COMPARISON OF HPGR - BALL MILL AND HPGR - STIRRED MILL CIRCUITS TO THE EXISTING AG/SAG MILL - BALL MILL CIRCUITS

by

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Abstract

In the past 20 years, the energy-efficient comminution technologies, including high pressure grinding rolls (HPGR) and high speed stirred mill, have been developed and adopted in the hard-rock mining operation in order to reduce the energy consumption and improve the process performance. The combination of HPGR and stirred mill in a single flowsheet without tumbling mills has been demonstrated to be technically feasible. This research focused on the energy and cost comparisons of the existing AG/SAG ball mill circuits with two proposed comminution circuits, including an HPGR - ball mill circuit and a novel HPGR - stirred mill circuit.

The main objective of this research was to advance the understanding of the potential benefits of the proposed HPGR stirred mill-based comminution circuits for low-grade, high - tonnage base metal operation. Samples and operating data were collected directly from the existing SAB/AGBC/SABC circuits to establish a base case for comparison. To support the base case, the existing circuits were fitted and simulated using a JK SimMet® model. Specific energy requirements for the proposed HPGR - ball mill circuit and HPGR - stirred mill circuit were determined from a pilot-scale HPGR and stirred mill test, in association with a JK SimMet® simulation.

Results obtained from the research showed that the HPGR - ball mill circuit and HPGR - stirred mill circuit achieved a substantial reduction in energy, with considerable cost advantage over the existing SAB/AGBC/SABC circuits.

Preface

Some of the results presented in this document were published in abbreviated form in the proceedings of the 45th Annual Meeting of Canadian Mineral Processors:

Wang, C., Nadolski, S., Mejia, O., Drozdiak, J., & Klein, B. (2013). Energy and cost comparisons of HPGR based circuits with the SABC circuit installed at the Huckleberry mine. 45th Annual Meeting of the Canadian Mineral Processors, Ottawa, ON, Canada.

I was responsible for developing the test program, conducting test work, and interpreting the results, under the supervision of Dr. Bern Klein, professor of the Norman B. Keevil Institute of Mining Engineering, University of British Columbia. Mr. Stefan Nadolski and Mr. Zorigtkhuu Davaanyam assisted with the HPGR testing and provided a summary for HPGR operating data. Mr. Amit Kumar and Mr. Javier Perez assisted with the IsaMill™ testing. Mr. Olav Mejia assisted with the assessment of capital and operating costs.

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Symbol	Description
B_{Wi}	Bond ball mill work index
C_{Wi}	Bond crusher work index
D	HPGR roll diameter (m)
E_{sp}	net specific energy consumption (kWh/t)
F	HPGR hydraulic pressing force (kN)
F_{80}	particle size at which 80% of particles pass in feed
F_{SP}	HPGR specific pressing force (N/mm^2)
L	HPGR roll length (m)
$\dot{m}$ or m-dot	HPGR specific throughput constant (ts/hm^3)
M	throughput (tph)
P_{80}	particle size at which 80% of particles pass in product
P_i	idle power draw
P_t	total main motor power draw (kW)
R^2	coefficient of determination
R_{Wi}	Bond rod mill work index
SI	stress intensity
v	HPGR roll speed (m/s)
ρ	media density

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CHAPTER 1: INTRODUCTION

1.1 Background

In the mining and mineral processing field, comminution refers to the size reduction of run-of-mine (ROM) ore, such as that accomplished in the crushing and grinding process, until the finely-disseminated minerals of interest are liberated from the gangue before concentration. Up until now, tumbling grinding mills, such as Autogenous/Semi-autogenous grinding mills (AG/SAG mills) and ball mills, have had a dominant bearing on the design and economics of comminution circuits. However, it is commonly agreed that the majority of employed comminution processes are both energy-intensive and energy-inefficient, are responsible for up to 80% of overall process plant energy consumption and have an efficiency of as low as 1% (Abouzeid & Fuerstenau, 2009; Fuerstenau & Abouzeid, 2002). Thus, in order to help the global mining industry to extract low-grade, high-tonnage, and complex mineral deposits in a more economical and environmental manner, the exploration of innovative technology to reduce energy demand and carbon emissions is becoming more essential. The U.S. Department of Energy reported that there is the potential to reduce energy consumption in the metals industry by up to 61% from current practice to best-estimated practical minimum energy consumption. Suggestions for doing so included the implementation of best practices, and the adoption of energy-efficient mining and mineral processing technologies, such as advanced blasting techniques, high pressure grinding rolls (HPGR), and stirred mills (U.S. DOE, 2007).

The concept of a combination of HPGRs and stirred mills in a single comminution flowsheet was proposed in order to achieve size reduction without the need for tumbling mills (Valery & Jankovic, 2002; Pease, 2007). The pilot-scale HPGR and high speed stirred mill testing facility at the UBC Norman B. Keevil Institute of Mining Engineering provided a very unique opportunity to assess the HPGR and/or stirred mill circuits, and to understand the potential benefits. In order to examine a combined HPGR and stirred mill circuit, both machines have to be operated outside of their currently optimal operating conditions. Drozdiak et al. (2011) demonstrated that an HPGR - stirred mill circuit is both technically feasible and showed promising benefits over the stage crushers - ball mill circuit and HPGR - ball mill circuits. In order to determine whether the novel HPGR - stirred mill circuit arrangement could achieve energy and cost benefits in comparison to conventional AG/SAG mill based circuits, the pilot-scale operation with large quantities of sample would provide a more reliable way to measure energy and directly compare results, thus strongly supporting the main findings. Four sets of samples, including soft and hard ores, from two copper porphyry operations were studied. The circuits' comparisons were evaluated in terms of comminution specific energy, total circuit energy, and capital and operating costs.

1.2 Thesis objectives

This thesis focused on the study of low-grade, high-tonnage hard-rock comminution (gold, copper, platinum, molybdenum, etc.). The primary objective of the research was to enhance the understanding of the potential benefits of the HPGR and/or stirred mill circuits. In order to achieve the primary objective, the following secondary objectives are targeted,

- Determination of the specific energy requirements to operate the existing AG/SAG ball mill circuit arrangements, based upon the provided samples and process data from the existing operations as well as the circuit simulation confirmation.
- Determination of the potential specific energy requirements necessary to operate the proposed HPGR - ball mill and HPGR - stirred mill circuits with the provided samples from the existing operations.

- Comparison of the overall circuit energy requirements including the major material handling equipment of the existing comminution circuits to that of proposed circuits for equivalent comminution duties.
- Comparison of the operating and capital costs of the existing comminution circuits to that of proposed circuits for equivalent comminution duties.

1.3 Thesis outline

Chapter 2 reviews the current literature related to the comminution fundamentals, high pressure grinding rolls, and stirred media mills. This section also covers history of comminution technology, and summarizes the energy efficiency and circuit design of comminution for the hard-rock mining industry.

Chapter 3 presents an overall experimental program for the evaluation of comminution circuits. The description of existing comminution circuits, collected samples, and equipment used for the program, as well as the experimental procedures, are presented.

Chapter 4 presents the results and discussion of pilot-scale HPGR and stirred mill testing.

Chapter 5 presents the circuit modeling and simulation.

Chapter 6 presents a thorough energy and cost comparison between the proposed circuits and the existing circuits.

Chapter 7 covers the main conclusions of the research, and recommendations for future work.

CHAPTER 2: LITERATURE REVIEW

2.1 Comminution

2.1.1 A short comminution history

From the 1920s to 1950s, a comminution circuit consisting of multiple-stage crushing, followed by rod and ball mills, was the most common circuit design. During the 1960s, the rod mills were gradually replaced by larger-diameter ball mills, accepting coarser feed and achieving operating cost savings. In the early 1970s, autogenous grinding (AG) mills and semi-autogenous grinding (SAG) mills started to gain favour with large installations in the based metal industry of North American, and soon, the circuits with AG/SAG mills and ball mills became industry standard universally, due to their simpler flowsheet and higher processing capacity. Until December 2010, over 1500 AG/SAG mills were sold globally and the total installed power was approximately 5,000 MW (Jones & Fresko, 2011).

However, for the treatment of high-tonnage hard competent ore, AG/SAG mill-based circuits have become extremely energy-inefficient (Morley & Staples, 2010). Over the past two or three decades, the mining industry has been searching for more energy-efficient comminution technology for hard-rock mining, due to the current desires of reducing energy consumption, carbon footprint, and greenhouse gas emissions (Norgate & Haque, 2010). Since the improvement of roll wear protection allowed HPGR to treat hard and abrasive materials, high pressure grinding rolls (HPGR) have become more attractive. There are a number of projects that are now using HPGRs in the comminution circuit instead of conventional AG or SAG mills. More recently, stirred mill technology has also been adapted into the minerals industry. Stirred mill technology shows better energy efficiency than ball mills for fine and ultrafine grinding applications and there has been an increasing interest in extending this technology to coarser grinding applications (Anderson & Burford, 2006).

2.1.2 Comminution methods

During the comminution process, external force applied by comminution equipment results in particle size reduction. There are different kinds of external forces, as shown in Figure 2.1. Crushing process mechanisms include impact and compression, and the products usually contain relatively coarser size fractions. On the other hand, the dominant breakage mechanisms in the grinding process are abrasion and attrition, which contribute to finer particle size of product.

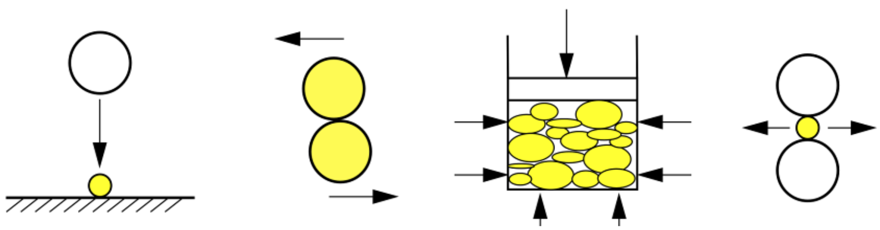

Figure 2.1 Grinding mechanisms

Comminution requires different types of crushing and grinding machines, depending on the feed size ranges and ore hardness, as well as the throughput requirement. It was reported that 89% energy in the comminution plant was consumed during the size reduction from about 20 mm to 100 μm (Powell, 2010). Therefore, the focus in this work is to improve the energy efficiency in that range of size reduction, thus the grinding process. According to the ways by which motion is imparted to the mill charge, grinding mills are generally categorized into two types: tumbling mills and stirred media mills (Wills & Napier-Munn, 2006). Figure 2.2 compares the theoretical size reduction and power ranges for different types of grinding mills. HPGR technology was included in the comparison, because it has a typical feed size of up to 70 mm and a product size no finer than 4 mm (van der Meer & Gruendken, 2010).

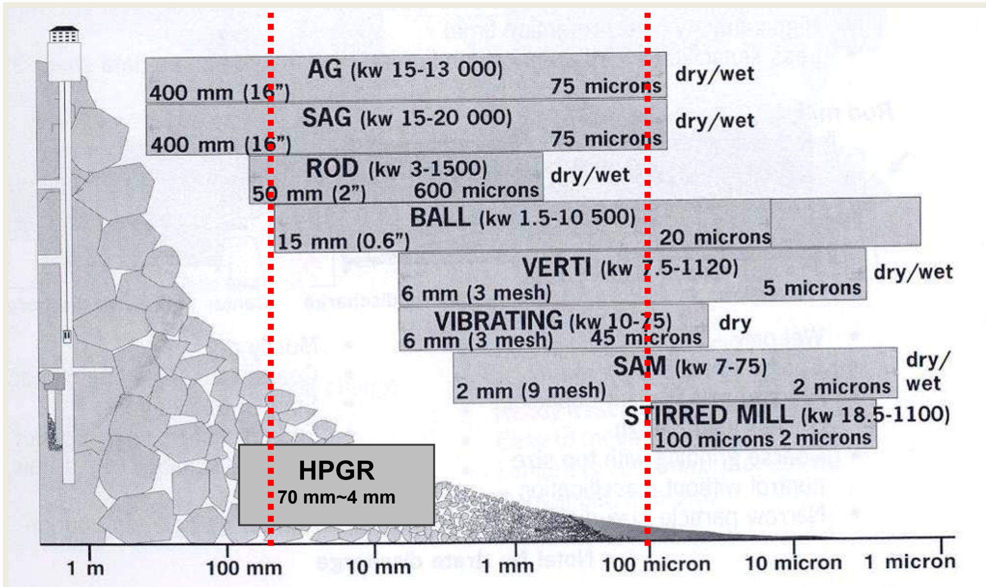

Figure 2.2 Theoretical size reduction per mill type (Metso, 2011)

There are many different types of tumbling mills. The most common units in hard-rock mining are AG, SAG, rod and ball mills. A tumbling mill consists of a metallic cylindrical drum rotating horizontally with internal wear liners and a charge of tumbling media. The tumbling media may be steel rods (rod mills), steel balls (ball mills), rock itself (AG mills) or a combination of rock and steel balls (SAG mills). The breakage mechanisms of impact and attrition are applied to fracture rock in the tumbling mills. Impact breakage is generated by the free-fall of the tumbling media above the mill load. Attrition breakage is achieved by the rolling movement of the load as the material lifts and slips together.

2.1.3 Comminution energy

Comminution is an energy-intensive process. It has been reported that over 3% of the electrical energy consumed worldwide in 1976 was used by crushing and grinding (Fuerstenau & Abouzeid, 2002). In 2001, comminution consumed approximately 1.5% of the national energy consumption in Australia; in the case of South Africa, 1.8% of total energy consumption was consumed during the comminution process in 2003 (Djordjevic, 2010). In a more recent study, estimations of energy consumption by the US Department of Energy (U.S. DOE, 2007) showed that comminution accounted for approximately 50% of the total mining energy in the USA (refer to Figure 2.3). In addition, it was reported that approximately 50% of comminution process costs are attributed to the energy usage, while the other half is attributed to liner/charge wear (Radziszewski, 2002).

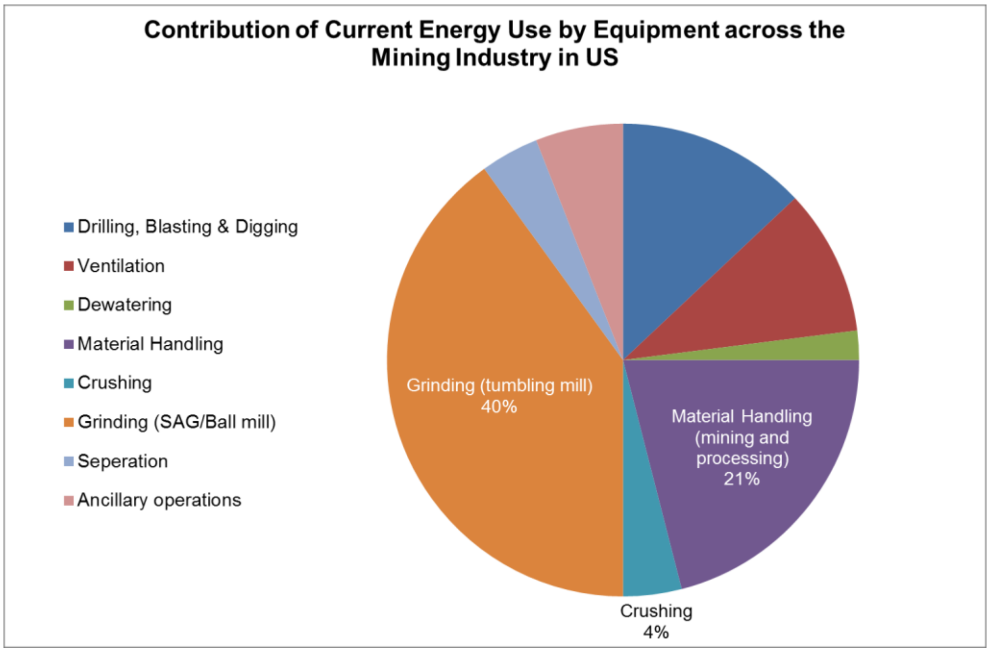

Figure 2.3 Energy use by equipment in US mining industry (U.S. DOE, 2007)

Conventional comminution methods are widely accepted as energy-inefficient processes. The traditional definition of absolute comminution efficiency is the ratio of the useful output energy for producing new surface to the total energy input. In the case of tumbling mills in particular, comminution efficiency refers to the energy required for size reduction, over the mechanical energy delivered to the system by rotating the mill chamber. By this definition, it has been estimated that energy efficiency ranges from 0.1% to 2% for the conventional grinding process, based on the generation of new surface area (Fuerstenau & Abouzeid, 2002; Tromans & Meech, 2004; Whittles et al., 2006). Operation of tumbling mills requires a substantial amount of energy to rotate the large cylindrical chambers filled with steel media and slurry, and most input energy is being dissipated in the form of heat and noise (Alvarado et al., 1998).

2.1.4 Ore characterization and specific energy determination

There are a number of bench-scale grindability tests that have been developed over the years for the design of grinding circuits and optimization of the existing operation.

- Bond work index test (Bond, 1961)
- JK Drop weight test (Napier-Munn et al., 1996)
- SMC Test® (Morrell, 2004)
- SPI test (Starkey & Dobby, 1996)
- SAGDesign Test (Starkey et al., 2006)

As shown in Table 2.1, ore characteristic parameters generated from the above tests are input into the following models to determine the grinding circuit specific energy:

- The Bond based methods, such as Millpower 2000 (Barratt, 1989)
- The proprietary test (e.g. SPI, SAGDesign, SMC Test®) based models
- The population balance/breakage model/classification model based methods (e.g. JK SimMet®)

Table 2.1 SAG models, classified by size parameters (Doll & Barratt, 2011)

Model	Type	Fine size	Medium size	Coarse size
Millpower 2000	3 parameter	B_{Wi}	R_{Wi}	C_{Wi}
SGS CEET2	3 parameter	B_{Wi}	SPI	CI
JK SimMet	2 parameter	B_{Wi}	A, b	-
SMC test	2 parameter	M_{ib}	DWI, A x b	-
SAGDesign	2 parameter	B_{Wi} (modified)	SAGDesign	-

There is single approach to design all comminution circuits, different methods were adopted by different groups of engineers or researchers depending on the circuits and special requirements. It is suggested that the grinding circuits should be designed by using a combination of the above mentioned design methodologies, with pilot plant confirmation sometimes required. The industry has widely relied on the ball mill work index for the design and analysis of ball mill circuits. Some modifications were made to this method for those treating AG/SAG mill or HPGR circuit products, which have a non-standard particle size distribution.

2.2 High pressure grinding rolls

As shown in Figure 2.4, a high pressure grinding roll (HPGR) is operated with a pair of counter-rotating rolls mounted in a sturdy frame. One roll rotates on a fixed axis, while the other is mounted on the floating bearings and moves horizontally. Hydraulic cylinders exert a compressive force (up to 300 N/mm^2) towards to the two rolls (Schoenert, 1987). Material is choke-fed by gravity from the feed hopper into the gap between the moveable roll and the fixed roll, creating a compressive bed of material. The material-bed is then comminuted by the mechanism of inter-particle breakage.

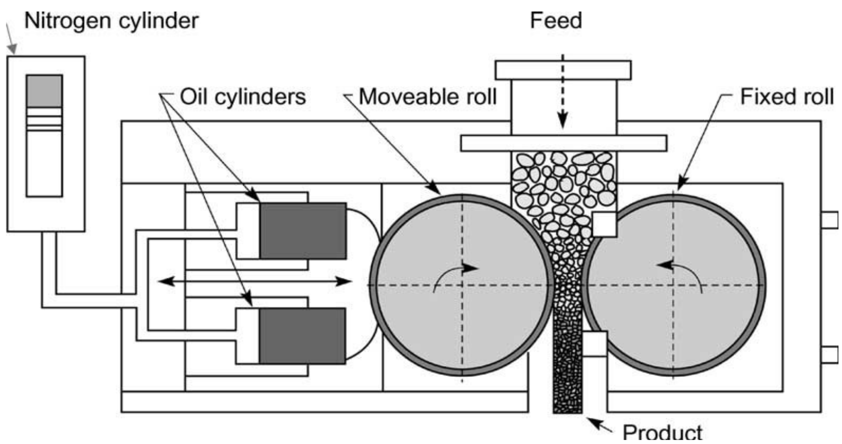

Figure 2.4 Schematic of an HPGR unit (Napier-Munn et al., 1996)

The major HPGR manufacturers in the global market for the mineral and cement industries are,

- ThyssenKrupp Polysius, Germany
- KHD Humboldt Wedag AG, Germany
- Koeppern GmbH & Co. KG, Germany
- CITIC HIC, China

Although there is no fundamental difference in the design principle, the detailed design of the machine does vary by manufacturer. Different aspect ratios between the roll diameter and roll length have been adopted by different manufacturers. Polysius and CITIC prefer a high aspect ratio, which provides a larger operating gap and a longer wear life, while KHD and Koeppern favor a low aspect ratio (Morley, 2010; CITIC HIC, 2012).

The roll surface wear life is the main cause affecting the availability of the HPGR operation (Morley, 2010). All manufacturers are able to provide roll liner designs in order to protect the rolls and to enhance the availability of HPGR operation. Currently, the most applicable roll liner for hard and abrasive materials is the studded lining. The tires with tungsten carbide studs improve wear life through the formation of an autogenous layer between studs. Koeppern has developed Hexadur® wear lining, a hard and abrasion-resistant material set into a softer matrix (Morley, 2010). This technology also promotes the formation of an autogenous layer, and thus protects the surface of rolls.

2.2.1 HPGR history

The design of HPGR was originally from roller presses in the area of coal briquetting in the early 1900s (Morley, 2006b). Professor Klaus Schoenert (Schoenert, 1979) used fundamental physics to develop a new comminution method, so-called inter-particle breakage, resulting from compressing a confined particle-bed. HPGR as an energy-efficient comminution equipment, is based on this inter-particle breakage principle, coupled with a modified cylindrical roll design of roller press.

HPGR was first introduced to treat soft cement clinker in the cement industry in 1985. The first HPGR for comminution of diamond was installed at the Premier Mine in South Africa in 1988 (Casteel, 2005). It was accepted that the HPGR preferentially breaks the kimberlitic host rock, and that the larger diamonds can be liberated with minimum damage (Daniel, 2007a). In the iron industry, the HPGR has been mainly applied to the comminution of iron concentrate for pellet feed production since 1994 (Casteel, 2005). The first HPGR in the iron ore processing plant was fully commissioned as a replacement of a tertiary crusher at CMH-Los Colorados in Chile in 1998 (Westermeyer & Cordes, 2000).

The first plant trial of using HPGR technology for hard abrasive ore-processing was at Cyprus Sierrita copper mine in 1995 (Thompsen et al., 1996). The HPGR unit was decommissioned after 18 months and this trial was considered not successful, due to the issue of high roll surface wear, resulting in significant downtime (Morley, 2010). Surface wear issues were also found at Argyle Diamond Mine in Western Australia in 1990 when treating a hard lamproite ore (Maxton et al., 2003). Since then, major improvements have been made in wear lining design and this has reduced wear significantly, allowing HPGR to be a successful candidate for hard-rock mining applications.

The first commercial installation of HPGR in both the copper industry and large-scale hard-rock mining was completed at Cerro Verde in Peru in 2006 (Vanderbeek et al., 2006; Koski et al., 2011). Other examples of HPGR applications are PTFI Grasberg copper-gold operation in 2007, Anglo-platinum Mogalakwena in 2008, Newmont's Boddington gold expansion project in 2009, and the Cerro Verde expansion project expected in 2014 (Burchardt et al., 2011).

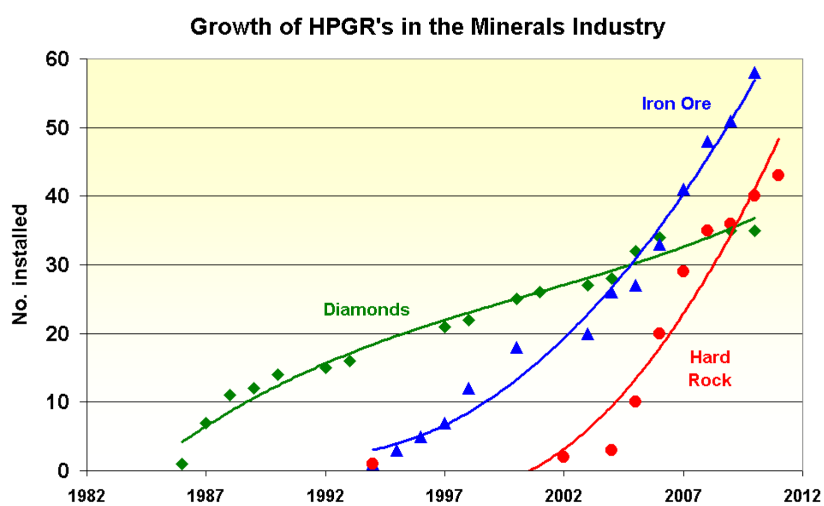

Figure 2.5 HPGR population growth (Burchardt et al., 2011)

Figure 2.5 shows the growth trend of HPGR units in the global minerals industry since 1986, and it can be seen that the application of HPGRs in hard-rock mining (such as gold, copper platinum, molybdenum, etc.) rapidly catches up to those in the diamond and iron ore industries. This trend is expected to continue in the near future.

2.2.2 HPGR operating parameters

The two most important operating parameters for HPGR operation are operating pressure and roll circumferential speed. These two parameters are directly linked to specific HPGR parameters, which have been established over the years and have become well-accepted in industry. Presently, there are no standard small-scale tests available to properly size and select an HPGR for commercial application. The only reliable way is to perform pilot-scale testing, and the obtained parameters will allow accurate prediction of energy and throughput for a given ore.

Specific throughput constant (m-dot)

The specific throughput constant ($\dot{m}$ or m-dot) represents the capacity of an HPGR with a roll diameter of 1 m, a length of 1 m, and a peripheral speed of 1 m/s for given feed materials. It is a key parameter used for sizing the roll dimensions for a given throughput, and allows comparison between any size rolls (Klymowsky et al., 2006). The m-dot is described by the following equation,

$$\dot{m} \text{ or m-dot} = M / (D * L * v) \qquad \text{(Equation 1)}$$

where M is the throughput [tph], D is the roll diameter [m], L is the roll length [m], and v is the peripheral speed [m/s] of the roll. Therefore, the unit for throughput constant is presented in ts/hm^3. Usually, the specific throughput rates on commercial HPGRs were found to be higher than those measured on lab scale or pilot scale units. As a consequence, 1:1 scale-up from pilot scale testing will normally result in a relatively conservative design for industrial HPGR operation (Burchardt et al., 2011).

Operating gap

The operating gap is defined as the smallest distance between the two rolls during the HPGR operation. The operating gap fluctuates because the floating roll moves dynamically. The actual size of the operating gap depends on several parameters, such as feed type, roll surface structure, and other operating conditions (Schoenert & Sander, 2002).

Specific pressing force

The specific pressing force (F_{SP}) is defined as the total force divided by the projected area of the roll. It is used as a normalized independent parameter for comparing the pressing force between different machine sizes (Bearman, 2006). The specific pressing force is expressed by the following equation,

$$F_{SP} = F / (D * L) \qquad \text{(Equation 2)}$$

where F is the hydraulic pressing force [N], D is the roll diameter [mm], and L is the roll length [mm]. The unit for specific pressing force is usually expressed as N/mm^2.

Net specific energy consumption

The net specific energy consumption (E_{sp}) refers to the net power input per ton of product, and thus is presented in kWh/t. Typical operating values lie at around 1 to 3 kWh/t (Morley, 2006b).The net specific energy consumption is used for motor sizing for an industrial unit. It can be calculated by the following equation,

$$E_{sp} = (P_t - P_i) / M \qquad \text{(Equation 3)}$$

where P_t is the total main motor power draw [kW], P_i is the idle power draw [kW], and M is the throughput [tph].

2.2.3 HPGR advantages

Whittles et al. (2006) claimed that the most energy-efficient form of comminution is slow compression of a single particle, followed by slow compression of a bed of particles. In the case of large-scale continuous application, slow compression of a particle-bed is a more practical and effective way of comminution than slow compression of a single particle (Schoenert, 1987). Thus, HPGR utilizes this concept to transfer energy directly from the rolls to the particle-bed and the particles themselves act as the stress transfer medium (Fuerstenau & Kapur, 1995). When the particle-bed is compacted, high inter-particle stresses generated by multiple-point contacts between particles lead to the fragmentation or deformation of particles. This unique process results in improved energy efficiency over traditional tumbling mills (Fuerstenau et al., 1991; Fuerstenau et al., 1996). As outlined in the literature, HPGR is generally reported to be between 10~50% more energy-efficient than conventional comminution circuits (Schoenert, 1987; Oestreicher & Spollen, 2006; Rosario & Hall, 2010; Drozdiak et al., 2011).

The reduced energy consumption provided by HPGR technology is able to offer significant operating cost savings. Unlike tumbling mills, HPGRs do not require steel grinding media. The elimination of steel grinding media also leads to operating cost savings. In comparison with conventional AG/SAG mills, the HPGR circuit throughput is less sensitive to changes in ore hardness. HPGR could significantly improve the economics of a project when taking into account ore hardness variability (Amelunxen et al., 2011).

Micro-cracking is another advantage of HPGR technology. Daniel (2007) has proved the existence of micro-cracking in the HPGR product. It was reported in much of the available literature that micro-cracks result in a reduction in the Bond ball mill work index of 10~25%, compared to a tertiary cone crusher product (Daniel, 2007a; Amelunxen et al., 2011; Patzelt et al., 2006). In the case of gold heap leaching, HPGR-treated ores show 5~25% increased extraction due to the presence of micro-cracks (Baum et al., 1997).

2.2.4 HPGR disadvantages

The most typical HPGR flowsheet requires a secondary crusher in closed circuit with a screen, and a screening circuit to classify the HPGR product prior to feeding the subsequent process. These specifications require an increased amount of materials handling equipment, thus higher capital costs are often associated with the circuit (Morley, 2010). For example, capital costs for the HPGR comminution circuit in Cerro Verde project were reported to be 23.5% higher than an equivalent SAG based circuit (Vanderbeek et al., 2006). Some flowsheet modifications, such as open circuit secondary crushing and open circuit grinding in HPGRs (in single pass or edge recycle mode), were often proposed to simplify the current "standard" HPGR circuit with reduced plant capital cost (Burchardt et al., 2011). However, currently, those potential approaches are often associated with other process limitations or disadvantages. It was projected that the next generation of HPGR flowsheets should have HPGR operated in open circuit, eliminating the need for auxiliary equipment (Morley & Daniel, 2009). This would result in significant capital cost savings and the reduction of the complexity of HPGR circuits. HPGR is a dry grinding technology. Therefore, when high moisture content is present in the feed, poor performance in terms of throughput and wear rates can be experienced. When processing wet material, the inability to produce a continuous autogenous layer on the roller surface can drastically decrease roller life (Morley, 2010).

2.2.5 HPGRs in hard-rock mining

Over the last few years, HPGRs began being adapted to hard-rock high-tonnage mining operations. Currently, the most functional HPGR flowsheet is based on secondary crushers operated in either normal or reversed closed circuit with dry screen, followed by tertiary

HPGRs in normal closed circuit with wet screen (Burchardt et al., 2011; Morley, 2006a). The widely recognized examples of HPGR installations in hard-rock mining are Cerro Verde (refer to Figure 2.6) and Newmont Boddington (refer to Figure 2.7).

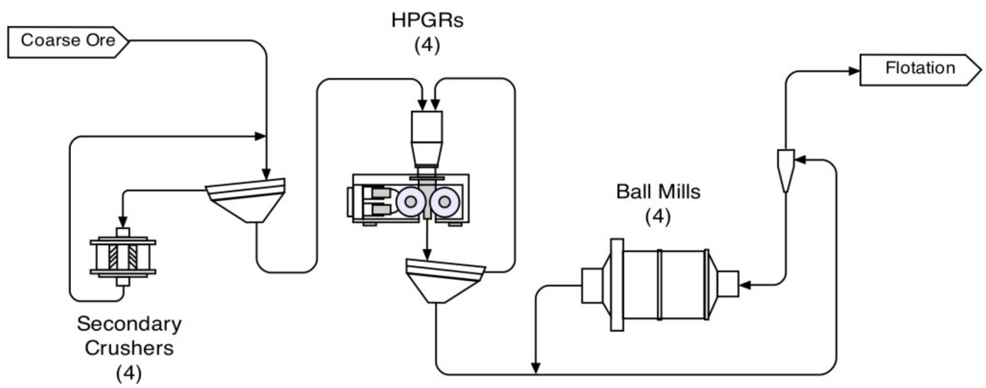

Figure 2.6 Cerro Verde flowsheet (Vanderbeek et al., 2006)

Cerro Verde currently processes 120,000 tpd of hard copper-molybdenum sulfide ores. The comminution circuit consists of one 60” x 113" primary gyratory crusher, four MP-1000 secondary cone crushers, four 2.4 m diameter (D) by 1.65 m length (L) 5.0 MW HPGRs, and four 7.3 m D x 11 m L 12 MW gearless ball mills (Koski et al., 2011). It was reported that HPGRs operating at Cerro Verde have reached 97% availability, and roll wear life exceeds 6,500 hours, higher than the expected 6,000 hours. Cerro Verde is currently undergoing a production expansion project with another eight HPGR units to be installed in 2014.

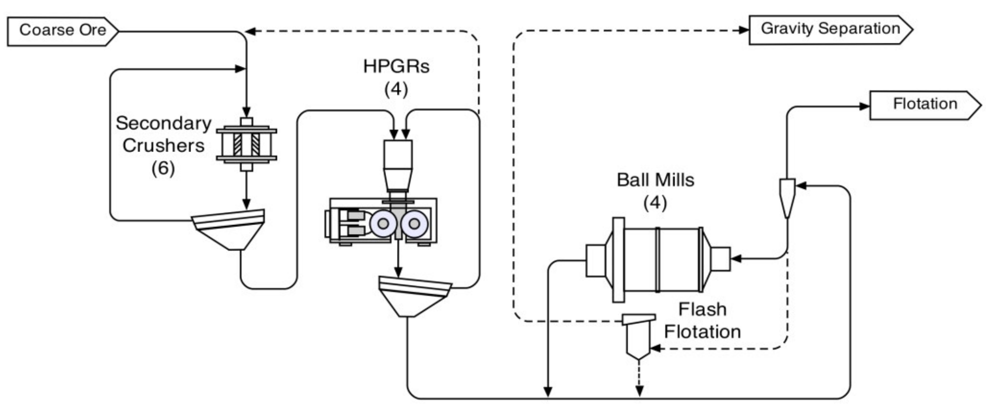

Figure 2.7 Boddington flowsheet (Hart et al., 2011)

Newmont Boddington gold project has a design capacity of 35 Mtpa or up to 105,000 tpd of very hard gold ores. The comminution circuit consists of two 60” x 113" primary gyratory crushers, six MP-1000 secondary cone crushers, four 2.4 m D x 1.65 m L 5.6 MW HPGRs, and four 7.9 m D x 13.4 m L 16 MW ball mills (Hart et al., 2011). The availability of HPGRs at Boddington is reported to be approximately 88% and the roll wear life lasts 4,400~5,700 hours, higher than the projected 4,250 hours.

2.3 Stirred media mills

As opposed to the tumbling mills, where motion is imparted to the charge via rotation movement of the grinding chamber, the stirred mill initiates the charge motion by the central-rotating shaft while the mill shell is stationary (Wills & Napier-Munn, 2006). Over the past two decades, stirred mill has rapidly developed as an efficient technology for fine (15~40 μm) and ultrafine (<15 μm) grinding in the minerals industry.

The history of stirred media mills dates back to the 1930s, when the mills were mostly used to condition the particle surface rather than for size reduction (Shi et al., 2009). As the technology continued developing during the 1950s, the first stirred mill for size reduction was introduced in Japan. In 1979, the tower mill was introduced into the U.S. market by Metso Grinding division for the grinding of limestone (Allen, 2011). At that time, the machine could not handle hard and abrasive metal ores. To adapt the mill to the metals industry, Svedala (acquired by Metso) fabricated a Vertimill® based on some modifications and improvements of the tower mill. Since then, over 300 Vertimills® have been sold around the world (Metso, 2012).

The horizontal stirred mill (IsaMill™) technology was first developed in the early 1990s under joint collaboration between Mount Isa Mines Ltd. of Australia (now a part of Xstrata) and Netzsch-Feinmahltechnik GmbH of Germany, to enable fine-grained deposits (Curry et al., 2005). This technology manufactured by Netzsch was originally used as a small batch grinding application for high-value manufactured products (Burford & Clark, 2007). Improvements have been made to increase the mill capacity, allowing for continuous operation for the metals industry. In 1994, the first full scale M3,000 IsaMill™ (1.1 MW) was installed at the Mount Isa Lead/Zinc Concentrator, which was then followed by four M3,000 IsaMills™ commissioned at the McArthur River Concentrator in 1995 (Curry et al., 2005). To date, over 100 IsaMills™ (over 190 MW) have been sold worldwide, and the treated materials include copper/gold, lead/zinc and platinum (Xstrata IsaMill, 2012b).

2.3.1 Technology overview

Generally, there are two main categories of stirred media mills based on the shell orientations, which are vertical mills, such as Vertimill® and stirred media detritor (SMD), or horizontal mills, such as IsaMill™ (Lichter & Davey, 2006).

Vertical stirred mill

As shown in Figure 2.8, the main examples of the vertical type of stirred mills are Vertimill® and stirred media detritor. The Vertimill® is equipped with a slow-rotating screw suspended into the grinding chamber, and is driven by a fixed speed motor (Metso, 2012). The Vertimill® is a gravity-induced type of stirred mill, where the grinding media is drawn up by the rotating screw and cascades over the edge of the screw, creating a downward flow of media along the mill perimeter (Allen, 2011). Slurry material, fed into the chamber at the top of the mill, spreads down along the mill perimeter and then is drawn back upward by the screw (Cleary et al., 2006). After continuous contact with grinding media, the particles are ground finer by the attrition breakage mechanism. The fine particles overflow the mill, while the coarse particles are re-circulated back to the mill. The Vertimill® can handle feed size as coarse as 6 mm and can produce products as fine as 20 μm, while providing higher efficiency compared to traditional ball mills (Allen, 2011).

Figure 2.8 Left: Vertimill®; right: stirred media detritor (Lichter & Davey, 2006)

In contrast, fluidized stirred mill technology such as SMD uses a rotational movement of shaft to create a fluidized media bed (Metso, 2012). The SMD consists of a vertical shaft mounted with pins, and operates at a relatively high speed (8 m/s tip speed). Slurry material is fed into the chamber at the top of the mill and passes through the fluidized bed, where size reduction is taking place under the high-intensity attrition breakage mechanism. Vertimill® is considered a more efficient technology at a coarser feed size, while SMD is an application for fine and ultrafine grinding (Metso, 2012). The SMD machine can provide products as fine as 98% passing 2 µm with feed size ranging from 100 to 15 µm (Lichter & Davey, 2006).

Horizontal stirred mill

Figure 2.9 shows the general components of the IsaMill™, a typical horizontal stirred mill that agitates fine inert media at high speed in a high power-intensive environment for efficient grinding. The IsaMill™ consists of 7-8 even-spaced grinding discs mounted on the shaft, rotating at a tip speed of 19-22 m/s (Anderson & Burford, 2006). A patented product separator is located at the discharge end of the mill as a self-classifier to retain the grinding media and coarse particles, while only allowing finished products to exit the mill. The IsaMill™ utilizes a high-intensity attrition breakage mechanism for size reduction (Arburo & Smith, 2009).

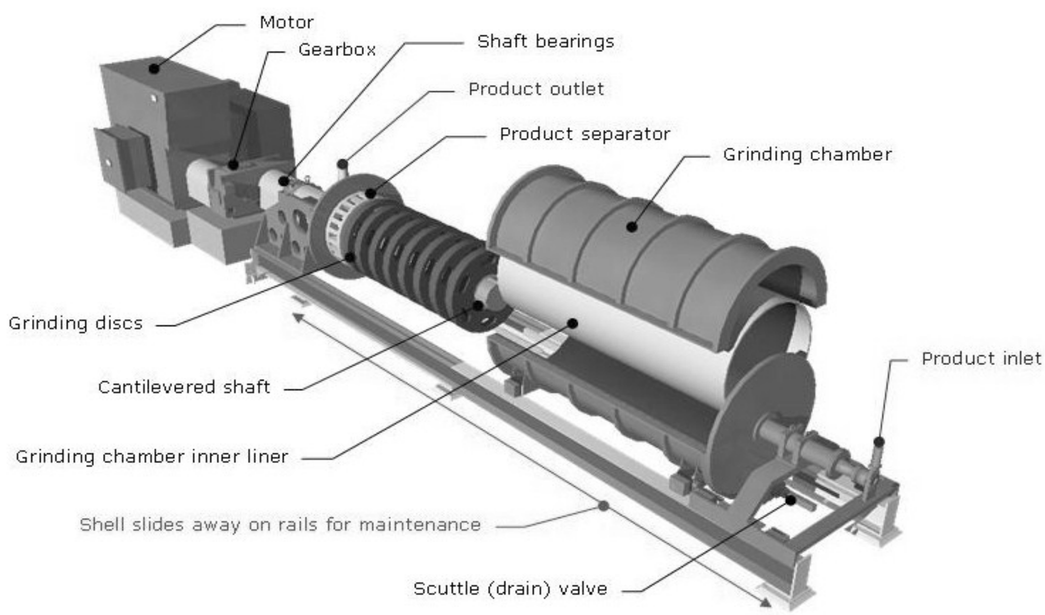

Figure 2.9 IsaMill™ main component (Arburo & Smith, 2009)

Figure 2.10 illustrates the process mechanism of IsaMill™. The shaft compresses the media between the grinding discs, creating 7-8 consecutive grinding stages to avoid short circulating in the mill. The slurry has to travel through each grinding chamber in a 'plug' flow pattern until it reaches the discharge end (Xstrata IsaMill, 2012a). Coarser particles are centrifuged out of the grinding chamber and into a zone of higher concentration of grinding media. This enables the IsaMill™ to operate in an open circuit, eliminating the pumping and piping system for recirculating streams (Arburo & Smith, 2009). Grinding media that reaches the product separator region is centrifuged towards the shell and then pushed back to the feed end with the coarse particles. The application of a dynamic classifier keeps the media inside the mill without the need for media retention screens, resulting in an increased throughput rate (Xstrata IsaMill, 2012a). The inert media environment provides clean mineral surfaces, resulting in improved performance of downstream flotation or leach processes (Arburo & Smith, 2009). The IsaMill™ produces a sharp product size distribution, which also assists in achieving a better performance in downstream (Pease, 2007).

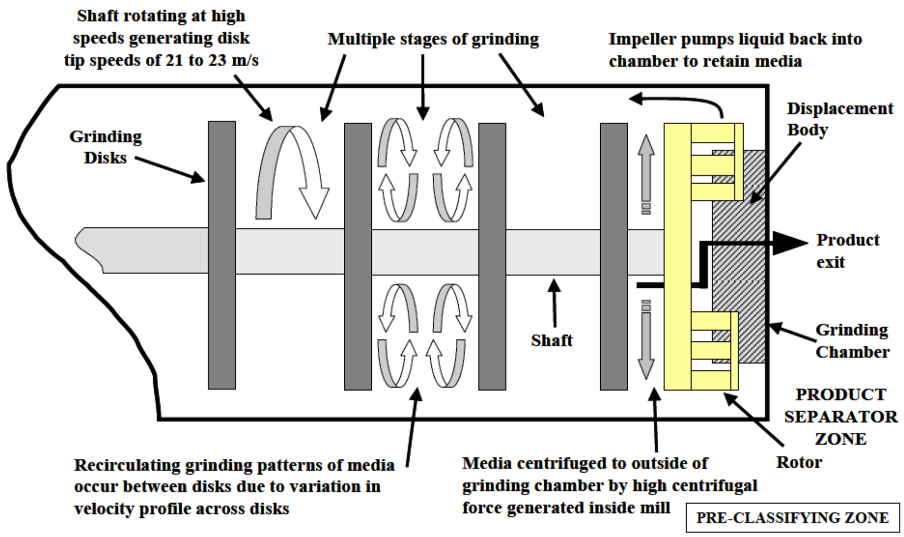

Figure 2.10 IsaMill™ grinding mechanism (Burford & Clark, 2007)

Thanks to the horizontal configuration, the grinding events in the IsaMill™ are evenly distributed throughout the grinding chamber, thus ensuring that the IsaMill™ scale-up is 100% direct from laboratory to full scale (Gao et al., 2002). Gao et al. (1999) determined the energy requirements for a commercial scale M4,000 mill based on the results obtained in a laboratory scale M4 mill, with a ratio of 1:1 scale-up. Curry et al. (2005) confirmed that the 1:1 energy scale-up also exists between an M4 mill and the largest IsaMill™ M10,000.

While the IsaMill™ technology has successfully demonstrated its ability for energy-efficient grinding in fine and ultrafine applications, the current IsaMills™ are making their way toward treating coarser materials with the development of increased grinding media size and increased mill capacity (Burford & Clark, 2007; Anderson et al., 2011; Larson et al., 2012). Table 2.2 summarizes a number of operation examples and laboratory trials with the IsaMill™ operating in coarse grinding.

Table 2.2 IsaMills™ in coarse grinding

Project	Model	Feed F_{80} [μm]	Target P_{80} [μm]	Media size [mm]
McArthur River	M10,000	300	55~60	5
Kumtor Gold trail	M10,000	135	62	2.5~3.5
Anglo Platinum pilot	M250	300	45	3.5
Teck Mesaba pilot	M20	340	75	6
Ernest Henry Mine	M10,000	300~350	45	6.5

2.3.2 Horizontal stirred mill operating parameters

There are several parameters that are critical when operating an IsaMill™ operation. Optimization of the operation is achieved by adjusting the operational parameters to produce a desired product with a reduction in energy and an increase in mill throughput. These parameters include the mill speed, feed density, flow rate and grinding media.

Mill speed

Although increasing mill speed can lead to an increase in centrifugal force, the experiments conducted by Larson et al. (2008) using an M4 mill (4-liter IsaMill™) showed that the mill speed has very little effect on energy efficiency. However, the mill speed has effect on the stress intensity, thus an increasing tip speed results in an increase in the stress intensity. The impeller speed also controls the breakage mechanisms of the mill, so that different ores can be ground by altering the speed (Kwade & Schwedes, 2002).

Feed density

Maintaining an appropriate feed pulp density is important to the energy efficiency of stirred mill operation. The pulp density is calculated by the weight percentage of the solids in the slurry. Pulp density that is too low results in lower energy efficiency; pulp density that is too high can also lower the energy efficiency, due to viscosity issues. Although there is an optimal feed density, it may depend on the material property and target grind size (Larson et al., 2008). In the case of sulphide ore, it is suggested that the pulp density should be maintained at between 50% and 65% solids to achieve good energy efficiency (Larson et al., 2008).

Volumetric flow rate

The volumetric flow rate is used in conjunction with the feed density and actual power draw to calculate the specific energy in kWh/t. Larson et al. (2008) claimed that changing the volumetric flow rate does not change the signature plot result, which means there is no effect on grinding energy efficiency. However, the flow rate has an effect on the residence time, which directly affects the size reduction.

Media volume

The media volume is defined as the volume of bulk media divided by the net volume of the mill (the shaft and discs volume subtracted from the total mill volume). The IsaMill™ can operate with a media volume of 80% while maintaining high grinding energy efficiency. Too-low media filling may cause insufficient grinding of the particles, thus the unbroken particles will build up and clog the mill (Larson et al., 2008). The excessive media filling may damage the mill lining and the agitator.

Media size

The grinding energy efficiency is mainly determined by the grinding media. As the media size decreases, the media surface area increases significantly, and the number of mechanical stress actions increases linearly. Thereby, the reduction of the media size is practical to reduce energy consumption (Lichter & Davey, 2006). However, when the media size is smaller than 0.8 mm, the benefit of decreasing media size becomes restricted (Gao et al., 1999). Consequently, it is critical to choose the optimal media size, which depends on the ability to match the media and the feed size. It is suggested that the optimum ratio of media size to a given feed size (80% passing) is approximately 20:1 for fine grinding (Mankosa et al., 1986; Yue & Klein, 2006).

2.3.3 Stirred mill energy saving

Improving the grinding energy efficiency is one of the main purposes of the development of stirred mill technology. Instead of rotating the entire grinding chamber like in a ball mill, the stirred mill only rotates the central agitator, which results in decreased energy requirements. Kwade and Schwedes (2002) stated that the stress intensity in a stirred mill can be expressed by the following relationship,

$$SI = d^3 * \rho * v^2 \qquad \text{(Equation 4)}$$

where SI is the stress intensity per media particle [N*m], d is the media diameter [m], ρ is the media density [kg/m^3], and v is the media velocity [m/s]. In an IsaMill™, enough energy in collisions can be achieved by increasing the media SG, media diameter or mill speed to break the feed. IsaMill™ is already operating at a very high speed, thus enabling it to treat coarse particles with small media. The original media had a low SG of 2.4 and small diameter (<1 mm), resulting in milling inefficiencies and limitation of feed size (Burford & Niva, 2008). With the improvements over the last 15 years, the grinding media with a higher SG of 3.7 and larger diameter (up to 6.5 mm) have advanced IsaMill™ to the point where it can readily treat up to 400 µm F_{80} feed (Larson et al., 2012).

With the combination of small grinding media and increased media velocity, stirred mill technology has been shown to improve the energy efficiency of grinding in particle sizes below an F_{80} of 150 µm, compared with traditional ball mills (Pease, 2007; Allen, 2011). The finer the product required, the more energy-efficient stirred mills will be than ball mills. In the case of coarser grinding comparison, work performed by Shi et al. (2009) showed that the vertical stirred mill achieved about 30% energy savings compared with ball mills when taking a coarse feed F_{80} of 3.35 mm and grinding to a P_{80} of 75 µm. David et al. (2011) investigated the energy requirements of an M4 IsaMill™ and a ball mill for processing a feed F_{80} of 400 µm to varying product sizes. As shown in Figure 2.11, the ball mill Levin tested showed slightly lower energy requirements for coarser grinds (P_{80}> 100 µm), but became less efficient with finer grinds. The intersection point may be shifted to coarser, if larger media is used to optimize the IsaMill™ process based on the 20:1 ratio of media size and feed size.

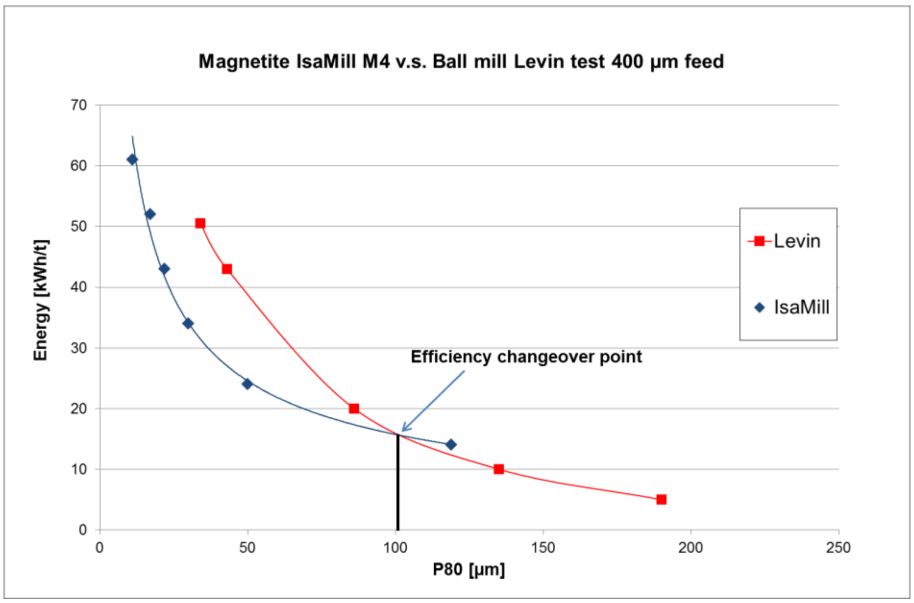

Figure 2.11 IsaMill™ / Levin test comparison (David et al., 2011)

Burford and Niva (2008) examined the energy efficiency between tower mills and the IsaMill™ and found that the IsaMill™ operated more effectively at finer sizes (<70 µm) while a tower mill became more efficient at coarser sizes. Although the tower mill operates more efficiently at coarser sizes, it has not been able to achieve a direct scale-up like the horizontal stirred mill (Pease et al., 2006).

2.4 HPGR - stirred mill circuit

Valery and Jankovic (2002) proposed the very first concept of HPGR and stirred mill combination in a single flowsheet to fully utilize the benefits of these two technologies. As displayed in Figure 2.12, a circuit with high-intensity blast, two-stage HPGR and a single stage Vertimill® was modeled as an alternative to the conventional SAG mill - ball mill circuit. The results from the simulation work indicated 45% energy savings provided by the novel circuit (Valery & Jankovic, 2002).

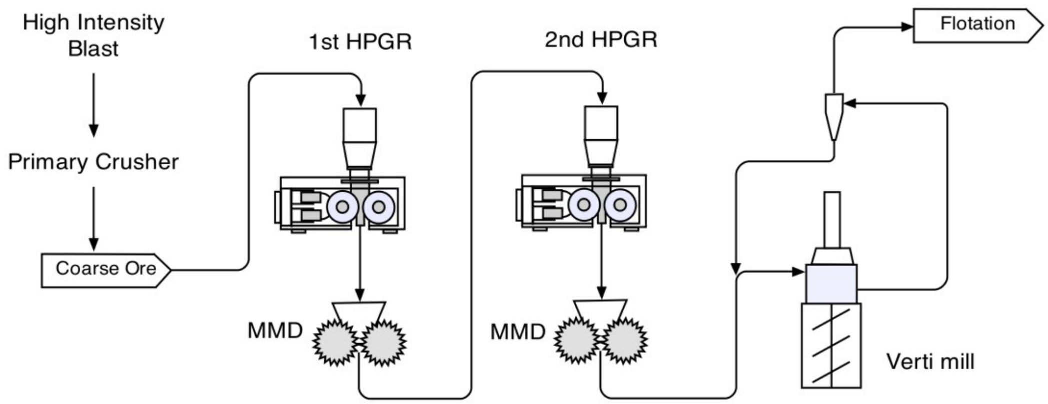

Figure 2.12 A proposed HPGR - stirred mill circuit (Valery & Jankovic, 2002)

Pease et al. (2006) proposed a hypothetical circuit of HPGR and IsaMill™ as a low-footprint, high energy-efficiency alternative, shown in Figure 2.13, which eliminates the need for tumbling mills. Pease (2007) then stated that further development in reducing the product size of HPGRs and/or increasing the feed size of IsaMill™ would enable this circuit for future comminution flowsheet design.

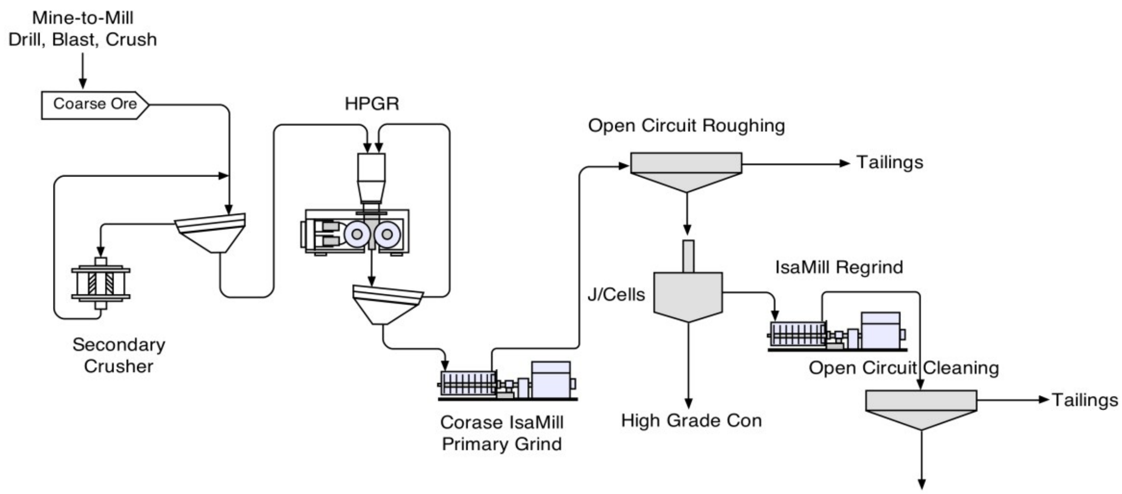

Figure 2.13 A proposed HPGR - IsaMill™ circuit (Pease, 2007)

The first pilot-scale HPGR - IsaMill™ circuit was operated continuously in the pilot plant at the Anglo Platinum Divisional Metallurgical Laboratory (Ayers et al., 2008). As shown in Figure 2.14, the HPGR - IsaMill™ circuit consists of a single HPGR in closed circuit with a dry screen at a cut size of 5 mm, followed by a wet screen at a cut size of 850 μm. The wet screen undersize was fed directly to an M250 IsaMill™ operating with 3.5 mm ceramic media for treating a coarse feed F_{80} of 300 μm and grinding to a product P_{80} of 45 μm. The operating results showed a higher total circuit specific energy consumption of ~80 kWh/t, compared to the average circuit energies of 30~35 kWh/t. The IsaMill™ operation was not considered optimized, thus larger diameter ceramic media and higher media density need to be investigated in order to improve the IsaMill™ efficiency.

