

MODELLING STUDIES OF A ROTARY OFFSET CRUSHER

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DECLARATION

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

Titus Nghipulile

Signed on ____11 October 2019_____

ABSTRACT

The quest for efficiency in comminution is an on-going concern as it usually constitutes a major cost component in the metal production industry. Such improvements can be made by circuit optimization or development of more efficient equipment. The rotary offset crusher (ROC), a novel crushing equipment, invented in 2002 by Michael Hunt, Henry Simonsen and Ian Sinclair is simple by design with only two moving parts (the cylindrical discs) that are parallel to each other, but not vertically aligned. This crusher employs the same crushing mechanism as the HPGR which is said to be energy efficient as compared to tumbling mills, with energy savings of 10 – 30 % reported. In addition, unlike the HPGR, in the ROC centrifugal motion guides the transportation of particles in the crushing zone and therefore, showing the potential to be a high throughput crusher. This study aimed at building the laboratory prototype to demonstrate the concept and study the principles guiding the operation of this equipment. The original design concept was re-ignited, and a redesigned laboratory crusher has been built. The crusher was instrumented with sensing devices to pick up signals that allow measurement of the speeds for the two discs, motor drive torque and mass on the conveyor belt and thereby allowing computation of the crusher power draw and feed rates. Laboratory breakage tests with the drop weight and piston-die apparatus on coal and quartz were conducted and the results were correlated to those of the ROC.

Following the fabrication and commissioning of the crusher, experiments were conducted with coal to investigate the effect of feed size distribution, horizontal offset of the discs and vertical exit gap between the top and bottom discs. The rotational speed was fixed at 330 rpm in all tests. The horizontal offsets were 5 and 10 mm while the vertical exit gaps were 1.5 and 3 mm. The two feed size fractions used were -13.2+9.5 mm and 19+13.2 mm. Results from the 8 experiments conducted showed no well-defined relationships between different operating

variables (horizontal offset, vertical exit gap and feed size distribution) and d_{80} and throughput. More experiments still need to be conducted to establish sustained trends. The size reduction ratios were in the range of 2 with acicular particle discharged from the crusher. Those results were attributed to the absence of corrugated profiles on the crushing surfaces as well as the mineralogical characteristics of coal.

To investigate the effects of rotational speed and feed rate, 8 experiments were conducted with quartz. The horizontal offset between the discs and vertical exit gap were fixed at 10 and 3 mm respectively. What came to light was that the performance of the ROC is highly dependent on the speed. The size reduction ratios as high as 7 were recorded at the speed of 550 rpm and feed rate of 1 tph. It is recommended that many experiments be conducted with quartz at this speed for various offsets, particle sizes, feed rates and crusher exit gaps to help with optimization of the crusher. Thereafter, higher speeds can be tried to establish the relationship between speed and size reduction.

The discrete element method (DEM) was used to study the transportation of particles in the ROC. Simulation results showed that the throughput is highly dependent on the rotational speed of the discs. This agrees with the experimental data generated using the laboratory prototype. Simulation using the DEM for various design configurations is worth considering in future as this would improve the understanding of the flow behaviours of the particles in the crusher. The design of disc profiles of various configurations needs to be undertaken using the DEM to study both the breakage and transportation of particles in the crusher. This would guide the future modifications of the crushing faces of the crusher. Importantly, it is recommended that comparative studies with competing comminution machines such as HPGR, short head cone crusher and Loesche mill be undertaken to establish benefits in terms of energy efficiency, throughput, size reduction, if any, for the ROC over the existing machines.

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“But thanks be to God! He gives us the victory through our Lord Jesus Christ.” 1 Corinthians 15:57

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LIST OF ABBREVIATIONS AND SYMBOLS

ROC	Rotary offset crusher
HPGR	High Pressure Grinding Roll
VRM	Vertical roller mill
AG	Autogenous grinding mill
SAG	Semi-Autogenous grinding mill
ROM	Run-of-mine
DEM	Discrete element method
W	Bond work input in kWh/t (kilowatt-hour per tonne)
W_i	Bond work index in kWh/t of the material
P_{80}	The 80 % passing size for the crusher product in Bond's law
F_{80}	The 80 % passing size for the crusher feed in Bond's law
M_i	Work index for Morrell formulation (Eq. (2.25))
τ	Residence time of particles in a comminution machine
DWT	Drop weight test
E_i	Input energy in the DWT
m_d	Mass of drop weight in Eq. (2.9)
E_{cs}	Specific comminution energy (in kWh/t)
t_{10}	percent passing the tenth of the feed size given by Eq. (2.11)
A and b	ore breakage parameters as defined in Eq. (2.11)
S_i	Selection function for size class i
S_j	Selection function for size class j
m_i	Mass of material in size class i
m_j	Mass of material in size class j
w_i	Mass fraction of material in size class i
w_j	Mass fraction of material in size class j
p_i	Mass of product for size class i

f_i	Mass of feed for size class i
b_{ij}	Breakage function as defined in Eq. (2.23)
B_{ij}	Cumulative breakage function as defined in Eq. (2.25)
β	Material-dependent breakage parameter in Eq. (2.25)
ϕ	Fraction of fine particles resulting from single breakage event
γ	Related to relative number of fines produced in a breakage event
R	Relative particle size (as defined in Eq. (2.25))
F	Force
F_n	Normal force acting on particles (as defined by Eq. (2.27))
F_t	Tangential force acting on particles (as defined by Eq. (2.28))
v_n	Normal relative velocity of the particles (in Eq. (2.27))
v_t	Tangential relative velocity of the particles (in Eq. (2.28))
K_n	Stiffness of the spring in normal direction (in Eq. (2.27))
K_t	Stiffness of the spring in tangential direction (in Eq. (2.28))
C_n	Normal damping coefficient in Eq. (2.27)
C_t	Tangential damping coefficient in Eq. (2.28)
ε	Coefficient of restitution
Δx	Amount of overlap of particles (in mm) in Eq. (2.27) and (2.28)
P	Power (in Watt)
n	Rotational speed in revolution per minutes (rpm)
T	Torque (in Nm)
ω	Angular velocity in radians per seconds (rad/s)
IR	Infrared rays
LED	Light emitting diode
HIGH	Digital output representing 1 or true
LOW	Digital output representing 0 or false
E_k	Stored kinetic energy in Joules
I	Moment of inertia

C_s	Coefficient of fluctuation of the speed
M_t	Mass on conveyor at time t
A_t	Analog output of the load cell at time t
G	Crusher gap
G_{in}	ROC input vertical gap
G_e	ROC exit vertical gap
h_c	Height of the comminution cavity
x_o	Discs offset in the x-direction
N_p	Number of particles during DEM simulation
m_t	Total mass of particles (balls) fed to the ROC during DEM simulation
m_s	Mass of a single particle (ball) fed to the ROC during the DEM simulation
d_i	Diameter of a ball (particle) for DEM simulation
ρ	Density of the balls (particles) fed to the ROC in kg/m^3
SG	Specific gravity
Q	Throughput in tonne per hour (tph) or kilogram per hour (kg/h)
F	Feed rate in tph or kg/h
ANOVA	Analysis of variance
P-value	Statistical value for hypothesis testing
$RMSE$	Root mean square error
R^2	Coefficient of correlation
d_{80}	80 % passing size
R_{80}	Reduction ratio calculated using the d_{80} of the feed and product
d_{50}	50 % passing size
R_{50}	Reduction ratio calculated using the d_{50} of the feed and product
d_M	Geometric mean size of the mono-size feed sample to the ROC

CHAPTER ONE: INTRODUCTION

1.1 Background and Motivation for the study

The mining industry consumes about 7 % of the global energy (Manouchehri, 2015), of which almost half is utilised in comminution circuits (Curry et al., 2014, Ballantyne and Powell, 2014, Napier-Munn, 2015 and Jeswiet and Szekeres, 2016). However, comminution is an inefficient process with only a small fraction of the energy meant for size reduction being used to effect breakage and the rest is lost in different forms of energies; heat, sound energies, mechanical losses and others. Traditionally, crushing is achieved using jaw, gyratory and cone crushers while milling is mostly in tumbling mills. The size reduction ratios of these machines are not that high (in the range of 3:1 to 10:1), (Gupta and Yan, 2006), and this leads to having comminution circuits with many stages of crushing and milling depending on the target product size. On top of low reduction ratios for conventional crushers and mills, their operating costs are too high due to the complexity associated with their designs, high power draw, media costs, and water requirement. Thus innovation to lower comminution costs and improving the overall efficiency remains the chief goal for the sustainability of the mineral industry.

Recently, there has been research interest in the application of dry comminution technologies such as the high pressure grinding roller (HPGR) and vertical roller mills (VRM), that were traditionally used in the cement industry, in the mineral industry. It is evident from the findings of many researchers that both HPGR and VRM can save about 10 – 30 % of energy as compared to conventional mills (Rosario and Hall, 2010; Saramak et al., 2017; Altun et al., 2015; Genç and Benzer, 2016; Reichert et al., 2015; Wang et al., 2009). Energy savings can be attributed to the fact that compression breakage, dominating in HPGR and VRM, is more energy efficient than impact breakage, dominating in the tumbling mills (Schönert, 1979).

These machines produce less fines, but achieve desired product size (without overgrinding) and also of importance is that their progeny particles have many micro cracks (Ozcan and Benzer, 2013) which suggests less comminution energy and improved grinding kinetics in the subsequent grinding machine such as a ball mill.

The rotary offset crusher (commonly known as ROC) is a novel crushing equipment invented in 2002 by three South African engineers (Michael Hunt, Henry Simonsen and Ian Sinclair) working for the innovation company called Black Knight Technologies (Hunt et al., 2002). Figure 1.1 shows the original ROC prototype, which is no longer available. The original design concept was recently rekindled in the School of Chemical and Metallurgical Engineering at the University of the Witwatersrand and a redesigned laboratory crusher (scaled up by a factor of 1.7 – a ratio of disc diameters) has been built.

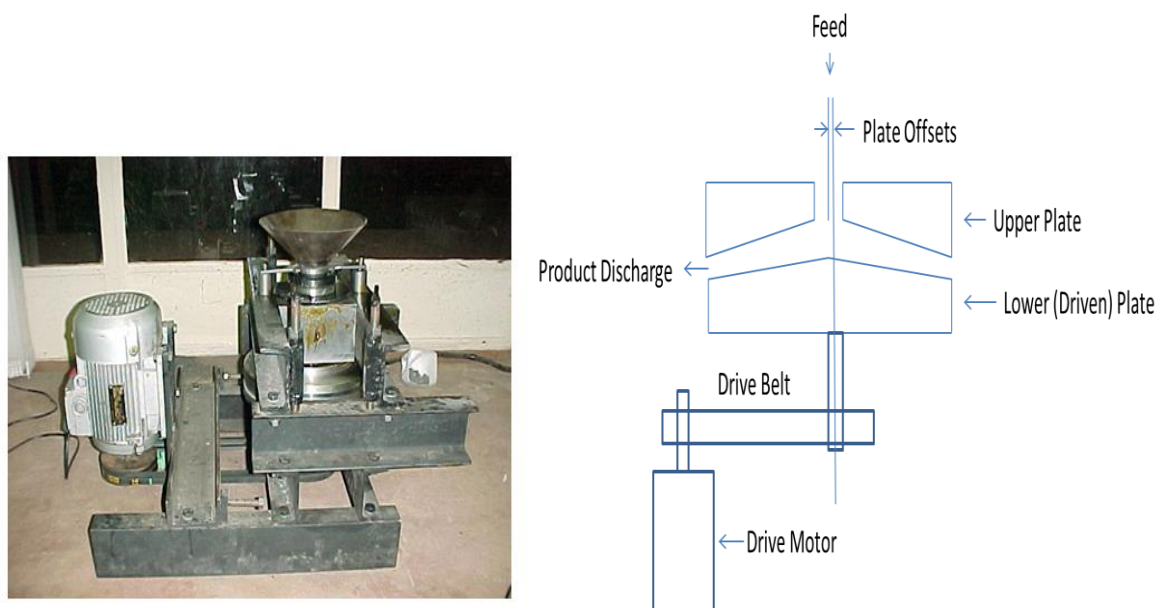


Figure 1. 1: A photograph of the original prototype and working diagram

An important design feature is the parallel cylindrical discs (labelled on the schematic diagram in Figure 1.1 as lower and upper plates) that are not mechanically linked (each has its own shaft) and, as the name of the crusher implies, the two discs (plates) are not vertically aligned, i.e. there is a horizontal offset between their respective vertical axes. In this design, only the

bottom disc is driven by the motor via the belt drive that provides an option to vary the speed by changing the diameters of the pulleys. The top disc moves due to the friction generated between its surface and the material nipped between the discs, i.e. speed synchronisation of the two discs is governed by particle locking. The effect of the rotational speed and disc offset on crushing efficiency, power draw and throughput are yet to be established.

The operating principle for the ROC is quite straightforward whereby the material is fed from the top and it gravitates through the feed chute and then it gets trapped in the crushing zone (the conical space) between the two discs. As the discs rotate, they are directly imparting energy to the particles resulting in breakage by impact, compression and abrasion and the centrifugal motion guides the transportation of particles from the centre of the discs to the periphery. The centrifugal acceleration is a function of the speeds and radii of the two discs. The higher the rotational speed, the higher the centrifugal acceleration which, in turn, implies high crusher throughput. This crushing technology employs the same breakage mechanism as the HPGR whereby particles are compressed between two discs. It is, therefore, expected that energy savings recorded for HPGR should characterise the ROC. With centrifugal motion, the ROC also promises to be a higher throughput crusher.

The maximum particle size of the crusher discharge depends, to a degree, on the space between the edges of upper and lower discs. Figure 1.2 shows the particle size distribution on the bottom disc after one of the preliminary tests. Significant high reduction ratio, approximately in the range of 15:1 to 20:1 can be estimated from this size distribution by comparing the size of the particles at the edge of the disc to those in the centre of the disc. This is compared to the reduction ratio in the range of 3:1 to 10:1 achievable with conventional crushers.



Figure 1. 2: Particle size distribution on the lower disc of the rotary offset crusher

In summary, the ROC is very simple by design, with only the discs moving, and because of this, the capital and maintenance costs are expected to be low when compared to other crushers with a similar footprint. This crusher promises a high degree of size reduction coupled with high throughput (due to the centrifugal motion) and a relatively small footprint and thus could be suitable for underground installations.

1.2 Research objectives and scope of the study

The main objective of the research was to build a laboratory prototype of the ROC to confirm the concept and deepen our understanding of the principles that describe this equipment. The ROC is a new machine and there are many aspects of this crusher that are not fully understood. Modelling of processes that affect the performance of this crusher was looked at. The kinetic approach that is based on the determined selection and breakage functions was one of the approaches considered. Measuring the energy input is one of the crucial bits of information unpacked in this study to form a basis of comparison with the conventional crushers and mills. Experiments were conducted with coal and quartz for various operating parameters (feed rate, feed size, discs offset, rotational speed, and vertical exit gap).

The review of the contribution of discrete element method (DEM) to comminution by Weerasekara et al. (2013) highlights the DEM's ability to predict the performance of the new equipment. As long as one can define the equipment internal geometry and the motion of solid bodies, then a record of interaction events can be captured to provide vast information. The information from the DEM includes; the distribution of collision energies, types of collisions, rate of collisions, the residence time of particles, and damage response of the rocks (Weerasekara et al., 2013). The DEM in our case was used to study the effect of operating parameters on the crusher throughput.

1.3 Layout of the Dissertation

The dissertation is organised in 8 chapters. The current chapter has presented the background, motivation and scope for the research.

The second chapter presents a review of literature on comminution machines in general, energy laws, fracture mechanics, single particle breakage, particles bed breakage, population balance modelling, DEM as a tool for studying transportation, power of rotating bodies, methodology for measuring the signals for load and speed, energy of flywheels and particle motions on rotating bodies. Hence, this chapter is a basis for the research work conducted and reported in the subsequent chapters.

Chapter 3 describes the major equipment and apparatus used to conduct the experiments, i.e. the ROC and its auxiliary components, the drop weight tester and piston die apparatus. The methodology employed for the preparation of the coal and quartz samples is explained as well as the experimental program followed.

Chapter 4, the first results chapter, is all about the crusher instrumentation. The circuits and Arduino codes for measuring the rotational speeds of the discs as well as for loads of the feeder and drive system are explained and some results are presented. The explanation entails the

measurement as well as the data capturing and storage. The methodologies of calculating the power draw and specific comminution energy as well as feed rate from the instrumentation data are discussed.

Chapter 5 seeks to explain the principles guiding the operation of the crusher with respect to size reduction and transportation. Effect of the disc offset on the geometry of the crushing chamber is modelled to show the variation in the exit gap. Transportation of particles in the crusher was studied using the DEM with some models described in this chapter.

Chapter 6 describes the breakage characteristics of the coal and quartz samples used for ROC experiments. The breakage characteristics were derived from the laboratory tests conducted with the drop weight tester and piston die apparatus for the impact and compression breakage respectively.

Chapter 7 presents and discusses experimental results in terms of size reduction, throughput and energy efficiency. The results were derived from the coal and quartz experiments with the ROC for various operating conditions (feed rate, feed size, disc offset, rotational speed, exit gap). Regression models for various responses (d_{80} , d_{50} , R_{80} , rate of breakage, and throughput) as a function of the operating parameters are presented. The suitability of the population balance model is also assessed. Energy balance in the ROC is conducted; primarily considering the rotational energy of the discs (calculated from the moment of inertia) and specific comminution energy.

Chapter 8, the last chapter, presents the main conclusions drawn from this research and offers suggestions for future work.

CHAPTER TWO: LITERATURE REVIEW

2.1 Comminution equipment and efficient use of energy

In the mineral industry, comminution or “size reduction” is the head operation which aims at liberating the valuable minerals from the waste constituents (gangue). Liberation is achieved by progressively reducing the size of the particles using the crushers and mills. In the crushers and mills, some forces such as compression must be applied to the rocks to ensure fragmentation happens. The breakage mechanisms encountered in comminution machines are discussed in section 2.2. Not only does comminution aim at liberating the different minerals in the ore matrix, but it also ensures the particle size suitable for subsequent separation and recovery processes is achieved. Size reduction in the mineral industry starts with blasting whereby explosives are used to effect fragmentation of the ore body. Post blasting, the run-of-mine (ROM) as large as 1.5 m is fed to a heavy duty primary crusher (jaw or gyratory crusher) and get reduced to 10 – 20 cm (Wills and Finch, 2016). The product of a primary crusher is fed to the secondary crusher and, if need be, later to the tertiary crusher. The standard and short-head cone crushers are used for secondary and tertiary crushing respectively. The product of cone crushers typically has a size ranging between 0.3 – 2 cm which is suitable for grinding (Wills and Finch, 2016).

Grinding is conventionally achieved with tumbling mills (usually fed to the rod mill operating in the open circuit which precedes a ball mill in closed circuit). Such traditional circuits are becoming less common nowadays. The current practice is to incorporate the Autogenous grinding (AG) and Semi-autogenous grinding (SAG) mills in comminution circuits to handle the primary or secondary crusher products and the AG/SAG discharge may be fed to the ball

mills for finishing or alternatively depending on the target d_{80} the AG/SAG product goes directly to the separation stage. However, problems such as the scarcity of water in some locations and high operating costs due to high energy demand and cost of grinding media in tumbling mills have necessitated the need to investigate the use of dry crushing technologies such as HPGR and VRM that were traditionally used in the cement industry (van der Meer and Maphosa, 2012; Altun et al., 2017). The operating principles for these two dry crushing technologies and cone crusher are discussed in Appendix A. Generally, the HPGR and VRM are similar in the fact that both employ the inter-particles breakage mechanism (Barrios and Tavares, 2016; Altun et al., 2017). There is now research interest in the application of both HPGR and VRM in the mineral industry because of benefits in terms of energy efficiency (Napier-Munn et al., 2005), improved degree of liberation (Reichert et al., 2015; Rosario and Hall, 2010) and increased capacity (Altun et al., 2011; van der Meer and Gruendken, 2010). The reasons for these benefits are discussed in subsection 2.1.1. These dry crushing technologies are poised to replace the secondary crushers and/or AG and SAG mills (Gupta and Yan, 2006).

2.1.1 Benefits of HPGR and VRM

The benefits of breakage mechanism (compression breakage, which is discussed in detail in section 2.2) encountered in the HPGR and VRM includes high comminution energy efficiency with researchers reporting energy savings of about 10 - 30 % compared to ball milling (Aydogan et al., 2006; Reichert et al., 2015; Altun et al., 2015; Wang et al., 2009) and improved degree of liberation (Ozcan and Benzer, 2013; Reichert et al., 2015). The energy efficiency can be explained with the early work of Schönert (1988) whose results are shown in Figure 2.1 as cited in the review paper by Rashidi et al. (2017). Schönert (1988) observed that confined particles-bed breakage encountered in HPGR is more energy efficient than impact breakage in

the tumbling mills. In the quest for energy efficient comminution machine, Schönert (1979) postulated that the most efficient method to break particles, in terms of the energy utilization, is to compress the particles in bed between two plates and this is what happens in the HPGR, VRM and ROC.

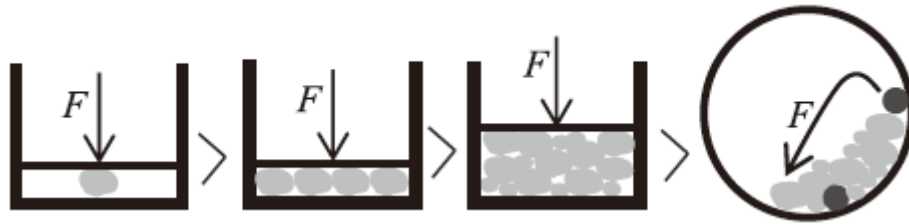


Figure 2. 1: The order of energy utilisation for various breakage mechanisms

Improved degree of liberation is attributed to the phenomenon of micro-cracking of daughter particles due to very higher stresses applied during comminution (Ozcan and Benzer, 2013; Saramak et al., 2017). The inter-particles breakage mechanism encountered in HPGR and VRM is characterised by micro-cracking along grain boundaries which ensures preferential liberation of valuable minerals and thus the product from these comminution machines is expected to respond favourably in the downstream processes such as leaching (Ghorbani et al., 2013), and flotation (Chapman et al., 2013; Saramak et al., 2017). The presence of micro cracks also suggests that if the product of HPGR and VRM is to be fed to the ball mill, for example, the energy required to effect breakage is reduced (Genc and Benzer, 2016). Barani and Balochi (2016) in their study with an iron ore also noted that pre-HPGR crushing over cone crushing improves the milling kinetics in the ball mill.

2.1.2 Why the ROC?

The compression (pressure) breakage mechanism dominates in the ROC given that particles are forced between two spinning discs which may enable this new crusher to have comparable energy savings as the HPGR and VRM. The high throughput associated with the ROC due to

the centrifugal motion of particles in the crushing zone poses an advantage over the existing crushers whose transportation primarily relies on gravity. This means that, for example, the same size reduction ratio can be achieved as with the ROC and HPGR, but the residence time for the particles is smaller in the ROC, i.e. high throughput. Energy requirement for the ROC must be quantified and compared to the existing crushers to substantiate the claim for energy savings.

2.2 Energy and comminution theory

2.2.1 Energy Theories in Comminution

Even though the comminution machine may achieve both high size reduction ratio and throughput, one of the important criteria that cannot be overlooked is energy consumption. Most of the energy input for a comminution machine is absorbed by the machine itself, and only a small fraction of the total energy is available for breaking the material (Wills and Napier-Munn, 2006). For example, the energy efficiency for the ball mill is just about 2 %, (Altun et al., 2011). It is, therefore, important that energy requirements for the comminution machines such as the ROC are correctly assessed whenever they are used whether for laboratory or pilot scale testing as the viability of the technology will heavily depend on energy efficiency. Various theories have been developed to quantify the energy responsible for effecting size reduction, i.e. the energy expended in creating new surfaces areas. This was done by deriving the relationship between the energy inputs with the particle size of the feed that give a particular product size from the comminution machine. All formulations can be described mathematically using a differential equation below that was proposed by Walker et al. (1937).

$$\frac{dE}{dL} = -kL^{-m} \quad (2.1)$$

In Eq. (2.1), dE is an infinitesimal change in specific energy, dL is infinitesimal size change, L is the particle size, k is a constant and m is a constant related to the material and the way it is broken (it takes three possible values (2, 1.5 and 1). In the following subsections, the laws defining comminution energy are discussed.

2.2.1.1 Rittinger, Kick and Bond laws

One of the earliest researchers to formulate the comminution law is Rittinger who, in 1867, postulated that the energy required for size reduction is directly proportional, not to the change in length dimension, but rather, to the change in surface area, (Wills and Napier-Munn, 2006). Replacing the constant m with 2 and integrating Eq. (2.1) results in Eq. (2.2) which is commonly known as Rittinger's law in comminution. Because the surface area is inversely proportional to the particle size, Rittinger's law is a reasonable estimation for energy when the machine is handling fine particles.

$$\int_0^{E_R} dE = -k_R \int_{d_f}^{d_p} \frac{dL}{L^2} \rightarrow E_R = k_R \left(\frac{1}{d_p} - \frac{1}{d_f} \right) \quad (2.2)$$

Where E_R is comminution energy, k_R is Rittinger's constant, d_f is the representative particle size (such as the mean) for the feed and d_p is the representative particle size for the product.

About twenty years later in 1885, Kick concluded that the energy required to reduce a material in size was directly proportional to the percent reduction, $\frac{dL}{L}$. Replacing m with 1 and integrating Eq. (2.1) results in Eq. (2.3) which is known as Kick's law. Kick's law applies to the particles that are more than 1 cm, (Wills and Napier-Munn, 2006).

$$\int_0^{E_K} dE = -k_K \int_{d_f}^{d_p} \frac{dL}{L} \rightarrow E_K = k_K \ln\left(\frac{d_f}{d_p}\right) \quad (2.3)$$

Where E_K is comminution energy and k_K is Rittinger's constant.

With the above formulations, it can be argued that neither Rittinger's law nor Kick's law is fully applicable in crushing and grinding operations because the particles, to and from the crushers and mills, are of the wider size range (with both coarse and fine particles). To overcome this shortcoming, the well-known Bond's equation was developed in the early 1950s. Bond states that the energy required in comminution is proportional to the new crack length created, (Wills and Napier-Munn, 2006) and this law covers a wide range of particle size, especially the typical particle size for the conventional mills (ball and rod mills). Bond's law is obtained by integrating Eq. (2.1) and assigning a value of 1.5 to the exponent m . After integration and replacing E with work input (W), the Bond Work Index equation (shown in Eq. (2.4)) may be derived.

$$W = 10W_i \left(\frac{1}{\sqrt{P_{80}}} - \frac{1}{\sqrt{F_{80}}} \right) \quad (2.4)$$

Where W is the work input (kWh/t), W_i is called work index (kWh/t), P_{80} is the 80 % passing size of the product and F_{80} is the 80 % passing size of the feed.

The work index is a comminution parameter which measures the grindability (hardness) of the material (Wills and Finch, 2016), but it is said to also include the mechanical efficiency of the machine (Acar, 2013). Numerically, the work index is kWh per tonne required to reduce the material from a theoretically infinite feed size to 80 % passing 100 μm (Napier-Munn et al, 2005). Bond has developed standard methods for determining the work index in the laboratory for the ball and rod mills that are listed in Gupta and Yan (2006). To estimate the crushability index, tests such as impact crushing tests of rock samples can be utilised, (Gupta and Yan, 2006).

2.2.1.2 New specific energy-particle size relationship

The Bond's equation can practically predict the specific comminution energy for size reduction with acceptable accuracy in the ball and rod mills, but as pointed out by Morrell (2004), Morrell

(2006), Morrell (2008), and Jankovic et al. (2010), this law tends to either over-predict or under-predict the specific comminution energy for Autogenous (AG) or semi-autogenous (SAG) mill, HPGR and conventional crushers. In the quest to develop an alternative equation that can be used to accurately estimate the specific energy, not only for tumbling mills, but also for jaw, gyratory and cone crushers as well as HPGR, Eq. (2.5) by Morrell, (2009) was formulated.

$$W = 4M_i(x_2^{f(x_2)} - x_1^{f(x_1)}) \quad (2.5)$$

Where W is the specific comminution energy in kWh/t, M_i is the work index related to the breakage property of an ore (kWh/t); for comminuting the product from the final stage of crushing to a P_{80} of 750 μm (coarse particles) the index is labelled M_{ia} and for size reduction from 750 μm to the final product P_{80} normally reached by conventional ball mills (fine particles) it is labelled M_{ib} and for conventional crushing M_{ic} is used while for HPGRs M_{ih} is used, x_2 is 80 % passing size for the product (μm), x_1 is 80 % passing size for the feed (μm)

The values for M_{ia} , M_{ic} and M_{ih} are obtained directly from the SMC Test[®], whilst M_{ib} values are obtained from the Bond ball work index test raw data. SMC Test[®], Morrell at The University of Queensland, is a laboratory comminution test which provides a range of information on breakage characteristics of rock samples for use in the minerals industry, (“SCM”, n.d.). For reasons of commercial confidentiality, the exact details of how to determine the M_{ia} , M_{ic} and M_{ih} values have not been published. However, there are laboratories around the globe accredited to conduct the SCM Test[®] on behalf of the SCM Test[®] (Pty) Ltd.

Comparing Rittinger, Kick and Bond's laws to Morrell's formulation, the difference is that the exponent m in Eq. (2.5) does not have a specific value, but it is rather a function of the 80 % passing sizes and according to Morrell (2006), the exponent $f(x_j)$ is estimated using Eq. (2.6).

$$f(x_j) = -(0.295 + \frac{x_j}{10^6}) \quad (2.6)$$

Where x_j is the 80 % passing size.

As discussed in Morrell (2008) it was noted that Eq. (2.5) can be modified for HPGR to be written as Eq. (2.7). The equation for estimating specific energy for HPGR is worth a look at in this study noting that the ROC is very different from conventional crushers and tumbling mills, but rather closely related to the HPGR.

$$W_h = 4K_3 M_{ih} (x_2^{f(x_2)} - x_1^{f(x_1)}) \quad (2.7)$$

Where K_3 is 1.0 for all HPGRs operating in closed circuit with a classifying screen or if the HPGR is in open circuit, K_3 takes the value of 1.19, and M_{ih} is HPGR ore work index and is provided directly by SMC Test[®].

2.2.1.3 Implication of energy laws for this study

To estimate the specific energy of the ROC, Eq. (2.7) can possibly be used given its similarity in the breakage mechanism to the HPGR. An alternative is to back-calculate the specific energy using load and no-load power values and calculating the specific energy using the operating time and feed mass as shown in Eq. (2.8). The calculated specific comminution energy can be used to validate Eq. (2.7) for the ROC application.

$$W_{ROC} = \frac{(P_L - P_{NL})}{M} \times \tau \quad (2.8)$$

Where W_{ROC} is the specific comminution energy of the ROC in kWh/t, P_L is load power (with material fed to the crusher), P_{NL} is no-load power (with no material fed to the crusher), M is mass of ore in tonne and τ is the operating time in hours.

2.2.2 Breakage mechanisms in comminution machines

Minerals are poly-crystals and it is assumed that, in comminution, they are brittle though, in reality, they can exhibit elastic behaviour (King, 2012; Wills and Napier-Munn, 2006). According to Griffith (1921), as quoted by Wills and Napier-Munn (2006), the degree of fracture caused by the applied stress does not only depend on the mechanical properties of the individual minerals in the ore, but more importantly upon the presence of cracks in the ore matrix. These cracks or flaws act as sites for stress concentration (Tavares & King, 1998) as depicted in Figure 2.2. Crack propagation, on which primary and secondary crushing rely, begins where the stress is concentrated and depending on the magnitude of the applied stress relative to the strength of the material, a degree of comminution is achieved.

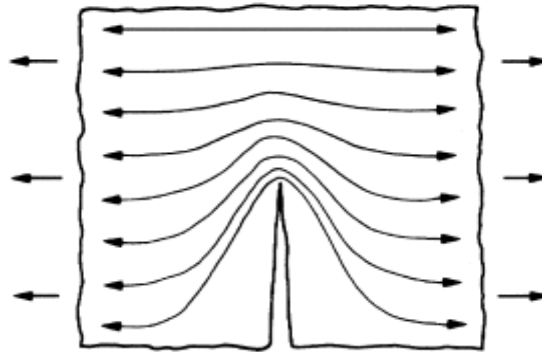


Figure 2. 2: Stress concentration at a crack tip (Wills and Napier-Munn, 2006)

According to King (2012), there are three mechanisms of fracture (shatter, cleavage and abrasion) that may be at play in comminution machines. Figure 2.3 taken from the published work of Hasan et al. (2017), reproduced from Kelly and Spottiswood (1982) depicts these three modes of breakage. These different breakage mechanisms never take place individually;

instead, they are associated with one another (Hasan et al., 2017). Moreover, the type of comminution device, operating conditions and the materials being handled greatly determine the extent of various breakage modes in a breakage event. The three breakage mechanisms are discussed in the following subsections.

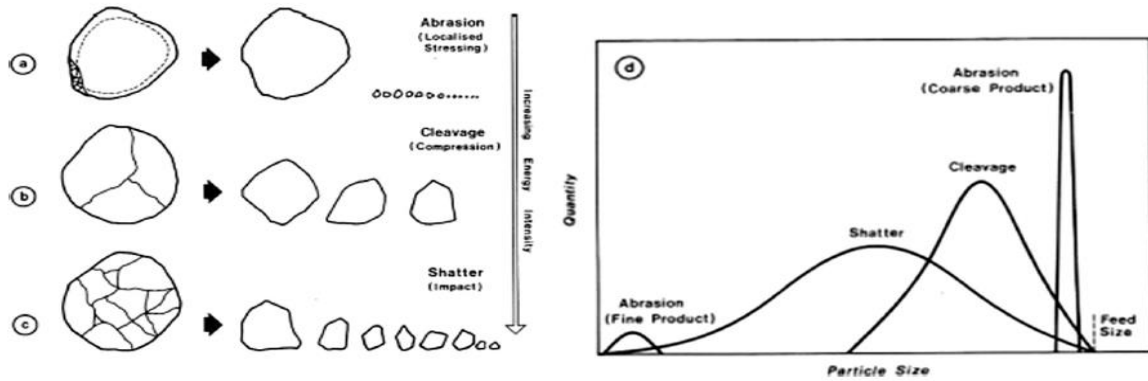


Figure 2. 3: Different breakage mechanisms and associated size distributions (Hasan et al., 2017)

2.2.2.1. Shatter

This is also called impact breakage. This mechanism of fracture happens when a larger compressive force (such as the sharp “blows” applied on the particles in impact crushers) is applied rapidly on the material resulting in the production of a broader spectrum of product sizes as shown in Figure 2.3. It is an unselective process such that multiple fracture processes occur so that progeny particles are immediately subject to further breakage by successive impacts (King, 2012). Shattering is the dominating mode of fracture in industrial tumbling mills (Wills and Finch, 2016). The key innovative feature for the ROC is the fact that the crushing action is not achieved by direct impact as is generally the case with conventional crushers (Hunt et al., 2002). Instead, it is achieved by the cyclical variation of the passage width between the opposing faces of the crushing discs. This cyclical variation suggests a combination of impact and abrasion.

2.2.2.2 Cleavage

Another name for this type of breakage mechanism is compression. According to the Oxford English dictionary, the word "cleavage" in fracture mechanics can be defined as "The splitting

of rocks or crystals in a preferred plane or direction". Those planes are mainly caused by the presence of weaker bonds between atoms in the crystal lattice and they are shown as lines on the particle in Figure 2.3. Unlike the shattering process, cleavage fracture tends to produce large daughter particles of the same grain size as the parent particles as well as much finer progeny particles that originate at the points of application of the stress.

As discussed in section 2.1.1, this type of breakage dominates in the HPGR. In HPGR, the particles fracture by compression in a packed particle bed (ensured by choke feeding), not by direct nipping of the individual particles between two rolls, (Gupta and Yan, 2006; Altun et al., 2017). This suggests that the applied load moves inwards from the roll-bed interfaces to the particles through the inter-particles breakage mechanism. As the particles get trapped between the two discs of the ROC, breakage by compression, can be expected.

2.2.2.3 Abrasion

Abrasion is regarded as a surface phenomenon which takes place when particles move parallel to their plane of contact (Napier-Munn et al., 2005) and it occurs when insufficient energy is applied to cause fracture of the particles. As parent particles are colliding with each other, small pieces are breaking off from their surfaces resulting in the generation of finer progeny particles. It is worth mentioning that the parent particles hardly change sizes even after the production of fines (see Figure 2.3) and this being the case, abrasion is a birth process because of the appearance of small particles, but unlike the situation with shatter and cleavage breakage there is no corresponding death process (King, 2012). As shown in Figure 2.3, this mode of breakage results in a bimodal particle size distribution which shows that, for a breakage event, progeny particles close in size to the parent size are produced along with fine particles. The common example of abrasion is attrition which is mainly encountered in Autogenous mills (AG and SAG) where large particles act as media. What happens in the mill is that fine particles are nipped between the large particles resulting in the generation of many fine particles. In the

ROC, abrasion is to be expected due to the discs-particles and particle-particle contacts as well as sliding. While abrasion breakage due to particles-particles contact is desirable, abrasion breakage due to discs-particles contacts may be detrimental because such contacts contribute to wear. It is not within the scope of this research to quantify the wear rate and costs thereof.

2.2.3 Single particle breakage

The most efficient size reduction method is single particle breakage as the energy losses by friction and unsuccessful impact events are minimized, or avoided altogether, and losses due to particle–particle interactions do not exist (Tavares, 2004). Some of the common devices used to conduct single particle impact breakage experiments differing from the mode of application of stresses and the number of contact points as summarised by Chikoshi (2017) include: (1) drop weight tester (DWT), (2) Split Hopkinson pressure bars (SHPB), (3) Impact load cell (ILC), and (4) Rotary breakage tester (RBT). The drop weight tester that was used to conduct experiments in this study is discussed in detail below.

2.2.3.1 Drop weight tester

A drop weight tester (DWT) is the simplest and most commonly used device for characterising materials in terms of breakage. During the drop weight test, fracture is achieved by dropping a weight (usually a steel disc) from a known height onto a single particle (or a bed of particles) that is placed on a hard surface (anvil). Figure 2.4 presents the working principle of a typical drop weight tester.

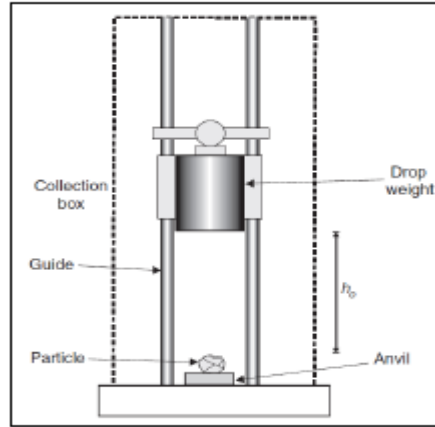


Figure 2. 4: Working diagram of the drop weight tester (Tavares, 2007)

The available potential energy, before the weight is released, is transferred to kinetic energy which is then transferred to a particle placed in the centre of the anvil. Depending on the magnitude of the impact energy and the hardness of the particle, breakage may or may not occur. The daughter particles are sieve analysed to determine the product size distribution. To quantify the energy input, Eq. (2.9) formulated by Napier-Munn et al., (1996) as cited in Genc et al. (2004) can be used, and by dividing the energy input by the sample mass, the specific comminution energy can be calculated using Eq. (2.10). By changing the drop weight and/or height, a wide range of input energies can be achieved. Coal and quartz samples (materials used to conduct test works with the ROC) were subjected to various impact energy levels (discussed in Chapter 3) and results are discussed in Chapter 6.

$$E_i = m_d g (h_i - h_f) \quad (2.9)$$

Where E_i is the energy input (J), m_d is the mass of the drop weight (kg), h_i is the initial height of the drop weight above the anvil (m) and h_f is the final height of the drop weight above the anvil (m).

$$E_{cs} = \frac{E_i}{m_p} \quad (2.10)$$

Where E_{cs} is the specific comminution energy and m_p is the mean particles mass (g).

2.2.3.2 The t_{10} breakage model

The data from the DWT can be used to determine the breakage and energy utilisation parameters for comminution modelling (Napier-Munn et al., 2005). While this is mainly important for SAG modelling, it can relatively give an indication of the hardness of the ore. One important parameter is the t_{10} parameter, known as a product fineness that is related to the specific comminution energy as shown in Eq. (2.11). The t_{10} parameter represents the cumulative mass passing the 1/10 of the feed size. For the mono-size classes, the t_{10} size is the 10th of the geometric mean for that size interval. Typically, in a crusher, t_{10} is 10 to 20 %, whereas in tumbling mills values in the range 20 to 50 % are expected, (Bearman et al., 1997). The relationship in Eq. (2.11) is graphically shown in Figure 2.5.

$$t_{10} = A \times (1 - e^{-bE_{cs}}) \quad (2.11)$$

Where A and b are material specific impact breakage parameters.

The impact breakage parameters, A and b, can be determined through the interpretation of a typical t_{10} - E_{cs} curve shown in Figure 2.5. Parameter A is the maximum value of t_{10} , i.e., the highest level of size reduction from a single impact event, typically varying from 35 to 70 %, (Magalhaes and Tavares, 2014). Parameter b is the slope of the linear part of the curve and that it is related to material stiffness (Napier-Munn et al., 2005). Stiffness is defined as the extent to which a material resists deformation in response to an applied force (Baumgart, 2000).

The product of the model parameters (A and b) is used as an indicator for the ore hardness. This product indicates the material's ability to resist impact breakage (Shi and Kojovic, 2007). The product of A and b can be obtained by differentiating Eq. (2.11) and equating the derivative to zero. This results in Eq. (2.12) which shows that the product of the breakage parameters (A and b) is the slope of the curve at 'zero' input energy. A lower A x b value shows that the ore has a high resistance to impact breakage, i.e. a hard ore. For two or more different ores, their

amenability to fragmentation by impact can be investigated and compared by calculating this product ($A \times b$). The products $A \times b$ were computed for the coal and quartz that were used to assess the ROC's efficiency and results are discussed in Chapter 6.

$$\lim_{E_{cs} \rightarrow 0} \frac{dt_{10}}{dE_{cs}} = A \times b \quad (2.12)$$

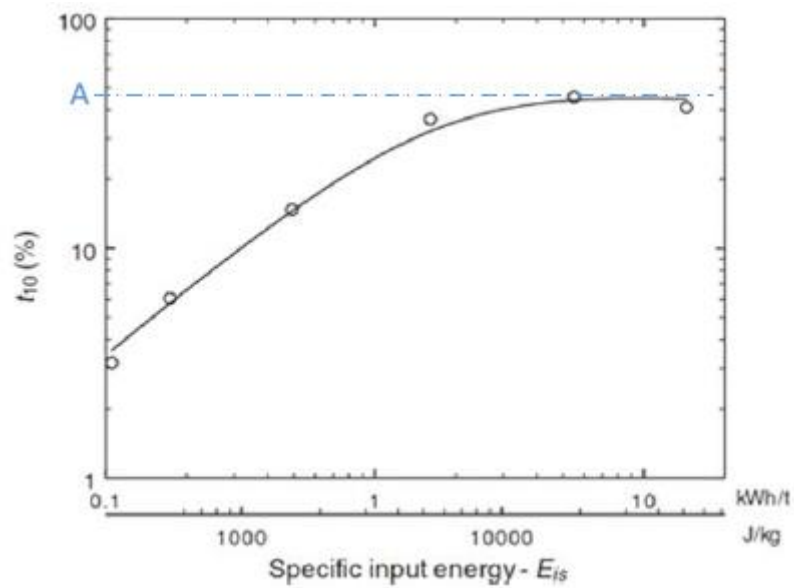


Figure 2. 5: Relationship between the parameter t_{10} and specific input energy (E_{cs} or E_{is}) (adapted from Tavares (2007))

2.2.4 Particles bed breakage

The drop weight tester discussed in section 2.2.3 above, is mainly used to characterise the ores in terms of impact breakage. In the case of compression breakage (schematically shown in Figure 2.6), a piston die apparatus is used. The piston die cell is used extensively in literature to study compression breakage (Fuerstenau et al., 1996, Viljoen et al., 2001, Oettel et al., 2001, Barrios et al., 2011, Esnault et al., 2015) or specifically for machines such as VRM (Shahgholi et al., 2017), in HPGR (Dundar et al, 2013, Davaanyam, 2015). Conducting experiments with

the piston die tester simply involves putting the weighed sample in the crushing chamber and applying a known pressure via the press.

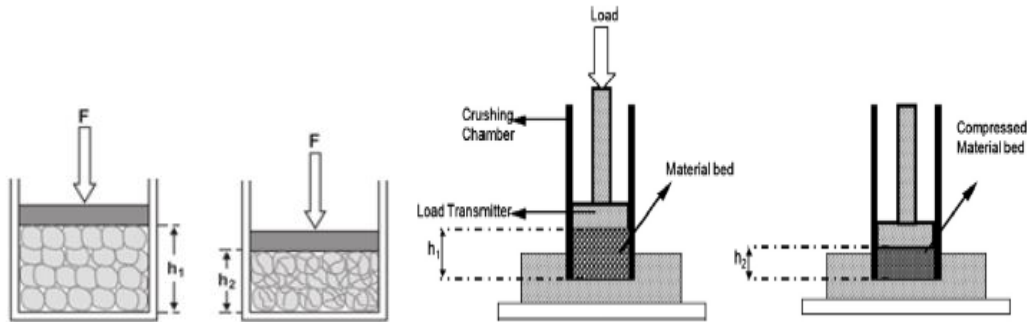


Figure 2. 6: Representation of the particle bed comminution and piston-die tester (adapted from Ozcan and Benzer, 2013)

The displacement, i.e. the difference between the initial and final heights (h_1 and h_2 respectively shown in Figure 2.6), is plotted versus the applied force to get the energy. A typical plot is shown in Figure 2.7. This energy, which is the work done on particle bed, is the area under the force-displacement curve which can simply be represented by Eq. (2.13).

$$E = \int_{h_1}^{h_2} f(x)dx \quad (2.13)$$

Where E is work done (Joules), dx is the differential displacement (m), $f(x)$ is the force as a function of displacement (Newton), h_1 is the typically 0 (the initial displacement) and h_2 is the final displacement (in m), for example on Figure 2.7, h_2 is equal to d_5 .

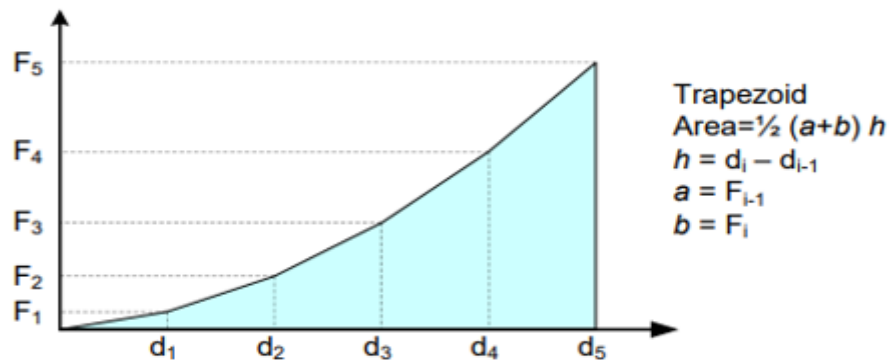


Figure 2. 7: Example showing application of numerical integration to evaluate the specific comminution energy (Davaanyam, 2015)

The typical piston die test yields the following results, (Esnault et al., 2015, Dundar et al., 2013, Davaanyam, 2015):

- Product size distribution.
- Specific comminution energy corresponding to the applied pressure.
- Thickness of the particle bed before and after compression.
- Bed porosity and hence density.

In this study, however, the analysis of the data from the piston die tests focused only on the product size distributions and specific comminution energy. These are compared to results obtained using the rotary offset crusher and drop weight tester.

2.3 Modelling grinding rate in comminution and the ROC approach

The study of grinding in the ROC can be treated as kinetics rate process on which population balance modelling (PBM) is based. The population balance model (PBM), introduced in comminution by Epstein (1947), as cited in Dundar et al. (2013), and Napier-Munn et al. (2005), is basically a simple mass balance for the size reduction process. Generally, the size reduction process in comminution machines is described mathematically by taking the following three steps into consideration (Gupta and Yan, 2006):

1. A particle is selected for breakage (selection, or disappearance function, or a breakage rate),
2. The broken particle produces a given distribution of fragment sizes (breakage or appearance function), and
3. Differential movement of particles through, or out, of a continuous mill (discharge rate or classification function).

In the case of batch grinding, as with the experiments conducted in this research, the size-mass balance (population balance model) is formulated as follows (Austin et al., 1984):

$$\frac{dm_i(\tau)}{d\tau} = -S_i m_i(\tau) + \sum b_{ij} S_j m_j(\tau) \quad (2.14)$$

Where m_i is the mass of material in size class i , S_i is the rate of breakage of size class i , b_{ij} is the breakage function for size class i , S_j is the rate of breakage of size class j , m_j is the mass of material in size class j , and τ is the grinding time.

The solution of Eq. (2.14) as reported in Austin (1971a) is shown in Eq. (2.15) which can be used to predict the product size distribution provided that the experimentally determined values for the selection and breakage functions are known.

$$p_i = f_i + (\sum_{j=1}^i b_{ij} S_j m_j) t - S_i m_i t \quad (2.15)$$

Where p_i and f_i represent the masses of product and feed for size class i respectively and t is the residence time.

In the following sub-sections (2.3.1 and 2.3.2), the mathematical formulations and implications for the selection and breakage functions are discussed.

2.3.1 Selection Function

The selection function is defined as the disappearance rate of a specific size after breakage and it is expressed as tonnes per hour broken per tonne in the mill ($\text{tph/t} = 1/\text{h}$), (Dundar et al., 2013). It is also called the breakage rate, or probability of breakage, of the material. With the above definition, the selection function of size class i can simply be expressed mathematically as:

$$S_j = \frac{m_i}{t \times m_t}, j > i \quad (2.16)$$

Where S_j is a selection function for size class j , m_i is a mass of a lower size class i , m_t is a total mass of the material and t is a residence time of the material in the mill.

According to Austin, (1971a) the rate of breakage can be calculated by assuming that the breakage process is analogous to a first-order chemical reaction. With this assumption, milling kinetic for a single size feed can be written as follows, (Austin, 1999):

$$\frac{dw_j(t)}{dt} = -S_j w_j t \quad (2.17)$$

Where S_j is the rate of disappearance of particles (or selection function) of size class j , w_j is the mass fraction present in the size interval j after crushing time t ; j is an integer defining the different size intervals, the largest being 1.

Rewriting Eq. (2.17) as Eq. (2.18) for the first size class, and carrying out integration results in Eq. (2.19) showing clearly the relationship between w_i , S_i and t .

$$\int_0^t \frac{dw_1(t)}{w_1(t)} = -S_1 \int_0^t dt \quad (2.18)$$

$$\log(w_1(t)) = \log(w_1(0)) - \frac{S_1 t}{2.3} \quad (2.19)$$

On the log-log scales, the milling time versus $\frac{w_1(t)}{w_1(0)}$ can be plotted as in Figure 2.8 for normal breakage. The breakage rate (S_j) can then be evaluated from the slope of the straight line. Graphs of various size classes need to be drawn and their respective slopes (S_j) calculated. Following this intensive process of milling and drawing the graphs, the selection functions may

be plotted against the particle size to show how the breakage rate changes from coarser to finer particles at the operating conditions. Example of such a plot is shown in Figure 2.10.

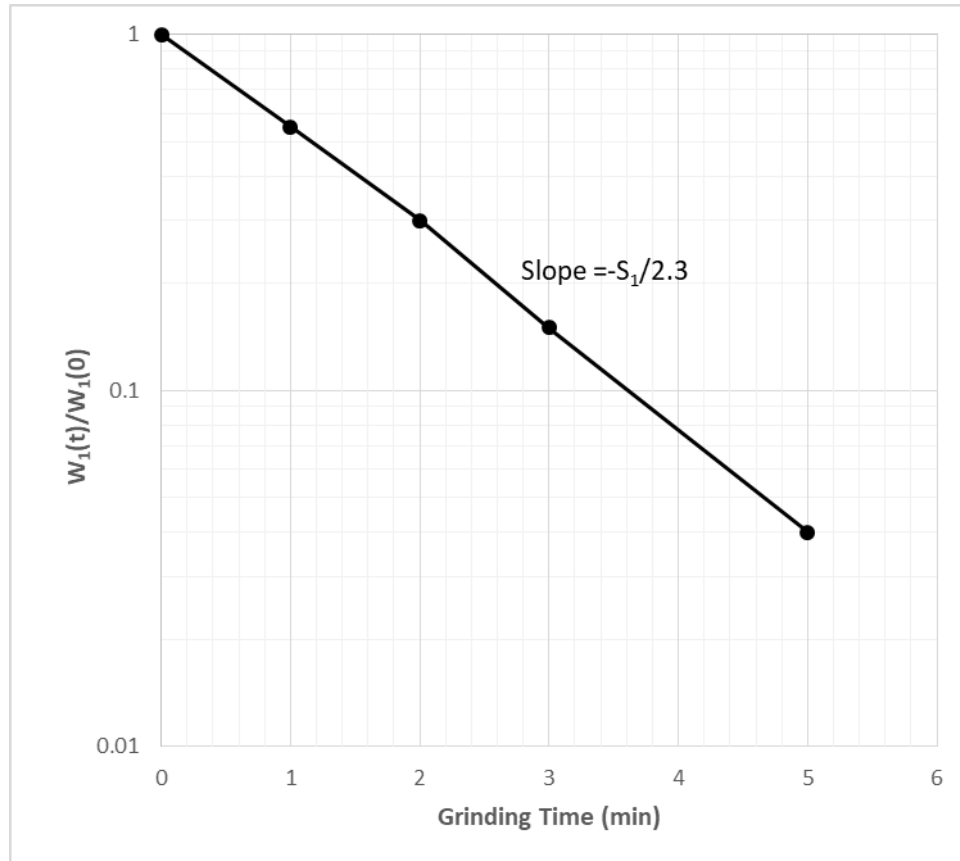


Figure 2. 8: First- order reaction model applied to milling normal breakage (redrawn after Austin et al., 1984)

2.3.1.1 Abnormal Breakage

It is worth noting that at times, especially with coarser particles, abnormal breakage is encountered. Abnormal breakage implies that the milling kinetic is not described by Eq. (2.17). Figure 2.9 summarises the non-first order milling kinetics from literature as contained in the literature survey for the PhD thesis of Chimwani (2014) that is cited from the work of Bilgili et al. (2006). Austin et al. (1973) attempted to investigate this abnormal breakage in ball mills. What came to light from their study was that abnormal breakage increases with particle size in proportion to the increase in ball and mill diameters.

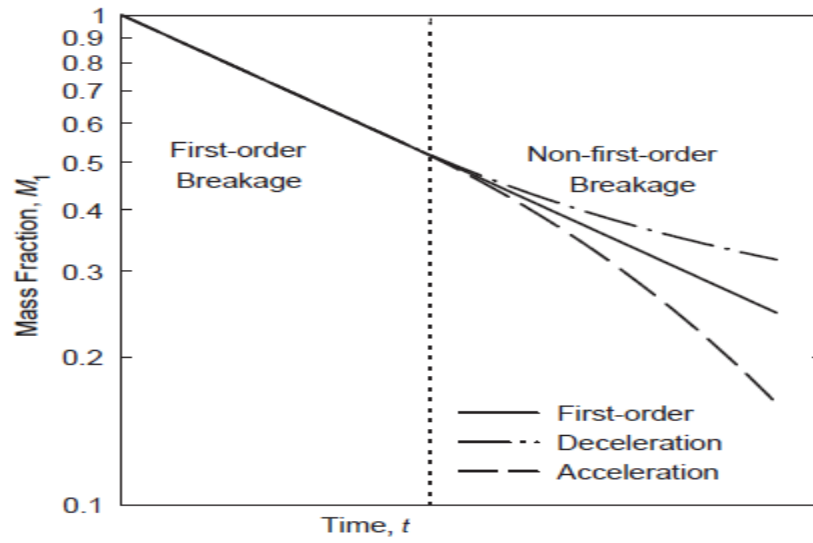


Figure 2. 9: Non-first order milling of narrow sized feed after Bilgili et al., 2006 (Chimwani, 2014)

To consider the deviation of the rate of breakage from Eq. (2.17), Austin et al. (1984) formulated Eq. (2.20) which is graphically shown in Figure 2.10. According to Austin et al. (1984), this deviation from the first-order kinetic equation is due to the conditions of the physical grinding environment.

$$S_j = ax_j^\alpha \frac{1}{1 + \left(\frac{x_j}{\mu}\right)^\wedge} \quad (2.20)$$

Where x_j is the maximum limit in screen size interval j in mm; $\wedge$ and α are positive constants which are dependent on material properties; a is a parameter dependent on mill conditions and material properties, and it shows the kinetic of the milling process, and μ is a parameter dependent on mill conditions.

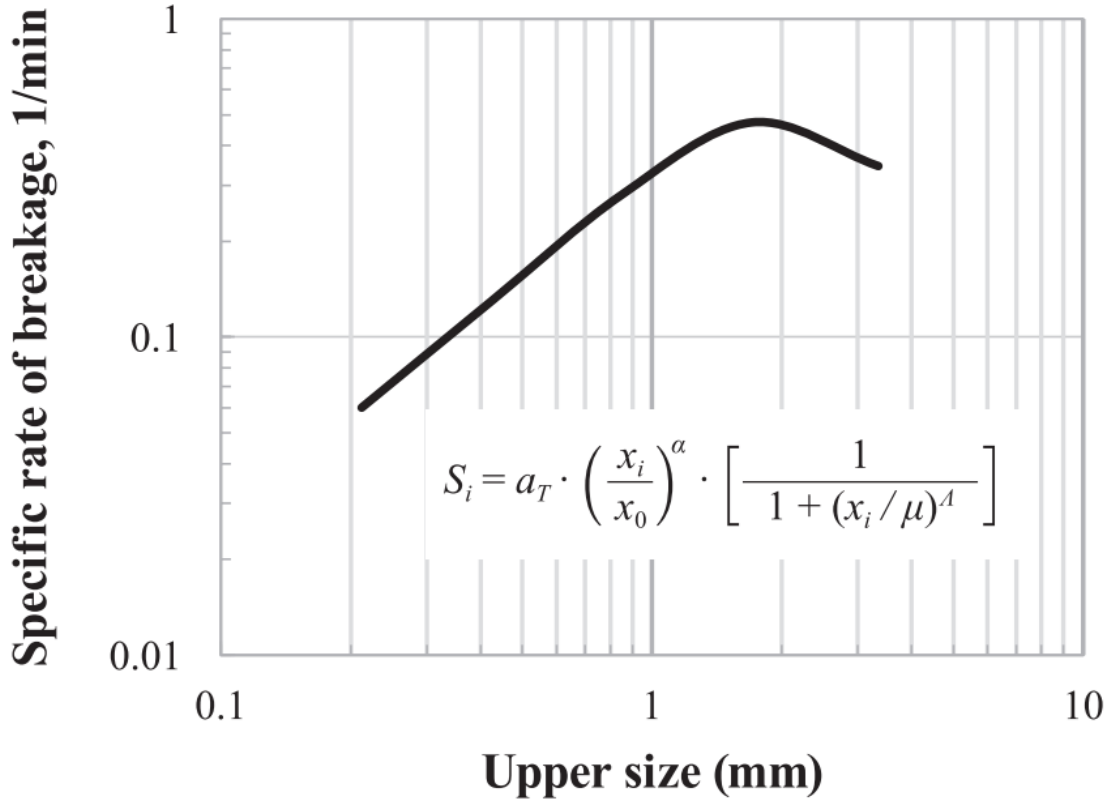


Figure 2. 10: Variation of selection function with particle size (Ozkan et al., 2003)

2.3.1.2 Back-calculation of rate of breakage

If the feed and product size distributions are known in addition to the breakage function, the selection function can be back-calculated using Eq. (2.15). This method of estimating the rate of breakage was initially proposed by Klimpel and Austin (1977) with their results, in Figure 2.11, showing a good correlation between experimental and calculated data. Recently, Dunder et al. (2013) used the same approach to back-calculate the rates of breakage for the cement samples in the HPGR and the validation tests showed high reliability of the calculated rates of breakage.

For example, to calculate the selection function of the first size class, Eq. (2.15) can be modified resulting in Eq. (2.21).

$$p_1 = f_1 - S_1 m_1 \tau \quad (2.21)$$

Making S_1 the subject of the formula, Eq. (2.22) is obtained which can be used to get the rate of breakage for the first size class.

$$S_1 = \frac{f_1 - p_1}{m_1 \tau} \quad (2.22)$$

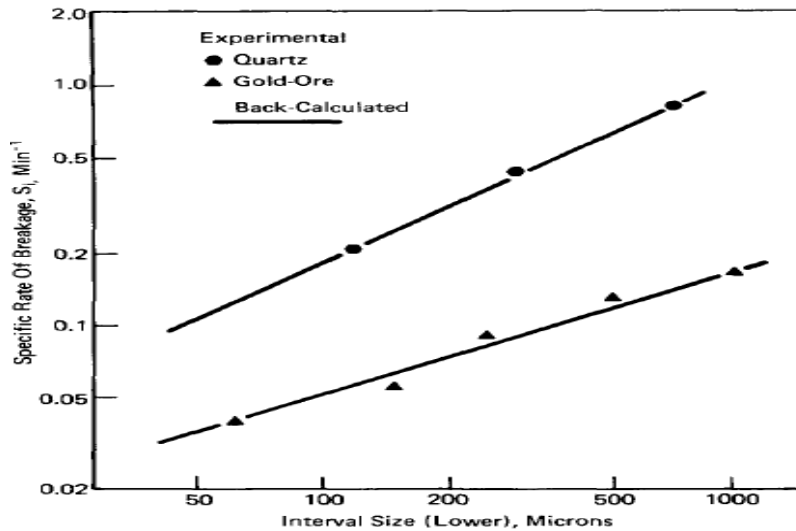


Figure 2. 11: Comparison of experimental and back-calculated rates of breakages (Klimpel and Austin, 1977)

2.3.2 Breakage function

Also called the primary distribution function, in simple term, this may be defined as the average size distribution resulting from the fracture of a single particle, (Kelly and Spottiswood, 1990). Usually, but not always, breakage pattern for a given material is uniform and, therefore, the material can be said to be “normalisable”, (Mulenga, 2012). With breakage pattern being a material property, it can be argued that it does not depend on the milling environment.

For each size class, after breakage, the relative size distribution of the product (denoted as $b_{i,j}$) is the basis of breakage function and it is represented symbolically in Eq. (2.23).

$$b_{i,j} = \frac{\text{mass of particles from class } j \text{ broken to size } i}{\text{mass of particles of class } j \text{ broken}}, \text{ where } i < j \quad (2.23)$$

Where i and j are size classes.

The breakage functions may directly be calculated knowing the feed mass and the size distribution after crushing or milling. Several methods were developed by several researchers of determining breakage functions in mills and they are summarised in Austin (1971b) and Gupta and Yan (2006). Some of the laboratory devices (listed in Table 2.1) have been developed for different modes of breakage and they can be used to estimate the breakage parameters. For examples in studies by Dundar et al., (2013), Esnault et al., (2015) and Shahgholi et al. (2017), the piston die tester was used to estimate the breakage parameters for some materials (cement clinker, quartz and limestone) under compression. In the case where the mode of breakage is not known for the machine, the laboratory tests may be conducted, and product size distributions are compared to the size distribution of machine discharge. This can give an indication of the dominating mode of fracture in that machine.

