International Journal of Engineering and Technical Research (IJETR)
ISSN: 2321-0869 (O) 2454-4698 (P), Volume-10, Issue-2, February 2020

Adaptive Cost Optimization Model for Open Pit Mining Operations

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***Abstract*—Mining is a complex operation composed by a set of interdependent processes, with clear interfaces between different unit operations, from the pit to the concentration. In such complex environment, adaptive cost-effective optimization strategies are required to increase productivity and revenue while minimizing environmental impact. Since the overall efficiency can be strongly impacted by the initial blast energy distribution, a simplified operating method for the total cost minimization is presented based on the searching of the ideal blast design. Using the blast parameters as decision variables, this methodology allows the evaluation of several blasting scenarios and their potential impacts on subsequent unit operations.**

***Index Terms*—Blasting, Fragmentation, Mining, Unit Operations, Optimization.**

I. Introduction

One of the most important requirements in any attempt of optimizing mining operations, or parts of it, is the ability to model and measure all unit processes involved. It requires a deep knowledge of all critical operations, their sensitivities and the establishment of clear parametrized interfaces between them. The use of special blasting techniques to drive mining cost optimization projects are supported by its efficiency in rock fragmentation and lower energy cost when compared with the processed rock, a part of being the easier stage to implement changes and modification [1].

The fundamental goal of blasting is to achieve an adequate rock fragmentation and heave in order to positively impact downstream unit operations. A balance of macro and micro fragmentation, together with the muckpile disposal and swelling, are demanded in order to achieve optimal results in term of cost and energy consumption downstream.

On the other hand, all efforts dedicated to the optimization works must be integrated under a holistic continuous improvement program where all involved counterparts act in close coordination [2, 3]. As part of this process, there is a clear demand for tools and models to measure and predict the potential impact of rock blast fragmentation in other unit operations. Once these models are calibrated, along each operational cycle, their prediction capability becomes more reliable, and the optimum mine energy configuration closer every round.

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Thus, this paper presents a simple adaptive mining cost optimization model with the intention of illustrating the importance of modelling different unit operations. In order to obtain the best blast energy distribution, which could lead to the optimum total mining cost, a non-linear optimization strategy is required. Additionally, while the ore extraction optimization includes operations like drilling, blasting, loading, transport and comminution, the waste rock optimization attains to only to drilling, blasting, loading and transport operations.

II. Cost Minimization Model

The objective of the model is to find the blast design which helps to minimize the total cost of the mining process. This is a complex task due to the vast number of variables and uncertainties involved in the operation. Since mining can be regarded as sequential multi-step operation, as schematically represented in Fig 1, tracing the potential impact of a blast on subsequent unit operations are possible. In order to drive blasting results into the mining optimization path, it is necessary to create parametric and calibratable equations, where the interfaces between each operation are clearly identified. Thus, once each unit operation is modeled, their respective cost structure can be stablished.

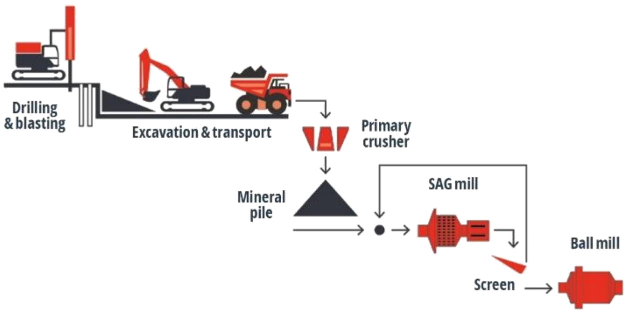

Fig 1: Simplified scheme of a typical metalliferous open pit mine configuration.

