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Method for estimating the size of mineable resource base in the concept of an effective exploitation parcel (EPE)

Introduction

Underground mining plants, to safely and efficiently extract raw materials from the deposit, must invest significant resources and time before they start generating a profit. In Poland, the discounted payback period of a mine power plant complex, which is the time

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it takes for the initial investment to be recouped, can be as long as 30 years (Michalak 2012). To achieve financial returns at a mining site, one must prepare a detailed action plan (Godoy 2018) for each of the five stages of the mining process: searching for raw materials, exploring the deposit, constructing the mine, extracting the deposit, and reclaiming the site. The initiation of each stage involves costs, and there is no guarantee of the level of profit until the decommissioning (reclamation) process is fully completed (Ranosz et al. 2024). This is due to the uncertainty of assessing the mining conditions and their variability over time (Newman et al. 2010).

When it comes to evaluating mining investment projects, a straightforward approach that compares revenues and costs is not enough. The complexity of the factors that influence the final value of such projects necessitates a holistic and realistic valuation. This approach must account for uncertainty, which is a direct result of the characteristics of the deposits and the broader economic, technological, and environmental context. Only by considering all these factors can a comprehensive and accurate valuation be achieved.

Both internal and external factors significantly influence the resource potential of deposits. Internal factors, which are a source of project-specific risk, include the geological structure of the deposit, mining and processing technology, the availability of production infrastructure, work organisation, and the competence of human resources. However, even more substantial impacts on the value of deposits can be caused by external factors, such as market and economic conditions, volatility of raw material prices, economic policy, currency risk, or environmental conditions of the investment location. Additionally, investment returns may be affected by unpredictable major events, such as natural disasters or pandemics. The need to adapt and be resilient in the face of these external factors is crucial for the success of a mining project.

Also significant are operational factors, such as the timeliness of deliveries and the management of planning, execution, and control processes (Rogowski 2004), as they translate into the efficiency of project implementation. Kopacz et al. (2018a) emphasize the need to consider the so-called secondary aspects, i.e., the cumulative risks resulting from the interaction between factors, which further complicate the assessment of project value.

It is worth noting that many analyzes in the literature focus on selected aspects – geological, mining, or economic – without a comprehensive interdisciplinary synthesis.

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This is due to both the limited availability of reliable data and the complexity of the issue. Geological risks are relatively well recognized. Deficiencies in data availability may be compensated for by geostatistical methods, which enable the estimation of resource size and quality (David 1982). The importance of factors such as natural hazards, sedimentary disturbances, seam depth and thickness, roof and floor conditions, and coal yieldability is highlighted by Sobczyk (2009), Zhu (2011), and Khanzode et al. (Khanzode et al. 2011). Olea et al. (2011) highlight the impact of resource estimation accuracy on project value, indicating that it depends on the degree of development. Kopacz (Kopacz et al. 2018) emphasizes the importance of mineralisation of the deposit and the amount of waste rock. A synthetic approach was proposed by Vladyko et al. (2025), where the authors evaluated the project by estimating performance indicators for exploration, equipment, technological, economic, and organizational factors.

The control of quality parameters, especially the calorific value of coal, is a key element of mine production planning (Hindistan et al. 2010). At the same time, few studies have been devoted to the impact of geological resource modeling – using digital tools, geostatistics, and mathematical models – on the economic efficiency of coal projects. Notably, Heriawan and Katsuaki (Heriawan and Katsuaki 2008) used geostatistical methods to model multilayer seam thicknesses and ash, sulphur, and sodium content. Srivastava (2013) pointed to kriging as an improvement over inverse distance methods. In contrast, Rademeyer et al. (2019) presented an NPV-optimizing model, which maximizes the net present value (NPV) of a project, showing that projects are more sensitive to external factors than to production costs.

The assessment of the mineral resource base forms the basis for decision-making regarding the exploitation of the deposit and individual parcels (Kopacz 2016; Maleki et al. 2021). Uncertainty, often equated with insufficient or poor quality of available data, remains an important aspect of this assessment. This applies to each of the risk areas and geological resource estimation. Nieć (1990) indicates that errors in assessing the average values of qualitative and structural parameters can exceed 40% for poorly recognized resources. This most often occurs due to the limited number of boreholes, the time spent preparing documentation, low core yield, or interpretation errors.

In summary, the valuation of mining investment projects requires a multifaceted approach, integrating technical and geological analysis with assessments of market, environmental, and organisational factors. Only a holistic approach, taking into account interactions between sources of risk and uncertainty, allows a realistic assessment of the resource base and investment value of a project. Supporting the process of estimating deposit economic potential with digital tools is becoming an everyday practice for professionals in the industry. In this study, the authors present a comprehensive method for evaluating the mining resource base, which takes into account changing external and internal factors. The method accounts for the impact of uncertainty and utilizes digital solutions to enhance the accuracy of estimating factors such as geological parameters or exploitation costs.

1. State of knowledge

1.1. Resources estimation

The estimation of recoverable resources, both in Poland and internationally, is subject to classification, technological, and economic principles. Although the scientific basis is similar, the classification systems and practices used differ between countries due to local legal, industrial, and environmental conditions (Camisani-Calzoari 2004; Grubert 2012; Nieć 2010; Nieć 2012; Rendu and Miskelly 2001). In Poland, national systems with strong state oversight and geological classification dominate, while transparent, market-based classification models (e.g., JORC Code, NI 43-101) are used worldwide (Miskelly, n.d.; Nieć 2019). Since 1994, the Polish principles for the classification of resources have been compiled in the Regulation of the Minister of the Environment "On detailed requirements to be met by geological documentation of mineral deposits". The latest update of this Regulation dates from 2011 and implements the Act of 9 June 2011 (Geological and Mining Law). The documentation system currently in force in Poland (along with the inherent exploration and prospecting processes) dates back to the times of a centrally controlled economy and its main result is the inventory (balancing) of resources with very little consideration given to their economic utility (Sieniawska and Wierchowiec 2016).

One of the international standards for documentation and classification is the JORC Code. Almost from its inception, the JORC Code gained recognition and approval from the financial community, which translated into its recognition by stock exchanges: Australian Stock Exchange (1989) and New Zealand Stock Exchange (1992) as a necessary standard for presenting mineral reserves (Saługa et al. 2015). Since then, it has become an informal international standard, with many countries utilizing it to update their national classifications and enhance transparency in the assessment of mineral resources (Galos et al. 2015). A distinctive feature of the JORC Code is the requirements that must be met by the data, based on which the resources of a deposit are estimated. Once compiled, these data are included in the Public Report. They must meet the conditions of transparency, relevance, and competence (Australasian… 2012). The JORC Code defines two categories: 'resources' and 'reserves'. Within the categories, resources were divided into three groups based on the core yield for the studied samples and their maximum allowable spacing of points of observation (drill holes or in situ profiling). The resource may be inferred, indicated, or measured, and the final classification is confirmed by a Competent Person (CP) in each case.