Table 2. 1: List of modes of breakage and testing devices (Weerasekara, et al., 2013)

Mode of breakage	Device	Mode of Operation	Particle size range (mm)	Energy range (kWh/t)
Single impact body breakage	JKDWT	Single particle	12 - 70	0.1 – 2.5
	JKBT	Single particle	5 - 45	0.1 – 3.5
Bed breakage	Piston and die	Many particles	2 - 20	0.1 – 2.0
Abrasion surface damage	Powell abrasion mill	Many particles	20 - 250	0.004 – 0.1

Conveniently, sometimes breakage functions are represented as cumulative breakage functions, defined by Eq. (2.24). Cumulative breakage function ($B_{i,j}$) is the cumulative mass fraction of particles passing the top size of interval i from breakage of particles of size j . Various methods developed to estimate the $B_{i,j}$ values are given in, Austin, (1971b).

$$B_{i,j} = \sum_{k=i+n}^n b_{k,j} \quad (2.24)$$

The cumulative breakage function of the material is said to be independent of the initial particle size (i.e. it is normalisable) and, therefore, generally it is fitted to the empirical model by shown below (Klimpel et al., (1984).

$$B_{i,j} = \Phi_j(R)^\gamma + (1 - \Phi_j)(R)^\beta \quad (2.25)$$

Where R is a relative particle size given by $\frac{x_i-1}{x_j}$ ($j > i$, where i and j represent size classes), β is a material-dependent parameter (whose values usually range between 2.5 and 5 according to Austin et al. (1984), but as reported in some studies β can be greater than 5, for example, Ozkan et al. (2003) reported the β value of 6.4 for the lignite coal and Petrakis et al. (2017) reported a β value of 5.8 for quartz), γ is also a material-dependent parameter with values between 0.5 and 1.5 according to Austin et al. (1984), and finally Φ_j is a fraction of fine particles resulting from a single breakage event and it is also dependent on the material. The value of γ is related to the relative number of fines produced and thus related to the milling efficiency (Petrakis et al., 2017) while Φ and β are related to the coarser end of the breakage distribution function and show how fast parent particles migrate to the smaller size classes.

These parameters can be experimentally determined by plotting $B_{i,j}$ values versus relative particle size (R in Eq. (2.25)) on log-log scales (as shown in Figure 2.12). This plot is plotted based on an assumption that the breakage distribution function is independent of the initial particle size; implying that material was assumed to be “normalisable” (Kelly and Spottiswood, 1990). Taking into account this assumption Φ is a constant value, i.e. it is not a function of particle size j . The slope of the lower straight-line part of the curve gives the value of β , the

slope of the upper part of the curve gives the value of γ , and Φ is the intercept, (Kelly and Spottiswood, 1990).

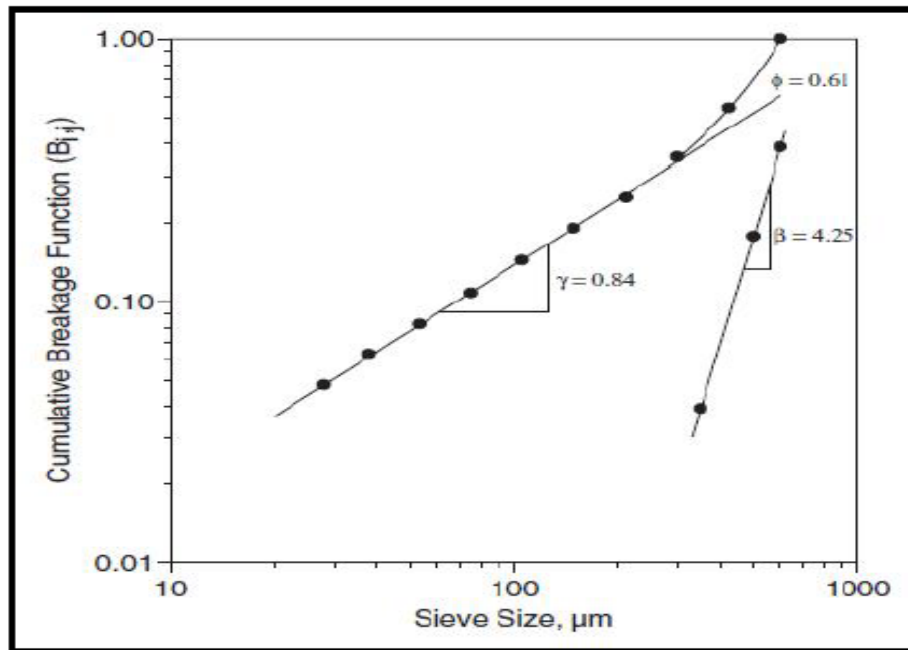


Figure 2. 12: An example of cumulative breakage function versus particle size (Mulenga, 2012)

2.4 DEM as a tool for studying transportation

The dynamic behaviour of the particles in the ROC can be studied using computational methods such as the discrete element method (DEM). The DEM is a numerical method that is widely used to study the motion and collisions of particles. The initial work on the DEM originated in the early 1970s with some work done on rocks modelled in 2D, (Jing and Stephansson, 2007). Since then the DEM has become widely accepted as an effective method of addressing engineering problems in granular and discontinuous materials, especially in granular flows, powder mechanics, rock mechanics, and comminution (Weerasekara et al., 2013).

The information that is generated using the DEM includes the distribution of collision energies, types of collisions, rate of collisions, residence time of particles, discharge rate from the device

and damage response of the rocks, which is used to predict the key quantities in comminution machines such as flow patterns, power consumption, impact and abrasion forces on particles and equipment, wear and stress patterns on surfaces, breakage rates (Weerasekara et al., 2013; Bharadwaj, 2014). Application of DEM to comminution machines include the studies by Bruchmuller et al., 2011, Cleary and Morrison, 2011, Weerasekara et al., 2016 who have used this numerical method to study breakage in tumbling mills. For crushers, Clearly and Sinnott (2015) used the DEM to study the particle flow and breakage in jaw, gyratory, cone and roll crushers while Quist and Evertsson (2010), Li (2013) and Johansson et al. (2017) focused only on cone crushers to understand the particle flow and breakage. In his PhD thesis, Quist (2017) compared the performance of cone crusher and HPGR using the DEM simulations. In our case, the DEM was only used to understand the particle flow in the ROC, i.e. breakage was not simulated. The simulation results were then validated using the data obtained from the experiments conducted with the laboratory crusher.

There are many open-source and commercial DEM codes available. Some of the DEM software are listed in Table B1 in Appendix B. The particle flow code (PCF3D) available at the University of Witwatersrand was used to study the particle flow behaviour in the ROC.

2.4.1 Spring-dashpot contact model

The DEM uses Newton's laws of motions to get information on particle flow as well as the contact laws to resolve contact forces (considering the contacts among the particles and those between particles and the surface of the equipment). The total force acting on individual particles, given by Eq. (2.26), is a summation of the contact and body forces.

The contact force is the summation of the tangential and normal forces acting on the particle (Thornton et al., 2013) while the body forces include all other forces such as gravity, electrostatic and magnetic forces.

$$\sum F = F_{\text{contact}} + F_{\text{body}} = m\bar{a} \quad (2.26)$$

Where m is the mass and $\bar{a}$ is the acceleration.

A review of the contact models by Thornton et al. (2013) highlighted that the most commonly used model in the DEM is the spring-dashpot (damped harmonic oscillator) model. What happens with this model is that particles are allowed to overlap and the amount of overlap (Δx) and the normal (v_n) and tangential (v_t) relative velocities determine the collision forces, i.e. normal and tangential forces that are calculated using Eq. (2.27) and (2.28) respectively.

$$F_n = -K_n \Delta x + C_n v_n \quad (2.27)$$

$$F_t = \min\{\mu F_n, \sum K_t v_t \Delta t + C_t v_t\} \quad (2.28)$$

Where F_n and F_t are normal and tangential forces to the contact plane, μ is the coefficient of static friction, K_n and K_t are stiffness of the spring in the normal and tangential directions respectively and they determine the maximum overlap between the particles and it is a function of particle size and material properties such as Young modulus and Poisson ratio (Bharadwaj, 2014), C_n and C_t are normal and tangential damping coefficients respectively and they relate to the coefficient of restitution ϵ . The integral term in Eq. (2.28) represents an incremental spring that stores energy from the relative motion and models the elastic tangential deformation of the contacting surfaces (Cleary and Sinnott, 2015).

The coefficient of restitution is defined as the ratio of the post-collisional to pre-collisional normal component of the relative velocity, (Cleary and Sinnott, 2015). Thornton et al., (2013) explained how the value of ϵ can be estimated. The equations for calculating the spring stiffness constants and damping coefficients as listed in Weerasekara et al. (2013) are given in Table B2 in Appendix B.

2.4.2 Simulation Methodology

To get started with DEM simulation, the first step is to create the internal geometry of the equipment using the CAD package and import this as a triangular surface mesh into the DEM software. The boundary motions of the moving parts such as the two discs for the ROC should then be defined. The next step is the release of the particles into the equipment within the domain at certain initial coordinates and then simulation advances using small incremental time steps. The behaviours of individual particles in the equipment are modelled to help predict the behaviour of the bulk. This is done by estimating the total force (given by Eq. (2.26)) acting on each particle and subsequently the accelerations, velocities and positions of those individual particles at every instant.

2.5 Power of Rotating Systems and Energy of flywheels

The power utilised in rotating object and systems, such as flywheels (the two discs for the ROC), is equal to a product of the torque and angular velocity. Noting that the speed of rotation is commonly expressed in revolution per minutes (rpm), the shaft power can be calculated as follows:

$$P = \frac{2\pi nT}{60} \quad (2.29)$$

Where P is the shaft power in Watts, n is a speed in rev/min and T is torque in Nm.

Torque can simply be defined as a measure of the forces that cause an object to rotate. There are two types of torque: reaction and rotational torques. While with the rotational torque the force applied rotates the load, with the reaction torque the force is acting on the object that is not free to move. The ROC involves the rotational torque with the discs freely spinning in the angular direction.

During a cycle of operation, both the angular velocity and torque can be varying with time even though most machines are designed to operate at constant or near-constant speeds for larger blocks of operating time, (Norton, 2006). Taking that into account, the average power can be calculated using the equation below.

$$P_{av} = T_{av}\omega_{av} \quad (2.30)$$

Where ω_{av} is the average angular velocity and T_{av} is the average of the torque during the cycle of operation.

2.5.1 Measurement of various signals

To calculate the mechanical power of the ROC, there is a need to measure the rotational speeds for the discs and the motor drive torque. Transducers (sensing devices) are used to make such measurements in real time. The IR-based sensor and strain gauge load cells installed on the crusher are discussed in the following subsections.

2.5.1.1 Using the IR rays to estimate the speed

The infrared (IR) sensor is made up of an emitter, the IR light emitting diode (LED), which sends the IR rays to the rotating shaft (having been designed to have both reflective and non-reflecting surfaces) such that when the light IR hits the reflective surface, it gets reflected to the receiver (also called photodiode or phototransistor) as shown in Figure 2.13. On the other hand, if the surface is non-reflective the emitted rays get absorbed by the surface implying that

the receiver does not get any signal. The digital signal is fed to the controller for processing and the time difference between either two successive HIGHS or LOWs can be used to estimate the rotational speed in rpm. Some advantages for the IR sensor include; non-contact operation (no wear or friction), high speed operation (up to 100 kHz), when packed it is immune to dust and water, can measure zero speed, and measurements are reproducible.

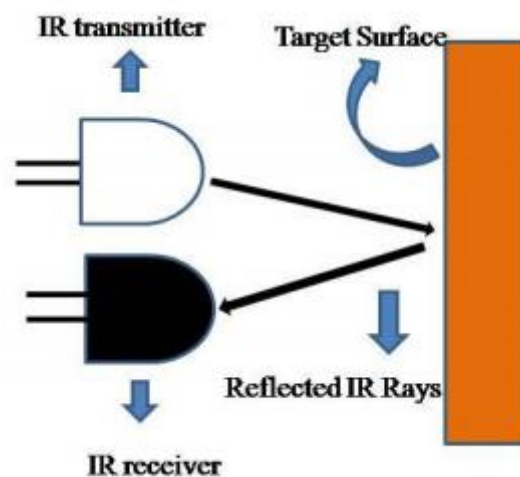


Figure 2. 13: Working principle of the IR sensor (Mollah, 2016)

2.5.1.2 Using strain gauge load cell to measure the load

Two strain gauge load cells were installed on the ROC for measuring the loads. One load cell is installed perpendicular to the axis of the driving pulley to measure the force required to keep the torque arm stationary. From measured force, the torque can be calculated. The other load cell is installed to measure the weight of the material on the conveyor. This enables the computation of the crusher feed rate. Chapter 4 discusses in detail the installation of the load cells on the crusher as well as the framework for capturing the data and how the torque and feed rate were calculated. The working principle of the load cell is discussed below.

The load cell has an elastic structure that deforms whenever a force is applied to it as shown in Figure 2.14 (The Institute of Measurement and Control, 2013). It is a common knowledge that

an elastic structure would give a linear relationship between the applied stress and the resulting strain for as long as the yield point on the stress-strain curve is not exceeded. The amount of force applied is calculated by taking the difference between compressed and uncompressed measurements. Hooke's law, in Eq. (2.31) is applied to compute the magnitude of the force.

$$F = k(x_c - x_0) \quad (2.31)$$

Where F is the applied force, k is the slope of the stress versus strain graph; x_0 and x_c are the uncompressed and compressed lengths respectively.

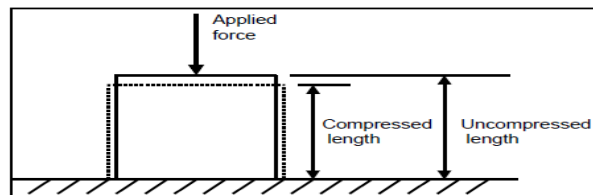


Figure 2. 14: Basic structures of an elastic element in a load cell (The Institute of Measurement and Control, 2013)

Within the load cell, there are electrical resistance strain gauges that are intimately bonded to the elastic structure making it possible to change the force applied into voltages. The strain gauges are connected in a four-arm Wheatstone bridge configuration (circuit diagram in Figure 2.15). The R_1 , R_2 , R_3 and R_4 in Figure 2.15 represent the strain gauges or just resistors (Meyer, 2016).

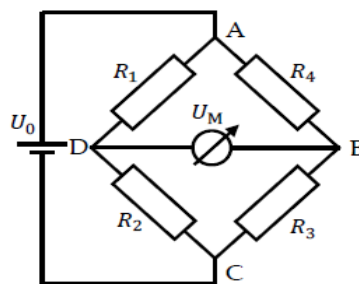


Figure 2. 15: Wheatstone bridge configuration (Meyer, 2016)

Considering the Wheatstone bridge circuit in Figure 2.15, the output voltage can be calculated. If there is no load, i.e. when the circuit is balanced, then $R_1R_3 = R_2R_4$ (Meyer, 2016). When a force is applied to the load cell, the Wheatstone bridge becomes unbalanced and give output voltage between B and D. With the voltage divider rule, this output voltage (U_M) can be calculated using Eq. (2.32).

$$U_M = U_o \left(\frac{R_1}{R_1 + R_2} - \frac{R_4}{R_3 + R_4} \right) \quad (2.32)$$

Where U_o is the DC voltage supplied and R_1, R_2, R_3, R_4 are the resistances for the four resistors in Figure 2.15.

2.5.2 Energy of the flywheels

A flywheel is defined as an energy storage device, i.e. it absorbs and stores kinetic energy when accelerated and returns energy to the system when needed by slowing its rotational speed, (Norton, 2006). The two discs for the ROC are themselves acting as flywheels. This section reviews how kinetic energy for the flywheels can be calculated. The kinetic energy of the rotating system (with a constant angular velocity) is given Eq. (2.33).

$$E_k = \frac{1}{2} I_m \omega^2 \quad (2.33)$$

Where I_m is a moment of inertia, in kg.m^2 , of all rotating mass on the shaft about the axis of rotation and ω is the angular velocity in rad/s.

The formulae for calculating the moment of inertia for solid objects relevant to the ROC such as the solid cylinder, hollow cylinder and solid cone are shown in Eq. (2.34) to (2.36), (Norton, 1998, Riley and Sturges, 1996)). These formulae were used to compute the moment of inertia for the crusher discs, as discussed in Chapter 7.

$$I_{m,scy} = \frac{1}{2} \pi \rho \delta r^4 \quad (2.34)$$

$$I_{m,hcy} = \frac{1}{2} \pi \rho \delta (r_o^4 - r_i^4) \quad (2.35)$$

$$I_{m,sc} = \frac{3}{10} M_c R_c^2 \quad (2.36)$$

Where $I_{m,scy}$ is the moment of inertia of solid cylinder, $I_{m,hcy}$ is the moment of inertia of hollow cylinder, $I_{m,sc}$ is the moment of inertia of the solid cone, ρ is the density of material of construction, δ is the thickness of the disk, r_o is the outside radius of the cylinder, r_i is the inside radius of the cylinder, R_c is the radius of the cone and M_c is the mass of the cone.

During the cycle of operation, the kinetic energy for a typical flywheel is said to fluctuate due to noticeable variations in the angular velocity of the shaft, (Shigley et al., 2004). The fluctuation in the angular velocity is defined in terms of what is called the coefficient of fluctuation of speed (C_s), defined in Eq. (2.37) during a cycle of operation. The typical values of C_s as summarised in the standard handbook for Machine design by Shigley et al. (2004) are listed in Table 2.2. Considering the fluctuation in the velocity, the fluctuation in the kinetic energy can be calculated using the Eq. (2.38).

$$C_s = \frac{\omega_{\max} - \omega_{\min}}{\omega_{av}} \quad (2.37)$$

Where $\omega_{\max}$ is the maximum angular velocity, $\omega_{\min}$ is the minimum angular velocity and ω_{av} is the average angular velocity.

$$E_k = \frac{1}{2} I_m (\omega_{\max}^2 - \omega_{\min}^2) \quad (2.38)$$

Table 2. 2: Suggested values for the coefficient of velocity fluctuation

Required speed uniformity	C_s
Very uniform	≤ 0.003
Moderately uniform	0.003-0.012
Some variations acceptable	0.012-0.05
Moderate variation	0.05-0.2
Large variation acceptable	≥ 0.2

2.8 Summary of literature review

Comminution is an inefficient process and consumes about 50 % of the energy in the mining industry. Thus there is a continuing search for energy efficient machine to ensure the sustainability of the minerals industry. The energy for effecting breakage can be estimated using well-established laws by Rittinger, Kick and Bond. Knowing the specific comminution energy, the energy efficiency of a machine such as the ROC can be evaluated. Studying the size reduction process in the ROC as the rate process is one objective of this study. This is done by applying the population balance model which requires experimentally determined selection and breakage functions. The breakage function (also called appearance function) is simply the size distribution of particles smaller than the parent particles while the selection function (commonly referred to as a rate of breakage) is the disappearance rate of the particles in a size class i , i.e. in a time of operation, i.e. the total mass fraction of particles in size classes below size class i after the breakage event.

The numerical tool called DEM is used extensively in comminution to study the particle flow and breakage in various machines. In our case, the DEM was used to study the transportation of particles in the ROC under various crusher settings (feed size, disc offset, rotational speed, exit gap). To evaluate the power draw of the crusher, the speeds of the discs and motor drive torque need to be measured. The common sensors are IR sensor and load cell for speed and load measurements respectively. The energy for flywheels, such as the discs of the ROC, can be calculated using the moment of inertia (dependent on the shape of the flywheel) and angular velocity. This would help in conducting the energy balance, as discussed in Chapter 7, to know how much energy is used for comminution as compared to the energy expended in rotating the discs.

CHAPTER THREE: RESEARCH METHODOLOGY

The purpose of this chapter is to outline the research methodology employed in the study to address the research objectives. The main equipment used, i.e. the rotary offset crusher (ROC), with its auxiliary equipment, is briefly described in this chapter. The description of the crusher entails the explanation of the design and operational features. Explanation of how the coal and quartz samples were prepared is included before discussing the comminution tests conducted using the ROC, drop weight tester and piston die apparatus. Finally, the design of experiments for studying transportation of particles in the crusher using the DEM is discussed.

3.1 Rotary offset Crusher

The rotary offset crusher is a new crushing technology which is simple in design with two spinning cylindrical discs. The working diagram is shown in Figure 3.1 and the two cylindrical discs before installation are shown in Figure C1 in Appendix C. The discs are made of mild steel and they are not mechanically linked. Both discs have a radius of 250 mm, with the bottom disc having a thickness of 50 mm while top disc has a thickness of 80 mm. The pictorial diagrams for the ROC and its auxiliary components are shown in Figures 3.2 and 3.3 while the drawing showing the dimensions of the ROC structure is in Appendix C. The whole structure is 1.1 m high and 1 m wide. The vibrations of the structure are damped by bolting it onto the floor.

As illustrated in Figures 3.1 and 3.3, the material is transported to the feed hopper of the crusher using a conveyor belt, it then gravitates in the chute to the crushing zone. The particles nipped between the spinning discs move outwards with the centrifugal acceleration until they are discharged unto the collection box. The material in the collection box is collected unto a container for sieve analysis. The largest particle in the crusher product depends, to a degree, on

the space between the edges of the two discs, i.e. the exit gap. This exit gap is set before the test by adjusting the nuts shown in Figure 3.2 to move the top disc in the vertical direction. The standard operating procedures in Appendix D explain in detail how such adjustments are made. The exit gap is measured after every run to check if there is any change. Similarly, the offset between the vertical axes of the two discs (shown in Figure 3.1), is varied by sliding the top disc. This is achieved by adjusting the nuts as depicted in Figure 3.2 and described in detail in Appendix D.

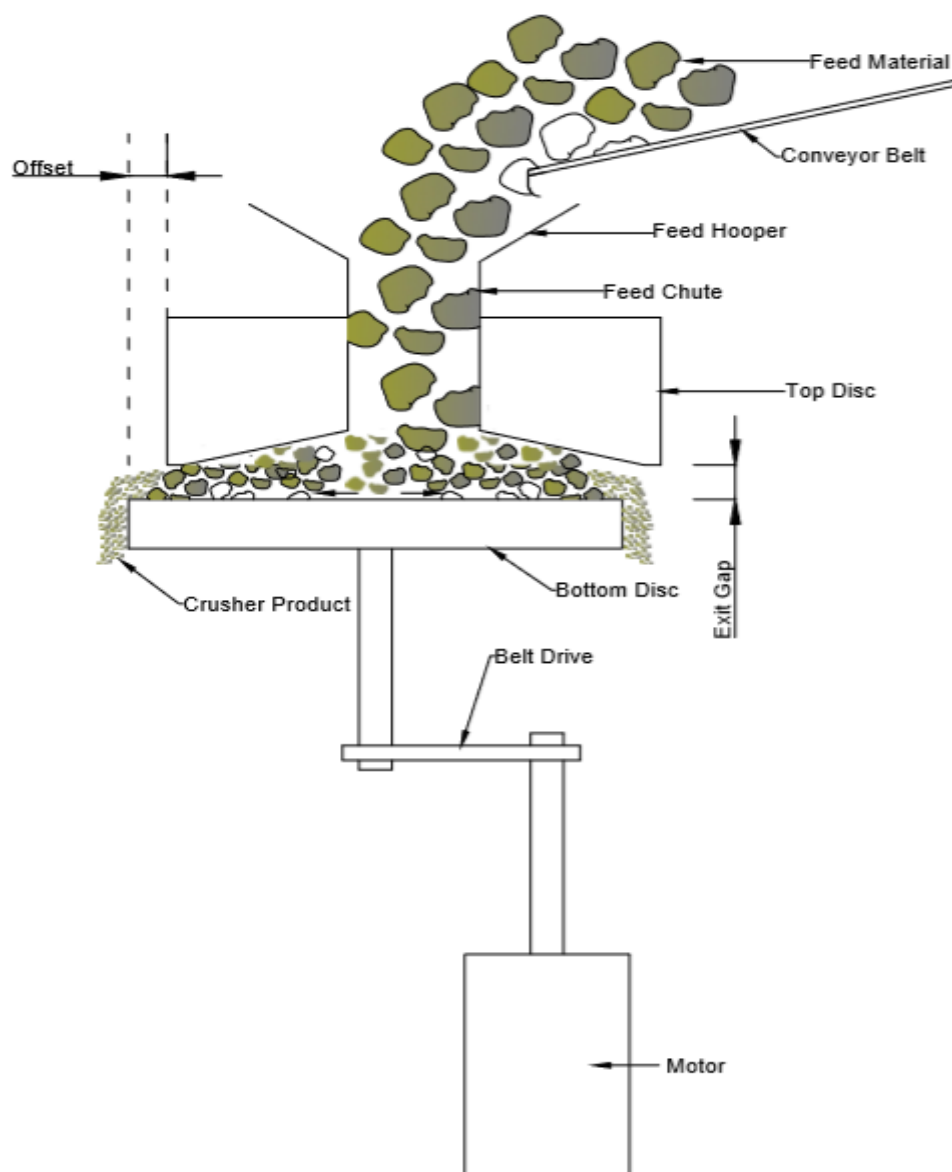


Figure 3. 1: The working diagram of the rotary offset crusher

The crusher is powered using a 3 kW three-phase induction motor. This motor has a full-load speed of 1420 rpm and using Eq. (2.29), its full-load torque is 20 Nm. Power transmission from the motor rotor to the shaft of the bottom disc is achieved with a V-belt (see Figures 3.1 and 3.3). During operations, the drive system is covered by the safety guard (labelled in Figure 3.2). The crusher is instrumented with sensing devices to pick up the signals for the speeds of the two discs, feeder load, and motor drive torque. The speed of the bottom disc and torque are used to calculate the mechanical power. The instrumentation circuits are discussed in much detail in Chapter 4.

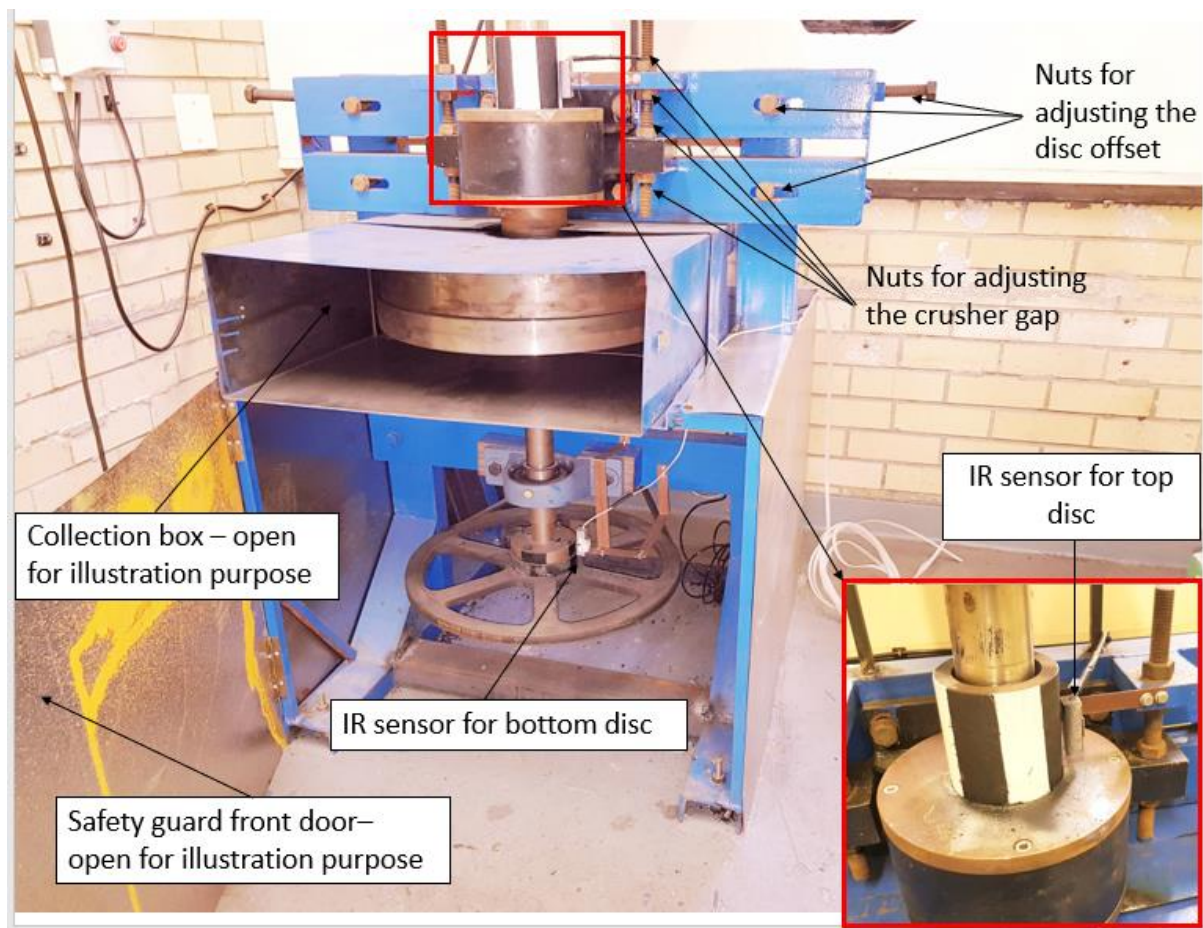


Figure 3. 2: The rotary offset crusher with all its auxiliary components

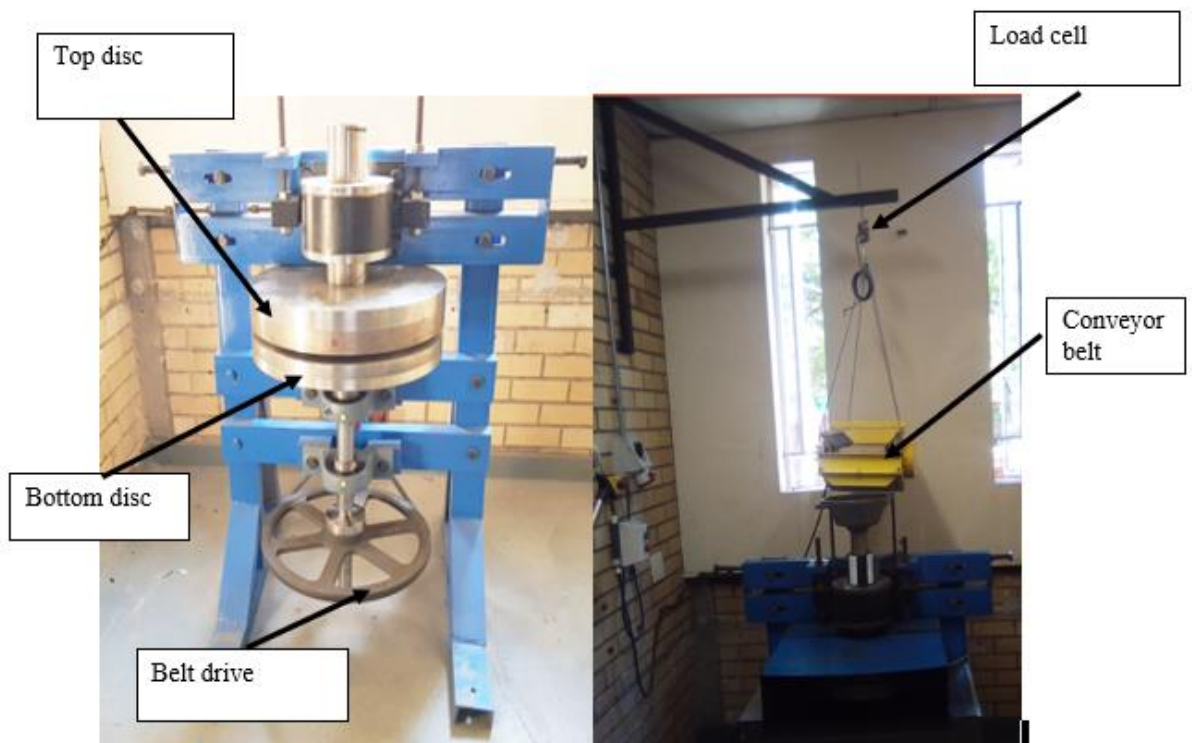


Figure 3. 3: Rotary offset crusher, illustration of the discs (on the left) and the feeder (on the right)

3.2 Coal comminution

3.2.1 Sample Preparation

Three coal samples (each with a mass of about 20 kg) with different density (SG) fractions (-1.32+1.3; -1.45+1.42 and -1.5+1.47 respectively) were homogenised. These samples are products of some gravity separation experiments. A representative sample was prepared for proximate analysis using the ten cups rotary splitter shown in Figure 3.4. Using this splitter 10 identical samples were obtained. The proximate analysis was done on one of the homogenised samples using the thermogravimetric analysis (TGA), discussed in Appendix E. The coal sample assayed 1 %, 25 %, 55 % and 19 % for moisture, volatile matter, fixed carbon and ash respectively.



Figure 3. 4: Rotary sampler used to split the samples

Using laboratory sieves, two narrow size fractions ($-19+13.2$ mm and $-13.2+9.5$ mm) of the coal sample were prepared as feed samples for the ROC experiments discussed in section 3.2.2. The two size fractions were also used for impact and compression breakage tests discussed in sections 3.4 and 3.5 respectively.

3.2.2 ROC crushing tests

The coal sample was used to conduct crushing tests at various crushing settings (rotational speed, discs offset and exit gap) with mono-sized feed charges. The rotary splitter was used to prepare five 1.5 kg identical feed samples for the $-19+13.2$ mm and $-13.2+9.5$ mm size fractions. Four of these samples, for each size class, were used in the ROC experiments shown

in Table 3.1 while the fifth sample was reserved for laboratory breakage tests discussed in sections 3.4 and 3.5.

During each ROC experimental run, the following data were captured: feeding time, residence time, real-time speeds and motor drive torque. These data were used to characterise the crusher in terms of size reduction, throughput and power draw as explained in Chapters 4 and 7. The crusher products for all tests were dry sieved using the sieve shaker shown in Figure 3.5 with sieves stacked in a series of $\sqrt{2}$: from 13.2 mm down to 38 μm and relevant size distributions were plotted together to establish the trends.

Table 3. 1: Test conditions for coal comminution with a speed of 330 rpm

Feed size, mm	Exit gap, mm	Offset, mm
-13.2+9.5	3	10
-19+13.2	3	10
-13.2+9.5	3	5
-19+13.2	3	5
-13.2+9.5	1.5	5
-19+13.2	1.5	5
-13.2+9.5	1.5	10
-19+13.2	1.5	10

3.3 Quartz Comminution

3.3.1 Sample preparation

The quartz sample was dry sieved to get the same narrow size fractions as those used for coal experiments, -19+13.2 mm and -13.2+9.5 mm and 1 kg sub-samples for the two size classes were prepared using the rotary splitter. The prepared samples were used for the crushing tests discussed in section 3.2.2 and breakage tests discussed in sections 3.4 and 3.5.

3.3.2 ROC crushing test

The experimental runs for quartz comminution using the rotary offset crusher are summarised in Table 3.2. The offset and exit gap were fixed at 3 mm and 10 mm respectively in all batch

tests while studying the effects of the feed size, feed rate and rotational speed of the discs on the crusher performance. As with coal experiments, the crusher products were collected and dry sieved to get the size distributions to compare the size reduction of the two materials (coal and quartz) as discussed in Chapter 7.

Table 3. 2: Quartz ROC comminution Characteristics

Feed size (mm)	Feed rate (kg/h)	Speed (rpm)
-13.2+9.5	1023	330
-19+13.2	1027	330
-13.2+9.5	1703	330
-19+13.2	1720	330
-13.2+9.5	1753	550
-19+13.2	1674	550
-13.2+9.5	1109	550
-19+13.2	970	550

3.4 Single particle impact using the drop weight tester

The purpose of the impact tests was to generate alternative data to compare with the results obtained with the rotary offset crusher. The experimental apparatus used (whose working diagram is shown in Figure 3.5) comprises the steel anvil, a drop weight (steel disc with a flat impact surface), and an electromagnet to hold and release the drop weight from the predetermined height. The input energy is evaluated using Eq. (2.9). The same size classes (-19+13.2 mm and -13.2+9.5 mm) used for coal and quartz comminution in the ROC tests were prepared for the drop weight tests. For each size class and material, about 1 kg of the sample was split into approximately 100 g identical samples for use in single particle impact breakage and particles bed compression tests (discussed in section 3.5). The test conditions for the impact breakage tests are summarised in Table 3.3. Given that for each run there are on average more than 50 particles, statistical variation is considered. After every run (particles in one size class

subjected to the same impact energy), the fragments were collected and dry sieved to get their size distributions. The results are discussed in Chapter 6.

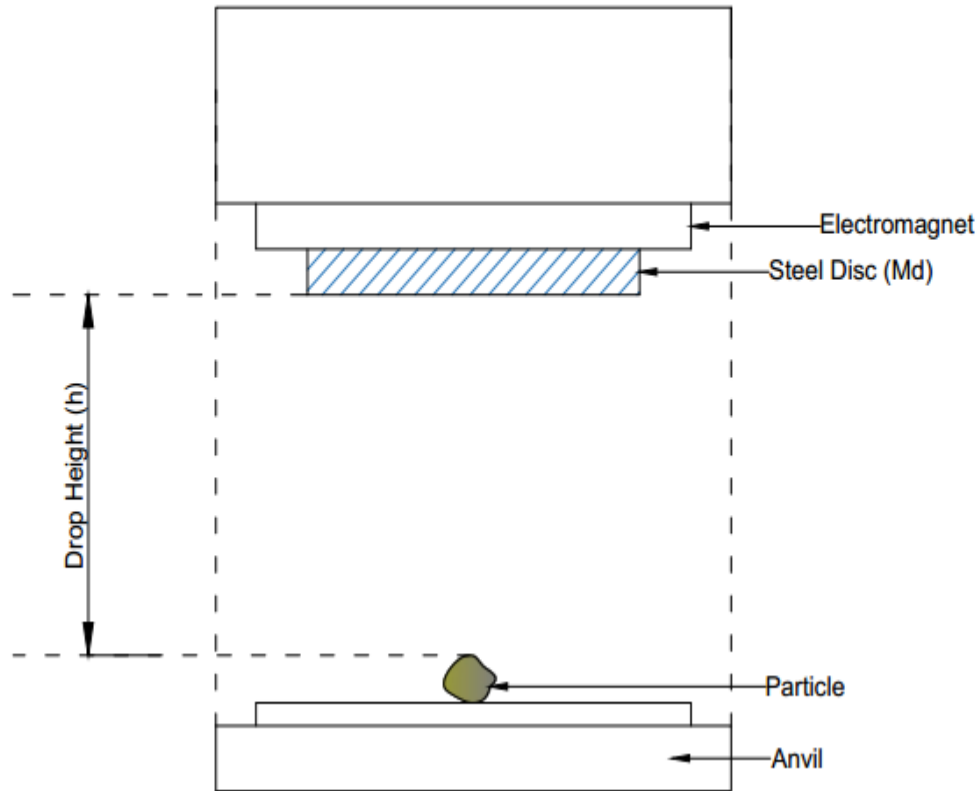


Figure 3. 5: The working diagram for the drop weight tester

Table 3. 3: Drop weight test conditions for coal and quartz impact breakage

Size range (mm)	-13.2+9.5				-19+13.2		
Impact energy (J)	1.72	2.99	4.82	7.23	1.72	2.99	4.82

3.5 Compression breakage tests using the piston-die apparatus

The piston die cell is used extensively in literature to study compression breakage (Fuerstenau et al., 1996, Viljoen et al., 2001, Oettel et al., 2001, Barrios et al., 2011, Esnault et al., 2015) or specifically for machines such as VRM (Shahgholi et al., 2017), in HPGR (Dundar et al, 2013, Davaanyam, 2015). The experimental setup is shown in Figure 3.6. The sample is put in the crushing chamber (inside the die) and the piston is placed on top of the sample before

applying pressure using the press machine. The force (measured using the load cell) and displacement are recorded in real time in Microsoft Excel[®] for the computation of the energy input. The four compression tests conducted are shown in Table 3.4. The maximum load and speed for applying the pressure were fixed at 50 kN and 3 mm/min respectively in all tests. The initial bed height was also kept constant (40 mm) for the two size classes (-19+13.2 and -13.2+9.5 mm) of both coal and quartz. The mass per test is a function of the 40 mm initial bed height. With the 40 mm bed height, the sample size per test ensures enough sample size to absorb efficiently the energy input and large enough relative to the grain size to ignore side effects. Results for the compression tests are discussed in Chapter 6.

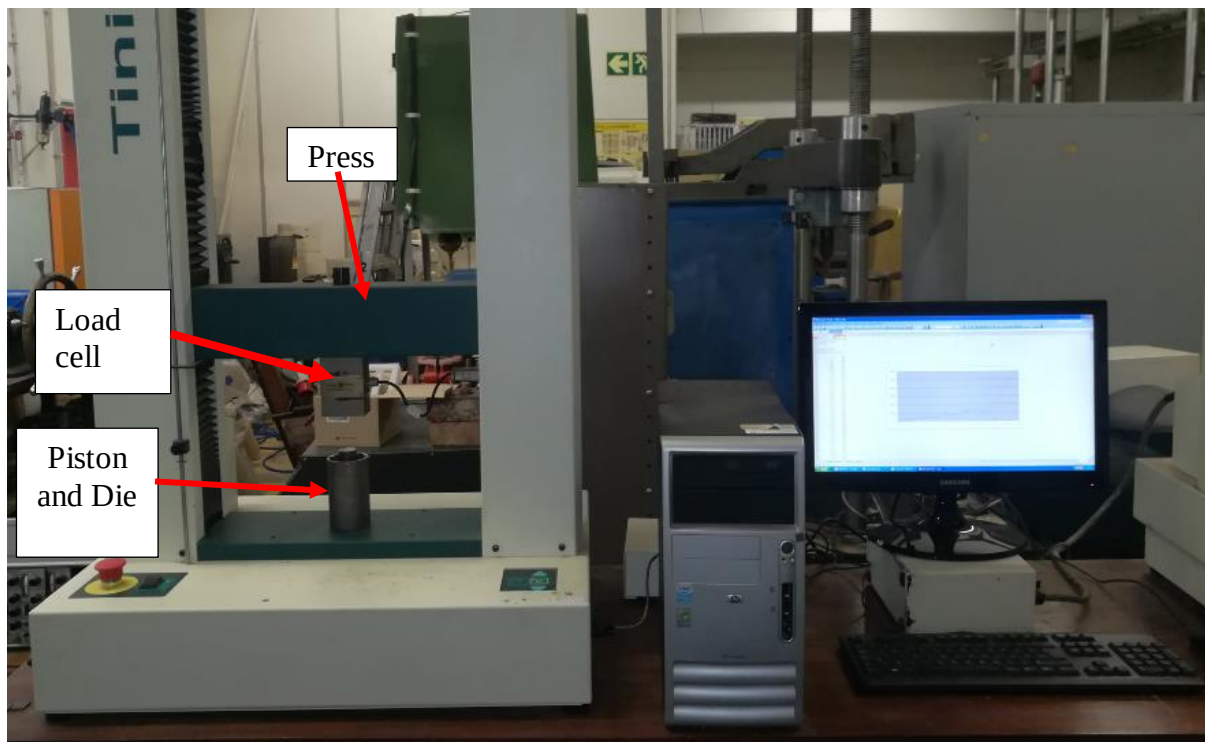


Figure 3. 6: Experimental setup for the compression tests

Table 3. 4: Test conditions for the particles bed breakage tests

Material	Coal		Quartz	
Size (mm)	-13.2+9.5	-19+13.2	-13.2+9.5	-19+13.2
Mass (g)	84.4	107.9	59.4	66.3

3.6 Simulations using the DEM PFC software

To study the behaviour of the particles in the rotary offset crusher focusing more on the transportation, the DEM particle flow code 3D (PFC3D) software was used to conduct simulations. The design of experiments (simulations) was conducted using the *Statgraphic 18*[®] statistical software choosing the randomised central composite design. Operating variables whose effects on transportation were investigated with DEM are shown in Figure 3.7 and the simulation “recipe” is listed in Table 3.5 with all resulting 18 treatments listed in Table F1 in Appendix F. For each simulation, 500 particles (steel balls with the density of 7700 kg/m³) were fed to the crusher.

Table 3. 5: Characteristics of DEM simulations

Factor	Unit	Levels
Ball size	mm	4, 7, 10
Speed	rpm	330, 765, 1200
Offset	mm	5, 10, 15
Exit gap	mm	4, 7, 10, 15
Throat diameter of chute	mm	100
Height of comminution cavity	mm	20
Interior flat edge of top disc	mm	10
Diameter for both discs	mm	250

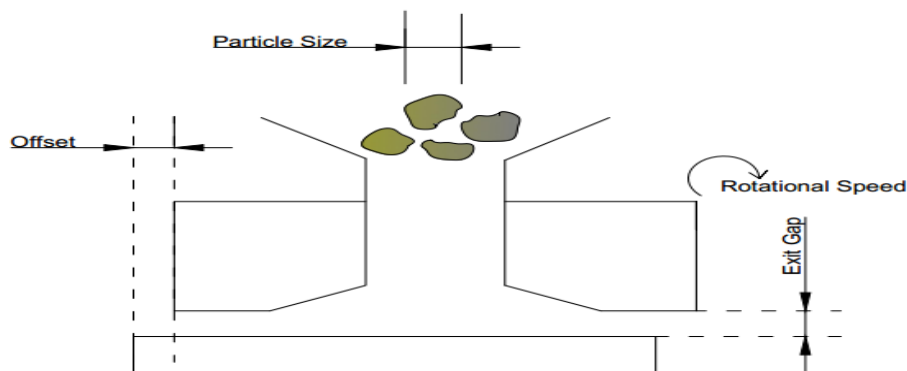


Figure 3. 7: Operating variables whose effects on transportation were investigated with DEM

CHAPTER FOUR: INSTRUMENTATION DEVELOPMENT AND METHODOLOGIES FOR CALCULATING THE FEED RATE AND POWER DRAW

4.1 Overview

The crusher is instrumented with four transducers, shown in Figure 4.1, namely: two IR sensors (shown as IR1 and IR2) directed to the shafts, driving the discs, to measure their rotational speeds and two strain gauge load cells for measuring the loads (weight of material conveyed to the crusher and the force required to keep the torque arm stationary). The measurement of the speeds of the discs and the motor drive torque are used to calculate the mechanical power of the system while the change in the mass of the material on the conveyor belt provides the feed rate. The microcontroller board called Arduino UNO was used to run codes for the circuits and give output (in terms of electrical signals) that is sent to the computer for data analysis. This microcontroller is discussed in detail in Appendix G. The electronic circuits, Arduino codes for speed and load measurements as well as the methodologies of calculating the power draw and feed rate are discussed in the subsequent sections of this Chapter.

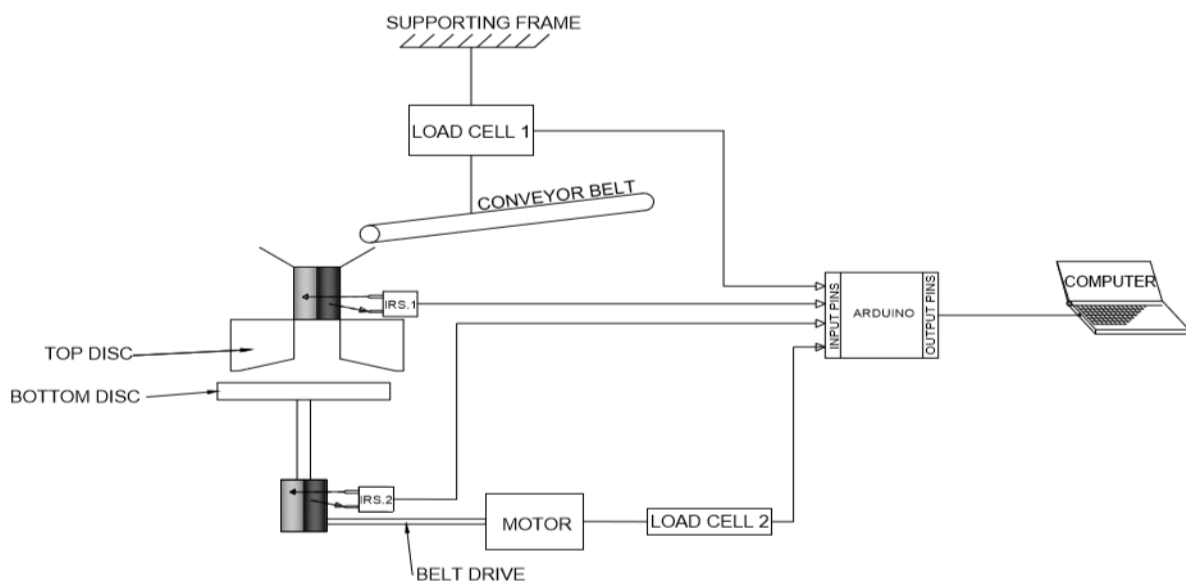


Figure 4. 1: Overview of the crusher instrumentation

4.2 Using the IR sensor to measure the rotational speeds of the discs

The working principle of the infrared (IR) sensor directed at the spinning shafts for the discs is shown in Figure 4.2. The shafts are painted in alternating patches of two colours (black and white) as shown in Figures 4.3 and 4.4. The KE0068-180802 IR sensor module, shown in Figure H1 in Appendix H, was used for speed measurement. The operating characteristics for this sensor module extracted from the datasheet are listed in Table H1 in Appendix H. Basically, it operates with a voltage supply of 5 V and gives the digital output, i.e. either 1 or 0. The sensor should be located about 1 to 3 cm from the spinning shaft. As shown in Figure 4.2, most of the infrared radiation directed to a black surface will be absorbed and thus will not become incident on the photodiode (IR receiver). The opposite happens when the rays encounter a white surface.

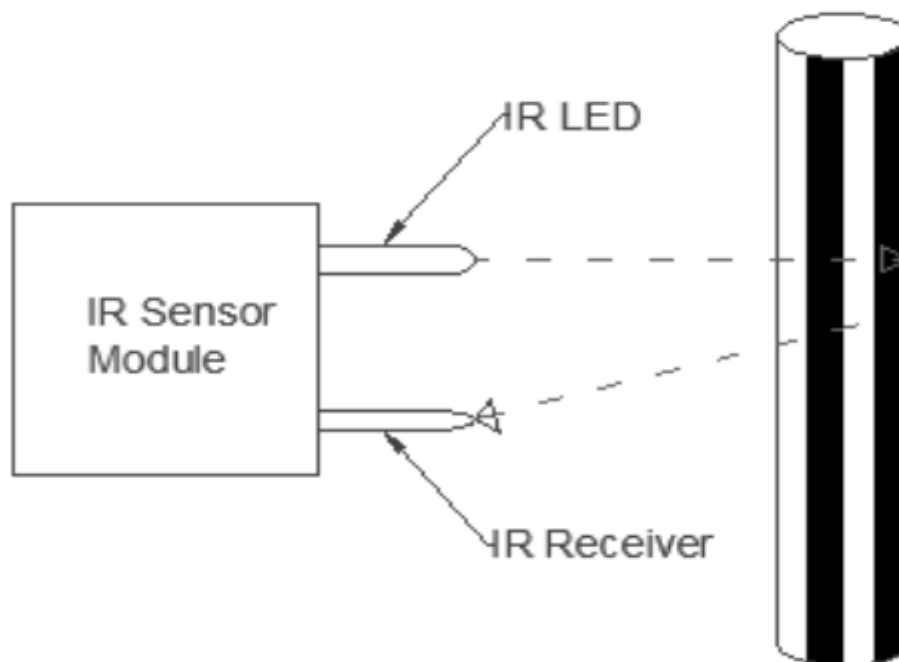


Figure 4. 2: Working principle of an IR sensor

The photo-coupler (a combination of the IR LED and receiver) is integrated with other electronic components such as an indicator, operational amplifier and potentiometer to make the sensor module shown in Figure H1 in Appendix H. The indicator, as shown in Figure 4.3,

gives a visual feedback (red) when some rays are reflected, i.e. when the transmitted rays encounter a white surface. The amplifier conditions the voltage signals received by the IR receiver to digital outputs (LOW or HIGH) which is fed to Arduino while the potentiometer (a variable resistor) is used to change the sensitivity of the sensor. The electrical characteristics of the amplifier and potentiometer are listed in Tables H2 and H3 respectively in Appendix H.



Figure 4. 3: No visual feedback when the IR sensor encounters black surface (on the left) and there is a red visual indication when the sensor encounters the white surface

4.2.1 Installation of the IR sensor on the ROC and digital output

The photographs in Figure 4.4 show how the IR sensor modules were integrated on the crusher to measure the speeds of the shafts for the bottom and top discs. The sensors are placed 1 cm from the shafts. Specially designed rectangular brackets were fabricated to fully enclose the sensors (except the side on which the IR emitter and receiver are pointing) and thereby prevent environmental interference on the signals. The shaft has equal, alternating patches of black and white (four for each colour). With eight strips (each with a width of 39 mm) on the shaft, it means that there are eight pulses in a signal for each revolution; with four rising edges and four falling edges.



Figure 4. 4: (On the left) The IR sensor placed 1 cm from the shaft of the top disc (with alternating patches of black and white) and (on the right) the IR sensor placed 1 cm from the shaft of the bottom disc (painted in a similar manner)

The circuit diagram for the ROC instrumentation is shown in Figure 4.5. When the infrared radiations encounter the black surface (a good absorber) all the transmitted radiations are absorbed, i.e. such that no rays are reflected to the receiver as shown in Figure 4.2, so the circuit is still open, i.e. a signal output of 1 (High-level signal). On the other hand, with the white colour (a bad absorber), radiations are reflected to the receiver and thereby closing the circuit, i.e. the IR receiver outputs a low-level signal (0). The Arduino code in Appendix I was written to read the digital signal output of the IR sensor when the discs are spinning. What the Arduino code for reading digital outputs does is simple: it reads the value of the digital outputs which are saved in real time in excel using the PLX DAQ excel add-on tool for data capturing in real time in Microsoft Excel®. The PLX DAQ is discussed in detail in Appendix G. The typical signal is presented in Figure 4.6. From Figure 4.6, it was observed that period for the “crest” (when the digital value is 1) are relatively similar. The same can be concluded for the “trough” (when the digital output is 0). This is despite the fact that the black and white patches are equal in length. As discussed in subsection 4.2.2, the periods for the “troughs” were used to continuously estimate the rotational speed for the discs during the cycle of operation. Validation measurements discussed in subsection 4.2.3 showed accuracy levels greater than 99 % despite the difference in the periods for the troughs and crests in Figure 4.6.

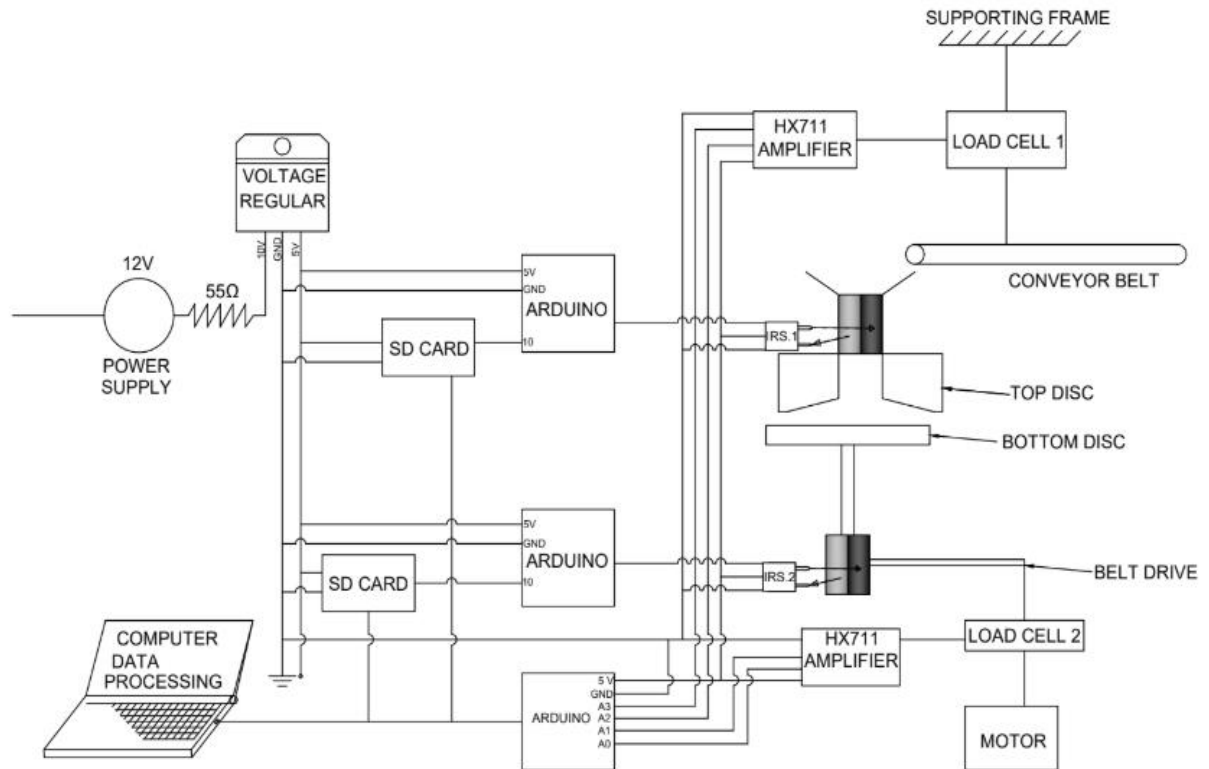


Figure 4. 5: Schematic diagram of the crusher instrumentation

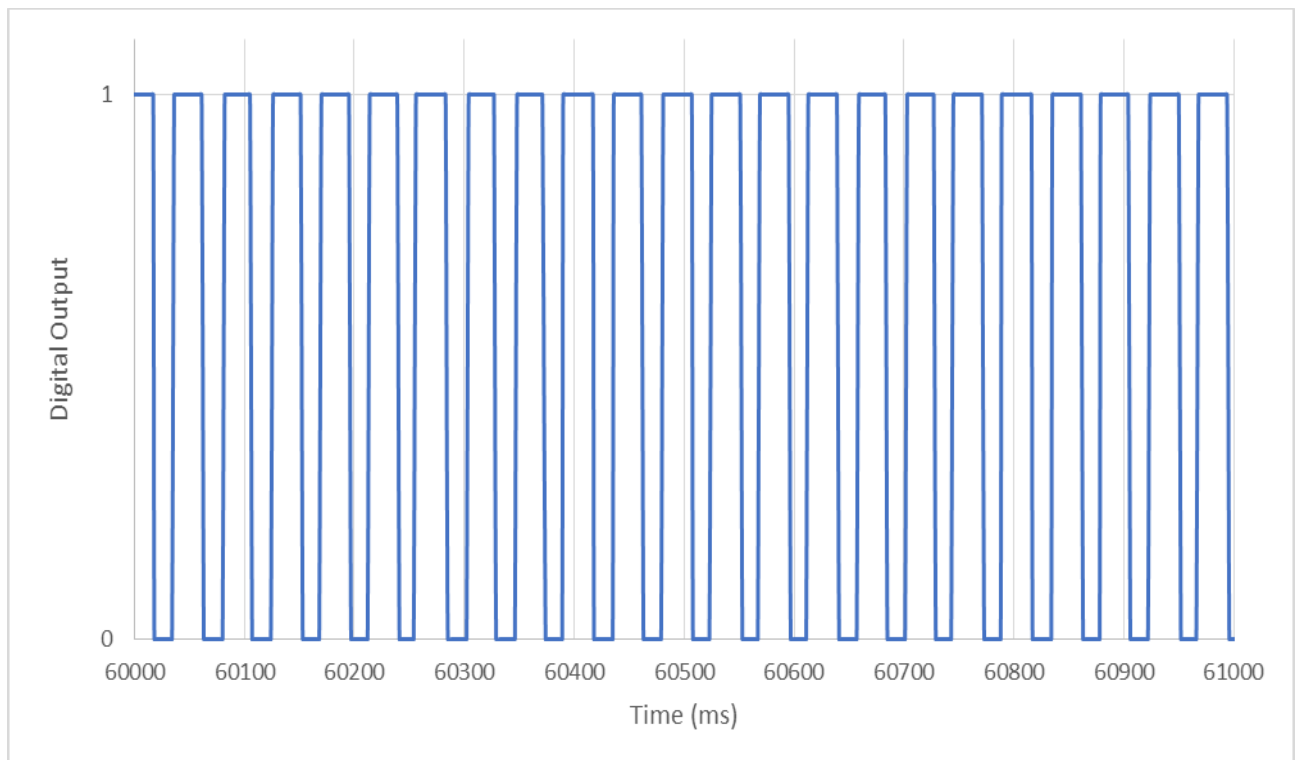


Figure 4. 6: Digital output of the IR sensor directed to the spinning bottom disc

4.2.4 ROC tachometer fabrication and calculation of rotational speed

It was demonstrated with digital outputs shown Figure 4.6 that a square wave is obtainable with the IR sensor encountering the spinning shaft. This served as a motivation to use the sensor module in Figure H1 in Appendix H for speed measurement. The monitoring circuit (in Figure 4.5) was then designed as part of the instrumentation for the crusher to measure the speeds in rpm for both discs. The circuit diagram in Figure 4.5 shows how different electronic devices are connected for measuring the digital output of the IR sensor, computation of the speed in rpm, data storage and processing. The digital output signal from the IR sensor is sent to the Arduino UNO. The circuit is powered using a 12 V power source. This power is further regulated to 5 V using the L7805 voltage regulator (its electrical characteristics are listed in Table L1 in Appendix L). Putting a pull-down resistor of 55 Ω in series with the input voltage pin for the L7805 regulator ensures the regulator is supplied with a safer voltage input of 10 V. The sensor modules directed at the shafts for the two discs are connected to pin 2 of their respective Arduino UNO boards. With the codes uploaded on the board, the periods for the troughs (when the digital output is 0) are continuously read for as long as the circuit is powered and used to calculate the speed (in rpm) using Eq. (4.1).

$$n_{\text{ROC}} = \frac{60 \times 10^6}{4 \times t_w} \quad (4.1)$$

Where n_{ROC} is the speed of the disc in rpm, t_w is the time in microseconds when the sensor encounters the white patch and constant 4 is the number of white patches in a revolution.

The Arduino code for this purpose is included in Appendix I. The code interprets input to provide a measurement of the period (in microseconds) when the sensor encounters the white colour. This is made possible by using an interrupt pin (digital pin 2 or 3). The interrupt pin enables the special function in Arduino programming language known as Interrupt () to be used

to record the time in microseconds using a `micros()` function for which the digital output is either LOW or HIGH. The rpm values are saved in the SD card and transferred to the computer for analysis in Microsoft Excel®. The SD card library that was used is discussed in detail in Appendix G. It is important that the rate at which the data is transferred to the Arduino UNO is high enough for accurate measurement and real-time data capturing. The communication rate (also called baud rate), measured in bits per second (bps), is a measure of how fast the data is sent over a serial line. The baud rate in Arduino UNO range between 480 and 2 000 000 bps. The fastest baud rate of 2×10^6 bps was chosen.

