the media and the particles, and consequently could alter the efficiencies of grinding. Therefore, an appropriate media size needs to be selected for a particular grinding application.

These results illustrate that the IsaMill™ performance is greatly affected by both media size and feed size and that an optimum ratio of media size to feed size exists in stirred mills. Therefore, correct media size should be selected for different IsaMill™ application for optimum utilization of energy in the mill. These findings are similar to those observed by Zheng *et al.,* (1996) and Jankovic (2003) in their separate studies. They concluded that smaller media become more efficient when grinding finer materials. The tendency of smaller media having better grinding efficiency would continue until the media becomes too small to cause particle breakage effectively.

4.1.4 Solids concentration

The concentration of solids is an important factor in wet-grinding operations because of the direct influence on the ground product's fineness and energy consumption. It has been reported that an optimum solids concentration exists for a particular grinding application in stirred mills; and this optimum will depend on the mill design and the required product fineness desired for that application (He *et al.*, 2006; Pease *et al.*, 2006). For this study, experiments were performed to investigate the effect of solids' concentration when grinding UG2 ore of feed size F_{80}=120μm at solids concentrations of 40%, 50% and 60%. The three levels of solids concentration were chosen to cover the operating range of IsaMill^{TM} applications in the processing of Platinum ore.

4.1.4.1 Effect of solids concentration on specific energy-particle size distribution relationship

Figure 4-26 shows the cumulative specific energy consumed at each pass, when treating slurries with three different solids concentrations (40% - 60% solids) with other operating variables held constant for all tests. Stirrer speed was 2100rpm, media size at 3.5mm, media load at 65%, feed with P_{80}=120μm and flowrate of 2.5l/min. It was observed that high slurry solids concentration resulted in lower energy consumption. The total cumulative specific energy consumptions for low (40%), medium (50%) and high (60%) solids concentration were 73.54kWh/t, 64.61kWh/t and 51.91kWh/t respectively. The decrease in specific energy consumption with increased solids concentration can be attributed to changes in slurry rheology (He *et al.*, 2006; Tangsathitkulchai, 2003). At higher solids concentration, the slurry viscosity increases and leads to coating of slurry, media and mill chamber thereby reducing the energy consumption.

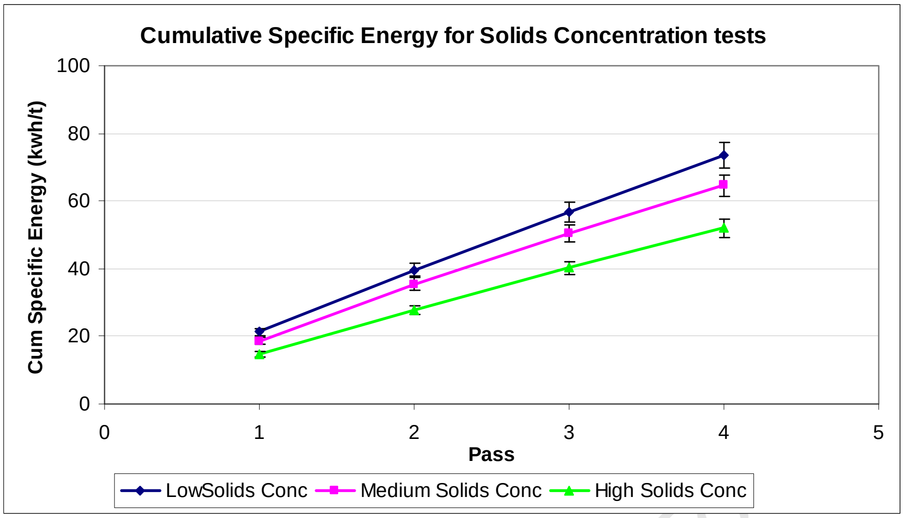

Figure 4-26: Relationship between solids concentration and cumulative specific energy consumption

4.1.4.2 Effect of solids concentration on particle-size distributions and reduction ratios

Figure 4-27 and Figure 4-28 shows the product size distributions obtained when treating UG2 ore of feed F_{80}=120µm at three different solids concentrations (40% - 60% solids) while other operating variables were held constant for all tests. Stirrer speed was 2100rpm, media size at 3.5mm, media load at 65%, feed with P_{80}=120µm and flowrate of 2.5l/min. It was observed that the product fineness increased with reduced solids concentrations -- with product particle sizes of P_{80}=42µm, P_{80}=52µm and P_{80}=64µm obtained in the first pass for 40%, 50% and 60% solids concentration tests. Similar trend was observed when looking at the reduction ratios achieved for tests performed at solids concentrations of 40%, 50% and 60% and presented in Figure 4-29. Figure 4-29 indicates that tests performed at low solids concentration exhibited a higher reduction ratio compared with those conducted at high and medium solids concentrations. A finer product was achieved for low solids concentrations because of the higher viscosity effects encountered with increased solids concentrations (He *et al.*, 2006). According to He *et al.*, (2006), slurry viscosity increases with higher solids concentration. The high viscosity dampens the motion of the grinding media in the mill and reduces the velocity and kinetic energy of the media significantly bringing about lower stress intensities of collisions among media/particle/chamber wall. Therefore, the captured particles cannot be ground effectively, thus causing an ineffectual grinding operation (He *et al.*, 2006; Zheng *et al.*, 1996).

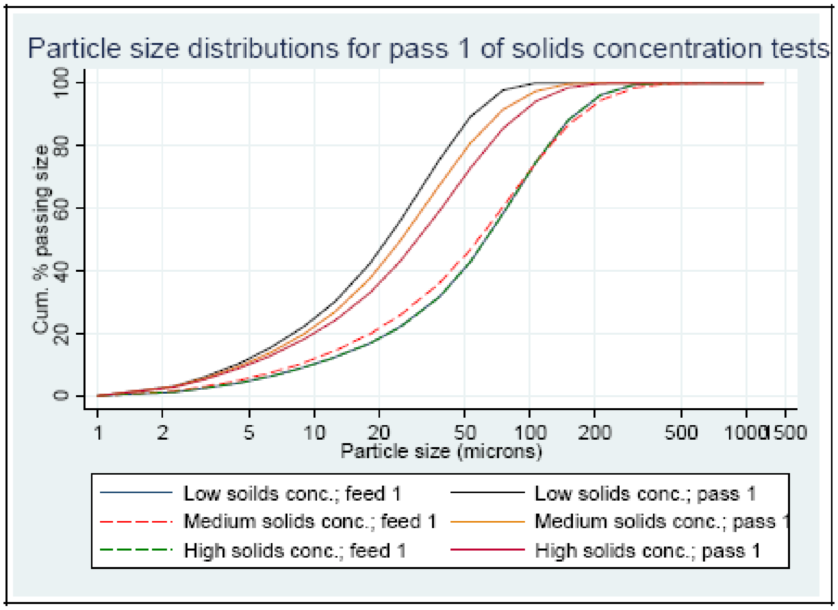

Figure 4-27: Pass-by-pass particle-size distributions obtained for Low solids conc. = 40%, Medium solids conc. = 50%, High solids conc. = 60% tests

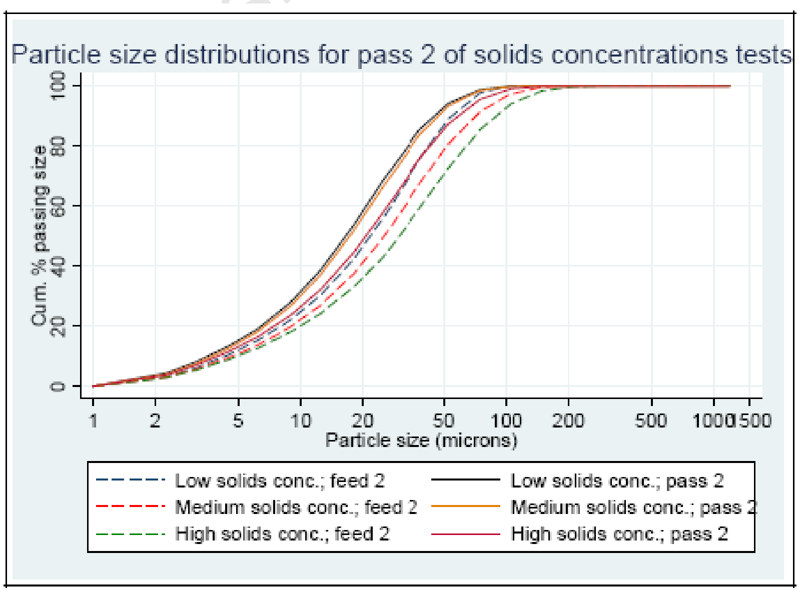

Figure 4-28: Pass-by-pass particle-size distributions obtained for Low solids conc. = 40%, Medium solids conc. = 50%, High solids conc. = 60% tests

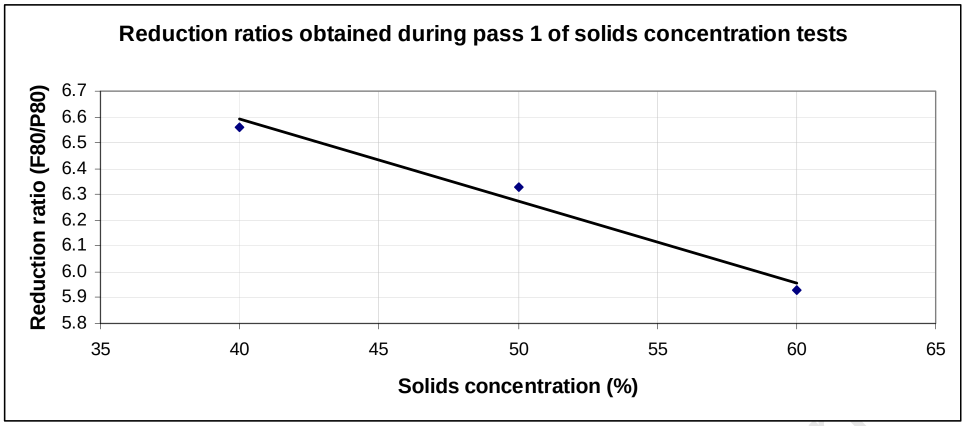

Figure 4-29: Effect of solids concentration on reduction ratio

4.1.4.3 Effect of solids concentration on specific energy particle size distribution relationship

The signature plots obtained from the experiments performed at various solids concentrations (40%, 50% and 60% solids) are presented in Figure 4-30. Other operating variables were kept constant; stirrer speed at 2100rpm, media size at 3.5mm, media load at 65%, feed with P_{80}=120µm and flowrate of 2.5l/min. For specific energy consumption higher than 30kWh/t, the signature plots in Figure 4-30 indicate that there are no significant differences in the grinding efficiency achieved at 40%, 50% and 60% solids concentration. However, when the specific energy consumption was lower than 30kWh/t the grinding efficiencies are seen to be different. For example; for 20kWh/t the product particle sizes obtained at low, medium and high solids concentration were 44µm, 50µm and 53µm respectively. This implies that the grinding efficiency at lower energy levels decreased with increasing solids concentration from 40% to 60%. This could probably be because of the high viscosities that occur at high solids concentration due to fine particle sizes and small media (He *et al.*, 2006; Tangsathitkulchai, 2003).

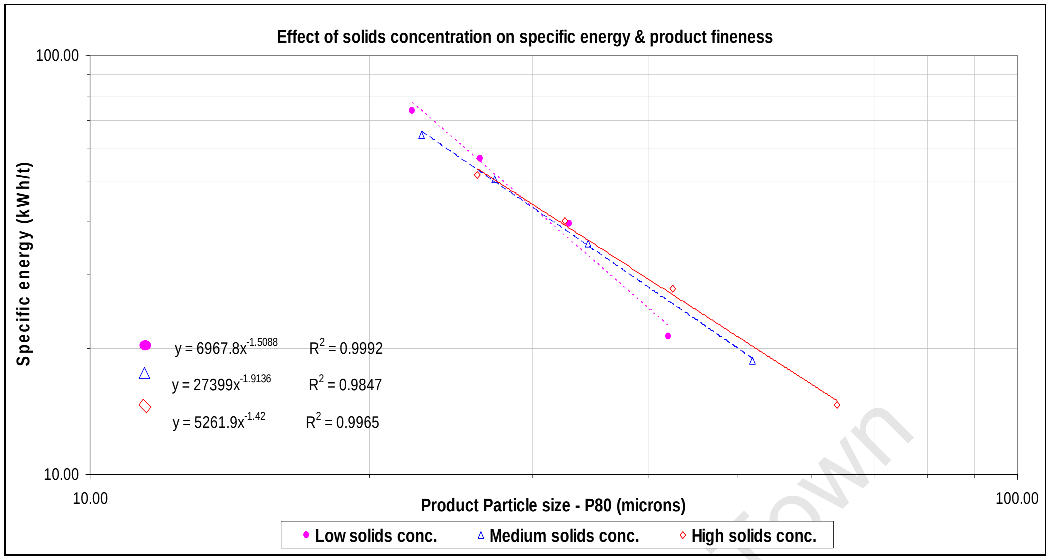

Figure 4-30: Effect of solids' concentration on specific energy and product fineness. Conditions: Low solids conc. = 40%, Medium solids conc. = 50%, High solids conc. = 60%.

Many researchers (Gao *et al.*, 2007; Weller & Gao, 1999; He *et al.*, 2006; Jankovic, 2003; Zheng *et al.*, 1996) have shown that solids concentration has a major influence on slurry viscosity which in turn influences the movement of the grinding media and stress intensity. According to Kwade (1999), the probability of stressing slurry particles at every media contact is increased with higher solids concentration. However, if the slurry viscosity becomes too high at high solids concentration, the motion of the grinding media in the mill would be dampened -- significantly reducing the velocity and kinetic energy of the media (He *et al.*, 2006; Tangsathitkulchai, 2003). This will ultimately reduce the grinding efficiency because the movement between media and particles decreases and parts of the grinding chamber can not be activated by the stirrer anymore (He *et al.*, 2006; Kwade, 1999).

In this project, the grinding efficiency was seen to decrease with increasing solids concentrations from 40 – 60%. This is in line with the work reported by Napier-Munn *et al.*, (1999) which indicated an on set of exponential increase of slurry viscosity as the solids concentration increases above 40% solids. The exponential increase in viscosity as solids concentration increases above 40% can be said to have detrimental effects on motion of charge and, therefore, the grinding efficiency as observed in Figure 4- 30. Thus, the grinding efficiency was seen to reduce with increased solids concentration from 40% to 60% solids. The results compare well with the findings of He *et al.* (2006) and Zheng *et al.* (1996) who suggested that the grinding efficiencies in stirred mills will continue to increase with increase in solids concentration up to solids concentrations of around 65%. They also indicated that efficiencies would tend to decrease beyond solids concentrations of 65%. This project was limited to investigating solids concentrations of up to 60% because of the stirrer used to maintain homogenous feed slurry was small for higher solids concentrations which led to line chockages as particles settled to the bottom of the feed tank.

4.2 M10 000 IsaMill experimental results

The purpose of this section is to present results obtained from the sampling campaigns conducted on industrial scale IsaMill^{TM} circuits and to verify that the performance of the M4 IsaMill^{TM} laboratory scale was comparable with full scale M10 000 IsaMill^{TM}. According to Weller et al., (1999), it is an industrial standard to use the M4 for characterization of industrial IsaMill^{TM} since results obtained in terms of specific energy consumption corresponds very closely to those achieved at industrial scale. Therefore, surveys were conducted on two Anglo platinum Concentrators; Waterval UG2 Concentrator and Western Limb Tailings Re-treatment Plant (WLTRP). The two concentrators were chosen in order to investigate industrial scale (M10 000) IsaMill^{TM} applications in main stream and ultra fine grinding at Waterval Concentrator and WLTRP respectively.

4.2.1 Waterval UG2 Concentrator

Sampling campaign was conducted at Waterval Concentrator with all operating variables such as feedrate, slurry density and mill power set to their basic case scenario. The average settings were feedrate 150 m^3/hr, slurry density 1.45 kg/l and mill power draw 2130 kW. The methodology and flowsheet of the site where the sampling campaign was conducted was discussed in Chapter 3.

A summary of the results obtained from the surveys conducted on IsaMill™ circuit are presented in Table 4-1. It can be seen that the two surveys performed at Waterval Concentrator were conducted under similar conditions and there appear to be no significant difference in the energy consumed and the achieved product fineness.

Table 4-1: Summary of results for Waterval concentrator IsaMill™ sampling campaign

	Feed size F80 (µm)	Product size P80 (µm)	Specific energy (kWh/t)
Survey 1	105	60	30.02
Survey 2	105	63	30.17

Figure 4-31 shows the particle size distributions for the M10 000 IsaMill™ feed and product samples. It can be seen that the IsaMill™ was able to produce a sharp product size distribution. This is a direct result of the effect of eight chambers operating in series, preventing short-circuiting of feed particles to the discharge end and the classification action of the product separator (Pease *et al.*, 2006; Shi *et al.*, 2010). According to Shi *et al.*, (2010), the product separator retains the media while allowing fine product to exit the mill. Any coarse particles and media which enter the product separator region are centrifuged towards the shell. The rotor acts like a centrifugal pump, pumping the liquid back to the feed end of the mill. This compresses the media between the discs, creating consecutive grinding zones between the discs which results in sharp product size distribution in open circuit.

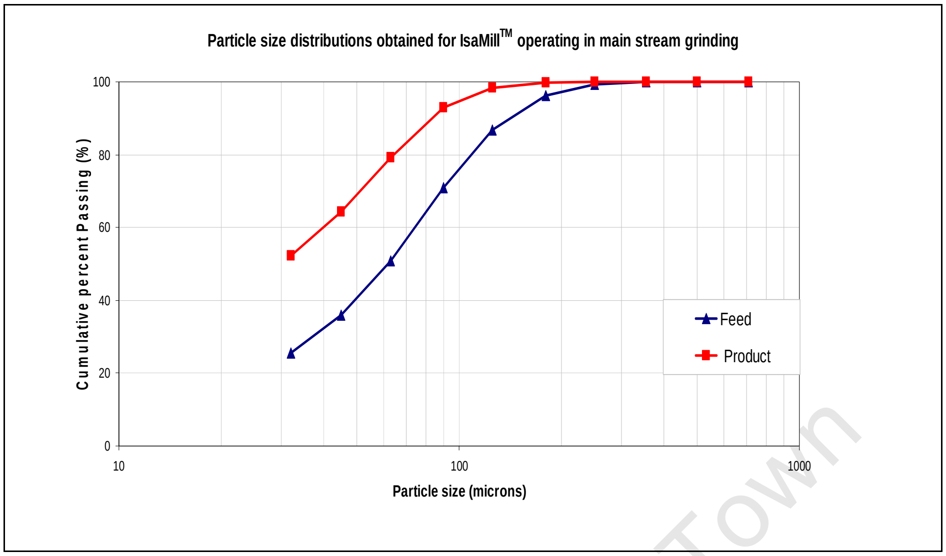

Figure 4-31: Particle-size distributions obtained for M10 000 IsaMill™ at WaterVal UG2 Concentrator operating in main stream grinding application

4.2.2 Western Limb Tailings Re-treatment Plant

Results obtained from the sampling campaigns performed on the IsaMill™ operating at WLTRP are presented in this section. All the operating variables -- except flow-rate -- were set to their basic case scenarios. However, the flowrate was set to different levels for each survey. A summary of the results obtained from these surveys is presented in Table 4-2.