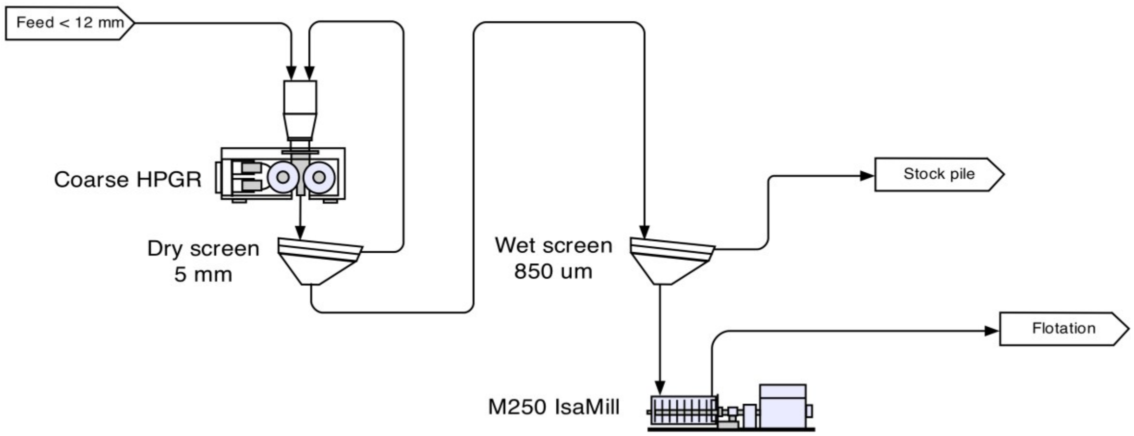

Figure 2.14 An HPGR - IsaMill™ circuit at Anglo Platinum pilot plant (Ayers et al., 2008)

Several publications have documented the effects of processing material through multiple passes of HPGR, and two passes were found to be most effective; a third pass continues to generate fines without substantially reducing the top size (Daniel, 2007b; Hilden & Powell, 2008). Drozdiak et al. (2011) concluded that a circuit would require two stages of HPGR comminution to achieve an acceptable feed size for coarse stirred milling.

Drozdiak et al. (2011) also conducted tests on pilot-scale testing of two stages of HPGR, followed by a horizontal stirred mill, to assess whether the first stage HPGR should be operated in closed circuit. As shown in Figure 2.15, two flowsheet options were examined based on energy consumption, as well as design and operating considerations. It was found that operating the first stage of HPGR in open circuit required less energy compared with operating in closed circuit with a screen (Drozdiak et al., 2011). Drozdiak et al. (2011) also demonstrated that the HPGR - stirred mill circuit is technically feasible, and the testing results showed that the novel HPGR - stirred mill circuit achieved an energy reduction of 16.7% and 9.2% over the conventional stage crushing - ball mill circuit and HPGR - ball mill circuit, respectively.

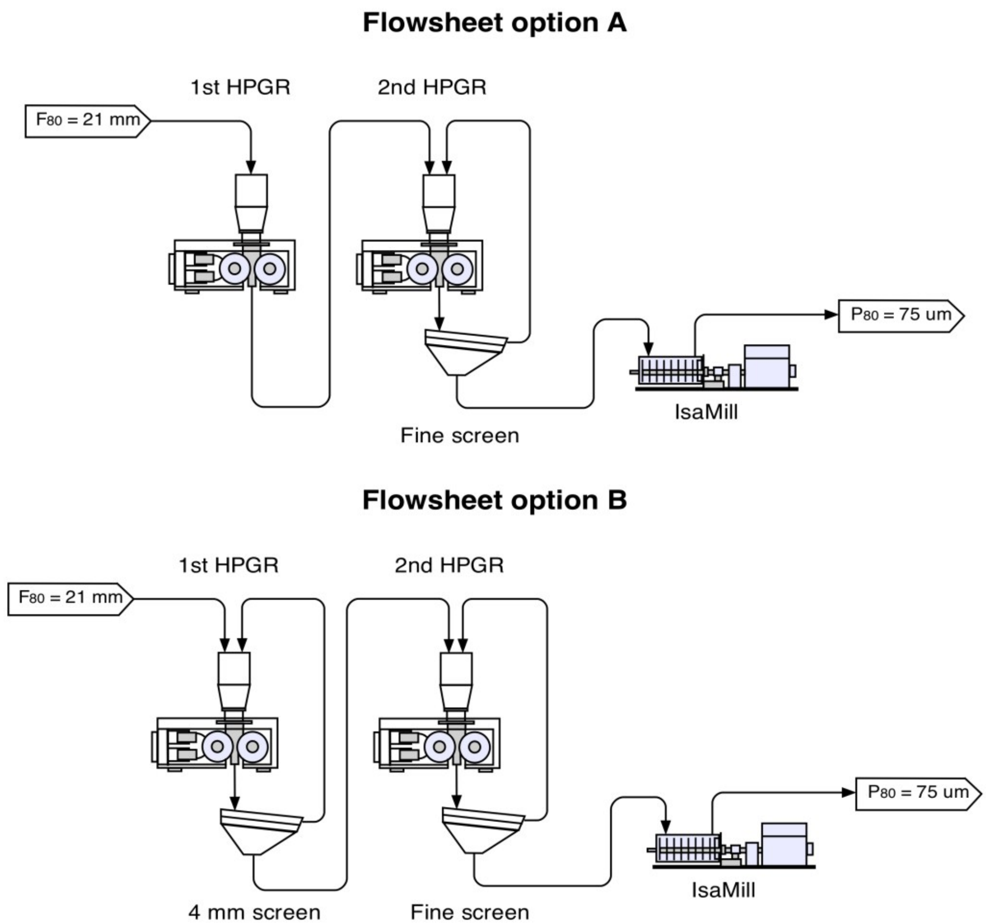

Figure 2.15 HPGR - IsaMill™ testing flowsheets (Drozdiak et al., 2011)

2.5 Summary of literature review

Both HPGR and stirred mill technology presents promising benefits, especially in energy savings and improved downstream performance. The combination of HPGR and stirred mill in a single flowsheet may serve as a potential energy-efficient comminution option in the future without tumbling mills. For the simulation of HPGR - stirred mill circuit, both machines are operating outside their respective industry optimal conditions. Continuous efforts have been made to bridge these two technologies. Based on the review of the literature, multiple passes of HPGRs to reduce the product size, and increasing the feed size of IsaMill™, are essential to enable the HPGR - stirred mill circuit. Two stages of HPGR are suggested, to prepare an acceptable feed size for coarse stirred milling. Utilizing large ceramic grinding media is the key to operating the coarse stirred milling efficiently.

Based on the literature review, a testing program involving a combination of laboratory scale testing and pilot-scale testing was developed to determine the appropriate design criteria for HPGR - stirred mill circuit. The key operating parameters and operating conditions for this testing program were identified in the literature. In order to address the objectives of this thesis, the JK SimMet® package was used as the tool for energy determination and flowsheet simulation, with the confirmation of plant survey data and the Bond-based benchmarking method.

CHAPTER 3: EXPERIMENTAL PROGRAM

This chapter describes the methodology and the equipment used to address the objectives of this research. The main objective was to evaluate the existing AG/SAG mill - ball mill circuits and the proposed HPGR-based comminution circuits. Lab and simulation work were carried out for the evaluation of existing circuits. In the case of the proposed circuits, a pilot-scale testing program was carried out on four sets of coarse ore samples from two copper-molybdenum operations in British Columbia, Canada. The battery limits for this comminution circuit evaluation are the coarse ore sample as the feed to the AG/SAG circuit and the ball mill circuit product as the feed to the flotation circuit (ball mill cyclone overflow). The main auxiliary equipment (conveyors, vibrating screens and feeders) will be taken into account to determine the least gross energy requirements. Further detail for the test flowsheet can be found in Appendix A.

3.1 Methodology

The major components of the experimental program are shown in Figure 3.1, for the evaluation of each comminution circuit outlined in this document. Sampling surveys around the grinding circuits were conducted by the mill technical groups from the two operations. Data representative of continuous mill operation, directly preceding mill shutdown and sample collection, was analyzed to confirm process stability and, subsequently, to determine the operating work index of the existing circuit. The collected bulk sample was analyzed using established comminution laboratory testing methodologies (such as Bond grindability tests, particle size analysis, etc.), characterizing the properties of the ore and slurry for modeling and simulation of the circuit. The modeling and simulation of the existing circuit, using JK SimMet® software, was carried out using known equipment data, process data, and ore characteristic parameters as inputs (Napier-Munn et al., 1996). The collected coarse ore sample was prepared for pilot-scale HPGR and stirred mill testing to determine the key operating parameters for flowsheet design and power-based calculations. Recycle tests were performed to simulate the HPGR performance in closed circuit with a screen and to determine the associated specific energy values. The main auxiliary equipment (conveyors, vibrating screens and feeders) was sized and added to the flowsheet to determine the gross energy consumption. Ultimately, the simulation and test results allowed for the direct comparison of the energy and costs of the three circuits: a conventional AG/SAG mill - ball mill circuit at the existing operation, an HPGR - ball mill circuit, and a novel HPGR - stirred mill circuit. For each analysis, grind size requirements and plant throughput were equated so that the comparison was based solely on the specific comminution energy, expressed as kWh/t.

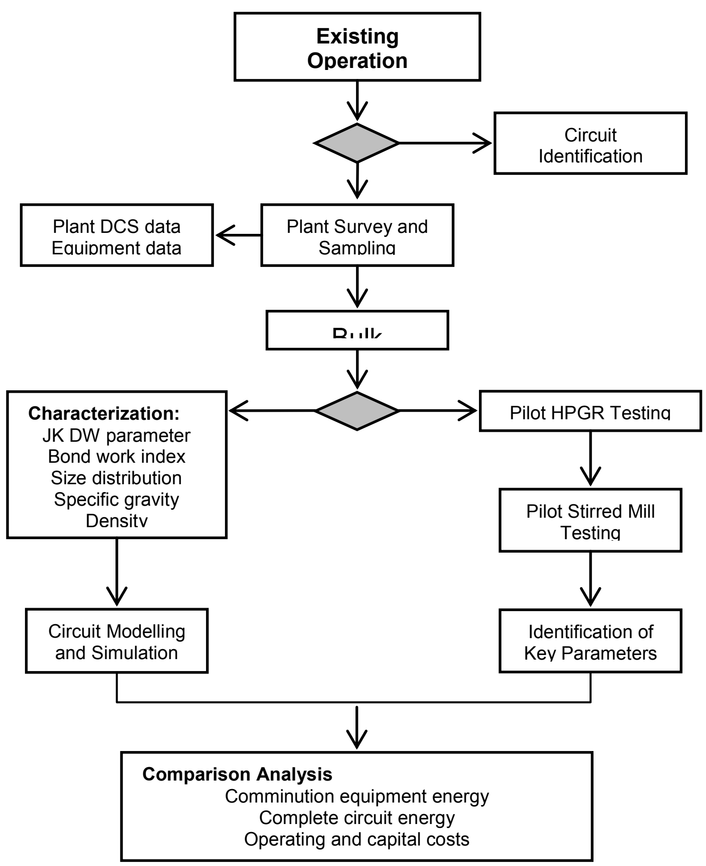

Figure 3.1 Experimental program break-down

3.2 Circuits description

3.2.1 Existing SAB circuits

Figure 3.2 and Figure 3.3 present the current process configurations of circuit A and circuit C, referred to as SAB-type comminution circuits. Circuit A is comprised of one 9.75 m D x 4.72 m L (32 x 15.5 ft) SAG mill and two 5.03 m D x 7.01 m L (16.5 x 23 ft) ball mills. The SAG mill product is pumped to a splitter and then onto two stationary screens. The SAG mill is driven by two 3350 kW (4400 hp) fixed-speed synchronous motors. Each ball mill is equipped with a single 3350 kW (4400 hp) fixed-speed synchronous motor.

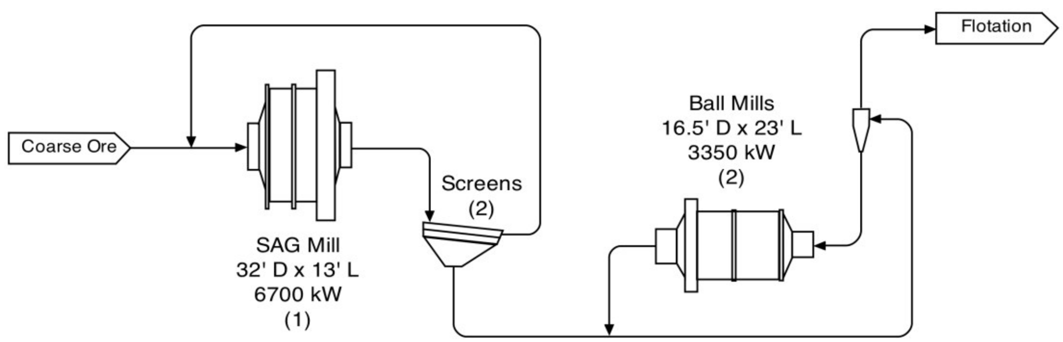

Figure 3.2 Schematic of circuit A

Circuit C consists of a 10.36 m D x 4.88 m L (34 x 16 ft) SAG mill and two 5.03 m D x 8.23 m L (16.5 x 27 ft) ball mills. The SAG mill discharge is pumped onto a single stationary screen and the undersize is split into two ball mills. The SAG mill is driven by two 4700 kW (6250 hp) variable-speed direct-current motors. One ball mill is powered by a 3350 kW (4500 hp) synchronous motor, and the other by 4700 kW (6250 hp) variable-speed direct-current motor.

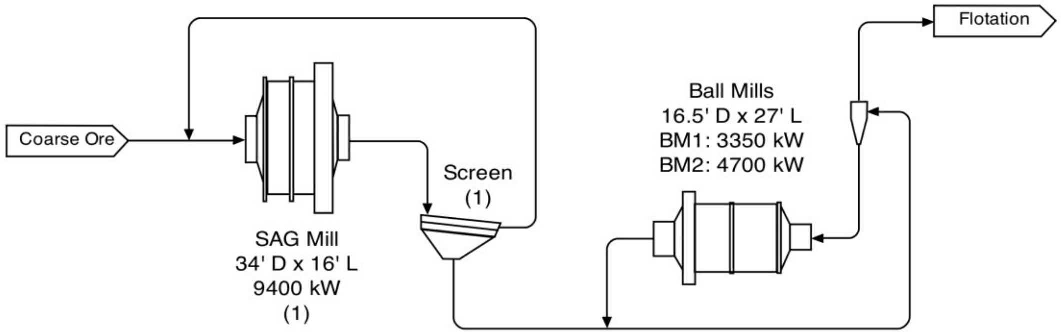

Figure 3.3 Schematic of circuit C

3.2.2 Existing AGBC circuit

Figure 3.4 shows the current process configuration of circuit D, referred to as an AGBC-type comminution circuit. This circuit consists of one 10.36 m D x 4.57 m L (34 x 15.8 ft) AG mill with a 2.13 m (7 ft) Symons short head crusher and a single 5.03 m D x 8.83 m L (16.5 x 29 ft) ball mill. The AG mill is driven by twin synchronous fixed-speed 3300 kW (4400 hp) motors. The pebble crusher has a maximum power draw of 261 kW (350 hp). The single ball mill is powered by a 4100 kW (5500 hp) synchronous motor.

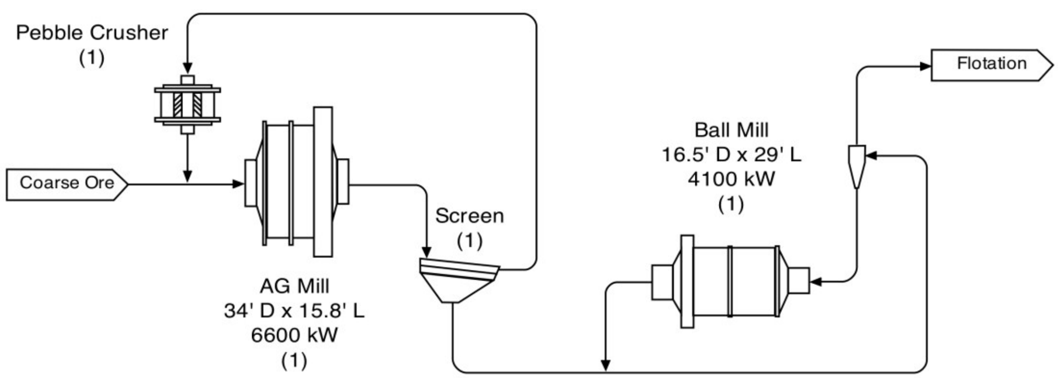

Figure 3.4 Schematic of circuit D

3.2.3 Existing SABC circuit

Figure 3.5 shows the current process configuration of circuit H, commonly referred to as a SABC-type comminution circuit. This circuit is comprised of one 9.75 m D x 4.57 m L (32 x

15 ft) SAG mill with a HP 500 pebble crusher, and two 5.03 m D x 9.14 m L (16.5 x 30 ft) ball mills. The SAG mill discharge is pumped onto a single stationary screen and the undersize is split into the two ball mills. The SAG mill is driven by two 4100 kW (5500 hp) quadramatic GE motors. Each ball mill is powered by one 4100 kW (5500 hp) quadramatic motor.

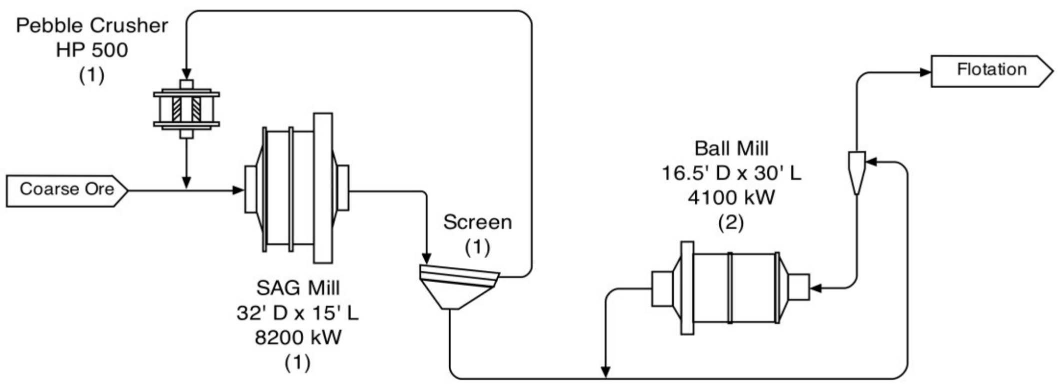

Figure 3.5 Schematic of circuit H

3.2.4 HPGR - ball mill circuit

The proposed HPGR - ball mill circuit (refer to Figure 3.6) comprises a secondary crushing circuit prior to a HPGR circuit, and followed by a ball mill circuit. The energy requirements for the secondary crushing stage and the ball mill circuit were determined using the Bond based method. Energy values obtained from pilot HPGR testing and laboratory testing were combined to determine the specific energy requirement for this circuit.

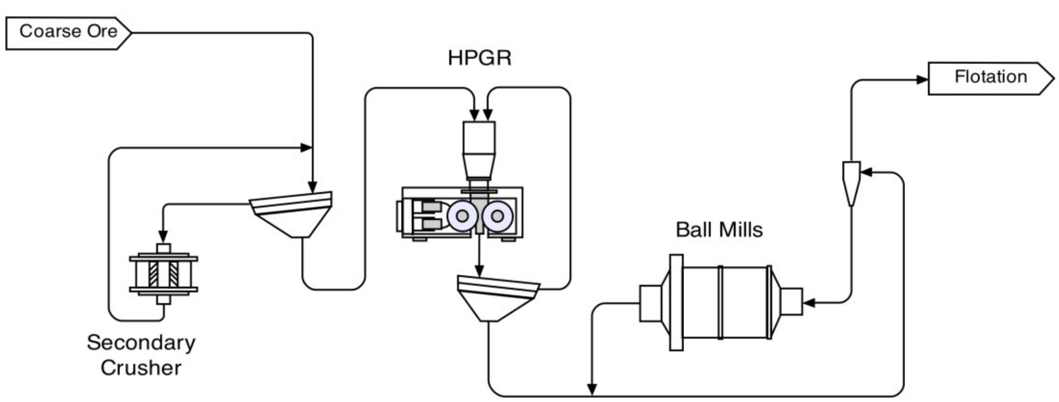

Figure 3.6 Proposed HPGR - ball mill circuit schematic

3.2.5 HPGR - stirred mill circuit

The proposed HPGR - stirred mill circuit is comprised of a secondary crushing circuit prior to an open HPGR circuit, and followed by a second HPGR in closed circuit to generate finer feed for high-speed stirred milling (refer to Figure 3.7). The energy requirements of the secondary crusher were determined using the Bond based method. Energy values obtained from pilot HPGR testing, stirred mill testing, and laboratory testing were combined to calculate the total specific energy requirement for this proposed circuit.

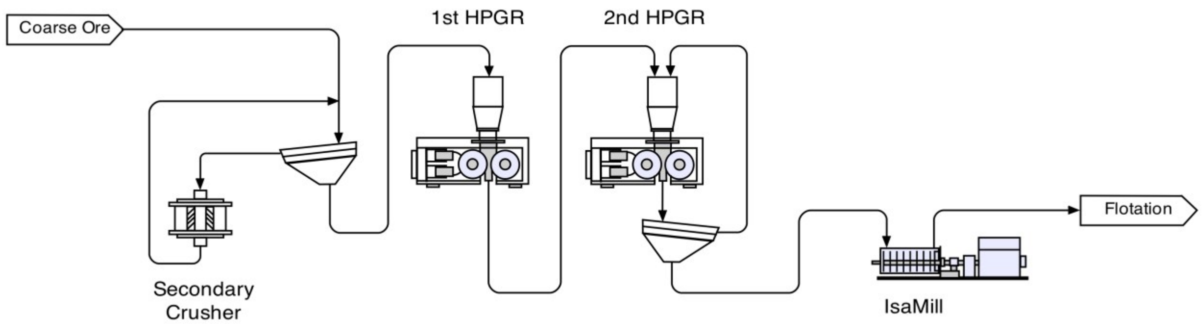

Figure 3.7 Proposed HPGR - stirred mill circuit schematic

3.3 Sample description

Coarse ore samples from each comminution circuit were received at UBC for the testing program. Four sets of samples were collected at the existing AG/SAG mill circuit feed belts during the sampling surveys. The existing circuits were labelled as “A”, “C”, “D” and “H” circuit after the abbreviation of the project names. It is important to note that there were not case B or case E, only these four cases were evaluated in this study. Table 3.1 presents the sample description, and the particle size distributions of the samples are shown in Figure 3.8. The composite of each sample was submitted to external labs for a full suite of grindability tests, including the JKTech drop-weight test and the Bond work indices grindability tests. The results are summarized in Table 3.2. Samples A, C and D would be considered moderately soft based on the results from several test protocols, while sample H would be considered much harder.

Table 3.1 Sample description

Sample ID	Origin	Ore type	Circuit type	Mass, [kg]	SG	% solids	F_{80}, [mm]
Sample A	Case A	Cu porphyry	SAB	~800	2.6	98.6	108.3
Sample C	Case C	Cu porphyry	SAB	~1000	2.6	99.4	91.5
Sample D	Case D	Cu porphyry	AGBC	~400	2.6	97.9	95.0
Sample H	Case H	Cu porphyry	SABC	~1000	2.6	97.0	64.8

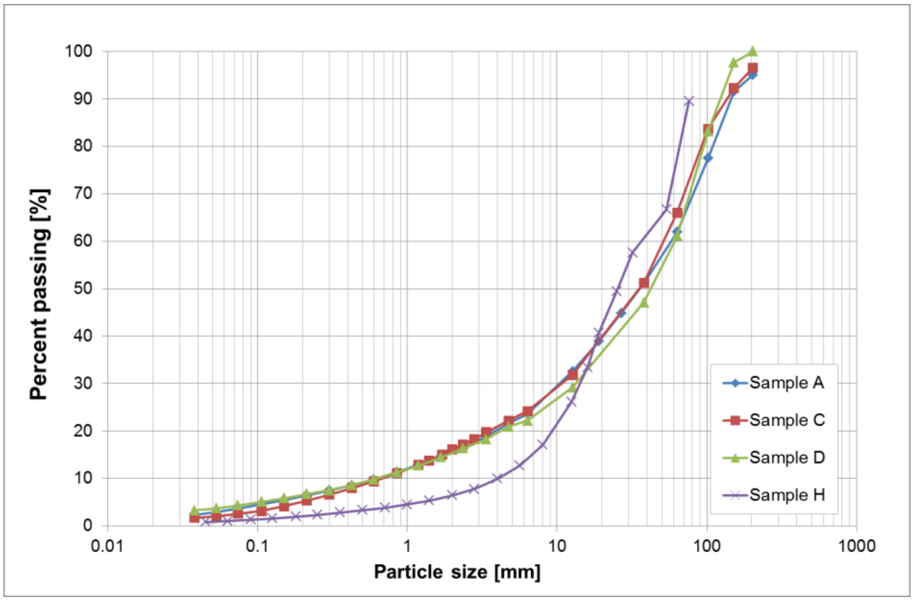

Figure 3.8 Particle size distributions of received samples

Table 3.2 Summary of ore characteristic test results

Grindability Tests		**Units**	**A**	**C**	**D**	**H**
Bond crusher work index	C_{Wi}	[kWh/t]	8.09	10.6	7.96	
JK Drop weight test	A	[-]	63.7	63.6	61.3	57.90
	b	[-]	1.02	1.02	1.21	0.54
	A x b	[-]	65.0	64.9	74.2	31.3
	t_a	[-]	0.45	0.31	0.58	0.59
Bond rod mill work index	R_{Wi}	[kWh/t]	10.3	12.3	11.1	
Bond ball mill work index	B_{Wi}	[kWh/t]	13.8	13.6	13.8	18.0
Bond abrasion test	Ai	[kWh/t]	0.328	0.267	0.210	

3.4 Equipment and procedure

3.4.1 High pressure grinding roll

A pilot-scale HPGR (refer to Figure 3.9), installed at the UBC NBK Institute of Mining Engineering, was manufactured by Koeppern Machinery Australia for the purpose of pilot-scale testing. This pilot unit was designed to provide operating data for sizing and selection of industrial machines. The rolls are fitted with Hexadur® WTII roll-wear protection, specifically designed by Koeppern for comminution of highly abrasive minerals. Table 3.3 summarizes the specifications of this HPGR unit.

Figure 3.9 Pilot HPGR installed at UBC

Table 3.3 Pilot-scale HPGR specifications

Parameter	Units	Value
Roll diameter	[mm]	750
Roll length	[mm]	220
Main motor power	[kW]	200
Maximum pressing force	[kN]	1600
Maximum specific pressing force	[N/mm^2]	8.5
Maximum roll speed	[m/s]	1.57
Wear surface	[-]	Hexadur® WTII

The procedure of a HPGR pilot test is described as follows,

1. screen the sample on the laboratory vibrating screen (32 mm aperture panel);
2. crush the screened oversize material until 100% passing 32 mm screen;
3. homogenize all -32 mm sample with the rotary splitter and obtain a representative sub-sample for feed characterization (moisture, PSD, etc.);
4. split samples into one 45-gallon drum (~300 kg) for one individual HPGR test;
5. adjust the test parameters, such as specific pressing force, feed moisture, and closed circuit testing, and then start the test and record the logged experimental data;
6. collect waste material during unstable operation and collect center product and edge product during stable operation;
7. weigh collected waste material, center product and edge product separately;
8. obtain representative sub-samples of center product and edge product, respectively;
9. determine the PSDs of the center product and edge product;
10. combine waste material, and remaining center and edge products;
11. screen the combined material on the laboratory vibrating screen;
12. repeat steps 5 to 12 using the combined screened oversize material and a portion of fresh feed when performing a closed circuit testing;
13. repeat steps 4 to 10 when performing a pressure or moisture testing.

During a test, the center portion is finer than the edge portion due to the edge effect. A diverter gate was installed on the end of the product conveyor to separate the product into approximately 30% edge product and 70% center product. In industrial units, the proportion of center and edge product is normally 10% edge and 90% center. Thus, all of the HPGR PSDs reported in this document account for the scaling of edge and center PSDs at a ratio of 1:9. After each pilot test, the logged data, in combination with machine data and sample data, allowed the calculation and determination of HPGR operational parameters for HPGR sizing and selection.

3.4.2 Horizontal stirred mill

A Netzsch M20 horizontal stirred mill (refer to Figure 3.10) was upgraded and configured with grinding discs to IsaMill™ specifications, based on recommendations from Xstrata Technology. Stirred mill testing was carried out to generate a log-log graph of energy input and product size, referred to as a signature plot. Signature plot tests are an established method for sizing IsaMills^{TM} based on laboratory scale test results; a scale-up ratio of 1:1 is associated with the method (Gao et al., 1999). Table 3.4 summarizes the specifications of the stirred mill unit.

Figure 3.10 M20 stirred mill installed at UBC

Table 3.4   M20 stirred mill specifications

Parameter	Units	Value
Total mill capacity	[L]	20
Effective mill volume	[L]	18.8
Main motor power	[kW]	18.6
Pump motor power	[kW]	1.5
Maximum Mill flowrate	[L/min]	~ 25
Maximum shaft speed	[rpm]	~ 1200
Volume of mix tank	[L]	180
Tank agitator power	[W]	250

The procedure of a signature pilot test is described as follows,

1. run the mill empty to warm up the machine, and measure the no-load power draw;
2. add ~100 kg dry solids and water into mix feed tank to make up the desired pulp density, and recirculate the slurry via the pump for mixing;
3. adjust the feed pump to the desired flowrate, and collect feed sample for PSD and density measurement;
4. add ceramic media to mill chamber to the desired charge volume;
5. start the mill at the desired speed, and start recording power draw and other data;
6. pass the slurry through to the mill, and discharge the product into the product tank;
7. collect product sample at the midway point of the pass for PSD and density measurement;
8. when finished one pass, switch the valves so the product tank becomes the feed tank;
9. repeat steps 6 to 8a select number of times.

After each signature plot test, a signature plot graph was plotted based on the recorded data, showing the relationship between energy input and product size. The specific energy requirement for an IsaMill™ in reducing particles to a desired product size could be extrapolated from the signature plot graph.

3.4.3 Other equipment

Laboratory-scale jaw and gyratory crushers were used to prepare the ore sample to a top size of 32 mm for HPGR testing. A rotary splitter was used to homogenize and split large samples. A 40-inch Sweco® vibrating screen was used as the process screen for the pilot-scale work. The particle size analysis work was carried out on dry and wet screen shaking apparatuses, according to the ATSM standard screening protocol. A standard Bond ball mill was used to determine the Bond ball mill work index for HPGR product.

Figure 3.11 Other equipment

CHAPTER 4: PILOT HPGR - STIRRED MILL TESTING AND RESULTS

The following chapter describes the HPGR - stirred mill pilot testing program and the test results. The pressure sensitivity tests were conducted to determine a suitable specific pressing force of HPGR process for the supplied sample. Closed circuit tests were performed to evaluate how comminution performance would be affected by operating the HPGR in closed circuit. A complete summary of HPGR test results, operating data and sample data can be found in Appendix B. Standard Bond ball mill tests were carried out on the HPGR product samples to determine the Bond ball mill work indices, and the results can be found in Appendix C. A complete summary of IsaMill™ signature plot test results, operating data, and sample data can be found in Appendix D.

4.1 HPGR feed samples

The four sets of samples received were screened and crushed to a top size of 32 mm for HPGR testing. They were then homogenized in bulk, and split into drums using a rotary splitter. For each sample set, a representative sub-sample was taken for determination of the particle size distribution, Proctor density and moisture content. Table 4.1 summarizes the HPGR feed material parameters. The HPGR feed PSDs are shown in Figure 4.1.

Table 4.1 HPGR feed material parameters

Item Description	Units	A	C	D	H
Moisture	[%]	1.4	0.6	2.1	3
Bulk Density	[t/m^3]	1.89	1.86	1.62	1.70
Proctor Density	[t/m^3]	2.12	2.08	2.08	2.10
F_{100}	[mm]	32	32	32	32
F_{80}	[mm]	19.7	21.9	21.9	23.6
F_{50}	[mm]	8.3	10.7	11.5	14.2

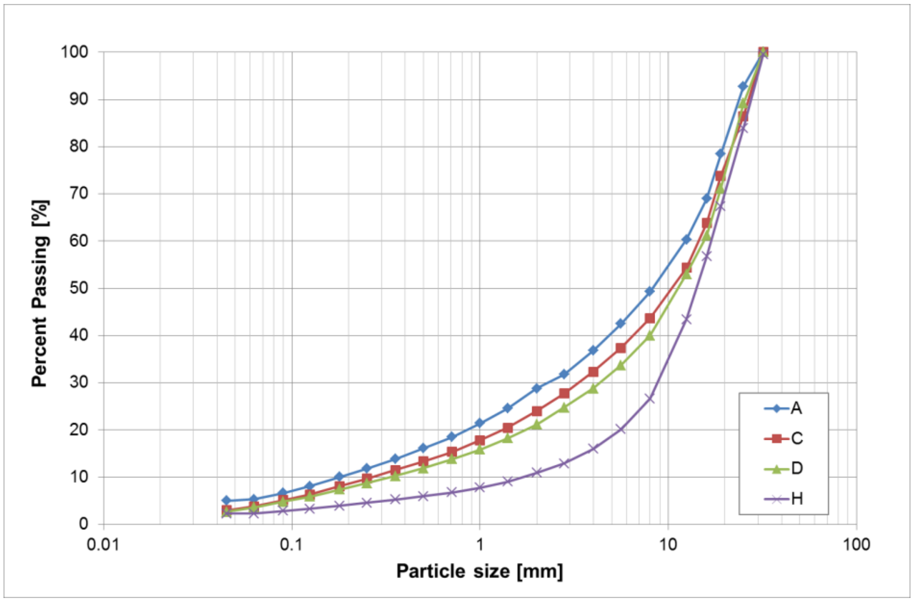

Figure 4.1 Particle size distributions of HPGR feed samples

4.2 HPGR testing results

Currently, pilot-scale testing is the only reliable approach for HPGR scale-up. HPGR pilot testing was required to provide the design parameters required to size the machine, and to determine the specific energy consumption and the target PSD for a commercial HPGR operation. Throughout this research, there were 19 pilot-scale HPGR tests performed. Table 4.2 is a reference legend for all HPGR tests, and the detailed test results are summarized in Table 4.3. The values included in the summary are considered to be critical indicators of comminution performance.

Table 4.2 HPGR tests reference legend

Test No.	Feed source	Test tag
A101	Screened & crushed sample A	Pressure test
A102	Screened & crushed sample A	Pressure test
A201	Combined A101 & A102 product	Closed circuit test cycle 1
A202	Screened A201 product plus fresh A201 feed	Closed circuit test cycle 2
A203	Screened A202 product plus fresh A201 feed	Closed circuit test cycle 3
C101	Screened & crushed sample C	Pressure test
C102	Screened & crushed sample C	Pressure test
C103	Screened & crushed sample C	Pressure test
C201	Combined C101, C102 & C103 product	Closed circuit test cycle 1
C202	Screened C201 product plus fresh C201 feed	Closed circuit test cycle 2
C203	Screened C202 product plus fresh C201 feed	Closed circuit test cycle 3
D101	Screened & crushed sample D	Pressure test
D201	Test D101 product	Closed circuit test cycle
H101	Screened & crushed sample H	Pressure test
H102	Screened & crushed sample H	Pressure test
H103	Screened & crushed sample H	Pressure test
H201	Combined H101, H102 & H103 product	Closed circuit test cycle 1
H202	Screened H201 product plus fresh H201 feed	Closed circuit test cycle 2
H203	Screened H202 product plus fresh H201 feed	Closed circuit test cycle 3

Table 4.3 Summary of HPGR test results

Test No.	F_{SP} [N/mm^2]	M-dot [ts/hm^3]	E_{SP} net [kWh/t]	Scaled HPGR product (90% Center, 10% Edge)	
				P_{80} [mm]	P_{50} [mm]
A101	4.0	259	1.70	4.90	1.31
A102	3.0	257	1.37	6.30	1.91
A201	4.0	178	2.66	1.95	0.46
A202	4.0	184	2.45	1.73	0.50
A203	4.0	191	2.22	1.67	0.54
C101	4.0	259	1.69	6.63	1.55
C102	3.0	266	1.23	6.54	1.58
C103	2.5	285	1.02	7.97	1.87
C201	4.0	157	2.87	2.57	0.91
C202	4.0	188	2.14	2.00	0.74
C203	4.0	208	1.87	1.88	0.76
D101	3.0	244	1.55	4.70	1.17
D201	4.0	142	2.90**	1.71	0.55
H101	2.5	194	1.51	7.37	3.18
H102	3.0	184	1.89	6.50	3.00
H103	4.0	172	2.56	6.26	2.66
H201	3.0	217	1.58	4.44	1.98
H202	3.0	213	1.53	3.97	1.88
H203	3.0	222	1.25	3.83	1.75

4.2.1 Pressure sensitivity tests

The pressure sensitivity testing was aimed at determining the appropriate specific pressing force to achieve the optimal balance of size reduction, specific throughput, and specific energy consumption. Different specific pressing forces of 2.5 N/mm^2, 3.0 N/mm^2 and 4.0 N/mm^2 were chosen for the pressure sensitivity testing for sample C and sample H. Tests for sample A were only performed at 3.0 N/mm^2 and 4.0 N/mm^2 due to the insufficient sample quantity.

The comparison of F_{80}/P_{80} and F_{50}/P_{50} reduction ratios at different specific pressing forces is shown in Figure 4.2. In the case of test A, both F_{80}/P_{80} and F_{50}/P_{50} were observed to increase when a higher specific pressing force was applied. In the case of tests C and H, both F_{80}/P_{80} and F_{50}/P_{50} increased with increasing specific pressing force from 2.5 to 3.0 N/mm^2. However, there was very little change above 3 N/mm^2.

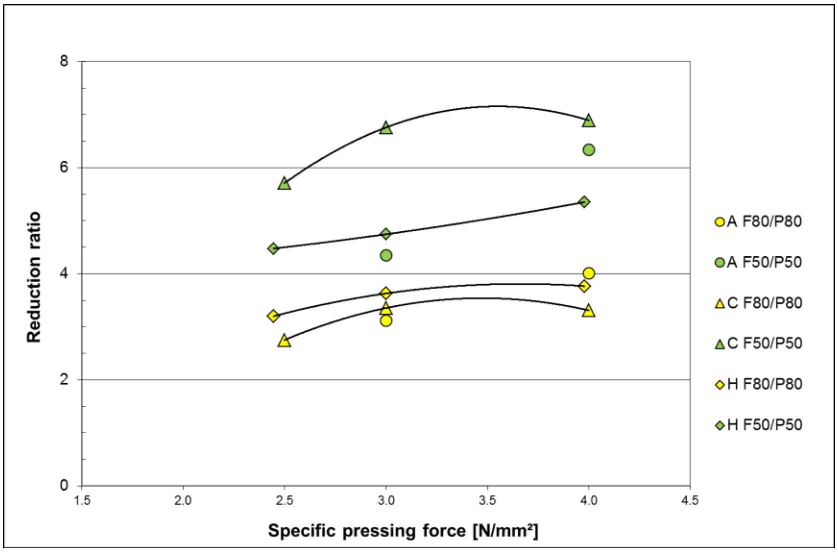

Figure 4.2 Effect of specific pressing force on size reduction ratio

The effect of pressure on specific throughput constant is shown in Figure 4.3. It appears that operating with higher specific pressing force slightly reduced the specific throughput constant rate due to the reduced operating gap.

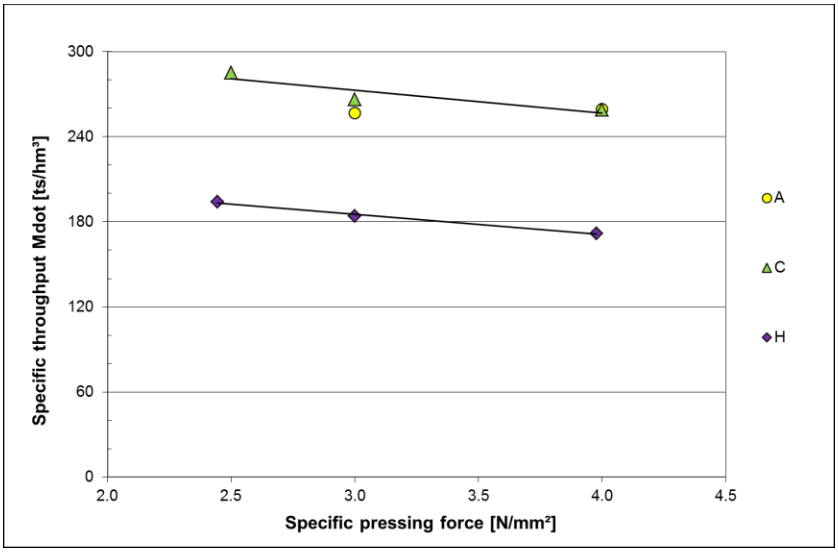

Figure 4.3 Effect of specific pressing force on specific throughput constant

A linear relationship between specific pressing force and specific energy was observed in the pilot-scale HPGR testing, as shown in Figure 4.4. The graph shows that a higher specific pressing force resulted in greater energy consumption.

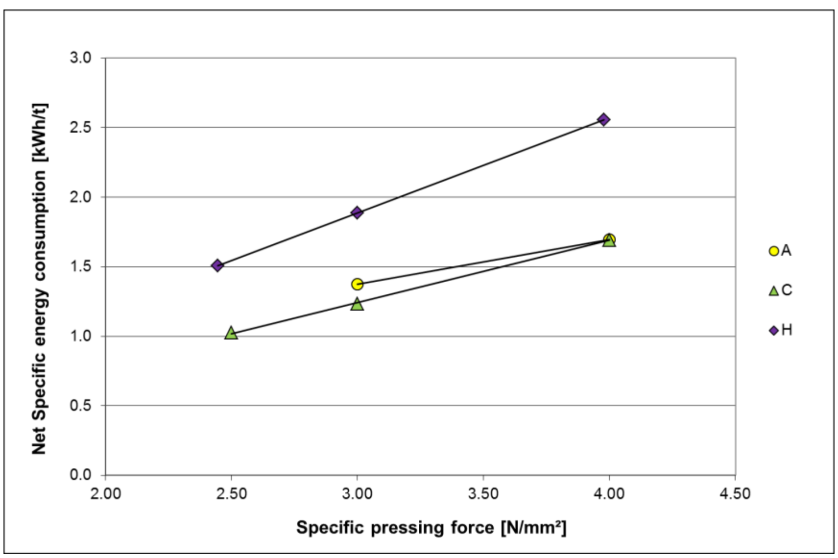

Figure 4.4 Effect of specific pressing force on net specific energy consumption

Based on the initial results from the pressure sensitivity tests for samples A and C, it appears that the optimal specific pressing force is about 3.0 N/mm^2 for the first stage HPGR operation, and 4.0 N/mm^2 for the subsequent closed circuit. Although there was not enough material available for test D, the findings from tests A and C would be applicable for sample D, because they were from the same operation, and can thus be expected to have similar material response to HPGR comminution. In the case of test H, a specific pressing force of 3.0 N/mm^2 was nominated as being most suitable for both the first stage open-circuit and the second stage closed-circuit HPGR operation.

4.2.2 Closed circuit testing

The product fineness of a closed HPGR circuit depends mainly on the screen aperture size. The smaller the screen aperture size, the higher the HPGR circulating load. To evaluate the effect of the closed circuit configuration of HPGR on comminution performance, a closed circuit testing program was carried out using 0.71 mm screen aperture size for tests A, C and H. However, there was insufficient quantity of material available to perform the closed circuit testing for sample D.

Since the second stage HPGR was operating with wet screening, water was added into the fresh product from the previous stage to prepare the feed material for closed circuit testing. Saturated tests were performed to determine the potential moisture content for the screen oversize in a closed-circuit operation. Approximately 5% moisture was determined and adjusted for each sample set, based on the measured saturated oversize moisture of ~13%, and calculated circulating load.

The first pass product from the second stage HPGR was screened, and the oversize material was mixed with a calculated amount of fresh product from the previous stage and fed through the HPGR. This procedure was repeated for a number of cycles in order to simulate the re-circulation in the plant. Product size, specific throughput constant and specific energy consumption for each cycle were compared. Results obtained from last recycle of HPGR closed circuit testing will be used for energy calculations.

The effect of closed circuit operation on product size P_{80} and P_{50} for each cycle is shown in Figure 4.5. The results showed that the introduction of a re-circulating load reduced the product size and the recycle operation started stabilizing after two cycles with signs of little reduction in product size observed.

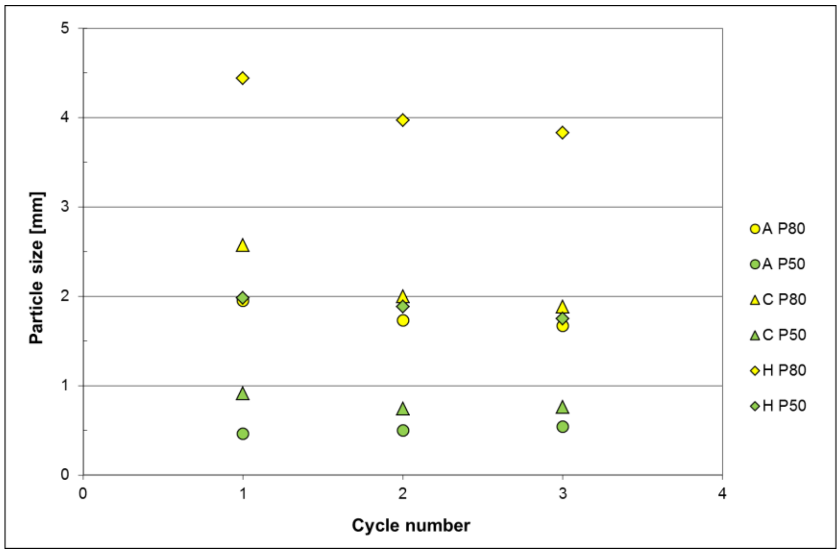

Figure 4.5 Product size for closed circuit testing

Figure 4.6 shows the effect of closed circuit operation on specific throughput constant for each cycle. It was found that the introduction of a re-circulating load had little effect on the specific throughput constant for tests A and H. However, there was an increase in specific throughput constant for test C due to the re-circulating load.

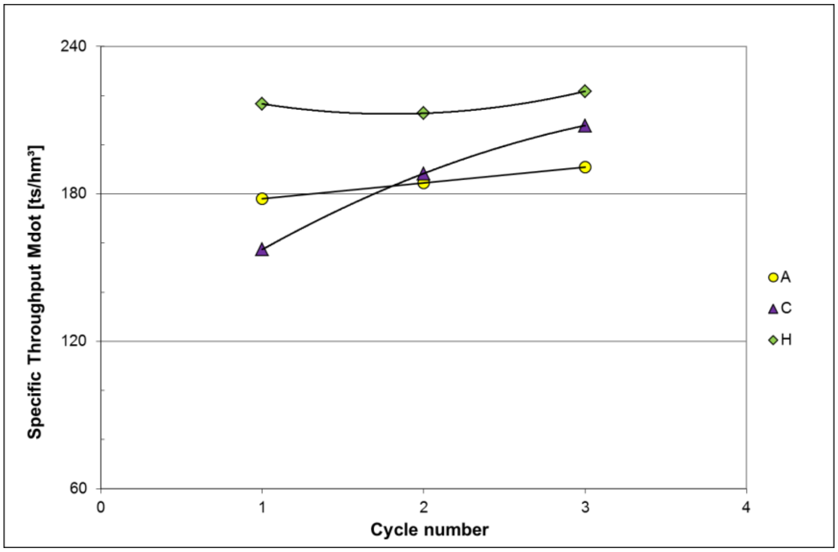

Figure 4.6 Specific throughput constant for closed circuit testing

The effect of closed circuit operation on specific energy consumption for each cycle is shown in Figure 4.7. It can be seen that an improvement in net energy consumption was achieved by the introduction of re-circulating load. Note that the effect of screen inefficiencies was not taken into account in the determination of the specific energy consumptions.

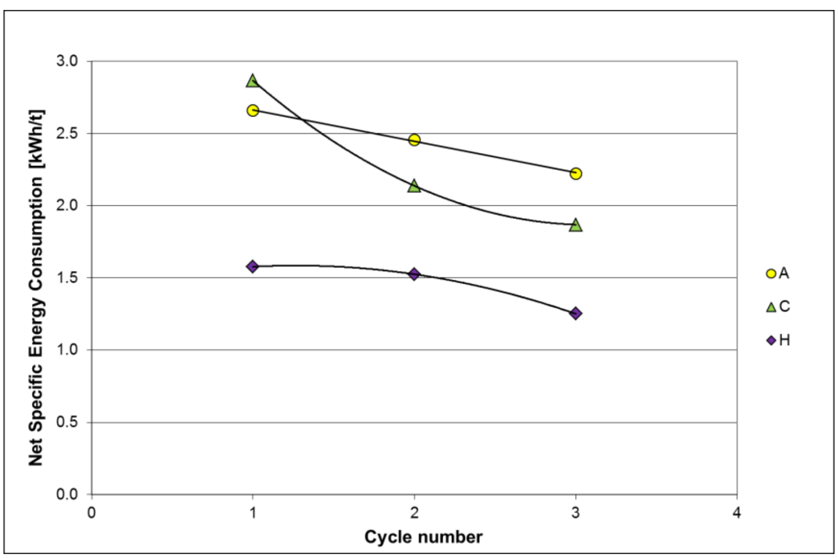

Figure 4.7 Net specific energy consumption for closed circuit testing

4.2.3 Bond ball mill work indices

Bond ball mill grindability tests were performed on selected first-stage HPGR product sample as compared to the AG/SAG feed Bond work indices. The test results are summarized in Table 4.4, and an order of 7% to 15% reduction in Bond work indices was observed due to potential micro fracturing, which is in agreement with the range reported in other studies (Daniel, 2007a; Amelunxen et al., 2011; Patzelt et al., 2006).

Table 4.4 Bond ball mill work indices

Circuit	RoM [kWh/t]	HPGR product [kWh/t]	Difference [%]
A	13.8	12.1	-12.3
C	13.6	12.6	-7.4
D	13.8	12.8	-7.2
H	18.0	15.4	-14.4

It is important to note that the Bond work indices reported above only accounted for coarse product size. It was observed that the Bond work indices increased slightly as the product size decreased for HPGR product, as shown in Appendix C. Since the data regarding Bond work indices at finer product size for the run-of-mine materials was not available, the values reported above would be applied to both coarse grind size and fine grind size, provided that the reductions in Bond work indices were maintained.

4.3 Stirred mill testing results

Increasing the feed size of IsaMill™ is essential to enable the HPGR - stirred mill circuit for future comminution design. It was stated in section 2.3.1 that the IsaMills™ are making their way toward treating coarser material, with the development of increased grinding media size and increased mill capacity. Currently, an IsaMill™ is able to handle feed F_{80} of 300-400 μm operating with large ceramic grinding media and is efficient at grinding products below 100 μm (Larson et al., 2012). The purpose of the test work program was to evaluate the horizontal stirred mill (IsaMill™) in treating coarse material prepared by HPGR, and to provide data for HPGR - IsaMill™ circuit design. With consideration to the coarseness of the feed, all experiments were carried out using a graded charge (50% 5.0-6.0 mm, 28.6% 4.5-5.5 mm, 14.3% 3.0-4.0 mm and 7.1% 2.0-3.0 mm) of large diameter Zirconium Silicate ceramic media manufactured by CENOTEC Co. Ltd., based on correspondence with Xstrata Technology. The operating conditions for each test are tabulated in Table 4.5.

Table 4.5 Test conditions for 710 μm signature plot

Description	Units	ISA A1 (case A)	ISA C1 (case C)	ISA D1 (case D)	ISA H1 (case H)
F_{100}	[μm]	710	710	1000	710
F_{80}	[μm]	310	326	420	343
Feed Wt.	[kg]	100	100	100	100
Solids Density	[%]	52	52	51	50
Flow rate	[L/min]	22	22	22	22
Media Volume	[%]	70	70	70	70
Mill Speed	[RPM]	900	900	900	1000

A signature plot for the testing of sample A is shown in Figure 4.8. An F_{80} of 310 μm corresponds to a media to particle size ratio of 19.4:1, which is in agreement with the ratio of 20:1 suggested by Mankosa et al. (1986). The graph shows that the first pass produced a product P_{80} of 114 µm. Therefore, a P_{80} of 100 µm was chosen to be the target product size, and an estimated specific energy consumption of 3.8 kWh/t was required.

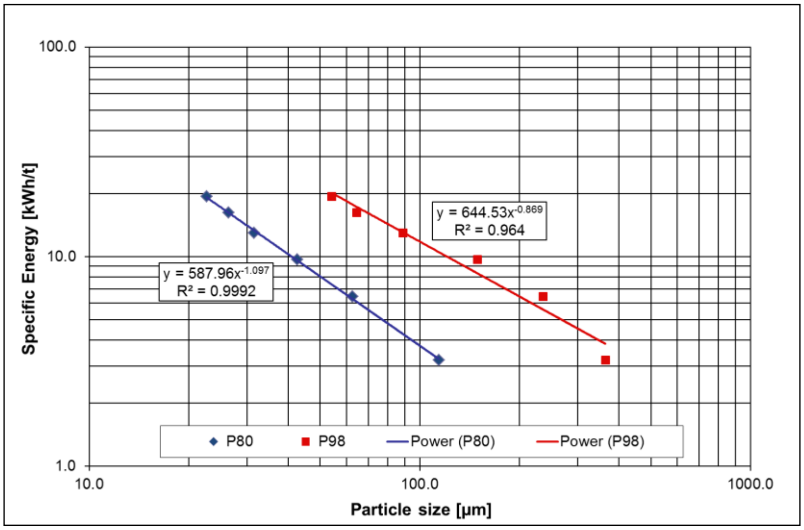

Figure 4.8 Signature plot of test ISA A1

A signature plot for the testing of sample C is shown in Figure 4.9. An F_{80} of 326 µm corresponds to a media to particle size ratio of 18.4:1. The graph shows that grinding to a product P_{80} of 100 µm required 4.4 kWh/t.

