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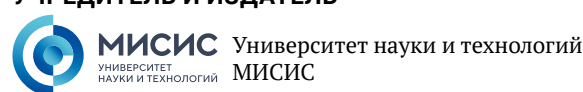
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GEOLOGY OF MINERAL DEPOSITS

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**UAV-based aeromagnetic surveying for exploration targeting at the Kargoba copper-molybdenum occurrence**

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Abstract

The Kargoba mineral occurrence in East Kazakhstan covers an area of 25.5 km² and is characterized by rugged mountainous terrain at elevations of 800–1,013 m above sea level. This study presents a methodology for high-resolution unmanned aerial vehicle (UAV)-based aeromagnetic surveying aimed at targeting copper-molybdenum mineralization. The study sought to identify geophysical targeting criteria and refine the boundaries of prospective zones. The survey was conducted using a DJI Matrice 300 RTK quadcopter equipped with a Geoshark MG30GM quantum magnetometer. Magnetic-field interpretation included derivative-based transforms, namely the tilt derivative (TDR), first vertical derivative (VDR), and total horizontal derivative (THDR), as well as three-dimensional modeling of magnetic sources. Six anomalous zones were delineated. Zone A-2 was identified as the highest-priority exploration target, exhibiting an anomaly amplitude of up to 500 nT and a spatial association with a granodiorite intrusion. The following criteria were established for targeting porphyry mineralization: anomaly amplitudes of 200–500 nT, horizontal gradients exceeding 3 nT/m, anomaly strike lengths of 0.5–2 km, spatial association of positive and negative magnetic anomalies, and spatial correlation with hydrothermal alteration zones. Comparison of the identified anomalies with published data from the Lavrion district in Greece supported the applicability of the selected approach. A key diagnostic feature is the spatial association of anomalies of opposite polarity, reflecting the structural and geological controls on mineralization. The high spatial resolution of the aeromagnetic survey at Kargoba enables more precise delineation of prospective mineralized zones and more effective positioning of exploration drillholes. The use of a UAV considerably facilitates and accelerates surveying in rugged and inaccessible terrain. Compared with conventional ground surveys and crewed airborne surveys, it reduces survey time and costs while maintaining high data quality. The proposed approach therefore offers a cost-effective, lower-risk means of targeting concealed mineralized bodies and extends the capabilities of mineral exploration in remote regions.

Keywords

porphyry mineralization, mineral exploration, exploration targeting, copper-molybdenum occurrence, magnetic anomaly, aeromagnetic survey, unmanned aerial vehicles (UAVs), East Kazakhstan

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ГЕОЛОГИЯ МЕСТОРОЖДЕНИЙ ПОЛЕЗНЫХ ИСКОПАЕМЫХ

Научная статья

Аэромагнитная разведка с применением беспилотного летательного аппарата при прогнозировании медно-молибденового рудопроявления Каргоба

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Аннотация

Участок рудопроявления Каргоба (Восточный Казахстан) площадью 25,5 км² характеризуется сильно расчленённым горным рельефом с абсолютными отметками 800–1013 м. В работе представлена методика высокоточной аэромагнитной съёмки с использованием беспилотного летательного аппарата для прогноза медно-молибденового оруденения. Цель исследования – выявление геофизических критериев рудоконтроля и уточнение границ перспективных зон. Съёмка выполнялась квадрокоптером DJI Matrice 300 RTK с квантовым магнитометром Geoshark MG30GM, проведена 3D-интерпретация магнитного поля с применением производных трансформант (TDR, VDR, THDR). В результате выделено шесть аномальных зон, из которых наиболее перспективной является зона А2 с амплитудой до 500 нТл и пространственной связью с гранодиоритовой интрузией. Уточнены критерии прогнозирования порфирового оруденения: амплитуда 200–500 нТл, горизонтальные градиенты свыше 3 нТл/м, протяжённость аномальных зон 0,5–2 км, сопряжённость положительных и отрицательных аномалий и их корреляция с зонами гидротермальных изменений. Сравнение выявленных аномалий с данными зарубежных исследований района Лаврион (Греция) подтвердило правильность выбранного метода. Ключевой особенностью здесь выступает сочетание разнознаковых аномалий, которое отражает структурно-геологические особенности контроля оруденения. Высокое разрешение магнитной съёмки на участке Каргоба позволяет точно локализовать рудные зоны и оптимизировать расположение разведочных скважин. Применение БПЛА значительно упрощает и ускоряет обследование труднодоступного горного района: сокращаются сроки выполнения магнитной съёмки и снижается её стоимость по сравнению с традиционными наземными маршрутами и пилотируемой авиацией, при этом обеспечивается сопоставимое или более высокое качество данных. Таким образом, предложенный подход позволяет решать задачи поисков скрытых рудных тел с меньшими затратами и рисками, расширяя возможности геологоразведки в удалённых районах.

Ключевые слова

порфировое месторождение, геологоразведка, прогнозирование рудных тел, медно-молибденовое рудопроявление, магнитная аномалия, аэромагнитная съёмка, беспилотные летательные аппараты (БПЛА), Восточный Казахстан

Финансирование

Статья выполнена в рамках грантового финансирования по научным и (или) научно-техническим проектам на 2024–2026 годы (Министерство науки и высшего образования Республики Казахстан) № AP23484205 «Анализ спектральных характеристик спутниковых и наземных данных с целью локализации поиска редкометалльного оруденения в пределах Центральной Калбы».

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Orazbekova G.B., Mataibayeva I.E., Zimanovskaya N.A., Ragdanova A.A., Sarsembayeva A.S. UAV-based aeromagnetic surveying for exploration targeting at the Kargoba copper-molybdenum occurrence. *Mining Science and Technology (Russia)*. 2026;11(2):107–121. <https://doi.org/10.17073/2500-0632-2025-06-993>



Introduction

The effective prediction of concealed orebodies in remote and inaccessible areas remains an important challenge in modern mineral exploration [1]. Conventional aeromagnetic surveys conducted using crewed aircraft have several major limitations, including high costs, complex logistics, and insufficient spatial resolution in areas of rugged terrain. These limitations can be addressed through the use of unmanned aerial vehicles (UAVs) equipped with high-sensitivity magnetometers [2]. In recent years, this approach has been increasingly adopted, substantially improving the accuracy and spatial resolution of aeromagnetic surveys [3–5].

Despite the clear advantages of UAVs, their use presents several methodological and technical challenges, including terrain-following flight-path planning, electromagnetic interference generated by the UAV itself, accurate positioning, and maintaining the specified flight altitude [4]. Furthermore, the interpretation of high-resolution magnetic data requires algorithms and software capable of identifying and reliably interpreting anomalies associated with different geological sources [6, 7].

Similar spatial analysis and mineral-resource prediction problems have also been successfully addressed for other types of mineral resources using computer modeling. For example, a GIS-based modeling methodology was developed for common non-metallic minerals hosted in the Cenozoic deposits of the Brest Region. The methodology can be used to predict mineral deposits and assess their development potential with due consideration of geological and land-use factors [8].

Alongside advances in geophysical exploration methods, automated mineralogical analysis techniques are also being actively developed. In particular, convolutional neural network-based approaches have been proposed for segmenting ore minerals in digital images of polished sections, enabling highly accurate identification of mineral phases [9]. These developments reflect a broader transition in mineral exploration toward quantitative, reproducible, and automated data-processing methods.

Thus, the aim of this study is to develop and test an integrated methodology combining high-resolution UAV-based aeromagnetic surveying, advanced data processing and interpretation techniques, and the geological characteristics of the Kargoba mineral occurrence.

To achieve this aim, the following tasks were addressed: UAV-based aeromagnetic surveying of the study area; magnetic data processing and transformation; construction of a 3D magnetization model to a depth of 500 m; delineation and classification of anomalous zones; and identification of priority targets for further exploration.

Methodology

Aeromagnetic surveying is based on measuring variations in the Earth's magnetic field caused by contrasts in the magnetic susceptibility of rocks. Under potential-field theory, anomaly sources can be represented by equivalent source bodies in three-dimensional space, enabling filtering and inversion methods to be used to reconstruct the source structure¹.

The method is particularly suitable for areas characterized by extensive magmatism and ore-related intrusions, such as the Kargoba area. Porphyry-system models indicate that hydrothermal-metasomatic alteration zones near the apices of intrusions display a marked geophysical contrast with the surrounding rocks and can therefore be effectively delineated by aeromagnetic surveying [10].

UAV-based aeromagnetic surveying was selected to obtain high-resolution magnetic-field data over rugged terrain that is difficult to cover using conventional crewed airborne surveys [11].

The use of a quantum magnetometer and magnetic-data processing techniques, including reduction to the pole, the first vertical derivative, the total horizontal derivative, and the tilt derivative (TDR), followed procedures previously applied in international studies [12–14].

The aeromagnetic data were processed in two stages. Initial processing and assessment of field-data quality were performed on site, while final data processing was completed after the field survey. Geosoft Oasis montaj² was used for data processing and routine quality control.

The contribution of this study is the adaptation of the established methodology to the mountainous terrain of East Kazakhstan, modifying an interpretive tomography algorithm used to construct the 3D model in Voxler to account for the dimensions and acquisition parameters of the survey block, and establishing criteria for identifying anomalies considered prospective for mineralization by comparison with known deposits of a similar type.

Equipment and survey procedure

A DJI Matrice 300 RTK multirotor UAV served as the airborne platform for the aeromagnetic survey³.

¹ Report on supplementary geological investigations and inferred resource assessment for map sheets L-44-V, XI, and XVII (Kazakhstan sector). Kapchagai Geological Exploration Expedition LLP, Almaty; 2016. Principal investigators: A. V. Dubinkin, A. K. Akhmatolla, V. V. Podkovyrov et al.

² Seequent. Oasis montaj. URL: <https://www.seequent.com/products-solutions/oasis-montaj/>

³ DJI. Matrice 300 RTK. URL: <https://www.dji.com/support/product/matrice-300>

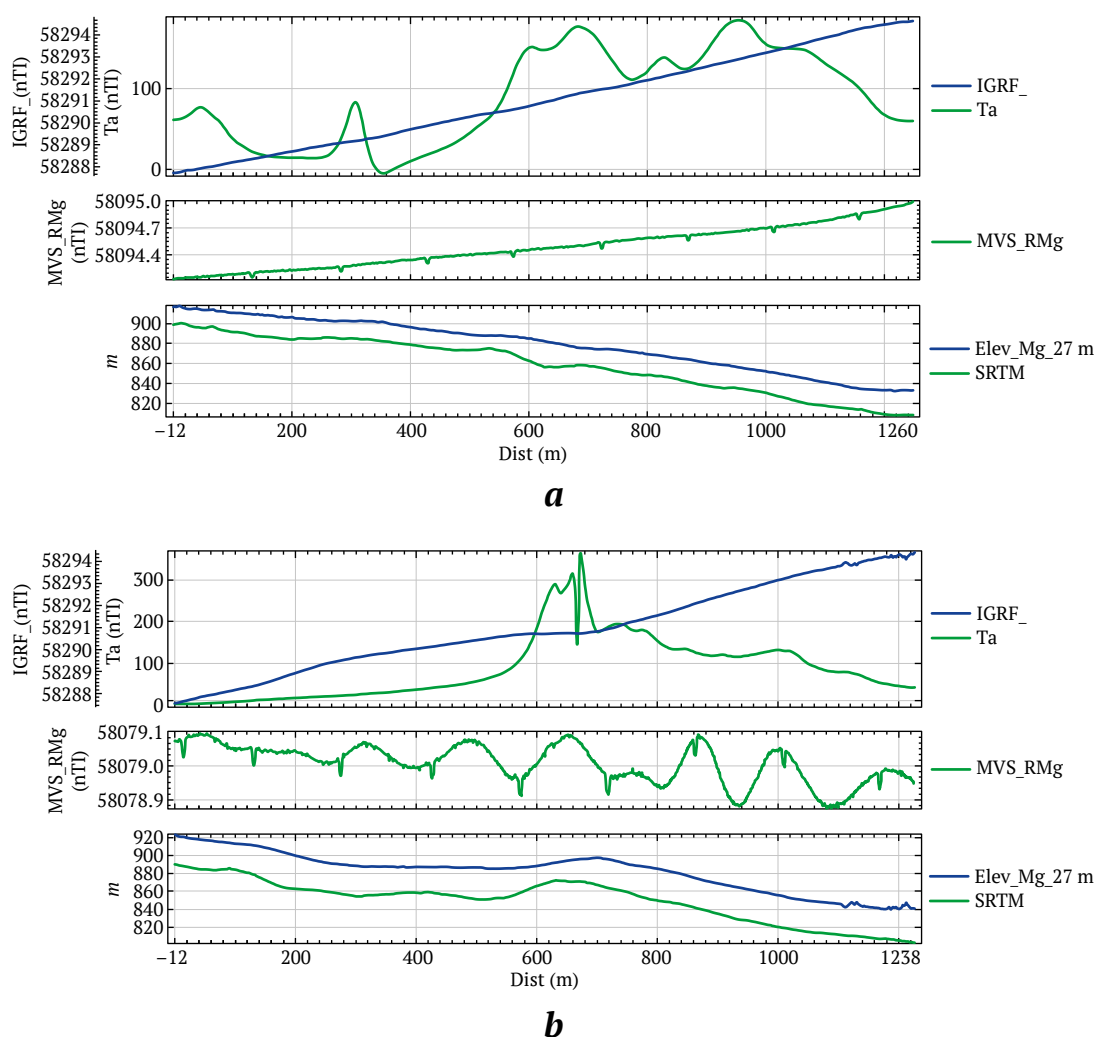


Fig. 1. Example visualization of aeromagnetic survey database elements for lines L1034 (a) and L1042 (b)

UgCS Pro software⁴ was used for flight planning, UAV control and monitoring, and the real-time display of telemetry data transmitted by the UAV.

Magnetic measurements were acquired with a Geoshark MG30GM rubidium quantum magnetometer mounted on the UAV [15].

Initial data processing was performed concurrently with data acquisition, on a daily basis after the field crew returned to the field base. Data from the airborne magnetometer and the magnetic base station were transferred to the processing computer and imported into Geosoft Oasis montaj databases. The data were edited by removing the run-in segments preceding each survey line and the turns between adjacent lines. The completeness and quality of the raw data were then visually checked, and a preliminary magnetic-field map was progressively updated as new data became available.

For visual quality control of the data acquired along each survey line, Geosoft Oasis montaj was used to display profiles of the measured total magnetic field (Ta), the International Geomagnetic Reference Field (IGRF) calculated at the magnetometer elevation, temporal variations in the geomagnetic field recorded at the magnetic base station (MVS_RMg), ground elevation derived from SRTM data, and magnetometer elevation (Elev_Mg_27 m). An example of these database profiles is shown in Fig. 1.

Aeromagnetic survey error was evaluated from differences between repeat-line and survey-line measurements and from crossover differences at intersections between survey lines and tie lines. The results are presented in Table 1.

Repeat-line measurements covered 14.9 line km, accounting for 6.2% of the combined length of the survey and tie lines, compared with the 5% specified in the technical requirements. Based on the differences between repeat-line and survey-line measurements, the estimated error of the Kargoba aeromagnetic survey

⁴ SPH Engineering. UgCS User Manual. URL: <https://manuals.ugcs.com/>



was ± 2.99 nT. The maximum permissible error specified in the technical requirements was ± 5 nT, increasing to ± 7 nT in areas of high magnetic-field gradients. Fig. 2 shows two representative profiles. Line L1036 displays a positive magnetic anomaly with an amplitude of approximately 200 nT, whereas line L1087 displays a high-amplitude anomaly of up to 500 nT.

Aeromagnetic survey error at crossover points between tie lines and survey lines was estimated for locations where the horizontal magnetic-field gradient was below 50 nT/km using the root-mean-square error formula:

$$m_1 = \pm \sqrt{\frac{\sum_{i=2}^n \delta_i^2}{2(n-1)}},$$

where δ_i is the difference between the final ΔT readings at the crossover points, nT; n is the number of δ differences.

Table 1
Aeromagnetic survey error estimated from repeat-line measurements

No.	Line	Line length, m	Mean difference, nT	Standard deviation, nT
1	L1036:0	1,231	0.24	2.23
2	L1087.2:0	1,844	2.04	4.53
3	L1092.2:0	2,122	-0.28	1.00
4	L1093.2:0	2,236	0.37	0.38
5	L1101.2:0	1,450	0.27	0.44
6	L1113:0	1,761	-3.34	5.48
7	L1122:0	2,145	-0.23	0.94
8	L1123:0	2,113	0.68	1.52
Total		14,903	0	2.99

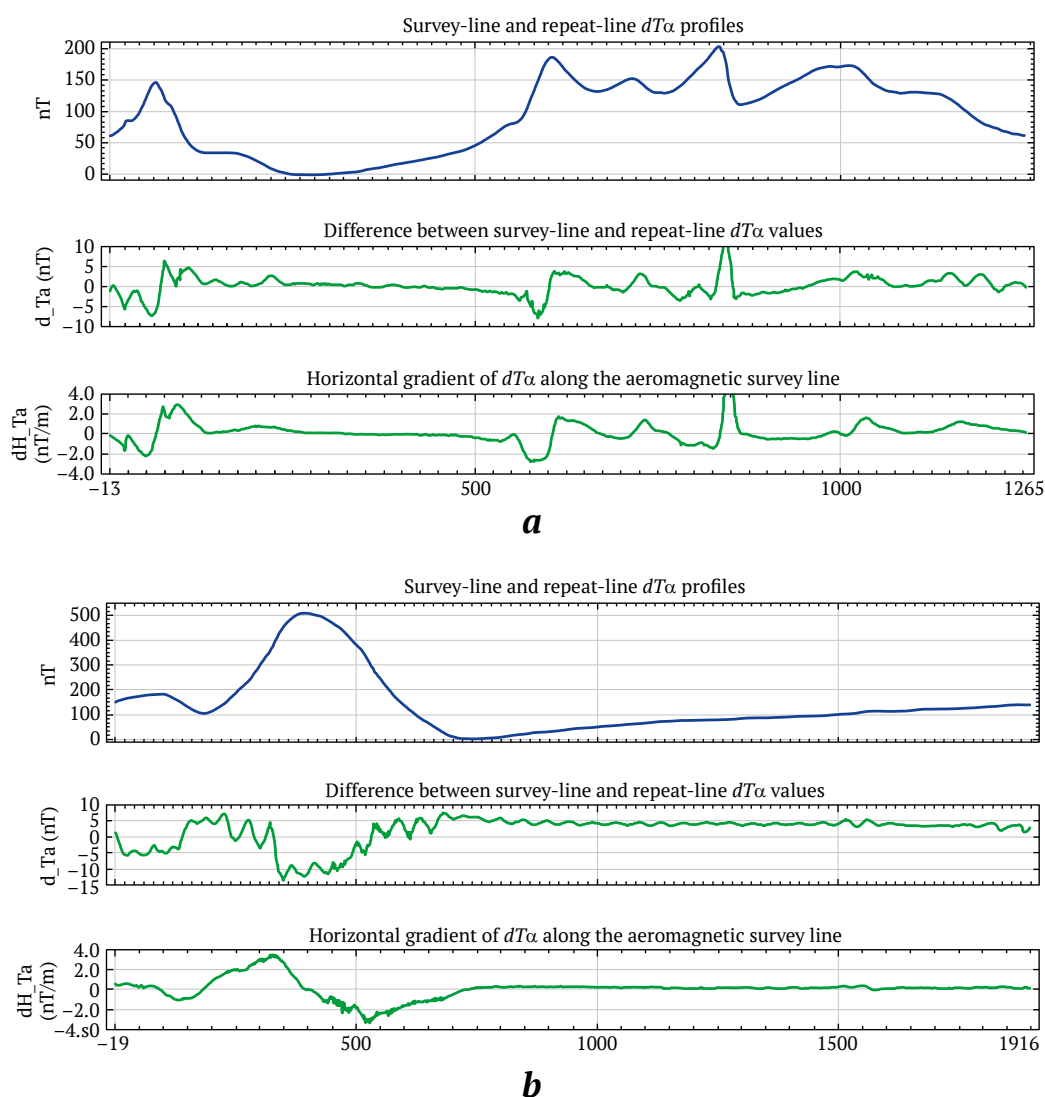


Fig. 2. Comparison of survey-line and repeat-line magnetic-field measurements for two representative profiles: L1036, showing a positive anomaly of moderate amplitude (a); and L1087, showing a high-amplitude anomaly (b)

Analysis of the magnetic-field values at 485 crossover points yielded a root-mean-square error of ± 3.46 nT.

Following final data processing, the complete magnetometric database was compiled, and grids of the anomalous magnetic field and its most informative transforms were calculated.

Maps of the magnetic field and its most informative transforms are presented in Figs. 3–9.

Fig. 3 shows the anomalous magnetic-field model for the Kargoba area as a contour map generated using Surfer software.

In accordance with the applicable mapping specifications, the map uses shades of red for negative anomalous magnetic-field values and shades of blue for positive values.

A profile map showing the amplitude and variation of ΔT along the survey lines was used for the initial assessment of data quality and the spatial distribution of the anomalous magnetic field. Analysis of the anomaly pattern identified zones with pronounced local peaks, potentially corresponding to geological bodies with strong magnetic susceptibility contrasts. Comparison of these zones with the magnetic-field transforms shown in Figs. 4–8 confirmed the presence of buried magnetic sources associated with intrusive bodies.

The anomalous magnetic field was also displayed using the Image Maps raster mode in Geosoft, pro-

viding a clearer representation of the overall anomaly morphology and its distribution across the survey area. At another stage of data processing, a Shaded Relief display was applied to better define the morphology of the major anomalies.

Before detailed interpretation, the anomalous magnetic field over the Kargoba area was reduced to the pole (see Fig. 4). Reduction to the pole transforms the observed magnetic field into the field that would theoretically be measured at the geomagnetic pole, where both the Earth's magnetic-field vector and the magnetization of the source rocks are vertical. In theory, the maxima and minima of reduced-to-the-pole anomalies generated by subsurface sources should be centered over the causative bodies, as is generally the case for gravity anomalies. The magnetic anomalies should also become approximately monopolar, with their associated opposite-polarity lobes largely suppressed, thereby providing a clearer overall anomaly pattern.

Reduction to the pole, like most other magnetic-field transformations, was performed using Geosoft Oasis montaj™. The inclination and declination used in the calculation were 68.65° and 5.21° , respectively.

The morphology of the anomalous magnetic field indicates that its sources occur at different structural levels. Accordingly, one of the principal interpretation tasks is to separate the magnetic-field contributions of sources located at different depths.

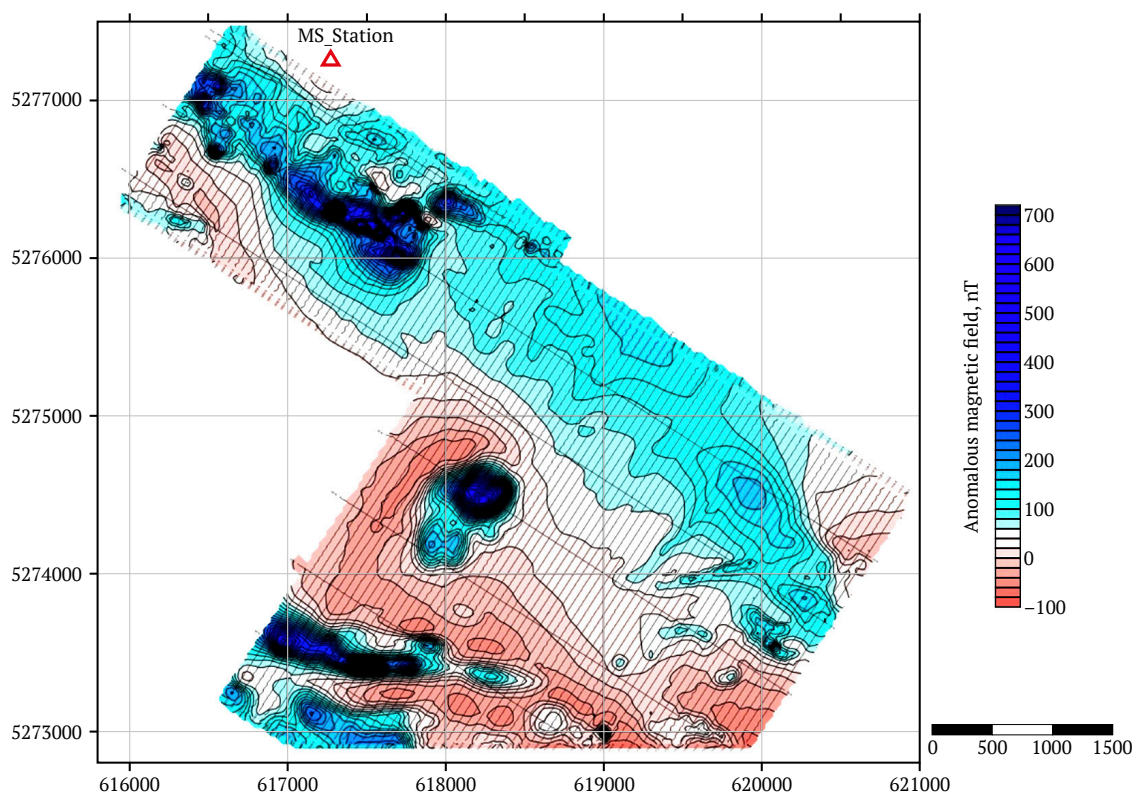


Fig. 3. Contour map of the anomalous magnetic field

The high-frequency component of the magnetic field is most clearly expressed in the first vertical derivative (VDR) (see Fig. 5). The vertical-derivative transform enhances the high-frequency component of the magnetic-field spectrum, emphasizes features associated with relatively shallow local sources, and suppresses the low-frequency component. Compared with the original field, the first vertical derivative provides sharper delineation of the upper edges of magnetically active bodies.

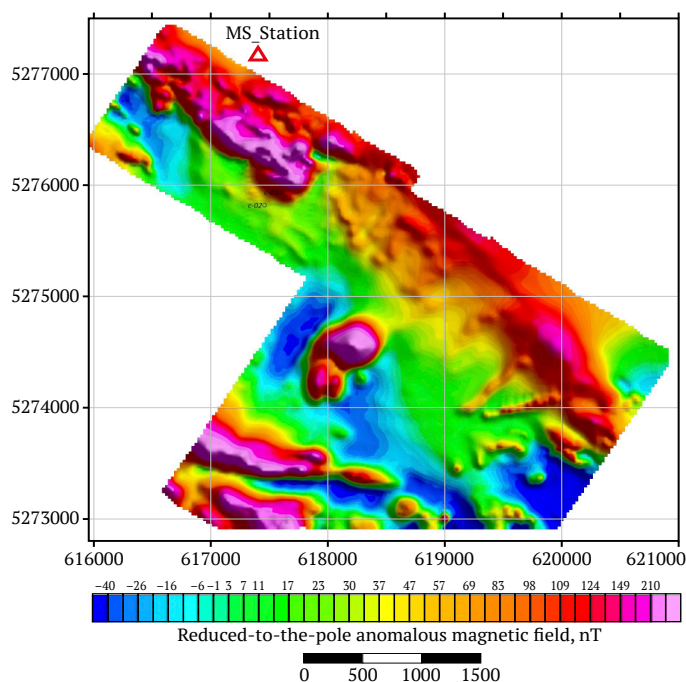


Fig. 4. Reduced-to-the-pole anomalous magnetic field

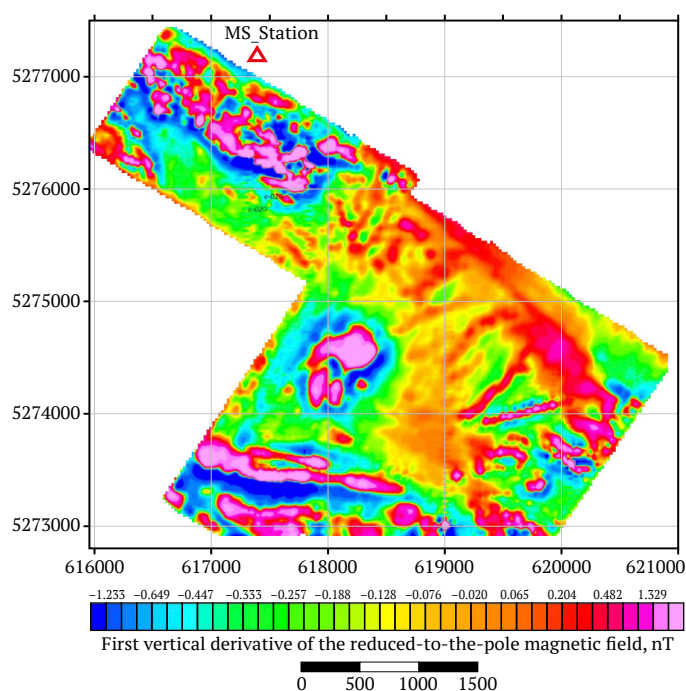


Fig. 5. First vertical derivative of the reduced-to-the-pole magnetic field

More accurate delineation of the boundaries of magnetic sources can be achieved by analyzing the distribution of the total horizontal derivative (*THDR*). Magnetic-field gradients are free from the linear background associated with large regional anomalies and the normal regional gradient. In the *THDR* field, the boundaries of anomalies with different amplitudes are marked by extrema, allowing the outlines of both weak and strong anomalies to be visualized simultaneously. Linear zones of maximum *THDR* values delineate anomalous areas, trace steps and inflection lines in the original magnetic field that can be correlated with extensive geological contacts and faults, and emphasize the boundaries of tectonic blocks⁵.

For comparison with the vertical components of the anomalies, the total horizontal derivative of the magnetic field was also calculated. This transform provided a more accurate estimate of the lateral boundaries of the sources; however, it proved less informative than the first vertical derivative (see Fig. 5) and the analytic signal (see Fig. 6).

Fig. 6 shows a map of the analytic signal amplitude of the magnetic field.

The analytic signal amplitude (*AS*) of the anomalous magnetic field is defined as:

$$AS = \sqrt{VDR^2 + THDR^2},$$

where *VDR* is the first vertical derivative and *THDR* is the total horizontal derivative of the magnetic field.

The analytic signal amplitude is useful for delineating magnetic-source boundaries and is independent of both the dip of the anomaly-producing geological bodies and the latitude of the survey area. Like the total horizontal derivative, the analytic signal can be used to trace contacts between magnetic units, refine source locations, and delineate the upper edges of magnetic bodies.

The Tilt derivative (*TDR*) is widely used as an alternative to the first vertical derivative (*VDR*) for enhancing fine-scale features of the magnetic field:

$$TDR = \tan^{-1}[VDR / THDR].$$

The tilt angle is expressed in radians. *TDR* extrema have nearly equal amplitudes of approximately ± 1.5 and clearly identify the maxima and minima of the original field regardless of their intensity. The *TDR* enables reliable mapping of the positions and edges of source bodies and facilitates the tracing of structural features within the survey area. *TDR* extrema correspond to the axes of the source bodies, whereas zero

⁵ Shadrintsev M.P., et al. Report on the results of reconnaissance geological and geophysical surveys conducted in the Tarbagatai area in 1974–1975 (map sheets L-44-9-G; L-21-B-b, g; L-44-22-A-a, v). Altai Integrated Geological and Geophysical Expedition, Opytnoye Pole, May 1976.

values mark their boundaries. The *TDR* is considerably less dependent on the amplitude of the original-field anomalies than the first vertical derivative, total horizontal derivative, and analytic signal. It therefore provides clearer delineation of the axes of weak anomalies associated with deeper sources. The *TDR* is comparable to a residual-anomaly field and enhances the fine structure of the magnetic field.

