SATBAYEV
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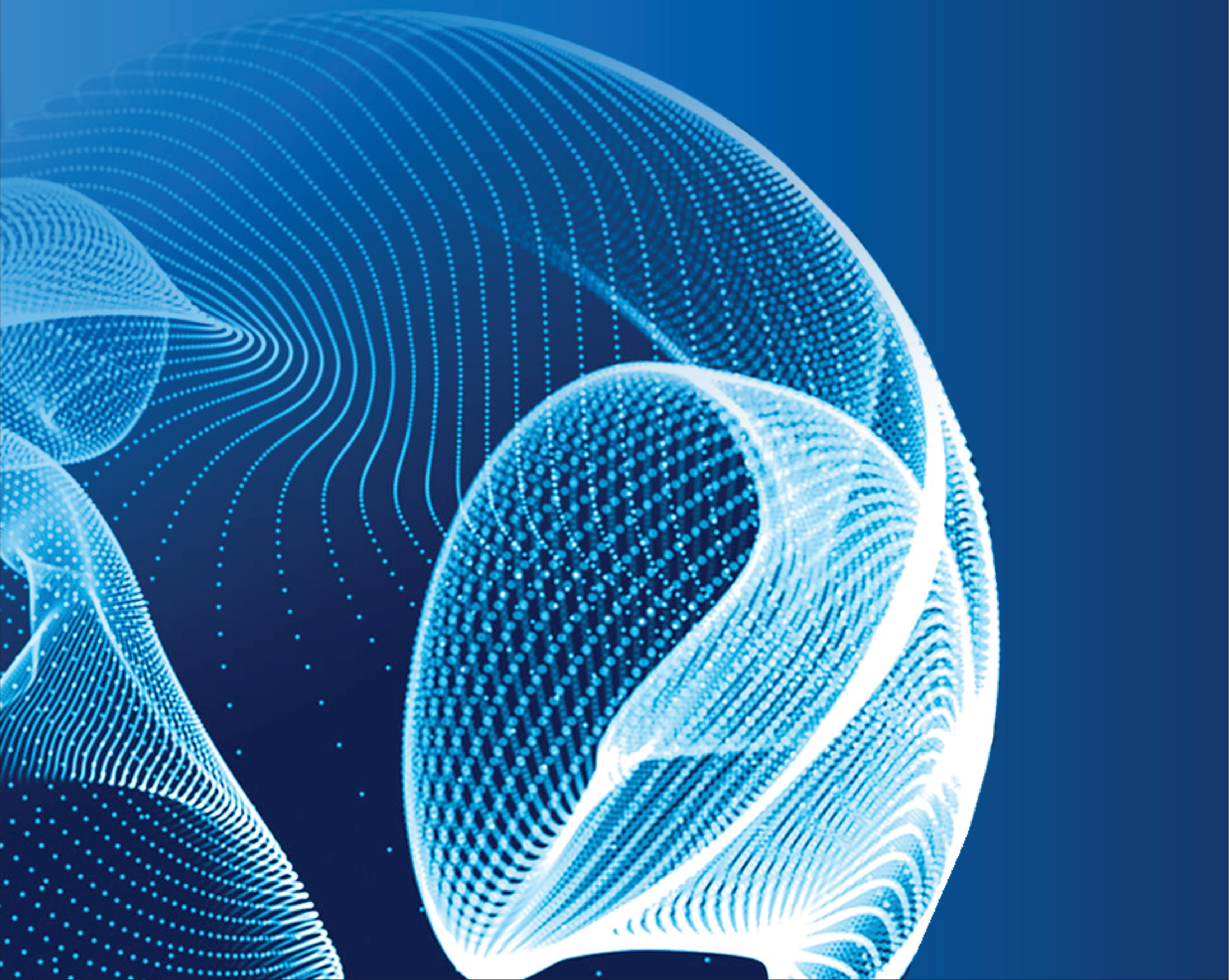
ISSN 2959-2348 (Online)

# ENGINEERING JOURNAL of Satbayev University

Volume 147 (Issue 5)
October 2025



**eISSN 2959-2348**

**eISSN 2959-2348**

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SATBAYEV UNIVERSITY

Engineering Journal of Satbayev University

*Volume 147 (2025), Issue 5, 1-9*

https://doi.org/10.51301/ejsu.2025.i5.01

# Determination of the optimal technological regime and parameters of the cyanidation process of the flotation concentrate obtained after the enrichment of resistant gold-bearing ores

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**Abstract.** As a result of comprehensive studies of the material composition of the flotation concentrate and the forms of gold presence, it was established that the chemical composition of the concentrate is 55.17% represented by lithophile components with the mass fraction of 55.17%. The main ones are silica and alumina with mass fractions of 36.4% and 8.26%, respectively. Ore mineralization of the flotation concentrate is represented by pyrite with the mass fraction of 40.7%. It has been established that gold in the flotation concentrate sample is present in native form. The bulk of gold grains is represented by particles with a size of 10-38 microns - 82.89%. About 63% of gold is in free form. The proportion of closed grains is 9.23%. The main mineral in the flotation concentrate sample, associated with gold, is pyrite – 25%. The portion of gold associated with quartz is 3.07%. For industrial operation, sorption type of cyanidation of flotation concentrates with a consumption of Norit RO 3520 activated carbon in an amount of 10% of the volume of the liquid phase is recommended as the optimal mode. The parameters for sorption cyanidation of flotation concentrate have been established: flotation concentrate size – P80 10 microns; sodium cyanide concentration – 0.1% (sodium cyanide consumption – 2.3 kg/t); pH – 10.5; pulp density – 40% (solid); duration of the process is 24 hours. With the established parameters, relatively high, at least 86%, extraction of gold into the solution was achieved.

***Keywords:*** *gold, flotation concentrate, cyanidation, sodium cyanide, concentration, activated carbon.*

Received: 25 July 2025
Accepted: 16 October 2025
Available online: 31 October 2025

## 1. Introduction

All over the world, the gold content in ore deposits is decreasing, and their mineralogy is becoming more complex and complex. Despite this, the process of producing gold by cyanidation remains dominant for many years due to the selectivity of cyanide to gold, the simplicity and efficiency of the process [1-4]. Today, the industry is adjusting its methods for extracting gold from ores and its enrichment products (concentrates, tailings), using more efficient processes and technologies.

For effective cyanidation leaching of refractory and/or complex ores, pre-treatment prior to leaching is necessary. When sulfides are present, they often prevent the cyanide from making physical contact with the gold particles, thereby preventing the gold from dissolving. Conversely, for ores containing carbonaceous matter, dissolved gold in the leach solution can be re-adsorbed to carbonaceous species, a process known as preg-robbing, which also interferes with gold recovery. To solve these problems, pre-treatments are used, which increase costs and technical requirements for the process flow [3, 4].

The characteristics of the ore dictate the choice of the correct cyanidation process design. Gold ores within a geological province/region (even the same deposit) may have different mineral components, different mineral concentrations and degrees of alteration or oxidation.

Gold in ores and concentrates is often found in combination with non-ferrous metals (Cu, Pb, Ag). Applying cyanide to these types of ores can present some difficulty as the variety of minerals present in these ores can make the application of cyanide unfavorable. This type of ore includes gold deposits in Central Kazakhstan with a content of 1-1.5 g/t gold. The complex nature of these ores necessitated pre-treatment operations using gravity and cyanidation flotation. Production is characterized by low gold recovery. In addition, the choice of the optimal concentration of sodium cyanide, which is used as a leaching reagent, requires clarification.

The reserves of rich and easy-to-process gold ores in Kazakhstan are being depleted, and ores belonging to the category of refractory ores are being involved in processing. The persistence of ores may be due to the fine dissemination of gold in carrier minerals, chemical depression of gold at the


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Engineering Journal of Satbayev University. eISSN 2959-2348. Published by Satbayev University

stage of metallurgical processing, or the presence of organic compounds that are active towards dissolved gold.

An important problem in the processing of refractory gold ores is the extraction of ultrafine gold, which is encapsulated in sulfide minerals and is difficult to extract by traditional methods. Minor concentrations of «invisible» gold, as established in [5], usually contain pyrrhotite, chalcopyrite, bornite and galena. Work [6] shows that finely dispersed gold can be associated not only with sulfide, but also with rock-forming minerals. The most common carrier minerals of «invisible» gold are arsenopyrite and arsenic pyrite. The gold content of arsenopyrite depends on its morphological variety [7].

In work [8], based on studies of the gold content of arsenopyrite in gold-sulfide deposits in Kazakhstan, it is shown that due to the high sorption activity of the carbonaceous substance, metallurgical processing of flotation sulfide concentrate is impossible when the Au/Corg ratio is <8 g/kg. If the ratio is not met, additional gravitational enrichment of concentrates is required.

Due to the unique structure of the carbonaceous substance, the question related to the particular component and which one has increased sorption activity towards dissolved gold remains poorly understood. The high sorption activity of carbonaceous matter has a significant impact on the technology of processing ores and concentrates using flotation, pyro- and hydrometallurgical methods [9, 10].

A promising method for extracting ultrafine gold from refractory ores due to its enlargement may be the processing of feedstock and enrichment products using various physical and energetic methods of influence [11-14]. However, these methods have not been widely used in practice.

For gold ores of Central Kazakhstan, cyanidation of current flotation concentrates containing refractory minerals and carbonaceous substances, the presence of which causes a «slowdown» of the kinetic patterns of leaching and a decrease in gold recovery, has a particular relevance from a technological and environmental point of view. Processing of these materials by cyanidation entails high consumption of cyanide and increased costs [15, 16].

The importance of studying this issue is enhanced by the fact that in the scientific literature there is only a limited number of information devoted to the study of gold extraction based on the specific properties of the ore. In addition, in publically known works it was not possible to achieve high gold extraction from such ores.

In this article, based on a comprehensive study of the material composition of flotation concentrate obtained from the beneficiation of ore in Central Kazakhstan, the influence of the concentrate size, sodium cyanide concentration and leaching duration on gold recovery in the cyanidation process is investigated.

## 2. Materials and methods

For laboratory hydrometallurgical studies, a flotation concentrate with a gold content of 17.9 g/t was used, obtained after gravity-flotation enrichment of ore from a deposit in Central Kazakhstan.

The material composition of flotation concentrate was studied using optical and electron microscopic mineralogical studies. Chemical, assay, X-ray spectral and sieve analyzes were performed.

The chemical composition of the flotation concentrate was determined using optical emission (ICP90, ICP40 analyses) and atomic absorption methods of analysis. To increase the reliability of the gold content results, direct assay analysis was carried out on four parallel samples.

Determination of the mineral composition of flotation concentrate samples was carried out using a complex of various studies, including: diffractometric, quantitative mineralogical analyses, optical study of heavy fractions and mapping of products on a Quanta FEG-650 F electron microscope as part of the automated mineralogical complex Qemscan.

The study of gold forms for the presence of large gold particles in the flotation concentrate was carried out using an Olympus SZX-7 stereomicroscope. To conduct research, a sample of flotation concentrate was divided into size classes using the sieve analysis method. From each size class, ½ part was submitted for assay analysis; briquette sections were made from the second part. The study of briquette sections was carried out using an automated mineralogical complex Qemscan based on an electron scanning microscope. The parameters of the complex were configured to search for valuable minerals (SMS – Specific Mineral Search), mapping was performed in automatic mode.

Laboratory hydrometallurgical studies were carried out using cyanide leaching in bottle-type agitators. External view of laboratory stands with agitators shown on Figure 1.

***Figure 1. Laboratory stand for conducting study on agitated leaching***

To assess the level of sorption activity of the flotation concentrate sample, tests were performed to determine the PRI index (preg-robbing index) analytical method. Agitated cyanidation of the flotation concentrate was carried out on material subjected to fine grinding in a ball mill to a particle size of P80 71 and 45 microns, as well as ultrafine grinding to a particle size of P80 30, 20, 10 and 7 microns.

Ultrafine grinding was carried out in a Netzsch IsaMill M4B laboratory mill, the general view of which is shown in Figure 2. In the test the feed charge with beads size 2.8 mm was used: 60% 2.5-2.8 mm; 30% 1.8-2.0 mm; 10% 1.4-1.6 mm; solid content of the feed was ~43%, quantity of grinding cycles – 8.

Grinding was carried out in the following mode: mass of crushed material – 200 g; loading of grinding medium – 72.5% of the mill volume; pulp density during grinding – 55.25% solid; impeller rotation speed – 1300 rpm.

*Figure 2. General view of Netzsch IsaMill M4 bead mills*

Estimated specific energy consumption when grinding flotation concentrate from an initial size of 80% 150 microns to a size of 80% 10.1 µm was 81.5 kWh/t.

To determine the optimal leaching parameters, tests were carried out with different sizes of grinded concentrate under cyanidation conditions without loading sorbent and sorption type leaching, with different concentrations of sodium cyanide, and process duration with the purpose of studying the dynamics of leaching.

During the tests, NaCN concentration was monitored as well as pulp pH. The cyanidation products were subjected to atomic absorption (solution) and assay (cake, coal) types of analysis.

## 3. Results and discussion

### 3.1. Study of the material composition of flotation concentrate

The chemical composition of the flotation concentrate, determined by optical emission (ICP90, ICP40 analyses) and atomic absorption analyzes is presented in Table 1.

*Table 1. Chemical composition of flotation concentrate*

| Element | Mass fraction, % | Element | Mass fraction, % |
|---|---|---|---|
| $SiO_2$ | 36.40 | $S_{total}$ | 22.90 |
| $Al_2O_3$ | 8.26 | $S_{sulf}$ | <0.25 |
| CaO | 1.50 | Pb | 0.015 |
| $K_2O$ | 1.12 | Zn | 0.048 |
| $Na_2O$ | 3.00 | Cu | 0.059 |
| MgO | 1.28 | As | 0.087 |
| MnO | 0.06 | Sb | 0.0066 |
| $P_2O_5$ | 0.07 | Ba | 0.022 |
| $TiO_2$ | 0.18 | Co | 0.012 |
| $C_{org}$ | 0.26 | Cr | 0.0017 |
| $C_{total}$ | 0.83 | Ni | 0.021 |
| $CO_2$ | 3.04 | Sr | 0.014 |
| $Fe_{total}$ | 20.60 | Ag, g/t | 2.90 |
| $Fe_{sulf}$ | 18.90 | Au, g/t | 17.30 |

To increase the reliability of the result on gold content, its determination was carried out on four parallel samples using direct assay analysis. The established average gold content (17.47 g/t) based on the gold content in four samples, g/t: 16.8; 17.2; 18.2 and 17.7 showed good correlation with the result given in Table 1.

The chemical composition of the flotation concentrate is represented mainly by lithophile components with the mass fraction of 55.17%. The main ones are silica and alumina with mass fractions of 36.4% and 8.26%, respectively.

The total proportion of oxides of alkali and alkaline earth metals, with a significant predominance of sodium oxide 3.0%, is ~6.90%.

### 3.2. Mineral composition of flotation concentrate

The mineral composition of the flotation concentrate is shown in Table 2.

*Table 2. Mineral composition of flotation concentrate*

| Mineral, group of minerals | Mass fraction in sample, % |
|---|---|
| Rock-forming minerals | |
| Plagioclases | 26.0 |
| Quartz | 18.0 |
| Mica (sericite, chlorite) | 9.0 |
| Carbonates | 5.0 |
| Gypsum, anhydrite, jarosite | - |
| Ore-forming minerals | |
| Pyrite | 40.7 |
| Sphalerite, galena, altaite | 0.1 |
| Faded ore, chalcopyrite | 0.3 |
| Arsenopyrite | 0.1 |
| Iron hydroxides | 0.5 |
| Accessory minerals | |
| Titanium minerals, apatite | 0.3 |
| Total | 100 |

From Table 2 it is clear that the mineral composition of the flotation concentrate samples consists of 58% rock-forming minerals. Ore minerals are represented mainly by pyrite 40.7%. Other sulfides are noted in the samples in amounts of tenths of a percent.

### 3.3. Forms of gold presence

The analysis of the forms of gold presence in the flotation concentrate sample was performed sequentially. First, using the Olympus SZX-7 stereomicroscope, an initial study was conducted to identify large, visible gold particles. Then, a more detailed study of the gold's forms was carried out using an electron microscope. The initial analysis did not reveal large visible gold particles. It was found that the bulk of the gold in the flotation concentrate, namely 99.96%, is represented as native gold. The remaining insignificant fraction (0.04%) of gold was found in the form of tellurides of various compositions. Due to the extremely small number of tellurides, the technological and mineralogical characteristics were studied only for native gold.

Figure 3 shows the granulometry of gold grains present in the flotation concentrate.

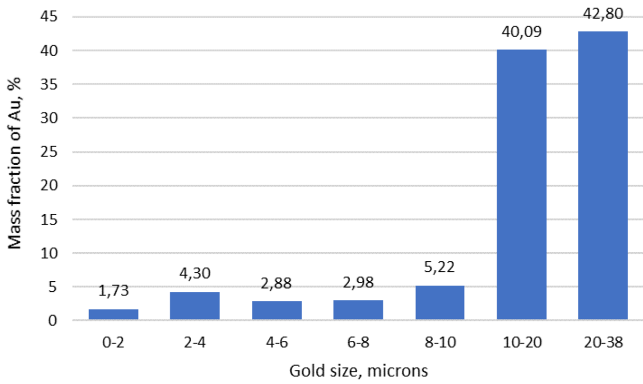


*Figure 3. Granulometric characteristics of gold*

The bulk of the valuable component consists of particles ranging in size from 10 to 38 microns, the total proportion of which reaches 82.89%. The number of small particles not exceeding 2 microns is insignificant and does not exceed 1.73%. In the size range from 2 to 10 microns, the gold content ranges from 2.88 to 5.22%. The results of the mineral associations of gold in the flotation concentrate sample, presented in Table 3, show that the background includes free gold and gold with a partially free surface. Thus, 69.15% of the grains of the valuable component have access of the solution and reagents to the grain surface.

***Table 3. Mineral associations of gold***

| Mineral, group of minerals | Mass fraction of gold, % |
|---|---|
| Background | 69.15 |
| Quartz | 3.07 |
| Mica | 0.11 |
| Chlorite | 0.60 |
| Feldspars | 0.05 |
| Tellurides | 0.86 |
| Pyrite | 25.03 |
| Chalcopyrite | 0.35 |
| Galena | 0.004 |
| Sphalerite | 0.11 |
| Arsenopyrite | 0.03 |
| Faded ore | 0.62 |
| Accessory minerals | 0.03 |
| Total: | 100 |

The main mineral associated with gold in the flotation concentrate sample is pyrite. The ratio of such intergrowth's accounts for 25.03% of the metal. In contact with quartz, the proportion of gold is 3.07%. In intergrowth with other minerals, the share of gold is tenths and hundredths of a percent. The established patterns are clearly illustrated by fragments of the map of gold mineral particles in the flotation concentrate sample and associations of gold with various minerals, shown in Figure 4.

Thus, the results of studies of the material composition of the flotation concentrate show that its chemical composition is represented by lithophilic components with a mass fraction of 55.17%. The mass fraction of silica and alumina in them is 36.4% and 8.26%, respectively. The total proportion of alkali and alkaline earth metal oxides, with a significant predominance of sodium oxide 3.0%, is ~6.90%. The ore-forming components are represented by iron and sulfur, where iron predominates in the sulfide form. The proportion of sulfur is 22.90%. The mineral composition of the flotation concentrate samples is 58% composed of rock–forming minerals in the following proportions: plagioclases – 26%; quartz – 18%; Mica – 9% and carbonates - 5%.

Ore mineralization of the flotation concentrate is represented mainly by pyrite with the mass fraction of 40.7%. Other sulfides are observed in tenths of a percent. Iron hydroxides make up 0.5% of the total sample mass.

It has been established that gold in the flotation concentrate sample is present in native form. The ratio of gold in the form of tellurides of various compositions accounts for no more than 0.04%. The bulk of gold grains is represented by particles with a size of 10-38 microns - 82.89%. About 63% of gold is in free form. The proportion of closed grains is 9.23%. The main mineral in the flotation concentrate sample, the gold is in association with, is pyrite – 25%. The ratio of gold associated with quartz is 3.07%. Tenths and hundredths of a percent of the total mass of native gold have been established in intergrowth with other minerals.

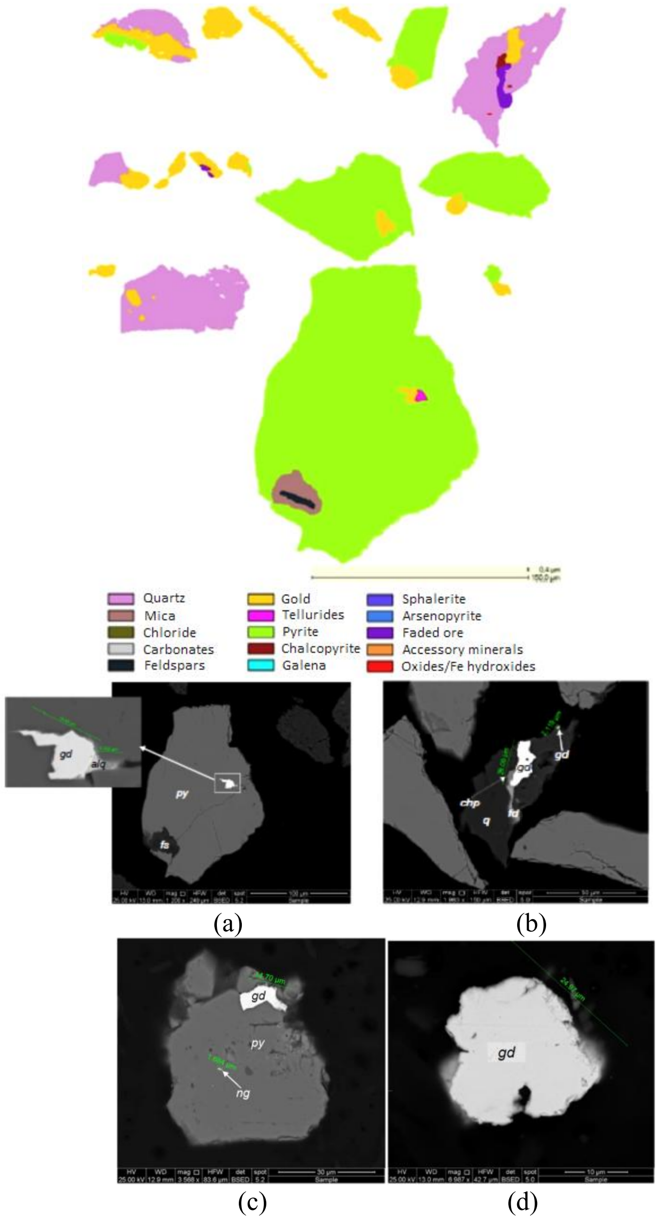


***Figure 4. Map of gold mineral particles in a flotation concentrate sample and photographs of gold grains in association with various minerals: (a) – Pyrite (py) with inclusions of feldspar (fs) and native gold (gd) in association with lead mineral (altaite (alq)); (b) – Native gold (gd) in association with quartz (q), chalcopyrite (chp), faded ore (fd); (c) – Pyrite (py) with gold inclusions of variable composition (native gold (gd), nagiagite (ng)); (d) – Free grain of native gold (gd)***

It should be expected that the resistance of gold to the cyanide process will be due to the presence of oxidized films on the surface of the metal and its association with carbonates and minerals insoluble in aqua regia.

### 3.4. Determination of sorption activity index (PRI test) of flotation concentrate

The level of sorption activity was assessed analytically by calculating the PRI (preg-robbing index). The method for determining the sorption activity index consists in analyzing the amount of gold extracted into the solution during cyanidation and the remaining amount in the solution after cyanidation, when a certain amount of gold in the form of a cyanide complex was previously added to the solution. Thus, each PRI test consists of two parts:

1. Cyanidation of a sample of the test material in a pure gold-free solution;

2. Cyanidation of a sample of the test material in a solution previously saturated with gold to a certain concentration.

For each part of the PRI tests, 100 g of pre-ground concentrate was used to a coarseness of P80 71, 30 and 10 microns. Cyanidation was performed in the mode without sorbent loading with the following parameters: test duration – 1 hour; pulp density – 33.3%; initial concentration of NaCN – 0.3%; concentration of NaOH – 0.1%.

A gold concentration of 1.71 mg/l was previously created in the solution for the second part of the PRI tests.

After a set cyanidation time, the solution was filtered out and the concentration of gold in it was determined by atomic absorption method. The sorption activity index was calculated using the formula:

$$PRI = 2 \times 1.71 + 2\cdot([Au]_{solution\ 1} - [Au]_{solution\ 2}), \tag{1}$$

where: PRI is sorption activity index; $[Au]_{solution\ 1}$ is concentration of gold in the solution of the first part of the test, mg/l; $[Au]_{solution\ 2}$ is concentration of gold in the solution of the second part of the test, mg/l.

Based on the calculated values of the PRI index sorption activity was assessed using the following criteria:

– PRI = 0 – the material does not have sorption activity;

– 0 < PRI < 1.0 – the material has low sorption activity;

– 1.0 ≤ PRI < 2.5 – the material has moderate sorption activity;

– 2.5 ≤ PRI ≤ 3.4 – the material has high sorption activity.

The results for determining the sorption activity index are shown in Table 4.

***Table 4. Results of the sorption activity index (PRI)***

| Concentrate grinding size $P_{80}$, microns | $[Au]_{solution\ 1}$, mg/l | $[Au]_{solution\ 2}$, mg/l | Assessment of sorption activity |
|---|---|---|---|
| 71 | 0.25 | 0.05 | Very high |
| 30 | 0.38 | 0.12 | Very high |
| 10 | 0.51 | 0.28 | Very high |

Analysis of the obtained data shows that the studied concentrate has a very high sorption activity at all degrees of grinding. The PRI index significantly exceeds 3.4 for all three fractions, which indicates a strong «preg-robbing» effect.

An increase in the degree of grinding leads to a slight decrease in the PRI index, although it remains in the range of «very high» sorption activity. This may be due to an increase in the surface area of the material with finer grinding, which contributes to a more intensive absorption of gold.

The high sorption activity of the concentrate can significantly reduce the efficiency of the cyanidation and gold extraction process. The material actively absorbs gold from the cyanide solution, preventing it from passing into solution and further extraction.

To increase the efficiency of gold extraction from this concentrate, it is necessary to apply methods aimed at reducing or blocking sorption activity. Such methods include pretreatment with activated carbon, the addition of special depressant reagents, or the use of higher concentrations of cyanide. The choice of the optimal method depends on the specific characteristics of the material and the process conditions.

In this work, Norit RO 3520 activated carbon was used as a sorbent.

### 3.5 Cyanidation of flotation concentrate of various sizes

The experiments were carried out in two modes – in the cyanidation mode without loading the sorbent and in the sorption cyanidation mode with the addition of activated carbon Norit RO 3520. The experimental conditions and parameters of agitation cyanidation of the flotation concentrate are presented in Table 5.

