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Daniel Kržanović[*1], *Radmilo Rajković*[*2], *Milenko Jovanović*[*3], *Ivana Jovanović*[*4], *Miomir Mikić*[*5]

THE IMPACT OF AN ERROR IN SELECTION THE PUSHBACKS IN THE LONG-TERM PLANNING OF THE OPEN PIT PRODUCTION ON PROJECT QUALITY, CASE STUDY: OPEN PIT VELIKI KRIVELJ, BOR, SERBIA[**]

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Abstract

Today, there is a large number of software in which various mathematical algorithms for optimization the open pits are implemented. However, the importance of experience of the mining engineers in planning is still crucial for the quality of mining projects. Proper selection of the pushbacks is one of the most important planning steps in a long-term planning of production at the open pit. Therefore, the mistakes that occur in this planning step have a great impact on the project quality. Using the example of the Veliki Krivelj project, this paper presents the importance of correct selection the Pushbacks from an aspect of satisfying the practical geometric mining constraints.

Keywords: *pushbacks, mining constraints, error, project quality, open pit Veliki Krivelj*

1 INTRODUCTION

Long-term planning of the open pit production plays a key role in evaluation the mining projects, with the main objective to maximize the value realized by the excavation and processing of the mineral resources. Usually, due to the complexity of problem, the planning process is divided into phases, generating three related problems that are solved sequentially to obtain a rough production plan, namely:

1) determination of the final pit, which consists of delimiting the subregion of the mine where the excavation will be carried out,
2) selection of pushbacks, that allows to guide the sequence of excavation and to control the design,
3) excavation dynamics, which defines in every pushback when different zones will be excavated.

The main drivers in the strategic mine planning are:

1) to improve income as much as possible, and
2) postpone the excavation of unnecessary mining waste while respecting the technical limitations such as the minimum width of the working

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floor, safe angles of working and final slopes, and planned ore mining capacity.

In addition to identify the optimum pit limit, the pit limit analysis is also used to identify a series of nested pits within the final pit limit. The purpose of these nested pits is to establish a transition from the most profitable material (highest value per unit mined) in the pit to the least profitable or break-even material, which occurs at the pit limit. This understanding will aid the planner in selecting where to begin mining, and in what sequence to mine the pit out in order to produce the highest NPV from the material within the final pit limit.

Pushbacks are nothing more than a sequence of the pit limits on the basis of the alternative economic scenarios. Simply pushbacks describe how a pit will expand as the value of recovered mineral increases. The progression of pushbacks or nested pit shells roughly corresponds to the optimal evolution of the mine over time, Figures 1 and 2.

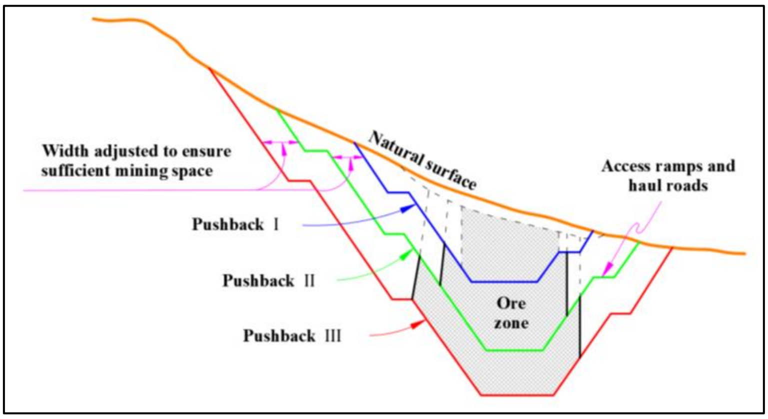

Figure 1 *Phased development of mining works at the open pit [1]*

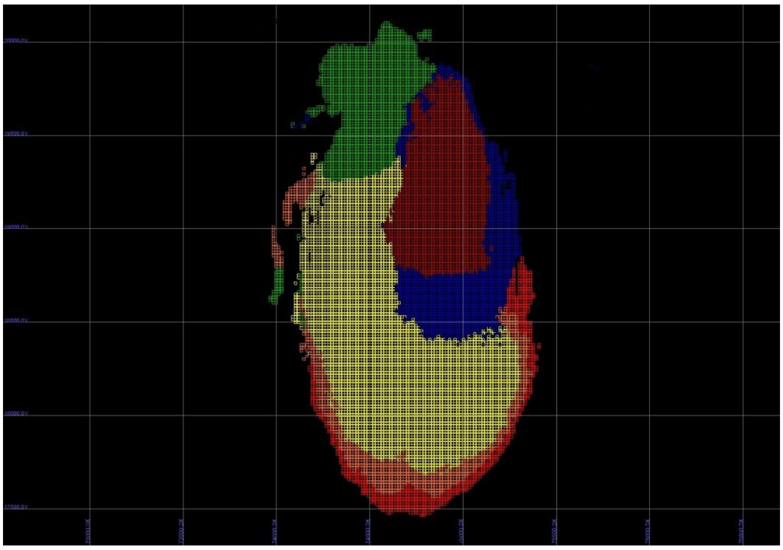

Figure 2 *Pushbacks generated in the Gemcom Gems software [2]*

2 SELECTION OF PUSHBACKS

Pushbacks (Figure 1) are essentially a series of manageable exploitation phases for an open pit mine. A pushback is ideally composed of a unique, spatially contiguous volume that can be mined with the available mining equipment and meets the practical geometric mining constraints.

Selection of a pushback is a key component to the long-term planning process of the open pit as it is critical to the final mine design and realized profit.

Pushbacks are designed to have a quick access to the high-grade ore zones of the deposit to maximize revenue in the early years. Since it is the main purpose of designing the Pushbacks, it also helps to reduce the investment risk. Another importance of using Pushbacks is that they can provide a safety zone for projects so that when the metal price on the market is not favorable, the exploitation of the ore can be temporarily stopped with the minimum losses for the mining company.

Practical considerations result in three categories of constraints on pushback design:

i. **Geotechnical constraints** – refers to the need to respect the general slope angle of an inclination in order to achieve the necessary safety for the work of people and equipment. In certain cases, when the lifetime of Pushback is shorter, it is possible to plan a steeper angle of the Pushback slope inclination, with the aim of reducing the amount of waste, i.e., the overburden coefficient.

ii. **Quality constraints** – quality and overall size constraints on the content of each pushback to meet production targets

iii. **Geometric constraints** - size and type of mining equipment - which ultimately determines the minimum working width of the stages in Pushback. Space is required for the haul road design and machine access.

Geometric constraints have a great impact on the design of Pushbacks, and if the mine planning engineers do not take these constraints into account, the mistakes occur that significantly impair the project quality. Therefore, the correct approach to this problem is to establish a compromise between the stated limiting conditions and desire to achieve the maximum NPV.

This paper is focused on the problem that exists in the selection of Pushbacks and mistakes that engineers can make in the design process when they do not take geometric constraints into account.

3 CASE STUDY

The authors of the paper have used a real example to show the impact of wrong selection of the Pushbacks by designers on the quality of the Veliki Krivelj project.

Two cases were analyzed. The Case 1, which represents the wrong selection of Pushbacks, from an aspect of not respecting the geometric constraints, and the Case 2, which correctly approaches the selection of pushbacks, taking into account the geometric constraints.

As a rule, the planning engineers are not specially trained for this type of work, but it is necessary to have a large amount of professional experience, as well as a significant knowledge of specialized software tools necessary for performing the precise and efficient analysis of design solutions.

The importance of experience in planning in mining projects comes to the fore especially when the quality of input parameters or, as in this case, geometric constraints, should be considered.

The mining plan of the deposit, which includes the optimization of the mine boundary, selection of pushbacks and optimization of mining dynamics, was carried out in the Whittle software by the mine planning engineers.

The pushback selection was carried out using a number of empirical rules on the nested pits, obtained using the methodology developed by Lerchs and Grossmann [3].

The mistake made by the mine planning engineers during the project design is related to the selection of Pushbacks. The mine planning engineers chose the Pushbacks that enable the maximum NPV, not taking into account the necessary width between Pushbacks, which enables smooth and safe operation of the mining machinery (Case 1). Figure 3 shows the contours of he selected pushbacks and final contours of the pit for the characteristic level k+245 m. The figure shows that the mining width between Pushback 1 and Oushback 2 is one block width (15 m). For the applied loading and transport machinery, the minimum safe working width of the floor is 30 m.

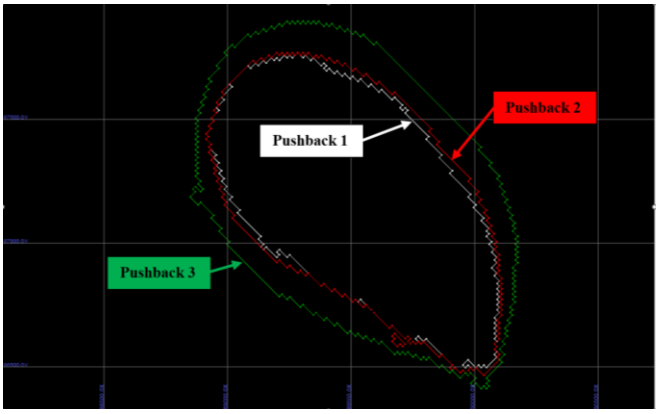

Figure 3 *Wrong selection of Pushbacks in relation to the geometric constraints, Case 1*

The correct selection of pushbacks, which also implies respect for geometric constraints, is shown in Figure 4.

In this case, the mining width between pushbacks is at least 35 m (Case 2).

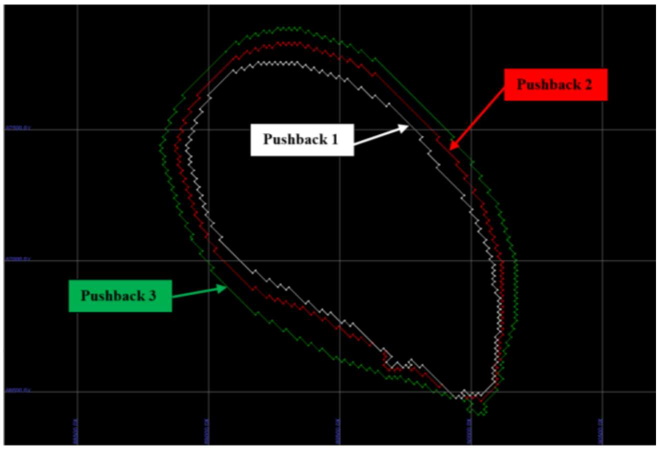

Figure 4 *Proper selection of Pushbacks in relation to the geometric constraints, Case 2*

In the optimization process of excavation dynamics, the simulation and Discounted Cash Flow Analysis (DCF) are performed in order to obtain the maximization of the net present value in a long-term planning of the open pit. The analysis is based on the Milawa algorithm, which is specifically designed to optimize the excavation dynamics of the long-term exploitation planning strategy.

Table 1 shows the obtained NPVs for the Case 1 and Case 2.

Table 1 *Maximum NPV values for the analyzed Cases*

	Case 1	**Case 2**
NPV, $	548,946,621	560,529,960

Graph in Figure 5 shows the cash flow for the Case 1 and Case 2.

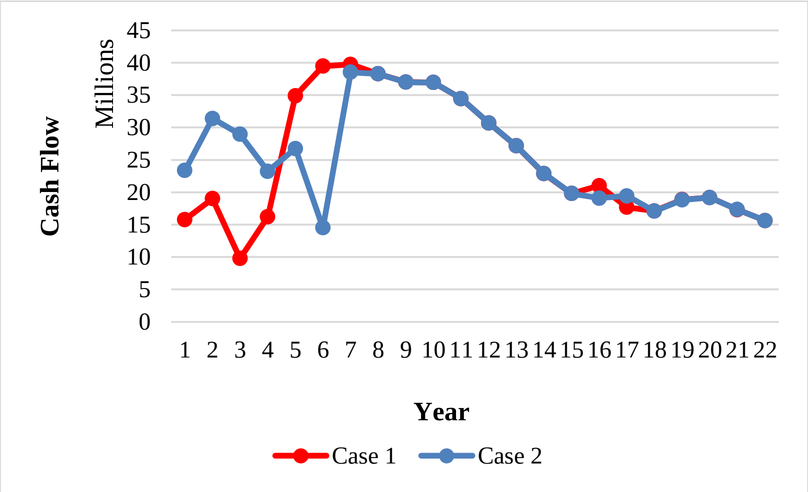

Figure 5 *View of the Cash Flow for the Case 1 and Case 2*

Thus, for Case 1, a higher NPV was obtained, but it is unrealistic considering that it is not possible to realize the planned mining activities due to the insufficient width between the pushbacks, which is necessary for the operation of the equipment.

4 CONCLUSION

In an effort to achieve the main goal in planning the mining activities, which is the maximization of Net Present Value, the mining engineers can make mistakes that affect the quality of mining projects, and therefore the impossibility of implementing such solutions in the mining processes. One such situation occurs when the mine planning engineers in selection the Pushbacks, do not respect the geometric constraints related to the minimum mining width between the Pushbacks.

Based on the analysis of two cases for the Veliki Krivelj project, Case 1, which represents the wrong selection of Pushbacks, from an aspect of non-compliance with the geometric constraints, and Case 2, which approaches the selection of Pushbacks in a correct way, taking into account the geometric constraints, it is shown that the selection of Pushbacks is one of the most important steps in the planning process of open pits and has a huge impact on the mining project quality.

The following NPV values were obtained using the DCF analysis:

- Case 1 560,529,960 $
- Case 2 548,946,621 $

Thus, for the Case 1, a higher NPV in the amount of $11,583,339 was obtained, but it is unrealistic considering that it is not possible to realize the planned mining activities due to the insufficient width between the Pushbacks, necessary for equipment operation.

In the end, it can be concluded that despite advances in the available algorithms, procedures, and software in the open pit planning, a human planner role is still necessary.

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*Saša Stepanović[*1], Nikola Stanić[*2], Miljan Gomilanović[*3], Aleksandar Milijanović[*4]*

DEFINING THE REMEDIATION MEASURES OF THE INNER WASTE DUMP SLOPES AT THE OPEN PIT GACKO-CENTRAL FIELD[**]

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Abstract

The open pit mining takes place in a heterogeneous working environment with known parameters with greater or lesser reliability. This is the main reason why the unexpected processes can occur at the open pts that endanger the safety of exploitation, people and machinery. Due to this reason, the interventions at the surface facilities in cases of occurrence of such processes are carried out with the basic function of ensuring the stability, safety of workers and mining equipment. These works require the highest priority, and they are carried out with equipment that is already present at the open pit or can be provided in a short term and based on the available information, because there is rarely enough time for the additional specific research and tests. The processes that lead to endangering the safety of exploitation are most often related to two types of phenomena, the appearance of slope instability and appearance of unforeseen amounts of water in the open pit contour, although they can also be related to the other situations (e.g., self-ignition of coal). First of all, the activities that must be carried out in these cases have to take into account two factors, the safety of works and limited time of implementation. This paper describes the phenomenon of instability of the inner waste dump at the open pit Gacko-Central Field, analysis of the phenomenon and procedure for defining the remediation measures.

Keywords: *surface exploitation, exploitation safety, slope stability, remediation works*

1 INTRODUCTION

The coal deposit Gacko, although it represents a unique complex layered deposit of lignite, is generally divided into four exploration and exploitation fields, the West (open pit Gračanica), Central, East and South Field (roof coal series). In the Western Field, the exploitation works have been completed, and currently the exploitation of lignite for the needs of the Thermal Power Plant Gacko is carried out within the limits of the Central and Southern Fields [1].

Works within the boundaries of the Central Field began in 2010, according to the Supplementary Mining Design for the expansion of the open pit Gračanica. In the beginning, the works were carried out in parallel on the exploitation of remaining reserves in the West Field, i.e., the excavation Field B of the open pit Gračanica, and since 2013, the works have been carried out exclu-sively within the boundaries of the Central Field. Solutions for this project covered the period until the end of 2015 [2].

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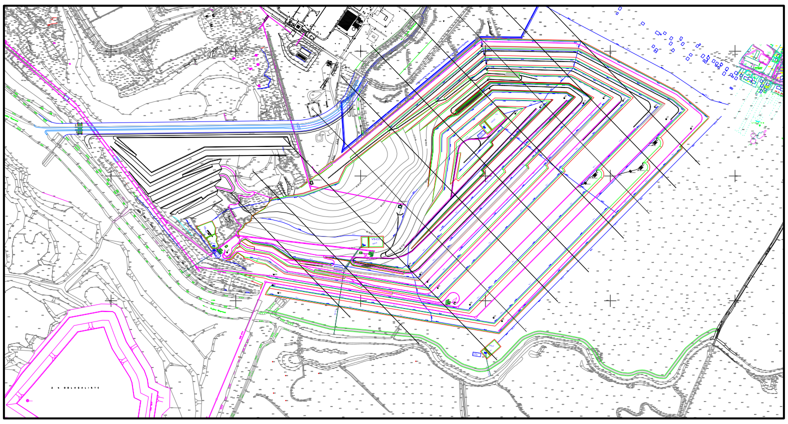

Figure 1 *Designed state of works for the end of 2015 according to the Supplementary Mining Design for the expansion of the pen pit Gračanica*

In 2016, the works were carried out according to the Simplified Mining Design, so that in 2017, the Main Mining Design of the open pit Gacko - Central Field for a capacity of $2.3 \cdot 10^6$ t/year of the run-off mine coal would be completed. This design has predicted the exploitation within the borders of the Central and South Field. Coal mining within the South Field or roof coal series is characterized above all by a favorable overburden coefficient, but also by lower coal quality, high stratification, i.e., the presence of numerous seams and inter-seams of waste within the coal series and overburden and waste with the unfavorable physical and mechanical characteristics [3].

Figure 2 *Front of the open pit mining works in the Central Field, 2015*

Currently, the works are being carried out according to the Supplementary Mining Design for exploitation the part C of the open pit Gračanica - Gacko surface. The basic design solution involves the coal mining at the roof coal series working site and central working site in a way and with equipment that is adapted to the conditions of exploitation. By carrying the out works at two sites, the possibility of coal homogenization for the needs of the Thermal Power Plant is ensured until the planned coal refining plant is put into operation [4].

Figure 3 *Front of the works of the central working site (a) and working site in the roof coal series (b) of the open pit Gacko-Central Field, 2022*

Due to the mentioned reason, it is necessary to provide all necessary conditions for safe and economical production at both mining sites.

As coal mining at the open pir Gacko-Central field is historically burdened with numerous problems, which are primarily reflected in insufficient capacities of the basic and especially auxiliary mining equipment, some of the technological solutions were forced in the previous period even though they were not originally designed [5]. As a consequence of the lack of transport capacity for waste, an inner waste dump was formed within the excavated area of the central working area, which, at the beginning, had a temporary character, but as time went on and in the conditions of deeper exploitation with an ever-increasing current coefficient of overburden, the problem of missing transport capacity was not solved, the disposed masses in the excavated area of the central working site remained permanently.

The inner waste dump was formed as a truck and bulldozer dump, and the formed floors had the parameters adopted for external waste dumps as the boundary structural parameters. Due to a lack of auxiliary equipment, the upper surfaces of the floors are only partially or not planned, and often the floors themselves deviate from their geometry. Due to the inadequate structural parameters and changed conditions of the working environment, there were phenomena indicating the occurrence of instability of the inner waste dump.

In order to systematically approach the remediation of the inner waste dump, the following phases of defining the solution, shown in Figure 4, were adopted.

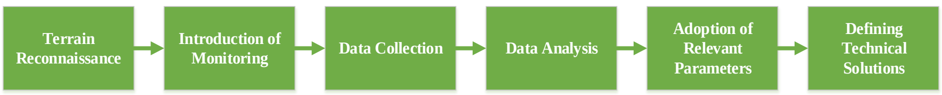

Figure 4 *Phases of defining the inner waste dump remediation measures*

2 PROCEDURES FOR DEFINING THE REMEDIATION MEASURE OF THE INNER WASTE DUMP SLOPES

2.1 Terrain reconnaissance

After the earthquake of April 21, 2022 in the region of Herzegovina at the Open Pit Gacko-Central Field and the external and internal waste dumps, the phenomena of instability the slopes of the formed waste dumps and transport roads and belt conveyor routes were observed. These phenomena of instability were particularly pronounced in the belt conveyor routes and transport roads, where the systems of cracks and fissures and mass sliding were clearly observed, which led to the violation of embankment integrity of the belt conveyor routes or transport roads.

Cracks, fissures and cracking zones were clearly visible (Figures 5 and 6).

Figure 5 *Displacement of the belt conveyor route, violation of the embankment integrity of the route and conveyor structure (April 28, 2022)*

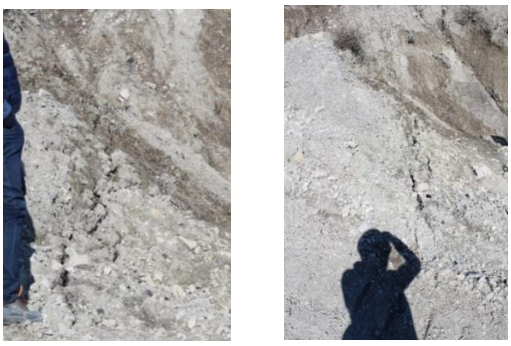

Figure 6 *Cracks visible on the slope of the East external waste dump in the immediate vicinity of the conveyor route (April 28, 2022)*

Apart from the local ones, no major stability violations of the structures and slopes of the open pit and waste dump were found.

Two months later, based on field observations, it was established that the engineering-geological processes leading to the instability have intensified (Figure 7).

Figure 7 *Cracks and cracking systems at the inner waste dump in the area of the VSC1 water collection pipeline route, on the waste dump slopes and transport roads (June 15, 2022)*

The reason for increase the intensity of processes leading to the slope instability are primarily:

- Earthquake of April 21, 2022. magnitude 5.7 on the Mercalli scale with an epicenter 16 km east of Stolac (B&H).
- Dumped mass of waste at the inner waste dump formed the slopes that in some cases exceed the designed slopes in height at the analogous external waste dumps.
- Surface of floors of the inner waste dump does not allow drainage of water that falls directly on them and is retained for a long period of time leading to this water remaining in the waste dump body and significantly affecting the change in physical and mechanical properties of deposited masses.
- Formation of floors of the inner waste dump was developed from the top to the bottom by shaking of trucks from the upper surface, which often exceeded the designed height of floors, instead of the less technologically favorable, but from the aspect of stability, safer formation from the lower level, starting from the floor of excavated coal, with a gradual increase in the height of floors.

It should be noted that apart from the earthquake, the other reasons are the result of permanent lack of the basic, and above all, auxiliary equipment at the open pit, as the result of which the waste dump was not formed and planned satisfactorily.

The intervention measures to reduce the negative effects, caused by movement of the inner waste dump, were the reconstruction of embankments and routes of belt conveyors and transport roads, their filling, planning and compaction, and then the load increase in the slope foot of the inner waste dump in the places where the sliding was the most intense (Figure 8) and establishing the monitoring of the inner waste dump stability (Figure 9).

Works on the conveyor belts and transport roads are still ongoing and the completion of planned activities is a prerequisite for the safe and reliable operation of the system. In addition to already completed or works that are in the final stage on belt conveyors of the I BTO system KLO 750 and KLM 93, it is necessary to urgently carry out the appropriate works on stabilization in the area of belt conveyor for coal transport TU 2 and drive station of belt conveyor TU 3.

