Master thesis:

Substantiation of the Efficient Technological Scheme of Development of Motronivka Placer (Malyshev Titanium-Zirconium Ore Deposit)

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AFFIDAVIT

I declare on oath that I wrote this thesis independently, did not use other than the specified sources and aids, and did not otherwise use any unauthorized aids.

I declare that I have read, understood, and complied with the guidelines of the senate of the Montanuniversität Leoben for "Good Scientific Practice".

Furthermore, I declare that the electronic and printed version of the submitted thesis are identical, both, formally and with regard to content.

Date 27.11.2019

Signature Author
Stanislav, Vasylchuk

Acknowledgment

First of all, I would like to thank my dad for never-ending support and faith in me. Secondly, I would like to thank my colleges and friends for their readiness to shear experience on how to get the best results. And finally, I would like to thank all the staff of Montanuniversität Leoben and Bergakademie Freiberg for the opportunity to study on Advance Mineral Recourses Development program. I got all the necessary help and support from them during studies and it is hard to overestimate the advantages of this experience for my carrier and life.

Abstract

During the development of the Malyshev titanium-zirconium ore deposit in Ukraine, the richest parts were worked out. Under the conditions of increasing demand on titanium in the world and the development of technologies, it was decided to mine a new section to ensure the necessary supply of the existing processing plant by ore. The purpose of this thesis is to justify the most effective technological scheme for the Motronivka-Annivka section development. The most effective is a technological scheme in which investments and natural resources are used as efficiently as possible. Therefore, the method used in this thesis is a comparison of possible technological schemes of the field development by their capital and operational expenditures. In the thesis, schemes using the following mining equipment are considered: bucket-wheel excavator, dragline excavators and hydraulic excavators. When developing technological schemes, the organization of the interconnected operation of the equipment within the whole open-pit area was analyzed. Also, in this work, technical and economic dependencies were studied and analyzed when changing work parameters. These studies have helped to justify the choice of equipment and the working schedule of the mine. Furthermore, these calculations have allowed to study the modes of mutual work of equipment to achieve to ensure its maximum performance. Engineering drawings of whole mine work organization for all considered technological schemes are presented in Attachments for this thesis.

Keywords:

open-pit mine, titanium, technological scheme, equipment selection, performance, pit calculation program

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Introduction

According to the research report on development of the world market for titanium dioxide (TiO_2) for 2018 - 2025 presented by Grand View Research Inc.[1], it is expected to grow steadily by more than 8.9% until the 2025 year. This trend is due to escalating demand from end-use industries that use titanium dioxide as a pigment. [1]

Titanium alloys with various metals (aluminum, molybdenum, iron) are as strong as steel. They are used in aircraft, spacecraft and missiles manufacturing owing to its low density and ability to resist extreme temperatures. Also, titanium alloys are used in medicine, ship and submarine construction, computer manufacturing etc. [2]

The Malyshev titanium-zirconium ore deposit consists of titanium minerals as follows: ilmenite, leucoxene, rutile, zircon, staurolite and tourmaline, as well as quartz sands suitable for use in foundry. As a result of depletion of the resources in western and central sections (developed by Vilnohirsk state mining and metallurgical plant since 1959), a mining project in the Motronivka-Annivka section has been implemented.

The objective of this work is to achieve the priority goal of mining planning: to develop the most cost-effective technological scheme of development of the Motronivka placer Malyshev titanium-zirconium ore deposit, which should:

- maximize the return of the monies invested, and
- maximize the recovery of the resource. [3]

In the present work comparative analysis of 3 different schemes of development of the deposit has been done:

1) Using a bucket-wheel excavator and draglines on lower benches;
2) Using draglines;
3) Using hydraulic excavators and draglines on the bottom bench.

[1]Grand View Research Inc. – American company for market research in the field of chemical, mineral, energy industry.

Chapter 1 General Information and Geology

This chapter includes the necessary data about Malyshev ore deposit for the analysis of technological schemes. All information in this chapter (except for figures) was taken from the report on geological surveys of Malyshev titanium-zirconium ore deposit [4].

1.1 General Information

Malyshev ore deposit of titanium-zirconium ores is located on the right bank of Dnieper River's middle current, in Samotkan River's riverhead, therefore its first name – Samotkanskoe.

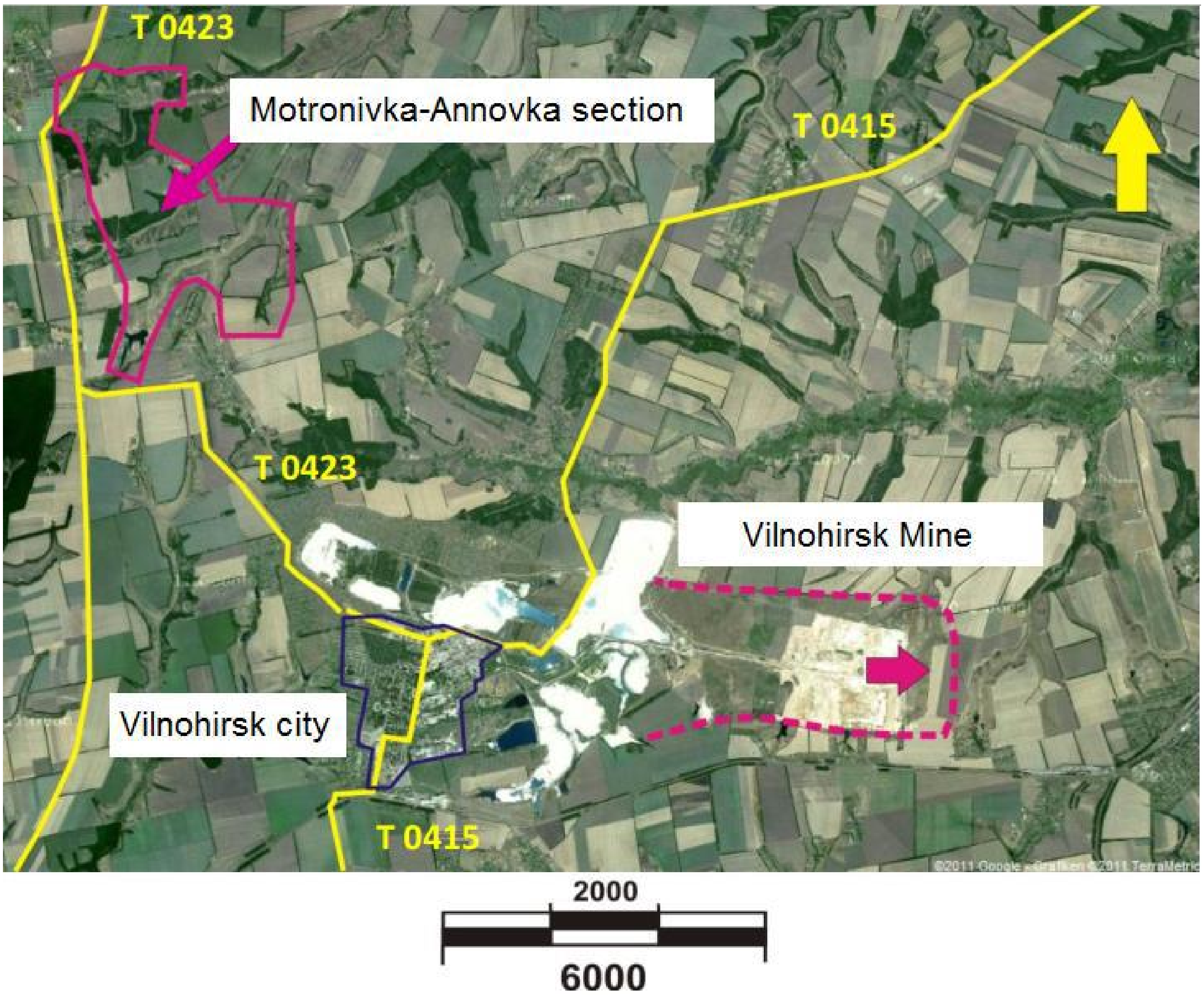


Figure 1 Satellite picture of Vilnohirsk city and the Motronivka-Annivka section [5]

Administratively, the deposit is located in the southeastern part of the Verkhnodniprovsk Raion, the north-eastern part of Piatykhatky Raion of Dnipropetrovsk Oblast. The total area is about 39 km^2.

The Motronivka-Annivka section is located in the northwestern part of the field, 1.5 -2.0 km from the Western section (Figure 1).

Along the southern border of the field passes the high-capacity double-track railway Kyiv-Dnipro. The nearest railway station is Vilnohirsk. The adjacent road of class III Vilnohirsk-Lihovka crosses the site from south to north and passes along the western border of the planned open-pit field. At the southern border of the field there is a road Vilnohirsk-Dmitrivka-Verkhnodniprovsk.

The site itself is characterized by a complete lack of roads in the autumn-winter-spring period and during rains in summer.

Motronivka-Annivka section is a steppe area with a highly broken relief. Absolute elevations range from +178 at water-parting lines to +110 at valley bottoms.

Figure 2 Views of the Motronivka-Annivka section [5]

About 15% of the field is occupied by rural areas (village Dmitrovka, Motronivka, Annivka). 50% of the site is arable land. The rest of the area is occupied by small flat-bottom valleys covered by meadows, pastures and planted forests (Figure 2).

The climate is temperate continental. The average annual temperature is about +8˚, the average in January is -6.5˚, in July +22˚. The average yearly rainfall is around 440mm, sometimes it reaches 540mm, in dry years it may be 254mm.

Geological exploration at the Malyshev deposit was carried out in 1955 – 1958 years. The general calculation of the resources of titanium and zircon in categories A + B + C1 was approved by Ukrainian Geological Administration in 1959.

Since 1962, Vilnohirsk mining and smelting complex is operating developing the approved resources of titanium-zirconium ores, and the city was built. Mining and production of concentrates at UMCC[2] were focused on stocks of industrial grades of the Sakmarian sub-panel. Over the entire period of mining, the richest West and Central sections have been developed.

The resources of C2 category of the North-Western zone are calculated also and approved in 1959 together with the industrial reserves of the Sakmarian sub-panel in amount of 841 million m^3 ore sands with an average content of collective concentrate of 55.11 kg/m^3, zircon 3,33 kg/m^3, rutile 8.04 kg/m^3 and ilmenite 32.58 kg/m^3.

It is planned to develop the resources of the North-Western zone of relatively poor stocks of titanium and zircon containing sand of the Poltava sub-panel. These areas were covered by a rare network (2000m x 200m) of exploration wells. In total, 76 wells were drilled in the Motronivka-Annivka area during this period.

Exploration work at the Motronivka-Annivka section was carried out (with some interruptions) by an order of the Ministry of Non-Ferrous Metallurgy of the USSR with the help of the state-owned exploration company and in coordination with the Ministry of Geography of the USSR in 1979-1990. Feasibility studies of permanent conditions for ore sands of the Poltava series were carried out by the Kryvyi Rih Design Institute in 2003-2004. Ukrainian Geological Administration approved final mining parameters for calculations of resources for these ore sends in 2005.

[2] UMCC – United Mining and Chemical Company

1.2 Topography

Typical topography of the Motronivka-Annivka section is numerous ravines, which are a result of erosion of river beds. The surface of the site is a wavy plain with levels from approximately +165 to +180 m. In the area of the three main ravines, which divide the plain from the south-west to the north-east, the relief drops to below +120 meters above sea level (Figure 3).

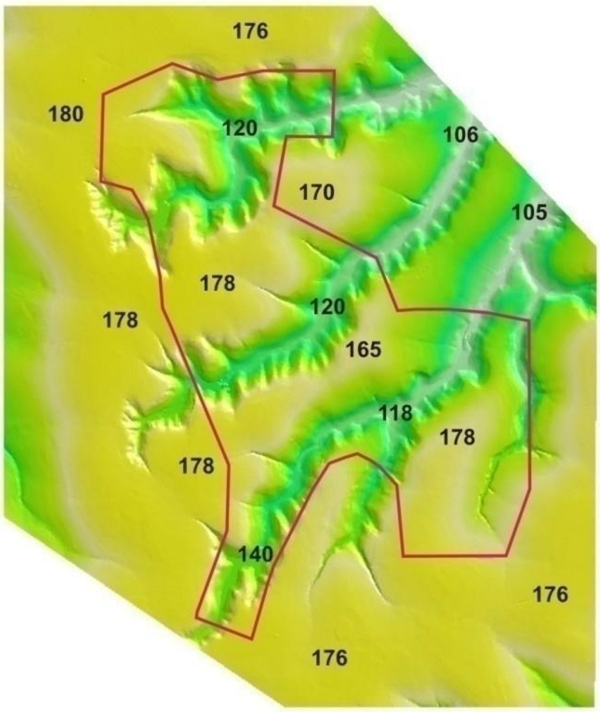


Figure 3 Topographic map of the Motronivka-Annivka section and a view on a typical ravine [5]

1.3 Geological Structure

The geological structure of the Motronivka-Annivka section is identical to the Malyshev deposit as it is an extension of the occurrence in north-west direction.

The most ancient measures situated at the base of the geological sheet consist of crystalline rocks of the Archaean rock system, represented mainly by gray plagiogranites and their migmatites. They lie mainly from 50 to 120 meters deep and are not exposed on the surface in the area of the section. These measures are situated most deeply along the southern border of the site, dropping to 25m above sea level, and are covered by loose sediments of the Paleogene Cenozoic system (Buchatska Series and Kyiv Stage sediments). The rise of the crystalline basement is observed in the northwestern and western margins of the site, where it rises to the levels of +94

+103m in and lies directly below the sandy deposits of the Poltava depositional sequence.

Crystalline rocks in the upper level are weathered and are covered by weathering crust up to 40m.

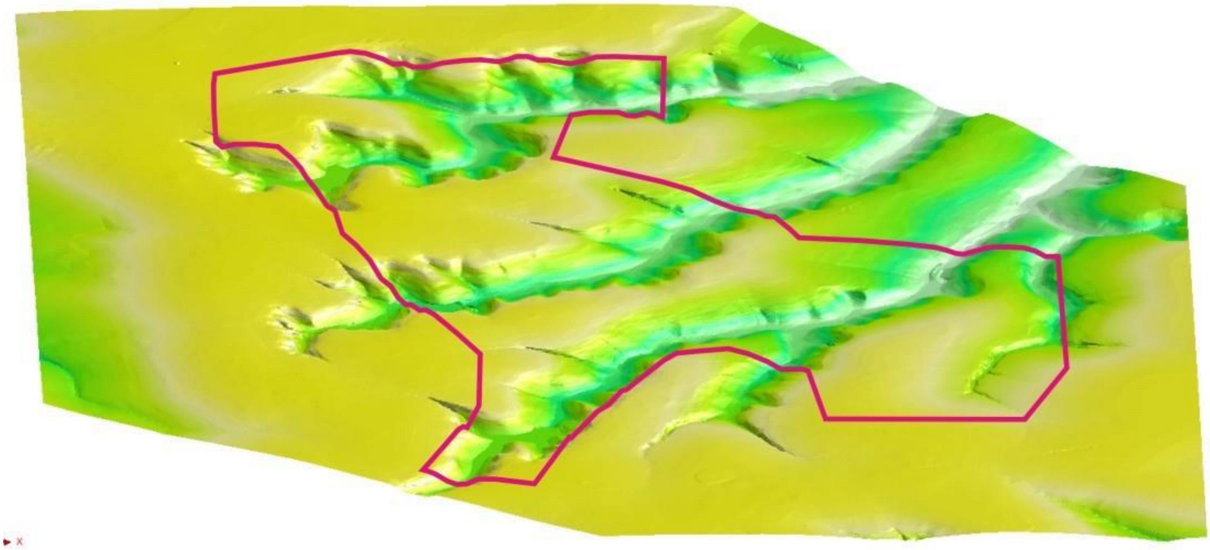


Figure 4 Side view on the Motronivka-Annivka section [5]

Buchatska Series sediments ***(P2bč)*** within the field have limited distribution and are elongated along the southern border. The total thickness of these deposits varies from 0 to 28 meters. These deposits are mainly represented by differently grained carbonaceous sands containing carbonaceous residues, gray plastic carbonaceous clays with interlayers of secondary kaolin and brown coal. The thickness of the lignite layer in some places in the eastern part reaches 5 meters.

Buchatska sediments are underlying by **Kyiv Stage** sediments ***(P2kv)*** (Upper Eocene). It is represented by a uniform layer of sandy-argillaceous sediments and have limited distribution as well. Their total thickness ranges from 0 to 15 m.

Deposits of **Kharkiv Stage** sediments ***(P3hr)*** (Upper Paleogene) have an almost continuous distribution and are absent only in the north-western part of the site. The thickness of the Kharkiv Stage is 15–18 meters. It is represented mainly by bright green glauconite quartz fine granular sands with a significant predominance of glauconite in the upper part and insignificant content in its lower part.

The mineralogical composition of the uppermost horizon of the Kharkiv Stage is characterized by a predominance of pyrite, ilmenite, leucoxene, muscovite, and garnet. Sillimanite, zircon, staurolite, and rutile are present in insignificant amounts.

The mineral content of the heavy fraction in the sands of the Kharkiv Stage does not exceed 0.5%. Only in the central part of the site the content of the heavy minerals rises to 3.5%.

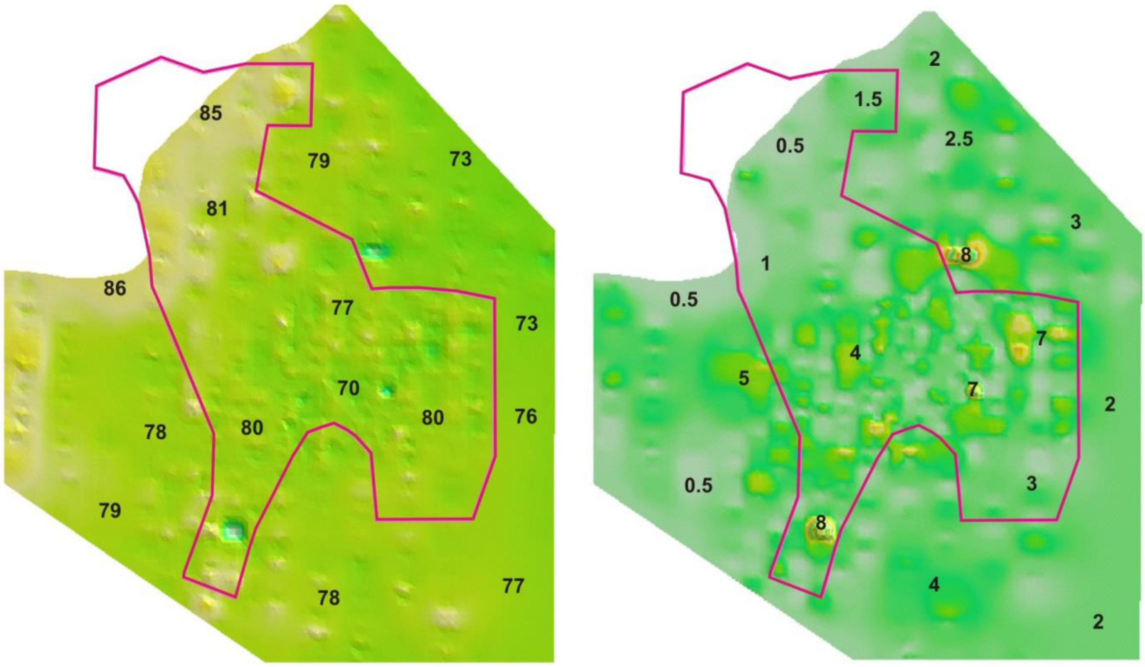


Figure 5 Kharkiv Stage: 1 - map of the bottom level [m.a.s.l]; 2 – map of thickness [m] [5]

Miocene deposits, represented by quartz sands of **Poltava Stage** sediments ***(N_1pl)***, overlie the sediments of the Kharkiv Stage (Figure 5). In a small area in the western part of the site, the sands of the Poltava series occur directly on the weathered crust of crystalline rocks. The contact of the sands of the Poltava Stage with the underlying sediments of the Kharkiv Stage is gradual. Three horizons are conventionally distinguished within the Poltava Stage and are evenly distributed across the field.

The lower horizon, with an average thickness of about 6m, is represented by greenish-gray fine-grained sand with glauconite. This horizon lies directly on the greenish-gray glauconite–quartz sand of the Kharkiv Stage and has a gradual transition with them.

In the north-western part of the site, where underlying rocks are weathered crust, this horizon is absent. The lower horizon of the Poltava Stage has an increased content of sillimanite, ilmenite, staurolite, tourmaline, rutile, zircon in the heavy fraction. Between the minerals listed, the concentration of pink garnet is observed in the heavy fraction and reaches in some samples 19.7%. The average content of heavy minerals in the samples of the lower horizon of the Poltava Stage does not exceed 0.3–0.6%. Still, there are separate isolated interlayers and lenses up to 3m thick, especially in the eastern part of the site, where concentration of heavy minerals sometimes reaches 4.0–4.7%. The number of such interlayers and lenses increases to the east, but they do not have industrial value.

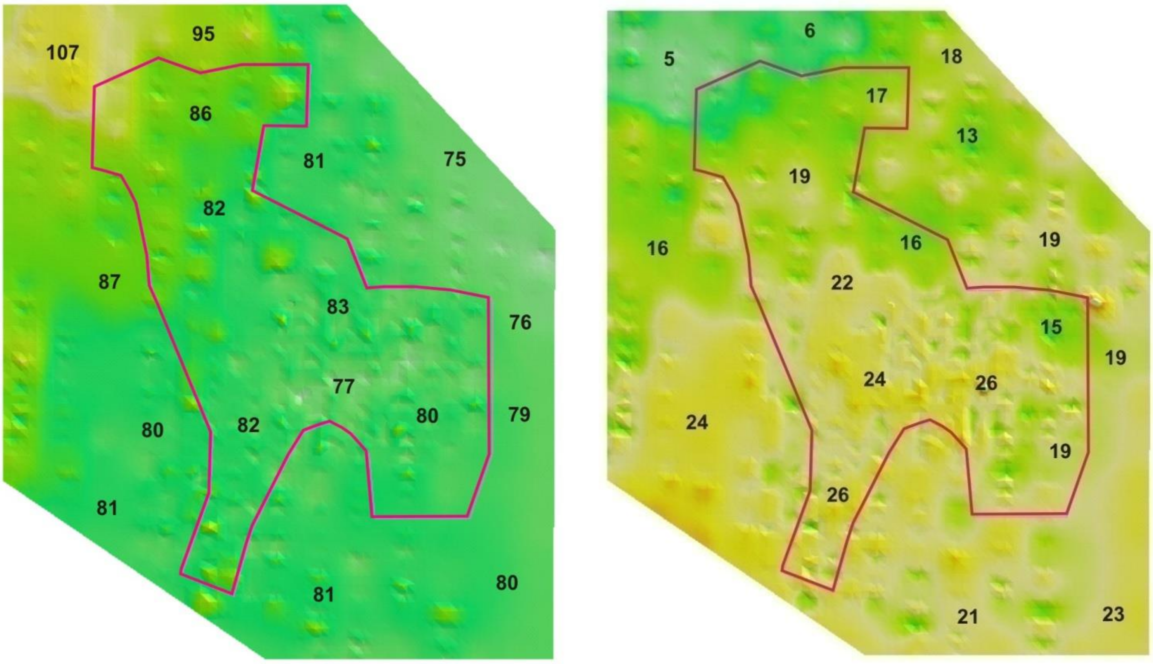


Figure 6 Poltava Stage: 1 - map of the bottom level [m.a.s.l]; 2 – map of thickness [m] [5]

Sometimes ore interlayers occur directly in contact with sediments of the Kharkiv Stage, in some places are rising up to 3-5m and gradually passing into the middle layer. The width of these interlayers is 500-600m and length – 500-1000m.

The middle horizon of the Poltava Stage lies above the greenish-gray sand of the lower horizon and has a gradual transition with it. It is represented by yellowish-gray fine quartz sand. Its thickness ranges from 6m to 8m in the eastern part, and 10-15m in the middle and western parts of the site. Separate lenses and interlayers, including ore minerals, the content of which reaches industrial one, are associated with this horizon. In some places, these interlayers and lenses are connected with an underlying reach area, forming a single enriched zone. Most of the rich interlayers and lenses were found in the south-eastern part of the site.

In general, middle horizon's sand is characterized by a low content of heavy minerals, usually not more than 5–10 kg/m^3. In enriched layers and lenses, the content of heavy minerals ranges from 17 to 73 kg/m^3. On average, it does not exceed 35-40 kg/m^3. Among minerals of Poltava Stage's heavy fraction prevail: ilmenite, rutile, kyanite-sillimanite, staurolite, tourmaline, zircon. Ratio between these minerals is: zircon - 4.4%, rutile - 14%, ilmenite - 45.5%, kyanite-sillimanite - 23.8%, tourmaline - 5.0%. Thickness of ore-bearing interlayers and lenses of the middle horizon varies from 2m to 12m (average 4.05m), with an average thickness of barren interbed 6.3m.

The upper horizon of the Poltava Stage sediments is represented by fine-grained quartz variegated cross-bedding sand. The main ore is present in this horizon. Its minimum thickness is 3.5m and the average - 7m.

The upper horizon of the ore-bearing sand lies above the middle horizon with absolute marks from +79m to +95m in the eastern part, and from +95m to +108m in the west. The elevation ranges from +89m to +105m in the east and from +105m to +114m in the west, respectively.

The upper ore-bearing horizon has a clear upper boundary, where fine-grained sands of the Sarmatian Stage cover it. There are exceptions in some wells in the western part of the site, where a poor layer of the Poltava Stage's quartz sand lies on the roof of ore-bearing sands. The maximum thickness of this interlayer does not exceed 6m.

The bottom of the upper horizon is distinguished by a cut-off grade of ilmenite 18 kg/m^3 for commercial reserves. The average content of the ore minerals range within this horizon in the whole area is 67.3 kg/m^3, including: zircon - 5.35 kg/m^3, rutile - 10.62 kg/m^3, ilmenite - 35.98 kg/m^3, staurolite - 9.87 kg/m^3, kyanite-sillimanite - 2.72 kg/m^3 and tourmaline - 2.76 kg/m^3. Spinel, chromite and monazite are present in quantities of single grains. The content of heavy minerals in separate samples ranges from 17.02 kg/m^3 to 399.22 kg/m^3. In general, the content of useful components is somewhat smoothed out and ranges from 21.5 kg/m^3 to 168.4 kg/m^3.

The dependence of the presence of useful minerals within the ore formation on the content of the heavy fraction is observed quite clearly.

In terms of lithology, sand of the Poltava Stage is uniform. Average grain composition is shown in Table 1.

Table 1 Grain composition of the ore containing sand within the Motronivka-Annivka section [4]

Size, mm	**>0,560**	**-0,560 +0,400**	**-0,400 +0,315**	**-0,315 +0,200**	**-0,200 +0,160**	**-0,160 +0,100**	**-0,100 +0,07**	**<0,071**	**sum**
Content, %	0,50	1,15	1,30	5,83	11,21	53,26	24,59	2,16	100

The light fraction of sand is represented by quartz grains (87% - 88%) and clay minerals (12 - 13%). Negligibly there are feldspars and mica. Quartz grains are mostly small and poorly rounded. The heavy minerals are smaller than quartz grains, as is shown by particle size analysis of concentrate.

The size of grains of the heavy fraction is less than quartz and it is mainly -0,100 to +0,056. The average grain size of ore minerals was calculated for 270 samples and is in mm: zircon - 0.0654, rutile - 0.0783, ilmenite - 0.0755, kyanite-sillimanite-0.0879, staurolite 0.0878, and tourmaline - 0.0888, the average grain size of quartz is 0.140 mm.

The Sarmatian Stage sediments ***(N_1s_2)*** is bedded above the deposits of the Poltava Stage on the entire area of the site (Figure 7). Their continuous distribution is interrupted only at places of gullies, especially in the northern part of the site.

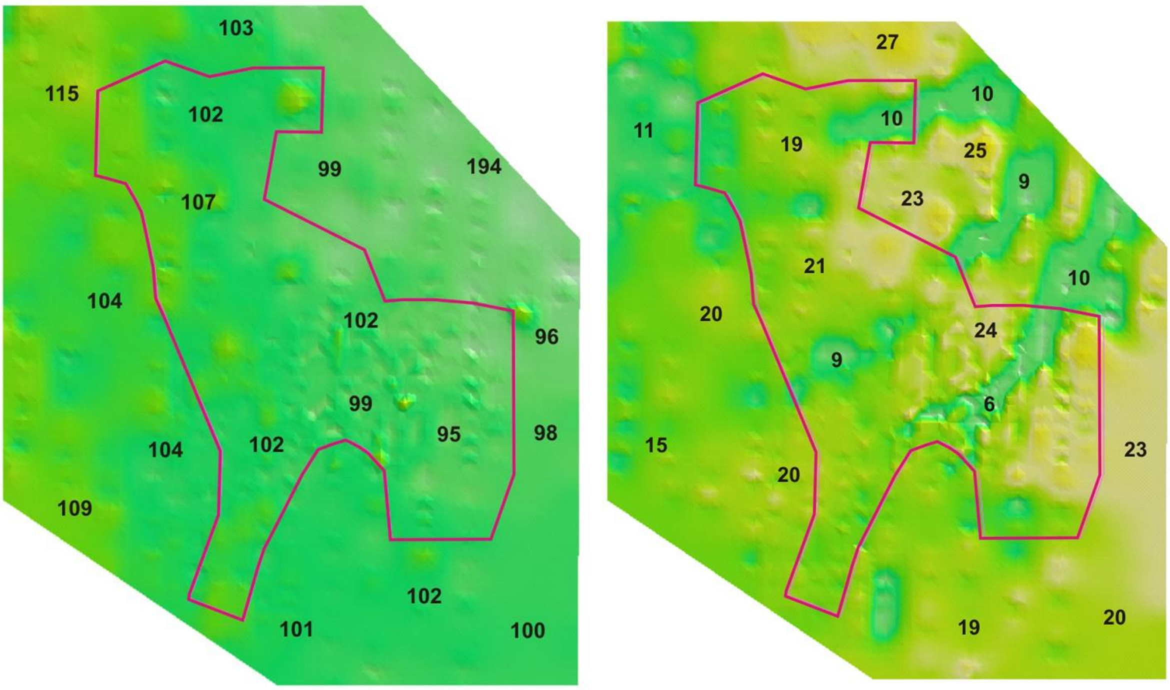


Figure 7 Sarmatian Stage: 1 - map of the bottom level [m.a.s.l]; 2 – map of thickness [m] [5]

The contact of the Sarmatian sands with underlying sediments of the Poltava Stage is slightly wavy, almost horizontal. Absolute marks of the bottom of the Sarmatian sands range from 89m to 114m. Thickness of this layer varies from 6.6m to 27.0m, with an average thickness of 16.7m. Lithologically, it is represented by fine-grained and medium-grained quartz sand with clay content ranging from 8-10% to 35-

40% and ore mineral content ranging from 0.1 to 0.7% with some exceptions, when the concentration of heavy minerals in the middle of the horizon rises up to 2-3%. Thickness of the layers with the heavy mineral content of more than 1% ranges from 0.9m to 7.7m, on average 2.8m. Rise of the heavy fraction within the Sarmatian Stage was found at the northwestern and southwestern flanks of the site. Ore interbeds in the Sarmatian sand occurring directly on the ore sand of the Poltava Stage with ilmenite content of 18 kg/m^3 and more.

The content of clay minerals in the Sarmatian Stage sands increases from bottom to top, forming a gradual transition to the greenish-gray variegated gypsum-bearing clay of the upper part. Thickness of this clay body is not uniform and decreases from 10–12 to 5–7m in the Eastern direction. The clay is dense and viscous. Therefore, the content of sand particles in it decreases from bottom to roof in reverse. Presence of clusters of individual gypsum crystals is characteristic.

Greenish-gray clay on the water parting spaces is covered with a thick layer of **Quaternary sediments *(Q)***. At the base of this stratum, red-brown clay has a gradual transition to the horizon of greenish-gray clay.

Both kinds, red-brown and greenish-gray clays, contain gypsum druses and carbonate concretions. Thickness of these clays ranges from 0 to 25m. Above the red-brown clay, having with it a gradual, barely perceptible transition, there are dense red-brown loams, 5-10m thick. It is present all over the site. Dense yellow-brown loams represent the upper part of the Quaternary stratum with small dense calcareous inclusions and pale-yellow loess-like light loams with a total thickness up to 20m.

An entire thickness of the Quaternary sediments and sands of the Sarmatian Stage are weathered to varying degrees along the beams during the formation of the modern relief. Within the valleys and beams, the modern Quaternary sediments are represented by diluvial loams with rare layers of diluvial sands. Their thickness ranges from 0 to 10m.

Within the site, erosion activity is well developed. Difference between the topographic elevation of the water-parting lines and thalwegs of the beams reaches 50-60m. The slopes of the beams are steep, turfy and usually covered by 1-10m of loam with rare layers of diluvial-alluvial sands.

1.4 Ore Qualitative and Processing Characteristics

The seam of ore sand is present at the upper horizon of the Poltava Stage and the lower part of the Sarmatian Stage. The ore horizon is represented by quartz gray or yellow-gray, occasionally dark gray, fine sand with a high content of heavy minerals.

The mineralogical composition of sand includes quartz, clay minerals, ilmenite, kyanite-sillimanite, rutile, zircon, staurolite, tourmaline, leucoxene, monazite, spinel, magnetite, chromite, limonite, carbonates, mica, manganese oxides, garnet, feldspars. The main components of the ore sand are quartz (80-87%) and the clay fraction (12-19%).

Table 2 Granulometric composition of the ore sand (results of various laboratories) [4]

Laboratory	Sampling material	Fraction size, mm									
		>0,560	-0,560 +0,400	-0,400 +0,315	-0,315 +0,200	-0,200 +0,160	-0,160 +0,100	-0,100 +0,071	-0,071 +0,056	<0,056	sum
UMCC 1979 - 1990	Mineral with tailings (289 samples)	0,50	1,15	1,30	5,82	11,21	53,26	24,45	2,16	0,14	100
UMCC 1979 - 1990	Samples without tailings	-	0,91	0,77	10,44	23,93	58,23	5,04	0,57	0,11	100
State-run enterprise «Yuzhukrgeologiy» 1981	Samples without tailings	0,49	0,88	0,89	4,05	8,53	67,03	17,71	0,27	0,15	100
UMCC	Heavy fraction from (7 samples)	-	-	0,01	0,29	1,17	9,12	33,59	50,31	5,50	100
«Giredmet»		>0,560	-0,560 +0,280	+0,280 -0,200	-0,200 +0,140	-0,140 +0,100	-0,100 +0,074	-0,074 +0,044	<0,044	sum	
«Giredmet»	Mineral with tailings	0,44	2,51	6,3	19,49	48,48	6,92	2,42	13,45	100	
«Giredmet»	Minerals without tailings	0,51	2,88	7,27	22,50	56,02	8,02	2,80	-	100	
Right-bank GRE 1955 - 1958		-0,800 +0,500	-0,500 +0,250	-0,250 +0,100	-0,100 +0,050	-0,050 +0,010	-0,010 +0,050	-0,005 +0,001	<0,001	sum	
Right-bank GRE 1955 - 1958	Mineral with tailings (30 samples)	0,01	3,61	20,73	49,21	6,44	7,27	3,94	8,19	100	
Right-bank GRE 1955 - 1958		>0,280	-0,280 +0,180	-0,180 +0,125	-0,125 +0,100	-0,100 +0,071	-0,071 +0,040	<0,040	sum		
Right-bank GRE 1955 - 1958	Concentrate	0,57	3,07	12,26	25,77	44,45	13,88	-	100		

The average content of the light fraction is 96% (including 82% sand and 14% clay), and the heavy fraction 4%. Dry sand has a volume weight 1.76 t/m^3 and can be easily loosened.

