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The first issue of the journal "Podzemni radovi" (Underground Mining Engineering) was published back in 1982. Its founders were: Business Association Rudis - Trbovlje and the Faculty of Mining and Geology Belgrade. After publishing only four issues, however, the publication of the journal ceased in the same year.

Ten years later, in 1992, on the initiative of the Chair for the Construction of Underground Roadways, the Faculty of mining and Geology as the publisher, has launched journal "Podzemni radovi". The initial concept of the journal was, primarily, to enable that experts in the field of underground works and disciplines directly connected with those activities get information and present their experiences and suggestions for solution of various problems in this scientific field.

Development of science and technique requires even larger multi-disciplinarity of underground works, but also of the entire mining as industrial sector as well. This has also determined the change in editorial policy of the journal. Today, papers in all fields of mining are published in the "Underground Mining Engineering", fields that are not so strictly in connection with underground works, such as: surface mining, mine surveying, mineral processing, mining machinery, environmental protection and safety at work, oil and gas engineering and many others.

Extended themes covered by this journal have resulted in higher quality of published papers, which have considerably added to the mining theory and practice in Serbia, and which were very useful reading material for technical and scientific community.

A wish of editors is to extend themes being published in the "Underground Mining Engineering" even more and to include papers in the field of geology and other geo-sciences, but also in the field of other scientific and technical disciplines having direct or indirect application in mining.

The journal "Underground Mining Engineering" is published twice a year, in English language. Papers are subject to review.

This information represents the invitation for cooperation to all of those who have the need to publish their scientific, technical or research results in the field of mining, but also in the field of geology and other related scientific and technical disciplines having their application in mining.

Editors

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Original scientific paper

RANKING OF THE MOST IMPORTANT CRITERIA FOR THE SELECTION OF THE MINING METHOD FOR NON-STRATIFIED DEPOSITS

Dejan Bogdanović[1], Hesam Dehghani[2], Farshad Saki[3], Slavica Miletic[4]

Received: February 21, 2024 **Accepted:** March 8, 2024

Abstract: Selecting and planning of the mining method for non-stratified deposits is the most delicate and complex process on which the success of the mine depends. This procedure is basically a multi-criteria decision making problem in which the aim is to select the best mining method from many alternatives. The aim of this paper is to show the influence of many factors (criteria) in the selecting of the most suitable mining method and to determine their influence on this process. The eight groups of influencing factors, i.e. criteria, were taken into account – geometric data on the ore body, the mechanical characteristics of the massif, the ore reserves, the situation on the surface of the terrain, the workforce, the possible environmental hazards, the market conditions and the safe working conditions. The AHP method was used for ranking these criteria. The ranking was carried out by mining experts and managers of various underground mines in Serbia using the group decision method. The results obtained show a clear distinction between the individual criteria when selecting the best mining method. Furthermore, the results clearly show the importance of the ranking process in determining the most influential criteria in this very complex process.

Keywords: Criteria Ranking; Mining Method; AHP; Non-Stratified Deposits

1 INTRODUCTION

Mining method selection is a time-consuming and difficult process that requires a high level of expertise and experience. This process can be a difficult task for mining engineers and managers. For a proper and effective evaluation, the decision maker may need to analyze a large amount of data and consider many factors – criteria.

The criteria that influence the selection of mining method are not all equally important, nor are they all equally reliable; some change, others are constant. Some parameters

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exclude some methods or technologies, in some situations some of them have no meaning and so on.

There are numerous methods that have been developed in the past for the selection of mining methods. The first numerical approach for the selection of mining methods was proposed by Nicholas (1981 and 1992). In this method, different mining methods are evaluated based on the ranking of certain input parameters –criteria. The mining method with the highest summed result is selected. Later, Nicholas proposed some modifications that include the weighting of various categories, such as ore geometry, ore zone, hangingwall and footwall.

Miller et al. (1995) developed the UBC method as a modification of the Nicolas method. The main weakness of these approaches is that the importance of the individual selection criteria was not taken into account.

A modern approach views the selection of mining methods as a multi-criteria decision problem (MCDM) with a finite number of alternatives that must be ranked taking into account many different and conflicting criteria. The advantage of these methods is that they can take into account both financial and non-financial criteria. The best known of these methods are scoring models, Analytic Hierarchy Process – AHP, Analytic Network Process – ANP, TOPSIS, ELECTRE, PROMETHEE, ELECTRE, MAUT, MACBETH, VIKOR, TODIM, Grey, MULTIMOORA, and MAHP. Multi-criteria decision making methods (MCDM) such as AHP and Fuzzy AHP, which are used in the literature for mining method selection, make the evaluations using the same rating scale and preference functions based on the criteria. Accordingly, Ataei et al. (2008) used the AHP approach for mining method selection. On the other hand, Bitarafan & Ataei (2004) used various fuzzy methods as an innovative tool for criteria aggregation in mining decision problems. Alpay & Yavuz (2009) have also proposed a combination of AHP and fuzzy logic methods for mining method selection. Samimi Namin et al. (2008) used fuzzy TOPSIS method for optimal mining method selection. Also, Bogdanovic et al. (2012) used integrated AHP and PROMETHEE method for selection of the most appropriate mining method. Saki et al. (2020) proposed a new methodology to find the most suitable MCDA techniques for selecting the optimal underground mining method. First, a list of fifty parameters, including geomechanical, geometrical, technical, economic, environmental and social parameters, were considered for the selection of the optimal mining method. Then the most influential parameters, including the thickness, the RMR value of the hanging wall and the production rate, were selected as the most important parameters according to the experts' opinions on the subject.

All this indicates that the selection of the optimal underground mining method depends primarily on influential factors – criteria. Therefore, it is very important to identify all criteria and determine the degree of their influence on the selection of mining method. In order to determine their influence, they must be ranked, which is done in this paper. At this point, it must be noted that the ranking was generally done for for non-stratified ore deposits using the group decision method. For each individual case of mining method selection, the ranking should be made according to the conditions in the particular mine or deposit.

2 MATERIALS AND METHODS

The studies are being carried out in various underground mines in Serbia and Iran. Different mining methods are used in these mines. The conditions are also very different, from ore geometry, ore type and reserves, mechanical properties of the rock to market conditions and safe working conditions. In addition, many mining experts and managers were involved in this study, providing a very good basis for obtaining high-quality results.

2.1 AHP method

AHP is a quantitative technique proposed by Saaty (1980). This technique develops and analyzes a multidimensional hierarchical structure of goals, criteria and alternatives. It calculates the strength of each criteria, compares the alternatives with each criteria, and ranks all alternatives. AHP uses a comparison matrix to evaluate each criteria and the alternatives based on scores from 1 to 9 (Table 1). On this basis, the evaluation leads to a final ranking of the alternatives.

Accordingly, only the first step is carried out in this paper with the aim of evaluating the most important group of criteria for the selection of the mining method for non-stratified deposits.

Table 1 Pair-wise comparison scale for AHP method

Verbal Judgement	Numerical Rating
Equally preferred	1
Moderately preferred	3
Strongly preferred	5
Very strongly preferred	7
Extremely preferred	9
2, 4, 6 and 8 are intermediate values	

2.2 The research method

The original research method was developed to rank the criteria and evaluate the degree of their influence on the selection of mining method for non-stratified deposits. The research method comprises the following four steps: (1) data collection, (2) defining the most important criteria for the selection of the mining method for non-stratified deposits, (3) AHP calculations and (4) results and discussion (Figure 1).

The research began with interviews with mining experts and managers. The questions were designed to collect the necessary data on the most important criteria for the selection of the mining method. The final list of the most important criteria was then determined. In the next step, the most important criteria were ranked using the AHP to determine their influence on the mining method selection process. Once the results of the ranking were obtained, the most important criteria were identified and analysed to provide a useful basis for future mining method selection and to better understand the priorities in this process.

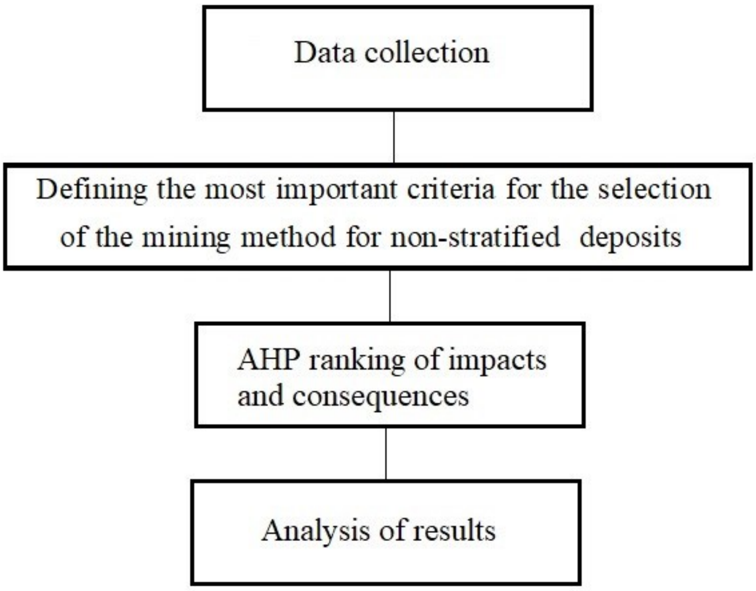

Figure 1 Schematic overview of the research method

3 THE OBITAINED RESULTS

3.1 Data collection

As already mentioned, the research began with interviews with mining experts and managers. This step took the longest and represents the basis for the further research steps, whereby the corresponding questionnaire was compiled in such a way that the most important criteria were achieved on the basis of the answers received.

3.2 Defining the most important criteria for the selection of the mining method for non-stratified deposits

In this step, the criteria identified are grouped according to their nature and their influence on the choice of mining method. This is done through a discussion between mining experts and managers until a consensus is reached for each group of criteria. On this basis, eight criteria groups were identified, namely:

- C1 – geometric data about the ore body (the shape – morphological type of the ore body, its dimensions and its spatial location),
- C2 – the physical-mechanical characteristics of the massif (strength and deformability, rupture assembly and stress state as well as hydrogeological conditions),
- C3 – the ore reserves, the content and distribution of the usable components of the ore, the mineralogical composition and other data that determine the value of the ore and its primary processing,
- C4 – the situation on the surface of the land (the presence of infrastructure, residential, industrial or other buildings under protection, as well as the presence of permanent or occasional watercourses and reservoirs),
- C5 – the workforce (presence of trained or untrained miners, their level of training and the cost of the workforce under the given conditions),
- C6 – the potential environmental hazards (impact on the existing ecosystem and hydrological system, etc.),
- C7 – the market conditions (value of the useful component, price stability, expected supply and demand, risks, etc.),
- C8 – the safe working conditions (safety conditions, occupational health and safety risks and their prevention, health protection of workers).

3.3 AHP calculations

The AHP calculations were carried out using the group decision method (authors with mining experts and managers of mining companies). The aggregation of individual judgments (AIJ) method was used for group decision making. Figure 2 shows the hierarchical structure of the AHP problem. The criteria were discussed and ranked until a consensus was reached for each evaluation using the scale shown in Table 1. The comparison matrix (8x8) is shown in Table 2. SuperDecisions software was used for the AHP calculations. Figure 3 shows the calculation results obtained from the comparison matrix.

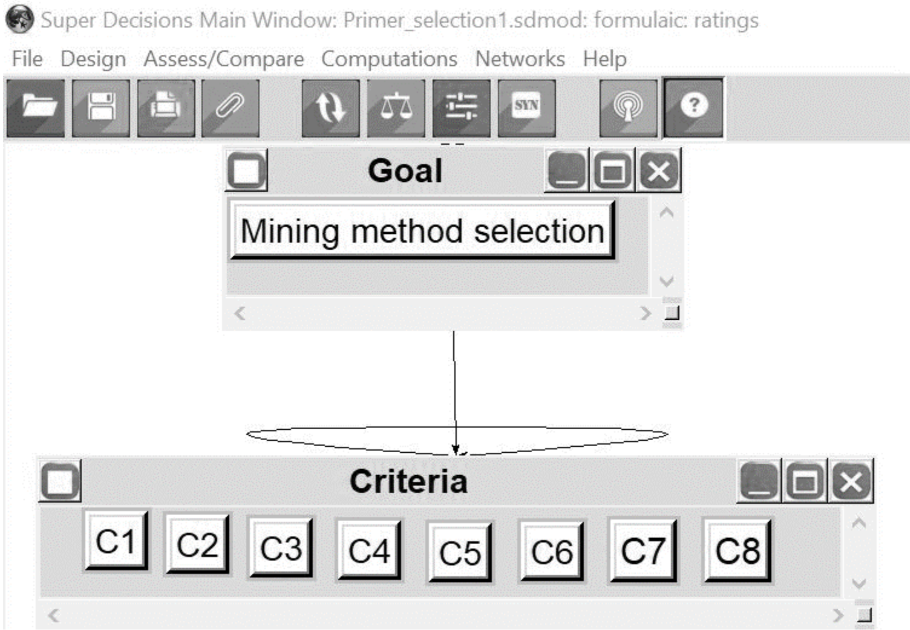

Figure 2 Hierarchical structure of the AHP problem

Table 2 Criteria comparison matrix

Criteria	C1	C2	C3	C4	C5	C6	C7	C8
C1	1	2	3	4	5	4	1	4
C2		1	3	4	4	4	2	3
C3			1	3	4	2	1/4	1
C4				1	1	1/2	1/6	1
C5					1	1/4	1/5	1/2
C6						1	1/2	1
C7							1	3
C8								1

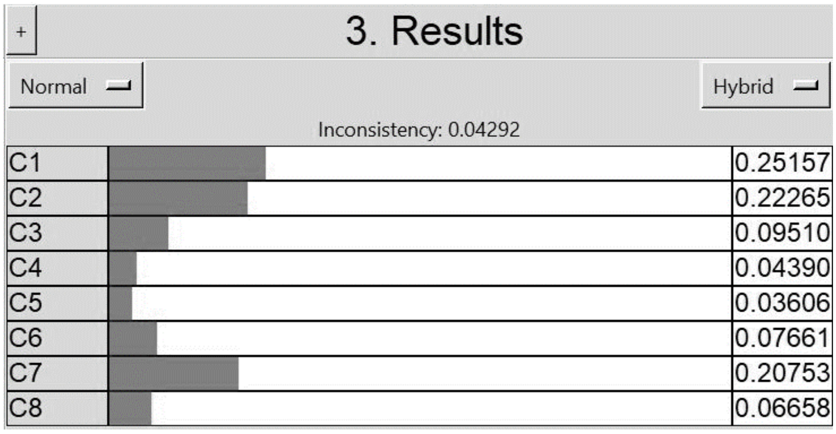

Figure 3 Results obtained by AHP calculations

3.4 Results and discussion

The results obtained clearly show which criteria have the greatest influence on the selection process of mining methods. The results show that it is possible to divide the criteria into three groups in terms of their influence on the mining method selection process. The first group includes criteria whose weight coefficients are greater than 0.2 (20%). These are the criteria C1 – geometric data about the ore body, C2 – physical-mechanical characteristics of the massif and C7 – market conditions. The second group of criteria comprises those whose weighting coefficients are between 0.05 and 0.1 (5% to 10%). These are C3 – ore reserves, C6 – potential environmental hazards and C8 – safe working conditions. The third group comprises the least influential criteria – C4 – the situation on the surface of the land and C5 – the workforce.

In the first group, the strongest criteria is C1 with a weighting coefficient of 0.25157. The shape of the ore body, its dimensions and its spatial location are of essential importance for the selection of the appropriate mining method, the construction and dimensions of the pit, its layout and the possible mining capacity. In second place is criteria C2 (weight coefficient 0.22265). Which mining method is used depends largely on the physical-mechanical characteristics of the massif (ore body and surrounding rock). For example, it is possible to use unsupported methods in hard and solid rock. Otherwise, it is possible to apply other methods (unsupported methods or caving methods) in accordance with the given characteristics of the rock massif. In third place is criteria C7 – the market conditions. The value of the useful component, price stability, expected supply and demand and risks are all elements that influence the choice of mining method. For example, the value of the useful component in the ore affects the selection process, so that richer deposits allow the use of more expensive methods that have a higher utilization rate, less impact on the surface of the land, less impact on the environment, etc., and vice versa.

The second group of criteria is significantly weaker than the first. Criteria C3 is in first place here. The ore reserves are an important factor in the selection of mining method, and it depends on them whether or not a method is used that enables a high production capacity. The planning process, the duration of mining, etc. also depend on the ore reserves. Criteria C6 is in second place. Potential environmental hazards are a factor that is becoming increasingly important. The aim is to operate mines in such a way that the impact on the environment is minimized. Accordingly, when selecting the mining method, preference should be given to the mining method that poses the least risk to the environment. Criteria C8 is in third place. Safe working conditions (safety conditions, occupational health and safety risks and their prevention, health protection) are a very important criteria that must be taken into account when selecting the excavation method. Which excavation method is chosen depends on which measures and how they are applied, how high the risks are in the workplace, etc.

The third group comprises the least influential criteria – C4 and C5. The situation on the surface of the land (the presence of infrastructure, residential, industrial or other buildings under protection, as well as the presence of permanent or occasional watercourses and reservoirs) is a factor that influences the selection of mining method. If it is necessary to protect surface objects, this significantly limits the choice of the optimal mining method. Otherwise, additional costs for relocation of facilities or reimbursement of costs will be incurred if methods are selected that jeopardize the surface of the site. As for the workforce, it was ranked as the least influential criteria. The reason for this is that this criteria can be highly influenced by the process of recruitment, provision of good training, adequate wages, good working conditions, etc.

4 CONCLUSION

Here, the AHP method was used to evaluate the most important criteria for selecting the mining method for non-stratified deposits. Eight criteria were considered: – C1 (geometric data on the ore body), C2 (physical-mechanical characteristics of the massif), C3 (the ore reserves), C4 (the situation on the surface of the land), C5 (the workforce), C6 (the potential environmental hazards), C7 (the market conditions) and C8 (the safe working conditions).

Based on the ranking results obtained, the criteria can be divided into three groups according to how influential they are on the selection of mining method. The first, most influential group includes the following criteria: C1 (geometric data about the ore body), C2 – physical-mechanical characteristics of the massif) and C7 (market conditions). The second group, which is in the middle in terms of influence on the choice of mining method, comprises the following criteria: C3 (ore reserves), C6 (potential environmental hazards) and C8 (safe working conditions). The third group comprises the least influential criteria: C4 (the situation on the surface of the land) and C5 (the workforce).

The results of the ranking of the most important criteria for the selection of the mining method for non-stratified deposits can serve as a guide for mining experts and managers for the correct selection of the optimal mining method in their mines.

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Original scientific paper

HYDROPHOBIZED LIMESTONE AS FILLER IN POLYMER MATERIALS

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Abstract: The article shows changes in certain mechanical properties of polyvinyl chloride (PVC) depending on the quality of added limestone as a filler. Natural limestone and limestone hydrophobicized with stearic acid were added. Modification experiments were carried out with "wet" and "dry" processes in order to find out the required amount of stearic acid for a complete surface coating of limestone - degree of coating 99.90%. Coating of the limestone surface was achieved in the "wet" process with 1.5% stearic acid, while in the "dry" process the same degree of coating was achieved with 3% stearic acid. A significant amount of both "wet" and "dry" modified limestone was prepared. Such a product was added to PVC mixture in order to investigate mechanical properties of the obtained PVC product. Research into the mechanical properties of PVC has shown that PVC containing limestone modified by the "wet" process exhibits better mechanical properties than that containing limestone modified by the "dry" process. For example, PVC obtained from a mixture containing limestone modified by the "wet" process with 1.5% stearic acid shows a better tensile strength of 54.20 MPa, while limestone modified by the "dry" process with 3% stearic acid shows a tensile strength 53.20 MPa.

Keywords: PVC; limestone; calcite; hydrophobization; mechanical properties

1 INTRODUCTION

Considering the structure of polymer materials, it is expected that in polymer without fillers, fracture occurs where its texture is weakest, i.e. at the highest strain spot, as a result of the deformation process. Local fracture further spreads out through the entire material, Figure 1. demonstrates breaking of Van der Waals connection between two chains of PVC under the influence of tension force on the example of polyvinyl chloride as polymer without filler.