4.2.3 Validation

The mechanical tachometer shown in Figure 4.7 was also used to measure the speed of the bottom disc and the measured values were compared to those obtained using the IR sensor module with Arduino microcontroller by calculating the percent error with Eq. (4.2). This was done by taking the value given by the mechanical tachometer as a true value and what is given by the ROC IR sensor as a measured value. Percent error, in Table 4.1, ranging between 0.4 and 0.9 % was recorded; implying the accuracy levels of over 99 %.

$$\text{Percent error} = \left| \frac{\text{True value} - \text{Measured value}}{\text{True value}} \right| \times 100 \quad (4.2)$$

Table 4. 1: Speeds measured using IR sensor and Tachometer

Speed (rpm)		Percent Error (%)
IR sensor	Tachometer	
545-548	550	0.4 - 0.9



Figure 4. 7: Mechanical hand tachometer type 2200

Following this validation, two ROC IR tachometers were fabricated. Soldering was done to make permanent connections between the sensor module, Arduino UNO board and power supply as shown in Figure 4.8. The typical speed signals are discussed in section 4.4.

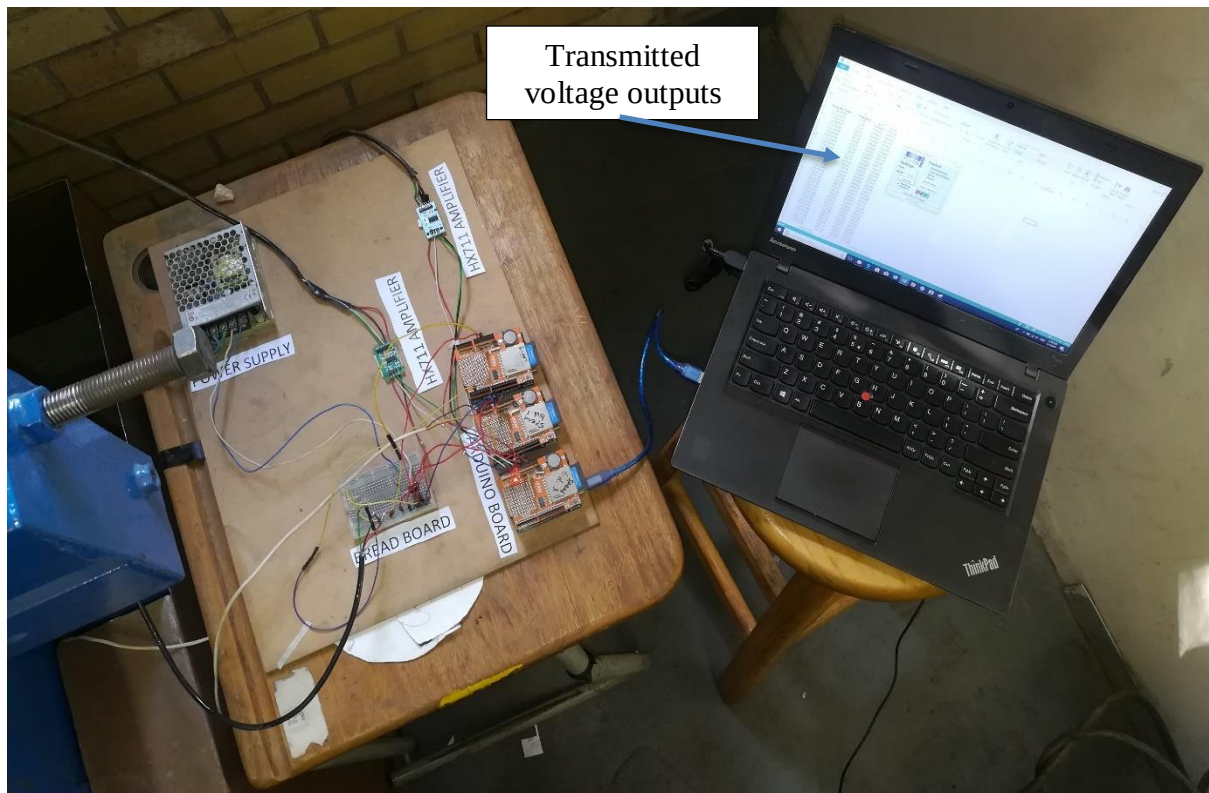


Figure 4. 8: A photograph of the circuit diagram for the crusher instrumentation

4.3 Load Measurement

As already discussed in section 4.1, two strain gauge load cells were installed on the crusher (see Figures 4.1 and 4.9): one is for measuring the motor drive torque and the other is for measuring the mass of the sample conveyed to the crusher hopper. The former measurement is important for the calculation of the mechanical power of the system while the latter is needed to calculate the feed rates. As stated in the literature review Chapter, load cells are transducers based on the four-arm Wheatstone bridge circuit that is intimately combined with an elastic element to be able to measure the loads in terms of voltage. A load cell with a rated capacity of 50 kg and safe overload of 150 %, shown in Figure J1 of appendix J was used in both cases. This load cell is suitable for applications involving both tension and compression, (“Zemic Europe B. V.”, 2017).

Unlike in the case of speed measurement where digital output is preferred, analog outputs are needed for load measurement. The load cells have the excitation voltage of 5 – 12 V and output sensitivity of 3 mV/V implying that only about 15 – 36 mV is available at full scale. The voltage output of the load cell is very small for processing by the Arduino, and hence a need for an amplifier. The instrumentation amplifier, commonly known as HX711, a 24-bit precision analog-to-digital converter with operating voltage ranging between 4.8 and 5.5 V and commonly used in weighing scales (“Mantech Electronics”, n.d.), was selected for signal conditioning. The board and electrical characteristics for the HX711 amplifier are discussed in Figure K1 in Appendix K.

The circuit diagram (Figure 4.5) shows how the load cells and amplifiers are connected to the Arduino UNO board. The regulated voltage supply is ensured with the L7805 voltage regulator that gives a fixed voltage output of 5.01 V. The amplified outputs of the two load cells are sent

to one Arduino UNO board and with the loaded code (in Appendix M), their respective outputs are sent in real time to Microsoft Excel[®]. In order to calculate the desired quantities (mass of material on the conveyor and motor drive torque), the load cells were calibrated following the procedures included in Appendix M with the calibration curves and relevant equations discussed in subsection 4.3.1.



Figure 4. 9: Illustrations of the load cells installed on the crusher to measure the load of the drive system (on the left) and the weight of the materials conveyed to the crusher (on the right)

4.3.1 Calibration of the load cells and framework for the data processing

As discussed in Appendix M, calibration of load cells was done using limestone samples of known masses. Two plots are shown in Figures 4.10 and 4.11 for the mass versus time and force versus time respectively.

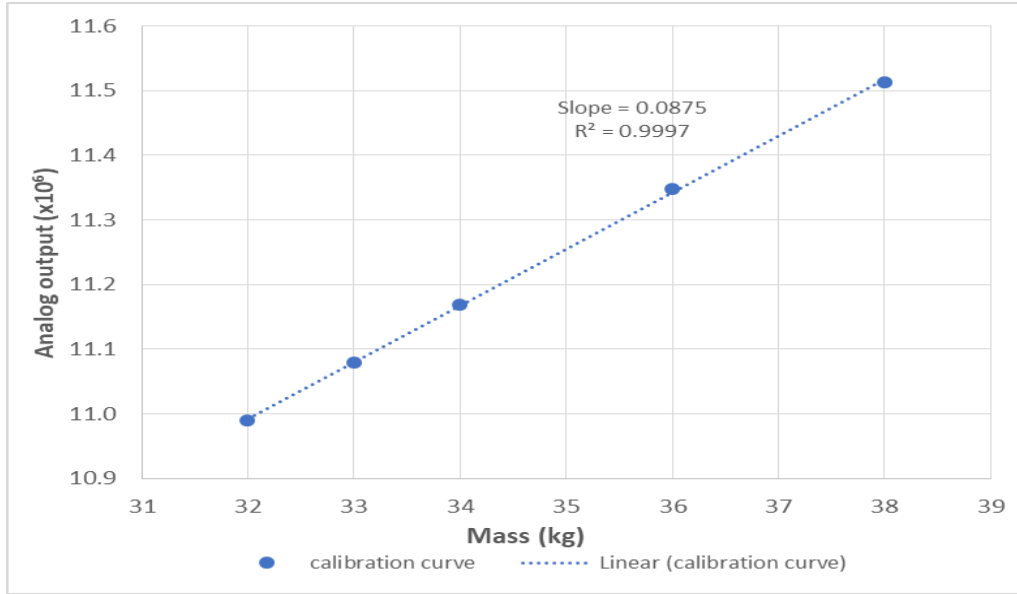


Figure 4. 10: Relationship between the mass and analog output of the load cell

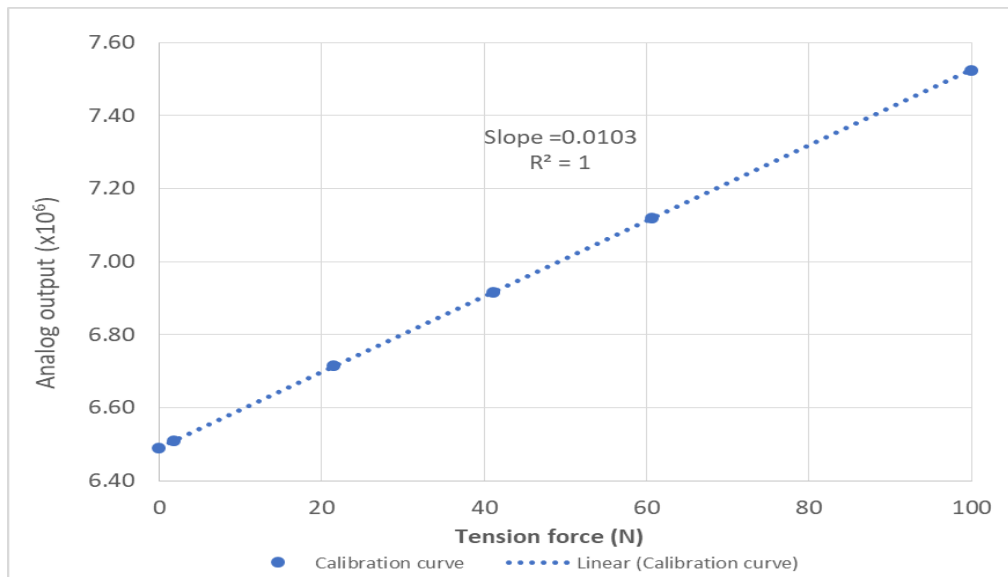


Figure 4. 11: Relationship between the forces exerted on the load cell and analog output of the load cell

4.3.1.1 Feed rate calculation

Using the relationship shown in Figure 4.10, the mass of the sample on the conveyor at any time can be computed using Eq. (4.3). The typical signal from one test conducted with quartz is shown in Figure 4.12. From the mass versus time graph, the feed rate can be calculated by dividing the total mass fed to the crusher with the total feeding time as shown by Eq. (4.4). For

example, using Figure 4.12, the feeding time is 3.6 seconds and feed mass 1.48 kg, which gives the feed rate of 1.48 t/h. Similar plots (in Appendix U) were obtained in the tests conducted with the ROC, and results are incorporated in subsequent chapters.

$$M_t = \frac{(A_t - A_{0,c})}{0.0875} \quad (4.3)$$

Where M_t is the mass of material on the conveyor at a given time t , A_t is the analog output of the load cell at time t and $A_{0,c}$ is the no load analog output of the conveyor, i.e. the load cell output when conveyor belt is empty.

$$F = \frac{M_{\text{batch}}}{t_f} \quad (4.4)$$

Where F is feeding rate (tph), M_{batch} is the total mass fed to the crusher (in ton) and t_f is the feeding time (in hours)

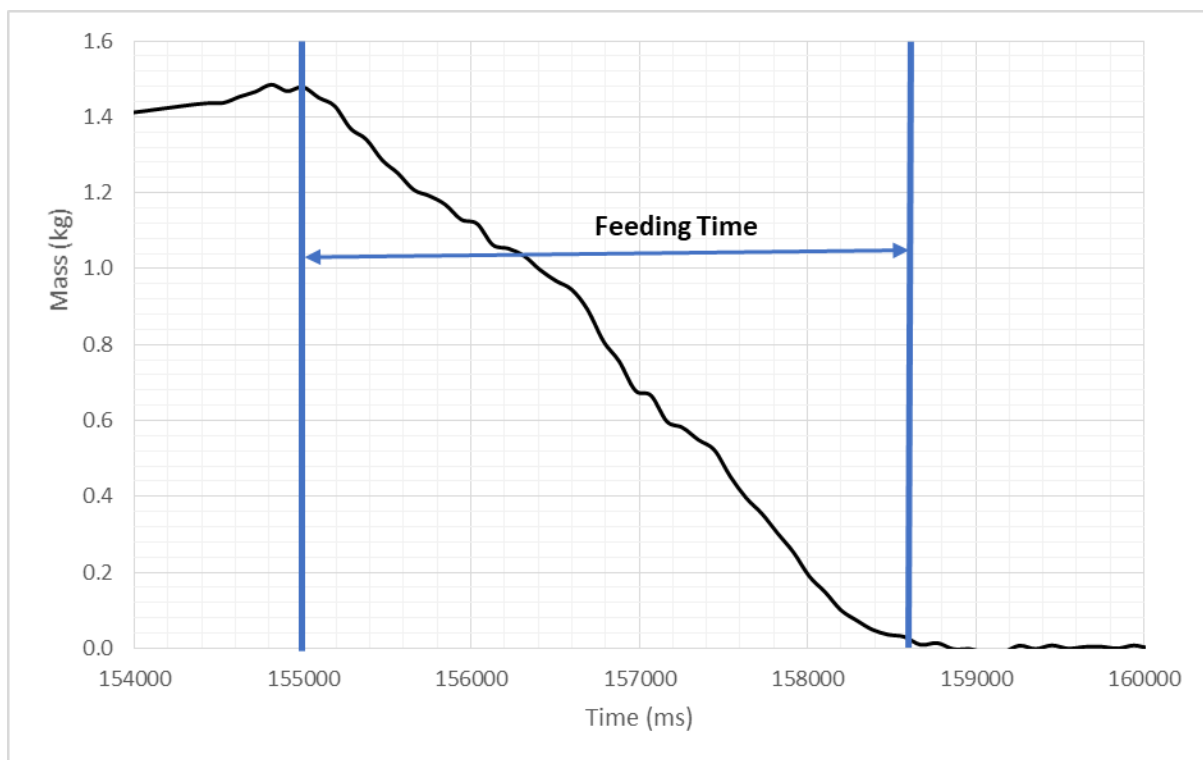


Figure 4. 12: Typical plot for the mass versus time during the ROC conveyor belt

4.3.1.2 Computation of motor drive torque

From the first principle, and for a simple case, the relationship between output voltage and force can be formulated as follows (linear as shown in Figure 4.11):

$$A_t = C_f F_t + A_0 \quad (4.5)$$

Where F_t is the force in Newtons, C_f is a slope of a straight line between voltage and force (= 0.0103 from Figure 4.11), A_t is the load voltage at time t while A_0 is the value of the no load voltage.

The torque can be calculated using Eq. (4.6) from force F_t and the perpendicular distance d between the point of force application for rotation to the axle of the bottom disc (as shown in Figure 4.13).

$$T_t = F_t \times d \quad (4.6)$$

Where T_t is the torque at time t .

Substituting F_t from Eq. (4.5) into Eq. (4.6) results in Eq. (4.7) which gives the torque.

$$T_t = \frac{(A_t - A_0)}{0.0103} \times d \quad (4.7)$$

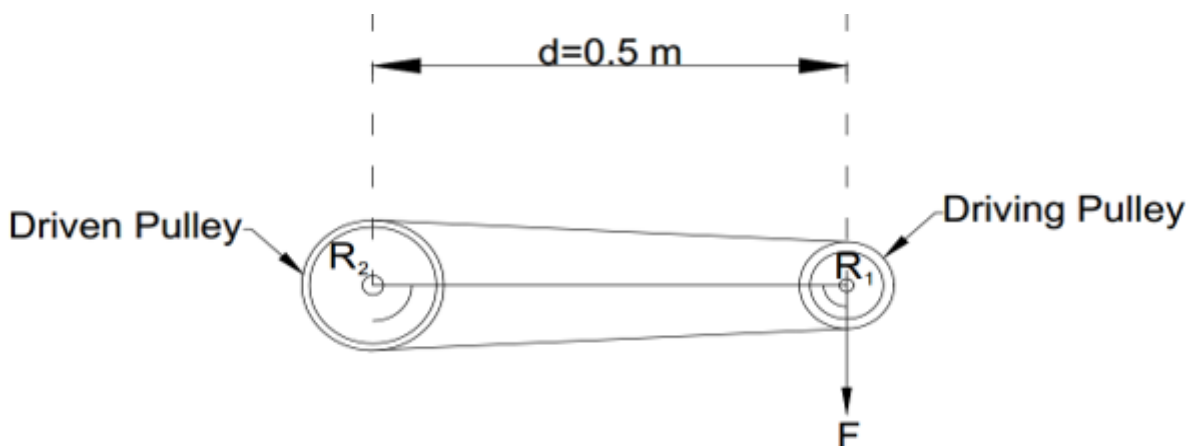


Figure 4. 13: Illustration of the measurement of the load for the ROC drive system; R_1 and R_2 are radius for the driving and driven pulley respectively, F is the measured force and d is the centre distance between the axes of the pulleys

In Figure 4.14, the torque signals for the drive system for two cases (bottom disc only and the two discs stacked together) are shown. Looking at the black signal, (for the bottom disc only), it is observed that during the start-up, a very high torque of 28 Nm was needed to get the disc spin at the operating speed (of 330 rpm). After 2 seconds, the steady state torque of about 4 Nm was recorded. In the case of the two discs stacked together (purple line), their combined start-up torque is 38 Nm while the steady state torque is 9 Nm. While the torque is directly proportional to the current drawn by the motor, such a relationship applies only before the motor is off. The torque recorded after the motor is off is because of the stored energy in the flywheels.

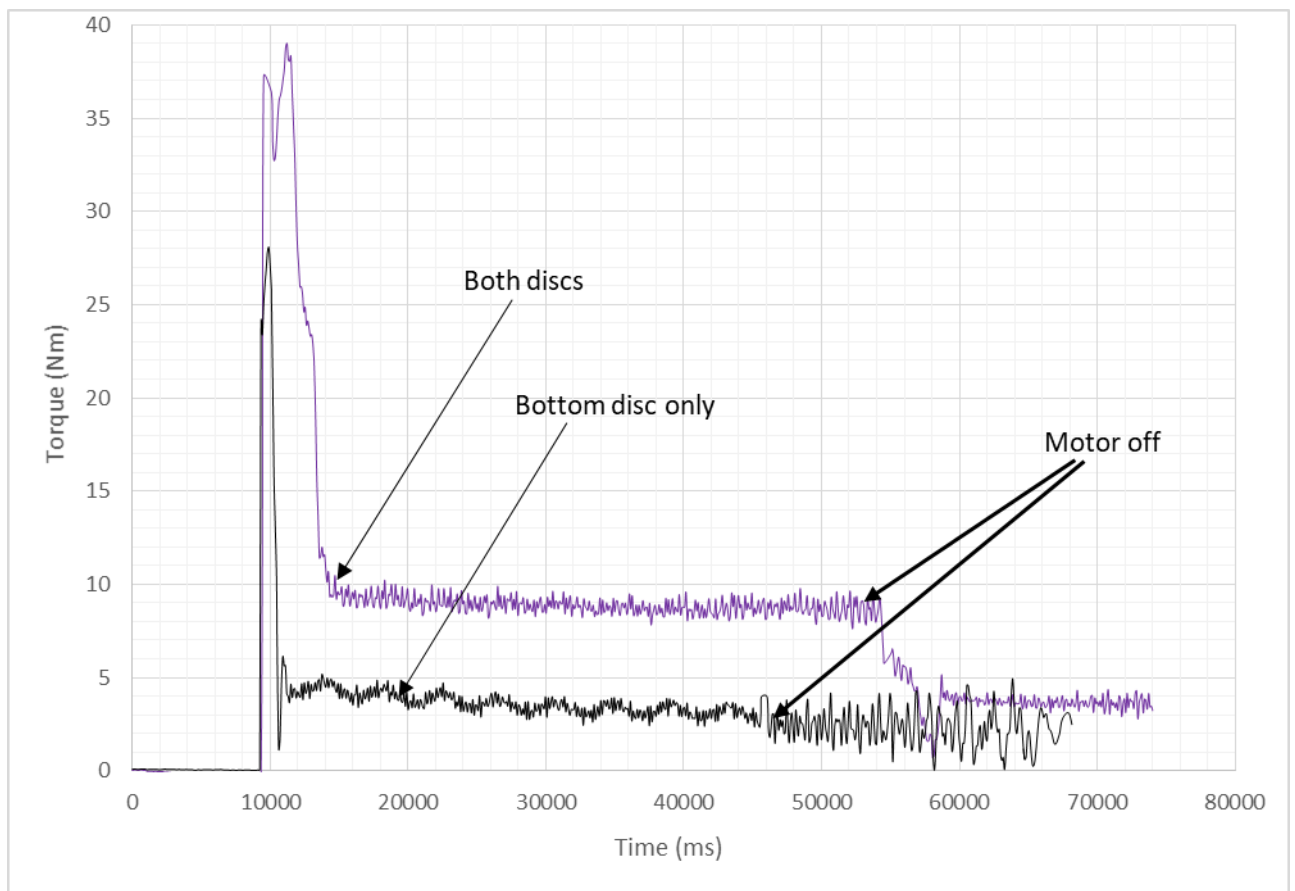


Figure 4. 14: The torque signals for evaluation of crusher power draw for the two cases: with a bottom disc only and with both discs stacked together

4.4 Crusher Power draw and Methodology for energy computation

The signals for the mechanical power can simply be calculated by multiplying the rotational speed and torque at any given time as shown by Eq. (2.29). This can simply be done by substituting Eq. (4.1) and Eq. (4.7) into Eq. (2.29) resulting in Eq. (4.8).

$$P_t = \frac{2\pi \times (A_1 - A_0)}{60 \times 0.0103} \times n_{ROC} d \quad (4.8)$$

Where P_t is the power draw by the crusher at time t .

4.4.1 Power draw of the discs

Eq. (4.8) was used to calculate the power draw for the two cases: (1) when only the bottom disc is spinning, and (2) when the two discs are stacked together to get the power of the crusher (with no particles). The difference between the powers for the two cases is the power draw by the top disc. The power signals for the two cases are depicted in Figures 4.15. The steady state power for case 2, i.e. two discs stacked together, is 320 W while for the bottom disc only is 120 W, this means the power draw by the top disc only is 200 W.

The start-up power in both cases is exceedingly higher than the steady state power. This is expected with flywheels; more power is needed to get the discs to reach the operational speed. During the start-up, the energy is stored in the discs, only to be utilised when not enough power is supplied or after the motor is off.

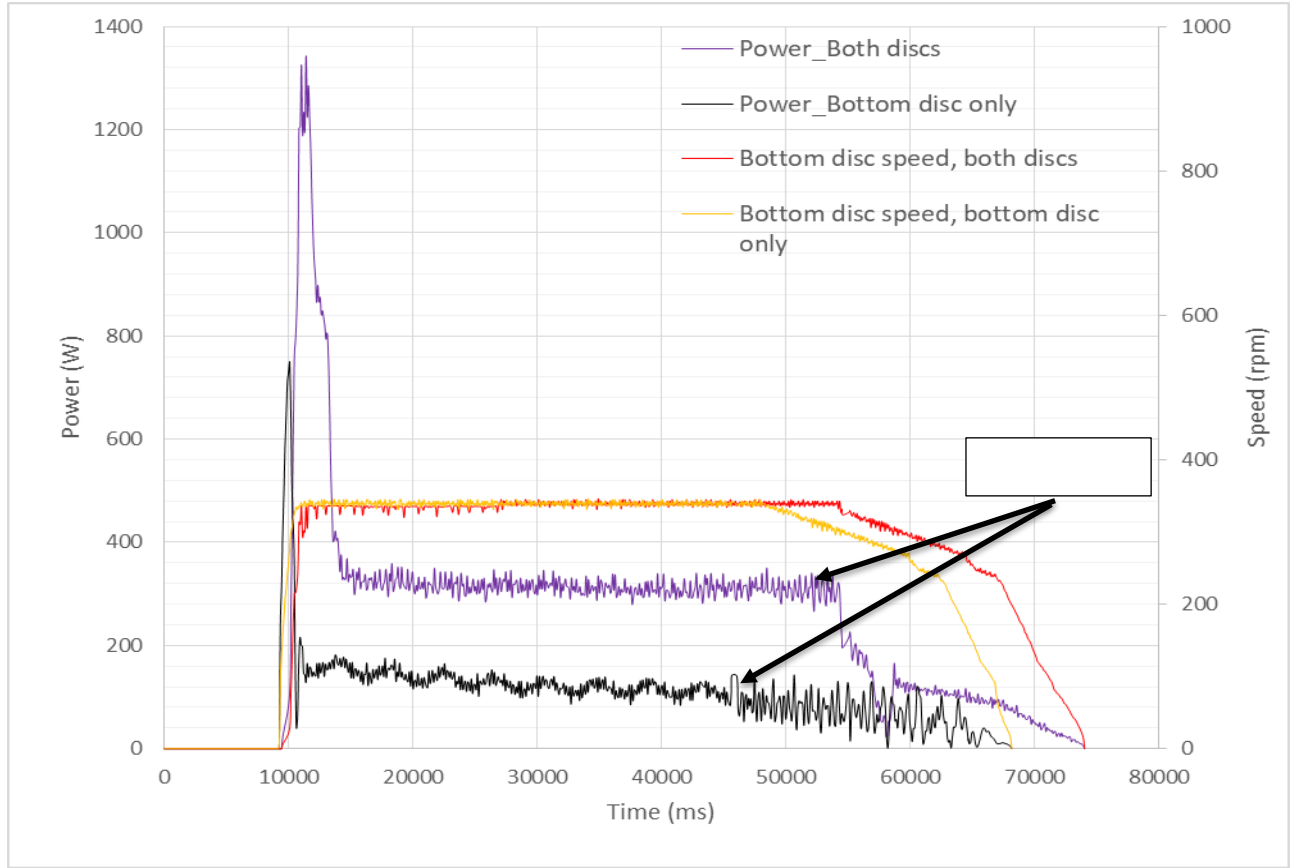


Figure 4. 15: The power signals for evaluation of crusher power draw for the two cases: with a bottom disc only and with both discs stacked together

The input currents for the motor were measured also measured using the clamp meter (as shown in Figure 4.16) for the case when the two discs are stacked together. The measured currents were used to compute the power input for the motor using Eq. (4.9). Results are presented in Figure 4.17. The power input during the steady state is 450 W as compared to 320 W (mechanical power draw by the discs). The motor has a power factor of about 0.82, implying its output power to be about 369 W. The 49 W difference between the motor output and mechanical power can be attributed to the power loss during transmission with a V-belt.

$$P_m = \sqrt{3} \times VI \quad (4.9)$$

Where P_m is the motor input power, V is the voltage in volts, I is the current in Amp.



Figure 4. 16: Current measurements for the live wire of the ROC motor using the clamp multi-meter

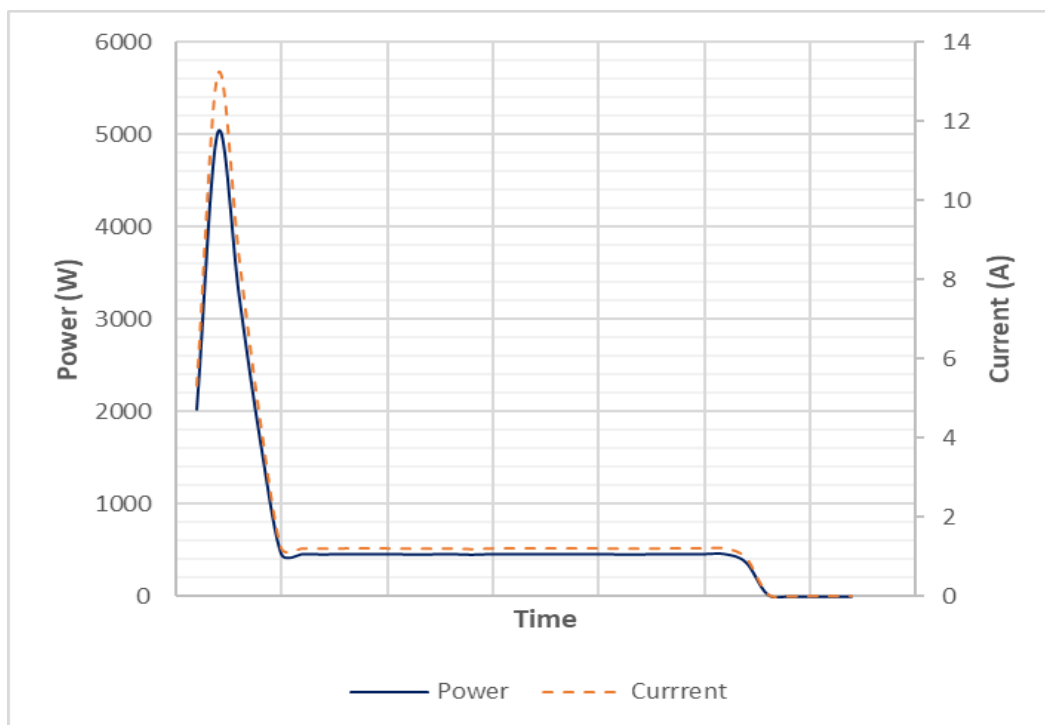


Figure 4. 17: Input power to the motor when the two discs are stacked together

4.4.2 Calculation of the specific comminution energy

After each ROC crushing test, the instrumentation data were used to calculate and plot signals for the speeds of the two discs, mass on the conveyor and power on the same set of axes. The typical plot is in Figure 4.18. The crushing time can be extrapolated from the plot for the top disc versus time. The plot for the crushing time is shown in Figure 4.19 (this is extracted from Figure 4.18). The total power (P_t) is taken from the curve of power-versus time during the time of crushing. The plot for the power (extracted from Figure 4.18) is shown in Figure 4.20. The “rise” in the power signal signifies locking of particle (s) between the two discs and commencing of breakage while the “drop” implies that breakage has happened, and the two discs are no longer locked together. In future, the semi-continuous operation should be considered to measure the power for say 10 minutes and assess whether the steady state can be reached.

The specific energies during crushing were estimated using Eq. (4.10). For each test, the total energy during crushing was calculated from the area under the power versus time curve using the numerical integration (trapezoidal rule). One such plot for the power versus time is shown in Figure 4.20.

$$E_{cs} = \frac{E_i}{3.6 \times m_s} \quad (4.10)$$

Where E_{cs} is the specific energy (in kWh/t), E_i is the input energy (in J) during crushing and m_s is the mass of the crushed rocks (in grams) and 3.6 is a factor for converting specific energy from J/g to kWh/t.

The framework for estimating the specific comminution energy is discussed further in section 7.3 of Chapter 7 with many operational signals given in Appendix U.

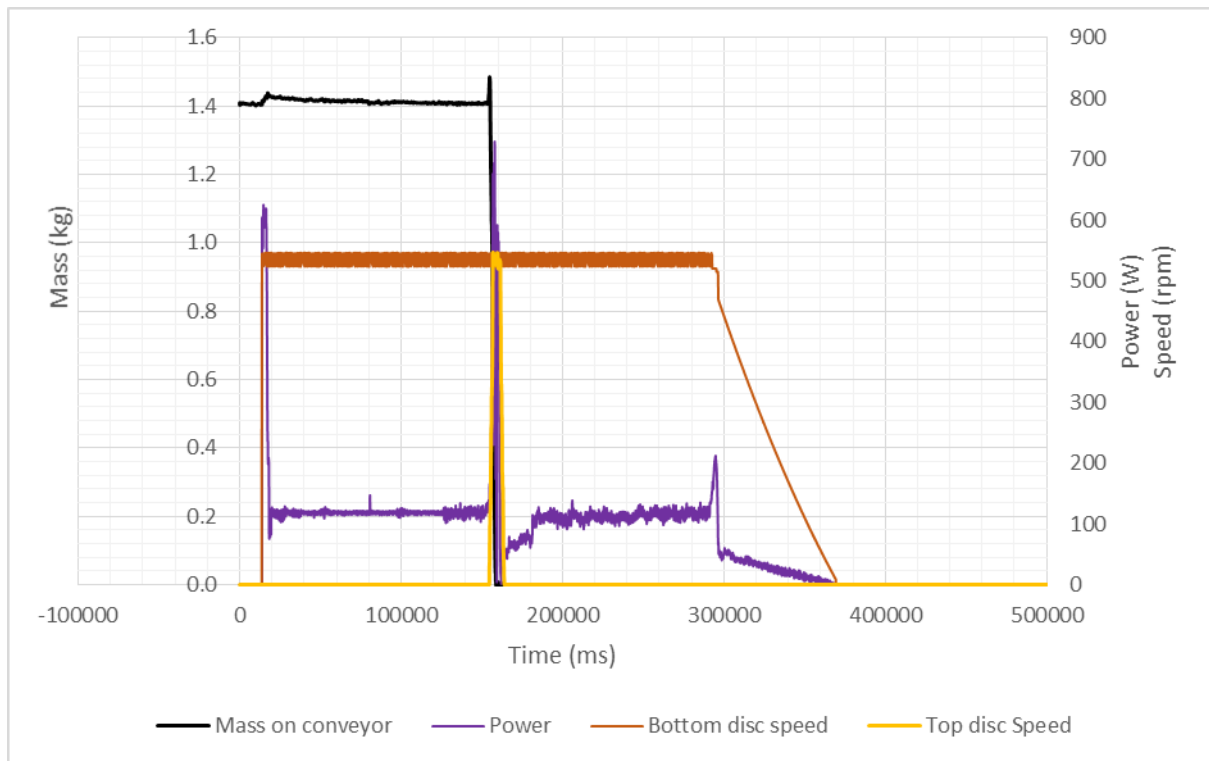


Figure 4. 18: The signals for the operating variables for one ROC crushing test

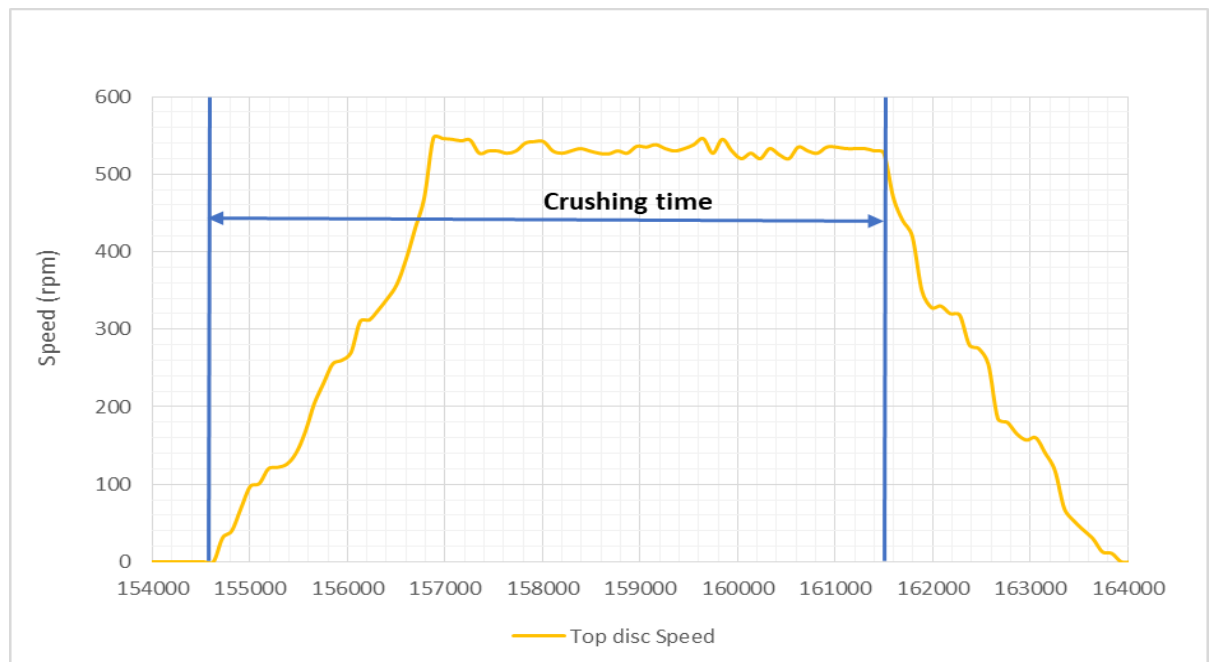


Figure 4. 19: Speed of top disc as a function of the operating time

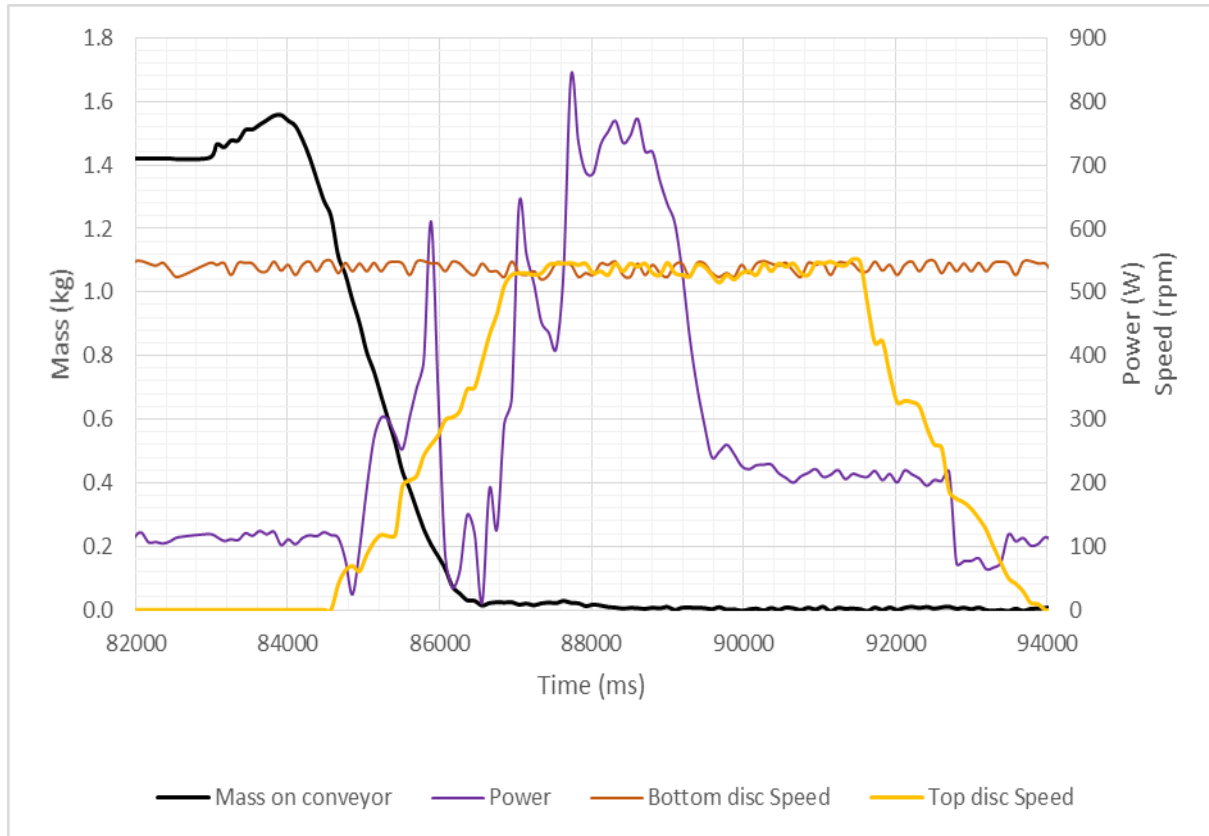


Figure 4. 20: Illustration of the changes in power, speeds and mass during comminution

4.6 Conclusions

In this chapter, the design considerations for the instrumentation circuits for the speed and load measurements were discussed. The use of the IR sensor to estimate the speeds of the discs have proven to be appropriate noting the accuracy level of 99 %. One improvement that may be considered, to ensure the accuracy is enhanced further, is to reduce the tolerance between the dimensions for the black and white patches (paintings of black and white) on both shafts to be at least ± 0.5 mm. Consideration of changing and optimising the sensitivity of the sensors may also improve the accuracy further. The use of the HX711 amplifier as a signal conditioner for the load cell outputs have proven to be suitable for ROC application especially for the measurement of the mass on the conveyor. Of critical importance for load measurement is the fixed voltage supply. The circuit needs to be investigated further to ensure that no instability

occurs. An investigation into the use of alternative signal conditioners for the load cell measuring the motor drive torque may be considered. More measurements for the input currents and voltage to the motor while in operation should be arranged during future tests to create a larger database that may be useful in conducting the energy balance of the ROC. The Arduino microcontroller, which is comparatively cheap and comes with free software, was found to be adequate for not so complex applications such as the laboratory ROC.

CHAPTER FIVE: ROC – ITS OPERATING PRINCIPLES AND TRANSPORTATION MODELLING

In Chapter 3, the fabricated ROC was described without addressing in detail the principles that govern its operation. This chapter explains the operating principles of the crusher; focusing on the crushing and transportation mechanisms. The influence of the offset between the vertical axes of the discs on the geometry of the crushing zone is also explained. Finally, modelling of the crusher throughput as a function of operating parameters (particle size, rotation speed, offset, and exit gap) was attempted using the data generated using the DEM software.

5.1 The operating principles of the ROC

The schematic diagram in Figure 5.1 shows the design and operational features for the built laboratory crusher. The conveyor belt is used to transport the feed material to the hopper. Under normal gravity, the material moves in the feed chute to the conical space (crushing zone) between the two discs. In the crushing zone, particles nipped between the fast spinning discs get comminuted by pressure (compression) breakage mechanism. In addition to compression, surface breakage (abrasion) takes place. Abrasion is due to bottom disc-particles, top disc-particles and particles-particles contacts. The discs-particles contacts are important noting that the top disc moves due to the friction between its interior surface and the particles trapped in the crushing zone. In other words, speed synchronization of the two discs is achieved with particle locking. While the particles-particles contacts are preferred, discs-particles contacts, especially with hard ore, may contribute to high operating costs as a result of high wear of the crushing surfaces. But this depends on the material of construction for the crushing surfaces. In our case, experimental time was too short to allow measurement of mass loss due to wear. Consideration of putting corrugated profiles such as a slight wave design in the future may be of benefits as they are said to ensure compound crushing by compression, tension and shearing

(Wills and Finch, 2016). Concentration of particles between the ridges is likely to increase inter-particles breakage which may result in higher reduction ratios.

Flexibility in the discs vertical movements also ensures impact breakage. While the flexibility in the disc movement can prevent jamming from occurring frequently, such as variation should be modest to ensure the particles are given sufficient residence time in the crushing zone to be reduced to the target product size. A modest variation in the space between the discs is adequate to bring about comminution of the inflexible rock particle. It is therefore important that the structure is robust enough to ensure the full transmission of energy to the particles in the crushing zone. The throw (vertical opening of any disc) of say more than 1 mm is not desirable as this may result in large flakes getting discharged.

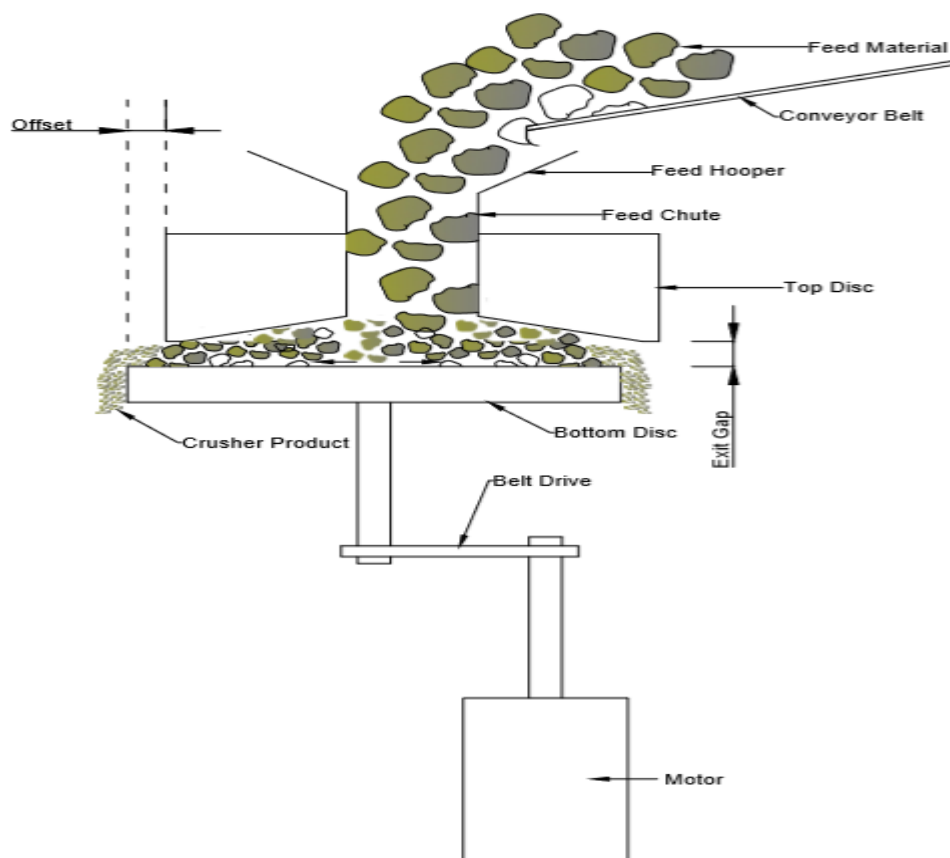


Figure 5. 1 The schematic showing the operating principle of the rotary offset crusher

5.1.1 Feed and product sizes

The largest particle size in the feed material is dependent on the throat (internal diameter) of the chute and most importantly on input gap (G_{in}) of the crusher. This input gap is a function of the angle of the comminution cavity as shown in Figure 5.2. A larger angle α suggests that coarser particles can be fed to the crusher. This angle greatly influences the geometry of the crushing chamber, and in turn, the capacity of the crusher. The angle of the comminution cavity for the current design used in this study is 4.76° . The influence of this angle on size reduction still needs to be investigated, perhaps using the DEM.

The largest particle discharged from the crusher is dependent, to a degree, on the exit gap (G_e). The two gaps are shown in Figure 5.2 and they are related by Eq. (5.1). To ensure safe operation, like in other crushers, the largest particle size must be smaller than the input gap. Other factors that are expected to affect the product size distribution are rotational speed, disc offset, profile design, feed rate and feed size. The effects of speed, offset, feed rate and feed size are discussed in Chapter 7.

$$G_i = h_c + G_e \quad (5.1)$$

Where h_c is the height of comminution cavity, G_e is the exit gap and G_i is the input gap.

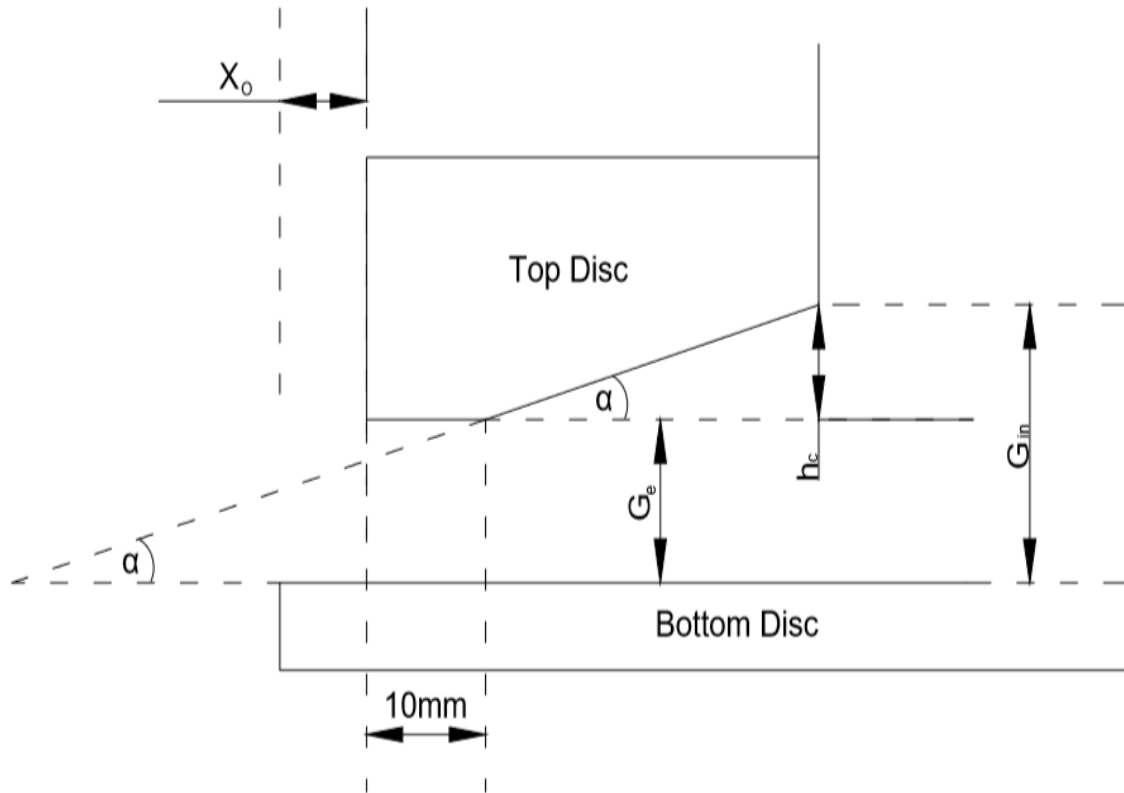


Figure 5. 2: Relationship between input and output gaps as well as the angle and the height of the comminution cavity, x_o is offset

5.2 Concept of horizontal offset explained

As shown in Figure 5.1, there is an offset between the vertical axes of the discs. This horizontal offset of the top disc relative to the bottom disc results in a change in the geometry of the crushing zone. The top view of the discs, in Figure 5.3, shows that in 180° , there is volume contraction, hence comminution predominantly happens in this half. On the other hand, there is volume expansion in the other 180° , implying that transportation is dominating. With the offset, as shown in Figures 5.3 and 5.4, the exit gap is smaller on the left side and larger on the right side. This implies that the largest particle that can be discharged from the crusher depends rather on the exit gap shown on the right side. The maximum exit gap ($G_{\theta, \max}$) is a function of the offset. The increase in the offset results in a larger $G_{\theta, \max}$. The effect of the variation in the

disc offset to the input and exit gaps was modelled and results are discussed in section 5.2.2. The effect of the variation in the discs offset on throughput is discussed in sections 5.4 and 5.5.

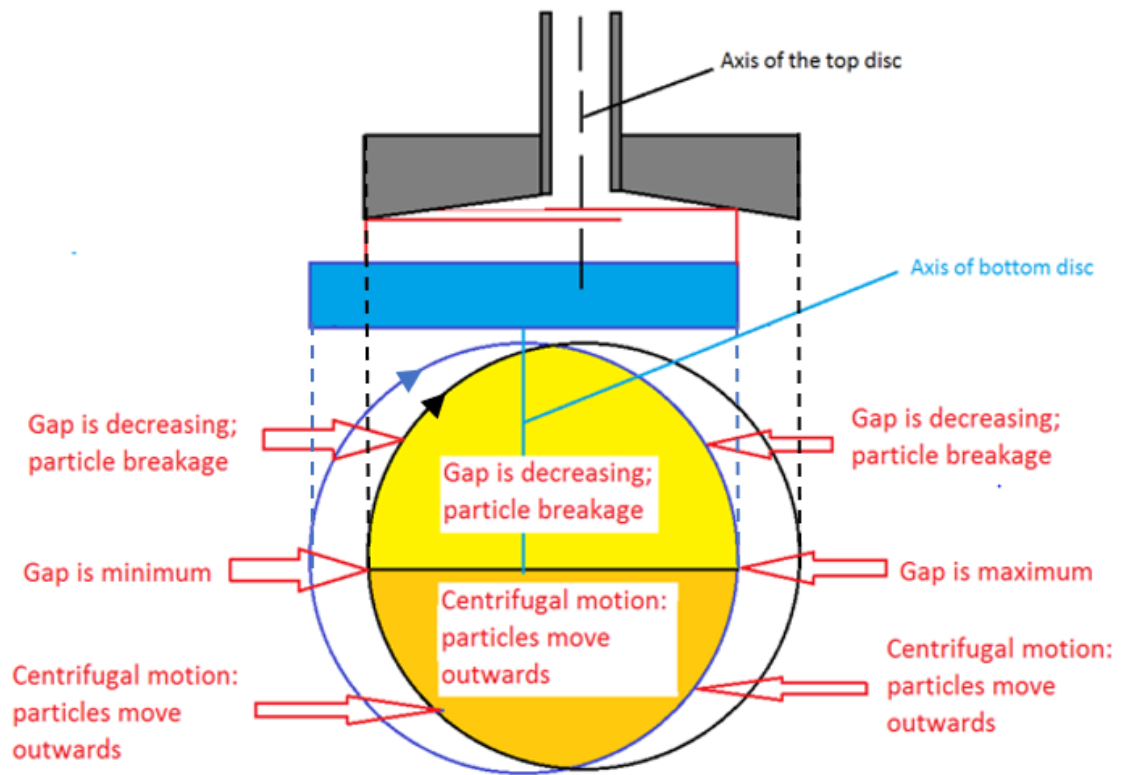


Figure 5. 3: Influence of disc offset on the geometry of the crushing zone and exit gap

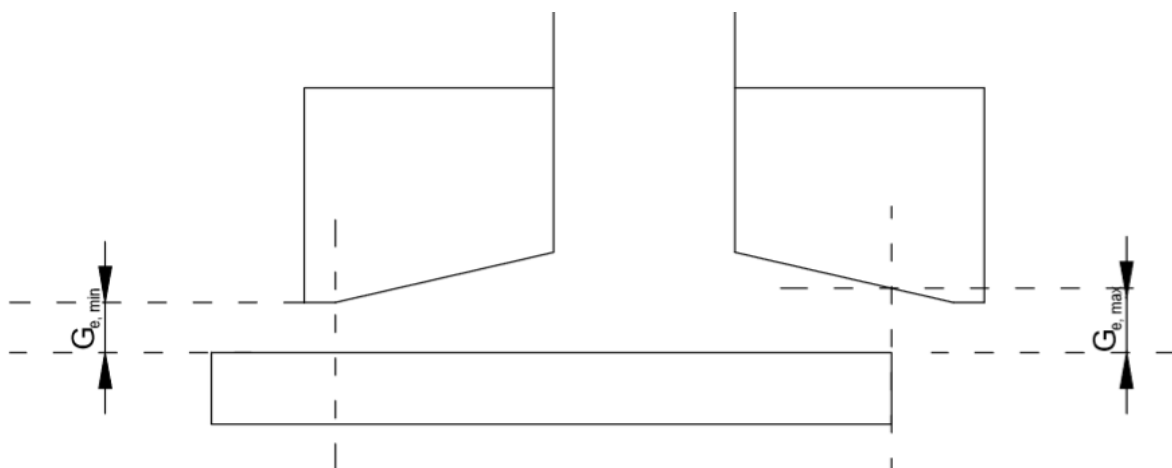


Figure 5. 4: Side view of the discs with offset greater than the interior flat edge of the top disc

It is important to note that there is only a variation in the input and exit gaps when the offset is greater than the interior flat edge of the top disc. This flat edge is 10 mm. With the offset equal to or less than the interior flat edge of the top disc, there is no variation in the exit gap as shown in Figure 5.5, but the volume expansion on the right half and volume contraction in the left half of the crushing zone still exists, i.e. comminution dominates in the left half while transportation is predominantly taking place in the right half of the conical space regardless of the offset.

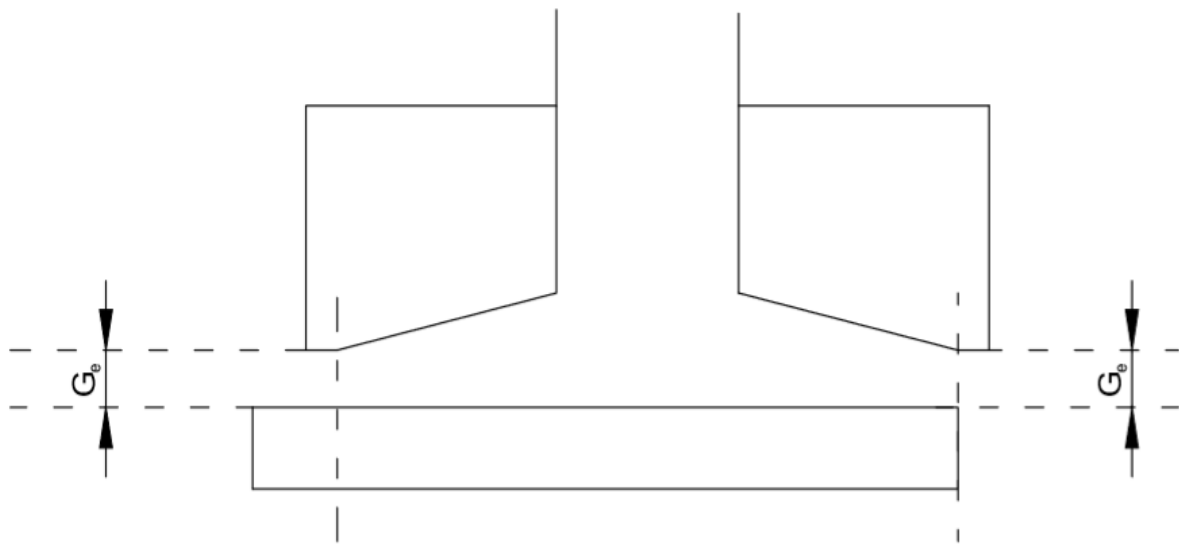


Figure 5. 5: Side view of the disc when offset is equal to interior flat edge of the top disc

5.2.1 Crushing actions

As discussed already, in each rotation, there is a closing event (crushing action/events) in 180° and opening event in the other half. What this implies is that the higher the speed of the discs, the higher the frequency of crushing actions. The number of crushing events for speeds in the range of 100 to 3000 rpm were computed by dividing the speed by 60 seconds/minutes and results are plotted in Figure 5.6. The data were fitted to the linear equation (shown as Eq. (5.2)) that can be used to find the number of closure or comminution events in a second for any speed

of rotation. For example, for the speed of 300 rpm, there are 5 closure/opening events in a second and for 1200 rpm there are 20 closure/opening events in a second. High speeds result in more crushing actions which suggest more breakage events by impact and compression while at low speeds abrasion tend to dominate. It is therefore important that the degree of size reduction be optimised by selecting the appropriate speed. Because the closure and opening events are equal in a rotation, Eq. (5.2) can as well be used to calculate the number of opening events in a second for any rotational speed.

$$N_E = 0.0167N \quad (5.2)$$

Where N_E is the number of closure/opening events and N is the speed of rotation in rpm.

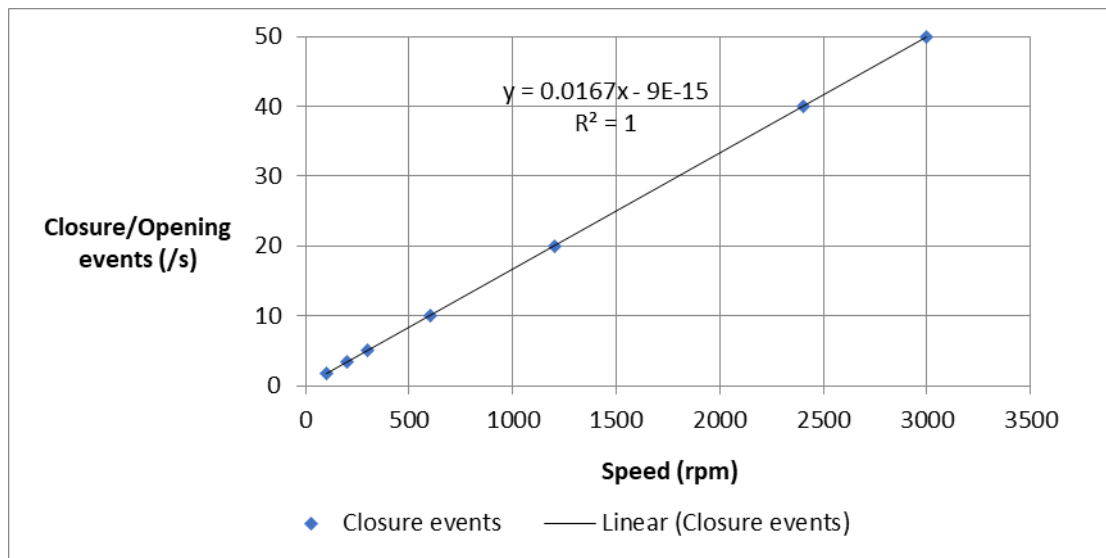


Figure 5. 6: Frequency of closure/opening events as a function of speed of rotation

5.2.2 Effect of offset on crushing chamber geometry

Drawing an x-y plane (centred at the origin of the bottom disc) between the two discs and reflecting their x and y values on this common plane, can help to find the differences in the x and y values for the two discs. With no offset between the z-axes of the two discs, the discs perfectly overlap each other, i.e. their x and y values (as shown in Figure 5.7 which depicts polar coordinates of the common plane) are equal and these values can be calculated using Eq. (5.3) and (5.4) respectively.

$$x = R \cos \theta \quad (5.3)$$

$$y = R \sin \theta \quad (5.4)$$

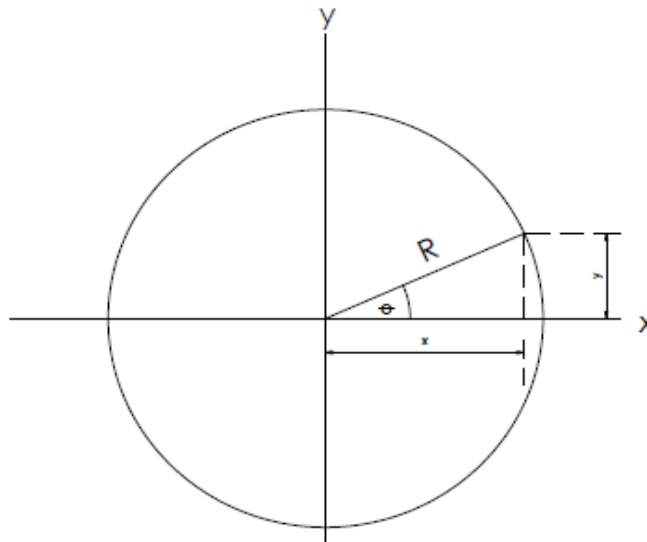


Figure 5. 7: Polar coordinates of the disc

With an offset, as shown in Figure 5.8, there are differences in the x and y values for the two discs relative to the x- and y axes of the bottom disc. The difference in the x and y values for the two discs can be calculated using Eq. (5.5) and Eq. (5.6) respectively.

$$x_{\text{diff}} = \pm R_{\text{diff}} \cos \theta \quad (5.5)$$

$$y_{\text{diff}} = \pm R_{\text{diff}} \sin \theta \quad (5.6)$$

Where R_{diff} is the radii difference (see Figure 5.8) and it is negative on the right side of the line of symmetry in Figure 5.8 and positive on the left side of the line of symmetry.