The minerals of the heavy fraction sands can be divided into electromagnetic and non-electromagnetic groups. Ilmenite, staurolite, tourmaline, chromite, magnetite, spinel, monazite and partially leucoxene have electromagnetic properties. Non-electromagnetic are rutile, zircon, sillimanite, most of the leucoxene, rarely andalusite, viridine.

The Poltava and Sarmatian sands are similar to each other by complex of heavy minerals. The distribution of ore minerals in the sands is uneven both in the horizontal and vertical dimensions. The maximum content is usually at the top and the middle part of the seam. There is a general decrease in the contents in the northwestern and southeastern directions. The western part of the site is divided by zone of barren sand.

Main ore minerals of the industrial importance are: ilmenite, rutile, zircon, kyanite-sillimanite and staurolite. The rest have not industrial value, due to its minor contents.

Table 3 Distribution of the main minerals by their fractions [4]

Fractions, mm	Content in %					
	Zircon	Rutile	Ilmenite	Disthene	Staurolite	Tourmaline
-0,315+0,200	-	-	0,02	0,13	0,25	-
-0,200+0,160	0,28	0,46	0,46	1,11	3,70	1,04
-0,160+0,100	6,83	11,97	8,46	12,76	20,99	11,75
-0,100+0,071	21,59	29,00	26,14	22,53	30,12	38,38
-0,071+0,056	42,80	45,97	52,89	27,02	42,72	44,91
-0,056+0,040	28,50	12,59	12,03	36,46	2,22	3,92
Average size, mm	0,0654	0,0783	0,0755	0,0879	0,0878	0,0888

The ratio of the contents of the main ore minerals over the site is fairly constant. The average grades of ore minerals in the heavy fraction are: ilmenite 55.4% (45.78 kg/m^3), rutile 15.6% (12.95 kg/m^3), zircon 8.2% (6.8 kg/m^3), disthen-sillimanite 13.6% (11.23 kg/m^3), staurolite 3.8% (3.15 kg/m^3).

Granulometric analysis of initial ore sand (without the clay component) was carried out by various organizations that had different sets of sieves available (Table 2). The average grain size of the sand varies slightly (from 0.126 to 0.146 mm), with a tendency for the size increase from south-east to north-west and north. The heavy fraction grains are almost two times smaller than quartz ones, on average 0.078 mm (Table 3). The particle size distribution of the heavy fraction is fairly constant.

Ilmenite is the most common titanium mineral in the sands of both the Poltava Stage and the Sarmatian Stage. The mineral is represented by flattened-tabular semi-rolled, less often angular, irregularly shaped black to almost yellow grains. The specific weight is 3.86 - 4.27 t/m^3.

Table 4 Chemical composition of ilmenite [4]

<table>
<tr><th rowspan="3">Element</th><th colspan="7">Content in %</th></tr>
<tr><th rowspan="2">Kyiv Geological Survey 1956 - 1958</th><th colspan="5">Laboratory №16, Giredmet</th><th rowspan="2">Average</th></tr>
<tr><th>Sample 1</th><th>Sample 4,5A</th><th>Sample 5</th><th>Sample 6</th><th>Sample 8</th></tr>
<tr><td>TiO_2</td><td>61,78</td><td>63,31</td><td>62,80</td><td>61,42</td><td>62,58</td><td>61,50</td><td>62,23</td></tr>
<tr><td>FeO_3</td><td>26,65</td><td>29,84</td><td>31,70</td><td>27,01</td><td>25,07</td><td>27,90</td><td>28,03</td></tr>
<tr><td>Al_2O_2</td><td>4,59</td><td>-</td><td>0,47</td><td>0,17</td><td>0,30</td><td rowspan="4">0,1-1,0</td><td>0,31</td></tr>
<tr><td>CaO</td><td>-</td><td>-</td><td>-</td><td>0,01</td><td>0,15</td><td></td></tr>
<tr><td>MgO</td><td>1,06</td><td>0,45</td><td>0,46</td><td>0,27</td><td>0,43</td><td>0,40</td></tr>
<tr><td>MnO</td><td>0,88</td><td>1,08</td><td>0,86</td><td>0,85</td><td>0,95</td><td>0,94</td></tr>
<tr><td>ZrO_2</td><td>-</td><td>-</td><td>-</td><td>0,045</td><td>0,02</td><td>0,01</td><td>0,03</td></tr>
<tr><td>HfO_2</td><td>-</td><td>-</td><td>-</td><td>0,0019</td><td>0,001</td><td>-</td><td></td></tr>
<tr><td>Ta_2O_5</td><td>0,01</td><td>0,0045</td><td></td><td>0,0035</td><td>0,0004</td><td></td><td>0,0045</td></tr>
<tr><td>Nb_2O_5</td><td>0,054</td><td>0,09</td><td></td><td>0,03</td><td>0,03</td><td></td><td>0,061</td></tr>
<tr><td>Sc</td><td></td><td></td><td></td><td>0,0059</td><td>0,0073</td><td></td><td>0.0066</td></tr>
<tr><td>$\sum TR_2O_3$</td><td></td><td></td><td></td><td>0,005</td><td>0,005</td><td></td><td></td></tr>
<tr><td>Ga</td><td></td><td></td><td></td><td>0,005</td><td>0,005</td><td></td><td></td></tr>
<tr><td>Ge</td><td></td><td></td><td></td><td><0,005</td><td><0,005</td><td></td><td></td></tr>
<tr><td>Jn</td><td></td><td></td><td></td><td><0,01</td><td><0,01</td><td></td><td></td></tr>
<tr><td>Sn</td><td></td><td></td><td></td><td><0,03</td><td><0,03</td><td></td><td></td></tr>
<tr><td>Pb</td><td></td><td></td><td></td><td>0,008</td><td>0,0005</td><td></td><td></td></tr>
<tr><td>Yi</td><td></td><td></td><td></td><td>0,01</td><td>0,01</td><td></td><td></td></tr>
<tr><td>SiO_2</td><td>1,00</td><td></td><td></td><td>0,20</td><td>0,65</td><td></td><td>0,04</td></tr>
<tr><td>V_2O_5</td><td></td><td></td><td></td><td>0,15</td><td>0,15</td><td></td><td>0,15</td></tr>
<tr><td>Co</td><td></td><td></td><td></td><td>0,006</td><td>0,005</td><td>0,01</td><td></td></tr>
<tr><td>Cr_2O_5</td><td>2,32</td><td>1,10</td><td>0,63</td><td>0,67</td><td>0,56</td><td>0,60</td><td>0,98</td></tr>
<tr><td>P_2O_5</td><td>0,22</td><td>0,19</td><td>0,20</td><td>0,17</td><td></td><td>0,15</td><td>0,20</td></tr>
<tr><td>FeO</td><td>-</td><td></td><td></td><td>1,35</td><td>1,09</td><td>1,60</td><td></td></tr>
</table>

Ilmenite from the Poltava Stage has a slightly lower content of TiO_2 comparing with Sarmatian Stage ilmenite, and increased content of magnesium, manganese, phosphorus, chromium, as well as increased content of niobium, tantalum, indium, gallium (Table 4).

Table 5 Chemical composition of rutile and leucoxene [4]

Elements	Content, %: rutile + leucoxene			Content, %: rutile			Content, %: leucoxene			Content, %
	Kyiv geological survey	Sample - 1	Sample - 4,5a	Sample - 5	Sample - 6	Sample - 8	Sample - 5	Sample - 6	Sample - 8	The average results of chemical, spectral and neutron activation analysis
ZrO_2	-	-	-	0,15	0,17	-	0,45	0,35	-	
HfO_2	-	-	-	0,06	0,07	-	0,0064	0,0054	-	
TiO_2	90,88	98,17	95,14	95,1	94,6	96,5	87,6	85,5	86,7	94.01
Fe_2O_3	2,36	0,80	0,90	0,23	0,27	-	3,5	1,1	-	1.11
AlO_3	4,63	-	0,13	0,04	0,027	0,01-0,1	2,30	1,80	0,1-1,0	-
CaO		-	-			-	0,35	0,17	0,1-1,0	-
MgO		-	0,2	0,05	0,03	0,01-0,1	0,15	0,14	0,1-1,0	0,02
MnO	-	-	-	0,02	0,05	0,01-0,1	0,09	0,15	0,1-1,0	0,02
Td_2O_5	0,02	0,015	0,016	0,016	0,013	-	0,015	0,012	0,1-1,0	0,015
Nb_2O_5	0,21	0,38	0,17	0,23	0,20	-	0,09	0,095	0,1-1,0	0,24
Sc	-	-	-	0,0016	0,0015		0,0051	0,0074	-	0,0015
$\sum TR_2O_3$				-	-		0,02	0,01		
Ga				<0,002	<0,002		0,003	0,004		
Ge				<0,005	<0,005		<0,005	<0,005		
Jn				<0,001	<0,01		<0,001	<0,01		
Ti				<0,03	<0,03		<0,003	<0,03		
Pb							0,02	0,01		
Li							0,03	0,02		
SiO_2	0,84			0,5	0,4		0,008	0,004		0,045
V_2O_5				0,33	0,30		0,15	0,15		0.275
Co				0,0003	0,0002		0,0005	0,0003		
FeO							0,3	0,5		
Cr_2O_3	0,33	-	0,07				-	-		
P_2O_5							0,1	-		
ThO_2							0,0246	0,0295		

Leucoxene is present in an amount of 3-7%. It is a product of ilmenite change. The mineral is non-magnetic. In the case of electromagnetic separation, it is concentrated in a non-electromagnetic fraction together with rutile and gives with it a typical industrial concentrate. The color under the microscope is greenish-brown, greenish-gray, cream; the mineral is opaque. It has a rough surface, poorly rounded. Relative density is 3.71-4.10 t/m^3, on average about 3.8 t/m^3. X-ray diffraction analysis shows the presence of a rutile structure. The content of TiO_2 is 85-87%. It is characterized by the presence of ferrous and ferric oxide, as well as water. According to the spectral analysis, leucoxene contains the same impurities as ilmenite.

Table 6 Chemical composition of zircon [4]

Elements	Content, %						
	Kyiv geological survey	Sample - 1	Sample -4,5a	Sample - 5	Sample - 6	Sample - 8	Average
ZrO_2	63,7	63,1	63,9	65,9	65,1	65,7	64,57
HfO_2	1,03	0,81	0,95	1,14	1,15	1,10	1,03
TiO_2	0,47	-	0,30	0,23	0,30	-	0,32
Fe_2O_3	0,23	0,07	0,50	0,07	0,08	-	0,19
AlO_3	3,28	-	0,30	0,13	0,17	-	0,20
CaO		-	-	0,05	0,05	0,01-0,1	-
MgO		0,35	0,05	0,04	0,045	0,01-0,1	-
MnO	-	-	-	0,006	0,007	0,01-0,1	-
Td_2O_5	0,018	-	0,001	<0,0007	<0,0006	-	0,001
Nb_2O_5		0,027	0,02	<0,01	<0,01	-	0.015
Sc				0,0074	0,0069	0,007	0.007
$\sum TR_2O_3$				0,10	0,10		
Ga				<0,001	<0,001		
Ge				<0,005	<0,005		
Jn				<0,01	<0,01		
Sn				<0,01	<0,01		0.007
Pb				<0,01	<0,01		
Li				0,03	0,02		
SiO_2	31,61	34,33		31,0	31,0	32,8	32,0
V_2O_5							
Co							
FeO							
Cr_2O_5	0,03	-	-	-	-	0,04	
P_2O_5	0,13	-		0,104	0,10		
ThO_2	-	-	-	-	-	-	-

Rutile is concentrated in the non-electromagnetic fraction, its content in heavy minerals ranges from 12 to 18%, on average 15.8%. It is represented by grains of irregular and rounded shape or needle-like, slightly rounded crystals. Color is yellow-red, red to dark cherry and black. The specific gravity is from 4.2 to 4.29 t/m^3. The reflectivity ranges from 8.7 to 22.7%, hardness from 588 to 1057 kg/mm^2.

Indium and gallium are present in small amounts. TiO_2 content fluctuations depend on the leucoxene content in the monomineral fraction of rutile, which reaches 3-5% or more.

Spectral analysis indicates the presence of zirconium, copper, lead, tin, nickel, uranium, cesium, lanthanum, phosphorus in rutile. X-ray diffraction analysis confirms the rutile structure.

Zircon is concentrated in non-electromagnetic fraction. Grains with a well-preserved form of crystals in the form of a tetrahedral prism in combination with a pyramid prevail. The ratio between the length and the width of the crystals is from 2:1 to 6:1. The color of the grains under the microscope is light pink to colorless, the grains often contain needle-like inclusions of opaque black crystals, due to which they fall into the electromagnetic fraction. The specific weight is 4,27. According to chemical analyses conducted by the Central Research Institute of Geological Prospecting for Base and Precious Metals laboratory (Moscow), zircon contains UO_3 in the range of 0.10%. Spectral analyses show low levels of yttrium, ytterbium, scandium, copper, lead, strontium, and traces of zinc.

Table 6 shows chemical analyses performed on zircon (ordinary samples) by Kyiv Geological Survey during exploration works in 1956-1958 and Giredmet in 1976-1991.

Table 7 Chemical composition of disthene-sillimanite [4]

Elements	Content, %				
	Sample - 1	Sample - 4,5a	Sample - 5,6	Sample - 8	Average
ZrO_2	0,036	0,19	0,16	-	0,19
TiO_2	0,082	0,20	0,50	-	0,20
Fe_2O_3	0,61	0,33	0,18	0,15	0,33
Al_2O_3	63,67	61,8	62,05	61,60	62.28
SiO_2	36,2	36,9	36,35	36,50	36,49
CaO		0,013	0,10	0,10	0,11
NnO		0,045	0,008	0,01	0,021
MgO		0,035	0,033	0,05	0,039
Cr_2O_3		0,06	0,006	0,001-0,1	0,06
Ni		0,08			
Cu		0,002			
Nb_2O_5	0,003	-			

Content of **disthene-sillimanite** is uneven and varies in the collective concentrate from 5 to 28%, on average equal to 11.6%. Sillimanite is quantitatively predominant. It is represented by needle-like columnar and flat elongated crystals, colorless, glassy, fragile, often contains black punctate inclusions. Disthene is represented by irregular tabular grains with a characteristic stepped end; colorless, sometimes bluish, also has dark dotted inclusions, sometimes in such quantity that it becomes opaque. Because of similar diagnostic features and the same chemical composition, estimation of their content in the sand is also carried out. Table 7 summarizes the results of spectral and neutron activation analyses of disthene-sillimanite.

Staurolite is represented by irregularly rounded grains ranging in size from 0.0722 mm to 0.1165mm in various shades of yellow and yellow-brown. It has a glass luster, uneven fracture, transparent. The content in the collective concentrate ranges from 1.5 to 8%, with an average of 3.1%. It is concentrated in electromagnetic fraction. The specific weight is 3.5-3.8 g/cm^3. The chemical composition of staurolite is given in Table 8 (according to Kyiv Geological Survey).

Table 8 Chemical composition of staurolite [4]

Elements	Content, %	
	Sample - 1	Sample - 4,5a
Al_2O_3	50.07	51.8
SiO_2	31.41	27.9
Fe_2O_3	10.54	15.1
TiO_2	1.16	-

In small quantities, staurolite also contains FeO, MnO, CaO, MgO, NiO.

Other minerals of the heavy fraction (garnet, chromite, spinel, monazite) have no industrial value due to their low content in the amount of 1-10%.

Quartz is the main mineral of the light fraction, is about 85% of the mineralogical composition of the ore sand. It is mainly concentrated in fraction –0.160 +0.071 mm. Its grains have an irregular shape, mainly semi-rolled. They are also rounded and not rounded at all. Basically, 60% of the grains are transparent, 38% - opaque and 1-2% of the grains are ferrules and have light-yellow color.

Laboratory tests of enriched quartz sand were carried out. They have shown that the sand has an application in glass foundry. The sand of the Sarmatian Stage of this with the most widely used small quality class for foundry can be assessed in this regard.

1.5 Hydrogeological and Engineering-Geological Characteristics

1.5.1 Hydrogeological Conditions of the Area

Hydrogeological conditions of the Motronivka-Annivka section are complex due to differences in sedimentary rocks in terms of lithological composition, filling conditions, circulation and discharge of groundwater, their hydrodynamic and hydrochemical features. Thus, the groundwater of the Motronivka-Annivka section is represented by the main aquifer in the Miocene sediments.

1.5.2 Engineering-Geological Characteristics

Novomoskovsk geological survey expedition drilled five wells, selected and analyzed 29 monoliths for a full range of physical and mechanical characteristics. Loamy sediments were tested to a depth of 25.0m. The thickness of these top sediments is maximum within limits of watersheds and is 15–20m, rarely up to 25–27m, decreases to 5–15m on the slopes, and alluvial-diluvial clay deposits are developed in the bottoms of beams.

The granulometric composition of loamy-clay deposits contains clay fractions 20-40%, dust 56-75%, sand 2.1-2.5%. The plasticity number varies from 0.11 to 0.31, humidity - 14-25%, the angle of internal friction ranges from 15˚ to 26˚.

According to test data for soils in dry and wet conditions, it can hold a load up to 2.0 kg/cm^2 when dry, and 1.0 kg/cm^2 when wet. Soils are characterized as having the medium bearing capacity, and when saturated with water - with weak bearing capacity.

Investigation of engineering-geological properties of rocks at the site was carried out in 2002 by a net of wells with a sampling interval of 70m.

The top layer of the area is represented by sandy-clay Cainozoic deposits. They are represented by the soil-vegetative layer, loess-like loams of yellow-brown, pale-yellow, brown and red-brown color, clays of red-brown and greenish-grey color. The Miocene rocks, mostly composed of almost homogeneous fine and fine-grained quartz sands.

The inclination of natural slope for this sand in the dry state is 30-34˚, underwater is up to 23˚, the filtration coefficient according to laboratory data varies from 0.08 to 4.52 m/day, in some cases up to 6.8 m/day. The density of dry soil in the extremely loose state - 1.22 g/cm^3, in extremely dense - 1.56 g/cm^3. The porosity coefficient in the extremely loose state is 1.18, in the extremely dense addition 0.70. Full moisture capacity - 31.1%. The maximum molecular moisture content is 5.8%.

1.5.3 Preliminary Estimation of Water Inflow

Experience in construction and operation of pits in similar conditions shows that the most extensive water inflows are observed in the initial period of development when the natural groundwater reserves are activated and depression cone is formed. This period lasts for 1.5-2.0 years according to the experience. After that, a noticeable decline in water inflow and its relative stabilization takes place.

According to a report on the geological study of the subsoil of the Motronivka-Annivka section, the following data are obtained:

- 751 m^3/day (31,3 m^3/hour) – water inflow from natural groundwater resources;
- 365 m^3/day (15,2 m^3/hour) – water inflow from atmospheric precipitation;
- 20 959 m^3/day (873 м^3/hour) – water inflow from showery rain precipitation.

Water-bearing rocks of aeolian-diluvial and quaternary sediments have, on average, a filtration coefficient 0.5 m/day. With a small thickness of these deposits and as they are processed as overburden, their water characteristics will not cause an increase in water inflow.

1.6 Mining Conditions

The mining and technical conditions of the Motronivka-Annivka section are favorable for the operation of opencast mining.

Overburden rocks are yellow-brown loam, red-brown loam and red-brown clay, brown quaternary, greenish-grey clay, and fine-grained sand of the Sarmatian Stage. Granulometric composition of loamy-clay deposits contains clay fractions 20-40%, dust 56-75% and sand 2.1-2.5%.

All overburden is a soft and loose rock. It is suitable for direct excavation without the necessity of blasting or another method of preliminary loosening. Volume weight of overburden rocks ranges from 1.54 t/m^3 (brown-yellow loess-like loam) to 2.05 t/m^3 (greenish-grey clay). Average - 1.90 t/m^3. Ore sand in its raw form - up to 2.0 t/m^3, and in its dry state - 1.76 t/m^3. The average coefficient of loosening according to practical experience data is 1.20. Rocks allow the use of different types of excavators: rotary, dragline, hydraulic excavators, but the specific pressure is not more than 2 kg/cm^2.

The overburden thickness within the field varies widely and depends on current topography: the minimum overburden thickness occurs within the valleys of the beams, and the maximum over the plateau watersheds. The minimum thickness is 15.4-30.0m; the maximum thickness reaches 77.0m. The average thickness of overburden is 52.2m, of which about 17m is fine and medium-grained sands of the Sarmatian Stage, which directly lie on the ore sand of the Poltava Stage.

Ore formation is fine and fine-grained, slightly clayey (about 13-14%). It is loose in its natural state and is denser when drying. By drillability, they loosen well, and when they are saturated with water, they turn into flowing send. The thickness of the ore sand within the ore deposit varies insignificantly.

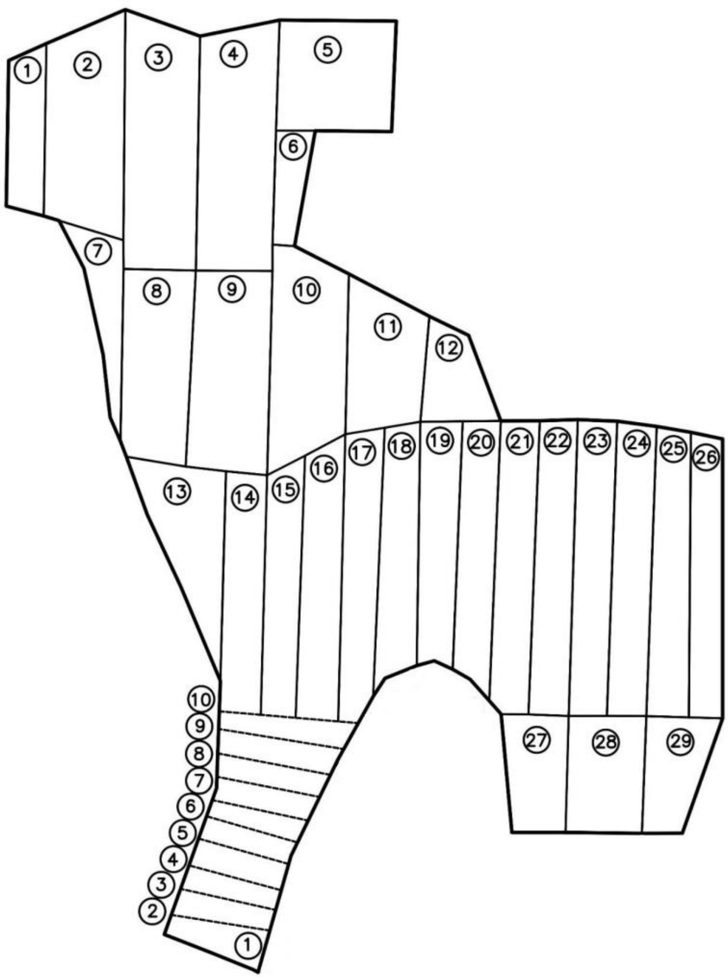


Figure 8 Motronivka-Annivka section divided into subsections [5]

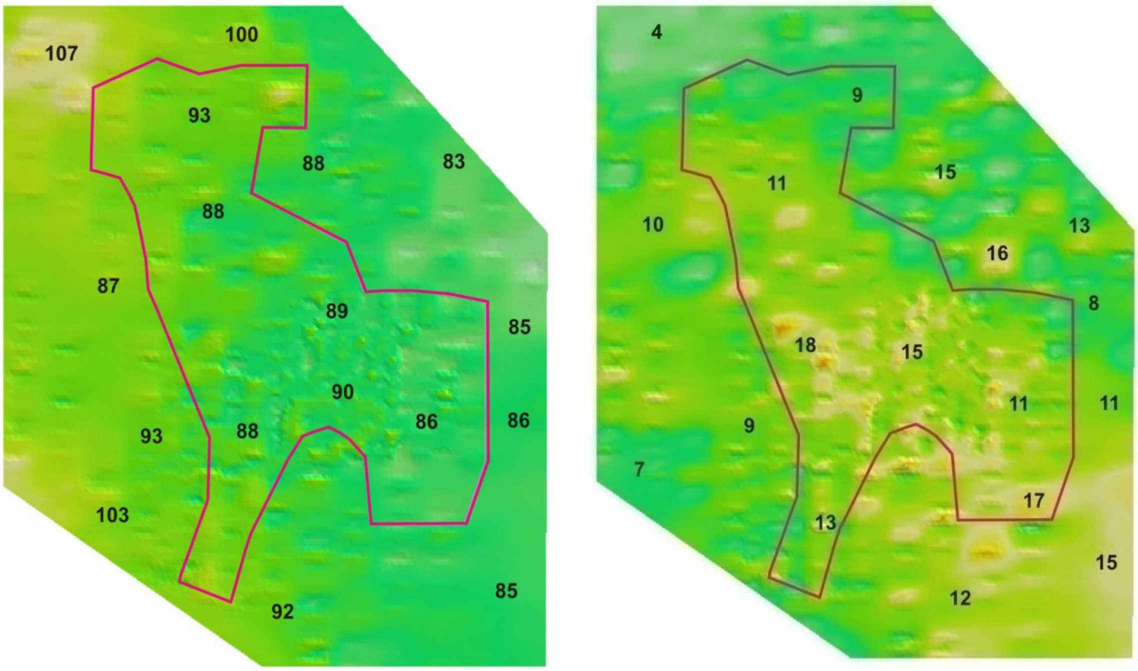


Figure 9 Main ore body: 1 - map of the bottom level [m.a.s.l]; 2 – map of thickness [m] [5]

The bed of ore sand lies almost horizontally with only some elevation in western and southern directions. The absolute elevation marks of the ore sand layer are presented in Figure 9.

Fluctuations in the stripping ratio due to significant changes in overburden thickness are great. The minimum values are observed in areas with insignificant overburden thickness, mainly in beams and are about 2.0 m^3/m^3. The maximum values of the stripping coefficient are noted in the southern part of the watersheds and reach 19-20 m^3/m^3 in individual wells (Figure 10).

Among the overburden sand of the Sarmatian Stage and the Poltava Stage ore layer, cementation takes place in the form of epigenetic carbonate-clay, carbonate or ferrous formations of spherical, elliptical, and other irregular forms of sandstones. The size of these formations, as it was observed in cores of exploration wells, does not exceed 30-40cm. Their strength is diverse and generally is insignificant.

As it was noted, the roof and bottom of the ore-bearing layer are sands which look similar to the ore, but poor of useful components. The sand of the Sarmatian Stage, which overlaps the ore layer, can also be a mineral for the production of molding and glass after washing out clays. Therefore, they could be worked out selectively from the rest of the overburden.

Table 9 Average overburden and ore layer thicknesses for every subsection of the field with strip ratio calculation [4]

Block number	Reserves category	Volume of overburden, thousand m^3	Volume of ore, thousand m^3	Thickness, m		Strip ratio
				overburden	ore	
1	C_1	16485,9	2543,9	66,1	10,2	6,48
2	C_1	39431,5	6271,7	59,1	9,4	6,29
3	C_1	36013,6	7095,9	47,2	9,3	5,08
4	C_1	36710,9	6917,7	46,7	8,8	5,31
5	C_1	19274,4	4069,0	36,0	7,6	4,74
6	C_1	9065,1	1223,4	61,5	8,3	7,41
7	C_1	13264,0	2472,9	58,4	10,9	5,36
8	C_1	38376,8	6497,1	69,7	11,8	5,91
9	C_1	45855,8	7874,2	69,3	11,9	5,82
10	C_1	38688,1	6679,1	61,4	10,6	5,79
11	C_1	19497,0	3742,7	44,8	8,6	5,21
12	C_1	9952,0	1728,0	42,5	7,4	5,76
13 top	B	20388,8	5640,8	46,5	12,9	3,61
13 bottom	B	8447,5	1411,0	51,4	8,6	5,99
14	B	45576,4	9937,7	53,2	11,6	4,59
15	B	33893,8	8437,4	50,8	12,7	4,00
16	B	29096,3	8289,7	44,4	13,1	3,51
17	B	9280,8	2879,3	41,2	12,75	3,245
18	B	9705,0	2643,9	47,2	12,9	3,68
19	B	9810,9	2666,2	49,4	13,4	3,69
20	B	9537,8	2522,1	45,9	12,2	3,78
21	B	10313,9	2815,5	43,3	11,8	3,67
22	B	11207,8	2761,7	46,6	11,5	4,06
23	B	22296,8	6041,7	46,5	12,6	3,69
24	B	24043,4	5503,9	49,8	11,4	4,37
25	B	24817,9	4797,5	53,8	10,4	5,17
26	C_1	20974,6	3587,9	53,9	9,3	5,80
27	C_1	20327,9	4283,5	68,3	14,4	4,75
28	C_1	25110,1	5563,3	66,8	14,8	4,51
29	C_1	17364,8	4214,9	61,2	14,9	4,12
Sum		733511,8	157438,2			
Average				52,2	11,2	4,66

There are two aquifers. The upper aquifer in loam reduces the carrying capacity of overburden rocks within it, which, as the experience of work on neighboring open pits shows, should be taken into account when choosing mining equipment and location of working sites.

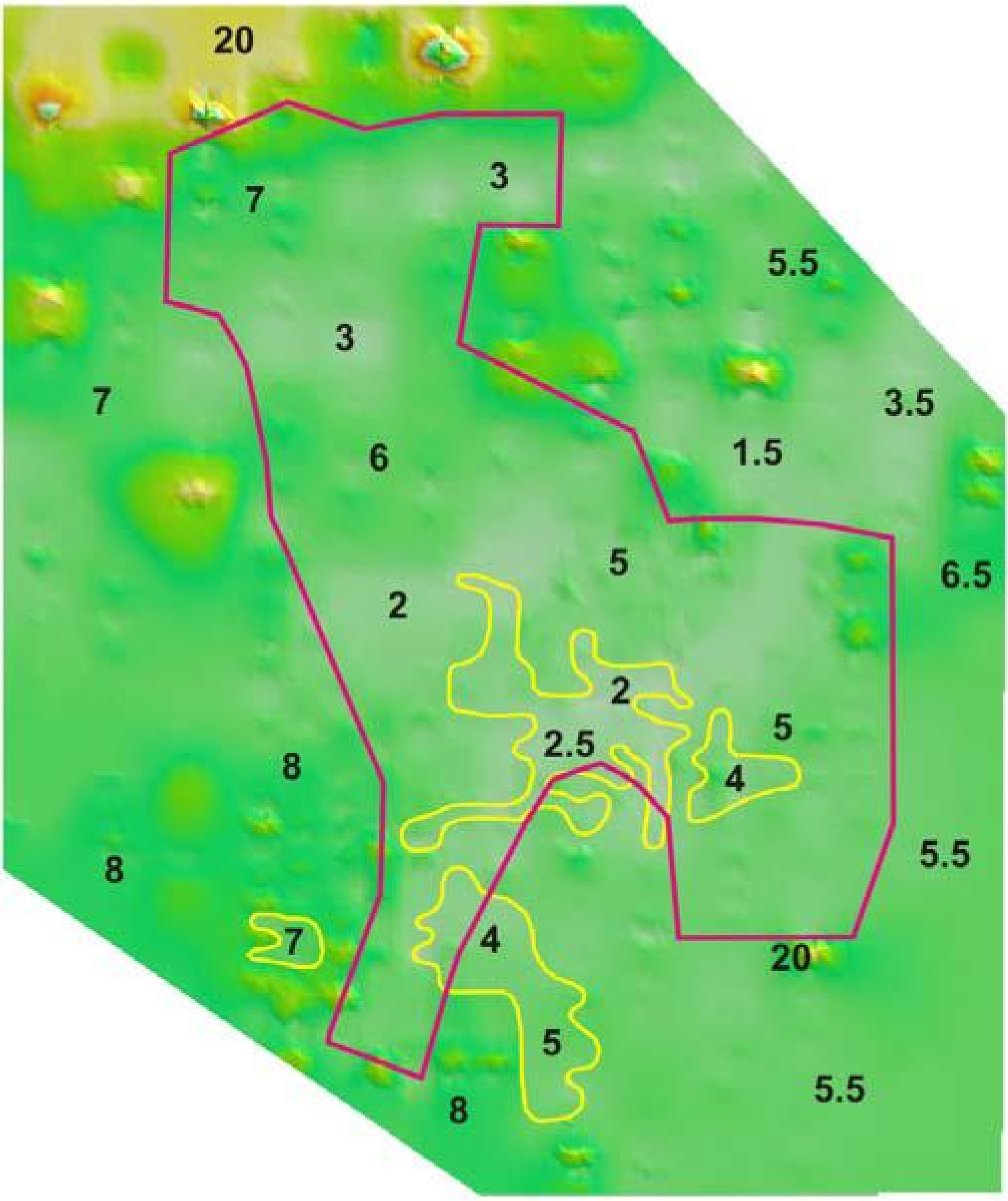


Figure 10 Stripping ratio of the whole ore body (main and lower) [5]

1.7 History of the Malyshev Ore Deposit Development

The deposit has been developed since 1959 for servicing a local processing plant, and then (since 1960) an experimental processing plant. From December 1961, a plant began to work with a capacity of 1 million m^3 of sand per year as a first phase. Since 1968, the second phase of the plant was reached with the rise of annual mining production up to 2.5 million m^3 of sand per year. Until 1987, the production and processing capacity increased to 4.5 million m^3 per year. Then it began to decline gradually. An especially significant decline in production occurred in 1994 when the production capacity dropped to 1.3 million m^3. By 2005, the production and processing surpassed the planned volume and reached 5.5 million m^3, but the content of ore minerals decreased significantly. When mining operations reached the Eastern section, the content of useful components was slightly lower than at the Central section. By the capacities of the Motronivka-Annivka section, the production of concentrate should be increased to 6.5 million m^3 per year.

Until 1986, bucket-wheel excavator K-300 of Czechoslovak production and PO-800 bucket-wheel excavator was used. Front shovel hydraulic excavator with an electrical drive with shovel volume 4.6m^3 EKG-4,6 (ЭКГ4,6) in combination with dump trucks were also used for sand extraction. In 1986, in one of the quarries of the Western section was applied dragline ESh-10-70 (ЭШ10/70).

Currently, a combined development system is used at all working mines on the deposit. Ahead of stripping front, black soil is removed by scrapers with the disposition of it into storage piles. After the workings, black soil will be evenly distributed all over the area.

Upper overburden bench represented by loam 20-25m thick is developed by bucket-wheel excavator KU-800 and RShR-1600-40/70 (РШР-1600-40/70) with conveyor system and reloader PVP-6600 (ПВП-6600), overburden spreaders ZP-6600 (ЗП-6600), OShR-5000/190 (ОШР-5000/190). The waste rock is to be dumped at the mined-out space – external top bench.

Benches below are processed in a combined way. The upper bench, represented mainly by red-brown clay and in a lesser amount of greenish-gray clay, is mined by EKG-8E (ЭКГ-8И) and EKG-10 (ЭКГ-10) excavators with loading in trucks with the further placement of it into internal worked out space - lower dump. The lower bench of red-brown and greenish-gray clays is processed by transportless mining system with ESh-10-70 (ЭШ 10/70) excavators and placed in the same internal dump or by the transport system by EKG-8E or EKG-10 excavators with loading rock in 40t BELAZ dump trucks. Rock is dumped in the same lower inner dump.

The ore layer is mined out by draglines ESh-6-45 (ЭШ 6/45) or ESh-10/70 with loading into vehicles and transportation of it to mobile pumping stations. Further, pulp formed by water jet is fed through the slurry pipelines (pipe Ø 630 mm) to a processing plant.