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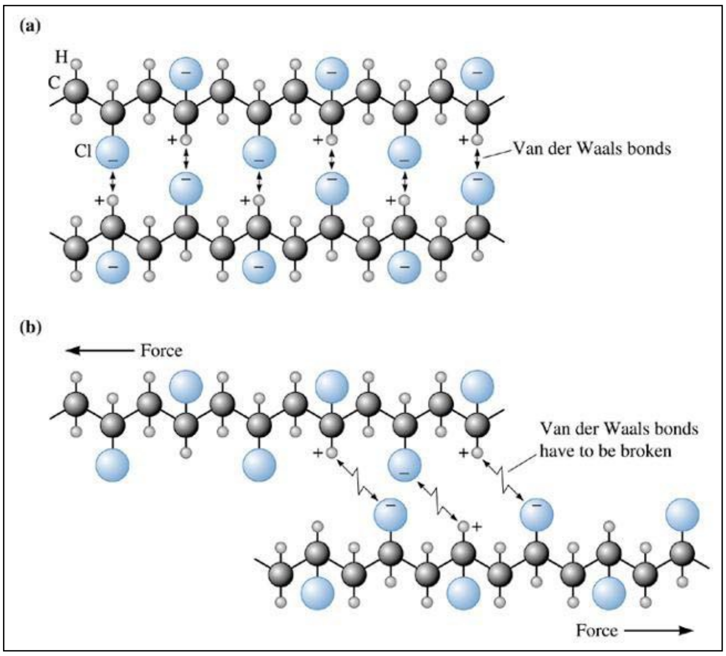

Figure 1 Scheme of molecular connections in polyvinyl chloride (PVC) a) Van der Waals connections between two PVC chains, b) breaking of Van der Waals connections between two PVC chains under the influence of tension force

In polymer with filler, fracture can occur in the matrix itself, in the interspace polymer/filler or within agglomerates for some reason formed in filler. Elongation limits greatly depend on the quantity of the filler, but not on the quantity of the additive for filler modification too. On the other hand, tensile strength and elongation at fracture are properties observed at large deformations in product, since they show its structural properties better there, (Kovačević, Lučić and Cerovečki, 1997; Kovačević et.al., 1996; Kovačević et.al., 1994). Phenomena and mechanisms occurring at calcite surface during limestone surface modification authors of this paper clarified in the paper, (Mihajlović et. al., 2009a; Mihajlović et. al., 2009b). This paper provides findings obtained by further observation of the limestone surface modification

2 EXPERIMENTAL

For experimental research of this paper limestone 95% -10 μm and of the following chemical composition: 53.77% CaO; 0.084% Fe_2O_3; 0.035% Al_2O_3; 1.79% MgO; 0.24% SiO_2; 0.027% Na_2O; 0.036% K_2O; 44.05% was used. Properties of stearic acid used for limestone surface modification in experiments were the following: Molecular formula $CH_3(CH_2)_{16}COOH$; molecular weight, g/mol 284.47; density, g/ml 0.847; dissociation constant, pKa 5.7; solubility in water, g/100 ml 0.034 (25ºC), 0.1 (37ºC); solubility in alcohol, g/100 ml 2.5 (cold); solubility in ether, $CHCl_3$, CCl_4, CS_2 very soluble.

3 RESULTS AND DISCUSSION

Since previous experiment showed that complete limestone surface coating with stearic acid is achieved with 1.5% of stearic acid by the “wet” process and 3% by the “dry” process, 5 kg of limestone modified this way were prepared, (Mihajlović, Sekulić and Petrov, 2005; Mihajlović et. al., 2009a; Mihajlović et. al., 2009b; Mihajlović et. al., 2012). This product was mixed with PVC in order to make panels, the mechanical properties of which were to be tested. This was done in order to investigate which modification process provides better ultimate effects in PVC final products, even though both were completely coated with stearic acid on the surface “TIRA-test 2300“ universal testing machine with 10 kN load cell, 0-10 kN testing range, accuracy class 1, was used for assessment of mechanical properties of PVC mixture. Testing material was prepared by milling process in the shape of type A test tube. Prepared samples were tested for tensile strength, tensile yield strength, tensile elongation and elongation at break according to ISO 527-2 standard. The obtained results are presented in Table 1.

Table 1 Mechanical properties of PVC mixtures

Mechanical properties	Method	Unit	Samples				
			PVC+C	PVC+CD-1.5	PVC+CW-1.5	PVC+CD-3	PVC+CW-3
Tensile strength	ISO 527-2	MPa	52.70	52.70	54.20	53.20	53.60
Tensile yield strength	ISO 527-2	MPa	36.50	36.20	38.20	37.90	39.10
Tensile elongation	ISO 527-2	%	4.30	4.40	4.35	4.46	4.30
Elongation at break	ISO 527-2	%	35.90	35.40	29.30	22.50	24.60

The samples are presented in Table 1 as follows: C- calcite; PVC+CD-1.5- mixture containing calcite modified with 1.5% stearic acid by “dry” process; PVC+CW-1.5- mixture containing calcite modified with 1.5% stearic acid by “wet” process; PVC+CD-3- mixture containing calcite modified with 3% stearic acid by “dry” process; PVC+CW-3- mixture containing calcite modified with 3% stearic acid by “wet” process.

3.1 Tensile strength and elongation

Limestone modification with stearic acid makes its surface hydrophobic and thus compatible with PVC basis, which decreases the degree of friction between polymer and filler, (Kovačević et.al., 1994; Mihajlović et. al., 2009a; Mihajlović, Sekulić and Petrov, 2005). This conclusion was confirmed by the results obtained from testing of PVC mixture mechanical properties in this paper. The "wet" process of calcite modification with stearic acid (Table 1) increases tensile strength from 52.70 MPa (sample of uncoated limestone) to 54.20 MPa (sample with 1.5% stearic acid). Further increase in the amount of stearic acid to 3% decreases tensile strength to 53.60 MPa. This can be explained by the fact that excess stearic acid in modification process causes bonding between filler particles and agglomerate. Large particles in material structure weaken the material and decrease its fluidity, thus leading to the decrease in tensile strength, (Mihajlović, Sekulić and Petrov, 2005).

In the "dry" process of calcite modification with stearic acid (Table1) the highest value of tensile strength of 53.20 MPa is achieved in sample with 3% stearic acid. This confirms the fact that tensile strength increases as the filler's hydrophobicity rises, and it is highest when complete coating of calcite with stearic acid is achieved. This explains why higher amount of stearic acid (3%) is needed for coating of calcite of 99.90% in the "dry" modification process in comparison to the "wet" process (1.5%), as already shown, (Mihajlović et. al., 2009a; Mihajlović et. al., 2009b). Analysis of the obtained results leads to conclusion that for the same amount of stearic acid in calcite modification process tensile strength is higher in PVC mixture containing calcite modified by the "wet" process related to the PVC mixture containing calcite modified by the "dry" process. Namely, when the amount of stearic acid in modification process is 1.5%, tensile strength of PVC mixture containing calcite modified by the "wet" process is by 2.85% higher related to the tensile strength of PVC mixture containing calcite modified by the "dry" process. Also, when the amount of stearic acid in modification process is 3%, tensile strength of PVC mixture containing calcite modified by the "wet" process is by 0.75% higher related to the PVC mixture containing calcite modified by the "dry" process.

Since tensile strength and elongation are inversely proportional, samples of PVC mixture with higher tensile strength will demonstrate lower elongation. Namely, PVC mixture containing calcite modified with 1.5% stearic acid by the "wet" process shows by 0.5% less elongation related to the elongation of PVC mixture containing calcite modified by the "dry" process regardless of higher tensile strength. Also, PVC mixture containing calcite modified with 3% stearic acid by the "wet" process shows by 0.2% less elongation related to the elongation of PVC mixture containing calcite modified by the "dry" process, although its tensile strength is higher.

3.2 Tensile yield strength and elongation at break

Tensile yield strength of PVC mixture when calcite is modified by the “wet” process increases with the increase of the degree of sample coating (Table 1). Namely, tensile yield strength increases from 36.50 MPa in starting sample to 38.20 MPa in the sample with 1.5%. Further tensile yield strength increase to 39.10 MPa in sample with 3% stearic acid is observed, but that change is minor related to the sample with 1.5% stearic acid with tensile yield strength of 38.20 MPa, so it can be neglected. In the “dry” process of calcite modification with stearic acid, tensile yield strength of PVC mixture also rises with the degree of coating of the sample just like in the “wet” process (Table 1). Considering the fact that in the “dry” process calcite is modified with stearic acid in inhomogenous conditions, it is necessary to add more stearic acid than in the “wet” process (i.e. 3%) in order to achieve the degree of coating of 99.90%. In accordance with that, the highest value of tensile yield strength of 37.90 MPa is achieved in sample with 3%, while in the sample modified with 1.5% stearic acid tensile yield strength is lower-36.20 MPa. Comparing tensile yield strength of PVC mixtures containing calcite modified by the “wet” process and tensile yield strength of PVC mixtures containing calcite modified by the “dry” process shows that higher tensile yield strength occurs in mixture containing calcite modified by the “wet” process (Table 1). Namely, when the amount of stearic acid in modification process is 1.5%, tensile yield strength of PVC mixture containing calcite modified by the “wet” process is by 5.5% higher related to tensile yield strength of PVC mixture containing calcite modified by the “dry” process. Also, when the amount of stearic acid in modification process is 3%, tensile yield strength of PVC mixture containing calcite modified by the “wet” process is by 3.2% higher related to PVC mixture containing calcite modified by the “dry” process. Tendency of a material to bend without breaking is decreased when calcite is modified both by the “wet” and the “dry” processes with increased degree of sample coating. This is expected considering the fact that strength and elongation of a material are inversely proportional. Namely, PVC mixtures with higher tensile yield strength will demonstrate lower tensile elongation. This is explained by the fact that modification of calcite mineral surface produces filler which increases system fluidity since its surface is hydrophobic, and as such compatible to PVC basis, (Mihajlović et. al., 2012). Elongation at break of PVC mixture decreases from 35.90% to 29.30% in sample with 1.5% stearic acid in the “wet” process of calcite modification. Increase in the amount of stearic acid to 3% in calcite modification process further decreases elongation at break to 24.60%. In the “dry” process of calcite modification elongation at break of PVC mixture also decreases with the increase of the degree of coating just like in the “wet” process. However, sudden drop in value of elongation at break with achieving maximum degree of coating is evident. Elongation at break decreases from 35.90% to 22.50% in sample with 3% stearic acid. Sudden drop in value of elongation at break from 35.40% to 22.50% is explained by the inhomogenous character of the “dry” process of calcite modification. The obtained results indicate that the investigated mechanical properties of PVC mixture depend on the strength of bond between calcite and the adsorbed organic component.

The sample modified with 1.5% stearic acid concentration by wet process, as well as the sample modified with 3% stearic acid concentration by dry process, are totally hydrophobic, but PVC mixture with filler obtained by wet process shows better mechanical properties than mixture with filler obtained by dry process. This leads to conclusion that using filler in which surface active material is chemically adsorbed on calcite provides better mechanical properties of PVC mixture, as is the case with the sample modified by wet process with 1.5% concentration of stearic acid. In other words, the chemisorbed stearate on calcite allows stronger interaction in calcite-stearic acid-PVC system, i.e. better mechanical properties. Poorer mechanical properties were attained with filler obtained by dry process with 3% stearic acid concentration related to the PVC mixture with the sample obtained by wet process with 1.5% stearic acid concentration. In the sample modified by wet process with 3% stearic acid concentration, larger amount of the physically adsorbed surface active material was detected apart from the chemisorbed stearate, which evidently influences the intensity of interactions in PVC mixture. Poorer mechanical properties of PVC mixture with calcite modified by dry process with 3% stearic acid concentration are the result of physical adsorption of surface active material on the mineral.

4 CONCLUSION

Complete surface modification of limestone by stearic acid is achieved with 1.5% of stearic acid by the “wet” process and with 3% stearic acid by the “dry” process. Previous research has shown that with the “wet” modification process, a more uniform (homogeneous) distribution of stearic acid is achieved on the calcite surface compared to the “dry” process. This is why mechanical properties of a PVC product are better when limestone modified by the “wet” process is used for mixture. It was also observed even in case of limestone complete coating by the “dry” process with 3% stearic acid, limestone coated by the “wet” process grants better mechanical properties of PVC final product. Namely, tensile strength of PVC mixture containing limestone modified with 1.5% stearic acid by the “wet” process is by 2.84% higher related to tensile strength of PVC mixture containing limestone modified with the same amount of stearic acid but by the “dry” process. When limestone is modified with 3% stearic acid, PVC mixture containing limestone modified by the “wet” process shows by 0.74% higher tensile strength than PVC mixture containing limestone modified by the “dry” process. Breaking strength of PVC mixture containing limestone modified with 1.5% stearic acid by the “wet” process is by 5.23% higher related to the breaking strength of PVC mixture containing limestone modified with the same amount of stearic acid by the “dry” process. When the amount of stearic acid in the modification process is increased to 3%, PVC mixture containing limestone modified by the ‘’wet’’ process shows by 3.07% higer tensile strength than PVC mixture containing limestone modified by the “dry” process.

The results obtained imply that tensile strength, tensile elongation and elongation at break of PVC mixture depend on the strength of connection between calcite and the adsorbed organic component. The filler obtained by wet modification method and with concentrations of stearic acid which enable forming of chemisorbed stearate monolayer on calcite was found out to be the best. These mechanical properties are the best in PVC mixture with calcite obtained by wet process with 1.5% stearic acid concentration, which was totally hydrophobic, and in which chemically adsorbed surface material dominates, which means that the chemisorbed stearate on calcite enables stronger interaction in calcite-stearic acid-PVC system.

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Original scientific paper

DETERMINATION OF PARAMETERS FOR WET GRINDING OF PHOSPHATES IN A LABORATORY BALL MILL AND CLASSIFICATION IN A HYDROCYCLONE

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Abstract: The company Elixir Prahovo uses dry grinding of raw phosphate in its production process. However, due to increased moisture in the raw phosphate causing difficulties in the dry grinding phase, the possibility of introducing wet grinding is being considered at the industrial plant in Prahovo. The moisture content of the incoming raw phosphate varies from barge to barge, ranging between 5-10% H_2O, which is very high for the dry grinding process and create certain problems in the transportation system (the system of apron conveyors).

To assess the possibility of wet grinding of phosphates, predefined laboratory tests were conducted at the Department of Mineral Processing of the Faculty of Mining and Geology in Belgrade. This paper presents a portion of the results obtained during the aforementioned tests, which relate to the tested sample and its processing, as well as the determination of the particle size distribution of the initial sample, density, and bulk mass of phosphate ore. T In further investigations, grindability tests were conducted in a laboratory ball mill, and hydrocyclone tests were conducted on the ground phosphate in a laboratory hydrocyclone. Based on the results of these investigations, the necessary technological parameters have been determined to propose a scheme for the technological process of wet grinding and classification of phosphate. These parameters primarily relate to the material distribution in the hydrocyclone, pulp density in classification, and the particle size distribution of the ground phosphate.

Keywords: wet grinding; phosphate; classification; hydrocyclone

1 INTRODUCTION

The Chemical Products Industry (IHP Prahovo) was established in 1960, initially as a superphosphate factory and later as a producer of various granulated mineral fertilizers (Pavlica & Draškić, 1997). The founder was the Mining and Smelting Basin Bor (RTB Bor), which, aiming to address the issue of sulfuric acid neutralization, built a factory to

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convert that environmental and financial problem into a commercial product – phosphoric acid, fertilizers, and other phosphorus-based products.

In August 2012, the Elixir Group privatized IHP Prahovo, with necessary investments in the reconstruction of production and storage capacities to ensure the start of phosphoric acid production and phosphorus-based product manufacturing.

Elixir Prahovo once again takes the lead in the production of phosphoric acid and mineral fertilizers, with ambitious investment plans for the upcoming period.

In this regard, this paper also presents a portion of the research results conducted at the Department of Mineral Processing of the Faculty of Mining and Geology in Belgrade, aiming to define the scheme of wet grinding and classification of phosphates. These research efforts stemmed from the necessity to address the issue of increased moisture in the imported raw phosphate (Lazić & Nikšić, 2023).

To explore the potential of wet grinding of phosphates, a series of tests were conducted, and this paper presents a portion of those results that we consider most relevant for defining the grinding scheme and equipment selection.

Certainly, one of the crucial parameters is the particle size distribution of the feed material (Kolonja & Knežević, 2000). It is notable that the feed phosphate's coarseness is below 5 mm, and in some phosphates, up to 25% of the feed material consists of ground product. Grindability tests in the laboratory ball mill have shown that a fineness of approximately 40% passing -0.074 mm is achieved at the mill outlet in a very short time (3 minutes).

Through hydrocyclone and classification tests in the mechanical classifier, it was determined that the pulp density at the inlet to the classification process should be around 30% solids by weight, while the overflow density should be around 23-25% solids by weight (Lazić & Nikšić, 2023).

2 RESEARCH SAMPLE

A sample of Syrian phosphate weighing approximately 100 kg was provided by the expert services of the Elixir company and delivered to the Department of Mineral Processing at the Faculty of Mining and Geology in Belgrade.

In Tables 1 and 2, the particle size distribution and chemical composition of the phosphate sample used for testing are presented, along with the bulk weight with and without shaking. We can observe a moisture content of approximately 5.96%, which is very high for dry grinding (Magdalinović ,1999).

Table 1 Particle size analysis of the phosphate sample used for testing

Sample	%, H_2O	%, P_2O_5 on dry mass	Bulk weight, g/dm^3		Sieve analysis (%) mm							
			Without shaking	With shaking	1.00	0.63	0.50	0.40	0.25	0.125	0.063	<0.063
Syrian phosphate	5.96	29.43	1413.00	1692.00	16.50	6.40	4.90	5.70	21.10	27.20	9.20	9.00

Table 2 Chemical composition of the phosphate sample used for testing

% Cl	% Al_2O_3	% Fe_2O_3	% MgO	% Na_2O	% F	Cd, ppm	As, ppm	% C org
0.077	0.28	0.21	0.65	0.73	3.47	4.98	3.13	0.20

3 SAMPLE PREPARATION FOR TESTING

The processing of the initial sample of Syrian phosphate weighing approximately 100 kg was carried out in accordance with the testing program, which included homogenization, quartering, and sampling for further testing using the "checkerboard" method. One half of the sample, approximately 50 kg, was set aside as a reserve, while the other half was used for testing purposes.

On samples weighing 1 kg, the particle size distribution of the initial sample was determined through wet sieving, along with density and bulk mass determination.

In Figures 1 and 2, the "cake" before quartering and the "cake" after sampling using the "checkerboard" method are depicted.

