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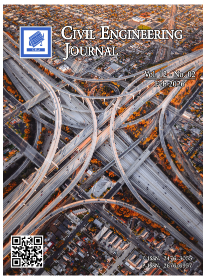

Optimization of Drilling and Blasting Parameters During the Drifting of Underground Mine Workings

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Abstract

The study aims to scientifically substantiate optimal drilling-and-blasting (D&B) parameters for driving underground mine workings under complex geological and mining conditions at the Akzhal deposit. The work addresses the selection of explosive types and the rational depth, configuration, and design of cut boreholes, together with their blasting pattern (BP), to improve rock-mass stability, operational safety, and advance efficiency. The methodology combines an assessment of geological-technical conditions with a review of current blasting practice, mathematical and numerical modelling of blast-induced face breakage, and pilot-scale industrial trials supported by statistical analysis and techno-economic evaluation under routine production constraints and reporting. The results show that optimization of the BP increases the borehole utilization factor (BUF) from $\eta = 0.85$ to $\eta = 0.98$. The locally produced Granulite A6 is proven effective, reducing blasting costs by 1.5 times relative to AS-8 while preserving the required energy characteristics. Charge optimization improves excavation-contour quality, enhances fragmentation uniformity, and reduces overbreak; the most rational solution is a rhombic cut combined with Granulite A6. Scientific novelty lies in integrating geological-geomechanical analysis, 3D modelling in Micromine, and industrial validation. Practical relevance is confirmed by decreasing cycle costs from USD 538.85 to USD 489.38, improving BUF, and enhancing contour quality.

Keywords: Drilling and Blasting Operations (D&B); Underground Mine Workings; Explosives; Borehole Utilization Factor (BUF); Blasting Pattern (BP); Granulite A6; Cut Borehole.

1. Introduction

Sustainable development of the mining industry at the present stage is driven by the need for rational utilization of the mineral resource base and the implementation of resource-saving, high-efficiency mining technologies. The Republic of Kazakhstan, possessing significant reserves of non-ferrous and polymetallic ores, shows a steady trend toward expanding the use of underground mining methods. This transition is driven by the gradual depletion of reserves suitable for open-pit mining, increasing geological complexity, and growing requirements for economic performance, technological reliability, and environmental safety of production processes. Under these conditions, optimizing the development of preparatory, cutting, and capital underground mine workings become increasingly important, since their parameters largely determine profitability, personnel safety, and the overall stability of mining operations [1, 2].

The selection of rational technological solutions for underground mine workings is shaped by a set of factors, including the geological structure of the deposit, rock physico-mechanical properties, hydrogeological conditions, and

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the mining-technical parameters of the mining system. At the Akzhal deposit, underground drivage is predominantly carried out using drilling and blasting (D&B). Its performance is governed by explosive type, charging and initiation parameters, borehole layout and geometry, and the design of cut holes. Integrated optimization of these parameters makes it possible to achieve the required fragmentation, increase the borehole utilization factor (BUF), reduce specific explosive consumption, and improve excavation contour quality by lowering overbreak formation and minimizing blast-induced damage in the near-contour rock mass [3-5].

In mining practice, uniform fragmentation and sizing of ore after blasting play a key role in subsequent processing stages, including beneficiation and the extraction of gold, silver, zinc, and other valuable components. The particle-size distribution determines mineral liberation, ore preparation quality, and the efficiency of hydrometallurgical and flotation processes. In Patent No. 35896 "Method for electrochemical extraction of gold and silver from thiosulphate solutions using a bipolar electrolysis cell" [6], it is shown that an optimal ore size significantly increases the efficiency of electrochemical recovery of precious metals. These findings are also relevant to the production stage, because blast fragmentation quality directly affects downstream processing efficiency. The reported evidence supports the importance of controlling ore size distribution, which is consistent with the objectives of the present study aimed at optimizing D&B parameters. Achieving the required fragmentation at the blasting stage creates favorable prerequisites for efficient subsequent processing, strengthening the technological and economic linkage between mining and beneficiation [7].

Xu et al. emphasize that the controllability of underground blasting is determined by coordinated charge parameters, borehole layout, and initiation timing [8]. However, in many publications the relationship "controlled breakage → BUF/contour quality → cycle economics" is discussed in a fragmented manner; therefore, a reproducible engineering procedure tailored to specific mine conditions remains insufficiently standardized [9].

Rakishev et al. proposed production-oriented automation for fragmentation assessment and approaches to forecasting particle-size distribution during drivage [10, 11]. Nevertheless, studies often focus either on measurement/forecasting of fragmentation or on isolated blasting parameters, while the transition to practical, transferable rules for cut design and charging that explicitly control BUF and contour quality is still limited.

From an economic perspective, integrated frameworks that link cost and critical rock-mass factors with explosive consumption and breakage outcomes are particularly promising [12-15]. At the same time, cost-focused assessments that jointly consider specific charge and D&B costs under varying rock-mass conditions are still not consistently supported by comparable field datasets, despite the growing methodological advances in this direction. Consequently, alternative schemes are not always compared under equivalent drivage conditions, which weakens the evidential basis for selecting an "optimal" solution for a particular horizon.

A key element of the face cycle is the formation of the cut cavity, which is sensitive to geometry, drilling accuracy, and rock-mass quality [16-19]. Yet the robustness of proposed cut designs across a series of cycles under realistic drilling deviations and rock-mass heterogeneity is not always demonstrated; this limits the adoption of such designs as an operational standard without additional verification.

For high in-situ stress conditions, stress-adaptive cut design supported by numerical modeling and field tests is actively developing [20-22]. Even so, operational procedures that systematically adapt cut geometry and charging parameters to stress-structural heterogeneity while delivering a predictable effect on BUF and contour stability are still described with insufficient detail.

Approaches to enhancing breakage efficiency include advanced charge designs (including dispersed charge) and explicit consideration of structural discontinuities when forecasting fragmentation [23-25]. However, in a number of studies the comparable impact of these solutions on BUF, contour quality, and cycle economics is treated separately, complicating integrated decision-making for D&B design.

Overbreak control and contour quality are mandatory conditions for stable and cost-effective drivage, and their dependence on borehole parameters and rock-mass structure is confirmed by recent studies [26-28]. Nevertheless, integration of perimeter blasting with cut parameters (as the primary mechanism creating a free face) and delay schemes often remain outside a unified optimization logic.

Intelligent overbreak control methods and physically based fracture models improve interpretability of design decisions [29-31]. Still, practical applicability is constrained by calibration requirements and limited availability of comparable in-situ datasets, which complicates transferring results to specific mine conditions without additional field verification.

Automation of blasting pattern (BP) preparation and digital optimization demonstrate potential for standardizing design and reducing human-factor variability [32-34]. Yet many solutions are oriented toward separate blocks (e.g., document generation or rock-mass classification), while an end-to-end chain "condition diagnosis → cut/charge/delay selection → drivage KPI control" is not always fully formalized.

Numerical studies of complex underground interactions and expert stability assessment approaches extend the toolbox for justifying decisions in confined conditions [35-37]. A persistent issue is aligning geomechanical stability assessments with D&B parameters as a controllable source of technogenic disturbance, so that design moves from qualitative risk statements to concrete parameter selection. Vibration and dynamic constraints significantly affect delay selection and charge design, as confirmed by studies on vibration characteristics and prediction in underground media [38-40]. However, formulations that optimize vibration constraints jointly with BUF, contour quality, and cycle cost are still relatively scarce.

Dynamic effects in the rock mass, including vibration amplification, and uncertainty in rockburst-related factors require explicit consideration when designing blasting under high stress [41-43]. Yet, embedding these constraints into a unified operational regulation for selecting cut parameters, charges, and delay intervals at the cycle level remains insufficiently widespread. Stress-management methods and preconditioning / destressing are viewed as tools to enhance safety and controllability [44, 45]. However, their coordination with stable contour formation and reproducible drivage performance, while maintaining economic efficiency, requires stronger techno-economic substantiation for specific mine conditions. Charge engineering offers solutions that improve controllability and reduce blast-induced damage in the near-contour zone, including directed and specialized charge configurations [46-48]. Nevertheless, selection criteria between conventional and advanced designs with simultaneous linkage to BUF, contour quality, and cycle cost are often described incompletely. Competition from mechanical excavation and growing requirements for initiation reliability increase the demand for reproducible D&B performance in drivage [49-51]. The influence of initiation reliability and process discipline on statistical variability of BUF and contour quality is frequently accounted for indirectly and should be incorporated more explicitly into comparable field protocols.

Finally, experimental-numerical studies of innovative cut schemes and face-parameter optimization show that even under similar conditions the outcome is highly sensitive to the “geometry-charge-initiation” combination [52, 53]. Practical value is typically limited when alternatives are compared using an incomplete KPI set or without a unified comparability criterion that links economics and drivage performance across a series of cycles.

Despite the substantial body of research on D&B optimization, several fundamental issues remain insufficiently addressed and require further scientific substantiation. The complex geological conditions of the Akzhal deposit necessitate a site-specific D&B design that accounts for structural characteristics of the rock mass, physico-mechanical properties of ore and host rocks, and the influence of tectonic disturbances and hydrogeological factors. D&B efficiency is controlled by a combination of rock-mass properties, explosive type and consumption, drilling parameters, initiation schemes, and BP configuration. Therefore, a relevant scientific and practical task is to develop D&B parameters that ensure high fragmentation quality while minimizing secondary damage, stabilizing excavation contours, increasing BUF, and reducing explosive consumption. Optimization of these parameters can improve productivity and techno-economic performance by reducing drilling and explosive-material costs.

The aim of this study is to develop and scientifically substantiate rational D&B parameters for underground mine workings under the conditions of the Akzhal deposit, considering its specific geological, geomechanical, and mining-technical features. To achieve this aim, the study addresses the following tasks: (i) a comprehensive analysis of structural-tectonic features and physico-mechanical properties of the rock mass in the context of D&B efficiency; (ii) development of a D&B optimization methodology focused on improving technical and economic performance of underground drivage; (iii) substantiation of rational borehole depth considering face geometry and geological conditions; (iv) selection of effective explosive types ensuring the required fragmentation quality and minimizing adverse blast effects; (v) investigation of borehole layout schemes and cut design parameters; (vi) optimization of BUF as an integral indicator of drivage cycle efficiency. The proposed solutions are verified by in-situ pilot trials followed by statistical processing and techno-economic assessment.

This study aims to improve underground drivage technology through comprehensive optimization of the D&B system, including geological-geomechanical analysis, 3D modeling in Micromine, and pilot-scale industrial trials. The scientific significance lies in establishing a systematic approach to selecting D&B parameters that increases rock-mass breakage efficiency while reducing undesirable effects, including excessive explosive consumption, overbreak formation, and deviation from the designed excavation contour. The practical significance is associated with implementing the proposed recommendations at mining enterprises in Kazakhstan, improving operational safety, increasing BUF, reducing drivage cycle costs, and enhancing techno-economic performance of underground mine workings and underground mining under complex conditions of the Akzhal deposit.

2. Geological and Technical Characteristics of the Deposit

As shown in Figure 1, the Akzhal polymetallic ore deposit is in the Shet District of the Karaganda Region, Republic of Kazakhstan, near the Akzhal. The deposit is characterized by a complex geological and structural setting and pronounced rock-mass heterogeneity, which directly influences the selection of drilling and blasting (D&B) parameters and, consequently, the efficiency of underground mining.

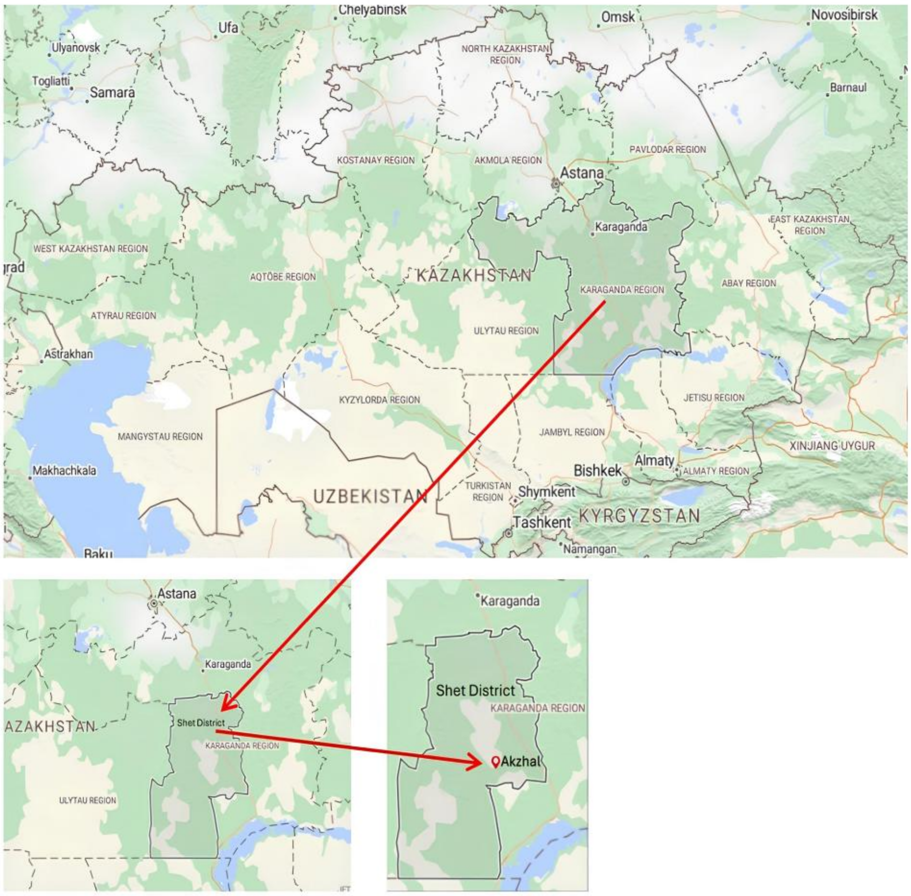


Figure 1. Location map of the Akzhal deposit

The geological framework comprises effusive-sedimentary sequences of the Middle and Upper Devonian and the Lower Carboniferous, formed within the junction zone of three major structures of the Dzungaro-Balkhash geosynclinal region. The northern part of the area corresponds to the southern limb of the Zhaman-Sarysu syncline; the southern part coincides with the northern limb of the Aktau-Mointinsky anticline; and the central zone is confined to the eastern segment of the Akzhal-Aksor syncline. This structural position exerts strong control on the localization of ore bodies, which is associated with a system of linear fractures, faults, and contact zones. These structures are key factors in reserve evaluation and in planning exploration and mining operations. Intrusive rocks are developed to a limited extent and are represented by granodiorites, diorite porphyries, and monzonite porphyries of the Late Permian complex. Their occurrence exerts local control on the physico-mechanical behavior of the rock mass by forming skarn-altered zones and modifying the degree of jointing and fracture intensity.