The cost model of each unit operation is coupled together to form the objective function $f(x)$ to be minimized. The non-linear optimization model with constrains intends to solve the problem by finding the solution vector x which locally minimizes the scalar function f, taking into account the allowed constrains of x. In this work, the cost function has the following form

$$\min f(x) = C_D + C_B + C_E + C_T + C_C + C_O \quad (1)$$

where $f(x)$ is the total cost and x is the vector containing the blast design configuration; C_D, C_B, C_E, C_T and C_C are the drilling, blasting, excavation, transport and comminution costs per cubic meter or metric tons. The comminution cost is the sum of all its individual processes; C_O would be other costs, such as administrative, structure, headcount and others, not considered in the model.

Since the model's strategy is the cost optimization through blasting results, the decision variables are defined as those that characterize the blast design, that is, diameter, burden, spacing, stemming, explosive's type, timing, and others. Therefore, the constrains of the model are defined based on the minimum and maximum allowable values of each variable, in reference to the burden size [4, 5], and safety and environmental limitations. Additionally, it is assumed that the operating parameters of others unit operations remains unmodified between one blast scenario and another. A simplified representation of the optimization cost model is presented in Fig 2.

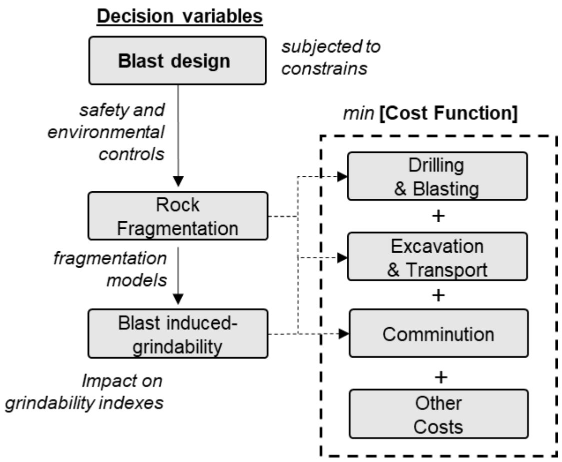

Fig 2: The proposed cost model structure.

The optimization model proposed in this work can also incorporate uncertainties from some variables of the problem, which can be treated by Monte Carlo simulations strategies. Typical field errors such as drilling deviation, error in the quantity of explosives and other downstream parameters, can be simulated.

A. Drilling and Blasting

Several blast fragmentation models have been proposed in the last years [6, 7, 8, 9, 10, 11, 12]. In this paper, the Kuz-Ram model [6, 7] is implemented in order to estimate the particle size distribution from blasting. Basically, the Kuz-Ram model combines the mean size prediction [13] and the uniformity index n with the cumulative distribution function of Rosin-Rammler [14] in order to describe the granulometric distribution curve of a given blast. The mean size formula $X50$ is

$$X50 = A\, P_f^{\,-4/5} Q^{1/6} \left(\frac{115}{RWS}\right)^{19/30} \qquad (2)$$

where A is the rock factor; P_f is the powder factor; Q is the amount of explosive per hole; and RWS is the relative weight strength (ANFO=100%).

The uniformity index n is given by

$$n = \left(2.2 - 14\frac{B}{D}\right)\sqrt{\frac{1+R}{2}}\left(1-\frac{W}{B}\right)\left(\frac{l}{L}+0.1\right)^{0.1}\frac{L}{H} \qquad (3)$$

where B is the burden (m); D is the diameter (mm); R is the ratio between spacing and burden; H is the bench height; W is the standard deviation of drilling (m); L is the length of the charge; and l is a relationship between column and bottom charge.

Thus, the particle size distribution is obtained by combining the equations (2) and (3) with the Rosin-Rammler distribution formula

$$P = 1 - \exp\left[-0.693\left(\frac{X}{X50}\right)^{n}\right] \qquad (4)$$

where P is the accumulated percentage associated with the X particle size.