Recoverable resource estimation is a key element of strategic planning for a country's economy, raw material geology, energy, and environmental protection. The current state of knowledge in recoverable resource estimation indicates that it is no longer solely a domain of geology, but rather an interdisciplinary field that combines science, industry, economics, and policy. The development of digital tools, automation, and advanced data analytics significantly increases the precision and efficiency of these processes (Dyczko 2023; Kulpa et al. 2024; Rendu and Miskelly 2001; Szamałek et al. 2021).

1.2. Mining process optimization

The mining process should be continuously optimized to ensure that the extraction of the desired volume of raw material is as economically viable, safe, and environmentally friendly as possible. IT tools support decision-making and further optimisation of the mining process on many levels (Alford et al. 2007). Among these are:

- selection of the location and grid of geological boreholes so as to identify the deposit as well as possible while minimising the costs incurred (Boucher et al. 2005),
- selection of the profitability threshold, i.e., the minimum content and quality of the raw material allowing its economically efficient extraction, both globally for the mine and for its parts (Ahmadi and Shahabi 2018; Will and Vendla 2018; Khan and Asad 2020),
- selection of the size and shape of the fields and excavations, depending on the exploitation system. These may include pillars, chambers, longwalls (Erdogan et al. 2017; Nhleko et al. 2018; Villalba Matamoros and Kumral 2019),
- design of access and preparatory excavations so as to minimize their excavation costs and subsequent exploitation costs (Brazil et al. 2002),
- selection of the exploitation method depending on the shape and depth of the deposit, yieldability of the rock mass, and natural hazards (King 2018; Kosenko 2025),
- optimisation of the exploitation schedule, allowing for the postponement of the incurred expenditures and the acceleration of revenues (Dimitrakopoulos and Jewbali 2013).

The most significant number of studies on optimising mining schedules concerns open-pit mines. A solution based on the mixed-integer programming (MIP) method was proposed in 2003 for the LKAB mine in Kiruna (Topal et al. 2003; Campeau et al. 2022). Mixed programming has been used in models to optimize mining production scheduling at the Stillwater plant (Carlyle and Eaves 2001), building a schedule that maximizes revenue from mined ore. This solution assumes deterministic geological data. A method that considers the uncertainty of the geological data is mixed stochastic programming (SIP) (Carpentier et al. 2016; Dimitrakopoulos and Jewbali 2013). A different approach is presented in Brzychczy and Wnuk-Lipiński (2013). This solution uses multi-criteria optimisation based on an evolutionary algorithm. It takes as objective functions the minimum standard deviation of the assumed periodically varying extraction volume and the minimum standard deviation of the extraction level itself. However, the solution does not consider the time required to carry out preparatory works for the scheduled longwalls. In this study, the authors attempted to develop a holistic method for evaluating a flexible resource base in response to changing factors. This method includes elements of uncertainty and makes use of digital solutions to increase the accuracy of estimating exploitation profit.

2. Scientific objectives, methodology and scope of work

2.1. Objectives

This article aims to present the concept of EPE, which is the amount of resources that can be profitably mined under given technical, economic, and mining conditions. EPE uses computer decision support tools for deposit modelling and production scheduling. The concept asserts that resources belong to EPE if they are mineable, meeting a boundary profitability condition with an NPV of 0, and with an optimized mining plan and schedule. Therefore, EPE presents a resource base that evolves in response to the considered conditions. In the proposed method, estimating the resource base requires determining the economic value of resources as well as the costs of their extraction. This is achieved by locating mining works in 3D space and interrogating the geology model for necessary qualities, for multiple possible variants of exploitation layouts. For comparison, frameworks such as JORC or NI 43-101 typically consider a limited number of layout variants, as a mining engineer must manually or semi-automatically prepare each one.

The scientific objective of this article is illustrated by a computational example, which presents the results of estimating the resource base under certain technical and economic extraction assumptions. The analysis was carried out for a base case scenario, which was presented with two variants of longwall mining schedules. In the first variant, authors assumed the depletion of resources, starting with parcels of the maximum profit potential and proceeding to those of the minimum profit potential. In the second variant, the economic result was averaged over time by simultaneously exploiting a longwall with the maximum and minimum values of the income potential. This issue is described in detail later in the paper. In addition, for each of these scenarios, the resource base was estimated as a function of the base price of coal, which varied from –20% to +50%, and the results of this analysis are presented in Chapter 4.

2.2. Methodology and scope of work

The proposed methodology for the determination of an effective exploitation parcel includes:

1. Determination of parcels that constitute an organized longwall system.
2. Automatic generation of many adjacent longwall layouts for each parcel, given the assumed constraints of the designed mining operation.
3. Estimation of the value of the coal to be extracted and the cost of extracting coal from the parcels, together with the allocation of expenditures related to the maintenance of the technological infrastructure of the mine.
4. Estimation of the income potential of longwalls and parcels.

5. Defining the order in which parcels are mined, with the aim of maximising economic efficiency indicators or averaging them over time.
6. Optimisation of mining schedules aiming for maximising the margin or revenue resulting from the sale of coal accumulated in the parcel.
7. Estimation of total risk for longwalls, parcels, and mines used to adjust the base discount rate in the hierarchical model.
8. Estimation of the economic efficiency indicators of the deposit included in the EPE scope, i.e., NPV (Net Present Value) and IRR (Internal Rate of Return).
9. Ranking of longwalls and plots' economic potential.
10. Generalization of conclusions and recommendations.

Under the proposed method, geological information from a digital model of the deposit is loaded into a planning software. Technical constraints for extraction and baseline economic data are entered, based on which it is possible to estimate the costs of extracting a section of the deposit and the revenue achievable from the sale of the extracted raw material. In addition, a nuisance model of geological and mining conditions that increases unit mining costs was prepared, which allowed for the determination of a variable discount rate for individual variants. The developed method was tested on the hard coal seam planned for extraction. In the process of an optimized mining schedule, the economic results of extracting the seam and the size of the resource base for eight price variants in the range of 0.8–1.5 times the base price of hard coal were determined.

2.3. Planning and optimisation of longwall layout

In the method, the design process starts with the delineation of the mining boundaries within the parcels. The plots are designed to be safely extracted. Therefore, planning begins with an analysis of the safety pillars and faults that constitute the parcel boundaries, as well as the natural hazards that may occur within it. The order in which the longwalls are picked may be determined by mining conditions, particularly the impact of mining on the surface and the hazards of water, fire, and rockbursts.

Planning and optimisation take place in a proprietary application developed for this purpose. Once the boundaries of the plots have been established, constraints are placed on the geometric parameters of the operation due to the natural hazards present. For longwall workings, the constrained variables are the azimuth of the longwall run and the width of the longwall. Then, the unit cost of the working day for the longwall complex is estimated, as well as the unit costs of cutting the necessary development tunnels, depending on their purpose, the annual cost of maintaining the mine, and the average daily advance of the longwall face. Income from coal sales is estimated based on the coal quality parameters of the longwall (parcel).

