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Advanced Simulation for Semi-Autogenous Mill Systems: A Simplified Models Approach

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1. Introduction

Modelling and simulation of semi-autogenous (SAG) mills are valuable tools for helping to design control laws for a given application and subsequently to optimise its performance and process control. SAG mills (see Figure 1) are presently one of the most widely used alternatives in the field of mineral size reduction as a result of their advantages such as higher processing capacity, lower physical space requirements, and lower investment and maintenance costs, as compared to conventional circuits (Salazar, et al., 2009).

Due to the size of SAG mills, pilot plants are usually used for research purposes to improve the control strategies. In cases where a pilot-scale is not available for test, simulations using models based on data from a wide range of full-scale plants are helpful and can significantly reduce risks for process control purposes. Simulations also provide an additional and very valuable crosscheck against the pilot results (Morell, 2004).

Fig. 1. Typical semi-autogenous (SAG) mills

This chapter presents a dynamic simulator of a semi-autogenous grinding operation deduced from first principles coupled to an on-line parameter estimation scheme able to simulate industrial operations for future control purposes. The proposed procedure for simulation purposes is as follows: Model equations are based on a conventional non-stationary population balance approach to develop the necessary dynamic model of the semi-autogenous mill operation. The presented models are able to predict the time-evolution of key operating variables such as product flow rate, level charge, power-draw,
load position and others, as functions of other important variables such as mill rotational speed and fresh feed characteristics. The set of ordinary differential equations was solved using MATLAB/SIMULINK as a graphic programming platform, a useful tool for understanding the grinding process.

Additionally, this work presents results using dynamic simulations from a 1700 t/h copper-ore mill showing the effectiveness of the system to track the dynamic behaviour of the variables.

The remainder of this chapter is organised as follows. Theory about specific models for SAG mill processes is presented in section 2. Simulations for the prescribed application are presented in section including the results using MATLAB/SIMULINK. The main conclusions of the chapter are provided in the final section, as well as ideas about future industrial applications of this work.

2. Models for semi-autogenous mills

Essentially, the modelling exercise consists in formulating non steady-state material balances in the milling equipment, along with force conservation relations and hydraulic considerations. The methodology used in this study has already been established by Magne (Magne et al., 1995) and Morrell (Morrell, 2004) and involves formulating particle inventories for each particle size inside the mill. The input variables are: water flow rate, mineral flow rate and size distribution, grinding media flow rate and the mill critical speed. The model output variables are: power-draw, load level, ball load, mineral discharge rate and size distribution, water discharge rate, ball throughput, bearing pressure, pebble throughput, and toe and shoulder angles of the internal load.

2.1 SAG mill model

The particles fed to the mill are ground in the milling chamber and subsequently downloaded into the discharge zone, where, according to a classification probability, they are either returned to the milling chamber for further grinding or become part of the mill output stream. For modelling purposes the mill is divided in two zones according to the process taking place (Fig. 2). The first zone encompasses the milling chamber where the particle reduction process is identified and modelled. In the second, the output zone, the material is internally classified and the final product is discharged. To complete the system

Fig. 2. Schematic representation of a SAG mill. (1) Mill. (2) Grinding Chamber. (3) Internal classifier
description it is necessary to consider the relationship between the feed stream and mill charge level. This relationship is known as the transport rate and is probably the least developed aspect in models proposed so far (Apelt et al., 2002 a,b).

### 2.2 Transport and water balance

The fictitious flow $P^*$ (Fig. 2.) that represents the amount of mineral in the internal charge that is handled by the classification grate or internal classification, is the representation of the mineral transport proposed by Magne (Magne et al., 1995). Several experimental studies have found the following rather unsatisfactory correlation of $P^*$ with the mass of the mineral retained in the mill $W$:

$$P^* = 29 \cdot W^{0.5}$$

(1)

Where $W$ is in tonnes (t) and $P^*$ in t/h.

