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The application of a simplified approach to modelling tumbling mills, stirred media mills and HPGR's

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ABSTRACT

Most modern population balance models for comminution invoke the concept of a specific breakage rate function and a breakage distribution function to describe breakage kinetics. One of the difficulties of this approach is that these functions are very difficult to measure directly. Consequently, it is usual to assume that these functions can be represented by simple equations with parameters that can easily be estimated from test data using back-calculation techniques. However, these estimates can be very sensitive to small measurement errors and are usually subject to very large variances. This paper presents a simplified approach to modelling comminution processes that invoke the concept of an energy-based cumulative breakage rate function to describe breakage kinetics. This function can be estimated directly from plant data and is well-suited to multi-component modelling of individual rock types and mineral species. Examples of the application of this simplified modelling approach are described for the treatment of platinum ores using ball mills, AG/SAG mills, HPGR's and stirred media mills.

1. Introduction

The steady-state and dynamic performance of grinding mills can be described using a population balance approach to quantify the size distribution and composition of material inside the mill. It is common to select sizes conforming to a root-two geometric progression given by x_i, where the subscript i serves as an index to reference each size class. The first or largest size, x_1, is normally selected to be larger than the largest particles likely to be encountered in the feed stream. The so-called sink size class n references all particles with sizes between zero and x_n. The composition of material in each size class can be expressed in term of rock type, assay value and mineral content. Consideration should also be given to the state of liberation of each species. Breakage behaviour inside the mill is usually quantified in terms of a specific breakage rate function and a separate breakage distribution function. Material transport behaviour is usually expressed in terms of a residence time distribution and a specific discharge rate function. In this introduction, the basic mathematical structure of "standard" population balance models will be reviewed to highlight some of the dilemmas that must be resolved to estimate model parameters accurately.

The specific breakage rate function for tumbling and stirred media mills, $S_{i,k}$, can be defined as the statistical average of the fractional rate (per unit time) that particles of composition or species type k break out of size class i. Because a time-based breakage rate function can be sensitive to mill geometry and operating conditions, it is desirable to make a transformation to an energy-normalised breakage rate function defined by $S^E_{i,k} = S_{i,k}M/P$ (Herbst and Fuerstenau, 1980) where P is the net mill power and M is the total mass of the material to be broken in the mill. The energy-normalised breakage rate function is generally insensitive to scale-up. In principle, it should be possible to measure the specific breakage rate function directly by using radioactive tracer techniques to "tag-and-track" each species of interest during its passage through the mill. This is very difficult and impractical to do on a routine basis. Consequently, simplifying assumptions must be made to obtain an estimate of the breakage rate function. A commonly made assumption is that the breakage rate function can be represented by a simple equation with parameters that can be back-calculated from test data (Austin et al., 1984).

The primary breakage distribution function for a tumbling or stirred media mill, $b_{i,j,k}$, is defined as the mass fraction of species k breaking out of size class j that appears in size class i before particles have had a chance to be re-broken. Again, in principle, it should be possible to estimate the breakage distribution function using radioactive tracer techniques, but such an approach has yet to be proven viable. To resolve this dilemma, it is possible to adopt a similar approach to that described above by assuming that the breakage distribution function can be represented by a simple equation with parameters that can be identified by back-calculation. An alternative approach is to derive a plausible breakage distribution function from laboratory dropweight and tumbling tests

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on small rocks in narrow size ranges (Napier-Munn et al., 1996). However, such tests may not reflect the environmental conditions likely to be encountered in a pilot or production mill where there can be a wide spectrum of particle sizes and modes of breakage. It follows that there is a high risk that the derived breakage distribution function does not reflect reality.

A popular approach to modelling material transport behaviour or residence time distributions in tumbling and stirred media mills is to partition the mill into one or more axial segments. Each segment is then treated as a perfectly mixed reactor. The number of segments required will depend on the aspect ratio of the mill (diameter to length ratio). The model can be refined by allowing for back-mixing between segments. In the case of a grate discharge mill, it is common to invoke the concept of a specific discharge rate function, $g_{i,k}$, defined as the fractional mass rate that particles of a given species type and size leave the discharge segment.

A generic population balance model for a tumbling or stirred media mill that can be treated as a single fully mixed reactor is given by the following equation:

$$accumulation = flow\ in - flow\ out - consumption + generation$$

$$\frac{dw_{i,k}}{dt} = f_{i,k} - g_{i,k}w_{i,k} - S^E_{i,k}\frac{P}{M}w_{i,k} + \sum_{\substack{j=1\\ i>1}}^{i-1} b_{i,j,k}S^E_{j,k}\frac{P}{M}w_{j,k} \tag{1}$$

where $w_{i,k}$ is the hold-up mass of size class i and rock type k particles, (t); $f_{i,k}$ is the mill inlet flow rate of size class i and rock type k particles, (t/h); t is time, (h); $S^E_{i,k}$ is the specific breakage rate for size class i and rock type k (kWh/t)$^{-1}$; $g_{i,k}$ is the specific discharge rate for size class i and rock type k (h)$^{-1}$; $b_{i,j,k}$ is the primary breakage distribution for rock type k from size j to i; P is mill power (kW); M is ore hold-up (t).

Eq. (1) takes the standard form of a state-variable description of a dynamical system where the dependent variables or states are given by the hold-ups $w_{i,k}$ inside the mill. The resulting set of differential equations can be solved numerically to simulate both the dynamic and steady-state behaviour of the mill. In the case of AG/SAG mills, the mill power P and total ore mass M are both functions of the state variables. Equations for estimating the power draw for tumbling and stirred media mills in terms of the state variables are well-established (Rowland and Kjos, 1980; Austin, 1990; Tüzün, 1993).