Table 4-2: Results summary for the WLTRP sampling campaign

IsaMill variable flowrate (t/hr)	Feed size F80 (µm)	Product size P80 (µm)	Specific energy (kWh/t)
39	40	25	49
47	40	24	39

Table 4-3 and Table 4-4 provide a summary of the Analysis of Variance (ANOVA) for the energy consumed during the two sampling campaigns conducted at WLTRP. It is observed that the energy utilized for the two sampling campaigns were significantly different as indicated by a low P-value (6.07 X 10^{-35}) and standard deviation = 5.24.

Table 4-3: Summary for energy consumed during the sampling campaigns at WLTRP

SUMMARY

Groups	*Count*	*Sum*	*Average*	*Variance*
Survey 1 - Energy (kWh/t)	31	1518.1	48.97	2.940565
Survey 2 - Energy (kWh/t)	31	1208.40	38.98	1.419904

Table 4-4: Analysis of variance for energy consumed during the sampling campaigns at WLTRP

ANOVA

Source of Variation	*SS*	*df*	*MS*	*F*	*P-value*	*F crit*
Between Groups	1547.30	1	1547.31	709.70	6.07E-35	4.00
Within Groups	130.81	60	2.18			
		SDev	5.24			
Total	1678.12	61				

It is observed from Table 4-2 that the specific energy consumption decreased from 49kWhr/t to 39kWhr/t when the feedrate was increased from 39t/hr to 47t/hr. Results also indicated increased fineness of grind as feedrate increased. These results are similar to observations made by Weller & Gao (1999) during their study of the IsaMill^{TM} performances. According to Weller & Gao (1999) the increase in flowrate influences the residence time of particles in the mill. Higher flowrates have shorter residence times than lower ones. They showed that the pressure in the mill chamber also increases with increasing flowrate leading to an increased rate of particle breakage. In our study at WLTRP, the product sizes achieved for the two surveys do not appear to be significantly different.

An attempt to conduct more surveys and with other variables, was made on the IsaMill^{TM} operating at WLTRP. The other variables investigated were stirrer speed at 254rpm, 242rpm and 238rpm and media load at two levels: high and low. The media load levels were indicated as being high and low levels, because the actual media level could not be measured. A correlation between the media load and power was, therefore, used to indicate the two levels. The basic case operating scenario corresponding to 1.7MW of mill power draw was taken as the high media load, while the low level was obtained by cutting off the media charge to the mill in order to drop the power draw down to 1.5MW. The mill was then maintained and stabilized at this power-draw level for the duration of the low media load tests. The particle size distributions obtained from the samples collected for the stirrer speed and media load tests indicated coarser distributions for the discharge, compared with the feed material. The coarser particle sizes in the product material could have been the abraded silica-sand media material in the product samples. The data obtained from these tests were, therefore, discarded and not used in the analysis of this work.

4.2.3 Comparison of M4 and M10 000 IsaMill™ performance

To verify that the performance of a laboratory IsaMill™ was comparable with the industrial scale mill, the M4 results were plotted on one graph and the data points from the M10 000 surveys were super-imposed on the same graph, as is illustrated in Figure 4-32. It was observed that the specific energy consumption increased with improved product fineness. The super-imposed data points obtained from sampling campaigns at WaterVal Concentrator and WLTRP exhibited a very consistent behavior of industrial scale IsaMill™ (M10 000) that is closely matched by results achieved using the M4. Figure 4-32 indicates that the signature plot could probably be used as a scale up indicator to predict the energy consumed per ton of ground material in an industrial scale IsaMill™. This result is in line with the findings of Weller *et al.,* (1999), who concluded that testing in the M4 IsaMill™ for industrial scale gives a fairly accurate estimate of the specific energy required to prepare a desired product size in an industrial scale mill. Therefore, the M4 IsaMill™ can be used as a tool for generation of data for design and estimating the specific energy needed to process the UG2 platinum ore in an industrial scale IsaMill™ because they appear to scale well to the M10 000.

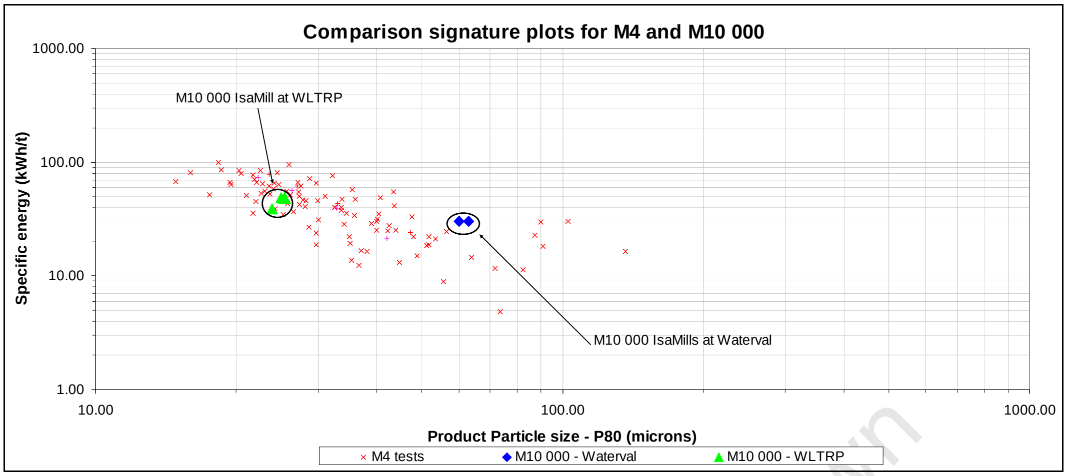

Figure 4-32: Comparison of signature plots obtained from the M4 and the M10 000 sampling campaigns

Figure 4-32 also indicates that the specific energy requirements increased with improved fineness of grind. According to many researchers (Wang & Forssberg, 2007; Curry *et al.*, 2005; Donovan, 2003; Lichter & Davey, 2002; Weller *et al.*, 1999), larger particles have many flaws that are broken down during breakage into smaller fragments. A reduced number of flaws in smaller particles lead to less chances of particle breakage. Therefore, an increase in energy input is necessary to raise the number of events whereby the smaller particles may be broken. This is achieved by increasing the stirrer speed and/or using smaller media in order to increase the grinding surfaces. (Jankovic, 2003; Pease *et al.*, 2006). The results obtained in this thesis indicate that the M4 IsaMill^{TM} can be used to characterize industrial IsaMill^{TM} at design or process optimization of already installed mills. This can be achieved by optimizing process variables such as media and solids concentration for given ore using M4 and transfer the learning to industrial scale mills.

5.0 Correlations of parameter effects on specific energy and product fineness

The results presented in chapter 4 indicated a correlation of IsaMill^{TM} operating variables to the fineness of grind and specific energy consumption. This relationship can be used to develop an empirical model for the prediction of IsaMill^{TM} grinding efficiency when changes in operating variables are made. Multiple regressions were performed to investigate the significance of the correlations of operating variables and grinding efficiency observed in chapter 4. All the variables investigated in this project were included in the first set of regressions. The parameters with negligible contribution were then removed, one by one, in order to check for any double-counting of the effects. The correlation equation was only selected when all the remaining variables appeared to have a significant effect.

5.1 Specific energy

The specific energy consumption in a stirred mill is affected by many design and operating variables (Jankovic, 2003). The effects of stirrer speed, media load, media size, feed size, solids' concentration and flowrate were investigated in this project. All the variables investigated were included in the regressions, in order to obtain the expression for the specific energy consumption.

Table 5-1 and Table 5-2 provide a summary of the regression statistics and the Analysis of Variance (ANOVA) obtained for the specific energy expression respectively. Both tables appear to indicate a good regression for the specific energy as indicated by a high R squared adjusted (0.77) and small significant F value (1.78493E-07) which is less than 0.05. This means that 77% of the variations observed in specific energy consumption during the experiments can be explained by changes in operational variables. Since significant F (1.78493E-07) is less than 0.05 it can be said with 95% confidence level that a relationship between operational variables (Media load, Stirrer speed, Solids flowrate and feed size) and specific energy consumption exists.

Table 5-1: Summary of specific energy regression statistics

Regression Statistics	
Multiple R	0.895636
R Square	0.802165
Adjusted R Square	0.766195
Standard Error	10.84029
Observations	27

Table 5-2: Analysis of variance (ANOVA) results obtained for specific energy regressions

ANOVA					
	df	*SS*	*MS*	*F*	*Significance F*
Regression	4	10482.49	2620.624	22.3009	1.78493E-07
Residual	22	2585.264	117.512		
Total	26	13067.76			

Figure 5-1 shows the correlation between the observed and the predicted specific energy after removing all the variables that had an insignificant contribution. There appears to be good agreement between the observed and the predicted values of specific energy consumption.

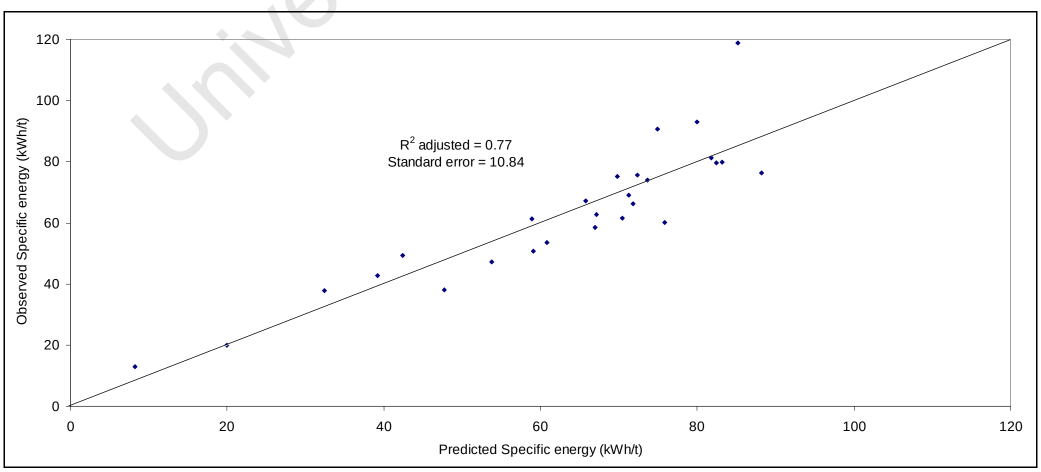

Figure 5-1: Correlation of observed and predicted specific energy for the M4 IsaMill™ tests

Table 5-3 shows a summary of the obtained regression coefficients, standard errors and P-values. A p-value cut off point of 0.05 was used in analyzing the results for this project. This means that a P-value less than 0.05 imply that there is a 5% chance that the relationship emerged randomly and a 95% chance that the relationship is real. As observed in Table 5-3, P-values of less than 0.05 were obtained for all regressed variables. It can, therefore, be said that there is a 95% chance that the regressed variables (media load, stirrer speed, solids flowrate and feed size) significantly influenced the specific energy consumption based on their small P-values which are less than 0.05.

Table 5-3: Summary of regression coefficients, standard errors and P-values for specific energy the expression

	Coefficients	*Standard Error*	*P-value*
Intercept	-124.4779226	33.9047311	0.0013396
Media load	0.7260932	0.2764942	0.0154268
Stirrer speed	0.0839043	0.0127597	0.0000013
Solids flowrate	-23.6552503	4.7642070	0.0000573
Feed size	0.0616241	0.0303008	0.0542116

$$\begin{aligned} Specific\ energy = -124.48 + 0.08 * Stirrer\ \ speed + 0.73 * Media\ \ load \\ +0.06 * Feed\ \ size - 23.65 * Solids\ \ flowrate \end{aligned}$$ **Equation 5-1**

Equation 5-1 was selected, which combines only the variables that had a significant effect and represented the specific energy better than the other equations tested. The variables with significant influence on specific energy for the range of variables tested are: stirrer speed, media load, feed size and flowrate. The other operational variables investigated such as media size, solids concentration -- for the range tested -- did not appear to have significant effects on the specific energy consumption.

5.2 Product fineness

According to many workers (Jankovic, 2003; Pease *et al.*, 2006; Weller & Gao, 1999; Zheng *et al.*, 1996) the product fineness is largely influenced by the mill design, and the media and slurry properties. In this project, the influences of stirrer speed, media load, media size, feed size, solids' concentration and flowrate were investigated and included in the regressions -- to obtain the expression for product fineness.

Figure 5-2 shows the correlation between the observed and the predicted product fineness. There appears to be good agreement between the observed and the predicted values of product fineness.

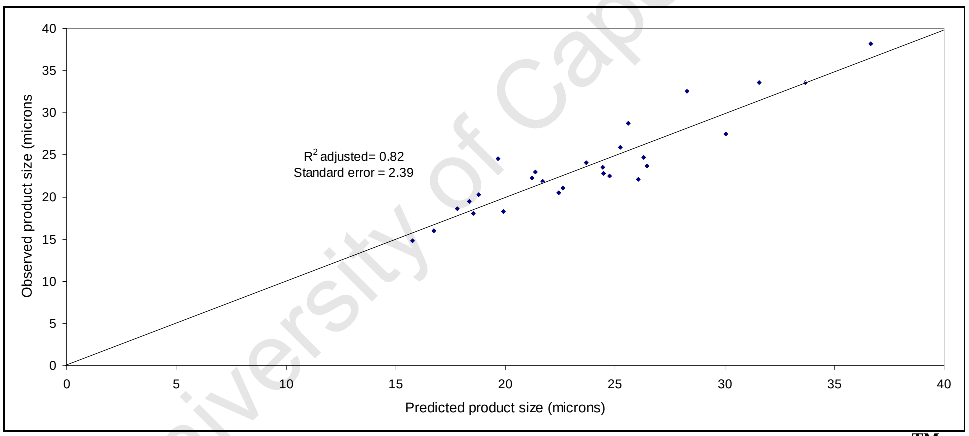

Figure 5-2: Correlation of observed and predicted product sizes for the M4 IsaMill^{TM} tests

Table 5-4, Table 5-5 and

The summary of regression coefficients, standard error and P-values are presented in Table 5-6. As observed in Table 5-6, P-values of less than 0.05 were obtained for all regressed variables. This means that there is a 95% chance that the relationship between operational variables (media size, media load, stirrer speed, solids flowrate and feed size) and product fineness statistically significant.

Table 5-6 shows summaries of regression statistics, an analysis of the variance and regression expression coefficients, respectively. The results indicate that the effects of media size, media load, stirrer speed, solids flowrate and feed size are statistically significant at 95% confidence level. As observed in Table 5-4, the adjusted R squared = 0.817, which suggests that 82% of the variance in the observed product fineness are explained by the changes made in the regressed operational variables.

Table 5-4: Summary of product fineness regression statistics

Regression Statistics	
Multiple R	0.923336
R Square	0.852549
Adjusted R Square	0.817442
Standard Error	2.386733
Observations	27

The Analysis of Variance (ANOVA) obtained for the product fineness expression and presented in Table 5-5 also appear to indicate a good regression of operational variables and product fineness based on small significant F value (4.5478E-08) which is less than 0.05. It can, therefore, be said with 95% confidence that the relationship between operational variables (Media load, Stirrer speed, Solids flowrate and feed size) and product fineness is statistically significant.

Table 5-5: Analysis of variance (ANOVA) results obtained for product fineness regressions

ANOVA					
	df	*SS*	*MS*	*F*	*Significance F*
Regression	5	691.671	138.3342	24.28409	4.54779E-08
Residual	21	119.6264	5.696494		
Total	26	811.2974			

The summary of regression coefficients, standard error and P-values are presented in Table 5-6. As observed in Table 5-6, P-values of less than 0.05 were obtained for all regressed variables. This means that there is a 95% chance that the relationship between operational variables (media size, media load, stirrer speed, solids flowrate and feed size) and product fineness statistically significant.

Table 5-6: Summary of regression coefficients, standard errors and P-values for product fineness the expression

	Coefficients	Standard Error	P-value
Intercept	47.43	8.41	1.34E-05
Solids conc.	0.38	0.07	1.7E-05
Media size	2.42	0.65	0.00135
Media load	-0.20	0.06	0.004044
Stirrer speed	-0.02	0.00	7.89E-07
Feed size	0.03	0.01	0.000162

Equation 5-2 was selected, which combines only the variables that had a significant effect and represented the product fineness better than the other equations tested. It appears that stirrer speed, media load, media size, feed size and flow-rate significantly influenced the product fineness.

$$\begin{aligned} \Pr oduct\ \ size &= 47.43 - 0.02 * Stirrer\ \ speed + 2.42 * Media\ \ size - \\ & 0.20 * Media\ \ load + 0.03 * Feed\ \ size + 0.38 * Solids\ \ conc. \end{aligned}$$ **Equation 5-2**

6.0 Observations, conclusions and recommendations

The significant conclusions of the study are summarised, with recommendations for further work discussed.

This study investigated the influence of operating variables on the operations of the IsaMill^{TM} in terms of energy utilization and product fineness. This was done using the M4 IsaMill^{TM} where a range of stirrer speeds, media sizes, media loads, feed sizes, solids' concentrations and flowrates tested. Sampling campaigns were also performed on M10 000 IsaMill^{TM} to investigate the IsaMill^{TM} operations at industrial scale.

The results obtained during the study have demonstrated that for the range of variables investigated; stirrer speed, media load and feed size significantly influence the IsaMill^{TM} specific energy and product fineness.

6.1 Observations

- This test work has shown that when the IsaMill^{TM} mill is operated at different speeds the energy required to grind UG2 ore of feed F_{80} = 120µm to a given product size (P_{80}) varies. For the same feed and media load the energy utilized for a product P_{80} = 38µm at stirrer speeds of 1500, 1800, and 2100 rpm were 30, 22, 26 kWh/t. This shows that at lower speeds (1500 rpm) more energy is required for given product size, increasing the speed to 1800 rpm resulted in a decrease in energy and further increase led to higher energy utilization. This indicates an existence of optimum speed when grinding UG2 ore of F_{80} = 120µm at 1800 rpm.

- In terms of the effect of media load, this project found that different media loads are required to efficiently grind various feed sizes (F_{80}) of UG2 ore to desired product. When grinding feed F_{80} = 250µm to product P_{80} = 38µm, the energy consumption reduced from 50 to 41 kWh/t with increasing media load from 50% to 65% but increased to 54 kWh/t as media load was increased from 65% to 80%. This implies that there is an optimum media load when grinding feed size F_{80} =

250µm which is close to 65% load. However, no significant differences in energy consumption was observed when grinding feed size F_{80} = 120µm where energies of 29, 27 and 29 kWh/t were consumed for media loads of 50%, 65% and 80% respectively. The energy consumed to grind feed F_{80} = 55µm to product P_{80} = 38µm was seen to increase from 18 to 30 kWh/t with increased media load from 50% to 80%. These results indicate that the IsaMill™ media load should be optimized at different levels for an efficient grinding process which depends on feed particle size.

