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DEVELOPING A METHODOLOGY THAT INCORPORATES
CERAMIC MEDIA PROPERTIES TO MODEL POWER DRAW IN A
M4 ISAMILL™.

By

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A thesis submitted to the University of Cape Town in fulfilment of the requirements for a
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Faculty of Engineering and the Built Environment
Department of Chemical Engineering
University of Cape Town

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"Knowledge has to be improved, challenged, and increased constantly, or it vanishes"

-- Peter Drucker

University of Cape Town

DECLARATION BY AUTHOR

This thesis is composed of my original work and contain no material previously published or written by another person except where due reference has been made in the text.

I have clearly stated the contribution of others to my thesis as a whole, including experimental design, data analysis, original research work and professional editorial advice used or reported in my research work. The content of my thesis is the result of research work carried out since commencing my higher degree studies and does not include a substantial part of work that has been submitted to qualify for the award of any other degree or diploma in any other university or tertiary institution.

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Statement of contributions to jointly authored works contained in this thesis.

1. Farber BY, **Durant B** and Bedesi N, *Effect of media size and mechanical properties on milling efficiency and media consumption*. Minerals Engineering Volume 24, Issues 3-4, February-March 2011, Page 367-372
2. **Durant B**, Rule C and Farber B, *Ceramic media selection for optimization of energy efficiency in IsaMillsTM*. Proceedings of the 7th Fine Grinding and Dispersion symposium held in Braunschweig Germany, 12-13 October 2011.

Statement of contribution by others to the thesis as a whole.

Dr Boris Farber, Senior Scientist of Zircoa Inc. USA played a significant role in the conception of the project of this thesis and acting as external advisor with respect to the experimental program, data analysis and interpretation of findings.

The method development in this thesis was, except as acknowledged, carried out by me. The input data for this thesis was obtained with assistance of the following sources:

- Historical ceramic media evaluation data was supplied courtesy of Mr Chris Rule and Mrs Nurie Bedesi of Anglo American's platinum operations.
- Data generated for this thesis specifically was, done at the Divisional Metallurgical Laboratory with the assistance of Ruan Fouchee and Nathaniel Komane.
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SYNOPSIS

Sustainability in the mineral processing industry is guaranteed by the efficient operation of unit processes to maximize profitability within the prevailing economic conditions at the time. Efficient operation can be measured based on cost, metallurgical and energy efficiency of the operation or as a combination of the three, with energy efficiency becoming increasingly more important in the world and current South African context even more so.

With ore bodies becoming more complex and finely disseminated the need for ultra fine grinding to maximise mineral recoveries in PGM concentrating operations by means of mainstream grinding has become more important. Conventional grinding technology was found to be inefficient to address this challenge. Introduction of new stirred milling technology for mainstream grinding using ceramic media successfully addressed the challenge posed by finely disseminated and complex PGM ore bodies.

Introduction of new technology also brings about optimisation opportunities on various levels to ensure that the new technology is performing optimally in order to maximise profitability. As relatively new technology, several potential optimisation areas for the horizontal stirred mill has been identified for study.

Operating costs for IsaMills™ horizontal mills can be classified into 3 groups i.e. media, maintenance and energy cost²⁹. Ceramic media type used can potentially play a role in determining the cost contribution of each of these groups to the overall mill operating costs. An area found very important to study, taking into consideration the enormous strain on the South African electricity supply system, was the impact of ceramic media properties on the energy efficiency of the horizontal stirred mill.

Historical ceramic media test data^{5, 14, 17} indicated general trends between ceramic media properties and energy consumption. However the datasets available contained little data on the physical properties of the media to conclusively define the relative role of the media properties on energy usage.

Five (5) different ceramic media samples from various sources were selected for further subsequent tests in an attempt to decouple the effect of the various media properties on mill power draw. Various physical properties of the media evaluated were measured during the study. Physical properties that were measured were: friction coefficient, size, density, and aspect ratio of every media. M4 IsaMill™ tests were also conducted in water and slurry environments to determine the relative contribution of the media properties on energy consumption.

Analysis of historical data and results from the water-media study indicates that media mass and density contributes minimally if at all to the energy consumption of the M4 IsaMill™. Energy consumption in the M4 IsaMill™, operating in a water ceramic media environment, was found to be driven by a combination of media load and machine contributing factors. Machine contributing factors can be defined as the No Load Power (NLP) of the mill. Media load factor was found to be proportional to the product of the friction coefficient of the media, the total surface area of the grinding media charge and the aspect ratio of the media.

From the results, a methodology was developed to predict the power draw of the M4 IsaMill™ in a water-ceramic media system that incorporates the media properties contributing to energy consumption. The model was successfully applied to a separate set of tests, with the average error in the predicted M4 IsaMill™ power draw being less than 2% of the measured mill power.

The model developed in a water-media system was modified to incorporate the friction coefficient measured in slurry and used to predict the power draw of the M4 IsaMill™ during slurry tests. The prediction of power using the modified model was inaccurate because ore particle properties and the energy requirements for particle breakage were not incorporated in the modified water model. Stress Intensity and stress number is proportional to specific energy required for breakage in horizontal stirred mills³³. As the ore and ceramic media properties will influence the stress intensity and the stress number the slurry model was expanded to incorporate a breakage energy component.

After fitting breakage (k_b) and media charge (k_c) constants to the model fairly accurate prediction of the power draw could be accomplished. Attempts were made to validate the model at lower volumetric media filling degrees with varying success. At very low media loads (50% less than model development data), the prediction was very inaccurate and based on the work from other authors⁴³, it was clear that the model requires further development.

Results of the study and the methodologies developed serves as a good foundation for future research into defining and quantifying the impact of ceramic media properties on all areas of horizontal stirred mill operation. The methodology also indicates promise in decoupling the factors contributing to the energy consumption of horizontal stirred mills. Further work is required to define the impact of particle size on media friction coefficient, the overall impact of ore to media ratio, and the effect of slurry rheology on energy consumption of horizontal stirred mills.

It also serves as a good foundation for future scale up studies to test the wider application of the methodology on pilot and industrial scale mills. If scale up is reasonable the models could be potentially incorporated into a process control system to provide online monitoring of mill load conditions in the mill.

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NOMENCLATURE

AR_{GM}	Aspect Ratio of the Grinding Media
C_p	Media Density contribution constant
C_v	Volume Solids Concentration in Mill Product
d	Media diameter (mm)
d_{50}	Size at which 50% of the particles measured are smaller than the indicated size.
$E_m(E_v)$	Specific Energy (kwh/t)
F_{80}	Size at which 80% of the particles in the feed are smaller than the states size.
F_{c_d}	Dry Friction Coefficient of the ceramic media.
F_{c_w}	Friction Coefficient of ceramic media measured in water.
F_{c_s}	Friction Coefficient of media measured in UG2 slurry.
$F_{c_{a,b}}$	Friction Coefficient of Ceramic media A and B
GC	Grinding Chamber of the stirred mill
GM	Grinding Media
N_c	Number of Grinding Media contacts
N_{GM}	Number of Grinding Media particles in mill
N_p	Number of feed particles in stirred mill.
P_{80}	Size at which 80% of the particles are smaller than the stated size.
P_s	Probability that particles are entrapped between two particles or media and sufficiently stressed at media contact.
SA_T	Total Surface Area of grinding media in grinding chamber.
SG	Specific Gravity
SI	Stress Intensity
SI_{GM}	Stress Intensity of the Grinding Media
SN	Stress Number
t	Time (min or sec)
u_s	Particle Sliding Coefficient.
v	Disc or agitator tip speed

$W_{a,b}$ Power draw of mill with Ceramic media A or B

GREEK LETTERS

ρ_{GM} Density of the ceramic grinding media.

$\rho_{a,b}$ Density of ceramic media A or B

Φ_{GC} Filling ratio of the media in the mill grinding chamber.

ε Porosity of the bulk of the grinding media

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LIST OF ABBREVIATIONS

Al	Alumina Media
ATZ	Alumina Toughened Zirconia media
CFD	Computational Fluid Dynamics
DEM	Discrete Element Method
DML	Divisional Metallurgical Laboratory
FCTR	Flotation Characterisation Test Rig
GZ	Glass Zirconia Media
MIG	Mainstream Inert Grinding
MIM	Mount Isa Mines
NLP	No Load Power
PEPT	Positron Emission Particle Tracking
PGM	Platinum Group Minerals
PSD	Particle Size Distribution
Stab Zr	Stabilised Zirconia Media
USA	United States of America
WLTR	Western Limb Tailings Retreatment Plant
ZrSi	Zirconium Silicate Media
ZTA	Zirconia Toughened Alumina Media.

CHAPTER 1 - INTRODUCTION

Sustainability in the mineral processing industry is guaranteed by the efficient operation of unit processes to maximize profitability within the prevailing economic conditions at the time. Efficient operation can be measured based on cost, metallurgical and energy efficiency of the operation or as a combination of the three. Energy efficiency of processes are becoming increasingly more important in the world and in the current South African context even more so.

Over the last two decades, the need for ultra fine grinding has become more important, due to the increasing complexity of ore bodies. The finer degree of dissemination of valuable minerals within the ore bodies requires finer grinding to be able to achieve economic degrees of liberation. X-Strata Technologies (formerly Mount Isa Mines Technologies) ²¹ successfully developed and implemented horizontal stirred milling technology to address the need for finer economically viable grinding.

In the recent past, Anglo American's platinum operations³³ successfully commissioned twenty two (22) horizontal stirred mills (IsaMills™) in a Mainstream Inert Grinding (MIG) application to improve recovery of Platinum Group Minerals (PGM) from its South African concentrator operations. They thus established the world's largest base of horizontal stirred mills. The aim of the technology was to ensure a finer final grind to secondary/scavenger flotation circuit, resulting in improved PGM recovery as a result of the improved degree of liberation in the feed to the circuit.

A major challenge with implementation and sustainability of new technology is to ensure optimum operation and to minimize operating costs after commissioning to maximize profitability in the shortest possible time. As the application of horizontal stirred milling in the coarse grinding application is relatively new, several areas for optimization of the technology has been identified that directly influence the unit operating cost.

Application of ceramic media for grinding minerals in a mainstream milling application is relatively new, but now standard practice. In the past, naturally occurring media such as silica sand or river pebbles was favoured as grinding media due to the high comparative cost of ceramic media. Constraints with respect to natural media supply, a decrease in ceramic media supply price as well as the successes in other industries such as the paint and pigment industries has initiated the conversion to ceramic media in horizontal stirred mills. Ceramic media, mainly alumina and zirconia toughened alumina media have been in use in horizontal stirred mills for the last couple of years. However several other types of ceramic media are available commercially and have to be given consideration with respect to optimisation of the horizontal stirred mill cost and performance.

The operating cost for horizontal stirred milling can be divided into three (3) cost contributing variables i.e. ceramic media, maintenance and electricity cost. Media cost forms a major portion of the overall operating cost but can potentially have a direct influence on maintenance and electricity cost as limited research has indicated^{16, 17}. As part of the horizontal stirred mill research initiative, the opportunity was identified to assess the effect of ceramic media type on energy consumption in horizontal stirred mills on a laboratory scale.

The key questions to ask are:

1. Do ceramic media properties influence the energy consumption of horizontal stirred mills?
2. Can ceramic media properties be used to differentiate between a “good” and bad media for milling in horizontal stirred mills with respect to energy consumption?

Farber et al (2008,2010)^{14,15} indicated that energy consumption is not directly related to media density, but is largely dependent on the friction coefficient of the media surface, when comparing the energy consumption of an alumina toughened zirconia media (ATZ) with that of a zirconia toughened alumina media (ZTA) in a M4 IsaMill™.

Therefore the hypothesis of this study is:

- 1. Surface and Physical properties of the grinding media such as friction coefficient, size and shape, that influences the surface area of the grinding media, is the key drivers of energy consumption in a laboratory horizontal stirred mill.**
- 2. The contribution of media density to the energy consumption of a laboratory horizontal stirred mill is minimal or even negligible.**

This thesis presents the results of a study performed using a M4 IsaMill™ to determine the energy consumption of various media types with varying physical and chemical properties in water and in slurry. As part of the study, beyond investigating the hypothesis mentioned above, an attempt will also be made to develop a methodology to model power draw in an IsaMill™ that incorporates ceramic media properties.

CHAPTER 2 –LITERATURE REVIEW.

2.1 The Development of the horizontal stirred mill (IsaMill™) and its benefits to the minerals processing industry.

2.1.1 The development history of the IsaMill™.

In the 1970's, Mount Isa Mines (MIM) in Australia, investigated the potential for fine grinding in conventional ball and tower mills with very little success when grinds smaller than 25µm was required for economic liberation^{18,20,21,34}. The limited success with the conventional technology at the time was due to the energy consumption required to achieve the required grind being extremely high. Conventional grinding technology also exhibited a negative impact on downstream flotation recovery as a result of steel grinding media wear. The results from the initial investigations were also confirmed during pilot testwork in the 1980's, proving that sub 25 micron grinds could not be achieved economically, even with the application of tower mills²¹.

“Out-of-the-box” thinking was therefore required to find a solution to economically grind to sub 25 micron. Research initiatives was extended to other industries, where it was proven that ultrafine grinding is economically possible, to serve as starting point for the development of a new grinding technology. This ultimately led to the horizontal stirred mill technology employed in the pharmaceutical and food industries, where albeit at smaller scale, it was shown that grinding with fine media stirred at high speed was extremely efficient.

In 1991 a Netzch 0.5 litre bench scale mill was introduced to the MIM site and this event is today seen as the turning point in fine and ultrafine grinding for the minerals industry^{20,21}. In 1992 it was demonstrated at the MIM pilot plant that economic grinding to grinds as low as 7µm was possible using a Netzch LME 100 horizontal stirred mill. Pilot trials also indicated associated benefits in downstream flotation recovery with the implementation of high speed stirred milling operating with inert media in the pilot plant circuit^{20, 21}.

The next step was the development of the IsaMill™ as part of a collaborative agreement between MIM and Netzch Feinmahltechnik GmbH of Germany over a period of 7 years. The outcome of the collaborative program was a machine that addressed the two crucial concerns when the technology was originally identified:

- How to scale up the machines from the relatively small units employed in the other industries to a size suitable for the mineral industries?
- Conversion from a batch to continuous process while maintaining the process advantages of the technology.

Subsequent scale up tests at the Hilton and MIM Copper Zinc concentrators resulted in the implementation of the first 1.1MW IsaMill™ at Mount Isa Mines^{20, 21}. Commercialization of the new technology was transferred from MIM Ltd to MIM Process Technologies (now X-Strata Technologies) and the technology was introduced to the mining industry as a technology capable of economically grinding to sub 10 micron.

Up to this stage of development the technology was only proven for application in side stream grinding of concentrate arising from mainstream flotation circuits, otherwise referred to as Ultra Fine Grinding (UFG), as illustrated in figure 1. The next challenge for the development team was to find a way to introduce the same processing advantages identified to mainstream grinding at much coarser feed PSD's. In response, the first M10 000 IsaMill™ was developed and installed in 2003. In the 16 years since the first 1.1 MW installation, the installed capacity per unit increased an amazing 16 fold from the initial 205kW units to the 3MW units installed today as the M10 000 IsaMill™^{20,21}.

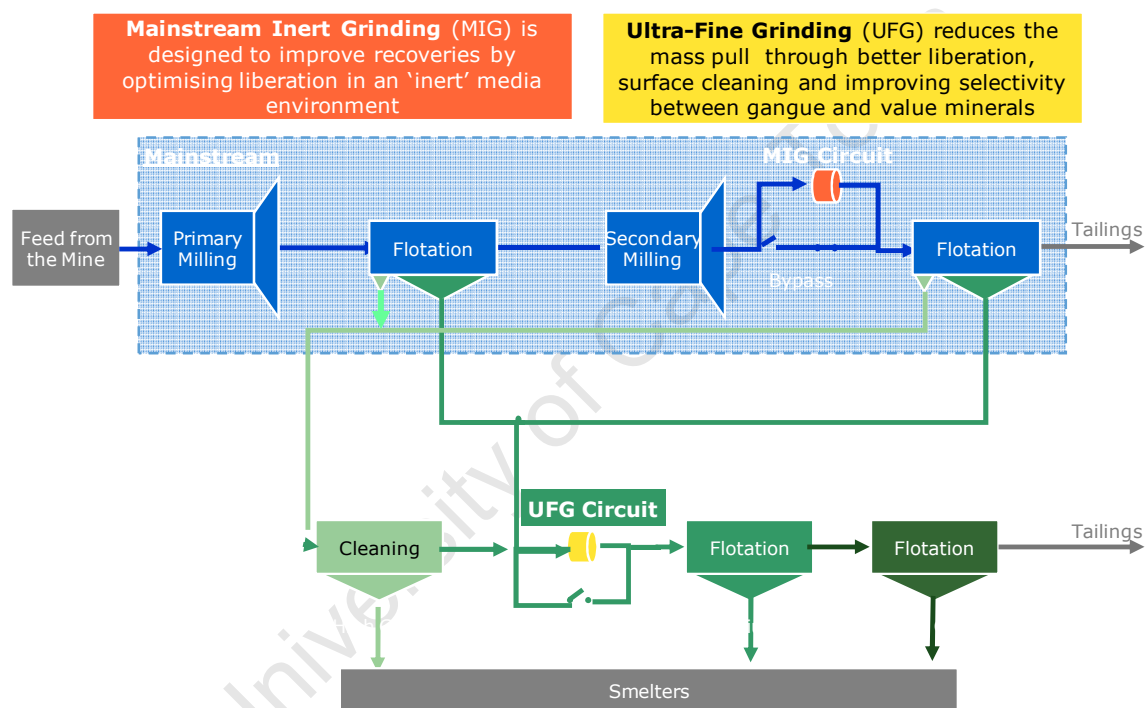


Figure 1: The application of IsaMill™ technology in the concentrator flowsheet (Rule).

By end 2010 there were 81 IsaMills™ in operation worldwide with a total installed power of over 130MW, treating feed sizes up to 250µm, with product sizes as low as 7µm being achieved. The technology has successfully been implemented in the copper, zinc, platinum, industrial minerals, iron and gold industries^{20, 21}.

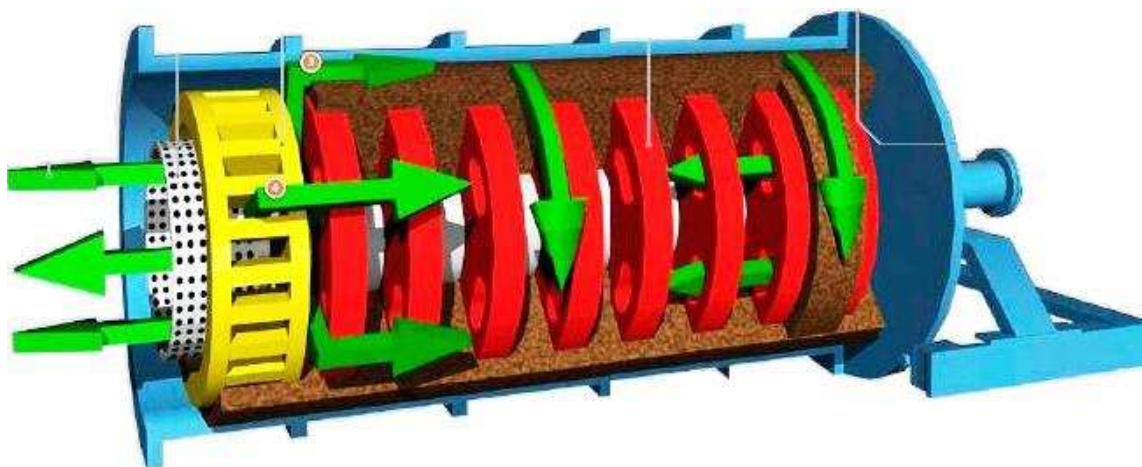


Figure 2: IsaMill™ internal flow pattern (XStrata Technology, 2009).

2.1.2 Benefits of the IsaMill™ technology to the minerals processing industry.

The IsaMill™ technology has the following distinct advantages when compared to other conventional grinding technologies available: ^{11, 14, 15, 18, 20, 21, 34, and 35.}

- High Intensity Grinding
- Improved Energy Efficiency
- Superior Product Size Distribution
- Improved Flotation Recovery.

The smaller footprint of the IsaMill™ and the high power installed results in a much higher power intensity, greater than $300\text{kW}/\text{m}^3$, when compared to other grinding technology as illustrated in Table 1^{20, 21}. Energy Efficiency is dictated by the grinding media size, as the smaller the media the more surface area is available for particle-media collisions to occur. The higher energy intensity of the IsaMill™ and the larger surface area of the media mean that small media can break coarse particles. The higher density of the ceramic media makes mainstream application of the technology possible

^{20, 21}.

The superior energy efficiency of the IsaMill™ can be seen in Table 2^{20, 21}.

Table 1: Comparison of the Power Intensity of different grinding technologies^{20, 21}

Technology	Mill Diameter (m)	Mill Length (m)	Installed Power (kW)	Mill Volume (m ³)	Power Intensity (kW/m ³)
AG Mill	10	4.5	6400	353	18
Ball Mill	5	6.4	2600	126	21
Regrind Ball Mill	3.2	4.8	740	39	29
Tower Mill	2.5	2.5	520	12	4
IsaMill™	2.0	4.7	3000	10	400
SMD	2.8(width)	6.0 (height)	355		150

Table 2: Comparison of the Energy Efficiency of different grinding technologies³⁷.

Technology	Power Intensity (kW/m ³)	Media Size (mm ³)	No of Balls (per m ²)	Surface Area (m ² /m ³)
Ball Mill	20	20	95,500	120
Tower Mill	4	12	442,000	200
IsaMill™	400	1	1,146,500,00	3600

A superior product size distribution is produced from the IsaMill™ due to the combination of the short residence time, plug flow, attrition grinding and the patented product separator²¹. The IsaMill™ operates by stirring fine media at high speeds (typical tip speeds of 20m.s⁻¹) and particles are broken by attritioning and abrasion. The patented product separator ensures that a sharp product size distribution is produced minimizing slime generation which in turn reduces reagent consumption^{20, 21}. Due to the elimination of slimes grinding or overgrinding of already sufficiently ground particles, energy wastage is reduced significantly in IsaMills™.

In return energy is utilized where it is needed the most i.e. to break coarse particles to the required size and is achieved due to the following reasons^{20, 21}:

- The small media size used favours attrition grinding.
- Grinding is essentially performed in 8 grinding chambers in series and short circuiting of material is much less than for other grinding technologies.
- Particles have to escape the high centripetal force of the product separator, which allows for internal classification of ore particles.
- The residence time is relatively short compared to other grinding technology. Average residence times of ninety (90) seconds are typical.

The final benefit of the technology is the major improvements in flotation recovery that were evident when incorporated into the flowsheet. This is due to the effects of milling occurring in an inert grinding environment. Therefore it is important to fully understand the impact of steel media on mineral systems in flotation^{19, 20, and 21}.

Electrical potential is assumed by a mineral or metal when immersed in water and the following 2 resultant reactions occurs:

Oxidation Reaction: Between minerals and water.
Reduction Reaction: Reducing of Oxygen.

When the mineral sulphide contacts with the steel grinding media in water, an electrochemical or galvanic cell is formed, where the element with the highest rest potential i.e. the steel media is the anode and the lowest is the cathode. The following reactions will then take place with steel media in water²²:

Oxidation Reaction: $Fe \rightarrow Fe^{2+} + 2e^- \dots \dots \dots [Eq. 2.1.1]$

Reduction Reaction: $4e^- + 2H_2O + O_2 \rightarrow 4OH^- \dots \dots \dots [Eq. 2.1.2]$

In other research^{8, 9, 14,19,21,35} it has been identified that steel grinding media can be detrimental to the flotation process for the following reasons:

- The reducing environment created lowers dissolved oxygen and Eh levels in slurry and collector adsorption onto mineral surfaces is compromised. For collector adsorption, oxidation of the xanthate to dixanthogen is required and the process is Eh dependent.
- Iron hydroxide coatings form on mineral surfaces reducing the selectivity of coarse and fine particles for flotation and this impact is enhanced as grinds become finer.
- Hydrophylic insoluble metal hydroxides precipitate on the mineral sulphide surfaces due to the reduction of mineral surfaces thereby rendering the sulphides unfloatable.

Therefore by grinding in an inert environment all 3 issues are addressed as there is no steel in the system that can contact with mineral surfaces.

2.1.3 Grinding Media Development by X-Strata Technology (ex MIM Process Technologies)

Curry and Clermont¹⁴ reported that media selection plays a major role on mill parameters such as energy efficiency, internal wear and operating costs. With the initial implementation of IsamillsTM, low cost, low quality media was used i.e. silica sand, river pebbles, smelter slag or primary mill scats.

The natural media generally exhibited poor energy efficiencies due to the angular nature of the media when compared to the energy efficiencies when operating on ceramic media¹⁴. The natural media used initially as stirred milling grinding media was silica sand or river pebbles. The natural grain size of the natural media also limited the maximum feed size that could be effectively ground in the stirred mill. Up to this stage of the mill development, the mill was still primarily installed in concentrate regrinding application.

Natural media was also predominantly used in the application as the low consumption rate, high energy efficiency and low unit cost for ceramic media was never possible, until recently. In the study¹⁴, several ceramic media types were classified based on media composition and the physical properties as presented in Tables 3 and 4.

The cost of ceramic media is closely linked to the Zirconia (ZrO_2) content of the media⁴, therefore the higher the Zirconia content the more expensive the media. The low cost alumina media was not found to be economical due to high consumption as a result of inconsistencies in the bead internal structure.

Table 3: Ceramic Media Classification by Composition¹⁴.

Media Type	Al_2O_3	SiO_2	ZrO_2	CeO_2	MgO	Y_2O_3
Alumina	>85%	Remainder	-	-	-	-
Yttrium Stabilised Zirconium Oxide	-	-	95%	-	-	5%
Cerium Stabilised Zirconium Oxide	-	-	80%	20%	-	-
Magnesium Stabilised Zirconium Oxide	-	-	97%	-	3%	-
Zirconium Silicates	-	31%	69%	-	-	-
Aluminium Silicates	34%	62%	-	-	-	-

Table 4: Physical Properties of different ceramic media types¹⁴.

Media Type	Bulk Density	Hardness (HV)	Fracture Toughness (K1C)
Alumina	2.0-2.1	1500-1700	3-5
Yttrium Stabilised Zirconium Oxide	3.6-3.8	1300-1400	13
Cerium Stabilised Zirconium Oxide	3.9-4.0	1100-1200	13
Magnesium Stabilised Zirconium Oxide	3.2-3.4	900-1100	6
Zirconium Silicates	2.2-2.4	600-800	3
Aluminium Silicates	1.7-1.8	800-900	3

Keramax[®] MT1 ceramic media was developed, based on this understanding of ceramic media, as part of a collaborative venture between Magotteaux and X-Strata technologies. Development was aimed at producing a ceramic media specifically for use in stirred milling applications in the minerals industry.

Keramax[®] MT1 was then evaluated against the following media types commonly used in the minerals industry as stirred milling grinding media:

- Two types of Alumina Media
- Australian River Pebbles
- Australian Silica Sand
- Nickel Smelter Slag.

The newly developed media was found to be the most economic media for the IsaMill[™]. Lower power draw recorded in a slurry environment was postulated to be as a result of the lower sliding friction coefficient of the media. A reduction of 50% in the energy was evident from the use of the newly developed media instead of the traditionally used natural media.

A secondary but very important finding was that the performance of the two Alumina media was significantly different, although the media composition was the same. This indicated that media should just not be selected based on the physical properties and/or the composition of the media, but that media evaluation should be done on laboratory scale and compared against a known benchmark¹⁴. The promising results lead to the media becoming the media of choice for IsaMill[™] installations worldwide.

2.2 Stirred Milling in the South African PGM Industry.

Rule (2011)³⁵ provides a detailed overview of the stirred milling technology capabilities in the South African PGM industry. Table 5 indicates that the number of horizontal stirred mill units installed by far outnumbers the vertical stirred mills in operation. Out of the total installed power of 71.18MW, horizontal stirred mills account for 93.5% of the installations. Anglo American's platinum operations accounts for 86% of the total installed power base of stirred mills in South Africa³⁵.

Table 5: Total Installed Base of Stirred Mills in the South African PGM producing operations³⁵.

Operator	Operation	Number of Stirred Mills	Type	Installed Power (kW)	Application
PGM Concentrators					
Anglo Platinum/JV's	WLTR	1	IsaMill™ M10 000	2600	MIG/UFG
	Waterval UG2	2	IsaMill™ M10 000	6000	MIG
	Mogalakwena South	3	IsaMill™ M10 000	9000	MIG
	Mogalakwena North	4	IsaMill™ M10 000	12000	MIG
	Waterval	4	IsaMill™ M10 000	12000	MIG
	Amandelbult	4	IsaMill™ M10 000	12000	MIG
	Amandelbult	1	IsaMill™ M3 000	1500	UFG
	Mogalakwena North	1	IsaMill™ M10 000	3000	UFG
	Waterval	1	IsaMill™ M10 000	3000	UFG
	Mototolo	1	Metso SMD™ 355	355	UFG
RBR	BRPM	1	IsaMill™ M10 000	3000	MIG
Lonmin	Eastern Platinum C	1	IsaMill™ M3 000	1000	UFG
	Eastern Platinum A/B	1	Metso SMD™ 355	355	UFG
Platmin	Boynton	1	IsaMill™ M3 000	1500	MIG
PGM Tailings Retreatment					
Impala	Rustenburg TRP	2	Metso SMD™ 355	710	UFG
Platinum Mile	Paardekraal TRP	2	Metso SMD™ 355	710	UFG
	Paardekraal TRP	2	Deswik - 2000litre	1065	MIG/UFG
Chromite Tailings Retreatment					
Aquarius/JV	RK1 Chromite Tails	1	Deswik - 1000litre	500	MIG
Sylvanie Resources	Steelpoort	2	Metso SMD™ 185/18.5	203.5	MIG/UFG
	Lannex	1	Metso SMD™ 185/18.5	185	MIG/UFG
	Lannex	1	Kings	75	MIG
	MilSell	2	Metso SMD™ 185/18.5	203.5	MIG/UFG
Tharisa	Brits Chrome TRP	1	Deswik - 5000litre	220	MIG

Anglo American's platinum operations has predominantly favoured the horizontal stirred milling technology; with only one vertical stirred mill installed in a concentrate regrind application in the Eastern Limb of the Bushveld Complex. Anglo American's platinum operations has also played a key role in the development of the technology for the mainstream inert grinding process^{11, 15, and 36}.

In the early 2000's, development of the stirred milling technology for Anglo American's platinum application was initiated with laboratory scale SMD and IsaMill™ testing at the company's in house research facilities^{11,15,36}. Positive results from the laboratory studies, on various ore types and process streams, initiated larger scale testing on 100 and 250 litre IsaMills™ at the pilot plant in Rustenburg to evaluate the potential recovery improvement by the application of horizontal stirred milling when treating dormant PGM tailings dump material. With the benefit realized at the larger pilot scale, the first stirred milling application was included into the process design for the Western Limb Tailings Retreatment (WLTR) concentrator in a concentrate regrind application. Coinciding with the decision to include the technology in the process design was the decision to also scale up the commercially available 1MW M3000 IsaMill™ unit for future enablement in mainstream applications.

The resultant 2.6MW M10000 IsaMill™ was commissioned in 2003 operating with silica sand as grinding media. Scale up from M3000 to M10 000 was fairly good^{11, 15}. Results after commissioning indicated that the cost and operating efficiencies of the scaled up unit outperformed that of a similar M3000 installation at a neighbouring mine^{11, 15}.

The foregoing discussion however was just the start of the wider application of stirred milling, and in particular horizontal stirred mills in the Anglo American's platinum concentrator operations. A research program was initiated to investigate the potential application to mainstream circuit regrind. On site pilot testing was conducted on all 3 reef types, at all the operations. A M20 IsaMill™, feeding an FCTR flotation rig was utilised to quantify the potential recovery improvement from mainstream inert grinding. Initially silica sand was used for the grinding media, but later replaced with a ceramic media for energy and cost effectiveness. Monthly plant feed, tailings and concentrate samples were analysed, in parallel to this test programme, on a size by assay and mineralogy by assay basis to provide further supporting data for the motivation to install the first M10000 IsaMill™ in a MIG application at Mogalakwena South C-Section in 2005³⁶.

The M10000 had an updated 3MW drive to compensate for the use of the higher density ceramic media. With the initial results indicating significant metallurgical improvement³², an aggressive roll out plan was initiated enabling the installation and commissioning of another 21 IsaMillsTM from 2006 to 2009. The significant improvement in metallurgical performance across all Anglo American's platinum operations to date has validated this aggressive roll-out of the new grinding technology.

2.3 The ceramic media manufacturing process and types of ceramic media commercially available.

Development of the Keramax[®] MT1 Media proved that ceramic media specifically designed for stirred milling applications resulted in significant improvements in both milling efficiency and overall mineral processing economics¹⁴. As part of Anglo American's platinum operations IsaMillTM implementation strategy, a strategic decision to establish a ceramic media evaluation facility at the Divisional Metallurgical Laboratory (DML) for the selection and development of suitable media for stirred milling applications was taken.

The ceramic media strategy implemented involved:

- Identifying potential suppliers of ceramic media based on a review of the world wide ceramic media market.
- Identifying supplier partners with the R&D capability to develop new media or enhance available media for stirred milling application.
- Understanding the physical and chemical properties of ceramic media.
- Understanding of the various ceramic media manufacturing processes.

Ceramic media is manufactured from a combination of the following raw materials⁵:

- Zirconia produced by the chemical or fusion route.
- Zirconium Silicate extracted from Zircon sand
- Reactive alumina extracted from cell/smelter grade alumina.

Manufacturing of the ceramic media normally follows one of 2 major production processes, the fusion or sintering process, as indicated in figure 3.

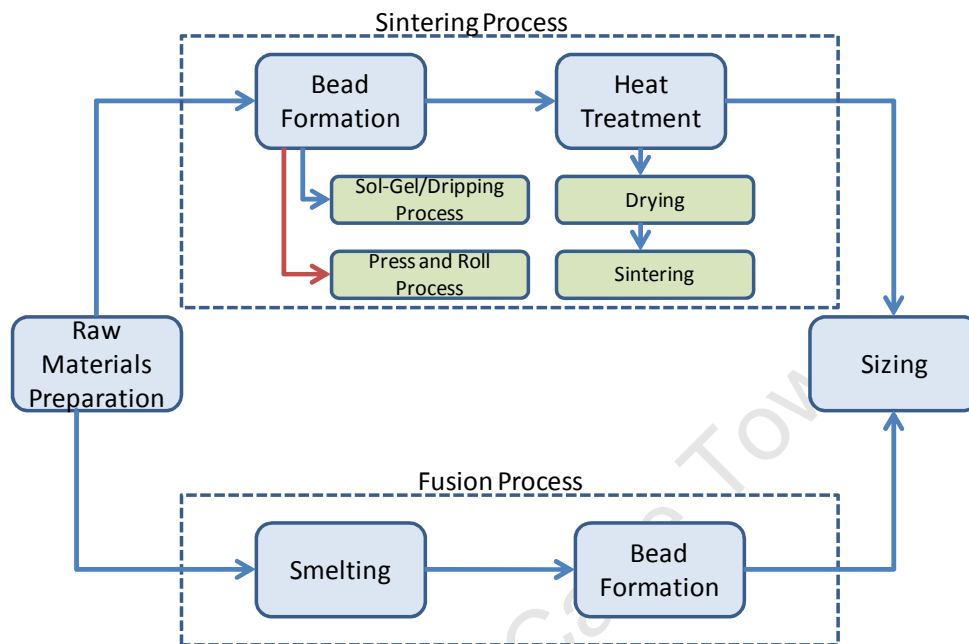


Figure 3: Typical Ceramic Media manufacturing processes.

Raw materials preparation is very similar and involves material to the required grind to ensure bead competency and good bead properties throughout the bead structure, normally smaller than $1\mu\text{m}$ ^{2, 3, and 41}.

For the sintering process, bead formation normally occurs by either the “sol-gel” or the “pressing and rolling” technique. The “pressing and rolling” technique involves densification of the dry raw material by means of a hydraulic or electrical press followed by a rolling process to shape the bead. The final process is heat treatment by sintering⁴.

For the “sol-gel” process the raw material is milled wet to form slurry or “slip” as it is commonly referred to in the ceramics industry before mixing with an alginate to control the slip viscosity for the dripping process⁴¹. If the viscosity of the slip is not carefully controlled, the bead size and shape is affected.

The mixed slip is then dropped through a paraffin header into a calcium chloride bath where transition from the liquid to the sol-gel phase takes place. The beads are then dewatered and subjected to heat treatment processes to produce dense ceramic media. The sintering process varies from 1350°C to 1700 °C depending on the material composition⁴¹. During this process densification takes place by material transport of diffusion at the grain boundaries as the beads shrink. This process is similar to the baking of a cake.

In the second production process, the fusion route, the raw material is smelted and homogenized at temperatures in excess of 2000°C in a kiln. Upon leaving the kiln, droplets of the homogenized melted material fall to surface where it is collected and densifies to round spheres as a result of the surface tensions of the liquid droplets. The atmospheric condition outside the kiln allows the formed droplets to solidify and stabilize.

Of the two processes the sintered process is preferred as the manufacturing route of choice. It has been identified that ceramic media produced by the fusion route has the following distinct disadvantages⁴¹:

- Inclusion of air bubbles cannot be avoided, reducing the specific weight of the media, resulting in pores in the media that will break when subjected to the stresses inside a stirred mill.
- Beads can stick to each other resulting in the formation of irregular shaped beads that will increase the energy consumption of stirred mills.
- Inhomogenisation of the beads can take place after forming and during the cooling process, resulting in weakened bead structures.

Commercially available ceramic media can be classified into several classes as per figure 4 based on the chemical composition of the media^{4, 41}.

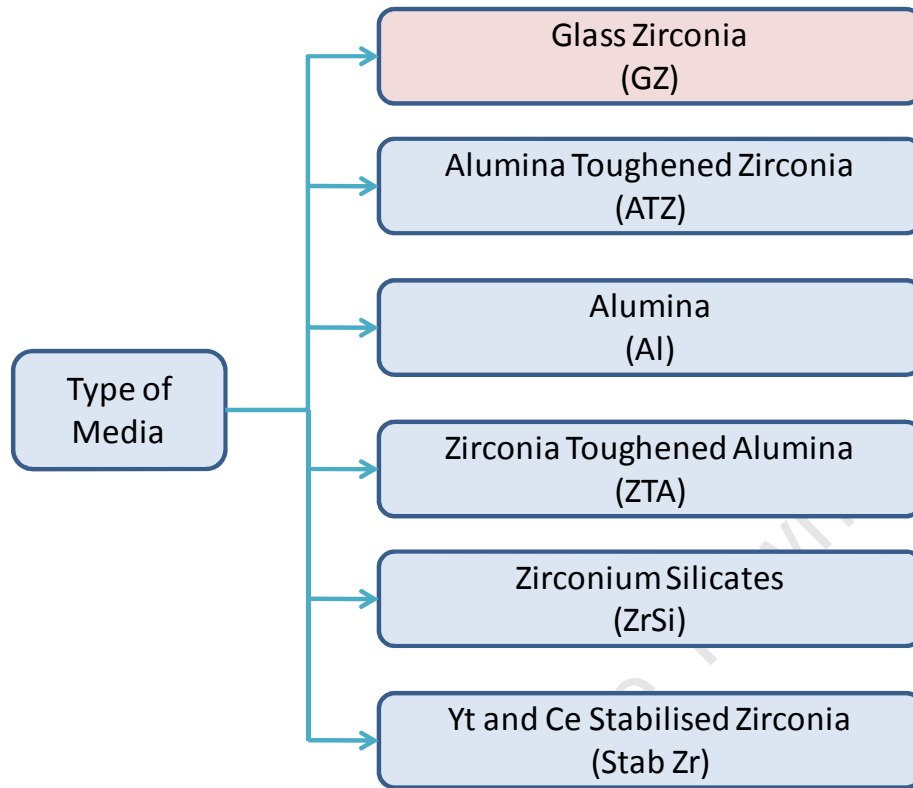


Figure 4: Media Classification based on composition⁵

The Glass Zirconia media that is produced using the fusion route have been found to be unsuitable for use in the stirred milling environment due to high wear rates and poor grinding efficiency.

2.4 Physical Properties of Ceramic media and its impact on stirred milling performance and operating costs.

Several authors^{2, 3, 6, 14, 16, 17, 31, 41} investigated the suitability of ceramic media for stirred milling processes and the following properties are considered as crucial properties when selecting the best ceramic media for the stirred milling application:

- Media Size and Density
- Media Shape and Competency
- Media Hardness (Function of chemical composition and grain size)
- Media Composition (Inertness of the media).
- Media Surface Properties

2.4.1 Media Size and Density.

The size of the media is important as it is the primary limitation of the fineness of the product that can be achieved in the stirred milling process. Kwade et al. (2002)³¹ postulated that the comminution process in stirred media mills is determined by how often each particle in the mill is stressed and how high the stress intensity at each event is. In the work, a mathematical description of the stress intensity was derived from the grinding mechanisms occurring during stirred mill operation. Grinding takes place in stirred media mills primarily due to the high velocity gradients near the stirrer and near the chamber walls resulting in the grinding media moving with different velocities. Grinding media with high and low velocity collide against each other, releasing some of the media's kinetic energy as energy for particle breakage or comminution.

Secondly, the media is based in a zone around the chamber wall as a result of the centrifugal acceleration occurring in the mill³¹. Pressure between the grinding media in this zone is stressing the ore particles and it was postulated that the stress intensity in this zone is much smaller than the stress intensity generated as a result of the kinetic energy of the media. Therefore, it was concluded that the stress intensity will be proportional to the kinetic energy of the grinding media.

Based on the following assumptions a characteristic parameter for stress intensity was then derived³¹:

- Only single particles are stressed intensively between the grinding media, and therefore the stressed particle volume does not depend on media size.
- The tangential velocity of the grinding media is proportional to the circumferential speed of the discs.
- The diameter of the discs is kept constant.

In the case of deagglomeration, disintegration, and grinding of weak or medium hard crystalline materials like limestone it can be assumed that the elasticity of the feed material is much smaller than that of the grinding media. Energy transferred to the particle when caught between two media will be proportional the stored energy in the grinding media. Therefore the energy transferred is not a function of the composition of the particles or the grinding media. Stress intensity³¹ (SI) is proportional to an expression called the stress intensity of the grinding media (SI_{GM}) as defined below in equation 2.4.1

$$SI \propto SI_{GM} = d^3 \times \rho_{GM} \times v_t^2 \dots \dots [Eq\ 2.4.1]$$

Where:

d = media diameter

ρ = media density

v = disc/agitator tip speed.

If all particles in the mill are taken into account the product of the stress intensity and the stress number (SN) is proportional to the total specific energy $\{E_M, (E_V)\}$ ³¹. This is the total energy input related to the total mass of volume of the feed material as illustrated in equation 2.4.2.

$$SI \times SN \propto E_m(E_V) \dots \dots [Eq\ 2.4.2]$$

Stress Number is the average number of stress events of each feed particle or agglomerate (and the fragments from it) in batch grinding³¹. Therefore the stress number is determined fundamentally by:

- The number of media contacts, N_c ,
- The probability that a particle is entrapped between particles and sufficiently stressed at a media contact, P_s .
- The number of feed particles in the mill, N_p .

$$SN = \frac{N_c \times P_s}{N_p} \dots \dots [Eq\ 2.4.3]$$

The number of media contacts can be considered to be proportional to:

- The number of stirrer revolutions, n ,
- The total grinding time, t .
- And the number of grinding media in the grinding chamber, N_{GM}

$$N_c \propto ntN_{GM} \propto \frac{V_{GC} \times \phi_{GM} \times (1 - \varepsilon)}{\frac{\pi}{6} \times d_{GM}^3} \dots \dots [Eq\ 2.4.4]$$

Where:

- n = number of stirrer revolutions per unit time (s^{-1})
- t = grinding time
- V_{GC} = Volume of the chamber (m^3).
- ϕ = filling ration of the grinding media.
- ε = porosity of the bulk of the grinding media.
- d = diameter of the grinding media.

The probability of a particle being caught and sufficient pressure being applied, in the case of grinding crystalline materials, is proportional to the active volume between two grinding media. It was also found that the pressure is therefore directly proportional to the diameter of the grinding media.

In the case of deagglomeration and cell disintegration during the comminution process the probability is proportional to the grinding media surface area. Sufficient energy exist in the sheer stresses between grinding media for this to occur and that only a small portion of the agglomerates or cells are stressed in the active volume. Because of the proportion of the cells stressed are so low and the fact that the active volume is part of the grinding surface area, stressing in the active volume do not need to be considered separately. Pressure applied to the particle can then be considered to be proportional to the square of the grinding media parameter.

The number of the feed particles is proportional to the overall volume ($V_{p,tot}$) of the feed particles as shown below in equation 2.4.5.

$$N_p \propto V_{p,tot} = V_{GC} \{1 - \phi_{GM}(1 - \varepsilon)\} C_v \dots \dots [Eq 2.4.5]$$

Where C_v is the volume solids concentration in the stirred mill product.

From all this, the proportionality for the reduced stress number for the grinding (SN_{GM}) of crystalline materials was developed as presented in equation 2.4.6³³.

$$SN \propto \frac{\phi_{GM}(1 - \varepsilon)}{\{1 - \phi_{GM}(1 - \varepsilon)\} C_v} \times \frac{nt}{d_{GM}^2} \propto C.SN_{GM} \dots \dots [Eq 2.4.6]$$

Therefore to achieve the same product fineness two of the three parameters, stress number, stress intensity or specific energy has to be constant. Media density and size are major contributors to SN and SI. As the specific energy of the process is the product of the parameters, it is therefore of major importance to understand the impact of density and size when selecting the correct media to optimize grinding efficiency and energy consumption.

Farber et al (2009)¹⁶ used the methodology to develop a model to predict the best possible media size for a media of a higher density (5.2g/cm^3) to maintain the same milling performance as a lower density media (3.8g/cm^3). The results indicated that the higher density media will give the same milling performance if a bead of 1.7mm was used.

Weber et al (2010)⁴¹ used the methodology proposed by Kwade to evaluate the performance of 5 different grinding media with two different bead sizes, when grinding a ground calcium carbonate powder with a mean particle size (d_{50}) of $17\mu\text{m}$ down to $2\mu\text{m}$. An optimized energy curve was generated by plotting the specific energy required against the SI and thereby identifying the most energy efficient media density and size for grinding the calcium carbonate powder to the specified grind.

Curry and Clermont¹⁴ also indicated that the use of natural media such as silica sand that that was initially used in stirred mills was limited by the fact that the sand media broke up to its natural grain size when exposed to stresses in the high intensity milling environment. The natural grain size therefore is a limiting factor to the feed size that could be successfully treated as the Stress Intensity and Stress Number will be limited to the natural grain size of the sand.

2.4.2 Media Shape and Competency

Curry and Clermont¹⁴ reported that angular natural media initially used in the stirred mills, had much lower energy efficiency when compared to spherical ceramic media. A large proportion of the energy was going into the media until a smooth graded charge was established in the stirred mill, and not into the breakage of ore particles. Even with the use of energy inefficient natural media, the stirred mill still proved to be much more efficient than conventional ball milling.

Figure 5 presents before and after images of silica sand after only 1 hour of conditioning in water clearly indicating how the silica sand particles became more spherical after milling.

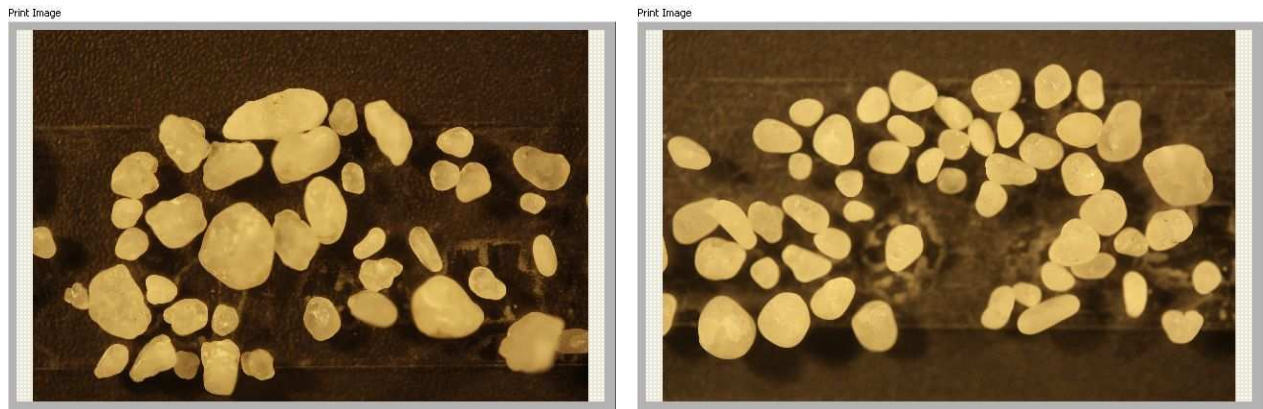


Figure 5: Silica Sand Before and After Water Conditioning.

It is also logical to assume that the angular shape of the natural media will result in accelerated wear of internal wear components of the mill due to the higher friction coefficient of the media. The sharp edges of the angular media will cut up the rubber wear components of the mill it comes in contact with.