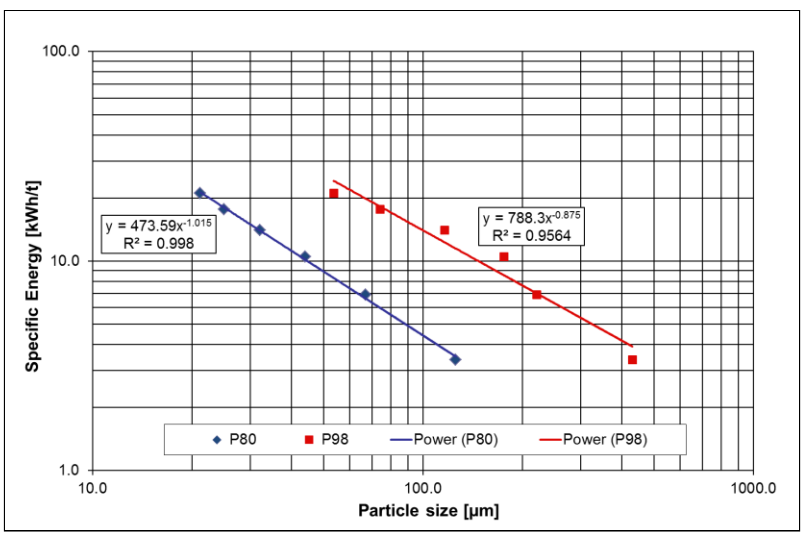

Figure 4.9 Signature plot of test ISA C1

A signature plot for the testing of sample D is shown in Figure 4.10. A top size of 1 mm was chosen to be fed to the IsaMill™. The graph shows that grinding to a product P_{80} of 100 μm required 5.0 kWh/t. However, the media top size appeared to have been undersized for this test, thus the IsaMill™ may not have been operated efficiently. Had the 20:1 ratio suggested by Mankosa et al. (1986) been applied, the media top size would have been 8.4 mm in order to treat an F_{80} of 420 μm. With the increase of the media diameter from 6 mm to 8.4 mm, the energy per top size particle should increase by about 170%. This provides a higher breakage rate for the coarse feed, thus lowering the overall net energy input and producing a coarser product.

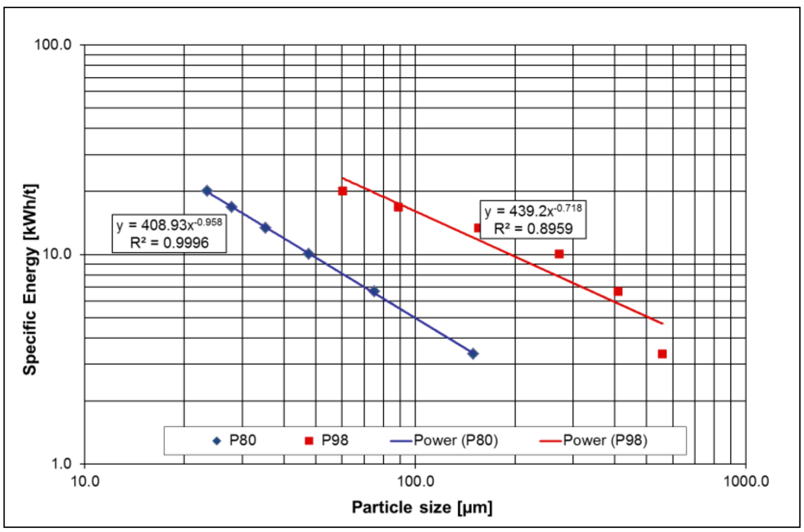

Figure 4.10 Signature plot of test ISA D1

A signature plot for the testing of sample H is shown in Figure 4.11. An F_{80} of 343 μm corresponds to a media to particle size ratio of 17.5:1. The graph shows that the first pass produced a product P_{80} of 82.3 μm. Therefore, it was not reliable to extrapolate the specific energy consumption to a grind size P_{80} of 100 μm. A P_{80} of 75 μm was chosen to be the target product size and an estimated specific energy consumption of 4.8 kWh/t was required.

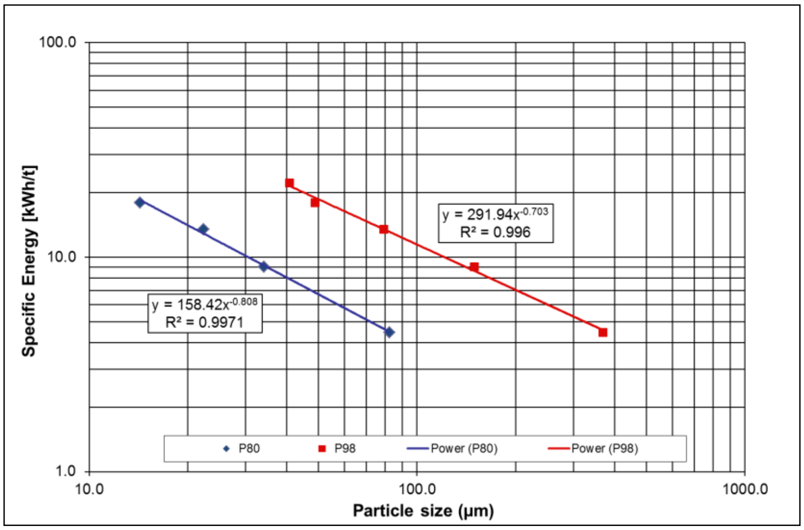

Figure 4.11 Signature plot of test ISA H1

The particle size measurements used to generate the signature plots were performed using wet mechanical screens in order to keep the size analysis consistent. The test results of all signature plots are summarized in Table 4.6.

Table 4.6 Summary of signature test results

Test Description	Units	ISA A1	ISA C1	ISA D1	ISA H1
Feed top size	[µm]	710	710	1000	710
F_{80}	[µm]	310	326	420	343
Target P_{80}	[µm]	100	100	100	75
Specific Energy	[kWh/t]	3.8	4.4	5.0	4.8
Media Consumption	[g/kWh]	6	7	5	3

4.4 Discussions

During the pressure sensitivity testing, it appeared that higher energy input could be achieved with higher specific pressing force, but the benefit of incremental size reduction would decrease. It was also found that higher specific pressing force resulted in smaller operating gap, which reduced the specific throughput rates. Therefore, appropriate specific pressing forces were determined based on the trade-off between each parameter, thus providing operating parameters for HPGR sizing and selection. The effect of feed moisture content was not evaluated due to the availability of sample quantity. However, it is generally reported that the introduction of moisture to a HPGR circuit has adverse effects on throughput and energy consumption. During the HPGR testing, the first stage operation was performed using the existing moisture for each sample. The second stage HPGR operation was performed with the inclusion of moisture. The HPGR closed circuit testing provided information for the second stage HPGR design and power calculation for HPGR - stirred mill circuit.

For the HPGR - stirred mill circuit, both machines were operating outside their respective industry standard conditions. Challenges were primarily associated with the nominated transfer size between the HPGR and stirred mill. For example, nomination of a coarser transfer size necessitated the use of larger stirred mill grinding media, and resulted in a reduction in stirred mill energy efficiency. Conversely, nomination of a finer cut-point was detrimental to the screening efficiency of HPGR product. During the screening process in HPGR closed circuit testing, it was observed that in order to achieve a suitable degree of screening efficiency, considerable effort was required to disperse the compacted HPGR product. In the laboratory, this was addressed through repeated screening and manual dispersion of material on the screen bed. However, continuous-scale industrial operation would require specially-designed material handling and classification equipment to efficiently separate the compacted material. Successful development of the HPGR - stirred mill circuit relies on further addressing the efficient separation of HPGR product at a suitable feed size for stirred mill operation.

At the current target grind, the existing AG/SAG mill - ball mill circuits were only compared to the HPGR - ball mill circuit. From the IsaMill™ testing results, reliable energy consumption values were determined from the signature plots with the first pass product finer than the existing grind targets. Thus, finer grind sizes were selected in order to compare all three different circuits for each case. A coarser product can be produced with a lower overall energy consumption, when coarser grinding media is in use.

CHAPTER 5: MODELING AND SIMULATION RESULTS

A JK SimMet® model was developed for the determination of specific energy requirements to accomplish the desired size reduction in the existing AG/SAG mill circuit. In conjunction with the pilot HPGR testing results, JK SimMet® was used to simulate the HPGR flowsheet, to determine the circuit energy consumption. The power requirement to the ball mill circuit was calculated using the Bond work index and Bond's third comminution theory, the application of the "phantom cyclone" technique to factor for extra fines produced by AG/SAG mill and HPGR (Napier-Munn et al., 1996). This chapter also summarizes the specific energy consumption calculation for the HPGR - stirred mill circuit, based on the pilot-scale testing work.

5.1 JK SimMet® for AG/SAG mill circuits simulation

The specific energy consumptions of the existing AG/SAG mill circuits were evaluated using a JK SimMet® model. The main model inputs were,

- Ore characteristic parameters derived from the JK drop weight test: A, b and t_a;
- Appearance function from JK full weight test;
- Bond work index C_{Wi}, R_{Wi} and B_{Wi};
- Machine specifications for the existing pebble crushers and AG/SAG mills;
- Operating parameters for the existing pebble crushers and AG/SAG mills;
- Material properties such as % solids density by weight and particle size distribution data obtained from surveying the plant.

The main process design criteria, including the plant throughput and grindability parameters of the ore, are summarized in Table 5.1. The existing equipment parameters, as well as the complete crusher table parameters from the JK drop-weight tests, including appearance function and breakage ECS data, can be found in Appendix F.

Table 5.1 Summary of AG/SAG mill circuit process design parameters

Description	Units	Case A	Case C	Case D	Case H
Throughput	[tph]	889	1332	765	766
Solid SG	[-]	2.6	2.6	2.6	2.7
Circuit F_{80}	[mm]	108.3	91.5	95.0	64.8
Crusher work index	[kWh/t]	8.09	10.6	7.96	-
JK parameter A x b	[-]	65.0	64.9	74.2	31.3
JK parameter t_a	[-]	0.45	0.31	0.58	0.59
Ball mill work index, ROM	[kWh/t]	13.8	13.6	13.8	18.0
Final product P_{80} (coarse)	[mm]	188	265	243	158
Final product P_{80} (fine)	[mm]	100	100	100	75

The JK SimMet® AG/SAG mill variable rates model was used to determine the total power requirement of the mill, and the size of mill discharge (Napier-Munn et al., 1996). This model estimated the gross power draw of a mill for a given dimension, operating with a particular charge and speed. Note that the gross power draw as calculated by this method refers to the power input to the mill motor, but the measured DCS power draw was the power output at the pinion. The listed main model inputs were used for the model-fit of mill. The AG/SAG mill breakage rates function and mill discharge rates function were developed and fitted based on the author's personal experience. A good fit was able to be achieved to the measured AG/SAG mill motor power draw and mill load volume. Pebble crusher modelling was developed based on JK SimMet® Andersen's model. The JK drop-weight test results and plant survey data were input into the model, and run several times, until the best fit was found. The JK SimMet® standard efficiency curve model was used to model-fit the AG/SAG mill screen to achieve the transfer size similar to that measured at the existing operation. The fitted model of the AG/SAG mill circuit was then used to simulate the process changes required to achieve a finer grind size. A summary of simulated power required for AG/SAG milling and pebble crushing to process ore from existing circuits at a desired process rate is given in Table 5.2. The model screen snapshots are shown in Figure 5.1 to Figure 5.8.

Table 5.2 Summary of AG/SAG mill circuits simulation results

Description	Throughput [tph]	Target P_{80} [μm]	AG/SAG power [kW]	Pebble crusher power [kW]	AG/SAG spec. energy [kWh/t]
Case A	889	188	6,293	-	7.08
Case A	889	100	6,262	-	7.04
Case C	1332	265	8,157	-	6.12
Case C	1332	100	8,135	-	6.11
Case D	765	243	5,949	103	7.78
Case D	765	100	5,810	107	7.59
Case H	766	158	7,859	122	10.26
Case H	766	75	8,120	60	10.60

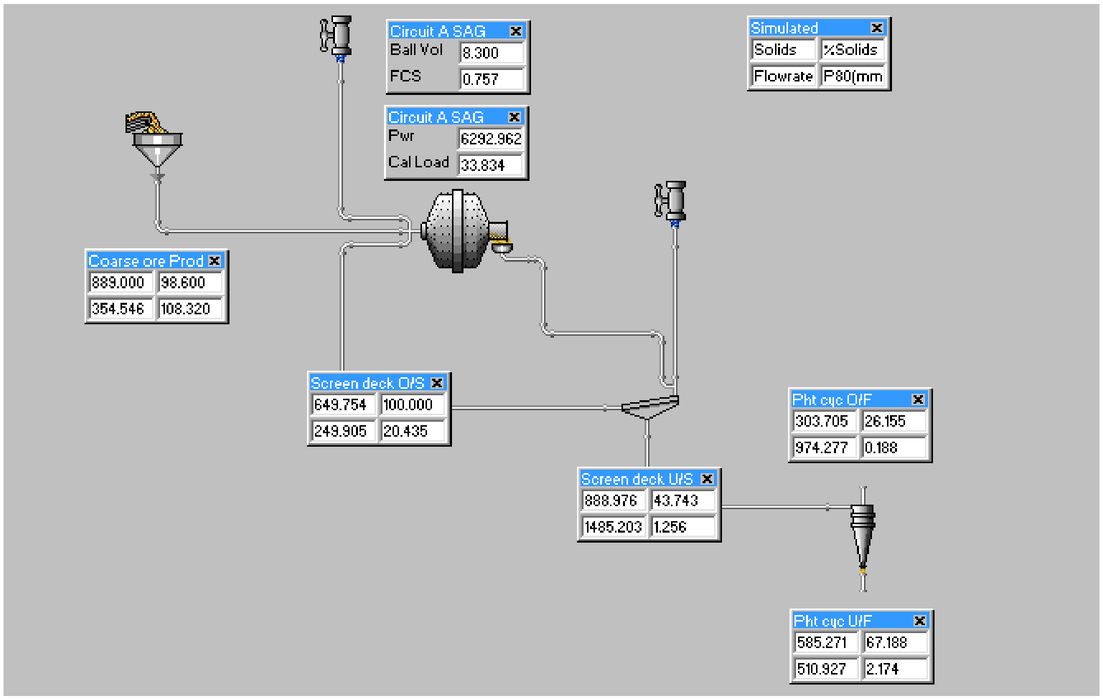

Figure 5.1 JK SimMet® screenshot of the SAG circuit simulation for case A

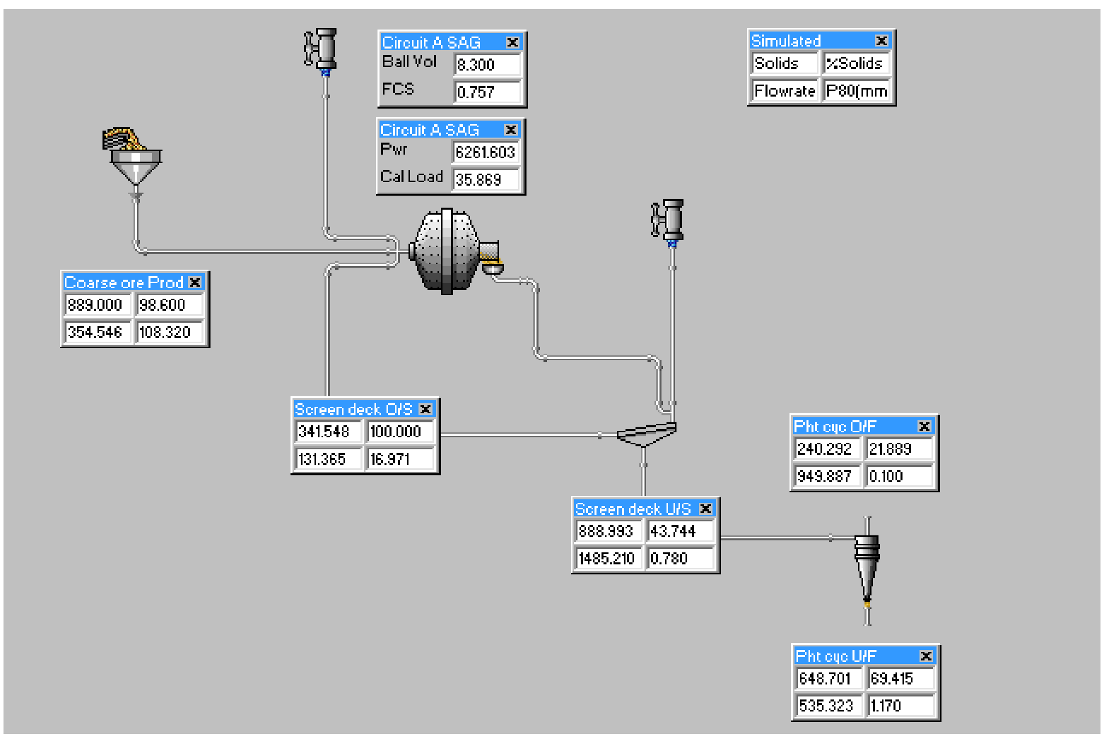

Figure 5.2 JK SimMet® screenshot of the SAG circuit simulation for case A (cont'd)

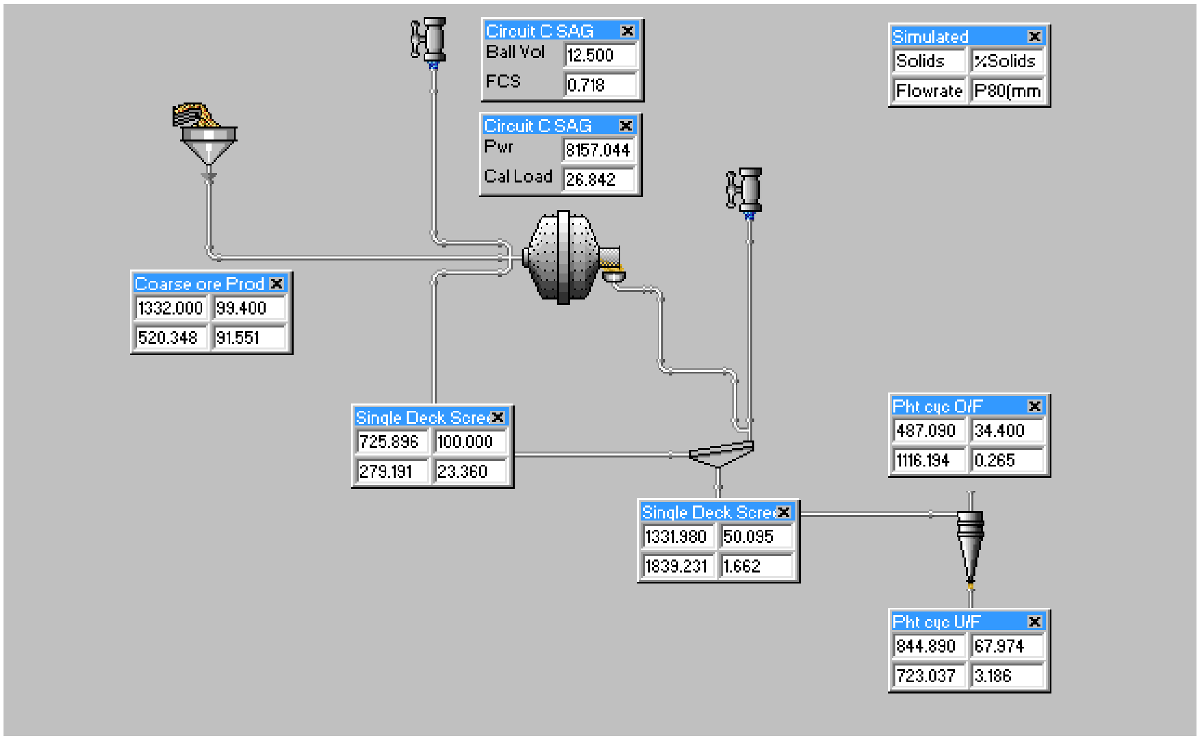

Figure 5.3 JK SimMet® screenshot of the SAG circuit simulation for case C

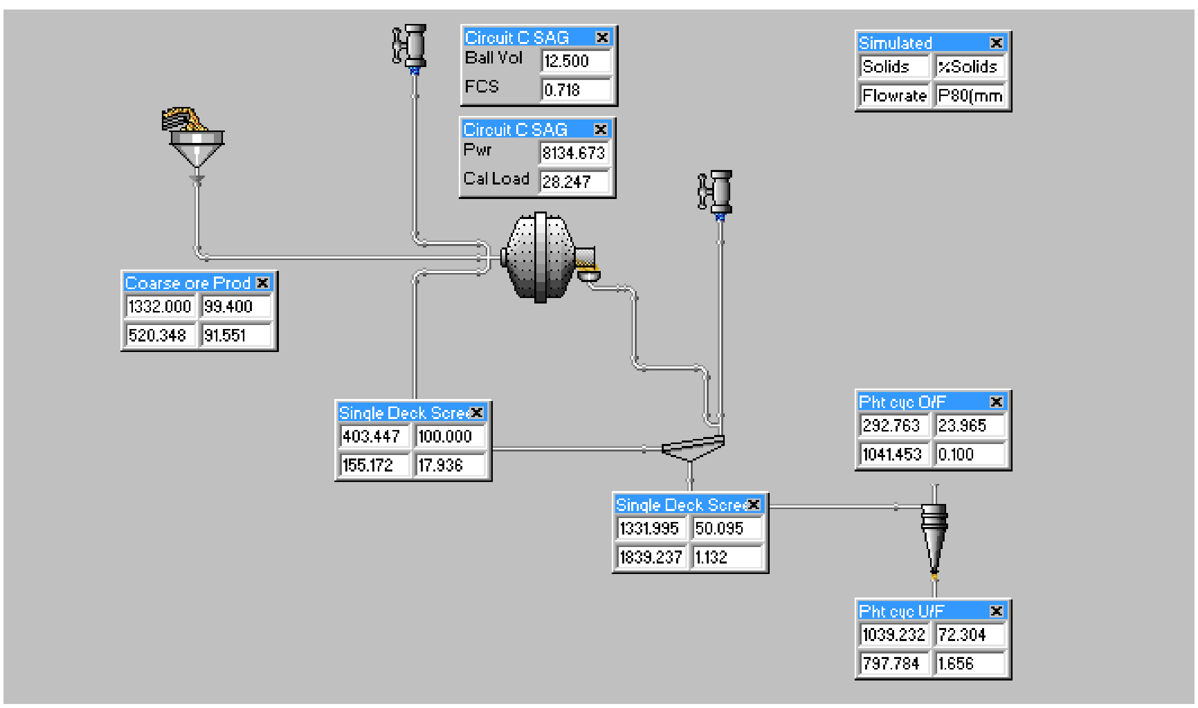

Figure 5.4 JK SimMet® screenshot of the SAG circuit simulation for case C (cont'd)

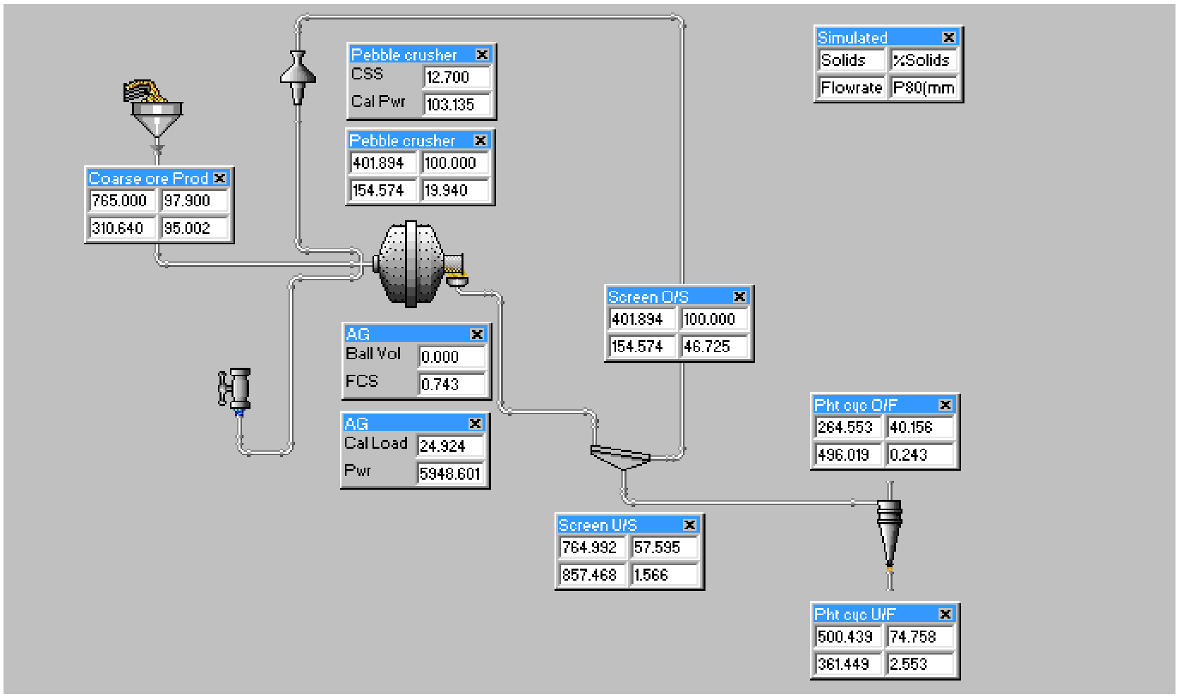

Figure 5.5 JK SimMet® screenshot of the SAG circuit simulation for case D

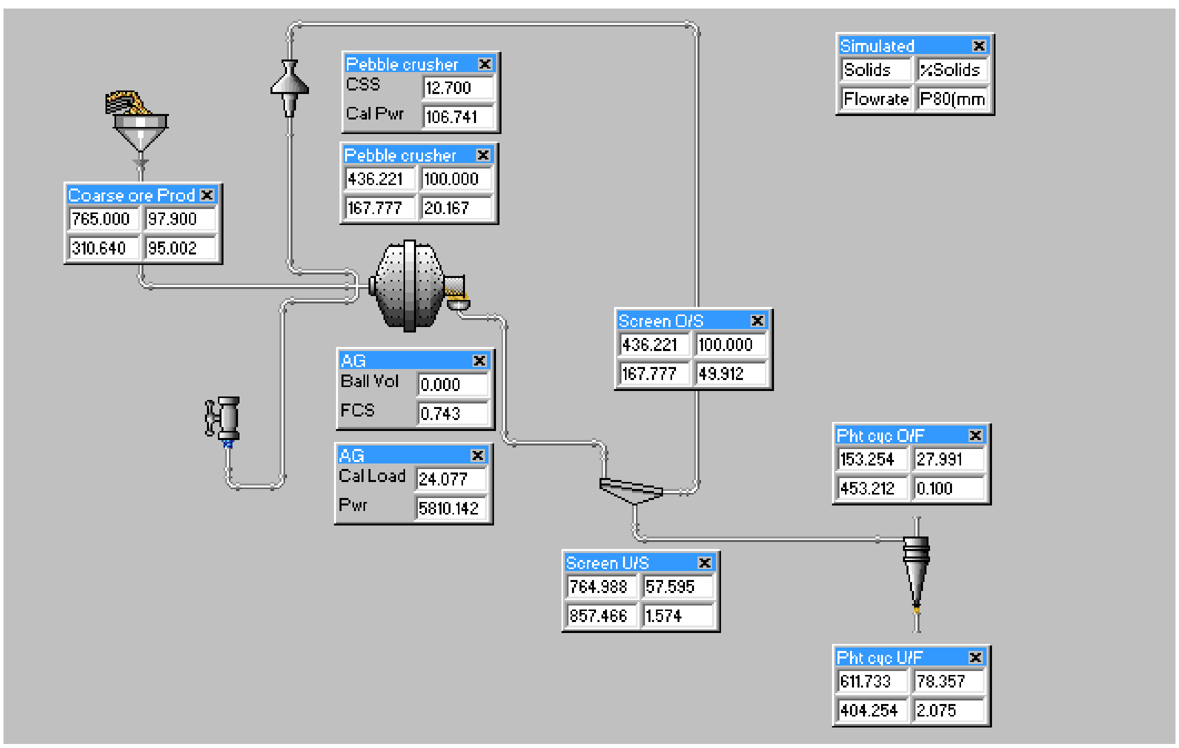

Figure 5.6 JK SimMet® screenshot of the SAG circuit simulation for case D (cont'd)

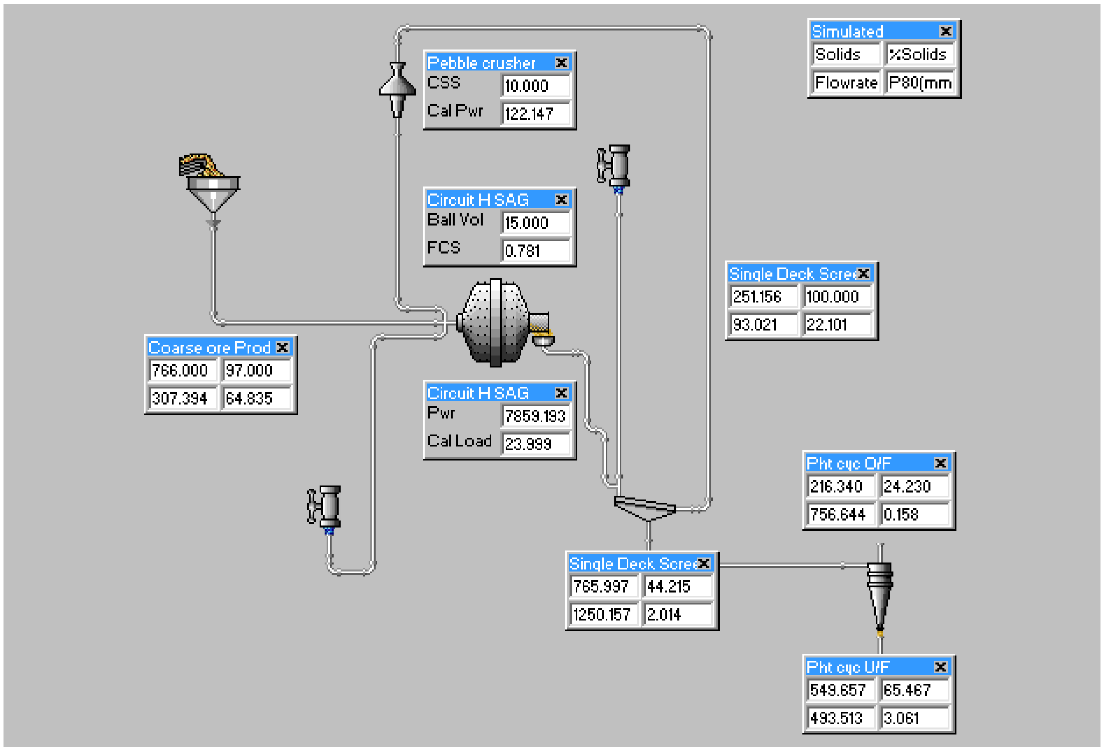

Figure 5.7 JK SimMet® screenshot of the SAG circuit simulation for case H

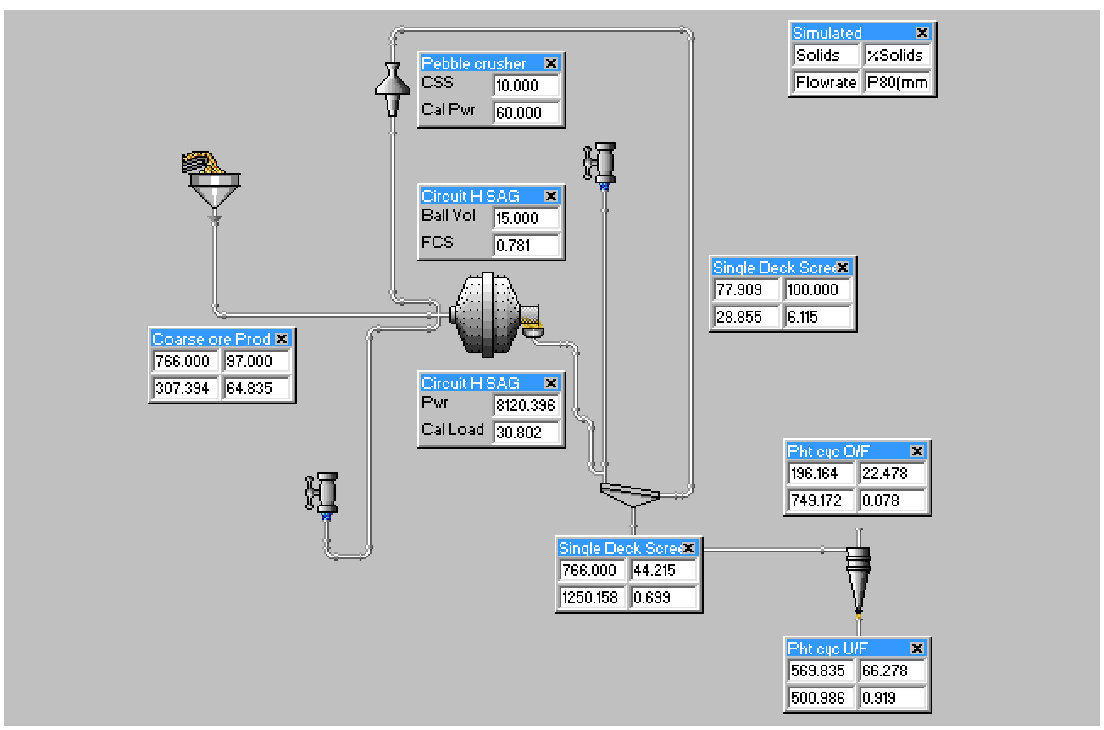

Figure 5.8 JK SimMet® screenshot of the SAG circuit simulation for case H (cont'd)

5.2 JK SimMet® for HPGR circuit simulation

The specific energy consumption of the the HPGR section in the HPGR - ball mill circuit was evaluated using a JK SimMet® model. The main model inputs were,

- Appearance function from JK drop weight test;
- Pilot-scale HPGR machine specifications;
- Pilot-scale HPGR operating parameters;
- HPGR pilot testing results (m-dot, PSD of feed and product, specific energy).

The main process design criteria, including the plant throughput and the HPGR modelling parameters, are summarized in Table 5.3. The complete crusher table parameters from the JK drop-weight tests, including appearance function and breakage ECS data, can be found in Appendix F.

Table 5.3 Summary of HPGR circuit process design parameters

Description	Units	Case A	Case C	Case D	Case H
Throughput	[tph]	889	1332	765	766
Solid SG	[-]	2.6	2.6	2.6	2.7
Circuit F_{80}	[mm]	108.3	91.5	95.0	64.8
Crusher work index	[kWh/t]	8.09	10.6	7.96	-
HPGR fresh feed F_{80}	[mm]	19.7	21.9	21.9	23.6
HPGR net spec. energy	[kWh/t]	1.37	1.23	1.55	1.89
Ball mill work index, ROM	[kWh/t]	13.8	13.6	13.8	18.0
Ball mill work index, HPGR product	[kWh/t]	12.1	12.6	12.8	15.4
Final product P_{80} (coarse)	[mm]	188	265	243	158
Final product P_{80} (fine)	[mm]	100	100	100	75

The specific energy requirements for the secondary crushing circuit operation were calculated using the Bond equation (Bond, 1961),

$$W = 10 \times C_{Wi} \times [1/\sqrt{P_{80}} - 1/\sqrt{F_{80}}] \qquad \text{(Equation 5)}$$

where W is the specific energy consumption [kWh/t], C_{Wi} is the Bond crusher work index [kWh/t], P_{80} [µm] is the particle size at which 80% of particles pass in product, and F_{80} [µm] is the particle size at which 80% of particles pass in feed.

HPGR modelling was developed based on the pilot-scale testing results and the model-fitting in JK SimMet® (Daniel & Morrell, 2004). The JK SimMet® standard efficiency curve model was used to determine the transfer size between the HPGR circuit and the subsequent ball mill circuit. The fitted model of the HPGR circuit was then used to simulate the process changes required to achieve a finer grind size. The screen snapshots of the models of the HPGR circuit are shown in Figure 5.9 to Figure 5.16. A summary of the simulation results is shown in Table 5.4. A factor of 120% of net specific energy was applied to calculate the total motor power draw of the HPGR for the process capacity. This value was consistent with that observed with other HPGR operations (Klymowsky et al., 2006).

Table 5.4 Summary of HPGR - ball mill circuits simulation results

Description	F_{80} [mm]	Target P_{80} [µm]	Sec. crusher [kWh/t]	HPGR spec. energy [kWh/t]	Scaled HPGR spec. energy [kWh/t]
Case A	19.7	188	0.33	2.42	**2.90**
Case A	19.7	100	0.33	2.42	**2.90**
Case C	21.9	265	0.37	2.27	**2.73**
Case C	21.9	100	0.37	2.27	**2.73**
Case D	21.9	243	0.28	2.96	**3.55**
Case D	21.9	100	0.28	2.96	**3.55**
Case H	23.3	158	0.47	3.32	**3.98**
Case H	23.3	75	0.47	3.32	**3.98**

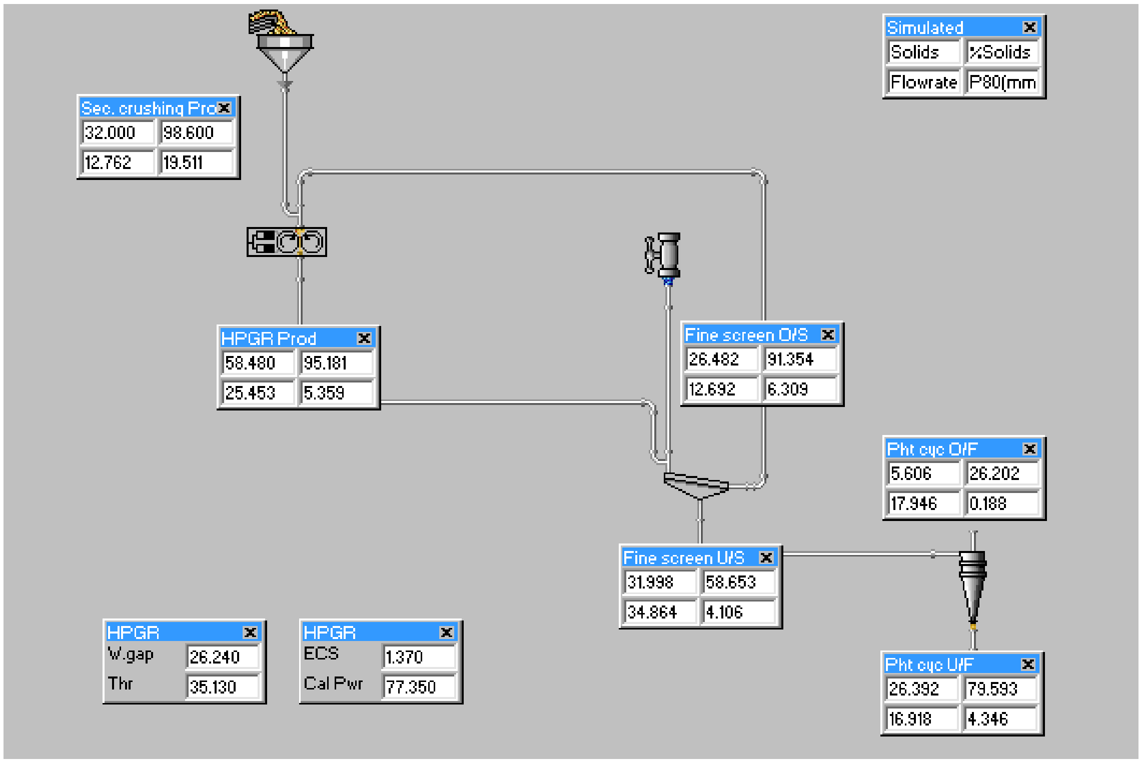

Figure 5.9 JK SimMet® screenshot of the HPGR circuit simulation for case A

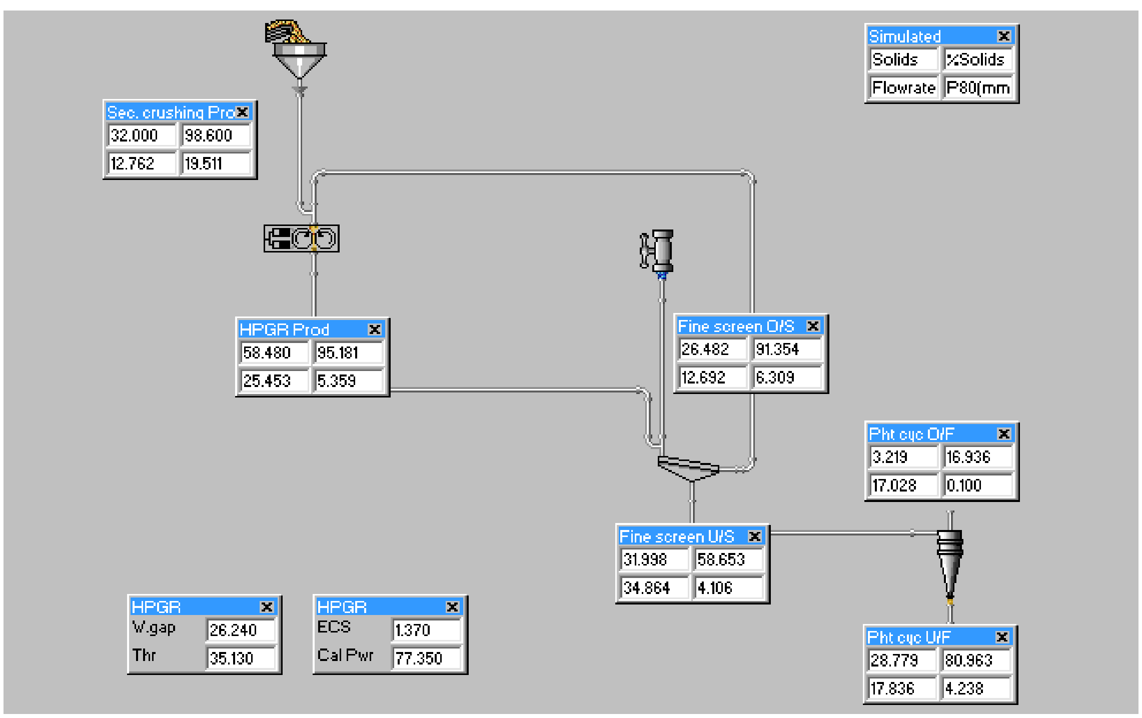

Figure 5.10 JK SimMet® screenshot of the HPGR circuit simulation for case A (cont'd)

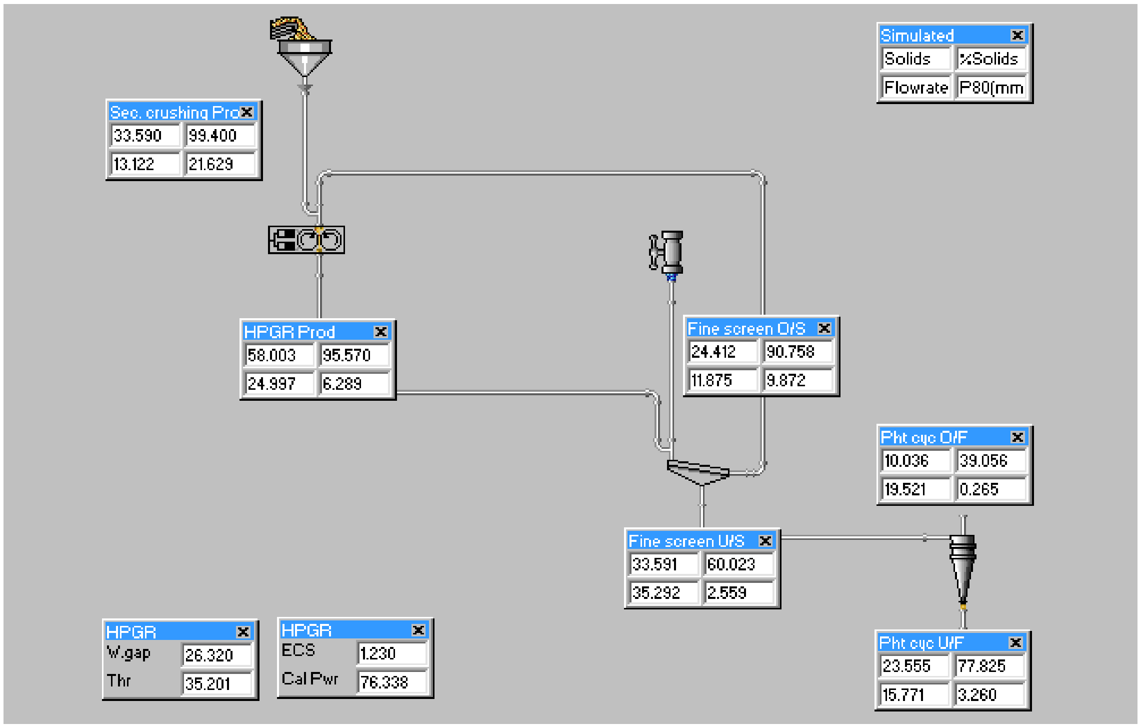

Figure 5.11 JK SimMet® screenshot of the HPGR circuit simulation for case C

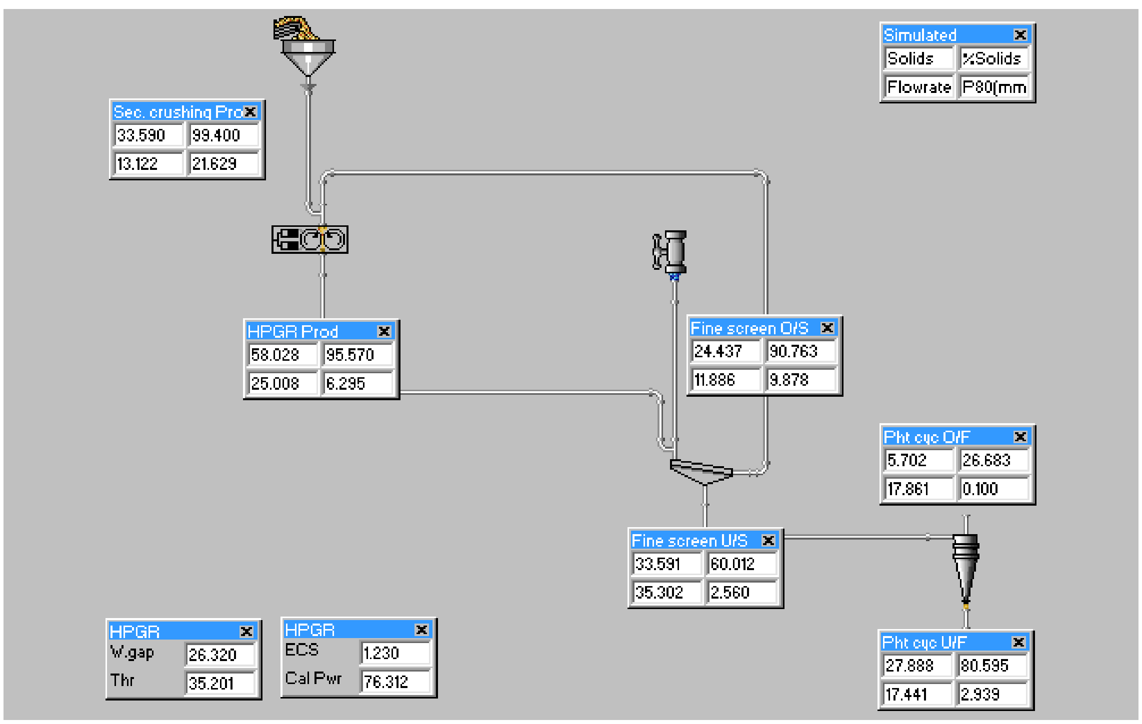

Figure 5.12 JK SimMet® screenshot of the HPGR circuit simulation for case C (cont'd)

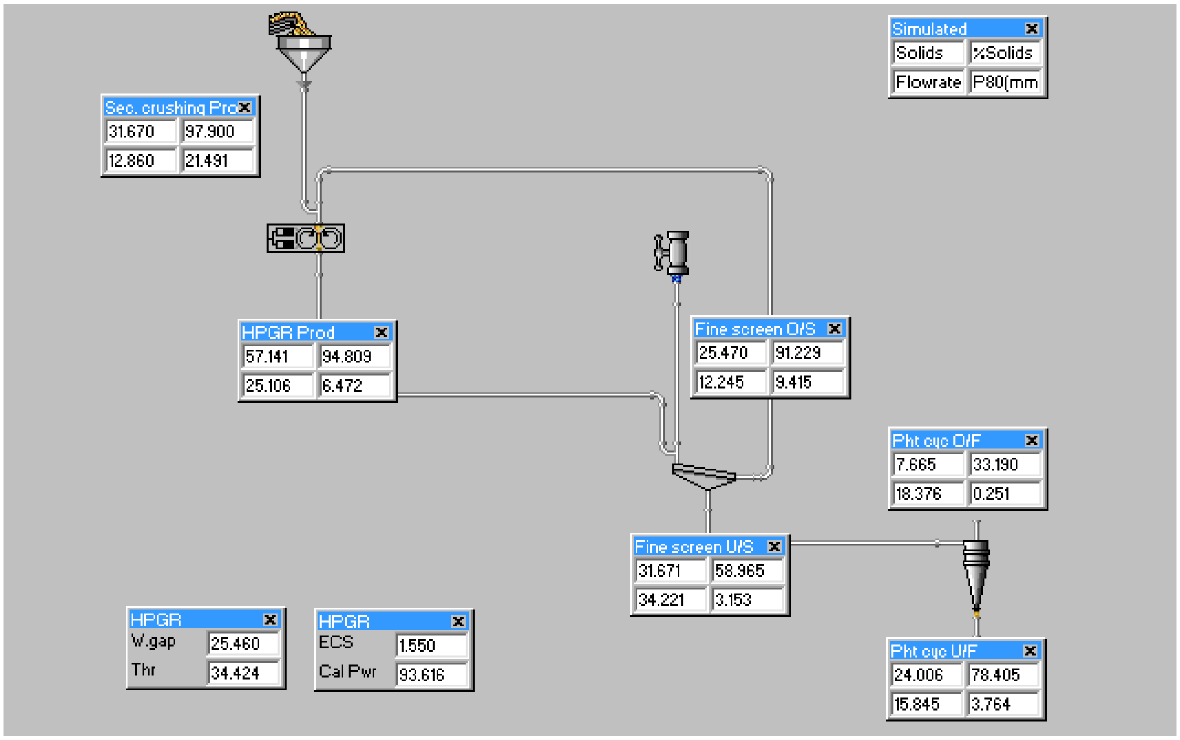

Figure 5.13 JK SimMet® screenshot of the HPGR circuit simulation for case D

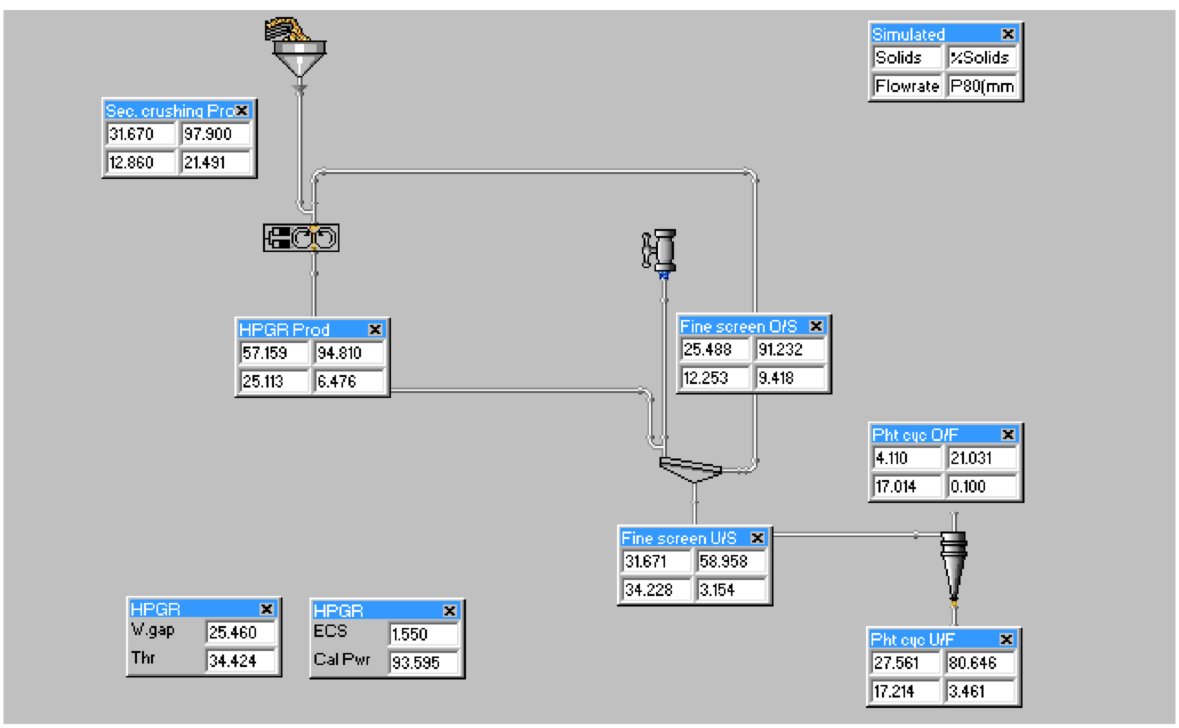

Figure 5.14 JK SimMet® screenshot of the HPGR circuit simulation for case D (cont'd)

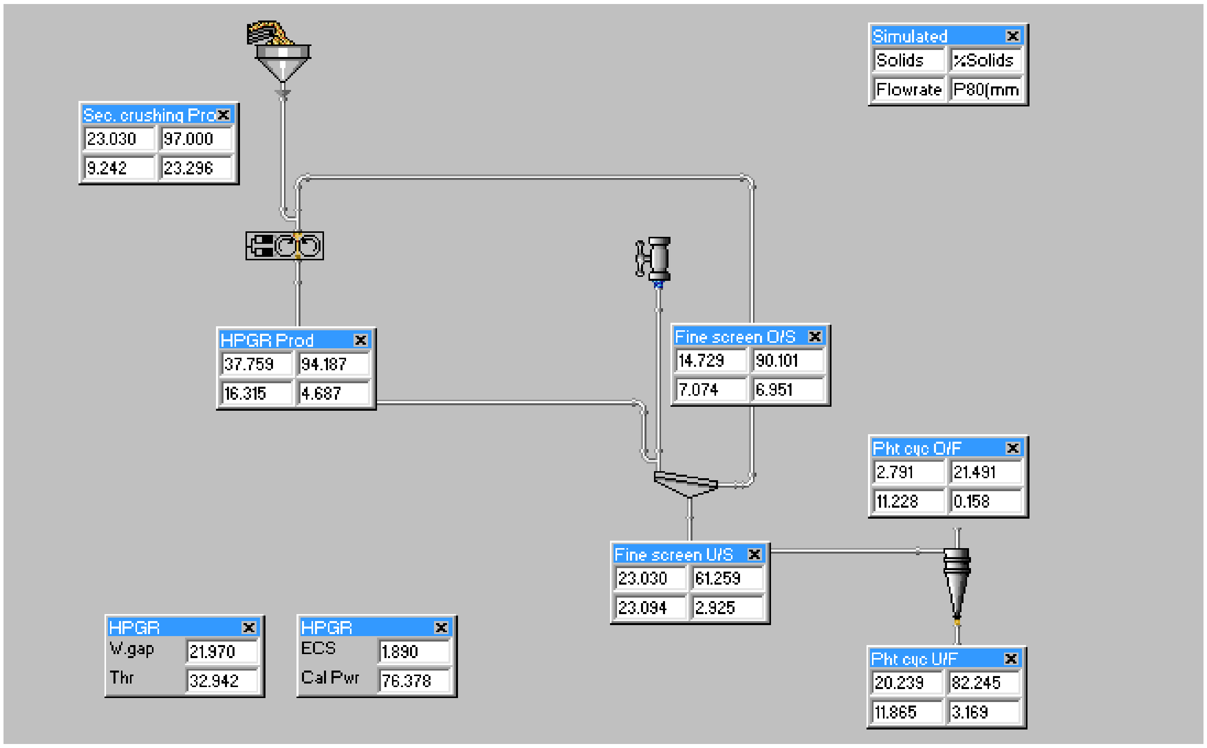

Figure 5.15 JK SimMet® screenshot of the HPGR circuit simulation for case H

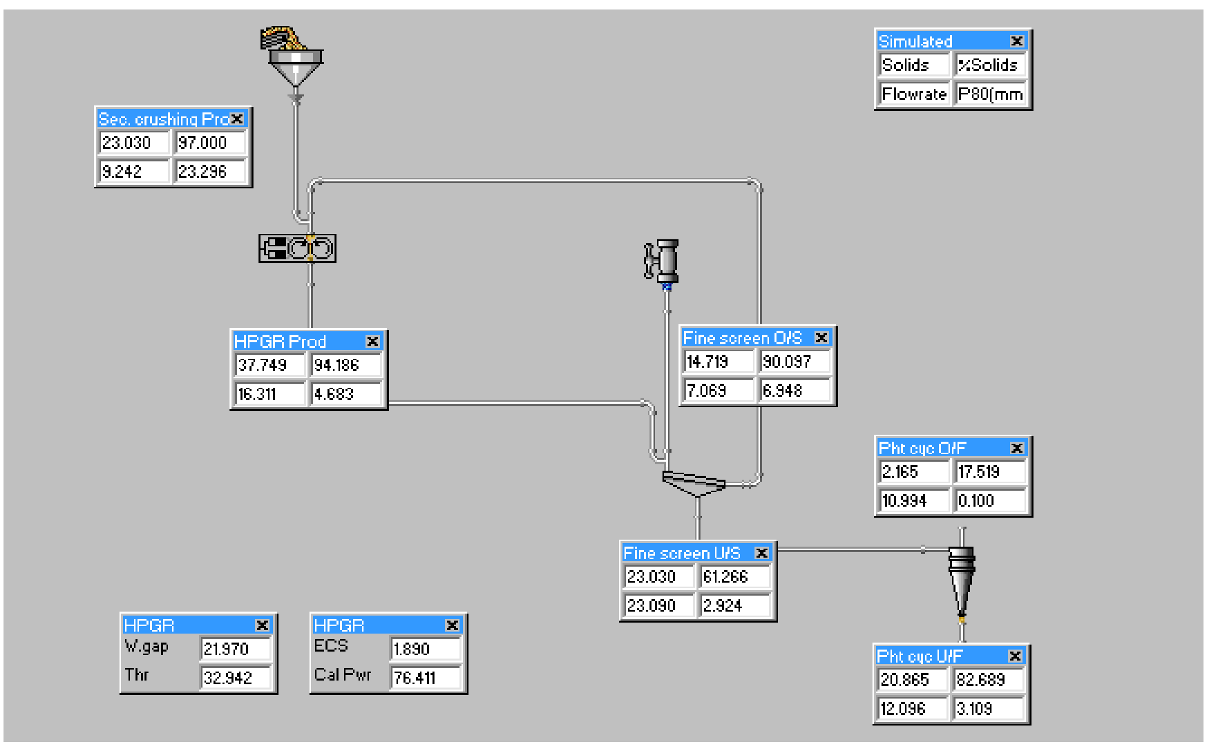

Figure 5.16 JK SimMet® screenshot of the HPGR circuit simulation for case H (cont'd)

5.3 Ball mill circuit energy requirements

In order to make a proper energy comparison of the AG/SAG mill - ball mill circuits and the HPGR - ball mill circuit, the ball mill circuit energy requirements were calculated via the "phantom cyclone" / Bond method (Napier-Munn et al., 1996; Doll et al., 2010). The Bond equation was originally developed for treating product from conventional crushers or rod mills. However, both AG/SAG mills and HPGR create more product fines than conventional crushers or rod mills. Applying the Bond equation directly to AG/SAG mills or HPGR product results in an overestimate of the power required for the secondary ball mills. Thus, the "phantom cyclone" was used to remove extra fines produced by AG/SAG mills or HPGR to make the product size distribution parallel to the conventional Rosin-Rammler distribution model (Doll et al., 2010). The "phantom cyclone" overflow represented the finished product from the ball mill circuit, which consumes no energy in the ball mill. The "phantom cyclone" underflow was considered as the effective tonnage feed (eff. tonnage) and F_{80} (eff. F_{80}) for the energy calculation. Using the Bond ball mill work indices and effective transfer size F_{80} determined in section 0 and section 5.2, the specific energy requirements for ball mills were calculated using the Bond equation (Bond, 1961).

$$W = 10 \times B_{Wi} \times [1/\sqrt{P_{80}} - 1/\sqrt{F_{80}}] \qquad \text{(Equation 6)}$$

where W is the specific energy consumption [kWh/t], B_{Wi} is the Bond ball mill work index [kWh/t], P_{80} [µm] is the particle size at which 80% of particles pass in product, and F_{80} [µm] is the particle size at which 80% of particles pass in feed.