The lithological succession comprises limestones, sandstones, tuffaceous sandstones, and siltstones characterized by contrasting density and fabric. The average rock density is 2.74 g/cm³ and the loosening (swell) factor is 1.5. Natural moisture content varies from 0.1 to 1.14%, indicating low water inflow and allowing the faces to be classified as predominantly dry for drill-and-blast operations. The reported values and subsequent interpretations are based on the synthesis of long-term site investigation materials and production observations for the deposit. The Akzhal deposit has been investigated from 2017 to the present, with a specific emphasis in successive research programs on systematizing the physico-mechanical properties of rocks across representative intervals and lithological units. To enhance reproducibility, a unified framework for sampling, specimen preparation, and laboratory testing was adopted in this study: core and sample selection, preparation of cylindrical specimens with prescribed geometric tolerances (including end parallelism requirements), and execution of laboratory determinations were performed in accordance with the applicable GOST standards; procedural consistency and terminology were additionally aligned with the ISRM Suggested Methods. Only relevant and representative results were extracted from the accumulated production and laboratory database and used for the present analysis.

The rock mass is distinctly jointed, with fractured intensity varying both with depth and between lithological varieties. The most intensive fracture development is observed at depths up to 100 m, while within tectonically disturbed zones it extends to approximately 200 m. The mean block size ranges from 0.32 to 0.72 m (Table 1). Increased jointing was found to promote more uniform fragmentation and to reduce specific explosive consumption, whereas higher rock strength increases resistance to blast-induced breakage.

Table 1. Characteristics of rock mass fracturing

Rocks (their strength)	Content (%) of blocks by size, cm							Average block diameter, m
	20	21-41	41-60	61-80	81-100	101-120	121	
Intensely fragmented secondary quartzites derived from effusive porphyries (f = 8); granodiorite porphyries (f = 12).	47.1	29.0	10.4	5.3	4.0	3.2	1.0	0.32
Secondary quartzites derived from granodiorite porphyries (f = 12-14): dense secondary quartzites derived from effusive porphyries (f = 12).	15.5	16.1	13.6	13.2	12.1	12.3	17.2	0.70
Marbleized limestone; coarse-grained limestone (f = 10-14).	30.0	20.8	14.3	11.4	9.9	8.1	5.5	0.50
Scarified limestone (f = 14-16).	19.8	17.2	15.5	9.7	8.2	10.5	19.1	0.66
Hornfelsed sandstone; tuffaceous sandstone; diabase and diorite porphyrites; granodiorites (f = 14-18).	28.5	20.5	13.0	11.2	11.0	9.6	6.2	0.52
Actinolite-garnet skarns; epidote-pyroxene skarns (ore-bearing) (f = 14-18); massive limestone (f = 10).	14.0	15.3	12.6	11.4	12.7	14.5	19.5	0.72

Hydrogeological conditions are assessed as favorable for underground operations. Groundwater is predominantly associated with fracture and fracture-karst systems developed in tuffs and limestones. The average hydraulic conductivity (filtration coefficient) is 0.081 m/day, which allows the host rocks to be classified as weakly water permeable. The presence of isolated karst cavities, less than 30 m deep and filled with clayey material, does not significantly affect drilling and blasting (D&B) parameters. The combined influence of geological and technical factors-lithological heterogeneity, spatial variations in fracturing, local tectonic disturbances, and low water inflow-defines the specific requirements for D&B design at the Akzhal deposit. A comprehensive consideration of these characteristics is required to develop rational borehole patterns, select appropriate explosive types, optimize charge parameters, and determine the required undercharging length, thereby improving blast performance and the stability of underground excavation contours.

The deposit is mined using a combined approach (open-pit and underground), employing a high-productivity sublevel caving method with forced caving and diesel-powered mobile equipment (DME). The efficiency and safety of mining operations-particularly during the development of preparatory and cut excavations-depend directly on the quality of D&B implementation. As shown in Figure 2, the subsequent investigation focuses on the haulage (orepass/transport) drift at the +285 m level in the Central sector. This excavation is a key element for analyzing and optimizing D&B, because its successful advance enables the first stage of underground mine development and the establishment of the mine's technological infrastructure.

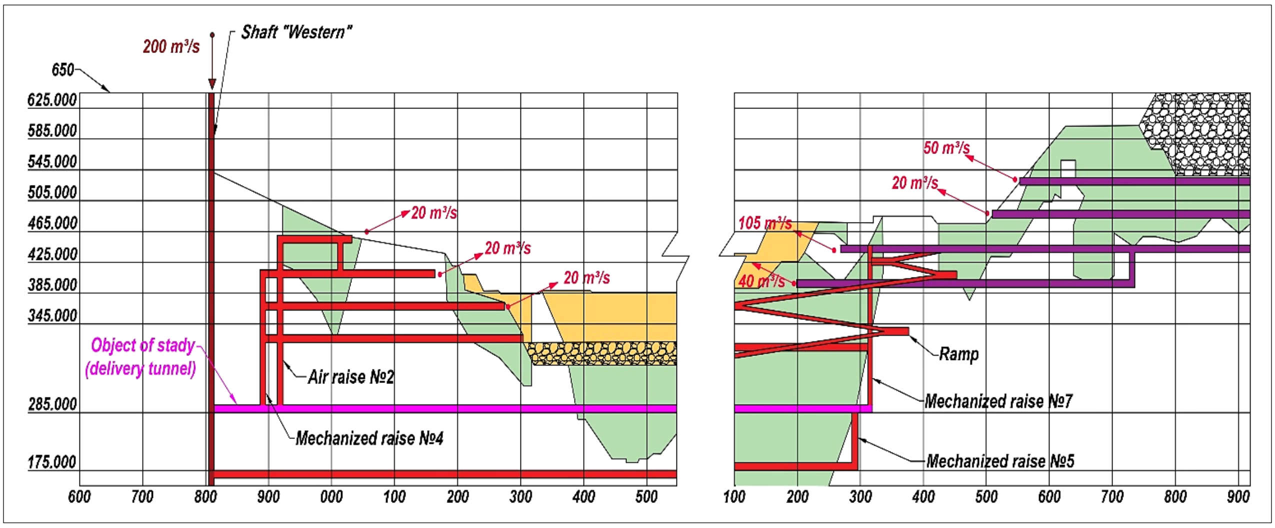


Figure 2. Mine access (opening-up) scheme of the Akzhal underground mine indicating the study object (delivery drift)

According to the mine design, the cross-sectional area of the delivery drift is S = 15 m². The underground mine workings have an arched profile, with a width of 4.10 m and a height of 3.88 m, and the total length of horizontal underground mine workings is approximately 1,500 m (Figure 3).

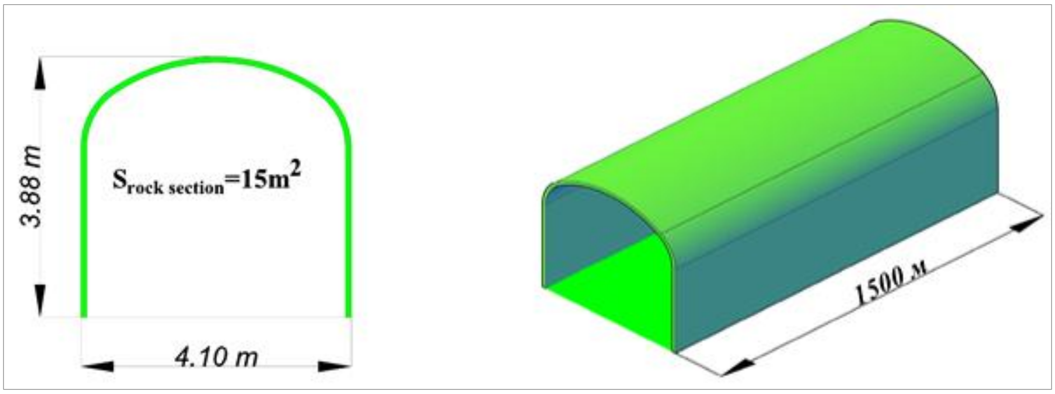


Figure 3. Main parameters of the object under study (delivery drift)

3. Quantitative Assessment of Influencing Factors for Substantiating Optimal Drilling-and-Blasting Parameters

The effectiveness of drilling and blasting (D&B) in underground mine workings is governed by a set of physico-mechanical properties of the rock mass, including density, uniaxial compressive and tensile strength, Young's modulus, the Protodyakonov strength index, and the degree of fracturing. The combined influence of these parameters controls the dominant breakage mechanism, the associated energy demand, and the fragmentation level achieved during blasting.

For the quantitative assessment, field measurement data collected at the Akzhal deposit were used. The Protodyakonov strength index varies from f = 6 to 15, corresponding to rocks of moderate to high strength. The highest values are typical of skarnified limestones (f = 15-20) and intrusive rocks (f = 12-15), whereas lead-zinc ores are characterized by lower strength (f = 7-8). In terms of abrasiveness, the rocks predominantly fall within Classes I-II (very low to low abrasiveness), which contributes to improved durability of drilling tools. The average rock density is 2.70-2.76 g/cm³, and the uniaxial compressive strength ranges from 78 to 83 MPa. For ore bodies (lead-zinc ores), the strength decreases to 58-60 MPa, facilitating breakage under identical charge parameters. The internal friction angle is 31-48°, cohesion is 11.5-25.7 MPa, and Young's modulus is (0.61-0.79) × 10^5 MPa. These indicators suggest that the rock mass includes both elastic and brittle structural domains, which necessitates a differentiated approach to selecting the borehole layout and blasting pattern (BP) for underground mine workings.

Fracturing is one of the key factors controlling blastability. A high density of microcracks promotes more uniform fragmentation and reduces specific explosive consumption, whereas a sparse network of macrofractures can result in oversized fragments and incomplete breakage. At the Akzhal deposit, fracture intensity varies from 2 to 8 fractures per meter; zones affected by tectonic disturbances typically exhibit more efficient fragmentation under identical charge parameters. To minimize the influence of external factors, standardized reference blasting conditions were adopted, corresponding to the breakage of 1 m³ of rock with six free faces and a fragmentation degree of $\eta = 2$. Under these conditions, a reference specific explosive consumption was calculated for Ammonite 6ZhV.

Comparison of the obtained data indicates that massive limestones and acidic tuffs require 15-20% higher explosive consumption than ore bodies, which is attributed to their higher density and Young's modulus. For lead-zinc ores, the optimal specific explosive consumption is 1.8-1.9 kg/m³, whereas for host rocks it is 2.0-2.2 kg/m³. These relationships confirm the dominant role of fracturing in achieving the required fragmentation degree and improving the energy efficiency of D&B in underground mine workings.

The results enabled the derivation of functional relationships between the physico-mechanical properties of the rock mass and breakage efficiency. This provided a basis for substantiating the optimal blasthole depth, charge configuration, and rational specific explosive consumption required to achieve the target fragmentation with minimal energy input.

4. Analysis and Selection of Explosives for the Conditions of the Akzhal Deposit

In the industrial development project for the Akzhal deposit (Kazgiprotsvetmet, Volume 2, Book 2), the baseline commercial explosives specified are Granulite AS-8 for stoping operations and Ammonite 6ZhV for drifting operations. When assessing their practical applicability, the main criteria were energy characteristics, technological suitability for use under underground conditions, safety of manufacture and charging, and economic efficiency (Tables 2 and 3).