### 2.3 Water balance

The following equation represents the experimental variation of the internal water load, $W_w(t)$, as a result of changes in input and output water flow rates, $F_w$ and $P_w$ (t/h), the latter being estimated by $P_w = C_w \cdot W_w$ (Magne et al., 1995):

$$\frac{dW_w}{dt} = F_a - C_w \cdot W_w$$

(2)

The parameter $C_w$ (h$^{-1}$), water output, has been correlated to the mass of mineral in the mill, $W$, according to the following relation (Magne et al., 1995):

$$C_w = \exp\left(64.41 - 19.56\ln(W) + 1.55\ln(W)^2\right)$$

(3)

The proposition that the classification system always allows particles of a size less than $X_m$ to pass (Fig. 3) is the basis for the development proposed by Morrell (Morrell, 2004), who, like Magne (Magne et al, 1995), considers that particles less than this size behave like water in the grinding chamber, i.e. all particles with less than a certain size pass through the grate with the same classification efficiency.

![Classification function against particle size](image-url)
This discharge function, constant for sizes less than $X_m$ and defined as $d_m$, is directly related to the flow of discharge of the pulp by the size of the mill, $p_i$, and the mass of the particles in the internal charge of the mill, $w_i$, according to:

$$d_m = \frac{\sum_{i} p_i}{\sum_{i} w_i}$$

(4)

In order to determine this discharge function, Morrell (Morrell, 2004) considers two effects. The first is the flow via grinding media interstices, and the second considers the flow via the slurry pool (where present). In addition, the contributions of Latchireddi (Latchireddi, 2002) have allowed this proposition to be studied in large-scale pilot models and to determine the influence of the design and the geometry of the mill pulp lifters. The results of the correlation between the fill level and discharge flow can be seen in the following general equation:

$$J = \eta \gamma^{n_1} A^{n_2} J_b^{n_3} \phi^{n_4} Q^{n_5} D^{n_6}$$

(5)

Where:
- $J$ is the net fractional slurry hold-up inside the mill;
- $A$ is the fractional open area;
- $J_b$ is the fractional grinding media volume;
- $\phi$ is the fraction of critical speed;
- $Q$ is the slurry discharge flowrate;
- $\gamma$ is the mean relative radial position of the grate holes;
- $\eta$ is the coefficient of resistance, which varied depending on whether flow was via the grinding media interstices or the slurry pool (where present); and
- $n_1-n_6$ are the models parameters.

The value of $\gamma$ is a weighted radial position, which is expressed as a fraction of the mill radius and is calculated using the formula:

$$\gamma = \frac{\sum_{i} r_i a_i}{r_m \sum_{i} a_i}$$

(6)

Where $a_i$ is the open area of all holes at a radial position $r_i$, and $r_m$ is the radius of the mill inside the liners.

Latchireddi’s (Latchireddi, 2002) contribution can be seen in the parameters $n_i$ and $\eta$ from equation (6) which shows the effect of the design of the pulp lifter. These were modeled according to:

$$n_i = n_g - k_i e^{(k_i \lambda)}$$

(7)

Where:
- $n_g$ are the parameter values for the grate-only condition;
- $\lambda$ is the depth of the pulp lifter expressed as a fraction of mill diameter; and
- $k_i$ and $k_j$ are constants.
For large-scale mills, the pulp discharge flow can be determined by combining equations (4) and (5) as follows:

\[ d_m = \frac{Q}{J} \]  

(8)

The calculation procedure can be transformed in an iterative numerical sequence.

A numerical approximation of the proposal by Gupta & Yan (Gupta & Yan, 2006) shows the product flow \((\text{m}^3/\text{h})\) from equation (5) as separate from the flow of the fluid through the zone of grinding medium (equation 9) and the flow from the pool zone (equation 10).

\[ Q_m = 6100 \gamma^{2.5} A J_H^{2.5} J_H < J_{MAX} \]

(9)

\[ Q_t = 935 \gamma^2 A J_D^{0.5} J_S = J_p, J_S > J_{MAX} \]

(10)