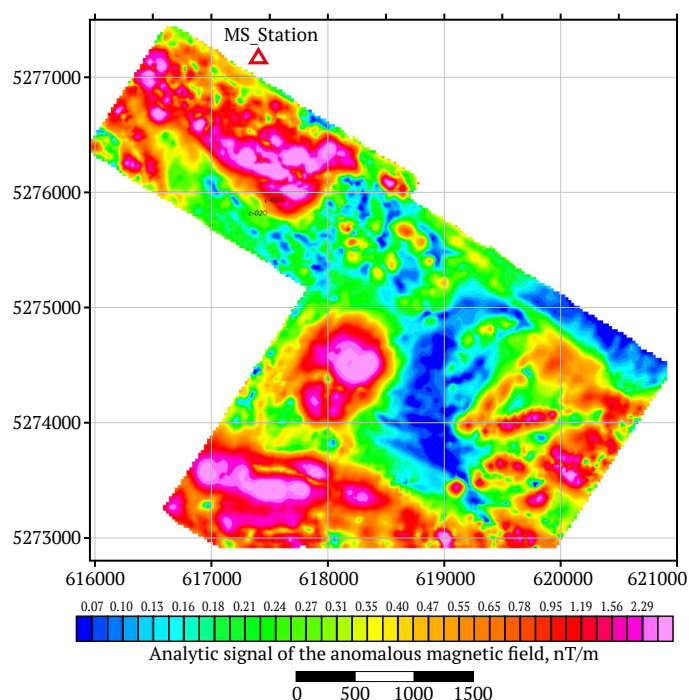


Fig. 6. Analytic signal of the anomalous magnetic field

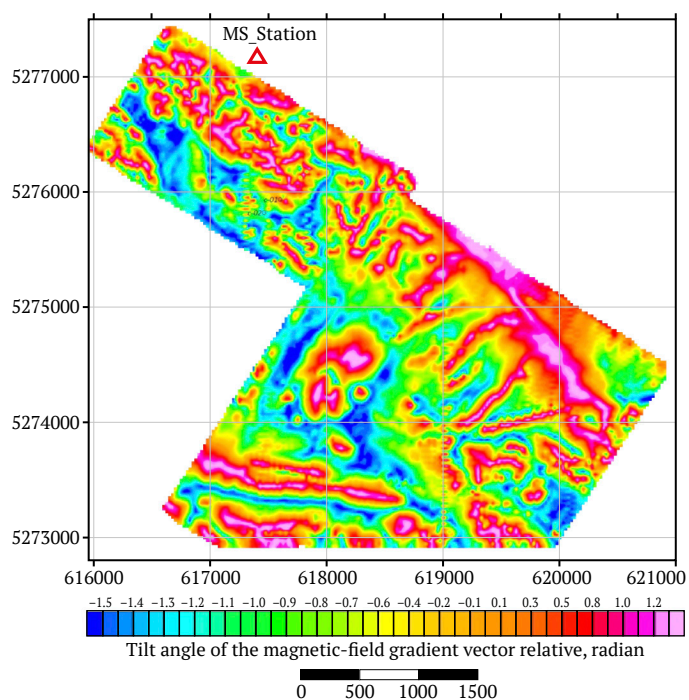


Fig. 7. Tilt angle of the magnetic-field gradient vector relative to the horizontal

The tilt-derivative map of the Kargoba area is shown in Fig. 7. The total horizontal derivative of the tilt angle (*TDR-THDR*) enhances and traces the edges of magnetic sources. This transform is not affected by ringing, which can generate spurious maxima, and therefore does not produce false edges. The amplitudes of the *TDR-THDR* maxima can be used to estimate the depths of source edges. The axes of zones with maximum *TDR-THDR* values trace the boundaries between bodies with contrasting magnetic properties.

To examine the three-dimensional distribution of magnetic properties in the Kargoba area, 3D modeling was performed using an interpretive tomography algorithm. A model representing the equivalent distribution of magnetic sources was calculated using specialized filtering procedures applied to the original magnetic field. These procedures are implemented in the *Express gravity-magnetic modeling and inversion* module of the *IP Seismic* software package. The method is classified as interpretive tomography and does not require prior information about the target⁶.

The 3D magnetic model of the Kargoba area was calculated over a depth interval of 0–1,000 m, with elementary cell dimensions $dX = 25$ m, $dY = 25$ m, and $dZ = 10$ m. The resulting model, initially generated in SEG-Y format, was exported as an ASCII text table and imported into the Geosoft Oasis montaj™ database. Depths were converted to absolute elevations. The 3D grid was generated using the 3D Griding/Kriging algorithm, and the model was visualized in Geosoft 3D. The cell dimensions of the Voxler 3D model of excess magnetization in the Kargoba area were $25 \times 25 \times 10$ m.

Elements of the 3D magnetization model for the aeromagnetic survey area covering the Kargoba mineral occurrence are shown in Figs. 8 and 9.

Magnetization values are expressed in arbitrary units. Higher values are shown in yellow, red, and pink, whereas lower values are shown in shades ranging from green and light blue to dark blue.

A schematic geological map of the Kargoba area at a scale of 1:25,000⁷ was used as the geological basis for interpreting the magnetic anomalies.

The schematic geological map of the Kargoba mineral occurrence is shown in Fig. 10.

⁶ Shadrintsev M.P., et al. Report on the results of reconnaissance geological and geophysical surveys conducted in the Tarbagatai area in 1974–1975 (map sheets L-44-9-G; L-21-B-b, g; L-44-22-A-a, v). Altai Integrated Geological and Geophysical Expedition, Opytnoye Pole, May 1976.

⁷ Report on supplementary geological investigations and inferred resource assessment for map sheets L-44-V, XI, and XVII (Kazakhstan sector). Kapchagai Geological Exploration Expedition LLP, Almaty; 2016. Principal investigators: A.V. Dubinkin, A.K. Akhmatolla, V.V. Podkovyrov et al.

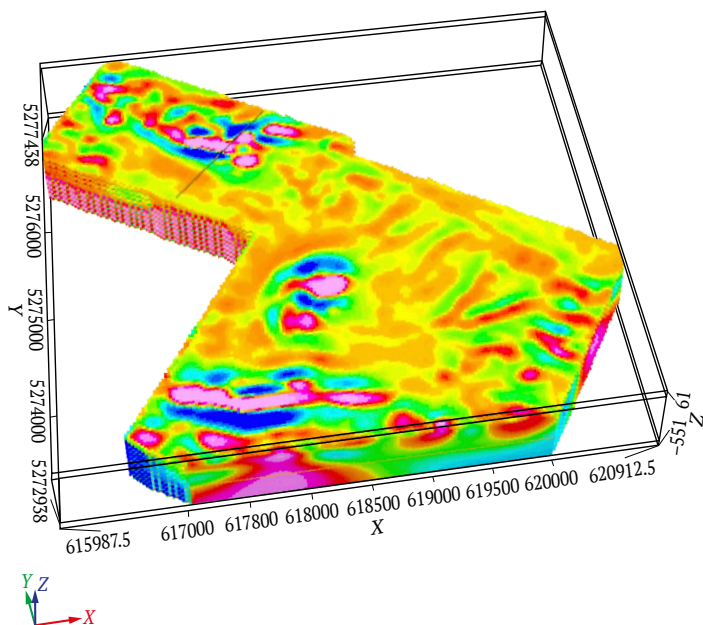


Fig. 8. General view of the 3D magnetization model at depths of 0–500 m

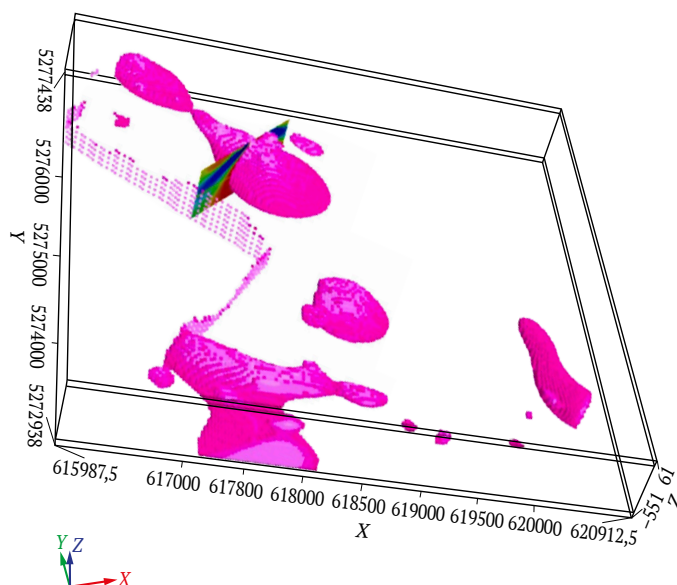


Fig. 9. Zones of increased magnetization, ($F > 11.4$) a.u.

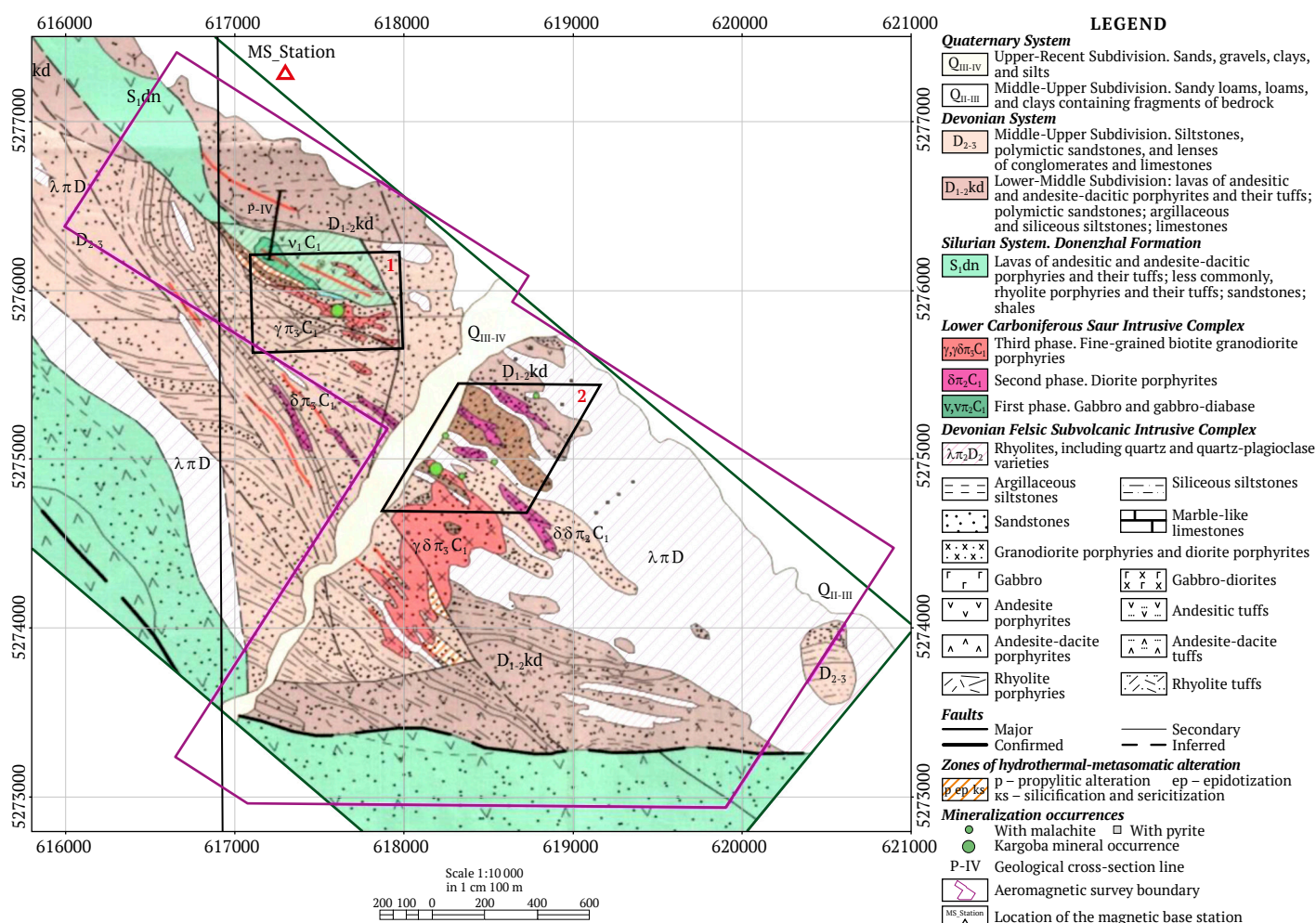


Fig. 10. Schematic geological map of the Kargoba mineral occurrence area

Source: Alimkhanov N. K., Bashkirtsev A. M., et al. Report on the project “Exploration of Prospective Areas in the Eastern Region”, 2005–2007. GRK Topaz LLP, Ust-Kamenogorsk, December 2007



Results and discussion

Unlike conventional ground magnetic surveying, UAV-based surveys provide a denser and more uniform observation grid, particularly in remote and poorly accessible areas. This substantially improves mapping resolution and the reliability of spatial analysis of magnetic sources. For example, in conventional surveys with a station spacing of 50–100 m, the detection of small, thin local bodies at depths of up to 150 m may be hindered by spatial smoothing of the signal and terrain effects [16, 17].

The combined use of high-frequency transforms, such as the tilt derivative (TDR) and analytic signal, with UAV magnetic data provides a spatial resolution of 25–30 m, comparable to the drill spacing used during resource evaluation. Furthermore, unlike conventional inversion approaches involving upward continuation, the proposed method employs a regularized inverse solution adapted to the near-field conditions characteristic of low-altitude measurements at flight heights of 35–50 m.

Previous studies of UAV applications in porphyry systems, including those conducted in Peru and Central Asia [18], have focused primarily on overall mapping accuracy or the detection of large-scale structures. By contrast, the present study demonstrates the potential to identify concealed, localized bodies of possible economic significance through integrated analysis of magnetic-field transforms, 3D modeling, and geological correlation. The study therefore contributes to the development of methods for local depth interpretation of UAV aeromagnetic data.

The effectiveness of the proposed approach was evaluated at the Kargoba mineral occurrence, one of the less extensively studied sites in East Kazakhstan exhibiting indications of porphyry mineralization. To test the methodology under actual geological and structural conditions, previously available data were analyzed together with the newly acquired aeromagnetic data.

The Kargoba mineral occurrence was discovered in 1957. Aerogeophysical Team No. 31 of the Chingiz Division of the Volkov Expedition identified two areas of copper mineralization represented by pyrite, chalcopyrite, malachite, and azurite in bleached sandstones, granodiorite porphyries, and quartz porphyries. Trenches were excavated, and four boreholes up to 100 m deep were drilled. The boreholes intersected several mineralized intervals containing 0.1–0.5% Cu at depths of 12–92 m⁸.

In 1974–1975, the Tarbagatai Team of the Altai Integrated Geological and Geophysical Expedition⁹ conducted a comprehensive exploration program to investigate and evaluate the mineral occurrence in greater detail. The program included induced-polarization and self-potential electrical surveying on a 250×25 m grid, a soil geochemical survey on the same grid, trenching and test-pitting, 890 m of shallow drilling for geological mapping, and two exploration boreholes up to 300 m deep. The work established a veinlet-disseminated copper-molybdenum style of mineralization and its genetic association with a granodiorite intrusion.

Further investigations of the Kargoba occurrence were conducted in 2005–2007¹⁰ over an area of 25.5 km². The work was intended to refine the geological structure, examine the controls on mineralization, trace the mineralized zones northwestward and southeastward, and explore and evaluate the right-bank part of the occurrence. It included reconnaissance geological traverses, magnetic surveying on a 1,000–500×50 m grid, 467 m³ of trenching, and geochemical exploration based on primary dispersion halos, including bedrock sampling along reconnaissance traverses and geological-geochemical profiles, with a total of 773 samples collected.

The combined geological, geochemical, and geophysical evidence indicates the potential for economically significant mineralization on both the left and right banks. At least four orebodies with high copper and molybdenum grades have been delineated at the surface and to depths of up to 160 m on the left bank. The present erosion level is interpreted to expose only the uppermost part of the mineralized system. Average grades and orebody thicknesses may therefore increase with depth, and the individual orebodies may merge into a single body at deeper levels.

Exploration on the right bank of the Kargoba River delineated a major zone of silicification comprising pyrite-bearing quartz-sericite metasomatic rocks. The zone can be traced southeastward from the river for approximately 1 km and is 0.5–0.7 km wide. High concentrations of ore-forming elements occur within this zone in the area of granodiorite-porphyry outcrops.

There is therefore a high probability of identifying a porphyry copper mineralized system within an area of almost 1 km² on the right bank, potentially comparable in scale and resources to that on the left bank.

The exploration program at the Kargoba copper-molybdenum occurrence delineated a highly prospective porphyry copper target recommended for follow-up prospecting, evaluation, and exploration.

⁸ Shadrinsev M.P., et al. Report on the results of reconnaissance geological and geophysical surveys conducted in the Tarbagatai area in 1974–1975 (map sheets L-44-9-G; L-21-B-b, g; L-44-22-A-a, v). Altai Integrated Geological and Geophysical Expedition, Opytnoye Pole, May 1976.

⁹ Ibid.

¹⁰ Alimkhanov N.K., Bashkirtsev A.M., et al. Report on the project “Exploration of Prospective Areas in the Eastern Region”, 2005–2007. GRK Topaz LLP, Ust-Kamenogorsk, December 2007.

The available data indicate that the granodiorite bodies are the principal source associated with copper-molybdenum mineralization. Their emplacement resulted in the formation of extensive zones of hydrothermal-metasomatic alteration containing ore mineralization. A belt of epidotized and silicified andesite and andesite-dacite lavas was also identified along the contact with the granite intrusion. The belt extends for 750–800 m in a near north-south direction and ranges from 10 to 80 m in width. Copper mineralization, represented by malachite along fractures in the rocks, is visible within this zone.

High molybdenum concentrations have also been recorded, and silver and gold may occur in potentially economic concentrations.

Despite the relatively small size of the survey area, its anomalous magnetic field is strongly differentiated. The lowest magnetic-field values coincide spatially with Middle-Upper Devonian siltstones and polymictic sandstones occupying the central part of the survey area and extending northwestward as a broad belt.

Devonian quartz and quartz-plagioclase rhyolite porphyries of the felsic subvolcanic intrusive complex, which are most widespread in the north-eastern part of the study area, are expressed as

a relatively low-gradient magnetic field of elevated intensity.

Mosaic-patterned positive magnetic-field values characterize areas underlain by lavas and tuffs of andesite and andesite-dacite porphyries of the Silurian Donenzhal Formation. The most intense local positive magnetic anomalies are associated with granodiorite porphyries of the third phase of the Lower Carboniferous Saur Intrusive Complex.

The zones of positive magnetic anomalies delineated within the Kargoba mineral occurrence were numbered and plotted on the interpretation map shown in Fig. 11.

The magnetic survey delineated six zones of positive magnetic anomalies within the Kargoba area.

A-1. This zone is located in the northwestern part of the survey area, measures approximately 2×0.5 km, and is elongated northwestward. Its position and strike are consistent with the geological boundaries shown on the schematic geological map of the Kargoba mineral occurrence. The anomalous zone has a complex, irregular mosaic pattern, with individual magnetic maxima reaching 400, 500, and 800 nT. The anomaly probably has a composite source comprising andesite and andesite-dacite porphyrite lavas together with granodiorites and diorites of the Saur Intrusive Complex.

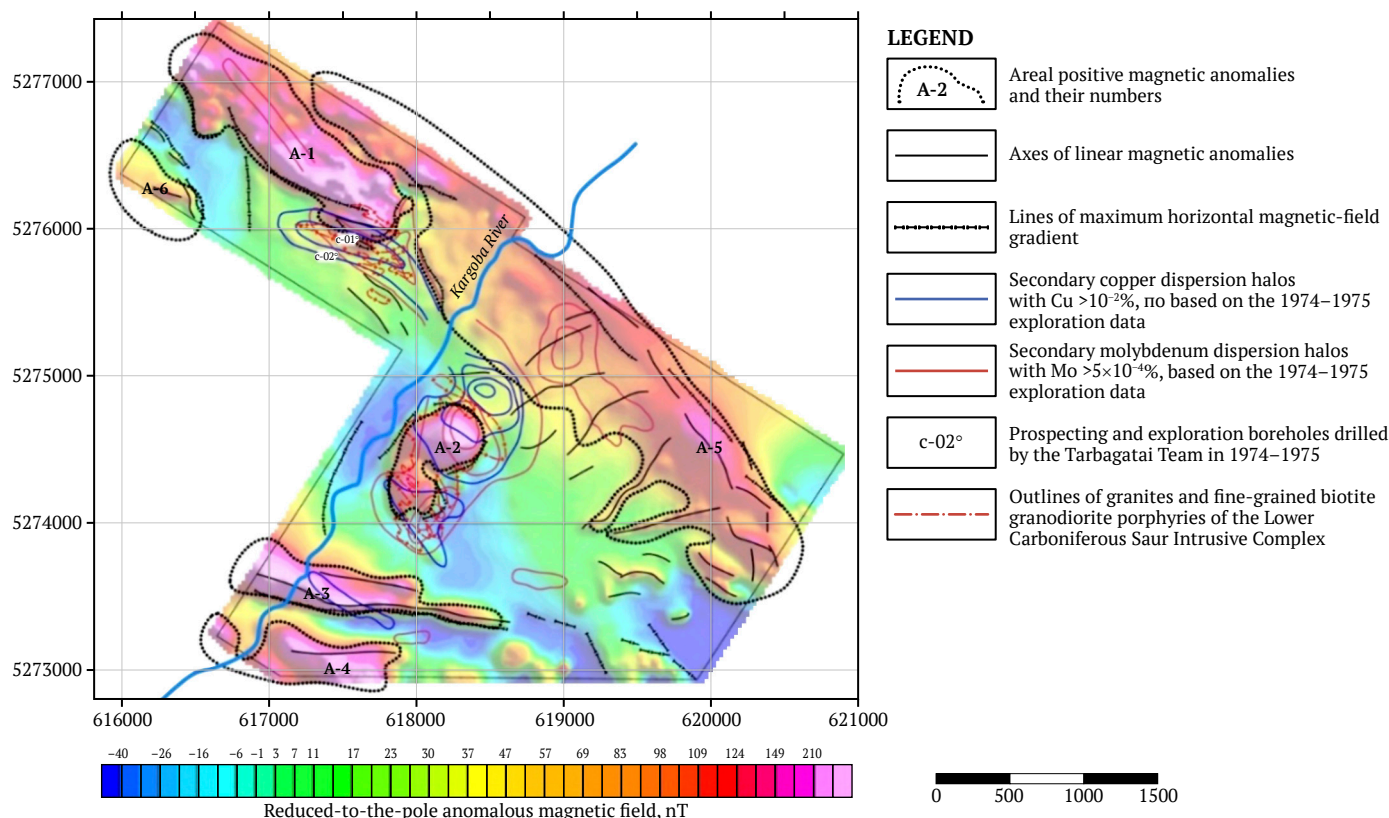


Fig. 11. Magnetic-field interpretation map showing exploration targets and major lineaments

Source: Dubinkin A. V., Akhmatolla A. K., Niyazov A. V., et al. Supplementary geological investigation and inferred resource assessment for map sheets L-44-V, XI, and XVII (Kazakhstan sector): unpublished geological report. Almaty: Kapchagai Geological Exploration Expedition LLP; 2016

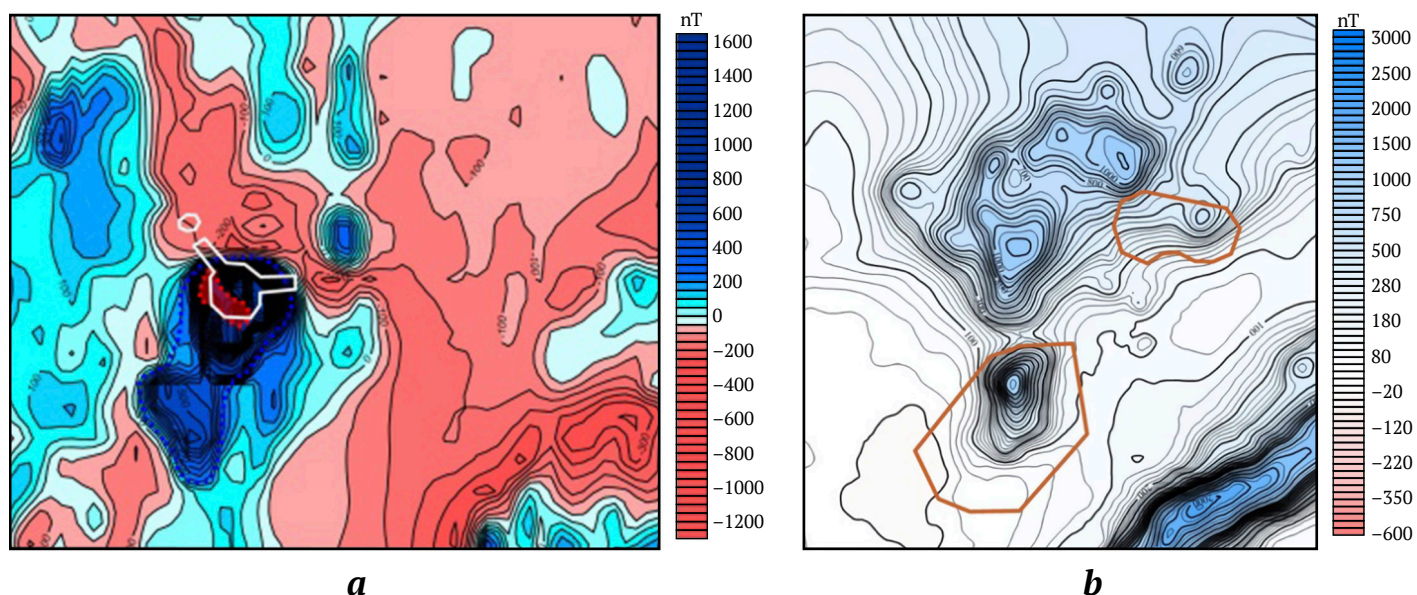


Fig. 12. Anomalous magnetic field over the Yubileynoye (a) and Raigorodok (b) gold deposits

A-2. This anomalous zone is located on the left bank of the Kargoba River and broadly coincides with the mapped exposure of a granodiorite-porphyry stock of the Saur Intrusive Complex. It comprises four discrete magnetic maxima. The northern maximum is approximately isometric in plan view, measures 400–500 m in diameter, and has an amplitude of 480 nT. Additional local maxima to the west and south are slightly elongated in a near north-south direction, measure approximately 100–150×150–250 m, and have amplitudes of 180–200 nT.

A-3. This zone occurs in the southern part of the survey area. It comprises a chain of local magnetic maxima forming a single east-west-trending linear zone measuring approximately 2,000×200–250 m. The local maxima have amplitudes ranging from 180 to 400 nT. Zone A-3 is spatially associated with the contact between andesite porphyrites of the Silurian Donenzhal Formation and the Middle-Upper Devonian sequence of siltstones and polymictic sandstones. The magnetic anomalies may be caused by granodiorites of the Saur Intrusive Complex emplaced along east-west-trending faults.

A-4. Approximately 400 m south of A-3, close to the boundary of the magnetic survey area, zone A-4 was delineated within an area underlain by Silurian andesite and andesite-dacite porphyrites. The zone measures approximately 1,200×250 m and contains individual positive magnetic maxima with amplitudes of up to 250 nT.

A-5. This northwest-trending linear zone, located in the northern part of the magnetic survey area, measures approximately 4,500×500–1,200 m and is

characterized by elevated magnetic-field values of up to 150–180 nT. On the geological map, the zone coincides with outcrops of rhyolite porphyries belonging to the Devonian felsic subvolcanic intrusive complex.

A-6. This zone represents part of a low-amplitude positive magnetic anomaly of 110–120 nT in the southwestern corner of the survey area. The geological map shows Middle-Upper Devonian polymictic sandstones in this area. The anomaly may be related to small Devonian subvolcanic rhyolite-porphyry bodies.

From an exploration perspective, A-2 is the most prospective zone for porphyry copper mineralization. The geometry and amplitude of the magnetic response associated with the granodiorite-porphyry stock are broadly comparable to the anomalies recorded over the Yubileynoye and Raigorodok gold deposits (Fig. 12). Exploration of this zone should therefore assess its potential for both copper and gold mineralization.

The Yubileynoye deposit is located in Aktope Region, Republic of Kazakhstan, on the western limb of the Kunduzdy Syncline. It is associated with a stock of biotite-hornblende plagiogranite porphyry intruding basalts and andesite-basalts of the Mugodzhar Formation and surrounded by intensely silicified rocks.

Gold mineralization occurs in hydrothermally altered rocks both along the peripheral part of the intrusion and within the intrusion itself. The orebodies comprise stockwork zones formed by a dense network of veinlets 0.1–10 cm thick. Locally, the veinlet network becomes sufficiently dense to form massive quartz-rich metasomatic rocks. The ores are of the



moderately sulfidic type. The principal minerals in the primary ores are chalcopyrite, pyrite, magnetite, native gold, and stibnite. Accessory ore minerals include sphalerite, galena, pyrrhotite, marcasite, and rutile. Gangue minerals comprise quartz, calcite, dolomite, epidote, albite, chlorite, actinolite, and sericite.

In addition to gold, copper, and silver, the disseminated quartz-sulfide ores contain 0.001–0.1% arsenic and antimony and 0.01–0.2% zinc. Molybdenum, lead, scandium, gallium, yttrium, ytterbium, bismuth, and tungsten occur sporadically at concentrations of up to several thousandths of a percent.

The deposit coincides with an intense, nearly isometric positive magnetic anomaly. Its high amplitude may reflect both the high magnetic susceptibility of the granodiorite-diorite intrusive rocks and hornfelsing of the overlying host rocks.

The Raigorodok ore field is spatially associated with local magnetic anomalies of 800–1,000 nT. The North Raigorodok ore field occurs at the contact between a gabbro-diorite-monzonite massif and an Upper Ordovician conglomerate sequence near an apical projection of the massif. The host rocks are intensely granitized and skarn-altered.

The orebodies range in strike length from several tens of meters to 400 m and in thickness from a few meters to several tens of meters. They are separated by barren rock packages and subeconomic interbeds whose thicknesses are comparable to those of the orebodies. Gold is distributed unevenly within the ores. Gold grades are generally low, commonly 0.5–2.5 g/t, while silver grades are of a similar order.

The South Raigorodok deposit is hosted in the endocontact and exocontact zones of a small dioritic to gabbro-dioritic intrusive body that cuts an Upper Ordovician sequence of coarse-clastic to boulder conglomerates. The orebodies comprise irregular zones of gold-bearing veinlet-disseminated sulfide mineralization containing pyrite, chalcopyrite, galena, and molybdenite. Their geometry is particularly complex at the intersections of fracture zones with different orientations. The mineralized zones range from 1–2 to 35–40 m in thickness, and gold grades range from trace levels to 150 g/t.

The Central Raigorodok mineral occurrence is associated with the supra-apical part of a small stock-like diorite body. The orebodies consist of zones of veinlet silicification 0.5–1 m thick and occur mainly along contacts between diorite and diorite-porphyry dikes and the enclosing conglomerates. Gold grades range from 0.3 to 10 g/t.

By analogy with these deposits, exploration is recommended in the area of magnetic anomaly A-2 at Kargoba for both copper and gold. The most pro-

spective targets are hydrothermally altered rocks both in the peripheral exocontact zone and within the intrusion itself. The geophysical exploration program should also include gamma-ray spectrometry to identify and map zones of secondary alteration and high-precision detailed gravity surveying to characterize the three-dimensional density structure of the intrusion.

This analogue-based approach to target delineation is supported by the results of digital elevation model analysis in the Aktogai ore field [19]. That study showed that major porphyry copper deposits, including Aktogai and Aidarly, are expressed in the topography as localized caldera-like depressions. Areas with similar morphometric characteristics were therefore considered prospective for further exploration. This interpretation is consistent with the delineated A-2 zone, which is likewise characterized by the combined presence of positive magnetic anomalies, hydrothermal alteration, and structurally controlled landforms.

Conclusion

The aeromagnetic survey provided a detailed 1:5,000-scale magnetic dataset to support further exploration at the Kargoba mineral occurrence.