***Table 5. Parameters of agitated cyanidation of flotation concentrate***

| Parameter | Unit | Measurement |
|---|---|---|
| Sodium cyanide concentration | % | 0.2 |
| pH | – | 10.5 |
| Pulp density during cyanidation | % solid | 40 |
| Coal loading (only for sorption type of cyanidation) | % of liquid phase volume | 10 |
| Type of sorbent | – | Activated carbon Norit RO 3520 |
| Leaching duration | hour | 24 |

Agitation cyanidation of the flotation concentrate was carried out on the material obtained after fine grinding in a ball mill to a size of P80 71 and 45 microns and ultrafine grinding in a bead mill to a size of P80 30, 20, 10 and 7 microns.

The results of cyanidation tests for flotation concentrate at different grinding sizes are shown in Table 6.

***Table 6. Results of cyanidation of flotation concentrate at different grinding sizes***

| Material size $P_{80}$, microns | Reagent consumption, kg/t | | | Au content, g/t | | Au recovery, % |
|---|---|---|---|---|---|---|
| | NaCN | | CaO | in the feed | in a cake | |
| | total | considering the residue | | | | |
| Cyanidation without sorbent loading | | | | | | |
| 71 | 3.3 | 0.7 | 1.5 | 17.9 | 7 | 60.89 |
| 45 | 3.8 | 1.1 | 1.5 | | 8.4 | 53.07 |
| 30 | 3.9 | 2 | 1.4 | | 5.2 | 70.95 |
| 20 | 3.8 | 1.8 | 1.4 | | 4.2 | 76.54 |
| 10 | 3.9 | 2.1 | 1.4 | | 3.4 | 81.01 |
| 7 | 3.9 | 2.2 | 1.5 | | 3.4 | 81.01 |
| Sorption type cyanidation | | | | | | |
| 71 | 3.5 | 1.2 | 2 | 17.9 | 5 | 72.07 |
| 45 | 3.9 | 1.3 | 2 | | 4.7 | 73.74 |
| 30 | 3.9 | 1.8 | 1.9 | | 3.3 | 81.56 |
| 20 | 3.8 | 2 | 1.9 | | 3 | 83.24 |
| 10 | 3.9 | 2.1 | 2 | | 2.4 | 86.59 |
| 7 | 4.4 | 3.1 | 1.9 | | 2.4 | 86.59 |

The results presented in Table 6 show that the flotation concentrate is favorable in relation to the process of agitated cyanidation. The recovery of gold from a concentrate with a particle size of P80 71 microns during cyanidation without loading a sorbent was 60.89%, and with sorption type cyanidation – 72.07%. A comparative analysis of the results of direct and sorption type cyanidation in the studied range of changes in P80 size from 71 to 7 μm indicates the presence of sorption activity of the flotation concentrate. Figure 5 visualizes the dependence of gold recovery on the size of the material.

Grinding the concentrate from a particle size of P80 71 microns to a particle size of P80 45, 30, 20, 10 and 7 microns allows increasing gold recovery in the sorption mode of cyanidation by 1.68%, 9.5%, 11.17%, 14.53% and 14.53% (abs.), respectively. At the same time, the gold content in the cake, compared with cyanidation without sorbent loading, is reduced by 0.3 g/t, 1.7 g/t, 2.0 g/t, 2.6 g/t and 2.6 g/t (abs.), respectively.

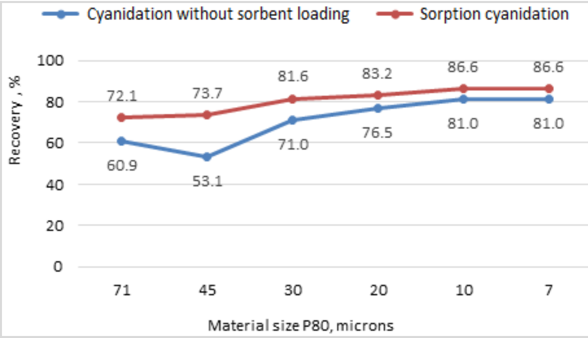


***Figure 5. Dependence of gold recovery on flotation concentrate size***

The maximum extraction of gold into the solution (86.6%) is achieved when the particle size of the flotation concentrate P80 is 10 microns. Finer grinding of the concentrate does not lead to an increase in gold extraction. Considering this fact, in subsequent experiments, the particle size of the flotation concentrate was fixed at the level of P80 10 microns.

As shown in Figure 5, with a comparable concentrate particle size, the gold recovery rates obtained in two different modes after the leaching process are significantly different. This highlights the importance of conducting additional studies to determine the sorption properties of the concentrate.

### 3.6 Effect of sodium cyanide concentration on gold recovery during cyanidation of flotation concentrate

To explore the possibility of reducing cyanide consumption during cyanidation of flotation concentrate, leaching tests were performed at different concentrations of sodium cyanide in solution: 0.3%, 0.2%, 0.15% and 0.1%.

Tests were carried out in sorption mode on material with a particle size of P80 10 µm. The pulp density during leaching is 40% solid. Sorption type cyanidation was carried out with coal loading in an amount of 10% of the volume of the liquid phase. Leaching duration is 24 hours. The test results are presented in Table 7.

***Table 7. Results of sorption type cyanidation of flotation concentrate at different concentrations of sodium cyanide in solution***

| Concentration | Reagent consumption, kg/t | | | Au content, g/t | | Recovery |
|---|---|---|---|---|---|---|
| | NaCN | | | | | |
| NaCN, % | total | considering the residue | CaO | in the feed | in a cake | Au, % |
| 0.1 | 2.3 | 1.7 | 1.9 | 17.9 | 2.4 | 86.59 |
| | 2.3 | 1.7 | 1.9 | | 2.5 | 86.03 |
| 0.15 | 2.8 | 1.6 | 1.9 | | 2.4 | 86.59 |
| | 2.8 | 1.6 | 1.9 | | 2.6 | 85.47 |
| 0.2 | 3 | 1.6 | 1.6 | | 2.4 | 86.59 |
| | 3 | 1.6 | 1.6 | | 2.4 | 86.59 |
| 0.3 | 4.5 | 1.5 | 1.6 | | 2.4 | 86.59 |
| | 4.5 | 1.6 | 1.7 | | 2.4 | 86.59 |

An increase in the concentration of sodium cyanide in solution does not affect the extraction of gold into solution (Table 7). The average values of gold extraction in the considered range of NaCN concentration changes (from 0.1 to 0.3%) practically remain at the same level and are: 86.03% – at NaCN = 0.1% and 86.59% – at NaCN = 0.3%.

The results indicate that flotation concentrate particle size is a more significant factor than the cyanide concentration used to determine the percentage dissolution of gold particles. At a cyanide concentration of 0.1% in solution, gold recovery was 86.32% (average). Increasing the concentration above a given limit only slightly increases recovery. This indicates that increasing the cyanide concentration above 0.1% does not have a significant effect on the rate of gold dissolution, and may even slow it down [17]. In addition, an increase in the content of metal cyanide complexes in the solution can complicate the cyanidation process and significantly increase the cost of gold production [18].

An increase in gold recovery with a decrease in the size of the flotation concentrate is fully explained by an increase in the contact surface area for leaching reactions to occur [19]. The obtained results convincingly show that the P80 concentrate size of 10 microns and the sodium cyanide concentration of 0.1% are the best combination of the concentrate cyanidation process providing a high gold recovery of 86.59%.

### 3.7. Effect of cyanidation process duration on gold extraction from flotation concentrate

Agitation cyanidation of the flotation concentrate was carried out in sorption mode on a material with a grain size of P80 10 microns at a constant concentration of sodium cyanide equal to 0.1%. The duration of the process was: 8, 12, 16, 20, 24, 32, 40 and 48 hours. The pulp density during leaching is 40% solid, pH = 10.5. The acidity level was maintained at 10.5 by adding lime. The results of the concentrate cyanidation tests at different process times are shown in Table 8.

***Table 8. Results of cyanidation of flotation concentrate at different process durations***

| Duration process, h. | Reagent consumption, kg/t | | | Au content, g/t | | Recovery |
|---|---|---|---|---|---|---|
| | NaCN | | | | | |
| | Total | considering the residue | CaO | in the feed | in the cake | Au, % |
| 8 | 1.9 | 1 | 1.9 | 17.9 | 2.6 | 85.47 |
| 12 | 1.9 | 1.4 | 1.9 | | 2.6 | 85.47 |
| 16 | 1.9 | 1.6 | 1.9 | | 2.5 | 86.03 |
| 20 | 1.9 | 1.6 | 1.9 | | 2.4 | 86.59 |
| 24 | 2.4 | 1.7 | 2 | | 2.4 | 86.59 |
| 32 | 2.6 | 1.9 | 2.4 | | 2.4 | 86.59 |
| 40 | 2.7 | 2.2 | 2.4 | | 2.5 | 86.03 |
| 48 | 2.7 | 2.2 | 2.4 | | 2.5 | 86.03 |

An analysis of the data obtained shows that the majority of gold is extracted in the first 8-12 hours of cyanidation, reaching 85.47%. Increasing the processing time to 32 hours has little effect on gold recovery, leading only to an increase of 1.12% (abs.).

To ensure the stability of technological parameters during the industrial processing of flotation concentrate obtained from stubborn ores of Central Kazakhstan, it is recommended to carry out cyanidation for at least 24 hours, which will allow 86.59% of gold to be extracted into the solution.

A further increase in the duration of cyanidation, as studies show, is not economically feasible. The cost of reagents, energy consumption and depreciation of equipment in this case exceeds a slight increase in gold extraction. Therefore, optimization of the cyanidation process should be aimed at maintaining optimal conditions during the first 24 hours, ensuring maximum speed and completeness of gold extraction.

The key factors influencing cyanidation efficiency are cyanide concentration and the pH of the medium. Maintaining the optimal acidity of the solution in the pH range of 10.5-11.5 contributes to the stable formation of soluble cyanide complexes of gold. The cyanide concentration should be sufficient to maintain the dissolution process, but not excessive, in order to avoid reagent loss and the formation of undesirable by-products.

Thus, in order to achieve optimal results in cyanidation of flotation concentrate from stubborn ores in Central Kazakhstan, it is necessary to carefully monitor and maintain optimal technological parameters during the first 24 hours of the process. This will ensure stable and economically profitable extraction of gold.

## 4. Conclusions

The flotation concentrate obtained from gold-bearing ore mined in Central Kazakhstan is a promising raw material for the extraction of gold by cyanidation. However, an analysis of the component composition of the concentrate and the forms of gold finding indicates potential difficulties in extracting stubborn forms, especially from sulfide minerals. This concentrate demonstrates favorable characteristics for the process of agitation cyanidation.

The determination of the PRI index showed that the material has a very high sorption activity in relation to gold. The PRI indices for the concentrate with P80 particle sizes of 71, 30, and 10 microns were 5.02, 4.08, and 3.66, respectively.

When cyanidation of a concentrate with a particle size of P80 71 microns without the use of a sorbent achieved gold recovery at the level of 60.89%. The use of sorption cyanidation with the addition of activated carbon made it possible to increase gold extraction to 72.07%.

A decrease in the particle size of the concentrate from P80 71 microns to P80 45, 30, 20, 10 and 7 microns, compared with cyanidation without the addition of activated carbon, leads to an increase in gold extraction in the sorption regime by 1.68%, 9.5%, 11.17%, 14.53% and 14.53% (abs.), respectively. At the same time, the gold content in the cake decreases by 0.3 g/t, 1.7 g/t, 2.0 g/t, 2.6 g/t and 2.6 g/t (abs.).

For industrial applications, sorption cyanidation of Norit RO 3520 activated carbon flotation concentrate is recommended under the following optimal conditions: particle size of the flotation concentrate – P80 10 microns; coal consumption of Norit RO 3520 – 10% of the volume of the liquid phase; concentration of sodium cyanide – 0.1%; consumption of sodium cyanide – 2.3 kg/t; pH – 10.5; density the pulp content is 40%; the duration of the process is 24 hours, which ensures the extraction of gold into solution at a level of at least 86%.

Optimization of the sorption cyanidation process using Norit RO 3520 activated carbon and ultrafine flotation concentrate grinding opens up new horizons for improving the efficiency of gold extraction. Reducing the particle size to P80 10 microns significantly increases the surface area available for interaction with cyanide and sorbent, which, in turn, contributes to a more complete release of gold from the mineral matrix.

The choice of Norit RO 3520 activated carbon as a sorbent is due to its high adsorption capacity and kinetic characteristics that are optimal for extracting gold from cyanide solutions. Maintaining the optimal concentration of sodium cyanide at 0.1% ensures the necessary concentration of cyanide ions for complexation with gold, while the consumption of sodium cyanide 2.3 kg/t is economically feasible and environmentally acceptable.

Maintaining the acidity of the solution at pH = 10.5 is necessary to ensure the stability of cyanide complexes and prevent cyanide hydrolysis. The pulp density of 40% is optimal to ensure good mixing and contact between the solid and liquid phases. The 24-hour process duration provides sufficient time to achieve maximum gold recovery.

The application of the proposed optimal conditions ensures the achievement of gold recovery in solution at a level of at least 86%, which significantly exceeds the indicators achieved by cyanidation without the use of a sorbent. This leads to a decrease in the gold content in the cake and an increase in the overall economic efficiency of the gold extraction process from the gold-bearing ores of Central Kazakhstan.

## Author contributions

Conceptualization: NKD; Data curation: ZAY; Formal analysis: EEZ; Funding acquisition: ZAY; Investigation: ZAY; Methodology: AAA; Project administration: VAK, NKD; Resources: ZAY; Software: EEZ, AAA; Supervision: VAK, NKD; Validation: NKD; Visualization: VAK, NKD; Writing – original draft: NKD, ZAY; Writing – review & editing: EEZ, AAA. All authors have read and agreed to the published version of the manuscript.

## Funding

This research received no external funding.

## Acknowledgements

The authors express their sincere gratitude to the editor and anonymous reviewers for their constructive comments and valuable suggestions, which have significantly improved the quality of this manuscript.

## Conflicts of interest

The authors declare no conflict of interest.

## Data availability statement

The original contributions presented in this study are included in the article. Further inquiries can be directed to the corresponding author.

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# Құрамында алтыны бар кендерін байытқаннан кейін алынған флотоконцентратты циандау процесінің оңтайлы технологиялық режимін және параметрлерін айқындау

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**Аңдатпа.** Флотоконцентраттың заттық құрамын және алтынның табылу нысандарын кешенді зерттеу нәтижесінде концентраттың химиялық құрамы 55.17%-ға литофильді компоненттермен ұсынылғаны анықталды, олардың массалық үлесі 55.17% құрайды. Олардың негізгілері сәйкесінше 36.4% және 8.26% массалық үлестері бар кремний диоксиді және глинозем болып табылады. Флотоконцентраттың кенді минералдануы пиритпен ұсынылған, оның массалық үлесі – 40.7%. Флотоконцентрат үлгісіндегі алтын табиғи түрде болатыны анықталды. Алтын дәндерінің негізгі бөлігі 10-38 мкм – 82.89% тұрады. Алтынның шамамен 63%-ы еркін күйде. Жабық дәндердің үлесі 9.23% құрайды. Алтын ассоциацияланған флотоконцентрат үлгісіндегі негізгі минерал – пирит - 25%. Кварцпен байланысты алтынның үлесі 3.07% құрайды. Өнеркәсіпте пайдалану үшін оңтайлы режим ретінде сұйық фаза көлемінің 10% мөлшерінде Norit RO 3520 белсендірілген көмір шығыны бар флотоконцентратты сорбциялық циандау ұсынылады. Флотоконцентратты сорбциялық циандау параметрлері белгіленді: флотациялық концентраттың ірілігі - P80 10 мкм; натрий цианидінің концентрациясы – 0.1% (натрий цианидінің шығыны – 2.3 кг/т); pH – 10.5; пульпаның тығыздығы - 40% (қатты); процестің ұзақтығы – 24 сағат, белгіленген параметрлерде жоғары, ерітіндіге кемінде 86% алтынды бөліп алу қол жеткізілді.

***Негізгі сөздер:*** *алтын, флотоконцентрат, циандау, натрий цианиді, концентрация, белсендірілген көмір, сорбциялық циандау.*

# Определение оптимального технологического режима и параметров процесса цианирования флотоконцентрата, полученного после обогащения упорных золотосодержащих руд

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**Аннотация.** В результате комплексных исследований вещественного состава флотоконцентрата и форм нахождения золота установлено, что химический состав концентрата на 55.17% представлен литофильными компонентами, массовая доля которых составляет 55.17%. Основными из них являются кремнезем и глинозем с массовыми долями 36.4% и 8.26%, соответственно. Рудная минерализация флотоконцентрата представлена пиритом, массовая доля которого составляет 40.7%. Установлено, что золото в пробе флотоконцентрата присутствует в самородной форме. Основная масса зерен золота представлена частицами размером 10-38 мкм – 82.89%. Порядка 63% золота находится в свободном виде. Доля закрытых зерен составляет 9.23%. Основным минералом в пробе флотоконцентрата, с которым золото находится в ассоциации, является пирит – 25%. Доля золота, ассоциированного с кварцем, составляет 3.07%. Для промышленной эксплуатации в качестве оптимального режима рекомендуется сорбционное цианирование флотоконцентрата с расходом активированного угля Norit RO 3520 в количестве 10% от объема жидкой фазы. Установлены параметры сорбционного цианирования флотоконцентрата: крупность флотационного концентрата – P80 10 мкм; концентрация цианида натрия – 0.1% (расход цианида натрия – 2.3 кг/т); pH – 10.5; плотность пульпы – 40% (твердого); продолжительность процесса – 24 ч. При установленных параметрах достигнуто высокое, не менее 86%, извлечение золота в раствор.

***Ключевые слова:*** *золото, флотоконцентрат, цианирование, цианид натрия, концентрация, активированный уголь, сорбционное цианирование.*

**Publisher's note**

SATBAYEV UNIVERSITY

Engineering Journal of Satbayev University

*Volume 147 (2025), Issue 5, 10-16*

https://doi.org/10.51301/ejsu.2025.i5.02

# Study of the acoustic properties of new smelted steels alloyed with chromium, vanadium, and manganese

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**Abstract.** The article presents the results of a comprehensive study of the acoustic, damping, and vibration properties of newly cast steels alloyed with vanadium, chromium, and manganese. The relevance of the work is determined by the growing industrial need to reduce impact noise and vibration generated during the operation of machinery, mining, and metallurgical equipment. Increased acoustic loads lead to accelerated wear of components, reduced reliability of units, and adverse effects on workers, which makes the search for new materials particularly important. The study includes an analysis of literature data on existing noise-reduction methods and demonstrates the advantages of using alloys with enhanced internal damping compared to traditional structural materials. The experimental part was carried out by modeling impact processes using specialized measuring equipment that made it possible to record sound pressure levels, frequency spectra, and vibration decay rates. Special attention was paid to the influence of chemical composition, phase structure, and grain morphology on the acoustic characteristics of the steels. It was established that the optimal combination of alloying elements promotes the formation of a structure that provides more efficient vibration attenuation. The results obtained can be used in the development of new materials and protective components aimed at reducing industrial noise and increasing equipment durability.

***Keywords:*** *impact noise, damping properties, acoustic properties, vanadium alloying, low-alloy steels, noise reduction.*

Received: 05 August 2025
Accepted: 16 October 2025
Available online: 31 October 2025

## 1. Introduction

Reducing noise levels in the environment is one of the pressing tasks of modern science and technology. Among the various sources of acoustic impact, industrial noise occupies a special place, the level of which has increased significantly in recent years due to the widespread introduction of high-performance machines and mechanisms, as well as an increase in the operating speeds of equipment. The most common and harmful type of industrial noise is mechanical noise of impact and impulse origin, whose levels can reach 120 dB [1-5].

Impulse and impact noises are widespread in metallurgy, mechanical engineering, and transportation. Their unexpected impact can cause stress reactions, increased blood pressure, rapid breathing, arrhythmia, and decreased mental performance. Therefore, the problem of reducing impact noise is considered one of the priority areas of research related to improving the reliability and safety of machines and mechanisms.

Modern noise control methods include sound insulation, noise source shielding, sound absorption, vibration absorption, increasing structural rigidity, using damping devices and materials, and rational equipment placement. However, these approaches often involve an increase in the weight, size, and cost of machines, as well as a reduction in the manufacturability of structures. In particular, the use of non-metallic sound-absorbing materials (rubber, polyurethanes, plastics) is limited by their low strength and thermal stability, which makes them unsuitable for high temperatures and aggressive environments typical of the metallurgical and machine-building industries [2, 3].

In this regard, the use of metallic materials with enhanced damping properties that can effectively reduce the level of sound emission at its source is of particular interest. Such materials are distinguished by their simplicity of design, stability of properties when the frequency of vibrations changes, high strength, wear resistance, and a wide range of operating temperatures. Nevertheless, the damping mechanism of metal alloys has not been fully studied, and attempts to establish a functional relationship between the loss coefficient and other physical and mechanical characteristics have not yet yielded conclusive results.

The most promising for research are low-alloy structural steels used for the manufacture of critical machine parts – shafts, axles, gears, bushings, and couplings. These materials include 16Cr4 (AISI 5115), 20Cr4 (AISI 5120), 20MnCr5 (AISI 8620), and 16CrV4 (AISI 6150), which are distinguished by their combination of strength, wear resistance, and toughness after heat treatment.

This study examines the acoustic and damping properties of new low-alloy steels alloyed with manganese, chromium, and vanadium. Vanadium, being a constant component of most steels, significantly affects the nature of non-metallic inclusions and microstructure, while chromium and manganese contribute to increasing the strength and damping characteristics of the material [1, 5].

The aim of this work is to establish the patterns of influence of vanadium, chromium, and manganese alloying on the acoustic and damping properties of steels used under impact and friction loads, with the goal of developing materials that reduce impact noise in mechanical engineering.

For many decades, research into reducing industrial noise and vibration has attracted the attention of researchers in the fields of mechanical engineering, materials science, and acoustics. The problem of impulse and impact noise generated by modern high-speed equipment is particularly acute. Analysis of the literature shows that the level of impact noise in industrial conditions can reach 110-120 dB, which significantly exceeds acceptable health standards [6, 7].

According to [8-12], standard industrial sound level meters do not provide sufficient accuracy when measuring short-term sound pulses, since their integration periods do not meet the requirements for recording instantaneous peak values. To correctly measure impulse noise affecting the hearing organs, it is necessary to use devices with a constant integration time of about 20 μs, capable of recording peak sound pressure. Studies [13] have shown that even short sound pulses with an amplitude of up to 140 dB can cause damage to the hearing organs, despite their subjectively perceived lower volume.

In mechanical engineering and metallurgy, the source of intense impact noise is the collision of machine parts and mechanisms, for example, in pipe rolling mills, where the sound level reaches 110-118 dB [14]. In this regard, active research is being conducted to develop materials and structures with enhanced damping properties that can effectively absorb acoustic energy.

The most widely used methods of noise reduction are sound insulation, shielding, sound absorption, and vibration damping [15-19]. However, each of these approaches has significant drawbacks: increased machine dimensions, reduced compactness of structures, increased metal consumption, and more complicated operation. The use of special damping devices increases the cost and complicates the maintenance of equipment, and the rational placement of noisy units requires significant capital investments in the reconstruction of existing workshops.

The use of non-metallic materials (rubber, plastics, polyurethanes) as sound-absorbing elements is limited by their low strength and thermal stability, which makes them unsuitable for use in high-temperature conditions and aggressive environments. Therefore, increasing attention is being paid to metal alloys with a high coefficient of internal friction, which have the ability to dissipate the mechanical energy of vibrations into heat.

The literature notes that high-damping alloys often contain chromium, manganese, and vanadium as the main alloying elements [1, 2]. Manganese increases viscosity and fatigue resistance, chromium contributes to the strengthening of ferrite without reducing plasticity, and vanadium improves the microstructure and regulates the amount of non-metallic inclusions that affect sound attenuation. According to [4], there are more than three thousand grades of steel containing chromium, which confirms its versatility as an alloying element for alloys with high damping properties.

Research [5] have examined alternative methods for improving damping characteristics, including changing the geometry of metal surfaces and creating bimetallic composite materials (45 steel-copper MT, X18H10T steel-AMGS alloy, X18H10T steel-copper M1, etc.) obtained by hot rolling and explosion welding. These materials have shown high efficiency in noise absorption, but their production is technologically complex and expensive.

Despite the results achieved, the problem of creating economical, strong, and durable materials with high internal damping remains unresolved. In particular, the damping mechanisms in low-alloy steels used in mechanical engineering, as well as the influence of vanadium alloying on their acoustic and physical-mechanical properties, have not been sufficiently studied. This circumstance determines the relevance of the present study, aimed at finding patterns that ensure a reduction in impact noise by optimizing the composition of steels and the structure of the material.

## 2. Materials and methods

Standard steels for castings of grades 16Cr4 – AISI 5115, 20Cr4 – AISI 5120, 20MnCr5 – AISI 8620, 16CrV4 – AISI 6150), used for shafts, axles, gears, and couplings, alloyed with chromium, manganese, and vanadium, whose mechanical characteristics are given.

Scrap metal, ferroalloys, waste from our own production, and armored iron were used as metal charge materials. A RelTEC crucible induction furnace was used for smelting. Sheet metal made of 10 steel served as the starting material. Alloying was carried out with 97.6% metallic manganese, 77.5% FeSi, and 99.98% metallic nickel. Synthetic cast iron with a carbon content of 3.9% served as a carbon-containing additive. The steel was cast into a metal mold measuring 210 x 115 x 115 mm.

Based on research into equipment for studying the acoustic proprieties (sound level, sound pressure level) of steels, a device was used for comprehensive research into the acoustic and vibration properties of plate and tubular steel samples, with further modernization. The operational scheme of the setup is shown in Figure 1.

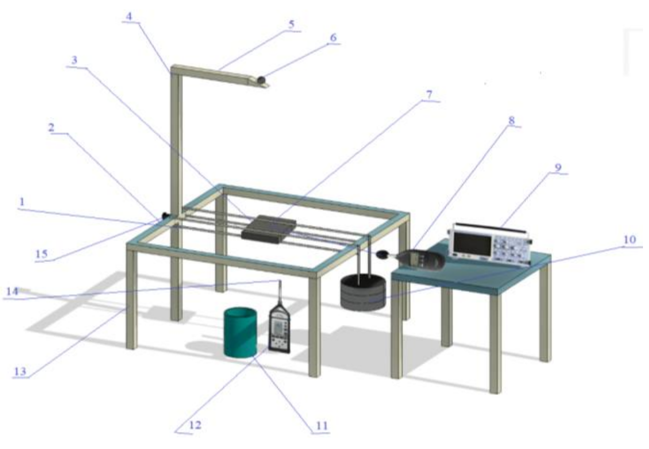

***Figure 1. Device for comprehensive acoustic investigation [13]: 1 – capron threads; 2 - frame; 3 – sample plate; 4 – frame stand; 5 – inclined plane; 6 – ball impactor; 7 – vibration sensor of the Bruel & Kjer sound level meter; 8 – Bruel & Kjer sound level meter model 2204 with octave filter model 1613; 9 – S-18 oscilloscope; 10 – load; 11 – ball receiver; 12 – «Octava 101A» sound level meter; 13 – frame stands; 14 – microphone of the «Octava 101A» sound level meter; 15 – screw for securing the impactor stand***

The ball impactor 6 was placed on the inclined plane 5, down which it rolled and then freely fell onto the geometric center of the plate sample 3. The ball impactor 6 rebounds from the plate sample and is caught in the ball receiver 11. The sound generated as a result of the collision between the ball impactor 6 and the sample 3 is recorded using the «Octava-101A» sound level meter 12.