Breakage in an HPGR is fundamentally different to a tumbling or stirred media mill in that the relationship between grind size and specific energy input is highly nonlinear. This arises from the fact that as material passes through the rolls, the void volume in the particulate bed becomes progressively smaller, resulting in disproportionately greater dissipation of energy in the form of rolling friction. As a result, the energy component that goes into particle breakage becomes progressively reduced. Fuerstenau et al. (1991) have indicated that grinding kinetics for HPGR's can best be represented by population balance equations expressed directly as a function of the net specific energy input ξ (kwh/t). If $w'_{i,k}$ is the mass fraction in the product stream for size class i and species k, then the population balance for breakage in an HPGR is given by:

$$\frac{dw'_{i,k}(\xi')}{d\xi'} = -S^E_{i,k}w'_{i,k}(\xi') + \sum_{j=1}^{i-1} S^E_{j,k}b_{i,j,k}w'_{j,k}(\xi') \tag{2}$$

where $\xi' = \xi^{1-\tau}/(1-\tau)$ and the exponent $0 \leqslant \tau \leqslant 1$ is an energy dissipation parameter. The net specific energy consumed in an HPGR is determined by the net power absorbed and throughput, which can be related to the geometry of the rolls, roll speed and set-up conditions (initial or zero gap setting and hydro-pneumatic pressures). Important parameters for scale-up from pilot test data are the specific pressure force, specific throughput and the specific power draw (Klymowsky et al., 2002). A more elaborate model for HPGR's has been described by Daniel and Morell (2004) that involves dividing the internals of the HPGR into a number of zones and allowing for roll edge effects. Although the modelling of HPGR's appears to be straightforward, very little seems to have be done to understand and quantify individual mineral deportment and liberation behaviour.

From the above discussion, it should be evident that mathematical structures for population balance models describing total solids and individual mineral behaviour for tumbling mills, stirred media mills and HPGR's are well-established. However, the accurate identification of model parameters can be a formidable task, even for models that keep track of a single species, let alone multiple species with multiple states of liberation. Accordingly, it is often necessary to make gross simplifying assumptions regarding the structure of the breakage rate and breakage distribution functions. Moreover, heavy reliance must be placed on back-calculation or nonlinear regression techniques to determine parameters. Small measurement errors can often lead to huge variances for the parameters being identified and it is doubtful that the estimates of the breakage rate and breakage distribution functions accurately reflect reality.

2. Simplified approach to modelling comminution circuits

The approach adopted by Mintek to design flowsheets for new concentrators or upgrading and optimizing existing plants is to conduct tests at laboratory or pilot-scale. At laboratory scale, it is possible to explore different circuit configurations using locked cycle techniques and at pilot-scale it is possible to simulate the performance of complete circuits operated continuously. Such circuits usually involve the integration of comminution, flotation and physical separation processes for the concentration of selected mineral species. The attractive feature of conducting tests at pilot-scale is that the risk of scale-up errors is relatively low. In the case of grinding circuits, scale-up is usually based on the assumption that specific energy input remains essentially constant with scale-up, although some corrections must be made to allow for mill geometry effects.

Obviously, it can be a costly exercise to pilot a large number of circuit configurations when seeking a solution to a design or optimization problem. This is where computer simulation studies can serve a useful role in guiding the selection of the most promising options to pilot. Over the last couple of decades, Mintek has coded a suite of population balance models of conventional crushers, tumbling mills, stirred media mills, HPGR's and various size classifiers (vibrating screens, hydrocyclones and elutriators). The comminution models not only include those described in the introductory discussion, but also those with a much simpler mathematical structure and with parameters that can be derived directly from pilot tests or from data generated from sampling campaigns conducted at production scale. These simplified models also allow for the behaviour of individual rock types or mineral species. Most of the concepts behind these simple models are well-described in the literature for tumbling mills, but have yet to be explored fully with regard to stirred media mills and HPGR's. All the simplified models are based on the assumption that grinding kinetics can be described by a single function, instead of two separate functions. This function is the cumulative breakage rate function, defined as the rate per unit mass that a given species coarser than a given size breaks to below that size. It plays an analogous role to a partition function used in physical separation processes in that parameters can be related to equipment design geometry and operating conditions in much the same way. Models based on the use of this function are collectively referred to as *Cumulative Rate* models.

2.1. Cumulative rate models for tumbling and stirred media mills

For conventional tumbling mills (rod, ball, and pebble mills), it has been found that models using a cumulative breakage rate function can easily be fitted to pilot or plant data and appear to be adequate for evaluating competing circuits (Hinde and King, 1978; Finch and Ramirez-Castro, 1980; Laplante et al., 1987). In these publications, cumulative breakage rates were expressed in the time domain, but it has been found to be more useful to use an energy-normalised form. For a perfectly mixed overflow mill where the size distribution of the mill charge and discharge stream are usually very similar, the population balance for material coarser than size x_i inside the mill is given by:

$$\begin{aligned}&\mathit{Accumulation} = \mathit{in} - \mathit{out} - \mathit{consumption}\\ &\frac{dMW_{i,k}}{dt} = F[(1 - F_{i,k}) - W_{i,k}] - \left(\frac{P}{M}\right)K^E_{i,k}MW_{i,k}\end{aligned} \tag{3}$$

where $W_{i,k}$ is the mass fraction of material coarser than size x_i in the mill; F is the feed flow rate (t/h); M is the mass hold-up of ore (t); $F_{i,k}$ is the mass fraction of feed finer than x_i; $K^E_{i,k}$ is the energy-normalised cumulative breakage rate function.

