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UPL determination of multi-element deposits with grade uncertainty using a new block economic value calculation approach

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Abstract

The block economic value (BEV) of a single-metal deposit is calculated based on the metal content and the related costs. The common methods available for calculating BEV are just based upon the profitable elements, and the effects of undesirable elements on BEV are not considered. However, in multi-element deposits, the effects of other elements existing in the blocks on BEV should be considered with the purpose of optimizing the blending. These elements and blending methods have considerable effects on the quality of the final product. In this paper, a new approach is introduced to determine BEV in multi-element deposit with two types of profitable and penalty elements by considering the effect of blending on BEV. Consequently, the ultimate pit limits (UPLs) will be determined based on these conditions. The developed model is tested in the Gol-e-Gohar No.2 iron-ore mine, and the mine UPLs is determined. The results obtained showed that the mineable reserve of the pit increased by 3% when the effects of both types of elements are considered. In order to investigate the effect of grade uncertainty on BEV, twenty realizations of the ore block are generated using the sequential Gaussian simulation approach. The UPLs of all the realizations are determined using the developed BEV-calculation method, and the pit limits with different probabilities of occurrence are determined. The total mineable reserve varied between 20,380 and 46,410 million tons. The exploitation of mine should start with the smallest pit (100% probability). The largest pit should be considered as a guide for surface-facility locating.

Keywords: *BEV, Multi-Element Deposits, Pit Limits, Grade Uncertainty, Open-Pit Mining.*

1. Introduction

One of the most important stages of an open-pit mine design is to determine UPL (ultimate pit limit). After finding the mine's final pit limits, it is possible to determine the size and location of the processing plant and the locations of the waste dumps and other facilities and equipment [1]. A pit outline is also required to determine the mine's production schedule. UPL is determined based on one or more of the following objectives:

- Maximizing the pit's Net Present Value (NPV)
- Maximizing the value per ton of the final product
- Maximizing the mine life
- Maximizing the metal content of the pit

Out of the mentioned objectives, in most cases, maximizing the pit NPV is the most important objective for an open-pit design [2].

In order to achieve this objective, the block economic value (BEV) of each block is required as the main input parameter for optimization. BEV is a monetary value that is assigned to each block, according to the estimated revenue from the ore content minus the costs involved in producing the final products.

The mathematical approaches for UPL optimization are divided into two groups: deterministic and probabilistic. In the deterministic models, all inputs are assumed to have fixed real values (known). However, the assumption of input certainty is not always realistic. In reality, some data, e.g. ore grades, future product demands, prices, and production costs can vary within certain limits [3]. In fact, uncertainties in the mining-engineering field are caused by insufficient and incomplete data. For example, owing to the presence of spatial grade uncertainty, the dynamic change of ore and waste material makes the prediction of the optimal mining sequence a challenging task [4–8].

Algorithms for solving an open-pit optimization problem have been developed by many researchers. In these algorithms, BEV is considered as the input data.

In the moving-cones approach presented by Pana, if the material inside a cone contains a profitable amount of ore, the material is removed from the block model. The process is repeated until no more profitable cones of material exist [9]. Lerchs and Grossman have used the concept of dynamic programming and introduced a 2D algorithm for finding the optimum pit limits [10]. They also developed a graph-theoretic approach to find an optimum pit limit. They converted the block model of a mine into a graph and determined the ultimate pit limits by solving for the maximum closure of the graph.

Other different heuristics and rigorous methods for ultimate pit-limit determination have been developed, e.g. maximum flow in the network [11–15], genetic algorithms [16], pit-parameterization algorithms [17], and decision-support-system algorithms [18]. In all of these algorithms, the optimization is done based on the economic value of the blocks. In the literature, BEV is calculated based on the main element content; the effect of the other existing elements is not considered, and it is assumed that they are undesirable elements and must be removed during the processing stage such as copper ore but in some cases such as iron ore, in addition to the iron content, other elements such as sulfur and phosphorous are important for the blending purpose to reach the destination/customer requirements.

Another issue for consideration is the BEV uncertainty due to the variation in the grades of the existing elements. Many methods consider the grade uncertainty in their calculations. Godoy has developed a method to quantify the geological uncertainty in long-term production scheduling of open-pit mines [19]. Menabde has treated the BEV uncertainty using a conditional simulation method [20]. Osanloo has considered the uncertainty in BEV using the probability models [3]. Kumral and Dowd have combined simulated annealing and Lagrangian parameterization for short-term planning in non-metallic mines [21].

The common formula for calculating BEV is the Whittle equation that is based upon the fact that the economic value of a block is the difference between the income and costs. The income of a block is directly related to the metal content (main element) that is recovered from the block. This does not take into account more complex situations, where the blending of different material types is required to obtain a product that can be economically processed and sold. In the case of multi-element deposits, the quality product, and consequently, the value of the final product is a function of the grades of the main element and other existing elements of the mine blocks. Therefore, the value of the final product should not be calculated solely on the basis of the main element (metal) content because the price of the final saleable product is affected by the existence of some deleterious or undesirable elements in the block. BEV is the basis for determining the ultimate pit limits and production scheduling of the mine in most developed algorithms in this regard. Also NPV of the project is calculated according to the economic value of the blocks. Miss estimation of the block value leads to a wrong UPL determination, and consequently, wrong production scheduling. Thus in this work, a linear formula is presented to calculate BEV in multi-element deposits by considering the effects of the existing elements (main elements, desirable and undesirable elements) in the orebody. In order to investigate the grades' uncertainty, twenty different realizations were generated using the sequential Gaussian simulation (SGS). The method applies grade uncertainty due to the variation in the elements' grades to the UPL optimization. The proposed approach was applied to an iron-ore deposit, and the BEV model of each realization was calculated using the proposed model.