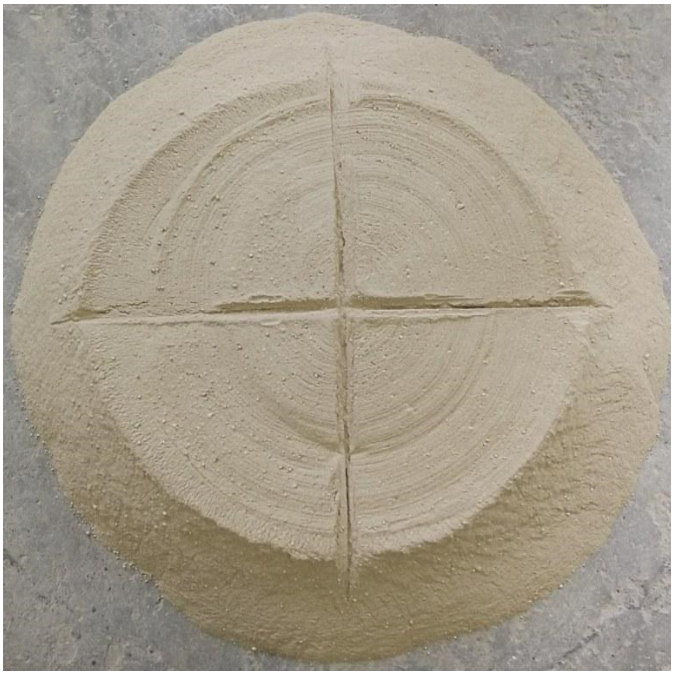

Figure 1 Cake before quartering

Figure 2 Cake after sampling using the checkerboard method

By sieving the initial sample through a series of standard sieves, the following particle size distribution was obtained:

Table 3 Particle size distribution of the initial sample before grinding

Size fraction [mm]	Mass [%]	Cumulative retention [%]	Cumulative passing [%]
+3.360	3.10	3.10	100.00
-3.360 +1.651	4.40	7.50	96.90
-1.651 +0.833	5.70	13.20	92.50
- 0.833 + 0.417	10.10	23.30	86.80
-0.417 +0.297	9.70	33.00	76.70
-0.297+0.208	13.4	46.40	67.00
-0.208 +0.104	23.00	69.40	53.60
-0.104 +0.074	5.50	74.90	30.60
-0.074 +0.043	0.8	75.70	25.10
-0.043 +0.00	24.30	100.00	24.30

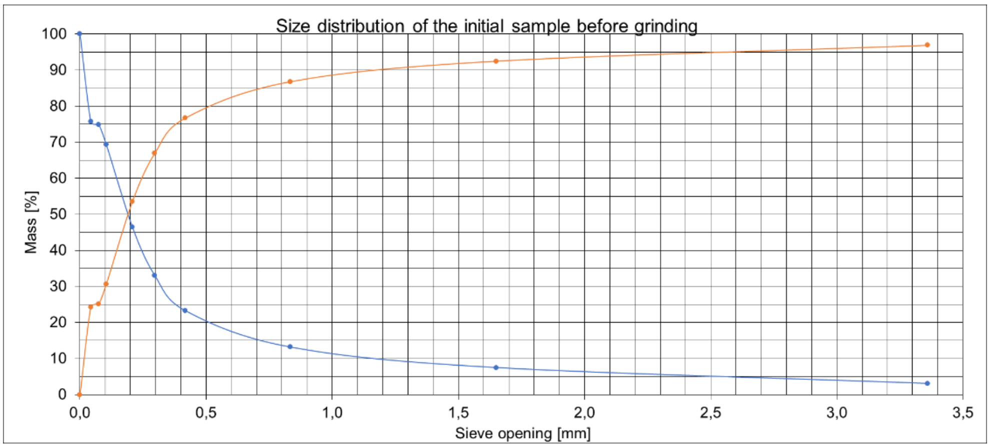

Figure 3 Particle size distribution of the Syrian phosphate sample examined in the study

Based on the particle size distribution curves, the d95 (particle size at 95% cumulative passing) of the tested phosphate is approximately 2.5 mm, with a median particle size of around 0.2 mm. The P80 (particle size at 80% cumulative passing) is 500 µm. Additionally, the percentage of the -0.074 mm class (finished product) is approximately 25%.

The density of the feed sample was determined using the pycnometer method and found to be 3.05 t/m³. The bulk mass was determined "without shaking" and amounts to 1.63 t/m³.

4 GRINDABILITY TESTS

Grindability tests were conducted using wet grinding in a Denver laboratory mill (Figure 4).

Figure 4 Denver laboratory mill (Deušić & Lazić, 2013)

The tests were conducted on samples weighing 1 kg with a ratio of solid to liquid (S:L) of 1:0.42, for durations of 5, 10, and 15 minutes. After grinding, wet sieving was performed using a 75 μm sieve, and a table and grindability curve were created, which are presented in the following text.

Table 4 Grindability of Syrian phosphate in a laboratory ball mill

Grinding time [min]	Oversieve mass [g]	Undersieve mass [g]	Size fraction [% - 74 μm]
0	749	251	25.10
5	540	460	46.00
10	421	579	57.90
15	343	657	65.70

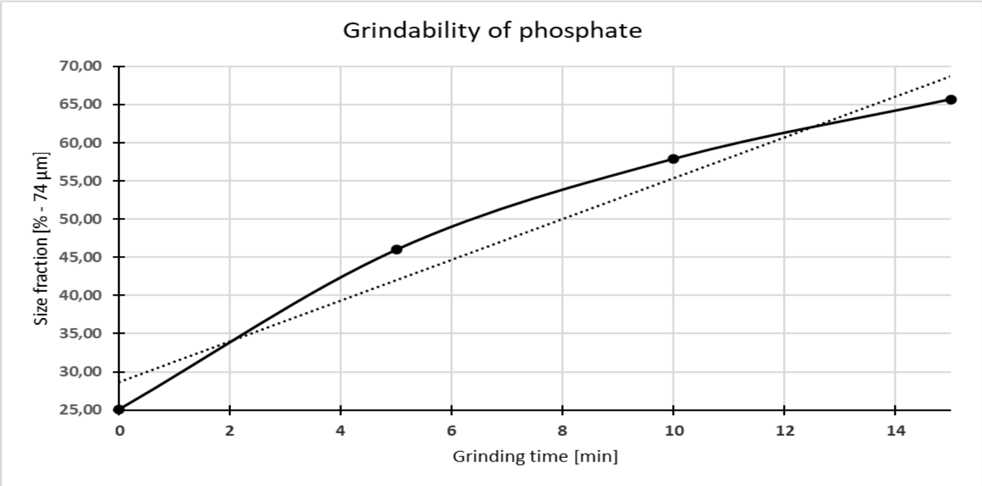

Figure 5 Grindability curve of Syrian phosphate

Based on the grindability curve, it can be observed that for a grinding fineness of approximately 40% passing -0.074 mm, the ore needs to be ground for only 3 minutes. This is understandable because the initial ore sample already contains about 25% of the ground product. The fineness of 40% passing -0.074 mm is determined based on literature data and experience with wet grinding of lead and zinc ores.

5 HYDROCYCLONE TESTS

Hydrocyclone tests on wet-ground phosphate were conducted on a laboratory setup depicted in Figure 6.

Figure 6 Laboratory setup for hydrocyclone testing

The tests were conducted by wet grinding 10 samples of 1 kg each in the laboratory ball mill to obtain the required ore mass for hydrocyclone testing. The test began by forming an inlet pulp with a density of 35% solid phase and taking a sample of the inlet pulp. Subsequently, classification was performed on a 100 mm diameter hydrocyclone with a sand nozzle size of 12 mm. During this process, samples of the hydrocyclone underflow (sand) and overflow were taken, their densities were measured, and their particle size distributions were determined.

Based on the particle size distributions of the feed, sand, and overflow, a mass balance was performed to separate the sand and overflow using the Grumbrecht method.

Table 5 Particle size distribution of the inlet pulp (35% solids), nozzle diameter d=12 mm

Size fraction [mm]	Mass [g]	Mass [%]	Cumulative retention [%]	Cumulative passing [%]
+0.833	4.00	1.82	1.82	100.00
- 0.833 + 0.417	5.00	2.27	4.09	98.18
-0.417 +0.297	10.00	4.55	8.64	95.91
-0.297+0.208	22.00	10.00	18.64	91.36
-0.208 +0.104	56.00	25.45	44.09	81.36
-0.104 +0.074	25.00	11.36	55.45	55.91
-0.074 +0.00	98.00	44.55	100.00	44.55

Table 6 Particle size distribution of the overflow (33% solids, inlet 35% solids), nozzle diameter d=12 mm

Size fraction [mm]	Mass [g]	Mass [%]	Cumulative retention [%]	Cumulative passing [%]
+0.833	0.00	0.00	0.00	100.00
- 0.833 + 0.417	2.00	0.61	0.61	100.00
-0.417 +0.297	9.00	2.75	3.36	99.39
-0.297+0.208	25.00	7.65	11.01	96.64
-0.208 +0.104	87.00	26.61	37.61	88.99
-0.104 +0.074	39.00	11.93	49.54	62.39
-0.074 +0.00	165.00	50.46	100.00	50.46

Table 7 Particle size distribution of the underflow (sand) 38% solids, (inlet 35% solids), nozzle diameter d=12 mm

Size fraction [mm]	Mass [g]	Mass [%]	Cumulative retention [%]	Cumulative passing [%]
+0.833	4.00	1.63	1.63	100.00
- 0.833 + 0.417	6.00	2.45	4.08	98.37
-0.417 +0.297	17.00	6.94	11.02	95.92
-0.297+0.208	34.00	13,88	24.90	88.98
-0.208 +0.104	65.00	26.53	51.43	75.10
-0.104 +0.074	24.00	9.80	61.22	48.57
-0.074 +0.00	95.00	38.78	100.00	38.78

Table 8 Distribution in the hydrocyclone determined by the Grumbrecht method

Size fraction [mm]	Inlet ↑	Overflow ↑	Underflow ↑				
+0.833	2	3	4	2-4	3-4	2-4x3-4	(3-4)^2
- 0.833 + 0.417	100.00	100.00	100.00	0.00	0.00	0.00	0.00
-0.417 +0.297	98.18	100.00	98.37	-0.19	1.63	0.00	3.00
-0.297+0.208	95.91	99.39	95.92	-0.01	3.47	0.00	12.00
-0.208 +0.104	91.36	96.64	88.98	2.38	7.66	18.00	59.00
-0.104 +0.074	81.36	88.99	75.10	6.26	13.89	87.00	193.00
-0.074 +0.00	55.91	62.39	48.57	7.34	13.81	101.00	191.00
+0.833	44.55	50.46	38.78	5.77	11.68	67.00	136.00
Sum						274.00	593.55

Based on the data presented in Table 8, the mass fraction of the overflow is calculated as Mpr overflow = (274 / 593.55) * 100 = 46.11%. The mass fraction of the sand is 53.89%.

After the first classification test, water was added to the cyclone pump's basket, thereby diluting the pulp to 32% solids. This was done because based on the density of the sand and overflow, it was determined that the previous classification test achieved a low classification efficiency (around 12%). At the same time, the sand discharge opening was increased from 12 to 18 mm.

Table 9 Particle size distribution of the inlet pulp (32% solids), nozzle diameter d=18 mm

Size fraction [mm]	Mass [g]	Mass [%]	Cumulative retention [%]	Cumulative passing [%]
+0.833	4.00	2.11	2.11	100.00
- 0.833 + 0.417	5.00	2.63	4.74	97.89
-0.417 +0.297	14.00	7.37	12.11	95.26
-0.297+0.208	24.00	12.63	24.74	87.89
-0.208 +0.104	52.00	27.37	52.11	75.26
-0.104 +0.074	21.00	11.05	63.16	47.89
-0.074 +0.00	70.00	36.84	100.00	36.84

Table 10 Particle size distribution of the overflow (21% solids, inlet 32% solids), nozzle diameter d=18 mm

Size fraction [mm]	Mass [g]	Mass [%]	Cumulative retention [%]	Cumulative passing [%]
+0.833	0.00	0.00	0.00	100.00
- 0.833 + 0.417	0.00	0.00	0.00	100.00
-0.417 +0.297	2.00	0.69	0.69	100.00
-0.297+0.208	3.00	1.04	1.74	99.31
-0.208 +0.104	30.00	10.42	12.15	98.26
-0.104 +0.074	29.00	10.07	22.22	87.85
-0.074 +0.00	224.00	77.78	100.00	77.78

Table 11 Particle size distribution of the underflow (sand) 48% solids, (inlet 32% solids), nozzle diameter d=18 mm

Size fraction [mm]	Mass [g]	Mass [%]	Cumulative retention [%]	Cumulative passing [%]
+0.833	4.00	1.27	1.27	100.00
- 0.833 + 0.417	8.00	2.55	3.82	98.73
-0.417 +0.297	25.00	7.96	11.78	96.18
-0.297+0.208	51.00	16.24	28.03	88.22
-0.208 +0.104	127.00	40.45	68.47	71.97
-0.104 +0.074	34.00	10.83	79.30	31.53
-0.074 +0.00	65.00	20.70	100.00	20.70

Table 12 Distribution in the hydrocyclone determined by the Grumbrecht method

Size fraction [mm]	Inlet ↑	Overflow ↑	Underflow ↑				
1	2	3	4	2-4	3-4	2-4x3-4	(3-4)^2
+0.833	100.00	100.00	100.00	0.00	0.00	0.00	0.00
- 0.833 + 0.417	97.89	100.00	98.73	-0.83	1.27	-1.00	2.00
-0.417 +0.297	95.26	100.00	96.18	-0.92	3.82	-3.00	15.00
-0.297+0.208	87.89	99.31	88.22	-0.32	11.09	-4.00	123.00
-0.208 +0.104	75.26	98.26	71.97	3.29	26.29	86.00	691.00
-0.104 +0.074	47.89	87.85	31.53	16.37	56.32	922.00	3172.00
-0.074 +0.00	36.84	77.78	20.70	16.14	57.08	921.00	3258.00
Sum						1921.00	7259.90

Based on the data presented in Table 12, the mass fraction of the overflow is calculated as Mpr = (1921 / 7259.9) * 100 = 26.47%. The mass fraction of the sand is complemented to 100%, which is 73.53%. Better classification efficiency (around 50%) was recorded, but a high overflow fineness of 77.78% passing -0.074 mm was obtained, which could pose a problem in further processing of the phosphate. In this regard, a third classification test was conducted with a pulp density at the hydrocyclone inlet of 30% solids, while simultaneously reducing the sand discharge opening to 10 mm.

Table 13 Particle size distribution of the inlet pulp (30% solids), nozzle diameter d=10 mm

Size fraction [mm]	Mass [g]	Mass [%]	Cumulative retention [%]	Cumulative passing [%]
+0.833	2.00	1.06	1.06	100.00
- 0.833 + 0.417	14.00	7.41	8.47	98.94
-0.417 +0.297	9.00	4.76	13.23	91.53
-0.297+0.208	18.00	9.52	22.75	86.77
-0.208 +0.104	49.00	25.93	48.68	77.25
-0.104 +0.074	18.00	9.52	58.20	51.32
-0.074 +0.00	79.00	41.80	100.00	41.80

Table 14 Particle size distribution of the overflow (23% solids, inlet 30% solids), nozzle diameter d=10 mm

Size fraction [mm]	Mass [g]	Mass [%]	Cumulative retention [%]	Cumulative passing [%]
+0.833	0.00	0.00	0.00	100.00
- 0.833 + 0.417	2.00	0.52	0.52	100.00
-0.417 +0.297	5.00	1.31	1.83	99.48
-0.297+0.208	14.00	3.66	5.48	98.17
-0.208 +0.104	92.00	24.02	29.50	94.52
-0.104 +0.074	44.00	11.49	40.99	70.50
-0.074 +0.00	226.00	59.01	100.00	59.01

Table 15 Particle size distribution of the underflow (sand) 46% solids, (inlet 30% solids), nozzle diameter d=10 mm

Size fraction [mm]	Mass [g]	Mass [%]	Cumulative retention [%]	Cumulative passing [%]
+0.833	4.00	1.19	1.19	100.00
- 0.833 + 0.417	9.00	2.67	3.86	98.81
-0.417 +0.297	51.00	15.13	18.99	96.14
-0.297+0.208	62.00	18.40	37.39	81.01
-0.208 +0.104	114.00	33.83	71.22	62.61
-0.104 +0.074	26.00	7.72	78.93	28.78
-0.074 +0.00	71.00	21.07	100.00	21.07

Table 16 Distribution in the hydrocyclone determined by the Grumbrecht method

Size fraction [mm]	Inlet ↑	Overflow ↑	Underflow ↑				
1	2	3	4	2-4	3-4	2-4x3-4	(3-4)^2
+0.833	100.0	100.00	100.00	0.00	0.00	0.0	0.00
- 0.833 + 0.417	98.94	100.00	98.81	0.13	1.19	0.0	1.00
-0.417 +0.297	91.53	99.48	96.14	-4.61	3.34	-15.0	11.00
-0.297+0.208	86.77	98.17	81.01	5.76	17.16	99.0	295.00
-0.208 +0.104	77.25	94.52	62.61	14.64	31.91	467.0	1018.0
-0.104 +0.074	51.32	70.50	28.78	22.54	41.71	940.0	1740.0
-0.074 +0.00	41.80	59.01	21.07	20.73	37.94	787.0	1439.0
Sum						2277.0	4504.5

Based on the data from Table 16, the mass fraction of the overflow is calculated as Mpr = (2277 / 4504.5) * 100 = 50.56%. The sand fraction is 49.44%, with a classification efficiency of 40%. In the following Table 17 and graphically in Figure 7, the cumulative overflow screening from the classification test and the expected screening (BM product) are shown.

Table 17 Particle size distribution of the hydrocyclone overflow obtained in the laboratory of FMG at different pulp densities at the hydrocyclone inlet (30% solids, 32% solids, and 35% solids) and different sand discharge openings (10 mm, 12 mm, and 18 mm)

Sieve opening, mm	Overflow 30%Č (10 mm)	Overflow 32%Č (18 mm)	Overflow 35%Č (12mm)	Sieve opening, mm	BM product
				1.00	100.00
0.833	100.00	100.00	100.00	0.63	99.80
0.417	99.48	100.00	99.39	0.50	99.30
0.297	98.17	99.31	96.64	0.40	98.70
0.208	94.52	98.26	88.99	0.25	90.00
0.104	70.50	87.85	62.39	0.13	57.50
0.074	59.01	77.78	50.46	0.063	2.44
0.000	0.00	0.00	0.00	0.00	0.00

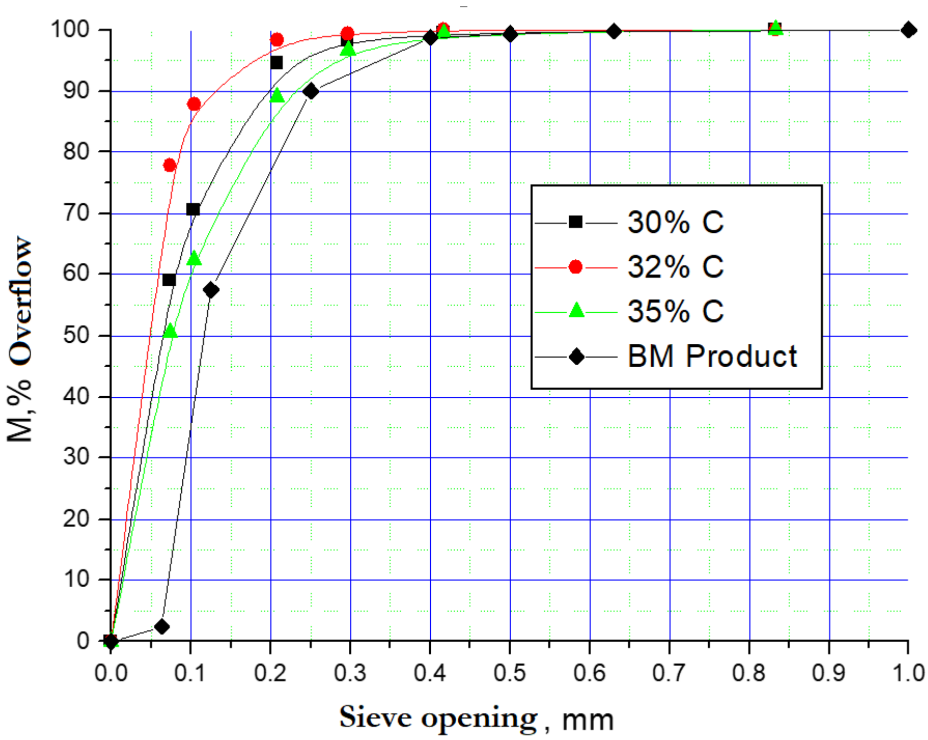

Figure 7 Cumulative overflow screening curves from the hydrocyclone and the expected curve (BM product)

Based on the overflows curves, it can be observed that all hydrocyclone overflows obtained in the FMG laboratory are finer than the expected curve (BM product). Additionally, it is evident that the hydrocyclone overflow obtained at a pulp density of 35% solids at the inlet and with a sand nozzle of 12 mm is the closest to the expected curve. For this case, the d90 of the overflow is approximately 208 µm, and the median diameter (d50) is around 74 µm. However, for this case, the classification efficiency is the lowest (12%).

For a pulp density at the hydrocyclone inlet of 32% solids and with a sand nozzle of 18 mm, the finest overflow was obtained, with a d90 of approximately 130 µm and a median diameter (d50) of around 50 µm. For a pulp density at the inlet of 30% solids and with a sand nozzle of 10 mm, the overflow with a coarseness of d90 around 200 µm and a median diameter of approximately 60 µm was obtained. In the case of the BM product curve, the d90 is around 250 µm, with a median diameter of approximately 120 µm.