At steady-state Eq. (3), simplifies to

$$P_{i,k} = \frac{F_{i,k} + \xi F_{i,k}}{1 + \xi K^E_{i,k}} \tag{4}$$

where $P_{i,k} = (1 - W_{i,k})$ is the mass fraction less than size x_i in the product stream and ξ is the specific energy input. For a plug flow residence time distribution the relationship between the product and feed size distribution is given by:

$$P_{i,k} = 1 - (1 - F_{i,k})\exp(-\xi K^E_{i,k}) \tag{5}$$

It should be evident that given the feed and product size distribution, together with the specific energy input, it is possible to determine the energy-normalised cumulative breakage rate function without recourse to back-calculation methods.

Fig. 1 shows the raw data and model fit for batch tests conducted in an instrumented ball mill with a motor shaft torque sensor and tachometer to allow for the accurate estimation of the net energy consumed by the mill charge.

A similar model can also be used to describe the behaviour of stirred media mills. In general, the simplified model provides a poorer fit than the standard model to test data. An improved fit can, however, be achieved by allowing a nonlinear relationship between breakage rates and specific energy input.

2.2. A simplified model for SAG mills

The application of the concept of a cumulative breakage rate function to the steady-state modelling of SAG mills (including fully autogenous and run-of-mine ball mills) has been described by Austin et al. (1993). The dynamic modelling of an open circuit SAG mill using an energy-normalised cumulative breakage rate function was first described in 1993 (Amestica et al., 1993) and again in 1996 (Amestica et al., 1996).

The specific cumulative breakage rate function, $K_{i,k}$, in the time domain is defined as the fractional rate at which particles of a given species or rock type, k, above a given size, x_i, in the mill break to below that size per unit time. A transformation relationship, similar to that used for the standard model, can used to define an energy-normalised cumulative breakage rate function, ($K^E_{i,k} = K_{i,k}M/P$).

Consider a population balance for a continuously fed mill (treated as a single fully mixed reactor) for particles in size class 1 and rock type k. If there are no particles coarser than size x_1 the specific breakage rate, $K^E_{1,k}$, of particles coarser than this size is undefined. It follows that the population balance equation with respect to time for size class 1 is given by:

$$\begin{aligned}&\mathit{accumulation} = \mathit{in} - \mathit{out} - \mathit{consumption}\\ &\frac{d(W_{2,k})}{dt} = F_{2,k} - P_{2,k} - W_{2,k}\frac{P}{M}K^E_{2,k}\end{aligned} \tag{6}$$

where $F_{2,k}$ and $P_{2,k}$ are the cumulative mass flows of material coarser than size x_2 in the feed and product streams, respectively. $W_{i,k}$ is now defined as the absolute mass of material coarser than size x_i inside the mill and $w_{i,k}$ is the absolute mass retained in size class i. For the first size class, Eq. (6) is identical to:

$$\begin{aligned}&\mathit{accumulation} = \mathit{in} - \mathit{out} - \mathit{consumption}\\ &\frac{d(w_{1,k})}{dt} = f_{1,k} - g_{1,k}w_{1,k} - w_{1,k}\frac{P}{M}K^E_{2,k}\end{aligned} \tag{7}$$

where $f_{i,k}$ is absolute mass flowrate for the inlet feed to the mill.

The first term on the right-hand side of Eq. (7) gives the flowrate of size class 1 and rock type k into the mill. The second

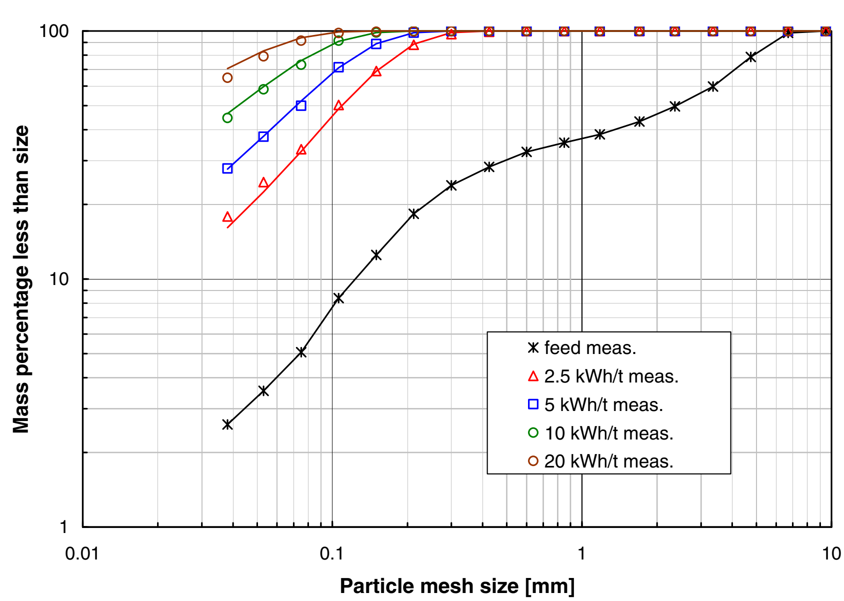

Fig. 1. Fit of simple model to tests conducted using a batch ball mill.

term expresses the discharge flow rate in terms of the value of the specific discharge function and mass hold-up for size class 1 and rock type k. The third term in Eq. (7) assumes that breakage rates are proportional to the product of the specific power input and the mass of size class 1 material inside the mill. When expressed in this form, the energy-based specific breakage rate function, $K^E_{i,k}$ $(\text{kWh/t})^{-1}$ is assumed to be insensitive to scale-up. It should be noted that the power draw of the mill is strongly dependent on the hold-up of rock and pebbles inside the mill, which makes it necessary to solve this equation by numerical methods.