2. Multi-element deposits

For particular types of minerals, e.g. iron ore, coal, phosphate, and bauxite, the product quality depends on the different elements that exist in the orebody. The consumption of these minerals is a function of the elements' grades. On the other hand, in many cases, the chemical properties of the material are dictated by mill or sale contracts, and they impose limitations on the grades of the main elements and the contaminants. In this situation, different blocks with various characteristics are blended together such that the resulting mixture satisfies the required quantity and quality. Therefore, the grades of the associated elements are important because of their effects on the production quality; consequently, their effects must be considered.

In multi-element deposits, different parts of a deposit may have different properties. For example, in a coal-mining operation, one seam may have a low calorific value and high ash content, while another seam may have a high calorific value and low ash content. Each of these parts may have no economic value if considered individually. However, if the materials in these two parts are considered as a complex that could be mixed together, it might be possible to increase the tonnage of saleable material that satisfies contractual agreements. For iron-ore mines, different blocks contain various amounts of iron ore (Fe) and other elements, e.g. sulfur (S) or phosphorous (P).

For iron-ore mines, payment is based upon the product grade, not only of the iron content but also of the contents of other elements. If each block is treated on its own, it may not satisfy the concentration plant or contract requirements, and be considered as a non-economic block. However, if a block with a low Fe and a high sulfur content is blended with a block with a high Fe and a low sulfur content, then a saleable product might be obtained that makes both blocks valuable.

Five blocks are considered as an example in Table 1. If the contract dictates that the product should contain 59% and 0.2% of Fe and S, respectively, then none of the mentioned blocks can satisfy the requirements because in blocks 1 to 3, the sulfur content is more than the acceptable limitation, and in blocks 4 and 5, the Fe content is less than expected. However, if all of the blocks are blended together, the resulting mixture will contain 59.6% and 0.20% of Fe and S, respectively, which is in the acceptable range.

Table 1. Block blending example.

Block	Fe (%)	S (%)	Tonnage
1	62	0.28	3220
2	61	0.21	3200
3	60	0.22	3100
4	58	0.16	3050
5	57	0.15	3200

3. Block economic value

Computer-based methods for mine modeling and design usually start by developing a model of the orebody, dividing it into blocks and assigning a grade to each block (i.e. geological block model). GBM is determined based on the exploration data and estimation techniques, e.g. inverse distance and kriging. The size of the blocks depends upon the exploration drilling pattern and the size of the mining equipment.

Once grades have been assigned to the ore-blocks, the net income of the blocks is calculated. The costs of mining, milling, etc. should be calculated before proceeding to the economic optimization. The block is given a value with the assumption that it has already been exposed for mining. All the information required to determine the block value has been summarized below [22]:

(1) Tonnage;
(2) Grade;
(3) Anticipated metallurgical recovery;
(4) Content of penalty/deleterious elements;
(5) Content of valuable by-products;
(6) Current mining cost (+ overheads); and
(7) Current metallurgical extraction costs (e.g. for pyro-metallurgical processing, this includes smelting and refining, plus delivering to market costs).

When dealing with single-metal deposits, the block value can be calculated via Equation 1.

$$BEV = \{[m \times y \times P] - (m_o)(C_o) - (m_w)(C_w)\} \quad (1)$$

where:
m: Metal content per block
y: Recovery
P: Price of saleable product ($/kg)
M_o: Processed ore (ton/block)
C_o: Cost of ore processing ($/ton)
M_w: Tonnage of rock per block
C_w: Cost of mining per ton of rock ($/ton).

The above formula is applicable to the minerals such as gold and copper. When it comes to minerals like iron ore, coal, limestone, and other industrial minerals, the formula should be changed to evaluate the economic value of the blocks. The reason is that the aim of mining for the latter group of minerals (i.e. iron ore, coal, and limestone) is to achieve a pre-determined quality; the price obtained for blended ore is constant [23].

For polymetallic deposits, e.g. copper-molybdenum, copper-gold, and lead-zinc deposits, which include at least two valuable mine products (one is the main product and the other one is the co-product or minor product), BEV and, consequently, the cut-off grade are calculated using the Net Smelter Return or Metal Equivalent, which have been thoroughly discussed by Osanloo and Ataei [24–26] and Rendu [27]. In addition, Kakaei and Ataei have presented an approach to

determine the optimum cut-off grades and production scheduling in a multi-product open-pit mine [28]. However, in multi-element deposits, the existing elements are not necessarily co-products; i.e. the mine product is composed of the main element, and the other elements' effects on the final product could be positive or negative.
In iron-ore mines, as a multi-element deposit, iron is the main element and the other existing elements are not valuable. However, the value of the blocks is not entirely determined by the iron content. While the iron content is important, it is not the most critical element, as heavy penalties are imposed if the ore contains certain undesirable impurities. On the other hand, it could have some benefits if the material contains a certain percentage of beneficial components.
Iron ores can be classified based on chemical characteristics for commercial purposes. If the composition of the material meets the market specifications as mined, it is classified as direct-shipping ore (DSO), which requires no treatment before selling. However, if the ore characteristics do not meet the market specifications, treatments or mineral processing are necessary before selling the mine product.
BEV of this type of material (i.e. a multi-element deposit) is a function of many properties including the following:

- Selling price of the final commodity;
- Percentage of main elements;
- Tonnage of processed ore;
- Mining recovery;
- Processing recovery (if required);
- Mining costs;
- Processing costs (if required);
- Freight and selling costs;
- Percentage of penalty elements;
- Percentage of useful elements;
- Environmental and rehabilitation costs;
- Engineering, consulting, and administration costs; and
- Royalties and government taxes.