Table 2. Primary physical and technical characteristics of Granulite AS-8 and Ammonite 6ZV

<table>
<tr><th>Parameters</th><th>Granulite AS-8</th><th colspan="4">Ammonite 6ZV</th></tr>
<tr><td>Explosive composition</td><td>Oiled granular ammonium nitrate with aluminium powder</td><td colspan="4">Cartridge-loaded powdered TNT-based mixture with water-resistant ammonium nitrate</td></tr>
<tr><td colspan="6">Calculated characteristics</td></tr>
<tr><td>Oxygen balance, %</td><td>-</td><td colspan="4">-0,53</td></tr>
<tr><td>Heat of explosion (water-steam), kcal/kg</td><td>1242</td><td colspan="4">1030</td></tr>
<tr><td>Total ideal work of explosion, kcal/kg</td><td>955</td><td colspan="4">850</td></tr>
<tr><td>Specific volume of gases, l/kg</td><td>847</td><td colspan="4">895</td></tr>
<tr><td colspan="6">Experimental characteristics</td></tr>
<tr><td>Loading density, g/cm^3</td><td>0.87-0.92</td><td colspan="4">-</td></tr>
<tr><td>Cartridge density, g/cm^3</td><td>(1.0-1.1)</td><td colspan="4">-</td></tr>
<tr><td>Work capacity, cm^3</td><td>-</td><td colspan="4">1.0-1.2</td></tr>
<tr><td>Brisance, mm</td><td>410-430</td><td colspan="4">360-380</td></tr>
<tr><td>Critical diameter of an unconfined charge, mm</td><td></td><td colspan="4">14-16</td></tr>
<tr><td>Compression of the lead cylinder in the rock-ring test, mm</td><td>80-100</td><td colspan="4">10-13</td></tr>
<tr><td rowspan="4">Detonation transfer distance between cartridges, cm</td><td rowspan="4">-</td><td>Dry</td><td colspan="3">After 1-hour immersion in water at a depth of 1 m</td></tr>
<tr><td colspan="4">Cartridge diameter, mm</td></tr>
<tr><td>32</td><td>36</td><td>32</td><td>36</td></tr>
<tr><td>5-9</td><td>7-12</td><td>3-6</td><td>4-10</td></tr>
<tr><td>Detonation velocity, km/s</td><td>3-3.6</td><td colspan="4">3.6-4.8</td></tr>
</table>

Table 3. Component composition of the industrial explosive Granulit A6

Parameter name	Parameter value for grades, %	
	PA, APV, AKP	**SAl**
Ammonium nitrate	90±3	90±3
Aluminium	6±2	-
Silicoaluminium (silicon-aluminium alloy)	-	6±2
Petroleum product (oil product)	4±1	4±1

Granulite AS-8 is a granular ammonium-nitrate explosive containing finely dispersed aluminium (≈8%), a petroleum product (≈3-4%), and ammonium nitrate (≈90%). In terms of energy performance, AS-8 is suitable for breaking strong to very strong rocks; however, its application in underground mine workings is constrained by increased fire and explosion hazards during manufacture and pneumatic charging. The presence of aluminium powder with a particle size of 2-10 μm increases electrostatic activity, which may elevate the risk of self-heating and ignition. In addition, the high cost of finely dispersed aluminium makes AS-8 economically inefficient for production at the point of consumption.

Ammonite 6ZhV is a medium-strength, powdered ammonium-nitrate explosive intended for manual charging in dry and water-bearing faces, provided that no gas or dust hazard is present. Despite satisfactory performance, its use is associated with higher operating costs, complex logistics (transportation, storage, and distribution chambers), and reduced operational flexibility under frequent charging cycles.

Considering these limitations, a locally produced formulation-Granulite A6-was proposed and tested under the conditions of the Akzhal deposit. Granulite A6 is produced using the UI-2 unit and subsequently cartridged on the UPP-2 system. Its composition (Table 3) includes ammonium nitrate (≈90%), a petroleum product (≈4%), and an aluminium or silicoaluminium fraction (≈6%). Mixing is performed in the UI-2 batch mixer (Figure 4), whereas cartridging is carried out on the semi-automatic UPP-2 unit, ensuring the required cartridge mass, density, and geometry.

Pilot-scale industrial trials demonstrated the following advantages of Granulite A6. The use of aluminium with a particle size of 50-500 μm reduces electrostatic activity and associated hygienic risks compared with AS-8. The batch process enables rapid adjustment of the formulation's energy parameters depending on mining and geological conditions, while local production reduces logistics costs and decreases dependence on factory-made cartridges.

To support pneumatic conveying under production conditions, Granulite A6 was pre-wetted to a moisture content of 2–3%, which was intended to mitigate electrostatic charge accumulation. Aluminium losses during pneumatic charging were 2–5% (a decrease in aluminium content from 6.0% to 5.7–5.9%), whereas losses of up to 30% were observed for AS-8, leading to an increased specific explosive consumption and higher associated costs. The specific energy of Granulite A6 remained comparable to that of factory-produced AS-8, confirming the technological feasibility of the local formulation when operated within the established production workflow.

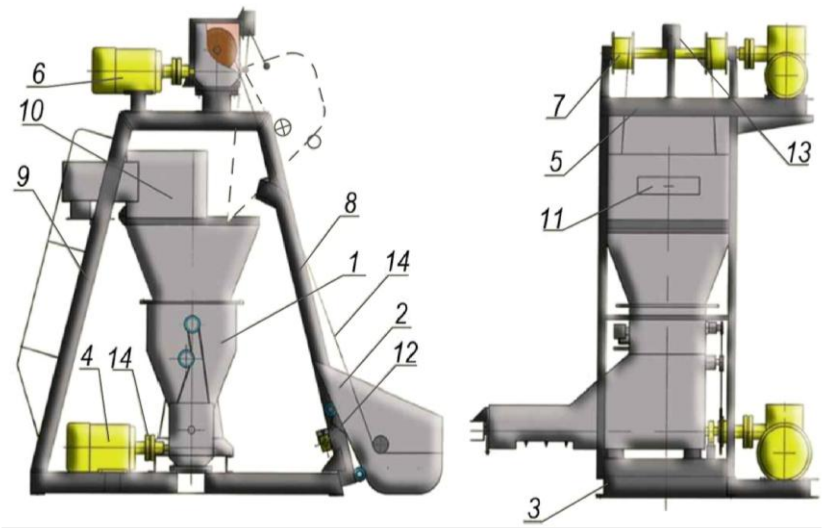


Figure 4. Process flow diagram for the preparation of Granulite A6. (1 - mixer; 2 - skip; 3 - base; 4 - screw rotation drive; 5 - upper frame; 6 - skip lift drive; 7 - drums; 8 - chute; 9 - ladder; 10 - exhaust hood; 11 - granular filter; 12 - limit switches; 13 - bracket; 14 – cable).

The UI-2 and UPP-2 units ensured an adequate level of industrial safety and provided stable product characteristics through routine quality control, including verification of moisture content, particle-size distribution, and the absence of self-heating. Regular incoming inspection of ammonium nitrate, fuel, and aluminium powder further supported the reproducibility of the formulation.

From a techno-economic standpoint, the use of Granulite A6-both bulk and cartridged-delivered comparable energy output while reducing production costs due to lower aluminium losses, local manufacture, and reduced logistics expenses. At the same time, the operational performance of A6 remained dependent on consistent adherence to moisture conditioning, antistatic precautions, and raw-material quality assurance.

The pilot-scale trials indicate that Granulite A6 is well suited to the conditions of the Akzhal deposit, providing the required energy performance, improved sanitary and hygienic working conditions, and economic advantages relative to conventional explosives (AS-8 and Ammonite 6ZhV). Accordingly, within the scope of this study, Granulite A6 produced on the UI-2 unit and cartridged on the UPP-2 system was selected as the primary explosive for implementing the designed drilling-and-blasting (D&B) parameters. Local production of Granulite A6 enabled reduced aluminium losses and achieved a production cost approximately 1.5 times lower than that of AS-8 without deterioration in explosive performance. Analyses of blasthole charges showed minimal aluminium losses (up to 0.3%), while the charge energy remained comparable to that of factory-produced AS-8.

A promising direction for further improving D&B efficiency is the redistribution of charge energy and reduction of energy losses through explicit consideration of geological and technical factors together with charge design. In this context, cartridged explosives of different diameters can improve handling and enhance flexibility when developing blasting-pattern (BP) solutions.

In the present implementation, cartridged Granulite A6 was used for both drivage and stoping operations, including water-bearing rock masses. Ammonite 6ZhV served as the primer cartridge: a single primer (36 mm diameter) for blasthole charges and a double primer for borehole charges in stoping operations.

The choice of the initiation system was guided by the requirements for operational reliability under variable ground conditions (including water-bearing zones), timing consistency, and compatibility with the mine’s approved blasting practices. All blasting activities were performed under an approved D&B passport and site safety procedures, with documented pre-blast checks and traceability of key initiation components to ensure repeatability and risk control During the pilot-scale trials, the emphasis was placed on operational reproducibility: consistency of explosive properties, traceability of initiation components, and standardized pre-blast checks under the mine’s safety management system. This approach strengthens the reliability of the observed performance differences and supports the transfer of the selected D&B parameters into routine practice.

5. Material and Methods

The methodological framework of this study was designed to develop and scientifically substantiate rational drilling-and-blasting (D&B) parameters for driving underground mine workings at the Akzhal deposit (Figure 5).

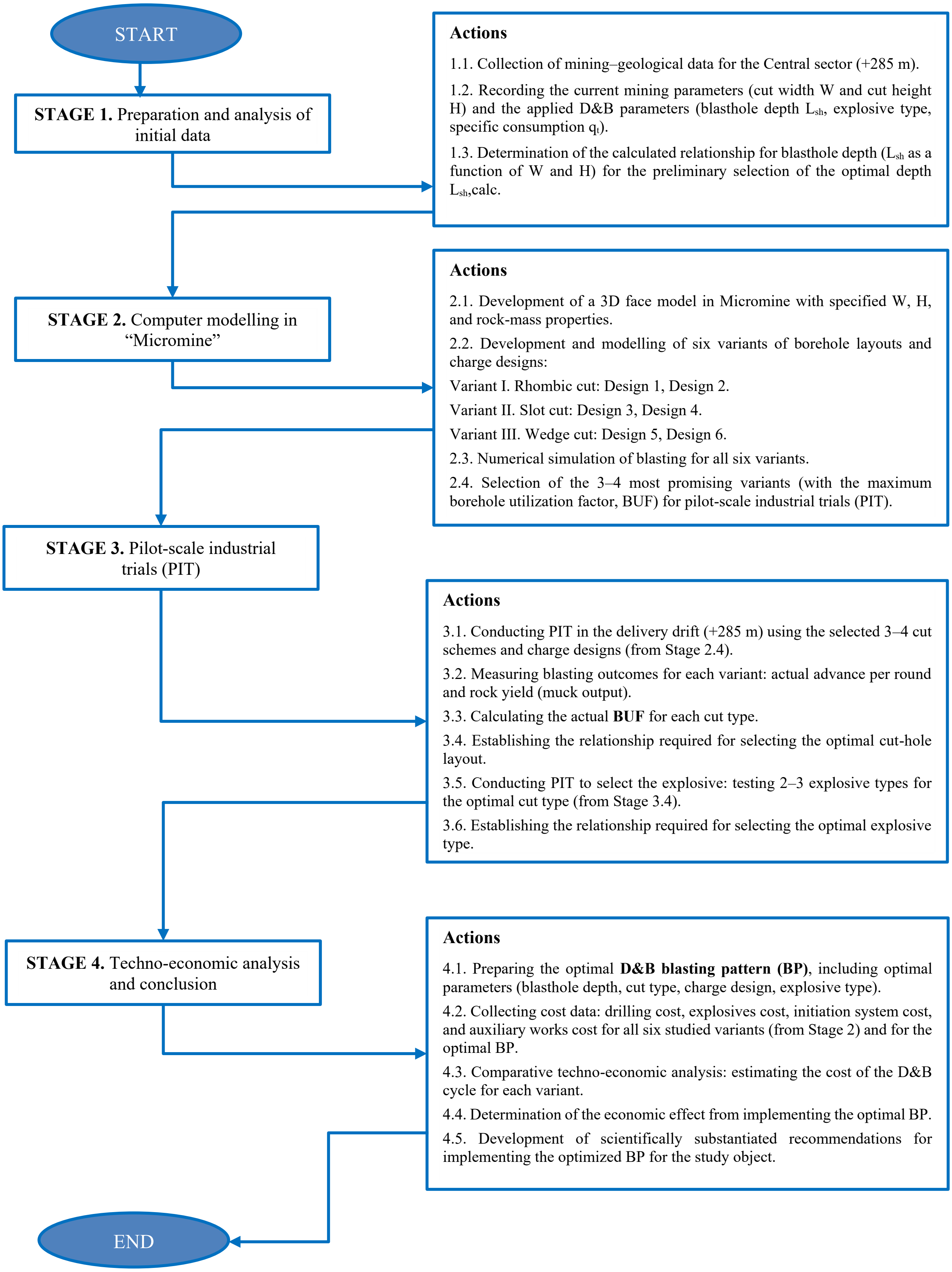


Figure 5. Flowchart of the algorithm for optimizing drilling-and-blasting (D&B) parameters for the delivery drift at the +285 m level

The research included a comprehensive assessment of geological and mining-technical conditions, including orebody geometry and configuration, depth of occurrence, lithological variability of the host rocks, rock-mass fracturing intensity, the presence of tectonic zones, and the groundwater inflow regime. Rock physico-mechanical properties-density, uniaxial compressive strength (UCS), the Protodyakonov strength index, and Young's modulus-were used to characterize rock-mass stability and to predict breakage efficiency under blast loading. To ensure reproducibility, sampling, specimen preparation, and laboratory testing procedures followed relevant national standards (GOST) and internationally accepted ISRM recommended practices for rock testing.

The influence of structural and strength characteristics on D&B performance was evaluated by analyzing functional relationships between blastability and rock-mass fracturing intensity, density, and strength. Increased fracturing was found to promote more uniform fragmentation and to reduce specific explosive consumption, whereas higher rock-mass strength increased resistance to blast-induced breakage. Based on these relationships, blasthole parameters, stemming (uncharged) length, and specific explosive consumption were empirically substantiated for the principal mining and technical conditions encountered at the deposit.

Optimization of blasthole parameters was performed considering key geological and technical factors, including rock strength and fracturing, water inflow conditions and the gas regime, the geometry and cross-section of mine workings, the type and capabilities of drilling equipment, and charge–rock contact conditions. This approach aimed to minimize total labor, material, and energy expenditures per meter of advance while maintaining contour stability and the required fragmentation quality. In addition, the optimization workflow explicitly treated the borehole utilization factor (BUF) and contour quality as operational KPIs, enabling direct comparison of alternative schemes under comparable drivage conditions.