The plate sample 3, while vibrating in the capron threads 1, causes vibrations that are measured by the Model 2204 apparatus 8 from the company «Bruel&Kjer». The tension of the sample using the capron threads 1 is stable because the load 10 controls the tension force.

The height of the ball's fall is adjusted using the screw for securing the impactor stand 15. The complex consisting of the fastening of the sample 3 and the ball impactor 6 is fixed to the frame 2, which is secured by stands 3 at the required height above the floor.

For the purpose of measurement, ball impactors (SHKh15) made of steel with diameters of 7 mm (1.40 g), 8 mm (2.09 g), 9 mm (2.97 g), and 11 mm (5.55 g) were used.

The study of steel plate samples (50 x 50 x 5) mm was carried out on the installation. The mass of the ball, the density of the sample, the distance from the point of impact to the sample, and the thickness of the sample are interrelated by the following ratio:

$$m \langle 4.6 \cdot \rho \cdot l \cdot h^2 , \tag{1}$$

where $m$ is the mass of the sample plate, g; $\rho$ is the density of the sample plate material, g/cm$^3$; $l$ is the distance from the point of impact to the nearest edge of the sample plate, cm; $h$ is the thickness of the sample plate, cm.

The width and length of the sample plate must be 5 times greater than its thickness. An experimental plate with dimensions of 50x50x5 mm meets these characteristics.

The sound pressure levels were measured in octave frequency bands ranging from 31.5 to 31 500 Hz, and the vibration acceleration levels were measured in the range from 31.5 to 31,500 Hz. The sound level was adjusted according to the «A» scale, and the overall vibration acceleration level was adjusted according to the «Lin» characteristic.

The ZG-10 sound generator was used to calibrate the sound signal studies. Corrections for changes in the sound signal depending on atmospheric pressure were made using a PF-101 pistonphone. A constant air temperature and humidity were maintained in the laboratory. Acoustic measurements were calculated as the arithmetic mean of five measurements [13]. The sound pulse generated by the collision of the sample under study with the striker was recorded using a storage oscilloscope, after which it was photographed and the damping parameters were determined: logarithmic decrement, sound attenuation rate, relative scattering, and internal friction.

## 3. Results and discussion

When developing high-damping alloys, one of the main criteria is that there should be no significant reduction in strength properties. Therefore, one of the reasons for choosing chromium, manganese, and vanadium as alloying elements for iron-carbon alloys was that, among the main alloying elements (the most commonly used), these elements strengthen ferrite more than others (Table 1).

***Table 1. Chemical composition and mechanical properties of the studied steels***

| Steel grade | Chemical composition, % by weight | | | | | | Mechanical properties | | | | |
|---|---|---|---|---|---|---|---|---|---|---|---|
| | C | Si | Mn | Cr | V | Other elements | $\sigma_t$, MPa | $\delta 5$, % | $\psi$, % | $a_n$, J/cm$^2$ | $\sigma_y$, MPa |
| 16Cr4 | 0.12-0.18 | 0.17-0.37 | 0.4-0.7 | 0.7-1.0 | - | ≤0.035 S; | 700 | 12 | 45 | 70 | 500 |
| 20Cr4 | 0.17-0.23 | 0.17-0.37 | 0.5-0.8 | 0.5-0.8 | - | ≤0.035 P; | 800 | 11 | 40 | 60 | 650 |
| 20MnCr5 | 0.15-0.21 | 0.17-0.37 | 0.9-1.2 | 0.9-1.2 | - | ≤0.30 Cu; | 900 | 10 | 40 | - | 750 |
| 16CrV4 | 0.12-0.18 | 0.17-0.37 | 0.4-0.7 | 0.8-1.1 | - | ≤0.3 Ni | 750 | 13 | 50 | 80 | 550 |
| BO-1 | 0.23 | 0.15 | 0.9 | 1.7 | 0.47 | 0.035 S; | 650 | 12 | 35 | 70 | 520 |
| BO-2 | 0.15 | 0.25 | 0.99 | 1.3 | 0.42 | 0.035 P; | 720 | 10 | 40 | 75 | 580 |
| BO-3 | 0.28 | 0.2 | 0.51 | 1.2 | 0.56 | 0.35 Cu; | 810 | 15 | 45 | 71 | 680 |
| BO-4 | 0.13 | 0.17 | 0.45 | 1.5 | 0.51 | 0.4 Ni | 880 | 11 | 50 | 78 | 720 |

Note: $\sigma_t$ is tensile strength, MPa; $\delta 5$ is relative elongation after rupture on five-fold length specimens, %; $\psi$ is relative reduction in area after rupture, %; $a_n$ is impact strength, J/cm$^2$; $\sigma_y$ is yield strength, MPa.

The lack of modern acoustic equipment capable of measuring peak values of impulse sound signals was an obstacle to solving this problem. In this study, sound pressure levels were studied during impact excitation of various alloys using a modern Octava-101A sound level meter, which allows measuring the peak value of a pulsed sound signal that characterizes the initial impact effect with a maximum sound pressure duration of $T = 20$ μs. The secondary effect of the collision was recorded on the «Impulse» scale with an integration time of $T = 35$ ms. This time characterizes the average time of sound perception by human auditory organs. To determine the time of registration of the impact sound pulse, the sound pressure levels and sound levels were studied on the same alloys with an integration time of 7-35 ms and a peak sound pulse delay of 20 μs. Table 2 shows the sound levels and sound pressures of alloys at different integration time constant values of the equipment used.

As can be seen from Table 2 when measuring a sound signal with a duration of $T = 20$ μs, which practically characterizes the instantaneous value of the impact sound pulse, the sound pressure levels for all the alloys studied are close to each other regardless of the chemical composition and type of heat treatment of the samples.

The sound levels of all alloys at this integration time are the same and amount to 125 dBA. Exceedance of peak sound pressure levels above the values on the «Impulse» scale was detected across the entire frequency range and averaged 15 dB, with a difference in sound levels of 13 dBA. However, Table 4 shows that the sound pressure levels at $T = 35$ ms for alloys BO-1 and BO-2 at a frequency of 8.16 kHz do not differ from the peak values of the sound signal at $T = 20$ μs. The reason for this is apparently the increased elastic modulus values for these steels.

***Table 2. Average values of sound levels and sound pressure levels of experimental alloys after various types of heat treatment and at different sound pulse integration times***

| Alloy number | Integration time | Sound pressure levels, dB, in octave bands with geometric mean frequencies, Hz | | | | | | Type of heat treatment | Sound level, dBA |
|---|---|---|---|---|---|---|---|---|---|
| | | 500 | 1000 | 2000 | 4000 | 8000 | 16000 | | |
| BO-1 | 35 ms | 62 | 67 | 63 | 88 | 112 | 112 | normalization | 111 |
| BO-1 | 20 µs | 76 | 85 | 97 | 105 | 121 | 120 | normalization | 125 |
| BO-2 | 35 ms | 56 | 63 | 68 | 87 | 112 | 113 | normalization | 112 |
| BO-2 | 20 µs | 77 | 84 | 97 | 105 | 120 | 119 | normalization | 125 |
| BO-3 | 35 ms | 70 | 73 | 75 | 79 | 98 | 102 | normalization | 109 |
| BO-3 | 20 µs | 76 | 84 | 96 | 105 | 119 | 125 | normalization | 125 |
| BO-4 | 35 ms | 56 | 63 | 64 | 81 | 110 | 97 | quenching, low tempering | 100 |
| BO-4 | 20 µs | 76 | 84 | 97 | 107 | 113 | 113 | quenching, low tempering | 125 |
| 16Cr4 | 35 ms | 50 | 69 | 73 | 78 | 107 | 112 | quenching | 110 |
| 16Cr4 | 20 µs | 77 | 84 | 97 | 105 | 121 | 119 | quenching | 125 |
| 20Cr4 | 35 ms | 68 | 69 | 77 | 91 | 110 | 107 | normalization | 113 |
| 20Cr4 | 20 µs | 82 | 86 | 97 | 107 | 125 | 121 | normalization | 125 |

While the chemical composition and type of heat treatment had virtually no effect on peak sound and sound pressure levels, changing the thickness of the sample from 5 to 4 mm contributed to an average increase in these properties of 3 dB.

Based on the results of the experiment, it can be concluded that the sound pressure levels 35 ms after collision characterize the dissipative properties of the colliding parts. The attenuation of sound vibrations in metal materials in our experiment over 35 ms ranges from 3.5 to 17.5 dBA at a sound attenuation rate of 100 to 750 dB/s, respectively. The acoustic properties were measured on a newly created setup.

Figure 2 show the average values of sound levels and sound pressure levels in octave bands of geometric mean frequencies of the studied steels after normalization.

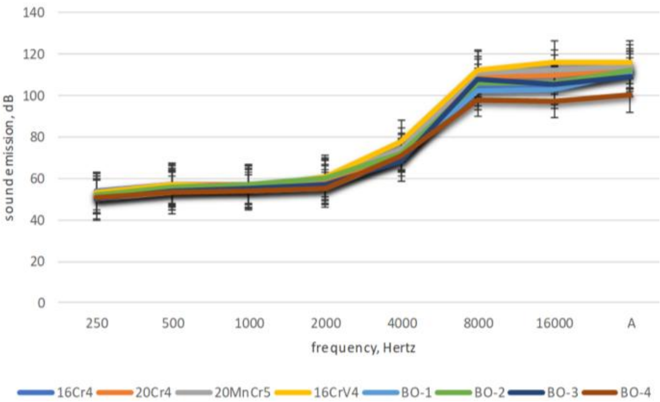


***Figure 2. Average values of sound levels and sound pressure levels of the investigated steels after normalization***

As can be seen from Figure 2, the peak of the SPL is at frequencies of 8000 and 16000 Hz (112-116 dB). The minimum of the SPL is at a frequency of 250 Hz (50-54 dB).

The sound levels of the alloys studied vary between 110 and 116 dBA. The «loudest» alloys during collisions are 16Cr4 (116 dBA), 18ХГ (114 dBA), and 20Cr4 (113 dBA). The BO-4 damping alloy stands out from all other alloys due to its low sound level (100 dBA) and relatively low sound pressure levels. The reason for the high damping properties of the BO-4 alloy (0.13% C; 0.17% Si; 0.45% Mn; 1.5% Cr; 0.51% V, the rest being iron) after normalization is structural damping due to the formation of large metal grains.

The above-mentioned alloys, after cementation and subsequent quenching and low tempering, exhibit increased damping properties compared to normalization. As can be seen in Figure 3, the maximum sound pressure levels in the octave bands of mean geometric frequencies are also at 8000 and 16000 Hz (95-110 dB). The minimum levels are at 250-500 Hz (50-58 dB).

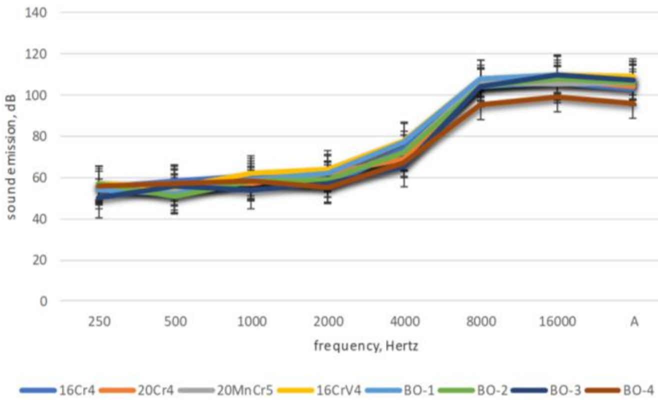


***Figure 3. Average values of sound levels and sound pressure levels of the investigated steels after carburizing, quenching, and low tempering***

Figure 3 clearly shows how the sound pressure levels change after normalization, carburizing followed by quenching, and low tempering. After carburizing followed by quenching and low tempering, the sound pressure levels of the alloys decrease by 4–8 dBA compared to normalization.

The BO-4 alloy is characterized by minimum sound levels (96 dB(A)) and sound pressure levels in octave frequency bands.

After cementation followed by quenching and low tempering, the sound pressure levels of the alloys decrease by (4-8 dBA) compared to normalization.

Figure 4 shows the microstructures of samples BO-3 and BO-4, respectively, after thermomechanical treatment for three cycles.

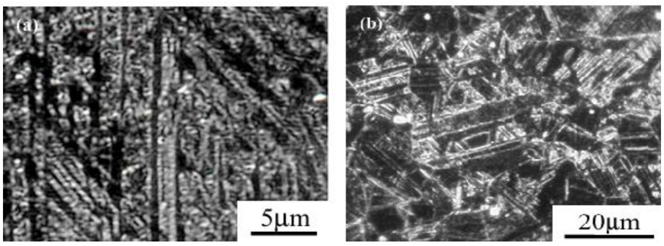


***Figure 4. Microstructure of samples BO-3 (a) and BO-4 (b)***

The effect of manganese content within the range of 0.5-1.0% on the acoustic properties of alloys is not noticeable. Although according to data, manganese and manganese-based alloys have high damping capacity at low and high deformation amplitudes ($\varphi$ = 7-40%), in Fe-Mn alloys -C alloys, the damping capacity is mainly determined by the carbon content and the influence of manganese on phase transformations.

Manganese, dissolving in ferrite and combining with carbon to form carbides, increases the hardness and strength of steel. Due to the fact that in the presence of manganese, transformations shift to lower temperatures and eutectoid forms at lower carbon concentrations, the structure of manganese steels is less differentiated. By causing significant supercooling and increasing the stability of austenite during isothermal transformation and in the upper and lower martensite temperature ranges, manganese contributes to the formation of large amounts of residual austenite and the possibility of a-phase formation. Since manganese forms carbides that are easily soluble in austenite, the steels under study, even with slight overheating (900-950°C, 1 hour in the furnace), had large and coarse grains, both in hypoeutectoid and hypereutectoid steel. However, the formation of large coarse grains led to an improvement in the damping capacity of the steel.

## 4. Conclusions

Based on experimental studies of the acoustic properties of alloyed steels, it has been established that sound pressure levels during impact excitation depend mainly on the structure and thickness of the sample, while the chemical composition and type of heat treatment have less influence on peak sound values. When measuring peak pulses with a duration of 20 µs, the sound pressure levels for all the steels studied are practically the same and amount to about 125 dBA, which corresponds to the instantaneous effect of collision.

Measurements taken at an integration time of 35 ms showed that the sound pressure values obtained characterize the dissipative (damping) properties of the alloys. The attenuation of sound vibrations in metallic materials over 35 ms is 3.5-17.5 dBA at an attenuation rate of 100-750 dB/s, which reflects the difference in the ability of alloys to absorb acoustic energy.

The alloy BO-4 (0.13% C; 0.17% Si; 0.45% Mn; 1.5% Cr; 0.51% V, the rest being iron), which exhibited the lowest sound level of 96-100 dBA after normalization and cementation followed by quenching and low tempering. The increased acoustic resistance of this alloy is explained by structural damping caused by the formation of a coarse-grained microstructure.

It has been shown that alloying steels with vanadium, chromium, and manganese improves their elastic and damping characteristics without significantly reducing their strength properties. It has also been established that increasing the manganese content within the range of 0.5-1.0% does not have a noticeable effect on acoustic characteristics, but contributes to the formation of residual austenite and improves viscosity.

Thus, the developed and researched vanadium-containing low-alloy steels have an optimal combination of strength and damping properties, which allows them to be recommended for use in mechanical engineering in the manufacture of parts subject to shock and vibration loads (gears, axles, shafts, couplings, etc.).

## Author contributions

Conceptualization: RZA, GAB; Data curation: RZA, AVB, GAB; Formal analysis: GAB; Funding acquisition: RZA, GAB; Investigation: RZA; Methodology: RZA, GAB; Project administration: RZA; Resources: RZA, GAB; Software: AVB, GAB; Supervision: GAB, RZA; Validation: RZA, GAB; Visualization: AVB, GAB; Writing – original draft: RZA, AVB, GAB; Writing – review & editing: RZA, AVB, GAB. All authors have read and agreed to the published version of the manuscript.

## Funding

This research was funded by Science Committee of the Ministry of Science and Higher Education of the Republic of Kazakhstan, grant number AP27511849 «Development of alloy steels with dissipative properties and their surface modification by deposition of nanostructured wear-resistant coatings for critical components in mechanical engineering».

## Acknowledgements

The authors express their sincere gratitude to the editor and anonymous reviewers for their constructive comments and valuable suggestions, which have significantly improved the quality of this manuscript.

## Conflicts of interest

The authors declare no conflict of interest.

## Data availability statement

The original contributions presented in this study are included in the article. Further inquiries can be directed to the corresponding author.

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# Хром ванадий марганецпен легирленген жаңа балқытылған болаттардың акустикалық қасиеттерін зерттеу

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**Аңдатпа.** Мақалада ванадий, хром және марганец элементтерімен легірленген жаңа балқытылған болаттардың акустикалық, демпферлік және дірілдік қасиеттерін кешенді зерттеу нәтижелері келтірілген. Жұмыстың өзектілігі өнеркәсіптің машина жасау, тау-кен және металлургия жабдықтарын пайдалану кезінде туындайтын соққы шуы мен діріл деңгейін төмендету қажеттілігінің артуына байланысты. Акустикалық жүктемелердің жоғарылауы бөлшектердің тез тозуына, қондырғылардың сенімділігінің төмендеуіне және жұмысшыларға жағымсыз әсерлерге әкеледі, бұл жаңа материалдарды іздеуді ерекше сұранысқа ие етеді. Зерттеу барысында шуды болдырмаудың қолданыстағы әдістері бойынша әдеби деректерге талдау жүргізілді және дәстүрлі құрылымдық материалдармен салыстырғанда ішкі демпфері жоғары қорытпаларды пайдаланудың артықшылықтары көрсетілді. Эксперименттік бөлік дыбыс қысымының деңгейін, жиілік спектрін және тербелістердің ыдырау жылдамдығын түсіруге мүмкіндік беретін арнайы өлшеу жабдығын қолдана отырып, соққы процестерін модельдеу негізінде орындалды. Химиялық құрамның, фазалық құрылымның және астық морфологиясының болаттардың акустикалық сипаттамаларына әсеріне ерекше назар аударылады. Легирлеуші элементтердің оңтайлы үйлесімі тербелістерді тиімдірек сөндіруді қамтамасыз ететін құрылымның қалыптасуына ықпал ететіні анықталды. Алынған нәтижелер өндірістік шуды азайтуға және жабдықтың беріктігін арттыруға бағытталған жаңа материалдар мен қорғаныс элементтерін әзірлеу кезінде пайдаланылуы мүмкін.

***Негізгі сөздер:*** *соққы шуы, демпферлік қасиеттер, акустикалық өнімділік, ванадиймен легірлеу, төмен легірленген болаттар, дыбыс шығаруды бәсеңдету.*

# Исследование акустических свойств новых выплавленных сталей, легированных хромом ванадием марганцем

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**Аннотация.** В статье представлены результаты комплексного исследования акустических, демпфирующих и вибрационных свойств новых выплавленных сталей, легированных ванадием, хромом и марганцем. Актуальность работы обусловлена возрастающей потребностью промышленности в снижении уровня ударного шума и вибраций, возникающих при эксплуатации машиностроительного, горнорудного и металлургического оборудования. Повышенные акустические нагрузки приводят к ускоренному износу деталей, снижению надёжности агрегатов и неблагоприятному воздействию на работников, что делает поиск новых материалов особенно востребованным. В ходе исследования проведён анализ литературных данных по существующим методам шумоподавления и продемонстрированы преимущества использования сплавов с повышенным внутренним демпфированием по сравнению с традиционными конструкционными материалами. Экспериментальная часть выполнялась на основе моделирования ударных процессов с применением специализированного измерительного оборудования, позволяющего фиксировать уровень звукового давления, частотный спектр и скорость затухания колебаний. Особое внимание уделено влиянию химического состава, фазовой структуры и морфологии зерна на акустические характеристики сталей. Установлено, что оптимальное сочетание легирующих элементов способствует формированию структуры, обеспечивающей более эффективное гашение колебаний. Полученные результаты могут быть использованы при разработке новых материалов и защитных элементов, направленных на снижение производственного шума и повышение долговечности оборудования.

***Ключевые слова:*** *ударный шум, демпфирующие свойства, акустические характеристики, легирование ванадием, низколегированные стали, снижение звукоизлучения.*

**Publisher's note**

**Engineering Journal of Satbayev University**
*Volume 147 (2025), Issue 5, 17-28*

https://doi.org/10.51301/ejsu.2025.i5.03

# Solutions to improve the mining technology diagram for ore bodies at Vi Kem Copper Mine, Lao Cai province, Vietnam

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**Abstract.** Designing and selecting a mining technology scheme for ore body conditions is complex because it depends on many factors. The technology schemes need to ensure efficiency and safety in the production process. Therefore, it is necessary to propose solutions to improve the mining technology scheme to exploit ore bodies to meet the current production requirements at underground mineral mines of the Vinacomin – Minerals Holding Corporation. Based on the current mining plan and mining technology design scheme of the Vi Kem Copper Mine, the authors conducted a survey, analysis and evaluation, thereby proposing solutions to improve the mining technology scheme for ore bodies at the Vi Kem Copper Mine and selected an area in the mine to conduct design calculations for the proposed solution. To achieve the research results presented in this article, the authors used methods such as data collection, analysis and synthesis, field surveys, analysis of results and evaluation, combined with theoretical approaches to calculate the experimental design area. The proposed options to improve the mining technology diagram are highly feasible when applied. There is a high possibility of applying mechanization in technological stages, increasing productivity and production efficiency of the mine, increasing ventilation capacity, and improving labor safety. The improved technology is basically the same as the preparation plan of the old mining technology diagram. However, each item will be optimized to increase the application of mechanization, bringing about production efficiency and labor safety. The research results show high feasibility when applying the improved technology diagram. Technical calculations show that the mining capacity of each chamber is 1.76 times higher, and labor productivity is nearly 3 times higher than the current technological scheme.

***Keywords:*** *mining technology, solution, ore body, improvement, supplement, Vi Kem Copper Mine.*

Received: 02 May 2025
Accepted: 16 October 2025
Available online: 31 October 2025

## 1. Introduction

The Vinacomin – Minerals Holding Corporation is the only national management unit assigned to manage and exploit major mineral mines (including all ore mines) in Vietnam. With the distribution characteristics and location of the ore bodies, the underground mining method has been applied in several mines, such as: Vi Kem Copper Mine in Lao Cai province; Cho Dien Zinc-lead mine in Thai Nguyen province; Lang Hit Zinc-lead Mine in Thai Nguyen province; Tay Nam Nui Phao Tin Mine and Cuc Duong Zinc-lead Mine in Thai Nguyen province. Vi Kem Copper Mine is one of the mines belonging to the Vinacomin – Minerals Holding Corporation. This is also one of the large underground mining ore mines in Vietnam.

Many mining technology diagrams and systems can be selected and applied to each specific ore body condition in different mines. However, to select a suitable mining technology scheme, it is necessary to analyze and evaluate the influencing factors, the main factors include thickness, slope angle of the ore body, stability of the ore body, mining depth, gas content in the mining area, the impact of mining on the surface works, as well as the conditions of the rock mass such as primary stress state, mechanical strength, layering, degree of cracking, properties of cracks.

The selection of an appropriate mining technology scheme is important because it affects the following issues:
– mineral recovery and ore impoverishment;
– the level of development of the necessary tunnels;
– capital requirements and operating costs;
– the type of necessary equipment selected and applied;
– cycle time and sequence of operations;
– annual mine production (tons/year);
– potential risks and occupational safety.

Actual mining operations at ore mines show that the collapse of rock blocks after mining ore blocks has caused deformation and tunnel collapse incidents, posing potential safety risks to people and equipment that may occur when mining ore bodies, especially at thick and steep ore bodies.

Today, many studies are related to designing mining technology diagrams, optimization, numerical modeling, assessments, and proposals for reasonable mining solutions for ore bodies in different mines and conditions. Some typical studies on development and solutions of ore mining technology [1, 2], studies related to forecasting models, the impact of mining technology on ore quality [3-6], improvement and development of ore mining technology [7-11], studies on ore mining technology in complex conditions, intelligent and automatic mining technology [12-17], forecasting blasting vibration and optimizing ore mining design [18-21], geological forecasting models of ore bodies [22-27], studies on assessing geological conditions and ore exploration [28-31], stability assessment of stopping operations and transportation equipment performance in ore mining [32-35].

In Vietnam, due to the conditions and characteristics of the distribution of ore types in different regions, the mines are all small, scattered, and not concentrated. Therefore, investment in research for mining ore mines using underground mining methods is still limited, mainly manual mining, drilling and blasting methods. A few projects are related to improving drilling and blasting for tunnels [36]; the Vietnam Institute of Mining Science and Technology implements some projects. In 2024 at the Vi Kem Copper Mine, ore bodies were exploited using two mining technology diagrams, including Ore shrinkage stoping and sublevel stoping mining technology diagrams. However, according to the assessment and analysis at the actual site, these mining technology diagrams have not yet brought about efficiency; some technological design steps are not reasonable, affecting labor productivity and mining output, potentially posing a risk to worker safety. In 2025, according to the exploitation plan, output still needs to be maintained and increased higher than in 2024, especially requiring increased output in the following years to meet the actual production situation of the Vi Kem Copper Mine. Thus, the research and improvement of the mining technology diagram for ore bodies at the ViKem Copper Mine is urgent for the current Vietnamese mineral mining industry. The selection of a suitable mining technology diagram for ore bodies is essential. It is the basis for the management agency and the mining mines to choose technology to ensure safety and improve production efficiency. This is a study of scientific and practical significance for the current Vietnamese mineral mining industry. The research results of the paper are applied to the Vi Kem Copper Mine of the Vinacomin – Minerals Holding Corporation to exploit ore bodies with medium to thick thickness to ensure safety and efficiency in the mining process.

## 2. Materials and methods

### 2.1. Study area

The research was conducted at the Vi Kem Copper Mine in Lao Cai province, Vietnam.

The surface boundary is defined in accordance with Mineral Exploitation License No. 1688/GP-BTNMT dated July 12, 2017, covering an area of 155 hectares. The boundary extends along the exploration axis from geological exploration line 65a (adjacent to the licensed iron ore mining area) to line 11d, with an average width of approximately 550 meters, ensuring that all mining works are encompassed within the licensed area.

The depth boundary of the mine is defined at the deepest reserve level of −90 m. The location of the study area is illustrated in Figure 1.


***Figure 1. Location of Vi Kem Copper Mine (modified from [37])***

The map (Figure 1) indicates the spatial position of the Vi Kem Mine in relation to nearby settlements and regional infrastructure, providing a geographical context for study.