Enrichment on the processing plant includes screening, disintegration and desliming of sand. The gravitational method is used to obtain a collective concentrate of heavy minerals. After drying, it is divided into commercial monoconcentrates using electrical and magnetic enrichment methods. The complexity of the technological enrichment scheme can be explained by the partial overlapping of the properties (mechanical, electrophysical, chemical) of various minerals and the presence of mutual transition forms of it.

The applied and constantly improving enrichment scheme ensures the production of high-quality commercial concentrates of zircon, rutile, ilmenite, kyanite-sillimanite, staurolite, and also molding sand. Thus, a high degree of recovery of the resources of the field is ensured. Currently, 100% of titanium and zircon are sold. Kyanite and staurolite concentrates are sold per 10-30%. Only 2-3% of molding sand can be sold due to the lack of consumers.

For the entire period of mining on the field, the balance reserves of sands of the Sarmatian Stage of the Western section are fully exhausted. The sand of the Sarmatian Stage of the Central section was mined out till early 2006. The remaining balance reserves of sands of the Sarmatian Stage of the Eastern section can ensure the operation of the plant only until 2021.

1.8 Reserves Estimation

The recalculation of reserves by new samples was done using computer technology. The conventional ilmenite was used for calculation (all other minerals were equated to it using appropriate coefficients). Based on the results, the ore contour in the wells was determined by the thickness of ore sands and overburden. The percentage of conditionality of each well with taking into account the stripping ratio was taken into account.

For the calculation of the thickness of ore sand in wells, interlayers of Sarmatian sands directly lying on Poltava Stage ore sand were added (it is up to 2m thick). However, in order to equalize the thickness of the ore sands, two wells with barren layers 2.0m and 3.0m are also included in the reserves calculation. In general, the number of samples with content less than 18 kg/m^3 is insignificant – just 0.44% of the total number.

The State Reserves Committee of Ukraine in 2005 approved the following conditions:

Commercial reserves:

1. Ultimate content of conditional ilmenite for outlining the ore formation by its thickness - 18 kg/m^3;

2. The coefficients for converting the contents into conditional ilmenite: rutile + leucoxene - 3.738, zircon - 4.311; ilmenite - 1.0; disthene + sillimanite - 0.378; staurolite - 0.061;

3. Minimal thickness of the ore layer is 2.0m;

4. Interlayers within the ore layer with less than ultimate content of conditional ilmenite is included in the calculation of reserves based on the actual thickness and actual content of conditional ilmenite;

5. For smoothing of the contours of the open-pit field, ore with the content of conditional ilmenite equal to the minimum industrial, but higher than ultimate was included even if its thickness is low.

Non-commercial reserves:

1. Ultimate content of conditional ilmenite for outlining the ore formation by its thickness – 12 kg/m^3;

2. Non-commercial reserves are calculated on the area outside the contour of commercial reserves.

For the field next reserves are allocated and calculated:

- commercial reserves within the contour of the open-pit field;
- commercial reserves under buildings (within 300m of sanitary protection zone);
- commercial reserves outside the contour of the open-pit field;
- non-commercial reserves.

Next components of the deposit are divided into commercial and non-commercial reserves:

- stocks of main components and its co-products: zircon, rutile, ilmenite, kyanite-sillimanite and staurolite;
- reserves of titanium dioxide associated with rutile and ilmenite, zirconium dioxide and hafnium associated with zircon.

The most important characteristics of the Motronivka-Annivka section are given in Table 10.

Table 10 Characteristics of the Motronivka-Annivka section [4]

Characteristics	Units	Total commercial and non-commercial reserves	Reserves within the open pit field
Area	thousand m^2	39558,8	14046,7
Overburden thickness	m	55,6	52,22
Volume of overburden	thousand m^3	2220748,1	733511,8
Thickness of ore send	m	9,8	11,21
Reserves of ore send	thousand m^3	389221,7	157438,2
Reserves of heavy fraction	thousand t	26194,4	13008,6
Reserves of zircon	thousand t	2080,8	1070,6
Reserves of rutile	thousand t	4134,5	2031,3
Reserves of ilmenite	thousand t	14005,4	7207,7
Content of heavy fraction	kg/m^3	67,3	82,63
Content of zircon	kg/m^3	5,35	6,80
Content of rutile	kg/m^3	10,62	12,95
Content of ilmenite	kg/m^3	35,98	45,78

1.9 Summary

The chosen most important data, which is necessary for the comparison of technological schemes, is presented below.

All overburden is a soft and loose rock. It is suitable for direct excavation without the necessity of preliminary loosening. Average volume weight of overburden rocks which may be used for calculations is 1.9 t/m^3, for ore sand - 2.0 t/m^3. The average coefficient of loosening for computations (both overburden and ore send) - 1.2 m^3/m^3. Rocks allow the use of different types of excavators: rotary, dragline, hydraulic excavators, but the specific pressure should be no more than 2 kg/cm^2.

The average thickness of overburden within the open pit field is 52.2m, the thickness of the ore sand layer – 11.2m. The estimated thickness of the black soil layer is 1m. The angle of repose is 35 degrees, but for the long-time stability, the angle of 21 degrees should be reached.

The field has an aquifer, which is situated 30m deep with an average thickness of 5m. Also, the ore send layer, and especially layer under it is waterlogged. Special measures should be taken to guarantee the safety of the work.

Chapter 2 Equipment Selection and Description of Technological Schemes

An engineering analysis of the possible technological schemes of the Motronivka-Annivka section development is done in this chapter. The purpose of these calculations is to determine the required number of the main and auxiliary equipment that will make it possible to carry out an economic evaluation of the schemes.

2.1 Calculation of Extraction Front Annual Advance and Rock Volumes

Calculation of the annual advance of the mining front is made according to the formula:

$$P_{year} = \frac{Q_{year}}{H_{o.l.} \cdot A_{e.f.}} \text{ [m]} \quad \textbf{(1)}$$

where: Q_{year} - annual production rate by ore sand, m^3;

$H_{o.l.}$ - average ore layer thickness, m;

$A_{e.f.}$ - average width of the extraction front, m.

The required annual production rate by ore send for the Motronivka-Annivka section - 2.7 million m^3. According to the geological report [4], the average thickness of the ore layer is 11.22 m. The designed width of the extraction front is 1600 m. Thus, the necessary yearly advance of the mining front is 150 m/year.

The waste rock volume of that should be mined out per year for the ore layer stripping is calculated by formula:

$$Q_{o.year} = H_o \cdot P_{year} \cdot A_o \text{ [m}^3\text{]} \quad \textbf{(2)}$$

where: H_o - average overburden thickness, m;

A_o - average width between open pit flanks, m.

The volume of overburden depends on applied technological schemes. The number of benches influences an average width of the open pit field. For servicing of every bench, it is necessary to construct roads, which requires some space on the side flanks. In addition, the volume of overburden depends on the bench's height and its advance front length, which are not the same in different technological schemes. Cross-sections of all technological schemes are presented in Attachments №1, №2 and №3.

The input data and results of the overburden rock volumes calculation for all technological schemes are presented in Attachments №4, №5 and №6. The sum of volumes that should be mined out per year on all benches in the technological scheme with the use of bucket-wheel excavator is 13 814,7 thousand m^3. The overall overburden volume in the technological scheme with the use of draglines is 14 461,8 thousand m^3. And the overall overburden volume in the technological scheme with the use of hydraulic excavators is 15 050,6 thousand m^3.

2.2 Estimation of Necessary Equipment Quantity

The rock volume which should be mined on every bench to reach the necessary annual advance of the mining front:

$$Q_{year} = H \cdot P_{year} \cdot A_{e.f.} \text{ [m}^3\text{/year]} \quad \textbf{(3)}$$

where: H - height of the bench, m;

P_{year} - annual advance of the mining front, m;

$A_{e.f.}$ - width of the extraction front, m.

The formula for the calculation of working hours of an excavator per year:

$$N_{h.y.} = (N_m \cdot N_d - N_h - N_s) \cdot n \cdot (T_{sh} - T_{sh.t.}) \text{ [h]} \quad \textbf{(4)}$$

where: N_m - number of working months per year, month;

N_d - average number of working days in a month, days;

N_h - number of holidays per year, days;

N_s - number of standstill days caused by whether conditions per year, days;

n - number of shifts, shifts;

T_{sh} - duration of the shift, hours;

$T_{sh.t.}$ - time, which is required per shift turnover, dinnertime and preventive maintenance of the equipment, hours.

Calculation of production rate per hour of hydraulic excavators and draglines is carried out according to the formula:

$$Q_h = \frac{(3600 - T_{p.b.f.} - T_m) \cdot V_b \cdot K_{v.e.}}{T_c \cdot K_f} K_r \text{ [m}^3\text{/h]} \quad \textbf{(5)}$$

where: $T_{p.b.f.}$ - consumption of time on preparation of the bench face per hour, seconds;

T_m - consumption of time on change of the excavator position (movement), seconds;

V_b - bucket capacity, m^3;

$K_{v.e.}$ - coefficient of volume efficiency of the bucket;

T_c - average cycle time for one bucket shoveling, seconds;

K_f - fragmentation index of rock in the bucket (the soil conversion factor);

K_r - coefficient of readiness of the excavator.

Calculation of production rate per hour of a bucket-wheel excavator:

$$Q_{h.bwe} = \frac{60E \cdot n_u \cdot K_p \cdot K_u}{K_f} \quad \textbf{(6)}$$

where: E - BWE bucket volume;

n_u - average number of unloadings per minute (according to manufacturing documentation);

K_p - coefficient of productivity;

K_u - coefficient of usage.

Calculation of the necessary number of excavators for the fulfillment of the annual plan:

$$N_{ex.} = \frac{Q_{year}}{Q_h \cdot N_{h.y.}} \text{ [unit]} \quad \textbf{(7)}$$

where: Q_{year} - annual production, m^3/year;

Q_h - production per hour of the excavator, m^3/hour;

$N_{h.y.}$ - working hours of the excavator per year, hours.

The working time of the trucks is the same as the excavator's working time this truck service. Therefore, for calculation $N_{h.y.}$ of the excavator will be used. The rock volume which should be transported during one shift is calculated according the formula:

$$Q_{t.sh.} = \frac{Q_{year} \cdot K_f \cdot (T_{sh} - T_{sh.t.})}{N_{h.y.}} \text{ [m}^3\text{/shift]} \quad \textbf{(8)}$$

where: K_f - coefficient of fragmentation;

T_{sh} - duration of the shift, hours;

$T_{sh.t.}$ - time, which is required per shift turnover, dinnertime and preventive maintenance of the truck, hours.

The time required per one trip of the truck:

$$T_t = 60 \cdot \frac{2L_{h.d.}}{V_t} + \frac{T_c \cdot n_b}{60} + T_{un} \text{ [min]} \quad \textbf{(9)}$$

where: $L_{h.d.}$ - hauling distance, km;

V_t - average speed of the truck, km/h;

T_c - average cycle time for one bucket shoveling, seconds;

n_b - necessary number of buckets to fill the truck, buckets;

T_{un} - unloading time, min.

Calculation of the maximum possible number of trips per shift:

$$N_{trips} = 60 \cdot \frac{(T_{sh} - T_{sh.t.})}{T_t} \text{ [min]} \quad \textbf{(10)}$$

The result should be rounded down.

Calculation of the necessary number of trucks for fulfillment of the yearly production plan:

$$N_{truck} = \frac{Q_{t.sh.}}{N_{trips} \cdot V_b \cdot K_{v.e.} \cdot K_r \cdot n_b} \text{ [trucks]} \quad \textbf{(11)}$$

where: $Q_{t.sh.}$ - rock volume that should be transported per shift, m^3;

N_{trips} - number of trips per shift, times;

V_b - bucket capacity, m^3;

$K_{v.e.}$ - coefficient of volume efficiency of the bucket;

K_r - coefficient of readiness of the truck.

The result should be rounded up.

All calculations of the necessary equipment for all technological schemes presented in Attachments №4, №5 and №6.

2.3 Technological Schemes Description

Plans and all necessary cross-sections of the three technological schemes are presented in Attachments №1, №2 and №3. Also, on sections A'-A' are shown non-miningflanks of the opencast, which are used for the calculation of rock volumes on every bench. For instance, the width of the first bench in the technological scheme with the bucket-wheel excavator use is calculated as 215.2 x 2 + 1600 = 2030,4m. Consequently, using the third formula, the volume of the rock is 304,5 thousand m^3. The results and all incoming data of the calculations of volumes by benches are demonstrated in Attachments №4, №5 and №6.

Heights of all benches were chosen to get a minimum quantity of excavators with its maximum use on every bench.

The choice of mining equipment was made considering the following factors:

1. Geological and mining conditions of the deposit;
2. Accessibility of information about use of this equipment on mines in Ukraine;
3. Availability of information about serviceability, repair capability and profitability of using this equipment.

It may be possible to use this method for comparison of other technological schemes on other mines using different kinds of equipment.

2.3.1 Technological Scheme with the use of Bucket-Wheel Excavator

2.3.1.1 Front Bench (scheme one)

In the first technological scheme, the removal of the potentially fertile layer is done by dragline ESh-14-50 with loading on trucks CAT 777. Also, this kind of equipment is used on all other benches (except the bench which is mined by the bucket-wheel excavator). This is made to ensure the interchangeability of the equipment. The work goes only during weather permitting conditions six months in a year (May to October). The technical specifications of this equipment are presented in Tables 11 and 12. The scheme of aimed loading of mining trucks by NKMZ draglines is shown in Figure 11. Dimensions of the CAT 777 truck are given in Figure 12.

Table 11 Technical specifications of ESh-14-50 dragline excavator [12]

Bucket capacity, m^3	14
Boot length, m	50
Average ground pressure, kPa at operation at walking	 82,84 133,2
Travel speed, m/sec	0,055
Maximum digging and dumping radius, m	57,87
Maximum dumping height, m	46,5
Maximum digging depth, m	20,5
Excavator weight, t	620

Figure 11 Scheme of aimed loading of a truck by ESh-14-50 dragline excavator [12]

Excavator ESh-14-50 is the most optimal choice in comparison with other accessible draglines. Relatively short boom makes easier the work of the machine operator when truck loading and the bucket volume of $14m^3$ provides high productivity.

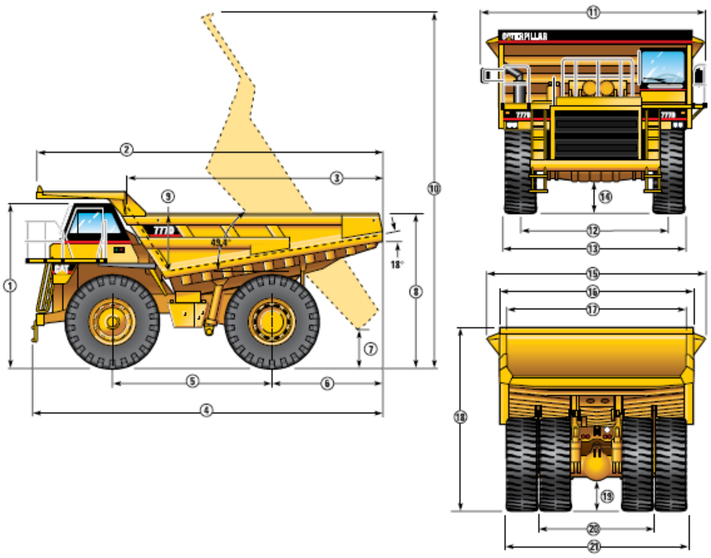
Moreover, the decrease of boom weight decreases the weight of the counterweight, which leads to a decrease in the overall weight. This decreases specific weight, which is critical in given geological conditions, and also affects the price.

The maximum possible specific pressure of the excavator at walking according to technical specification is 133,2 kPa. This equals 1,36 kg/cm^2, which does not exceed the maximum allowed specific pressure in given geological conditions 2 kg/cm^2.

According to practical experience from the mines in identical geological conditions, coefficient of volume efficiency of the bucket is 1.0. To fill the truck's body, it is necessary four excavator's buckets (4 x 14 x 1.0 = 56 m^3, less than nominal capacity 64,1m^3). Fragmentation index is 1.2, so the average weight of the loaded rock is 56 ÷ 1,2 x 1,9 = 88,7t, and the maximal weight is 56 ÷ 1,2 x 2 = 93,3t, what is approximate to the nominal load-carrying capacity 90,8t and does not exceed the maximum load-carrying capacity 99,8t.

Table 12 Technical specifications of CAT 777 dump truck [13]

Nominal Payload, t	90,8
Maximum Payload, t	99,8
Body capacity, m^3	64,1
Target Gross Machine weight, kg	164 654



1	Height to Top of ROPS	4730 mm
2	Overall Body Length	9830 mm
3	Inside Body Length	6580 mm
4	Overall Length	10 535 mm
5	Wheelbase	4560 mm
6	Rear Axle to Tail	3062 mm
7	Ground Clearance	896 mm
8	Dump Clearance	965 mm
9	Loading Height – Empty	4380 mm
10	Inside Body Depth – Maximum	1895 mm
11	Overall Height – Body Raised	9953 mm
12	Operating Width	6687 mm
13	Front Tire Width	4170 mm
14	Engine Guard Clearance	864 mm
15	Overall Canopy Width	6200 mm
16	Outside Body Width	5524 mm
17	Inside Body Width	5200 mm
18	Front Canopy Height	5200 mm
19	Rear Axle Clearance	902 mm
20	Rear Dual Tire Width	3576 mm
21	Overall Tire Width	5223 mm

Figure 12 Dimensions of CAT 777 dump truck [13]

The fertile layer is moved into internal stockpiles and is used for recultivation. Surface forming is to be done by bulldozer CAT-D8R. For servicing of mining roads, grader CAT-16M is used.

The thickness of the first bench is up to 1 m. The total volume of overburden within the front bench with the necessary annual advance of the extraction front is 304,5 thousand m^3.

2.3.1.2 Second Stripping Bench (scheme one)

Second striping bench 28m thick is mined by bucket-wheel excavator SRs-2000 (Figure 13). The technical specifications are presented in Table 14. The scheme of work on the mining face is shown in Figure 14. Rock, after extraction, is loaded on face conveyer, and passing trunk and stacking conveyers comes to overburden spreader A2Rs. The spreader is forming a dumping bench, keeping the natural order of rock position (Attachment №1).

Table 13 List of equipment used on the second bench

Equipment		Number
Bucket-wheel excavator SRs-2000		1
Face conveyer	Conveyer	2
	Receiving hopper	2
Trunk conveyer	Conveyer	2
	Receiving hopper	2
Stacking conveyer	Conveyer	2
	Receiving hopper	2
	Throw-off carriage	2
Cable transporter		1
Overburden spreader		1
Reloader		3
Belt shifter		3
Excavator CAT-312 for conveyer belt cleaning		1

To provide the necessary annual advance of the extraction front, yearly productivity of the bucket-wheel excavator should be not less than 7 606,2 m^3. Therefore, the excavator should work year-round, 24 hours per day. The entire list of equipment to service workings on the second stripping bench is listed in Table 13.

2.3.1.3 Third and Fourth Stripping Benches (scheme one)

Third and fourth stripping benches are mined by draglines ESh-14-50 with loading on trucks CAT 777.

Bulldozers CAT-D8R and graders CAT-16M are used for the maintenance of roads and approaches. Dumping of the rock mined from the fourth bench is to be done before dumping of the rock from the third bench (Attachment №1).

There is an aquifer within the third bench. The working machine level is one meter above the aquifer, which ensures optimal working conditions for ESh-14-50 and CAT 777. Thickness of the aquifer reaches 5m. Drainage trenches are used to drain the mined-out rocks. The scheme of development of the aquifer bearing bench is in Figure 15. Also, the organization of work on the third bench is presented in Attachment №1.

Thicknesses of third and fourth overburden benches are 12 and 9 meters. Annual production is 5 639,0 thousand m^3.

Table 14 Technical specifications of SRs-2000 bucket-wheel excavator [18]

Maximum bench height, m	28
Digging radius, m	41,3
Bucket capacity, m^3	1,7
Average number of unloading per minute, times	26
Weight of excavator, t	2160

Figure 13 SRs-2000 bucket-wheel excavator [14]

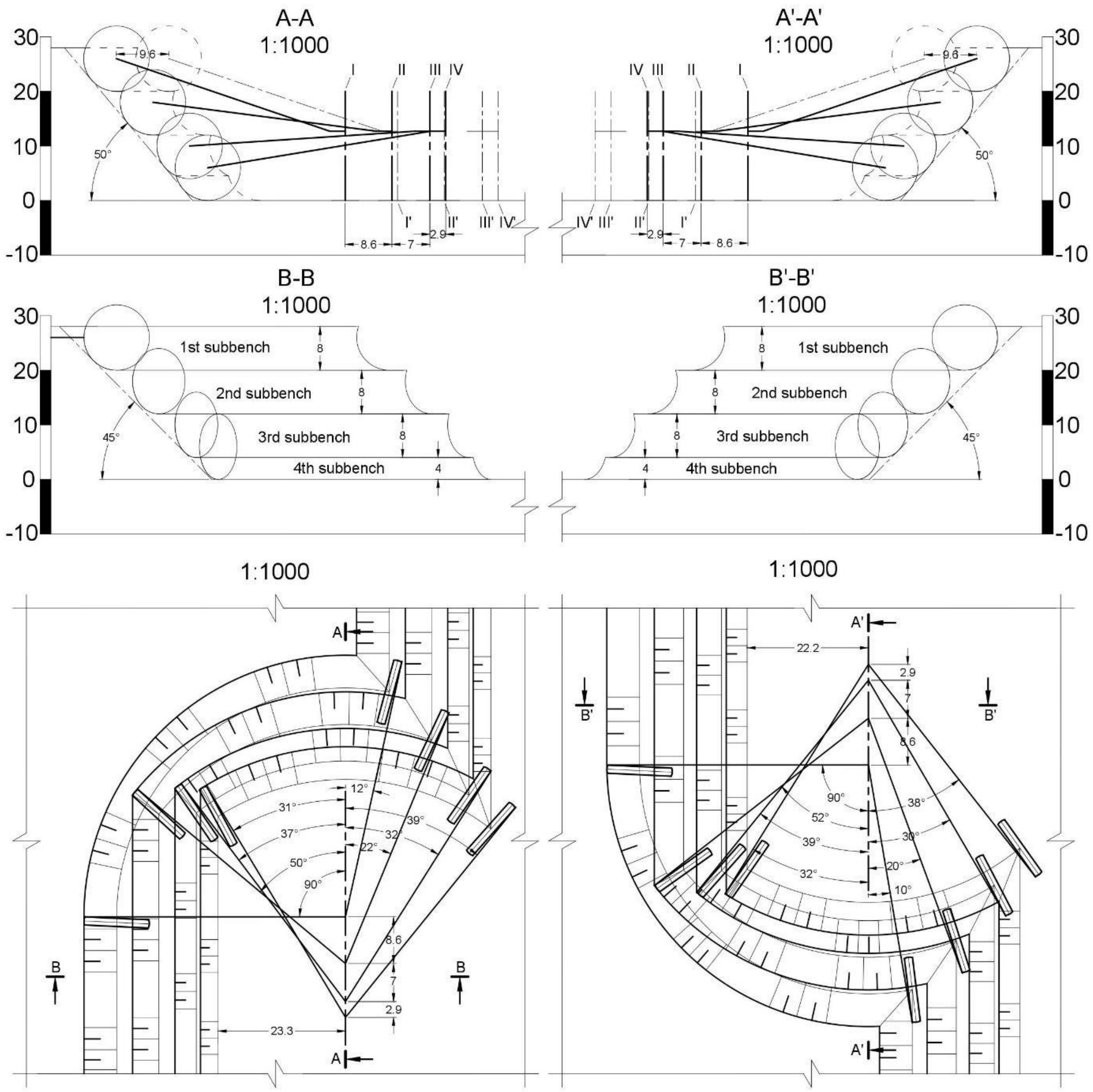


Figure 14 Scheme of SRs-2000 bucket-wheel excavator operation in mine face

2.3.1.4 Fifth Extraction Bench (scheme one)

The fifth bench includes an interlayer of the overburden of 2m above 11.2m thick ore layer. The ore mineral sand layer is waterlogged, so the interlayer of the overburden ensures dryness on the working machine level. The development of overburden is to be done by dragline ESh-14-50 employing direct dumping method. Waste rock is dumped directly in в mined-out space, and ore rock is loaded in trucks CAT 777 to be delivered on an interim dump place. The scheme of extraction bench development is shown in Figure 16.

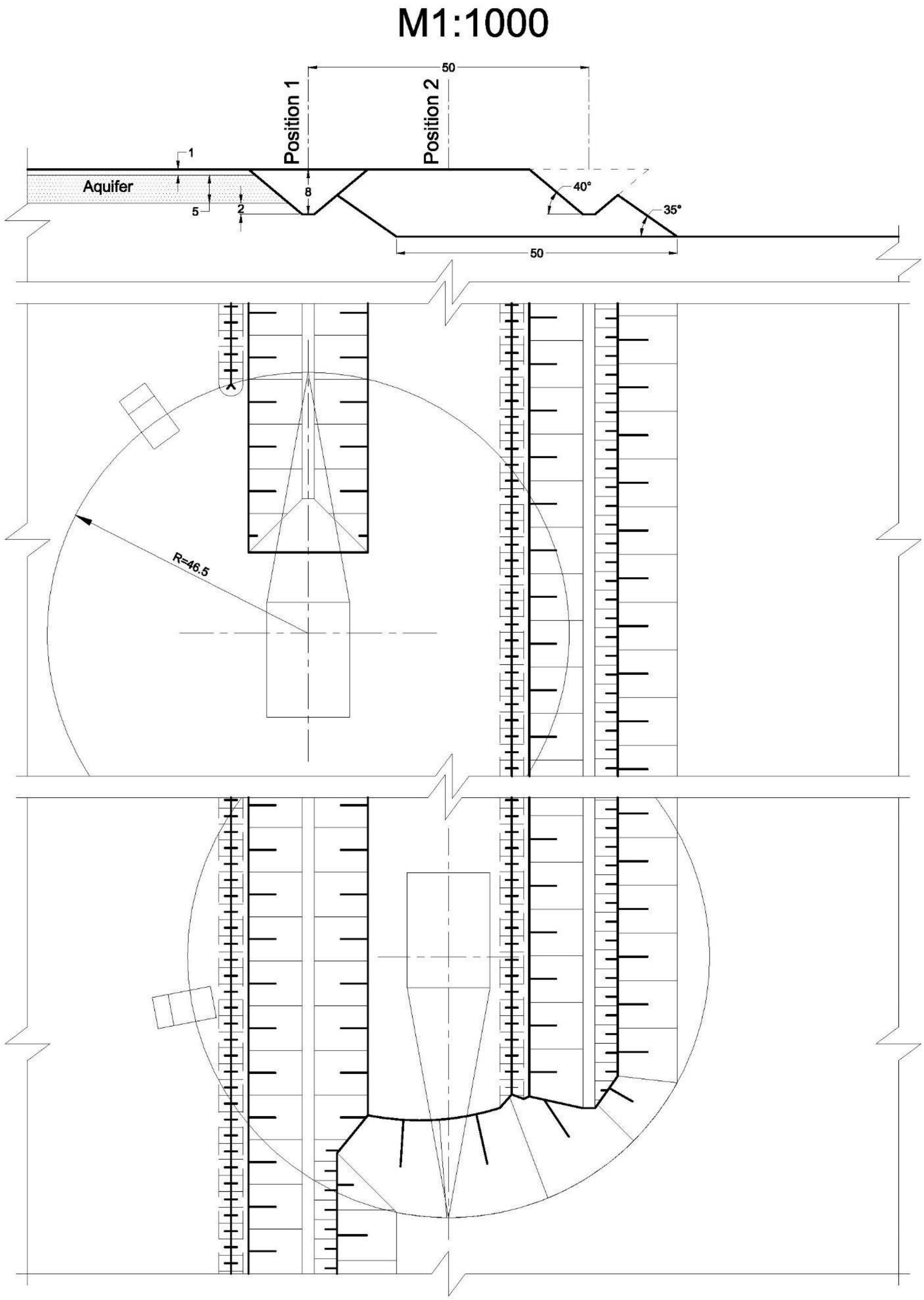


Figure 15 Scheme of the aquifer bearing bench development by ESh-14-50 dragline excavator

The overall thickness of the extraction bench is 13,2m. Annual production is 3 203,4 thousand m^3, including 2 714,7 thousand m^3 of ore and 488,7 thousand m^3 of waste rock per year.

2.3.2 Technological Scheme with the use of Dragline Excavators

2.3.2.1 Front Bench (scheme two)

First bench in the technological scheme with the use of draglines is developed the same way as in the technological scheme with the use of bucket-wheel excavator using the same time schedule. Stripping of the potentially fertile layer is done by dragline ESh-14-50 with loading on trucks CAT 777. Thickness is up to 1m and annual productivity is 316,7 thousand m^3.

2.3.2.2 Second, Third and Fourth Overburden Benches (scheme two)

In the given technological scheme, all striping and extraction benches are developed by ESh-14-50 with loading in CAT 777.

Maintenance of roads and approaches is supposed to be done by bulldozers CAT-D8R and graders CAT-16M. Disposition of the waste rocks in dumping benches is done according to its natural order.

Thicknesses of second, third and fourth overburden benches are 7, 8 and 13 meters respectively. The overall volume of overburden rocks that should be mined out per year in these benches is 8 241,2 thousand m^3.

2.3.2.3 Fifth and Sixth Overburden Benches (scheme two)

Fifth and sixth overburden benches in the technological scheme with the use of draglines are developed identically to third and fourth benches in the technological scheme with the use of bucket-wheel excavator by draglines ESh-14-50 with loading in trucks CAT 777. Construction and servicing of roads are carried out by bulldozers CAT-D8R and graders CAT-16M.

Thicknesses of fifth and sixth overburden benches are 12 and 9 meters, with the average annual productivity of 4 921,1 thousand m^3.

2.3.2.4 Seventh Extraction Bench (scheme two)

The development of the seventh extraction bench is identical to the development of the fifth extraction bench in the technological scheme with the use of a bucket-wheel excavator. Interlayer of the waste rock of 2m it dumped in the mined-out space by excavator ESh-14-50. The ore send is loaded in trucks CAT 777. The overall thickness of the extraction bench is 13,2m. Annual production is 3 203,4 thousand m^3, including 2 714,7 thousand m^3 of ore and 488,7 thousand m^3 of waste rock per year.

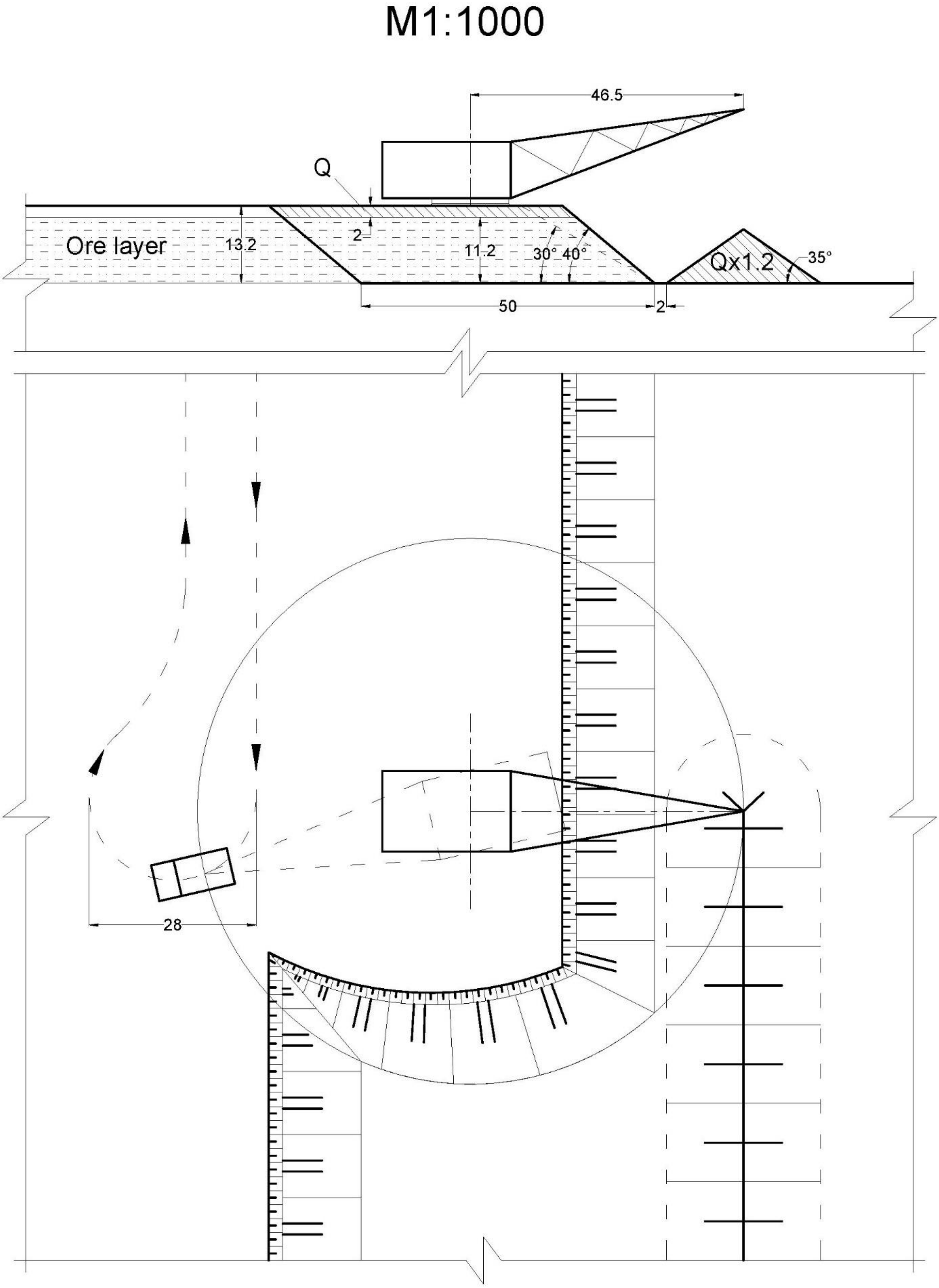


Figure 16 Scheme of the extraction bench development by ESh-14-50 dragline excavator

2.3.3 Technological Scheme with the use of Hydraulic Excavators

2.3.3.1 Front Bench (scheme three)

Hydraulic backhoe excavator CAT 375L is used for the removal of the potentially fertile layer in this technological scheme with loading in CAT 725. The main technical specifications of this equipment are shown in Tables 15 and 16. Working parameters of CAT 375L are presented in Figure 17. Dimensions of the truck CAT 725 are given in Figure 18. The same equipment is used for drainage trenches construction on the fifth bench, which will ensure the interchangeability of the equipment.

According to practical experience from the mines in identical geological conditions, the average coefficient of volume efficiency of the hydraulic excavator's bucket is 1.0. For filling of the dump truck's body, it is necessary five buckets (5 x 2,4 x 1.0 = 12 m3, less than nominal capacity 14,3 m3). The fragmentation index is 1.2, therefore, average weight of loaded rock is 12 ÷ 1,2 x 1,9 = 19t and maximal 12 ÷ 1,2 x 2 = 20t, what does not exceed the nominal load-carrying capacity 23,6t. Also, if required, it is possible to load an additional 0,5-0,8 of the bucket.

The potentially fertile layer is moved into internal dumps and is used for the recultivation of disturbed lands. Bulldozers CAT-D8R is to be used for the evening of the surface. For the road's maintenance CAT-16M is used.