Where:
\(\gamma\) is the mean relative radial position of the grate apertures;
\(A\) is the total area of all apertures \((\text{m}^2)\);
\(\phi\) is the fraction of the critical speed of the mill;
\(D\) is the mill diameter \((\text{m})\);
\(Q_m\) is the volume flow rate through the grinding media zone \((\text{m}^3/\text{h})\);
\(Q_t\) is the volume flow rate of slurry through the pool zone, \((\text{m}^3/\text{h})\);
\(J_H\) is the net fraction of slurry hold-up within the interstitial spaces of the grinding media;
\(J_S\) is the net fractional volume of slurry in the slurry pool;
\(J_{MAX}\) is the maximum net fraction of slurry in the grinding zone; and
\(J_p\) is the net fraction of the mill volume occupied by pulp.

2.4 Internal classification and power-draw

For the internal classifier (Fig. 2.), the balance is carried out by defining a classification efficiency vector, \(c_i\) (fraction), which includes two effects: one produced by the mill’s internal grate and the other by the pulp evacuation system (Magne et al., 1995). Thus, \(c_i\) is defined by:

\[ 1 - c_i = \frac{P_i}{P_i^*} \]

(11)

Where:
\(P_i\) is the product flow rate from the mill and \(P_i^*\) is the product flow rate from the grinding chamber (fictitious flow).

For each size class \(i\), the mill chamber feed flow rate, \(f_i^*\) \((\text{t/h})\), is obtained by adding the mill feed flow rate, \(f_i\) \((\text{t/h})\), to the internal recirculation flow rate (Fig. 2):

\[ f_i^* = f_i + c_i P_i^* \]

(12)

Under experimental considerations (Magne et al., 1995) it is possible to find the following expression of the classification efficiency vector, where \(x_i\) is the size of particle, \(c_i\) is the solid pulp percentage and \(\beta\) is a parameter.
For each size class $i$, Magne’s (Magne et al, 1995) proposed model relates the mass variation in the milling chamber (Fig. 2) to the feed flow rate to the grinding chamber, $f_i^*$ (t/h), to the product flow rate from the grinding chamber, $p_i^*$ (t/h), and to the comminution kinetics, as follows:

$$\frac{dw_i}{dt} = f_i^* - p_i^* - K_i w_i - \left( K_i - K_{i-1} \right) \sum_{i=1}^{I-1} w_i$$  \hspace{1cm} (16)

Where $K_i$ (h$^{-1}$) denotes the effective parameter (corresponding to $S_i$ in conventional grinding) and $w_i$ is the weight of size $i$ particles in the mill charge (t).

The effective parameter, $K_i$, is defined as the fraction of specific power supplied to the mill:

$$K_i = K_i^e \frac{M_p}{W}$$  \hspace{1cm} (17)

Where $K_i^e$ is defined as the specific grinding rate constant (t/kWh), $M_p$ is the power-draw (kW) and $W$ the total ore weight in the chamber (t). The equation used to predict the power consumed by the mill (power-draw), $M_p$, is based on a modification of Bond’s Law (Austin, 1990):

$$M_p = K_p D^{2.3} L (1 - A) \left( \frac{W}{V} \right) \phi_c \left[ 1 - \frac{0.1}{2^{9.1 \ln \phi_c}} \right]$$  \hspace{1cm} (18)

Where $D$ (m) and $L$ (m) are the mill dimensions, $V$ (m$^3$) is the mill effective volume, and $K_p$ and $A$ are parameters. The ratio between the internal load mass and the mill volume, $(W/V)$, is related to the percentage of mill capacity by the following equation:

$$\frac{W}{V} = (1 - \varepsilon_b) \rho_s (1 + w_c) + 0.6 J_b (\rho_b - \rho_s (1 + w_c))$$  \hspace{1cm} (19)

Where $\varepsilon_b$ is the porosity of the mill internal load (void fraction), $\rho_s$ (t/m$^3$) and $\rho_b$ (t/m$^3$) are the density of mineral and balls respectively, $w_c$ is the mill water/mineral mass ratio, and $J_b$ (fraction) is the ball weight fraction.