Additionally, the interval for changing the input parameters (longwall length, longwall width, position of the layout starting point, and azimuth of the longwall run) is specified.

For each combination of parameters, the application creates a new mining variant, draws the outlines of the longwall workings and then tests the geological model for the volume and quality of the contained raw material and, on this basis, determines the value of the raw material, the cost of its extraction and, further, the profit and margin resulting from the extraction of the parcel. All variants, together with the economic parameters, can then be exported in tabular form or analyzed in the software with their graphical interpretation. A conceptual diagram of the process is shown in Figure 1.

The following optimisation issue is determining the order in which parcels and longwalls are extracted. The authors propose two key approaches, consisting of:

- maximising the value of the deposit (venture) for the level of the margin or profit;
- averaging the value of the deposit (venture) by the level of the margin or profit.

The approach described above enables the creation of two schedules: in the first case, a schedule that prioritizes the highest-margin parcels, and in the second, a schedule that maintains a stable margin throughout the operation's life. This optimisation is carried out considering technical, organisational, mining, and technological constraints. Of all the longwalls and deposit parcels examined, those for which the margin is negative are rejected, meaning that their selection under the given market conditions will only worsen the outcome of the mining project and is unprofitable.

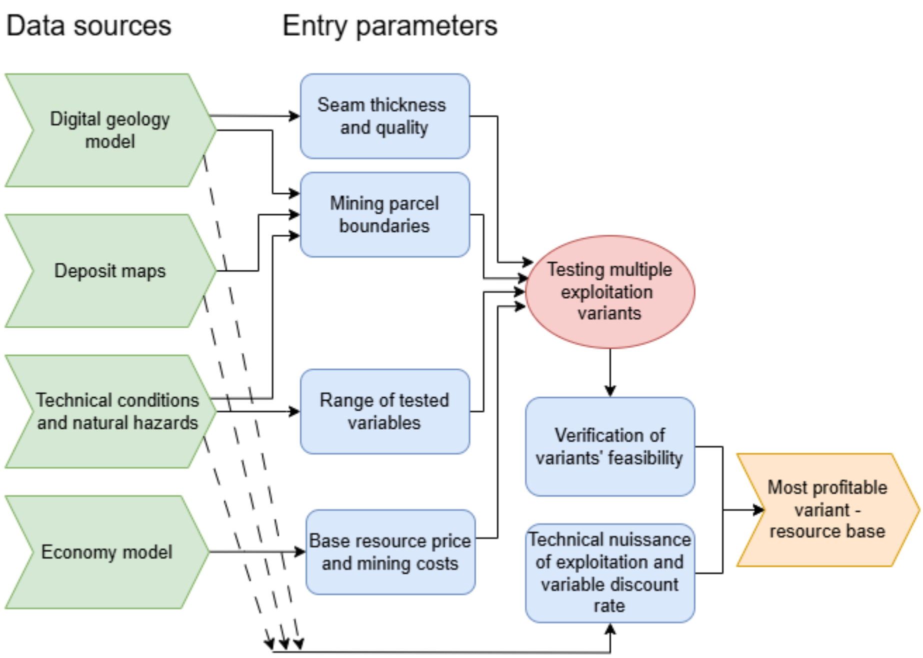

Fig. 1. Base elements of the method
Source: own work

Rys. 1. Bazowe elementy metodyki

2.4. Raw material volume, capital expenditure and operating costs

In selecting the most favourable option for exploiting the parcel, several economic indicators should be considered. Firstly, based on the data taken from the deposit model and the assumed base price of coal, the estimated value of the resources (coal) that can be extracted in each variant should be determined.

Value of hard coal

The value of hard coal (thermal coal) W_s available for mining in each variant was determined using formula:

$$W_s = p_b \cdot \sum_{i=1}^{n} A_i \cdot h_i \cdot d_i \cdot \left(\frac{qia_i}{25} - 11(sa_i - 1) - 0.4(aa_i - 16) \right) \tag{1}$$

- W_s – sale value of hard coal to be extracted in each variant from a longwall set (PLN),
- p_b – base price of raw material with standard quality parameters (PLN/Mg),
- A_i – area of the elementary part of coal seam (i) to be mined (m^3),
- h_i – average thickness of i (m),
- d_i – average density of i (Mg/m^3),
- qia_i – calorific value of coal in i (Mj/kg),
- sa_i – sulphur content of the mineral in i (%),
- aa_i – ash content of the mineral in i (%).

Equation (1) considers the correction of the raw material price based on the deviations of its quality parameters from the benchmark parameters, using a modification of the formula of Grudziński (Grudziński 2009) adapted to the hard coal standards of the CIF ARA index for the year 2024.

Capital expenditure

The capital expenditure of a given longwall (mining parcel) was accounted for and capitalized over time through depreciation of the mine's basic fixed assets (in the mine under review, the value of annual expenditures corresponded to the level of annual balance sheet depreciation).

Operating costs

In the next step, the mining costs of the plots at a given longwall layout were estimated. The mining costs were assigned to the plots by dividing them into three cost groups:

- direct costs of coal extraction at the longwall,
- the costs of excavating roadways necessary for mining plots, together with the costs of cutting longwall galleries,
- costs constituting a cost markup for the maintenance of the technical infrastructure of the plant and processes supporting mining.

The first group comprizes the direct costs at the longwall, which include the total operating costs of extraction, primarily the costs of wages and salaries, as well as their derivatives, material and energy consumption, and third-party services. The Formula defines these:

$$C_{dlj} = c_{lj} \cdot \sum_{j=1}^{n} \frac{l_j}{p} + \left(c_{rf} + c_{lq}\right) \cdot n \tag{2}$$

C_{dlj} – direct costs of the longwall j (PLN),
c_{lj} – unit cost of extraction in the longwall (PLN/day),
l_j – length of the longwall j (m),
p – average (daily) advance assumed for the longwall (m/day),
c_{rf} – average cost of reinforcing the longwall (PLN),
c_{lq} – average cost of liquidation of the longwall gallery and casing in the longwall (PLN),
n – number of longwalls.

The second group is the cost of excavating the tunnels lining the parcel, calculated individually for each variant:

$$C_d\left(w.\right) = \sum_{j=1}^{m} \left(C_{D,k} \cdot L_k\left(w.\right)\right) \tag{3}$$

$C_d(w.)$ – costs of digging development for a mining parcel in variant (PLN),
$C_{D,k}$ – unit costs of tunnelling a heading, depending on its purpose (PLN/m),
$L_k(w.)$ – summary length of headings of particular purpose needed in variant (m).

The costs determined by Formula (3) were estimated as the sum of the product of the unit costs of headings j of a given cross-section and purpose ($C_{D,k}$) and the length of workings ($L_k(w.)$) in a given variant of development of mining plots. For each longwall development variant, the costs of cutting longwall galleries were also considered. The total length of the longwall galleries was taken as the product of the number of longwalls (n) in a variant and the width of a given longwall ($L_{kw.}$).

$$L_g = n \cdot L_{lw.} \tag{4}$$

The total length of the headgates and tailgates was determined based on the contour runs of the variant longwalls, for each excavation, adding 50 m of the protective pillar of the transport gates.