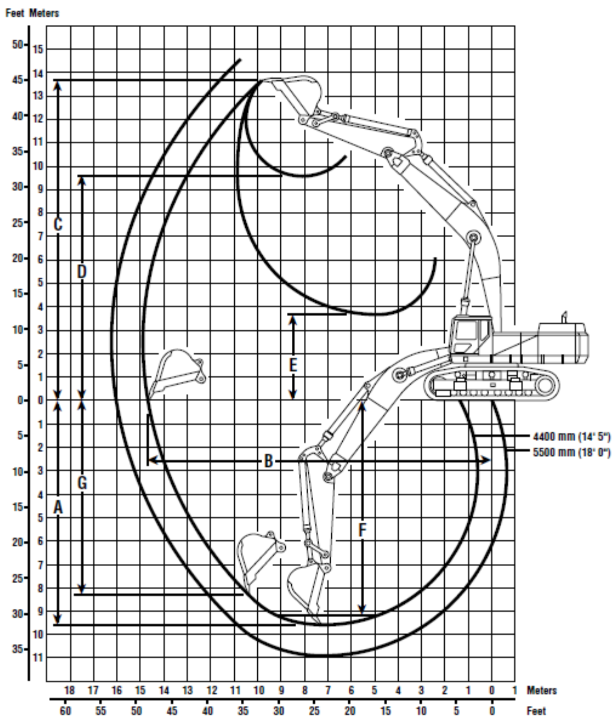
2.3.3.2 Second, Third and Fourth Overburden Benches (scheme three)

All overburden benches before an aquifer bearing bench are developed by hydraulic excavators CAT 6018 FS with loading into dump trucks CAT 777. The choice of track is conditioned by the possibility to interchange them with the tracks that are used on the ore extraction bench in combination with draglines ESh-14-50. This decision supports the optimization of maintenance and repair of transporting equipment. Technical specifications of CAT 6018 FS are given in Table 17 and its working parameters and dimensions of the excavator presented in Figures 19 and 20 respectively.

If the fragmentation index is 1.2, it is necessary five buckets to fill the truck (5 x 10 = 50 m^3, less than truck's body capacity 64.1m^3). Therefore, average weight of the loaded rock is 50 ÷ 1,2 x 1,9 = 79,2 t and the maximum 50 ÷ 1,2 x 2 = 83,3 t, what does not exceed the nominal payload 90,8t.

Table 15 Technical specifications of CAT 375L hydraulic excavator [15]

Bucket capacity, m^3	2,4
Engine power, kW	319
Shipping Height, m	5.31
Shipping Length, m	14.65
Shipping width, m	3.50
Maximum digging depth, m	11
Weight of excavator, kg	82,380



Stick	R5.5H 5500 mm (18'1")
Bucket	2.8 m^3 (3.75 yd^3)
A Maximum digging depth	10 840 mm (35'7")
B Maximum reach at ground level	15 960 mm (52'4")
C Maximum cutting height	14 500 mm (47'7")
D Maximum loading height	10 350 mm (33'11")
E Minimum loading height	2460 mm (8'1")
F Maximum digging depth at 2440 mm (8') level bottom	10 750 mm (35'3")
G Maximum vertical wall digging depth	9390 mm (30'10")
Bucket forces	282 kN (63,400 lb)
Stick forces	207 kN (46,400 lb)

Figure 17 Working parameters of CAT 375L hydraulic excavator [15]

Maintenance of roads, approaches and places for the dumping of the rock is done by bulldozers CAT-D8R and graders CAT-16M. Disposition of the waste rocks in dumping benches is done according to its natural order.

Thicknesses of the second, third and fourth benches are 9, 9 and 10 meters respectively. The overall volume of annual production on these benches is 8 591,7 thousand m^3.

2.3.3.3 Fifth and Sixth Overburden Benches (scheme three)

There is an aquifer within the fifth bench. Its thickness is up to 5 meters. Hydraulic excavator back shovel CAT 375L constructs drainage tranches that are used for draining the mined-out rocks with loading in trucks CAT 725. Scheme of development if this bench by hydraulic excavators is given in Figure 21. Also, the organization of work on this bench is presented in Attachment №3. Main rock volumes are developed by the same equipment as on all others overburden benches.

Table 16 Technical specifications of CAT 725 dump truck [16]

Nominal loading capacity, t	23,6
Body capacity, m^3	14,3
Engine power, kW	230
Target Gross Machine weight, kg	45 850

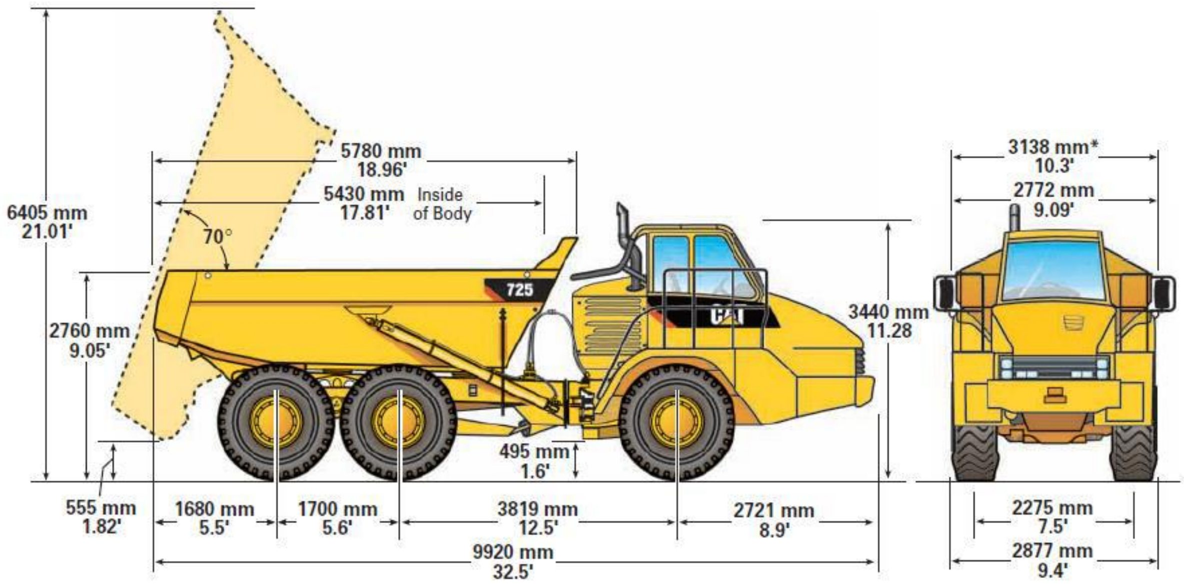
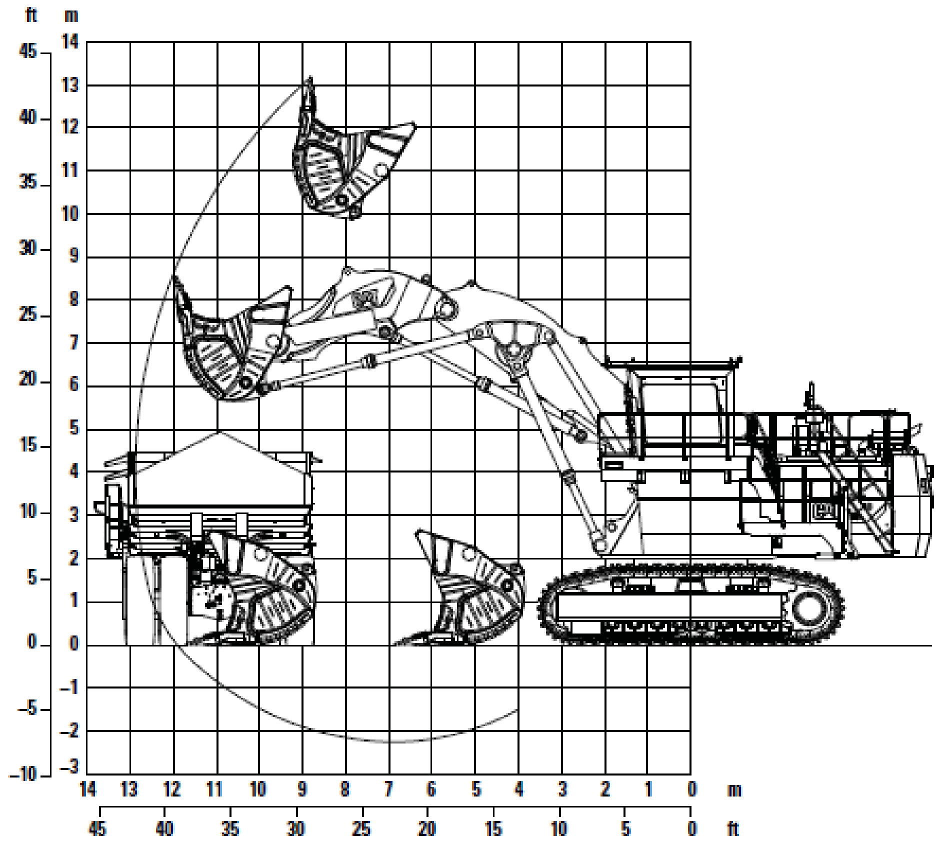


Figure 18 Dimensions of CAT 725 dump truck [16]

Thicknesses of the fifth and sixth overburden benches are 12 and 9 meters respectively. The overall volume of annual production is 5 640,3 thousand m^3.

Table 17 Technical specifications of CAT 6018 FS hydraulic excavator [17]

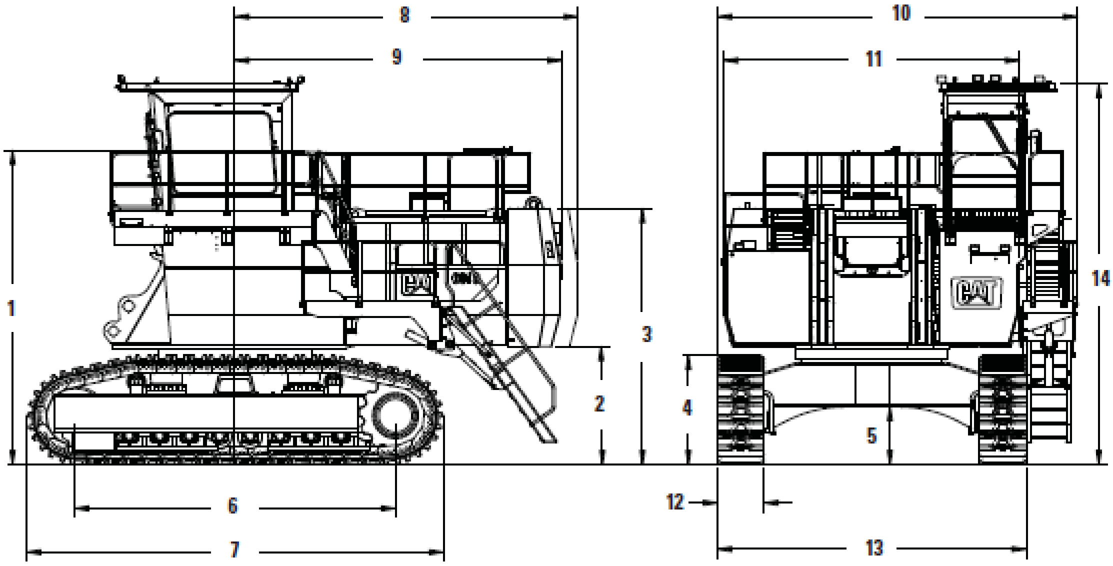
Bucket capacity, m^3	10,0
Engine power, kW	858
Maximum digging height, m	13,2
Maximum digging radius, m	12,9
Maximum digging depth, m	2,3
Maximum unloading height, m	10,1
Weight of excavator, t	183



Boom	6.35 m	20 ft 10 in
Stick	4.1 m	13 ft 5 in
Digging Forces		
Maximum crowd force	910 kN	204,500 lbf
Maximum crowd force at ground level	810 kN	182,030 lbf
Maximum breakout force	730 kN	164,050 lbf

Working Range		
Maximum digging height	13.2 m	43 ft 4 in
Maximum digging reach	12.9 m	42 ft 4 in
Maximum digging depth	2.3 m	7 ft 7 in
Maximum dumping height	10.1 m	33 ft 2 in
Crowd distance on level	4.8 m	15 ft 9 in

Figure 19 Working parameters of CAT 6018 FS hydraulic excavator [17]



1	5400 mm	17 ft 9 in	8	5890 mm	19 ft 4 in
2	2040 mm	6 ft 8 in	9	5610 mm	18 ft 5 ft
3	4400 mm	14 ft 5 in	10	6190 mm	20 ft 4 in
4	1880 mm	6 ft 2 in	11	5080 mm	16 ft 8 in
5	1000 mm	3 ft 3 in	12	800 mm	2 ft 7 in
6	5500 mm	18 ft 1 in	13	5300 mm	17 ft 5 in
7	7120 mm	23 ft 4 in	14	6560 mm	21 ft 6 in

Figure 20 Dimensions of CAT 6018 FS hydraulic excavator [17]

2.3.3.4 Seventh Extraction Bench (scheme three)

Development of the extraction bench in the technological scheme with the use of hydraulic excavators is identical to the development of this bench in the technological scheme with the use of bucket-wheel excavator. Use of dragline excavators for the extraction bench development is conditioned by its height and water-bearing nature of the rock below the ore layer. An interlayer of overburden rock is dumped directly in the worked-out space by draglines ESh-14-50. Ore is to be loaded in trucks CAT 777.

The overall thickness of this bench is 13,2m. Annual production is 3 203,4 thousand m^3, including 2 714,7 thousand m^3 of ore and 488,7 thousand m^3 of waste rock.

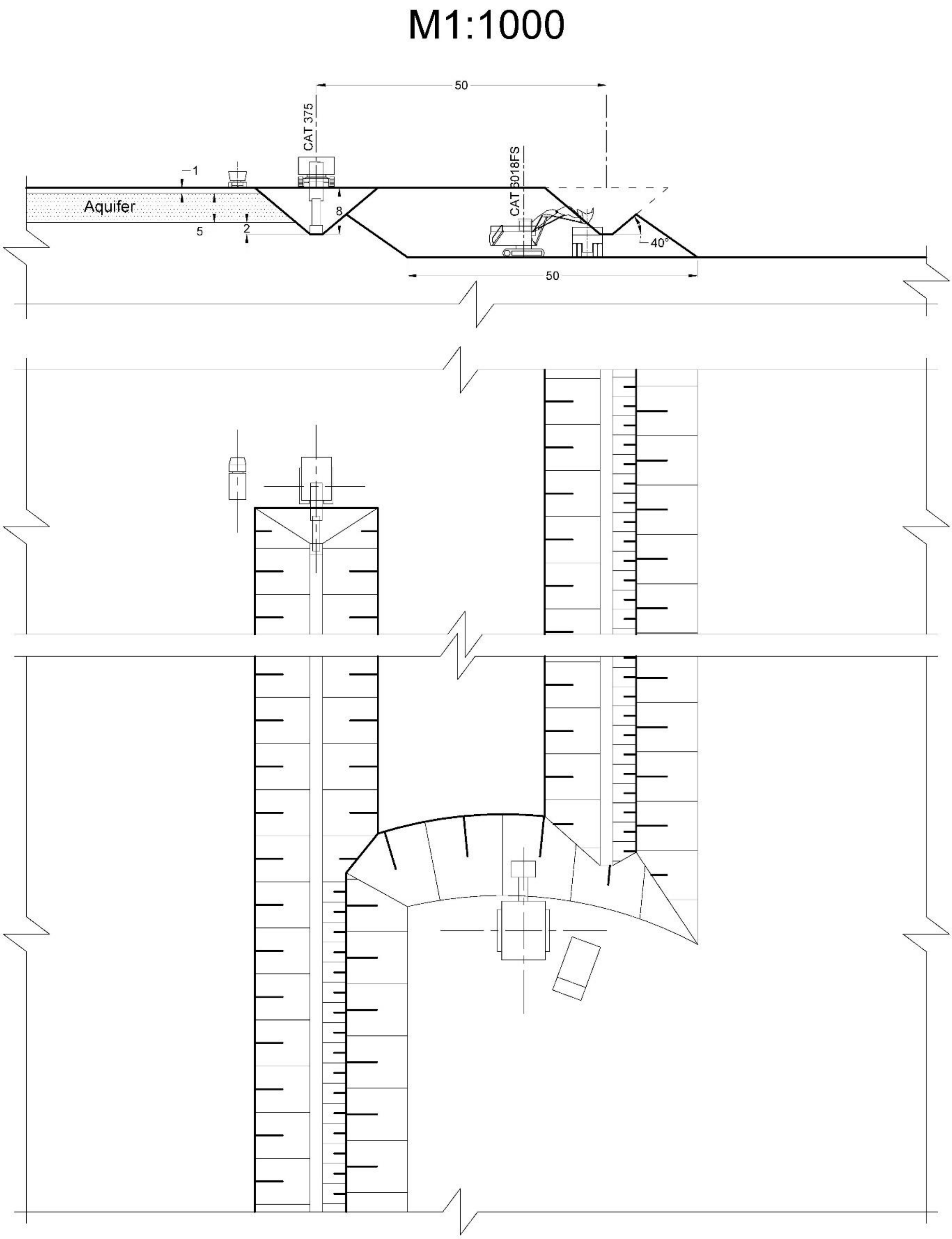


Figure 21 Scheme of the aquifer bearing bench development by hydraulic excavators (CAT 6018 & CAT 375L)

Chapter 3 Economic Evaluation and Technological Schemes Comparison

This chapter describes the economic evaluation of the schemes. For their comparison will be used capital and operational expenditures on purchase and supply of main and ancillary equipment.

3.1 Calculation of Capital Expenditures

3.1.1 Cost Estimation of Technological Scheme with the use of Bucket-Wheel Excavator

Overall capital expenditures on the acquisition of all the necessary equipment on the bench developed by BWE include bucket-wheel excavator itself, overburden spreader, conveying system and ancillary equipment (belt shifters, reloaders etc.). The whole range of equipment, their necessary quantity and prices are listed in Table 18.

Table 18 Capital expenditures on the bucket-wheel excavator and it's ancillary equipment

Equipment		Price per unit [thousand dollars]	Quantity [units]	Total cost [thousand dollars]
Bucket-wheel excavator SRs-2000		38 360	1	38 360
Face conveyer	Conveyor	4 790	2	9 580
	Receiving hopper	165	2	330
Trunk conveyer	Conveyor	4 790	2	9 580
	Receiving hopper	165	2	330
Stacking conveyer	Conveyor	4 790	2	9 580
	Receiving hopper	165	2	330
	Throw-off carriage	2 330	2	4 660
Cable transporter		3 420	1	3 420
Overburden spreader		17 810	1	17 810
Reloader		5 480	3	16 440
Belt shifter		600	3	1 800
Excavator CAT-312 for conveyor belt cleaning		190	1	190
Overall				112 410

The full list of main and ancillary equipment on all benches used in the first technological scheme is given in Table 19.

Table 19 Overall capital expenditures on the technological scheme with the use of bucket-wheel excavator

	Equipment	Price per unit [thousand dollars]	Quantity [units]	Total cost [thousand dollars]
Set of main and ancillary equipment on the bench developed by bucket-wheel excavator		112 410	1	112 410
Main	ESh-14-50 (NKMZ)	8 100	9	72 900
	CAT 777	980	25	24 500
Ancillary	Bulldozer CAT D8R	530	8	4 240
	Grader CAT 16M	450	3	1350
	Mobile workshop (adverse terrain vehicle)	150	2	300
	Vulcanizing equipment	400	2	800
	Crawler crane 100t	820	1	820
	Tire crane 40t	400	2	800
	Tire crane 25t	270	2	540
Overall				218 660

Overall equipment cost in technological scheme with the use of bucket-wheel excavator is 218 660 thousand dollars.

3.1.2 Cost Estimation of Technological Scheme with the use of Dragline Excavators

In the given technological scheme on all benches draglines ESh-14-50 in combination with trucks CAT 777 are used. Overall expenses on the acquisition of the main and ancillary equipment in the second technological scheme are given in Table 20.

Overall equipment cost in the technological scheme with the use of draglines is 184 240 thousand dollars.

3.1.3 Cost Estimation of Technological Scheme with the use of Hydraulic Excavators

In the given technological scheme on all overburden benches are used hydraulic excavators CAT 6018 FS with loading in trucks CAT 777. Black soil striping and dragline trances construction is carried out by hydraulic excavators CAT 375L in combination with trucks CAT 725. The extraction bench is to be developed by draglines ESh-14-50 with loading in trucks CAT 777.

All expenses on the acquisition of the main and ancillary equipment in the third technological scheme are given in Table 21.

Table 20 Overall capital expenditures on the technological scheme with the use of draglines

Equipment		Price per unit [thousand dollars]	Quantity [units]	Total cost [thousand dollars]
Main	ESh-14-50 (NKMZ)	8 100	16	129 600
	CAT 777	980	47	46 060
Ancillary	Bulldozer CAT D8R	530	9	4 770
	Grader CAT 16M	450	3	1350
	Mobile workshop (adverse terrain vehicle)	150	2	300
	Crawler crane 100t	820	1	820
	Tire crane 40t	400	2	800
	Tire crane 25t	270	2	540
Overall				184 240

Overall equipment cost in the technological scheme with the use of bucket-wheel excavator is 95 780 thousand dollars.

3.1.4 Comparison of Technological Schemes by its Capital Expenditures

All capital expenditures of all three technological schemes are presented on a graph in Figure 22.

A comparison of technological schemes shows that the most expensive by its initial investment will be the technological scheme with the use of bucket-wheel excavator. The minimal capital expenses will be in the technological scheme with the use of hydraulic excavators (twice cheaper).

Table 21 Overall capital expenditures on the technological scheme with the use of draglines

Equipment			Price per unit [thousand dollars]	Quantity [units]	Total cost [thousand dollars]
Main	Excavators	ЭШ14/50 (NKMZ)	8 100	3	24 300
		CAT 6018 FS	2 720	5	13 600
		CAT 375L	1 200	2	2 400
	Trucks	CAT 777	980	44	43 120
		CAT 725	270	14	3 780
Ancillary	Bulldozer CAT D8R		530	9	4 770
	Grader CAT 16M		450	3	1350
	Mobile workshop (adverse terrain vehicle)		150	2	300
	Crawler crane 100t		820	1	820
	Tire crane 40t		400	2	800
	Tire crane 25t		270	2	540
Overall					95 780

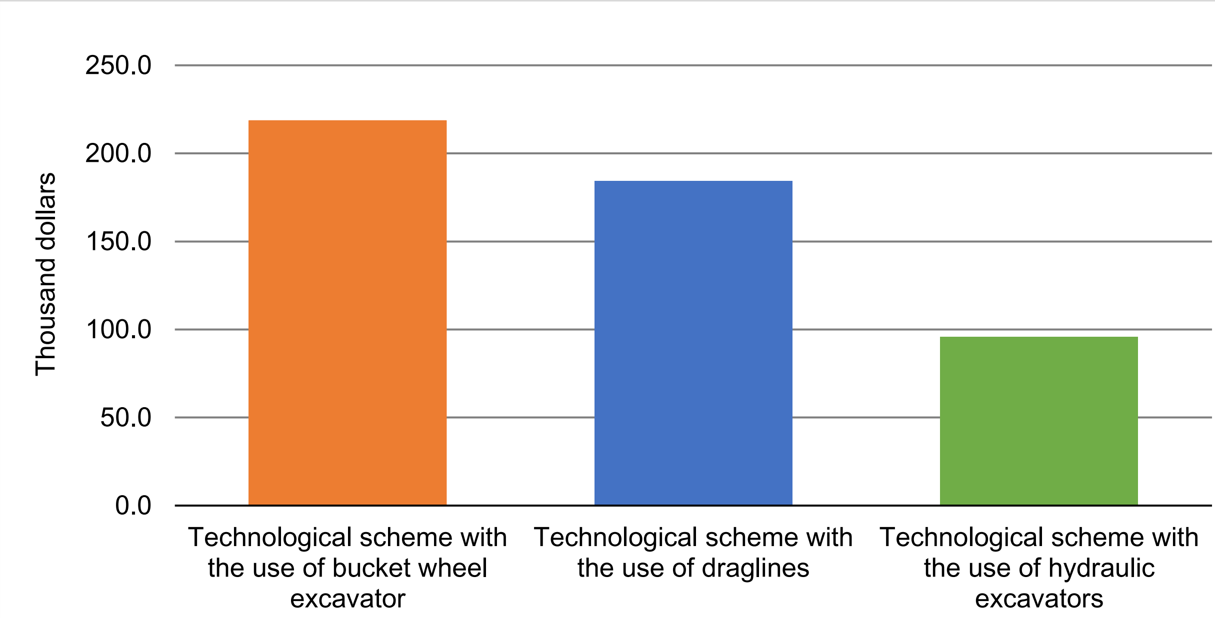


Figure 22 Comparison of the technological schemes by their capital expenditures

3.2 Calculation and Comparison of Operational Expenditures

The accuracy of estimation of capital and operating costs depends on the quality of the technical assessment and knowledge of expected mining and mineral processing conditions [3]. For calculation of operational costs will be used methodologies from O'Hara and Suboleski.

The cost guides provided in the form *cost* = KT^x compute the cost of each mining activity as a function of the tons T of ore mined and express this cost as a cost per day of mining activity. The cost per ton can be derived from the formula by dividing the cost per day by the tons mined per day, so that *cost per ton* = KT^x/T [3].

The costs estimated in a preliminary feasibility study are unlikely to be more accurate than ± 20%, and this degree of accuracy is insufficient to provide a sound basis for major mine financing or confident assurance of a profitable mining operation [3].

All the formulas below were taken from SME Mining Engineering Handbook by Howard L. Hartman (1992) [3].

The cost of open pit mining can be assessed against the total ore and waste tonnage (T_p) mined daily.

Loading cost per day = \$2.67 T_p *0.7*

Haulage cost per day = \$18.07 T_p *0.6*

General services cost per day = \$6.65 T_p *0.7*

The open pit general services cost includes the cost of pit maintenance, road grading, waste dump grading, pumping, and open pit supervision.

The number of mine personnel required in open pit mines with competent soft rock may be estimated from the following formula in which op N_{op} is a number of open pit personnel and T_p is tons of ore and waste mined daily.

$N_{op} = 0.024\ T_p^{0.8}$

The number of service personal for open pit mining low-grade ore N_{sv} is to be estimated as a percentage of mine personal as shown below:

$N_{sv} = 25.4\ \%\ of\ N_{op}$

The number of administrative and technical personnel N_{at} required for open pit mines may be estimated as a percentage of the total required for mining, milling, and services: *Nat = 11% of* ($N_{op} + N_{sv}$)

The average cost per day for mine personnel required on open pit mines in Ukraine is 50 dollars. For service personal - 45 dollars. For administrative and technical personnel, the average daily cost is 75 dollars. The results of calculations, including cost per day of mining activity and the cost per ton for every technological scheme, are presented in Table 22. In the table the first goes technological scheme with the use of bucket-wheel excavator, the second is the technological scheme with the use of draglines, and the third the technological scheme with the use of hydraulic excavators.

It is clear that when exploring this method, the difference between the operational costs of different technological schemes is conditioned by difference between rock volumes that should be stripped out to reach the ore layer. This volume depends on the overall slope inclination, which differs depending on used equipment (Attachments №1, №2 & №3) operational expenditures will have the technological scheme with the use of bucket-wheel excavators. Still, according to this calculation method, the difference between them is not significant.

Table 22 Results of operational expenditures estimation

Parameter or mining activity	scheme one	scheme two	scheme three
Volume of the rock mined per day, m^3	49 790	51 740	53 510
Average specific weight, t/m^3	2	2	2
Rock mined per day, t	99 580	103 480	107 020
Loading cost per day, dollars	8418	8648	8854
Haulage cost per day, dollars	18 024	18 445	18 821
General services cost per day, dollars	20 967	21 539	22 052
Number of mine personnel, person	240	247	254
Cost of mine personnel per day, dollars	12 000	12 350	12 700
Number of service personal per day, person	61	63	65
Cost of service personnel per day, dollars	2 745	2 835	2 925
Number of administrative and technical personnel, person	34	35	36
Cost of administrative and technical personnel per day, person	2 550	2 625	2 700
Overall operational expenditures, dollars	64 705	66 441	68 052
Cost per ton, dollars/t	0.650	0.642	0.636

Chapter 4 Technical and Economic Dependencies

In this chapter, the economic and technological dependencies when changing the technological parameters of the mine (transportation distance, operating mode, equipment parameters, etc.) are studied and annualized. Also, the substantiates the choice of main mining equipment considering the optimal use financial investments is carried out.

4.1 Justification for the Main Mining Equipment Selection

4.1.1 Selection of a Bucket-Wheel Excavator

To compare equipment in this chapter, a simplified version of calculating the volume of rocks is used. The average overburden coefficient for the Motronivka placer is five m^3/m^3.

In this paper, in consideration of information availability, four options for bucket-wheel excavators were compared: KR5600Nk-2, SRs1500, SRs2000 and SRs2400. The calculated data and the technical specifications are presented in Table 23. The graph is shown in Figure 23.

Table 23 Characteristics of the possible options for a bucket-wheel excavator

Bucket-wheel excavators	KR5600Nk-2	SRs1500	SRs2000	SRs2400
Hourly performance of a bucket-wheel excavator, m^3/hour	4400	3200	4300	5100
Utilization factor	0.7	0.7	0.7	0.7
Availability rate	0.85	0.85	0.85	0.85
Fragmentation index of the rock in the bucket	1.2	1.2	1.2	1.2
Maximum annual productivity of the bucket-wheel excavator, thousand m^3/year	14 500,0	10 500,0	14 200,0	16 800,0
Nominal bench height, m	20	24	32	35
Weight of the excavator, t	1500	2600	2910	4238
Price of the excavator, thousand dollars	19 770	34 270	38360	55 870
Required number of machines, units	2	2	1	1
Overall costs of the excavators, thousand dollars	39 540	68 540	38 360	55 870

The average thickness of overburden in the field is approximately 50m. The bucket-wheel excavator may be used on the upper 28m above the aquifer (Attachment №1). Only excavators SRs2000 and SRs4000 can mine it by one bench. Between these two excavators, the SRs2000 has lower capital costs; therefore, it is rational to use this particular machine. In terms of capital costs, the acquisition of SRs2000 is approximately equal to the costs for the purchase of two KR5600Nk-2. But, considering the higher operating costs (energy costs, staff remuneration, etc.), it makes sense to choose SRs2000.

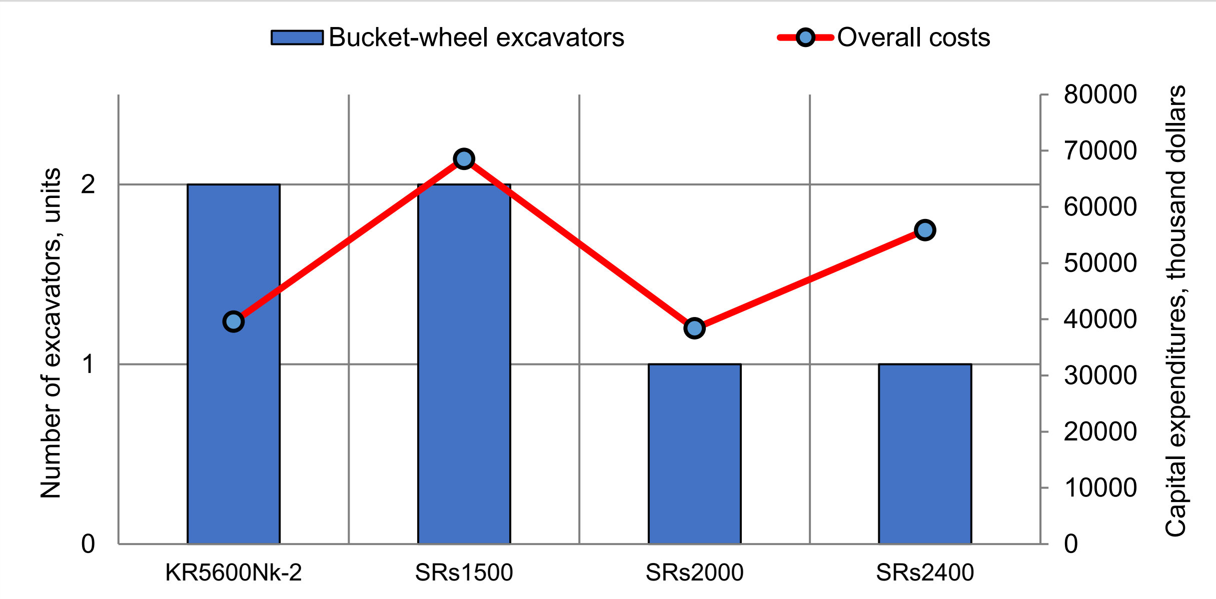


Figure 23 Comparison of bucket-wheel excavators

4.1.2 Selection of Dragline Excavators

In this paper, draglines produced by a local manufacturer NKMZ [12] are compared. Considering the low bearing capacity of the rocks in the given geological conditions, excavators ESh-6.5-45, ESh-10-50, ESh-14-50, which can mine the necessary bench height, are considered. Excavators ESh-10-70, ESh-20-65, ESh-15-90N and ESh-20-90N are not considered as loading into dump trucks with such a long boom is too difficult. In addition, a longer boom means a larger counterweight, which in turn affects the overall weight of the excavator. This increases the specific pressure, which for ESh-15-90N and ESh-20-90N [12] excavators exceed the maximum allowable for given geological conditions.

Caterpillar Inc. dump trucks are considered for rock transportation. The machines of this manufacturer in terms of price and technical specifications approximately correspond to other leading world brands (Komatsu Ltd., Volvo Group, etc.).

The comparison of draglines is made considering their necessary quantity and capital expenditures for the development of the entire mine (approximately 16 200 m^3/year). The general calculation equation of the annual production of the excavators is presented below.

$$Q_{ex.year.} = \frac{3600 \cdot V_b \cdot K_{v.e.} \cdot K_u \cdot (T_{sh} - T_{sh.t}) \cdot n \cdot N_d}{t_c \cdot K_f} \text{ [m}^3\text{/year]} \qquad \textbf{(12)}$$

where: V_b - bucket capacity, m^3;

$K_{v.e.}$ - coefficient of volume efficiency of the bucket;

K_u - utilization factor of the excavator;

T_{sh} - duration of the shift, hour;

$T_{sh.t}$ - time, which is necessary per shift turnover, dinnertime and preventive maintenance of the equipment, hour;

n - number of shifts, shift;

N_d - average number of working days per year, day;

t_c - average cycle time for one bucket shoveling, sec;

K_f - fragmentation index of the rock.

With the work schedule of 2 shifts per 12 hours each, a cycle time of 60 seconds and 310 working days per year, the annual output of ESh-6.5-45 is 1,170.0 thousand m^3/year; ESh-10-50 - 1800.0 thousand m^3/year and ESh-14-50 - 2 520.0 thousand m^3/year (Attachment №7). Therefore, for the development of the necessary rock volume are required: 14 excavators ESh-6.5-45; 9 excavators ESh-10-50 or 7 excavators ESh-14-50.

The calculation of the time required per one trip of the truck and the calculation of the possible number of trips per shift are done according to the equations №9 and

№10 respectively (page 33). The rough calculation of the required number of trucks from the annual production of every machine and average way distance is given below.

$$N_{truck} = \frac{Q_{year} \cdot K_f}{N_d \cdot N_{trips} \cdot V_b \cdot n_b \cdot K_{v.e.} \cdot K_u \cdot n} \text{ [units]} \quad \mathbf{(13)}$$

where: N_{trips} - calculation of the possible number of trips per shift, trips.

Therefore, the time required per one trip of CAT 772 and CAT 777 when working with the selected excavators is the same - 29 minutes, for CAT 773 - 28 minutes. Then, CAT 772 and CAT 777 can perform 20 trips per shift, and CAT 773 - 21 trip. Consequently, the required number of CAT 772 dump trucks when loading by ESh-6.5/45 excavators is 104 vehicles, CAT 773 when loading by ESh-10-50 excavators - 86 vehicles and CAT 777 when loading ESh-14-50 excavators - 49 vehicles.

