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## Optimizing the performance of high-pressure grinding roll based ore enrichment circuits

### Introduction

The progressive exploitation of ore deposits results in extracting run-of-mine (ROM) ore with a lower grade of useful mineral. Over the last four decades, average ore grades of selected base minerals have decreased as follows:

- copper: from 1.5 to 1% (however an increase of up to 2.5 % was observed in the period 1990–2000),
- lead: from nearly 8 to 3.5%,
- nickel: from 4 to 1%,
- gold: from 3.5 to 2 g/Mg (however an increase of over 4 g/Mg in the period 1980–1985).

Finer mineralisation requires adjusting the technological circuit to the new conditions. This usually results in the need to apply operations enabling a more intensive ore size reduction. Extended comminution operations are necessary to maintain the useful mineral weight recovery grade at a certain level and to achieve satisfactory economic indices. The costs of ore enrichment (comminution operations) – constituting the highest share in total processing costs – increase regardless, making it necessary to improve the effectiveness of crushing and grinding operations by introducing appropriate comminuting devices into the circuit. The circuit's modernization should cover all operations from coarse crushing to a fine

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grinding stage. Modern comminution devices often do not operate optimally, leaving room for additional efficiency studies. HPGR devices usually result in beneficial energy savings; but on the plant scale, this might not be always true.

Industrial operations involving ROM ore size reduction are the most energy intensive processes in the ore processing technological circuit, amounting to between 30 and 45% of total costs in the mining industry. Around 0.39% of national energy consumption in the USA is used in comminution. In Canada and the Republic of South Africa, this index amounts to nearly 2%, while in Australia it reaches 1.5% (Tromans 2008). In many other countries, ore pre-treatment technology consumes even more energy. It is estimated that industrial crushing processes consume 3 to 5% of global electric energy production. In order to limit the energy consumption of comminution operations, it is necessary to reduce such grinding operations in favour of intensifying the fine crushing stage. To achieve that, modern crushing devices which increase the size reduction ratio at lower energy consumption and higher capacity are being applied.

Most experts agree that the development of beneficiation technology – in the form of mechanical devices – has reached its limits. Nowadays, it is hard to invent some breakthrough device which would significantly reduce processing costs. Rather, savings should be sought in the proper selection of equipment at individual processing stages and by optimization of the entire circuit operation (i.e. the limitation of unnecessary material streams, minimizing energy losses and idle runs, etc.).

Considering the above, the process modeling for HPGR-based comminution circuits is significant in terms of optimizing the entire technological circuit's operation (Napier-Munn et al. 1996; Morrell et al. 1997; Daniel and Morrell 2004; Cleary et al. 2008; Morrell 2009; Saramak et al. 2010).

## 1. Methodology

In terms of the above aspects, an optimizational approach to HPGR-based circuits' performance should start from the analysis of potential benefits from the application of the high-pressure roller press into a specific technological circuit. A key role here is played by the type of feed material and the required quality of the final product (concentrate). For a specific ore type, two basic properties determine its size reduction effectiveness – particle size distribution and moisture content (Fuerstenau and Abouzeid 1998; Tavares 2005; Fuerstenau, Abouzeid 2007; Saramak 2011). The above properties also influence the unfavorable phenomenon of flake formation. A part of the HPGR product leaves the press in the form of high-density flakes, reaching up to 90% of the material's specific gravity. Flakes usually break on the HPGR screen surface or even in transfer chutes, bins, and conveyors before reaching the screen. However, for certain types of hard ores, flakes might be too competent to de-agglomerate on the screen and, as a result, they are recycled to the HPGR press with the screen oversize product (Saramak 2012). This is an undesirable effect,

as the fine material constituting a flake still remains in the grinding circuit instead of being directed to the downstream beneficiation processes. This increases the HPGR circuit load, decreasing both energy efficiency and throughput rates. The results of studies (Saramak, Kleiv 2012) show that the relationship between HPGR product particle size distribution and feed moisture content can be described by means of a parabolic function.