The accumulation or rate of change of the hold-up of material coarser than x_i ($i = 2, 3, \ldots, n-1$) is given by:

$$\frac{d(W_{i,k})}{dt} = F_{i,k} - P_{i,k} - W_{i,k}\frac{P}{M}K^E_{i,k} \tag{8}$$

where the first term on the right-hand size of this equation is the inlet flowrate of rock type k coarser than size x_i and the second term is the discharge flowrate of material coarser than size x_i. The third term is the rate at which material coarser than size x_i breaks to below this size. A similar equation can be applied to material coarser than size x_{i+1}:

$$\frac{d(W_{i+1,k})}{dt} = F_{i+1,k} - P_{i+1,k} - W_{i+1,k}\frac{P}{M}K^E_{i+1,k} \tag{9}$$

Subtracting Eq. (8) from Eq. (9) gives the accumulation of material within size class i:

$$\begin{aligned} &\mathit{accumulation} = \mathit{in} - \mathit{out} + (\mathit{generation} - \mathit{consumption}) \\ &\tfrac{dw_{i,k}}{dt} = f_{i,k} - g_{i,k}w_{i,k} + \left\{W_{i,k}\tfrac{P}{M}K^E_{i,k} - (w_{i,k} + W_{i,k})\tfrac{P}{M}K^E_{i+1,k}\right\} \end{aligned} \tag{10}$$

The mass balance for particles in size class n (the sink interval with particle sizes between zero and x_n) is given by:

$$\frac{d(w_{n,k})}{dt} = f_{n,k} - g_{n,k}w_{n,k} + W_{n,k}\frac{P}{M}K^E_{n,k} \tag{11}$$

The water balance is given by:

$$\begin{aligned} &\mathit{accumulation} = \mathit{in} - \mathit{out} \\ &\tfrac{dw_0}{dt} = f_0 - g_0 w_0 \end{aligned} \tag{12}$$

The above equations can be solved numerically to simulate both the dynamic and steady-state behaviour of the mill for any circuit configuration.

The total mass of ore inside the mill is obviously a function of the hold-ups of particles of all rock types and all sizes. The power is a function of the hold-ups of water, ore and grinding media. A number of model equations are available in the literature for predicting the power draw, such as the one developed for SAG mills given in Eq. (13) (Austin, 1990):

$$\begin{aligned} P = {} & 10.6D^{2.5}L(1 - 1.03J)[(1 - \varepsilon_c)(\hat{\rho}_r/w_c)J \\ & + 0.6J_B(\rho_B - \hat{\rho}_r/w_c)](\phi_c)\left(1 - \frac{0.1}{2^{9-10\phi_c}}\right) \end{aligned} \tag{13}$$

where J and J_B are the static fractional volumetric filling of the mill for the total charge and for the balls, respectively. The effective porosity of the charge is given by ε_c and w_c is the weight of the ore expressed as a fraction of the total mass of ore and water in the mill. The density of the balls and average density of the ore (t/m^3) are given by ρ_B and $\hat{\rho}_r$, respectively, and φ_c is the mill speed expressed as a fraction of the critical speed.

The Mintek version of the cumulative rates model described above caters for the effects of different aspect ratios (D/L) when scaling up pilot data by treating the production mill as a number of fully mixed reactors in series with size-independent back-mixing between adjacent reactors.

It can be shown from the population balance equations for the solids given above (Eqs. (7), (10), and (11)) that the cumulative specific breakage rate function can be obtained directly from plant data. By definition, there are no particles coarser than x_1, so $K^E_{1,k}$ is undefined. The specific breakage rates for particles coarser than size x_2 (particles in size class 1 only) can be obtained at steady-state by equating the derivative in Eq. (7) to zero and rearranging terms:

$$K^E_{2,k} = \frac{f_{1,k} - p_{1,k}}{w_{1,k}P/M} \tag{14}$$

After equating the derivatives to zero in Eq. (10), values for the cumulative breakage rate function can be calculated recursively for $i = 2, 3, \ldots, (n-1)$:

$$K^E_{i+1,k} = \frac{f_{i,k} - p_{i,k} + W_{i,k}(P/M)K^E_{i,k}}{(w_{i,k} + W_{i,k})P/M} \tag{15}$$

From Eq. (11) (for $i = n$):

$$K^E_{n,k} = -\frac{f_{n,k} - p_{n,k}}{W_{n,k}P/M} \tag{16}$$

In a pilot mill, all the terms on the right-hand side of the above equations are directly measurable.

Fig. 2 shows the result of replicate measurements of the cumulative breakage rate function for a pilot SAG mill operating in closed circuit with a vibrating screen. This function can be approximated by the following equation:

$$K^E_{i,k} = \frac{\kappa_{1,k}\hat{x}_i^{\alpha_{1,k}}}{1 + (\hat{x}_i/\mu_k)^{\lambda_k}} + \kappa_{2,k}\hat{x}_i^{\alpha_{2,k}} \tag{17}$$

where $\kappa_{1,k}$, $\kappa_{2,k}$, $\alpha_{1,k}$, $\alpha_{2,k}$, μ_k, and λ_k are model parameters. An attractive feature of the Mintek pilot SAG mill is that it is equipped with an advanced control system based on proprietary software that can be used to help facilitate the estimation of the relationship between the parameters given in Eq. (17) and operating conditions. It is interesting to note that for open circuit SAG milling the rock charge can become unstable with throughput increasing with decreasing rock charge or power draw. In spite of the simple structure of the Mintek SAG model, it has provided useful insight into system dynamics.