According to the data obtained, the most appropriate choice under these conditions is ESh-14-50 in combination with CAT 777 dump trucks (Table 24, Figure 24). Thought the overall cost of draglines ESh-6.5-45 with trucks CAT 772 is lower, their operational expenditures will be significantly higher because of the bigger required equipment number.

Table 24 Characteristics of the possible options of draglines

Dragline:	ESh-6.5-45	ESh-10-50	ESh-14-50
Bucket capacity, m^3	6.5	10	14
Maximum digging depth, m	22	21	21
Weight of the excavator, t	280	622	620
Price of the excavator, thousand dollars	3 660	8 125	8 100
Required number of excavators, units	14	9	7
Overall costs of the excavators, thousand dollars	51 212	73 135	56 700
Dump truck:	CAT 772	CAT 773	CAT 777
Body capacity, m^3	31.3	41.9	64.1
Payload, t	46	55	90.8
Necessary number of buckets to fill the truck, bucket	4	3	4
Weight of the truck, t	82.1	102.74	164.65
Price of the truck, thousand dollars	490	610	980
Required number of trucks, units	104	86	49
Overall costs of the trucks, thousand dollars	50 960	52 460	48 020
Overall costs of the main equipment, thousand dollars	102 173	125 595	104 720

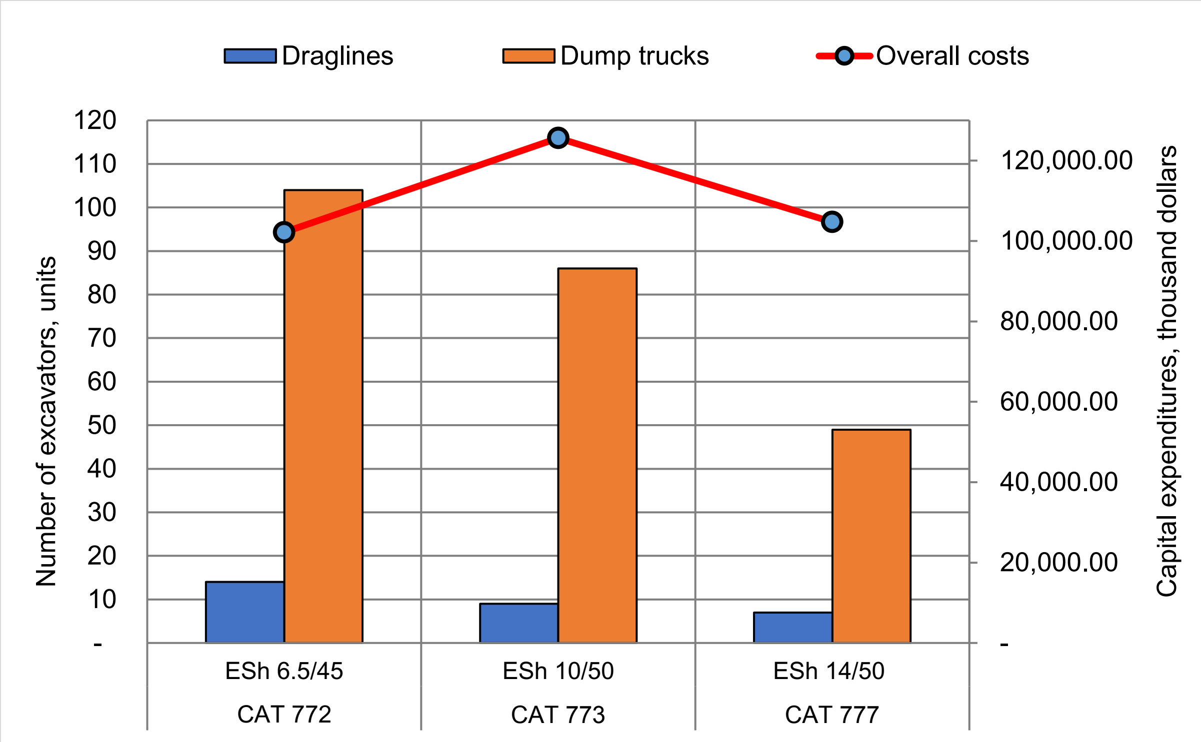


Figure 24 Comparison of dragline excavators

4.1.3 Selection of Hydraulic Excavators

According to the geological conditions on the Motronivka placer, it is possible to use a front shovel hydraulic excavator on the upper benches above the aquifer without drainage measures. In this work, the excavators produced by Caterpillar Inc. with bucket volume from 5.7m^3 (CAT 385C FS) to 16.5m^3 (CAT 6030 FS) are considered. The machines of this manufacturer, in terms of price and technical specifications, approximately correspond to other leading world brands (Komatsu Ltd., Volvo Group, etc.), which makes the selection of equipment relatively objective. Excavators with large bucket volumes are not offered because the dump trucks to serve such excavators exceed the allowable value of specific ground pressure. Caterpillar Inc. dump trucks are also considered for rock transportation.

The calculation of the required number of hydraulic excavators in combination with the corresponding dump trucks is carried out by analogy to the calculation of the required number of draglines. The rock volume (to ensure the required annual ore production of 2,700.0 m^3/year) is approximately 7,600.0 thousand m^3.

Table 25 Characteristics of the possible options for hydraulic excavators

Hydraulic excavator:	CAT 385C FS	CAT 6015 FS	CAT 6018 FS	CAT 6030 FS
Bucket capacity, m^3	5.7	8.1	10	16.5
Maximum digging height, m	11.2	11.0	13.0	13.8
Weight of the excavator, t	90.6	140	183	294
Price of the excavator, thousand dollars	1 350	2 080	2 720	4 370
Required number of excavators, units	5	4	3	2
Overall costs of the excavators, thousand dollars	6 750	8 320	8 160	8 740
Dump truck:	CAT 770	CAT 773	CAT 777	CAT 785
Body capacity, m^3	25.1	35.8	64.1	78
Payload, t	36.3	55	90.8	131
Necessary number of buckets to fill the truck, bucket	3.5	3.5	5	4.5
Weight of the truck, t	71.2	102.7	164.7	249.5
Price of the truck, thousand dollars	425	610	980	1485
Required number of trucks, units	61	43	25	17
Overall costs of the trucks, thousand dollars	25 925	26 230	24 500	25 245
Overall costs of the main equipment, thousand dollars	32 675	34 550	32 660	33 985

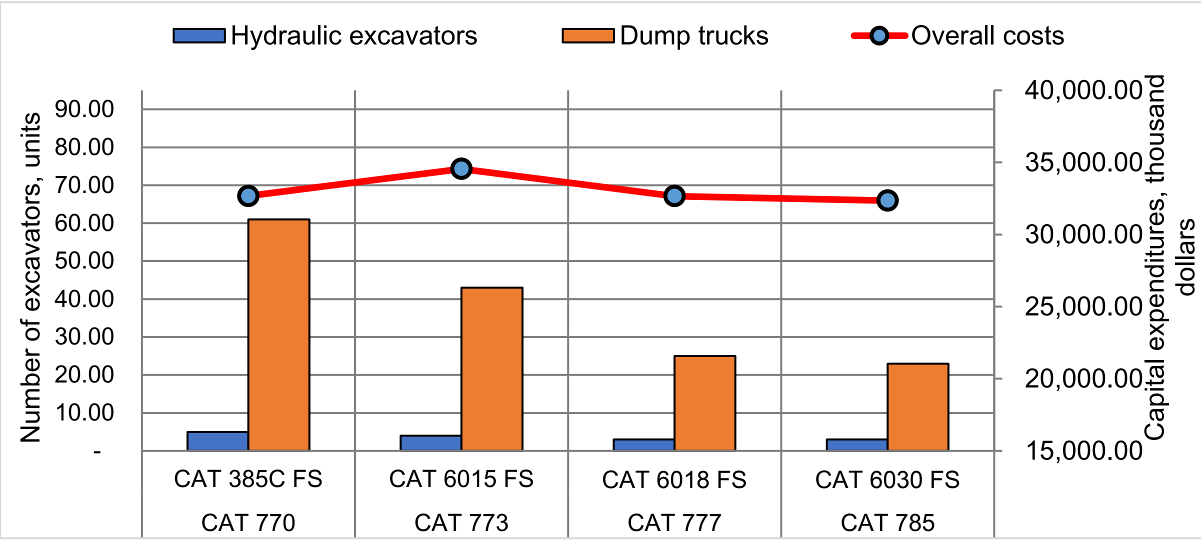


Figure 25 Comparison of hydraulic excavators

With the work schedule of 2 shifts per 12 hours, 310 working days a year, fragmentation index of 1.3 and the cycle time of 40 seconds, the annual production capacities of the hydraulic excavators are: CAT 385C FS - 1540.0 m^3/year, CAT 6015

FS – 2190.0 m^3/year; CAT 6018 FS - 2700 m^3/year; CAT 6030 FS - 4460 m^3/year (Attachment №7).

Therefore, to ensure the necessary overburden productivity, it is necessary to use 5 CAT 385C FS; 4 excavators CAT 6015 FS; 3 excavators CAT 6018 FS and 2 excavators CAT 6030 FS.

The time spent on one trip by CAT 770 and CAT 773 dump trucks, which can work together with excavators CAT 385C FS and CAT 6015 FS respectively, is 27.67 minutes. The time of CAT 777 dump truck that can serve excavator CAT 6018 FS is 28.33 minutes. CAT 785 dump trucks, which may be combined with excavators CAT 6030 FS, can perform one trip also in 28.33 minutes (Attachment №7).

Therefore, dump trucks CAT 770 and CAT 773 can make 22 trips per shift. Dump trucks CAT 785 and CAT 777 – 21 trips.

In order to ensure the required overburden productivity, it is necessary to use 61 trucks CAT 770 and 43 trucks CAT 773. There are necessary 25 trucks CAT 777 when working in combination with excavators CAT 6018 FS and 17 trucks CAT 785 are enough when working excavators CAT 6030 FS.

According to the calculations, the most optimal option is the CAT 6018 FS excavator in combination with CAT 777 dump trucks (Table 25, Figure 25). With the relatively low number of excavators, this option is chipper comparing to more productive excavators. The percentage of usage decreases significantly with increasing excavator capacity. Also, this option, when compared with less productive excavators, can be serviced by fewer dump trucks, which reduces overall operating costs.

4.2 Dependence of the Required Main Mining Equipment Number on Work Schedule

In this work, 2 possible work schedules of the mine are compared:

- twenty-four-hour work on 2 shifts per 12 hours without days off; and
- twenty-four-hour work on 3 shifts per 8 hours without days off.

4.2.1 Comparison of the Required Equipment Number for Technological Scheme with the use of Bucket-Wheel Excavator

The average total volume of overburden, taking into account the average stripping ratio of 5 m^3/m^3, is 13,500.0 thousand m^3 per year. It is also necessary to take into account the volume of mineral resources of 2,700.0 thousand m^3. Of these, 7,600.0 thousand m^3 is being developed by a bucket-wheel excavator. Consequently, 8,600.0 thousand m^3 are mined by draglines. The required number of excavators ESh-14-50 and dump trucks CAT 777 is calculated according to the equations presented in subsection 4.1.2.

Table 26 Necessary number of equipment to ensure the required annual productivity for different work schedules for the technological scheme with the use of bucket-wheel excavator (scheme one)

Work schedule	The necessary number of equipment to ensure the required annual productivity		
	Bucket-wheel excavator SRs 2000	Draglines ESh-14-50	Dump trucks CAT 777
2 shifts per 12 hours	1	4	26
3 shifts per 8 hours	1	4	29

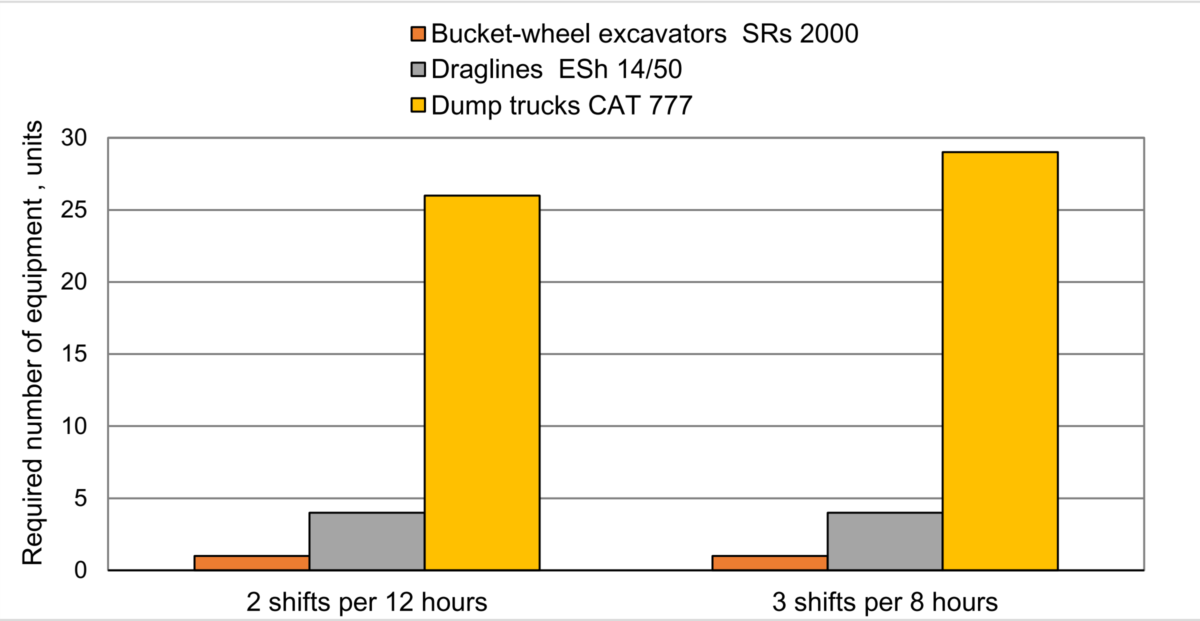


Figure 26 Required number of main mining equipment to ensure annual productivity for the technological scheme with the use of bucket-wheel excavator

With the work schedule 2 on shifts per 12 hours each, an average cycle time of 60 seconds and 310 working days per year, the annual output of ESh-14-50 is 2520 m^3/year (Attachment №7). Consequently, 4 excavators are needed to develop the required rock volume.

The required time for one trip of CAT 777 dump truck is 29 minutes, and it can make 20 trips per shift. Therefore, 26 trucks CAT 777 are needed.

With the work schedule on 3 shifts per 8 hours each, an average cycle time of 60 seconds and 310 working days per year, the annual output of ESh-14-50 is 2270 m^3/year (Attachment №7). Consequently, 4 excavators are needed to develop the required rock volume.

The required time for one trip of CAT 777 dump truck is 29 minutes, and it can make 12 trips per shift. Therefore, 29 trucks CAT 777 are needed.

The comparison of the required number of equipment for various work schedules for the technological scheme with the use of bucket-wheel excavator is shown in Figure 26. The calculation results are presented in Table 26.

4.2.2 Comparison of the Required Equipment Number for Technological Scheme with the use of Dragline Excavator

In the second variant of the technological scheme, the entire volume of overburden and ore mineral (16,200.0 thousand m^3) is developed by draglines ESh-14-50 in combination with trucks CAT 777.

With the work schedule on 2 shifts per 12 hours each, an average cycle time of 60 seconds and 310 working days per year, the annual output of ESh-14-50 is 2520 m^3/year (Attachment №7). Consequently, 7 excavators are needed to develop the required rock volume.

The required time for one trip of CAT 777 dump truck is 29 minutes, and it can make 20 trips per shift. Therefore, 49 trucks CAT 777 are needed.

With the work schedule on 3 shifts per 8 hours each, an average cycle time of 60 seconds and 310 working days per year, the annual output of ESh-14-50 is 2270 m^3/year (Attachment №7). Consequently, 8 excavators are needed to develop the required rock volume.

The required time for one trip of CAT 777 dump truck is 29 minutes, and it can make 12 trips per shift. Therefore, 54 trucks CAT 777 are needed.

The comparison of the required number of equipment for various work schedules for the technological scheme with the use of draglines is shown in Figure 27. The calculation results are presented in Table 27.

Table 27 Necessary number of equipment to ensure the required annual productivity for different work schedules for the technological scheme with the use of draglines

Work schedule	The necessary number of equipment to ensure the required annual productivity	
	Draglines ESh-14-50	Dump trucks CAT 777
2 shifts per 12 hours	7	49
3 shifts per 8 hours	8	54

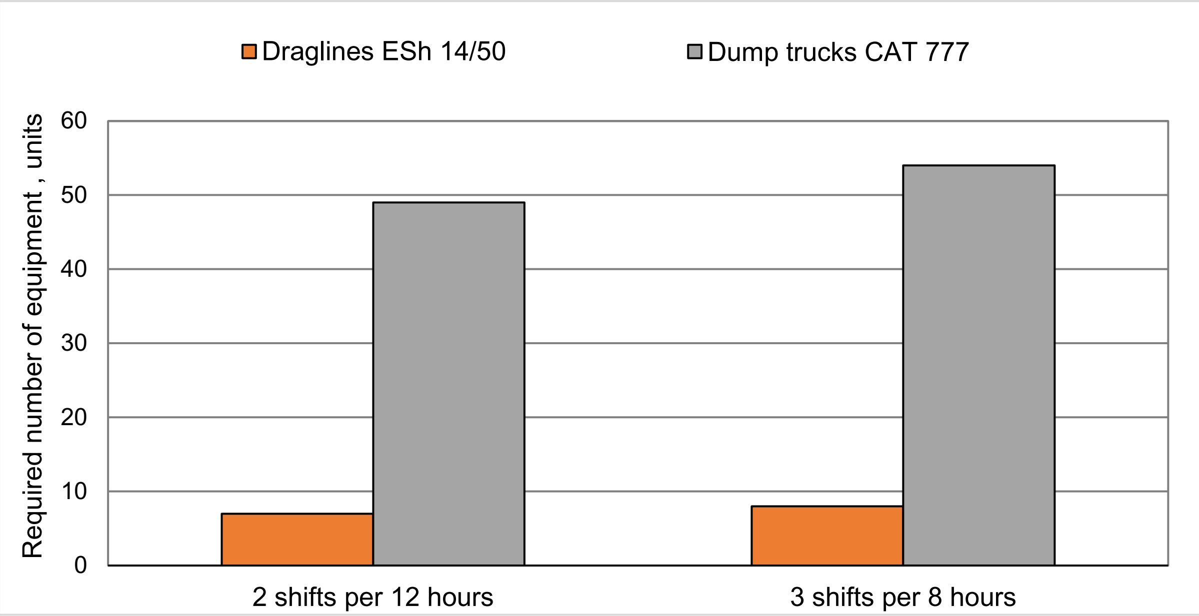


Figure 27 Required number of main mining equipment to ensure annual productivity for the technological scheme with the use of draglines

4.2.3 Comparison of the Required Equipment Number for Technological Scheme with the use of Hydraulic Excavators

In this chapter, in the third variant of mining equipment is considered the option without drainage trench construction. The upper horizons above the aquifer (7,600.0 thousand m^3) are developed by CAT 6018 FS hydraulic excavators, which work together with CAT 777 dump trucks. All the waterlogged rocks below this level are mined by draglines ESh-14-50 in combination with CAT 777 dump trucks (8,600.0 thousand m^3). The accuracy obtained allows drawing the necessary conclusions.

With the work schedule on 2 shifts per 12 hours each, an average cycle time of 40 seconds and 310 working days per year, the annual output of CAT 6018 FS is 2700

m^3/year (Attachment №7). Consequently, 3 excavators are needed to develop the required rock volume.

The required time for one trip of CAT 777 dump truck is 28.33 minutes, and it can make 21 trips per shift. Therefore, 25 trucks CAT 777 are needed.

With the work schedule on 3 shifts per 8 hours each, an average cycle time of 40 seconds and 310 working days per year, the annual output of CAT 6018 FS is 2430 m^3/year (Attachment №7). Consequently, 4 excavators are needed to develop the required rock volume.

The required time for one trip of CAT 777 dump truck is 28.33 minutes, and it can make 12 trips per shift. Therefore, 29 trucks CAT 777 are needed.

With the work schedule on 2 shifts per 12 hours each, an average cycle time of 60 seconds and 310 working days per year, the annual output of ESh-14-50 is 2520 m^3/year (Attachment №7). Consequently, 4 excavators are needed to develop the required rock volume.

The required time for one trip of CAT 777 dump truck is 29 minutes, and it can make 20 trips per shift. Therefore, 26 trucks CAT 777 are needed.

With the work schedule on 3 shifts per 8 hours each, an average cycle time of 60 seconds and 310 working days per year, the annual output of ESh-14-50 is 2270 m^3/year (Attachment №7). Consequently, 4 excavators are needed to develop the required rock volume.

The required time for one trip of CAT 777 dump truck is 29 minutes, and it can make 12 trips per shift. Therefore, 29 trucks CAT 777 are needed.

Therefore, the total required number of CAT 777 dump trucks in the technological scheme with the use of hydraulic excavators is 51 units for the work schedule on 2 shifts per 12 hours, and 58 units for the work schedule on 3 shifts per 8 hours.

The comparison of the required quantity of equipment for various work schedules for the technological scheme with the use of hydraulic excavators is shown in Figure 28. The calculation results are presented in Table 28.

Table 28 Necessary number of equipment to ensure the required annual productivity for different work schedules for the technological scheme with the use of hydraulic excavators

Work schedule	The necessary number of equipment to ensure the required annual productivity		
	Hydraulic excavators CAT 6018 FS	Draglines ESh-14-50	Dump trucks CAT 777
2 shifts per 12 hours	3	4	51
3 shifts per 8 hours	4	4	58

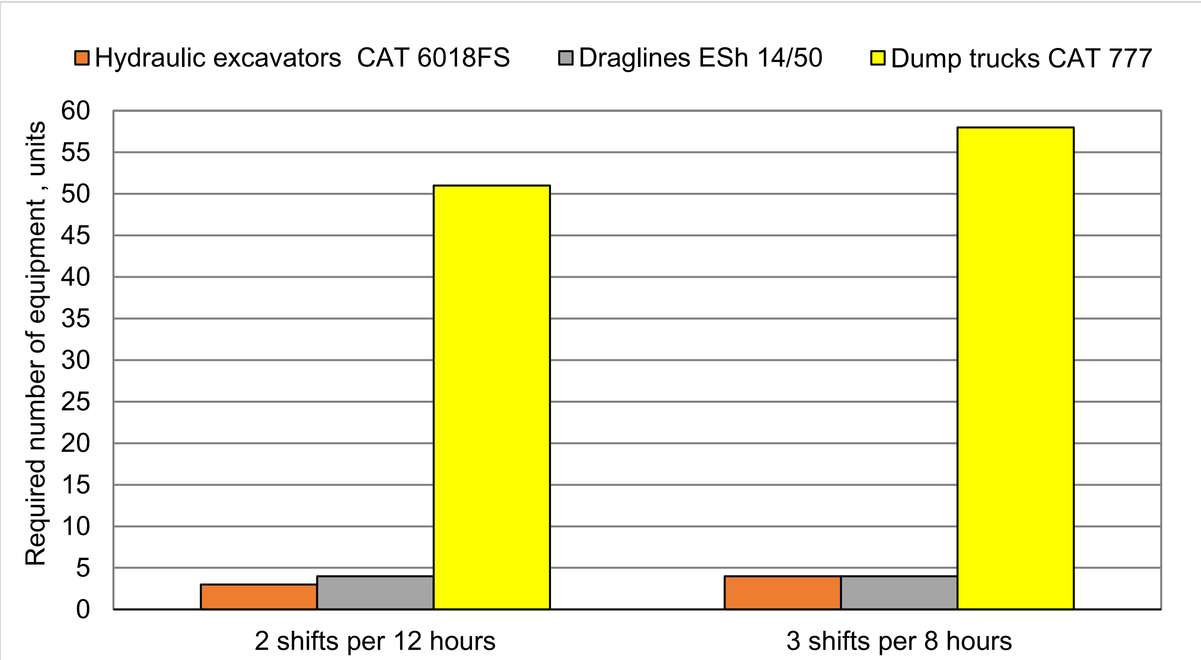


Figure 28 Required number of main mining equipment to ensure annual productivity for the technological scheme with the use of hydraulic excavators

4.2.4 Conclusions about Dependence of Required Equipment Number on Work Schedule

According to the calculations, the 2-shift of 12 hours work schedule requires the smaller number of the main mining equipment compering to the 3-shift of 8 hours each. In this mode, the maximum possible use of equipment in time is achieved. Therefore, there will be the lowest capital and operating expenditures will be when choosing this mode of operation. Many mining enterprises in Ukraine use this work schedule (Poltava GZK, Pokrovskiy GZK, Vilnohirsk GZK etc.)

The work schedule on 3 shifts per 8 hours each is more expensive, which can be explained by the additional loss of time on shift changes, which reduces the utilization of equipment, and therefore its performance. This, in turn, leads to the necessity of purchasing additional units of equipment to ensure the required annual productivity.

4.3 Dependence of Capital Expenditures on Annual Output of the Mine

In this subsection, the capital expenditures on the main mining equipment for every technological scheme are examined. The range of possible annual output is from 2.5 to 5 million m^3 of ore per year, therefore, taking into consideration the stripping ratio, from 12.5 to 25.0 million m^3 of overburden per year. All calculation results are presented in Attachment №7.

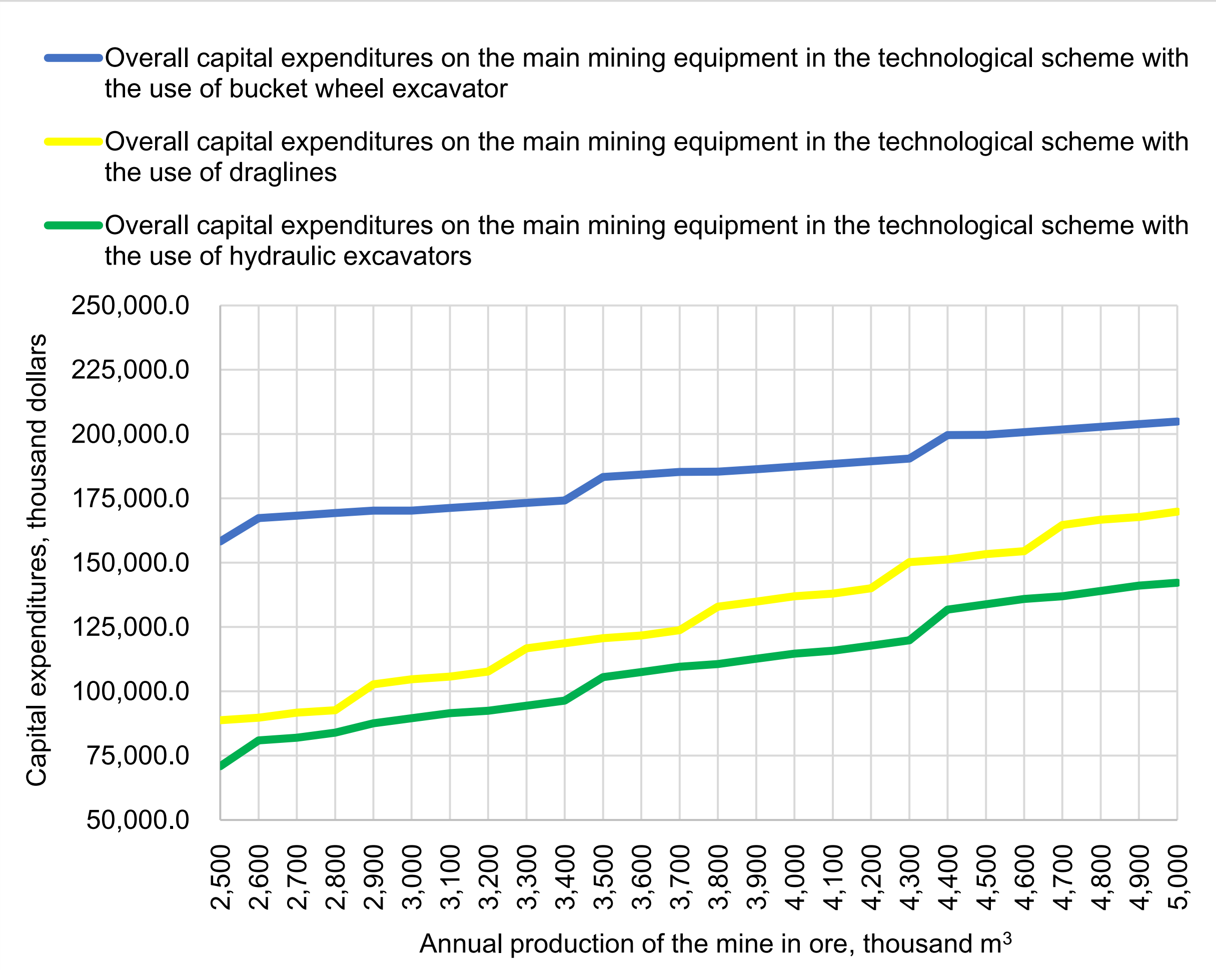


Figure 29 Dependence of the total capital expenditures on the annual output of the mine

The dependence of capital costs on annual ore output is presented in the graph in Figure 29. Also, in Figure 30 is presented a sum of unused capital expenditures, which shows how much the possible productivity of main mining equipment exceeds the required productivity for the annual overburden and mining plan fulfillment. Or, in other words, this indicator describes the effectiveness of invested funds usage.

According to the calculated data, the technological scheme with the use of hydraulic excavators is the most economical in terms of capital expenditures, regardless of the required annual output of the mine.

The efficiency of capital investment usage depends on the degree of application of this equipment to ensure the required annual ore productivity. But at the same time, some reserve for the productivity of the main mining equipment (accordingly, a small amount of unused capital investment) should be provided as an insurance against unforeseen circumstances (weather conditions, major emergency repairs of equipment, problems with the spare parts supply, etc.). This reserve will ensure the stable and uninterrupted operation of the processing plant and the enterprise as a whole.

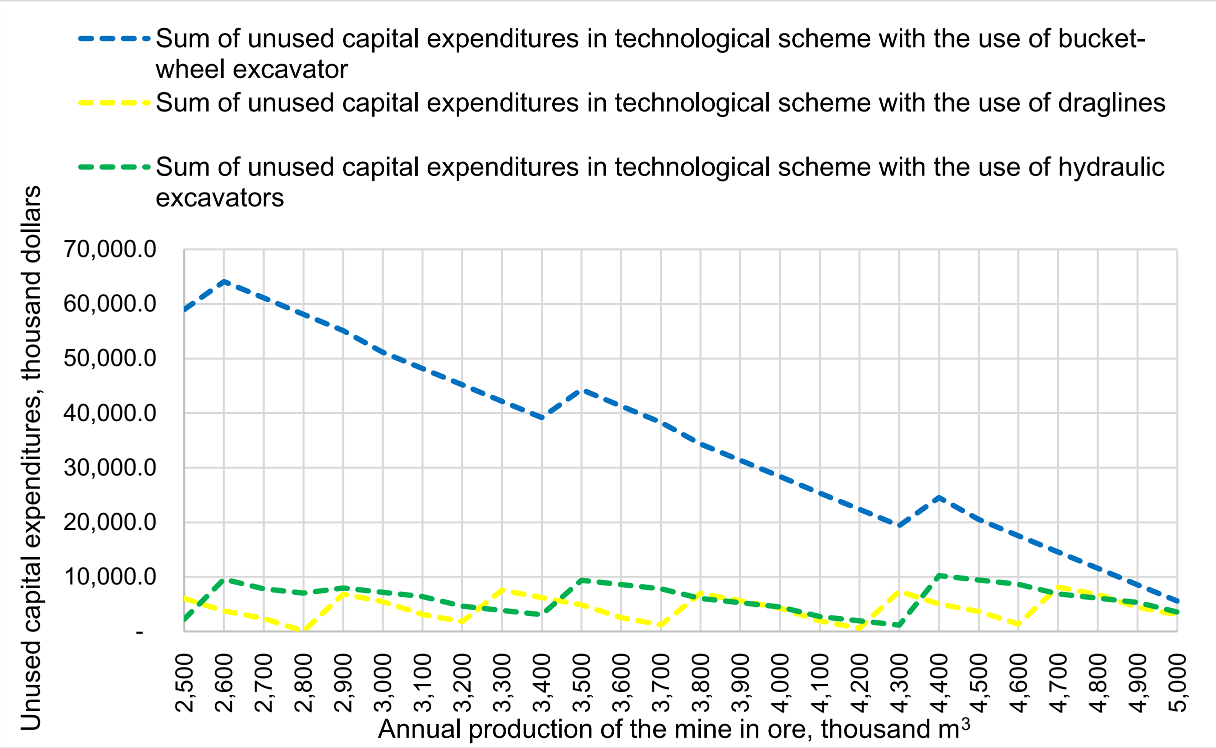


Figure 30 Dependence of the sum of unused capital expenditures on the annual output of the mine

4.4 Dependence of Equipment Usability Factor on Transportation Distance

The purpose of this section is to study the dependence of the excavator utilization factor on the rock transportation distance. The results were analyzed to obtain the optimal number of dump trucks to ensure maximum equipment performance.

4.4.1 Determination of the Most Optimal Transportation Distance

This subsection considers the utilization rate of an excavator to the utilization rate of dump trucks depending on transportation distance in a range from 1.5 to 3 km. The purpose of these calculations is to select the number of dump trucks at which the utilization rates of the excavator and dump trucks in this range will be as high as possible (Attachment №8). At the same time, in these theoretical calculations, the equipment availability factor is not taken into account.

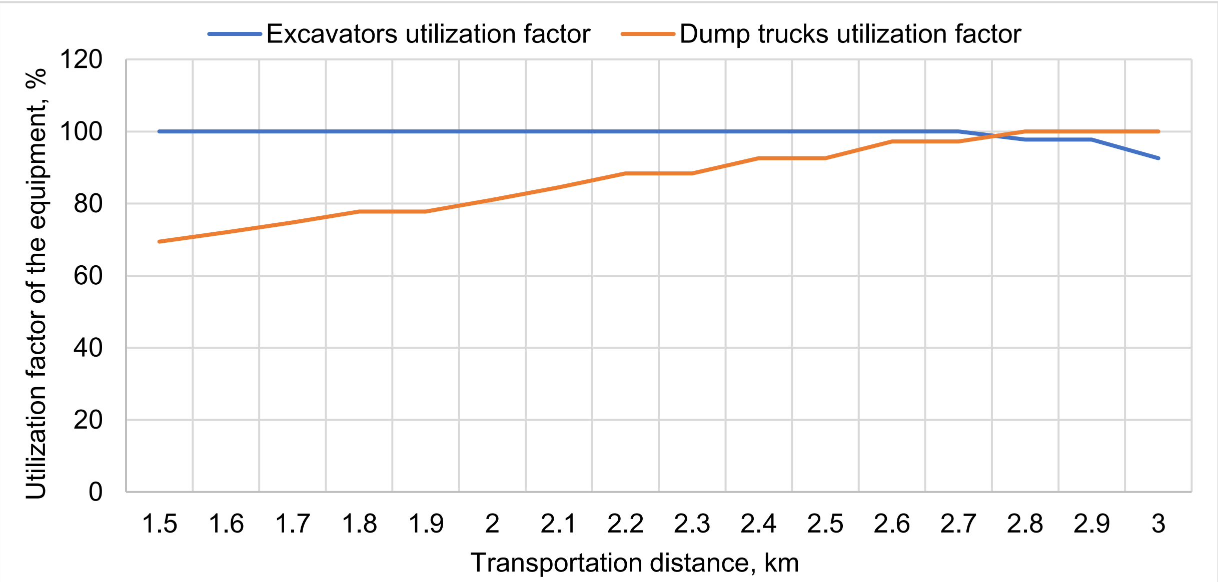


Figure 31 Dependence of equipment utilization factors on transportation distance when using 5 dump trucks (CAT 6018 FS / CAT 777)

According to the data obtained, when using an excavator CAT 6018 FS in combination with 5 trucks CAT 777, the optimal operation is observed on a transportation distance of 2.7 - 2.8 km. The utilization rates of the equipment at this distance are approximately equal and reaching 100%. (Figure 31). With the shorter transportation distance, dump trucks are not fully employed. At the greater distance, the excavator is idle. In the considered range of transportation distances (1.5 to 3 km), the minimum utilization rate of the dump trucks does not fall below 69%, and for the excavator below 93%.