Activities to stabilize the moving masses of the inner waste dump are based on load increasing in the slope foot of this waste dump.

Figure 8 *Remediation zones on the plateaus and slopes of the inner waste dump (June 15, 2022)*

2.2 Inner waste dump monitoring

Systematic monitoring and mapping of the instability phenomena of the inner waste dump, applied to the situational plan of the waste dump, was also introduced.

The directions of cracks and cracking zones, and the main directions of mass movement, were registered and marked (Figure 9).

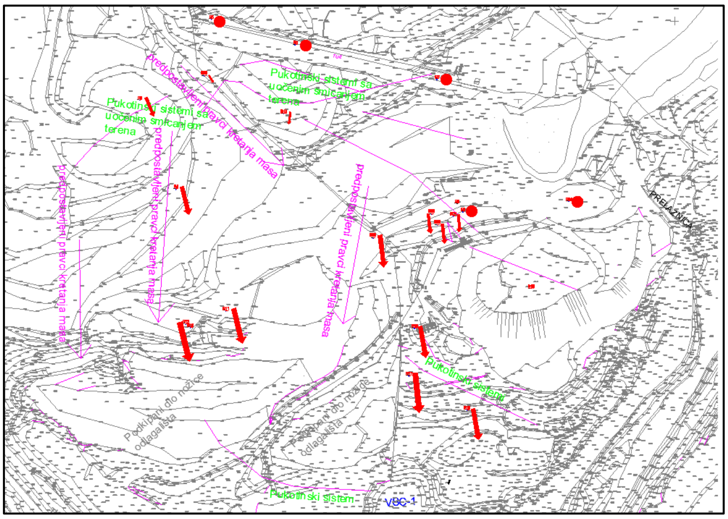

Figure 9 *Network of benchmarks (red markers) for monitoring the inner waste dump with directions of movement proportional to the intensity of movement (June 15, 2022)*

The benchmark system was established on May 31, 2022, and Table 1 gives the coordinates of reference points and displacements in a direction of all three axes for the period until June 14, 2022.

Table 1 *Results of the inner waste dump monitoring*

Benchmark	x1	y1	z1	x2	y2	z2	dx	dy	dz
R1	6541677.038	4780179.349	940.089	6541677.048	4780179.310	940.088	-0.01	0.039	0.001
R2	6541502.077	4780180.557	939.926	6541502.094	4780180.538	939.929	-0.017	0.019	-0
R3	6541716.191	4779899.039	952.022	6541716.261	4779898.932	951.960	-0.07	0.107	0.062
R4	6541762.133	4779785.438	934.533	6541762.272	4779784.807	934.389	-0.139	0.631	0.144
R5	6541807.648	4780002.655	934.016	6541807.682	4780002.641	934.029	-0.034	0.014	-0.01
R6	6541915.002	4779959.842	925.379	6541915.042	4779969.812	925.412	-0.04	0.03	-0.03
R7	6542096.526	4779925.671	909.317	6542096.549	4779925.627	909.335	-0.023	0.044	-0.02
R8	6542129.640	4779756.249	898.416	6542129.631	4779756.170	898.422	0.009	0.079	-0.01
R9	6542121.789	4779766.219	899.024						
R10	6542014.698	4779722.518	898.340	6542014.823	4779721.947	898.210	-0.125	0.571	0.13
R11	6541826.139	4779629.302	909.614	6541826.289	4779628.666	909.525	-0.15	0.636	0.089
R12	6541780.641	4779608.044	909.765	6541780.779	4779607.414	909.674	-0.138	0.63	0.091
R13	6541774.893	4779612.484	909.465	6541775.054	4779611.870	909.362	-0.161	0.614	0.103
R14	6542264.951	4779769.218	890.369	6542264.970	4779769.196	890.416	-0.019	0.022	-0.05
R15	6542216.479	4779657.402	893.023						
R16	6542067.535	4779607.151	874.706	6542067.634	4779606.675	874.658	-0.098	0.476	0.048
R17	6542060.293	4779545.954	861.547	6542060.471	4779545.386	861.467	-0.178	0.568	0.08
R18	6542134.859	4779502.437	850.821	6542135.070	4779502.016	850.757	-0.211	0.421	0.064
R19	6541900.299	4779881.608	932.236	6541900.298	4779881.460	932.215	0.001	0.148	0.021
R20	6541798.494	4779930.091	933.917	6541798.557	4779929.988	933.928	-0.063	0.103	-0.01
R21	6542117.277	4779750.417	898.093	6542117.295	4779750.247	898.019	-0.018	0.17	0.074
R22	6542089.296	4779753.119	897.980	6542089.418	4779752.760	897.843	-0.122	0.359	0.137
R23	6542096.579	4779739.231	897.951	6542096.713	4779738.925	897.825	-0.134	0.306	0.126

As there is a major infrastructure facility in the hinterland of deposited masses of the inner waste dump of the open pit, a displaced river bed of the river Gračanica, the monitoring also included the characteristic points of the river bed and, in the monitoring process, no changes in the position of reference points along the river bed were noted.

The monitoring results (Figure 10) also indicate that in the hinterland of the inner waste dump, the waste dump level has decreased, and there has been a rise on the slopes and in the foot zone. The total area affected by the movement is about 280,000 m^2 (28 ha). It is estimated that the movement included at least 1,500,000 m^3 of deposited masses.

The condition of the mined bottom coal, in a part between the front of works or water reservoir in the deepest part of the open pit and waste dump, indicates that under the pressure of moved masses the cracks they were formed in the unmined parts of the bottom coal. Considering the structure of coal series and coal seams, that marls, banded coal marls and marly clays with weaker strength parameters than coal and bottom coal seams and inter-seam waste, it can be assumed that the discontinuities arose precisely in these working environments, and what, in some places, was also observed on the ground.

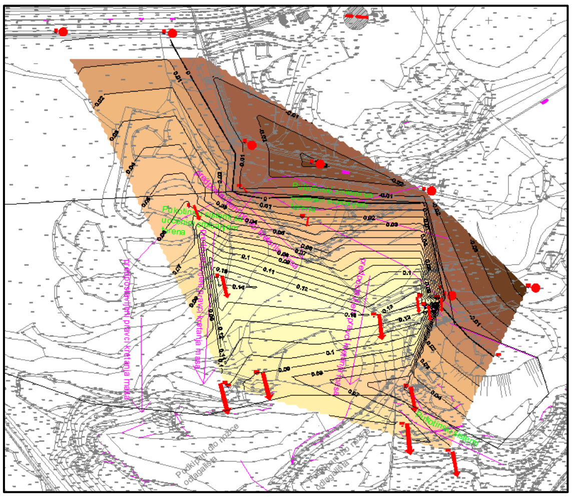

Figure 10 *Isolines of changes in the level of benchmark (in a direction of the z axis)*

2.3 Data collection

During the monitoring process itself, a significant volume of data was collected in terms of determining the dynamics of movement the unstable masses of the inner waste dump. At the same time, the collection of other important data was carried out, which are contained in the geological studies or other studies and the results of specific research and tests relating to the considered area and environment. Considering the results of specific tests, there are, first of all, the results of physical-mechanical and hydrogeological characteristics of the environment. The mentioned data, together with data on geological structure of the working environ-ment and conditions of works that preceded the observed phenomena, represent the basis to define the necessary remediation measures.

2.4 Data analysis and adoption the relevant values of the working environment parameters

The observed and mapped cracks and cracking systems indicate that the masses move in the form of hectometer-sized blocks in a direction of the work front, that is, along the gradient of the excavated coal bottom (Figures 9 and 10).

In order to more accurately describe the existing slope of the inner waste dump for characteristic sections, that is, for slopes at places of maximum height and slope, a slope stability check was carried out. Figure 11 shows two characteristic cross-sections on which calculations of the slope safety factor of the inner waste dump were done.

The most important input data for calculation the slope stability at characteristic sections refer to the physical and mechanical parameters of deposited masses, their structure, degree of cracking and presence of water in deposited material.

Currently available data, specifically related to the deposited waste, are insufficient, unreliable and often very old, and do not reflect the real state of changes in the working environment during the century of exploitation and with the progress of works. In order to solve this problem, it is necessary to take a critical look at the available data necessary for the required stability analysis and check them using one of the fast test methods, or apply the reverse analysis method, compare the available values of physical-mechanical parameters with the values for similar materials at the other open pits, or literary va-lues.[6] Only after a comparative analysis, a closer idea can be obtained on the input data quality necessary for the calculations as well as an assessment of the result reliability of the slope stability check [7].

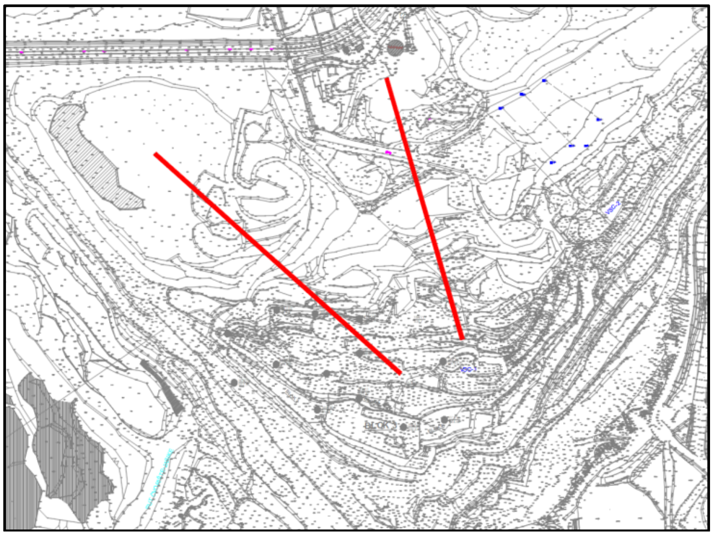

Figure 11 *Isolines of changes of the benchmark level (in a direction of the z axis)*

2.5 Defining the technical remediation measures

Considering the importance of the observed engineering-geological phenomena on:

- development of the production process and necessary provision the appropriate amount of higher quality coal for the needs of the Thermal Power Plant,
- safety of people at the open pit,
- safety of equipment and facilities, primarily the water reservoirs, transport roads and main belt conveyors for transport the overburden and coal,
- size of the affected area,
- volume of the moved deposited masses,

the activities on stabilization the inner waste dump are defined.

In this particular case, the following is recommended:

1. To continue works on increasing the load on slope foots of the inner waste dump by planning with a bulldozer. Masses for loading the slope foot should be provided by planning the upper surfaces of floors of the waste dump and softening the slope system to an angle of 12 to a maximum of 14^o, as much as the available space allows.
2. By planning the waste dump surface and construction of channels, to ensure the drainage of water that falls directly on the waste dump surface.
3. During planning, if possible, to fill cracks and cracking zones, and by multiple passes of the equipment, to increase the compaction of material in those zones in order to prevent the infiltration of surface water during precipitation into the waste dump body.
4. Deposit the waste in the zone between the existing southern slope and inner waste dump up to the level of belt conveyor of the combined waste system. Disposal should be carried out in the extreme western part with formation the upper edge up to the geological profile 76-76'.

5. To bring the pumping station on a reservoir to the designed state and thus ensure a low level of water in the reservoir. If possible, clean the water reservoir and deepen it in order to ensure better drainage of the working environment in the vicinity of water reservoir.
6. Continuous monitoring the inner waste dump, with time shortening between two readings.
7. The mentioned activities should be accompanied by preparation the appropriate technical documentation that will elaborate in detail the development of works in the area of inner waste dump in the function of its stabilization.
8. It is necessary to urgently prepare an Elaborate on the stability of the open pit and waste dump slopes as a part of regular monitoring the condition of mining works and facilities at the open pit and development a project for additional engineering-geological and hydrogeological explorations.

3 CONCLUSION

Implementation of emergency measures at the open pits, in cases of unexpected disruptions to the stability and safety of production, must be implemented in accor-dance with all prescribed safety measures and in the shortest possible time. The problem is when the stability of surface exploitation facilities has already been disturbed, and they must be brought to a stable state by remedial measures. In such cases, a systematic approach to the planning, designing and implementing the remediation activities is necessary including the phases in Figure 4.

The applied procedures, techniques and technologies, type, characteristics and method of application the equipment, must be previously checked on the basis of all available, and if this proves to be insufficient, the additional data and information must be provided so that the planned works can be carried out in a safe manner.

Another important factor, time, is directly related to the costs of implementation of these works, and the most common rule is that the partial solutions give partial results and reduction of the costs of remediation works later results in delay the basic mining works, while a comprehensive and initially more expensive solution enables faster and safer works on reaching the full production capa-city.

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SYSTEM FOR WATER COLLECTION THAT GRAVITIES INTO THE OLD BED OF THE RIVER ČEHOTINA[**]

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Abstract

Functioning of the drainage system has a great importance for the efficient and continuous exploitation at the open pits. Modern exploitation is realized at the open pits with increasing depth and surface, so it is of a crucial importance to accept as much water as possible before it enters the open pit contour. By this way, the loading of a drainage system inside the open pit is greatly reduced, but it also prevents the negative effects of water on the massif. This paper, using the example of the open pit Potrlica, presents how the already existing system, which fully solves all problems in a dynamic environment such as surface mining of lignite, can be improved. Timely improvement of the system can greatly relieve the existing drainage system and improve the exploitation system.

Keywords: *surface mine, drainage, river bed regulation, drainage factors, pumping station*

1 INTRODUCTION

Protection of the open pit from water is an important technological operation, and the exploitation of mineral raw materials at the open pits largely depends on its application. Therefore, the investments in the drainage system are necessary. The existence and functioning of the drainage system at the open pit directly generates costs, as well as the system of excavation, transport and disposal of overburden. A good drainage system generates costs that are fully in line with the system work. A bad system not only generates costs but can also cause a temporary suspension of works. In those situations, in addition to the costs, the suspension of work prevents the generation of profits for a long period of time.

Correct dimensioning and timely investment in order to form a drainage system for a longer period of time can reduce the drainage system costs. In addition to reduce the operating costs of the functioning of drainage system, it is possible to create such conditions in the deposit that will affect the increase in profits; some examples are given in the work of El Idrysy and Connelly [1]. In this paper, the drainage system at the Pljevlja surface mine is presented. A special

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review was made on the way the new technical solution functions after the relocation the main watercourse in the wider area of the open pit and improvement the system in the newly created situation.

2 DESCRIPTION OF THE CURRENT STATE

The open pit Potrlica is located in the territory of the municipality of Pljevlja in the northern part of Montenegro. The process of exploitation has been going on since the 80s of the 20th century, and the general direction of development the works is the northeast-southwest. The previous process of exploitation from the point of view of drainage had before it several key elements that affect its functioning, namely:

- Periodically increasing the depth of the open pit from the moment of opening.
- The position of the Ćehotina watercourse in the area where exploitation is planned.
- Large inflow of underground water from the Tvrdaš and Kutlovača streams.

Figure 1 (TK 1:25000) shows the position of the natural bed of the Ćehotina river (purple line), position of the Ćehotina river bed after relocation in 2004 (blue line) and position of the new projected route of the Ćehotina river (red line). Figure 1 also shows the position of the Tvrdaš outcrop, as well as the boundary of the Potrlica-Kalušići deposit.

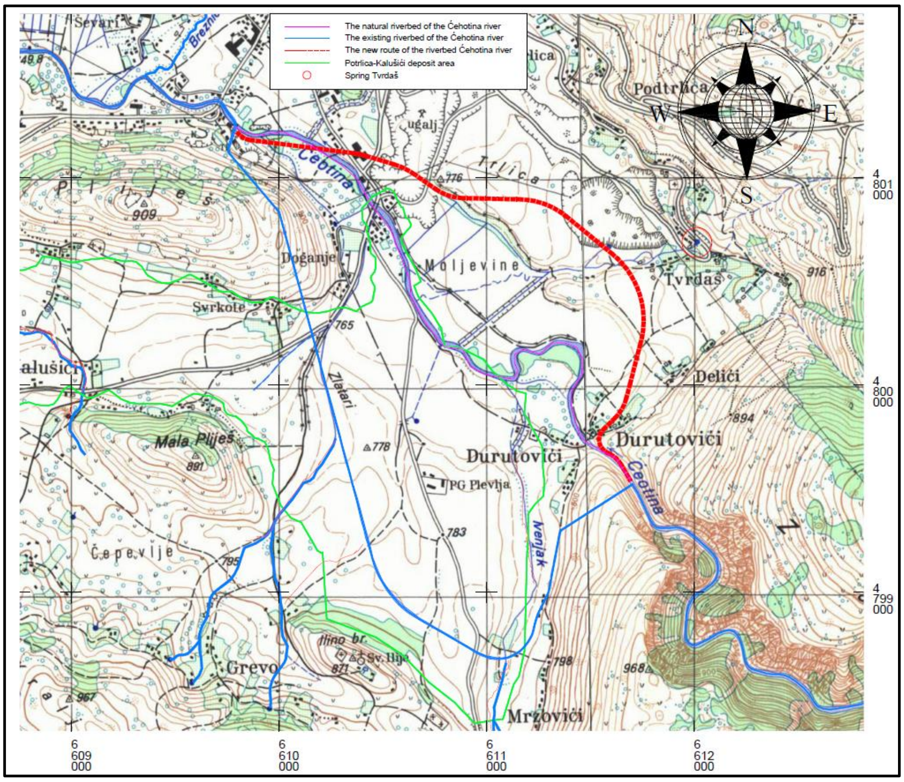

Figure 1 *Position of the bed of the Ćehotina river in different stages of exploitation at the open pit Potrlica-Kalušići*

The exploitation process as well as the drainage system successfully fit into the mentioned natural conditions, so that during the exploitation process there were no major problems from the point of view of drainage. In 2004, in order to ensure the continuation of exploitation, the first relocation of the Ćehotina river was carried out to the position where it is today. The increased inflow of water from the outcrops, which at one point was 400 l/s seconds from the Tvrdaš outcrop [2], as well as the problem of permanent increase the depth of the open pit, was solved by the correct dimensioning of water evacuation system from the open pit. The values of the amount of pumped water for the period from 1998 to 2018, shown in Table 1, were taken from the Supplementary Mining Design for the coal exploitation at the open pit Potrlica - Pljevlja for the period from 2020 to 2025 [3].

Table 1 *Quantities of water pumped from the central part of the open pit Potrlica in the period 1998-2018 [3]*

Godina	Mesečne količine ispumpane vode (m^3)												Ukupno (m^3)
	I	II	III	IV	V	VI	VII	VIII	IX	X	XI	XII	
1998.	809.178	776.208	665.994	657.516	986.274	673.530	621.720	543.534	637.734	618.894	1.120.980	1.058.808	9.170.370
1999.	876.060	878.886	1.042.794	866.640	786.570	711.210	669.762	559.548	390.930	332.526	585.924	1.080.474	8.78.1324
2000.	1.035.258	938.232	1321.626	1.309.380	1.010.766	684.834	519.984	438.972	383.394	433.320	410.712	474.768	8.961.246
2001.	803.526	727.224	1.000.404	821.424	1.026.780	788.454	690.486	682.950	668.820	972.144	862.872	913.740	9.958.824
2002.	761.136	895.842	888.306	1.079.532	990.042	743.238	720.630	848.742	820.482	1.379.088	987.216	873.234	10.987.488
2003.	1.344234	1.172.790	1.090.836	1.182.210	1.011.708	832.728	725.340	649.980	664.110	646212	815.772	907.146	11.043.066
2004.	1.139.173	1.146.398	1.370.380	1.131.145	1.199.383	1.147.201	1.074.146	1.098.198	830272	923303	1.061.575	1.400.566	13.521.740
2005.	1.196.024	1.037.362	1.584.716	1.526.094	1227.884	958.349	892.717	835.369	943.693	1.091.524	971.730	1.211.317	13.476.779
2006.	1.118286	923.940	1.542.661	1.520.996	1.438.160	1.114.463	1.096.940	864.043	797.137	820.076	873.601	943.056	13.053.359
2007.	1.117.012	1.093.435	1.220.875	1239.354	778.658	1.043.734	1.072.408	996.581	782.482	1.038.636	1.310.720	1.589.814	13283.709
2008.	1.408.849	1.176.271	1.446.444	1364.245	1348.952	1.055.203	1.296.065	1.171.811	851299	905.461	1.016.971	1.672.650	14.714.221
2009.	1.512.076	1.412.035	1.628.046	1.607.656	1359.148	1.193.476	1.329.836	1.117.012	973.642	1.122.746	1.452.818	1.468.746	16.177.237
2010.	1.487.016	1.373.472	1.712.052	1290.708	1350.216	1.274.292	1.149.804	1.186.740	1.038.312	1.042.632	1.162.944	1.582.200	15.650.388
2011.	1.394.712	1.215.936	1.373.112	1230.552	1376.064	1.264.896	1.222.200	1.218.240	1.010.376	1.022.904	1.037.880	1.100.664	14.467.536
2012.	1.069 200	1.177.200	2.123.280	1.969.200	1.804.320	1.535.040	1.386.000	1.173.600	1.102.320	1.125360	1.129.680	1.270.800	16.866.000
2013.	1.622.160	1.770.480	2.163.600	1.910.880	1.889424	1.562.400	1.364.400	1.231.200	1.033.200	1.069.920	1.021.680	1.044.000	17.645.040
2014.	1.251.360	1202.400	1.265.040	1.643.040	2.471.760	2.061.360	1.824.480	1.468.080	1.211.040	1.501200	1.398.240	1.928.880	19.226.880
2015.	1.699200	1.933.200	2.278.800	2236.320	2.008.080	1.548.000	1.528.740	1.433.664	1.026.684	1.177.848	1.280.448	1.192.896	19.343.880
2016.	1.205.892	1.440.504	2.283.876	2.057.472	1.828.332	1.772.928	1.746.936	1.601.244	1.303.020	1.359.792	1.673.748	1.501.380	19.775.124
2017.	1.254.456	1.495.908	1.699.740	1.571.832	1.936.404	1.692.216	1.396.044	1.417.932	1.212.048	790.305	817.650	1.296.260	16.580.795
2018.	1.282.435	1.159.596	1.370.730	1.786.464	1.461.816	1.435.140	1.545.948	1.410.444	1.174.140	964.458	844.938	1136970	15.573.079
Ukupno	**25.387.243**	**24.949.319**	**31.073.312**	**30.002.660**	**29.252.437**	**25.092.692**	**23.874.586**	**21.947.884**	**18.855.135**	**20.338.349**	**21.838.099**	**25.648.369**	**298.260.085**

Currently, the process of exploitation, that is, the position of the working levels of the open pit are in the immediate vicinity of the active bed of the Ćehotina river (at about 150 m), so it is necessary to relocate it as soon as possible. Figure 2 shows the position of excavation levels of the open pit and position of the existing Ćehotina river bed, as well as the new route of the river bed after relocation.