Assuming that the mill chamber behaves like a perfectly mixed reactor (Whiten, 1974), $p_i^*$ can be related to particle size $i$ mill charge by:

$$p_i^* = w_i \left( \frac{p'}{W} \right)$$  \hspace{1cm} (20)
Where $P^*$ is the contribution of the total internal flow rate to the product stream (t/h). The relation between $P^*$ and $W$ can be obtained assuming that there is no recycling of fines from the internal classifier. This assumption simplifies the mass balance equation and allows the calculation of $P^*$ on the basis of the product flow rate of fine particles, $p_n$ (t/h), and the mass of fines in the internal charge, $w_n(t)$, as shown in equation (21):

$$P^* = W \left( \frac{P_n}{w_n} \right)$$

From equations (12), (16), and (21), the following expression is then obtained for the dynamic mass balance of size $i$ particles in the milling chamber:

$$\frac{dw_i}{dt} = \left( \frac{P^*}{W} \right) (1 - c_i) w_i - K_i w_i - \left( K_i - K_{i-1} \right) \sum_{t=1}^{i-1} w_i + f_i$$

As in equation (16), Morrell’s (2004) proposal for the comminution process gives a similar relationship as follows:

$$\frac{dw_i}{dt} = f_i - p_i + \sum_{j=t}^{i} r_{ij} w_j a_{ij} - r_i w_i$$

Where $r_i$ is the breakage rate of particles of size $i$, $d_i$ is the discharge rate of particles of size $i$ and $a_{ij}$ is the breakage distribution function.

The breakage rate function, $r_i$, can be obtained using data fitting techniques or full-scale mills with the general form being as follows:

$$\ln(r_i) = k_{i1} + k_{i2} D_b + k_{i3} \phi + k_{i4} I$$

Where $D_b$ is make-up ball size, $\phi$ is the mill rotational rate and $k_{i1-i4}$ are constants. The breakage distribution function, $a_{ij}$, is obtained via the specific comminution energy, $E_{cs}$ (kWh/t) and the $t_{10}$ parameters estimated, used to generate a size distribution. This equation is:

$$t_{10} = A \left( 1 - e^{-b E_{cs}} \right)$$

Where $A$ and $b$ are parameters of rock breakage.

The mill power-draw studied by Morrell (Morrell, 2004) is similar to that used by Austin (Austin, 1990) and considers the individual power requirements for the cylindrical section and the conical sections. The mill power, $P_m$ (kW), is then the sum of the net power, $P_{net}$ (kW) and the no load power, $P_{nl}$ (kW). Thus:

$$P_m = P_{net} + P_{nl}$$

$$P_{nl} = 1.68 D^{2.05} \left( \phi_{c} \left( 0.667 L_{cone} + L_{cyl} \right) \right)^{0.82}$$
\[
P_{\text{net}} = 7.98D^{2.5}L_{\text{c}}\rho J \left( \frac{5.97\phi_c - 4.34\phi_c^2 - 0.985 - J}{5.97\phi_c - 4.34\phi_c^2 - 0.985} \right) \phi_c \left( 1 - (1 - 0.954 + 0.135J) e^{-19.52(0.954 + 0.135J)} \right) (29)
\]

\[
L_{\text{d}} = L \left( 1 + 2.28(1 - J) \frac{L_{\text{cyl}}}{L} \right) (30)
\]

Where \( L_{\text{d}} \) (m) is the medium size of the final section of the conical zone, and \( L_{\text{cone}} \) and \( L_{\text{cyl}} \) are the sizes (m) of the conical and cylindrical sections of SAG mill.

### 2.5 Grinding media, bearing pressure and load position

The mass of grinding media inside the chamber is determined by a mass balance considering the ball replacement rate and the metal consumption rate; this latter parameter is proportional to the mass of mineral in the mill (Salazar et al., 2009):

\[
\frac{dW_b}{dt} = F_b - \chi (W + W_b) (31)
\]

Where \( W_b \) is the ball mass (t) in the mill, \( F_b \) the ball replacement rate (t/h), \( \chi \) a ball wear constant (h\(^{-1}\)) and \( W \) the total internal mineral load (t).