When using 4 dump trucks, the optimal equipment performance is observed at a distance of 1.9 - 2.0 km (Figure 32). In transportation distances from 1.5 to 3 km, the minimum excavator utilization factor is 74% and the minimum dump truck utilization factor is 74%. All calculations and data are presented in Attachment №8.

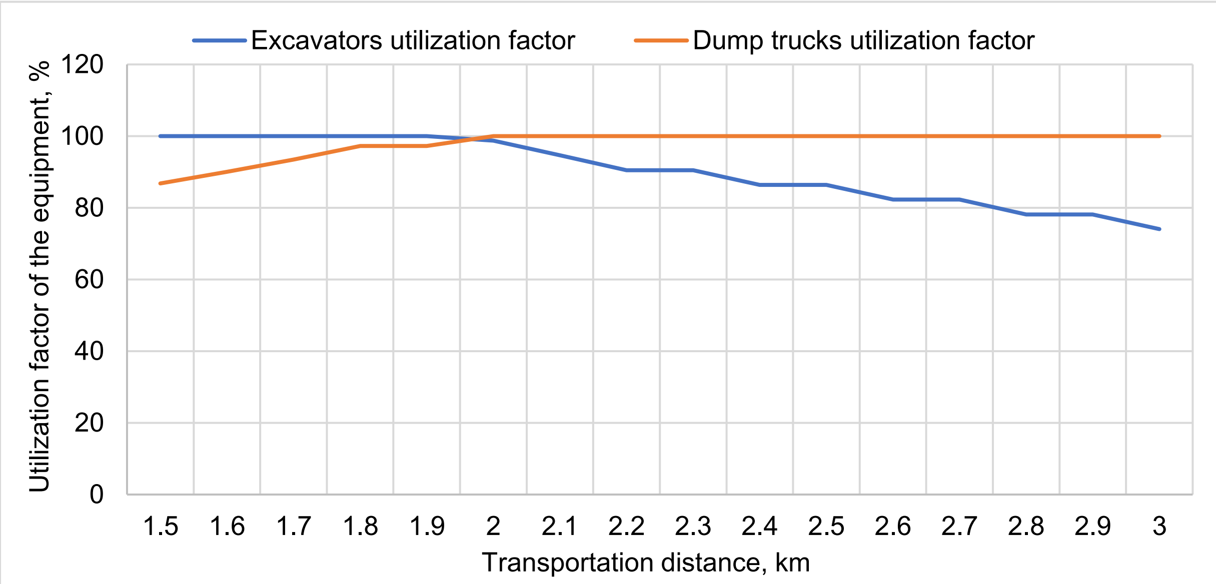


Figure 32 Dependence of equipment utilization factors on transportation distance when using 4 dump trucks (CAT 6018 FS / CAT 777)

On the benches where hydraulic excavators are used, the number of dump trucks can be adjusted depending on the transportation distance. For example, at the distance of up to 2.4km, 4 dump trucks should be used, and at a distance of more than 2.4 km, the usage of 5 dump trucks is more advisable.

This approach is actively used in the mines of JSC "Pokrovskiy GZK". The company has one common vehicle fleet for all mines, which ensures certain maneuverability in relation to the required number of dump trucks. Every mine, depending on current conditions, orders the necessary number of dump trucks per shift.

According to the data obtained, when using an ESh-14-50 excavator in combination with 3 trucks CAT 777, the optimal operation is observed at the rock mass transportation distance of 2 km, where the utilization rates of all equipment reach 100%. (Figure 33). As in the case of the hydraulic excavator, with the shorter transportation distance, dump trucks are not fully utilized; and with the greater distance, the excavator is idle. In the considered range of transportation distances (1.5 to 3 km), the minimum utilization rate of dump trucks does not fall below 87%, and the minimum excavator's utilization factor is 77% (Attachment №8).

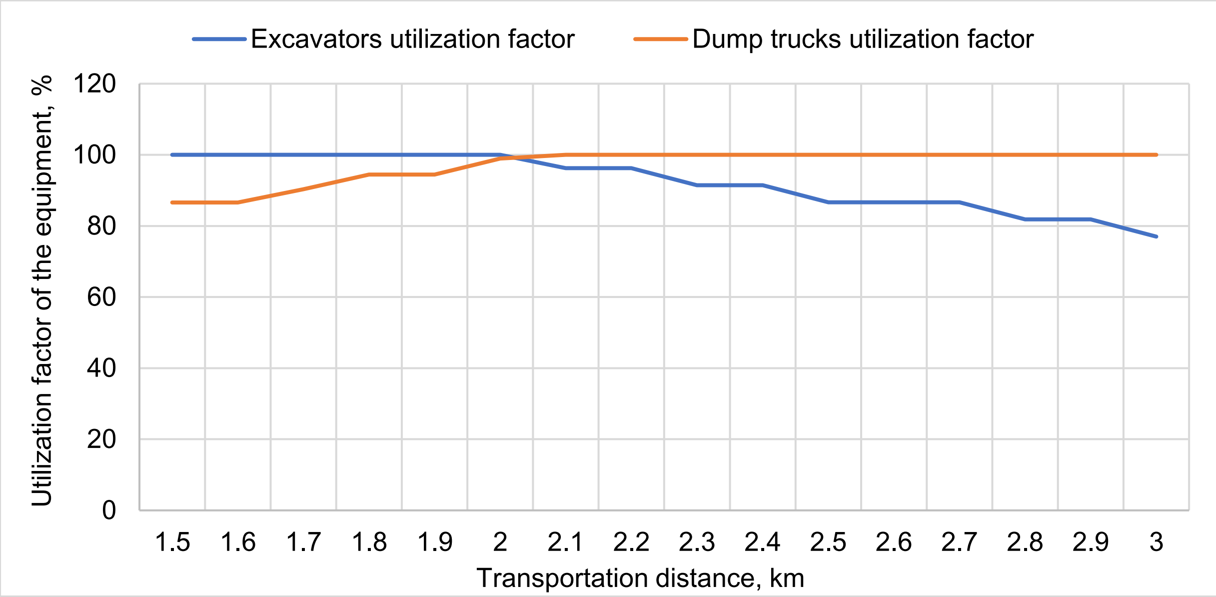


Figure 33 Dependence of equipment utilization factors on transportation distance when using 3 dump trucks (ESh-14-50 / CAT 777)

When calculating the excavator ESh-14-50 in combination with 2 dump trucks, over the entire range of distances, the dump trucks are used at 100%, while the excavator's utilization rate is from 51% to 77% (Attachment №8). Due to the fact that the optimal practical utilization factor of the equipment should be 70-90%, this option does not meet these requirements on most of the distances. When calculating the utilization rate of the excavator in combination with 4 dump trucks, the excavator is used at 100%, while the utilization factor of dump trucks is in the range from 65% to 97% (Attachment №8), so this option may be used on the larger distances.

4.4.2 Selection of Optimal Dump Trucks Number to ensure Maximum Equipment Utilization

In this subsection, the utilization factor of the excavator CAT 6018 FS and the dump trucks CAT 777 are calculated depending on the transportation distance, while the number of dump trucks is changed to ensure the most optimal operation of the equipment. For illustrative purposes, a range of 1 to 5 km of transportation distance is considered. In these theoretical calculations, the equipment availability factor is not taken into account. All calculated data are presented in Attachment №8.

Figure 34 shows how the utilization rate of the dump trucks changes when the transportation distance changes under the condition of maximum use of the excavator (100%). From the results obtained, it can be concluded that while ensuring the maximum possible productivity of the excavator, dump trucks are idle. The lowest utilization rate of the dump trucks is observed on the distances where additional trucks are just added. Then the utilization rate gradually increases with the distance of transportation increase. The utilization rate of dump trucks ranges from 76% to 98%. Also, the graph shows a general trend that the utilization rate increase and the amplitude of the oscillations decrease with the distance increase.

The second graph in Figure 35 shows how the utilization rate of the excavator changes when the transportation distance changes under the condition of maximum use of the dump trucks. Excavator's utilization rate changes from 68% to 90.5% here. Unlike in the first graph (Figure 34), the utilization factor of the excavator is maximal on the distances where additional trucks are just added and gradually decreases with the increase of the distance. But the general trend that the utilization rate increase and the amplitude of the oscillations decrease with the distance increase stays the same.

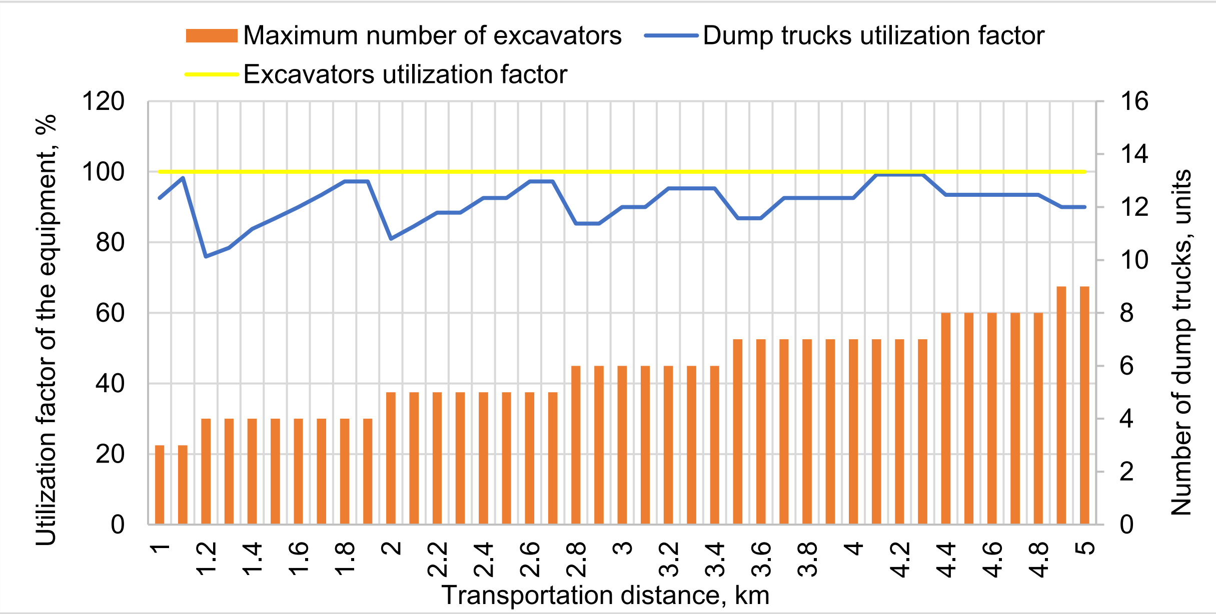


Figure 34 Dependence of utilization rate of the dump trucks on transportation distance with the maximum use of the excavator (CAT 6018 FS / CAT 777)

In both cases, the utilization rates of the equipment are mostly higher than the minimum allowable level of 70%. Therefore, both options are viable. The first option can be used to provide the maximum possible productivity, and the second is more economical in terms of capital and operating costs.

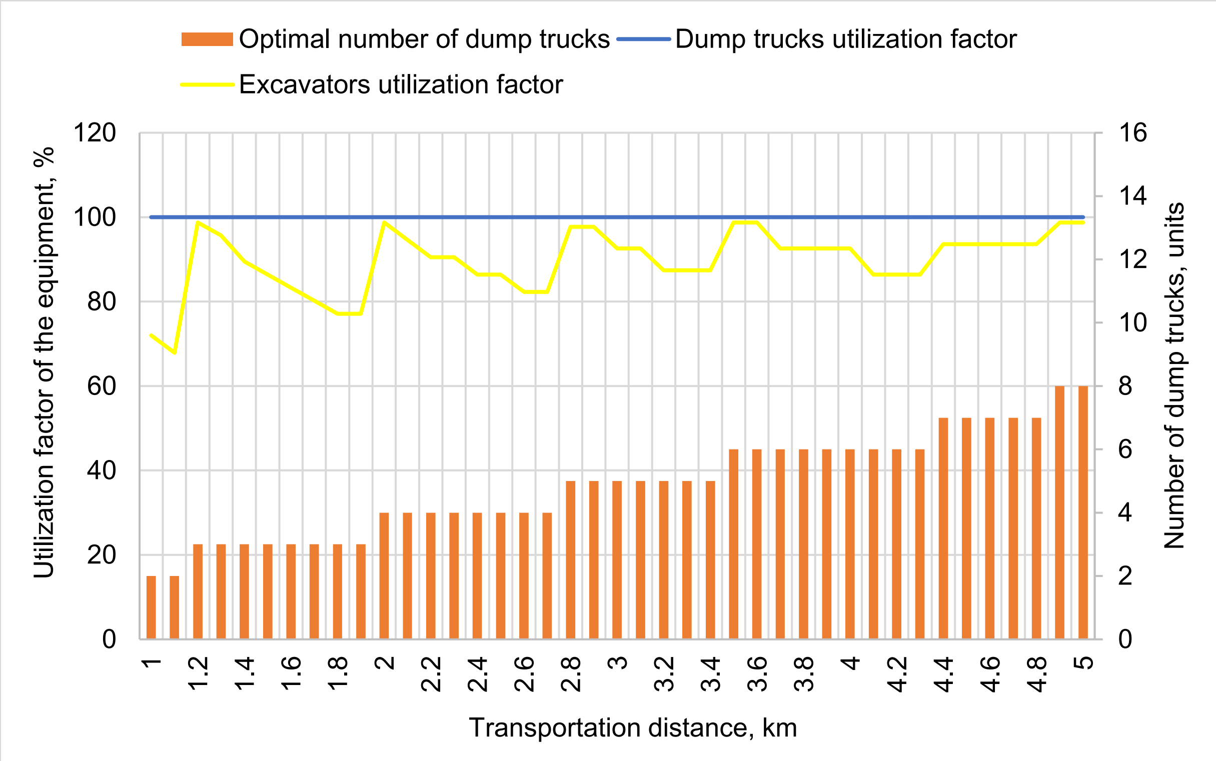


Figure 35 Dependence of utilization rate of the excavator on the transportation distance with the maximum use of the dump trucks (CAT 6018 FS / CAT 777)

Conclusion

In this thesis, three technological schemes of development of the Motronivka-Annivka section of Malyshev titanium-zirconium ore deposit were analyzed, including a technological scheme with the use of bucket-wheel excavator, with the use of draglines and with the use of hydraulic excavators. All options are viable.

The comparison of the technological schemes was made by their capital and operational expenditures. For the estimation of the capital expenditures were used real prices on the equipment or prices achieved by proportional comparisons with similar equipment. For the estimation of the operational expenditures was used O'Hara and Suboleski methodology [3] because of the difficulty of all necessary data collection for using more precise methods.

In order to facilitate the work, all calculations were made using the spreadsheet program Excel. This calculation system may be used for the estimation of other projects. Also, engineering drawings of whole mine work organization for all three technological schemes and the most complicated elements of these schemes were carried out in the AutoCAD program.

The selection of equipment is carried out ensuring its maximum possible interchangeability, which can provide additional work stability and should reduce the maintenance and repair costs.

According to the economic evaluation of the Motronivka-Annivka section development, the most cost-effective is the technological scheme with the use of hydraulic excavators on the overburden benches. The ore extraction bench is to be developed by dragline excavators. As the draglines can use downward digging to take the whole bench, using the given equipment guarantees a dry surface of the working floor, which is critical in given geological conditions.

An excavator ESh-14-50 is the most optimal choice comparing to other NKMZ draglines (the most widespread and accessible draglines producer in Ukraine). The relatively short boom makes the work of the operator during truck loading as comfortable and safe as possible. The bucket volume of $14 m^3$ provides high productivity. In addition, the decrease of the boom weigh proportionally decreases the weight of the counterweight, which, in turn, decreases the weight of the whole excavator. This reduces the specific weight, which is crucial considering the soft underlying rocks, and also has an impact on the price.

Economic efficiency of hydraulic excavators on the overburden benches is conditioned by their relatively low cost comparing to draglines. The bucket volume of CAT 6018 FS is lower than that of the draglines ESh-14-50, but this is compensated by a lower cycle time on the rock mass loading. Also, an important role plays the mobility of hydraulic excavators and their independence from an electricity network.

The negative side of this scheme is the pollution of the local environment by exhaust gases. Therefore, all equipment was only selected if their engines satisfy Euro V emission standard requirements.

Also, in this work, technical and economic dependencies were studied and analyzed when changing the parameters of mining equipment. Research results are presented in the 4th chapter. These studies have helped to justify the choice of equipment and the working schedule of the mine. Furthermore, these calculations have allowed to study the modes of mutual work of equipment to achieve to ensure its maximum performance.

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List of Figures

List of Tables

Attachment №1

Work Organization in Technological Scheme with the use of Bucket-Wheel Excavator

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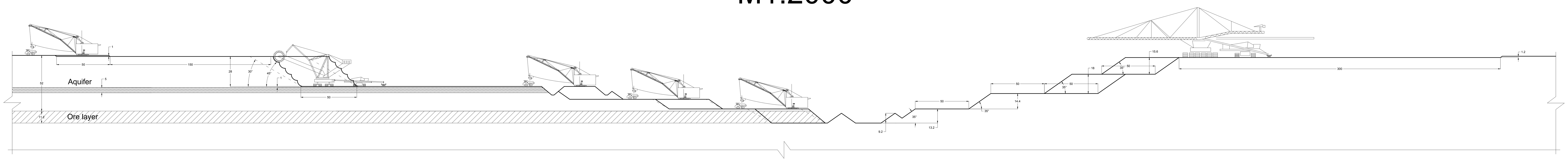

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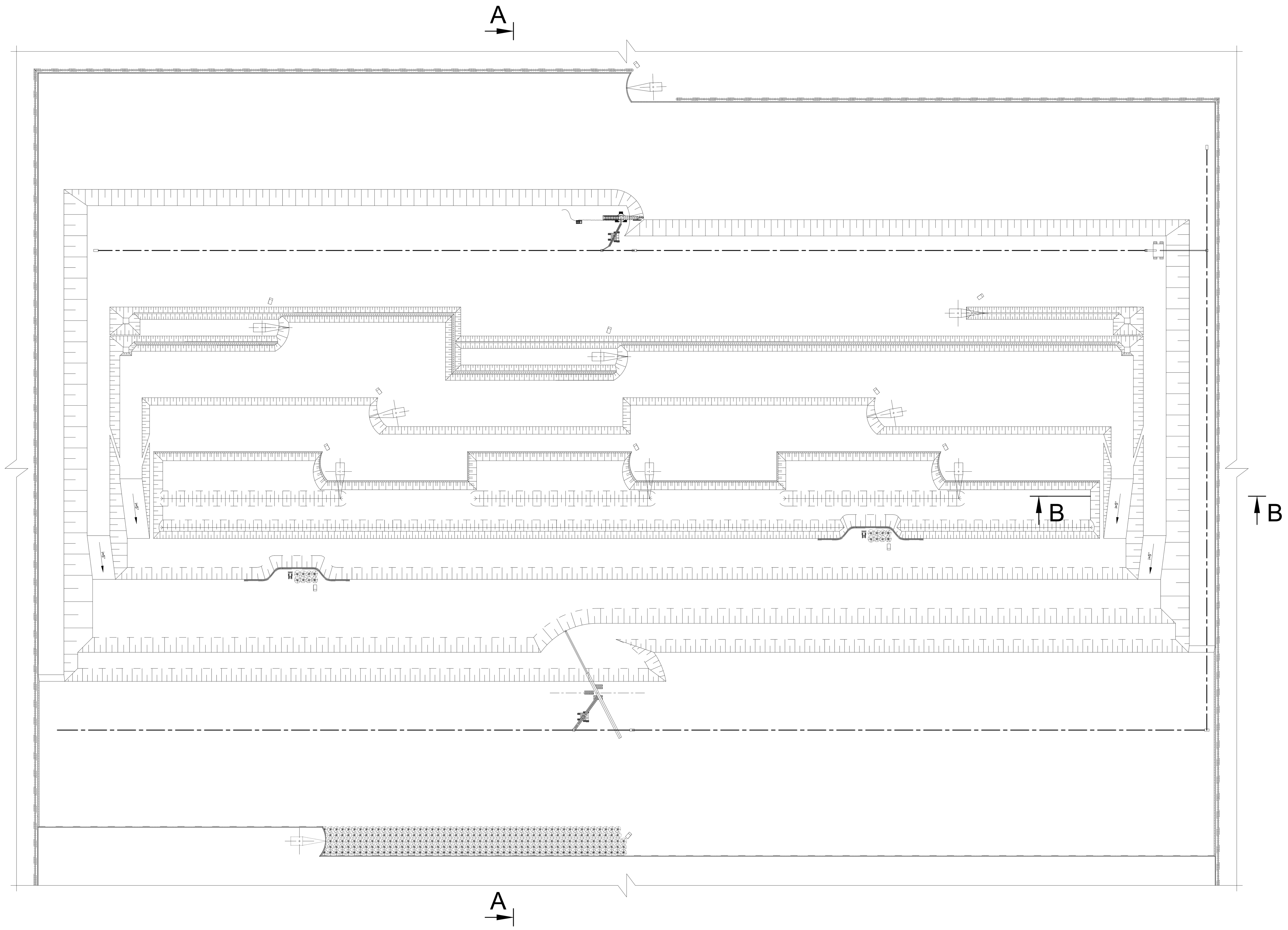

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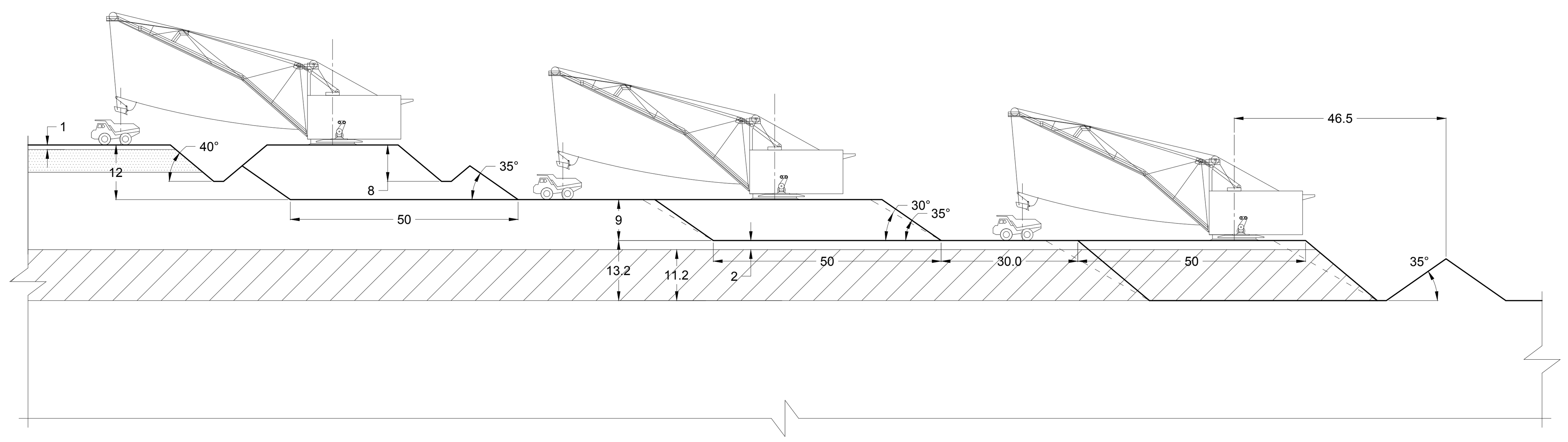

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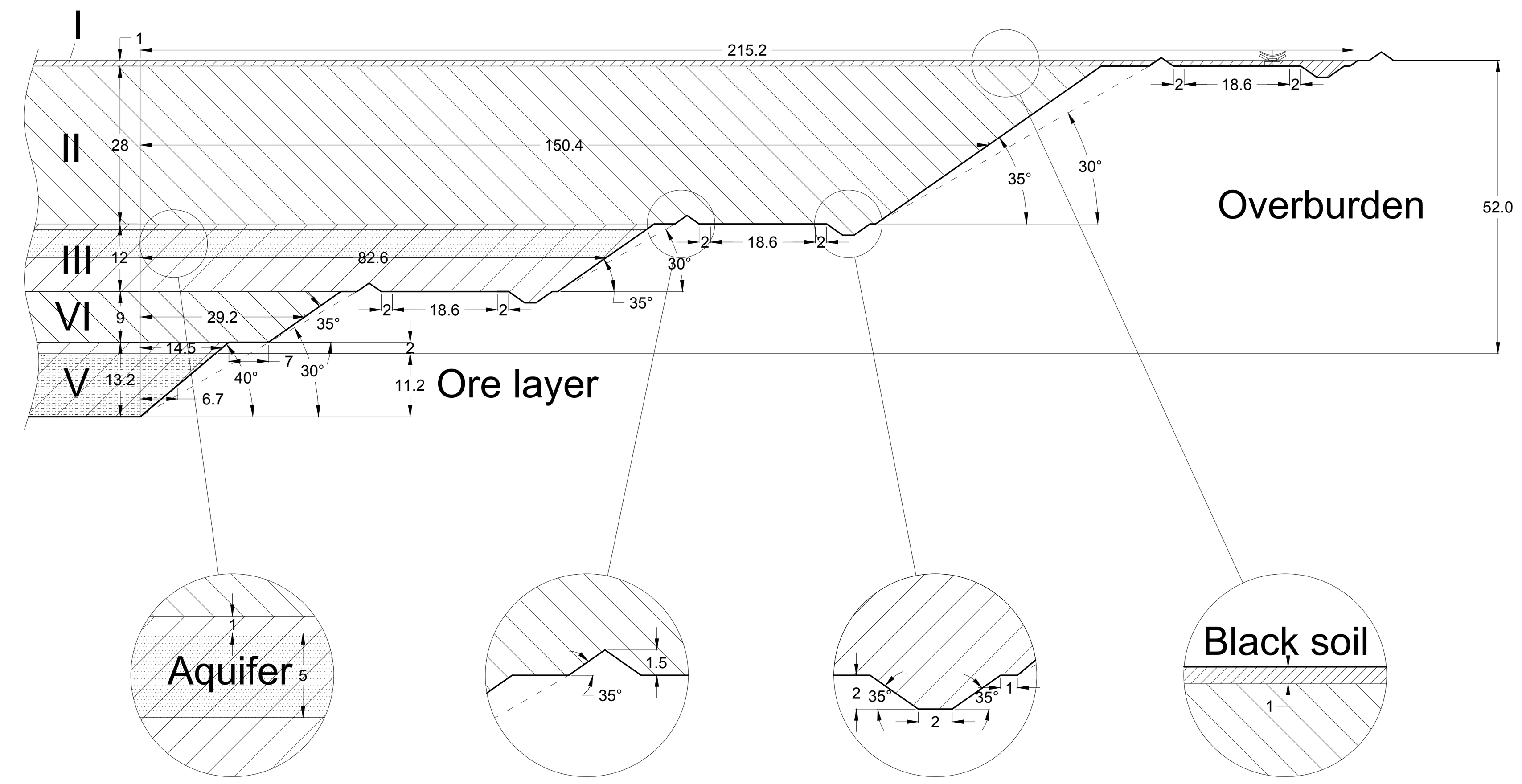
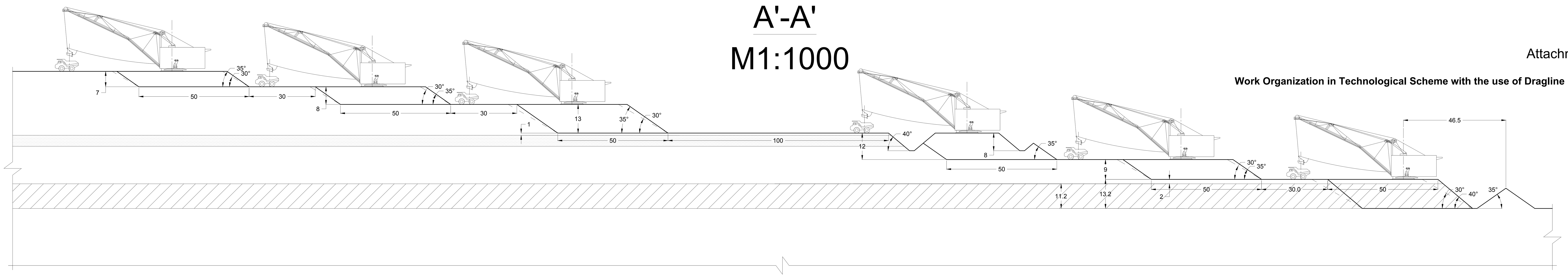
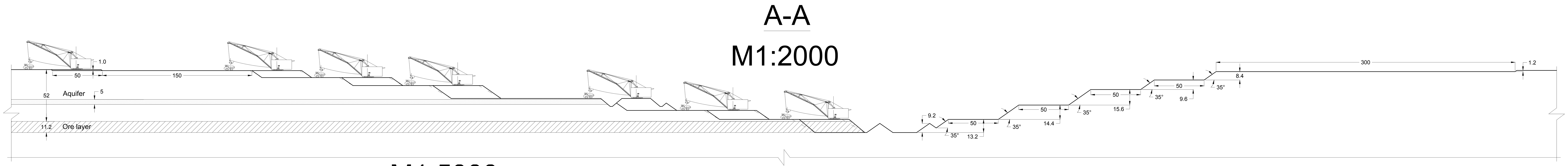


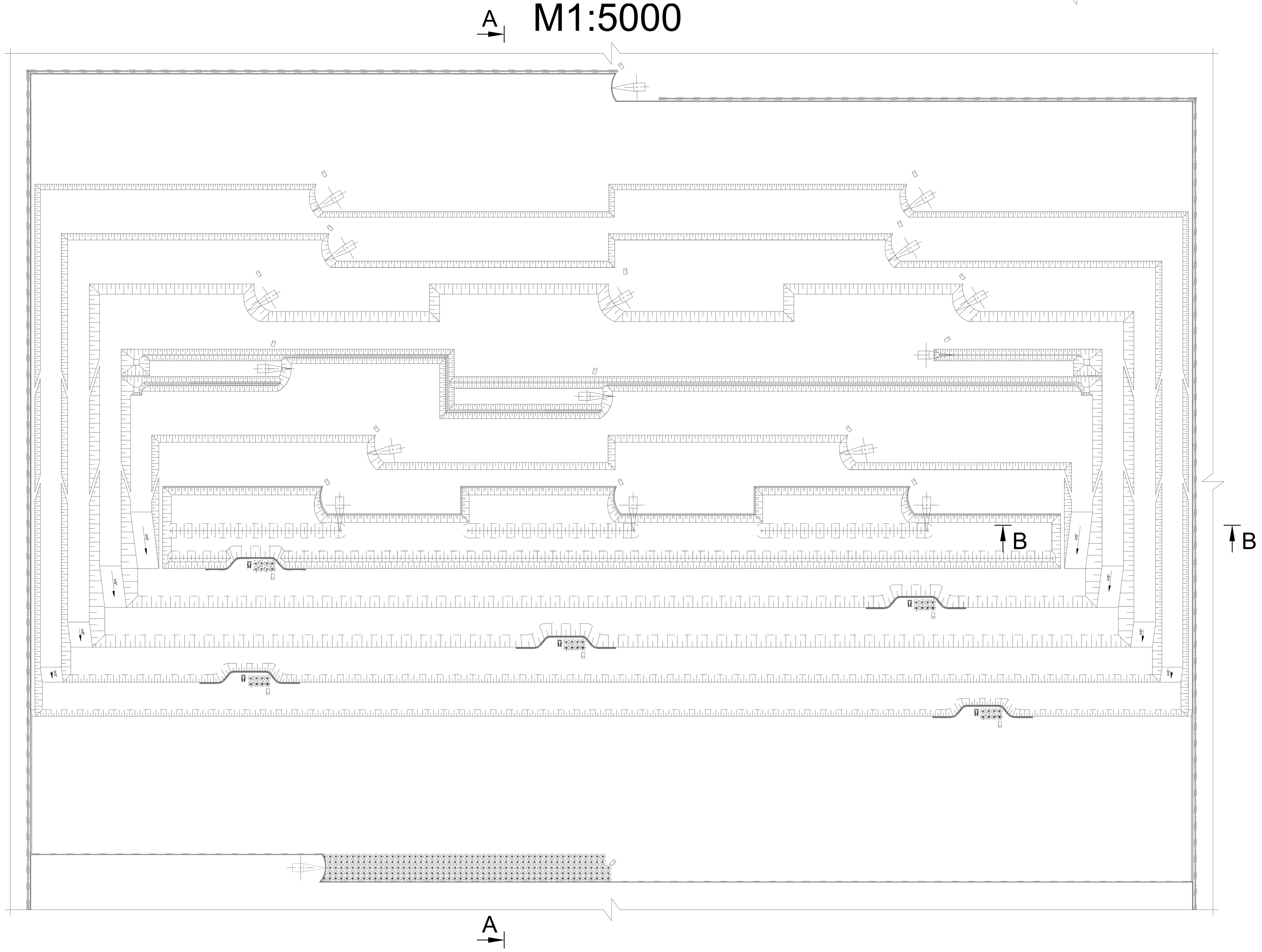
Master thesis: Substantiation of the Efficient Technological Scheme of Development of Motronivka Placer (Malyshev Titanium-Zirconium Ore Deposit)