- This test work also indicates that the performance of the IsaMill™ is greatly affected by the media size and feed particle size distribution. For the same stirrer speed, media load and slurry percent solids, it was found that the best grinding efficiency when grinding UG2 ore of feed sizes F_{80} = 55µm and F_{80} = 120µm were achieved when the 2mm media was used. However, the 3.5mm media was the most efficient media to grind the UG2 ore of F_{80} = 250µm. This implies that the ratio of media to feed size must be matched and optimized in order to maximize the grinding efficiencies.

- Comparison of the data obtained from the sampling campaigns conducted on M10000 and the M4 IsaMill™ test work exhibited a consistent behaviour of industrial IsaMill™ that is closely matched by results achieved using the M4 mill. This entails that the M4 laboratory scale IsaMill™ can be utilized to accurately estimate the energy required to prepare desired product fineness in M10000 industrial scale IsaMill™. Therefore, the M4 can be used to generate data for design and operations optimization of industrial scale IsaMill™.

- Regression analysis that was performed to investigate the influence of operating variables on specific energy consumption and product fineness indicated that:

- Specific energy was significantly influenced by stirrer speed, media load, feed size and solids' flowrate**.**
- Stirrer speed, media size, media load, feed size and solids' concentration were seen to significantly affect the product fineness.

6.2 Conclusions

In addressing the hypotheses for this thesis the following conclusions were drawn:

- The efficiency of grinding UG2 ore of feed F_{80} = 120µm varied for different stirrer speeds (1500rpm, 1800rpm and 2100rpm) while maintaining other operational variables such as media size, media load, slurry percent solids and flowrate constant. Stirrer speed of 1800 rpm was found to be optimum when grinding UG2 ore of feed F_{80}=120µm with energy consumption of 22kWh/t to achieve P_{80} = 38µm whereas energies of 30 and 26 kWh/t were consumed for similar product size at 1500 and 2100 rpm respectively.
- This study indicates that different media loads are required to efficiently grind various feed sizes (F_{80}) to desired product.
  - For grinding UG2 ore of feed F_{80}=250µm to product size P_{80} = 38µm, results indicate that the energy consumption reduced from 50 to 41 kWh/t with increasing media load from 50% to 65% but increased to 54 kWh/t as media load was increased from 65% to 80%. This suggests that there is an optimum media load when grinding feed size F_{80} = 250µm which is close to 65% load.
  - The energy consumed when grinding UG2 ore of F_{80} = 55µm to product size P_{80} = 38µm was seen to increase from 18 to 30 kWh/t with increased media load from 50% to 80%. This implies that higher energy is required when media load is increased from 50 to 80% for grinding UG2 ore of feed F_{80} = 55µm to product P_{80} = 38µm.

- This test work shows that selection of media size for optimum energy utilization during the grinding process in IsaMill™ depends on particle size distribution of feed material.
    - It was found that the lowest energy consumption when grinding UG2 ore of feed sizes F_{80} = 55µm and F_{80} = 120µm to product size of P_{80} = 30µm were achieved when the 2mm media was used; 19 and 37 kWh/t for F_{80} = 55µm and F_{80} = 120µm feed sizes respectively.
    - When grinding the UG2 ore of F_{80} = 250µm to product P_{80} = 30µm, the 3.5mm media was found to be the most efficient media size; consuming the lowest energy (60 kWh/t) compared to 98 and 80 kWh/t utilized for tests conducted with 2mm and 5mm media size respectively.

6.3 Recommendations for further work

Based on the extent to which the objectives of this research work have been fulfilled, the following recommendations, with respect to future research on the IsaMill™ application in the platinum industry are made:

- It is proposed that further studies be conducted to investigate the interactions between operational variables which were not considered in this work. According to Jankovic (2003), the interactions between variables can be very strong and therefore, the effect of one parameter can not be seen in isolation.

- It is proposed that the investigation of effect of coarser feed size material (P_{80}> 120µm) and media size on IsaMill™ performance be extended on pilot and industrial scale mills to consolidate data obtained in this work for IsaMill™ mainstream grinding applications.

- Mineralogical analysis should be performed on the feed and product samples in order to obtain information on liberation and mineral deportment. This information could be helpful in establishing the optimum grind required to achieve optimum liberation of PGMs, and thereby avoiding over-grinding.

References

Austin, L.G. 1990. *Ball Mills, Semi-Autogenous Mills and High-Pressure grinding Rolls.* The Pennsylvania State University

Austin, L.G., Klimpel, R.R. & Luckie, P.T. 1984. *Process Engineering of size reduction Ball milling.* Society of Mining Engineers.

Austin, L.G. & Shah, I. 1983. *A method for inter convension of microtrac and sieve size distributions*.

Becker, M.; Kwade, A. & Schwedes, J. 2001. *Stress intensity in stirred media mills and its effect on specific energy requirement.* International Journal of Mineral Processing, 61:189-208.

Bedesi, N.; Rule, C.M. & Knopjes, L. 2008. *Anglo Platinum's development of a ceramic grinding media selection protocol for MIG and UFG Isa milling.* Minerals Engineering, Comminution 08 Conference

Blecher, L.; Kwade, A. & Schwedes, J. 1996. *Motion and Stress Intensity of Grinding Beads in a Stirred Media Mill.Energy Density Distribution and Motion of Single Grinding Beads.* Powder Technology, 86(1):59-68

Burford, B.D. & Clark, L.W. 2007. *IsaMill Technology Used in Efficient Grinding Circuits.* [Homepage of Xstrata], [Online]. Available: www.isamill.com [2008, February/06].

Clark, L.W. & Burford, B.D. 2004. *Fine grinding and project enhancement.* [Homepage of Xstrata], [Online]. Available: www.isamill.com [2008, 02/06].

Clearly, P. W. 2001. *Charge behaviour and power consumption in ball mills: sensitivity to mill operating conditions, liner geometry and charge composition.* International Journal of Mineral processing, 63:79 – 114

Curry, D.C. & Clermont, B. 2006. *Improving the efficiency of fine grinding – Developments in ceramic media technology*. [Homepage of Xstrata], [Online]. Available: www.isamill.com [2008, February/06].

Curry, D.C.; Clark, L. W. and Rule, C. 2005. *Collaborative technology development – Design and operation of the world's largest stirred mill.* [Homepage of Xstrata], [Online]. Available: www.isamill.com [2008, February/06].

Datta, A.; Rajamani, R.K. 2001. *A direct approach of modeling batch grinding mills using population balance principles and impact energy distribution.* International Journal of Mineral processing, 64:181-200.

Donovan, J.G. 2003. *Fracture toughness based models for the prediction of power consumption, product size and capacity of Jaw Crushers*. PhD Thesis, Virginia Polytechnic Institute and State University.

Ellis, S. 2004. *Ultra-Fine Grinding - A Practical Alternative to Oxidative Treatment of Refractory Gold Ores.* [Homepage of Xstrata], [Online]. Available: www.isamill.com [2008, February/06]

Fourie, N.K. 2008. *Maximising value through IsaMill grinding technology*. Smart ideas Conference, Anglo Platinum

Fuerstenau, D.W. & Abouzeid, A.Z.M. 2002. *The energy efficiency of ball milling in comminution.* International Journal of Mineral processing, 67:161-185.

Gao, M., Holmes, R. & Pease, J. 2007. Plenary lecture: *The latest developments in fine grinding and ultrafine grinding technologies*

Gao, M.; Young, M. & Allum, P. 2002. *IsaMill fine grinding technology and its industrial application at Mount Isa Mines.* [Homepage of Xstrata], [Online]. Available: www.isamill.com [2008, 02/06]

Gao, M.W.; Forssberg, K.S.E. & Weller, K.R. 1996. *Power predictions for a pilot scale stirred ball mill.* International Journal for Mineral processing, 44(45):641-652

Gao, M. and Forssberg, E. 1994. *Prediction of product size distributions for a stirred ball mill.* Powder Technology 84 (2), 101–106

Harbort, G.; Murphy, A.; Vargas, A. and Young, M. 1999. The introduction of the IsaMill for ultra-fine grinding in the Mt Isa Lead/Zinc Concentrator

Hassibi, M.; Rodgers, K. J. & Yang, M. 1999. Advances in fine grinding and mill system application in the FGD industry. Technical paper, CHEMCO Equipment Company

He, M; Wang Y & Forssberg E, 2006. *Parameter effects on wet ultrafine grinding of limestone through slurry rheology in a stirred media mill.* Powder Technology, 161:10-21.

Hennart, S.L.A.; Wildeboer, W.J.; Van Hee, P. & Meesters, G.M.H. 2009. *Identification of the grinding mechanisms and their origin in a stirred ball mill using population balances.* Chemical Engineering Science 4123 – 4130.

Hogg, R. 1999. *Breakage mechanism and mill performance in ultrafine grinding.* Powder Technology, 105:135-140

Jain, S.K. 2001. *Mineral Processing.* 2nd ed.India: CBS Publishers & Distributors.

Jankovic 2001. *Media stress intensity analysis for vertical stirred mills.* Minerals Engineering, 14(10):1177-1186

Jankovic, A. 2003. Variables affecting the fine grinding of minerals using stirred mills. Minerals Engineering, 16:337-345

Jankovic, A.; Dundar, H. & Mehta, R. 2008. Relationships between comminution energy and product size for a magnetite ore. Minerals Engineering, Comminution 08 Conference

Jayasundara, C.T.; Yang, R.Y.; Guo, B.Y.; Yu, A.B.; Govengre, I.; Mainza, A. Van der Westhuizen, A.; Rubenstein; J. 2010. *CFD-DEM modelling of particle flow in IsaMills – Comparison between simulations and PEPT measurements.* Minerals Engineering.

Kwade, A. 2004. *Mill selection and optimization using a physical grinding model.* International Journal of Mineral processing, 74S:S93-S101

Kwade, A.; Schwedes, J. & Joerg, 2002. *Breaking characteristics of different materials and their effect on stress intensity and stress number in stirred media mills.* Powder Technology, 122:109-121

Kwade, A. 1999. *Wet comminution in stirred media mills — research and its practical application.* Powder Technology, 105:14-20

Lane, G.L. 1999, *CFD modelling of a stirred bead mill for fine grinding.* Second international conference on CFD in minerals and process industries

Lee, C.A. 1996, *A review of Mineralization in the Bushveld Complex and some other Layered Intrusions,* Elsevier Science, 103-145.

Lichter, J. K. H. & Davey, G. 2002, *Selection and sizing of ultrafine and stirred grinding mills, Mineral Processing Plant Design, Practice and Control.* Proceedings, 1

Lofthouse, C.H. & Johns, F.E. 1999. *The Svedala (ECC International) Detritor and the metals industry*. Minerals Engineering, 12(2):205-217

Mainza, N. A. 2006. *Contribution to the understanding of the three-product cyclone on the classification of a dual density platinum ore.* PhD Thesis, University of Cape Town

Morrel, S. 2007. *A method for predicting the specific energy requirement of comminution circuits and assessing their energy-utilisation efficiency.* Minerals Engineering, 21:224-233

Napier-Munn, T.J. and others. 1999. *Mineral comminution circuits, their operation and optimization.* 1st ed.JKMRC.

Neesse, T.; Schaaff, F. & Tiefel, H. 2004. *High-performance attrition in stirred mills.* Minerals Engineering, 17:1163-1167

Partyka, T. and Yan, D. 2007. *Fine grinding in a horizontal ball mill.* Minerals Engineering 20: 320 - 326

Pease, J. 2007. *Increasing the Energy Efficiency of Processing.* [Homepage of Xstrata], [Online]. Available: www.isamill.com [2008, 02/06]

Pease, J. 2007. Case Study - Coarse IsaMilling at McArthur River. [Homepage of Xstrata] [Online]. Available: www.isamill.com [2008, 02/06]

Pease, J.; Young, M. & Curry, D.C. 2005. Fine Grinding as Enabling Technology – The IsaMill. [Homepage of Xstrata], [Online]. Available: www.isamill.com [2008, 02/06]

Pease, J.; Curry, D.C.; Barns, K.E.; Young, M.F.; Rule, C. 2006. Transforming flow-sheet design with inert grinding – the IsaMill. [Homepage of Xstrata], [Online]. Available: www.isamill.com [2008, 02/06]

Peukert, W. 2004. *General concepts in nanoparticle technology and thie possible implication on cultural science and philosophy.* Powder Technology 158: 133 –140

Powell, M.; Mainza, A.N. 2002. *Potgietesrust Platinum Limited AG mill survey for the development of the multi-component AG model.* UCT – JKTech reports

Redner, S. 1990. *Statistical model for the fracture disordered media.* Hulin, J.P. (Ed.), Fragmentation. Elsevier, London, pp. 321–328.

Rodrigues, E.M.; Matias, A.; Godefroid, L.B.; Bastian, F.L.; Al-Rubaie, K.S. 2005. *Fatigue crack growth resistance and crack closure behavior in two aluminum alloys for aeronautical applications.* Mat. Res. vol.8 no.3

Rule, C.M.; Knopjes, L.; Clermont, B. and Philippe, C. 2008. *Fine grinding – Developments in ceramic media technology and resulting improved plant performance at Anglo Platinum*

Shi, F.; Morrison, R.; Cervellin, A.; Burns, F. and Musa, F. 2009. *Comparison of energy between ball mills and stirred mills in coarse grinding.* Proceedings of Comminution 08 Conference.

Stender, H.H.; Kwade, A. & Schwedes, J. 2004. *Stress energy distribution in different stirred media-mill geometries.* International Journal of Mineral processing, 74S:S103-S117

Tangsathitkulchai, C. 2003. *Effects of slurry concentration and powder filling on the net mill power of a laboratory ball mill.* Powder Technology, 137: 131 - 138

Tromans, D. 2007. *Mineral comminution: Energy efficiency considerations.* Minerals Engineering

Tuzun, M.A.; Loveday, B.K.; Hinde, A.L. 1994. *Effect of pin tip velocity, ball density and ball size on grinding kinetics in a stirred ball mill.* International Journal of Mineral processing, 43:171-191

Varinot, C.; Berthiaux, H. & Dodds, J. 1999. *Prediction of the product-size distribution in associations of stirred bead mills.* Powder Technology, 105:228-236

Wang, Y. & Forssberg, E. 2000. *Technical note – Product-size distribution in stirred media mills.* Minerals Engineering, 13(4):459-465

Wang, Y. & Forssberg, E. 2007. *Enhancement of energy efficiency for mechanical production of fine and ultra-fine particles in comminution*. China particuology, 5:193-201.

Weller, K.R., Morrell, S. and Gottlieb, P., 1996. *Use of grinding and liberation models to simulate tower mill circuit performance in a lead/zinc concentrator to increase flotation recovery* International Journal of Mineral Processing, 44-45, pp. 683-702.

Weller, K.R. & Gao, M. 1999. *Ultra-fine grinding.* AJM Crushing and Grinding Conference, Kalgoorlie

Weller, K.R.; Gao, M. & Bowen, P. 1999. *Scaling up horizontal stirred mills from a 4-litre test mill to a 4000-litre "IsaMill".* Powder Technology Symposium, Pennsylvania State University

Wills, B.A. 1997. *Mineral processing Technology.* 6th ed.Britain: Butterworth - Heinemann

Wills, B.A. & Napier-Munn, 2006. *Mineral-processing Technology. 7th ed.* Elsevier Science and Technology Books

Yang, R.Y.; Jayasundara, C.T.; Yu, A.B.; Curry, D. 2006. *DEM simulation of the grinding media in IsaMill*. Minerals Engineering, 19:984-994

Yue, J. 1983. *Rheological effects on ultra-fine grinding in stirred mills.* MSc. Thesis. University of British Columbia

Yue, J. & Klein, B. 2004. *Particle breakage kinetics in horizontal stirred mills.* Minerals Engineering, 18:325-331

Zheng, J.; Harris, C.C. & Somasundaran, P. 1996. *A study on grinding and energy input in stirred media mills.* Powder Technology, 86:171-178.

Appendices

Appendix 1: M4 IsaMill test matrix

	Feed Size (µm)	Density (kg/l)	Flowrate (l/min)	Stirrer speed (rpm)	Media size (mm)	Media load (ml)
1	120	1.42	2.5	2100	3 - 4	2000
2	120	1.60	2.5	2100	3 - 4	2000
3	120	1.80	2.5	2100	3 - 4	2000
4	120	1.42	2.0	2100	3 - 4	2000
5	120	1.42	3.0	2100	3 - 4	2000
6	120	1.42	2.5	1800	3 - 4	2000
7	120	1.42	2.5	1500	3 - 4	2000
8	120	1.42	2.5	2100	1.8 - 2.2	2000
9	120	1.42	2.5	2100	5	2000
10	120	1.42	2.5	2100	3 - 4	1500
11	120	1.42	2.5	2100	3 - 4	2500
12	230	1.42	2.5	2100	3 - 4	2000
13	55	1.42	2.5	2100	3 - 4	2000
14	230	1.60	2.5	2100	3 - 4	2000
15	230	1.80	2.5	2100	3 - 4	2000
16	230	1.42	2.5	2100	3 - 4	1500
17	230	1.42	2.5	2100	3 - 4	2500
18	230	1.42	2.5	2100	1.8 -2.2	2000
19	230	1.42	2.5	2100	5	2000
20	55	1.60	2.5	2100	3 - 4	2000
21	55	1.80	2.5	2100	3 - 4	2000
22	55	1.42	2.5	2100	1.8 - 2.2	2000
23	55	1.42	2.5	2100	5	2000
24	55	1.42	2.5	2100	3 - 4	1500
25	55	1.42	2.5	2100	3 - 4	2500
26	120	1.60	2.5	1500	3 - 4	2000
27	120	1.80	2.5	2100	3 - 4	1500