Pressure is the key operating parameter of the HPGR which determines the quality of comminution products. The influence of pressure ( $F_{sp}$ ) on the crushing product size reduction ratio ( $S_x$ ) can be determined by an exponential function (Saramak 2010):

$$S_x = A \cdot F_{sp}^B, \quad 0 < B < 1 \quad (1)$$

where  $A$  and  $B$  – parameters.

Practice shows that the size reduction ratio is insufficient in a qualitative description of comminution products. It is then recommended to build a model based on the function of crushing product particle size distribution (Saramak 2013), or at least to be able to calculate weight recoveries of specific particle size fractions (especially the finest fractions). As a result of approximating the HPGR crushing product through the use of selected theoretical distribution, it is possible to make the parameters of the distribution conditional on the operating pressure value. For example, for truncated Weibull's distribution, which has its justification in the description of comminution products, pressure  $F_{sp}$  can be presented as an independent variable of individual parameters:

$$d_{\max}, n, c = f(F_{sp}) \quad (2)$$

therefore:

$$\Phi(d) = f(F_{sp}) = \Phi(d; F_{sp}) \quad (3)$$

In such a model, it is possible to control the basic technical parameter of the press in order to obtain a specific quality of a crushing product and, as a result, to determine the operational conditions of downstream beneficiation processes.

The material's particle size is a key feature influencing the effects of flotation, and it can be different for various types of ore. In general, particles below and above a certain value cause a significant decrease in flotation effectiveness (Brożek, Młynarczykowska 2010). A sample relationship between flotation recovery and material particle size for sulphide copper ores obtained on the plant scale is presented in Fig. 1.

For Polish copper ores, the most favourable particle size of feed for flotation operations is below 0.07 mm (Fig. 1). In a multi-stage ore comminution process, material from 0.02 to 0.07 mm may appear in the circuit even at the fine crushing stage. It can then be extracted from the crushing circuit and sent directly to flotation, bypassing the grinding operations and

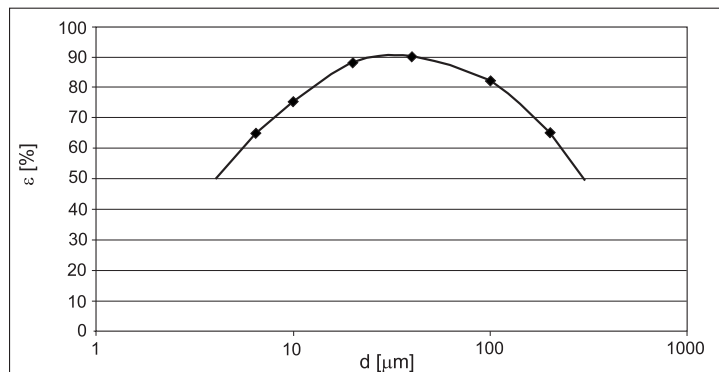


Fig. 1. Relationship between the metal recovery in flotation operations and particle size distribution for sulphide copper ores (Potulska 2008)

Rys. 1. Zależność pomiędzy wielkością ziarna a efektywnością flotacji dla siarczkowych rud miedzi (Potulska 2008)

thus lowering their energy-consumption. Test results showed that up to 15% more material below 0.07 mm can be generated in roller presses when compared to rod mills (author's unpublished work).

Participation of very fine particles in grinding products – mainly an effect of the over-grinding phenomenon – significantly decreases, in turn, flotation effectiveness. The aim, therefore, is to limit their formation during the grinding and regrinding stages of the comminution circuit. Taking into account the above issues, the enrichment process for sulphide copper ores should be designed to maximize the weight recovery of the particle size fraction between 0.02 and 0.07 mm, and to minimize the content of particle fractions finer than 0.02 mm (Trahar 1981).

The modeling approach, presented in this paper, assumes making the qualitative properties of comminution products conditional on the technical parameters of a specific crusher. Following the technical parameters of an individual device affects the particle size distribution of comminution products:

- in the crusher: gap width ( $s$ ), jaw jump ( $e$ ), frequency ( $f$ );
- in the HPGR: operating pressure ( $F_{sp}$ ), speed of rolls ( $v$ );
- in the mills: frequency of rotations, the mill's capacity, feed density;
- in classification operations: cut-point particle size ( $d_T$ ).