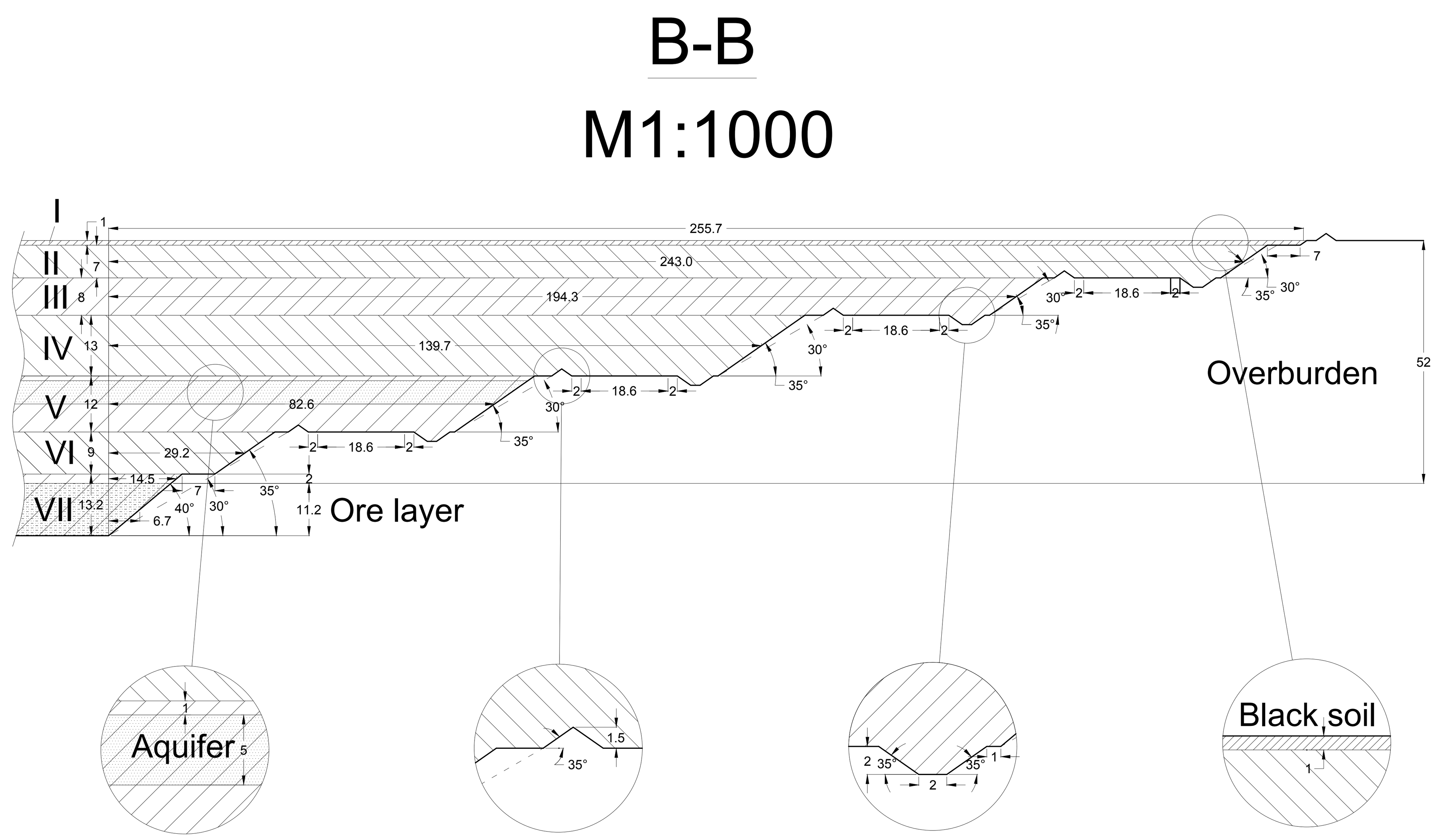
Attachment №2

Work Organization in Technological Scheme with the use of Dragline Excavators









Master thesis: Substantiation of the Efficient Technological Scheme of Development of Motronivka Placer (Malyshev Titanium-Zirconium Ore Deposit)

Attachment №3

Work Organization in Technological Scheme with the use of Hydraulic Excavators

Master thesis: Substantiation of the Efficient Technological Scheme of Development of Motronivka Placer (Malyshev Titanium-Zirconium Ore Deposit)

Attachment №4

Required Equipment Number Calculation in Technological Scheme with the use of Bucket-Wheel Excavator

Input data	Symbols	Units	1st bench ESh-14-50 CAT 777	2nd bench SRs 2000 Conveyers A2Rs	3rd bench: Rock excavation ESh-14-50 CAT 777	3rd bench: Drainage trench construction ESh-14-50 CAT 777	4th bench ESh-14-50 CAT 777	5th bench: Overburden ESh-14-50 Dumping in the mined-out space	5th bench: Ore ESh-14-50 CAT 777
Annual prodaction on a bench									
Width of the extraction front	A e.f.	m	2030	1811	1765	1765	1658	1629	1613
Annual advance of the mining front	P year	m	150	150	150	35	150	150	150
Height of the bench	H	m	1	28	12	8	9	2	11.22
Volume of rock that should be mined on a bench (annual capacity)	Q year	m3	304500	7606200	3177000	494200	2238300	488700	2714679
* Annual production without trench volumes	-	m3	-	-	2682800	-	-	-	-
Number of working hours of excavator per year									
Number of working months in year	N m	mounth	6	12	12	12	12	12	12
Average number of working days in month	N d	day	30	30	30	30	30	30	30
Number of holydays per year	N h	day	4	8	8	8	8	8	8
Number of standstill days caused by whether conditions per year	N s	day	10	20	20	20	20	20	20
Number of shifts	n	shift	2	2	2	2	2	2	2
Duration of the shift	T sh	hour	12	12	12	12	12	12	12
Time, which is necessary per shift turnover, dinnertime and preventive maintenance of the equipment	T sh.t.	hour	2	2	2	2	2	2	2
Working hours per year	N h.y.	hour	3320	6640	6640	6640	6640	6640	6640

Production rate per hour

Input data	Symbols	Units	1st bench ESh-14-50 CAT 777	3rd bench: Rock excavation ESh-14-50 CAT 777	3rd bench: Drainage trench construction ESh-14-50 CAT 777	4th bench ESh-14-50 CAT 777	5th bench: Overburden ESh-14-50 Dumping in the mined-out space	5th bench: Ore ESh-14-50 CAT 777
Seconds in hour	-	sec	3600	3600	3600	3600	3600	3600
Loss of time per preparation of the bench face per hour	T p.b.f.	sec	180	180	180	180	180	180
Loss of time for change of excavator position (movement)	T m	sec	0	0	0	0	0	0
Average cycle time for one bucket shoveling	T c	sec	140	140	140	140	140	140
Bucket capacity	V b	m3	14.0	14.0	14.0	14.0	14.0	14.0
Coefficient of volume efficiency of the bucket	K v.e.	-	1.0	1.0	1.0	1.0	1.0	1.0
Fragmentation index of the rock in the bucket	K f	-	1.2	1.2	1.2	1.2	1.2	1.2
Coefficient of readiness of excavator	K r	-	0.7	0.7	0.7	0.7	0.7	0.7
Production rate per hour	Q h	m3/hour	200	200	200	200	200	200

Input data	Symbols	Units	2nd bench SRs 2000 Conveyers A2Rs
Bucket volume	E	m3	1.7
Average number of unloadings per minute (according to manufacturing documentation)	n u	times	26
Coefficient of productivity	K p	-	0.7
Coefficient of usage	K u	-	0.85
Fragmentation index for bucket wheel excavator	K f	-	1.2
Production rate per hour of the bucket wheel excavator	Q h.b.v.	m3/hour	1315

Input data	Symbols	Units	1st bench ESh-14-50 CAT 777	2nd bench SRs 2000 Conveyers A2Rs	3rd bench: Rock excavation ESh-14-50 CAT 777	3rd bench: Drainage trench construction ESh-14-50 CAT 777	4th bench ESh-14-50 CAT 777	5th bench: Overburden ESh-14-50 Dumping in the mined-out space	5th bench: Ore ESh-14-50 CAT 777
Calculation of the necessary number of excavators									
Annual production on a banch	Q year	m3	304500	7606200	2682800	494200	2238300	488700	2714679
Production per hour of the excavator	Q h	m3/hour	200	1315	200	200	200	200	200
Working hours of excavator per year	N h.y.	hour	3320	6640	6640	6640	6640	6640	6640
Necessary number of excavators	N ex.	unit	0.46	0.87	2.03	0.37	1.69	0.37	2.05
*Actual number of excavators	-	unit	1	1	3		2	3	

Input data	Symbols	Units	1st bench ESh-14-50 CAT 777	3rd bench: Rock excavation ESh-14-50 CAT 777	3rd bench: Drainage trench construction ESh-14-50 CAT 777	4th bench ESh-14-50 CAT 777	5th bench Ore ESh-14-50 CAT 777
Rock volume which should be transported during one shift							
Annual production on a banch	Q year	m3	304500	2682800	494200	2238300	2714679
Coefficient of rock fragmentation	K f	-	1.2	1.2	1.2	1.2	1.2
Duration of the shift	T sh	hour	12	12	12	12	12
Time, which is necessary per shift turnover, dinnertime and preventive maintenance of the truck	T sh.t.	hour	2	2	2	2	2
Working hours per year	N h.y.	hour	3320	6640	6640	6640	6640
Rock volume should be transported per shift	Q sh.	m3/shift	1101	4848	893	4045	4906
Time required per one trip of the truck							
Number of minutes in hour	-	min	60	60	60	60	60
Number of times to make a way to the hauling point	-	times	2	2	2	2	2
Hauling distance	L h.d.	km	3.5	2.4	2.4	2	3
Average speed of the truck	V t	km/hour	15	15	15	15	15
Average cycle time for one bucket shoveling	T c	sec	140	140	140	140	140
Necessary number of buckets to fill the truck	n b	bucket	4	4	4	4	4
Unloading time	T un	min	5	5	5	5	5
Time required per one trip of the truck	T t	min	42	34	34	30	38
Calculation of the possible number of trips per shift							
Number of minutes in hour	-	min	60	60	60	60	60
Duration of the shift	T sh	hour	12	12	12	12	12
Time, which is necessary per shift turnover, dinnertime and preventive maintenance of the truck	T sh.t.	hour	2	2	2	2	2
Time required per one trip of the truck	T t	min	42	34	34	30	38
Number of trips per shift	N trips	times	14.17	17.89	17.89	19.78	15.65
*Actual number of trips per shift	-	times	14	17	17	19	15
Calculation of the necessary number of trucks							
Volume of rock that should be transported per shift	Q sh	m3/shift	1101	4848	893	4045	4906
Number of trips per shift	N trips	times	14	17	17	19	15
Bucket capacity	V b	m3	14.0	14.0	14.0	14.0	14.0
Coefficient of volume efficiency of the bucket	K v.e.	-	1.0	1.0	1.0	1.0	1.0
Coefficient of readiness of the truck	Kr	-	0.75	0.75	0.75	0.75	0.75
Necessary number of buckets to fill the truck	n b	bucket	4	4	4	4	4
Number of trucks	N truck	unit	1.87	6.79	1.25	5.07	7.79
*Actual number of trucks	-	unit	2	7	2	6	8

General number of equipment, units	
SRs 2000	1
ESh-14-50	9
CAT 777	25

Attachment №6

Required Equipment Number Calculation in Technological Scheme with the use of Hydraulic Excavators

Input data	Symbols	Units	1st bench CAT 375L CAT 725	2nd bench CAT 6018FS CAT 777	3rd bench CAT 6018FS CAT 777	4th bench CAT 6018FS CAT 777	5th bench		6th bench CAT 6018FS CAT 777	7th bench	
							Rock excavation CAT 6018FS CAT 777	Drainage trench construction CAT 375L CAT 725		Overburden ESh-14-50 Dumping in the mined-out space	Ore ESh-14-50 CAT 777
Annual production on a bench											
Width of the extraction front	A e.f.	m	2181	2153	2051	1946	1836	1836	1730	1629	1613
Annual advance of the mining front	P year	m	150	150	150	150	150	35	150	150	150
Height of the bench	H	m	1	9	9	10	12	8	9	2	11.22
Volume of rock that should be mined on a bench (annual capacity)	Q year	m3	327150	2906550	2768850	2919000	3304800	514080	2335500	488700	2714679
* Annual production without trench volumes	-	m3	-	-	-	-	2790720	-	-	-	-
Number of working hours of excavator per year											
Number of working months in year	N m	mounth	6	12	12	12	12	12	12	12	12
Average number of working days in month	N d	day	30	30	30	30	30	30	30	30	30
Number of holydays per year	N h	day	4	8	8	8	8	8	8	8	8
Number of standstill days caused by whether conditions per year	N s	day	10	20	20	20	20	20	20	20	20
Number of shifts	n	shift	2	2	2	2	2	2	2	2	2
Duration of the shift	T sh	hour	12	12	12	12	12	12	12	12	12
Time, which is necessary per shift turnover, dinnertime and preventive maintenance of the equipment	T sh.t.	hour	2	2	2	2	2	2	2	2	2
Working hours per year	N h.y.	hour	3320	6640	6640	6640	6640	6640	6640	6640	6640
Production rate per hour											
Seconds in hour	-	sec	3600	3600	3600	3600	3600	3600	3600	3600	3600
Loss of time per preparation of the bench face per hour	T p.b.f.	sec	300	300	300	300	300	300	300	180	180
Loss of time for change of excavator position (movement)	T m	sec	300	300	300	300	300	300	300	0	0
Average cycle time for one bucket shoveling	T c	sec	35	45	45	45	45	45	45	100	140
Bucket capacity	V b	m3	2.4	10.0	10.0	10.0	10.0	2.4	10.0	14.0	14.0
Coefficient of volume efficiency of the bucket	K v.e.	-	1.0	1.0	1.0	1.0	1.0	1.0	1.0	1.0	1.0
Fragmentation index of the rock in the bucket	K f	-	1.2	1.2	1.2	1.2	1.2	1.2	1.2	1.2	1.2
Coefficient of readiness of excavator	K r	-	0.85	0.8	0.8	0.8	0.8	0.85	0.8	0.7	0.7
Production rate per hour	Q h	m3/hour	146	444	444	444	444	113	444	279	200
Calculation of the necessary number of excavators											
Annual production on a banch	Q year	m3	327150	2906550	2768850	2919000	2790720	514080	2335500	488700	2714679
Production per hour of the excavator	Q h	m3/hour	146	444	444	444	444	113	444	279	200
Working hours of excavator per year	N h.y.	hour	3320	6640	6640	6640	6640	6640	6640	6640	6640
Necessary number of excavators	N ex.	unit	0.68	0.98	0.94	0.99	0.95	0.68	0.79	0.26	2.05
*Actual number of excavators	-	unit	1	1	1	1	1	1	1	3	
Rock volume which should be transported during one shift											
Annual production on a banch	Q year	m3	327150	2906550	2768850	2919000	2790720	514080	2335500	2714679	
Coefficient of rock fragmentation	K f	-	1.2	1.2	1.2	1.2	1.2	1.2	1.2	1.2	
Duration of the shift	T sh	hour	12	12	12	12	12	12	12	12	
Time, which is necessary per shift turnover, dinnertime and preventive maintenance of the truck	T sh.t.	hour	2	2	2	2	2	2	2	2	
Working hours per year	N h.y.	hour	3320	6640	6640	6640	6640	6640	6640	6640	
Rock volume should be transported per shift	Q sh.	m3/shift	1182	5253	5004	5275	5043	929	4221	4906	
Time required per one trip of the truck											
Number of minutes in hour	-	min	60	60	60	60	60	60	60	60	
Number of times to make a way to the hauling point	-	times	2	2	2	2	2	2	2	2	
Hauling distance	L h.d.	km	3.6	3	2.9	2.7	2.4	2.2	1.9	3	
Average speed of the truck	V t	km/hour	15	15	15	15	15	15	15	15	
Average cycle time for one bucket shoveling	T c	sec	35	45	45	45	45	45	45	140	
Necessary number of buckets to fill the truck	n b	bucket	5	5	5	5	5	5	5	4	
Unloading time	T un	min	5	5	5	5	5	5	5	5	
Time required per one trip of the truck	T t	min	37	33	32	30	28	26	24	38	
Calculation of the possible number of trips per shift											
Number of minutes in hour	-	min	60	60	60	60	60	60	60	60	
Duration of the shift	T sh	hour	12	12	12	12	12	12	12	12	
Time, which is necessary per shift turnover, dinnertime and preventive maintenance of the truck	T sh.t.	hour	2	2	2	2	2	2	2	2	
Time required per one trip of the truck	T t	min	37	33	32	30	28	26	24	38	
Number of trips per shift	N trips	times	16.34	18.32	18.78	19.77	21.47	22.77	25.05	15.65	
*Actual number of trips per shift	-	times	16	18	18	19	21	22	25	15	
Calculation of the necessary number of trucks											
Volume of rock that should be transported per shift	Q sh	m3/shift	1182	5253	5004	5275	5043	929	4221	4906	
Number of trips per shift	N trips	times	16	18	18	19	21	22	25	15	
Bucket capacity	V b	m3	2.4	10.0	10.0	10.0	10.0	2.4	10.0	14.0	
Coefficient of volume efficiency of the bucket	K v.e.	-	1.0	1.0	1.0	1.0	1.0	1.0	1.0	1.0	
Coefficient of readiness of the truck	Kr	-	0.75	0.75	0.75	0.75	0.75	0.75	0.75	0.75	
Necessary number of buckets to fill the truck	n b	bucket	5	5	5	5	5	5	5	4	
Number of trucks	N truck	unit	8.21	7.78	7.41	7.40	6.40	4.69	4.50	7.79	
*Actual number of trucks	-	unit	9	8	8	8	7	5	5	8	

General number of equipment, units	
ESh-14-50	3
CAT 6018FS	5
CAT 375L	2
CAT 777	44
CAT 725	14

Attachment №5

Required Equipment Number Calculation in Technological Scheme with the use of Dragline Excavators

Input data	Symbols	Units	1st bench ESh-14-50 CAT 777	2nd bench ESh-14-50 CAT 777	3rd bench ESh-14-50 CAT 777	4th bench ESh-14-50 CAT 777	5th bench: Rock excavation ESh-14-50 CAT 777	5th bench: Drainage trench construction ESh-14-50 CAT 777	6th bench ESh-14-50 CAT 777	7th bench: Overburden ESh-14-50 Dumping in the mined-out space	7th bench: Ore ESh-14-50 CAT 777
Annual production on a bench											
Width of the extraction front	A e.f.	m	2111	2086	1989	1879	1765	1765	1658	1629	1613
Annual advance of the mining front	P year	m	150	150	150	150	150	35	150	150	150
Height of the bench	H	m	1	7	8	13	12	8	9	2	11.22
Volume of rock that should be mined on a bench (annual capacity)	Q year	m3	316650	2190300	2386800	3664050	3177000	494200	2238300	488700	2714679
* Annual production without trench volumes	-	m3	-	-	-	-	2682800	-	-	-	-
Number of working hours of excavator per year											
Number of working months in year	N m	mounth	12	12	12	12	12	12	12	12	12
Average number of working days in month	N d	day	30	30	30	30	30	30	30	30	30
Number of holydays per year	N h	day	8	8	8	8	8	8	8	8	8
Number of standstill days caused by whether conditions per year	N s	day	20	20	20	20	20	20	20	20	20
Number of shifts	n	shift	2	2	2	2	2	2	2	2	2
Duration of the shift	T sh	hour	12	12	12	12	12	12	12	12	12
Time, which is necessary per shift turnover, dinnertime and preventive maintenance of the equipment	T sh.t.	hour	2	2	2	2	2	2	2	2	2
Working hours per year	N h.y.	hour	6640	6640	6640	6640	6640	6640	6640	6640	6640
Production rate per hour											
Seconds in hour	-	sec	3600	3600	3600	3600	3600	3600	3600	3600	3600
Loss of time per preparation of the bench face per hour	T p.b.f.	sec	180	180	180	180	180	180	180	180	180
Loss of time for change of excavator position (movement)	T m	sec	0	0	0	0	0	0	0	0	0
Average cycle time for one bucket shoveling	T c	sec	140	140	140	140	140	140	140	140	140
Bucket capacity	V b	m3	14.0	14.0	14.0	14.0	14.0	14.0	14.0	14.0	14.0
Coefficient of volume efficiency of the bucket	K v.e.	-	1.0	1.0	1.0	1.0	1.0	1.0	1.0	1.0	1.0
Fragmentation index of the rock in the bucket	K f	-	1.2	1.2	1.2	1.2	1.2	1.2	1.2	1.2	1.2
Coefficient of readiness of excavator	K r	-	0.7	0.7	0.7	0.7	0.7	0.7	0.7	0.7	0.7
Production rate per hour	Q h	m3/hour	200	200	200	200	200	200	200	200	200
Calculation of the necessary number of excavators											
Annual production on a banch	Q year	m3	316650	2190300	2386800	3664050	2682800	494200	2238300	488700	2714679
Production per hour of the excavator	Q h	m3/hour	200	200	200	200	200	200	200	200	200
Working hours of excavator per year	N h.y.	hour	6640	6640	6640	6640	6640	6640	6640	6640	6640
Necessary number of excavators	N ex.	unit	0.24	1.65	1.80	2.77	2.03	0.37	1.69	0.37	2.05
*Actual number of excavators	-	unit	1	2	2	3	3		2	3	
Rock volume which should be transported during one shift											
Annual production on a banch	Q year	m3	316650	2190300	2386800	3664050	2682800	494200	2238300	2714679	
Coefficient of rock fragmentation	K f	-	1.2	1.2	1.2	1.2	1.2	1.2	1.2	1.2	
Duration of the shift	T sh	hour	12	12	12	12	12	12	12	12	
Time, which is necessary per shift turnover, dinnertime and preventive maintenance of the truck	T sh.t.	hour	2	2	2	2	2	2	2	2	
Working hours per year	N h.y.	hour	6640	6640	6640	6640	6640	6640	6640	6640	
Rock volume should be transported per shift	Q sh.	m3/shift	572	3958	4313	6622	4848	893	4045	4906	
Time required per one trip of the truck											
Number of minutes in hour	-	min	60	60	60	60	60	60	60	60	
Number of times to make a way to the hauling point	-	times	2	2	2	2	2	2	2	2	
Hauling distance	L h.d.	km	3.6	3.2	2.9	2.6	2.4	2.4	2	3	
Average speed of the truck	V t	km/hour	15	15	15	15	15	15	15	15	
Average cycle time for one bucket shoveling	T c	sec	140	140	140	140	140	140	140	140	
Necessary number of buckets to fill the truck	n b	bucket	4	4	4	4	4	4	4	4	
Unloading time	T un	min	5	5	5	5	5	5	5	5	
Time required per one trip of the truck	T t	min	43	40	38	35	34	34	30	38	
Calculation of the possible number of trips per shift											
Number of minutes in hour	-	min	60	60	60	60	60	60	60	60	
Duration of the shift	T sh	hour	12	12	12	12	12	12	12	12	
Time, which is necessary per shift turnover, dinnertime and preventive maintenance of the truck	T sh.t.	hour	2	2	2	2	2	2	2	2	
Time required per one trip of the truck	T t	min	43	40	38	35	34	34	30	38	
Number of trips per shift	N trips	times	13.91	15.03	15.99	17.08	17.89	17.89	19.78	15.65	
*Actual number of trips per shift	-	times	13	15	15	17	17	17	19	15	
Calculation of the necessary number of trucks											
Volume of rock that should be transported per shift	Q sh	m3/shift	572	3958	4313	6622	4848	893	4045	4906	
Number of trips per shift	N trips	times	13	15	15	17	17	17	19	15	
Bucket capacity	V b	m3	14.0	14.0	14.0	14.0	14.0	14.0	14.0	14.0	
Coefficient of volume efficiency of the bucket	K v.e.	-	1.0	1.0	1.0	1.0	1.0	1.0	1.0	1.0	
Coefficient of readiness of the truck	Kr	-	0.75	0.75	0.75	0.75	0.75	0.75	0.75	0.75	
Necessary number of buckets to fill the truck	n b	bucket	4	4	4	4	4	4	4	4	
Number of trucks	N truck	unit	1.05	6.28	6.85	9.27	6.79	1.25	5.07	7.79	
*Actual number of trucks	-	unit	2	7	7	10	7	2	6	8	

General number of equipment, units	
ESh-14-50	16
CAT 777	47

Theoretical Calculation of the Annual Production Rate of Dragline and Hydraulic Excavators and Dump Trucks

Attachment №7

Input data	Units	ESh-6.5-45	ESh-10-50	ESh-14-50	CAT 385C FS	CAT 6015 FS	CAT 6018 FS	CAT 6030 FS	ESh-14-50	CAT 6018 FS
Bucket capacity	m3	6.5	10	14	5.7	8.1	10	16.5	14	10
Coefficient of volume efficiency of the bucketa	-	0.9	0.9	0.9	0.9	0.9	0.9	0.9	0.9	0.9
Utilization factor of the excavator	-	0.7	0.7	0.7	0.7	0.7	0.7	0.7	0.7	0.7
Duration of the shift	hour	12	12	12	12	12	12	12	8	8
Time, which is necessary for shift turnover, dinnertime and preventive maintenance of the equipment	hour	2	2	2	2	2	2	2	2	2
Number of shifts	shift	2	2	2	2	2	2	2	3	3
Number of working days in a year	day	310	310	310	310	310	310	310	310	310
Average cycle time for one bucket shoveling	sec	60	60	60	40	40	40	40	60	40
Fragmentation index of the rock in the bucket	-	1.3	1.3	1.3	1.3	1.3	1.3	1.3	1.3	1.3
Annual production of the excavators	thousand m3/year	1171.8	1802.8	2523.9	1541.4	2190.4	2704.2	4461.9	2271.5	2433.7
Appropriate dump truck	-	CAT 772	CAT 773	CAT 777	CAT 770	CAT 773	CAT 777	CAT 785	CAT 777	CAT 777
Avarage hauling distance	km	2.5	2.5	2.5	2.5	2.5	2.5	2.5	2.5	2.5
Average speed of the truck	km/hour	15	15	15	15	15	15	15	15	15
Necessary number of buckets to fill the truck	bucket	4	3	4	3.5	3.5	5	4.5	4	5
Unloading time	min	5	5	5	5	5	5	5	5	5
Time required for one trip of the truck	min	29.00	28.00	29.00	27.67	27.67	28.33	28.33	29.00	28.33
Calculated number of trips during one shift	times	20.69	21.43	20.69	21.69	21.69	21.18	21.18	12.41	12.71
Actual number of trips	times	20	21	20	21	21	21	21	12	12
Annual production of one dump truck (loosened rock)	thousand m3/year	203.1	246.1	437.5	163.6	232.5	410.1	609.0	393.7	351.5

4.3 Dependence of Capital Expenditures on Annual Output of the Mine

Annual mineral output of the open pit	thousand m3/year	2,500.0	2,600.0	2,700.0	2,800.0	2,900.0	3,000.0	3,100.0	3,200.0	3,300.0	3,400.0	3,500.0	3,600	3,700	3,800	3,900	4,000	4,100	4,200	4,300	4,400	4,500	4,600	4,700	4,800	4,900	5,000
Annual overburden output	thousand m3/year	12,500.0	13,000.0	13,500.0	14,000.0	14,500.0	15,000.0	15,500.0	16,000.0	16,500.0	17,000.0	17,500.0	18,000.0	18,500.0	19,000.0	19,500.0	20,000.0	20,500.0	21,000.0	21,500.0	22,000.0	22,500.0	23,000.0	23,500.0	24,000.0	24,500.0	25,000.0
Annual mining front advance	m	139.0	145.0	150.0	156.0	162.0	167.0	173.0	178.0	184.0	189.0	195.0	201	206	212	217	223	228	234	240	245	251	256	262	267	273	279
Technological scheme with the use of bucket-wheel excavator																											
Calculated number of the bucket-wheel excavators SRs 2000	unit	0.49	0.51	0.52	0.54	0.56	0.58	0.60	0.62	0.64	0.66	0.68	0.70	0.72	0.74	0.76	0.78	0.80	0.82	0.84	0.86	0.87	0.89	0.91	0.93	0.95	0.97
Actual number of the bucket-wheel excavators SRs 2000	unit	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1
Utilization rate of the bucket-wheel excavator SRs 2000	%	48.6	50.5	52.5	54.4	56.4	58.3	60.3	62.2	64.1	66.1	68.0	70.0	71.9	73.9	75.8	77.7	79.7	81.6	83.6	85.5	87.5	89.4	91.4	93.3	95.2	97.2
Prise of one SRs 2000 bucket-wheel excavator	thousand dollars	112,410.0	112,410.0	112,410.0	112,410.0	112,410.0	112,410.0	112,410.0	112,410.0	112,410.0	112,410.0	112,410.0	112,410.0	112,410.0	112,410.0	112,410.0	112,410.0	112,410.0	112,410.0	112,410.0	112,410.0	112,410.0	112,410.0	112,410.0	112,410.0	112,410.0	112,410.0
Overall costs of the acquisition of SRs 2000	thousand dollars	112,410.0	112,410.0	112,410.0	112,410.0	112,410.0	112,410.0	112,410.0	112,410.0	112,410.0	112,410.0	112,410.0	112,410.0	112,410.0	112,410.0	112,410.0	112,410.0	112,410.0	112,410.0	112,410.0	112,410.0	112,410.0	112,410.0	112,410.0	112,410.0	112,410.0	112,410.0
Sum of unused capital expenditures on the acquisition of excavator SRs 2000	thousand dollars	57,788.2	55,603.4	53,418.5	51,233.6	49,048.8	46,863.9	44,679.0	42,494.1	40,309.3	38,124.4	35,939.5	33,754.7	31,569.8	29,384.9	27,200.1	25,015.2	22,830.3	20,645.4	18,460.6	16,275.7	14,090.8	11,906.0	9,721.1	7,536.2	5,351.3	3,166.5
Calculated number of the draglines ESh 14/50	unit	2.89	3.01	3.12	3.24	3.36	3.47	3.59	3.70	3.82	3.93	4.05	4.17	4.28	4.40	4.51	4.63	4.74	4.86	4.98	5.09	5.21	5.32	5.44	5.55	5.67	5.79
Actual number of the draglines ESh 14/50	unit	3	4	4	4	4	4	4	4	4	4	5	5	5	5	5	5	5	5	5	6	6	6	6	6	6	6
Utilization rate of the draglines ESh 14/50	%	96.4	75.2	78.1	81.0	83.9	86.8	89.7	92.6	95.5	98.4	81.0	83.3	85.6	87.9	90.3	92.6	94.9	97.2	99.5	84.9	86.8	88.7	90.6	92.6	94.5	96.4
Prise of one ESh 14/50 excavator	thousand dollars	8,100.0	8,100.0	8,100.0	8,100.0	8,100.0	8,100.0	8,100.0	8,100.0	8,100.0	8,100.0	8,100.0	8,101.0	8,102.0	8,103.0	8,104.0	8,105.0	8,106.0	8,107.0	8,108.0	8,109.0	8,110.0	8,111.0	8,112.0	8,113.0	8,114.0	8,115.0
Overall costs of the acquisition of the draglines ESh 14/50	thousand dollars	24,300.0	32,400.0	32,400.0	32,400.0	32,400.0	32,400.0	32,400.0	32,400.0	32,400.0	32,400.0	40,500.0	40,505.0	40,510.0	40,515.0	40,520.0	40,525.0	40,530.0	40,535.0	40,540.0	48,654.0	48,660.0	48,666.0	48,672.0	48,678.0	48,684.0	48,690.0
Sum of unused capital expenditures on the ESh 14/50 draglines acquisition	thousand dollars	867.9	8,030.6	7,093.3	6,156.0	5,218.7	4,281.4	3,344.1	2,406.9	1,469.6	532.3	7,695.0	6,758.5	5,821.9	4,885.0	3,947.8	3,010.4	2,072.8	1,135.0	196.9	7,367.6	6,430.1	5,492.3	4,554.3	3,616.1	2,677.6	1,738.9
Calculated number of the dump trucks CAT 777	unit	21.66	22.53	23.40	24.26	25.13	26.00	26.86	27.73	28.60	29.46	30.33	31.19	32.06	32.93	33.79	34.66	35.53	36.39	37.26	38.13	38.99	39.86	40.73	41.59	42.46	43.33
Actual number of the dump trucks CAT 777	unit	22	23	24	25	26	26	27	28	29	30	31	32	33	33	34	35	36	37	38	39	39	40	41	42	43	44
Utilization rate of the dump trucks CAT 777	%	98.5	98.0	97.5	97.1	96.7	100.0	99.5	99.0	98.6	98.2	97.8	97.5	97.2	99.8	99.4	99.0	98.7	98.4	98.1	97.8	100.0	99.7	99.3	99.0	98.7	98.5
Prise of one CAT 777 dump truck	thousand dollars	980.0	980.0	980.0	980.0	980.0	980.0	980.0	980.0	980.0	980.0	980.0	981.0	982.0	983.0	984.0	985.0	986.0	987.0	988.0	989.0	990.0	991.0	992.0	993.0	994.0	995.0
Overall costs of the acquisition of the dump trucks CAT 777	thousand dollars	21,560.0	22,540.0	23,520.0	24,500.0	25,480.0	25,480.0	26,460.0	27,440.0	28,420.0	29,400.0	30,380.0	31,392.0	32,406.0	32,439.0	33,456.0	34,475.0	35,496.0	36,519.0	37,544.0	38,571.0	38,610.0	39,640.0	40,672.0	41,706.0	42,742.0	43,780.0
Sum of unused capital expenditures on the CAT 777 dump trucks acquisition	thousand dollars	330.2	461.0	591.8	722.6	853.4	4.2	135.0	265.8	396.6	527.4	658.2	789.8	921.7	70.9	202.3	334.0	465.9	598.1	730.6	863.3	6.4	138.6	271.2	404.0	537.1	670.4
Overall capital expenditures on the main mining equipment in the technological scheme with the use of bucket-wheel excavator	thousand dollars	158,270.0	167,350.0	168,330.0	169,310.0	170,290.0	170,290.0	171,270.0	172,250.0	173,230.0	174,210.0	183,290.0	184,307	185,326	185,364	186,386	187,410	188,436	189,464	190,494	199,635	199,680	200,716	201,754	202,794	203,836	204,880
Sum of unused capital expenditures in technological scheme with the use of bucket-wheel excavator	thousand dollars	58,986.3	64,094.9	61,103.6	58,112.2	55,120.9	51,149.5	48,158.2	45,166.8	42,175.5	39,184.1	44,292.8	41,303.05	38,313.38	34,340.74	31,350.13	28,359.56	25,369.03	22,378.54	19,388.07	24,506.65	20,527.26	17,536.90	14,546.58	11,556.30	8,566.05	5,575.84
Technological scheme with the use of dragline excavators																											
Calculated number of the draglines ESh 14/50	unit	5.36	5.57	5.79	6.00	6.21	6.43	6.64	6.86	7.07	7.29	7.50	7.71	7.93	8.14	8.36	8.57	8.79	9.00	9.21	9.43	9.64	9.86	10.07	10.29	10.50	10.71
Actual number of the draglines ESh 14/50	unit	6	6	6	6	7	7	7	7	8	8	8	8	8	9	9	9	9	9	10	10	10	10	11	11	11	11
Utilization rate of the draglines ESh 14/50	%	89.3	92.9	96.4	100.0	88.8	91.8	94.9	98.0	88.4	91.1	93.8	96.4	99.1	90.5	92.9	95.2	97.6	100.0	92.1	94.3	96.4	98.6	91.6	93.5	95.5	97.4
Prise of one ESh 14/50 excavator	thousand dollars	8,100.0	8,100.0	8,100.0	8,100.0	8,100.0	8,100.0	8,100.0	8,100.0	8,100.0	8,100.0	8,100.0	8,101.0	8,102.0	8,103.0	8,104.0	8,105.0	8,106.0	8,107.0	8,108.0	8,109.0	8,110.0	8,111.0	8,112.0	8,113.0	8,114.0	8,115.0
Overall costs of the acquisition of the draglines ESh 14/50	thousand dollars	48,600.0	48,600.0	48,600.0	48,600.0	56,700.0	56,700.0	56,700.0	56,700.0	64,800.0	64,800.0	64,800.0	64,808.0	64,816.0	72,927.0	72,936.0	72,945.0	72,954.0	72,963.0	81,080.0	81,090.0	81,100.0	81,110.0	89,232.0	89,243.0	89,254.0	89,265.0
Sum of unused capital expenditures on the ESh 14/50 draglines acquisition	thousand dollars	5,207.1	3,471.4	1,735.7	-	6,364.3	4,628.6	2,892.9	1,157.1	7,521.4	5,785.7	4,050.0	2,314.6	578.7	6,945.4	5,209.7	3,473.6	1,737.0	-	6,370.6	4,633.7	2,896.4	1,158.7	7,532.6	5,795.0	4,057.0	2,318.6
Calculated number of the dump trucks CAT 777	unit	40.12	41.72	43.33	44.93	46.54	48.14	49.74	51.35	52.95	54.56	56.16	57.77	59.37	60.98	62.58	64.19	65.79	67.40	69.00	70.61	72.21	73.82	75.42	77.02	78.63	80.23
Actual number of the dump trucks CAT 777	unit	41	42	44	45	47	49	50	52	53	55	57	58	60	61	63	65	66	68	70	71	73	74	76	78	79	81
Utilization rate of the dump trucks CAT 777	%	97.8	99.3	98.5	99.8	99.0	98.2	99.5	98.7	99.9	99.2	98.5	99.6	99.0	100.0	99.3	98.7	99.7	99.1	98.6	99.4	98.9	99.8	99.2	98.7	99.5	99.1
Prise of one CAT 777 dump truck	thousand dollars	980.0	980.0	980.0	980.0	980.0	980.0	980.0	980.0	980.0	980.0	980.0	981.0	982.0	983.0	984.0	985.0	986.0	987.0	988.0	989.0	990.0	991.0	992.0	993.0	994.0	995.0
Overall costs of the acquisition of the dump trucks CAT 777	thousand dollars	40,180.0	41,160.0	43,120.0	44,100.0	46,060.0	48,020.0	49,000.0	50,960.0	51,940.0	53,900.0	55,860.0	56,898.0	58,920.0	59,963.0	61,992.0	64,025.0	65,076.0	67,116.0	69,160.0	70,219.0	72,270.0	73,334.0	75,392.0	77,454.0	78,526.0	80,595.0
Sum of unused capital expenditures on the CAT 777 dump trucks acquisition	thousand dollars	865.5	272.9	660.3	67.7	455.2	842.6	250.0	637.4	44.8	432.3	819.7	227.3	615.8	22.0	411.0	800.8	205.4	595.8	987.0	390.0	781.8	183.3	575.7	968.8	368.7	762.5
Overall capital expenditures on the main mining equipment in the technological scheme with the use of dragline excavators	thousand dollars	88,780.0	89,760.0	91,720.0	92,700.0	102,760.0	104,720.0	105,700.0	107,660.0	116,740.0	118,700.0	120,660.0	121,706.0	123,736.0	132,890.0	134,928.0	136,970.0	138,030.0	140,079.0	150,240.0	151,309.0	153,370.0	154,444.0	164,624.0	166,697.0	167,780.0	169,860.0
Sum of unused capital expenditures in technological scheme with the use of draglines	thousand dollars	6,072.6	3,744.3	2,396.0	67.7	6,819.4	5,471.2	3,142.9	1,794.6	7,566.3	6,218.0	4,869.7	2,541.9	1,194.5	6,967.4	5,620.7	4,274.4	1,942.4	595.8	7,357.6	5,023.7	3,678.2	1,342.0	8,108.2	6,763.8	4,425.7	3,081.0
Technological scheme with the use of hydraulic excavators																											
Calculated number of the hydraulic excavators CAT 6018FS	unit	1.73	1.79	1.86	1.93	2.00	2.07	2.14	2.21	2.28	2.35	2.42	2.48	2.55	2.62	2.69	2.76	2.83	2.90	2.97	3.04	3.11	3.17	3.24	3.31	3.38	3.45
Actual number of the hydraulic excavators CAT 6018FS	unit	2	2	2	2	3	3	3	3	3	3	3	3	3	3	3	3	3	3	3	4	4	4	4	4	4	4
Utilization rate of the hydraulic excavators CAT 6018FS	%	86.3	89.7	93.2	96.6	66.7	69.0	71.3	73.6	75.9	78.2	80.5	82.8	85.1	87.4	89.7	92.0	94.3	96.6	98.9	75.9	77.6	79.4	81.1	82.8	84.5	86.3
Prise of one CAT 6018FS excavator	thousand dollars	2,720.0	2,720.0	2,720.0	2,720.0	2,720.0	2,720.0	2,720.0	2,720.0	2,720.0	2,720.0	2,720.0	2,721.0	2,722.0	2,723.0	2,724.0	2,725.0	2,726.0	2,727.0	2,728.0	2,729.0	2,730.0	2,731.0	2,732.0	2,733.0	2,734.0	2,735.0
Overall costs of the acquisition of the hydraulic excavators CAT 6018FS	thousand dollars	5,440.0	5,440.0	5,440.0	5,440.0	8,160.0	8,160.0	8,160.0	8,160.0	8,160.0	8,160.0	8,160.0	8,163.00	8,166.00	8,169.00	8,172.00	8,175.00	8,178.00	8,181.00	8,184.00	10,916.00	10,920.00	10,924.00	10,928.00	10,932.00	10,936.00	10,940.00
Sum of unused capital expenditures on the CAT 6018FS hydraulic excavators acquisition	thousand dollars	748.0	560.3	372.6	185.0	2,717.3	2,529.6	2,341.9	2,154.2	1,966.6	1,778.9	1,591.2	1,404.04	1,216.73	1,029.29	841.72	654.00	466.15	278.15	90.02	2,630.76	2,443.35	2,255.81	2,068.12	1,880.30	1,692.35	1,504.25
Calculated number of the draglines ESh 14/50	unit	2.89	3.01	3.12	3.24	3.36	3.47	3.59	3.70	3.82	3.93	4.05	4.17	4.28	4.40	4.51	4.63	4.74	4.86	4.98	5.09	5.21	5.32	5.44	5.55	5.67	5.79
Actual number of the draglines ESh 14/50	unit	3	4	4	4	4	4	4	4	4	4	5	5	5	5	5	5	5	5	5	6	6	6	6	6	6	6
Utilization rate of the draglines ESh 14/50	%	96.4	75.2	78.1	81.0	83.9	86.8	89.7	92.6	95.5	98.4	81.0	83.3	85.6	87.9	90.3	92.6	94.9	97.2	99.5	84.9	86.8	88.7	90.6	92.6	94.5	96.4
Prise of one ESh 14/50 excavator	thousand dollars	8,100.0	8,100.0	8,100.0	8,100.0	8,100.0	8,100.0	8,100.0	8,100.0	8,100.0	8,100.0	8,100.0	8,101.0	8,102.0	8,103.0	8,104.0	8,105.0	8,106.0	8,107.0	8,108.0	8,109.0	8,110.0	8,111.0	8,112.0	8,113.0	8,114.0	8,115.0
Overall costs of the acquisition of the draglines ESh 14/50	thousand dollars	24,300.0	32,400.0	32,400.0	32,400.0	32,400.0	32,400.0	32,400.0	32,400.0	32,400.0	32,400.0	40,500.0	40,505.00	40,510.00	40,515.00	40,520.00	40,525.00	40,530.00	40,535.00	40,540.00	48,654.00	48,660.00	48,666.00	48,672.00	48,678.00	48,684.00	48,690.00
Sum of unused capital expenditures on the ESh 14/50 draglines acquisition	thousand dollars	867.9	8,030.6	7,093.3	6,156.0	5,218.7	4,281.4	3,344.1	2,406.9	1,469.6	532.3	7,695.0	6,758.55	5,821.87	4,884.95	3,947.81	3,010.43	2,072.82	1,134.98	196.91	7,367.61	6,430.07	5,492.31	4,554.31	3,616.08	2,677.62	1,738.93
Calculated number of the dump trucks CAT 777	unit	41.35	43.00	44.65	46.31	47.96	49.62	51.27	52.92	54.58	56.23	57.89	59.54	61.19	62.85	64.50	66.16	67.81	69.46	71.12	72.77	74.42	76.08	77.73	79.39	81.04	82.69
Actual number of the dump trucks CAT 777	unit	42	44	45	47	48	50	52	53	55	57	58	60	62	63	65	67	68	70	72	73	75	77	78	80	82	83
Utilization rate of the dump trucks CAT 777	%	98.4	97.7	99.2	98.5	99.9	99.2	98.6	99.9	99.2	98.7	99.8	99.2	98.7	99.8	99.2	98.7	99.7	99.2	98.8	99.7	99.2	98.8	99.7	99.2	98.8	99.6
Prise of one CAT 777 dump truck	thousand dollars	980.0	980.0	980.0	980.0	980.0	980.0	980.0	980.0	980.0	980.0	980.0	981.0	982.0	983.0	984.0	985.0	986.0	987.0	988.0	989.0	990.0	991.0	992.0	993.0	994.0	995.0
Overall costs of the acquisition of the dump trucks CAT 777	thousand dollars	41,160.0	43,120.0	44,100.0	46,060.0	47,040.0	49,000.0	50,960.0	51,940.0	53,900.0	55,860.0	56,840.0	58,860.00	60,884.00	61,929.00	63,960.00	65,995.00	67,048.00	69,090.00	71,136.00	72,197.00	74,250.00	76,307.00	77,376.00	79,440.00	81,508.00	82,585.00
Sum of unused capital expenditures on the CAT 777 dump trucks acquisition	thousand dollars	639.8	979.0	338.2	677.4	36.6	375.8	715.0	74.2	413.4	752.6	111.8	451.43	791.77	149.81	490.54	831.97	188.08	529.89	872.39	226.58	569.46	913.04	265.31	609.27	953.92	304.26
Overall capital expenditures on the main mining equipment in the technological scheme with the use of hydraulic excavators	thousand dollars	70,900.0	80,960.0	81,940.0	83,900.0	87,600.0	89,560.0	91,520.0	92,500.0	94,460.0	96,420.0	105,500.0	107,528.0	109,560.0	110,613.0	112,652.0	114,695.0	115,756.0	117,806.0	119,860.0	131,767.0	133,830.0	135,897.0	136,976.0	139,050.0	141,128.0	142,215.0
Sum of unused capital expenditures in technological scheme with the use of hydraulic excavators	thousand dollars	2,255.7	9,569.9	7,804.2	7,018.4	7,972.6	7,186.8	6,401.1	4,635.3	3,849.5	3,063.7	9,398.0	8,614.0	7,830.4	6,064.1	5,280.1	4,496.4	2,727.0	1,943.0	1,159.3	10,224.9	9,442.9	8,661.1	6,887.7	6,105.7	5,323.9	3,547.4