The actual power required at the pinion for the ball mill was calculated by multiplying the specific energy consumption by the effective tonnage. The gross energy required for the motor of the ball mill was calculated by taking into account the assumed 5% electrical and mechanical efficiency losses. Thus, the ball mill circuit specific energy consumptions were calculated by dividing the gross power consumption by the actual process throughput. The calculation results are summarized in Table 5.5 and Table 5.6 for the AG/SAG mill - ball mill circuit and the HPGR - ball mill circuit, respectively. It was noticed that the HPGR screen undersize products were coarser than the product of the AG/SAG mill circuits, while having a lower associated Bond ball mill work index, as mentioned in section 4.2.3. Overall, the net effect of changing these two material attributes, size and work index, was an increase in required ball mill specific energy for the HPGR - ball mill circuit.

Table 5.5 Calculation of ball mill circuit specific energy for AG/SAG mill circuits

Description	Bond BWi [kWh/t]	Target P_{80} [µm]	Actual tonnage [tph]	Eff. F_{80} [µm]	Eff. Tonnage [tph]	Motor input [kWh/t]	Spec. W [kWh/t]
Case A	13.8	188	889	2,174	585	4,366	**4.91**
Case A	13.8	100	889	1,170	649	6,652	**7.48**
Case C	13.6	265	1332	3,186	845	5,274	**3.96**
Case C	13.6	100	1332	1,656	1,039	11,191	**8.40**
Case D	13.8	243	765	2,553	500	3,217	**4.20**
Case D	13.8	100	765	2,075	612	6,918	**9.04**
Case H	18.0	158	766	3,060	550	6,386	**8.34**
Case H	18.0	75	766	919	570	8,886	**11.60**

Table 5.6 Calculation of ball mill circuit specific energy for HPGR - ball mill circuits

Description	Bond BWi [kWh/t]	Target P_{80} [µm]	Actual tonnage [tph]	Eff. F_{80} [µm]	Eff. Tonnage [tph]	Motor input [kWh/t]	Spec. W [kWh/t]
Case A	12.1	188	889	4,346	733	5,381	**6.05**
Case A	12.1	100	889	4,328	799	8,614	**9.69**
Case C	12.6	265	1332	3,260	934	5,427	**4.07**
Case C	12.6	100	1332	2,939	1106	11,932	**8.96**
Case D	12.8	243	765	3,764	580	3,729	**4.87**
Case D	12.8	100	765	3,461	666	7,427	**9.71**
Case H	15.4	158	766	3,169	673	6,726	**8.78**
Case H	15.4	75	766	3,109	694	10,945	**14.29**

5.4 Measured energy versus simulated energy for existing circuits

The benchmarking energy calculations based on the Millpower 2000 method can be found in Appendix F (Barratt, 1989). It is important to note that the benchmark method specifies power “at the mill pinion.” Thus, 5% electrical and mechanical efficiency losses were applied to determine the motor input. Table 5.7 shows the specific energy values of the benchmarking, JK SimMet® simulation, and the existing plant. It was found that both benchmarking and simulation required less energy than that actually consumed in the plant. Thus, using the simulated energy values for energy comparison is a relatively conservative approach, assuming that the existing circuits were operating efficiently when the mill survey and site data were recorded.

Table 5.7 Benchmarking and simulated energy versus measured energy

	Target P80 [µm]	Benchmark energy [kWh/t]	Simulated energy [kWh/t]	Measured energy [kWh/t]
Case A	188	10.50	11.99	14.24
Case A	100	15.01	14.53	-
Case C	265	9.05	10.08	10.90
Case C	100	15.39	14.51	-
Case D	243	-	12.11	12.21
Case D	100	-	16.78	-
Case H	158	16.49	18.76	20.56
Case H	75	24.06	22.28	-

5.5 HPGR - stirred mill circuit energy requirements

The process parameters shown in Table 5.8 were applied for evaluation of the HPGR - stirred mill circuit. The HPGR process parameters were determined from HPGR pilot test results. A factor of 120% of net specific energy was applied to calculate the total motor power draw of the HPGR for the process capacity (Klymowsky et al., 2006). Based on correspondence with Xstrata Technology, the total required IsaMill™ motor power was determined by applying a motor efficiency of 95% to the specific energy values referenced from the signature plot.

Table 5.8 Summary of HPGR - stirred mill process design parameters

Description	Units	Case A	Case C	Case D	Case H
Throughput	[tph]	889	1332	765	766
Solid SG	[-]	2.6	2.6	2.6	2.7
Circuit F_{80}	[mm]	108.3	91.5	95.0	64.8
Crusher work index	[kWh/t]	8.09	10.6	7.96	-
HPGR fresh feed F_{80}	[mm]	19.7	21.9	21.9	23.6
1^{st} stage HPGR specific pressing force	[N/mm^2]	3	3	3	3
2^{nd} stage HPGR specific pressing force	[N/mm^2]	4	4	4	3
2^{nd} stage HPGR screen aperture size	[mm]	0.71	0.71	1 .0	0.71
% passing screen size in HPGR product	[%]	57.9	48.2	64.6	27.4
Assumed HPGR screen efficiency	[%]	90	90	90	90
1^{st} stage HPGR net specific energy	[kWh/t]	1.37	1.23	1.55	1.89
2^{nd} stage HPGR net specific energy	[kWh/t]	2.44	2.29	2.88**	1.45
2^{nd} stage HPGR net specific energy @ screen	[kWh/t]	4.69	5.29	4.95	5.89
Stirred mill feed F_{80}	[µm]	310	326	420	342.6
Stirred mill solid density	[%]	51.5	51	51	50
Stirred mill speed	[RPM]	900	900	900	1000
Stirred mill flowrate	[L/min]	23	23	23	23
Stirred mill media charge	[%]	65	65	65	65
Targeted stirred mill product P_{80}	[µm]	100	100	100	75
Stirred mill specific energy	[kWh/t]	3.8	4.4	5.0	4.8

** simulate the re-circulation process using data from A and C

CHAPTER 6: CIRCUITS ENERGY COMPARISON

The following chapter presents the comminution energy comparison and complete energy comparison between the proposed circuits and the existing circuits. Capital and operating cost comparison was performed for case H as an example to demonstrate the financial benefits of the proposed circuits.

6.1 Comminution equipment energy

Based on the pilot testing results, JK simulation results, and the "phantom cyclone" ball mill energy calculations, the total energy requirements for comminution equipment were determined for each circuit, and summarized in Table 6.1 to Table 6.4. The specific comminution energy requirement for each HPGR-based circuit was plotted against the existing AG/SAG mill-based circuits, for the equivalent comminution duty. Figure 6.1 clearly shows that HPGR-based circuits achieved significant reductions (between 11% and 36%) in comminution energy, as opposed to the existing operations.

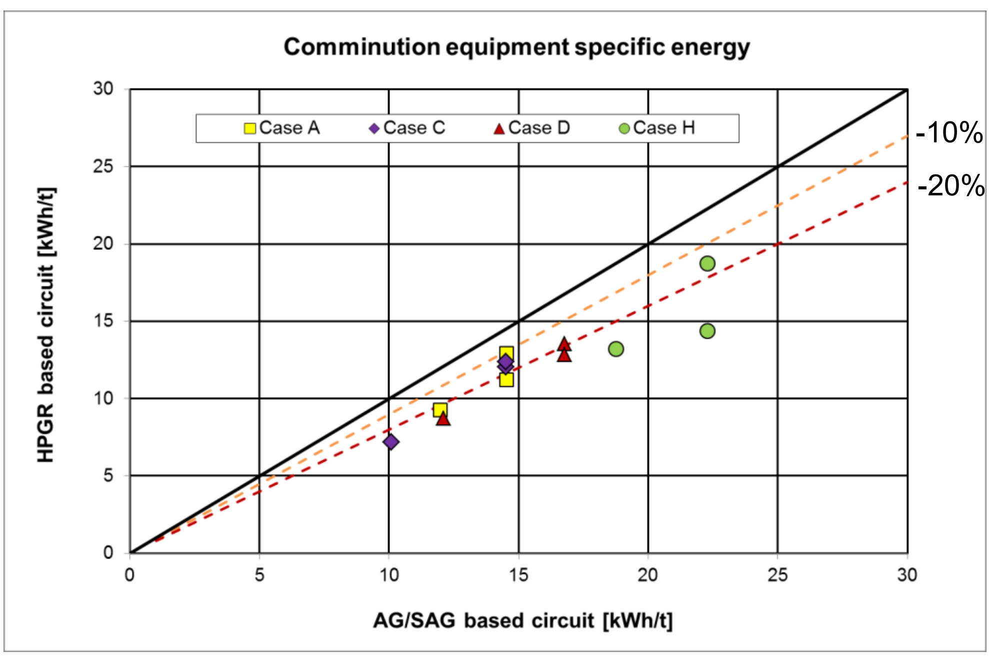

Figure 6.1 HPGR based circuit total kWh/t versus AG/SAG based circuit total kWh/t

For producing coarse grind size, the HPGR - ball mill circuit required 23-30% less energy than the AG/SAG mill - ball mill circuit. The main savings resulted from the lower energy required by the HPGR as compared to the AG/SAG mill. However, an additional secondary crusher and conveyor system were required to facilitate the HPGR circuit. The HPGR also produced a coarser product than the SAG mill. Thus, the energy needed for crushing, ball milling, and material handling was higher for the HPGR - ball mill circuit than the AG/SAG mill - ball mill circuit. When extending the target size to a finer grind size, the energy savings of the HPGR - ball mill circuit reduced to the order of 11-19%. The novel two-stage HPGR - stirred mill circuit showed a reduction in energy of 15-36%, as compared to the AG/SAG mill - ball mill circuit.

Table 6.1 Comminution energy for case A

Description	Qt.	Unit inst. [kW]	Unit simu. [kW]	Total consumption [kW]	Specific energy [kWh/t]
889 tph throughput					
92% circuit overall availability -19,629 tpd					
SAB base case - 188 µm					
SAG mill - 9.75 m D x 4.25 m EGL	1	6,700	6,293	6,293	7.08
Ball mill - 5.0 m D x 7.0 m L	2	3,350	2,182	4,365	4.91
				10,658	**11.99**
HPGR - ball mill - 188 µm					
Secondary crusher - MP 800	1	600	294	294	0.33
HPGR - 2.2 m D x 1.55 m W	1	3,000	2,574	2,574	2.90
Ball mill - 5.0 m D x 7.0 m L	2	3,350	2,689	5,378	6.05
				8,247	**9.28**
Specific Energy Difference					**23%**
SAB base case - 100 µm					
SAG mill - 9.75 m D x 4.25 m EGL	1	6,700	6,262	6,262	7.04
Ball mill - 5.0 m D x 7.0 m L	2	3,350	3,326	6,651	7.48
				12,913	**14.53**
HPGR - ball mill - 100 µm					
Secondary crusher - MP 800	1	600	294	294	0.33
HPGR - 2.2 m D x 1.55 m W	1	3,000	2,574	2,574	2.90
Ball mill - 5.0 m D x 7.0 m L	2	3,350	4,307	8,614	9.69
				11,483	**12.92**
Specific Energy Difference					**11%**
HPGR - stirred mill - 100 µm					
Secondary crusher - MP 800	1	600	294	294	0.33
1st HPGR - 1.7 m D x 1.2 m W	1	1,500	1,462	1,462	1.64
2nd HPGR - 2.4 m D x 1.65 m W	1	5,600	5,003	5,003	5.63
IsaMill™ - M10,000	2	2,000	1,605	3,209	3.61
				9,968	**11.21**
Specific Energy Difference					**23%**

Table 6.2 Comminution energy for case C

Description	Qt.	Unit inst. [kW]	Unit simu. [kW]	Total consumption [kW]	Specific energy [kWh/t]
1332 tph throughput					
92% circuit overall availability -29,410 tpd					
SAB base case - 265 µm					
SAG mill - 10.36 m D x 4.57 m EGL	1	9,400	8,157	8,157	6.12
Ball mill - 5.0 m D x 8.23 m L	2	4,700	2,637	5,275	3.96
				13,432	**10.08**
HPGR - ball mill - 265 µm					
Secondary crusher - MP 800	1	600	488	488	0.37
HPGR - 2.4 m D x 1.65 m W	1	4,000	3,636	3,636	2.73
Ball mill - 5.0 m D x 8.23 m L	2	4,700	2,711	5,421	4.07
				9,545	**7.17**
Specific Energy Difference					**29%**
SAB base case - 100 µm					
SAG mill - 10.36 m D x 4.57 m EGL	1	9400	8135.7	8,136	6.11
Ball mill - 5.0 m D x 8.23 m L	2	4,700	5,596	11,191	8.40
				19,327	**14.51**
HPGR - ball mill - 100 µm					
Secondary crusher - MP 800	1	600	488	488	0.37
HPGR - 2.4 m D x 1.65 m W	1	4,000	3,636	3,636	2.73
Ball mill - 5.0 m D x 8.23 m L	2	4,700	5,967	11,935	8.96
				16,059	**12.06**
Specific Energy Difference					**17%**
HPGR - stirred mill - 100 µm					
Secondary crusher - MP 800	1	600	488	488	0.37
1st HPGR - 1.85 m D x 1.5 m W	1	1,500	1,966	1,966	1.48
2nd HPGR - 2.2 m D x 1.25 m W	2	5,000	4,228	8,456	6.35
IsaMill™ - M10,000	2	3,000	2,784	5,568	4.18
				16,477	**12.37**
Specific Energy Difference					**15%**

Table 6.3 Comminution energy for case D

Description	Qt.	Unit inst. [kW]	Unit simu. [kW]	Total consumption [kW]	Specific energy [kWh/t]
765 tph throughput					
92% circuit overall availability -16,891 tpd					
AGBC base case - 243 µm					
AG mill - 10.36 m D x 4.24 m EGL	1	6,600	5,948	5,948	7.78
Pebble crusher - 7' Symons	1	350	103	103	0.13
Ball mill - 5.0 m D x 8.84 m L	1	4,100	3,213	3,213	4.20
				9,264	**12.11**
HPGR - ball mill - 243 µm					
Secondary crusher - MP 800	1	600	214	214	0.28
HPGR - 2.2 m D x 1.5 m W	1	3,000	2,716	2,716	3.55
Ball mill - 5.0 m D x 8.84 m L	1	4,100	3,726	3,726	4.87
				6,656	**8.70**
Specific Energy Difference					**28%**
AGBC base case - 100 µm					
AG mill - 10.36 m D x 4.24 m EGL	1	6,600	5,810	5,810	7.59
Pebble crusher - 7' Symons	1	350	107	107	0.14
Ball mill - 5.0 m D x 8.84 m L	1	4,100	6,918	6,918	9.04
				12,835	**16.78**
HPGR - ball mill - 100 µm					
Secondary crusher - MP 800	1	600	214	214	0.28
HPGR - 2.0 m D x 1.5 m W	1	3,000	2,716	2,716	3.55
Ball mill - 5.0 m D x 8.84 m L	1	4,100	7,428	7,428	9.71
				10,358	**13.54**
Specific Energy Difference					**19%**
HPGR - stirred mill - 100 µm					
Secondary crusher - MP 800	1	600	214	214	0.28
1st HPGR - 1.7 m D x 1.2 m W	1	1,500	1,423	1,423	1.86
2nd HPGR - 2.4 m D x 1.65 m W	1	5,600	4,544	4,544	5.94
IsaMill™ - M10,000	2	2,200	1,817	3,634	4.75
				9,815	**12.83**
Specific Energy Difference					**24%**

Table 6.4 Comminution energy for case H

Description	Qt.	Unit inst. [kW]	Unit simu. [kW]	Total consumption [kW]	Specific energy [kWh/t]
766 tph throughput					
92% circuit overall availability -16,913 tpd					
SABC base case - 160 µm					
SAG mill - 9.76 m D x 4.11 m EGL	1	8,200	7,859	7,859	10.26
Pebble crusher - HP 800	1	300	123	123	0.16
Ball mill - 5.00 m D x 9.14 m L	2	4,100	3,194	6,388	8.34
				14,370	**18.76**
HPGR - ball mill - 160 µm					
Secondary crusher - MP 800	1	600	356	356	0.47
HPGR - 2.4 m D x 1.7 m W	1	4,000	3,049	3,049	3.98
Ball mill - 5.00 m D x 9.14 m L	2	4,100	3,363	6,725	8.78
				10,130	**13.23**
Specific Energy Difference					**30%**
SABC base case - 75 µm					
SAG mill - 9.76 m D x 4.11 m EGL	1	8,200	8,120	8,120	10.60
Pebble crusher - HP 800	1	300	61	61	0.08
Ball mill - 5.00 m D x 9.14 m L	2	4,100	4,443	8,886	11.60
				17,066	**22.28**
HPGR - ball mill - 75 µm					
Secondary crusher - MP 800	1	600	356	356	0.47
HPGR - 2.2 m D x 1.55 m W	1	4,000	3,049	3,049	3.98
Ball mill - 5.00 m D x 9.14 m L	2	4,100	5,473	10,946	14.29
				14,351	**18.74**
Specific Energy Difference					**16%**
HPGR - stirred mill - 75 µm					
Secondary crusher - MP 800	1	600	356	356	0.47
1st HPGR - 1.7 m D x 1.4 m W	1	1,500	1,737	1,737	2.27
2nd HPGR - 2.2 m D x 1.55 m W	2	2,800	2,707	5,414	7.07
IsaMill™ - M10,000	2	2,600	1,746	3,493	4.56
				11,001	**14.36**
Specific Energy Difference					**36%**

6.2 Complete comminution circuit energy

The design of the conveyors, feeders, screens, and pumps systems was included in the flowsheet in order to perform the energy requirement comparison for the complete comminution circuit. The refined flowsheets for the HPGR - ball mill circuit and the HPGR - stirred mill circuit, with the inclusion of the main auxiliary components, are shown in Figure 6.3 and Figure 6.4.

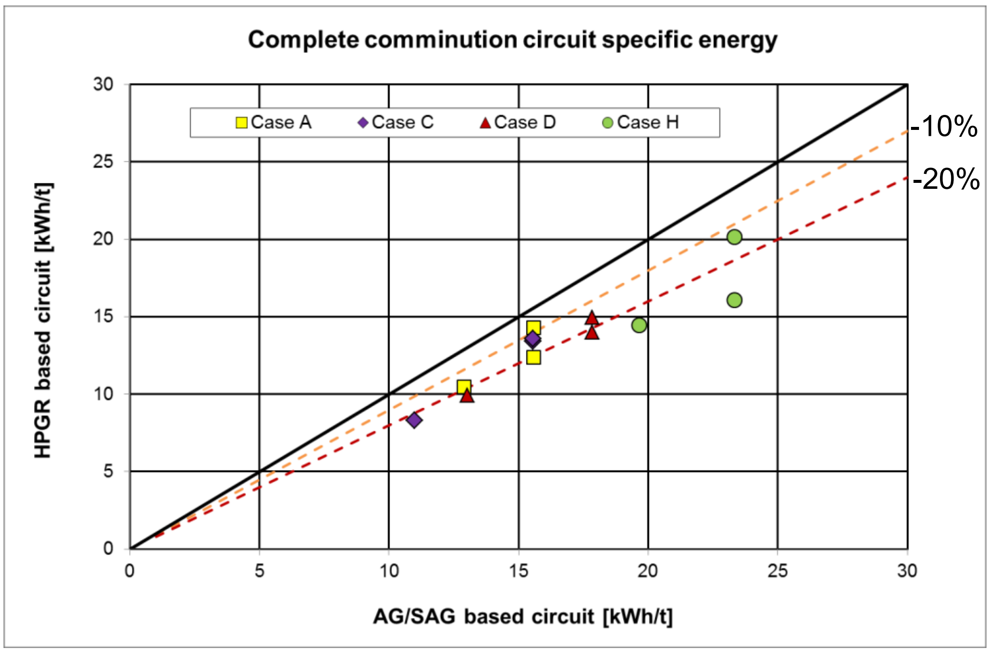

Figure 6.2 HPGR based circuit total kWh/t versus AG/SAG based circuit total kWh/t

Based on the established circuits, the estimation of the energy usage for the entire comminution circuit for each case is summarized in Table 6.5. Although the magnitude of the savings decreased when including the major auxiliary equipment, considerable energy reductions in the order of 8-31% were still achieved by using the HPGR-based circuits, as shown in Figure 6.2. The detailed breakdown of the energy consumption can be found in Appendix H.

For producing coarse grind size, the HPGR - ball mill circuit required 19-27% less energy as compared to the AG/SAG mill - ball mill circuit. When extending the target size to a finer grind size, the energy savings of the HPGR - ball mill circuit reduced to the order of 8-16%. The two-stage HPGR - stirred mill circuit showed a reduction in energy of 13-31%, as compared to the AG/SAG mill - ball mill circuit.

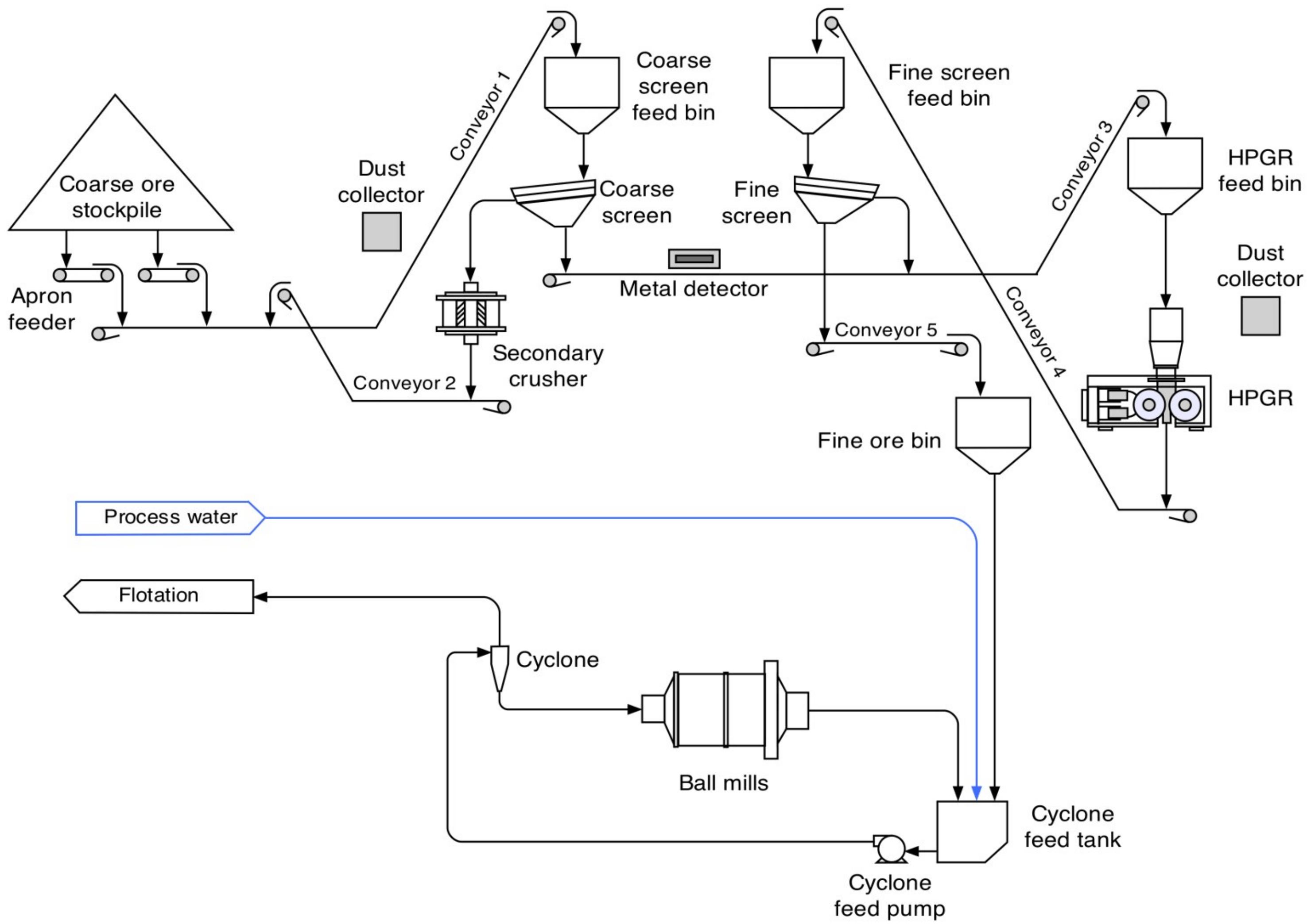

Figure 6.3 HPGR - ball mill circuit simplified flowsheet

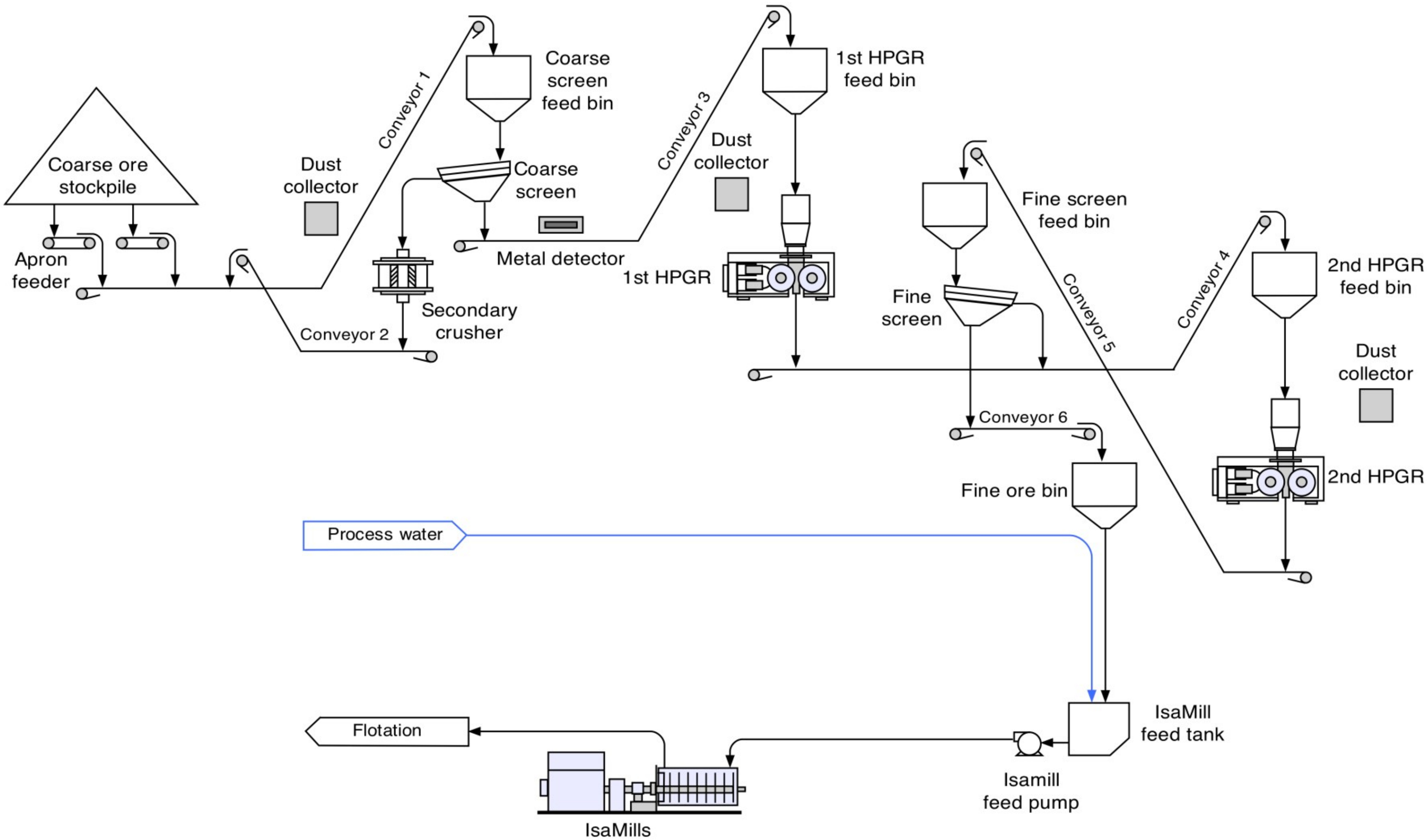

Figure 6.4 HPGR - stirred mill circuit simplified flowsheet

Table 6.5 Complete circuit energy

Description	Case A		Case C		Case D		Case H	
	Total power [kW]	Specific energy [kWh/t]	Total power [kW]	Specific energy [kWh/t]	Total power [kW]	Specific energy [kWh/t]	Total power [kW]	Specific energy [kWh/t]
AG/SAG ball mill (coarse grind)								
Comminution equipment	10,658	11.99	13,432	10.08	9,264	12.11	14,370	18.76
Auxiliary equipment	814	0.92	1,204	0.90	700	0.92	689	0.90
	11,472	**12.90**	14,635	**10.99**	9,964	**13.03**	15,060	**19.66**
HPGR - ball mill (coarse grind)								
Comminution equipment	8,247	9.28	9,545	7.17	6,656	8.70	10,130	13.23
Auxiliary equipment	1,055	1.19	1,521	1.14	934	1.22	932	1.22
	9,302	**10.46**	11,066	**8.31**	7,590	**9.92**	11,063	**14.44**
Specific Energy Difference		**19%**		**24%**		**24%**		**27%**
AG/SAG ball mill (fine grind)								
Comminution equipment	12,913	14.53	19,327	14.51	12,835	16.78	17,066	22.28
Auxiliary equipment	930	1.05	1,383	1.04	821	1.07	797	1.04
	13,843	**15.57**	20,711	**15.55**	13,655	**17.85**	17,863	**23.32**
HPGR - ball mill (fine grind)								
Comminution equipment	11,483	12.92	16,059	12.06	10,358	13.54	14,351	18.74
Auxiliary equipment	1,233	1.39	1,787	1.34	1,087	1.42	1,086	1.42
	12,715	**14.30**	17,846	**13.40**	11,446	**14.96**	15,437	**20.15**
Specific Energy Difference		**8%**		**14%**		**16%**		**14%**
HPGR - stirred mill (fine grind)								
Comminution equipment	9,968	11.21	16,477	12.37	9,815	12.83	11,001	14.36
Auxiliary equipment	1,028	1.16	1,640	1.23	892	1.17	1,321	1.72
	10,996	**12.37**	18,117	**13.60**	10,707	**14.00**	12,321	**16.09**
Specific Energy Difference		**21%**		**13%**		**22%**		**31%**

6.3 Capital and operating cost

To complete the comparison of the process options, a preliminary level of capital and operating cost assessment was performed for case H (Klein et al., 2012). The accuracy of capital and operating cost estimation is within plus or minus 50%. The capital cost estimates were based on base data of the second quarter of 2012 for major equipment, including HPGR and stirred mills. The indirect cost was estimated by applying a factor of 45% to the direct capital cost, which was considered to be within industry standards for the purpose of this research (Klein et al., 2012). The capital cost estimates are summarized in Table 6.6.

Table 6.6 Summary of capital cost estimate

Description	SABC 160 μm [M$]	SABC 75 μm [M$]	HPGR - ball mill [M$]	HPGRs - stirred mill [M$]
Equipment cost	36	38	39	47
Total direct costs	109	115	125	149
Total indirect costs	49	52	56	67
Total capital costs	**158**	**167**	**181**	**216**

The operating cost estimates in units of cost per tonne are summarized in Table 6.7. The average unit power cost used was USD$ 0.05 kWh. The maintenance cost was estimated by applying a factor (3%) to the direct capital cost for that area. The consumables cost, including liners, wear parts, and overhaul requirements for the crushing plant was estimated using Bruno simulation software (Klein et al., 2012). The grinding media consumption for the SAG mill and ball mills was determined using the Bond formulas and corrections by Rowland (Rowland & Rowland Jr., 2002). Technology improvements were also factored into the grinding media consumption (Klein et al., 2012). Labor cost was estimated based on 8 workers for the existing SABC circuit, and 10 workers for the HPGR - ball mill circuit and the HPGR - stirred mill circuit options, at a cost of USD$ 50,000 per worker per year.

Table 6.7 Summary of operating cost estimate

Description	Unit	SABC 160 µm	HPGR - ball mill 160 µm	SABC 75 µm	HPGR - ball mill 75 µm	HPGRs - stirred mill 75 µm
Power	[$/t]	1.08	0.86	1.17	1.08	0.77
Maintenance	[$/t]	0.54	0.62	0.57	0.62	0.74
Consumables	[$/t]	1.98	1.22	2.11	1.56	1.46
Labor	[$/t]	0.07	0.08	0.07	0.08	0.08
Total operating cost	**[$/t]**	**3.67**	**2.78**	**3.92**	**3.34**	**3.05**

As shown in Figure 6.5, it was found that both HPGR - ball mill and HPGR - stirred mill circuits had higher associated capital costs than the SABC option in case H. Conversely, operating costs for the two proposed circuits were substantially lower, which related directly to the lower energy consumption and consumables.

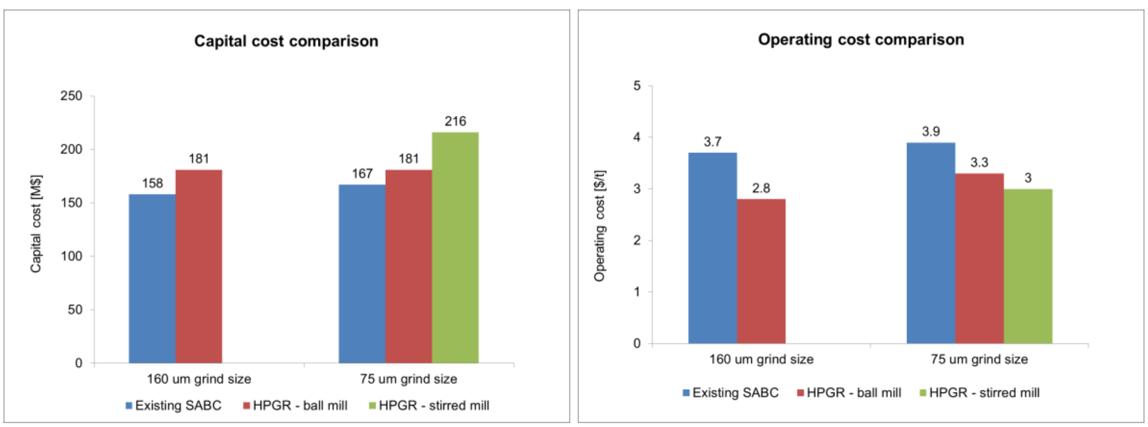

Figure 6.5 Left: capital cost comparison; right: operating cost comparison

As summarized in Table 6.8, the trade-off economics were evaluated on the basis of net present value (NPV). A discount rate of 5% and a 15-year mine life were assumed. At a grind size of 160 µm, the HPGR - ball mill circuit showed significant cost advantage over the SABC circuit with a NPV of $33 million, and an IRR of 22%. At a grind size of 75 µm, both options had cost advantages over the SABC option, although the HPGR - ball mill circuit had lower overall costs than the HPGR - stirred mill circuit. In general, a finer grind size would not be selected unless it resulted in significant recovery improvements. Since the copper-molybdenum recovery versus grind-size information was not available, such a comparison was not possible for the present study. However, in cases where a finer primary grind is needed to achieve high metal recoveries, the two-stage HPGR - stirred mill process demonstrates significant energy savings that would be reflected in the NPV.

Table 6.8 Net present value and internal rate of return

Description	Unit	HPGR - ball mill v.s. SABC @ 160 μm	HPGR - ball mill v.s. SABC @ 75 μm	HPGR - stirred mill v.s. SABC @ 75 μm
CAPEX difference	[M$]	-23	-14	-50
OPEX difference	[M$]	5	4	5
NPV @ 5%, 15 years	[M$]	33	22	5
IRR @ 5%, 15 years	[%]	22	23	7

6.4 Discussions

Results obtained from the research showed that the HPGR - ball mill circuit achieved a substantial reduction in energy with significant cost advantage over the existing SAB/AGBC/SABC circuits at coarser grind duties. The HPGR - stirred mill circuit became more favorable at finer grind duties.

Both the higher energy efficiency and elimination of steel grinding media associated with the HPGR-based circuit significantly reduced the determined operating costs. It is important to note that the energy evaluation did not take into account the power requirement of de-agglomerating equipment, which would be required to disperse HPGR product prior to being transported to the screens. However, the implementation of potentially effective dispersing equipment, such as a vertical shaft impactor (VSI), would certainly not exceed the predicted reduction in energy.

In addition, although the upfront capital requirements for the HPGR-based circuits are higher than the capital required for a conventional SABC circuit, the present study shows that a financial benefit can still be realized. Energy and consumables requirements are the main differences among the comminution circuits that offset the higher capital cost for HPGR and stirred mills. The shorter lead and erection times of an HPGR - ball mill circuit and an HPGR - stirred mill circuit versus a SAG mill circuit will result in earlier cash flow, which may also offset the disadvantage of higher capital costs, partially, if not completely.

CHAPTER 7: CONCLUSIONS AND RECOMMENDATIONS

7.1 Main research findings

This thesis focused on the study of low-grade, high-tonnage hard-rock comminution, and the major objective of the research was to enrich the understanding of the potential benefits of the HPGR and/or stirred mill circuits, as opposed to the conventional comminution circuit. In order to achieve the objective, literature was reviewed to identify the critical design and operating considerations for the construction of an evaluation program. The evaluation of the proposed circuits and existing circuits were supported by a combination of laboratory scale testing, pilot-scale testing and simulation work. The conclusions of the evaluation program can be summarized as follows,

- The combination of HPGR and stirred mill in a single flowsheet, without tumbling mills, has been demonstrated to be technically feasible, with the implementation of two passes of HPGRs in the flowsheet, and large-diameter ceramic media in IsaMill™ coarse stirred milling.
- The work has demonstrated that the HPGR - ball mill circuit as an alternative to existing SAB/AGBC/SABC comminution circuits has significant potential as an energy-efficient alternative. In the comparison of pure comminution equipment energy, the HPGR - ball mill circuit required an order of 23-30% less energy than the AG/SAG mill - ball mill circuit at coarse grind size. Energy savings of 19-27% were indicated using the HPGR - ball mill circuit when additional equipment was added.
- The work has shown that combining the two comminution technologies, HPGR and stirred mill, has considerable potential as an energy-efficient approach to grinding metallic ores to a fine grind size. When extending the target size to a finer grind size, the energy savings of the novel two-stage HPGR - stirred mill circuit were an order of

15-36%, as compared to the AG/SAG mill - ball mill circuit. Lower energy savings of 13-31% were indicated using the novel two-stage HPGR - stirred mill circuit when additional equipment was included.

- The work has also presented the financial evaluation for case H. The results showed that the HPGR - ball mill circuit had significant cost advantage over the SABC circuit with a NPV of $33 million and an IRR of 22% at the current grind size. At a finer grind size, both the HPGR - ball mill circuit and the HPGR - stirred mill circuit had cost advantages over the existing SABC option, although the HPGR - ball mill circuit had lower overall costs than the HPGR - stirred mill circuit.

Overall, the proposed HPGR - ball mill circuit and HPGR - stirred mill circuit have demonstrated significant potential as a means to grind more efficiently, this attribute being increasingly important as the mining industry is faced with extracting metals from harder and more complex deposits. Project economics would be further improved in regions where energy supply is more expensive than the relatively low energy unit costs used as a basis for this evaluation.

7.2 Future research opportunities

Some future opportunities are proposed as follows,

- Evaluation of the ore hardness variability effect on energy requirements and overall project economics for different circuit options. HPGRs are less sensitive to variation in ore hardness when compared to AG/SAG mills, thus an improved circuit performance can be realized when considering ore hardness in project evaluation.
- Examination of the differences in liberation characteristics from a SAG mill - ball mill operating in closed circuit with a classifying cyclone, and the HPGR - stirred mill operating in open circuit. With differences in particle breakage mechanisms, as well as mineral particle size distributions, it would not be surprising to find differences in degree of liberation.
- Further work is required to improve the classification of HPGR product. This is challenging when taking into account the detrimental effect of moisture on HPGR performance, thereby necessitating a dry classification process to limit the amount of moisture returned to the HPGR grinding section with oversize particles. To further develop the second-stage HPGR closed circuit, HPGR needs to produce finer product and seek alternatives to classification technologies, such as air classifier, etc.
- Evaluation of feed top size for coarse stirred milling with different media size. A media size of up to 8 mm should be tested on a top size of up to 1.2 mm feed, and signature plot comparison should be made to determine the optimal grinding efficiency for coarse stirred milling.
- Further work is also required to test other possible HPGR - stirred mill flowsheets, to identify the changes in performance when different flowsheets are in use.

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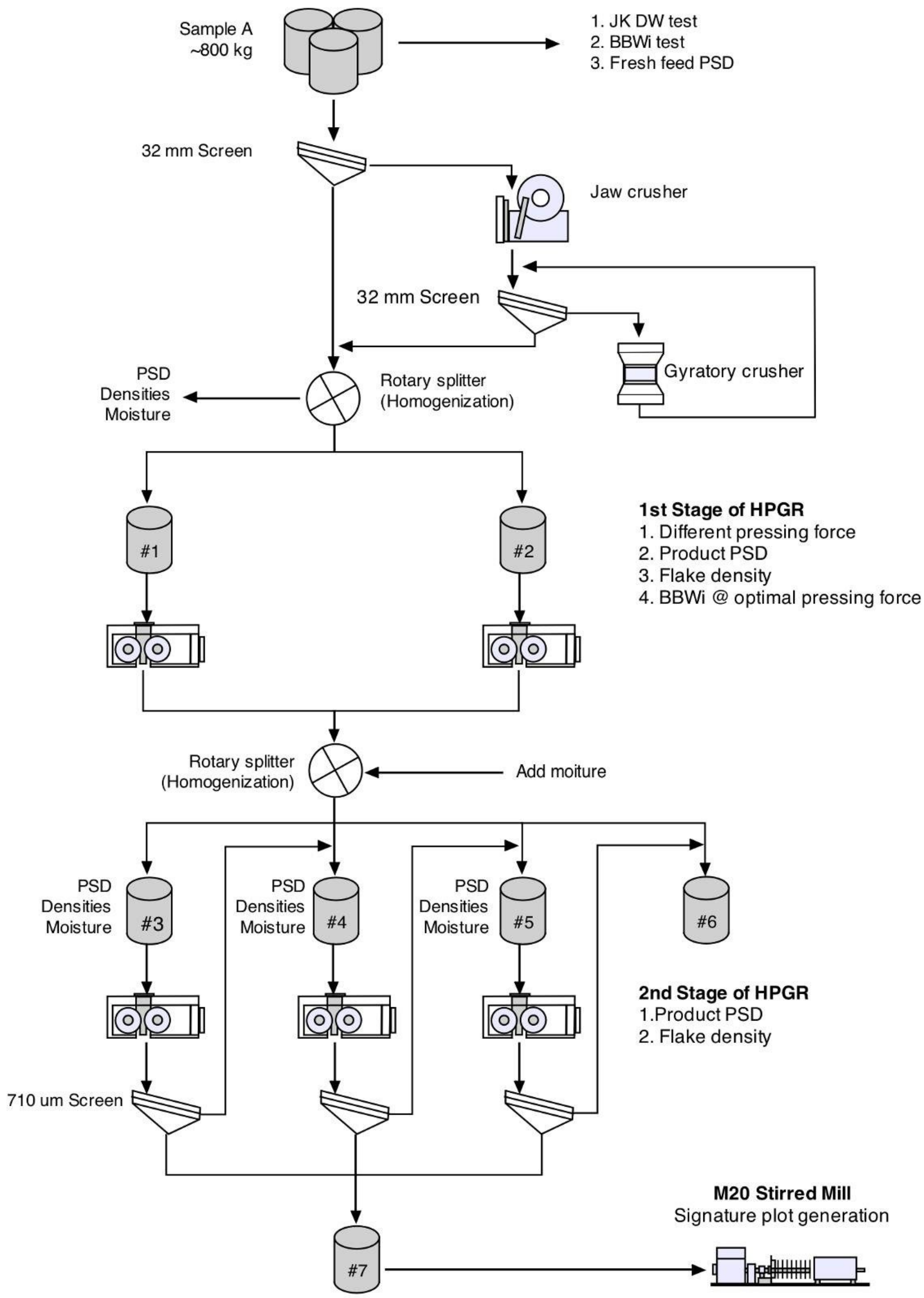
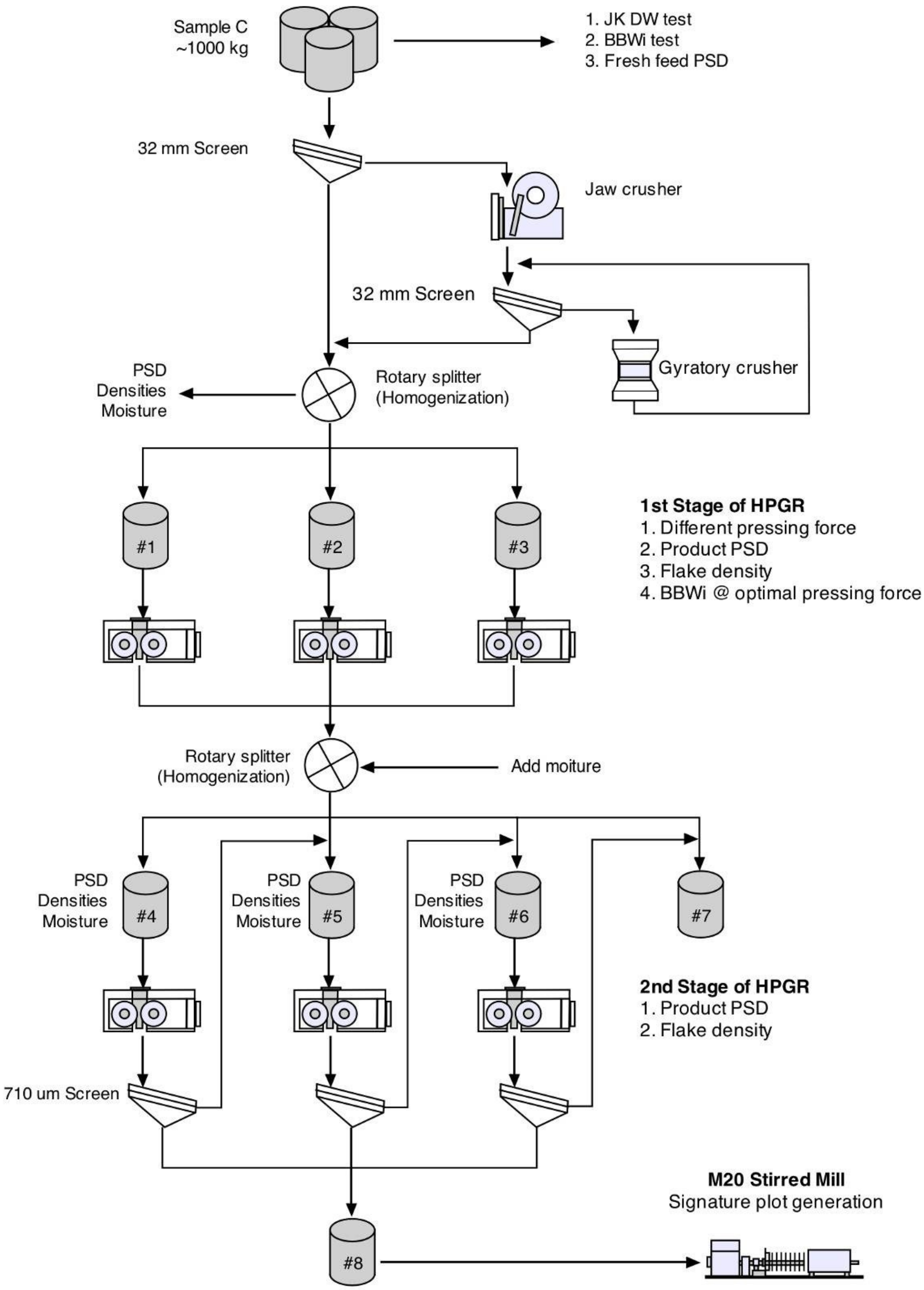
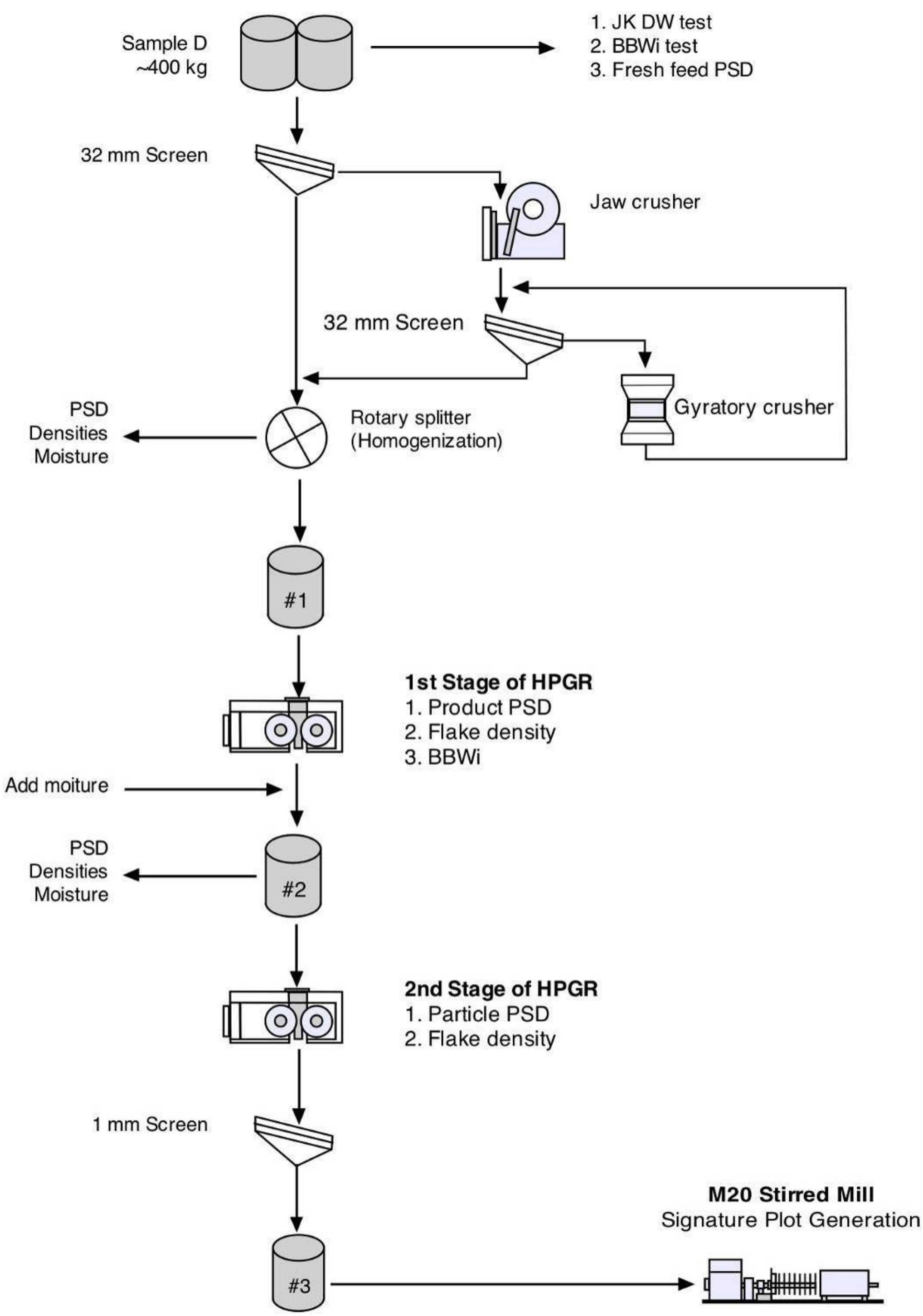
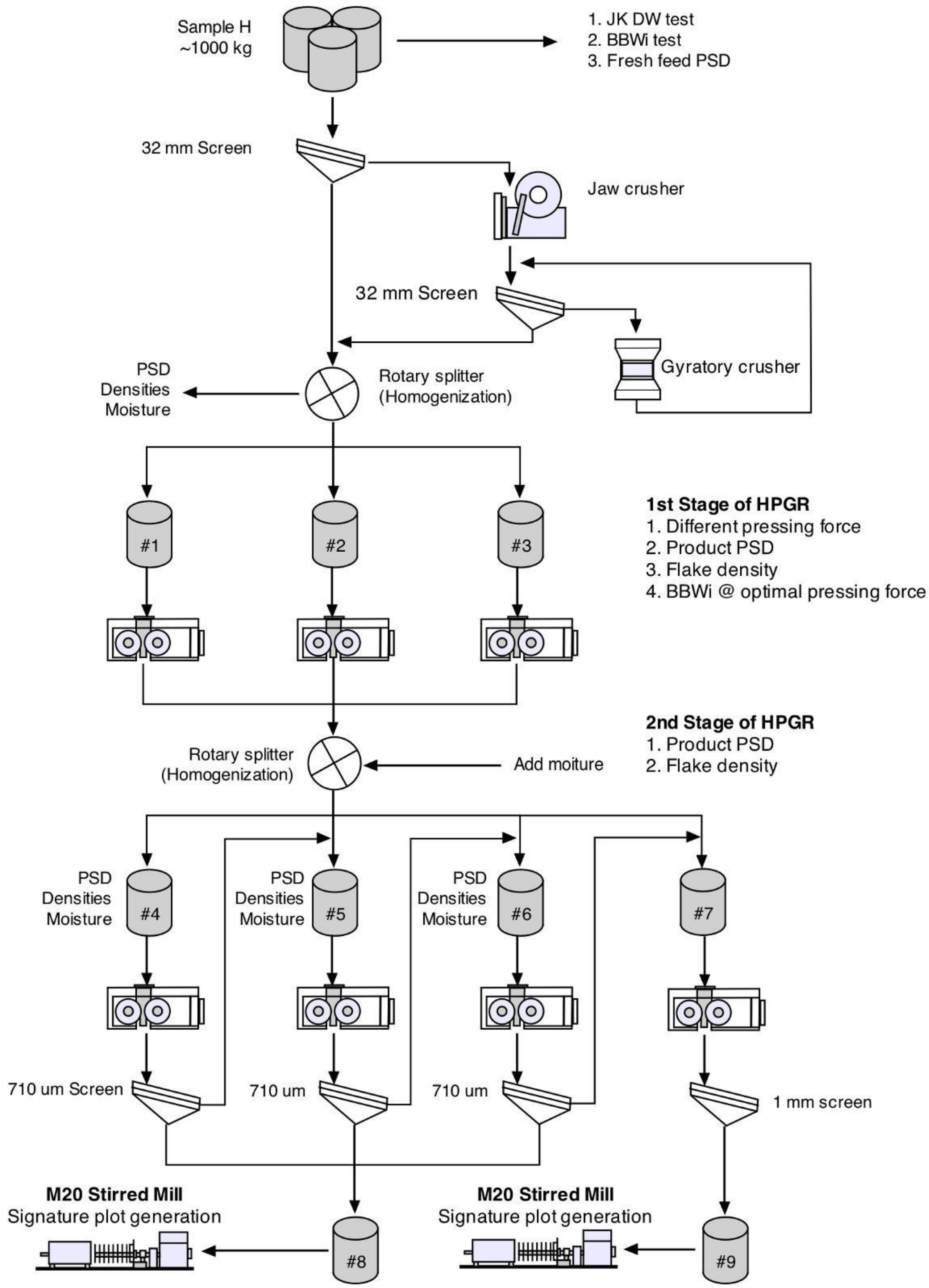
Appendices