6 CONCLUSION

Based on the results presented in this study, the following conclusions can be drawn:

The moisture content in the initial sample of around 6% is high for dry grinding, so considering wet grinding of phosphate is entirely justified.

The initial phosphate sample is very fine (d95= 2.5 mm) and contains approximately 25% of the -74 micrometer class (ground material), which will have a positive impact on the wet grinding process.

By wet grinding the tested phosphate sample in a laboratory ball mill for 3 minutes, the required fineness of grinding of approximately 40% passing -0.074mm is achieved, which is sufficient for the classification process in the hydrocyclone.

By classifying the ground phosphate in the hydrocyclone at different pulp densities, various finenesses of the ground product are obtained. The optimal grinding fineness of approximately 60% passing -0.074mm has been selected for further production process, which is achieved at a pulp density of 30% solids at the hydrocyclone inlet. The mass distribution of sand and overflow at this pulp density at the inlet is approximately half-half (50:50%), with a classification efficiency of around 40%.

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Original scientific paper

UNDERGROUND LOADING-HAULAGE EQUIPMENT SELECTION WITH APPLICATION OF TOPSIS METHOD WITH DIFFERENT WEIGHTING METHODS OF CRITERIA

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Abstract: The choice of adequate mining machines is one of the most complex problems in the underground exploitation of mineral raw materials. The weight of such a decision is reflected in the high prices of these machines and the occurrence of even greater costs during their use if their choice was wrongly conceived. There are numerous examples from practice in which such decisions were made solely on the basis of comparing the prices of machines, without considering any of the remaining important parameters that over time can bring significant benefits or harm to the investor. In this paper, by applying the TOPSIS method with different weighting methods of criteria the most suitable loader for work in the underground mine in question was selected. Based on 7 different criteria, a selection was made between 6 different loaders, which were produced by 3 different renowned manufacturers of mining equipment. The final benefit of this work, in addition to the selection of the optimal loader, is that with some corrections, it can also be applied to the selection of other mining equipment in underground mines such as: trucks, boomers or bolters.

Keywords: loading-haulage equipment; selection; TOPSIS; entropy; standard deviation; criterion weight

1 INTRODUCTION

The selection of underground loading-haulage equipment in the mining industry represents a vital task for mining engineers, especially in greenfield projects. Selection of adequate loading-haulage equipment is important because it directly affects the productivity of the underground mine. In greenfield projects, beside common parameters considered in alternative ranking such as capacity of the loading-haulage equipment, price and engine emission class other criteria such as delivery time of equipment, terms of payment (percentage of advance payment), possibility of remote control and the state of maintenance network should be evaluated. As with any decision-making process,

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these criteria have different weights. Often it is a common practice that the decision maker (expert), or a decision-making team defines these weights based on their expert opinion, making the alternatives ranking prone to subjectivism. For ranking to be more objective weights of criteria should be determined by some objective method.

In the mining industry many different methods are used for multi-criteria decision making. For example, when selecting the underground mining method methods such as VIKOR, TOPSIS, PROMETHEE, ELECTRA, AHP and others are often used (Mijalkovski et al., 2022; Saki et al., 2020; Ali et al., 2021; Mijalkovski et al., 2023). For selection maintenance strategy in mining industry Pourjavad et al. (2013) used ANP and TOPSIS method. Koçali (2023) used TOPSIS method for personal protective equipment selection. Fuzzy TOPSIS method was also used for human health and safety risk management in underground coal mines by Mahdevari et al. (2014). The same method was used for equipment selection by Yavuz (2016) and for selection of loading-haulage equipment in open pit mines by Bazzazi et al. (2008). TOPSIS was used for selection of a production drill rig by Chanda (2019) and for haulage system selection among TOPSIS, VIKOR and AHP were used by Ghasvereh et al. (2019).

Based on literature review it can be concluded that the TOPSIS method is scientifically verified for problem solving in the mining industry. In this paper a ranking of the best loading-haulage equipment will be conducted using TOPSIS method with expert given criteria weight (assigned weights) as well as objective criteria weight calculated by entropy and standard deviation method.

2 TOPSIS METHOD APPLICATION

TOPSIS method or The Technique for Order of Preference by Similarity to Ideal Solution is one of multi-criteria decision analysis method developed by Hwang and Yoon (1981). This technique focuses on determining the most desirable alternative by comparing its proximity to the positive ideal solution and distance from the negative ideal (anti-ideal) solution. The ideal solution is derived by amalgamating the best attributes from each criterion, while the anti-ideal solution comprises the worst attributes. It is important to note that this technique is applicable only to numerical datasets, where the criterion weights are known or defined based on expert opinions. By considering the assigned weights, the ranking results can be obtained (Tzeng & Huang, 2011; Uzun et al., 2021.; Amudha et al., 2021). As previously mentioned, weights defined by experts can be prone to subjectivism. To make ranking objective, the weights in this paper will be determined using entropy weight method and standard deviation method (Li et al., 2011; Wang & Luo, 2010) to rank the alternatives with TOPSIS method.

Figure 1. shows the steps in TOPSIS MCDM process.

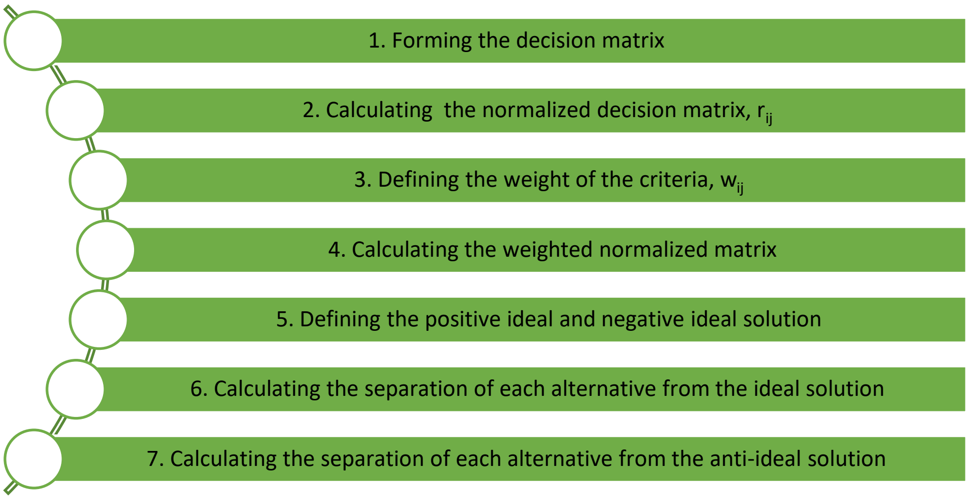

Figure 1 Steps in TOPSIS MCDM process

2.1 Forming the decision matrix

Decision matrix in its basic form is presented in following equation:

$$X = \begin{array}{c} \\ A_1 \\ \vdots \\ A_m \end{array} \begin{array}{c} \begin{array}{ccc} C_1 & \cdots & C_n \end{array} \\ \begin{bmatrix} x_{11} & \cdots & x_{1n} \\ \vdots & \ddots & \vdots \\ x_{m1} & \cdots & x_{mn} \end{bmatrix} \end{array} \tag{1}$$

Decision matrix for underground loading-haulage equipment selection will have 6 alternatives and 7 criteria as shown in figure 2. In these six alternatives, there are two loaders from each of the renowned manufacturers of mining equipment Sandvik, Epiroc and GHH, but due to the confidentiality of business data, it was not possible to show them with the names of manufacturers and models in this paper. Given that this choice of equipment was conceived for the needs of participating in the tender announced by the mine owner, it was necessary to take several criteria for choosing the optimal solution. For the investor, the most important criteria were the delivery date of the machine and the engine class, while for the contractor as a buyer and user of this machine, the other criteria were also very important. In addition to the price and method of payment for the machine, it is very important for the buyer how he will be able to maintain the machine, and the possibility of remote control will allow him to use the machine in unsupported underground facilities.

	C1: Price (€)	C2: Bucket capacitiy (t)	C3:Delivery period (weeks)	C4: Engine class (TIER)	C5: Terms of payment (advance payment %)	C6: State of maintenance network	C7: Possibility of remote control
A1:OPTION 1	733.000	10	52	III	35	Does not exist	Yes
A2:OPTION 2	865.000	14	48	III	40	Does not exist	No
A3:OPTION 3	965.000	14	24	V	20	Good	Yes
A4:OPTION 4	876.000	14	36	IV	30	Good	No
A5:OPTION 5	1.330.000	17	32	IV	20	Very good	No
A6:OPTION 6	1.120.000	21	32	IV	25	Very good	No

Figure 2 Decision matrix

As seen from Figure 2, some criteria are quantitative (C1, C2, C3, C4, C5) and some are qualitative (C6 and C7). To be able to proceed to step 2 of the TOPSIS algorithm, qualitative scale needs to be defined for C6 and C7. For criteria C6: State of maintenance network qualitative scale is transformed to quantitative values according to table 1.

Table 1 Qualitative and quantitative values of criteria C6

Qualitative values	Quantitative values
Does not exist	1
Poor	2
Average	3
Good	4
Very good	5

For criteria C7: Possibility of remote control is transformed from qualitative to quantitative values according to table 2.

Table 2 Qualitative and quantitative values of criteria C7

Qualitative values	Quantitative values
No	0.0001*
Yes	1

*Value 0.0001 instead of 0 is used so entropy method for criteria weight can be calculated

Now our decision matrix has quantitative values for all criteria, as shown in Figure 3.

	C1: Price (€)	C2: Bucket capacity (t)	C3: Delivery period (weeks)	C4: Engine class (TIER)	C5: Terms of paymet (advance payment %)	C6: State of maintenance network	C7: Possibility of remote control
A1:OPTION 1	733.000,00	10	52	3	35	1	1
A2:OPTION 2	865.000,00	14	48	3	40	1	0,0001
A3:OPTION 3	965.000,00	14	24	5	20	3	1
A4:OPTION 4	876.000,00	14	36	4	30	3	0,0001
A5:OPTION 5	1.330.000,00	17	32	4	20	5	0,0001
A6:OPTION 6	1.120.000,00	21	32	4	25	5	0,0001

Figure 3 Decision matrix with all quantitative values

2.2 Calculation of normalized decision matrix

Normalization is technique that is used to standardize the values, making them to a common scale with values ranging from 0 to 1. There are several normalization techniques that are used for multi-criteria decision making (Çelen, 2014). In this paper vector normalization is performed. Transformation from decision matrix to normalized decision matrix is done by following equation:

$$\|X\| \rightarrow \|R\| \tag{2}$$

Where:

$$\|R\| = \left\|r_{ij}\right\|_{m\times n} \tag{3}$$

Where:

$$r_{ij} = \frac{x_{ij}}{\sqrt{\sum_{i=1}^{m} x_{ij}^2}} \tag{4}$$

Following previous equations, we get the normalized decision matrix as shown on figure 4.

	C1: Price (€)	C2: Bucket capacity (t)	C3: Delivery period (weeks)	C4: Engine class (TIER)	C5: Terms of paymet (advance payment %)	C6: State of maintenance network	C7: Possibility of remote control
OPTION 1	0,30	0,27	0,55	0,31	0,49	0,12	0,71
OPTION 2	0,35	0,37	0,51	0,31	0,56	0,12	0,00
OPTION 3	0,39	0,37	0,25	0,52	0,28	0,36	0,71
OPTION 4	0,36	0,37	0,38	0,42	0,42	0,36	0,00
OPTION 5	0,54	0,45	0,34	0,42	0,28	0,60	0,00
OPTION 6	0,46	0,56	0,34	0,42	0,35	0,60	0,00

Figure 4 Normalized decision matrix (vector normalization)

2.3 Defining weight of criteria

Weight of criteria has very high influence on the final ranking of the alternatives. In this paper weight will be assigned by a team of decision makers, relying on their expertise. Weights of criteria assigned like this can be subjective and sometimes may lead us to wrong conclusion. For this reason, two commonly used methods for objective criteria weight calculation (entropy and standard deviation) will be used to compare the rankings of the alternatives. In all cases the sum of all criteria weights must be 1.

2.3.1 Subjective criteria weight

The criteria weight assigned by the expert team for decision making is presented in figure 5.

	C1: Price (€)	C2: Bucket capacity (t)	C3: Delivery period (weeks)	C4: Engine class (TIER)	C5: Terms of paymet (advance payment %)	C6: State of maintenance network	C7: Possibility of remote control
Weight	0,2	0,25	0,13	0,09	0,12	0,11	0,1

Figure 5 Subjective criteria weight

2.3.2 Entropy method

The entropy weight function is based on the discrete probability distribution:

$$e_j = \frac{-1}{\ln(m)} \sum_{i=1}^{m} r_{ij} \ln(r_{ij}) \tag{5}$$

The degree of diversity (d) possessed by each criteria is evaluated as:

$$d_j = 1 - e_j, j = 1, 2, 3 \tag{6}$$

The weight objective for each criteria is given by:

$$W_j = \frac{d_i}{\sum_{i=1}^{m} d_i} \tag{7}$$

For entropy method calculation a vector normalized matrix cannot be used, so another normalized matrix, a linear one was calculated using the equations 2, 3 and the following equation:

$$r_{ij} = \frac{x_{ij}}{\sum_{i=1}^{m} x_{ij}} \tag{8}$$

Linear normalized matrix is presented in figure 6.

	C1: Price (€)	C2: Bucket capacity (t)	C3: Delivery period (weeks)	C4: Engine class (TIER)	C5: Terms of paymet (advance payment %)	C6: State of maintenanc e network	C7: Possibility of remote control
OPTION 1	0,12	0,11	0,23	0,13	0,21	0,06	0,50
OPTION 2	0,15	0,16	0,21	0,13	0,24	0,06	0,00
OPTION 3	0,16	0,16	0,11	0,22	0,12	0,17	0,50
OPTION 4	0,15	0,16	0,16	0,17	0,18	0,17	0,00
OPTION 5	0,23	0,19	0,14	0,17	0,12	0,28	0,00
OPTION 6	0,19	0,23	0,14	0,17	0,15	0,28	0,00

Figure 6 Normalized decision matrix (linear normalization)

The calculated values for weight criteria by entropy method is presented in figure 7.

	C1: Price (€)	C2: Bucket capacity (t)	C3: Delivery period (weeks)	C4: Engine class (TIER)	C5: Terms of paymet (advance payment %)	C6: State of maintenance network	C7: Possibility of remote control
Weight	0,01	0,02	0,02	0,01	0,02	0,12	0,79

Figure 7 Criteria weight by Entropy method

2.3.3 Standard deviation method

The standard deviation method determines the weights of the criteria in two steps, by the following equations (Odu, 2019):

$$\sigma_j = \sqrt{\frac{\sum_{i=1}^{m}\left[r_{ij} - \overline{r_j}\right]^2}{m}}, i = 1, \ldots, m; j = 1, \ldots, n \tag{9}$$

Therefore

$$W_j = \frac{\sigma_j}{\sum_{j=1}^{n}\sigma_j} \tag{10}$$

Where σ_j is the standard deviation for criteria j.

The calculated values for weight criteria by standard deviation is presented in figure 8.

	C1: Price (€)	C2: Bucket capacity (t)	C3: Delivery period (weeks)	C4: Engine class (TIER)	C5: Terms of paymet (advance payment %)	C6: State of maintenance network	C7: Possibility of remote control
Weight	0,06	0,07	0,08	0,06	0,09	0,18	0,46

Figure 8 Criteria weight by standard deviation

Figure 9. presents the comparative preview of weight of criteria by different approaches (subjective, entropy and standard deviation).

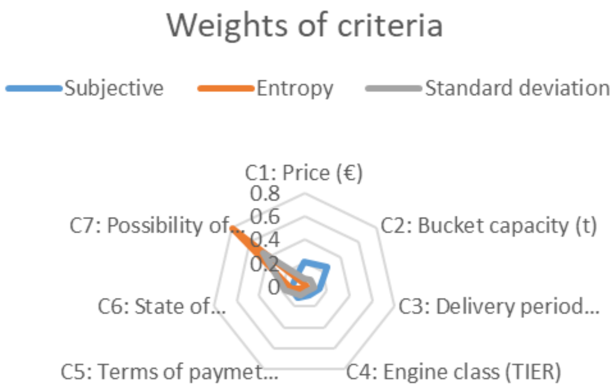

Figure 9 Comparative preview of weight of criteria by different approaches

2.4 Calculation of the weighted normalized matrix

Now we can form weighted decision matrix following next equation:

$$V_{ij} = W_j \cdot r_{ij} \tag{11}$$

Where W_j is the weight of the j-th criteria, $\sum_{j=1}^{n} w_j = 1$.

Now we get weighted decision matrixed for all our three cases (with subjective weights, with entropy calculated weights and with standard deviation calculated weights). On figures 10-12. weighted matrixes are presented.

	C1: Price (€)	C2: Bucket capacity (t)	C3: Delivery period (weeks)	C4: Engine class (TIER)	C5: Terms of paymet (advance payment %)	C6: State of maintenance network	C7: Possibility of remote control
OPTION 1	0,06	0,07	0,07	0,03	0,06	0,01	0,07
OPTION 2	0,07	0,09	0,07	0,03	0,07	0,01	0,00
OPTION 3	0,08	0,09	0,03	0,05	0,03	0,04	0,07
OPTION 4	0,07	0,09	0,05	0,04	0,05	0,04	0,00
OPTION 5	0,11	0,11	0,04	0,04	0,03	0,07	0,00
OPTION 6	0,09	0,14	0,04	0,04	0,04	0,07	0,00

Figure 10 Subjective weighted matrix

	C1: Price (€)	C2: Bucket capacity (t)	C3: Delivery period (weeks)	C4: Engine class (TIER)	C5: Terms of paymet (advance payment %)	C6: State of maintenance network	C7: Possibility of remote control
OPTION 1	0,00	0,00	0,01	0,00	0,01	0,01	0,56
OPTION 2	0,00	0,01	0,01	0,00	0,01	0,01	0,00
OPTION 3	0,01	0,01	0,01	0,01	0,01	0,04	0,56
OPTION 4	0,00	0,01	0,01	0,00	0,01	0,04	0,00
OPTION 5	0,01	0,01	0,01	0,00	0,01	0,07	0,00
OPTION 6	0,01	0,01	0,01	0,00	0,01	0,07	0,00

Figure 11 Entropy weighted matrix

	C1: Price (€)	C2: Bucket capacity (t)	C3: Delivery period (weeks)	C4: Engine class (TIER)	C5: Terms of paymet (advance payment %)	C6: State of maintenance network	C7: Possibility of remote control
OPTION 1	0,02	0,02	0,05	0,02	0,04	0,02	0,32
OPTION 2	0,02	0,03	0,04	0,02	0,05	0,02	0,00
OPTION 3	0,03	0,03	0,02	0,03	0,02	0,06	0,32
OPTION 4	0,02	0,03	0,03	0,02	0,04	0,06	0,00
OPTION 5	0,03	0,03	0,03	0,02	0,02	0,11	0,00
OPTION 6	0,03	0,04	0,03	0,02	0,03	0,11	0,00

Figure 12 Standard deviation weighted matrix

2.5 Defining the ideal and anti-ideal solution

For each criteria the ideal and anti-ideal solution are calculated according to the type of the criteria (MIN or MAX). In the table 3. are listed types of the criteria.

Table 3 Types of criteria

Criteria	Criteria type
C1: Price (€)	MIN
C2: Bucket capacity (t)	MAX
C3: Delivery period (weeks)	MIN
C4: Engine class (TIER)	MAX
C5: Terms of payment (advance payment %)	MIN
C6: State of maintenance network	MAX
C7: Possibility of remote control	MAX

The ideal solution is the solution that maximizes the benefit criteria (MAX type) and minimizes the cost criteria (MIN type) whereas the anti-ideal solution maximizes the cost criteria (MIN type) and minimizes the benefit criteria (MAX type) (Roszkowska, 2011).