Many iron mines produce iron-ore lumps as a final product. Many mines also have a concentration plant. Therefore, the BEV calculation models are developed based on the two mentioned classes, i.e. the DSO and non-DSO (NDSO) deposits.

3.1. DSO deposits

The processing cost is not included in the BEV calculation for DSO deposits. Thus the profit of the blocks is calculated based on the metal content and other existing elements (including penalty and profitable elements) and the related costs (e.g. mining, environmental, engineering, royalty, and freight costs).
In these types of deposit, it is important to find the destination that is most compatible with the ore characteristics. Markets have different quotations for iron ore. The specifications of the required ore are determined for each quotation. Then it is important to find a suitable destination for the mine product. Consequently, BEV should be calculated based on the most compatible quotation along with maximizing the total pit value. BEV for DSO deposits can be calculated using Equations 2 through 6.

$$\mathrm{BEV} = \mathrm{income} - \mathrm{Costs} \quad (2)$$

where:

$$\mathrm{Income} = \mathrm{AP} \times B_T \times R_m \quad (3)$$

$$\mathrm{AP} = \mathrm{DP} + \mathrm{AF} \quad (4)$$

$$\mathrm{AF} = [(g_{Fe} - Fe_{min}) \times \beta_{Fe}] + [\sum (i_{max} - i) \times \beta_i] \quad (5)$$

$$\mathrm{Costs} = (M_c + Er_c + En_c + Cr_c + Bc + Rc + Fr_c) \times B_T \quad (6)$$

where:
AP: Adjusted price ($/ton)
B_T: Total tonnage of block (ton)
R_m: Exploitation recovery (%)
DP: Declared price ($/ton)
AF: Adjustment factor ($/ton)
g_{Fe}: Iron grade in block (%)
Fe_{min}: Minimum allowable grade of Fe (%)
β_{Fe}: Fe adjustment factor
i_{Max}: Allowable grade of element i (%)
i: Grade of element i in block (%)
β_i: Element i adjustment factor
Mc: Mining costs ($/ton)
Erc: Environmental costs ($/ton)
Enc: Engineering costs ($/ton)
Crc: Crushing and grinding costs ($/ton)
Bc: Blending costs ($/ton)
Rc: Royalties and government taxes ($/ton)
Frc: Freight and selling costs ($/ton).
The formula for BEV calculation in DSO deposit is as follows:

$$\mathrm{BEV} = \left[\mathrm{DP} + [(g_{Fe} - Fe_{min}) \times \beta_{Fe} + [\Sigma (i_{max} - i) \times \beta_i] \times R_m \times B_T\right] - [(M_C + Er_c + En_c + Cr_c + Bc + Rc + Fr_c) \times B_T] \quad (7)$$

The grades of the different elements are specified in the quotations for iron ore. If the product grade exactly matches the specified grades, then the price equals the price declared in the contract.

However, if the grades are more or less than the specified grades, various clauses determine the rewards or penalties applied to the price. In this case, an adjustment factor is used to determine the price to calculate BEV.

3.2. NDSO deposits

In NDSO iron-ore deposits, the Fe grade is less than the required properties or the grades of some undesirable elements are beyond the acceptable limits. In this case, an enrichment operation should be conducted to achieve the required properties. This operation affects the project's cost and, consequently, the economic value of the blocks. The BEV calculation is based upon the grades of the different elements and the costs related to reducing these elements to an acceptable limit. In this case, the general formula for BEV is the same as the formula for DSO deposits (Equation 2); however, the related parameters in this formula are calculated via Equations 8 through 12.

$$\text{Income} = AP \times T_c \tag{8}$$

$$AP = Pr_c + AF \tag{9}$$

$$AF = (Fe_E - Fe_c) \times \beta_{Fe} \tag{10}$$

$$\begin{aligned}\text{Costs} = {} & (M_C + Er_c + En_c + Cr_c + Bc + Rc + Fr_c) \times T \\ & + T_C \times CF_c + P_c\end{aligned} \tag{11}$$

$$P_c = (C_i \times T \times R_m) \tag{12}$$

where:
T_c: Concentrate tonnage in the block
Pr_C: Concentrate price ($/ton)
Fe_E: Enriched Fe grade (%)
Fe_C: Target Fe grade in concentrate (%)
P_c: Processing costs ($/ton)
CF_c: Freight costs of concentrate ($/ton)
C_i: Unit cost of element i removal ($/ton).
The formula for BEV calculation in NDSO deposit is as follows:

$$\begin{aligned}BEV = {} & [Pr_c + ((Fe_E - Fe_c) \times \beta_{Fe})] - \\ & [(M_C + Er_c + En_c + Cr_c + Bc + Rc + Fr_c) \times T \\ & + (T_C \times CF_c) + (C_i \times T \times R_m)]\end{aligned} \tag{13}$$