Chadwick (2007)¹² in an article in the Mining Journal indicated that grinding media should be well rounded with minimal cracks and flaws as imperfections result in the accelerated breakdown of the SMD, increased media consumption and accelerated component wear. Changes in viscosity and operational efficiency are also evident with high media wear rates of the non-spherical poor quality media.

Bedesi et al (2008)⁴ reported that media should be mechanically sound to withstand the forces in the mill under normal operating conditions. Incompetent media will not only lead to higher than expected media consumption but also to higher wear of the mill internals and discs. This is as a result of sharp shards of media caught in the mill after disintegration of the media. The sharp shards can increase wear of internal rubber components by cutting up the rubber components.

Figure 6 to 9 presents some of the flaws and imperfections identified in media as part of the DML ceramic media development program.



Figure 6:: “Soft Centres” in media



Figure 7: Internal Cracking



Figure 8: “Onion skin” Effect



Figure 9: Air Inclusions in bead

The “soft centre” referred to is when there is an area filled with air in the centre of the ceramic bead. Once the bead is worn down to the point where the layer of ceramic media around the “soft centre” cannot withstand the forces in the mill anymore the bead will collapse and disintegrate in the mill. The internal cracking is as a result of incomplete drying prior to sintering. If the pressure in the mill is high enough, the bead will break along the crack resulting in sharp pieces that can cut up rubber components in the mill and reduce milling efficiency.

A common phenomenon of the pressed and rolled manufacturing method is the “onion skin” effect where the various layers exhibit varying hardness. Most commonly the hardness of the layers decreases from the edge to the centre of the bead. If the initial evaluation was done over a short period of time, the true wear characteristics of the bead will not be seen as the bead wear rate will increase the smaller the bead gets. If the bead wear rate is abnormally high it can also impact on the milling efficiency as the rapidly wearing beads may alter the viscosity of the slurry.

Control of the slip viscosity during the needle dripping and “sol gel” process is extremely important to form a round bead of the correct size and to allow all air to escape from the gel bead during curing in the calcium chloride bath. Alginate is used for this purpose during the manufacturing process and a change in the alginate can have a major impact on the viscosity of the slip and the quality of the bead. Rapid wear of media can be associated with the air inclusions, as the media will break into sharp angular pieces that can reduce the life of wear components in the mill

Data from X-Strata Technology Maintenance Records¹ for the M10000 Mill at the Western Limb Tailings Retreatment (WLTR) operation indicates the reduction in the disc wear when poor quality media 5 mm media was replaced with good quality fresh 5mm media as depicted in Figure 10.

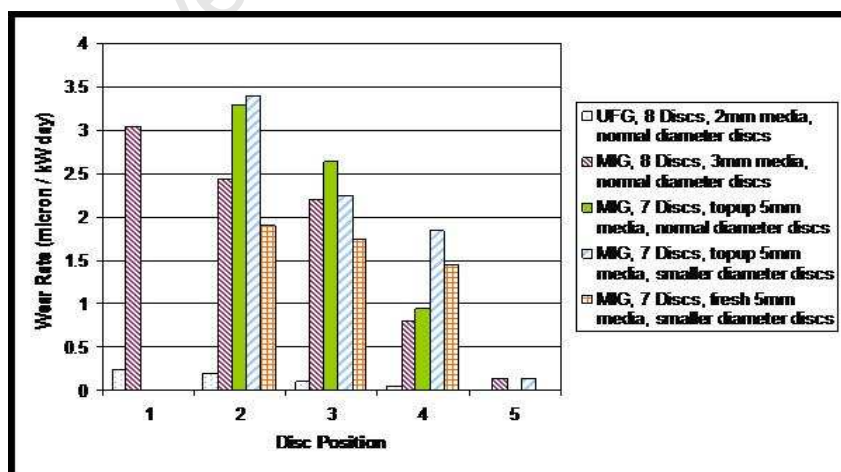


Figure 10: Disc Wear data from WLTR M10000 IsaMill™ (Anderson¹)

2.4.3 Media Hardness:

The hardness of the media is critical for the milling efficiency as the grinding media hardness need to be higher than the hardness of the milled particles. The chemical composition and the grain size of the ceramic media determine the hardness of the media and is also an indication of the wear resistance of the media. The general trend is that the hardness and wear resistance of the media increase as the zirconia content of the media increase but this is accompanied by an increase in price per unit mass for the media.

Becker and Schwedes (1999, 2001)^{2, 3} studied the effect of media and feed material properties on the energy consumption of stirred mills and the bead wear rate during milling. In the study, Fused Corundum (Al_2O_3) and Silicon Carbide (SiC) were ground with the following grinding beads in a laboratory stirred mill:

- Aluminium Oxide (5 Types)
- Zirconium Oxide (2 Types)
- Mixed Oxide (2 Types)
- Silicon Nitride
- Steatite
- Titanium Oxide and Steel

The fused corundum and the silicon carbide median particle size was $33\mu\text{m}$ and $40\mu\text{m}$ respectively with a target grind of 100% $1.5\mu\text{m}$ aimed for during the milling process. Energy requirements to reach the target grind were significantly higher when steel media was used instead of ceramic media. It was postulated by the author that high surface area particles are removed from the bead surface during the grinding process, resulting in a reactive bead surface. When reacting with water to form Fe_2O_3 and hydrogen gas. The gas formation therefore occupies volume in the mill and hinders media movement, resulting in higher energy requirements to reach the same target grind.

Another postulation was that the plastic deformation of the bead surface during a stress event consumes energy for deformation and therefore reduces the amount of energy available for particle breakage. The media tested were classified into 2 groups based on the observation that the specific energy requirements to reach the desired fineness of grind differed significantly between the various types of beads:

- Group 1 included the 86.5% Al_2O_3 , Steatite, Mixed Oxide; Yttrium stabilized zirconium oxide and the silicon nitride media.
- Only 2 media tested were classified into Group 2, the 99.7% and 92% Aluminium oxide media. Results indicated early flattening of the specific energy versus grind relationship as well as a limitation in the achievable fineness of grind above the desired target.
- Group 2 media all had microstructures comprising of 5-10 μm particles which resulted in whole particles breaking of the surface of the media during the comminution process.

Overall final grind was also found to be dependent on the microstructure fineness of the bead as the finer the microstructure the finer the overall grind achieved. Due to the wear of grinding media the solids increase in the suspension in the mill. Therefore power is not only required to comminute the feed material but also the abraded media particles resulting in increased specific energy required to reach the target fineness of grind if the bead wear is higher.

Specific energy was therefore found to be the energy input into the grinding chamber related to the mass of the product plus half of the mass of the grinding media wear. Subsequent tests on Group 1 media indicated that for a constant specific energy the same product fineness was obtained with different types of media, but that the higher the wear rate of the bead, the higher the total energy would be to obtain the same specific energy.

Therefore in summary, to minimize the specific energy required for grinding, a media with high hardness composed of primary particles smaller than $1.5\mu\text{m}$ should be used to grind the fused corundum and silicon carbide to the required target grind of 100% - $1.5\mu\text{m}$.

2.4.4 Media Composition (Inertness of the media).

Inert properties of the grinding media, is very important for processes where metallurgical performances of downstream process after stirred milling can be negatively affected by the degradation/wear products of grinding media. The effect of the wear of steel balls in conventional milling and its effect on the downstream flotation process are well described in the chapter 2.1.2.

2.4.5 Media Surface Properties:

Curry and Clermont¹⁴ proposed the use of the sliding friction coefficient as a new parameter to characterize stirred mill media. During the evaluation of the Keramax MT1 media it found was that the grinding efficiency was better than predicted from SI calculations. It was also found that the power draw when running on a water media system was more than for a media slurry system for natural medias and that the reverse was true for the newly developed Keramax MT1 media. From these results, it is hypothesized that slurry lubricates a typical media charge as the frictional loss between media particles in motion is lower in the presence of feed slurry.

Through DEM simulations it was proved that the surface rolling coefficient (μ_r) does not have a measurable effect on energy distribution, shaft torque or media velocity or force. Therefore for the DEM simulations μ_r was fixed at 0.005 as was consistent with earlier work. Further DEM simulations with two varying sliding coefficients (μ_s) revealed that media with the low sliding factor exhibited much larger forces and should therefore consume less energy during grinding operations. Simulations of the resultant torque acting on the IsaMillTM shaft indicated a drop of up to 50% in power consumption with the media with the lower sliding coefficient.

Farber et al (2010)¹⁷ investigated the effect of media size, density and friction on stirred milling performance on a UG2 ore. A model was developed, that incorporates media properties to predict the milling performance and can be used to design and develop milling media for specific applications. Farber found in the initial work that the power consumption in a laboratory mill was approximately 40% lower for a higher density media at the same filling degree. This was due to the significantly lower friction coefficient measured for the higher density media. This indicated that power consumption is primarily a function of the media density and the friction coefficient of the media and the following formula was developed to predict the relative power consumption of two media as presented in equation 2.4.7.

$$\frac{W_a}{W_b} = C_p \frac{\rho_a}{\rho_b} + (1 - C_p) \frac{Fc_a}{Fc_b} \dots \dots \dots [Eq 2.4.7]$$

Where:

- C_p = Contribution of media density effect.
- $W_{a,b}$ = Power draw of mill with media A and B
- $\rho_{a,b}$ = Density of Media A and B
- $Fc_{a,b}$ = Friction Coefficient of media A and B.

Farber et al (2010)¹⁷ in subsequent work investigated the effect of media type on mill internal wear, by measuring the wear on the impeller of a LMZ-03 Netsch Mill operating under the same conditions with different types of ceramic media. Fundamentally it can be expected that a media with a lower coefficient of friction will result in a lower wear on internal components of the mill. Results indicated a reduction of almost 70% in impeller wear with the use of a higher density media with a lower friction coefficient.

Figure 11 and 12 present's high definition photographs of a low density media with a high friction coefficient and a high density media with a low friction coefficient. Note the high glossy polished surface of the media with the low coefficient of friction in figure 12.



Figure 11: High Friction Coefficient media



Figure 12: Low friction Coefficient media

2.5 Modelling of media behaviour based on the media properties.

Kwade (2007)³³, in chapter 6 of the Handbook of Powder Technology, explains the concept of media packing and positioning in a horizontal stirred mill. The mill used for the work by Kwade did not include the centrifugal separator that was incorporated in the design of the IsaMill™ for the mineral processing industry.

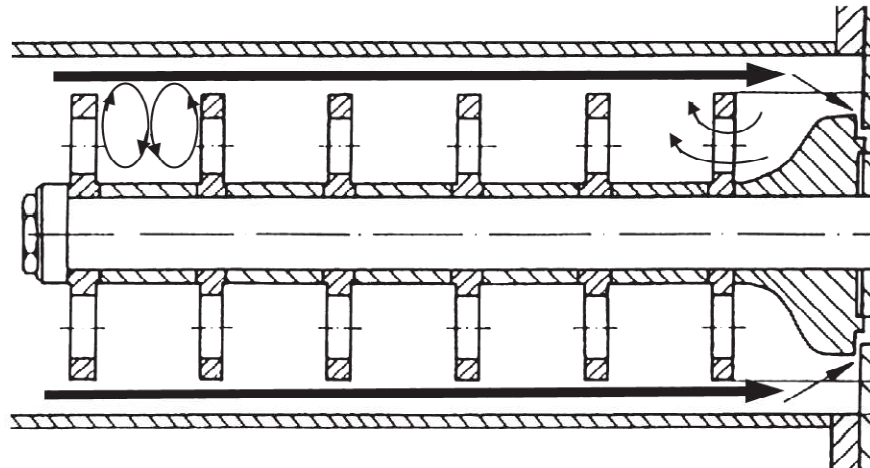


Figure 13: Flows and forces in a stirred media mill (Kwade 2007)³³

Tests done to assess the location and packing density in the mill were done using a radiator-detector arrangement along the axial direction of the mill as indicated in figure 14. The principle is that the grinding media in the path of the gamma rays emitted by the detector will weaken the signal detected by the detector and the more media in the path of the gamma rays the weaker the signal detected.

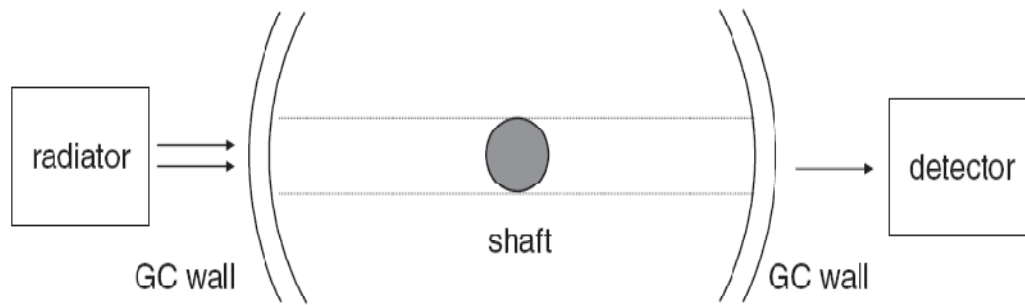


Figure 14: Measuring media packing using radio densitometry (Kwade 2007)³³

Results indicated that the media packing increase towards the discharge end of the mill as presented in figure 15.

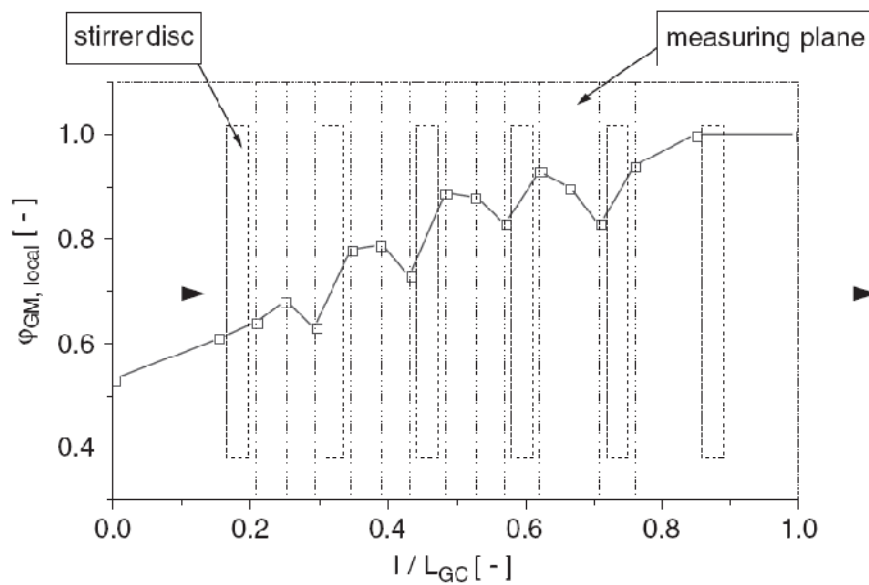


Figure 15: Radiometric densitometry measurements of the Grinding media in the mill

(Kwade 2007)³³

From these results a model was developed of the grinding media packing as illustrated in figure 16. This model assumes that a grinding chamber with length L can be divided in two zones with different filling ratios of grinding media. In zone 1 the filling ratio of grinding media is lower than the average filling ratio ($\phi_{GM1} < \phi_{GM}$). The grinding media is densely packed in zone 2 with the characteristic packing filling ratio ϕ_{GM2} .

This indicates that in a stirred mill with no centrifugal separator that the media added to the mill is not evenly distributed in the axial direction and that packing occurs towards the end of the mill. The degree of media packing could be controlled by controlling the feed flow and throughput and the stirrer speed in conventional stirred mills.

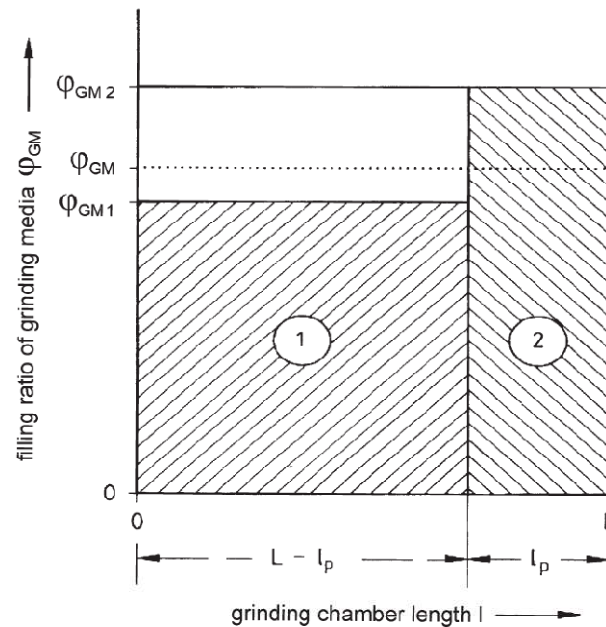


Figure 16: Radiometric densitometry measurements of the Grinding media in the mill

(Kwade 2007)³³

The IsaMill™ however differs slightly from this approach as the design of the mill includes a centrifugal separator. The separator classifies the slurry stream at the discharge of the mill to return grinding media and coarse particles to the grinding chamber, to produce a closely sized PSD in the discharge for downstream processes.

Discs turning at 19-22 m.s.⁻¹ tip speeds provide agitation and impart centrifugal force to the media load and slurry inside the mill. A forward flow is introduced through the mill by the slurry feed flowing from the mill feed end to the discharge end with a counter current flow being produced by the centrifugal separator. This classifies the slurry to prevent oversize ore particles and grinding media from exiting the mill.

If one considers the question: How is the media distributed in the mill? Information from X-Strata^{20, 21} indicates an even distribution of media in the mill resulting in 8 consecutive grinding chambers formed within the mill which increases the energy intensity of the mill way beyond that of other grinding technologies. DEM/CFD and PEPT modelling has been done on a simplified IsaMillTM by various authors²⁴⁻²⁸. The mill only had 3 discs and the modelling did not take into account the forward flow of the feed slurry or the counter-current flow generated by the centrifugal separator.

Jyasundara et al.²⁴⁻²⁸ utilised DEM simulations to investigate the effect of particle properties on the particle flow in the simplified IsaMillTM, in the radial direction mostly in the first work. The simulation was conducted at a tip speed 50% lower than that employed at during standard ceramic media evaluation at laboratory scale and at an 80% solid loading of the simplified mill.

Increasing the sliding friction coefficient of the media, the spatial distribution of the media velocities indicated a more vigorous movement of the media and the velocity gradient is higher for the low friction coefficient media. There was a higher porosity distribution of media along the chamber wall for the lower sliding friction coefficient due to the unconfined movement of the media. Grinding is a result of the shear imparted on particles as various layers of media move over each other and at the lower velocity gradient the media will move as a solid bed that will lower grinding performance.

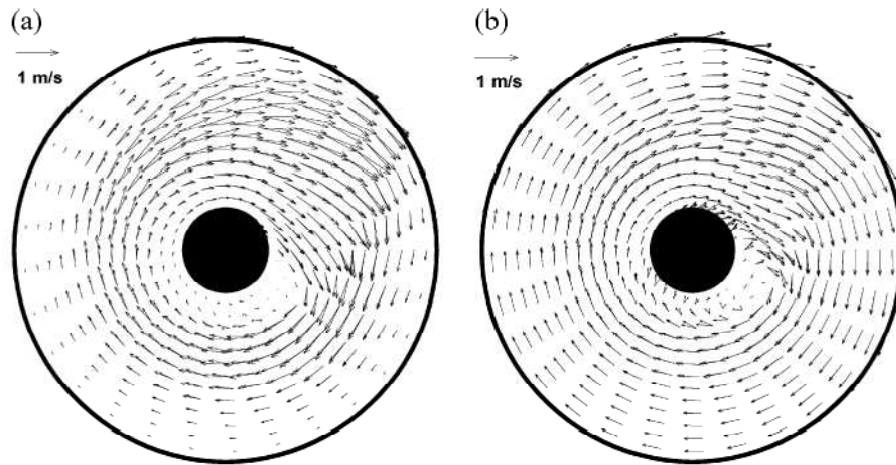


Figure 17: Spatial distribution of velocities for different particle/particle sliding friction coefficients: (a) $\mu_{s,pp} = 0.01$ and (b) $\mu_{s,pp} = 1.0$

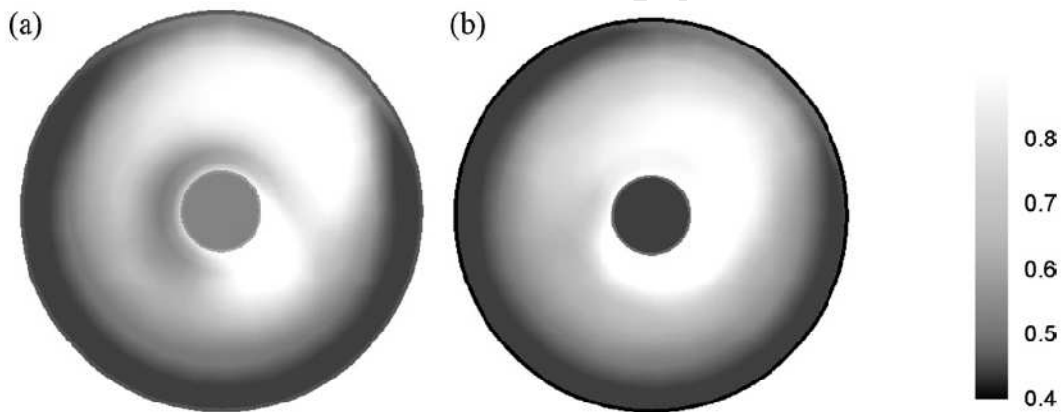


Figure 18: Spatial Distribution of porosity for different particle/particle sliding friction coefficients: (a) $\mu_{s,pp} = 0.01$ and (b) $\mu_{s,pp} = 1.0$

The restitution coefficient had a negligible impact on the velocity and porosity distributions in the mill. Higher restitution coefficients led to less energy dissipation on collision and therefore more vigorous movements and collisions that could be beneficial for grinding. Power draw remained constant and it is known in general that the harder the media the higher the restitution coefficient.

Media density also exhibited very little impact in the radial direction on the spatial distribution of velocities and porosities. It was however noted that an increase in the particle density increased the collision frequency and energy of the mill and collision energy can be considered proportional to the media density. An increased density indicated an increased power draw due to the effect of the stirrer discs lifting the media to the chamber wall. This is however expected if the sliding coefficient of all the densities is seen as constant. The work done by Farber^{16, 17} has proven that higher density media has superior friction coefficients and can therefore draw less power in a stirred mill than lower density media of higher sliding friction coefficients.

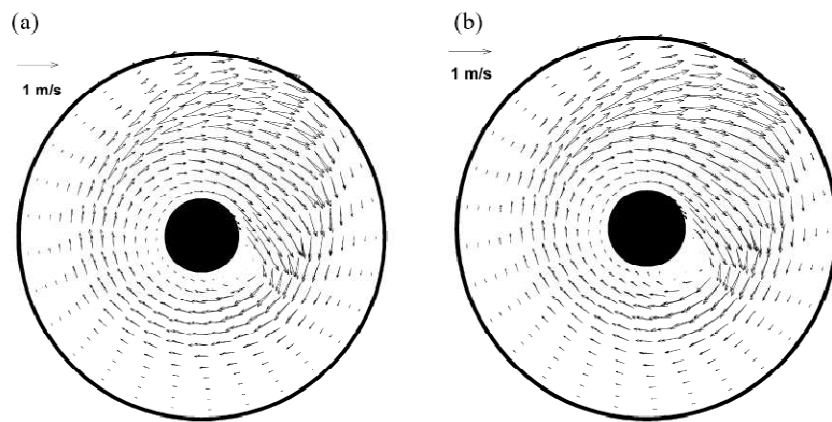


Figure 19: Spatial Distribution of velocities for different particle density: (a) $\rho = 1000 \text{ kg.m}^{-3}$ and (b) $\rho = 3500 \text{ kg.m}^{-3}$

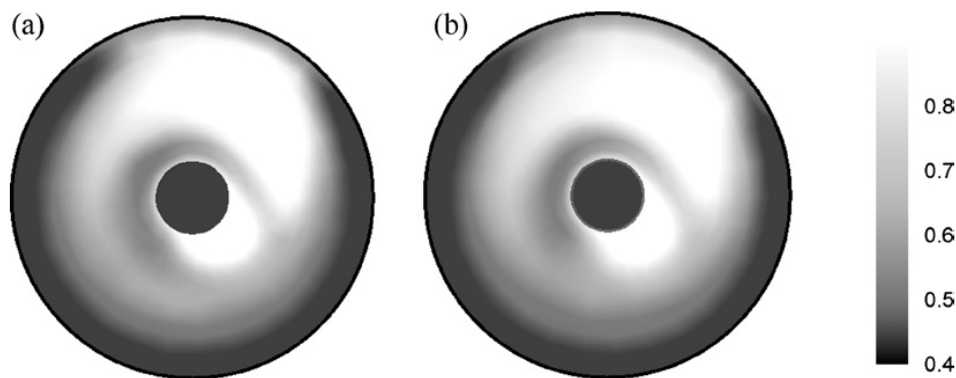


Figure 20: Spatial Distribution of porosity for different particle density: (a) $\rho = 1000 \text{ kg.m}^{-3}$ and (b) $\rho = 3500 \text{ kg.m}^{-3}$

With respect to media size, the spatial distribution and velocities of media at different sizes differed significantly. In the axial direction circulating flow of 2mm particles between the discs was noticed while the collisions were five (5) times more for the 2mm media when compared to 4mm media. Lower power draws was simulated for the larger media at the same filling, the surface area of the media will be significantly less and therefore the effect of the sliding friction coefficient will be less.

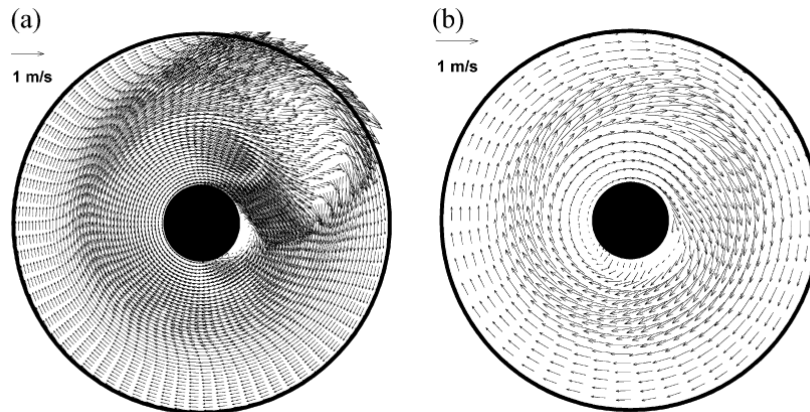


Figure 21: Spatial Distribution of velocities for different particle sizes: (a) $d_p = 2\text{mm}$ (b) $d_p = 4\text{mm}$

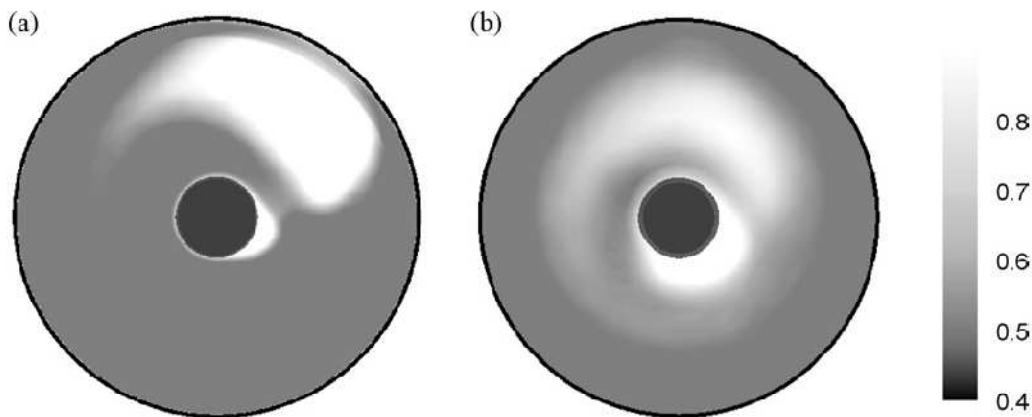


Figure 22: Spatial Distribution of porosity for different particle sizes: (a) $d_p = 2\text{mm}$ (b) $d_p = 4\text{mm}$

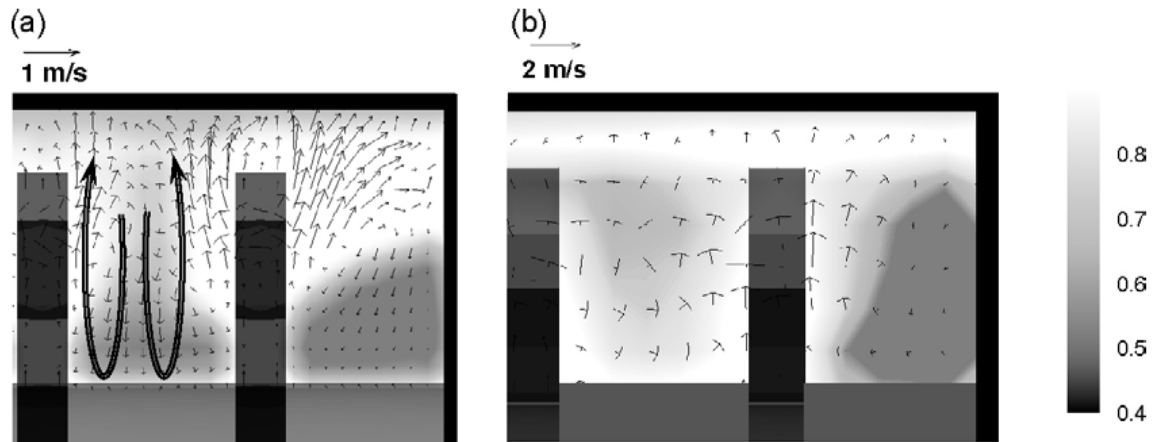


Figure 23: Spatial Distribution of velocity and porosity for different particle sizes in the upper regional sectional elevation at XX': (a) $d_p = 2\text{mm}$ (b) $d_p = 4\text{mm}$

In the second set of work by Jyasundara et al.²⁵, the effect of mill loading and rotational speed on the media motion was investigated in the simplified mill. The tests were again done at 1200 rpm for the mill load tests, which is significantly lower than the M4 test condition of 2100rpm. Media was concentrated toward the bottom of the grinding chamber for the lower mill fillings and not evenly distributed between the top and the bottom of the mill.

This indicates that at the speed tested in the lower mill filling; much lower velocity gradients and low energy collisions would occur that would result in poor grinding performance. For the higher loading, the media was more evenly distributed through the mill and higher number of collisions and higher energy collisions is possible, resulting in improved grinding performance.

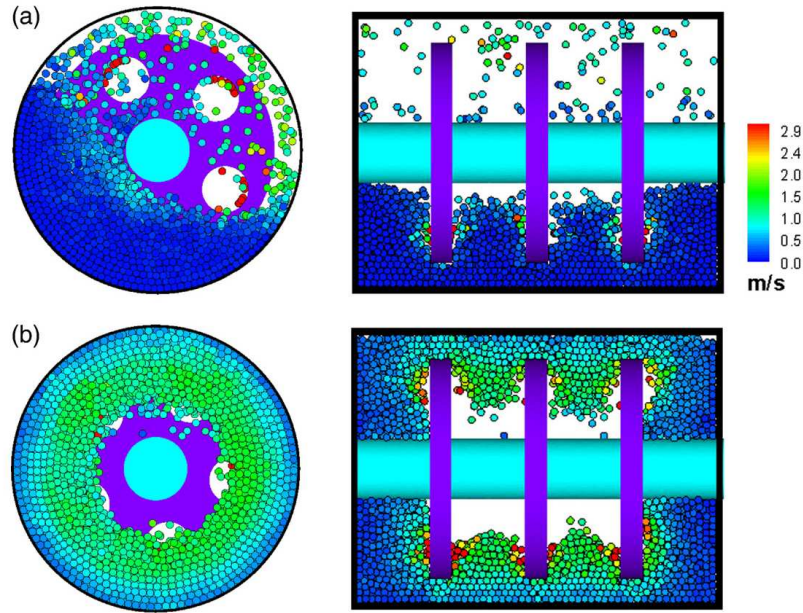


Figure 24: Particle flow in the radial and axial directions for different loadings: (a) $J=40\%$ and (b) $J=80\%$ at 1200rpm.

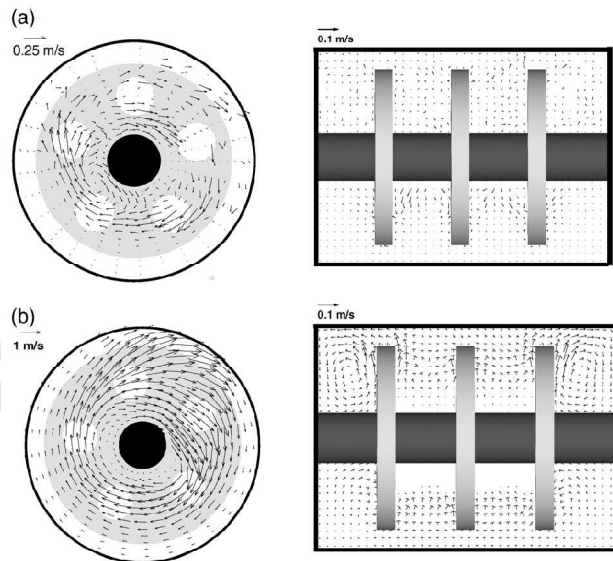


Figure 25 Spatial distribution of particle velocity in the radial and axial direction for different loadings: (a) $J=40\%$ and (b) $J=80\%$ at 1200rpm

Higher mill speeds indicated more evenly distributed media load in the mills at an 80% filling as would be expected at higher loads. Most of the findings from the DEM simulations were later validated by PEPT at the University of Cape Town by Van der Westhuizen et al²⁸.

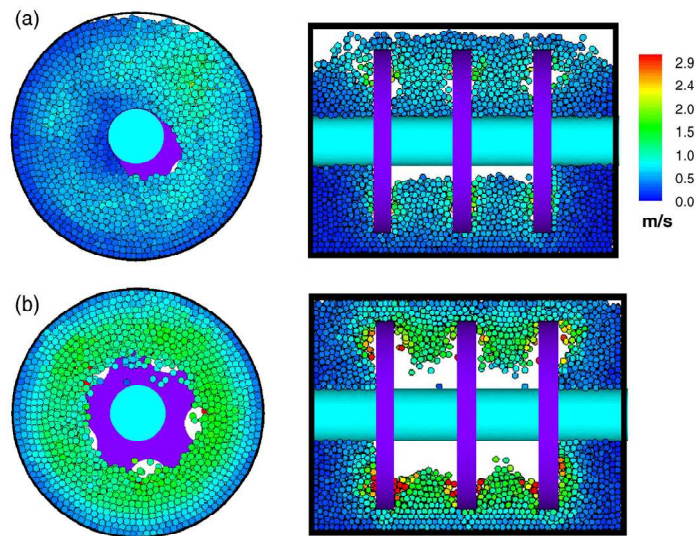


Figure 26: Partical flows in the radial and axial directions for different mill speeds: (a) 500rpm and (b) 1200rpm at 80% mill loading.

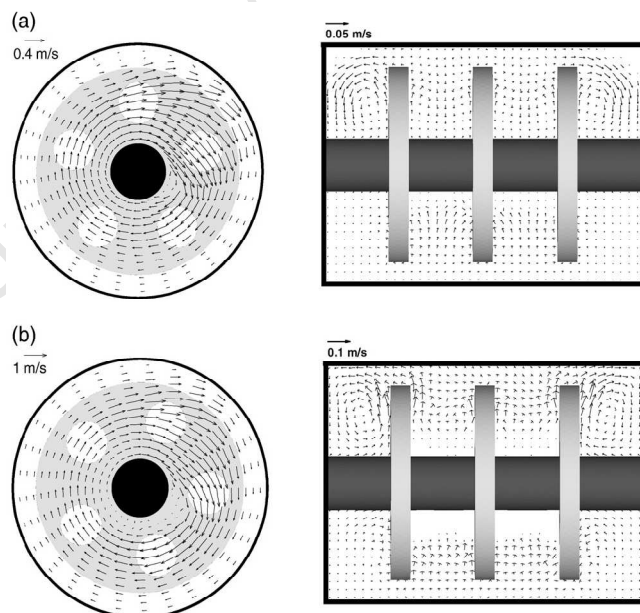


Figure 27: Spatial distribution of particle velocity in the radial and axial direction for different mill speeds: (a) 500rpm and (b) 1200rpm at 80% mill loadings

CHAPTER 3 – RESEARCH METHODOLOGY.

This chapter of the thesis details the individual phases of the research methodology and experiments performed.

The Research Methodology that was applied can be divided into 6 different stages set out below and each of the stages will be described in detail later in this section.

1. Analysis of the historical ceramic media evaluation data, produced internally in Anglo American's platinum operations, following a standardised ceramic media evaluation methodology⁴. The data was used to test the applicability of Farber's¹⁷ methodology with respect to surface friction across several media types. The media evaluation methodology⁴ was developed by Anglo American's platinum operations to evaluate media of different compositions and from different sources to identify the best possible media for IsaMill™ application.
2. Measuring the friction coefficient of the selected ceramic media by Dr Boris Farber at the Zircoa Inc. Laboratory in the USA.
3. Preconditioning of selected media in water to remove surface defects and substandard quality beads from the bead samples.
4. Performing water tests of selected media in a Netsch four (4) litre IsaMill™ at various media loads as per the standard developed water testing procedures⁴ adopted at the Divisional Metallurgical Laboratory (DML).
5. Slurry testing of the selected media to establish the impact of the selected media on grinding efficiency at a constant feed size to determine the most energy efficient media.
6. Determining the physical properties of the media such as size, shape/aspect ratio and density at the DML Ceramic Media Laboratory in Rustenburg.

3.1 Analysis of historical ceramic media data.

From the ceramic media evaluation database, nine (9) sets of results were selected for analysis to investigate the effect of media type on energy efficiency and to test the application of Farber's methodology¹⁷ across media types. The selection was done to ensure that at least five (5) media types with different chemical compositions were included in the analysis. Four (4) media with similar composition, but slightly different manufacturing processes were also included in the evaluation.

From the media selected the effect for various media compositions or classes, as well as variation within ceramic media classes, on the energy consumption of the M4 IsaMill™ will be evaluated.

The following was considered during analysis of the historical data base:

- Media Chemical Properties
- Media Physical Properties
- Energy Consumption in Water
- Energy Consumption in Slurry
- Signature plots illustrating grinding efficiency of the media.
- Specific energy required to obtain a target grind P80 of 50µm.

3.2 Preconditioning of Media prior to Water and Slurry Testing.

To ensure that surface defects and substandard ceramic media were excluded, the media first had to be first preconditioned for one (1) hour in water in the horizontal stirred mill. Two (2) litres of media was weighed and charged into the mill, where it was preconditioned in water for 1 hour. The water flowrate was kept constant at 2.6 l/min, with a constant agitator speed of 2100 rpm, as per the standard DML procedure⁴. A total of 10 to 12 litres of media for each of the media types selected were preconditioned in 2 litre batches. During the hour long preconditioning, the power draw and discharge temperature of the mill was recorded every 5 minutes.

The conditioned batch of media was then dried and weighed to determine the preconditioning mass loss. During the milling process surface defects are removed and substandard beads will break up into smaller pieces and if small enough will automatically exit through the centrifugal separator. After preconditioning the media is removed from the mill and dried. The dried media is then screened using a 2mm screen to remove any sub 2mm particles that was formed due to beads failing but too big to leave the mill via the centrifugal separator. The batches of conditioned media for each type were then stored in 20 litre buckets for use during the water and slurry testing phases of the study.



Figure 28: The Netsch M4 Mill with Agitated Tanks

3.3 Water Testing of Media in a 4-litre horizontal stirred mill.

As indicated in table 6, a specified volume of ceramic media was measured and charged to the mill through the media addition port for each of the water tests.

Table 6: Volumes of Ceramic media used for water testing

Volume of Media Used for Water Testing (ml)
1000
1300
1500
1700
2000

The mill was started up and adjusted to 2100 rpm while water was fed to the mill at a rate of 2.6 l/min. For the full period of the test, one (1) hour, the power draw and discharge temperature of the mill was recorded every 5 minutes.

3.4 Slurry Testing of Media in a 4-litre horizontal stirred mill.

Preconditioned media prepared as described in section 3.2 was used for the slurry test runs and the test matrix presented in Table 7 was followed for the test program.

Table 7 Slurry Test Matrix

Media Type	Media Volume		
	1000ml	1500ml	2000ml
Ceramic A	No	No	Yes
Ceramic B	No	No	Yes
Ceramic C	Yes	Yes	Yes
Ceramic D	No	No	Yes
Ceramic E	No	No	Yes

For all media types, two (2) litres of ceramic media was measured and loaded into the mill for the slurry test. For the Zirconium Silicate Type 1 media (Ceramic C), tests were also conducted using 1,0 and 1,5 litres of ceramic media respectively, to assess the impact of media load on grinding efficiency.

The UG2 slurry used as feed was produced at the DML Pilot Plant in Rustenburg. A 100% -12 mm ore feed was milled using two stage ball milling (Figure 29). This produced a feedstock of UG2 ore for laboratory tests with a P80 passing size of 105.8 μ m for the tests.

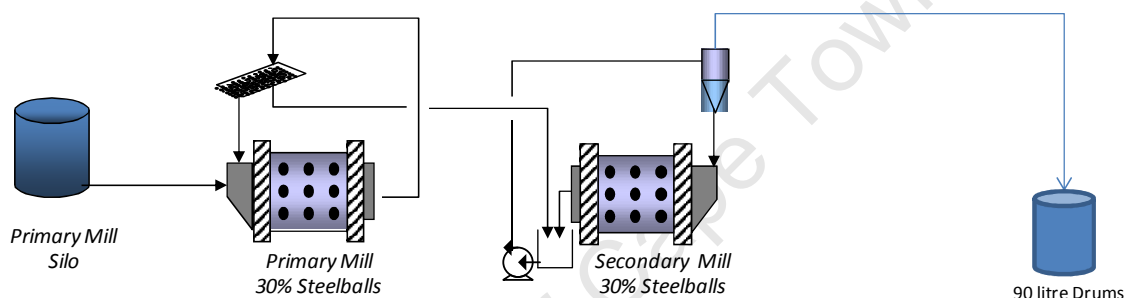


Figure 29: Flowsheet for preparation of ore for slurry testing

A thousand (1000) litre bulk sample of the secondary cyclone overflow was collected, by diverting the whole overflow stream into ninety (90) litre dustbins. The solids were then allowed to settle and all water decanted prior to transportation to the metallurgical laboratory for further preparation. The bulk samples were filtered, dried and then put through the lump breaker prior to blending and splitting. Blending was done using the standard DML bulk blending and odds-and-evens blending techniques given in Appendix A. After blending, the bulk sample was split into forty (40) kilogram aliquots for slurry make up, using a 10 way rotary splitter.

For every test, a forty (40) kilogram aliquot was used to make up feed slurry containing 40% solids in a stainless steel agitated tank. The slurry was allowed to circulate a few times to ensure that the slurry is well mixed prior to commencing with the test.

After ensuring that the slurry was well mixed, 5 feed samples were collected for particle size distribution (PSD) analysis using a Malvern Mastersizer 2000 laser diffraction particle size analyser. Once mixed the horizontal stirred mill was started and slurry pumped through the mill in a single pass for 30 minutes during which time energy, flowrate, pressure and temperature were measured. After every 5 minutes a sample was collected for PSD analysis using a Malvern Mastersizer 2000 laser diffraction particle size analyser for every pass of the batch through the mill. All PSD samples were later analysed off-line. The discharge from each pass was collected in an agitated tank and fed to the mill for the next batch to be passed through the mill. In total the slurry material is passed through the mill four times during one slurry test as illustrated in the series of flow diagrams in figure 30-33.

During the 4th pass the mill discharge was discarded and upon completion the mill was drained and flushed with potable water. This was to ensure that all residual solids in the system are removed prior to the commencement of the next test. At the completion of the slurry test, the media is washed, dried and weighed to determine mass loss during the test.

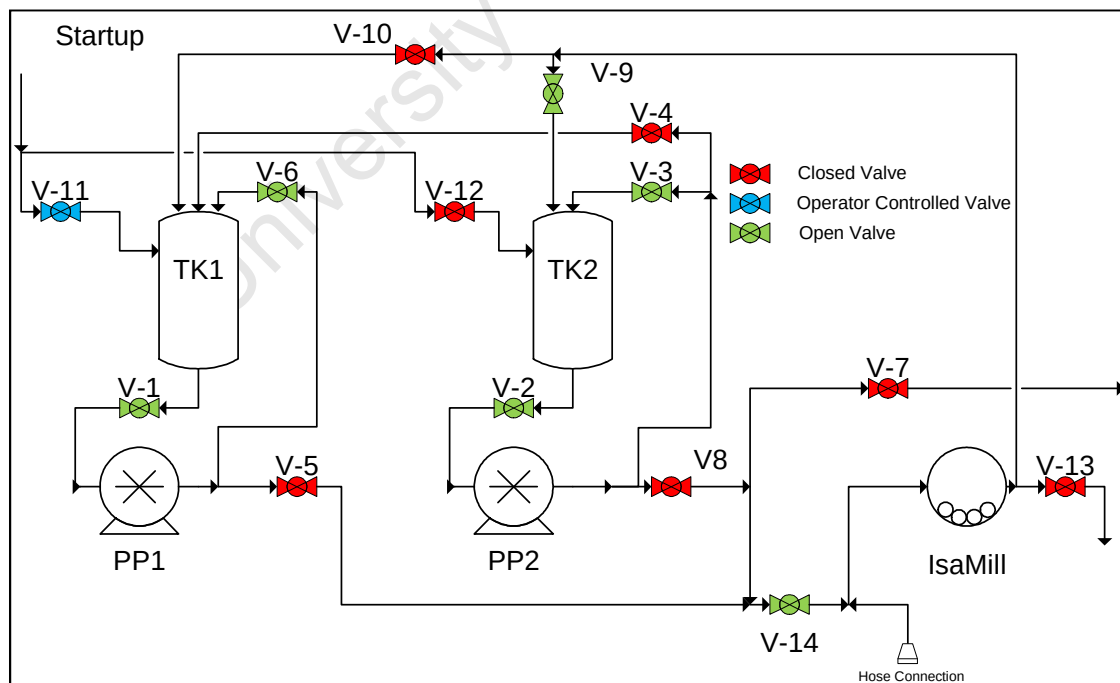


Figure 30: M4 Start-up Valve Operation

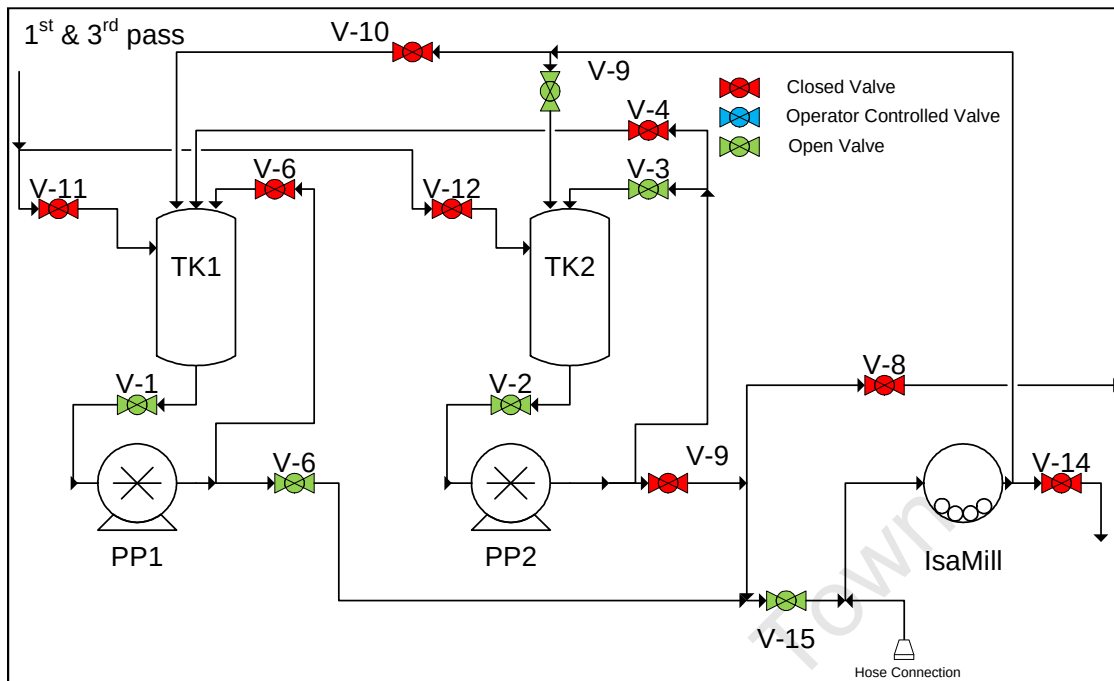


Figure 31: M4 IsaMill™ 1st and 3rd Pass Valve Operation

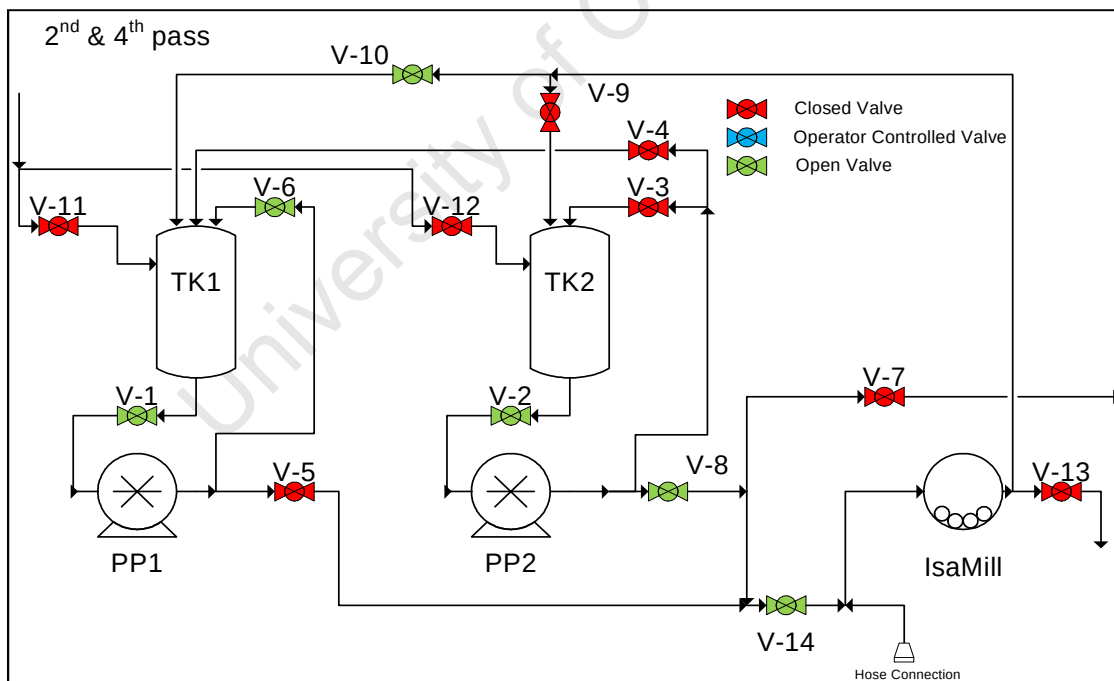


Figure 32: M4 IsaMill™ 2nd and 4th Pass Valve Operation

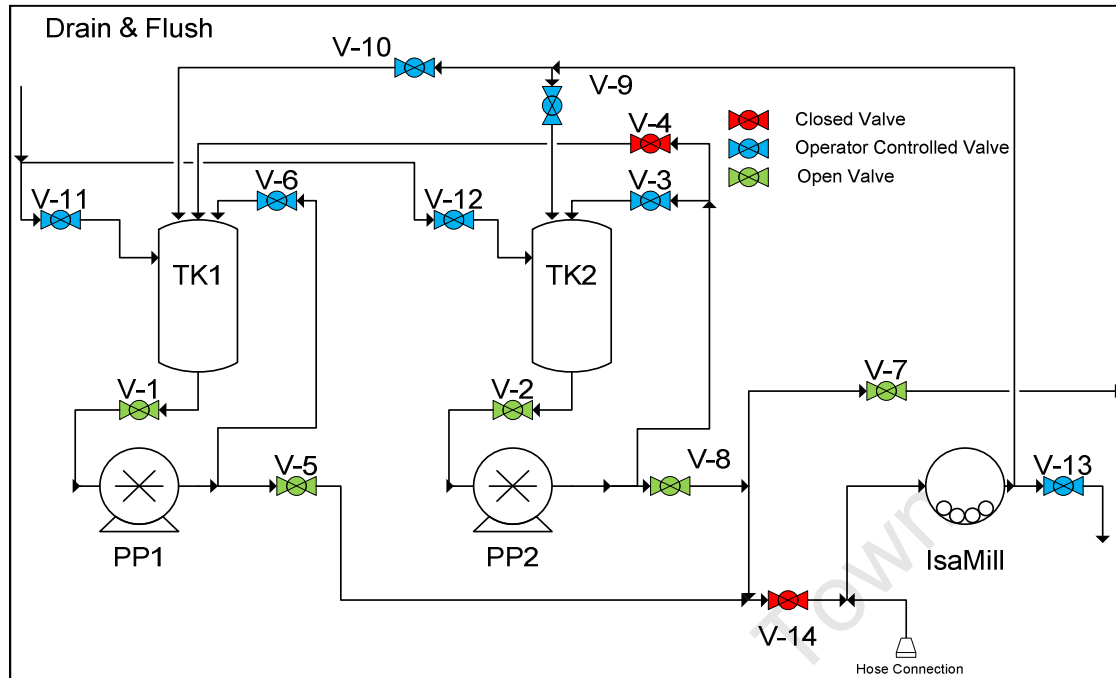


Figure 33: M4 IsaMill™ Drain and Flush Valve Operation

The P_{80} values from each pass through the mill are plotted against their respective net energies to produce a signature plot, which is a measure of grinding efficiency^{11, 14,15,20,21}. To produce a signature plot covering a wide range of grind sizes and specific energy with relative ease 4 passes of the material through the mill is required.