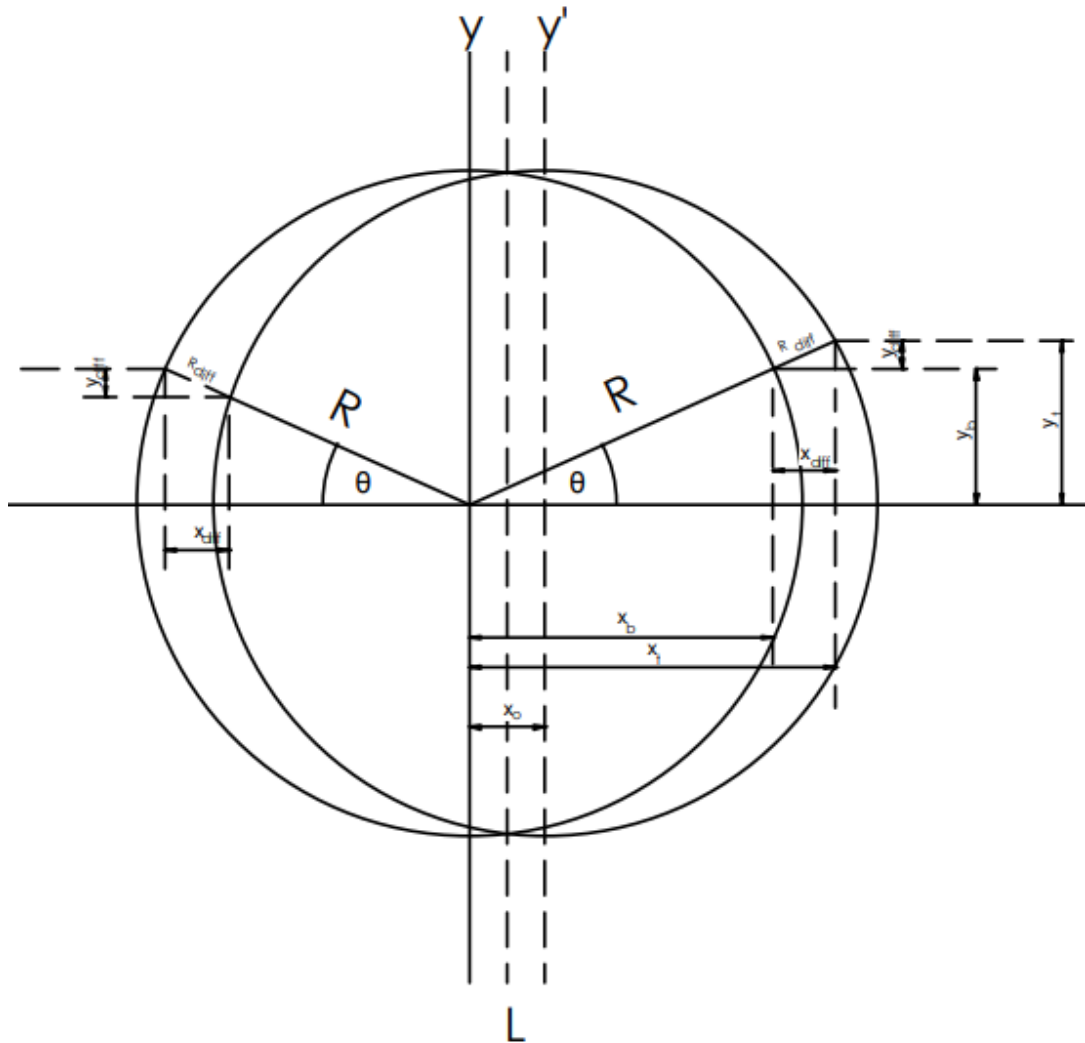


Figure 5. 8: Polar and Cartesian coordinates of the discs

5.2.2.1 Point difference with the offset in the x-direction

The point differences in the x-direction were calculated at three offset levels of 5, 10, 15 mm and a fixed value of 3 mm for the exit gap. Results are plotted in Figure 5.9 from the data in Appendix N. It is observed that the higher the horizontal discs offset the larger the x_{diff} values suggesting an expansion in the volume of the crushing chamber. While the change in the disc offset signifies either an expansion or contraction of the crushing chamber depending on

whether the offset is increased or decreased, the most significant information that can be derived from the x_{diff} is the variation in the input and exit gap of the crusher. This is discussed in the next sub-section.

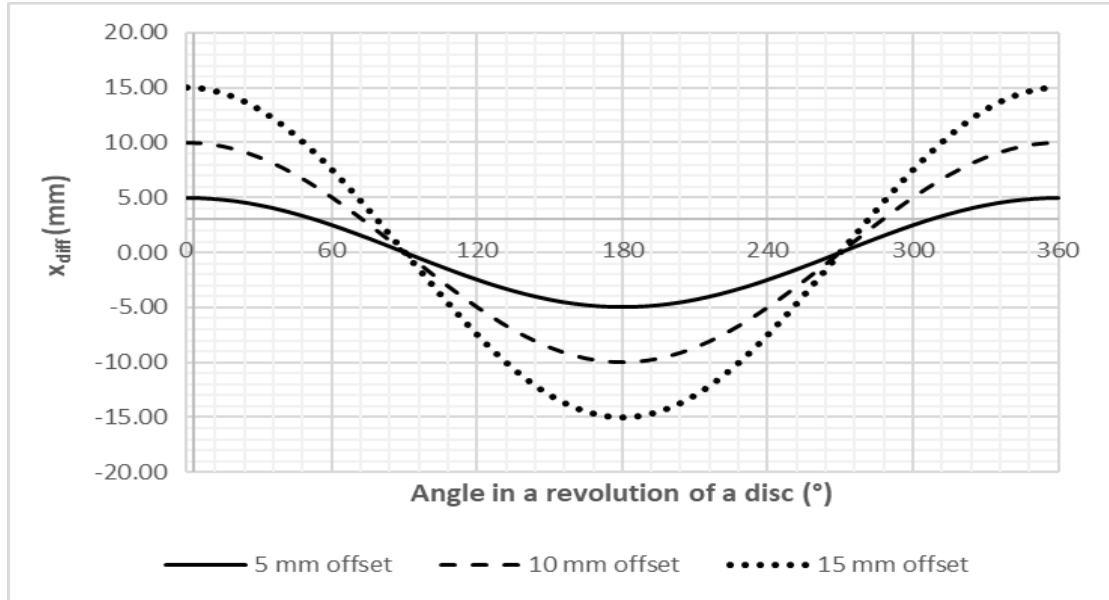


Figure 5. 9: Point difference for x values of the crusher discs at various crusher offsets and exit gap of 3 mm

5.2.2.1 Change in the input and exit gaps

The point differences for x values for the two discs help to estimate the expected variation in the crusher input and exit gaps. As shown in Figures 5.3 and 5.4, the exit gap (and hence the input gap) ranges between a minimum and maximum value when the offset is greater than 10 mm. The input gap (in the z-direction) can be related by simple geometry to the x values as shown in Figure 5.2 provided that the angle of the comminution cavity is known. This is done by defining the tangent of the angle α . For any disc offset, the input gap can be solved using Eq. (5.7). Because the x_{diff} is a function of angle $\theta \in [0, 2\pi]$, the variation of in the input gap can also be computed for angle $\theta \in [0, 2\pi]$.

$$z_{\text{in},x_{\text{diff}}} = G_i + x_{\text{diff}} \tan \alpha \quad (5.7)$$

Where $z_{\text{in},x_{\text{diff}}}$ is a new input gap given the horizontal offset in the x-direction.

The variations in the gaps were modelled using 15, 20 and 25 mm offsets and results are plotted in Figure 5.10 from the data in Appendix N. From Figure 5.10, as expected, it is shown that the variation in the input gap is directly proportional to the offset of the top disc in the x-direction. With the offset of 25 mm, the maximum variation in the input gap is 6 % (of the input gap of 23 mm). With a relationship between the input and exit gaps (shown in Eq. (5.8)), the exit gaps for the three offsets (15, 20 and 25 mm) were also calculated with the computations shown in Appendix N and results are plotted in Figure 5.14. It is worth analysing Figure 5.11 together with Figures 5.6 and 5.7. At 0°, the exit gap is maximum and at 180°, the exit gap is minimum. It was observed from Figure 5.11 that the offsets of 15, 20 and 25 mm, the exit gap increases in the two quadrants on the right side of the y-axis (see Figure 5.11) with a maximum at 0° increasing by about 2, 4 and 6 % respectively, implying that changing the offset (above 10 mm) by 5 mm translates into a 2 % increase in the exit gap at 0°.

$$G_{\theta} = z_{in,x_{diff}} - h_c \quad (5.8)$$

Where G_{θ} is the actual exit gap relative to a point at angle θ on the x-y plane and h_c is the height of comminution cavity, i.e. 20 mm for the current design.

With these results (in Figure 5.10 and 5.11), it can be hypothesised that increasing the offset ensures that particles are transported relatively faster in the crushing chamber, i.e. higher throughput. On the power draw, the hypothesis can be that the torque is inversely proportional to the offset of the top disc. This is because, given a larger offset (and hence a larger gap), the probability of particles getting released when they are arrested is high. It should, however, be stated that a very high offset would ensure high throughput but may be detrimental to the size reduction. In section 5.4 and Chapter 7, simulation and experimental results were used to test these hypotheses.

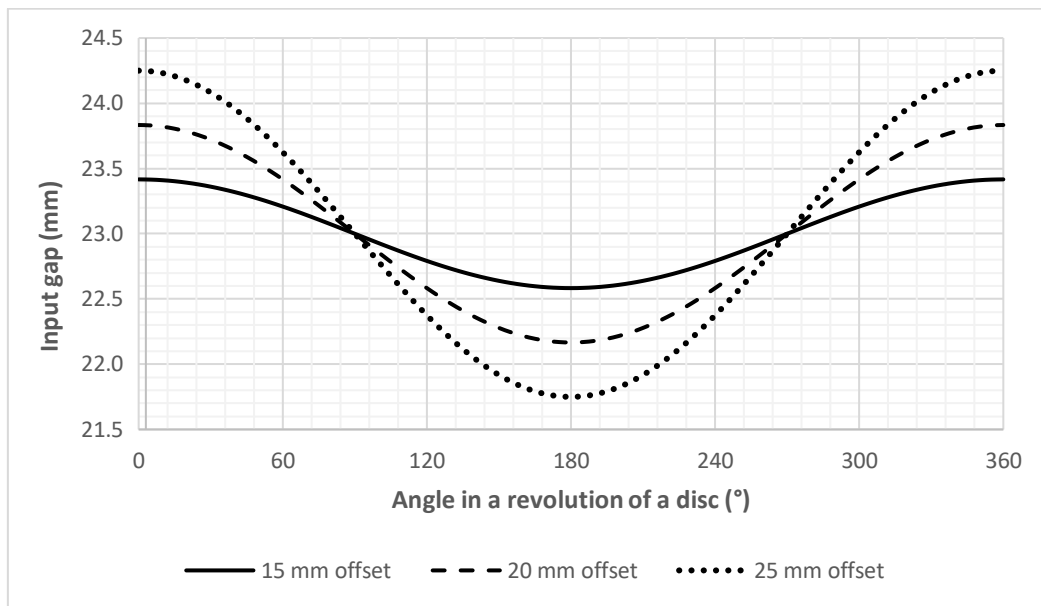


Figure 5. 10: The variation in input gap as a function of discs offset in x direction of crushing chamber geometry

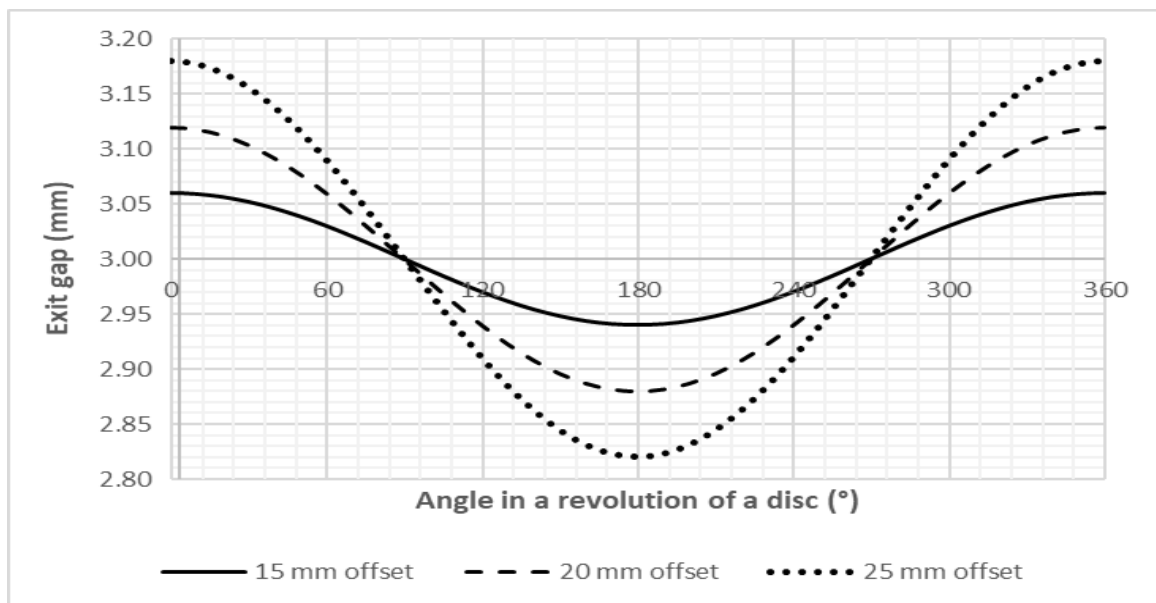


Figure 5. 11: The change in exit gap as a function of discs offset in x direction of crushing chamber geometry

5.3 Transportation in the feeding and crushing zones

In the crushing zone, the particles move to the periphery as illustrated in Figure 5.12 with centrifugal acceleration (given by Eq. (5.9)) while in the feeding zone (specifically in the chute) particles fall with a constant acceleration due to gravity (9.81 m/s^2).

$$a_c = \omega^2 r \quad (5.9)$$

Where ω is the angular velocity (in rad/s) for the discs and r is the radius of the discs.

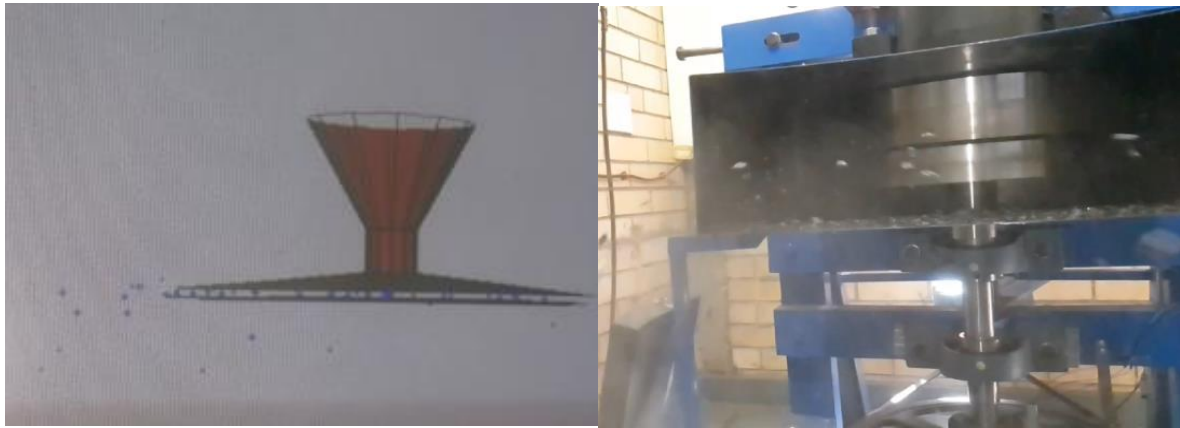


Figure 5. 12: A photograph from DEM showing the progression of particles in the comminution cavity with centrifugal acceleration (on the left) and a photograph showing the trajectories of the coal particle leaving the crushing zone (on the right)

Using Eq. (5.9), the centrifugal acceleration in the crushing zone as a function of the radius of the disc were evaluated for the speeds of 330 and 550 rpm (the two speeds considered for the ROC crushing tests as discussed in section 3.3.2 in Chapter 3). Results are plotted in Figure 5.13 and calculations are shown in Appendix O. As Eq. (5.9) shows the centrifugal acceleration increases with the radius of the discs. It is observed that increasing the speed by a factor of 1.7 (i.e. 550 divided by 330) results in the centrifugal acceleration increasing by a factor of 2.8. This factor was obtained by dividing the slopes of the two graphs.

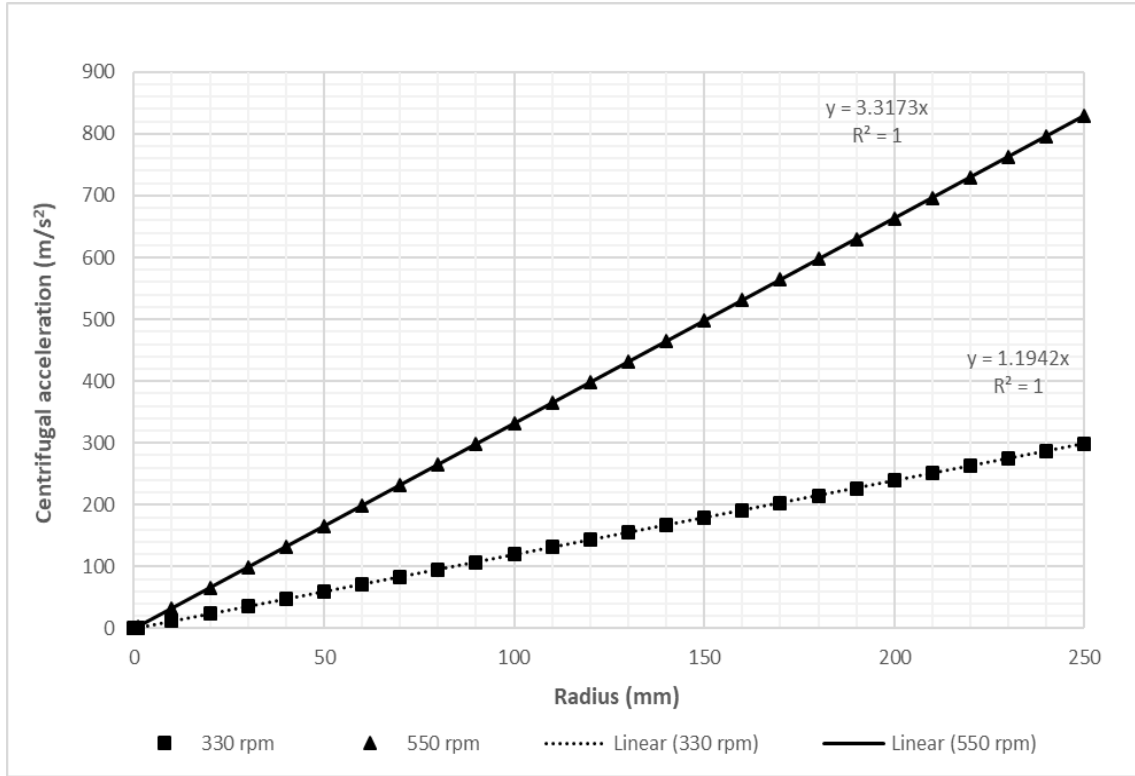


Figure 5. 13: Relationship between the centrifugal acceleration acting on particles in the crushing chamber of the rotary offset crusher and the radius of the discs at various speeds: 330 and 550 rpm

To find out how fast the particles are, in the crushing zone move with centrifugal acceleration as compared to the particles falling under normal gravity in the feeding zone (feed chute), the ratios of the calculated centrifugal acceleration and acceleration due to gravity (9.81 m/s^2), were computed using Eq. (5.14) and the values obtained are listed in Table O1 in Appendix O and plotted in Figures 5.14 for the speeds of 330 and 550 rpm.

$$R_{\text{centr./g}} = \frac{\omega^2 r}{g} \quad (5.10)$$

Where $R_{\text{centr./g}}$ is the ratio of the acceleration of particles in the crushing zone and the acceleration of particles in the feeding shaft, g is the acceleration due to gravity in m/s^2 and r is the radius of the disc in m along the comminution cavity.

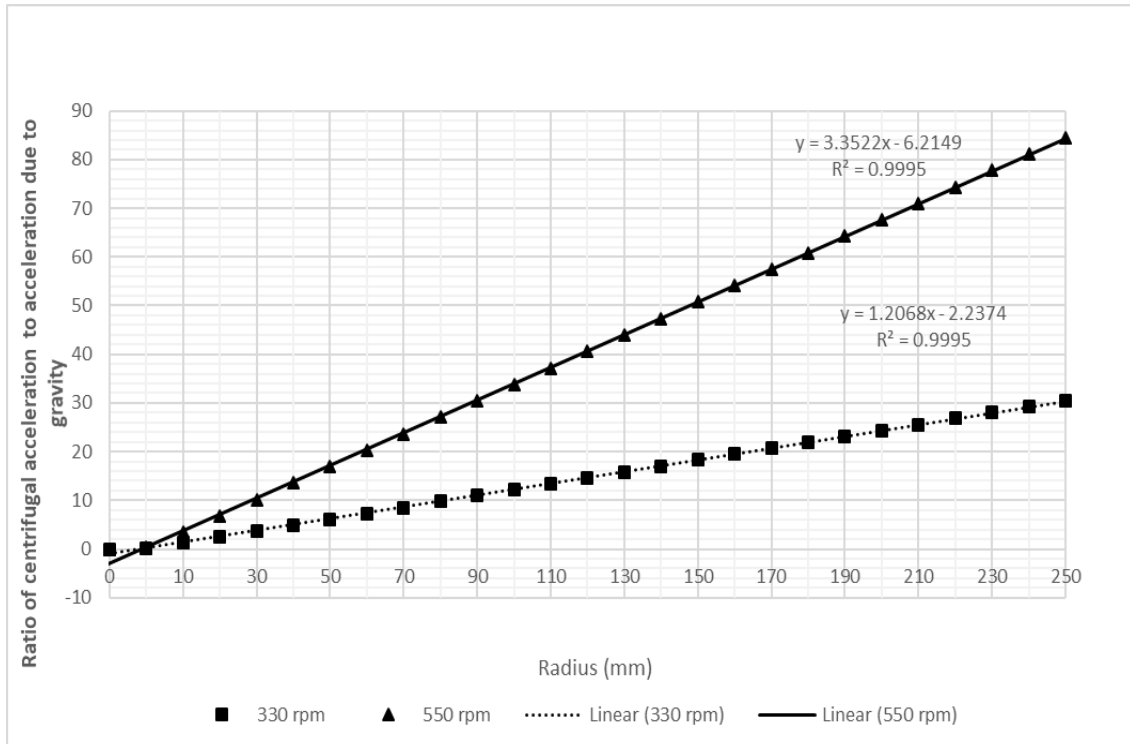


Figure 5. 14: Ratio of centrifugal acceleration and acceleration due to gravity as a function of the radius of the disc at speeds of 330 and 550 rpm

Regardless of the speed, with these modelling results in Figure 5.14, it can be argued that the ROC has potential to be a high throughput crusher (especially at high speeds), but there is a need to ensure that the feeding rate is not a constraint during the operation of the crusher. Fast delivery of the material to the crushing zone is, therefore, a requirement to ensure steady state operation. The y-intercepts for the straight lines (trendlines) for both speeds are not zero, which suggests that there is a curvature in the data. This curvature can be observed for radii less than 10 mm. However, for the purpose of evaluating the trends in the motion of particles nipped between the discs (radius: 50 – 250 mm), the linear fit is still adequate.

5.4 Effect of operating parameters on throughput

Overall, the ROC capacity is a function of the following: feeding conditions (especially the feed rate), material properties (size, density and hardness), feed hopper capacity, feed chute dimensions (throat diameter and height), geometry of the crushing zone (angle α and height h ,

disc offset, input and exit gaps), rotational speed, and discs profiles. Of all these factors affecting the capacity of the crusher, the DEM was used to understand the influence of particle size, rotational speed, offset and exit gap. Results are discussed in the following subsections.

5.4.1 Effect of Particle size

The influence of particle size on the throughput was simulated using 4, 7 and 10 mm steel balls while keeping the offset at 10 mm, the speed at 765 rpm and exit gap at 15 mm. As a basis for comparison, same mass (1.5 kg) each ball size was used. To get the number of balls for each ball size that gives 1.5 kg mass, Eq. (5.11) is used which divides the target mass (1.5 kg) by the unit mass (given by Eq. (5.12))

$$N_p = \frac{m_t}{m_s} \quad (5.11)$$

Where N_p is the number of particles, M_t is the total input mass of balls and M_s is the mass of a single particle (ball) given by Eq. (5.12).

$$M_s = \frac{\pi d_i^3 \rho}{6} \quad (5.12)$$

Where d_i is the diameter of the ball (in m) and ρ is the density of the steel ball used (7700 kg/m³)

The distributions for the three ball sizes are shown in Figure 5.15. It is observed that there is a positive relationship between the ball size and throughput, i.e. large balls get discharged faster from the crusher. The 10 mm balls are expected to travel faster in both feed chute and crushing zone than 4 mm balls. It is, however, observed that for the time less than 0.6 second, the 10 mm ball graph is between the graphs for the 4 and 7 mm balls. A possible reason for this flow behaviour is that the simulations were conducted with no deflector (small cone in the centre of the bottom disc in the laboratory crusher). Simulations with a deflector are being conducted by

William Gumbi, as part of his MSc and it is not expected for graphs of various ball sizes to intersect.

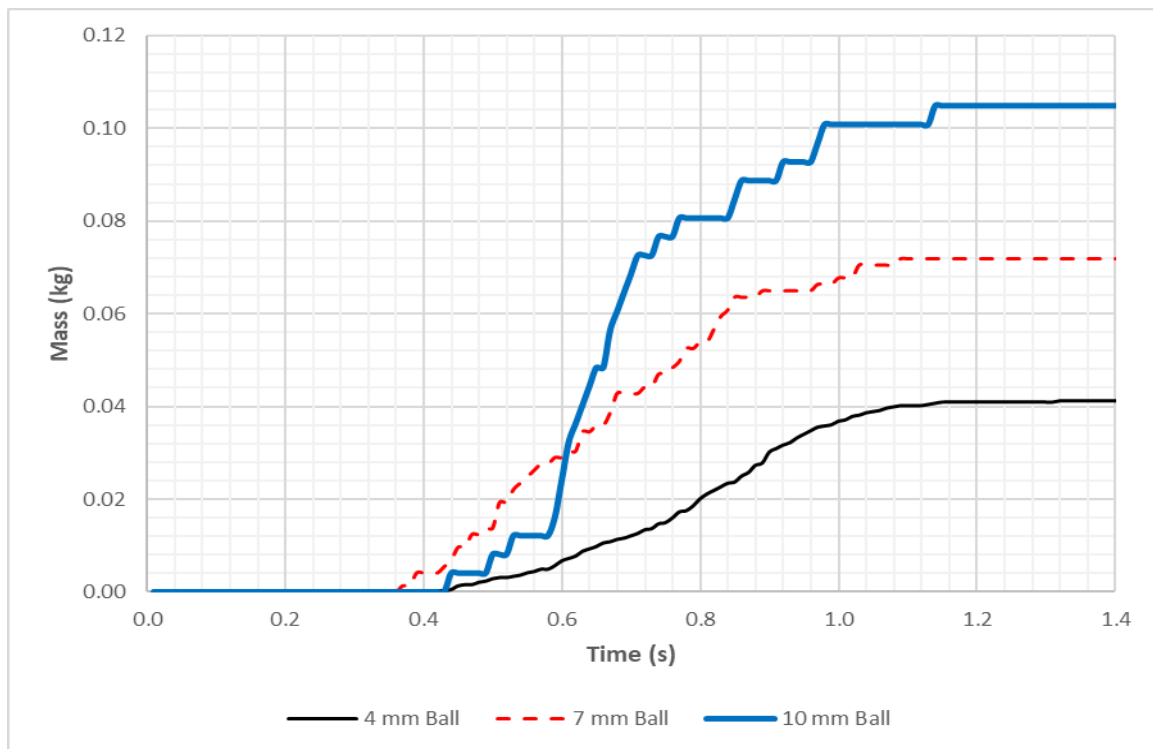


Figure 5. 15: Effect of ball size on transportation of particles in the rotary offset crusher operating with a rotation speed 765 rpm, the offset of 10 mm and exit gap of 15 mm

5.4.2 Effect of rotational speed

The effect of the rotational speed of the disc on transportation was investigated at the levels of 330, 765 and 1200 rpm, with the ball size of 4 mm, offset of 10 mm and exit gap of 7 mm. The same number of balls are fed to the crusher at three speeds. Results are plotted in Figure 5.16. Increasing the speed from 330 rpm to 765 rpm resulted in many balls being discharged. This agrees with what is shown in Figures 5.13 and 5.14, i.e. doubling the speed results in fast transportation of particles in the crushing zone. With 330 rpm, the maximum number of balls discharged from the crusher is 35 as compared to 49 balls when the speed was increased to 765 rpm. However, a further increase in the rotational speed to 1200 rpm did not result in a significant increase in the number of balls discharged.

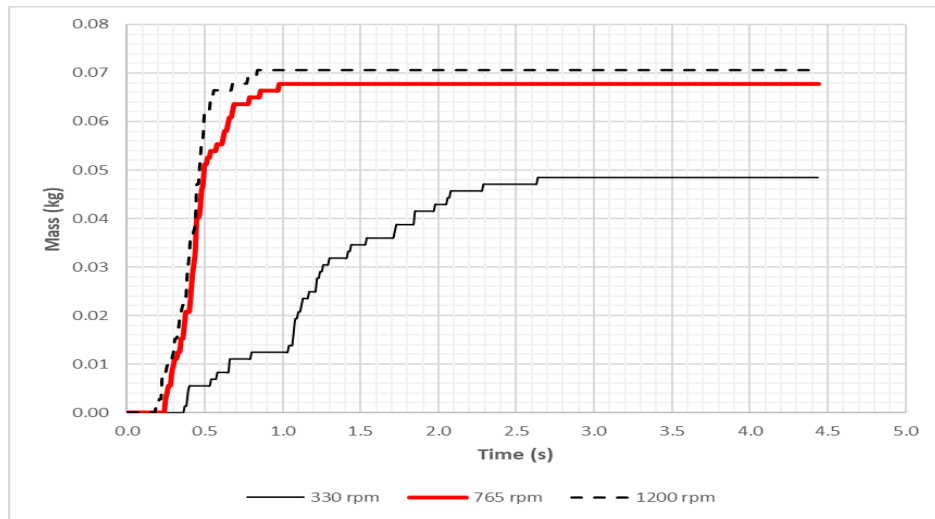


Figure 5. 16: Effect of speed on crusher throughput operating with ball size of 7 mm, offset of 10 mm and exit gap of 7 mm

5.4.3 Effect of offset

Understanding the effect of offset on the capacity of the crusher is one of the objectives of this study. The offset of the vertical axes for the two discs was varied at three levels: 5, 10 and 15 mm. The rotational speed of the discs was kept at 765 rpm, ball size was 4 mm and the vertical exit gap was fixed at 10 mm. From Figure 5.17, it is observed that the offset had no significant influence on the number of particles discharged from the crusher at the crusher settings considered. This contradicts what has been stated in section 5.2 that the throughput increases with offset. Such an anomaly in the results can be attributed to two reasons:

- (1) With disc offset value less than or equal to 10 mm, the exit gap is the same (see Figure 5.5) and hence no influence on transportation.
- (2) The absence of breakage in the ROC for the DEM simulations conducted. With the offset of 15 mm, the exit gap increases by 2 % as discussed in section 5.2.2, with breakage that increase in the exit gap is expected to result in a relative increase in the crusher throughput. It is therefore recommended that breakage be incorporated in the DEM simulation in the future

and similar simulations be conducted to ascertain whether this claim is true. The DEM top disc also needs to be driven by particles as it is the case with a laboratory prototype.

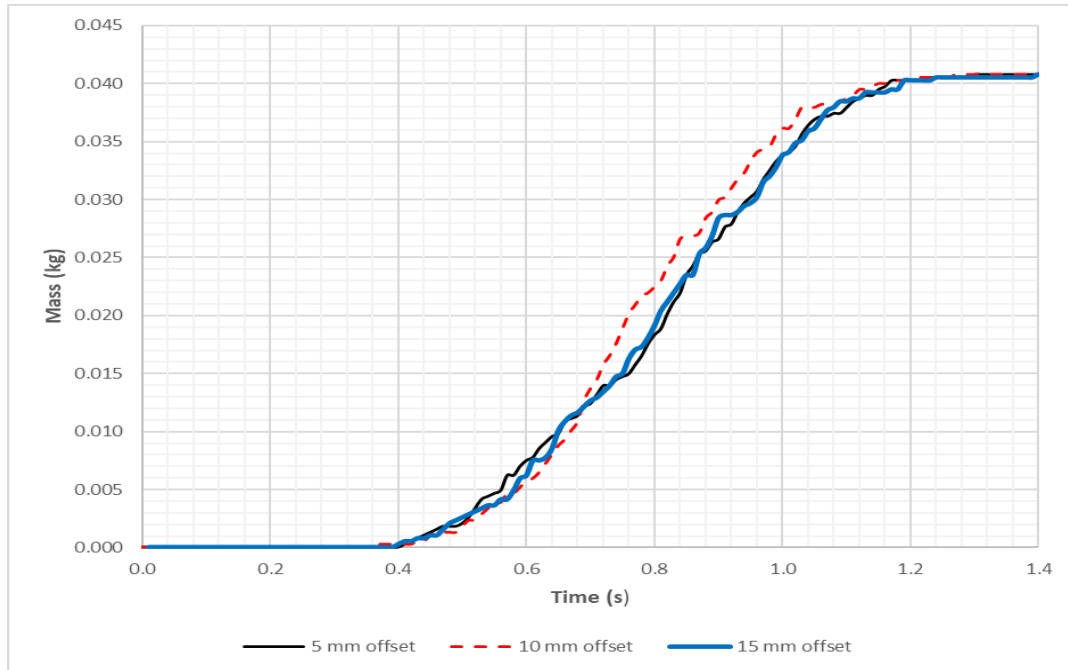


Figure 5. 17: Effect of offset on the crusher throughput with ball size of 4 mm, speed 765 rpm and gap 10 mm

5.4.4 Effect of Exit gap

The effect of the exit gap was investigated while keeping the speed, offset and ball size at 765 rpm, 10 mm and 4 mm respectively. Figure 5.18 shows the distributions for the three exit gaps considered (5, 10 and 15 mm). A trend that is expected is that increasing the exit gap of the crusher results in many balls (particles) discharged, i.e. throughput increases with the increase in the exit gap. While such a trend is observed for the 5 and 10 mm exit gaps, further increase in the exit gap to 15 mm did not show improvement in the crusher throughput.

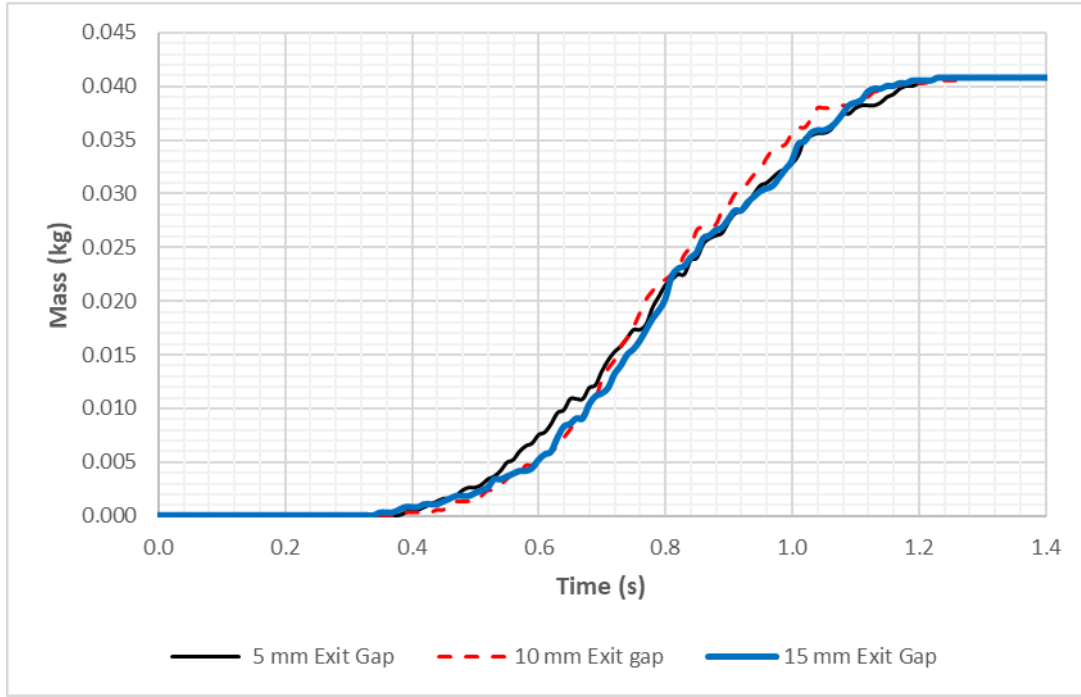


Figure 5. 18: Effect of exit gap on crusher throughput with ball size of 4 mm, speed of 765 rpm and offset of 10 mm

5.5 Regression modelling of crusher throughput

Using the DEM, 18 simulations at the operating conditions discussed in Chapter 3, i.e. feed size (4, 7 and 10 mm), speed (330, 765 and 1200 rpm), offset (5, 10 and 15 mm) and exit gap (4, 7 and 10 mm) were run and the throughputs were computed at various combination of crusher settings. The calculated values are summarised in Table O2 in Appendix O. Multiple regression modelling was conducted using the *Statgraphics 18*[®] statistical software and the fitted model is shown in Eq. (5.13).

$$Q = -0.143 + 0.0486d_i + 0.000425n - 0.00343x_0 - 0.0343G_e \quad (5.13)$$

Where Q is the throughput in tph, d_i is the ball size in mm, n is the rotational speed of the discs in rpm, x_0 is the offset and G_e is the exit gap.

To assess the reliability of the fitted model, the predicted throughputs were plotted against the simulated throughputs as depicted in Figure 5.19. The R^2 statistic indicates that the model as fitted explains 88 % of the variability in throughput.

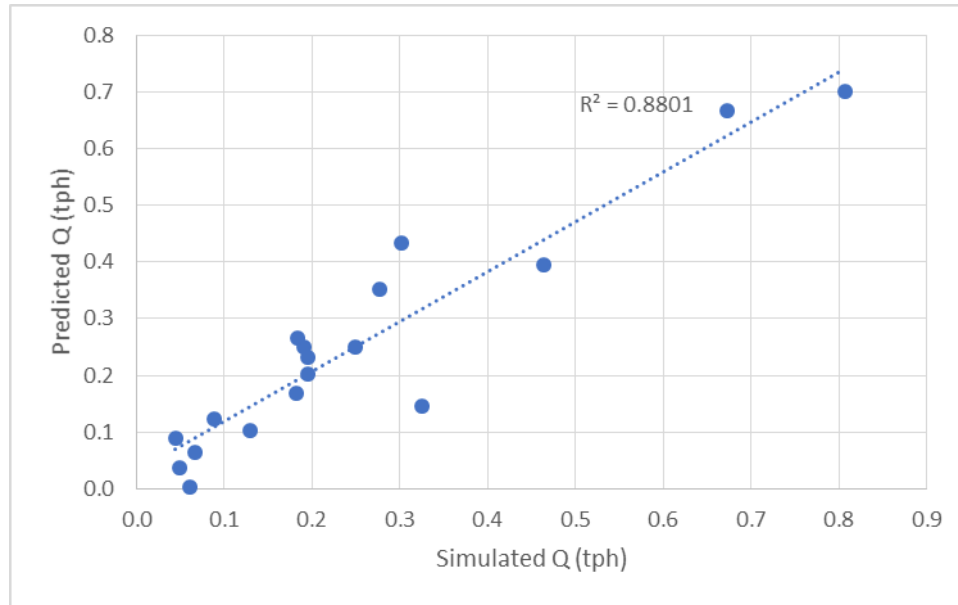


Figure 5. 19: Simulated versus model throughput values

More statistical data for this model are listed in Appendix O. Since the P-value (equal to 0.00) in the ANOVA table (Table O3 in Appendix O) is less than 0.05, there is a statistically significant relationship between the variables at the 95 % confidence level. In determining whether the model can be simplified, it was observed that the highest P-value (in Table 5.1 below) on the independent variables is 0.5303, belonging to the disc offset. Since the P-value is greater than 0.05, that term is not statistically significant at the 95 % or higher confidence level. This agrees with what is shown in Figure 5.17 that there is no definite relationship between the offset and the throughput. But as already stated, this is only true for the offset values smaller than or equal to 10 mm and for the offset greater than 10 mm, results are expected to be different if breakage is incorporated in the DEM simulation recipe.

Table 5. 1: Standard errors and P-values for the estimated coefficients of throughput model

<i>Parameter</i>	<i>Estimate</i>	<i>Standard Error</i>	<i>P-Value</i>
CONSTANT	-0.143	0.114	0.231
Feed size (mm)	0.0486	0.00879	0.0001
Speed (rpm)	0.000425	0.0000606	0.0000
Offset (mm)	-0.0034	0.00527	0.5303
Exit gap (mm)	-0.0343	0.00879	0.0018

5.7 Summary

The rotary offset crusher is explained in much detail with the operating principles guiding the size reduction and transportation put into perspectives. Comminution happens in the crushing zone by compression, impact and abrasion and it is mainly dependent on the rotational speed of the discs. The offset between the vertical axes of the discs ensures the change in the geometry of the crushing chamber. This happens in such a way that in 180° of a revolution there is volume contraction (ensuring comminution) and in the other 180°, there is volume expansion (facilitating transportation of the material). The exit gap changes with offset values that are greater than 10 mm in such a way that a 5 mm change in the offset equals a 2 % increase in the exit gap for the half of the crushing zone. From the DEM results, the speed of discs has proven to be the chief operating variable affecting the crusher throughput. As expected, based on the results for the DEM simulations, the offset (with values equal to or less than 10 mm) does not have a significant effect on the throughput. At higher offset values (more than 10 mm), it is expected that there is a positive relationship between the disc offset and throughput given the variation in the exit gap of the crusher. But that would come to light only if breakage is incorporated into the DEM “recipe” for the ROC.

CHAPTER SIX: BREAKAGE CHARACTERISATION OF COAL AND QUARTZ

6.1 Overview

The two materials (coal and quartz) used to assess the Rotary Offset Crusher's efficiency in breakage were characterised using the laboratory breakage testing devices. The drop weight tester was used for single particle breakage while the piston and die tester was used for compressed bed breakage. The aim of these experiments was to characterise these materials in terms of breakage and relate the results to those of the crusher. The Bond work indices for coal and quartz are 13 kWh/t and quartz 13.57 kWh/t respectively, (Wills and Finch, 2016). Noting that the Bond work index, by definition, is the kWh/t per tonne required to reduce the infinite feed size to the product with a d_{80} of 100 μm , (Napier-Munn et al., 2005), these values suggest that coal and quartz have comparatively the same grindability.

6.2 Single particle impact breakage

For both coal and quartz, the -19+13.2 mm particles were subjected to three energy levels while the -13.2+9.5 mm were subjected to four levels. In each test (for each energy level), on average, more than 50 particles were used. The single particle impact breakage tests were evaluated through the size distributions of the progeny particles shown in Figure 6.1. For quartz, the breakage size distributions showed that increasing impact energy results in a finer product. In the case of -19+13.2 mm of coal, the increase of the impact energy increases the fraction of +2 mm daughter particles while decreasing the fraction of -2 mm progeny particles in the product. For example, the -19+13.2 mm of coal at a low energy level of 0.23 kWh/t has more fines (-2 mm) than all other (same size class, but high specific energy levels). For the -13.2+9.5 mm coal particles, the increase in breakage energy level increases the fineness of the breakage

distribution for the +500 μm product size fractions. Considering the -500 μm size range, the curves for the low energy levels (0.58 and 1.01 kWh/t) are on the right of the curves for high energy levels (1.54 and 2.33 kWh/t). While a typical breakage behaviour is expected for the same material, anomaly results due to inherent variation in the microcracks can be recorded. To counter the influence of variation in distributions of cracks in particles of same size class, many particles (more than 50 per test) were individually impacted with their consolidated daughter particles screened to have the average breakage distributions shown in Figure 6.1. Genc et al. (2004) reported similar results for the copper ore and they attributed such deviations to inhomogeneity of the ore and thus implying the difference in microstructure, mineralogical composition and distributions microcracks. Relationships between input energy and dependent variables (d_{80} and t_{10}) were established from the experimental data as discussed in subsections 6.2.1 and 6.2.2.

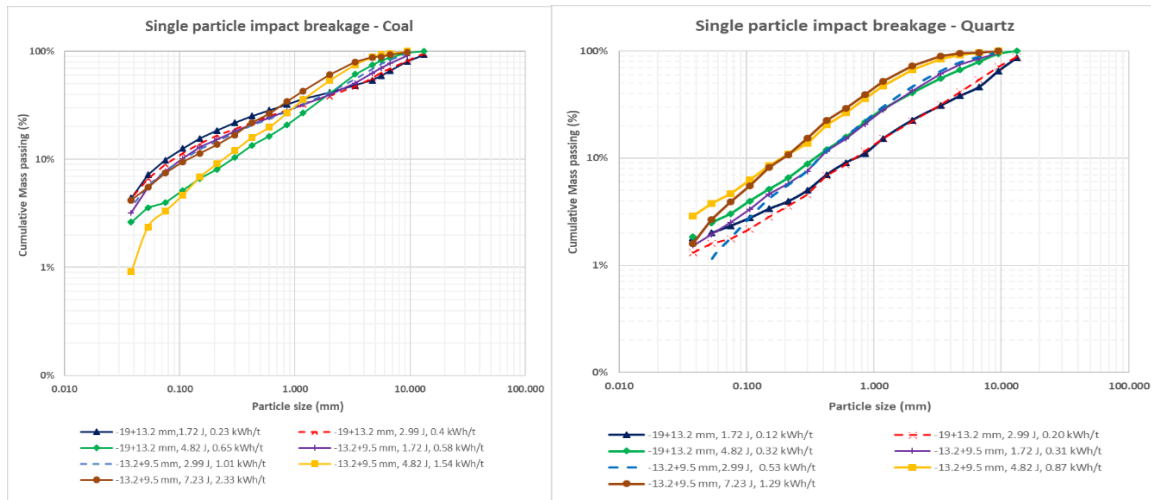


Figure 6. 1: Size distributions of -19+13.2 mm and -13.2+9.5 mm coal and quartz samples subjected to single particle impact breakage at various energy levels

6.2.1 The d_{80} size as a function of impact energy

The 80 % passing sizes (d_{80}) from the cumulative size distributions in Figure 6.1 were extrapolated and plotted against the input energy (in kWh/t). The specific comminution energies were evaluated using Eq. (2.10). The results are shown in Figures 6.2 and 6.3 for -

19+13.2 mm and -13.2+9.5 mm respectively, for both coal and quartz. As expected, the d_{80} decreases (implying finer product) with the increase in impact energy. What can be deduced from Figures 6.2 and 6.3, is that quartz requires less energy to get broken than coal. This is seen from the quartz curves being below those of coal. This suggests that coal is relatively stronger (tougher) than quartz (brittle material).

The data were fitted to the power function with the general equation taking the form shown in Eq. (6.1).

$$d_{80} = pE_{cs}^{-q} \quad (6.1)$$

Where p and q are model parameters.

The exponent q , which indicates the decay rate of the power function converges to 0.6 for both coal and quartz. Since constant q represents the slope of the linearized power function, same q values for both coal and quartz suggests that the two material have a relatively same relationship between the product size distribution (in terms of d_{80}) and input impact energy (in kWh/t). It is recommended that many more experiments are conducted at a wide range of energy levels to substantiate the above claim. On the other hand, the p values for the power functions of quartz are relatively smaller than those for the coal graphs. Noting that p is the y-intercept of the linearized Eq. (6.1), it can be deduced that quartz would generally have smaller d_{80} (finer product) if subjected to the same impact energy as coal. This argument can be supported by the fact the graphs of quartz are below those of coal. The results in Figures 6.2 and 6.3 were correlated to the ROC results in Chapter 7 to assess its energy efficiency.

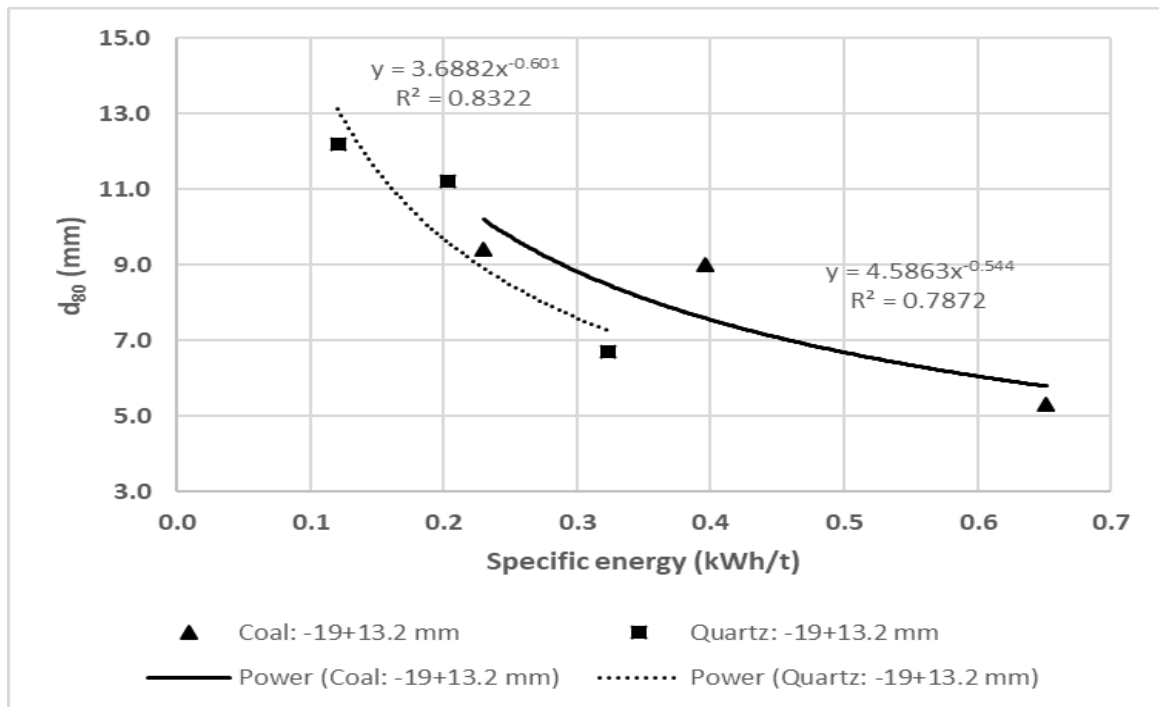


Figure 6. 2: Relationship between the d_{80} sizes and input impact energy for the -19+13.2 mm of coal and quartz

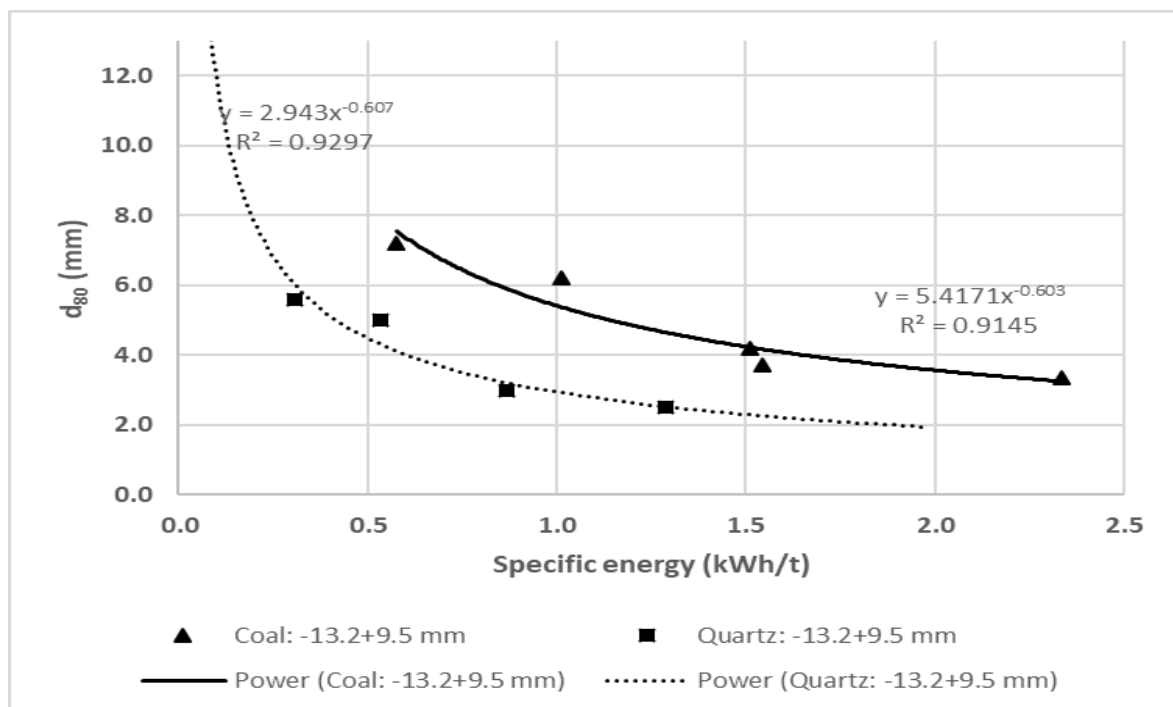


Figure 6. 3: Relationship between the d_{80} sizes and input impact energy for the -13.2+9.5 mm of coal and quartz

6.2.2 Product fineness

The fineness of the products for the single impact breakage tests is usually measured in terms of breakage index, t_{10} , defined in Eq. (2.11). The t_{10} values were interpolated from the cumulative size distributions in Figure 6.1. To do interpolation, the t_{10} sizes (the 10th of the feed size ranges) were obtained by dividing the geometric mean of the feed size range by 10, as shown in Eq. (6.2).

$$t_{10,size} = \frac{\sqrt{x_{bottom} \times x_{top}}}{10} \quad (6.2)$$

Where x_{bottom} and x_{top} are bottom and top screen sizes respectively for the size range.

The t_{10} values as a function of input impact energy are plotted in Figure 6.4 for both coal and quartz. The expected trend is that increasing the impact energy results in a higher generation of fine products and hence large t_{10} values. Such a trend is observed only for the -13.2+9.5 mm particles for both coal and quartz. For the -19+13.2 mm, this relationship is contrary. These results are consistent with what was observed for the product size distributions in Figure 6.1. It is observed that the coal products have higher values of t_{10} . While that is true for all energy levels considered for the -19+13.2 mm size class, the same is not true for the -13.2+9.5 mm. For impact energy greater than 2.9 J, as Figure 6.4 shows, more fines were produced with the -13.2+9.5 mm of quartz as compared to the -13.2+9.5 mm of coal.

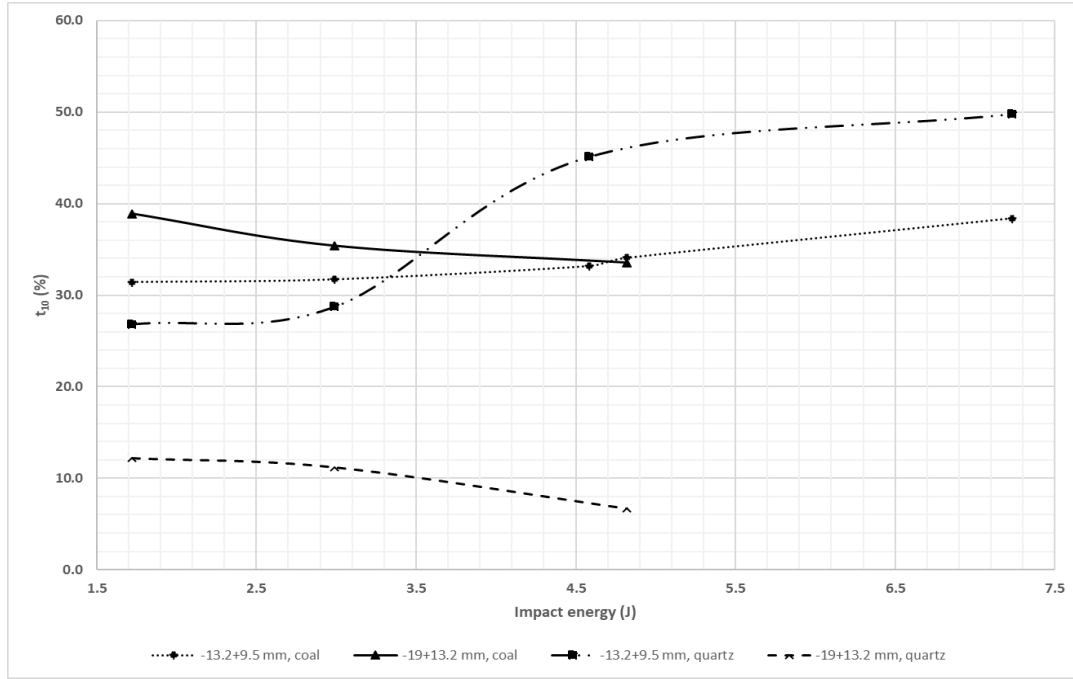


Figure 6. 4: The relationship between the product fineness (t_{10}) and impact energy

The experimental data in Figure 6.4 were fitted to Eq. (2.11) to estimate the model parameters (A and b). This was done using the iteration method with the set objective of minimising the sum of square errors. The data are summarised in Table P13 in Appendix P. The product of A and b is useful in comparing the competency of various ores, i.e. the ability to resist breakage (Shi and Kojovic, 2007). The products of A and b are listed in Table 6.1. The comments on the relative hardness of the two size classes for both coal and quartz are summarised in Table 6.1. The -19+13.2 mm are relatively weaker in strength than -13.2+9.5 mm, for both coal and quartz. As pointed out by Tavares and King (1998), material strength decreases with the coarseness of particles, i.e. coarser particles tend to be less resistant to breakage. This is because the crack densities of coarser particles tend to be greater than for small particles. The products of A and b in Table 6.1 is a good summary of the results shown in Figures 6.1 to 6.4.

Table 6. 1: The product of A and b parameters for the t_{10} model

Material	Size (mm)	A×b	Comments based on A×b
Coal	-13.2+9.5	127	Harder than -19+13.2 mm coal; softer than -13.2+9.5 mm quartz; and relatively of same hardness as -19+13.2 mm coal
	-19+13.2	639	Softer than -13.2+9.5 mm coal and quartz
Quartz	-13.2+9.5	97	Harder than -19+13.2 mm quartz and coal
	-19+13.2	131	Softer than -13.2+9.5 mm quartz; harder than -19+13.2 mm coal; and relatively of same hardness as -13.2+9.5 mm coal

6.1.3 Estimation of the breakage function parameters

The cumulative breakage functions given by Eq. (2.24) were computed from the breakage functions (b_{ij}) of the two mono-size ranges (-13.2+9.5 mm and -19+13.2 mm) for both coal and quartz with the aim of estimating the breakage function parameters in Eq. (2.25). The cumulative breakage functions, which take the same shape as the cumulative mass passing shown in Figures 6.1 and 6.2, are shown in Appendix P as Figures P1 and P2 for coal and quartz respectively.

To calculate the breakage distribution parameters (ϕ , γ and β), the cumulative breakage functions were fitted to Eq. (2.25) using the iteration method. For fitting parameters, the objective function was set to measure the minimum root mean square error (*RMSE*). The *RMSE* is given by Eq. (6.3) and it generally indicates the agreement between experimental and model data (Vining and Kowalski, 2006).

$$RMSE = \sqrt{\frac{1}{N} (\sum_{i=1}^N (y_i - y'_i)^2)} \quad (6.3)$$

Where N is the length of the input vector, i.e. the number of size classes in the case of cumulative breakage functions modelling, y_i is the experimental value for class size i and y'_i is the model value for size class i .

Since breakage parameter β is generally accepted as a material's characteristic (Austin et al., 1984), it was kept constant for both coal and quartz and thereby allowing investigation of the sensitivities of the other two parameters (ϕ and γ) to the change in the impact energy. The β value of 3.35 as suggested by Kwon et al. (2004) for low rank coal was adopted. Mulenga (2009) reports a β value of 3.2 for the South African coal which is close enough to the adopted value. For quartz, a β value of 5.8, as suggested by Petrakis et al. (2017) to be the best estimate in the literature, was adopted.

The best solutions (of the breakage parameters ϕ and γ) giving minimum *RMSE* for coal and quartz are listed in Table 6.2 and 6.3 respectively. Relationships between breakage parameters and impact energy are shown in Figures 6.5 and 6.6 for ϕ and γ respectively. The cumulative breakage functions were re-calculated using the estimated parameters and then the experimental versus predicted cumulative breakage functions were plotted to establish the coefficients of correlation. Those plots are listed as Figures P3 to P6 in Appendix P. The coefficients of correlation (R^2) are above 95 % in all cases, which indicates the good reliability of the estimated breakage parameters.

Table 6. 2: Breakage distribution parameters for coal sample

Size range (mm)	-13.2+9.5				-19+13.2		
Impact Energy (J)	1.7221	2.9888	4.8202	7.2304	1.7221	2.9888	4.8202
Specific energy (kWh/t)	0.58	1.01	1.51	2.33	0.23	0.4	0.65
Breakage parameters							
ϕ	0.98	1.02	1.42	1.42	0.92	0.95	1.47
γ	0.48	0.52	0.66	0.57	0.38	0.43	0.67
β	3.35	3.35	3.35	3.35	3.35	3.35	3.35

Table 6. 3: Breakage distribution parameters for quartz sample

Size range (mm)	-13.2+9.5				-19+13.2		
Impact Energy (J)	1.7221	2.9888	4.8202	7.2304	1.7221	2.9888	4.8202
Specific energy (kWh/t)	0.31	0.53	0.82	1.29	0.12	0.2	0.32
Breakage parameters							
ϕ	1.26	1.28	1.17	1.43	0.91	1.03	1.25
γ	0.70	0.69	0.62	0.55	0.68	0.74	0.63
β	5.8	5.8	5.8	5.8	5.8	5.8	5.8

From the results obtained (in Tables 6.2 and 6.3 for coal and quartz respectively), it is observed that there is no significant change between breakage parameters for the two size classes at different energy levels for both coal and quartz. This reaffirms the fact that the breakage distribution function, and hence the parameters for estimating it, is a material property. It can be observed from Figure 6.5 that ϕ increases with the increase in impact energy. This is expected noting that this breakage parameter (ϕ) represents the fraction of fines produced in a single fracture event (Austin et al., 1984). While the ϕ values for coal are comparatively the same for both size classes (see Figure 6.5), this was not the case with quartz. For quartz, the finer size class (-13.2+19 mm) has higher ϕ values. This suggests that -13.2+9.5 mm of quartz subjected to same impact energy as the -19+13.2 mm of quartz particles, has a higher proportion of finer progeny particles, which agrees with what is shown by the t_{10} plot versus energy input (see Figure 6.4).

From Figure 6.6, it can be seen that γ values for coal are smaller than those for quartz. Noting that γ relates to the finer size classes of the product (see Figure 2.12), this suggests that proportionally more fines are being produced from the quartz impact breakage events. While this is supported by the plots of t_{10} for the -13.2+9.5 mm in Figure 6.4, the same is not true for the -19+13.2 mm size classes, i.e. in Figure 6.4, the -19+13.2 mm of coal shows higher t_{10} values compared to quartz. Such an anomaly can best be explained with comments listed in

Table 6.1. The average breakage function parameters (ϕ , and γ) for coal are 1.09 and 0.53 respectively. For quartz, ϕ and γ take the averages of 1.19 and 0.65 respectively. Mulenga (2009) reported the γ value of 0.53 from ball milling experiments with coal which is equal to the average value obtained from the drop weight tests (0.53).