Appendix 2: Calculation of solids concentrations for the M4 IsaMill tests

Test number	Lable	Container mass	Mass Container + wet sample	Wet sample mass	Dry sample mass	% Solids
1	BMC/ISAFEED001-T1	94.30	2606.30	2512.00	1006.50	40.07
	BMC/ISADISCH002-T1	91.20	2077.30	1986.10	830.30	41.81
	BMC/ISADISCH003-T1	91.20	2020.10	1928.90	791.50	41.03
	BMC/ISADISCH004-T1	91.20	2169.20	2078.00	858.90	41.33
	BMC/ISADISCH005-T1	156.10	2391.40	2235.30	933.70	41.77
2	BMC/ISAFEED006-T2	58.10	2924.10	2866.00	1449.70	50.58
	BMC/ISADISCH007-T2	63.00	2634.10	2571.10	1318.40	51.28
	BMC/ISADISCH008-T2	57.80	2614.20	2556.40	1268.80	49.63
	BMC/ISADISCH009-T2	58.50	2628.60	2570.10	1282.60	49.90
	BMC/ISADISCH010-T2	62.80	2582.90	2520.10	1265.50	50.22
3	BMC/ISAFEED011-T3	57.80	2573.30	2515.50	1516.90	60.30
	BMC/ISADISCH012-T3	58.80	3040.40	2981.60	1809.60	60.69
	BMC/ISADISCH013-T3	57.90	2907.80	2849.90	1694.40	59.45
	BMC/ISADISCH014-T3	58.40	2821.20	2762.80	1645.10	59.54
	BMC/ISADISCH015-T3	59.40	2877.70	2818.30	1689.50	59.95
4	BMC/ISAFEED016-T4	59.80	2078.40	2018.60	771.90	38.24
	BMC/ISADISCH017-T4	58.10	2324.80	2266.70	876.10	38.65
	BMC/ISADISCH018-T4	62.90	2275.40	2212.50	860.10	38.87
	BMC/ISADISCH019-T4	57.40	2261.60	2204.20	870.90	39.51
	BMC/ISADISCH020-T4	57.80	2402.80	2345.00	931.60	39.73
5	BMC/ISAFEED021-T5	62.80	2098.90	2036.10	816.30	40.09
	BMC/ISADISCH022-T5	61.40	2351.50	2290.10	937.70	40.95
	BMC/ISADISCH023-T5	61.70	2407.80	2346.10	918.40	39.15
	BMC/ISADISCH024-T5	58.60	2401.10	2342.50	938.10	40.05

	BMC/ISADISCH025-T5	58.10	2398.40	2340.30	941.70	40.24
6	BMC/ISAFEED026-T6	62.80	2160.10	2097.30	816.90	38.95
	BMC/ISADISCH027-T6	62.90	2378.70	2315.80	938.20	40.51
	BMC/ISADISCH028-T6	62.80	2278.90	2216.10	853.40	38.51
	BMC/ISADISCH029-T6	61.80	2242.10	2180.30	856.20	39.27
	BMC/ISADISCH030-T6	63.10	2278.60	2215.50	875.40	39.51
7	BMC/ISAFEED031-T7	57.80	2190.40	2132.60	884.50	41.48
	BMC/ISADISCH032-T7	58.90	2574.00	2515.10	1037.90	41.27
	BMC/ISADISCH033-T7	57.50	2530.00	2472.50	1056.60	42.73
	BMC/ISADISCH034-T7	63.00	2588.80	2525.80	1072.70	42.47
	BMC/ISADISCH035-T7	58.60	2514.30	2455.70	1055.70	42.99
8	BMC/ISAFEED036-T8	64.20	2576.20	2512.00	1026.40	40.86
	BMC/ISADISCH037-T8	58.40	2516.40	2458.00	957.70	38.96
	BMC/ISADISCH038-T8	58.60	2491.40	2432.80	942.50	38.74
	BMC/ISADISCH039-T8	57.50	2500.30	2442.80	952.30	38.98
	BMC/ISADISCH040-T8	58.10	2483.00	2424.90	964.10	39.76
9	BMC/ISAFEED041-T9	58.60	2511.00	2452.40	956.00	38.98
	BMC/ISADISCH042-T9	58.70	2518.10	2459.40	953.60	38.77
	BMC/ISADISCH043-T9	58.30	2466.30	2408.00	938.50	38.97
	BMC/ISADISCH044-T9	63.00	2485.10	2422.10	948.60	39.16
	BMC/ISADISCH045-T9	62.80	2428.80	2366.00	936.00	39.56
10	BMC/ISAFEED046-T10	62.80	2197.20	2134.40	865.80	40.56
	BMC/ISADISCH047-T10	58.40	2461.10	2402.70	974.90	40.58
	BMC/ISADISCH048-T10	58.40	2375.40	2317.00	915.30	39.50
	BMC/ISADISCH049-T10	65.00	2365.50	2300.50	928.70	40.37
	BMC/ISADISCH050-T10	61.50	2474.30	2412.80	985.90	40.86
11	BMC/ISAFEED051-T11	57.70	2579.70	2522.00	996.50	39.51
	BMC/ISADISCH052-T11	58.30	2419.60	2361.30	981.10	41.55
	BMC/ISADISCH053-T11	57.40	2402.60	2345.20	950.50	40.53
	BMC/ISADISCH054-T11	63.20	2359.20	2296.00	936.20	40.78
	BMC/ISADISCH055-T11	62.90	2350.80	2287.90	941.10	41.13

12	BMC/ISAFEED056-T12	62.90	2389.80	2326.90	937.20	40.28
	BMC/ISADISCH057-T12	61.80	2327.50	2265.70	914.30	40.35
	BMC/ISADISCH058-T12	58.60	2404.10	2345.50	952.40	40.61
	BMC/ISADISCH059-T12	62.30	2408.90	2346.60	936.70	39.92
	BMC/ISADISCH060-T12	62.90	2465.80	2402.90	942.80	39.24
13	BMC/ISAFEED061-T13	58.00	2465.10	2407.10	937.20	38.93
	BMC/ISADISCH062-T13	63.00	2372.40	2309.40	895.40	38.77
	BMC/ISADISCH063-T13	58.20	2315.70	2257.50	849.40	37.63
	BMC/ISADISCH064-T13	61.80	2340.10	2278.30	864.50	37.94
	BMC/ISADISCH065-T13	58.70	2333.30	2274.60	866.70	38.10
14	BMC/ISAFEED066-T14	57.70	2842.00	2784.30	1451.90	52.15
	BMC/ISADISCH067-T14	62.90	2282.50	2219.60	1162.80	52.39
	BMC/ISADISCH068-T14	64.00	2338.90	2274.90	967.40	42.52
	BMC/ISADISCH069-T14	63.00	2549.90	2486.90	1040.10	41.82
	BMC/ISADISCH070-T14	58.60	2537.10	2478.50	1041.50	42.02
15	BMC/ISAFEED071-T15	59.80	3316.00	3256.20	2018.20	61.98
	BMC/ISADISCH072-T15	64.20	1753.50	1689.30	952.00	56.35
	BMC/ISADISCH073-T15	59.10	1143.70	1084.60	529.30	48.80
	BMC/ISADISCH074-T15	58.70	1970.60	1911.90	936.50	48.98
	BMC/ISADISCH075-T15	58.90	1487.90	1429.00	685.80	47.99
16	BMC/ISAFEED076-T16	57.70	2590.90	2533.20	1081.10	42.68
	BMC/ISADISCH077-T16	63.50	2385.80	2322.30	815.40	35.11
	BMC/ISADISCH078-T16	57.90	2338.30	2280.40	798.10	35.00
	BMC/ISADISCH079-T16	59.00	2341.80	2282.80	815.90	35.74
	BMC/ISADISCH080-T16	58.80	2339.60	2280.80	821.20	36.00
17	BMC/ISAFEED081-T17	58.60	2509.20	2450.60	989.80	40.39
	BMC/ISADISCH082-T17	58.10	2410.40	2352.30	958.50	40.75
	BMC/ISADISCH083-T17	58.90	2324.70	2265.80	898.70	39.66
	BMC/ISADISCH084-T17	57.40	2290.50	2233.10	883.60	39.57
	BMC/ISADISCH085-T17	58.10	2304.20	2246.10	885.90	39.44
18	BMC/ISAFEED086-T18	58.90	2512.90	2454.00	983.50	40.08

	BMC/ISADISCH087-T18	58.40	2494.10	2435.70	823.20	33.80
	BMC/ISADISCH088-T18	63.80	1818.40	1754.60	583.10	33.23
	BMC/ISADISCH089-T18	61.70	1829.50	1767.80	589.30	33.34
	BMC/ISADISCH090-T18	62.90	2500.20	2437.30	816.50	33.50
	BMC/ISAFEED091-T19	61.50	2600.80	2539.30	1018.30	40.10
	BMC/ISADISCH092-T19	59.70	2608.30	2548.60	1032.10	40.50
19	BMC/ISADISCH093-T19	58.20	2524.80	2466.60	977.10	39.61
	BMC/ISADISCH094-T19	58.10	2474.30	2416.20	956.50	39.59
	BMC/ISADISCH095-T19	63.00	2398.70	2335.70	926.80	39.68
	BMC/ISAFEED096-T20	58.10	2901.70	2843.60	1478.40	51.99
	BMC/ISADISCH097-T20	61.70	2671.10	2609.40	1349.90	51.73
20	BMC/ISADISCH098-T20	60.20	2638.00	2577.80	1305.50	50.64
	BMC/ISADISCH099-T20	63.30	2770.10	2706.80	1375.60	50.82
	BMC/ISADISCH100-T20	58.40	2771.70	2713.30	1386.00	51.08
	BMC/ISAFEED101-T21	57.50	3214.50	3157.00	1900.50	60.20
	BMC/ISADISCH102-T21	58.10	3034.20	2976.10	1781.10	59.85
21	BMC/ISADISCH103-T21	57.10	2994.50	2937.40	1721.20	58.60
	BMC/ISADISCH104-T21	61.30	3133.80	3072.50	1803.50	58.70
	BMC/ISADISCH105-T21	58.20	2471.90	2413.70	1425.00	59.04
	BMC/ISAFEED106-T22	62.80	2590.60	2527.80	1075.30	42.54
	BMC/ISADISCH107-T22	58.10	2546.60	2488.50	1056.90	42.47
22	BMC/ISADISCH108-T22	59.70	2532.70	2473.00	1030.40	41.67
	BMC/ISADISCH109-T22	58.90	2597.00	2538.10	1053.50	41.51
	BMC/ISADISCH110-T22	58.20	2532.50	2474.30	1036.60	41.89
	BMC/ISAFEED111-T23	58.70	2565.70	2507.00	1011.70	40.36
	BMC/ISADISCH112-T23	57.30	2527.00	2469.70	992.00	40.17
23	BMC/ISADISCH113-T23	58.60	2504.20	2445.60	951.50	38.91
	BMC/ISADISCH114-T23	57.90	2485.30	2427.40	953.00	39.26
	BMC/ISADISCH115-T23	127.10	2461.70	2334.60	915.90	39.23
24	BMC/ISAFEED116-T24	62.80	2578.90	2516.10	1007.20	40.03
	BMC/ISADISCH117-T24	63.00	2456.10	2393.10	948.00	39.61

	BMC/ISADISCH118-T24	63.00	2438.50	2375.50	908.50	38.24
	BMC/ISADISCH119-T24	58.30	2501.60	2443.30	940.70	38.50
	BMC/ISADISCH120-T24	58.70	2418.70	2360.00	912.60	38.67
25	BMC/ISAFEED121-T25	58.20	2597.30	2539.10	1037.40	40.86
	BMC/ISADISCH122-T25	58.60	2467.00	2408.40	990.00	41.11
	BMC/ISADISCH123-T25	63.20	2520.00	2456.80	997.10	40.59
	BMC/ISADISCH124-T25	61.70	2594.40	2532.70	1044.80	41.25
	BMC/ISADISCH125-T25	63.10	2547.00	2483.90	1033.90	41.62
26	BMC/ISAFEED126-T26	58.00	2871.80	2813.80	1477.10	52.49
	BMC/ISADISCH127-T26	62.80	2873.80	2811.00	1459.30	51.91
	BMC/ISADISCH128-T26	58.00	2834.80	2776.80	1403.10	50.53
	BMC/ISADISCH129-T26	58.50	2806.10	2747.60	1390.40	50.60
	BMC/ISADISCH130-T26	61.90	2825.60	2763.70	1400.00	50.66
27	BMC/ISAFEED131-T27	58.70	3150.10	3091.40	1882.90	60.91
	BMC/ISADISCH132-T27	58.40	3212.60	3154.20	1913.10	60.65
	BMC/ISADISCH133-T27	63.00	3117.90	3054.90	1802.80	59.01
	BMC/ISADISCH134-T27	64.20	3146.20	3082.00	1834.90	59.54
	BMC/ISADISCH135-T27	58.60	3154.00	3095.40	1855.70	59.95

Appendix 3: Test conditions for the M4 IsaMill tests

Test number	Lable	Solids concentration	Net power draw (kW)	Media size	Media load (%)	Vol flowrate (l/min)	Stirrer speed (rpm)	Calc. solids flow (kg/min)	Specific energy (kWh/t)	Cum. specific energy (kWh/t)
1	BMC/ISAFEED001-T1	40.07								
	BMC/ISADISCH002-T1	41.81	1.8	3.5	65	2.5	2100	1.48	20.21	20.21
	BMC/ISADISCH003-T1	41.03	1.5	3.5	65	2.5	2100	1.46	17.16	37.38
	BMC/ISADISCH004-T1	41.33	1.4	3.5	65	2.5	2100	1.47	15.90	53.28
	BMC/ISADISCH005-T1	41.77	1.4	3.5	65	2.5	2100	1.48	15.74	69.01
2	BMC/ISAFEED006-T2	50.58								
	BMC/ISADISCH007-T2	51.28	2.2	3.5	65	2.5	2100	2.05	17.88	17.88
	BMC/ISADISCH008-T2	49.63	1.9	3.5	65	2.5	2100	1.99	15.95	33.83
	BMC/ISADISCH009-T2	49.90	1.7	3.5	65	2.5	2100	2.00	14.19	48.02
	BMC/ISADISCH010-T2	50.22	1.6	3.5	65	2.5	2100	2.01	13.28	61.30
3	BMC/ISAFEED011-T3	60.30								
	BMC/ISADISCH012-T3	60.69	2.3	3.5	65	2.5	2100	2.73	14.04	14.04
	BMC/ISADISCH013-T3	59.45	2	3.5	65	2.5	2100	2.68	12.46	26.49
	BMC/ISADISCH014-T3	59.54	1.9	3.5	65	2.5	2100	2.68	11.82	38.31
	BMC/ISADISCH015-T3	59.95	1.8	3.5	65	2.5	2100	2.70	11.12	49.43
4	BMC/ISAFEED016-T4	38.24								
	BMC/ISADISCH017-T4	38.65	1.8	3.5	65	2	2100	1.10	27.33	27.33
	BMC/ISADISCH018-T4	38.87	1.6	3.5	65	2	2100	1.10	24.15	51.48
	BMC/ISADISCH019-T4	39.51	1.4	3.5	65	2	2100	1.12	20.79	72.28
	BMC/ISADISCH020-T4	39.73	1.4	3.5	65	2	2100	1.13	20.68	92.96
5	BMC/ISAFEED021-T5	40.09								
	BMC/ISADISCH022-T5	40.95	2.2	3.5	65	3	2100	1.74	21.02	21.02
	BMC/ISADISCH023-T5	39.15	1.7	3.5	65	3	2100	1.67	16.99	38.01
	BMC/ISADISCH024-T5	40.05	1.5	3.5	65	3	2100	1.71	14.65	52.67
	BMC/ISADISCH025-T5	40.24	1.5	3.5	65	3	2100	1.71	14.58	67.25
6	BMC/ISAFEED026-T6	38.95								

	BMC/ISADISCH027-T6	40.51	1	3.5	65	2.5	1800	1.44	11.59	11.59
	BMC/ISADISCH028-T6	38.51	0.8	3.5	65	2.5	1800	1.37	9.75	21.34
	BMC/ISADISCH029-T6	39.27	0.7	3.5	65	2.5	1800	1.39	8.37	29.71
	BMC/ISADISCH030-T6	39.51	0.7	3.5	65	2.5	1800	1.40	8.32	38.03
7	BMC/ISAFEED031-T7	41.48								
	BMC/ISADISCH032-T7	41.27	0.6	3.5	65	2.5	1500	1.46	6.83	6.83
	BMC/ISADISCH033-T7	42.73	0.5	3.5	65	2.5	1500	1.52	5.49	12.32
	BMC/ISADISCH034-T7	42.47	0.4	3.5	65	2.5	1500	1.51	4.42	16.74
	BMC/ISADISCH035-T7	42.99	0.3	3.5	65	2.5	1500	1.53	3.28	20.02
8	BMC/ISAFEED036-T8	40.86								
	BMC/ISADISCH037-T8	38.96	2	2	65	2.5	2100	1.38	24.10	24.10
	BMC/ISADISCH038-T8	38.74	1.6	2	65	2.5	2100	1.38	19.39	43.49
	BMC/ISADISCH039-T8	38.98	1.4	2	65	2.5	2100	1.38	16.86	60.35
	BMC/ISADISCH040-T8	39.76	1.3	2	65	2.5	2100	1.41	15.35	75.70
9	BMC/ISAFEED041-T9	38.98								
	BMC/ISADISCH042-T9	38.77	1.9	5	65	2.5	2100	1.38	23.01	23.01
	BMC/ISADISCH043-T9	38.97	1.5	5	65	2.5	2100	1.38	18.07	41.07
	BMC/ISADISCH044-T9	39.16	1.4	5	65	2.5	2100	1.39	16.78	57.86
	BMC/ISADISCH045-T9	39.56	1.3	5	65	2.5	2100	1.40	15.43	73.29
10	BMC/ISAFEED046-T10	40.56						0.00		
	BMC/ISADISCH047-T10	40.58	1.5	3.5	50	2.5	2100	1.44	17.36	17.36
	BMC/ISADISCH048-T10	39.50	1.2	3.5	50	2.5	2100	1.40	14.26	31.62
	BMC/ISADISCH049-T10	40.37	0.9	3.5	50	2.5	2100	1.43	10.47	42.08
	BMC/ISADISCH050-T10	40.86	1	3.5	50	2.5	2100	1.45	11.49	53.57
11	BMC/ISAFEED051-T11	39.51						0.00		
	BMC/ISADISCH052-T11	41.55	2.1	3.5	80	2.5	2100	1.47	23.73	23.73
	BMC/ISADISCH053-T11	40.53	1.8	3.5	80	2.5	2100	1.44	20.85	44.58
	BMC/ISADISCH054-T11	40.78	1.6	3.5	80	2.5	2100	1.45	18.42	63.00
	BMC/ISADISCH055-T11	41.13	1.6	3.5	80	2.5	2100	1.46	18.26	81.26
12	BMC/ISAFEED056-T12	40.28						0.00		
	BMC/ISADISCH057-T12	40.35	2	3.5	65	2.5	2100	1.43	23.27	23.27
	BMC/ISADISCH058-T12	40.61	1.9	3.5	65	2.5	2100	1.44	21.97	45.24