## 2. Work models of ore crushing circuits

### 2.1. Economic model

The target function  $M$ , constituting a basis for comminution circuit work optimization, in its general form can be written as the difference between the total income obtained from

production and the total cost of plant operation. In prior studies (Svedensten, Evertsson 2005; Tumidajski 2012), target functions were presented wherein the final products of the enrichment circuit were also the final trade products (e.g. aggregate production industry). A general form of the target function for ore beneficiation processes can be as follows:

$$M = \sum_{j=1}^k \sum_{i=1}^n Q_i \cdot \gamma_j \cdot p_j - \sum_{j=1}^k \sum_{i=1}^n (FC_i + VC_i \cdot Q_i) \cdot \gamma_j \quad (4)$$

where:

- $n$  – number of technological operations in a given processing circuit,
- $k$  – number of separated size fractions of a product,
- $\gamma_j$  – weight recovery of the size fraction  $j$ , [%],
- $p_j$  – value (price) of the particle fraction  $j$ , [PLN],
- $FC_i$  – fixed cost of technological operation  $i$ , [PLN],
- $VC_i$  – variable cost of technological operation  $i$ , [PLN/Mg],
- $Q_i$  – capacity of technological operation  $i$ , where  $\sum_{i=1}^n Q_i = Q$ , [Mg/h].

The target function (4) takes into account the cost of each technological operation separately and considers all determined particle size fractions of the product. However, the model is ineffective when applied to the ore processing sector, given that a final product of comminution circuits  $\gamma_i$  is at the same time the flotation feed and has no trade (commercial) value  $c_j$ . Besides that, only the material size fraction 0.02–0.07mm effectively influences the increasing efficiency of flotation operations (Fig. 2). Therefore, only the mentioned size fraction generates income. The material particle size fraction below 0.02 mm generates paymetal losses, while the material coarser than 0.07 mm is recycled to the regrinding operations. Accurate assessment of a respective particle size fraction is also impossible. Some agreed values can be accepted by attributing weights to individual particle size fractions which, for example, correspond to the load of the circuit. The particle size fraction value can be roughly calculated considering the useful mineral grade multiplied by its market price and the metal recovery index. The attributed weights have a positive or negative sign which clearly categorizes the individual particle size fraction as income or cost in the target function.

The income component in the target function (4) for a multi-stage ore enrichment process can be formulated as a value of the useful ore component recovered in the technological process (in currency units):

$$I_1 = \gamma_{0.02-0.07} \cdot \beta_{0.02-0.07} \cdot Q \cdot \varepsilon_{0.02-0.07} \cdot P \quad (5)$$

where:

- $\varepsilon_{0.02-0.07}$  – a percentage recovery of metal from concentrate (metallurgical recovery) in the size fraction 0.02–0.07 mm,

- $\beta_{0.02-0.07}$  – useful mineral grade in the size fraction 0.02–0.07 mm,  
 $\gamma_{0.02-0.07}$  – concentrate weight recovery in the size fraction 0.02–0.07 mm,  
 $Q$  – circuit's capacity (total amount of processed feed),  
 $p$  – unit price of the useful component.

The cost component constitutes total spendings on the feed upgrading at all stages of the technological circuit for each particle size fraction of the product:

$$C_1 = \sum_{j=1}^k \sum_{i=1}^n (FC_i + VC_i \cdot Q_i) \cdot \gamma_j \quad (6)$$

For the ore comminution technological circuit, a target function maximizing the profit is presented using equation (7):

$$M_1 = \max \left( \gamma_{0.02-0.07} \cdot \beta_{0.02-0.07} \cdot Q \cdot \varepsilon_j \cdot p - \sum_{j=1}^k \sum_{i=1}^n (FC_i + VC_i \cdot Q_i) \cdot \gamma_j \right) \quad (7)$$