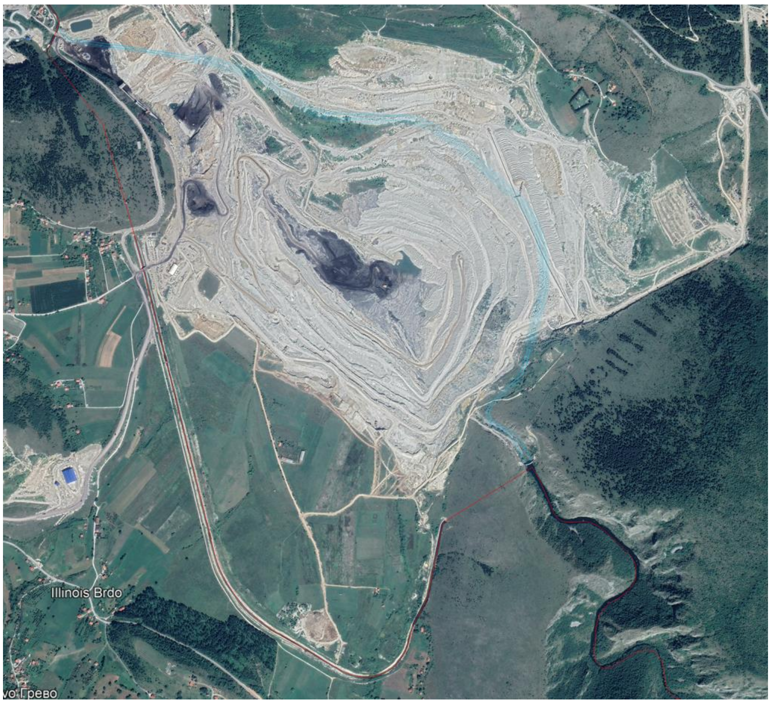

Figure 2 *State of works at the Potrlica open pit (Google Earth satellite image)*

3 DESCRIPTION OF THE PROBLEM

3.1 Introduction to the problem

The current situation requires the continuation of exploitation in the Coal Basin Pljevlja in the coming period with realization the significant investment steps such as the relocation of the Ćehotina river. The primary reason for realization of this investment is to ensure the safe continuation of exploitation at the Potrlica deposit, as well as the creation of conditions for expansion of the open pit in the area of Kalušić, and thus the extension of the mine life.

From the point of view of functioning the drainage system, the relocation of the Ćehotina river will bring significant changes, not so much to the way of its function, but to the change of water flow into the contour of the open pit. Position of the objects to be defended, but in the configuration of the surrounding terrain, the surface, cover, direction and fall that gravitate towards and from the objects, will also depend on the amount of water that comes into the contour of the surface mine [4]. Changing the flow direction of the main recipient completely changes the orientation of catchment areas that gravitate towards the open pit. Depending on the position of exploitation objects in space, size, slope and type of surface, an analysis of distribution the catchment areas and water inflow from them into contour of the open pit was carried out for the condition before and after relocation of the Ćehotina river.

As it can be seen in Figure 3, before relocation the Ćehotina River into the open pit area, water flows from the catchment areas P2, P3 and P4, while water from the catchment area P1 flows directly into the Ćehotina River.

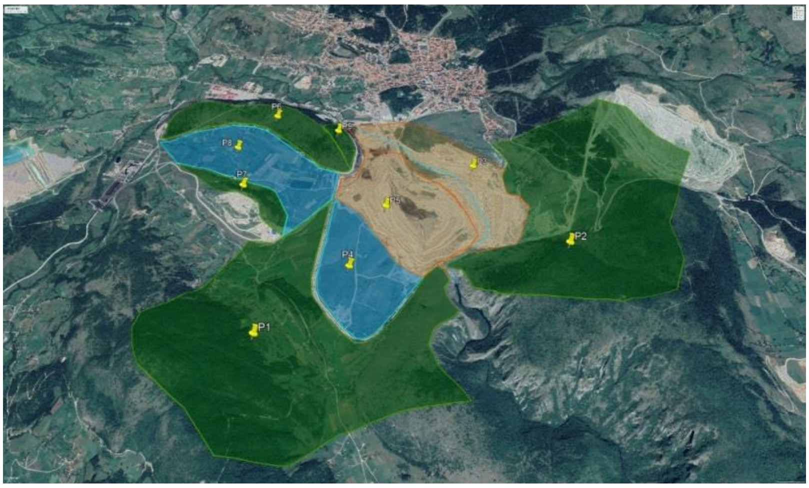

Figure 3 *Positions of catchment areas in the wider area of the open pits Potrlica-Kalušići after relocation of the Ćehotina river (Google Earth satellite image)*

After relocation of the Ćehotina river, the distribution of catchment areas is different; the new bed will collect water that gravitates from the catchment area P2 and P3, as well as a part of water from the catchment area P9. The old bed of the Ćehotina river will collect water from the catchment slope P1 and part of P4, P6 and P8. Figure 4 shows the layout of the catchment areas after relocation of the Ćehotina River.

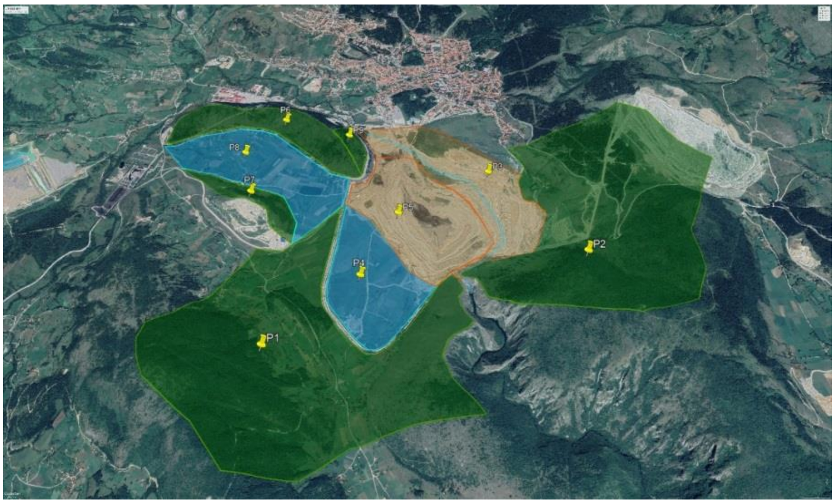

Figure 4 *Positions of catchment areas in the wider area of the open pits Potrlica-Kalušići after relocation of the Ćehotina river (Google Earth satellite image)*

3.2 Elaboration of the problem

The old river bed, through which the river flowed before relocation, retains its hydrological function even after relocation, until it is completely cut by the exploitation works and its connection with the tunnel under the Velika Pliješ hill is cut off. In accordance with the dynamics of development in order to continue the exploitation, the existing bed will be cut by the mining works. From this moment, all water from the eastern catchment area (P1), as well as several streams that flow into the bed, will be directed towards the contour of the open pit. During the forecast calculation of the flow of water into the intersected bed, a shortened version of the catchment area P1, position is shown in Figure 4, was considered. For the purposes of the analysis, a catchment area of 2.5 km^2 was adopted, as well as a runoff coefficient of 0.6. Since the expected groundwater inflow is not known, it is estimated at 50 l/s. The expected inflow of water into the cut bed for rainfall with a return period of 50 years and rainfall intensity lasting 8 hours is 4.47 m^3/s.

This amount is not negligible, but it is certainly about precipitation that can be expected rarely, once in 50 years. Certainly, the climate change has affected the changes related to the precipitation. So, in a situation where the bed is left open towards the open pit contour and working floors, all water that falls on the aforementioned catchment area would directly flow into the open pit. On that occasion, there would not be a successive inflow of water into the open pit contour, but water would flow into the open pit contour via the working floors in the form of watercourse of a temporary nature. The inflow of water into the open pit contour in this way and in such amount would certainly adversely affect the exploitation process. This would result in the reduced mobility of equipment in the inflow zone, but also deterioration of the geomechanical characteristics of the environment in the wider inflow zone. Also, all water that reaches the open pit in this way, which is about $2x10^6$ m^3/year, would have to be evacuated from the main reservoir to the recipient. In this way, in addition to the amounts shown in Table 1 (in the continuation of exploitation, a slightly smaller amount of water for pumping is expected), it would be necessary to pump an additional 2×10^6 m^3 of water/year with a pumping height of 100 m.

4 ALTERNATIVE SOLUTIONS

The possibility of free flow of collected water to the existing watercourse after the mining works cut the old river bed of the Ćehotina River is no longer possible, and as an alternative to evacuating the collected water there are two ways (variants). Controlled discharge of water into the open pit contour and subsequent pumping of water from the bottom of the opent pit to the recipient (V1) or by direct pumping of water from the old river bed of the Ćehotina river to the recipient (V2). In order to avoid the additional pumping of significant amounts of water from the bottom of the open pit, as well as the additional installation of pumping units and pipelines, but also the generation of significant costs, the variant V2 was taken into consideration, which is the direct pumping of water from the old bed of the Ćehotina river.

In this variant, the entire system was considered as a drainage system inside the open pit, so during the calculation, the system was dimensioned in accordance with the way of dimensioning the water reservoirs and pumping stations inside the open pit. [5] In accordance with the calculation for reception of this water, it is necessary to provide a reception area of 100,000 m^3 and it is necessary to install on it a pumping station with a capacity of 290 l/s, with an additional reserve in the capacity of 90 l/s, and the expected evacuation time of collected water is 290 h.

In accordance with the conditions on the ground, the expected dynamics of development the mining works and request for the necessary accommodation capacity in order to prevent the inflow of collected water into the contour of the mine, it is necessary to block the bed with an embankment and carry out its additional arrangement. Evacuation of the collected water to the recipient is carried out by a pump-pipeline system.

By dividing the bed with an embankment and installing a pumping station, it can be said that a water reservoir is formed on the terrain surface. This kind of phenomenon may not be common for the open pit exploitation, but it is certainly possible to come across such cases. The open pit Gacko drainage system has one object of this type in its system [6].

4.1 Additional arrangement of the old bed of the river Ćehotina

In order to ensure the formation of a reception area in the old bed of the Ćehotina river, it is necessary to carry out the following works:

- Construction of the Embankment-1 in order to close the bed and possibility of forming a temporary reservoir.
- Construction of the Embankment-2 in the area of the "Ivenjak" stream in order to ensure favorable conditions for pumping out water.
- Revision the bottom of the riverbed from Embankment-2 to the concrete part of the riverbed.
- Elevation of the embankment in order to secure the floodplain up to the level of 779 m above sea level.

Figure 5 *Position of the Embankment-1, Embankment-2 and embankment overhangs of the Ćehotina river bed*

4.2 Construction of the Embankment-1 and Embankment-2

The barrier Embankment-1 ensures the closure of the old bed of the Ćehotina river and enables the formation of water accumulation. Position of the Embankment-1 was chosen in accordance with the developed dynamics of works until the 10th year, but also with the requirements of the reservoir volume.

According to the shown calculation, the required capacity of the water reservoir is 100,000 m^3, and the reservoir volume from the intended position of the embankment to the Durutovići tunnel for the water mirror level up to the elevation of 779 m above sea level is about 127,000 m^3. This volume refers to the space without priorarrangement of the bottom and construction of the Embankment-2.

The barrier Embankment-1 will be built up to the level of the flooded area (elevation 779) in two sub-levels on the side towards the open pit and one level towards the water accumulation. In order to ensure the water tightness, the embankment will be made of clay and protective film. For the purposes of forming the Embankment-1, it is necessary to install 3235 m^3 of clay and 1900 m^2 of protective film. Figure 2 shows the cross-section of the Embankment-1 barrier.

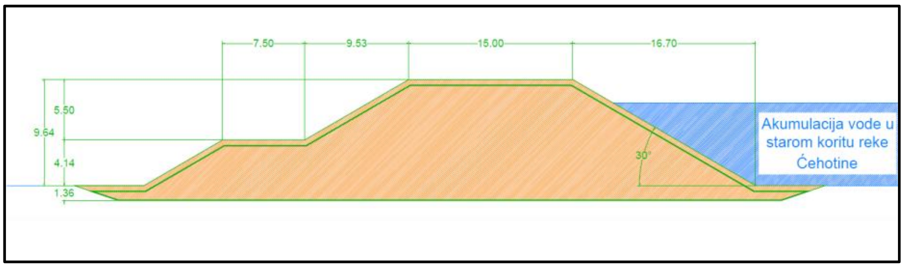

Figure 6 *Layout of the Embankment-1 barrier*

Due to the construction of the Embankment-1, the deepest part of the bed is located next to the embankment itself, and thus the position for installation the pumping station is predetermined. The long distance from the recipients would require large investments to form a pumping station, but also frequent displacement of pipelines due to the progress of exploitation works.

In order to avoid this unfavorable situation, and also to avoid a large retention of water in the event that the pumping station is moved to the location upstream, it is necessary to construct the barrier Embankment-2 in the area of the Ivenjak stream and arrange the riverbed from the Embankment-2 to the concrete part of the riverbed. The works on the arrangement of the riverbed involve changing the longitudinal slope of the riverbed towards the Durutovići tunnel. In order to ensure the connection of a bed part between the Embankment-1 and Embankment-2 with a bed part in which the pumping plant is located, the barrier Embankment-2 will be built up to the level of 778 m above sea level. The remaining water from the first part of river bed, after lowering the water level below the elevation 778, will be pumped into the river bed part where the pumping station is installed.

By changing the longitudinal dip of the bed, a local depression is formed, suitable for positioning the pumping station for water evacuation to the new bed of the Ćehotina river. The depth of bed in this part after the reconstruction is 5.9 m. The barrier Embankment-2 will be constructed in the same way as the barrier Embankment-1, which requires 2500 m^3 of clay and 1800 m^2 of protective film.

Unlike the barrier embankments, the excavated material in the open pit from the higher floors will be used to arrange the bottom of the river bed. The surface layer (about 50 cm) of material, used to arrange the riverbed, will be made of clay in the same way as in the case of creating the barrier embankments. In order to realize these works, it is necessary to provide 15,000 m^3 of material.

Analyzing the configuration of the embankment around the bed of the Ćehotina river and predicted flood area up to the level of 779 m above sea level, it was observed that in the part of bend near the "Ivenjak" stream, the embankment is significantly lower compared to the predicted flood area. In order to prevent water from flowing out, it is necessary to build the embankment in this part. The total length of the embankment that needs to be raised is 150 m, and the maximum height of overhang is 3 m. For realization of these works, it is necessary to install 5000 m^3 from the excavation from the open pit. During installation, the material should be spread in layers of up to 0.25 m and passed with a flat roller 6 times over the full surface of the poured material.

5 COMPARISON OF ALTERNATIVES

Table 2 shows a comparison of the electricity amount consumed for water evacuation that reaches the open pit contour (V1) from the catchment area P1, that is, the old bed of the Ćehotina River (V2).

Table 2 *Comparison of the evacuation methods from the catchment area P1*

	V1	V2
Pumping station capacity, m^3/h	2304	1044
Amount to pump out, m^3/year	2,000,000	2,000,000
Number of discharge hours h	868	1,916
Installed power of pumping station, kW	2,440	165
Total electricity consumption, kWh	2,118,056	316,092

As it can be seen from Table 2, the Variant V2 is significantly more favorable. The main reason for this is that with the application of this variant, the pumping units of lower power are engaged.

In comparison the two mentioned variants, the designed capacities and pumping stations without the required legal capacity reserve were taken into account. Also, in comparison the variants, the necessary investments were not considered, which are significant in order to arrange the bed of the old bed of the Ćehotina river and make an embankment, as well as for formation a new pumping station. As the pumping out analysis did not include the entire drainage system, but only the water from the catchment area P1, the comparison did not take into consideration the investments for expansion the pumping station inside the open pit in the Variant V1.

If the necessary investments were included in the comparison process, the difference between the variants would certainly be smaller, but even in that case the Variant V2 would be more favorable than the Variant V1.

6 CONCLUSION

The water evacuation system at the open pit Potrlica surface mine can be characterized as a good one. In addition to all mentioned aggravating factors, the large inflow of groundwater, its replenishment due to the great proximity of watercourses, as well as the permanent deepening of the pit and increase in the pumping height, it fully met all the requirements that were put before it.

This paper provides an example of how such system can be improved and become even better. By accepting the water that gravitates towards the bottom of the open pit before it enters the contour of the open pit (or the higher levels of the mine), it significantly reduces the cost of water evacuation, but also facilitates the entire process of exploitation. One of the negative things that follows this way of working is the wide spread of drainage facilities, not all of them are centralized, which certainly requires the additional costs.

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ANALYTICAL MODEL FOR DETERMINING THE AVAILABILITY OF CONTINUOUS SURFACE EXPLOITATION SYSTEMS, CASE STUDY: I BTD SYSTEM OF THE OPEN PIT DRMNO[**]

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Abstract

This paper presents an analytical model for determining the availability of a continuous coal system at the open pit Drmno. A continuous coal-fired system is a technical system with sequential connection of elements in terms of reliability and functioning. Realization of the safe production is the main goal of continuous systems at the open pits. In order to achieve this goal, it is necessary to determine the availability of a continuous system. To determine the availability, data related to the I BTD system of the open pit Drmno Kostolac, more precisely the base with various downtimes over a period of 3 years (2016-2018), was used. This paper presents a case study for determining the availability of a continuous system at the coal open pit Drmno using an analytical model. The model can be applied to determine the availability of continuous systems at the open pit in the function of design, planning and implementation of production and maintenance systems and, as such, is applicable in the other industrial areas as well.

Keywords: *availability, continuous system - I ECC system, open pit*

1 INTRODUCTION

At the open pits of the Electric Power Company of Serbia, the continuous systems are used for coal mining. These are high-capacity, complex mining systems, the operation of which is extremely important for the reliable supply of coal to the Thermal Power Plant. Continuous surface mining systems represent the systems where the flow of material is continuous. The application of high capacitive continuous systems reduces the costs of transport, and therefore the total costs of exploitation itself. In this regard, the availability of one system was analyzed on an example of the open pit Drmno.

Coal mining in the Kostolac Basin began in 1870. The open pit Drmno is the only active mine in the Kostolac basin. The open pit Drmno produces 25% of coal (lignite) in Serbia, see [1].

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Based on the long-term monitoring and development of the lignite open pits, it has been proven that the continuous systems with rotary excavators and belt conveyors are the most efficient loading and transport system for these needs, see [2]. The mechanization, applied as a part of these systems, is complex and made according to the special requirements because these systems must satisfy and be adapted to the specific working conditions [3].

This paper presents a case study for determining the availability of a continuous coal system at the open pit Drmno consisting of the following elements (subsystems): rotary excavator SRs 400, self-propelled conveyor BRs 2400, a series of conveyors and crushing plant. The following figures show the elements of the I BTD system of the open pit Drmno.

Figure 1 *Rotary excavator SRs 400*

Figure 2 *Self-propelled conveyor BRs 2400*

Figure 3 *Belt conveyors*

Figure 4 *Crushing plant*

2 AVAILABILITY

According to the ISO-IEC standard, the availability is defined as: "The ability of a technical system to be in a state in which it can perform the required function, under the given conditions and at a given moment of time, i.e., during a given time interval, assuming that the necessary supply is provided (external resources)" [4,5].

The availability is calculated on the basis of a time state picture, in which times when the system is in the "up-time" differ with the times when the system is in the "down-time" [6,7]. The temporal picture of the state is shown in Figure 5.

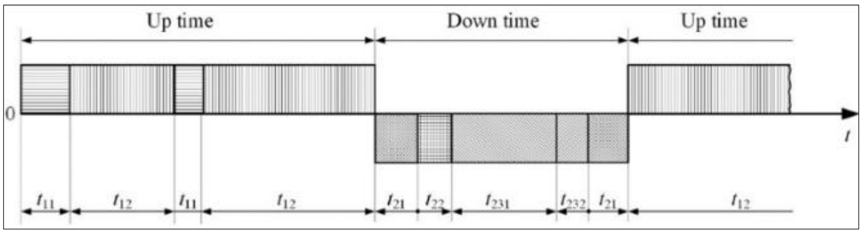

Figure 5 *Time picture of the state [6,7]*

The availability is also determined as the quotient of the total time during which the system is in a correct state and the total time that makes up the time in correct operation and the time in failure [8,9]:

$$A(t) = \frac{\sum t_{11}, t_{12}}{\sum t_{11}, t_{12}, t_{21}, t_{22}, t_{231}, t_{232}} \quad (2.1)$$

Operational availability $A_o(t)$ from a denominator above the mentioned equation (down time) excludes losses of an organizational and logistical nature.

$$A_{\mathrm{o}}(t) = \frac{\sum t_{11}, t_{12}}{\sum t_{11}, t_{12}, t_{231}, t_{232}} \quad (2.2)$$

The internal availability is obtained when only the active corrective maintenance time $A_i(t)$ is taken into account:

$$A_i(t) = \frac{\sum t_{11}, t_{12}}{\sum t_{11}, t_{12}, t_{231}} \quad (2.3)$$

The availability can also be shown as a ratio of *MTBF* and *MDT* indicators,

$$A = \frac{MTBF}{MTBF + MDT} \quad (2.4)$$

- $MTBF$ *- mean time between failure*
- MDT *- mean down time in failure*

It is usual to display the availability as a number or coefficient k_A, but in certain situations and under certain assumptions, the availability can be displayed in the form of function *A(t).* In that case, the assumptions about the exponential distribution of reliability $R(t) = e^{-\lambda t}$ and the convenience of maintenance $M(t) = 1 - e^{\mu t}$ are used, where λ and μ are the failure and maintenance intensities determined by:

$$\lambda = \frac{1}{MTBF} \quad \text{and} \quad \mu = \frac{1}{MDT}. \quad (2.5)$$

The availability function *A(t)*, then takes the form:

$$A(t) = \frac{\mu}{\lambda + \mu} + \frac{\lambda}{\lambda + \mu} \cdot e^{-(\lambda + \mu) \cdot t}, \quad (2.6)$$

from where the stationary availability value is obtained as:

$$A = k_A = \lim_{t \to \infty} A(t) = \frac{\mu}{\lambda + \mu} = \frac{1}{1 + \frac{\lambda}{\mu}} = $$
$$= \frac{1}{1+B}, \quad (2.7)$$

where k_A represents the availability coefficient and is obtained when $A(t)$ is calculated for $t \to \infty$, i.e., when the availability value becomes stationary [9].

3 ANALYTICAL MODEL TO DETERMINE THE AVAILABILITY

Based on the data obtained from the Electric Power Company of Serbia and the open pit Drmno, surface mine, a database was created related to the electrical (cable break, TT connection break, etc.), mechanical (damage to the superstructure bearings, broken tracks, replacement of teeth, etc.) and other failures (repair, service, etc.) of the I BTD system in a period of three years (2016-2018). Table 1 presents a part of the failure database that was used for the model. Table 2 presents the failures of the I BTD system at the open pit Drmno for a period of three years (2016-2018).

Table 1 *Presentation of a part of database on failures of the I BTD System*

Date	Month	Year	System	Facility	Failure	Start of failure	End of failure	Time in failure	Total time in failure (min.)	Note	Shift
1/1/2016	January	2016	I BTD	RB SRs-400	Electrical	10:00:00	10:50:00	00:50	50	/	1
1/1/2016	January	2016	I BTD	Crushing Plant	Others	13:00:00	14:30:00	01:30	90	/	1
1/1/2016	January	2016	I BTD	RB SRs-400	Electrical	19:00:00	19:10:00	00:10	10	/	2

Figure 6 presents a view of the regular connection of the I BTD system at the open pit Drmno.