4. Grade uncertainty

According to the Vallee's report, 60% of the surveyed mines had an average rate of less than 70% of the designed production capacity during the first year of operation after start-up. Geological uncertainty was identified as a major contributor to these shortfalls [29]. Indeed, geological risks that originate from geological uncertainties cannot be eliminated; however, they can be minimized by gathering more exploratory data during the mining-development period [3].
One of the most widely used methods for grade estimation is kriging. Kriging is a geostatistical method that estimates the grade so that the mean squared error is minimized; therefore, the variance of the value estimated by kriging is smaller than the real but unknown variance. This smoothing of the true variability of the grade leads to overestimating the low grades and underestimating the high grades [30].
The best way to quantify grade uncertainty is with a conditional simulation. A conditional simulation is a generalization of the Monte Carlo-type simulation approach, which considers 3D spatial correlations [31, 32]. Sequential Gaussian simulation (SGS) is a solution to the kriging's smoothing problem. SGS is based upon a multi-Gaussian random function that is fully characterized by a single variogram function. The general steps involved in SGS are as follow [33]:
(1) Transform data to "normal space;"
(2) Establish grade network and coordinate system;
(3) Decide whether to assign data to the nearest grid node or keep separate;
(4) Determine a random path through all the grid nodes;
 (a) Search for nearby data and previously simulated grid nodes;
 (b) Construct the conditional distribution by kriging;
 (c) Draw a simulated value from the conditional distribution;
(5) Back transform and check results.
SGS is a rapid method for simulating a multivariate field. In the sequential conditional simulation, each entity is simulated sequentially, based on its normal conditional cumulative distribution. In this sequential method, the conditional state is generated using the original data as well as the simulated data from previous stages that are situated in the neighboring locations. It produces independent images of in-situ orebody grades called realizations [34]. These models represent the same deposit, and are all constrained to (a) reproduce all available information, and (b) be equally probable representations of the actual deposit.
After that, a pit limit will be optimized for each realization. This means that it is possible to define

several pit limits with respect to the number of realizations (Figure 1). In the traditional approach, a single-pit limit is determined. However, in uncertainty-based approaches, the single-pit limit is replaced with many different possible pit limits.

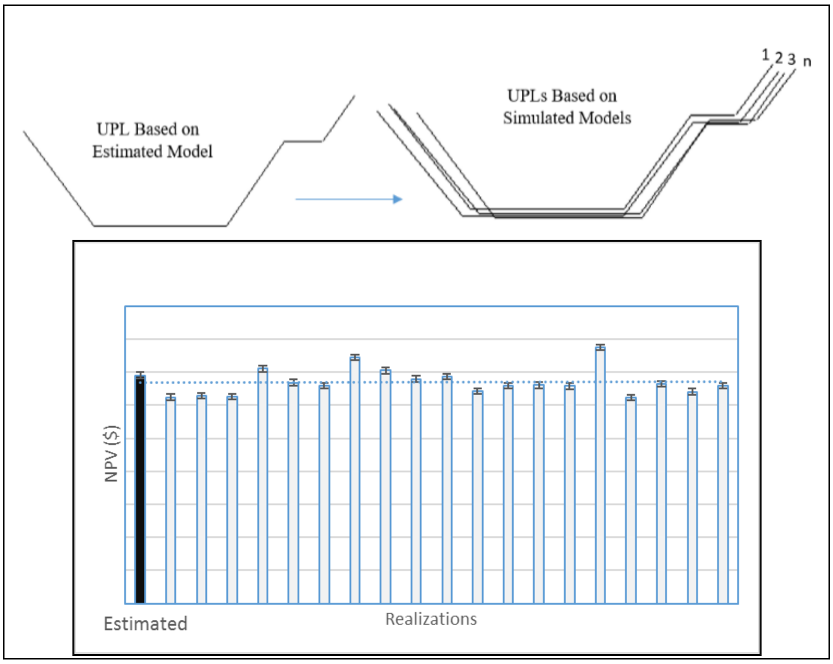

Figure 1. Simulation Concept.

5. UPL determination in estimated model

In this work, the Gol-e-Gohar iron-ore mine No. 2 was selected for determining the pit limits. The block dimensions were calculated to be 10×10×15 m. The number of blocks on each axis was 210×140×20 blocks (i, j, and k axes, respectively). The grades of P, S, and Fe were determined for each block. The shape of the deposit is depicted in Figure 2.

The block model includes up to 588,000 blocks, including ore and waste blocks, and the mineralization can be divided into two parts. In one part, the deposit has a low sulfur content, and can be considered DSO. Another part of the deposit contains a high percentage of sulfur, which does not meet the market requirements; hence, it is necessary to send these blocks to the concentration plant. The second part of the deposit is considered NDSO. Thus we selected this deposit as a case study to use both the above-mentioned formulae (i.e. DSO and NDSO) to calculate BEV.

The deposit has a high grade of Fe content, and it is possible to find a quotation compatible with the average grade of the deposit. Using the economic block value, each positive block is further checked to see whether its value can pay for the removal of overlying waste blocks. The related costs and recoveries are given in Table 2.

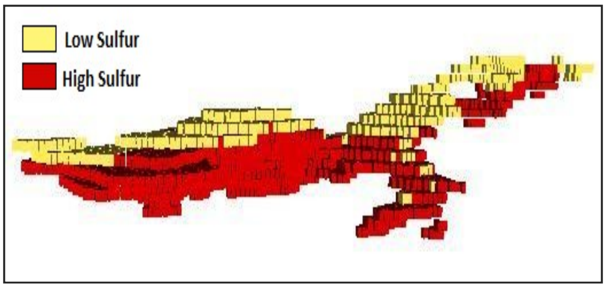

Figure 2. Orebody Model of Gol-e-Gohar Mine No. 2.

Table 2. Costs and recoveries.