The survey was conducted using a modern low-altitude UAV-based acquisition system comprising a DJI Matrice 300 RTK and a rubidium quantum magnetometer. This technology combines the high maneuverability of a UAV with the high sensitivity of the magnetometer. It enables high-precision aeromagnetic surveying at very low altitudes with detailed terrain following over both level ground and rugged terrain. Compared with ground magnetic surveying, the new system increased survey productivity severalfold and provided substantially greater detail and spatial resolution than conventional aeromagnetic surveys conducted using crewed aircraft.

The magnetic data were processed and interpreted using modern techniques and software. The calculated magnetic-field transforms enabled the separation of components associated with structural elements of different scales and source depths within the magnetic model.

The study achieved its objective by demonstrating the effectiveness of high-resolution UAV-based aeromagnetic surveying for delineating concealed zones of porphyry mineralization. New qualitative and quantitative data were obtained, including a 3D magnetization model extending to a depth of 500 m. Six anomalous zones were delineated from this model, of which A-2 was identified as the priority target for further investigation. The inferred orebody



parameters include depths of 50–150 m and a strike length of approximately 2 km. The spatial association between the magnetic anomalies and the granodiorite intrusion was confirmed. The identified geophysical criteria – anomaly amplitudes of 200–500 nT, strike lengths of 0.5–2 km, the occurrence of paired anomalies, and their spatial association with granodiorites –

support the prospectivity of the area for porphyry copper-molybdenum mineralization.

The results demonstrate that the proposed methodology can be applied in comparable geological settings and provide a reliable basis for subsequent drill testing, with a high likelihood of intersecting potentially economic mineralization.

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Design rationale for a milling drum used in layer-by-layer surface milling of peat deposits containing wood inclusions

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Abstract

This article addresses the technological foundations and design requirements for developing a machine intended for layer-by-layer surface milling of stump-containing peat deposits. The issue is particularly relevant for small-scale enterprises employing adaptive peat production technologies, where traditional stump removal methods are economically and technically impractical due to high preparatory costs and the need for bulky equipment. The study aims to substantiate the design parameters of a milling machine capable of simultaneously comminuting peat mass and embedded wood inclusions without their preliminary extraction. This approach is intended to enhance production efficiency, improve product quality, and reduce negative environmental impact. The research tasks included analysing milling regimes, interaction between cutting elements and wood inclusions, and developing proposals for a drive system with increased reliability under variable loads. A comprehensive methodology was applied: generalisation of data on physico-mechanical properties of peat and wood, analysis of literary and experimental sources, and theoretical substantiation of cutting parameters and milling drum design. A dual-motor hydraulic drive configuration was considered, wherein an auxiliary motor is activated upon increased cutting resistance, ensuring stable machine operation under deposit heterogeneity. As a result, optimal parameters of the milling drum were substantiated (working width 4.5 m, diameter 0.335 m, rotational speed 149 s^{-1} , peripheral velocity 25 m/s, feed per knife 0.015 m), providing peat crumbs with a particle size fraction of 10–25 mm and comminution of wood to dimensions suitable for subsequent separation, peat processing, and waste utilisation. The proposed approach facilitates accelerated drying of peat crumbs under natural conditions, reduced waste volumes, and rational resource utilisation. It is concluded that the proposed design demonstrates high efficiency for operation at small peat deposits with areas up to 100 ha. The obtained results will be applied in the design of a prototype and further development of adaptive peat production technology.

Keywords

adaptive technology, layer-by-layer surface milling, embedded wood, peat, particle size distribution, milling machines, production efficiency

For citation

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ГОРНЫЕ МАШИНЫ, ТРАНСПОРТ И МАШИНОСТРОЕНИЕ

Научная статья

Инженерное обоснование конструкции фрезерного барабана для поверхностно-послойного фрезерования торфяных залежей с древесными включениями

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Аннотация

Технологические основы и конструктивные требования к созданию машины для поверхностно-послойного фрезерования пнистых торфяных залежей особенно актуальны для малых предприятий, применяющих адаптивную технологию производства торфа, где традиционные методы корчевки



экономически и технологически нецелесообразны из-за высокой стоимости подготовительных работ и необходимости использования громоздкой техники. Цель работы – обоснование параметров конструкции фрезерной машины, способной одновременно измельчать торфяную массу и погребённые в ней древесные включения без предварительного извлечения последних. В работе применён комплексный подход: обобщение данных по физико-механическим характеристикам торфа и древесины, анализ литературных и экспериментальных источников, теоретическое обоснование параметров резания и конструкции фрезерного барабана. В результате обоснованы оптимальные параметры фрезерного барабана: ширина захвата 4,5 м, диаметр 0,335 м, частота вращения 149 с^{-1} , окружная скорость 25 м/с, подача на нож 0,015 м, обеспечивающие получение крошки фракции 10–25 мм и дробление древесины до размеров, пригодных после сепарации для последующей переработки торфа и утилизации отходов. Предложенный подход позволяет вести добычу торфа на залежах с древесными включениями без предварительной корчевки, что снижает затраты на подготовительные работы и расширяет ресурсную базу малых предприятий.

Ключевые слова

адаптивная технология, поверхностно-послойное фрезерование, погребенная древесина, торф, фракционный состав, фрезерующие машины, эффективность производства

Для цитирования

Gamayunov S.N., Zhigulskaya A.I., Zyuzin B.F. Design rationale for a milling drum used in layer-by-layer surface milling of peat deposits containing wood inclusions. *Mining Science and Technology (Russia)*. 2026;11(2):122–129. <https://doi.org/10.17073/2500-0632-2025-06-994>

Introduction

Most peat producers currently operate on a relatively small scale, with production geared primarily toward local consumption [1]. In this context, study [2] demonstrated the need to develop and implement an adaptive peat production technology at peat sites of up to 100 ha. This requires the development of multifunctional machinery, including equipment capable of simultaneously milling the peat deposit and the wood inclusions embedded within it.

One of the main ways to reduce the cost of developing small peat sites is to develop mechanized equipment [3] capable of milling peat together with wood inclusions without prior stump removal. This approach opens up new opportunities for layer-by-layer surface milling of peat deposits to produce milled peat of a specified quality and ensure the efficient use of biogenic resources [4].

Milling establishes the process conditions for all subsequent production stages, including drying, harvesting, and processing of the milled peat [5]. Various types of peat milling machines are used for this purpose [6]. These machines are designed to operate under harsh conditions [7]. Innovative solutions in this area are technically feasible and economically viable and can support the sustainable development of small peat producers [8].

The aim of this study was to justify the design parameters of a milling drum capable of minimizing energy consumption while producing milled peat with a specified particle size distribution suitable for subsequent production and processing operations.

The objectives of the study were as follows:

1. To analyze published and experimental data on the physical and mechanical properties of peat and wood.

2. To justify the cutting parameters required to obtain milled peat particles in the 10–25 mm size range, which is considered optimal for subsequent production stages.

3. To develop design proposals for the milling drum and consider a dual-motor hydraulic drive configuration to improve reliability under variable loads.

The proposed approach builds on the research reported in [2] and will be applied in the design of a milling drum for operation in peat deposits with a high stump content under challenging peat extraction conditions.

Study rationale

The development of small peat deposits covering up to 100 ha requires economically viable solutions, since conventional stump grubbing is not justified because of its high cost and labor intensity [2]. Practical experience [9] shows that the use of high-capacity machinery at small deposits with limited working areas is inefficient: its full production capacity cannot be utilized, while the associated capital costs substantially increase the unit cost of production.

Simultaneous milling of peat and stumps embedded in the deposit eliminates the need for additional operations and avoids the cost of purchasing and operating numerous specialized machines. It also allows the same set of machinery to be used under different operating conditions, in accordance with the concept of adaptive technology [2].

The concept of continuous milling of a peat deposit together with wood inclusions was first considered in the mid-twentieth century [10]. However, it was not widely adopted because of technical limitations and the insufficient power of the tractors available at the time. Modern engineering solutions,



including a dual-motor hydraulic drive and optimization of the cutting-element parameters, allow this concept to be revisited and implemented under current operating conditions.

Layer-by-layer surface milling is a key operation on which all subsequent stages of the production process depend, including drying, spreading, harvesting, and utilization [5]. Milling quality directly affects the particle size distribution of the milled peat, the drying process, and the condition of the deposit surface after machine operation [3]. It is therefore essential to select cutting elements and milling parameters [4, 11] that produce a uniform milled peat spread and ensure effective comminution of the peat while simultaneously reducing the stumps to fragments no smaller than 10 mm, allowing them to be efficiently removed using standard separators.

Studies have shown that the specific energy consumption of wood milling is substantially higher than that of peat milling [4, 12]. A key optimization objective is therefore to reduce failures and overloads, which can be achieved using the dual-motor hydraulic drive proposed in [13]. The proposed machine should be mobile, cost-effective, and suitable for commercial production, as supported by studies on machinery design for small peat deposits [14]. It must also be suitable for peat extraction conditions, including operation under variable loads and in high-moisture environments. The interaction between the cutting elements and the heterogeneous medium is stochastic and therefore requires an adequate mathematical description of the cutting forces [15].

Current conditions therefore call for new engineering solutions that allow milling to be performed without prior preparation of the peat deposit. Such operations require robust and reliable equipment capable of withstanding variable loads while producing material in the specified particle size range.

Methodology

The study was based on an integrated analysis of the literature and synthesis of reliable data on the physical and mechanical properties of peat and wood. The available data were considered sufficient to justify the design parameters without the need for additional experimental studies.

A particle size range of 10–25 mm for the milled peat was adopted as the optimization criterion for the cutting conditions, since this range facilitates faster drying and efficient separation [16]. The resulting wood fraction can also be used as fuel [17]. Machines based on a similar operating principle – forestry mulchers – have demonstrated high efficiency in comminuting woody residues and stumps [18].

Mathematical modeling was used to determine the drum parameters from the following kinematic equations:

$$v_{per} = \frac{\pi D n}{60}, \quad s = \frac{v_f}{nz},$$

where v_{per} is the peripheral speed, m/s; D is the drum diameter, m; n is the drum rotational speed, rpm; s is the feed per knife, m; v_f is the forward speed, m/s; z is the number of knives.

To obtain the specified particle size range, the milling depth, feed per knife, drum peripheral speed, and the number and geometry of the cutting elements were calculated. The resulting values formed the basis for the milling drum design, which was intended to ensure consistent comminution of both the peat and the wood inclusions.

Results and discussion

The exploitation of small peat deposits of up to 100 ha is becoming increasingly important in many regions of Russia, as it helps meet local demand for peat used both as fuel and for agricultural purposes. The diversification of peat applications, primarily toward further processing that preserves some of its natural properties, has necessitated the development of new peat production machinery and technologies capable of producing material with the required particle size characteristics [2].

Layer-by-layer surface milling of a peat deposit is the first and key operation in the milled peat production cycle [2]. The quality of this operation largely determines the performance of the entire cycle, including drying time, peat yield per cycle, moisture content, and other properties of peat produced under natural field conditions [5]. The milling operation must therefore create a layer of milled peat that dries as rapidly as possible under the prevailing weather conditions. This is the primary technical requirement for the milling operation.

Milling quality is assessed by the particle size distribution of the resulting milled peat and its drying behavior during the production cycle [16]. Another important criterion is the condition of the production field surface after milling [19]. The particle size distribution of the milled peat is also influenced by the milling depth, peat deposit properties, and the peripheral speed of the cutter, which corresponds to the cutting speed [7]. An optimal peripheral speed of 25 m/s was established, providing stable fragmentation of the wood without excessive comminution of the peat.

The analysis showed that the 10–25 mm size fraction provides an optimal balance between drying rate and the efficiency of separating wood inclusions.

Particles of this size, spread in a layer 1–1.5 particles deep, exhibit the highest rate of moisture loss while offering minimal resistance to airflow [3, 5, 16]. Larger peat particles were found to dry more rapidly (Fig. 1), thereby reducing the time required to reach the harvesting moisture content.

A layer of fine particles dries more slowly than a layer of coarser particles because its smaller pores reduce the rate of moisture evaporation. Fine particles at the surface form a dense layer with low thermal conductivity, limiting the penetration of solar radiation into the underlying material and slowing the drying of the lower layers. The 10–25 mm size fraction therefore substantially improves drying efficiency and the reliability of the milled peat production process.

A properly formed milled peat spread is characterized by a uniform depth of the milled layer and

a homogeneous particle size distribution dominated by coarser peat particles [20]. For milled peat intended for agricultural use, a direct relationship has also been established between particle size and substrate properties, including air capacity and water retention [21]. The cutter design and knife geometry should therefore be selected to produce a particle size distribution favorable for drying in the spread layer while minimizing the spread nonuniformity coefficient.

Feed per knife is one of the principal parameters determining both the milling process and the design of the cutter elements [6, 7]. The proportion of the 10–25 mm fraction, (F_{10-25}), varies nonlinearly with feed per knife (Fig. 2). The maximum value of (F_{10-25}), reaching 58.3%, was obtained using straight knives of triangular cross-section at a feed per knife of 0.015 m [11, 20].

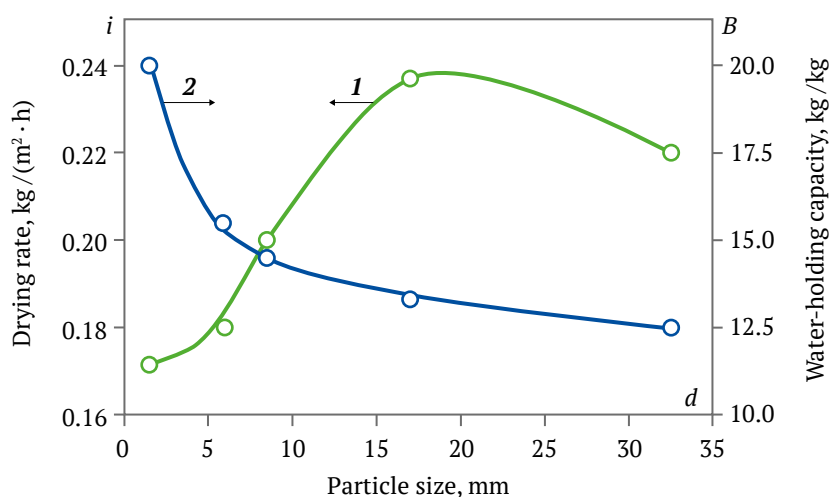


Fig. 1. Dependence of drying rate (i) and water-holding capacity (B) on milled peat particle size (d) [5, 20]

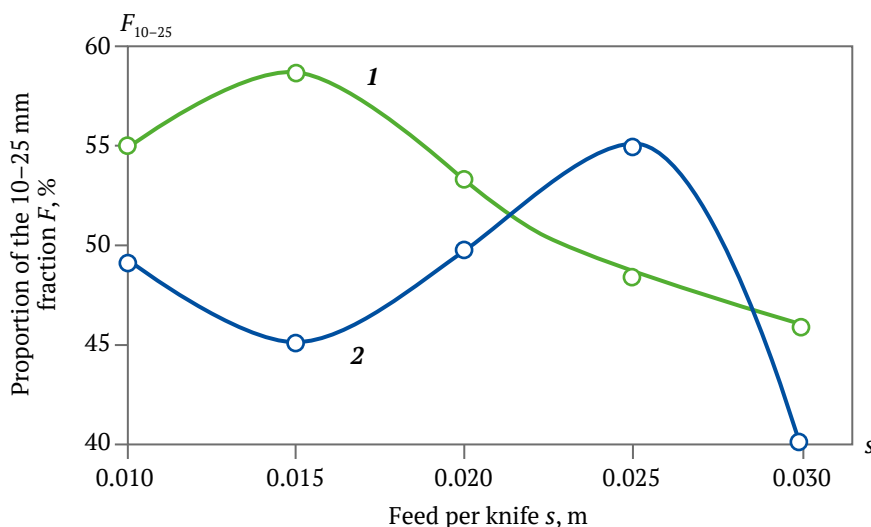


Fig. 2. Dependence of the proportion of the 10–25 mm fraction F_{10-25} in milled raised-bog peat on feed per knife s for milling drums fitted with straight knives of triangular cross-section (1) and longitudinal knives (2) [11, 20]



The difference in performance between straight and through knives is attributable to the physical processes involved in chip formation and in drawing the detached material through the gap between the cutter and the peat deposit. The quality of the milled peat spread depends on the type of milling equipment and the cutting elements fitted to it.

The specific energy consumption for peat milling ranges from 0.14 to 1.1 MJ/m³, whereas that for milling green wood is considerably higher, ranging from 12.6 to 14.4 MJ/m³ [4, 17, 22]. The mean specific energy consumption for milling peat together with stumps is 5.76 MJ/m³ [13], indicating that this process can be implemented without a substantial increase in energy demand. The separated wood fraction can be used as fuel. Thermal conversion of blended fuels based on peat and comminuted wood can yield biochar with favorable fuel properties [23]. In Europe, the replacement of peat with locally available biomass, including comminuted wood, is also being considered [24].

The most promising technical solution for layer-by-layer surface milling of a peat deposit together with stumps was proposed by M.I. Sarmatov [13] based on laboratory and theoretical studies. This work provided a justification for the cutter geometry and kinematic parameters and introduced a calculation model for determining the power required to drive the drum knives [25, 26]. Further developments were reported in [27, 28], which presented analytical relationships for calculating the increase in resisting torque when the cutter encounters a stump and for determining the active width of the cutter. These relationships were used to establish the optimal number and arrangement of knives on the drum and to justify the advantages of a continuous cutting edge.

Milling units with a larger working width are known to be less maneuverable, more material-intensive, and more labor-intensive to maintain [6, 29]. A working width of 4.5 m was therefore selected for the cutter. This width permits a whole number of passes across the production field, providing high productivity while maintaining sufficient machine maneuverability.

According to [30], the knife design, the material from which the knives are made, and their arrangement on the drum are critical to the efficient operation of the milling unit. The knives must be made of high-strength, wear-resistant materials suitable for the intended operating conditions [31]. This is particularly important when milling wood inclusions, which substantially increase cutting resistance.

The peripheral speed required to fragment the wood can be achieved either by increasing the drum rotational speed [32], which is subject to technical limitations, or by increasing the cutter diameter, which is a more practicable solution. Because the cutter mass is inversely proportional to the square of the peripheral speed, reducing its mass at a constant rotational speed requires an increase in cutter diameter [33]. Based on the design proposed in [13], the cutter diameter measured over the knife tips was set at 0.335 m.

The proposed machine is expected to employ a dual-motor hydraulic drive in which the second motor is automatically engaged when cutting resistance increases, for example when the cutter encounters a large stump [13]. This configuration is expected to improve operational reliability and reduce the risk of equipment failure.

The following key parameters should be adopted in designing a milling drum for layer-by-layer surface milling of peat deposits with a high stump content to obtain milled peat in the required 10–25 mm size range:

- working length of the drum, 4.5 m;
- cutter diameter measured over the knife tips, 0.335 m;
- drum rotational frequency, 149 s⁻¹;
- peripheral speed at the cutting edges, 25 m/s;
- feed per knife, 0.015 m;
- knife working height (milling depth), 0.020 m;
- knife working width, 0.025 m;
- number of cutting elements per cutting plane, 6;
- machine travel speed, 2.14 m/s;
- number of hydraulic drive motors, 2.

Promising directions for further development include the introduction of systems for monitoring and automated control of cutter operating parameters [34]. Further research should also investigate the use of composite materials for cutting elements capable of withstanding higher loads with minimal wear.

Conclusion

The engineering solutions presented in this article are aligned with current trends in mining technology aimed at improving the efficiency and environmental safety of mineral deposit development.

The study demonstrated the feasibility of developing a specialized milling machine for layer-by-layer surface milling of peat deposits with a high stump content. The identified design parameters and features of the milling unit will enable simultaneous comminution of the peat and wood inclusions, pro-



duction of milled peat in the 10–25 mm size range, and reliable equipment operation.

The key engineering solutions include a milling drum with the following optimal parameters: a working width of 4.5 m, a diameter of 0.335 m, a rotational frequency of 149 s^{-1} , a peripheral speed of 25 m/s, and a feed per knife of 0.015 m. These parameters ensure the required quality of the milled peat. The dual-motor hydraulic drive is expected to improve machine reliability and reduce the risk of equipment failure when the cutter encounters large stumps.

The study findings are of practical value and will be used to design a prototype for operation under actual field conditions at peat extraction sites. By

eliminating preliminary stump grubbing, the proposed design can reduce production costs and improve the environmental performance of the peat production process. Future work will focus on developing a prototype fitted with heavy-duty cutting elements made of composite materials and integrating load-monitoring systems to optimize operating parameters. The peat and comminuted wood produced during milling can be separated and subsequently used as feedstock for biochar production by pyrolysis, thereby broadening the options for waste utilization. The findings provide a basis for developing an adaptive peat production technology that accommodates different scenarios for the commercial use of the available resources.

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Research paper

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**Mine design and process engineering solutions
for copper-nickel ore quality management in underground mining**N. A. Turtygina¹ , A. V. Okhrimenko² , A. V. Glinsky² , K. A. Ryzhenkov¹ ¹ N.M. Fedorovsky Polar State University, Norilsk, Russian Federation² PJSC MMC Norilsk Nickel, Norilsk, Russian Federation natyrtigina@mail.ru**Abstract**

Declining ore grades at the Norilsk deposits are accompanied by increasing variability in the quality of mined ore and deterioration in the quality of saleable products. These trends necessitate a revision of the fundamental approaches to optimizing mining systems for ore-stream quality management. This study aimed to advance the theoretical framework and improve the methodology for substantiating process engineering solutions for integrated ore quality management in underground mining through innovative modernization of mine flowsheets. An integrated research approach was employed, including a review of previous studies; analysis of Russian and international experience in ore-stream management; field sampling of ore in deposits and along ore streams, followed by geological and mineralogical testing and correlation analysis; production-scale investigations of particle-size segregation during mining; analytical studies based on probability theory and mathematical statistics; and modern modeling methods. An updated classification of methods for assuring and controlling ore quality in underground mining is proposed. Blending methods are classified as blending and stabilization methods aimed at achieving and maintaining the required quality of mined ore, whereas segregation and separation methods are intended to divide, separate, or sort the material. Production-scale investigations confirmed that considerable potential exists for more effective control of ore-stream quality. Fundamental principles were formulated for developing process engineering solutions at the underground-mine design stage to manage copper-nickel ore quality and substantially improve the quality of mine products.

Keywords

underground mine, particle-size segregation, ore quality parameters, ore quality management, mining technology, mining operations, grade stabilization, mineral composition

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РАЗРАБОТКА МЕСТОРОЖДЕНИЙ ПОЛЕЗНЫХ ИСКОПАЕМЫХ

Научная статья

**Проектно-технологические решения для управления качеством
медно-никелевых руд на подземном руднике**Н. А. Туртыгина¹ , А. В. Охрименко² , А. В. Глинский² , К. А. Рыженков¹ ¹ Заполярный государственный университет имени Н.М. Федоровского,
г. Норильск, Российская Федерация² ГМК «Норильский никель», г. Норильск, Российская Федерация natyrtigina@mail.ru**Аннотация**

В условиях обеднения минерально-сырьевой базы норильских месторождений наблюдается увеличение размаха колебаний качественных характеристик в объёмах добычи и снижение качества товарной продукции, что требует актуализации принципиальных положений в области оптимиза-



ции горнотехнических систем для управления качеством рудопотоков. Цель исследования – развитие теории и совершенствование методологии обоснования технологических решений системного обеспечения качества при подземной добыче руд на основе инновационных решений по модернизации технологических схем рудников. В работе использован комплексный метод исследований, включающий обобщение результатов ранее выполненных научных работ, анализ отечественного и зарубежного опыта по управлению рудопотоками, натурное опробование руды в залежах и рудопотоке с геолого-минералогическими испытаниями и корреляционным анализом, производственные исследования по изучению сегрегации рудной массы при добыче, аналитические исследования с использованием методов теории вероятностей и математической статистики, а также современные методы моделирования. Представлена актуализированная систематизация методов обеспечения и контроля качества руд при подземной добыче: способы усреднительного характера рассматриваются как усреднительно-стабилизационные (смесительные) методы, направленные на повышение и поддержание качества добытых руд, а сегрегационно-разделительные (сепарационные) – на разделение или сепарацию. Производственными исследованиями подтверждено наличие значительных резервов для эффективного формирования качества рудопотоков. Сформированы принципиальные положения по разработке технологических решений в проекте подземного рудника для управления качеством медно-никелевых руд, способных существенно улучшить продукцию горнодобывающего предприятия.

Ключевые слова

подземный рудник, сегрегация, качественные показатели, методы управления, технология, горные работы, стабилизация, вещественный состав

Для цитирования

Turtygina N.A., Okhrimenko A.V., Glinsky A.V., Ryzhenkov K.A. Mine design and process engineering solutions for copper-nickel ore quality management in underground mining. *Mining Science and Technology (Russia)*. 2026;11(2):130–140. <https://doi.org/10.17073/2500-0632-2025-07-1003>

Introduction

One of the principal requirements imposed by consumers on mineral raw materials is the consistent quality of the material extracted from the subsurface. Underground mining of ore deposits amid declining ore grades is generally accompanied by deterioration in the quality of mined ore, requiring mining companies to improve their competitiveness. The Norilsk copper-nickel deposits are no exception in this respect. For the underground mines of the Norilsk Division of PJSC MMC Norilsk Nickel, addressing this challenge primarily requires meeting market demand for higher-quality saleable ore and increased metal output while expanding the mining and processing of ore with progressively deteriorating quality characteristics. Research aimed at advancing the theory and technologies of integrated ore quality management in underground mining is therefore both relevant and timely. The present study is driven by the need to identify process engineering solutions capable of substantially improving the quality of mine products under conditions of steadily declining grades in the remaining ore reserves.

Research aim and objectives

The following objectives were formulated:

- to develop an overall research methodology and select the study object;
- to investigate the geological variability in ore quality within the Norilsk copper-nickel ore deposits and assess the complexity of their occurrence;

- to conduct in-mine investigations of ore segregation within the mine flowsheet and examine changes in ore quality parameters during mining;

- to update and systematize methods for ore-stream quality management;

- to develop a system of mining, process engineering, organizational, and technical measures capable of improving ore quality and maintaining the stability of the mineral composition of the mined material within a combined mine system for production quality management.

At present, the high-grade ore deposits of the Talnakh ore cluster are being mined most intensively. The mineral material extracted from the subsurface comprises three commercial types of copper-nickel ore – high-grade, cuprous, and disseminated ores – as well as diluting material consisting of waste rock and cemented backfill. The run-of-mine material produced by the mines is supplied as feed to the Norilsk and Talnakh concentrators. The chemical composition of the feed material significantly affects concentrate production, the metallurgical process as a whole, and metal recovery.

Statistical processing of a large dataset was used to forecast changes in ore production and the recoverable value of the mined ore. The analysis indicates that over the next 20 years of underground mining at the Norilsk ore deposits, the total value of recoverable components per tonne of copper-nickel ore will decline (Fig. 1).

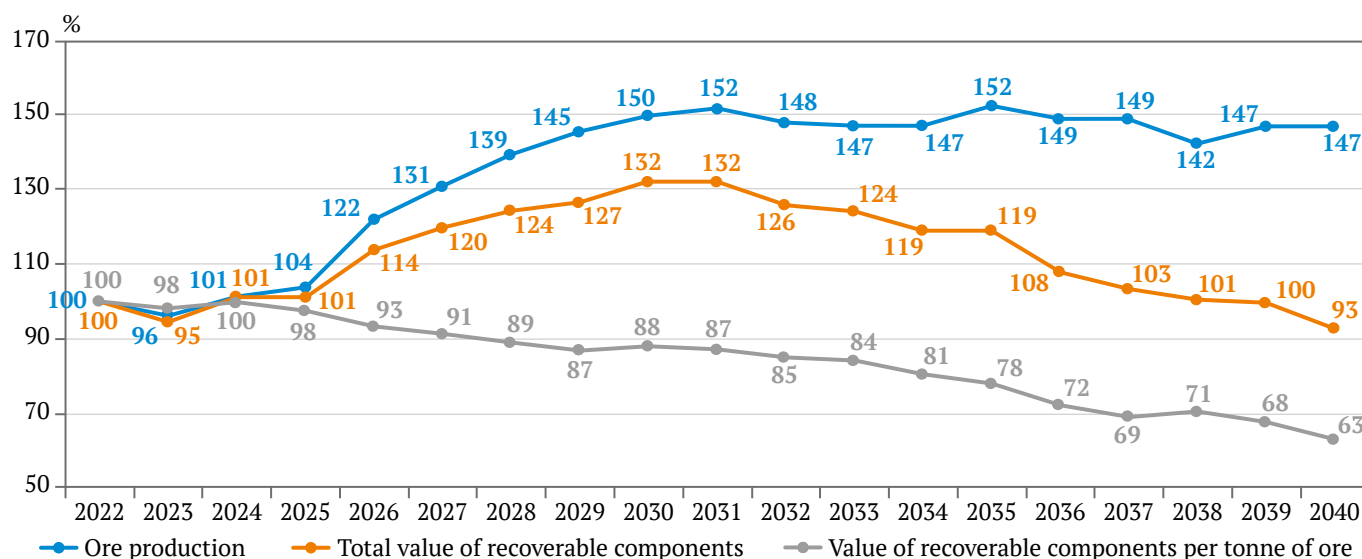


Fig. 1. Forecast changes in ore production and the value of recoverable components during development of the Norilsk ore deposits

Against this background, the planned 30% increase in mine production relative to the current level, together with the gradual introduction of lower-grade disseminated and refractory ores, is expected to reduce the recoverable value of the mined ore by 35% or more. Thus, as the quality of the commercially mineable reserves of the principal ore types declines, a comprehensive basis is required for mine design solutions aimed at optimizing underground grade-stabilization systems. Such solutions should combine innovative modernization of mine flowsheets with additional process engineering measures designed to improve the quality of the mined material. This makes it necessary to revise several fundamental principles of ore quality management in mining operations at the Norilsk deposits [1].

Research methodology and study objects

An integrated research methodology was employed, including a synthesis of findings from previous studies; analysis of Russian and international experience in ore-stream management; in-situ sampling of ore within the deposits and along the ore streams, followed by geological and mineralogical characterization and correlation analysis; production-scale investigations of ore segregation during mining; analytical studies based on probability theory and mathematical statistics; and modern modeling methods. The authors' contribution included establishing correlations between metal grades in the ore and particle-size classes under the operating conditions of the mines, as well as updating and systematizing methods for ore quality management in underground mining. For the first time, analytical studies and field observations were conducted to assess the blending capacity

of individual stages of the mine production chain, and computer modeling was performed for the preconcentration of valuable components in the ore. A mine flowsheet was developed for an underground system designed to establish and stabilize ore quality during mining. The content and architecture of a mine-wide information and control system for real-time ore quality monitoring throughout the entire mining production chain were also substantially redesigned.

1. Theoretical studies

Ore quality management during mining is inherently a techno-economic problem and is generally achieved through process control, ore blending, real-time control of mining operations, separation of ore streams into different material-handling routes as the extracted mineral material is delivered from the mine, and other measures [2].

In the authors' view, the multistage mine planning process is initially intended to establish the target average metal grade in the planned ore output, whereas organizational and operational processes are intended to ensure that this grade is distributed uniformly throughout the ore streams by mechanical mixing. Mined ore can be mixed in load-haul-dump and haulage equipment, mechanical crushers, ore passes, underground bins, stockpiles, and storage facilities. Taken together, these processes provide mined mineral material with the optimum average quality¹. Ore quality management should therefore be understood as a set of organizational and production measures

¹ Regulations on the Department of Production Automation and Digitalization, SP-132-2021, Polar Division of PJSC MMC Norilsk Nickel. Norilsk; 2021. 17 p. (In Russ.)



implemented within the underground mine system to establish the optimum average quality of the ore during mining and ensure that this quality is distributed uniformly throughout the mined material [3, 4]. Overall, blending and separation operations form the basis of mineral raw material quality management during deposit development [5]. In view of these considerations, the methods used to assure and control ore quality in underground mining were updated and systematized (Fig. 2).