	BMC/ISADISCH059-T12	39.92	1.5	3.5	65	2.5	2100	1.42	17.64	62.88
	BMC/ISADISCH060-T12	39.24	1.4	3.5	65	2.5	2100	1.39	16.75	79.63
	BMC/ISAFEED061-T13	38.93						0.00		
	BMC/ISADISCH062-T13	38.77	1.5	3.5	65	2.5	2100	1.38	18.16	18.16
13	BMC/ISADISCH063-T13	37.63	1.3	3.5	65	2.5	2100	1.34	16.22	34.38
	BMC/ISADISCH064-T13	37.94	1.2	3.5	65	2.5	2100	1.35	14.85	49.23
	BMC/ISADISCH065-T13	38.10	1	3.5	65	2.5	2100	1.35	12.32	61.55
	BMC/ISAFEED066-T14	52.15						0.00		
	BMC/ISADISCH067-T14	52.39	2.2	3.5	65	2.5	2100	2.10	17.50	17.50
14	BMC/ISADISCH068-T14	42.52	1.6	3.5	65	2.5	2100	1.70	15.68	33.17
	BMC/ISADISCH069-T14	41.82	1.4	3.5	65	2.5	2100	1.67	13.95	47.12
	BMC/ISADISCH070-T14	42.02	1.3	3.5	65	2.5	2100	1.68	12.89	60.01
	BMC/ISAFEED071-T15	61.98								
	BMC/ISADISCH072-T15	56.35	2.4	3.5	65	2.5	2100	2.54	15.77	15.77
15	BMC/ISADISCH073-T15	48.80	2.1	3.5	65	2.5	2100	2.20	15.94	31.71
	BMC/ISADISCH074-T15	48.98	1.8	3.5	65	2.5	2100	2.20	13.61	45.32
	BMC/ISADISCH075-T15	47.99	1.7	3.5	65	2.5	2100	2.16	13.12	58.44
	BMC/ISAFEED076-T16	42.68								
	BMC/ISADISCH077-T16	35.11	1.6	3.5	50	2.5	2100	1.25	21.39	21.39
16	BMC/ISADISCH078-T16	35.00	1.3	3.5	50	2.5	2100	1.24	17.44	38.83
	BMC/ISADISCH079-T16	35.74	1.1	3.5	50	2.5	2100	1.27	14.45	53.28
	BMC/ISADISCH080-T16	36.00	1	3.5	50	2.5	2100	1.28	13.04	66.32
	BMC/ISAFEED081-T17	40.39								
	BMC/ISADISCH082-T17	40.75	2.5	3.5	80	2.5	2100	1.45	28.80	28.80
17	BMC/ISADISCH083-T17	39.66	1.5	3.5	80	2.5	2100	1.41	17.75	46.56
	BMC/ISADISCH084-T17	39.57	1.3	3.5	80	2.5	2100	1.40	15.42	61.98
	BMC/ISADISCH085-T17	39.44	1.2	3.5	80	2.5	2100	1.40	14.28	76.27
	BMC/ISAFEED086-T18	40.08								
	BMC/ISADISCH087-T18	33.80	4.1	2	65	2.5	2100	1.20	56.95	56.95
18	BMC/ISADISCH088-T18	33.23	1.6	2	65	2.5	2100	1.18	22.60	79.56
	BMC/ISADISCH089-T18	33.34	1.5	2	65	2.5	2100	1.18	21.13	100.68
	BMC/ISADISCH090-T18	33.50	1.3	2	65	2.5	2100	1.19	18.22	118.90

	BMC/ISAFEED091-T19	40.10								
	BMC/ISADISCH092-T19	40.50	2.5	5	65	2.5	2100	1.44	28.98	28.98
19	BMC/ISADISCH093-T19	39.61	2	5	65	2.5	2100	1.41	23.70	52.69
	BMC/ISADISCH094-T19	39.59	1.7	5	65	2.5	2100	1.41	20.16	72.85
	BMC/ISADISCH095-T19	39.68	1.5	5	65	2.5	2100	1.41	17.75	90.60
	BMC/ISAFEED096-T20	51.99								
	BMC/ISADISCH097-T20	51.73	1.6	3.5	65	2.5	2100	2.07	12.89	12.89
20	BMC/ISADISCH098-T20	50.64	1.5	3.5	65	2.5	2100	2.03	12.34	25.23
	BMC/ISADISCH099-T20	50.82	1.4	3.5	65	2.5	2100	2.03	11.48	36.71
	BMC/ISADISCH100-T20	51.08	1.3	3.5	65	2.5	2100	2.04	10.60	47.31
	BMC/ISAFEED101-T21	60.20								
	BMC/ISADISCH102-T21	59.85	1.9	3.5	65	2.5	2100	2.69	11.76	11.76
21	BMC/ISADISCH103-T21	58.60	1.7	3.5	65	2.5	2100	2.64	10.75	22.50
	BMC/ISADISCH104-T21	58.70	1.6	3.5	65	2.5	2100	2.64	10.10	32.60
	BMC/ISADISCH105-T21	59.04	1.6	3.5	65	2.5	2100	2.66	10.04	42.64
	BMC/ISAFEED106-T22	42.54								
	BMC/ISADISCH107-T22	42.47	1.6	2	65	2.5	2100	1.51	17.69	17.69
22	BMC/ISADISCH108-T22	41.67	1.4	2	65	2.5	2100	1.48	15.77	33.46
	BMC/ISADISCH109-T22	41.51	1.3	2	65	2.5	2100	1.47	14.70	48.17
	BMC/ISADISCH110-T22	41.89	1.3	2	65	2.5	2100	1.49	14.57	62.73
	BMC/ISAFEED111-T23	40.36								
	BMC/ISADISCH112-T23	40.17	1.8	5	65	2.5	2100	1.43	21.04	21.04
23	BMC/ISADISCH113-T23	38.91	1.6	5	65	2.5	2100	1.38	19.31	40.35
	BMC/ISADISCH114-T23	39.26	1.5	5	65	2.5	2100	1.39	17.94	58.28
	BMC/ISADISCH115-T23	39.23	1.4	5	65	2.5	2100	1.39	16.75	75.04
	BMC/ISAFEED116-T24	40.03								
	BMC/ISADISCH117-T24	39.61	1.3	3.5	50	2.5	2100	1.41	15.41	15.41
24	BMC/ISADISCH118-T24	38.24	1.1	3.5	50	2.5	2100	1.36	13.50	28.91
	BMC/ISADISCH119-T24	38.50	0.9	3.5	50	2.5	2100	1.37	10.97	39.88
	BMC/ISADISCH120-T24	38.67	0.9	3.5	50	2.5	2100	1.37	10.93	50.81
25	BMC/ISAFEED121-T25	40.86								
	BMC/ISADISCH122-T25	41.11	2.1	3.5	80	2.5	2100	1.46	23.98	23.98

	BMC/ISADISCH123-T25	40.59	1.7	3.5	80	2.5	2100	1.44	19.67	43.65
	BMC/ISADISCH124-T25	41.25	1.6	3.5	80	2.5	2100	1.46	18.21	61.86
	BMC/ISADISCH125-T25	41.62	1.6	3.5	80	2.5	2100	1.48	18.05	79.91
26	BMC/ISAFEED126-T26	52.49								
	BMC/ISADISCH127-T26	51.91	0.5	3.5	65	2.5	1500	2.08	4.01	4.01
	BMC/ISADISCH128-T26	50.53	0.4	3.5	65	2.5	1500	2.02	3.30	7.31
	BMC/ISADISCH129-T26	50.60	0.4	3.5	65	2.5	1500	2.02	3.29	10.61
	BMC/ISADISCH130-T26	50.66	0.3	3.5	65	2.5	1500	2.03	2.47	13.07
27	BMC/ISAFEED131-T27	60.91								
	BMC/ISADISCH132-T27	60.65	1.8	3.5	50	2.5	2100	2.73	10.99	10.99
	BMC/ISADISCH133-T27	59.01	1.6	3.5	50	2.5	2100	2.66	10.04	21.03
	BMC/ISADISCH134-T27	59.54	1.4	3.5	50	2.5	2100	2.68	8.71	29.74
	BMC/ISADISCH135-T27	59.95	1.3	3.5	50	2.5	2100	2.70	8.03	37.77

Appendix 4: Pass by pass power draw readings

Test #		0 mins	5mins	10mins	15mins	20mins	25mins	30mins	Average Power	Standard Dev
1	Pass 1	2.9	3.2	3.2	3.2	3.1	3.1	3.1	3.11	0.11
	Pass 2	3.1	2.9	2.8	2.8	2.8	2.8	2.8	2.86	0.11
	Pass 3	2.8	2.7	2.7	2.7	2.7	2.7	2.7	2.71	0.04
	Pass 4	2.7	2.7	2.7	2.7	2.7	2.7	2.7	2.70	0.00
2	Pass 1	2.2	3.5	3.5	3.5	3.5	3.5	3.5	3.31	0.49
	Pass 2	3.5	3.2	3.2	3.1	3.1	3.1	3.1	3.19	0.15
	Pass 3	3.1	3	3	3	3	3	3	3.01	0.04
	Pass 4	3	2.9	2.9	2.9	2.9	2.9	2.9	2.91	0.04
3	Pass 1	2.1	3.6	3.6	3.6	3.6	3.6	3.6	3.39	0.57
	Pass 2	3.6	3.4	3.4	3.3	3.3	3.3	3.3	3.37	0.11
	Pass 3	3.3	3.2	3.2	3.2	3.2	3.2	3.2	3.21	0.04
	Pass 4	3.2	3.1	3.1	3.1	3.1	3.1	3.1	3.11	0.04
4	Pass 1	2.1	2.3	2.3	2.3	2.3	2.3	2.3	2.27	0.08
	Pass 2	2.3	2.2	2.1	2.1	2.1	2.1	2.1	2.14	0.08
	Pass 3	2.1	2.1	2	2	2	2	2	2.03	0.05
	Pass 4	2	2	2	2	2	2	2	2.00	0.00
5	Pass 1	2.3	3.6	3.6	3.5	3.4	3.4	3.4	3.31	0.46
	Pass 2	3.3	3	3	3	3	3	3	3.04	0.11
	Pass 3	3	2.8	2.8	2.8	2.8	2.8	2.8	2.83	0.08
	Pass 4	2.8	2.8	2.8	2.8	2.8	2.8	2.7	2.79	0.04
6	Pass 1	2.2	3	3.1	3.1	3.1	3.1	3.1	2.96	0.34
	Pass 2	3.1	2.9	2.9	2.9	2.8	2.8	2.8	2.89	0.11
	Pass 3	2.8	2.8	2.7	2.7	2.7	2.7	2.7	2.73	0.05
	Pass 4	2.7	2.7	2.7	2.7	2.7	2.7	2.7	2.70	0.00
7	Pass 1	1.5	1.8	1.9	1.9	1.9	1.9	1.9	1.83	0.15
	Pass 2	1.9	1.8	1.8	1.8	1.8	1.8	1.8	1.81	0.04
	Pass 3	1.7	1.7	1.7	1.7	1.7	1.7	1.7	1.70	0.00

	Pass 4	1.7	1.6	1.6	1.6	1.6	1.6	1.6	1.61	0.04
8	Pass 1	2.5	3.2	3.3	3.3	3.3	3.3	3.4	3.19	0.31
	Pass 2	3.4	3.1	2.9	2.9	2.9	2.9	2.9	3.00	0.19
	Pass 3	2.9	2.8	2.7	2.7	2.7	2.7	2.7	2.74	0.08
	Pass 4	2.7	2.7	2.6	2.6	2.6	2.6	2.6	2.63	0.05
9	Pass 1	2.2	3.1	3.2	3.2	3.2	3.2	3.2	3.04	0.37
	Pass 2	3.2	3	2.8	2.8	2.8	2.8	2.8	2.89	0.16
	Pass 3	2.8	2.7	2.7	2.7	2.7	2.7	2.7	2.71	0.04
	Pass 4	2.7	2.6	2.6	2.6	2.6	2.6	2.6	2.61	0.04
10	Pass 1	2	2.8	2.8	2.8	2.8	2.8	2.8	2.69	0.30
	Pass 2	2.8	2.7	2.5	2.5	2.5	2.5	2.5	2.57	0.13
	Pass 3	2.5	2.2	2.2	2.2	2.2	2.2	2.2	2.24	0.11
	Pass 4	2.2	2.3	2.3	2.3	2.3	2.3	2.3	2.29	0.04
11	Pass 1	2.4	3.5	3.5	3.4	3.4	3.4	3.4	3.29	0.39
	Pass 2	3.4	3.1	3.1	3	3	3	3	3.09	0.15
	Pass 3	3	2.9	2.9	2.9	2.9	2.9	2.9	2.91	0.04
	Pass 4	2.9	2.9	2.9	2.9	2.9	2.9	2.9	2.90	0.00
12	Pass 1	2.2	3.3	3.3	3.2	3.1	3.6	5.5	3.46	1.00
	Pass 2	4.4	3.8	3.4	3.2	3	3	3	3.40	0.53
	Pass 3	3	2.9	2.8	2.8	2.8	2.8	2.8	2.84	0.08
	Pass 4	2.8	2.7	2.7	2.7	2.7	2.7	2.7	2.71	0.04
13	Pass 1	1.9	2.8	2.8	2.8	2.8	2.8	2.8	2.67	0.34
	Pass 2	2.8	2.6	2.6	2.6	2.6	2.6	2.6	2.63	0.08
	Pass 3	2.6	2.5	2.5	2.5	2.5	2.5	2.5	2.51	0.04
	Pass 4	2.5	2.3	2.3	2.3	2.3	2.3	2.3	2.33	0.08
14	Pass 1	2.4	3.8	3.9	3.7	3.6			3.48	0.61
	Pass 2	2.5	2.9	2.9	2.9	2.9	2.9	2.9	2.84	0.15
	Pass 3	2.9	2.8	2.7	2.7	2.7	2.7	2.7	2.74	0.08
	Pass 4	2.7	2.6	2.6	2.6	2.6	2.6	2.6	2.61	0.04
15	Pass 1	2.1	3.4	3.7	3.7	3.7			3.32	0.69
	Pass 2	2.2	3.4	3.4	3.4	3.4			3.16	0.54

	Pass 3	3.4	3.2	3.1	3.1	3.1			3.18	0.13
	Pass 4	3.1	3.1	3	3	3			3.04	0.05
16	Pass 1	2.1	2.6	2.6	2.6	2.9	4.4		2.87	0.79
	Pass 2	2.2	2.5	2.7	2.7	2.7	2.6	2.6	2.57	0.18
	Pass 3	2.6	2.6	2.6	2.5	2.3	2.3	2.3	2.46	0.15
	Pass 4	2.3	2.3	2.3	2.3	2.3	2.3	2.3	2.30	0.00
17	Pass 1	2.3	3.1	3.7	3.7	3.8	4.4	4.6	3.66	0.78
	Pass 2	2.6	2.7	2.8	2.8	2.8	2.8	2.8	2.76	0.08
	Pass 3	2.8	2.6	2.5	2.5	2.5	2.5	2.5	2.56	0.11
	Pass 4	2.5	2.5	2.5	2.5	2.5	2.5	2.5	2.50	0.00
18	Pass 1	2.5	3.3	5.5	5.5	5.5	5.5		4.63	1.37
	Pass 2	2.3	2.7	2.8	2.9	2.9	2.9	2.9	2.77	0.22
	Pass 3	2.9	2.8	2.8	2.8	2.7	2.7	2.7	2.77	0.08
	Pass 4	2.7	2.6	2.6	2.6	2.6	2.6	2.6	2.61	0.04
19	Pass 1	2.2	3.9	3.9	3.8	3.8	3.8	3.8	3.60	0.62
	Pass 2	2.6	3.2	3.3	3.3	3.3	3.3	3.4	3.20	0.27
	Pass 3	2.3	2.9	3	3	3	3	3	2.89	0.26
	Pass 4	3	2.9	2.8	2.8	2.8	2.8	2.8	2.84	0.08
20	Pass 1	2.2	3	3	2.9	2.9	2.9	2.9	2.83	0.28
	Pass 2	2.9	2.8	2.8	2.8	2.8	2.8	2.8	2.81	0.04
	Pass 3	2.8	2.7	2.7	2.7	2.7	2.7	2.7	2.71	0.04
	Pass 4	2.7	2.6	2.6	2.6	2.6	2.6	2.6	2.61	0.04
21	Pass 1	2.2	3.3	3.2	3.2	3.1	3.1	3.1	3.03	0.37
	Pass 2	3.1	3	3	2.9	2.9	2.9	2.9	2.96	0.08
	Pass 3	2.9	2.9	2.9	2.9	2.9	2.9	2.9	2.90	0.00
	Pass 4	2.9	2.9	2.9	2.9	2.9	2.9	2.9	2.90	0.00
22	Pass 1	2.3	2.9	2.9	2.9	2.9	2.9	2.9	2.81	0.23
	Pass 2	2.9	2.8	2.7	2.7	2.7	2.7	2.7	2.74	0.08
	Pass 3	2.7	2.6	2.6	2.6	2.6	2.6	2.6	2.61	0.04
	Pass 4	2.6	2.6	2.6	2.6	2.6	2.6	2.6	2.60	0.00
23	Pass 1	2.4	3.1	3.1	3.1	3.1	3.1	3.1	3.00	0.26

	Pass 2	3.1	2.9	2.9	2.8	2.8	2.8	2.8	2.87	0.11
	Pass 3	2.8	2.8	2.8	2.8	2.8	2.8	2.8	2.80	0.00
	Pass 4	2.8	2.7	2.7	2.7	2.7	2.7	2.7	2.71	0.04
24	Pass 1	2.2	2.5	2.5	2.6	2.6	2.6	2.6	2.51	0.15
	Pass 2	2.6	2.4	2.4	2.4	2.3	2.3	2.3	2.39	0.11
	Pass 3	2.3	2.3	2.2	2.2	2.2	2.2	2.2	2.23	0.05
	Pass 4	2.2	2.2	2.2	2.2	2.2	2.2	2.2	2.20	0.00
25	Pass 1	2.6	3.4	3.4	3.4	3.4	3.3	3.3	3.26	0.29
	Pass 2	3.3	3.1	3	3	3	3	3	3.06	0.11
	Pass 3	3	3	2.9	2.9	2.9	2.9	2.9	2.93	0.05
	Pass 4	2.9	2.8	2.9	2.9	2.9	2.9	2.9	2.89	0.04
26	Pass 1	1.5	1.8	1.8	1.8	1.8	1.9	1.9	1.79	0.13
	Pass 2	1.9	1.8	1.8	1.8	1.7	1.7	1.7	1.77	0.08
	Pass 3	1.7	1.7	1.7	1.7	1.7	1.7	1.7	1.70	0.00
	Pass 4	1.7	1.6	1.6	1.6	1.6	1.6	1.6	1.61	0.04
27	Pass 1	2.5	3.1	3.1	3.1	3.1	3.1	3.1	3.01	0.23
	Pass 2	3.1	2.9	2.9	2.9	2.9	2.9	2.9	2.93	0.08
	Pass 3	2.9	2.8	2.7	2.7	2.7	2.7	2.7	2.74	0.08
	Pass 4	2.7	2.7	2.6	2.6	2.6	2.6	2.6	2.63	0.05