Subsection 4.4.1 Determination of the Most Optimal Transportation Distance (CAT 6018FS / CAT 777)

Attachment №8

Input data	Units	Transportation distance, km															
		1.5	1.6	1.7	1.8	1.9	2	2.1	2.2	2.3	2.4	2.5	2.6	2.7	2.8	2.9	3
Seconds in hour	sec	3600	3600	3600	3600	3600	3600	3600	3600	3600	3600	3600	3600	3600	3600	3600	3600
Loss of time for preparation of the bench face per hour	sec	300	300	300	300	300	300	300	300	300	300	300	300	300	300	300	300
Loss of time for excavator's movements per hour	sec	300	300	300	300	300	300	300	300	300	300	300	300	300	300	300	300
Average cycle time for one bucket shoveling	sec	45	45	45	45	45	45	45	45	45	45	45	45	45	45	45	45
Bucket capacity	m3	10	10	10	10	10	10	10	10	10	10	10	10	10	10	10	10
Coefficient of volume efficiency of the bucket	-	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1
Fragmentation index of the rock in the bucket	-	1.2	1.2	1.2	1.2	1.2	1.2	1.2	1.2	1.2	1.2	1.2	1.2	1.2	1.2	1.2	1.2
Production rate of the excavator per hour (in the solid)	m3/hour	555.6	555.6	555.6	555.6	555.6	555.6	555.6	555.6	555.6	555.6	555.6	555.6	555.6	555.6	555.6	555.6
Duration of the shift	hour	12	12	12	12	12	12	12	12	12	12	12	12	12	12	12	12
Time required for the shift turnover, equipment inspection and dinnertime	min	60	60	60	60	60	60	60	60	60	60	60	60	60	60	60	60
Time required for the preventive maintenance of the excavator	min	30	30	30	30	30	30	30	30	30	30	30	30	30	30	30	30
Remaining working time of the excavator	hour	10.5	10.5	10.5	10.5	10.5	10.5	10.5	10.5	10.5	10.5	10.5	10.5	10.5	10.5	10.5	10.5
Production rate of the excavator per shift (in the solid)	m3	5833.33	5833.33	5833.33	5833.33	5833.33	5833.33	5833.33	5833.33	5833.33	5833.33	5833.33	5833.33	5833.33	5833.33	5833.33	5833.33
Volume of the truck body according to producer's information	m3	64.1	64.1	64.1	64.1	64.1	64.1	64.1	64.1	64.1	64.1	64.1	64.1	64.1	64.1	64.1	64.1
Calculated number of buckets that can be loaded into the truck	bucket	6.41	6.41	6.41	6.41	6.41	6.41	6.41	6.41	6.41	6.41	6.41	6.41	6.41	6.41	6.41	6.41
Possible number of buckets that can be loaded into the truck	bucket	6	6	6	6	6	6	6	6	6	6	6	6	6	6	6	6
Rock volume that can be loaded into the dump truck	m3	60	60	60	60	60	60	60	60	60	60	60	60	60	60	60	60
Number of times to make a way to the hauling point in one trip	trip	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2
Average speed of the truck	km/hour	15	15	15	15	15	15	15	15	15	15	15	15	15	15	15	15
Time required for the positioning of the dump truck for loading	sec	20	20	20	20	20	20	20	20	20	20	20	20	20	20	20	20
Loading time	sec	270	270	270	270	270	270	270	270	270	270	270	270	270	270	270	270
Unloading time	sec	300	300	300	300	300	300	300	300	300	300	300	300	300	300	300	300
Time required for one trip of the truck	hour	0.36	0.38	0.39	0.40	0.42	0.43	0.44	0.46	0.47	0.48	0.50	0.51	0.52	0.54	0.55	0.56
Working time of the dump truck (rock transportation)	hour	10.5	10.5	10.5	10.5	10.5	10.5	10.5	10.5	10.5	10.5	10.5	10.5	10.5	10.5	10.5	10.5
Calculated number of trips of the dump truck per shift considering the transportation distance	trip	28.85	27.84	26.88	26.00	25.17	24.39	23.65	22.96	22.31	21.70	21.12	20.57	20.04	19.54	19.07	18.62
Actual number of trips of the dump truck per shift	trip	28	27	26	25	25	24	23	22	22	21	21	20	20	19	19	18
Rock volume that can be moved by one dump truck per shift	m3	1680	1620	1560	1500	1500	1440	1380	1320	1320	1260	1260	1200	1200	1140	1140	1080
Utilization rate of the dump trucks	%	100	100	100	100	100	100	100	100	100	100	100	100	100	100	100	100
Utilization rate of the excavator	%	28.80	27.77	26.74	25.71	25.71	24.69	23.66	22.63	22.63	21.60	21.60	20.57	20.57	19.54	19.54	18.51
Rock volume that can be moved by two dump trucks per shift	m3	3360	3240	3120	3000	3000	2880	2760	2640	2640	2520	2520	2400	2400	2280	2280	2160
Utilization rate of the dump trucks	%	100	100	100	100	100	100	100	100	100	100	100	100	100	100	100	100
Utilization rate of the excavator	%	57.60	55.54	53.49	51.43	51.43	49.37	47.31	45.26	45.26	43.20	43.20	41.14	41.14	39.09	39.09	37.03
Rock volume that can be moved by three dump trucks per shift	m3	5040	4860	4680	4500	4500	4320	4140	3960	3960	3780	3780	3600	3600	3420	3420	3240
Utilization rate of the dump trucks	%	100	100	100	100	100	100	100	100	100	100	100	100	100	100	100	100
Utilization rate of the excavator	%	86.40	83.31	80.23	77.14	77.14	74.06	70.97	67.89	67.89	64.80	64.80	61.71	61.71	58.63	58.63	55.54
Rock volume that can be moved by four dump trucks per shift	m3	6720	6480	6240	6000	6000	5760	5520	5280	5280	5040	5040	4800	4800	4560	4560	4320
Utilization rate of the dump trucks	%	86.81	90.02	93.48	97.22	97.22	100	100	100	100	100	100	100	100	100	100	100
Utilization rate of the excavator	%	100	100	100	100	100	98.74	94.63	90.51	90.51	86.40	86.40	82.29	82.29	78.17	78.17	74.06
Rock volume that can be moved by five dump trucks per shift	m3	8400	8100	7800	7500	7500	7200	6900	6600	6600	6300	6300	6000	6000	5700	5700	5400
Utilization rate of the dump trucks	%	69.44	72.02	74.79	77.78	77.78	81.02	84.54	88.38	88.38	92.59	92.59	97.22	97.22	100	100	100
Utilization rate of the excavator	%	100	100	100	100	100	100	100	100	100	100	100	100	100	97.71	97.71	92.57

Subsection 4.4.1 Determination of the Most Optimal Transportation Distance (Esh-14-50 / CAT 777)

Input data	Units	Transportation distance, km															
		1.5	1.6	1.7	1.8	1.9	2	2.1	2.2	2.3	2.4	2.5	2.6	2.7	2.8	2.9	3
Seconds in hour	sec	3600	3600	3600	3600	3600	3600	3600	3600	3600	3600	3600	3600	3600	3600	3600	3600
Loss of time for preparation of the bench face per hour	sec	180	180	180	180	180	180	180	180	180	180	180	180	180	180	180	180
Loss of time for excavator's movements per hour	sec	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0
Average cycle time for one bucket shoveling	sec	120	120	120	120	120	120	120	120	120	120	120	120	120	120	120	120
Bucket capacity	m3	14	14	14	14	14	14	14	14	14	14	14	14	14	14	14	14
Coefficient of volume efficiency of the bucket	-	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1
Fragmentation index of the rock in the bucket	-	1.2	1.2	1.2	1.2	1.2	1.2	1.2	1.2	1.2	1.2	1.2	1.2	1.2	1.2	1.2	1.2
Production rate of the excavator per hour (in the solid)	m3/hour	332.5	332.5	332.5	332.5	332.5	332.5	332.5	332.5	332.5	332.5	332.5	332.5	332.5	332.5	332.5	332.5
Duration of the shift	hour	12	12	12	12	12	12	12	12	12	12	12	12	12	12	12	12
Time required for the shift turnover, equipment inspection and dinnertime	min	60	60	60	60	60	60	60	60	60	60	60	60	60	60	60	60
Time required for the preventive maintenance of the excavator	min	30	30	30	30	30	30	30	30	30	30	30	30	30	30	30	30
Remaining working time of the excavator	hour	10.5	10.5	10.5	10.5	10.5	10.5	10.5	10.5	10.5	10.5	10.5	10.5	10.5	10.5	10.5	10.5
Production rate of the excavator per shift (in the solid)	m3	3491.25	3491.25	3491.25	3491.25	3491.25	3491.25	3491.25	3491.25	3491.25	3491.25	3491.25	3491.25	3491.25	3491.25	3491.25	3491.25
Volume of the truck body according to producer's information	m3	64.1	64.1	64.1	64.1	64.1	64.1	64.1	64.1	64.1	64.1	64.1	64.1	64.1	64.1	64.1	64.1
Calculated number of buckets that can be loaded into the truck	bucket	4.58	4.58	4.58	4.58	4.58	4.58	4.58	4.58	4.58	4.58	4.58	4.58	4.58	4.58	4.58	4.58
Possible number of buckets that can be loaded into the truck	bucket	4	4	4	4	4	4	4	4	4	4	4	4	4	4	4	4
Rock volume that can be loaded into the dump truck	m3	56	56	56	56	56	56	56	56	56	56	56	56	56	56	56	56
Number of times to make a way to the hauling point in one trip	trip	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2
Average speed of the truck	km/hour	15	15	15	15	15	15	15	15	15	15	15	15	15	15	15	15
Time required for the positioning of the dump truck for loading	sec	20	20	20	20	20	20	20	20	20	20	20	20	20	20	20	20
Loading time	sec	480	480	480	480	480	480	480	480	480	480	480	480	480	480	480	480
Unloading time	sec	300	300	300	300	300	300	300	300	300	300	300	300	300	300	300	300
Time required for one trip of the truck	hour	0.42	0.44	0.45	0.46	0.48	0.49	0.50	0.52	0.53	0.54	0.56	0.57	0.58	0.60	0.61	0.62
Working time of the dump truck (rock transportation)	hour	10.5	10.5	10.5	10.5	10.5	10.5	10.5	10.5	10.5	10.5	10.5	10.5	10.5	10.5	10.5	10.5
Calculated number of trips of the dump truck per shift considering the transportation distance	trip	24.87	24.11	23.39	22.72	22.08	21.48	20.91	20.37	19.85	19.36	18.90	18.46	18.03	17.63	17.24	16.88
Actual number of trips of the dump truck per shift	trip	24	24	23	22	22	21	20	20	19	19	18	18	18	17	17	16
Rock volume that can be moved by one dump truck per shift	m3	1344	1344	1288	1232	1232	1176	1120	1120	1064	1064	1008	1008	1008	952	952	896
Utilization rate of the dump trucks	%	100	100	100	100	100	100	100	100	100	100	100	100	100	100	100	100
Utilization rate of the excavator	%	38.50	38.50	36.89	35.29	35.29	33.68	32.08	32.08	30.48	30.48	28.87	28.87	28.87	27.27	27.27	25.66
Rock volume that can be moved by two dump trucks per shift	m3	2688	2688	2576	2464	2464	2352	2240	2240	2128	2128	2016	2016	2016	1904	1904	1792
Utilization rate of the dump trucks	%	100	100	100	100	100	100	100	100	100	100	100	100	100	100	100	100
Utilization rate of the excavator	%	76.99	76.99	73.78	70.58	70.58	67.37	64.16	64.16	60.95	60.95	57.74	57.74	57.74	54.54	54.54	51.33
Rock volume that can be moved by three dump trucks per shift	m3	4032	4032	3864	3696	3696	3528	3360	3360	3192	3192	3024	3024	3024	2856	2856	2688
Utilization rate of the dump trucks	%	86.59	86.59	90.35	94.46	94.46	98.96	100	100	100	100	100	100	100	100	100	100
Utilization rate of the excavator	%	100.00	100.00	100.00	100.00	100.00	100.00	96.24	96.24	91.43	91.43	86.62	86.62	86.62	81.80	81.80	76.99
Rock volume that can be moved by four dump trucks per shift	m3	5376	5376	5152	4928	4928	4704	4480	4480	4256	4256	4032	4032	4032	3808	3808	3584
Utilization rate of the dump trucks	%	64.94	64.94	67.76	70.85	70.85	74.22	77.93	77.93	82.03	82.03	86.59	86.59	86.59	91.68	91.68	97.41
Utilization rate of the excavator	%	100	100	100	100	100	100	100	100	100	100	100	100	100	100	100	100
Rock volume that can be moved by five dump trucks per shift	m3	6720	6720	6440	6160	6160	5880	5600	5600	5320	5320	5040	5040	5040	4760	4760	4480
Utilization rate of the dump trucks	%	51.95	51.95	54.21	56.68	56.68	59.38	62.34	62.34	65.63	65.63	69.27	69.27	69.27	73.35	73.35	77.93
Utilization rate of the excavator	%	100	100	100	100	100	100	100	100	100	100	100	100	100	100	100	100

Subsection 4.4.2 Selection of Optimal Dump Trucks Number to ensure Maximum Equipment Utilization

Input data	Units	Transportation distance, km																				
		1	1.1	1.2	1.3	1.4	1.5	1.6	1.7	1.8	1.9	2	2.1	2.2	2.3	2.4	2.5	2.6	2.7	2.8	2.9	3
Seconds in hour	sec	3600	3600	3600	3600	3600	3600	3600	3600	3600	3600	3600	3600	3600	3600	3600	3600	3600	3600	3600	3600	3600
Loss of time for preparation of the bench face per hour	sec	300	300	300	300	300	300	300	300	300	300	300	300	300	300	300	300	300	300	300	300	300
Loss of time for excavator's movements per hour	sec	300	300	300	300	300	300	300	300	300	300	300	300	300	300	300	300	300	300	300	300	300
Average cycle time for one bucket shoveling	sec	45	45	45	45	45	45	45	45	45	45	45	45	45	45	45	45	45	45	45	45	45
Bucket capacity	m3	10	10	10	10	10	10	10	10	10	10	10	10	10	10	10	10	10	10	10	10	10
Coefficient of volume efficiency of the bucket	-	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1
Fragmentation index of the rock in the bucket	-	1.2	1.2	1.2	1.2	1.2	1.2	1.2	1.2	1.2	1.2	1.2	1.2	1.2	1.2	1.2	1.2	1.2	1.2	1.2	1.2	1.2
Production rate of the excavator per hour (in the solid)	m3/hour	555.6	555.6	555.6	555.6	555.6	555.6	555.6	555.6	555.6	555.6	555.6	555.6	555.6	555.6	555.6	555.6	555.6	555.6	555.6	555.6	555.6
Duration of the shift	hour	12	12	12	12	12	12	12	12	12	12	12	12	12	12	12	12	12	12	12	12	12
Time required for the shift turnover, equipment inspection and dinnertime	min	60	60	60	60	60	60	60	60	60	60	60	60	60	60	60	60	60	60	60	60	60
Time required for the preventive maintenance of the excavator	min	30	30	30	30	30	30	30	30	30	30	30	30	30	30	30	30	30	30	30	30	30
Remaining working time of the excavator	hour	10.5	10.5	10.5	10.5	10.5	10.5	10.5	10.5	10.5	10.5	10.5	10.5	10.5	10.5	10.5	10.5	10.5	10.5	10.5	10.5	10.5
Production rate of the excavator per shift (in the solid)	m3	5833.33	5833.33	5833.33	5833.33	5833.33	5833.33	5833.33	5833.33	5833.33	5833.33	5833.33	5833.33	5833.33	5833.33	5833.33	5833.33	5833.33	5833.33	5833.33	5833.33	5833.33
Volume of the truck body according to producer's information	m3	64.1	64.1	64.1	64.1	64.1	64.1	64.1	64.1	64.1	64.1	64.1	64.1	64.1	64.1	64.1	64.1	64.1	64.1	64.1	64.1	64.1
Calculated number of buckets that can be loaded into the truck	bucket	6.41	6.41	6.41	6.41	6.41	6.41	6.41	6.41	6.41	6.41	6.41	6.41	6.41	6.41	6.41	6.41	6.41	6.41	6.41	6.41	6.41
Possible number of buckets that can be loaded into the truck	bucket	6	6	6	6	6	6	6	6	6	6	6	6	6	6	6	6	6	6	6	6	6
Rock volume that can be loaded into the dump truck	m3	60	60	60	60	60	60	60	60	60	60	60	60	60	60	60	60	60	60	60	60	60
Number of times to make a way to the hauling point in one trip	trip	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2
Average speed of the truck	km/hour	15	15	15	15	15	15	15	15	15	15	15	15	15	15	15	15	15	15	15	15	15
Time required for the positioning of the dump truck for loading	sec	20	20	20	20	20	20	20	20	20	20	20	20	20	20	20	20	20	20	20	20	20
Loading time	sec	270	270	270	270	270	270	270	270	270	270	270	270	270	270	270	270	270	270	270	270	270
Unloading time	sec	300	300	300	300	300	300	300	300	300	300	300	300	300	300	300	300	300	300	300	300	300
Time required for one trip of the truck	hour	0.30	0.31	0.32	0.34	0.35	0.36	0.38	0.39	0.40	0.42	0.43	0.44	0.46	0.47	0.48	0.50	0.51	0.52	0.54	0.55	0.56
Working time of the dump truck (rock transportation)	hour	10.5	10.5	10.5	10.5	10.5	10.5	10.5	10.5	10.5	10.5	10.5	10.5	10.5	10.5	10.5	10.5	10.5	10.5	10.5	10.5	10.5
Calculated number of trips of the dump truck per shift considering the transportation distance	trip	35.33	33.81	32.42	31.14	29.95	28.85	27.84	26.88	26.00	25.17	24.39	23.65	22.96	22.31	21.70	21.12	20.57	20.04	19.54	19.07	18.62
Actual number of trips of the dump truck per shift	trip	35	33	32	31	29	28	27	26	25	25	24	23	22	22	21	21	20	20	19	19	18
Rock volume that can be moved by one dump truck per shift	m3	2100	1980	1920	1860	1740	1680	1620	1560	1500	1500	1440	1380	1320	1320	1260	1260	1200	1200	1140	1140	1080
Calculated number of dump trucks to ensure the maximum possible performance of the excavator	unit	2.78	2.95	3.04	3.14	3.35	3.47	3.60	3.74	3.89	3.89	4.05	4.23	4.42	4.42	4.63	4.63	4.86	4.86	5.12	5.12	5.40
Actual number of dump trucks to ensure the maximum possible performance of the excavator (N trucks 1.1)	unit	3	3	4	4	4	4	4	4	4	4	5	5	5	5	5	5	5	5	6	6	6
Production rate of the dump trucks with maximum possible utilization of the excavator	m3	6300	5940	7680	7440	6960	6720	6480	6240	6000	6000	7200	6900	6600	6600	6300	6300	6000	6000	6840	6840	6480
Utilization rate of the dump trucks providing the maximum possible usage of the excavator (Ku 1.2)	%	92.59	98.20	75.95	78.41	83.81	86.81	90.02	93.48	97.22	97.22	81.02	84.54	88.38	88.38	92.59	92.59	97.22	97.22	85.28	85.28	90.02
Actual number of the dump trucks to ensure the maximum possible performance of the dump trucks (N trucks 2.1)	unit	2	2	3	3	3	3	3	3	3	3	4	4	4	4	4	4	4	4	5	5	5
Production rate of the dump trucks with maximum possible utilization of the dump trucks	m3	4200	3960	5760	5580	5220	5040	4860	4680	4500	4500	5760	5520	5280	5280	5040	5040	4800	4800	5700	5700	5400
Utilization rate of the excavator providing the maximum possible usage of the dump trucks (Ku 2.2)	%	72.00	67.89	98.74	95.66	89.49	86.40	83.31	80.23	77.14	77.14	98.74	94.63	90.51	90.51	86.40	86.40	82.29	82.29	97.71	97.71	92.57

Input data	Units	3.1	3.2	3.3	3.4	3.5	3.6	3.7	3.8	3.9	4	4.1	4.2	4.3	4.4	4.5	4.6	4.7	4.8	4.9	5
Seconds in hour	sec	3600	3600	3600	3600	3600	3600	3600	3600	3600	3600	3600	3600	3600	3600	3600	3600	3600	3600	3600	3600
Loss of time for preparation of the bench face per hour	sec	300	300	300	300	300	300	300	300	300	300	300	300	300	300	300	300	300	300	300	300
Loss of time for excavator's movements per hour	sec	300	300	300	300	300	300	300	300	300	300	300	300	300	300	300	300	300	300	300	300
Average cycle time for one bucket shoveling	sec	45	45	45	45	45	45	45	45	45	45	45	45	45	45	45	45	45	45	45	45
Bucket capacity	m3	10	10	10	10	10	10	10	10	10	10	10	10	10	10	10	10	10	10	10	10
Coefficient of volume efficiency of the bucket	-	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1
Fragmentation index of the rock in the bucket	-	1.2	1.2	1.2	1.2	1.2	1.2	1.2	1.2	1.2	1.2	1.2	1.2	1.2	1.2	1.2	1.2	1.2	1.2	1.2	1.2
Production rate of the excavator per hour (in the solid)	m3/hour	555.6	555.6	555.6	555.6	555.6	555.6	555.6	555.6	555.6	555.6	555.6	555.6	555.6	555.6	555.6	555.6	555.6	555.6	555.6	555.6
Duration of the shift	hour	12	12	12	12	12	12	12	12	12	12	12	12	12	12	12	12	12	12	12	12
Time required for the shift turnover, equipment inspection and dinnertime	min	60	60	60	60	60	60	60	60	60	60	60	60	60	60	60	60	60	60	60	60
Time required for the preventive maintenance of the excavator	min	30	30	30	30	30	30	30	30	30	30	30	30	30	30	30	30	30	30	30	30
Remaining working time of the excavator	hour	10.5	10.5	10.5	10.5	10.5	10.5	10.5	10.5	10.5	10.5	10.5	10.5	10.5	10.5	10.5	10.5	10.5	10.5	10.5	10.5
Production rate of the excavator per shift (in the solid)	m3	5833.33	5833.33	5833.33	5833.33	5833.33	5833.33	5833.33	5833.33	5833.33	5833.33	5833.33	5833.33	5833.33	5833.33	5833.33	5833.33	5833.33	5833.33	5833.33	5833.33
Volume of the truck body according to producer's information	m3	64.1	64.1	64.1	64.1	64.1	64.1	64.1	64.1	64.1	64.1	64.1	64.1	64.1	64.1	64.1	64.1	64.1	64.1	64.1	64.1
Calculated number of buckets that can be loaded into the truck	bucket	6.41	6.41	6.41	6.41	6.41	6.41	6.41	6.41	6.41	6.41	6.41	6.41	6.41	6.41	6.41	6.41	6.41	6.41	6.41	6.41
Possible number of buckets that can be loaded into the truck	bucket	6	6	6	6	6	6	6	6	6	6	6	6	6	6	6	6	6	6	6	6
Rock volume that can be loaded into the dump truck	m3	60	60	60	60	60	60	60	60	60	60	60	60	60	60	60	60	60	60	60	60
Number of times to make a way to the hauling point in one trip	trip	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2
Average speed of the truck	km/hour	15	15	15	15	15	15	15	15	15	15	15	15	15	15	15	15	15	15	15	15
Time required for the positioning of the dump truck for loading	sec	20	20	20	20	20	20	20	20	20	20	20	20	20	20	20	20	20	20	20	20
Loading time	sec	270	270	270	270	270	270	270	270	270	270	270	270	270	270	270	270	270	270	270	270
Unloading time	sec	300	300	300	300	300	300	300	300	300	300	300	300	300	300	300	300	300	300	300	300
Time required for one trip of the truck	hour	0.58	0.59	0.60	0.62	0.63	0.64	0.66	0.67	0.68	0.70	0.71	0.72	0.74	0.75	0.76	0.78	0.79	0.80	0.82	0.83
Working time of the dump truck (rock transportation)	hour	10.5	10.5	10.5	10.5	10.5	10.5	10.5	10.5	10.5	10.5	10.5	10.5	10.5	10.5	10.5	10.5	10.5	10.5	10.5	10.5
Calculated number of trips of the dump truck per shift considering the transportation distance	trip	18.19	17.78	17.39	17.01	16.65	16.31	15.98	15.66	15.35	15.06	14.78	14.50	14.24	13.99	13.75	13.51	13.28	13.06	12.85	12.64
Actual number of trips of the dump truck per shift	trip	18	17	17	17	16	16	15	15	15	15	14	14	14	13	13	13	13	13	12	12
Rock volume that can be moved by one dump truck per shift	m3	1080	1020	1020	1020	960	960	900	900	900	900	840	840	840	780	780	780	780	780	720	720
Calculated number of dump trucks to ensure the maximum possible performance of the excavator	unit	5.40	5.72	5.72	5.72	6.08	6.08	6.48	6.48	6.48	6.48	6.94	6.94	6.94	7.48	7.48	7.48	7.48	7.48	8.10	8.10
Actual number of dump trucks to ensure the maximum possible performance of the excavator (N trucks 1.1)	unit	6	6	6	6	7	7	7	7	7	7	7	7	7	8	8	8	8	8	9	9
Production rate of the dump trucks with maximum possible utilization of the excavator	m3	6480	6120	6120	6120	6720	6720	6300	6300	6300	6300	5880	5880	5880	6240	6240	6240	6240	6240	6480	6480
Utilization rate of the dump trucks providing the maximum possible usage of the excavator (Ku 1.2)	%	90.02	95.32	95.32	95.32	86.81	86.81	92.59	92.59	92.59	92.59	99.21	99.21	99.21	93.48	93.48	93.48	93.48	93.48	90.02	90.02
Actual number of the dump trucks to ensure the maximum possible performance of the dump trucks (N trucks 2.1)	unit	5	5	5	5	6	6	6	6	6	6	6	6	6	7	7	7	7	7	8	8
Production rate of the dump trucks with maximum possible utilization of the dump trucks	m3	5400	5100	5100	5100	5760	5760	5400	5400	5400	5400	5040	5040	5040	5460	5460	5460	5460	5460	5760	5760
Utilization rate of the excavator providing the maximum possible usage of the dump trucks (Ku 2.2)	%	92.57	87.43	87.43	87.43	98.74	98.74	92.57	92.57	92.57	92.57	86.40	86.40	86.40	93.60	93.60	93.60	93.60	93.60	98.74	98.74