Robust HPGR model calibration using genetic

Robust HPGR model calibration using genetic algorithms

abstract

Mathematical modeling and simulation techniques are widely used to design and optimize comminution circuits in mineral processing plants. However, circuit performance predictions are prone to errors due to inaccurate calibration of models used in simulations. To address this problem, the authors applied a method based on genetic algorithms (GA) for estimation of HPGR (high pressure grinding rolls) model parameters. In this research, a simulation algorithm was developed and implemented in MATLAB™ based on published HPGR models to test and demonstrate GA application for model calibration. The GA toolbox of MATLAB was used to obtain the optimal values of HPGR model parameters. The authors successfully validated simulator predictions against HPGR data sets at laboratory and industrial scales. The results indicate that GA is a robust and powerful search method to find the best values of HPGR model parameters that lead to more reliable simulation predictions.

1. Introduction

Comminution circuits have the highest level of energy consumption in mineral processing plants. Therefore, improving design and operation of comminution devices to optimize performance and energy consumption are always an important part of manufacturers and process engineers’ researches. HPGR is the result of basic changes in roller crushers due to Schönert (1986). Subsequently, the comminution mechanism was changed in the new crusher due to its high pressures (Schönert, 1979,1986). The high throughput of HPGR units and their low specific energy consumption made them increasingly suitable for use in comminution circuits.

With respect to modeling of high pressure grinding rolls, the most considerable works were done by Morrell and co-workers (Morrell et al., 1997; Daniel and Morrell, 2004). Recently, Torres and Casali (2009), following the work done by Morrell and Daniel, developed a new method for modeling of HPGR. In the model presented by Daniel and Morrell, outputs of drop-weight apparatus are used for ore characterization and their model is fitted to laboratory data through limiting the number of model parameters. In next step, the obtained model parameters are kept constant and used in a scale-up procedure to predict the performance of fullscale units. But in Torres and Casali new method, established functional expressions of breakage and selection functions are used for this purpose and also there is no scale-up procedure involved.

In HPGR units, comminution mechanism basically differs from that of media mills due to the high pressure applied on particles. For this reason, the breakage and selection functions obtained using media mills such as a ball mill or a rod mill and other devices cannot be used for simulation of HPGR units. As a solution, these parameters can be estimated simultaneously by fitting the measured data with corresponding models. Considering a normalizable breakage function and only the first two terms of the logarithmic polynomial expression used to define selection function, there will be totally six parameters which must be back calculated from actual HPGR data sets in order to determine breakage function and selection functions.

A programming environment equipped with powerful search tools for function optimization is a prerequisite for model fitting purposes. In a previous paper, successful application of genetic algorithm search method to optimize comminution circuits was reported by Farzanegan and Vahidipour (2009). Therefore, MATLAB environment and also its genetic algorithms toolbox were selected for implementation of HPGR simulation model, optimal calibration of model parameters and finally prediction of HPGR product size distribution in an open circuit.

2. Description of HPGR model

Authors programmed the HPGR mathematical model explained by Torres and Casali (2009) as the main part of simulator structure.

This model includes a set of equations that is based on ore characteristics, equipment dimensions and operating conditions. Given the required input, the model is able to predict throughput, power consumption and particle size distribution of HPGR product.

The equations are based on physical phenomena that control the operation: mass balances for throughput estimation, physics equations for the power consumption and population balance model for prediction of particle size distribution of product.

In this study, authors used particle size distribution sub-model to predict HPGR product size distribution under various operating conditions, because product size distribution is an important result of modeling and optimization researches.

2.1. Particle size distribution model

In models presented by Daniel and Morrell (2004) and Torres and Casali (2009), comminution between rolls includes two stages: single-particle comminution (pre-crushing) and particle-bed comminution (Fig. 1).