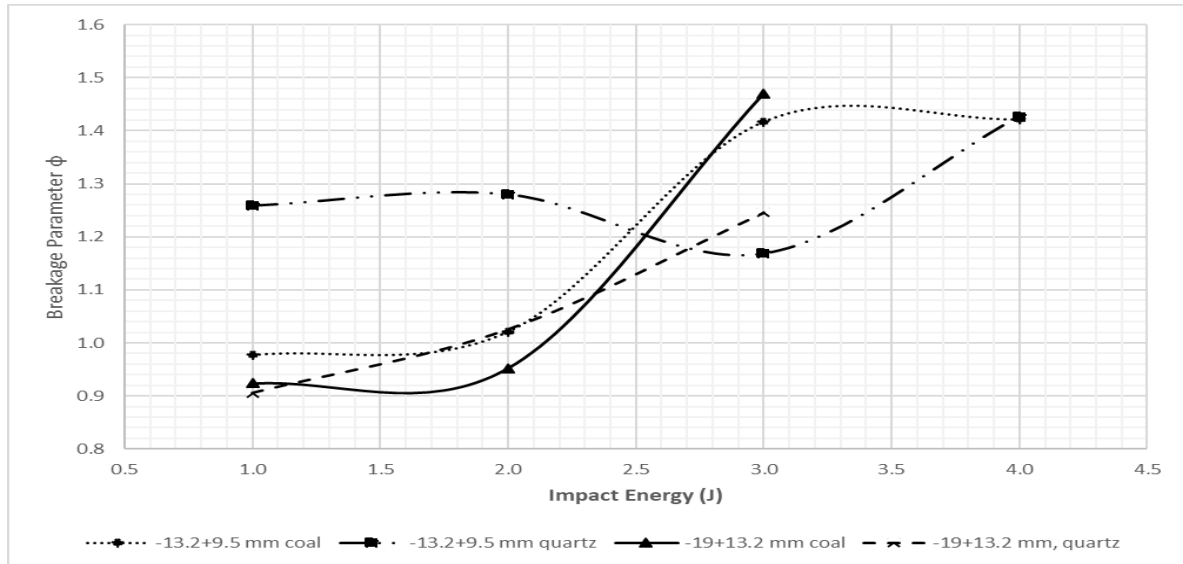


Figure 6. 5: Relationship between the breakage parameter ϕ and the input impact energy for single particle breakage tests

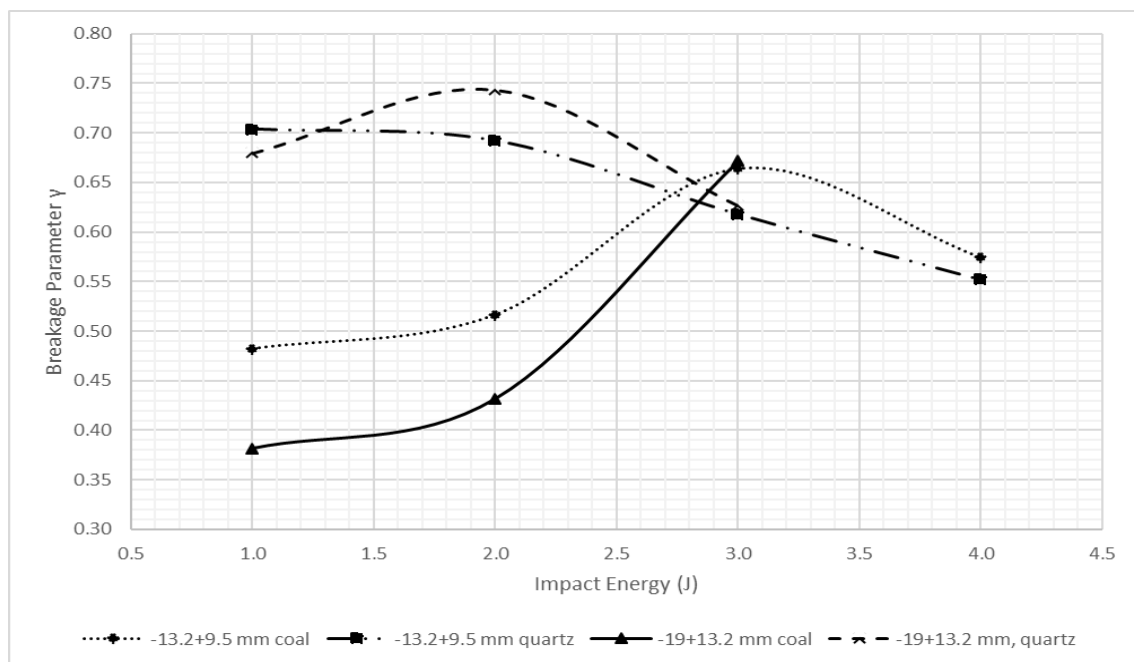


Figure 6. 6: Relationship between the breakage parameter γ and the input impact energy for single particle breakage tests

6.2 Compressed bed breakage

The size distributions for the four tests described in Chapter 3, i.e. for the two size classes (-13.2+9.5 mm and -19+13.2 mm) for both coal and quartz conducted with a maximum load of 50 kN, are shown in Figure 6.7. As expected, the products for the finer size class (-13.2+9.5 mm) have more coarse particles than -19+13.2 mm size class as it can be observed from the peaks of the product density functions. The d_{80} values (listed in Table 6.4) were extrapolated from the plots for cumulative passing in Figure Q1 in Appendix Q.

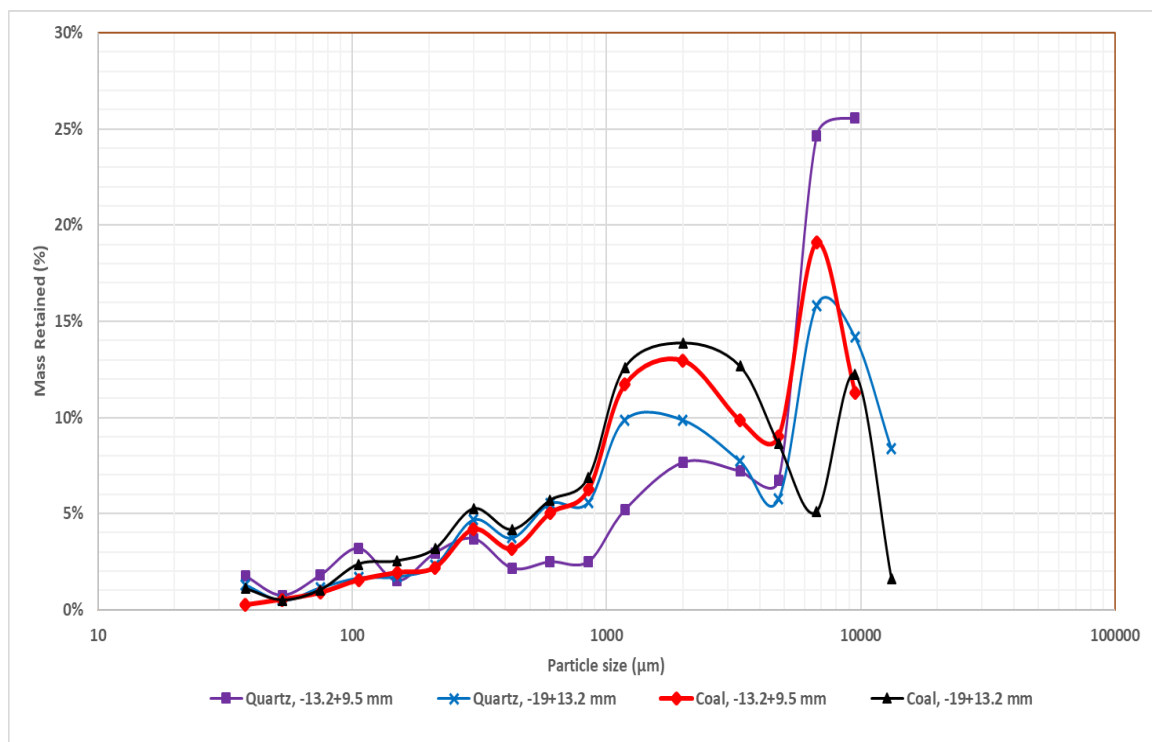


Figure 6. 7: Size distributions for the products of compression breakage tests

The experimental data for the forces and displacements were plotted to get the force-displacement curves with an example shown in Figure 6.8 for the first test while the plots for the other three tests are listed in Figures Q2 to Q4 in Appendix Q. As shown in Figure 6.8, the data were fitted to the polynomial of the order 3, with the coefficient of correlation greater than 0.99, implying that the model equation accounts for over 99 % variations in the experimental

data. Energies absorbed by the particles were calculated by finding the integral of the model equation between the initial displacement (0 mm) and maximum displacement from the curve. For the curve in Figure 6.8, the integration is shown in Eq. (6.4). This gives the total energy of 144 J. The specific comminution energies were evaluated using Eq. (6.5). The work done and specific comminution energies for the other three tests were calculated using the same method and the summary is given in Table 6.4.

$$E = \int_0^{0.0121} (7 \times 10^{10}x^3 - 9 \times 10^8x^2 + 5 \times 10^6x - 1707.4) dx \quad (6.4)$$

$$. E_{cs} = \frac{E}{3.6 \cdot M_s} \quad (6.5)$$

Where E is the work done on the bed of particles, i.e. the area under the force-displacement curve, M_s is the mass of sample in a test in grams and 3.6 is conversion factor from J/g to kWh/t.

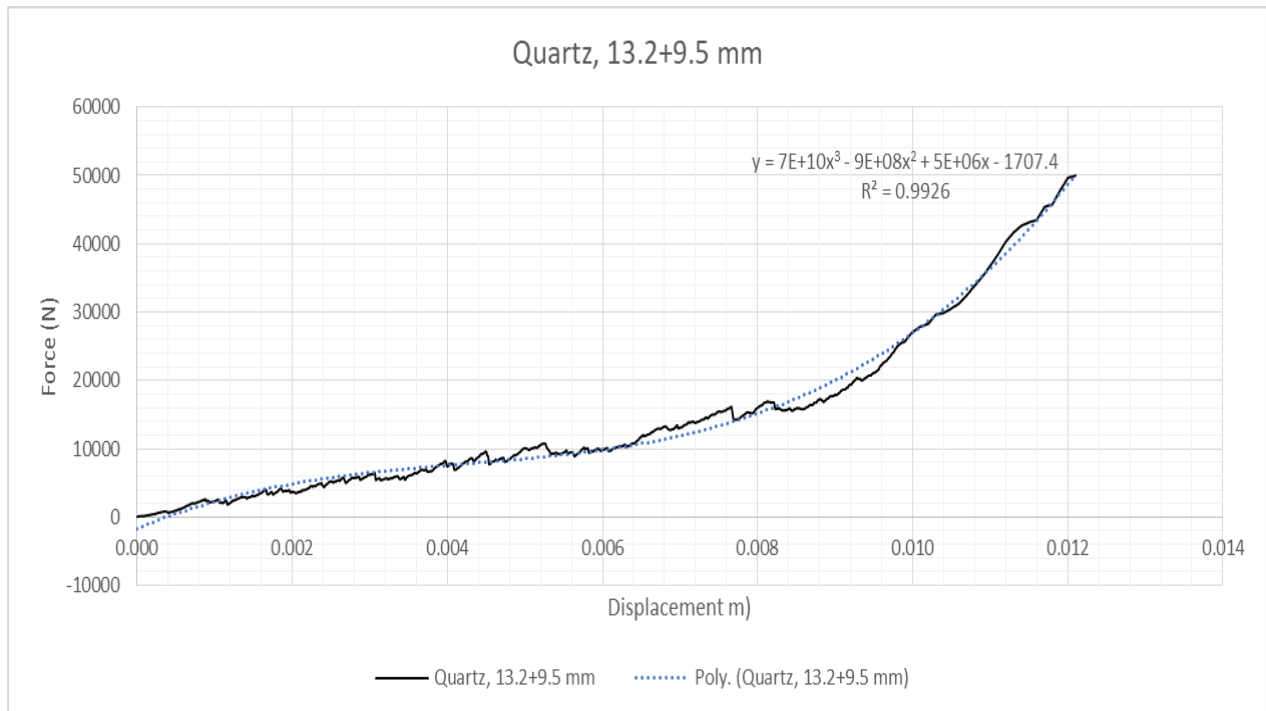


Figure 6. 8: Force-displacement curve and trend line of polynomial degree 6

Table 6. 4: Summary of results for compression tests

Test #	Material	Size (mm)	Mass (g)	d ₈₀ (mm)	R ₈₀	d ₅₀ (mm)	R ₅₀	E (J)	kWh/t
1	Quartz	-13.2+9.5	84.37	10	1.8	6.75	2.4	144.1	0.474
2		-19+13.2	107.96	10	1.8	3.6	4.5	205.6	0.529
3	Coal	-13.2+9.5	59.31	8.3	2.1	3.2	5.0	156.5	0.733
4		-19+13.2	66.33	6.3	2.8	2.4	6.7	166.8	0.698

As for the single particle drop weight tests, the breakage functions parameters were computed. Same iteration method as discussed in section 6.2.3 was used. The estimated parameters for both coal and quartz are shown in Table 6.5. The ϕ values, for both coal and quartz, are smaller for the -13.2+9.5 mm size class, which suggests that proportionally more progeny particles to size classes below the feed size class were generated from the -19+13.2 mm particles. This agrees with the theory that coarser particles are weak in strength and hence can easily break (Tavares and King, 1998). The ϕ values for coal are relatively larger than for quartz, which suggests that the probability for the generation of progeny particles is proportionally higher for coal than quartz subjected to pressure breakage mechanism. This contrasts with comparatively same ϕ values for both coal and quartz subjected to single impact breakage tests (see Tables 6.2 and 6.3); highlighting that the mode of breakage has pronounced effect on the generation of progeny particles. It is observed that for quartz, γ for -13.2+9.5 mm is relatively smaller than that of -19+13.2 mm. The opposite is true for coal. But generally, it can be said that the γ values for both coal and quartz are equal. This can even be observed in Figure 6.7 for the size classes less than 400 μm , the graphs for both coal and quartz are overlapping. The results for the compression tests were correlated to those of ROC as discussed in Chapter 7.

Table 6. 5: Breakage parameters estimated from the compression tests

Material	Quartz		Coal	
Size range (mm)	-13.2+9.5	-19+13.2	-13.2+9.5	-19+13.2
Breakage parameters				
ϕ	0.792	1.049	1.112	1.278
γ	0.477	0.559	0.683	0.585
β	5.8	5.8	3.35	3.35

6.3 Summary

It has been shown that the -19+13.2 mm are weaker in strength (less resistant to breakage) than the -13.2+9.5 mm for both coal and quartz. This is what is expected because crack density increases with increase in the particle size. The breakage functions parameters for the breakage tests and the relationships between the d_{80} and energy input (kWh/t) have been established and these would be correlated to the results of the ROC in Chapter 7.

CHAPTER SEVEN: CRUSHER PERFORMANCE EVALUATION AND MODELLING

This chapter presents the results derived from the experiments discussed in Chapter 3 that were conducted using the laboratory ROC. Initially, coal was used to investigate the effects of feed size, offset and exit gap on the crusher performance and the results are discussed in section 7.1. Lastly, the effects of feed rate and rotational speed on the size reduction, throughput and power draw were investigated by conducting experiments with silica (quartz) and the results are discussed in section 7.2.

7.1 Coal comminution

The coal was crushed in the ROC at various crusher settings (exit gap of 1.5 and 3 mm and the offset of 5 and 10 mm). The mono-size samples, -19+13.2 mm and -13.2+9.5 mm (about 1.5 kg for each run), were fed to the crusher using the conveyor belt. The feed rates and speed remained fixed. The results are discussed in the following subsections.

7.1.1 Effect of disc offset and exit gap on size reduction

The fragments size distributions are plotted in Figure 7.1 and 7.2 for the tests conducted with the exit gap of 1.5 and 3 mm respectively. It is observed, from Figure 7.1, that with the exit gap of 3 mm, for both feed size classes, the offset of 5 mm produced finer products. This suggests that increasing the offset to 10 mm when the exit gap is 3 mm is not beneficial for size reduction. Considering the minus 5 mm particle sizes (in Figure 7.1), it can be argued that the feed size has no effect on the production of progeny particles less than 5 mm, as it can be seen that the plots for each offset are overlapping.

For the exit gap of 1.5 mm (Figure 7.2), the offset of 10 mm produced finer products for both size classes. This suggests that the offset is directly proportional to the reduction ratios which contradicts what is observed for the exit gap of 3 mm. These results are an indication of the dynamics involved in the operation of the ROC. It is recommended that more experiments be conducted to establish sustained trends for the effects of offset and exit gap on size reduction. Nevertheless, regression modelling of size reduction ratios and throughput (discussed in section 7.1.3) was conducted for qualitative analysis of the results obtained.

Crushing coal in the ROC was not promising considering the d_{80} sizes for the crusher products in Figures 7.1 and 7.2. With the exit gap of 1.5 mm, the d_{80} are all greater than 9 mm, implying the size reduction ratios of about 2 for the -19+13.2 mm feed size and size reduction ratios of about 1.5 for the -13.2+9.5 mm feed size. The relevant calculations for the size reduction ratios are discussed in Appendix R and the calculated R_{80} as well as the d_{80} are summarised in Table R1 in Appendix R.

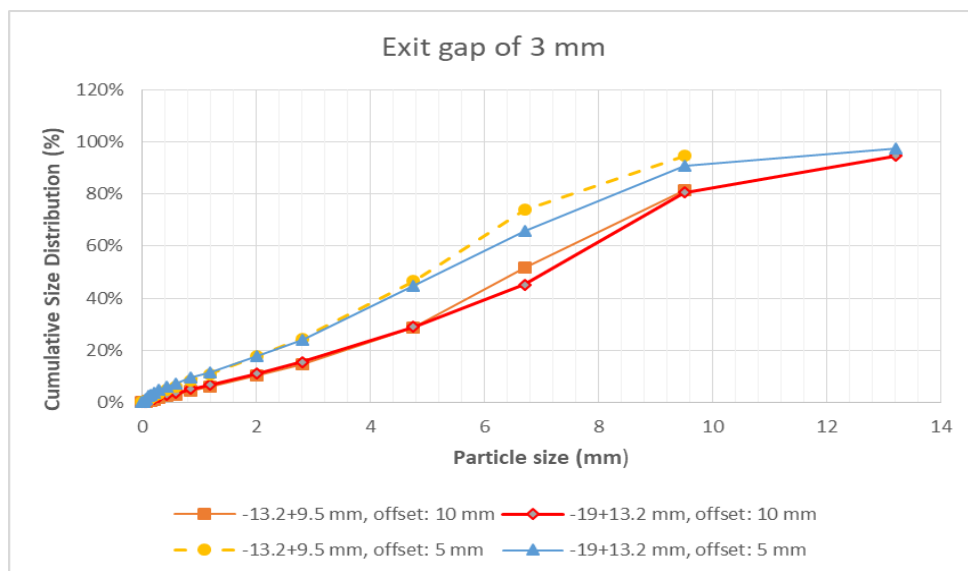


Figure 7. 1: Cumulative Mass passing Size distribution of the crusher products for the mono-sized coal samples crushed at various offset for the exit gap of 3 mm, rotational speed of 330 rpm and with feed rate of 5.4 t/h

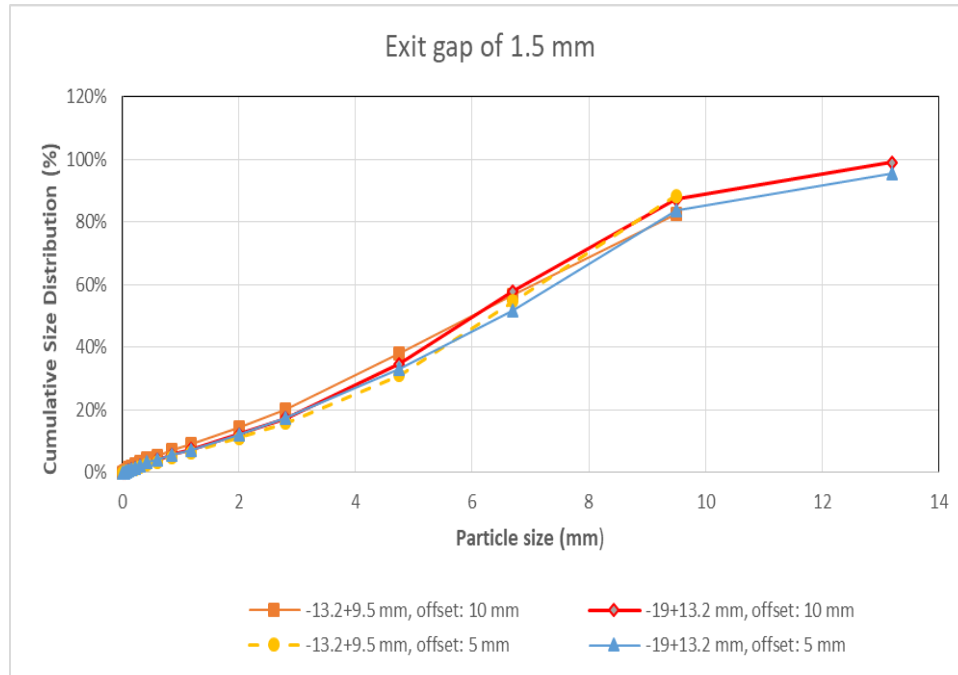


Figure 7. 2: Cumulative Mass passing Size distribution of the crusher products for the mono-sized coal samples crushed at various offset for the exit gap of 1.5 mm, rotational speed of 330 rpm and with feed rate of 5.4 t/h

7.1.2 Characterisation of coarser coal particles from the crusher

As a way of characterising the coal particles discharged from rotary offset crusher, the thicknesses and diameters for the coarser particles (those retained on screens above 3.35 mm (inclusive)) were measured using a vernier calliper and plotted relative to the exit gap as shown in Figure 7.3. Figure 7.4 is a photograph of the showing the top and side views of the particles discharged from the crusher (whose dimensions are plotted n Figure 7.3). These results are for a test conducted with the offset of 10 mm, exit gap of 3 mm and feed size range of -19+13.2 mm. The thickness of the coarse crusher products ranges between 2.7 and 3.7 mm, suggesting that some particles discharged from the crusher have thicknesses which are about 23 % larger than the exit gap. But as shown in Figure 5.5 in Chapter 5, there is no gap variation when the offset is less than or equal to 10 mm to account for that 0.7 mm. Such a larger variation in the gap can only be attributed to the flexibility in the disc vertical movements. There is a need to

strengthen the support structures for the discs to ensure that no flexure takes place. The rigidity of the structure would ensure the full transmission of energy to the particles, and in turn, achieve higher size reduction ratios.

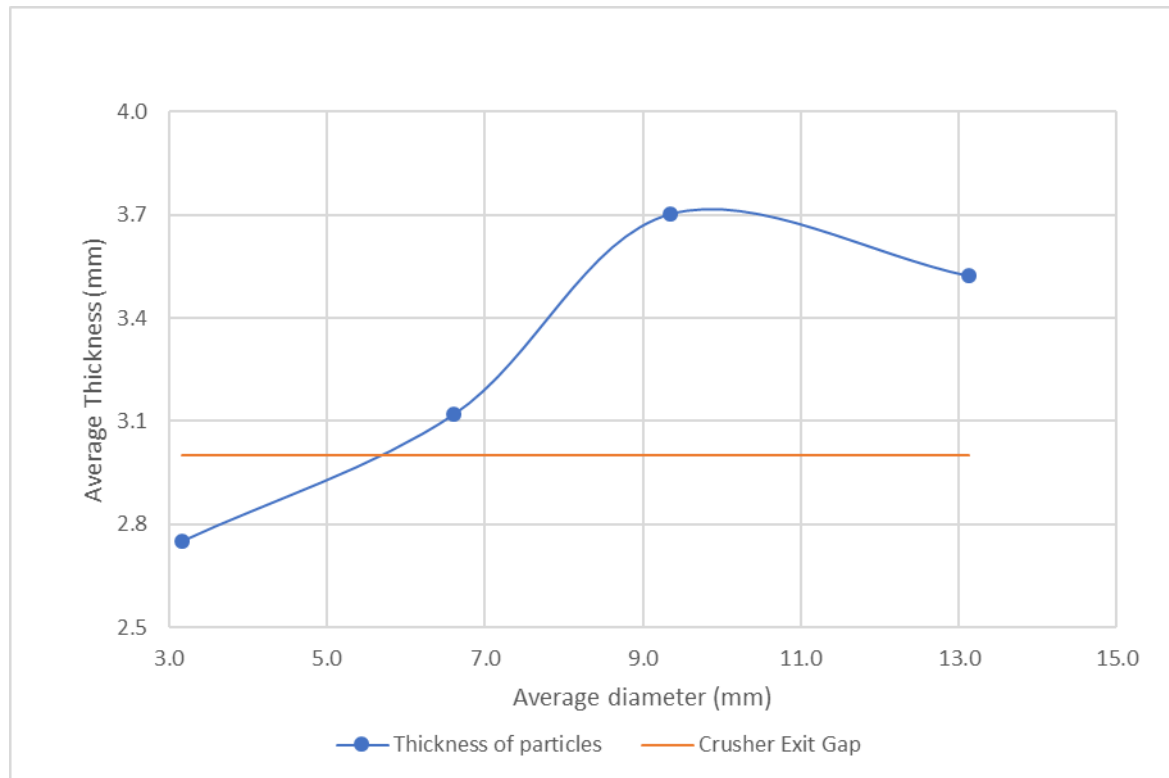


Figure 7. 3: Dimensions of coarse crusher products relative to the crusher exit gap of 3 mm



Figure 7. 4: Acicular coal particles from the crusher

Figure 7.3 shows that the diameters of the crusher products are as big as 13 mm. This suggests that acicular (flat) particles are discharged (see Figure 7.4), i.e. particles get discharged for as long as they are equal to the exit gap which implies that crushing is one dimensional. Production of acicular particles can be attributed to the mineralogical and deformation

characteristics of coal. Coal, particularly the low grade, is classified as a sedimentary rock (Akinyemi et al., 2012). This means that coal is deposited in layers, probably coal interspersed with shale. When comminuted such multiple composition particles can separate exposing the sheet-like structure of the shale (probably coarser acicular particles, but this claim needs to be confirmed by analytic techniques). With regard to deformation characteristics, Xu et al. (2017) pointed out that the deformation and failure process of coal under compression happens in three stages, namely: compaction, elastic deformation and plastic deformation. During the compaction stage, the internal defects and voids are being closed. In the elastic deformation stage (can be called the pre-weakening stage), the applied load brings about changes in the axial and radial strains of the particles, but no initiation of micro cracks. In the plastic deformation stage, the initiation of micro cracks starts and increase resulting in actual breakage. The implication of these stages is that coal under compression requires the prolonged application of stress to fracture. It can be concluded that coal is thus probably unsuitable for ROC crushing without special disc profiles. Corrugated profiles on the crushing surfaces are said to effect compound crushing, i.e. by compression, tension and shear, and in turn, increase the reduction ratio (Wills and Finch, 2016).

As discussed in section 7.2, with quartz (a brittle material), no such acicular particles were discharged from the crusher. The effects of operating parameters such as rotational speed and feed rate, which were not considered during the coal experiments, have significant effects on the size reduction as discussed in section 7.2.

7.1.3 Regression modelling

Regression modelling of size reduction ratios and throughput as functions of the independent variables (feed size, offset and crusher exit gap) was conducted and the models and their statistical implications are discussed in subsections 7.1.3.1 and 7.1.3.2 respectively.

7.1.3.1 Size reduction

To understand if there are linear relationships between operating variables (feed size, offset and gap) and responses (d_{80} and R_{80}), the coefficients of correlations were computed, and they are listed in Table 7.1. As expected, there is a strong positive correlation between the feed particle size and reduction ratio. The next strong relationship exists between the offset and d_{80} . While it was shown that with DEM simulations (discussed in Chapter 5), there is no strong relationship between offset and throughput, there exists a definite relationship between the d_{80} size and offset. But as pointed out already, more experiments are needed to establish sustained trends.

Table 7. 1: Coefficients of correlations between factors and responses

	<i>Mean feed size (mm)</i>	<i>Offset (mm)</i>	<i>Gap (mm)</i>	<i>d₈₀ (mm)</i>	<i>R₈₀</i>
Mean feed size (mm)	1				
Offset (mm)	0	1			
Gap (mm)	0	0	1		
d₈₀ (mm)	0.144	0.637	-0.226	1	
R₈₀	0.922	-0.242	0.102	-0.248	1

Multiple linear regression modelling was conducted in Microsoft Excel[®] using the experimental data in Table R1 in Appendix R. The model equations for d_{80} and R_{80} are given below.

$$d_{80} = 7.6025 + 0.0377d_M + 0.155x_o - 0.183G_e \quad (7.1)$$

$$R_{80} = 0.586 + 0.112d_M - 0.0343x_o + 0.114G_e \quad (7.2)$$

Where d_M is the geometric mean size of the feed in mm, x_o is the offset in the x direction and G_e is the exit gap.

To assess the robustness of the obtained equations, the experimental and predicted data were plotted in Figures 7.5 and 7.6 for the d_{80} and R_{80} respectively. The data used for plotting these two graphs are listed in Table R2 in Appendix R. While the R^2 value of over 90 % was obtained (showing the high predictive power of the model) for Eq. (7.2)), the d_{80} equation has a predictive power of less than 50 %. The scatter in the data points indicates that d_{80} is not highly dependent on the factors (offset, feed size, and gap) considered during coal comminution experiments. As it would be seen in section 7.2, the d_{80} is affected mostly by the speed and feed rate.

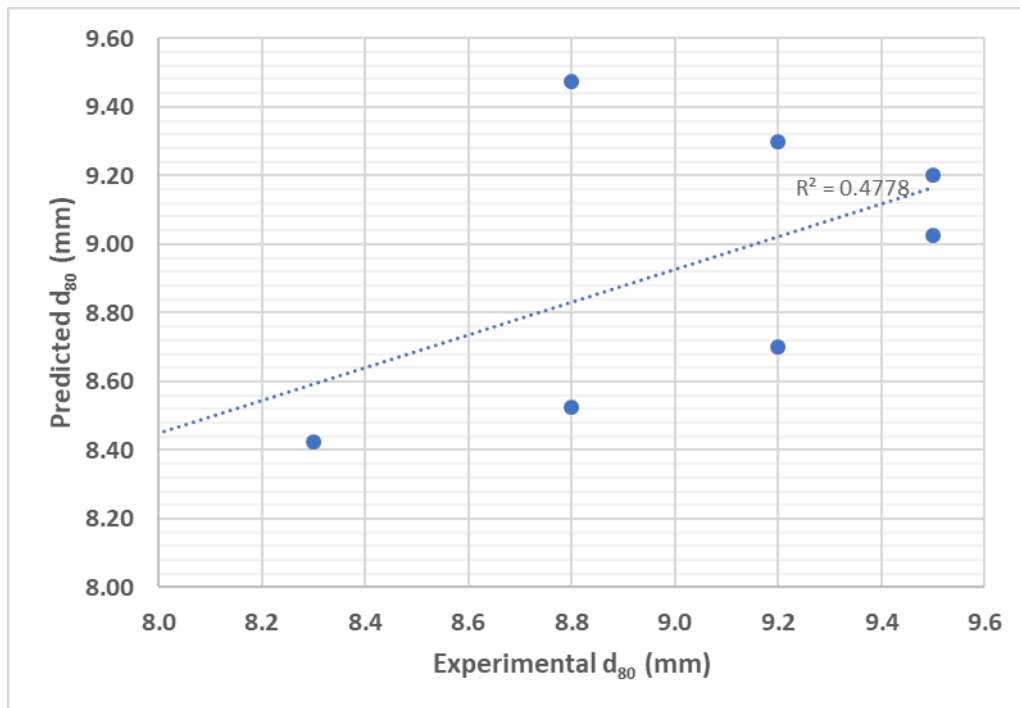


Figure 7. 5: Experimental versus predicted d_{80} sizes

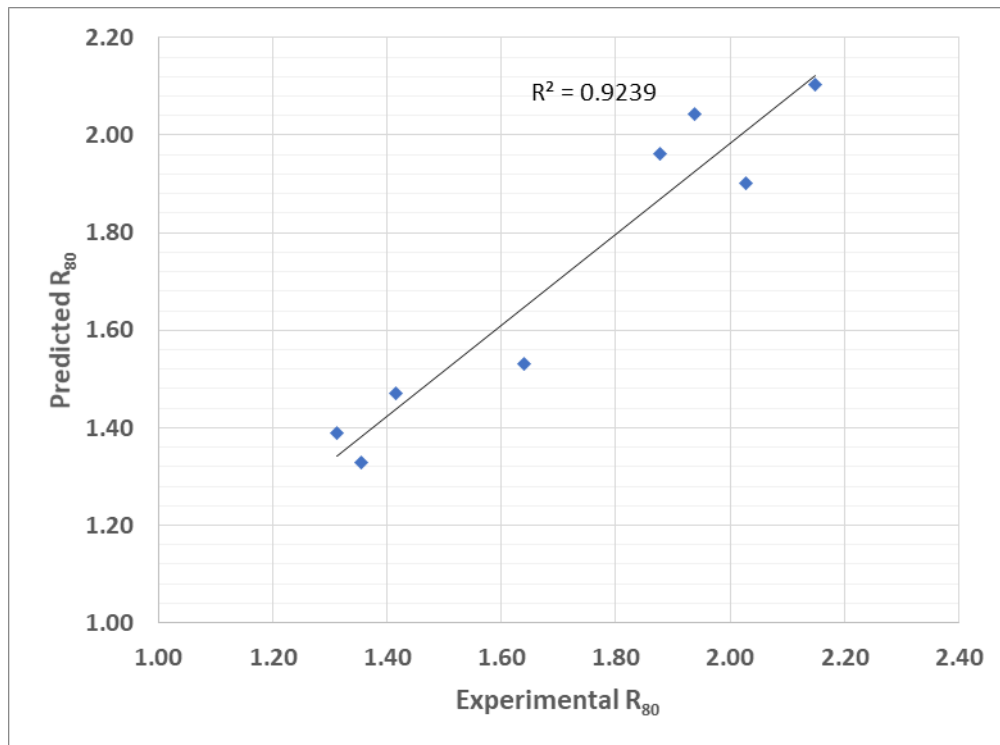


Figure 7. 6: Experimental versus predicted R_{80}

7.1.3.2 Crusher throughput

The throughput is computed from the mass per batch test and the residence time. The residence time is taken to be the time when the particles used in the batch test are in the crushing zone, i.e. time difference between when the top disc starts spinning and when its speed starts to decrease. One drawback in measuring the residence time was that sometimes few particles (say 2 % of the feed) get stuck between the discs and with particle locking, the top disc keeps on spinning and thereby misleading the measurement of some of the residence times (as for the values shown in red in Table R3 in Appendix R). It is important to note that such an operational issue of few particles stuck between the discs was only associated with coal experiments, i.e. no quartz particles were stuck between the discs. Nevertheless, the throughputs were computed, and they are summarised in Table R3 in Appendix R.

The multiple linear regression model describing the relationship between throughput (tph) and 3 independent variables (feed size, offset and exit gap) was attempted and the equation of the fitted model is shown below.

$$Q = 0.044 - 0.00163 \times d_M - 0.0005 \times x_o + 0.0173 \times G_e \quad (7.3)$$

Where Q is throughput in tph.

The predicted and experimental throughputs were plotted in Figure 7.7 to assess the robustness of Eq. (7.3). The R^2 statistic indicates that the model as fitted explains 40 % of the variability in Q (tph). Since the P-value in Table 7.2 is greater or equal to 0.05, there is no statistical significance in the relationship between the variables at the 95 % or higher confidence level. However, the model was assessed qualitatively, by looking at the P-values of each independent variables (shown in Table 7.3). The highest P-value belongs to the offset. As concluded already from the DEM results presented in sections 5.4.3 and 5.5 in Chapter 5, the offset values of less than or equal to 10 have no influence on the crusher throughput. The throughput is greatly affected by the rotational speed (see Figure 5.19) and the exit gap (with lower P-value in Table 7.3). As it would be seen in section 7.2 (quartz experiments), the crusher throughput is highly dependent on the rotational speed and feed rate. Incorporating these two variables (speed and feed rates) in the model equation resulted in improved accuracy as discussed in section 7.2.2.

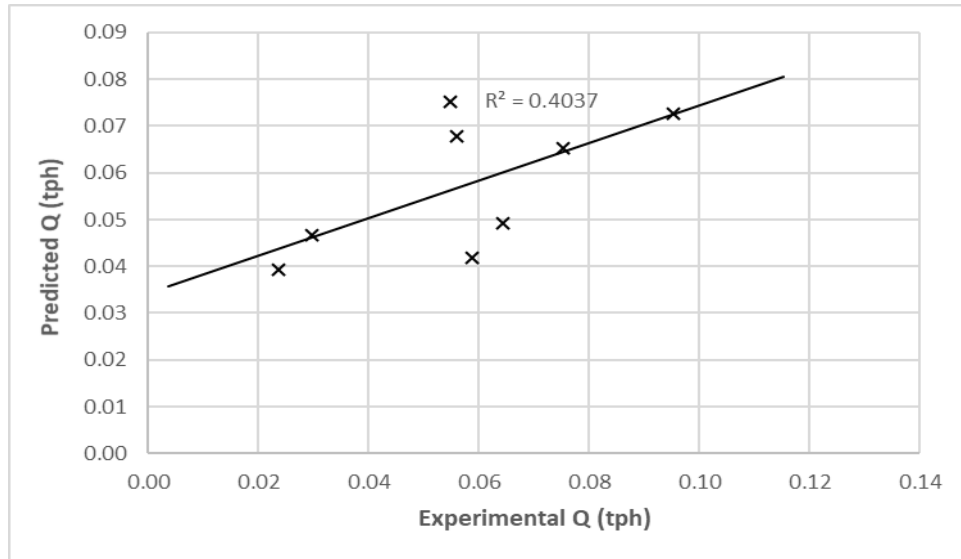


Figure 7. 7: Experimental versus predicted throughputs for bituminous coal crushing tests using the ROC

Table 7. 2: Analysis of variance for the multiple regression modelling of throughput

Source	Sum of Squares	Df	Mean Square	F-Ratio	P-Value
Model	0.001477	3	0.000492	0.91	0.512
Residual	0.00217	4	0.000542		
Total (Corr.)	0.00364	7			

Table 7. 3: Statistics data for the regression modelling of throughput

Parameter	Estimate	Standard Error	P-Value
CONSTANT	0.044	0.06016	0.505
Mean size (mm)	-0.00163	0.00358	0.672
Offset (mm)	-0.0005	0.00329	0.887
Exit Gap (mm)	0.0173	0.011	0.189

7.1.4 Estimation of breakage distribution parameters

As it was done for the breakage tests in Chapter 6, the cumulative breakage functions were also computed from the breakage functions (b_{ij}) using the size distributions in Figures 7.1 and 7.2 with the aim of estimating the breakage function parameters. The cumulative breakage distributions are shown in Figures R1 and R2 in Appendix R for the 1.5 and 3 mm exit gaps respectively. As discussed for drop weight and compression tests in Chapter 6, the experimental

data in Figures R1 and R2 in Appendix R were fitted to Eq. (2.25) to get the breakage parameters using the iteration method with the set objective function that minimise the *RMSE* between the experimental and model data. This was done using Microsoft Excel® *Solver* add-in tool. The estimated breakage parameters (in Table 7.4) were used to predict the cumulative breakage functions which were compared to the experimental data. Plots for experimental versus predicted cumulative breakage functions are shown in Figures R3 and R4 in Appendix R for the exit gap of 1.5 and 3 mm respectively. For all test conditions, the R^2 statistics are above 0.98, showing a good agreement between the experimental and predicted B_{ij} values. This confirms that cumulative breakage function established from mono-sized particles crushed in the ROC can be assumed to be correct and can be applied to model different operating conditions.

Table 7. 4: Coal breakage distribution parameters obtained using the ROC

Size range (mm)	-13.2+9.5	-19+13.2	-13.2+9.5	-19+13.2	-13.2+9.5	-19+13.2	-13.2+9.5	-19+13.2
Offset (mm)	10	10	5	5	5	5	10	10
Exit gap (mm)	3	3	3	3	1.5	1.5	1.5	1.5
ϕ	0.88	1.58	1.08	1.58	0.82	1.73	0.92	1.86
γ	1.25	1.12	1.11	1.12	1.21	1.44	1.02	1.48
β	3.35	3.35	3.35	3.35	3.35	3.35	3.35	3.35

The breakage parameters were plotted against offset in Figures 7.8 and 7.9 for -13.2+9.5 mm and -19+13.2 mm respectively. For the -13.2+9.5 mm (in Figure 7.8), γ increases with the increase in the offset at the exit gap of 3 mm, but at the smaller exit gap (1.5 mm), the opposite happens, i.e. γ decreases with the increase in the offset. Parameter ϕ , which is a fraction of generation of progeny particles smaller than the feed, decreases with the increase in the offset (from 5 to 10 mm) when the exit gap is 3 mm, which suggests that relatively a smaller proportion of fine particles were produced at a higher offset of 10 mm. This agrees with what is shown in Figure 7.1, i.e. fineness of product size is inversely proportional to the disc offset. With a reduced exit gap (1.5 mm), parameter ϕ increases with the increase in the disc offset.

Such a trend agrees with the size reduction ratio (and hence the d_{80}) in Table R1 in Appendix R.

For the -19+13.2 mm (Figure 7.9), the breakage parameters are relatively constant, which suggests that breaking the -19+13.2 mm was not affected by the change in exit gap and offset. This implies that the breakage parameters (for this size class for coal) are not dependent on the operating conditions; which agrees with what other studies have demonstrated for the existing comminution devices, i.e. breakage distribution parameters are material properties, (Fuerstenau et al., 2003, Petrakis et al., 2017).

Comparing the average breakage functions calculated from the ROC data with those of drop weight and compression tests as shown in Table 7.5, it is observed that ϕ is relatively similar, but γ from the ROC are double those from the impact and compression tests. Since γ is related to the finer size classes (see Figure 2.12), the difference can be attributed to the fact that in the ROC, abrasion is a dominant process in this crusher.

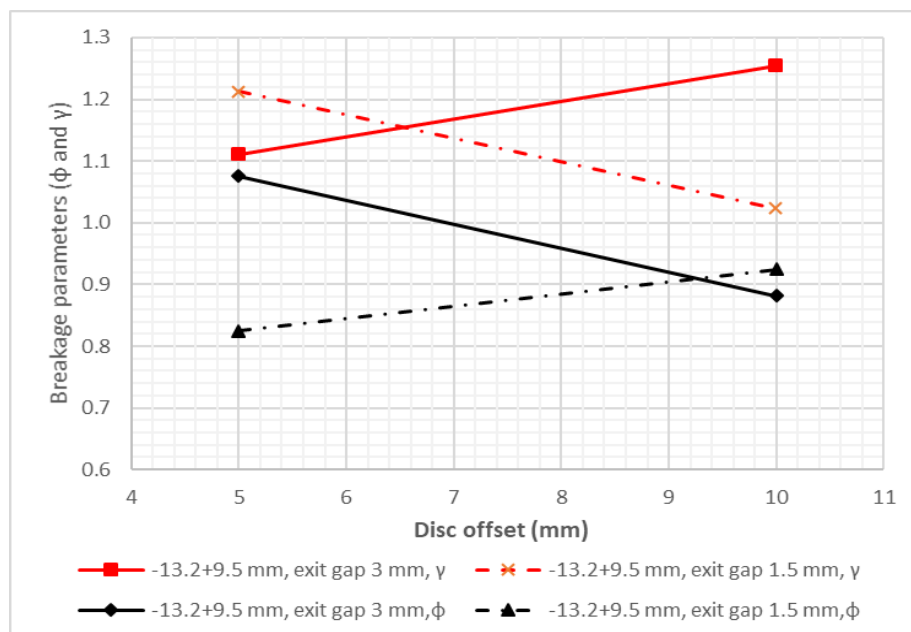


Figure 7. 8: Relationships among breakage function parameters and crusher settings with -13.2+9.5 mm coal particles

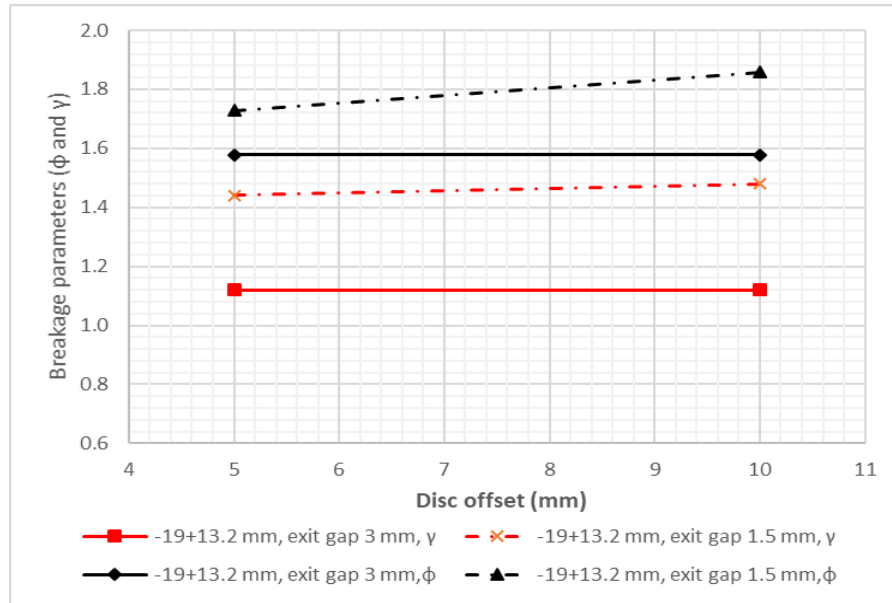


Figure 7. 9: Relationships among breakage function parameters and crusher settings with -19+13.2 mm coal particles

Table 7. 5: Average breakage function parameter of coal from ROC, DWT and Compression data

Equipment/Apparatus	ϕ	γ
DWT	1.09	0.53
Compression	1.19	0.63
ROC	1.30	1.22

7.2 Quartz comminution

While valuable information was derived from the ROC experiments with coal as discussed in section 7.1, low size reduction ratios (of about 2) were recorded. Some important operating variables that were not considered for coal experiments are feed rate and rotational speed. These two factors were studied using quartz and results are discussed in the following subsections.

7.2.1 Effect of feed rate and rotational speed

The disc offset and exit gap were kept constant at 10 mm and 3 mm respectively while two feed size classes (-13.2+9.5 mm and -19+13.2 mm) were considered. The feed rate was changed between 1 and 1.7 tph while two rotational speeds (330 and 550 rpm) were used.

The size distributions for the two feed sizes at various feed rates and speeds are shown in Figure 7.10. It is observed that finer products are discharged from the crusher operating at a speed of 550 rpm. This suggests that the higher the speed the more the crushing frequency (as discussed in section 5.2.1 in Chapter 5). It is also observed that at both speeds, low feed rate results in finer products. It is apparent that with fast transportation of the material to the crushing zone, there is a tendency of flexure in structure resulting in the change of the geometry of the crushing zone and thereby permitting oversize particles to slip through. As already pointed out, the structure needs to be strengthened to ensure the full transmission of energy to the particles nipped between the discs. While no consistent relationship could be derived between offset and product size distribution (as shown in Figures 7.1 and 7.2), the relationship between the two operating variables (feed rate and rotational speed) and size reduction is consistent for both feed sizes. To increase the database for modelling and optimization of the crusher, it is recommended that many experiments be conducted at the current speed and wide ranges of feed rate, offset and exit gap, and thereafter the higher speeds can also be attempted. It is expected that increasing the speed further would ensure not only higher throughputs, but higher size reduction ratios can also be expected.

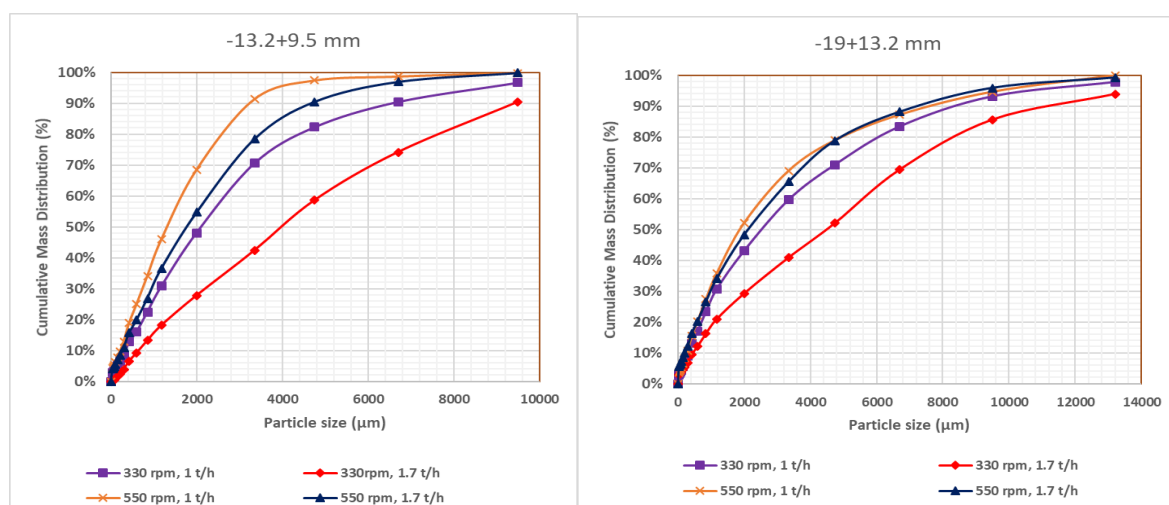


Figure 7. 10: Product size distributions for the -13.2+9.5 mm and -19+13.2 mm of quartz crushed in the ROC at speeds of 330 rpm and 550 rpm and feed rates of 1000 and 1700 kg/h

7.2.1.1 Reduction ratios

The d_{80} and d_{50} sizes were extrapolated from the size distributions in Figure 7.10 and plotted against the operating variables as shown in Figure 7.11. As already stated, increasing the speed from 330 rpm to 550 rpm resulted in a finer product. For the -19+13.2 mm at 550 rpm, the feed rate had no influence on the d_{80} size (horizontal yellow line). But for the d_{50} sizes (graph on the right) for the two speeds, increasing the feed rate is detrimental to the size reduction process in the crusher. As stated already, this can be attributed to the flexibility in the structure. The size reduction ratios computed from the d_{50} and d_{80} are listed in Table 7.6. Unlike for the coal, where the reduction ratios were around 2, with quartz, the size reduction ratios are as high as 6.9. This suggests that the comminution of particles in the ROC is highly dependent on the rotational speed and feed rate as opposed to the offset and exit gap. Regarding the influence of the speed on size reduction, this agrees with what was discussed in section 5.2.1 in Chapter 5. Regression modelling of the d_{80} and d_{50} sizes as functions of operating variables is discussed in section 7.2.2.

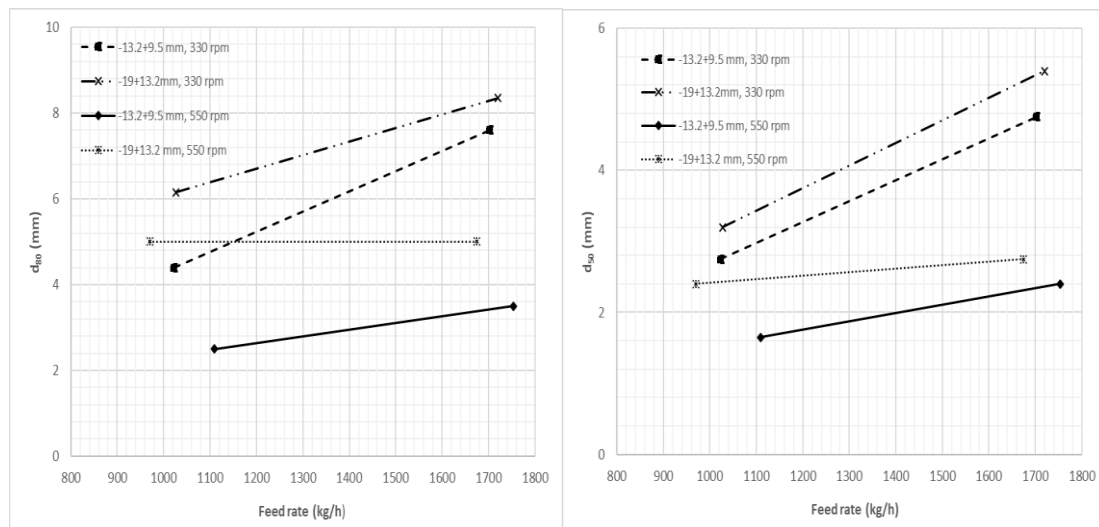


Figure 7. 11: Relationships between the d_{80} , d_{50} , speed and feed rate for the -19+13.2 mm and -13.2+9.5 mm of quartz

Table 7. 6: Summary of reduction ratios for ROC quartz experiments

Test #	Size range (mm)	Speed (rpm)	Feed rate (kg/h)	Throughput (kg/h)	d_{80} (mm)	R_{80}	d_{50} (mm)	R_{50}
1	-13.2+9.5	330	1023	194	4.4	2.8	2.8	4.1
2	-19+13.2	330	1027	199	6.2	2.9	3.2	5.0
3	-13.2+9.5	330	1703	725	7.6	1.6	4.8	2.4
4	-19+13.2	330	1720	652	8.4	2.1	5.4	3.0
5	-13.2+9.5	550	1753	1461	3.5	3.6	2.4	4.7
6	-19+13.2	550	1674	1217	5.0	3.6	2.8	5.9
7	-13.2+9.5	550	1109	693	2.5	5.0	1.7	6.9
8	-19+13.2	550	970	661	5.0	3.6	2.4	6.7

7.2.1.2 Throughput

Overall, the ROC capacity is a function of the following: the feed rate, material properties (size, density and hardness), feed hopper capacity, feed chute dimensions (throat diameter and height), geometry of the crushing zone (angle α and height h_c , disc offset as well as the vertical exit gap), rotational speed and discs profiles. Only the effects of particle sizes and rotational speed on throughput have been investigated thus far. The throughputs for the 8 tests are listed in Table 7.6 and plotted as a function of operating parameters in Figure 7.12. For the two feed sizes, their plots at both speeds and feed rates are overlapping, which suggests that the feed size has no influence on the throughput. As expected, fast feeding results in higher crusher throughput, but then, as pointed out already, this negatively affects the size reduction as the discs tend to move apart. The flexing is an indication of limited capacity of the crushing zone for the speeds attempted. When the structure is strengthened, the full potential of the crusher in terms of capacity will be established. Increasing the speed further would result in higher throughput as Figure 12 shows. As the DEM results in Figure 5.16 in Chapter 5 showed already, the rotational speed of the discs has a significant influence on the crusher throughput. Regression modelling of the throughput as a function of operating variables is discussed in section 7.2.2.

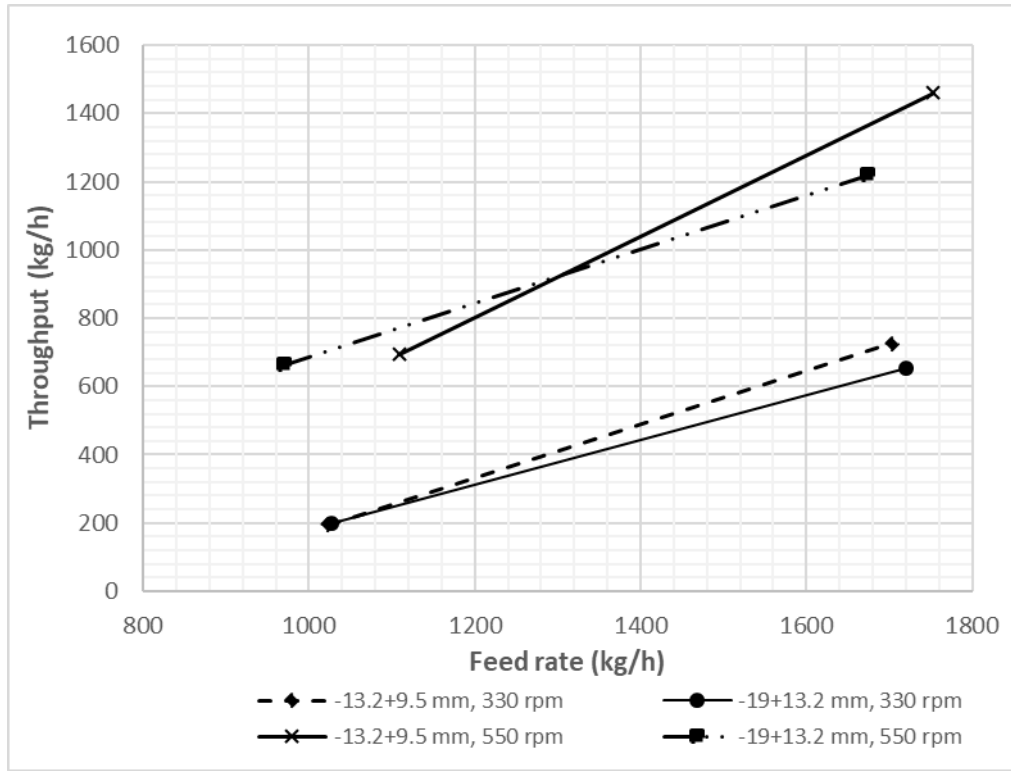


Figure 7. 12: Crusher throughput as a function of feed size, feed rate and rotational speed

7.2.2 Regression modelling of size reduction and throughput

As with coal data, the experimental data from the quartz experiments were used to develop regression models for the d_{80} , d_{50} and throughput (Q) that are shown in Eq. (7.4) to (7.6) respectively.

$$d_{80} = 62.3 - 0.09 \times n - 0.004 \times F + 0.215 \times d_M - 0.704 \times \tau \quad (7.4)$$

$$d_{50} = 32.6 - 0.05 \times n - 0.001 \times F + 0.06 \times d_M - 0.358 \times \tau \quad (7.5)$$

$$Q = -3660.5 - 5.50 \times n - 0.557 \times F + 36.9 \times \tau \quad (7.6)$$

Where n is the rotational speed (in rpm), F is the feed rate (in kg/h), d_M is the geometric mean of the feed size range (in mm) and τ is the residence time of the particles in the crusher (in seconds).

To assess the robustness of the model, the experimental values were plotted against the predicted values as shown in Figure 7.13 for the d_{80} and d_{50} and in Figure 7.14 for the

throughput. The R^2 values are all over 94 %, which suggests good reliability of Eq. (7.4) to (7.6). This is in comparison to R^2 values of less than 50 % obtained with coal experiments as shown in Figures 7.5 and 7.7. Such a big difference in the R^2 values for models obtained with coal and quartz is attributed to the fact that the variables on which size reduction and throughput are greatly dependent on, i.e. speed and feed rate, were not considered during the coal experiments and thus limited for predictions. More statistical data for the d_{80} , d_{50} and throughput models are listed in Tables S2 to S4 in Appendix S. The P-values for speed are the lowest (in those Tables S2 to S4 in Appendix S), which implies that speed is the most important variable that affects the crusher performance. However, it is important, that this be tested at more speeds to establish more definite trends. Raising the speed from 330 rpm to 550 rpm was done cautiously for safety reasons.

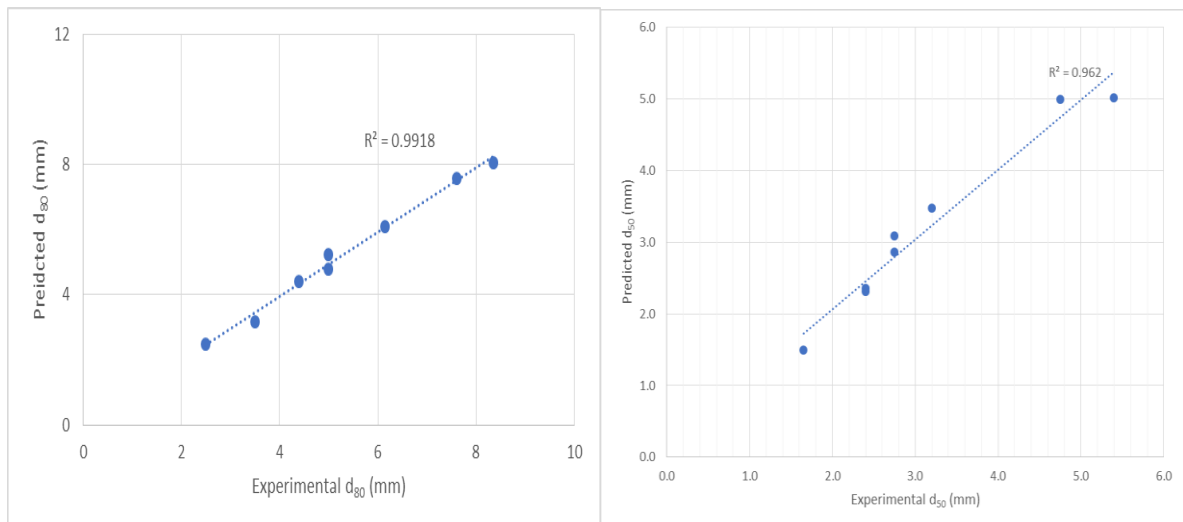


Figure 7. 13: Experimental versus predicted d_{80} values for quartz crushing with ROC

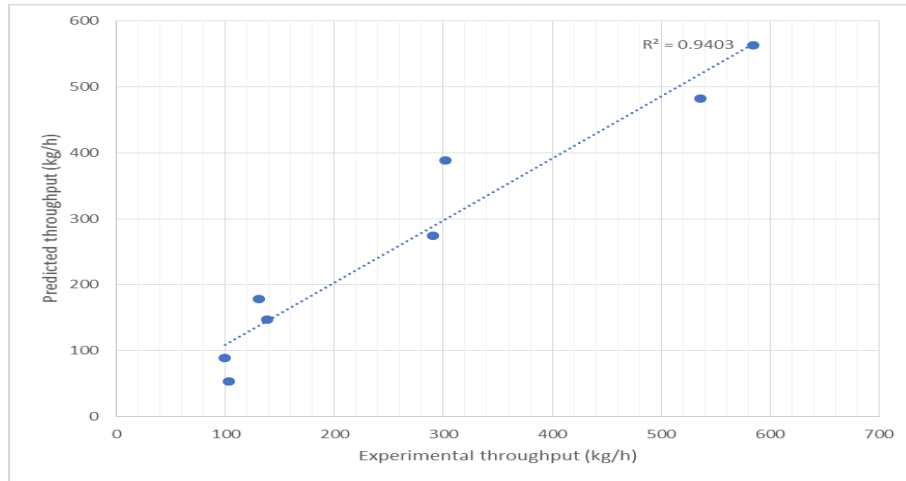


Figure 7. 14: Experimental crusher throughput versus the predicted throughput

7.2.3 Breakage parameters

The breakage distribution parameters were computed using the same technique as for coal experiments discussed in section 7.1.4. They are summarised in Table 7.7. The γ values are comparatively similar for both combinations of feed rates and speeds. With the speed of 330 rpm, the ϕ values at a higher feed rate are relatively smaller than at lower feed rate, which suggests that proportionally slow feeding ensures fragmentation of particles. This is what has been shown from the experiments (see Figure 7.11). The ϕ values at the 550 rpm speed are similar with both slow and fast feeding. This highlights the superiority of the rotational speed on the crusher performance.