## 2.2. Technological model

The final product of the comminution circuit is, in general, a composition of the following three particle size fractions: the fraction below 0.02 mm which is generally the waste, the fraction 0.02–0.07 mm which is the optimal feed for flotation operations, and, finally, the fraction over 0.07 mm which is recycled to the circuit. The approach to ore comminution circuit optimization, based on the technological criterion (maximization of the useful component weight recovery), assumes the following form of the target function:

$$M_2 = \max \left\{ \begin{array}{l} \gamma_{0.02-0.07} \cdot \beta_{0.02-0.07} \cdot Q + (1-w) \cdot [1 - (\gamma_{0.02-0.07} + \gamma_{0-0.02})] \cdot \mu_{+0.07} \cdot Q + \\ -\gamma_{0-0.02} \cdot \beta_{0-0.02} \cdot Q - w \cdot [1 - (\gamma_{0.02-0.07} + \gamma_{0-0.02})] \cdot \mu_{+0.07} \cdot Q \end{array} \right\} \quad (8)$$

where:

- $\beta_{0.02-0.07}$  – useful mineral grade in the product particle size fraction 0.02–0.07 mm,  
 $\beta_{0-0.02}$  – useful mineral grade in the product particle size fraction 0–0.02 mm,  
 $\mu_{+0.07}$  – useful mineral grade in the product particle size fraction +0.07 mm,

$$w = \frac{\gamma_{-0.02}}{\gamma_{-0.02} + \gamma_{0.02-0.07}} \quad \text{– a proportion in which the size fraction +0.07 mm}$$

is divided between fractions below 0.02 mm and 0.02–0.07 mm.

Individual components of the target function (8) describe a useful mineral grade in three particle size fractions. In the fraction 0.02–0.07 mm, the grade is denoted as  $\beta$  because the fraction is regarded as a final product of the circuit (a concentrate). In the fraction 0–0.02mm, in turn, the grade is denoted as  $\vartheta$  because it is treated as the waste. Finally in the fraction +0.07 mm, the grade is general content  $\mu$  because it is neither the final product nor the waste. The  $w$  value determines the proportion of the material separation into fractions 0–0.02 mm and 0.02–0.07 mm.

As the fraction 0.02–0.07 mm is the most favourable feed for flotation operations for Polish copper ores (Potulska 2008), the first component of formula (8) has a positive sign. The second component, in turn, has a negative sign (minus) because, as it was stated earlier, the size fraction 0–0.02 mm generally decreases flotation effectiveness. The third component also has a negative sign because it describes the material which will pass to the waste from the material recycled.

Functional relationships between the parameters of particle size distribution curves of comminution products and technical parameters of crushing devices are the limitations in target functions  $Z_1$  and  $Z_2$ . For the roller press, the operational pressure  $F_{sp}$  is a major technical parameter:

$$\Phi(d; n, c, d_{\max}) = \Phi(d; F_{sp}) \quad (9)$$

where:

- $n, c, d_{\max}$  – parameters of the function approximating the empirical distribution curve,
- $F_{sp}$  – operational pressure in the roller press.

Weight recoveries of individual particle size fractions of comminution products will be described by the following functions:

$$\gamma_{0-0.02} = \Phi(0.02; F_{sp}) \quad (10)$$

$$\gamma_{0.02-0.07} = \Phi(0.07; F_{sp}) - \Phi(0.02; F_{sp})$$

$$\gamma_{+0.07} = \Phi(d_{\max}; F_{sp}) - \Phi(0.07; F_{sp})$$

with the presence of limitations:

$$F_{sp-\min} \leq F_{sp} \leq F_{sp-\max} \quad (11)$$

$$Q_{\min} \leq Q \leq Q_{\max}$$

### 3. The energy consumption of ore processing circuit

This section presents a sample analysis of operation effects obtained for the modernized ore crushing circuit. It analyzes the energy consumption of two technological circuits – the first circuit based on conventional tumble mills, and the second an HPGR unit installed at the fine crushing stage. Both circuits are presented in Fig. 2.