3.5 Measuring the friction coefficient of the selected ceramic media.

Friction coefficient measurements on the selected media was performed by Dr Farber of Zircoa Inc , USA using the same custom built friction coefficient tester utilized in his previous research work¹⁴. In Figure 34 the detail of the friction coefficient tester can be seen. The procedure entailed positioning of an individual bead in contact with a polished section of another bead. A normal force was applied to the test bead and friction force was recorded during lateral bead motion. Friction coefficient measurements were done under dry sliding conditions, in water and in an ore slurry environment.

For the tests in UG2 ore slurry (a platinum bearing ore with a high chromitite content originating from the Bushveld complex in South Africa), a pre-milled sample with a particle size (P_{80}) of approximately $9\text{ }\mu\text{m}$ was used. Tests conditions were as follows:

Normal Force = 300 mN ,

Lateral speed was approximately $10\text{ }\mu\text{m/s}$.

Travel distance $\sim 100\text{ }\mu\text{m}$.

The grind of the slurry for the friction tests was 100% $-9\mu\text{m}$. An ultrafine grind is used to ensure that at the small quantities used, a homogenous mix of particle sizes is present during every friction coefficient measurement. If a closely sized fine slurry is not used the friction coefficient result may be biased due to varying particle sizes in the slurry. At least 2 beads were measured for each ceramic sample, while each bead was in contact with 3 polished sections. Approximately 10 measurements for each polished section were made.

Customised Friction Coefficient Tester



Illustration of Measurement Technique

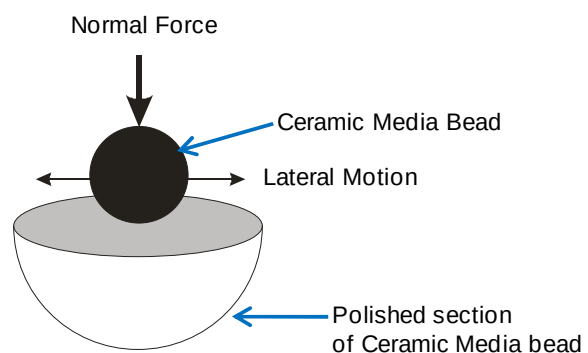


Figure 34: Test setup for ceramic bead friction coefficient measurement.

3.6 Determining the physical properties of the selected ceramic media.

The following physical properties of the selected ceramic media were measured at the DML Ceramic Media Quality laboratory in Rustenburg:

1. Specific Gravity (SG).
2. Media Size and Sphericity.

The specific gravity of the media was determined using a Micromeritics AccuPyc II 1340 Gas Pycnometer (figure 35). The Pycnometer uses helium gas to determine the skeletal density of the ceramic media, as the gas penetrates the pores of the ceramic media and therefore the true density of the ceramic media is determined. This method of SG determination is much more accurate than the more commonly practiced Archimedes method, where water is utilized to determine the media density. The water is not in all cases able to penetrate the media pores and generally the ceramic media density is understated using the Archimedes method.



Figure 35: Micromeritics Pycnometer

Determination of the media size and sphericity factor was done using equipment that was custom built for this purpose by Dr Boris Farber of Zircoa Inc in the USA for the DML Ceramic Media Quality Control Laboratory (Figure 38).

A sample of the conditioned media was secured on a glass slide with two sided tape. A photograph of the media is taken by using backlighting illumination with a high resolution camera (figure 36) and the Image Analysis program then detects the bead shape by analysing the bead black level (figure 37)

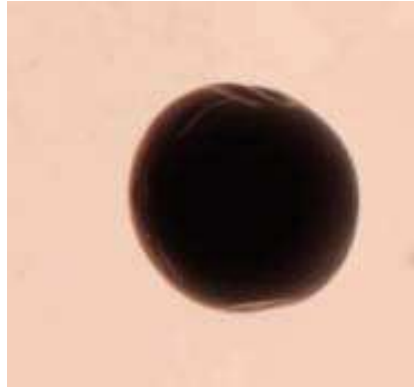


Figure 36: Photographic Image of bead with Backlight Illumination.

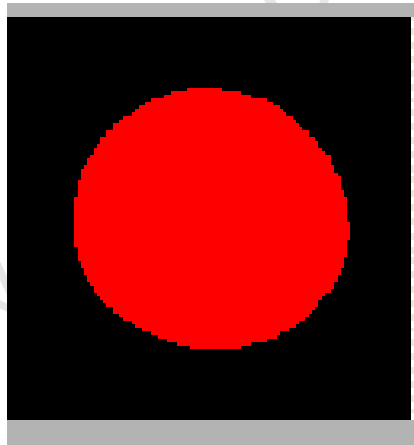


Figure 37: Bead Black level Image used for analysis



Figure 38: Custom Built Ceramic Media Sizing Equipment

The bead image is represented by an ellipse with an area equivalent of the image projected by the bead. For every bead analysed the major (a) and minor (b) axis is measured as illustrated in Fig 39 below.

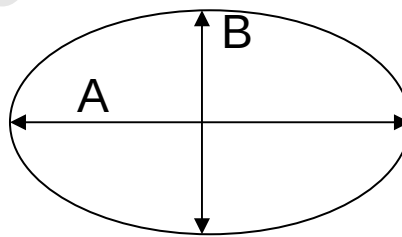


Figure 39: Defining the Major (a) and Minor (b) axis of the ellipse.

Aspect ratio, which is a description of the sphericity of the media, is calculated using equation 3.1

$$\text{Aspect Ratio (AR)} = \frac{\text{Major Axis (a)}}{\text{Minor Axis (b)}} \dots \dots \dots [\text{Eq 3.1}]$$

After analysing 100 beads, the imaging software produces a detailed analysis report as shown in figure 40. The following information is contained in the report:

- Particle Size Distribution of Beads
- Frequency distribution of Major and Minor Axis measured with statistical average and standard deviation.
- Frequency distribution of the calculated aspect ratio of the beads measured.

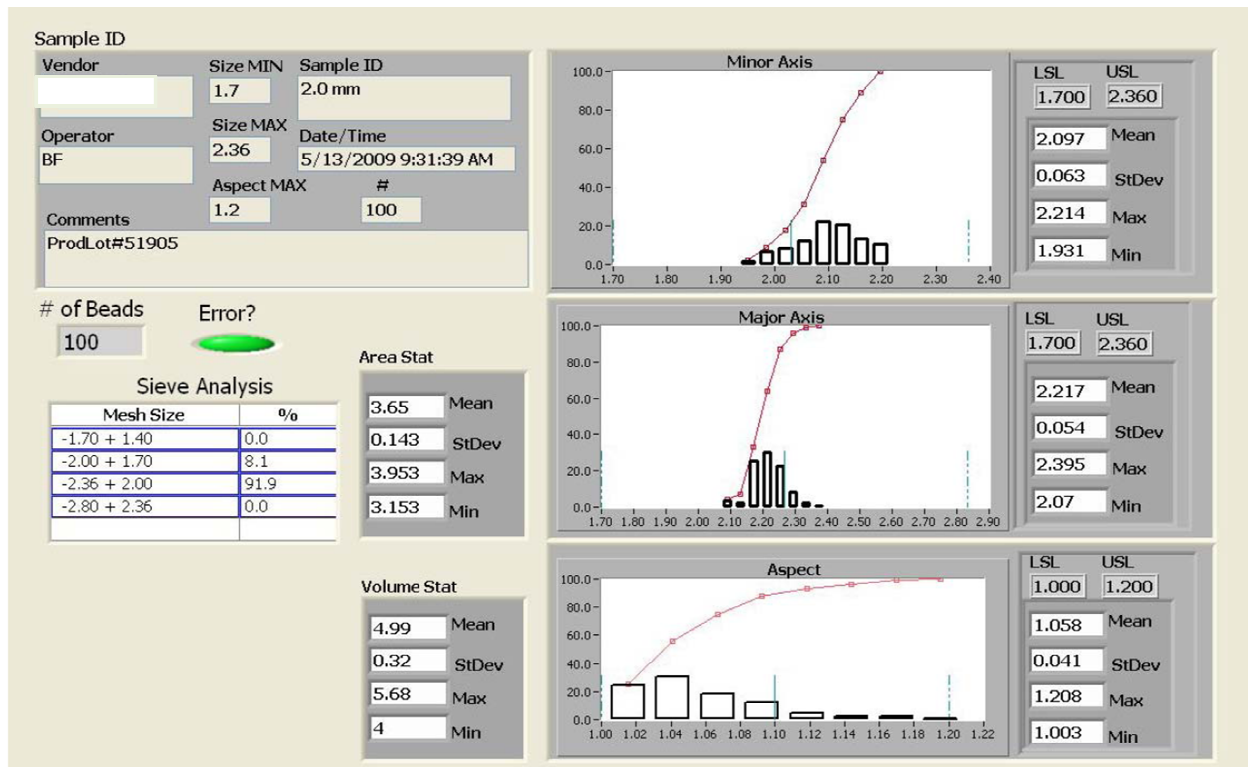


Figure 40: Example of the Imaging Report

The research methodology was designed to produce reliable data on the ceramic media properties and to obtain reliable IsaMill™ energy data under varying conditions. The data generated using the research methodology will be used to investigate the relationship between media properties and power consumption of an M4 IsaMill™.

CHAPTER 4 – RESULTS AND DISCUSSION

Chapter 4 of the thesis details the results and findings of both the historical data analysis and the experimental program followed.

4.1 Analysis of DML Ceramic Media database data.

The first stage of the research methodology was to evaluate the applicability of Farber's methodology¹⁷ by applying it to a ceramic database, containing a wide range of media types. The data contained in the database used in this analysis was generated using the methodology described in Chapter 3. This is the method used for evaluating and selecting ceramic media for IsaMill™ applications, in Anglo American's platinum operations⁴.

A total of five (5) different media types of varying composition were selected from the database for this analysis. Included in the analysis, were four (4) media of the same composition from two (2) different suppliers, which only differed due to minor modifications in the manufacturing process of the ceramic media beads. The properties of the media selected are presented in table 8.

Figures 41 and 42 indicate the relationship between the power draw of the M4 mill and the density of the ceramic media. It can be seen that there is no relationship between the power draw and media density in a water or slurry environment exists. The full set of results analysed for the selected media is given in Appendix B.

Table 8: Physical and Chemical Properties of media selected for analysis from the AAP Database.

Media Type	Physical Properties				Chemical Properties (%)		
	Size (mm)	Bulk Density (kg/l)	Specific Gravity (g/cm ³)	Vickers Hardness	ZrO ₂	Al ₂ O ₃	SiO ₂
Alumina	3.56	2.23	3.57	1116	-	90	10
Zirconia Toughened Alumina A1*	3.60	2.46	3.81	1274	14	79	6.5
Zirconia Toughened Alumina A2*	3.45	2.47	3.82	1363	14	79	6.5
Zirconia Toughened Alumina B1*	3.44	2.44	3.81	1277	13	80	7
Zirconia Toughened Alumina B2*	3.13	2.46	3.81	1314	13	80	7
Zirconium Silicate 1	3.36	2.50	4.00	1020	62	-	33
Zirconium Silicate 2	3.14	1.98	3.20	1206	30	25	40
Alumina Toughened Zirconia	3.06	3.36	5.31	1270	70	30	-
Ceria Stabilised Zirconia**	2.96	3.94	6.23	1178	98	-	-

*A and B Denotes that the media was obtained from different suppliers.

** Chemical properties as received from supplier indicated a total ZrO₂ and Ceria content of 98%. No detailed analysis of ZrO₂ and Ceria and Ceria content was provided.

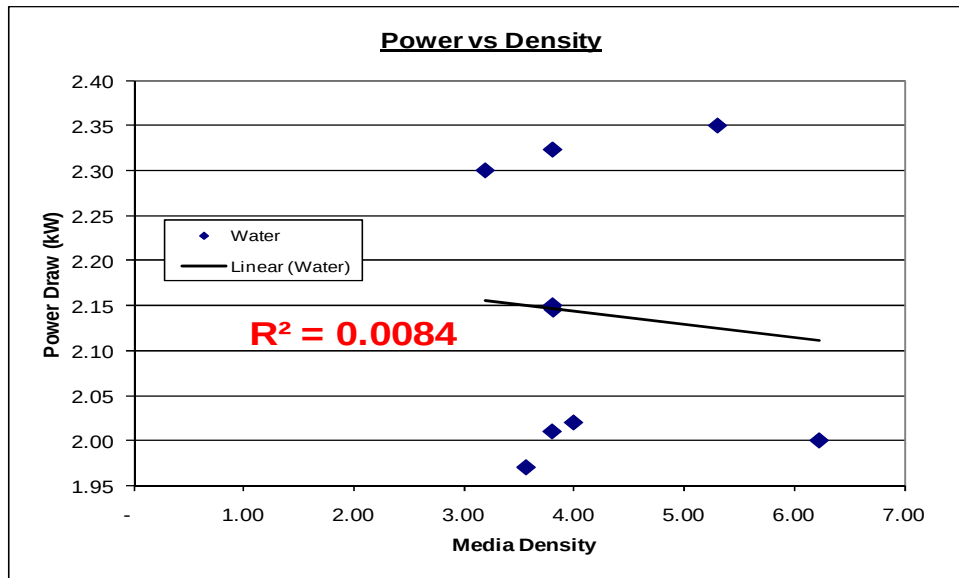


Figure 41: Relationship between media density and power draw as measured in water

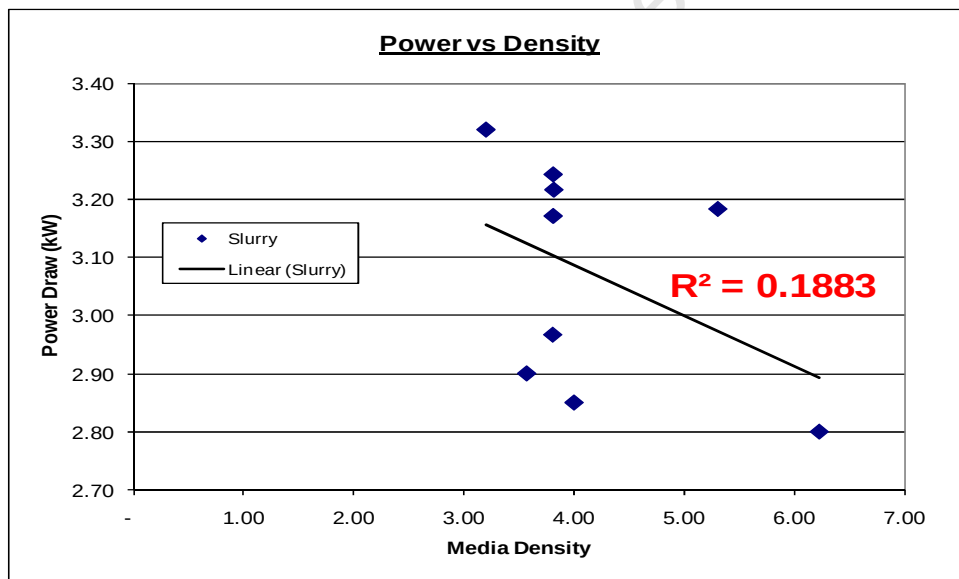


Figure 42: Relationship between media density and power draw as measured in slurry.

The next variable to consider was whether the media mass added to the mill could be related to the power draw. The following reasoning was applied: If the power draw was related to the mass, then the density and power draw would also have exhibited a correlation. Mill filling for all tests was kept constant at a 2 litre fill.

If the reasoning was true the media with the highest density and mass filling would exhibit the highest power draw. Figure 43 and 44 indicate that there is no relationship between media mass and power. This is unexpected as conventional milling theory and DEM simulations²⁶ indicated an increase in power draw with an increase in media density.

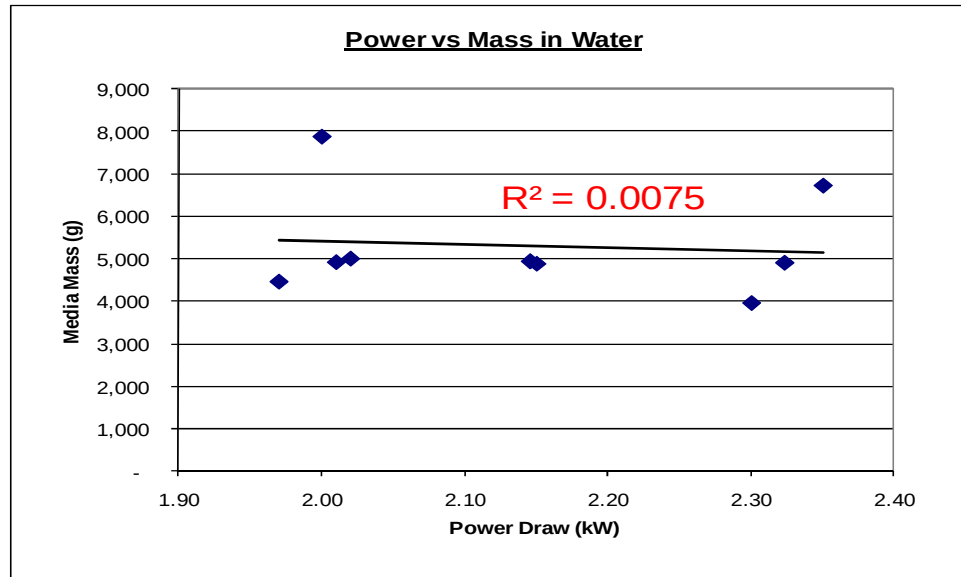


Figure 43: Relationship between media mass and power draw as measured in water.

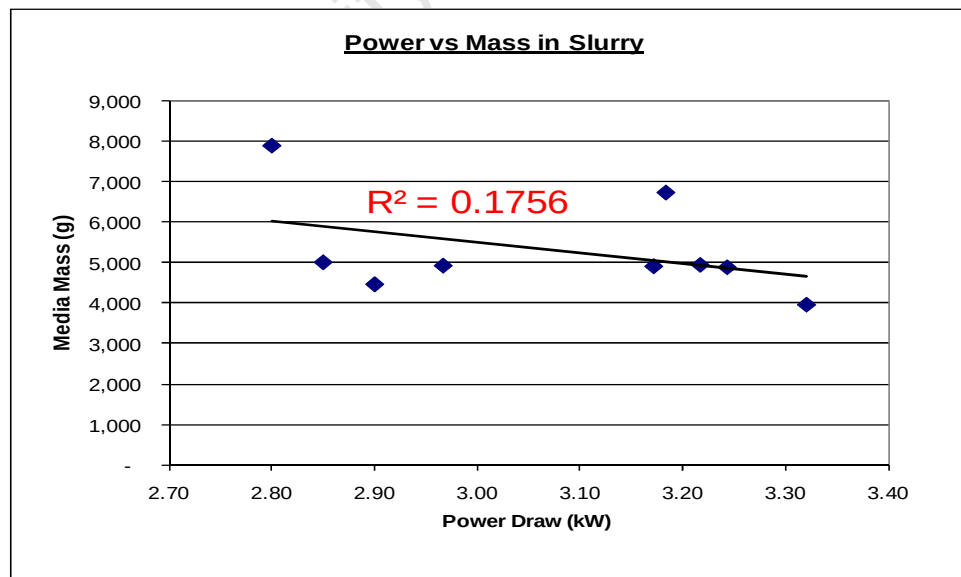


Figure 44: Relationship between media mass and power draw as measured in slurry.

Another way of analysing the data is to normalize the power draw by calculating a power draw per kilogram of media added to the M4 IsaMill™ in operation. When plotted against media density as presented in figure 45, it indicates that the higher the media density the lower the kW consumed per kg of media. If the mass and density of the media was a major energy driver the reverse would be true and the highest density media should have used the most kW per kilogram of media.

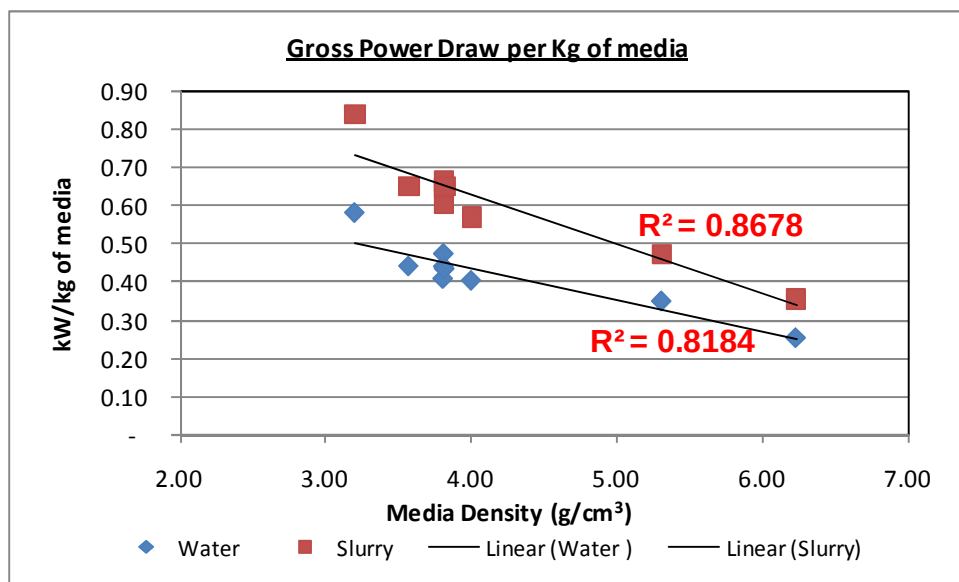


Figure 45: Power draw per kg of media.

The results also indicate that even media with the same densities and mass exhibit differences in power draw per kg of media added to the horizontal stirred mill. Therefore it is plausible to hypothesize that Farber's¹⁷ conclusion that media surface friction coefficient is the major contributor towards power draw in a horizontal stirred mill is true.

Using Farber's¹⁷ methodology the surface friction coefficient for the media types selected for the analysis was calculated using equation 2.4.7. For calculation purposes it was assumed that the contribution of density (C_p) of 0.15 determined using the method developed by Farber¹⁷ holds true for all media types. The calculated coefficient of friction for the different media types is presented in figure 46.

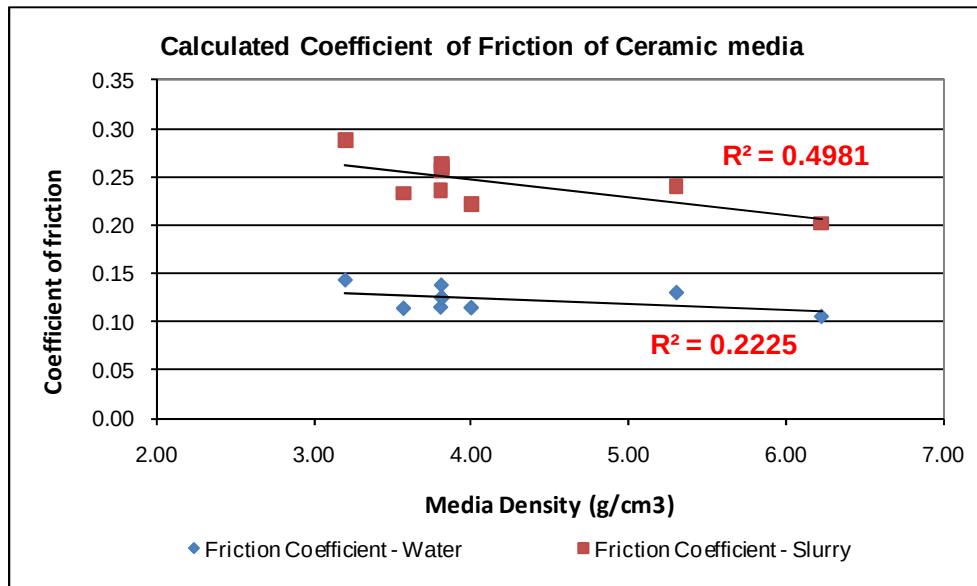


Figure 46: Surface Friction Coefficient Calculation for Media in Water and Slurry.

Results are in agreement with the findings of Farber's earlier work and indicated the difference in the friction coefficient between media. The work also indicates the large difference in the friction coefficient between slurry and water, which has an effect on energy consumption with the addition of ore particles to the system. The R^2 values were also low indicating that Farber's¹⁷ methodology may not be fully describing the influence of the various media properties on energy consumption.

The assumption of a uniform C_p of 0.15 has not been validated yet. An attempt to do that is made in the second phase of this work. After establishing that power draw in a horizontal stirred mill is predominantly dependent on the friction coefficient of media in water and slurry as presented in Figure 47, it is reasonable to evaluate how the difference in power draw affects the grinding of ore particles. The primary purpose of the mill is to liberate valuable minerals from gangue minerals and this requires sufficient energy input^{16, 17}.

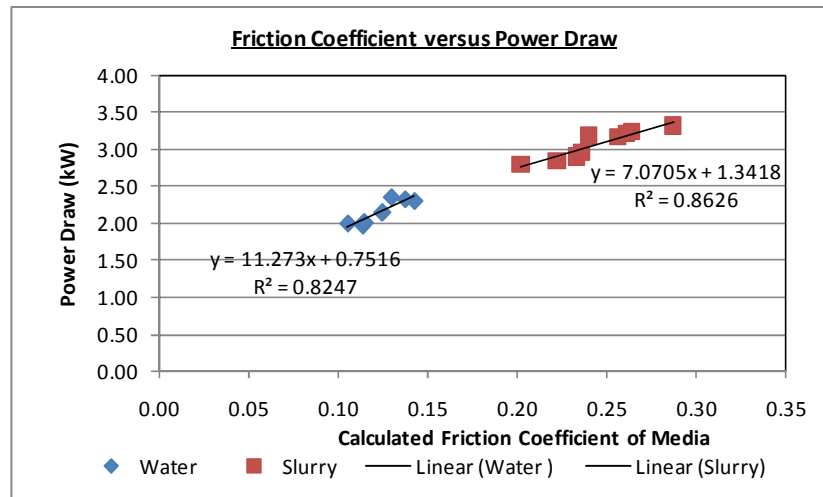


Figure 47: Relationship between media friction coefficient and mill power draw (kW).

Data from the IsaMill™ tests performed using different ceramic types was used to determine the effect of power draw on product size. The grind data was generated from the recycle tests performed under identical operating conditions where the material is recycled through the IsaMill™ four times. The feed and discharge PSD is then used to determine the 80% passing size in μm (P_{80}) of the original feed and the discharge of every pass.

The specific energy requirement to reach the target grind was calculated based on the net power draw of mill and the feed solids flowrate to the mill. The net power draw of the mill was calculated by subtracting the No Load Power from the total power measured when operating the mill on slurry.

The P_{80} is then plotted against the specific energy calculated for each pass on a log-log plot and a trendline fitted to data to construct a signature plot that defines the grinding efficiency of the mill under those conditions. Using the fitted trendline the specific energy (kWh/t) required to reach a target P_{80} grind of $50\mu\text{m}$ was calculated. The signature plots developed from the slurry tests are presented in Figure 48.

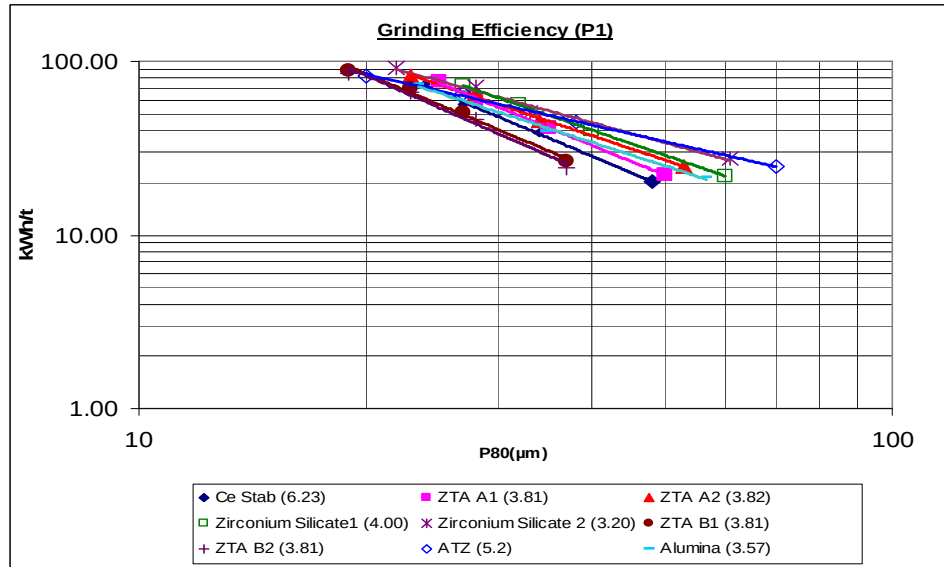


Figure 48: Signature plots for UG2 Slurry testing.

Table 9 presents the grind and energy consumption data of the media as calculated from data collected during the slurry test. The ZTA media A1 was chosen as the standard media for comparative purposes because that was the media used in industrial stirred mill applications. The kWh/t of energy required to obtain a discharge grind P_{80} of 50 μ m was calculated by fitting a straight line to the signature plot data.

Table 9 indicates that the specific energy required for grinding the feed to a target P_{80} of 50 μ m was different for the various media types tested. It was very difficult to attribute the difference in specific energy requirements to the media type difference, because the feed size used in the testwork was different. This was due to the method used by DML historically to collect the bulk samples for media evaluation. Bulk final tailings samples was taken from the Waterval UG2 Concentrator in 90litre drums and on arrival at DML the solids were allowed to settle and the water decanted. Thereafter water was added to make up slurry to correct density for media evaluation slurry tests, normally 40% solids concentration. It is well known that industrial milling and flotation processes are subjected to variable operation from time to time and this is evident from the variation in the feed P_{80} of the slurry used in the media evaluation.

Table 9: Calculated Specific energy, wear and costs for historical test data.

Media Type	Specific Energy for P_{80} of 50μm (kWh/t)	Difference in Specific energy (%)	Feed P_{80} (μm)	Media Consumption (g/kWh)	Relative Media Cost (R/kg)
Ce Stabilised Zirconia	18.79	-14.6	135	1.14	2.64
Zirconium Toughened Alumina A1	22.02	0	110	12.32	1.00
Zirconium Toughened Alumina A2	26.88	22.1	111	6.38	1
Zirconium Silicate 1	28.66	30.2	170	3.12	0.61
Zirconium Silicate 2	33.89	53.9	167	4.86	0.86
Zirconium Toughened Alumina B1	15.82	-28.1	96	10.99	0.55
Zirconium Toughened Alumina B2	18.98	-13.8	100	14.52	0.55
Alumina	24.88	13	125	11.18	0.56
Alumina Toughened Zirconia	34.38	56.1	110	5.92	0.53

Chaponda (2011)¹³ in his studies using a 4-litre horizontal stirred mill showed that the grinding efficiency is very sensitive to feed size and that the signature plot shifts with a change in feed F_{80} as illustrated in figure 49.

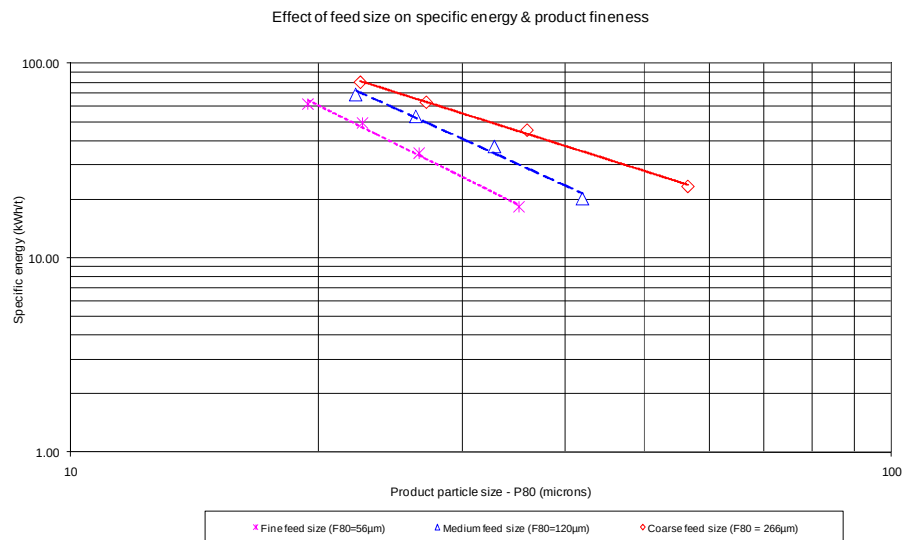


Figure 49: Effect of feed size on specific energy and product fineness (Chaponda 2011)¹³

The work by Chaponda¹³ indicated that the specific energy requirements can be over or understated depending on how the feed size changed. Farber^{16, 17} also indicated that media size required reaching a target P_{80} is also dependant on the density of the media and feed size. The higher the media density the smaller the top size of media required. If similar size media data is considered and taking only tests performed at the same media load into account, the high energy requirements for the Zirconium Silicate media can be attributed to the coarser feed particle size distribution. It is difficult to identify the most energy efficient media for grinding UG2 ore if the feed size is varying.

Therefore for subsequent media evaluation and the practical phase of the testwork, a new methodology was implemented to prepare a feed sample for laboratory media evaluation. The new methodology required a 100% -12mm crushed UG2 ore to be milled under carefully controlled conditions at DML's pilot plant facility through a primary and secondary ball mill to produce a feedstock with a consistent grind for the laboratory tests (See Chapter 3).

Bulk samples were collected of the secondary cyclone overflow, as soon as the plant is operating under stable conditions, in ninety (90) litre drums and allowed to settle before the water was decanted from the samples. The sample was then filtered and dried before lump breaking in disc type lump breakers. The bulk coarse screened sample was then blended and odds-and-evens portions were re-blended before the sample was split into 40kg aliquots that were stored for subsequent media evaluation trials.

Implementation of this methodology ensured that a closely sized feed was used for all subsequent experiments and that all data generated during media evaluation trials can be used and direct comparison of all media made. As long as the plant operation is stable at the specified operating conditions, the same product size can be produced on different days. This would ensure the PSD is reproducible for future sample batches. This is very useful for future test work as bulk feed samples can be reproduced to within a narrow specification for all future M4 IsaMill™ work.

From the analysis of the data, Farber's¹⁷ methodology can be applied to all types of media under controlled circumstances but that the contribution of density factor should still be validated for all media types. Farber's methodology¹⁷ is limited to predicting the power of an alternative media at the same volumetric media fill as the model does not cater for varying loads. The formula can also not predict the power draw of the same media at a different volumetric fill. The analysis also highlighted some shortcomings with the previous testing methodology that required to be corrected prior to commencing with the practical portion of the study.

4.2 Physical and Chemical Properties of selected media.

Five different ceramic media types with different chemical compositions and densities of similar size were selected for subsequent testwork as presented in Table 10.

Table 10: Physical and Chemical Properties of Selected Media.

Name	Chemical Composition			Media Type	Physical Properties		
	% ZrO ₂	% SiO ₂	% Al ₂ O ₃		Density (kg/l)	Size (mm)	Aspect Ratio
Ceramic A	14	6.5	79.5	Zirconia Toughened Alumina	3.75	3.57	1.11
Ceramic B	13	7	80	Zirconia Toughened Alumina	3.91	3.51	1.17
Ceramic C	62	33	-	Zirconium Silicate	4	3.35	1.01
Ceramic D	65	35	-	Zirconium Silicate	3.2	3.28	1.01
Ceramic E	98*	-	-	Ceria Stabilized Zirconia (Ce-TZP)	6.2	2.96	1.06

*The 98% is quoted for Ceramic E indicates the total ZrO₂ and CeO₂ content, as a detailed breakdown of the two elements were not provided by the supplier.

Media from 3 different media classes or types was included in the evaluation. In two of the classes a 2nd media of the same class, with similar chemical composition, but from alternative suppliers were also chosen for the evaluation. Varying degrees of sphericity, as indicated by the aspect ratio obtained from the imaging exercise, was evident for all the media with the Zirconium silicate media being the only media that exhibited a shape very close to a sphere.

The size distribution of the media is presented in figure 50. It can be seen that although the average size of the media was similar that distinct differences in the media size distributions existed. The two Zirconium Silicate media, C and D, had similar particle size distributions. Size was used in development of the models by using the average media size to calculate the number of media particles and subsequently the total surface area of the media in the mill.

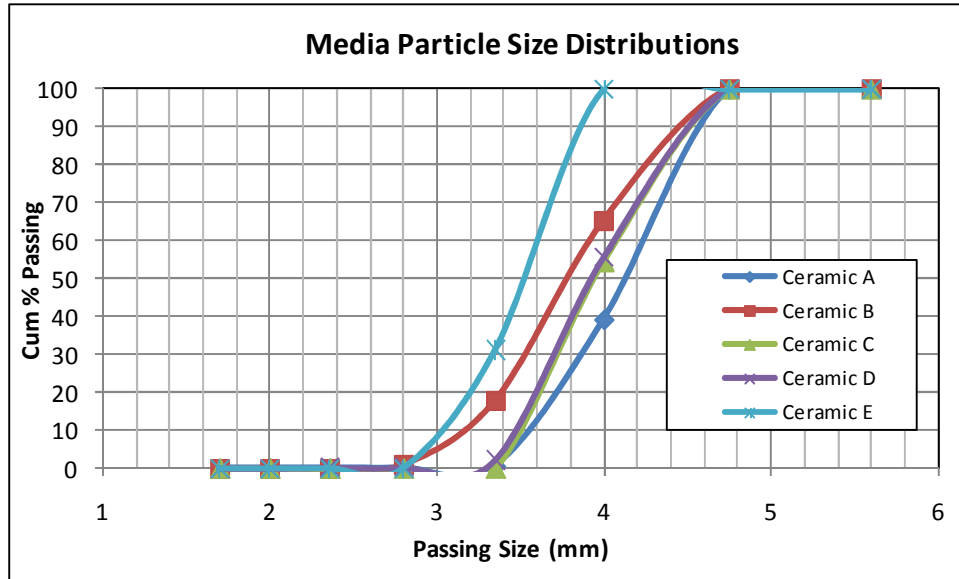


Figure 50: Size Distributions of the selected media

Table 11 indicates that the selected media had varying hardness (HV) and fracture toughness (K1C) as measured using the Vickers Hardness tester.

Table 11: Hardness and Fracture Toughness of Selected Media.

Media Type	Hardness (HV)		Fracture Toughness (K1C)	
	GPa	Std Dev	MPa*m ^{1/2}	Std Dev
Ceramic A	12.16	0.18	5.23	0.70
Ceramic B	11.51	0.80	5.60	0.79
Ceramic C	9.81	0.16	1.68	0.18
Ceramic D	8.5	0.44	2.45	0.66
Ceramic E	11.56	0.10	12.78	1.22

Hardness of the Ceramic E and the ZTA media (A and B) were similar, but the fracture toughness of Ceramic E was superior to that of the ZTA media. The zirconium silicate media (C and D) had the lowest hardness and fracture toughness of all 3 media types. Table 12 presents the Friction Coefficient measurement results of the selected media in a dry, water and slurry environment.

Table 12: Friction Coefficient measurements of Selected Media.

Media Type	FC Dry	Error Dry	FC Water	Error Water	FC Slurry	Error Slurry
Ceramic A	0.14	0.005	0.18	0.006	0.33	0.007
Ceramic B	0.13	0.004	0.15	0.003	0.28	0.005
Ceramic C	0.11	0.004	0.14	0.004	0.30	0.007
Ceramic D	0.17	0.008	0.19	0.009	0.33	0.008
Ceramic E	0.11	0.005	0.13	0.003	0.24	0.007

Results indicate that in different environments the media exhibited different friction coefficient measurements with the highest increase in the friction coefficient noted in a slurry environment. Media with similar chemical composition (A&B and C&D) also exhibited significantly different friction coefficients in both the water and slurry environments.

In the dry environment however there was some unexpected results as ceramic A and B exhibited similar friction coefficients, while ceramic C and D's friction coefficients was significantly different. In all 3 environments tested Ceramic E, exhibited the lowest friction coefficient.

4.3 Preconditioning Results and assessment of the repeatability of the experimental procedure.

To ensure that only competent media was used in the water and slurry testing a bulk sample of each media type was preconditioned in 2l batches to produce a bulk sample of competent media to be used for water and slurry testing. In an effort to reduce the number of tests, the multiple pre conditioning tests for the media was used to ascertain how repeatable the test procedure is.

As power draw measurements were recorded both by means of the kWh totaliser on the mill and by manually recording the instantaneous power draw every 5 minutes. An assessment was done to evaluate if the manual readings taken can be averaged over a 1 hour period to give a representative power measurement. This was done by comparing the averaged totaliser values to the averaged instantaneous readings.

Table 13 presents the comparison of the two power measurements for all 5 ceramic media types tested. Results indicate that over the multiple preconditioning steps the repeatability of the test procedure is within acceptable limits. Very low standard deviations were recorded for both the totaliser and average instantaneous power draw measurements. Both power draw measurements are recorded to only one decimal but since multiple measurements were taken manually. This allows for a more accurate calculation of the overall power draw. The error between the two readings is also less than 6% and therefore it was decided to use the results of the instantaneous measurements taken manually for the remainder of the data analysis.

Table 13: Comparison of power measurements for Preconditioning Tests.

Media Type	Totaliser Power Draw (kWh)		Manual Reading Calculated Power Draw (kWh)		% Error between Power Measurements
	Average	Standard Deviation	Average	Standard Deviation	
Ceramic A	1.900	0.100	1.931	0.077	3.077
Ceramic B	1.980	0.110	2.037	0.083	5.692
Ceramic C	1.820	0.045	1.828	0.054	0.769
Ceramic D	2.020	0.045	2.048	0.041	2.769
Ceramic E	1.900	2.03E-13	1.942	0.016	4.154

The results from the preconditioning tests indicate different power draws for different types of media. Even media with the same composition, such as media types A and B, and media from the same media class such as C and D exhibited different power draws.

Another test for repeatability of the procedure and competency of the media is to analyse the mass loss or wear data associated with each media type. As the wear characteristics of the media may vary within a batch of media, the preconditioning data generated was also used to evaluate the homogeneity of the bulk sample and to ascertain whether the two litres aliquots obtained from the bulk samples were representative of the bulk. Table 14 presents the wear analysis for the preconditioning tests.

Table 14: Comparison of wear losses during Preconditioning Tests.

Media Type	% Wear Loss	
	Average	Standard Deviation
Ceramic A	0.26%	0.00044
Ceramic B	0.83%	0.00148
Ceramic C	0.06%	0.00011
Ceramic D	0.32%	0.00166
Ceramic E	0.02%	0.00008

During the preconditioning phase the media exhibited different wear behaviour as expected. The general trend as reported in literature was validated by lower wear rates with increased zirconia content³. The low standard deviation on the wear loss tests suggests that the smaller aliquots was representative of the bulk sample and that the blending and splitting produced representative sub samples for the testwork. It is however very important to also keep in mind that the Zirconia content is not the only parameter determining the wear rate of the ceramic media beads, but factors such as the size to which the raw material was ground also plays a significant role in the wear resistance of media^{2,3}. The wear results also indicate that the media with similar compositions (A and B) or media from the same ceramic media category (C and D) can exhibit differential wear during stirred milling. The degree of confidence that the testwork could be repeated reduced the amount of testwork required in subsequent stages, where the effect of power draw at various mill fillings was evaluated.

4.4 Evaluation of the relationship between media properties and power draw in a water system.

4.4.1 The relationship between ceramic media density and power draw.

The first step in analysing the data in an attempt to link the physical properties of the media to the power draw of the M4 IsaMill™, was to test the hypothesis that the density contribution of the media to power draw of the M4 IsaMill™ is minimal. Grinding in a stirred mill takes place in a 3 phase system of ore, water and media and it is difficult to completely decouple the influence of the ore and slurry properties and the ceramic media properties on power draw from each other. Therefore the first phase of the evaluation focussed on a water and ceramic media system only.

Figure 51 presents the relationship between media density and power draw recorded for the M4 Ismill™ for all media types. The trend indicates that the power draw decreases with increasing media density at all volumetric fillings.