The total length of the contour workings (main workings) was determined as twice the sum of the lengths of the parcel longwalls, bearing in mind that this value is greater and depends on the location of the parcel in relation to the plant's capital workings. However, the value determined approximates the part of the main workings, whose location directly depends on how the parcel is cut.

The third group includes all costs that are not directly dependent on the excavation and mining process. These costs, typical of a given cutting variant, are determined by the Formula:

$$C_z\left(w.,d\right) = C_z \cdot LLE \tag{5}$$

$C_z(w.,d)$ – plant infrastructure maintenance cost markup in group III for w. during the extraction period (PLN),

C_z – other plant operating costs included in group III during the extraction period (PLN/year),

LLE – number of years of operation, calculated according to the following Formula:

$$LLE = \frac{\sum_{i=1}^{n} \frac{l_j}{ad}}{365 \cdot n_p} \tag{6}$$

l_j – length of longwall j in the variant (m),

ad – average daily advance assumed for the walls in the variant (m),

n_p – number of longwalls mined in parallel.

Table 1. Process-based allocation of mining site costs

Tabela 1. Podział kosztów zakładu górniczego w ujęciu procesowym

Direct costs of longwall coal extraction	Cost of development tunnels	Plant intependent costs
• P_2. Coal extraction in longwalls • P_3.1. Run-of-mine transport from longwall face • P_4.2. Logwall reinforcement and deconstruction	• P_1. Digging of tunnels (tailgates, headgates, galleries, main gates for the plot • P_3.2. Run-of-mine transport from tunnel faces • P_4.1. Reinforcement, reconstruction and deconstruction of plot tunnels • P_4.3. Floor scraping	• P_1. Digging of plant's main gates • P_3.3. Run-of-mine haulage to shafts • P_3.4. Shaft transport (including maintenance) • P_5. Energy • P_6. Water pumps • P_7. Ventilation, air conditioning and hazard management • P_10.1. Mineral processing • P_10.2. Coal transport on surface • P_10.3. Waste rock management • P_ (8; 9; 11–22). Other processes

Source: own work based on (Kopacz 2017)

The costs of group three include costs that are not directly dependent on mining, i.e. the processes listed in column 3 of Table 1. The total number of separated processes is 22 (processes P_1 to P_22 according to the placement cost division in coal mines).

Using the revenue and costs associated with extracting resources through a given tunnel layout, the profit and margin were calculated. Margin was understood as the difference between revenue and costs in a longwall or parcel, per tonne of coal. Profit, on the other hand, was the difference between the total revenues and costs of a longwall (parcel) for a given mining variant (schedule).

The margin criterion was adopted as the most relevant for indicating the most economically viable longwall workings layout in a given parcel.

2.5. Exploitation risk – specific risk of a mine

Within prepared schedules, it is possible to determine the NPV of the different alternatives based on the variable discount rate concept developed in MEERI PAS (Sobczyk et al. 2024). To assess the risk level associated with underground coal mining, a methodology has been proposed. It considers the impact of risk factors on the mining process, resulting from geological and mining conditions, including safety, economic efficiency, and rational resource use, using the mathematical multi-criteria decision-making method AHP (Analytic Hierarchy Process). To objectively assess the level of risk in the mining process, particularly in terms of the impact on the production cost of longwalls, the following procedure was developed:

a) Determination of the risk structure model of the longwall mining process

The hierarchical model for assessing the risk of increased unit exploitation costs in longwalls is presented in Figure 3 of Chapter 4.

The hierarchical model consists of 3 levels. The first level contains the main objective of the task: assessing the level of risk of increased unit exploitation costs in longwalls. The second level of the model is represented by three main groups of risk factors, which include:

- mining factors,
- geological factors,
- natural hazards.

On the lower, third level of the hierarchical model, sub-criteria were introduced, representing a more detailed development of the Level II criteria, specific to the analyzed mining plant.

In the group of mining factors, three sub-criteria were considered:

- longwall length (m),
- longwall width (m),
- distance from the descent shaft (m).

In the group of geological factors, four sub-criteria were distinguished:
- seam thickness (m),
- reserves (thousand Mg),
- tectonic disturbance (index),
- seam depth (m).

In the category of natural hazards, the following were introduced:
- water hazard (grades),
- coal susceptibility to spontaneous combustion (group),
- rockburst hazard (grades).

b) Calculation of the weights (priority valuation) of the individual criteria building the model

The impact of the individual variables was analyzed with the help of experts, and the impact of the individual criteria was assessed on Saaty's nine-point scale (Saaty 1987). Adjusting the project's specific risk was based on the hypothesis that the variability of the individual geological and mining factors can be aggregated in the form of the variability of project risk *Rp* using the developed risk index *RF*. In the weighted cost of capital (*WACC*) concept, project risk is understood as a component of the cost of equity (*COE*). An adjustment of the *WACC* has been proposed so that the specificity and variability of individual geological and mining parameters can be appropriately reflected in the process of assessing the economic efficiency of coal deposits (Sobczyk and Kopacz 2018). At the basis of the concept of *WACC* adjustment is the assumption that, with a specific generalisation, the components of both components of *WACC*: the cost of equity and the cost of debt, can be juxtaposed, performing an ordering and simplification of the components of the formula for *WACC* for the purposes of further analyzes (Sobczyk et al. 2024). This results in a formula in general form:

$$WACC \cong COE + COD = \left(\beta(Rp) + rf\right) + \left(rf + Rp + P\right) \quad (7)$$

↳ *WACC* – weighted cost of capital (PLN),
COE – cost of equity (PLN),
COD – cost of debt (PLN),
β – a measure of the systematic risk of an asset, i.e. its volatility compared to the market as a whole (%),
Rp – specific project risk (–),
rf – market equivalent of non-diversified (ambient) risk, which can be expressed for projects up to one year, e.g. by the WIBOR rate (%), or for medium- and long-term projects by the interest rate on government bonds, with an appropriate maturity (%),
P – profit (%).

After considering equity and debt shares as: A and respectively: $(1 - A)$, simplifying the sides towards R_p, the *WACC* can be presented as:

$$WACC = Rp \cdot (A \cdot \beta + 1 - A) + W \cdot (1 - A) + rf \tag{8}$$

The weighted cost of capital, *WACC*, provides a useful tool for calculating the cost of equity individually for each parcel and the entire deposit (mine/investment project), by appropriate adjustment, determined by the formula:

$$Rp = (A \cdot \beta + 1 - A)\frac{WACC - W \cdot (1 - A) - r_f}{A \cdot \beta + 1 - A} \tag{9}$$

The relative difference of the *RF* index was determined for each parcel according to the formula:

$$\delta_i = \frac{(RF_i - RF_c)}{RF_c} \tag{10}$$

↳ RF_i – risk index estimated for the parcel (–),
RF_c – risk index estimated for the deposit (–).