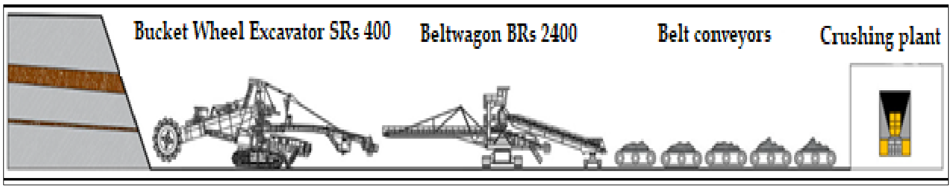

Figure 6 *Layout of the BTD system at the open pit Drmno [7,10]*

Table 2 *Failures of the BTD system at the open pit Drmno for a period of three years (2016-2018)*

Ord. No.	System	Type of failure	Sample size	Parameter μ	Parameter λ
1.	I BTD system	Electrical failures	983	0.01778	0.00163
2.	I BTD system	Mechanical failures	1504	0.01476	0.00226
3.	I BTD system	Other failures	2414	0.00679	0.00230
4.	I BTD system	All (EMO) failures	4901	0.00956	0.00474

λ *– parameter – failure intensity*
μ *– parameter – maintenance intensity*

Figure 7 presents the distribution of the I BDT system failures by types of failures in total failures for the period 2016-2018.

Figure 8 presents the distribution of the I BTD system failures by facilities for different types of down time for the period 2016-2018.

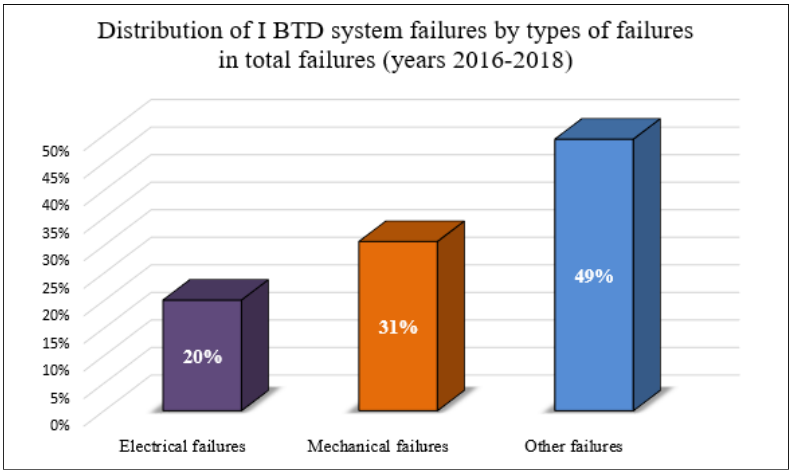

Figure 7 *Distribution of the I BDT system failures by types of failures in total failures for the period 2016-2018*

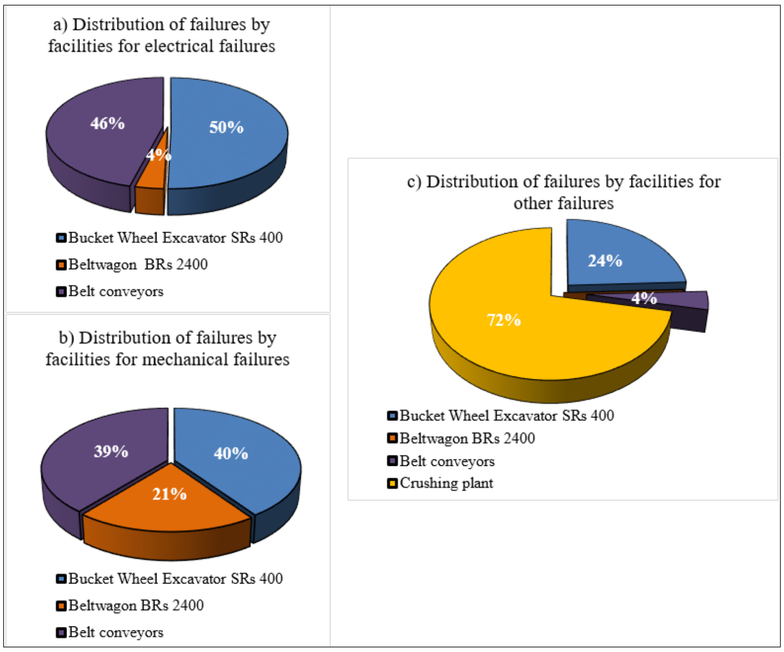

Figure 8 *Distribution of the I BTD system failures by facilities for different types of down time*

Based on the results of statistical processing of data on the operation time until failure and repair time, the analytical expression for availability is of the form:

$$A(t) = \frac{\mu_{EMO}}{\lambda_{EMO}+\mu_{EMO}} + \frac{\lambda_{EMO}}{\lambda_{EMO}+\mu_{EMO}} \cdot e^{-(\lambda_{EMO}+\mu_{EMO})\cdot t}, \quad (3.1)$$

$$A(t) = \frac{0.00956}{0.00474 + 0.00956} + \frac{0.00474}{0.00474 + 0.00956} \cdot e^{-(0.00474+0.00956)\cdot t},$$

$$A(t) = 0.66854 + 0.33146 \cdot e^{-0.0143\cdot t},$$

where: $k_A = \frac{\mu_{EMO}}{\lambda_{EMO}+\mu_{EMO}} = \mathbf{0.66854}$ – availability coefficient, i.e., stationary value of the availability A.

4 CONCLUSION

The availability of the I BTD system at the open pit Drmno, calculated by this model, corresponds to the real state of this system in the field. For the high-capacity mining systems such as the continuous coal mining (I BTD), it is important to determine its availability in order to define the picture of the system state necessary in the planning phase. The time in which the system is not in operation entails the high economic and production costs. This model has a role to help responsible persons (engineers) at the open pit in planning and control of exploitation, adopting the appropriate maintenance strategy, all with the aim of stable coal production and cost reduction. The availability of the specific system as a whole is the basic input data for the production planning at the open pit Drmno, but also other activities in the field of planning, production monitoring or equipment maintenance.

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SELECTION OF A RATIONAL SOLUTION FOR TRANSPORT OF THE BY-PRODUCTS FROM THE COMBUSTION PROCESS IN TPP PLJEVLJA TO THE MALJEVAC LANDFILL[**]

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Abstract

This paper presents an analysis of selection the optimal model for transportation of by-products from the Thermal Power Plant Pljevlja to the Maljevac landfill. The analysis was done for two types of transport, truck transport and transport with belt conveyors. The paper presents the results of analysis the technological process of transport the by-products from the Thermal Power Plant Pljevlja to the Maljevac landfill, and evaluation the most important criteria for selection the optimal solution. During the selection of criteria and evaluation, three criteria were singled out as follows: ecology, economy and reliability of the system. The analysis showed that the most favorable variant from the aspect of capital costs and aspect of reliability is the variant of truck transport. The variant of continuous transport is more favorable from the aspect of normative costs. The environmental criteria had to be met by both variants.

Keywords: *transport, transport optimization, cost rationalization*

1 INTRODUCTION

The location of the Thermal Power Plant Pljevlja is situated in the industrial zone of the town of Pljevlja, on the fourth kilometer of the road Pljevlja - Djurdjevića Tara - Žabljak, at the altitude of 760 m.

The Maljevac landfill belongs to the group of wet landfills because finely ground waste (slag and ash) is hydraulically transported and disposed of in the form of hydro mixture. The mixture, transported by pipelines, consists of water and ash (in a ratio of 1:6 to 1:10). Through the pipeline, the mixture is brought to the landfill where the ash is deposited. Through the overflow structure, located on the right side of the landfill, the water from the landfill surface is drained by gravity to the dredging station, thus forming a closed system of recirculation the transport water.

Construction of a partition took place in two phases. In the first phase, the basic dam was built, with a crown elevation of 790.5 meters above sea level, and in the second phase, the embankments 1, 2, 3 and 4 steps were successively constructed with a final maximum elevation of 810 meters above sea level.

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Currently, the works are being carried out on remediation the Cassettes 1 and 2, while ash disposal is being carried out in the area of Cassette 3. According to the valid documentation, the Cassette is being built in several phases, and the construction phases of a landfill up to the levels 817 and 821 have been completed. Currently, the works on construction the embankment at level 824 are being completed.

Based on the examples from the region for the thermal power plants of similar or the same capacity that burn coal, the transport of by-products from the thermal power plant to the ash landfill can be divided into two parts:

- Internal transport.
- External transport.

Internal transport means the transport of ash, slag and gypsum from the place of production to the place of storage inside the thermal power plant. For ash, it is usually compressed air, which is transported to the silo where it is reloaded. Slag and gypsum are most often transported by conveyors with a rubber belt.

External transport means the transport of by-products from the place of storage inside the thermal power plant to the place of permanent disposal (ash landfill). Different modes of transport are used in practice. Table 1 shows the types of transport with examples from the region. The locations of the ash landfills are situated in the immediate vicinity of the thermal power plant.

Table 1 *Types of transport with examples from the region*

Location	Gacko	Ugljevik	Kostolac	Kolubara	Obrenovac
Ash	Hydro mixture	Truck	Hydro mixture	Hydro mixture	Hydro mixture
Slag	Truck	Truck	Belt conveyor /Truck	Truck	Truck
Gypsum	-	Truck	Belt conveyor		

2 ANALYSIS OF VARIANT SOLUTIONS

As a part of the Analysis, two variants of the transport of ash, slag and gypsum from the Pljevlja Thermal Power Plant to the Maljevac ash landfill, i.e., to the Cassette 3, were considered.

As a part of the first variant, the transport by trucks with a carrying capacity of 25 t is planned.

In the second variant, the transport is provided by conveyors with a rubber belt.

In both cases, the transport from three separate silos located in the vicinity of the Pljevlja Thermal Power Plant to the Cassette 3, the Maljevac ash landfill, was considered for a work system in four brigades (3 working, 1 on vacation). The capacity of silos for ash storages, slag and gypsum is shown in Table 2.

Table 2 *Characteristics of the silo after reconstruction the Thermal Power Plant Pljevlja*

	Slag	Ash	Gypsum
Silo capacity, m^3	400	3200	600
Discharge rate, m^3/h	50	200	83
Bulk density, t/m^3	0.95	0.8	1.2

Dimensioning of the transport system, i.e., its capacity, should ensure the transport of by-products of the Thermal Power Plant Pljevlja throughout the year, for the estimated operating time of the Thermal Power Plant Pljevlja after reconstruction is about 7500 h/year. For the purposes of the analysis, the annual amount of by-products that need to be taken to the landfill was adopted. The expected annual quantities of materials are:

Ash	420,000.00 t
Slag	70,000.00 t
Gypsum	154,000.00 t

Variant 1 - Truck transport of by-products of the Thermal Power Plant Pljevlja

In the Variant 1, the truck transport of by-products of the Thermal Power Plant Pljevlja to the Maljevac ash landfill, i.e., to the Cassette 3, was analyzed. For the purposes of the analysis, the dump trucks with a carrying capacity of 25 t and box volume of 15 m^3 were adopted. Comprehensive recommendations on selection the machine types for defining the discontinuous loading and transport systems, are given in the Manual for the Surface Mining [1]. Calculation of transport costs is a direct function of distance between the current position of the loading vehicle and unloading point.

The technological processes included in this analysis are:

- Loading
- Transport outside the ash landfill contour
- Transport within the ash landfill contour
- Disposal of by-products
- Leveling and planning of disposed material

The material is loaded into trucks after placing the truck under the silo opening. The material is loaded into the means of transport by pouring it directly into truck box using a funnel that avoids dust emission.

Transport outside the ash landfill takes place on separately constructed roads for two-way traffic, and the maximum slope of transport roads is below 8%. The total height difference of transport outside the contour of the ash landfill is 66 m. The total length of transport from the place of loading is 2235 m.

Transport of ash inside the ash landfill is carried out along the already formed embankments and surfaces of dam and cassettes to the Cassette 3 embankment, where the material is unloaded. The transport of combustion products is planned to be carried out according to a transport scheme with a loop; the same route is used for the full and empty trucks.

During the analysis for transport calculation, a Mercedes-Benz Actros 4141 truck or trucks with similar characteristics were used. Figure 1 shows a view of the Mercedes-Benz Actros 4141 truck.

Figure 1 *Truck Mercedes-Benz Actros 4141*

Material at the truck unloading point is planned with a bulldozer or crawler loader. In order to prevent the dust raising during the truck unloading along the embankment, and in the places predicted for unloading, it is necessary to install a dewing system.

Figure 2 shows the transport route for trucks.

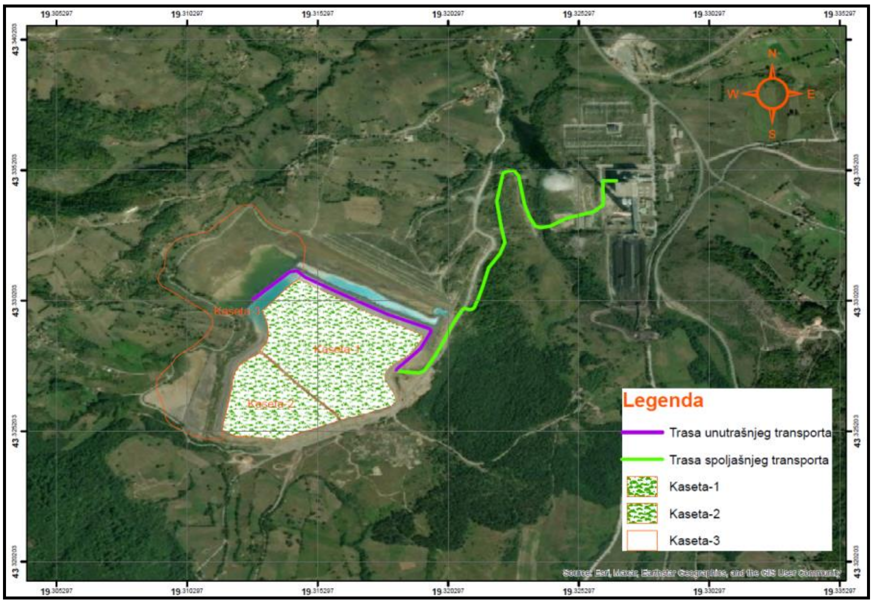

Figure 2 *Route of transport roads*

Numerical analysis of ash, slag and gypsum transport capacity - Variant 1

To meet the annual capacity needs and working conditions, transport routes, transport cycle time, work organization, etc., the MERCEDES-BENZ ACTROS 4141 trucks were used.

Analysis of the transport capacity of ash, slag and gypsum

The transport calculation was done using the simulation model-software Talpac 10.2. The software Talpac simulates the technological phases of loading and transport, based on the operational and technological parameters of these phases, and results in the operational capacities of loading and transport machinery. The software package Talpac represents a simulation model of the loading and transport process at the open pits. The software enables optimization of the transport fleet, calculation the technical and economic parameters of the equipment's operation, such as the cycle length, capacity, etc. [2,3,4]

The analysis of the transport system was carried out for the following initial conditions:

- Operating hours of the Thermal Power Plant, 7500 h/year,
- Shift duration, 8 hours,
- Available time of the effective operation of a truck, 5300 h/year,

- Annual ash capacity, 420,000 t, 525,000 m^3 (γ=0.8 t/m^3),
- Annual slag capacity, 70,000 t, 73,684 m^3 (γ=0.95 t/m^3),
- Annual gypsum capacity, 154,000 t, 128,333 m^3 (γ=1.2 t/m^3),
- Maximum route slope, < 8%,
- Minimum bend radius, 15 m,
- Trucks with a carrying capacity of 25 t (box volume 15 m^3).

Ash transport calculation

The analysis was performed individually for each material separately. Considering the requirements regarding the ash transport capacity, the transport capacity for one to four trucks was analyzed.

The maximum amount of material that the truck transports in one cycle can be limited either by the load capacity of the truck or volume of its box. In this case, the box volume is the upper limit. Table 3 shows the results of ash transport calculations.

Table 3 *Summary of the ash transport calculation results*

Truck No.	Truck capacity (t/h)	System capacity (t/h)	Required capacity (t/year)	Required time to realize the required capacity (h/year)
1	51.26	51.26	420,000	8193
2	50.99	101.98	420,000	4119
3	50.89	152.7	420,000	2751
4	50.87	203.48	420,000	2064

Slag transport calculation

Considering the requirements regarding the slag transport capacity, the transport capacity for one truck was analyzed.

The results of calculation the slag transport are shown in Table 4.

Table 4 *Results of the slag transport calculations*

Truck No.	Truck capacity (t/h)	System capacity (t/h)	Required capacity (t/year)	Required time to realize the required capacity (h/year)
1	59.93	59.93	700,00	1168

Gypsum transport calculation

Considering the requirements regarding the gypsum transport capacity, the transport capacity for one truck was analyzed.

The results of calculation thegypsum transport are shown in Table 5.

Table 5 *Results of the gypsum transport calculations*

Truck No.	Truck capacity (t/h)	System capacity (t/h)	Required capacity (t/year)	Required time to realize the required capacity (h/year)
1	7422.	74.22	154,000	2075

Reliability, the probability of no-failure operation of the truck transport system is given in Table 6. Reliability is calculated for different levels of reliability of individual trucks and a system in which they are in a parallel connection.

Table 6 *Reliability of the parallel transport system with n trucks (n = 1 – 4)*

Reliability of one truck	0.6		0.65		0.7		0.75		0.8		0.85		0.9	
Truck No.	System reliability	Q(t/year)	System reliability	Q(t/year)	System reliability	Q(t/year)	System reliability	Q(t/year)	System reliability	Q(t/year)	System reliability	Q(t/year)	System reliability	Q(t/year)
1	0.6000	162,180	0.6500	175,695	0.7000	189,210	0.7500	202,725	0.8000	216,240	0.85	229,755	0,9000	243.270
2	0.7200	389,232	0.7638	412,883	0.8050	435,183	0.8438	456,131	0.8800	475,728	0.91375	493,973	0,9450	510.867
3	0.7920	642,233	0.8282	**671,594**	0.8610	**698,185**	0.8906	**722,208**	0.9173	**743,866**	0.941375	**763,361**	0,9630	**780.897**
4	0.8376	**905,613**	0.8674	937,838	0.8937	966,295	0.9170	991,452	0.9376	1,013,733	0.955905	1,033,524	0,9722	1.051.170

The Variant 2 is a variant that was considered using the continuous transport with belt conveyors. The reduced flexibility and strict structure of continuous systems significantly reduces the set of potential solutions, and thus the space for eventual improvements and optimization of the transport system [5]. The continuous transport system will consist of a receiving conveyor to which material from the silo feeder is added, four belt conveyors, three main (stationary) and a disposal conveyor that will change its length and position depending on a disposal front. In the case of the variant solution of continuous transport, it is necessary to place bars with a pneumatic feeder or auger on the silos, as in the case of truck transport. Belt conveyors are placed after the bars, which will have a loading funnel placed on them to accept the material. The material from the hopper will fall onto the belt conveyor. From each silo one conveyor will be placed with a feeder to transport the material from the silo to the receiving belt.

The belt conveyors will be of the closed type (covered) in order to increase the time utilization and reduce the negative impact on the environment. Figure 3 shows the layout of the belt conveyor.

A self-propelled conveyor will be attached to a disposal conveyor having the function of continuous disposal of material - disposer, characteristic of the BRs-1200.

Figure 3 *View of a belt conveyor*

Selection of a self-propelled conveyor that will work as a disposer was made on the basis of the necessary structural parameters that allow the material to be deposited with an appropriate radius, i.e., at a distance enabling a safe position of the self-propelled conveyor and less movement of the conveyor belts within the system.

Figure 4 *Self-propelled conveyor BRs-1200*

Figure 5 shows a scheme of a belt conveyor for transport of gypsum, ash and slag to the place of deposition.

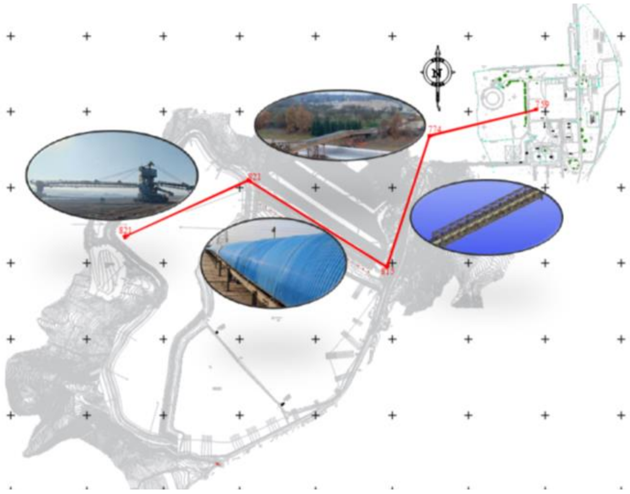

Figure 5 *Scheme of a belt conveyor for transport of ash, slag and gypsum*

The characteristics of the belt conveyor line are shown in Table 7.

Table 7 *Belt conveyor line characteristics*

Belt conveyor designation	Power station elevation (m)	Return station elevation (m)	Height difference	Material lifting height at the loading point (m)	Belt conveyor length (m)	Belt conveyor slope	Belt speed (m/s)	Belt width (mm)
T1	774	759	15	1.4	350	4.3%	2.09	800
T2	815	774	41	1.4	455	9.0%	2.09	800
T3	821	815	6	1.4	540	1.1%	2.09	800
T4	821	821	0	1.8	445	0.0%	2.09	800

Calculation of the conveyor drive group

Calculation of the belt conveyor was performed according to the appropriate SRP standard, and the calculation results are shown in Table 8.

Table 8 *Summary of the belt conveyor calculations*

Belt conveyor designation	Belt conveyor length (m)	Required motor power (kW)	Installed motor power (kW)	Total no. of motors (n x 75 kW)
T1	350	38.02	44.73	1
T2	455	42.99	50.57	1
T3	540	55.86	65.72	1
T4	445	46.30	54.47	1

3 ANALYSIS OF THE OBTAINED RESULTS

Evaluation of the variant solutions was given on the basis of an economic analysis that included the capital and operating costs.

Cost analysis of the Variant 1

Table 9 shows the investments required for the purchase of equipment, facilities and works in the Variant 1.

Table 9 *Investments for the transport system in Variant 1*

Variant 1			
	Pcs.	**€**	**€**
Trucks	3	140,000	420,000
Bridge over the river	1	100,000	100,000
Construction of the road route	1	300,000	300,000
Construction of the bearing layer of road	1	80,000	80,000
Equipment for silos	3	15,000	45,000
Preparatory and auxiliary works	1	17,000	17,000
Unforeseen expenses	1	85,800	85,800
Lighting system along the road route			223,500
TOTAL		**737,800**	**1,271,300**

Table 10 shows the standardized costs of materials and energy for the truck transport.

Table 10 *Consumption standards for truck transport*

Standard	Unit (€/m^3)	Costs (€/m^3)
Fuel standard	0.641	0.1428
Oil and lubricant standard	0.051	0.0114
Standard of spare parts	0.005	0.0011
Tire standard	0.000	0.00003
TOTAL STANDARD COSTS OF TRANSPORT		**0.698**

Table 11 shows the total standard costs of the transport system.

Table 11 *Total standard costs of the truck transport system*

Operation	Standard (€/m^3)
Truck transport standard	0.698
Bulldozer operation standard	0.231
Grader operation standard	0.064
Tank operation standard	0.04
TOTAL	1.033

Table 12 shows the annual labor costs.