Fig. 3 shows the measured specific discharge rate function for the SAG mill, which is identical to that used in the standard model.

2.3. A cumulative rates model for a HPGR

It has been found that the product size distribution from an HPGR can be related to the feed size distribution $F_{i,k}$ and specific energy input ξ using the following equation:

$$P_{i,k} = 1 - (1 - F_{i,k})\exp\left(-\frac{\xi^{1-\tau}}{1-\tau}K^E_{i,k}\right) \tag{18}$$

where $K^E_{i,k}$ is the energy-normalised cumulative breakage rate function. This function and the energy dissipation parameter can be obtained directly from test data. It has been found that the energy-normalised breakage distribution function can be approximated by a polynomial function given by:

$$K^E_{i,k} = \kappa_k[\exp(a_k\ln(x_i) + b_k(\ln(x_i))^2 + c_k(\ln(x_i))^3 + \cdots)] \tag{19}$$

In practice, a reasonable fit to the breakage distribution function can be achieved using a polynomial of order three or less. Fig. 4 shows the relations between the measured size distributions of products obtained in a Polysius Labwal HPGR with a 250 mm diameter by 100 mm long rolls.

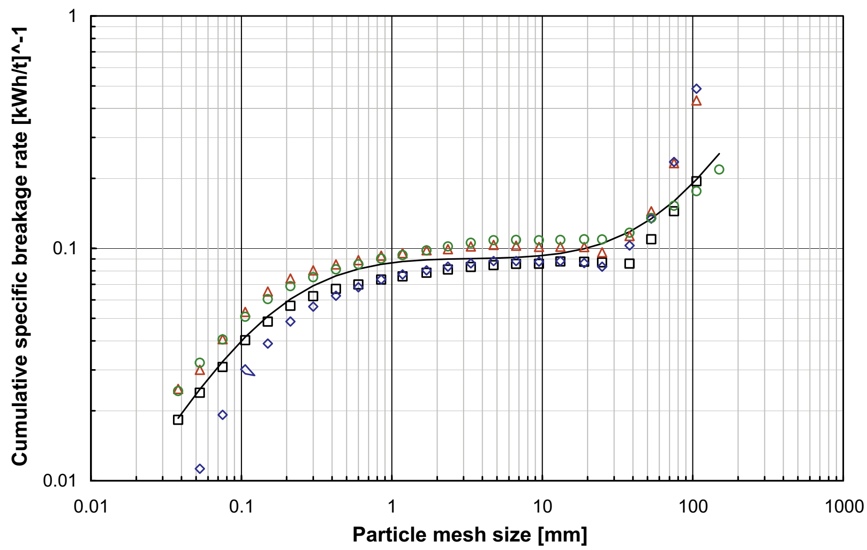

Fig. 2. Cumulative energy-normalised breakage rate function for SAG mill.

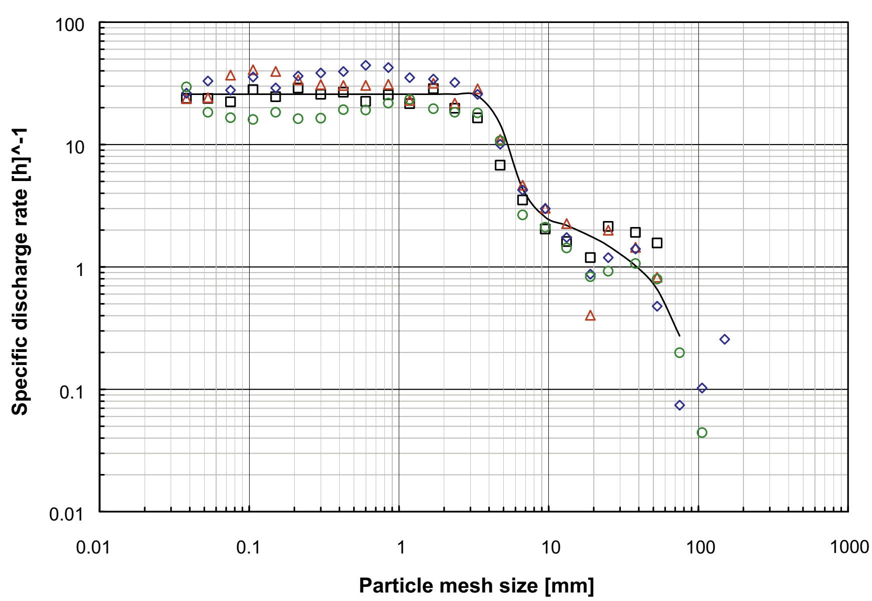

Fig. 3. Specific discharge rates for pilot SAG mill with pebble ports.

3. The application of simplified models to the comminution of platinum ores

Mintek has been actively involved in the development of process flowsheets for the platinum industry for more than two decades. In the 1980's, the major source of platinum was from the Merensky reef on the Western limb of the Bushveld Complex. At that time, there was a clear vision that there would be an increased demand for platinum to support the expected growth in the autocatalyst industry. It was considered unlikely that the demand could be met by the depleting Merensky ore reserves and the decision was taken to exploit UG2 ores, which are very high in chromite content. This ore type is currently the major source of South Africa's platinum ore reserves. At the time, conventional methods for concentrating and smelting UG2 ores were problematical due to the very fine grain size of the platinum-bearing minerals and the high chromite content of the ore. Mintek, in collaboration with leading platinum producers, subsequently developed the "Mintek Process" flowsheet (Deeplaul and Bryson, 2004) involving two stages of milling and flotation. The mill-float–mill-float (MF2) concept is now widely adopted by the South African platinum mining industry. Over the last decade, most of the new platinum concentrators have been designed and built on the basis of laboratory and piloting tests conducted at Mintek. There is little doubt that the simple model structures described in this paper have provided a useful role in the development of innovative solutions to some of the problems that have been encountered, especially with regard to UG2 ores. A couple of examples of problems that were resolved are presented below for primary and secondary milling circuits.