Appendix 5: particle size distribution results for M4 IsaMill tests

	Malvern Results (μm)																			
Test number	**1**	**2.25**	**3.12**	**4.5**	**6.25**	**9**	**12.5**	**18**	**25**	**38**	**53**	**75**	**106**	**150**	**212**	**300**	**425**	**600**	**850**	**1200**
1	0.00	1.28	2.49	4.24	6.23	9.09	12.33	16.77	21.92	31.55	42.95	58.33	74.68	88.16	96.17	99.36	99.99	100.00	100.00	100.00
	0.00	3.14	6.11	10.43	15.28	22.14	30.22	42.11	55.75	75.43	89.14	97.74	99.98	100.00	100.00	100.00	100.00	100.00	100.00	100.00
	0.00	4.26	8.00	13.24	19.20	28.04	38.73	53.62	68.42	85.34	94.34	98.82	99.96	100.00	100.00	100.00	100.00	100.00	100.00	100.00
	0.00	5.01	9.31	15.45	22.72	33.73	46.67	63.28	77.87	91.94	97.86	99.89	100.00	100.00	100.00	100.00	100.00	100.00	100.00	100.00
	0.00	5.89	10.87	18.11	26.80	39.73	54.11	71.03	84.31	95.37	99.17	100.00	100.00	100.00	100.00	100.00	100.00	100.00	100.00	100.00
2	0.00	1.54	2.98	5.02	7.34	10.63	14.42	19.72	25.75	36.00	46.82	60.41	74.55	86.54	94.37	98.29	99.72	100.00	100.00	100.00
	0.00	2.90	5.60	9.46	13.76	19.81	26.90	37.31	49.35	67.26	80.85	91.49	97.41	99.64	100.00	100.00	100.00	100.00	100.00	100.00
	0.00	3.94	7.48	12.53	18.30	26.84	37.16	51.65	66.33	83.66	93.29	98.40	99.92	100.00	100.00	100.00	100.00	100.00	100.00	100.00
	0.00	4.82	9.01	15.03	22.12	32.81	45.40	61.76	76.42	90.98	97.41	99.83	100.00	100.00	100.00	100.00	100.00	100.00	100.00	100.00
	0.00	5.81	10.73	17.87	26.40	39.06	53.20	70.03	83.47	94.95	99.03	100.00	100.00	100.00	100.00	100.00	100.00	100.00	100.00	100.00
3	0.00	1.28	2.49	4.24	6.23	9.09	12.33	16.77	21.92	31.55	42.95	58.33	74.68	88.16	96.17	99.36	99.99	100.00	100.00	100.00
	0.00	2.81	5.32	8.83	12.71	18.08	24.18	32.86	42.94	59.00	72.80	85.44	94.09	98.42	99.78	100.00	100.00	100.00	100.00	100.00
	0.00	3.64	6.91	11.50	16.57	23.75	32.23	44.42	57.71	75.60	87.52	95.52	99.08	99.99	100.00	100.00	100.00	100.00	100.00	100.00
	0.00	4.64	8.65	14.19	20.38	29.38	40.05	54.70	69.12	85.57	94.31	98.66	99.78	99.81	99.81	99.87	99.95	100.00	100.00	100.00
	0.00	5.28	9.82	16.26	23.72	34.74	47.50	63.80	78.16	92.13	98.01	99.91	100.00	100.00	100.00	100.00	100.00	100.00	100.00	100.00
4	0.00	1.25	2.41	4.08	6.01	8.79	11.91	16.09	20.82	29.67	40.52	55.67	72.23	86.21	94.80	98.53	99.60	99.84	99.96	100.00
	0.00	3.42	6.56	11.02	16.08	23.53	32.67	46.06	60.47	78.96	90.31	97.10	99.59	100.00	100.00	100.00	100.00	100.00	100.00	100.00
	0.00	4.73	8.95	15.06	22.28	33.10	45.77	62.09	76.62	90.99	97.34	99.78	100.00	100.00	100.00	100.00	100.00	100.00	100.00	100.00
	0.00	5.70	10.67	18.11	27.16	40.53	55.20	72.12	85.13	95.73	99.26	100.00	100.00	100.00	100.00	100.00	100.00	100.00	100.00	100.00
	0.00	7.02	13.12	22.19	32.94	48.01	63.31	79.33	90.27	97.88	99.84	100.00	100.00	100.00	100.00	100.00	100.00	100.00	100.00	100.00
5	0.00	1.22	2.40	4.09	6.04	8.80	11.93	16.27	21.46	31.40	43.20	58.90	75.29	88.55	96.30	99.35	99.99	100.00	100.00	100.00
	0.00	2.86	5.56	9.42	13.75	19.96	27.43	38.64	51.58	70.29	83.78	93.58	98.43	99.89	100.00	100.00	100.00	100.00	100.00	100.00
	0.00	4.25	7.95	13.12	19.05	28.00	38.93	54.12	69.01	85.59	94.02	97.90	98.64	98.64	98.64	98.83	99.05	99.27	99.53	99.79
	0.00	4.96	9.24	15.39	22.76	34.00	47.23	64.14	78.84	92.71	98.29	99.95	100.00	100.00	100.00	100.00	100.00	100.00	100.00	100.00
	0.00	5.88	10.88	18.17	26.96	40.07	54.65	71.71	84.99	95.85	99.40	100.00	100.00	100.00	100.00	100.00	100.00	100.00	100.00	100.00
6	0.00	1.25	2.44	4.15	6.12	8.91	12.05	16.39	21.59	31.74	43.90	59.97	76.45	89.38	96.64	99.37	99.95	100.00	100.00	100.00
	0.00	2.92	5.65	9.57	13.94	20.11	27.47	38.42	51.07	69.52	83.05	93.10	98.23	99.86	100.00	100.00	100.00	100.00	100.00	100.00

	0.00	3.87	7.35	12.26	17.87	26.26	36.59	51.34	66.43	84.20	93.88	98.75	99.96	100.00	100.00	100.00	100.00	100.00	100.00	100.00
	0.00	4.75	8.82	14.54	21.27	31.52	43.83	60.16	75.09	90.25	97.11	99.77	100.00	100.00	100.00	100.00	100.00	100.00	100.00	100.00
	0.00	5.38	9.98	16.66	24.73	36.94	50.94	68.06	82.13	94.51	99.00	100.00	100.00	100.00	100.00	100.00	100.00	100.00	100.00	100.00
7	0.00	1.54	2.98	5.02	7.34	10.63	14.42	19.72	25.75	36.00	46.82	60.41	74.55	86.54	94.37	98.29	99.72	100.00	100.00	100.00
	0.00	1.81	3.48	5.90	8.65	12.49	16.79	22.89	30.45	44.61	59.47	75.91	89.34	97.14	99.80	100.00	100.00	100.00	100.00	100.00
	0.00	2.67	5.19	8.82	12.88	18.55	25.20	35.09	46.89	65.20	79.66	91.23	97.64	99.82	100.00	100.00	100.00	100.00	100.00	100.00
	0.00	3.46	6.62	11.03	15.94	23.04	31.73	44.66	58.98	77.94	89.88	97.10	99.70	100.00	100.00	100.00	100.00	100.00	100.00	100.00
	0.00	3.86	7.37	12.30	17.91	26.29	36.65	51.56	66.87	84.84	94.44	99.07	100.00	100.00	100.00	100.00	100.00	100.00	100.00	100.00
8	0.00	1.25	2.43	4.13	6.06	8.77	11.79	16.00	21.28	32.03	45.00	61.76	78.25	90.54	97.07	99.42	99.96	100.00	100.00	100.00
	0.00	4.03	7.57	12.53	18.04	25.88	35.01	47.72	60.98	78.03	88.96	96.09	99.20	99.99	100.00	100.00	100.00	100.00	100.00	100.00
	0.00	5.66	10.49	17.40	25.46	37.23	50.45	66.62	80.22	92.87	98.09	99.90	100.00	100.00	100.00	100.00	100.00	100.00	100.00	100.00
	0.00	6.64	12.37	20.85	30.96	45.38	60.43	76.77	88.43	97.12	99.68	100.00	100.00	100.00	100.00	100.00	100.00	100.00	100.00	100.00
	0.00	7.99	15.01	25.48	37.64	53.93	69.44	84.37	93.50	98.95	100.00	100.00	100.00	100.00	100.00	100.00	100.00	100.00	100.00	100.00
9	0.00	1.25	2.41	4.08	6.01	8.79	11.91	16.09	20.82	29.67	40.52	55.67	72.23	86.21	94.80	98.53	99.60	99.84	99.96	100.00
	0.00	3.19	6.10	10.23	14.82	21.29	28.93	40.11	52.83	71.09	84.20	93.73	98.44	99.88	100.00	100.00	100.00	100.00	100.00	100.00
	0.00	4.18	7.85	12.98	18.80	27.43	37.97	52.87	67.94	85.41	94.68	99.12	100.00	100.00	100.00	100.00	100.00	100.00	100.00	100.00
	0.00	5.08	9.35	15.35	22.35	32.93	45.52	62.02	76.86	91.50	97.75	99.89	100.00	100.00	100.00	100.00	100.00	100.00	100.00	100.00
	0.00	5.61	10.32	17.09	25.17	37.33	51.26	68.32	82.36	94.63	99.02	100.00	100.00	100.00	100.00	100.00	100.00	100.00	100.00	100.00
10	0.00	1.25	2.44	4.15	6.12	8.91	12.05	16.39	21.59	31.74	43.90	59.97	76.45	89.38	96.64	99.37	99.95	100.00	100.00	100.00
	0.00	2.66	5.45	9.61	14.37	20.98	28.40	38.78	50.47	67.87	81.27	91.85	97.68	99.75	100.00	100.00	100.00	100.00	100.00	100.00
	0.00	3.71	7.17	12.06	17.57	25.62	35.40	49.49	64.24	82.26	92.44	97.75	99.27	99.60	99.96	100.00	100.00	100.00	100.00	100.00
	0.00	4.33	8.31	14.03	20.76	30.96	43.21	59.62	74.82	90.36	97.31	99.85	100.00	100.00	100.00	100.00	100.00	100.00	100.00	100.00
	0.00	5.06	9.55	16.10	24.01	36.01	49.89	67.10	81.44	94.19	98.89	100.00	100.00	100.00	100.00	100.00	100.00	100.00	100.00	100.00
11	0.00	1.26	2.47	4.21	6.15	8.77	11.64	15.85	21.54	33.49	47.38	64.30	79.95	91.03	96.75	98.92	99.63	99.90	99.99	100.00
	0.00	3.41	6.50	10.83	15.68	22.74	31.31	43.84	57.54	75.78	87.72	95.59	99.05	99.97	100.00	100.00	100.00	100.00	100.00	100.00
	0.00	5.06	9.33	15.28	22.22	32.71	45.11	61.23	75.75	90.34	96.97	99.64	100.00	100.00	100.00	100.00	100.00	100.00	100.00	100.00
	0.00	5.83	10.81	18.12	26.95	40.03	54.48	71.35	84.52	95.45	99.20	100.00	100.00	100.00	100.00	100.00	100.00	100.00	100.00	100.00
	0.00	6.86	12.77	21.61	32.20	47.17	62.51	78.68	89.82	97.70	99.80	100.00	100.00	100.00	100.00	100.00	100.00	100.00	100.00	100.00
12	0.00	1.08	2.08	3.47	4.96	7.01	9.32	12.49	15.92	21.20	26.45	33.68	43.57	56.37	70.77	84.25	94.01	98.87	100.00	100.00
	0.00	3.02	5.74	9.56	13.79	19.69	26.54	36.43	47.70	64.51	77.72	88.79	95.74	98.97	99.90	100.00	100.00	100.00	100.00	100.00
	0.00	4.36	8.13	13.37	19.23	27.67	37.63	51.38	65.27	81.99	91.72	97.40	99.55	100.00	100.00	100.00	100.00	100.00	100.00	100.00
	0.00	5.32	9.79	16.03	23.23	33.90	46.34	62.40	76.77	91.08	97.42	99.83	100.00	100.00	100.00	100.00	100.00	100.00	100.00	100.00

	0.00	5.91	10.91	18.20	26.88	39.66	53.84	70.57	83.86	95.13	99.10	100.00	100.00	100.00	100.00	100.00	100.00	100.00	100.00	100.00
13	0.00	2.76	5.33	9.00	13.12	18.93	25.70	35.52	46.79	63.86	77.50	89.03	96.18	99.30	100.00	100.00	100.00	100.00	100.00	100.00
	0.00	3.85	7.31	12.22	17.80	26.06	36.15	50.54	65.32	82.96	92.86	98.19	99.85	100.00	100.00	100.00	100.00	100.00	100.00	100.00
	0.00	4.83	9.02	15.07	22.28	33.21	46.12	62.78	77.52	91.81	97.84	99.89	100.00	100.00	100.00	100.00	100.00	100.00	100.00	100.00
	0.00	5.51	10.25	17.25	25.77	38.58	52.99	70.14	83.74	95.17	99.13	100.00	100.00	100.00	100.00	100.00	100.00	100.00	100.00	100.00
	0.00	6.55	12.14	20.45	30.46	44.93	60.19	76.85	88.72	97.39	99.77	100.00	100.00	100.00	100.00	100.00	100.00	100.00	100.00	100.00
14	0.00	0.88	1.70	2.81	4.00	5.68	7.61	10.28	13.14	17.53	22.18	29.33	39.94	54.07	69.72	83.85	93.73	98.61	99.95	100.00
	0.00	2.44	4.55	7.51	10.83	15.46	20.67	27.73	35.50	47.94	59.85	73.07	84.95	93.48	98.01	99.67	100.00	100.00	100.00	100.00
	0.00	4.24	7.88	12.88	18.34	25.97	34.84	47.27	60.45	77.66	88.77	95.96	98.95	99.58	99.58	99.64	99.84	100.00	100.00	100.00
	0.00	4.75	8.82	14.47	20.85	30.17	41.26	56.36	71.01	87.26	95.52	99.31	100.00	100.00	100.00	100.00	100.00	100.00	100.00	100.00
	0.00	5.61	10.31	16.99	24.82	36.46	49.81	66.43	80.49	93.41	98.48	99.95	100.00	100.00	100.00	100.00	100.00	100.00	100.00	100.00
15	0.00	0.76	1.48	2.45	3.49	4.92	6.58	8.88	11.38	15.25	19.28	25.43	34.78	47.86	63.36	78.62	90.53	97.39	99.88	100.00
	0.00	2.08	3.83	6.26	8.93	12.62	16.74	22.24	28.08	37.25	46.53	58.30	71.16	82.98	91.69	96.86	99.27	99.98	100.00	100.00
	0.00	3.78	7.08	11.67	16.66	23.52	31.30	42.24	54.28	71.35	83.80	93.21	98.20	99.86	100.00	100.00	100.00	100.00	100.00	100.00
	0.00	4.72	8.74	14.28	20.43	29.25	39.58	53.73	67.80	84.28	93.49	98.44	99.95	100.00	100.00	100.00	100.00	100.00	100.00	100.00
	0.00	5.47	10.04	16.43	23.71	34.35	46.58	62.25	76.26	90.34	96.73	99.28	99.60	99.60	99.60	99.69	99.80	99.91	99.99	100.00
16	0.00	1.19	2.37	4.02	5.79	8.14	10.74	14.27	18.06	23.87	29.72	37.93	49.09	62.87	77.14	89.14	96.71	99.71	100.00	100.00
	0.00	2.64	5.04	8.42	12.09	17.10	22.76	30.79	39.91	53.80	65.30	76.01	84.40	90.56	94.93	97.94	99.61	100.00	100.00	100.00
	0.00	3.51	6.76	11.37	16.45	23.55	31.81	43.66	56.70	74.54	86.71	95.08	98.94	99.97	100.00	100.00	100.00	100.00	100.00	100.00
	0.00	3.96	7.63	12.84	18.67	27.03	36.94	50.85	65.15	82.50	92.50	98.06	99.84	100.00	100.00	100.00	100.00	100.00	100.00	100.00
	0.00	4.92	9.08	14.81	21.39	31.28	43.20	59.22	74.18	89.71	96.87	99.69	100.00	100.00	100.00	100.00	100.00	100.00	100.00	100.00
17	0.00	1.22	2.34	3.87	5.53	7.77	10.28	13.80	17.94	25.36	33.67	44.94	58.26	72.05	84.10	92.85	97.78	99.69	100.00	100.00
	0.00	2.49	4.62	7.56	10.78	15.24	20.30	27.32	35.26	48.13	60.33	73.57	85.13	93.25	97.60	99.38	99.94	100.00	100.00	100.00
	0.00	3.95	7.40	12.17	17.40	24.80	33.55	46.07	59.55	77.32	88.82	96.23	99.32	99.99	100.00	100.00	100.00	100.00	100.00	100.00
	0.00	5.08	9.33	15.10	21.58	31.16	42.63	58.12	72.83	88.57	96.20	99.49	100.00	100.00	100.00	100.00	100.00	100.00	100.00	100.00
	0.00	5.51	10.18	16.83	24.64	36.31	49.77	66.56	80.73	93.66	98.65	100.00	100.00	100.00	100.00	100.00	100.00	100.00	100.00	100.00
18	0.00	0.67	1.29	2.11	2.99	4.22	5.66	7.69	9.93	13.82	18.79	27.50	40.88	58.03	75.52	89.49	97.55	100.00	100.00	100.00
	0.00	2.04	5.91	8.25	11.38	14.36	19.47	24.52	32.34	39.91	49.44	60.67	72.81	84.04	92.76	98.06	99.98	100.00	100.00	100.00
	0.00	6.30	11.37	18.17	25.45	35.45	46.52	60.67	73.78	88.11	95.55	99.19	99.99	100.00	100.00	100.00	100.00	100.00	100.00	100.00
	0.00	6.52	11.84	19.14	27.27	38.72	51.32	66.66	79.72	92.24	97.70	99.81	100.00	100.00	100.00	100.00	100.00	100.00	100.00	100.00
	0.00	8.23	15.08	24.81	35.73	50.39	64.91	79.94	90.23	97.62	99.74	100.00	100.00	100.00	100.00	100.00	100.00	100.00	100.00	100.00
19	0.00	1.33	2.58	4.31	6.19	8.71	11.51	15.54	20.60	30.12	40.57	53.62	67.23	79.49	88.87	95.01	98.30	99.59	99.91	99.99