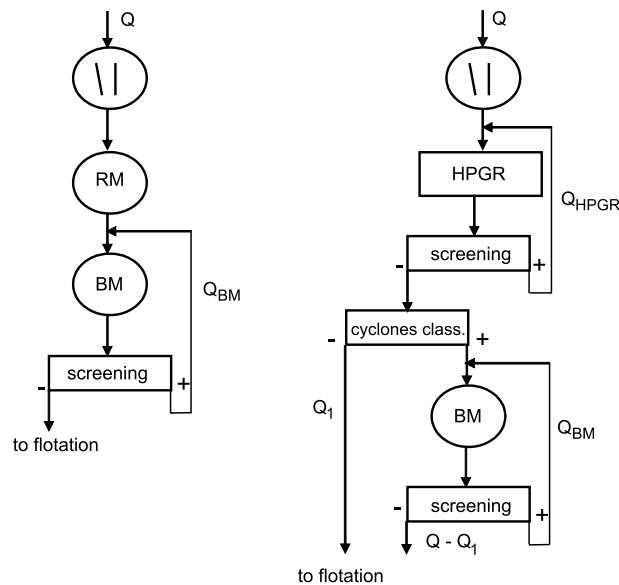


Fig. 2. Technological comminution circuit before (left) and after (right) modernization  
*RM* – rod mill, *BM* – ball mill, *Q* – circuit's capacity, *Q<sub>BM</sub>* – ball mill recycle material stream,  
*Q<sub>HPGR</sub>* – HPGR recycle material stream,  
*Q<sub>1</sub>* – material stream directed to flotation, bypassing the grinding stage

Rys. 2. Technologiczny układ rozdrabniania przed (z lewej) i po (z prawej) modernizacji

The following control variables connected with individual devices can be distinguished in the first circuit:

- crusher gap width,
- screening cut-point in separation operations,
- grinding time.

The first crushing stage is the same for both circuits; therefore, the analysis includes only the stages which differ from each other in both variants. Total energy consumption in the first variant can be determined as follows:

$$P_1 (\text{kWh}) = E_{SP\_crusher} \cdot Q + \sum_{i=1}^{N_1} E_{SP\_RM\_i} \cdot Q_i + \sum_{j=1}^{N_2} E_{SP\_BM\_j} \cdot (Q_j + Q_{BM1}) - i \quad (12)$$



while in the second circuit:

$$\begin{aligned}
 P_2 \text{ (kWh)} = & E_{SP\_crusher} \cdot Q + \sum_{k=1}^{N_3} E_{SP\_HPGR\_k} \cdot (Q_k + Q_{HPGR}) + \\
 & + \sum_{l=1}^{N_4} E_{SP\_BM\_l} \cdot (Q_l - Q_1 + Q_{BM2})
 \end{aligned} \tag{13}$$

where:

- $Q$  – circuit's capacity,
- $Q_i$  – capacity of a single rod mill,
- $Q_j$  – capacity of a single ball mill in first variant,
- $Q_1$  – capacity of a single ball mill in second variant,
- $Q_k$  – capacity of the HPGR unit,
- $E_{SP\_crusher}$  – crusher's unit energy consumption,
- $E_{SP\_RM\_i}$  – rod mill unit energy consumption ( $i = 1, \dots, N_1$ ),
- $E_{SP\_BM\_j}$  – ball mill unit energy consumption in first variant ( $j = 1, \dots, N_2$ ),
- $E_{SP\_BM\_l}$  – ball mill unit energy consumption in second variant ( $l = 1, \dots, N_4$ ),
- $E_{SP\_HPGR\_k}$  – HPGR unit energy consumption ( $k = 1, \dots, N_3$ ),
- $Q_{HPGR}$  – HPGR recycle material stream,
- $Q_{BM1}$  – ball mill recycle material stream in the first variant,
- $Q_{BM2}$  – ball mill recycle material stream in the second variant,
- $Q_1$  – material stream bypassing the grinding stage in the second variant,
- $N_1$  – number of rod mills in the circuit in the first variant,
- $N_2$  – number of ball mills in the circuit in the first variant,
- $N_3$  – number of HPGR units in the circuit in the second variant,
- $N_4$  – number of ball mills in the circuit in the second variant.