3.1. De-bottlenecking a primary ROM mill

The Mintek cumulative rates model for AG/SAG milling was first exploited in the late 1990s (Hinde and Pearson, 2001) to increase the capacity of Impala Platinum's UG2 plant. In 1997, Mintek was contracted by Impala to do laboratory and piloting tests aimed

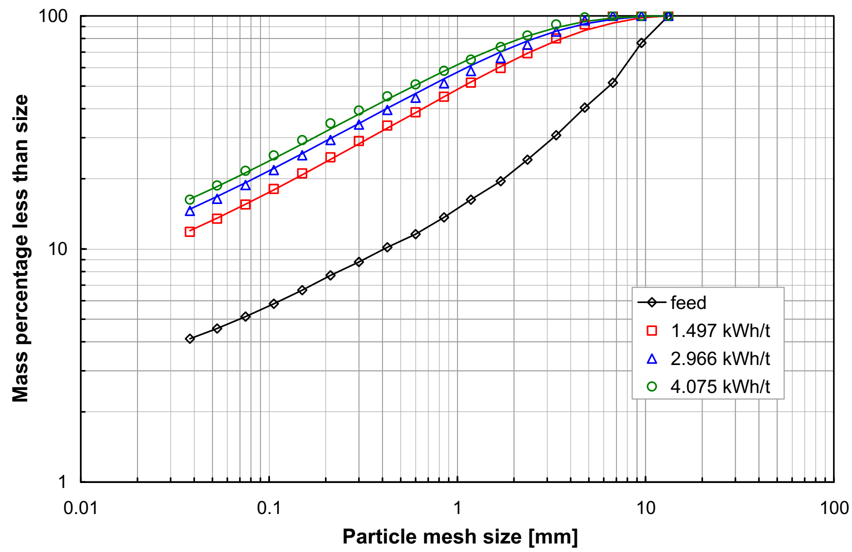

Fig. 4. Fit of simple model to tests conducted with a laboratory HPGR.

at identifying ways of increasing the capacity of the plant by more than 30%, without purchasing new grinding mills. The plant comprised primary run-of-mine ball mills operating in closed circuit with hydrocyclones and secondary ball mills, also operating in closed circuit with hydrocyclones.

In order to formulate a solution to the problem, it was first necessary to understand the geology and mineralogy of the UG2 ore mined at Impala. The UG2 reef exists in the form of a main band or seam of chromite (typically 600–700 mm in width). Above the main seam is a parting of pyroxenite and above that, thin leaders of chromitite. Below the main seam is a band of pegmatoidal pyroxenite. Although the leaders can carry PGM, they are not normally mined at Impala. Care is taken to avoid breaking into the hanging wall pyroxenite to maintain safe working conditions and to avoid diluting the ore with waste. The pegmatoid can carry small but economically extractable PGM. The pegmatoid and some barren footwall anorthosite must be mined to gain access to the working face.

The main chromitite seam is made up of chromite grains up to a few hundred microns in size bonded to each other in a silicate-rich interstitial matrix. A large portion of the platinum group metals minerals (PGM's) are associated with base metal sulphides in this matrix. Much of the chromitite is very friable and can easily be disaggregated at low energy inputs to liberate the chromite grains and some of the interstitial sulphides. The PGM minerals are often only a few microns in size and their recovery is best achieved by milling and floating in two stages. Because of the presence of geological discontinuities and the secondary alteration of many minerals the UG2 reef at Impala, is very difficult to mine selectively and to treat metallurgically. The task is complicated further by the fact that both Merensky and UG2 ores are mined at the same time. This leads to logistical problems for hoisting and transporting the two reef-types separately. Consequently, some cross-tramming and contamination of UG2 ore with Merensky ore is unavoidable.

Because of the large amounts of waste present in the ore, it was thought that pre-concentration of the ore prior to milling might be a viable option using dense media separation techniques. However, it quickly became evident that the PGM grade of the discard fraction would be unacceptably high, probably because of the presence of the Merensky ore with densities similar to that of the UG2 waste. Nonetheless, it turned out from laboratory tests that higher overall PGM recoveries could be achieved by treating the DMS floats and sinks separately. It was then argued that the separation of the ore into Merensky-rich and UG2-rich fractions could be exploited to good effect if a simple way could be found to achieve this separation. The approach that was adopted was to see if advantage could be taken of the differences in the friability of the different rock types by milling the ore fully autogenously in open circuit.

As a result of piloting tests and computer simulation studies using the cumulative rates model described above, it was found that the target throughputs for the primary mills could be achieved by installing pebble ports to reject hard critical-size material that would otherwise build-up in the mill and limit throughput. Because the secondary mills were grossly underutilized, it was possible to de-bottleneck the complete grinding circuit. An important outcome of the pilot studies was that the increase in new feed rate with pebble ports doubled that achievable without pebble ports. However, the flowrate of pebbles was only 15% of the new feed rate. In other words, it made much more sense to crush the discharged pebbles rather than to crush a portion of the ROM feed prior to milling.

A crushing plant was installed to crush the pebbles to −12 mm before being fed to the low-grade secondary mill. The discharge pulp from the primary mills was screened at about 450 μm with the undersize going to the high-grade secondary mill.