### 2.2. Geological characteristics of the study area

#### 2.2.1. Characteristics of lithological distribution

In the exploration area, the following main types of soil and rock are identified: cover layer, weathered layer, gneiss-biotite, granitogneiss, amphibolite, two-mica quartz schist, metamorphic rocks, and several other rocks interbedded with the above formations. The cover layer extends across the entire surface of the exploration area, with a thickness ranging from 5 to 15 m. It consists primarily of red-brown, gray-brown, and black-brown claystone and gravel. The layer is soft and friable, prone to landslides when saturated with water or disturbed by construction activities.

Gneiss-biotite rock interbedded with two-mica quartz schist occupies approximately 55% of the exploration area. The rock is gray-white to ash-gray in color, with a medium- to fine-grained texture, banded structure, and blocky composition comprising quartz, feldspar, and biotite.

Gneiss-biotite associated with migmatite typically surrounds and hosts the ore bodies. It is hard, brittle, fractured, and broken, with low engineering stability in fractured zones, exhibiting a compressive strength below 900 kg/cm$^2$. Rocks located farther from the ore bodies are generally compact, less fractured, and display a durable structural integrity.

Metamorphic rocks constitute the principal ore-bearing formations within the exploration area. Their thickness varies from 0.5 to 30 m. These rocks are typically gray-black or blue-black, containing copper mineralization filling fractures and cavities. Ore-bearing metamorphic rocks are dense, compact, and less fractured, indicating high compressive resistance and stable structural characteristics.

Granitogneiss covers about 20% of the exploration area. It is gray-white, with a granular texture and blocky structure, composed mainly of quartz and biotite. The formation is typically located along the margins of ore bodies.

Fractured sections of granitogneiss show a compressive strength below 863 kg/cm$^2$, whereas intact zones have a stable structure with compressive strengths reaching up to 2156 kg/cm$^2$ in some locations. Additionally, minor occurrences of amphibolite, granite, and recrystallized limestone are observed within the area. These formations are 0.5-5 m thick, exhibit a blocky structure and granular texture, and are generally dense and compact.

### 2.2.2. Stratigraphic characteristics and physical properties of soil and rock

Based on the mechanical strength and the results of the analysis of the physical properties of soil and rock, the relative stability with water, combined with the naked eye observation and the ore-bearing characteristics of soil and rock according to the geological perspective, the soil and rock of the mine are divided into the following layers.

Layer 1. Rock with poor geological structure.

This layer includes the entire weathered crust extending over the exploration area. The composition consists mainly of reddish-brown and gray clay, two-mica quartz schist, and gray-white to yellow-gray gneiss-biotite. Due to intense weathering, the material is soft, loose, and highly susceptible to collapse when exposed to water or subjected to dynamic construction loads. The layer's average thickness ranges from 5 to 15 m, reaching up to 52 m in some locations.

Layer 2. Rock layer above the ore body.

This layer represents a relatively stable geological structure, composed mainly of gneiss-biotite, granitogneiss, two-mica quartz schist, and other interbedded rocks. The thickness of the layers varies between 0.5 and 30 m. The rocks are slightly weathered, exhibiting cracking and fracturing in some sections – areas with increased fracturing show poorer structural stability. Groundwater may be present under slight pressure within fractures and cavities, but its quantity and impact on construction activities are negligible.

Layer 3. Ore-bearing rock layer.

This layer consists of metamorphic rocks and migmatized gneiss-biotite, which form the primary ore-hosting formation. The rocks are dense, compact, and slightly fractured, displaying granular and banded textures. Most layers are structurally solid and coherent, although some localized zones show moderate to intense fracturing.

Layer 4. Rock layer below the ore body.

This layer comprises gneiss-biotite, granitogneiss, amphibolite, and lithified limestone. The rocks are generally solid and compact, with few fractures and a durable geological structure, providing a stable foundation for construction and mining activities.

The average analytical results of physical and mechanical properties for these layers are presented in Table 1 [38].

***Table 1. The average results of the analysis indicators***

| № | Indicator names | Unit | Value | | |
|---|---|---|---|---|---|
| | | | Layer 2 | Layer 3 | Layer 4 |
| 1 | Humidity: | | | | |
| | Dry humidity, $W_{kg}$ | % | 0.163 | 0.163 | 0.131 |
| | Moisture absorption, $W_{hn}$ | % | 0.43 | 0.296 | 0.295 |
| 2 | Porosity, $n$ | % | 1.136 | 0.835 | 0.83 |
| 3 | Density, $\nabla$ | g/cm$^3$ | 2.729 | 2.859 | 2.788 |
| 4 | Weight of volume: | | | | |
| | – dry wind, $\gamma_{kg}$ | g/cm$^3$ | 2.701 | 2.836 | 2.766 |
| | – saturation, $\gamma_{bh}$ | g/cm$^3$ | 2.711 | 2.845 | 2.774 |
| | – absolutely dry, $\gamma_c$ | g/cm$^3$ | 2.699 | 2.835 | 2.764 |
| 5 | Compressive strength: | | | | |
| | – dry wind, $\delta_n$ | kg/cm$^2$ | 760 | 1008 | 1203 |
| | – saturation, $\delta_{bh}$ | kg/cm$^2$ | 721.4 | 960 | 1150 |
| 6 | Tensile strength, $\delta_k$ | kg/cm$^2$ | 75.82 | 94.85 | 109.2 |
| 7 | Internal friction angle, $\phi$ | degree | 36.55 | 37.20 | 37.31 |
| 8 | Cohesion, $C$ | kg/cm$^2$ | 138.5 | 177.8 | 211.6 |
| 9 | Firmness coefficient, $f$ | | 7.439 | 9.046 | 10.21 |
| 10 | Soft deformation coefficient, $k$ | | 0.946 | 0.95 | 0.951 |

### 2.3. Mine's field reserves

The underground mine comprises 18 identified copper ore bodies, designated as TQ1.1, TQ1.2, TQ2.1, TQ2.2, and TQ1a.1, TQ1a.2, TQ5.1, TQ7.1, TQ7.2, TQ7a.1, TQ9.1, TQ9.2, TQ9.3, TQ10.1, TQ11.1, and TQ12.1.

Among these, eight ore bodies – TQ1.1, TQ1.2, TQ2.1, TQ2.2, TQ1a.1, TQ1a.2, TQ5.1, and TQ7.1 – have been classified as industrial ore bodies with approved reserves. The remaining bodies have been identified only as copper ore resources.

The calculation of copper ore reserves and resources was conducted based on geological data presented in the Additional Exploration Report of Copper Ore and Minerals in the Vi Kem Area, Bat Xat District, Lao Cai Province.

The total copper ore reserves (sulfur ore type) amount to 5.154 million tons, of which 0.247 million tons belong to grade 121 and 4.908 million tons to grade 122. The detailed reserves of individual ore bodies at the Vi Kem Copper Mine are presented in Table 2 [38].

***Table 2. The reserves of the ore bodies at the Vi Kem Copper Mine***

| № | Ore body name | Ore block area, m$^2$ | Reserve thickness, m | Volume of the block, m$^3$ | Dry ore reserves, t | Copper content, % | Metal reserves, t |
|---|---|---|---|---|---|---|---|
| 1 | TQ1.1 | 163.697 | 3.51 | 574.194 | 1900.583 | 0.66 | 12.631 |
| 2 | TQ1.2 | 86.188 | 4.87 | 419.504 | 1388.559 | 1.01 | 14.050 |
| 3 | TQ2.1 | 52.015 | 2.94 | 152.675 | 450.390 | 0.65 | 2.942 |
| 4 | TQ2.2 | 47.555 | 3.74 | 177.765 | 524.405 | 0.56 | 2.957 |
| 5 | TQ1a.1 | 18.954 | 3.21 | 60.867 | 201.468 | 0.91 | 1.843 |
| 6 | TQ1a.2 | 24.554 | 3.65 | 89.740 | 297.038 | 0.94 | 2.792 |
| 7 | TQ5.1 | 18.378 | 2.96 | 54.414 | 160.521 | 0.50 | 806 |
| 8 | TQ7.1 | 26.367 | 2.77 | 72.948 | 231.245 | 0.47 | 1.083 |
| Grade 121 Reserve | | 17.946 | 4.15 | 74.474 | 246.509 | 0.88 | 2.166 |
| Grade 122 Reserve | | 419.762 | 3.64 | 1527.632 | 4907.701 | 0.75 | 36.938 |
| Total | | 437.708 | 3.66 | 1602.106 | 5154.210 | 0.76 | 39.104 |

These results were approved by the Mineral Reserves Assessment Council under Decision No. 837/QĐ-HĐTLKS, dated November 4, 2011.

### 2.4. Assessment of the current status of the mining technology diagram of the ore bodies

#### 2.4.1. Opening up the mine's field

The Vi Kem Copper Mine is developed through a pair of inclined shafts for ore transport, material haulage, and mine ventilation, combined with crosscuts connecting the working horizons. After the construction of the inclined shaft station system at level +30, a haulage crosscut is excavated at the same level toward the TQ1.1 ore body to establish the transport horizon for the mine. The central ventilation system is formed by driving a ventilation crosscut at level +150 toward the TQ1.1 ore body, then excavating a ventilation tunnel at the same level in the TQ2.1 ore body, thereby completing the mine’s ventilation horizon.

#### 2.4.2. Mine's field preparation

The mine field is divided into two main areas, the Southeast and the Northwest, separated by the geological line T.53[a]. Within each area, preparation works are organized according to ore body groups by excavating transport and ventilation decline zones and longitudinal levels corresponding to the transport horizons. The initial preparation phase is conducted simultaneously at three ore bodies: TQ2.1, TQ2.2, and TQ1.1, comprising a total of six mining rooms: two at TQ2.1 (2.1-27-BLQ and 2.1-33-BLQ), two at TQ2.2 (2.2-4-BLQ and 2.2-10-DVPT), and two at TQ1.1 (1.1-32-BLQ and 1.1-35-BLQ). The rooms are developed according to the technological schemes of ore shrinkage stoping and sublevel stoping. Manual drilling and blasting are applied to ore shrinkage stopping, while mechanized extraction is used for sublevel stopping. The average mining capacity of each room using ore shrinkage stoping technology reaches about 45000 t/year, whereas mechanized sublevel stoping can achieve up to 125000 t/year.

#### 2.4.3. Mining plan and order

The mining sequence of the mine is carried out on the following principle [39].

With the characteristics of the geotechnical conditions of the mine, the project mobilizes 08 ore bodies that are qualified to be included in the mining design, the reserves of the ore bodies are scattered and not concentrated, and the metal content of each ore body varies. To neutralize the metal content and ensure that the mining content is brought to the processing plant evenly in each period, 2 to 3 ore bodies will be exploited simultaneously in a year. Mobilize and exploit first areas with certain and favorable geological conditions (thickness, slope angle, metal content) and mobilize other areas later.

In a group of ore bodies for the same level, the upper ore body is exploited first, and the lower ore body is exploited later. However, depending on the geological structure and the distance between the ore bodies, the lower ore body can be exploited first if it does not affect the mining room of the upper ore body. In an ore body, the upper layer is exploited first, and the lower layer is exploited later. Reserve areas near the tunnel protection pillars and constructions are arranged to be exploited later.

#### 2.4.4. General assessment of the mining technology of Vi Kem Copper Mine

Currently, the primary mining method applied at the Vi Kem Copper Mine is ore shrinkage stoping, with geological ore reserves extracted by this technology amounting to 3.185 million tons, representing 62% of the total mine reserves. The remaining 38% of the reserves are mined using the ore sublevel stoping method. After implementing these technologies in production, a comprehensive assessment of the mining technology applied at the Vi Kem Copper Mine has been conducted, summarized below [39].

– For the ore shrinkage stoping mining technology diagram: applied to ore bodies and blocks with a 1.2 to 3.5 m ore body thickness.

This is a manual mining technology, low output, very high risk of unsafety because the ore bodies at Vi Kem Copper Mine are a group of ore bodies and have a vein shape, many layers, especially the junction between the ore body and the roof rock, floor rock. In the ore shrinkage stoping mining technology, workers must stand and work directly under the ore block, the rock has been affected by drilling and blasting and cracked above. Implementing temporary support is relatively complex, and there are no specific studies and assessments, so there are many potential risks of falling, rocks, landslides, and unsafe labor.

The ore shrinkage stoping mining technology diagram is only applied to ore bodies with a thickness of less than 3.5 m (no calculation of the roof pressure to choose the appropriate room width). However, in reality, the thickness of the ore body at Vi Kem mine is unstable. Many locations form nests, lenses with local thickness greater than 3.5 m. Therefore, when working through these areas, the roof or floor of the ore body is suspended on the side of the storage room. Due to the separation of layers between the rock and the ore body, it is easy for the roof and side of the storage room to collapse, causing safety issues and loss of resources during the mining process.

The construction organization is challenging to carry out as designed, specifically: the size of the ore after blasting is uneven, the ore discharge work is difficult due to the manual design of the discharge door, so there is a risk of the discharge door being blocked; when removing the ore, the amount of ore is uneven, so the floor of the room is uneven, leading to difficulties in organizing the next construction cycle.

– For the ore sublevel stoping mining technology diagram: pplied to ore bodies and blocks with a thickness of 3.5 to 7.8 m, with a length in the mining block direction of a strike line greater than 300 m.

Manual reverse charge dynamite design calculations are not feasible (using DK52-64 reverse jumbo, drilling depth 12 m). In reality, there is no investment in jumbo, because the cross-section of the tunnels has been reduced from 8.4 to 6.5, so the appropriate drilling machine has not been selected to put into the tunnel. Thus, this technology cannot be applied to production. In addition, the design has not calculated or designed mining technology for ore bodies with a thickness greater than 7.8 m.

### 2.5. Proposing solutions to improve the mining technology diagram for ore bodies at the Vi Kem Copper Mine

Based on the analysis and evaluation of geological conditions, hydrogeology and especially the rock structure of the ore bodies. The selection of mining technology diagram, as well as the arrangement of excavation of tunnels depends on factors such as thickness, slope angle of the ore body, stability of the ore body, cross-section shape of the tunnel, rock mass conditions, primary stress state in the rock mass, mechanical strength, layering, degree of cracking, properties of cracks, water content of the rock mass, depth of the tunnel, impact of mining, surrounding underground works, dynamic loads (if any), etc. Therefore, analyzing and evaluating influencing factors will help select and improve the mining technology diagram more effectively.

Based on the current conditions and the actual operational situation of the mine, the authors propose improvements to the mining technology diagram of the ore bodies in two main directions: for the ore shrinkage stoping mining technological scheme and for the ore sublevel caving mining technological scheme.

#### 2.5.1. Ore shrinkage stoping mining technological diagram

This mining technological scheme is appropriate for ore bodies characterized by steep dips exceeding 75° and small thickness. However, several improvements are required to enhance operational safety, efficiency, and mechanization. In particular, it is necessary to introduce temporary support systems, mechanize the drilling and floor-leveling processes, and improve the handling of hanging rock. The proposed modifications are as follows.

First, it is essential to supplement the calculation of rock pressure in the cutting chamber, including the pressure on the roof, floor, and working face. These calculations should be based on a cutting chamber height of less than 2 m and a width of less than 3.5 m, similar to the methodology used for tunnel pressure assessment. The results will be the basis for selecting an appropriate containment and support solution.

Second, a temporary support system must be incorporated into the continuous blasting shrinkage stoping process. During drilling, creating separation zones in the fractured and non-cohesive rock mass causes ore fragments to detach and fall freely toward the drilling area, posing significant safety risks. Therefore, it is necessary to design and install temporary reinforcement measures suitable for such conditions.

Third, it is necessary to mechanize the leveling of the cutting chamber floor after each ore extraction and discharge cycle. The current manual leveling approach proposed in the design is inefficient and unsafe, as it requires significant labor and time while reducing productivity. To address this, the authors recommend using self-propelled drilling rigs and compact excavators equipped with bucket attachments, long chisels, or scraper blades to level the chamber floor and assist in removing hanging rock. Given that the geological conditions at the Vi Kem Mine differ from those at other sites, the traditional method of poking hanging rock with a drill bit has proven ineffective.

Finally, it is proposed to design a mechanized ore discharge system employing a hydraulic mechanism or an equivalent automated solution to replace the manual ore discharge method currently in use. This improvement will prevent frequent ore blockages during unloading operations, reduce steel consumption for hopper structures, lower manual labor intensity, and significantly enhance safety during ore handling.

### 2.5.2. Ore sublevel caving mining technological diagram

For the ore sublevel caving mining technological diagram, several key parameters are proposed to be adjusted and refined to improve safety, adaptability, and efficiency under the Vi Kem Copper Mine's specific geological and technical conditions.

First, it is recommended that the stratification height be recalculated and optimized, selecting a value smaller than that adopted in the current design. Reducing the stratification height will improve blasting control, ensure more uniform ore fragmentation, and enhance the stability of the remaining rock mass. To implement this, a compact and flexible self-propelled drilling rig with adjustable drilling angles is proposed, allowing greater precision and adaptability to the variable geometry of the ore body.

Second, it is necessary to replace the manual ore discharge method, which is unsuitable for the Vi Kem Copper Mine conditions. In the sublevel caving method, the blasted ore fragments are significantly larger than those obtained through shrinkage stoping, and manual handling of such material is unsafe and inefficient. Furthermore, after blasting, personnel are prohibited from entering the blasted voids to deal with oversized ore due to safety concerns. Therefore, a mechanized ore discharge system should be introduced, capable of handling large rock fragments and ensuring continuous, safe ore flow from the drawpoints without manual intervention.

## 3. Results and discussion

### 3.1. Justification and selection of design area

Based on the Vi Kem Copper Mine's current opening-up and preparation diagram, the selected design area includes ore blocks 13-122-1.1, 14-122-1.1, and 2-121-1.1 of the TQ1.1 ore body. These ore blocks are soon scheduled for extraction and have been accessed through the main haulage crosscut at level +30, extending from the shaft station to the TQ1.1 ore body. The corresponding ventilation system for this area is established through a ventilation crosscut at level +150 and a ventilation incline connecting levels +110 to +150.

The selected reserve blocks are distributed within the elevation range from –50 to +110, between the exploration lines T.47 and T.50-1. Therefore, the choice of these blocks for design calculation and trial implementation is entirely consistent with the current mine opening-up layout and development plan.

All selected ore blocks belong to steeply dipping ore bodies, with an average dip angle of about 75° and ore thickness ranging from 2.98 to 3.29 m, averaging 3.1 m. The main geological and structural characteristics of these ore blocks are presented in Table 3. In contrast, the longitudinal cross-section of the reserve block in the selected design area is shown in Figure 2.

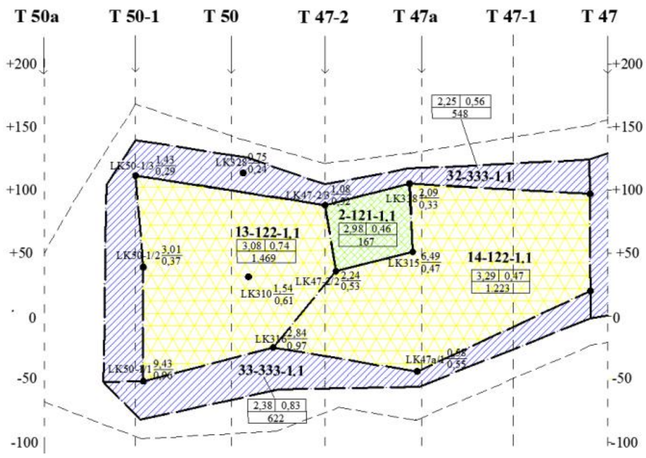



*Figure 2. Longitudinal cross-section of the reserve block in the selected design area*

The rock layer above the ore (roof rock). The rock block has a relatively stable geological structure, including Gneisbiotite, Granitogneis, 2-mica quartz schist, and other rocks interspersed. The layers are from 0.50m to 30m thick.

*Table 3. Characteristics of the selected ore blocks for design area*

| № | Ore body name | Ore block number | Ore block area, $m^2$ | Reserve thickness, m | Volume of the block, $m^3$ | Dry ore reserves, t | Copper content, % | Metal reserves, t |
|---|---|---|---|---|---|---|---|---|
| 1 | TQ1.1 | 2-121-1.1 | 3.691 | 2.98 | 10.981 | 36.348 | 0.46 | 167 |
| 2 | TQ1.1 | 13-122-1.1 | 19.373 | 3.08 | 59.697 | 197.599 | 0.74 | 1.469 |
| 3 | TQ1.1 | 14-122-1.1 | 24.109 | 3.29 | 79.387 | 262.770 | 0.47 | 1.223 |
| Total | | | 47.173 | | 150.066 | 496.717 | | 2.859 |

The rock blocks are affected by weak weathering and are cracked and broken in some places. The cracked and broken places have a generally poor geological structure. The rock in this layer contains local pressure water in cracks and holes, but it is poor and has a negligible impact on the passing construction.

The ore-bearing rock layer includes metamorphic rocks and micmatized Gneisbiotite. The rock is compressed, has minor cracking, has a granular structure, scales and a strip-like structure, and is solid in the block, but has some places with strong cracking and breaking.

The rock layer under the ore (floor rock) includes Gneisbiotite, Granitogneis, Amphibol, calcified limestone, and others. The rocks in this layer are usually solid, with few cracks, and have a durable geological structure. The average results of the analytical indicators are shown in Table 1.

The design selection area is currently prepared with a haulage level at +70 level and a ventilation level at +110 level. Both of these tunnels are dug along the ore body. The connection between the haulage level of +70 and the main haulage level of +30 is by a central incline from +30 to +70. The connection between the +110 and the ventilation level +150 is by a ventilation incline from the +110 to the +150 levels. Thus, the boundary division diagram of the mining layer has been formed, which is very suitable for the mining preparation work. The vertical height of the mining layer is 40m. The length in the direction of a strike line of the design ore block area is 350 m. The haulage and ventilation levels are dug with a dome-shaped cross-section, supported by anchors or without support. The height of the tunnel is 2.5 m, the width is 3.5 m, and its area is 7.5 m2. The current status of the haulage and ventilation levels of the design area is shown in Figure 3.

*Figure 3. Current status of the haulage and ventilation levels in the design area*

The preparation of the mining room at these ore blocks has not been carried out (the room incline has not been dug yet). Therefore, the preparation according to the proposed system diagram and mining technology is favorable and is not affected by the old mining technology diagram (the old mining technology diagram applied to this condition is the mining system of the ore shrinkage stoping, the ore extraction by drilling and blasting, ore recovery through the discharge door onto the trolley at the haulage level of +70 level). The longitudinal cross-section of the current status diagram of the preparation tunnels and the plan of the preparation tunnels of the design area are shown in Figures 4 and 5.

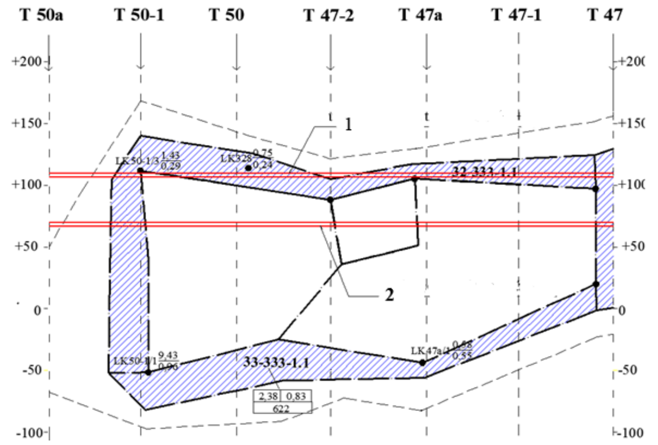



***Figure 4. Longitudinal cross-section of the current status diagram of the design area: 1 – ventilation level at +110; 2 – haulage level at +70***

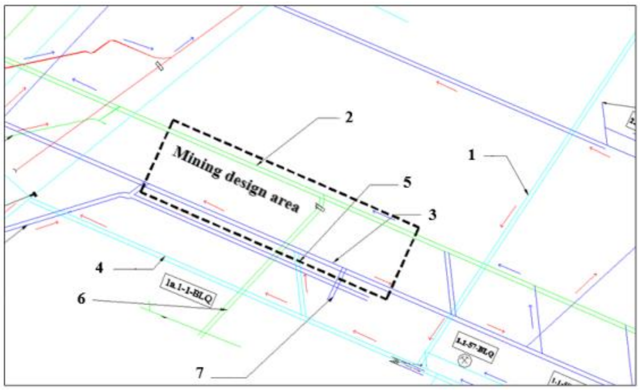



***Figure 5. Plan of the preparation tunnels in the design area: 1 – main haulage crosscut at +30 level to the TQ1.1 ore body; 2 – ventilation level at +110 of the TQ1.1 ore body; 3 – haulage level at +70 of the TQ1.1 ore body; 4 – haulage level at +30 of the TQ1.1 ore body; 5 – central haulage brake incline; 6 – crosscut at +110 level from the TQ1.1 to the TQ1a.1 ore body; 7 – crosscut at +70 level from the TQ1.1 to the TQ1a.1 ore body***

### 3.2. Selecting a mining technology diagram suitable for the conditions of the design area

As analyzed above, the system diagram and sublevel caving are selected in the design area to improve the mining technology diagram.

According to the distribution characteristics of the ore blocks in the design area, the length in the direction of a strike line is 350 m. To ensure the complete ore exploitation and not need to leave pillars to protect the tunnels, at level +110, a ventilation level is dug in the floor rock. At the same time, it is parallel and 10m away from the longitudinal level dug in the ore body. At level +70, a haulage level is also dug in the floor rock, about 10 m away from the ore body. The connection between the longitudinal level of the rock and the longitudinal level of the ore body is made by a cross-measure drift. In the center of the actual design area, an inclined tunnel is dug from level +70 to +110, connecting the longitudinal level of the rock at level +70 to the longitudinal level of the rock at level +110 to serve the transportation of equipment between the stratified longitudinal levels. The inclined tunnel is dug at a slope angle of 150 to facilitate equipment transportation. This tunnel is dug according to each stratified height. The height of each mining horizon is selected to correspond to the drilling equipment, expected to be 10-12 m.

To prepare the mining sublevel, from the +70/+110 inclined tunnel, the cross-measure drifts are dug into the ore body. The cross-measure drifts are dug at the corresponding high levels of the mining sublevel in the design area. When the cross-measure drifts meet the ore body, the longitudinal level of the mining sublevel is dug out towards the border of the design area. At the +70 longitudinal level of rock and the longitudinal level of the last mining sublevel, excavate the ore chute and cross-measure drifts from the longitudinal level of the ore body at level +70 to prepare for ore discharge during the mining process. The mining technology diagram is shown in Figure 6.