Appendix A - Test flowsheet




Sample C
~1000 kg
1. JK DW test
2. BBWi test
3. Fresh feed PSD
32 mm Screen
Jaw crusher
32 mm Screen
Gyratory crusher
PSD
Densities
Moisture
Rotary splitter
(Homogenization)
#1
#2
#3
1st Stage of HPGR
1. Different pressing force
2. Product PSD
3. Flake density
4. BBWi @ optimal pressing force
Rotary splitter
(Homogenization)
Add moiture
PSD
Densities
Moisture
#4
PSD
Densities
Moisture
#5
PSD
Densities
Moisture
#6
#7
2nd Stage of HPGR
1. Product PSD
2. Flake density
710 um Screen
M20 Stirred Mill
Signature plot generation
#8


Sample D
~400 kg
1. JK DW test
2. BBWi test
3. Fresh feed PSD
32 mm Screen
Jaw crusher
32 mm Screen
Gyratory crusher
PSD
Densities
Moisture
Rotary splitter
(Homogenization)
#1
1st Stage of HPGR
1. Product PSD
2. Flake density
3. BBWi
Add moiture
PSD
Densities
Moisture
#2
2nd Stage of HPGR
1. Particle PSD
2. Flake density
1 mm Screen
M20 Stirred Mill
Signature Plot Generation
#3


Sample H
~1000 kg
1. JK DW test
2. BBWi test
3. Fresh feed PSD
32 mm Screen
Jaw crusher
32 mm Screen
Gyratory crusher
PSD
Densities
Moisture
Rotary splitter
(Homogenization)
#1
#2
#3
1st Stage of HPGR
1. Different pressing force
2. Product PSD
3. Flake density
4. BBWi @ optimal pressing force
2nd Stage of HPGR
1. Product PSD
2. Flake density
Rotary splitter
(Homogenization)
Add moiture
PSD
Densities
Moisture
#4
PSD
Densities
Moisture
#5
PSD
Densities
Moisture
#6
#7
710 um Screen
710 um
710 um
1 mm screen
M20 Stirred Mill
Signature plot generation
#8
M20 Stirred Mill
Signature plot generation
#9

Appendix B - HPGR experiment data

Data	Description	Symbol	Unit	A101	A102	A201	A202	A203
Press Constants	Roller Diameter (D)		[m]	0.750				
	Roller Width (W)		[m]	0.220				
Data	**Description**	**Test Number:**		**A101**	**A102**	**A201**	**A202**	**A203**
		Symbol	**Unit**					
Process Set Points	Speed	**n**	[m/s]	0.75	0.75	0.75	0.75	0.75
		n	[rpm]	19.10	19.10	19.10	19.10	19.10
	Static Gap	X_0	[mm]	9.0	9.0	9.0	9.0	9.0
	Hydraulic Pressure	**P**	[bar]	82	62	82	82	82
	Pressing Force	**F**	[kN]	660.0	495.0	660.0	660.0	660.0
	Specific Pressing Force	F_{SP}	**[N/mm^2]**	**4.0**	**3.0**	**4.0**	**4.0**	**4.0**
Process Data	**Test Time**	t	[s]	22.81	19.61	20.20	21.81	23.18
	Average Actual Speed:	w_{AV}	[m/s]	0.76	0.76	0.73	0.80	0.80
	Standard Deviation	s_w		0.07	0.10	0.12	0.29	0.30
	Actual Roller gap (average)	X_{gAV}	**[mm]**	**23.14**	**23.65**	**14.18**	**16.79**	**17.50**
	Standard Deviation	s_X		0.79	1.22	0.59	0.35	0.24
	Actual Hydraulic Pressure (average)	P_{AV}	**[bar]**	**80.8**	**62.4**	**81.1**	**82.0**	**81.8**
	Standard Deviation			0.81	0.43	0.75	0.28	0.43
	Actual Pressing Force (average)	F_{AV}	[kN]	650	502	652	659	658
	Actual Specific Pressure (average)	F_{SPAV}	**[N/mm^2]**	**3.95**	**3.05**	**3.96**	**4.01**	**4.00**
	Idle Power Draw	Pi	[kW]	11.57	11.56	11.57	10.14	11.77
	Power Draw	P	[kW]	66.28	55.66	68.05	69.48	67.41
	Total Specific Energy Consumption	E_{SP}	[kWh/t]	2.06	1.73	3.20	2.87	2.69
	Net Specific Energy Consumption	$E_{SP\ net}$	**[kWh/t]**	**1.70**	**1.37**	**2.66**	**2.45**	**2.22**
	Press throughput	W	[t/h]	32.23	32.09	21.24	24.18	25.02
	Specific Throughput Constant	**m dot**	**[ts/hm^3]**	**259**	**257**	**178**	**184**	**191**
Material Data	**Average Flake Density**	r_F	**[t/m^3]**	**2.23**	**2.25**	**N/A**	**2.25**	**2.24**
	Standard Deviation			0.02	0.04	N/A	0.01	0.01
	Flake Thickness Average	X_F	**[mm]**	**26.17**	**26.24**	**N/A**	**19.30**	**22.04**
	Standard Deviation			1.27	1.35	N/A	0.19	2.08
	Feed Moisture		**[%]**	**1.4%**	**1.4%**	**5.1%**	**4.2%**	**4.1%**
	Proctor Density (wet)		[t/m^3]	2.12	2.12	2.22	2.27	2.27
	Proctor Density (dry)		[t/m^3]	2.15	2.15	2.37	2.40	2.40
	Particle Size Distribution							
	Feed: 100% Passing Size	F_{100}	[mm]	32.00	32.00	19.00	19.00	19.00
	Feed: 80% Passing Size	F_{80}	[mm]	19.66	19.66	6.28	4.76	4.45
	Feed: 50% Passing Size	F_{80}	[mm]	8.30	8.30	1.62	1.23	1.06
	Centre: 80% Passing Size	P_{80}	[mm]	4.44	5.76	1.89	1.70	1.63
	Centre: 50% Passing Size	P_{50}	[mm]	1.18	1.78	0.45	0.49	0.54
	Edge: 80% Passing Size	P_{80}	[mm]	9.55	10.72	2.47	1.99	2.02
	Edge: 50% Passing Size	P_{50}	[mm]	3.28	3.91	0.67	0.57	0.52
	Combined 90% Center & 10% Edge: 80% Passing Size	P_{80}	[mm]	4.90	6.30	1.95	1.73	1.67
	Combined 90% Center & 10% Edge: 50% Passing Size	P_{50}	[mm]	1.31	1.91	0.46	0.50	0.54
	Reduction Ratio F80/P80 (Scaled Product)			4.01	3.12	3.22	2.75	2.66
	Reduction Ratio F50/P50 (Scaled Product)			6.34	4.35	3.52	2.46	1.96
	4 mm % Passing (Scaled Product)		[%]	75.5	67.0	91.0	92.8	92.3
	0.71 mm % Passing (Scaled Product)		[%]	34.6	33.0	61.7	58.4	57.9
	Mass Balance							
	Total Feed Material	M_F	[kg]	345	343	304	285	283
	Total Centre Product	M_C	[kg]	144.9	124.1	82.7	95.5	106.2
	Centre Product % of Centre & Edge Material	**MCE%**	[%]	71.0%	71.0%	69.3%	65.2%	65.9%
	Total Edge Product	M_E	[kg]	59.3	50.7	36.6	51.0	54.9
	Edge Product % of Centre & Edge Material	$M_{EF\%}$	[%]	29.0%	29.0%	30.7%	34.8%	34.1%
	Total Waste Product	M_W	[kg]	128	154	169	126	109
	Waste Product % of Total Feed	$M_{WF\%}$	[%]	37.1%	45.0%	55.6%	44.3%	38.3%
	Total Recovered Product	M_P	[kg]	332	329	289	273	270
	Mass Reconciliation (+ "gain; - "loss")	$M_{PF\%}$	[%]	-3.7%	-4.0%	-5.2%	-4.2%	-4.8%

Data	Description	Symbol	Unit	C101	C102	C103	C201	C202	C203
Press Constants	Roller Diameter (**D**)	[m]	0.750						
	Roller Width (**W**)	[m]	0.220						
Data	**Description**	**Test Number:**		**C101**	**C102**	**C103**	**C201**	**C202**	**C203**
		Symbol	**Unit**						
Process Set Points	Speed	**n**	[m/s]	0.75	0.75	0.75	0.75	0.75	0.75
		n	[rpm]	19.10	19.10	19.10	19.10	19.10	19.10
	Static Gap	X_0	[mm]	9.0	9.0	9.0	9.0	9.0	9.0
	Hydraulic Pressure	**P**	[bar]	82	61.5	51.3	82.1	82.1	82.1
	Pressing Force	**F**	[kN]	660.0	495.0	412.5	660.0	660.0	660.0
	Specific Pressing Force	F_{SP}	**[N/mm^2]**	**4.0**	**3.0**	**2.5**	**4.0**	**4.0**	**4.0**
Process Data	**Test Time**	t	[s]	19.59	19.59	19.20	28.81	19.00	20.01
	Average Actual Speed:	w_{AV}	[m/s]	0.74	0.77	0.75	0.75	0.77	0.75
	Standard Deviation	S_w		0.06	0.21	0.08	0.06	0.20	0.04
	Actual Roller gap (average)	X_{gAV}	**[mm]**	**21.51**	**23.66**	**24.35**	**12.56**	**16.62**	**17.50**
	Standard Deviation	s_X		0.50	0.60	0.97	1.00	0.22	0.36
	Actual Hydraulic Pressure (average)	P_{AV}	**[bar]**	**81.3**	**60.1**	**49.0**	**81.5**	**81.2**	**81.2**
	Standard Deviation			0.54	2.26	2.09	0.96	0.60	0.64
	Actual Pressing Force (average)	F_{AV}	[kN]	654	483	394	656	653	653
	Actual Specific Pressure (average)	F_{SPAV}	**[N/mm^2]**	**3.97**	**2.94**	**2.40**	**3.98**	**3.97**	**3.97**
	Idle Power Draw	Pi	[kW]	11.00	10.81	10.88	7.99	11.18	10.96
	Power Draw	P	[kW]	64.70	52.27	46.84	63.62	61.85	59.15
	Total Specific Energy Consumption	E_{SP}	[kWh/t]	2.04	1.56	1.33	3.28	2.61	2.29
	Net Specific Energy Consumption	$E_{SP\ net}$	**[kWh/t]**	**1.69**	**1.23**	**1.02**	**2.87**	**2.14**	**1.87**
	Press throughput	W	[t/h]	31.73	33.59	35.11	19.42	23.70	25.79
	Specific Throughput Constant	**m dot**	**[ts/hm^3]**	**259**	**266**	**285**	**157**	**188**	**208**
Material Data	**Average Flake Density**	r_F	**[t/m^3]**	**2.25**	**2.25**	**2.24**	**2.23**	**2.24**	**2.25**
	Standard Deviation			0.02	0.03	0.02	0.02	0.02	0.01
	Flake Thickness Average	X_F	**[mm]**	**25.09**	**26.32**	**27.82**	**19.31**	**21.80**	**22.99**
	Standard Deviation			2.00	1.96	2.30	1.29	0.50	0.48
	Feed Moisture		**[%]**	**0.6%**	**0.6%**	**0.6%**	**5.7%**	**4.4%**	**4.2%**
	Proctor Density (wet)		[t/m^3]	2.08	2.08	2.08	2.28	2.28	2.25
	Proctor Density (dry)		[t/m^3]	2.09	2.09	2.09	2.47	2.42	2.38
	Particle Size Distribution								
	Feed: 100% Passing Size	F_{100}	[mm]	32	32	32	19	19	19
	Feed: 80% Passing Size	F_{80}	[mm]	21.93	21.93	21.93	5.48	4.20	4.66
	Feed: 50% Passing Size	F_{80}	[mm]	10.68	10.68	10.68	1.35	1.29	1.41
	Centre: 80% Passing Size	P_{80}	[mm]	6.08*	6.01	7.39	2.54*	1.97*	1.85
	Centre: 50% Passing Size	P_{50}	[mm]	1.41*	1.43	1.72	0.91*	0.75*	0.77
	Edge: 80% Passing Size	P_{80}	[mm]	11.07*	10.58	11.97	2.92*	2.42*	2.40
	Edge: 50% Passing Size	P_{50}	[mm]	3.5*	3.28	4.27	0.88*	0.66*	0.73
	Combined 90% Center & 10% Edge: 80% Passing Size	P_{80}	[mm]	6.63	6.54	7.97	2.57	2.00	1.88
	Combined 90% Center & 10% Edge: 50% Passing Size	P_{50}	[mm]	1.55	1.58	1.87	0.91	0.74	0.76
	Reduction Ratio F80/P80 (Scaled Product)			3.31	3.35	2.75	2.13	2.10	2.48
	Reduction Ratio F50/P50 (Scaled Product)			6.89	6.76	5.71	1.48	1.74	1.86
	4 mm % Passing (Scaled Product)		[%]	67.8	67.6	63.9	90.3	97.0	93.8
	0.71 mm % Passing (Scaled Product)		[%]	37.1	35.6	33.7	45.0	48.9	48.2
	Mass Balance								
	Total Feed Material	M_F	[kg]	302	310	316	247	241	243
	Total Centre Product	M_C	[kg]	128.9	131.6	133.9	51.0	67.8	100.7
	Centre Product % of Centre & Edge Material	**MCE%**	[%]	75.2%	72.5%	72.0%	55.5%	54.2%	70.3%
	Total Edge Product	M_E	[kg]	42.4	49.8	52.2	40.9	57.3	42.6
	Edge Product % of Centre & Edge Material	$M_{EF\%}$	[%]	24.8%	27.5%	28.0%	44.5%	45.8%	29.7%
	Total Waste Product	M_W	[kg]	121	119	120	147	110	91
	Waste Product % of Total Feed	$M_{WF\%}$	[%]	40.0%	38.5%	38.0%	59.6%	45.6%	37.5%
	Total Recovered Product	M_P	[kg]	292	301	306	239	235	234
	Mass Reconciliation (+ "gain; - "loss")	$M_{PF\%}$	[%]	-3.3%	-3.0%	-3.0%	-3.1%	-2.4%	-3.4%

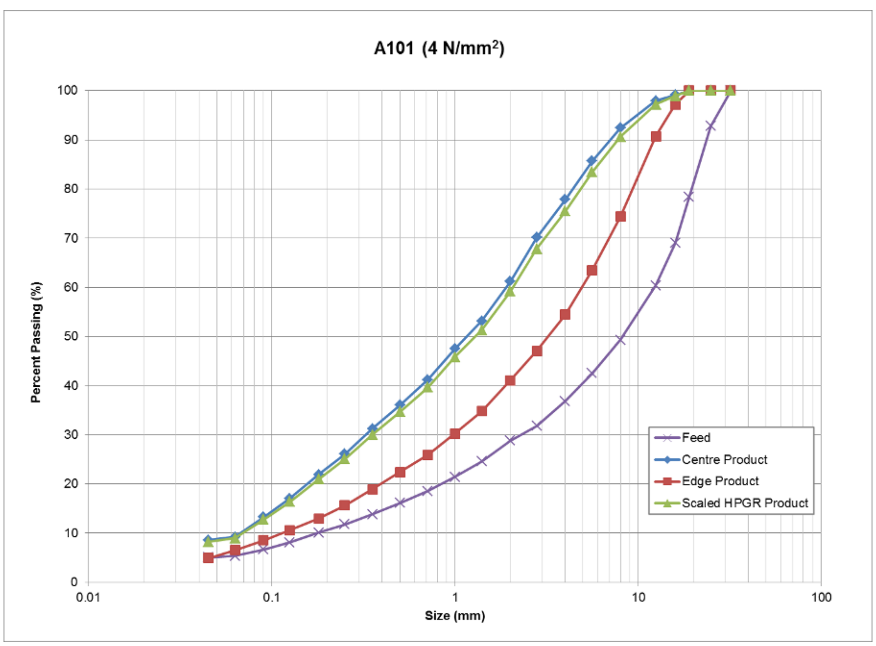
Press Constants	Roller Diameter (**D**)	[m]	0.750		
	Roller Width (**W**)	[m]	0.220		
Data	**Description**	**Test Number:**		**D101**	**D201**
		Symbol	**Unit**		
Process Set Points	Speed	**n**	[m/s]	0.75	0.75
		n	[rpm]	19.10	19.10
	Static Gap	$\mathbf{X_0}$	[mm]	9.0	9.0
	Hydraulic Pressure	**P**	[bar]	62	82.1
	Pressing Force	**F**	[kN]	495.0	660.0
	Specific Pressing Force	$\mathbf{F_{SP}}$	**[N/mm^2]**	**3.0**	**4.0**
Process Data	**Test Time**	t	[s]	20.04	16.46
	Average Actual Speed:	w_{AV}	[m/s]	0.79	0.78
	Standard Deviation	s_w		0.27	0.24
	Actual Roller gap (average)	$\mathbf{X_{gAV}}$	**[mm]**	**23.32**	**12.10**
	Standard Deviation	s_X		0.69	0.39
	Actual Hydraulic Pressure (average)	$\mathbf{P_{AV}}$	**[bar]**	**61.8**	**82.2**
	Standard Deviation			0.60	0.40
	Actual Pressing Force (average)	$\mathbf{F_{AV}}$	[kN]	497	661
	Actual Specific Pressure (average)	$\mathbf{F_{SPAV}}$	**[N/mm^2]**	**3.02**	**4.02**
	Idle Power Draw	Pi	[kW]	9.97	11.99
	Power Draw	P	[kW]	59.09	77.78
	Total Specific Energy Consumption	E_{SP}	[kWh/t]	1.87	4.25
	Net Specific Energy Consumption	$\mathbf{E_{SP\,net}}$	**[kWh/t]**	**1.55**	**3.60**
	Press throughput	W	[t/h]	31.67	18.29
	Specific Throughput Constant	**m dot**	**[ts/hm^3]**	**244**	**142**
Material Data	**Average Flake Density**	$\mathbf{r_F}$	**[t/m^3]**	**2.25**	**2.24**
	Standard Deviation			0.02	0.01
	Flake Thickness Average	$\mathbf{X_F}$	**[mm]**	**25.46**	**17.87**
	Standard Deviation			1.20	0.43
	Feed Moisture		**[%]**	**2.1%**	**5.3%**
	Proctor Density (wet)		[t/m^3]	2.08	2.33
	Proctor Density (dry)		[t/m^3]	2.10	2.52
	Particle Size Distribution				
	Feed: 100% Passing Size	F_{100}	[mm]	32	16
	Feed: 80% Passing Size	F_{80}	[mm]	21.94	6.03
	Feed: 50% Passing Size	F_{80}	[mm]	11.46	1.41
	Centre: 80% Passing Size	P_{80}	[mm]	4.30	1.66
	Centre: 50% Passing Size	P_{50}	[mm]	1.03	0.55
	Edge: 80% Passing Size	P_{80}	[mm]	8.68	2.05
	Edge: 50% Passing Size	P_{50}	[mm]	3.00	0.56
	Combined 90% Center & 10% Edge: 80% Passing Size	P_{80}	[mm]	4.70	1.71
	Combined 90% Center & 10% Edge: 50% Passing Size	P_{50}	[mm]	1.17	0.55
	Reduction Ratio F80/P80 (Scaled Product)			4.67	3.53
	Reduction Ratio F50/P50 (Scaled Product)			9.79	2.56
	4 mm % Passing (Scaled Product)		[%]	76.3	92.3
	1 mm % Passing (Scaled Product)		[%]	47.8	64.6
	Mass Balance				
	Total Feed Material	$\mathbf{M_F}$	[kg]	293	256
	Total Centre Product	$\mathbf{M_C}$	[kg]	123.5	54.2
	Centre Product % of Centre & Edge Material	**MCE%**	[%]	70.1%	64.8%
	Total Edge Product	$\mathbf{M_E}$	[kg]	52.8	29.4
	Edge Product % of Centre & Edge Material	$\mathbf{M_{EF\%}}$	[%]	29.9%	35.2%
	Total Waste Product	$\mathbf{M_W}$	[kg]	111	164
	Waste Product % of Total Feed	$\mathbf{M_{WF\%}}$	[%]	38.0%	64.2%
	Total Recovered Product	$\mathbf{M_P}$	[kg]	288	248
	Mass Reconciliation (+ "gain; - "loss")	$\mathbf{M_{PF\%}}$	[%]	-1.7%	-3.1%

Data	Description	Symbol	Unit	H101	H102	H103	H201	H202	H203
Press Constants	Roller Diameter (D)		[m]	0.750					
	Roller Width (W)		[m]	0.220					
		Test Number:							
		Symbol	**Unit**						
Process Set Points	Speed	**n**	[m/s]	0.75	0.75	0.75	0.75	0.75	0.75
		n	[rpm]	19.10	19.10	19.10	19.10	19.10	19.10
	Static Gap	X_0	[mm]	9.0	9.0	9.0	9.0	9.0	9.0
	Hydraulic Pressure	**P**	[bar]	51.3	61.5	82	61.5	61.5	61.5
	Pressing Force	**F**	[kN]	412.5	495.0	660.0	495.0	495.0	495.0
	Specific Pressing Force	F_{SP}	**[N/mm²]**	**2.5**	**3.0**	**4.0**	**3.0**	**3.0**	**3.0**
Process Data	**Test Time**	t	[s]	19.40	18.60	20.60	20.01	22.00	19.45
	Average Actual Speed:	w_{AV}	[m/s]	0.75	0.76	0.76	0.76	0.76	0.76
	Standard Deviation	s_w		0.08	0.17	0.15	0.12	0.14	0.15
	Actual Roller gap (average)	X_{gAV}	**[mm]**	**18.06**	**16.83**	**15.72**	**16.49**	**16.96**	**17.81**
	Standard Deviation	s_X		0.61	0.48	0.40	0.61	0.21	0.72
	Actual Hydraulic Pressure (average)	P_{AV}	**[bar]**	**50.0**	**61.4**	**81.4**	**60.9**	**67.4**	**61.5**
	Standard Deviation			1.27	0.59	0.65	0.53	5.72	0.92
	Actual Pressing Force (average)	F_{AV}	[kN]	402	494	654	490	542	494
	Actual Specific Pressure (average)	F_{SPAV}	**[N/mm²]**	**2.45**	**3.00**	**3.98**	**2.98**	**3.29**	**3.00**
	Idle Power Draw	Pi	[kW]	10.84	10.81	10.64	10.35	10.72	10.67
	Power Draw	P	[kW]	46.79	54.24	65.35	52.80	51.23	45.21
	Total Specific Energy Consumption	E_{SP}	[kWh/t]	1.96	2.36	3.05	1.96	1.93	1.64
	Net Specific Energy Consumption	$E_{SP\ net}$	**[kWh/t]**	**1.51**	**1.89**	**2.56**	**1.58**	**1.53**	**1.25**
	Press throughput	W	[t/h]	23.85	23.03	21.41	26.91	26.56	27.61
	Specific Throughput Constant	**m dot**	**[ts/hm³]**	**194**	**184**	**172**	**217**	**213**	**222**
Material Data	**Average Flake Density**	r_F	**[t/m³]**	**2.38**	**2.39****	**2.39**	**2.34**	**2.36**	**2.34**
	Standard Deviation			0.02	**NA**	0.04	0.02	0.01	0.01
	Flake Thickness Average	X_F	**[mm]**	**20.06**	**21.97**	**22.5**	**17.54**	**22.10**	**21.89**
	Standard Deviation			1.81	4.17	1.8	2.41	1.69	1.65
	Feed Moisture		**[%]**	**3.0%**	**3.0%**	**3.0%**	**5.0%**	**5.0%**	**5.0%**
	Bulk Density (wet)		[t/m³]	1.70	1.70	1.70	1.74	1.74	1.74
	Proctor Density (wet)		[t/m³]	2.10	2.10	2.10	2.38	2.38	2.38
	Particle Size Distribution								
	Feed: 100% Passing Size	F_{100}	[mm]	32.00	32.00	32.00	32.00	32.00	32.00
	Feed: 80% Passing Size	F_{80}	[mm]	23.58	23.58	23.58	7.42	5.20	4.97
	Feed: 50% Passing Size	F_{80}	[mm]	14.23	14.23	14.23	3.06	2.45	2.38
	Centre: 80% Passing Size	P_{80}	[mm]	7.04	6.18	6.05	4.44	3.96	3.84
	Centre: 50% Passing Size	P_{50}	[mm]	3.04	2.88	2.58	2.00	1.89	1.76
	Edge: 80% Passing Size	P_{80}	[mm]	10.11	8.93	7.69	4.43	3.99	3.75
	Edge: 50% Passing Size	P_{50}	[mm]	4.76	4.17	3.50	1.87	1.76	1.67
	Combined 90% Center & 10% Edge: 80% Passing Size	P_{80}	[mm]	7.37	6.50	6.26	4.44	3.97	3.83
	Combined 90% Center & 10% Edge: 50% Passing Size	P_{50}	[mm]	3.18	3.00	2.66	1.98	1.88	1.75
	Reduction Ratio F80/P80 (Scaled Product)			3.20	3.63	3.77	1.67	1.31	1.30
	Reduction Ratio F50/P50 (Scaled Product)			4.48	4.74	5.35	1.55	1.30	1.36
	4 mm % Passing (Scaled Product)		[%]	58.7%	60.8%	64.4%	76.8%	80.4%	82.1%
	0.71 mm % Passing (Scaled Product)		[%]	18.9%	20.2%	22.4%	26.2%	25.5%	27.4%
	Mass Balance								
	Total Feed Material	M_F	[kg]	249	246	236	310	257	202
	Total Centre Product	M_C	[kg]	81.0	75.5	84.5	73.4	82.9	92.0
	Centre Product % of Centre & Edge Material	**MCE%**	[%]	63.0%	63.4%	69.0%	62.0%	51.5%	62.4%
	Total Edge Product	M_E	[kg]	47.5	43.5	38.0	45.0	78.0	55.4
	Edge Product % of Centre & Edge Material	$M_{EF\%}$	[%]	37.0%	36.6%	31.0%	38.0%	48.5%	37.6%
	Edge Product % of Centre Product	$M_{EC\%}$	[%]	59%	58%	45%	61%	94%	60%
	Total Waste Product	M_W	[kg]	111	123	102	179	88	51
	Waste Product % of Total Feed	$M_{WF\%}$	[%]	44.6%	49.8%	43.3%	57.8%	34.3%	25.1%
	Total Recovered Product	M_P	[kg]	240	242	225	297	249	198
	Mass Reconciliation (+ "gain; - "loss")	$M_{PF\%}$	[%]	4.0%	1.9%	4.9%	4.2%	3.1%	1.8%

HPGR Test A101

Pressure	4.0	N/mm2	Feed	344.8	Kg
Moisture	1.4	%	Center Product	144.9	Kg
			Edge Product	59.3	Kg
			Waste	128	Kg
			Center Product %	71.0%	
			Edge Product %	29.0%	

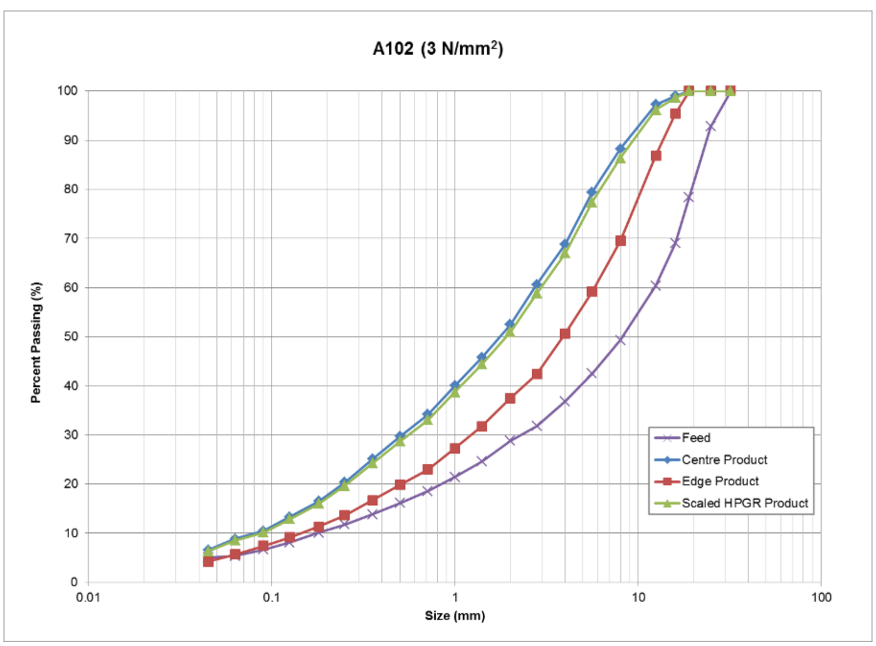
Sample No.		**Feed**		**Centre Product**		**Edge Product**		**Experimental Full PSD**	**Scaled HPGR Product**
A101									**90% Center + 10% Edge**
Screen Size	Particle Size	Retained	Cum. Passing	Retained	Cum. Passing	Retained	Cum. Passing	Cum. Passing	Cum. Passing
[mm]	**[mm]**	**[g]**	**[%]**	**[g]**	**[%]**	**[g]**	**[%]**	**[%]**	**[%]**
-35.5 to +32	32	0.0	100	0.0	100	0.0	100	100	100
-32 to +25	25	470.1	92.8	0.0	100	0.0	100	100	100
-25 to +19	19	937.6	78.4	0.0	100	0.0	100	100	100
-19 to 16	16	615.5	69.0	60.3	99.1	168.3	97.2	98.5	98.9
-16 to +12.5	12.5	562.5	60.3	79.8	97.9	385.6	90.7	95.8	97.2
-12.5 to +8	8	722.2	49.3	370.9	92.4	978.3	74.3	87.2	90.6
-8 to +5.6	5.6	442.7	42.5	451.8	85.7	649.8	63.5	79.2	83.5
-5.6 to +4	4	369.4	36.8	527.7	77.8	537.7	54.5	71.1	75.5
-4 to +2.8	2.8	329.2	31.8	518.0	70.1	445.1	47.0	63.4	67.8
-2.8 to +2	2	195.5	28.8	602.9	61.2	357.8	41.0	55.3	59.2
-2 to +1.4	1.4	276.1	24.5	541.8	53.1	374.1	34.8	47.8	51.3
-1.4 to +1	1	207.2	21.4	377.9	47.5	269.9	30.2	42.5	45.8
-1 to +.71	0.71	185.5	18.5	426.1	41.2	260.7	25.9	36.7	39.6
-.71 to +.5	0.5	159.3	16.1	347.0	36.0	209.8	22.4	32.0	34.6
-.5 to +.355	0.355	146.7	13.8	321.0	31.2	206.3	18.9	27.6	30.0
-.355 to +.25	0.25	133.8	11.8	345.3	26.1	198.7	15.6	23.0	25.0
-.25 to +.18	0.18	112.6	10.0	284.0	21.9	158.1	12.9	19.3	21.0
-.18 to +.125	0.125	127.2	8.1	325.9	17.0	142.3	10.6	15.1	16.4
-.125 to +.09	0.09	94.4	6.6	255.0	13.2	121.2	8.5	11.9	12.8
-.09 to +.063	0.063	87.2	5.3	270.7	9.2	119.9	6.5	8.4	8.9
-.063 to +.045	0.045	20.6	5.0	44.1	8.5	96.2	4.9	7.5	8.2
-0.045	Pan	325.7		574.8		293.5			
Total		6521		6725		5973.3			
Size Distribution Interpolations									
Linear P50	[mm]		**8.30**		**1.18**		**3.28**	**1.58**	**1.31**
Linear P80	[mm]		**19.66**		**4.44**		**9.55**	**5.83**	**4.90**



HPGR Test A102

Pressure	3.0	N/mm2	Feed	342.9	Kg
Moisture	1.4	%	Center Product	124.1	Kg
			Edge Product	50.7	Kg
			Waste	154.4	Kg
			Center Product %	71.0%	
			Edge Product %	29.0%	

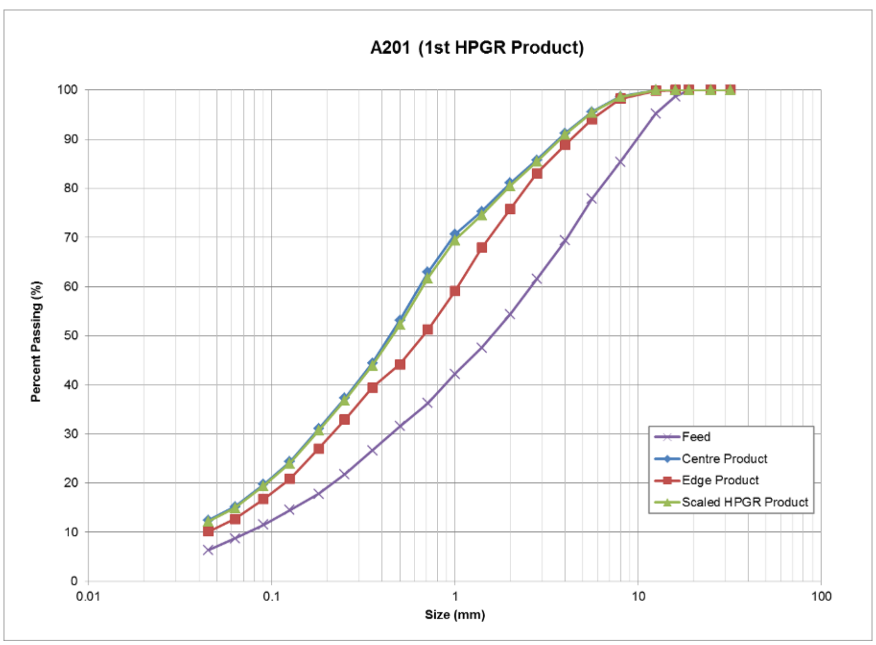
Sample No.		Feed		Centre Product		Edge Product		Experimental Full PSD	Scaled HPGR Product
A102									90% Center + 10% Edge
Screen Size	Particle Size	Retained	Cum. Passing	Retained	Cum. Passing	Retained	Cum. Passing	Cum. Passing	Cum. Passing
[mm]	[mm]	[g]	[%]	[g]	[%]	[g]	[%]	[%]	[%]
-35.5 to +32	32	0	100	0.0	100	0.0	100	100	100
-32 to +25	25	470.1	92.8	0.0	100	0.0	100	100	100
-25 to +19	19	937.6	78.4	0.0	100	0.0	100	100	100
-19 to 16	16	615.5	69.0	75.0	99.0	282.6	95.4	97.9	98.6
-16 to +12.5	12.5	562.5	60.3	130.6	97.2	521.2	86.9	94.2	96.2
-12.5 to +8	8	722.2	49.3	669.1	88.2	1068.2	69.5	82.8	86.3
-8 to +5.6	5.6	442.7	42.5	653.1	79.4	632.9	59.1	73.5	77.4
-5.6 to +4	4	369.4	36.8	784.8	68.8	521.1	50.6	63.6	67.0
-4 to +2.8	2.8	329.2	31.8	610.4	60.6	508.5	42.4	55.3	58.8
-2.8 to +2	2	195.5	28.8	604.4	52.5	300.9	37.4	48.1	51.0
-2 to +1.4	1.4	276.1	24.5	499.9	45.7	353.0	31.7	41.7	44.3
-1.4 to +1	1	207.2	21.4	428.0	40.0	274.5	27.2	36.3	38.7
-1 to +.71	0.71	185.5	18.5	432.6	34.1	260.6	23.0	30.9	33.0
-.71 to +.5	0.5	159.3	16.1	330.7	29.7	191.3	19.8	26.8	28.7
-.5 to +.355	0.355	146.7	13.8	343.4	25.1	194.6	16.7	22.6	24.2
-.355 to +.25	0.25	133.8	11.8	350.2	20.3	185.9	13.6	18.4	19.7
-.25 to +.18	0.18	112.6	10.0	285.9	16.5	144.7	11.3	15.0	16.0
-.18 to +.125	0.125	127.2	8.1	236.6	13.3	130.5	9.2	12.1	12.9
-.125 to +.09	0.09	94.4	6.6	212.9	10.4	108.9	7.4	9.5	10.1
-.09 to +.063	0.063	87.2	5.3	119.8	8.8	107.9	5.6	7.9	8.5
-.063 to +.045	0.045	20.6	5.0	166.9	6.6	87.9	4.2	5.9	6.3
-0.045	Pan	325.7		487.5		256.5			
Total		6521		7421.8		6131.7			
Size Distribution Interpolations									
Linear P50	[mm]		**8.30**		**1.78**		**3.91**	**2.21**	**1.91**
Linear P80	[mm]		**19.66**		**5.76**		**10.72**	**7.28**	**6.30**



HPGR Test A201

Pressure	4.0	N/mm2	Feed	304.2	Kg
Moisture	5.1	%	Center Product	82.7	Kg
			Edge Product	36.6	Kg
			Waste	169.2	Kg
			Center Product %	69.3%	
			Edge Product %	30.7%	

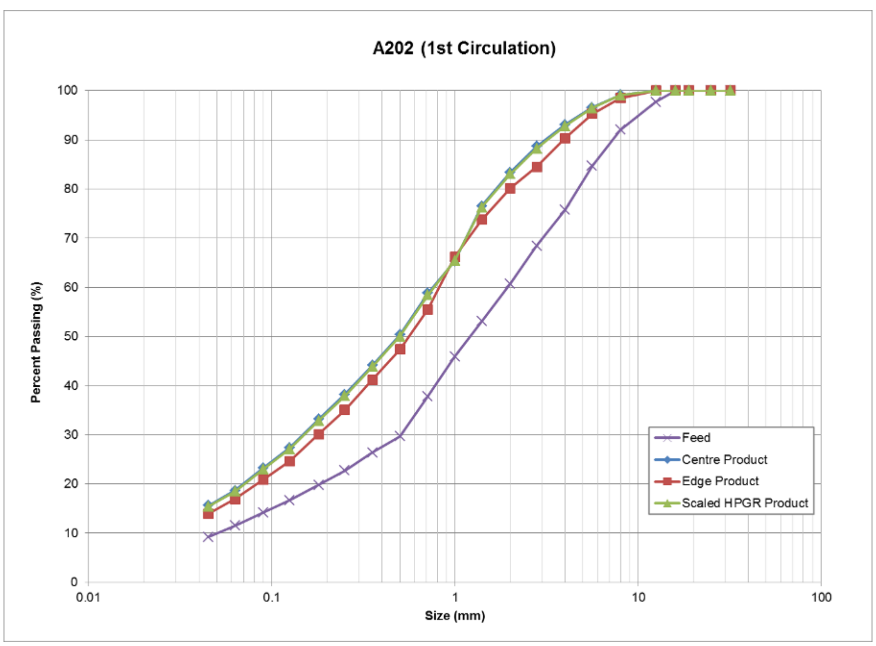
Sample No.		Feed		Centre Product		Edge Product		Experimental Full PSD	Scaled HPGR Product
A201									90% Center + 10% Edge
Screen Size	Particle Size	Retained	Cum. Passing	Retained	Cum. Passing	Retained	Cum. Passing	Cum. Passing	Cum. Passing
[mm]	[mm]	[g]	[%]	[g]	[%]	[g]	[%]	[%]	[%]
-35.5 to +32	32	0.0	100	0.0	100	0.0	100	100	100
-32 to +25	25	0.0	100	0.0	100	0.0	100	100	100
-25 to +19	19	0.0	100	0.0	100	0.0	100	100	100
-19 to 16	16	84.9	98.7	0.0	100	0.0	100	100	100
-16 to +12.5	12.5	224.6	95.2	0.0	100	7.0	100	100	100
-12.5 to +8	8	640.8	85.4	68.4	98.7	70.8	98.2	98.6	98.7
-8 to +5.6	5.6	483.7	77.9	164.7	95.6	182.8	94.1	95.1	95.4
-5.6 to +4	4	554.2	69.4	230.0	91.2	228.8	88.8	90.5	91.0
-4 to +2.8	2.8	506.4	61.5	290.6	85.7	255.7	83.0	84.9	85.4
-2.8 to +2	2	463.3	54.4	245.7	81.1	316.1	75.8	79.4	80.5
-2 to +1.4	1.4	447.1	47.5	306.0	75.3	346.5	67.9	73.0	74.5
-1.4 to +1	1	345.7	42.2	245.0	70.6	389.1	59.0	67.1	69.5
-1 to +.71	0.71	383	36.3	406.4	62.9	344.7	51.2	59.3	61.7
-.71 to +.5	0.5	308	31.5	517.5	53.1	306.6	44.2	50.4	52.2
-.5 to +.355	0.355	317.7	26.6	458.6	44.4	206.8	39.5	42.9	43.9
-.355 to +.25	0.25	317.2	21.7	378.7	37.2	288.6	32.9	35.9	36.8
-.25 to +.18	0.18	259	17.8	325.9	31.1	257.5	27.0	29.8	30.7
-.18 to +.125	0.125	211.8	14.5	354.5	24.3	272.1	20.8	23.3	24.0
-.125 to +.09	0.09	194.7	11.5	243.8	19.7	178.9	16.7	18.8	19.4
-.09 to +.063	0.063	181.2	8.7	240.4	15.2	177.3	12.7	14.4	14.9
-.063 to +.045	0.045	154.1	6.3	143.9	12.4	112.8	10.1	11.7	12.2
-0.045	Pan	409.9		655.8		444.0			
Total		6487.3		5275.9		4386.1			
Size Distribution Interpolations									
Linear P50	[mm]		**1.62**		**0.45**		**0.67**	**0.49**	**0.46**
Linear P80	[mm]		**6.28**		**1.89**		**2.47**	**2.08**	**1.95**



HPGR Test A202

Pressure	4.0	N/mm2	Feed	284.8	Kg
Moisture	4.2	%	Center Product	95.5	Kg
			Edge Product	51	Kg
			Waste	126.3	Kg
			Center Product %	65.2%	
			Edge Product %	34.8%	

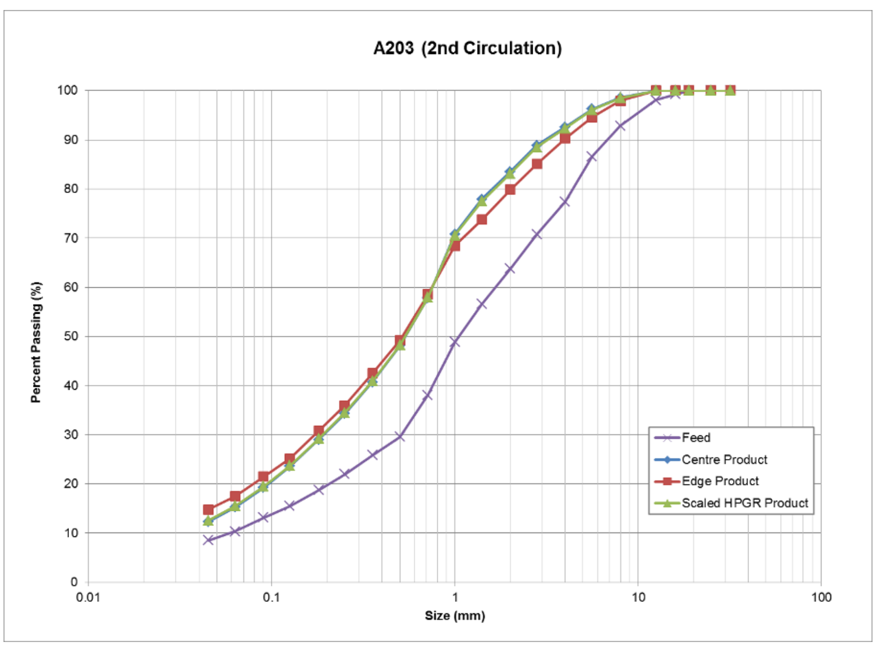
Sample No.		**Feed**		**Centre Product**		**Edge Product**		**Experimental Full PSD**	**Scaled HPGR Product**
A202									**90% Center + 10% Edge**
Screen Size	Particle Size	Retained	Cum. Passing	Retained	Cum. Passing	Retained	Cum. Passing	Cum. Passing	Cum. Passing
[mm]	**[mm]**	**[g]**	**[%]**	**[g]**	**[%]**	**[g]**	**[%]**	**[%]**	**[%]**
-35.5 to +32	32	0.0	100	0.0	100	0.0	100	100	100
-32 to +25	25	0.0	100	0.0	100	0.0	100	100	100
-25 to +19	19	0.0	100	0.0	100	0.0	100	100	100
-19 to 16	16	0.0	100	0.0	100	0.0	100	100	100
-16 to +12.5	12.5	102.7	97.7	0.0	100	0.0	100	100	100
-12.5 to +8	8	252.3	92.0	49.4	99.1	93.0	98.5	98.9	99.0
-8 to +5.6	5.6	327.8	84.7	138.2	96.6	198.0	95.3	96.1	96.5
-5.6 to +4	4	397	75.8	192.3	93.1	301.6	90.3	92.1	92.8
-4 to +2.8	2.8	327.4	68.4	245.2	88.7	356.5	84.5	87.2	88.3
-2.8 to +2	2	342.6	60.7	290.8	83.4	267.2	80.2	82.3	83.1
-2 to +1.4	1.4	340	53.1	377.9	76.5	395.0	73.7	75.6	76.3
-1.4 to +1	1	320.4	45.9	617.5	65.3	459.4	66.2	65.7	65.4
-1 to +.71	0.71	359.9	37.8	363.0	58.8	659.2	55.5	57.6	58.4
-.71 to +.5	0.5	362.8	29.7	464.8	50.3	500.2	47.3	49.3	50.0
-.5 to +.355	0.355	145.7	26.4	342.7	44.1	375.0	41.2	43.1	43.8
-.355 to +.25	0.25	163.7	22.7	325.6	38.2	379.1	35.0	37.1	37.9
-.25 to +.18	0.18	130.8	19.8	279.7	33.1	299.7	30.1	32.1	32.8
-.18 to +.125	0.125	138.4	16.7	318.4	27.4	335.4	24.7	26.4	27.1
-.125 to +.09	0.09	111.1	14.2	228.8	23.2	232.1	20.9	22.4	23.0
-.09 to +.063	0.063	119.1	11.5	253.1	18.6	240.8	16.9	18.0	18.5
-.063 to +.045	0.045	103.7	9.2	168.0	15.6	185.5	13.9	15.0	15.4
-0.045	Pan	409.2		860.0		853.1			
Total		4454.6		5515.4		6130.8			
Size Distribution Interpolations									
Linear P50	[mm]		**1.23**		**0.49**		**0.57**	**0.52**	**0.50**
Linear P80	[mm]		**4.76**		**1.70**		**1.99**	**1.80**	**1.73**



HPGR Test A203

Pressure	4.0	N/mm2	Feed	283.2	Kg
Moisture	4.1	%	Center Product	106.2	Kg
			Edge Product	54.9	Kg
			Waste	108.5	Kg
			Center Product %	65.9%	
			Edge Product %	34.1%	

Sample No.		Feed		Centre Product		Edge Product		Experimental Full PSD	Scaled HPGR Product
A203									90% Center + 10% Edge
Screen Size	Particle Size	Retained	Cum. Passing	Retained	Cum. Passing	Retained	Cum. Passing	Cum. Passing	Cum. Passing
[mm]	[mm]	[g]	[%]	[g]	[%]	[g]	[%]	[%]	[%]
-35.5 to +32	32	0.0	100	0.0	100	0.0	100	100	100
-32 to +25	25	0.0	100	0.0	100	0.0	100	100	100
-25 to +19	19	0.0	100	0.0	100	0.0	100	100	100
-19 to 16	16	37.6	99.2	0.0	100	0.0	100	100	100
-16 to +12.5	12.5	60.1	98.0	0.0	100	0.0	100	100	100
-12.5 to +8	8	259.4	92.8	48.1	98.6	130.3	98.0	98.4	98.5
-8 to +5.6	5.6	309.1	86.6	79.8	96.2	216.6	94.6	95.7	96.1
-5.6 to +4	4	455.9	77.4	124.4	92.6	274.4	90.3	91.8	92.3
-4 to +2.8	2.8	331	70.8	125.9	88.9	333.7	85.1	87.6	88.5
-2.8 to +2	2	346.9	63.8	182.9	83.5	333.3	79.8	82.2	83.1
-2 to +1.4	1.4	360.8	56.5	190.4	77.9	391.9	73.7	76.5	77.5
-1.4 to +1	1	383.2	48.8	241.0	70.8	336.5	68.4	70.0	70.6
-1 to +.71	0.71	535.6	38.0	439.8	57.8	634.4	58.5	58.1	57.9
-.71 to +.5	0.5	424	29.5	330.9	48.1	594.3	49.2	48.5	48.2
-.5 to +.355	0.355	183.3	25.8	251.7	40.7	428.6	42.5	41.3	40.9
-.355 to +.25	0.25	194.1	21.9	217.6	34.3	418.0	36.0	34.9	34.5
-.25 to +.18	0.18	158.4	18.7	180.5	29.0	331.2	30.8	29.6	29.2
-.18 to +.125	0.125	163.1	15.5	184.7	23.5	360.9	25.1	24.1	23.7
-.125 to +.09	0.09	117.4	13.1	147.2	19.2	235.9	21.5	20.0	19.4
-.09 to +.063	0.063	139	10.3	134.9	15.2	255.3	17.5	16.0	15.5
-.063 to +.045	0.045	88.6	8.5	100.0	12.3	171.9	14.8	13.1	12.5
-0.045	Pan	423.6		417.8		943.8			
Total		4971.1		3397.6		6391			
Size Distribution Interpolations									
Linear P50	[mm]		**1.06**		**0.54**		**0.52**	**0.53**	**0.54**
Linear P80	[mm]		**4.45**		**1.63**		**2.02**	**1.77**	**1.67**

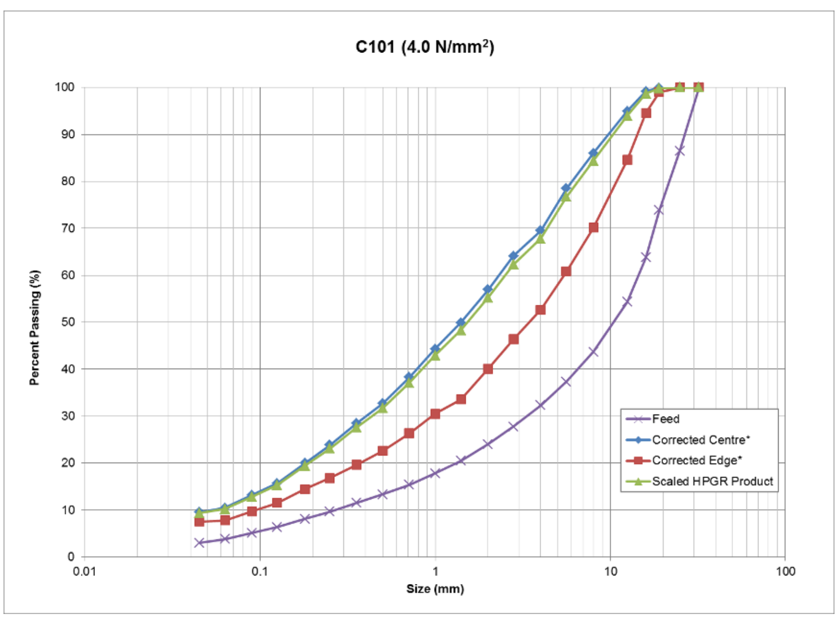


HPGR Test C101

Pressure	4.0	N/mm2	Feed	302	Kg
Moisture	0.6	%	Center Product	128.9	Kg
			Edge Product	42.4	Kg
			Waste	120.8	Kg
			Center Product %	75.2%	
			Edge Product %	24.8%	