Table 12 *Labor costs at the annual basis*

Job position	Operator No.	Educational background	GROSS SALARY €
Manager	1	Secondary vocational education	1,286
Supervisor	4	Secondary vocational education	5,714
Truck driver	12	Highly qualified	15,429
Bulldozer operator	4	Highly qualified	5,143
Grader operator and tanker driver	8	Highly qualified	9,143
Total	29		36,714
Total for a year			**440,571**
Standard labor cost			**0.606**

The analysis of the Variant 1 has established that at the annual basis:

Standard costs €	751,009
Labor costs €	440,571

and the planned investments are realized during the first year and amount to € 1,271,300.

Figure 6 shows the sensitivity analysis of investments, standard costs and labor costs. Based on the analysis results, shown in the graphic and table, it can be concluded that the standard costs have the highest sensitivity in the Variant 1.

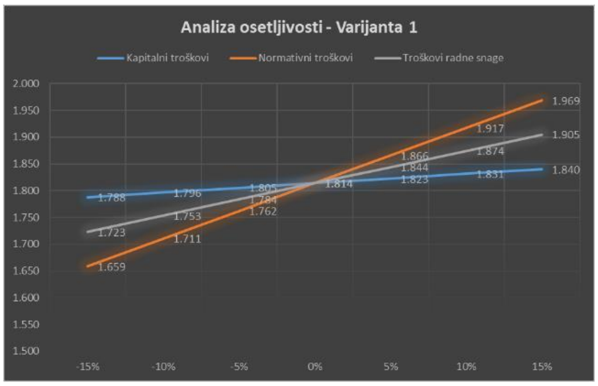

Figure 6 *Sensitivity analysis graph for the Variant 1*

Cost analysis of the Variant 2

Table 13 shows the investments required for the purchase of equipment, facilities and works in the Variant 2.

Table 13 *Investments for the transport system in the Variant 2*

	Type of cost	km		Pcs.	Price per unit	Total
	Equipment for silos			3	15,000	45,000
	Receiving belt with loading hoppers			1	60,000	60,000
T1	Belt conveyor	0.35		1	595,000	595,000
	Power station			1	270,000	270,000
T2	Belt conveyor	0.455		1	773,500	773,500
	Power station			1	270,000	270,000
T3	Belt conveyor	0.54		1	918,000	918,000
	Power station			1	270,000	270,000
T4	Belt conveyor	0.445		1	756,500	756,500
	Power station			1	270,000	270,000
	Loading trolley			1	70,000	70,000
	Dumper (Self-propelled dumper)			1	1,150,000	1,150,000
	Construction of a route for a conveyor belt			1	80,000	80,000
	Control and automation			1	812,700	812,700
	Bridge over asphalt road			1	50,000	50,000
	Unforeseen expenses					512,704
	Lighting system along the conveyor route					179,000
TOTAL						**7,082,404**

Table 14 shows the standardized costs of materials and energy for transport by belt conveyors.

Table 14 *Standards transport consumption by belt conveyors in the Variant 2*

Type	Standard (unit/m^3)	Quantity (unit)	Unit price (€/unit)	Unit costs (€/m^3)
Electrical energy(kWh/m^3)	2.9307		0.15	0.4396
Oil (kg/ m^3)	0.0200		5.5	0.1100
Lubricants (kg/m^3)	0.0200		5.5	0.1100
Stacks of rollers (pcs./m^3)	10%	143.2	850	0.0166
Stacks of lower rollers (pcs./m^3)	10%	71.6	550	0.0054
Stacks of damping rollers (pcs./m^3)	25%	12	950	0.0016
Drums (pcs./m^3)	10%	2	8500	0.0023
Rubber belt B=800 mm (m/m^3)	10%	179	1250	0.0304
Wipers (pcs.)	100%	2	300	0.0001
Bumper plates (pcs./m^3)	100%	2	350	0.0001
Sealing rubber (pcs.)	100%	2148	15	0.0044
TOTAL				**0.7204**

The total standard costs of material disposal in the Variant 2 are given in Table 15.

Table 15 *Total standard costs in the Variant 2*

Operation	Standard (€/m^3)
Standards of transport consumption by belt conveyors	0.7204
Standards for bulldozer operation	0.231
Standards of tank operation	0.04
TOTAL	**0.9914**

For the total masse of 727,018 m^3 (644,000 t) of ash, slag and gypsum, the total costs of standardized material during transport by belt conveyors amount to € 720,765.

Table 16 shows the annual labor costs.

Table 16 *Labor costs in the Variant 2*

Job position	Operator No.	Educational background	GROSS SALARY €
Manager	1	SSS	1,286
Supervisor	4	SSS	5,714
Belt conveyor operator	16	VKV	20,571
Dump truck operator	4	VKV	5,143
Bulldozer operator	4	VKV	5,143
Tank driver	1	VKV	1,143
Total	30		39,000
Total for a year			**468,000**
Standard labor cost			**0.643**

The analysis of the Variant 2 has established that at the annual basis:

Standard costs €	720,765
Labor costs €	468,000

and the planned investments are realized during the first year and amount to € 7,082,404.

Figure 7 shows the sensitivity analysis of investments, standard costs and labor costs. Based on the analysis results, shown in the graphic and table, it can be concluded that the standard costs have the highest sensitivity in the Variant 2.

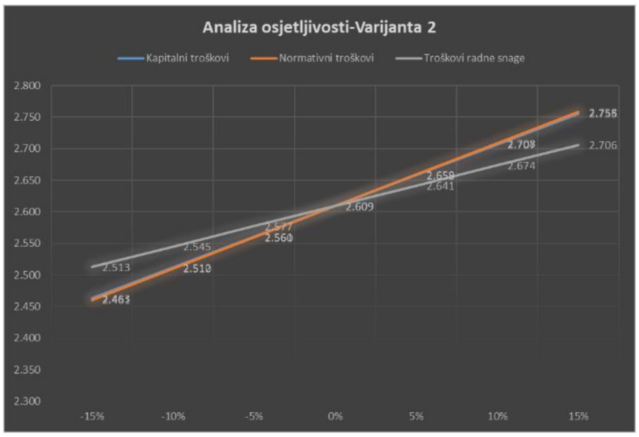

Figure 7 *Sensitivity analysis graph for the Variant 1*

4 DISCUSSION

Total investment and specific investment and operating costs are given in Table 17.

Table 17 *Total investment and specific investment and operating costs*

Comparison of variants					
	Investments (€)	Specific KAPEX (€/m^3)	Specific OPEX (€/m^3)	Labor force	Total (OP+KAP) (€/m^3)
Variant 1	1,271,300	0.1749	1.0330	0.6060	1.8139
Variant 2	7,082,404	0.9496	0.9914	0.6437	2.5847

Based on the values, shown in Table 17, it is concluded that the total specific costs expressed per m^3 in the Variant 2 are 42% higher compared to the Variant 1, which would represent the basic economic parameter for selection a more favorable variant.

In addition to this parameter, the formation of a new continuous transport system would also imply formation of the new specific capacities in a part of equipment maintenance, which represents an additional investment cost as well as the need to hire the additional personnel. From this aspect, the Variant 1 is more favorable.

5 CONCLUSION

The analysis carried out according to the basic techno-economic parameters gives preference to the Variant 1, but in addition to the economic assessment, the following two aspects of the transport system are also important, namely the environmental and system reliability in terms of the system readiness to respond to the request of the thermal power plant in real time with the appropriate capacity. From the ecological aspect, both variants are acceptable, that is, both modes of transport can be arranged with minimization the impact on the environment. When it comes to reliability, the reliability of these systems was specially analyzed, where the basic requirement was that it must be in a continuous operation in parallel with the Thermal Power Plant, that is, ensure all times the removal of slag, gypsum and ash with the required capacity considering the small volumes of silos.

Continuous operation equipment is characterized by a high degree of reliability, which ranges from 0.92 to 0.95 for the belt conveyors and from 0.9 to 0.95 for the conveyors (self-propelled conveyors) with a belt.

A continuous system on transport is a system consisting of 6 elements connected in series. In the case of high reliability of each system elements (0.95 belt conveyors, 0.92 depositor), the reliability of continuous system is 0.712. Figure 9 shows the schematic presentation of the serial continuous transport system.

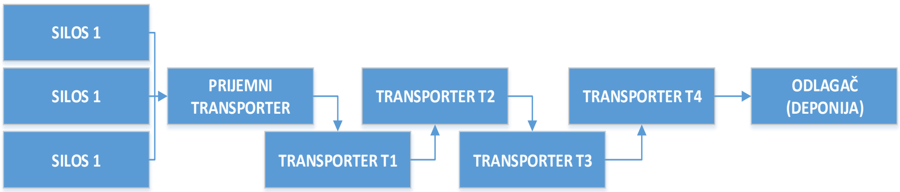

Figure 8 *Layout of a serial continuous transport system*

Contrary to this system, a discontinuous truck transport system enables the system to operate even in case of partial failure, i.e., malfunction, of one or two trucks (Figure 9).

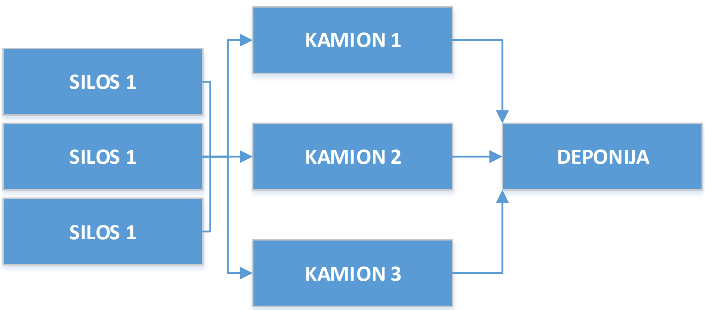

Figure 9 *Layout of a parallel discontinuous transport system*

Since this parallel structure of the truck transport system enables more reliable functioning of the system (reliability for three trucks and an individual probability of operation of 79% is 96%), it represents a significant advantage in selection a transport system in conditions of very limited material bin capacities.

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SELECTION THE OPTIMAL TRUCK MODEL FOR TRANSPORT OF BY-PRODUCTS FROM THE TPP PLJEVLJA TO THE MALJEVAC LANDFILL[**]

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Abstract

This work presents an analysis of selection the optimal truck model for transport of ash, gypsum and slag from the Thermal Power Plant Pljevlja to the Maljevac landfill. The capacity of trucks for transport of by-products was calculated applying the Talpak program package for different types of trucks, taking into account different engagement times. The analysis was done for three types of trucks: Kamaz 53605-A5 (4x2), Mercedes-Benz Actros 4141 and Volvo FMX 520 10X4, for operation in one, two and three shifts. The results of analysis were used to select the optimal type of truck.

Keywords: *by-product, transport, optimization*

1 INTRODUCTION

Ash, slag and gypsum are the by-products occurring in the process of electricity production in the thermal power plants during the coal combustion and desulfurization of combustion gases. In practice, there are several different ways of transport and disposal of this material. Through this work, three variants of the truck transport of by-products from the silos at the Pljevlja Thermal Power Plant to the Maljevac – Cassette 3 ash landfill were compared. The analyzes were considered for the transport of by-products after the environmental reconstruction of the Thermal Power Plant Pljevlja.

This work presents an analysis of transport the by-products from the Thermal Power Plant Pljevlja to the Maljevac landfill, that is the Cassette 3. The scheme of the technological process included in this analysis is shown in Figure 1.

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Figure 1 *Technological process of loading, transport and disposal of by-products*

Truck transport is increasingly used. Manufacturers produce more and more types of trucks suitable for working in different conditions and overcoming different exploitation capacities [1].

All kinds of materials can be transported by the means of truck transport, regardless of their physical and mechanical properties. Great possible mobility, flexibility, outstanding maneuverability during operation and great independence from energy sources are characteristics that come to full expression when using vehicles of auto transport and promise the best economy [2].

One of the most comprehensive reviews of influencing factors on the mechanization selection was given by the authors [3,4]. According to the mentioned authors, the selection of equipment is influenced by the following factors:

- Organizational factors,
- Required equipment flexibility
- Technical characteristics of the equipment,
- Planned production targets,
- Experience in working with a certain type of equipment,
- Lifetime of the equipment,
- Capital and operating costs,
- Reputation of the manufacturer and history of reliability the specific type of equipment,
- The possibility and term of procurement, as well as the manufacturer's guarantee,
- Type of drive (drive fuel),
- Required level of maintenance,
- The need to hire the additional auxiliary equipment,
- Degree of automation,
- Level of safety and comfort when handling the equipment,
- Plan of transport routes and structural parameters, section lengths, slopes, curves,
- The quality of the road surface,
- Speed, load capacity and cycle time of truck movement,
- Tire wear and rolling resistance,
- Construction of a waste dump/landfill,
- Waiting time for loading/unloading.

For most of the authors, see [5, 6, 7, 8, 9], the term optimal equipment actually means a technological system, which within the limitations of the working environment fulfills the production targets with ensuring the minimum costs.

Dimensioning of the truck transport system, i.e. its capacity, should ensure the removal of by-products of the Thermal Power Plant Pljevlja throughout the year, for the estimated operating time of the Pljevlja Thermal Power Plant after reconstruction of about 7,500 h/year. The characteristics of the silo after reconstruction are shown in Table 1.

Table 1 *Characteristics of the silo after reconstruction the Pljevlja Thermal Power Plant*

	Slag	Ash	Gypsum
Silo capacity, m^3	400	3200	600
Discharge rate, m^3/h	50	200	83
Bulk density, t/m^3	0.95	0.8	1.2

For the purposes of an analysis, the annual amount of by-products that need to be transported to the Maljevac landfill was adopted:

- Ash 420,000.00 t
- Slag 70,000.00 t
- Gypsum 154,000.00 t

2 ANALYSIS OF A RATIONAL TRUCK MODEL FOR TRANSPORT OF BY-PRODUCTS

Transport calculation was done using a simulation model-software Talpac 10.2. The software Talpac simulates the technological phases of loading and transport on the basis of the operational and technological parameters of these phases and results in the operational capacities of loading and transport machinery. The software package Talpac represents a simulation model of the loading and transport process at the open pits. The software enables optimization of the transport fleet, calculation the technical and economic parameters of equipment operation such as cycle length, capacity, etc. [10, 11, 12]

The analysis of transport system was performed for the following initial conditions:

- Operating hours of the Thermal Power Plant, 7500 h/year
- Shift duration, 8 hours
- Annual ash capacity, 420,000 t, 525,000 m^3 (γ=0.8 t/m^3)
- Annual slag capacity, 70,000 t, 73,684 m^3 (γ=0.95 t/m^3)
- Annual gypsum capacity, 154,000 t, 128,333 m^3 (γ=1.2 t/m^3)
- Maximum slope of the route, < 8%
- Minimum bend radius, 15 m
- Trucks of different load capacities

Variant 1 - Truck Kamaz 53605-A5 (4x2)

Table 2 shows the view and technical characteristics of the Kamaz 53605-A5 (4x2) truck.

Table 2 *View and technical characteristics of the Kamaz 53605-A5 (4x2) truck*

Engine power, kW (KS)	221 (300)
Load capacity, kg	11300
Box capacity, m^3	5.6

The results of capacity calculation for the Kamaz 53605-A5 truck are shown in Table 3.

Table 3 *Results of capacity calculation for the Kamaz truck (Report from Talpac)*

Production Summary - Full Simulation		
Truck		*[PRJ] Kamaz Kamaz 53605-A5 (4x2)*
Availability	%	70.00
Payload in Template	ton	6.72
Operating hours per Year	Oph/year	5,302.50
Average Payload	ton	6.89
Production per Operating Hour	ton	30.28
Production per Loader Operating Shift	ton	159
Production per Year	ton	160,556
Queue Time at Loader	min/ Cycle	0.05
Spot Time at loader	min/ Cycle	0.40
Average Loading Time	min/ Cycle	0.13
Travel Time	min/ Cycle	11.94
Spot Time at Dump	min/ Cycle	0.30
Average Dump Time	min/ Cycle	0.20
Average Cycle Time	min/ Cycle	13.03
Fleet Size		5
Average No. of Bucket Passes		2.00
Haulage System		
Production per Year	ton/Year	802,778
Excavation Target	t	644,000.00
Loader hrs to move Target	OphYear	6,077
Total Truck hrs to move Target	Oph/Year	21,269

Variant 2 - Truck Mercedes Actros 4141

Table 4 shows the view and technical characteristics of the Mercedes Actros 4141 truck.

Table 4 *View and technical characteristics of the Mercedes Actros 4141 truck*

Engine power, kW (KS)	**300 kW-410 KS**
Load capacity, kg	25000
Box volume, m^3	15

The results of capacity calculation for the Mercedes Actros 4141truck are shown in Table 5.

Table 5 *Results of calculation the transport capacity for the Mercedes Actros truck (Report from Talpac)*

Production Summary - Full Simulation		
Truck		*[PRJ] Mercedes Actros 25 t*
Availability	%	70.00
Payload in Template	ton	12.00
Operating hours per Year	Oph/year	5,302.50
Average Payload	ton	12.29
Production per Operating Hour	ton	50.89
Production per Loader Operating Shift	ton	267
Production per Year	ton	269,831
Queue Time at Loader	min/ Cycle	0.04
Spot Time at loader	min/ Cycle	0.33
Average Loading Time	min/ Cycle	0.40
Travel Time	min/ Cycle	12.36
Spot Time at Dump	min/ Cycle	0.33
Average Dump Time	min/ Cycle	0.33
Average Cycle Time	min/ Cycle	13.80
Fleet Size		3
Average No. of Bucket Passes		4.00
Haulage System		
Production per Year	ton/Year	809,494
Excavation Target	t	644,000.00
Loader hrs to move Target	Oph/Year	189.51
Total Truck hrs to move Target	Oph/Year	8,253

Variant 3 - Truck Volvo FMX 520

Table 6 shows the view and technical characteristics of the Volvo FMX 520 truck.

Table 6 View and technical characteristics of the Volvo FMX 520 truck

**(image downloaded from the site BAS World, Volvo FMX 520 10X4 Tipper Truck New Tipper Truck - BAS World)*

Engine power, kW (KS)	**370 (520)**
Load capacity, kg	49500
Box capacity, m^3	30

The results of capacity calculation for the Volvo FMX 520 10X4 are shown in Table 7.

Table 7 *Results of calculation the transport capacity for the Volvo FMX 520 truck (Report from Talpac)*

Production Summary - Full Simulation		
Truck		*[PRJ] VOLVO FMX 520 50 t*
Availability	%	70.00
Payload in Template	ton	36.00
Operating hours per Year	Oph/year	5,302.50
Average Payload	ton	36.25
Production per Operating Hour	ton	125.48
Production per Loader Operating Shift	ton	659
Production per Year	ton	665,349
Queue Time at Loader	min/ Cycle	0.05
Spot Time at loader	min/ Cycle	0.33
Average Loading Time	min/ Cycle	1.07
Travel Time	min/ Cycle	14.40
Spot Time at Dump	min/ Cycle	0.33
Average Dump Time	min/ Cycle	0.33
Average Cycle Time	min/ Cycle	16.51
Fleet Size		2
Average No. of Bucket Passes		9.01
Haulage System		
Production per Year	ton/Year	1,330,698
Excavation Target	t	644,000.00
Loader hrs to move Target	Oph/Year	3,666
Total Truck hrs to move Target	Oph/Year	5,132

The analysis of transport the by-products of combustion in the Thermal Power Plant Pljevlja included three variants in which different types of trucks were considered where the main parameter being their carrying capacity. The basic parameters of the variant analysis are shown in the following table (Table 8).

Table 8 *Basic parameters of the analyzed equipment*

	Truck type	**Load capacity, t**	**Box volume, m^3**	**Required number of trucks**	**Possible capacity, t/year**	**Required working time for realization the planned capacity, h/year**	**System utilization, (%)**
1.	Volvo FMX 520	50	30	2	1,330,698	2,566	0.48
2.	Mercedes Actros 4141	25	15	3	809,494	2,751	0.79
3.	Kamaz 53605-A5	11.3	5.6	5	802,778	4,253	0.80

Comparing the shown variants, the advantage is given to the variant 2, that is, the variant for the Mercedes Actros 4141 truck with a capacity of 25 t. The advantage of this variant is reflected in the fact that the Investor has already the trucks of this type, but also has an organized maintenance system, experience in the exploitation and maintenance of this type of truck. From the aspect of analyzed parameters, the Investor has the best characteristics in terms of system utilization and required number of truck engagements, which is and directly related to the number of engaged drivers.

3 ANALYSIS OF TRANSPORT FROM THE ASPECT OF TIME ENGAGEMENT

The analysis of transport the by-products of combustion in the Thermal Power Plant Pljevlja also included the three sub-variants in which different time engagement of trucks for transport, i.e., for work in one, two and three shifts, were considered. The analysis was performed for the selected type of Mercedes Actros 4141 truck. The results of analysis are shown in the following table (Table 9).

Table 9 *Basic parameters of the analyzed transport system for different number of shifts*

Subvariant	Shift No.	Required number of trucks	Possible capacity, t/year	Required working time for realization the planned capacity, h/year	System utilization, (%)
1.	3	3	809,494	2,751	0.79
2.	2	4	915,353	2,386	0.70
3.	1	7	797,247	1,369	0.80

The following tables (Tables 10, 11 and 12) show the results of the transport capacity calculation for the Mercedes Actros truck for different shifts (Reports from the Talpac software package).

Table 10 *Results of calculation the transport capacity of the Mercedes Actros truck for 1 shift*

Production Summary - Full Simulation		
Haulage System: Haulage System-1		*Haul Cycle: [PRJ] Route 1*
Material: [PRJ] P+S+G		*Roster: [PRJ] PV 1 SHIFT*
Truck		*[PRJ] Mercedes Actros 25 t*
Availability	%	70.00
Payload in Template	ton	18.00
Operating hours per Year	OpHr/year	1,695.75
Average Payload	ton	18.40
Production per Operating Hour	ton	67.16
Production per Loader Operating Shift	ton	447
Production per Year	ton	113,892
Queue Time at Loader	min/ Cycle	0.18
Spot Time at loader	min/ Cycle	0.33
Average Loading Time	min/ Cycle	0.53
Travel Time	min/ Cycle	12.85
Spot Time at Dump	min/ Cycle	0.33
Average Dump Time	min/ Cycle	0.33
Average Cycle Time	min/ Cycle	14.56
Fleet Size		7
Average No. of Bucket Passes		4.97
Haulage System		
Production per Year	ton/Year	797,247
Excavation Target	t	644,000.00
Loader hrs to move Target	Oph/Year	1,957
Total Truck hrs to move Target	Oph/Year	9,589

Table 11 *Results of calculation the transport capacity of the Mercedes Actros truck for 2 shifts*

Production Summary - Full Simulation		
Haulage System: Haulage System-1		*Haul Cycle: [PRJ] Route 1*
Material: [PRJ] P+S+G		*Roster: [PRJ] PV 2 SHIFTS*
Truck		*[PRJ] Mercedes Actros 25 t*
Availability	%	70.00
Payload in Template	ton	18.00
Operating hours per Year	Oph/year	3,391.50
Average Payload	ton	18.40
Production per Operating Hour	ton	67.48
Production per Loader Operating Shift	ton	449
Production per Year	ton	228,846
Queue Time at Loader	min/ Cycle	0.08
Spot Time at loader	min/ Cycle	0.33
Average Loading Time	min/ Cycle	0.53
Travel Time	min/ Cycle	12.85
Spot Time at Dump	min/ Cycle	0.33
Average Dump Time	min/ Cycle	0.33
Average Cycle Time	min/ Cycle	14.45
Fleet Size		4
Average No. of Bucket Passes		4.97
Haulage System		
Production per Year	ton/Year	915,383
Excavation Target	t	644,000.00
Loader hrs to move Target	Oph/Year	3,409
Total Truck hrs to move Target	Oph/Year	9,544

Table 12 *Results of calculation the transport capacity of the Mercedes Actros truck for 3 shifts*

Production Summary - Full Simulation		
Truck		*[PRJ] Mercedes Actros 25 t*
Availability	%	70.00
Payload in Template	ton	12.00
Operating hours per Year	Oph/year	5,302.50
Average Payload	ton	12.29
Production per Operating Hour	ton	50.89
Production per Loader Operating Shift	ton	267
Production per Year	ton	269,831
Queue Time at Loader	min/ Cycle	0.04
Spot Time at loader	min/ Cycle	0.33
Average Loading Time	min/ Cycle	0.40
Travel Time	min/ Cycle	12.36
Spot Time at Dump	min/ Cycle	0.33
Average Dump Time	min/ Cycle	0.33
Average Cycle Time	min/ Cycle	13.80
Fleet Size		3
Average No. of Bucket Passes		4.00
Haulage System		
Production per Year	ton/Year	809,494
Excavation Target	t	644,000.00
Loader hrs to move Target	Oph/Year	189.51
Total Truck hrs to move Target	Oph/Year	8,253

Comparing the shown subvariants, the advantage is given to the subvariant 2, that is, the subvariant for two shifts. The advantage of this sub-variant is reflected in the fact that the smallest number of trucks is engaged enabling the constant removal of combustion by-products dynamically aligned with production capacities. This plays a particularly important role when separate transport and deposition of gypsum and slag and ash is carried out on the other side, that is, when a simultaneous transport of different types of materials to different deposit sites is required.