The particles coarser than a definite size are broken in single-particle compression zone in which comminution is done directly by rolls surfaces, as would occur in conventional roller crushers. The definite size is equal to the length of the boundary line between the two comminution zones and is called critical distance, xc, (Klymowsky et al., 2002). In HPGR units, the main process of particle breakage takes place in the particle-bed compression zone which has been depicted in Fig. 1.

The distance between rolls surfaces, s(a), can be calculated geometrically as a function of angle a using the following equation:

equation 1

where s0 is the least distance between rolls (rolls gap) and D is roll diameter. The value of xc can be obtained using the following equation:

equation 2

where aIP is the angle that refers to the beginning of inter-particle compression zone. The theoretical throughput, Gs (t/h), treated by an HPGR can be calculated using mass conservation law. Eq. (3) is used for calculation of throughput as a function of angle a(Torres and Casali, 2009):

equation 3

where q(a) is ore band density at angle a (t/m3), L is roll length (m) and U is peripheral velocity (m/s). Under steady-state conditions, flow rates (t/h) of the beginning and the end of compression zone are equal. In addition, the flow rate is equal to throughput. Therefore, using Eqs. (1) and (3) and also considering that q(aIP) = qa and q(a = 0) = d, Eq. (4) is obtained in which the unknown value is inter-particle compression angle (aIP):

equation 4

where d is ore band density in extrusion zone and qa is feed bulk density. The solution to Eq. (4) results in two roots values that only the higher one is the acceptable value for the angle (the smaller root gives too high angles). The acceptable value for cosaIP can be calculated by the following equation:

equation 5

The particles coarser than xc are comminuted in single-particle compression zone and the product size distribution of this zone is calculated by the following equation (Torres and Casali, 2009):

equation 6

where bil is non-cumulative breakage function, f SP l is the mass fraction in size class l entering the single-particle compression zone (class of size x > xc) and N is the number of size classes coarser than xc.

The final product of HPGR is composed of two different particle size distributions (Klymowsky et al., 2002; Daniel and Morrell, 2004; Patzelt et al., 2006). Lubjuhn (1992) explained this phenomenon when he found that the pressure profile exerted over the rolls is similar to a parabola, as it is shown in Fig. 2.

Following this approach, the force depending roll where the pressure is applied was divided into NB blocks. In each one, a different compression on the pressure profile is applied. The product of single-particle compression zone rejoins with the materials that are equal or lesser than xc and will be comminuted in particle bed compression zone. The product of particle-bed compression zone is the final product and is calculated using the following equation:

equation 7 y 8

where Pi,k is remained mass on ith screen on block k, vz is the downward velocity of flake, f IP i is the fraction in size class of the mineral going to particle-bed compression zone, z⁄ is the vertical distance from beginning of particle compression bed zone to extrusion zone.

Also Sj,k is the selection function value of size class i in block k. The parameter z⁄ is calculated geometrically as follows:

equation 9

For the cumulative breakage function elements, Bi, the following functional expression (Eq. (10)) is used (Austin and Luckie, 1972):

equation 10

where a1, a2, and a3 are model parameters. For calculation of noncumulative breakage function elements, bi,j, Eq. (11) can be used:

equation 11

To represent selection function, Herbst and Fuerstenau (1980) have proposed the following logarithmic functional expression:

equation 12

where f1, f2 and SE 1 are model parameters that should be calculated.

To obtain selection function value for each block, Si,k, Eq. (13) (Herbst and Fuerstenau, 1980) is used, assuming a parabola pressure profile exerted over the rolls:

equation 13

where Hk is the hold up of each block and Pk is power consumption of block k that changes by each block. The hold up of each block k is calculated using the following equation:

equation 14

where Gs is throughput (t/h) and can be calculated in extrusion zone using the following equation:

equation 15

According to pressure profile exerted over the rolls (Lubjuhn, 1992), the power consumption of each block k is calculated using the following equation (Torres and Casali, 2009):

equation 16

where yk (m) is center position of block k which is geometrically calculated using the following equation (see Fig. 2):

equation 17

Moreover, F is compression force (kN) and is presented in the following equation (Torres and Casali, 2009):

equation 18

where Rp is roll pressure (bar).