Table 7. 7: Breakage distribution parameters for quartz crushed in the ROC

Size range (mm)	- 13.2+9.5	-19+13.2	- 13.2+9.5	- 19+13.2	-13.2+9.5	- 19+13.2	- 13.2+9.5	- 19+13.2
Speed (rpm)	330	330	330	330	550	550	550	550
Feed rate (kg/h)	1023	1027	1703	1720	1753	1674	1109	970
ϕ	1.36	1.30	1.17	1.18	1.40	1.31	1.47	1.30
γ	0.70	0.61	0.87	0.71	0.66	0.56	0.60	0.56
β	5.8	5.8	5.8	5.8	5.8	5.8	5.8	5.8

7.2.4 Estimation of the selection functions

As discussed in Chapter 2, the selection function can be back-calculated using the population balance model (Eq. (2.18)). Dundar et al. (2013) used the same method to estimate the rates of breakage of cement samples crushed using the HPGR. Noting that mono-size particles were crushed using the ROC, Eq. (2.15) can be used to estimate the rate of breakage for the two size classes. The variables f_i and m_i in Eq. (2.15) are equal if mono-size feeds are used, which in turn, implies that Eq. (2.15) can be written as:

$$S_i = \frac{(1 - \frac{p_i}{m_i})}{\tau} \quad (7.7)$$

Where $\frac{p_i}{m_i}$ is the mass fraction of the feed size class in the crusher product.

The relationships between the back-calculated S_i and operating variables are shown in Figure 7.15. With a low speed of 330 rpm, the rates of breakage for the two size classes are very low, equal and not dependent on the feed rate. Increasing the speed to 550 rpm shows a drastic increase in the rates of breakage and the influence of the feed size as well as feed rate coming out clearly. While this back-calculation method showed the general trend in the rates of breakage as a function of operating variables, it is recommended that a more reliable experimental method of determining the S_i values be developed specifically for the ROC. Perhaps, the review of the methodologies employed for estimating the rate of breakage in the HPGR (which is comparable in operation to the ROC) can be undertaken to form as a basis.

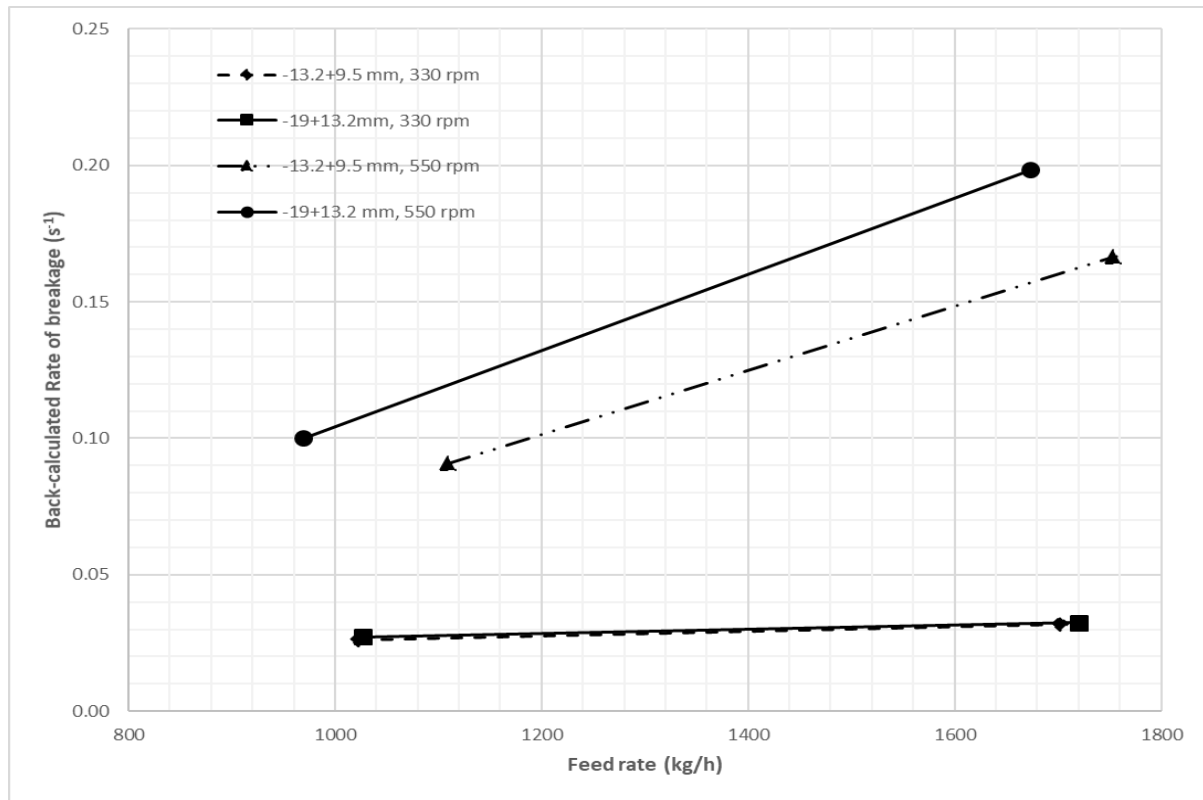


Figure 7. 15: Relationship between the rate of breakage and the operating variables of the ROC

Two ROC crushing tests were purposely conducted with mixed feeds (-19+13.2 mm and -13.2+9.5 mm), as shown in Figures 7.16 and 7.17, and the back-calculated S_i values, as well as the breakage functions (b_{ij}) obtained from the DWT, compression tests and ROC experiments, were used to estimate the product size distributions (that were compared to the experimental product size distributions shown as solid lines). This was to assess the reliability of the calculated rates of breakage as well as the breakage functions estimated using the laboratory breakage tests. The population balance model (Eq. (2.15)) was used to calculate the product size distributions. The size distributions estimated using the average b_{ij} from the DWT are closer to the experimental distribution, which is what is expected because individual particles were subjected to one breakage event and therefore, good estimates of the actual breakage functions for the broken material. The development of a novel methodology for

estimating the rate of breakage of particles comminuted in the ROC would go a long way in improving the accuracy for the application of the PBM to the ROC. Noting that the appropriate device to estimate the breakage parameters for the material crushed in the ROC is the piston and die, it is recommended that many compression tests be conducted (with replications) for the computation of the average breakage function.

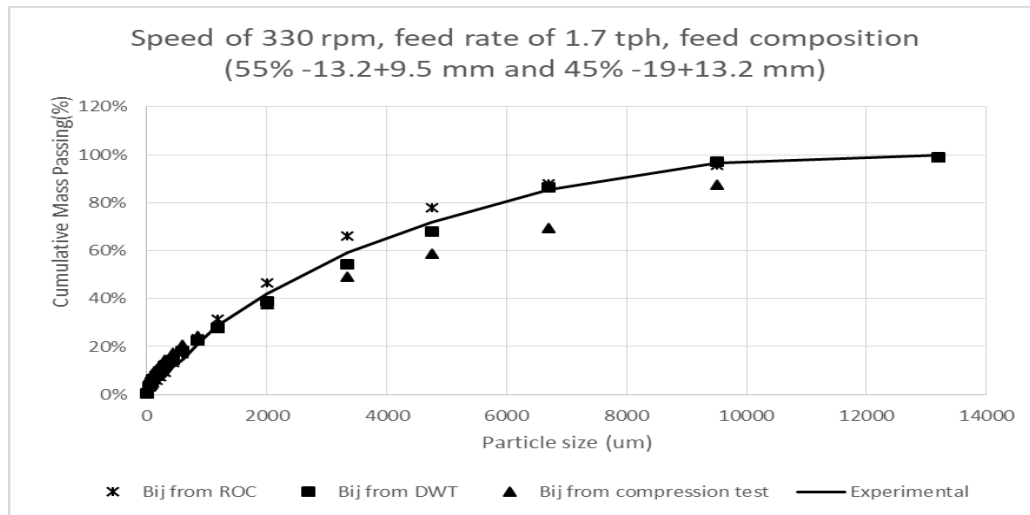


Figure 7. 16: Relationship between experimental product size distribution and predicted product size distributions using back-calculated S_i and B_{ij} from DWT and compression tests

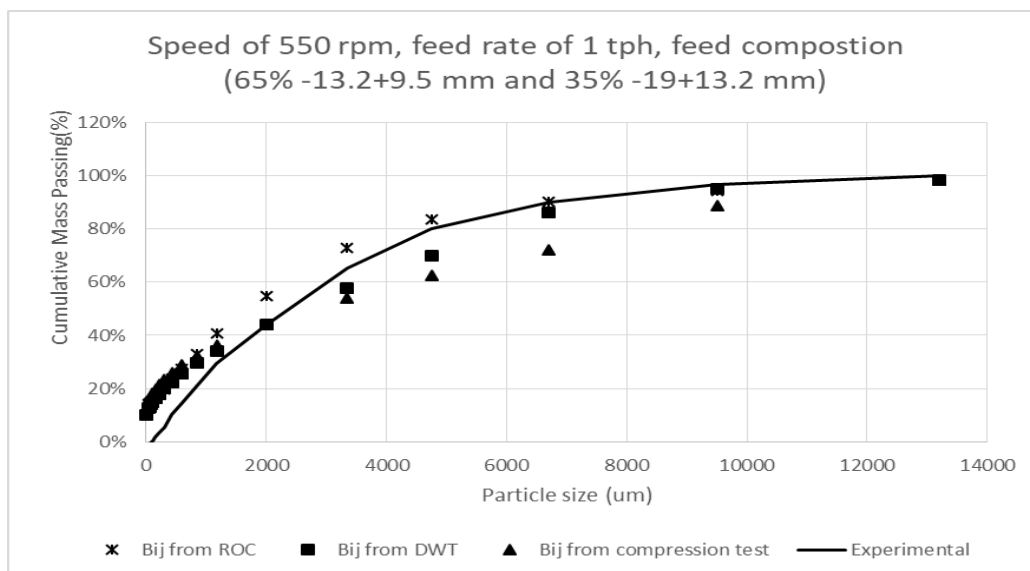


Figure 7. 17: Relationship between experimental product size distribution and predicted product size distributions using back-calculated S_i and B_{ij} from DWT and compression tests

7.3 Energy considerations in the ROC

The two major energies in the ROC are the rotational energy (the energy stored in discs) and the specific comminution energy (responsible for creating new surface areas). The rest is the friction in the system. These two types of energy are discussed in the following subsections.

7.3.1 Stored energy in the discs

As stated in Chapter 2, the energy of the discs (flywheels) can be calculated using Eq. (2.33). Considering the shapes of the discs Eq. (2.34) to Eq. (2.36) were used to derive the formulae that can be used to calculate the moment of inertia for the two discs. The derivations are discussed in Appendix T. The obtained equations for the top and bottom discs were substituted into Eq. (2.33) to get Eq. (7.8) and (7.9) which give the stored kinetic energy for the bottom and top disc respectively. The symbols in these equations are shown in Figure 7.18.

$$E_{k,bd} = \frac{1}{4} \pi \rho h_{bd} \omega_{bd,ave}^2 (R_{o,bd}^4 - R_{i,bd}^4) \quad (7.8)$$

$$E_{k,td} = \frac{1}{4} \pi \rho \omega_{td}^2 (h_{td} (r_{o,td}^4 - r_{i,td}^4) - \frac{2\pi \rho h_{c1} (R_{c1}^5 - R_{c2}^5)}{R_{c1}} + h_{fs} (r_{o,fs}^4 - r_{i,fs}^4)) \quad (7.9)$$

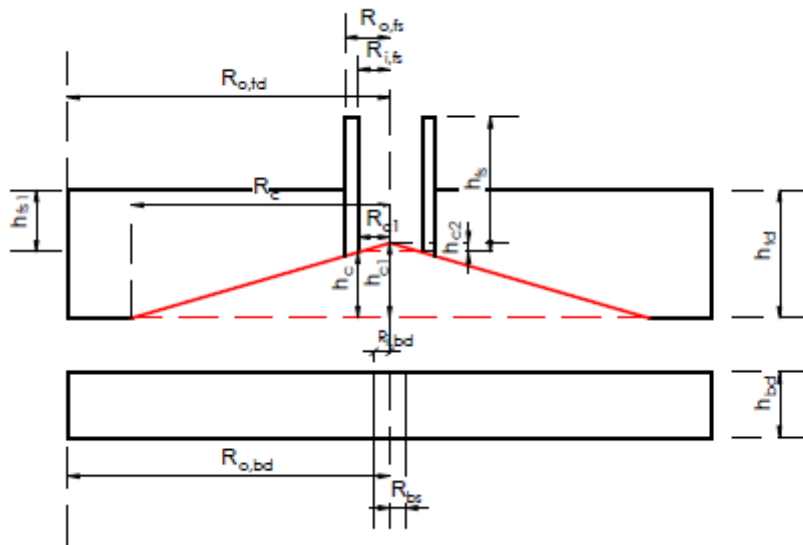


Figure 7. 18: Dimensions of the ROC discs and shafts

The amounts of energy stored in the discs at the two speeds are shown in Table 7.8. The higher the speed, the more energy stored in the flywheel. As Eq. (2.34) to (2.36) show, the moment of inertia is not a function of the speed, but rather merely dependent on the mass of the discs, hence the same moment of inertia at the two speeds. The top disc has a higher moment of inertia, hence a larger amount of energy stored in the disc. The energy is stored in the discs in the form of rotational momentum, which ensures “nearly constant” speeds of the discs. While the moment of inertia is not a function of the rotational speed, the energy stored in the discs is proportional to the square of the angular velocity as Eq. (2.33) shows, hence larger energies at 550 rpm in Table 7.8.

Table 7. 8: Energy stored in the flywheels of the ROC rotating at 330 and 550 rpm

Speed (rpm)	Disc	Moment of inertia (kg.m ²)	Energy stored (J)
330	Bottom	2.30	1373
	Top and feed chute	3.50	2091
550	Bottom	2.30	3814
	Top and feed chute	3.50	5809

7.3.2 Specific comminution energy

As stated in Chapter 4, the specific comminution energies were estimated using Eq. (4.10) using the experimental data. This was done for the ROC crushing tests (discussed in section 7.2 for quartz). The summary of the computations is given in Table U1 in Appendix U. The power signals are listed in Figures U1 to U9 in Appendix U. The relationship between the product size (d_{80}) and input energy in kWh/t is shown in Figure 7.19. The existing formulae (Bond in Eq. (2.4) and Morrell’s in Eq. (2.5)) for estimating the specific comminution energy were also assessed for the ROC application and results plotted on the same Figure 7.19. In

addition, the relationships between the d_{80} sizes and specific comminution energy from the DWT (as shown in Figure 6.2 in Chapter 6) were also used to estimate the energy input for the experimental d_{80} sizes obtained from the ROC and the results are added to Figure 7.19.

As it can be observed from the coefficient of correlation (R^2) values, there exist well-defined relationships (fitting the power function) between the product size and specific energy. The single particle breakage (such as with the DWT) is the most efficient breakage mechanism in terms of energy utilization (Tavares, 2004). This can be observed from Figure 21 that the specific comminution energies predicted using the relationship established from the DWT experiment are smaller than those predicted by Bond and Morrell's equations. The curve (in red curve) for experimental data is between the relationships for the DWT and Bond for product with d_{80} size larger than 6 mm and correlating to what is predicted using Bond equation for product with d_{80} sizes less than 6 mm. The estimated values using the Bond equation are lower than those by Eq. (2.5) for Morrell (2004). It is important to note that for both Eq. (2.4) and (2.5), the Bond work index of 13.57 kWh/t for quartz (as reported in Wills and Finch, 2016) was used. The most reliable work index for the rocks crushed in the ROC (comparable to the HPGR) could perhaps be obtained from the SCM test[®] (Morrell, 2009) and this is likely to be smaller than 13.57 kWh/t noting that the ROC (as a secondary or tertiary crusher) would not be targeting 100 μm (the standard closing size for estimating the Bond work indices for the conventional mills). The same material (quartz) needs to be crushed in other comminution machines such as cone crusher and HPGR to generate the data that would serve as a basis for comparing the performance of the ROC to the existing crusher and highlight the benefits, if any.

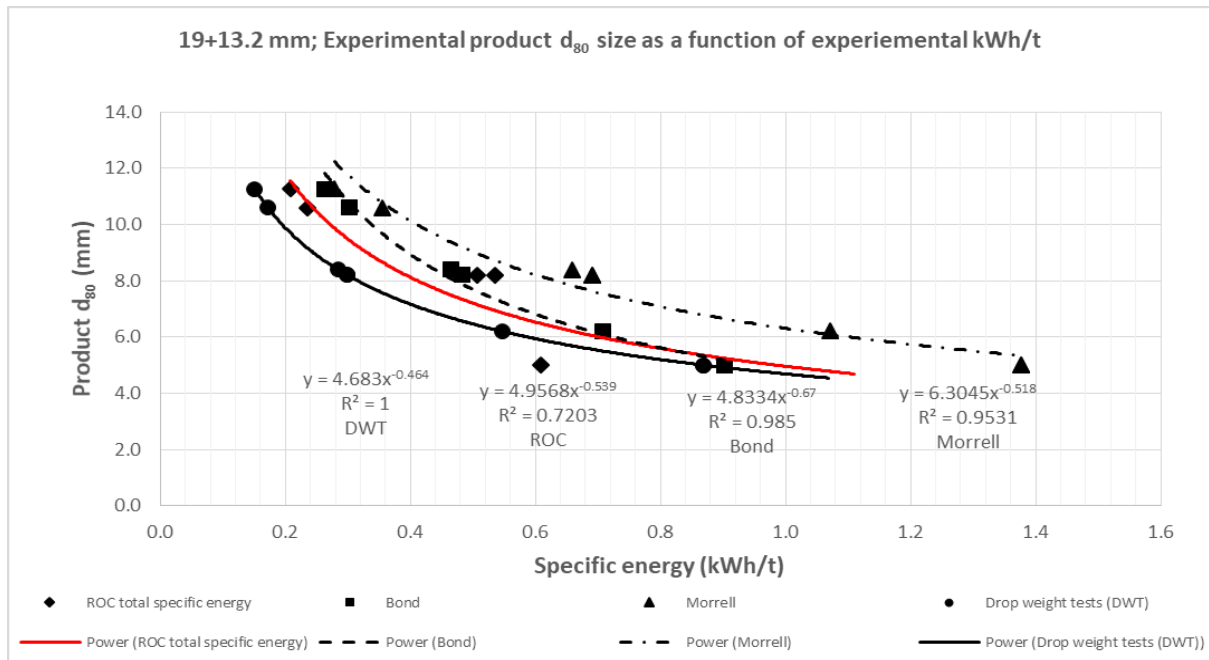


Figure 7. 19: Relationships between the specific comminution energy estimated using various methods and d_{80} size for the crusher product

7.4 Summary

This chapter presented and discussed the key findings of this project. Coal comminution in the ROC has proven not to be possible with the current design that has no corrugated profiles as evident from the production of acicular particles resulting in very low reduction ratios in the range of 1.5 to 2. On the other hand, quartz breakage was achieved with reduction ratio as high as 7 recorded when feeding at 1 tph and speed of 550 rpm. The rotational speed is the most important operating parameter affecting the performance of the crusher. The next important parameter is the feed rate. The investigation of the effect of offset on size reduction has not resulted in a reliable trend. Further experiments are recommended in that regard to establish sustained trends. With promising results achieved with the speed of 550 rpm as compared to results at 330 rpm, further experiments are needed at various operating parameters in order to improve the accuracy of the derived relationships between the dependent variables (d_{80} , d_{50} , throughput) and independent variables (feed size, feed rate, speed, offset, exit gap). The ROC promises to be higher throughput crusher that is as well as characterized by higher size

reduction ratios, but the structure needs to be strengthened to ensure the full transmission of energy to the particles. While the method of back-calculating the rates of breakage proven to have higher accuracy levels, it is recommended that a direct method of computing the selection function for the particles crushed in the ROC be developed. Perhaps, the methodologies for computing the selection function in the HPGR and VRM may be reviewed to form as the basis.

CHAPTER EIGHT: CONCLUSIONS AND RECOMMENDATIONS

8.1 Conclusions

The working prototype for the laboratory ROC has been built and it is instrumented with sensors to pick up signals for speeds and load measurements. The equipment has just been commissioned thus only a limited run of experiments have been performed. The indications so far have highlighted that the disc speed is a key factor affecting the performance. The study of the effects of the horizontal offset between the discs, vertical exit gap and feed size distribution was affected by the structural flexure of the equipment, which will have to be addressed in future. From the trends established so far, the small offset (5 mm) as compared to the offset of 10 mm when the exit gap is 3 mm produced finer products. Many experimental runs are needed to establish sustained trends. The application of the population balance modelling was attempted using the experimentally determined breakage functions.

8.1.1 Coal comminution

The coal comminution in the ROC resulted in low reduction ratios (about 2). These results were attributed to the mineralogical and deformation characteristics of coal. Many acicular (flat) particles were discharged from the crusher. Consideration of modifying surface profile of the crusher in future may address this problem.

8.1.2 Quartz comminution

Size reduction ratios as high as 7 were recorded from quartz experiments at a speed of 550 rpm. But this is obtainable only at a feed rate of about 1 tph. High feed rates proved to be detrimental as particles tend to push the discs apart. As pointed out already, there is a need to strengthen the structure to ensure that no significant flexure happens and thereby ensuring that the energy meant for breaking the particles is efficiently used. With low feed rate (of about 1 tph), such a

flexure does not happen or rather it is minimised, and this allows the particles to get comminuted as they go along the comminution cavity.

8.2 Recommendations and future work

There is a need to address the structural weakness to ensure the full transmission of energy to the particles nipped between the crusher discs. The rigidity of the structure would ensure that the full potential of the crusher is established. While the experiments conducted with this new crusher have given some insights about the effect of operating variables on the crusher performance, there is still a need for more test works. Hence, the following future works are recommended:

- More experiments with quartz (and any other materials) need to be conducted at the speed of 550 rpm and then higher speeds should also be tried to establish sustained trends and help in optimisation of the crusher performance. Tests at higher offset values (say 15, 20 and 25 mm) would yield more useful information on the effect of horizontal offset between the discs on size reduction, throughput and power draw.
- Simulation using the DEM for various design configurations such as throat diameter of the feed chute, height of feed chute, feeding conditions, interior flat edge of the top disc, would improve the understanding of flow behaviours for the particles in the crusher.
- Design of disc profiles of various configurations needs to be undertaken using the DEM to study both the breakage and transportation of particles in the crusher. This would guide the future modifications of the crushing faces of the crusher.
- Comparative studies with competing comminution machines such as HPGR, short head cone crusher and Loesche mill to establish benefits in terms of energy efficiency, throughput, size reduction, if any, for the ROC over the existing machines.

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Appendices

Appendix A: Cone crushers, HPGR and VRM

A1. Cone crushers

The cone crushers, lighter than primary crushers, can reduce the size further to 0.3 – 2 cm, either in one stage or multi-stages. Essentially, the cone crusher is a modified gyratory crusher, with a shorter and unsuspended spindle as shown in Figure A1 (Wills and Napier-Munn, 2006). The spindle is free to turn on its axis in the eccentric sleeve so that during crushing the lumps are compressed between the rotating head and top shell segments. The spindle is supported in a curved bearing below the head and the crushing shell flares outwards allowing for the swell of the broken particles by providing an increasing cross sectional (annular) area towards the discharge.

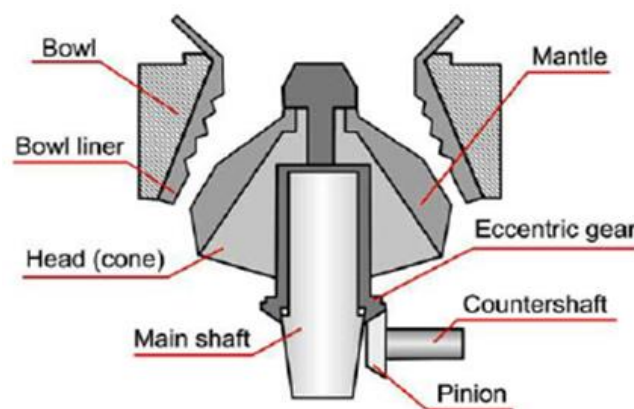


Figure A 1: Cone crusher functional diagram

A2. High Pressure Grinding rolls

The schematic diagram of the HPGR is shown in Figure A2. The working principle of the HPGR is a very straightforward in that the material is compressed between two counter-rotating rolls. The hydraulic cylinders apply very high pressure to the system causing inter-particle breakage to take place as the feed travels between the two rolls.

A3. Vertical Roller Mills (VRM)

The working principle for the VRM is shown in Figure A3. A notable design feature of the VRM is that milling, and classification is combined in a single unit, thereby simplifying the milling circuit. During the operation, the material is initially fed to the middle of the rotating table, then it moves towards the edge of the table where the rollers exert pressure resulting in breakage. The ground particles are lifted to the classification zone where the target product size is obtained, and the reject material is reported back to the mill body with the fresh feed.

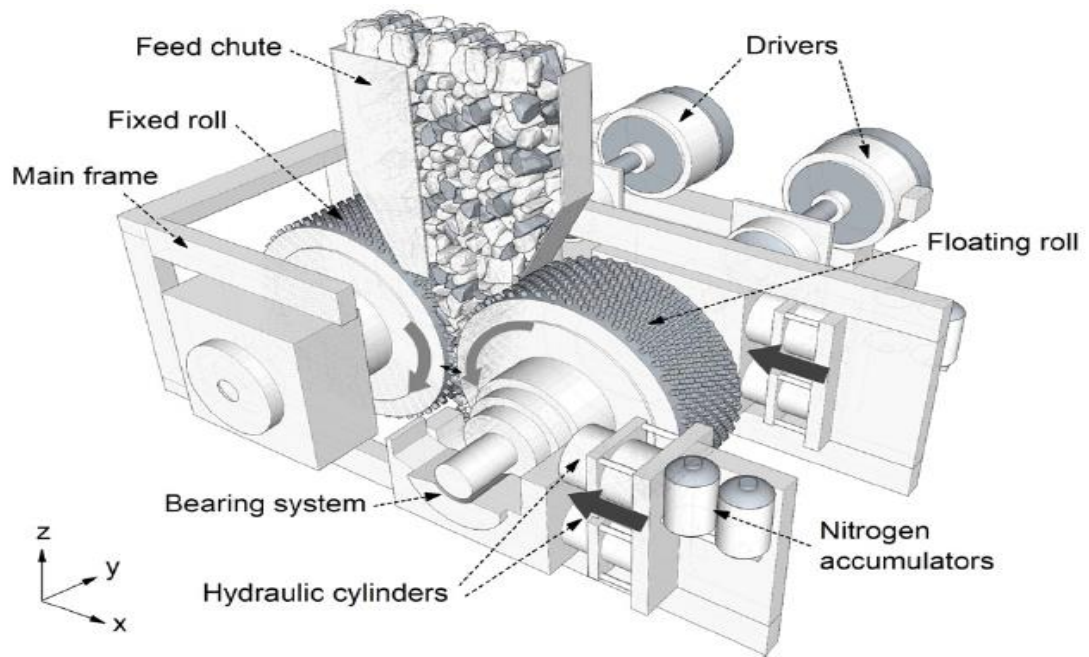


Figure A 2: Schematic of a HPGR (Barrios & Tavares, 2016)

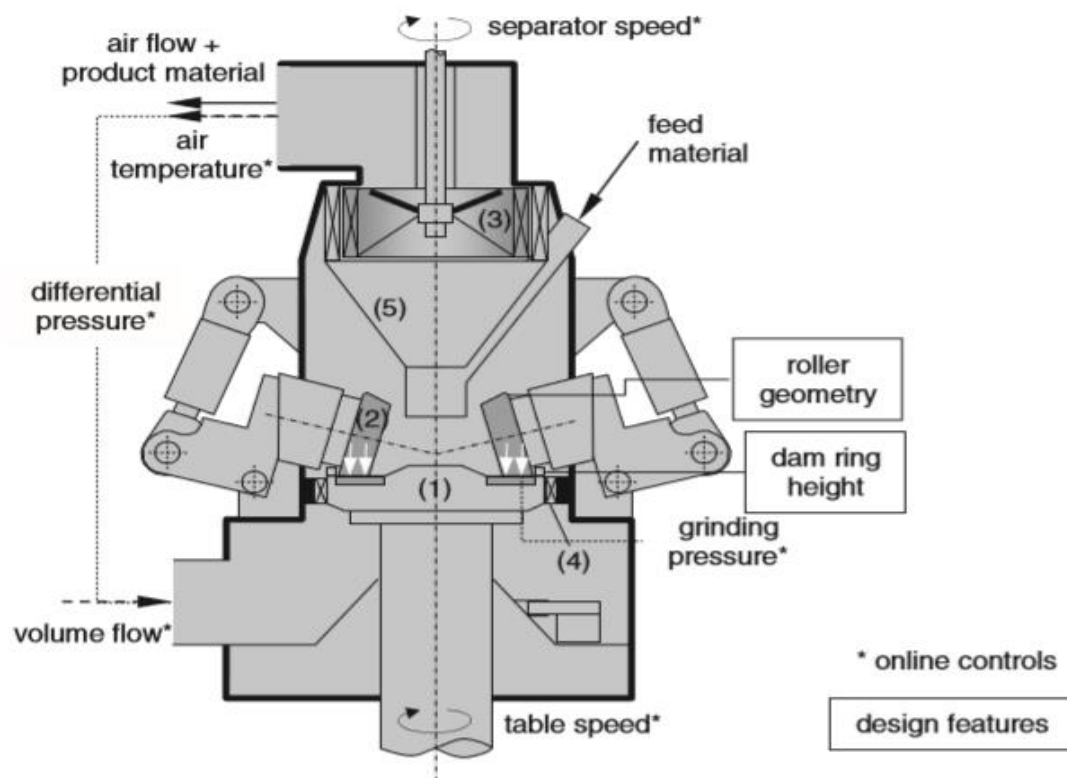


Figure A 3: Schematic diagram for the air swept mode VRM (Reichert et al., 2015)

Appendix B: DEM codes

Table B 1: Examples of DEM codes (Bharadwaj, 2014)

Codes	Few examples	Advantages	Disadvantages
Open-source	LIGGGHTS, YADE-OPEN, ESys-Particle, LMGC90	- No licence cost - Full control of source - Multiple users and simulations. - Cost effective scaling for large organisations	- Programming and DEM expertise needed - Support group needed for multiple users - XML based inputs and outputs
Commercial	Rocky-ESSS, PCF3D, EDEM, Chute Maven, Chute Analyst, PASSAGE	- No programming experience needed - Better graphic user interface - Support and training	- Expensive license cost, especially for multiple users and multi-core processor licenses - Limited control over implementation but code is thoroughly tested for quality - Custom defined force models and measurements might not be trivial to implement

Table B 2: Spring stiffness and damping coefficients used in the contact model

	Normal direction (n)	Tangential direction (t)
Spring stiffness constant (K)	$K_n = \frac{4}{3} E^* \sqrt{R^* \delta_n}$	$K_t = 8 G^* \sqrt{R^* \delta_t}$
Damping coefficient (C)	$C_n = 2 \sqrt{\frac{5}{6}} \beta \sqrt{S_n m^*}$	$C_t = 2 \sqrt{\frac{5}{6}} \beta \sqrt{K_t m^*}$
$S_n = 2 E^* \sqrt{R^* \delta_n} \ln \varepsilon$ $\beta = \frac{\ln \varepsilon}{\sqrt{\ln^2 \varepsilon + \pi^2}}$ E* is the effective Young modulus G* is the effective shear modulus m* is the effective mass		

Appendix C: The discs before installation and drawing for the ROC

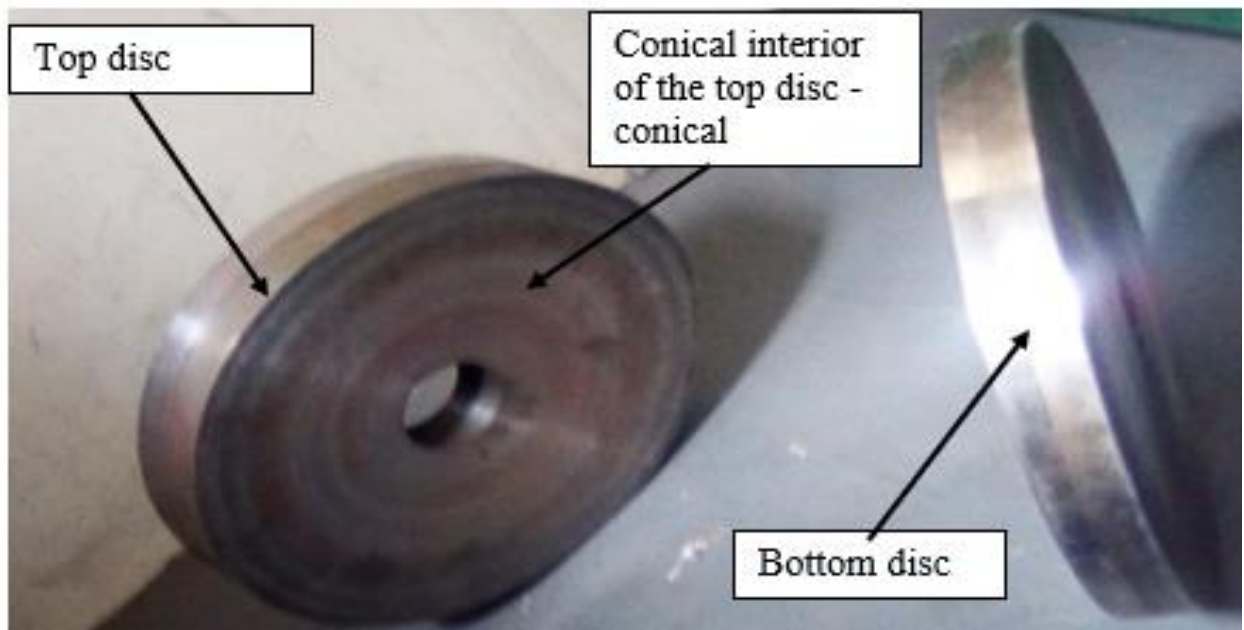


Figure C 1: The crusher discs before installation

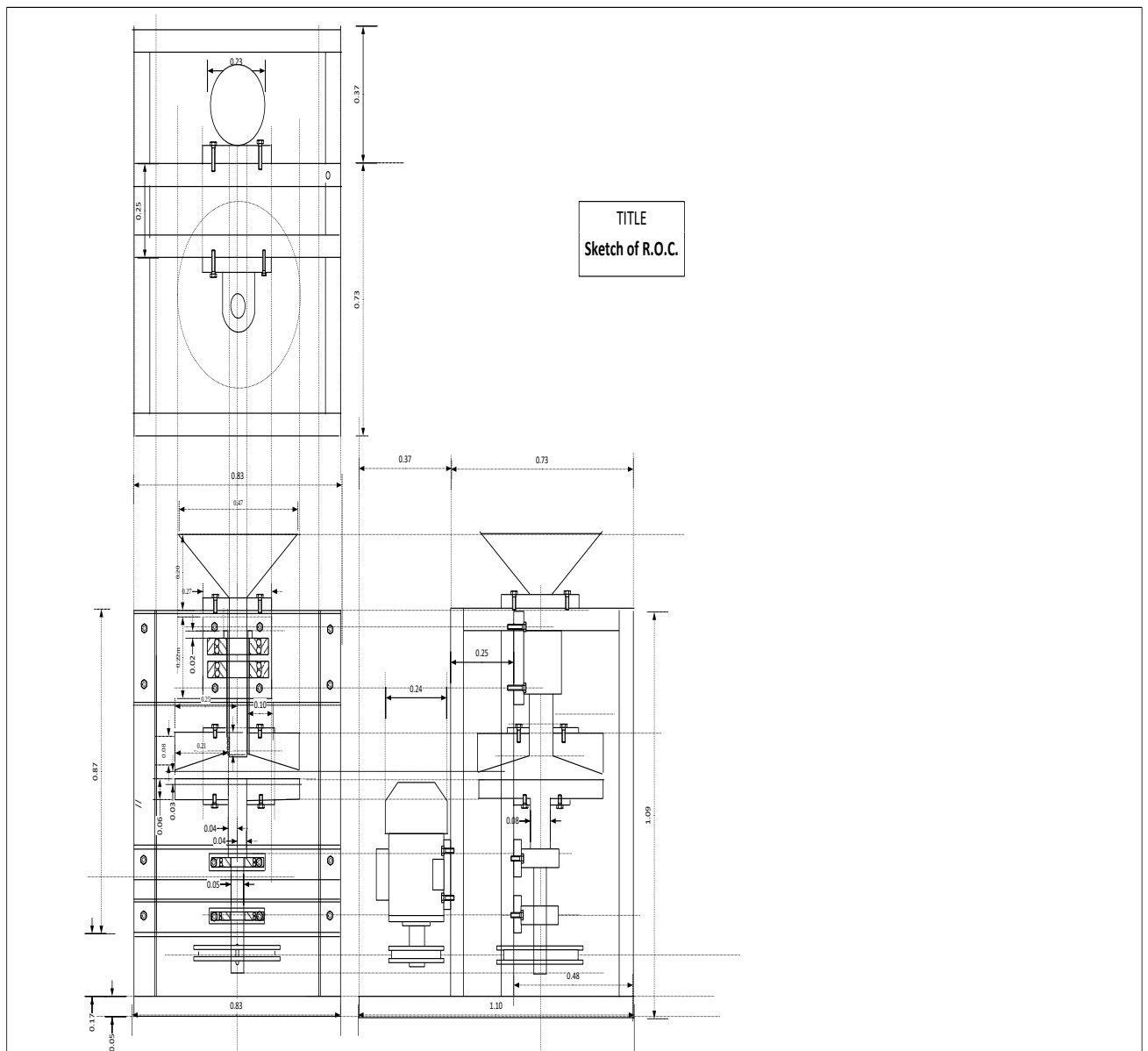


Figure C 2: Drawing for the ROC

Appendix D: Standard operating procedures for the Rotary offset crusher

SAFETY

It is very important that a principle of “take 2 minutes” to think through the task is always practised. The operator must put on the PPEs (hard gloves, lab coat, closed shoes (preferably the safety boots), goggles, ear plugs, dust masks). When the crusher is in operation and during discharging, the dust extractor must be ON. House-keeping is key.

PROCEDURES

1. The offset and disc gap are set, (see Figure D1 for the nuts that are adjusted for the setting the offset and gap) and crusher is inspected to make sure all nuts are tight. The collection box is then closed.
2. The bed dimensions for the grate on the conveyor are adjusted (see Figure D2)
3. The dimensions for the grate that controls the bed width and height of the material fed to the crusher are adjusted. The weighed sample is then placed on the conveyor belt.
4. The instrumentation circuits (for speed and load measurement) are connected to the computer and tested before commencing with the test work.
5. Take readings (outputs for the load cells of the feeder and motor drive as well as speeds for the discs) before feeding the material
6. The crusher is put into operation. Take some readings for few seconds before feeding the material to the crusher.
7. The material is then fed to the crusher by starting the conveyor.
8. Continuously take readings for the load cells, speeds for both discs.
9. When the top disc stops, put the power supply for the crusher off by pressing the emergency button. The conveyor belt can be stopped as soon as feeding is done
10. Collect the material in the collection box and in the crushing chamber (if any)
11. Dry sieve the crusher product.

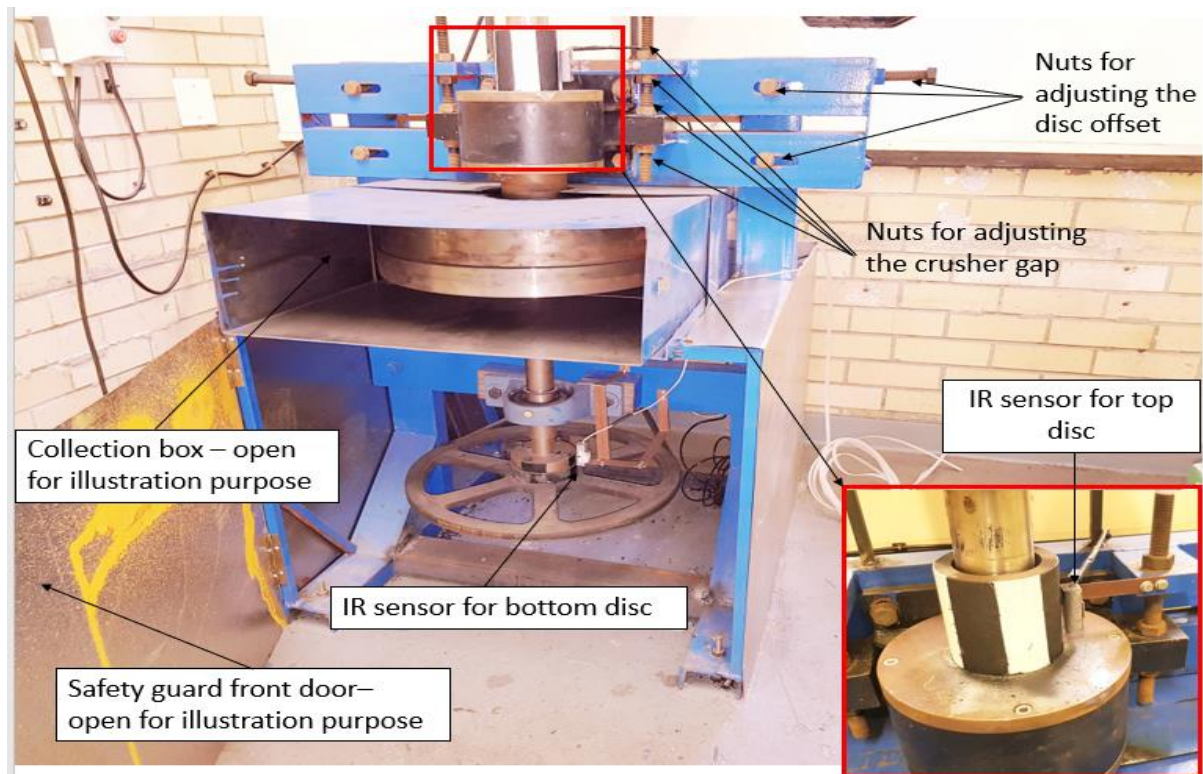


Figure D 1: The nuts for changing the offset and exit gap of the ROC



Figure D 2: The grate on the conveyor belt to control the feed rate

Appendix E: Proximate Analysis

In TGA, the sample contained in the pan of a sensitive analytical balance is heated and the weight change is recorded. To obtain a complete proximate analysis in a single TGA experiment, the system is programmed to hold initially at 200 °C in nitrogen, then to jump to 900 °C and hold for a specified time in nitrogen before switching to oxygen. Figure E1 illustrates the typical TGA profile obtained. The three weight loss steps correspond to moisture, volatiles and fixed carbon respectively. The remaining weight at 900 °C in oxygen is ash. Results for the coal sample used to conduct experiments are shown in Figure E2.

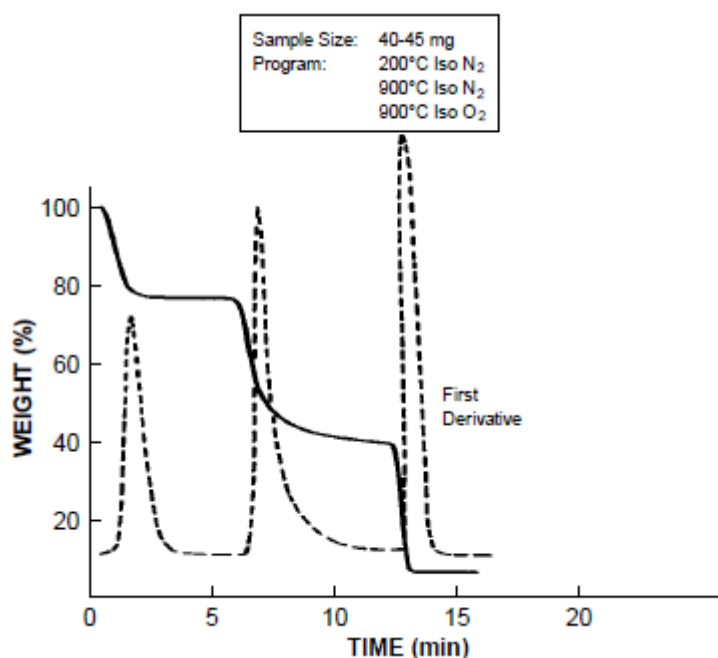


Figure E 1: Steps for performing the proximate analysis using TGA (taken from a document accessed on <http://www.tainstruments.com/pdf/literature/TA129.pdf>)

Sample: TN Coal Sample 1
Size: 10.4570 mg
Method: Pyrolysis Biomass Janet 2018
Comment: Thermal Decomposition

DSC-TGA

File: \\...\\Data\\Sanchol\\TN Coal Sample 1.001
Operator: Janet
Run Date: 02-Nov-2018 16:03
Instrument: SDT Q600 V20.9 Build 20

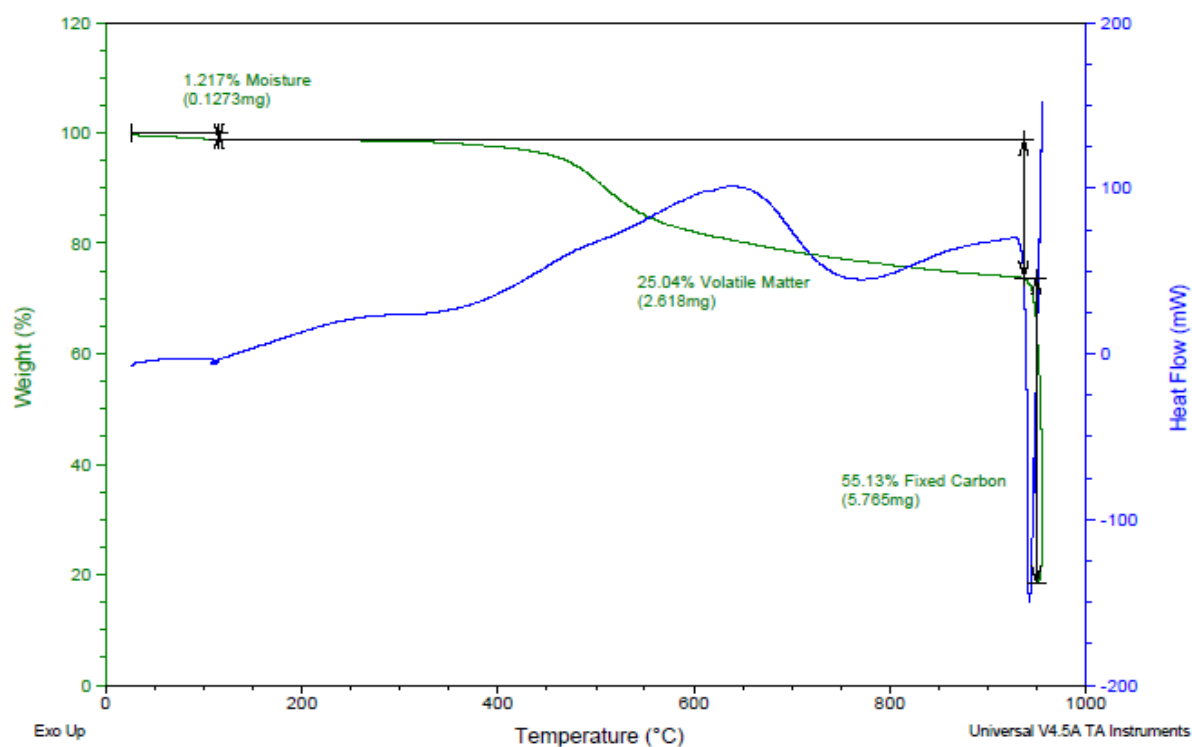


Figure E 2: Proximate analysis of the coal sample crushed with the rotary offset crusher

Appendix F: Treatments for the DEM simulations

Table F 1: Experimental treatments for DEM simulations

Run	Ball size (mm)	Speed (rpm)	Offset (mm)	Speed (rpm)
1	4	330	15	4
2	7	330	10	7
3	7	765	10	4
4	10	330	15	10
5	7	765	10	10
6	4	330	5	4
7	10	1200	15	4
8	10	765	10	7
9	7	765	5	7
10	7	765	15	7
11	7	765	10	7
12	7	1200	10	7
13	4	765	10	7
14	7	765	10	7
15	10	1200	5	4
16	4	1200	15	10
17	4	1200	5	10
18	10	330	5	10

Appendix G: Arduino Microcontroller

A microcontroller is defined by Wikipedia as a small computer on a single integrated circuit containing a process core, memory and programmable input and output peripherals. Arduino is one of the popular microcontroller boards based on ATmega328. The popularity of Arduino in engineering projects can be attributed to the fact that it has an open source software, and it is cheaper and readily available in many electronic shops. Its programming language does not require the user to be an expert and there are many documented open source materials written on its application in various engineering projects which served as a basis for its application in the crusher instrumentation. The software can freely be downloaded on the internet from <http://arduino.cc/en/Main/Software>

Description of the Arduino UNO board

Retrieved from <http://www.mantech.co.za/Datasheets/Products/ST1025.pdf>

The family of Arduino microcontrollers include Arduino UNO, Arduino Leonardo, Arduino Lily Pad, Arduino Mini, Arduino Mini Pro, Arduino BT, Arduino Nano and Arduino Mega, but among all these, Arduino UNO and Arduino Mega are the popular versions. The typical Arduino UNO board is shown in Figure G1. Boards for all other types of Arduino resemble this board, with the differences being in the number of input and output pins.

The board has 14 digital pins which can either be used as input or output pins and each of these digital pins operate at 5 volts, with some having specialized functions, for example pins 0 and 1 are for serial data communication. Pins 2 and 3 are for external interruption, which makes it possible for them to be configured to trigger an interrupt in value. In addition to digital pins, the board has 6 analog inputs, labelled A0 through A5, each of which provide 10 bits of resolution (i.e. 1024 different values), giving values spanning between 0 and 5 V. In addition to the digital, analog and power pins, the board has a 16 MHz oscillator. All other pins are labelled on Figure G1.

Power Supply

The Arduino can be powered via the USB connection or with an external power supply. If the power is supplied via the USB cable from a computer, the voltage is automatically set to 5 V with the help of on-board voltage regulator. The battery is commonly used to supply external (non-USB) power. Leads from a battery can be inserted in the GND and Vin pin headers of the power connector. The board can operate on an external supply of 6 to 20 volts, but if it is supplied with less than 7V, however, the 5V pin may supply less than five volts and the board becomes unstable (“Arduino Uno”, n.d.). On the other hand, if it is supplied with more than 12 V, the voltage regulator may overheat and in turn damage the board. It is, therefore, recommended that the voltage supply be in the range of 7 to 12 V.

Programming

The circuit developed for a specific function and connected to Arduino board cannot respond to the change in the environment and gives output if there is no program uploaded on the board. First, the software must be downloaded into the computer, and the code (commonly known as a sketch) can then easily be written in what is called integrated development environment (IDE), shown in Figure G2, using the C++ language. All codes have two main parts: void setup () and void loop ().

- The void setup () function follows the declaration of any variable at the very beginning of the sketch and as the name implies, it is the function that prepares the sketch and it runs only once. It is used to set modes (input or output) of the pins and/or initialise serial communications.
- Following the preparation of the sketch the void loop () function follows and this has commands that are executed continuously, for example triggering outputs. It simply allows the sketch to change, respond and control the Arduino board and the associated circuit. This function is the core of all sketches and it does the bulk of the work.

Arduino UNO board comes pre-burned with a boot loader that allows the user to upload a new sketch to it without the use of an external hardware programmer. Once the sketch is uploaded on the board and the circuit is connected to the board as well, the sketch starts to run and gives output.

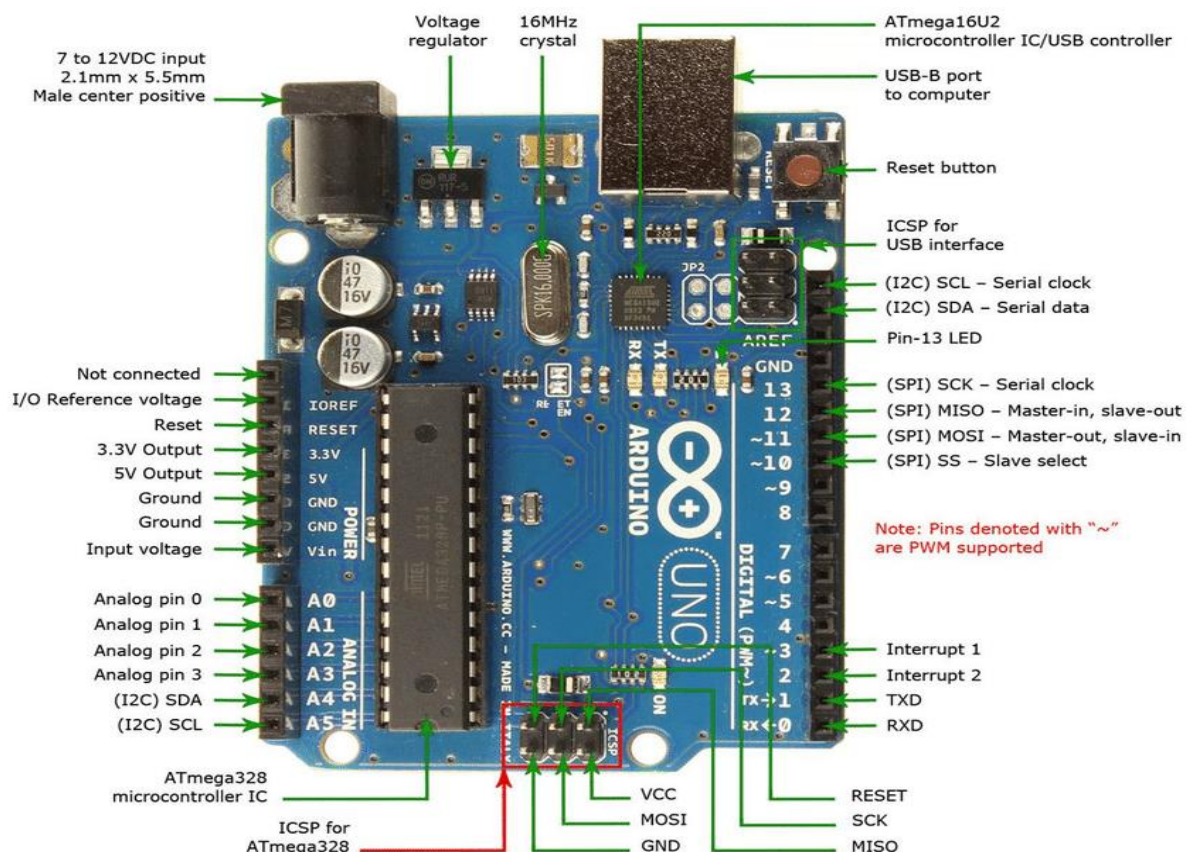


Figure G 1: Arduino UNO Schematic

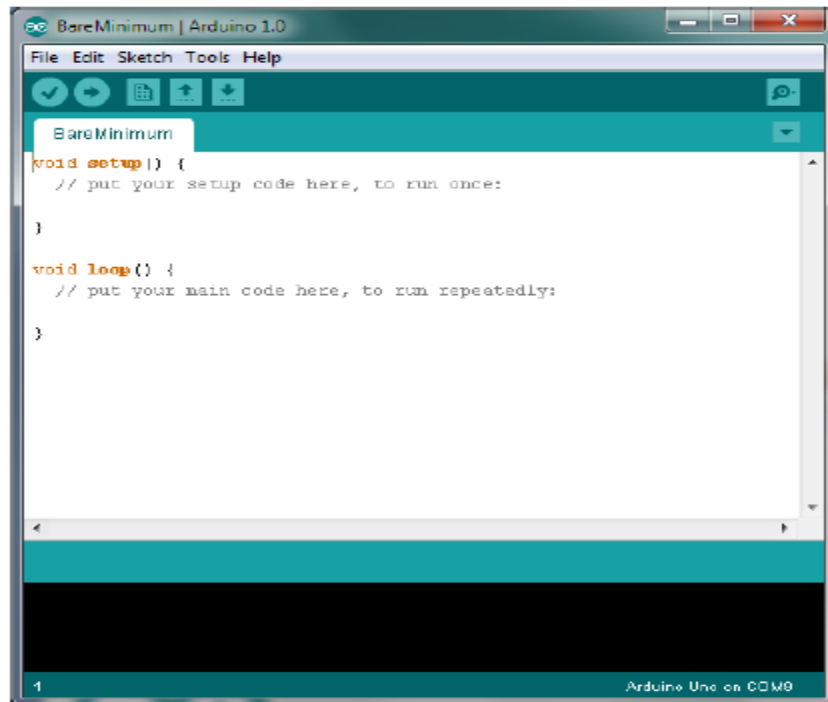


Figure G 2: Parts of Arduino IDE

Serial Communication

Serial is used to communicate between Arduino board and other devices such as computers, other Arduino, LCD screen, OLED screen and SD card micro board. Serial uses a serial port, also called universal asynchronous receiver-transmitter (UART) to transmit and receive information. It is worth noting that information cannot be received and transmitted at the same time and not more than two devices can be connected to the Arduino for serial communication. The Arduino receives (RX) and transmits (TX) information through digital pins 0 and 1 respectively. If the computer is connected to the Arduino board using a USB cable the information can appear on the serial monitor.

Data capturing

There are three common ways of capturing data that are listed below:

1. Display on the serial monitor that can be opened in the IDE (the Arduino board must be connected to the computer),
2. Send to the Microsoft excel using the data acquisition addon tool called PLX DAQ.
3. Save in the SD card (using the data logger shield board)

1. Serial monitor

The monitor can be opened in the tools (see Figure G2). One example is shown in Figure G3.

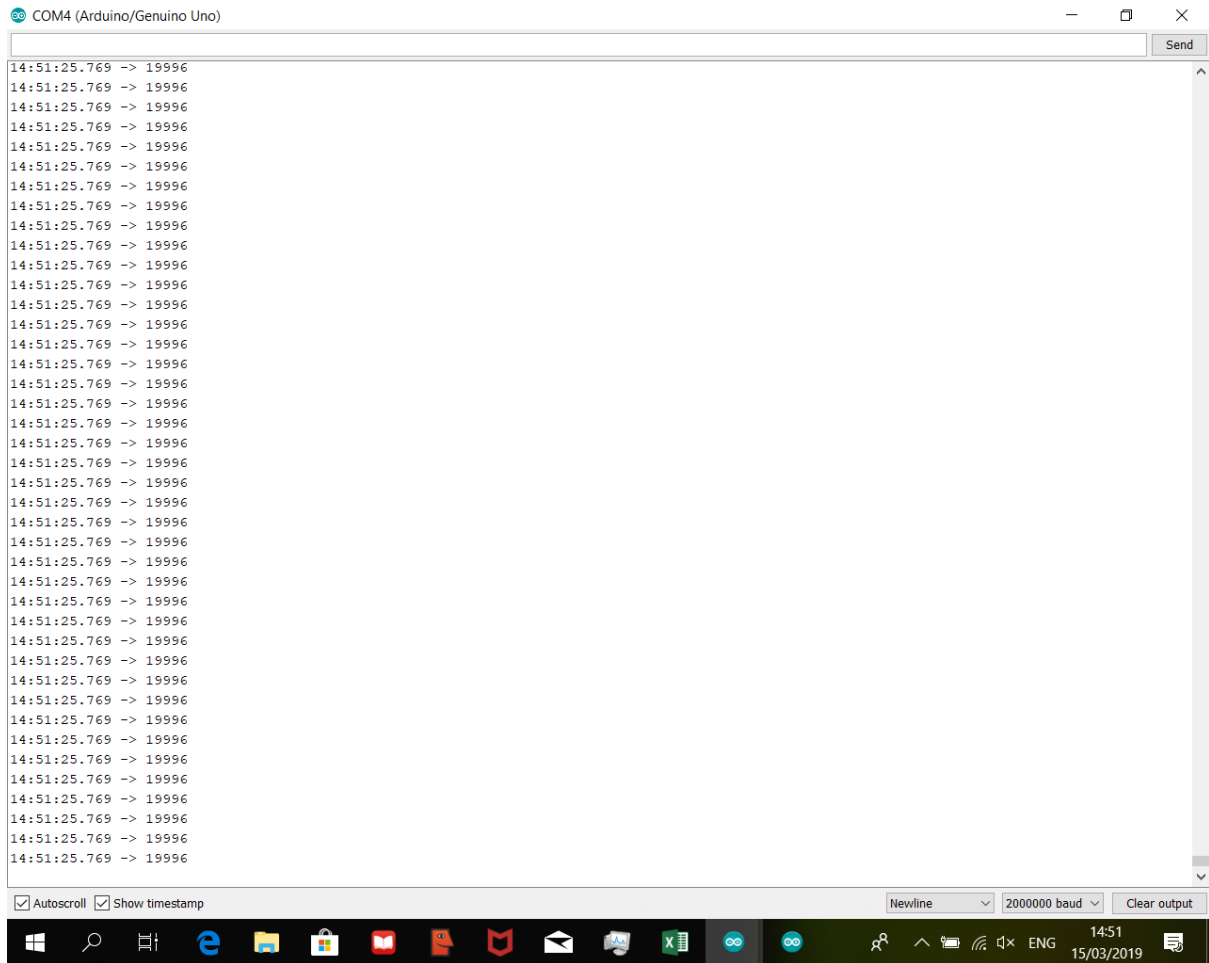


Figure G 3: An illustration of data displayed on the serial monitor

2. PLX-DAQ for data acquisition

This is a Parallax microcontroller data acquisition add-on tool for Microsoft Excel®, which can be used with any microcontroller connected to the sensor and serial port of the computer that, to record the output data in real time. PLX denotes Parallax and DAQ stands for data acquisition.

More information can be accessed on the following websites:

<https://www.instructables.com/id/sending-data-from-Arduino-to-excel-and-plotting-it/>

<https://www.parallax.com/downloads/plx-daq/>

<https://forum.arduino.cc/index.php?topic=209056.0>

Common features of PLX DAQ worth noting includes:

- Plot data as it arrives in real time using Microsoft Excel®.
- Record up to 26 columns of data.
- Mark data with real time (hh:mm:ss or seconds) since reset.
- Read/write any cell on a worksheet.
- Read/set any of the four checkboxes to control the interface.
- Baud rates up to 128 000.

- Support COM 1 to 15.
- It works best with Microsoft Excel 2000 – 2003, but also tested on Microsoft excel 2010.
- It is tested on Window 98 and XP. Used with Window 10 in this research.

The procedures for using the PLX DAQ with Arduino for data capturing are as follows:

1. You simply download the software from <https://www.parallax.com/downloads/plx-daq>, install it and open the sample excel file then you access the visual basics for application (VBA) code. The Arduino IDE code should be uploaded on Arduino board. Common syntaxes are described in the Arduino code for load measurement discussed in Appendix H. PLX DAQ relies on UART serial communication thus you must use the Serial.print() or other functions from the serial class.
2. In excel open the file “PLX DAQ, then an automatic notification pops up asking if you want to run the Active X macro, click Ok, then you get a view such as what is shown in Figure G4. Choose the correct port of the computer to which the Arduino is connected as well as the baud rate selected in the program and press connect.
3. Data transmission should have begun by now, displayed in columns as instructed in the code. One example is shown in Figure G5.