2.6. Data sources

For the purposes of the study, data from mine *X*, which plans to mine between 2027 and 2049, were used. The geological data were obtained from a previously prepared digital model of the deposit, which comprized 66 historical test holes drilled during the initial reconnaissance stage of the deposit and 10 roadway profiles drilled in the last year (Figure 2). During the modeling, primarily 2D seam maps and data on quality parameters were used. In addition, maps and cross-sections from the geological documentation of the 'Y' coal seam were used for comparison.

For the quality parameter model, 42 samples were used, which included ash content (AA), calorific value (QIA), sulphur content (SA), bound moisture (WA), bulk density (DA), and volatile matter content (VA).

Technical parameters, such as the achieved longwall workings advance and maximum working dimensions, were estimated based on the results achieved at neighbouring mines from 2021 to 2024.

The costs of operation and maintenance of the plant were estimated based on the data provided by mine X and modified accordingly. The costs of excavation were estimated based on a quote from a mining company that performs roadway excavations.

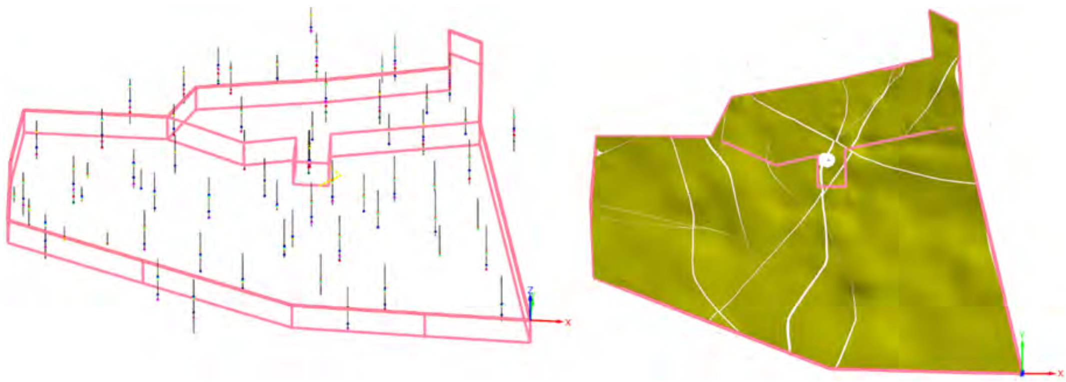

Fig. 2. Visualisation of surface drillholes and profiling (against the background of the seam 118)
Source: own study

Rys. 2. Wizualizacja otworów powierzchniowych oraz profilowań (na tle bryły pokładu 118)

3. Computational example

The ‘X1’ deposit area is located in the eastern part of the Upper Silesian Coal Basin, on the northern slope of the Main Basin. The strike of the strata has a direction from S-N to NWW-SEE, and the dip directed to E and NE is from 2 to 10°. Productive Carboniferous sediments represent the Carboniferous within the depth range of the boreholes, i.e., up to approx. 1,000 m. Formations represented by the Libiąskie (Westfal D), Łaziskie (Westfal C), and Orzeskie (Westfal B) strata. According to the Deposit Development Project, deposit 118 (Libiąskie strata) was selected for exploitation.

An outcrop is present in the extreme north-western part of the deposit. Thickness in the seam varies from 1.3 m in hole Bhl-VIk to 4.4 m in hole G-15. The seam covers an area of over 36 km^2 and is characterized by an inclination of approximately 2–7°.

The deposit is cut by a series of faults with dislocations reaching up to 280 m. Some of the dislocations are known in terms of their course and throw values (in areas of neighbouring deposits), while others run between drill holes and their course can only be determined by mine workings.

Within the ‘X1’ deposit, following an analysis of the deposit structure, tectonics, natural hazards, and regulatory safety pillars (for regional faults, mine and surface infrastructure), nine mining plots of seam 118 were chosen for analysis. The previously planned layout of shafts and capital tunnels was used to determine the distance to the shaft. The deposit is accessed via one shaft and one ramp, both of which lead from the surface. Its exploitation employed a longwall system with roof collapse, utilizing two longwalls excavated in parallel in separate plots. Due to the high risk of coal spontaneous combustion, it was decided that no coal fences would be left between successive longwall workings. To prevent water hazards, it was decided that the longwall workings must not be carried out in the direction of the dip. The remaining calculation assumptions are summarized in Table 2.

Table 2. Calculation assumptions for the presented example

Tabela 2. Założenia obliczeniowe przykładu pracy na metodzie

Variable	Value
Base price of coal (PLN)	456
Cost of driving one metre of headgate (PLN/m)	15,150
Cost of driving one metre of longwall gallery (PLN/m)	21,450
Cost of driving one metre of main heading (PLN/m)	48,000
Cost of longwall reinforcement (PLN thou.)	21,000
Cost of closing the longwall and moving the support out (PLN thou.)	6,000
Independent costs of the plant (PLN thou./year)	670,242
Average daily advance of longwall exploitation (m/d)	3
Range of permissible longwall run azimuth (°)	15–195
Range of permissible longwall widths (m)	16–250
Minimum longwall length (m)	500
Change in longwall run azimuth (°)	5
Change in position of layout start point (m)	10
Change in longwall length (m)	10
Change in longwall width (m)	5

Source: own study

As mentioned in the description of the study's scientific objectives, simulations of the resource base size and NPV ratios were conducted for deviations from the base coal price value in the range of –20% to +50%, in 10 percentage point increments. This made it possible to generate a total of eight variants of schedules for a given sequence of exploiting individual longwalls and parcels.

The results of the analysis are preceded by a detailed description of the identified longwalls and parcels, presenting their value-generating potential in Table 3. In addition, Figure 3 and Table 4 present the analytics used to estimate the discount rate adjustments for individual longwalls and parcels, which were considered when estimating the NPV and the size of the resource base.

4. Results

After calculating the margin for each parcel, the variants whose extraction would generate the highest margin were selected. On this basis, the plots were sorted according to the criterion of the margin level. The results are summarized in Table 3.

Table 3. Economic results of extracting the parcels

Tabela 3. Wyniki ekonomiczne wybrania parcel

Parcel	Longwall	Parcel margin (PLN/Mg)	Longwall margin (PLN/Mg)	Parcel profit (PLN)	Longwall profit (PLN)
C	Longwall 2 C	60.799	65.3	147,371,669	116,422,090
	Longwall 1 C		48.28		30,949,579
D	Longwall 2 D	34.204	46.117	152,830,080	65,476,278
	Longwall 3 D		42.648		57,720,281
	Longwall 1 D		28.098		24,895,115
	Longwall 4 D		5.857		4,738,407
G	Longwall 3 G	23.829	36.009	141,527,220	102,026,498
	Longwall 2 G		23.449		56,364,241
	Longwall 1 G		–24.016		–16,863,519
A	Longwall 1 A	23.712	24.902	51,237,802	31,370,724
	Longwall 2 A		22.048		19,867,078
B	Longwall 4 B	16.104	43.858	78,440,522	84,747,968
	Longwall 3 B		24.795		35,224,294
	Longwall 2 B		–3.092		–3,022,804
	Longwall 1 B		–71.295		–38,508,936
E	Longwall 4 E	–12.140	10.139	–55,753,411	15,878,670
	Longwall 3 E		–3.315		–4,482,990
	Longwall 2 E		–22.61		–24,624,871
	Longwall 1 E		–72.706		–42,524,220
B1	Longwall 1 B	–14.636	–14.636	–9,090,857	–9,090,857
H	Longwall 1 H	–21.573	–2.139	–84,343,041	–3,150,440
	Longwall 2 H		–17.902		–25,162,591
	Longwall 3 H		–54.328		–56,030,009

Table 3. cont.