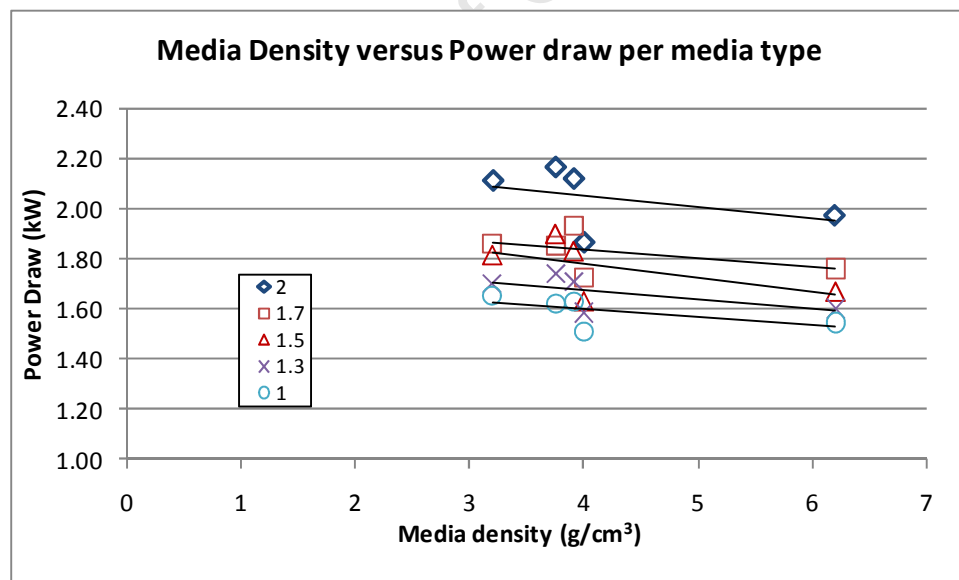


Figure 51: Density versus Power draw for different mill fillings.

Conventional ball milling energy consumption is however driven by the total mass of the grinding media in the mill and therefore a trend might not be noticeable when interpreting the relationship between media density and power draw. In the M4 IsaMill™ the media density and the media fill determine the total mass of media. If the mill mass was a predominant factor in determining the power draw of the M4 IsaMill™, a strong relationship between total media mass for all media types across a range of fills should be evident. This is not the case as shown in figure 52, where a poor linear relationship between media mass and power draw is exhibited with a R^2 equal to 0.1332. Mass contribution to power draw is very low and illustrated by the flat slope of the regression line in figure 52.

Ceramic D was identified as an outlier as the power draw measured was always lower than the linear trendline. As indicated in section 4.3 the media exhibited much higher wear rates than what was expected based on the Zirconia content of the media. It is postulated that the higher wear rate could be because of the grind of the raw material during manufacturing.

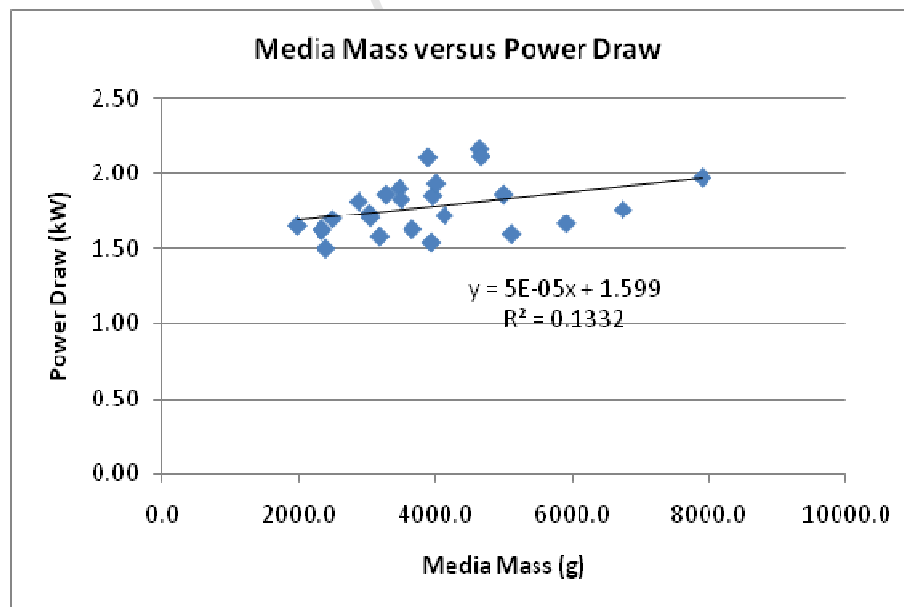


Figure 52: Media Mass versus Power draw for different mill fillings.

Based on the results presented Farber's theory¹⁷, that the density of the media plays a minimal role in the energy consumption of the mill, and his finding that the C_p of 0,15 is applicable to all media types regardless of the properties of the media could now be evaluated.

4.4.2 Density contribution to power draw for different media types.

As indicated in section 2.4.5, Farber¹⁷ developed a relationship to predict the power draw of an M4 IsaMill™ based on the density and friction coefficient of the ceramic media using equation 2.4.7. The shortcomings of the methodology were that it was based on an extremely limited database and that stated C_p of 0.15 applied to all media types was never tested experimentally or validated.

Using the results obtained from this study, Farber's methodology¹⁷ was applied to predict the power draw of the M4 mill using Ceramic Media A as benchmark media for prediction. Figure 53 presents the parity chart for the predicted and measured power draw for all the ceramic media types tested at the various filling degrees.

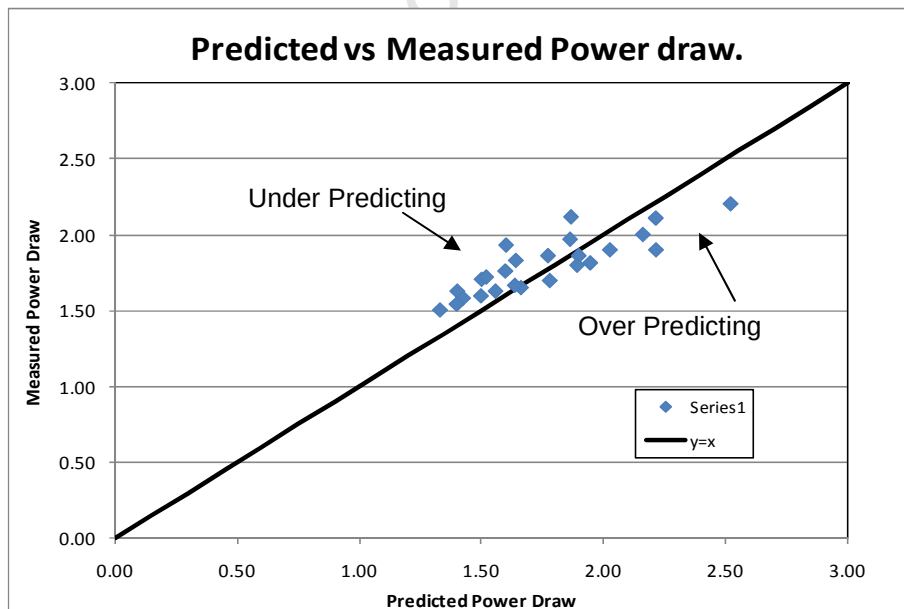


Figure 53: Experimental versus Predicted Power draw when applying Farber's methodology assuming a C_p of 0.15

The results indicate that a relatively good approximation of the mill power draw can be obtained for the M4 IsaMill™ by assuming $C_p = 0.15$. The parity chart indicates that the model under predicts the power of the lower end and over predicts at the higher power draw end.

The question however remains whether the prediction is good enough to be used in stirred mill modelling and simulations. Accurate grind predictions will rely on accurate power predictions to calculate the energy (kWh/t) applied to the ore for breakage.

The C_p for each media type was then individually fitted to obtain a correlation of the measured and predicted power draw and is presented in Table 15. The purpose of the fitting exercise was to show that using comparative tests and applying Farber's methodology¹⁷ that the density contribution is not constant for all media types.

Table 15: Density Contribution Constant (C_p) fitted to dataset using Farber's formula.

Media Type	Fitted C_p
Ceramic A	0.18
Ceramic B	0.15
Ceramic C	0.14
Ceramic D	0.19
Ceramic E	0.13

The results indicate that to obtain accurate power draw predictions the C_p for each media type need to be individually fitted. A constant C_p cannot be applied for all media types. However, it has indicated that the C_p contribution can be constant for the same media at different media fill. A laborious and time consuming exercise therefore need to be undertaken if this model is to be considered as the bases for the power draw model for the M4 mill.

4.4.3 The influence of other media properties on the power draw of an M4 Mill.

Several other media and media load properties other than the properties already discussed were measured during the experimental program, such as friction coefficient, size and aspect ratio. Other possible contributing variables of the media load such as the total surface area of the media could be calculated and the effect on mill power draw investigated.

Media friction coefficient's impact on power draw of the mill was the first parameter to be investigated as it appeared to have a significant impact on the energy efficiency of the mill as discussed in Chapter 2. If friction coefficient was the sole driver to determine power draw of the mill, then the power draw at different volumetric fills for the same friction coefficient would be the same. However, figure 54 indicates that changing the volumetric filling resulted in different power draws for the same media.

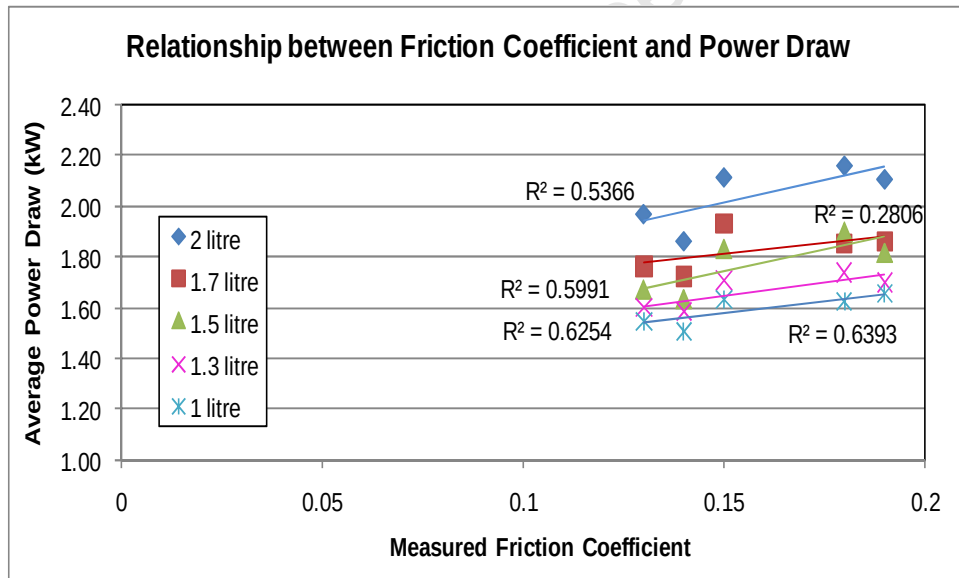


Figure 54: Friction Coefficient versus Power Draw for all media loads.

The discussion in section 4.4.1 indicated that the total mass of the media is not a significant contributor to power draw of an M4 Isamill™. The mill is known for its high intensity grinding due to the large surface area of the media in the mill, and therefore the surface area should also play a significant role in media movement in the mill as

well as the power required to impart and sustain motion to the media, the shaft and the grinding discs.

Surface area of an average size bead was approximated using the average minor and major axis dimension of the beads measured by the customised imaging equipment as detailed in section 3.6. Since the surface area of an ellipsoid can not be represented by elementary functions, the surface area of an average size bead was calculated using the Knud Thomsen formula (equation 4.4.1),

$$\text{Surface Area of ellipsoid (cm}^2\text{)} = \sim 4 \times \pi \times \left[\frac{2 \times a^p \times b^p + b^{2p}}{3} \right]^{\frac{1}{p}} \dots \dots \dots [\text{Eq 4.4.1}]$$

Where a is the major axis and b is the minor axis as indicated in figure 36.

The original formula for approximation of surface area Thomsen was first proposed by Thomsen with $p=1.58496$ for degenerate ellipsoid where 1 dimension is equal to zero. The p-value was later revised by Thomsen to 1.6075 as his investigations found that this results in the least relative error ($\pm 1.061\%$) when approximating the ellipsoid surface area. David Cantrell (2004) found that $p = 8/5$ makes the approximation nearly optimal. The maximum approximation error was found to be -1.178% . For the purpose of calculation during this study $p=1.6075$ was used to approximate the surface areas of media beads.

The volume of an average size bead was calculated for each media type using equation 4.4.2, where a and b are the major and minor axes, respectively.

$$\text{Volume of Ellipsoid (cm}^3\text{)} = \frac{4}{3} \times \pi \times a \times b^2 \dots \dots \dots [\text{Eq 4.4.2}]$$

The total number of beads in the load to the mill was then calculated using the mass of the media loaded into the mill, the media density (ρ) and the volume of an average size bead loaded to the mill (equation 4.4.3).

$$\text{Nr of beads} = \frac{\text{Total Mass of Media added to Mill(g)}}{\text{Media Density } (\rho) \times \text{Volume of an Average bead (cm}^3\text{)}} \cdot [\text{Eq4.4.3}]$$

Total media surface area of the media load in the mill was then approximated by multiplying the number of beads with the approximated surface area of an average size bead loaded into the mill. Figure 55 presents the relationship between power draw and the total surface area of media loaded into the mill.

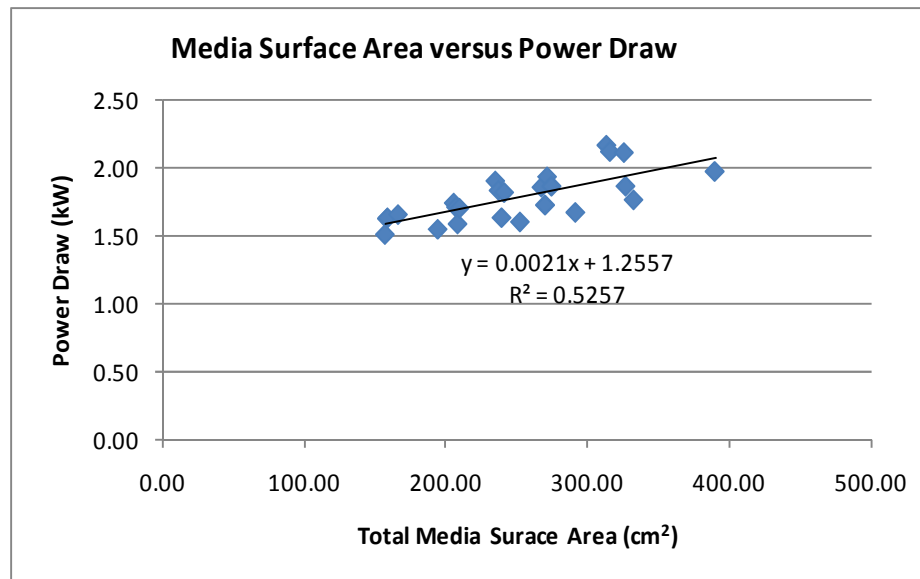


Figure 55: Relationship between total media surface area and power draw.

Total surface area of the media appears to have a much better correlation with power draw. This indicated that the media surface area in the mill plays a dominant role in the energy consumption during grinding.

If total surface area of the media is an important driver in energy consumption in the M4 IsaMill™, then combining that with the friction coefficient may provide a way of predicting energy consumption in the stirred mill. It is logical to assume that for the same surface area, more energy will be required to operate the mill at the same settings if a media with a higher friction coefficient is utilised. Figure 56 presents the relationship between mill power draw and the product of the total media surface area and the media friction coefficient.

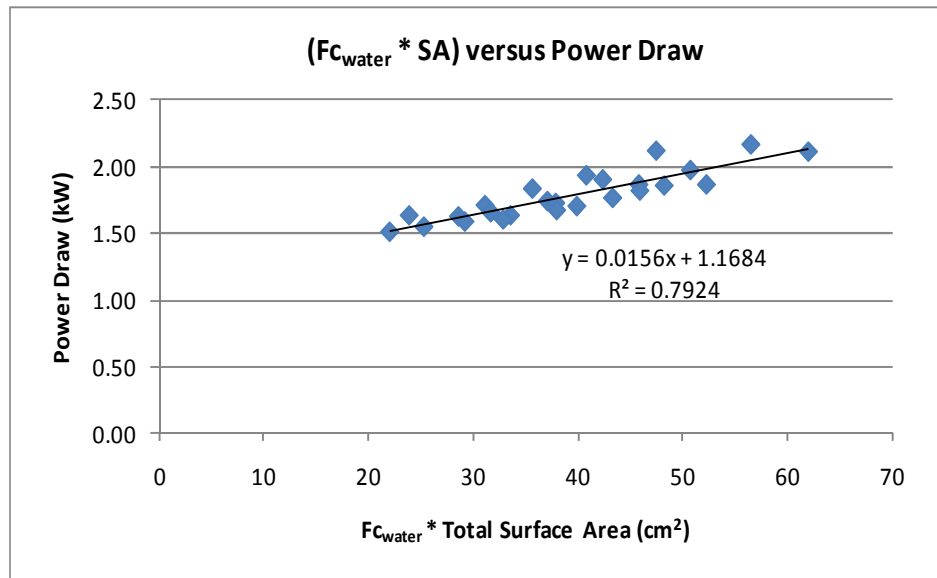


Figure 56: $F_{c_w} * SA$ versus Mill Power Draw.

A fairly good fit was obtained indicating that the media surface properties and the total surface area of the media are major contributors to energy consumption in the M4 IsaMill™. Figure 57 shows the relationship between power draw and the product of the media surface properties and friction coefficient for the various media types tested. The results indicate that although the slopes of the lines were very similar, the two Zirconia Toughened Alumina (ZTA) medias (A and B), was offset when compared to the other 3 media types tested.

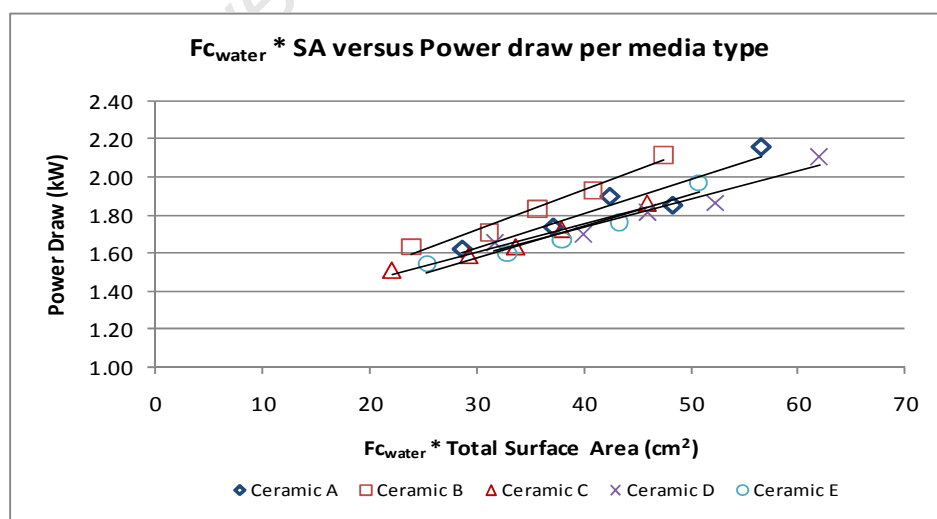


Figure 57: $F_{c_w} * SA$ versus Power draw per media type in a water environment.

Table 16 indicates the correlation coefficients, y-intercepts and the slopes of the regression lines fitted to the data for each media type. A good fit was achieved for every media type, if each media type is considered as a separate entity.

Table 16: Results of linear regression per media type

<u>Media Type</u>	<u>Slope (m)</u>	<u>Y-intercept (x=0)</u>	<u>R²</u>
Ceramic A	0.0180	1.0887	0.8971
Ceramic B	0.0209	1.096	0.9727
Ceramic C	0.0151	1.1539	0.9771
Ceramic D	0.0148	1.1403	0.9349
Ceramic E	0.0165	1.0829	0.9276

A comparison of the media properties not considered up to this point indicated that the two ZTA media had a higher aspect ratio than the other media types tested. Aspect ratio was incorporated into the analysis by plotting the product of the media friction coefficient, the total surface area of the media and the aspect ratio against the power draw of the mill per media type as depicted in fig 58. The aspect ratio was calculated as detailed in section 3.6 and the results presented in Table 10.

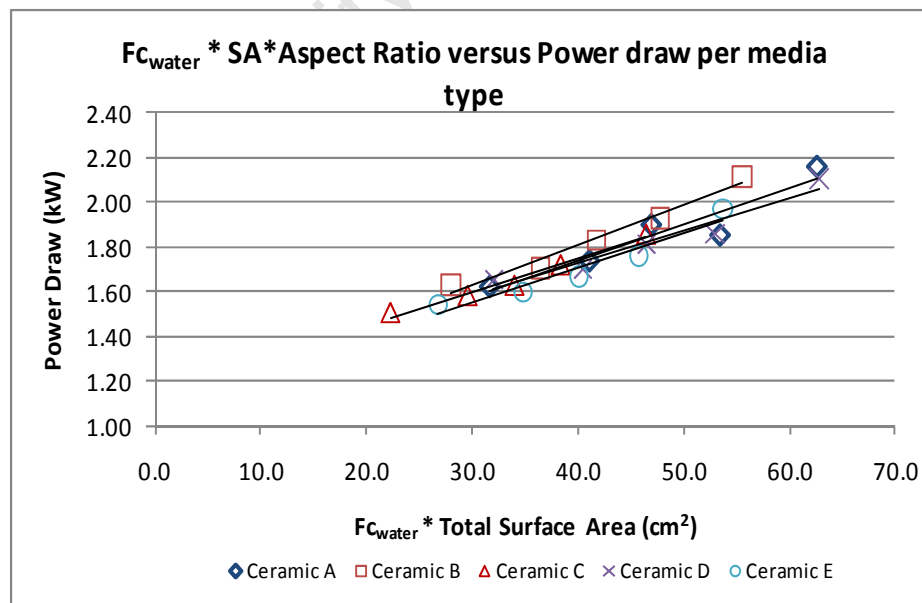


Figure 58: $F_{c_w} * SA * AR$ versus Power draw per media type in a water environment.

The individual regression for each media type indicates that all media types exhibited very similar behaviour using this methodology. It can be seen from Table 17 that the y-intercept of the regression line is similar to the NLP of the mill. During the experimental program the NLP was measured as 1.1kW for the M4 mill. This should be correct because if there is no media in the mill then only the power required to turn the shafts and discs in the water would be drawn.

Table 17: Results of linear regression per media type after incorporating the aspect ratio.

<u>Media Type</u>	<u>Slope (m)</u>	<u>Y-intercept (x=0)</u>	<u>R²</u>
Ceramic A	0.0163	1.0887	0.8971
Ceramic B	0.0178	1.096	0.9727
Ceramic C	0.0149	1.1539	0.9771
Ceramic D	0.0146	1.1403	0.9349
Ceramic E	0.156	1.0829	0.9276

Regression analysis of the combined data set, regardless of the media type indicated a very strong relationship between the product of the mentioned media properties and the mill power draw as indicated in Figure 59. The y-intercept of the regression line also correlated well with the No Load Power (NLP) of the mill.

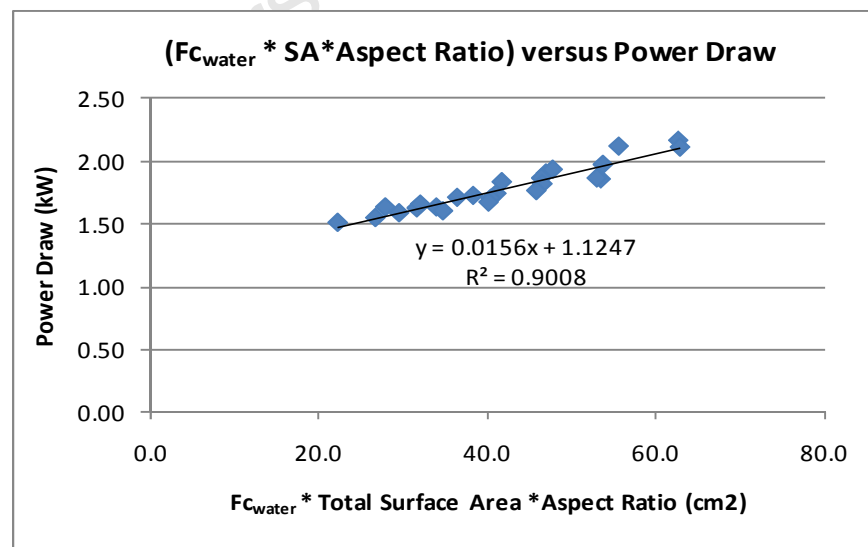


Figure 59: Relationship between the product of media friction coefficient, total surface area and media aspect ratio and mill power draw.

Based on the data of the tests, when operating in a simple system of water and ceramic grinding media only, the power draw of the mill can be predicted using equation 4.4.4 if the media properties are known:

$$\text{M4 Power Draw (kW)} = 0.0156 \times (Fc_w \times SA_T \times AR_{GM}) + NLP \dots [\text{Eq 4.4.4}]$$

Equation 4.4.4 was used to calculate the power draw for all the tests performed using the media friction coefficient, total media surface area and media aspect ratio. The same data used to develop the model was used to predict the power draw. Figure 60 presents the parity chart between the measured and experimental power draw for all the tests. The standard error determined by regression analysis of the predicted versus measured power draw was 5.67% with an R^2_{adj} of 0.95737 (Appendix K). This indicates that accurate predictions of the power draw on an M4 IsaMill™ can be done if the physical and surface properties of the ceramic media are known.

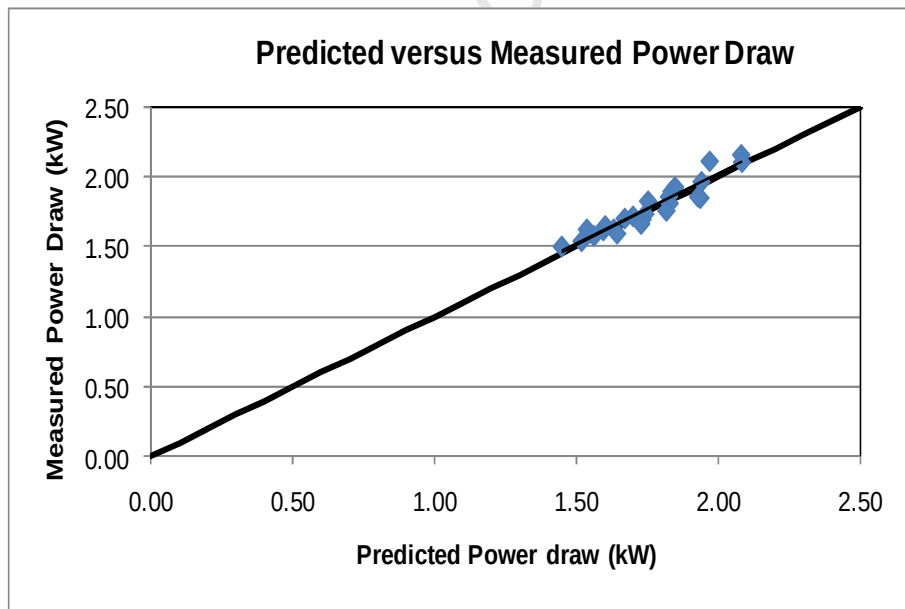


Figure 60: Predicted versus Measured Power draw using model development data

To validate the model a further five tests were performed using fixed mass media load of 4.5kg. Use of the fixed mass approach resulted in varying media loads tested due to the varying density of the media. The model was then used to predict the power draw and was compared to the measured power draw as shown in Figure 61. Good prediction of power draw with a standard error of 5.00% and an R^2_{adj} of 0.7494 was obtained through regression analysis (Appendix J) when comparing the predicted and measured power draws.

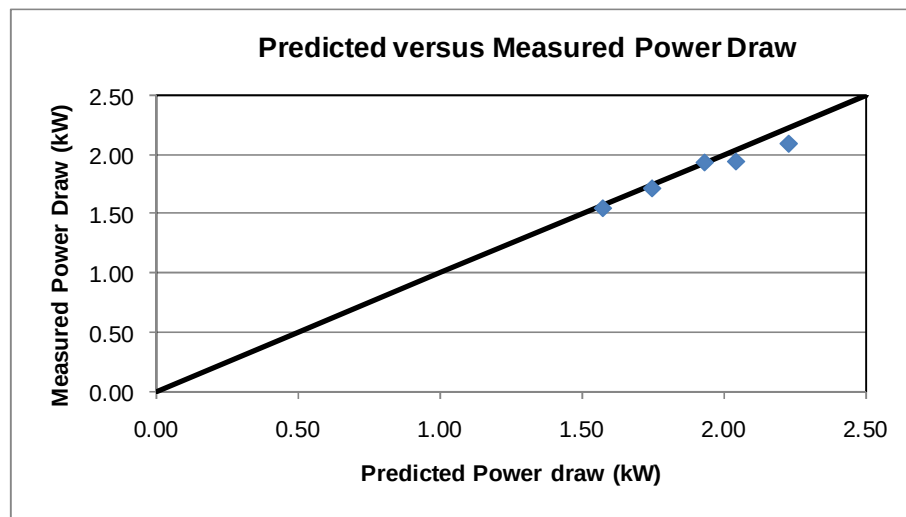


Figure 61: Predicted versus Measured Power draw for validation tests

It should however be noted that the model holds true only for the media fill ranges tested and that further test work is required to evaluate the system response at higher filling degrees. If one calculates the % mill filling, using 3.2 litres as the active volume of the mill chamber, the media loads during the experiments varied from 31.25% to 62.5%. Tests were limited to evaluate the impact of media properties on energy consumption without encountering complicating factors such as media compression. Power draw is expected to increase at extreme filling conditions when media compaction is experienced and therefore the relationship is not expected to remain linear for all media filling degrees.

Another area for future investigation is to determine if the same methodology is applicable to media with a wide size distribution as the ceramic media used during this investigation was similar not only in average size but all media types had very narrow size distributions. Applicability of the methodology when using different sizes including smaller ones (1-2mm) require validation. Applicability of this methodology needs to be tested at larger scales to determine scale up factors required, if any, on larger pilot scale and industrial size stirred mills.

The results indicate that the total surface area of the grinding media, the media friction coefficient and the aspect ratio of the media are the dominant factors influencing the mill power draw. Media density and mass do not play a significant role with respect to mill power draw. The effect of media size on energy consumption did not form part of the scope of this study, but has been included in the surface area calculation.

4.5 Evaluation of the relationship between the dominant media properties, defined in a water system, and the power draw of an M4 mill in a slurry system.

After defining the major media properties driving the energy consumption in a simple system, it is pertinent to test if the findings are valid for more complex systems, which includes ore particles. Ore particles are introduced with the water and based on the properties of the ore it can change the properties of the fluid (slurry) into the mill. As discussed in chapter 2, the IsaMill™ was originally designed for concentrate regrind applications, which in the PGM industry would treat a F_{80} of 30-40 μm ^{14, 15, and 36}. The technology was later evolved in the PGM industry to be used as mainstream grinding technology to increase PGM liberation for the particles reporting to flotation, where the feed streams has a F_{80} of 100-110 μm ³⁶.

Since a range of variables can affect the performance of the mill, the purpose of this work was to test the concept developed for the M4 IsaMill™ operated with water and media on a system that includes slurry. Figure 62 presents a schematic of the simplified factors contributing to energy consumption in horizontal stirred mills:

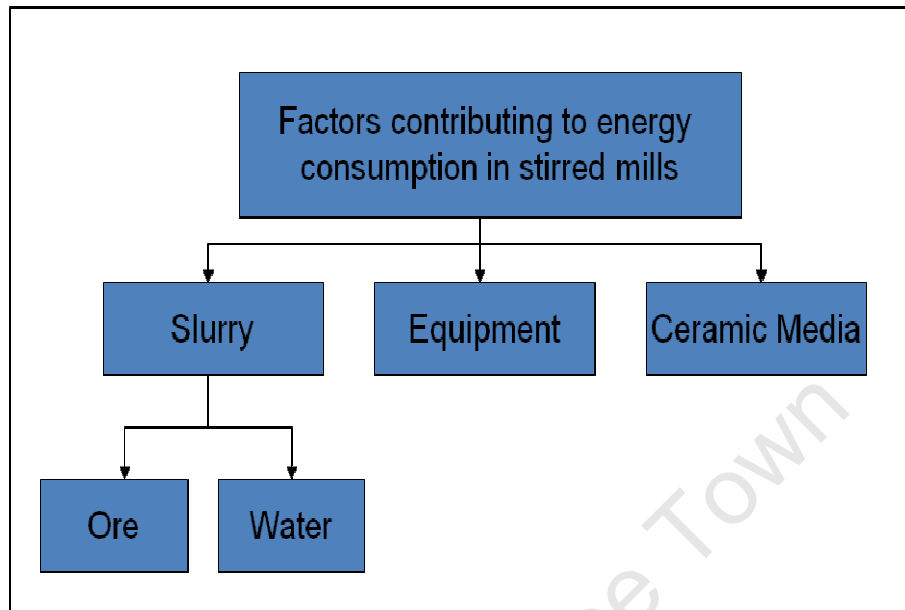


Figure 62: Classification of factors contributing to the energy consumption in IsaMills™

As observed during the water testing the equipment factor can be defined as the NLP of the IsaMill™. Results from the water tests indicated that the media friction coefficient ($F_{c_{GM}}$), the total surface area of media in the IsaMill™ (SA_T) and the aspect ratio of the media (AR_{GM}), are the media properties that affects energy consumption in the M4 IsaMill™.

If the ore particles had no influence on the media behaviour, then the friction coefficient of the media measured in water and in slurry environments should be the same. However the information presented in Table 12 showed that the friction coefficient of all the media types evaluated was significantly higher in a slurry environment compared to water only.

As explained in Chapter 3 and section 4.1, extreme care was taken in preparation of the ore feed to the mill to ensure that the slurry properties of the feed to the mill were consistent for all tests. The consistency of the feed particle size distribution is illustrated

in figure 63. The particle size distributions from several feed samples were similar, indicating that the procedure implemented for feed preparation resulted in a closely sized feed to the M4 IsaMill™.

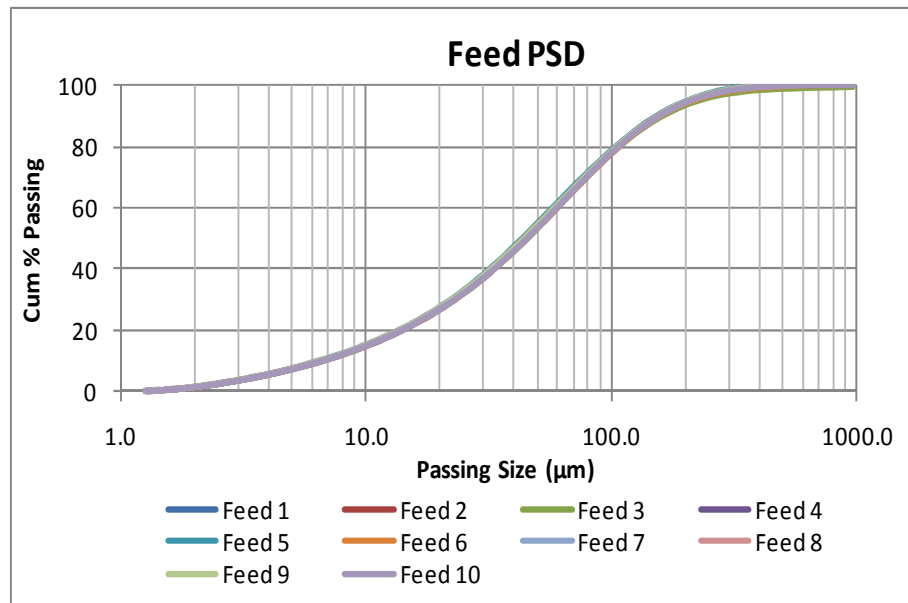


Figure 63: M4 Feed Particle Size Distributions

Analysis of the signature plots (Figure 64), generated by passing the material through the IsaMill™ 4 times, and indicates differences in the grinding behaviour and product sizes produced from the M4 IsaMill™. Varying grinding behaviour is expected during milling, if differences in media size and density between the different media tested exist. As discussed in section 2.4.1, the specific energy required for breakage is proportional to the Stress Intensity (SI) and the Stress Number (SN). SI and SN further influences both SN and SI, and therefore need to be incorporated in a complex system where breakage occurs. This will be discussed in detail later in this chapter.

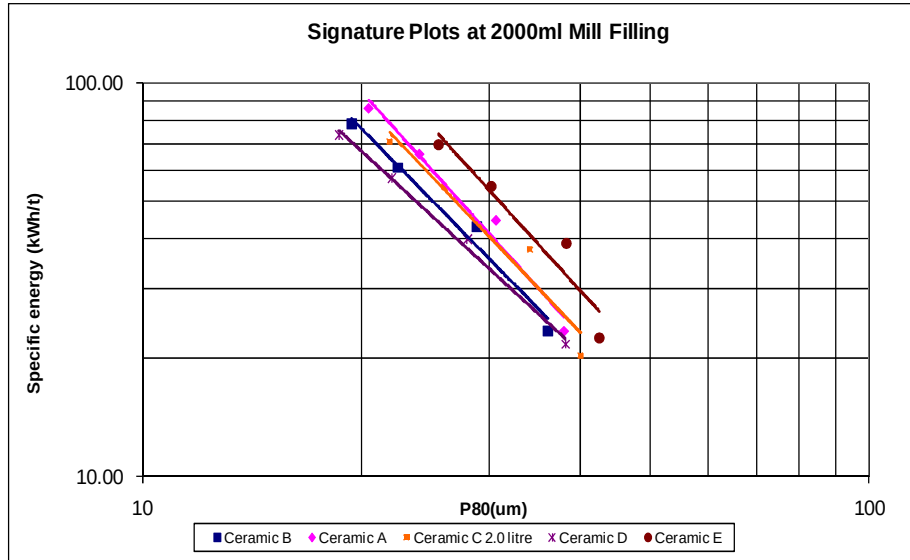


Figure 64: Signature plots for slurry testing at 2000ml fill.

As industrial mills are designed to only allow one pass of the material through the mill, it was initially decided to use the 1st pass only to evaluate the impact of media properties on energy consumption. However, the data from all 4 passes was used to obtain sufficient data for the model development. For the model development only the data produced during the slurry tests with a 2000ml media filling was considered. The test data from tests at lower media fillings, although limited was then subsequently used in an attempt to validate the model.

The first step in the model development was to adjust the water model by replacing the friction coefficient measured in water, with the coefficient measured in a slurry environment as shown in Equation 4.5.1.

$$M4 \text{ Power Draw (kW)} = 0.0156 \times (F_{c_s} \times SA_T \times AR_{GM}) + NLP \dots [Eq 4.5.1]$$

Where:

F_{c_s} = Friction Coefficient Measured in UG2 slurry

SA_T = Total Surface Area of the Grinding Media

AR_{GM} = Aspect Ratio of the Grinding Media

NLP = Measured No Load Power of the Equipment.

To be able to apply the equation to all 4 passes of the mill, the surface area of the media in the mill during each pass had to be considered. The experimental procedure did not allow for media wear determination between each pass in the slurry testing procedure. Based on the low wear rates of the media during the 2hr slurry trials shown in table 18, the media surface area and aspect ratio was assumed to be constant for the duration of a slurry test.

Table 18: Wear rates of media Recorded during Slurry Tests.

Ceramic Media Type	Mill Volumetric Filling (ml)	% Wear
Ceramic A	2000	0.83
Ceramic B	2000	0.99
Ceramic C	2000	0.24
Ceramic D	2000	0.84
Ceramic E	2000	0.13

Prediction of power using equation 4.5.1 against the measured power for the slurry tests is presented in figure 65. Linear regression indicated unadjusted correlation coefficient (R^2_{adj}) of 0.9423 and a standard error of 19.61% when comparing the predicted and measured power draw data. Fitting a new constant k , resulted in no improvement in the prediction as a new fitted k value was not significantly different from that of the water model constant.

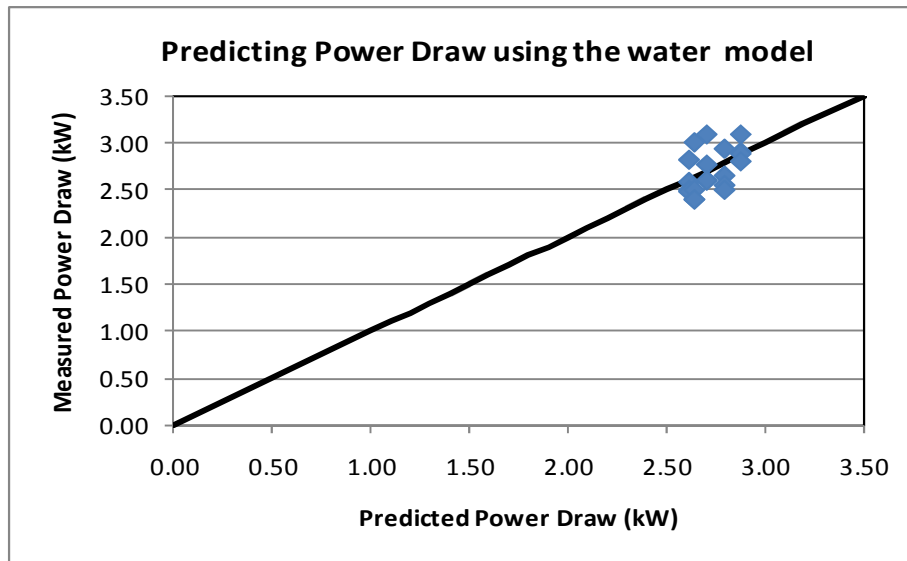


Figure 65: Predicted versus actual Power draw using the modified model.

Due to the poor prediction of the modified model, a closer analysis of the prediction error as indicated in figure 66 revealed that the error was not a standard error for all the tests. Although the average error was low, the error varied between 0.1 and 15% for the various passes through the mill. This is an indication that other factors not considered in the model are contributing to the power draw.

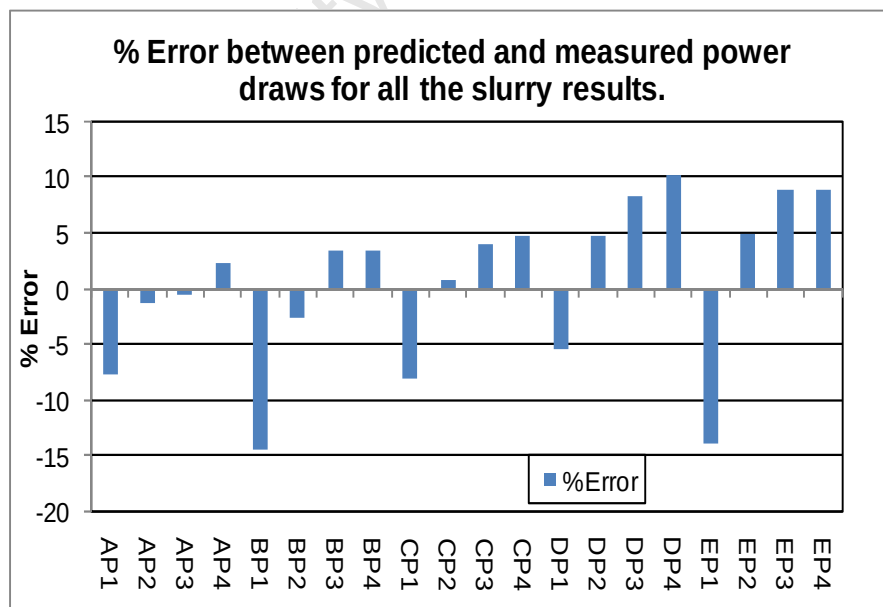


Figure 66: % Error in power prediction for every slurry test pass.

Chaponda (2011)¹³ indicated clear differences in the grinding efficiency at different feed size and the energy requirements to reach a specific grind with the same size media. As explained in chapter 2, Kwade (2002) indicated that the specific energy required for breakage is proportional to the product of the stress intensity and stress number as shown in equation 2.4.2. In equation 2.4.3 the stress number is defined as a function of the number of media contacts and the number of particles in the feed to the mill. For a given feed solids flow, the smaller the P_{80} of the feed the more solids particles will be in the feed. Therefore the number of feed particles can be considered to be inversely proportional to the F_{80} size of the feed as shown in equation 4.5.2.

$$N_p \propto \frac{1}{F_{80}} \dots [Eq 4.5.2]$$

Where:

N_p = Number of Particles in the Feed

F_{80} = Size at which 80% of the particles are smaller than the quoted size.

Based on this assumption the stress number will be proportional to the number of media particles times the feed P_{80} .

$$SN_a \propto \frac{N}{\frac{1}{F_{80}}} \dots [Eq 4.5.3]$$

Where:

SN_a = Stress Number

F_{80} = Size at which 80% of the particles are smaller than the quoted size.

N = Number of grinding media beads in the mill.

Figure 67 shows the relationship between the product of the number of media particles and the P_{80} of the feed and the power draw of the mill. The results indicate a strong relationship between the number of media particles and the size of the feed fed to the mill. For a consistent media load the larger the feed particle size the lower the available specific energy for breakage.

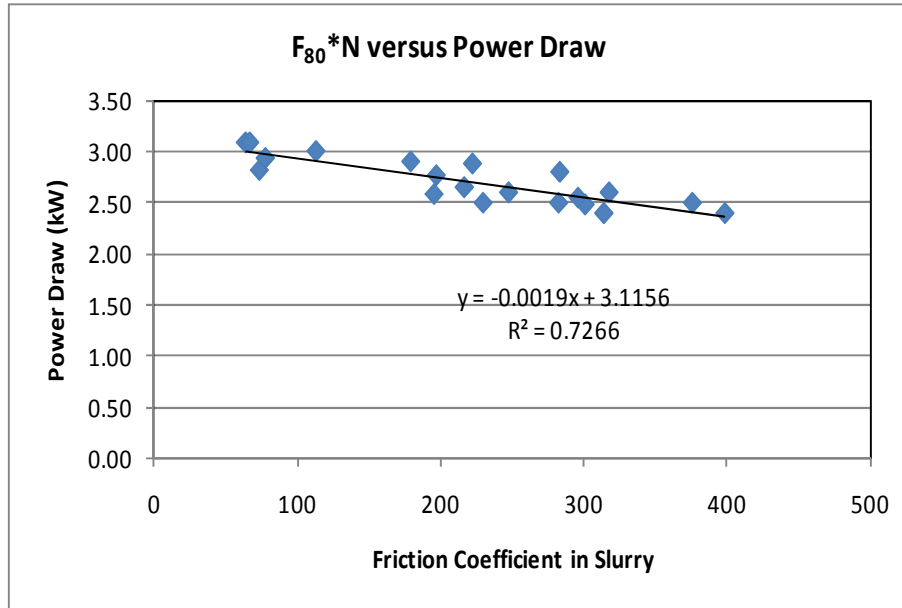


Figure 67: Relationship between the number of media beads and the feed particles size and power draw for 2l slurry tests.

Since the media size and density vary from media type to media type, and the speed of the mill was kept constant, the stress intensity formula as presented earlier in equation 2.4.1 can be simplified to the formula presented in equation 4.5.4.

$$SI \propto SI_a = \frac{d^3 \times \rho_{GM}}{1000} \dots \dots \dots [Eq 4.5.4]$$

Where:

- SI = Stress Intensity of the grinding media.
- SI_a = Adjusted Stress Intensity for power model
- d = Grinding media Diameter
- ρ_{GM} = Density of the Grinding Media

This indicates that a portion of the power drawn by the mill will be used for breakage and a portion for keeping the charge in motion. Power draw in a slurry-media system can therefore be presented by equation 4.5.5. Machine factor have already been established in the water tests as the NLP of the IsaMill™.

Power Draw (kW)

$$= \text{Breakage Power draw} + \text{Charge Power Draw} \\ + \text{Machine Power Draw} \dots \dots \dots [\text{Eq 4.5.5}]$$

In chapter 2 it was indicated that the change to ceramic media from natural media such as silica sand, resulted in an improvement in the energy efficiency of IsaMills™. A sample of silica sand was then used to determine the friction coefficient of silica sand and the results are presented in Table 19. It can be seen that silica sand exhibited much higher friction coefficients than any of the ceramic media evaluated (Table 12).

Table 19: Friction Coefficients of Silica Sand.

Media Type	FC Dry	Error Dry	FC Water	Error Water	FC Slurry	Error Slurry
Silica Sand	0.14	0.006	0.22	0.010	0.49	0.008

Friction coefficient measurements in the slurry were done on very fine UG2 slurry to with a size of 9µm to avoid any bias that bigger particles may introduce. The effect of particle size on the friction coefficient was not evaluated and no body of work was found on this topic. Therefore ore particle size and its effect on the friction coefficient was not considered for this work and included in the scope.

Ore Particles size can impact the process in 2 main areas namely energy consumption and slurry viscosity. Even though the slurry density was kept constant for all tests, the series of grinding steps for each media type test, the feed size was continually changed introducing a 2nd variable during slurry testing.

Changes in feed particle size distribution would alter the viscosity of the slurry entering the mill. It is a well established fact that slurry viscosity increase with an increase in the fines of grind of the solids in the stream. Slurry testing involves multiple passes of the same slurry material and therefore with every pass the feed will become finer and

viscosity will increase. Increased viscosity will in turn result in increased resistance to media motion in the mill, leading to higher energy consumption.