To make the energy consumption of the second circuit lower than in the first variant, the difference  $P_1 - P_2$  must be larger than zero. Assuming  $N_2 = N_4$  and  $Q_{MK1} = Q_{MK2}$  (i.e. the number of ball mills and ball mill recycle streams are the same for both circuits) we obtain:

$$P_1 - P_2 = \sum_{i=1}^{N_1} E_{SP\_RM\_i} \cdot Q_i - \sum_{k=1}^{N_3} E_{SP\_HPGR\_k} \cdot (Q_k + Q_{HPGR}) + \sum_{l=1}^{N_4} E_{SP\_BM\_l} \cdot Q_1 \tag{14}$$

It was also assumed that each of the three rod mills can be replaced with a single HPGR unit in terms of capacity and the HPGR recycle stream amounts to as much as 50% of capacity  $Q$ . Energy consumption of individual units were also empirically determined:

$$E_{SP\_HPGR} = 2 \text{ kWh/Mg} \quad (15)$$

$$E_{SP\_BM} = 2.2 \text{ kWh/Mg}$$

$$E_{SP\_RM} = 5 \text{ kWh/Mg}$$

Particle size distribution of the HPGR and rod mill products was also analyzed. As was mentioned earlier, the difference in weight recoveries of the fine particles – which can be directed to the flotation operation bypassing the grinding stage – is about 15% in favour of the HPGR unit. Taking into account the above relationship, equation (14) can be written in the following form:

$$P_1 - P_2 = \sum_{i=1}^{N_1} E_{SP\_RM\_i} \cdot Q_i - \frac{1}{2} \sum_{k=1}^{N_3} E_{SP\_HPGR\_k} \cdot Q_k + \sum_{l=1}^{N_4} E_{SP\_BM\_l} \cdot Q_l \quad (16)$$

Analyzing the model, it can be observed that energy savings at the second crushing stage can already be achieved when the HPGR energy consumption is not larger than double the energy consumption of the rod mill. An additional energetic gain is obtained at the grinding stage in ball mills because the material stream  $Q_1$  is bypassing the grinding stage and is sent directly to flotation. The final profit (17) amounts to 1.2 kWh/Mg for each replaced rod mill and 0.85 kWh/Mg for each operating ball mill in the second variant of the circuit (Fig. 2):

$$P_1 - P_2 = Q(1.2N + 0.85N_4) \quad (17)$$

The assumed values (14) and the relationship of capacities of the HPGR to the crusher (or mills) differ from each other depending on the type of crushed ore and the technological enrichment process run. The calculations (16) should then be made separately for each case. This model considers only energy savings, but some additional gain can be obtained by the lower consumption of grinding media (rods and balls), due to the withdrawing of rod and ball mills from the circuit. Rod mills are replaced by HPGR units, while the number of ball mills can be reduced due to the lower capacity ( $Q_k - Q_1$ ) of the modernized grinding circuit.

### Summary and conclusions

The effect of the modernization presented in section 4 does not take into account any technical issues connected with circuit modification, and possibly enables a suitable separation of the  $Q_1$  mass stream to take place. Rebuilding the circuit will also require taking into consideration additional material streams (i.e. the HPGR recycle stream), which results in increasing the circuit's energy consumption. The increase, however, will be of rather

insignificant importance when compared to the energy savings achieved after the new crushing device application.

Optimal values of target functions (7) and (8) can be obtained by applying the mathematical programming theory, due to the model's nonlinearity. The obtained results are sufficient to determine the values of the HPGR technical parameters which maximize either the profit or concentrate the grade with the presence of the model's limitations, given by equations (10) and (11).

Based on the results of the study presented in this paper, one can say that it is possible to build a simulation model of an ore processing plant operation, relying on the technical parameters of comminution devices. The parameters, in turn, can be combined with the comminution process' effects by the parameters in approximation functions of theoretical particle size distributions. It proves the thesis that the ore processing plant operation can be effectively controlled in terms of the assumed technological or economic criteria.

The following general conclusion can be formulated on the basis of investigation results presented in this paper:

- comminution circuit optimizational approach should consider the technological circuit as a whole instead of analyzing individual crushing and grinding stages separately.
- physico-mechanical feed properties initially determine the comminution effects as well as influence significantly the flotation effectiveness,
- the HPGR operating press force is a parameter of major importance in terms of the comminution product's particle size distribution,
- an individual crushing device work model can be determined by means of the approximation of comminution product particle size distribution and linking the parameters of the approximation formula with the device's technical parameters.