At Stage 1, the initial dataset was collected and systematized (Figure 5), including mining-geological and mining-technical parameters of the deposit and documentation of current D&B practices for subsequent comparative assessment. The condition of underground mine workings and existing drilling and charging procedures was evaluated. Using the collected data, a theoretical calculation of the optimal blasthole depth was performed to establish a baseline for subsequent computer modelling and to determine a depth range that maximizes breakage efficiency while preserving the design contour.

At Stage 2, 3D modelling of the face geometry was performed, and six experimental D&B schemes were developed, differing in cut type (rhombic, slot, and wedge) and charge design. For each scheme, numerical analyses were used to compare expected blast-response characteristics (stress redistribution trends, fragmentation behavior, and the predicted BUF). Based on the modelling outputs, three to four options were shortlisted as the most promising in terms of maximizing BUF, improving energy efficiency, and ensuring more uniform breakage.

At Stage 3, pilot-scale industrial trials of the selected schemes were conducted under operating mine conditions. During the trials, performance indicators were recorded, including advance per round, blasted rock output, BUF, explosive consumption, excavation contour quality, wall and roof stability, and sanitary–hygienic conditions at the face. The key outcomes (advance, BUF, and contour quality) were documented using routine mine surveying and face-mapping practices to ensure that the evaluation procedure remained suitable for operational replication. In parallel, additional tests of explosive types were carried out to select a formulation providing an appropriate balance between energy performance and technological practicality. The field results were used to refine charge parameters and to optimize stemming length; the trial dataset was then summarized using descriptive statistics to support consistent comparisons across the tested schemes.

At Stage 4, the accumulated evidence was integrated: an optimal blasting pattern (BP) was finalized; the D&B cycle cost was calculated for each option; technical and economic indicators were compared; and the expected economic effect of implementing the proposed solution was evaluated. The final outcome was a set of scientifically substantiated recommendations for introducing the optimized D&B scheme into routine production (Figure 5).

For underground operations, locally produced Granulite A6 was adopted as the primary commercial explosive. Manufacturing on the UI-2 unit and cartridging on the UPP-2 system ensured stable energy characteristics, reduced aluminium consumption, improved operational safety, and enhanced sanitary and hygienic working conditions. Process control and incoming inspection of raw materials were implemented to support formulation reproducibility and to maintain stable quality parameters relevant to underground application.

A key element of the methodology was pilot-scale testing of three cut types-rhombic, vertical wedge, and slot-using both bulk Granulite A6 and waterproof cartridged Granulite A6. The comparative analysis showed that a rhombic cut with bulk A6 is the most cost-effective option in dry faces (263,163 KZT per blast cycle), whereas in water-bearing conditions the optimal solution is waterproof cartridged A6 (266,365 KZT per blast cycle). Further optimization of stemming length reduced explosive consumption by 401.3–432.1 kg per cycle while maintaining the required blast energy and high fragmentation performance.

To facilitate practical implementation, the final recommendations were issued as operational blast design sheets and decision rules for selecting the cut type, charge design, and stemming length within the blasting pattern (BP) for dry and water-bearing faces, thereby supporting repeatability and stable KPIs (advance, BUF, contour quality, and cycle cost).

Overall, the results confirm the effectiveness of the proposed methodology, the reproducibility of the experimental dataset, and its applicability to improving D&B technologies in underground mines with comparable geological and mining-technical conditions. The proposed approach supports scientifically grounded decision-making aimed at increasing productivity, reducing costs, and ensuring safe working conditions under routine operational constraints. In addition, the workflow is scalable: the same comparison logic can be applied to new faces by updating rock-mass inputs and operational constraints, preserving methodological consistency across local variability. The resulting blasting pattern (BP) documentation provides traceable design rationale, improving auditability and facilitating knowledge transfer between shifts and sites. Finally, production-round feedback (KPIs and observed deviations) can be incorporated into subsequent designs without changing the overall decision structure, enabling continuous improvement and better control of performance variability.

6. Results and Discussion

6.1. Optimization of Borehole Parameters

Borehole depth is a key parameter of the drilling and blasting (D&B) system because it governs the duration of the drivage cycle, labor intensity, advance per round, and the overall cost of constructing underground mine workings. Rational selection of borehole depth should ensure the target advance while minimizing specific time, material, and labor inputs per 1 m of drivage.

Substantiation of borehole depth requires comprehensive consideration of geological, technical, and organizational factors. Geological factors include rock strength and fracturing, bedding (stratification), the presence of tectonically disturbed zones, water inflows, and the gas regime. Technical factors comprise the applied drivage scheme, the shape and cross-sectional area of the underground mine workings, the type and performance of drilling equipment, and the characteristics of explosives and initiation systems. Organizational factors are defined by the drivage-cycle duration, the productivity of development crews, equipment operating schedules, and the logistics of auxiliary operations.

At insufficient borehole depths (l = 1.0-1.5 m), the share of auxiliary activities in the drivage cycle increases markedly (face ventilation; preparatory and completion operations during drilling, loading, charging, and blasting). This results in a higher number of cycles per unit time, greater time losses due to repeated procedures, and lower overall techno-economic efficiency. In addition, shallow drilling complicates stable formation of the cut cavity and the required free volume, thereby adversely affecting the borehole utilization factor (BUF) and the achieved advance per cycle.

At excessively large borehole depths (l = 3.5-4.5 m), the mechanical drilling rate typically decreases and borehole quality deteriorates (deviation from the design direction and increasing layout errors). At the same time, the likelihood of incomplete utilization of blast energy increases, fragmentation uniformity decreases, and uncontrolled crack development in the near-contour zone becomes more pronounced. As a result, BUF decreases, the time required to advance 1 m of underground mine workings increases, and specific explosive consumption rises, thereby reducing the economic efficiency of drivage operations.

Therefore, borehole depth should be selected to achieve the required advance while accounting for the performance of drilling and load-haulage equipment, as well as the geometric parameters of the face (Figure 6). In general form, borehole depth is determined by the condition of attaining the required advance rate of underground mine workings:

$$l_b = \frac{L}{N i n_m n_c \eta}, m, \tag{1}$$

where, L - length of underground mine workings, m; N- number of working days per month, units; i - the time allocated for driving the underground mine workings, months; nm - the number of working shifts per day, units; nc - the number of cycles per shift, units; η - borehole utilization factor (BUF).

Dependence of borehole depth *l* on the excavation width *B* and excavation height *H* of underground mine workings (as excavated), where Series H2–H5 represent the corresponding calculated curves.

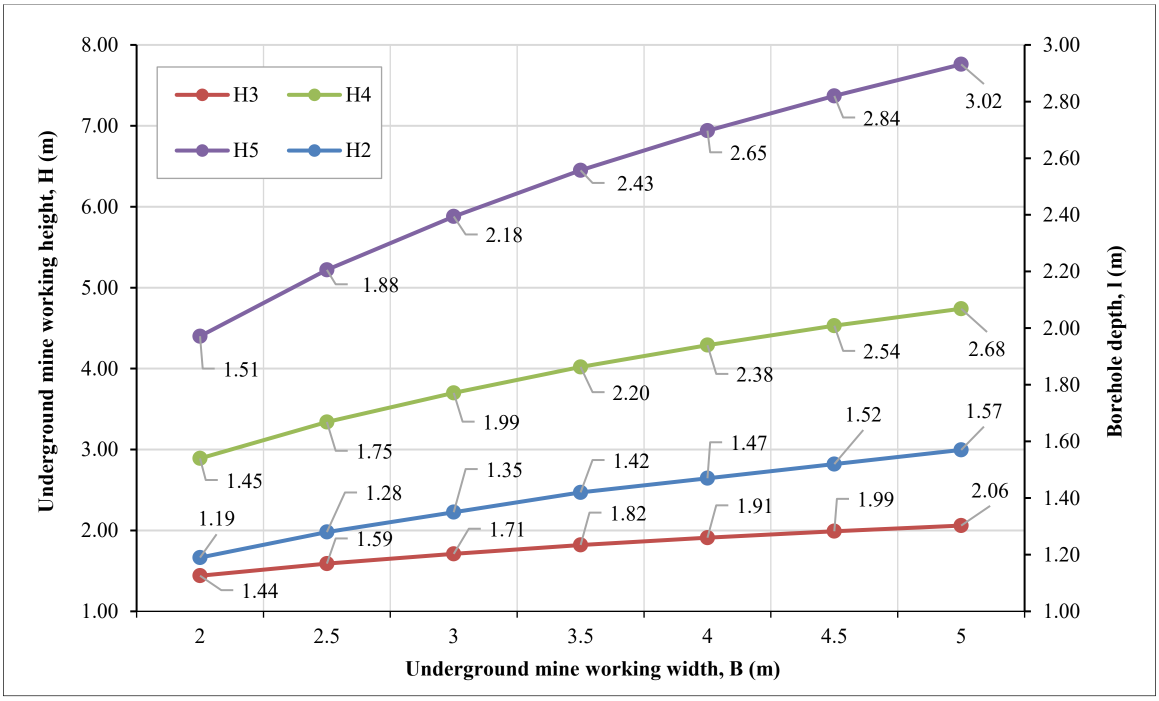


Figure 6. Relationship between borehole depth l and the excavation width B and excavation height H of underground mine workings, where Series H2–H5 denote the corresponding calculated curves

The obtained relationships are consistent with the empirical correlation l = (0.6-0.7)B, which is widely used in cut design, and they confirm that borehole depth should be assigned with explicit consideration of working geometry rather than based on averaged values. Insufficient drilling depth leads to inadequate cut opening, a reduction in BUF, and under-advance per cycle, whereas an excessive increase in l results in overdrilling, higher costs for preparatory operations, and intensified blast-induced damage in the near-contour zone. Therefore, use of these relationships enables technically substantiated optimization of drilling parameters, improves the utilization of explosive energy, enhances the contour quality of underground mine workings, and supports stable advance rates while meeting industrial safety requirements.

The results also demonstrate high reproducibility of the identified trend and its suitability for practical application in developing the blasting pattern (BP). Under the conditions of the Akzhal deposit, a technologically justified borehole-depth range is approximately 2.8-3.2 m, providing higher BUF, lower specific explosive consumption, improved geometric stability of the design contour of underground mine workings, and reduced labor intensity of drilling operations, thereby increasing the overall techno-economic efficiency of drivage.

6.2. Modelling and Development of a Drilling and Blasting Passport in Micromine Software

A digital D&B model was developed using Micromine, which enables construction of three-dimensional face models, design of borehole-layout parameters, and subsequent preparation of the BP. At the first stage, a detailed 3D model of the delivery drift face was created, incorporating the actual face geometry, cross-sectional parameters, and the structural features and physico-mechanical properties of the rock mass. The use of digital modelling ensured close alignment of design solutions with the real mining and geological conditions and increased the reliability of the calculated drilling and charging parameters.

Based on the geometric model, three cut-layout variants were designed, differing in cut type: rhombic (Figure 7-a), vertical slot (Figure 7-b), and wedge (Figure 7-c). For each variant, two charge designs were specified, resulting in six experimental D&B schemes intended for comparative evaluation of their technical and economic performance. The variability of the design solutions was achieved by changing: (1) the cut type (rhombic, vertical slot, wedge); and (2) the charge design, including two explosive application modes: Variant 1-cartridged Granulite A6; Variant 2-bulk (granulated) Granulite A6.

The borehole layouts and charge designs developed in Micromine provided the basis for preparing the blasting pattern (BP), including the spatial arrangement of boreholes, drilling parameters, and charging solutions. For each scheme, numerical modelling was performed to determine the following: the extent of the rock-mass breakage zone; the shape and dimensions of the cut cavity; loosening patterns and local confinement effects in the surrounding rock mass; and the predicted borehole utilization factor (BUF). Based on comparative analysis, the most efficient solutions were selected and recommended for subsequent pilot-scale industrial trials.

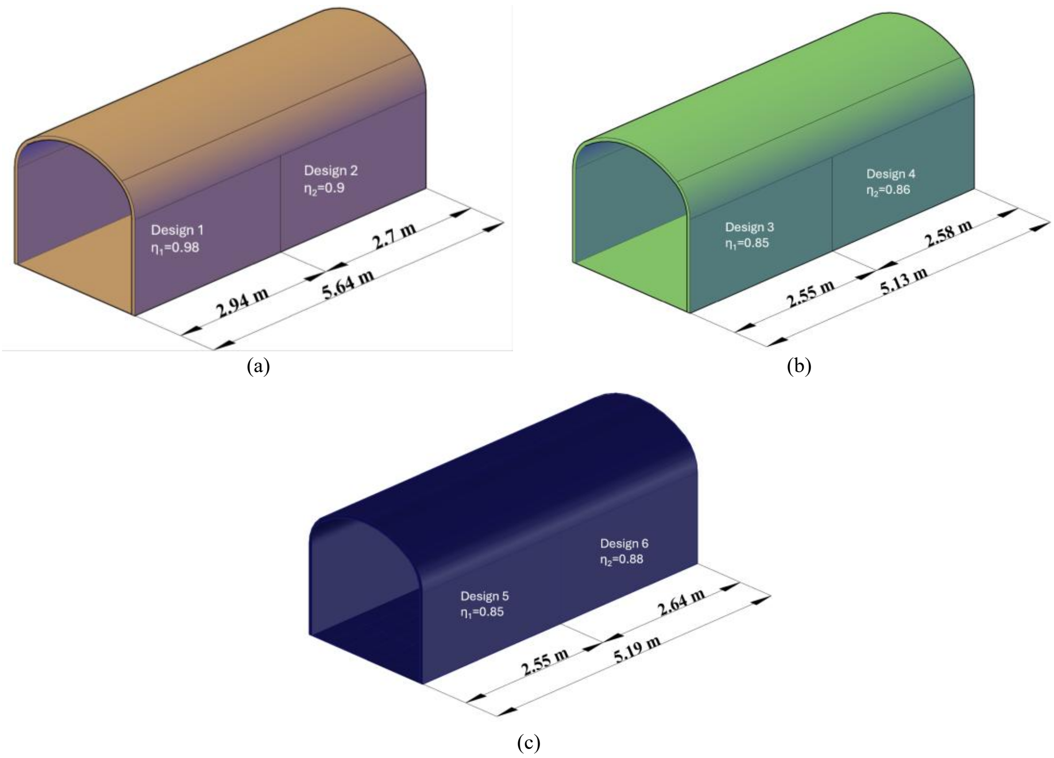


Figure 7. Sequence of modelling and testing the location of cut-boreholes designs

The energy potential of charges is governed by the specific energy and density of the explosive. Substantiation of charge mass and its distribution along boreholes requires accounting for the rock volume to be broken-its lithology, fracturing, and strength characteristics-as well as face geometry and the required cut-cavity configuration. Therefore, an integrated approach was used to evaluate rational D&B parameters by combining underground mine workings geometry, rock-mass properties, explosive application mode, and charge placement pattern. Considering the mining-geological and mining-technical conditions of the Akzhal deposit and preliminary results, three baseline cut schemes were selected: rhombic, slot, and wedge. For each scheme, calculation relationships for assigning charge parameters were developed. The modelling results indicated that the variants differ in their ability to achieve the required advance per round, maintain stable formation of the initial free volume, and ensure uniform fragmentation of the rock mass under the recommended parameters.