Ideal solution A^+ has the form:

$$A^+ = \left\{\left(\left(MAX_i v_{ij} \middle| j \in K'\right)\right), \left(MIN_i v_{ij} \middle| j \in K''\right)\right\} = \{v_1^+, v_2^+, \dots, v_n^+\}, (i = 1,2, \dots m) \quad (12)$$

Anti-ideal solution A^- has the form:

$$A^- = \left\{\left(\left(MIN_i v_{ij} \middle| j \in K'\right)\right), \left(MAX_i v_{ij} \middle| j \in K''\right)\right\} = \{v_1^-, v_2^-, \dots, v_n^-\}, (i = 1,2, \dots m) \quad (13)$$

Where:

$K' \subseteq K \to K'$ is a subset of set K who makes the MAX type criteria,

$K'' \subseteq K \to K''$ is a subset of set K who makes the MIN type criteria.

According to equations 12 and 13 we can now calculate the ideal and anti-ideal solutions for subjective assigned weights of criteria, entropy calculated weights of criteria and standard deviation calculated weights of criteria (Figure 13).

	C1: Price (€)	C2: Bucket capacity (t)	C3: Delivery period (weeks)	C4: Engine class (TIER)	C5: Terms of paymet (advance payment %)	C6: State of maintenance network	C7: Possibility of remote control
Subjective							
Ideal	0,060	0,139	0,033	0,047	0,033	0,066	0,071
Anti-ideal	0,109	0,066	0,072	0,028	0,067	0,013	0,000
Entropy							
Ideal	0,004	0,010	0,006	0,006	0,007	0,070	0,559
Anti-ideal	0,007	0,005	0,013	0,004	0,014	0,014	0,000
Standard deviation							
Ideal	0,019	0,041	0,021	0,030	0,024	0,106	0,324
Anti-ideal	0,035	0,019	0,046	0,018	0,048	0,021	0,000

Figure 13 Ideal and anti-ideal solutions for each criteria regarding the different weight approach

2.6 Calculation of each alternative distance from ideal solution

The separation distance of each alternative from the ideal solution is calculated according to equation:

$$S_i^+ = \sqrt{\sum_{j=1}^{k}(v_{ij} - v_j^+)^2}, (i = 1,2, \dots m) \tag{14}$$

Now we can calculate the separation distance for every alternative for our case. On Figure 13. Alternative distance from ideal solution is presented for all three cases.

	C1: Price (€)	C2: Bucket capacity (t)	C3: Delivery period (weeks)	C4: Engine class (TIER)	C5: Terms of paymet (advance payment %)	C6: State of maintenance network	C7: Possibility of remote control	S+
Subjective								
A1: OPTION 1	0,00000	0,00533	0,00148	0,00036	0,00063	0,00277	0,00000	**0,10280**
A2: OPTION 2	0,00012	0,00216	0,00109	0,00036	0,00112	0,00277	0,00500	**0,11227**
A3: OPTION 3	0,00036	0,00216	0,00000	0,00000	0,00000	0,00069	0,00000	**0,05665**
A4: OPTION 4	0,00014	0,00216	0,00027	0,00009	0,00028	0,00069	0,00500	**0,09288**
A5: OPTION 5	0,00237	0,00071	0,00012	0,00009	0,00000	0,00000	0,00500	**0,09104**
A6: OPTION 6	0,00100	0,00000	0,00012	0,00009	0,00007	0,00000	0,00500	**0,07922**
Entropy								
A1: OPTION 1	0,00000	0,00003	0,00005	0,00001	0,00003	0,00311	0,00000	**0,05677**
A2: OPTION 2	0,00000	0,00001	0,00004	0,00001	0,00005	0,00311	0,31279	**0,56214**
A3: OPTION 3	0,00000	0,00001	0,00000	0,00000	0,00000	0,00078	0,00000	**0,02812**
A4: OPTION 4	0,00000	0,00001	0,00001	0,00000	0,00001	0,00078	0,31279	**0,56000**
A5: OPTION 5	0,00001	0,00000	0,00000	0,00000	0,00000	0,00000	0,31279	**0,55930**
A6: OPTION 6	0,00000	0,00000	0,00000	0,00000	0,00000	0,00000	0,31279	**0,55929**
Standard deviation								
A1: OPTION 1	0,00000	0,00045	0,00062	0,00015	0,00032	0,00712	0,00000	**0,09310**
A2: OPTION 2	0,00001	0,00018	0,00046	0,00015	0,00057	0,00712	0,10511	**0,33705**
A3: OPTION 3	0,00004	0,00018	0,00000	0,00000	0,00000	0,00178	0,00000	**0,04474**
A4: OPTION 4	0,00001	0,00018	0,00011	0,00004	0,00014	0,00178	0,10511	**0,32769**
A5: OPTION 5	0,00025	0,00006	0,00005	0,00004	0,00000	0,00000	0,10511	**0,32481**
A6: OPTION 6	0,00010	0,00000	0,00005	0,00004	0,00004	0,00000	0,10511	**0,32456**

Figure 14 Separation distance from ideal solution for every alternative regarding the different weight approach

2.7 Calculation of each alternative distance from anti-ideal solution

The separation distance of each alternative from the anti-ideal solution is calculated according to equation:

$$S_i^- = \sqrt{\sum_{j=1}^{k}(v_{ij} - v_j^-)^2}, (i = 1,2, \dots m) \tag{15}$$

Now we can calculate the separation distance for every alternative for our case. On Figure 15. Alternative distance from anti-ideal solution is presented for all three cases.

	C1: Price (€)	C2: Bucket capacity (t)	C3: Delivery period (weeks)	C4: Engine class (TIER)	C5: Terms of paymet (advance payment %)	C6: State of maintenance network	C7: Possibility of remote control	S-
Subjective								
A1: OPTION 1	0,00237	0,00000	0,00000	0,00000	0,00007	0,00000	0,00500	**0,08627**
A2: OPTION 2	0,00144	0,00071	0,00003	0,00000	0,00000	0,00000	0,00000	**0,04664**
A3: OPTION 3	0,00089	0,00071	0,00148	0,00036	0,00112	0,00069	0,00500	**0,10120**
A4: OPTION 4	0,00137	0,00071	0,00048	0,00009	0,00028	0,00069	0,00000	**0,06019**
A5: OPTION 5	0,00000	0,00216	0,00076	0,00009	0,00112	0,00277	0,00000	**0,08301**
A6: OPTION 6	0,00029	0,00533	0,00076	0,00009	0,00063	0,00277	0,00000	**0,09934**
Entropy								
A1: OPTION 1	0,00001	0,00000	0,00000	0,00000	0,00000	0,00000	0,31279	**0,55929**
A2: OPTION 2	0,00001	0,00000	0,00000	0,00000	0,00000	0,00000	0,00000	**0,00340**
A3: OPTION 3	0,00000	0,00000	0,00005	0,00001	0,00005	0,00078	0,31279	**0,56007**
A4: OPTION 4	0,00001	0,00000	0,00002	0,00000	0,00001	0,00078	0,00000	**0,02860**
A5: OPTION 5	0,00000	0,00001	0,00003	0,00000	0,00005	0,00311	0,00000	**0,05655**
A6: OPTION 6	0,00000	0,00003	0,00003	0,00000	0,00003	0,00311	0,00000	**0,05652**
Standard deviation								
A1: OPTION 1	0,00025	0,00000	0,00000	0,00000	0,00004	0,00000	0,10511	**0,32464**
A2: OPTION 2	0,00015	0,00006	0,00001	0,00000	0,00000	0,00000	0,00000	**0,01488**
A3: OPTION 3	0,00009	0,00006	0,00062	0,00015	0,00057	0,00178	0,10511	**0,32921**
A4: OPTION 4	0,00014	0,00006	0,00020	0,00004	0,00014	0,00178	0,00000	**0,04863**
A5: OPTION 5	0,00000	0,00018	0,00032	0,00004	0,00057	0,00712	0,00000	**0,09071**
A6: OPTION 6	0,00003	0,00045	0,00032	0,00004	0,00032	0,00712	0,00000	**0,09100**

Figure 15 Separation distance from anti-ideal solution for every alternative regarding the different weight approach

2.8 Calculation of the relative closeness of alternative to the ideal solution

Calculation of the relative closeness of alternative to the ideal solution can be done by following equation:

$$C_i = \frac{S_i^-}{S_i^- + S_i^+}, 0 \leq C_i \leq 1 \tag{16}$$

Regarding that $C_i = 0$ represents anti-ideal solution and $C_i = 1$ represents the ideal solution. Figure 16. represents the relative closeness of alternatives to the ideal solution according to the previous equation.

	Subjective	Entropy	Standard deviation
OPTION 1	0,456	0,908	0,777
OPTION 2	0,293	0,006	0,042
OPTION 3	0,641	0,952	0,880
OPTION 4	0,393	0,049	0,129
OPTION 5	0,477	0,092	0,218
OPTION 6	0,556	0,092	0,219

Figure 16 Relative closeness of the alternatives to the ideal solution with respect to the weighting method

Figures 17-19. Represent relative closeness of every alternative to ideal solution for three different weighting methods.

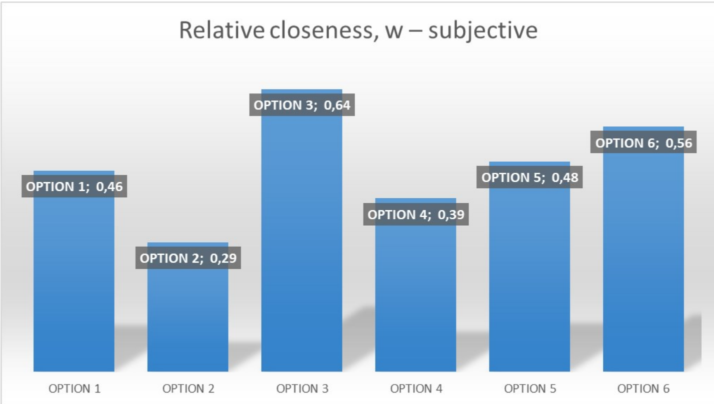

Figure 17 Relative closeness of the alternatives to the ideal solution, subjective weighting

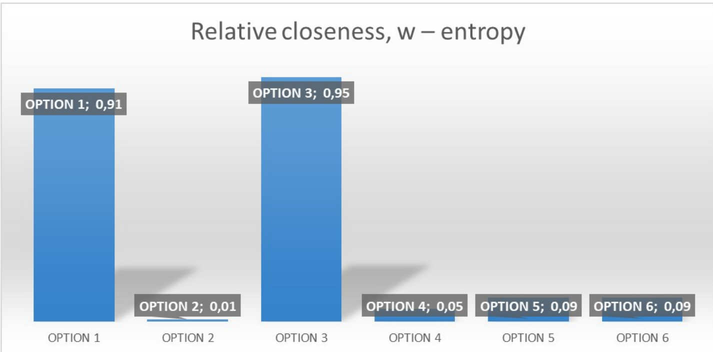

Figure 18 Relative closeness of the alternatives to the ideal solution, entropy weighting

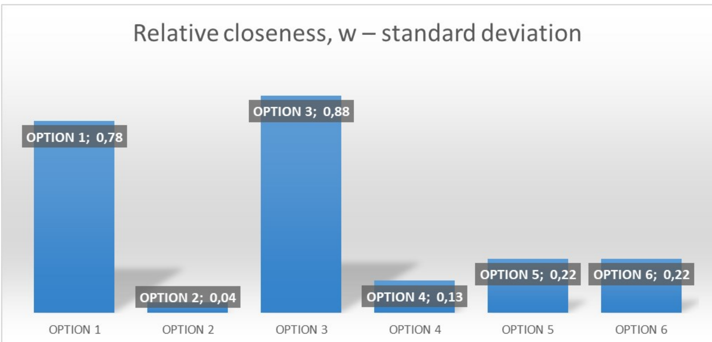

Figure 19 Relative closeness of the alternatives to the ideal solution, standard deviation weighting

2.9 Ranking the preference order

Ranking of the alternatives is done according to the descending order of the C_i. Now with known values of relative closeness of each alternative to the ideal solution ranking of the alternatives can be done. Figure 20. shows the ranking of the alternatives regarding the weighting method.

Alternative	Ci	Rank
A3: OPTION 3	0,64	1
A6: OPTION 6	0,56	2
A5: OPTION 5	0,48	3
A1: OPTION 1	0,46	4
A4: OPTION 4	0,39	5
A2: OPTION 2	0,29	6

Subjective weighted

Alternative	Ci	Rank
A3: OPTION 3	0,95	1
A1: OPTION 1	0,91	2
A5: OPTION 5	0,09	3
A6: OPTION 6	0,09	4
A4: OPTION 4	0,05	5
A2: OPTION 2	0,01	6

Entropy weighted

Alternative	Ci	Rank
A3: OPTION 3	0,88	1
A1: OPTION 1	0,78	2
A6: OPTION 6	0,22	3
A5: OPTION 5	0,22	4
A4: OPTION 4	0,13	5
A2: OPTION 2	0,04	6

Standard deviation weighted

Figure 20 Ranking the preference order

3 DISCUSION

The initial assumption of this paper was that all six alternatives of analyzed loaders do not exceed the limit values given in the mining design with their external dimensions. The choice is further narrowed by the condition that the maximum deadline for the delivery of a new machine must not exceed one year. Three renowned manufacturers of mining equipment submitted their offers, which are shown in Figure 2. Seven different criteria were selected for the evaluation of these offers. The application of the TOPSIS method of multi-criteria decision-making resulted in the recommendation that the optimal solution should be the loader, which is designated as Option 3 in this paper.

It is interesting to point out that in all three cases of ranking the alternatives using the weight method, alternative 3 was chosen as the optimal, while alternative 2 was chosen as the least desirable. The loader marked as alternative 3 had an 11.6% higher price than the loader marked as alternative 2 and it was fourth in terms of price. If the choice of the machine were to be carried out, as is the practice in a large number of cases, based on the price alone, an alternative 3 would certainly not be considered as an acceptable option.

It is clear that some other criteria were decisive for alternative 3 to be chosen as the optimal. The facts that this alternative had the shortest delivery time, was the only one with a TIER V engine class and shared the best values with one of other alternatives in three other criteria were crucial for it to be chosen as the optimal.

By choosing alternative 3, the investor will receive a new machine in the shortest possible time and will be able to activate it immediately and realize benefits in the form of increased production in his mine. The bidding company has good after-sales support in the mine region, which will greatly reduce machine downtime in the future. However, as the most significant value of the selected alternative, the possibility of remote control of this loader should be singled out, which gives more opportunities for its use in various operations in the mine and drastically affects the operator's safety when performing risky loading and transport operations in underground mines.

4 CONCLUSION

The choice of mining machinery should be “protected” from the subjective decisions of investors or managers and be based on the application of mathematical methods in order to find the optimal solution that will enable safer work and a healthier environment for operators and finally to the improvement of productivity in the operation of the mine in terms of the reliability of the selected equipment during its lifetime. The application of TOPSIS or some other method of multi-criteria decision-making should become a standard “tool” when selecting mining machinery. It is certain that if decision-makers were presented with the possibilities of this way of applying mathematical methods and were shown the real and long-term benefits for their business, the vast majority would correct their previous principles when choosing equipment for their mines.

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Original scientific paper

OPTIMIZATION AND ANALYSIS OF DRILLING AND BLASTING PARAMETERS USING O-PITBLAST SOFTWARE

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Abstract: The development of modern technologies has enabled the application of specialized softwares in the field of mining through various segments and processes, including in works related to drilling and blasting. The application of specialized software for drilling and blasting with the use of modern technologies such as photogrammetry enabled more accurate data from the field for later processing. By processing data and creating a 3D model of the terrain for drilling and blasting works, it can obtain precise data, which serve us for the optimization of drilling and blasting parameters, as well as their analysis. The application of this type of specialized software, in this case, the O-Pitblast software, provides an overview of all parameters, as well as possible corrections and different scenarios regarding the results of the blasting itself. Optimization and analysis through this software can improve the results of blasting in terms of fragmented material size, and reduce the unwanted effects of blasting, such as the flyrock or blast vibration. The software has different types of tools that allow a detailed review and insight into the entire process from the beginning of planning drilling and blasting works to the very end and also printing of reports after the work is done.

Keywords: optimization; 3D model; drilling; blasting; O-Pitblast; software

1 INTRODUCTION

Mining industries and others have developed computer softwares for solving various engineering tasks, such as in the design phase but also the exploitation process, to evaluate the performed works and optimize different operations (Crnogorac L. et al., 2019). During blasting at quarries, parameters related to drilling and blasting work need to be optimized, to see the real situation from the field now it is enabled with the revolution of specialized softwares development. The specialized softwares for drilling and blasting which are currently on the market were developed on the principle of recording aerial images with the help of unmanned aerial vehicles and processing images

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with the photogrammetry method. By processing photos, software obtains a cloud of points which, as a document, contains information about every point of the part of the terrain on which the blast is carried out or the entire open pit, according to X, Y, and Z coordinates. By importing data into the software that is specialized for drilling and blasting, it generates a 3D model and gets a detailed overview of the bench on which the blasting is being done and then can perform optimization and analysis of the drilling and blasting parameters. The combination of software with modern technologies such as UAV technology (unmanned aerial vehicle) or drones is also significant because the software can finally process the precise input data from the field (Milanović S. et al., 2019). Through this paper, an example of the use of O-Pitblast software at a quarry is given for optimization and analysis of parameters during drilling and later during blasting operations.

With the O-Pitblast software, users can monitor and analyze various parameters that affect blasting results. It is possible to analyze the fragmentation of rock material, which is affected by the parameters of drilling and blasting, such as the geometry of the blast field, the amount of explosives, delay time, etc. These parameters can be analyzed in the software, which also has tools for analyzing the blast vibration, predicting the strength of the blast vibration and its impact on the environment.

Through the analysis of the blast vibration, with the help of this software, can be imported vibration values in the database and analyzed, and the quantity of explosives per delay can be predicted and in that way avoid possible damage to buildings due to blasting. Detailed analysis is shown through diagrams according to the appropriate standards as well as by determining the attenuation law.

2 DATA IN O-PITBLAST SOFTWARE

Through the O-Pitblast software, the database allows us to enter various parameters such as the characteristics of the rock material, the technical characteristics of the explosives as well as the means for the initiation system of the explosives, and the prices of the costs for drilling and blasting works, which allows the software to choose the most suitable possible solution for the drilling and blasting parameters. For this example, the following characteristics of the rock material and explosives in the quarry were used, imported into the software database, and used through further analysis.

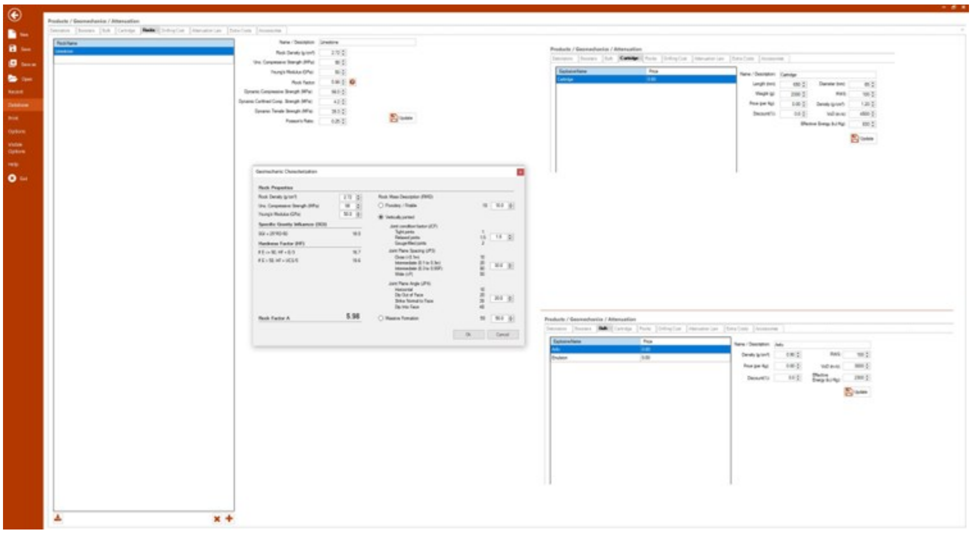

Figure 1 Creating rock material and explosive properties in the O-Pitblast software

The creation of a 3D terrain model is done based on the input of a large number of points, which are generated based on the processing of photos taken from an unmanned aerial vehicle. 3D models can be generated through the software based on a large number of points positioned in X, Y, and Z coordinates. In this case, a model of the bench was generated on which the blasting was carried out, in order to first establish the drilling parameters in the blast field, but also to optimize the blasting parameters.