The bearing pressure, \( P_b \) (psi) is estimated as a linear function of the total weight of the milling chamber (balls, water and mineral) as shown in equation (32) (Salazar et al., 2009), where \( \alpha \) and \( \lambda \) are fitted parameters. The load position is expressed in terms of toe and shoulder angles, which are calculated by relations (33 to 35) (Apelt et al., 2001):

\[
P_b = \alpha + \lambda (W + W_w + W_b) (32)
\]

\[
\theta_t = 2.5307(1.2796 - J) \left( 1 - e^{-19.42(0.954 + 0.135J)} \right) + \frac{\pi}{2} (33)
\]

\[
\theta_s = \frac{\pi}{2} \left( \theta_t - \frac{\pi}{2} \right) \left( (0.3386 + 0.1041\phi_c) + (1.54 - 2.5673\phi_c) J \right) (34)
\]

\[
\phi = 0.35(3.364 - J) (35)
\]

Where \( \theta_t \) is the toe angle (radians), \( \theta_s \) the shoulder angle (radians).

### 3. Simulation

#### 3.1 SAG in Matlab-Simulink

The numerical solution of the set of algebraic–differential equations (model) described in the previous section, is obtained through a system in MATLAB/SIMULINK (Figure 3). Simulink is a programming system structured in blocks, which allows the solution of differential equations as well as the programming of user-blocks through S-functions. This feature, together with the possibility of using Matlab’s specific toolboxes, makes it a powerful platform for the development of prototypes. The present model can be seen as a more complex simulation block compatible with this simulation strategy in (Salazar et al., 2009).
3.2 Results
An example of the simulation results is presented in Figs. 4 to 7. These figures respectively show the response of the power-draw and the fill level for the Magne approach (Magne et al., 1995) in Figures 4 and 5, and for the Morell approach (Morrell, 2004) in Figures 6 and 7. The results are the product of 10% flow change related to the nominal operation conditions (1700 t/h).

Fig. 3. SAG mill simulator in Matlab-Simulink

Fig. 4. Magne’s model power-draw response
Fig. 5. Magne's model fill level response

Fig. 6. Morell's model power-draw response
4. Conclusion

Advanced simulation for semi-autogenous mill systems has been presented in the context of a simplified models approach that incorporated developments of (Magne et al., 1995) and Morrell (Morrell, 2004) among the others. A main focus has also been a comparison of these two models. This comparison showed that both models provided good predictive capability of two very important process variables, power draw and fill-level, especially under the same simulation conditions.

It is interesting to note that despite differences in the theoretical background for these approaches, the results of dynamic simulations under industrial operational conditions are similar. Thus, these results validate adequately the comminution process in the SAG mill, and in the future, these models could be combined for industrial purposes. With these results we believe that it is possible to scale-up from pilot plant simulation and to optimise existing circuits for process control purposes using combinations of these models to reduce risks and improve performance.

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Whiten, W.J. (1974). A matrix theory of comminution machines. *Chemical Engineering Science*, 29 (2), 589–599.
When talking about modelling it is natural to talk about simulation. Simulation is the imitation of the operation of a real-world process or systems over time. The objective is to generate a history of the model and the observation of that history helps us understand how the real-world system works, not necessarily involving the real-world into this process. A system (or process) model takes the form of a set of assumptions concerning its operation. In a model mathematical and logical assumptions are considered, and entities and their relationship are delimited. The objective of a model – and its respective simulation – is to answer a vast number of “what-if” questions. Some questions answered in this book are: What if the power distribution system does not work as expected? What if the produced ships were not able to transport all the demanded containers through the Yangtze River in China? And, what if an installed wind farm does not produce the expected amount of energy? Answering these questions without a dynamic simulation model could be extremely expensive or even impossible in some cases and this book aims to present possible solutions to these problems.

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