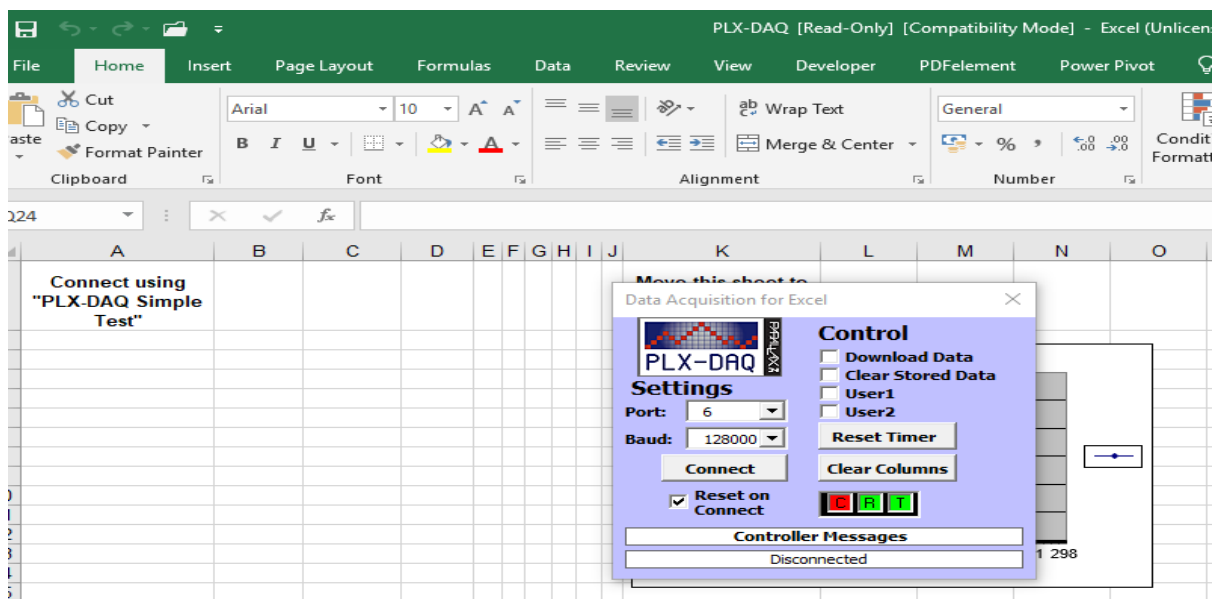


Figure G 4: Illustration of the view of the PLX DAQ spreadsheet with a control panel

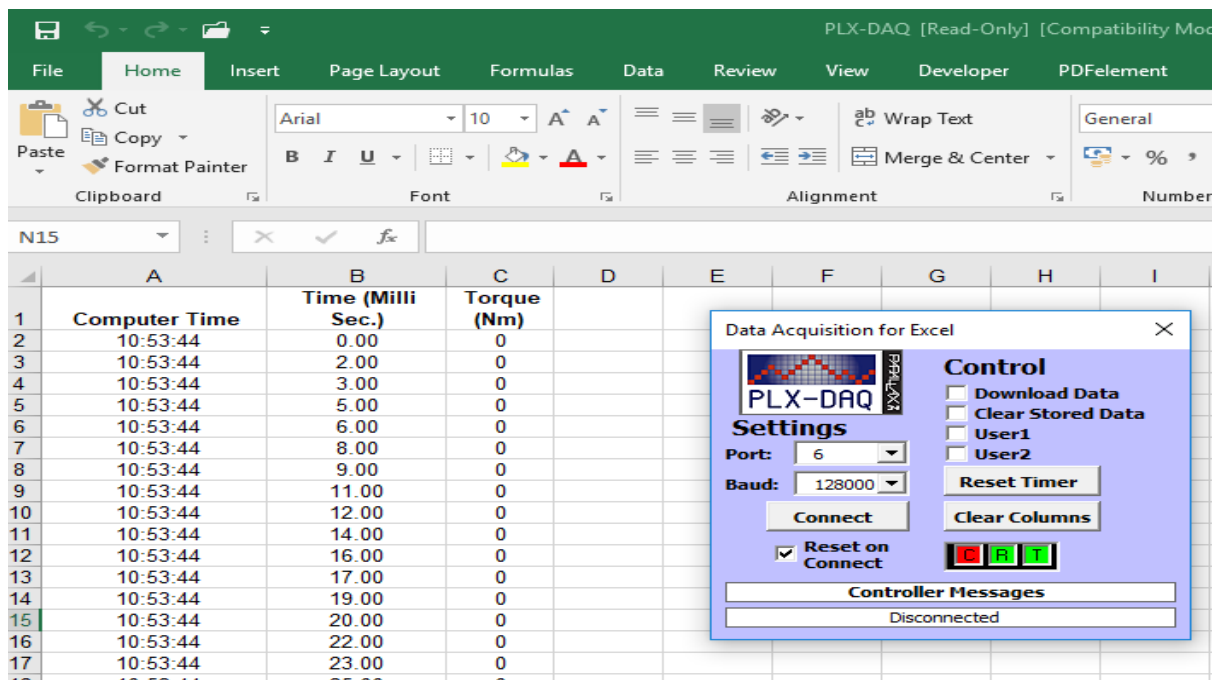


Figure G 5: PLX DAQ Spreadsheet with the control panel and illustration of some data

3. Description of SD card code features

The communication between the microcontroller and the SD card uses SPI, which takes place on digital pins 11, 12, and 13 (on most Arduino boards) or 50, 51, and 52 (Arduino Mega). Additionally, another pin must be used to select the SD card. This can be the hardware SS pin - pin 10 (on most Arduino boards) or pin 53 (on the Mega) - or another pin specified in the call to `SD.begin()`. Note that even if you don't use the hardware SS pin, it must be left as an output or the SD library won't work. Different boards use different pins for this functionality, so be sure you have selected the correct pin in `SD.begin()`. The board used for crusher instrumentation is shown in Figure G6. More information can be accessed on this website <https://www.arduino.cc/en/Reference/SDCardNotes>

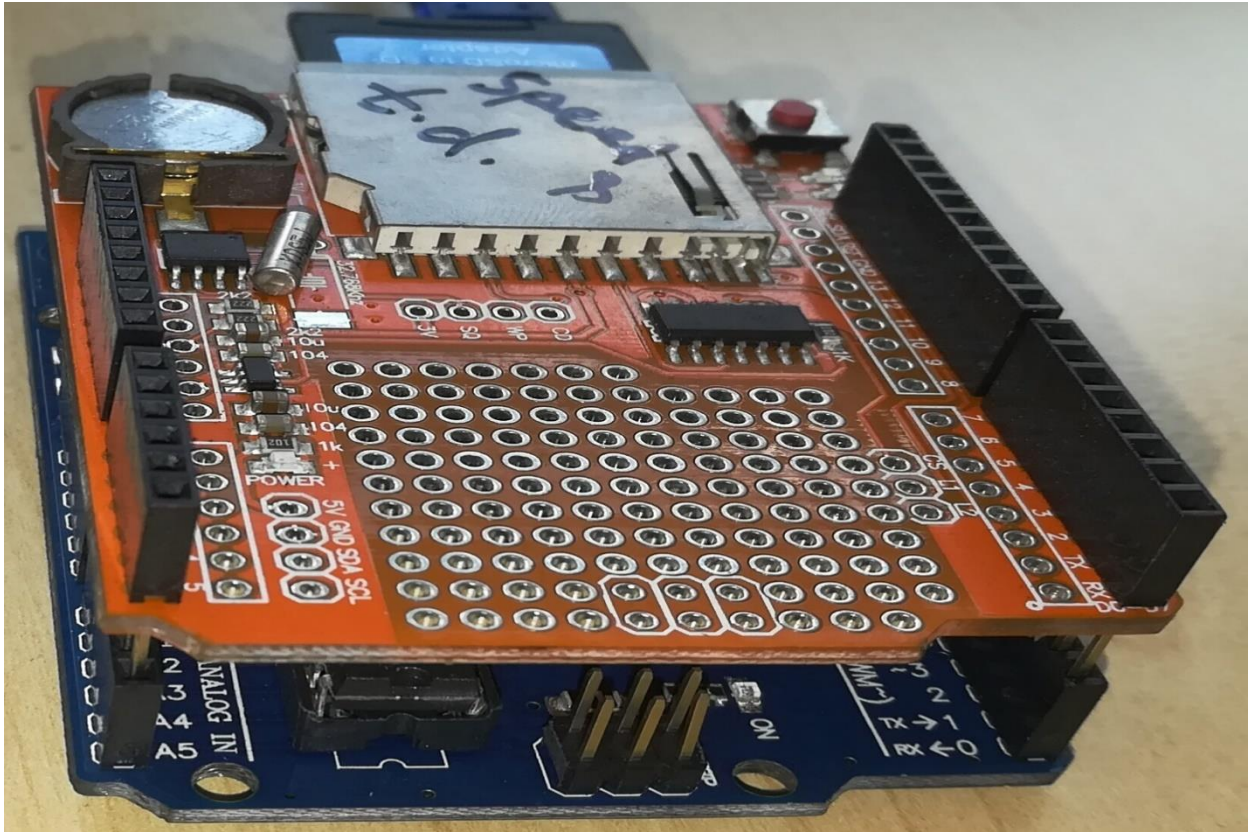


Figure G 6: The datalogging shield combined with the Arduino UNO

Appendix H: Descriptions and specifications IR sensor and auxiliary components

This was used for measuring the speed of the spinning discs of the crusher. It is shown in Figure H1 below and its specifications are listed in Table H1.

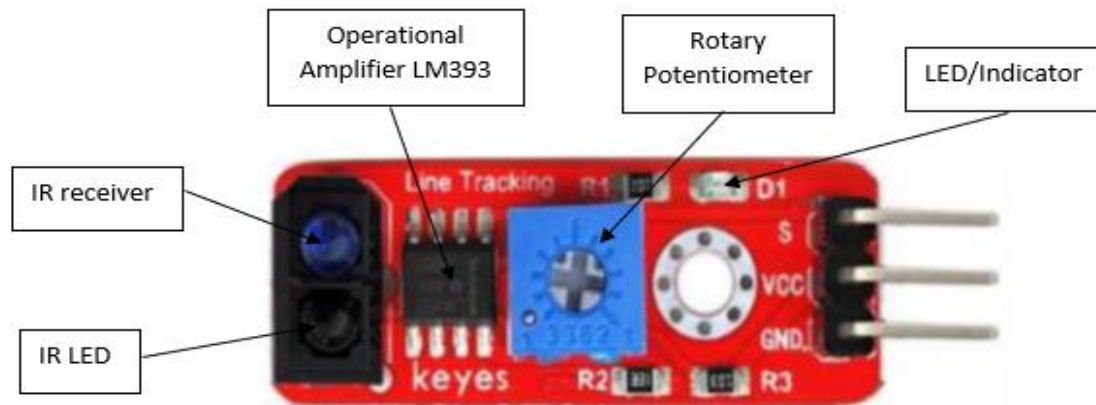


Figure H 1: KE0068-180802A IR line tracking sensor module ("Mantech Eletronics", n.d.)

Table H 1: Specifications of the IR line sensor board

Working voltage	3.3 – 5 V DC
Interface	3 Pins Interface
Output	Digital signal
Detection distance	0 – 3 cm
Mass	2.3 g

The Operational Amplifier (LM393)

Retrieved from <http://www.mantech.co.za/Datasheets/Products/LM393-ON-7B.pdf>

The signal output of the IR receiver needs to be conditioned for the Arduino to be able to make computations. The operational amplifier on the IR sensor board in Figure H1 amplify the digital output of the receiver and either LOW or HIGH output is fed to the Arduino. The electrical characteristics for LM393 amplifier are listed in Table H2.

Table H 2: Electrical characteristics of LMXXX amplifier

Characteristic	Symbol	LM293, LM393, LM393E			LM2903/E/V, NCV2903, NCV2903V			Unit
		Min	Typ	Max	Min	Typ	Max	
Input Offset Voltage (Note 6) $T_A = 25^\circ\text{C}$ $T_{\text{low}} \leq T_A \leq T_{\text{high}}$	V_{IO}	-	± 1.0	± 5.0 ± 9.0	-	± 2.0 ± 9.0	± 7.0 ± 15	mV
Input Offset Current $T_A = 25^\circ\text{C}$ $T_{\text{low}} \leq T_A \leq T_{\text{high}}$	I_{IO}	-	± 5.0	± 50 ± 150	-	± 5.0 ± 50	± 50 ± 200	nA
Input Bias Current (Note 7) $T_A = 25^\circ\text{C}$ $T_{\text{low}} \leq T_A \leq T_{\text{high}}$	I_{IB}	-	20	250 400	-	20 20	250 500	nA
Input Common Mode Voltage Range (Note 7) $T_A = 25^\circ\text{C}$ $T_{\text{low}} \leq T_A \leq T_{\text{high}}$	V_{ICR}	0 0	- -	$V_{CC} - 1.5$ $V_{CC} - 2.0$	0 0	- -	$V_{CC} - 1.5$ $V_{CC} - 2.0$	V
Voltage Gain $R_L \geq 15\text{ k}\Omega$, $V_{CC} = 15\text{ Vdc}$, $T_A = 25^\circ\text{C}$	A_{VOL}	50	200	-	25	200	-	V/mV
Large Signal Response Time $V_{in} = \text{TTL Logic Swing}$, $V_{ref} = 1.4\text{ Vdc}$ $V_{RL} = 5.0\text{ Vdc}$, $R_L = 5.1\text{ k}\Omega$, $T_A = 25^\circ\text{C}$	-	-	300	-	-	300	-	ns
Response Time (Note 9) $V_{RL} = 5.0\text{ Vdc}$, $R_L = 5.1\text{ k}\Omega$, $T_A = 25^\circ\text{C}$	t_{TLH}	-	1.3	-	-	1.5	-	μs
Input Differential Voltage (Note 10) All $V_{in} \geq \text{GND}$ or V^- Supply (if used)	V_{ID}	-	-	V_{CC}	-	-	V_{CC}	V
Output Sink Current $V_{in} \geq 1.0\text{ Vdc}$, $V_{in+} = 0\text{ Vdc}$, $V_O \leq 1.5\text{ Vdc}$, $T_A = 25^\circ\text{C}$	I_{SINK}	6.0	16	-	6.0	16	-	mA
Output Saturation Voltage $V_{in} \geq 1.0\text{ Vdc}$, $V_{in+} = 0$, $I_{SINK} \leq 4.0\text{ mA}$, $T_A = 25^\circ\text{C}$ $T_{\text{low}} \leq T_A \leq T_{\text{high}}$	V_{OL}	- -	150 -	400 700	- -	- 200	400 700	mV
Output Leakage Current $V_{in-} = 0\text{ V}$, $V_{in+} \geq 1.0\text{ Vdc}$, $V_O = 5.0\text{ Vdc}$, $T_A = 25^\circ\text{C}$ $V_{in-} = 0\text{ V}$, $V_{in+} \geq 1.0\text{ Vdc}$, $V_O = 30\text{ Vdc}$, $T_{\text{low}} \leq T_A \leq T_{\text{high}}$	I_{OL}	- -	0.1 -	- 1000	- -	0.1 -	- 1000	nA
Supply Current $R_L = \infty$ Both Comparators, $T_A = 25^\circ\text{C}$ $R_L = \infty$ Both Comparators, $V_{CC} = 30\text{ V}$	I_{CC}	- -	0.4 -	1.0 2.5	- -	0.4 -	1.0 2.5	mA

Product parametric performance is indicated in the Electrical Characteristics for the listed test conditions, unless otherwise noted. Product performance may not be indicated by the Electrical Characteristics if operated under different conditions.

LM293 $T_{\text{low}} = -25^\circ\text{C}$, $T_{\text{high}} = +85^\circ\text{C}$

LM393, LM393E $T_{\text{low}} = 0^\circ\text{C}$, $T_{\text{high}} = +70^\circ\text{C}$

LM2903, LM2903E $T_{\text{low}} = -40^\circ\text{C}$, $T_{\text{high}} = +105^\circ\text{C}$

LM2903V & NCV2903 $T_{\text{low}} = -40^\circ\text{C}$, $T_{\text{high}} = +125^\circ\text{C}$

NCV2903V $T_{\text{low}} = -40^\circ\text{C}$, $T_{\text{high}} = +150^\circ\text{C}$

NCV2903 and NCV2903V are qualified for automotive use.

- The maximum output current may be as high as 20 mA, independent of the magnitude of V_{CC} , output short circuits to V_{CC} can cause excessive heating and eventual destruction.
- At output switch point, $V_O = 1.4\text{ Vdc}$, $R_\theta = 0\ \Omega$ with V_{CC} from 5.0 Vdc to 30 Vdc, and over the full input common mode range (0 V to $V_{CC} - 1.5\text{ V}$).
- Due to the PNP transistor inputs, bias current will flow out of the inputs. This current is essentially constant, independent of the output state, therefore, no loading changes will exist on the input lines.
- Input common mode of either input should not be permitted to go more than 0.3 V negative of ground or minus supply. The upper limit of common mode range is $V_{CC} - 1.5\text{ V}$.
- Response time is specified with a 100 mV step and 5.0 mV of overdrive. With larger magnitudes of overdrive faster response times are obtainable.
- The comparator will exhibit proper output state if one of the inputs becomes greater than V_{CC} , the other input must remain within the common mode range. The low input state must not be less than -0.3 V of ground or minus supply.

The Potentiometer

The function of the rotary potentiometer (variable resistor) labelled on Figure H1 is to adjust the sensitivity of the sensor to the environment and thereby varying the detection distance. Its specifications are listed in Table H3.

Table H 3: Specifications for 10 k Ω Potentiometer

Tolerance	
$100\ \Omega \leq R_n \leq 1\ M\Omega$	$\pm 20\%$
$1\ M\Omega \leq R_n \leq 5\ M\Omega$	$\pm 30\%$
Maximum voltage	200 VDC (linear)
Nominal Power at 50 °C	0.15 W (linear)
Residual resistance	$\leq 5 \times 10^{-3} R_n$ (2 Ω minimum)
Equivalent Noise Resistance	$\leq 3\ \% R_n$ (3 Ω minimum)
Operating Temperature	-25 °C to 70 °C
Mechanical rotational angle	235 $\pm$ 5 °
Electrical rotational angle	220 $\pm$ 20 °
Torque	0.4 to 2 Ncm
Stop torque	>5 Ncm

Appendix I: Arduino codes for speed measurement

Two Arduino codes were written: the first one is for reading the digital output values for the sensor and the other codes is the one for calculating the speed of the shafts for the discs.

Arduino code for Reading the digital outputs of the IR sensor

```
/*This simple program uses IR sensor to read the digital value of output voltage
```

```
* for the spinning discs of the ROC
```

```
* Data are saved in PLX DAQ Spreadsheet in real time
```

```
*/
```

```
int IRSensor = 2;
```

```
unsigned long milli_time;
```

```
void setup()
```

```
{
```

```
Serial.begin(128000); //initialise the serial communication
```

```
Serial.println("CLEARDATA");
```

```
Serial.println("LABEL, Computer Time, Time (ms), Vout");
```

```
Serial.println("RESENTTIMER");
```

```
pinMode(IRSensor, INPUT);
```

```
}
```

```
void loop()
```

```
{
```

```
milli_time=millis();
```

```
float Vout=digitalRead(2);
```

```
Serial.print("DATA, TIME,");
```

```
Serial.print(milli_time);
```

```
Serial.print(",");
```

```
Serial.println(Vout);
```

```
}
```

Arduino code for RPM calculations data displayed on serial monitor

```
int IRSensor=2; //IR sensor INPUT

volatile long oldtime; //To store time

volatile long timeperiod; //To store time period for the silver patches


int rpm; //RPM value


volatile boolean currentstate;//current state of IR input scan
volatile boolean prevstate;//State of IR sensor in previous scan


void setup() {
  Serial.begin(2000000);
  pinMode(IRSensor, INPUT);
  oldtime=0;
  prevstate=HIGH;

}


void loop() {
  //RPM Calculation
  currentstate=digitalRead(IRSensor); //Read IR sensor state
  if(prevstate!=currentstate)//If there is a change in input (from 1 to 0)
  {
    if(currentstate==LOW) //Comparing right and left sides with the answer being true or false
    {timeperiod=(micros()-oldtime); //Time period in microseconds when the condition (currentstate==LOW) is true
      rpm=(60000000/timeperiod/4);//rpm=(1/duration)*1000*1000*60/(number of silver patches)
      oldtime=micros();
    }
  }
  prevstate=currentstate;//store this previous scan data for next scan
  {
    Serial.println(rpm);
  }
}
```

Arduino code for RPM calculations data displayed on LCD screen

```
#include<LiquidCrystal.h>

LiquidCrystal lcd(12,11,6,5,4,3);

float IRsensor=2;

int rpm;

int oldtime=0;

int timeperiod;

volatile boolean currentstate;//current state of IR input scan

volatile boolean prevstate;//State of IR sensor in previous scan


void setup()
{
  lcd.begin(16,2);      //initialize LCD Display
  pinMode(IRsensor, INPUT) ;
  prevstate=HIGH;
}

void loop() {
  delay (1000);
  currentstate=digitalRead(IRsensor); //Read IR sensor state
  if(prevstate!=currentstate)//If there is a change in input (from 1 to 0)
  {
    if(currentstate==LOW) //Comparing right and left sides with the answer being true or false
    {timeperiod=(micros()-oldtime); //Time period in microseconds when the condition (currentstate==LOW) is true
      rpm=(60000000/timeperiod/4);//rpm=(1/duration)*1000*1000*60/(number of silver patches)
    }
  }
  oldtime=micros();
  prevstate=currentstate;
  lcd.clear();
  lcd.setCursor(0,0);
  lcd.print("___ TACHOMETER ___");
  lcd.setCursor(0,1);
  lcd.print(  rpm);
  lcd.print(" RPM");
  lcd.print(" ");
}
```

Arduino code for RPM calculations data saved in SD card

```

#include <SPI.h>;

#include <SD.h>;

const int chipSelect=10;


int IRsensor=2; //IR sensor INPUT

volatile long prevtime; //To store time from previous scan

volatile long timeperiod; //To store time period for the white patches


int rpm; //RPM value


volatile boolean currentstate;//current state of IR input scan

volatile boolean prevstate;//State of IR sensor in previous scan


File dataFile;


void setup() {
  Serial.begin(2000000);
  pinMode(IRsensor, INPUT);
  prevtime=0;
  prevstate=HIGH;


  while (!Serial)
  {
    ;
  }

  Serial.println ("Initialising SD card");

  //see if the card is present and can be initialised

```

```

if (!SD.begin(chipSelect))
{
    Serial.println("Card failed to initialise");
    while(1);
}
Serial.println("Card initialised");
}

void loop() {

//make a string for assembling data to log
String dataString="";

//open the file to write on it
File dataFile=SD.open("textFile.txt", FILE_WRITE);

//if file is available, write to it
if (dataFile)
{
    dataFile.println(dataString);
    dataFile.close();
}
else
//if the file is not open,pop up an error
Serial.println ("error opening datalog.txt");

//RPM Calculation
currentstate=digitalRead(IRsensor); //Read IR sensor state
if(prevstate!=currentstate)//If there is a change in input (from 1 to 0)

```

```

    { if(currentstate==LOW) //Comparing right and left sides with the answer being true or
false

    timeperiod=(micros()-prevtime); //Time period in microseconds when the condition
(currentstate==LOW) is true

    rpm=(60000000/timeperiod/4);//rpm=(1/duration)*1000*1000*60/(number of white
patches)

    prevtime=micros();

    prevstate=currentstate;//store this previous scan data for next scan

    dataFile.println(rpm);

    }

}

```

Appendix J: Zemic load cell

The 50 kg load cell used for load measurement (both for the feeder and motor drive) is shown in Figure I1 and its characteristics are listed in Table I1.



Figure J 1: The 50 kg Zemic load cell used for load measurement

Table J 1: Operating data for the 50 kg Zemic load cell

Output sensitivity (FS)	mV/V	3.0 ± 0.008
Combined errors	% of FS	≤ 0.020
Minimum dead load	% of $^aE_{\max}$	0
Safe overload	% of $E_{\max}$	150
Ultimate overload	% of $E_{\max}$	300
Zero balance	% of FS	
Excitation voltage	V	5 – 12
Operating temperature	°C	-35 ~ +65
Element material		Nickel plated alloy steel

$^aE_{\max}$ is the rated capacity (50 kg)

Appendix K: The 24 Bit High precision Analog to Digital Converter

This amplifier is commonly used in electronic scales. It is best coupled with the load cell to improve the very small voltage output (in mV) of the load cell to high voltage values (in volts between 0 to 5 V) that the Arduino can handle. The HX711 board comes in three parts (board, 6 female pins header and 4 female pins header) as depicted in Figure K1 below. Descriptions of the pins for the HX711 chip shown in Figure K2 are listed in Table K1 while the electrical characteristics for the HX711 boards are listed in Table K2.

The HX711 amplifier has a gain factor of 178. This amplifier has interfaces that can easily be connected to the load cell. The load cell has four interfaces: a red wire for the power input to the load cell, the black or earth wire, as well as the green and white wires for signal outputs. These four wires are connected to their corresponding wires on the amplifier board (see Figure 4.5). The amplifier is then powered via the power pins (V_{cc} and GND). Serial communication between the HX711 and Arduino board is done using SCK (A0 and A2 analog pins on Arduino boards) and DOUT pins (analog pins A0 and A3 on Arduino UNO board). SCK and DOUT communication is also possible with the digital pins. The Arduino UNO, with a loaded program, outputs the analog values corresponding to the exerted load. The data is saved in real time in the Microsoft Excel® with the help of the data acquisition tool (PLX DAQ). However, the magnitude of the force can only be accurately calculated if the calibration curve relating the output voltage and the force is drawn. The calibration process and results are discussed in section 4.3.

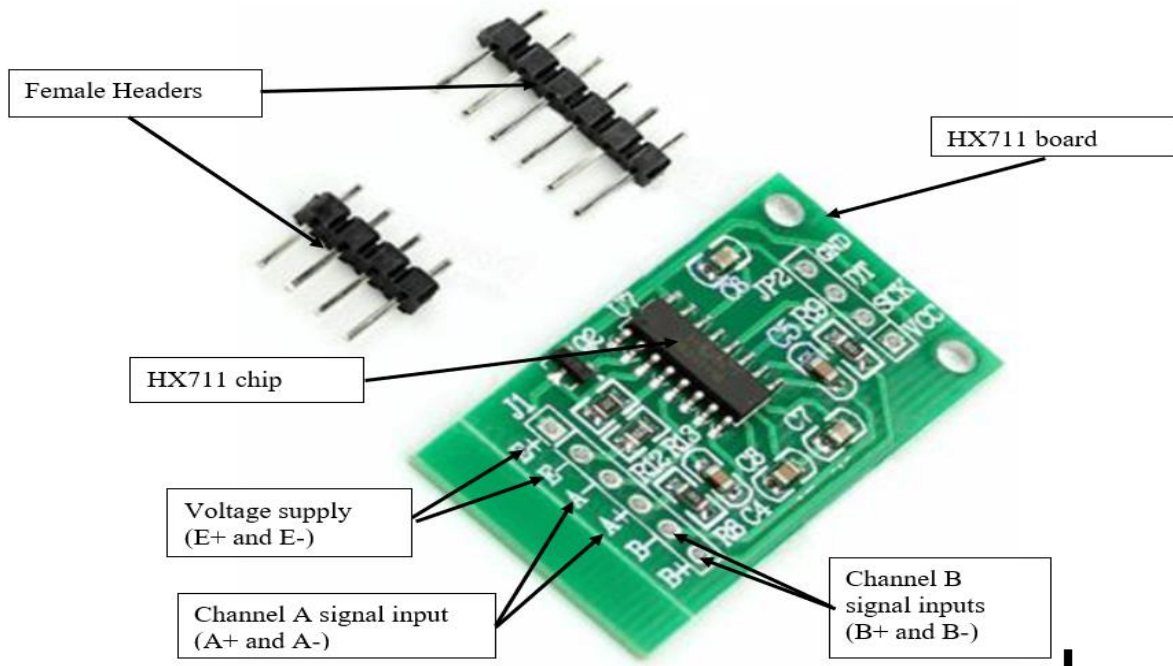


Figure K 1: The photographs of the 24 Bit high precision A/D converter (bridge amplifier) and the female headers for connection

Regulator Power	VSUP	1	16	DVDD	Digital Power
Regulator Control Output	BASE	2	15	RATE	Output Data Rate Control Input
Analog Power	AVDD	3	14	XI	Crystal I/O and External Clock Input
Regulator Control Input	VFB	4	13	XO	Crystal I/O
Analog Ground	AGND	5	12	DOUT	Serial Data Output
Reference Bypass	VBG	6	11	PD_SCK	Power Down and Serial Clock Input
Ch. A Negative Input	INNA	7	10	INPB	Ch. B Positive Input
Ch. A Positive Input	INPA	8	9	INNB	Ch. B Negative Input

Figure K 2: The Schematic diagram for the HX711 chip

Table K 1: Descriptions of pins for the HX711 chip

Pin #	Name	Function	Description
1	VSUP	Power	Regulator supply: 2.7 ~ 5.5V
2	BASE	Analog Output	Regulator control output (NC when not used)
3	AVDD	Power	Analog supply: 2.6 ~ 5.5V
4	VFB	Analog Input	Regulator control input (connect to AGND when not used)
5	AGND	Ground	Analog Ground
6	VBG	Analog Output	Reference bypass output
7	INA-	Analog Input	Channel A negative input
8	INA+	Analog Input	Channel A positive input
9	INB-	Analog Input	Channel B negative input
10	INB+	Analog Input	Channel B positive input
11	PD_SCK	Digital Input	Power down control (high active) and serial clock input
12	DOUT	Digital Output	Serial data output
13	XO	Digital I/O	Crystal I/O (NC when not used)
14	XI	Digital Input	Crystal I/O or external clock input, 0: use on-chip oscillator
15	RATE	Digital Input	Output data rate control, 0: 10Hz; 1: 80Hz
16	DVDD	Power	Digital supply: 2.6 ~ 5.5V

Table K 2: Key electrical characteristics for the HX711 boards

Parameter	Notes	MIN	TYP	MAX	UNIT
Full scale differential input range	$V(\text{inp}) - V(\text{inn})$	$\pm 0.5(AVDD/GAIN)$			V
Common mode input		AGND+1.2		AVDD-1.3	V
Output data rate	Internal Oscillator, RATE = 0		10		Hz
	Internal Oscillator, RATE = DVDD		80		
	Crystal or external clock, RATE = 0		$f_{clk}/1,105,920$		
	Crystal or external clock, RATE = DVDD		$f_{clk}/138,240$		
Output data coding	2's complement	800000		7FFFFFFF	HEX
Output settling time ⁽¹⁾	RATE = 0		400		ms
	RATE = DVDD		50		
Input offset drift	Gain = 128		0.2		mV
	Gain = 64		0.4		
Input noise	Gain = 128, RATE = 0		50		nV(rms)
	Gain = 128, RATE = DVDD		90		
Temperature drift	Input offset (Gain = 128)		±6		nV/°C
	Gain (Gain = 128)		±5		
Input common mode rejection	Gain = 128, RATE = 0		100		dB
Power supply rejection	Gain = 128, RATE = 0		100		dB
Reference bypass (V _{BG})			1.25		V
Crystal or external clock frequency		1	11.0592	20	MHz
Power supply voltage	DVDD	2.6		5.5	V
	AVDD, VSUP	2.6		5.5	
Analog supply current (including regulator)	Normal		1400		μA
	Power down		0.3		
Digital supply current	Normal		100		μA
	Power down		0.2		

(1) Settling time refers to the time from power up, reset, input channel change and gain change to valid stable output data.

Appendix L: The L7805 voltage regulator

Retrieved from <http://www.mantech.co.za/Datasheets/Products/L78XX.pdf>

To supply 5 V to Arduino UNO and IR sensor, the voltage regulator, shown in Figure L1, was used. Its electrical characteristics are presented in Table L1.

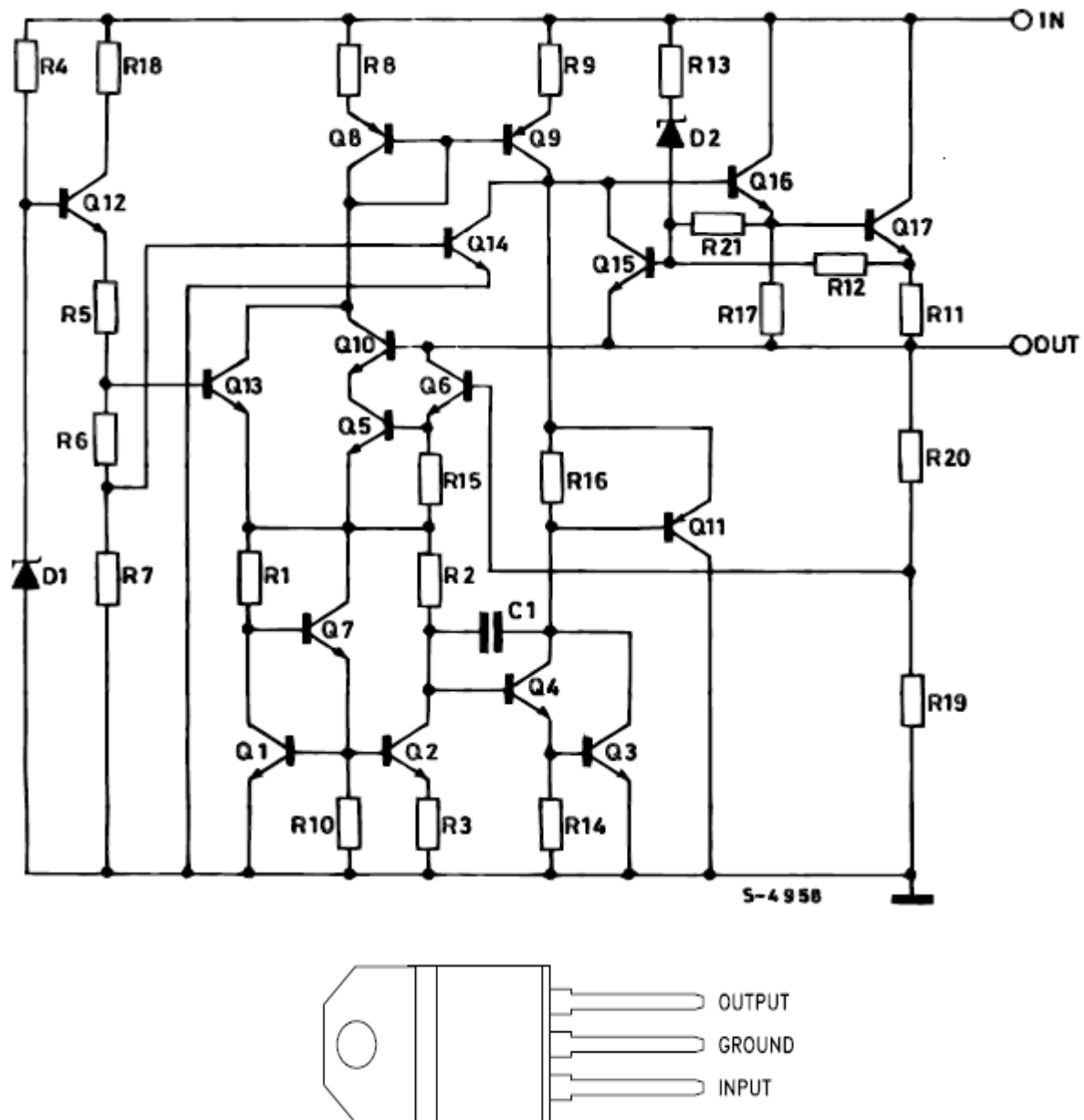


Figure L 1: Circuit diagram (above) and schematic diagram (below) of L7805 voltage regulator ("Mantech Electronics⁵", n.d.)

Table L 1: Electrical characteristics of the L7805 voltage regulator (refer to test circuits; $T_J = 0$ to 125°C , $V_I = 10\text{ V}$, $I_O = 500\text{ mA}$, $C_1 = 0.33\mu\text{F}$, $C_o = 0.1\mu\text{F}$ unless otherwise specified).

Symbol	Parameter	Test Conditions	Min.	Typ.	Max.	Unit
V_O	Output Voltage	$T_J = 25^\circ\text{C}$	4.8	5	5.2	V
V_O	Output Voltage	$I_O = 5\text{ mA to }1\text{ A}$ $P_O \leq 15\text{ W}$ $V_I = 7\text{ to }20\text{ V}$	4.75	5	5.25	V
$\Delta V_O(^*)$	Line Regulation	$V_I = 7\text{ to }25\text{ V}$ $T_J = 25^\circ\text{C}$		3	100	mV
		$V_I = 8\text{ to }12\text{ V}$ $T_J = 25^\circ\text{C}$		1	50	
$\Delta V_O(^*)$	Load Regulation	$I_O = 5\text{ mA to }1.5\text{ A}$ $T_J = 25^\circ\text{C}$			100	mV
		$I_O = 250\text{ to }750\text{ mA}$ $T_J = 25^\circ\text{C}$			50	
I_d	Quiescent Current	$T_J = 25^\circ\text{C}$			8	mA
ΔI_d	Quiescent Current Change	$I_O = 5\text{ mA to }1\text{ A}$			0.5	mA
		$V_I = 7\text{ to }25\text{ V}$			0.8	
$\Delta V_O/\Delta T$	Output Voltage Drift	$I_O = 5\text{ mA}$		-1.1		mV/ $^\circ\text{C}$
eN	Output Noise Voltage	$B = 10\text{ Hz to }100\text{ KHz}$ $T_J = 25^\circ\text{C}$		40		$\mu\text{V}/V_O$
SVR	Supply Voltage Rejection	$V_I = 8\text{ to }18\text{ V}$ $f = 120\text{ Hz}$	62			dB
V_d	Dropout Voltage	$I_O = 1\text{ A}$ $T_J = 25^\circ\text{C}$		2		V
R_O	Output Resistance	$f = 1\text{ KHz}$		17		m Ω
I_{sc}	Short Circuit Current	$V_I = 35\text{ V}$ $T_J = 25^\circ\text{C}$		0.75		A
I_{scp}	Short Circuit Peak Current	$T_J = 25^\circ\text{C}$		2.2		A

(*) Load and line regulation are specified at constant junction temperature. Changes in V_O due to heating effects must be taken into account separately. Pulse testing with low duty cycle is used.

Appendix M: Calibration of the load cells and Arduino code

Calibration was done by placing the known mass on the conveyor belt (mass known) for the feeder load cell (as shown in Figure M1) and in the pre-weighed bucket for the motor drive load cell (as shown in Figure M2). The calibration data for the load cell of the feeder are listed in Table M1 while those for the motor drive torque are listed in Table M2. The Arduino code for calculating the loads (weight of feeder and sample and torque of the driving pulley) is also included in this Appendix.



Figure M 1: A conveyor belt with limestone sample (evenly spread) during calibration

Table M 1: Calibration data for load cell of the feeder

Description	Mass (kg)	Analog output
Empty conveyor belt	32	10990194
Belt+1kg	33	11079370
Belt+2kg	34	11169257
Belt+4kg	36	11347815
Belt+6kg	38	11513535

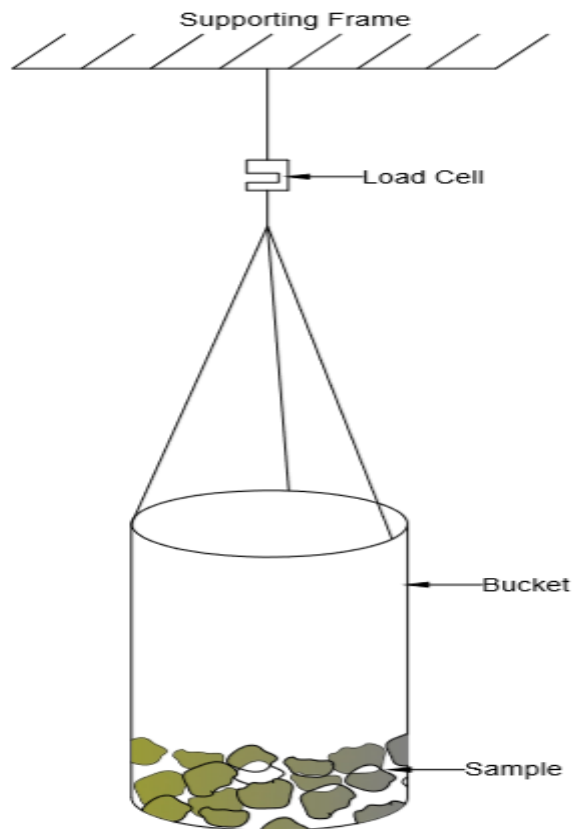


Figure M 2: Schematic showing the calibration of the load cell

Table M 2: Calibration data for load cell of the motor drive torque

Description	Load Mass, g	Calculated Force, N	Analog output
No load	0	0.00	6490738
Empty bucket	186	1.82	6509336
Bucket+2kg limestone	2186	21.44	6714334
Bucket+4kg limestone	4186	41.06	6915401
Bucket+6kg limestone	6186	60.68	7118660

Arduino code for load measurement (both for the feeder and motor drive torque)

```
#include <Q2HX711.h>

const byte hx711_data_pin = A1;
const byte hx711_clock_pin = A0;
Q2HX711 hx711f(A1, A0);

float lc_f;

const byte hx711_data_pin1 = A3;
const byte hx711_clock_pin1 = A2;
Q2HX711 hx711m(A3, A2);

float lc_m;

unsigned long milli_time;

void setup()
{
  Serial.begin(128000);
  Serial.println("CLEARDATA");
  Serial.println("LABEL, Computer Time, Time (ms), lc_f, lc_m");
  Serial.println("RESENTTIMER");

}

void loop()
{
  milli_time=millis();
  lc_f=hx711f.read();
  lc_m=hx711m.read();
  Serial.print("DATA, TIME,");
  Serial.print(milli_time);
  Serial.print(",");
  Serial.print(lc_f);
  Serial.print(",");
  Serial.println(lc_m);

}
```

Appendix N: Modelling data for Change in Geometry of the crushing chamber

X offset	mm	15		20		25	
Exit gap	mm	3		3		3	
Input gap	mm	23		23		23	
Angle		Input gap (mm)	Exit gap (mm)	Input gap (mm)	Exit gap (mm)	Input gap (mm)	Exit gap (mm)
Degrees	Radians						
0	0.00	23.417	3.060	23.833	3.120	24.250	3.180
5	0.09	23.415	3.060	23.830	3.120	24.245	3.179
10	0.17	23.410	3.059	23.821	3.118	24.231	3.177
15	0.26	23.402	3.058	23.805	3.116	24.207	3.174
20	0.35	23.392	3.056	23.783	3.113	24.175	3.169
25	0.44	23.378	3.054	23.755	3.109	24.133	3.163
30	0.52	23.361	3.052	23.722	3.104	24.083	3.156
35	0.61	23.341	3.049	23.683	3.098	24.024	3.147
40	0.70	23.319	3.046	23.638	3.092	23.958	3.138
45	0.79	23.295	3.042	23.589	3.085	23.884	3.127
50	0.87	23.268	3.039	23.536	3.077	23.803	3.116
55	0.96	23.239	3.034	23.478	3.069	23.717	3.103
60	1.05	23.208	3.030	23.417	3.060	23.625	3.090
65	1.13	23.176	3.025	23.352	3.051	23.528	3.076
70	1.22	23.143	3.021	23.285	3.041	23.428	3.062
75	1.31	23.108	3.016	23.216	3.031	23.324	3.047
80	1.40	23.072	3.010	23.145	3.021	23.217	3.031
85	1.48	23.036	3.005	23.073	3.010	23.109	3.016
90	1.57	23.000	3.000	23.000	3.000	23.000	3.000
95	1.66	22.964	2.995	22.927	2.990	22.891	2.984
100	1.75	22.928	2.990	22.855	2.979	22.783	2.969
105	1.83	22.892	2.984	22.784	2.969	22.676	2.953
110	1.92	22.857	2.979	22.715	2.959	22.572	2.938
115	2.01	22.824	2.975	22.648	2.949	22.472	2.924
120	2.09	22.792	2.970	22.583	2.940	22.375	2.910
125	2.18	22.761	2.966	22.522	2.931	22.283	2.897
130	2.27	22.732	2.961	22.464	2.923	22.197	2.884
135	2.36	22.705	2.958	22.411	2.915	22.116	2.873
140	2.44	22.681	2.954	22.362	2.908	22.042	2.862
145	2.53	22.659	2.951	22.317	2.902	21.976	2.853
150	2.62	22.639	2.948	22.278	2.896	21.917	2.844
155	2.71	22.622	2.946	22.245	2.891	21.867	2.837
160	2.79	22.608	2.944	22.217	2.887	21.825	2.831

165	2.88	22.598	2.942	22.195	2.884	21.793	2.826
170	2.97	22.590	2.941	22.179	2.882	21.769	2.823
175	3.05	22.585	2.940	22.170	2.880	21.755	2.821
180	3.14	22.583	2.940	22.167	2.880	21.750	2.820
185	3.23	22.585	2.940	22.170	2.880	21.755	2.821
190	3.32	22.590	2.941	22.179	2.882	21.769	2.823
195	3.40	22.598	2.942	22.195	2.884	21.793	2.826
200	3.49	22.608	2.944	22.217	2.887	21.825	2.831
205	3.58	22.622	2.946	22.245	2.891	21.867	2.837
210	3.67	22.639	2.948	22.278	2.896	21.917	2.844
215	3.75	22.659	2.951	22.317	2.902	21.976	2.853
220	3.84	22.681	2.954	22.362	2.908	22.042	2.862
225	3.93	22.705	2.958	22.411	2.915	22.116	2.873
230	4.01	22.732	2.961	22.464	2.923	22.197	2.884
235	4.10	22.761	2.966	22.522	2.931	22.283	2.897
240	4.19	22.792	2.970	22.583	2.940	22.375	2.910
245	4.28	22.824	2.975	22.648	2.949	22.472	2.924
250	4.36	22.857	2.979	22.715	2.959	22.572	2.938
255	4.45	22.892	2.984	22.784	2.969	22.676	2.953
260	4.54	22.928	2.990	22.855	2.979	22.783	2.969
265	4.63	22.964	2.995	22.927	2.990	22.891	2.984
270	4.71	23.000	3.000	23.000	3.000	23.000	3.000
275	4.80	23.036	3.005	23.073	3.010	23.109	3.016
280	4.89	23.072	3.010	23.145	3.021	23.217	3.031
285	4.97	23.108	3.016	23.216	3.031	23.324	3.047
290	5.06	23.143	3.021	23.285	3.041	23.428	3.062
295	5.15	23.176	3.025	23.352	3.051	23.528	3.076
300	5.24	23.208	3.030	23.417	3.060	23.625	3.090
305	5.32	23.239	3.034	23.478	3.069	23.717	3.103
310	5.41	23.268	3.039	23.536	3.077	23.803	3.116
315	5.50	23.295	3.042	23.589	3.085	23.884	3.127
320	5.59	23.319	3.046	23.638	3.092	23.958	3.138
325	5.67	23.341	3.049	23.683	3.098	24.024	3.147
330	5.76	23.361	3.052	23.722	3.104	24.083	3.156
335	5.85	23.378	3.054	23.755	3.109	24.133	3.163
340	5.93	23.392	3.056	23.783	3.113	24.175	3.169
345	6.02	23.402	3.058	23.805	3.116	24.207	3.174
350	6.11	23.410	3.059	23.821	3.118	24.231	3.177
355	6.20	23.415	3.060	23.830	3.120	24.245	3.179
360	6.28	23.417	3.060	23.833	3.120	24.250	3.180

Appendix O: Transportation modelling

Table O 1: Modelling data for centrifugal acceleration calculations

Speed	rpm	330		550	
	rad/s	34.557519		57.5958653	
g	m/s/s	9.81			
Radius -mm		a	Number of G	a	Number of G
0		0.00	0.00	0.00	0.00
1		1.19	0.12	3.32	0.34
10		11.94	1.22	33.17	3.38
20		23.88	2.43	66.35	6.76
30		35.83	3.65	99.52	10.14
40		47.77	4.87	132.69	13.53
50		59.71	6.09	165.86	16.91
60		71.65	7.30	199.04	20.29
70		83.60	8.52	232.21	23.67
80		95.54	9.74	265.38	27.05
90		107.48	10.96	298.56	30.43
100		119.42	12.17	331.73	33.82
110		131.36	13.39	364.90	37.20
120		143.31	14.61	398.07	40.58
130		155.25	15.83	431.25	43.96
140		167.19	17.04	464.42	47.34
150		179.13	18.26	497.59	50.72
160		191.08	19.48	530.77	54.10
170		203.02	20.69	563.94	57.49
180		214.96	21.91	597.11	60.87
190		226.90	23.13	630.28	64.25
200		238.84	24.35	663.46	67.63
210		250.79	25.56	696.63	71.01
220		262.73	26.78	729.80	74.39
230		274.67	28.00	762.98	77.78
240		286.61	29.22	796.15	81.16
250		298.56	30.43	829.32	84.54

Table O 2: DEM simulation data and results

Run	Ball size (mm)	Speed (rpm)	Offset (mm)	Exit gap (mm)	Mass out (kg)	Residence time (s)	Throughput (tph)
1	4	330	15	4	0.02735	1.63	0.06
2	7	330	10	7	0.0484	2.64	0.066
3	7	765	10	4	0.06776	0.88	0.277
4	10	330	15	10	0.06451	5.23	0.044
5	7	765	10	10	0.06776	0.75	0.325
6	4	330	5	4	0.02245	1.66	0.049
7	10	1200	15	4	0.10079	0.54	0.672
8	10	765	10	7	0.09273	0.72	0.464
9	7	765	5	7	0.06776	1.33	0.183
10	7	765	15	7	0.06914	1.28	0.194
11	7	765	10	7	0.06776	1.28	0.191
12	7	1200	10	7	0.07053	0.84	0.302
13	4	765	10	7	0.03896	1.08	0.13
14	7	765	10	7	0.06776	0.98	0.249
15	10	1200	5	4	0.10079	0.45	0.806
16	4	1200	15	10	0.04051	0.8	0.182
17	4	1200	5	10	0.03999	0.74	0.195
18	10	330	5	10	0.04032	1.63	0.089

Multiple Regression - Q (tph)

Table O 3: Analysis of Variance for throughput regression modelling

Source	Sum of Squares	Df	Mean Square	F-Ratio	P-Value
Model	0.663522	4	0.165881	23.86	0.0000
Residual	0.0903948	13	0.00695345		
Total (Corr.)	0.753917	17			

Appendix P: Drop weight tests data and results

Table P 1: The t_{10} fitting parameters obtained using the iteration method

Fitting t_{10} and kWh/t to model equation defining t_{10}				
Model equation:		$t_{10}=A(1-e^{-bEcs})$		
Parameters		A	b	A*B
-13.2+9.5mm:coal		34.81275304	3.64155501	
Ecs (kWh)	exp t_{10} (%)	model t_{10} (%)	Error	Error squared
0.576506	31.440	30.55	0.893	0.79734239
1.01256	31.730	33.94	-2.211	4.888755997
1.509	33.190	34.69	-1.496	2.239359239
1.543035	34.080	34.81	-0.726	0.526604148
2.33437	38.395	34.81	3.589	12.88412969
Sum of squares				21.33619147
-13.2+9.5mm:quartz		55.78608733	1.730563879	
0.3070	26.85	22.99	3.86	14.90409863
0.5337	28.75	33.63	-4.89	23.88313561
0.8677	45.13	43.36	1.77	3.131456661
1.2894	49.77	49.80	-0.02	0.000516829
Sum of squares				41.91920773
-19+13.2mm:coal		36.00334035	17.74125979	
0.230098	38.88	35.40	3.48	12.13849257
0.395635	35.38	35.97	-0.59	0.349434304
0.651384	33.53	36.00	-2.47	6.115707358
Sum of squares				36.34489402
-19+13.2mm:quartz		85.69036987	1.529795674	
0.12089	18.809756	14.47	4.34	18.84867202
0.203028	18.614634	22.88	-4.26	18.17782816
0.323517	34.553659	33.45	1.10	1.214601979
Sum of squares				39.77089784

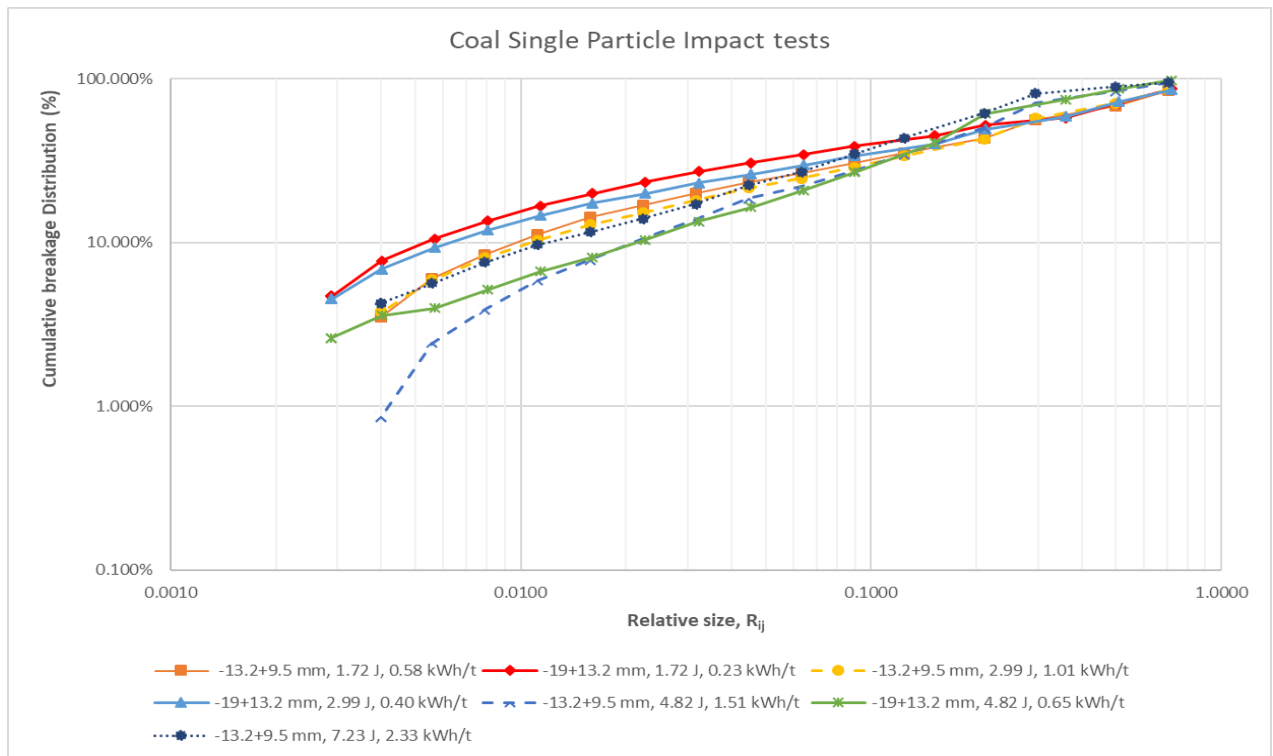


Figure P 1: Normalised cumulative size distributions for coal sample various impact energy levels

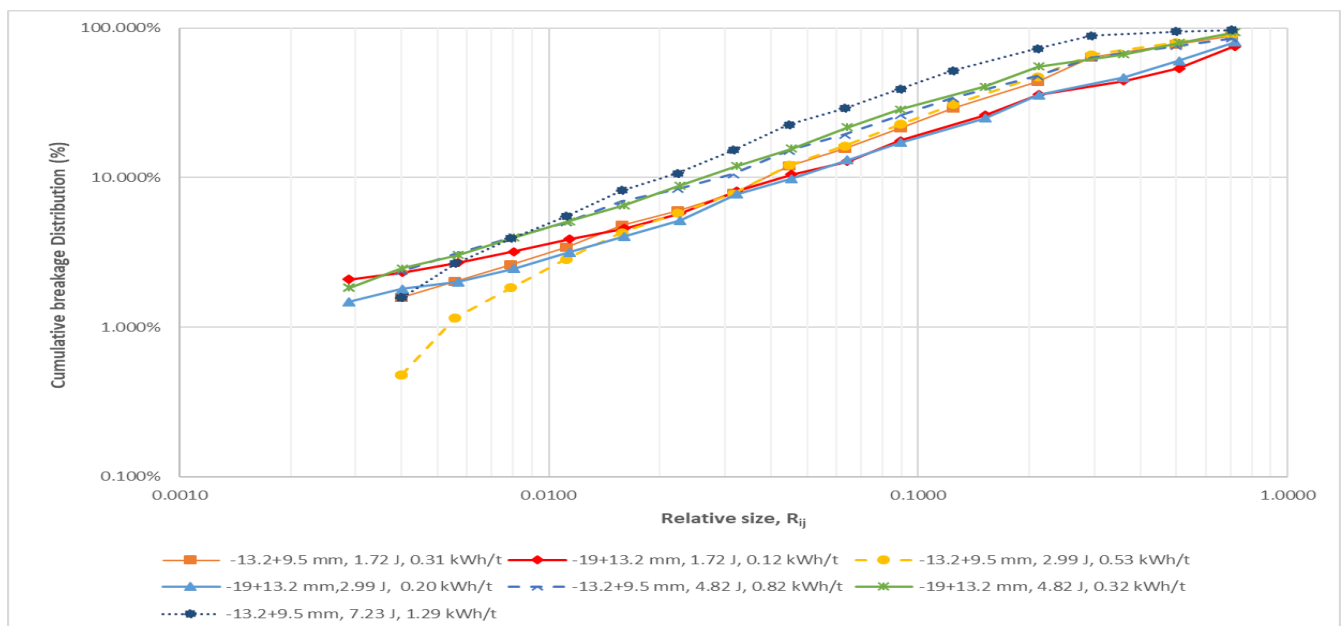


Figure P 2: Normalised cumulative size distributions for the quartz various impact energy levels

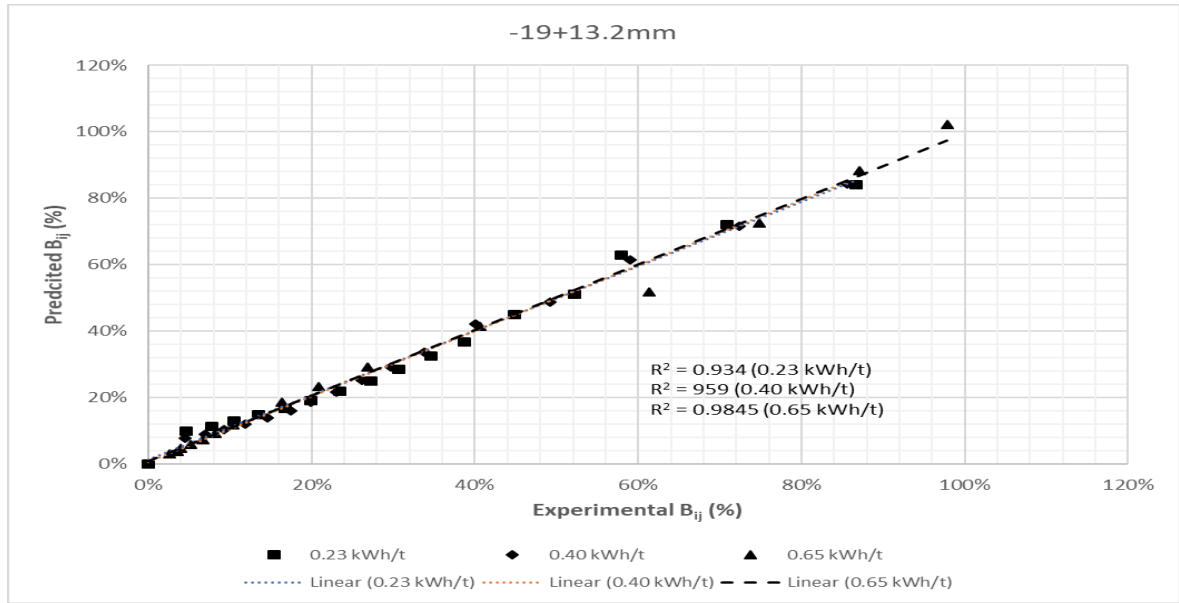


Figure P 3: Experimental versus predicted B_{ij} values for the coal in the size range of -19+13.2 mm

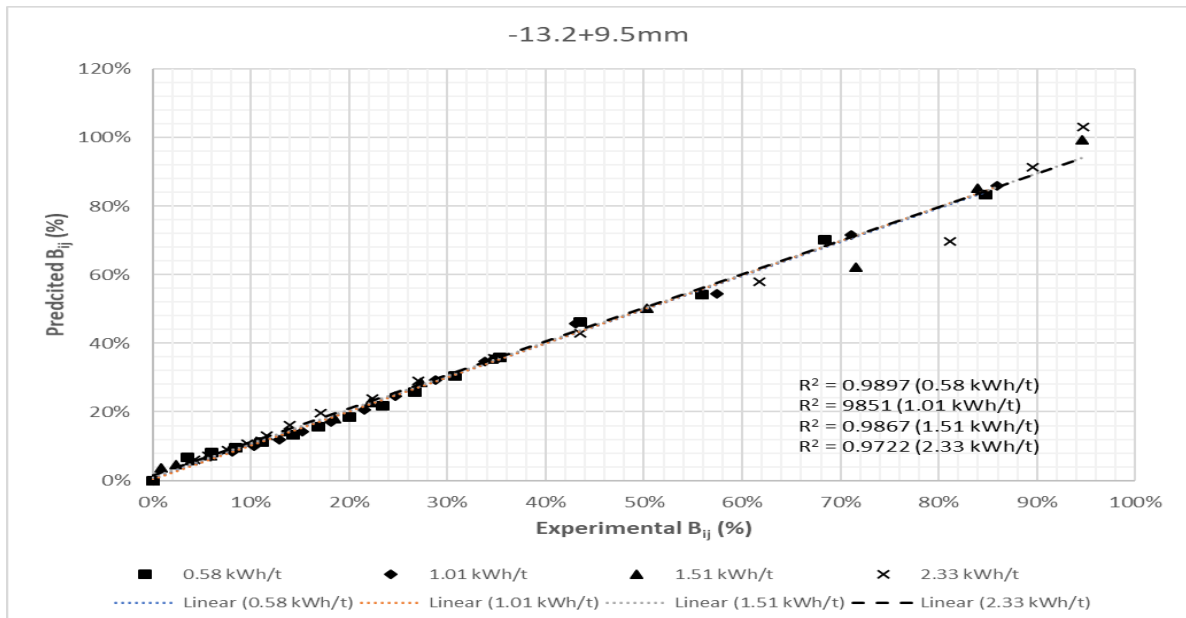


Figure P 4: Experimental versus predicted B_{ij} values for the coal in the size range of -13.2+9.5 mm