For calculating D&B parameters for underground mine workings with a cross-sectional area of S = 15 m^2 , the input specifications of the Boomer-104 drilling rig were adopted (borehole diameter d = 45 mm, borehole length l = 3.0 m, BUF ≈ 0.85-0.98). The broken-rock volume and explosive consumption were calculated using empirical relationships (Equations 4.1-4.6), considering the rock-structure coefficient S_1 = 1.4 and the relative explosive performance coefficient e_1 = 0.95. As stemming and for filling the borehole collar space, a sand-clay slurry with S:C:W = 1:3:0.6 was proposed, placed mechanized using the PPM-90 unit. The calculation relationships for three cut types-rhombic, wedge, and slot-are summarized in Table 4. The proposed equations are empirical-analytical and intended for practical application in BP design.

Table 4. Calculated parameters for three types of cuts (empirical-analytical relationships)

Parameters	Formulas	Note
Rhombic		
Specific consumption of explosives	$q = q_p \cdot S_1 \cdot V_1 \cdot e_1, kg/m^3$	S_1 - rock structure coefficient, for rocks with low fracturing, for the Akzhal deposit, S1=1.4 can be used; V_1- rock clamping coefficient with one open surface
Explosive consumption per cycle	$Q_c = v_{r.m} \cdot q \cdot \eta, kg,$	η - borehole utilization factor;
Number of boreholes at the face	$N = \frac{Q_c}{q_c}, p.$	q_c - charge mass in one borehole, kg.
Line of least resistance	$W = \sqrt{\frac{q_n}{qm}}, m$	m - charge spacing (convergence) coefficient (for strong rocks, m = 1.0 may be adopted); q_n - capacity of one borehole, kg
number of cut boreholes	$n_c = \frac{0,5N}{3}, \ p.$	Empirical estimate of the proportion of cut boreholes.
Number of compensation boreholes	$n_{com} = \frac{0,5N}{n_c}, p$	Compensation (empty / uncharged) boreholes.

Vertical wedge cut		
Number of boreholes in faces of all mine workings	$N = \frac{1,27qS_W}{\Delta d^2 K_f}$,	Δ - explosive density in the borehole/cartridge, kg/m^3; d - borehole (or explosive cartridge) diameter, m; K_f - borehole filling coefficient;
Number of compensation holes	$N_o = \frac{\left(\eta \cdot l_b/_A\right)^3}{V_o}$	l_b - borehole depth, m; A=9.35 - scale coefficient; V_0 - volume of an empty borehole, m^3,
The line of least resistance	$W = [^p/_{qm}]^{1/2}, m$	p - capacity (charge per 1 m of borehole), kg/m; q_{m1}- capacity of 1 m of borehole (under charging conditions).
Required consumption of explosives	$Q = q \cdot S_w \cdot L_b,\ kg$	q - specific explosive consumption, kg/m³.
Slotting cut		
Number of boreholes at the face	$N = N_c + \frac{q \cdot S_W}{f_1}$.	N_C - number of cut boreholes, pcs.; S_W - cross-sectional area, m²; f_l- rock confinement coefficient for one free face.
Number of non-charged, compensating holes in a slotted cut with the same diameter	$N_{n-c} = \frac{0,85S_W}{2 \cdot (W_1 + \eta)} - f_1$	W1 - distance between charged and compensation boreholes (typically 0.3 m); η – BUF; f_l - rock clamping coefficient with one open surface
Specific consumption of explosives	$q = q_n \cdot S_1 \cdot f_1 \cdot e_1, kg/m^3$	q_n - normal specific explosive consumption (for rocks with f = 12-14, q_n = 1.02 may be adopted); S_1 - rock structure coefficient; e_1 - relative explosive performance coefficient.
Required consumption of explosives	$Q = q \cdot S_w \cdot L_b$, кг	S_m - - working cross-sectional area, m²; L_b - borehole depth, m.

Note: SI units are used in the calculations; coefficient values are selected in accordance with the mining-geological conditions and the characteristics of the applied explosive.

The empirical-analytical relationships in Table 4 were used to justify D&B parameters during BP design for the three cut types. Using input data on underground mine workings geometry, rock-mass characteristics, and explosive parameters, the specific explosive consumption, total explosive consumption per cycle, required number of boreholes, and cut-zone parameters (including the line of least resistance and the number of compensation boreholes) were determined. These calculated values formed the basis for developing borehole layouts and charging schemes in Micromine and for subsequent comparative assessment of the variants with respect to advance per round, stability of cut opening, contour quality of underground mine workings, and predicted BUF.

6.3. Comparative Analysis of Cut Schemes Based on 3D Modelling Results

An additional study was conducted to substantiate the rational cut type using three-dimensional modelling of borehole layouts and analysis of charge influence zones. In Micromine, a 3D model of the delivery-drift face was constructed and analysed, enabling explicit consideration of the actual geometry of the underground mine workings and the spatial configuration of the blasting pattern (BP). Within the modelling framework, three technologically relevant cut types were evaluated: rhombic, vertical slot, and wedge. For each cut type, two charge designs were specified, resulting in six alternative drilling and blasting (D&B) schemes (Figures 8 to 10).

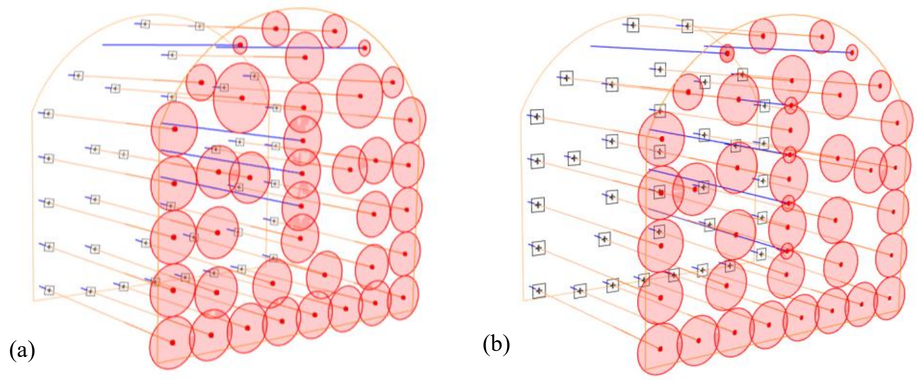


Figure 8. Borehole layout at the face for a rhombic cut with different charge designs: (a) Design 1; (b) Design 2 (Variant I)

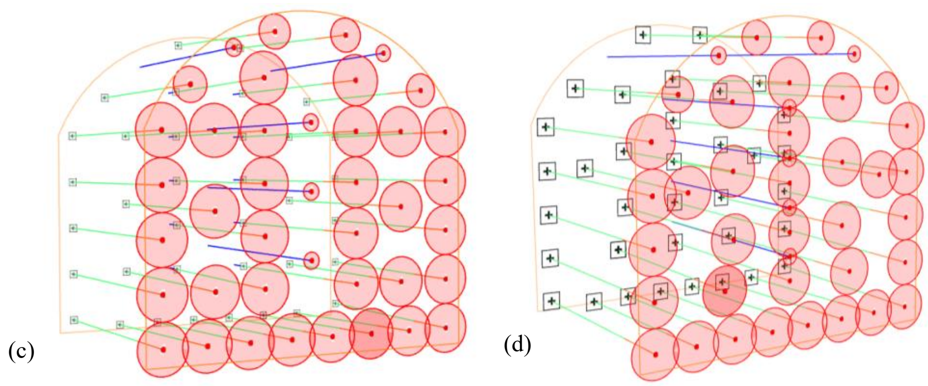


Figure 9. Borehole layout at the face for a vertical slot cut with different charge designs: (c) Design 3; (d) Design 4 (Variant II)

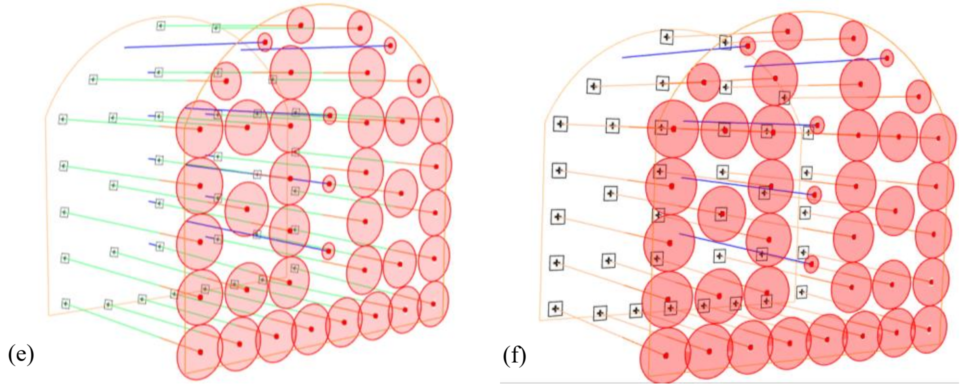


Figure 10. Borehole layout at the face for a wedge cut with different charge designs: (e) Design 5; (f) Design 6 (Variant III)

Attention was given to uncharged (empty) boreholes, which act as an artificial free surface and a guide for rock breakage. In Figures 8 to 10, these boreholes are highlighted in purple. Their use reduces rock-mass confinement during cut initiation, promotes smoother formation of the design contour, decreases the occurrence of bootlegs, and limits blast-induced cracking in the near-contour zone. Consequently, a larger proportion of explosive energy is directed to rock breakage, enabling more rational explosive consumption.

The rhombic cut (Variant I) is widely regarded as an effective method for creating the initial free volume in strong and fractured rocks. For this cut type, Design 1 (cartridged Granulite A6) and Design 2 (bulk/granulated Granulite A6) were modelled (Figure 8). The rhombic configuration concentrates explosive energy in the central zone of the face and promotes stable opening of the cut cavity, thereby creating favorable conditions for subsequent breakage by the production boreholes.

The vertical slot cut (Variant II) is characterized by the formation of a vertically oriented slot-shaped free surface created by uncharged boreholes. In this study, Design 3 (cartridged Granulite A6) and Design 4 (bulk/granulated Granulite A6) were evaluated (Figure 9). In this configuration, charged boreholes operate in conjunction with empty ones, which improves controllability of rock breakage and partially limits damage beyond the design contour, thereby enhancing contour quality. However, the effectiveness of the slot cut strongly depends on drilling accuracy and on consistency between charging parameters and face geometry.

The wedge cut (Variant III) is formed by two or more pairs of converging boreholes that create a wedge-shaped cavity (Figure 10). Design 5 (cartridged Granulite A6) and Design 6 (bulk/granulated Granulite A6) were evaluated. The effectiveness of the wedge cut is governed by the convergence angle of the boreholes, the spacing between collar positions, and the relative positioning of the charges. Deviations from the design borehole orientation and insufficient drilling accuracy can deteriorate cut opening and reduce breakage efficiency, which is particularly critical in strong rocks and under limited free-surface conditions.

A comparative assessment of charge-overlap zones, blast-energy distribution, and the predicted borehole utilization factor (BUF) indicated that the rhombic cut has a clear advantage over the vertical slot and wedge variants. This advantage is attributed to a more uniform and concentrated energy distribution in the central zone of the face, which promotes stable formation of the initial free volume and increases the expected BUF under the conditions of the delivery drift at the +285 m level. Based on the modelling results, the rhombic cut was adopted as the baseline scheme for pilot-scale industrial trials.