The updated systematization of ore quality management methods was based on the findings of previous studies, together with the authors' field observations and assessments. It differs from commonly used classifications by dividing ore quality management methods into three groups: organizational, operational and technical, and process engineering methods. Blending operations are classified as blending and stabilization methods intended to improve ore quality and maintain it within the ore streams.

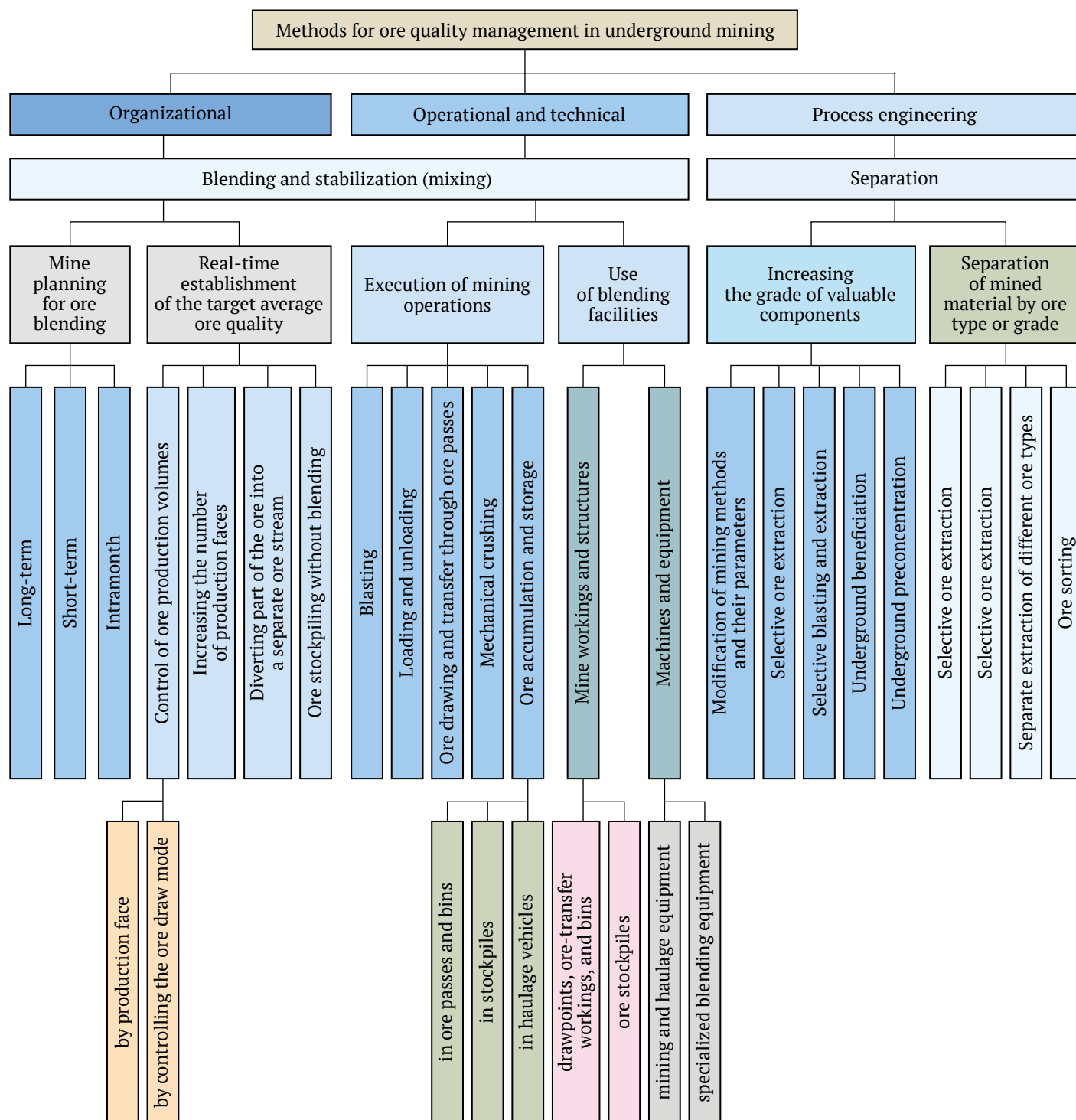


Fig. 2. Structuring of updated methods for ore quality assurance and control in underground mining

The process engineering methods comprise segregation-based technologies intended for material separation. Segregation and separation methods of ore quality management are used to separate or sort the mined material or remove some of its constituents from the production volume [2, 6]. Selected portions of the mined material are separated to increase the valuable-component grade in a specified portion of the production output. Equipment used for this purpose may include load-haul-dump and haulage machines, real-time monitoring devices, batch or particle sorting units, screens, crushers, and feeders² [2]. Segregation-based separation methods, in turn, are intended to increase the grade of valuable components in the mined ore or sort the material using modern X-ray radiometric ore-sorting and preconcentration equipment. Segregation-based flowsheets involving small-batch separation and small-batch preconcentration incorporate X-ray radiometric separators, which can reject more than 40% of the low-quality fraction from the total mined material and almost double the grade of the valuable component in the preconcentrate [2].

2. Empirical studies

To establish design requirements for mine systems capable of maintaining stable ore quality during underground mining of copper-nickel ores, the natural variability in ore quality at the Norilsk deposits was quantified using the index of geological complexity, λ . The analysis was based on geological reporting data generated in the Geologist automated workstation database environment (version 13.1) and in Micromine software (version 2023.5), which provides an integrated framework for data collection, storage, and processing.

The geological variability in the quality characteristics of deposits within the Norilsk mineral resource complex was investigated at the company's underground mines. The Zapolyarny Mine develops the Norilsk-1 deposit using a mining method involving caving of the overlying rock mass. The Komsomolsky Mine extracts several commercial types of copper-nickel ore from the Talnakh and Oktyabrskoye deposits. The higher-value, high-grade ores of the Talnakh deposit are mined selectively using mining methods with complete cemented backfilling of the mined-out areas³.

² Regulations on month-ten-day period-day-shift mine planning by the Planning Center at the Komsomolsky Mine. Norilsk; 2019. 10 p. (In Russ.)

³ Regulations on month-ten-day-period-day-shift mine planning by the Planning Center at the Komsomolsky Mine. Norilsk; 2019. 10 p. (In Russ.)

Strategy for the development of sulfide and copper-nickel ore deposits of the Talnakh ore cluster and the Norilsk-1 deposit by the mining enterprises of the mineral resource complex of the Polar Division of OJSC MMC Norilsk Nickel. Mining Administration, Polar Division of OJSC MMC Norilsk Nickel. Norilsk; 2013. (In Russ.)

To provide a comprehensive basis for mine design and process engineering solutions concerning the grade-stabilization capacity of underground mine processes, changes in ore quality parameters along the ore streams were investigated. The field studies showed that the natural quality of copper-nickel ores is affected most strongly by the combined process engineering and technical measures associated with mining operations and the transfer of mined material through the underground material-handling system. In particular, the formation of ore quality characteristics within the ore streams was examined by correlation analysis using the primary data from the site-specific design for extraction of a stope, together with data from Micromine (Fig. 3), MineSched, and the ten-day and shift scheduling system DSG 2.0. The latter is a digital solution integrated into the business processes of Norilsk Nickel's mines to improve production quality and ensure compliance with production schedules. The specialized GEOVIA MineSched software package (version 2021) supports mine planning and production optimization for both open-pit and underground mining operations.

The pattern of within-block variability in metal grades in the stope is shown in Fig. 4.

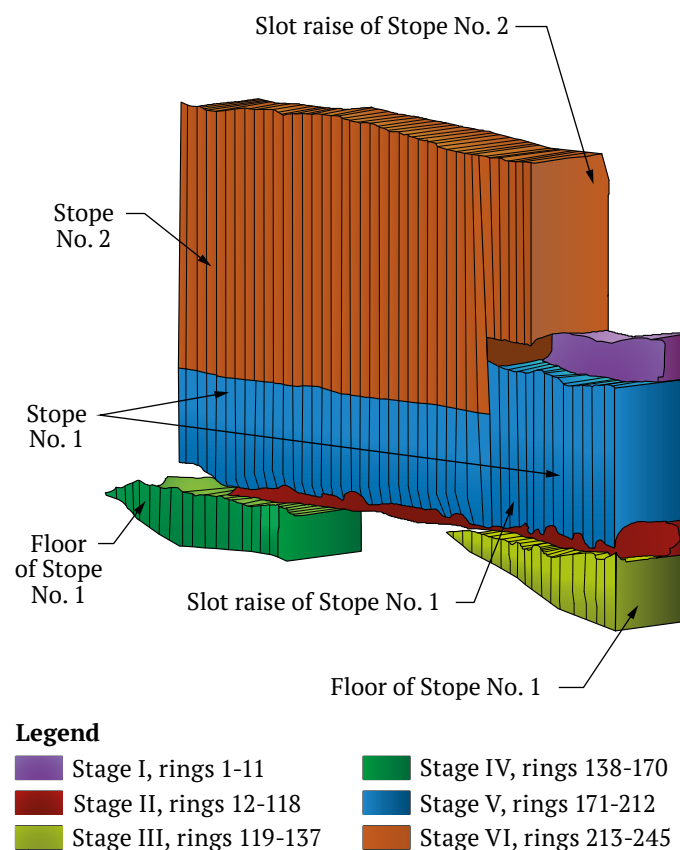


Fig. 3. General view of the stope in the detailed stope extraction design developed in the Micromine mining and geological information system

In mining practice, an ore pass is often regarded as an effective underground blending facility. During the extraction of disseminated ores in particular, however, an ore pass may cause deblending because of particle-size segregation within the mined material. Particle-size segregation of the rock-ore material during development of a copper-nickel ore deposit arises from the inverse relationship between the size of the blasted ore fragments and their metal grades (Fig. 5). Field investigations of the ore stream were conducted to examine this segregation phenomenon. The sampling scheme included sampling of the muckpile, car-by-car sampling of each ore train on the haulage levels,

and sampling of ore from the bucket of a load-haul-dump machine (LHD). During the observation period, 1.926 thsd t of ore passed through the haulage-level workings. Thus, as the ore fragment size in the muckpile decreased from 350 to 50 mm, the nickel grade in the mined ore increased from 0.19 to 0.34%, while the copper grade increased from 0.22 to 1.30%.

Analysis of the identified statistical relationship demonstrates the need for new mine design and process engineering solutions for copper-nickel ore quality management. These solutions should ensure stable average metal grades in the segregated mined material delivered to the concentrator.

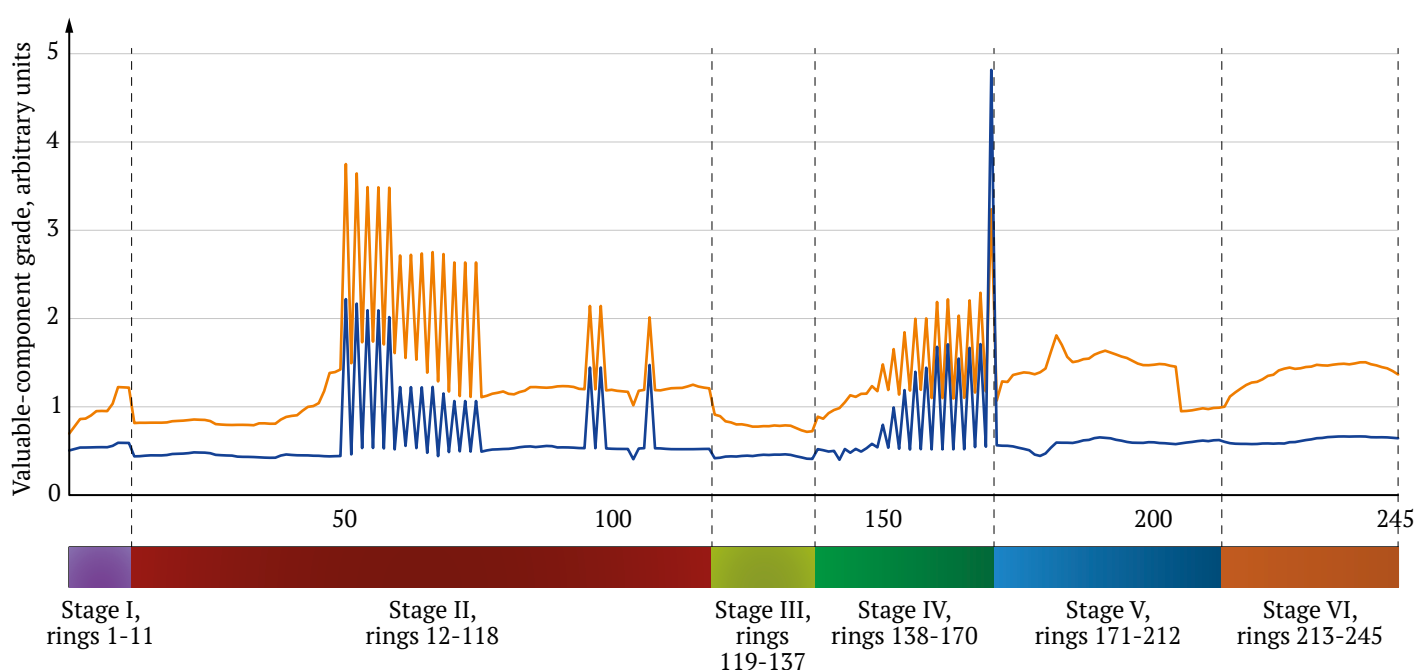


Fig. 4. Variability in the grades of metals A and B by stope extraction stage

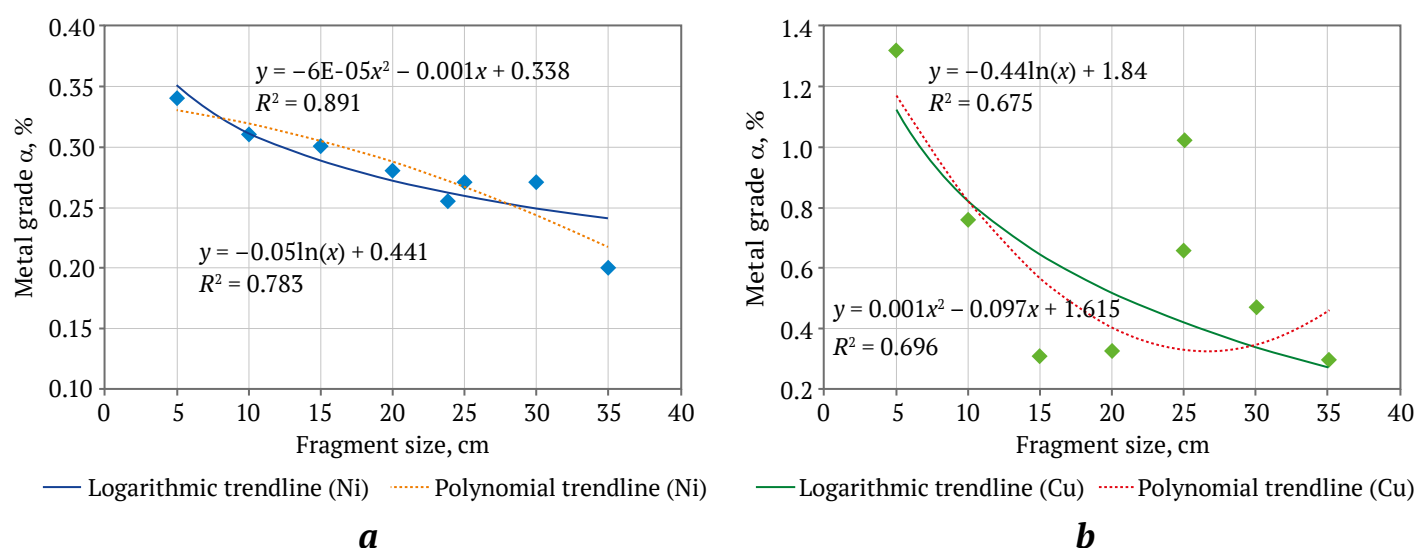


Fig. 5. Inverse relationship between blasted ore fragment size and nickel (a) and copper (b) grades



Results

An assessment of the complexity of orebody occurrence at the Norilsk copper-nickel deposits showed that the deposits do not fall into the complex category ($\lambda = 0.01–0.5$). However, mining conditions deteriorate as the orebody occurrence complexity index increases. Consequently, the predicted ore loss and dilution rates for individual mining units increase, thereby reducing ore recovery during mining and widening the discrepancy between planned and actual mining performance under the changed mining conditions [7].

To investigate changes in ore quality parameters along the ore stream, Micromine was used to calculate indices of ore quality variability during mining operations (Table 1). The results indicate that metal grades in the ore output will be unstable as the ore is blasted in the stope through successive blast-hole rings.

The standard deviations of nickel and copper grades attributable to within-block variability were predicted to exceed 0.41 and 0.55%, respectively. The production-scale investigations therefore showed that the composition of the mined material is partially destabilized at individual stages of the production chain, reducing the overall grade-stabilization capacity of the mine flowsheet.

For example, particle-size segregation during the transfer of mined material through ore passes causes deblending, that is, separation by particle size, and reduces the blending coefficient. Thus, segregation of the mined ore disrupts the natural grade-stabilization effect of the mine flowsheet while simultaneously creating favorable conditions for separation technologies. The higher the ore contrast index, the more favorable the conditions for separation and, consequently, for the use of underground segregation-based technologies [8]. In-mine investigations were conducted to determine ore contrast. Under the conditions of the Norilsk underground mines, rock-ore fragments within the specified size range were classified as having low contrast at $M = 0.4–0.8$ and moderate contrast at $M = 0.8–1.1$. Despite the relatively low contrast of the Norilsk ores, computer modeling of preconcentration of the mineral components in the mined ore was performed [2, 9]. Numerical modeling and statistical processing of the test data were carried out using STATISTICA 10.0 and Microsoft Office Excel. The modeling results showed that this solution reduces the volume of saleable ore by 36% by removing lower-grade fractions from the total mined material. As a result, the average nickel grade in the preconcentrate increases by 63% relative to the feed, from 0.26 to 0.41%, that is, by more than 1.5 times. The average copper grade in the preconcentrate increases from 0.38 to 0.60%, corresponding to an absolute increase of 0.22 percentage points, at a metal recovery of 94%.

Table 1

Statistical indices of ore quality variability

Variability indices in the stope orebody according to the mine design		Predicted indices of process-induced variability during blasting in the stope	
Ni	Cu	Ni	Cu
Coefficient of variation, v , %			
64.76	41.08	89.24	51.01

Discussion

The integrated study showed that sulfide ores within the Norilsk underground mining area are heterogeneous in terms of valuable-component grades, modes of occurrence, spatial distribution, and orebody structure [10]. Because the ore deposits of the Norilsk mineral resource complex exhibit high natural variability in in-situ ore quality characteristics ($\sigma = 0.07–1.40$; $v = 39.30–111.50$), they may be regarded as geologically complex from the standpoint of maintaining a stable mineral composition of the ore during underground mining. Therefore, the conceptual basis for developing underground systems for copper-nickel ore quality management should incorporate an updated set of methods and models. Their use in mine design and process engineering decision-making can improve the quality of the mined material despite the rapid deterioration in the quality of commercially mineable reserves of the principal ore types.

The range of ore quality fluctuations within the ore streams can be reduced most effectively within the underground mining system. For example, after blasting, the range of variation in the quality characteristics of blasted disseminated ore is smaller at the production face and in LHD buckets than in rail cars. One of the principal causes of ore quality destabilization during the extraction of disseminated ores is the use of a caving method, for which ore loss and dilution rates reach 15–20% or more. Another cause is natural particle-size segregation during underground mining operations and material-handling processes.

Under mine operating conditions, particle-size segregation should therefore be considered as a phenomenon that can be used beneficially to increase the grade of valuable components in the mined ore by rejecting waste rock and part of the subeconomic ore at the mining stage.

It should be taken into account that, under actual underground mining conditions, in-mine preconcentration is expected to reduce the ore tonnage delivered to the processing plant by more than 50%. The in-mine preconcentration model also did not

account for the decrease in metal grades in the mined material from the production face to the surface or the resulting decline in ore-grade contrast. Consequently, when copper-nickel ores exhibit relatively low grade contrast, the use of preconcentration alone within the mine production chain is unlikely to achieve adequate throughput.

For complex-structured orebodies, a promising approach should therefore combine separation with the controlled addition of subeconomic ore from the orebody contact zone. As demonstrated in [11], accounting for the exponential decline in valuable-component grades within the contact zone provides a sound basis for incorporating part of the subeconomic ore into the main ore stream without reducing concentrate quality, thereby increasing the overall recovery of the ore reserves. Accordingly, optimization of technologies intended to improve ore quality requires an integrated solution aimed at establishing a combined production quality management system that accounts for segregation of the mined material. The studies described above therefore provide a basis for formulating the fundamental principles underlying process engineering solutions incorporated into underground mine designs for copper-nickel ore quality management [12, 13].

A classification of equipment and methods for ore grade stabilization, including their respective indices and operating principles, was proposed in [12]. A substantial natural blending effect was shown to occur when mined ore is stored in stockpiles, ore passes, or bins [5, 14]. On this basis, the present study recommends converting the underground mine crushing and storage complex into a blending and stabilization facility with a predicted blending coefficient of $K_\beta = 1.6\text{--}2.8$ (Fig. 6). The underground ore-stream quality management system is designated as 1–A–A₇–2–B–B₁₄. This designation indicates that, at the first management stage, organizational measures are implemented to establish the target average ore quality in specified quantities of mined ore through real-time mine planning and hourly intrashift control. At the second management stage, production and process engineering measures ensure the uniform distribution of the established average metal grades throughout the mine-wide ore stream through the use of a specially designed stockpile-based blending facility [4].

The overall underground ore quality management system should include an information and control architecture comprising two subsystems: an information subsystem for real-time ore quality management and a production and process engineering subsystem (Fig. 7).

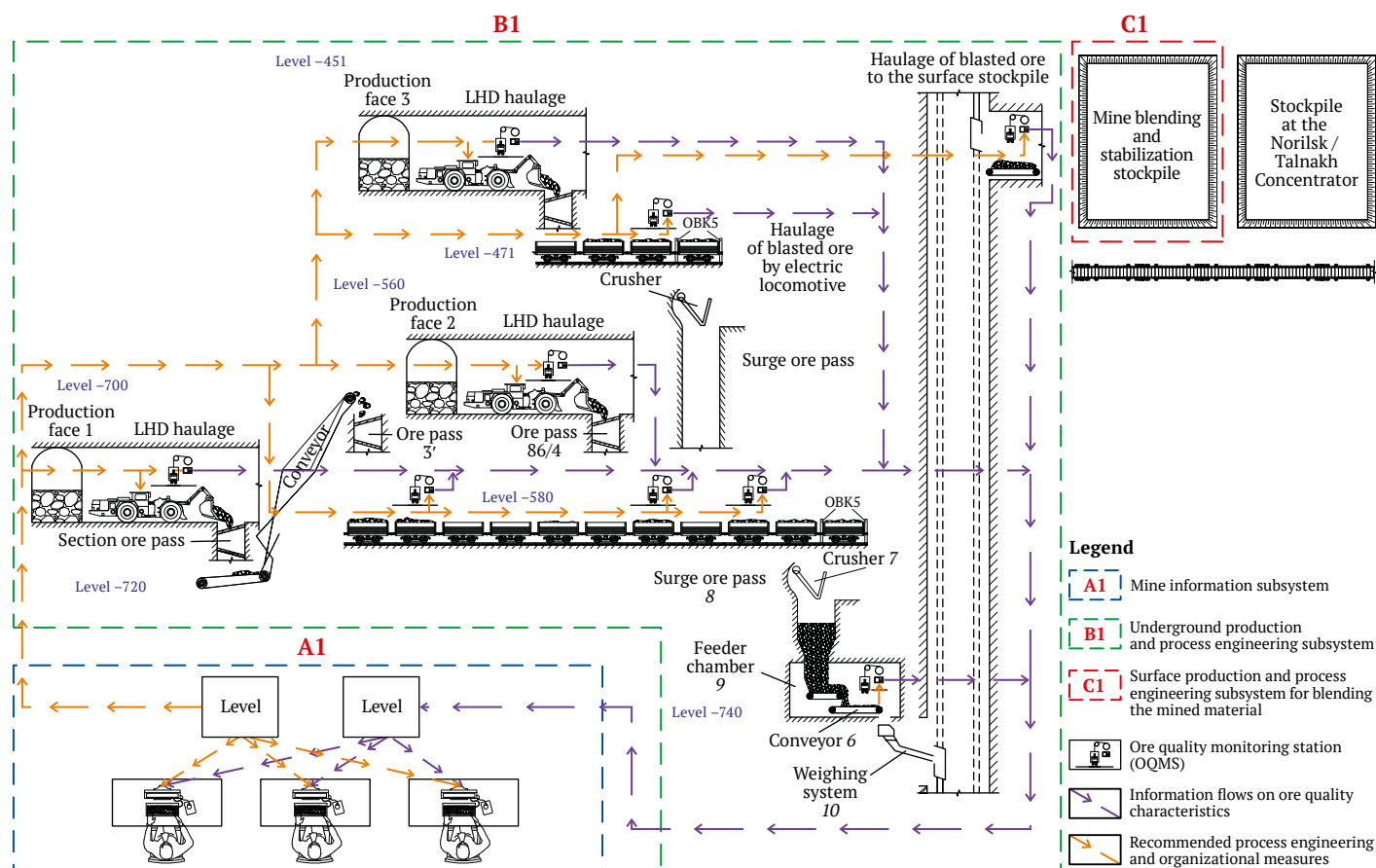


Fig. 6. Flowsheet of the underground ore-stream quality management system

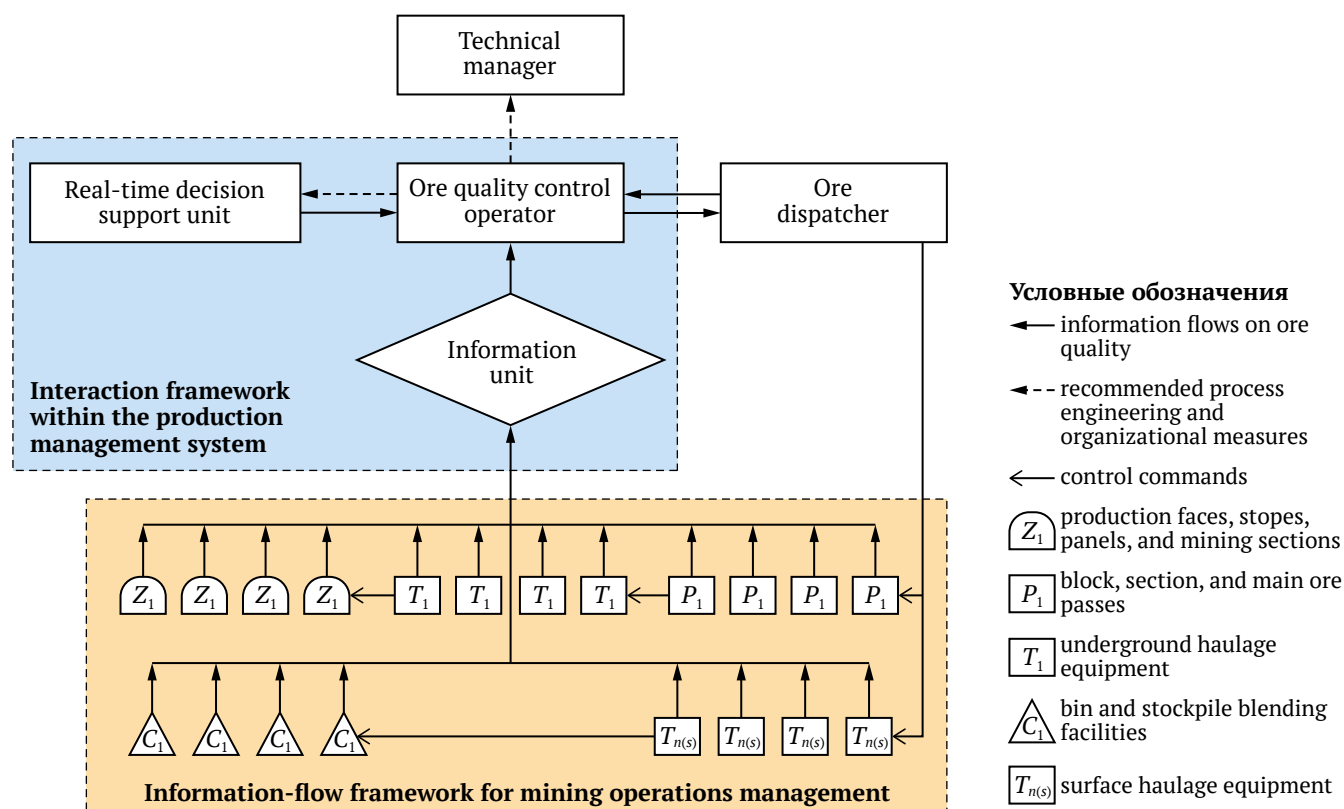


Fig. 7. Model of the information and control architecture of the underground ore quality management system during mining

The production and process engineering subsystem has a conventional configuration and encompasses the organization of production operations and processes required for mining. By contrast, the modern real-time information and control subsystem must be developed substantially anew using radiometric and electronic equipment functionally and structurally integrated into the mine production chain. This subsystem should provide centralized real-time monitoring of metal grades throughout the mining process: at the production faces; upstream and downstream of ore passes; in rail cars on haulage levels; in skips before hoisting; and in bins, material-transport streams, and stockpiles. Ore quality monitoring stations may be used for this purpose [15, 16].

Introducing real-time ore quality monitoring equipment into mining operations and integrating it into a unified underground mine production system will accelerate data processing, analysis, and operational response by improving the mobility and access to information of engineering and technical personnel. This will, in turn, improve real-time intrashift planning and control over production processes aimed at meeting the planned ore quality and production-volume targets.

Key findings

To ensure the long-term security of the mineral resource base of the Norilsk mines, the fundamental principles of the strategic approach to optimizing process engineering solutions for copper-nickel ore quality management in underground mining should be reconsidered.

As the Norilsk deposits are progressively depleted and ore quality variability increases during underground mining of copper-nickel ores, improvements in mined ore quality require an innovative and integrated approach. This approach should involve modernization of mine flowsheets through the combined use of blending and stabilization methods and segregation-based separation methods for ore quality management during mining.

Underground mines should make use of their available process capacity for blending the mined material in order to manage product quality. The most promising option is to locate blending and stabilization facilities at the surface, at the end of the mine production chain.

Under specific mine conditions, particle-size segregation should be used beneficially to reject the low-quality fraction from the total mined material and increase the grade of valuable components in the ore streams.



Ore quality management during mining should be implemented as a multistage process. Accordingly, the information and control subsystem of a modern underground mine should be upgraded to provide mine management personnel with data on changes in ore quality variability parameters throughout the mining process, from the working face to the surface. Digitalization of data flows in mine production management will create an integrated information environment and ensure the transparency and accessibility of reliable ore quality data. These data provide the basis for process engineering solutions incorporated into mine designs to increase metal recovery by stabilizing grades in the mill feed.

A comprehensive basis for mine design and process engineering solutions aimed at optimizing the formation and stabilization of mined ore quality

is provided by establishing a combined production quality management system in the underground mine. Such a system should account for segregation of the mined material and incorporate underground preconcentration technology to increase the grade of valuable components in the mined ore.

Conclusion

The fundamental principles developed from the research findings and engineering studies provide a conceptual basis for mine design solutions incorporating innovative modernization of process flow-sheets and optimization of underground systems for copper-nickel ore quality management. This approach can improve the economic performance of the entire mining and metallurgical complex of the Norilsk industrial region.