At coarse feed grinds the angular particles will increase power draw due to the high friction coefficient of the coarse particles. With each pass particles will become more rounded and require less power for movement as rounded particles will be easier to move around in the mill. Therefore feed particle size can play a significant role in the energy consumption of the mill. Impact of grind and viscosity on power draw did not form part of this study and discussion given above is based on basic principles of solid fluid operation and on the author's experience from operating M4 IsaMills™.

The power draw formula for a mill on slurry was therefore formulated as shown in equation 4.5.6. k_b and k_c can be defined as the breakage contribution constant and charge contribution constant respectively.

$$\text{Power Draw (kW)} = k(SN_a * SI_a)^c \times (Fc_s \times SA_T \times AR_{GM}) + NLP.. [Eq 4.5.6]$$

k and C was fitted to the dataset and the best fit was obtained at values of 0.0264 and -0.160 for k and C , respectively. Regression analysis comparing the predicted versus experimental data indicated an R^2_{adj} of 0.945 and standard error of 12.00% (Appendix K).

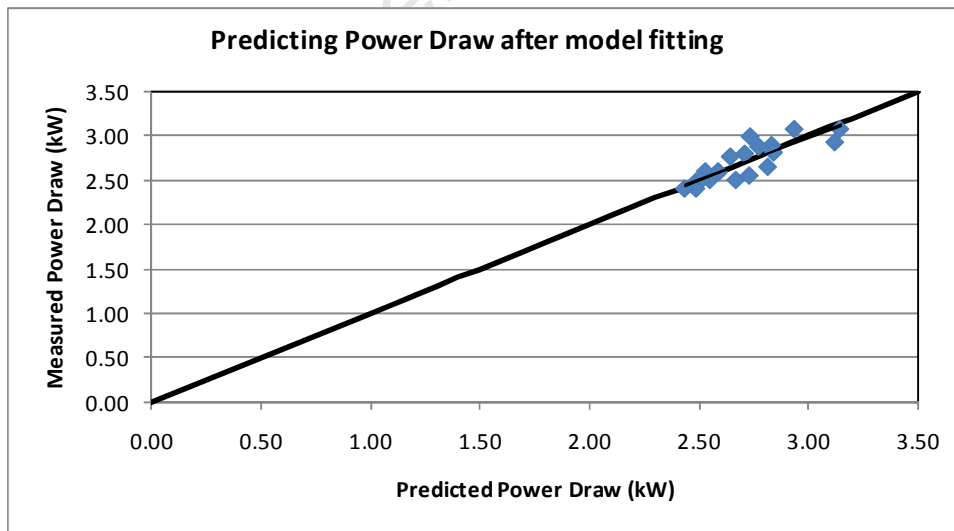


Figure 68: Predicted versus actual Power draw using the slurry power model

The results indicate fairly good prediction of power draw as shown in figure 68 and 69. The prediction can be significantly improved if the effect of the grind size on mineral slurry rheology and friction in the system are incorporated. Errors could have also been introduced by assuming the media surface area constant due to the low wear over the 2 hour test.

Media size was not measured after tests to ascertain whether the average media size was reduced. Reduction in media size will have a major impact on the total media surface area of the mill and subsequently power draw.

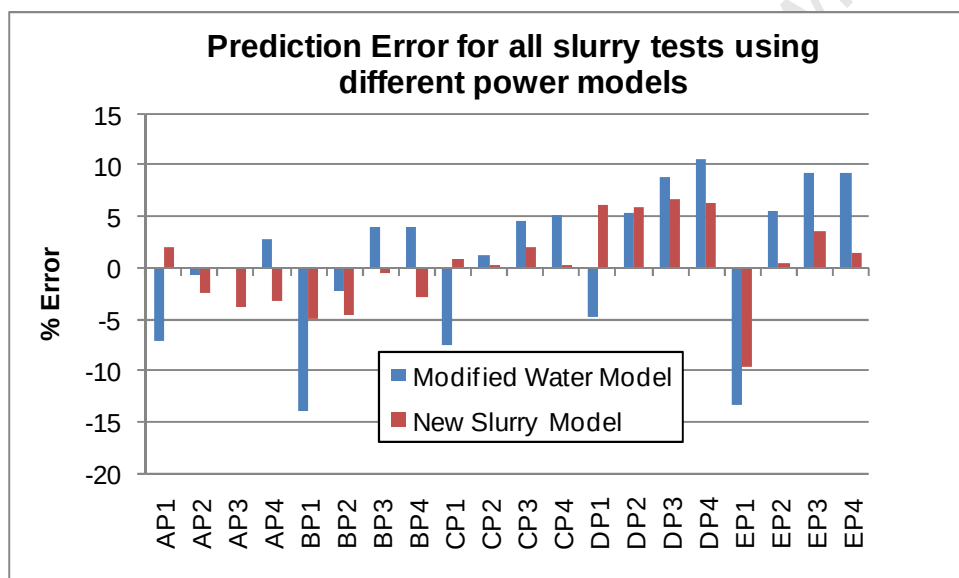


Figure 69: % Error in Power Draw prediction using both power draw models.

Slurry tests were conducted at 1000ml and 1500ml fills and the proposed model used to predict the IsaMill™ power draw. At the lower load, the stress intensity would remain constant but the SN would decrease due to the lower number of available media contacts. This would reduce the grinding efficiency of the mill at the same operating conditions as for the 2000ml mill filling as presented in figure 70.

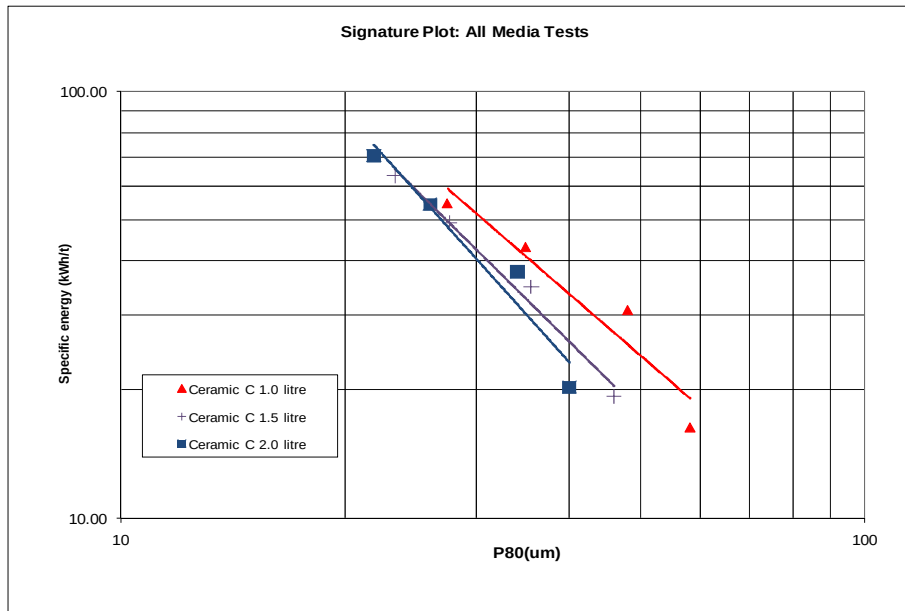


Figure 70: Signature plots for Ceramic C at various Mill Fills.

Figure 70 indicates that there was very little difference in the grinding efficiency at the fine end of the spectrum when comparing the 2000ml and the 1500ml mill fill tests. At the coarser end, there was a pronounced difference in grind noticed. This could be due to inefficient breakage of large particles due to the lower number of media contacts at the lower filling degree. At the lowest mill filling degree of 1000ml one the worst grinding efficiency was indicated with much higher specific energy required to reach the same grinds as for the higher mill filling tests.

At the lowest mill filling, the effect of ore particles retained in the mill due to inefficient breakage is more pronounced. The retention of coarse angular ore particles will result in higher energy consumption due to the coarse nature of the particles and a lower ratio of media beads to ore particles in the mill.

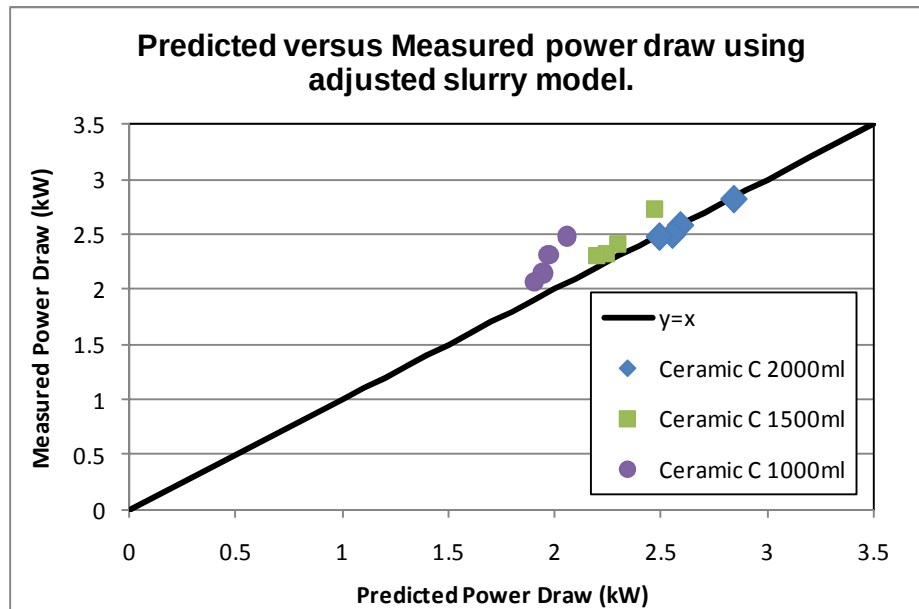


Figure 71: Predicted versus Measured Power draw for Media C at different mill fills.

Figure 71 shows the comparison between the predicted and measured values at 3 different mill fills for Ceramic C using the slurry power draw model. It must be noted that the 2000ml data is the same as that was used for model development. It was included in the graph for reference purposes to better illustrate the trend with different mill fills. It can be seen from the results that the model proposed under predicted power draw at lower loads. Further slurry tests are recommended at different loads with different media types to further refine the proposed model.

What the testwork has indicated was the impact of ore particle properties and the proportions of the ore and media particles in the mill and its influence on mill power draw. At the lower media fills the ratio of media beads to ore particles are also higher. Ore particles should therefore contribute more to the power draw component of the mill charge. It was seen from friction coefficient tests on silica sand that angular silica particles has much higher friction coefficients than the ceramic media.

Therefore due to the larger proportion of angular ore particles in the charge, the friction coefficient measured in 9µm UG2 slurry may be understated. The impact of inadequate top size breakage and coarse material in the mill and its impact on energy efficiency were discussed by Young et al. in 2011⁴³, indicating that with inadequate breakage coarse material will be retained in the mill. Comparable to the critical size in SAG mills the coarse particles will reduce grinding efficiency and increase power consumption⁴³.

Figure 72 shows that in water the slope of the regression lines for power at different fills on silica sand and ceramic media C was different. From this it can be postulated that the charge power draw will be affected at different ore particle to media particle ratios and that it would not only affect the breakage energy as initially indicated. Further work is recommended to evaluate the influence of media particle ratios to provide sufficient data for model testing and validation.

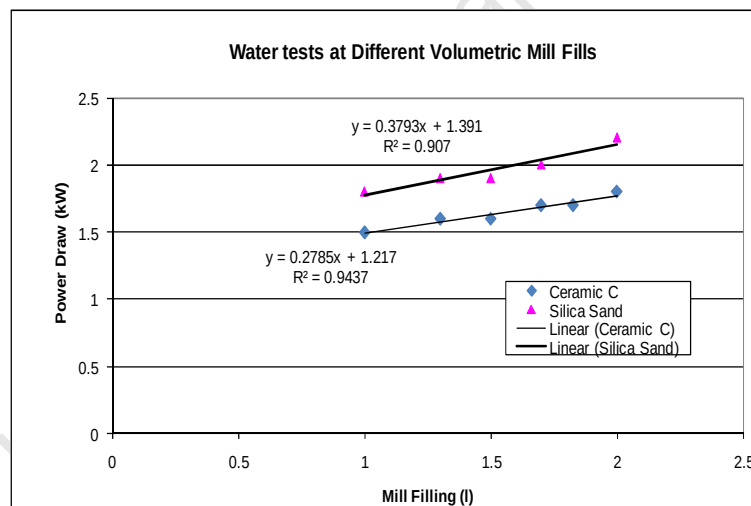


Figure 72: Signature plots for Ceramic C at various Mill Fills

In summary, power draw when operating on slurry is a combination of the energy required for breakage, the energy required to keep the charge in motion and the energy required by the equipment. The suggested power draw model is capable of predicting mill power for all media types tested at the same volumetric filling if the media properties

in slurry and the mill feed F_{80} is known. Further validation of the model is required particularly on material different to those used to develop the models.

Therefore to further refine the slurry power draw model to allow for wide application particularly at different mill fillings, future studies should concentrate on:

1. Defining the impact of ore particle size on friction coefficient, as the friction coefficient component is a major contributor to power draw.
2. The impact of media to solids ratios and its impact on breakage and charge power draw components of the model.
3. Investigate the impact of slurry rheology on power draw.
4. Once the additional factors (1-3 above) have been established and the power draw model has been refined, scaling up of the model to larger units can be investigated

5. CONCLUSIONS AND RECOMMENDATIONS.

The work presented in the preceding chapters provided details of the project with regards to the following hypothesis tested that:

1. Surface and Physical properties of the grinding media such as friction coefficient, size and shape, that influences the surface area of the grinding media, are the key drivers of energy consumption in a laboratory horizontal stirred mill.
2. The contribution of media density to the energy consumption of a laboratory horizontal stirred mill is minimal or even negligible.

5.1 Conclusions.

The main conclusions from this project can be summarised as follows:

5.1.1 Ceramic Media – Water System.

1. In a ceramic media – water system, it was proved that the contribution of media density to the energy consumption of a horizontal stirred mill was minimal. This is due to the differences in media friction coefficient of the media evaluated that plays a much larger role in energy consumption than media density.
2. The density media contribution constant (C_p) proposed by Farber (2010)¹⁷ is not constant for all media types. Different media exhibited different friction coefficient which in turn will affect the contribution of density to energy consumption.
3. In a ceramic media water system, the major contributors to energy consumption can be separated in to media load and machine contributors.
4. The media load contribution was found to be proportional to the total surface area of the media load (SA_T), the friction coefficient of the media measured in water (F_{c_w}), and the aspect ration of the media (AR_{GM}).

$$\text{Media charge Contribution} \propto SA_T \times F_{c_w} \times AR_{GM} \dots \dots 5.1.1$$

5. The machine contribution was defined as the NLP of the mil. Machine energy from basic engineering theory for IsaMillsTM is the power required to turn the shaft and discs at the required speed. The linear regression line fitted when investigating the media load contribution to the power draw of the mill also

intersected with the power draw at 1.12 kW at zero media charge contribution. This was similar to the measured no load power of the mill.

6. A water power draw model (Equation 4.4.4) was developed which gave fairly good power draw for a laboratory M4 IsaMill™ to within 2% of the measured power draw.

5.1.2. Ceramic Media Slurry System.

1. Application of the water model to the ceramic media slurry system was done, by replacing the friction coefficient measured in water, with the friction coefficient measured in slurry. Prediction of power was poor and further model development was required to accurately predict power draw on the M4 IsaMill™. This was due to the fact that the water model did not incorporate energy required for breakage of ore particles in the mill or the impact of particle size in the mill feed in the model.
2. A new power draw model was developed during the study which incorporates the required breakage energy. This was accomplished by incorporating the Stress Intensity (SI) and Stress Number (SN) as an additional factor in power draw in a slurry-media system. The power draw model was modified to allow for 3 contributors to energy consumption i.e. breakage energy factor, media load factor and machine factor.
3. Constants to the power draw model was fitted to the data and the modified model was used to predict the M4 power draw with relatively good accuracy.
4. Validation of the model was attempted using slurry test data conducted at 25% and 50% less media volumetric filling. At 25% less volume fill the prediction was still fairly accurate but the model was found to be inaccurate at 50% lower volumetric fill.
 - o It is postulated that the inaccuracy was because the model do not completely incorporate the impact of varying particle to media ratios. Since

no viscosity measurements were made during the experimental program, hydrodynamic effects on power draw was not incorporated into the model.

- o Underprediction of power draw can also be potentially attributed to undergrinding of the ore particles, resulting in a larger proportion of coarse particles being retained in the mill.
5. The model has the potential to decouple the effect of the various factors contributing to energy consumption in horizontal stirred mills. Further refinement of the model is however required as suggested in chapter 5.2 in future work.

5.2 Recommendations.

There appear to be some promise for use of the methodology developed to predict the power draw of horizontal stirred mills. However further validation and development of both the methodology and proposed models is required. For future testwork the following is recommended:

5.2.1 Water Model.

1. Evaluate the applicability of the water model to media loads composed of smaller beads.
2. Evaluate the applicability of the water model on “seasoned” media loads. This can be accomplished by conducting water tests on media loads with different proportions of different media sizes.
3. To test scale up of the model to larger mills. If scale up is reasonable, the model can potentially be incorporated in the plant operating methodology during start up to ensure the optimum media load fill in the mill.

5.2.2 Slurry Model

1. Investigate the influence of ore particle sizes on the friction coefficient of ceramic media measured in slurry.
2. Investigate the impact of media to ore ratio in the mill and its impact on energy consumption.
3. Investigate the impact of slurry rheology on energy consumption.

4. Once the abovementioned 3 points has be refined and included where applicable into the model, scale up to larger mills can be investigated.

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Appendix A: Bulk Blending and Odds and Evens Blending Procedures

Bulk Blending Procedure

Purpose

To perform initial bulk blending on the finely crushed or milled ore.

Benefit

It is very important to blend a sample prior to any subsample being extracted from the bulk. A non-representative feed sample will lead to non-representative results.

Prerequisites

Any sample being presented to this method must have been representatively sampled (BT 2), jaw-crushed (BT 3), and cone-crushed (BT 4).

Equipment

20 Clean Rubbish Bins with lids	Waterproof marking pen
Aluminium Sample Pan	Steel-capped boot
Leather gloves	Sample Labels
Goggles	String
Respirator	
Overalls	

Safety

This process generates some dust, and uses rubbish bins to contain the sample at whatever stage. Accordingly use eye protection, steel-capped boots, respirator and leather gloves.

Method

The description given is for 4 bins of primary sample. This amount may vary. If this amount exceeds 4, use steps 1-16 to reduce the bulk primary sample to 4 bins. Then store the rest of the primary sample awaiting further decisions.

1. The rubbish bins containing the various products from milling process are for explanation purposes stored in 4 90l dustbins.
2. Transport these 4 bins outside the laboratory using a two-wheeled cart.
3. Arrange the bins in a row on the cement surface outside the laboratory and label it from 1 to 4.
4. Place 4 clean empty rubbish bins in a row parallel to the bins labelled 1 to 4 in step 3. Label these bins 5-8 individually.
5. You now have a total of 8 bins. These are in two rows of four. The rows are parallel to each other. Bins 1-4 are full of sample. Bins 5-8 are empty.
6. Pick up the aluminium sample pan and approach bin # 1.
7. Fill the aluminium sample pan with and carry it to bin # 5. Drop the material from the pan into bin # 5.
8. Return to bin # 1 and again fill the pan with material. Carry it to bin # 6 and drop the material inside this bin.
9. Return to bin # 1 and fill the pan with material. Carry the material to bin # 7. Drop the material into bin # 7.
10. Return to bin # 1 and again fill the pan with - 2 mm material. Carry it to bin # 8. Drop the material into bin # 8.
11. Repeat steps 6-10 until bin # 1 is empty.
12. Repeat steps 6-10 using the material in bin # 2, until bin # 2 is empty.
13. Repeat steps 6-10 using the material in bin # 3, until bin # 3 is empty.
14. Repeat steps 6-10 using the material in bin # 4, until bin # 4 is empty.
15. You now have bins 5-8 full with sample, and bins 1-4 empty.
16. Make sure that all your dust bins are clearly identified with their correct labels.
17. The dust bins you are using are now ready for odds and evens blending.

Note

The critical features of this procedure are shown pictorially for a total of 4 bins of primary sample only.

Preparation Stage

Milled Material



FULL



EMPTY

Milled Material



FULL



EMPTY

Milled Material



FULL



EMPTY

Milled Material

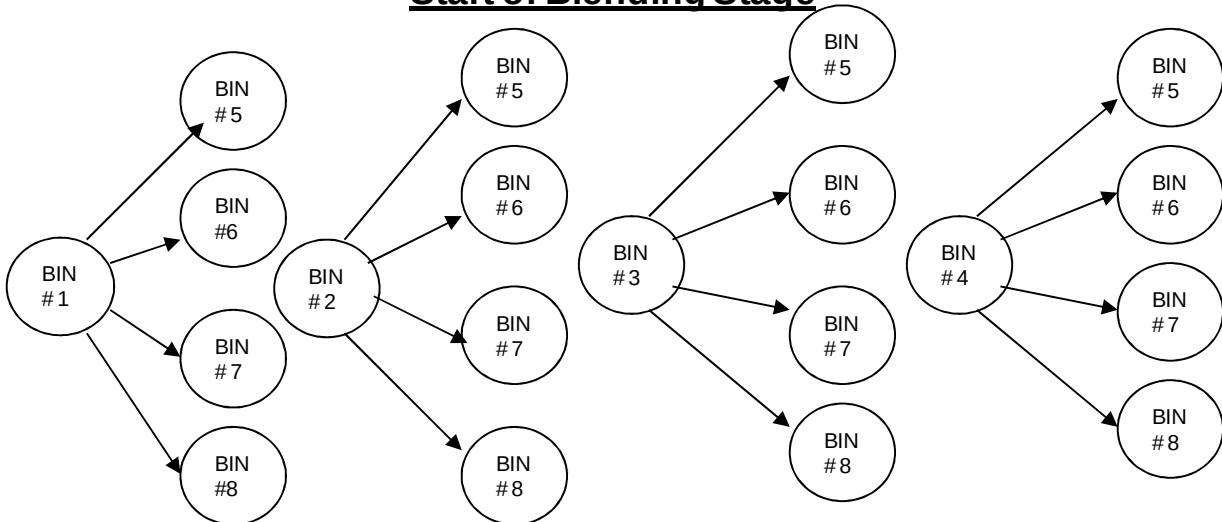


FULL



EMPTY

Start of Blending Stage



Pour one pan of sample from BIN #1, BIN #2, BIN #3, BIN #4, at a time UNTIL EMPTY

Preparation Stage

Milled Material



EMPTY



FULL

Milled Material



EMPTY



FULL

Milled Material



EMPTY



FULL

Milled Material



EMPTY



FULL

University of Cape Town

Odds-and-Evens Blending Procedure

Purpose

To obtain a well-blended sample such that the group and segregation errors are minimised. The Merensky ores and flotation products have several different PGE-bearing minerals of different sizes and densities, and are prone to this error.

Benefit

Samples blended in this way obtain precise agreement between replicate fire assays, i.e. the scatter is reduced.

Prerequisite

This method is essential for correct blending. Do not use any other method.

Equipment List

10-way spinning riffler

6 sample buckets (20-litre capacity)

Identified sample

Respirator

Goggles

Ear Protection

Overalls

Compressed air

Dust Extraction Fan

Leather Gloves

Safety

The spinning riffler is classified as moving machinery; therefore no work may be done on the moving parts whilst the equipment is being operated. The only permitted activity is the handling of the sample into the distribution cone above the rotating cascade. Wear all specified PPE.

Method

1. Put on your goggles, ear protection and respirator. Make sure that the extraction fan is switched on.
2. Clean the spinning riffler using compressed air. The subsample pots must be immaculately clean; otherwise the sample to be handled will become contaminated.
3. Clean the 4 sample buckets using compressed air. These buckets must be immaculately clean; otherwise the sample to be handled will become contaminated.
4. Mark one of the six sample buckets "first pass odd".
5. Mark one of the remaining five sample buckets "first pass even".
6. Mark one of the remaining four sample buckets "second pass odd - # 1".
7. Mark one of the remaining three sample buckets "second pass odd - # 2".
8. Mark one of the remaining two sample buckets "second pass even - # 1".
9. Mark the remaining sample bucket "second pass even - # 2".
10. At this stage you should now have the following sample buckets with labels:

Bucket #	Label
1	First Pass Odd
2	First Pass Even
3	Second Pass - Odd # 1
4	Second Pass - Odd # 2
5	Second Pass - Even # 1
6	Second Pass - Even # 2

11. The rotating cascade of the spinning riffler has 10 subsample pots. Number these as 1, 2, 3, 4, 5, 6, 7, 8, 9, and 10.
12. Odd numbers are 1, 3, 5, 7, and 9.
13. Even numbers are 2, 4, 6, 8, and 10.
14. It is now evident that there are 5 odd-numbered subsample pots, and there are 5 even-numbered subsample pots.
15. Load the 10 subsample pots into the spinning riffler cascade as follows: from the red dot marked 1 in the direction of the arrow - 1, 2, 3, 4, 5, 6, 7, 8, 9 and 10.

16. Place the sample to be blended near to the spinning riffler to be used. This is called the "original sample".
17. Load the original sample into the spinning riffler feed funnel. Do not overfill the funnel.
18. Place the sample label on to the centre of the spinning riffler cascade, so that the sample identity is clear during the blending process. If necessary use some adhesive tape to secure this arrangement.
19. Check that no person is close to the spinning riffler machinery; the area must be clear of unauthorised persons before start-up.
20. Start the spinning riffler by switching on the motor that turns the cascade, and then the vibrator. This sequence ensures that the sample will not reach the subsample pots before the cascade is turning.
21. The sample will run through the feed funnel and become subdivided proportionally into the 10 subsample pots. When this is complete, switch the spinning riffler motor and vibrator off.
22. Wait for the spinning riffler motion to stop naturally.
23. Wait until the spinning riffler cascade has completely stopped movement.
24. Rotate the spinning riffler cascade manually until subsample pot labelled "1" is in front of you.
25. Remove the subsample pot numbered "1" and empty its contents into the sample bucket labelled "first pass odd".
26. Rotate the spinning riffler cascade manually until the subsample pot labelled "2" is in front of you.
27. Remove the subsample pot labelled "2" and empty its contents into the sample bucket labelled "first pass even".
28. Repeat steps 24-25 for the odd subsample pots 3, 5, 7 and 9.
29. Repeat steps 26-27 for the even subsample pots 4, 6, 8, and 10.
30. At this stage half of the original sample is present in the sample bucket labelled "first pass odd", and the other half, in the sample bucket labelled "first pass even".
31. Repeat step 15. Then proceed to step 32.

32. Place the bucket labelled "second pass odd - # 1" and the bucket labelled "second pass even - # 1" in the area of the spinning riffler.
33. Repeat steps 16-23 using the sample contained in the bucket labelled "first pass odd".
34. Rotate the spinning riffler cascade manually until the subsample pot labelled "1" is in front of you.
35. Remove the subsample pot numbered "1" and empty its contents into the sample bucket labelled "second pass odd - #1".
36. Rotate the spinning riffler cascade manually until the subsample pot labelled "2" is in front of you.
37. Remove the subsample pot labelled "2" and empty its contents into the sample bucket labelled "second pass even - #1".
38. Repeat steps 34-35 for the odd subsamples pot 3, 5, 7 and 9.
39. Repeat steps 36-37 for the even subsamples pots 4, 6, 8 and 10.
40. Place the bucket labelled "second pass odd - # 2" and the bucket labelled "second pass even - # 2" in the area of the spinning riffler.
41. Repeat steps 16-23 using the sample contained in the bucket labelled "first pass even".
42. Rotate the spinning riffler cascade manually until the subsample pot labelled "1" is in front of you.
43. Remove the subsample pot numbered "1" and empty its contents into the sample bucket labelled "second pass odd - #2".
44. Rotate the spinning riffler cascade manually until the subsample pot labelled "2" is in front of you.
45. Remove the subsample pot labelled "2" and empty its contents into the sample bucket labelled "second pass even - #2".
46. Repeat steps 42-43 for the odd subsamples pots 3, 5, 7 and 9.
47. Repeat steps 44-45 for the even subsamples pots 4, 6, 8 and 10.

48. You now have the following buckets containing their described contents:

Bucket Label	Contents
Original Sample	Empty
First Pass Odd	Empty
First Pass Even	Empty
Second Pass Odd # 1	One Quarter of original sample
Second Pass Even # 1	One Quarter of original sample
Second Pass Odd # 2	One Quarter of original sample
Second Pass Even # 2	One Quarter of original sample

49. Take the sample bucket labelled "second pass odd # 1" and add its contents to the bucket labelled "original sample".

50. Take the sample bucket labelled "second pass even # 2" and add its contents to the sample bucket labelled "original sample".

51. Take the sample bucket labelled "second pass even # 1" and add its contents to the sample bucket labelled "original sample".

52. Take the sample bucket labelled "second pass odd # 2" and add its contents to the sample bucket labelled "original sample".

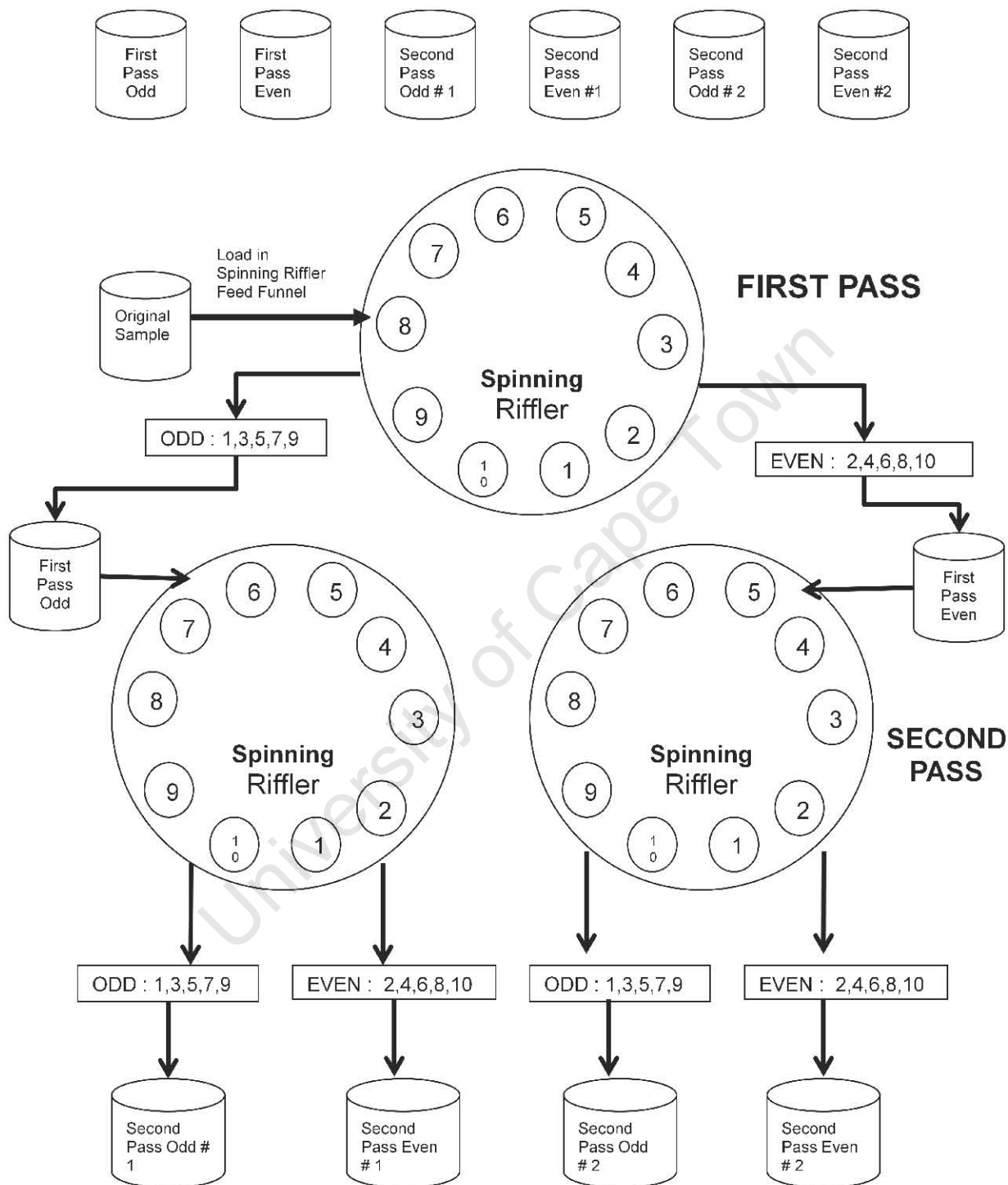
53. There is now only one sample bucket containing sample. This is labelled "original sample".

54. This is the blended sample.

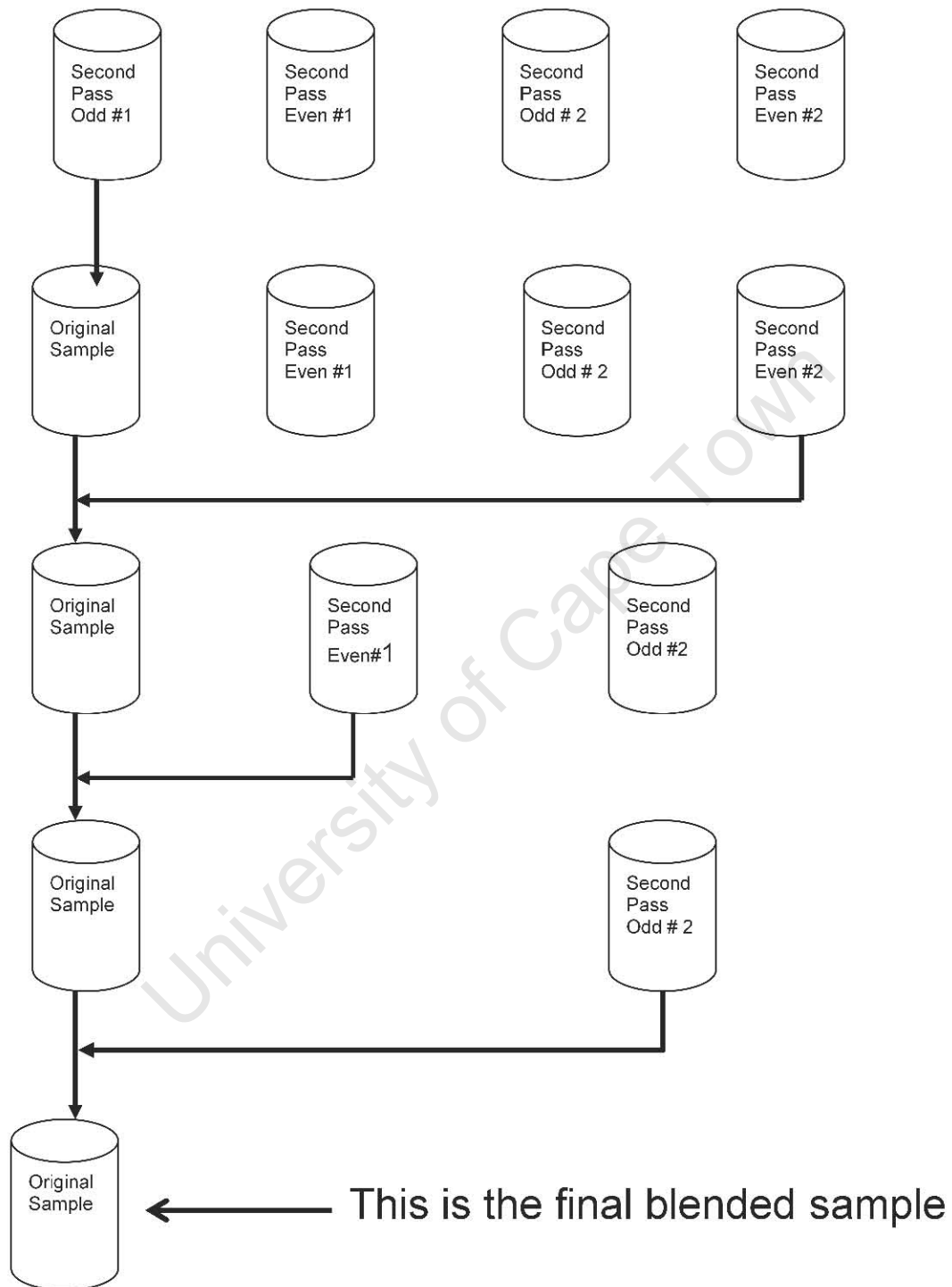
55. Before using or despatching this sample for any purpose, it must proceed to BT 7 for subsampling.

Note

The critical features of this procedure are shown pictorially in the diagrams below.



Final Blending Stage



Appendix B: DML Historical Ceramic Media Test Data

DML Database Data:

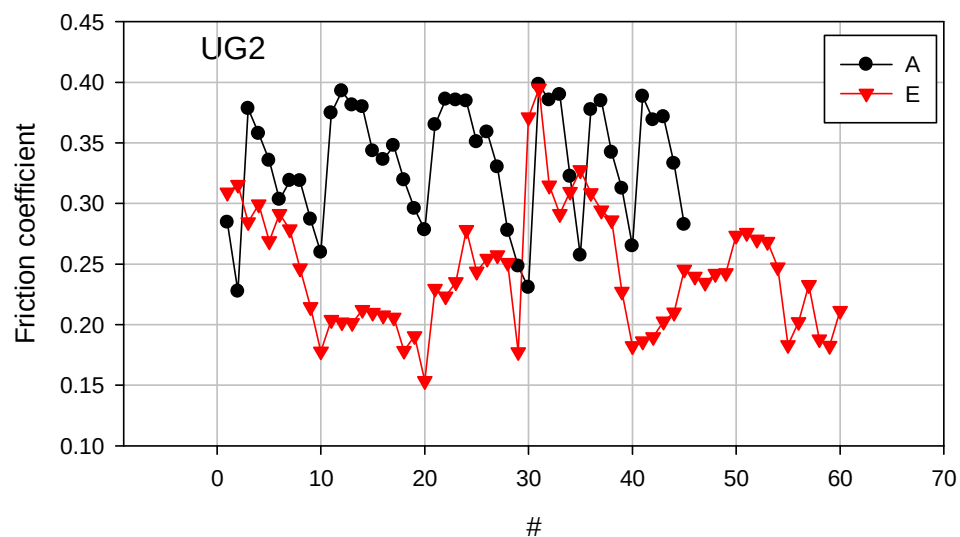
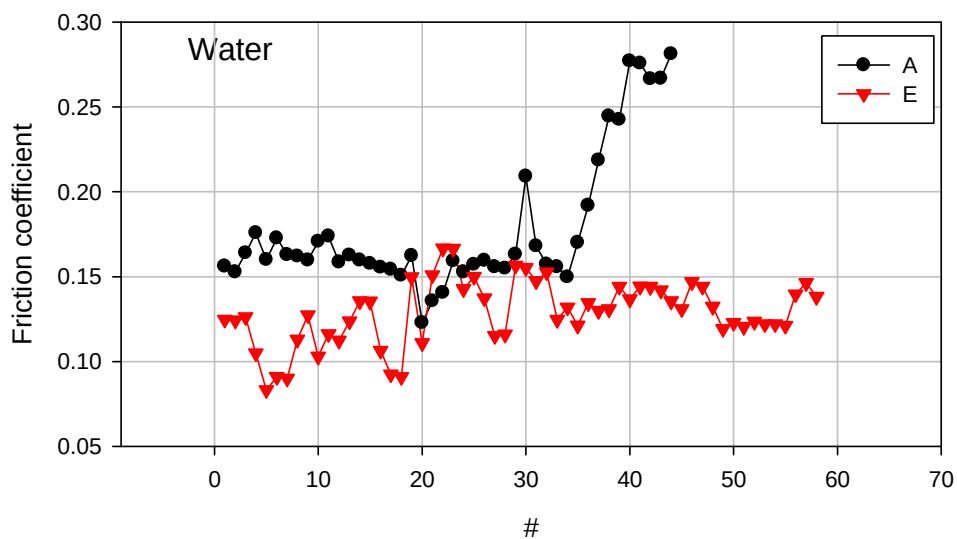
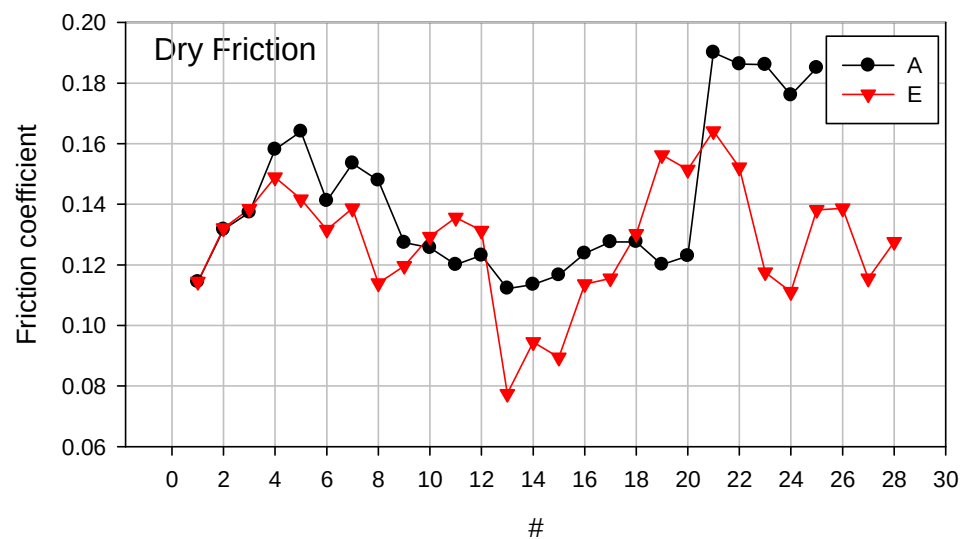
	<u>Physical Properties</u>				<u>Chemical Properties (%)</u>				
	Size (mm)	Bulk density (kg/l)	Specific gravity (g/cm3)	Vickers Hardness	ZrO2	Y2O3	CeOs	Al2O3	SiO2
Ceria Stabilised Zirconia	2.96	3.94	6.23	1178	98	-		-	-
Zirconia Toughened Alumina A1	3.60	2.46	3.81	1274	14	-	-	79	6.5
Zirconia Toughened Alumina A2	3.45	2.47	3.82	1363					
Zirconium Silicate1	3.36	2.50	4.00	1020	62	-	-	-	33
Zirconium Silicate 2	3.14	1.98	3.20	1206	30	-	-	25	40
Zirconia Toughened Alumina B1	3.44	2.44	3.81	1277	13	-	-	80	7
Zirconia Toughened Alumina B2	3.13	2.46	3.81	1314					
Alumina	3.56	2.23	3.57	1116	-	-	-	90	10
Alumina Toughened Zirconia	3.06	3.36	5.31	1270	70	-	-	30	-

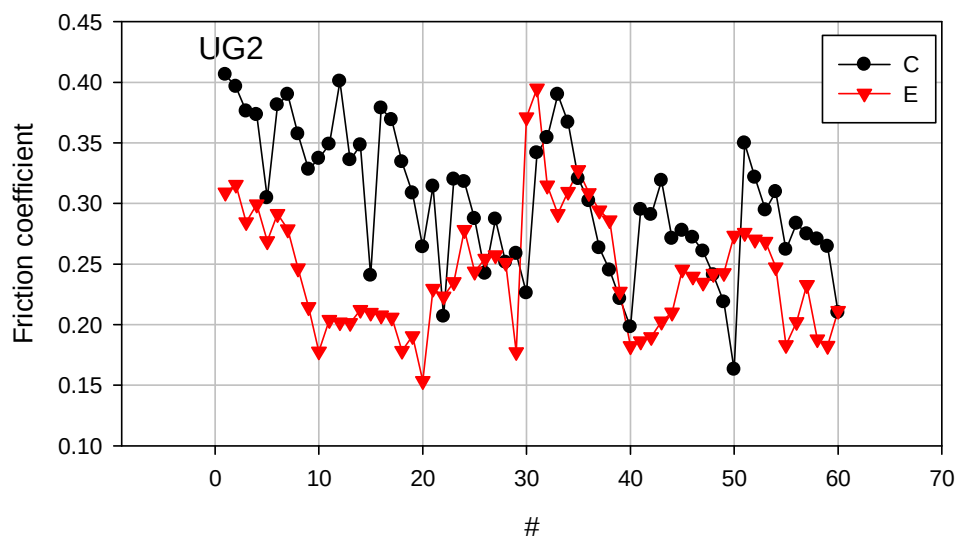
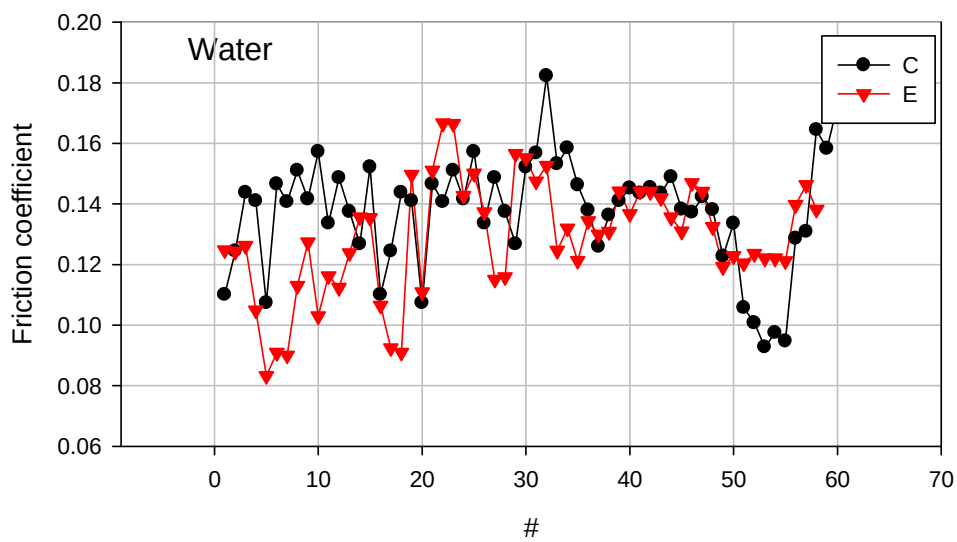
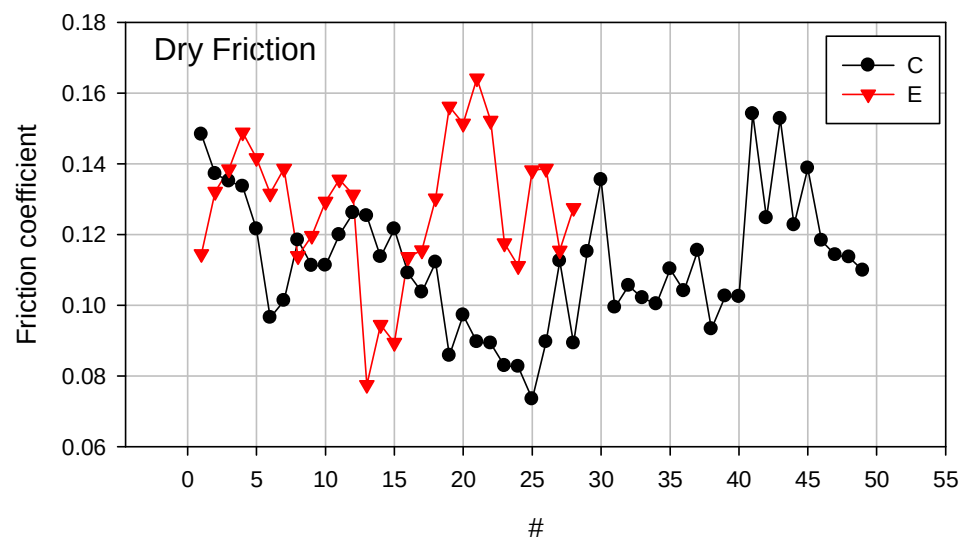
Media Type				Media data						
	<u>Mill Feed Properties</u>			<u>Pre Conditioning Test</u>				<u>Slurry Test</u>		
	Flowrate (l/h)	Solids SG	Slurry SG	Mass In (g)	Mass Out (g)	Volume (ml)	Bulk (SG)	Mass In (g)	Mass Out (g)	Volume (ml)
Ce Stab (6.23)	2.6	3.8	1.42	7872.0	7868.1	2000	3.94	7868.1	7864.4	1998.1
ZTA A1 (3.81)	2.6	3.8	1.42	4923.8	4903.3	2000	2.46	4903.3	4861.1	1974.5
ZTA A2 (3.82)	2.6	3.8	1.42	4945.1	4924.7	2000	2.47	4924.7	4901.3	1982.3
Zirconium Silicate1 (4.00)	2.6	3.8	1.40	5007.0	5005	2000	2.50	5005	4995.5	1995.4
Zirconium Silicate 2 (3.20)	2.6	3.8	1.40	3965.4	3963.4	2000	1.98	3963.4	3944.6	1989.5
ZTA B1 (3.81)	2.6	3.8	1.40	4886.8	4853.7	2000	2.44	4853.7	4813	1969.8
ZTA B2 (3.81)	2.6	3.8	1.42	4910.6	4826.6	2000	2.46	4826.6	4773.4	1944.1
Alumina (3.57)	2.6	3.8	1.42	4467.4	4430.6	2000	2.23	4430.6	4393.9	1967.1
ATZ (5.2)	2.6	3.8	1.42	6721.0	6717.1	2000	3.36	6717.1	6695.3	1992.4