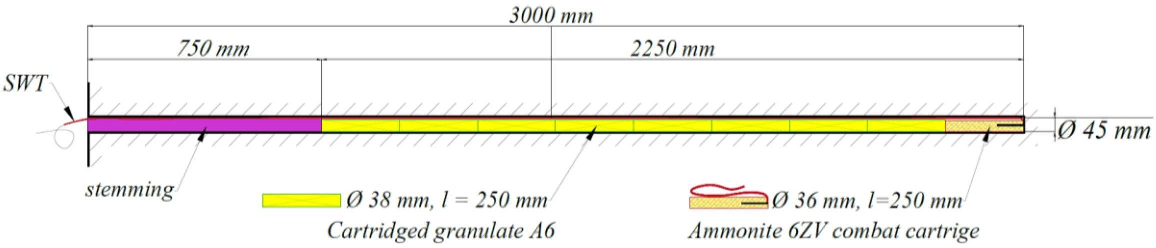
Summary: Based on the 3D-modelling results, the most rational borehole-layout scheme was identified as Design 1 of the rhombic cut (Figure 8a), which provides an optimal combination of outcomes: higher rock-mass breakage efficiency, improved contour quality of underground mine workings, and reduced adverse effects (bootleg formation and excessive disturbance of the near-contour zone). In contrast, under the conditions considered, the vertical slot cut exhibited less stable opening of the cut cavity and lower technological performance, which limits its applicability. The wedge cut also did not achieve the required efficiency and was inferior to the rhombic option in terms of cut-zone quality and fragmentation uniformity, indicating that its use is justified only under suitable conditions and when high drilling accuracy can be ensured. Application of 3D modelling in Micromine enabled development of directly comparable drilling and charging schemes and supported a well-grounded selection of the rational cut type at the blasting pattern (BP) design stage. In terms of initial free-volume formation and predicted BUF, the rhombic cut demonstrated the highest effectiveness due to a more favorable distribution of blast energy in the central zone of the face. The use of uncharged boreholes improves controllability of rock breakage and enhances contour quality of underground mine workings by reducing bootlegs and limiting blast-induced disturbance in the near-contour zone. For the delivery drift at the +285 m level, Design 1 of the rhombic cut is therefore considered the rational solution and is recommended for pilot-scale industrial trials and subsequent industrial application under similar mining and geological conditions.

6.4. Pilot-Scale Industrial Investigations under the Study-Object Conditions

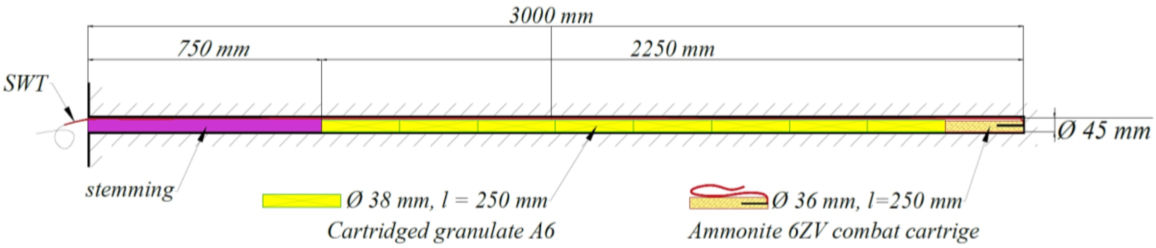
Following assessment of the factors governing drilling and blasting (D&B) performance during the construction of underground mine workings (including physico-mechanical properties of the rock mass; borehole diameter, depth, and number; cut type; explosive characteristics; charge design and charge placement; blast-process modelling; and related parameters), six pilot-scale industrial blasting trials were conducted at the study object-the drift at the +285 m level of the Akzhal deposit. During the trials, the cut scheme and charge design were varied while maintaining comparable mining-technical conditions, ensuring valid comparisons between the experimental variants.

The performance of each variant was evaluated using a set of technological and quality indicators, including the borehole utilization factor (BUF), advance per cycle, fragmentation characteristics (lump-size distribution) of the blasted rock, and contour quality of underground mine workings (bootleg formation and the degree of blast-induced cracking in the near-contour zone). This integrated approach enabled identification of a rational cut scheme and an appropriate application mode of Granulite A6, improving both breakage efficiency and key parameters of the drivage cycle.

For the underground conditions of the Akzhal deposit, locally produced Granulite A6 was found to be the most suitable option because it provides sufficient specific energy and relatively low dust generation, thereby improving sanitary and hygienic conditions at the face. In dry faces, bulk (granulated) Granulite A6 with mechanized charging is recommended (Figure 11-b), whereas in water-bearing boreholes, cartridged Granulite A6 with a diameter of 38 mm is advisable (Figure 11-a), ensuring stable charging and reducing the risk of charge washout.



(a)



(b)

Figure 11. Charge designs used in the experimental programme: (a) design with cartridged Granulite A6; (b) design with bulk (granulated) Granulite A6

The trial results confirmed the superiority of short-delay blasting (SDB) over instantaneous initiation. The use of SDB increases advance rates, reduces drilling volume and explosive consumption by 10-20%, improves fragmentation, mitigates seismic effects, and increases the borehole utilization factor (BUF) by 10-15% due to controlled energy release and sequential formation of free surfaces. The rational delay intervals are 20-30 ms between cut and production boreholes and 10-20 ms between production and contour boreholes. With increasing rock strength, the delay intervals should be reduced to ensure stable opening of the cut cavity. To prevent excessive muck throw along the axis of the underground mine workings under SDB, a 10-15% reduction in specific explosive consumption is recommended while maintaining the required breakage effectiveness.

The Granulite A6 initiation scheme was implemented using non-electric initiation systems (NeIS), including SINV (FGUP "NMZ Iskra", Novosibirsk, Russian Federation) and EXEL (Orica-Kazakhstan JSC, Ust-Kamenogorsk). These systems provide reproducible timing sequences at short delays, which is critical for effective SDB implementation and for controlling the rock-breakage process. If additional redundancy is required, the scheme can be supplemented by electric initiation (instantaneous electric detonators and detonating cord), thereby increasing robustness and operational flexibility under production conditions.

To obtain a reliable assessment of D&B performance, trials were conducted using three cut variants: (1) rhombic cut (Test I-cartridged Granulite A6; Test II-bulk/granulated Granulite A6); (2) vertical slot cut (Test III-cartridged Granulite A6; Test IV-bulk/granulated Granulite A6); and (3) wedge cut (Test V-cartridged Granulite A6; Test VI-bulk/granulated Granulite A6). The comparative indicators derived from the trial blasts are presented in Figure 12.

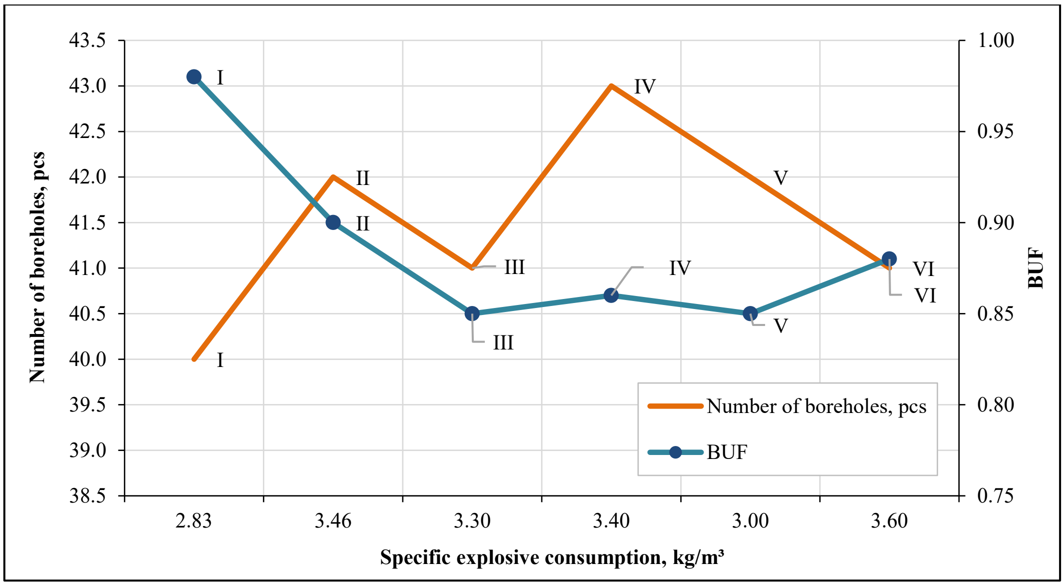


Figure 12. Comparative assessment of drilling and blasting performance based on pilot-scale industrial trials: variation of the borehole utilization factor (BUF) and drivage-cycle parameters as a function of specific explosive consumption

Figure 12 summarizes the results of six trials (I-VI) and illustrates how BUF and key drivage-cycle parameters vary with specific explosive consumption. In Trial I, at a specific consumption of 2.83 kg/m³, the highest performance was achieved, with BUF = 0.98 and the largest advance over two cycles (5.88 m). This indicates highly effective utilization of blast energy for rock breakage and stable formation of the initial free volume.

In Trials II-III, increasing specific explosive consumption to 3.46 and 3.30 kg/m³ led to a decrease in BUF to 0.90 and 0.85, respectively, accompanied by reduced advance. This trend indicates less favorable cut-opening conditions and increased ineffective energy losses. In Trials IV-VI, at 3.4-3.6 kg/m³, BUF stabilized within 0.85-0.88, while advance did not exceed 3.6 m. These results confirm that higher explosive consumption does not necessarily translate into higher blasting efficiency and indicate the onset of a technological saturation regime. Therefore, further increases in specific explosive consumption are unlikely to improve D&B performance and may intensify blast-induced damage in the near-contour zone and deteriorate the contour quality of underground mine workings.

The identified relationship can be applied as an engineering criterion for optimizing D&B parameters during blasting pattern (BP) development, enabling substantiated selection of cut type, charge design, and rational specific explosive consumption to achieve maximum BUF and the required advance while minimizing adverse effects.

6.5. Fragmentation Analysis of the Blasted Rock Mass

During the pilot-scale industrial trials, fragmentation of the blasted rock mass was evaluated as a key D&B quality indicator because it governs the efficiency of load-and-haul operations and the stability of the drivage cycle. Post-blast fragmentation was assessed using a linear measurement approach by recording characteristic fragment dimensions within the blasted zone, followed by processing of the collected data.

The most favorable fragmentation was obtained for the rhombic cut with Charge Design 1, where the mean linear fragment size was approximately 60 mm. For the rhombic cut with Charge Design 2, slightly larger-but technologically acceptable values were recorded, at approximately 85 mm. The remaining cut variants produced coarser fragmentation, which reduces drivage effectiveness and may increase the proportion of oversized fragments (Figure 13).

Figure 13 summarizes the results of Trials I-VI and illustrates the dependence of rock fragmentation (mean fragment size) and the borehole utilization factor (BUF) on specific explosive consumption. The data show that, as specific explosive consumption increases from 2.83 kg/m³ (Trial I) to 3.60 kg/m³ (Trial VI), the mean fragment size increases from 60 mm to 135 mm. This trend indicates that, under the considered mining and geological conditions, increasing specific explosive consumption does not intensify fragmentation; instead, it produces a coarser size distribution. A plausible explanation is redistribution of blast energy into ineffective losses (gas-dynamic effects, throw, and local stress relief), together with less favorable cut-opening conditions associated with a non-rational blasting pattern (BP) configuration.

The behavior of BUF exhibits an opposing trend: the maximum values (≈0.95-0.98) are achieved at the lowest specific explosive consumption in Trial I, after which the indicator decreases and stabilizes within 0.85-0.92 at higher consumptions. This confirms that increasing energy input by raising explosive consumption does not improve breakage efficiency and may be accompanied by less complete borehole utilization, increased disturbance of the near-contour zone, and reduced contour quality of underground mine workings.

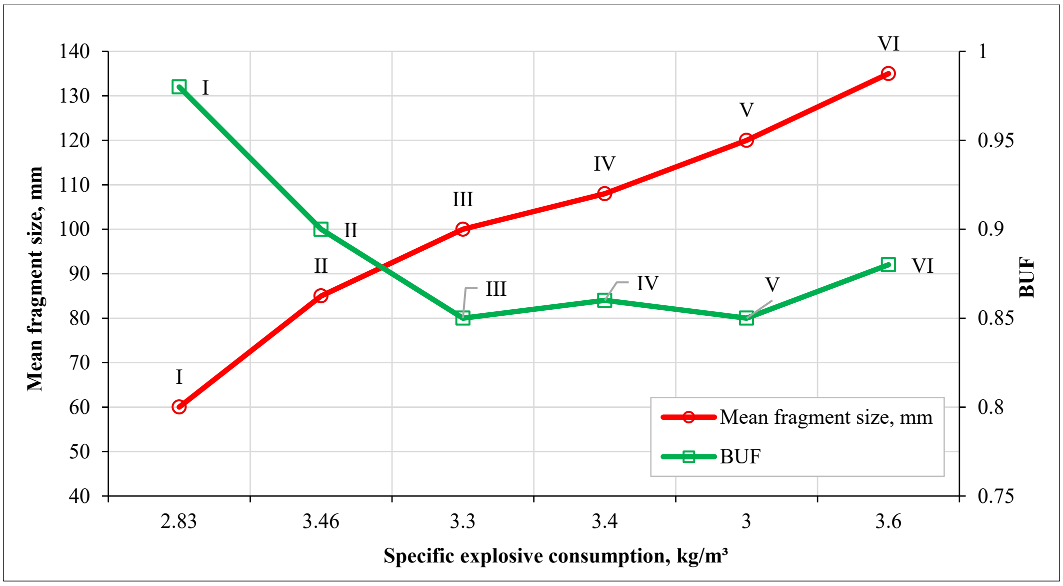


Figure 13. Effect of specific explosive consumption on rock fragmentation (mean fragment size) and the borehole utilization factor (BUF) based on pilot-scale industrial trials

Overall, the results indicate that optimal fragmentation is achieved not by maximizing specific explosive consumption, but by a rational combination of D&B parameters, including cut type, charge design, and initiation scheme. Under the conditions of the Akzhal deposit, the most favorable option is the rhombic cut (Design 1), which provides the finest fragmentation while maintaining high BUF, thereby improving loading productivity, drive-cycle stability, and the overall techno-economic performance of D&B.

6.6. Assessment of the Contour Quality of Underground Mine Workings after Face Blasting

Contour quality of underground mine workings after face blasting is a key criterion for evaluating D&B performance because it governs near-contour rock-mass stability, the extent of bootleg formation, the need for additional scaling/trim blasting, and the scope of ground-support works. Within the pilot-scale industrial program, a geometric assessment of the actual contours obtained under different D&B variants was performed to identify the scheme that most accurately reproduces the design profile while minimizing blast-induced damage in the near-contour zone.