Item	Unit	Value
Mining costs	$/ton	4.5
Crushing and grinding costs	$/ton	1.9
Environmental costs	$/ton	0.01
Engineering costs	$/ton	0.1
Blending costs	$/ton	0.1
Royalties and government taxes	$/ton	0.005
Waste removal cost	$/ton	2.8
Freight cost of ore	$/ton	10
Mining recovery	%	95
Processing costs	$/ton	12
Ore freight cost	$/ton	2
Concentrate freight cost	$/ton	10
Enrichment factor Fe	%	1.2
Processing grade recovery	%	88

5.1. UPL in DSO part

Based on the ton-grade curve of the Gol-e-Gohar mine (Figure 3), the Fe grade is between 52% and

62%. Therefore, it is possible to find different quotations for the mine product in the DSO part; however, it is important to select the one that is the most compatible with the deposit characteristics. In this regard, the deposit BEV is calculated based upon all the four quotations mentioned in Table 3 using Equations 3–6 to consider the given costs and recoveries. The adjustment factors for different elements are given in Table 4, and the summary results of the UPL determination are shown in Table 5 based upon the four intended destinations. Therefore, quotation No. 3, which gives the maximum pit value, is selected as a suitable destination for selling the mine product.

Table 3. Quotes for Iron ore [35].

Quotation	Fe (%)	S (%)	P (%)	Price ($/T)
1	61	<0.2	<0.2	52
2	60	<0.2	<0.2	50
3	59	<0.3	<0.3	48
4	58	<0.4	<0.4	44

Table 4. Price adjustment coefficients [36].

Element	Adjustment factor
Fe	1 dollar for each 1 percent
S	0.03 dollar for each 0.1 percent
P	0.03 dollar for each 0.1 percent

Table 5. Results of UPL determination.

Destination	Total NPV ($)
No. 1	65,175,730
No. 2	65,565,374
No. 3	66,226,504
No. 4	49,222,275

5.2. UPL in NDSO part

In the NDSO cases, the block value is determined based on the income of selling the products minus the costs. The costs of this part of the deposit include the concentration and enrichment costs, in addition to the costs considered in the DSO part.

The enrichment unit increases the Fe content of the blocks and reduces the amount of undesirable elements. The average percent of P is under the maximum allowable content declared in the quotes, so the aim of the enrichment process is to reduce the S content. The output material from the enrichment unit should have a lower S and increased Fe content. As the average S content is about 2.5%, a flotation process is required to remove or decrease S. The final product of this part of the mine (i.e. the NDSO part) is the iron-ore concentrate. It is assumed that the Fe and S grades in the concentrate are 67% and 0.2%, respectively. The average grade of Fe in the NDSO part is about 55% and the enrichment factor for Fe is 1.2, which is calculated as follows:

$$En_{Fe} = \frac{Fe_C}{g_{Fe}} = \frac{67}{55} = 1.2 \tag{14}$$

where:

En_{Fe}: Enrichment factor of element Fe

Fe_C: Target Fe grade in concentrate

g_{Fe}: Iron grade in block (%).

As an example, consider a block that contains 54% Fe. The Fe content of this block, after processing, would be 64.8%, which is calculated as follows:

$$Fe_E = g_{Fe} \times En_{Fe} = 54 \times 1.2 = 64.8$$

where:

Fe_E: Enriched Fe grade.

The price of iron-ore concentrates with an Fe content of 67% is about $86/ton. Therefore, it is possible to calculate the price of an ore block considering the concentrate price, Fe grades, mining, and processing recovery. The data required for calculating BEV in this part of the mine are given in Table 6. All other related costs and recovery elements are the same as the DSO part presented in Table 2.

Table 6. Costs and recovery.

Item	Unit	Value
Processing costs	$/ton	12
Ore freight cost	$/ton	2
Concentrate freight cost	$/ton	10
Enrichment factor Fe	%	1.2
Processing grade recovery (R_p)	%	88

Calculation of the mentioned block income is presented below:

$$AF = (Fe_E - Fe_c) \times \beta_{Fe}) = [(64.8 - 67) \times 1] = -2.2$$

$$AP = 86 - 2.2 = \$83.8 .$$

Equation 15 is used to calculate the weight of the concentrate obtained from each block.

$$R_p = \frac{Fe_C \times T_C}{g_{Fe} \times B_T \times R_m} \tag{15}$$

The weight of the considered block is 6150 tons, and the weight of the concentrate is

$$0.88 = \frac{67 \times T_C}{54 \times 6150 \times 0.95} \rightarrow T_C = 4143 \text{ (Tons).}$$

Equation 13 is used to calculate the concentrate weight obtained from each block in this part of the mine. As the weight and price of the block are determined, it is possible to calculate the related income achieved from each block.

$$Income = AP \times T_c = 83.8 \times 4143 = \$\ 347183$$

The related costs of all the blocks consist of two main parts: the costs that are not dependent on the grade (i.e. mining, crushing, taxes, blending, environmental, engineering, and ore transport costs) and the costs that change based on the grade of the undesirable elements. As the purpose of the flotation process is to reduce the S content in the product, the grade of S affects the cost of this operation. The cost of reducing the grade of S to an acceptable limit is calculated based upon the average grade of S in the feed material (i.e. 2.5%). The cost of flotation for reducing the S content from 2.5 to 0.2% is $12/ton of feed material. Thus the cost of reducing the sulfur content of blocks with 2% would be $9.6/ton, which is calculated as follows:

$$C_i = \frac{2}{2.5} \times 12 = \$9.6/\text{ton}$$

The independent costs, based on the information given in Table 2, consider the mining, crushing, taxes, blending, environmental, engineering, and ore transport costs, and total $8.615/ton. The costs that depend on the grade of sulfur are calculated based on the sulfur content of each block; for example, a block with 2% sulfur is $9.60/ton. Therefore, the costs related to the mentioned block are:

$$\text{Costs} = (8.615 + 9.6) \times 6150 + 4143 \times 10 = \$153252$$

Using Equation 2, BEV of the mentioned block can be calculated as follows:

$$\text{BEV} = \text{income} - \text{Costs} = 347183 - 153452 = \$193731$$

The NDSO part includes 5,758 blocks. The BEV of these blocks is calculated likewise; then UPL is determined using the NPV (net present value) scheduler software. In fact, the UPL determined in this part is an aggregate of the UPLs in the DSO and NDSO parts. In this case, by considering the effect of the existing elements in the orebody, the mineable reserve is 43,854,640 tons.

6. UPL without considering the elements' effects

The UPL of the Gol-e-Gohar mine was determined without considering the presence of existing elements to clarify the effect of considering their presence in the orebody. In this regard, the information given in Table 7 was used. It is basically the average costs related to mining and processing the ore blocks in the mine. This data was used to determine the UPL using the NPV scheduler software. The orebody was divided into two parts, and the related cost and income were adjusted based on the product type. For comparison purposes, the costs and prices are the same as in the previous part.

The total ore located in UPL is 42,465,750 tons. It is 1,388,890 tons less than the case where the effect of the multi-elements in the deposit was considered. This means the final pit limits are different when the effects of the existing elements in the orebody are considered. Taking the effects into account leads to a better recognition of UPL and, consequently, could help achieve a better production schedule.

Table 7. Information for UPL determination.

Item	Unit	Value
Mining costs	$/ton	4.5
Crushing and grinding costs	$/ton	1.9
Environmental costs	$/ton	0.01
Engineering costs	$/ton	0.1
Blending costs	$/ton	0.1
Royalties and government taxes	$/ton	0.005
Processing costs	$/ton	12
Waste removal cost	$/ton	2.8
DSO product price	$/ton-ore	48
NDSO product price	$/ton-con	86
Mining recovery	%	95
Processing recovery	%	88

7. Grade uncertainty in UPL

The initial information for generating the block model was obtained through exploration drilling. The grade of the blocks was estimated using methods such as kriging and inverse distance. However, the volume of the exploration work is limited and it causes the estimated block model to be non-deterministic. Thus several block models can be generated in dealing with this uncertainty. All generated models represent the deposit under study with equal probability.

Conditional simulation is an effective method of generating realizations of the orebody. Out of the conditional simulation algorithms, e.g. sequential Gaussian simulation (SGS), probability field simulation (PFS), and simulated annealing (SA), SGS is the most efficient method for obtaining the grade distribution. Therefore, in this work, SGS was used to generate twenty different deposit realizations. In these realizations, the grades of Fe, S, and P were generated for each simulated model. The grade of each element contained uncertainty, which was reflected by the variation from one simulated realization to another.

An important point is that the block density depends on the percent of existing elements. The relation between the density and the existing elements is calculated using multiple variable regression on the data obtained from exploration boreholes. Then the density of the blocks in each realization is calculated based on the grades of the elements. The density is important for calculating the tonnage of each block, and it affects the total tonnage of the deposit and the UPL as well. The relationship using multiple variable regression between the density and grades of different elements can be expressed as Equation 16.

$$\rho = -0.00086(P) - 0.00436(S) + 0.0367(Fe) + 2.184 \quad (16)$$

where:
ρ: Density (ton/m^3).

In all realizations, the density of each block is calculated using Equation 14 based on the grades of each element. Then BEVs are calculated using Equations 3 to 6 in the DSO part and Equations 7 to 11 in the NDSO part. The UPL of each realization is determined based on the BEVs where the grades of existing elements are considered. The generated economic block models are used to determine UPL using the NPV scheduler software. The total ore tonnages of the deposit in the estimated model and the realizations are shown in Figure 3.

Table 8. Tonnage of scenarios.

Confidence Level	Tonnage (1000 ton)			Stripping Ratio
	DSO part	NDSO part	Total	
100%	13,387	6,993	20,380	3.51
80%	16,557	13,858	30,414	3.27
60%	17,222	18,156	35,378	2.97
40%	17,755	23,068	40,823	2.72
20%	18,370	28,040	46,410	2.53

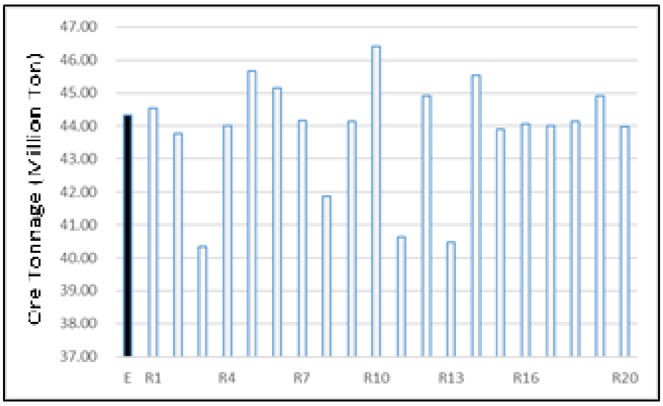

Figure 3. Total Ore Tonnage in UPL (E: estimate, R_i: Realization i).