Tabela 3. cd.

Parcel	Longwall	Parcel margin (PLN/Mg)	Longwall margin (PLN/Mg)	Parcel profit (PLN)	Longwall profit (PLN)
J	Longwall 2 J	–23.558	17.266	–346,846,571	14,975,495
	Longwall 4 J		7.977		20,001,553
	Longwall 5 J		–0.581		–1,422,515
	Longwall 3 J		–2.197		–2,061,774
	Longwall 1 J		–7.501		–4,699,272
	Longwall 6 J		–13.671		–29,142,567
	Longwall 7 J		–28.108		–51,084,023
	Longwall 8 J		–55.972		–82,771,205
	Longwall 9 J		–94.443		–105,177,536
	Longwall 10 J		–132.81		–105,464,726

Source: own study

To determine the economic outcome of the different scheduling options, the NPV ratio, estimated based on cash flows updated with a variable discount rate, was used. The variable discount rate was determined from the risk model for deposit "X1", the factors of which are summarized in Figure 3.

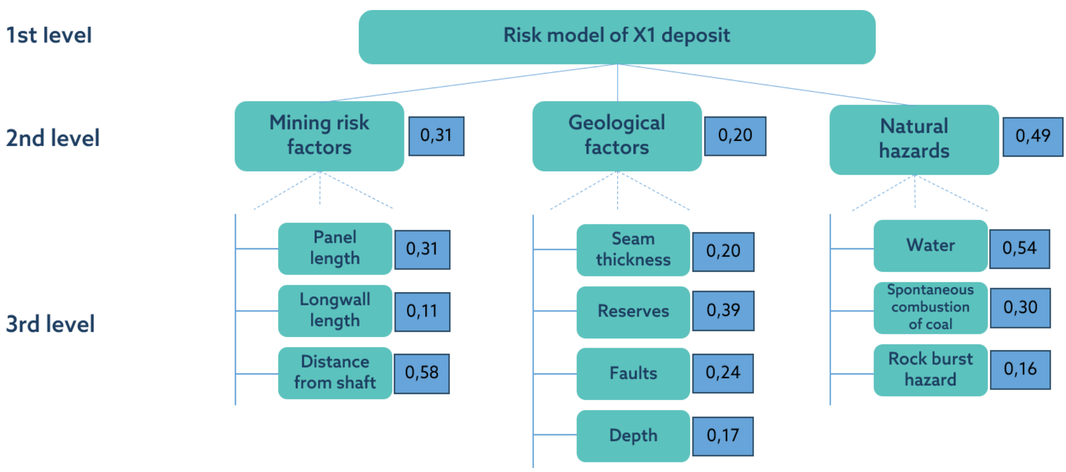

Fig. 3. Model for assessing the level of risk of an increase in unit longwall operating costs
Source: own study

Rys. 3. Model oceny poziomu ryzyka wzrostu jednostkowych kosztów eksploatacji w ścianach

Construction of the exploitation risk indicator (RI)

The obtained weights of the criteria building the hierarchical model of evaluating the level of risk of increased unit exploitation costs at longwalls were used to construct the exploitation risk index, RI. This indicator is the sum of the ratio of the weights of individual criteria to the normalized values of those criteria at the longwalls analyzed:

$$RI = \sum_{i=1}^{n} N_i \cdot z_i \tag{11}$$

RI – value of the exploitation risk index,
i – index of statistical feature,
n – number of statistical features,
N_i – weight of the statistical feature j,
z_i – value of the normalized feature.

For the construction of the index, risk factors were used, the orders of magnitude of which had to be standardized and brought to comparability through normalisation.

The quotient transformation (Sokołowski 1982) was used:

- for stimulants:

$$z_{ij} = \frac{\left(x_{ij} - \min x_{ij}\right)}{\left(\max x_{ij} - \min x_{ij}\right)} \tag{12}$$

- for destimulants:

$$z_{ij} = \frac{\left(\max x_{ij} - x_{ij}\right)}{\left(\max x_{ij} - \min x_{ij}\right)} \tag{13}$$

i – index of deposit parcel,
j – index of statistical feature (criterion),
x_{ij} – value of the feature j in deposit parcel i,
$\min\{x_{ij}\}$ – minimum value (lower reference point),
$\max\{x_{ij}\}$ – maximum value (upper reference point),
z_{ij} – transformed values.

The values of the RI risk index obtained using the AHP method were calculated for 34 longwalls located in nine mining plots of the hard coal deposit. The index consists of ten risk factors grouped into three groups: mining factors, geological factors, and natural hazards.

Table 4. Values of RI for individual longwalls and mining plots

Tabela 4. Wartości wskaźnika RI dla poszczególnych ścian i parcel eksploatacyjnych

Longwall name	Longwall RI	Parcel name	Parcel RI
Longwall 1 A	0.7279	A	0.6222
Longwall 2 A	0.4877		
Longwall 1 B	0.6763	B	0.6553
Longwall 2 B	0.6527		
Longwall 3 B	0.6451		
Longwall 4 B	0.6635		
Longwall 1 B1	0.6625	B1	0.6703
Longwall 1 C	0.4387	C	0.4169
Longwall 2 C	0.4116		
Longwall 3 C	0.4924		
Longwall 1 D	0.3765	D	0.3704
Longwall 2 D	0.3707		
Longwall 3 D	0.3756		
Longwall 4 D	0.3981		
Longwall 1 E	0.5490	E	0.3912
Longwall 2 E	0.4376		
Longwall 3 E	0.3993		
Longwall 4 E	0.3591		
Longwall 1 G	0.2759	G	0.2753
Longwall 2 G	0.2667		
Longwall 3 G	0.2677		
Longwall 4 G	0.3198		
Longwall 1 H	0.2720	H	0.2441
Longwall 2 H	0.2637		
Longwall 3 H	0.2891		
Longwall 1 J	0.5065	J	0.1991
Longwall 2 J	0.4426		
Longwall 3 J	0.4109		
Longwall 4 J	0.3700		
Longwall 5 J	0.3451		
Longwall 6 J	0.2956		
Longwall 7 J	0.2084		
Longwall 8 J	0.1756		
Longwall 9 J	0.2024		
AVG for deposit	0.3760		

Source: own study

The values of the individual risk factors in the analyzed parts of the deposit and longwalls are shown in Table 4.