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OPTIMALIZACJA PRACY UKŁADÓW PRZERÓBKI MECHANICZNEJ RUD  
OPARTYCH NA WYSOKOCIŚNIENIOWYCH PRASACH WALCOWYCH

Słowa kluczowe

Rozdrabnianie, optymalizacja, wysokociśnieniowe prasy walcowe

Streszczenie

Artykuł dotyczy zagadnień związanych z modelowaniem i optymalizacją pracy układów rozdrabniania rud. Przeanalizowano typowy kilkustopniowy technologiczny układ rozdrabniania rud, w którym wysokociśnieniowa prasa walcowa została zainstalowana na drugim stopniu rozdrabniania. Produkt końcowy analizowanego układu był jednocześnie nadawą do procesów flotacji, a skład ziarnowy nadawy determinował wstępnie efektywność procesów flotacyjnych. Aby określić efektywność pracy układu technologicznego przeróbki rud zbudowano odpowiedni model matematyczny pracy układu rozdrabniania, w którym funkcja celu została powiązana z efektywnością procesów flotacyjnych. Kryterium decydującym o jakości produktów rozdrabniania była wielkość ziarna i związane z tym wychody poszczególnych frakcji ziarnowych. Wychody poszczególnych frakcji produktów rozdrabniania zostały uzależnione od technicznych parametrów pracy urządzeń rozdrabniających.

W pierwszym wariantcie funkcja celu maksymalizowała efekt ekonomiczny, czyli zysk, natomiast drugi wariant zakładał maksymalizację odzysku składnika użytecznego w koncentracie.

Aplikacje przemysłowe wysokociśnieniowych pras walcowych pozwalają na osiągnięcie wymiernych oszczędności energetycznych w modernizowanych układach rozdrabniania rud, co zostało przedstawione poprzez analizę porównawczą energochłonności dwóch układów rozdrabniania rud. W skład pierwszego układu rozdrabniania wchodziła kruszarka na etapie kruszenia wstępnego, młyn prętowy na etapie kruszenia drobnego oraz młyn kulowy na etapie mielenia. W drugim, zmodernizowanym układzie młyny prętowe zostały zastąpione wysokociśnieniowymi prasami walcowymi. Korzyści osiągnięte przy takiej modernizacji polegały na obniżeniu energochłonności etapu kruszenia drobnego prawie o połowę oraz zmniejszenie przerobu na etapie mielenia w młynach kulowych poprzez skierowanie części produktu prasy walcowej do procesu flotacji wstępnej.

#### OPTIMIZING THE PERFORMANCE OF HIGH-PRESSURE GRINDING ROLL BASED ORE ENRICHMENT CIRCUITS

##### Key words

Comminution, optimization of comminution circuit as integrity, high-pressure grinding rolls

##### Abstract

This article concerns the issues of modeling and the optimizational approach for the performance of ore comminution circuits. A typical, multi-stage comminution circuit was analyzed with the high-pressure grinding rolls unit operating at a fine crushing stage. The final product of the circuit under investigation was, at the same time, a flotation feed in which particle size distribution initially determined the effectiveness of flotation operations. In order to determine the HPGR-based comminution circuit performance, a suitable mathematical model was built wherein the target function was linked directly with the effectiveness of the flotation processes. The target function in the presented model considers the issue in terms of the flotation operation's effectiveness. The particle size distribution of individual comminution products and resulting from the weight recoveries of individual size fractions were criteria determining the quality of the comminution product. Weight recoveries of individual size fractions, in turn, were tied with the technical operating parameters of individual comminution devices. In the first model, profit maximization was the target function, while the second variant of the model took into account maximization of the useful mineral weight recovery in the concentrate.

The HPGR application into ore processing circuits also results in energy saving benefits which were presented in a comparative analysis of the energy consumption of two comminution circuits – the first based on conventional crushing devices, and the second on the HPGR unit application which replaced the rod mills. The main benefit of such a modernization was almost two times lower energy consumption by the fine crushing stage and a decrease in the ball mills' grinding operations load through bypassing a part of the material directly for the rough flotation operations.