The total ore tonnage varies between 40 and 46.5 million tons. The mean tonnage is 43.8 million with a standard deviation of 1.6 million tons. Based on the mean and standard deviation, it is possible to calculate the confidence interval using Equation 17:

$$(\bar{T} - \frac{\sigma}{\sqrt{n}} Z_{1-\frac{\alpha}{2}}, \bar{T} + \frac{\sigma}{\sqrt{n}} Z_{1-\frac{\alpha}{2}}) \quad (17)$$

where:
$\bar{T}$: Tonnage mean
σ:: Standard deviation
n: Number of estimates
1-α: Desired confidence level.
For the Gol-e-Gohar iron-ore mine, the ore tonnage is in the range of 40.6 to 47 million tons with a confidence level of 95 percent. To define the UPL probability with different confidence levels, the UPL with the maximum occurrence probability is the one that is extracted by all the realizations. It means that these blocks can be extracted with the maximum confidence level. The UPL of scenarios with different probabilities including 100, 80, 60, 40, and 20% are illustrated in Figure 4. A summary of the results for the UPLs corresponding to 100, 80, 60, 40, and 20% is given in Table 8. Figure 5 shows the stripping ratio (W/O) of the different scenarios. The W/O of the mine is between 3.2 and 3.6 with an average of 3.4. The stripping ratio of the UPL with the maximum confidence level is 3.5.

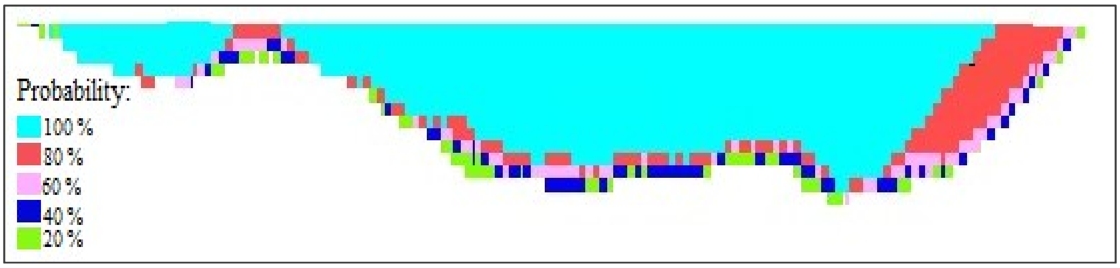

Figure 4. UPL of Scenarios.

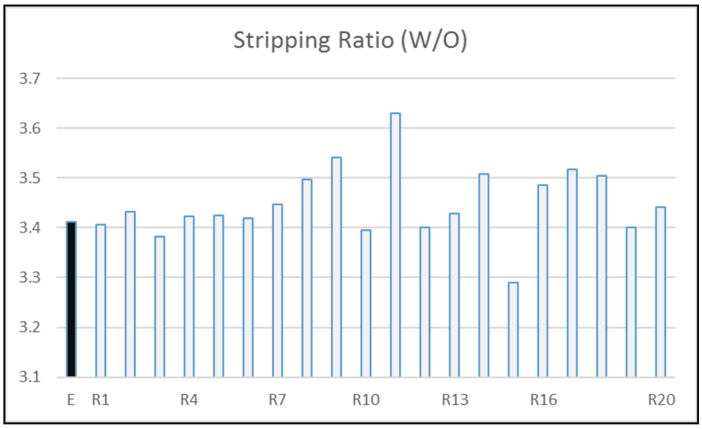

Figure 5. Stripping Ratio (E: estimate, R_i: Realization i).

8. Discussion

In this paper, the effects of the different elements on the final product and the block economic value were described considering the blending of low-grade and high-grade materials in different elements. In the exist models for BEV calculation, the value of each block was calculated individually but in the multi-element deposits, the blocks were blended to achieve a pre-determined quality and the value of the block had to be considered based on the resulting blend.

Therefore, two models were developed for iron-ore deposits in the cases of direct-shipping and non-direct-shipping ores. In these models, Fe is the main element and P and S are the penalty elements. These models consider the positive effect of the main element and the negative effects of the penalty elements on the block value. The results obtained show that this method produces a more realistic mineable reserve and pit value.

The model is applied in the case study of an iron-ore deposit. The mineable reserve of the deposit is evaluated using the proposed method and commonly available methods. The ultimate pit limit of the deposit, which was determined using the available methods underestimated the minable reserve by approximately 1.3 million tons. In the proposed method, a more realistic minable reserve was determined, which means that the resource efficiency was improved.

Moreover, pit optimization in the case of multi-element deposits was studied in two respects. The first option was the assumption of direct shipping, where the extracted material required no more treatment. In the second case, the treatment option was considered, and then the mineable reserve became 43,854,640 tons.

The direct-shipping material has several selling opportunities. Each customer requires a product with a specific quality and price. At the time of mine planning and design, the mine designer should consider all possible opportunities to select the most profitable option that best fits the mine condition. The approach presented here considers different destinations/customers for the ore blocks. It enables a mine designer to select the most profitable destination for a product. In this work, four different destinations were analyzed and the one most compatible with the studied deposit was determined.

Furthermore, pit optimization was studied in the presence of grade and tonnage uncertainty. Sequential Gaussian simulation was a suitable tool for generating the different ore-deposit realizations. The method was used to generate 20 equally probable ore-deposit realizations. Three elements including Fe, P, and S were simulated. Then the density of each block was calculated in each realization with respect to the simulated grades. This means that each block could have a different grade and tonnage in the realizations and it might affect the pit-limit optimization.