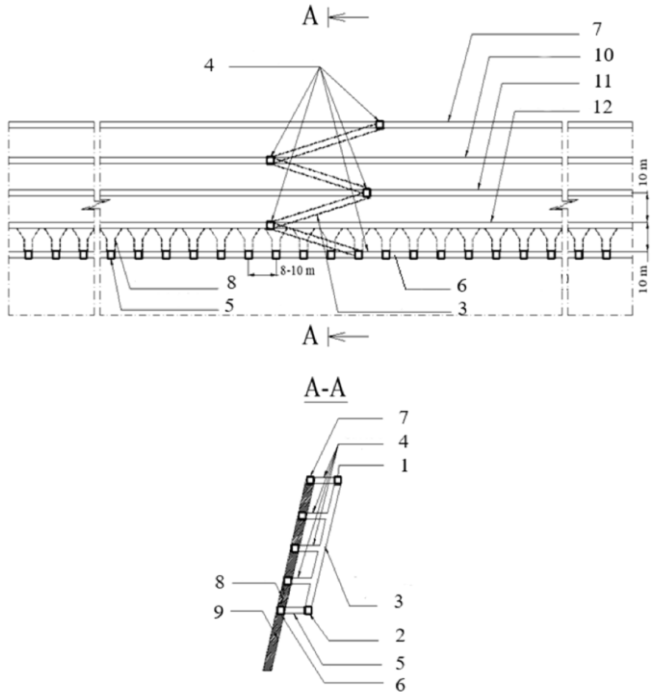



***Figure 6. Diagram of the preparation of the mining room in the design area: 1 – ventilation level at +110 (in floor); 2 – haulage level at +70 (in floor); 3 – inclined tunnel connecting +70 to +110 levels; 4 – cross-measure drifts; 5 – cross-measure drift (ore discharge); 6 – haulage level at +70 (in ore body); 7 – ventilation level at +110 (in ore body); 8 – ore chute; 9 – ore body; 10 – stratified longitudinal level at +100; 11 – stratified longitudinal level at +90; 12 – stratified longitudinal level at +80***

### 3.3. Calculation of the main parameters of the selected mining technology diagram

According to the adopted mining technology diagram, the ore body's geological characteristics and the mining area's corresponding technical parameters are determined as follows. The average ore body thickness is 3.12 m, and the average length of a cutting room is 10.4 m. The height of the mining stratification is set at 10 m, while the ore hardness ranges from f = 10 to 16, indicating a complex to tough rock type. The fundamental parameters of the ore drilling and blasting passport developed for the selected design area are presented in Table 4.

The preparation tunnels in the design area include inclined tunnels for transporting equipment from level +70 to +110, and longitudinal levels in the rock at level +70 and +110. The tunnels are stratified into longitudinal levels, cross-measure drifts, and the ore chute. In practice, the rock and ore in the mine area are excellent and stable.

After excavation, the tunnels do not need to be supported. In particular, only a few tunnel sections excavated through faults and weak geology need to be reinforced with steel or wood.

***Table 4. Basic parameters of the ore drilling and blasting passport***

| № | Names of basic parameters | Unit | Value |
|---|---|---|---|
| 1 | Mining room width | m | 3 |
| 2 | Mining room height | m | 10 |
| 3 | Length of one blasting in the cutting room | m | 10.4 |
| 4 | Unit explosive index | kg/m$^3$ | 1.61 |
| 5 | Distance between rows of drill holes | m | 1.9 |
| 6 | Number of rows of holes drilled per blasting | row | 04 |
| 7 | Number of holes drilled in the cutting face | holes | 6 |
| 8 | Number of holes drilled in a blasting section | holes | 24 |
| 9 | Amount of explosives per explosion | kg | 502 |

The results of calculating the number of holes drilled on the prepared tunnel face are shown in Table 5. The drilling and blasting passport are shown in Figure 7, and some economic and technical indicators of the prepared tunneling technology are shown in Table 6.

***Table 5. Number of holes drilled on the prepared tunnel face***

| № | Names of basic parameters | Symbol | Unit | Value |
|---|---|---|---|---|
| 1 | Average excavation cross-sectional area | $S_d$ | m$^2$ | 6.25-8.4 |
| 2 | Rock strength coefficient | $f$ | - | 8-10 |
| 3 | Unit explosive index | $q$ | kg/m$^3$ | 2.35 |
| 4 | Tunnel width | $B$ | mm | 2.640 |
| 5 | The coefficient depends on the shape of the tunnel | $C$ | - | 3.86 |
| 6 | Distance between border boreholes | $r_b$ | mm | 45 |
| 7 | Explosive cost per meter boreholes | $\gamma_0$ | kg/m | 0.20 |
| 8 | Explosive cost per meter of boreholes | $\gamma$ | kg/m | 0.45 |
| 9 | Explosive density | $\Delta$ | kg/m$^3$ | 1100 |
| 10 | Volume of explosives in 1m boreholes | $V$ | m$^3$ | 0.0008 |
| 11 | Explosive charge coefficient | $a$ | - | 0.60 |
| 12 | Explosive compaction coefficient in the boreholes | $b$ | - | 1.00 |
| 13 | Diameter of explosive stick | $d$ | m | 0.032 |
| 14 | Total number of calculated drill holes | $N^{tt}{}_g$ | holes | 37 |
| - | Number of holes drilled at the side of the tunnel | $N_b$ | holes | 17 |
| - | Number of holes drilled, cut, and the floor of the tunnel | $N_{r,f}$ | holes | 20 |
| 15 | Total number of drilled holes arranged on the tunnel face | $N_g$ | holes | 38 |
| - | Number of holes drilled at the side of the tunnel | $N_b$ | holes | 14 |
| - | Number of empty boreholes | $N_{tr}$ | holes | 1 |
| - | Number of drilled cut holes | $N_r$ | holes | 6 |
| - | The number of holes drilled to break the floor and create water channels | $N_{f,n}$ | holes | 17 |

***Table 6. Summary of technical and economic indicators for tunnel preparation***

| № | Names of basic parameters | Unit | Value |
|---|---|---|---|
| 1 | Cross-sectional area of tunnel excavation | m$^2$ | 6.25-8.4 |
| 2 | Usable cross-sectional area of the tunnel | m$^2$ | 5.28 |
| 3 | Perimeter of tunnel | m | 6.1 |
| 4 | Tunnelling step | m/cycle | 1.53 |
| 5 | Number of shifts per day and night | shift | 3 |
| 6 | Number of shifts completing a cycle | shift | 1.5 |
| 7 | Cycle completion coefficient | - | 0.85 |
| 8 | Tunnel digging speed per day and night | m/d-n | 2.6 |
| 9 | Tunnel-digging speed per month | m/mon | 68 |
| 10 | Number of people working day and night | worker | 15 |
| 11 | Direct labour productivity | m/w-sh | 0.17 |

The design calculations show that 38 blast holes are arranged on the tunnel face, including 14 side, 6 cut, 17 floor, and 1 empty hole. The tunnel's cross-sectional area ranges from 6.25 to 8.4 m$^2$, with a rock strength coefficient of 8-10 and a unit explosive index of 2.35 kg/m$^3$. The explosive density is 1100 kg/m³, and the average explosive consumption per meter of borehole is 0.45 kg/m.

The usable tunnel area reaches 5.28 m$^2$, and the excavation perimeter is 6.1 m. Each drilling-and-blasting cycle advances the face by 1.53 m, taking about 1.5 shifts, with a completion coefficient of 0.85. With three shifts per day, the tunneling rate is 2.6 m/day or 68 m/month, achieved by 15 workers with a productivity of 0.17 m per worker-shift.

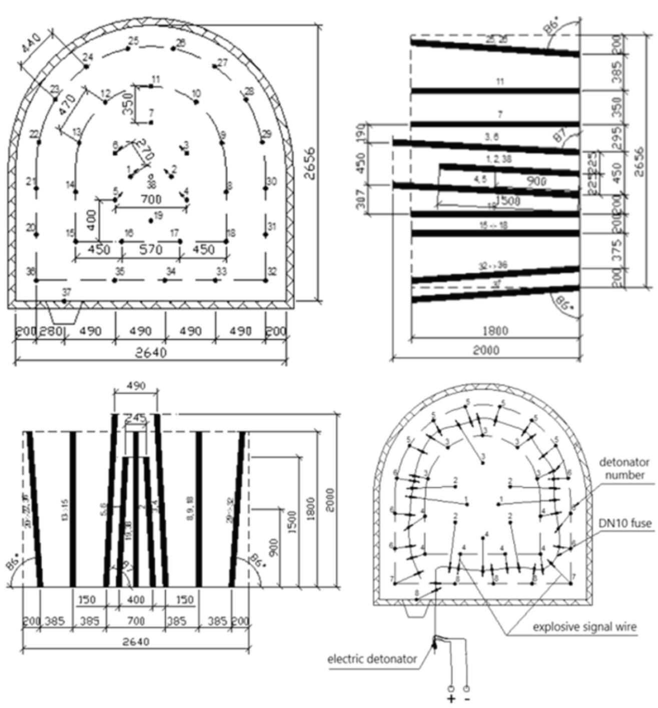


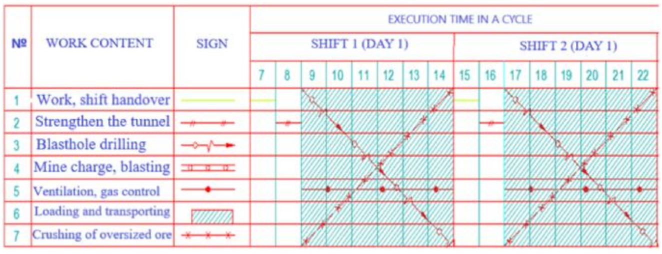

***Figure 7. Diagram of the arrangement of drilled holes on the face and diagram of the detonator cap connection***

The diagram illustrates the layout of the blast holes on the tunnel face and the corresponding connection scheme of detonator caps. The figure shows the numbering of boreholes, spacing parameters, and sequence of detonation, ensuring controlled rock fragmentation and uniform advance of the mining face.

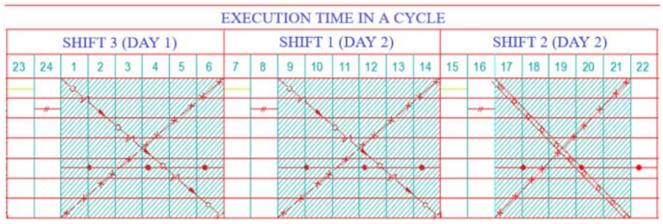
## 3.4. Production organization in the ore mining technology diagram

The production process in the selected ore mining technology diagram is organized in three working shifts daily, each lasting eight hours. The primary operations performed within one complete production cycle include: inspection and reinforcement of the stratified longitudinal level; breaking of oversized ore fragments; drilling of blast holes; loading and blasting operations followed by ventilation; loading of ore onto haulage trucks; ore transportation; and rock processing to create additional mining space.

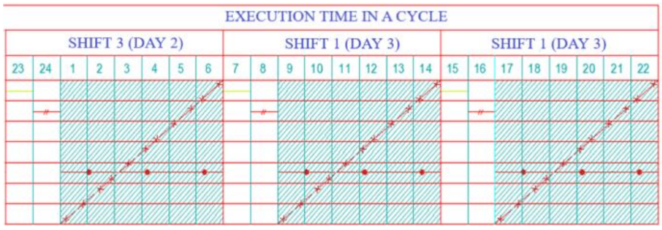
The production cycle is organized over eight working shifts based on the total workload and operational sequence. The design of the production cycle organization and the corresponding labor and equipment arrangement are illustrated in Figures 8 and 9.




(a)

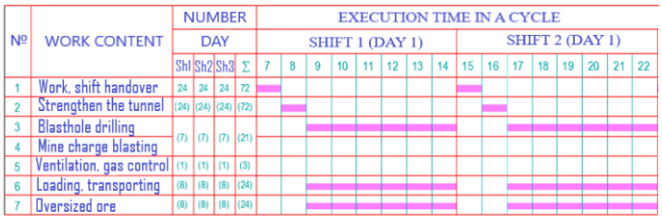



(b)

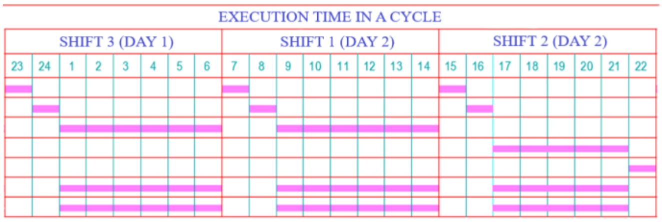



(c)

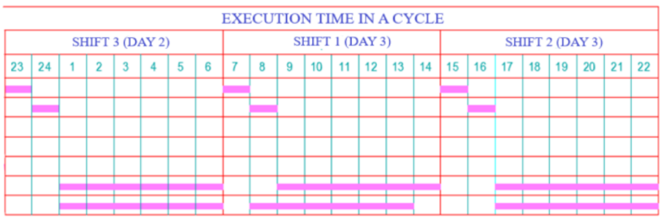
***Figure 8. Production cycle organization chart in the mining face: (a) – tasks performed during shifts 1 and 2 of day 1; (b) – tasks performed during shift 3 of day 1 and shifts 1 and 2 of day 2; (c) – tasks performed during shift 3 of day 2 and shifts 1 and 2 of day 3***




(a)




(b)




(c)

***Figure 9. Human resource arrangement chart in the mining face: (a) – number of personnel assigned during shifts 1 and 2 of day 1; (b) – number of personnel assigned during shift 3 of day 1 and shifts 1 and 2 of day 2; (c) – number of personnel assigned during shift 3 of day 2 and shifts 1 and 2 of day 3***

### 3.5. Calculation of economic and technical indicators of the mining technology diagram

#### 3.5.1. Ore output in a blasting cycle

The ore output in a single blasting cycle is determined using the following formula:

$$Q_c = B \cdot r \cdot h \cdot \gamma_t \cdot k_{rec} \cdot k_{conv}, \tag{1}$$

where $B$ is width of the mining room (3.0 m); $r$ is depth of one blasting round (10.4 m); $h$ is height of the mining room (10 m); $\gamma_t$ is volumetric (bulk) weight of ore (3.3 t/m$^3$); $k_{rec}$ is ore recovery coefficient, reflecting extraction efficiency (0.9); $k_{conv}$ is conversion coefficient accounting for losses and dilution; since the total loss and dilution ratio is 5% (1.0).

Substituting these values into Equation (1), the ore output per blasting cycle is 926.6 tons.

The daily ore output is determined as:

$$Q_d = \frac{Q_{ck}}{n_{cyc}} \cdot n_{shift} \cdot k_{cyc}, \tag{2}$$

where: $n_{cyc}$ is the number of shifts required to complete one full cycle (8 shifts); nshift is the number of shifts operated per day (3 shifts); kcyc is the cycle completion coefficient, representing the effective completion rate (0.8).

Substituting the values into Equation (2), the ore output per day and night equals 278 tons.

Monthly ore production is calculated by:

$$Q_m = Q_d \cdot n_{work}, \tag{3}$$

where: $n_{work}$ is the number of working days per month (25 days).

Substituting the values into Equation (3), the monthly ore output is 6.95 thousand tons.

The annual production capacity of the design area is calculated using:

$$Q_y = Q_m \cdot n_{month} \cdot k_{tr}, \tag{4}$$

where: $n_{month}$ is the number of working months in a year (12 months); ktr is the coefficient accounting for time spent on transition between stopes or mining areas (0.95).

Substituting the values into Equation (4), the annual ore output of the design area is approximately 79.2 thousand tons per year. The economic and technical indicators of the applied mining technology diagram in the selected design area are summarized in Table 7.

***Table 7. Leading economic and technical indicators***

| № | Name of indicators | Unit | Value |
|---|---|---|---|
| 1 | Width of mining room | m | 3 |
| 2 | Height of the mining room | m | 10 |
| 3 | Volumetric weight of ore | ton/m$^3$ | 3.3 |
| 4 | Depth of one blasting of a cycle | m/ck | 10.4 |
| 5 | Number of shifts operating a day and night | shift | 3 |
| 6 | Number of shifts completing a cycle | shift | 8 |
| 7 | Working time during the year | day | 300 |
| 8 | Time utilization coefficient in the year | - | 0.85 |
| 9 | Ore recovery coefficient in the mining room | - | 0.95 |
| 10 | Ore output in a blasting cycle | ton | 926.6 |
| 11 | Ore mining coefficient | - | 0.90 |
| 12 | The largest annual ore output | ton/year | 79.230 |
| 13 | Number of workers directly employed in the mining area | worker | 72 |
| 14 | Direct labor productivity | ton/worker | 12.8 |
| 15 | Loss and impoverishment rates due to drilling and blasting technology | % | 5.0 |

### 3.6. Discussion

This research proposes improvements to the mining technological diagram to enhance the efficiency of ore extraction at the Vi Kem Copper Mine.

For the existing ore mining technology diagram, the study suggests optimizing the ore recovery system by using ore chutes and branch tunnels, and introducing wheel loaders to improve loading efficiency, labor safety, and the handling of oversized ore. This approach eliminates the current manual ore discharge method through small discharge doors, thereby addressing safety risks and preventing blockages caused by large rock fragments. As a result, it reduces the volume of oversized ore that must be crushed in the stope and shortens the preparation time for subsequent cutting cycles.

The study proposes adopting an ore sublevel caving mining technological diagram for medium-thick ore bodies and unprepared tunnel sections, where applying the shrinkage stoping method is difficult. In this system, the workers' positions are located safely within the stratified longitudinal level, enabling the mechanization of drilling operations, facilitating the movement of equipment between sublevels, improving ventilation efficiency, and increasing ore discharge capacity while reducing the overall volume of inclined tunnels.

For thick ore bodies, it is recommended to apply the layered sublevel caving method for lens-shaped ore blocks, with ore transportation directly along stratified longitudinal levels. For ore bodies of significant strike length, the sublevel caving diagram should be further optimized by incorporating inclined tunnels for equipment transfer, minimizing the excavation of inclined tunnels, and improving ore recovery in the cutting rooms through the use of ore chutes and mechanized excavator loading. Compact jumbos for drilling blast holes from upper to lower sublevels ensure high operational flexibility and safety.

The proposed improvements are technically and economically feasible. They enable more mechanization across mining stages, increasing productivity, mine efficiency, ventilation capacity, and labor safety. Considering the current operational conditions at the Vi Kem Copper Mine, the modernization of the mining technological system is both timely and necessary. The improved technological diagram remains consistent with the fundamental layout of the existing system but optimizes each component to maximize mechanization and operational efficiency.

The analysis confirms the high feasibility of the proposed technological improvements. Calculations show that the mining capacity per cutting room can be increased by 1.76 times, while labor productivity can be enhanced nearly threefold compared with the current system. The new approach eliminates the need for ore discharge through steep ore chutes, which often causes production delays and safety hazards. Drilling is performed by self-propelled rigs in a top-down direction, ensuring operational convenience and safety. Since the working face is confined within the stratified longitudinal level, temporary supports, manual ore crushing, and manual leveling of the stope floor are no longer required. During mining, the ore naturally accumulates in the shrinkage chamber. It is gradually discharged through an improved ore chute system, without requiring manual discharge doors, and loaded by an excavator onto transport vehicles, resolving the significant challenges in the ore handling and transportation stages.

## 4. Conclusions

The analysis of the current mining system at the Vi Kem Copper Mine showed that the existing technological diagrams, mainly ore shrinkage stoping and ore sublevel caving, require modernization to increase efficiency, safety, and mechanization under the mine's specific geological and technical conditions.

The proposed improvement of the ore shrinkage stoping technological diagram includes the introduction of mechanized drilling and floor-leveling systems, temporary support solutions, and hydraulic ore discharge mechanisms. These modifications enhance operational safety, reduce manual labor intensity, and minimize ore blockages and oversized rock formations. A sublevel caving mining diagram is proposed for medium-thick and unprepared ore bodies. This system ensures safe working conditions for personnel, allows for mechanized drilling and flexible equipment movement between sublevels, improves ventilation, and reduces the excavation volume of inclined tunnels.

A layered sublevel caving scheme is recommended for thick and lens-shaped ore bodies, integrating inclined tunnels for equipment transfer and direct ore transportation along longitudinal levels. This approach improves ore recovery, enables the use of small jumbos for drilling from upper to lower sublevels, and further increases mechanization.

Technical calculations confirm the feasibility of the improved technological diagrams. The mining capacity per cutting room increases by 1.76 times, and labor productivity rises nearly threefold compared with the current system. Additionally, the proposed solutions enhance ventilation, reduce preparation time, and significantly improve labor safety. Implementing the improved mining technological diagrams at the Vi Kem Copper Mine is technically feasible, economically efficient, and timely. The proposed system maintains compatibility with the existing infrastructure while optimizing key operational elements to achieve higher productivity, safety, and sustainability of underground copper ore extraction.

## Author contributions

Conceptualization: TTV, PQL; Data curation: DTL, DTTV; Formal analysis: DTL, PQL; Investigation: DTTV, TTV; Methodology: TTV; Project administration: TTV, PQL; Resources: DTL, DTTV; Supervision: TTV; Validation: PQL, DTL; Visualization: DTTV; Writing – original draft: TTV; Writing – review & editing: TTV, PQL. All authors have read and agreed to the published version of the manuscript.

## Funding

This research received no external funding.

## Acknowledgements

Hereby, the authors express their gratitude to the staff of the Department of Engineering and Technology of the Vinacomin – Minerals Holding Corporation for creating favorable conditions for data collection and field survey. At the same time, the authors also thank the Editors of the Engineering Journal of Satbayev University, as well as anonymous reviewers for useful suggestions and recommendations taken into consideration during revision.

## Conflicts of interest

The authors declare no conflict of interest.

## Data availability statement

The original contributions presented in this study are included in the article. Further inquiries can be directed to the corresponding author.

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# Вьетнам, Лаокай провинциясы, Ви Кем мыс кенішінде кен денелерін игерудің технологиялық схемасын жетілдіру бойынша шешімдер

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**Аңдатпа.** Кен денелерінің пайда болу жағдайларына байланысты технологиялық даму схемасын жобалау және таңдау қиын міндет болып табылады, өйткені ол көптеген факторларға байланысты. Мұндай схемалар өндіріс процесінің тиімділігі мен қауіпсіздігін қамтамасыз етуі керек. Сондықтан Vinacomin-Minerals Holding Corporation корпорациясының жерасты кеніштерінің заманауи өндірістік талаптарын қанағаттандыру үшін кен денелерін өндірудің технологиялық схемасын жетілдіруге бағытталған шешімдерді ұсыну қажет. Тау-кен жұмыстарының қолданыстағы жоспары мен мыс кенішінің жобалық технологиялық схемасы негізінде авторлар зерттеу, талдау және бағалау жүргізді, олардың нәтижелері бойынша кен денелерін игерудің жетілдірілген схемасын ұсынды және жобалық есептеулер жүргізу үшін кеніштің учаскесін таңдады. Жұмыста деректерді жинау, талдау және синхрондау әдістері, далалық зерттеулер және эксперименттік учаскенің теориялық есептеулері қолданылады. Ұсынылған шешімдердің іске асырылу деңгейі жоғары: технологиялық операцияларды механикаландыру, өндірістің өнімділігі мен тиімділігін арттыру, желдетуді жақсарту және еңбек қауіпсіздігін арттыру мүмкіндігі қамтамасыз етіледі. Жетілдірілген технология тұтастай алғанда бұрынғы схеманың құрылымын сақтайды, бірақ әрбір элемент механикаландыруды қолдануды кеңейту үшін оңтайландырылған, бұл өндірістік тиімділік пен қауіпсіздіктің өсуін қамтамасыз етеді. Техникалық есептеулер көрсеткендей, әр камераның өндірістік қуаты 1.76 есе, ал жұмыс өнімділігі қолданыстағы даму схемасымен салыстырғанда үш есе артады.

***Негізгі сөздер:*** *тау-кен технологиялық схемасы, жетілдіру, кен өндіру, кен денесі, Ви Кем, Вьетнам.*

# Решения по совершенствованию технологической схемы разработки рудных тел на медном руднике Ви Кем, провинция Лаокай, Вьетнам

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**Аннотация.** Проектирование и выбор технологической схемы разработки в зависимости от условий залегания рудных тел представляет собой сложную задачу, поскольку она зависит от множества факторов. Такие схемы должны обеспечивать эффективность и безопасность производственного процесса. Поэтому необходимо предложить решения, направленные на совершенствование технологической схемы добычи рудных тел, чтобы удовлетворить современные производственные требования подземных рудников корпорации Vinacomin – Minerals Holding Corporation. На основе действующего плана горных работ и проектной технологической схемы медного рудника Ви Кем авторы провели обследование, анализ и оценку, по результатам которых предложили усовершенствованную схему разработки рудных тел и выбрали участок рудника для проведения проектных расчётов. В работе применены методы сбора, анализа и синтеза данных, полевые обследования и теоретические расчёты экспериментального участка. Предложенные решения обладают высокой степенью реализуемости: обеспечивается возможность механизации технологических операций, повышение производительности и эффективности добычи, улучшение вентиляции и повышение безопасности труда. Усовершенствованная технология в целом сохраняет структуру прежней схемы, однако каждый элемент оптимизирован для расширения применения механизации, что обеспечивает рост производственной эффективности и безопасности. Технические расчёты показывают, что производственная мощность каждой камеры увеличивается в 1.76 раза, а производительность труда – почти в три раза по сравнению с действующей схемой разработки.

***Ключевые слова:*** *горнотехнологическая схема, совершенствование, добыча руды, рудное тело, Ви Кем, Вьетнам.*

**Publisher's note**

SATBAYEV UNIVERSITY
Engineering Journal of Satbayev University
*Volume 147 (2025), Issue 5, 29-34*

https://doi.org/10.51301/ejsu.2025.i5.04

# Optimizing Kazakhstan's water budget through subsurface floodwater recharge

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**Abstract.** The study is dedicated to analyzing the economic efficiency of Managed Aquifer Recharge (MAR) systems as a method of sustainable water supply under the conditions of climate change and water resource scarcity. The work explores the fundamental principles of MAR, its advantages in reducing water supply costs, minimizing water losses, improving resilience to extreme climatic conditions, and maintaining ecosystem balances. Special attention is given to the economic and social benefits that the use of MAR can provide to agriculture and local communities. The research methodology includes an analysis of existing practices, economic modeling, and an assessment of the impact of implementing MAR technologies at the regional level. The results demonstrate that MAR is an effective and economically advantageous strategy for water supply, promoting sustainable use of water resources and improving quality of life. The scientific novelty lies in justifying the economic feasibility of applying MAR systems in the context of climate change, while the practical significance of the study lies in the potential to implement the findings into real-world water resource management practices.

***Keywords:*** *Managed Aquifer Recharge, economic efficiency, water supply, climate change, water resources, sustainable development.*

Received: 29 March 2025
Accepted: 16 October 2025
Available online: 31 October 2025

## 1. Introduction

Central Asia (CA) is one of the most vulnerable regions in the world to the consequences of climate change and water scarcity. The region faces changes in precipitation patterns, increased frequency of extreme temperatures, and growing aridity, which negatively affect agriculture, ecosystems, and food security [1]. In Kazakhstan, the problem is exacerbated by a pronounced seasonal water imbalance: during spring, there is an excess of water due to snowmelt and floods, while in the summer months, the country suffers from a shortage of water resources [2]. Traditionally, the focus has been on surface water resources, while the potential of groundwater has been insufficiently utilized [3].