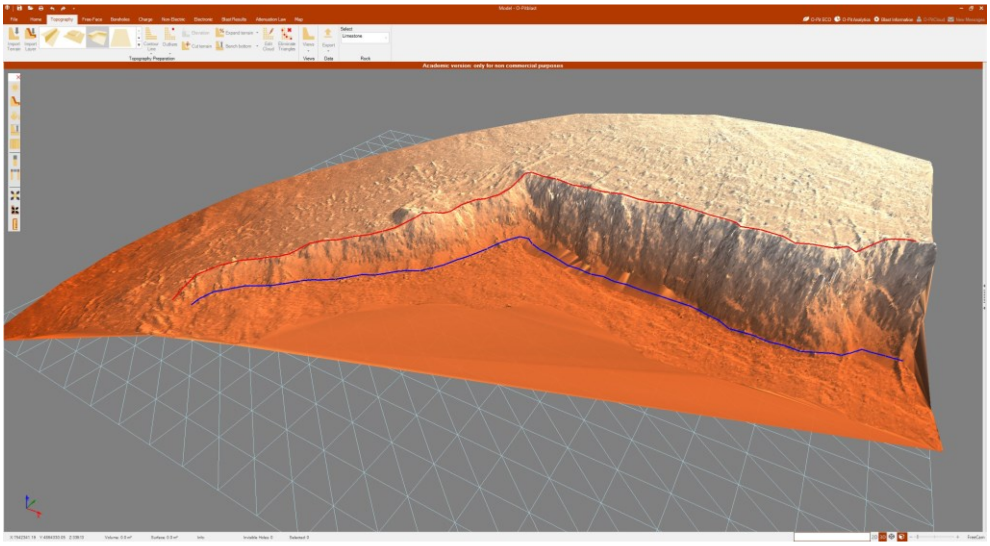

Figure 2 3D model of terrain with crest and toe of the bench in O-Pitblast software

After creating the terrain model, before designing the drilling and blasting works, it would be good to define the crest and toe of the bench, so that the drilling angle and the position of the blastholes can be checked later in the software. This tool is very important because by adjusting the angle, it can correct the burden, which is very important when blasting, especially in the first row of the blast field. Also, using tools for correcting and checking the drilling angle, can prevent the unwanted effects of blasting, such as the flyrock, but also contribute to a better fragmentation of the material and prevent the appearance of boulders.

3 DESIGNING DRILLING AND BLASTING PARAMETERS

3.1 Blastholes and designed parameters

During the blasting on the quarry, the designed parameters were used, which are shown in the following Table 1. These parameters were entered into the software so that further analysis and optimization could be carried out. Blastholes were created before entering the parameters. Blastholes are entered by using already existing templates to create positions or by entering a file with the exact positions of the blastholes in the appropriate coordinate system. The following picture shows the designed blastholes, as well as additional blastholes that would be drilled to reduce the burden size in places where it is necessary.

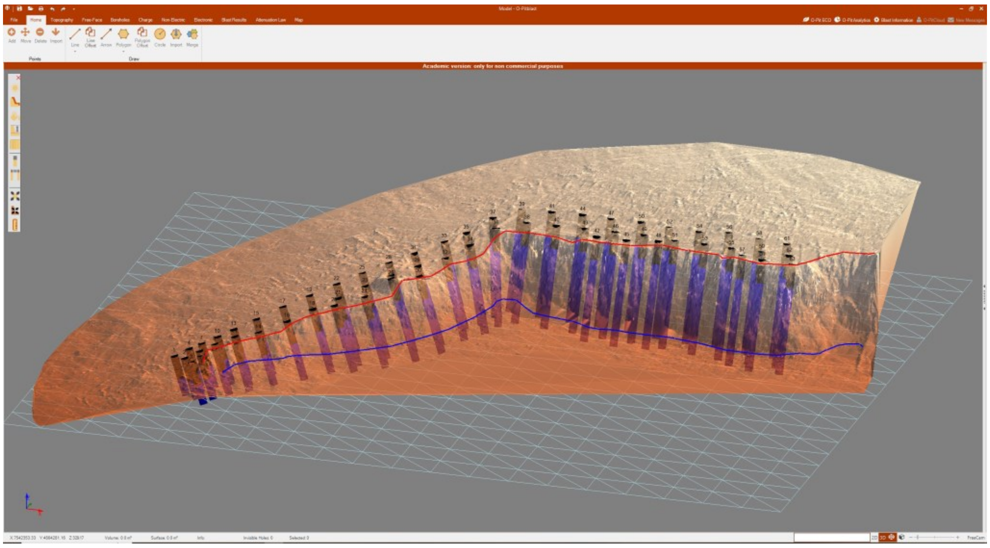

Figure 3 Blastholes for designed parameters of drilling in O-Pitblast software

The parameters of drilling and blasting on the quarry are given in such a way as to satisfy the appropriate requirements regarding fragmentation, loading, and transport mechanization, but in this example, their analysis and optimization will be performed through software. The following Table 1 shows the design parameters of drilling and blasting.

Table 1 Designed drilling and blasting parameters

Parameters	Value
Burden	3 m
Spacing	3.6 m
Inclination of blasthole	75^0
Stemming	3 m
Bench height	8.5 m
Drilled length	685.6 m
Design volume	4,935 m^3
Number of blastholes	63

The designed drilling and blasting parameters entered into the software must be analyzed in the software to determine whether they can be further optimized, to obtain the best possible fragmentation of the material, reduce the unwanted effects on the environment, as well as prevent the flyrock. First, the designed parameters are entered, where the software will give the final results in terms of the burden size, as in the fragmentation size, and it can compare the obtained results with the optimized parameters processed later to obtain the difference. We entered the designed parameters according to the existing data taken during blasting so far, then we analyzed them in the software.

3.2 Blastholes information and blast pattern

When entering the designed blastholes into the software, it allows further access to each blasthole on the blastfield, where it can fill those blasthole from the database that was previously created. In addition to entering data on explosives, we also have data on in-hole detonation delay time that we must also enter, then data on blastholes such as position, angle, depth, the quantity of explosives, etc. The biggest advantage of the software is precisely in the optimization of drilling parameters, where for this purpose the correction of the blasthole inclination can be performed separately or only the first row of the blastholes, etc. In addition to the optimization of drilling parameters, the software has tools for different types of connections in the blastfield and various scenarios, as well as analysis of the blasting results. Analysis of blast vibration due to demining, analysis of the fragmentation, the possibility of controlling the overlap of delay time, and other various options. The following Figure 4 shows the bench where the blast charge can be seen for a specific blasthole.

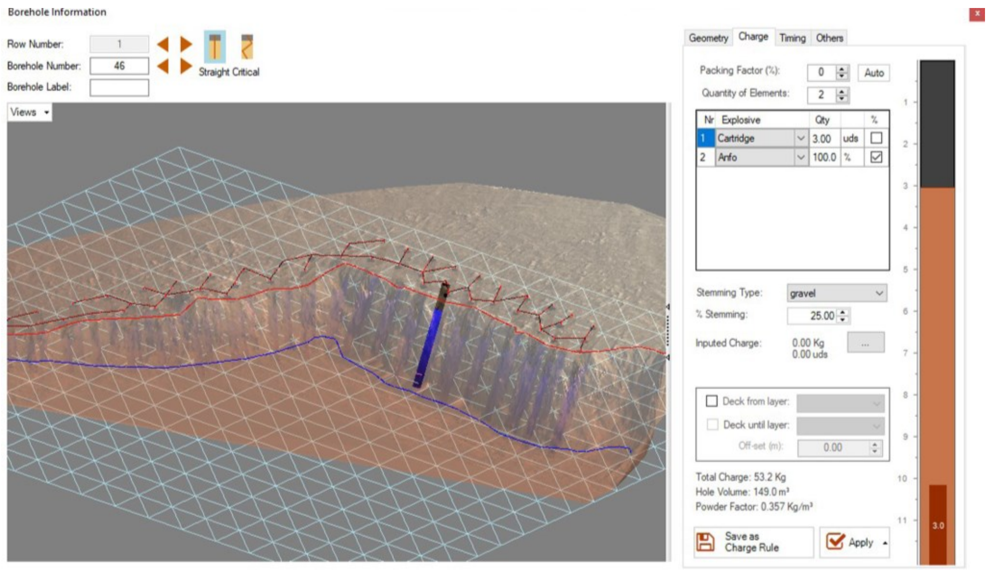

Figure 4 Blasthole charge according to designed parameters in O-Pitblast software

Blasthole is carried out as planned according to the designed parameters, and the data on the quantity and characteristics of the explosives used are shown in the following Table 2 and were also entered into the database during processing in the software. After entering and reviewing the blasthole charges, it is possible to proceed to the creation of the blast pattern, which can be used in the database for the non-electric initiation of blastholes. The Nonel system provides adequate safety during initiation, reduction of blast vibration, a combination of different delay times, use in blastholes filled with water, etc (Milanović S. et al., 2023).

Table 2 Characteristics of explosives (https://www.maxamcorp.com/en)

Parameters	ANFO	Emulsion
Density (kg/l)	0.8-0.95	1.1-1.3
Detonation speed (m/s)	2500-3500	2800-6000
Gas volume (lit/kg)	970	921
Specific energy (kJ/kg)	2260	2590

When entering data on the connection system of detonators, as well as on blasthole delay time, software can check the initiation system with the help of simulation, as well as whether any of the blastholes on the blastfield are initiated at the same time. Initiation of blasthole at the same time can lead to unwanted effects such as an increase of the blast vibration, so the delay time can be checked here, i.e. whether the appropriate amount that is designed for the delay time does not have the same initiation time as some of the other blasthole on the blastfield. The blast pattern, delay time, as well as delay time control system with the simulation of blasting are shown in the following Figure 5.

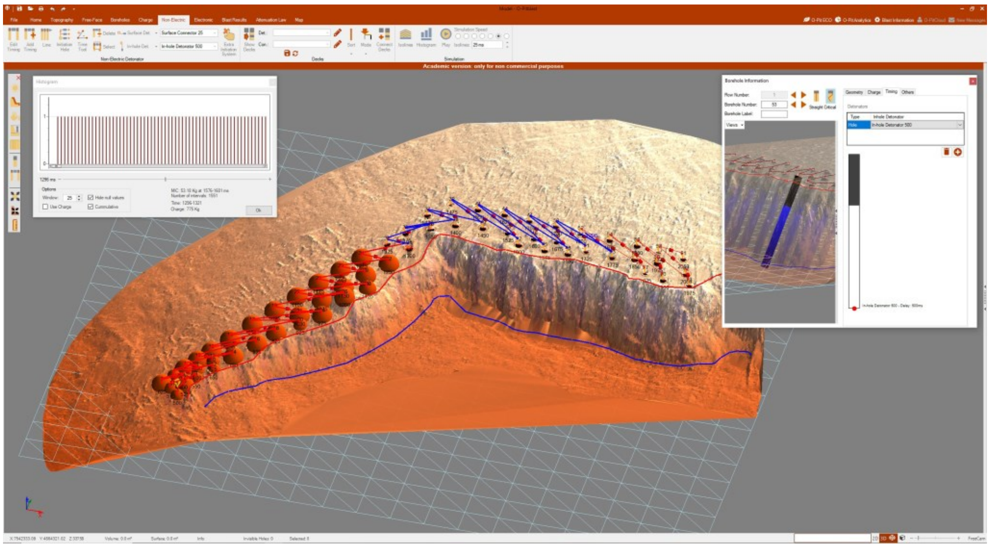

Figure 5 Simulation of blasting with the delay time in the O-Pitblast software

3.3 Burden distribution

As already mentioned in the previous part of this paper, the main advantage of the O-Pitblast software is the correction of the drilling inclination to optimize the burden size, i.e. first row in the blastfield. The burden is an important parameter in blasting, and when it is close to the designed one along the entire blasthole, the quality of the blast increases, in terms of the size of the fragmented material and also of the reduced possibility of flyrock. Correct designed burden guarantees high efficiency of rock mass blasting, and burden distribution in this software provides data of a proposed blasthole pattern by blasting documentation guideline (Pyra J. and Gądek K., 2020). During blasting in the quarry, the front of the bench is often irregularly shaped, so that in most cases the designed one burden deviates a lot, so this software enables precise data of the place of bench burden that is equal to or deviated one. The following Figure 6 shows a model with differently colored zones depending on the size of the line of smallest burden.

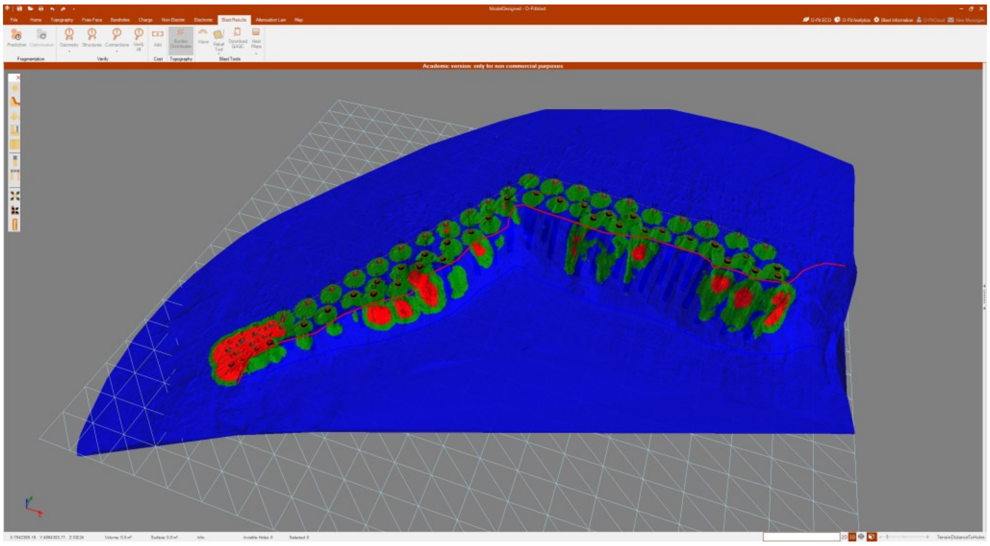

Figure 6 Burden distribution with designed parameters in the O-Pitblast software

The green color represents the zones where the burden value is close to the designed one, and the blue color represents the zones where there are larger burden deviations (the value of the burden is higher than the designed one), and the red zones represent the zones where there are smaller burden values than the designed one (Milanović S. et al., 2023). Based on this option, it is possible to predict even in which place in the blastfield there will be a problem from the eventual flyrock or even from the appearance of oversized rock material. There are three types of flyrock during blasting, which appear by lack of confinement of the energy in the explosive column, and also in the condition when there is a small burden size for the blasthole diameter (Negovanović, M. et al., 2022). Different types of flyrock are shown in Figure 7.

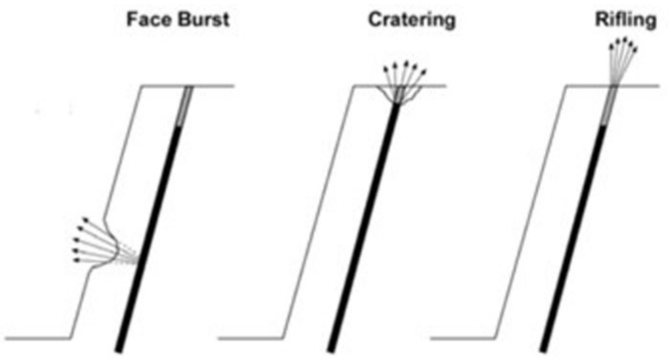

Figure 7 The three key mechanisms of flyrock (https://www.o-pitblast.com/)

By using the software, the inclination can be corrected before the drilling, and after the inspection with the use of burden distribution in the software, the correct values of the drilling inclination can be obtained according to which the drilling can be done, so that the results of the blasting later are as satisfactory as possible. When calculating the zone of flyrock, the software uses three possible types of flyrock, i.e. face burst, cratering and rifling, which can be calculated using the following equations (https://www.o-pitblast.com/).

- Face burst

$$L = \frac{k^2}{g} \cdot \left(\frac{\sqrt{m}}{B}\right)^{2.6} \tag{1}$$

- Cratering

$$L = \frac{k^2}{g} \cdot \left(\frac{\sqrt{m}}{SH}\right)^{2.6} \tag{2}$$

- Rifling

$$L = \frac{k^2}{g} \cdot \left(\frac{\sqrt{m}}{SH}\right)^{2.6} \sin 2\theta_0 \tag{3}$$

Where:

θ - drilling angle,
L - maximum throw (m),
m - charge mass (kg/m),
B - burden (m),
SH - stemming height (m),
g - gravitational constant,
k - rock coefficient.

3.4 Fragmentation analysis

The software can estimate the fragmentation through the analysis of the entered drilling and blasting parameters, i.e. estimate the size of fragmented rock material. In this example, it will also compare the fragmentation size before and after the optimization of the drilling and blasting parameters. Fragmentation of rock material (size and shape) is generated depending on the rock material properties, blasting pattern (where in open pit mines often is a blast pattern with parallel blastholes, because is easier to control fragmentation), and type of explosives (Lapčević, V. et al., 2023).

The O-Pitblast software uses the Kuz-Ram mathematical model for the calculation and evaluation of the fragmentation, which takes into account several parameters in the calculation, which can be seen in the equations listed below. A correlation between the detonation energy and average fragmentation size applied per rock volume was developed by Kuznetsov (1973) as a function of the rock type. This equation was modified by Cunningham (1983) and is given by the next equation (Fonseca de Castro, D., 2021).

$$X_{50} = A \cdot K^{-0,8} \cdot Q^{\frac{1}{6}} \cdot \left(\frac{115}{RWS}\right)^{\frac{19}{30}} \tag{4}$$

Where:

X_{50} - the average particle size (mm),
A - the rock factor,
K - the load ratio (kg/m^3),
Q - the quantity of the explosive (kg),
RWS - the relative weight energy (RWS)

Uniformity index equation, this expression was developed through field tests by Cunningham (1987). It correlates all the geometric parameters of the blast pattern, as follows (Fonseca de Castro, D., 2021).

$$n = \left[2.2 - 14\left(\frac{B}{D}\right)\right] \cdot \left[\frac{1+\frac{S}{B}}{2}\right] \cdot \left(1 + \frac{W}{B}\right) \cdot \left(\frac{|L_B - L_C|}{L} + 0,1\right)^{0,1} \cdot \frac{L}{H} \tag{5}$$

Where:

B - the burden (m),
S - the spacing (m),
D - the diameter of the hole (mm),
W - the deviation (m),
L - the total load length (m),
H - the height of the bench (m).

According to the designed parameters of drilling and blasting, the software assessed the fragmentation size, which is shown in Figure 8.

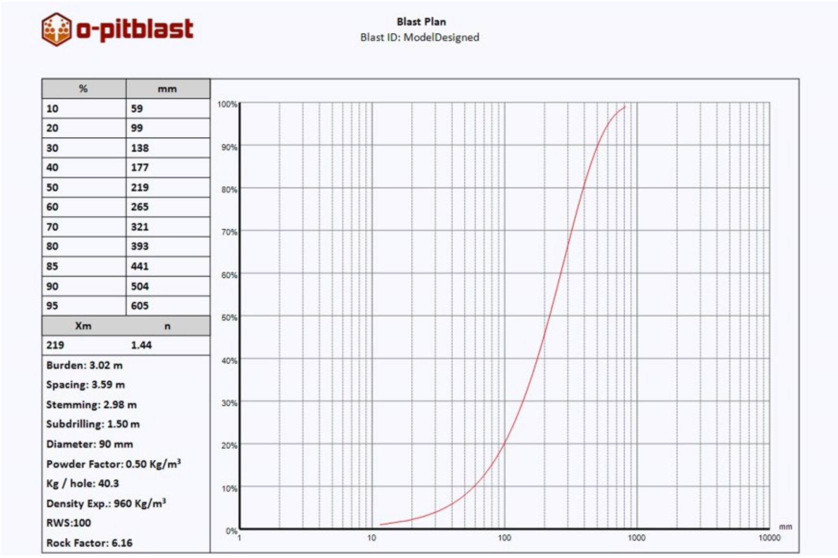

Figure 8 Fragmentation size from designed parameters in the O-Pitblast software

4 OPTIMIZING DRILLING AND BLASTING PARAMETERS

Based on the arrangement of blasthole that was made according to the designed parameters of drilling and blasting in Figure 3, where two rows of blasthole were made, and the blastfield was designed so that the last row of blasthole leaves a flat bench, and because of this, as you can see in the picture, we have a higher distance from the free face and the bigger burden. In places where the burden is bigger, auxiliary blastholes were drilled to correct the burden size and achieve better blasting results. Figure 6, shows the exact distribution of the burden on the slope of the bench, where they were drilled according to the designed inclination, while the result of the fragmentation size of the blasting is shown in Figure 8 by processing through the software.