Further pilot milling and flotation tests were conducted to establish design parameters for the upgrading of the complete circuit. Simulation studies using the cumulative rates ball mill model were also conducted to establish the grind sizes and PGM recoveries that could be expected. The target throughputs for the production plant were subsequently achieved along with a significant increase in PGE recoveries, without sacrificing concentrate grades.

3.2. Optimising secondary ball mill to maximise flotation recoveries

One of the lessons learnt from the Impala study was a quantitative understanding of the deportment of the chromite and silicates in the secondary mills treating UG2 ores. These mills were operated in closed circuit with hydrocyclones. From sampling campaigns at Impala's UG2 plant and other concentrators, it was clear that after primary milling and flotation, the chromite was effectively liberated and virtually barren of PGM. However, most of the PGM's were associated and locked in the silicate minerals.

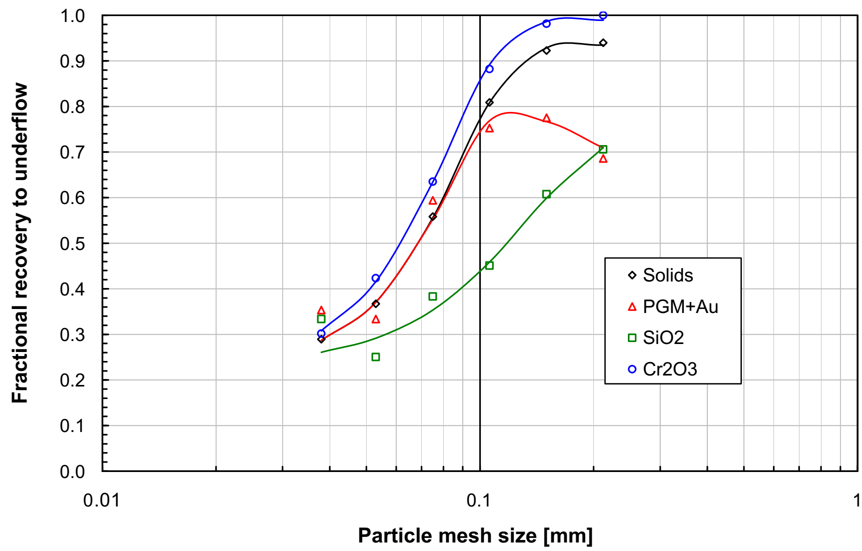

Fig. 5. Partition plots for classification in hydrocyclone.

Because the liberated chromite has a much higher density than the silicates, it tends to report preferentially to the hydrocyclone underflow. The lighter silicates containing the locked PGM tend to report to the hydrocyclone overflow stream and bypass the milling process. Fig. 5 shows measured estimates of a hydrocyclone partition function for silicates, chromite, PGM and total solids. It can be seen that above about 100 μm the partition plot for the PGM starts to decrease rapidly with increasing particle size and asymptote to values approximating the partition function for the silicates. This is to be expected if the PGM are locked in the silicates at the coarser sizes.

Fig. 6 shows estimates of the energy-normalised cumulative rates of breakage for the different assay types based on size distribution and assay measurements of the mill inlet and discharge streams. This information can be used to simulate different circuit configuration scenarios. In order to maximise PGM flotation recoveries, it is obviously desirable to achieve a fine grind for the silicates and to minimize the energy expended milling the barren chromite. Simulations showed that a finer silicate grind could be achieved by milling the ore in open circuit, rather than in closed circuit with a hydrocyclone (Fig. 7). Open circuit milling also offers the benefit of a much coarser grind for the chromite (Fig. 8), which is desirable because chromite slimes can easily become entrained with the flotation concentrate.

The results of the plant sampling campaigns and simulation studies have led to several plants changing from closed circuit milling to open circuit with hydrocyclones being used as water–solids separators to optimize mill feed pulp densities to the mill. The benefits of open circuit milling for UG2 ores depend on the amount of waste rock present. For mines that are able to minimize the amount of waste dilution, the benefits are significant with increases in PGM recovery amounting to several percentage points.

It was very clear from the simulation studies that there is considerable benefit to be gained by milling in closed circuit with fine-mesh screens. Over the years, many attempts have been made to replace hydrocyclones with vibrating screens on platinum mines treating UG2 ores. Although the potential energy-saving benefits have been demonstrated, available technology had not proven to

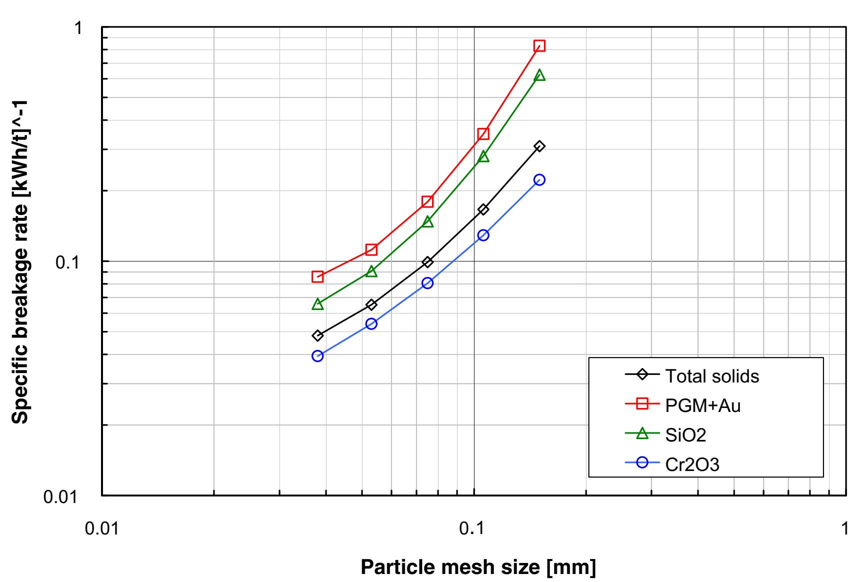

Fig. 6. Energy-based cumulative breakage rates.