The drilling and blasting cost approach is strictly based on a rock fragmentation prediction model, since a specific blast design is required to achieve the target fragmentation. The drilling operation is usually evaluated in terms of specific drilling, which measures the total meters drilled per volume of blasted rock. Knowing the unit cost of drilling per linear meter, the volumetric drilling cost C_D can be estimated. On the other hand, the blasting cost C_B is estimated by considering the total amount of explosive and consumption of accessories per volume of blasted rock and their respective unit costs.

B. Excavation and Transport

The following unit operation after blasting is typically the excavation and transport operations. In this step, the digging performance can be directly influenced by the particle size distribution, shape and swelling of the muckpile while better fragmentation improves the usage of trucks capacities. Although other important factors can also influence the overall performance of excavation and transport, such as operator's skills, floor and whether conditions, and others, specific studies have proven the impact of blasting results on these operations [15, 16, 17, 18].

Based on the measured digging performance, a site-specific diggability model can be developed. In this effort, some relevant parameters of the blast and excavator equipment are used to proper fit the model. Thus, the following site-specific model can be used

$$Q_L = V^{\alpha_1} Bc^{\alpha_2} H^{\alpha_3} A'^{\alpha_4} E_B{}^{\alpha_5} f(x) \qquad (5)$$

where Q_L is the digging productivity (m^3/h); V is the blast volume (m^3); H is the bench height (m); Bc is the bucket capacity (t); E_B is the blast specific energy (kJ/m^3); A' is the rock factor, as proposed by [15]; α_{1-5} = adjustable parameters; $f(x)$ = other parameters or calibration functions.

In terms of transport, a direct consequence of a higher (lower) loading productivity on the transport operation, assuming that all the other parameters are kept constant, such as the skill and efficiency of the truck operator, trip distances and others, is that the truck would need to be at the loading position for a smaller (larger) amount of time. Thus, it is reasonable to think that is the step in the total truck cycle time which may be directly influenced by blasting results.

Finally, the volumetric cost model for loading C_E and transport C_T is estimated based on the digging and transport productivities and their unit hourly cost.

C. Comminution Circuit

It is necessary to develop predictive and calibratable models for each stage of the comminution circuit. However, it is essential to identify the interface parameters between one stage and another, so that a change occurred in a previous process may reflect a consequence in the subsequent one. For simplicity, this work adopts simplified models for crushing, SAG and ball mills. However, the methodology allows the incorporation of more fundamental models, such as those based on the population balance approach [19].

The impact of dynamic forces in rock-blast fragmentation process has been the subject of investigation for years. Nevertheless, while the influence of the macro-fragmentation in downstream operations is evident, the role of micro-cracking is still not fully understood [19, 20, 21, 22]. Reference [19] suggested that both macro-fragmentation and the internal weakness of individual particles can affect crushing and grinding effectiveness.

Numerous studies show evidences on how grindability indexes can be used to quantify the potential impact of blasting in the processes of internal weakening of individual particles [23, 19]. The most common relationship is usually reported by relating the impact of powder factor on the work index [19, 20, 22, 21]. In this paper, this effect is modeled as presented in [2].

The third theory of comminution of Bond [25] allows the estimation of the energy required to reduce 80% of the feed to the 80% of the product. This is described by

$$W = 10Wi\left(\frac{1}{\sqrt{P_{80}}} - \frac{1}{\sqrt{F_{80}}}\right) \tag{6}$$

where W is the energy consumed in the process (kwh/ton); F_{80} and P_{80} are the passing sizes at 80% from the feed and product, respectively; Wi is the Bond work index.

The total energy consumed in the comminution circuit is estimated by assessing the contribution of each stage, from crushing to milling, as

$$E_C = E_{c1} + E_{c2} + \cdots + E_{cn} \tag{7}$$

where $E_{c1} \ldots E_{cn}$ are the energies consumed in each one of the n stages of the comminution circuit. In this paper, only the crusher, SAG and Ball mills machines are considered, as described next.