Sample No.		Feed		Centre Product		Corrected Centre*	Edge Product		Corrected Edge*	Experimental Full PSD	Scaled HPGR Product
C101											90% Center + 10% Edge
Screen Size	Particle Size	Retained	Cum. Passing	Retained	Cum. Passing	Cum. Passing	Retained	Cum. Passing	Cum. Passing	Cum. Passing	Cum. Passing
[mm]	[mm]	[g]	[%]	[g]	[%]	[%]	[g]	[%]	[%]	[%]	[%]
-35.5 to +32	32	0.0	100	0.0	100	100	0.0	100	100	100	100
-32 to +25	25	2105.1	86.5	0.0	100	100	0.0	100	100	100	100
-25 to +19	19	1966.2	73.8	51.2	100	99.9	36.5	100	99.0	100	100
-19 to 16	16	1562.5	63.8	216.6	98.4	99.2	295.9	96.9	94.6	98.0	98.7
-16 to +12.5	12.5	1474.2	54.3	852.3	93.3	95.0	774.4	89.7	84.6	92.4	93.9
-12.5 to +8	8	1660.1	43.7	1510.3	84.2	86.0	1533.0	75.5	70.1	82.1	84.4
-8 to +5.6	5.6	982.5	37.3	1308.0	76.3	78.5	886.8	67.3	60.8	74.1	76.7
-5.6 to +4	4	787.2	32.3	1483.1	67.4	69.5	912.6	58.9	52.6	65.3	67.8
-4 to +2.8	2.8	709.1	27.7	863.5	62.2	64.1	747.2	51.9	46.4	59.7	62.3
-2.8 to +2	2	582.6	24.0	1183.5	55.1	56.9	687.4	45.6	40.1	52.8	55.3
-2 to +1.4	1.4	546.4	20.5	1151.7	48.2	49.9	747.3	38.6	33.5	45.8	48.3
-1.4 to +1	1	414.9	17.8	848.5	43.1	44.3	488.9	34.1	30.5	40.9	42.9
-1 to +.71	0.71	380.1	15.4	968.1	37.3	38.3	520.0	29.3	26.3	35.3	37.1
-.71 to +.5	0.5	320.9	13.3	929.6	31.7	32.7	411.7	25.5	22.6	30.2	31.7
-.5 to +.355	0.355	284.8	11.5	680.1	27.6	28.4	376.8	22.0	19.6	26.2	27.5
-.355 to +.25	0.25	282.1	9.7	753.8	23.1	23.8	337.1	18.9	16.8	22.1	23.1
-.25 to +.18	0.18	241.6	8.1	616.5	19.4	19.9	297.3	16.1	14.4	18.6	19.4
-.18 to +.125	0.125	277.1	6.3	718.9	15.1	15.7	300.7	13.3	11.5	14.6	15.3
-.125 to +.09	0.09	188	5.1	398.8	12.7	13.2	229.2	11.2	9.7	12.3	12.8
-.09 to +.063	0.063	207.7	3.8	438.5	10.0	10.4	235.7	9.0	7.8	9.8	10.2
-.063 to +.045	0.045	125.2	3.0	143.1	9.2	9.5	54.9	8.5	7.5	9.0	9.3
-0.045	Pan	462.5		1529.8			916.5				
Total		15560.8		16645.9			10789.9				
Size Distribution Interpolations											
Linear P50	[mm]		**10.68**		1.55	**1.41**		2.56	**3.50**	**1.76**	**1.55**
Linear P80	[mm]		**21.93**		6.72	**6.08**		9.41	**11.07**	**7.38**	**6.63**

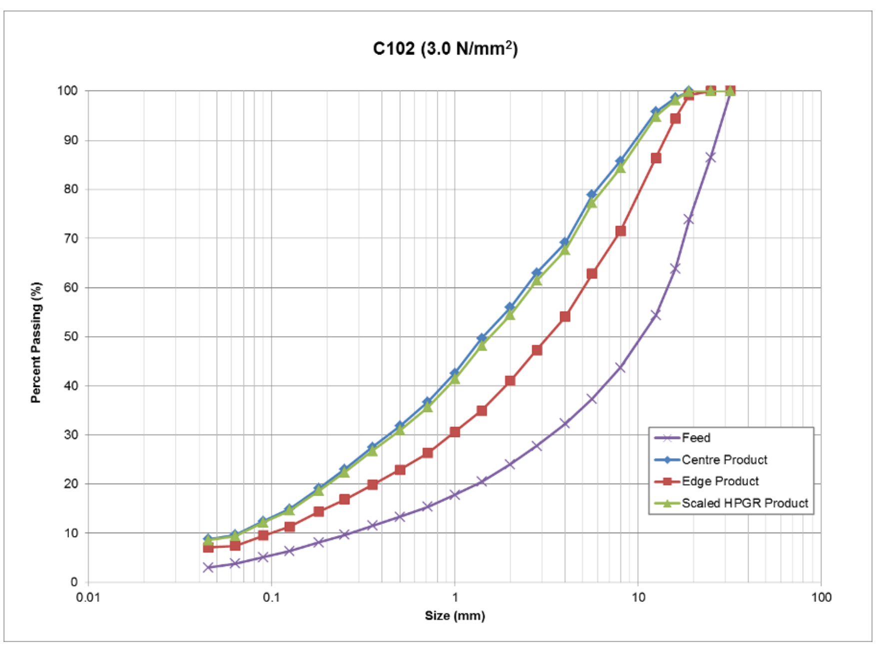
*assume that the ratio of centre over full PSD per size fraction for test C102 is the same for test C101 in order to adjust the centre and edge product PSD due to the side effect



HPGR Test C102

Pressure	3.0	N/mm2	Feed	310	Kg
Moisture	0.6	%	Center Product	131.6	Kg
			Edge Product	49.8	Kg
			Waste	119.4	Kg
			Center Product %	72.5%	
			Edge Product %	27.5%	

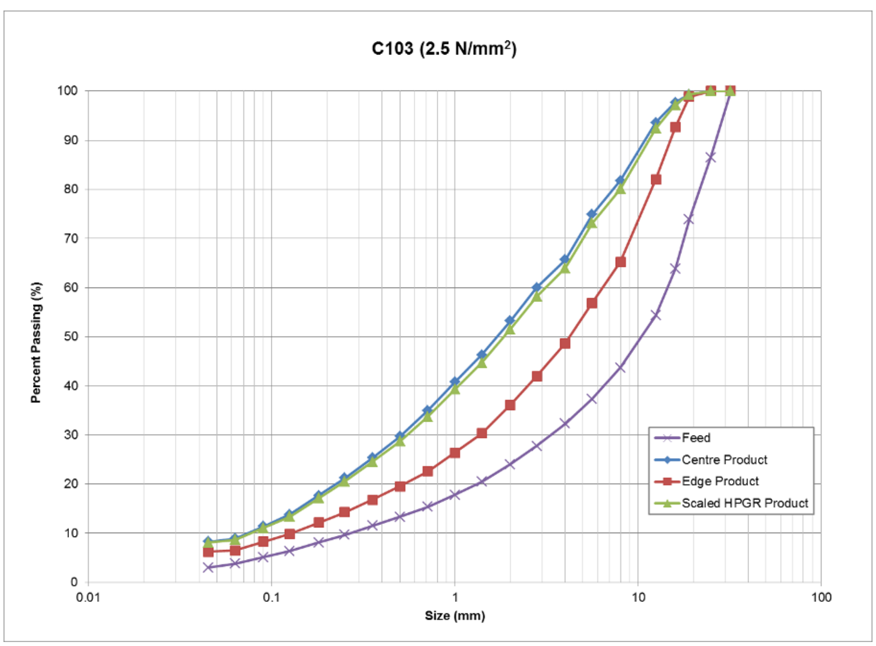
Sample No.		Feed		Centre Product		Edge Product		Experimental Full PSD	Scaled HPGR Product
C102									90% Center + 10% Edge
Screen Size	Particle Size	Retained	Cum. Passing	Retained	Cum. Passing	Retained	Cum. Passing	Cum. Passing	Cum. Passing
[mm]	[mm]	[g]	[%]	[g]	[%]	[g]	[%]	[%]	[%]
-35.5 to +32	32	0.0	100	0.0	100	0.0	100	100	100
-32 to +25	25	2105.1	86.5	0.0	100	0.0	100	100	100
-25 to +19	19	1966.2	73.8	0.0	100	107.3	99.2	100	99.9
-19 to 16	16	1562.5	63.8	225.9	98.6	624.1	94.5	97.5	98.2
-16 to +12.5	12.5	1474.2	54.3	455.4	95.8	1071.7	86.3	93.2	94.8
-12.5 to +8	8	1660.1	43.7	1629.1	85.7	1961.3	71.5	81.8	84.3
-8 to +5.6	5.6	982.5	37.3	1119.5	78.8	1152.3	62.8	74.4	77.2
-5.6 to +4	4	787.2	32.3	1566.1	69.1	1154.4	54.0	65.0	67.6
-4 to +2.8	2.8	709.1	27.7	998.5	63.0	893.4	47.3	58.7	61.4
-2.8 to +2	2	582.6	24.0	1144.7	55.9	831.7	41.0	51.8	54.4
-2 to +1.4	1.4	546.4	20.5	1016.1	49.6	797.6	34.9	45.6	48.2
-1.4 to +1	1	414.9	17.8	1143.6	42.6	575.0	30.6	39.3	41.4
-1 to +.71	0.71	380.1	15.4	953.9	36.7	563.8	26.3	33.8	35.6
-.71 to +.5	0.5	320.9	13.3	795.0	31.8	451.4	22.9	29.3	30.9
-.5 to +.355	0.355	284.8	11.5	691.1	27.5	405.8	19.8	25.4	26.7
-.355 to +.25	0.25	282.1	9.7	732.1	23.0	390.5	16.9	21.3	22.4
-.25 to +.18	0.18	241.6	8.1	631.0	19.1	336.0	14.3	17.8	18.6
-.18 to +.125	0.125	277.1	6.3	671.8	14.9	396.1	11.3	14.0	14.6
-.125 to +.09	0.09	188	5.1	402.2	12.5	243.9	9.5	11.6	12.2
-.09 to +.063	0.063	207.7	3.8	460.2	9.6	274.2	7.4	9.0	9.4
-.063 to +.045	0.045	125.2	3.0	139.4	8.8	40.2	7.1	8.3	8.6
-0.045	Pan	462.5		1418.0		938.4			
Total		15560.8		16193.6		13209.1			
Size Distribution Interpolations									
Linear P50	[mm]		**10.68**		**1.43**		**3.28**	**1.82**	**1.58**
Linear P80	[mm]		**21.93**		**6.01**		**10.58**	**7.41**	**6.54**



HPGR Test C103

Pressure	2.5	N/mm2	Feed	315.5	Kg
Moisture	0.6	%	Center Product	133.9	Kg
			Edge Product	52.2	Kg
			Waste	119.8	Kg
			Center Product %	72.0%	
			Edge Product %	28.0%	

Sample No.		**Feed**		**Centre Product**		**Edge Product**		**Experimental Full PSD**	**Scaled HPGR Product**
C103									**90% Center + 10% Edge**
Screen Size	Particle Size	Retained	Cum. Passing	Retained	Cum. Passing	Retained	Cum. Passing	Cum. Passing	Cum. Passing
[mm]	**[mm]**	**[g]**	**[%]**	**[g]**	**[%]**	**[g]**	**[%]**	**[%]**	**[%]**
-35.5 to +32	32	0.0	100	0.0	100	0.0	100	100	100
-32 to +25	25	2105.1	86.5	0.0	100	0.0	100	100	100
-25 to +19	19	1966.2	73.8	116.6	99.3	144.7	98.8	99.2	99.3
-19 to 16	16	1562.5	63.8	283.5	97.7	779.3	92.6	96.3	97.2
-16 to +12.5	12.5	1474.2	54.3	715.9	93.6	1338.6	82.0	90.3	92.4
-12.5 to +8	8	1660.1	43.7	2045.0	81.7	2109.3	65.2	77.1	80.1
-8 to +5.6	5.6	982.5	37.3	1189.9	74.9	1045.3	56.9	69.8	73.1
-5.6 to +4	4	787.2	32.3	1598.6	65.7	1034.1	48.6	60.9	63.9
-4 to +2.8	2.8	709.1	27.7	977.4	60.0	845.2	41.9	54.9	58.2
-2.8 to +2	2	582.6	24.0	1176.8	53.2	727.8	36.1	48.4	51.5
-2 to +1.4	1.4	546.4	20.5	1201.5	46.3	728.1	30.3	41.8	44.7
-1.4 to +1	1	414.9	17.8	963.0	40.7	492.7	26.4	36.7	39.3
-1 to +.71	0.71	380.1	15.4	992.4	35.0	481.7	22.6	31.5	33.7
-.71 to +.5	0.5	320.9	13.3	922.8	29.7	383.8	19.5	26.8	28.6
-.5 to +.355	0.355	284.8	11.5	748.7	25.3	339.3	16.8	22.9	24.5
-.355 to +.25	0.25	282.1	9.7	718.8	21.2	318.6	14.3	19.2	20.5
-.25 to +.18	0.18	241.6	8.1	610.7	17.7	270.2	12.1	16.1	17.1
-.18 to +.125	0.125	277.1	6.3	671.9	13.8	291.3	9.8	12.7	13.4
-.125 to +.09	0.09	188.0	5.1	412.2	11.4	195.4	8.2	10.5	11.1
-.09 to +.063	0.063	207.7	3.8	439.5	8.9	222.0	6.5	8.2	8.6
-.063 to +.045	0.045	125.2	3.0	104.2	8.3	32.7	6.2	7.7	8.1
-0.045	Pan	462.5		1431.1		778.8			
Total		15560.8		17320.5		12558.9			
Size Distribution Interpolations									
Linear P50	[mm]		**10.68**		**1.72**		**4.27**	**2.19**	**1.87**
Linear P80	[mm]		**21.93**		**7.39**		**11.97**	**8.99**	**7.97**

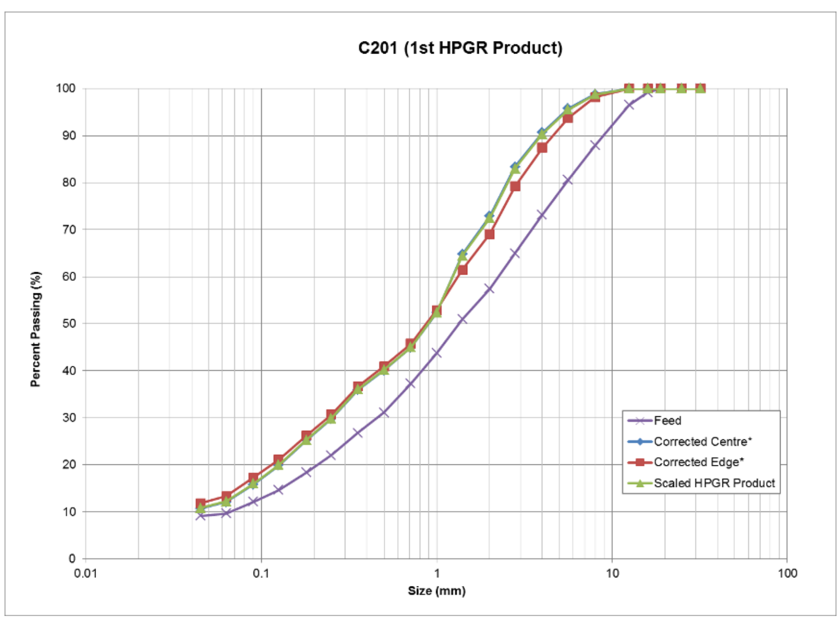


HPGR Test C201

Pressure	4.0	N/mm2	Feed	246.6	Kg
Moisture	5.7	%	Center Product	51	Kg
			Edge Product	40.9	Kg
			Waste	147	Kg
			Center Product %	55.5%	
			Edge Product %	44.5%	

Sample No.		**Feed**		**Centre Product**		**Corrected Centre***	**Edge Product**		**Corrected Edge***	**Experimental Full PSD**	**Scaled HPGR Product**
C201											**90% Center + 10% Edge**
Screen Size	Particle Size	Retained	Cum. Passing	Retained	Cum. Passing	Cum. Passing	Retained	Cum. Passing	Cum. Passing	Cum. Passing	Cum. Passing
[mm]	**[mm]**	**[g]**	**[%]**	**[g]**	**[%]**	**[%]**	**[g]**	**[%]**	**[%]**	**[%]**	**[%]**
-35.5 to +32	32	0.0	100	0.0	100	100	0.0	100	100	100	100
-32 to +25	25	0.0	100	0.0	100	100	0.0	100	100	100	100
-25 to +19	19	0.0	100	0.0	100	100	0.0	100	100	100	100
-19 to 16	16	88.1	99.2	0.0	100	100	0.0	100	100	100	100
-16 to +12.5	12.5	294.2	96.5	0.0	100	100	0.0	100	100	100	100
-12.5 to +8	8	933.6	87.9	74.4	98.7	98.8	167.8	98.2	98.2	98.5	98.7
-8 to +5.6	5.6	806.2	80.6	184.2	95.6	95.7	411.5	93.9	93.7	94.9	95.5
-5.6 to +4	4	803.1	73.2	298.6	90.6	90.6	605.9	87.5	87.4	89.2	90.3
-4 to +2.8	2.8	899.1	65.0	493.9	82.3	83.4	648.7	80.6	79.2	81.5	83.0
-2.8 to +2	2	819.6	57.4	665.0	71.0	72.9	868.8	71.4	69.1	71.2	72.5
-2 to +1.4	1.4	711.2	50.9	434.6	63.7	64.8	810.7	62.8	61.5	63.3	64.4
-1.4 to +1	1	783.4	43.7	654.8	52.6	52.4	977.3	52.5	52.9	52.6	52.4
-1 to +.71	0.71	706.4	37.3	412.7	45.7	44.9	729.0	44.8	45.7	45.3	45.0
-.71 to +.5	0.5	670.6	31.1	313.0	40.4	40.1	401.5	40.5	40.9	40.5	40.2
-.5 to +.355	0.355	478.3	26.7	233.9	36.4	35.9	429.8	36.0	36.7	36.2	36.0
-.355 to +.25	0.25	518.7	22.0	355.0	30.4	29.8	583.0	29.8	30.7	30.2	29.9
-.25 to +.18	0.18	401.3	18.3	264.2	26.0	25.2	439.9	25.2	26.2	25.6	25.3
-.18 to +.125	0.125	401.4	14.6	321.1	20.6	19.8	477.1	20.1	21.1	20.4	19.9
-.125 to +.09	0.09	277.4	12.1	254.5	16.3	15.8	320.8	16.7	17.3	16.5	16.0
-.09 to +.063	0.063	270.9	9.6	254.4	12.0	12.1	309.8	13.5	13.3	12.6	12.2
-.063 to +.045	0.045	49.3	9.2	117.8	10.0	10.7	72.1	12.7	11.8	11.2	10.8
-0.045	Pan	999.5		591.4			1200.3				
Total		10912.3		5923.5			9454				
Size Distribution Interpolations											
Linear P50	[mm]		**1.35**		**0.89**	**0.91**		**0.91**	**0.88**	**0.90**	**0.91**
Linear P80	[mm]		**5.48**		**2.64**	**2.54**		**2.75**	**2.92**	**2.68**	**2.57**

*assume that the ratio of center over full PSD per size fraction for test C203 is the same for test C201 in order to adjust the center and edge product PSD due to the side effect

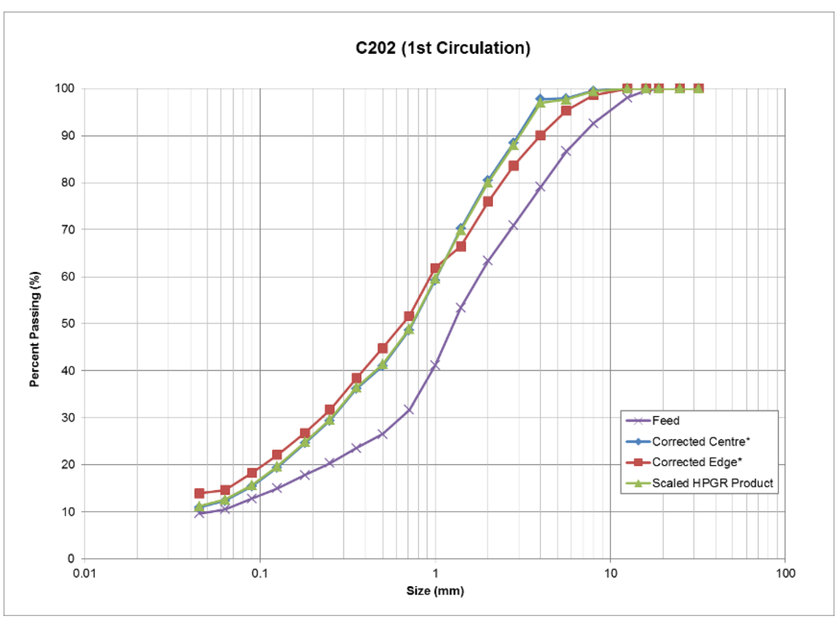


HPGR Test C202

Pressure	4.0	N/mm2	Feed	240.5	Kg
Moisture	4.4	%	Center Product	67.8	Kg
			Edge Product	57.3	Kg
			Waste	109.7	Kg
			Center Product %	54.2%	
			Edge Product %	45.8%	

Sample No.		**Feed**		**Centre Product**		**Corrected Centre***	**Edge Product**		**Corrected Edge***	**Experimental Full PSD**	**Scaled HPGR Product**
C202											**90% Center + 10% Edge**
Screen Size	Particle Size	Retained	Cum. Passing	Retained	Cum. Passing	Cum. Passing	Retained	Cum. Passing	Cum. Passing	Cum. Passing	Cum. Passing
[mm]	**[mm]**	**[g]**	**[%]**	**[g]**	**[%]**	**[%]**	**[g]**	**[%]**	**[%]**	**[%]**	**[%]**
-35.5 to +32	32	0.0	100	0.0	100	100	0.0	100	100	100	100
-32 to +25	25	0.0	100	0.0	100	100	0.0	100	100	100	100
-25 to +19	19	0.0	100	0.0	100	100	0.0	100	100	100	100
-19 to 16	16	31.4	100	0.0	100	100	0.0	100	100	100	100
-16 to +12.5	12.5	122.8	98.1	0.0	100	100	0.0	100	100	100	100
-12.5 to +8	8	437.1	92.6	62.1	99.3	100	74.3	98.9	98.6	99.1	99.4
-8 to +5.6	5.6	474	86.7	185.4	97.1	98.0	181.2	96.4	95.3	96.7	97.7
-5.6 to +4	4	606.5	79.1	72.7	96.2	97.8	309.9	92.0	90.1	94.3	97.0
-4 to +2.8	2.8	653.4	70.9	823.4	86.4	88.4	425.5	85.9	83.6	86.2	87.9
-2.8 to +2	2	598.6	63.4	656.4	78.6	80.5	542.4	78.2	76.0	78.4	80.0
-2 to +1.4	1.4	797.9	53.4	838.3	68.7	70.3	695.0	68.3	66.4	68.5	69.9
-1.4 to +1	1	980.4	41.1	771.5	59.5	59.3	476.4	61.6	61.9	60.4	59.5
-1 to +.71	0.71	753.5	31.7	882.0	49.0	48.6	732.4	51.1	51.6	50.0	48.9
-.71 to +.5	0.5	415	26.5	643.5	41.4	41.0	479.9	44.3	44.8	42.7	41.4
-.5 to +.355	0.355	237.3	23.5	414.8	36.5	36.1	445.9	38.0	38.4	37.2	36.4
-.355 to +.25	0.25	256	20.3	562.6	29.8	29.4	476.8	31.2	31.7	30.4	29.6
-.25 to +.18	0.18	203.4	17.8	401.8	25.0	24.6	350.2	26.2	26.7	25.6	24.8
-.18 to +.125	0.125	223.7	15.0	428.2	19.9	19.4	336.8	21.4	22.1	20.6	19.7
-.125 to +.09	0.09	170.7	12.8	327.6	16.0	15.4	276.9	17.5	18.3	16.7	15.7
-.09 to +.063	0.063	187.3	10.5	265.8	12.9	12.3	251.5	13.9	14.6	13.4	12.5
-.063 to +.045	0.045	66.1	9.7	123.2	11.4	10.9	44.9	13.3	13.9	12.3	11.2
-0.045	Pan	771.4		961.8			935.0				
Total		7986.5		8421.1			7035				
Size Distribution Interpolations											
Linear P50	[mm]		**1.29**		**0.74**	**0.75**		**0.67**	**0.66**	**0.71**	**0.74**
Linear P80	[mm]		**4.20**		**2.14**	**1.97**		**2.19**	**2.42**	**2.16**	**2.00**

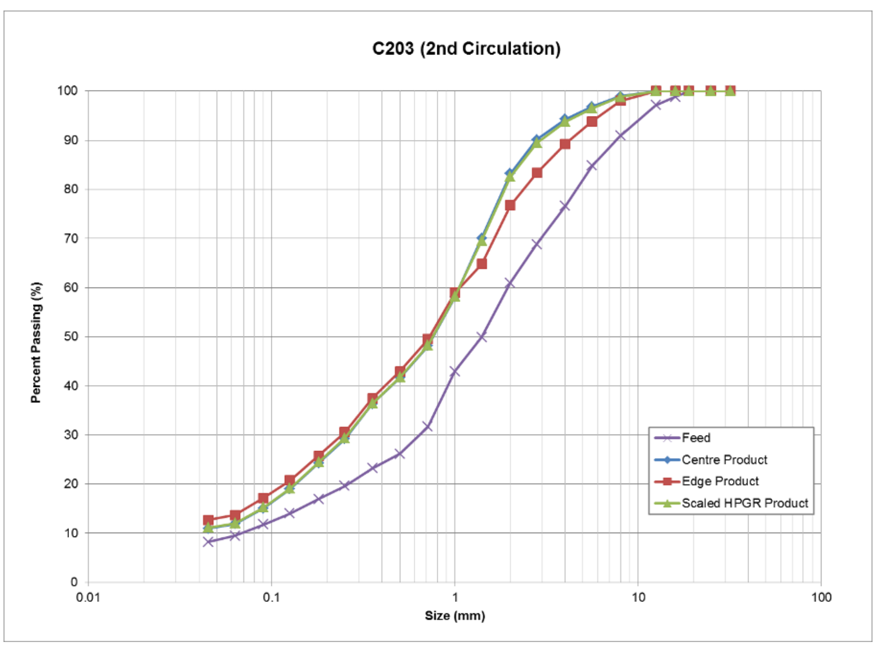
*assume that the ratio of center over full PSD per size fraction for test C203 is the same for test C202 in order to adjust the center and edge product PSD due to the side effect



HPGR Test C203

Pressure	4.0	N/mm2	Feed	242.5	Kg
Moisture	4.2	%	Center Product	100.7	Kg
			Edge Product	42.6	Kg
			Waste	91	Kg
			Center Product %	70.3%	
			Edge Product %	29.7%	

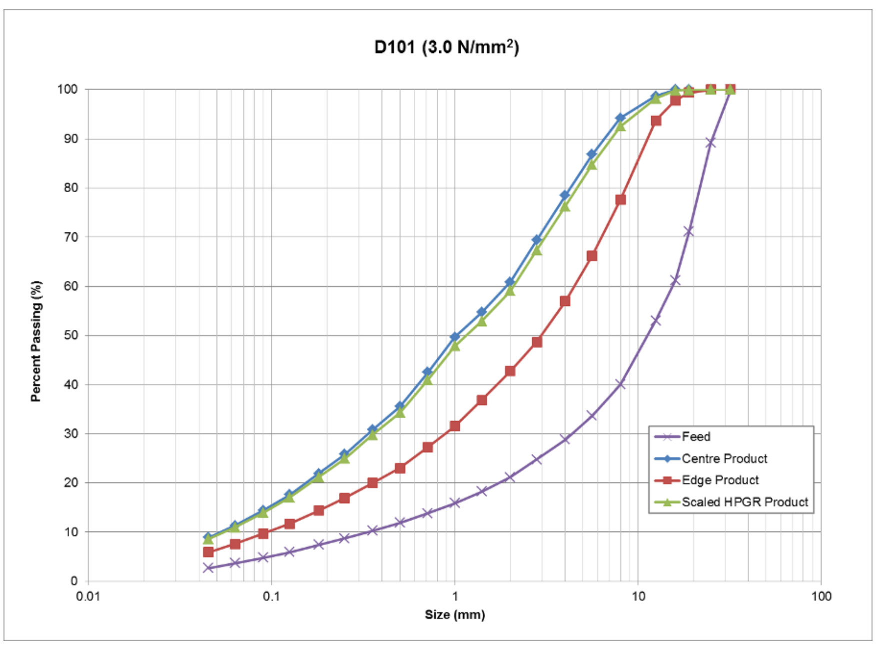
Sample No.		Feed		Centre Product		Edge Product		Experimental Full PSD	Scaled HPGR Product
C203									90% Center + 10% Edge
Screen Size	Particle Size	Retained	Cum. Passing	Retained	Cum. Passing	Retained	Cum. Passing	Cum. Passing	Cum. Passing
[mm]	[mm]	[g]	[%]	[g]	[%]	[g]	[%]	[%]	[%]
-35.5 to +32	32	0.0	100	0.0	100	0.0	100	100	100
-32 to +25	25	0.0	100	0.0	100	0.0	100	100	100
-25 to +19	19	0.0	100	0.0	100	0.0	100	100	100
-19 to 16	16	80.2	98.8	0.0	100	0.0	100	100	100
-16 to +12.5	12.5	114.8	97.1	0.0	100	0.0	100	100	100
-12.5 to +8	8	422.9	90.9	65.4	98.9	102.0	98.0	98.6	98.8
-8 to +5.6	5.6	409.5	84.8	125.0	96.8	210.0	93.8	95.9	96.5
-5.6 to +4	4	560	76.6	155.5	94.3	234.3	89.2	92.8	93.8
-4 to +2.8	2.8	525.8	68.8	249.3	90.1	298.4	83.3	88.1	89.4
-2.8 to +2	2	530.8	61.0	417.6	83.2	333.2	76.7	81.3	82.5
-2 to +1.4	1.4	751.2	49.9	790.0	70.1	605.3	64.7	68.5	69.5
-1.4 to +1	1	472.6	42.9	723.5	58.1	294.9	58.9	58.3	58.2
-1 to +.71	0.71	762.4	31.7	600.7	48.1	477.7	49.5	48.5	48.2
-.71 to +.5	0.5	375	26.1	390.6	41.6	329.8	42.9	42.0	41.7
-.5 to +.355	0.355	196.8	23.2	320.2	36.3	279.2	37.4	36.6	36.4
-.355 to +.25	0.25	247.3	19.6	427.9	29.2	346.9	30.6	29.6	29.3
-.25 to +.18	0.18	180.3	16.9	296.8	24.3	244.0	25.7	24.7	24.4
-.18 to +.125	0.125	196.3	14.0	321.6	18.9	254.0	20.7	19.5	19.1
-.125 to +.09	0.09	152.8	11.8	235.6	15.0	179.1	17.2	15.7	15.2
-.09 to +.063	0.063	157.2	9.4	192.0	11.8	177.7	13.7	12.4	12.0
-.063 to +.045	0.045	84.9	8.2	54.1	10.9	49.8	12.7	11.5	11.1
-0.045	Pan	554.2		659.2		640.8			
Total		6775		6025.0		5057.1			
Size Distribution Interpolations									
Linear P50	[mm]		**1.41**		**0.77**		**0.73**	**0.75**	**0.76**
Linear P80	[mm]		**4.66**		**1.85**		**2.40**	**1.94**	**1.88**



HPGR Test D101

Pressure	3.0	N/mm2	Feed	292.5	Kg
Moisture	2.1	%	Center Product	123.5	Kg
			Edge Product	52.8	Kg
			Waste	111.2	Kg
			Center Product %	70.1%	
			Edge Product %	29.9%	

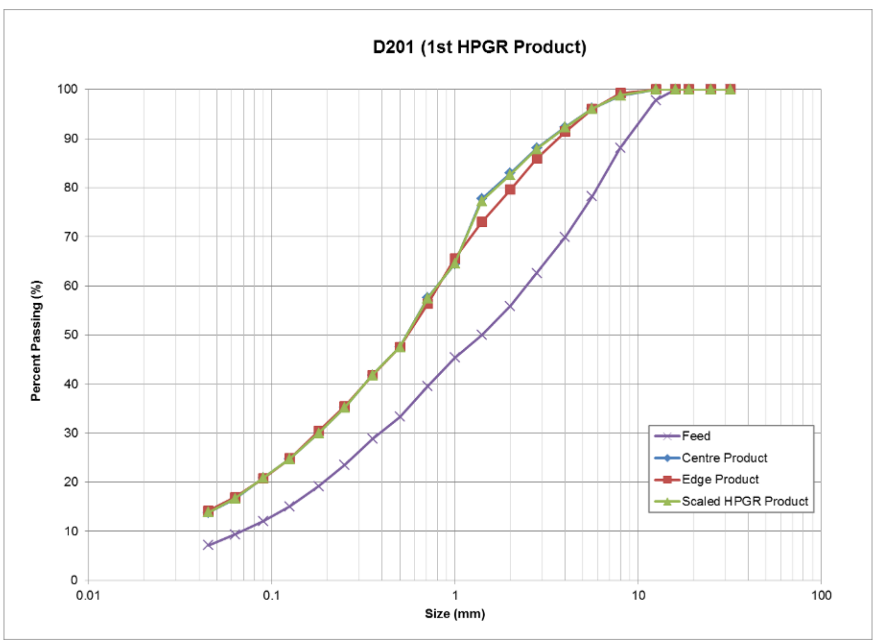
Sample No.		Feed		Centre Product		Edge Product		Experimental Full PSD	Scaled HPGR Product
D101									90% Center + 10% Edge
Screen Size	Particle Size	Retained	Cum. Passing	Retained	Cum. Passing	Retained	Cum. Passing	Cum. Passing	Cum. Passing
[mm]	[mm]	[g]	[%]	[g]	[%]	[g]	[%]	[%]	[%]
-35.5 to +32	32	0.0	100	0.0	100	0.0	100	100	100
-32 to +25	25	611.5	89.2	0.0	100	0.0	100	100	100
-25 to +19	19	1014.3	71.2	0.0	100	40.4	99.4	100	99.9
-19 to 16	16	566.8	61.1	0.0	100	99.3	97.8	99.3	99.8
-16 to +12.5	12.5	460.6	53.0	48.6	98.7	256.0	93.7	97.2	98.2
-12.5 to +8	8	731.6	40.0	170.2	94.2	1019.5	77.6	89.2	92.5
-8 to +5.6	5.6	358.5	33.7	277.4	86.8	718.4	66.2	80.6	84.7
-5.6 to +4	4	275	28.8	315.0	78.4	583.8	56.9	72.0	76.3
-4 to +2.8	2.8	229.5	24.7	339.6	69.4	521.1	48.7	63.2	67.3
-2.8 to +2	2	201.4	21.2	321.0	60.9	373.0	42.7	55.4	59.1
-2 to +1.4	1.4	164.8	18.2	232.1	54.7	373.6	36.8	49.3	52.9
-1.4 to +1	1	136.6	15.8	189.7	49.6	334.7	31.5	44.2	47.8
-1 to +.71	0.71	114.4	13.8	270.1	42.5	269.1	27.2	37.9	40.9
-.71 to +.5	0.5	107.3	11.9	260.5	35.5	266.0	23.0	31.8	34.3
-.5 to +.355	0.355	94.1	10.2	177.7	30.8	192.9	20.0	27.6	29.7
-.355 to +.25	0.25	86.9	8.7	188.2	25.8	193.6	16.9	23.1	24.9
-.25 to +.18	0.18	74.6	7.4	147.5	21.9	159.4	14.4	19.6	21.1
-.18 to +.125	0.125	82.6	5.9	160.4	17.6	171.0	11.7	15.8	17.0
-.125 to +.09	0.09	62.8	4.8	121.5	14.4	125.9	9.7	13.0	13.9
-.09 to +.063	0.063	65.4	3.6	115.1	11.3	133.1	7.6	10.2	11.0
-.063 to +.045	0.045	55	2.7	92.8	8.9	102.9	5.9	8.0	8.6
-0.045	Pan	149.8		333.4		373.6			
Total		5643.5		3760.8		6307.3			
Size Distribution Interpolations									
Linear P50	[mm]		**11.46**		**1.03**		**3.00**	**1.46**	**1.17**
Linear P80	[mm]		**21.94**		**4.30**		**8.68**	**5.48**	**4.70**



HPGR Test D201

Pressure	4.0	N/mm2	Feed	255.6	Kg
Moisture	5.3	%	Center Product	54.2	Kg
			Edge Product	29.4	Kg
			Waste	164	Kg
			Center Product %	64.8%	
			Edge Product %	35.2%	

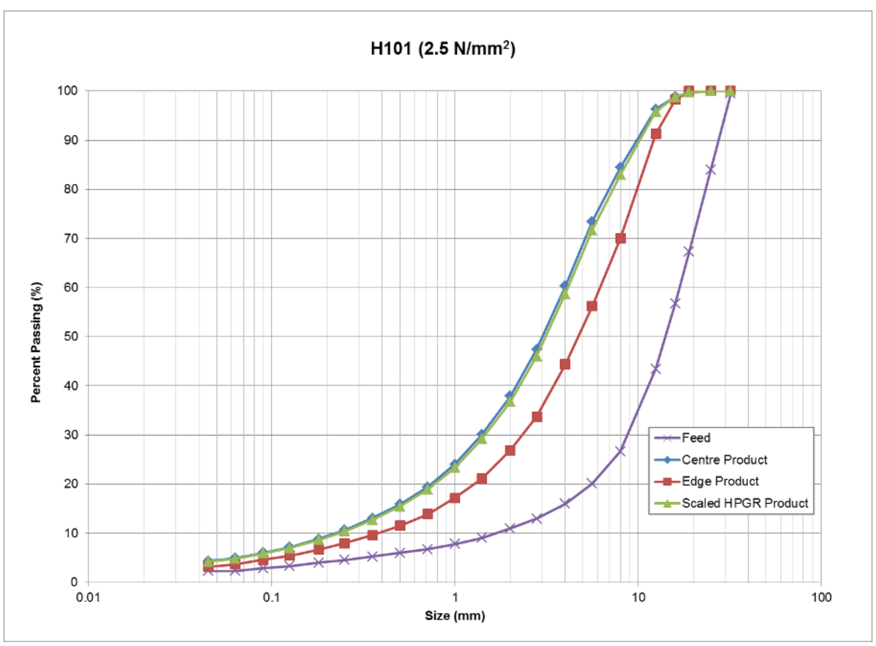
Sample No.		Feed		Centre Product		Edge Product		Experimental Full PSD	Scaled HPGR Product
D201									90% Center + 10% Edge
Screen Size	Particle Size	Retained	Cum. Passing	Retained	Cum. Passing	Retained	Cum. Passing	Cum. Passing	Cum. Passing
[mm]	[mm]	[g]	[%]	[g]	[%]	[g]	[%]	[%]	[%]
-35.5 to +32	32	0.0	100	0.0	100	0.0	100	100	100
-32 to +25	25	0.0	100	0.0	100	0.0	100	100	100
-25 to +19	19	0.0	100	0.0	100	0.0	100	100	100.0
-19 to 16	16	0.0	100	0.0	100	0.0	100	100	100
-16 to +12.5	12.5	93.9	97.8	0.0	100	0.0	100	100	100
-12.5 to +8	8	421.2	88.1	64.2	98.7	37.2	99.2	98.9	98.8
-8 to +5.6	5.6	426.4	78.2	133.6	96.1	159.6	96.0	96.1	96.1
-5.6 to +4	4	359.2	69.9	190.9	92.4	226.2	91.4	92.0	92.3
-4 to +2.8	2.8	317.4	62.6	218.4	88.1	269.2	85.9	87.3	87.8
-2.8 to +2	2	290.8	55.9	258.5	83.0	309.4	79.6	81.8	82.6
-2 to +1.4	1.4	257.3	49.9	266.5	77.7	324.0	73.0	76.1	77.3
-1.4 to +1	1	198.5	45.3	674.8	64.4	364.3	65.6	64.8	64.6
-1 to +.71	0.71	249.4	39.6	347.6	57.6	451.6	56.4	57.2	57.5
-.71 to +.5	0.5	271.3	33.3	511.9	47.5	434.1	47.5	47.5	47.5
-.5 to +.355	0.355	193.9	28.8	291.3	41.8	283.9	41.8	41.8	41.8
-.355 to +.25	0.25	229.6	23.5	336.0	35.2	309.1	35.5	35.3	35.2
-.25 to +.18	0.18	190.1	19.1	266.1	30.0	247.1	30.4	30.1	30.0
-.18 to +.125	0.125	175.7	15.1	264.2	24.8	277.3	24.8	24.8	24.8
-.125 to +.09	0.09	130.2	12.1	195.3	20.9	196.2	20.8	20.9	20.9
-.09 to +.063	0.063	119.2	9.3	219.0	16.6	189.2	16.9	16.7	16.6
-.063 to +.045	0.045	92.6	7.2	142.5	13.8	135.9	14.2	13.9	13.8
-0.045	Pan	309.8		701.6		695.8			
Total		4326.5		5082.4		4910.1			
Size Distribution Interpolations									
Linear P50	[mm]		**1.41**		**0.55**		**0.56**	**0.55**	**0.55**
Linear P80	[mm]		**6.03**		**1.66**		**2.05**	**1.81**	**1.71**



HPGR Test H101

Pressure	2.5	N/mm2	Feed	249	Kg
Moisture	3.0	%	Center Product	81	Kg
			Edge Product	47.5	Kg
			Waste	111	Kg
			Center Product %	63.0%	
			Edge Product %	37.0%	

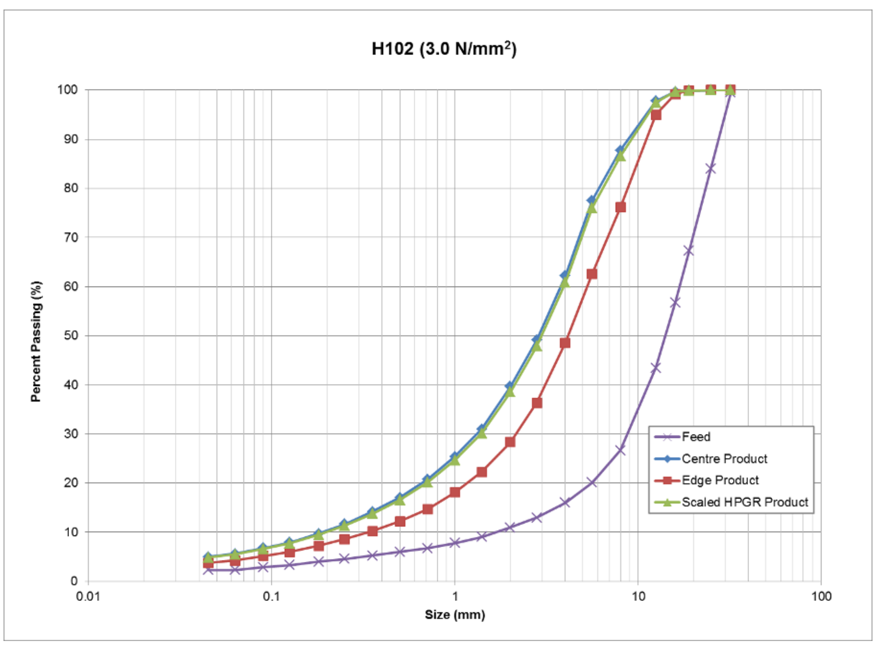
Sample No. H101		Feed		Centre Product		Edge Product		Experimental Full PSD	Scaled HPGR Product 90% Center + 10% Edge
Screen Size	Particle Size	Retained	Cum. Passing	Retained	Cum. Passing	Retained	Cum. Passing	Cum. Passing	Cum. Passing
[mm]	[mm]	[g]	[%]	[g]	[%]	[g]	[%]	[%]	[%]
-35.5 to +32	32	62.5	100	0.0	100	0.0	100	100	100
-32 to +25	25	2013.3	83.9	0.0	100	0.0	100	100	100
-25 to +19	19	2146.8	67.3	31.5	100	0.0	100	100	99.7
-19 to 16	16	1368.5	56.7	94.1	98.9	193.5	98.2	98.6	98.8
-16 to +12.5	12.5	1722	43.4	285.1	96.3	766.9	91.3	94.5	95.8
-12.5 to +8	8	2166.2	26.6	1327.1	84.4	2357.6	70.0	79.1	83.0
-8 to +5.6	5.6	844.7	20.1	1224.0	73.4	1525.1	56.2	67.1	71.7
-5.6 to +4	4	533.4	16.0	1460.7	60.3	1308.2	44.4	54.4	58.7
-4 to +2.8	2.8	398.1	12.9	1441.1	47.4	1176.8	33.7	42.3	46.0
-2.8 to +2	2	257.2	10.9	1062.2	37.9	757.4	26.9	33.8	36.8
-2 to +1.4	1.4	243	9.0	868.3	30.1	638.7	21.1	26.7	29.2
-1.4 to +1	1	160.6	7.8	673.7	24.0	433.5	17.2	21.5	23.3
-1 to +.71	0.71	136.1	6.7	511.7	19.4	361.7	13.9	17.4	18.9
-.71 to +.5	0.5	102.8	5.9	400.2	15.8	266.6	11.5	14.2	15.4
-.5 to +.355	0.355	91.1	5.2	310.9	13.0	213.4	9.5	11.8	12.7
-.355 to +.25	0.25	90	4.5	268.2	10.6	179.9	7.9	9.6	10.4
-.25 to +.18	0.18	76.8	3.9	202.2	8.8	142.6	6.6	8.0	8.6
-.18 to +.125	0.125	83.2	3.3	191.1	7.1	134.7	5.4	6.5	6.9
-.125 to +.09	0.09	58.8	2.8	119.3	6.0	90.7	4.6	5.5	5.9
-.09 to +.063	0.063	71.4	2.3	130.4	4.9	100.3	3.7	4.4	4.8
-.063 to +.045	0.045	0.8	2.3	62.8	4.3	56.0	3.2	3.9	4.2
-0.045	Pan	294.7		480.3		350.9			
Total		12922		11144.9		11054.5			
Size Distribution Interpolations									
Linear P50	[mm]		**14.23**		**3.04**		**4.76**	**3.56**	**3.18**
Linear P80	[mm]		**23.58**		**7.04**		**10.11**	**8.27**	**7.37**



HPGR Test H102

Pressure	3.0	N/mm2	Feed	246	Kg
Moisture	3.0	%	Center Product	75.5	Kg
			Edge Product	43.5	Kg
			Waste	122.5	Kg
			Center Product %	63.4%	
			Edge Product %	36.6%	

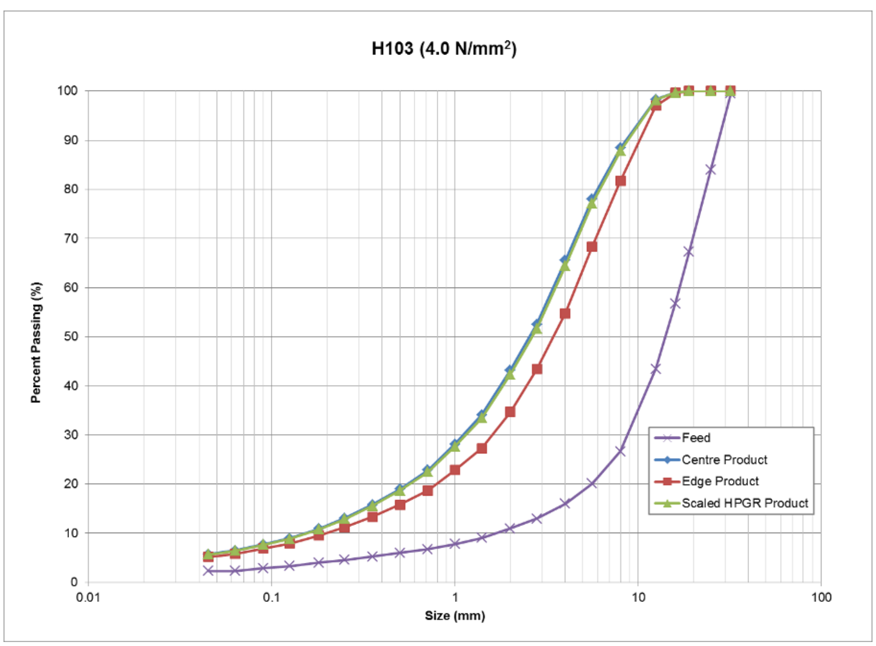
Sample No.		Feed		Centre Product		Edge Product		Experimental Full PSD	Scaled HPGR Product
H102									90% Center + 10% Edge
Screen Size	Particle Size	Retained	Cum. Passing	Retained	Cum. Passing	Retained	Cum. Passing	Cum. Passing	Cum. Passing
[mm]	[mm]	[g]	[%]	[g]	[%]	[g]	[%]	[%]	[%]
-35.5 to +32	32	62.5	100	0.0	100	0.0	100	100	100
-32 to +25	25	2013.3	83.9	0.0	100	0.0	100	100	100
-25 to +19	19	2146.8	67.3	10.8	100	8.6	100	100	99.9
-19 to 16	16	1368.5	56.7	18.6	100	75.8	99.2	99.5	99.7
-16 to +12.5	12.5	1722	43.4	194.6	97.7	448.2	94.9	96.7	97.5
-12.5 to +8	8	2166.2	26.6	993.5	87.7	1984.5	76.1	83.5	86.6
-8 to +5.6	5.6	844.7	20.1	1011.6	77.5	1421.5	62.6	72.1	76.1
-5.6 to +4	4	533.4	16.0	1526.2	62.2	1480.9	48.5	57.2	60.8
-4 to +2.8	2.8	398.1	12.9	1296.9	49.1	1281.7	36.3	44.4	47.8
-2.8 to +2	2	257.2	10.9	939.1	39.7	848.9	28.3	35.5	38.5
-2 to +1.4	1.4	243	9.0	858.4	31.0	634.4	22.2	27.8	30.1
-1.4 to +1	1	160.6	7.8	558.5	25.4	434.0	18.1	22.7	24.7
-1 to +.71	0.71	136.1	6.7	456.0	20.8	361.2	14.7	18.6	20.2
-.71 to +.5	0.5	102.8	5.9	374.7	17.0	264.1	12.2	15.2	16.5
-.5 to +.355	0.355	91.1	5.2	284.2	14.1	207.8	10.2	12.7	13.8
-.355 to +.25	0.25	90	4.5	253.8	11.6	174.1	8.6	10.5	11.3
-.25 to +.18	0.18	76.8	3.9	190.7	9.7	139.3	7.2	8.8	9.4
-.18 to +.125	0.125	83.2	3.3	176.4	7.9	134.3	6.0	7.2	7.7
-.125 to +.09	0.09	58.8	2.8	112.1	6.8	84.4	5.1	6.2	6.6
-.09 to +.063	0.063	71.4	2.3	115.6	5.6	98.6	4.2	5.1	5.5
-.063 to +.045	0.045	0.8	2.3	64.6	5.0	54.4	3.7	4.5	4.8
-0.045	Pan	294.7		491.5		389.0			
Total		12922		9927.8		10525.7			
Size Distribution Interpolations									
Linear P50	[mm]		**14.23**		**2.88**		**4.17**	**3.32**	**3.00**
Linear P80	[mm]		**23.58**		**6.18**		**8.93**	**7.27**	**6.50**



HPGR Test H103

Pressure	4.0	N/mm2	Feed	235.5	Kg
Moisture	3.0	%	Center Product	84.5	Kg
			Edge Product	38	Kg
			Waste	102	Kg
			Center Product %	69.0%	
			Edge Product %	31.0%	

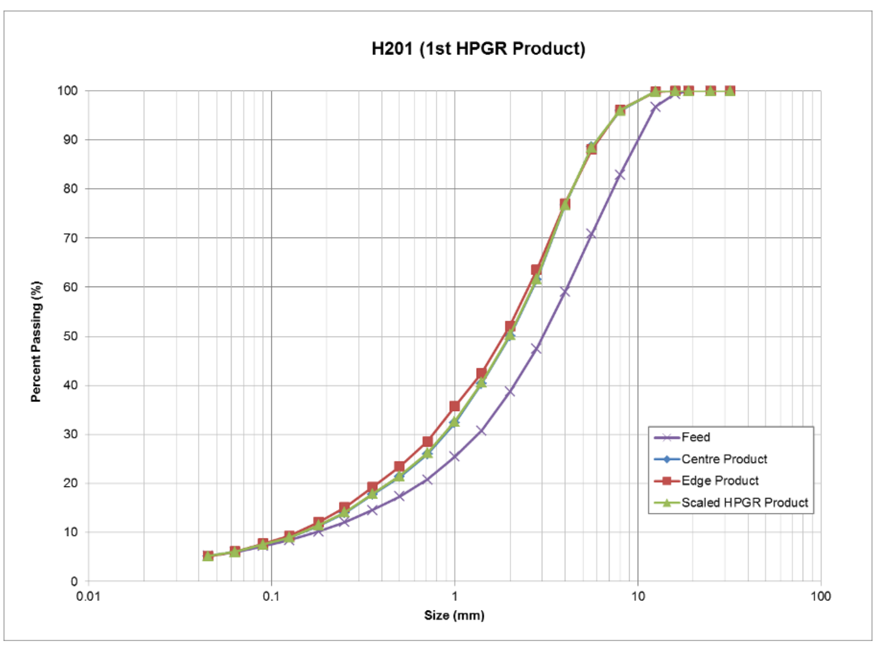
Sample No.		**Feed**		**Centre Product**		**Edge Product**		**Experimental Full PSD**	**Scaled HPGR Product**
H103									**90% Center + 10% Edge**
Screen Size	Particle Size	Retained	Cum. Passing	Retained	Cum. Passing	Retained	Cum. Passing	Cum. Passing	Cum. Passing
[mm]	**[mm]**	**[g]**	**[%]**	**[g]**	**[%]**	**[g]**	**[%]**	**[%]**	**[%]**
-35.5 to +32	32	62.5	100	0.0	100	0.0	100	100	100
-32 to +25	25	2013.3	83.9	0.0	100	0.0	100	100	100
-25 to +19	19	2146.8	67.3	0.0	100	0.0	100	100	100
-19 to 16	16	1368.5	56.7	32.8	100	27.8	100	99.7	99.7
-16 to +12.5	12.5	1722	43.4	133.6	98.3	183.7	97.0	97.9	98.2
-12.5 to +8	8	2166.2	26.6	990.0	88.4	1074.0	81.7	86.4	87.8
-8 to +5.6	5.6	844.7	20.1	1042.3	78.0	941.5	68.3	75.0	77.1
-5.6 to +4	4	533.4	16.0	1255.1	65.5	953.0	54.8	62.2	64.4
-4 to +2.8	2.8	398.1	12.9	1300.0	52.5	803.7	43.3	49.7	51.6
-2.8 to +2	2	257.2	10.9	933.3	43.2	611.8	34.6	40.5	42.3
-2 to +1.4	1.4	243	9.0	910.0	34.1	521.0	27.2	32.0	33.4
-1.4 to +1	1	160.6	7.8	599.1	28.1	307.2	22.8	26.5	27.6
-1 to +.71	0.71	136.1	6.7	527.8	22.8	292.8	18.7	21.5	22.4
-.71 to +.5	0.5	102.8	5.9	389.3	19.0	201.4	15.8	18.0	18.6
-.5 to +.355	0.355	91.1	5.2	314.8	15.8	178.6	13.3	15.0	15.6
-.355 to +.25	0.25	90	4.5	278.7	13.0	148.8	11.2	12.4	12.8
-.25 to +.18	0.18	76.8	3.9	214.3	10.9	117.0	9.5	10.5	10.7
-.18 to +.125	0.125	83.2	3.3	193.3	8.9	113.5	7.9	8.6	8.8
-.125 to +.09	0.09	58.8	2.8	122.9	7.7	69.9	6.9	7.5	7.6
-.09 to +.063	0.063	71.4	2.3	127.0	6.5	79.1	5.8	6.2	6.4
-.063 to +.045	0.045	0.8	2.3	73.2	5.7	41.5	5.2	5.6	5.7
-0.045	Pan	294.7		572.8		363.5			
Total		12922		10010.3		7029.8			
Size Distribution Interpolations									
Linear P50	[mm]		**14.23**		**2.58**		**3.50**	**2.83**	**2.66**
Linear P80	[mm]		**23.58**		**6.05**		**7.69**	**6.65**	**6.26**



HPGR Test H201

Pressure	3.0	N/mm2	Feed	309.83	Kg
Moisture	5.0	%	Center Product	73.4	Kg
			ND Edge Product	45	Kg
			Waste	179	Kg
			Center Product %	62.0%	
			Edge Product %	38.0%	