4 CONCLUSION

Based on the comparison of the presented variants and sub-variants, it is concluded that according to the criteria of the number of engaged trucks, system capacity and time engagement of the trucks, the variant 2 is optimal, which implies the operation of the Mercedes Actros 4141 truck in sub-variant 2, with work in 2 shifts. When choosing the variant, it was taken into account that the Investor already has trucks of this type, an organized maintenance system and experience in the exploitation and maintenance of this type of truck. Another advantage of this variant is from the aspect of analyzed parameters, where the variant 2 has the best characteristics in terms of system utilization and required number of truck engagements. In this variant, there is a good dynamic compatibility between the production of by-products in the thermal power plant and capacity of removal to the ash, slag and gypsum landfills. An important aspect of considering the overall issue of optimizing the transport of ash, slag and gypsum as a potential man-made mineral raw material for production the building materials is that the thermal power plant work products are deposited in different locations, that the capacities for transport different types of materials differ significantly, and that the adopted system with its flexibility fully meets these requirements.

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STUDIES ON AVAILABILITY OF THE MINING EQUIPMENT- AN OVERVIEW[**]

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Abstract

Mining mechanization works in very difficult working conditions, where high productivity, availability, reliability and safety are constantly expected from this mechanization, as the carrier of production. The availability of mining mechanization is a key factor for stable exploitation, and this feature of the equipment is the subject of a long-term and detailed study. In this review, the published scientific, professional articles and doctoral dissertations on the topic of the availability of mining mechanization are presented. These works show the measures that need to be applied in order to improve the availability of mechanization and thus increase the stability of production. These works also analyze the key influencing factors of availability, determine the essential elements of system maintenance and management in order to increase the availability of mining machinery.

Keywords: *availability, mining mechanization, safety of functioning, effectiveness*

1 INTRODUCTION

The effects of operation the mining mechanization depend on the reliability, availability, their functioning, technical-technological performance, handling, maintenance, logistic support, adaptability - harmony of the relationship between the performance of machines and characteristics of the working environment [1].

Availability is a measure of the usable quality of technical systems. Increasing the availability of mining mechanization enables the safe and stable production.

It is of great importance for mining mechanization to determine the availability of mechanization intended for exploitation in order to reduce the high economic and production costs and ensure the stable exploitation.

The basic division of mining machinery is into basic and auxiliary mechanization. It is made on the basis of the function of specific equipment in the production process. Together, both of these groups make up the complex of mining mechanization at the mine. Mechanization in mining works in the specific, variable and difficult working environment conditions. These machines work in extremely variable operating modes, which affects their working life. One of the consequences of adapting to the conditions

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of the working environment and production requirements is that the mining mechanization is very diverse from different aspects, constructively, the way it affects the working environment, flow of materials, type of driving energy, etc. Figure 1 shows the examples of different types of mining mechanization.

Bucket Wheel Excavator ER 1250 17/1.5

Loader WA 700 Komatsu

Hydraulic backhoe excavator PC 2000 Komatsu

Bulldozer Komatsu 155

Figure 1 *Examples of different types of mining mechanization at the mine*

2 AVAILABILITY

The availability represents the probability that the technical system will be able to work correctly at any moment of time, that is, to be included in the work and remain within the allowed deviations of the assigned functions in a given time period and working environment conditions, see [2, 3].

The ISO-IEC standard defines availability as: "The ability of a technical system to be in a state in which it can perform the required function, under given conditions and at a given moment of time, i.e., during a given time interval, assuming that the necessary supply is provided (external resources)" [2, 4].

The availability can also be calculated based on a temporal state picture [5, 6], in which times in good state alternate with times in failure [5, 7]. The temporal picture of the system state is shown in Figure 2.

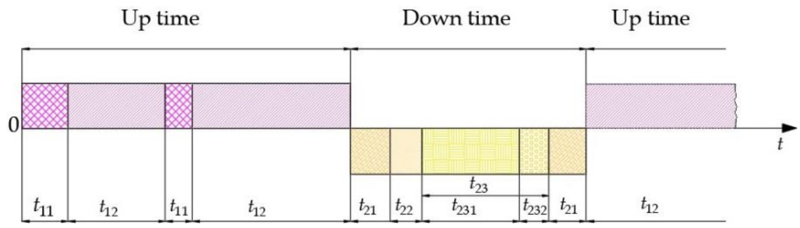

Figure 2 *Time picture of the system state [7,8]*

Considering the broader terms such as the operational reliability and effectiveness, the availability is often considered in the literature as a specific element of these broader terms.

Operational reliability is a common term used to describe the concept of availability and factors that affect it: reliability, maintenance convenience and level of maintenance support [3, 5].

The term technical system effectiveness consists of availability, reliability and functional convenience. Figure 3 shows the definition of effectiveness [2].

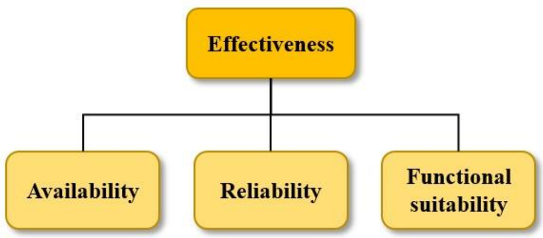

Figure 3 *Effectiveness of the technical system [2, 3]*

3 REVIEW OF WORKS ON THE AVAILABILITY OF MINING MECHANIZATION

This review paper presents the works, dissertations for determining the condition parameters - the availability of mechanization in mining. In recent years, an increasing number of authors have written works on this topic. Examples of analyzed issues are given below.

In the paper entitled "Determining the Availability of Continuous Systems at the Open Pits Applying the Fuzzy Logic" [9] Gomilanović et al. present a model for determining the availability of continuous systems at the open pits using the fuzzy logic, fuzzy inference system. The applied model was formed by the synthesis of partial indicators of availability. The model is based on an expert system for assessment the availability of continuous mining systems. The availability, as a complex state parameter, is decomposed into partial indicators, reliability and maintenance convenience, and the phase compositions used for the integration of partial indicators are the max-min and min-max compositions. The main advantage of this model compared to the conventional models is that it takes into account the impact of partial indicators of availability and does not require a long-term monitoring and records necessary to determine the temporal picture of the system state.

Djurić, R. in his doctoral dissertation "The Concept of Availability in Defining the Effective Maintenance of the Auxiliary Mechanization at the Open Pits" [10] shows the analysis of partial indicators of availability and model of their synthesis at the level of availability. The input data for this model are of a hybrid nature (expert ratings and measured values). Partial indicators of the availability are reliability, convenience of maintenance and functionality of technical systems. The synthesis of the mentioned partial indicators was carried out applying the theory of fuzzy logic to the level of availability. Two approaches were developed on the basis of the expert assessment and measured input data. Dissertation presents a case study of evaluation the availability of auxiliary mechanization - bulldozer.

In the paper "Readiness as a Feature of a Rotary Excavator Maintenance" [11] Jovančić, P. shows the analysis of machine downtime from the aspect of readiness as an initial, basic indicator of the maintenance system. The concept of availability is similar to the concept of readiness. Readiness as opposed to the availability does not include the time when a technical system has been in storage. The analysis was performed on the example of the SchRs 800 rotary excavator, which excavates coal at the open pit Drmno for a period of three years (1997-1999). In this paper, the authors conclude that the proper and timely maintenance should be imperative for all maintainers in order to better time and capacity utilization of the equipment.

In the papers "Fuzzy Logic Model for Evaluation the Safety of Mechanization Functioning at the Open Pits" [12] and "Application of the Fuzzy Logic Modeling in Evaluation the Safety of Mechanization Functioning at the Open Pits" [13] Jagodić, D. e al. defined a mathematical-conceptual model for evaluation the safety of machine functioning of the auxiliary mechanization at the open pits of Elektroprivreda Srbije, which enables the analysis and structuring of partial indicators of reliability, convenience of maintenance and logistic support for maintenance and their synthesis to the level of operational safety. Functional safety is the most complete term to describe the availability of a technical system, as a measure of its usable quality. The obtained results of the evaluation model of safety of functioning and indicators of bulldozer work point to the necessary correction of the mechanization control, maintenance policy, identification and diagnosis of defects, critical failures, etc.

Bugarić, U. et al. in the paper "Analytical Determination of the Availability of a Rotary Excavator as a Part of Coal Mining System- Case Study: Rotary Excavator SchRs 800.15/1.5 of the Drmno open pit" [14] present a model for analytical determination of the availability of a rotary excavator SchRs 800.15 /1.5 as a part of the first BTD system at the open pit Drmno. The application of this method for determining the availability allows efficiently determining the influencing factors of the system operation as a function of time. By modeling the work process as a function of time, using the appropriate statistical methods, the functional dependence of parameters, such as availability, duration of failure, duration of work as a function of time, is defined. The parameter values, obtained by the statistical analysis, show in which stage of life the rotary excavator is. The obtained parameters serve to determine the availability of the rotary excavator.

The paper "The Effect of the Maintenance System Control on the Operational Availability of the Technical Systems" by Avdić, H. et al. [15] presents the effect of the maintenance system control on the operational availability in the Banovići brown coal mine. Increased requirements for the rationalization of resources, as well as the requirements for increasing the utilization of production capacities, place high demands on the availability and reliability of the technical systems. The basis of the paper is a proposal for control the maintenance system in the Banovići brown coal mine in order to efficiently perform the production tasks and reduce the mechanization failures. The proposed way of control the maintenance system with default operational availability in the complex systems such as the maintenance system in the Banovići brown coal mine implies the establishment of a pyramidal maintenance organization. The proposed maintenance organization would involve the unification of the mechanical and electrical sectors.

In the paper "Predicting the Availability of Continuous Mining Systems Using the LSTM Neural Network" [8] Gomilanović, M. and co-authors deal with the development of a model for predicting the availability of continuous mining systems at the open pits using the artificial neural networks. The main idea of this paper is the improvement of the analytical approach, the starting assumption of which is that the distribution of the length of time of the system in failure has an exponential distribution. In this work, data related to the I BTD system of the open pit Drmno Kostolac was used. This work aims to improve already existing models for predicting the availability of continuous systems at the open pits. Based on the RMSE, MAE and R2 values presented in this paper, it is concluded that the model obtained using the neural network has a higher predictive power compared to the analytical approach. Based on the presented model, a suitable simulation is created giving the availability range of considered system. Based on simulation, a more accurate picture of the availability of continuous systems at the open pits is given. The availability of continuous systems obtained by this model well reflects the current state of the I BTD system at the open pit Drmno.

Biserčić, A. in his master thesis entitled "Methodology for Predicting the Availability of Mining Systems at the Open Pits" [16] presents an innovative methodology for predicting the availability of mining systems at the open pits. In this paper, the new methodologies are developed for online and offline reliability and availability prediction, the success of which will be evaluated precisely by how well they approximate the real intensity function (that is, the intensity of failures and repairs in different time windows over time) as well as how well they predict the failure types which will happen.

Jagodić, D. in his doctoral dissertation entitled "Development of a Model of the Usability Quality of Auxiliary Mechanization at the Lignite Open Pits" [2] presents an innovative mathematical-conceptual model for evaluating the usability quality of auxiliary mechanization at thehe coal open pis. This model includes the analysis and synthesis of the following indicators: reliability, convenience of maintenance, functionality, logistical support for maintenance, severity of failure, frequency of occurrence of failure, detectability, effectiveness, availability, safety of functioning, risk, costs. The model is based on a phase logic. The phase model was formed in three levels applying the appropriate models of phase composition (max-min, min-max, Cartesian product-ordered pairs) to the higher levels of synthesis to the overall rating of usable quality. Verification of the model was carried out through a case study, evaluation the use quality of a dozer. The presented concept of evaluation the use quality of auxiliary mechanization provides guidelines for optimization the process of equipment control at the coal open pits in order to better use the auxiliary mechanization as well as the entire system of coal exploitation.

Ivezić, D. et al. in the paper "A Fuzzy Expert Model for Availability Evaluation" [17] present the concept of availability in the auxiliary mining machines. In this paper, the authors created an expert phase model that analyzes and integrates the reliability, convenience of maintenance and functionality of three types of bulldozers (Liebherr, Dressta, Caterpillar) that work at the coal open pits (Drmno, Tamnava West Field and Field D) within the Elektroprivreda Srbije. Based on the evaluation results, a comparison of the mentioned auxiliary equipment - bulldozers was made. Conclusions are given that can be useful for improving the convenience of maintenance, logistics and during the purchase of the new machines.

In the paper "Development of the Availability Concept Using the Fuzzy Theory with the AHP Correction, Case Study: Bulldozers at the Lignite Open Pit" Djenadić S. et al. [7] developed a model for defining the availability of auxiliary mechanization that relies on the phase theory and a multi-criteria method in evaluating the AHP (method of analytical hierarchical processes). The basis of the work is the expert evaluation of the formed partial indicators that enter the availability structure. In this work, the structure of availability is constructed in the form of three partial indicators that have a direct impact on availability, namely: reliability, maintainability and supportability. In the expert evaluation, the evaluations of four experts were taken, who evaluated each of the three analyzed machines from the area of three previously defined partial indicators assigning the grades-linguistic variables (A-best grade, B, C, D, E-worst grade) that have their own class membership functions. Two machine states were analyzed, when the machines were used for 2 years (new machines, in the warranty period) and when the machines were used for seven years (machines before scrapping). The authors used the obtained results on expert opinion to form a phase model with a max-min composition. The case study was done on the example of bulldozers working at the coal open pits. The model itself was verified by compareson with the conventional method of assessment the availability.

The authors Tanasijević, M. et al. in the paper "A Fuzzy-Based Decision Support Model for Effectiveness Evaluation - Case Study of the Investigation of Bulldozers" [18] present the effectiveness as a comprehensive concept and measure of the level of usability of the observed technical system. This paper presents the analysis and structuring of partial indicators. The model was developed to synthesize them to the level of effectiveness. The paper used indicators of a hybrid nature, such as the measured values and expert evaluations. A phased reasoning model for their processing and integration into effectiveness is presented. This concept gives the possibility to evaluate the technical system in terms of making decisions about the remaining possibilities and optimizing the costs of the life cycle. The model is applied on the example of auxiliary machines, bulldozers. The case study covers two approaches. The first approach is based on the expert assessments, and the second on the measurement and statistical data processing.

In the paper "Analysis of the Availability and Utilization of Dragline for Enhancement the Productivity at the Open Pits - Case Study", Mohammadi, M. [19] presents a case study for dragline excavators at one of the largest coal open pit in India. The study shows trends in the availability of these excavators over 11 years. Also, this study highlights the reasons for changing the availability and ineffective operation of draglines.

4 DISCUSSION AND CONCLUSION

This review paper presents the published scientific, professional works and doctoral dissertations on the topic of the availability of mining mechanization. Also, in these papers, the key influential factors of availability are analyzed, the essential elements of system maintenance and management are determined in order to improve the availability of mining machinery. Based on this review, it can be concluded that the availability of mining mechanization is extremely important for the exploitation process itself. In all the presented papers, the authors emphasized the importance of the availability of mining mechanization as a complex concept of the technical system state. Its determination is a complex task and is considered from two aspects. The first is that the availability itself is a factor that affects the security of functioning and effectiveness and represents their essential element. On the other hand, the availability itself is a function of a greater number of sub-indicators of the technical system state. These sub-indicators are obtained by a detailed analysis of the individual production systems or entire complex of mechanization in mines, and the value is determined by the synthesis of these indicators. The result, that is, the availability value, was determined in several different ways, using an analytical model for determining the availability, using a phase model and a model based on an artificial neural network. All of these ways of determining the availability require a long-term observation of the time picture of the system state or the use of expert evaluation.

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FACTORS AND INDICATORS OF THE GEOLOGICAL AND ECONOMIC ASSESSMENT OF THE COAL RESOURCES IN THE REPUBLIC OF SRPSKA AS THE STARTING BASIS FOR A DEPOSIT EVALUATION AS A FUNCTION OF THE NATIONAL AND COMMERCIAL PROFITABILITY

Abstract

The geological-economic evaluation of coal deposits in the Republic of Srpska includes two ways of evaluation the deposits: evaluation of coal deposits without taking the time factor into account (analysis the parameters of the static evaluation methods: economy and profit rate) and evaluation the three largest coal deposits (Gacko, Ugljevik and Stanari) with taking the time factor into account, evaluation the economic profitability (commercial and national). The following dynamic methods are used to assess the commercial profitability: Net present value, internal rate of return and cash flow discounting method for determining the return period of investments. The analysis of the economic effects of the investment project (assessment of deposit) is performed using a "cost-benefit" analysis. This analysis was performed using the following methods: NPV, IRR and DCFROI. As a part of the economic analysis, the analysis of the conditions of uncertainty is processed, i.e., sensitivity analysis of coal deposits, namely: static sensitivity analysis (determining the breaking point of profitability and critical price of a unit of mineral raw material) and dynamic sensitivity analysis, directed at the results of changes in parameters (income, costs, investments and legal obligations) to the changes in the calculated internal rate of return [2]. For the assessment of national profitability, the basic criterion of national profitability, net added value - NVA, is applied. In addition to the direct effects of coal production in the Republic of Srpska, on the national economy (financial income and expenses), the additional criteria were also analyzed: effects on employment, foreign currency inflow and international competitiveness, i.e., applying a "cost-benefit" analysis, an analysis of indirect effects is also performed (effects) of coal production on the national economy of the Republic of Srpska.

Keywords: *geological-economic assessment, rating factors, rating indicators, national profitability, commercial profitability*

1 INTRODUCTION

Geological-economic assessment of the coal resources includes:

a) complex analysis of all factors;
b) general expression of rating factors through one or more synthetic value indicators, with or without taking the time factor into account, and
c) comparing the synthetic value indicators with individual value and natural indicators in order to establish the certain relationships, characteristic for the assessed deposit.

The geological-economic evaluation relies on a number of factors and indicators, which

* *Mine and TPP Gacko*

together represent a whole. Evaluation factors and indicators are used in all phases of the geological-economic evaluation, where in the preliminary exploration the phase natural indicators, derived from the appropriate factors, dominate, while in the detailed exploration phase of the deposit preparation and exploitation, the value indicators and factors, related to them, are of increasing importance.

2 FACTORS OF THE GEOLOGICAL AND ECONOMIC ASSESSMENT

The methodology of the geological-economic assessment of coal resources in the Republic of Srpska includes the analysis of the following assessment factors: mineragenetic, geological, legal, technical-exploitation, technological, regional, market, geoecological and social-political-economic-strategic.

2.1 Mineragenetic factors

The assessment of potential of a certain area, characterized by the specific mineragenetic characteristics, is carried out in order to single out the most favorable parts of the terrain in terms of possibility of finding the new coal deposits. The evaluation of potential in certain coal deposits in the Republic of Srpska was carried out on the basis of the results of a certain stage and type of geological explorations.

2.2 Geological factors

From the aspect of geological-economic evaluation the coal deposits of the Republic of Srpska, the geological factors, having a natural character, can be expressed through the economic type of the raw material, economic type of the deposit, morphological characteristics and degree of the concentration of reserves.

Coal, as the primary energy raw material in the Republic of Srpska, can be divided, according to the economic type of raw material into:

a) Economic type of lignite coal raw material of Gacko and Stanari, and
b) Economic type of brown coal raw material of Ugljevik and Miljevina.

According to the economic type, the coal deposits in the Republic of Srpska can be divided into three groups:

a) Deposits of the primary economic type: Gacko, Ugljevik and Stanari;
b) Deposits of the secondary economic type, Miljevina, and
c) Deposits of the tertiary economic type: Majevičko, Teslićko, Lješljansko, Banja Luka, Mesićko, Bukovačko, Šipovsko, Omarsko and Kotorvaroško.

The coal deposits in the Republic of Srpska are layered, simple (Stanari) to the complex (Gacko, Ugljevik and Miljevina) structures.

2.3 Legislative and legal factors

Without the certain legal regulations, it is not possible to optimally manage the coal energy resources. The geological-economic assessment of the coal deposits of the Republic of Srpska from the perspective of legislation includes the analysis of [7]:

- Legislative-legal factor and exploration of coal deposits;
- Legislative-legal factor and exploitation of coal deposits, and
- Legislative-legal factor in the production and consumption of thermo-electric energy in the Republic of Srpska.

2.4 Technical-exploitation factors

The technical-exploitation factors of the geological-economic assessment of the coal deposits of the Republic of Srpska include the analysis of:

- Coal mine from the aspect of exploitation conditions, exploitation system, coal production and overburden excavation, losses and dilution during coal exploitation;

- Use of accompanying mineral raw materials/resources through the complex utilization of coal deposits in the Republic of Srpska.

Considering the coal mines in the Republic of Srpska, two groups of deposits are distinguished as the objects of coal exploitation:

- Deposits where coal is already or has been exploited (Gacko, Ugljevik, Stanari, Miljevina, Kotor Varoš, Lješljani and Ramići) and
- Deposits where it is possible to open the new mines (Majevica, Teslić, Mesići, Bukovica).

Starting from the fact that the coal deposits are the non-renewable energy resources with a limited life and that until 2013 the coal mines with surface exploitation (Gacko) were mainly exploited the coal seams with less stratification, and lately, by conducting the exploitation and detailed phase of geological exploration of the coal reserves of larger stratification, the expansion of the mineral-raw material base of the existing mine was done as well as increase of the exploitation life.

2.5 Technological factors

For assessment the quality of the raw material (coal), its calorific value has the greatest importance. The use of raw lignite in a lump form in wide and general consumption and industry is justified only within the economic radius of transport, because its calorific value is low. This means that by selling the presorted raw lignite as a fuel in smaller urban settlements (closer to the mine), the positive economic effects would be achieved. The largest part of the raw lignite, produced in the Republic of Srpska, however, is used in the thermal power plants for the production of electricity. Technological factors of coal resources define:

- Technological type of raw material;
- Technological process of preparation, processing and use of raw materials, and
- Better thermal utilization of coal from the Republic of Srpska.