To obtain better blasting results and reduce possible drilling costs, the parameters were optimized through software, where it will present the optimized drilling and blasting parameters, as well as the results, in the following text. The first step was carried out in correcting the blasthole pattern, with the help of the blasthole optimization tool, i.e. the first row of blastholes according to the drawn crest and toe of the bench. Such an arrangement of blasholes with the same drilling angle as with the designed parameters contributes to the fact that we have a precisely determined burden size on the entire slope. In this case, the second row of blastholes follows the first, so that the last row would leave a flat bench, the addition of blastholes was made, but this time in the last row, which is shown in Figure 9.

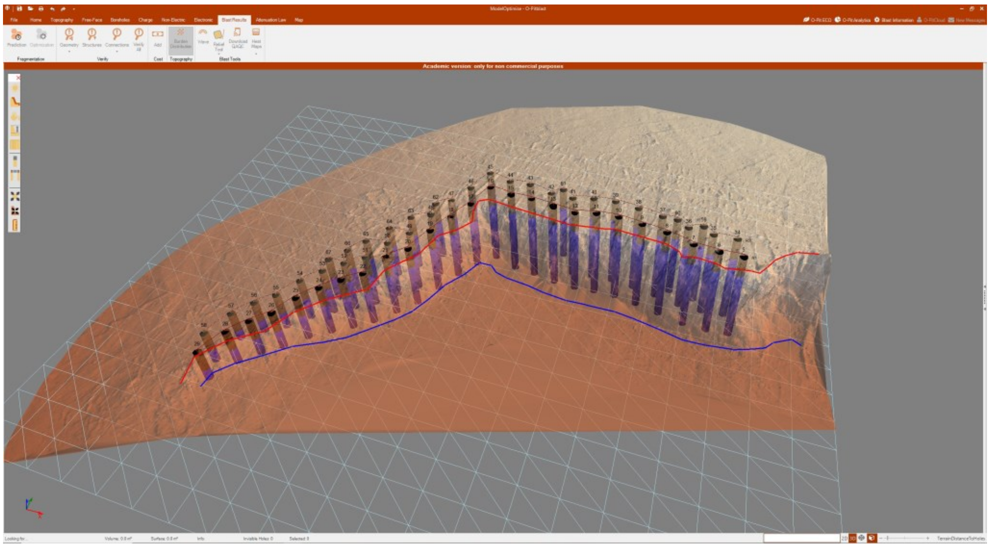

Figure 9 Blast pattern of optimized parameters in the O-Pitblast software

In this case, the subsequent design of the additional blastholes in the first row was not performed, but the software gave us an optimized first row of blastholes according to the model of the level and slope that we entered from the field surveys. This method prevents flyrock because by subsequently drilling holes in front of the first row, it is difficult to adjust the inclination and there is a possibility of pieces flyrock, where the burden is smaller. The following Figure 10 shows the distribution of zones where the burden is larger and smaller than the designed one, where it can be seen that the software gave better parameters than in the case of the previously designed blastfield.

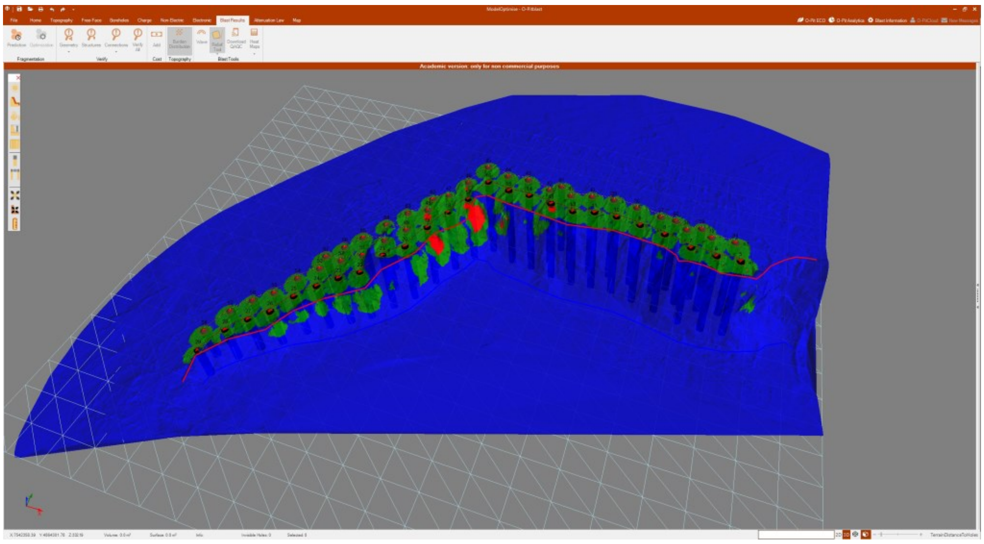

Figure 10 Burden distribution with optimized parameters in the O-Pitblast software

In the case of the designed parameters and zones, the distribution can be seen where the burden is smaller than the given one, which is indicated by the red zone, which represents the zone of possible flyrock, shown in Figure 6. In Figure 10, with the optimized parameters, that zone is reduced to a minimum. As a result of the optimization, a re-evaluation of the fragmentation size of Figure 11 was made, which shows the difference in the designed parameters from Figure 7. The parameters resulting from the optimization are shown in the following Table 3.

Table 3 Optimized drilling and blasting parameters

Parameters	Value
Burden	2.9 m
Spacing	3.3 m
Inclination of borehole	75^0
Stemming	3 m
Bench height	8.5 m
Drilled length	606.5 m
Design volume	4,662 m^3
Number of holes	59

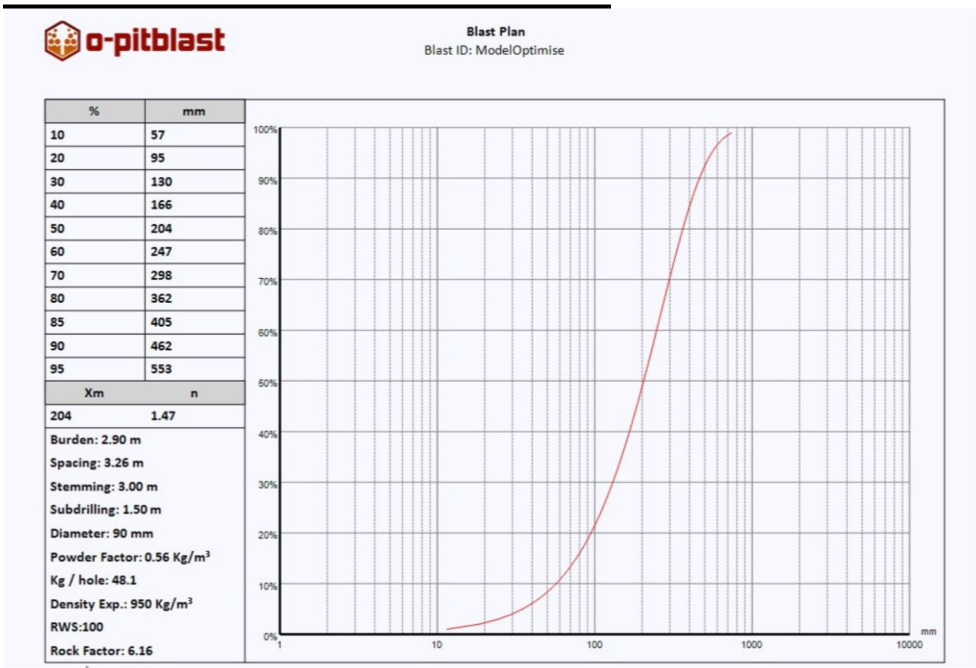

Figure 11 Fragmentation size from optimized parameters in the O-Pitblast software

5 RESULTS OF COMPARISON BETWEEN DESIGNED AND OPTIMIZED PARAMETERS

The difference between the designed parameters of drilling and blasting compared to the optimized ones is represented by the graphic shown in the following Figure 12, where both curves of fragmentation are entered, to compare the results. A comparative analysis between these parameters shows that the average fragment size is better in the case of optimized parameters by 10.2% compared to the designed blasting parameters. In the case of optimized parameters, the drilling geometry has been corrected, while the angle of the blastholes is the same.

A significant difference between these parameters is also observed by comparing the display of the free surface, i.e. the front of the slope with the value zones of the burden size, where it can be seen that in the first case, there are zones of possible flyrock, which is not the case with the optimized parameters. It can also be seen from the analysis that in the case of optimized parameters, the volume of blasted rock material increases, for a smaller number of blasthole, which also reduces drilling costs.

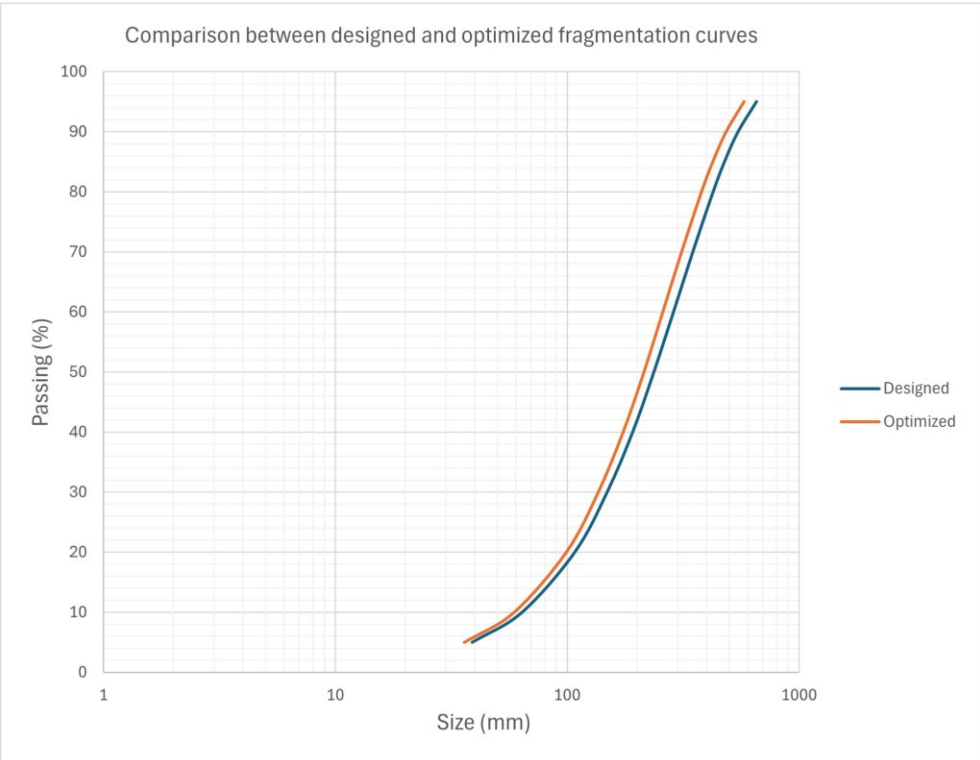

Figure 12 Comparison between designed and optimized fragmentation curves

In addition to the comparative analysis of the granulometric curve for the designed and optimized parameters, a comparison of other parameters such as the volume of blasted material and drilling length was also performed, where the differences are represented by the graphic in the following Figure 13, which clearly shows the difference between the deviation of the values of the designed (blue color) and optimized parameters (orange color). The volume of blasted material is 5.7% higher with the designed parameters compared to the optimized ones, but the drilling length is reduced with the optimized parameters by 12.2% compared to the designed ones.

The parameters optimized in this way ensure a better fragmentation of rock material, with a reduced drilling length (affecting also the reduction of drilling costs), for the small difference between the volume of the blasted material. In the next figure, we can see the difference between optimized and designed parameters which were analyzed throught this paper.

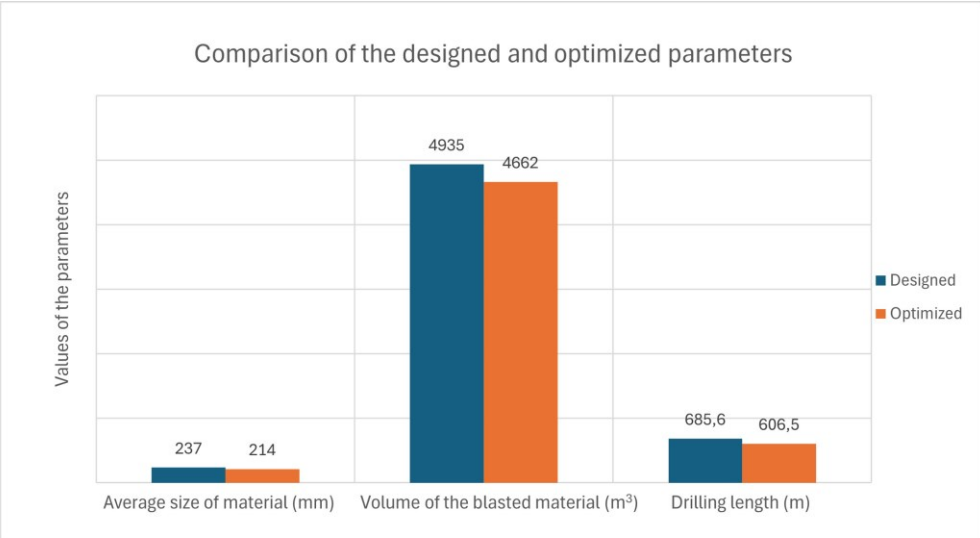

Figure 13 Comparison between designed and optimized parameters

6 CONCLUSION

By the obtained results, it can be established that the fragment size distribution curve is better in the case after the optimization of the drilling and blasting parameters through the software, in this case, the difference is also presented graphically, where can see the exact comparison of the designed results and the parameters after the optimization. First of all, the use of the software enables us to have a detailed overview of all segments from the beginning of the drilling plan to the final part of blasting and the very results that have been achieved.

In addition to the comparative analysis of the fragmentation as shown in the paper and the correction of the first order of drilling to prevent pieces from flyrock, the software enables several additional analyses to monitor blast vibration and develop the attenuation law and similar solutions. So, there are many tools in the software itself to help prevent the side effects of blasting as well as improve blasting results.

With the appearance of this software such as O-Pitblast, the control of the drilling itself has been facilitated, that is, the possibility of designing blastholes according to a real 3D model from the field and the condition of the slope, which enables the borehole to be designed with the exact burden size. By correcting the position of the blastholes, an accurate georeferenced drilling plan is obtained, which is easier to use in the field, and thus improves the quality of blasting and the results in terms of material fragmentation.

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Original scientific paper

DEFINITION OF CRITERIA AND ALTERNATIVES FOR CHOOSING THE OPTIMAL MINING METHOD DEPOSITS WHEN APPLYING MULTI – CRITERIA OPTIMIZATION

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Abstract: When solving real problems, and to make a quality decision, it is necessary to consider a great number of often complex parameters. For these reasons, the development of decision-making process modeling has seen significant growth in recent years, and multi-criteria optimization models have stood out among them as useful for solving complex and conflicting phenomena. Multi-criteria optimization models make it easier for decision-makers to find the optimal solution in situations where there are many different criteria, which can often conflict with each other. The choice of the appropriate method of exploitation of mineral deposits follows the consideration of the problem and the approach to further development, which is primarily the determination of the criteria that influence the choice of the optimal alternative.

Keywords: multi-criteria decision-making; definition of criteria; underground exploitation

1 INTRODUCTION

The methods of underground exploitation include all technological stages of preparation and excavation of part of the block or the whole deposit. Excavation takes place according to a technological process that most often includes drilling, blasting, ore crushing, ventilation, loading and export of ore. In addition, backfilling or excavation operations may be required (depending on the excavation method). Depending on the shape of the deposit, its size, depositing conditions, physical and mechanical properties of the ore and accompanying rocks, hydrological conditions, sensitivity of the surface to mining operations, mineralogical and chemical composition of the ore, distribution of minerals and the value of the raw material will refer to which method of excavation will be used during exploitation of a c definite ore deposit. As a rule, ore bodies of irregular

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shape usually must be excavated by some method with backfilling. The size of the ore body is often a decisive factor, as it reflects the amount of ore reserves.

The most successful method will be the one that gives the highest production with the most useful components in the shortest time, with the lowest consumption of energy and materials, with complete safety for employees, and without unfavorable consequences for the further development of the mine. The methods used in underground exploitation are adapted to the rock conditions, shape, dimensions, strength and stability of the ore body. For successfully exploitation, infrastructure is needed for access to workplaces, production and transport of ore, ventilation, drainage, as well as equipment maintenance.

Choosing the most suitable excavation method in underground mining is not a simple process, because it requires working with a large amount of information about the characteristics of individual excavation methods, and very often there are several possible solutions for a specific application. Multi-criteria optimization methods have proven to be very useful for ranking alternatives, especially in cases when several complex criteria need to be considered at the same time.

Multi-criteria analysis represents a sub-discipline or branch of operational research, and essentially deals with the design of mathematical and computer models that serve as support for the evaluation of a final set of alternatives in the space of a set of criteria by one or more decision makers. Since this approach analyzes many often-conflicting criteria, its primary goal is to develop a methodology that will enable the aggregation of a set of criteria based on the subjective preferences of decision makers. Achieving the mentioned goal requires, most often, the application of complex procedures and methodologies.

Multicriteria analysis methods are designed to rank alternatives based on several selected criteria. They enable the comparison of both quantitative and qualitative criteria, and at the same time include in the analysis criteria that are expressed in different units of measure. Among the many methods of multi-criteria analysis that were developed to solve real problems, the following methods stood out: AHP (Analytic hierarchy process), TOPSIS (Technique for Order Performance by Similarity to Ideal Solution), PROMETHEE (Performance Ranking Organization Method for Enrichment Evaluations), ELECTRE (Elimination and Choice Expressing the REALITY) and VIKOR (Multicriteria Optimization and Compromise Solution), because they have proven to be useful tools when modeling problems in different fields of application.

2 DETERMINING THE CRITERIA THAT INFLUENCE THE CHOICE OF EXCAVATION METHOD

The problem related to the "Borska reka" copper deposit is to choose the appropriate method of mineral raw materials exploitation from deposits. In order to achieve the goal, it is necessary to consider the problem and approach further development, which is primarily to determine criteria and alternatives. Based on the researched literature related to the choice of methods in underground exploitation and the most significant factors that influence the choice of a suitable method, three criteria are distinguished: technical, production and economic (Ataei et al. 2008; Yazdani-Chamzini 2012).

The method of mining the deposit depends on its shape, size, depositing conditions, physical and mechanical properties of the ore and accompanying rocks, hydrological conditions, sensitivity of the surface to mining operations, mineralogical and chemical composition of the ore, distribution of minerals and the value of the raw material. Therefore, all these characteristics are extremely important and should be taken into account when making decisions about the optimal excavation method.

The method of excavation mining deposits depends on its shape, size, depositing conditions, physical and mechanical properties of the ore and accompanying rocks, hydrological conditions, sensitivity of the surface to mining operations, mineralogical and chemical composition of the ore, distribution of minerals and the value of the raw material. Therefore, all these characteristics are extremely important and should be taken into account when making decisions about the optimal excavation method.

In mining, we often encounter complex structured problems where the selection of the best from a group of possible alternative solutions is performed based on several criteria. The choice of criteria depends both on the natural conditions of the deposit and on techno-economic factors.

As a very specific ore deposit due to its location at a great depth and the copper content in the ore, which is low, it requires the consideration of many criteria. Criteria and sub-criteria that affect the choice of the optimal alternative are defined. Three criteria were singled out: technical, production and economic.

The given criteria are divided into sub-criteria, and in this case eighteen sub-criteria are defined, shown in table 1. Because these are different types of criteria, which are in opposition to each other, the application of multi-criteria decision-making (MCD) methods in the process of their prioritization is completely logical and justified.