Figure 14 shows the contours of the underground mine workings formed using a rhombic borehole layout and Charge Design 1, which were identified as optimal based on the modelling and field-trial results.

(a)

(b)

Figure 14. Geometric assessment of the contour configuration of underground mine workings after drilling and blasting at the Akzhal mine: (a) actual cross-section; (b) comparative assessment of roof contours (I-optimized scheme; II-baseline blasting pattern (BP)).

Figure 14-a shows the actual cross-section of the drift after blasting. The actual contour (highlighted in red) exhibits close agreement with the design profile, indicating rational distribution of blast energy within the face and effective formation of the initial free volume. The minimal deviation of the actual contour from the design geometry implies a lower likelihood of local overstressed zones that are most susceptible to rockfalls and slabbing. Accordingly, the optimized drilling and blasting (D&B) parameters promote a more uniform distribution of impulsive loading and stable underground mine workings geometry, which is critical for improving operational stability and reducing the scope and cost of subsequent auxiliary operations.

Figure 14-b presents a comparative assessment of roof geometry along a reference section by juxtaposing two contour variants. Contour I corresponds to the proposed scheme (rhombic cut, Charge Design 1), whereas Contour II reflects the outcome of the baseline BP used in routine practice at the Akzhal deposit. The comparison indicates that Contour I reproduces the design geometry more accurately and exhibits smaller deviations, which implies a more controlled breakage mechanism and reduced blast-induced disturbance of the near-contour rock mass. In contrast, Contour II is characterized by local overbreak beyond the design boundary and pronounced geometric distortions, indicating less coordinated redistribution of the blast-induced stress field under the conventional scheme and a higher likelihood of slabbing and increased damage in the near-contour zone.

Therefore, the rhombic borehole layout combined with the optimized Charge Design 1 improves contour quality, reduces the risk of roof and sidewall deformation, and increases the technological reliability of the drivage process. The geometric contour assessment confirms reduced bootleg formation and limited blast-induced cracking in the near-contour zone when the optimized D&B scheme is applied.

6.7. Relationship between Borehole Utilization Factor, Specific Explosive Consumption, and Actual Advance Length

Figure 15 illustrates the relationship between the borehole utilization factor (BUF), specific explosive consumption, and the actual advance per cycle. Analysis of the experimental results (Trials I-VI) indicates that the best drivage-cycle performance is achieved in Trial I: BUF = 0.98 at the lowest specific explosive consumption (2.83 kg/m³) and the maximum actual advance over two cycles (approximately 5.88 m, based on Figure 15 and the pilot-scale industrial measurements). This operating regime corresponds to the most rational distribution of blast energy, ensuring high utilization of the borehole layout and minimizing energy losses.

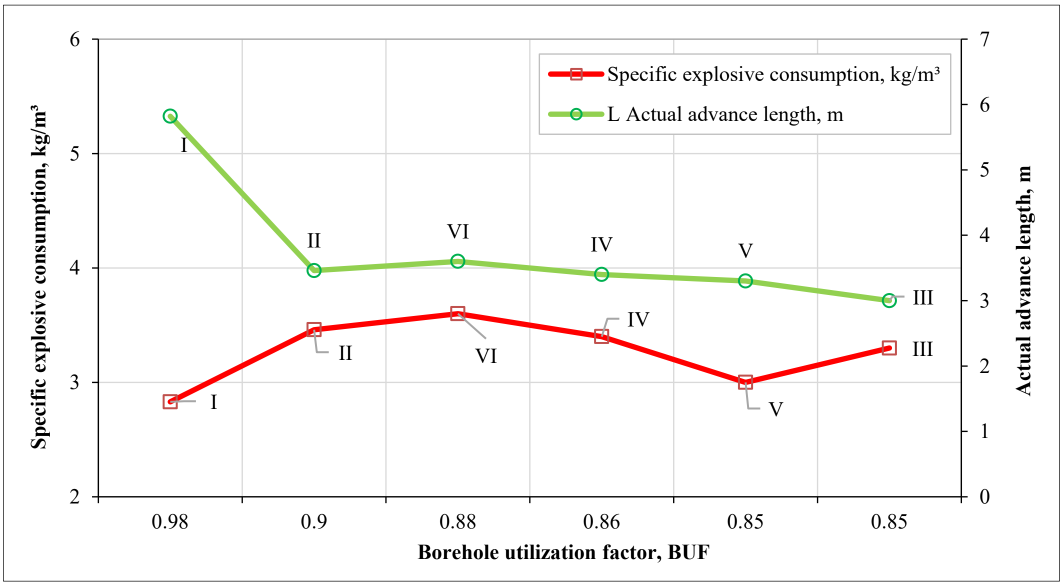


Figure 15. Relationship between the borehole utilization factor (BUF), specific explosive consumption, and actual advance length based on pilot-scale industrial trials

Moving from Trials II to VI, a consistent decline in performance is observed: despite an increase in specific explosive consumption to 3.0-3.6 kg/m³, the actual advance decreases (to approximately 5.1 m), while BUF drops to 0.85-0.90, indicating reduced useful blast work and increased inefficient energy losses. Even in Trial VI, where a partial increase in BUF is observed, the achieved advance does not reach the level of Trial I, confirming the existence of an optimal range of specific explosive consumption and the need to standardize it for the study object. The identified relationships show that increasing specific explosive consumption beyond the rational level does not improve D&B performance and may be accompanied by deterioration of contour quality of underground mine workings, increased disturbance of the near-contour rock mass, and reduced overall drivage-cycle productivity. Therefore, the operating regime of Trial I can be considered a technological benchmark, providing an optimal balance between resource inputs and achieved performance indicators.

The maximum drivage productivity is attained at BUF ≈ 0.98 and a specific explosive consumption of 2.83 kg/m³, which yields the greatest actual advance (≈5.88 m over two cycles). Increasing specific explosive consumption to 3.0-3.6 kg/m³ does not lead to higher advance and is accompanied by a reduction in BUF, highlighting the need to optimize D&B parameters based on energy adequacy and controllability of rock breakage.

Summary: Pilot-scale industrial trials confirmed that D&B performance is governed not only by specific explosive consumption, but also by rational selection of the cut type, charge design, and initiation parameters that ensure stable formation of the initial free volume and complete utilization of the borehole layout. Under the conditions of the +285 m level at the Akzhal deposit, the highest performance was achieved in Trial I at a specific explosive consumption of 2.83 kg/m³, where the maximum BUF = 0.98 and the greatest advance (5.88 m over two cycles) were recorded.

It was established that increasing specific explosive consumption to 3.3-3.6 kg/m³ does not improve performance and is accompanied by decreased BUF and reduced advance. This confirms that D&B parameters should be optimized according to the criterion of energy adequacy rather than by maximizing charge mass. Application of short-delay blasting (SDB) with rational delay intervals of 20-30 ms between cut and production boreholes and 10-20 ms between production and contour boreholes increase controllability of breakage, improves fragmentation, and contributes to higher BUF. Locally produced Granulite A6 is technologically and economically justified for the underground conditions of the Akzhal deposit. Selection of the application mode (cartridged or bulk/granulated) should be determined by hydrogeological conditions at the face and charging stability requirements. Fragmentation analysis showed that, within the specific explosive consumption range of 2.83-3.60 kg/m³, the mean fragment size increased from 60 to 135 mm, indicating that increasing explosive consumption is ineffective as a means of improving fragmentation under the considered conditions.

Overall, the rhombic cut with Charge Design 1 was identified as the most rational option based on the combined criteria “BUF-fragmentation-contour quality”. This scheme minimizes the share of oversized fragments, ensures stable advance rates, improves contour quality of underground mine workings, and reduces adverse effects (bootleg formation, blast-induced cracking, and the risk of roof and sidewall deformation). These outcomes create prerequisites for increasing drivage productivity while simultaneously improving underground operational safety.

The obtained results underpin the study methodology by providing a scientific basis for adopting the rhombic cut (Design 1) as the rational geometric scheme for driving the delivery drift. The established parameters form the basis for developing an optimal blasting pattern (BP) in which cut geometry, initiation sequence, and specific explosive consumption are integrated into a single, energetically and geomechanically consistent system. The proposed relationships and criteria can be applied as an engineering tool for BP design under comparable mining and geological conditions and for selecting rational D&B parameters.

6.8. Comparison with Previous Pilot-Scale Industrial Trials

To strengthen the validity of the obtained results, the performance indicators derived in the present study were compared with data from earlier pilot-scale industrial trials conducted at the Akzhal mine (+425 m level, delivery drift, S = 15 m²). In the earlier trials, Granulite AS-8 was used and vertical wedge and slot cut schemes were evaluated. Because baseline conditions in terms of rock class and working geometry are broadly comparable, this comparison enables a consistent assessment of the influence of cut geometry, blasting pattern (BP) configuration, and explosive application mode on D&B performance (Table 5).

Table 5. Comparative techno-economic analysis and estimated D&B cycle cost as a function of cut type and charge design

Data source	Level	Variant	Cut type / charge design	Explosive used	BUF (η)	Face advance per cycle (l_y), m	Overall assessment
Results of previous pilot-scale industrial trials							
Results of previous pilot-scale industrial trials	+425 m	PSI-1	Vertical wedge cut	Granulite AS-8	0,81	2,59	Baseline performance level
Results of previous pilot-scale industrial trials	+425 m	PSI-2	Vertical wedge cut (+ 3 compensation boreholes)	Granulite AS-8	0,84	2,68	Improvement due to an additional free surface
Results of previous pilot-scale industrial trials	+425 m	PSI-3	Slot cut (+ 2 compensation boreholes)	Granulite AS-8	0,74	2,35	Reduced performance
Results of the present study							
Present study	+285 м	I	Rhombic cut, Design 1	Bulk (granulated) Granulite A6	0,98	2,94	Optimal (recommended)
Present study	+285 м	II	Rhombic cut, Design 2	Cartridged Granulite A6	0,90	2,7	Rational (application-dependent)
Present study	+285 м	III	Slot cut, Design 3	Bulk (granulated) Granulite A6	0,85	2,55	Efficiency is limited
Present study	+285 м	IV	Slot cut, Design	Cartridged Granulite A6	0,86	2,58	Economically inferior to the rhombic scheme
Present study	+285 м	V	Vertical wedge cut, Design 5	Bulk (granulated) Granulite A6	0,85	2,55	Least rational in terms of cost
Present study	+285 м	VI	Vertical wedge cut, Design 6	Cartridged Granulite A6	0,88	2,64	High cost with moderate effect

The earlier trial results indicate that the vertical wedge scheme without compensation boreholes achieved BUF = 0.81 with an advance of $l_γ$ = 2.59 m per cycle. Introducing three compensation boreholes increased BUF to 0.84 and the advance to $l_γ$ = 2.68 m, confirming the role of an artificially created free surface in improving borehole-layout utilization. In contrast, the slot scheme-even with compensation boreholes- showed a decrease in BUF to 0.74 and a reduction in advance to $l_γ$ = 2.35 m, indicating less stable cut opening and a less effective transfer of blast energy to rock breakage.

In the present study (+285 m level), adopting a rhombic cut combined with an optimized charge design provided markedly higher efficiency: the maximum BUF reached 0.98 (Variant I), while the estimated cost of the D&B cycle was the lowest among the evaluated schemes (USD 489.38). These findings indicate that the increased drilling effort and higher explosive consumption associated with the slot and vertical wedge schemes are not accompanied by proportional gains in BUF or advance under the conditions of the studied level.

Comparison with the earlier trials (Table 5) shows that optimizing cut-zone geometry and charge design provides a technologically meaningful improvement in key D&B indicators. Specifically, moving from the vertical wedge cut with compensation boreholes (BUF = 0.84) to the rhombic cut (Design 1) increased BUF to 0.98 (+16.7%). In parallel, the advance per cycle increased from 2.68 to 2.94 m (+9.7%), while the estimated cycle cost decreased from USD 538.85 to USD 489.38 (−USD 49.47, −9.18%).

These results confirm that improvements in D&B efficiency are achieved primarily through rational creation of the initial free volume and consistency between the cut scheme and BP configuration, rather than by increasing specific explosive consumption. The rhombic cut (Design 1) provides the most favorable combination of BUF, fragmentation, contour quality of underground mine workings, and overall performance; therefore, it is recommended as the baseline solution for developing the BP for the delivery drift at the +285 m level under comparable mining and geological conditions.

7. Techno-Economic Efficiency of Drilling and Blasting Operations

The techno-economic efficiency of drilling and blasting (D&B) was evaluated for driving horizontal underground mine workings with a cross-sectional area of S = 15 m² at the Akzhal deposit. A comparative analysis was performed for three cut types (rhombic, slot, and vertical wedge) combined with two charge-design variants that differ in the explosive application mode: bulk (granulated) Granulite A6 and cartridged Granulite A6. The summary indicators for each variant are presented in Table 6, and a comparative visualization of the total D&B cycle cost is provided in Figure 16.

Table 6. Comparative techno-economic analysis and estimated D&B cycle cost as a function of cut type and charge design

Cost item	Cycle costs, *USD*					
	Type of cut - rhombic		Type of cut - slotted		Type of cut - vertical wedge	
	Design I (1)	Design II (2)	Design III (3)	Design IV (4)	Design V (5)	Design VI (5)
Borehole drilling, $/linear meter	226,77	226,77	238,85	239,60	244,65	251,72
Explosive consumption, $/kg: Granulite A6	26,22	29,93	31,40	32,91	33,98	35,57
Ammonite 6ZV	10,0	10,0	11,20	11,20	12,0	12,0
Non-electric initiation system (NeIS), $/unit	39,0	39,0	40,15	40,15	40,53	40,53
Detonating cord, $/m	2,65	2,65	2,75	2,75	2,80	2,85
Electric detonators, $/unit	1,31	1,31	1,40	1,40	1,45	1,45
Overhead costs 6% of total expenses, $	183,43	185,66	186,38	186,40	193,27	194,73
Total:	**489,38**	**495,32**	**512,13**	**514,41**	**528,68**	**538,85**

Note: The USD-to-KZT exchange rate was taken as the official rate as of 15 September 2025 (1 USD = 538 KZT).