	0.00	1.90	3.56	5.90	8.53	12.25	16.52	22.52	29.42	41.14	53.16	67.43	81.14	91.59	97.42	99.62	100.00	100.00	100.00	100.00
	0.00	3.57	6.73	11.18	16.12	23.13	31.41	43.38	56.58	74.65	86.93	95.28	99.04	99.99	100.00	100.00	100.00	100.00	100.00	100.00
	0.00	4.45	8.28	13.62	19.65	28.57	39.34	54.33	69.20	86.13	94.98	99.17	100.00	100.00	100.00	100.00	100.00	100.00	100.00	100.00
	0.00	5.35	9.85	16.19	23.58	34.62	47.52	64.04	78.54	92.45	98.18	99.93	100.00	100.00	100.00	100.00	100.00	100.00	100.00	100.00
20	0.00	3.18	6.09	10.26	14.97	21.72	29.63	40.82	52.94	69.68	81.73	91.05	96.47	98.80	99.44	99.62	99.85	100.00	100.00	100.00
	0.00	4.06	7.72	12.89	18.73	27.22	37.32	51.40	65.66	82.69	92.41	97.87	99.76	100.00	100.00	100.00	100.00	100.00	100.00	100.00
	0.00	4.77	8.97	14.94	21.86	32.10	44.13	59.94	74.50	89.57	96.62	99.56	100.00	100.00	100.00	100.00	100.00	100.00	100.00	100.00
	0.00	5.62	10.38	17.19	25.22	37.15	50.70	67.31	81.11	93.60	98.47	99.93	100.00	100.00	100.00	100.00	100.00	100.00	100.00	100.00
	0.00	5.96	11.24	19.11	28.53	42.17	56.85	73.51	86.13	96.21	99.44	100.00	100.00	100.00	100.00	100.00	100.00	100.00	100.00	100.00
21	0.00	3.01	5.80	9.83	14.35	20.79	28.32	39.06	50.93	67.80	80.38	90.41	96.40	99.01	99.72	99.89	100.00	100.00	100.00	100.00
	0.00	3.96	7.55	12.63	18.36	26.63	36.44	50.14	64.16	81.29	91.44	97.41	99.61	100.00	100.00	100.00	100.00	100.00	100.00	100.00
	0.00	4.86	9.09	15.02	21.77	31.68	43.29	58.65	73.01	88.35	95.92	99.35	100.00	100.00	100.00	100.00	100.00	100.00	100.00	100.00
	0.00	5.61	10.38	17.11	24.90	36.36	49.44	65.74	79.67	92.77	98.14	99.89	100.00	100.00	100.00	100.00	100.00	100.00	100.00	100.00
	0.00	5.98	11.09	18.58	27.48	40.49	54.79	71.50	84.58	95.49	99.23	100.00	100.00	100.00	100.00	100.00	100.00	100.00	100.00	100.00
22	0.00	2.85	5.49	9.30	13.59	19.68	26.79	36.94	48.32	65.05	78.10	89.05	95.90	98.99	99.82	99.94	100.00	100.00	100.00	100.00
	0.00	4.79	8.95	14.83	21.58	31.53	43.20	58.63	73.03	88.35	95.86	99.27	99.99	100.00	100.00	100.00	100.00	100.00	100.00	100.00
	0.00	6.24	11.52	19.14	28.18	41.36	55.73	72.29	85.05	95.55	99.17	100.00	100.00	100.00	100.00	100.00	100.00	100.00	100.00	100.00
	0.00	7.26	13.61	23.09	34.32	49.85	65.30	81.01	91.34	98.21	99.88	100.00	100.00	100.00	100.00	100.00	100.00	100.00	100.00	100.00
	0.00	8.43	15.96	27.31	40.36	57.32	72.80	86.93	95.00	99.35	100.00	100.00	100.00	100.00	100.00	100.00	100.00	100.00	100.00	100.00
23	0.00	2.96	5.62	9.37	13.55	19.44	26.28	36.01	46.92	62.98	75.59	86.30	93.19	96.52	97.59	97.86	98.08	98.52	99.10	99.64
	0.00	4.16	7.80	12.88	18.58	26.93	37.02	51.23	65.76	83.16	93.01	98.34	99.95	100.00	100.00	100.00	100.00	100.00	100.00	100.00
	0.00	5.27	9.68	15.82	22.92	33.49	45.87	61.94	76.37	90.80	97.24	99.73	100.00	100.00	100.00	100.00	100.00	100.00	100.00	100.00
	0.00	5.80	10.64	17.56	25.74	37.89	51.67	68.46	82.27	94.47	98.93	100.00	100.00	100.00	100.00	100.00	100.00	100.00	100.00	100.00
	0.00	6.52	11.96	19.81	29.17	42.85	57.67	74.55	87.30	97.28	99.98	100.00	100.00	100.00	100.00	100.00	100.00	100.00	100.00	100.00
24	0.00	2.90	5.61	9.48	13.81	19.91	26.99	37.08	48.32	64.77	77.58	88.40	95.35	98.66	99.66	99.85	99.97	100.00	100.00	100.00
	0.00	3.99	7.52	12.41	17.83	25.67	35.13	48.69	63.01	80.91	91.56	97.66	99.76	100.00	100.00	100.00	100.00	100.00	100.00	100.00
	0.00	4.68	8.70	14.21	20.48	29.89	41.33	57.05	72.15	88.40	96.22	99.52	100.00	100.00	100.00	100.00	100.00	100.00	100.00	100.00
	0.00	5.22	9.66	15.89	23.18	34.21	47.25	64.04	78.75	92.69	98.29	99.94	100.00	100.00	100.00	100.00	100.00	100.00	100.00	100.00
	0.00	5.38	10.04	16.85	25.11	37.60	51.90	69.24	83.24	95.12	99.19	100.00	100.00	100.00	100.00	100.00	100.00	100.00	100.00	100.00
25	0.00	1.43	2.84	4.87	7.14	10.30	13.86	18.70	24.11	33.27	43.20	56.23	70.55	83.48	92.64	97.67	99.64	100.00	100.00	100.00
	0.00	3.28	6.34	10.67	15.47	22.31	30.50	42.53	55.91	74.22	86.61	95.08	98.97	99.99	100.00	100.00	100.00	100.00	100.00	100.00
	0.00	4.11	8.07	13.87	20.64	30.67	42.45	58.08	72.72	88.30	95.89	99.31	100.00	100.00	100.00	100.00	100.00	100.00	100.00	100.00

	0.00	5.35	9.95	16.65	24.70	36.80	50.58	67.41	81.30	93.75	98.54	99.94	100.00	100.00	100.00	100.00	100.00	100.00	100.00	100.00
	0.00	6.23	11.72	19.86	29.61	43.66	58.57	75.11	87.28	96.67	99.58	100.00	100.00	100.00	100.00	100.00	100.00	100.00	100.00	100.00
26	0.00	1.74	3.32	5.55	8.05	11.60	15.73	21.58	28.19	38.88	49.39	61.85	74.50	85.50	93.35	97.94	99.85	100.00	100.00	100.00
	0.00	2.39	4.65	7.89	11.52	16.56	22.24	30.18	39.26	53.88	67.20	80.64	91.12	97.26	99.60	100.00	100.00	100.00	100.00	100.00
	0.00	2.87	5.54	9.37	13.64	19.57	26.37	36.12	47.31	64.42	78.18	89.76	96.75	99.55	100.00	100.00	100.00	100.00	100.00	100.00
	0.00	3.38	6.49	10.88	15.71	22.49	30.45	42.02	55.02	73.27	86.02	94.91	99.00	99.99	100.00	100.00	100.00	100.00	100.00	100.00
	0.00	3.86	7.40	12.33	17.77	25.50	34.67	47.80	61.83	79.83	90.89	97.45	99.76	100.00	100.00	100.00	100.00	100.00	100.00	100.00
27	0.00	1.35	2.66	4.53	6.66	9.67	13.13	18.05	23.88	34.15	44.95	58.16	71.69	83.44	91.70	96.38	98.54	99.40	99.76	99.92
	0.00	2.23	4.33	7.39	10.84	15.64	21.06	28.81	38.08	53.62	67.81	81.59	91.70	97.32	99.46	99.95	100.00	100.00	100.00	100.00
	0.00	3.06	5.86	9.84	14.23	20.31	27.35	37.64	49.59	67.43	80.86	91.10	96.55	98.48	98.72	98.72	98.83	99.17	99.52	99.79
	0.00	3.73	7.09	11.78	16.93	24.19	32.85	45.48	59.39	77.86	89.55	96.61	99.13	99.47	99.47	99.48	99.59	99.73	99.88	99.99
	0.00	4.45	13.59	19.44	27.93	36.72	52.55	67.26	84.67	94.18	98.92	99.98	100.00	100.00	100.00	100.00	100.00	100.00	100.00	99.73

Appendix 6: Test conditions for the sampling campaign at Waterval Concentrator

	MIG IsaMill A	MIG IsaMIll B
Volumetric flowrate (m3/h)	139.8	148.6
Solids concentration	48.6	45.7
Solids flowrate (tph)	100.42	98.20
Slurry density	1.48	1.45
80% passing feed size (F_{80})	105	105
80% passing product size (P_{80})	60	65
Power (kW)	2123.6	2215.7
Specific energy (kWh/t)	15.19	14.91

Appendix 7: Particle size distribution results for the sampling campaigns at Waterval Concentrator

Size (µm)	IsaMill A feed	IsaMill A product	IsaMill B feed	IsaMill B product
1400	100.00	100.00	100.00	100.00
1000	100.00	100.00	100.00	100.00
710	100.00	100.00	100.00	100.00
500	99.99	100.00	99.99	100.00
355	99.92	100.00	99.94	99.98
250	99.31	99.97	99.48	99.95
180	96.16	99.73	96.77	99.77
125	86.85	98.34	88.16	98.70
90	70.85	92.82	72.97	93.90
63	50.90	79.29	53.55	81.27
45	35.96	64.20	38.52	65.98
32	25.42	52.18	27.48	51.70

Appendix 8: Test conditions for the sampling campaign at Western Limb Tailings Re-treatment Plant

	Low flowrate	Medium flowrate	High flowrate
feed flow (m3/h)	76.19	81.19	99.73
feed density (t/m3)	1.35	1.34	1.35
feed t/h	38.75	39.99	43.46
Power (MW)	1.96	1.92	1.95
80% passing feed size (F80)	44.34	39.25	32.73
80% passing product size (P80)	24.00	25.39	24.98
Specific energy (kWh/t)	51.78	47.82	39.34

Appendix 9: Particle size distributions for the western Limb Tailings Re-treatment Plant sampling campaigns

size (µm)	Low flowrate feed	Low flowrate product	Medium flowrate feed	Medium flowrate product	High flowrate feed	High flowrate product
1200.00	100	100	99.91	100	100	100
850.00	100	100	99.81	100	100	100
600.00	100	100	99.67	100	100	100
425.00	99.39	100	99.43	100	100	100
300.00	98.57	100	99.02	100	99.9	100
212.00	98.44	99.94	98.48	100	99.53	100
150.00	98.43	99.87	97.96	99.92	99.05	100
106.00	97.13	99.8	97.55	98.97	98.61	99.73
75.00	93.09	99.31	94.84	97.82	97.78	99.2
53.00	85.73	97.2	89.11	94.65	93.8	96.44
38.00	75.81	92.43	79.17	89.98	85.87	90.57
25.00	60.81	81.69	63.51	79.69	71.4	80.04
18.00	48.63	69.91	50.73	68.47	58.19	68.65
12.50	34.83	53.06	36.31	52.45	42.07	52.51

9.00	26.56	41.18	27.69	41.11	31.96	41.15
6.25	18.11	27.91	18.87	28.34	21.44	28.42
4.50	12.21	18.32	12.69	18.96	14.12	19.09
3.12	7.07	10.19	7.34	10.85	7.92	10.99
2.25	3.55	4.96	3.72	5.5	3.9	5.6
1.00	0	0	0	0	0	0

University of Cape Town

Faculty of Engineering & the Built Environment

Certificate of Corrections

CANDIDATE	
STUDENT NUMBER	
THESIS TITLE	
DEPARTMENT	

I/We, the undersigned supervisor/s, hereby certify that the above-mentioned candidate has completed the corrections to the Masters dissertation to my/our satisfaction in accordance with the recommendations of examiners.

A schedule of the completed corrections is attached. Where any corrections have not been made, an explanation is given.

One unbound copy and cd-rom of the dissertation are returned herewith and may now be deposited in the Library, subject to the Dean's approval of the abovementioned corrections.

Print your name and sign

	Name	Signature	Date
Supervisor 1			
Supervisor 2			
Head of Department			
Approved by the Dean			

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03 November 2011

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Dear Mr Chaponda

The Examinations Committee of the Faculty of Engineering & the Built Environment completed its assessment of the examiners' reports on your Masters dissertation and the Chair requested me to inform you of the Committee's decision.

The Committee resolved that the dissertation be passed subject to you (1) revise the unbound copy of your dissertation under the direction of your supervisor, incorporating the changes required by the examiner(s) as specified in the relevant report(s); (2) submitting the revised copy to this Office together with a short report summarising the changes made. If the changes are largely corrections of spelling or typographical errors etc. then a short paragraph confirming completion of such corrections will be sufficient. If, however, the changes involve more substantial revisions then the report should be longer (one or two pages) and must describe these changes and explain their significance; (3) submitting one electronic version of your dissertation in PDF format on CD rom for the Library. Your report must be signed by yourself and attached to the certification of corrections form which has been submitted to your supervisor.

Please note that the following requirements must be satisfied in order for you to be considered for graduation in **December 2011**:-

(i) one unbound, revised copy of your dissertation must be submitted to the Faculty Office together with the report on your revisions and one CD rom by not later than **Thursday, 10 November 2011 at 14h00;**

(ii) you are required to submit a summary of the key aspects of the dissertation in the form of a Paper which is, potentially, of publishable standard and endorsed by your supervisor by no later than 30 April (in the hope of June graduation) or 10 November 2011 or as soon as possible (in the hope of December graduation).

We also ask you to note that in terms of Faculty policy, a candidate who is required to correct/revise his/her dissertation must do so within one year of the date of having been informed of this requirement.

Yours sincerely

Sandra Naidoo

Ms Sandra Naidoo
Postgraduate Assistant
For Professor FW Petersen
Dean of the Faculty of Engineering & the Built Environment
d18(a)

IMPORTANT NOTES:

1. The University will not permit degree/diploma qualifiers to graduate if they have any outstanding University fees, fines, interest or dues. The date for payment of outstanding amounts is 30 April in the case of qualifiers for June graduation and 31 October in the case of qualifiers for December graduation.

2. Information on the graduation ceremony is sent to finalists in the first week of May (in the case of those expecting to graduate in June) and the first week in October in the case of those expecting to graduate in December. Contact the Students Records Office, (021) 650 5986/7, if you have any queries about the graduation ceremony or if you have not received this information, or click on http://www.uct.ac.za/students/graduation/

cc Head of Department, Chemical Engineering

Overall View

In my view the student has met the requirements for the award of the degree of MSc (Eng) by the University of Cape Town as specified in the "Guidelines for Examiners of Masters' Dissertation" made available to me.

There are however, some editorial and other more substantive matters that the candidate needs to address to the satisfaction of the Internal Examiner BEFORE the award can be made, and these are elaborated below. I have also posted the hard copy of the thesis I had access to, into which I hand-wrote my comments which the candidate can use to effect the corrections as the comments will be easier to follow.

By way of a good conclusion to this interesting work, **it would have been good to apply the generated regression analysis models to independent data to establish their usefulness, and expand the number of full scale data sets collected.**

Corrections

1. Page (ii) para 2: move para 3 to join para 2

2. Page (ii) para 3 line (1) "This study, has focused only on ...... product fine as" Justify the statement.

3. Page (ii) para 4 , the last sentence should be pushed to the next para.

4. Page (iii) para 5 insert "while the" and move para 5 to join para 4.

5. Page (iii) para 1: line 1: " .. test work has... delete 'has' and replace with 'have'

6. Page (iii) para 1 line 8: The last sentence is a conclusion which should be recognized as such in this text.

7. Page (iii) para 3 second sentence, delete entails and replace with 'suggests'.

8. The conclusion of the abstract should be a bit stronger than what is currently given.

9. Page (ix) Figure 3-15 use a capital V, in vibratory....

10. Page 1: line 3 the location of "Mount Isa" must be stated.
Line 5 ".... grinding application ...insert 'elsewhere'

11. Page 1: Last two sentences should be moved to the Abstract

12. Page 2: para 1: The third sentence is not clear it should be recast.

13. Page 2: para 3 line 3: Second sentence : reference to a photo, or diagram of the IsaMill given somewhere in the text is necessary here for the reader to relate to.

14. Page 3: para 1: line 3 change provides to read 'providing'.

15. Page 3: para 2: line 1 "... platinum industry ... " add 'in SA'

16. Page 3: para 2: line 3 remove "...and therefore, requires.." to read requiring.

17. Page 3: para 2 line 4 "... lock..." to read locked.

18. Page 3: para 2 line 9 remove "...the...", replace with 'facilitate'

19. Page 3: para 3 line 1 include: "**In this work**, studies to investigate …."
20. Page 4: para 1 to read dominant instead of "… dominating
21. Page 4: para 2 line 5: "… are large enough to have…." include 'a' after 'have'
22. Page 4: para 2 line 6: "…effectively caught and be…" remove **be**
23. Page 4: para 3 bullet point 1 line 2: "…media size…" add "for optimum operation"
24. Page 6: para 1 line 2: remove better, replace 'good'
25. Page 6: para 1 line 3: "…..Stirred mills are discussed in greater depth…. " include 'by'
26. Page 6: para 2 line 2: "……..a product size with desired liberation characterists…" include 'coma'
27. Page 6: para 2 line 2 Therefore, different remove Therefore and include capital 'D' in different……..
28. Page 6: para 2 line 3 "…..operating conditions" remove 'will' ……….
29. Page 6: para 3 line 1 "The expansion in the minerals"remove the and include capital E 'expansion'
30. Page 6: para 3 line 1 "…..processing of ores …"remove with and include 'that have'
31. Page 6: para 3 line "….liberation and improved recovery of …. " include 'these finely disseminated'
32. Page 6: para 3 line 7 "……(Napier-Munn et al., 1999) However, they are extremely ……" include 'coma' after (1999) and remove capital H to small 'h'
33. Page 6: para 3 line 8 " …….utilization and could not, therefore, be ……"include 'economically'
34. Page 7: para 1 line "……research to investigate technologies that" remove could and replace 'can'

35. Page 7: para 2 line 11 "……machine and operating variables that can" remove assist in achieving and replace 'be used to deliver the' ……."benefits obtained" remove in to replace 'from' ……"fine grinding" include 'coma'

36. Page 7: para 2 line 12 remove achieving and replace deliver the, same line remove in and replace from and coma

37. Page 8: para 4 line "……energy required to produce new surfaces." remove However, and include capital T to 'There'

38. Page 17: para 2 line 1 "……The horizontal stirred mills uses…." include 'a'

39. Page 17: para 2 line 6 "……..as high as the mechanical design of the mill……"include coma after 'mill'

40. Page 19: para 4 move to join para 3

41. Page 20: para 1 line 1 "….. in grinding" include, 's' to mill

42. Page 21: para 2 line 4 The sentence is confusing re cast

43. Page 22: para 1 line 2 "….particles are also alleged to have many" include 'more'

44. Page 22: para 1 line 6 – 11, 'sentence incomplete'

45. Page 22: para 2 line 1 – 2, 'incomplete'

46. Page 23: para 1 line 1 – 6, 'not well written it seems to be a repeat – recast'

47. Page 23: para 1 line 14 – 16 of last para, 'how does this affect the energy?'

48. Page 24: para 1 line 5, needs a definition

49. Page 25, on the diagram "Surface Area m" of what?

50. Page 27: para 1 line 1, "energy efficiency" why?

51. Page 29: para 2 line 8, "…..inError! Reference source not found"correct

52. Page 35: para 1 line 4 – 5, "…..grinding media to capture effectively the particles" not clear

53. Page 37: para 2 line 4 "…….(Kwade, 1999)." include, in the low size range

54. Page 37: last para, "There is no strong basis for the choice of the two parameters"

55. Page 39: para 3 line 4 "……is significantly higher than that" include 'of'

56. Page 39: para 4 line 1, "…..It is" remove alleged replace 'stated'

57. Page 40: para 1 line 8"……the density were running as close to a" 'steady state as possible' was this timed?

58. Page 44: para 2 line 6, "…… media load and also to introduce the" 'top size' meaning?

59. Page 46: para 1, diagram, 'improve the quality of the photos'

69. Page 46: para 2, line 6 – 7 "…Thereafter, the mill was started up and material introduced into the mill for four passes. The mill was then started up and the feed" 'clarify'

70. Page 49: para 1 line 2 "…. In product from the complete pass" remove can replace 'could'

71. Page 49: para 3 line 5, "….. four passes constituting a test." 'Standard deviations,' how?

72. Page 50: para 1 line 6 "…..conducted at the following stirrer speeds; 1500rpm, 1800rpm and 2100rpm." 'What was the figure obtained?'