In this paper, the criteria are defined based on certain characteristics of the ore deposit and underground excavation methods that are selected as potential. An operational model is presented that includes a combination of technical, production and economic criteria for choosing the optimal method of copper mining. An overview of criteria and sub-criteria is given in table 1 (Bajić, 2020).

Table 1 Review of criteria and sub-criteria

Criteria	Sign	Sub-criteria	Sign
Technical	T	Depth	T_1
		Thickness of ore body	T_2
		Shape of ore body	T_3
		Value of ore	T_4
		Slope angle	T_5
		Rock hardness and stability	T_6
		Ore body form and contact with adjacent rocks	T_7
		Mineral and chemical composition of ore	T_8
Production	P	Productivity of the mining technology and production capacity	P_1
		Safety at work	P_2
		Environmental impact	P_3
		Ore dilution	P_4
		Ore impoverishment	P_5
		Ventilation	P_6
		Hydrology	P_7
Economic	E	Capital expenditure	E_1
		Excavation costs	E_2
		Maintenance costs	E_3

Technical criteria represent one of three groups of criteria and include the geological conditions of the deposit, such as the depth and thickness of the ore deposit, the shape of the deposit or layers, as well as their extension and dip. The physical and mechanical characteristics of the ore and the surrounding rocks, as well as the mineralogical and chemical composition of the ore, also have an important role. **Deposit depth -** In many mines of underground exploitation of mineral deposits, the ore for exploitation is located at a depth of several meters or even 100 meters. For ore bodies that are hundreds or even thousands of meters below the surface, only some mining methods can be applied, considering their depth. (Javanshirgiv & Safari, 2017; Gluščević, 1974; Genčić, 1973; Đorđević, 2018; Torbica & Petrović, 1997). At greater depths, work with open excavations should be avoided and backfilling methods should be applied, and planned excavation should also be carried out. **Thickness of ore body -** Only certain excavation methods can be applied to narrower ore deposits, while all excavation methods can be applied to more powerful deposits. Deposits of small dimensions cannot be mined by highly mechanized technological processes, due to high investments in mining machinery and accompanying equipment and short exploitation time, which negatively affect the economy of production. **The shape of the deposit -** It is a very important parameter that should be considered, because it directly affects the choice of excavation method. Irregularly shaped ore bodies usually must be mined by some backfill method, and the size of the ore body is often a determining factor, as it reflects the amount of ore reserves (Javanshirgiv & Safari, 2017). **Value of ore -** If it is a valuable ore, a less effective method is often preferred, which gives a significantly higher utilization of the ore mass over an effective method with greater ore mass losses. If the ore has a high value, it is necessary to choose a method that will achieve as little losses as possible. If the ore is poor, the losses may be greater, but the impoverishment of the ore must be as little as possible. **Physical and mechanical characteristics of ore and surrounding rocks -** The strength of ore rocks represents the ability of the massif to resist crushing of ore for a certain period. It usually depends on the hardness of the rock. Physical and mechanical characteristics are of great importance for the choice of excavation method because the extent of excavation, the method of securing, the choice of appropriate equipment for drilling and ore loading, the dimensions of safety pillars, etc., depend on them. **Ore body form and contact with adjacent rocks -** Ore bodies with a clear contact and regular ore bodies enable the application of any method if the other conditions suitable for the mining method in question are met. If the supporting rocks contain a certain percentage of metal, sublevel caving mining method will be preferred. In the case of ore bodies that have an irregular shape and where the contact is not clear, the application of some methods, such as magazine or undercutting, is excluded, or difficulties are created during work and additional costs for preparation. For irregular ore bodies, methods with backfilling of empty spaces, or methods with roof demolition of barren rocks can be applied. **The mineralogical and chemical composition of the ore** is significant due to the presence of pyrite and pyrrhotite. In the case of ores that contain large amounts of pyrite and pyrrhotite, less preference is given to shrinkage stopping and sublevel caving mining method, and a relatively higher preference to one of the methods with backfilling. In addition to the mentioned difficulties, ores with a higher amount of pyrite and pyrrhotite, if they are exposed to air and moisture for a long time, oxidize, create difficulties in the flotation process and reduce the utilization of metal.

Production criteria represent important conditions that must be considered when evaluating and making a decision in choosing the optimal mining method for a selected ore deposit. When choosing an excavation mining method, the safety of employees must be taken into account, to ensure safety against fire in the pit, intrusion of underground and surface water, as well as to ensure good ventilation in the underground mine premises. Ensuring the appropriate production capacity in terms of volume and quality has a great impact on the price of the product, in this case - mined ore, as well as on the provision of the entire production plan. With low production costs, a greater economic effect of the mine is achieved. Lower production (excavation) costs are achieved if less material and manpower are used per unit of product (tons of mined ore). The workforce in underground mining exploitation is one of the most significant costs of production, which is why it is necessary to mechanize production processes as much as possible and to use mechanization to the greatest extent possible (Torbica & Petrović, 1997). **Productivity of the mining method and production capacity -** is expressed by the production intensity coefficient, which represents the ratio of ore production in one block during the year to per unit of excavation area. It can also be defined as the speed (required time) of excavation of individual excavation blocks or parts of the ore deposit. The productivity of the excavation mining method is of great importance in the economy of a mine, given that a higher productivity of the excavation method results in a higher production capacity. Excavation mining methods in which the entire ore surface or the ore surface on several horizons can be excavated at the same time can have a high general productivity or capacity, while the intensity of the applied method itself can be low. Conversely, if mining is done on only one part of the ore surface, or the entire surface, the method used may be very intensive, and the overall productivity or capacity of the mine may be low. However, high-productivity (mass) methods of high intensity are applied to large areas of the deposit, so the capacity of the mine is also large, even though excavation is performed only on one part of the ore surface. general productivity. Safety at work Safety at work is the main requirement for any excavation method, which must be met. The cost-effectiveness of operation must not allow the method to become dangerous to the life and work of people and call into question the safety of mine installations. When choosing a method of excavation, there must be safety that the application of a certain method will not cause fire, " break-in " of groundwater and surface water, destruction of above ground and underground walls of Mines and facilities in the mine and they will not endanger mining activities and miners. **Environmental impact** damage to the soil, which reduces the surface area and at the same time changes the quality and fertility of the soil, which occurs as a result of carrying out works in the underground exploitation of deposits of mineral resources, using certain excavation methods. This disrupts the natural whole. **Ore dilution** - represents a reduction in the content of useful components in the mass of mined ore compared to the content of useful components in the body of ore before mining.

Ore impoverishment during the exploitation of an ore deposit represents the ratio of the amount of tailings that got into the mine ore to the total amount of mine ore. Methods with the destruction of excavated spaces and shrinkage stopping method are characterized by greater impoverishment. Similarly, cutting and filling methods, in which ore is loaded by mechanical means directly from the backfill (scrapers or loading shovels), have greater impoverishment. Methods cutting and filling and block caving methods as a rule, have less impoverishment than other methods. **Ventilation -** It is necessary to provide good ventilation, i.e. supply of fresh air in sufficient quantities for normal operation and extraction of mine air, i.e. harmful gases and dust. Ventilation is the primary method of removing unsafe gases and dust from underground mining operations, such as drilling and blasting, from diesel equipment (carbon monoxide) or gases originating from rocks (e.g. radon gas). Hydrology In order to secure the surface from collapse and water penetration into the pit, it is necessary to apply methods with hydraulic-cement backfilling of empty spaces, or methods with leaving permanent safety pillars. These methods must also be applied when there are important objects above the ore deposit. If there are important objects on the surface above the ore deposit, such as a railway line, main road, aqueducts, etc., the application of sublevel caving methods is excluded. Sometimes it is more economical to move those objects, divert a railway or a road, and apply some highly productive methods than the cutting and filling method, which in that case would have to be applied.

Economic criteria include mining costs such as investment costs, excavation costs and maintenance costs. The assessment of these costs is necessary for the selection of underground mine excavation methods. Investment costs are defined as the amount of investment required before the mine starts to generate income. These are the costs of creating the opening rooms, the costs of excavation equipment as well as ventilation and drainage equipment. Investments in the purchase of mining machinery during the opening of an underground mine represent the largest investment expenditure.

Excavation costs include all costs: materials needed for preparation rooms, consumables, preparation and distribution of backfill paste in the pit, manpower for ore exploitation, drainage and ventilation.

Maintenance represents constant control over all means of work, as well as the performance of certain repairs and preventive actions, the goal of which is the constant, functional training and preservation of production equipment, plants and other machines and devices. Assets wear out over time and their working capacity decreases, as well as they are subject to breakdowns, breakages and damages, so interruptions in work occur. This causes the appearance of costs due to replacement and repair of parts, but also costs due to downtime in the production process (Jovančić, 2014). **Investment costs** are defined as the amount of investment required before the mine starts to generate income. This includes research, preparation, opening...Costs of construction of the opening premises, equipment for excavation and transport such as various machines, equipment for aeration (fan) and drainage. **Excavation costs -** materials for the construction of preparatory rooms, which include costs of consumables and energy for excavation, costs of making and distributing backfill pastes in the pit, costs of crushing, transport by belt conveyors and export of ore, labor costs for ore exploitation. With low production costs, a greater economic effect of the mine is achieved. Lower production (excavation) costs are achieved if less material and work force are used per unit of product (tons of ore mined). **Maintenance costs** include the maintenance costs of machines, equipment, existing buildings and plants, and installations, mine depreciation, mineral treasure maintenance costs, administrative overhead costs...

Considering all the previous facts and considering that the "Borska reka" copper deposit belongs to the group of ore bodies with a relatively high copper content, as well as due to the depth of the deposit at which it is located and the existence of objects on the surface of the terrain above the ore body, the paper justifies 5 applicable types methods, as possible for the excavation of such a deposit. For the exploitation of the "Borska Reka" deposit, highly capacitive and highly productive mining methods were considered, which would enable economically profitable mining of ore with a low metal content.

Five different alternatives have been defined in the form of five different methods of excavation in the underground exploitation of mineral deposits. Therefore, the system of methods of excavation of underground ore deposits is shown in table 2 (Bajić, 2020).

Table 2 Proposal of alternatives as potential methods of excavation ore deposit

Alternative 1	Sublevel caving
Alternative 2	Cutting and filling
Alternative 3	Shrinkage stopping
Alternative 4	Block caving
Alternative 5	Vertical crater retreat (VCR) mining

3 MULTI-CRITERIA OPTIMIZATION

If we start from the assumption that for the majority of decisions in concrete situations, the previously defined variant of the decision-making process can be valid when breaking down a decision into its parts, as well as that decisions need to be made on the basis of arguments, it can be stated that mathematical models and optimization methods have a significant, and in some cases an irreplaceable role in the most important stages of this process. The problem of managing a certain system is often indicated as unsolvable, however, with further study, it is often established that a solution exists, and even that there are several possible solutions. Then one encounters the problem of determining the "best" solution, i.e. optimization of that system.

The multi-criteria approach is a way to describe each specific problem as realistically as possible. The task of optimization is to choose the best variant, that is, the best solution from a number of possible ones, i.e. favorable variants for the adopted criteria. The best variant is the optimal solution of the optimization task and represents a compromise between the desire, i.e. criteria, and possibilities, i.e. restrictions. Optimization is performed using different methods, depending on the type of relations in the mathematical model, criterion function and limitations (Nikolić & Borović, 1996).

The word "optimum" is a synonym for maximum good or minimum bad (Opricović, 1992). To describe and achieve the best, optimization theory or decision theory is concerned. The decision-making process contains three general steps, namely: getting to know the system, determining the measure of effectiveness and optimization.

In addition to the fact that optimization theory is a numerical procedure for determining the optimum, it also deals with those problems that are not completely mathematically formulated.

Optimization models help in the decision-making process by enabling the expert to connect all the data and relationships in each situation, and the result should enable the choice of a good one, ie. optimal alternatives, while reducing all the complexities of the task. By applying optimization methods, the expert receives information that indicates the consequences and impacts of the chosen decision.

The application of the optimization method starts from a real problem that needs to be solved. Optimization models use the approach of "discrete models" when variant solutions are projected instead of creating a comprehensive mathematical model (Opricović, 1992).

When solving the multi-criteria decision-making problem, goals are defined, i.e. with which underground mining method to excavate the ore deposit, criteria are chosen to measure the achievement of goals, alternatives are specified, i.e. which methods come into consideration, the performance of alternatives is transformed according to different criteria so that they have the same metric, they are assigned weighting coefficients criteria in order to determine their relative importance, the appropriate multi-criteria decision-making method for ranking alternatives is chosen and finally the best alternative is determined, i.e. the optimal method for the appropriate area is chosen, i.e. a proposal for the final solution is given (Petković, 2016; Opricović, 1998) .

Figure 1 shows the levels (processes) at which multi-criteria optimization takes place.

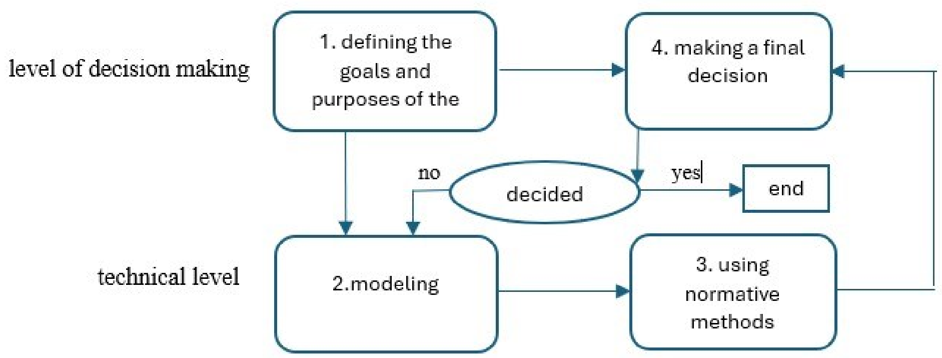

Figure 1 Schematic representation of the optimization process

At the decision-making level, the main role is played by the "decision maker". The technical level suggests a set of good decisions (alternative solutions) to the decision maker, while make it easier the final decision. This implies that the proposed solutions should be clearly, briefly and precisely explained, as well as that their number should be relatively small. In the interactive process between these two levels, the proposed solutions are modified and generally the process converges on the final solution.

The procedure for solving multi-criteria decision-making tasks depends on the "intensity of conflict" of the criteria.

Defining the problem is the first and most important step in the selection of materials using multi-criteria decision-making (Rao, 2007). After the sets of criteria and alternatives are defined, a decision-making matrix is formed, which represents the basis for the evaluation of alternatives. The second step represents the definition of the preference regarding the importance of the selected criteria by the decision maker. These preferences are expressed through weight coefficients that range from 0 to 1, where a lower value of the weight coefficient means a lower relative importance of the criteria and conversely. It should be noted that the sum of all weighting coefficients of the criteria is equal to 1. When evaluating the weights of the criteria and deciding on the optimal solution, the values of the weighting factors are determined based on subjective opinion, by ranking information by priority and importance. With this approach, the decision-maker gives his opinion on the importance of the criteria for a given decision-making process in accordance with his system of preferences.

The third step is the selection of the solution method and the determination of the aggregate function in the material selection process using multi-criteria decision-making. The decision maker has at his disposal many methods for multi-criteria decision-making. Choosing a particular method is not a simple task and depends on the specific decision problem being solved and the goals set by the decision maker. The simpler the method, the better, however, complex decision problems may require the application of complex methods. The mathematical model, which is characteristic for each method, determines the aggregate function, the so-called decision rule that shows the overall assessment of the alternative, using data from the initial decision matrix, as well as the decision maker's preferences, expressed through the weight coefficients of the criteria. Based on these functions, it is possible to perform a complete ranking of the alternatives.

In the fourth step, the stability of the obtained solution, i.e. the ranking of alternatives, can be determined by the sensitivity analysis procedure. In this step, the decision maker can analyze whether a change in the value of the weighting coefficients of the criteria leads to a significant change in the ranks of the alternatives. If the decision maker is completely sure about the importance of the criteria, then this step can be omitted.

The last step is to choose the best alternative, ie. "optimal" alternatives. The choice is simple when some alternative, according to the aggregation function, dominates over the others. However, such situations are relatively rare, so in certain situations, two or more alternatives can be proposed as a solution to the multi-criteria decision-making problem. After choosing the final solution, the chosen solution is implemented, and the effects of its implementation are monitored and analyzed (Nikolić & Borović, 1996).

During the evaluation and optimization of excavation methods in the underground exploitation of mineral ore deposits, an example of the procedure for solving problems by applying multi-criteria decision-making using the FAHP method was given. The algorithm is shown in Figure 2.

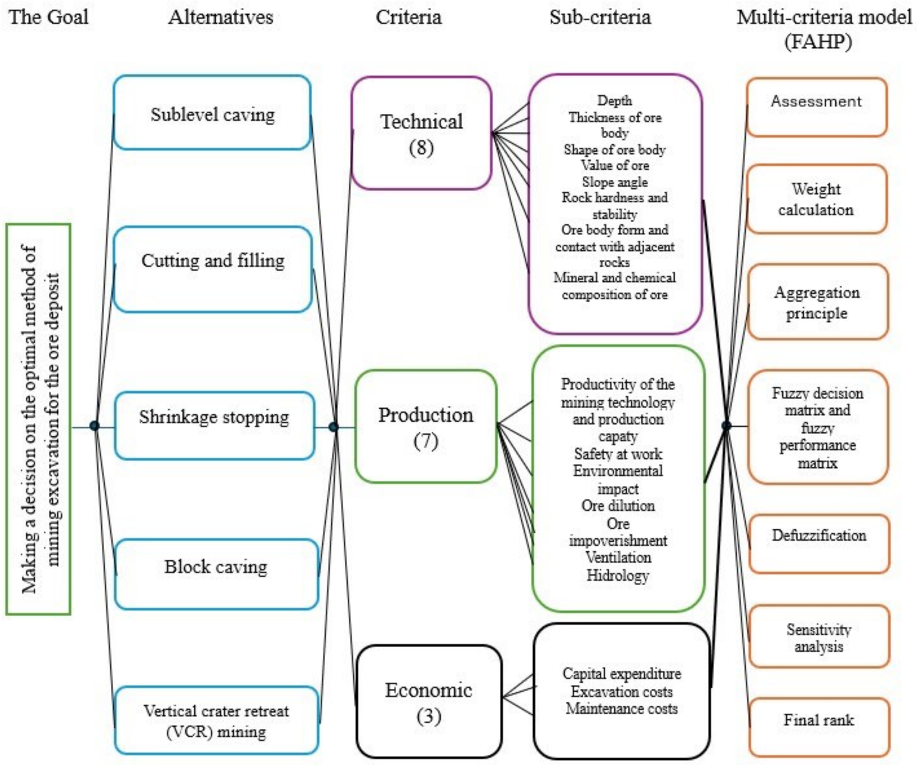

Figure 2 Algorithm for multi-criteria decision-making when choosing the optimal excavation method using the FAHP method as an example

4 CONCLUSION

The methods of underground exploitation include all technological stages in the preparation and excavation of a specific ore deposit. Deciding on the optimal method of forging in underground mines represents a series of interconnected technological processes where the efficiency of the entire process depends on the efficiency of each of the processes individually. This means that many influencing factors should be considered, starting with the characteristics of the deposit. In addition to studying the characteristics of the deposits, it is necessary to take care of the safety of the workers during the exploitation of the mine, contribute to the low ore losses, ensure the necessary production capacity, as well as the low production costs. In this regard, it can be said that deciding related to the determination of the excavation method is not at all simple.

The "Borska reka" copper deposit represents the research area during the preparation of this paper. In this example, possible criteria and alternatives are defined in order to choose the optimal excavation method during the underground exploitation of mineral deposits.

Considering that the mining process of ore deposits can be carried out in various ways, many different methods could be applied. The possibility of ore extraction from the "Borska Reka" ore body was discussed with highly capacitive and highly productive mining methods. In a view of the ore body "Borska reka" belongs to ore bodies with a low copper content, as well as it is located at a certain depth of deposit, 5 applicable types of underground mining methods were studied: sublevel caving, cutting and filling, shrinkage stopping, block caving and vertical crater retreat (VCR) mining.

Various multicriteria optimization methods are used for efficiency and to simplify the decision- making process One of them is the FAHP method, suitable for understanding imprecise and incomplete data, as well as for discovering mutual relationships between these data.

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