The average RI value for the entire deposit is 0.3760, with the three mining plots analyzed (J, H, and G) having lower RI values. In contrast, the remaining plots (A, B, B1, C, D, and E) have a risk index value significantly higher than the average value for the deposit. Parcel B1 has the highest RI risk index value with a score of 0.67. Equally high RIs were estimated for parcel A (RI = 0.66) and for parcel B (RI = 0.67). These values show the dominant influence of water and fire hazards on the RI value compared to the other parcels. Such a high RI is also influenced by the considerable distance from the descent shaft. The lowest risk is found in parcel J with a score of RI = 0.20. Such a low value of the risk index is mainly influenced by the smallest distance from the descent shaft in relation to the other lots, the high volume of reserves, the relatively long longwalls, and the least constraints due to natural hazards.

The index values determined in this manner formed the basis for recalculating the cost of equity for each parcel. It was assumed that the base WACC value would be 10%. Using the following formulae (7–10), a value per Rp for the entire deposit was calculated at 3.64%. The values for individual plots were already determined according to formula (14) and are presented in Table 5.

$$Rp_i = Rp\left(1+\delta_i\right) \tag{14}$$

In turn, substituting all the variables, along with the newly estimated values, enabled the determination of the adjusted weighted cost of capital for individual parcels and the entire deposit. Summaries of Rp_i and $WACC_i$ values for individual parcels and the entire mine are presented in Table 5 below.

After considering the nuisance of the geological and mining conditions, and after applying an adjustment to the discount rate for this, Table 6 presents the estimation of the size of the resource base of seam 118, the NPV, and the mining time in this seam (*LLE* – number of years of mining). These values were estimated based on mining schedules constructed individually for deviations of the average price value (±20% to ±50%) from the assumed price base value: 456 PLN/Mg of coal. It can be noted that for the base case scenario, the estimated NPV was close to PLN 442 million, which would indicate the economic viability of extracting only 19.4 million Mg of coal in the extracted seam of the "X1" deposit, and would indicate a mine life of 9 years – increasing the price by 50% results in an increase of the NPV by nearly PLN 6 billion, and continuing the exploitation of the seam 118 over the next 25 years. A decrease in price by more than 10% from the base value results in a loss of project profitability and a depleted resource base. Figure 4 presents the location of longwalls for the exemplary variant – base price +10% on the background of parcel borders. The colours in the figure represent the margin expressed in PLN per tonne.

Table 5. Estimated values of relative difference (δ_i) of Rp_i factor as well as WACC for individual parcels and the entire deposit

Tabela 5. Oszacowane wartości względnej różnicy (δ_i) wskaźnika Rp_i oraz WACC dla poszczególnych parcel i całego złoża

Parcel	RF risk factor	δ_i relative difference of RF (%)	Rp_i (%)	$WACC_i$ (%)
A	0.6222	82.2	6.63	13.29
B	0.6553	91.9	6.98	13.68
B1	0.6703	96.3	7.14	13.85
C	0.4169	22.1	4.44	10.88
D	0.3704	8.5	3.94	10.34
E	0.3912	14.6	4.17	10.58
G	0.2753	–19.4	2.93	9.22
H	0.2441	–28.5	2.60	8.86
J	0.1991	–41.7	2.12	8.33
Deposit	RF_c = 0.3415		3.64	$WACC_c$ = 10%

Source: own study

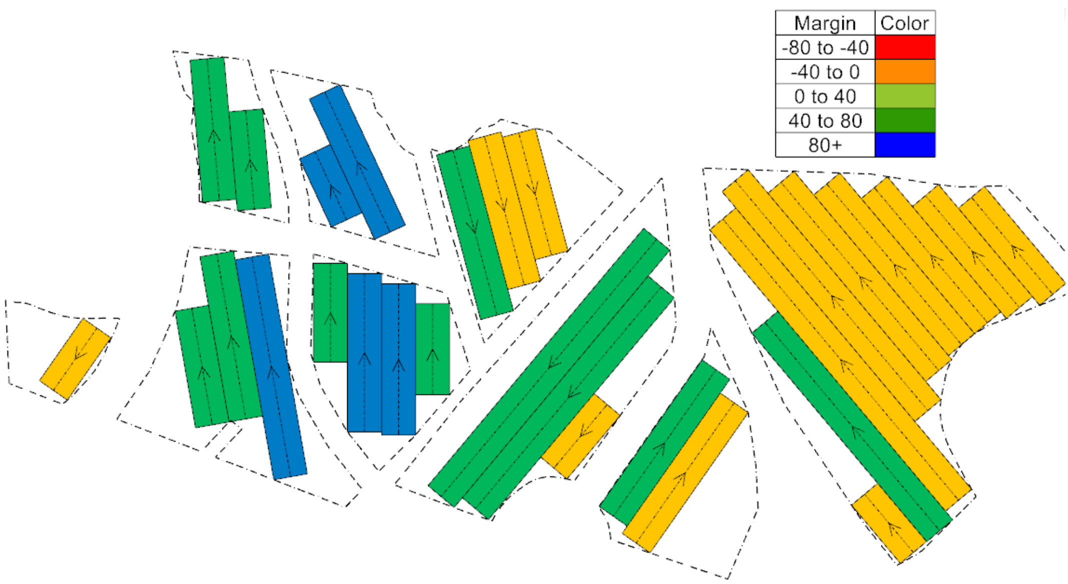

Fig. 4. Location of longwalls for scenario of price 10% higher than base
Source: own study

Rys.4. Lokalizacja ścian wydobywczych dla scenariusza ceny wyższej o 10% od ceny bazowej

Table 6. Results of estimating the size of the resource base of seam 118, NPV and time of extraction in this seam for the MAX variant

Tabela 6. Wyniki szacowania wielkości bazy zasobowej pokładu 118, NPV oraz czas wydobycia w tym pokładzie dla wariantu MAX

Price		Resource base (mln Mg)	NPV (thou. PLN)	NPV/Resources	LLE
SCHEDULE VARIANT (MAX)	Base	19.39	441,536	23	9
	10%	41.60	1,448,941	35	20
	20%	43.79	2,962,342	68	24
	30%	44.10	4,563,711	103	21
	40%	43.53	5,931,199	136	25
	50%	44.86	7,709,968	172	25
	–10%	4.85	45,728	9	3
	–20%	0.00	0	–	0

Source: own study

Table 7 presents the results of the scenario analysis for the resource averaging option. It can be observed that the baseline NPV was approximately PLN 446 million for a resource base of 19.4 million metric tonnes. The maximum NPV reaches PLN 7.8 billion for a resource

Table 7. Results of estimating the resource base of seam 118, NPV and mining time in this seam for the AVG variant

Tabela 7. Wyniki szacowania wielkości bazy zasobowej pokładu 118, NPV oraz czas wydobycia w tym pokładzie dla wariantu uśredniania zasobów

Price		Resource base (mln Mg)	NPV (thou. PLN)	NPV/Resources	LLE
SCHEDULE VARIANT (AVG)	Base	19.39	446,112	23	10
	10%	41.60	1,570,117	38	19
	20%	43.79	3,092,234	71	20
	30%	44.10	4,664,619	106	21
	40%	43.53	6,189,502	142	20
	50%	44.86	7,829,810	175	21
	–10%	4.85	45,153	9	3
	–20%	0.00	0	–	0

Source: own study

base of 44.9 million Mg in seam 118. The maximum mining time in the seam then amounts to 21 years. Interestingly, the NPV of each variant in the scenario with resource averaging is higher than in the scenario with longwall extraction, according to the highest margin. This is the result of the very characteristics of the discounting process and the NPV method. With the highest price value, mining from seam 118 would be 4 years shorter than in the highest-margin parcel-picking scenario, and the amount of the depleted resources would reach 49.9 million Mg.