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Pilot testing of sylvinite ore reserves re-mining method

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Abstract

As part of the previously justified mining method for the Verkhnekamsk potash salt deposit (VPSD) – which involves the re-extraction of reserves from the KrII sylvinite layer in areas previously mined with “yielding” pillars – pilot tests of the method were conducted. Their primary objective was to assess the technical feasibility and safety of stoping in the KrII* layer – a natural-technogenic rock mass – as well as to refine the mining-geological and mining-technical conditions for re-mining. The results of the pilot testing and the analysis of the observations of the workings condition in the re-mining area showed that, when exposed, the weakly consolidated caved rock is prone to inrush. The most dangerous areas in this regard are sections where old workings have been undercut by inclines, as well as areas near abandoned pillars with increased rigidity. Cyclic variability in the physical and mechanical properties was observed in the KrII* natural-technogenic layer. Specifically, the rock strength ranges 0.4 to 3.8 MPa in the zone of rock caving to 16.2 to 24.0 MPa in pillars. The displacement rates of a working contour in the KrII* natural-technogenic layer are generally 22–28 mm/year and fall within the range typical of the standard mining-geological conditions of the VPSD. The stability of the workings enclosing (surrounding) rock mass was assessed as satisfactory. The implementation of the re-mining method opens a prospect of increasing the recovery of sylvinite ore from the KrII layer.

Keywords

saliferous rocks, re-mining, natural-technogenic layer, room-and-pillar mining method, pilot testing, physical and mechanical properties, monitoring/observation, deformation and stability of mine workings

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РАЗРАБОТКА МЕСТОРОЖДЕНИЙ ПОЛЕЗНЫХ ИСКОПАЕМЫХ

Научная статья

Опытно-промышленные испытания технологии повторной отработки запасов сильвинитовой руды

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Аннотация

В рамках ранее обоснованной системы разработки Верхнекамского месторождения калийных солей, заключающейся в повторной выемке запасов сильвинитового пласта КрII на участках, отработанных с «податливыми» целиками, проведены ее опытно-промышленные испытания (ОПИ). Их основной



целью являлась оценка технической возможности и безопасности проходки очистных выработок в сформированном природно-техногенном массиве пласта КрII*, а также в уточнении горно-геологических и горнотехнических условий повторной отработки. Результаты ОПИ и анализ наблюдений за состоянием выработок на участке повторной отработки показали, что при вскрытии слабоконсолидированные обрушенные породы склонны к вывалообразованию. Наиболее опасными в этом отношении являются участки подсечки старых выработок уклонами, а также зоны вблизи оставленных целиков повышенной жесткости. Для природно-техногенного пласта КрII* установлена циклическая изменчивость физико-механических свойств. В частности, предел прочности пород изменяется от 0,4–3,8 МПа в зоне обрушенных пород до 16,2–24,0 МПа в целиках. Скорости смещения контура выработок в природно-техногенном пласте КрII* в основном составляют 22–28 мм/год и укладываются в диапазон значений, характерный для типовых горно-геологических условий ВКМКС. Устойчивость приконтурного массива пройденных выработок оценивается как удовлетворительная. Реализация технологии повторной отработки открывает перспективу повышения извлечения сylvинитовой руды с пласта КрII.

Ключевые слова

соляные породы, повторная отработка, природно-техногенный пласт, камерно-столбовая система разработки, опытно-промышленные испытания, физико-механические свойства, мониторинг, деформация и устойчивость выработок

Финансирование

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Introduction

The term “re-mining (re-extraction) of mineral resources” refers to the extraction of reserves that were previously lost (undermined, caved, or left in pillars) or left in the subsoil because their initial extraction was uneconomical at operating or closed mines [1, 2]. Depending on the minerals being extracted and the primary mining method used, different re-stopping processes are employed at different deposits. For instance, the re-mining of ore deposits is primarily based on the additional extraction of caved rock mass from stoping zones [3–5]. To date, this method has found the widest application at coal deposits mined using both breast stoping (longwall system) [6, 7] and the Silesia method [8, 9].

The extraction of mineral using the Silesia/room-and-pillar mining methods opens up broad prospects for the use of methods that allow for the re-extraction of ores left in rib, barrier, and panel pillars [10–12]. There are quite a few technical approaches to implementing re-extraction in such conditions [13, 14], including two-stage stoping [15–17].

In the 1980s and 1990s, attempts were made to carry out planned two-stage mining of sylvinite deposits within the Verkhnekamsk potassium salt deposit (VPSD), the largest of its kind in Europe. The essence of this method is as follows [17]: during the

initial stage of layer mining, wide “rigid” rib pillars were left in place, after which the stopes were filled with salt waste, primarily using a hydraulic method. In the second stage, after the filling mass has reached the required strength, the rib pillars were partially extracted. It should be noted that two-stage stoping has not become widespread at the Verkhnekamsk potash salt deposit mines. This is due to a combination of factors: the complexity of coordinating stoping and backfilling operations; the deterioration of the physical and mechanical properties of the rock in the pillars as a result of wetting by backfill brines; the complex nature of deformation in the heterogeneous pillars formed during the second stage; the need to maintain the development workings for an extended period; and so on.

In addition to the extraction of pillars resulting from the commissioning of the second stage, re-mining methods also include the recovery of reserves previously classified as non-commercial due to unfavorable economic conditions and/or technological limitations. Specifically, for the Starobinsky potash salt deposit (Belarus), a study was conducted on the feasibility of extraction of the partially mined out 4th sylvinite layer, which was left behind during the extraction of the underlying layers 2, 2–3, and 3 of the Third Potash Horizon [18–20].

Another approach within the framework of potash deposit re-mining method involves the extraction of reserves from a natural-technogenic layer under VPSD conditions [21]. This layer was formed in areas where room-and-pillar mining system was used to develop the KrII sylvinite layer, with leaving “yielding” rib pillars in the mining field of the BKPRU-2 mine operated by PJSC Uralkali. Under the influence of rock pressure, these pillars deform and fracture, causing the stopes to be filled primarily with their fragments and resulting in the formation of a continuous rock mass with periodically varying physical and mechanical properties. As shown in [21], the secondary natural-technogenic KrII* layer has sufficient recoverable thickness and acceptable strength characteristics that creates the conditions for the re-extraction of sylvinite ore reserves under VPSD conditions. This method is particularly relevant for mines in the final stages of operation, as it is designed to extend the mine's operational life. In addition, its implementation will make it possible to include reserves previously classified as production losses in the extraction process, thereby increasing the overall recovery factor for a mineral from a producing layer.

When selecting process flow sheets for re-mining of the KrII* natural-technogenic layer using room-and-pillar method, the key issues are determining the bearing capacity and deformation behavior of heterogeneous rib pillars composed of rock masses with varying strengths. This is essential for ensuring safe conditions for re-mining of the overlying watertight rock strata and for protecting a mine from flooding. A solution to these problems obtained for the first time on the basis of quantitative estimates

using numerical simulation methods [21] provides the scientific basis for demonstrating the feasibility of re-mining of sylvinite ore reserves.

The implementation of the proposed method for stoping within the KrII* natural-technogenic layer requires the preparation of a “Safety Justification for a Hazardous Production Facility” and the conduct of pilot tests. This work was carried out between 2022 and 2024 and included scientific support, which involved monitoring the condition of the mine workings. The main objective of the testing was to assess the technical feasibility and safety of driving and stoping within the KrII* natural-technogenic layer rock mass, as well as to clarify the mining-geological and mine engineering conditions for driving and stoping.

In light of this, the following main objectives were set for the testwork:

- 1) determining the optimal conditions for implementing the method for the re-extraction of reserves from the KrII* natural-technogenic layer;
- 2) clarification of the geological features and physical and mechanical properties of the KrII* layer;
- 3) monitoring deformation processes in mine workings and assessing their stability.

Testing conditions and key technical solutions

A 170-by-200-meter area within the mining field of the BKPRU-2 mine was selected for pilot testing of the method for re-mining of the KrII* natural-technogenic layer. The testing area is located in a zone where the initial mining of two gently dipping, closely spaced sylvinite layers was performed in the first half of the 1970s, leaving “yielding” rib pillars in the KrII layer with the parameters shown in Fig. 1, *a*.

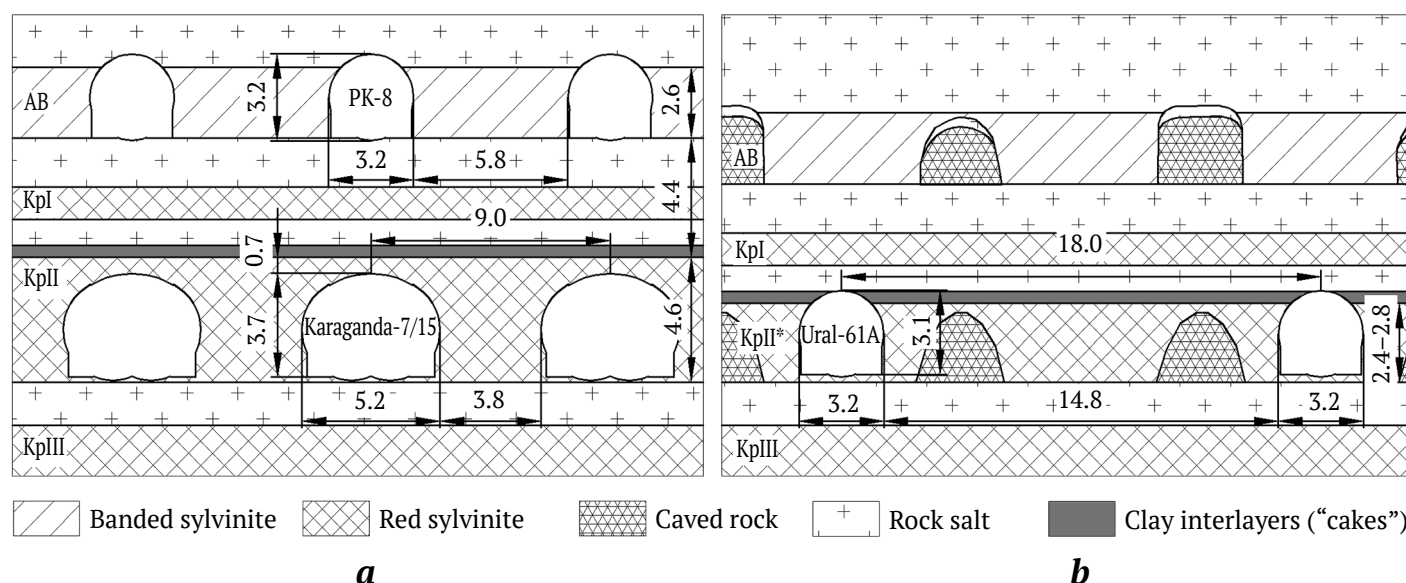


Fig. 1. Parameters of the initial mining of the KrII and AB layers (*a*) and the re-mining of the natural-technogenic KrII* layer (*b*) in the testing area

During the initial mining of the KrII layer in the area under consideration, a protective silvinite member approximately 0.7 m thick was left in the roof. Subsequently, the stopes were filled primarily with fragments of this layer and the marginal parts of the pillars, and to a lesser extent with host rocks. Under the influence of moisture and rock pressure, the caved rocks eventually consolidated, forming a continuous and relatively monolithic rock mass. Thus, the naturally and technologically formed KrII* layer 2.4–2.8 m thick composed of alternating of caved rock and remnants of pillars, has a KCl content close to that found in natural potash layers, which makes it commercially viable for re-mining.

The paper [21] examines various options for re-extraction of the KrII* natural-technogenic layer using room-and-pillar method. The calculations showed that, at the testing area, safe conditions for re-mining of the KrII* layer were achieved by driving single-pass stopes with an Ural-61A shearer, coaxially with first-stage rib pillars with a twofold increase in the inter-axis distance. The parameters for implementing this process flow are shown in Fig. 1, b.

The pilot tests involved the re-extraction of reserves from the testing area – occurring in the first-stage pillars – by driving ten testing stopes in the meridian direction. The area preparation involved driving transport and ventilation inclines from the main-direction workings, which had been driven through the underlying rock salt layer, as well as an

extraction drift through the KrII* layer. To maintain stability, the extraction drift is located in the rigid pillars area between the starting workings of the primary mining stage. At the end of a pass, the stopes were connected by ventilation connections. In addition, several stopes were connected by research connections in order to conduct field research. In order to experimentally evaluate a stable mine working roof span within the natural-technogenic layer, some connections were driven in two passes across the width, with the passes overlapping. The location of the research workings relative to the previously driven development workings and stopes is shown in Fig. 2.

The stability of the workings in the natural-technogenic layer was ensured by undercutting all three clay “cakes” in the roof, as well as by using roof bolting at the junctions of the workings and in sections where the working width exceeded one shearer pass, when excavating caved unstable rock in former stopes.

To monitor the condition of the workings and the load-bearing elements of the room-and-pillar system for re-mining, underground observation stations (OS) were set up at the testing area to monitor the displacement of the workings surrounding rock mass. In addition, the scientific support for the testing project included a detailed study of the geological setting and physical and mechanical properties of the rocks in the KrII* natural-technogenic layer, as well as an assessment of their gas-bearing capacity.

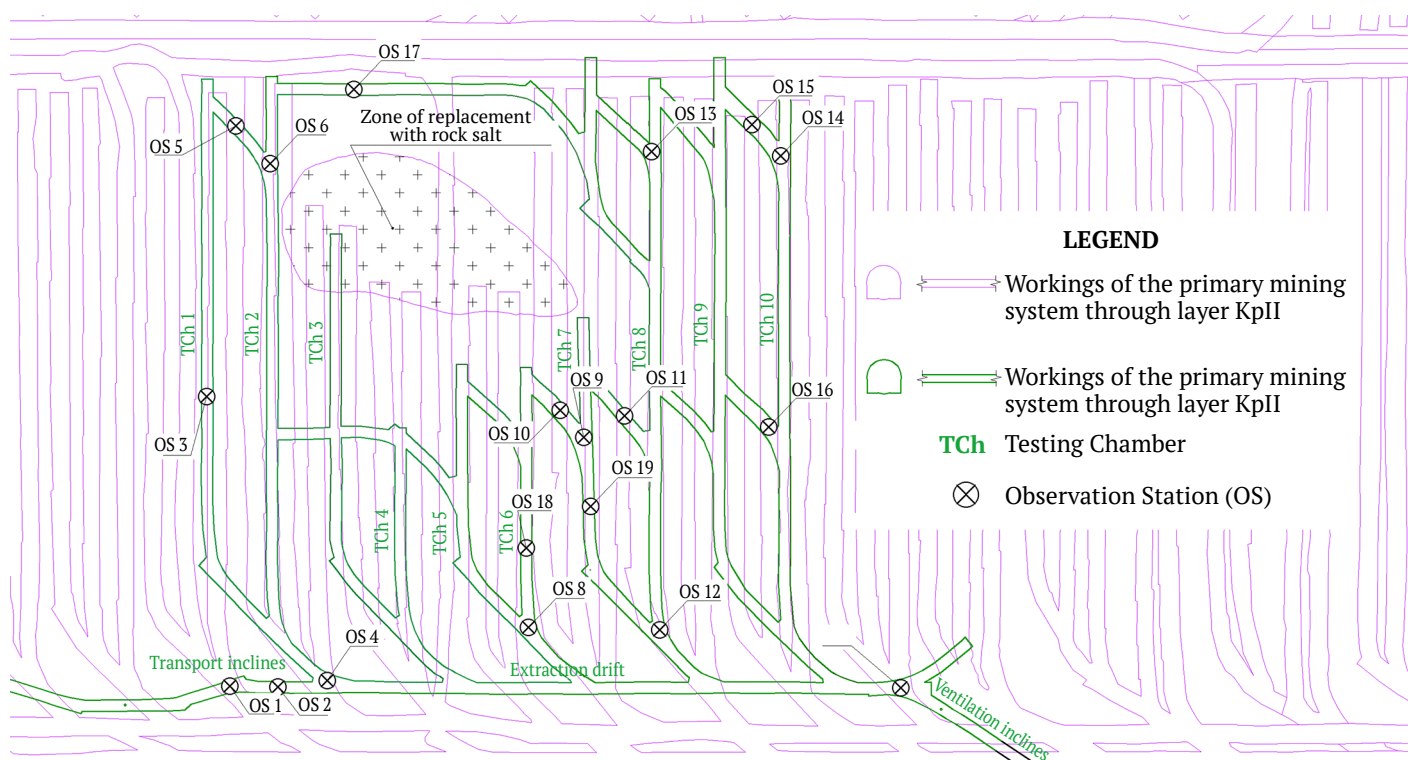


Fig. 2. Testing area workings layout

Features of driving workings in the natural-technogenic layer rock mass

During driving the research workings within the KrII* natural-technogenic layer at the testing area, it was found that the voids in the previous stopes are currently almost entirely filled with blocky material that was cemented to varying degrees (Fig. 3). The worked-out stopes were filled in several stages. First, the protective member of sylvinitic layer left in the roof fails. Next, vertical compression of a pillar caused it to expand laterally. As a result, the margin parts were broken down into a system of contour sub-parallel prismatic blocks. These blocks shifted and tilted into the mined-out space, dislodging cuttings from the floor and causing it to accumulate in the center of a stope.

The formation of a system of subvertical fractures in the central part of a pillar results in the formation of a trapezoidal skeleton bounded by faults, within which the lower part of the KrII layer (sublayers, SL 5–7 in Fig. 3) remains virtually unchanged in thickness. At the same time, the reduction in the height of the pillar is primarily due to creep deformations, which predominate in the upper part of the layer (sublayers, SL 1–4), where more than half of the layer's thickness is lost. At the same time, the rock filling the stopes contains virtually no material from the overlying layers, indicating that the AB-KrII* technical interlayer has not caved. Thus, the stopes were filled primarily through the redistribution of rocks from the KrII sylvinitic layer.

As a result of transverse deformation, the width of the pillars increased to 5.0–6.0 m across the layer floor, while the width of the caved rock area at the site of the former stopes was 3.0–4.0 m. The thickness of the KrII layer within the contour of the former stopes decreased to 2.4–2.6 m, and, in the pil-

lars, to 2.6–2.8 m, which represented 52–57% and 57–61%, respectively, of the initial thickness of 4.6 m (see Fig. 1).

In the course of re-mining at the testing area, field tests were conducted on large-scale prismatic specimens measuring 400×200×200 mm to determine their uniaxial compressive strength, in accordance with the methodology described in [21].

The results of the field tests on large-scale specimens at the testing area revealed significant variation in the strength characteristics of the KrII* natural-technogenic layer. For the rock caved into the stopes, the compressive strength ranges from 0.4 to 3.8 MPa, with an average value of 2.0 MPa. The strength of the rocks of rib pillars ranges from 16.2 to 24.0 MPa, with an average of 20.6 MPa, which roughly corresponds to the natural properties of the intact KrII layer as determined during background testing.

Thus, within the re-mining area with newly driven stopes, the strength properties of the KrII* natural-technogenic layer are characterized by an extremely heterogeneous distribution. In the central part of the pillars, the strength is at the background levels for the VPSD and decreases by nearly an order of magnitude in areas of caved rock.

A geological study of core samples from boreholes drilled into the roof of the workings at the testing area revealed zones of stratification and loosening in the rocks of the AB-KrII* parting above the former stopes. These often contain accumulations of condensate brines that have migrated from the floor of the stopes created in the overlying AB layer. In addition to the parting, local brine seepage is observed when the shearer cuts through rock that has caved into the stopes. The nature of the brines is due to the long-term condensation of moisture from the air in the old mine workings.

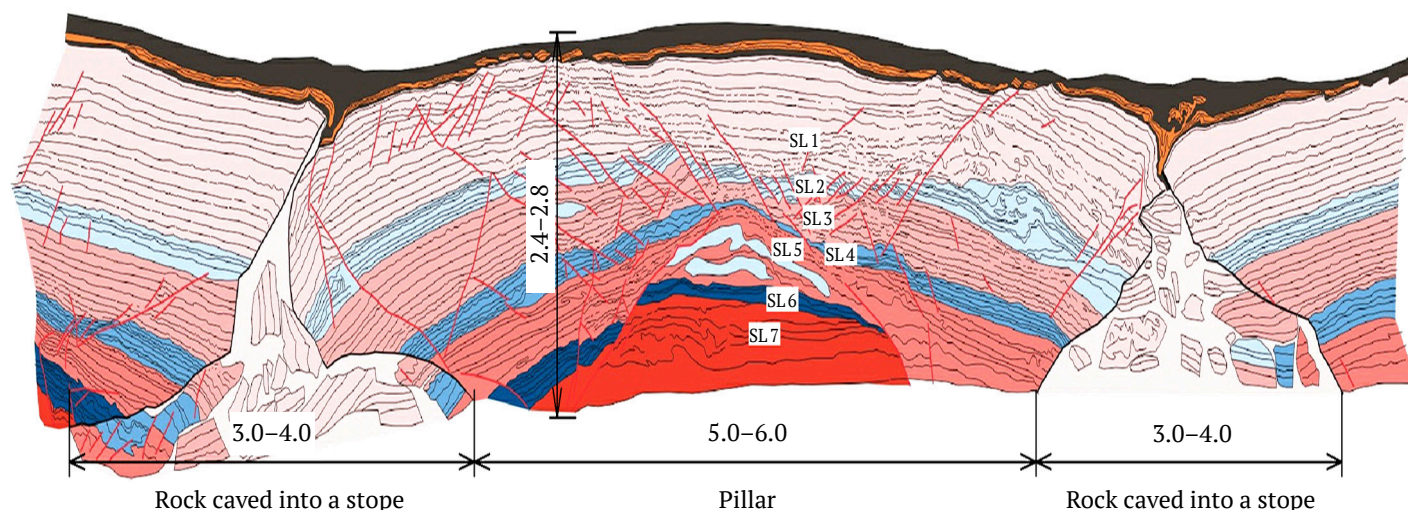


Fig. 3. Structure of the KrII* Natural-Anthropogenic layer

During the research driving at the testing area, it was determined that various metallic objects may be found in the caved rock; these include fragments of mechanisms, assemblies, and equipment parts left during the initial mining of the panel, as well as other debris (conveyor belts, wood, etc.). This leads to increased wear on the shearer's teeth and makes driving more difficult. In addition, the increased clay content and wetting by condensation brines contribute to the high viscosity of the rock that caved into the stopes. As a result, the driving rate in the layer decreases and the number of tool-related incidents increases.

The complex hypsometry of the KrII* natural-technogenic layer, which differs significantly from the natural conditions under which the VPSD sylvinitic layers occur, also complicates driving and stopping.

The natural-technogenic layer contains voids ranging from small, irregularly or round-shaped caverns with volumes of several cubic centimeters to voids larger than one cubic meter, which are parts of the first mining stage workings left unfilled with rock mass fragments (Fig. 4). The following factors are likely causes of the void formation:

- uneven tilting of the opposite walls of the primary stopes during the deformation of the yielding rib pillars due to the heterogeneous geologic and mining conditions of the KrII layer (folding, variations in occurring depth, thickness, etc.) and local deviations from the pillar design parameters;

- the location of the primary workings near high-stiffness (rigid) pillars, which thus protect them.

If large voids are reopened or if they are located close to the research workings, there is a risk of acce-

lerated deformation of the contour, leading to a loss of its integrity in the form of fracturing, delamination, and roof breaks. Safe driving in the specified zones of the natural-technogenic layer must include the following measures:

- the creation of a safety zone near the contour of a rigid pillar, within which mining operations are prohibited during re-extraction;

- use of a semi-automatic (remote) control mode for a shearer;

- the use of additional methods to monitor the condition of the workings enclosing rock mass (drilling blast holes and performing video endoscopy of the rock mass, monitoring displacements of the workings contours).

Monitoring the condition of workings in a natural-technogenic layer

Monitoring of the displacement of the workings enclosing rock in the KrII* natural-technogenic layer was carried out at underground observation (monitoring) stations (OS) set up in the testing area. The locations of the stations within the testing area are shown in Fig. 2. A long-term observation station consists of deep and contour benchmarks (mine survey plugs) installed within a working cross-section (Fig. 5). The free end of the benchmarks has a mounting hole for attaching them to a measuring device – a belt extensometer. To assess the impact of various factors (the presence of rock caved into the stopes, the width of the working face, support, etc.) on the magnitude of rock displacements in the surrounding rock mass, the stations were installed under a wide range of geological and mining conditions.



a



b

Fig. 4. Appearance of the workings driven through the KrII* natural-technogenic layer:

a – an intersection of rock caved into the stope on the right wall of the working;

b – an outcrop of weakly consolidated rock on the left wall of the working

Monitoring of the rock mass displacement was performed by periodically measuring the distances between benchmarks installed in opposite parts of a working (roof-floor, left-right walls). During the initial stage of the observations, periodic measurements were taken once every 3 months on average. Subsequently, the frequency of measurements decreased depending on the intensity of the rock mass displacement.

Based on the observation results, graphs were plotted showing the displacements of the benchmarks as a function of the time elapsed since a station was established. In addition, the average annual displacement rate of the enclosing rock mass was calculated for all observation stations.

As an example, Fig. 6, *a* shows graphs of the vertical displacements of the roof rock at observation station 4, located at the junction of the extraction drift and the starting research stopes 1–2 (see Fig. 2). An analysis of the monitoring data revealed that the maximum rate of vertical displacement of the roof strata took place during the initial observation period. Subsequently, the deformation rate gradually decreases, stabilizing at 22–28 mm/year. A relationship was also established between the magnitude of the displacements and the distance to the roof of a working: the minimum values were recorded at a distance of 6.0 m from the roof, while the maximum values were recorded directly at a working contour.

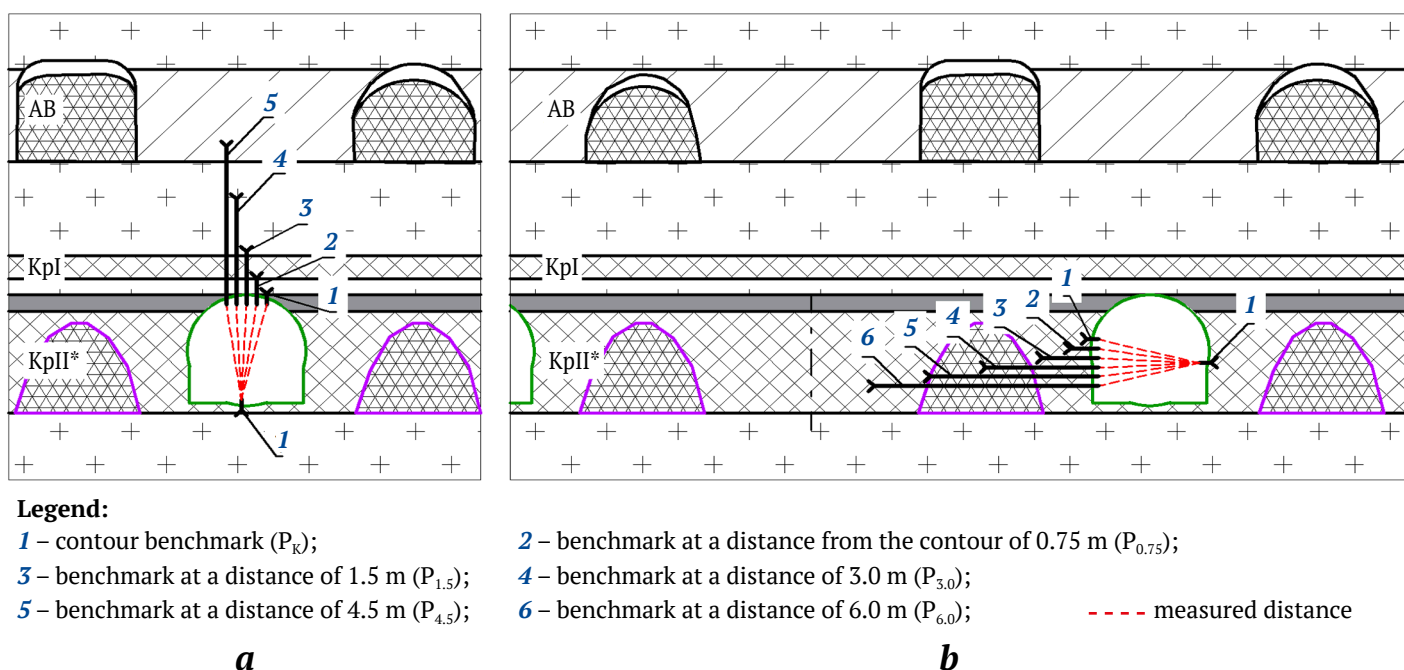


Fig. 5. Scheme of stations positioning for monitoring the displacement of the enclosing rock mass: *a* – mine workings roof; *b* – rib pillar

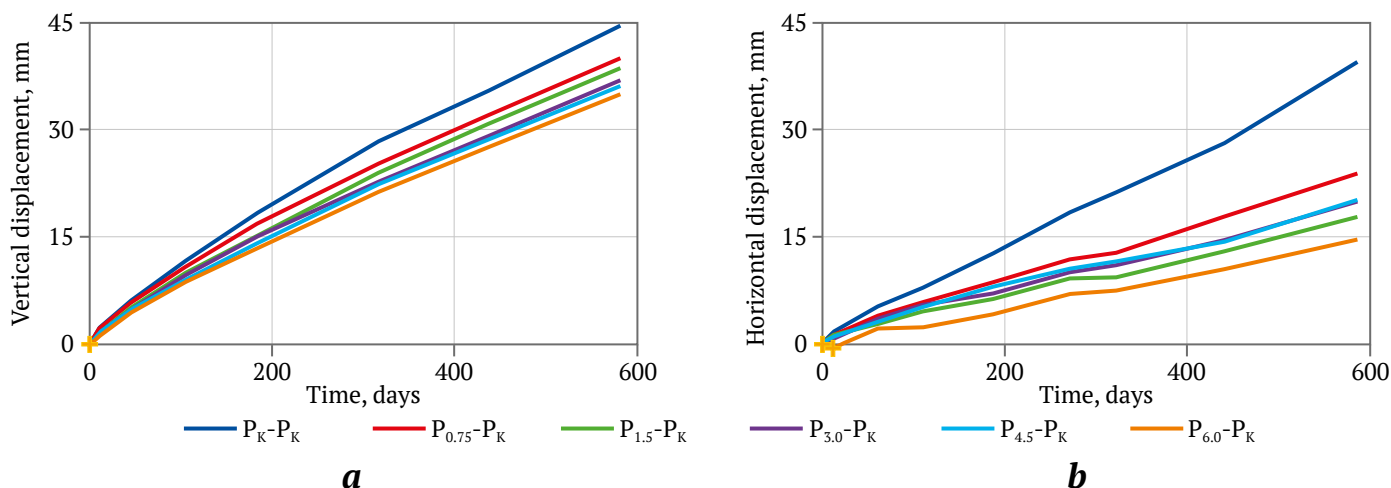


Fig. 6. Rock mass displacement as a function of time:

a – vertical roof displacements at the junctions of workings (OS 4); *b* – horizontal displacements of the rib pillar (OS 3)



A qualitatively similar pattern of deformation in the roof rock, despite quantitative differences in the absolute values of the displacements, was observed at all observation stations equipped with vertically oriented deep benchmarks according to scheme shown in Fig. 5, *a*. At the same time, an analysis of the monitoring results showed that the displacement rates of the contours of the research workings at the points where they intersect caved rock are generally lower than in the previously left pillars in the KrII* layer. This observed pattern is due to the vastly different deformation properties of these structural elements within the natural-anthropogenic layer, which cause a heterogeneous stress field to form within it. As a result, stress concentrations in the primary mining pillars lead to increased rates of displacement of the workings contours relative to parts of former stopes filled with caved rock.