At the technological design stage, detailed D&B costing is often challenging because of variability in mining and geological conditions, differences in equipment fleets, and the lack of a unified costing methodology. In this study, the economic assessment was based on actual production data and included a cost breakdown across the main categories: drilling, explosives, non-electric initiation system (NeIS) components, and auxiliary materials/overheads.

According to the calculations, the total cost of one D&B cycle ranges from USD 489.38 to USD 538.85, depending on cut type and charge design. The absolute difference is USD 49.47, which corresponds to approximately 10.1% relative to the minimum value (Table 6).

Regarding the concise costing methodology, the estimated cost for each variant was determined using a unified approach based on the drilling and charging parameters obtained from 3D modelling and pilot-scale industrial trials. For each scheme, drilling volume, charge mass, consumption of initiation components, and auxiliary materials were quantified; costs were then aggregated across the principal categories (drilling, explosives, NeIS/detonating cord/electric detonators, and overheads). Comparability among variants was ensured by identical mining-technical conditions ($S = 15\ m^2$, borehole length $l = 3.0$ m, borehole diameter $d = 45$ mm) and a single currency conversion rate. This calculation algorithm minimized external influences and isolated the contribution of the key design and technological parameters (cut type and charge design) to the variation in total cycle cost.

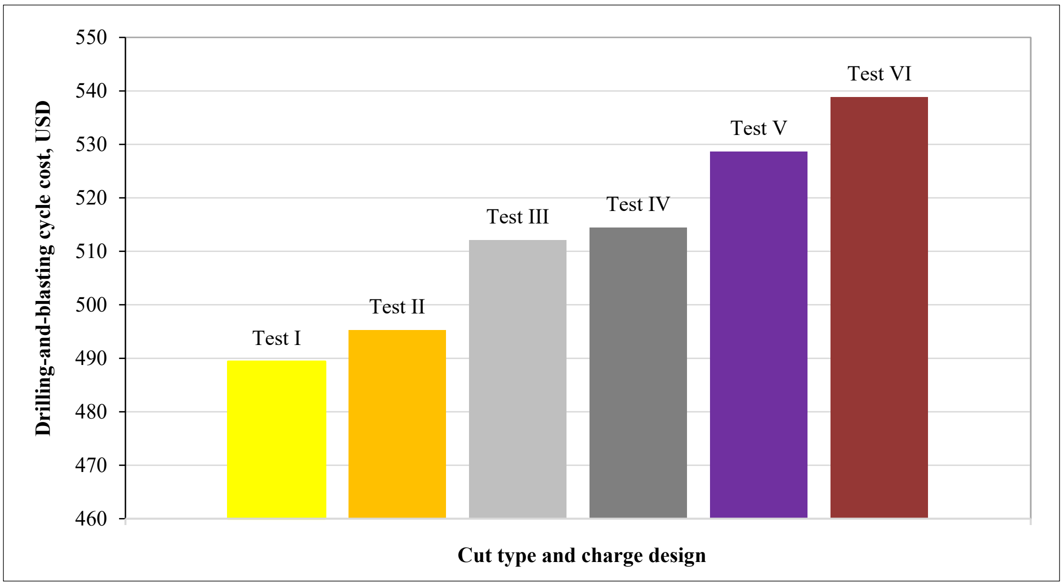


Figure 16. Comparative assessment of the estimated drilling-and-blasting (D&B) cycle cost for different cut types and charge designs

7.1. Influence of Cut Type and Charge Design on the Estimated Cycle Cost and Interpretation of Results

The minimum estimated cost of the D&B cycle was obtained for the rhombic cut (Design 1) and amounted to USD 489.38, which is the lowest value among all considered variants. The economic advantage of this option is attributed to rational formation of the initial free volume, a higher BUF, and reduced overall consumption of materials while maintaining the required technological performance of the drivage cycle.

Under water-bearing conditions, the use of bulk (granulated) explosives may be constrained due to reduced detonation stability and lower breakage effectiveness. In such cases, the use of cartridged Granulite A6 (rhombic cut, Design 2) is more appropriate, with the cycle cost increasing to USD 495.32. The difference between Designs 1 and 2 is only USD 5.94 (≈1.2%), confirming the robustness of the rhombic scheme with respect to explosive application mode and demonstrating its technological reliability. For the slot cut, the cycle cost increases to USD 512.13-514.41, which is USD 16.81-25.03 (≈3.4-5.1%) higher than the rhombic scheme. This increase is primarily driven by higher drilling costs and increased explosive consumption, reflecting less efficient utilization of blast energy and higher material intensity.

The highest costs are associated with the vertical wedge cut, where the cycle cost reaches USD 528.68 (Design 5) and USD 538.85 (Design 6). The cost premium relative to the rhombic scheme is USD 39.30-49.47 (≈8.0-10.1%), indicating the lowest economic feasibility of this option under the conditions of the considered level (Figure 16).

The comparative analysis (Figure 16) indicates that cut type and charge design directly affect total material costs because they govern the blasting pattern (BP) configuration, drilling volume, blast-energy distribution, and specific explosive consumption. The rhombic scheme promotes efficient formation of the initial free volume and a more complete transfer of blast energy to rock breakage, which reduces total material consumption and makes this option economically preferable to the slot and vertical wedge schemes.

Accordingly, under the conditions of the Akzhal deposit, the rhombic cut combined with bulk (granulated) Granulite A6 can be considered the optimal option for dry faces, whereas in water-bearing zones it is more rational to apply cartridged Granulite A6 to ensure charging stability and reproducibility of the blasting effect.

An additional factor improving both the efficiency and safety of D&B is the use of short-delay blasting (SDB). The application of rational delay intervals increases the controllability of rock breakage through coordinated stress-wave interaction and reduces seismic effects on adjacent underground mine workings. As a result, the technological stability of the drivage process improves, and the likelihood of secondary deformation in the near-contour zone decreases (Table 7).

Table 7. Integrated comparative assessment of the technological and economic efficiency of D&B variants (Trials I-VI)

Scheme (trial)	Cut type (design)	Cycle cost, USD	BUF	Face advance over two cycles, m	D&B efficiency assessment
I	Rhombic, D1	489,38	0.98	5,88	Optimal (recommended) option.
II	Rhombic, D2	495,32	0.90	5,4	Rational, particularly under water-bearing conditions.
III	Slot, D3	512,13	0.85	5,1	Lower efficiency: higher cost without a proportional performance gain.
IV	Slot, D4	514,41	0.86	5,16	Limited efficiency; economically inferior to the rhombic scheme
V	Vertical wedge, D5	528,68	0.85	5,1	Least rational based on the overall set of indicators.
VI	Vertical wedge, D5	538,85	0.88	5,28	Least rational based on the overall set of indicators.

Note: In all trials, the drilled borehole length at the face was taken as lb = 3 m; therefore, for cost calculations, the total borehole length over two cycles was assumed to be 6 m.

Comparison of the six experimental schemes (Table 7) shows that the lowest D&B cycle cost is achieved for the rhombic cut (Design 1), which simultaneously provides the highest BUF and the greatest advance (per cycle or over two cycles). When transitioning to the slot and vertical wedge cuts, costs increase by approximately 4-10%, whereas advance decreases and borehole utilization deteriorates, indicating reduced energy and technological efficiency. Therefore, for the +285 m level at the Akzhal deposit, the rational option is the rhombic scheme combined with Granulite A6; the choice of explosive application mode (bulk/granulated versus cartridged) should be governed by hydrogeological conditions at the face.

Summary:

- It was established that, for S = 15 m², the total D&B cycle cost varies within USD 489.38-538.85, and the difference between the minimum and maximum variants is approximately 10%, confirming a significant effect of cut type and charge design on D&B economic performance.
- The most economically efficient option is the rhombic cut (Design 1), which yields the minimum cost while maintaining high technological performance indicators.
- Under water-bearing conditions, the use of cartridged Granulite A6 is preferable; the associated cost increase relative to the bulk/granulated mode does not exceed ≈1.2%, demonstrating the robustness of the rhombic scheme with respect to the explosive application mode.
- The slot and vertical wedge cuts are characterized by a 4-10% cost increase, primarily due to higher drilling volumes and less efficient utilization of blast energy.
- The use of short-delay blasting (SDB) is technologically justified because it improves controllability of rock breakage, reduces adverse impacts on the near-contour rock mass, and increases overall efficiency of the drivage cycle.

The techno-economic results obtained substantiate rational D&B parameters for the Akzhal deposit and demonstrate that cost minimization is achieved not by simply reducing charge mass, but by ensuring consistency among cut scheme, charge design, and initiation parameters. In this study, six alternative D&B schemes were evaluated under comparable conditions using an integrated criterion of "cycle cost-BUF-actual advance-contour quality." It was confirmed that the rhombic scheme (Design 1) simultaneously provides the minimum estimated cost and the best technological performance, indicating the most effective conversion of blast energy into useful rock breakage. The established relationships form a methodological basis for engineering selection of D&B schemes according to energy adequacy and economic feasibility and can be applied in developing blasting patterns (BP) for underground mine workings of comparable complexity.

8. Conclusions

8.1. Research Novelty

The scientific novelty of this study lies in the comprehensive substantiation of rational drilling and blasting (D&B) parameters for the Akzhal deposit through the integrated use of geological-geomechanical analysis, three-dimensional digital modelling in Micromine, empirical-analytical calculations, and pilot-scale industrial trials. For the first time for the delivery drift at the +285 m level, six alternative D&B schemes were evaluated on a comparable basis, differing in cut type (rhombic, vertical slot, and wedge) and charge design (cartridged and bulk/granulated Granulite A6). Their effects were quantified in terms of the borehole utilization factor (BUF), actual advance, fragmentation of the blasted rock mass, and contour quality of underground mine workings.

It was established that rock-breakage efficiency is controlled not only by specific explosive consumption but also by cut-zone geometry, the spatial structure of the blasting pattern (BP), and the initiation regime. In combination, these factors ensure stable formation of the initial free volume and reduce blast-induced cracking in the near-contour zone. The advantage of the rhombic scheme (Design 1) with Granulite A6 was confirmed experimentally, enabling BUF = 0.98 and a reduction in the estimated D&B cycle cost.

The developed BP solutions and parameter sets, grounded in the identified regularities of controlled fragmentation, can be applied not only to drivage of underground mine workings but also, with appropriate adaptation, to production blasting in ore blocks. Achieving the required fragmentation at the blasting stage creates favorable conditions for subsequent loading and haulage, crushing, and beneficiation, strengthening the technological and economic linkage between mining and mineral processing. Overall, the results contribute to both the scientific basis and the applied practice of D&B in underground mining.

8.2. Practical Significance

The practical significance of the study is associated with the direct implementability of the proposed, scientifically substantiated recommendations for designing and executing D&B operations in underground mine workings under complex geological conditions. It was shown that applying the optimized D&B parameters-including a rational selection of explosive type and the geometry and layout of cut holes-can increase BUF from 0.85 to 0.98, reduce direct material costs per D&B cycle from USD 538.85 to USD 489.38, and improve excavation contour quality. The use of locally produced Granulite A6 enhances both technological and economic performance of drivage operations and improves occupational safety conditions. The results can be applied at the Akzhal deposit and adapted for implementation at other mines with comparable geological and mining-technical settings.

8.3. Research Limitations

The immediate applicability of the reported results is largely constrained by the specific geological and mining-technical conditions of the Akzhal deposit, under which the experimental and pilot-scale investigations were conducted. Additional limitations relate to the characteristics of the explosives used, the initiation systems, and the specific design configurations of the cut-hole layout. When transferring the proposed methodology to other deposits, it is necessary to account for variations in lithology, fracturing intensity, rock mechanical properties, hydrogeological conditions, and the available mining equipment. Such variations may require recalibration of the calculation inputs and iterative D&B design parameters to achieve comparable efficiency and reproducibility.

8.4. Future Research Directions

Future work will focus on extending the applicability of the developed methodology to other areas of the deposit that differ in geological, geomechanical, and hydrogeological conditions. Further research is planned to evaluate new and modified explosive formulations and to quantify their combined effects on technological, economic, and geomechanical D&B performance indicators. Another priority direction is the advancement of digital geomechanical modelling of the rock mass, explicitly accounting for structural heterogeneity and in situ stress state. In addition, the integration of machine learning and intelligent data analytics is envisioned to improve the prediction accuracy of optimal D&B parameters, enable adaptive control of blasting operations, and enhance the efficiency of planning and the operational stability of drivage cycles.

9. Declarations

9.1. Author Contributions

Conceptualization, T.A. and D.S.; methodology, T.A. and D.S.; software, R.Z. and D.S.; validation, T.A., D.S., and R.Z.; formal analysis, T.A. and D.S.; investigation, T.A. and R.Z.; resources, T.A. and D.S.; data curation, T.A.; writing—original draft preparation, T.A. and D.S.; writing—review and editing, T.A., R.Z., D.S., and A.O.; visualization, T.A. and A.O., supervision, T.A. and D.S.; project administration, T.A.; funding acquisition, T.A. All authors have read and agreed to the published version of the manuscript.

9.2. Data Availability Statement

The data presented in this study are available in the article.

9.3. Funding

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9.4. Conflicts of Interest

The authors declare no conflict of interest.

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