Conclusions

The paper presents a method for determining the resource base based on market conditions for raw materials. The proprietary software used in the method, which optimizes production scheduling considering market conditions, enables the comparison of thousands of longwall layouts and the identification of the layout with the highest margin.

Based on the calculation example, it can be concluded that a slight change in the price of raw material can significantly affect the amount of the resource base and the raw material available for economically viable extraction. In the calculation example, the resource base varied from 0 to nearly 45 million Mg, with an NPV variation of 0 to 7.8 billion PLN. Moreover, the life of the mine varied from 0 (no resources in the scenario) to a maximum of 25 years, assuming efficient mining with two parallel longwalls. In the baseline scenario, the size of the resource base reached 19.4 million metric tonnes, which was only 42% of the size of the proven reserves in the analyzed seam scheduled for extraction.

The fluctuations in resource base and NPV are huge for the adopted set of base parameter values. This is generally in line with the reasonable conclusions presented in the literature, which suggest that the key criterion for the value of mining investment projects is the price of the resource. However, its high volatility provokes both the question and its answer: how should the base value of this price be determined in the medium- and long-term horizons of the analysis, if it has such a significant impact on the value of mining projects?

The research method presented here, therefore, aligns directly with the requirements of the JORC Code resource evaluation system, offering the possibility of analyzing multiple layouts simultaneously with the use of computer tools.

An essential element of the method is the inclusion in the valuation of the impact of negative geological and mining factors, the materialisation of which may significantly affect the economic efficiency and technical feasibility of the entire mining project. In this regard, the authors proposed an original method of estimating the risk index RI for individual longwalls and parcels using a hierarchical model that analyzes the impact of geological and mining factors and natural hazards on the economic results of the mine (test deposit 'X1'). Using the RI, the cost of equity, a component of the WACC, is then adjusted as an individual assessment of the risks associated with a particular project. On this basis, a base discount rate is then adjusted, which, along with the mining schedule, changes in subsequent years of operation. As a result, the risk load aligns with its materialization over time, according to the deposit's depletion concept.

The method developed is new and has not been known in this form in the literature. It presents the authors' approach to the problem of deposit valuation, drawing on their extensive experience and expertise. Ultimately, it will be fully integrated into a single IT application, enabling a comprehensive assessment of the value of mineral deposits.

The Authors have no conflict of interest to declare.

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METHOD FOR ESTIMATING THE SIZE OF MINEABLE RESOURCE BASE IN THE CONCEPT OF AN EFFECTIVE EXPLOITATION PARCEL (EPE)

Keywords

coal mining, resource estimation,
production planning and optimisation, effective exploitation parcel

Abstract

The paper presents an authorial method of determining the deposit resource base considering changing market conditions. To achieve this goal authors built the dedicated software, which optimizes mining schedules taking into account the expected impact of geological and mining factors and natural hazards on the exploitation process. The research method developed, and the IT tool built, enables the analysis of thousands of variants of cutting the mining plots in order to identify ones that maximize the profit potential of longwalls and parcels. In this method the potential is measured by the value of the operating margin and profit, which include capital expenditure and operating costs of coal production. The impact of geological, mining and natural hazard factors is identified in a multi-criteria method. Then, by estimating the authorial risk index RI, the cost of equity – the weighted component of the mining company's cost of capital – is adjusted.

The results of the presented example show that even small changes in raw material price may significantly affect the amount of recoverable resources and the economic value of the project (NPV). Resource base in the analyzed cases ranges from 0 to 45 million Mg and the NPV from 0 to 7.8 billion PLN. For the base price scenario effective resources amount to 19.4 million Mg, which represents only 42% of the proven recoverable resources. The approach used is in line with the requirements of the JORC Code and enables multivariate analysis of tunnel layouts using advanced digital tools.

METODA SZACOWANIA WIELKOŚCI ZASOBÓW WYDOBYWANYCH W KONCEPCJI EFEKTYWNEJ PARCELI EKSPLOATACYJNEJ (EPE)

Słowa kluczowe

wydobycie węgla, szacowanie zasobów, planowanie i optymalizacja produkcji, efektywna parcela eksploatacyjna

Streszczenie

W artykule zaprezentowano autorską metodę określania wielkości bazy zasobowej złoża w zależności od zmiennych warunków rynkowych. Do realizacji tego celu zastosowano dedykowane oprogramowanie do optymalizacji harmonogramów produkcji górniczej z uwzględnieniem oczekiwanego wpływu czynników geologicznych, górniczych i zagrożeń na wydobycie węgla kamiennego. Opracowana metoda badawcza i zbudowane narzędzie informatyczne umożliwiają analizę tysięcy wariantów rozcinki złoża i parcel eksploatacyjnych celem identyfikacji wariantów zagospodarowania maksymalizujących potencjał dochodowy ścian i parcel. W niniejszej metodzie etapie potencjał ten jest mierzony wartością marży operacyjnej i zysku szacowanego po uwzględnieniu nakładów inwestycyjnych i kosztów operacyjnych produkcji węgla. Wpływ czynników geologicznych, górniczych i zagrożeń naturalnych jest identyfikowany w metodzie wielokryterialnej, a następnie poprzez szacowanie autorskiego indeksu ryzyka RI korygowany jest koszt kapitałów własnych – składnik ważonego kosztu kapitału przedsiębiorstwa górniczego.

Wyniki przykładu obliczeniowego wskazują, że nawet niewielkie zmiany cen surowca mogą znacząco wpływać na opłacalność eksploatacji, skalę zasobów możliwych do pozyskania oraz wartość ekonomiczną projektu (NPV), przy czym w analizowanym przypadku baza zasobowa wahała się od 0 do 45 mln Mg, a NPV od 0 do 7,8 mld PLN. Dla scenariusza bazowego oszacowana wielkość zasobów efektywnych ekonomicznie sięgnęła 19,4 mln Mg, co stanowi zaledwie 42% zasobów udokumentowanych w kategorii zasobów operatywnych (wydobywalnych). Zastosowane podejście wpisuje się w wymagania systemu klasyfikacji zasobów JORC Code i umożliwia wielowariantową analizę geometrii wyrobisk z wykorzystaniem zaawansowanych narzędzi cyfrowych.