This same reason explains the nonuniform nature of the horizontal displacements of the benchmarks shown in Fig. 6, *b*, which were installed in the re-formed rib pillar according to the scheme in Fig. 5, *b*. Thus, the greatest rock displacements – 39.4 mm – over the observation period were recorded at the pillar contour. As the distance from the contour exceeds 0.75 m, the displacements decrease sharply, stabilizing in the range of 14.5 to 23.8 mm. At the same time, the measured displacement value at a distance of 1.5 m is lower than the values observed in the 3.0–4.5 m distance range. This deviation from the general pattern is due to the fact that the benchmark P_{1.5} is located in the caved rock mass with different deformation characteristics.

Based on the results of workings condition observations in the testing area, the following conclusions can be drawn. When exposed, weakly consolidated caved rock exhibits low stability. The most dangerous areas in this regard are sections where old workings have been undercut by inclines, as well as areas near abandoned pillars with increased rigidity. For the most part, the observed workings contour displacements fall within the range of values typical of the standard mining and geological conditions at VPSD. The stability of the workings enclosing rock mass in the KrII* natural-technogenic layer is assessed as satisfactory.

Conclusion

In order to evaluate the new mining method for the Verkhnekamsk potash salt deposit, which involves the re-extraction of reserves from the KrII sylvinite layer, pilot tests of this method were conducted. The tests were carried out within the mining field of the BKPRU-2 mine operated by Uralkali PJSC in the area where room-and-pillar mining method with “yielding” pillars was previously used.

The tests conducted confirmed the practical feasibility and specific characteristics of driving workings within the KrII* natural-technogenic layer. The technical solutions adopted for the conditions of the testing area ensure the stability of the mine workings throughout the entire period of mining.

The main requirements governing safe mining conditions during the re-extraction of reserves from the KrII* natural-technogenic layer include the following:

- a sufficient degree of compactness and continuity of the formed rock mass, which results from complete backfilling of stopes when using a primary mining method with “yielding” pillars;
- selection of re-mining parameters that ensure rigid support of the overlying watertight strata;
- driving stopes coaxially with the rib pillars of the primary mining system;
- the creation of a safety zone near the contour of a rigid pillar, within which mining operations are prohibited during re-extraction;
- use of the semi-automatic (remote) control mode for a shearer.

Based on the results of a study of the geological structure of the KrII* natural-technogenic layer, sporadic voids were discovered, consisting of areas of the first stage of mining that were not filled with rock fragments. Cyclic variability in the physical and mechanical properties of the KrII* layer has also been established. Specifically, the rock strength ranges from 0.4–3.8 MPa in the zone of caved rock to 16.2–24.0 MPa in pillars.

An analysis of the monitoring data on deformation processes in the mine workings driven in the KrII* natural-technogenic layer showed that the rates of displacement of their contours are generally 22–28 mm/year and fall within the range typical of the standard mining-geological conditions of the VPSD. The stability of the workings enclosing (surrounding) rock mass is assessed as satisfactory.

Despite the broad prospects for applying the re-mining method in the BKPRU-2 mine field, most of which has been mined-out with leaving “yielding” pillars in the KrII layer, additional studies are needed to calculate recoverable reserves to be involved, with a focus on assessing the feasibility of implementing this method at specific stoping areas. This is due to the variability of geological and mining conditions, which are determined by factors such as the parameters of primary mining, the degree of compactness and continuity of the formed rock mass of the natural-technogenic layer, the presence of geological anomalies in the structure of the watertight strata, proximity to the perma-



nent boundaries of mining operations, the presence of protected sites on the Earth's surface, and so on.

Thus, the implementation of this method will make it possible to involve previously written-off

reserves into extraction and increase the overall sylvinite ore recovery factor from the KrII layer. Under conditions similar to those at the testing area, the expected increase in the recovery factor averages 16% [21].

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Mathematical modeling of the reliability of mine ventilation systems using integro-differential equations with additional invariance

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Abstract

Modern mining operations are conducted under complex geological conditions characterized by variable gas-dynamic and aerodynamic parameters, placing stringent demands on the reliability and stability of mine ventilation systems. This study presents the results of mathematical modeling based on integro-differential equations describing unsteady airflow distribution and accounting for distributed parameters, including airway geometry, air leakage, and heat and mass transfer. A non-Lie method was used to identify an additional invariance that cannot be detected by classical Lie–Ovsiannikov symmetry analysis. A conserved quantity – the generalized circulation $\Gamma(t)$ – was identified. It comprises the classical circulation term and an additional term accounting for flow unsteadiness, thereby enabling airflow reversal in diagonal connections to be predicted without iterative recalculation of the entire ventilation network. A numerical algorithm combining finite-difference and finite-element methods was developed. Application of the model to the airflow distribution network of the No. 15 West longwall face identified critical diagonal branches with a high risk of airflow reversal and determined the ranges of the stability indices. The findings provide a theoretical basis for systems designed to provide early warning of emergency ventilation modes and for adaptive ventilation control, with the potential to improve mining safety.

Keywords

mining enterprise, ventilation, modeling, equations, invariance, circulation, method, algorithms, stability, prediction

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ТЕХНОЛОГИЧЕСКАЯ БЕЗОПАСНОСТЬ

Научная статья

Математическое моделирование надёжности вентиляционных систем шахт на основе интегро-дифференциальных уравнений с учётом дополнительной инвариантности

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Аннотация

Современные горные предприятия функционируют в сложных горно-геологических условиях, характеризующихся изменчивостью газодинамических и аэродинамических параметров, что предъявляет повышенные требования к надёжности и устойчивости шахтных вентиляционных систем. В работе



представлены результаты математического моделирования, основанного на интегро-дифференциальных уравнениях нестационарного воздушораспределения с учётом распределённых параметров – геометрии выработок, утечек, тепло- и массообмена. Для исследования скрытых свойств системы применён нелиевский метод, позволивший выявить дополнительную инвариантность, не обнаруживаемую классическим подходом Ли. Установлена сохраняющаяся величина – обобщённая циркуляция $\Gamma(t)$, включающая классический член и добавку, учитывающую нестационарность потока, что даёт возможность прогнозировать опрокидывание струй в диагональных соединениях без итерационного пересчёта всей вентиляционной сети. Разработан численный алгоритм на основе комбинации методов конечных разностей и конечных элементов. На примере схемы воздушораспределения 15-й западной лавы выявлены критические диагонали с высоким риском опрокидывания, определён диапазон изменения показателей устойчивости. Полученные результаты создают теоретическую основу для разработки систем раннего предупреждения аварийных режимов и адаптивного управления вентиляцией, что в перспективе повышает безопасность ведения горных работ.

Ключевые слова

горное предприятие, вентиляция, моделирование, уравнения, инвариантность, циркуляция, метод, алгоритмы, устойчивость, прогнозирование

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Introduction

The reliability of mine ventilation systems is essential to the safety and efficiency of mining operations. Under variable and often extreme external conditions, including fluctuations in pressure, temperature, humidity, and air composition, maintaining stable ventilation performance presents a complex engineering challenge. Integro-differential equations are widely used to describe the processes occurring in such systems because they can account for both instantaneous and cumulative effects, as well as interactions among multiple physical factors. Several issues remain unresolved, including the absence of invariants that remain conserved under abrupt changes in airway geometry, the difficulty of predicting airflow reversal under multiple simultaneous disturbances, and the poor convergence of numerical methods for stiff systems. This study focuses on the first two of these issues.

Conventional analysis and modeling methods do not always provide the required accuracy or solution stability under uncertain and unsteady conditions. In this context, the additional invariance of integro-differential equations is of particular interest. An equation is said to possess additional invariance when certain characteristics of the system remain unchanged under transformations outside the classical symmetry group. This approach can reveal hidden relationships, improve the stability of numerical methods, and provide more robust criteria for the design and operation of ventilation systems.

This study aims to develop and theoretically substantiate a mathematical model of mine ventilation system reliability based on integro-differential equations with additional invariance and to assess its ef-

fectiveness in predicting the stability of airflow distribution under time-varying external disturbances.

Accordingly, the invariant $\Gamma(t)$ is introduced to predict airflow reversal. The convergence of the numerical methods lies outside the scope of this study and requires separate investigation.

To achieve this aim, the study pursued the following objectives:

1. To construct a system of integro-differential equations describing airflow dynamics in a mine ventilation network while accounting for distributed parameters, including changes in airway geometry, air leakage, and heat and mass transfer.
2. To identify additional invariance operators for the resulting system of equations and determine the associated conservation laws.
3. To develop a numerical algorithm for solving the integro-differential equations using a combination of the finite difference and finite element methods.
4. To evaluate the model reliability through a preliminary qualitative comparison with known physical laws and literature data.

The study applies the theory of additional invariance of integro-differential equations to the reliability analysis of mine ventilation systems, providing a potential basis for safer and more efficient ventilation control [1–3].

Materials and methods

The model represents the mine ventilation system as a network of mine workings, including shafts, crosscuts, drifts, and longwall faces, together with fan installations, airflow regulators, and stoppings. The network is treated as a distributed-parameter system. The model describes unsteady aerodynamic processes



involving changes in pressure, airflow rate, temperature, and air composition in each network branch.

The study used an integrated approach combining theoretical analysis, mathematical modeling, numerical calculations, and field validation to investigate the additional invariance of integro-differential equations and its application to mine ventilation system reliability.

Theoretical analysis. Existing integro-differential models of airflow distribution in mine ventilation networks were reviewed. Particular attention was given to the conditions under which solutions remain invariant under specified transformations, allowing additional symmetries and the associated conservation laws to be identified.

Mathematical modeling. Integro-differential models were developed from the fundamental equations of aerodynamics and heat transfer, accounting for mine-specific conditions, including changes in airway geometry, air leakage, temperature, and humidity. The mine ventilation network was represented as a directed graph. Its edges correspond to mine workings with specified aerodynamic resistances, whereas its nodes represent airway junctions, splits, and intersections. For each edge, the model includes a momentum equation for airflow based on a generalized Darcy–Weisbach equation incorporating inertial terms, a continuity equation, and a heat-balance equation accounting for heat exchange with the surrounding rock mass and moisture release. Invariance was analyzed using group analysis of differential equations and Lie group theory.

Numerical methods. Numerical algorithms combining the finite difference and finite element methods were developed and implemented to solve the resulting integro-differential equations. The model inputs include airway geometry (length, cross-sectional area, and shape); the aerodynamic resistances of fixed and adjustable network components; fan characteristics expressed as pressure-quantity characteristic curves; and the initial and boundary conditions, including pressure, temperature, and humidity at the surface and at the working face. The model outputs are the time-dependent distributions of airflow rate, static pressure, temperature, and methane concentration across the network branches, together with stability indicators such as the pressure-based stability margin and the risk of airflow reversal in diagonal branches. A series of numerical tests was performed for different ventilation-network configurations to assess the stability and reliability of the solutions.

Qualitative reliability assessment. The obtained theoretical and numerical results were compared with known physical laws and literature data on airflow parameters in coal mines. This allowed

a preliminary conclusion on the qualitative agreement of the proposed model with the physics of the processes. A full-scale verification with quantitative comparison against field data requires a separate study and was not performed in this work.

Together, these methods provide a framework for evaluating the role of additional invariance in mine ventilation system reliability [2, 4, 5].

Theoretical framework: Fundamental approaches to mathematical modeling of mine ventilation systems

General description of the modeled system

A mine ventilation system is a complex distributed network of mine workings, including shafts, cross-cuts, drifts, and longwalls. Unsteady airflow through this network is driven by pressure differences generated by fan installations, natural ventilation pressure, and heat release. The mathematical description of such a system includes:

- a momentum equation for airflow based on the generalized Darcy–Weisbach relation with inertial terms;
- a continuity equation;
- a heat-balance equation accounting for heat exchange with the surrounding rock mass and moisture release;
- an equation of state for air.

In compact operator form, the system can be written as:

$$L_0(\partial)u(t, x) = 0, \quad L_1(\partial)u(t, x) = 0,$$

where $u = (Q, p, \rho, T)^T$ is the vector of unknown field variables – airflow rate, pressure, density, and temperature – and L_0 and L_1 are differential operators containing partial derivatives with respect to time and spatial coordinates. Additional relations, including junction conditions at the network nodes and boundary conditions at the surface and the working face, define the structure of the ventilation graph.

Classical Lie–Ovsiannikov and non-Lie methods for symmetry analysis

The Lie–Ovsiannikov method is traditionally used to analyze the symmetries of differential equations. It seeks invariance under sets of **first-order differential operators**. Although this approach is effective for a broad class of systems, some equations possess hidden symmetries that cannot be detected by the classical method.

Studies [4–6] proposed an alternative to the Lie–Ovsiannikov method, hereinafter referred to as the non-Lie method, for identifying additional invariance in systems of differential equations. This method has revealed additional invariance in the Dirac and Max-

well equations [7, 8], as well as in several other Poincaré-invariant equations.

Definition. Let $L(x, p)$ be a linear differential operator. The equation

$$L(x, p)\psi = 0$$

is said to be invariant under a set of operators $Q = \{Q_A\}$, if, for each A ,

$$L(x, p)(Q_A\psi) = 0$$

provided that ψ satisfies the original equation. The set Q is referred to as a set of additional invariance operators if its elements cannot be obtained by the classical Lie–Ovsianikov method, for example, when Q_A are integro-differential operators or differential operators of order higher than one.

Unlike classical symmetries, additional invariance describes properties of solutions that remain unchanged under transformations outside the Lie–Ovsianikov symmetry group of the equation. These properties are often associated with integral conservation laws that do not follow directly from the equations of motion.

Application of the non-Lie method to mine ventilation problems

The system of equations describing unsteady airflow distribution in a mine ventilation network, written in operator form, possesses additional invariance. The principal steps of the non-Lie method used to identify this invariance are described in [7, 8–10].

The original system of operator equations is first transformed into a canonical form, namely, a decoupled system in which the individual components of the vector-valued solution no longer depend on one another. This is achieved through an invertible transformation $U(p)$ acting on the solution vector. The transformation substantially simplifies the subsequent invariance analysis.

The solution set of the decoupled system is then examined to identify operators that commute, in a generalized sense, with the differential operator defining the original equation. The analysis shows that such operators exist and form a Lie algebra. In particular, for system (1) with $(m = 0)$, a case analogous to electrodynamics, a nine-dimensional Lie algebra of additional invariance operators was identified. Its basis elements Q_{ab} , where $(a, b = 1, 2, 3)$, are integro-differential operators. Their action on a solution therefore involves both differentiation and integration with respect to the spatial variables.

Explicit expressions for the operators Q_{ab} are provided in [11–13]. For the purposes of this study, their physical implications for mine ventilation are more important than their analytical form.

Physical interpretation

For a closed contour C in the ventilation network, that is, a sequence of mine workings forming a cycle, invariance under the operators Q_{ab} implies the conservation of a quantity referred to here as the generalized circulation:

$$\Gamma(t) = \oint_C (vdl) + \frac{1}{c^2} \int_S \frac{\partial v}{\partial t} ds,$$

where v is the air velocity in the mine working, directed along its axis; S is any surface bounded by the contour C ; and c is the speed of sound in air, approximately 340 m/s under standard conditions.

The first term,

$$\oint_C (vdl),$$

is the classical velocity circulation known from fluid mechanics. The second term,

$$\frac{1}{c^2} \int_S \frac{\partial v}{\partial t} ds,$$

arises from the additional invariance and is absent from conventional ideal-fluid models. It accounts for unsteady effects: changes in air velocity over time produce an additional contribution to the conserved quantity.

Practical significance

The quantity $\Gamma(t)$ is constant in time for any solution of the system. Thus, if $\Gamma(0)$ is known at the initial time, for example, under normal ventilation conditions, $\Gamma(t)$ remains unchanged as the network parameters vary, including airway resistances, fan operating modes, and thermal effects. This remains valid until the solution ceases to exist, for example, when airflow reversal occurs.

This invariant provides a practical tool for mine ventilation analysis. Consider a diagonal connection consisting of two airflow paths and a cross-connection, where airflow reversal may occur. Under the conventional approach, the entire network must be recalculated repeatedly as the resistance of one branch changes in order to determine when stability is lost. With the invariant $\Gamma(t)$, it is sufficient to calculate the invariant once for the critical contour and then monitor the departure of the current system state from the state that would preserve Γ . Once this departure exceeds a specified threshold, the system is approaching airflow reversal.

Applications in numerical modeling and reliability assessment

Additional invariance offers the following practical applications:

1. Verification of numerical schemes. When the equations are discretized, the numerical scheme

should preserve the invariant $\Gamma(t)$ within a small error margin. If the calculated value of $\Gamma(t)$ changes by more than a specified threshold, for example, 0.1%, this indicates insufficient integration accuracy or an inappropriate step size.

2. Prediction of airflow reversal. In diagonal connections, the onset of instability can be determined from the behavior of $\Gamma(t)$ without iterative recalculation of the entire network, substantially reducing computation time.

3. Reliability criteria. Airflow distribution is considered reliable if $\Gamma(t)$ remains within specified limits for all critical contours C over the prescribed range of parameter perturbations.

The following section presents specific examples of using the invariant $\Gamma(t)$ to analyze the stability of the ventilation network of the No. 15 West longwall face (Fig. 1) and compares the results with conventional ventilation pressure analysis.

Results

The effectiveness of the intelligent ventilation control system was evaluated in terms of temperature, diffusion, pressure, pressure drop, gas and dust conditions, including methane emissions and dust concentration, airflow distribution, and equipment operating status.

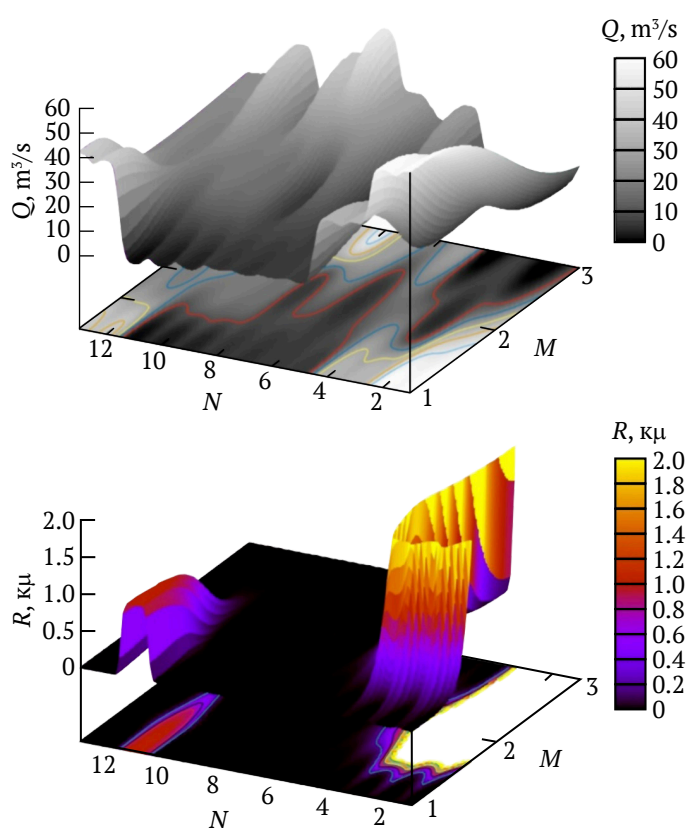


Fig. 1. Aerodynamic parameters of loops decomposed into components of bounded complexity

The decomposition method was refined and applied to determine the qualitative and quantitative characteristics of the ventilation layouts in the production districts under study.

The proposed method identifies loops of varying complexity more efficiently and supports the selection of an appropriate strategy for controlling the ventilation systems of production districts [11–14].

Analysis of the loops in the airflow distribution layout of the No. 15 West longwall face (see Fig. 1) identified 12 active diagonal branches with a wide range of stability-index variation. Airflow reversal was possible in 10 diagonal branches. A further 54 diagonal branches were classified as non-realizable because their stability-index ranges lay outside the resistance adjustment range of the mine airflow regulators. Five diagonal branches were located in return airways. The second stage of the study examined the airflow distribution layout of the No. 15 East longwall face.

Aerodynamic parameters were measured at selected sections of the mine workings.

For each modified loop, aerodynamic resistance was calculated from the corresponding ventilation pressure drop, and the stability criterion was determined at the onset of airflow reversal. Airflow reversal adversely affects the ventilation of production districts, particularly under emergency conditions, and poses a safety hazard. Improper placement of airflow control devices increases the risk of reversal [7, 15].

The ventilation control method is implemented in dedicated software that supports both ventilation control and the selection of mine ventilation layouts. The software also calculates aerodynamic parameters for selected airflow distribution configurations under operating conditions.

The following organizational and technological measures were recommended:

- designing mine ventilation network configurations with specified reliability indices;
- selecting, from the available structural configurations, the one providing the highest operational reliability;
- selecting, from among the available structural configurations, the one providing the highest operational reliability;
- identifying the weakest components of technical systems, adapting integrated reliability assessment methods, and evaluating the effectiveness of structural redundancy;
- adjusting the operating parameters of the main ventilation fans, including temperature and pressure, as a function of the drive-motor rotational speed;



- providing bypasses arrangements, including dedicated bypass channels, alternative paths, and backup routes, to maintain system operation under abnormal or emergency conditions;

- using an isenthalpic throttling process;

- increasing the circulation rate through the heat exchanger in which the compressed air is cooled, using a pump;

- selecting system components from different manufacturers that operate on different physical principles and incorporate different materials, manufacturing processes, and software;

- monitoring indicators of the technological, technical, and technogenic stability of complex technical systems with a variable structure, together with their availability factors;

- monitoring aerodynamic parameters related of air quality;

- using hardware and microcontroller algorithms to adjust the parameters of positive and negative airflow regulators in real time.

Discussion

The findings offer a new perspective on the stability of mine ventilation systems. Unlike conventional approaches based on steady-state ventilation pressure calculations and iterative methods [1, 3, 10], the proposed invariant-based approach provides a quantitative estimate of the stability margin of diagonal connections without repeatedly recalculating the entire network. This shifts ventilation management from reactive response to predictive control, in line with current trends in mine ventilation risk management [2, 6].

When the model was applied to optimize ventilation under increasing production rates, the system demonstrated considerable adaptability to changes in production conditions. This helped prevent system overload and maintain acceptable working conditions. These findings support the conclusions reported in [8, 9] regarding the need to account for the dynamics of gas-dynamic processes.

The physical interpretation of the generalized circulation $\Gamma(t)$ warrants particular attention. Classical circulation reflects steady vortical structures, whereas the additional term containing the time derivative of velocity captures flow inertia and changes in kinetic energy. The invariant $\Gamma(t)$ is sensitive to transient dynamics that conventional models either disregard or can represent only through complex calculations [10, 14]. The model therefore incorporates effects that are difficult to predict within conventional frameworks and builds on the non-Lie method of symmetry analysis [7, 11].

Under geologically complex conditions characterized by frequent pressure fluctuations, the model adapted successfully and helped maintain stable ventilation and safe working conditions. This result is consistent with field observations reported in [5, 13], which showed that conventional ventilation layouts are sensitive to external fluctuations.

Analysis of the loops in the No. 15 West long-wall ventilation network identified 12 active diagonal branches with wide ranges of stability-index variation. Airflow reversal was possible in 10 diagonal branches, five carried return air, and 54 were classified as non-realizable because their stability-index ranges lay outside the resistance adjustment range of the mine airflow regulators. These findings indicate that the invariant-based approach effectively identifies critical network components requiring priority monitoring, consistent with the decomposition methodology used in [11–14].

The adaptive model reduced energy consumption by 10–15% compared with conventional methods, which is consistent with the results of fan operating mode optimization reported in [16, 17].

It should be emphasized that the proposed invariant criterion should be regarded as an indicator rather than a definitive condition of airflow stability. A deviation of $\Gamma(t)$ from its conserved value is highly likely to indicate impending airflow reversal; however, preservation of the invariant does not guarantee stability under abrupt changes in airway geometry, such as roof falls or blockages [10, 15]. Future research should combine the invariant-based approach with catastrophe theory and fuzzy logic methods [2, 9] to develop integrated reliability criteria that account for both deterministic and stochastic factors.

From an engineering perspective, the most promising application is to incorporate the calculation of $\Gamma(t)$ into an automated ventilation control system. This would require monitoring air velocity in the key branches of critical contours using ultrasonic or thermal anemometric sensors and performing numerical integration with a time step matched to the sensor sampling interval [4, 14]. If $\Gamma(t)$ moves outside the specified threshold band, the system could issue a warning or initiate a preemptive adjustment of fan operating conditions [12, 16, 17]. As demonstrated in [18], appropriate selection of motor ratings and operating modes for ventilation fans can substantially improve energy efficiency and reduce operating costs, which is consistent with the proposed adaptive-control approach. The model thus provides both a theoretical basis and a practical framework for developing early-warning systems for emergency conditions in coal mines [8, 14].



Conclusions

1. The study developed a system of integro-differential equations describing unsteady airflow distribution in a mine ventilation network. The system accounts for flow inertia, distributed parameters such as airway geometry and air leakage, and heat exchange with the surrounding rock mass, and can therefore represent transient conditions caused by changes in branch resistance.

2. The non-Lie method revealed additional invariance in the system. For a closed contour, the analysis identified a conserved quantity, the generalized circulation $\Gamma(t)$, which comprises the classical circulation term and an additional term accounting for flow unsteadiness. This invariant cannot be detected by the classical Lie–Ovsiannikov method, demonstrating the novelty of the approach for mine ventilation analysis.

3. A numerical algorithm combining finite differences in time with finite elements in space was developed to calculate the distributions of airflow rate,

pressure, temperature, and methane concentration across the network branches. The algorithm also estimates the pressure stability margin and the risk of airflow reversal in diagonal branches.

4. The invariant-based approach requires less computation than conventional ventilation pressure analysis because the stability of diagonal branches can be assessed without iterative recalculation of the entire network as the parameters vary.

5. A preliminary qualitative assessment of the model reliability was carried out based on comparison with known physical laws and literature data. Quantitative verification of the developed model against field data was not performed in this work and requires further research. The practical significance of the results lies in establishing a theoretical framework for predictive stability analysis of ventilation systems: the invariant $\Gamma(t)$ can be used in early warning algorithms for airflow reversal and adaptive fan control, which can ultimately improve mine safety.

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Acoustic methods for detecting and locating fire sources in coal seams: classification and future directions

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Abstract

Underground coal mining is currently facing major challenges; however, demand for coal as an energy source and chemical feedstock is expected to increase in the near future. Accidents remain frequent in coal mining, with approximately 40–50 incidents reported each year. Among the most hazardous accidents are fires caused by spontaneous coal combustion, which can result in fatalities among mine workers, damage to mining equipment, and the loss of mine sections or reserves. Early detection and localization of spontaneous-combustion fires enable fire sources to be identified at an early stage, thereby improving the effectiveness of fire-control and mitigation measures. This analytical review proposes a classification of acoustic methods for detecting and locating fire sources in coal seams, with the origin of the recorded acoustic signal used as the classification criterion. For each method, applicability limits were defined, and its strengths and weaknesses were identified. The results may be used to select the most suitable diagnostic approach for spontaneous-combustion fires under specific operating conditions. Acoustic diagnostics of fire sources in coal seams can detect combustion, determine the coordinates of fire sources, track the spatial extent and temporal evolution of fire boundaries, and identify the fire stage. Based on Professor N.F. Kusov's hypothesis, an instrument was developed to support acoustic monitoring and single-station acoustic direction finding of fire sources in a coal-bearing rock mass. Spontaneous coal fires may also contribute to the formation of zones with elevated radionuclide concentrations. In such cases, acoustic diagnostics may be used to determine the coordinates of these zones.

Keywords

acoustic diagnostics, fire sources, coal seams, single-station acoustic source localization, single-station acoustic direction finding, natural enrichment

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СВОЙСТВА ГОРНЫХ ПОРОД. ГЕОМЕХАНИКА И ГЕОФИЗИКА

Научная статья

Классификация и перспективы акустической диагностики очагов пожара в угольных пластах

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Аннотация

Добыча угля подземным способом в настоящее время переживает кризис, однако в ближайшей перспективе ситуация изменится, так как спрос на энергетические ресурсы и химическое сырье будет только расти. В процессе добычи угля регулярно происходят аварии. Примерно 40–50 случаев в год. Наиболее опасный вид аварии – эндогенный пожар, может привести не только к гибели горнорабочих и уничтожению горного оборудования, но и к потере целого месторождения. Диагностика эндогенных пожаров позволяет на ранней стадии обнаружить очаг, что существенно повышает эффективность мероприятий по локализации и ликвидации аварии. Проведённый аналитический обзор акустических способов диагностики очагов пожаров в угольных пластах позволил разработать классификацию. В ней



классификационным признаком является причина появления регистрируемого акустического излучения. У каждого способа определены границы возможного применения, а также выделены сильные и слабые стороны. Результаты исследований позволяют выбрать оптимальный вариант диагностики эндогенного пожара под свои условия. Акустическая диагностика очагов пожаров в угольных пластах позволяет установить факт горения, определить координаты очагов и пространственно-временное положение их границ, а также стадию пожара. На основе гипотезы профессора Н. Ф. Кусова для реализации акустического мониторинга был изобретен прибор, который позволяет пеленговать из одной точки очаги пожара в угольном массиве. В перспективе эндогенные угольные пожары могут быть источником радионуклидов. Акустическая диагностика в данном случае может быть использована для определения координат зон с самоповышающейся концентрацией радионуклидов.

Ключевые слова

акустическая диагностика, очаги пожара, угольные пласты, оптимальный, концепция неразнесённого слежения, пеленгация с одной точки, естественное обогащение

Для цитирования

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Introduction

Mine fires are among the most hazardous accidents in underground mining. Large amounts of combustible material, including coal and timber supports, together with high-velocity ventilation airflow in mine workings, create conditions for rapid fire development. Toxic and asphyxiant gases can then spread quickly through the mine workings [1].

For any scientific or technological development, defining its scope of application is as important as developing the method itself. This study does not consider methods based on physical principles other than acoustics for detecting fire sources in coal seams. Instead, it focuses on acoustic diagnostics, which cannot fully replace the methods currently used but can provide an important complementary tool. Interest in acoustic methods for detecting and locating fire sources in coal seams has increased in recent years both in Russia [2] and internationally [3–5], including studies involving artificial intelligence [6]. Despite this growing interest, these methods have not yet been systematically described, and no classification has been proposed. As a result, it remains unclear which methods are most appropriate under specific operating conditions. Detecting and locating fires in coal seams is important not only for mine safety but also for related applications, including the identification of zones with elevated radionuclide concentrations.

This study aimed to systematize acoustic methods for detecting and locating fire sources in coal seams and to define the conditions under which they can be applied.

The objectives of the study were as follows:

1. To classify acoustic methods for detecting and locating fire sources in coal seams.
2. To identify criteria evaluating these methods under specific operating conditions.

3. To compare the identified categories of methods according to these criteria.

4. To consider single-station acoustic source localization, including single-station direction finding, as a promising direction in acoustic diagnostics.

5. To examine the identification of zones with elevated radionuclide concentrations in coal as a potential application of acoustic diagnostics of fire sources in coal seams.

Methods

This study reviews existing acoustic methods for detecting and locating fire sources in coal seams, proposes their classification, and provides a comparative analysis. The study combined literature analysis with a synthesis of results from laboratory and field studies, including mine-based tests, conducted by the authors and by other researchers. Acoustic monitoring devices were developed and patented for the authors' experiments. Particular attention is given to single-station acoustic source localization, an approach that has so far received insufficient attention in the literature but offers new prospects for acoustic diagnostics.

Results

Acoustic methods for detecting and locating fire sources in coal seams can be divided into three categories according to the origin of the recorded acoustic signal: passive, active, and combined methods. Passive methods involve recording acoustic emission generated directly in the rock mass by physical processes accompanying combustion, including vibrations of condensed-medium surfaces. Active methods involve recording incident, reflected, and refracted acoustic waves that propagate through the rock mass in response to an external probing pulse, such as an impact or explosion. Combined methods involve recording acoustic emission generated when combustion thermally activates an acoustic emitter, or indicator.



Passive methods

This category includes methods that record acoustic emission generated by processes accompanying combustion, including rock failure, such as fracturing and roof collapse [7], whistling associated by gas release, boiling of water, and similar phenomena.