2.6 Regional factors

Regional factors play an important role in the geological-economic assessment of the coal deposits in the Republic of Srpska. In many cases, they significantly affect the economic indicators of the deposit being evaluated. The most important regional factors are: transport conditions, energy sources, climatic conditions, water supply conditions, and population.

2.7 Market factors

The market of mineral raw materials is specific in many characteristics compared to the market of the other industrial products. Market factors have an extraordinary effect on the geological-economic evaluation of the coal deposits of the Republic of Srpska and the achieved economic effects, as a result of the evaluation of deposits. Coal deposits in the Republic of Srpska are exploited in various, mostly difficult conditions, where the mines, with the current sales prices, operate at the limit of profitability. Although the planned predictions regarding the increase in production have not been realized, it can be stated that the surface exploitation is intensively developing, which in the current conditions provides significantly more competitive coal for the market. This is the result of investments, procurement of modern equipment and selection of more favorable deposits for this type of exploitation, that is, the opening of the new surface mines. The geologically and economically interesting for the future thermal energy planning are the coal mines, that is the lignite deposits, due to the cleanliness, less environmental pollution, low sulfur content, and due to significantly larger geological reserves. The mine locations are determined by the location of the deposits themselves.

2.8 Geological factors

The economic valorization of coal resources (exploration, exploitation, processing, post-exploitation period) results in the negative consequences for the environment. They are manifested in all spheres: earth, water and air [5]. Geoecological factors are especially important for the problem of coal resources of the Republic of Srpska: geoecological type of ore and deposits, changes in the state of the geological environment under the impact of geological exploration, exploitation and preparation of mineral raw materials, the impact of geological exploration, exploitation, preparation of mineral raw materials and accompanying processes on changes in the state of air, water, soil and plant and animal life, measures of recultivation and revitalization of the geological environment, geoecological condition of waste, possibility of using fly ash as the technogenic raw material, monitoring of the geological environment in the immediate vicinity of active coal mines, geoecological conservation of coal deposits, geoethical factors and their impact on evaluation the coal deposits, impact of infrastructural facilities on the geological environment in the immediate vicinity of coal mines, sustainable use and indicators of sustainable use of the coal resources, post-exploitation use of the open pits, pollution prevention measures in various process-work phases and environmental costs and their impact on profitability of the coal production [6].

2.9 Social-political-economic-strategic factors

Energy is an activity that has a significant impact on socio-economic development and environment. Therefore, it is the first area that should be established in the Republic of Srpska on the basis of sustainable development. The basis is the introduction of such economic incentives by the state, which will orient the producers and consumers of electricity towards increasing energy efficiency applying not only technical but also the organizational measures. The state task is to introduce measures to increase the energy efficiency, enable entrepreneurs to keep their energy usage costs from rising. The state should monitor whether this has been achieved, by measuring every year:

- energy profitability (a measure of increase the economic profit in relation to the increase in the energy consumption),
- coefficient of price elasticity (a measure of reduction the energy consumption caused by the increased energy prices and supported by the economic stimulus of the state), and
- annual emission into the atmosphere of key pollutants (monitoring of the approach to obligations from the international agreements on the environmental protection).

The coal reserves proven so far, its quality, the existing capacities of the Gacko and Ugljevik mines and thermal power plants, as well as the capacities of the Stanari and Miljevina mines, are of great importance not only for the Republic of Srpska, but also for the surrounding regions. There are conditions to expand the capacities of both mines and thermal power plants, which would enable a long-term and orderly supply of electricity to the economy and population in a wider area.

3 INDICATORS OF THE GEOLOGICAL - ECONOMIC EVALUATION OF COAL RESOURCES IN THE REPUBLIC OF SRPSKA WITH THE METHODOLOGY OF DEPOSIT EVALUATION AS A FUNCTION OF THE NATIONAL AND COMMERCIAL PROFITABILITY

In the closest connection with the factors of geological-economic assessment, there are various indicators, which can be grouped into the natural, valuable and synthetic. Indicators of geological-economic evaluation are the basis for a comparative analysis and selection the priority deposits for exploration and production activation, but they are also used in investment into the natural resources, long-term planning of the use of mineral resources, and then in determining the optimal investment options in the mineral economy. The process of geological-economic assessment of coal deposits can provide the useful data for the concrete expression of economic, social and socio-political effectiveness of geological explorations, specific analyzes of the current and future costs of such explorations, i.e., for a more complete overview and quantification the complex expressions of a mining rent.

3.1 Natural indicators

Considering the specificity of material being processed, a special attention is paid to the following natural indicators of geological-economic assessment: coal reserves of the Republic of Srpska, average quality of coal reserves, total energy potential of coal resources in the Republic of Srpska, production capacity and exploitation life.

Coal reserves with the average quality of reserves by deposits are shown with the balance on 12/31/ for the year in which the geological-economic assessment is carried out.

The following parameters are used to calculate the energy potential of the coal reserves of the Republic of Srpska, E (10^x Wh): m – mass of reserves (10xg), mean lower calorific value (kJ/kg), conversion coefficient of the calorific value of 1MJ into the energy value of 1kWh.

The energy potential is calculated according to the following formulas:

$$E\ (MJ) = m\ (10^{x} kg) \times DTE\ (MJ/kg)$$

$$1MJ = 1/3.6\ (kWh) = 0.2778\ (kWh)$$

$$E\ (10^{x}\ kWh) = E\ (MJ) \times 0.2778\ (kWh)$$

The exploitation life of the coal deposits (Gacko, Stanari and Ugljevik) is observed as a function of the amount of exploitation reserves and planned production capacity according to the demand for energy fuel of current and planned thermal power plants. For other coal deposits in the Republic of Srpska, the exploitation life depends on the household demand and industrial consumption (mainly heating plants).

In the Republic of Srpska, coal (which is intended as the thermal energy fuel) is currently exploited at three active mines: Gacko, Ugljevik, and Stanari. The capacity of coal exploitation should be designed so as to provide the sufficient quantities of coal for a minimum of 25 years of operation (lifetime of the thermal power plant) plus 15 years after the revitalization of the thermal power plant (most of the placement of coal produced in active mines is planned for the thermal energy facilities) [1]. Since the exploitation life is aligned with the life of the thermal power plant, it is necessary to adjust the depreciation of the equipment at the mines to the duration of the thermal power plant operation. By increasing the amortization period, the amortization rate will decrease, and thus the cost of produced coal. One of the key elements of the financial/cost and investment assessment is the construction of the new thermal power plants, Gacko 2.

3.2 Value indicators

The analysis of value indicators as the important parameters of economic value of the deposit is performed on the basis of the financial effects realized in the coal mines of the Republic of Srpska in the previous year, with a projection for 2020, 2025 and 2030. Also, an analysis of the production and sales price of 1 ton of coal in the previous period of at least three years, with a projection for the period up to 2030, is carried out. Operating costs are planned for the period up to 2030 on the basis of the certain elements (regulations, workforce, maintenance, depreciation) [4].

Economic analysis of the active coal mines in the Republic of Srpska has the task of processing the information on relevant economic facts. The analysis takes into account the business results from the previous period, and based on them, a business plan for the next period is drawn up. The analysis does not correct the business failures in the previous period, but creates the preconditions to avoid them in the future. These economic indicators, along with the economic analysis of the mine for the previous period with projections for 2020, 2025 and 2030, are necessary for the calculation of NPV and IRR [3].

3.3 Synthetic indicators

Synthetic indicators are the most complex indicators of the geological-economic assessment. Their main goal is to determine the economic (value) assessment of the mineral deposits. The economic (value) assessment of the coal energy resources of the Republic of Srpska includes:

- evaluation of the coal deposits without taking the time factor into account, and
- evaluation of the coal deposits taking the time factor into account.

For each investment project (deposit), determining the cash inflow and cash outflow in each individual year is not a simple operation. In the mineral economy, where the effect of exploitation is largely dependent on the natural conditions, this determination is even more complicated and carries a relatively high degree of uncertainty. The impact of inflation should be added to this.

3.3.1 Evaluation of the coal deposits without taking the time factor into account

In practice, the evaluation of a particular deposit or ore body is often expressed through the difference between the value of the useful components in the deposit (or those components that can be used from the deposit), and the costs that must be incurred to obtain them. The value of deposits and reserves of coal energy resources of the Republic of Srpska, without taking the time factor into account, which is done in the current year, is shown with the state of December 31 of the previous year. The following formula is used to calculate the value of coal deposits in the Republic of Srpska:

$$Vu = (Vi - Ti) \times (R - G)$$

where:

- Vu - conditional value of the deposit without taking the time factor into account (KM),
- Vi - value of the useful components that the deposit contains (KM/t)
- Ti - costs required to obtain the useful components (KM/t),
- R - reserves of mineral raw materials in the deposit (t),
- G - actual or planned losses of mineral raw materials (t).

3.3.2 Evaluation of the coal deposits taking the time factor into account

Exploration and exploitation of mineral deposits takes place over a certain period of time. The economic effects of exploitation also take place in a limited period of time and are subject to the effect of various impacts, depending on the length of exploitation life of the deposit, i.e., relationship between exploration works and production capacity. In principle, in a longer period of exploitation, it is realistic to expect a more pronounced effect of various risks, within which the geological, mining and economic risks dominate, and which must be taken into account in a certain way when calculating the economic (value) assessment of deposits and mines. Evaluation of the coal deposits of the Republic of Srpska, taking the time factor into account, includes the calculation of economic profitability, i.e., evaluating the commercial and national profitability of coal production from coal mines in the Republic of Srpska.

4 COMMERCIAL PROFITABILITY

The commercial profitability analysis is the first step in the economic evaluation of the project. It is concentrated on assessment the feasibility of a new project from the aspect of financial results. The direct bene fits and costs of the project are therefore expressed in the monetary terms at prevailing expected market prices. This analysis is applied both to the evaluation of justification and acceptability of a project, as well as to the ranking of projects based on their profitability. Commercial profitability analysis includes: investment profitability analysis and financial analysis. These two types of analysis are complementary and not interchangeable. Both must be carried out as they focus on different aspects of investment evaluations. Analyzing the profitability of investment is the measurement of profitability the resources employed on the project, direct return on invested capital, regardless the forms of financing sources. Therefore, the analysis of profitability the investment is an assessment the potential returns of resources related to the project, regardless the financial transactions that occurred during the project life. On the other hand, the financial analysis takes into account the financial aspects of the project in order to ensure that the available financial resources in the future enable the easy implementation and operation of the project. The final assessment of commercial viability of coal production in the assessed deposits in the Republic of Srpska is carried out analyzing the economic effects that the project (assessed coal deposit) brings.

The following dynamic methods are used to evaluate the commercial profitability: Net Present Value, Internal Rate of Return and cash flow discounting method for determining the return period of investments. The analysis of the economic effects of the investment project (assessed deposit) is performed applying the "cost-benefit" analysis. This analysis was performed using the following methods: NPV, IRR, DCFROI [9]. As a part of the economic analysis, an analysis of the conditions of uncertainty is also carried out, that is, the sensitivity analysis for the coal deposits, namely: static sensitivity analysis (determining the break-even point of profitability and critical price of a unit of mineral raw material) and dynamic sensitivity analysis, which considers the results of changing parameters (income, costs, investments and legal obligations) to the changes of the calculated Internal Rate of Return.

The dynamic methods (assessment the value of deposits taking the time factor into account) take into account the effect of time factor, that is, they start from the fact that money has a time value, and is determined by reduction the future economic effects of production using the appropriate discount rate on the day of assessment. Based on the input parameters, i.e., assessment of inputs (production, costs, investments, etc.) and expected results based on them (income, profit, etc.) for the period up to 2030, the net present value and internal rate of return are calculated for deposits with the active coal mines. Based on the current trends in coal prices in the previous period, the price of coal (KM/t) is planned and projected. Through the sensitivity analysis, we monitor the financial effects in the active coal mines are monitores, so the adopted price is only a starting point for further analyses.

5 NATIONAL PROFITABILITY

The analysis of national profitability is similar in form to the analysis of commercial profitability, since both try to identify the costs and benefits and, by their comparison, evaluate the profitability of the proposed investment [8]. A commercial cost-effectiveness analysis is the first step towards a national cost-effectiveness analysis. A country overall development strategy usually requires the fulfillment of several goals. Namely, it is necessary to evaluate the social validity of the project - from the aspect of effects on the economy as a whole, and through the special aspects of economic life in the context in which the project will be treated.

Consequently, in addition to the basic criterion - additional value as a way to evaluate the main impact of the project on the economy - a number of additional indicators are given to measure the certain implications of the investment project, such as: effects on employment, foreign exchange inflow and international competitiveness. For other implications that cannot be measured in the quantitative terms, a qualitative analysis with the additional consideration of the impact on infrastructure, technical know-how and environment is recommended. The final evaluation of the national profitability of coal production from deposits in the Republic of Srpska is performed by analyzing the results of the basic criterion of national profitability - net additional value and indirect impacts (effects) of coal production on the national economy of the Republic of Srps

6 FINAL CONSIDERATIONS

The strength of energy potential of the Republic of Srpska is based on balance reserves of coal which amount to ~700 Mt, potential reserves of oil which amount to 50 Mt and potential reserves of U_3O_8 of 6,000 t. The average coal quality of the balance reserves of the Gacko, Ugljevik, Miljevina and Stanari coal deposits is 11,300 kJ/kg. Coal from the Gacko and Ugljevik deposits is mostly used (about 95%) as an energy source for the production of electricity, while coal from the Stanari deposit is mostly used for the industry and household needs. Coal production in the active mines enables meeting the needs of households at the level of ~200,000 t/year. About 2,000 workers are employed in the active coal mines Gacko, Ugljevik and Stanari, and ~300 workers in the Miljevina mine. Investments in the Gacko, Ugljevik and Stanari coal mines [10] are mainly focused on the acquisition of mining equipment and machinery, and part of them on preparation the project documentation and preparatory activities for opening the new coal mines.

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Materials Science

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SYNTHESIS AND TESTING OF THE ELECTRO-CATALYTIC MATERIALS FOR THE HYDROGEN-PEROXIDE REDUCTION

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Abstract

In this paper, the synthesis, characterization and electrochemical testing of carbon material (graphene) modified with the Pd nanoparticles in the hydrogen peroxide reduction reaction was performed. Graphene was synthesized from glucose as a precursor, using $FeCl_3$ as a catalyst. The reduction of hydrogen peroxide on the synthesized material was studied by the cyclic voltammetry and square wave voltammetry.

Keywords: *synthesis, characterization, carbon material, graphene, palladium.*

1 INTRODUCTION

Carbon materials, as carriers of metal nanoparticles, offer a wide range of applications in both analytical and industrial electrochemistry. This broad applicability is precisely that stems from the carbon itself that is largely derived from their structural diversity and stability, exchange and chemically complex surfaces, as well as the strong carbon-carbon bonds within the carbon material. Compared to the other materials, they have several advantages in the electrode fabrication, evidenced by their wide range of applicable potentials, low cost, relative inertness, and pronounced electrocatalytic activity.

This study presents the results of synthesis, characterization, and electrochemical investigation of the carbon material (graphene), modified with the composite nanoparticles through the hydrogen peroxide reduction. Graphene was synthesized from glucose as a precursor, using $FeCl_3$ as a catalyst. The reduction of hydrogen peroxide on the synthesized material was studied using the cyclic voltammetry and square wave voltammetry methods.

2 EXPERIMENTAL PART

2.1 Materials and methods

2.1.1 Synthesis of graphene (G)

The procedure for graphene preparation in this study is based on the method proposed by Zhang et al [1]. Six grams of glucose and six grams of $FeCl_3 \cdot 6H_2O$ were measured and dissolved in 10 mL of deionized water. The water mixture was heated at 80°C for 24 hours. Carbonization of the obtained product was performed at 700 K for 3 hours under an inert nitrogen (N_2) atmosphere. The material was then ground and transferred to 100 mL of concentrated

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HCl (36%) with stirring for 6 hours. After that, the suspension was filtered and washed with 300 mL of deionized water and 80 mL of acetone. The resulting material was dried at 70°C for 1 hour [2, 3].

2.1.2 Synthesis of graphene-palladium composites

One hundred milligrams of graphene were measured and dispersed in a 20 mL mixture of ethanol/water (50%, v/v). The graphene mixture was then subjected to the ultrasonication for 5 minutes. Subsequently, 25 mg of cetyltrimethylammonium bromide (CTAB) was added, and the same ultrasonication procedure was repeated for another 5 minutes.

In a separate vial, a mixture of 50 mg $PdCl_2$ and 0.05 mL concentrated HCl was prepared and heated at 60°C for 30 minutes. After cooling, solution was diluted with H_2O to 2 mL and added to the graphene suspension. It was left at room temperature for 1 hour before adding 1 mL of $NaBN_4$ solution. The resulting suspension was transferred to a stainless-steel autoclave and heated at 160°C for 90 minutes. After hydrothermal treatment, the suspension was centrifuged and the liquid phase was decanted. The residue was washed several times with deionized water. Finally, the obtained material was dried in an oven at approximately 60°C and labeled as G-Pd [2, 4].

2.2 Characterization

The characterization of the synthesized graphene-palladium (G-Pd) composite, as well as graphene, was performed using X-ray diffraction (XRD) and scanning electron microscopy (SEM) techniques. The SEM micrographs were obtained using a Zeiss Leo Gemini 1530 instrument with an Everhart-Thornley detector for collecting the secondary electrons. The acceleration voltage was set to 10 kV. The XRD diffraction patterns were obtained using an STADI P instrument with a Mythen1K detector [4].

2.3 Cleaning and preparation of a glassy carbon electrode (GCE)

First, a glassy carbon electrode (GCE) with a geometric surface area of 0.0706 cm2 was polished with 0.05 mm Al_2O_3 powder on a polishing cloth, followed by rinsing with deionized water and ethanol in an ultrasonic bath. After the cleaning process, 5 mg of G-Pd was dispersed in 1 mL of ethanol/water mixture (40%, v/v), and the resulting suspension was homogenized in an ultrasonic bath for 30 minutes. 5 mL of the suspension (equivalent to 0.025 mg of material) was transferred onto the surface of the GCE and dried under a stream of N_2. Once a thin layer of electrocatalyst was formed, 5 mL of 0.05 wt.% Nafion solution in ethanol was applied and dried again under a stream of N_2. The modified GCE was used to investigate the electroreduction of hydrogen peroxide (H_2O_2).

The nominal metal loading in the Pd-based electrocatalyst was 82 μg cm-2. The thickness of Nafion was estimated to be approximately 0.2 μm using the following equation:

$$t = m/(A\rho) \qquad (1)$$

where m is the mass of ejected Nafion (2.500·10−6 g), A is the film area, and ρ is the approximate film density (1.980 g cm^{-3}).

It is worth nothing that oxygen and hydrogen peroxide can penetrate the Nafion membranes of a few microns in thickness and thus can be reduced at the electrode surface. The electroreduction of H_2O_2 was studied using the cyclic voltammetry (CV) and square wave voltammetry (SWV).

The three-electrode system consisted of a G-Pd modified GCE as the working electrode, saturated calomel electrode (SCE) as the reference electrode, and graphite rod as the auxiliary electrode. 5 mM H_2O_2 solution in 0.1 M KOH was used to study the electrocatalytic behavior of G-Pd. Purified nitrogen was bubbled through the electrolyte for thirty minutes before the addition of the appropriate amount of hydrogen peroxide.

The electrolytic cell was shielded from light with aluminum foil. Prior to the electrochemical measurements, the activation of each electrocatalytic film was performed by continuous potential cycling with respect to the SCE in a N_2-saturated 0.1 M KOH solution until a stable and reproducible cyclic voltammogram was recorded.

All measurements were carried out under the same conditions for each electrode to allow for comparison of the obtained results. Electrochemical investigations were conducted using cyclic voltammetry and square wave voltammetry methods [2, 4].

3 RESULTS AND DISCUSSION

3.1 Characterization of the graphene and graphene-palladium composites

The characterization of graphene and graphene-palladium was performed using the XRD, and the corresponding diffractograms are shown in Figure 1.

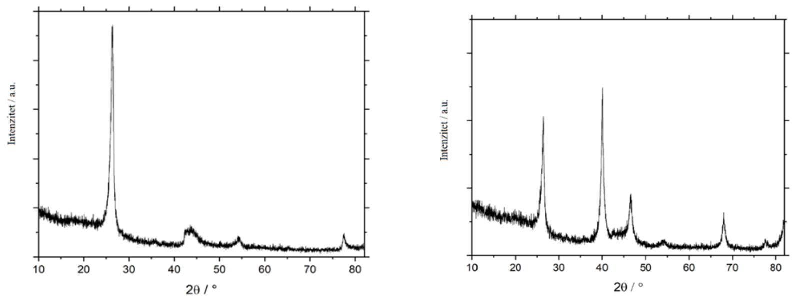

Figure 1 *XRD diagram of graphene (left) and graphene-palladium (right)*

The sharp peak at about 26.3° belongs to the (002) crystallographic plane of the graphene nanoparticles, composed of several layers of graphene sheets (Figure 1, left). The corresponding spacing between the graphene layers is 0.38 nm, and is obtained using the Bragg equation:

$$2d \sin\theta = n\lambda \qquad (2)$$

After the deposition of palladium particles on graphene under the hydrothermal conditions, the (002) peak is observed at the same 2θ value (Figure 4.1.1, right) indicating that the palladium particles were not incorporated between the graphene layers. In Figure 1, on the right, the diffraction peaks at 2θ ~ 40.2˚, 46.8˚, 68.1˚ and 82.3˚ are observed, which correspond to the Pd (111), (200), (220) and (311) crystallographic planes of the centered cubic lattice of palladium.

The morphology of graphene and graphene-palladium was analyzed using the SEM and corresponding photomicrographs are shown in Figure 2.

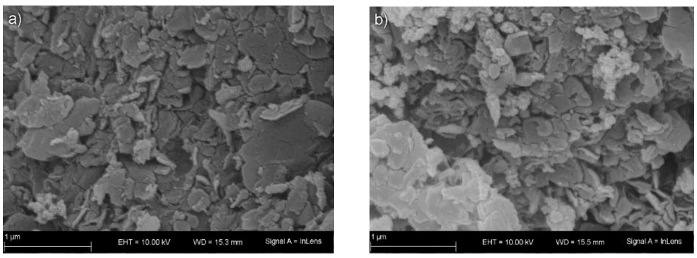

Figure 2 *SEM micrographs of (a) graphene; (b) graphene-palladium [2]*

The SEM microphotographs show that the synthesized graphene (Figure 2.a) has a sheet-like morphology consisting of multiple layers, which provides more active sites for palladium nucleation. After the deposition of palladium nanoparticles, graphene retained the same morphology (Figure 2. b).

3.2 Electrochemical tests

Electrochemical testing of the hydrogen peroxide reduction reaction was carried out using the cyclic voltammetry and square wave voltammetry methods. A 0.1 M potassium hydroxide solution was used as the background electrolyte. The experiments were first conducted in a pure electrolyte without hydrogen peroxide, and then in electrolytes containing hydrogen peroxide at concentrations of 5, 10, 15, 20, and 25 mM. During the electrochemical measurements, nitrogen gas was purged into the electrolyte to remove oxygen. Cyclic voltammograms were recorded at the scan rates of 0.01, 0.02, 0.05, 0.08, and 0.1 V/s.