Historically, Kazakhstan and other countries in the region have actively used dams and reservoirs to regulate river flow, irrigate land, and manage flooding. However, the modern practice of dam construction and operation is showing increasing limitations and risks. Key issues include sediment deposition, which leads to a reduction in reservoir capacity by up to 1% annually [4], as well as significant environmental impacts on biodiversity [5]. Furthermore, the construction of large dams involves socio-economic costs, including population displacement and the loss of agricultural land [6]. Climate change exacerbates these problems, causing reservoirs to become less resilient to droughts and extreme precipitation events [7].

In this context, there is a need to explore more sustainable and adaptive approaches to water resource management. One such promising technology is Managed Aquifer Recharge (MAR), which involves collecting and storing surplus surface water underground for future use. Global experience, particularly research from Australia, demonstrates the high efficiency of MAR in agricultural regions with similar conditions. For example, MAR systems in the Little Para and Lockyer valleys contribute up to 8.9 million cubic meters of water annually, with an efficiency rate of up to 67% [8]. In addition to increasing available water reserves, MAR contributes to improved crop yields.

For Kazakhstan, MAR offers an opportunity to simultaneously reduce flood risks, accumulate strategic water reserves, and improve water quality through natural filtration in aquifers. Research conducted in the Zhambyl region has confirmed the presence of areas with high potential for MAR implementation, based on an analysis of soil-hydrogeological conditions [2].

In the context of intensifying climate change and growing demand for water resources, the MAR technology can ensure more sustainable water supply for agriculture, reduce evaporation losses, and minimize ecological risks. The implementation of MAR in Kazakhstan opens up prospects for the sustainable development of the agricultural sector and increased food security in the country.


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Engineering Journal of Satbayev University. eISSN 2959-2348. Published by Satbayev University

The goal of this study is to assess the economic feasibility and benefits of using Managed Aquifer Recharge (MAR) technologies as a solution for sustainable water resource management in Kazakhstan. The research focuses on integrating international MAR experience and spatial modeling methods to identify regions where surplus surface water during flood periods can be stored underground for future agricultural use. The overall aim is to provide scientific support for integrating MAR into Kazakhstan's national water resource strategies and raise awareness of MAR as a promising tool for ensuring long-term agricultural and environmental sustainability.The scientific novelty of this research lies in the pioneering application of the economic efficiency assessment method for water storage through Managed Aquifer Recharge (MAR) technologies in Kazakhstan, where water scarcity, particularly in the southern and central regions, is an urgent issue.

## 2. Materials and methods

The methodological framework of this study focuses on a comparative analysis of international experience in implementing Managed Aquifer Recharge (MAR) technologies, with an emphasis on their economic efficiency and applicability to the conditions of Kazakhstan. The study explores the experiences of Australia, Europe, and the USA, where MAR systems have been successfully integrated into agricultural water supply. Special attention is given to economic indicators cost-benefit ratios, the efficiency of using existing infrastructure, and the impact on crop yield growth.

The implementation of Managed Aquifer Recharge (MAR) technologies in Australia, Europe, and the USA has demonstrated significant economic benefits, particularly in agriculture and urban water management.

In Australia, the active development of MAR began after the 1994 drought, which triggered a major reform in the water sector. The creation of the National Water Plan and the Water Resources Commission contributed to improving the efficiency of water management, reducing water supply costs, and increasing resilience to droughts. Research shows that MAR projects in the Lockyer Valley and the Little Para area achieved up to 67% system efficiency, with annual replenishment volumes ranging from 0.6 to 8.9 million $m^3$ [8]. These systems not only enhance water security but also contribute to increased agricultural productivity, especially in viticulture [9].

In Europe, countries such as Spain, and in comparative terms, Israel, have focused on MAR. For example, in Israel, around 90% of wastewater is recycled and mainly used for agricultural needs, significantly reducing dependence on freshwater and lowering water supply costs [10]. In Spain, the experience of using infiltration basins and aquifer recharge demonstrated that MAR directly contributes to the economic stability of irrigated agriculture, mitigating the effects of droughts and optimizing water use [11].

In the United States, MAR systems are actively supported by the government through subsidies and special programs. This is particularly evident in California, where Flood-MAR projects aim to capture excess surface water during floods and store it in underground aquifers. These projects not only reduce flood risks but also provide long-term economic benefits through stable water supply for agriculture and reduced costs associated with building expensive surface reservoirs [12].

Managed Aquifer Recharge (MAR) is increasingly seen as an effective tool for sustainable water resource management, particularly under the growing pressure on groundwater. Research in this area focuses on both the economic efficiency and institutional feasibility of MAR projects. The analysis of scientific literature highlights three key approaches to evaluating such projects: through the lens of institutional regulation, economic feasibility at global and local levels, and the assessment of social and environmental benefits.

Reznik emphasize the importance of the institutional environment for the successful implementation of MAR in California, USA. Based on hydroeconomic modeling, the authors demonstrate that the effectiveness of MAR projects directly depends on water rights, coordination between agencies, and management flexibility [13]. The introduction of laws such as the Sustainable Groundwater Management Act (SGMA) creates new incentives for investment in MAR, but in the absence of institutional cooperation, even technically and economically viable projects may be inefficient. This study highlights the role of institutional structures as a key factor in the sustainability of MAR systems.

Ross provides a comparative analysis of 21 MAR schemes in 15 countries, evaluating the levelized cost and the benefit-cost ratio (BCR) [14]. The author shows that systems using natural replenishment through infiltration basins or filtration from rivers exhibit the highest profitability. MAR systems using recycled water are costlier but also yield positive effects. Special attention is given to the unquantified ecological and social benefits, such as groundwater level preservation, water quality improvement, and energy savings. This approach expands the understanding of the cost benefits of MAR beyond direct economic gains.

Halytsia, in a study conducted in Poland, present a rare example of a local economic evaluation of a MAR project considering social and ecological aspects [15]. The authors apply a combined approach: classic cost-benefit analysis (CBA), contingent valuation for calculating the willingness of the population to pay for water quality preservation, and sensitivity analysis to account for uncertainties. The results show that even considering social discounting and potential risks, expanding MAR systems remains beneficial. This study is valuable as an example of a comprehensive approach, including indirect and non-material benefits, and provides a foundation for future policies in sustainable water supply.

In summary, the research underscores that successful MAR implementation depends not only on hydrological and technical conditions but also on institutional support, economic viability, and community engagement. The studies demonstrate a wide range of evaluation methods – ranging from modeling and global meta-analyses to localized practical cases.

Thus, the international experience confirms that MAR systems are an economically efficient solution, contribute to increased agricultural productivity, and support sustainable water resource management. This makes MAR particularly relevant for arid and semi-arid regions like Kazakhstan.

## 3. Study area

The Chilik River (also known as Shelek) is located in southeastern Kazakhstan, within Almaty Region. It originates on the southern slopes of the Zailiysky Alatau range, in the zone of glaciers and eternal snow, and belongs to the Lake Balkhash basin. The river’s length is approximately 245 km, and its catchment area covers about 5090 km$^2$ [16]. The basin's relief varies from high-mountainous upstream reaches to lowland and foothill terrain in the middle and lower courses, determining both the river's flow regime and the natural conditions affecting land use and water management [17].

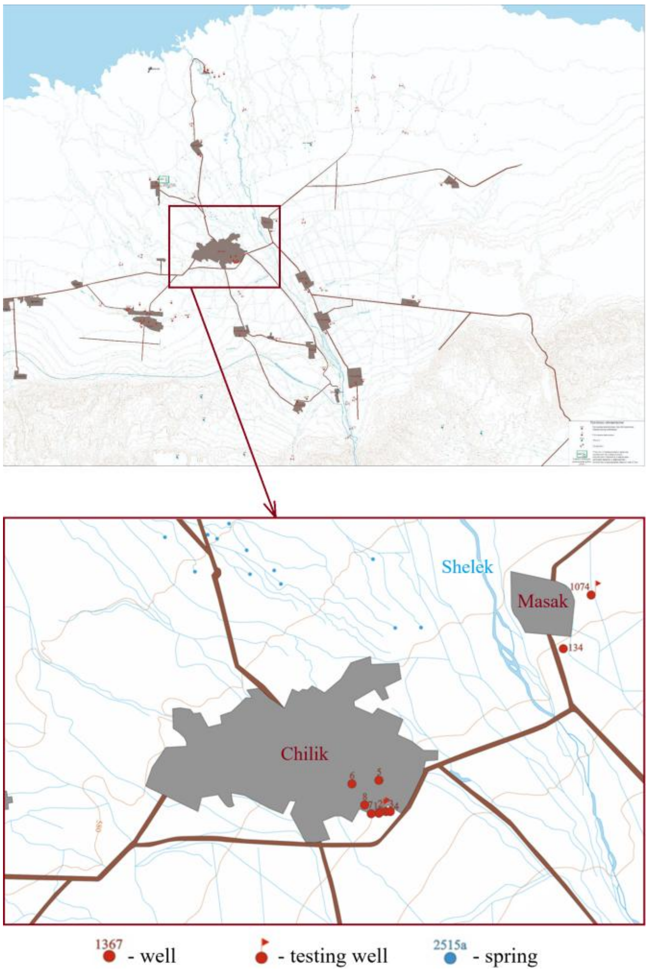


***Figure 1. Maps of actual material of Chilik groundwater deposit***

The Chilik's flow is primarily fed by glacial and snowmelt. Peak discharge occurs in the spring-summer period (May-July), coinciding with intensive agricultural water use. Mean annual discharge ranges between 38 and 42 m$^3$/s, yielding an annual runoff volume of about 1.2-1.3 km$^3$ [18]. Mineralization is low especially in the upper reaches making the water suitable for drinking, household, and irrigation needs [19].

The Chilik River is a key water source for irrigated agriculture in the Chilik Valley, particularly the Shelek Oasis. Water is supplied via an extensive irrigation network, including the main Chilik Canal [20]. Major crops include maize, vegetables, fodder crops (alfalfa), as well as fruit orchards and greenhouse farming [21]. Settlements such as Shelek, Enbek, and Bayseit depend heavily on the river for their livelihoods.

To evaluate the economic efficiency of a hypothetical Managed Aquifer Recharge (MAR) project in the Chilik Basin, we adopted indicative parameters based on empirical data, regional analogies, and international methodologies proposed by Ross [13], Maliva (2014) [2], and Halytsia et al. (2022) [15].

An investment of USD 500000 covers design, geotechnical surveys, construction of infiltration fields, intake channels, pumping stations, and filtration facilities. This estimate aligns with typical MAR project costs in regions with comparable hydro-climatic conditions. Annual operating expenses are set at USD 20000, covering energy, maintenance, staffing, and routine water-quality monitoring. Such figures are corroborated by data from rural water-supply projects and regional irrigation modernization programs in Kazakhstan.

The project assumes 350000 m$^3$/year of recharge. This is based on spring flood runoff in the Chilik, of which at least 0.5% can be sustainably diverted for infiltration effectively utilizing surplus surface water while minimizing evaporation and downstream losses.

Total annual benefits are estimated at USD 122500, using a tariff of USD 0.35/m$^3$. This figure reflects not only the market value of drinking water but also indirect benefits: reduced water shortages, improved quality, enhanced agricultural resilience, and ecosystem services (soil degradation prevention). A discount rate of 5% recommended by the European Commission for water-resource projects with a 30-year planning horizon is applied. These parameters follow international cost-benefit analysis standards and are widely used in GIZ, UNDP, and EU Green Deal-supported projects.

## 4. Results and discussion

Hypothetical efficiency calculation of a MAR system in Almaty region (Chilik River basin) using Ross methodology [14] represented on Table 1.

***Table 1. Input conditions (hypothetical MAR project on the Chilik River)***

| Parameter | Source/Notes |
|---|---|
| Capital expenditure ($C_{cap}$), USD | Infiltration fields, intake structures, filtration setup [14] |
| Annual operating expenditure ($C_{op}$), USD | Pump maintenance, power, staffing, monitoring |
| Annual recharge volume ($V_{annual}$), m$^3$ | Spring flood runoff; 0.5% of ~70-80 million m$^3$ seasonal flow |
| Annual economic benefit ($B_{annual}$), USD | 350 000 m$^3$ × USD 0.35/m$^3$ + ecosystem services |
| Discount rate ($r$), % | Standard for long-term water projects (EU Commission) |
| Project lifetime ($n$), years | Typical lifespan of water-infrastructure [15] |

Calculation of capital recovery factor ($CRF$):

$$CRF = \frac{r(1+r)^n}{(1+r)^n - 1}. \tag{1}$$

This factor annualizes the capital cost over the project lifetime. After substituting the parameters, the calculated value is $CRF \approx 0.065$, which means that about 6.5% of the capital investment is recovered annually over the project lifetime.

Levelised cost of water (LCW):

$$LCW = \frac{C_{cap} \cdot CRF + C_{op}}{(V_{annual})}. \tag{2}$$

Based on these parameters, the levelised cost of water equals $LCW \approx 0.15$ USD per m$^3$, which represents the average cost of recharging or producing one cubic meter of water over the project's lifetime.

Benefit-cost ratio (BCR):

$$BCR = \frac{B_{annual}}{\left(C_{cap} \cdot CRF \cdot C_{op}\right)}. \quad (3)$$

The resulting value is $BCR \approx 2.0$, which means that the annual economic benefits are approximately twice the annualized costs. Economic Conclusion: $BCR > 1$ indicates the MAR project is economically viable and sustainable under these assumptions.

The project provides several additional advantages. It effectively utilizes spring floodwater that would otherwise be lost through runoff, thereby improving the overall efficiency of water resource management. At the same time, it enhances groundwater quality through the process of natural filtration as the recharged water percolates through soil and aquifer layers. The system also strengthens agricultural resilience, as it reduces farmers' dependence on irregular surface water supplies and mitigates the risks associated with seasonal droughts. Finally, the project creates opportunities for research and education, encouraging collaboration with local colleges and research institutes in studying and improving Managed Aquifer Recharge (MAR) technologies.

***Table 2. Justification of parameter estimates***

| Parameter | Value | Justification / source |
|---|---|---|
| Capital expenditure, USD | 500000 | Typical cost for infiltration-field construction, intakes, and pipelines; comparable to GIZ Central Asia and India cases [14, 15] |
| Operating expenditure, USD/year | 20000 | Includes power, staffing, pump and filter maintenance; aligns with rural water-supply budgets in Kapchagay & Turkestan regions |
| Annual recharge volume, m$^3$ | 350000 | 0.5% of spring runoff (10-15 m$^3$/s over 60 days ≈ 7-80 million m$^3$) |
| Annual economic benefit, USD | 122500 | Based on USD 0.30-0.35/m$^3$ local tariff plus 5-10% bonus for ecosystem and social benefits |
| Discount rate, % | 5 | In line with EU Commission, GIZ, and UNDP cost-benefit analysis standards |
| Project lifetime, years | 30 | Standard life cycle for water and geotechnical infrastructure |

For the efficiency calculation of the hypothetical MAR system in Almaty Region (Chilik River basin), we used parameters based on international best practice and adapted to the Kazakh context. A capital investment of USD 500000 reflects the average cost of constructing infiltration facilities including geotechnical surveys, earthworks, and connection to the existing water supply network and is comparable to similar projects in India, Poland, and Central Asia (GIZ) [13, 15]. Annual operating expenses of USD 20000 cover electricity consumption, equipment depreciation, operator salaries, and periodic monitoring. Similar figures appear in the budgets of Kazakh water-supply projects in the southern regions (Kapchagay, Turkestan Region).

The annual recharge volume is assumed to be 350000 m$^3$/year, based on estimates of spring floodflows in the Chilik River which average 10-15 m$^3$/s over two months. Even diverting just 0.5% of this resource makes MAR implementation feasible.

Economic benefit is calculated using the local drinking-water tariff (≈ 0.30-0.35 USD/m$^3$) plus an allowance for quality and reliability improvements. The analysis also applies the European Commission's recommended 5% discount rate and a standard 30-year infrastructure lifespan.

A comparative analysis highlighted both the advantages and the current limitations of various MAR implementation models in Kazakhstan, the USA, and Europe. Global experience demonstrates MAR's high economic efficiency in agriculture and its resilience to climate-driven risks.

Kazakhstan's Potential and Local Case Studies Modeling and field studies indicate that Kazakhstan has significant potential for MAR, especially in arid zones and areas with pronounced seasonal water imbalances. A regional study in the Zhambyl Region identified soils and hydrogeological conditions highly favorable to large-scale MAR deployment.

The next research step involves conducting field measurements of flow rates and water quality in the Chilik River basin. These data will make it possible to transition from hypothetical estimates to a calibrated economic efficiency model for the Managed Aquifer Recharge (MAR) system. The planned activities include measuring seasonal discharge variations in cubic meters per second, assessing the suitability of floodwater for infiltration based on its chemical composition and turbidity, and characterizing the aquifer properties in potential recharge zones to ensure accurate modeling and practical applicability.

Collecting such empirical data will make it possible to perform a correlation analysis between natural variables such as spring runoff volume, flood duration, and mineralization levels and the modeled economic indicators, including the levelised cost, benefit-cost ratio, and discounted returns. This approach not only enhances the accuracy of the assessment but also adapts the model to the specific hydrogeological conditions of the basin. Consequently, real hydrometric data from the Chilik River will provide a solid foundation for validating the proposed MAR model, refining water availability estimates, reducing uncertainty, and identifying the main factors that determine project profitability. Given the climatic and hydrological variability of the region, this method ensures both scientific rigor and practical applicability of the MAR economic model for the Almaty Region and other areas with similar conditions.

The study will also investigate how integrating Managed Aquifer Recharge (MAR) systems with terrain-based digital elevation models (DEMs) and field data can optimize water capture and underground storage, reduce flood risks, and enhance overall climate resilience. The primary goal of the project is to develop practical solutions for sustainable water-resource management in Kazakhstan's semi-arid regions, where seasonal water scarcity and climate variability create significant challenges. By drawing on international MAR case studies from Spain to Australia the research aims to formulate evidence-based recommendations for adapting and scaling MAR technologies across Kazakhstan and the broader Central Asian region.

## 5. Conclusions

The hypothetical analysis of the economic efficiency of implementing a Managed Aquifer Recharge (MAR) system in the Chilik River basin shows that such projects have high potential for profitability and resilience under the conditions of Kazakhstan's southern regions. The calculated parameters, based on international methodologies and adapted to local conditions, indicate a cost level comparable to MAR projects carried out abroad, while the economic benefit expressed through drinking-water tariffs and additional ecosystem services yields a benefit-cost ratio (BCR) greater than 2.0, confirming the project's viability.

Using spring floodwaters as the infiltration source allows the surplus surface water to be managed efficiently, reducing losses from evaporation and runoff. At the same time, the project helps improve groundwater quality through natural filtration and strengthens water security and agricultural sector resilience.

Despite the hypothetical nature of the current assessment, the proposed methodology demonstrates the applicability of MAR systems to the Almaty Region's conditions. The next step should be to conduct field measurements of hydrological and geological parameters to refine the model's inputs, calibrate its outputs, and prepare a detailed techno-economic feasibility study based on actual data.

In this way, MAR technology represents a promising tool for sustainable water-resource management amid climate variability and growing water scarcity. If a pilot project in the Chilik River basin proves successful, this model could be scaled up and adapted to other regions of Kazakhstan with similar natural conditions.

**Author contributions**

Conceptualization: JS, RSA; Data curation: RSA, IKR; Formal analysis: ZAO, RSA; Funding acquisition: JS, RSA; Investigation: RSA, IKR; Methodology: RSA, IKR; Project administration: JS; Resources: JS; Software: ZAO, IKR; Supervision: RSA, JS; Validation: DKS, RSA; Visualization: DKS, RSA; Writing – original draft: RSA, DKS; Writing – review & editing: RSA, DKS. All authors have read and agreed to the published version of the manuscript.

**Funding**

This research was funded by grant financing of the project BR27197639 «Flood-drought mitigation innovations with managed aquifer recharge hydrogeological strategies for the Zhambyl, Almaty, Zhetysu, Abay, and East Kazakhstan regions» of the Ministry of Science and Higher Education of the Republic of Kazakhstan for 2024-2026.

**Acknowledgements**

The authors express their sincere gratitude to the editor and anonymous reviewers for their constructive comments and valuable suggestions, which have significantly improved the quality of this manuscript.

**Conflicts of interests**

The authors declare no conflict of interest.

**Data availability statement**

The original contributions presented in this study are included in the article. Further inquiries can be directed to the corresponding author.

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# Жер асты қабаттарында тасқын суларын жинау арқылы Қазақстанның су бюджетін оңтайландыру

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**Аңдатпа.** Зерттеу климаттың өзгеруі мен су тапшылығы жағдайында тұрақты сумен жабдықтау әдісі ретінде басқарылатын Сулы горизонтты толықтыру жүйелерінің (managed Aquifer Recharge, MAR) экономикалық тиімділігін талдауға бағытталған. Жұмыста MAR негізгі принциптері, оның сумен жабдықтау шығындарын азайтудағы, су шығынын азайтудағы, экстремалды климаттық жағдайларға төзімділікті арттырудағы және экожүйелердің тепе-теңдігін сақтаудағы артықшылықтары қарастырылады. MAR қолдану ауыл шаруашылығы мен жергілікті қауымдастықтарға бере алатын экономикалық және әлеуметтік артықшылықтарға ерекше назар аударылады. Зерттеу әдістемесі қолданыстағы тәжірибені талдауды, экономикалық модельдеуді және аймақтық деңгейде MAR технологиясын енгізудің әсерін бағалауды қамтиды. Нәтижелер MAR су ресурстарын тұрақты пайдалануға және өмір сүру сапасын жақсартуға ықпал ететін тиімді және үнемді су стратегиясы екенін көрсетеді. Ғылыми жаңалық-климаттың өзгеруі жағдайында MAR жүйелерін қолданудың экономикалық орындылығын негіздеу, ал зерттеудің практикалық маңыздылығы - алынған нәтижелерді су ресурстарын басқару тәжірибесіне енгізу мүмкіндігінде жатыр.

***Негізгі сөздер:*** *басқарылатын су өткізгіш қабаттарды қайта толтыру, экономикалық тиімділік, су жабдықтау, климат өзгерісі, су ресурстары, тұрақты даму.*

# Оптимизация водного бюджета Казахстана за счёт накопления паводковых вод в подпочвенных горизонтах

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**Аннотация.** Исследование посвящено анализу экономической эффективности систем управляемого пополнения водоносных горизонтов (Managed Aquifer Recharge, MAR) как метода устойчивого водоснабжения в условиях изменения климата и нехватки водных ресурсов. В работе рассматриваются основные принципы MAR, его преимущества в снижении затрат на водоснабжение, минимизации потерь воды, повышении устойчивости к экстремальным климатическим условиям и поддержании баланса экосистем. Особое внимание уделено экономическим и социальным выгодам, которые применение MAR может предоставить сельскому хозяйству и местным сообществам. Методология исследования включает анализ существующей практики, экономическое моделирование и оценку влияния внедрения технологий MAR на региональном уровне. Результаты демонстрируют, что MAR является эффективной и экономически выгодной стратегией водоснабжения, способствующей устойчивому использованию водных ресурсов и повышению качества жизни. Научная новизна заключается в обосновании экономической целесообразности применения систем MAR в контексте изменения климата, а практическая значимость исследования – в возможности внедрения полученных результатов в практику управления водными ресурсами.

***Ключевые слова:*** *управляемое пополнение водоносных горизонтов, экономическая эффективность, водоснабжение, изменение климата, водные ресурсы, устойчивое развитие.*

SATBAYEV UNIVERSITY

Engineering Journal of Satbayev University

Volume 147 (2025), Issue 5, 35-41

https://doi.org/10.51301/ejsu.2025.i5.05

# Petrogenesis and mineralogical characteristics of copper-bearing rocks in the Koldar massif, Central Kazakhstan

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**Abstract.** This paper analyzes the increasing global demand for mineral resources, particularly non-ferrous metals, and the growing challenge of finding new deposits as easily accessible ones are depleted. The study focuses on copper-porphyry deposits, which are crucial for meeting this demand. It highlights the importance of improving exploration methods and integrating various geological, geophysical, and geochemical techniques. The research emphasizes the unique metallogenic potential of Kazakhstan, specifically the Dzungar-Balkhash folded region, which hosts significant copper deposits like Kounrad, Bozshakol, Koksai, Aktogay, and Aidarly. The study uses the Koldar massif as a reference site to analyze the geological and mineralogical characteristics of these deposits. The methodology involved field studies, sample collection, and macro- and microscopic mineralogical analysis. A key finding of the study is the identification of widespread kalishpatization and argillitization zones within the Kyzylkiya deposit, which were confirmed through both spectral analysis and micromineralogical studies. The research also details the mineralogical and geochemical characteristics of different types of copper-porphyry ores, linking them to specific petrological features of the ore-bearing plutonism. The paper concludes by presenting prospect and prospecting signs for copper-porphyry deposits in various geological settings.

***Keywords:*** *petrogenesis, copper, mineralogy, metallogenic models, non-ferrous metal ores.*

Received: 23 January 2025
Accepted: 16 October 2025
Available online: 31 October 2025

## 1. Introduction

The increasing demand for mineral raw materials, and especially for non-ferrous metal ores, determines the further strengthening and expansion of their mineral resource base and an increase in exploration reserves, primarily in the areas of existing mining enterprises and newly created territorial production complexes, for the long term. Currently, the fund of easily discoverable deposits located on the surface or at shallow depths is mostly exhausted. This requires constant improvement of the genetic and geological foundations of regional, small- and medium-scale and local forecasting, the formation of geological prospecting models of different metallogenic categories, the rational use and combination of geological, geophysical, geochemical and mining methods based on the development of their effective complexes.

### 1.1. Geological context of non-ferrous metal deposits

Recent research in the Chinese Altai highlights the significance of integrating zircon U–Pb geochronology with isotopic and geochemical data to better understand the petrogenesis and tectonic settings of granitic complexes associated with mineralization [1]. In Eastern Kazakhstan, studies of Early Triassic magmatism reveal a genetic link between monzonite–granite intrusions and the activity of the Siberian Large Igneous Province, suggesting a broader geodynamic control on ore formation processes [2]. Investigations in Central Lhasa further demonstrate that Cretaceous magmatic events played a key role in the development of Fe–Cu skarn systems, emphasizing the metallogenic potential of post-collisional magmatism [3]. Meanwhile, data from Northwestern Mongolia provide insight into the evolution of orogenic belts and show how regional tectonic regimes influence the emplacement and mineralization of granitoid massifs [4].