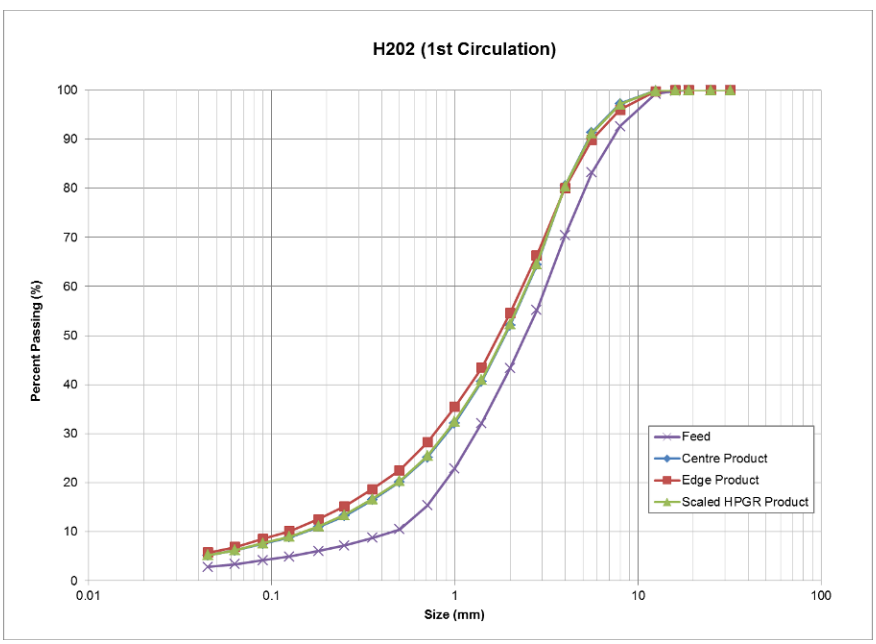
Sample No.		Feed		Centre Product		Edge Product		Experimental Full PSD	Scaled HPGR Product
H201									90% Center + 10% Edge
Screen Size	Particle Size	Retained	Cum. Passing	Retained	Cum. Passing	Retained	Cum. Passing	Cum. Passing	Cum. Passing
[mm]	[mm]	[g]	[%]	[g]	[%]	[g]	[%]	[%]	[%]
-35.5 to +32	32	0	100	0.0	100	0.0	100	100	100
-32 to +25	25	0	100	0.0	100	0.0	100	100	100
-25 to +19	19	0	100	0.0	100	0.0	100	100	100
-19 to 16	16	95.9	99.4	0.0	100	0.0	100	100	100
-16 to +12.5	12.5	397.2	96.8	12.3	100	16.5	100	99.9	99.9
-12.5 to +8	8	2127.7	82.9	519.4	96.0	378.9	96.1	96.0	96.0
-8 to +5.6	5.6	1857.6	70.8	978.4	88.6	826.0	88.1	88.4	88.6
-5.6 to +4	4	1817.1	59.0	1570.7	76.8	1130.9	77.0	76.9	76.8
-4 to +2.8	2.8	1776.1	47.5	2020.3	61.5	1386.3	63.5	62.3	61.7
-2.8 to +2	2	1347	38.7	1518.3	50.1	1171.2	52.1	50.8	50.3
-2 to +1.4	1.4	1218.9	30.8	1287.4	40.3	982.4	42.5	41.2	40.6
-1.4 to +1	1	812.1	25.5	1065.7	32.3	699.6	35.7	33.6	32.6
-1 to +.71	0.71	717.9	20.8	842.3	25.9	731.8	28.5	26.9	26.2
-.71 to +.5	0.5	529.8	17.3	611.4	21.3	519.5	23.5	22.1	21.5
-.5 to +.355	0.355	429.9	14.6	481.2	17.7	434.4	19.2	18.3	17.9
-.355 to +.25	0.25	375.7	12.1	491.4	14.0	420.9	15.1	14.4	14.1
-.25 to +.18	0.18	294.5	10.2	346.2	11.4	310.0	12.1	11.7	11.5
-.18 to +.125	0.125	272.8	8.4	320.7	9.0	286.9	9.3	9.1	9.0
-.125 to +.09	0.09	180	7.2	186.7	7.6	162.4	7.7	7.6	7.6
-.09 to +.063	0.063	195	6.0	201.8	6.0	160.3	6.1	6.1	6.0
-.063 to +.045	0.045	112.9	5.2	116.9	5.1	105.9	5.1	5.1	5.1
-0.045	Pan	804.4		682.5		523.1			
Total		15362.5		13253.6		10247			
Size Distribution Interpolations									
Linear P50	[mm]		**3.06**		**2.00**		**1.87**	**1.95**	**1.98**
Linear P80	[mm]		**7.42**		**4.44**		**4.43**	**4.43**	**4.44**



HPGR Test H202

Pressure	3.0	N/mm2	Feed	256.5	Kg
Moisture	5.0	%	Center Product	82.9	Kg
			ND Edge Product	78	Kg
			Waste	88	Kg
			Center Product %	51.5%	
			Edge Product %	48.5%	

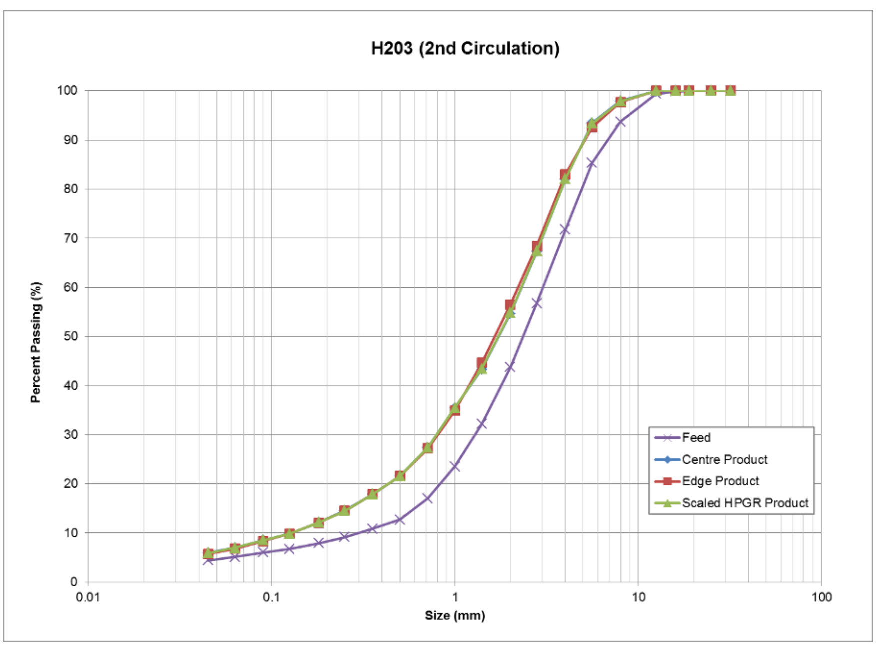
Sample No.		Feed		Centre Product		Edge Product		Experimental Full PSD	Scaled HPGR Product
H202									90% Center + 10% Edge
Screen Size	Particle Size	Retained	Cum. Passing	Retained	Cum. Passing	Retained	Cum. Passing	Cum. Passing	Cum. Passing
[mm]	[mm]	[g]	[%]	[g]	[%]	[g]	[%]	[%]	[%]
-35.5 to +32	32	0.0	100	0.0	100	0.0	100	100	100
-32 to +25	25	0.0	100	0.0	100	0.0	100	100	100
-25 to +19	19	0.0	100	0.0	100	0.0	100	100	100
-19 to 16	16	28.9	100	0.0	100	0.0	100	100	100
-16 to +12.5	12.5	81.5	99.3	13.9	100	30.5	100	99.8	99.9
-12.5 to +8	8	1046.3	92.6	288.3	97.3	526.3	96.0	96.7	97.1
-8 to +5.6	5.6	1478.2	83.2	650.4	91.4	859.2	89.9	90.7	91.2
-5.6 to +4	4	2020	70.4	1203.8	80.5	1377.4	80.1	80.3	80.4
-4 to +2.8	2.8	2386	55.2	1778.9	64.4	1922.3	66.3	65.3	64.6
-2.8 to +2	2	1870.8	43.3	1366.1	52.0	1657.3	54.5	53.2	52.3
-2 to +1.4	1.4	1751.1	32.1	1257.5	40.6	1558.6	43.4	42.0	40.9
-1.4 to +1	1	1453	22.9	938.2	32.1	1114.5	35.4	33.7	32.5
-1 to +.71	0.71	1180.9	15.4	771.9	25.2	1011.9	28.2	26.6	25.5
-.71 to +.5	0.5	766.4	10.5	556.3	20.1	797.2	22.5	21.3	20.4
-.5 to +.355	0.355	279.2	8.7	406.7	16.4	545.8	18.6	17.5	16.7
-.355 to +.25	0.25	240.4	7.2	352.6	13.3	488.9	15.1	14.2	13.4
-.25 to +.18	0.18	178.2	6.0	252.1	11.0	366.1	12.5	11.7	11.1
-.18 to +.125	0.125	170.9	5.0	231.1	8.9	338.6	10.1	9.5	9.0
-.125 to +.09	0.09	116.7	4.2	145.3	7.6	223.0	8.5	8.0	7.7
-.09 to +.063	0.063	130.7	3.4	155.5	6.2	230.3	6.9	6.5	6.2
-.063 to +.045	0.045	85.37	2.8	107.9	5.2	163.4	5.7	5.4	5.2
-0.045	Pan	444.9		572.2		801.0			
Total		15709.5		11048.7		14012.3			
Size Distribution Interpolations									
Linear P50	[mm]		**2.45**		**1.89**		**1.76**	**1.83**	**1.88**
Linear P80	[mm]		**5.20**		**3.96**		**3.99**	**3.98**	**3.97**



HPGR Test H203

Pressure	3.0	N/mm2	Feed	201.5	Kg
Moisture	5.0	%	Center Product	92	Kg
			Edge Product	55.4	Kg
			Waste	50.5	Kg
			Center Product %	62.4%	
			Edge Product %	37.6%	

Sample No.		Feed		Centre Product		Edge Product		Experimental Full PSD	Scaled HPGR Product
H203									90% Center + 10% Edge
Screen Size	Particle Size	Retained	Cum. Passing	Retained	Cum. Passing	Retained	Cum. Passing	Cum. Passing	Cum. Passing
[mm]	[mm]	[g]	[%]	[g]	[%]	[g]	[%]	[%]	[%]
-35.5 to +32	32	0	100	0.0	100	0.0	100	100	100
-32 to +25	25	0	100	0.0	100	0.0	100	100	100
-25 to +19	19	0	100	0.0	100	0.0	100	100	100
-19 to 16	16	17	100	0.0	100	0.0	100	100	100
-16 to +12.5	12.5	53.4	99.4	5.1	100	0.0	100	100	100
-12.5 to +8	8	628.4	93.7	258.6	97.9	303.4	97.7	97.8	97.9
-8 to +5.6	5.6	921.8	85.4	570.1	93.5	659.7	92.6	93.1	93.4
-5.6 to +4	4	1514.9	71.7	1462.3	82.0	1242.6	83.0	82.4	82.1
-4 to +2.8	2.8	1657.8	56.8	1880.9	67.2	1897.7	68.4	67.7	67.4
-2.8 to +2	2	1440.8	43.8	1611.6	54.6	1547.5	56.5	55.3	54.8
-2 to +1.4	1.4	1283.6	32.2	1445.1	43.3	1549.2	44.6	43.8	43.4
-1.4 to +1	1	962.9	23.5	993.8	35.5	1255.2	34.9	35.3	35.4
-1 to +.71	0.71	714.6	17.0	1032.7	27.4	1001.8	27.2	27.3	27.4
-.71 to +.5	0.5	485.6	12.7	737.9	21.6	730.5	21.6	21.6	21.6
-.5 to +.355	0.355	209.3	10.8	482.8	17.8	487.2	17.8	17.8	17.8
-.355 to +.25	0.25	181.7	9.1	420.9	14.5	438.1	14.5	14.5	14.5
-.25 to +.18	0.18	139	7.9	305.8	12.1	315.3	12.0	12.1	12.1
-.18 to +.125	0.125	130.1	6.7	291.4	9.8	291.3	9.8	9.8	9.8
-.125 to +.09	0.09	79.9	6.0	166.6	8.5	192.1	8.3	8.4	8.5
-.09 to +.063	0.063	98.4	5.1	200.6	7.0	199.4	6.8	6.9	6.9
-.063 to +.045	0.045	75.5	4.4	129.3	5.9	138.0	5.7	5.9	5.9
-0.045	Pan	490.5		757.3		743.0			
Total		11085.2		12752.8		12992.0			
Size Distribution Interpolations									
Linear P50	[mm]		**2.38**		**1.76**		**1.67**	**1.72**	**1.75**
Linear P80	[mm]		**4.97**		**3.84**		**3.75**	**3.81**	**3.83**



Appendix C - Standard bond ball mill work index data

Standard Bond Ball Mill Grindability Test

Bond Test #:	**BWIA1**	Aperture Test Sieve:	**180**	microns
Project:	MASc Research Thesis	Test Feed Density:	**2.60**	g/cc
Date:	26-Jun-12	Undersize in the Test Feed:	**28.60**	%
Performed by:	Chengtie Wang	Mill Solid Load:	**1162.6**	g
Ore Type	Copper Porphyry	Ideal Potential Product:	**332.2**	g
Sample Source:	A HPGR Product (3.0 N/mm2)	Ideal Circulating Load:	**830.4**	g

Cycle	Test Feed Added	Number of Revs.	Weight of Oversize	Weight of Undersize				
				Feed	Discharge	Net Product	Net / Rev	Circulating Load Ratio
1	1162.6	100	615.3	332.5	547.3	214.8	2.15	112
2	547.3	82	824.8	156.5	337.8	181.3	2.22	244
3	337.8	106	841.1	96.6	321.5	224.9	2.12	262
4	321.5	113	832.7	91.9	329.9	238.0	2.10	252
5	329.9	113	829.5	94.3	333.1	238.8	2.10	249
6	333.1	113	830.9	95.3	331.7	236.4	2.10	250
7	331.7	113	830.1	94.9	332.5	237.6	2.10	250

BOND'S WORK INDEX FORMULA

$$Wi = 44.5 / (Pi^{.23} \times Gpb^{.82} \times (10/\sqrt{P} - 10/\sqrt{F}))$$

Pi = Sieve size tested	180	microns
Gpb = Net Undersize produced per revolution of mill	2.10	grams
P = 80% passing size of test prodcut	141	microns
F = 80% passing size of test feed	1891	microns

WORK INDEX (Wi)

11.96	kw-hr/ton
13.15	kw-hr/tonne

NB: Gbp = Average of last 3 Net/Rev Cycles

Standard Bond Ball Mill Grindability Test

Bond Test #:	**BWIA2**	Aperture Test Sieve:	**250**	microns
Project:	MASc Research Thesis	Test Feed Density:	**2.60**	g/cc
Date:	2-Jul-12	Undersize in the Test Feed:	**34.60**	%
Performed by:	Chengtie Wang	Mill Solid Load:	**1162.6**	g
Ore Type	Copper Porphyry	Ideal Potential Product:	**332.2**	g
Sample Source:	A HPGR Product (3.0 N/mm2)	Ideal Circulating Load:	**830.4**	g

Cycle	Test Feed Added	Number of Revs.	Weight of Oversize	Weight of Undersize				
				Feed	Discharge	Net Product	Net / Rev	Circulating Load Ratio
1	1162.6	50	620.2	402.3	542.4	140.1	2.80	114
2	542.4	52	827.6	187.7	335.0	147.3	2.86	247
3	335.0	76	835.7	115.9	326.9	211.0	2.79	256
4	326.9	79	831.0	113.1	331.6	218.5	2.78	251
5	331.6	78	831.5	114.7	331.1	216.4	2.77	251
6	331.1	79	830.8	114.6	331.8	217.2	2.76	250
7	331.8	79	830.1	114.8	332.5	217.7	2.77	250

BOND'S WORK INDEX FORMULA

Wi = 44.5 / (Pi^.23 x Gpb^.82 x (10/√P - 10/√F))

Pi = Sieve size tested	250	microns
Gpb = Net Undersize produced per revolution of mill	2.76	grams
P = 80% passing size of test prodcut	191	microns
F = 80% passing size of test feed	1891	microns

WORK INDEX (Wi)

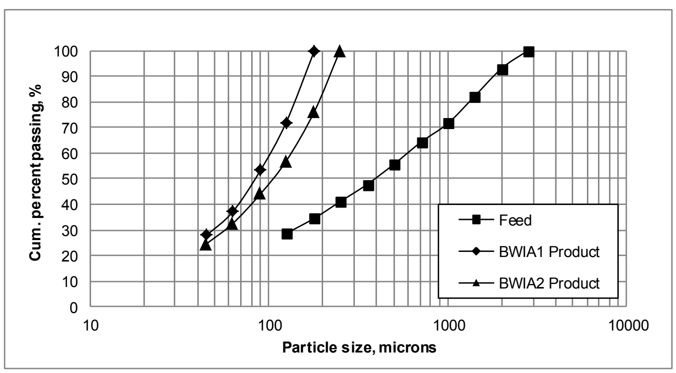
<u>11.00</u>	kw-hr/ton
<u>12.10</u>	kw-hr/tonne

NB: Gbp = Average of last 3 Net/Rev Cycles

Standard Bond Ball Mill Grindability Test Size Analysis

Project: MASc Research Thesis Sample Source: A HPGR Product (3.0 N/mm2)

Size		Feed		BWIA1 Product		BWIA2 Product	
[mesh]	[microns]	Weight [g]	Cum. Passing [%]	Weight [g]	Cum. Passing [%]	Weight [g]	Cum. Passing [%]
5	4000	0.0	100.0				
7	2800	402.5	93.0				
10	2000	645.5	81.9				
14	1400	597.2	71.5				
18	1000	417.3	64.3				
25	710	502.4	55.6				
35	500	452.6	47.8				
45	355	393.0	41.0				
60	250	371.2	34.6			0.0	100.0
80	180	347.2	28.6	0.0	100.0	156.6	76.3
120	125	1653.7	0.0	92.4	71.9	128.2	56.8
170	90			61.1	53.3	84.0	44.1
230	63			53.3	37.1	76.4	32.5
325	45			30.3	27.9	52.1	24.7
	Pan			91.9		162.7	
Total mass		**5782.6**		**329.0**		**660.0**	
Interpolations		F80	1891.0	P80	140.8	P80	191.0
		F50	558.6	P50	84.4	P50	106.2



Standard Bond Ball Mill Grindability Test

Bond Test #:	**BWIC1**	Aperture Test Sieve:	**180**	microns
Project:	MASc Research Thesis	Test Feed Density:	**2.60**	g/cc
Date:	28-Mar-12	Undersize in the Test Feed:	**19.37**	%
Performed by:	Chengtie Wang	Mill Solid Load:	**1162.0**	g
Ore Type	Copper Porphyry	Ideal Potential Product:	**332.0**	g
Sample Source:	C HPGR Product (3.0 N/mm2)	Ideal Circulating Load:	**830.0**	g

Cycle	Test Feed Added	Number of Revs.	Weight of Oversize	Weight of Undersize				
				Feed	Discharge	Net Product	Net / Rev	Circulating Load Ratio
1	1162.0	100	688.8	225.1	473.2	248.1	2.48	146
2	473.2	97	860.0	91.7	302.0	210.3	2.17	285
3	302.0	126	853.0	58.5	309.0	250.5	1.99	276
4	309.0	137	830.5	59.9	331.5	271.6	1.98	251
5	331.5	135	828.3	64.2	333.7	269.5	2.00	248
6	333.7	134	829.3	64.7	332.7	268.0	2.00	249
7	332.7	134	830.2	64.5	331.8	267.3	2.00	250

BOND'S WORK INDEX FORMULA

$$Wi = 44.5 / (Pi^{.23} \times Gpb^{.82} \times (10/\sqrt{P} - 10/\sqrt{F}))$$

Pi = Sieve size tested	180	microns
Gpb = Net Undersize produced per revolution of mill	2.00	grams
P = 80% passing size of test prodcut	143	microns
F = 80% passing size of test feed	2434	microns

WORK INDEX (Wi)

12.03	kw-hr/ton
13.23	kw-hr/tonne

NB: Gbp = Average of last 3 Net/Rev Cycles

Standard Bond Ball Mill Grindability Test

Bond Test #:	**BWIC2**	Aperture Test Sieve:	**250**	microns
Project:	MASc Research Thesis	Test Feed Density:	**2.60**	g/cc
Date:	22-Jun-12	Undersize in the Test Feed:	**24.08**	%
Performed by:	Chengtie Wang	Mill Solid Load:	**1162.0**	g
Ore Type	Copper Porphyry	Ideal Potential Product:	**332.0**	g
Sample Source:	C HPGR Product (3.0 N/mm2)	Ideal Circulating Load:	**830.0**	g

Cycle	Test Feed Added	Number of Revs.	Weight of Oversize	Weight of Undersize				
				Feed	Discharge	Net Product	Net / Rev	Circulating Load Ratio
1	1162.0	50	720.0	279.8	442.0	162.2	3.24	163
2	442.0	70	849.1	106.4	312.9	206.5	2.97	271
3	312.9	86	864.7	75.3	297.3	222.0	2.57	291
4	297.3	101	841.9	71.6	320.1	248.5	2.45	263
5	320.1	104	825.5	77.1	336.5	259.4	2.49	245
6	336.5	101	824.4	81.0	337.6	256.6	2.55	244
7	337.6	98	829.8	81.3	332.2	250.9	2.55	250

BOND'S WORK INDEX FORMULA

Wi = 44.5 / (Pi^.23 x Gpb^.82 x (10/√P - 10/√F))

Pi = Sieve size tested	250	microns
Gpb = Net Undersize produced per revolution of mill	2.53	grams
P = 80% passing size of test prodcut	197	microns
F = 80% passing size of test feed	2434	microns

WORK INDEX (Wi)

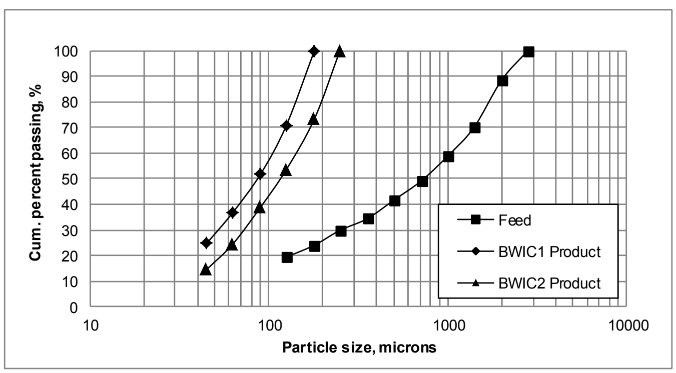
11.46	kw-hr/ton
12.61	kw-hr/tonne

NB: Gbp = Average of last 3 Net/Rev Cycles

Standard Bond Ball Mill Grindability Test Size Analysis

Project: MASc Research Thesis　　Sample Source: C HPGR Product (3.0 N/mm2)

		Feed		BWIC1 Product		BWIC2 Product	
Size		Weight	Cum. Passing	Weight	Cum. Passing	Weight	Cum. Passing
[mesh]	[microns]	[g]	[%]	[g]	[%]	[g]	[%]
5	4000	0.0	100.0				
7	2800	128.8	88.5				
10	2000	206.1	70.0				
14	1400	124.1	58.9				
18	1000	108.7	49.1				
25	710	83.4	41.7				
35	500	80.9	34.4				
45	355	50.7	29.9				
60	250	64.5	24.1			0.0	100.0
80	180	52.5	19.4	0.0	100.0	264.3	73.4
120	125	55.6	14.4	289.5	70.6	198.2	53.4
170	90	160.6		185.1	51.8	144.3	38.9
230	63			147.6	36.8	143.0	24.5
325	45			118.8	24.8	98.6	14.6
	Pan			243.9		144.9	
Total mass		**1115.9**		**984.9**		**993.3**	
Interpolations		F80	2433.7	P80	142.6	P80	197.4
		F50	1035.9	P50	86.7	P50	116.7



Standard Bond Ball Mill Grindability Test

Bond Test #:	**BWID1**	Aperture Test Sieve:	**180**	microns
Project:	MASc Research Thesis	Test Feed Density:	**2.60**	g/cc
Date:	5-Jul-12	Undersize in the Test Feed:	**30.67**	%
Performed by:	Chengtie Wang	Mill Solid Load:	**1115.9**	g
Ore Type	Copper Porphyry	Ideal Potential Product:	**318.8**	g
Sample Source:	D HPGR Product (3.0 N/mm2)	Ideal Circulating Load:	**797.1**	g

Cycle	Test Feed Added	Number of Revs.	Weight of Oversize	Weight of Undersize				
				Feed	Discharge	Net Product	Net / Rev	Circulating Load Ratio
1	1115.9	100	568.0	342.3	547.9	205.6	2.06	104
2	547.9	73	788.9	168.1	327.0	158.9	2.17	241
3	327.0	101	813.7	100.3	302.2	201.9	2.00	269
4	302.2	113	794.0	92.7	321.9	229.2	2.03	247
5	321.9	108	795.0	98.7	320.9	222.2	2.05	248
6	320.9	108	792.5	98.4	323.4	225.0	2.09	245
7	323.4	105	794.0	99.2	321.9	222.7	2.12	247

BOND'S WORK INDEX FORMULA

$$Wi = 44.5 / (Pi^{.23} \times Gpb^{.82} \times (10/\sqrt{P} - 10/\sqrt{F}))$$

Pi = Sieve size tested	180	microns
Gpb = Net Undersize produced per revolution of mill	2.09	grams
P = 80% passing size of test prodcut	143	microns
F = 80% passing size of test feed	1681	microns

WORK INDEX (Wi)

12.43	kw-hr/ton
13.68	kw-hr/tonne

NB: Gbp = Average of last 3 Net/Rev Cycles

Standard Bond Ball Mill Grindability Test

Bond Test #:	**BWID2**	Aperture Test Sieve:	**250**	microns
Project:	MASc Research Thesis	Test Feed Density:	**2.60**	g/cc
Date:	4-Jul-12	Undersize in the Test Feed:	**36.78**	%
Performed by:	Chengtie Wang	Mill Solid Load:	**1115.9**	g
Ore Type	Copper Porphyry	Ideal Potential Product:	**318.8**	g
Sample Source:	D HPGR Product (3.0 N/mm2)	Ideal Circulating Load:	**797.1**	g

Cycle	Test Feed Added	Number of Revs.	Weight of Oversize	Weight of Undersize				
				Feed	Discharge	Net Product	Net / Rev	Circulating Load Ratio
1	1115.9	50	579.9	410.4	536.0	125.6	2.51	108
2	536.0	48	771.1	197.1	344.8	147.7	3.05	224
3	344.8	63	825.3	126.8	290.6	163.8	2.60	284
4	290.6	82	795.1	106.9	320.8	213.9	2.62	248
5	320.8	77	797.1	118.0	318.8	200.8	2.62	250
6	318.8	77	797.2	117.2	318.7	201.5	2.62	250
7	318.7	77	797.0	117.2	318.9	201.7	2.62	250

BOND'S WORK INDEX FORMULA

$$Wi = 44.5 / (Pi^{.23} \times Gpb^{.82} \times (10/\sqrt{P} - 10/\sqrt{F}))$$

Pi = Sieve size tested	250	microns
Gpb = Net Undersize produced per revolution of mill	2.62	grams
P = 80% passing size of test prodcut	188	microns
F = 80% passing size of test feed	1681	microns

WORK INDEX (Wi)

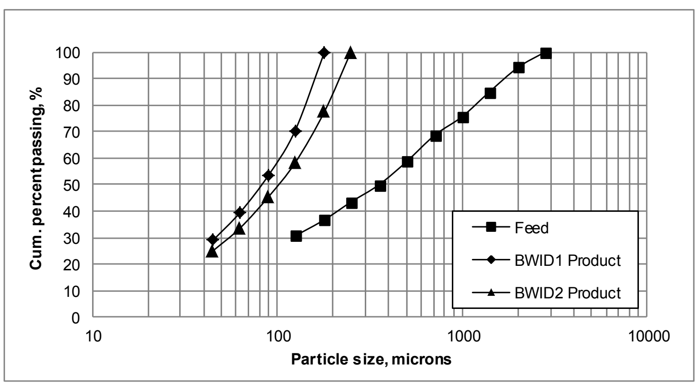
11.66	kw-hr/ton
12.82	kw-hr/tonne

NB: Gbp = Average of last 3 Net/Rev Cycles

Standard Bond Ball Mill Grindability Test Size Analysis

Project: MASc Research Thesis Sample Source: D HPGR Product (3.0 N/mm2)

Size		Feed		BWID1 Product		BWID2 Product	
[mesh]	[microns]	Weight [g]	Cum. Passing [%]	Weight [g]	Cum. Passing [%]	Weight [g]	Cum. Passing [%]
5	4000	0.0	100.0				
7	2800	376.0	94.2				
10	2000	595.9	84.9				
14	1400	596.8	75.7				
18	1000	452.3	68.6				
25	710	642.8	58.7				
35	500	561.1	50.0				
45	355	422.1	43.4				
60	250	428.3	36.8			0.0	100.0
80	180	393.4	30.7	0.0	100.0	213.0	77.5
120	125	1977.1	0.0	284.8	70.4	183.0	58.3
170	90			161.1	53.7	120.6	45.5
230	63			136.4	39.5	112.8	33.6
325	45			101.0	29.1	81.3	25.1
	Pan			279.9		237.9	
Total mass		**6445.8**		**963.2**		**948.6**	
Interpolations		F80	1681.0	P80	142.8	P80	187.7
		F50	500.7	P50	82.9	P50	102.3



Appendix D - Stirred mill experiment data

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ISAMILL

IsaMill Grinding Test Report

Project Name	M.A.Sc. Thesis	**Date(s) Tested**	26-Jun-12
Duty Description	Primary Grinding	**Date Issued**	03-Jul-12
Ore /Conc. Type	Copper Porphyry	**Location**	UBC CMP
Company	University of British Columbia	**IsaMill Type**	Netzsch M20
Contact Person	Chengtie Wang	**Media Spec.**	Cenotec 6.5 Graded Charge
Contact Details	chengtie@interchange.ubc.ca	**Test Number**	ISA A1

Test Data

Solids SG (t/m3): 2.60 | **Media Vol (L):** 13 | **Media g (Start):** 30800 | **Media g (End):** 30774

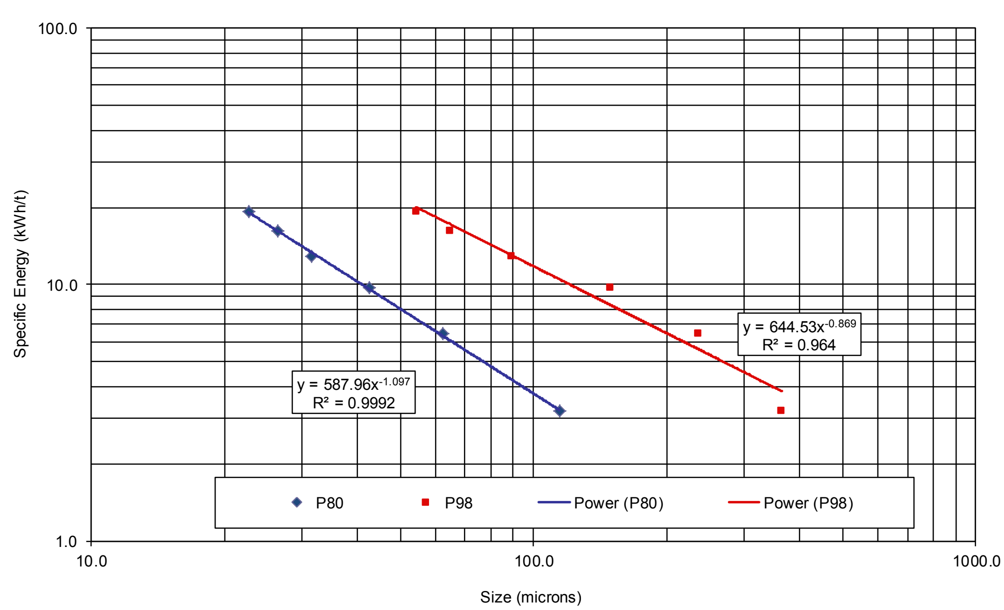
Pass #	N (rpm)	NLP (kW)	Q (sec/L)	Pump %	kg/L	Temp C	E (Wh)	Time (h)	Comments
1	880	4.93	2.56	100	1.46	26.0	679	0.081	
2	880	4.93	2.56	100	1.46	29.7	618	0.074	
3	882	4.93	2.56	100	1.46	33.1	637	0.076	
4	881	4.93	2.56	100	1.46	36.3	622	0.074	
5	882	4.93	2.56	100	1.46	39.2	593	0.071	
6	881	4.93	2.56	100	1.46	41.9	567	0.069	
7									

Calculated Data

Pass #	Gross kW	Net kW	Q (m3/h)	% Solids	M (t/h)	E (kWh/t)	Cumul. E	P80	P98	CSI
Feed								309.8	612.5	2.0
1	8.34	3.41	1.408	51.7%	1.064	3.2	3.2	114.2	364.2	3.2
2	8.37	3.44	1.408	51.8%	1.065	3.2	6.4	62.5	235.7	3.8
3	8.37	3.44	1.408	51.4%	1.057	3.2	9.7	42.6	149.2	3.5
4	8.39	3.46	1.408	51.4%	1.057	3.3	13.0	31.5	89.0	2.8
5	8.34	3.41	1.408	51.5%	1.058	3.2	16.2	26.5	64.6	2.4
6	8.26	3.33	1.408	51.6%	1.060	3.1	19.3	22.7	54.2	2.4
7										

Target P80 Size (if applic.): 100 | **kWh/t @ Target:** 3.8 | **Media Consumption (g/kWh):** 6

Signature Plot



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IsaMill Grinding Test Report

Project Name	M.A.Sc. Thesis	**Date(s) Tested**	26/06/2012
Duty Description	Primary Grinding	**Date Issued**	03/07/2012
Ore /Conc. Type	Copper Porphyry	**Location**	UBC CMP
Company	University of British Columbia	**IsaMill Type**	Netzsch M20
Contact Person	Chengtie Wang	**Media Spec.**	Cenotec 6.5 Graded Charge
Contact Details	chengtie@interchange.ubc.ca	**Contact Person**	ISA A1

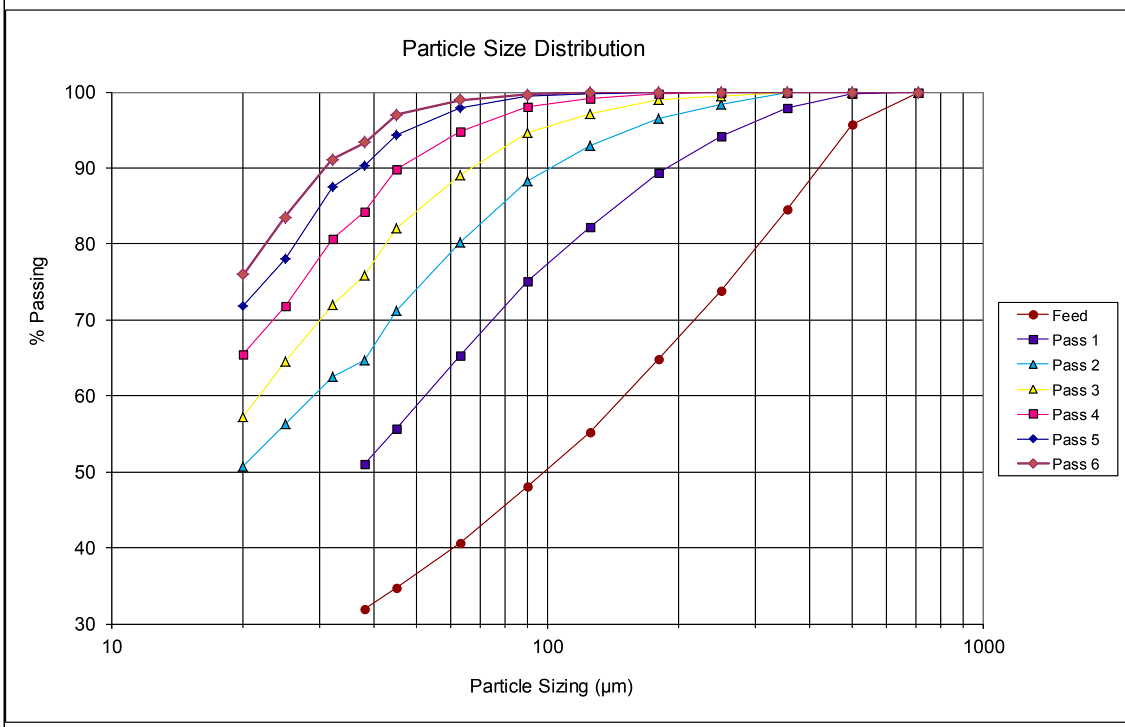
Sizing Analysis

Sizing Method	Mechanical Wet Screening	**Performed by**	Chengtie Wang

Sizing Data

Percent Passing

	Feed		**Pass 1**		**Pass 2**		**Pass 3**		**Pass 4**		**Pass 5**		**Pass 6**	
Size	Retained	Passing	Retained	Passing	Retained	Passing	Retained	Passing	Retained	Passing	Retained	Passing	Retained	Passing
[µm]	[g]	[%]	[g]	[%]	[g]	[%]	[g]	[%]	[g]	[%]	[g]	[%]	[g]	[%]
710	0.0	100	0.0	100		100		100		100		100		100
500	6.4	96	0.2	100		100		100		100		100		100
355	16.5	85	1.8	98		100		100		100		100		100
250	15.8	74	3.5	94	1.0	98	0.3	100	0.1	100		100		100
180	13.4	65	4.5	89	1.1	97	0.3	99	0.1	100		100		100
125	14.4	55	6.8	82	2.2	93	1.1	97	0.4	99	0.1	100		100
90	10.6	48	6.7	75	2.9	88	1.5	95	0.8	98	0.2	99	0.2	100
63	11.1	41	9.2	65	4.9	80	3.4	89	2.4	95	0.9	98	0.5	99
45	8.7	35	9.1	56	5.5	71	4.3	82	3.7	90	2.0	94	1.3	97
38	4.2	32	4.3	51	4.0	65	3.8	76	4.2	84	2.4	90	2.4	93
32					1.4	62	2.4	72	2.7	81	1.6	87	1.5	91
25					3.7	56	4.5	65	6.5	72	5.4	78	5.1	84
20					3.5	51	4.5	57	4.7	66	3.6	72	5.0	76
Pan	47.4		48.2		31.0		34.8		48.7		41.2		50.7	
Total	148.5		94.3		61.2		60.8		74.3		57.4		66.7	
P98		**612.5**		**364.2**		**235.7**		**149.2**		**89.0**		**64.6**		**54.2**
P80		**309.8**		**114.2**		**62.5**		**42.6**		**31.5**		**26.5**		**22.7**



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IsaMill Grinding Test Report

Project Name	M.A.Sc. Thesis	**Date(s) Tested**	16-Feb-12
Duty Description	Primary Grinding	**Date Issued**	08-Mar-12
Ore /Conc. Type	Copper Porphyry	**Location**	UBC CMP
Company	University of British Columbia	**IsaMill Type**	Netzsch M20
Contact Person	Chengtie Wang	**Media Spec.**	Cenotec 6.5 Graded Charge
Contact Details	chengtie@interchange.ubc.ca	**Test Number**	ISA C1

Test Data

Solids SG (t/m3): 2.60 | **Media Vol (L):** 13 | **Media g (Start):** 30800 | **(Media g (End):** 30773

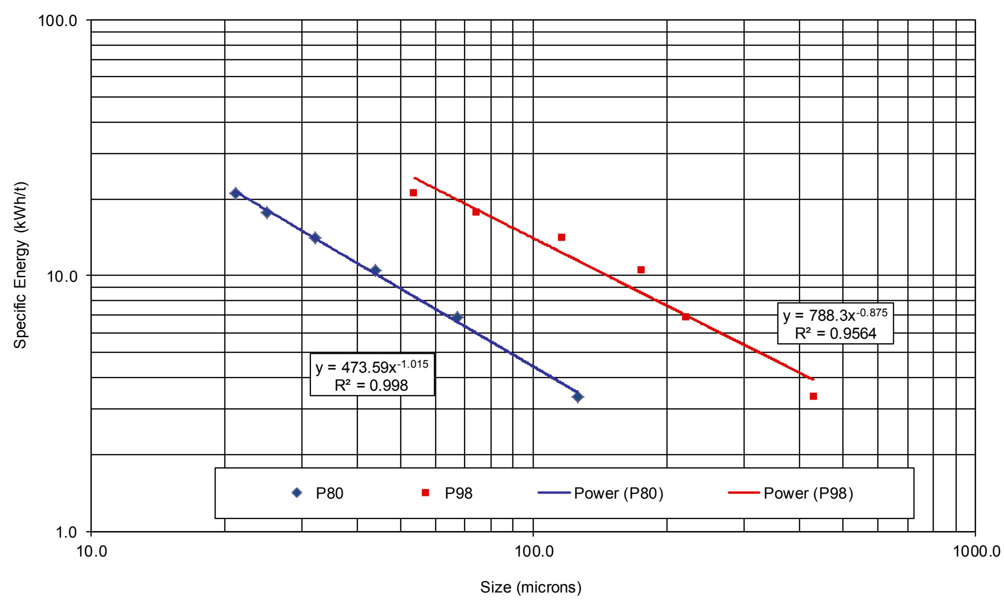
Pass #	N (rpm)	NLP (kW)	Q (sec/L)	Pump %	kg/L	Temp C	E (Wh)	Time (h)	Comments
1	874	4.96	2.56	100	1.46	18.7	686	0.080	
2	879	4.96	2.56	100	1.46	22.2	633	0.073	
3	879	4.96	2.56	100	1.46	25.7	635	0.073	
4	880	4.96	2.56	100	1.46	28.9	608	0.070	
5	879	4.96	2.56	100	1.46	31.8	616	0.071	
6	879	4.96	2.56	100	1.46	34.5	586	0.068	
7									

Calculated Data

Pass #	Gross kW	Net kW	Q (m3/h)	% Solids	M (t/h)	E (kWh/t)	Cumul. E	P80	P98	CSI
Feed								325.7	633.5	1.9
1	8.54	3.58	1.408	51.7%	1.063	3.4	3.4	125.6	429.7	3.4
2	8.69	3.73	1.408	51.6%	1.062	3.5	6.9	67.1	221.0	3.3
3	8.73	3.77	1.408	51.3%	1.054	3.6	10.5	43.9	176.0	4.0
4	8.72	3.76	1.408	51.4%	1.057	3.6	14.0	32.1	116.2	3.6
5	8.74	3.78	1.408	51.2%	1.053	3.6	17.6	25.0	74.2	3.0
6	8.61	3.65	1.408	51.4%	1.057	3.5	21.1	21.2	53.7	2.5
7										

Target P80 Size (if applic.): 100 | **kWh/t @ Target:** 4.4 | **Media Consumption (g/kWh):** 7

Signature Plot



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IsaMill Grinding Test Report

Project Name	M.A.Sc. Thesis	**Date(s) Tested**	16/02/2012
Duty Description	Primary Grinding	**Date Issued**	08/03/2012
Ore /Conc. Type	Copper Porphyry	**Location**	UBC CMP
Company	University of British Columbia	**IsaMill Type**	Netzsch M20
Contact Person	Chengtie Wang	**Media Spec.**	Cenotec 6.5 Graded Charge
Contact Details	chengtie@interchange.ubc.ca	**Contact Person**	ISA C1

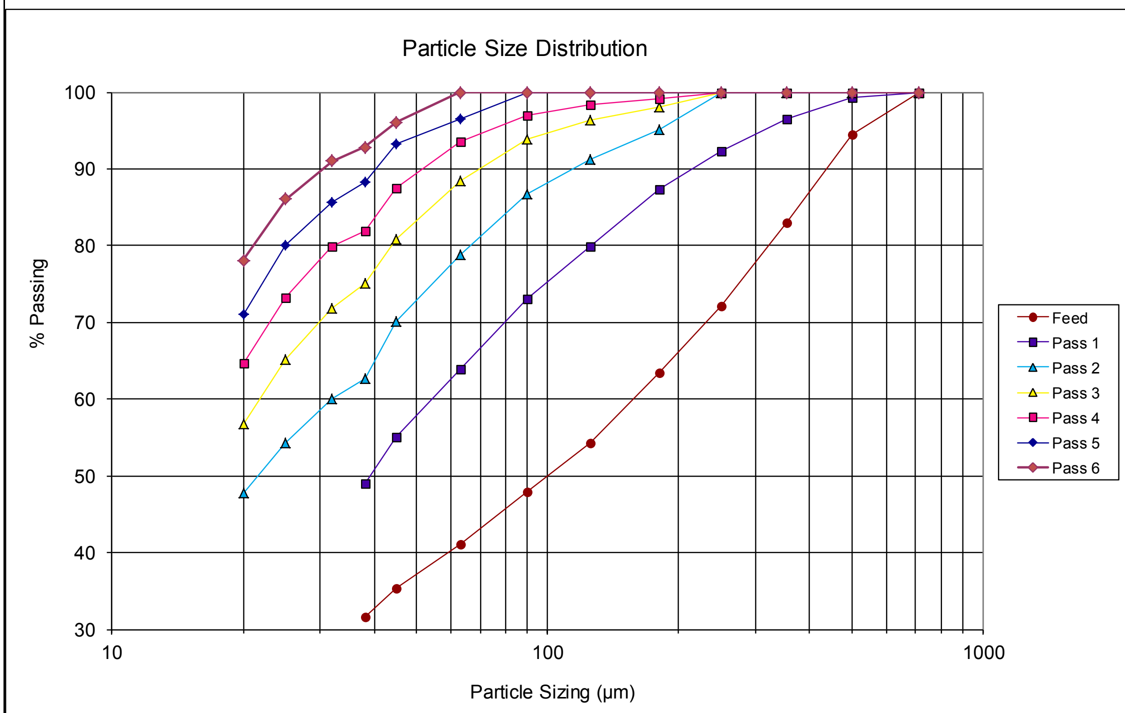
Sizing Analysis

Sizing Method	Mechanical Wet Screening	**Performed by**	Chengtie Wang

Sizing Data

Percent Passing

	Feed		**Pass 1**		**Pass 2**		**Pass 3**		**Pass 4**		**Pass 5**		**Pass 6**	
Size [µm]	Retained [g]	Passing [%]	Retained [g]	Passing [%]	Retained [g]	Passing [%]	Retained [g]	Passing [%]	Retained [g]	Passing [%]	Retained [g]	Passing [%]	Retained [g]	Passing [%]
710		100		100		100		100		100		100		100
500	7.0	95	0.8	99		100		100		100		100		100
355	14.5	83	3.1	97		100		100		100		100		100
250	13.8	72	4.9	92		100		100		100		100		100
180	11.1	63	5.7	87	3.0	95	1.1	98	0.5	99		100		100
125	11.5	54	8.4	80	2.4	91	1.1	96	0.5	98		100		100
90	8.0	48	7.8	73	2.9	87	1.4	94	0.8	97		100		100
63	8.7	41	10.5	64	4.9	79	3.1	89	2.1	94	2.1	97		100
45	7.3	35	10.1	55	5.4	70	4.4	81	3.7	88	2.1	93	2.4	96
38	4.6	32	7.0	49	4.7	63	3.4	75	3.5	82	3.1	88	2.0	93
32					1.6	60	1.9	72	1.2	80	1.6	86	1.1	91
25					3.6	54	3.8	65	4.1	73	3.5	80	3.0	86
20					4.0	48	4.9	57	5.3	65	5.7	71	4.8	78
Pan	40.1		55.9		29.8		32.8		39.4		44.5		47.5	
Total	126.5		114.1		62.3		57.8		61.0		62.7		60.8	
P98		633.5		429.7		221.0		176.0		116.2		74.2		53.7
P80		325.7		125.6		67.1		43.9		32.1		25.0		21.2



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IsaMill Grinding Test Report

Project Name	M.A.Sc. Thesis	**Date(s) Tested**	10-Jul-12
Duty Description	Primary Grinding	**Date Issued**	13-Jul-12
Ore /Conc. Type	Copper Porphyry	**Location**	UBC CMP
Company	University of British Columbia	**IsaMill Type**	Netzsch M20
Contact Person	Chengtie Wang	**Media Spec.**	Cenotec 6.5 Graded Charge
Contact Details	chengtie@interchange.ubc.ca	**Test Number**	ISA D1

Test Data

Solids SG (t/m3): 2.60 | **Media Vol (L):** 13 | **Media g (Start):** 30800 | **(Media g (End):** 30775

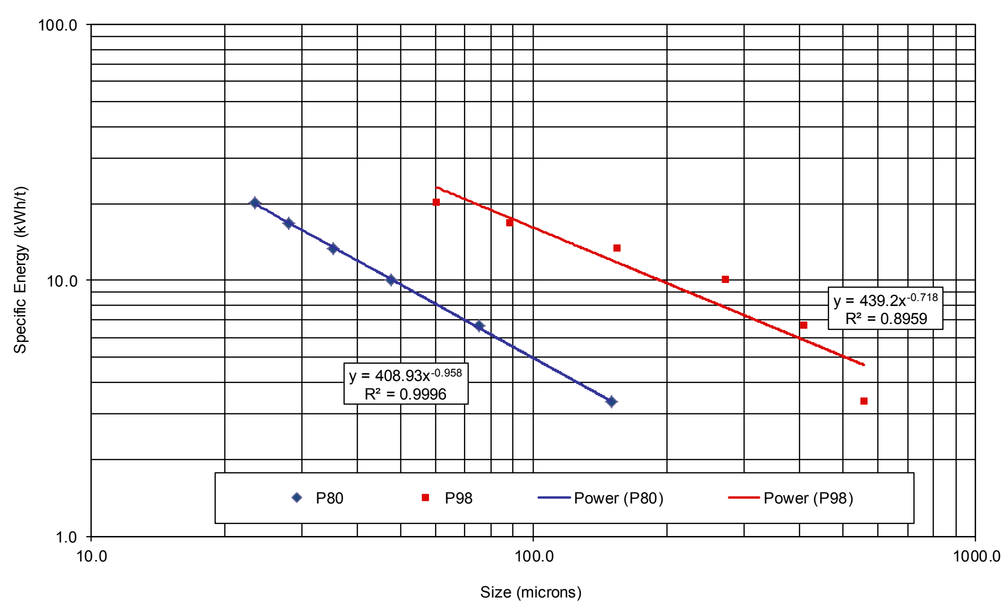
Pass #	N (rpm)	NLP (kW)	Q (sec/L)	Pump %	kg/L	Temp C	E (Wh)	Time (h)	Comments
1	869	4.87	2.56	100	1.46	26.5	689	0.082	
2	880	4.87	2.56	100	1.46	30.1	625	0.075	
3	879	4.87	2.56	100	1.46	33.4	631	0.075	
4	875	4.87	2.56	100	1.46	36.8	597	0.071	
5	876	4.87	2.56	100	1.46	39.9	610	0.073	
6	880	4.87	2.56	100	1.46	42.1	597	0.072	
7									

Calculated Data

Pass #	Gross kW	Net kW	Q (m3/h)	% Solids	M (t/h)	E (kWh/t)	Cumul. E	P80	P98	CSI
Feed								419.6	850.9	2.0
1	8.41	3.54	1.408	51.3%	1.055	3.4	3.4	149.7	558.9	3.7
2	8.37	3.50	1.408	51.1%	1.051	3.3	6.7	75.2	410.2	5.5
3	8.39	3.51	1.408	50.8%	1.044	3.4	10.0	47.6	271.8	5.7
4	8.36	3.49	1.408	50.7%	1.042	3.3	13.4	35.2	155.3	4.4
5	8.42	3.54	1.408	50.6%	1.041	3.4	16.8	27.9	88.7	3.2
6	8.33	3.45	1.408	50.9%	1.045	3.3	20.1	23.5	60.3	2.6
7										

Target P80 Size (if applic.): 100 | **kWh/t @ Target:** 5.0 | **Media Consumption (g/kWh):** 5

Signature Plot



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IsaMill Grinding Test Report

Project Name	M.A.Sc. Thesis	**Date(s) Tested**	10/07/2012
Duty Description	Primary Grinding	**Date Issued**	13/07/2012
Ore /Conc. Type	Copper Porphyry	**Location**	UBC CMP
Company	University of British Columbia	**IsaMill Type**	Netzsch M20
Contact Person	Chengtie Wang	**Media Spec.**	Cenotec 6.5 Graded Charge
Contact Details	chengtie@interchange.ubc.ca	**Contact Person**	ISA D1

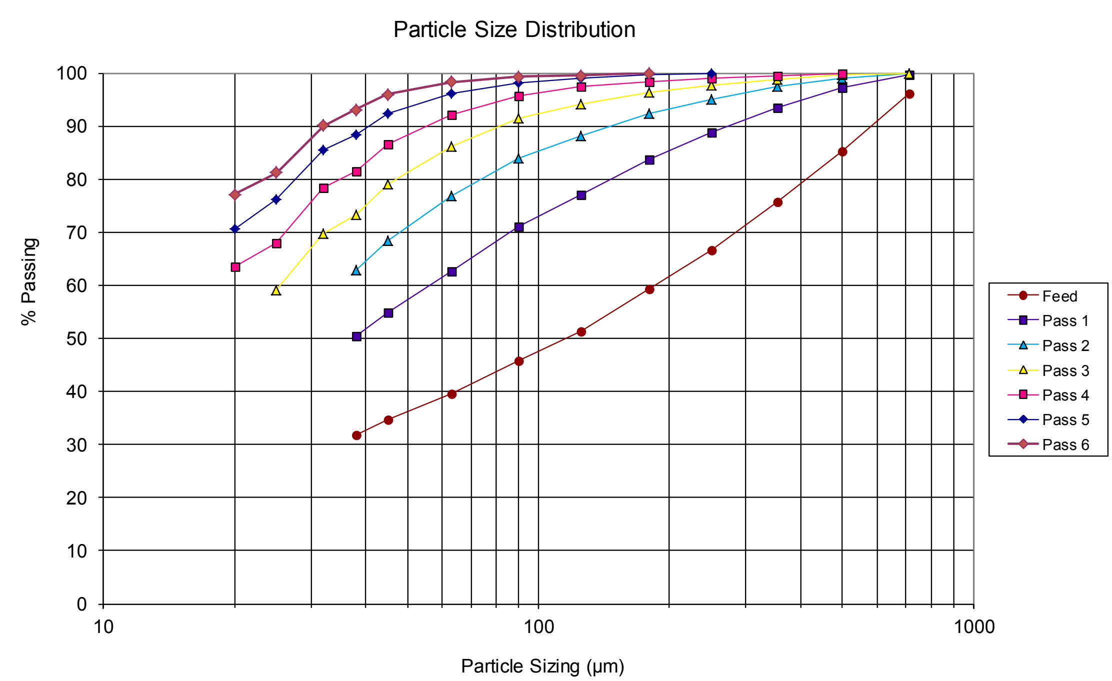
Sizing Analysis

Sizing Method	Mechanical Wet Screening	**Performed by**	Chengtie Wang

Sizing Data

Percent Passing

	Feed		Pass 1		Pass 2		Pass 3		Pass 4		Pass 5		Pass 6	
Size	Retained	Passing	Retained	Passing	Retained	Passing	Retained	Passing	Retained	Passing	Retained	Passing	Retained	Passing
[µm]	[g]	[%]	[g]	[%]	[g]	[%]	[g]	[%]	[g]	[%]	[g]	[%]	[g]	[%]
1000	0.0	100	0.0	100	0.0	100	0.0							
710	7.0	96	0.4	100	0.0	100	0.0	100						
500	19.3	85	3.2	97	1.2	99	0.3	100	0.0	100				
355	17.5	76	5.1	94	2.0	97	1.0	99	0.4	100				
250	16.2	67	6.3	89	2.9	95	1.3	98	0.4	99	0.0	100		
180	13.1	59	7.0	84	3.3	92	1.6	96	0.6	98	0.2	100	0.0	100
125	14.4	51	9.1	77	4.9	88	2.5	94	0.8	98	0.7	99	0.4	100
90	10.2	46	8.1	71	5.3	84	3.2	92	1.5	96	0.7	98	0.2	99
63	11.2	40	11.3	63	8.7	77	6.3	86	3.2	92	1.7	96	0.8	98
45	8.8	35	10.4	55	10.2	68	8.3	79	4.8	87	3.1	92	2.0	96
38	4.8	32	6.1	50	6.8	63	6.5	73	4.6	81	3.3	88	2.5	93
32							4.3	70	2.8	78	2.6	85	2.5	90
25							12.3	59	9.0	68	7.7	76	7.5	81
20									3.9	64	4.7	71	3.5	77
Pan	57.5		68.1		76.6		68.9		56.0		59.5		65.5	
Total	180.0		135.1		121.9		116.5		88.0		84.2		84.9	
P98		**850.9**		**558.9**		**410.2**		**271.8**		**155.3**		**88.7**		**60.3**
P80		**419.6**		**149.7**		**75.2**		**47.6**		**35.2**		**27.9**		**23.5**