Finally for calculation of particle size distribution of total product, PHPGR i , Eq. (19) is used:

equation 19

where PHPGR is the remained mass on ith screen and NB is the number of blocks.

The application of a simplified approach to modelling tumbling mills,

stirred media mills and HPGR’s

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

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 xi, where the subscript i serves as an index to reference each size class. The first or largest size, x1, 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 xn. 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, Si,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 SEi ;k ¼ Si;kM=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, bi,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 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, gi,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:

equation 1

where wi,k is the hold-up mass of size class i and rock type k particles, (t); fi,k is the mill inlet flow rate of size class i and rock type k particles, (t/h); t is time, (h); SEi; k is the specific breakage rate for size class i and rock type k (kWh/t)1; gi,k is the specific discharge rate for size class i and rock type k (h)1; bi,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 wi,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 n (kwh/t). If w0

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:

equation 2

where n0 = n1s/(1 s) and the exponent 0 6 s 6 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.

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 Fi,k and specific energy input n using the following equation:

eqation 18

where KEi;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 energynormalised breakage distribution function can be approximated by a polynomial function given by:

equation 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.

Chapter 108

Study on the Crush Model

of High-Pressure Grinding Rolls

Study on the Crush Model of High-Pressure Grinding Rolls

Abstract

A high-pressure grinding roll is energy efficient crush equipment.

On the basis of mass balance, energy conservation and overall balance, this paper puts forward the intersected idea due to the uneven stress in the roller surface.

The grinding and comminution model of roller press is deeply investigated. It puts forward crush model using mathematic theory. The simulated specific energy consumptions and particle size distributions, compared with the experiment data, were considered good enough. The model was able to predict adequately throughput capacity, specific energy consumption and particle size distributions of the edge, center and total products.

108.1 Introduction

The first commercial application of high-pressure grinding roll (HPGR) was in 1985 and its success resulted in increasing numbers of applications since then. The classical theory is bed compression crush theory. It can be described: mineral particle is crushed under lots of bed compression condition. Every crush machine cannot reach the ideal bed compression condition because of different crush condition. Reference [1] studies the effect law which load—displacement curves, bed height and load characteristic direct crush on the basis of bed compression grinding theory. Reference [2] gets a new concept of the differential shear theory, which based on the study of process and result of the high-pressure double roller cracker for brittleness ore crush. Most of the researchers follow the crush of a single particle theory of crack propagation and energy balance principle and the experimental test data for various amendments; there is no good explanation for formation mechanism of the cake under the high-pressure crush.

108.2 Modeling

HPGR rely on pure compressive stress to squeeze broken. The breakage of ore occurred in a short time. Its energy consumption is mainly reflected by the amount of the pressure. This paper puts forward the crush model of HPGR on the basis of ore characteristics, equipment dimensions and operating conditions. The model structure is developed by Daniel and Morrell [3] on the basis of conservation of mass, conservation of energy and overall balance. So it can predict throughput, power consumption and particle size distribution of the product. Figure 108.1 is crush schematic diagram of HPGR.

108.2.1 Throughput Model

The center of mass of a band of ore with width s(a) is defined by the position of vector r(a), diameter D(m), length F(m) and operating gap s0(m). So:

equation 108.1

The density of the ore band at any angle a is r(a). At the beginning of the particle bed compression zone, r(a) is equivalent to the feed bulk density r(a). The tonnage of the ore band, Gs (t/h), with r(a) expressed in t/m3, is written as a function of the angle a in Eq.

equation 108.2

The width of the ore band as a function of the angle a can be expressed as:

108.3

Under steady-state condition, if the amount of material loss is zero, i.e. the feed is equal to the product; we can get the next Eq.

108.4

We can find Eq. (108.4) is a quadratic equation of cos aIP. Because D ¼ b2 4ac[0; the equation has two roots. We can get Eq. (108.5) according to actual situation.