The expansion and strengthening of the raw material base of non-ferrous metals in the Republic of Kazakhstan is primarily possible due to the discovery and study of deposits of the leading geological and industrial types-copper-nickel, copper-porphyry, copper-pyrite, copper sandstones and shales. The specifics of deposits are determined by their paleotectonic and geological position, spatiotemporal and genetic relationships with various structural and material complexes (formations), the size and morphology of mineralization zones and ore bodies, the quality, mineral and material composition of ores. Copper deposits are quite widespread around the globe and are found on almost all continents. They are also diverse in time of formation (from the Precambrian to the Cenozoic), and each age epoch is characterized by the predominance of one or two geological and industrial types. In the Mesozoic, unlike the PaleoTriassicterozoic, there was a slight decrease in copper reserves, and the main reserves of this period were concentrated in copper-porphyry and copper-nickel deposits, which were developed mainly in the CIS


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Engineering Journal of Satbayev University. eISSN 2959-2348. Published by Satbayev University

countries, being associated with the Triassic trap formation. The Cenozoic era is characterized by a sharp increase in copper accumulation due to copper-porphyry deposits, confined mainly to the Pacific (94% of reserves) and partly to the Mediterranean (6% of reserves) folded belts [5].

Kazakhstan is a unique ore province, which contains deposits of non-ferrous and ferrous metals that are diverse in composition and origin [6]. In the Balkhash ore province, copper deposits are developed in the form of a sublatitudinal strip extending along the Northern Balkhash region for more than 500 km with a width of 40-50 km [5].

Copper porphyry deposits are characterized by large masses of calcified rocks with a veined-interspersed, stockwork character of mineralization and are mainly suitable for mining by large quarries. By origin, these deposits are hydrothermal: plutogenic and volcanogenic, which indicates a fairly wide range of temperatures at which their formation took place. Ore-bearing rocks can be coarse- and fine-grained diorites, granodiorites, granodiorite porphyries and plagiogranite porphyries, which break through older (from Silurian to Carboniferous) formations, both volcanogenic-sedimentary and intrusive. The main ore components are copper and molybdenum. The primary ore minerals are pyrite, chalcopyrite, bornite, molybdenite, and magnetite. Malachite and azurite develop in the oxidation zone, and chalcosine ores with bornite develop in the secondary enrichment zone. Hydrothermal changes consist of calypathization and biotitisation [5].

Mineralization, as a rule, is located on the contact of the intrusions that caused it with the host rocks (Aktogay, etc.) or on the periphery of granitoid massifs (Aksu, Koksai). At the same time, ore-bearing intrusions are confined to deep fault zones, as well as the widespread development of hydrothermal changes, up to the transformation of rocks into secondary quartzites. There are a number of areas within the Dzungar-Balkhash folded region where a combination of deep fault zones with manifestations of intrusive activity is favorable for mineralization, especially at its junction with the Teniz and Zhezkazgan depressions in the west, the Kokshetau block in the west and northwest, the Zharkyn synclinor structure in the east, and the Shu syneclise and Ili depression in the south. Studying various research cases of copper deposits around the world, it was interesting to review the case of the Shagai belt, Pakistan, with the resulting RSG picture of the massif [7] (Figure 1).

***Figure 1. RSG of the Reko Dik copper and sapphire deposit, Pakistan***

On the territory of the Republic of Kazakhstan, the research group accepted the copper deposits of the Koldar massif as the reference study site.

### 1.2 Geological description of the research object

The Dzungar-Balkhash folded region is located to the north, northwest and southeast of the lake. Balkhash, on three sides (except the southeast), it represents a trough formed by precipitation from the Middle Devonian to Permian inclusive. Folding, accompanied by intrusions, manifested itself in the Middle and Late Carboniferous; in its first stage, moderately acidic granitoid intrusions were introduced, and in the second, potassium granites mainly along faults. During the consolidation process in the Late Permian-Early Mesozoic, intense terrestrial volcanic activity was manifested along the periphery of the region in connection with the above-mentioned deep faults, which led to the development of not only tuff-effusive formations but also subvolcanic intrusions. The latter are associated with the formation of secondary quartzites and molybdenum-copper deposits of the stockwork type near volcanic apparatuses [8]. The formation of subgeosynclinal and skarn polymetallic deposits is associated with late Carboniferous small intrusions, and stockwork and vein deposits of tungsten and molybdenum are associated with early Carboniferous and late Carboniferous intrusions.

Five Paleozoic deposits are known in the Dzungara-Balkhash folded region in Central Kazakhstan: Kounrad, Bozshakol, Koksai, Aktogay, and Aidarly. The Aktogay deposit is located to the east of the lake. Balkhash is associated with volcanogenic formations of the Upper Paleozoic age, broken near the axis of the anticlinal fold by a number of small subvolcanic intrusions of the Upper Carboniferous age, represented by granodiorites and quartz diorites. The copper mineralization is confined to the apical part of one of the massifs and partially extends into its roof. In the central part of the deposit there is an oreless quartz core surrounded by a zone of kalishpatization, which is associated with the main mineralization [9]. The ore body has a horseshoe shape with a maximum width of up to 800 m. Mineralization has been traced to a depth of 400-600 m. It has a veined-interspersed character. Ore minerals are chalcopyrite, pyrite and less commonly bornite and molybdenite. Magnetite, pyrrhotite, sphalerite and galena are even rarer. The copper content in the ores is low and does not exceed 0.6%. Molybdenum is low. The Aidarly field is located next to the Aktogay field and is similar in structure to it.

## 2. Materials and methods

The sites were studied and sampling points were identified for macro- and microscopic mineralogical analysis of the selected sites (Figure 2). The general requirements for the cartographic foundations of predictive research are determined by the need to display a set of direct and indirect signs of predicted objects on graphic materials. The places of birth, attributed to the copper-porphyry family by a complex of geological, genetic and geological industrial characteristics, are located in specific geostructures – volcanic-plutonic belts of two types:

1) Basaltoid rocks formed in the outer (barrier) zones of island arcs, at the end of the early stages of the development of eugeosynclines;

2) Andesitoids, which are formed in the orogenic activation regime on a substrate of various compositions and times of occurrence.

The allocation of provinces or regions requires determining the position of the corresponding volcanic-plutonic belts in the general zonality of geosynclinal folded systems.

Basaltoid volcanic-plutonic belts in marginal continental geosynclinal folded systems are naturally located between the frontal troughs adjacent to the troughs and the inner or rear zones of the island-arc space.

In intracontinental geosynclinal folded systems, basaltoid belts are located either directly above the actual eugeosynclinal bends or on their flanks. This situation is typical for Central Kazakhstan and the Central Ural uplift, which have experienced the regeneration of the geosynclinal regime [10].

In the basaltoid volcanic-plutonic belts, the copper-porphyry and gold-copper-porphyry deposits themselves are associated with a volcanic-plutonic association formed by basalt-andesite-basalt volcanogenic and gabbro-diorite-quartz-diorite (plagiogranite) plutogenic formations. This association occurs after an undifferentiated basalt, contrasting basalt-rhyolite, and continuous basalt-andesite-dacite-rhyolite formation. After productive association, volcanogenic molassoids of Grauwacke and carbonate strata usually accumulate.

The appearance of copper-porphyry deposits is preceded by the formation of pyrite family deposits associated with differentiated volcanogenic formations and located in adjacent metallogenic zones [11]. Copper-porphyry deposits arose after ferromanganese volcanogenic ores. Vein polymetallic deposits are formed synchronously with copper-porphyry ores or somewhat later.

In andesitoid volcanic-plutonic belts, the composition of ore-bearing volcanic-plutonic associations depends on the nature of the substrate. In epicraton belts, molybdenum-porphyry deposits are associated with the diorite-granodiorite-granite formation, which is preceded by the usually sparsely distributed dacite-rhyolite. In epimiogeosynclinal belts, copper-molybdenum-porphyry deposits accompany associations formed by andesite-dacite-rhyolite and diorite-granodiorite-monzonite formations [12]. In the epicheosynclinal belts, molybdenum-copper-porphyry deposits are associated with volcanic-plutonic associations, which include andesite volcanogenic and gabbro-diorite-granodiorite plutogenic formations. These associations belong to the initial stages of the formation of volcanic-plutonic belts. Later formations are represented by ignimbrite, liparitoid, trachytoid, and trachyte volcanogenic formations accompanied by granite-leucogranite, subalkaline, alkaline, and alaskite intrusive complexes.

The epicraton-type birthplace is characterized by a significant amount of molybdenum, less often copper-molybdenum ore composition and is most similar to molybdenum-porphyry objects according to classification V. Popov. In the ores of these deposits, the ratio of copper to molybdenum varies from 0.4 to 20 (about 13 on average) with non-industrial copper contents [13]. The deposits contain practically no gold and silver.

The gold-bearing ores of epimiogeosynclinal type deposits have a molybdenum-copper composition; the Cu ratio: The Mo ranges from 15 to 40 with a weighted average of about 23. The ores are weakly gold-bearing and silver-bearing [14].

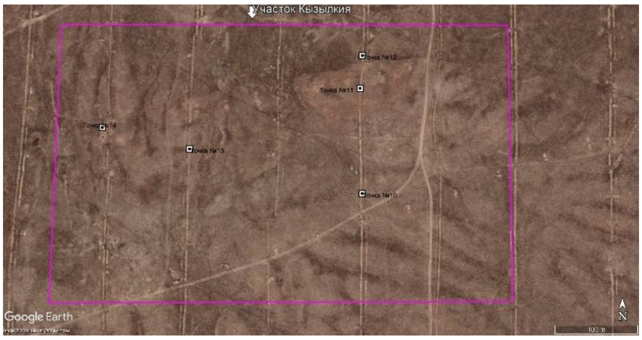
Epigeosynclinal deposits are characterized by gold-bearing molybdenum-copper ores. Cu Relations: Mo ranges from 30 to 235 at concentrations of Mo, sometimes of no industrial significance [15-24]. The gold content of ores is increased and can reach 20 g per 1 t of copper reserves.

Thus, copper-porphyry deposits are clearly divided into molybdenum-, gold-bearing copper-molybdenum-, gold-bearing molybdenum-copper and gold-copper-porphyry deposits, which correlates with the petrological characteristics of ore-bearing plutonism, depending on the types of metallogenic zones. Table 1 shows the conditions of occurrence of deposits of the copper-porphyry family.

***Table 1. Conditions of occurrence of molybdenum-porphyry deposits***

| № | Metallogenic and geological characteristics of typical environments | Types of deposits and reference objects |
|---|---|---|
| | Molybdenum-copper-porphyry (deposits of the Balkhash region) | |
| 1 | Metallogenic provinces | Marginal and intracontinental orogenic-activated volcanic-plutonic belts on the eugeosynclinal basis |
| 2 | Metallogenic (structural and formation) zones: | |
| 2.1 | Paleotectonic position | The central parts of the epigeosynclinal volcanic-plutonic belts with the development of an association of formations: andesite-dacite and andesite-rhyodacite volcanogenic and gabbro-diorite-granodiorite and plutogenic.<br>The central parts of the belts with the development of ore-bearing volcanic-plutonic associations |
| 2.2 | Formations containing ore-bearing plutogenic formations | Andesite-dacite, andesite-rhyodacite, and eugeosynclinal belt base complexes |
| 2.3 | Ore-bearing formations | Gabbro-diorite-granodiorite potassium-sodium ($K_2O$:$Na_2O$ from 0.5 to 0.8) |
| 3 | The position of the ore regions | Multiphase plutons or areas of development of porphyritic phases of an ore-bearing formation corresponding to groups of magmatogenic-hydrothermal systems |
| 4 | Position of ore fields and deposits | Porphyry stocks and adjacent structures of their frames are controlled by inherited paleodepositions of the base of the belts (sometimes stratovolcanoes) and corresponds to single magmatogenic-hydrothermal systems |
| 5 | Ore-bearing structures and the position of ore bodies in ore-bearing formations and massifs | Apical parts of rod-shaped and dike-shaped porphyry intrusions, sometimes containing breccia tubes of granodiorite porphyries |
| 6 | Forms of ore bodies | Conformal to porphyry intrusions: cones, truncated cones, hollow thick-walled cones, straight and inverted, plates, wedges; disconformal: inverted vertical and inclined cones, groups of cones and funnel-shaped bodies |
| 7 | Geochemical characteristics of ores | Cu:Mo from 200:1 to 30:1 |
| 8 | Ash content of hydrothermal-metasomatic changes (from the center to the periphery of the rods) | Biotitisation-sericitization-propylitization |

In the course of the study, the rock sections and polished sections were taken from control points, prepared and studied (Figure 2). Rock sections are thin sections of rocks examined under a microscope in transmitted light to study their mineral composition and texture. Polished sections are polished samples that are examined under a microscope in reflected light, which makes it possible to analyze ore minerals and their relative positions.




(a)


(b)

***Figure 2. Overview diagram of control points: (a) – initial condition of the study area before mining operations; (b) – open-pit view with control points where rock and polished sections were collected for mineralogical analysis***

## 3. Results and discussion

An important result of the research was the identification of the zone of spread of kalishpatization, which is widespread in the Kyzylkiya deposit. The samples taken from this zone also confirmed the results of spectral analysis. Argillitization, widespread in the territory of Kyzyl Kiya, was clearly expressed on the map, compiled according to the results of the spectral method, and confirmed by microminerological studies.

Thin sections are magnified by 10X. From left to right, the top row contains grains of biotite, quartz, plagioclase, sericitized, and potassium feldspar, in crossed nichols; unchanged quartz in granodiorite, about 5 mm in size, in crossed nichols. From left to right, the bottom row shows hydrothermal changes in feldspar, in crossed nichols; a cryptocrystalline glassy mass with fragments of pyroclasts and quartz fragments, in crossed nichols (Figure 3).

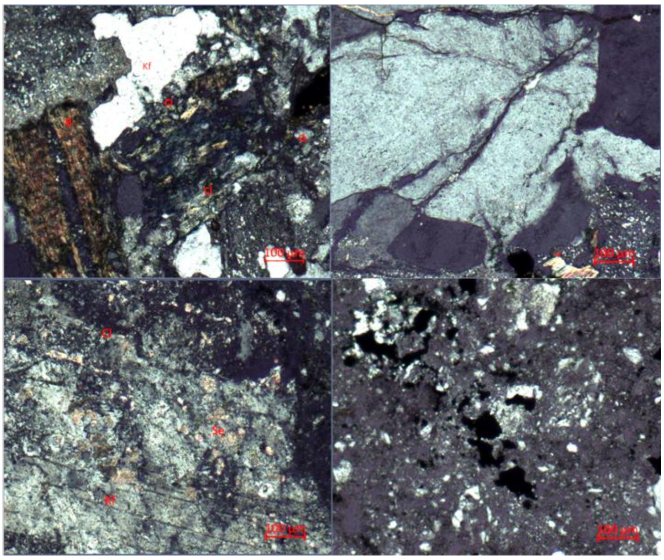


***Figure 3. Microscopic images of rock thin sections showing mineral composition and key geological features of the rock samples***

The mineralogical composition and geochemical features of ores from various types of copper-porphyry deposits are summarized in Table 2 where I-IV are types of deposits:
– I-molybdenum-porphyry,
– II-gold-bearing copper-molybdenum-porphyry,
– III-gold-bearing molybdenum-copper-porphyry,
– IV-gold-copper-porphyry

***Table 2. Mineralogical and geochemical characteristics of the main types of ores of copper-porphyry deposits***

| Ore type | Minerals | | | | Mineral associations | Impurity elements | | The predominance in various types of deposits | | | |
|---|---|---|---|---|---|---|---|---|---|---|---|
| | Main | Secondary | Rare | Gangue | | Main | Secondary | I | II | III | IV |
| Magnetite-pyrite | py, mag | hm, pyr | ilm, cub, chp, mo, rt | q, chl | py, mag-py | Co(py)*, Se(py) | Te, Re, Bi, Au, Ag | + | + | + | +++ |
| Molybdenum | mo, py | chp, bn, pyr, hb, wf, mag | gal, sph, bit, hm, bl, cs | q, anh, bi | mo, q-mo, q-py-mo | Re(mo) | Se, Te | +++ | ++ | + | + |
| Chalcopyrite-bornite | bn, chp, py | mo, chils, mag | sph, gal, ars, bl, bit, Au, el, Ag, an, tel, Au, Ag | q, pot, anh, bi, ser | mo-chp-bn, mo-py-chp-bn, bn-chp, py-chp-bn | Re(chp), Au(chp,bn), Ag(chp,bn, sph, gal) | Se, Bi, Pt(chp) | – | ++ | + | – |
| Pyrite-chalcopyrite | chp, py | bn, mo, mag, sph, chls | sph, gal, ars, bl, bit, Au, el, Ag, tel, Au, Ag, cv, an | q, pot, anh, bi, ser | mag-py-chp, mo-bn-py-chp, mo-py-chp, py-chp | Re(chp,mo), Au(chp,bn), Ag(chp,bn, sph, gal) | Se, Bi, Pt(chp) | ++ | +++ | +++ | +++ |
| Polysulfide | py, chp, gal, sph, bl | an, Au, Ag, tel, u sel Au, Cu, Pb | gs, ars, pyr | q, car, bar, zl | py-chp-bl-gal-sph, py-chp-sph-ars, gal-sph-py-chp | Au(py, chp, gal, bl) Ag(chp, sph, gal, bl) | Se, Te Bi (gal, sph) In, Cd, Ga | + | ++ | ++ | + |
| Enargite-polysulfide | py, en | luz, ars, bn, rdhr, bl, S, gal, sph, chp | spec, cv | q, car, zl | en-chp-gal-sph, py-en-chls, en-ars-py, en-rdhr-cv-py-S, luz-en | Ag(bl, gal, sph) | –, – As | – | + | + | – |
| Hypergene | chlz, bn, cv, as | py, chp, mal, Au, Cu, mo | | – | – | Au, Ag | – | + | ++ | ++ | ++ |

*The minerals that carry the elements of impurities are indicated in parentheses

The prevalence of the main types of ores is characterized as follows: "+++" indicates occurrence in significant amounts, including at the level of major industrial concentrations; "++" denotes constant presence in various quantities; "+" marks rare occurrence in small volumes; "–" signifies absence.

Mineral designations:

Ad – native silver; ap – argentite; anh – anhydrite; ars – arsenopyrite; as – azurite; Ai – native gold; bar – barite; bi – biotite; bl – pale ore; bit – bismuthinite; bn – bornite; sag – carbonate; chl – chlorite; chr – chalcopyrite; chls – chalcosine; cs – cassiterite; Si – native copper; cub – cubanite; cup – cuprite; cv – covelline; el – electrum; en – enargite; gal – galena; gs – hessite; hb – hubnerite; hm – hematite; ilm – ilmenite; luz – luconite; mag – magnetite; mal – malachite; mo – molybdenite; pot – potassium feldspar; py – pyrite; pyr – pyrrhotite; q – quartz; rdhr – rhodochrosite; rt – rutile; S – native sulfur; sel – selenides (of gold, silver, copper, lead); ser – sericite; spec – specularite; sph – sphalerite; tel – tellurides; wf – wolframite; zl – zeolites.

The characteristics of the prospect and prospecting conditions are summarized in Table 3.

***Table 3. Prospect and prospecting signs of copper-porphyry deposits in various settings***

| Elements of field models | Environments-according to the options for the position of the erosional truncation and the possibilities of detecting ore bodies | | |
|---|---|---|---|
| | Favorable, I, II | Moderately favorable, III | Unfavorable, IV |
| Supramineral space: Propylitized rocks of the intrusive frame | +c | +c | - |
| Zones: | | | |
| Pyritization | +c | +c | - |
| Tourmalinization | +- u | +- u | - |
| Argillitization | +c | +c | - |
| brecciated tubes | +-a | +-a | - |
| Supramineral lithogeochemical envelope | +c | +c | - |
| Vein polymetallic mineralization | +c | +c | - |
| Vein and impregnated copper-arsenic mineralization | +-c | +c | - |
| Ore – hosting space | | | |
| Propylitized rocks of the intrusive frame | +c | - | - |
| Pyritization zones | +c | - | - |
| Porphyritic intrusive bodies | +c | - | - |
| Brecciated tubes | +-c | - | - |
| External lithogeochemical envelope | +c | - | - |
| Vein polymetallic mineralization | +-c | - | - |
| Zones: | | | |
| Argillitization | +-c | - | - |
| Silicification and sericitization | +c | - | - |
| Ore bodies | +u | - | - |
| K-feldspathization and biotitization | +-c | - | - |
| sub-ore lithogeochemical envelope | +-c | - | - |
| sub-ore space: | | | |
| Propylitized rocks of the intrusive frame | +-c | - | +-c |
| Pyritization zones | +-c | - | +-c |
| Porphyritic intrusive bodies | +c | - | +c |
| External lithogeochemical envelope | +c | - | +c |
| K-feldspathization and biotitization | +-c | - | +-c |
| Quartzy core | +-u | - | +-u |
| Internal lithogeochemical envelope | +-c | - | +c |

The designations used in Table 3 are as follows: +" presence of a feature on the section; "–" absence of a feature; "+/–" possible presence of a feature; "u" unambiguous identification of the corresponding part of the geological space; "a" ambiguous identification; "c" identification in combination with other features.

## 4. Conclusions

An important result of the microminerological studies was the identification of the zone of spread of kalishpatization, which is widespread in the Kyzylkiya deposit. The samples taken from this zone also confirmed the results of spectral analysis. Argillitization, widespread in the territory of Kyzyl Kiya, was clearly expressed on the map, compiled according to the results of the spectral method, and confirmed by microminerological studies. Mineralogical and geochemical characteristics of the main types of ores of copper-porphyry deposits were identified. Prospect and prospecting signs of copper-porphyry deposits in various settings were detected.

## Author contributions

Conceptualization: DEU; Data curation: AAB; Formal analysis: EOO; Funding acquisition: EOO; Investigation: AAB; Methodology: DEU; Project administration: DEU; Resources: AAB; Software: TLA; Supervision: EOO; Validation: TLA; Visualization: DEU; Writing – original draft: DEU; Writing – review & editing: EOO. All authors have read and agreed to the published version of the manuscript.

## Funding

This research was funded by the Committee of Science of the Ministry of Science and Higher Education of the Republic of Kazakhstan (Grant No. BR21882179).

## Acknowledgements

The authors express their sincere gratitude to the editor and anonymous reviewers for their constructive comments and valuable suggestions, which have significantly improved the quality of this manuscript.

## Conflicts of interest

The authors declare no conflict of interest.

## Data availability statement

The original contributions presented in this study are included in the article. Further inquiries can be directed to the corresponding author.

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# Орталық Қазақстандағы Қолдар массивінің мыс-кенді тау жыныстарының петрогенезисі және минералогиялық сипаттамалары

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**Аңдатпа.** Бұл жұмыс минералдық шикізатқа, әсіресе түсті металл кендеріне деген жаһандық сұраныстың артуын және оңай табылатын кен орындарының сарқылуына байланысты жаңа кен орындарын іздеудегі қиындықтарды талдайды. Зерттеу осы сұранысты қанағаттандыруда маңызды рөл атқаратын мыс-порфирлі кен орындарына бағытталған. Ол барлау әдістерін жетілдірудің және әртүрлі геологиялық, геофизикалық және геохимиялық әдістерді біріктірудің маңыздылығын көрсетеді.Зерттеу Қазақстанның бірегей металлогендік әлеуетін, атап айтқанда Қоңырад, Бозшакөл, Қоксай, Ақтоғай және Айдарыл сияқты маңызды мыс кен орындары орналасқан Жоңғар-Балқаш қатпарлы аймағын ерекше атап көрсетеді. Зерттеу жұмысы осы кен орындарының геологиялық және минералогиялық сипаттамаларын талдау үшін Қолдар массивін эталонды зерттеу алаңы ретінде пайдаланады. Әдістеме далалық зерттеулерді, үлгілерді жинауды, сондай-ақ макро- және микроскопиялық минералогиялық талдауды қамтиды.Зерттеудің негізгі нәтижесі – Қызылқия кен орнында кең таралған калишпаттану және сазға айналу аймақтарын анықтау болып табылады, бұл спектрлік талдаулармен және микроминералогиялық зерттеулермен расталды. Жұмыс сондай-ақ мыс-порфирлі кендердің әртүрлі түрлерінің минералогиялық және геохимиялық сипаттамаларын, олардың кен түзуші интрузиялардың петрологиялық ерекшеліктерімен байланысын егжей-тегжейлі сипаттайды. Қорытынды бөлімде әртүрлі геологиялық жағдайлардағы мыс-порфирлі кен орындарының іздестіру белгілері ұсынылған.

***Негізгі сөздер:*** *петрогенезис, мыс, минералогия, металлогендік модельдер, түсті металл кендері.*

# Петрогенезис и минералогические характеристики медьсодержащих пород массива Колдар, Центральный Казахстан

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**Аннотация.** В данной работе анализируется растущий мировой спрос на минеральные ресурсы, особенно на руды цветных металлов, и усиливающаяся проблема поиска новых месторождений по мере истощения легкодоступных залежей. Исследование сфокусировано на медно-порфировых месторождениях, которые имеют решающее значение для удовлетворения этого спроса. Подчеркивается важность совершенствования методов разведки и интеграции различных геологических, геофизических и геохимических подходов.Исследование акцентирует внимание на уникальном металлогеническом потенциале Казахстана, в частности, на Джунгаро-Балхашском складчатом регионе, где расположены такие значимые медные месторождения, как Коунрад, Бозшаколь, Коксай, Актогай и Айдарыл. В качестве эталонного участка для анализа геологических и минералогических характеристик этих месторождений используется массив Колдар. Методология включала полевые исследования, отбор проб, а также макро- и микроскопический минералогический анализ.Ключевым результатом исследования стало выявление обширных зон калишпатизации и аргиллизации в пределах месторождения Кызылкия, что было подтверждено как спектральным анализом, так и микроминералогическими исследованиями. В работе также подробно описаны минералогические и геохимические характеристики различных типов медно-порфировых руд, связывая их с конкретными петрологическими особенностями рудоносного плутонизма. В заключении представлены поисковые признаки медно-порфировых месторождений в различных геологических обстановках.

***Ключевые слова:*** *петрогенезис, медь, минералогия, металлогенические модели, руды цветных металлов.*

**Publisher's note**

Engineering Journal of Satbayev University
Volume 147 (2025), Issue 5, 42-48

https://doi.org/10.51301/ejsu.2025.i5.06

# The role of environmental and industrial safety during subsoil development

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**Abstract.** The article examines the environmental problems of the territories of the Republic of Kazakhstan. These problems are associated with the depletion of accessible mineral resources and the growing amount of man-made waste (MMW), which increases the land area occupied by dumps and landfills as well as the negative impact on the environment. Accumulated waste is, on the one hand, the main pollutant of the environment, and on the other hand, a valuable material potentially suitable for processing and reuse in the production of widely demanded building materials. The need for processing technogenic mineral formations is dictated by the fact that they occupy vast territories, are easily dispersed by wind, and pose an increased environmental risk for mining and metallurgical regions. In this regard, the scientific foundations of integrated methodologies for the environmental assessment of MMW and their recycling in the production of building materials and products were developed. The study substantiates the possibility of using these wastes as raw materials for building products while simultaneously addressing environmental challenges in the affected territories. Methods for the disposal of man-made waste and a quantitative assessment of the prospects for introducing resource-saving technologies are presented.