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IsaMill Grinding Test Report

Project Name	Master Thesis HPGR/IsaMill	**Date(s) Tested**	16-Nov-11
Duty Description	Primary Grinding	**Date Issued**	23-Dec-11
Ore /Conc. Type	Copper Porphyry	**Location**	UBC CMP
Company	University of British Columbia	**IsaMill Type**	M20
Contact Person	Chengtie (Fisher) Wang	**Media Spec.**	Cenotec 6.5 Graded Charge
Contact Details	chengtie@interchange.ubc.ca	**Test Number**	ISA H1

Test Data

Solids SG (t/m3): 2.76 | **Media Vol (L):** 13 | **Media g (Start):** 30800 | **(Media g (End):** 30784

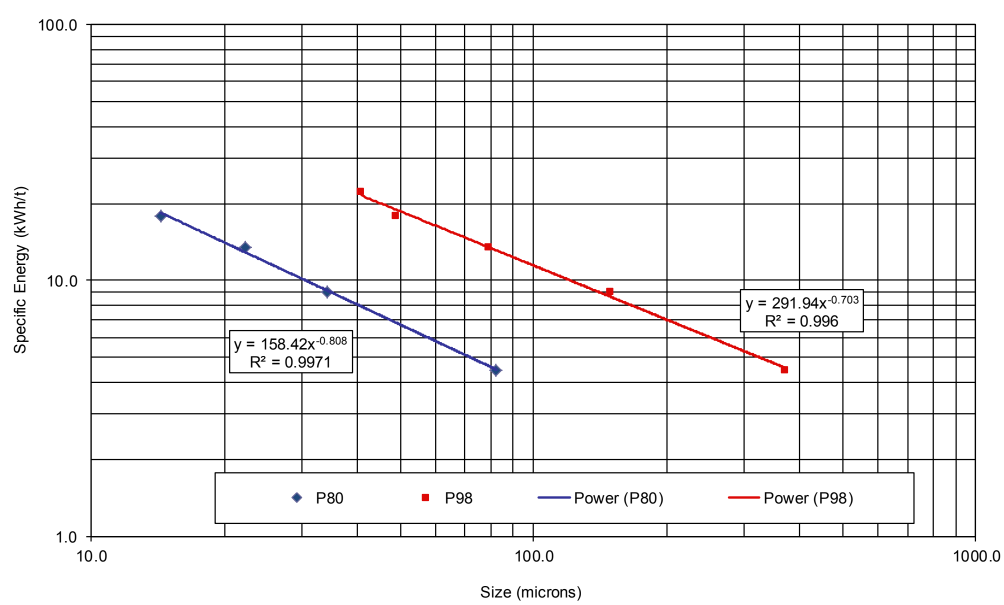
Pass #	N (rpm)	NLP (kW)	Q (sec/L)	Pump %	kg/L	Temp C	E (Wh)	Time (h)	Comments
1	964	5.44	2.56	100	1.47	21.5	812	0.081	
2	975	5.44	2.56	100	1.47	24.6	749	0.073	
3	976	5.44	2.56	100	1.47	28.7	734	0.073	
4	975	5.44	2.56	100	1.47	32.2	737	0.074	
5	975	5.44	2.56	100	1.47	35.8	712	0.073	
6	975	5.44	2.56	100	1.47	38.8	700	0.072	
7									

Calculated Data

Pass #	Gross kW	Net kW	Q (m3/h)	% Solids	M (t/h)	E (kWh/t)	Cumul. E	P80	P98	CSI
Feed								342.6	632.8	1.8
1	10.08	4.63	1.408	50.1%	1.038	4.5	4.5	82.3	369.4	4.5
2	10.21	4.77	1.408	50.4%	1.044	4.6	9.0	34.1	149.3	4.4
3	10.04	4.60	1.408	49.7%	1.029	4.5	13.5	22.3	79.1	3.5
4	10.01	4.56	1.408	49.9%	1.033	4.4	17.9	14.3	48.9	3.4
5	9.83	4.39	1.408	49.8%	1.031	4.3	22.2		40.8	
6	9.69	4.25	1.408	50.0%	1.035	4.1	26.3			
7										

Target P80 Size (if applic.): 75 | **kWh/t @ Target:** 4.8 | **Media Consumption (g/kWh):** 3

Signature Plot



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IsaMill Grinding Test Report

Project Name	Master Thesis HPGR/IsaMill	**Date(s) Tested**	16/11/2011
Duty Description	Primary Grinding	**Date Issued**	23/12/2011
Ore /Conc. Type	Copper Porphyry	**Location**	UBC CMP
Company	University of British Columbia	**IsaMill Type**	M20
Contact Person	Chengtie (Fisher) Wang	**Media Spec.**	Cenotec 6.5 Graded Charge
Contact Details	chengtie@interchange.ubc.ca	**Contact Person**	ISA H1

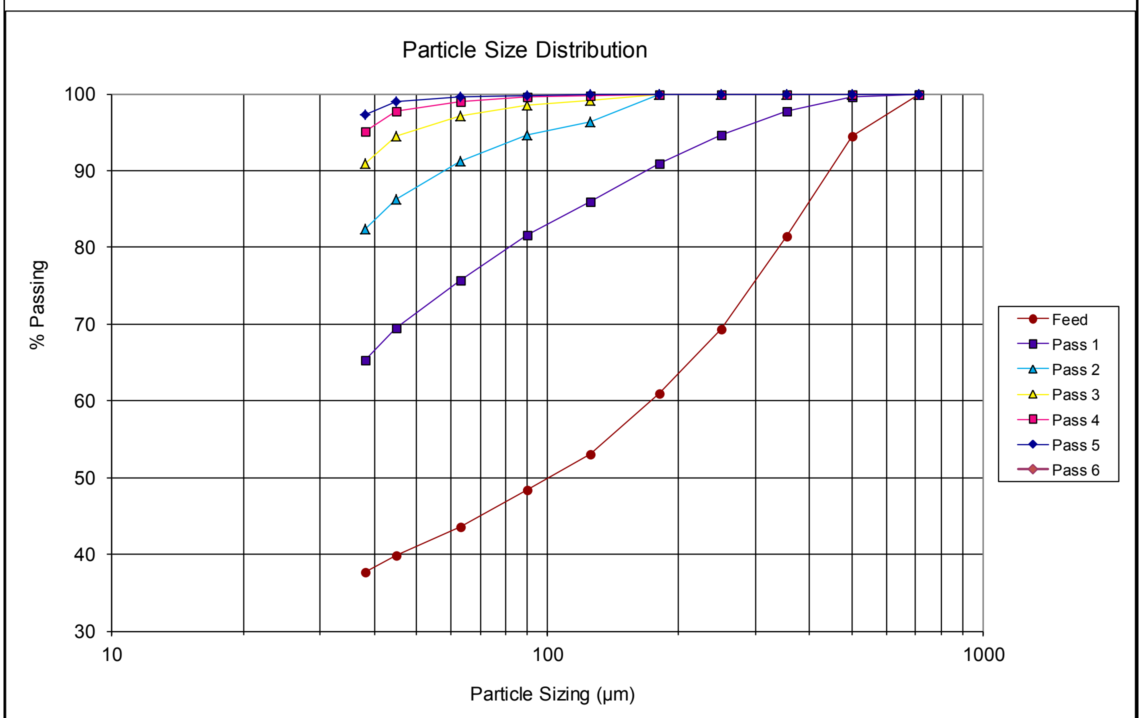
Sizing Analysis

Sizing Method	Mechanical Wet Screening	**Performed by**	Chengtie Wang

Sizing Data

Percent Passing

	Feed		**Pass 1**		**Pass 2**		**Pass 3**		**Pass 4**		**Pass 5**		**Pass 6**	
Size [µm]	Retained [g]	Passing [%]	Retained [g]	Passing [%]	Retained [g]	Passing [%]	Retained [g]	Passing [%]	Retained [g]	Passing [%]	Retained [g]	Passing [%]	Retained [g]	Passing [%]
710	0.0	100	0.0	100	0.0	100	0.0	100	0.0	100	0.0	100		
500	12.9	95	0.8	100	0.0	100	0.0	100	0.0	100	0.0	100		
355	31.2	81	4.5	98	0.0	100	0.0	100	0.0	100	0.0	100		
250	28.5	69	7.8	95	0.0	100	0.0	100	0.0	100	0.0	100		
180	20.2	61	8.8	91	0.0	100	0.0	100	0.0	100	0.0	100		
125	18.7	53	12.0	86	8.8	96	2.0	99	0.4	100	0.2	100		
90	11.1	48	10.4	82	4.3	95	1.4	99	0.5	100	0.2	100		
63	11.3	44	14.2	76	8.1	91	3.1	97	1.4	99	0.4	100		
45	9.1	40	15.2	70	12.2	86	6.4	94	3.3	98	1.4	99		
38	5.1	38	10.1	65	9.4	82	8.2	91	6.2	95	4.1	97		
Pan	89.6		157.7		201.1		213.7		225.5		223.9			
Total	237.7		241.4		243.9		234.8		237.1		230.2			
P80		**632.8**		**369.4**		**149.3**		**79.1**		**48.9**		**40.8**		
P98		**342.6**		**82.3**		**34.1**		**22.3**		**14.3**				



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IsaMill Grinding Test Report

Project Name	Master Thesis HPGR/IsaMill	**Date(s) Tested**	09-Nov-11
Duty Description	Primary Grinding	**Date Issued**	15-Nov-11
Ore /Conc. Type	Copper Porphyry	**Location**	UBC CMP
Company	University of British Columbia	**IsaMill Type**	M20
Contact Person	Chengtie (Fisher) Wang	**Media Spec.**	Cenotec 6.5 Graded Charge
Contact Details	chengtie@interchange.ubc.ca	**Test Number**	ISA H2

Test Data

Solids SG (t/m3): 2.76 **Media Vol (L):** 14 **Media g (Start):** 33239 **(Media g (End):** 33199

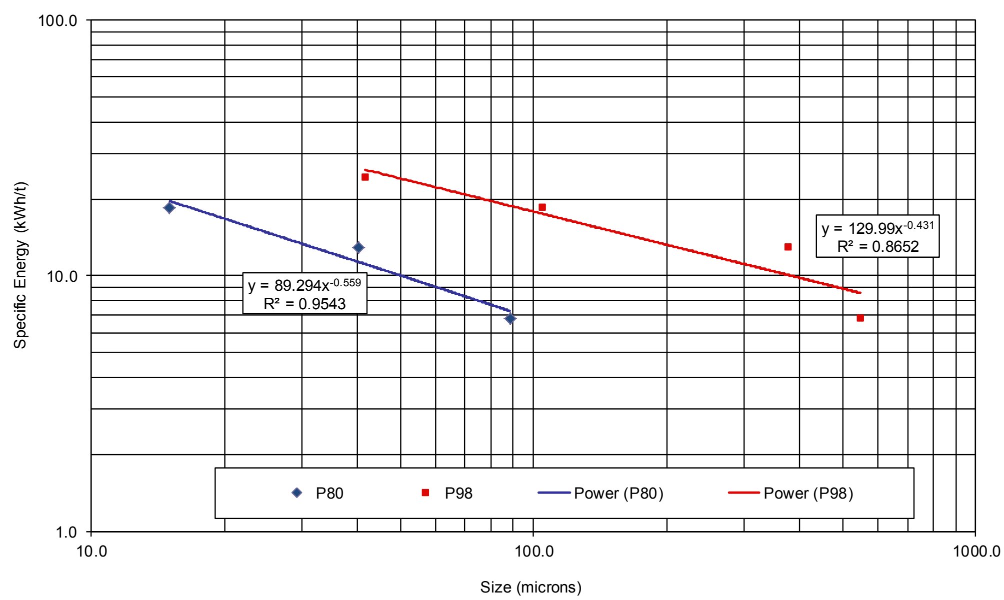
Pass #	N (rpm)	NLP (kW)	Q (sec/L)	Pump %	kg/L	Temp C	E (Wh)	Time (h)	Comments
1	974	5.49	2.56	100	1.55	22.2	1019	0.074	
2	982	5.49	2.56	100	1.55	30.6	948	0.073	
3	974	5.49	2.56	100	1.55	36.8	770	0.062	
4	975	5.49	2.56	100	1.55	41.3	816	0.066	
5	969	5.49	2.56	100	1.55	46.0	793	0.065	
6	972	5.49	2.56	100	1.55	49.9	807	0.068	
7									

Calculated Data

Pass #	Gross kW	Net kW	Q (m3/h)	% Solids	M (t/h)	E (kWh/t)	Cumul. E	P80	P98	CSI
Feed								354.1	746.8	2.1
1	13.74	8.25	1.408	55.6%	1.213	6.8	6.8	88.5	548.1	6.2
2	12.92	7.43	1.408	56.4%	1.231	6.0	12.8	40.2	377.9	9.4
3	12.43	6.94	1.408	55.7%	1.215	5.7	18.5	15.1	105.2	7.0
4	12.30	6.80	1.408	55.6%	1.213	5.6	24.2		41.7	
5	12.15	6.66	1.408	55.7%	1.216	5.5	29.6			
6	11.90	6.41	1.408	55.8%	1.217	5.3	34.9			
7										

Target P80 Size (if applic.): 75 **kWh/t @ Target:** 8.0 **Media Consumption (g/kWh):** 6

Signature Plot



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IsaMill Grinding Test Report

Project Name	Master Thesis HPGR/IsaMill	**Date(s) Tested**	09/11/2011
Duty Description	Primary Grinding	**Date Issued**	15/11/2011
Ore /Conc. Type	Copper Porphyry	**Location**	UBC CMP
Company	University of British Columbia	**IsaMill Type**	M20
Contact Person	Chengtie (Fisher) Wang	**Media Spec.**	Cenotec 6.5 Graded Charge
Contact Details	chengtie@interchange.ubc.ca	**Contact Person**	ISA H2

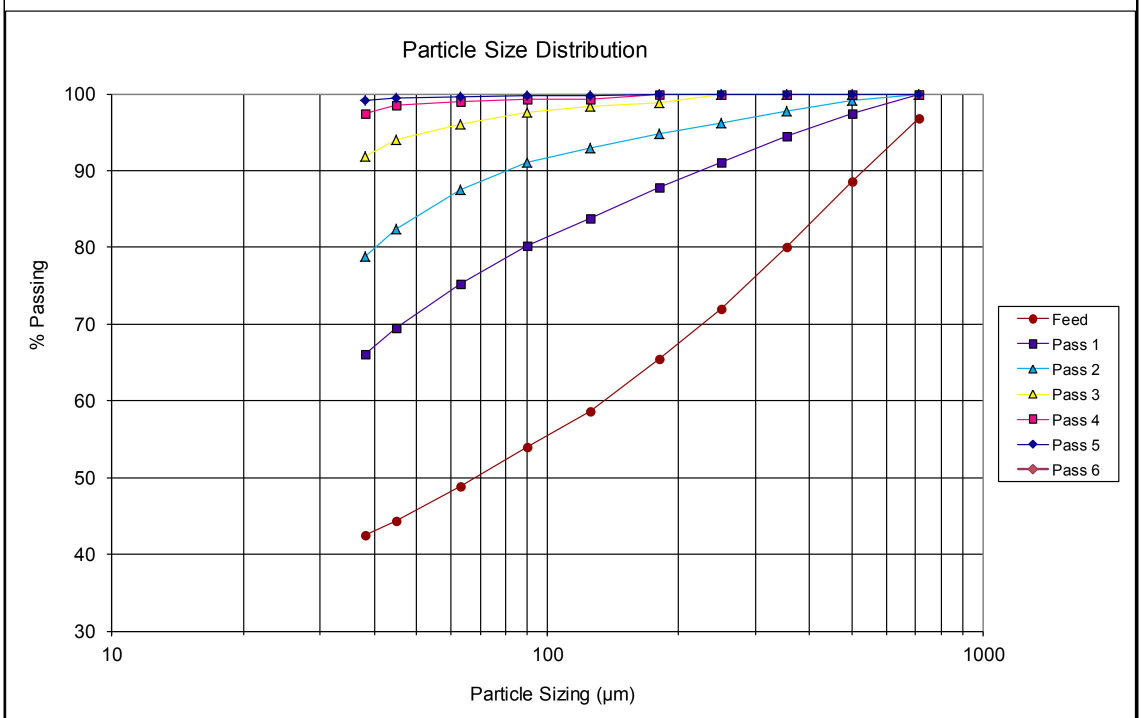
Sizing Analysis

Sizing Method	Mechanical Wet Screening	**Performed by**	Chengtie Wang

Sizing Data

Percent Passing

	Feed		**Pass 1**		**Pass 2**		**Pass 3**		**Pass 4**		**Pass 5**		**Pass 6**	
Size	Retained	Passing	Retained	Passing	Retained	Passing	Retained	Passing	Retained	Passing	Retained	Passing	Retained	Passing
[µm]	[g]	[%]	[g]	[%]	[g]	[%]	[g]	[%]	[g]	[%]	[g]	[%]	[g]	[%]
710	10.8	97	0.0	100	0.0	100	0.0	100	0.0	100	0.0	100		
500	27.8	89	6.3	97	1.4	99	0.0	100	0.0	100	0.0	100		
355	29.1	80	6.9	95	2.0	98	0.0	100	0.0	100	0.0	100		
250	27.4	72	8.2	91	2.4	96	0.0	100	0.0	100	0.0	100		
180	22.3	65	7.9	88	2.2	95	1.4	99	0.0	100	0.0	100		
125	23.1	59	9.9	84	2.8	93	0.4	98	0.7	99	0.5	100		
90	15.8	54	8.4	80	3.0	91	0.9	98	0.1	99	0.1	100		
63	17.3	49	12.2	75	5.5	88	1.8	96	0.3	99	0.2	100		
45	15.2	44	13.7	70	8.0	82	2.4	94	0.6	98	0.4	100		
38	6.5	42	8.5	66	5.5	79	2.5	92	1.2	97	0.9	99		
Pan	144.3		159.4		122.6		106.1		113.4		250.7			
Total	339.6		241.4		155.5		115.6		116.3		252.8			
P80		**746.8**		**548.1**		**377.9**		**105.2**		**41.7**		**n/a**		
P98		**354.1**		**88.5**		**40.2**		**15.1**		**n/a**		**n/a**		



Appendix E - JK SimMet® models Inputs

Equipment parameters

		A Circuit	C Circuit
Primary Mills	Description	SAG in closed circuit with stationary screen	Variable speed SAG in closed circuit with stationary screen
	Diameter (inside liners)	9.44 m	10.00 m
	Belly Length (inside liners)	4.25 m	4.57 m
	Feed Trunnion Diameter	2.16 m	2.13 m
	Feed and End Cone Angle	17 degrees	17 degrees
	Grate Opening (slot style)	22.23 mm	Outer = 28.58 mm, Inner = 22.23 mm
	Grate Open Area	7.9%	6.9%
	Mean Rel. Radial Pos. of the Grate Aperture	0.80	0.78
	Ball Load Estimate (%by vol.)	8.3%	12.5%
	Ball Top Size	127 mm	127 mm
	Mill Load at Time of Survey (%by vol.)	29.6%	26.3%
	Fraction of Critical Speed	75.7%	71.8%
	Ball Specific Gravity	7.8	7.8
	Measured Power	6726 kW	8474 kW
Stationary Screen	Aperture Opening Size (slots)	11.1 mm	15.9 mm
	Dimensions	8' x 20'	4' x 15'
	Open Area	35-40% new, but blind to as low as 10%	35-40% new, but blind to as low as 10%
Pebble Crushers	Description		
	Crusher CSS (estimate)		
	Eccentric throw		
	Liner Life (hrs)		
	Survey Power Draw (kW)		
	Max Power Draw (kW)		
	Max Power Draw (hp)		
	No Load Power (kW)		
Ball Mills	Description	Overflow BM in closed circuit with cyclones	Overflow BM in closed circuit with cyclones
	Number of ball mills	2	2
	Internal Diameter	A1 = 4.82 m, A2 = 4.81 m	C1 = 4.74 m, C2 = 4.85 m
	Internal Length	6.80 m	8.08 m
	Fraction of Critical Speed	A1 = 0.801, A2 = 0.800	C1 = 0.761, C2 = 0.770
	% Ball load	A1 = 35.60%, A2 = 35.55%	C1 = 35.36%, C2 = 35.68%
	Ball size	50% 63.5 mm / 50% 50.8 mm	50% 76.2 mm / 50% 63.5 mm
	Operating Power draw (kWh)	A1 = 2985 kW, A2 = 3091 kW	C1 = 2800 kW, C2 = 3242 kW
	Feed/Discharge Cone Angle	16 degrees	16 degrees
	Trunion Diameter	1.40 m	1.40 m
Cyclones	Number of cyclones per BM	10	10
	# of Operating Cyclones per BM during Survey	5	5
	Cyclone diameter	0.762 m	0.762 m
	Equivalent Inlet Diameter	0.305 m	0.305 m
	Vortex finder diameter	0.229 m	0.254 m
	Spigot / Apex diameter	0.145 m	C1 = 0.140 m, C2 = 0.142 m
	Length of cylindrical area	0.559 m	0.559 m
	Cone angle	16 degrees	16 degrees
	Survey Operating Pressure	see survey DCS data	see survey DCS data

		D Circuit	H Circuit
Primary Mills	Description	AG in closed circuit with vibrating screen and pebble crusher	Svedala SAG in closed circuit with vibrating screen and pebble crusher
	Diameter (inside liners)	10.08 m	9.75 m
	Belly Length (inside liners)	4.24 m	4.17 m
	Feed Trunnion Diameter	2.13 m	2.18 m
	Feed and End Cone Angle	17 degrees	15 degrees
	Grate Opening (slot style)	63.5 mm	25.4 mm & 38.1 mm combo grates
	Grate Open Area	6.3%	6.0%
	Mean Rel. Radial Pos. of the Grate Aperture	0.85	0.84
	Ball Load Estimate (%by vol.)	n/a	15.0%
	Ball Top Size	n/a	5"
	Mill Load at Time of Survey (%by vol.)	35.7%	25.0%
	Fraction of Critical Speed	74.3%	78.1%
	Ball Specific Gravity	n/a	7.8
	Measured Power	5662 kW	7452 kw
Stationary Screen	Aperture Opening Size (slots)	9.5 mm	variety of configurations: 10 mmx38 mm, 7 mmx37 mm & 9.5 mmx62 mm
	Dimensions	8' x 14'	21' 3" x 8'
	Open Area	30%	29.2%
Pebble Crushers	Description	7' Symons short head crusher	HP 500
	Crusher CSS (estimate)	12.7mm (estimate - no leading done)	10 mm
	Eccentric throw	???	n/a
	Liner Life (hrs)	???	419.12
	Survey Power Draw (kW)	76	n/a
	Max Power Draw (kW)	261	45 Amps
	Max Power Draw (hp)	350	
	No Load Power (kW)	14	8 Amps
Ball Mills	Description	Overflow BM in closed circuit with cyclones	Svedala Ball Mills
	Number of ball mills	1	2
	Internal Diameter	4.91 m	4.9 m
	Internal Length	8.78 m	9.144 m
	Fraction of Critical Speed	0.774	81%
	% Ball load	35.83%	33.9% for #1BM & 32.3% for #2BM
	Ball size	50% 76.2mm / 50% 63.5mm	76.2 mm
	Operating Power draw (kWh)	3607 kW	4015 for BM#1 and 4152 for BM#2
	Feed/Discharge Cone Angle	16 degrees	15 degrees
	Trunion Diameter	1.40 m	0.90 m
Cyclones	Number of cyclones per BM	10	6
	# of Operating Cyclones per BM during Survey	6	4
	Cyclone diameter	0.762 m	0.673 m
	Equivalent Inlet Diameter	0.305 m	0.219 m
	Vortex finder diameter	0.254 m	0.286 m
	Spigot / Apex diameter	0.142 m	0.152 m
	Length of cylindrical area	0.559 m	0.610 m
	Cone angle	16 degrees	11 degrees
	Survey Operating Pressure	see survey DCS data	see survey DCS data

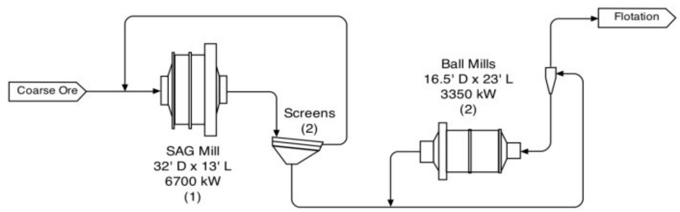
Crusher/HPGR parameters from JK Drop-weight tests

Case A - Appearance function					
Value of t10	t75	t50	t25	t4	t2
10	2.760	3.450	5.240	23.21	57.45
20	5.410	6.860	10.59	44.38	88.25
30	8.230	10.51	16.21	62.85	100.0
Case A - Breakage ECS data					
Value of t10	14.50	20.60	28.90	Initial particle size [mm]	
10	0.210	0.170	0.160	ECS [kWh/t]	
20	0.470	0.380	0.340		
30	0.780	0.650	0.570		
Case C - Appearance function					
Value of t10	t75	t50	t25	t4	t2
10	2.920	3.630	5.430	22.41	54.30
20	5.520	7.000	10.72	44.03	84.69
30	8.180	10.48	16.31	63.48	97.83
Case C - Breakage ECS data					
Value of t10	14.50	20.60	28.90	Initial particle size [mm]	
10	0.200	0.170	0.160	ECS [kWh/t]	
20	0.440	0.380	0.360		
30	0.740	0.640	0.610		
Case D - Appearance function					
Value of t10	t75	t50	t25	t4	t2
10	2.830	3.540	5.380	22.44	54.50
20	5.490	6.950	10.71	43.87	84.90
30	8.260	10.50	16.20	63.08	97.94
Case D - Breakage ECS data					
Value of t10	14.50	20.60	28.90	Initial particle size [mm]	
10	0.180	0.170	0.150	ECS [kWh/t]	
20	0.390	0.370	0.330		
30	0.670	0.610	0.560		
Case H - Appearance function					
Value of t10	t75	t50	t25	t4	t2
10	2.630	3.300	5.060	25.84	63.41
20	4.750	6.050	9.670	48.64	92.23
30	6.890	8.830	14.40	67.79	99.43
Case H - Breakage ECS data					
Value of t10	14.50	20.60	28.90	Initial particle size [mm]	
10	0.450	0.430	0.400	ECS [kWh/t]	
20	1.020	0.910	0.850		
30	1.810	1.460	1.360		

Appendix F - Benchmarking energy calculation

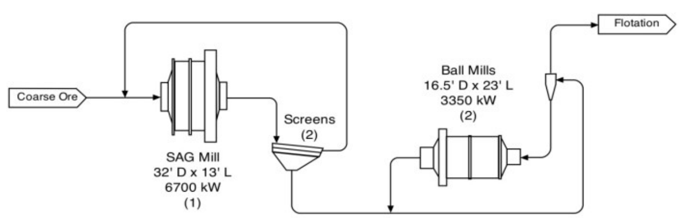
Circuit A

Circuit Feed			**Transfer**			**Product**		
F80=	108,300	um	T80=	1,500	um	P80=	188	um
CWi:	**8.09**	kWh/t	RWi:	**10.30**	kWh/t	BWi:	**13.80**	kWh/t



Primary mill					**Secondary mill**		
Easag=	5.14	kWh/t			Ebm=	4.86	kWh/t
Essbm=	8.69	kWh/t					
Total circuit power=	Wssbm+15%=	Easag+Ebm=	10.00	kWh/t			
Ball mill operating Wio=	10.31	kWh/t	74.7%	of BWi			
Circuit throughput:	889	tph					
EF4RM=	1.00	Only apply when >1					
EF4BM=	1.00	Only apply when >1					
EF5BM=	1.00	Only apply when P80 < 75um					

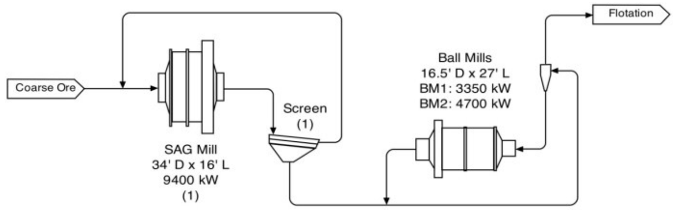
Circuit Feed			**Transfer**			**Product**		
F80=	108,300	um	T80=	1,500	um	P80=	100	um
CWi:	**8.09**	kWh/t	RWi:	**10.30**	kWh/t	BWi:	**13.80**	kWh/t



Primary mill					**Secondary mill**		
Easag=	5.14	kWh/t			Ebm=	9.15	kWh/t
Essbm=	12.43	kWh/t					
Total circuit power=	Wssbm+15%=	Easag+Ebm=	14.29	kWh/t			
Ball mill operating Wio=	12.34	kWh/t	89.4%	of BWi			
Circuit throughput:	889	tph					
EF4RM=	1.00	Only apply when >1					
EF4BM=	1.00	Only apply when >1					
EF5BM=	1.00	Only apply when P80 < 75um					

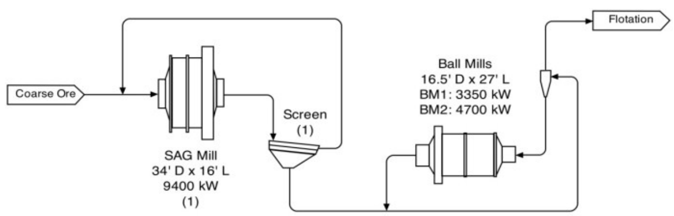
Circuit C

Circuit Feed				**Transfer**			**Product**		
F80=	91,500	um		T80=	2,000	um	P80=	265	um
CWi:	**10.60**	kWh/t		RWi:	**12.30**	kWh/t	BWi:	**13.60**	kWh/t



Primary mill							**Secondary mill**		
Easag=	5.21	kWh/t					Ebm=	3.41	kWh/t
Essbm=	7.50	kWh/t							
Total circuit power=	Wssbm+15%=	Easag+Ebm=	8.62	kWh/t					
Ball mill operating Wio=	8.73	kWh/t	64.2%	of BWi					
Circuit throughput:	1332	tph							
EF4RM=	1.00	Only apply when >1							
EF4BM=	1.00	Only apply when >1							
EF5BM=	1.00	Only apply when P80 < 75um							

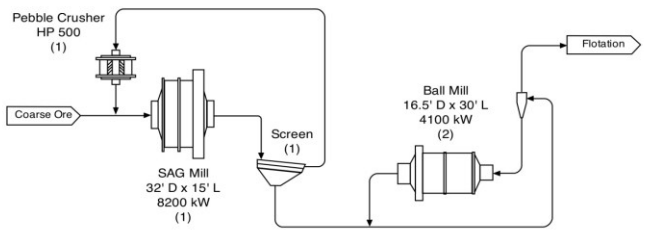
Circuit Feed				**Transfer**			**Product**		
F80=	91,500	um		T80=	2,000	um	P80=	100	um
CWi:	**10.60**	kWh/t		RWi:	**12.30**	kWh/t	BWi:	**13.60**	kWh/t



Primary mill							**Secondary mill**		
Easag=	5.21	kWh/t					Ebm=	9.44	kWh/t
Essbm=	12.74	kWh/t							
Total circuit power=	Wssbm+15%=	Easag+Ebm=	14.66	kWh/t					
Ball mill operating Wio=	12.16	kWh/t	89.4%	of BWi					
Circuit throughput:	1332	tph							
EF4RM=	1.00	Only apply when >1							
EF4BM=	1.00	Only apply when >1							
EF5BM=	1.00	Only apply when P80 < 75um							

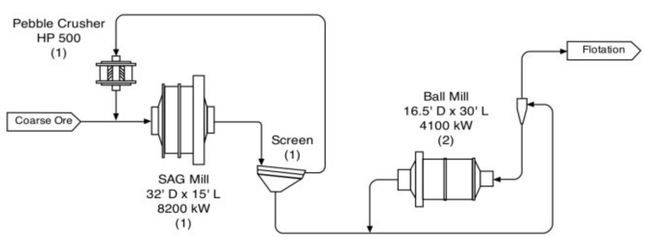
Circuit H

Circuit Feed				Transfer			Product		
F80=	64,800	um		T80=	1,800	um	P80=	160	um
CWi:	**20.00**	kWh/t		RWi:	**22.00**	kWh/t	BWi:	**18.00**	kWh/t



Primary mill				Secondary mill			Secondary mill		
Easag=	8.39	kWh/t		Epeb=	0.20	kWh/t	Ebm=	7.31	kWh/t
Essbm=	14.28	kWh/t							
Total circuit power=	Wssbm+10%=	Easag+Epeb+Ebm=	15.71	kWh/t					
Ball mill operating Wio=	13.18	kWh/t	73.2%	of BWi					
Circuit throughput:	766	tph							
EF4RM=	1.00	Only apply when >1							
EF4BM=	1.00	Only apply when >1							
EF5BM=	1.00	Only apply when P80 < 75um							

Circuit Feed				Transfer			Product		
F80=	64,800	um		T80=	1,800	um	P80=	75	um
CWi:	**20.00**	kWh/t		RWi:	**22.00**	kWh/t	BWi:	**18.00**	kWh/t



Primary mill				Secondary mill			Secondary mill		
Easag=	8.39	kWh/t		Epeb=	0.20	kWh/t	Ebm=	14.52	kWh/t
Essbm=	20.83	kWh/t							
Total circuit power=	Wssbm+10%=	Easag+Epeb+Ebm=	22.91	kWh/t					
Ball mill operating Wio=	15.80	kWh/t	87.8%	of BWi					
Circuit throughput:	766	tph							
EF4RM=	1.00	Only apply when >1							
EF4BM=	1.00	Only apply when >1							
EF5BM=	1.00	Only apply when P80 < 75um							

Appendix G - Equipment sizing

Sizing and selection of HPGRs for HPGR - ball mill circuit

Description	Units	Case A	Case C	Case D	Case H
Manufacturer	[-]	Polysius	Polysius	Polysius	Polysius
Model No.	[-]	Polycom 22/16	Polycom 24/17	Polycom 20/15	Polycom 22/16
Number required	[-]	1	1	1	1
Roll diameter	[mm]	2,200	2,400	2,000	2,200
Roll width	[mm]	1,550	1,650	1,500	1,550
Required throughput	[tph]	1625	2299	1380	1258
Specific pressing force	[N/mm^2]	3	3	3	3
Specific throughput constant (m-dot)	[ts/hm^3]	257	266	244	184
Net specific energy consumption	[kWh/t]	1.37	1.23	1.55	1.89
Installed motor power	[kW]	3,000	4,000	3,000	4,000
Operating motor power	[kW]	2,671	3,394	2,567	2,852

Sizing and selection of HPGRs for HPGR - stirred mill circuit

Description	Units	Case A		Case C	
HPGR Stage	[-]	First stage	Second stage	First stage	Second stage
Manufacturer	[-]	Polysius	Polysius	Polysius	Polysius
Model No.	[-]	Polycom 17/12	Polycom 24/17	Polycom 19/15	Polycom 22/16
Number required	[-]	1	1	1	2
Roll diameter	[mm]	1,700	2,400	1,850	2,200
Roll width	[mm]	1,200	1,650	1,500	1,500
Required throughput	[tph]	889	1725	1332	3100
Specific pressing force	[N/mm^2]	3	4	3	4
Specific throughput constant (m-dot)	[ts/hm^3]	257	191	266	208
Net specific energy consumption	[kWh/t]	1.37	2.22	1.23	1.87
Installed motor power	[kW]	1,600	5,600	3,700	2,500
Operating motor power	[kW]	1,462	4,596	1,966	3,478

Sizing and selection of HPGRs for HPGR - stirred mill circuit (cont'd)

Description	Units	Case D		Case H	
HPGR Stage	[-]	First stage	Second stage	First stage	Second stage
Manufacturer	[-]	Polysius	Polysius	Polysius	Polysius
Model No.	[-]	Polycom 17/12	Polycom 24/17	Polycom 17/14	Polycom 22/16
Number required	[-]	1	1	1	2
Roll diameter	[mm]	1,700	2,400	1,700	2,200
Roll width	[mm]	1,200	1,650	1,400	1,550
Required throughput	[tph]	765	1315	766	3100
Specific pressing force	[N/mm^2]	3	4	3	3
Specific throughput constant (m-dot)	[ts/hm^3]	244	142	184	222
Net specific energy consumption	[kWh/t]	1.55	2.9	1.89	1.25
Installed motor power	[kW]	1,600	2,800	1,600	2,800
Operating motor power	[kW]	1,423	4,576	1,737	2,325

Appendix H- Power consumption comparison

Power consumption comparison for case A

Description	Qt.	Unit inst. [kW]	Unit simu. [kW]	Total consumption [kW]	Specific energy [kWh/t]
889 tph throughput					
92% circuit overall availability -19,629 tpd					
SAB base case - 188 µm					
Comminution equipment					
SAG mill - 9.75 m D x 4.25 m EGL	1	6,700	6,293	6,293	7.08
Ball mill - 5.0 m D x 7.0 m L	2	3,350	2,182	4,365	4.91
Conveyors and feeders					
Coarse ore feeder	2	26	18	36	0.04
SAG mill feed conveyor	1	45	36	36	0.04
SAG mill screen O/S conveyor	3	14	13	40	0.05
Screens and pumps					
SAG discharge vibrating screen	1	36	18	18	0.02
SAG mill screen U/S discharge pump	1	300	241	241	0.27
Ball mill cyclone feed pump	2	300	222	445	0.50
TOTAL				**11,472**	**12.90**
SAB base case - 100 µm					
Comminution equipment					
SAG mill - 9.75 m D x 4.25 m EGL	1	6,700	6,262	6,262	7.04
Ball mill - 5.0 m D x 7.0 m L	2	3,350	3,326	6,651	7.48
Conveyors and feeders					
Coarse ore feeder	2	26	18	36	0.04
SAG mill feed conveyor	1	45	36	36	0.04
SAG mill screen O/S conveyor	3	14	7	22	0.03
Screens and pumps					
SAG discharge vibrating screen	1	36	18	18	0.02
SAG mill screen U/S discharge pump	1	300	241	241	0.27
Ball mill cyclone feed pump	2	300	289	578	0.65
TOTAL				**13,843**	**15.57**
HPGR - ball mill - 188 µm					
Comminution equipment					
Secondary crusher - MP 800	1	600	294	294	0.33
HPGR - 2.2 m D x 1.55 m W	1	3,000	2,574	2,574	2.90
Ball mill - 5.0 m D x 7.0 m L	2	3,350	2,689	5,378	6.05
Conveyors and feeders					
Coarse ore feeder	2	26	18	36	0.04
Coarse screen feed conveyor	1	75	66	66	0.07
Secondary crusher discharge conveyor	1	45	23	23	0.03

Coarse screen U/S to HPGR feed bin conveyor	1	200	162	162	0.18
HPGR discharge to fine screen conveyor	1	150	108	108	0.12
HPGR screen U/S discharge conveyor	1	45	19	19	0.02
Screens and pumps					
Coarse ore vibrating screen	1	90	36	36	0.04
HPGR discharge vibrating screen	2	90	36	71	0.08
Ball mill cyclone feed pump	2	300	267	533	0.60
TOTAL				**9,302**	**10.46**
HPGR - ball mill - 100 µm					
Comminution equipment					
Secondary crusher - MP 800	1	600	294	294	0.33
HPGR - 2.2 m D x 1.55 m W	1	3,000	2,574	2,574	2.90
Ball mill - 5.0 m D x 7.0 m L	2	3,350	4,307	8,614	9.69
Conveyors and feeders					
Coarse ore feeder	2	26	18	36	0.04
Coarse screen feed conveyor	1	75	66	66	0.07
Secondary crusher discharge conveyor	1	45	23	23	0.03
Coarse screen U/S to HPGR feed bin conveyor	1	200	162	162	0.18
HPGR discharge to fine screen conveyor	1	150	108	108	0.12
HPGR screen U/S discharge conveyor	1	45	19	19	0.02
Screens and pumps					
Coarse ore vibrating screen	1	90	36	36	0.04
HPGR discharge vibrating screen	2	90	36	71	0.08
Ball mill cyclone feed pump	2	400	356	711	0.80
TOTAL				**12,715**	**14.30**
HPGR - stirred mill - 100 µm					
Comminution equipment					
Secondary crusher - MP 800	1	600	294	294	0.33
1st HPGR - 1.7 m D x 1.2 m W	1	1,500	1,462	1,462	1.64
2nd HPGR - 2.4 m D x 1.65 m W	1	5,600	5,003	5,003	5.63
IsaMill™ - M10,000	2	2,000	1,605	3,209	3.61
Conveyors and feeders					
Coarse ore feeder	2	26	18	36	0.04
Coarse screen feed conveyor	1	75	66	66	0.07
Secondary crusher discharge conveyor	1	75	23	23	0.03
Coarse screen U/S to 1st HPGR feed bin conveyor	1	75	62	62	0.07
1st HPGR discharge to 2nd HPGR feed bin conveyor	1	250	220	220	0.25
2nd HPGR discharge to fine screen conveyor	1	200	165	165	0.19
2nd HPGR screen U/S discharge conveyor	1	75	21	21	0.02
Screens and pumps					
Coarse ore vibrating screen	1	90	36	36	0.04
2nd HPGR discharge vibrating screen	6	25	22	133	0.15
IsaMill™ feed pump	2	222	133	267	0.30
TOTAL				**10,996**	**12.37**

Power consumption comparison for case C

Description	Qt.	Unit inst. [kW]	Unit simu. [kW]	Total consumption [kW]	Specific energy [kWh/t]
1332 tph throughput					
92% circuit overall availability -29,410 tpd					
SAB base case - 265 µm					
Comminution equipment					
SAG mill - 10.36 m D x 4.57 m EGL	1	9,400	8,157	8,157	6.12
Ball mill - 5.0 m D x 8.23 m L	2	4,700	2,637	5,275	3.96
Conveyors and feeders					
Coarse ore feeder	2	30	27	53	0.04
SAG mill feed conveyor	1	45	53	53	0.04
SAG mill screen O/S conveyor	3	20	15	44	0.03
Screens and pumps					
SAG discharge vibrating screen	1	36	27	27	0.02
SAG mill screen U/S discharge pump	1	400	360	360	0.27
Ball mill cyclone feed pump	2	400	333	666	0.50
TOTAL				**14,635**	**10.99**
SAB base case - 100 µm					
Comminution equipment					
SAG mill - 10.36 m D x 4.57 m EGL	1	9,400	8,136	8,136	6.11
Ball mill - 5.0 m D x 8.23 m L	2	4,700	5,596	11,191	8.40
Conveyors and feeders					
Coarse ore feeder	2	30	27	53	0.04
SAG mill feed conveyor	1	75	53	53	0.04
SAG mill screen O/S conveyor	3	14	8	24	0.02
Screens and pumps					
SAG discharge vibrating screen	1	36	27	27	0.02
SAG mill screen U/S discharge pump	1	400	360	360	0.27
Ball mill cyclone feed pump	2	500	433	866	0.65
TOTAL				**20,711**	**15.55**
HPGR - ball mill - 265 µm					
Comminution equipment					
Secondary crusher - MP 800	1	600	488	488	0.37
HPGR - 2.4 m D x 1.65 m W	1	4,000	3,636	3,636	2.73
Ball mill - 5.0 m D x 8.23 m L	2	4,700	2,711	5,421	4.07
Conveyors and feeders					
Coarse ore feeder	2	30	27	53	0.04
Coarse screen feed conveyor	1	150	77	77	0.06
Secondary crusher discharge conveyor	1	45	23	23	0.02
Coarse screen U/S to HPGR feed bin conveyor	1	300	229	229	0.17

HPGR discharge to fine screen conveyor	1	200	156	156	0.12
HPGR screen U/S discharge conveyor	1	45	24	24	0.02
Screens and pumps					
Coarse ore vibrating screen	1	90	53	53	0.04
HPGR discharge vibrating screen	2	90	53	107	0.08
Ball mill cyclone feed pump	2	450	400	799	0.60
TOTAL				**11,066**	**8.31**
HPGR - ball mill - 100 μm					
Comminution equipment					
Secondary crusher - MP 800	1	600	488	488	0.37
HPGR - 2.4 m D x 1.65 m W	1	4,000	3,636	3,636	2.73
Ball mill - 5.0 m D x 8.23 m L	2	4,700	5,967	11,935	8.96
Conveyors and feeders					
Coarse ore feeder	2	30	27	53	0.04
Coarse screen feed conveyor	1	150	77	77	0.06
Secondary crusher discharge conveyor	1	45	23	23	0.02
Coarse screen U/S to HPGR feed bin conveyor	1	300	229	229	0.17
HPGR discharge to fine screen conveyor	1	200	156	156	0.12
HPGR screen U/S discharge conveyor	1	45	24	24	0.02
Screens and pumps					
Coarse ore vibrating screen	1	90	53	53	0.04
HPGR discharge vibrating screen	2	90	53	107	0.08
Ball mill cyclone feed pump	2	600	533	1,066	0.80
TOTAL				**17,846**	**13.40**
HPGR - stirred mill - 100 μm					
Comminution equipment					
Secondary crusher - MP 800	1	600	488	488	0.37
1st HPGR - 1.85 m D x 1.5 m W	1	1,500	1,966	1,966	1.48
2nd HPGR - 2.2 m D x 1.25 m W	2	5,000	4,228	8,456	6.35
IsaMill™ - M10,000	2	3,000	2,784	5,568	4.18
Conveyors and feeders					
Coarse ore feeder	2	30	27	53	0.04
Coarse screen feed conveyor	1	150	77	77	0.06
Secondary crusher discharge conveyor	1	45	23	23	0.02
Coarse screen U/S to 1st HPGR feed bin conveyor	1	150	96	96	0.07
1st HPGR discharge to 2nd HPGR feed bin conveyor	1	450	399	399	0.30
2nd HPGR discharge to fine screen conveyor	1	400	302	302	0.23
2nd HPGR screen U/S discharge conveyor	1	45	37	37	0.03
Screens and pumps					
Coarse ore vibrating screen	1	90	53	53	0.04
2nd HPGR discharge vibrating screen	12	25	17	200	0.15
IsaMill™ feed pump	2	222	200	400	0.30
TOTAL				**18,117**	**13.60**

Power consumption comparison for case D

Description	Qt.	Unit inst. [kW]	Unit simu. [kW]	Total consumption [kW]	Specific energy [kWh/t]
765 tph throughput					
92% circuit overall availability -16,891 tpd					
AGBC base case - 243 µm					
Comminution equipment					
AG mill - 10.36 m D x 4.24 m EGL	1	6,600	5,948	5,948	7.78
Pebble crusher - 7' Symons	1	350	103	103	0.13
Ball mill - 5.0 m D x 8.84 m L	1	4,100	3,213	3,213	4.20
Conveyors and feeders					
Coarse ore feeder	2	26	15	31	0.04
AG mill feed conveyor	1	45	31	31	0.04
AG mill screen O/S to pebble crusher conveyor	2	14	11	23	0.03
Pebble crusher discharge conveyor	1	14	11	11	0.02
Screens and pumps					
AG discharge vibrating screen	1	36	15	15	0.02
AG mill screen U/S discharge pump	1	250	207	207	0.27
Ball mill cyclone feed pumps	1	400	383	383	0.50
TOTAL				**9,964**	**13.03**
AGBC base case - 100 µm					
Comminution equipment					
AG mill - 10.36 m D x 4.24 m EGL	1	6,600	5,810	5,810	7.59
Pebble crusher - 7' Symons	1	350	107	107	0.14
Ball mill - 5.0 m D x 8.84 m L	1	4,100	6,918	6,918	9.04
Conveyors and feeders					
Coarse ore feeder	2	26	15	31	0.04
AG mill feed conveyor	1	45	31	31	0.04
AG mill screen O/S to pebble crusher conveyor	2	14	13	27	0.04
Pebble crusher discharge conveyor	1	14	13	13	0.02
Screens and pumps					
AG discharge vibrating screen	1	36	15	15	0.02
AG mill screen U/S discharge pump	1	229	207	207	0.27
Ball mill cyclone feed pump	1	600	497	497	0.65
TOTAL				**13,655**	**17.85**
HPGR - ball mill - 243 µm					
Comminution equipment					
Secondary crusher - MP 800	1	600	214	214	0.28
HPGR - 2.2 m D x 1.5 m W	1	3,000	2,716	2,716	3.55
Ball mill - 5.0 m D x 8.84 m L	1	4,100	3,726	3,726	4.87
Conveyors and feeders					
Coarse ore feeder	2	26	15	31	0.04
Coarse screen feed conveyor	1	100	67	67	0.09

Secondary crusher discharge conveyor	1	45	23	23	0.03
Coarse screen U/S to HPGR feed bin conveyor	1	200	141	141	0.18
HPGR discharge to fine screen conveyor	1	150	96	96	0.13
HPGR screen U/S discharge conveyor	1	45	26	26	0.03
Screens and pumps					
Coarse ore vibrating screen	1	90	31	31	0.04
HPGR discharge vibrating screen	2	90	31	61	0.08
Ball mill cyclone feed pump	1	500	459	459	0.60
TOTAL				**7,590**	**9.92**
HPGR - ball mill - 100 µm					
Comminution equipment					
Secondary crusher - MP 800	1	600	214	214	0.28
HPGR - 2.0 m D x 1.5 m W	1	3,000	2,716	2,716	3.55
Ball mill - 5.0 m D x 8.84 m L	1	4,100	7,428	7,428	9.71
Conveyors and feeders					
Coarse ore feeder	2	26	15	31	0.04
Coarse screen feed conveyor	1	100	67	67	0.09
Secondary crusher discharge conveyor	1	45	23	23	0.03
Coarse screen U/S to HPGR feed bin conveyor	1	200	141	141	0.18
HPGR discharge to fine screen conveyor	1	150	96	96	0.13
HPGR screen U/S discharge conveyor	1	45	26	26	0.03
Screens and pumps					
Coarse ore vibrating screen	1	90	31	31	0.04
HPGR discharge vibrating screen	2	90	31	61	0.08
Ball mill cyclone feed pump	1	700	612	612	0.80
TOTAL				**11,446**	**14.96**
HPGR - stirred mill - 100 µm					
Comminution equipment					
Secondary crusher - MP 800	1	600	214	214	0.28
1st HPGR - 1.7 m D x 1.2 m W	1	1,500	1,423	1,423	1.86
2nd HPGR - 2.4 m D x 1.65 m W	1	5,600	4,544	4,544	5.94
IsaMill™ - M10,000	2	2,200	1,817	3,634	4.75
Conveyors and feeders					
Coarse ore feeder	2	26	15	31	0.04
Coarse screen feed conveyor	1	100	67	67	0.09
Secondary crusher discharge conveyor	1	45	23	23	0.03
Coarse screen U/S to 1st HPGR feed bin conveyor	1	100	58	58	0.08
1st HPGR discharge to 2nd HPGR feed bin conveyor	1	250	179	179	0.23
2nd HPGR discharge to fine screen conveyor	1	200	133	133	0.17
2nd HPGR screen U/S discharge conveyor	1	45	28	28	0.04
Screens and pumps					
Coarse ore vibrating screen	1	90	31	31	0.04
2nd HPGR discharge vibrating screen	6	25	19	115	0.15
IsaMill™ feed pump	2	222	115	230	0.30
TOTAL				**10,707**	**14.00**

Power consumption comparison for case H

Description	Qt.	Unit inst. [kW]	Unit simu. [kW]	Total consumption [kW]	Specific energy [kWh/t]
766 tph throughput					
92% circuit overall availability -16,913 tpd					
SABC base case - 160 µm					
Comminution equipment					
SAG mill - 9.76 m D x 4.11 m EGL	1	8,200	7,859	7,859	10.26
Pebble crusher - HP 800	1	300	123	123	0.16
Ball mill - 5.00 m D x 9.14 m L	2	4,100	3,194	6,388	8.34
Conveyors and feeders					
Coarse ore feeder	2	26	15	31	0.04
SAG mill feed conveyor	1	45	31	31	0.04
SAG mill screen O/S to pebble crusher conveyor	2	14	8	15	0.02
Pebble crusher discharge conveyor	1	14	8	8	0.01
Screens and pumps					
SAG discharge vibrating screen	1	36	15	15	0.02
SAG mill screen U/S discharge pump	1	229	207	207	0.27
Ball mill cyclone feed pump	2	222	192	383	0.50
TOTAL				**15,060**	**19.66**
SABC base case - 75 µm					
Comminution equipment					
SAG mill - 9.76 m D x 4.11 m EGL	1	8,200	8,120	8,120	10.60
Pebble crusher - HP 800	1	300	61	61	0.08
Ball mill - 5.00 m D x 9.14 m L	2	4,100	4,443	8,886	11.60
Conveyors and feeders					
Coarse ore feeder	2	26	15	31	0.04
SAG mill feed conveyor	1	45	31	31	0.04
SAG mill screen O/S to pebble crusher conveyor	2	14	4	8	0.01
Pebble crusher discharge conveyor	1	14	8	8	0.01
Screens and pumps					
SAG discharge vibrating screen	1	36	15	15	0.02
SAG mill screen U/S discharge pump	1	229	207	207	0.27
Ball mill cyclone feed pump	2	222	249	498	0.65
TOTAL				**17,863**	**23.32**
HPGR - ball mill - 160 µm					
Comminution equipment					
Secondary crusher - MP 800	1	600	356	356	0.47
HPGR - 2.4 m D x 1.7 m W	1	4,000	3,049	3,049	3.98
Ball mill - 5.00 m D x 9.14 m L	2	4,100	3,363	6,725	8.78
Conveyors and feeders					
Coarse ore feeder	2	26	15	31	0.04
Coarse screen feed conveyor	1	45	67	67	0.09

Secondary crusher discharge conveyor	1	45	23	23	0.03
Coarse screen U/S to HPGR feed bin conveyor	1	150	130	130	0.17
HPGR discharge to fine screen conveyor	1	75	89	89	0.12
HPGR screen U/S discharge conveyor	1	45	26	26	0.03
Screens and pumps					
Coarse ore vibrating screen	1	90	31	31	0.04
HPGR discharge vibrating screen	2	90	38	77	0.10
Ball mill cyclone feed pump	2	300	230	460	0.60
TOTAL				**11,063**	**14.44**
HPGR - ball mill - 75 µm					
Comminution equipment					
Secondary crusher - MP 800	1	600	356	356	0.47
HPGR - 2.2 m D x 1.55 m W	1	4,000	3,049	3,049	3.98
Ball mill - 5.00 m D x 9.14 m L	2	4,100	5,473	10,946	14.29
Conveyors and feeders					
Coarse ore feeder	2	26	15	31	0.04
Coarse screen feed conveyor	1	75	67	67	0.09
Secondary crusher discharge conveyor	1	45	23	23	0.03
Coarse screen U/S to HPGR feed bin conveyor	1	150	130	130	0.17
HPGR discharge to fine screen conveyor	1	150	89	89	0.12
HPGR screen U/S discharge conveyor	1	75	26	26	0.03
Screens and pumps					
Coarse ore vibrating screen	1	90	31	31	0.04
HPGR discharge vibrating screen	2	90	38	77	0.10
Ball mill cyclone feed pump	2	400	306	613	0.80
TOTAL				**15,437**	**20.15**
HPGR - stirred mill - 75 µm					
Comminution equipment					
Secondary crusher - MP 800	1	600	356	356	0.47
1st HPGR - 1.7 m D x 1.4 m W	1	1,500	1,737	1,737	2.27
2nd HPGR - 2.2 m D x 1.55 m W	2	2,800	2,707	5,414	7.07
IsaMill™ - M10,000	2	2,600	1,746	3,493	4.56
Conveyors and feeders					
Coarse ore feeder	2	26	15	31	0.04
Coarse screen feed conveyor	1	100	67	67	0.09
Secondary crusher discharge conveyor	1	45	23	23	0.03
Coarse screen U/S to 1st HPGR feed bin conveyor	1	75	58	58	0.08
1st HPGR discharge to 2nd HPGR feed bin conveyor	1	450	399	399	0.52
2nd HPGR discharge to fine screen conveyor	1	400	302	302	0.39
2nd HPGR screen U/S discharge conveyor	1	45	28	28	0.04
Screens and pumps					
Coarse ore vibrating screen	1	90	31	31	0.04
2nd HPGR discharge vibrating screen	12	15	13	153	0.20
IsaMill™ feed pump	2	200	115	230	0.30
TOTAL				**12,321**	**16.09**