108.5

So Eq. (108.2) can estimate throughput in any position. Specifically at the extrusion zone (a = 0), the throughput can be calculated as:

108.6

108.2.2 Power Draw Model

The force applied to the material at the particle bed compression zone is called the compression force, F (kN). The rolls operating pressure is RP (bar). The total power draw is P (kW). Since the HPGR is operated in a choke fed condition, the applied pressure is distributed only in the upper right half of the roll. Then the projected area considered should be D/2\*L

108.7

Then torque t (kN/m) can be written as:

108.8

As the power required to spin both rolls is equal to twice the torque multiplied by the rolls angular velocity, then P can be written as:

108.9

The energy consumption W(kWh/t)is written as the ratio between the power draw (kW) and the throughput (t/h):

108.10

108.2.3 Particle Size Distribution Model

The HPGR is considered as a series of two size reduction stages. The first stage is single particle compression. The next stage is particle bed compression.

The single particle compression zone is located between the angle aSP and aIP. Particles larger than a certain size Cc are broken. The critical size Cc is got by replacing the inter particle compression angle aIP in Eq. (108.3).

108.1

The product of the single particle compression zone rejoins with the fraction of material of material of size lesser than or equal to the critical size Cc, forming beds of particles with a particle size distribution. Then it starts particle bed compression. According to previous sum up, the particle bed compression is broken into two areas, namely the central area and edge zone. It is also known as edge effects.

The pressure profile exerted over the rolls is similar to a parabola, as it is shown in Fig. 108.2. In Fig. 108.2, rolls are divided into NB equal portions. Because every NB equal portion is different, every power of NB equal portions is different and every crush rate of NB equal portions is different. The intensive property mik (mass fraction retained by weight in size class i, in each block k) is a function of the vertical position. According to mass balance principle, the model equation consists in a system of N0 NB differential equations, each one for the size class i ði ¼ 1; . . .; NÞ in each block k ðk ¼ 1; . . .; NBÞ. As shown in Eq. (108.12)

108.12

where Sjk is the rate of breakage of particles of size i in each block k; bij is the fraction of particles of size ‘‘i’’. To solve these equations the following border conditions are used:

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where pi,k is the mass fraction retained by weight in size class i, in the product of each block k. Z\* is the vertical distance from the entrance to the particle bed compression zone to the extrusion zone.

108.13

Equation (108.12) seems to be the batch grinding kinetic equation, which has been solved analytically by Reid [5]. The solution is written (Fig. 108.2):

108.14

108.15

In mathematics, [E] is defined as the largest integer less than or equal to E. [E] is defined as the smallest integer not less than E. So the particle size distribution of the edges product pEi can be calculated as:

108.16

The particle size distribution of the total product pHPGR i is :

108.17

The particle size distribution of the center product is:

108.18

EXPERIMENT

To test the HPGR model, we used a HPGR manufacturing company, Changsha. Its diameter is 600 mm and length is 200 mm. The test uses two kinds of ore. Data are shown in Table 108.1.

GRAFICO

These results show that for different operating pressures RP, different product size distributions are obtained, as can be observed from the comparison of A1 versus A6 and B1 versus B5, under the same ore and different RP. It is also evident that if the ore is different but the operating pressure is the same, different product size distributions are obtained. However, the effect is very is small, especially with a higher RP, as can be observed from the comparison of A6 versus B5. With a lower RP the differences are more evident (A1 vs. B1) (Fig. 108.3).

108.4 Conclusions

In bed compression crush, the crush of bed affects every particle crush. Model reflects the particle breakage and crush energy absorption material obey the basic principles. The smaller particle size, the greater power consumed. In crush study, equal portions, NB, of the roller can make throughput, power draw and the particle size more accurate. It is also evident that if the ore is different but the operating pressure is the same, different product size distributions are obtained. However, the effect is very is small, especially with a higher RP. With a RP the differences are more evident.