***Keywords:*** *environmental problems, the resource approach, man-made wastes, building materials, wastes.*

Received: 15 April 2025
Accepted: 16 October 2025
Available online: 31 October 2025

## 1. Introduction

The mineral resource complex plays a crucial role in ensuring the economic stability and sustainable development of the Republic of Kazakhstan. The country possesses vast reserves of mineral raw materials, including both metallic and non-metallic ores, which constitute the foundation of industrial growth and export potential. However, the effective utilization of these resources requires not only geological exploration and extraction, but also a well-balanced state policy aimed at rational use, technological modernization, and environmental protection.

Based on the importance of the mineral resource complex for the national economy, the strategy for its development should be implemented throughout the country and, therefore, must be a state-level strategy. The ultimate goal of this state strategy is to identify and implement optimal solutions that would satisfy Kazakhstan's internal demand for minerals, enhance the competitiveness of its mining and metallurgical sectors, and ensure a stable and secure position in the global mineral market.

To achieve this goal, it is necessary to address a series of fundamental questions [1-3]:

1. What types of mineral resources are advisable to mine directly on the territory of the Republic of Kazakhstan?
2. What volumes of mineral extraction, and in what time frame, should be ensured?
3. Which mineral deposits, including technogenic and secondary ones, should be prioritized for exploitation?
4. What technological solutions should be implemented in the extraction and processing of minerals and mineral raw materials?
5. Which beneficiation and enrichment technologies should be applied to maximize recovery and minimize losses?
6. What human resources are required both in quantity and qualification to ensure sustainable industry development?
7. To what extent should financial resources be mobilized (public and private investment) to implement this strategy effectively?

Finding comprehensive answers to these questions is an extremely complex task. The economy of Kazakhstan relies on more than one hundred types of minerals that vary in quality, origin, technological processing, and industrial application. These factors create a wide spectrum of possible development pathways for the national system of mineral and energy supply. Therefore, there is a need for a scientifically grounded methodology that would enable the reduction of numerous possible options to a rational and economically justified set, allowing the preparation of reliable feasibility studies for mining projects and technological solutions.

An important part of this comprehensive approach is the study of waste and secondary mineral resources, which represent a significant reserve for increasing the efficiency of the mineral sector.


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Engineering Journal of Satbayev University. eISSN 2959-2348. Published by Satbayev University

Tailings from ore enrichment and man-made deposits contain valuable components that can be reused in various technological cycles, including in the production of construction materials. Research in this area not only contributes to resource conservation and circular economy principles but also reduces the environmental burden on mining regions.

The object of this study is enrichment tailings and embedded mixtures with and without additives based on them, while the subject of research is the physical and mechanical properties and hardening behavior of these embedded mixtures under natural moist conditions.

## 2. Materials and methods

### 2.1. Research materials

Scientific publications of foreign and domestic researchers in the field of using mining and metallurgical waste as secondary raw materials were analyzed. The tailings of the processing plants Achpolymetal, Donskoy GOK, Kazakhmys Corporation and the Akzhal mine of Nova-Zinc LLP were studied, and the ash and slag waste from the Almaty thermal power plant, which uses coals from the Ekibastuz deposit, was also studied.

By burning coal, thermal power plants receive thermal energy and generate electrical energy. The negative side of this process is the formation of by-products of coal combustion – fly ash and slag. The composition of the ash and slag material was determined by the quantitative ratio of the minerals included in it, which depend on the mineralogical composition of the initial part of the fuel. Ordinary samples weighing from 3-5 to 15-16 kg were taken at the ash dumps, of which group samples were subsequently compiled.

### 2.2. Research methods

To determine characteristics of input materials and composition of embedded mixtures and their physical and mechanical properties, standard methods were used, and XRF and IR were used to identify their physical and chemical properties.  X-ray phase analysis (XPA) was carried out on a DRON-3M X-ray installation (RF) and X-ray structural analysis was carried out with a JCXA-733 «Superprobe» microanalyzer (Japan) with software, scientific research to study waste structure using laboratory polarizing microscope Leica ICH DM2500 (Switzerland), equipped with a powerful 100 W illuminator, which allows you to comfortably work with differential interference contrast; differential thermal analyzes (DTA) were carried out on a derivatographic device MOM-1500 D (Hungary); chemical analysis and microhardness tester PMT-3 (RF). Particle size distribution analysis was carried out using Analizette 22 Micro Tec Fritsch GmbH (Germany) device.

## 3. Results and discussion

As all industries grow, the consumption of the resources of the Earth's interior, including mineral resources, increases, which leads to the formation of a large amount of man-made waste (MMW) production. In this regard, environmental problems are increasing. The construction materials industry is one of the few that can dispose of high-tonnage waste from other industries. Large-scale utilization of MMW can significantly expand the raw material base for the production of building materials, as well as contribute to improving the environmental situation in the regions.

At the same time, it must be considered that the scale and depth of the negative impact of mining on the environment causes great concern to society. Even poets write with excitement: «There is less and less surrounding nature, More and more environment».

It has been established that the impact of mining on the environment must be identified a priori in order to have a reserve of time to develop the most technologically and economically effective methods for eliminating and minimizing this impact.

When developing technological processes for mining production, priority should be given to those solutions that ensure the absence or minimum generation of production waste (Figure 1).

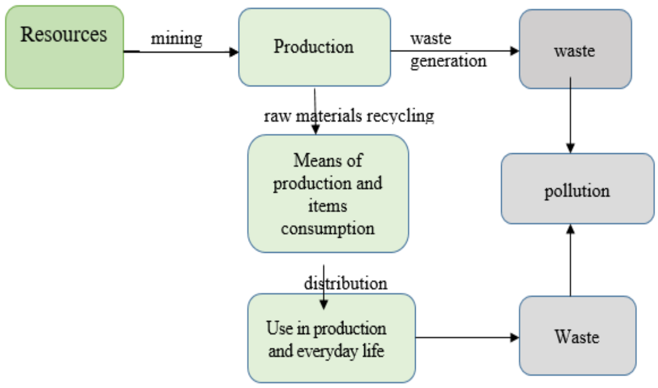



***Figure 1. Conversion of subsoil resources into environmental pollution***

The idea of creating waste-free (ecologized) subsoil use involves the development and implementation of methods and means of their organizational and economic support, allowing modern mining production to fit into the natural geochemical cycle, thereby turning it into a geochemically closed system. Closed green mining production is based on the following principles:

– minimum losses of matter and energy at the stages of their removal from the natural system and subsequent use in mining;

– maximum use of mining waste in other economic systems and to restore the disturbed ecological balance of the natural system.

These principles should be considered when developing the scientific basis for an environmental strategy for the development of the mineral resource base and mining industry in the Republic of Kazakhstan. All methods of mining are characterized by an impact on the biosphere, polluting almost all its elements, including the subsoil.

Subsoil is the object and operational basis of mining production and is subject to the greatest impact. Since subsoil belongs to elements of the biosphere that do not have the ability to naturally renew in the foreseeable future, their protection should include ensuring scientifically sound and economically justified completeness and complexity of use.

The impact of mining on the biosphere is manifested in various sectors of the national economy and is of great social and economic importance.

At present, it is not possible to give a comparative quantitative assessment of the impact of subsoil crawling and other types of human activities on the environment, since there is no scientific and methodological basis for such a comparison. The various private criteria used do not allow us to obtain an unambiguous answer to this question.

In Table 1, a comparative qualitative assessment of the environmental impact of certain types of industrial production is given [4]. As can be seen from this table, mining has the widest impact on the biosphere, affecting almost all its elements. We analyzed the results of various programs and projects for environmental restoration in areas of closed mines for the extraction of uranium, ore and non-metallic minerals and coal.

***Table 1. Comparative assessment of the impact of various types of industrial production on the environment***

| Industry | Impact of industries on elements of the biosphere | | | | | | |
|---|---|---|---|---|---|---|---|
| | Airpool | Water pool | | Earthly surface | | Flora, fauna | Subsoil |
| | | Superficial water | Underground water | Soil cover | Landscape | | |
| Chemical and petrochemical | $S_t$ | $S_t$ | $M_e$ | $M_e$ | Mi | $M_e$ | Mi |
| Metallurgical | $S_t$ | $S_t$ | Mi | $M_e$ | Mi | $M_e$ | O |
| Fuel and energy | $S_t$ | $S_t$ | Mi | Mi | Mi | Mi | O |
| Construction | Mi | Mi | Mi | $M_e$ | M | Mi | Mi |
| Transport | $M_e$ | $M_e$ | Mi | Mi | Mi | Mi | O |
| Mining | $M_e$ | $S_t$ | $S_t$ | $S_t$ | $S_t$ | $M_e$ | $S_t$ |

*Note. O – no impact; Mi – minor impact, $M_e$ – medium impact, $S_t$ – strong impact

Activities that were carried out during the implementation of these programs included: inventory of areas of underground and open-air mining complexes, creation, management and visualization of a database, assessment of environmental impact, forecasts of future changes in this impact, classification and prioritization of individual and group underground and above-ground objects complexes, development of technical solutions for the liquidation or conservation of objects with subsequent technical and biological reclamation of disturbed areas, management and purification of mine and drainage waters and simultaneous environmental monitoring.

For example, mining enterprises annually extract over 200 million tons of minerals and host rocks from the depths of the Republic of Kazakhstan. The extraction and processing of such large volumes of mineral resources have a negative impact on the environment due to the generation of mining waste. Subsoil use waste is stored off-balance ore, overburden rock, intermediate waste from the process of ore preparation and hydrometallurgy, waste tailings from the flotation and sorption process, slag from the pyrometallurgy process, liquid cyanide containing waste, which is placed on internal and external dumps, tailings and ash and slag storage facilities and other storage facilities.

Mining in Kazakhstan continues to have a negative impact on environmental components. During the extraction of mineral deposits, huge amounts of hazardous waste are produced, which detonate in the dumps of mines and mines, as well as in the tailings of processing plants and hydrometallurgical plants.

Solution to environmental problems in the areas of mines and processing plants cannot be delayed. Intervention at the highest levels of government is urgently needed. Modern society is seriously concerned about solving problems of mining ecology and industrial safety, on which the well-being of the current and future generations involved in subsoil use depends [5].

Thus, subsoil use objects are the main cause of negative impacts on the environment and mining safety. Neutralization of such impacts during the modernization of the mining industry will play a positive role in the formation of a strategy for the development of the mineral resource base; it requires, on the one hand, a certain organization of production and, on the other hand, the use of a method of rational development of subsoil resources, taking into account the problems of mining ecology and industrial safety, which is positive will influence the choice of priorities for the development of the mining industry.

Negative impact of industry is expressed in the impact on specific parts of nature and on the biosphere in whole from processes of extraction and processing of natural resources. Production and consumption waste are sources of anthropogenic environmental pollution on global scale and arise as inevitable result of consumer attitudes and unacceptably low resource utilization rates [6].

For example, in the USSR, non-ferrous metallurgy consumed about 2 billion tons of rocks per year, and commercial output accounted for 1%. In the Russian Federation, one way or another, 90-95%, or from 80 billion to 120 billion tons, end up as waste. More than a billion of them are toxic. Every year, the area occupied by waste increases by 250 thousand hectares. The main suppliers of waste are the mining, chemical, metallurgical, fuel and energy industries.

Depending on the possibility of use, distinction is made between recyclable and non-recyclable waste. For the former, there is technology for processing and inclusion in economic circulation, for the latter it is currently not available. Classification of industrial waste [7] wastes are often chemically heterogeneous, complex polycomponent mixtures of substances with various chemical and physical properties, presenting toxic, chemical, biological, corrosive, fire and explosion hazards.

Number of waste classifications are known in modern scientific and technical literature N.F. Reimers [8] divides waste into industrial, agricultural, construction and municipal. M.E. Pevzner [4] considers only MMC waste and divides it into solid, liquid and gaseous. Ch. Mantell [9] identifies 6 classes of waste: agricultural, livestock, metallurgical, construction, chemical and municipal. The works of Z.A. Estemesov and T.K. Sultanbekov [10] proposed classification of urban construction waste into 13 groups.

In this work, to understand mechanism of occurrence, control, and management, as well as to eliminate shortcomings noted above, classification of production and consumption waste was made according to physical condition, by sources, chemical and morphological composition, as well as by industries and regions (Figure 2).

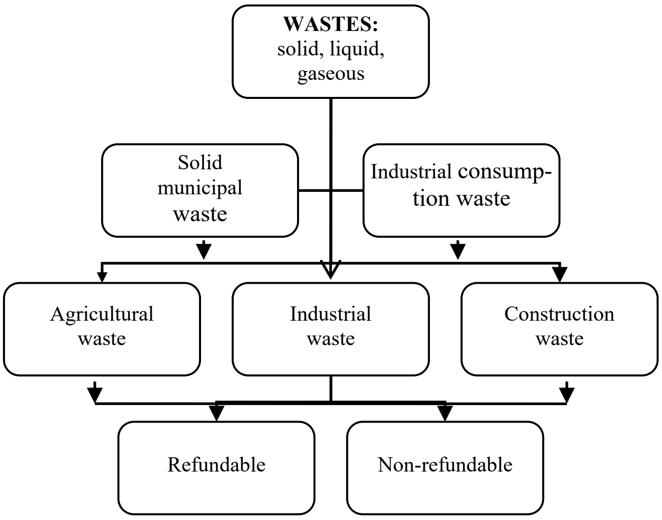



***Figure 2. Classification of waste by characteristics physical condition and education***

The main causes of waste generation in the mineral resource sector are multifactorial and systemic in nature [11]. They can be summarized as follows:

– irrational management practices, which have become the norm for many enterprises that continue to operate with outdated production and processing technologies;

– an obsolete regulatory framework, which does not correspond to the modern requirements of environmental protection and circular economy principles;

– ineffective supervision and control by central and local environmental and health authorities, as well as by other sectoral government bodies responsible for monitoring waste management practices;

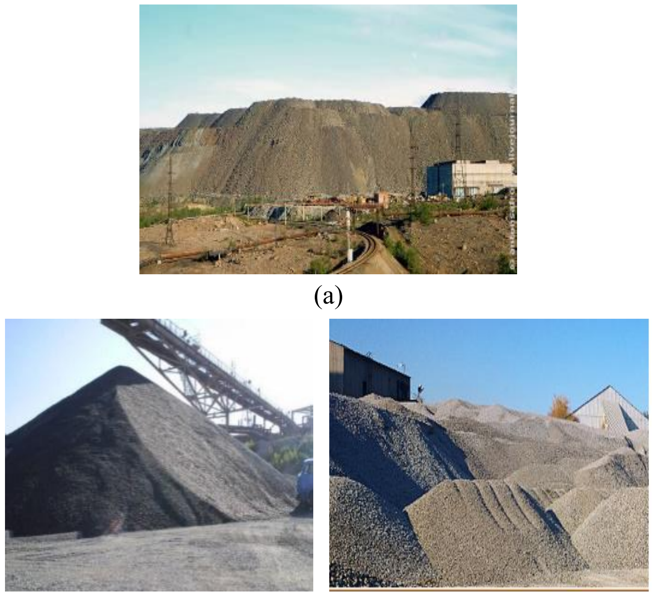
These factors collectively hinder the transition to sustainable resource management and lead to the accumulation of large volumes of industrial and mineral processing waste, creating both environmental and economic challenges for the country. The absence of a specialized law that would comprehensively regulate relations in the field of waste generation, utilization, and disposal. Most cities with a population of over 100 thousand inhabitants, especially megacities like Almaty, are experiencing an environmental crisis due to production and consumption waste.

The high concentration of population in megacities leads to increase in anthropogenic load on environment, which consists in increasing the production and consumption of various energy values. This leads to environmental pollution and disruption of ecology of the urban landscape. Disposal of waste generated by the city to landfills leads to pollution and irrational use of land, the atmosphere, surface and ground waters, increased transportation costs and irretrievable loss of valuable materials and substances.

The solution to this problem is currently urgent due to the manifold increase in the volume of solid waste. In this regard, the end of the twentieth century was marked by the holding of several International Environmental Congresses in the following countries: Brazil, Canada, Japan, Portugal, Sweden, and Russia. Several laws and codes have been adopted in Kazakhstan, which confirms the global nature of environmental problems associated with the lithosphere, atmosphere, hydrosphere and biosphere. The scientific foundations and practical recommendations for environmentally sound management of industrial waste are formulated in the works of scientists from Kazakhstan and abroad [12-17].

However, despite certain achievements in this area, the problems of waste burial, storage, neutralization, and disposal remain among the most critical environmental challenges. Insufficient attention is still paid to the processing and reuse of mining, construction, and municipal waste. Over many decades, the mining and metallurgical complex of the Republic of Kazakhstan has accumulated vast amounts of overburden materials, enrichment tailings, and metallurgical slags. As a result, millions of tons of harmful substances are annually released into the atmosphere, while hundreds of millions of cubic meters of contaminated wastewater are discharged into natural water bodies.

The accumulation of such waste not only leads to the degradation of landscapes and soil contamination but also poses risks to groundwater quality, biodiversity, and public health. Furthermore, the inefficient management of industrial waste represents a significant economic loss, as many of these materials contain valuable components that could be recycled or reused within the framework of a circular economy. Addressing this issue requires the development of innovative technologies for waste utilization, including the creation of construction materials and binder mixtures based on tailings and other by-products of mineral processing. (Figure 3).



(a)

(b) (c)

***Figure 3. General view of industrial waste: (a) – overburden dumps formed during open-pit mining; (b) – accumulation of ore beneficiation tailings near processing facilities; (c) – storage of metallurgical slag and coarse waste materials at industrial sites.***

Scale of the impact of industrial waste on environment is comparable to geological processes. Large-tonnage industrial wastes are accumulations of waste from the mining, metallurgical, energy and chemical industries containing useful components or minerals. The bulk of mining waste in Kazakhstan is generated in mining enterprises (73%), processing plants (25%) and metallurgical plants (2%). According to modern estimates, the enterprises of the mining complex of Kazakhstan have accumulated over 50 billion tons of industrial waste and occupy vast territories (more than 200 km$^2$ of area). Every year the amount of industrial waste increases by approximately 1,5 billion tons, and at the same time, the level of use of solid waste is currently low. The total reserves and areas occupied by tailings dumps within the regions are given in Table 2.

*Table 2. Processing waste by regions of Kazakhstan*

| Regions | Quantity | Reserves, thousand tons | Area km$^2$ |
|---|---|---|---|
| Akmola | 11 | 76834.50 | 12.30 |
| Aktobe | 8 | 30675.30 | 6.30 |
| Almaty | 5 | 47914.90 | 2.99 |
| East Kazakhstan | 39 | 887914.57 | 19.57 |
| Zhambulskaya | 6 | 44188.93 | 1.58 |
| Karaganda | 37 | 2809342.13 | 89.20 |
| Kostanayskaya | 4 | 611101.70 | 27.45 |
| Pavlodar | 2 | 8770.86 | 1.23 |
| South Kazakhstan | 5 | 142355.30 | 3.52 |

Moreover, the largest reserves are concentrated in tailings dumps. The need to involve enrichment tailings in production is dictated by the following circumstances:

– service life of tailings dumps is limited; the filling of many has already been completed or will end in the coming years;

– tailings occupy vast territories and, because they are finely dispersed and easily blown away material, are source of increased environmental risk for regions where mining and processing complexes operate.

Since enrichment waste is finely ground product that does not require additional grinding before use, this reduces economic costs. In addition, the process of ore enrichment ensures the homogeneity of the material both in chemical and mineralogical composition. The total reserves of mining and processing waste of some large mining enterprises in Kazakhstan are given in Table 3.

*Table 3. Reserves of technogenic waste at large mining enterprises*

| Name of the enterprise | Reserves, thousand tons | |
|---|---|---|
| | Technogenic dumps | CP dumps |
| JSC "Achpolitmetal" | – | 142570.1 |
| Belogorsky GOK | 24406.0 | 10067.8 |
| Donskoy GOK | 81447.7 | 38280.4 |
| Kazakhmys LLP | 973114.7 | 1674691.5 |
| Zhairemsky GOK | 6354.8 | 3188.8 |
| Tekeli Mining and Processing Plant | 15723.9 | 40360.5 |
| JSC "Kazzinc" | – | 373147.1 |
| Zhezdinsky GOK | 89.7 | 3173.2 |
| JSC "Kostanay Minerals" | – | 2038.3 |

According to the bodies of State Control and Supervision of Natural Resources, the share of waste used in the republic is currently 18-20%. It remains extremely low compared to world practice. In Western Europe (France, Germany, Italy, England) this figure is up to 58%, in North America (USA, Canada) – up to 63%, in Japan – up to 87%, China – up to 37% [18].

Accumulated waste is, on the one hand, the main pollutant of environment, and on the other hand, it represents valuable products that are potentially suitable for processing and reuse to obtain popular building materials.

The need for development (processing) of technogenic mineral formations is dictated by the fact that they occupy vast territories, easily blown away material, and are a source of increased environmental risk for the mining and metals industry regions. The constant increase in the volumes of various types of waste generated in the mining and processing industries and their storage in storage facilities and the experience of using such objects in industry allows us to consider them as sources for obtaining secondary raw materials and building materials.

Technologies development for production of building materials based on technogenic waste, contributing to development of industrial and innovative potential of state, careful attitude towards natural resources and the environment, should be considered as the most important scientific and practical task, the solution of which is directly related to environmental safety when disposing of billions of tons of waste in industrial regions [19, 20].

## 4. Conclusions

The conducted research confirms that waste generated by mining and metallurgical complexes (MMC) and thermal power plants (CHP) can serve as a valuable secondary raw material for various industries, particularly for the production of construction materials. These technogenic formations, traditionally considered an environmental burden, actually represent a significant reserve of useful components suitable for reuse. Their rational involvement in industrial circulation not only reduces the area occupied by dumps and tailings but also prevents the accumulation of new waste, contributing to the transition toward waste-free and environmentally sustainable production.

The technological benefit of such an approach lies in the implementation of modern waste-processing methods within the framework of Kazakhstan's national program for industrial and innovative development. Economically, it allows for the production of additional building materials and related products, expanding the raw material base of the construction industry. From a social perspective, this process stimulates the creation of new jobs in regions adjacent to mining and metallurgical enterprises, improves working conditions, and enhances environmental safety.

The environmental effect is particularly significant: it manifests in the reduction of land areas allocated for dumps, the restoration of disturbed ecosystems, and the mitigation of harmful emissions and discharges. Thus, the integrated recycling of man-made waste contributes simultaneously to resource efficiency, economic diversification, and the improvement of the ecological situation in industrial regions of Kazakhstan.

## Author contributions

Conceptualization: MMB, MBN; Data curation: TBN, DMK; Formal analysis: TBN, DMK; Funding acquisition: YIK, MBN; Investigation: YIK, MBN; Methodology: MMB, MBN, TBN; Project administration: MMB, YIK; Resources: MBN, TBN; Software: DMK; Supervision: MMB, MBN; Validation: MBN, TBN; Visualization: MBN, DMK; Writing – original draft: MMB, MBN; Writing – review & editing: MBN, TBN. All authors have read and agreed to the published version of the manuscript.

## Funding

This research was funded by grant funding from the Ministry of Science and Higher Education of the Republic of Kazakhstan BR21882292 «Integrated development of a sustainable construction industry: innovative technologies, production optimization, efficient use of resources and creation of a technology park».

## Acknowledgements

The authors sincerely thank the editor and reviewers for their insightful comments and valuable recommendations, which greatly contributed to enhancing the clarity and overall quality of this manuscript.

### Conflicts of interest

The authors declare no conflict of interest.

### Data availability statement

The original contributions presented in this study are included in the article. Further inquiries can be directed to the corresponding author.

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# Жер қойнауын игерудегі экологиялық және өнеркәсіптік қауіпсіздіктің рөлі


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**Аңдатпа.** Мақала Қазақстан Республикасы аумақтарының экологиялық проблемаларын қарастырады. Олар техногендік қалдықтар (ТҚҚ) санының ұлғаюымен, үйінділер мен полигондар алып жатқан аумақтардың өсуімен, сондай-ақ қоршаған ортаға теріс әсерімен бірге жердің минералды ресурстарының қолда бар қорларының азаюымен байланысты. Жинақталған қалдықтар, бір жағынан, қоршаған ортаны ластаушы болып табылады, ал екінші жағынан, сұранысқа ие құрылыс материалдарын алу үшін қайта өңдеуге және қайта пайдалануға болатын құнды өнімдер болып табылады. Техногендік минералды түзілімдерді игеру (қайта өңдеу) қажеттілігі олардың кең аумақтарды алып жатқандығына, желдің оңай үрленуіне және тау-кен және металлургия өнеркәсібі аймақтары үшін экологиялық қауіптің жоғарылау көзі болып табылатындығына байланысты. Осыған байланысты мақалада техногендік қалдықтарды экологиялық бағалаудың және оларды құрылыс материалдары мен бұйымдарын өндіруде қайта өңдеудің кешенді әдістемесінің ғылыми негіздері әзірленді, аумақтардың экологиялық мәселелерін бір мезгілде шеше отырып, аталған қалдықтарды оларды өндіру үшін шикізат ретінде пайдалану мүмкіндігі негізделді. Техногендік қалдықтарды кәдеге жарату әдістері және ресурс үнемдеуші технологияларды енгізу перспективаларын сандық бағалау ұсынылған.



***Негізгі сөздер:*** *аумақтардың экологиялық проблемалары, ресурстық тәсіл, техногендік қалдықтарды кәдеге жарату, құрылыс материалдары мен бұйымдары, қалдықтардың көлемі.*

# Роль экологической и промышленной безопасности при освоении недр

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**Аннотация.** Статья рассматривает экологические проблемы территорий Республики Казахстан. Они связаны с сокращением доступных запасов минеральных ресурсов Земли при одновременном увеличении количества техногенных отходов (ТГО), ростом площадей, занимаемых отвалыми и полигонами, а также отрицательным воздействием на окружающую среду. Накопленные отходы, с одной стороны, являются основным загрязнителем окружающей среды, а с другой — представляют собой ценные продукты, потенциально пригодные для переработки и повторного использования для получения востребованных строительных материалов. Необходимость разработки (переработки) техногенных минеральных образований диктуется тем, что они занимают обширные территории, легко раздуваются ветром и являются источником повышенного экологического риска для регионов горнодобывающей и металлургической промышленности. В этой связи в статье разработаны научные основы комплексной методики экологической оценки техногенных отходов и их переработки в производстве строительных материалов и изделий, обоснована возможность использования указанных отходов в качестве сырья для их производства при одновременном решении экологических проблем территорий. Представлены методы утилизации техногенных отходов и количественная оценка перспектив внедрения ресурсосберегающих технологий.

***Ключевые слова:*** *экологические проблемы, ресурсный подход, утилизация техногенных отходов, строительные материалы, отходы.*

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СОДЕРЖАНИЕ

***Учредитель:*** Satbayev University

***Регистрация:***
Министерство информации и общественного развития Республики Казахстан
№ KZ19VPY00056529 от 30.09.2022

**Официальный сайт**: https://vestnik.satbayev.university/index.php/journal/

**Основан в августе 1994 г. Выходит 6 раз в год**

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г. Алматы, ул. Сатпаева,
22 тел.: 292-63-46