The maximum crusher capacity Q_{c1}(t/h) can be estimated through the model proposed by [26]

$$Q_{c1} = kv\sqrt{L_T}W(2L_m + L_T)\sqrt{\frac{R_C}{R_C - 1}}\rho_s f(P_K) f(\beta) S_C \tag{8}$$

where $k = 2820$; v is the ratio between the current and critical rotation velocities; L_T is the throw; W is the width of jaw plates; L_m is the closed set (m); R_C is the machine reduction ratio (gape/set); P_K is a parameter dependent of the feed fragmentation; β is the a parameter dependent of $X50$; and S_C is the calibration factor.

The estimation of the SAG mill capacity Q_{c2} (t/h) is based on the Austin model [27, 28] as follows

$$Q_{c2} = 10.6D^{3.5}(1 - AJ_C)\rho_c \frac{\left(\frac{L}{D}\right)\varphi_c\left(1 - \frac{0.1}{2^{9-10\varphi_c}}\right)}{C_F 10Wi\left(\frac{1}{\sqrt{P_{80}}} - \frac{1}{\sqrt{F_{80}}}\right)} \tag{9}$$

where D and L are the inner diameter and length of the mill; φ_c is the ratio between the current and critical rotation speed of the mill; C_F is the calibration coefficient; A is a constant of 1.03; J_C is the fractional volume of the cylindrical portion of the mill filled with all charge. The value of ρ_c is the calculated by

$$\rho_c = (1 - \varphi)\left(\frac{\rho_R}{M_R}\right)J_C + 0{,}6J_B\left(\rho_b - \frac{\rho_R}{M_R}\right) \tag{10}$$

where J_B is the fractional volume filled with balls; M_R is the mass fraction of rock in the total charge; φ is the bed porosity; ρ_R and ρ_b are the densities of the rock and balls, respectively.

The ball mill capacity Q_{c3} (t/h) can be modeled depending on its size. According to [27], the following equation can be used to calculate the ball mill capacity

$$Q_{c3} = k_b\left(\frac{L}{D}\right)\rho_b \frac{(J_B - 0.937J_B^2)\varphi_c\left(1 - \frac{0.1}{2^{9-10\varphi_c}}\right)}{C_F 10Wi\left(\frac{1}{\sqrt{P_{80}}} - \frac{1}{\sqrt{F_{80}}}\right)} \tag{11}$$

where $k_b = 6.13$ if the mill diameter D is < 3.81 m and $k_b = 8.01$ otherwise. The other parameters are the same of the SAG mill model.

Finally, the total comminution cost C_C is calculated by adding all individual costs from each comminution stage. These individual costs may consider both the energy and fixed-operational costs.

III. Results

The proposed adaptive cost optimization model is illustrated for a typical open pit gold mine operation. The mine produces approximately 7.0M tons of ore per year through a typical open pit mine configuration, composed by drilling, blasting, loading, transport, comminution and concentration. For the purpose of this study, however, stages including screens, hydrocyclones or pebble crushers are excluded, as well as the ore concentration stages.

The current mine configuration was audited in order to establish the working conditions of the operation. The ore material with density of 2700kg/m^3 and uniaxial compressive strength of 100MPa, presented a rock factor of 11. The typical production blast was designed with a powder factor of 0.70 kg/m^3 and boreholes of 165mm diameter, having an average X80 of 680mm. Based on the primary crusher configuration, around of 14-15% of oversizes were reported. The oversize was treated with secondary breakage techniques.

On the other hand, loading machines with payload of 45tons and average digging rates of 3020 t/h were used together with trucks of 181tons capacity. The primary crusher with a gap of 900mm, CSS=90mm and OSS=200mm was operating at an average capacity of 843 t/h whereas the SAG and ball mills were at 908 t/h and 1142 t/h, respectively.

Together with the equipment description, this information was used to calibrate the respective machinery models.

Additionally, uncertainties related with geotechnical characteristics, blasting, loading, transport and comminution configurations, and others, were also included in this study.