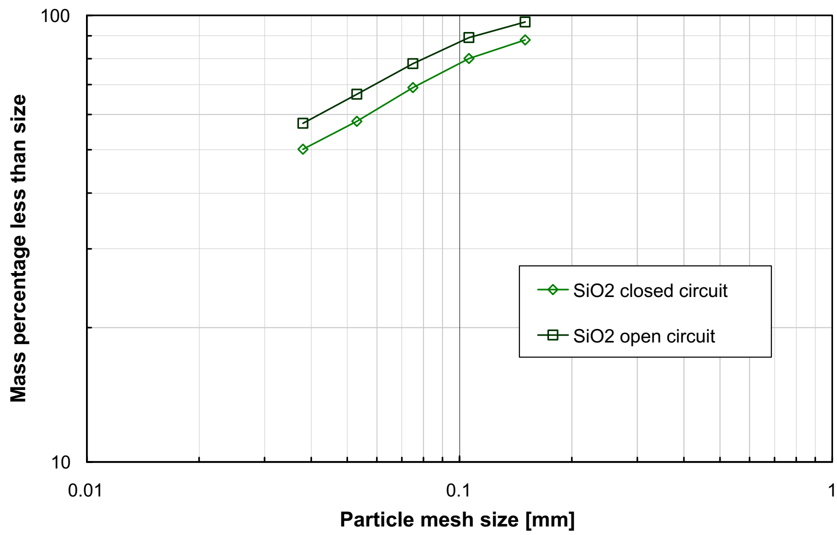

Fig. 7. Effect of open circuit and closed circuit milling on silicate grind.

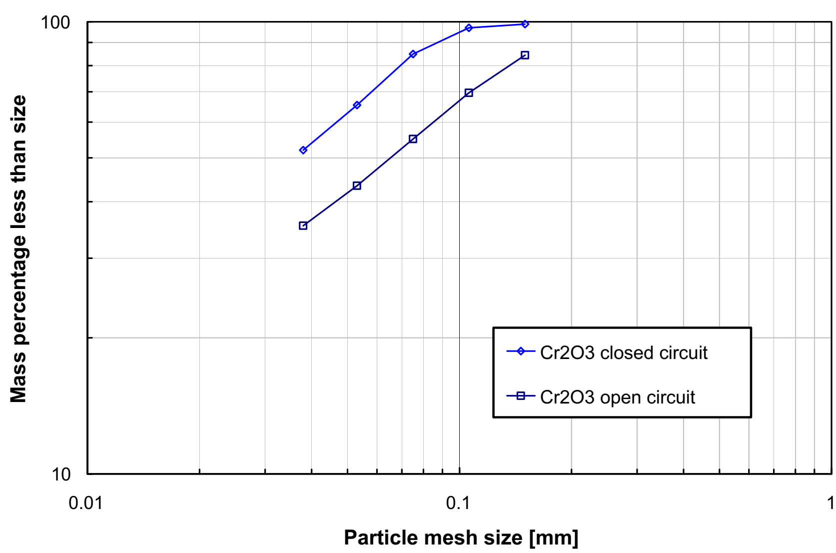

Fig. 8. Effect of open circuit and closed circuit milling on chromite grind.

be practical. Major problems encountered relate to mechanical robustness, mesh blinding and high wear rates of the steel mesh. A significant breakthrough in fine screening has been the development of the StackSizer by the Derrick Corporation in the USA. Apart from its small footprint, the novel feature of this screen is the use of urethane screening panels that are non-blinding, have high open areas, and a life span more than an order of magnitude longer than that achievable with conventional wire-mesh screens. Mintek has recently conducted tests using a Derrick test rig to establish partition function parameters for UG2 applications. Techno-economic studies have shown that the benefits of closing secondary ball milling circuits with vibrating screens at mesh sizes of about 100 μm are considerable for many mines.

4. Concluding comments

This paper has endeavoured to show that comminution models which use an energy-normalised cumulative breakage rate function to describe breakage kinetics have wide application. These *cumulative rates* models, with parameters that can be derived directly from routine pilot or production scale tests, can easily be integrated with models for flotation and physical separation processes. This can lead to the development of innovative flowsheets resulting in major savings in energy and downstream metallurgical benefits.

The simplified models, however, do have some flaws. One of them is that the cumulative rates of breakage of ore above a given size are assumed to be unaffected by the detailed structure of its size distribution above this size and the grinding environment. It turns out that the parameters of the cumulative breakage rate function can depend on the grinding environment. In the case of SAG mills, the cumulative breakage rates can be significantly affected by the ratio of the hold-ups of the ore and grinding media. It follows that the size dependency of the breakage rate function for an AG mill is very different from that of a run-of-mine ball mill. However, the same complications arise for the more sophisticated models where the parameters of both the breakage rate and breakage distribution function are affected by the grinding environment. Nevertheless, the "simple model" technique is so easy to apply that it may be possible to generate the required set of empirical relations by applying the method to all sets of pilot-scale data that have the required information. This is a very valid approach that may turn out to be the quickest way to describe the operation of these complex mills accurately. But this does not obviate the need for the long-term development of more fundamentally correct models, especially those that allow for the effects of liberation (Powell and Morrison, 2007).

To conclude, it is perhaps pertinent to quote Occam's razor: *The explanation of any phenomenon should make as few assumptions as possible, eliminating or "shaving off" those that make no difference in the observable predictions of the explanatory hypothesis or theory.*

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