The ultimate pit limit of all the realizations was determined based on the most profitable economic setting, which resulted in 20 pits. Some blocks were located in all the pits, and some blocks could not be profitably mined by all the pits. With this in mind, the probability of each block being mined could be calculated. Thus five pits were defined for different probabilities of 20, 40, 60, 80, and 100%. The pit corresponding to 100% was the smallest pit and the one corresponding to 20% had the largest pit limit. The ore extraction should start in the pit with 100% probability. This pit contains 20.3 million tons of ore with a stripping ratio of 3.5. We were more confident starting the mining operation within the smallest pit, while the largest pit should be considered as a guide for surface-facility locating.

9. Conclusions

Ultimate pit-limit determination is one of the major steps in mine planning and design. The pit limit is the borderline, indicating that the blocks within this limit are profitable for extraction. In addition to the economic parameter, the pit limit is important for the site selections of dumps and infrastructures. A missing pit-limit determination could impose additional costs on the project. Different methods exist for pit-limit determination and most of them use the economic block model as the input data. For single-metal deposits, the block economic value is calculated based on the income of the metal content and related costs. However, multi-element deposits (e.g. iron ore, coal or feed material for cement manufacture) include an inherent task of blending the run-of-mine materials in such a manner that the resulting mix meets the quality and quantity specifications.

This paper presented a new methodology for calculating the block economic value based on the grade of the main element as well as the grades of other existing elements. The grade of the other elements could have a negative or positive impact on the mine product and the value of the blocks as well. Therefore, in the proposed methodology, an adjustment factor was used for the mine product income, and the economic value of the blocks was calculated accordingly.

The developed model was applied to an iron-ore deposit. The deposit consisted of two parts. One part had a low sulfur content and could be considered as a direct-shipping ore. Another part had a high sulfur grade and, consequently, could not be sold without processing. Then the block economic values for the two different parts were calculated based on the ore characteristics, and the ultimate pit limits were determined.

The results obtained were compared with the pit limit that did not consider the effect of the contaminating elements; it was found that considering the effect of other elements resulted in different limits. In this case, the ultimate pit limits increased when the effects of the other elements were considered.

Moreover, the effects of the grade and tonnage uncertainty were investigated by generating 20 equal-probability realizations of the block model using sequential Gaussian simulation. For all the realizations, the pit limits were determined, and the confidence levels of the pits were calculated. The smallest pit with 100% confidence included 20.3 million tons of ore, and the largest pit with 20% confidence contained approximately 46 million tons of ore. The smallest pit was suitable for starting a mining operation and the largest pit could be used to select sites for mine infrastructures and dumps.

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تعیین محدوده نهایی در معادن چند عنصری با استفاده از یک روش جدید برای محاسبه ارزش اقتصادی بلوک‌ها تحت شرایط عدم قطعیت عیار

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چکیده:

در معادن تک فلزی، ارزش اقتصادی بلوک‌ها بر اساس میزان فلز محتوی و هزینه‌های مرتبط محاسبه می‌شود. در این معادن محاسبه ارزش اقتصادی بلوک‌ها تنها بر اساس عنصر سودآور محاسبه شده و تأثیر سایر عناصر همراه در نظر گرفته نمی‌شود. در معادن چند عنصری، درصد سایر عناصر موجود در بلوک نیز در ارزش اقتصادی بلوک‌ها تأثیرگذار است و با در نظر گرفتن اختلاط بلوک‌ها، باید تأثیر سایر عناصر همراه نیز لحاظ شود. درصد عناصر همراه و روش اختلاط تأثیر قابل توجهی در کیفیت و ارزش محصول نهایی معدن دارد. در این پژوهش، روش جدیدی برای محاسبه ارزش اقتصادی بلوک‌ها در معادن چند عنصری با در نظر گرفتن دو گروه عناصر سودآور و نامطلوب و همچنین تأثیر اختلاط ارائه شده است. با محاسبه ارزش اقتصادی بلوک‌ها با روش جدید، محدوده نهایی معدن با در نظر گرفتن این شرایط تعیین می‌شود. مدل ارائه شده در معدن سنگ آهن گل گهر شماره ۲، استفاده شده و محدوده نهایی آن تعیین شده است. با استفاده از روش جدید محاسبه ارزش اقتصادی بلوک‌ها، ذخیره قابل استخراج معدن، افزایش ۳ درصدی دارد. به منظور بررسی تأثیر عدم قطعیت عیار در ارزش اقتصادی بلوک‌ها، با استفاده از روش شبیه‌سازی گوسی متوالی، بیست تحقق مختلف از ذخیره معدن، تولید شد. ارزش اقتصادی بلوک‌های این تحقق‌ها با مدل ارائه شده، محاسبه و سپس محدوده نهایی آن‌ها تعیین شد. بر اساس شبیه‌سازی‌های انجام شده، ذخیره قابل استخراج معدن بین ۲۰,۳۸۰ تا ۴۶,۴۱۰ میلیون تن متغیر است. از این اطلاعات می‌توان برای برنامه‌ریزی شروع استخراج و جانمایی تأسیسات سطحی استفاده کرد. به این ترتیب که کوچک‌ترین پیت (پیت با احتمال وقوع ۱۰۰٪) برای شروع عملیات استخراج منظور می‌شود و جانمایی تأسیسات سطحی باید خارج از بزرگ‌ترین پیت ممکن انجام شود.

کلمات کلیدی: ارزش اقتصادی بلوک‌ها، ذخایر چند عنصری، محدوده نهایی، عدم قطعیت عیار، معدن روباز.