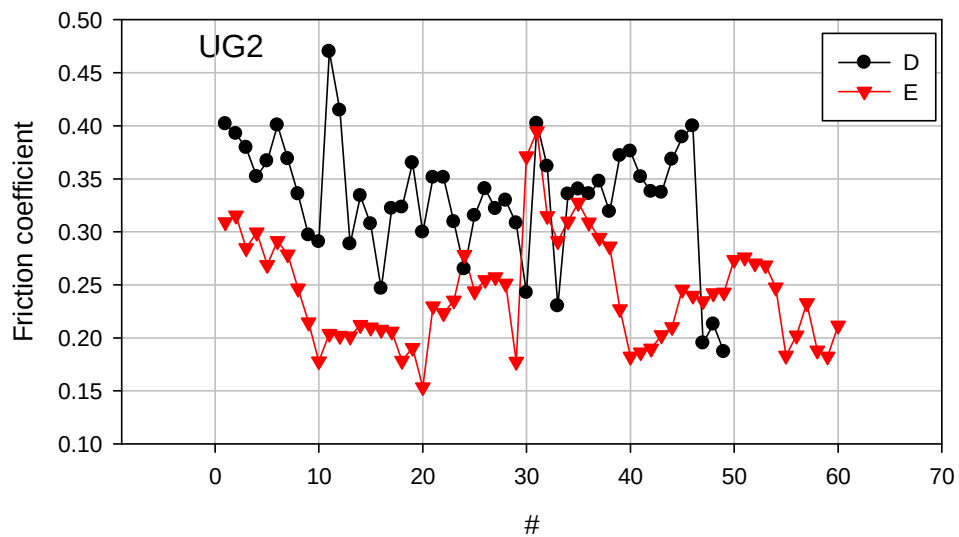
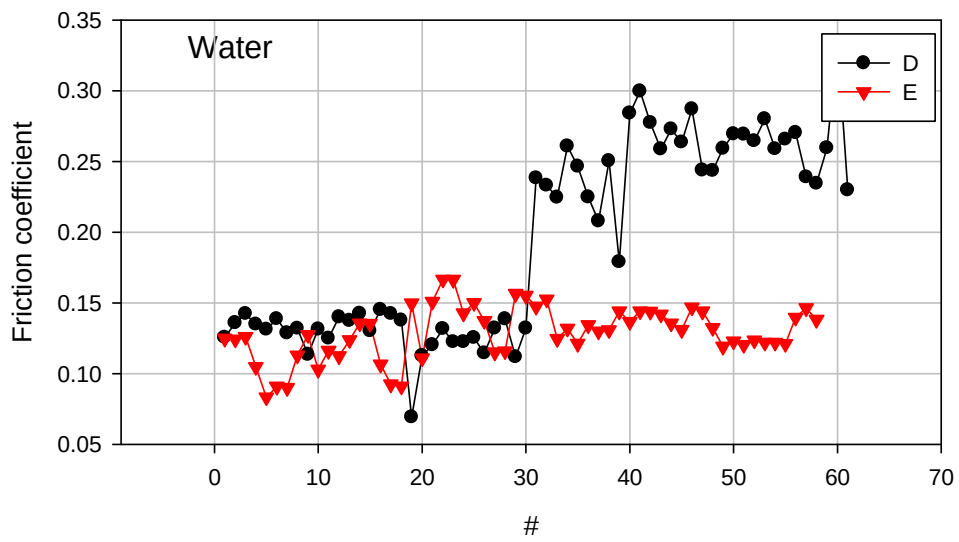
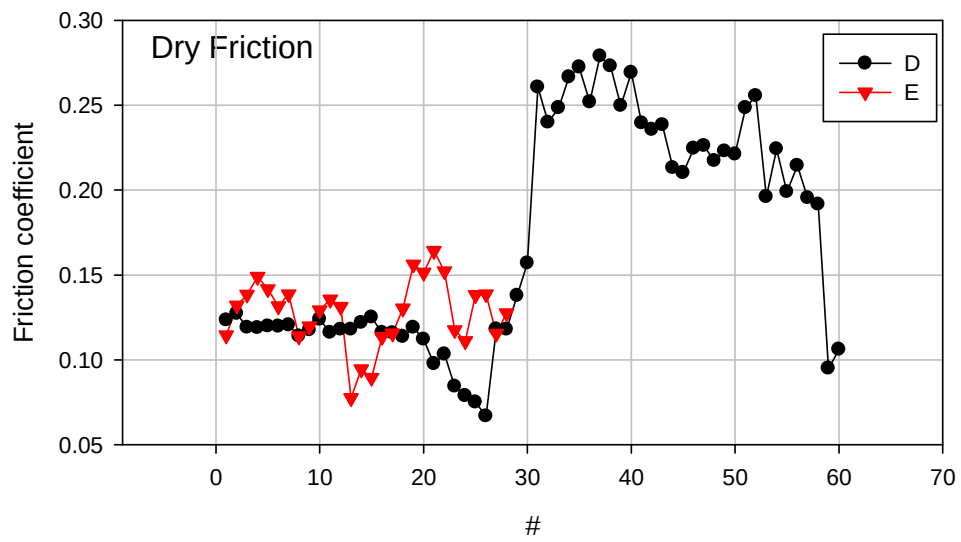
	Power Draw Data						
	<u>Preconditioning Test</u>		<u>Slurry Test</u>		<u>Equipment</u>		
Media Type	Power Draw (kW)	Power draw per kg Media (kW/kg)	Power Draw (kW)	Power draw per kg Media (kW/kg)	No Load Power (kW)	Friction Coefficient - Water	Friction Coefficient - Slurry
Ce Stab (6.23)	2.00	0.254	2.80	0.356	1.10	0.11	0.20
ZTA A1 (3.81)	2.01	0.408	2.97	0.605	1.10	0.12	0.24
ZTA A2 (3.82)	2.15	0.434	3.22	0.653	1.10	0.12	0.26
Zirconium Silicate1 (4.00)	2.02	0.403	2.85	0.569	1.10	0.11	0.22
Zirconium Silicate 2 (3.20)	2.30	0.580	3.32	0.838	1.10	0.14	0.29
ZTA B1 (3.81)	2.15	0.440	3.24	0.668	1.10	0.12	0.26
ZTA B2 (3.81)	2.32	0.473	3.17	0.657	1.10	0.14	0.26
Alumina (3.57)	1.97	0.441	2.90	0.655	1.10	0.11	0.23
ATZ (5.2)	2.35	0.350	3.18	0.474	1.10	0.13	0.24

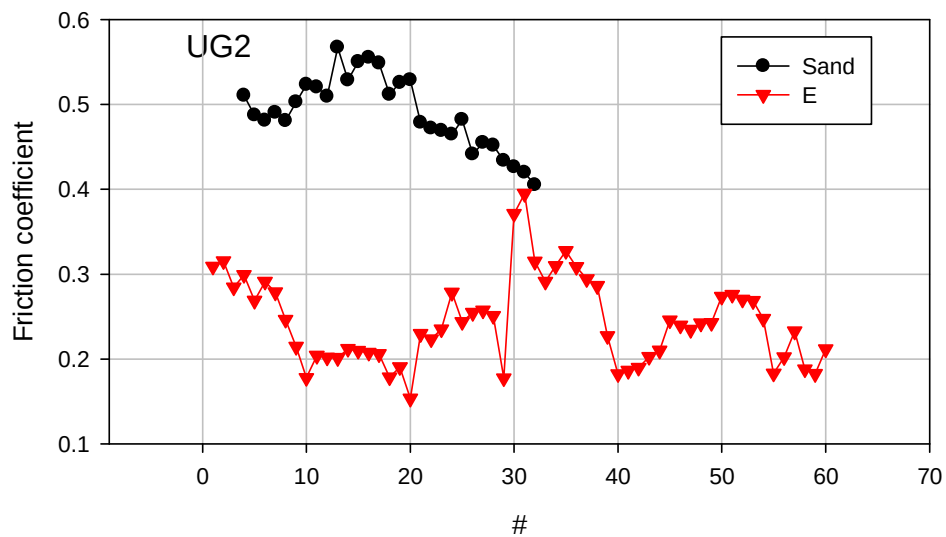
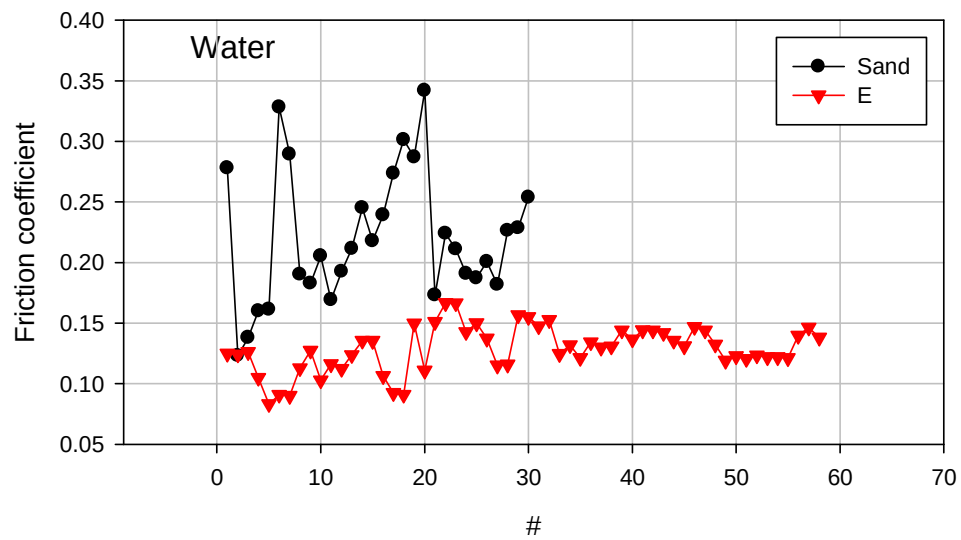
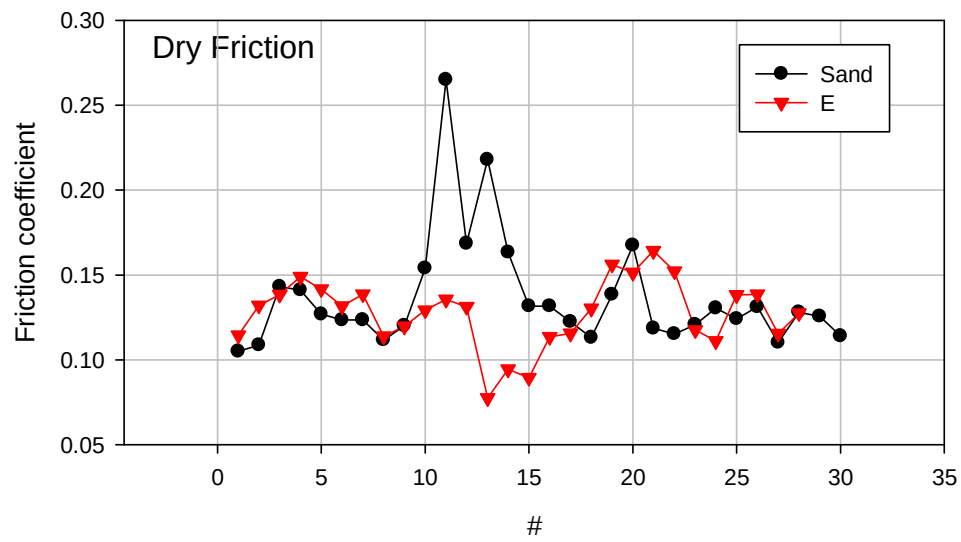
Measured Friction
Coefficient

Appendix C: Friction Coefficient Measurement Graphs



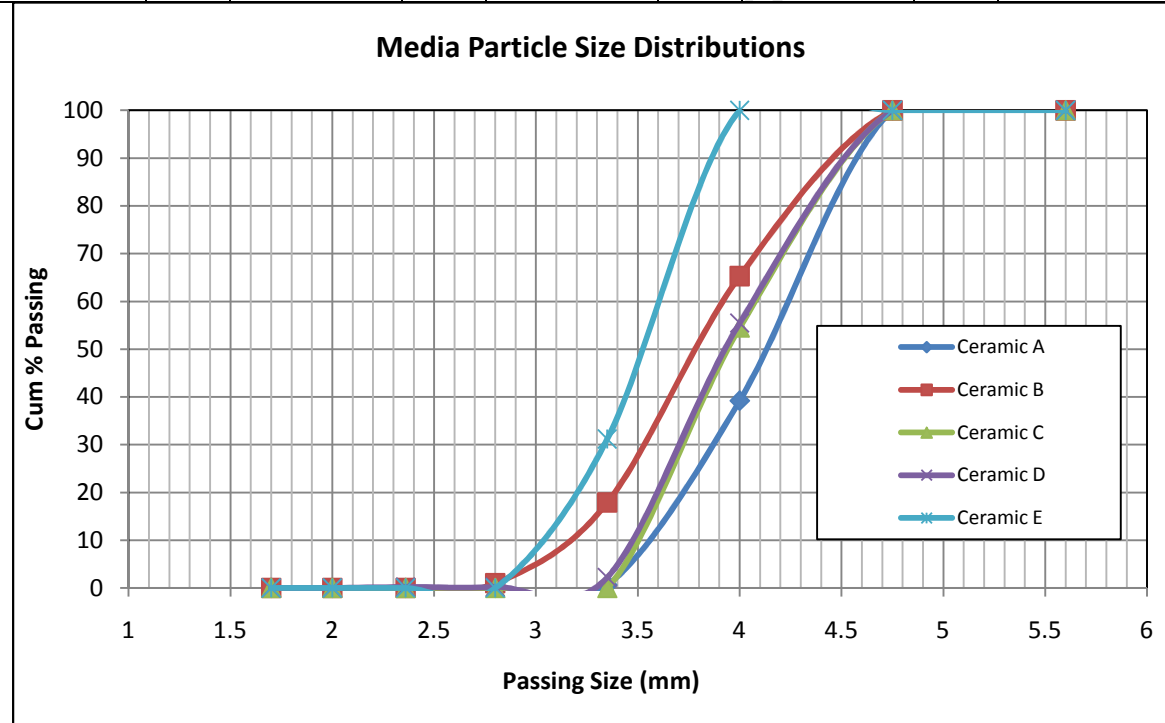






Appendix D: Ceramic Media Sizing Data

Size (mm)	Median Size (mm)	Ceramic A		Ceramic B		Ceramic C		Ceramic D		Ceramic E	
		%	Cum % Passing	%	Cum % Passing	%	Cum % Passing	%	Cum % Passing	%	Cum % Passing
-1.7	1.7	0	0	0	0	0	0	0	0.00	0.00	0.0
-2.00 +1.70	2	0	0	0	0	0	0	0.2	0.00	0.00	0.0
-2.36 + 2.00	2.36	0	0	1	0	0	0	0	0.2	0.00	0.0
-2.80 + 2.36	2.8	0.6	0.00	16.9	1	0	0	2	0.2	31.20	0.0
-3.35 + 2.80	3.35	38.6	0.6	47.4	17.9	54.5	0	53.3	2.2	68.80	31.2
-4.00 + 3.35	4	60.8	39.2	34.7	65.3	45.5	54.5	44.5	55.5	0.00	100.0
-4.75 + 4.00	4.75	0	100	0	100	0	100	0	100	0.00	100.0
-5.60 + 4.75	5.6	0	100	0	100	0	100	0	100	0.00	100.0



Ceramic A Sizing Data					
Minor Axis	Major Axis	Aspect Ratio	Area	Volume	Size
3.449	3.66	1.06	9.91	21.53	3.55
3.501	3.703	1.06	10.18	22.41	3.60
3.239	3.845	1.19	9.78	19.96	3.54
3.515	3.689	1.05	10.18	22.51	3.60
3.561	3.863	1.08	10.8	24.12	3.71
3.304	3.942	1.19	10.22	21.24	3.62
3.454	3.789	1.1	10.27	22.31	3.62
3.214	3.869	1.2	9.76	19.78	3.54
3.293	3.522	1.07	9.11	18.96	3.41
3.357	4.018	1.2	10.59	22.32	3.69
3.366	3.561	1.06	9.41	19.99	3.46
3.385	3.612	1.07	9.6	20.5	3.50
3.51	3.781	1.08	10.42	22.97	3.65
3.421	3.843	1.12	10.32	22.2	3.63
3.244	4.75	1.46	12.1	24.46	4.00
3.478	3.666	1.05	10.01	21.91	3.57
3.754	4.002	1.07	11.8	27.64	3.88
3.459	3.687	1.07	10.01	21.8	3.57
3.348	3.91	1.17	10.28	21.63	3.63
3.605	3.894	1.08	11.02	24.89	3.75
3.374	3.627	1.07	9.61	20.45	3.50
3.366	3.575	1.06	9.45	20.07	3.47
3.144	3.946	1.26	9.74	19.31	3.55
3.334	3.542	1.06	9.27	19.52	3.44
3.408	3.651	1.07	9.77	20.98	3.53
3.222	3.611	1.12	9.13	18.6	3.42
3.337	3.625	1.09	9.5	20	3.48
3.242	3.77	1.16	9.59	19.62	3.51
3.253	3.754	1.15	9.59	19.67	3.50
3.268	3.447	1.05	8.84	18.3	3.36
3.387	4.14	1.22	11.01	23.36	3.76
3.254	3.874	1.19	9.89	20.28	3.56

Ceramic A Sizing Data					
Minor Axis	Major Axis	Aspect Ratio	Area	Volume	Size
3.493	3.705	1.06	10.16	22.32	3.60
3.266	3.844	1.18	9.85	20.27	3.56
3.518	3.566	1.01	9.85	21.83	3.54
3.374	3.832	1.14	10.15	21.54	3.60
3.74	3.759	1	11.04	25.86	3.75
3.169	3.71	1.17	9.23	18.48	3.44
3.527	3.576	1.01	9.9	22	3.55
3.285	3.546	1.08	9.14	18.99	3.42
3.254	3.548	1.09	9.06	18.66	3.40
3.413	3.446	1.01	9.23	19.91	3.43
3.525	3.783	1.07	10.47	23.17	3.65
2.994	3.57	1.19	8.39	15.95	3.28
3.369	3.518	1.04	9.31	19.81	3.44
3.671	4.075	1.11	11.74	26.92	3.87
3.307	3.584	1.08	9.31	19.44	3.45
3.361	3.384	1.01	8.93	18.99	3.37
3.605	3.688	1.02	10.44	23.64	3.65
3.464	3.707	1.07	10.08	21.98	3.59
3.314	3.399	1.03	8.84	18.56	3.36
3.493	3.787	1.08	10.38	22.79	3.64
3.493	3.67	1.05	10.06	22.12	3.58
3.313	3.732	1.13	9.7	20.27	3.52
3.377	3.63	1.08	9.62	20.49	3.50
3.324	3.756	1.13	9.8	20.53	3.54
3.414	3.921	1.15	10.51	22.53	3.67
3.476	3.535	1.02	9.65	21.15	3.51
3.13	3.702	1.18	9.1	18.01	3.42
3.673	3.983	1.08	11.48	26.37	3.83
3.428	3.626	1.06	9.76	21.08	3.53
3.211	3.785	1.18	9.54	19.33	3.50
3.496	3.87	1.11	10.62	23.3	3.68
3.34	3.815	1.14	10	21.03	3.58
3.339	3.714	1.11	9.74	20.49	3.53
3.627	3.752	1.03	10.68	24.31	3.69
3.709	3.889	1.05	11.32	26.27	3.80

Ceramic A Sizing Data					
Minor Axis	Major Axis	Aspect Ratio	Area	Volume	Size
3.266	3.907	1.2	10.02	20.6	3.59
3.435	3.889	1.13	10.48	22.62	3.66
3.212	3.903	1.22	9.84	19.92	3.56
3.498	3.581	1.02	9.83	21.67	3.54
3.254	3.837	1.18	9.8	20.09	3.55
3.389	3.98	1.17	10.59	22.52	3.68
3.124	3.362	1.08	8.24	16.35	3.24
3.217	3.769	1.17	9.52	19.32	3.49
3.512	3.616	1.03	9.97	22.04	3.56
3.411	3.55	1.04	9.5	20.46	3.48
3.446	3.675	1.07	9.94	21.57	3.56
3.45	3.618	1.05	9.8	21.31	3.53
3.672	3.997	1.09	11.52	26.44	3.83
3.545	3.615	1.02	10.06	22.44	3.58
3.267	3.803	1.16	9.75	20.08	3.54
3.284	3.626	1.1	9.35	19.39	3.46
3.44	4.056	1.18	10.95	23.61	3.75
3.425	4.024	1.17	10.82	23.24	3.72
3.314	3.747	1.13	9.75	20.37	3.53
3.347	3.652	1.09	9.6	20.26	3.50
3.392	3.805	1.12	10.13	21.62	3.60
3.426	3.92	1.14	10.54	22.68	3.67
3.256	3.702	1.14	9.46	19.45	3.48
3.376	3.967	1.17	10.51	22.29	3.67
3.142	3.6	1.15	8.88	17.67	3.37
3.321	3.995	1.2	10.41	21.72	3.66
3.156	3.427	1.09	8.49	17	3.29
3.735	3.769	1.01	11.05	25.85	3.75
3.057	3.774	1.23	9.06	17.51	3.42
3.368	3.502	1.04	9.26	19.71	3.44
3.299	3.844	1.17	9.95	20.67	3.57
3.58	4.091	1.14	11.5	25.72	3.84
3.384	3.817	1.13	10.14	21.59	3.60
3.653	3.982	1.09	11.42	26.08	3.82

Ceramic B					
Minor Axis	Major Axis	Aspect Ratio	Area	Volume	Size
3.474	3.667	1.06	10	21.87	3.57
3.552	4.32	1.22	12.04	26.67	3.94
3.018	4.171	1.38	9.88	18.79	3.59
3.448	3.643	1.06	9.86	21.42	3.55
3.615	3.922	1.09	11.13	25.2	3.77
3.356	3.734	1.11	9.84	20.8	3.55
3.521	3.625	1.03	10.02	22.21	3.57
2.771	3.72	1.34	8.09	14.25	3.25
1.783	1.978	1.11	2.77	3.25	1.88
3.169	3.722	1.17	9.26	18.54	3.45
2.748	3.347	1.22	7.22	12.67	3.05
3.101	4.069	1.31	9.91	19.35	3.59
3.046	3.579	1.18	8.56	16.53	3.31
2.921	3.689	1.26	8.46	15.68	3.31
3.49	4.213	1.21	11.54	25.18	3.85
2.934	3.851	1.31	8.87	16.47	3.39
3.134	3.635	1.16	8.94	17.73	3.38
3.67	4.246	1.16	12.23	27.97	3.96
3.011	3.905	1.3	9.23	17.56	3.46
3.671	3.929	1.07	11.32	25.99	3.80
2.958	3.386	1.14	7.86	14.8	3.17
3.196	3.338	1.04	8.37	16.99	3.27
2.944	3.705	1.26	8.56	15.98	3.32
3.389	4.123	1.22	10.97	23.29	3.76
3.218	3.547	1.1	8.96	18.24	3.38
3.361	4.498	1.34	11.87	24.9	3.93
3.925	4.122	1.05	12.7	31	4.02
3.621	3.95	1.09	11.23	25.44	3.79
3.204	3.555	1.11	8.94	18.13	3.38
3.194	3.743	1.17	9.38	18.92	3.47

Ceramic B					
Minor Axis	Major Axis	Aspect Ratio	Area	Volume	Size
2.77	3.984	1.44	8.66	15.21	3.38
2.831	3.474	1.23	7.72	13.92	3.15
3.381	3.651	1.08	9.69	20.66	3.52
3.411	3.516	1.03	9.41	20.27	3.46
3.239	3.633	1.12	9.24	18.9	3.44
3.149	3.658	1.16	9.04	18.01	3.40
3.731	4.09	1.1	11.98	27.87	3.91
3.398	4.003	1.18	10.68	22.77	3.70
3.292	3.677	1.12	9.5	19.75	3.48
3.135	3.314	1.06	8.15	16.24	3.22
3.278	3.717	1.13	9.56	19.77	3.50
3.155	3.811	1.21	9.44	18.8	3.48
3.084	3.985	1.29	9.65	18.77	3.53
3.061	3.88	1.27	9.32	18.03	3.47
3.431	3.655	1.07	9.85	21.29	3.54
3.295	4.052	1.23	10.48	21.69	3.67
3.204	4.078	1.27	10.26	20.66	3.64
3.162	3.717	1.18	9.23	18.44	3.44
3.495	3.869	1.11	10.61	23.28	3.68
2.863	3.857	1.35	8.67	15.73	3.36
3.214	4.054	1.26	10.23	20.67	3.63
3.735	4.242	1.14	12.44	28.92	3.99
3.663	3.925	1.07	11.29	25.87	3.79
3.587	4.203	1.17	11.83	26.49	3.90
3.509	3.777	1.08	10.4	22.94	3.64
3.82	4.237	1.11	12.71	30.18	4.03
3.705	4.271	1.15	12.42	28.65	3.99
2.961	3.682	1.24	8.56	16.07	3.32
3.805	4.075	1.07	12.17	28.86	3.94
3.145	3.504	1.11	8.65	17.25	3.32
3.371	4.126	1.22	10.92	23.07	3.75
3.495	3.532	1.01	9.69	21.36	3.51
3.726	3.922	1.05	11.47	26.72	3.82
3.592	4.135	1.15	11.66	26.16	3.86
3.41	4.044	1.19	10.83	23.15	3.73
2.956	3.771	1.28	8.75	16.39	3.36
3.318	3.622	1.09	9.44	19.77	3.47
2.741	3.566	1.3	7.67	13.4	3.15
3.05	3.612	1.18	8.65	16.71	3.33

Ceramic B					
Minor Axis	Major Axis	Aspect Ratio	Area	Volume	Size
3.091	3.436	1.11	8.33	16.35	3.26
2.951	3.943	1.34	9.14	17.04	3.45
3.075	4.092	1.33	9.88	19.14	3.58
2.914	3.323	1.14	7.6	14.12	3.12
2.913	3.844	1.32	8.79	16.22	3.38
2.987	3.333	1.12	7.82	14.86	3.16
3.1	4.239	1.37	10.31	20.1	3.67
2.131	2.947	1.38	4.93	6.81	2.54
3.097	4.24	1.37	10.31	20.07	3.67
2.948	3.103	1.05	7.18	13.52	3.03
3.331	3.842	1.15	10.05	21.07	3.59
3.04	3.813	1.25	9.1	17.49	3.43
2.836	3.05	1.08	6.79	12.33	2.94
3.229	3.528	1.09	8.94	18.28	3.38
2.913	3.805	1.31	8.7	16.06	3.36
3.264	4.176	1.28	10.7	21.91	3.72
3.47	3.719	1.07	10.13	22.11	3.59
3.438	3.837	1.12	10.36	22.38	3.64
3.53	4.548	1.29	12.6	27.66	4.04
3.396	3.643	1.07	9.71	20.8	3.52
3.193	3.601	1.13	9.03	18.24	3.40
3.683	4.204	1.14	12.16	27.9	3.94
3.107	3.79	1.22	9.24	18.15	3.45
3.58	4.217	1.18	11.85	26.47	3.90
3.643	3.949	1.08	11.29	25.74	3.80
3.299	3.721	1.13	9.64	20.05	3.51
3.05	3.386	1.11	8.11	15.72	3.22
2.975	3.63	1.22	8.48	16	3.30
3.216	3.475	1.08	8.77	17.88	3.35
3.726	3.905	1.05	11.42	26.6	3.82
3.01	3.527	1.17	8.33	15.93	3.27
3.135	3.703	1.18	9.11	18.06	3.42

Ceramic C Sizing Data					
Minor Axis	Major Axis	Aspect Ratio	Area	Volume	Size
3.437	3.49	1.02	9.42	20.44	3.46
3.505	3.542	1.01	9.75	21.53	3.52
3.305	3.326	1.01	8.63	18.08	3.32
3.375	3.394	1.01	8.99	19.21	3.38
3.327	3.366	1.01	8.79	18.53	3.35
3.316	3.333	1.01	8.68	18.23	3.32
3.41	3.436	1.01	9.2	19.82	3.42
3.233	3.255	1.01	8.26	16.96	3.24
3.299	3.318	1.01	8.59	17.97	3.31
3.222	3.251	1.01	8.22	16.83	3.24
3.451	3.499	1.01	9.48	20.65	3.48
3.238	3.279	1.01	8.34	17.14	3.26
3.535	3.581	1.01	9.94	22.13	3.56
3.315	3.337	1.01	8.68	18.24	3.33
3.151	3.169	1.01	7.84	15.73	3.16
3.194	3.243	1.02	8.13	16.5	3.22
3.47	3.489	1.01	9.5	20.81	3.48
3.254	3.319	1.02	8.48	17.5	3.29
3.42	3.444	1.01	9.25	19.99	3.43
3.473	3.481	1	9.49	20.8	3.48
3.242	3.264	1.01	8.31	17.1	3.25
3.491	3.547	1.02	9.72	21.39	3.52
3.454	3.459	1	9.38	20.46	3.46
3.394	3.418	1.01	9.11	19.54	3.41
3.258	3.28	1.01	8.39	17.35	3.27
3.266	3.292	1.01	8.44	17.49	3.28
3.173	3.22	1.01	8.02	16.18	3.20
3.406	3.451	1.01	9.23	19.86	3.43
3.367	3.404	1.01	9	19.17	3.39
3.383	3.387	1	9	19.26	3.39

Ceramic C Sizing Data					
Minor Axis	Major Axis	Aspect Ratio	Area	Volume	Size
3.314	3.384	1.02	8.8	18.48	3.35
3.213	3.249	1.01	8.19	16.72	3.23
3.209	3.273	1.02	8.24	16.8	3.24
3.492	3.553	1.02	9.74	21.43	3.52
3.386	3.413	1.01	9.07	19.43	3.40
3.286	3.305	1.01	8.52	17.76	3.30
3.3	3.377	1.02	8.75	18.28	3.34
3.232	3.253	1.01	8.26	16.95	3.24
3.206	3.244	1.01	8.17	16.64	3.23
3.253	3.291	1.01	8.4	17.35	3.27
3.512	3.547	1.01	9.78	21.64	3.53
3.426	3.444	1.01	9.26	20.06	3.44
3.475	3.491	1	9.52	20.88	3.48
3.144	3.179	1.01	7.85	15.7	3.16
3.315	3.346	1.01	8.71	18.29	3.33
3.329	3.359	1.01	8.78	18.51	3.34
3.352	3.387	1.01	8.91	18.91	3.37
3.223	3.279	1.02	8.29	16.97	3.25
3.576	3.597	1.01	10.1	22.72	3.59
3.456	3.498	1.01	9.49	20.7	3.48
3.572	3.593	1.01	10.07	22.65	3.58
3.17	3.207	1.01	7.98	16.09	3.19
3.298	3.342	1.01	8.65	18.09	3.32
3.18	3.243	1.02	8.1	16.37	3.21
3.418	3.458	1.01	9.28	20.04	3.44
3.258	3.324	1.02	8.5	17.57	3.29
3.386	3.454	1.02	9.18	19.65	3.42
3.388	3.417	1.01	9.09	19.47	3.40
3.233	3.269	1.01	8.3	17.03	3.25
3.466	3.504	1.01	9.53	20.85	3.49
3.179	3.21	1.01	8.01	16.2	3.19
3.178	3.2	1.01	7.98	16.14	3.19
3.208	3.262	1.02	8.21	16.74	3.24
3.223	3.28	1.02	8.3	16.98	3.25
3.465	3.507	1.01	9.54	20.86	3.49
3.198	3.255	1.02	8.17	16.6	3.23
3.373	3.438	1.02	9.1	19.41	3.41
3.44	3.473	1.01	9.38	20.37	3.46
3.248	3.291	1.01	8.39	17.3	3.27

Ceramic C Sizing Data					
Minor Axis	Major Axis	Aspect Ratio	Area	Volume	Size
3.189	3.273	1.03	8.19	16.6	3.23
3.523	3.593	1.02	9.94	22.04	3.56
3.201	3.249	1.02	8.16	16.61	3.23
3.241	3.245	1	8.25	16.99	3.24
3.164	3.23	1.02	8.02	16.14	3.20
3.392	3.434	1.01	9.14	19.61	3.41
3.374	3.453	1.02	9.15	19.51	3.41
3.06	3.082	1.01	7.4	14.45	3.07
3.3	3.347	1.01	8.67	18.13	3.32
3.247	3.262	1	8.32	17.15	3.25
3.404	3.449	1.01	9.22	19.83	3.43
3.272	3.35	1.02	8.61	17.85	3.31
3.157	3.186	1.01	7.89	15.86	3.17
3.164	3.177	1	7.89	15.89	3.17
3.507	3.551	1.01	9.78	21.61	3.53
3.362	3.484	1.04	9.19	19.54	3.42
3.214	3.319	1.03	8.37	17.08	3.27
3.299	3.332	1.01	8.63	18.05	3.32
3.439	3.516	1.02	9.49	20.6	3.48
3.27	3.296	1.01	8.46	17.55	3.28
3.147	3.2	1.02	7.9	15.83	3.17
3.59	3.602	1	10.15	22.93	3.60
3.234	3.286	1.02	8.34	17.13	3.26
3.399	3.431	1.01	9.15	19.67	3.42
3.493	3.518	1.01	9.65	21.25	3.51
3.201	3.22	1.01	8.09	16.46	3.21
3.356	3.396	1.01	8.95	19	3.38
3.221	3.287	1.02	8.31	16.99	3.25
3.307	3.352	1.01	8.7	18.23	3.33
3.515	3.531	1	9.74	21.59	3.52
3.368	3.401	1.01	8.99	19.17	3.38
3.551	3.616	1.02	10.08	22.53	3.58

Ceramic D Sizing Data					
Minor Axis	Major Axis	Aspect Ratio	Area	Volume	Size
3.399	3.444	1.01	9.19	19.75	3.42
3.227	3.268	1.01	8.28	16.97	3.25
3.167	3.233	1.02	8.04	16.19	3.20
3.398	3.417	1.01	9.11	19.58	3.41
3.551	3.579	1.01	9.98	22.31	3.57
3.182	3.22	1.01	8.04	16.28	3.20
3.031	3.11	1.03	7.4	14.31	3.07
3.323	3.349	1.01	8.74	18.39	3.34
3.388	3.435	1.01	9.13	19.56	3.41
3.139	3.173	1.01	7.82	15.63	3.16
3.247	3.305	1.02	8.42	17.36	3.28
3.23	3.286	1.02	8.33	17.09	3.26
3.29	3.342	1.02	8.63	18	3.32
3.181	3.229	1.02	8.06	16.31	3.21
3.337	3.369	1.01	8.83	18.65	3.35
3.545	3.585	1.01	9.98	22.27	3.57
3.412	3.453	1.01	9.25	19.94	3.43
3.275	3.284	1	8.44	17.55	3.28
3.275	3.352	1.02	8.62	17.89	3.31
3.158	3.192	1.01	7.91	15.9	3.18
3.101	3.105	1	7.56	14.94	3.10
3.325	3.358	1.01	8.76	18.46	3.34
3.391	3.416	1.01	9.09	19.5	3.40
3.282	3.308	1.01	8.52	17.74	3.30
3.229	3.262	1.01	8.27	16.95	3.25
3.283	3.37	1.03	8.69	18.07	3.33
3.4	3.438	1.01	9.18	19.72	3.42
3.194	3.208	1	8.04	16.34	3.20
3.344	3.425	1.02	8.99	19.02	3.38
3.271	3.286	1	8.44	17.52	3.28

Ceramic D Sizing Data					
Minor Axis	Major Axis	Aspect Ratio	Area	Volume	Size
3.153	3.239	1.03	8.02	16.07	3.20
3.474	3.489	1	9.51	20.86	3.48
3.257	3.283	1.01	8.39	17.35	3.27
3.134	3.154	1.01	7.76	15.48	3.14
3.272	3.334	1.02	8.56	17.77	3.30
3.484	3.504	1.01	9.58	21.06	3.49
3.121	3.14	1.01	7.69	15.29	3.13
3.552	3.611	1.02	10.07	22.5	3.58
3.259	3.334	1.02	8.53	17.62	3.30
3.123	3.176	1.02	7.79	15.49	3.15
3.361	3.377	1	8.91	18.96	3.37
3.531	3.586	1.02	9.94	22.1	3.56
2.839	2.878	1.01	6.41	11.68	2.86
3.095	3.102	1	7.54	14.88	3.10
3.277	3.314	1.01	8.52	17.72	3.30
3.254	3.294	1.01	8.41	17.37	3.27
3.69	3.729	1.01	10.8	25	3.71
2.67	2.697	1.01	5.65	9.73	2.68
3.212	3.258	1.01	8.22	16.76	3.24
3.335	3.364	1.01	8.81	18.6	3.35
3.22	3.249	1.01	8.21	16.8	3.23
3.203	3.22	1.01	8.1	16.48	3.21
3.216	3.228	1	8.15	16.65	3.22
3.168	3.175	1	7.9	15.92	3.17
3.099	3.119	1.01	7.59	14.99	3.11
3.374	3.414	1.01	9.04	19.3	3.39
3.281	3.332	1.02	8.58	17.85	3.31
3.14	3.185	1.01	7.85	15.69	3.16
3.218	3.247	1.01	8.2	16.77	3.23
3.122	3.197	1.02	7.84	15.57	3.16
3.275	3.283	1	8.44	17.54	3.28
3.226	3.246	1.01	8.22	16.85	3.24
3.347	3.383	1.01	8.89	18.83	3.37
3.06	3.088	1.01	7.42	14.49	3.07
3.053	3.087	1.01	7.4	14.42	3.07
3.4	3.44	1.01	9.18	19.73	3.42
2.997	3.019	1.01	7.1	13.61	3.01
3.391	3.446	1.02	9.17	19.66	3.42
3.249	3.278	1.01	8.36	17.24	3.26

Ceramic D Sizing Data					
Minor Axis	Major Axis	Aspect Ratio	Area	Volume	Size
3.455	3.491	1.01	9.47	20.65	3.47
3.41	3.453	1.01	9.24	19.92	3.43
3.247	3.271	1.01	8.34	17.18	3.26
2.886	2.928	1.01	6.63	12.27	2.91
3.391	3.404	1	9.06	19.44	3.40
3.311	3.343	1.01	8.69	18.23	3.33
3.162	3.219	1.02	7.99	16.07	3.19
3.094	3.181	1.03	7.73	15.23	3.14
3.084	3.165	1.03	7.66	15.06	3.12
3.174	3.199	1.01	7.97	16.1	3.19
3.36	3.413	1.02	9	19.14	3.39
3.627	3.658	1.01	10.41	23.73	3.64
3.571	3.629	1.02	10.17	22.85	3.60
3.202	3.247	1.01	8.16	16.61	3.22
2.182	2.227	1.02	3.82	5.44	2.20
3.539	3.574	1.01	9.93	22.13	3.56
3.238	3.25	1	8.26	16.99	3.24
3.088	3.1	1	7.51	14.79	3.09
3.12	3.148	1.01	7.71	15.33	3.13
3.097	3.184	1.03	7.74	15.27	3.14
3.405	3.47	1.02	9.28	19.95	3.44
3.21	3.3	1.03	8.32	16.95	3.26
3.406	3.445	1.01	9.21	19.83	3.43
3.675	3.71	1.01	10.7	24.68	3.69
3.425	3.524	1.03	9.47	20.48	3.47
3.346	3.384	1.01	8.89	18.83	3.37
3.173	3.21	1.01	7.99	16.13	3.19
3.238	3.286	1.01	8.35	17.17	3.26
3.423	3.437	1	9.24	19.98	3.43
3.229	3.275	1.01	8.3	17.02	3.25
3.14	3.171	1.01	7.81	15.62	3.16
3.259	3.471	1.07	8.88	18.33	3.37

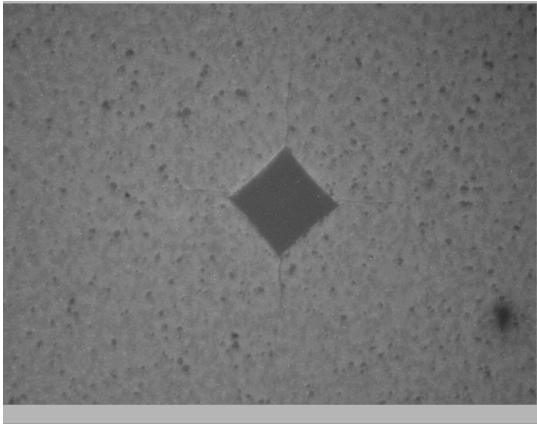
Ceramic E Sizing Data					
Minor Axis	Major Axis	Aspect Ratio	Area	Volume	Size
2.98	3.05	1.02	7.13	13.58	3.02
2.83	3.03	1.07	6.73	12.17	2.93
2.76	2.95	1.07	6.38	11.31	2.85
2.86	3.10	1.08	6.96	12.73	2.98
3.04	3.13	1.03	7.46	14.47	3.08
2.84	2.93	1.03	6.52	11.86	2.88
2.99	3.08	1.03	7.24	13.85	3.04
2.94	3.05	1.04	7.02	13.17	2.99
3.01	3.12	1.04	7.37	14.13	3.06
2.87	2.91	1.01	6.55	12.06	2.89
2.79	3.05	1.10	6.68	11.92	2.92
2.90	3.02	1.04	6.88	12.77	2.96
2.90	2.91	1.00	6.61	12.27	2.90
2.95	3.04	1.03	7.03	13.27	2.99
3.04	3.15	1.04	7.51	14.53	3.09
2.67	2.93	1.10	6.12	10.49	2.80
2.83	3.08	1.09	6.84	12.38	2.95
2.89	3.03	1.05	6.88	12.72	2.96
2.89	3.00	1.04	6.80	12.58	2.94
2.85	2.99	1.05	6.68	12.20	2.92
2.95	3.05	1.03	7.05	13.27	3.00
2.99	3.10	1.04	7.28	13.92	3.05
2.88	3.08	1.07	6.95	12.79	2.98
2.79	3.12	1.12	6.82	12.17	2.95
2.73	2.99	1.09	6.40	11.22	2.86
2.85	3.01	1.06	6.74	12.28	2.93
3.01	3.01	1.00	7.12	13.69	3.01
2.86	3.16	1.11	7.09	12.96	3.01
2.96	3.11	1.05	7.21	13.62	3.03
2.75	3.14	1.14	6.78	11.94	2.95

Ceramic E Sizing Data					
Minor Axis	Major Axis	Aspect Ratio	Area	Volume	Size
2.74	3.12	1.14	6.70	11.76	2.93
2.84	2.97	1.05	6.62	12.06	2.91
3.07	3.14	1.02	7.56	14.76	3.10
2.86	3.05	1.06	6.85	12.54	2.95
3.02	3.06	1.01	7.27	14.03	3.04
2.91	2.96	1.02	6.76	12.57	2.93
2.79	3.02	1.08	6.61	11.80	2.90
2.93	3.10	1.06	7.12	13.31	3.01
2.73	2.89	1.06	6.20	10.89	2.81
2.79	2.91	1.04	6.39	11.45	2.85
2.91	3.00	1.03	6.86	12.80	2.96
2.91	2.97	1.02	6.79	12.64	2.94
2.93	3.01	1.03	6.91	12.94	2.97
2.85	3.03	1.07	6.78	12.36	2.94
2.79	3.02	1.08	6.62	11.85	2.91
2.82	2.97	1.05	6.58	11.90	2.90
2.82	2.99	1.06	6.62	11.95	2.91
2.84	3.13	1.10	6.97	12.64	2.98
2.73	2.94	1.08	6.31	11.07	2.84
2.80	3.04	1.09	6.68	11.97	2.92
2.85	3.03	1.06	6.78	12.39	2.94
2.80	3.06	1.09	6.74	12.10	2.93
2.87	3.00	1.05	6.76	12.40	2.93
2.88	2.99	1.04	6.76	12.48	2.94
2.83	3.04	1.08	6.74	12.21	2.93
2.81	2.89	1.03	6.38	11.52	2.85
3.03	3.13	1.03	7.45	14.42	3.08
2.88	3.02	1.05	6.83	12.60	2.95
2.92	3.06	1.05	7.02	13.10	2.99
2.99	3.08	1.03	7.23	13.81	3.03
2.93	3.06	1.04	7.03	13.17	2.99
2.75	2.95	1.07	6.36	11.24	2.85
2.98	3.04	1.02	7.12	13.58	3.01
2.95	3.07	1.04	7.12	13.43	3.01
2.86	3.15	1.10	7.06	12.91	3.00
2.86	3.09	1.08	6.93	12.67	2.97
2.93	3.09	1.05	7.11	13.32	3.01
2.91	3.00	1.03	6.85	12.74	2.95
2.94	3.06	1.04	7.06	13.27	3.00

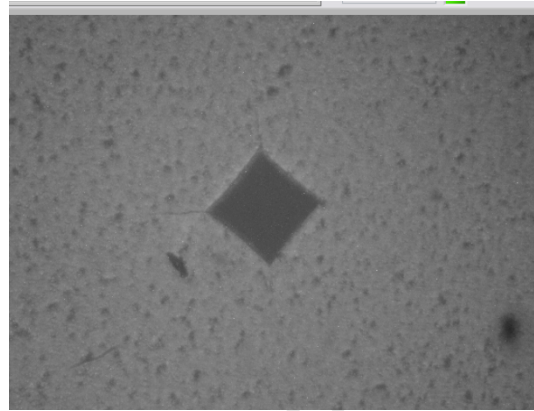
Ceramic E Sizing Data					
Minor Axis	Major Axis	Aspect Ratio	Area	Volume	Size
2.83	3.15	1.11	7.00	12.67	2.99
3.02	3.08	1.02	7.31	14.10	3.05
2.90	3.04	1.05	6.92	12.82	2.97
2.93	3.30	1.13	7.58	14.16	3.11
2.87	3.03	1.06	6.83	12.56	2.95
2.79	3.05	1.09	6.68	11.94	2.92
2.82	3.06	1.08	6.76	12.21	2.94
2.88	3.13	1.09	7.06	13.00	3.00
2.81	3.04	1.08	6.72	12.12	2.93
2.93	3.05	1.04	7.02	13.17	2.99
2.96	3.05	1.03	7.09	13.41	3.01
2.85	3.07	1.08	6.87	12.54	2.96
2.73	2.93	1.07	6.27	10.98	2.83
2.80	3.07	1.10	6.73	12.07	2.93
3.02	3.10	1.03	7.35	14.18	3.06
2.91	3.05	1.05	6.97	12.95	2.98
3.05	3.08	1.01	7.38	14.37	3.07
2.81	2.97	1.06	6.57	11.85	2.89
3.00	3.01	1.01	7.10	13.60	3.01
2.77	3.03	1.09	6.59	11.71	2.90
2.91	2.99	1.03	6.82	12.68	2.95
2.80	3.06	1.09	6.73	12.07	2.93
2.97	3.07	1.03	7.16	13.60	3.02
2.83	3.01	1.07	6.68	12.10	2.92
2.77	3.03	1.09	6.58	11.67	2.90
2.90	3.06	1.05	6.97	12.94	2.98
2.97	3.08	1.04	7.18	13.62	3.03
2.88	3.18	1.10	7.19	13.24	3.03
3.03	3.33	1.10	7.92	15.26	3.18
2.91	3.00	1.03	6.85	12.73	2.95
2.92	3.06	1.05	7.02	13.12	2.99
2.81	3.10	1.10	6.84	12.30	2.95

**Appendix E: Hardness and Fracture Toughness Measurement results and
Polished Section micrographs**

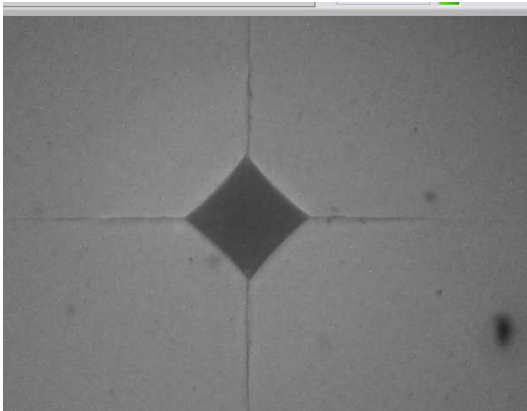
Test results on Samples A & E



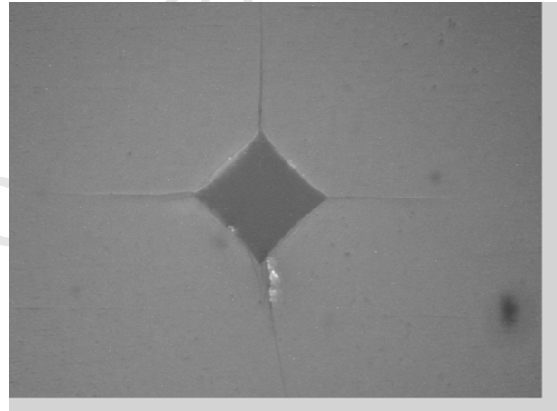
Sample A



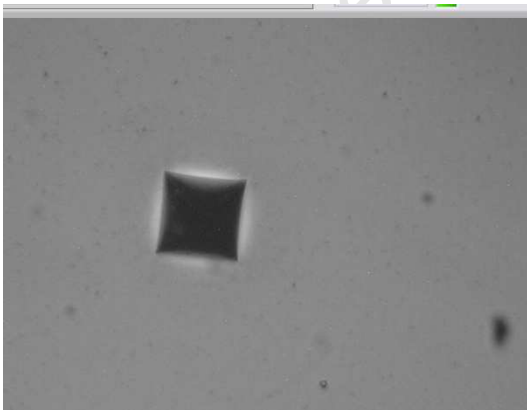
Sample B



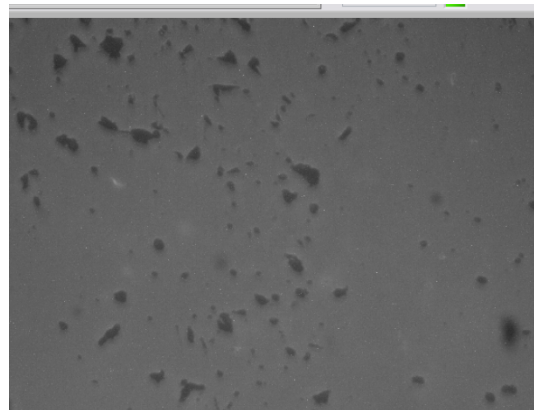
Sample C



Sample D



Sample E



Silica Sand

Hardness Measurement Results:

Sample	Hardness HV, GPa	StdDev	Fracture Toughness K1C MPa*m^{1/2}	StdDev
A	12.16	0.18	5.23	0.7
B	11.51	0.8	5.6	0.79
C	9.81	0.16	1.68	0.18
D	8.5	0.44	2.45	0.66
E	11.56	0.1	12.78	1.2

Appendix F: M4 IsaMill™ Feed Particle Size Distribution Data

		Cum % Smaller than specified Size									
Sample Name		Feed 1	Feed 2	Feed 3	Feed 4	Feed 5	Feed 6	Feed 7	Feed 8	Feed 9	Feed 10
Size (µm)	1.3	0.04	0.04	0.04	0.04	0.04	0.04	0.04	0.04	0.04	0.04
	1.4	0.26	0.25	0.25	0.25	0.26	0.26	0.26	0.26	0.26	0.26
	1.7	0.62	0.61	0.62	0.62	0.63	0.63	0.63	0.63	0.64	0.62
	1.9	1.11	1.09	1.10	1.10	1.13	1.13	1.12	1.13	1.13	1.11
	2.2	1.71	1.68	1.71	1.70	1.75	1.74	1.72	1.74	1.75	1.71
	2.5	2.41	2.37	2.41	2.40	2.47	2.44	2.42	2.45	2.46	2.41
	2.9	3.21	3.15	3.21	3.19	3.28	3.24	3.21	3.26	3.26	3.19
	3.3	4.09	4.02	4.10	4.07	4.19	4.12	4.09	4.16	4.16	4.07
	3.8	5.06	4.98	5.08	5.04	5.19	5.10	5.05	5.14	5.14	5.04
	4.4	6.13	6.03	6.15	6.10	6.28	6.15	6.11	6.22	6.22	6.09
	5	7.29	7.18	7.33	7.26	7.47	7.31	7.27	7.40	7.39	7.25
	6	8.56	8.44	8.60	8.52	8.77	8.56	8.53	8.68	8.67	8.50
	7	9.94	9.81	9.99	9.90	10.17	9.93	9.91	10.07	10.06	9.88
	8	11.45	11.30	11.50	11.40	11.71	11.43	11.42	11.59	11.58	11.38
	9	13.11	12.95	13.17	13.04	13.38	13.06	13.07	13.26	13.25	13.03
	10	14.93	14.74	14.99	14.85	15.21	14.85	14.88	15.08	15.09	14.83
	11	16.93	16.71	16.99	16.82	17.22	16.81	16.86	17.07	17.10	16.82
	13	19.13	18.86	19.20	19.00	19.42	18.96	19.02	19.26	19.31	18.99
	15	21.56	21.23	21.64	21.40	21.85	21.32	21.39	21.68	21.74	21.37
	17	24.23	23.83	24.34	24.04	24.53	23.92	23.98	24.33	24.42	23.98
	20	27.17	26.70	27.31	26.96	27.50	26.77	26.84	27.27	27.37	26.85
	23	30.41	29.85	30.60	30.19	30.78	29.92	29.97	30.51	30.61	30.00
	26	33.96	33.33	34.20	33.75	34.40	33.38	33.42	34.08	34.17	33.46
	30	37.82	37.13	38.13	37.63	38.35	37.17	37.21	37.97	38.04	37.23
	35	41.99	41.26	42.35	41.85	42.62	41.28	41.33	42.18	42.22	41.33
	40	46.44	45.68	46.85	46.34	47.18	45.70	45.76	46.66	46.69	45.74
	46	51.13	50.37	51.55	51.08	51.94	50.37	50.46	51.35	51.38	50.40
	52	56.00	55.24	56.38	55.97	56.84	55.22	55.36	56.18	56.23	55.27
	60	60.99	60.21	61.27	60.95	61.76	60.18	60.38	61.08	61.16	60.26
	69	66.00	65.21	66.12	65.92	66.62	65.17	65.43	65.95	66.08	65.29
	79	70.93	70.12	70.87	70.79	71.35	70.07	70.39	70.71	70.90	70.24
	91	75.68	74.86	75.40	75.47	75.85	74.80	75.16	75.28	75.54	75.00
	105	80.13	79.32	79.64	79.86	80.07	79.24	79.64	79.59	79.90	79.48
	120	84.18	83.42	83.51	83.87	83.94	83.31	83.72	83.56	83.90	83.56
	138	87.74	87.07	86.94	87.43	87.44	86.91	87.34	87.11	87.46	87.18
	158	90.76	90.22	89.89	90.49	90.50	90.00	90.42	90.19	90.54	90.27
182	93.24	92.85	92.34	93.03	93.11	92.55	92.98	92.78	93.10	92.83	
209	95.19	94.95	94.30	95.06	95.25	94.59	95.01	94.88	95.16	94.87	
240	96.68	96.58	95.83	96.63	96.92	96.15	96.57	96.51	96.75	96.46	
275	97.79	97.78	96.96	97.78	98.16	97.32	97.72	97.74	97.92	97.63	
316	98.60	98.64	97.77	98.60	99.03	98.16	98.55	98.62	98.75	98.48	
363	99.19	99.22	98.33	99.14	99.57	98.76	99.13	99.23	99.31	99.08	
417	99.59	99.60	98.71	99.49	99.89	99.18	99.52	99.62	99.67	99.47	
479	99.87	99.83	98.96	99.69	99.98	99.48	99.77	99.86	99.88	99.73	
550	99.97	99.96	99.14	99.81	100.00	99.68	99.92	99.97	99.97	99.86	
631	100.00	100.00	99.27	99.87	100.00	99.84	100.00	100.00	100.00	99.94	
724	100.00	100.00	99.38	99.92	100.00	99.93	100.00	100.00	100.00	99.99	
832	100.00	100.00	99.48	99.96	100.00	99.99	100.00	100.00	100.00	100.00	
955	100.00	100.00	99.59	99.98	100.00	100.00	100.00	100.00	100.00	100.00	

Sample Name	Feed 1	Feed 2	Feed 3	Feed 4	Feed 5	Feed 6	Feed 7	Feed 8	Feed 9	Feed 10	Average	Std Dev	% RSD
d(0.800)	104	107	106	105	104	107	106	106	105	106	106	1.028693	0.972323

Appendix G: M4 IsaMill™ Product Particle Size Distribution Data

	μm					
Sample Name	d (0.1)	d (0.2)	d (0.5)	d (0.8)	d (0.9)	Average d80 per pass
Ceramic A. 26/10. #1C - Average	3.95	7.53	21.29	42.78	57.83	
Ceramic A. 26/10. #1D - Average	3.63	6.69	18.34	36.98	49.88	
Ceramic A. 26/10. #1E - Average	3.45	6.24	16.91	34.22	46.29	37.99
Ceramic A. 26/10. #2A - Average	3.40	6.06	16.05	32.44	43.89	
Ceramic A. 26/10. #2B - Average	3.35	5.96	15.64	31.13	41.77	
Ceramic A. 26/10. #2C - Average	3.31	5.83	15.21	30.44	40.90	
Ceramic A. 26/10. #2D - Average	3.30	5.78	15.13	30.30	40.56	
Ceramic A. 26/10. #2E - Average	3.22	5.62	14.49	28.96	38.97	30.65
Ceramic A. 26/10. #3A - Average	2.99	5.00	12.34	24.53	33.09	
Ceramic A. 26/10. #3B - Average	2.98	4.97	12.20	24.23	32.60	
Ceramic A. 26/10. #3C - Average	2.97	4.96	12.17	24.23	32.85	
Ceramic A. 26/10. #3D - Average	2.96	4.92	11.98	23.76	32.02	
Ceramic A. 26/10. #3E - Average	2.93	4.85	11.88	23.64	31.87	24.08
Ceramic A. 26/10. #4A - Average	2.78	4.46	10.51	20.82	28.25	
Ceramic A. 26/10. #4B - Average	2.75	4.39	10.39	20.54	27.75	
Ceramic A. 26/10. #4C - Average	2.78	4.45	10.46	20.82	28.50	
Ceramic A. 26/10. #4D - Average	2.73	4.35	10.29	20.19	27.06	
Ceramic A. 26/10. #4E - Average	2.74	4.36	10.24	20.16	27.22	20.51

	μm					
Sample Name	d (0.1)	d (0.2)	d (0.5)	d (0.8)	d (0.9)	Average d80 per pass
Ceramic B. #1C - Average	3.78	7.10	20.20	40.66	54.62	
Ceramic B. #1D - Average	3.54	6.49	17.81	35.41	47.27	
Ceramic B. #1E - Average	3.40	6.14	16.33	32.48	43.59	36.18
Ceramic B. #2A - Average	3.30	5.81	15.03	29.71	39.72	
Ceramic B. #2B - Average	3.27	5.75	14.79	29.14	38.89	
Ceramic B. #2C - Average	3.23	5.64	14.37	28.31	37.82	
Ceramic B. #2D - Average	3.20	5.58	14.24	28.51	38.95	
Ceramic B. #2E - Average	3.20	5.57	14.12	28.52	39.66	28.84
Ceramic B. #3A - Average	2.94	4.88	11.79	23.07	30.93	
Ceramic B. #3B - Average	2.90	4.79	11.54	22.56	30.24	
Ceramic B. #3C - Average	2.89	4.76	11.44	22.37	30.03	
Ceramic B. #3D - Average	2.88	4.72	11.34	22.23	29.85	
Ceramic B. #3E - Average	2.87	4.70	11.31	22.15	29.66	22.48
Ceramic B. #4A - Average	2.71	4.29	10.00	19.56	26.29	
Ceramic B. #4B - Average	2.68	4.25	9.85	19.08	25.54	
Ceramic B. #4C - Average	2.70	4.27	9.89	19.38	26.23	
Ceramic B. #4D - Average	2.72	4.31	10.02	19.96	27.62	
Ceramic B. #4E - Average	2.69	4.23	9.74	18.96	25.54	19.39

Sample Name	μm					Average d80 per pass
	<u>d</u> <u>(0.1)</u>	<u>d</u> <u>(0.2)</u>	<u>d</u> <u>(0.5)</u>	<u>d</u> <u>(0.8)</u>	<u>d</u> <u>(0.9)</u>	
Ceramic C (1L) #1C - Average	4.60	9.41	30.52	66.67	92.39	
Ceramic C (1L) #1D - Average	4.14	8.15	26.16	56.60	78.08	
Ceramic C (1L) #1E - Average	3.97	7.73	24.28	51.51	70.40	58.26
Ceramic C (1L) #2A - Average	4.44	8.92	26.97	54.27	73.04	
Ceramic C (1L) #2B - Average	4.25	8.44	25.79	52.53	70.84	
Ceramic C (1L) #2C - Average	4.01	7.86	23.94	48.82	65.68	
Ceramic C (1L) #2D - Average	3.78	7.23	21.78	44.36	59.57	
Ceramic C (1L) #2E - Average	3.58	6.69	19.65	40.16	53.91	48.03
Ceramic C (1L) #3A - Average	3.65	6.81	18.73	35.93	46.82	
Ceramic C (1L) #3B - Average	3.64	6.80	18.78	36.68	48.45	
Ceramic C (1L) #3C - Average	3.53	6.53	18.17	35.53	46.84	
Ceramic C (1L) #3D - Average	3.47	6.40	17.57	34.46	45.73	
Ceramic C (1L) #3E - Average	3.36	6.08	16.59	32.50	42.95	35.02
Ceramic C (1L) #4A - Average	3.40	6.16	15.99	30.62	40.17	
Ceramic C (1L) #4B - Average	3.37	6.09	15.72	29.99	39.36	
Ceramic C (1L) #4C - Average	3.33	5.99	15.46	29.72	39.08	
Ceramic C (1L) #4D - Average	3.24	5.79	15.05	28.86	37.93	
Ceramic C (1L) #4E - Average	3.18	5.64	14.59	28.17	37.18	29.47

Sample Name	μm					Average d80 per pass
	<u>d</u> <u>(0.1)</u>	<u>d</u> <u>(0.2)</u>	<u>d</u> <u>(0.5)</u>	<u>d</u> <u>(0.8)</u>	<u>d</u> <u>(0.9)</u>	
Ceramic C. 1.5L. #1C - Average	4.13	8.01	24.47	51.23	69.97	
Ceramic C. 1.5L. #1D - Average	4.06	7.84	22.85	45.77	61.31	
Ceramic C. 1.5L. #1E - Average	3.83	7.34	20.84	40.97	54.57	45.99
Ceramic C. 1.5L. #2A - Average	3.76	7.05	19.36	38.16	50.85	
Ceramic C. 1.5L. #2B - Average	3.77	7.06	18.94	36.81	48.70	
Ceramic C. 1.5L. #2C - Average	3.69	6.91	18.56	36.10	47.76	
Ceramic C. 1.5L. #2D - Average	3.59	6.56	17.39	33.90	44.85	
Ceramic C. 1.5L. #2E - Average	3.51	6.39	16.90	32.88	43.52	35.57
Ceramic C. 1.5L. #3A - Average	3.36	5.93	14.86	28.83	38.48	
Ceramic C. 1.5L. #3B - Average	3.28	5.79	14.54	27.94	36.90	
Ceramic C. 1.5. #3C - Average	3.27	5.74	14.31	27.53	36.41	
Ceramic C. 1.5L. #3D - Average	3.22	5.63	14.00	26.94	35.67	
Ceramic C. 1.5L. #3E - Average	3.21	5.57	13.93	26.98	35.75	27.64
Ceramic C. 1.5L. #4A - Average	3.07	5.20	12.45	23.95	31.84	
Ceramic C. 1.5L. #4B - Average	3.04	5.12	12.26	23.53	31.24	
Ceramic C. 1.5L. #4C - Average	3.01	5.06	12.10	23.27	30.90	
Ceramic C. 1.5L. #4D - Average	2.98	4.99	11.99	23.05	30.62	
Ceramic C. 1.5L. #4E - Average	2.98	4.99	11.91	22.88	30.39	23.33

Sample Name	μm					Average d80 per pass
	<u>d</u> <u>(0.1)</u>	<u>d</u> <u>(0.2)</u>	<u>d</u> <u>(0.5)</u>	<u>d</u> <u>(0.8)</u>	<u>d</u> <u>(0.9)</u>	
Ceramic C. 2L. #1C - Average	4.11	7.97	22.45	44.54	60.00	
Ceramic C. 2L. #1D - Average	3.69	6.90	19.14	38.18	51.37	
Ceramic C. 2L. #1E - Average	3.66	6.82	18.66	37.45	50.56	40.06
Ceramic C. 2L. #2A - Average	3.85	7.17	18.81	36.73	48.94	
Ceramic C. 2L. #2B - Average	3.71	6.84	17.97	34.87	46.17	
Ceramic C. 2L. #2C - Average	3.82	7.15	18.45	35.61	47.26	
Ceramic C. 2L. #2D - Average	3.51	6.35	16.39	31.99	42.54	
Ceramic C. 2L. #2E - Average	3.46	6.18	15.89	31.24	41.66	34.09
Ceramic C. 2L. #3A - Average	3.25	5.61	13.83	26.81	35.71	
Ceramic C. 2L. #3B - Average	3.23	5.55	13.50	26.26	35.04	
Ceramic C. 2L. #3C - Average	3.21	5.48	13.39	26.07	34.76	
Ceramic C. 2L. #3D - Average	3.20	5.46	13.21	25.58	34.04	
Ceramic C. 2L. #3E - Average	3.15	5.34	13.07	25.45	33.82	26.04
Ceramic C. 2L. #4A - Average	2.97	4.88	11.51	22.35	29.87	
Ceramic C. 2L. #4B - Average	2.94	4.82	11.36	22.02	29.43	
Ceramic C. 2L. #4C - Average	2.94	4.83	11.38	21.89	29.11	
Ceramic C. 2L. #4D - Average	2.91	4.74	11.12	21.57	28.84	
Ceramic C. 2L. #4E - Average	2.92	4.74	11.10	21.49	28.70	21.86

Sample Name	μm					Average d80 per pass
	<u>d</u> <u>(0.1)</u>	<u>d</u> <u>(0.2)</u>	<u>d</u> <u>(0.5)</u>	<u>d</u> <u>(0.8)</u>	<u>d</u> <u>(0.9)</u>	
Ceramic D. #1C - Average	3.93	7.47	21.49	43.48	58.69	
Ceramic D. #1D - Average	3.64	6.76	18.70	37.26	49.97	
Ceramic D. #1E - Average	3.51	6.42	17.23	33.94	45.16	38.22
Ceramic D. #2A - Average	3.33	5.88	14.97	29.41	39.24	
Ceramic D. #2B - Average	3.32	5.82	14.78	29.05	38.97	
Ceramic D. #2C - Average	3.23	5.66	14.29	27.92	37.19	
Ceramic D. #2D - Average	3.13	5.41	13.68	26.86	35.84	
Ceramic D. #2E - Average	3.12	5.37	13.58	26.74	35.69	27.99
Ceramic D. #3A - Average	2.93	4.85	11.62	22.60	30.24	
Ceramic D. #3B - Average	2.91	4.77	11.36	22.19	29.71	
Ceramic D. #3C - Average	2.89	4.74	11.27	22.02	29.54	
Ceramic D. #3D - Average	2.86	4.67	11.12	21.75	29.19	
Ceramic D. #3E - Average	2.87	4.70	11.13	21.70	29.13	22.05
Ceramic D. #4A - Average	2.67	4.19	9.68	18.76	25.08	
Ceramic D. #4B - Average	2.67	4.21	9.67	18.77	25.20	
Ceramic D. #4C - Average	2.67	4.19	9.60	18.68	25.15	
Ceramic D. #4D - Average	2.66	4.18	9.57	18.52	24.82	
Ceramic D. #4E - Average	2.65	4.15	9.52	18.48	24.82	18.64

<u>Sample Name</u>	<u>μm</u>					<u>Average d80 per pass</u>
	<u>d (0.1)</u>	<u>d (0.2)</u>	<u>d (0.5)</u>	<u>d (0.8)</u>	<u>d (0.9)</u>	
Ceramic E. #1C - Average	4.088	7.941	22.774	46.37	63.225	
Ceramic E. #1D - Average	3.887	7.388	21.147	43.146	58.731	
Ceramic E. #1E - Average	3.605	6.683	18.752	38.109	51.607	42.54
Ceramic E. #2A - Average	3.964	7.546	21.128	42.586	57.848	
Ceramic E. #2B - Average	3.814	7.174	19.552	38.575	51.538	
Ceramic E. #2C - Average	3.824	7.17	19.343	38.903	52.543	
Ceramic E. #2D - Average	3.665	6.809	18.433	36.223	48.294	
Ceramic E. #2E - Average	3.552	6.516	17.341	35.106	47.754	38.28
Ceramic E. #3A - Average	3.441	6.149	15.89	31.184	41.344	
Ceramic E. #3B - Average	3.402	6.085	15.623	30.513	40.622	
Ceramic E. #3C - Average	3.354	5.976	15.43	30.187	40.135	
Ceramic E. #3D - Average	3.303	5.826	14.953	29.479	39.308	
Ceramic E. #3E - Average	3.283	5.794	14.891	29.5	39.79	30.17
Ceramic E. #4A - Average	3.309	5.693	13.501	25.536	33.987	
Ceramic E. #4B - Average	3.154	5.402	13.319	25.918	34.416	
Ceramic E. #4C - Average	3.16	5.405	13.275	25.773	34.282	
Ceramic E. #4D - Average	3.103	5.284	12.976	25.304	33.715	
Ceramic E. #4E - Average	3.1	5.275	12.899	25.19	33.645	25.54

Appendix H: Preconditioning Test Data

Preconditioning Data - Ceramic A							
Sample	Volume (L)	Mass Before (g)	Mass After (g)	% Loss	Totaliser Data		
					kWh before	kWh after	kWh used
A1	2	4632.4	4618.8	0.29%	2131.6	2133.6	2
A2	2	4638.3	4624.9	0.29%	2133.6	2135.6	2
A3	2	4672.1	4658.1	0.30%	2135.6	2137.5	1.9
A4	2	4704.2	4694.1	0.21%	2137.5	2139.3	1.8
A5	2	4695	4684.9	0.22%	2139.3	2141.1	1.8

	Average	Stdev
Power Draw	1.900	0.1
Wear Loss	0.26%	0.00044

Instantaneous Measurements - Ceramic A Preconditioning															
Min	A1			A2			A3			A4			A5		
	kW	Temp	RPM	kW	Temp	RPM	kW	Temp	RPM	kW	Temp	RPM	kW	Temp	RPM
0	2.20	30.0	2101	2.20	28.0	2098	2.20	34.0	2102	2.10	40.0	2102	2.10	41.0	2102
5	2.00	35.5	2102	2.10	32.5	2099	2.00	41.0	2103	1.90	46.0	2103	2.00	47.5	2104
10	2.00	37.0	2103	2.00	33.5	2100	1.90	42.5	2090	1.90	46.0	2104	1.90	49.0	2087
15	2.00	38.0	2104	2.10	35.0	2101	1.90	42.5	2087	1.80	46.0	2087	1.80	49.0	2088
20	2.00	39.0	2104	2.00	36.0	2101	2.00	43.5	2104	1.80	46.0	2087	1.80	49.0	2088
25	2.00	40.0	2105	2.00	37.0	2102	2.00	44.5	2104	1.80	46.0	2088	1.80	49.5	2088
30	2.00	41.0	2105	2.00	38.0	2102	2.00	45.0	2104	1.80	47.0	2088	1.80	50.0	2088
35	1.90	42.0	2105	2.00	39.0	2103	1.90	46.0	2104	1.90	47.0	2088	1.80	50.0	2088
40	1.90	42.0	2105	2.00	40.0	2103	2.00	46.5	2104	1.80	47.5	2088	1.80	50.0	2089
45	1.90	43.0	2105	2.00	40.5	2104	1.90	47.0	2104	1.80	48.0	2088	1.90	50.0	2089
50	1.90	44.0	2105	2.00	41.0	2104	1.93	47.0	2104	1.80	48.5	2088	1.80	50.0	2089
55	1.90	45.0	2106	2.00	42.0	2104	1.93	48.0	2104	1.80	48.5	2088	1.80	50.0	2089
60	1.80	45.5	2106	1.90	42.5	2104	1.93	48.0	2104	1.80	49.0	2088	1.80	51.0	2089
Ave	1.96	40.2	2104	2.02	37.3	2102	1.97	44.3	2101	1.85	46.6	2091	1.85	48.9	2091

Preconditioning Data - Ceramic B							
Sample	Volume (L)	Mass Before (g)	Mass After (g)	% Loss	Totaliser Data		
					kWh before	kWh after	kWh used
B1	2	4785.2	4748.1	0.78%	2233.3	2235.4	2.1
B2	2	4646.8	4603.8	0.93%	2235.4	2237.4	2
B3	2	4579.6	4535	0.97%	2237.4	2239.4	2
B4	2	4738.4	4709.7	0.61%	2239.4	2241.4	2
B5	2	4505.2	4464.9	0.89%	2241.4	2243.2	1.8

	Average	Stdev
Power Draw	1.980	0.10954
Wear Loss	0.83%	0.00148

Instantaneous Measurements - Ceramic B Preconditioning															
Min	B1			B2			B3			B4			B5		
	kW	Temp	RPM	kW	Temp	RPM	kW	Temp	RPM	kW	Temp	RPM	kW	Temp	RPM
0	2.30	31.0	2100	2.10	33.0	2100	2.10	35.0	2100	2.10	35.0	2100	1.90	38.0	2103
5	2.20	39.0	2101	2.10	38.0	2100	2.00	39.0	2101	2.20	40.0	2101	2.00	42.0	2103
10	2.20	41.0	2101	2.10	43.0	2102	2.00	39.0	2103	2.00	42.0	2103	2.00	46.0	2103
15	2.20	41.0	2102	2.10	43.0	2102	2.00	41.0	2103	2.00	42.0	2103	1.90	48.0	2104
20	2.10	42.0	2102	2.10	44.0	2103	2.00	41.0	2104	2.00	44.0	2103	2.00	48.0	2105
25	2.10	42.0	2103	2.10	44.0	2103	2.00	42.0	2104	2.00	47.0	2103	1.90	48.0	2105
30	2.10	43.0	2103	2.10	45.0	2103	2.00	42.0	2104	2.00	47.0	2104	1.90	47.0	2105
35	2.10	43.0	2103	2.10	45.0	2103	2.00	43.0	2104	2.00	48.0	2104	1.90	47.0	2105
40	2.10	43.0	2103	2.00	45.0	2103	2.00	43.0	2104	2.00	48.0	2105	1.90	47.0	2105
45	2.10	43.0	2103	2.10	45.0	2103	2.00	43.0	2104	2.00	48.0	2105	1.90	47.0	2105
50	2.10	44.0	2103	2.10	45.0	2103	2.00	44.0	2104	2.00	48.0	2105	1.90	47.0	2105
55	2.10	44.0	2103	2.10	44.0	2103	2.00	44.0	2104	2.00	48.0	2105	1.90	47.0	2105
60	2.10	44.0	2103	2.10	43.0	2103	2.00	45.0	2104	2.00	48.0	2105	1.90	47.0	2105
Ave	2.14	41.5	2102	2.09	42.8	2102	2.01	41.6	2103	2.02	45.0	2104	1.92	46.1	2104

Preconditioning Data - Ceramic C							
Sample	Volume (L)	Mass Before (g)	Mass After (g)	% Loss	Totaliser Data		
					kWh before	kWh after	kWh used
C1	2	4973	4969.4	0.07%	2141.1	2142.9	1.8
C2	2	4942.5	4938.8	0.07%	2142.9	2144.7	1.8
C3	2	4892.9	4889.9	0.06%	2215.6	2217.5	1.9
C4	2	4925.7	4923.3	0.05%	2217.5	2219.3	1.8
C5	2	4897.9	4895.1	0.06%	2219.3	2221.1	1.8

	Average	Stdev
Power Draw	1.820	0.04472
Wear Loss	0.06%	0.00011

Instantaneous Measurements - Ceramic C Preconditioning															
Min	C1			C2			C3			C4			C5		
	kW	Temp	RPM	kW	Temp	RPM	kW	Temp	RPM	kW	Temp	RPM	kW	Temp	RPM
0	1.90	39.0	2102	1.80	41.0	2087	2.00	29.0	2100	2.00	31.0	2101	1.90	34.0	2101
5	1.80	42.0	2087	1.80	47.0	2088	2.00	30.0	2101	1.90	35.0	2102	1.90	38.0	2102
10	1.80	42.5	2088	1.80	48.0	2089	2.00	32.0	2101	1.90	35.0	2103	1.80	40.0	2102
15	1.80	43.0	2088	1.80	48.0	2089	1.90	32.0	2102	1.90	36.0	2103	1.80	40.0	2102
20	1.80	43.5	2088	1.80	48.5	2089	1.90	32.0	2102	1.80	38.0	2103	1.80	41.0	2103
25	1.80	44.0	2089	1.80	49.0	2089	1.90	33.0	2103	1.80	38.0	2103	1.80	41.0	2103
30	1.80	44.5	2089	1.70	49.0	2089	1.90	33.0	2103	1.80	38.0	2103	1.80	42.0	2104
35	1.80	45.0	2089	1.80	50.0	2090	1.90	34.0	2103	1.80	38.0	2103	1.80	42.0	2104
40	1.80	45.5	2089	1.70	50.0	2089	1.90	34.0	2103	1.80	39.0	2103	1.80	42.0	2104
45	1.70	46.0	2089	1.80	50.0	2089	1.80	35.0	2103	1.80	39.0	2103	1.80	42.0	2104
50	1.80	46.5	2089	1.80	50.0	2090	1.90	35.0	2104	1.80	39.0	2104	1.80	42.0	2104
55	1.80	47.0	2089	1.70	50.0	2090	1.90	36.0	2104	1.80	40.0	2104	1.80	42.0	2104
60	1.70	47.0	2089	1.80	51.0	2090	1.90	36.0	2104	1.80	40.0	2104	1.80	42.0	2104
Ave	1.79	44.3	2090	1.78	48.6	2089	1.92	33.2	2103	1.84	37.4	2103	1.82	40.6	2103

Preconditioning Data - Ceramic D							
Sample	Volume (L)	Mass Before (g)	Mass After (g)	% Loss	Totaliser Data		
					kWh before	kWh after	kWh used
D1	2	3862.9	3839.2	0.006135	2221.1	2223.1	2
D2	2	3875.6	3866.9	0.002245	2223.1	2225.2	2.1
D3	2	3915.3	3905.4	0.002529	2225.2	2227.2	2
D4	2	3914.3	3904.7	0.002453	2227.2	2229.2	2
D5	2	3903	3893.4	0.00246	2229.2	2231.2	2
D6	2	3884.4	3861.7	0.005844	2231.2	2233.3	2.1

	Average	Stdev
Power Draw	2.020	0.04472
Wear Loss	0.32%	0.00166

Instantaneous Measurements - Ceramic D Preconditioning																		
Min	D1			D2			D3			D4			D5			D6		
	kW	Temp	RPM	kW	Temp	RPM	kW	Temp	RPM	kW	Temp	RPM	kW	Temp	RPM	kW	Temp	RPM
0	2.10	31.0	2100	2.30	25.0	2100	2.10	31.0	2100	2.10	38.0	2100	2.10	41.0	2100	2.10	25.0	2100
5	2.10	40.0	2102	2.20	33.0	2100	2.10	36.0	2100	2.10	41.0	2102	2.10	44.0	2102	2.10	30.0	2100
10	2.00	40.0	2103	2.10	34.0	2101	2.10	36.0	2100	2.10	41.0	2102	2.10	44.0	2102	2.20	32.0	2100
15	1.90	41.0	2104	2.10	35.0	2101	2.10	37.0	2102	2.10	41.0	2102	2.00	45.0	2104	2.10	34.0	2100
20	1.90	41.0	2104	2.10	36.0	2101	2.10	38.0	2103	2.10	42.0	2103	2.00	45.0	2104	2.10	37.0	2100
25	2.00	42.0	2104	2.10	36.0	2102	2.10	38.0	2103	2.10	42.0	2103	2.00	45.0	2104	2.10	38.0	2101
30	2.00	42.0	2104	2.10	36.0	2103	2.10	39.0	2103	2.00	43.0	2103	2.00	45.0	2104	2.10	39.0	2101
35	2.00	43.0	2105	2.10	36.0	2103	2.10	39.0	2103	2.00	43.0	2103	2.00	45.0	2104	2.10	39.0	2102
40	2.00	43.0	2105	2.10	36.0	2103	2.00	40.0	2103	2.00	43.0	2103	2.00	45.0	2104	2.10	39.0	2102
45	2.00	43.0	2105	2.10	36.0	2103	2.00	40.0	2103	2.00	44.0	2103	2.00	46.0	2105	2.10	40.0	2102
50	2.00	44.0	2105	2.10	37.0	2103	2.00	41.0	2103	2.00	44.0	2103	2.00	46.0	2105	2.10	40.0	2103
55	2.00	44.0	2105	2.00	38.0	2103	2.00	41.0	2103	2.00	44.0	2103	2.00	46.0	2105	2.10	40.0	2103
60	2.00	44.0	2105	2.00	38.0	2103	2.00	41.0	2103	2.00	45.0	2104	2.00	46.0	2105	2.10	41.0	2103
Ave	2.00	41.4	2104	2.11	35.1	2102	2.06	38.2	2102	2.05	42.4	2103	2.02	44.8	2104	2.11	36.5	2101

Preconditioning Data - Ceramic E							
Sample	Volume (L)	Mass Before (g)	Mass After (g)	% Loss	Totaliser Data		
					kWh before	kWh after	kWh used
E1	2	8102.4	8101.4	0.000123	2317.1	2319	1.9
E2	2	8106.2	8104.6	0.000197	2319	2320.9	1.9
E3	2	8102.6	8101	0.000197	2320.9	2322.8	1.9
E4	2	8109.2	8106.6	0.000321	2322.8	2324.7	1.9
E5	2	8111	8108.8	0.000271	2324.7	2326.6	1.9

	Average	Stdev
Power Draw	1.900	2E-13
Wear Loss	0.02%	0.00008

Instantaneous Measurements - Ceramic E Preconditioning															
Min	E1			E2			E3			E4			E5		
	kW	Temp	RPM	kW	Temp	RPM	kW	Temp	RPM	kW	Temp	RPM	kW	Temp	RPM
0	2.3	30	2100	2	30	2101	2.1	33	2100	2.3	34	2100	2.1	31	2100
5	2.1	35	2100	2.1	37	2102	2	41	2101	2.1	38	2101	2.1	39	2101
10	2	36	2101	2.1	39	2102	1.9	41	2101	2.1	38	2101	2.1	40	2101
15	1.9	36	2101	1.9	40	2103	1.9	37	2101	1.9	38	2102	2.1	40	2102
20	1.9	37	2101	1.9	41	2103	1.9	37	2101	1.9	38	2102	2.1	41	2102
25	1.9	37	2102	1.9	41	2104	1.9	38	2101	1.9	39	2102	1.9	42	2102
30	1.9	37	2102	1.9	41	2104	1.9	38	2101	1.9	39	2102	1.9	42	2102
35	1.9	38	2102	1.9	41	2104	1.9	38	2101	1.9	39	2102	1.8	43	2102
40	1.9	38	2102	1.9	41	2104	1.9	39	2102	1.9	41	2102	1.8	43	2102
45	1.9	38	2103	1.9	43	2105	1.9	39	2102	1.9	41	2102	1.8	43	2102
50	1.9	39	2103	1.9	43	2105	1.9	39	2102	1.9	41	2102	1.8	43	2102
55	1.9	39	2103	1.9	43	2105	1.9	39	2102	1.9	41	2102	1.8	44	2103
60	1.9	39	2104	1.9	43	2105	1.9	39	2102	1.9	41	2102	1.8	44	2103
Ave	1.95	36.8	2102	1.94	40.2	2104	1.92	38.3	2101	1.96	39.1	2102	1.93	41.2	2102

Preconditioning Data - Silica Sand							
Sample	Volume (L)	Mass Before (g)	Mass After (g)	% Loss	Totaliser Data		
					kWh before	kWh after	kWh used
S1	2	3079.8	2210.1	28.24%	2415.2	2417.3	2.1
S2	2	3119.4	2178	30.18%	2417.3	2419.3	2
S3	2	3078.1	2158.4	29.88%	2419.3	2421.3	2
S4	2	3078.5	2226.2	27.69%	2421.3	2423.1	1.8
S5	2	3071.2	2163.3	29.56%	2423.1	2425.2	2.1
S6	2	3077.8	2187.2	28.94%	2425.2	2427.3	2.1
S7	2	3109.3	2191.6	29.51%	2413.2	2415.2	2
S8	2	3113.7	2193.7	29.55%	2427.3	2429.4	2.1

	Average	Stdev
Power Draw	2.025	0.10351
Wear Loss	29.19%	0.00850

Instantaneous Measurements - Silica Sand															
Min	S1			S2			S3			S4			S5		
	kW	Temp	RPM	kW	Temp	RPM	kW	Temp	RPM	kW	Temp	RPM	kW	Temp	RPM
0	1.8	24	2100	2	31	2100	2	26	2100	2.1	36	2100	2.2	24	2100
5	2.2	29	2100	2.3	33	2102	2.1	37	2102	2.1	42	2103	2.4	30	2100
10	2.3	32	2100	2.3	36	2103	2.1	39	2103	2.2	44	2104	2.3	31	2100
15	2.3	32	2100	2.3	36	2103	2.2	41	2104	2.2	44	2104	2.2	31	2101
20	2.2	32	2101	2.1	38	2104	2.2	42	2105	2	45	2104	2.1	32	2101
25	2.2	32	2101	2.1	38	2104	2.2	43	2105	2	45	2104	2.1	32	2102
30	2.2	32	2101	2.1	39	2105	2	43	2105	2	45	2104	2.1	32	2102
35	2.1	34	2103	2	39	2105	2	43	2105	1.9	46	2104	2	33	2103
40	2.1	34	2103	2	39	2105	2	44	2105	2	46	2104	2	33	2103
45	2.1	34	2103	2	30	2105	2	44	2106	2	46	2104	2	33	2103
50	2.1	35	2104	1.9	33	2105	1.9	44	2106	1.9	46	2104	2	34	2103
55	2	35	2104	1.9	37	2105	1.9	44	2106	1.9	47	2104	2	35	2104
60	2	35	2104	1.9	40	2105	1.9	44	2106	1.9	47	2104	2	36	2104
Ave	2.12	32.3	2102	2.07	36.1	2104	2.04	41.1	2104	2.02	44.5	2104	2.11	32.0	2102

Instantaneous Measurements - Silica Sand									
Min	S6			S7			S8		
	kW	Temp	RPM	kW	Temp	RPM	kW	Temp	RPM
0	2.2	30	2100	1.5	27	2100	2.2	30	2101
5	2.3	35	2103	2.2	31	2100	2.3	38	2103
10	2.3	35	2103	2.1	34	2102	2.3	39	2104
15	2.2	35	2104	2.1	36	2103	2.3	40	2104
20	2.1	35	2104	2.1	36	2103	2.1	40	2104
25	2.1	36	2105	2.1	36	2103	2.1	40	2104
30	2.1	36	2105	2.1	36	2103	2.1	40	2104
35	2.1	36	2105	2	36	2104	2.1	40	2105
40	2	36	2105	2	37	2104	2	40	2105
45	2	37	2105	2	37	2104	1.9	40	2105
50	2	37	2105	2	37	2105	1.9	40	2105
55	1.9	37	2105	1.9	37	2105	1.9	41	2105
60	1.9	37	2105	1.9	37	2105	1.9	41	2105
Ave	2.09	35.5	2104	2.00	35.2	2103	2.08	39.2	2104

Appendix I: Water Test Data

Water Model Development Data

<u>Media Name</u>	<u>Aspect Ratio</u>	<u>Bead Volume (cm³)</u>	<u>Bead Surface Area (cm²)</u>	<u>Bead Mass (g)</u>	<u>Media Vol (l)</u>	<u>Media Mass (g)</u>	<u>No of Beads</u>	<u>Total Surface Area (cm²)</u>	<u>Water Power Draw (kW)</u>	<u>FC_{water}* SA</u>	<u>FC_{water}*SA* AR</u>	<u>Predicted Power</u>	<u>% Power Difference</u>
Ceramic A	1.107	0.180	0.046	0.677	2	4640.0	6858.8	313.87	2.16	56	62.6	2.08	-4.0
Ceramic A	1.107	0.180	0.046	0.677	1.7	3961.4	5855.7	267.97	1.85	48	53.4	1.93	4.3
Ceramic A	1.107	0.180	0.046	0.677	1.5	3479.9	5144.0	235.40	1.90	42	46.9	1.83	-3.6
Ceramic A	1.107	0.180	0.046	0.677	1.3	3046.2	4502.9	206.06	1.74	37	41.1	1.74	0.1
Ceramic A	1.107	0.180	0.046	0.677	1	2348.1	3470.9	158.84	1.62	29	31.7	1.59	-1.8
Ceramic B	1.170	0.166	0.044	0.649	2	4662.8	7181.0	316.38	2.12	47	55.5	1.97	-7.1
Ceramic B	1.170	0.166	0.044	0.649	1.7	4007.9	6172.4	271.95	1.93	41	47.7	1.84	-4.5
Ceramic B	1.170	0.166	0.044	0.649	1.5	3502.7	5394.4	237.67	1.83	36	41.7	1.75	-4.4
Ceramic B	1.170	0.166	0.044	0.649	1.3	3059.3	4711.5	207.58	1.71	31	36.4	1.67	-2.3
Ceramic B	1.170	0.166	0.044	0.649	1	2347.0	3614.5	159.25	1.63	24	27.9	1.54	-5.8
Ceramic C	1.012	0.156	0.041	0.625	2	5002.7	8004.2	327.31	1.86	46	46.4	1.82	-2.0
Ceramic C	1.012	0.156	0.041	0.625	1.7	4134.8	6615.6	270.52	1.72	38	38.3	1.70	-1.5
Ceramic C	1.012	0.156	0.041	0.625	1.5	3665.8	5865.2	239.84	1.63	34	34.0	1.63	0.0
Ceramic C	1.012	0.156	0.041	0.625	1.3	3190.2	5104.2	208.72	1.58	29	29.6	1.56	-1.5
Ceramic C	1.012	0.156	0.041	0.625	1	2407	3851.1	157.48	1.51	22	22.3	1.45	-4.0
Ceramic D	1.013	0.146	0.039	0.469	2	3891.6	8303.7	326.19	2.11	62	62.8	2.08	-1.3
Ceramic D	1.013	0.146	0.039	0.469	1.7	3281.5	7001.9	275.06	1.86	52	52.9	1.93	3.5
Ceramic D	1.013	0.146	0.039	0.469	1.5	2882.8	6151.2	241.64	1.82	46	46.5	1.83	0.6
Ceramic D	1.013	0.146	0.039	0.469	1.3	2505.6	5346.3	210.02	1.70	40	40.4	1.73	1.8
Ceramic D	1.013	0.146	0.039	0.469	1	1988.4	4242.7	166.67	1.65	32	32.1	1.60	-3.2
Ceramic E	1.058	0.106	0.032	0.657	2	7897.2	12023.6	390.31	1.97	51	53.7	1.94	-1.6
Ceramic E	1.058	0.106	0.032	0.657	1.7	6738.9	10260.1	333.06	1.76	43	45.8	1.81	3.0
Ceramic E	1.058	0.106	0.032	0.657	1.5	5907.8	8994.7	291.98	1.67	38	40.1	1.73	3.4
Ceramic E	1.058	0.106	0.032	0.657	1.3	5114.6	7787.0	252.78	1.60	33	34.8	1.64	2.6
Ceramic E	1.058	0.106	0.032	0.657	1	3939.2	5997.5	194.69	1.55				

Water Model Validation Data

<u>Media Name</u>	<u>Aspect Ratio</u>	<u>Bead Volume (cm³)</u>	<u>Bead Surface Area (cm²)</u>	<u>Bead Mass (g)</u>	<u>Media Vol (l)</u>	<u>Media Mass (g)</u>	<u>No of Beads</u>	<u>Total Surface Area (cm²)</u>	<u>Water Power Draw (kW)</u>	<u>FC_{water} *SA</u>	<u>FC_{water}* SA*AR</u>	<u>Predicted Power</u>	<u>% Power Difference</u>
Ceramic A	1.107	0.180	0.046	0.677	1.93	4500.0	6651.9	304.40	1.94	55	60.7	2.05	5.6
Ceramic B	1.170	0.166	0.044	0.649	1.94	4500.0	6930.3	305.34	1.93	46	53.6	1.94	0.3
Ceramic C	1.012	0.156	0.041	0.625	1.827	4500.0	7199.9	294.42	1.71	41	41.7	1.75	2.4
Ceramic D	1.013	0.146	0.039	0.469	2.3	4500.0	9601.8	377.19	2.09	72	72.6	2.23	6.8
Ceramic E	1.058	0.106	0.032	0.657	1.1	4500.0	6851.3	222.41	1.54	29	30.6	1.58	2.4

Water Test Data - Ceramic A							
Sample	Volume (L)	Mass Before (g)	Mass After (g)	% Loss	Totaliser Data		
					kWh before	kWh after	kWh used
A9	2	4640.0	4613.6	0.57%	2243.2	2245.5	2.3
A10	1.7	3961.4	3946.7	0.37%	2245.5	2247.4	1.9
A11	1.5	3479.9	3467.7	0.35%	2249.1	2251	1.9
A12	1.3	3046.2	3036.8	0.31%	2247.4	2249.1	1.7
A13	1	2348.1	2342.3	0.25%	2251	2252.6	1.6
A14	1.93	4500	4481.1	0.42%	2252.6	2254.6	2

Instantaneous Measurements - Ceramic A Water Test																		
Min	A9			A10			A11			A12			A13			A14		
	kW	Temp	RPM	kW	Temp	RPM	kW	Temp	RPM	kW	Temp	RPM	kW	Temp	RPM	kW	Temp	RPM
0	2.2	28	2096	2	27	2101	2	27	2099	1.7	37	2103	1.7	31	2101	2	36	2101
5	2.3	33	2098	1.9	32	2102	2	29	2100	1.7	38	2104	1.7	32	2101	2	38	2102
10	2.2	34	2099	1.9	34	2102	2	30	2101	1.7	38	2105	1.6	33	2088	2	39	2102
15	2.2	35	2100	1.9	34	2102	1.9	30	2101	1.7	38	2105	1.7	33	2086	2	40	2102
20	2.2	32	2100	1.9	35	2102	1.9	30	2102	1.7	39	2106	1.6	34	2100	1.9	40	2103
25	2.2	32	2101	1.9	35	2102	1.9	31	2102	1.7	39	2106	1.6	34	2088	1.9	40	2103
30	2.1	32	2101	1.8	36	2103	1.9	31	2102	1.7	39	2106	1.6	35	2090	1.9	41	2103
35	2.1	32	2102	1.8	36	2104	1.9	32	2102	1.7	39	2106	1.6	35	2089	2	41	2103
40	2.2	32	2102	1.8	37	2104	1.9	32	2102	1.7	39	2106	1.6	36	2087	1.9	41	2103
45	2.1	32	2103	1.8	38	2105	1.9	32	2102	1.7	39	2106	1.6	36	2086	1.9	42	2103
50	2.1	33	2103	1.8	38	2105	1.8	32	2102	1.9	29	2098	1.6	36	2086	1.9	42	2103
55	2.1	33	2103	1.8	39	2105	1.8	33	2103	1.9	29	2099	1.6	37	2103	1.9	42	2103
60	2.1	33	2103	1.8	39	2105	1.8	34	2103	1.8	29	2100	1.6	37	2088	1.9	42	2103
Ave	2.16	32.4	2101	1.85	35.4	2103	1.90	31.0	2102	1.74	36.3	2104	1.62	34.5	2092	1.94	40.3	2103

Water Test Data - Ceramic B							
Sample	Volume (L)	Mass Before (g)	Mass After (g)	% Loss	Totaliser Data		
					kWh before	kWh after	kWh used
B6	2	4662.8	4639.1	0.51%	2275.5	2277.6	2.1
B7	1.7	4007.9	3995.8	0.30%	2277.6	2279.5	1.9
B8	1.5	3502.7	3488.2	0.41%	2279.5	2281.3	1.8
B9	1.3	3059.3	3051.4	0.26%	2281.3	2283	1.7
B10	1	2347.0	2331.3	0.67%	2283	2284.6	1.6
B11	1.94	4500	4471.2	0.64%	2284.6	2286.5	1.9

Instantaneous Measurements - Ceramic B Water Test																		
Min	B6			B7			B8			B9			B10			B11		
	kW	Temp	RPM	kW	Temp	RPM	kW	Temp	RPM	kW	Temp	RPM	kW	Temp	RPM	kW	Temp	RPM
0	2.3	27	2100	2	35	2101	1.9	34	2100	1.8	40	2101	1.7	37	2101	2.1	37	2101
5	2.1	31	2100	2	37	2102	1.9	40	2101	1.7	42	2101	1.7	37	2101	2	41	2101
10	2.1	34	2100	2.1	39	2102	1.9	40	2101	1.7	44	2102	1.7	39	2101	2	41	2102
15	2.1	37	2100	2.1	39	2103	1.9	41	2101	1.7	44	2102	1.7	39	2102	1.9	41	2102
20	2.1	37	2101	1.9	41	2103	1.8	41	2101	1.7	44	2102	1.6	39	2102	1.9	43	2102
25	2.1	37	2101	1.9	41	2103	1.8	41	2101	1.7	44	2102	1.6	40	2102	1.9	43	2102
30	2.1	37	2102	1.9	41	2103	1.8	42	2101	1.7	44	2103	1.6	40	2102	1.9	45	2104
35	2.1	38	2102	1.9	41	2103	1.8	42	2101	1.7	44	2103	1.6	40	2103	1.9	45	2104
40	2.1	38	2102	1.9	41	2103	1.8	43	2102	1.7	41	2103	1.6	40	2103	1.9	45	2104
45	2.1	38	2103	1.9	41	2103	1.8	43	2102	1.7	41	2103	1.6	41	2103	1.9	45	2104
50	2.1	38	2103	1.9	41	2103	1.8	44	2102	1.7	40	2103	1.6	41	2103	1.9	45	2104
55	2.1	39	2103	1.8	41	2103	1.8	44	2102	1.7	40	2103	1.6	41	2103	1.9	45	2104
60	2.1	39	2103	1.8	42	2103	1.8	44	2102	1.7	40	2103	1.6	41	2103	1.9	45	2104
Ave	2.12	36.2	2102	1.93	40.0	2103	1.83	41.5	2101	1.71	42.2	2102	1.63	39.6	2102	1.93	1.9	2103