Figure 3 shows the obtained cyclic voltammograms for the commercial glassy carbon electrode and modified electrode in an alkaline electrolyte without the presence of hydrogen peroxide.

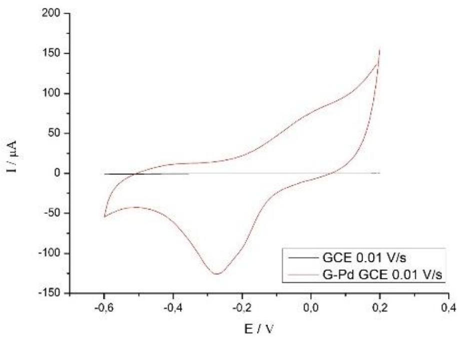

Figure 3 *Cyclic voltammograms for a commercial glassy carbon electrode (GCE) and modified electrode (G- Pd GCE) in 0.1 M KOH solution, at a potential change rate of 0.01 V/s*

Figure 4 shows the obtained cyclic voltammograms for a commercial glassy carbon electrode and modified electrode in an alkaline electrolyte containing 5 mM hydrogen peroxide.

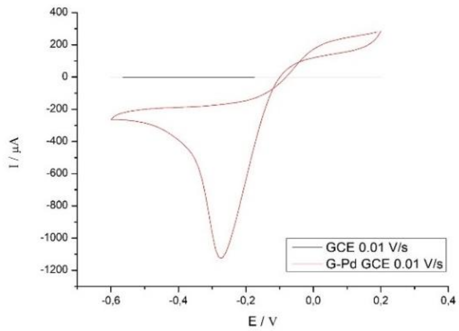

Figure 4 *Cyclic voltammograms for a commercial glassy carbon electrode (GCE) and modified electrode (G- Pd GCE) in 5 mM H2O2 , 0.1 M KOH , at a potential change rate of 0.01 V/*

The data obtained and presented in the previous figures demonstrate that the modified electrode (G-Pd GCE) exhibits a superior electrocatalytic activity compared to the commercial glassy carbon electrode. The shape of cyclic voltammogram, obtained in the electrolyte without hydrogen peroxide (Figure 3), is characterized by a prominent cathodic voltametric peak, which is attributed to the reduction of trace amounts of oxygen adsorbed on the palladium particles.

The modified electrode (G-Pd GCE) shows a strong reduction peak in the presence of 5 mM H_2O_2 at a scan rate of 0.01 V/s (Figure 5). The cathodic peak corresponds to a peak current value of Ip = -1075 μA, while the potential value of Ep = -0.270 V corresponds to the position of the main reduction peak in the voltammogram, obtained in a 0.1 M KOH solution without the presence of hydrogen peroxide.

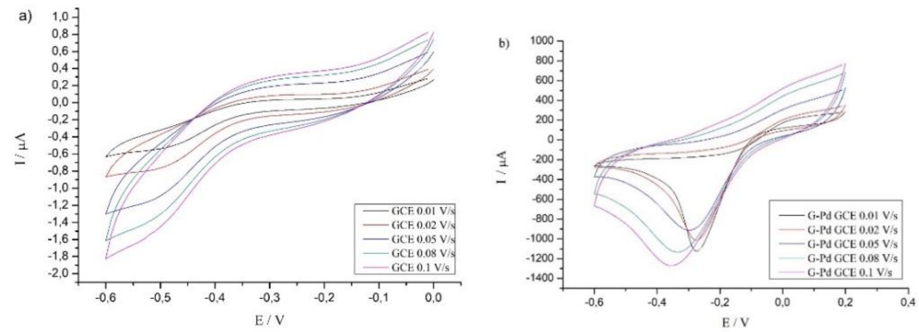

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Figure 5 *Cyclic voltammograms for: a) commercial glassy carbon electrode (GCE), b) modified electrode (G-Pd GCE) in 5 mM H_2O_2, 0.1 M KOH*

From the shown cyclic voltammograms, it is evident that the modified electrode exhibits the clear peaks at all scan rates, resulting in the higher cathodic current values compared to the cathodic current values, obtained from the cyclic voltammograms of the commercial glassy carbon electrode. Table 1 presents the parameters obtained from the cyclic voltammetry method in 5 mM H_2O_2, 0.1 M KOH, at a scan rate of 0.01 V/s.

Table 1 *Electrochemical parameters obtained by a cyclic voltammetry in 5 mM H_2O_2, 0.1 M KOH, at a potential change rate of 0.01 V/s.*

E electrode	E_p (V)	I_p (μA)
GCE	-0.480	-0.1
G-Pd GCE	-0.270	-1075

As it can be seen, the cathodic peak current of the modified electrode (-1075 μA) has a significantly higher value than the cathodic peak current of the commercial electrode. The peak has shifted from -0.480 V for the commercial glassy carbon electrode to -0.270 V for the G-Pd modified electrode. These parameters indicate the pronounced electrocatalytic activity of the G-Pd material in the HPRR process.

Electrochemical investigation of the hydrogen peroxide reduction reaction was performed using the square wave voltammetry under the same conditions as the cyclic voltammetry. The background electrolyte was 0.1 M KOH, which did not contain hydrogen peroxide in one cycle of the experiment, contained 5 mM H_2O_2 in the next cycle, and H_2O_2 concentration was increased in the third cycle of the experiment. During all experiments, nitrogen gas was introduced into the electrolyte to remove oxygen.

The parameters for obtaining the square wave voltammograms were as follows: amplitude value of 0.25 V, frequency of 2 Hz, and potential step of 0.01 V.

Figure 6 shows the obtained square wave voltammograms for the commercial glassy carbon electrode and modified electrode (G-Pd) in an alkaline electrolyte without the presence of hydrogen peroxide.

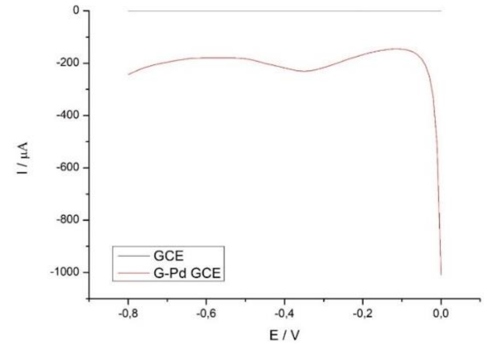

Figure 6 *Square wave voltammograms for the commercial glassy carbon electrode (GCE) and modified electrode (G-Pd GCE) in 0.1 M KOH solution*

As it can be seen in the obtained voltammogram, the modified electrode gives a clearly visible peak compared to the commercial glassy carbon electrode under the conditions without the presence of H_2O_2.

The following Figure 7 compares two voltammograms, obtained by the square wave voltammetry method for a commercial electrode made of glassy carbon, and a modified electrode, in the conditions when the electrolyte contains 5 mM H_2O_2.

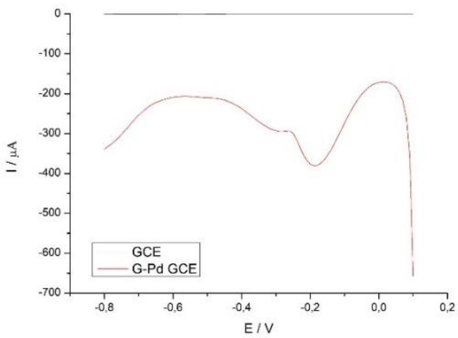

Figure 7 *Square wave voltammograms for the commercial glassy carbon electrode (GCE) and modified electrode (G-Pd GCE) in 5 mM H_2O_2, 0.1 M KOH*

From the presented figure, it is clearly seen that the modified electrode gives a visible peak compared to the commercial glassy carbon electrode when the electrochemical test was performed in an electrolyte containing 5 mM H_2O_2.

The third cycle of experiments refers to the study of changes in H_2O_2 concentration in electrolyte. Figure 8 shows the obtained voltammograms for the commercial electrode from glassy carbon and modified electrode (G-Pd), respectively.

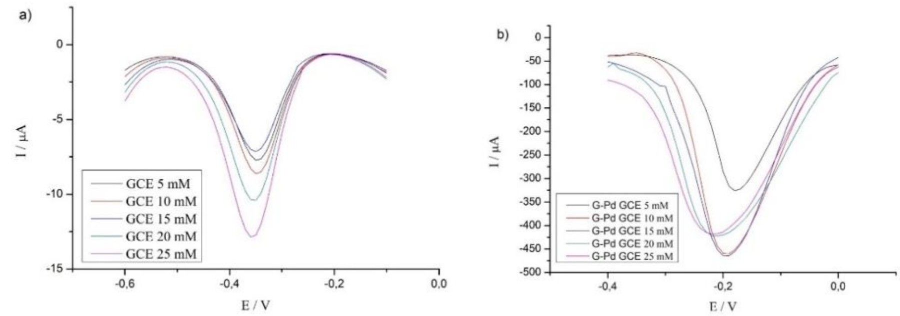

Figure 8 *Square wave voltammograms for: a) commercial glassy carbon electrode (GCE), b) modified electrode (G-Pd GCE) in 0.1 M KOH at different concentrations of H_2O_2*

It could be seen from presented figures that with a change in H_2O_2 concentration, the peak current also changes. Tables 2 and 3 show the parameters obtained by the square wave voltammetry for a commercial glassy carbon electrode and modified electrode (G-Pd), respectively. The change in peak current intensity with increasing hydrogen peroxide concentration is not linear due to the adsorption phenomena and secondary chemical processes that lead to degradation of H_2O_2.

Table 2 *Electrochemical parameters for a commercial glassy carbon electrode, obtained by square wave voltammetry in 0.1 M KOH at different concentrations of H_2O_2*

Concentration (mM)	E_p (V)	I_p (µA)	FWHM (V)
5	-0.44	-0.574	0.128
10	-0.45	-0.572	0.129
15	-0.45	-0.863	0.132
20	-0.45	-0.993	0.127
25	-0.448	-1.164	0.132

Table 3 *Electrochemical parameters for the modified electrode (G-Pd GCE), obtained by square wave voltammetry in 0.1 M KOH at different concentrations of H_2O_2*

Concentration (mM)	E_p (V)	I_p (µA)	FWHM (V)
5	-0.186	-197.44	0.227
10	-0.175	-167.54	0.266
15	-0.180	-163.77	0.237
20	-0.170	-151,598	0.225
25	-0.167	-127,642	0.201

4 CONCLUSION

On the basis of the obtained results and conducted discussion, the following conclusions can be made:

- The G-Pd hybrid material was successfully synthesized using a simple hydrothermal method and used as an electrocatalyst for HPRR. The obtained material exhibited significant electrocatalytic activity towards the electrochemical reduction of H_2O_2 compared to the unmodified and modified glassy carbon electrode under alkaline conditions. The investigation showed that the modified glassy carbon electrode had much higher sensitivity to the presence of H_2O_2 than the unmodified GCE.
- The nanohybrid G-Pd material was successfully synthesized, as confirmed by the analysis of microphotographs obtained through the scanning electron microscopy (SEM). A layered Pd-G structure was observed, and X-ray diffraction (XRD) confirmed the presence of peaks attributed to the graphene crystallographic planes and immobilized Pd nanoparticles [2].
- The electrocatalytic mechanism was predominantly controlled by a strong adsorption of hydrogen peroxide and oxygen on palladium nanoparticles. It was found that there were traces of adsorbed oxygen on the surface of palladium nanoparticles even in the absence of dissolved oxygen in the electrolyte.
- The electroreduction reaction of H_2O_2 on G-Pd was investigated using the cyclic voltammetry (CV) and square wave voltammetry (SWV) in the presence and absence of oxygen to uncover the main aspects of the electrocatalytic mechanism. A highly pronounced electrocatalytic activity of G-Pd was observed, manifested by a very high value of the reduction peak current and positive shift in the reduction potential value.
- From the obtained results, it can be also concluded that the experimental results agree with the theoretical results [5].

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ANALYSIS OF THE VERIFICATION CRITERIA OF TESTING METHODS BY TENSION OF STEEL WIRES[**]

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Abstract

One of the criteria in the laboratory practice for laboratory accreditation according to the requirements of the SRPS ISO/IEC 17025:2017 standard is the verification of methods. Verification - standard methods of testing the steel wires, has shown the verification and confirmation of methods in the specific laboratory test conditions, specified in the requirements of the SRPS ISO/IEC 17025:2017 standard. The requirements of the test method standards by verification have proved to the service user to have a confidence in the result obtained by its application. In the paper, the authors present the verification of the test method by tension of metal wires (ropes) at room temperature, Method B, according to the requirements of the SRPS EN ISO 6892-1:2020 standard.

Keywords: *analysis, criteria, method verification, tensile tests, steel wire*

1 INTRODUCTION

Accreditation of a laboratory is a means of confirming the technical competence of a laboratory that can meet the needs of users on the market. The laboratory gains its competence with accreditation to coordinate its products and services with the specific requirements and procedures of the international standard SRPS ISO/IEC 17025:2017. The accredited laboratory ensures its technical competence and quality of service provision for material testing by implementation the quality management system (QMS). To satisfy the user, the laboratory must establish, document, implement a quality management system under certain requirements of the standard [1]. Quality control can be divided into internal controls, where laboratories daily ensure the quality of work, and external controls, where many laboratories perform them, and their results are statically compared and evaluated to check the proficiency [2, 3].

One of the criteria for laboratory accreditation is the verification of standard methods. During method selection, the laboratory must use the appropriate methods and procedures for performing laboratory activities. The SRPS ISO/IEC 17025:2017 standard has requirements from the accredited laboratories

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that the standard methods must be checked and verified in application the standard methods in order to demonstrate their reliability. Checking of the standard method – verification proves that all the requirements of the standard of selected method are met. While the validation is a documented procedure that determines the suitability of measurement system for obtaining the useful analytical data [4].

During the rope operation in mining, there are changes in unfavorable conditions, because at high load there are broken wires and reduction in the cross-section S in the rope itself. To prevent a disaster, the rope must be constantly controlled. Control of the ropes must be constant during installation and operation. Steel rope is subjected to the stretching, compression, twisting and bending, so it requires the constant testing. Testing of the ropes for further use or rejection must be according to the instructions of equipment supplier and standards. The rope tested by the accredited laboratory receives a Rope Test Report where the results are given and by which method the tests were performed. Before testing the ropes, the user sends a Quality Certificate [3].

In this work, the standard method for testing the steel wires (ropes) is used by tensionning at room temperature, Method B, according to the requirements of the SRPS EN *ISO 6892-1:2020* standard. Tensile testing is one of the most important mechanical tests of materials, because it determines the most important parameters, such as: breaking (maximum) force Fm, tensile strength Rm, elasticity modulus, elongation (stretching), shrinkage, etc. The first experiments with wire tension to determine the tensile strength Rm were performed and described by Leonardo da Vinci [5].

Analysis of performance the verification of standard method for testing the steel wires by tension at room temperature, Method B, according to the requirements of the SRPS EN *ISO 6892-1:2020* standard is the input data for determining the dimensional measurement uncertainty.

The standard method is a method developed by the international organizations (SRPS ISO/IEC 17025, 2017).

The non-standard methods are methods developed in the laboratory and available in the scientific journals, as well as the modified standard methods [6-8].

Verification of the method shows that the method is reliable and accurate and that the results are consistent with the other laboratories using the same method, inter-laboratory comparative tests. The inter-laboratory comparative tests (eng. Proficiency testing schemes - PT schemes) are an external quality control of the method. The Pt schemes, developed with introduction the ISO 17025:1999 standard, became popular because they are mandatory in the accreditation process.

The criteria that determine the competence of a technical laboratory are also the appropriate management system and procedures, impartiality, confidentiality, responsibility and authorization, competent and experienced staff, appropriate calibrated equipment, valid test methods, inter-laboratory verification, control charts, measurement uncertainty and constant quality system verification (QMS) [9].

Integrated management system (ISM) as a process of quality management system integration, QMS in a modern business has become an obligation of every company in order to survive on the market. The results are better functioning, more organized processes and procedures, improved company performance and increased profits [10].

In the laboratories, where the method verification checks have been carried out, they are ready to perform the competent and reliable laboratory tests. Such laboratories are accredited by the Accreditation Body of Serbia (ATS) and are ready to work on the world market.

The organizer or providers of the PT schemes are obliged to prepare a report that includes the results achieved by the participating laboratories. The report also includes the methods used for testing and analyzing and the expected values for each measured quantity in each tested sample. The report should be precise and accurate without bias.

The organizer distributes the results of all participants in the PT scheme, as well as evaluates the reliability of the results of each participating laboratory. In the event that the test results are outside the range of expected and target values, the laboratories are forced to review the way of their work and determine the cause of deviation.

2 EXPERIMENTAL PART

2.1 Description of the method

This method is used to determine the tensile strength in tensioning the test samples of steel wires. The samples are loaded to breaking in a tensile strength testing device by tension, in accordance with the SRPS ISO 6892-1: 2020. The highest load, force Fm - breaking force (maximum force) are recorded and the tensile strength Rm of wire is calculated.

2.2 Description of the verification procedure

Evaluation of the laboratory work quality is carried out:

- By determining the individual competence of the employees in the laboratory on the basis of checking the acceptability of results of the test method over the critical interval CR $_{0.95}$.

The method for checking the acceptability of test results obtained under the repeatability conditions is based on comparison the range (x_{max}- x_{min}) of test results with the critical interval

CR $_{0.95}$ calculated from Table 1 for the corresponding n.

If the range does not exceed the critical interval, then the arithmetic mean of all n results is considered an acceptable measurement result.

Table 1 *Factors for a critical interval f(n) [11]*

n	f(n)	n	f(n)	n	f(n)	n	f(n)
2	2.8	14	4.7	25	5.2	37	5.4
3	3.3	15	4.8	26	5.2	38	5.5
4	3.6	16	4.8	27	5.2	39	5.5
5	3.9	17	4.9	28	5.3	40	5.5
6	4.0	18	4.9	29	5.3	45	5.5
7	4.2	19	5.0	30	5.3	50	5.6
8	4.3	20	5.0	31	5.3	60	5.8
9	4.4	21	5.0	32	5.3	70	5.9
10	4.5	14	4.7	33	5.4	80	5.9
11	4.6	22	5.1	34	5.4	90	6.0
12	4.6	23	5.1	35	5.4	100	6.1
13	4.7	24	5.1	36	5.4		

Verification of the method was performed by repeating the 8 measurements of tensile strength in tensioning the tested samples of metal wire, ropes.

The preparation procedure consists in the rope degreasing in its total length and washing well in gasoline, then unraveling into waists and further unraveling the waists into wires. Waists are marked with Roman numerals (most often from I to XVIII). Then the wires, each individually from the first coil, are well washed in gasoline or petroleum, manually straightened and numbered with numbers from I - 1, to I - n, where n is the total number of wires in the coil (easiest with a tape with a wire label).

The identical procedure of preparing and marking the wires is repeated with each subsequent waist.

After degreasing of wires, the next activity is cutting the wires to dimensions, i.e., length of samples necessary for testing the same on a tensile testing device, according to the requirements of the standard.

A part of each wire must remain, which is marked and kept in case of a need to repeat the mentioned test.

Record the maximum breaking force Fm, expressed in kN, tensile strength Rm in MPa (N/mm^2) and wire diameter d (r) mm.

Tensile strength during tensile testing must be expressed to the nearest 0.1 MPa (N/mm^2).

Test results are displayed by recalculating the measured values.

The obtained results are expressed by [12]:

- Average value of repeated tests $\bar{x}$;
- Standard deviation S_r;
- Repeatability limit r;
- Critical interval for n=8 tests at 95 % of probability level $CR_{0.95}$ (n).

3 RESULTS AND DISCUSSION

3.1 Factors affecting the accuracy of results

Since the tensile strength and breaking force are determined during the tensioning of test samples of steel wires, the factors that affect the accuracy of measurement are the error of micrometer, tensile testing device, as well as the error of operator.

After calculating the tensile strength and breaking force of steel wires, the obtained values are entered in Tables 1 and 2.

Table 2 *The obtained values of tensile strength Rm*

Standard: SRPS ISO 6892-1: 2020	**Method: Metallic materials - Tensile testing - Test method at room temperature, Method B**
Test No:	Test results Rm: in repeatability conditions
1	1706
2	1708
3	1711
4	1710
5	1707
6	1691
7	1675
8	1689
$\bar{x}$	1699.625
n	8
No. degrees of freedom – ν=n-1	7
$(x_{max} - x_{min})$	36
Standard deviation S_r	13.07055
f (n)	4.3
Limit of repeatability r $=1.96\sqrt{n}Sr$	r = 1.96*2.8284*13.07055 = 72.4587
Limit of reproducibility R $=1.96\sqrt{n}S_R$	/
Critical interval $CR_{0.95}$ (n) = f(n) s	4.3*13.07055 = 56.2
Note: *Relation* $(x_{max} - x_{min})$ *and* $CR_{0.95}$ (n)	
Conclusion:	The calculated critical interval is greater than the difference between the maximum and minimum determination values $(x_{max} - x_{min})$, which confirms the acceptability of results obtained by the applied test method
	56.2>36

Table 3 *The obtained values for breaking force Fm*

Standard: **SRPS ISO 6892-1: 2020**	**Method:** **Metallic materials - Tensile testing - Test method at room temperature, Method B**
Test No.:	Test results Rm: in repeatability conditions
1	5253
2	5259
3	5268
4	5265
5	5256
6	5259
7	5210
8	5253
$\bar{x}$	5252.875
n	8
No. degrees of freedom – ν =n-1	7
(x_{max}- x_{min})	58
Standard deviation S_r	18.122
f (n)	(4.3)
Limit of repeatability $r = 1.96\sqrt{n}Sr$	1.96*2.8284*18.122= 100.462
Limit of reproducibility $R = 1.96\sqrt{n}S_R$	/
Critical interval $CR_{0.95}$ (n) = f(n) s	4.3*18.122 = 77.92
Note: *Relation* (x_{max}- x_{min}) *and* $CR_{0.95}$ (n)	
Conclusion:	The calculated critical interval is greater than the difference between the maximum and minimum determination values (x_{max} - x_{min}), which confirms the acceptability of the results obtained by the applied test method.
	77.92 > 58

Analysis

The calculated critical interval is greater than the difference between the maximum and minimum determination values (x_{max}-x_{min}), which confirms the acceptability of results obtained in the test by the standard method: Metallic materials - Tensile testing Part 1: Test method at room temperature, Method B.

CONCLUSION

This work provides a calculation of the standard verification method: verification: Metallic materials - Tensile testing - Test method at room temperature, Method B.

Verification of the method was carried out by repeating the 8 measurements by tensioning the steel wires, where the tensile strength Rm and the breaking force Fm were determined, when the tested samples were tensioned, steel wires (ropes).

The calculated critical interval is greater than the difference between the maximum and minimum determination values (x_{max}-x_{min}) for tensile strength Rm and breaking force Fm. This confirms the acceptability of the obtained results for the test standard method: Metallic materials - Tensile testing Part 1: Test method at room temperature, Method B.

The analysis of verification criteria for the steel wire tension testing method confirms the acceptability of the obtained results. This means that the standard method for testing the steel wires meets the requirements set by the standard. The analysis of results has shown that the test method is applicable for testing the steel wires (ropes) by tension.

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