73. Page 53: para 2 the diagram. Not properly accounted for

74. Page 53: para 3, what of the de-chipping?

75. Page 55: para 3, diagram, which is on this graph?

76. Page 61: para 2 line 2, procedure

77. Page 71: para 1 line 8 remove little and replace limited

78. Page 85: para 1 line 7 remove inherently to read inherent

79. Page 89: para 2 line 3 mill to read mills

80a. Page 106: The constants in equation 5.2 do not match those given in Table 5.6 why?

80b. Page 106: **It would have been good to apply the generated regression analysis models to independent data to establish their usefulness.**

81. Page 107: para 4 line 1 remove project and replace study
82. Page 108: para 1 line 7 include an

83. Page 109: para 2 line 1, which is what? Restate it here

84. Page 109: para 2 line 8, remove project, replace study

85. Page 109: second last page, You are stating the obvious here not deducing anything from the data. Reconsider this point.

86. Page 110: para 1 bullet 1; so what does this mean?

87. Page 110: para 2 line 1, remove project replace work

88. Page 110: last para, line 1 remove tests

89. Page 113 on Reference, change font size

90. Page 137: Appendix 7 should go on the next page

Examination of Masters dissertation : Brian Chaponda
Dissertation title : Effect of operating variables on isamill performance using platinum bearing ores.

The parameters investigated are :
- Stirrer speed
- Grinding media size and load
- Feed size distribution
- Solids concentration
- Feed flowrate

The main findings are :
- For UG2 ore with a F80=120µm, the optimum speed is about 1800 rpm for the LM4
- The optimum media load depends on feed particle size to grind the ore to desired product.
- The optimum media size depends on feed size :
  If F80 = 55 to 120 µm then 2 mm media size is the optimum one
  If F80 = 250 µm then 3.5 mm media size
- The investigation on solids concentration effect leads to a better grinding efficiency when slurry viscosity increases. This should be also highlight in the conclusions 6.2.
- Good match about energy requirement between M4 and M10000. I will come back on this issue later.

<u>See 2.4.1 p18</u> : It will be better to modify the general feature of stirred mills. Indeed, if it is true that the fundamental differences between vertical and horizontal stirred mills are in the way grinding media and the ground product is separated; however, we could see high stirrer tip speed in both configurations. The author should distinguish :
- Low speed mill such as Tower mill (tip speed at about 3 m/s; vertical stirred mills)
- High speed mill (tip speed>10m/s) with vertical shaft such as detritor mill and with horizontal shaft such as Isa mill.

About separation system, detritors are designed with screens. The separation system is also specific for each configuration (vertical or horizontal shaft)

<u>See 2.4.3 p24 :</u> *"No mention is made on what would happen if, for example similar media sizes were used for tower mill and Isa mill"*
The author shows a thorough understanding of the scientific method and an adequate acquaintance with the relevant literature in the dissertation. He should also indicate that all grinding devices have physical limitation in media size. The author indicated about media size restriction in tumbling mill! It will be also important to indicate in the text the restriction for Tower mill and other mills (detritor, isa mill). It is not physically possible to use a media size of 1 mm in tower mill for instance. There for, media size limitation in each grinding comminution device could not allow the comparison as indicated in the author's sentence.

<u>See 2.5.2 p 27</u> : I highlight the Author's comment on figure 2-5. It is really great to show sometime to time the contradiction between different results on same parameters investigation. It keeps thinking about it and the complexity of such research.

See 2.5.6 p34 : *" Increases in solids concentration tend to increase the specific energy consumption required for a comminution process"*
I am very surprise!!! This affirmation should be cross-checked as this is not what I red and saw many times at pilot plant and in the industry. There is always an optimum for a specific grinding target. Is this sentence not taken out of the context?

See 2.5.6 p35 Line 4 : I recommend when putting number of solids content (%) to indicate clearly if it is percentage in weigh or in volume? Generally speaking, it makes big difference for a specific ore and when comparing different ore. % in volume is maybe more relevant than anything else.

See 2.6 Summary of literature review.
The Author showed a very good understanding of what has been written on this subject. The no-mature technology is highlighted clearly.

See 3.2.3 p44 : Table 3-1 should be reviewed as there is a mistake in this table about media chemical composition. % Zr_2O_2 should be changed to % ZrO_2.

See 3.2.4 p44 : Media preparation
It is more a question than a comment: Why not preparing more media than needed in the preconditioning phase? For instance, load a volume of media at about 80% in volume and run with pulp at about 70% in volume. No need to add fresh media between pre-conditioning phase and the run with pulp.

See Table 3.2 p 48
I think that everything should respect the nomenclature p.xiii. Better to put "F80" instead of "P80" when referring to the feed. You will find the same "mistake" some time to time in the text. If possible, this should be reviewed for better reader's understanding.

See 4.1.2.3. : The author is investigating the influence or effect of media load on specific energy by using media of 3.5 mm and F80=120μm to grind to a product size up to 20μm. It is not really clear in the text when the author further investigates the effect of media load, which media sizes were used.
It should be noted also that if media size is far from the optimum one for such grinding duty, the results should be taken with care or at least the context well indicated in the text. It could require more investigation to see the effect of media load by testing with different media size for instance. Maybe, the workload is too much for such thesis. But the author could write such recommendation as a further work.

See 4.2.3 : Comparison of M4 and M10 000 Isamill performance.
The figure 4-32 shows a huge variation of results in M4. Does the author plot all the points from M4 with many variation of operating parameters? Scientifically, with such results showing a cloud of points and considering the log axis, I really do not agree with such conclusion that M4 and M10000 results fit together looking on those results!
To get a grind of P80 = 45 μm, M4 results show variation from 13 kWh/t to 60 kWh/t!!!

Will it not be possible to filter M4 results by taking into account operating conditions close to M10000 one and then conclude to a good fitting. Otherwise, this conclusion is more commercial than scientific!

In conclusion,
I find this Master degree dissertation as a good job undertaken by the Author:
- The literature review shows a very good understanding of this topic. The Author highlights also some contradiction in results, he is critical. This is very important when facing to new technology.
- The Author masters the methodology for M4 test work.
- The interpretation of results shows the ability of independent thought. Again, even if previous published paper doesn't confirm some outcome, the results are as they are.
- The Author is also using all relevant methodology to assess the interpretation.

Corrections made for Masters dissertation: Brian Chaponda

Title: Effect of operating variables on IsaMill^{TM} performance using platinum bearing ores.

1. Page(ii) Para 2: Paragraph 3 has been moved to join with paragraph 3
2. Page (ii) Para 3line (1): The statement has been changed to read; This study, however, has focused only on variables that influence specific energy consumption and the product fineness of grind.
3. Page(ii) Para 4: The last sentence has been moved to the next paragraph
4. Page (iii) Para 5: Inserted "while"…The statement now reads; While the results obtained from the test work……….. Paragraph 5 has been joined with paragraph 4.
5. Page (iii) Para 1, line 1: Deleted has…. And replaced with have… Statement now reads; …. test work have shown that when the IsaMill^{TM} mill is operated….
6. Page (iii) Para 1, line 8: Statement changed to read; This study has indicated that the optimum media load is dependant on feed particle size.
7. Page (iii) Para 3, second sentence: Deleted "entails" and replaced with "suggests"
8. Conclusion of the abstract should be a bit stronger: This has been changed to include the following sentences;
9. Page (ix) Figure 3-15: Capital "V" has been used in…. Vibratory….
10. Page 1, line 3: Location of Mount Isa has been included to read; …. at Mount Isa Mines in Australia….. Line 5: "elsewhere" has been inserted in statement to read; …..mainstream grinding application elsewhere.
11. Page 1: Last two sentences have been moved to become the first two sentences in the abstract.
12. Page 2, Para 1: Statement has been changed to read; Grinding is an energy-intensive operation and is the highest consumer of power in mineral processing applications (Tromans, 2007).
13. Page 2, Para 3, line 3: Reference to Figure 3- 1, the IsaMill^{TM} schematic has been included in the statement.
14. Page 3, Para 1, line 3: provides has been changed to read "providing"

15. Page 3, Para 2, line 1: South Africa has been added to read; ….platinum industry in South Africa….
16. Page 3, Para 2, line 3: Statement has been changed to read; …… gangue minerals requiring fine grinding…….
17. Page 3, Para 2, line 4: …lock… has been changed to read….locked…
18. Page 3, Para 2, line 9: ….the …. Has been replaced with ….facilitate …..
19. Page 3, Para 3, line 1: Statement has been changed to read; In this work, studies to investigate effects………
20. Page 4, Para 1: …dominating….has been changed to …..dominant….
21. Page 4, Para 2, line 5: "a" has been included to read…. large enough to have a high……
22. Page 4, Para 2, line 6: "be" has been removed. Statement now reads…… effectively caught and broken by the media.
23. Page 4, Para 3, bullet point 1, line 2: Has been changed to read ….. required media size for optimum operation
24. Page 6, Para 1, line 2: "better" has been replaced with "good"
25. Page 6, Para 1, line 3: "by" has been included to read….. greater depth by looking at…..
26. Page 6, Para 2, line 2: comma has been included in this statement; ….desired liberation characteristics, for the subsequent recovery process.
27. Page 6, Para 2, line 2: "Therefore" has been removed now statement reads: Different mill characteristics…
28. Page 6, Para 2, line 3: "will" has been removed. Statement now reads…. operating conditions affect the product…….
29. Page 6, Para 3, line 1: "The" has been removed to read: Expansions in the mineral industry……
30. Page 6, Para 3, line 1: "with" has been removed and replaced with "that have" in the statement…… of ores that have fine particle intergrowth……
31. Page 6, Para 3: Statement has been changed to read; …… recovery of these finely disseminated minerals (Becker *et al.*, 2001; Rule *et al.*, 2008).

32. Page 6, Para 3, line 7: Sentences have been combined to read…. accepted (Napier-Munn *et al.*, 1999), however, they……
33. Page 6, Para 3, line 8: "economically" has been included to read…. therefore, be economically used for……
34. Page 7, Para 1: "could" have been replaced with "can" to read…. technologies that can be applied……
35. Page 7, Para 2, line 11: statement re-written to read; …… operating variables that can be used to deliver the benefits obtained from fine grinding, when these units are applied in mainstream grinding.
36. Page 7, Para 2, line 12: Statement changed to read as above in 35.
37. Page 8, Para 4: "However" has been removed and statement now reads; There are many other energy……
38. Page 17, Para 2, line 1: "a" has been included in statement to read …. The horizontal stirred mills use a closed milling chamber…….
39. Page 17, Para 2, line 6: comma has been included in the statement ….. mechanical design of the mill, and can allow…..
40. Page 19: Paragraph 4 has been moved to join paragraph 3
41. Page 20, Para 1, line 1: "s" has been included to mill, statement now reads; … grinding mechanism in grinding mills under given conditions…….
42. Page 21, Para 2, line 4: Statement has been changed to read: The increase in energy consumption may be attributed to the fact that larger particles have a greater chance of being captured and broken than the smaller ones.
43. Page 22, Para 1, line 2: "more" has been included to the statement ….. alleged to have many more flaws that are broken…….
44. Page 22, Para 1, line 6-11: statement now reads: The profile of specific energy and product-size relationship, according to Wang and Forssberg (2007) is dependent on the design of the mill and the macroscopic conditions that operate relevant to the individual particle properties. The relationship between specific energy and product size is very vital in the design and evaluation of stirred mills. This plot shown in Figure 2-4 is referred to as a signature plot, is very important in assessing the performance of the mill under different grinding conditions

(Lichter & Davey, 2002; Weller & Gao, 1999). The specific energy and product size relationship remains constant during the process of mill scale-up and is unique to pulp conditions and the media selected (Curry *et al.*, 2005; Weller *et al.*, 1999). In this project, the signature plot will be used in assess the performances of the IsaMill^{TM} at different operating conditions.

45. Page 22, Para 2, line 1-2: Statement now reads: Although it is a common practice in the minerals industry to define the product fineness by the particle size at which 80 percent of the particle mass is smaller (P_{80}), it does not give a true representation of the mill product size distribution. Therefore, in some other grinding applications such as the paint and pharmaceuticals industry, the product fineness is defined with tight specifications such as maintaining a constant ratio of P_{98}/P_{80} (Curry *et al.*, 2005).
46. Page 23, Para 1, line 1-6: Confirmed as repeat, part of the statement re-written on page 22, paragraph 1.
47. Page 23, Para 1, line 14-16: New sentences has been included to clarify effect on energy efficiency; … The use of stirrers leads to higher power intensity and better energy utilization for the IsaMill^{TM}.
48. Page 24, Para 1, line 5: Reference has been added to define the statement…. more efficient than both the tower mill and ball mill (Pease *et al.*, 2006).
49. Page 25: New table has been inserted;

	Power Intensity (kW/m^3)	Media Size (mm)	No. Balls/m^3	Media Surface area (m^3)
Ball Mill	20	20	95, 500	120
Tower Mill	40	12	440, 000	200
IsaMill^{TM}	280	1	1, 150, 000, 000	3600

50. Page 27, Para 1, line 1: I have not seen anything wrong with the statement: The reduced energy efficiency could be attributed to the increased energy consumption with increasing stirrer speed.
51. Page 29, Para 2, line 8: Correct reference has been sourced; …… presented in Table 2-2.

52. Page 35, Para 1, line 4-5: Statement changed to read; For low solids concentration, (usually below 40% solids) there is large inter-particle distance making it difficult for the grinding media to capture effectively the particles (Gao *et al.*, 2007; Napier-Munn *et al.*, 1999).
53. Page 37, Para 2, line 4: Statement changed to include …" in the low size range" and now reads; ….. conventional tumbling mills in the low size range (Kwade, 1999).
54. Page 37, last paragraph: Reference has been included to strengthen the basis for choice of parameters: ….. are influenced by the stirrer speed and media size (Pease *et al.,* 2006).
55. Page 39, Para 3, line 4: "of" has been included in the statement ….. significantly higher than that of any other…..
56. Page 39, Para 4, line 1: Removed "alleged" and replaced with "stated" in the statement; It is stated that IsaMill^{TM} high energy……
57. Page 40, Para 1, line 8: Can not see what the examiner's is referring to here; the section is talking about stress energy and velocities and not densities as alleged by the examiner's statement.
58. Page 44, Para 2, line 6: "Top size" has been clarified in the statement; Fresh media amounting to the mass loss was then added to the conditioned media to maintain the required media load and also to introduce the media top size.
59. Page 46, Para 1: Quality of photo has been improved.
69. Page 46, Para 2, line 6-7: Second sentence was a repetition and has been deleted.
70. Page 49, Para 1, line 2: "can" has been replaced with "could" in the statement; ….. complete pass could be pumped back through….
71. Page 49, Para 3, line 5: New sentence has been added to clarify use of standard deviations; Lower standard deviations for the power drawn during each pass meant better stability in mill operation.
72. Page 50, Para 1, line 6: No load results have been included in the statement; The no-load tests that were conducted achieved the following results; 1.2kW, 1.0kW and 0.9kW for stirrer speeds 1500rpm, 1800rpm and 2100rpm respectively.

73. Page 53, Para 2: Reference has now been made to the diagram in the statement; … flow-sheet shown in Figure 3-7 consists of the….
74. Page 3, Para 3: The De-chipping cyclone and woodchip screen has now been discussed in the statement; The de-chipping cyclone together with the wood chip screen is used to screen out all the wood chips from the circuit.
75. Page 55, Para 3: Graph has been changed to include the legend
76. Page 61, Para 2, line 2: Line has been deleted and line 1 modified to give a clear picture of the procedure; In order to achieve a representative sample for the survey period, sampling for each test was conducted over a period of 60 minutes, and at 15-minute intervals.
77. Page 71, Para 1, line 8: "little" has been replaced with "limited" in the statement; …. limited media-particle interaction……
78. Page 85, Para 1, line 7: "inherently" changed to read "inherent"
79. Page 89, Para 2, line 3: "mill" has been changed to read "mills"
80. Page 106: Table 5-6 has been changed to match with constants in equation 5-2.

	Coefficients	Standard Error	P-value
Intercept	47.43	8.41	1.34E-05
Solids conc.	0.38	0.07	1.7E-05
Media size	2.42	0.65	0.00135
Media load	-0.20	0.06	0.004044
Stirrer speed	-0.02	0.00	7.89E-07
Feed size	0.03	0.01	0.000162

Regression analysis model application to independent data: This work was meant to validate the model; hence, the experiments conducted were limited assessing the effects of operational variables on IsaMillTM performance. However, it could be interesting conduct more work for model validation and I recommend that this be looked at for future research work.

81. Page 107, Para 4, line 1: "project" has been replaced by "study"
82. Page 108, Para 1, line 7; "an" has been included in the statement; … different levels for an efficient grinding process…..

83. Page 109, Para 1, line 2: The different speeds have been restated; …. different stirrer speeds (1500rpm, 1800rpm and 2100rpm) while…..
84. Page 109, Para 2, line 8: “project” has been replaced with “study”
85. Page 109, second last page: What seems like the “obvious” has been stated because it’s a tradition in the industry to use one media size and this is not usually changed or optimised when conditions of the feed in terms of particle size distribution changes.
86. Page 110, Para 1, bullet 1: This is a supporting statement for the need to optimise the media size based on feed size distribution outlined in the main bullet on Page 109.
87. Page 110, Para 2, line 1: “project” has been replaced with “work”
88. Page 110, last paragraph, line 1: “tests” have been removed, statement now reads; Mineralogical analysis should be performed…….
89. Page 113: Font size for references has been changed to 12
90. Page 137: Appendix 7 has been moved to the next page (Page 138).

All the comments made by the examiners have been considered and changes in the thesis document have been made accordingly.

- The issue of a detailed comparison of the IsaMill^{TM} with other stirred mills such as tower mills could not be done as this was not the focus of this thesis. I would recommend future work on the comparison of various stirred mills to assess their efficiencies and limitations.
- The conclusion made on solids concentration is based on the range of solids concentration tested in this study. If a wider range is investigated, probably the results would be different from those presented in this work. I recommend that future work investigate a wider range of solids concentration to ascertain its impact of mills performance.