Acoustic emission during coal combustion under laboratory conditions has been studied by Russian [8] and international researchers [9] over the past two decades. Research output on acoustic emission in mining has grown by more than two orders of magnitude since 2010, with the vast majority of studies conducted by Chinese researchers [10]. More recently, research has begun to focus on infrasonic pulses generated during coal self-heating, that is, at the stage preceding flaming combustion [11–13].

Stationary methods

Acoustic emission accompanying coal combustion is recorded by sensors installed at known locations. The fire source is located using acoustic source-localization techniques based on differences in the arrival times of characteristic pulses.

This approach is justified for powerful low-frequency pulses, for example, during roof collapse above a burned-out area, when waves with frequencies of 2.0–4.5 Hz are generated [14]. For this approach to be effective in detecting fires at early stages, sensors must be installed throughout the mine workings during mine development.

Mobile methods

This approach uses portable equipment. A single kit carried by one person can be used to perform several dozen measurements at different points within several hours, that is, during one work shift. This method, developed by one of the authors of the present study, was successfully tested in an operating mine at an active fire site [15].

Active methods

Transmission-based acoustic testing

Fire sources are assumed to lie between the acoustic source and the acoustic receivers. This configuration corresponds to acoustic logging.

Study [16] demonstrated the feasibility of monitoring the combustion front using surface-based acoustic localization. However, acoustic localization requires special high-power sources of pulsed elastic vibrations with a resonant frequency of approximately 1 kHz. In addition, surface-based localization is possible only in the absence of strong acoustic-impedance contrasts above the coal seam under investigation.

During mining operations at the Moszczenica coal mine in Poland, patchy burned-out zones were identi-

fied in Carboniferous strata. In these zones, the physical properties of the rocks had been altered. These changes allow seismic methods to be used in coalfield studies to monitor zones where coal has disappeared from the seam [17].

Based on theoretical modeling of the wavefield, criteria were developed for identifying coal-free zones. In the model of an undisturbed seam, the wavefield is characterized by intense single and multiple reflections. Removal of coal-seam fragments leads to attenuation of the recorded signal because of multiple wave reflections.

Mine-based experimental studies confirmed the validity of the proposed criteria for identifying coal-free zones detected in deep geological boreholes.

Reflection-based acoustic testing

The acoustic source and receivers are located on the same side of the target objects, that is, the fire sources.

Study [18] used a pointed sledgehammer. Good results were obtained in a number of mine workings in the Donetsk and Ural basins. However, the method failed when extraneous voids, such as the goaf, were present along the acoustic path between the target cavities in the rock mass and the sensor installation points. In such cases, acoustic waves followed indirect propagation paths through the host rocks, which sharply reduced the diagnostic value of the signals. Acoustic-wave propagation in loose coal was examined in [19], which is relevant because loose coal, together with fragments of the host rocks, may be present in the goaf.

Combined methods

The position of the combustion front can be determined by direction finding of seismoacoustic pulses generated in the seam roof above the combustion front, using a system of spatially distributed geophones and arrival-time differences [7].

According to the patent¹, to broaden the applicability of this approach, boreholes are drilled and temperature indicators are placed in them at points with specified coordinates. When the specified rock temperature is exceeded in the monitored area, this event is recorded and a signal is transmitted to the ground surface. The position of the combustion front is then determined from the signal of the triggered indicator and its coordinates.

¹ Rzhnevsky V.V., Yamshchikov V.S., Shkuratnik V.L., Potapov S.L. Method for monitoring the position of the combustion front during underground coal gasification. USSR Patent No. 1173751. Appl. No. 3706101/03, 29 February 1984. Publ. 27 November 1995. (In Russ.)



In addition, to improve the accuracy and reliability of combustion-front monitoring, explosive charges with a specified ignition temperature are used as temperature indicators. They are combined into groups by means of pyrotechnic delay elements; the groups differ in the number of charges, their yield, and the delay interval. The resulting acoustic signals serve as indication signals; their amplitudes and the time intervals between them are measured.

The design of the indicators and the installation procedure were subsequently improved².

Among the methods considered, combined methods provide the most reliable, accurate, and unambiguous identification of combustion at specified coordinates. However, the accuracy of fire-source delineation directly depends on the number of indicators used: the greater their number, the higher the cost. Therefore, this approach is justified for monitoring the spatial position and temporal evolution of the combustion front during underground coal gasification, whereas its use for determining the coordinates of fire sources should be decided on a case-by-case basis. In any event, this approach can be combined with other methods, which, depending on the specific conditions, can improve overall diagnostic performance.

Discussion

This section consists of two parts: a comparison of different approaches to acoustic diagnostics of fire. This section compares the main categories of acoustic methods for detecting and locating fire sources in coal seams and discusses prospects for their further development.

The comparison is limited to acoustic methods. The study does not introduce a single integral measure of method performance because such a measure would require direct comparison of parameters that differ in nature. Cost, implementation time, accuracy, and detection range can be expressed quantitatively. By contrast, applicability, environmental impact, and the ability to determine the fire stage can only be assessed qualitatively. In addition, method performance depends strongly on specific operating conditions. It is therefore more appropriate to compare the relative advantages and limitations of each

method according to a set of predefined criteria. In this study, the performance of acoustic methods for detecting and locating fire sources in coal seams was assessed by determining how well each method met the proposed criteria under different operating conditions.

The simplest assessment scheme uses three categories: high, medium, and low. In a specific situation, these categories may be compared with traffic-light colors: good (green), moderate (yellow), and poor (red). According to Professor L.I. Baron [20], values in mining are generally considered accurate when their coefficient of variation does not exceed 20%. It is therefore sufficient to distinguish five assessment categories: high, above medium, medium, below medium, and low.

To provide a quantitative basis for comparing acoustic methods for detecting and locating fire sources in coal seams, numerical scores were introduced (Table 1).

As shown in Table 1, two scoring scales are used: 0–4 and 1–5. The 0–4 scale is applied to criteria for which no result may be achieved in principle, for example, when a method cannot be applied under certain conditions. The 1–5 scale is used for criteria for which a zero value is impossible, such as cost.

Table 1

Numerical values
used for comparative assessment

Assessment category	Interpretation	Assessment value	
		on the 0–4 scale	on the 1–5 scale
High	The criterion is clearly met; for a quantitative characteristic, this corresponds to the maximum value	4	5
Above medium	The criterion is met to a high degree, but not at the maximum possible level	3	4
Medium	The criterion is met, but with relatively low reliability or unambiguity	2	3
Below medium	The criterion can be met in principle, but with reservations	1	2
Low	The criterion cannot be met, or can be met only with substantial reservations	0	1

² Gladun Yu.V. Development of seismoacoustic methods for monitoring the boundaries of the goaf during underground coal combustion. [Diss. abstract ... Cand. Sci. (Eng.)]. Moscow: Moscow Mining Institute; 1990. 19 p. (In Russ.)

Potapov S.L. Monitoring the movement of the combustion front and the condition of the roof during underground coal combustion using explosive indicators. [Diss. ... Cand. Sci. (Eng.)]. Moscow: Moscow Mining Institute; 1990. 164 p. (In Russ.)



The acoustic methods considered in this study are characterized by parameters of different types, including capabilities and costs. A higher capability score indicates an advantage of a method, whereas a higher cost score indicates a disadvantage. To account for both types of parameters in the comparison, positive scores were assigned to characteristics reflecting advantages, and negative scores

were assigned to characteristics reflecting limitations or costs.

As noted in the Introduction, it is important to define the applicability limits of a particular technical solution, method, or broader diagnostic approach. The results of the comparative analysis of acoustic methods for detecting and locating fire sources in coal seams are therefore presented in tabular form (Table 2).

Table 2

Comparative assessment of acoustic diagnostic approaches for fire sources in coal seams

Criterion	Active	Passive		Combined*
		mobile	stationary	
Applicability	High (4): applicable with almost no restriction	Medium (3): limited by access to measurement points near the fire zone	High (5): applicable with almost no restrictions	Low (1): limited by whether indicators can be installed
Applicability to fire detection in the goaf	Limited (0): no substantial change in continuity occurs	Broad (3): limited by access to measurement points near the fire zone	Medium (2): depends on sensor location	Broad (4): limited by whether indicators can be installed
Reliability	Low (1): discontinuities are not necessarily caused by fire	High (5): acoustic field has characteristic parameters	Medium (3): additional data are required	High (5): indicators are activated at a preset temperature
Detection range	Long (4): almost unrestricted	Short (2): limited by attenuation of acoustic emission from the fire source or combustion front	Long (5): almost unrestricted	Medium (3): limited by indicator size
Timeliness	Medium (3): depends on the time required for sensor installation	High (4): depends on the time required for field and laboratory measurements	Low (1): depends on the time required for collapse above the burned-out area	High (5): information is transmitted almost immediately, at the speed of sound
Delineation accuracy	Medium (3): depends on sensor spacing and wave diffraction around obstacles	High (4): depends on the distance to the fire source	Low (1): affected by wave diffraction around obstacles	High (5): depends on the coordinates of indicator placement
Ability to determine the fire stage	Narrow (0): insensitive to the self-heating stage	Broad (4): established during calibration	Narrow (0): insensitive to the self-heating stage	Medium (2): depends on indicator design
Labor input	Medium (–3): mainly associated with sensor installation	Low (–1): mainly associated with field and laboratory measurements	Low (–2): mainly associated with installation of the recording system	High (–5): mainly associated with manufacture and installation of indicators
Cost	Medium (–3): capital costs: equipment kit; operating costs: sensor installation and data interpretation	Low (–1): capital costs: equipment kit; operating costs: field and laboratory measurements	Medium (–3): capital costs: equipment kit and sensor installation; operating costs: data interpretation	High (–5): capital costs: equipment kit and fabrication and installation of indicators; operating costs: data interpretation
Environmental compatibility	Medium (–2): installation of a large number of sensors and additional disturbance of the rock mass	High (0): virtually no environmental impact	High (0): virtually no environmental impact	Low (–4): installation of a large number of indicators and additional disturbance of the rock mass

* Starting from the second row, it is assumed that the indicators are installed in the combustion zone.

Table 2 shows that at least one acoustic diagnostic approach performs well for each criterion considered. The main limitation is the accuracy of fire-source delineation when the distance between the combustion front and the sensors is large. The number of measurements is limited primarily by cost considerations, whereas the exact coordinates of sensor installation are not subject to strict constraints. In practice, situations in which sensors cannot be placed within several hundred meters of a fire source are relatively uncommon. Measurements can therefore usually be performed at distances not exceeding 200–300 m from the fire source. As data are accumulated from a series of measurements taken at different locations, the search area is progressively narrowed. To adapt the assessment to specific operating conditions, weighting coefficients can be introduced to reflect the relative importance of each criterion. The preferred diagnostic approach is then selected by summing the products of these coefficients and the scores assigned to the corresponding cells in Table 2. The highest total score indicates the most suitable option.

The comparison presented in Table 2 indicates that the methods considered should be combined so that the limitations of one method are offset by the strengths of another. Acoustic diagnostics of fire sources in coal seams can detect combustion, determine fire-source coordinates, track the spatial extent and temporal evolution of fire boundaries, and identify the fire stage. The choice of a diagnostic method, or a combination of methods, therefore depends on the specific operating conditions. Numerical scores are assigned to the criteria relevant to the problem under consideration, additional weighting coefficients are introduced where necessary, and the most suitable option is selected on the basis of the resulting total score.

Future directions

The use of more advanced technical solutions expands diagnostic capabilities and improves the performance of acoustic methods for detecting and locating fire sources in coal seams. One less obvious but promising direction, particularly in passive acoustic diagnostics, is single-station acoustic source localization. This approach involves determining the bearing of an acoustic signal source in a rock mass using sensors installed at a single measurement station rather than sensors distributed spatially across the seam. More than 40 years ago, researchers outside Russia examined this problem [21] and concluded that the direction of arrival of a continuous acoustic signal could not be determined from single-station measurements.

In the early twenty-first century, however, Russian scientist Professor Nikolai F. Kusov proposed a different hypothesis. He suggested that the direction of arrival could be determined because a compressional wave propagates outward from the source in all directions and, in the case of a point source, radially. According to this hypothesis, particles on the surface of a solid medium are displaced not normal to the undisturbed surface but along the radius from the wave source. This differs from the behavior of a liquid, where particles at any point on the free surface are displaced vertically. This principle is shown schematically in Fig. 1. The gray half-plane represents the solid medium, that is, the rock mass; the black dot indicates the acoustic emission source; the concentric circles show the acoustic wavefronts at equal time intervals; the red arrows indicate the radius vectors of the points under consideration on the solid-medium surface relative to the wave source; the black arrows show particle displacement vectors on the solid-medium surface; the white circles with a solid outline indicate the initial particle positions in the area under consideration; the white circles with a dashed outline indicate the particle positions after displacement caused by the first arrival of the acoustic wave; and the numbers indicate the arbitrary numbering of particles.

Based on this hypothesis, a method was developed to provide an empirical technical implementation of single-station acoustic source localization [22]. The method is based on placing several sensing elements with different spatial orientations at the recording station. For continuous processes, such as fire, at least two sensing elements are required: one with a fixed spatial orientation and another whose orientation is varied during recording (Fig. 2).

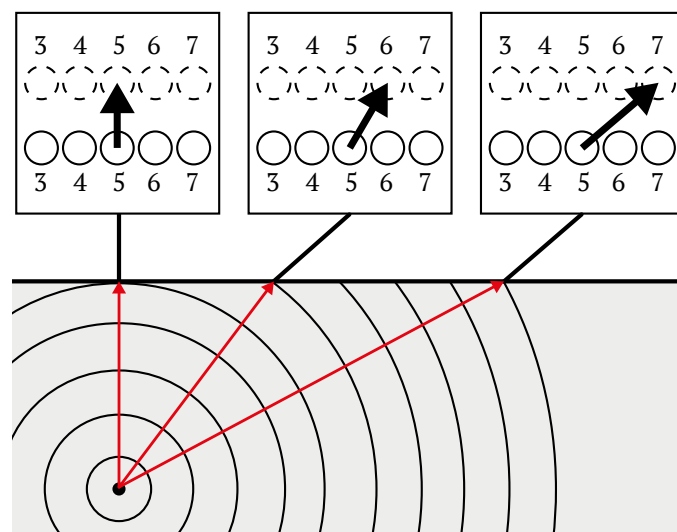


Fig. 1. N. F. Kusov's hypothesis (explained in the text)

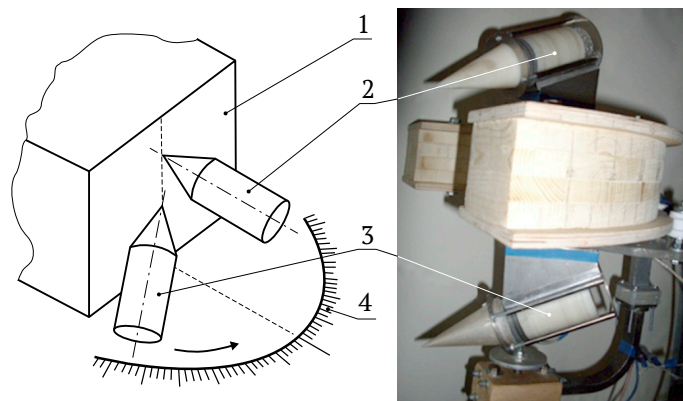


Fig. 2. Schematic diagram of the method:

- 1 – working surface of the medium;
- 2 – fixed sensing element; 3 – movable sensing element;
- 4 – protractor for measuring the angular position of the sensing elements

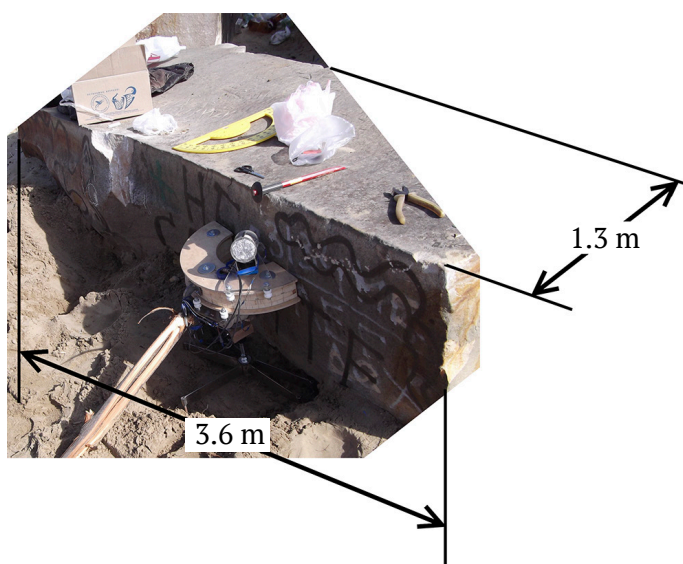


Fig. 3. General view of the sandstone block used in the test and the contact cone of the experimental setup; the photograph is rotated, which accounts for the cropped corners [22]

The bearing of the acoustic-wave source is determined by comparing the amplitudes of the first arrivals of acoustic pulses recorded by the movable cone with pulses previously recorded by the same cone and with pulses recorded simultaneously by the fixed cone. If the acoustic-pulse parameters remain constant over time, one cone is sufficient. If they vary, as is generally the case, two cones are required. For a planar case, such as a coal seam, two cones are sufficient. To determine the bearing of acoustic-wave sources in three-dimensional space, either a third cone should be used or the position of the movable cone should be varied sequentially in two planes.

The method was tested on a model object, namely a sandstone block in which mechanical impacts served as the acoustic source (Fig. 3), and under industrial conditions at an open-pit coal mine with an artificial combustion source.

The test procedure was as follows. The sensing elements of the measurement system were placed on the prepared surface of the test object, namely the vertically oriented surface of the rock sample, with contact provided by the cone tips. Controlled hammer impacts were then applied to the opposite side of the test object at the same point. The impacts were applied in series, with ten impacts in each series. The first cone remained in the same position throughout the test, whereas the second cone was rotated by 5° in the specified direction after each series of impacts. For the proposed method of determining the bearing of an acoustic emission source, impact intensity, whether from one impact to another or from one series to another, is not critical. The key parameter is not the absolute amplitude but the relative amplitude of the signals generated in the two sensing elements when each acoustic pulse reaches them.

The test results confirmed N. F. Kusov's hypothesis (Fig. 4).

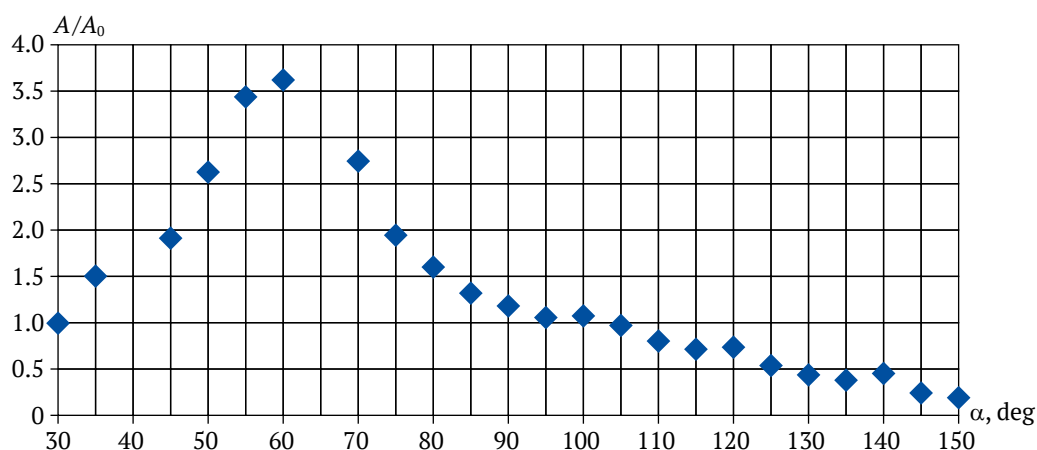


Fig. 4. Relative amplitudes of the first arrivals of acoustic pulses as a function of the angular position of the movable sensing element: acoustic-pulse source bearing, 60° ; angular position of the fixed sensing element, 112° [22]



The scatter between series ranged from 4 to 25%. Fig. 4 shows the values averaged for each series.

After the bearing of an acoustic source has been determined from several points, its coordinates can be calculated. Thus, highly directional sensing elements installed in an undisturbed coal pillar or coal seam can be used to determine the bearing of fire sources that generate characteristic acoustic emission from a single point in the bearing plane, that is, the seam plane.

The concept of single-station acoustic source localization is not limited to bearing determination. If longitudinal and transverse waves are recorded in a medium with a known sound velocity, this approach can also be used to determine the distance to the acoustic source. Sound velocity depends on several factors, including moisture content, pressure, and fracturing, which can be accounted for by introducing appropriate corrections. A mathematical description of single-station acoustic source localization remains a subject for further research.

Economic implication

The economic effect of specific acoustic methods for detecting and locating fire sources in coal seams, or combinations of such methods, can be calculated accurately only for diagnostic operations that have already been implemented, when the scope of the measures taken is known and the associated costs can therefore be determined. In general terms, it is more appropriate to discuss the sources of this effect.

From a safety perspective, accurate detection and localization of fire sources and their timely suppression can reduce the cost of fire-control measures, such as nitrogen injection into mine workings. At the same time, the value of human life cannot be reduced direct economic losses and the cost of replacing workforce.

When the value of mineral resources affected by fire is considered, quantitative estimates are fully justified. With regard to coal preservation, some reports indicate that in the People's Republic of China, coal losses caused by spontaneous-combustion fires alone amount to hundreds of billions of dollars annually. However, many fire sources may exist simultaneously, be at different stages, and develop at different rates. Most importantly, diagnostic conditions may differ substantially. Reliable numerical estimates therefore require the analysis of specific cases.

Beyond coal losses, fire sources in coal seams have another aspect that is highly promising for further research. Coal contains various chemical elements, including radioactive elements [23]. Their concentrations may differ by a factor of 1,000 or more between deposits [24]. In some cases [25–27], uranium and thorium contents may vary substantially within the same coal seam. It has been established [28] that

the tendency of coal to self-heat increases as the degree of coal metamorphism decreases. The lower the degree of coal metamorphism, the higher the content of radioactive elements in coal [29]. In other words, the tendency of coal to self-heat increases with increasing uranium content. During coal combustion, the concentration of radioactive elements in ash increases. At thermal power plants, this increase is typically three- to fourfold and may reach fifteenfold [30], whereas in fire sources in coal seams it may increase by orders of magnitude [31].

Conclusion

This analytical review examined acoustic methods for detecting and locating fire sources in coal seams and proposed their classification. The methods were compared in tabular form. The results show that combinations of different acoustic methods can be used to identify and locate fire sources in coal seams under virtually any real operating conditions. The prospects for passive acoustic diagnostics based on the concept of single-point acoustic source localization are also highlighted.

The main conclusions are as follows:

1. Acoustic methods for detecting and locating fire sources in coal seams were classified according to the origin of the recorded acoustic emission. On this basis, passive, active, and combined methods were identified.
2. Criteria were identified for evaluating acoustic methods for detecting and locating fire sources in coal seams under specific operating conditions. These criteria include applicability, applicability to detecting fires in the goaf, reliability, distance, timeliness, delineation accuracy, ability to determine the fire stage, labor input, cost, and environmental impact.
3. Acoustic diagnostics of fire sources in coal seams can detect combustion, determine fire-source coordinates, track the spatial extent and temporal evolution of fire boundaries, and identify the fire stage. The diagnostic method, or combination of methods, is selected according to the specific operating conditions. For this purpose, numerical scores are assigned to the criteria relevant to the problem under consideration, and additional weighting coefficients are introduced where necessary. The most suitable option is then selected on the basis of the resulting total score.
4. Highly directional sensing elements installed in an undisturbed coal pillar or coal seam can be used to determine the bearing of fire sources that generate characteristic acoustic emission from a single point in the bearing plane, that is, the seam plane.
5. Acoustic diagnostics of fire sources in coal seams can be used to determine the coordinates of zones with elevated radionuclide concentrations.



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Measurement of the attenuation coefficient of acoustic signals generated by an operating roadheader in a coal mine

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Abstract

This study is motivated by the need to improve the reliability of predictions of gasogeodynamic phenomena (GGP) in coal mines by refining acoustic prediction methods. This requires methods for determining the key parameters governing acoustic-wave propagation through the coal-rock mass, including the acoustic attenuation coefficient. The objective was to justify an algorithm for measuring the acoustic attenuation coefficient under mine conditions over the frequency range used for GGP prediction. To this end, the noise spectrum generated by an operating roadheader was recorded at a Kuzbass mine using the multichannel Mikon-GEO system and geophones installed 11, 21, 31, 47.7, 57.7, and 67.7 m from the roadway face. The results showed that, owing to interference among longitudinal and shear waves and waves reflected from the coal seam-host rock interfaces, the attenuation coefficient calculated from a line spectrum by summing harmonic amplitudes within 20-Hz frequency windows did not increase linearly with frequency, as expected for an unbounded homogeneous solid, but instead varied nonmonotonically. This introduces errors into rock-mass stress estimates obtained using methods based on spectral analysis of noise generated by operating mining equipment. To eliminate this source of error, it was proposed that the attenuation coefficient be determined after summing the harmonic amplitudes within frequency windows at least 200 Hz wide. It was also proposed that the averaging time for the amplitudes of the spectral components correspond to the roadheader operating interval between successive stoppages for roadway support installation. Under the experimental conditions, this approach produced a linear increase in the acoustic attenuation coefficient over the extrapolated frequency range of 0–700 Hz, with a coefficient of determination of 0.997.

Keywords

gasogeodynamic phenomena, operational prediction, hazard criterion, acoustic signals, attenuation coefficient determination algorithm, attenuation coefficient, wave interference, spectral analysis, time window, frequency window

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СВОЙСТВА ГОРНЫХ ПОРОД. ГЕОМЕХАНИКА И ГЕОФИЗИКА

Научная статья

Измерение коэффициента затухания акустического излучения от работающего проходческого комбайна в угольной шахтеА. В. Шадрин   *Федеральный исследовательский центр угля и углехимии СО РАН (ФИЦ УУХ СО РАН),
г. Кемерово, Российская Федерация* *avsh-357@mail.ru***Аннотация**

Актуальность данного исследования обусловлена необходимостью повышения достоверности прогноза геогазодинамических явлений в угольных шахтах акустическими методами путем их совершенствования. Для этого необходимы методы определения основных параметров, определяющих распространение акустических волн по углепородному массиву. К их числу относится коэффициент затухания звука. Цель: обосновать алгоритм измерения в шахтных условиях коэффициента затухания акустического излучения в диапазоне частот, используемом для прогноза ГДЯ. Для достижения указанной цели на одной из шахт Кузбасса с помощью многоканальной аппаратуры Микон-ГЕО регистрировался спектр «шума» работающего проходческого комбайна сейсмоприемниками, установленными на расстоянии соответственно 11; 21; 31; 47,7; 57,7 и 67,7 м от забоя выработки. Установлено, что благодаря интерференции продольной и поперечной волн и их отражений от границ раздела угольного пласта и вмещающих пород зависимость коэффициента затухания звука линейчатого спектра с окном суммирования гармоник в 20 Гц не возрастает линейно с частотой, как это имеет место в однородном твердом теле безграничных размеров, а приобретает немонотонный характер. Это приводит к погрешности оценки напряженного состояния массива методами, основанными на спектральном анализе шума работающего горного оборудования. Для устранения указанного недостатка предложено определять коэффициент затухания в частотных окнах, соответствующих одной гармонике, равных не менее чем 200 Гц. Причем время усреднения амплитуд спектральных составляющих предложено брать равным интервалу между циклами остановки комбайна для крепления бортов выработки. Для условий эксперимента это позволило получить линейный характер возрастания коэффициента затухания звука в экстраполированном диапазоне частот 0–700 Гц с коэффициентом детерминации 0,997.

Ключевые слова

геогазодинамические явления, текущий прогноз, критерий опасности, акустическое излучение, алгоритм определения, коэффициент затухания, интерференция волн, спектральный анализ, временное окно, частотное окно

Финансирование

Работа выполнена в рамках государственного задания ФГБНУ «Федеральный исследовательский центр угля и углехимии Сибирского отделения Российской академии наук» проект FWEZ-2026-0002 «Разработка методов комбинированного применения геофизических систем прогноза, контроля и управления состоянием горного массива, а также технологий предотвращения газодинамических явлений и мониторинга проведенных мероприятий по их реализации» (рег. № 1023032200092-2-1.5.1;2.7.5).

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Introduction

Acoustic methods are widely used in mining to investigate rock-mass structure and properties, monitor various technological processes, and assess safety under the influence of various factors [1–3]. Among these methods, those used for operational prediction of gasogeodynamic phenomena (GGP)

form a distinct group. This group, in turn, comprises two main subgroups. The first is based on recording acoustic emission from the near-face region of workings being driven [4–6]. The second relies on spectral analysis of acoustic signals generated either by operating mining equipment or by an impact source [7–9]. The impact source may be a sledge-



hammer or hammer¹, or a purpose-built drop-weight device in which a metal weight of known mass is dropped from a fixed height [10].

The main parameters used in acoustic investigations for regional, local, and operational GGP prediction are oscillation frequency and energy, wave propagation velocity, and attenuation in the monitored medium. These parameters must therefore be known for computer modeling of acoustic-wave propagation aimed at improving GGP prediction methods. In addition to the characteristics of the acoustic source, however, these parameters depend strongly on the structure, composition, physical and mechanical properties, and stress state of the specific part of the rock mass under investigation. These factors govern the generation, propagation, and attenuation of acoustic waves. Methods capable of rapidly determining these parameters experimentally are therefore required.

The operating frequency range is a key characteristic of acoustic signals. On the one hand, it determines the resolution of the method when monitoring rock-mass structure: the higher the frequency, the smaller the discontinuities that can be identified. On the other hand, the frequency range determines the acoustic attenuation coefficient and, consequently, the effective range of the monitoring method. The higher the frequency, the greater the acoustic attenuation coefficient and the shorter the effective monitoring range. For example, earthquakes are recorded at large source–receiver distances at very low frequencies of 0.001–16 Hz, whereas rockbursts within a mine field are recorded at frequencies ranging from a few hertz to several hundred hertz [11, 12]. Operational GGP prediction in coal mine workings using the method based on the parameters of an artificial acoustic signal employs a frequency range from a few hertz to 5 kHz when a piezoelectric transducer is used for signal acquisition [7–9]. In practice, however, strong signal attenuation at high frequencies and the operating characteristics of the acoustic receivers limit the usable range to 1.5–2.0 kHz. Thus, the acoustic attenuation coefficient depends on the type of acoustic receiver used, the rock-mass structure, its strength properties, and the stress state of the specific part of the coal–rock mass under investigation.

The objective of this study was to justify an algorithm for measuring the acoustic attenuation coefficient under mine conditions over the frequency range used for GGP prediction.

The following tasks were addressed:

1. Identify the factors and parameters governing the attenuation coefficient of acoustic signals generated by an operating roadheader in a coal-rock mass.
2. Synchronously record, using several sensors installed at different distances from the face of a development heading, the time history of acoustic signals generated by the roadheader cutting head.
3. Determine the spectrum of the recorded signals from the data obtained.
4. Analyze the results and propose an algorithm for determining the effective attenuation coefficient of the noise generated by an operating roadheader.

Parameters and factors governing acoustic-wave attenuation in a rock mass

The acoustic sources monitored using the above GGP prediction methods – cracks undergoing discontinuous growth and the noise generated by mining equipment cutting coal or rock – emit longitudinal and shear waves over a broad frequency range. These wave types differ in propagation velocity and attenuation coefficient, both of which depend strongly on the physical and mechanical properties, thermodynamic state, and structural characteristics of the rock mass [11–13]. Various sources report that the longitudinal-wave velocity is 1.5–2.5 times [14] or 1.7–1.9 times [15] the shear-wave velocity. The attenuation coefficient in solids and rock masses is approximately directly proportional to frequency [15–17] and inversely proportional to stress in the rock mass [18]. The attenuation coefficient of shear waves is generally lower than that of longitudinal waves [19]. For example, experiments have shown that “the shear-wave amplitude along the axis of the main lobe of its directivity pattern is generally 1.5–2 times that of the longitudinal wave in the same direction” [14].