Water Test Data - Ceramic C							
Sample	Volume (L)	Mass Before (g)	Mass After (g)	% Loss	Totaliser Data		
					kWh before	kWh after	kWh used
C6	2	5002.7	5001.3	0.03%	2254.6	2256.4	1.8
C7	1.7	4134.8	4134.3	0.01%	2256.4	2258.1	1.7
C8	1.5	3665.8	3665.6	0.01%	2258.1	2259.7	1.6
C9	1.3	3190.2	3189.6	0.02%	2259.7	2261.3	1.6
C10	1	2407.0	2406.6	0.02%	2261.3	2262.8	1.5
C11	1.83	4500	4499.2	0.02%	2262.8	2264.5	1.7

Instantaneous Measurements - Ceramic C Water Test																		
Min	C6			C7			C8			C9			C10			C11		
	kW	Temp	RPM	kW	Temp	RPM	kW	Temp	RPM	kW	Temp	RPM	kW	Temp	RPM	kW	Temp	RPM
0	2	29	2100	1.8	32	2100	1.8	29	2100	1.7	34	2100	1.6	36	2100	1.8	36	2102
5	1.9	31	2100	1.8	36	2101	1.7	37	2101	1.6	39	2102	1.5	39	2102	1.7	41	2103
10	1.9	32	2100	1.8	36	2102	1.7	39	2101	1.6	39	2102	1.5	40	2102	1.7	41	2103
15	1.9	32	2100	1.7	37	2102	1.6	40	2101	1.6	39	2102	1.5	40	2102	1.7	41	2103
20	1.9	33	2101	1.7	37	2102	1.6	40	2102	1.6	40	2102	1.5	40	2103	1.7	41	2103
25	1.9	33	2101	1.7	38	2102	1.6	40	2102	1.6	40	2102	1.5	40	2103	1.7	42	2104
30	1.9	34	2102	1.7	38	2102	1.6	41	2102	1.5	40	2102	1.5	40	2103	1.7	42	2104
35	1.8	34	2102	1.7	39	2102	1.6	41	2102	1.5	40	2103	1.5	40	2103	1.7	42	2104
40	1.8	35	2103	1.7	39	2102	1.6	41	2103	1.6	40	2103	1.5	40	2103	1.7	43	2104
45	1.8	35	2103	1.7	39	2102	1.6	41	2103	1.6	40	2103	1.5	40	2104	1.7	43	2104
50	1.8	36	2104	1.7	40	2102	1.6	41	2103	1.6	41	2103	1.5	40	2104	1.7	43	2105
55	1.8	37	2104	1.7	40	2103	1.6	42	2103	1.6	41	2103	1.5	40	2104	1.7	44	2105
60	1.8	37	2104	1.7	40	2103	1.6	42	2103	1.5	41	2103	1.5	40	2104	1.7	44	2105
Ave	1.86	33.7	2102	1.72	37.8	2102	1.63	39.5	2102	1.58	39.5	2102	1.51	39.6	2103	1.71	1.7	2104

Water Test Data - Ceramic D							
Sample	Volume (L)	Mass Before (g)	Mass After (g)	% Loss	Totaliser Data		
					kWh before	kWh after	kWh used
D7	2	3891.6	3887.1	0.12%	2264.5	2266.6	2.1
D8	1.7	3281.5	3281.3	0.01%	2266.6	2268.4	1.8
D9	1.5	2882.8	2876.2	0.23%	2268.4	2270.2	1.8
D10	1.3	2505.6	2501.1	0.18%	2270.2	2271.9	1.7
D11	1	1988.4	1984.4	0.20%	2271.9	2273.5	1.6
D12	2.31	4500	4492.1	0.18%	2273.5	2275.5	2

Instantaneous Measurements - Ceramic D Water Test																		
Min	D7			D8			D9			D10			D11			D12		
	kW	Temp	RPM	kW	Temp	RPM	kW	Temp	RPM	kW	Temp	RPM	kW	Temp	RPM	kW	Temp	RPM
0	2.2	29	2100	2	32	2101	1.9	38	2101	1.7	39	2101	1.7	34	2101	2.2	39	2101
5	2.2	35	2100	2	39	2102	1.9	41	2101	1.7	40	2102	1.7	37	2101	2.1	42	2102
10	2.1	37	2100	1.9	40	2102	1.8	43	2101	1.7	38	2102	1.7	37	2102	2.1	44	2103
15	2.1	37	2102	1.9	40	2102	1.8	43	2102	1.7	38	2102	1.7	37	2102	2.1	44	2103
20	2.1	39	2102	1.9	41	2102	1.8	43	2102	1.7	38	2103	1.7	39	2102	2.1	44	2103
25	2.1	39	2102	1.9	41	2102	1.8	44	2102	1.7	38	2103	1.7	39	2103	2.1	44	2103
30	2.1	39	2103	1.8	41	2102	1.8	44	2102	1.7	38	2103	1.7	39	2103	2.1	44	2103
35	2.1	40	2103	1.8	41	2102	1.8	44	2103	1.7	38	2103	1.6	40	2103	2.1	44	2103
40	2.1	40	2103	1.8	42	2102	1.8	44	2103	1.7	38	2103	1.6	40	2103	2.1	46	2103
45	2.1	40	2103	1.8	42	2102	1.8	44	2103	1.7	38	2103	1.6	40	2103	2.1	46	2103
50	2.1	40	2103	1.8	42	2102	1.8	44	2103	1.7	38	2103	1.6	41	2103	2.1	46	2104
55	2.1	40	2103	1.8	43	2102	1.8	44	2103	1.7	38	2103	1.6	41	2103	2	46	2104
60	2	41	2103	1.8	43	2102	1.8	44	2103	1.7	38	2103	1.6	41	2103	2	46	2104
Ave	2.11	38.2	2102	1.86	40.5	2102	1.82	43.1	2102	1.70	38.2	2103	1.65	38.8	2102	2.09	2.1	2103

Water Test Data - Ceramic E							
Sample	Volume (L)	Mass Before (g)	Mass After (g)	% Loss	Totaliser Data		
					kWh before	kWh after	kWh used
E6	2	7897.2	7896.7	0.01%	2326.6	2328.5	1.9
E7	1.7	6738.9	6737.7	0.02%	2328.5	2330.2	1.7
E8	1.5	5907.8	5907.6	0.00%	2330.2	2331.9	1.7
E9	1.3	5114.6	5113.6	0.02%	2331.9	2333.5	1.6
E10	1	3939.2	3938.3	0.02%	2333.5	2335	1.5
E11	1.11	4500	4499.8	0.00%	2335	2336.5	1.5

Instantaneous Measurements - Ceramic E Water Test																		
Min	E6			E7			E8			E9			E10			E11		
	kW	Temp	RPM	kW	Temp	RPM	kW	Temp	RPM	kW	Temp	RPM	kW	Temp	RPM	kW	Temp	RPM
0	2.1	26	2100	1.9	32	2101	1.9	31	2100	1.8	32	2101	1.6	33	2101	1.7	38	2102
5	2	33	2100	1.8	38	2103	1.8	38	2101	1.7	37	2101	1.6	39	2102	1.6	43	2102
10	2	35	2100	1.8	38	2103	1.7	40	2101	1.6	39	2101	1.6	39	2102	1.6	43	2103
15	2	35	2100	1.8	39	2104	1.7	40	2102	1.6	41	2102	1.6	41	2103	1.6	43	2103
20	2	36	2101	1.8	39	2104	1.7	41	2102	1.6	41	2102	1.6	41	2103	1.5	43	2104
25	2	37	2101	1.8	39	2104	1.7	41	2103	1.6	43	2102	1.6	42	2104	1.5	45	2104
30	2	37	2101	1.8	40	2104	1.6	41	2103	1.6	43	2103	1.5	43	2104	1.5	45	2104
35	2	38	2101	1.7	40	2104	1.6	41	2103	1.6	43	2103	1.5	43	2104	1.5	45	2104
40	1.9	38	2102	1.7	41	2104	1.6	42	2103	1.6	44	2103	1.5	44	2104	1.5	45	2104
45	1.9	38	2102	1.7	41	2104	1.6	42	2103	1.5	44	2103	1.5	44	2104	1.5	45	2104
50	1.9	38	2103	1.7	41	2104	1.6	42	2103	1.5	44	2103	1.5	44	2104	1.5	45	2104
55	1.9	38	2103	1.7	41	2104	1.6	42	2103	1.6	44	2103	1.5	45	2104	1.5	45	2104
60	1.9	38	2103	1.7	41	2104	1.6	42	2103	1.5	44	2103	1.5	45	2104	1.5	45	2104
Ave	1.97	35.9	2101	1.76	39.2	2104	1.67	40.2	2102	1.60	41.5	2102	1.55	41.8	2103	1.54	1.5	2104

Water Test Data - Silica Sand							
Sample	Volume (L)	Mass Before (g)	Mass After (g)	% Loss	Totaliser Data		
					kWh before	kWh after	kWh used
S9	2	3442.6	2745.2	20.26%	2436.3	2438.5	2.2
S10	1.7	2834.1	2296.5	18.97%	2438.5	2440.5	2
S11	1.5	2516.7	2039.9	18.95%	2440.5	2442.4	1.9
S12	1.3	2178.4	1701.7	21.88%	2442.4	2444.3	1.9
S13	1	1713.0	1291.9	24.58%	2444.3	2446.1	1.8
S14	2.91	4500	3646.5	18.97%	2446.1	2448.6	2.5

Instantaneous Measurements - Ceramic E Water Test																		
Min	S9			S10			S11			S12			S13			S14		
	kW	Temp	RPM	kW	Temp	RPM	kW	Temp	RPM	kW	Temp	RPM	kW	Temp	RPM	kW	Temp	RPM
0	2.4	28	2100	2.2	30	2100	2	35	2100	2.1	25	2100	1.3	32	2100	2.9	26	2100
5	2.4	37	2102	2.1	41	2103	2	38	2100	2	30	2100	1.9	32	2100	2.6	28	2100
10	2.4	40	2102	2	43	2104	1.9	42	2102	2	31	2100	1.9	34	2100	2.6	34	2101
15	2.4	41	2103	2	43	2104	2	42	2102	2	32	2100	1.9	36	2101	2.6	41	2101
20	2.4	41	2103	2	43	2104	1.9	43	2103	2	32	2100	1.9	36	2101	2.6	41	2102
25	2.3	42	2104	2	43	2104	1.9	44	2103	2	32	2100	1.9	36	2102	2.5	41	2102
30	2.3	42	2104	2	44	2104	1.9	44	2104	1.9	34	2100	1.9	37	2102	2.5	41	2103
35	2.3	42	2104	1.9	44	2104	1.9	44	2104	1.9	34	2100	1.9	37	2102	2.5	41	2103
40	2.1	43	2104	1.9	44	2104	1.9	45	2104	1.9	34	2100	1.8	37	2102	2.4	41	2103
45	2.1	43	2104	1.9	44	2105	1.9	45	2105	1.9	35	2100	1.8	38	2102	2.4	41	2103
50	2	43	2105	1.9	44	2105	1.9	45	2105	1.9	35	2101	1.8	38	2103	2.4	35	2103
55	2	43	2105	1.9	44	2105	1.9	46	2105	1.9	36	2101	1.7	38	2103	2.3	35	2103
60	2	43	2105	1.9	45	2105	1.9	46	2105	1.9	36	2101	1.7	39	2103	2.3	36	2103
Ave	2.24	40.6	2103	1.98	42.5	2104	1.92	43.0	2103	1.95	32.8	2100	1.80	36.2	2102	2.51	2.5	2102

Appendix J: Slurry Test Data

Slurry Model Development Data

<u>Media Name</u>	<u>Pass #</u>	<u>F80</u>	<u>P80</u>	<u>Reduction Ratio per pass</u>	<u>% Wear over test</u>	<u>Aspect Ratio</u>	<u>Bead Volume (cm³)</u>	<u>Bead Surface Area (cm²)</u>	<u>Bead Mass (g)</u>	<u>Media Vol (l)</u>	<u>Media Mass (g)</u>	<u>No of Beads</u>	<u>Total Surface Area (cm²)</u>	<u>No Load Power (kW)</u>	<u>Slurry Power Draw (kW)</u>	<u>FC_{slurry}*S A</u>	<u>FC_{slurry}*S A*AR</u>
Ceramic A	Pass 1	106	37.99	2.79	0.83	1.107	0.180	0.046	0.677	2	4613.5	6819.6	312.08	1.10	3.08	103	114.1
Ceramic A	Pass 2	37.99	30.65	1.24	0.83	1.107	0.180	0.046	0.677	2	4613.5	6819.6	312.08	1.10	2.90	103	114.1
Ceramic A	Pass 3	30.65	24.07	1.27	0.83	1.107	0.180	0.046	0.677	2	4613.5	6819.6	312.08	1.10	2.88	103	114.1
Ceramic A	Pass 4	24.07	20.5	1.17	0.83	1.107	0.180	0.046	0.677	2	4613.5	6819.6	312.08	1.10	2.80	103	114.1
Ceramic B	Pass 1	106	36.18	2.93	0.99	1.170	0.166	0.044	0.649	2	4636.2	7140.0	314.58	1.10	3.08	88	103.1
Ceramic B	Pass 2	36.18	28.83	1.25	0.99	1.170	0.166	0.044	0.649	2	4636.2	7140.0	314.58	1.10	2.77	88	103.1
Ceramic B	Pass 3	28.83	22.47	1.28	0.99	1.170	0.166	0.044	0.649	2	4636.2	7140.0	314.58	1.10	2.60	88	103.1
Ceramic B	Pass 4	22.47	19.385	1.16	0.99	1.170	0.166	0.044	0.649	2	4636.2	7140.0	314.58	1.10	2.60	88	103.1
Ceramic C	Pass 1	106	40.06	2.65	0.24	1.012	0.156	0.041	0.625	2	4899.6	7839.2	320.56	1.10	2.82	96	97.3
Ceramic C	Pass 2	40.06	34.09	1.18	0.24	1.012	0.156	0.041	0.625	2	4899.6	7839.2	320.56	1.10	2.58	96	97.3
Ceramic C	Pass 3	34.09	26.04	1.31	0.24	1.012	0.156	0.041	0.625	2	4899.6	7839.2	320.56	1.10	2.50	96	97.3
Ceramic C	Pass 4	26.04	21.86	1.19	0.24	1.012	0.156	0.041	0.625	2	4899.6	7839.2	320.56	1.10	2.48	96	97.3
Ceramic D	Pass 1	106	38.22	2.77	0.84	1.013	0.146	0.039	0.469	2	3883.7	8286.8	325.53	1.10	2.93	107	108.8
Ceramic D	Pass 2	38.22	27.99	1.37	0.84	1.013	0.146	0.039	0.469	2	3883.7	8286.8	325.53	1.10	2.65	107	108.8
Ceramic D	Pass 3	27.99	22.05	1.27	0.84	1.013	0.146	0.039	0.469	2	3883.7	8286.8	325.53	1.10	2.55	107	108.8
Ceramic D	Pass 4	22.05	18.64	1.18	0.84	1.013	0.146	0.039	0.469	2	3883.7	8286.8	325.53	1.10	2.50	107	108.8
Ceramic E	Pass 1	106	42.54	2.49	0.13	1.058	0.106	0.032	0.657	2	7895.9	12021.6	390.24	1.10	3.00	94	99.1
Ceramic E	Pass 2	42.54	38.28	1.11	0.13	1.058	0.106	0.032	0.657	2	7895.9	12021.6	390.24	1.10	2.50	94	99.1
Ceramic E	Pass 3	38.28	30.17	1.27	0.13	1.058	0.106	0.032	0.657	2	7895.9	12021.6	390.24	1.10	2.40	94	99.1
Ceramic E	Pass 4	30.17	25.54	1.18	0.13	1.058	0.106	0.032	0.657	2	7895.9	12021.6	390.24	1.10	2.40	94	99.1

<u>Media Name</u>	<u>Slurry Model 1</u>		<u>1/P80</u>	<u>N*P80</u>	<u>SI_N</u>	<u>SN*SI</u>	<u>New Slurry Model</u>	
	<u>Predicted Power</u>	<u>Predicted Power Error</u>					<u>Predicted Power</u>	<u>Error</u>
Ceramic A	2.88	-7.090162	0.009434	64.33616205	0.170622349	10.97718708	3.12	1.03
Ceramic A	2.88	-0.722639	0.026323	179.5112708	0.170622349	30.62863465	2.88	-0.82
Ceramic A	2.88	-0.028	0.032626	222.5002668	0.170622349	37.96351812	2.79	-3.34
Ceramic A	2.88	2.7505555	0.041545	283.3250177	0.170622349	48.34157998	2.66	-5.23
Ceramic B	2.71	-13.87634	0.009434	67.3586377	0.16922684	11.39888943	2.92	-5.44
Ceramic B	2.71	-2.180928	0.02764	197.3470314	0.16922684	33.39641458	2.70	-2.58
Ceramic B	2.71	3.9745495	0.034686	247.6592298	0.16922684	41.91058895	2.61	0.35
Ceramic B	2.71	3.9745495	0.044504	317.7577034	0.16922684	53.77313215	2.49	-4.56
Ceramic C	2.62	-7.577924	0.009434	73.95498307	0.150112321	11.10155414	2.81	-0.20
Ceramic C	2.62	1.333857	0.024965	195.7067157	0.150112321	29.37798929	2.61	0.98
Ceramic C	2.62	4.5166358	0.029335	229.9608736	0.150112321	34.51996043	2.55	2.04
Ceramic C	2.62	5.1531916	0.038407	301.0849422	0.150112321	45.19655942	2.43	-2.03
Ceramic D	2.80	-4.857044	0.009434	78.17750004	0.112610608	8.803615794	3.04	3.61
Ceramic D	2.80	5.2711928	0.026164	216.8188122	0.112610608	24.41609823	2.85	7.12
Ceramic D	2.80	8.8458648	0.035724	296.0422622	0.112610608	33.33749908	2.74	7.09
Ceramic D	2.80	10.633201	0.045348	375.7920062	0.112610608	42.31816622	2.64	5.13
Ceramic E	2.65	-13.41072	0.009434	113.4113459	0.161445628	18.30976592	2.80	-6.97
Ceramic E	2.65	5.4910664	0.023507	282.5952673	0.161445628	45.62377026	2.56	2.45
Ceramic E	2.65	9.2714237	0.026124	314.0554427	0.161445628	50.70287803	2.52	4.68
Ceramic E	2.65	9.2714237	0.033143	398.4278011	0.161445628	64.32442637	2.40	-0.12

Slurry Model Validation Data

<u>Media Name</u>	<u>Pass #</u>	<u>Feed (P80)</u>	<u>Aspect Ratio</u>	<u>Bead Volume (cm³)</u>	<u>Bead Surface Area (cm²)</u>	<u>Bead Mass (g)</u>	<u>Media Vol (l)</u>	<u>Media Mass (g)</u>	<u>No of Beads</u>	<u>Total Surface Area (cm²)</u>	<u>No Load Power (kW)</u>	<u>Slurry Power Draw (kW)</u>	<u>FC_{slurry} * SA</u>	<u>FC_{slurry} * SA * AR</u>
Ceramic C	Pass 1	106.00	1.012	0.156	0.041	0.625	1	2406.6	3850.5	157.45	1.10	2.48	47	47.8
Ceramic C	Pass 2	58.26	1.012	0.156	0.041	0.625	1	2406.6	3850.5	157.45	1.10	2.32	47	47.8
Ceramic C	Pass 3	48.03	1.012	0.156	0.041	0.625	1	2406.6	3850.5	157.45	1.10	2.15	47	47.8
Ceramic C	Pass 4	35.02	1.012	0.156	0.041	0.625	1	2406.6	3850.5	157.45	1.10	2.07	47	47.8
Ceramic C	Pass 1	106.00	1.012	0.156	0.041	0.625	1.5	3665.1	5864.1	239.79	1.10	2.73	72	72.8
Ceramic C	Pass 2	45.99	1.012	0.156	0.041	0.625	1.5	3665.1	5864.1	239.79	1.10	2.42	72	72.8
Ceramic C	Pass 3	35.57	1.012	0.156	0.041	0.625	1.5	3665.1	5864.1	239.79	1.10	2.32	72	72.8
Ceramic C	Pass 4	27.64	1.012	0.156	0.041	0.625	1.5	3665.1	5864.1	239.79	1.10	2.30	72	72.8
Ceramic C	Pass 1	106	1.012	0.156	0.041	0.625	2	4899.6	7839.2	320.56	1.10	2.82	96	97.3
Ceramic C	Pass 2	40.06	1.012	0.156	0.041	0.625	2	4899.6	7839.2	320.56	1.10	2.58	96	97.3
Ceramic C	Pass 3	34.09	1.012	0.156	0.041	0.625	2	4899.6	7839.2	320.56	1.10	2.50	96	97.3
Ceramic C	Pass 4	26.04	1.012	0.156	0.041	0.625	2	4899.6	7839.2	320.56	1.10	2.48	96	97.3

<u>Media Name</u>	<u>1/P80</u>	<u>N*P80</u>	<u>SI_N</u>	<u>SN*SI</u>	<u>Predicted Power (kW)</u>	<u>Prediction Error %</u>
Ceramic C	0.0094	36.325	0.150	5.453	1.94	-27.8
Ceramic C	0.0172	66.092	0.150	9.921	1.89	-22.7
Ceramic C	0.0208	80.170	0.150	12.034	1.87	-15.1
Ceramic C	0.0286	109.954	0.150	16.506	1.82	-13.9
Ceramic C	0.0094	55.321	0.150	8.304	2.38	-14.7
Ceramic C	0.0217	127.507	0.150	19.140	2.26	-7.1
Ceramic C	0.0281	164.874	0.150	24.750	2.20	-5.6
Ceramic C	0.0362	212.128	0.150	31.843	2.12	-8.5
Ceramic C	0.0094	73.955	0.150	11.102	2.81	-0.2
Ceramic C	0.0250	195.707	0.150	29.378	2.61	1.0
Ceramic C	0.0293	229.961	0.150	34.520	2.55	2.0
Ceramic C	0.0384	301.085	0.150	45.197	2.43	-2.0

Detailed Slurry Testing Data:

CERAMIC MEDIA WITH SLURRY							
Date:				Qslurry:	2.6		
Media type:	Ceramic A			Pump speed:	638.4		
Initial Media Mass:	4613.5			Mill speed:	2100		
Final Media Mass:	4575.3			Slurry SG:	1.40		
PASS 1				PASS 2			
Time	Power	Temp	Pressure	Time	Power	Temp	Pressure
minutes	kW	°C	bar	minutes	kW	°C	bar
0	3.0	28	0.4	0	2.8	35	0.4
5	3.5	32	0.7	5	2.9	42	0.5
10	3.3	36	0.7	10	2.9	42	0.5
15	3.1	36	0.5	15	3	42	0.5
20	3	36	0.5	20	2.9	43	0.5
25	2.8	35	0.4	25	2.9	43	0.5
30	2.8	35	0.4	30	2.8	43	0.4
Average	3.08	34.0	0.51	Average	2.90	41.4	0.47
PASS 3				PASS 4			
Time	Power	Temp	Pressure	Time	Power	Temp	Pressure
minutes	kW	°C	bar	minutes	kW	°C	bar
0	2.8	42	0.4	0	2.8	47	0.4
5	2.9	46	0.4	5	2.8	50	0.4
10	2.9	46	0.4	10	2.8	50	0.4
15	2.9	46	0.4	15	2.8	50	0.4
20	2.9	46	0.4	20	2.8	50	0.4
25	2.9	47	0.4	25	2.8	50	0.4
30	2.8	47	0.4	30	2.8	50	0.4
Average	2.88	46.3	0.40	Average	2.80	50.0	0.40
Media loss (g)	38.2			kWh reading start	3208.5		7.3
Media loss (%)	0.83%			kWh reading end	3215.8		3.7

CERAMIC MEDIA WITH SLURRY							
Date:				Qslurry:	2.6		
Media type:	Ceramic B			Pump speed:	638.4		
Initial Media Mass:	4636.2			Mill speed:	2100		
Final Media Mass:	4590.2			Slurry SG:	1.40		
PASS 1				PASS 2			
Time	Power	Temp	Pressure	Time	Power	Temp	Pressure
minutes	kW	°C	bar	minutes	kW	°C	bar
0	3.4	30	0.7	0	2.7	37	0.5
5	3.4	38	0.7	5	2.8	43	0.5
10	3.2	39	0.7	10	2.8	45	0.5
15	3.1	39	0.7	15	2.8	45	0.5
20	3	38	0.7	20	2.8	45	0.5
25	2.9	38	0.5	25	2.7	45	0.5
30	2.9	37	0.5	30	2.7	44	0.5
Average	3.08	37.0	0.64	Average	2.77	43.4	0.50
PASS 3				PASS 4			
Time	Power	Temp	Pressure	Time	Power	Temp	Pressure
minutes	kW	°C	bar	minutes	kW	°C	bar
0	2.6	44	0.5	0	2.6	47	0.3
5	2.6	48	0.5	5	2.6	50	0.3
10	2.6	48	0.5	10	2.6	51	0.3
15	2.6	48	0.5	15	2.6	51	0.3
20	2.6	48	0.5	20	2.6	51	0.3
25	2.6	48	0.5	25	2.6	51	0.3
30	2.6	47	0.5	30	2.6	51	0.3
Average	2.60	47.8	0.50	Average	2.60	50.8	0.30
Media loss (g)	46			kWh reading start	3215.8		7.1
Media loss (%)	0.99%			kWh reading end	3222.9		3.5

CERAMIC MEDIA WITH SLURRY							
Date:				Qslurry:	2.6		
Media type:	Ceramic C (1l)			Pump speed:	638.4		
Initial Media Mass:	2406.6			Mill speed:	2100		
Final Media Mass:	2400			Slurry SG:	1.40		
PASS 1				PASS 2			
Time	Power	Temp	Pressure	Time	Power	Temp	Pressure
minutes	kW	°C	bar	minutes	kW	°C	bar
0	2.2	30	0.5	0	2.3	35	0.6
5	2.6	34	0.7	5	2.4	40	0.6
10	2.6	36	0.7	10	2.4	40	0.6
15	2.5	36	0.7	15	2.4	40	0.6
20	2.5	36	0.7	20	2.3	40	0.6
25	2.4	36	0.7	25	2.2	40	0.6
30	2.3	35	0.7	30	2.2	40	0.6
Average	2.48	34.7	0.67	Average	2.32	39.3	0.60
PASS 3				PASS 4			
Time	Power	Temp	Pressure	Time	Power	Temp	Pressure
minutes	kW	°C	bar	minutes	kW	°C	bar
0	2.2	40	0.5	0	2.1	43	0.3
5	2.2	42	0.5	5	2.1	44	0.3
10	2.2	42	0.5	10	2.1	44	0.3
15	2.2	43	0.5	15	2.1	44	0.3
20	2.1	43	0.5	20	2.1	45	0.3
25	2.1	43	0.5	25	2.0	45	0.3
30	2.1	43	0.5	30	2.0	45	0.3
Average	2.15	42.7	0.50	Average	2.07	44.5	0.30
Media loss (g)	6.6			kWh reading start	3222.9		5.8
Media loss (%)	0.27%			kWh reading end	3228.7		2.9

CERAMIC MEDIA WITH SLURRY							
Date:	13-May-09		Qslurry:	2.6			
Media type:	Ceramic C (1.5l)		Pump speed:	638.4			
Initial Media Mass:	3665.1		Mill speed:	2100			
Final Media Mass:	3654.9		Slurry SG:	1.40			
PASS 1				PASS 2			
Time	Power	Temp	Pressure	Time	Power	Temp	Pressure
minutes	kW	°C	bar	minutes	kW	°C	bar
0	2.5	28	0.4	0	2.4	33	0.5
5	3	32	0.7	5	2.5	38	0.5
10	2.9	34	0.7	10	2.5	39	0.5
15	2.9	34	0.7	15	2.4	39	0.5
20	2.7	34	0.7	20	2.4	39	0.5
25	2.5	34	0.7	25	2.4	40	0.5
30	2.4	33	0.7	30	2.3	40	0.5
Average	2.73	32.7	0.66	Average	2.42	38.3	0.50
PASS 3				PASS 4			
Time	Power	Temp	Pressure	Time	Power	Temp	Pressure
minutes	kW	°C	bar	minutes	kW	°C	bar
0	2.3	39	0.5	0	2.3	42	0.3
5	2.4	41	0.5	5	2.3	45	0.3
10	2.3	42	0.5	10	2.3	45	0.3
15	2.3	42	0.5	15	2.3	46	0.3
20	2.3	42	0.5	20	2.3	46	0.3
25	2.3	42	0.5	25	2.3	46	0.3
30	2.3	42	0.5	30	2.3	46	0.3
Average	2.32	41.8	0.50	Average	2.30	45.7	0.30
Media loss (g)	10.2	kWh reading start		3228.9	2.6		
Media loss (%)	0.28%	kWh reading end		3231.5	1.3		

CERAMIC MEDIA WITH SLURRY

Date:	
Media type:	Ceramic C (2l)
Initial Media Mass:	4899.6
Final Media Mass:	4887.8

Qslurry:	2.6
Pump speed:	638.4
Mill speed:	2100
Slurry SG:	1.4

PASS 1			
Time	Power	Temp	Pressure
minutes	kW	°C	bar
0	2.9	28	0.4
5	3.2	35	0.7
10	3.2	36	0.7
15	2.8	36	0.7
20	2.6	36	0.7
25	2.6	36	0.7
30	2.5	36	0.7
Average	2.82	35.8	0.70

PASS 2			
Time	Power	Temp	Pressure
minutes	kW	°C	bar
0	2.6	35	0.5
5	2.6	41	0.5
10	2.7	42	0.5
15	2.6	42	0.5
20	2.6	42	0.5
25	2.5	42	0.5
30	2.5	42	0.5
Average	2.58	41.8	0.50

PASS 3			
Time	Power	Temp	Pressure
minutes	kW	°C	bar
0	2.5	42	0.5
5	2.5	45	0.5
10	2.5	46	0.5
15	2.5	46	0.5
20	2.5	46	0.5
25	2.5	46	0.5
30	2.5	46	0.5
Average	2.50	45.8	0.50

PASS 4			
Time	Power	Temp	Pressure
minutes	kW	°C	bar
0	2.5	46	0.3
5	2.5	49	0.3
10	2.5	50	0.3
15	2.5	50	0.3
20	2.5	50	0.3
25	2.5	50	0.3
30	2.4	49	0.3
Average	2.48	49.7	0.30

Media loss (g)	11.8	kWh reading start	3242.1	6.8
Media loss (%)	0.24%	kWh reading end	3248.9	3.4

CERAMIC MEDIA WITH SLURRY

Date:	
Media type:	Ceramic D
Initial Media Mass:	3883.7
Final Media Mass:	3851

Qslurry:	2.6
Pump speed:	638.4
Mill speed:	2100
Slurry SG:	1.40

PASS 1			
Time	Power	Temp	Pressure
minutes	kW	°C	bar
0	2.7	25	0.4
5	3.2	32	0.7
10	3.1	34	0.7
15	3	34	0.7
20	2.9	34	0.7
25	2.7	34	0.7
30	2.7	34	0.7
Average	2.93	32.4	0.66

PASS 2			
Time	Power	Temp	Pressure
minutes	kW	°C	bar
0	2.7	34	0.5
5	2.7	40	0.5
10	2.7	41	0.5
15	2.7	41	0.5
20	2.6	41	0.5
25	2.6	41	0.5
30	2.6	41	0.5
Average	2.65	39.9	0.50

PASS 3			
Time	Power	Temp	Pressure
minutes	kW	°C	bar
0	2.5	41	0.5
5	2.6	44	0.5
10	2.6	45	0.5
15	2.6	45	0.5
20	2.5	45	0.5
25	2.5	45	0.5
30	2.5	45	0.5
Average	2.55	44.8	0.50

PASS 4			
Time	Power	Temp	Pressure
minutes	kW	°C	bar
0	2.5	47	0.3
5	2.5	48	0.3
10	2.5	49	0.3
15	2.5	49	0.3
20	2.5	49	0.3
25	2.5	49	0.3
30	2.5	49	0.3
Average	2.50	48.8	0.30

Media loss (g)	32.7	kWh reading start	3235.1	7.0
Media loss (%)	0.84%	kWh reading end	3242.1	3.5

CERAMIC MEDIA WITH SLURRY

Date:	
Media type:	Ceramic E
Initial Media Mass:	7895.9
Final Media Mass:	7885.6

Qslurry:	2.6
Pump speed:	638.4
Mill speed:	2100
Slurry SG:	1.40

PASS 1			
Time	Power	Temp	Pressure
minutes	kW	°C	bar
0	3.2	31	0.4
5	3.4	38	0.7
10	3.2	40	0.7
15	3	40	0.7
20	2.9	39	0.5
25	2.8	39	0.5
30	2.7	38	0.5
Average	3.00	37.9	0.57

PASS 2			
Time	Power	Temp	Pressure
minutes	kW	°C	bar
0	2.4	38	0.4
5	2.6	43	0.5
10	2.5	44	0.5
15	2.5	44	0.5
20	2.5	44	0.5
25	2.5	44	0.5
30	2.4	44	0.5
Average	2.50	43.0	0.49

PASS 3			
Time	Power	Temp	Pressure
minutes	kW	°C	bar
0	2.4	43	0.5
5	2.4	46	0.5
10	2.4	46	0.5
15	2.4	47	0.5
20	2.4	47	0.5
25	2.4	47	0.5
30	2.4	47	0.5
Average	2.40	46.7	0.50

PASS 4			
Time	Power	Temp	Pressure
minutes	kW	°C	bar
0	2.4	47	0.3
5	2.4	50	0.3
10	2.4	50	0.3
15	2.4	50	0.3
20	2.4	50	0.3
25	2.4	50	0.3
30	2.4	50	0.3
Average	2.40	50.0	0.30

Media loss (g)	10.3	kWh reading start	3248.9	6.8
Media loss (%)	0.13%	kWh reading end	3255.7	3.4

CERAMIC MEDIA WITH SLURRY

Date:	
Media type:	Silica Sand
Initial Media Mass:	3428.5
Final Media Mass:	3260.4

Qslurry:	2.6
Pump speed:	638.4
Mill speed:	2100
Slurry SG:	1.40

PASS 1			
Time	Power	Temp	Pressure
minutes	kW	°C	bar
0	2.3	29	0.4
5	2.8	36	0.9
10	2.6	37	0.9
15	2.5	37	0.9
20	2.4	37	0.9
25	2.4	36	0.9
30	2.3	36	0.7
Average	2.50	35.4	0.80

PASS 2			
Time	Power	Temp	Pressure
minutes	kW	°C	bar
0	2.3	35	0.7
5	2.6	41	0.7
10	2.5	42	0.7
15	2.4	42	0.7
20	2.4	42	0.7
25	2.3	42	0.7
30	2.3	42	0.7
Average	2.42	40.9	0.70

PASS 3			
Time	Power	Temp	Pressure
minutes	kW	°C	bar
0	2.4	41	0.7
5	2.4	44	0.7
10	2.4	44	0.7
15	2.4	44	0.7
20	2.3	44	0.7
25	2.3	44	0.7
30	2.3	44	0.7
Average	2.35	44.0	0.70

PASS 4			
Time	Power	Temp	Pressure
minutes	kW	°C	bar
0	2.3	43	0.5
5	2.4	46	0.5
10	2.3	47	0.5
15	2.3	47	0.5
20	2.3	47	0.5
25	2.3	47	0.5
30	2.2	47	0.7
Average	2.30	46.8	0.53

Media loss (g)	168.1	kWh reading start	665.4	6.7
Media loss (%)	4.90%	kWh reading end	672.1	3.4

Media:

Ceramic A

Pass	Time	Gross Power	Q	NLP	N _{mill}	Slurry SG	Temp	Net Power	Q	%	Mass flow	kWh/t	ΣkWh/t	P80
#	min	kW	l/min	kW	rpm	kg/l	°C	kW	m ³ /h	Solids	t/h			μm
1	30	3.08	2.60	1.10	2100	1.4	34	1.98	0.16	39%	0.08	23.4	23.4	37.99
2	30	2.90	2.60	1.10	2100	1.4	41	1.80	0.16	39%	0.08	21.3	44.7	30.65
3	30	2.88	2.60	1.10	2100	1.4	46	1.78	0.16	39%	0.08	21.1	65.7	24.07
4	30	2.80	2.60	1.10	2100	1.4	50	1.70	0.16	39%	0.08	20.1	85.8	20.5

Media:

Ceramic B

Pass	Time	Gross Power	Q	NLP	N _{mill}	Slurry SG	Temp	Net Power	Q	%	Mass flow	kWh/t	ΣkWh/t	P80
#	min	kW	l/min	kW	rpm	kg/l	°C	kW	m ³ /h	Solids	t/h			μm
1	30	3.08	2.60	1.10	2100	1.4	37	1.98	0.16	39%	0.08	23.4	23.4	36.18
2	30	2.77	2.60	1.10	2100	1.4	43	1.67	0.16	39%	0.08	19.7	43.1	28.83
3	30	2.60	2.60	1.10	2100	1.4	48	1.50	0.16	39%	0.08	17.7	60.8	22.47
4	30	2.60	2.60	1.10	2100	1.4	51	1.50	0.16	39%	0.08	17.7	78.5	19.385

Media:

Ceramic C 1.0l

Pass	Time	Gross Power	Q	NLP	N _{mill}	Slurry SG	Temp	Net Power	Q	%	Mass flow	kWh/t	ΣkWh/t	P80
#	min	kW	l/min	kW	rpm	kg/l	°C	kW	m ³ /h	Solids	t/h			μm
1	30	2.48	2.60	1.10	2100	1.40	35	1.38	0.16	39%	0.08	16.3	16.3	58.26
2	30	2.32	2.60	1.10	2100	1.40	39	1.22	0.16	39%	0.08	14.4	30.7	48.03
3	30	2.15	2.60	1.10	2100	1.40	43	1.05	0.16	39%	0.08	12.4	43.1	35.02
4	30	2.07	2.60	1.10	2100	1.40	45	0.97	0.16	39%	0.08	11.4	54.5	27.47

Media:

Ceramic C 1.5I

Pass	Time	Gross Power	Q	NLP	N _{mill}	Slurry SG	Temp	Net Power	Q	%	Mass flow	kWh/t	ΣkWh/t	P80
#	min	kW	l/min	kW	rpm	kg/l	°C	kW	m ³ /h	Solids	t/h			μm
1	30	2.73	2.60	1.10	2100	1.40	33	1.63	0.16	39%	0.08	19.3	19.3	45.99
2	30	2.42	2.60	1.10	2100	1.40	38	1.32	0.16	39%	0.08	15.5	34.8	35.57
3	30	2.32	2.60	1.10	2100	1.40	42	1.22	0.16	39%	0.08	14.4	49.2	27.64
4	30	2.30	2.60	1.10	2100	1.40	46	1.20	0.16	39%	0.08	14.2	63.4	23.33

Media:

Ceramic C 2.0 I

Pass	Time	Gross Power	Q	NLP	N _{mill}	Slurry SG	Temp	Net Power	Q	%	Mass flow	kWh/t	ΣkWh/t	P80
#	min	kW	l/min	kW	rpm	kg/l	°C	kW	m ³ /h	Solids	t/h			μm
1	30	2.82	2.60	1.10	2100	1.40	36	1.72	0.16	39%	0.08	20.3	20.3	40.06
2	30	2.58	2.60	1.10	2100	1.40	42	1.48	0.16	39%	0.08	17.5	37.8	34.09
3	30	2.50	2.60	1.10	2100	1.40	46	1.40	0.16	39%	0.08	16.5	54.3	26.04
4	30	2.48	2.60	1.10	2100	1.40	50	1.38	0.16	39%	0.08	16.3	70.7	21.86

Media:

Ceramic D

Pass	Time	Gross Power	Q	NLP	N _{mill}	Slurry SG	Temp	Net Power	Q	%	Mass flow	kWh/t	ΣkWh/t	P80
#	min	kW	l/min	kW	rpm	kg/l	°C	kW	m ³ /h	Solids	t/h			μm
1	30	2.93	2.60	1.10	2100	1.40	32	1.83	0.16	39%	0.08	21.6	21.6	38.22
2	30	2.65	2.60	1.10	2100	1.40	40	1.55	0.16	39%	0.08	18.3	40.0	27.99
3	30	2.55	2.60	1.10	2100	1.40	45	1.45	0.16	39%	0.08	17.1	57.1	22.05
4	30	2.50	2.60	1.10	2100	1.40	49	1.40	0.16	39%	0.08	16.5	73.6	18.64

Media:

Ceramic E

Pass	Time	Gross Power	Q	NLP	N _{mill}	Slurry SG	Temp	Net Power	Q	%	Mass flow	kWh/t	ΣkWh/t	P80
#	min	kW	l/min	kW	rpm	kg/l	°C	kW	m ³ /h	Solids	t/h			μm
1	30	3.00	2.60	1.10	2100	1.40	38	1.90	0.16	39%	0.08	22.4	22.4	42.54
2	30	2.50	2.60	1.10	2100	1.40	43	1.40	0.16	39%	0.08	16.5	39.0	38.28
3	30	2.40	2.60	1.10	2100	1.40	47	1.30	0.16	39%	0.08	15.4	54.3	30.17
4	30	2.40	2.60	1.10	2100	1.40	50	1.30	0.16	39%	0.08	15.4	69.7	25.54

Media:

Sillica Sand

Pass	Time	Gross Power	Q	NLP	N _{mill}	Slurry SG	Temp	Net Power	Q	%	Mass flow	kWh/t	ΣkWh/t	P80
#	min	kW	l/min	kW	rpm	kg/l	°C	kW	m ³ /h	Solids	t/h			μm
1	30	2.50	2.60	1.10	2100	1.40	35	1.40	0.16	39%	0.08	16.5	16.5	70.88
2	30	2.42	2.60	1.10	2100	1.40	41	1.32	0.16	39%	0.08	15.5	32.1	64.16
3	30	2.35	2.60	1.10	2100	1.40	44	1.25	0.16	39%	0.08	14.8	46.8	53.89
4	30	2.30	2.60	1.10	2100	1.40	47	1.20	0.16	39%	0.08	14.2	61.0	48.84

Appendix K: Regression Analysis Results

Regression Analysis: Water Model Development - Predicted versus Experimental

SUMMARY

OUTPUT

<i>Regression Statistics</i>	
Multiple R	0.999518
R Square	0.999037
Adjusted R Square	0.95737
Standard Error	0.056641
Observations	25

5.66%

ANOVA

	<i>df</i>	<i>SS</i>	<i>MS</i>	<i>F</i>	<i>Significance F</i>
Regression	1	79.85289	79.85289	24890.38	2.077E-36
Residual	24	0.076996	0.003208		
Total	25	79.92988			

	<i>Coefficients</i>	<i>Standard Error</i>	<i>t Stat</i>	<i>P-value</i>	<i>Lower 95%</i>	<i>Upper 95%</i>	<i>Lower 95.0%</i>	<i>Upper 95.0%</i>
Intercept	0	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A
X Variable 1	1.013144	0.006422	157.7669	1.03E-37	0.999889761	1.026398	0.99989	1.026398

Regression analysis - Water Model Validation - Predicted versus Experimental

SUMMARY

OUTPUT

<i>Regression Statistics</i>	
Multiple R	0.999708
R Square	0.999416
Adjusted R Square	0.749416
Standard Error	0.050021
Observations	5

5.00%

ANOVA

	<i>df</i>	<i>SS</i>	<i>MS</i>	<i>F</i>	<i>Significance F</i>
Regression	1	17.13454	17.13454	6848.089	3.89E-06
Residual	4	0.010008	0.002502		
Total	5	17.14454			

	<i>Coefficients</i>	<i>Standard Error</i>	<i>t Stat</i>	<i>P-value</i>	<i>Lower 95%</i>	<i>Upper 95%</i>	<i>Lower 95.0%</i>	<i>Upper 95.0%</i>
Intercept	0	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A
X Variable 1	0.963115	0.011638	82.75318	1.28E-07	0.930801	0.995428	0.930801	0.995428

Regression Analysis - Modified Water Model - Predicted versus Experimental

SUMMARY OUTPUT

<i>Regression Statistics</i>	
Multiple R	0.997510024
R Square	0.995026249
Adjusted R Square	0.94239467
Standard Error	0.196103572
Observations	20

19.61%

<i>ANOVA</i>					
	<i>df</i>	<i>SS</i>	<i>MS</i>	<i>F</i>	<i>Significance F</i>
Regression	1	146.1756688	146.1757	3801.054	2.13328E-22
Residual	19	0.730675607	0.038457		
Total	20	146.9063444			

	<i>Coefficients</i>	<i>Standard Error</i>	<i>t Stat</i>	<i>P-value</i>	<i>Lower 95%</i>	<i>Upper 95%</i>	<i>Lower 95.0%</i>	<i>Upper 95.0%</i>
Intercept	0	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A
X Variable 1	0.989821578	0.016054799	61.65269	2.38E-23	0.956218497	1.023424658	0.956218497	1.023424658

Regression Analysis - Slurry Model Development - Predicted versus Experimental

SUMMARY OUTPUT

<i>Regression Statistics</i>	
Multiple R	0.999064341
R Square	0.998129557
Adjusted R Square	0.945497978
Standard Error	0.120416745
Observations	20

12.04%

ANOVA

	<i>df</i>	<i>SS</i>	<i>MS</i>	<i>F</i>	<i>Significance F</i>
Regression	1	147.0177895	147.0178	10139.02	3.20019E-26
Residual	19	0.275503655	0.0145		
Total	20	147.2932932			

	<i>Coefficients</i>	<i>Standard Error</i>	<i>t Stat</i>	<i>P-value</i>	<i>Lower 95%</i>	<i>Upper 95%</i>	<i>Lower 95.0%</i>	<i>Upper 95.0%</i>
Intercept	0	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A
X Variable 1	1.000379235	0.009934971	100.6927	2.19E-27	0.979585101	1.021173368	0.979585101	1.021173368

Regression Analysis - New Slurry Model Validation 1000ml mill fills - Predicted versus Experimental

SUMMARY OUTPUT

<i>Regression Statistics</i>	
Multiple R	0.9991073
R Square	0.99821539
Adjusted R Square	0.66488206
Standard Error	0.09609366
Observations	4

9.61%

<i>ANOVA</i>					
	<i>df</i>	<i>SS</i>	<i>MS</i>	<i>F</i>	<i>Significance F</i>
Regression	1	15.49504	15.49504	1678.044	0.0005954
Residual	3	0.027702	0.009234		
Total	4	15.52275			

	<i>Coefficients</i>	<i>Standard Error</i>	<i>t Stat</i>	<i>P-value</i>	<i>Lower 95%</i>	<i>Upper 95%</i>	<i>Lower 95.0%</i>	<i>Upper 95.0%</i>
Intercept	0	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A
X Variable 1	0.87067049	0.021255	40.96394	3.2E-05	0.803028989	0.938312	0.803029	0.938312

Regression Analysis - New Slurry Model Validation 1500ml mill fills - Predicted versus Experimental

SUMMARY OUTPUT

<i>Regression Statistics</i>	
Multiple R	0.99962056
R Square	0.99924127
Adjusted R Square	0.66590793
Standard Error	0.07324045
Observations	4

7.32%

<i>ANOVA</i>					
	<i>df</i>	<i>SS</i>	<i>MS</i>	<i>F</i>	<i>Significance F</i>
Regression	1	21.19357	21.19357	3950.956	0.000253007
Residual	3	0.016092	0.005364		
Total	4	21.20967			

	<i>Coefficients</i>	<i>Standard Error</i>	<i>t Stat</i>	<i>P-value</i>	<i>Lower 95%</i>	<i>Upper 95%</i>	<i>Lower 95.0%</i>	<i>Upper 95.0%</i>
Intercept	0	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A
X Variable 1	0.94007414	0.014956	62.85663	8.87E-06	0.892477957	0.98767	0.892478	0.98767

Regression Analysis - New Slurry Model Validation All lower mill fills - Predicted versus Experimental

SUMMARY OUTPUT

<i>Regression Statistics</i>	
Multiple R	0.99867948
R Square	0.99736069
Adjusted R Square	0.85450355
Standard Error	0.11768484
Observations	8

12%

ANOVA

	<i>df</i>	<i>SS</i>	<i>MS</i>	<i>F</i>	<i>Significance F</i>
Regression	1	36.63546	36.63546	2645.213	3.62526E-09
Residual	7	0.096948	0.01385		
Total	8	36.73241			

	<i>Coefficients</i>	<i>Standard Error</i>	<i>t Stat</i>	<i>P-value</i>	<i>Lower 95%</i>	<i>Upper 95%</i>	<i>Lower 95.0%</i>	<i>Upper 95.0%</i>
Intercept	0	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A
X Variable 1	0.90813889	0.017657	51.43163	2.75E-10	0.866386234	0.949892	0.866386	0.949892