A. Model Simulation

In this exercise, 1000 different scenarios covering a wide range of powder factors, resulting from different blast designs, were simulated through Monte Carlo technique in order to find out the candidate blast design configuration, which could drive a major impact in the total mine cost. Fig 3 shows all individual unit operations costs alongside with the total mine cost as a function of the blast powder factor. The variability of the results for a given powder factor is a response of the model to the uncertainties of the problem.

The model outputs indicate an average optimum scenario when the powder factor reaches 1.26 kg/m^3. This level of energy delivers a fragmentation size of 392mm in 80% passing, which supposes an improvement of 42% in comparison with the standard blast configuration. This level of fragmentation eliminates extra costs with secondary breakages and, in addition, suggest significant improvements on downstream unit operations. In the comminution circuit, for example, the SAG throughput could improve up to 946 t/h, which represents an increment of 4.2%, while the ball mill could reach 4.5%, increasing its capacity to 1193 t/h.

In terms of total costs, an average reduction of 4.6% is expected, which could represent potential savings of 1.54 ± 0.45 M USD/year. Additionally, the increment of throughput could lead extra benefits by increasing the amount of ore processed and recovered, even if higher energy blast configuration could lead to extra dilution or wall control problems. However, these undesired effects must be fully controlled during the implementation of the optimized blast design.

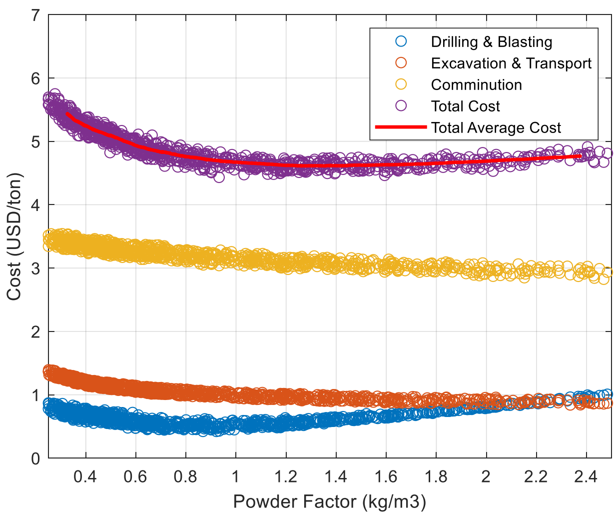

Fig 3: Mining cost distribution versus powder factor.

B. Model Considerations

Higher energy blasting could produce some adverse side-effects, such as ore dilution/loss, higher risk of fly rocks, wall control problems and higher ground vibrations levels, and should only be applied where downstream results have shown a benefit to the overall process. The challenge for quantifying the benefit of optimized blasting configurations, a part of the full environmental and safety controls, lies in measuring the actual results in order to recalibrate the prediction models. For example, maintaining unchanged the comminution circuit configuration such as rotation speed of grinding mills, concentration of water, solids, etc., one would ensure that actual productivity and consumption readings are compatible with the prediction criteria used in the optimization model. However, production realities may not allow for that and incorrect assumptions as to the impact of high-energy blasting can be made. Thus, the implicit and explicit limitations of the predictive models must be considered when deploying an optimization program [3] as proposed in this paper.

IV. Conclusion

Long-term mining costs optimization strategies involve continuous ability to model, measure and calibrate all related technical and operational performances along of the mine cycle. Taking this goal into account, a simplified optimization model for the total mining cost minimization was presented. The model allows the use of different predictive sub-models for each one of the unit operations, enlarging its flexibility for various mine configurations. The proposed optimization method has been illustrated for a standard open-pit gold mine, where the best blast energy distribution, required to potentially optimize the total mining costs, is identified. Results for each mining stage must be compared to the respective predictions, in order to allow the adaptive calibration of each sub-model and thus, to improve their predictive capacities at each operating cycle towards the optimum mining energy configuration.

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