The acoustic signal from a monitored source, as recorded by seismic receivers, is affected by numerous factors, including the layered structure of the rock mass, the presence of discontinuities and various heterogeneities, the stress state, the operational parameters of mining activities, and the simultaneous operation of other acoustic sources [20–22].

The layered structure of the coal-rock mass causes the coal seam to act as a waveguide. If the acoustic source is also located within the coal seam, most of the energy emitted by the source is concentrated in the seam rather than in the host rocks. Interference among longitudinal and shear waves and their reflections from interfaces between layers of contrasting acoustic impedance, together with possible mode conversion, gives rise to a channel wave in the coal seam [1].

¹ Guidelines. Method for acoustic sounding of a rock mass using a hardware and software system. Moscow: Intersectoral Research and Engineering Laboratory for the Development, Manufacture, and Implementation of Automated Systems in the Mining Industry (MNTL RIVAS); 2016. 36 p. (In Russ.)



The waveguide behavior of the coal seam also filters the full spectrum of the emitted signal, preferentially transmitting the harmonics with the lowest attenuation. Seam thickness determines the predominant frequency range recorded because it is an integer multiple of the half-wavelengths of the least attenuated harmonics. Accordingly, as seam thickness increases, the spectrum of harmonics filtered by the rock mass shifts toward lower frequencies. Conversely, as seam thickness decreases, the filtered harmonics shift toward higher frequencies.

Real acoustic sources, however, have relatively broad directivity patterns. Part of the acoustic energy is therefore refracted at the interfaces between the coal seam and the host rocks. These waves are subsequently reflected from interfaces between layers of contrasting acoustic impedance or from various heterogeneities and return to the coal seam. Because their propagation paths are longer than those of the direct longitudinal and shear waves, they arrive later. This time delay is evident when the signal source is impulsive. In particular, these additional oscillations in the tail of seismic signals generated by powerful impulsive sources such as earthquakes are known as coda waves [23].

With a continuous source and at short distances from the source, the longitudinal, shear, and coda waves cannot be resolved separately. Their harmonic components nevertheless interfere to form the resultant wavefield.

Because the resultant wavefield is produced by interference among these wave types, signal attenuation is evaluated in terms of an effective attenuation coefficient [1].

Acoustic-wave attenuation during propagation through a rock mass is caused primarily by energy absorption due to internal friction, energy scattering by heterogeneities in the medium, and geometrical spreading governed by the source radiation pattern [19]. Taking these factors into account, the signal amplitude at a given distance from the source can be expressed either as a function of the propagation time t to the receiver or as a function of the source-receiver distance r . In the first case, the amplitude of the resultant seismic wave $A(f, t)$ at frequency f and time t can be written as follows [11]:

$$A(f, t) = S(f, t) \cdot t^{-\gamma} \cdot \exp\left[\frac{-\pi f t}{Q(f)}\right], \quad (1)$$

where $S(f, t)$ is the source time function at frequency f and time t ; γ is a dimensionless parameter characterizing geometrical spreading and is equal to 1.0, 0.5, or 0.75 for body, surface, and diffuse waves, respectively [11]; and $Q(f)$ is the effective seismic quality factor.

The function $t^{-\gamma}$ is determined by the source radiation pattern and describes the geometrical spreading of radiated energy with time. At a fixed harmonic, the quality factor characterizes the relative loss of wave energy due to internal friction over one oscillation period. The quality factor increases as the relative energy loss decreases. The effective seismic quality factor characterizes this process for the entire wave packet.

In the second case, the amplitude of the resultant wave $A(f, r)$ at distance r from the source can be written as follows [19]:

$$A(f, r) = A_0(f) \cdot R(r) \cdot \exp[-\alpha_{\text{eff}}(f) \cdot r], \quad (2)$$

where $A_0(f)$ is the resultant-wave amplitude at the source; $R(r)$ is the geometrical-spreading function; $\alpha_{\text{eff}}(f)$ is the effective attenuation coefficient of the resultant wave produced by interference.

The exponential terms in square brackets in Eqs. (1) and (2) describe the same attenuation in the two representations of the resultant-wave amplitude and can therefore be equated. Substituting $t = r/V$, into the exponent in Eq. (1), where V is the effective velocity of the resultant wave, yields the following relationship between the effective attenuation coefficient and the effective seismic quality factor:

$$\alpha_{\text{eff}}(f) = \frac{\pi f}{QV}. \quad (3)$$

Modeling acoustic-wave attenuation with distance requires knowledge of the function $R(r)$. For an acoustic source with a small radiating area located on the surface of a mine working, the radiation pattern at low frequencies can be approximated as spherical; that is, sound propagates in nearly all directions. In this case, $R(r) = 1/r$ [19]. At high frequencies, the spreading function more closely approximates that of a plane wave [14], for which $R(r) \approx 1$ [19]. Because the resultant wave generated by a crack undergoing discontinuous growth or by operating mining equipment contains a broad frequency spectrum, the spreading function may be represented as:

$$R(r) = r^{-n}, \quad (4)$$

where $0 < n < 1$.

Based on experimental findings, cylindrical geometrical spreading, for which $R(r) = 1/\sqrt{r}$, was assumed in Refs. [1, 24]. Those studies, however, primarily examined ultrasonic waves. It may therefore be assumed that, in the frequency range of 0–1.5 kHz, a roadheader cutting head or drill rod generates a wave whose radiation pattern, depending on the harmonic frequency, is intermediate between spherical and cylindrical.

Eqs. (1) and (2) indicate that the resultant-wave amplitude decreases monotonically with increasing

distance from the source. Experimental data reported in several studies, however, show departures from this relationship. In ultrasonic sounding of a rock mass, such departures are particularly pronounced at short source-receiver distances [14]. A nonmonotonic variation in wave amplitude with distance should produce a corresponding nonmonotonic variation in the effective attenuation coefficient. This behavior is demonstrated below using experimental results obtained over the frequency range employed for operational GGP prediction using the method based on artificial acoustic signal parameters.

Experimental determination of the attenuation coefficient

During operational GGP prediction at a Kuzbass mine using the method based on artificial acoustic signal parameters, a nonmonotonic variation in the amplitude of the noise generated by an operating roadheader was experimentally recorded over source-receiver distances ranging from 11 to 67.7 m. Roadheader noise was recorded using the Mikon-GEO hardware and software system and six seismic receivers (SRs) installed 11, 21, 31, 47.7, 57.7, and 67.7 m from the roadway face. The experimental conditions are described in Ref. [25].

The signal amplitude recorded by SR 6, the most distant receiver, was found to be comparable to the background acoustic noise level measured when the roadheader was not operating. Signals from this receiver were therefore excluded from further analysis. Fig. 1 shows the initial signal spectra from the remaining five SRs before the subsequent averaging of harmonic amplitudes over the frequency and time windows considered below. The spectra were obtained using a fast Fourier transform with a frequency resolution of 1 Hz, and the amplitudes of the corresponding harmonics were averaged over 1-min time windows.

The data also show that, under the experimental conditions, the signal amplitude at frequencies above 1200 Hz was comparable to the noise level recorded when the roadheader was not operating. Subsequent analysis was therefore restricted to the frequency range of 1–1200 Hz.

Fig. 1 shows, for example, that in the low-frequency range of 0–100 Hz, the harmonic amplitudes recorded by SR 2, which was farther from the source, were greater than those recorded by the closer SR 1. The amplitudes recorded by SR 5 also exceeded those recorded by SRs 3 and 4. As the frequency increased and the harmonic amplitudes recorded by all SRs decreased, the departure from a monotonic decrease in amplitude with increasing r became less pronounced.

When the distance dependence of the resultant-wave amplitude $A(f, r)$ is expressed as in Eq. (2), the effective attenuation coefficient is calculated as follows [1]:

$$\alpha_{i1} = \frac{\ln\left(\frac{A_i}{A_1}\right)}{r_{i1}}, \quad i \in [2; 5], \quad (5)$$

where α_{i1} is the effective attenuation coefficient over the section between SR 1 and SR i , m^{-1} ; A_1 and A_i are the harmonic amplitudes recorded by SRs 1 and i respectively, a.u.; r_{i1} is the distance between SR i and SR 1, m.

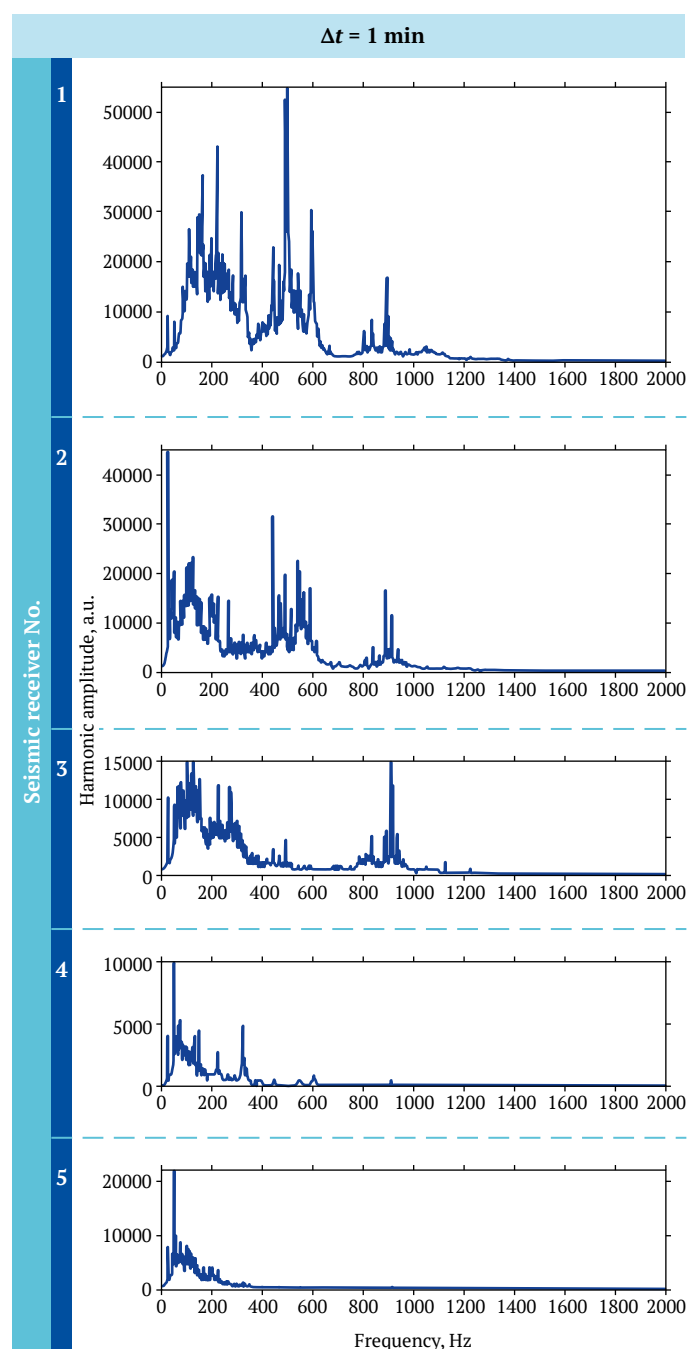


Fig. 1. Initial signal spectra recorded by SRs 1–5



If the attenuation coefficient is determined from the spectrum shown in Fig. 1, it is negative at frequencies for which the amplitude recorded by a more distant SR exceeds that recorded by a closer SR. This contradicts the physical meaning of signal attenuation. To eliminate this paradoxical result, the harmonic amplitudes were averaged over frequency windows wider than 1 Hz.

Another source of variability in attenuation-coefficient estimates is the dependence of the signal

spectrum on the position of the cutting head on the roadway face. Reference [25] showed that the spectra recorded during the first, third, and sixth minutes of roadheader operation differed substantially. To eliminate this effect, the spectral amplitudes were averaged over longer time windows. However, even averaging over a 9-min window did not eliminate the effect. The signal amplitude during the second 9-min interval was approximately twice that during the first interval (Fig. 2).

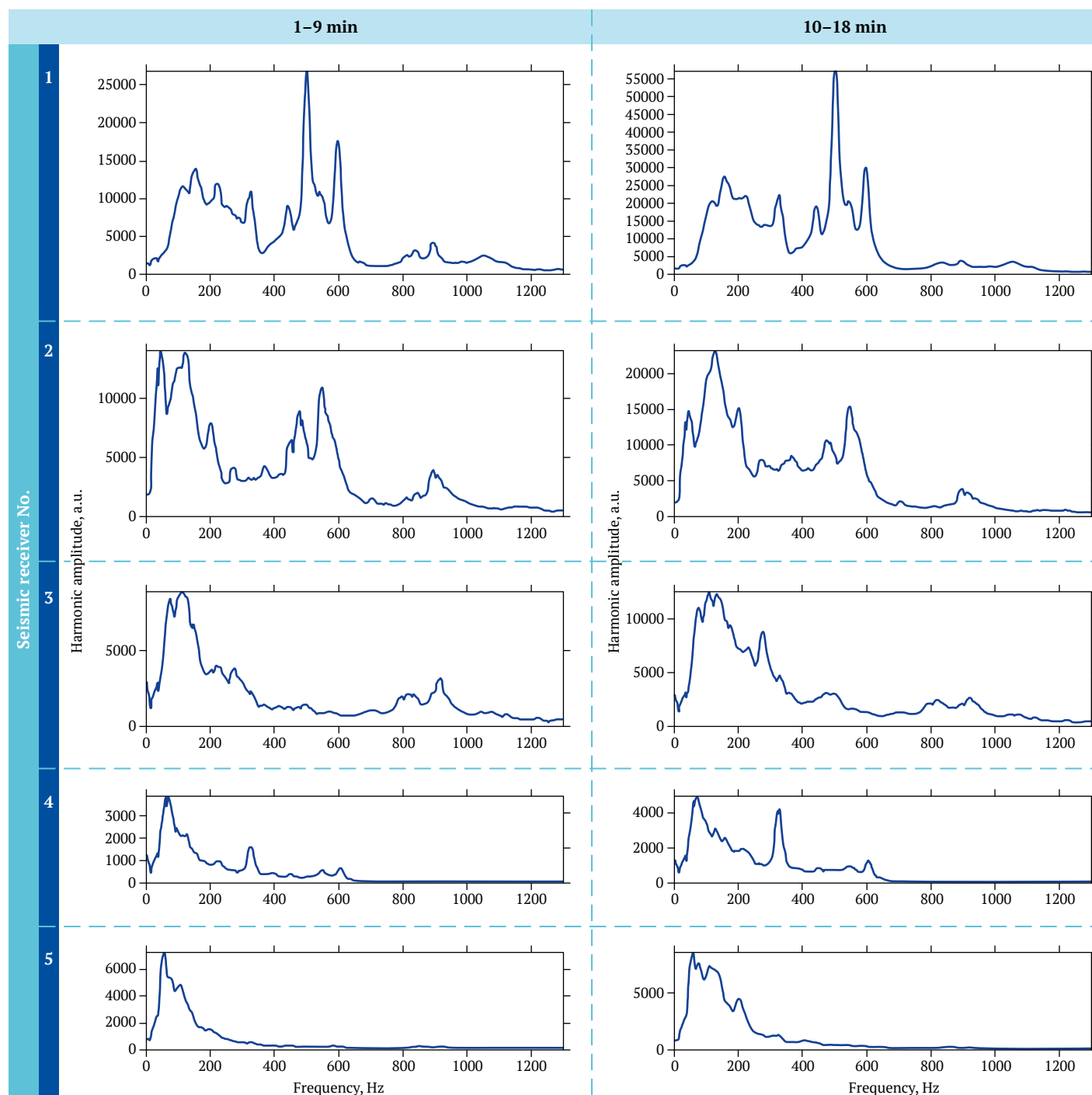


Fig. 2. Signal line spectra with harmonic amplitudes averaged within 20-Hz frequency windows and over 9-min time windows

The substantial difference between the signal spectra recorded during the first and second 9-min intervals was mainly attributable to changes in the distance between the acoustic source and receiver as the cutting head moved across the roadway face, variations in rock-mass fracturing as the cutting head penetrated deeper into the rock mass, possible variations in the force exerted by the cutting head on the rock mass, and the transition from cutting coal to cutting the host rock. These factors introduce variability into operational GGP prediction results. Their influence can be minimized by increasing the time over which the measured spectra are averaged. For a valid comparison of GGP prediction results obtained using geophysical and instrument-based methods, the averaging time should approximately correspond to the time required for the face to advance by the characteristic distance used in the instrument-based prediction method. This distance is 1 m – the length of the test-borehole interval used to measure the initial gas emission rate and/or drill-cuttings yield as the borehole is drilled in successive intervals. In practice, when prediction is performed using the method based on artificial acoustic signal parameters, a convenient averaging period is the duration of roadheader operation between successive roadway-support cycles, because the advance per cycle is close to 1 m. A roadheader generally requires approximately 30 min to advance the roadway by this distance.

To illustrate how the spectra change as the averaging time increases, Fig. 3 shows the spectrum of the same signal as in Fig. 2, but averaged over an 18-min interval.

Fig. 3 shows that doubling the duration of the time-averaging window did not substantially change the distribution of harmonic amplitudes across frequency, but somewhat reduced the variability in spectral amplitudes caused by movement of the cutting head during roadheader operation between stoppages for roadway support installation. Nevertheless, the pattern persisted in the low-frequency portion of the spectrum: at some frequencies, the harmonic amplitudes recorded by more distant SRs exceeded those recorded by SRs closer to the source. Increasing the width of the frequency-averaging window from 1 Hz (Fig. 1) to 20 Hz (Figs. 2 and 3) made this pattern less pronounced but did not eliminate it. The frequency-averaging windows were therefore widened further. Fig. 4 shows the effect of this increase. The experimental points are plotted at the midpoint frequencies of the corresponding windows.

Fig. 4 shows that, with a 100-Hz averaging window, the sum of the harmonic amplitudes recorded by

SR 2 in the 0–100 and 200–300 Hz windows exceeded the corresponding sums recorded by SR 1 (Fig. 4, *a*). A similar pattern was observed for SR 3 relative to SR 1 in the 200–300 Hz window. When the averaging-window width was increased to 200 Hz (Fig. 4, *b*), this pattern almost disappeared. It was completely eliminated when the frequency-averaging window was increased to 300 Hz (not shown).

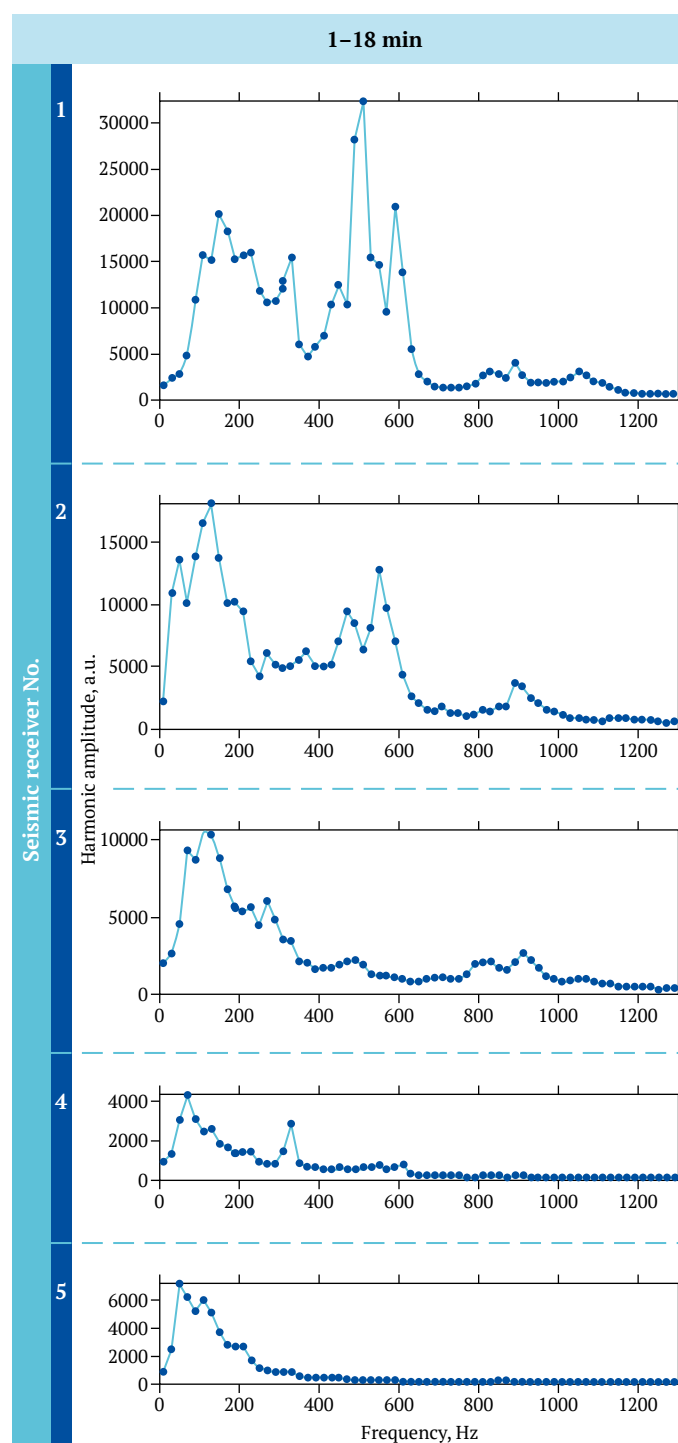


Fig. 3. Signal line spectra with harmonic amplitudes averaged within 20-Hz frequency windows and over an 18-min time window

By widening the frequency windows over which the harmonic amplitudes were averaged, the above nonmonotonic behavior of the roadheader-noise spectrum with distance from the source was eliminated. This made it possible to determine the effective signal attenuation coefficient for each section between SR 1 and SR i using Eq. (5). The results are shown in Fig. 5. As shown in the figure, the effective attenuation coefficient increases linearly with frequency over the range of 0–500 Hz.

Fig. 6 shows the 100–500 Hz segment of the $\alpha_{avg}(f)$ curve presented in Fig. 5, together with the corresponding linear trend line extrapolated to 0–700 Hz ($R^2 = 0.997$).

We previously proposed expressing the dependence of the attenuation coefficient on frequency and stress state as follows [8]:

$$\alpha_i = \alpha_0 + \beta \frac{f_i}{f_0} \frac{\sigma_{lim}}{\sigma_{cur}}, \quad i \in (M_{min}; M), \quad (6)$$

where α_i is the attenuation coefficient of the i th harmonic, m^{-1} ; α_0 is the attenuation coefficient of the lowest-frequency harmonic within the geophone operating range in the absence of stress, m^{-1} ; f_i and f_0 are the frequency of the i th harmonic and the lowest frequency within the operating range of the equipment, respectively, Hz; β is a parameter referred to here as the dynamic attenuation coefficient and determined by the properties of the rock mass, m^{-1} ; σ_{lim} and σ_{cur} are the mean limiting and current stresses in the rock mass, respectively, Pa; M_{min} and M are the harmonic numbers corresponding to the lowest and highest frequencies within the geophone operating range, respectively.

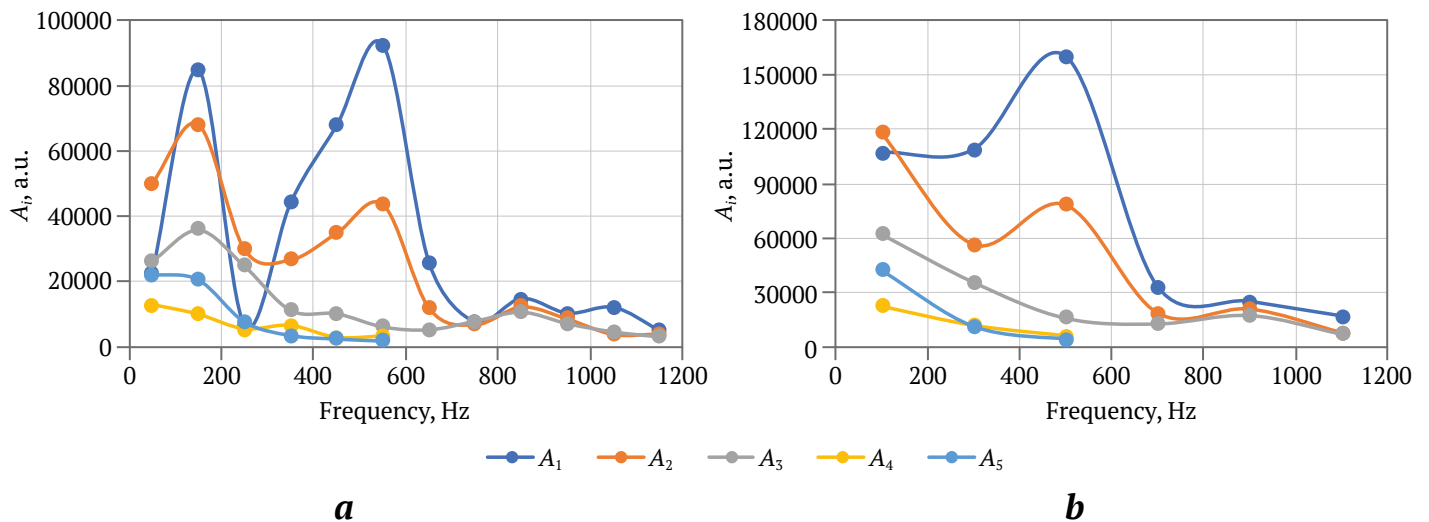


Fig. 4. Harmonic amplitudes recorded by SRs 1–5 after averaging over 18 min and within frequency windows of (a) 100 Hz and (b) 200 Hz

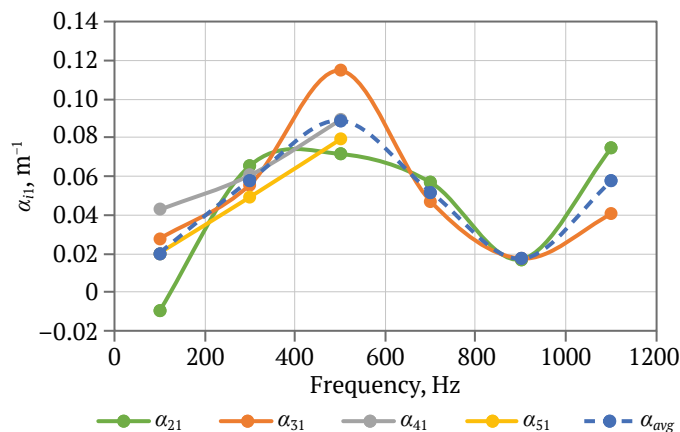


Fig. 5. Frequency dependence of the effective attenuation coefficient of roadheader noise, α_{i1} , for the sections between SR 1 and SR i , and the corresponding mean value, α_{avg}

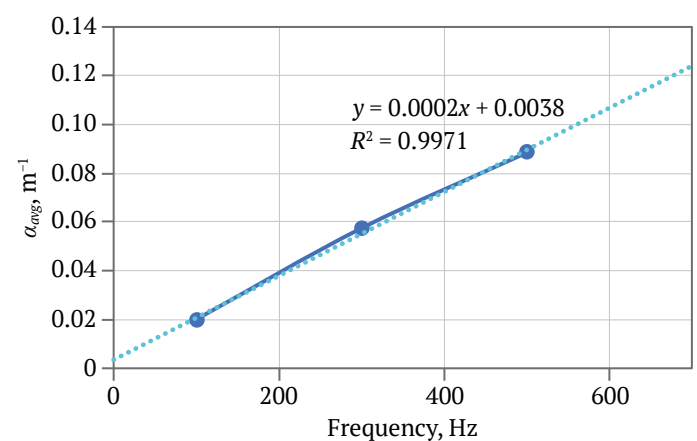


Fig. 6. Mean attenuation coefficient α_{avg} versus frequency f over 100–500 Hz (solid line) and the corresponding trend line extrapolated to 0–700 Hz (dashed line)



The extrapolation equation obtained in Micro-soft Excel and shown in Fig. 6 indicates that, in this case, $\alpha_0 = 0,0038 \text{ м}^{-1}$.

The dynamic attenuation coefficient β can be determined using the measured effective attenuation coefficients at the beginning and end of the linear segment of $\alpha_{\text{avg}}(f)$.

Rearranging Eq. (6) gives:

$$\beta = \frac{(\alpha_i - \alpha_0)f_0\sigma_{\text{cur}}}{f_i\sigma_{\text{lim}}}. \quad (7)$$

According to the data in Fig. 6, $\alpha_0 \approx 0.0038 \text{ м}^{-1}$ at a frequency close to $f_0 = 1 \text{ Hz}$, whereas $\alpha_{600} = 0.1238 \text{ м}^{-1}$ at a frequency close to $f = 600 \text{ Hz}$, corresponding to the 500–700 Hz window. Substituting these values into Eq. (7) yields, for the experimental conditions,

$$\beta = 0.0002 \frac{\sigma_{\text{cur}}}{\sigma_{\text{lim}}}. \quad (8)$$

The experiment was conducted in a section classified as nonhazardous using an instrument-based method for operational GGP prediction. Accordingly, the ratio $\sigma_{\text{cur}}/\sigma_{\text{lim}}$ should also correspond to a non-hazardous stress state. Assuming that $\sigma_{\text{cur}}/\sigma_{\text{lim}} = 0.6$, substitution into Eq. (8) gives $\beta = 0.00012 \text{ м}^{-1}$. For the seam section considered and the equipment used to record operating-equipment noise over the frequency range of 1–600 Hz, with $f_0 = 1 \text{ Hz}$, Eq. (6) then takes the form:

$$\alpha_i = 0.0038 + 0.00012 f_i \frac{\sigma_{\text{lim}}}{\sigma_{\text{cur}}}, \text{ м}^{-1}; \quad (9)$$

$$f_0 = 1 \text{ Hz}; f_i \in (0; 600 \text{ Hz}).$$

It should be noted, however, that when Eq. (9) is used in algorithms for determining the GGP hazard indicator and criterion in prediction methods based on spectral analysis of operating-equipment noise, the harmonic amplitudes of the line spectrum should be averaged within frequency windows at least 200 Hz wide.

Conclusion

Acoustic signals generated by the roadheader cutting head and propagating through the near-face region of a mine working carry information about the stress state of the rock mass. The principal characteristics of these signals used to assess the rock-mass stress state are the operating frequency spectrum and the dependence of the attenuation coefficient on frequency and ground pressure.

The decrease in acoustic-signal amplitude with increasing distance from the cutting head is non-monotonic and includes an oscillatory component. The signal spectrum also changes substantially as the cutting head moves across the roadway face and penetrates deeper into the rock mass. This reduces the reliability of attenuation-coefficient estimates. To overcome this limitation, it is proposed that the attenuation coefficient be determined from time-averaged data, with the harmonic amplitudes averaged within frequency windows.

The algorithm for determining the effective attenuation coefficient of noise generated by an operating roadheader comprises the following steps: installing several seismic receivers with identical specifications in the rib of the monitored roadway, 20–50 m from the face; simultaneously recording the acoustic signals generated by the cutting head; averaging the harmonic amplitudes over a time window corresponding to the duration of roadheader operation between successive stoppages for roadway rib support installation and within frequency windows 200–300 Hz wide; calculating the attenuation coefficient for the sections between the receiver closest to the face and each of the remaining receivers, followed by averaging the values obtained; and using these data to construct the frequency dependence of the mean attenuation coefficient and its linear extrapolation over the frequency range of 0–700 Hz, or over a wider range in which the attenuation coefficient increases with frequency. This extrapolation is then used to determine the parameters α_0 and β in Eq. (6), which describes the dependence of the effective attenuation coefficient on frequency and the stress state of the near-face region of the roadway.

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