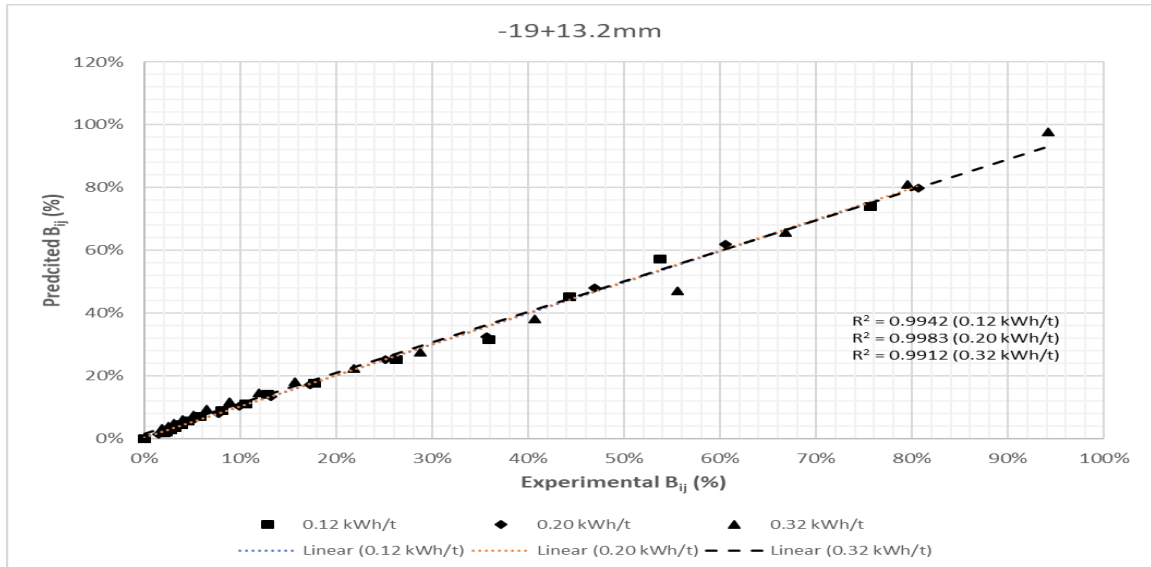


Figure P 5: Experimental versus predicted B_{ij} values for the quartz in the size range of -19+13.2 mm

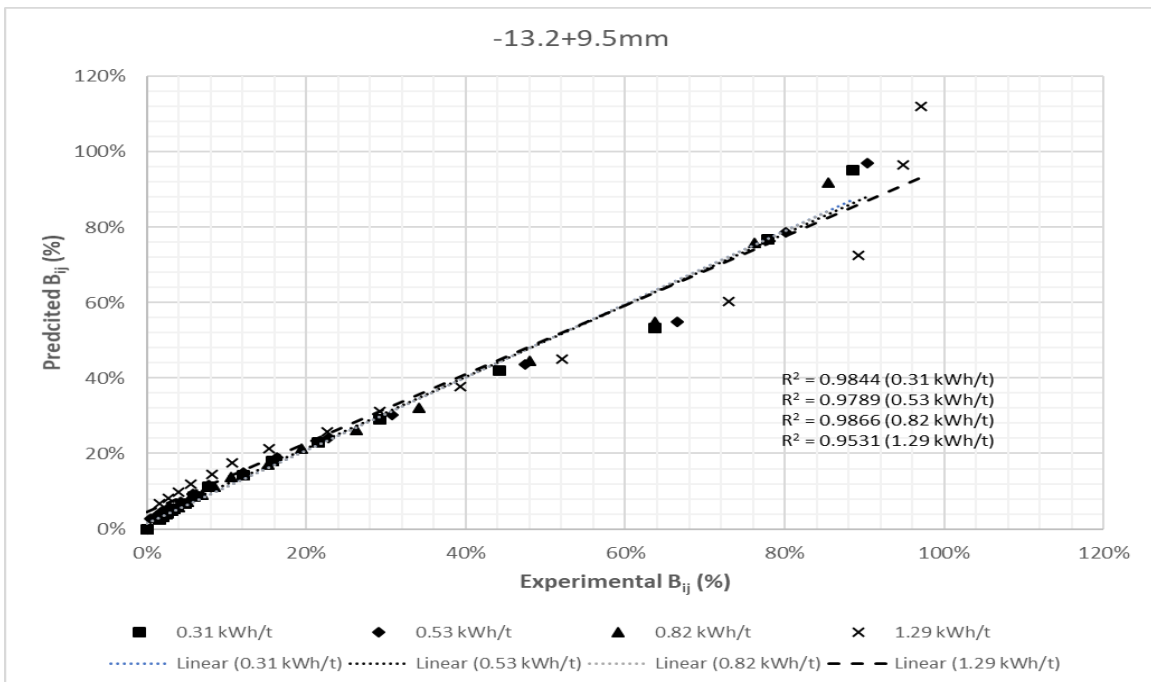


Figure P 6: Experimental versus predicted B_{ij} values for the quartz in the size range of -13.2+9.5 mm

Appendix Q: Compression tests data and results

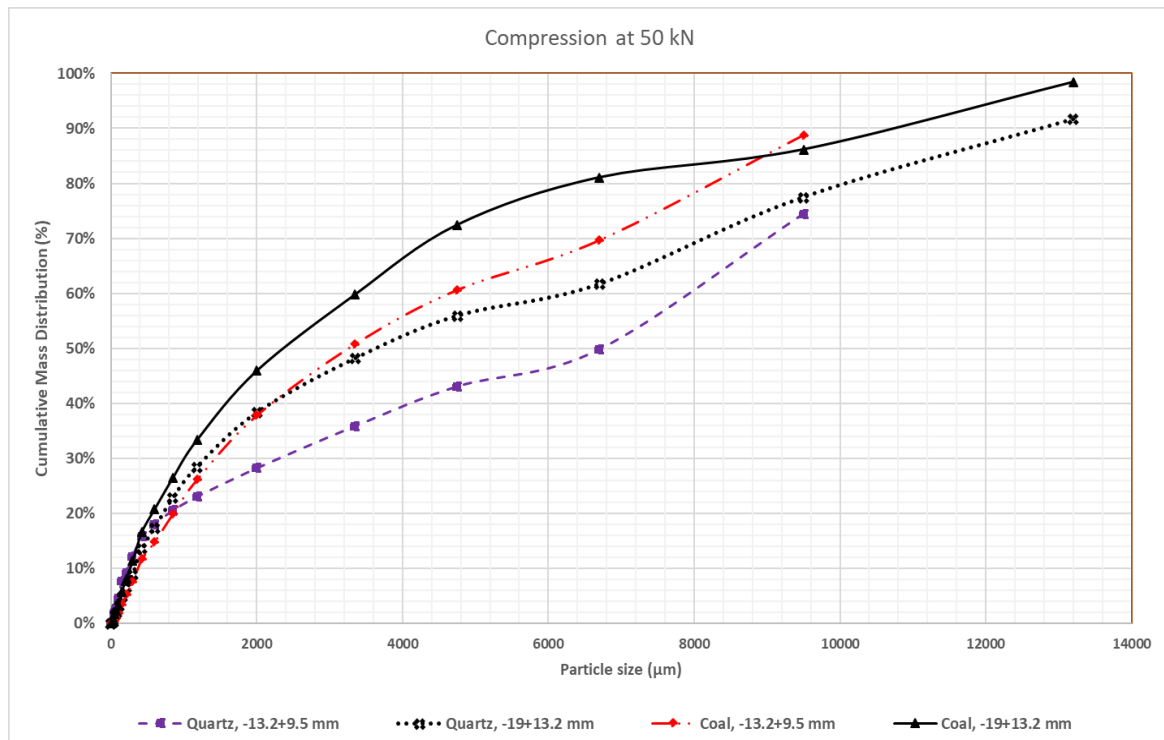


Figure Q 1: Cumulative size distributions for the compression of coal and quartz at 50 kN

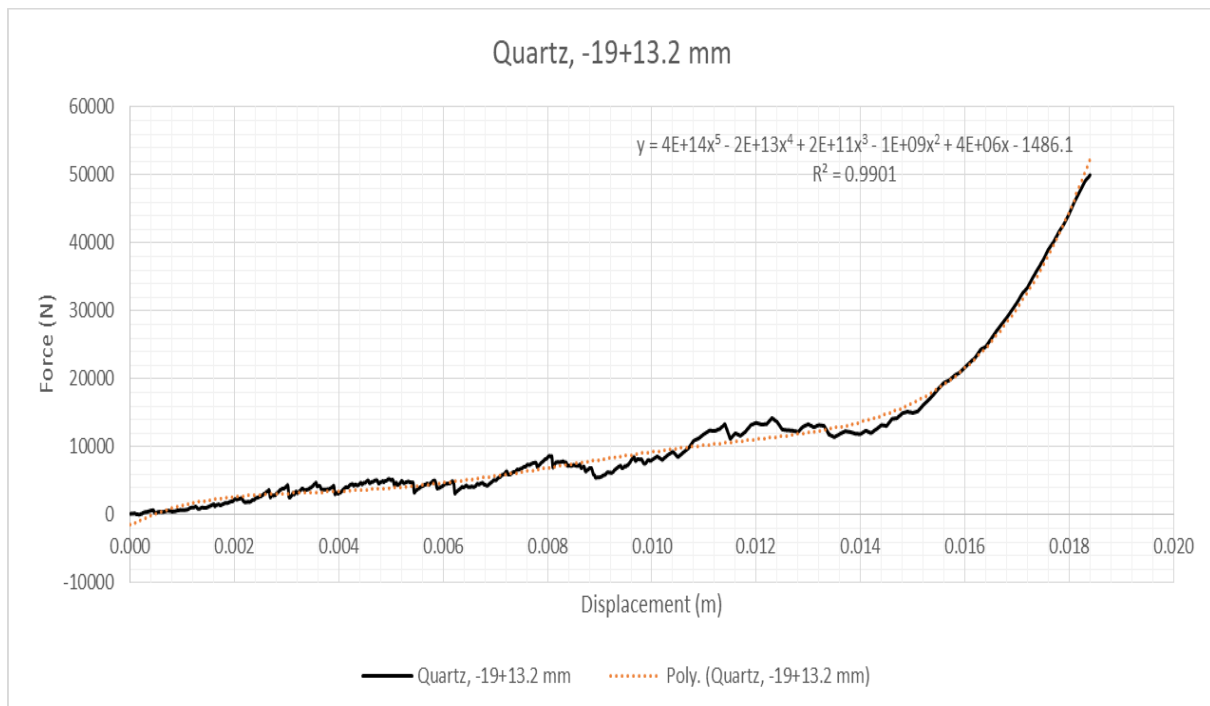


Figure Q 2: Fitting of force-displacement data for -19+13.2 mm of quartz to polynomial degree 6

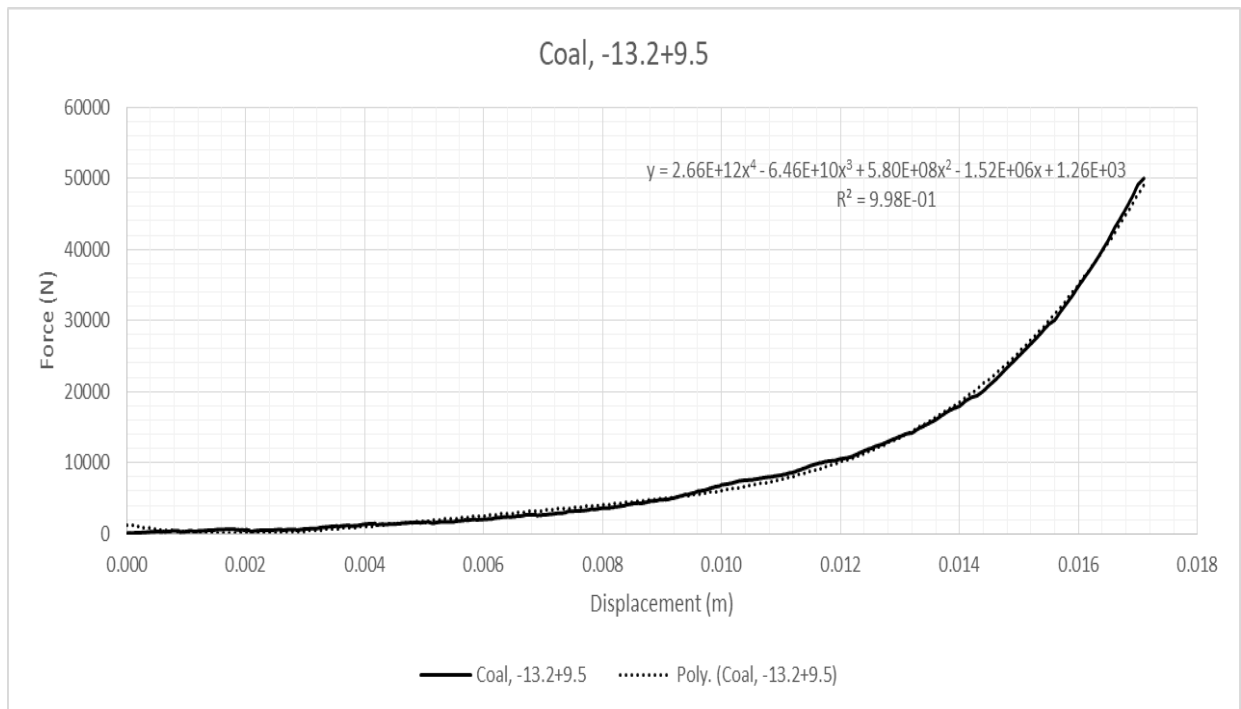


Figure Q 3: Fitting of force-displacement data for -13.2+9.5 mm of coal to polynomial degree 6

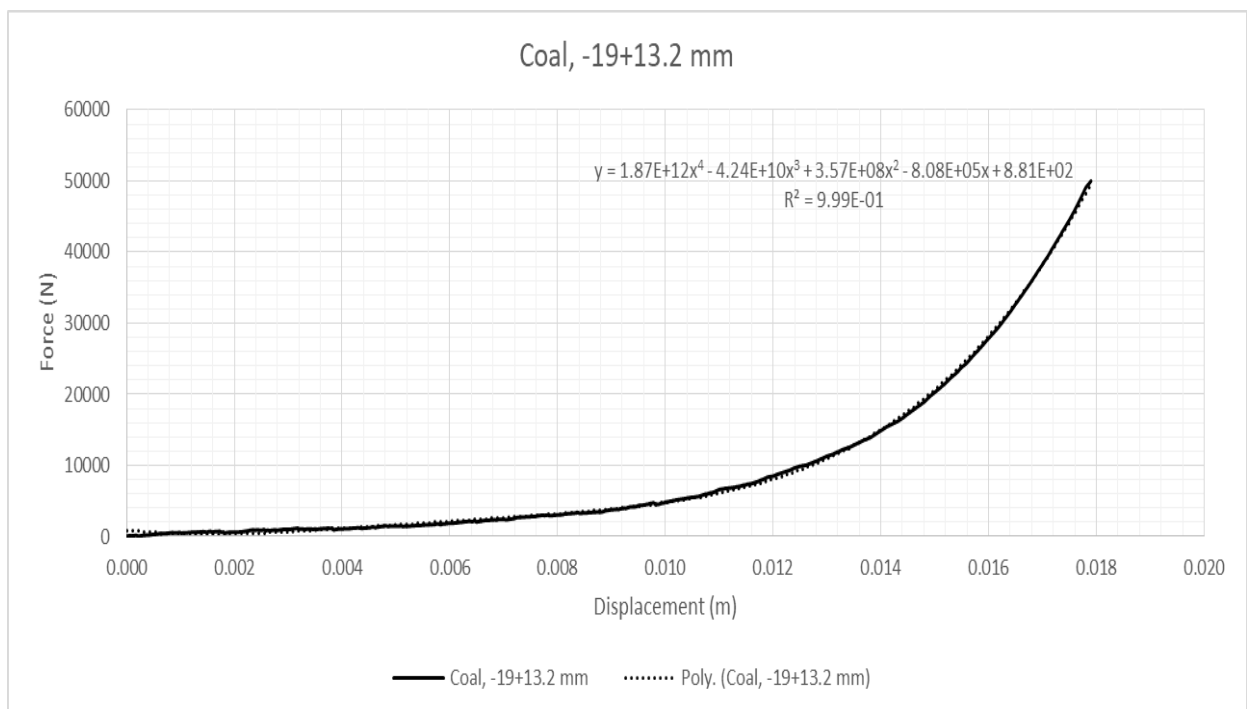


Figure Q 4: Fitting of force-displacement data for -19+13.2 mm of coal to polynomial degree 6

Appendix R: Coal comminution data and results

Size reduction ratios calculation and regression modelling

The reduction ratios were calculated by dividing the d_{80} values with the geometric mean of the feed size range as illustrated by Eq. (R1). The d_{80} values were interpolated from Figures 7.1 and 7.2.

$$R_{80} = \frac{d_{80}}{\sqrt{x_{bottom} \times x_{top}}} \quad (R1)$$

Table R 1: Size reduction ratios at various crusher settings

Test Number	Particle size (mm)	Mean size (mm)	Offset (mm)	Gape (mm)	d_{80} (mm)	R_{80}
1	-13.2+9.5	11.2	10.0	3.0	9.5	1.31
2	-19+13.2	15.8	10.0	3.0	9.5	1.88
3	-13.2+9.5	11.2	5.0	3.0	7.6	1.64
4	19+13.2	15.8	5.0	3.0	8.3	2.15
5	-13.2+9.5	11.2	5.0	1.5	8.8	1.42
6	-19+13.2	15.8	5.0	1.5	9.2	1.94
7	-13.2+9.5	11.2	10.0	1.5	9.2	1.35
8	-19+13.2	15.8	10.0	1.5	8.8	2.03

Table R 2: Modelling data for d_{80} and R_{80} for coal experiments

R-Squared Values				0.4778		0.9239	
Test Number	Mean Particle size (mm)	Offset (mm)	Gap (mm)	Actual d_{80} (mm)	Predicted d_{80} (mm)	Actual R_{80}	Predicted R_{80}
1	11.2	10	3.0	9.5	9.02	1.31	1.39
2	15.8	10	3.0	9.5	9.20	1.88	1.96
3	11.2	5	3.0	7.6	8.25	1.64	1.53
4	15.8	5	3.0	8.3	8.43	2.15	2.10
5	11.2	5	1.5	8.8	8.52	1.42	1.47
6	15.8	5	1.5	9.2	8.70	1.94	2.04
7	11.2	10	1.5	9.2	9.30	1.35	1.33
8	15.8	10	1.5	8.8	9.48	2.03	1.90

Table R 3: Experimental and predicted throughputs for coal experiments at 330 rpm

Test #	Size range (mm)	Mean size (mm)	Offset (mm)	Exit Gap (mm)	Mass (g)	Residence time (s)	Q (tph)	Predicted Q (tph)
1	-13.2+9.5	11.2	10	3	1005.49	38	0.095	0.0727
2	-19+13.2	15.8	10	3	982.21	47	0.075	0.0653
3	-13.2+9.5	11.2	5	3	928.02	61	0.055	0.0752
4	-19+13.2	15.8	5	3	1010.74	65	0.056	0.0678
5	-13.2+9.5	11.2	5	1.5	945.92	53	0.064	0.0493
6	-19+13.2	15.8	5	1.5	1062.09	65	0.059	0.0418
7	-13.2+9.5	11.2	10	1.5	992.93	120	0.030	0.0468
8	-19+13.2	15.8	10	1.5	987.78	150	0.024	0.0393

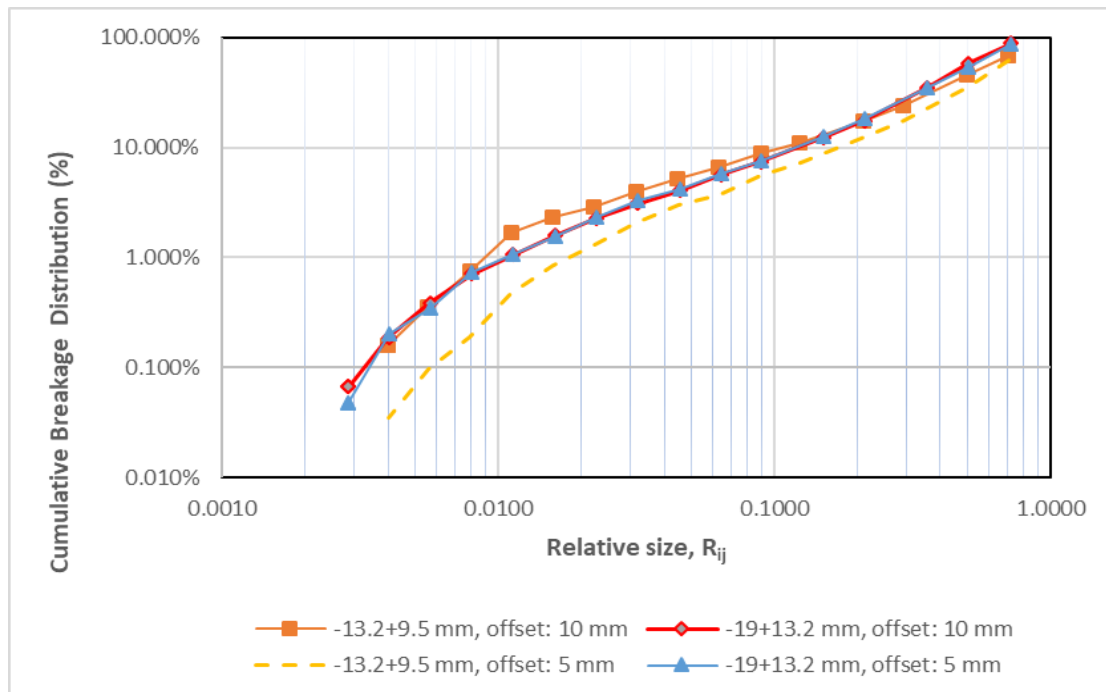


Figure R 1: Cumulative breakage functions for the mono-sized particle of coal comminuted in the rotary offset rusher at exit gap of 1.5 mm

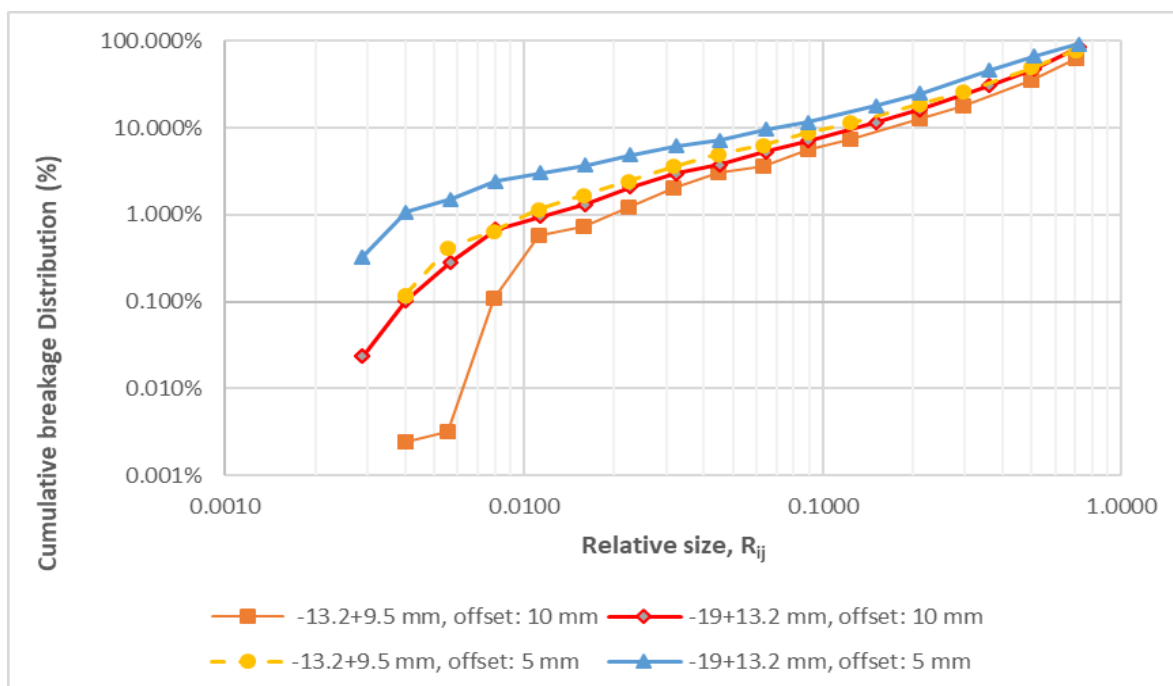


Figure R 2: Cumulative breakage functions for the mono-sized particle of coal comminuted in the rotary offset rusher at exit gap of 3 mm

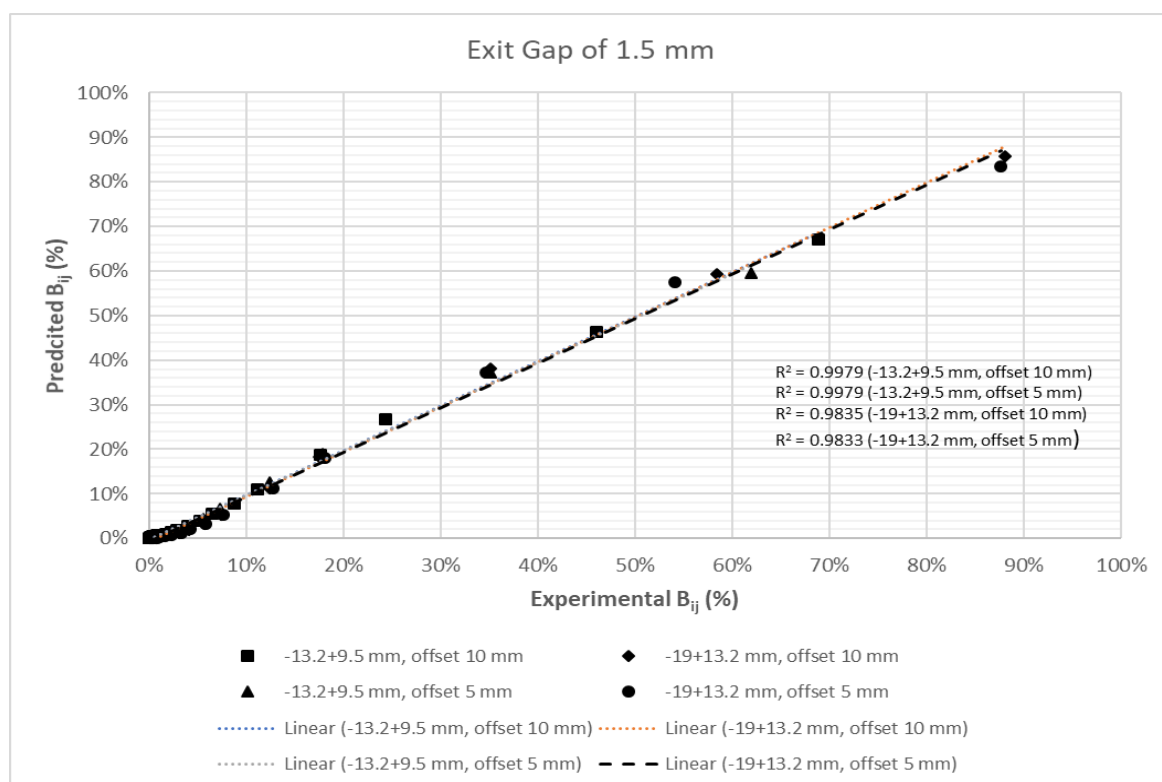


Figure R 3: Experimental versus predicted cumulative breakage functions for the mono-sized particle of coal comminuted in the rotary offset rusher at exit gap of 1.5 mm

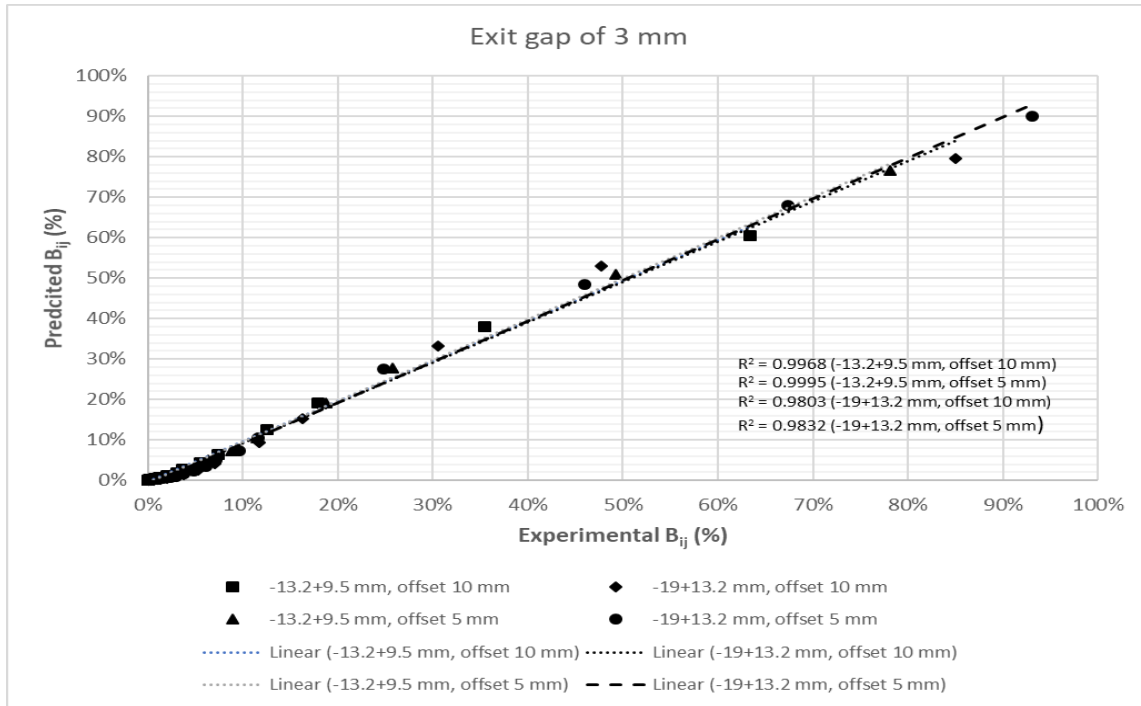


Figure R 4: Experimental versus predicted cumulative breakage functions for the mono-sized particle of coal comminuted in the rotary offset rusher at exit gap of 3 mm

Appendix S: Quartz comminution data and results

Table S 1: Regression data for d_{80} modelling

Regression Statistic: d_{80} model								
Multiple R	0.99593203							
R Square	0.99188061							
Adjusted R Square	0.98105476							
Standard Error	0.27223723							
Observations	8							
ANOVA								
	<i>df</i>	<i>SS</i>	<i>MS</i>	<i>F</i>	<i>Significance F</i>			
Regression	4	27.16141068	6.790353	91.62148	0.001820137			
Residual	3	0.222339321	0.074113					
Total	7	27.38375						
	<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	62.3043325	9.076747006	6.86417	0.006331	33.41807251	91.19059245	33.41807251	91.19059245
Speed	-0.0902804	0.011839006	-7.62567	0.004681	-0.127957404	-0.052603405	-0.127957404	-0.052603405
Feed rate	-0.00430773	0.001032607	-4.17171	0.025077	-0.00759395	-0.00102152	-0.00759395	-0.00102152
Feed size	0.21537946	0.048575112	4.433947	0.02132	0.060791771	0.369967146	0.060791771	0.369967146
Crushing Time	-0.70357081	0.106140648	-6.62867	0.006994	-1.041357725	-0.365783897	-1.041357725	-0.365783897

Table S 2: Regression data for d_{50} modelling

SUMMARY OUTPUT: d50								
Regression Statistics								
Multiple R	0.981012							
R Square	0.962384							
Adjusted R Square	0.912229							
Standard Error	0.376726							
Observations	8							
ANOVA								
	<i>df</i>	<i>SS</i>	<i>MS</i>	<i>F</i>	<i>Significance F</i>			
Regression	4	10.89298	2.723245	19.18823	0.017827			
Residual	3	0.425768	0.141923					
Total	7	11.31875						
	<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	32.59258	12.57844	2.591147	0.080994	-7.43762	72.62278	-7.43762	72.62278
Mean Feed	0.057662	0.067264	0.857248	0.454331	-0.1564	0.271727	-0.1564	0.271727
Speed (rpm)	-0.04774	0.016406	-2.90996	0.061999	-0.09995	0.00447	-0.09995	0.00447
Feed rate	-0.00142	0.001431	-0.98989	0.395204	-0.00597	0.003137	-0.00597	0.003137
Crushing time	-0.35803	0.147079	-2.43427	0.092974	-0.8261	0.110041	-0.8261	0.110041

Table S 3: Regression data for throughput modelling

SUMMARY OUTPUT:throughput									
<i>Regression Statistics</i>									
Multiple R	0.96969								
R Square	0.940299								
Adjusted R Square	0.895523								
Standard Error	62.83702								
Observations	8								
<i>ANOVA</i>									
	<i>df</i>	<i>SS</i>	<i>MS</i>		<i>F</i>	<i>Significance F</i>			
Regression	3	248755.9	82918.63		21.00008	0.006548			
Residual	4	15793.96	3948.491						
Total	7	264549.8							
	<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	-3660.52	1739.216	-2.1047		0.103106	-8489.36	1168.318	-8489.36	1168.318025
Speed	5.503299	2.36437	2.327596		0.080464	-1.06125	12.06784	-1.06125	12.06784362
Feed rate	0.557385	0.206273	2.702179		0.053972	-0.01532	1.13009	-0.01532	1.130089711
Crushing t	36.85741	21.17584	1.74054		0.156739	-21.9362	95.65098	-21.9362	95.65097536

Appendix T: Derivations of moment of inertia formulae for the discs

Figure T1 illustrates the dimensions (in terms of symbols) of the discs, and shafts. Considering the bottom disc (a hollow cylinder), Eq. (2.35), for the mass moment of inertia for the hollow cylinder, is substituted into Eq. (2.33) to get its rotational energy, as shown in Eq. (T1).

$$E_{k,bd} = \frac{1}{4} \pi \rho h_{bd} \omega_{bd,ave}^2 (R_{o,bd}^4 - R_{i,bd}^4) \quad (T1)$$

Where $E_{k,bd}$ is the rotational energy of the bottom disc, ρ is the density of material of construction, h_{bd} is the thickness of the disc, $R_{o,bd}$ and $R_{i,bd}$ are outside and inside radii respectively and $\omega_{bd,ave}$ average angular velocity of the bottom disc during steady state. Similarly, the rotational energy of the feeding shaft is calculated by substituting the dimensions of the shaft into Eq. (T2) as shown below:

$$E_{k,fs} = \frac{1}{4} \pi \rho h_{fs} \omega_{fs,ave}^2 (R_{o,fs}^4 - R_{i,fs}^4) \quad (T2)$$

Where $E_{k,fs}$ is the rotational energy of the feeding shaft, ρ is the density of material of construction, h_{fs} is the thickness of the disc, $R_{o,fs}$ and $R_{i,fs}$ are outside and inside radii respectively and $\omega_{fs,max}$ is the average angular velocity of the feeding shaft (same as those for the top disc).

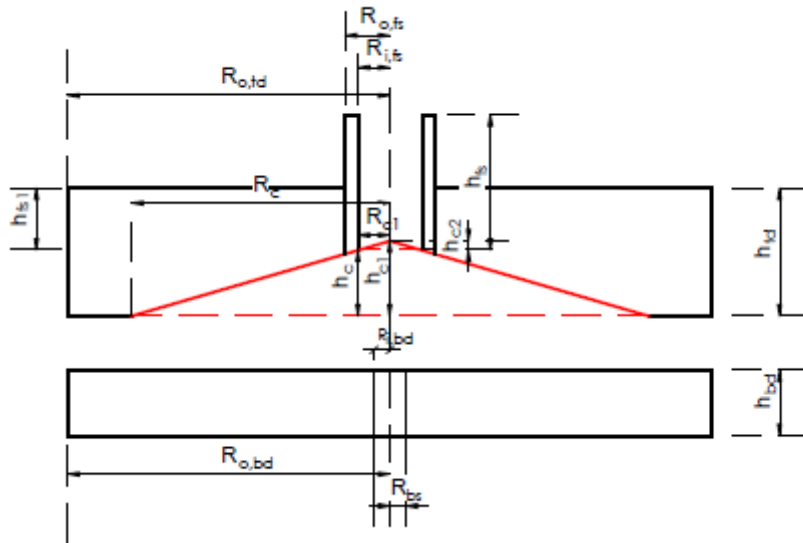


Figure T 1: Geometry for the ROC discs and shafts

To calculate the mass moment of inertia of the (hollow) top disc, first Eq. (2.35) can be used to calculate the moment of inertia assuming that the top disc has no cone (comminution cavity) is shown in Eq. (T3).

$$I_{m,t,td} = \frac{1}{2} \pi \rho h_{td} (R_{o,td}^4 - R_{i,td}^4) \quad (T3)$$

Considering the conical shape (truncated), shown in Figures T1 and T2, Eq. (2.36) was used to find the difference between moment of inertia of the large and small cone as shown in Eq. (T4).

$$I_{m,cc} = \frac{3}{10} (M_{c1} R_{c1}^2 - M_{c2} R_{c2}^2) \quad (T4)$$

Where $I_{m,cc}$ is the volume of the crushing chamber (assumed to be solid), M_{c1} is the mass of the large cone, M_{c2} is the mass of the small cone, R_{c1} is the radius of the large cone and R_{c2} is the radius of the small cone.

From Figure T2, the following relationship can be written:

$$h_{c1} = h_{c2} + h_c \quad (T5)$$

Using properties of similar shapes, the ratios of heights and radii of the larger and small cones, Eq. (T6) can be written.

$$\frac{h_{c1}}{h_{c2}} = \frac{R_{c1}}{R_{c2}} \quad (T6)$$

From Eq. (O6), the following relationship is obtained:

$$h_{c2} = \frac{R_{c2} h_{c1}}{R_{c1}} \quad (T7)$$

Replacing M_c and M_{c1} in Eq. (T4) with their respective equivalent expressions (density $\times$ volume) results in Eq. (T8). Substituting Eq. (T7) into Eq. (T8) results in Eq. (T9) which can be used to evaluate the mass moment of inertia of the truncated cone (assumed to be solid) knowing the radii of the large and small cone.

$$I_{m,cc} = \frac{\pi \rho}{10} (R_{c1}^4 h_{c1} - R_{c2}^4 h_{c2}) \quad (T8)$$

$$I_{m,cc} = \frac{2\pi \rho h_{c1} (R_{c1}^5 - R_{c2}^5)}{R_{c1}} \quad (T9)$$

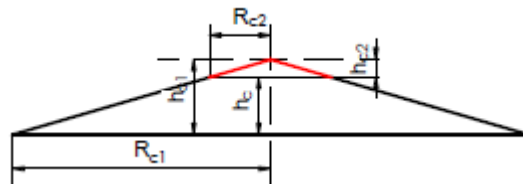


Figure T 2: Dimensions of the comminution cavity

The rotational energy of the top disc is then obtained by subtracting Eq. (T9) from Eq. (T3) and substituting the resultant equation for the moment of inertia in Eq. (2.33). Adding Eq. (T2) to the difference obtained result in Eq. (T10) that can be used to calculate the rotational energy of the top disc with its shaft.

$$E_{k,td} = \frac{1}{4} \pi \rho \omega_{td}^2 (h_{td} (r_{o,td}^4 - r_{i,td}^4) - \frac{2\pi \rho h_{c1} (R_{c1}^5 - R_{c2}^5)}{R_{c1}} + h_{fs} (r_{o,fs}^4 - r_{i,fs}^4)) \quad (T10)$$

Appendix U: Instrumentation plots for quartz comminution tests

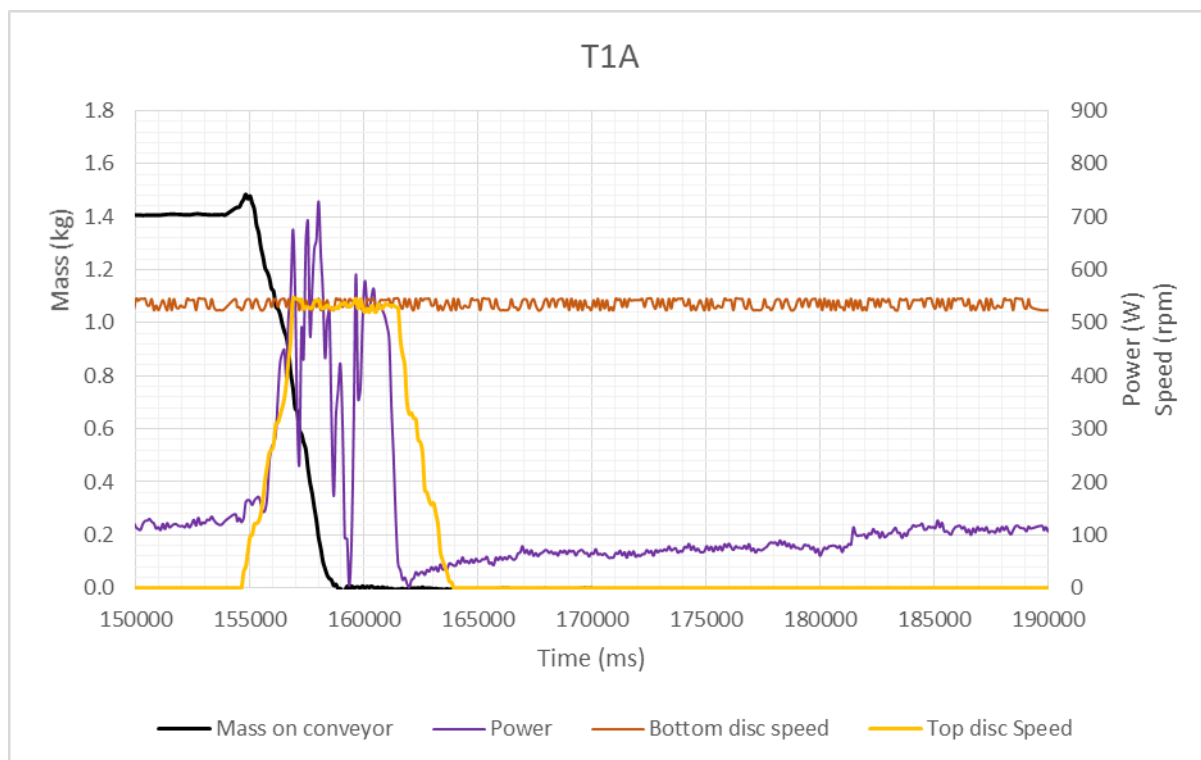


Figure U 1: The signals operating variables for T1A (a repeat of T1B) at the speed of 550 rpm



Figure U 2: The signals operating variables for T1B (a repeat of T1A) at the speed of 550 rpm

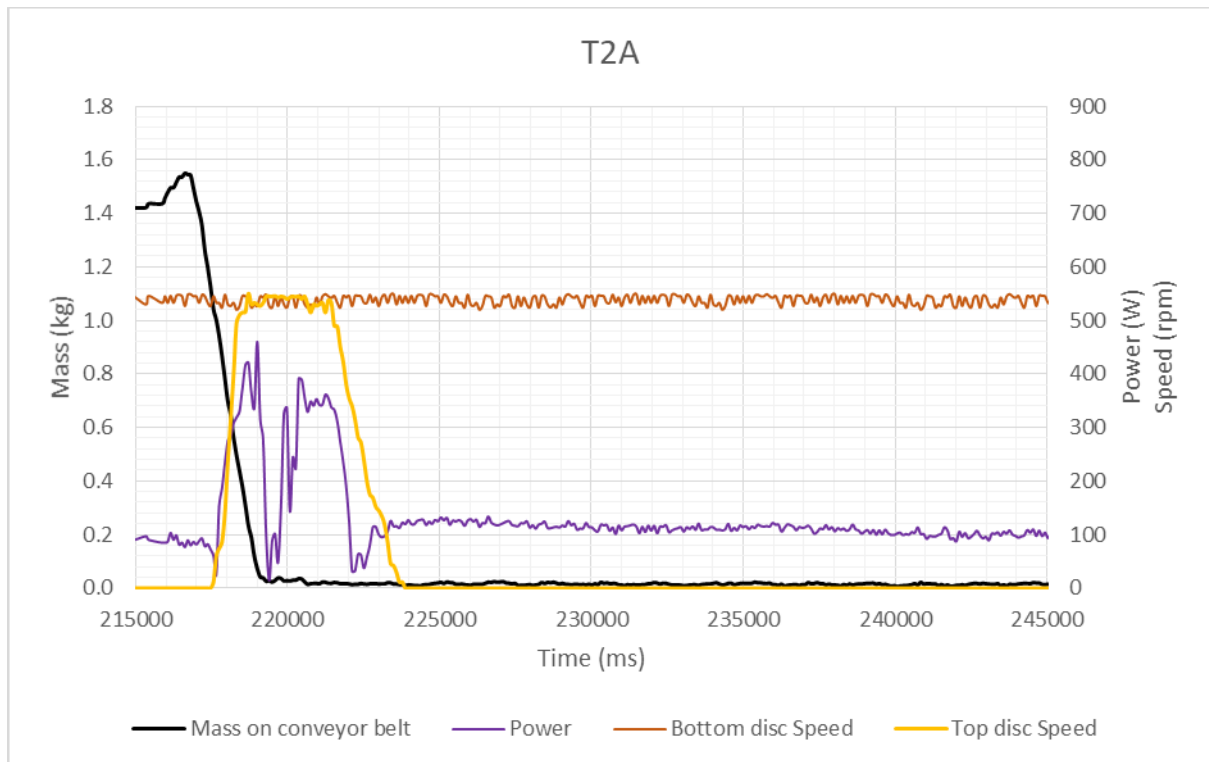


Figure U 3: The signals operating variables for T2A (a repeat of T2B) at the speed of 550 rpm

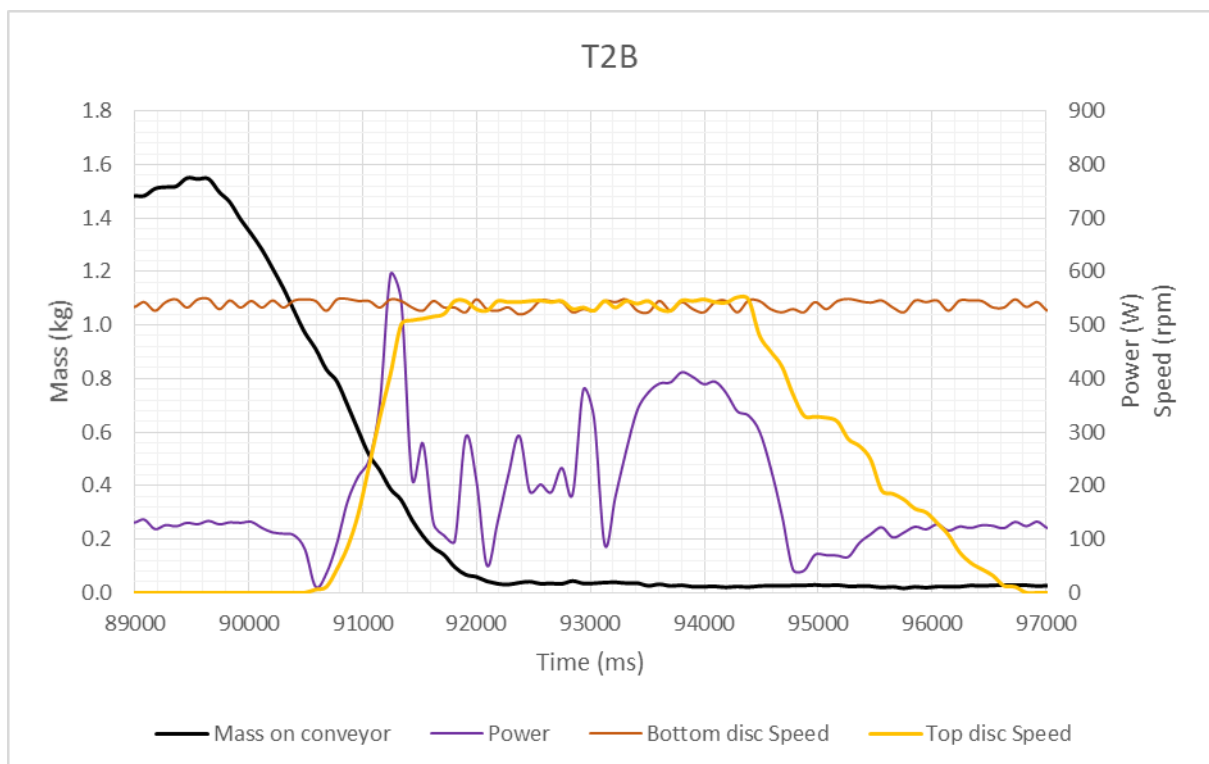


Figure U 4: The signals operating variables for T2B (a repeat of T2A) at the speed of 550 rpm

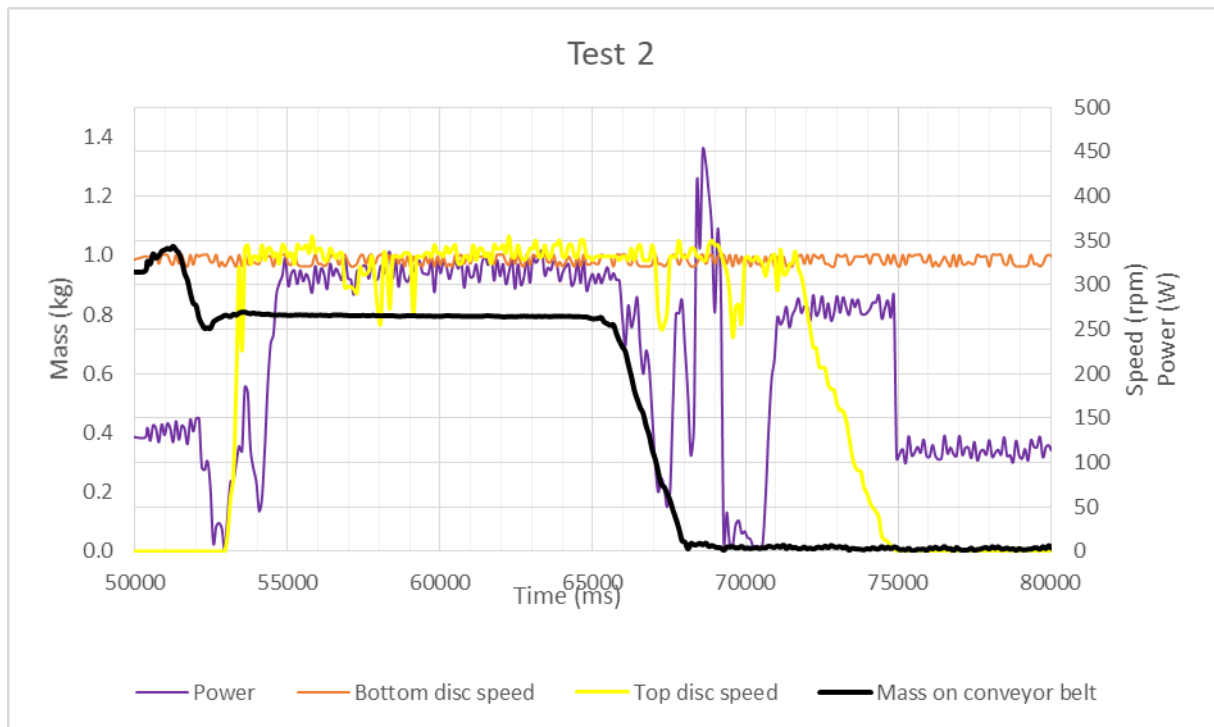


Figure U 5: The power signal for test 2 at the speed of 330 rpm

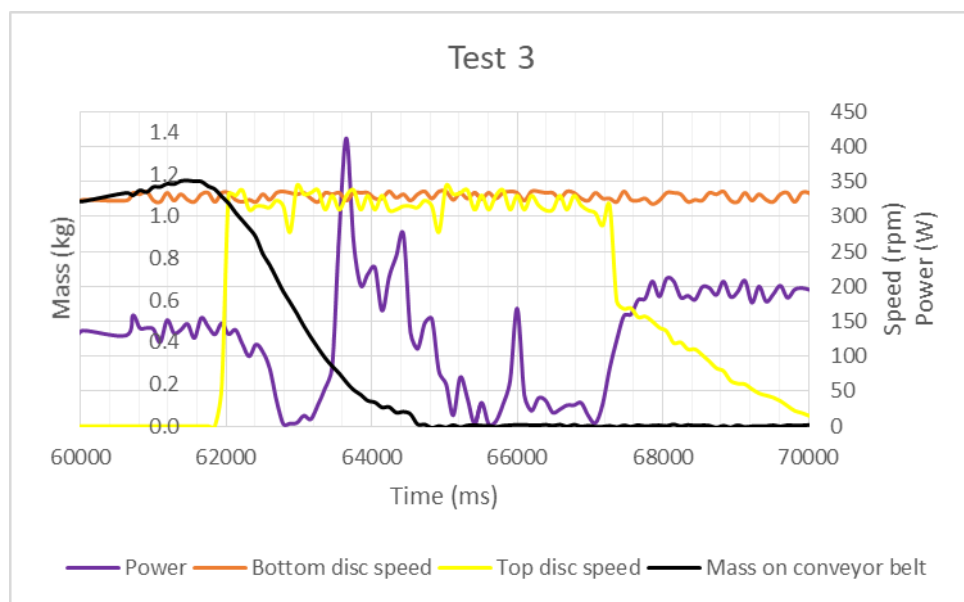


Figure U 6: The power signal for test 3 at the speed of 330 rpm

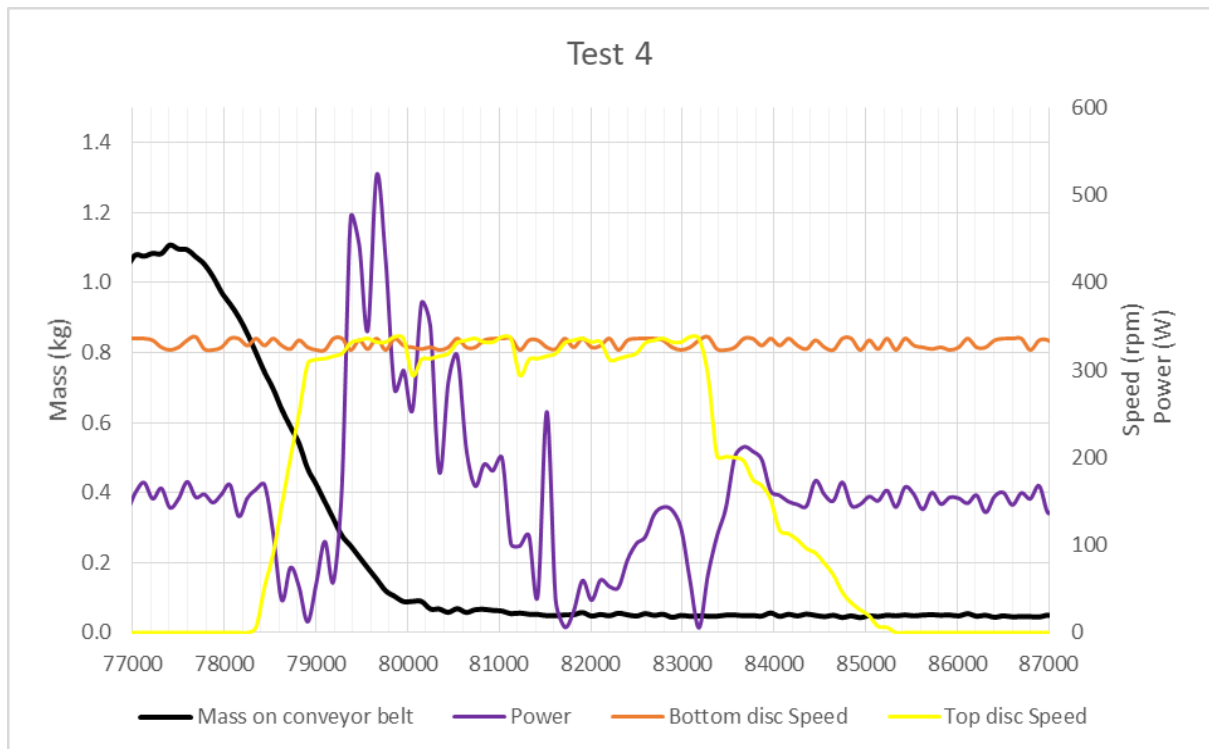


Figure U 7: The power signal for test 4 at the speed of 330 rpm

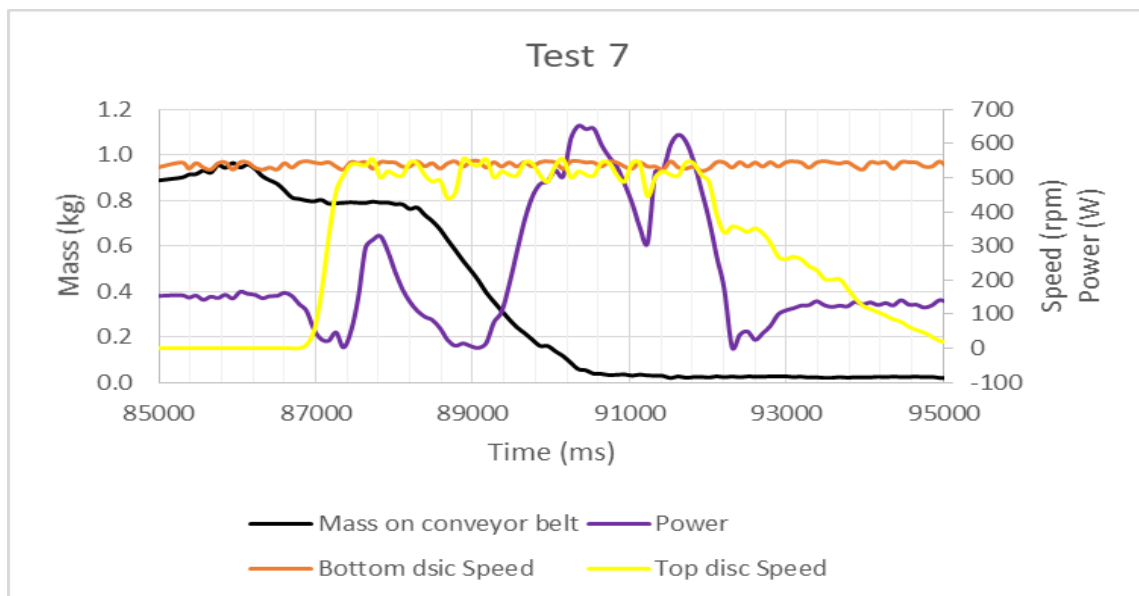


Figure U 8: The power signal for test 7 at the speed of 550 rpm

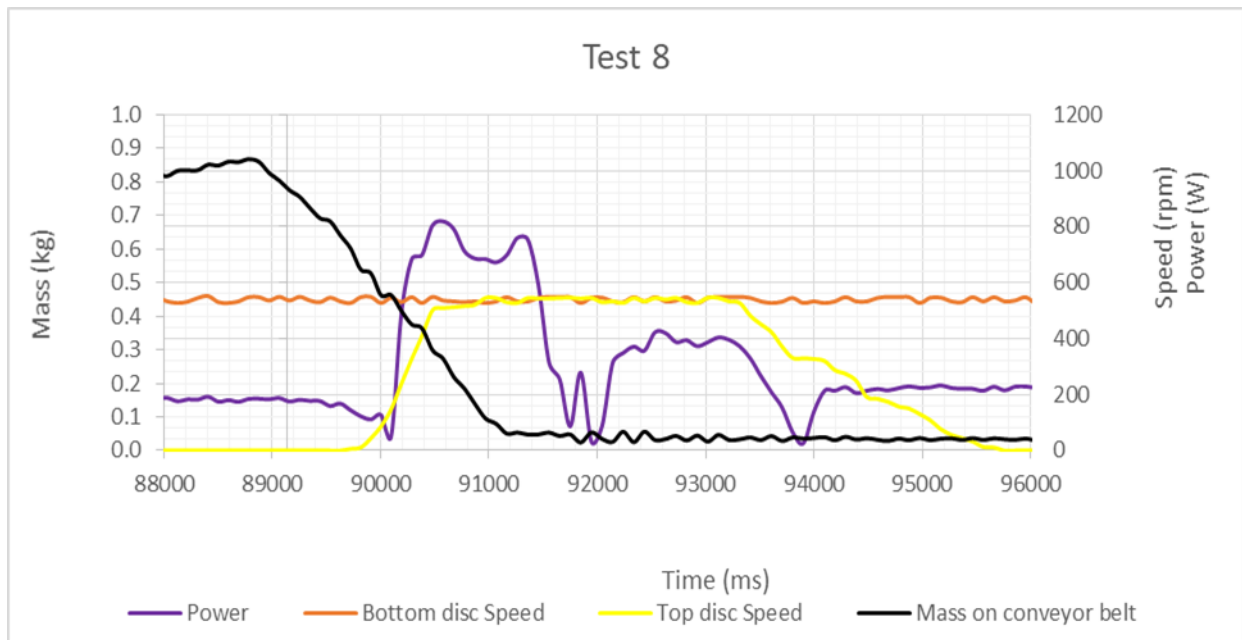


Figure U 9: The power signal for test 8 at the speed of 550 rpm

Table U 1: Summary of computation of specific comminution energy for the ROC experiments

Test #	Size range (mm)	Mean size (mm)	Mass (g)	Speed (rpm)	Feeding Time (s)	Crushing time (s)	d ₈₀ (mm)	R ₈₀	d ₅₀ (mm)	R ₅₀	Bond		Morrell				DWT		ROC Instrumentation							
											E (kWh/t)	Model d ₈₀ (mm)	f(x ₁)	f(x ₂)	E (kWh/t)	Model d ₈₀ (mm)	E (kWh/t)	Model d ₈₀ (mm)	Area under the curve (J)	Total specific energy (kWh/t)	Crushing time (s)	Frictional Energy (J)	Comminution Energy (J)	Specific comminution Energy (J/g)	Specific comminution Energy (kWh/t)	ROC Model d ₈₀ (mm)
2	-19+13.2	15.8	1027	330	3.2	18.602	6.2	2.9	3.2	5.0	0.707	6.095	-0.313	-0.301	1.071	6.084	0.546	6.200	5710.838		18.602	3720.4	1990.438	1.938	0.538	5.6
4	-19+13.2	15.8	1051	330	2.5	4.935	8.4	2.1	5.4	3.0	0.465	8.078	-0.313	-0.303	0.659	7.824	0.284	8.400	953.2259		4.8	960	2.2259	0.002		
6	-19+13.2	15.8	744	550	1.6	5	5.0	3.6	2.8	5.9	0.903	5.175	-0.313	-0.300	1.376	5.344	0.868	5.000				0	0	0.000		
8	-19+13.2	15.8	808	550	2.4	10	5.0	3.6	2.4	6.7	0.903	5.175	-0.313	-0.300	1.376	5.344	0.868	5.000	1771.828	0.609	3.4	680	1091.828	1.351	0.375	6.5
2A	-19+13.2	15.8	1459	550	2.2	3.4	10.6	1.7	5.0	3.2	0.302	10.779	-0.313	-0.306	0.355	10.784	0.172	10.600	1237.625	0.236	3.4	680	557.625	0.382	0.106	10.6
2B	-19+13.2	15.8	1459	550	2.3	3.2	11.3	1.6	6.0	2.7	0.263	11.815	-0.313	-0.306	0.278	12.232	0.151	11.250	1091.286	0.208	3.2	640	451.286	0.309	0.086	11.5
1A	-19+13.2	15.8	1458	550	3.6	6.9	8.2	2.2	3.6	4.5	0.483	7.875	-0.313	-0.303	0.691	7.634	0.299	8.200	2661.122	0.507	6.9	1380	1281.122	0.879	0.244	7.6
1B	-19+13.2	15.8	1432	550	3.5	6.9	8.2	2.2	3.6	4.5	0.483	7.875	-0.313	-0.303	0.691	7.634	0.299	8.200	2759.434	0.535	6.9	1380	1379.434	0.963	0.268	7.4